NI 43-101 Technical Report

Feasibility Study

Curipamba El Domo Project

Central Ecuador

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This Report, following National Instrument 43-101 rules and guidelines, was prepared for Adventus Mining Corporation (Adventus or the Company) by DRA Global Limited (DRA). The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved in DRA services, based on:

i) Information available at the time of preparation,

ii) Data supplied by outside sources, and

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1 EXECUTIVE SUMMARY

1.1 Summary Results

Description	Unit	Value
Project Economics		
Average annual EBITDA	$M	103
Pre-tax NPV 8% / After-tax NPV 8%	$M	426 / 259
Pre-tax IRR / After-tax IRR	%	45 / 32
Undiscounted operating pre-tax cash flow / after-tax cash flow	$M	749 / 497
Production Profile		
Total tonnes of mineralise ore mined and processed	Million tonnes	6.48
Total capitalised pre-stripping tonnes	Million tonnes	15.25
Total LOM tonnes waste mined	Million tonnes	57.08
Operating strip ratio (following capitalised pre-stripping)	waste:ore	6.46
Overall Strip ratio	waste:ore	8.81
Average tonnes mined per year (waste and ore)	Million tonnes	5.15
Peak tonnes mined per year (waste and ore)	Million tonnes	11.95
Peak tonnes mineraliseore mined per year	Million tonnes	0.98
Mine life	years	10
Net average payable metal recovery to concentrates (2)		
Gold (Au)	%	51.8
Silver (Ag)	%	63.6
Copper (Cu)	%	87.5
Lead (Pb)	%	30.3
Zinc (Zn)	%	84.7
Unit Operating Costs		
LOM AISC(1)	$/lb Cu Eq.	1.26

1.2 Introduction

1.2.1 THE ISSUER

This Report has been prepared for Adventus Mining Corporation (TSX-V: ADZN, OTCQX: ADVZF) ("Adventus" or the "Company"), a mining company listed on the Toronto Stock Exchange (TSX), with headquarters at 550-220 Bay Street, Toronto, M5J 2W4, Canada, and Salazar Resources Limited ("Salazar") (TSX-V: SRL, OTCQB: SRLZF) a mining company listed on the Toronto Stock Exchange (TSX) with headquarters at Suite 1305, 1090 West Georgia St., Vancouver, BC, V6E 3V7, Canada.

Adventus and Salazar (collectively the "Partners") are in the process of advancing and evaluating the potential development of the El Domo volcanogenic massive sulphide deposit, located within the 21,537-hectare Curipamba Project (the "Project") in central Ecuador. The relevant holding company for the Curipamba properties and permits is Curimining SA ("Curimining"), an Ecuadorian subsidiary of Salazar. Curimining' s current involvement includes the overseeing of the exploration and core logging works. Curimining is mentioned throughout the report mostly when associated with site related support.

Following positive economic results from previous Preliminary Economic Assessment ("PEA") study in 2019, the Feasibility Study ("FS") was mandated by Adventus and led by DRA Global Ltd ("DRA") to further advance the engineering, design and future construction of an open pit mine and facilities to produce concentrates of copper, zinc, and lead with significant gold and silver credits.

This work has been completed as part of Adventus's option agreement with Salazar whereby Adventus may earn a 75% ownership interest in the Curipamba Project with a preferential 95% payback of future cash flows until its investment has been fully repaid.

1.2.2 PURPOSE AND SCOPE

The purpose of the FS is to review and define the optimum configuration for the mine and processing arrangement based on the latest available test work and Mineral Resource Estimates (MRE). The FS further provides engineering definition through mine design, project infrastructure definition and optimised operations descriptions. Following completion of the engineering deliverables, a capital and operating cost estimate was prepared as well as a subsequent economic evaluation to determine the Project's viability. The definitions are followed by estimation and confirmation of project economics. The FS is based on a Class 3 type estimate as per the Association for the Advancement of Cost Engineering (AACE) Recommended Practice 47R-11 with a target accuracy of ±15%.

Additional to the FS, a preliminary PEA for underground mining following completion of the open pit mining operation, as summarised in the Adventus press release of October 26, 2021, is presented as an opportunity study in Section 24 of this Report. This PEA is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be categorized as mineral reserves.

1.3 Terms of Reference

1.3.1 Effective Date

This Report is considered effective as of October 26, 2021 and is in support of the Adventus's press release, dated October 26, 2021, entitled "*Adventus and Salazar Announce Feasibility Study Results and Updated Mineral Resources for the Curipamba Copper-Gold Project*."

1.3.2 Units and Currencies

In this Report, all currency amounts are in US Dollars ("**USD**" or "**$**") unless otherwise stated. Quantities are generally stated in *Système international d'unités* ("**SI**") metrics units, the standard Canadian and international practices, including metric tonne ("**tonne**", "**t**") for weight, and kilometre ("**km**") or metre ("**m**") for distances.

1.4 Property Description and Location

The 21,537-hectare Curipamba Project is located in Ecuador approximately 150 km south-southwest of the capital city, Quito, and approximately 150 km north-northeast of Guayaquil in the provinces of Bolivar and Los Rios. The closest town to the Project is Ventanas, which is approximately 20 km to the southwest and, in 2010, had a population of approximately 38,000 people.

1.5 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

1.5.1 Accessibility

International access to Ecuador is primarily through the airports in Guayaquil or Quito with daily flights to many international destinations. Road access to the area is excellent along paved roads, which branch off at Ventanas and Zapotal from Highway 25 that connects Quito and Guayaquil. Driving time from Guayaquil to the Project is approximately 2.5 hrs. Numerous well-maintained gravel roads provide access throughout most of the Project area, especially in the resource area. Salazar and Adventus have made improvements to certain gravel roads as part of the community outreach programs as well as to improve general access to the area for exploration and drill staff. Certain areas in the northern part of the Project can only be reached by mule or on foot.

1.5.2 Climate

The climate at the Project is tropical, humid, and hot most of the year. The wet season lasts from December to May, with the rest of the year considered to be the dry season. The average annual rainfall ranges from 2,200 mm to 2,500 mm, with most of the precipitation falling during the wet season. The climate has little effect on the operating season and exploration activities can be carried out year-round.

1.5.3 Local Resources

The Project area is near the towns of Ventanas, Quevedo, and Babahayo, as well as a number of smaller villages from where a general labour force and non-specialized supplies can be sourced. The local economy is largely agricultural in nature, and there are no large gold or base metal mines operating in this part of Ecuador. However, the Project borders a privately owned open-pit kaolinite clay mine at its southern boundary. Specialized contractors, skilled labour, heavy mining equipment, and other mining and exploration specific items required for the Project are likely to be acquired from outside the local region.

1.5.4 Infrastructure

There is basic infrastructure in the Project area such as good road access and household electricity (110V). The national power grid, with access to higher voltage supply than 110V, is within 20 km of the El Domo area in the Echeandía Canton.

1.5.5 Physiography

The Project is located in the transition zone between the Western Mountain Range (Cordillera Occidental) and the adjacent coastal lowlands. The physiography is characterized by floodplains to the west and moderate to steep-sloped hills to the east, with elevations ranging from 100 Masl to 1,000 Masl in less than seven (7) km of horizontal distance.

Vegetation in the area consists of plantations of banana, cacao, and oranges, cleared pastures for cattle, and forests.

Local drainage is provided by small rivers off the west side of the foothills. Primary drainage is through the Quevedo River that drains into the Babahayo River, which ultimately empties into the Gulf of Guayaquil.

1.6 History

The exploration history of the Project dates back to 1991, when the first reported exploration activity occurred. In 1991, RTZ Mining PLC Inc. (RTZ) conducted a regional stream sediment reconnaissance survey near the Project, collecting 548 samples. Results from this survey were in the public domain by 2004.

The Las Naves concessions were obtained by Mr. Leiva Ivan Santillan from the government in 2003. Subsequently, in 2005, he transferred the properties to Amlatminas, a private Ecuadorian company owned by Mr. Salazar.

In 2004, Mr. Salazar and Mr. Geovani staked 16 claims comprising the original property. The claims were held under Amlatminas. In September 2006, those claims were transferred to Salazar, a company at the time owned by Mr. Salazar and Mr. Acosta. Mr. Salazar and Mr. Acosta subsequently agreed to sell their shares in Salazar to Consolidated Kookaburra Resources Ltd (Consolidated Kookaburra). In March 2007, Consolidated Kookaburra changed its name to Salazar.

Between late 2007 and April 2008, initial core drilling (Phase I) was completed. A total of 51 core boreholes for 10,003 m tested 11 target areas (Buckle, 2009). Borehole CURI-39 intersected 12.22 m of massive sulphide mineralization at 1.20% Cu, 4.54% Zn, 3.62 g/t Au, and 51.89 g/t Ag at El Domo in February 2008. Between 2007 and 2008. Curimining also completed stream sediment sampling, consisting of 24 samples.

Between September 2010 and August 2011, Curimining completed a third drill program (Phase III drilling) comprising 84 core boreholes for a total of 15,582.9 m. Drilling was focused on the El Domo deposit.

Between August 2011 and April 2012, Curimining completed a fourth drilling program (Phase IV drilling), again targeting the El Domo deposit. The program comprised 51 core boreholes for a total of 10,248.8 m.

In 2015, Salazar requested that, according to the Mining Law, the exploration status be upgraded to Advanced Exploration. This change in status was granted by the government. As part of the status change, Salazar relinquished certain parts of the property, resulting in a slightly smaller, overall tenement.

Between February 2016 and September 2017, Curimining completed a fifth drilling program (Phase V drilling), comprising 33 core boreholes for a total of 9,757.4 m. The drilling focused on the El Domo deposit, specifically on the eastern edge of the massive sulphide mineralization, as well as on mineralization along the southwestern edge of known massive sulphide mineralization.

In 2018 and 2019, Curimining completed a Phase VI drilling campaign comprising of 100 core boreholes totalling 18,944 m. The drilling focused on in-fill drilling of the El Domo deposit in order to upgrade the classification of the Mineral Resource estimate.

1.7 Geological Setting and Mineralisation

The Project is located in the Macuchi Terrane, a volcano sedimentary island arc sequence that is part of an assemblage of accreted terranes that formed between the Late Jurassic and Eocene along the western edge of South America. The namesake Macuchi Group represents an intra-oceanic island arc volcanic sequence comprising predominately volcaniclastic and epiclastic rocks, including lithic-rich sandstone and breccia with accessory siltstone and chemical sediments, as well as basaltic and andesitic domes and flows. The Project is hosted in a volcanic pile comprising a basal rhyodacite unit overlain by two interfingering volcaniclastic sequences, and two coherent younger lithofacies, which intruded the sequence in both the north and south of the property. Mineralization is primarily located along the contact between a rhyodacite and volcanoclastic rocks.

The El Domo deposit is a gold-rich, polymetallic VMS deposit. Mineralization is largely flat-lying, stratiform and stratabound and occurs in one main massive sulphide lens, a directly overlying talus, or breccia zone, and a number of smaller, mineralised lenses primarily in the footwall of the main lens. The geology is complicated by a number of sub-vertical faults that offset the strata by up to approximately 50 m vertically. The deposit has a lateral extent of approximately 1,300 m by 1,100 m.

Mineralization can be divided into five types, where sphalerite, chalcopyrite, and pyrite are the principal sulphide minerals:

1. Massive sulphides with indistinct texture. In some places, a fragmental texture can be seen within the sulphides, suggesting that they may be formed by the replacement of lapilli tuff.
2. Sulphide-altered lapilli tuffs and peperites.
3. Transported sulphide fragments within polymictic lapilli tuffs.
4. Sulphide “pseudo”-fragments within polymictic lapilli tuffs.
5. Rare, thinly laminated siliceous chert with banded sulphides.

Gold was identified within sphalerite + galena + barite mineralization, where it occurs as minute inclusions in sphalerite. Accessory minerals include galena, tennantite/tetrahedrite, covellite, chalcocyanite, and barite, with barite being the principal gangue mineral.

1.8 Exploration Work

In early 2020, Adventus implemented a target generation initiative (TGI) for the Curipamba Project. The goal of the TGI was to synthesize the airborne MobileMT survey from 2019 with the historical work on the property. The magnetotelluric (MT) and magnetic data were examined in conjunction with historical drilling, mapping, prospecting, and soil and stream sediment sampling data. A total of 15 priority targets were identified for further investigation as detailed in a January 21, 2020 news release.

A program of mapping and prospecting was undertaken during the 2020/2021 field program. A total of 124 rock/chip samples were collected for analysis. Most samples were collected in the area around the Agua Santa target, located 4.5 km southwest of the El Domo deposit, with scattered coverage near other targets identified by the TGI. Results from Agua Santa returned a maximum of 14.7% Cu, with eight of 83 samples >1% Cu. They also returned a maximum of 11.4 g/t Au, with eight of 83 samples >1 g/t Au. Six samples returned >25% Zn, with 19 of 83 samples >1% Zn. A maximum of 120 g/t Ag was returned, with 12 of 83 samples >10 g/t Ag. Only one sample returned >1% Pb (1.2%). There were no strongly anomalous results from Cu, Au, Zn, Ag, or Pb from the 41 samples collected elsewhere on the property.

A soil sampling program was also completed in 2020/2021. A total of 639 samples were collected for analysis. The majority of samples collected were in the Agua Santa and surrounding area (569 samples). Results show a strong anomalous trend extending south from Agua Santa. The trend ends to the north of Agua Santa. There are some strong isolated anomalies to the east. The other area of focus was the La Vaquera target and surrounding area, approximately 6 km south-southwest of El Domo deposit, where a total of 70 samples were collected. There is a strong soil anomaly that justifies following up with a tighter grid spacing. A program of regional drilling was undertaken between November 14, 2020 to January 10, 2021, and April 22 to September 24, 2021. A total of 18 holes were drilled on priority targets identified during the TGI; a total of 5,582 metres were drilled.

1.9 Mineral Processing and Metallurgical Testing

Metallurgical test work for Curipamba and the El Domo deposit included historical work completed during 2009 – 2019, and testwork completed by Base Metallurgical Laboratories Ltd. (BML) during 2020 – 2021 under DRA supervision.

The work completed by BML during 2020 – 2021 was done on ore samples from previous testing campaign, and material sampled during 2020-2021 drilling program completed by Adventus.

The 2020-2021 testing program included assays for payable metals and deleterious elements, comminution and flotation tests, tailings settling and pressure filtration, gravity, leach and

Sulphidization-Acidification-Recycle-Thickening (SART) tests on the composites and lithological variability samples.

The work also included mineralogical characterization of the ore and flotation products, gold mineralogy studies, and assessment of the ore amenability to ore sorting and dense media separation.

In QP's opinion:

- The test work conducted sufficiently supports the feasibility study process design and identified the physical and metallurgical properties of the orebody ore and ore variability within the deposit.
- The drill core material used for the test work is representative of the deposit and is well documented with detailed records appended in the relevant test work reports.
- Metallurgical testing data supports the metal recovery assumptions contained in the LOM plans and metal recovery projections.

1.10 Mineral Resources Estimate

The Mineral Resource estimate presented in this report is a third estimate prepared by SLR (acquired Roscoe Postle Associates Inc. in 2019) and is a reasonable representation of the Mineral Resources of the Project at the current level of sampling. The Mineral Resources conform to CIM (2014) definitions and are reported in accordance with the NI 43-101. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Open pit Mineral Resources have been constrained within a Whittle pit shell. A summary of the Mineral Resources is presented in Table 1.1.

Table 1.1– Mineral Resource Statement

Resource Category	Tonnes (Mt)	Grade					Contained Metal				
		Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu (kt)	Pb (kt)	Zn (kt)	Au (koz)	Ag (koz)
Open Pit Resources											
Measured	3.2	2.61	0.2	2.50	3.03	45	84.9	7.7	81.1	316	4,704
Indicated	3.8	1.38	0.3	2.77	2.29	52	52.6	11.3	105.2	280	6,370
M+I	**7.1**	**1.95**	**0.3**	**2.64**	**2.63**	**49**	**137.5**	**19.0**	**186.3**	**596**	**11,074**
Inferred	0.3	0.34	0.2	1.01	1.34	39	1.2	0.7	3.5	15	430
Underground Resources											
Indicated	1.9	2.72	0.14	2.38	1.37	31	51.9	2.6	45.4	84	1,895
Inferred	0.8	2.31	0.11	2.68	1.74	29	17.3	0.8	20.1	42	688
Total Mineral Resources											
Measured	3.2	2.61	0.2	2.50	3.03	45	84.9	7.7	81.1	316	4,704
Indicated	5.7	1.83	0.24	2.64	1.98	45	104.5	13.9	150.6	364	8,265
M+I	**9.0**	**2.11**	**0.24**	**2.59**	**2.36**	**45**	**189.4**	**21.6**	**231.7**	**680**	**12,969**
Inferred	1.1	1.72	0.14	2.18	1.62	32	18.5	1.5	23.6	57	1,118

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are reported above an NSR cut-off value of US$29/t for potential open pit Mineral Resources and the underground portion of the 2021 Mineral Resources are reported with mining shapes which were generated using an NSR cut-off value of US$105/t NSR.
3. The NSR value is based on estimated metallurgical recoveries, assumed metal prices, and smelter terms, which include payable factors treatment charges, penalties, and refining charges.
4. Mineral Resources are estimated using the metal price assumptions: US$4.00/lb Cu, US$1.05/lb Pb, US$1.30/lb Zn, US$1,800/oz Au, and US$24/oz Ag.
5. Metallurgical recovery assumptions were based on three mineral types defined by the metal ratio Cu/(Pb+Zn):
   a. Zinc Mineral (Cu/(Pb+Zn) <0.33): 86% Cu, 90% Pb, 97% Zn, 68% Au, and 78% Ag;
   b. Mixed Cu/Zn Mineral (0.33≤ Cu/(Pb+Zn) ≤3.0): 86% Cu, 82% Pb, 95% Zn, 55% Au, and 67% Ag;
   c. Copper Mineral (Cu/(Pb+Zn) >3.0): 80% Cu, 37% Pb, 36% Zn, 14% Au, and 29% Ag;
6. NSR factors were also based on the metal ratio Cu/(Pb+Zn):
   a. Zinc Mineral (Cu/(Pb+Zn) <0.33): 53.41 US$/% Cu, 7.99 US$/% Pb, 13.47 US$/% Zn, 30.91 US$/g Au, and 0.39 US$/g Ag,
   b. Mixed Cu/Zn Mineral (0.33≤ Cu/(Pb+Zn) ≤3.0): 58.99 US$/% Cu, 7.05 US$/% Pb ,13.41 US$/% Zn, 25.12 US$/g Au, and 0.34 US$/g Ag;
   c. Copper Mineral (Cu/(Pb+Zn) >3.0): 57.83 US$/% Cu, 6.84 US$/g Au, and 0.19 US$/g Ag.
7. Bulk density interpolated on a block per block basis using assayed value, the correlation between measured density values and iron content, and base metal grade.
8. Mineral Resources are inclusive of Mineral Reserves.
9. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
10. The underground portion of the Mineral Resources are reported within underground reporting shapes and include low grade blocks falling within the shapes.
11. Numbers may not add due to rounding.

1.11 Mineral Reserve Estimate

The Minerals Reserves for the Curipamba Project were estimated using HxGN Mine Plan's MSOPit module to determine the ultimate pit limits. Only Measured and Indicated Mineral Resources categories were used in the Mineral Reserves Estimate. A standard open pit truck and shovel operation was assumed, with a 0.67 Mt/year throughput at the mill. An optimised pit shell was generated and used to create a final pit design, including access ramps and ensuring the minimum mining width is respected. The access ramps were design with a 12 m overall width.

The Mineral Reserves are estimates at 6.48 Mt of proven Proven and Probable Reserves with an average grade of 2.52 g/t Au, 45.69 g/t Ag, 0.25% Pb, 2.49% Zn, and 1.93% Cu (and average NSR of 185.4 $/t). To access the Mineral Reserves, a total of 55.34 Mt of waste will need to be extracted, resulting in an 8.54 stripping ratio. A summary of the Mineral Reserve Estimate is presented in Table 1.2

Table 1.2 – Mineral Reserve

Classification	Ore Type	Tonnage (kt)	NSR ($/t)	Grades					Metal				
				Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Cu (%)	Au (koz)	Ag (koz)	Pb (kt)	Zn (kt)	Cu (kt)
Proven Reserves	High Zn	370.8	222.5	3.56	67.42	0.60	4.96	0.92	42.4	803.7	2.2	18.4	3.4
	Mixed Zn/Cu	1,676.9	227.3	3.15	48.77	0.23	2.66	2.30	169.8	2,629.4	3.9	44.6	38.6
	High Cu	1,087.9	173.2	2.09	21.21	0.06	0.83	3.35	73.1	741.8	0.7	9.0	36.4
	Subtotal	*3,135.5*	*207.9*	*2.83*	*41.42*	*0.21*	*2.30*	*2.50*	*285.4*	*4,174.9*	*6.7*	*72.0*	*78.4*
Probable Reserves	High Zn	999.3	197.6	2.92	72.66	0.51	4.47	0.86	93.8	2,334.5	5.1	44.7	8.6
	Mixed Zn/Cu	2,068.0	149.7	1.94	41.98	0.20	2.04	1.44	129.0	2,791.1	4.1	42.2	29.8
	High Cu	275.5	152.9	1.89	24.41	0.07	0.91	2.93	16.7	216.2	0.2	2.5	8.1
	Subtotal	*3,342.8*	*164.3*	*2.23*	*49.70*	*0.29*	*2.68*	*1.39*	*239.5*	*5,341.9*	*9.4*	*89.4*	*46.4*
Total Reserves	**High Zn**	**1,370.1**	**204.3**	**3.09**	**71.24**	**0.53**	**4.61**	**0.88**	**136.1**	**3,138.1**	**7.3**	**63.2**	**12.1**
	Mixed Zn/Cu	**3,744.9**	**184.5**	**2.48**	**45.02**	**0.22**	**2.32**	**1.82**	**298.6**	**5,420.4**	**8.2**	**86.9**	**68.2**
	High Cu	**1,363.4**	**169.1**	**2.05**	**21.86**	**0.06**	**0.84**	**3.27**	**89.9**	**958.2**	**0.8**	**11.5**	**44.6**
	Total	**6,478.4**	**185.4**	**2.52**	**45.69**	**0.25**	**2.49**	**1.93**	**524.6**	**9,516.7**	**16.2**	**161.4**	**124.9**

Notes:
1. The effective date of the Mineral Reserve Estimate is October 26, 2021.
2. Mineral Reserves are reported in accordance with CIM guidelines.
3. An NSR cut-off grade of $32.99 was used for all material.
4. Mineral reserves were estimated at a gold price of 1,630 $/oz, a silver price of 21.00 $/oz, a lead price of 0.92 $/lb, a zinc price of 1.16 $/lb, and a copper price of 3.31 $/lb; they include modifying factors related to mining cost, dilution, mine recovery, process recoveries and costs, G&A, royalties, and rehabilitation costs.
5. Figures have been rounded to an appropriate level of precision for the reporting of Mineral Reserves.
6. Due to rounding, some columns or rows may not compute exactly as shown.
7. The Mineral Reserves are stated as dry tonnes processed at the crusher.
8. Tonnages are presented in metric tonnes.
9. Ounces are presented in troy ounces.

1.12 Mining Methods

1.12.1 Geotechnical Parameters

The open pit slope design parameters for the Project are based on geotechnical site investigations (completed by DRA, Adventus and Curimining), available local and regional geological data, drilling data and well-established geotechnical design methods

DRA used site observations, data, and statistical analyses to define geotechnical domains and select representative geomechanical properties. An appropriate quantity of quality data was collected to characterise the geological units of the study area and support FS-level slope designs. The Curipamba pit has been divided into different geotechnical domains based on lithologies, the different structural components present in the area, and the anticipated main orientations of the proposed pit walls

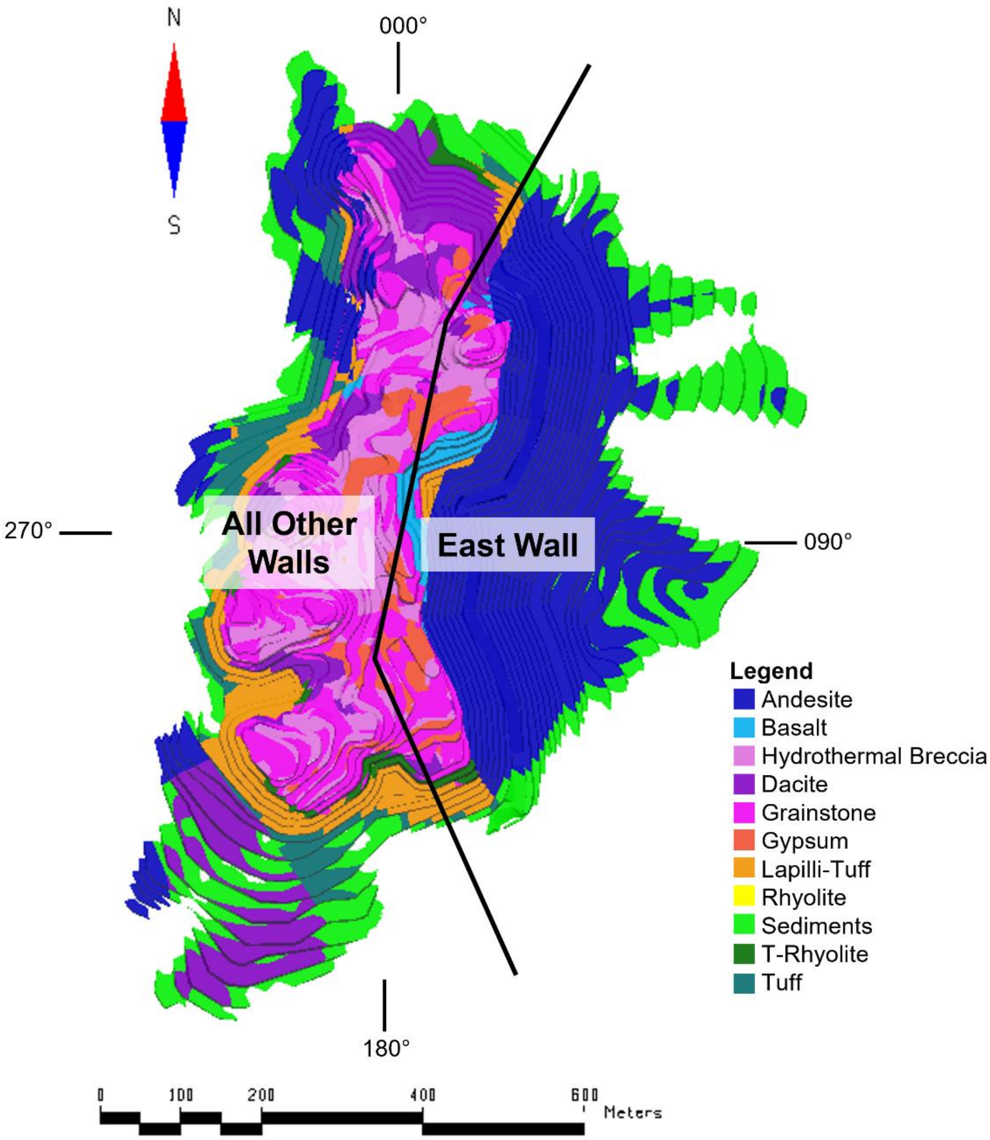
Recommended inter-ramp angles vary between 46.1° and 54°, based on wall orientation, overall wall height, geotechnical domain, and controls on slope stability. Inter-ramp slope heights are limited to 70 m, after which a geotechnical berm (or ramp) with a minimum width of 12 m or 14 m, depending on the area, is required. The inter-ramp height limits and geotechnical berms provide flexibility in the mine plan to mitigate potential slope instability, access slope monitoring installations, working space for in-pit wells, drains and other water management infrastructure. The Curipamba pit slope designs are presented in Table 1.3 and Figure 1.1.

Table 1.3 – Curipamba Pit Slope Design

Domain	Lithology	Wall Facing Direction	BFA	Bench Height	Planned Berm Width	Design IRA	Stack Height	Geotechnical Berm Width	Overall Slope Angle
		(°)	(°)	(m)	(m)	(°)	(m)	(m)	(°)
East Wall	Andesite	50-140	80	10	5.5	54	70	14	50.7
	Tuff	090-130	80	10	6	52.2	70	14	49.2
	Lapilli Tuff								
All other Walls	Andesite	260-280	70	10	5.5	47.6	70	12	45.5
	All other lithologies	180-320	70	10	6	46.1	70	12	44.2
Pit's Edge	Overburden and Saprolite		75	5	12	20.5		20 offset from edge of pit	

Notes:
1. The slope angles are for fully drained slopes
2. Pit wall directions are based on the following directions: North, 0°; West, 90°; South, 180°; East, 270°
3. The East wall was used to evaluate the maximum wall height, which was used to determine the stack height
4. The design IRA is a toe-to-toe angle
5. Pre-splitting of the final pit wall to maximize effective berm width and limit breakback, as well as careful blasting, are required to minimize damage to the final pit wall slopes
6. The 20 m offset for the overburden and saprolite is intended to allow equipment access to remove any sloughed material

Figure 1.1 – Plan View of the Ultimate Pit Showing the Geotechnical Domains



Source: DRA, 2021

1.12.2 Open Pit Mining

Conventional open pit mining with trucks, hydraulic shovels and loaders was chosen for the Project. Ore material from the pit will be loaded onto a truck by a loader and transported to either the mill, or to the ore stockpile where it will later be rehandled and sent to the mill. Waste material will be loaded on trucks by an excavator and transported to either the overburden stockpile, the saprolite stockpile, the tailings storage facility, or the waste rock facility, as appropriate.

The pit will be mined in five (5) phases over the 10-year mine life, with an additional 1.5 years of pre-production. The mine will be operated by a contractor seven (7) days a week, 24 hours a day in two (2) shifts of twelve (12) hours. Two (2) weeks of weather delays are considered; therefore, the mine will be operating 350 days a year.

The 666 kt/year ore requirement at the mill guides the mine plan. A total of 6.48 Mt of ore is mined and processed over the life-of-mine. A summarised annual production schedule is presented in Table 1.4.

Table 1.4 –Annual Production Plan

Description	Unit	Material Movement by Year												
		-2	-1	1	2	3	4	5	6	7	8	9	10	*Total*
Pit to Mill	kt	0	0	494	362	545	537	575	412	575	556	523	162	*4,742*
Pit to Ore Stockpile	kt	0	6	485	100	90	139	186	112	148	319	154	0	*1,737*
Ore Stockpile to Mill (Rehandle)	kt	0	0	95	304	121	129	91	254	91	110	143	399	*1,737*
Pit to Overburden Stockpile	kt	111	131	66	43	56	68	3	0	0	0	0	0	*478*
Pit to Saprolite Stockpile	kt	2,608	4,493	2,471	1,488	2,263	2,135	2,322	1,150	570	215	241	24	*19,979*
Pit to Tailings Storage Facility	kt	1,355	5,066	2,152	1,120	1,577	1,621	2,331	2,383	1,105	500	477	44	*19,730*
Pit to Waste Rock Facility	kt	349	2,254	2,388	3,084	1,848	1,870	992	1,981	321	31	34	3	*15,156*
Total	***kt***	***4,420***	***11,950***	***8,150***	***6,500***	***6,500***	***6,500***	***6,500***	***6,292***	***2,810***	***1,731***	***1,572***	***633***	***63,559***

Due to rounding, some columns or rows may not compute exactly as shown

1.13 Recovery Methods

The Curipamba concentrator operation was designed to process 666,000 tonnes per year of ore into copper, lead and zinc concentrates. The design criteria was developed based upon the test work results, the Life of Mine (LOM) plan, process design calculations, and vendor budget quotations.

The process definition has been completed based on results of the of techno-economical trade-off studies which resulted in the following plant setup based on the financial assessment and risks evaluation:

- Flotation concentrator producing copper, lead and zinc concentrates;
- Crushing area - two stage crushing with skid built direct powered crushing and screening modules and modular belt conveyors. Use of the front end loaders to feed the grinding area instead of the crushed ore stockpile tunnel or crushed ore silo;
- Grinding circuit configuration of a single stage grate discharge ball mill in closed circuit with hydrocyclones was chosen based on the ore comminution properties, plant throughput required, and DRA's previous comminution design experience.
- Use of conventional flotation cells for the flotation circuits.

The process flowsheet is based on proven technology operated by numerous massive sulphide ore processing operations around the world.

The plant consists of the following process circuits:

- Primary and secondary crushing;
- Grinding;
- Bulk flotation and regrind;
- Copper, lead and zinc flotation;
- Copper, zinc and lead, concentrate dewatering, storage and loadout;
- Tailings management;
- Reagents and consumables;
- Water and air services;

The mineral processing plant is designed to operate 365 d/y including planned maintenance time. The crushing area have been specified with an operating availability of 70%, equivalent to 17 h/d of operation. The concentrator plant has been designed for an availability of 92%, 24 h/d.

1.14 Project Infrastructure

Project infrastructure related to the mine, process plant and support services were designed and estimated to FS level.

Refer to Figure 1.2 for a layout of the site and mentioned facilities.

1.14.1 Project Roads

The site requires road connection to the main Ecuadorian highway system in order to deliver concentrate and receive consumable deliveries with majority of traffic flowing to the port city area of Guayaquil 150 km away. Site access will be improved through the upgrading of public road and bridges for a specific roads section between the villages of El Pasaje B, past El Congreso, up to the mine property access point. From the access point, a new access road section will be built that connects to the site haul road network

Site roads were designed along established road corridors to limit impact as far as practicable. New site roads are required to access the distributed terraces and gain access to the El Silencio valley floor in order to construct the waste rock and tailings facilities. Haul roads and site roads will typically be 12 m and 8 m wide respectively. A shared traffic model has been adopted due to the terrain limitations and is deemed a manageable operational risk.

1.14.2 Site Terraces and Infrastructure

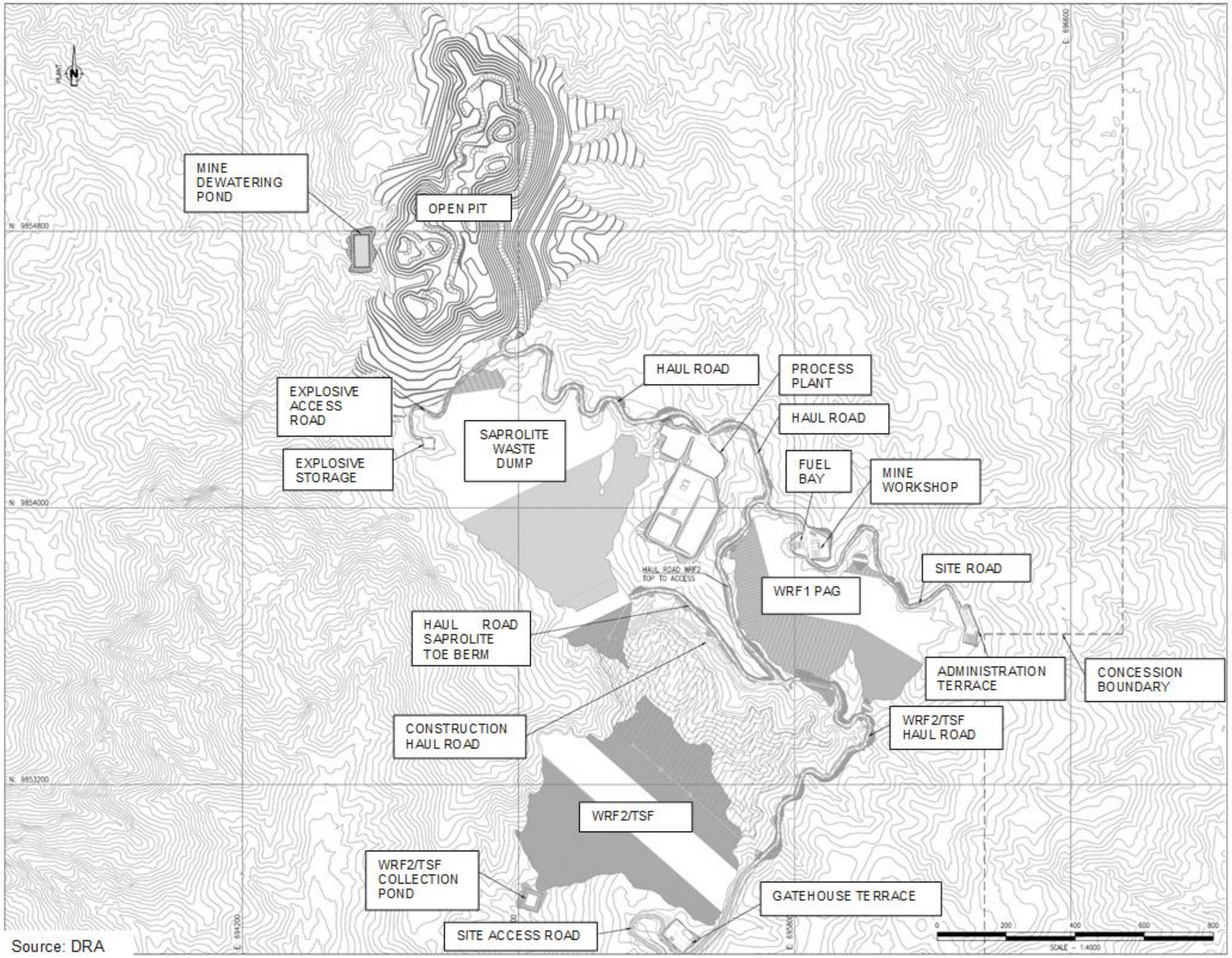
The steep terrain proved challenging in establishing terraces without major earthmoving and engineering fill work, resulting a distributed terrace approach. The Project infrastructure is distributed across three (3) main terraces.

The mine workshop terrace is located in a relatively flat terrain area that suits the service and logistics requirement of the mine haul fleet and contains the main mine workshop, a wash bay, fuel storage bay, and fuelling area.

The administration terrace will support an administration and training building, and a camp that will be erected to provide temporary accommodation during the construction phase and converted to offices prior to the start of operations.

The Gatehouse terrace is located near the Tailings Storage Facility (TSF) where the access road connects to the Project site. The Gatehouse terrace will serve as the main access point to the Project site and will house the gatehouse and security offices, as well as a delivery warehouse and the tailings facility water treatment plant.

Figure 1.2 – Overall Site Layout



Source: DRA

The process plant terrace supports the process equipment required to produce saleable concentrates from the Run-of-Mine (ROM) ore. The terrace has an approximate footprint of 150 m by 200 m divided into three (3) stepped sub-terraces. The process related structures are supported with further buildings: workshops, control rooms, electrical motor control rooms, process administration, reagent mixing and storage, concentrate storage and a laboratory. Process related structures are mostly uncovered to allow for mobile equipment maintenance access. Terrace logistics for consumable reagents and concentrate export has been considered. The terrace has a single vehicular entry and access point due to terrain restrictions. The process terrace will be constructed through the removal of mostly saprolite material. Engineering fills have been avoided where possible due to their complexity in the steep terrain.

A small terrace for explosive storage has been established in a suitable location near the open pit mine.

1.14.3 Camps, Accommodation and Catering

The construction phase of the Project will see the peak requirement for personnel and contractor accommodation. As part of early works preparation, Adventus purchased a one hundred (100) man prefabricated accommodation camp with sanitary facilities during Q2, 2021 (Adventus new release, dated July 14, 2021). It is envisaged to establish this camp to serve as limited site accommodation during the early works and construction phase for essential personnel. Each contractor have been requested to provide suitable temporary accommodation and catering as part of their construction quotations for peak periods of construction where overflow capacity is required. Most contractors have agreed that offsite accommodation will be preferred due to the site terrain limitations. Contractors aim to establish temporary accommodation on rented properties for the duration of their scope. The Owner's teams will be housed in locally available accommodation during the construction phase that Adventus will rent and maintain. The current Curimining canteen facilities may be expanded to accommodate the construction workforce.

At the commencement of operations, the purchased one hundred (100) man accommodation camp buildings will either be converted to office and administration buildings, or remain as an accommodation camp for the production period. The camp will require augmentation with a canteen and catering facility.

1.14.4 Site Power Supply

Adventus has opted for grid power supply to the Project from the main Ecuadorian public utility. Preliminary applications with the relevant utilities have progressed to secure the power allocation required for the Project totalling 10MW of consumed power. Approximately 80% to 90% of Ecuador's power grid is supplied from hydroelectric supply sources.

The nearest suitable transmission line connection point is located seven (7) km to the West of the Project site. Adventus will be required to construct a suitable 69 kV overhead line to the Project site. Adventus retained Engywork, a local Ecuadorian design firm, to design and cost the 7 km Project site power line for which an allowance has been included in the Capex.

Additional to the new site power line, the local grid powerline requires upgrading to manage the increased load demand. The Echeandia- Las Naves power line will require upgrading from 13.8 kV to 69 kV to allow Adventus to connect to the 69 kV infrastructure.

Initial indications are that the Ecuadorian government will invest to upgrade the Echeandia-Las Naves power line to 69 kV. A potential option may exist for Adventus to fund the construction of the required powerline against a tax incentive or other repayment arrangement to be agreed with the relevant Ecuadorian authorities. There is no current cost allowance included for such in the FS Capex.

1.14.5 Communications

The process plant and remote terraces will be connected through a fibre optic network for local telephone, internet and data services during the operational phase. On-site communications for operational personnel will be predominantly by wireless hand-held radios and cell phones using a local wireless cell phone service provider. A local wireless provider connection will be engaged during the early works phase to provide additional tower infrastructure near the mine site. In addition, when site Project network hardware and internet connections are commissioned, a local wireless network will establish Wi-Fi internet connection

With the installation of the overhead power grid infrastructure, a fibre optic line internet line is being investigated as part of the infrastructure investment

1.14.6 GEOTECHNICAL INVESTIGATIONS

Geotechnical investigations on the main process terrace platform were undertaken to establish founding conditions for the process plant equipment. Results indicated that the terrace will be mostly housed on compacted saprolite that may be susceptible to settling. Local earthworks improvements underneath major structures were allowed for to mitigate this risk.

Geotechnical investigations for the site roads were not undertaken during the FS. The Project site roads follow established road corridors, but also require new road construction. The El Domo deposit is connected to nearby towns by three (3) existing local roads. A site visit was conducted to inspect and understand local geotechnical conditions. Further regionally typical geotechnical and stability works parameters were agreed between AOC Ingenieria (AOC), a local civil engineering company retained to design the site access road, and DRA as inputs for the FS road design and estimate.

The waste facility geotechnical program is well developed and contained multiple test pits, boreholes, piezometer installations, and related tests. The site investigation program was prepared by Klohn Crippen Berger S.A. (KCB) using preliminary layouts of the mine waste and plant facilities and reviewed/modified as the design and investigations evolved. Investigations were constrained in some zones where access to areas within in the tailings and waste dump were not available at the time of the program. Results indicate that the residual soils and saprolite are the main foundation materials. It was observed that residual soils and saprolite are thicker (up to 35 m) on high ground and shallower (5 m approx.) near creeks.

1.15 Market Studies and Contracts

Adventus retained Exen Consulting Service (Exen) to prepare an initial independent market study (Exen, 2020) for the planned concentrates. Exen conducted a preliminary investigation as to best suited markets and potential smelter purchasing terms for each concentrate.

Contained metals within the concentrates are well-known and traded freely on metals exchanges globally. There are multiple smelters in different markets that can accept the three (3) concentrates for refinement.

Following the initial market study (Exen, 2020), Adventus conducted further term negotiations with specific off-takers to refine the market study terms. Preliminary terms were received from specific off-takers (Adventus ST, 2021) and evaluated for optimal project revenue.

The payable terms and relevant reference for each concentrate is depicted in

Table 1.5

Table 1.5 – Concentrate Payable Terms

Concentrate Payable %	Payable	Minimum Deductions
Cu Concentrate Payable (Adventus ST, 2021)		
Payable Au	96.0% ≥ 20 g/dmt 95.0% < 20 g/dmt	
Payable Ag	90.0% ≥ 30 g/dmt 0% < 30 g/dmt	
Payable Cu	96.5% ≥ 20% dmt 96.5% < 20% dmt	1.0% when grade above 20% dmt 1.2% when grade below 20% dmt
Pb Concentrate Payable (Exen, 2020)		
Payable Au	95.0%	1.0 g/dmt deduction
Payable Ag	95.0%	50 g/dmt deduction
Payable Pb	95.0%	3 unit deduction
Payable Cu	100%	2 unit deduction
Zn Concentrate Payable (Adventus ST, 2021)		
Payable Au	75.0% ≥ 1 g/dmt 0% < 1 g/dmt	1 g/dmt
Payable Ag	75.0% ≥ 3 oz/dmt 0% < 3 oz/dmt	3 oz/dmt
Payable Zn	85%	8 unit minimum deduction

The sales terms for each concentrate are depicted in Table 1.6.

Table 1.6 – Concentrate Sales Terms

Category	Terms
Transport (Inland, Shipping, Port Charges, Insurance) (Adventus ST, 2021)	
Cu Concentrate	$US 71.74 / wmt conc
Pb Concentrate	
Zn Concentrate	
Treatment Charges	
Cu Concentrate	$US 80.00 / dmt conc
Pb Concentrate	$US 180.00 / dmt conc
Zn Concentrate	$US 220.00 / dmt conc
Refining Cost	
Au	$US 5.00 /oz in copper concentrate $US 15.00 / oz in lead concentrate
Ag	$US 0.50 /oz in copper concentrate $US 1.50 / oz in lead concentrate
Cu	$US 0.08 /lb in copper concentrate $US 0.41 /lb in lead concentrate
Pb	$US 0.00
Cu Concentrate (Adventus ST, 2021)	Pb + Zn $3.00/dmt for each 1.0% Pb+Zn > 4.0% As $3.00/dmt for each 0.10% As ≥ 0.20% Sb $5.00/dmt for each 0.10% Sb ≥ 0.05% Hg $2.00/dmt for each 10 ppm Hg ≥ 10 ppm Cd $3.00/dmt for each 0.01% Cd ≥ 0.03%
Pb Concentrate (Exen, 2021)	$2.00/dmt for each 1.0% Zn > 5.0% As $1.50/dmt for each 0.10% As > 0.50% Sb $1.50/dmt for each 0.10% Sb > 0.50% Bi $1.50/dmt for each 0.01% Bi > 0.10% Hg $2,00/dmt for each 10 ppm Hg > 50 ppm
Zn Concentrate (Adventus ST, 2021)	Cd $1.50/dmt for each 0.1% Cd ≥ 0.3%

The indicated terms were utilised to prepare the Net Smelter Return (NSR) as input to the financial model. There are no material contracts or agreements in place as of the effective date of this Report. Adventus has not hedged, nor committed any of its production pursuant to an off-take agreement.

1.16 Environmental Studies, Permitting and Social or Community Impact

An Environmental and Social Impact Assessment (ESIA) has been completed for the Project that will be submitted to the Ecuadorian government for review. It represents the first step in acquisition of the necessary permits to construct and operate the Project. The ESIA complies with Ecuadorian law, and details the baseline condition for biophysical and socio-economic factors. The ESIA also includes a substantial commitment to avoidance and mitigation of negative impacts.

The Project location is in an area that has largely been altered by previous agricultural and other human activities. Only small patches of original forest remain. Despite this, a number of important floral and faunal species have been identified which have been incorporated into monitoring and management planning. Most measured biophysical parameters, including air quality, noise, and vibration, fall below Ecuadorian limits.

Curimining has completed several years of community engagement activities to ensure that exploration activities and eventually mine development is understood locally. With a few exceptions, local communities accept the Project, and recognise it as a source of employment, economic development, and improvement on local infrastructure.

1.16.1 Waste Management

Curipamba will have two (2) main waste dump facilities and a Tailings Storage Facility (TSF):

- A waste facility for storage of overburden and saprolite waste (Saprolite Waste Dump, SWD),
- Waste Rock Facility 1 (WRF1) for storage of majority of Potentially Acid Generating (PAG) and unsuitable waste rock; and,
- TSF/WRF2's embankment waste dump for the storage of majority suitable Non Acid Generating (NAG) material to build an embankment that will store process tailings in a lined facility formed behind the waste embankment.

The SWD will store saprolite and overburden waste from mine pre-stripping and mine infrastructure construction. The natural conditions of the saprolite at Curipamba show high in-situ water content, which will make it difficult to traffic and compact as engineered fill. To improve the workability of the saprolite and increase the stability of the dump, the outer zone of the dump is designed to allow co-mingling of saprolite and waste rock. The SWD is not designed to store water and an underdrain is proposed at the bottom to lower the phreatic surface within the body of the dump.

The tailings storage facility will contain the process tailings in a lined dam created by the construction of a TSF embankment from mine waste rock. The TSF embankment will be progressively constructed by ongoing placement of mine waste in the waste storage facility WRF2 which will buttress the TSF.

The TSF/WRF2 is designed to provide storage of tailings, temporary excess water storage (before water treatment and discharge) and the design flooding containment. Tailings will be discharged from the crest of the embankment and from the toe of the SWD to promote the formation of a tailings beach while maintaining the decant pond away from the embankment.

Tailings are considered PAG and their potential Acid Rock Drainage (ARD) will be managed in the TSF impoundment, which will be lined to prevent seepage. The TSF/WRF2 embankment dump includes a starter dam, subsequent raises using competent rockfill on the upstream slope and general mine waste rock on the downstream slope. Slope stability analysis were completed using geotechnical models including the topography, design configuration, foundation conditions, and geotechnical parameters assigned to the foundation and construction material. The performance of the dam under seismic conditions was assessed using pseudo static analysis used as a preliminary screening tool.

A collection pond located downstream the TSF/WRF2 embankment dump collects run-off water for monitoring and treatment. During operations, the TSF/WRF2 impoundment will be the main repository of surface run-off and precipitation water from the site's catchment. The TSF/WRF2 will collect the water, recycle the required volumes for process and provide temporary storage of excess water to maintain a monthly peak water rate of 470 m^3/h (130 L/sec) to the Effluent Treatment Plant (ETP).

Curipamba will produce both PAG and NAG mine waste materials that require separate storage. NAG waste rock will preferably be used for construction of the TSF/WRF2 embankment. NAG waste rock will also be used to encapsulate any good quality PAG rockfill considered for TSF/WRF2 embankment dump construction, with the objective being to avoid using any PAG rockfill in the TSF/WRF2 should the site conditions allow it. WRF1 is proposed for storage of poor quality and the majority (or all) the PAG waste rock not sent to construction of TSF/WRF2 and not used for co-mingling in the SWD. WRF1 has no waste rock zones in its design. As a general recommendation, the waste dump can be built in with "ascending platforms" built and with an "end -dumping" method (Method IV) (Hawley and Cunning, 2017), contained behind toe berms which will initially act as an "impact berms" to reduce risks of boulders rolling downstream.

Ore stockpiles will be required for process blending purposes throughout the LOM. Three (3) long term stockpiles are required to be built for the LOM. A suitable area has been identified that is located between the open pit and the process plant along the main haulage route. Although the ore stockpiles are designed to be temporary, the storage area will be utilized throughout the LOM. The ore material is expected to be acid generating as confirmed by geochemical test work (Phase, 2020) and will require suitable containment to avoid groundwater contamination. The stockpile floor area will be suitably sealed with a layered saprolite designed to capture and channel seepage water and run-off water toward the Saprolite waste facility for temporary water storage and treatment in the tailings facility water treatment plant. The maximum combined stockpile size is 400 kt in Year 9 of production.

1.16.2 WATER MANAGEMENT

1.16.2.1 Hydrogeology

El Domo is a stratiform and largely stratabound volcanic massive sulphide (VMS) deposit with an overlying zone of brecciated mineralised fragments (RPA 2019), within a Paleocene-Eocene submarine arc of the Macuchi Formation (Phase 2020). The preliminary mine design in this FS involves open pit mining for ten (10) years.

The Site straddles the watershed divide between the Naves Chico basin and El Silencio basin. Generally, the groundwater system in the Naves Chico and El Silencio Basins are expected to be recharged by precipitation and runoff in the higher elevations of the basin and discharge in the creek valleys at lower elevations. Examples of inferred groundwater recharge zones are the east and west valley ridges of both the El Silencio and Naves Chico basins. Examples of groundwater discharge zones are the creek valleys incised by the Quebrada Naves Chico and the Estero El Silencio. This is generally supported by shallow depths to groundwater, flowing artesian conditions or upward vertical hydraulic gradients reported at standpipes installed adjacent to stream channels.

Groundwater flows through the pit footprint (on the west-facing slope of the Naves Chico Valley) from recharge zones along the eastern ridge of the Naves Chico Valley toward the northwest. Horizontal groundwater flow occurs predominantly in the weathered bedrock and along the base of the saprolite. However, this predominant groundwater flow pattern may be locally altered by the network of faults that transect the El Domo pit. Preliminary groundwater quality data suggests that the groundwater quality evolves from a calcium bicarbonate facies (typical of precipitation) to a calcium sulphate facies adjacent the Quebrada Naves Chico at the northern extent of the site.

Groundwater recharge in the El Silencio Valley originates as rainfall infiltrating along the east and west valley ridges and passing vertically downward through an unsaturated (vadose) zone and eventually recharging the underlying unconfined aquifer in the weathered bedrock. Following local horizontal and downward vertical hydraulic gradients, groundwater flow in the unconfined aquifer is inferred to be directed southeast (from the west valley slopes) or southwest (from the east valley slopes) and eventually discharging to the Estero El Silencio in the lower reaches of the watershed. The groundwater quality along the flow path is inferred to evolve from a calcium-bicarbonate facies in the recharge zones to a sodium-bicarbonate facies within the discharge zones.

1.16.2.2 Geochemistry

Mined material lithologies were identified and the geochemical characteristics of each rock type were determined. Adventus retained pHase Geochemistry Inc. (pHase) to determine characterise and develop source terms for the mined material. Chemical load predictions for the open pit, tailings beach and waste facilities were determined (Phase A, 2021) (Phase B, 2021).

Prediction results indicate that the Saprolite Dump and Waste Rock Facility 1 (storage of Potentially Acid Generating PAG rock) will generate acidic conditions in the pH 2 to 3 range should the full volume of each of these facilities be available for oxidation. This reflects the limited neutralization potential of the Non Acid Generating (NAG) rock to provide any meaningful alkalinity to buffer acidity produced from the PAG rock despite the volumetrics being dominantly NAGnon-PAG. Due to the high sulphide content and low acid neutralisation potential of the PAG rockpH values, contact water is predicted to be characterised by low-pH with associated high concentrations of sulphate and metals, concentrations for nearly all parameters including aluminum, arsenic, cadmium, copper, iron, lead, manganese, nickel, and zinc, requiring containment and water treatment.

Modelling scenarios for Waste Rock Facility (WRF) 2, the TSF/WRF2 source terms , which assumed only rock scheduled for placement on the outer layer stages of the facility would be ofbeof the dump was exposed to oxidation and that these stages wouldcould be managed to be NAG (i.e. no PAG placement in the outer portion of this facility)material portions of andesite, tuff, rhyolite tuff or basalt,. Predictions indicated that seepage from the TSF/WRF2 should remain neutral (pH 7.2 to 7.7), but have elevated sulphate, arsenic, cadmium, and zinc. Based on the modelled scenarios, theAs noted, the neutralization alkalinity potential of the NAG rock is low and may not be sufficient should the PAG rock be present in these later stages of construction, without PAG/NAG management during construction, which will result in leachate similar to observations in humidity cell testing and the contact water from this facility leachatefacilityleachate would be predicted to become acidic (pH 2.9). Therefore, it is recommended that operational waste segregation and sorting protocols be developed for waste going to final facility layer stages of the TSF/WRF2 facility (e.g. sulphide analyses and cut-off criteria for units such as basalt and tuff) should also be considered during operations.

1.16.2.3 Water Management Infrastructure

Water management infrastructure is a function of the site layout and topography.

The water management criteria confirms that precipitation run-off and underground seepage water entering the project infrastructure will be collected, contained, treated and discharged within the respective natural catchments basins of El Silencio (TSF, WRF1 and 2, Plant, Infrastructure) or Naves Chico (Open Pit Mine). Domestic water sources will be isolated and collected via a piped or tank-and-truck method and discharged to a singular point for domestic water treatment.

The Naves Chico basin contains the open pit mine and related mining infrastructure. Contact water will enter the pit through precipitation and ground water inflows and collect in the bottom of the phase one pit sump. From here the water will be pumped directly to the water treatment plant and then discharged to a mine water pond for normalisation from where it will naturally overflow into the natural valley drainage system.

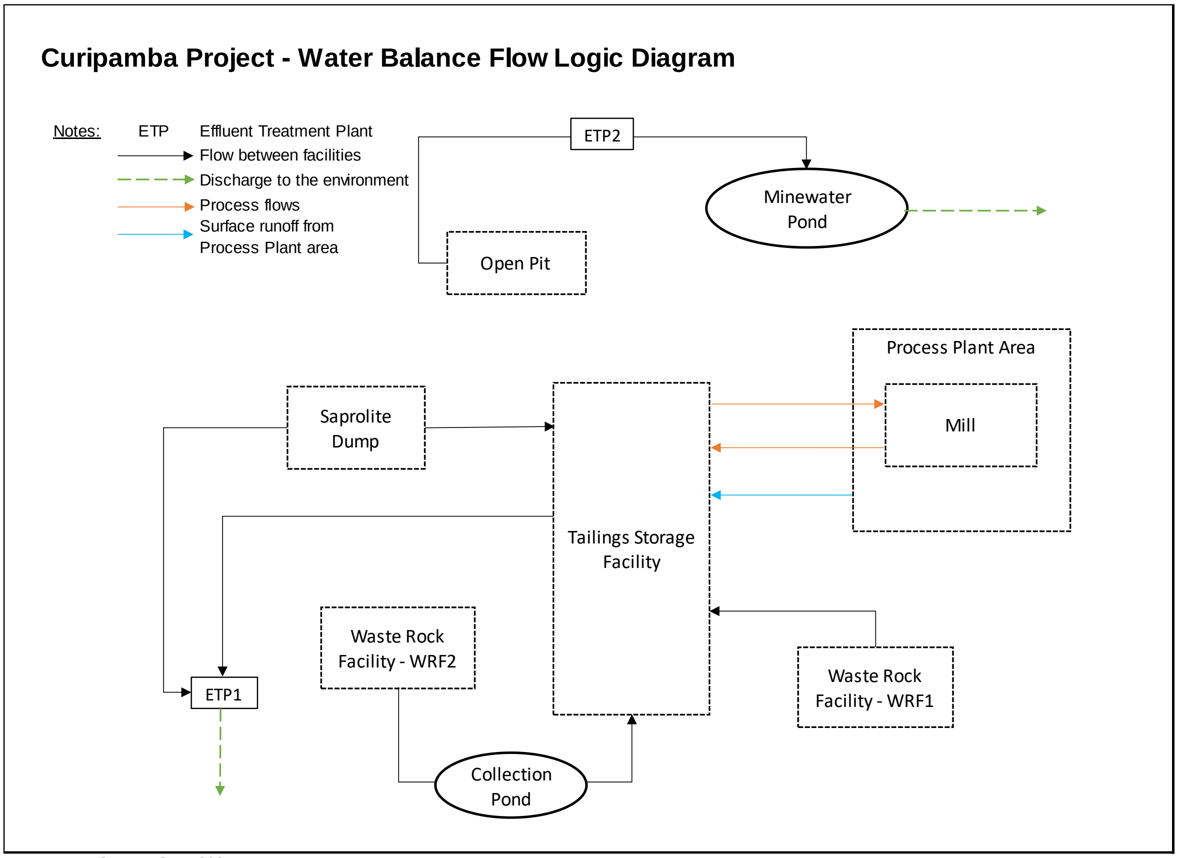
The El Silencio basin contains the majority of the project infrastructure namely: waste rock facilities, process plant, site support infrastructure and tailings facilities. The tailings facility will be constructed in the natural drainage path of the basin and will collect all precipitation water, contact and non-contact water. Diversion of non-contact water with berms and channels were evaluated, but was found to be too technically challenging to construct and maintain in the steep valley terrain.

Contact water from the process plant, site infrastructure and waste facilities WRF1 and Saprolite Dump will naturally drain into the tailings pond via constructed and suitable channels to avoid ground seepage.

Water will be pumped from the TSF through the water treatment plant and discharged via the WRF2 diversion berms into the natural valley drainage system.

The site wide water balance is depicted in Figure 1.3.

Figure 1.3 – Site Wide Water Balance Schematic



Source: SLR, 2021

1.16.2.4 Water Quality and Treatment

DRA retained SLR Consulting Canada Ltd. (SLR) and Minnow Environmental Inc (Minnow) to conduct initial water balance (SLR, 2021) and water quality predictions (Minnow, 2021) respectively, for the FS.

Probabilistic site-wide water balance modelling was conducted to calculate water discharge rates to the environment under variable rainfall conditions, and calculate flows associated with the open pit and waste management facilities to support the water quality modelling.

Initial water quality predictions were prepared on the assumption that all contact water will be contained, treated and discharged in the respective catchment basins of either Naves Chico, or El Silencio (Minnow, 2021).

The water quality predictions suggest that the untreated effluent that is pumped from the open pit will be acidic (pH 2.60 to 4.86) with elevated concentrations of sulphate and metals. Parameters that are predicted to be greater than the maximum effluent discharge limits include: sulphate, aluminum, arsenic, cadmium, cobalt, iron, lead, manganese, and zinc. The design includes a water treatment plant that neutralizes the acidic water and treats the parameters noted to allow discharge of clean water that is within environmental discharge limits.

Similar to the pit sump chemistry during operations, the water quality predictions suggest that the untreated effluent from the Saprolite Dump and TSF pond will be acidic (pH 2.30 to 3.13) with elevated concentrations of sulphate and metals. Parameters that are predicted to be greater than the maximum effluent discharge limits include: sulphate, aluminum, arsenic, boron, cadmium, copper, iron, lead, manganese, nickel, and zinc. The design includes a water treatment plant that neutralizes the acidic water and treats the parameters noted to allow discharge of clean water that is within environmental discharge limits.

The low pH and elevated concentrations of sulphate and metals reflect the reactive nature of the PAG rock and limited neutralizing nature of the NAG rock types to offset acid generation.

Careful management of NAG rock type placement unto the final outer layers of the waste rock facilities will be critical to prevent long-term active closure requirement. Introduction of oxygen impregnable layering is recommended as part of the detailed design to isolate internal rock from oxygen sources.

1.16.3 Mine Closure

The Project's conceptual closure approach is to rehabilitate the mine site so that it is physically and chemically stable and compatible with the intended future land use.

The establishment of a permanent pit lake is planned that will passivate remaining exposed reactive rock type surfaces over an active closure period currently estimated at 2 years. The natural pit lake level will be elevated through the construction of a dam wall from suitable mine waste material. Exposed pit areas that will not be passivated by the final pit lake will be suitably passivated with designed closure covers, and rehabilitated. The operational phase water treatment plant will remain active on site to treat the pit lake water over an estimated active closure period of more than 2 years while the pit water quality improves.

The preferred closure strategy for the TSF includes a dry closure cover over the exposed tailings and construction of a closure spillway channel to be constructed in the left abutment of the dump that will leave a 200 m minimum width of tailings beach upstream of the tailings dam to augment the long-term stability of the dam. The closure cover will be mostly built using the materials excavated for construction of the spillway which is assumed as saprolite which will act as an oxygen barrier. Tailings beach reclamation will occur by placement of reinforcement geogrids followed by placement of closure fill and revegetation by hydroseeding. The exterior rockfill slope of the embankment is considered acceptable for permanent closure. As part of the waste facility construction, PAG waste rock will be encapsulated within NAG waste rock, in strategic zones within the dump to reduce the potential for ARD in the long-term.

The Saprolite Waste Dump can be progressively closed at the end of construction. Progressive closure activities include revegetation of the exposed downstream slope and top surface. The long-term stability of the dump should be monitored and reviewed during operations.

Process related equipment and infrastructure will be dismantled and sold for remaining value and re-use. Non-saleable equipment and materials will be classified and correctly disposed of. All surface concrete structures will be demolished up to a suitable depth and disposed of in the waste rock facilities. Compacted soils will be ripped, topsoil introduced and rehabilitated through revegetation.

1.17 Capital Cost Estimate

1.17.1 Capital Estimate

The Capex includes the material, equipment, labour and freight required for the mine, process facilities, infrastructure and services necessary to support the operation and includes for the estimates developed and provided by external consultants: KCB for tailings storage and waste rock facilities, AOC for access roads, and EngyWorks for offsite power line estimates.

The Capex prepared for this FS is based on a Class 3 type estimate as per the Association for the Advancement of Cost Engineering (AACE) Recommended Practice 47R-11 with a target accuracy of ±15%. The estimate reference date is August 31, 2021

The Initial Capital phase is the period where the main project construction expenditure will commence. The Initial Capital phase commences upon receipt of all relevant construction and environmental permits and is planned as an 18-month construction period prior to commencement of production. Capital expenditure prior to the Initial Capital phase is considered exploration, development, and pre-construction capital; this cost includes for a planned early works construction preparation package. Sustaining Capex commences upon production of concentrate and continues throughout the mine life. Closure capital is planned toward the end of mine life in the final production year with capital cost for closure operations past the completion of the production phase.

Table 1.7 – Project Capex Summary by Major Area and Phase (US$ M)

Description	Pre-Construction Capital	Initial Capital	Sustaining	Closure	LOM Total Capex
Direct					
Pre-construction Early Works Program	22				22
Mining - Open Pit		52			52
Processing Plant		67	6	15	89
Mining Site - Infrastructure		2			2
Tailings / Waste Rock		34	12	9	54
Surface Infrastructure - Buildings, Roads, Water Treatment		12	8	2	22
Subtotal Direct	**22**	**167**	**26**	**26**	**241**
Indirect		-			
Construction Indirect		18			18
Freight & Logistics, Plant Equipment		7			7
Owner's Costs, EPCM, Taxes		10			10
Project Contingency		22	3	8	32
Refundable VAT on Initial Capital		25			25
Salvage Value		-		(10)	(10)
Subtotal Indirect	**0**	**82**	**3**	**-2**	**82**
Total	**22**	**248**	**29**	**24**	**323**

1.17.2 Operating Cost Estimate

The Opex estimate was developed to support the operating phase of the project with required personnel, equipment, infrastructure and services. The Opex estimate was developed following the completion of mine, process and infrastructure designs that provided the necessary consumable, personnel and service requirements for the FS estimating.

The Opex estimate is based on quantified consumable rates, derived personnel estimates, and received contractor and supplier quotations for both mining and process operations.

Adventus and DRA jointly develop the General & Administration (G&A) cost for the project indicating allowances for expected annual expenditures and determining required support personnel complement.

The Opex is estimated at $364M over the life of mine or $56.21/t of ore processed during the ten years of operation. Table 1.4 summarises the Opex by discipline area over the Project LOM.

Table 1.8 – Opex Summary by Area

Description by Area	Average Annual Costs (US $M)	Total Cost LOM (US $M)	Cost / t moved (US $/t)	Cost / t ore processed (US $/t)	Copper Equivalent cost $/lb Copper Eq.[3]
Mining	16	$157	3.35	24.53	0.34
Process	**15**	**150**		**22.74**	**0.32**
Process Plant	13	125		19.08	0.27
Water Treatment	2	24		3.66	0.05
G&A	6	56		8.95	0.12
Total[2]	**36**	**364**		**56.21**	**0.77**

1 A portion of owners team mining is captured under G&A totalling $3.2M over LOM, equivalent to $0.06/t moved($0.50/t milled). The mining cost is indicated in cost/t moved.
2 Figures may not total due to rounding
3 *CuEq is calculated as follows:*
(Payable Metals NSR Ag,Zn,Pb,Au, Ag)/(Payable Metals NSR Cu) (2.205 lb/kg)*(Payable Copper/lb)*

1.18 Economic Analysis

The economic analysis provides financial results against agreed metals price points, net smelter payables and penalties, taking recoveries, capital and operating costs into account. Annual cashflows are discounted to include an 18-month pre-production construction period. Key economic results are summarised in Table 1.9.

The open-pit mine generates an after-tax Net Present Value (NPV) of $ 259 M at an 8% discount rate, an after-tax Internal Rate of Return (IRR) of 32% and the after-tax payback period is 2.6 years from commencement of production.

An economic analysis based on the production and cost parameters of the Project was carried out and all figures are in $ USD currency.

Table 1.9 – Economic Summary

Description	Unit	Value
Metal Price Predictions for Economic Base Case		
Metal Prices – Cu	$US/lb	3.50
Metal Prices – Zn	$US/lb	1.20
Metal Prices - Au	$US/oz	1,700.00
Metal Prices - Ag	$US/oz	23.00
Metal Prices – Pb	$US/lb	0.95
Production Profile - Open Pit Only		
Total tonnes of mineraliseore mined and processed	Million tonnes	6.48
Total capitalised pre-stripping tonnes	Million tonnes	15.25
Total LOM tonnes waste mined	Million tonnes	57.08
Operating strip ratio (following capitalised pre-stripping)	waste:ore	6.46
Overall Strip ratio	waste:ore	8.81
Average tonnes mined per year (waste and ore)	Million tonnes	5.15
Peak tonnes mined per year (waste and ore)	Million tonnes	11.95
Peak tonnes mineraliseore mined per year	Million tonnes	0.98
Mine life	years	10

Description	Unit	Value
Payable Metal Recoveries [(2)]		
Net average payable metal recovery to concentrates - Au	%	51.8
Net average payable metal recovery to concentrates - Ag	%	63.6
Net average payable metal recovery to concentrates - Cu	%	87.5
Net average payable metal recovery to concentrates - Pb	%	30.3
Net average payable metal recovery to concentrates - Zn	%	84.7
Unit Operating Costs		
LOM AISC[(1)]	$/lb Cu Eq.	1.26
Project Economics		
Royalties - Ecuadorian Government	%	4
Royalties - Altius	%	2
Average annual EBITDA	$M	103
Pre-tax NPV 8% / After-tax NPV 8%	$M	426 / 259
Pre-tax IRR / After-tax IRR	%	45 / 32
Undiscounted operating pre-tax cash flow / after-tax cash flow	$M	749 / 497

1) This Report contains certain non-GAAP (Generally Accepted Accounting Principles) measures such as cash cost and ASIC. All-in sustaining cost copper equivalent for the Project represents mining, processing, site general and administrative costs ("G&A"), water treatment costs, royalties, treatment and refining charges and sustaining capital, divided by equivalent copper tonnes, and excludes corporate G&A. Such measures have non-standardized meaning under GAAP and may not be comparable to similar measures used by other issuers

(2) Net recoveries only include metals payable in their respective concentrates

The annual revenue by metal is presented in Figure 1.4. The annual Project cash flows are presented in Figure 1.5.

Figure 1.4 – Annual Revenue by Metal

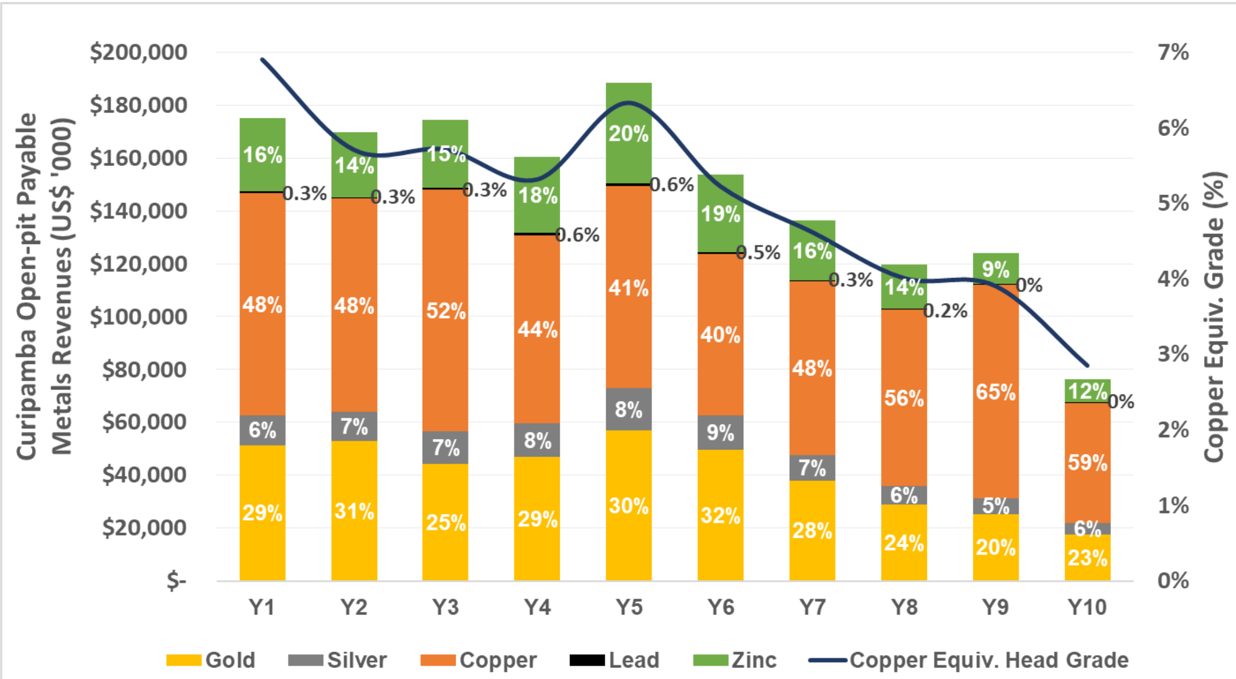
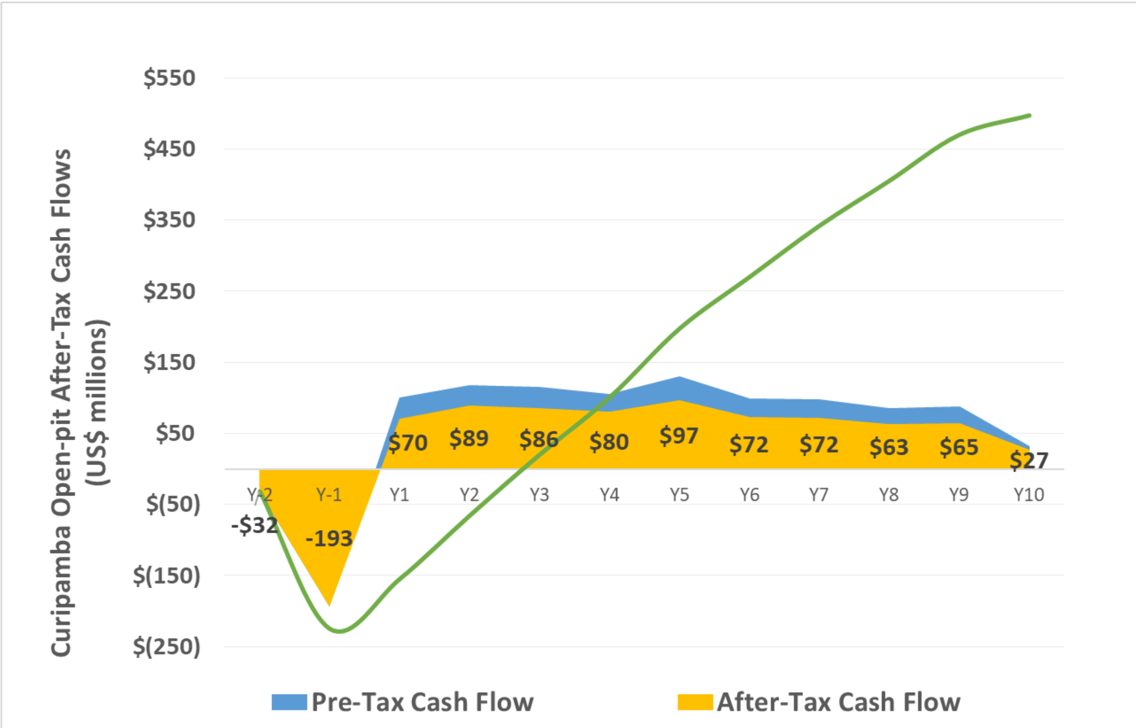


Figure 1.5 – Project Discounted Cash Flow



1.19 Project Schedule and Execution

1.19.1 Critical Construction Activities and Key Dates

The Project has developed an execution schedule suitable for the FS stage. Worldwide supply chain disruptions associated with the COVID-19 pandemic are extending fabrication and delivery times and must be noted as a risk. The commencement of construction activities remains dependent on the receipt of required permits from local authorities.

1.19.2 Project Implementation

The FS estimate considers the appointment of an Engineering, Procurement and Construction Management (EPCM) as the main contractor to provide the necessary technical and engineering design details for the project. The EPCM contractor will oversee the construction activities and be responsible for the supply of construction services for the project. Adventus plans to bolster an experienced EPCM construction team with locally employed personnel. In country, and specifically local to the mainly agriculturally based project location, skilled construction and operational personnel may challenging to source, requiring development. Adventus is engaging with local institutions to promote and establish mining related skills development in anticipation of the Project execution.

1.19.3 Early Works

An early works package is planned to address critical construction phase support project components. The implementation of the early works package will assist the Project execution schedule by alleviating construction pressure and execution economic risk. Progression of detailed geotechnical and survey information will provide the detailed engineering phase with required information to improve design accuracy and costing. Planned road upgrades and access road construction will reduce logistics risks and ensure larger equipment can be delivered to site without special transportation requirements.

1.19.4 Construction Phase Water Supply and Management

Adventus hold current surface water extraction permits which is deemed adequate for the planned early works. Application for further borehole extraction permits is required for construction phase. The construction phase will require contact water containment and treatment capabilities.

The Project will expedite the water treatment plant construction as a priority to ensure sufficient treatment capacity exists when the tailings facility is closed for water containment.

1.20 Operational Readiness Gap Analysis and Planning

An FS level Operational Readiness Plan (ORP) gap analysis was performed to commence ORP for the Project. An initial ORP plan was developed for implementation during the next project phase. The ORP indicates the necessary actions to prepare Adventus for operations and have been scheduled to coincide with the Project execution schedule.

1.21 Project Risks and Opportunities

Risks and Opportunity registers were prepared and maintained during the FS. Main risks are associated with permitting and ESIA approvals, project power supply infrastructure upgrading and funding, and mitigation of geochemical and long-term acid rock drainage. Recommended actions were noted.

The FS produced a suitable project infrastructure design and progressed the understanding of project opportunities with integration of test work and design data. Main opportunities involve the review of waste management to best mitigate impact and closure risk, reduction of water treatment requirements through diversion of non-contact water, and further development of the underground expansion opportunity.

1.22 Underground Expansion Opportunity

A Preliminary Economic Assessment (PEA) level study (±50% accuracy) was prepared to evaluate the merits of mine life extension through further underground mining following end of the open pit mine life.

The mine design is based on a single decline with a 15% gradient excavated in the final open pit east wall at an elevation of 850 m. The deposit will be mined using drift and fill with cemented rock fill at a production rate of approximately 0.67 Mt/year. A NSR cut-of of US$105.2/t of mineralised material was used for the stopes. Cemented Rock Fill (CRF) is recommended to reduce the width of the exposed roof span and thus guarantee the stability of the stopes. The mine will contain a ventilation system. The main decline will be the escape route to surface, and the main ventilation raise will be equipped to serve as a secondary egress.

Initial evaluation of the underground opportunity indicates a positive economic contribution to the Project and extended use of fixed capital invested in the open pit phase. More value is also gained from the available metal resource toward optimising the resource potential.

1.23 Concluding Remarks

1.23.1 Recovery Methods

A concentrator plant capable of handling all three ore types that will be mined at El Domo has been designed during the feasibility study based on extensive testwork and pilot plant data.

A conventional sequential flotation circuit has been selected to produce separate coper, lead and zinc concentrates using a robust and flexible process flowsheet. The testwork has been optimised to produce saleable concentrates.

Process design has had sufficient information sourced from the testwork summarised in Section 13, equipment vendor data, information provided by Adventus, and DRA in-house design data.

A skid-built crushing circuit has been selected rather thana stationary crushing circuit to reduce capital expenditure. These units will also be used for aggregates production during the plant construction.

The grinding circuit uses a single stage grate discharge ball mill in closed circuit with the classification and cyclones.

The 2019-2021 testwork results suggested an opportunity to increase a bulk rougher flotation feed size (P_{80}) from 75 to 125 microns.

Vertical stirred mill with ceramic media is suggested for the bulk concentrate regrind.

Tank cell technology was recommended for the bulk, copper, and zinc rougher flotation circuits to ensure a high unit equipment throughput, metals recovery, and control over the process flow.

Control over the cleaner feed grade and tonnage was offered by introducing a system of concentrate re-direction either to the copper circuit of to the zinc circuit by means of the launders and mining hoses in the bulk rougher flotation.

Conventional forced air mechanical cells were recommended for the cleaner circuits and represent a proven low-cost technical solution.

High-rate steel tank bolted thickeners were chosen for final tailings and concentrates settling and water recovery to provide an optimal capital cost and reduce equipment installation time. Any solids carryover from the tailings thickener will be captured within the process water settling pond with intermittent purge of the collected solids into the final tailings tank by means of the dedicated pump.

Vertical plate and frame filter presses were recommended for the concentrates dewatering as a conventional cost efficient equipment solution.

Water reticulation of the plant has been designed to minimize a raw water usage by means of the water reclaiming from the tailings dam. The process design resulted in freshwater usage which minimized the need for fresh water to less then 0.5 m^3 per tonne of mill feed ore.

1.23.2 INFRASTRUCTURE

The project infrastructure designs have been sufficiently progressed for FS level estimation. The noted infrastructure is of suitable design for the site and size of operations.

Geotechnical and founding condition designs have been sufficiently progressed for an FS and relevant assumptions noted.

Early construction of the site power infrastructure will be required to support the construction phase of the project. Temporary construction power generation on site may be needed.

Adventus may have the option to fund the 69kV Echeandia-Las Naves upgrade on an incentive basis agreement with relevant authorities.

Off-site local municipal infrastructure in towns such a Las Naves is suitable to regionally support the project. Local municipal infrastructure and authorities may require additional support during the construction phase and operational phase to ensure support to the Project.

1.23.3 Market Studies

Preliminary market studies and potential off taker terms were obtained for all three concentrates that indicate saleable concentrates with multiple off-taker potential. Payables, penalties and refinement charges have been reviewed. Transportation and freight costs were evaluated and included for all three concentrates.

1.23.4 Environmental

It is expected that the ESIA will proceed through review by the Ecuadorian government and the public, culminating with issuance of an Environmental License in second half of 2022. Subsequently individual permits can be acquired for all aspects of construction and operation.

1.23.5 Waste Management

The designed facilities meet the mine waste storage requirements for the FS mine plan. Saprolite waste will be comingled with waste rock for improved stability in a dedicated Saprolite waste dump. A PAG WRF1 is established above the TSF. The TSF will be established through the development of a downstream waste rock embankment that will contain mainly NAG material with some encapsulated PAG material within.

Mine ore stockpiles will be required for the life of mine with a maximum size of 400 kt in Year 9. The ore stockpiles will be located near the haul road between the open pit and processing plant. Run-off and seepage water will be directed toward the saprolite facility for collection and treatment in the TSF pond.

The disposal of other waste such as domestic, hazardous, and biological waste will be properly stored and disposed of offsite at registered disposal facilities.

1.23.6 Water Management

Water management infrastructure for each water catchment area consist of two (2) main containment ponds namely: the TSF for the El Silencio basin and the Open Pit sump for the Naves Chico basin. Other site facilities and contact water drain into these facilities from where the water will be treated prior to environmental discharge.

Mine material geochemistry was modelled and used to determine chemical characterisation and source terms for the various project elements. The geochemistry results indicated that the mine water is expected to be acidic and contain sulphate and heavy metals that will require treatment prior to discharge. Suitable water treatment plants were designed and costed for the FS.

1.23.7 Mine Closure

A conceptual mine closure plan was developed that aimed to return accessible land to pre-mining condition and minimise long term active closure requirements. Closure operations will commence in the final year of production and continue for an additional 2-year period.

The open pit will be closed with a pit lake that is supported by an embankment construction and operational phase water treatment plant. Other exposed areas of the mine pit will be suitably covered, sealed, and rehabilitated to avoid long-term acid generation. Water treatment will continue until the pit lake water quality improves and stabilises below required discharge limits.

The waste rock facilities will be closed with final NAG outer layers of mine waste. The TSF beach will be reclaimed and revegetated, and a spill way introduced to control the TSF level and exposed beach head. Water treatment will continue until the TSF pond water quality is stabilised below discharge limitations.

1.23.8 Other Relevant Data and Information

The Project has developed a conceptual execution schedule suitable for the FS stage.

The earthworks quantities are significant and will require a dedicated earthmoving fleet to complete the construction timeously. Sourcing of construction material was evaluated and deemed sufficient for the FS. The planned early works package will prepare the site for commencement of construction and mine pre-production. Sufficient water extraction is currently permitted for the early works. The construction phase will require contact water containment and treatment capabilities. Adventus will be relying on grid power as far as possible. Basic 7.69 kV power line infrastructure is present on-site and is deemed sufficient for basic power supply during early works, but is not deemed adequate for the construction phase of the Project. Construction power will require the upgrading of local power infrastructure, or supplementation with generator power until the main 69 kV power supply is commissioned on-site.

Adventus performed and initial Operational Readiness assessment which is deemed sufficient for the level of study.

An initial Preliminary Economic Assessment (PEA) level study (±50% accuracy) on mine expansion through underground mining indicates a positive economic contribution to the Project at PEA level that requires further definition.

1.24 Recommendations

1.24.1 Recovery Methods

During the next phase of the Project, it is recommended to update the process mass and water balances with the new test work data produced from the testing campaign executed post feasibility study, any new information with regard to the mining plan, and any new (post FS) data provided by the equipment suppliers.

When evaluating the purchase of the mechanical and electrical equipment, emphasis should be placed on the possible minimising of equipment suppliers in order to obtain the lowest pricing, the interchangeability of parts and components and service plans.

1.24.2 Infrastructure

The site infrastructure was developed and designed appropriately to suit the FS level cost estimate. Further detailed design development may reduce sizing and improve terrace sizing during detailed design.

The site infrastructure requires further geotechnical and survey information to progress and optimize detailed engineering designs and estimates.

Further development of the site accommodation strategy can follow the operational readiness planning performed during the feasibility study.

Engagement with local authorities on power supply will progress in the next stage and clarity gained on funding requirements. Telecommunication and fibre optic internet service providers will be engaged as part of the electrical infrastructure design.

Off-site infrastructure uses can be evaluated to reduce risk and create opportunities for the Project.

1.24.3 Environmental

It is recommended to complete the permitting process initiated with the submission of the ESIA in order to obtain the approvals necessary to initiate construction. Concomitant with this, ongoing environmental and community relations monitoring and activities should be maintained.

1.24.4 Waste Management

Further detailed design development and optimization of the Waste Rock and Tailings Facilities is recommended as part of the detailed design. Further testing of construction material characterisations and saprolite comingling testing will affirm the FS design assumptions.

Opportunities exist to review and optimise the layout and elements of the of the waste facilities to further reduce environmental impact and infrastructure risk and associated economics.

1.24.5 Hydrogeology

A conceptual understanding of the groundwater has been obtained for the mine site. A detailed numerical groundwater model must be developed during the next project phase to improve predication accuracy. As input to the development of such a model, future drilling to characterise groundwater flow patterns in the pit area is recommended.

Additional geological mapping is recommended to determine if there is presence of additional faults, which could change the water transport conditions. Installation of further pumping wells will better characterise the hydraulic conductivities and continuity of the in-pit fault network. Installation of additional piezometers is recommended to characterise groundwater quality adjacent the pit. Further vertical hydraulic conductivity tests are recommended in the residual soils and saprolite in the Naves Chico and El Silencio valleys.

1.24.6 Geochemistry

The current geochemical test work results were based on a relatively small test sample of available material for certain lithological units. A more comprehensive test work program is recommended during the next project phase to improve the understanding of the increase accuracy and acid generation and metal leaching potentials determination of the various lithologies.

1.24.7 Water Quality and Treatment

It is recommended that further geochemical test work be undertaken in the next project phase to improve the understanding of PAG and NAG lithologies and subsequently improve water quality model predications. Further detailed design development and characterized waste rock placement sequencing can improve the outer layer composition of the facilities and subsequently improve water quality. A balanced surface water quality model must be established during the next project phase that will improve input in water treatment designs. Base line water quality data must continue to be collected to provide longitudinal baseline information for discharge comparisons.

1.24.8 Mine Closure

The current conceptual closure plan described here is at a FS level.

It is recommended to develop a detailed and phased closure plan during the next project phase to improve early rehabilitation of the mine site and reduce contact water surfaces throughout the LOM.

1.24.9 Other Relevant Data and Information

The early detailed development and implementation of the operational readiness plan will benefit the Project and reduce human resource risk.

Adventus will be relying on grid power as far as possible. It is recommended to progress a detailed study of existing local power infrastructure and evaluate the potential to upgrade and utilise it for construction power.

Risk and opportunity registers were maintained along recommendations for the next phase with main risks and opportunities noted.

The underground opportunity indicates positive preliminary economic assessment results. It is recommended to continue with planned infill drilling programs. Continue with rock mechanic drilling, interpretations and design to confirm mine stability and study potential geological faults present in the rock. Further progression of the mine design into PFS or FS phase will improve definition and cost estimating. Improving mine scheduling and integration with the open pit mining plan may produce opportunities to commence earlier with underground mining and provide blended material to the mill toward optimising metallurgical recoveries.

2 INTRODUCTION

2.1 The Issuer

This Report has been prepared for Adventus Mining Corporation (TSX-V: ADZN, OTCQX: ADVZF) (“Adventus” or the “Company”), a mining company listed on the Toronto Stock Exchange (TSX), with headquarters at 550-220 Bay Street, Toronto, M5J 2W4, Canada, and Salazar Resources Limited (“Salazar”) (TSX-V: SRL, OTCQB: SRLZF) a mining company listed on the Toronto Stock Exchange (TSX) with headquarters at Suite 1305, 1090 West Georgia St., Vancouver, BC, V6E 3V7, Canada.

2.2 Business Case

Adventus and Salazar (collectively the “Partners”) are in the process of advancing and evaluating the potential development of the El Domo volcanogenic massive sulphide deposit, located within the 21,537-hectare Curipamba Project (the “Project”) in central Ecuador. The relevant holding company for the Curipamba properties and permits is Curimining SA, an Ecuadorian subsidiary of Salazar. Curimining’ s current involvement includes the overseeing of the exploration and core logging works. Curimining is mentioned throughout the report mostly when associated with site related support.

Following positive economic results from the previous Preliminary Economic Assessment (“PEA”) study completed in 2019, the Feasibility Study (“FS”) was mandated by Adventus and led by DRA Global Ltd (“DRA”) to further advance the engineering, design and future construction of an open pit mine and facilities to produce concentrates of copper, zinc, and lead with significant gold and silver credits.

This work has been completed as part of Adventus’s option agreement with Salazar whereby Adventus may earn a 75% ownership interest in the Curipamba Project with a preferential 95% payback of future cash flows until its investment has been fully repaid.

The FS has indicated positive economic results which are detailed in this Report.

A preliminary PEA for underground mining following completion of the open pit mining operation, as summarised in the Adventus press release of October 26, 2021, is presented as an opportunity study in Section 24 of this Report. This PEA is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be categorized as mineral reserves.

2.3 Previous Study

2.3.1 Preliminary Economic Assessment (PEA) 2019

In 2019, Adventus commissioned RPA (now SLR) and Knight Piésold (KP) to complete a PEA study as a pre-cursor to a Feasibility Study that forms part of the earn-in agreement between Adventus and Salazar. The PEA involved the development of an initial mineral resource estimate and initial processing and infrastructure development. The PEA considered both open pit and underground ore mining to maximise resource access. Initial cost development followed the completion of conceptual designs. The PEA returned positive economic results that recommended the continuation of the next study phase.

The PEA made the following recommendations:

- Further exploration expenditures to progress MRE certainty. Further footwall drilling was recommended toward additional infill and exploration drilling.
- The open pit and underground mine designs including a crown pillar below the open pit will require additional geomechanical, geotechnical, and groundwater studies to develop and optimize the next stages of the Project.
- There are opportunities for optimising the process flow sheet and reagent scheme to maximise the recovery of valuable metals while minimising costs of consumables and reagents.
- Although the Project can access relatively low-cost grid power, grid instability suggests that a fully redundant site back-up power plant may be required. Evaluate generators in the next phase as well as grid power stability.

2.4 Feasibility Study Purpose

The purpose of the FS is to review and define the optimum configuration for the mine and processing arrangement based on the latest available test work and Mineral Resource Estimates (MRE). The FS further provides engineering definition through mine design, project infrastructure definition and optimised operations descriptions. Following completion of the engineering deliverables, a capital and operating cost estimate was prepared as well as a subsequent economic evaluation to determine the Project's viability. The definitions are followed by estimation and confirmation of project economics. The FS is based on a Class 3 type estimate as per the Association for the Advancement of Cost Engineering (AACE) Recommended Practice 47R-11 with a target accuracy of ±15%.

2.5 Scope of Work

The FS was sub-divided into an initial Project Definition phase followed by the FS phase. Functionally, the initial Project Definition phase served as a condensed internal pre-feasibility study for Adventus which then bridged engineering, design, and cost estimates from the PEA to FS levels.

The scope of the Project Definition and FS phases are outlined in this section.

2.5.1 Project Definition Phase Scope Summary

- Evaluate the resource, processing, and support infrastructure options through a series of trade-offs and option evaluations to determine the best economic and social outcome with reasonable and manageable environmental and regional impact from commencement through to closure.
- Confirm the optimum technical configuration that will result in low construction and operational risk.
- The Project Definition phase evaluated Project related trade-offs with descriptions and outcomes indicated in Table 2.1.

Table 2.1 – Project Definition Phase Evaluations and High-Level Outcomes

No.	Title	Description and Work	Outcome and Recommendations
1	Process Flowsheet Evaluations	• Evaluation of possible process flowsheets to determine the best payable metallurgical recoveries. • Consider the addition of a hydrometallurgical (SART - sulphurisation-acidification-recycling-thickening) circuit as an expansion post-commissioning of the flotation concentrator in order to increase gold and silver recoveries • Gravity testing to evaluate the possibility of gravity recovery of gold and silver from bulk concentrates into a saleable product or intensive leach feed material	• Selected flowsheet will produce Copper, Zinc and Lead concentrate/. Flowsheet has simplistic process design, and higher payable recoveries. • No hydrometallurgical processes will be considered. • Further recovery test work will be undertaken for the FS phase.
2	Mobile vs. Permanent Process Crushing and aggregate production	• Comparing fixed and modular crushing equipment and their ability to produce aggregate for the Project construction phase.	• Modular crushing configuration is technically feasible and most economical. • Crushers are suitable for aggregate production with the addition of screening and washing equipment.
3	Conveyor mill feed vs. Front end loaders	• Comparisons of fixed tipper bin and front-end-loader mill feeding was compared to determine the most economical solution without incurring significant operational and safety risk.	• Mill feed bin will be loaded with a front-end loader either purchased, financed, or rented.
4	Flotation Technology	• Conventional flotation technology was compared with advanced technology flotation cells. • Bench scale testing of flotation technologies undertaken during the feasibility study test work to identify any potential advantages compared to conventional tank cell flotation with respect to recovery and flotation selectivity.	• Conventional cells will be used for FS design. • Continue with further bench scale testing during FS phase.
5	Ore Sorting / DMS	• Evaluation of ore sorting and DMS technology as a pre-crushing and milling step to reduce plant throughput and tailings volumes.	• Ore is not amenable to ore sorting technologies. Continue with crushing and milling as main comminution circuit.

No.	Title	Description and Work	Outcome and Recommendations
6	Optimized Throughput	• An integrated mine production, blending and processing scheduling exercise undertaken to determine the optimal throughput requirement for the Project.	• 1,850 tpd process throughput is selected. • Smaller mining pit is selected. • Explore underground mining possibilities from open pit.
9	Plant location	• Seven plant locations were compared to determine an optimal location, balancing construction, production, environmental and social requirements. Plant location 1 is recommended.	• Selected property is located on Adventus property with good proximity to mine. Improved shielding of communities to noise and light impact
8	Pit slope	• Current pit slope angles and associated assumptions were reviewed and determined adequate. • Further geotechnical drilling was requested to be performed as part of FS.	• Update FS mine design with final slope determinations.
9	UG Mining	• Initial study to investigate potential access can be gained from the open pit once exhausted.	• Separate PEA investigation as to feasibility of underground to be undertaken in parallel to open pit FS. • Refer to Section 24.4 for outcome.
10	Makeup Water Source	• Water balance and sourcing was evaluated to determine the need for water extraction off site. On-site water catchment and borehole options were compared with off-site river water extraction.	• Initial water balance modelling indicates a positive water balance for the site. • Borehole option and pit dewatering is deemed sufficient. water sources during construction phase. The Tailings Storage Facility (TSF) reservoir will provide sufficient water storage year round for the process water requirements. ; • River pump station contemplated in the PEA (RPA, 2019) is removed for the FS phase. The TSF will act as main water source for process water.

No.	Title	Description and Work	Outcome and Recommendations
11	Access Road Design Criteria	• The trade-off study compared the logistical requirements for the Curipamba project against the access road design criteria to determine the best access route to site. Existing and new road construction was evaluated.	• Selected route involves public road upgrading and new road section construction. Route is technically and economically feasible to proceed into FS design. • Explore possibility of an off-site consolidation yard during the FS to standardise and limit access road traffic to site.
12	Accommodation and Catering	• Various accommodation models on and off-site were evaluated for the construction and operations phases.	• Utilise regional infrastructure and accommodation as far as possible. • For Construction Personnel, a hybrid approach between on-site tented accommodation and off-site contracted accommodation is proposed as the most practical for the project execution phase. • For Operations Personnel, the self-accommodation is recommended toward providing the work force with flexibility in choosing their own settlement strategy. • Potential for a minor camp to remain on site for the operations phase to be confirmed during the FS.
13	Grid vs. On-site power generation	• Evaluate the potential for diesel generation vs. grid power.	• Results indicated that diesel generation will be more expensive and impact emission targets. • FS will proceed with grid power from early works stages to reduce construction phase generator dependency, if possible.

2.5.2 FS Phase:

- Develop the Project definition phase recommendations into engineered and estimated deliverables:
  - Mine Design;
  - Process and facility design
  - Procurement, market quotations and estimate development.
  - Design and estimating of infrastructure to support site access, mining, and processing.
- Updating of the MRE (RPA, 2019) with additional in-fill drilling data from 2019 to 2021 – referred to as "Block Model 2021".
- Pit optimisation and mine design based on updated FS phase Block Model 2021.
- Conducting further metallurgical test work, interpretation and further test work recommendations.
- Conduct initial geotechnical investigations and determine suitable engineering designs for earthworks and related infrastructure.
- Conduct and interpret further geotechnical drilling for mine pit slope design.
- Project execution scheduling and planning.
- Project capital estimation (Capex).
- Operating cost estimates (Opex) and operational readiness plan.
- Financial modelling.
- Risk and security recommendations.
- Recommended future workplans for subsequent study phases and execution preparation.

2.6 Sources of Information

DRA is the overall lead consultant in collaborating with other consultants (for scope that fall outside of DRA's responsibility) and is responsible to compile a final report that is inclusive of the work and deliverables performed by such consultants.

This Report relies on other expert consultants for the description of project elements indicated in Table 2.2. It is noted that the mention of the relevant consulting entity is to indicate the sources of information and does not necessarily indicate responsibility.

Table 2.2 – Other Experts Consultants

Project Element Involvement	Expert Entity Details
Geology and Mineral Resources	SLR Consulting (Canada) Ltd (SLR)
Tailings Storage	Klohn Crippen Berger (KCB)
Waste Facilities and Hydrogeology	Klohn Crippen Berger (KCB)
Sitewide Water Balance	SLR Consulting (Canada) Ltd
Water Quality Modelling	Minnow Aquatic Environmental Services (Minnow)
Geochemistry	pHase Geochemistry Ltd (pHase)
Environmental and Social	Knight Piésold (KP)
Other Properties	Adventus Mining Corporation (Adventus)
Public road upgrades and access road design	AOC Ingereia (AOC)
Power supply overhead powerline design	Engywork Cia Ltda (EngyWork)

This Report is based in part on technical reports, maps, published government reports, and public information, as listed in Section 27 "References" of this Report. Sections from reports, authored by other consultants listed above, may have been directly quoted/extracted, translated and/or summarised in this Report, and are so indicated, where appropriate.

2.7 Qualified Persons

At the request of Adventus, DRA has prepared the FS with the participation of specialised consultants overseen by Qualified Persons. Table 2.3 depicts the roles and responsibilities of qualified persons for the FS.

Table 2.3 – Qualified Persons

Section	Title of Section	Qualified Person
1	Summary	Each QP for related summary sections
2	Introduction	Philip de Weerdt, Pr.Eng, PMP, MBA (Project Manager, DRA)
3	Reliance on Other Experts	Philip de Weerdt, Pr.Eng, PMP, MBA (Project Manager, DRA)
4	Property Description and Location	Dorota El Rassi, M.Sc., P.Eng (Consultant Geologist, SLR)
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Dorota El Rassi, M.Sc., P.Eng (Consultant Geologist, SLR)
6	History	Dorota El Rassi, M.Sc., P.Eng (Consultant Geologist, SLR)
7	Geological Setting and Mineralisation	Dorota El Rassi, M.Sc., P.Eng (Consultant Geologist, SLR)
8	Deposit Types	Dorota El Rassi, M.Sc., P.Eng (Consultant Geologist, SLR)

Section	Title of Section	Qualified Person
9	Exploration	Dorota El Rassi, M.Sc., P.Eng (Consultant Geologist, SLR)
10	Drilling	Dorota El Rassi, M.Sc., P.Eng (Consultant Geologist, SLR)
11	Sample Preparation, Analysis and Security	Dorota El Rassi, M.Sc., P.Eng (Consultant Geologist, SLR)
12	Data Verification	Dorota El Rassi, M.Sc., P.Eng (Consultant Geologist, SLR)
13	Mineral Processing and Metallurgical Testing	Volodymyr Liskovych, Ph.D., P. Eng (Principal Process Engineer, DRA)
14	Mineral Resources Estimates	Dorota El Rassi M.Sc., P.Eng (Consultant Geologist, SLR)
15	Mineral Reserve Estimates	Daniel Gagnon, P.Eng (VP Mining & Geology, DRA)
16	Mining Methods	Daniel Gagnon, P.Eng (VP Mining & Geology, DRA)
16.3	Geotechnical	Claude Bisaillon P.Eng (Senior Geotechnical Engineer, DRA)
16.4	Pit Dewatering	Philip de Weerdt, Pr.Eng, PMP, MBA (Project Manager, DRA)
17	Recovery Methods	Volodymyr Liskovych, Ph.D., P. Eng (Principal Process Engineer, DRA)
18	Project Infrastructure	Philip de Weerdt, Pr.Eng, PMP, MBA (Project Manager, DRA)
19	Market Studies and Contracts	Philip de Weerdt, Pr.Eng, PMP, MBA (Project Manager, DRA)
20.1-20.7	Environmental Studies, Permitting and Social or Community Impact	Ken Embree P.Eng. (President, KP Canada)
20.8	Waste Facilities	Brett Stephens, RPEQ, P.Eng, P.E., (Senior Geotechnical Engineer, Principal, KCB)
20.9.1	Site Hydrology	Brett Stephens, RPEQ, P.Eng, P.E., (Senior Geotechnical Engineer, Principal, KCB)
20.9.2	Ground Water	Brett Stephens, RPEQ, P.Eng, P.E., (Senior Geotechnical Engineer, Principal, KCB)
20.9.3	Geochemical	Shannon Shaw, B.Sc., M.Sc., P.Geo (Principal, pHase Geo)
20.9.4-20.9.8	Water Management	Philip de Weerdt, Pr.Eng, PMP, MBA (Project Manager, DRA)
20.10.1 and 20.10.4	Closure – Open Pit and Mine	Philip de Weerdt, Pr.Eng, PMP, MBA (Project Manager, DRA)
20.10.2 and 20.10.3	Closure – Waste Facilities	Brett Stephens, RPEQ, P.Eng, P.E., (Senior Geotechnical Engineer, Principal, KCB)
21 all except for 21.3.3 and 21.3.4	Operating Cost Estimates	Philip de Weerdt, Pr.Eng, PMP, MBA (Project Manager, DRA)

Section	Title of Section	Qualified Person
21.3.3	Opex – Mining	Daniel Gagnon, P.Eng (VP Mining & Geology, DRA)
21.3.4	Opex – Process	Volodymyr Liskovych, Ph.D., P. Eng (Principal Process Engineer, DRA)
22	Economic Analysis	Philip de Weerdt, Pr.Eng, PMP, MBA (Project Manager, DRA)
23	Adjacent Properties	Philip de Weerdt, Pr.Eng, PMP, MBA (Project Manager, DRA)
24 all except for 24.7	Other Relevant Data and Information	Philip de Weerdt, Pr.Eng, PMP, MBA (Project Manager, DRA)
24.7	Underground Opportunity PEA level evaluation	Andre-François Gravel, P.Eng (Senior Mining Engineer, DRA)
25	Interpretation and Conclusions	Each QP for related summary sections
26	Recommendations	Each QP for related summary sections
27	References	Each QP for related summary sections

2.8 Site Visit

This Section provides details of the personal inspection of the Property by Qualified Persons.

Table 2.4 – Site Visit by Qualified Persons

Qualified Person	Site Visit	Date of Site Visit
Philip de Weerdt (DRA)	Yes	July 28-30, 2021
Daniel Gagnon (DRA)	Yes	July 28-30, 2021
Claude Bisaillon (DRA)	Yes	July 28-30, 2021
Volodymyr Liskovych (DRA)	No	-
Andre-François Gravel (DRA)	No	-
Dorota El Rassi (SLR)	Yes	January 8 to 10, 2019 July 28-30, 2021
Brett Stephens (KCB)	Yes	October 20-22, 2021
Ken Embree (KP)	No	-
Shannon Shaw (pHase Geo)	No	

2.9 Effective Date and Declaration

This Report is considered effective as of October 26, 2021 and is in support of the Adventus' press release, dated October 27, 2021, entitled "*Adventus and Salazar Announce Feasibility Study Results and Updated Mineral Resources for the Curipamba Copper-Gold Project*."

This Technical Report has the following effective dates:

- Technical Report: 26 October, 2021;
- Mineral Resource Estimate: 26 October, 2021;
- Mineral Reserve Estimate: 26 October, 2021.

2.10 Units and Currency

In this Report, all currency amounts are in US Dollars ("**USD**" or "**$**") unless otherwise stated. Quantities are generally stated in *Système international d'unités* ("**SI**") metrics units, the standard Canadian and international practices, including metric tonne ("**tonne**", "**t**") for weight, and kilometre ("**km**") or metre ("**m**") for distances.

Table 2.5 lists the abbreviations may be used in this Report.

Table 2.5 – Acronyms and Abbreviations

Abbreviation	Meaning or Units
'	Feet
"	Inch
$	Dollar Sign
$/m²	Dollar per Square Metre
$/m³	Dollar per Cubic Metre
$/t	Dollar per Tonne
%	Percent
% w/w	Percent Solid by Weight
¢/kWh	Cent per Kilowatt hour
°	Degree
°C	Degree Celsius
2D	Two Dimensions
3D	Three Dimensions
ADR	Adsorption, Desorption, Recovery
Adventus	Adventus Mining Corporation
Ag	Silver
AISC	All-In-Sustaining Costs
AOC	AOC Ingereia
AP	Acid Potential
ARD	Acid Rock Drainage
ARD	Acid Rock Drainage
As	Arsenic
As	Arsenic
ASL	Above Sea Level
Au	Gold
AuEq	Equivalent Gold
AWG	American Wire Gauge
Az	Azimuth
bcm	Bank Cubic Metre
BFA	Bench Face Angle

Abbreviation	Meaning or Units
Bi	Bismuth
BML	Base Metallurgical Laboratories Ltd
BoQ	Bill of Quantities
BSG	Bulk Specify Gravity
BSTP	Biological Sewerage Treatment Plant
BTU	British Thermal Unit
BWI	Bond Ball Mill Work Index
Ca	Calcium
$CaCO_3$	Calcium Carbonate
CAD	Canadian Dollar
CAGR	Compound Annual Growth Rate
CAPEX	Capital Expenditure or Capital Cost Estimate
Cd	Cadmium
Ce	Cesium
CEC	Cation Exchange Capacity
cfm	Cubic Feet per Minute
CFR	Cost and Freight
CIC	Carbon-in-Column
CIF	Cost Insurance and Freight
CIL	Carbon-in-Leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	Carbon-in-Pulp
cm	Centimetre
CM	Construction Management
Co	Cobalt
CO	Carbon Monoxide
CO_2	Carbon Dioxide
COG	Cut-Off Grade

Abbreviation	Meaning or Units
COV	Coefficient of Variation
Cu	Copper
CWi	Crusher Work Index
dB	Decibel
dBA	Decibel with an A Filter
DCF	Discounted Cash Flow
DEM	Digital Elevation Model
DRA	DRA Global Limited
DWI	Drop Weight Index
DWT	Drop Weight Test
DXF	Drawing Interchange Format
E	East
EA	Environmental Assessment
EHS	Environmental, Health and Safety
EIA	Environmental Impact Assessment
ELVs	Emission Limit Values
EMP	Environmental Management Plan
EP	Engineering and Procurement
EPA	Environmental Protection Agency
EPCM	Engineering, Procurement and Construction Management
ER	Electrical Room
Fe	Iron
FEED	Front End Engineering Design
FEL	Front End Loader
FeS_2 or Py	Pyrite
FOB	Free on Board
FS	Feasibility Study
ft	Feet
g	Gram
G&A	General and Administration
g/L	Grams per Litre

Abbreviation	Meaning or Units
g/t	Grams per tonne
gal	Gallons
GCW	Gross Combined Weight
GDP	Gross Domestic Product
GEIA	General Environmental Impact Assessment
GPS	Global Positioning System
H	Horizontal
h	Hour
h/a	Hour per Annum
h/d	Hours per Day
H_2	Hydrogen
H_2O	Water
H_2SO_4	Sulphuric Acid
ha	Hectare
HCl	Hydrochloric Acid
HDPE	High Density PolyEthylene
HF	Hydrofluoric Acid
HFO	Heavy Fuel Oil
Hg	Mercury
HVAC	Heating Ventilation and Air Conditioning
HW	Hanging Wall
Hz	Hertz
I/O	Input / Output
ICMC	International Cyanide Management Code
IEC	International Electro-Technical Commission
in	Inches
IRR	Internal Rate of Return
ISO	International Standards Organisation
IT	Information Technology

Abbreviation	Meaning or Units
JORC	Joint Ore Reserves Committee
JV	Joint Venture
K	Kelvin
KCB	Klohn Crippen Berger
KCl	Potassium Chloride
kg	Kilogram
kg/L	Kilogram per Litre
kg/t	Kilogram per tonne
kL	Kilolitre
km	Kilometre
km/h	Kilometre per Hour
koz	Kilo ounce (troy)
KP	Knight Piésold Consulting
kPa	Kilopascal
kt	Kilotonne
kV	Kilovolt
kVA	Kilovolt Ampere
kW	Kilowatt
kWh	Kilowatt-Hour
kWh/t	Kilowatt-Hour per tonne
L	Litre
L/h or Lph	Litre per Hour
lbs	Pounds
LCT	Locked Cycle Test
LFO	Light Fuel Oil
LG	Low Grade
Li	Lithium
LOM	Life of Mine
Ltd	Limited
LV	Low Voltage
m	Metre
M USD	Million United States Dollars

Abbreviation	Meaning or Units
m/h	Metre per Hour
m/s	Metre per Second
m^2	Square Metre
m^3	Cubic Metre
m^3/d	Cubic Metre per Day
m^3/h	Cubic Metre per Hour
m^3/y	Cubic Metre per Year
mA	Milliampere
Masl	Meters Above Sea Level
MCC	Motor Control Centre
mg	Milligram(s)
Mg	Magnesium
mg/kg	Miligram per Kilogram
mg/L	Milligram per Litre
$mg/m^2/day$	Milligram per Square Metre per Day
min	Minute
min/shift	Minute per Shift
Minnow	Minnow Aquatic Environmental Services
mL	Millilitre
ML	Metal Leaching
mm	Millimetre
mm/d	Millimetre per Day
Mm^3	Million Cubic Metres
Mn	Manganese
Mt	Million Tonne
Mtpy or Mt/y	Million Tonne per Year
MV	Medium Voltage
MVA	Mega Volt-Ampere
MW	Megawatts
MWh/d	Megawatt Hour per Day
My	Million Years

Abbreviation	Meaning or Units
N	Nitrogen
N	North
NAAQS	National Air Quality Standards
NaCN	Sodium Cyanide
NAG	Non-Acid Generating
NaHS	Sodium Hydrosulfide
NE	Northeast
NFPA	National Fire Protection Association
NGO	Non-Governmental Organisation
NGR	Neutral Grounding Resistor
Ni	Nickel
NI 43-101	National Instrument 43-101
Nm^3/h	Normal Cubic Metre per Hour
NNE	North - Northeast
NNP	Net Neutralisation Potential
NP	Neutralisation Potential
NPV	Net Present Value
NSR	Net Smelter Return
NTP	Normal Temperature and Pressure
NTS	National Topographic System
NW	North West
O/F	Overflow
OPEX	Operating expenditure / Operating cost estimate
oz	Troy ounce (31.1034768 grams)
p	Pressure
P&ID	Piping and Instrumentation Diagram
Pa	Pascal
PAG	Potential Acid Generating
Pb	Lead
Pd	Palladium
PF	Power Factor

Abbreviation	Meaning or Units
PFC	Power Factor Correction
PGE	Platinum-Group Element
Ph	Phase (electrical)
pH	Potential of Hydrogen
pHase	pHase Geochemistry Ltd
PLC	Programmable Logic Controller
ppm	Parts per Million
psi	Pounds per Square Inch
Pt	Platinum
P-T	Pressure-Temperature
PVC	Polyvinyl Chloride
QA/QC	Quality Assurance / Quality Control
QP	Qualified Person
RAP	Resettlement Action Plan
RF	Revenue Factor
RFQ	Request for Quotation
ROM	Run of Mine
rpm	Revolutions per Minute
S	South
S	Sulphur
s	Second
S/R or SR	Stripping Ratio
SABC	SAG and Ball (Milling) Circuit
SAG	Semi Autogenous Grinding
Salazar	Salazar Resources Limited
SART	
Sb	Antimony
scfm	Standard Cubic Feet per Minute
SCIM	Squirrel Cage Induction Motors
SE	South East
SG	Specific Gravity
SLR	SLR Consulting (Canada) Ltd

Abbreviation	Meaning or Units
SO_2	Sulphur Dioxide
SoW	Scope of Work
SPI	SAG Power Index
SW	South West
SWF	Sapolite Waste Facility?
t	Tonnes
t/d or tpd	Tonne per Day
t/h or tph	Tonne per Hour
t/h/m	Tonne per Hour per Metre
t/h/m²	Tonne per Hour per Square Metre
t/m or tpm	Tonne per Month
t/m²	Tonne per Square Metre
t/m³	Tonne per Cubic Metre
Ta	Tantalum
ton	Short Ton
tonne or t	Metric Tonne
ToR	Terms of Reference
TOS	Trade-Off Study
tpy or t/y	Tonne per year
TSF	Tailings Storage Facility
TSP	Total Suspended Particulates
TSS	Total Suspended Solids
TSX	Toronto Stock Exchange
U	Uranium
U/F	Under Flow
U/S	Undersize
ULC	Underwriters Laboratories of Canada
US, USA	United States (of America)
USD, $ USD, US$	United States Dollar
USGPM	US Gallons per Minute
USGS	United States Geological Survey

Abbreviation	Meaning or Units
UTM	Universal Transverse Mercator
V	Vertical
V	Volt
VFD	Variable Frequency Drive
VLF	Very Low Frequency
W	Watt
W	West
w/o	Waste / Ore
w/w	Weight per weight
WBS	Work Breakdown Structure
WRF	Waste Rock Facility
WTP	Water treatment plant
WWTP	Waste water treatment plant
X	X Coordinate (E-W)
XPS	Xstrata Process Support
XRD	X-Ray Diffraction
XRF	X-Ray Fluorescence
y	Year
Y	Y coordinate (N-S)
Z	Z coordinate (depth or elevation)
Zn	Zinc
µg	Microgram(s)
µg/m³	Micrograms per cubic metre
µm	Microns, Micrometre
µPa	Micropascal

3 RELIANCE ON OTHER EXPERTS

The QPs prepared this Report using reports and documents as noted in Section 27. The authors wish to make clear that they are QPs only in respect to the areas in this Report identified in their "Certificates of Qualified Person", submitted with this Report to the Canadian Securities Administrators.

The QPs of this Report are not qualified to provide extensive commentary on legal issues associated with Adventus's' Ecuadorian partners or presence, or the legal rights to the mineral properties. Adventus has provided certain information, reports and data to DRA in preparing this Report which, to the best of Adventus' knowledge and understanding, is complete, accurate and true.

The QPs who prepared this Report relied on information provided by experts who are not QPs. The QPs who authored the sections in this Report believe that it is reasonable to rely on these experts, based on the assumption that the experts have the necessary education, professional designations, and relevant experience on matters relevant to the Technical Report.

The QPs used their experience to determine if the information from previous reports was suitable for inclusion in this Technical Report and adjusted information that required amending. This Report includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

DRA has relied upon market studies provided by Adventus. Section 19 summarises the key information regarding the metals market overview and outlook. DRA has reviewed the content of the market study presentation and believes that it provides a reasonable overview of the current market as well as projections according to various recognised sources.

Although DRA did receive supporting documentation and consultant opinions as part of internal verification processes, the mentioned information has not been independently verified by DRA and no independent legal review was conducted. Where support documentation was received, these are referenced into Section 27.

4 PROPERTY DESCRIPTION AND LOCATION

The Project is located in Ecuador approximately 150 km south-southwest of the capital city of Quito, and approximately 150 km north-northeast of Guayaquil in the provinces of Bolivar and Los Rios. The closest town to the Project is Ventanas, which is approximately 20 km to the southwest and, in 2010, had a population of approximately 38,000 people (Figure 4.1).

4.1 Land Tenure

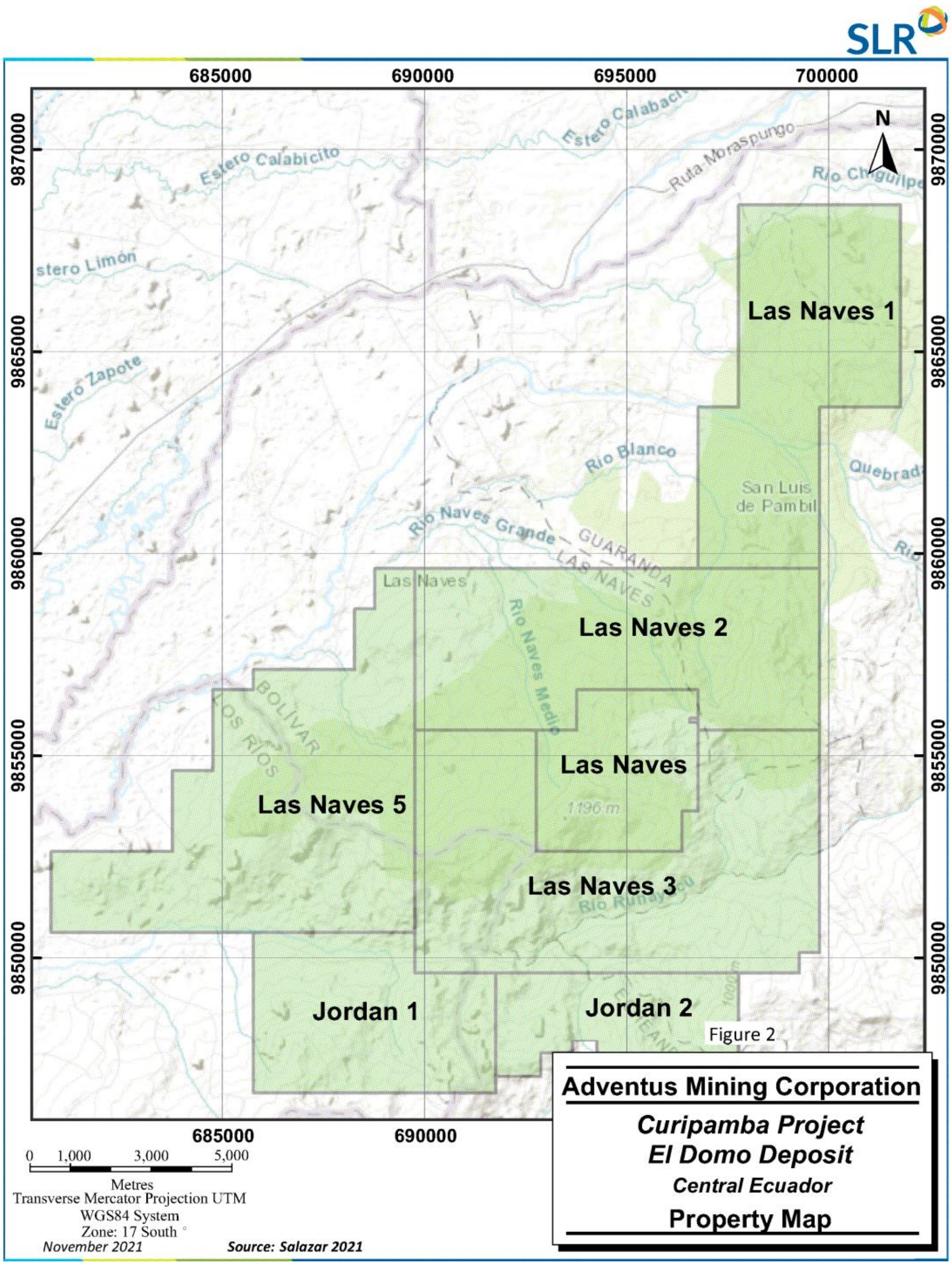
The Project comprises seven contiguous concessions with a total area of 21,537.48 ha (Figure 4.1 and Figure 4.2). The tenements are wholly owned by Curimining, one of the Ecuadorian subsidiaries of Salazar. Adventus reports that all tenements are in good standing and free of liens or encumbrances. The tenements were originally registered to Salazar between 2003 and 2006; following the enactment of the Ecuadorian Mining Act, the tenement titles were replaced by new titles on the dates shown in Table 4.1. The titles grant an exclusive right to perform mining activities, including exploration, exploitation, and processing of minerals over the area covered by the titles. Titles were granted for a period of 25 years, except for Las Naves, for which the title was granted for 22 years, 11 months, and 22 days.

Table 4.1 – Tenement Information

Tenement	Tenement Number	Area (ha)	County	Province	Date Mining Title issued	Date Registered in Mining Register	Expiry
Jordan 1	700918	2,200.00	Ventanas, Echeandía	Los Ríos, Bolívar	April 27, 2010	May 10, 2010	April 26, 2035
Jordan 2	200652	1,639.48	Echeandía, Guaranda	Bolívar	March 15, 2010	March 19, 2010	March 14, 2035
Las Naves	200508	1,458.00	Las Naves Guaranda	Bolívar	March 15, 2010	March 19, 2010	March 6, 2033
Las Naves 1	200627	3,200.00	Guaranda	Bolívar	March 15, 2010	March 19, 2010	March 14, 2035
Las Naves 2	200628	3,700.00	Guaranda, Las Naves	Bolívar	March 15, 2010	March 19, 2010	March 14, 2035
Las Naves 3	200629	4,815.00	Las Naves, Guaranda, Ventanas, Echeandía	Bolívar, Los Ríos	March 15, 2010	March 19, 2010	March 14, 2035
Las Naves 5	700885	4,525.00	Ventanas, Las Naves	Los Ríos, Bolívar	April 27, 2010	May 10, 2010	April 26, 2035

Figure 4.1 – Project Location

Figure 4.2 – Property Map



The Ecuadorian Mining Act establishes mining phases for the general mining regime, which is based on a strictly staged exploration schedule. In light of these restrictions, the tenements were classified under the small-scale mining regime in 2016, which allows for simultaneous exploration and exploitation activities without consideration of the mining phases of the general regime.

4.2 Underlying Agreement

In September 2017, Adventus entered into an agreement with Salazar whereby Adventus may earn a 75% interest in Salazar's wholly-owned Curipamba Project by funding exploration and development expenditures of US$25 million over five years, including the completion of an FS on the El Domo deposit, which was expected to be completed within three years. Adventus and Salazar subsequently and mutually agreed to extend the FS completion requirement to the first half 2022 due to expanded exploration programs at Curipamba and the COVID-19 global pandemic. Under the agreement, Adventus is required to fund 100% of the development expenditures to commercial production. Once commercial production has been achieved, Adventus will receive 95% of the dividends from the Project until its aggregate investment, including the US$25 million, has been recouped minus the approximate Salazar carrying value of US$18.2 million, after which dividends will be shared on a 75%/25% pro-rata basis with Salazar. During the option period, Salazar will help manage exploration and stakeholder relations relating to the Project in return for a 10% management fee worth a minimum of US$350,000 per year. In addition, Adventus will provide Salazar with a US$250,000 per year advance payment until the achievement of commercial production, to a maximum cumulative total of US$1.5 million. The advance payment is to be repaid preferentially to Adventus upon start of commercial production.

4.3 Permits and Authorization

Salazar, through Curimining, has complied with all of its obligations to pay the conservation patent fees and to submit annual exploration reports for the tenements for each year dating back to at least 2014. Curimining holds an environmental licence to carry out exploration activities; environmental impact studies submitted by Curimining have been approved by the environmental authorities. Curimining has been granted permits to use water and rainwater for industrial mining purposes and has complied with all obligations of filing quarterly reports with local, regional, and national authorities in order to maintain such permits in good standing. In addition, all requirements for community involvement have been met by Curimining, and Curimining has been involved in community outreach programs for the past 10 years.

Between 2007 and 2021, Curimining has been able to secure all surface rights to the tenements (Table 4.2).

Table 4.2 – Summary of Surface Access

Agreement	Seller	Date	Region	Area (ha)
Purchase	Jacinto Joselito Guarnizo Torres Marcia Soledad Peña Barrera	2007-01-24	Sesmo Sur-El Congreso	10.58
Purchase	Jeferson Hernán Barragan Gusman Diana Alexandra Chango Morales Jerson Oswaldo Barragan Guzman	2007-02-08	Sesmo Sur-El Congreso	6
Purchase	Segundo Manuel Rueda Aumala Nicanor Rueda Aumala Víctor Vicente Rueda Aumala América Bertali Rueda Aumala Richard Alberto Rueda Ramos Nicanor Temistocles Rueda Ramos Cleopatra Natair Rueda Ramos Arcesio Nicanor Rueda Gavilanes Nelly Jimena Verdezoto Paredes	2007-02-27	Sesmo Sur-El Congreso	6.28
Purchase	Curia Diocesana de Guaranda	2007-03-07	El Congreso de Runayacu	8.47
Purchase	José Franklin Ulloa Salazar y Mariuxi Paola Quijije Menendez	2011-09-20	Sesmo Sur-El Congreso	7.06
Purchase with clarification of area	Agueda Manobanda Manobanda	2018-04-16	Sesmo-Sur El Congreso	2.05
Purchase with clarification of area	Agueda Manobanda Manobanda	2018-04-16	Sesmo-Sur El Congreso	8.23
Purchase	Raúl Ramón Quilligana Urbina Blanca Dorila Gavilanes Rosero	2017-11-20	Recinto Nave Chico – Las Naves	10.54
Unification of Lots	Predios propiedad de CURIMINING S.A	2019-08-30	Las Palmeras	0.46
Purchase	Eugenio Ramiro Sarmiento Mercedes Leonila Vargas Bonilla	2017-10-18	Agua Santa	15.19
Purchase	Carmen Italia Guapulema Julio Cesar Guapulema Flores Susana Elena Gáleas Bonilla	2017-12-28	Recinto Jerúsalem	6.35
Purchase	Elsy Tatiana Morán Guapulema Helen Roxana Morán Guapulema Julio César Guapulema y Susana Elena Gáleas Bonilla	2017-12-28	Jerusalén. Predio denomiado "Dos Hermanas"	6.35
Purchase	Julio César Guapulema Susana Elena Gáleas Bonilla	2017-12-28	Jerusalén Alto.	56.45
Purchase	Julissa Jazmin Guapulema	2017-12-28	Jerusalén Alto. Predio denominado "El Recuerdo "	6.35

Agreement	Seller	Date	Region	Area (ha)
Purchase	Lucila Margot Guapulema Galeas	2017-12-28	Jerusalén Alto. Predio denominado "El Tesoro"	6.35
Purchase	Lucina Meriza Guapulema Galeas Graciela Judith Guapulema Maldonado Marcia Silvana Guapulema Maldonado Manuel Jesús Guapulema Maldonado	2017-12-29	Jerusalén Alto. Predio denominado "La Fortuna"	6.35
Purchase	Marco Gilberto Guapulema Gáleas	2017-12-28	Jerusalén Alto. Predio denominado "San Marcos"	6.35
Purchase	Maria Petita Guapulema Gáleas	2017-07-28	Jerusalén Alto. Predio denominado "Tres Hermanas"	6.35
Purchase	Leidy Lisseth Guapulema Peñaloza Ginger Jessica Guapulema Peñaloza Jenny Betzaida Guapulema Peñaloza	2017-12-28	Jerusalén Alto. Predio denominado "El Recuerdo"	6.35
Purchase	Fernando Bladimir Águila Vera	2018-06-29	Las Mercedes. Predio denominado "Centro de Acopio Las Naves"	0.5
Purchase	Mario Isidro Albán Cuzco María Mercedes Santillán Moposita	2018-09-28	Selva Alegre. Predio denominado como " Finca La Maravilla "	20.46
Unification of Lots	Predios propiedad de CURIMINING S.A	2019-09-30	La Playita	2.36
Unification of Lots	Predios propiedad de CURIMINING S.A	2020-03-10	Sector La Esperanza	44.45
a) Cancellation of mortgage and sale b) Resolution to update Boundaries	Angel Enrique García Ortiz	2020-03-05	Naves Chico	70.56
Purchase	Hector Fermín Guapulema Galeas Blanca Juduth Peñaliza Barragan	2018-01-23	Jerusalén Alto. Lote denominado "San Fermín "	14.11
Purchase	García Ortiz Hector Filemon Pacheco Martinez Paula Carmen	2019-12-09	Las Naves Chico	35.28
Purchase	García Ortiz Hector Filemon Pacheco Martinez Paula Carmen	2019-11-21	Las Naves Chico	17.64

Agreement	Seller	Date	Region	Area (ha)
Purchase	Segundo Manuel Rueda Aumala Israel Nicanor Rueda Aumala Víctor Vicente Rueda Aumala América Bertali Rueda Aumala Richard Alberto Rueda Ramos Nicanor Temistocles Rueda Ramos Cleopatra Natair Rueda Ramos Arcesio Nicanor Rueda Gavilanes Nelly Jimena Verdezoto Paredes	2007-02-26	El Congreso. Predio denominado Runayacu	26.55
Purchase	Angel Ovidio Ibarra Robayo Transito Elvira Robayo Andrade	2007-05-18	El Congreso - Guabito	37.43
Purchase	Ulloa Chasi Cesar Gonzalo Quinatoa Ulloa Luz Maria Ulloa Ulloa Carlos Critobal Borja Borja Clara Elena Ulloa Ulloa Holger Rubén Guerrero Ulloa Belgica Clemencia Borja Borja Elsa Marina	2007-05-14	Recinto El Congreso. Predio denominado como "Runayacu"	11.29
Purchase	Rea Llumitaxi Carlos Gerardo Cando Quinaloa Maria Zoila	2007-06-15	Recinto El Congreso.	7.06
Purchase	Nuñez Bonilla Gaston Tucapel Ases Flores Araceli Ofir	2007-07-24	Recinto El Guabito	12.7
Purchase	Flores Aguilar Gloria Maria Ases Flores Silvia Alicia Ases Flores Delida Lastenia Ases Flores Edison Lenin Ases Flores Neicer Ledy	2007-09-04	Parroquia Salinas. Predio denominado "El Pasaje"	63.5
Purchase	Angel Bélico García Ortiz María Angélica Tocta Camacho	2013-01-08	Naves Chico / La Merced	42.34
Purchase	Fernando Bladimir Águila Vera	2011-04-29	San Vicente	NA
Purchase	García Pacheco Blanca Isabel	2019-11-21	Naves Chico. Predio denominado "Finca Ana Paula"	17.64
Purchase	García Pacheco Flor María	2019-11-21	Recinto Naves Chico. Predio denominado "Finca Flor María"	17.64
Purchase	Carvajal Barcenes Fanny Alexandra	2021-07-26	El Congreso	27.91
Assignment of possessional rights	Miranda Escobar Leonardo Aurelio	2021-07-28	El Guabito, El Zapotal	69.76

The QP is not aware of any environmental liabilities on the property. Adventus has all required permits to conduct the proposed work on the property. The QP is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work program on the property.

4.4 Mining Rights in Ecuador

The following section has been extracted largely from thelawreviews.co.uk and BISA (2014a).

Ecuador passed a new Mining Law in 2009. The law was amended on July 16, 2013. Mining concessions can be obtained through public tender or auction processes. The public auction processes are for those mining areas that the state decides to delegate to a private party and that have not been subject to prior concession processes, while the process of public tender of mining concessions applies to those concessions that have expired or have been returned or reverted to the state. An exception for public tender or auction processes exists for right of the national mining company or foreign state companies or their subsidiaries, to acquire mining concessions directly from the state. In addition to the public tender and auction processes, the mining concession titles can be transferred between private parties, with previous written consent granted by the Ministry of Mines and Non-Renewable Natural Resources.

Mining concessions are granted for a term of up to 25 years; terms may be renewed for equal periods provided that, prior to its expiration, the mining concessionaire has presented a written petition to the Ministry of Mines. Mining concessions may be revoked by the state if minimum investments have not been made during any one year. Similarly, concession rights may be revoked if, after completion of one mining phase, the title holder does not apply for a change in concession status to reflect a more advanced mining phase.

Mining concessionaires have the exclusive right to prospect, explore, exploit, benefit, smelt, refine, market, and dispose of all mineral substances that may exist and may be obtained in the mining concessions. Currently, exploration and mining concessions are being tendered by the government to bidders under consideration of best practice in exploration and exploitation, financial and technical capacity, and commitment to high environmental standards.

The life cycle of mining concessions is broadly divided into an exploration and an exploitation phase. The exploration period is divided into three sub-periods including an initial exploration phase with a length of up to four years, an advanced exploration phase with a length of up to four years, and an economic evaluation phase of two years with the option to extend to a total of four years. In addition, mining concessions are divided into large-scale, medium-sized, and small-scale mining regimes (Table 4.3). Holders of concessions that fall under the small-scale mining regime do not need to follow the structured sequence of exploration, advanced exploration, etc., effectively allowing exploration and exploitation activities concurrently. The type of regime dictates royalties and other fees payable by the concession holder.

Table 4.3 – Summary of Fees Related to Mining Concessions

Regime Type	Production Volume	Royalties	Annual Patent Payments	Labour profit-sharing	Taxes
Small	Up to 300 tpd underground, up to 1000 tpd open pit, up to 1,500 tpd alluvial mining	3%	2% of minimum wage	10% workers, 5% state	Income Tax (25%) VAT (12%, recoverable for mineral exporters as of Jan 1, 2018) Capital Outflow Tax (5%)
Medium	301 tpd to 1000 tpd Underground. 1,001 tpd to 2,000 tpd open pit, 1,501 tpd to 3,000 tpd per day in alluvial mining	3%-8%	2.5%, 5% and 10% of minimum wage per mining phase	5% workers, 10% state	
Large Scale	More than 1,000 tpd underground, more than 2,000 tpd open pit, and more than 3,000 tpd per day in alluvial mining	5%–8% (Au, Ag, Cu)	2.5%, 5% and 10% of minimum wage per mining phase	3% workers, 12% state	Sovereign Adjustment (this is not a tax per se, but it is an economic compensation to the state to fulfil the 50/50 profit distribution among the concessionaire and the state)

Mining rights are independent of the surface rights, however, mining concessionaires or mining rights titleholders have the right to acquire, buy, rent, lease, or lend the surface lands required for the development of the mining projects or related infrastructure.

The primary permits to conduct exploration include an environmental permit, granted by the Ministry of Environment and Water, as well as a number of permits issued by the National Water Secretariat (SENAGUA).

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

The following section has been extracted largely from BISA (2014a).

5.1 Accessibility

International access to Ecuador is primarily through the airports in Guayaquil or Quito with daily flights to many international destinations. Road access to the area is excellent along paved roads, which branch off at Ventanas and Zapotal from Highway 25, the Pan-American Highway, that connects Quito and Guayaquil. Driving time from Guayaquil to the Project is approximately 2.5 hours. Numerous well-maintained gravel roads provide access throughout most of the Project area, especially in the resource area. Curimining has made improvements to certain gravel roads as part of the community outreach programs, as well as improving general access to the area for exploration and drill staff. Certain areas in the northern part of the Project can only be reached by mule or on foot.

5.2 Climate

The climate at the Project is tropical, humid, and hot most of the year. The wet season lasts from December to May, with the rest of the year considered to be the dry season. The average annual rainfall ranges from 2,200 mm to 2,500 mm, with most of the precipitation falling during the wet season. The climate has little effect on the operating season and exploration activities can be carried out year-round. Early works and site clearing will be restricted to the dry season.

5.3 Local Resources

The Project area is near the towns of Ventanas, Quevedo, Babahoyo, Las Naves and Echeandia, as well as a number of smaller villages from where a general labour force and non-specialized supplies can be sourced easily. The local economy is largely agricultural in nature, and there are no large gold or base metal mines operating in this part of Ecuador; contractors, skilled labour, heavy mining equipment, and other mining and exploration specific items would therefore need to be acquired elsewhere.

5.4 Infrastructure

There is basic infrastructure in the Project area such as road access and low voltage electricity (110 V). The access road is being upgraded to suit construction, and mine operations. The local government is currently planning to construct a new 69 kV power line between the towns of Echeandia and Las Naves. El Domo will tie into this line and construct a new power line to the Project approximately seven kilometres long, where the power will then be stepped down to the 13.8 kV on-site distribution voltage.

5.5 Physiography

The Project is located in the transition zone between the Western Mountain Range (Cordillera Occidental) and the adjacent coastal lowlands. The physiography is characterized by floodplains to the west and moderate to steep-sloped hills to the east, with elevations ranging from 100 metres above sea level (Masl) to 1,000 Masl in less than seven kilometres of horizontal distance.

Vegetation in the area mostly consists of intervened areas that have been cleared for plantations of banana, cacao, and oranges, and pastures for cattle. There are some remnants of native forest, however, very few areas remain untouched.

Local drainage is provided by small rivers off the west side of the foothills on the right bank of the Oncebí River and the left bank of the Naves Grande River. Primary drainage is through the Zapotal, Ventanas and Catarama Rivers, which drain into the Babahoyo River, which ultimately empties into the Gulf of Guayaquil.

6 HISTORY

The following section has been partially extracted from BISA (2014a) and references therein.

6.1 Exploration and Ownership History

The exploration history of the Project dates back to 1991, when the first reported exploration activity occurred. In 1991, RTZ Mining PLC Inc. (RTZ) conducted a regional stream sediment reconnaissance survey near the Project, collecting 548 samples. Results from this survey were in the public domain by 2004.

The Las Naves concessions were obtained by Mr. Leiva Ivan Santillan from the government in 2003. Subsequently, in 2005, he transferred the properties to Amlatminas, a private Ecuadorian company owned by Mr. Salazar.

In 2004, Mr. Salazar and Mr. Geovani staked 16 claims comprising the original property. The claims were held under Amlatminas. In September 2006, those claims were transferred to Salazar, a company at the time owned by Mr. Salazar and Mr. Acosta. Mr. Salazar and Mr. Acosta subsequently agreed to sell their shares in Salazar to Consolidated Kookaburra Resources Ltd (Consolidated Kookaburra). In March 2007, Consolidated Kookaburra changed its name to Salazar.

In 2005, a regional mapping and rock chip sampling program consisting of 124 rock samples was completed over parts of all 16 claims.

On March 13, 2006, Newmont Mining Corporation (Newmont) was granted a three-month exclusive access to the Project and agreed to possible joint venture (JV) terms in a Letter of Intent. Newmont conducted a Bulk Leachable Extractable Gold (BLEG) stream sediment survey comprising 225 samples. The Newmont BLEG stream sediment survey identified significant gold concentrations in streams draining mineralised areas, however, Newmont did not reach a final agreement and did not retain any interest in the property. According to Lahti (2006), the BLEG method provided a good indication of epithermal gold mineralization. The method identified the claims above the Umbe River as having potential for gold and copper mineralization. The survey also identified the southern two-thirds of the Project as being enriched in gold, silver, arsenic, and, to a much lesser degree, copper. The enrichment in arsenic, silver, mercury, and molybdenum indicates an epithermal signature over and adjacent to the gold-silver mineralization at Sesmo Sur, Las Naves Central, Roble, Caracol, and Piedras Blancas (Lahti, 2006).

In 2006, regional mapping and rock sampling continued. A total of 541 rock/chip samples were collected across the property by Amlatminas (137 samples), Newmont, Skeena Resources Limited, Pan American Silver Corporation, and Canberra and Skeena Resources.

In April 2007, Salazar contracted Geofísica Consultores of Peru to carry out induced polarization (IP) and magnetometer studies in the area where stream sediments, rock, and soil anomalies had been detected. A total of 36 lines spaced 100 m apart were surveyed for IP, for a total of 55.7 km. As a result of this work, 13 chargeability and resistivity anomalies were identified in the El Domo and surrounding areas. A total of 42 lines were surveyed by magnetometer for a total of 76.16 km.

In 2007, a broad soil survey was completed totalling 1,004 samples. Regional mapping and prospecting continued with 1070 rock/chip samples collected.

Between late 2007 and April 2008, initial core drilling (Phase I) was completed. A total of 51 core boreholes for 10,003 m tested 11 target areas (Buckle, 2009). Borehole CURI-39 intersected 12.22 m of massive sulphide mineralization at 1.20% Cu, 4.54% Zn, 3.62 g/t Au, and 51.89 g/t Ag at El Domo in February 2008.

Between 2007 and 2008. Curimining also completed stream sediment sampling, consisting of 24 samples.

On April 18, 2008, Ecuador's Constitutional Assembly passed a Constituent Mandate Resolution (the Mining Mandate) that provided, among other provisions, for the suspension of mineral exploration activities for 180 days or until a new Mining Law was approved. In January 2009, the new Mining Law was passed into law. The new Mining Law states that each company must negotiate an exploitation contract with the government.

In April and May 2008, Salazar retained Dr. Warren Pratt of Specialized Geological Mapping Ltd. to map the area of the El Domo (Las Naves) prospect, log drill core, and establish the volcanic lithostratigraphy (Pratt, 2008).

In 2008 and 2009, Curimining carried out detailed mapping and core logging to establish in more detail the local stratigraphy and improve understanding of the geology in light of the newly discovered massive sulphide mineralization. In 2008, regional mapping and rock/chip sampling resulted in 213 samples collected. Soil sampling also continued with 222 samples collected. In 2009, a total of 57 rock/chip samples were collected for analysis.

Subsequent to January 29, 2009, the new mining regulations were implemented. Between March and May 2010, the government reissued titles of the Project according to the new Mining Law to Salazar.

In June 2009, Dr. Pratt returned to conduct geological mapping and core logging at the Sesmo Sur prospect, approximately seven kilometres south of El Domo (Las Naves). The major objective of the visit was to map Sesmo Sur and prepare geological sections based on the map and core logging.

It was also proposed to create a lithostratigraphy that would allow correlation with the massive sulphide horizon at El Domo (Pratt, 2009).

In July 2009, Salazar retained Dr. Jim Franklin of Franklin Geosciences Ltd. to review the geological attributes of the newly discovered VMS deposit and occurrences at the Project. The objectives of the visit were to examine the overall setting of the occurrences, review selected drill holes to provide comment on the lithologies and principal controls on the occurrences, and to provide suggestions for further work to constrain the controls on mineralization (Franklin, 2009).

On June 3, 2010, Salazar received official notice from the Minister of Mines and Petroleum of Ecuador authorizing the restart of field operations. The notice granted Salazar the right to continue its exploration program in five properties within the Curipamba Project (Las Naves, Las Naves 2, Las Naves 3, Jordan 1, and Jordan 2) in Central West Ecuador, subject to receipt of certain permits. On January 14, 2010, Salazar received its water permit and filed an updated environmental impact assessment.

Between June 4 and September 23, 2010, a second drill program (Phase II drilling) was carried out at El Domo comprising 20 core boreholes totalling 3,241.4 m.

Limited prospecting in 2010 collected 42 rock/chip samples for analysis.

Between September 2010 and August 2011, Curimining completed a third drill program (Phase III drilling) comprising 84 core boreholes for a total of 15,582.9 m. Drilling was focused on the El Domo deposit.

In 2011, more extensive mapping and geochemical surveying programs were completed. The prospecting/mapping program collected 166 rock/chip samples for analysis. Soil sampling was also carried out, with a total of 338 samples collected. A broad stream sediment survey was completed across the property, with 135 samples collected for analysis.

Between August 2011 and April 2012, Curimining completed a fourth drilling program (Phase IV drilling), again targeting the El Domo deposit. The program comprised 51 core boreholes for a total of 10,248.8 m.

Only minor geochemical survey work was completed in 2012, with 37 rock/chip samples and 21 soil samples collected for analysis.

In 2013, only six rock/chip samples were collected for analysis.

In 2015, Salazar requested that, according to the Mining Law, the exploration status be upgraded to Advanced Exploration. This change in status was granted by the government. As part of the status change, Salazar relinquished certain parts of the property, resulting in a slightly smaller, overall tenement.

In 2016, regional mapping and prospecting resulted in the collection of 46 rock/chip samples.

Between February 2016 and September 2017, Curimining completed a fifth drilling program (Phase V drilling), comprising 33 core boreholes for a total of 9,757.4 m. The drilling focused on the El Domo deposit, specifically on the eastern edge of the massive sulphide mineralization, as well as on mineralization along the southwestern edge of known massive sulphide mineralization.

In early 2017, Curimining commissioned a resistivity-IP survey over the Barranco Colorado target and interpretation of collected data. The survey consisted of seven survey lines with a total length of 9,650 m, spaced 200 m apart.

A total of 109 rock/chip samples were collected during regional mapping and prospecting in 2017.

In 2018, 149 rock/chip samples were collected during regional mapping and prospecting and 326 soil samples were collected during a broad soil geochemical survey.

In 2018 and 2019, Curimining completed a Phase VI drilling campaign comprising of 100 core boreholes totalling 18,944 m. The drilling focused on in-fill drilling of the El Domo deposit in order to upgrade the classification of the Mineral Resource estimate.

In 2019, a total of 111 rock/chip samples were collected for analysis during regional mapping/prospecting programs. The broad soil geochemical sampling program continued, with 452 samples collected.

In 2019, Expert Geophysics Limited in alliance with MPX Geophysics Limited conducted a helicopter-borne MobileMT electromagnetic and magnetic survey totalling 2,323 line-km over the Project at a 100 m line spacing. A total of 15 high priority exploration targets were identified from the results for follow-up exploration.

6.2 Historical Mineral Resource Estimates

A "historical estimate" as defined by NI 43-101 is an estimate of the quantity, grade, or metal or mineral content of a deposit that the company has not verified as a current Mineral Resource or Mineral Reserve, and which was prepared before the issuer acquiring, or entering into an agreement to acquire, an interest in the property that contains the deposit.

The estimate in Table 6.1 is considered to be historical in nature and should not be relied upon. A QP has not completed sufficient work to classify the historical estimate as a current Mineral

Resource or Mineral Reserve and Adventus is not treating the historical estimates as current Mineral Resources or Mineral Reserves.

On March 10, 2009, an initial resource estimate for the El Domo deposit was prepared by Peter Karelse for John Buckle (Buckle, 2009). The resource was estimated using a 1.0 g/t Au cut-off grade to model the mineralised zone. The 13 drill holes that had been completed on the El Domo deposit to that time, were used to create the model of the deposit, with each hole given a 150 m zone of influence. Results of the resource estimation are presented in Table 6.1.

Table 6.1 – El Domo 2009 Mineral Resource Historical Estimate

Category	Tonnes (Mt)	Copper (%)	Lead (%)	Zinc (%)	Gold (g/t)	Silver (g/t)
Inferred	4.083	2.51	0.47	4.55	3.48	76.3

Notes:
1. CIM (2005) definitions were followed for Mineral Resources.
2. Using a 4.2 density.
3. g/t Au cut-off grade.
4. 150 m zone of influence.
5. Assays weighted by length.

6.3 Previous Mineral Resource Estimates and Engineering Studies

Several mineral resource estimates were carried out between 2010 and 2019. All these estimates are superseded by the current estimate in Mineral Resource estimate section of this report. Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standard Definitions for Mineral Resources and Mineral Reserves dated November 27, 2005 (CIM (2005) definitions) or May 10, 2014 (CIM (2014) definitions) were followed for all previous estimates.

In August 2010, Salazar commissioned Scott Wilson Roscoe Postle Associates Inc. (Scott Wilson RPA), now part of SLR, to complete an initial resource estimate of the El Domo deposit. A Technical Report in support of the disclosure of the Mineral Resource estimate was filed on SEDAR on October 13, 2010 (Valliant et al., 2010). Results of the resource estimation are summarised in Table 6.2.

Table 6.2 – El Domo Mineral Resource Estimate – September 24, 2010

Category	Tonnes (Mt)	Copper (%)	Lead (%)	Zinc (%)	Gold (g/t)	Silver (g/t)
Indicated	0.623	3.70	0.41	4.16	3.0	98.0
Inferred	2.499	3.24	0.37	4.28	4.3	79.5

Notes:
1. CIM (2005) definitions were followed for Mineral Resources.
2. Mineral Resources are estimated based on massive and semi-massive sulphide log interpretation at an NSR cut-off value of US$50/t.
3. Metal prices used are US$3.50/lb Cu, US$1.15/lb Zn, US$1.15/lb Pb, US$1,400/oz Au, and US$26.00/oz Ag.
4. Metallurgical recovery factors assumed were 72% Cu, 57% Zn, 50% Pb, 25% Au, and 20% Ag.
5. Common industry values for smelter terms were assumed.
6. Bulk density was estimated by lens, based on specific gravity determinations for each rock type.
7. A minimum thickness of 2.0 m was used.

In 2011, SLR completed an updated resource model of the El Domo deposit. This model incorporated results from the Phase III drilling. A Technical Report in support of the disclosure of the updated Mineral Resource estimate was filed on SEDAR on February 12, 2012 (Lavigne and McMonnies, 2011). Results of the Mineral Resource estimate are presented in Table 6.3.

Table 6.3 – El Domo Mineral Resource Estimate – September 29, 2011

Category	Tonnes (Mt)	Copper (%)	Lead (%)	Zinc (%)	Gold (g/t)	Silver (g/t)
Indicated	5.53	2.4	0.3	2.5	2.8	48.4
Inferred	1.46	1.9	0.3	2.8	2.4	52.5

Notes:
1. CIM (2005) definitions were followed for Mineral Resources.
2. Mineral Resources are estimated based on massive and semi-massive sulphide log interpretation at an NSR cut-off value of US$50/t.
3. Metal prices used are US$3.50/lb Cu, US$1.15/lb Zn, US$1.15/lb Pb, US$1,400/oz Au, and US$26.00/oz Ag.
4. Metallurgical recovery factors assumed were 72% Cu, 57% Zn, 50% Pb, 25% Au, and 20% Ag.
5. Common industry values for smelter terms were assumed.
6. Bulk density was estimated by lens, based on specific gravity determinations for each rock type.
7. A minimum thickness of 2.0 m was used.

In April 2013, Salazar commissioned BISA to complete a PEA (BISA PEA) of the El Domo deposit. The BISA PEA was supported by an updated Mineral Resource model that incorporated results from the Phase IV drilling. The results of the Mineral Resource estimate, with an effective date of December 15, 2013, are summarised in Table 6.4.

Table 6.4 – El Domo Mineral Resource Estimate – December 15, 2013

Unit	Category	Tonnes (Mt)	Copper (%)	Lead (%)	Zinc (%)	Gold (g/t)	Silver (g/t)
VMS	Indicated	5.468	2.52	0.30	3.27	3.23	59.19
Grainstone	Indicated	0.216	0.92	0.12	1.01	1.09	27.91
Breccia	Indicated	0.345	0.49	0.13	1.33	0.76	26.91
Gypsum	Indicated	0.051	0.94	0.03	0.39	0.34	7.40
Total	Indicated	6.080	2.33	0.28	3.06	2.99	55.81
VMS	Inferred	3.093	1.75	0.19	2.59	2.38	49.45
Grainstone	Inferred	0.170	0.96	0.10	0.69	1.00	19.24
Breccia	Inferred	0.370	0.53	0.07	0.83	0.78	24.89
Gypsum	Inferred	0.249	1.13	0.01	0.26	0.27	4.80
Total	**Inferred**	**3.882**	**1.56**	**0.16**	**2.19**	**2.03**	**42.92**

Notes:

1. CIM (2005) definitions were followed for Mineral Resources.
2. Mineral Resources are estimated based on 3D geological modelling of the VMS deposit. Four mineralised units with an NSR cut-off value of US$30/t were considered as mineral resource.
3. Metal prices used are US$2.95/lb Cu, US$0.91/lb Zn, US$0.91/lb Pb, US$1,200/oz Au, and US$20.00/oz Ag.
4. Metallurgical recovery factors assumed were based on three mineral types defined by the metal ratio Cu/(Zn+Pb):
   - Zinc Mineral (Cu/(Pb+Zn)<0.3): 15% Cu, 90% Zn, 40% Pb, 50% Au, and 65% Ag,
   - Mixed Cu/Zn Mineral (0.3≤Cu/(Pb+Zn)≤3.0): 75% Cu, 50% Zn, 0% Pb, 55% Au, and 65% Ag,
   - Copper Mineral (Cu/(Pb+Zn)>3.0): 90% Cu, 0% Zn, 0% Pb, 30% Au, and 40%.
5. Common industry values for smelter terms were assumed.
6. Bulk density was estimated based on specific gravity determinations for each lithological unit.
7. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Bulk density was estimated based on specific gravity determinations for each lithological unit.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The conclusions and recommendations related to the engineering work in the BISA study were as follows:

- The mining methods study proposes open pit mining for the western domain of the deposit and underground mining for the eastern domain.
- The economic and financial assessment considers a total 14 year life of mine, including nine years of open pit mining, with production of 2,000 tpd, followed by five years of underground mining with production of 1,000 tpd.

- The total mining production, considering the mine plans for both open pit and underground production and the reported metallurgical ore types, is:
  - Zinc mineral type: 1.74 Mt;
  - Mixed mineral Cu/Zn type: 3.00 Mt;
  - Copper mineral type: 2.18 Mt;
  - 46 Mt of waste rock and 0.98 Mt of low-grade material.
- A preliminary evaluation of infrastructure alternatives has been carried out: seven alternatives for tailings dams, five areas for the waste rock dumps, two deposits for topsoil, three alternative locations for the processing plant, four alternatives for facilities, three alternatives for the water adduction line, two alternatives for the water pipeline, five alternatives for the tailings transport system, three alternatives for the power supply system, and three alternatives for access to the mine. No major drawbacks are noted, and some work will have to be done to ensure the services required for the Project are in place.
- The preliminary estimate of pre-production capital investment totals US$110.3 million, an amount consistent with the current costs of mining and construction equipment. Operation using a mining contractor has been considered; this option reduces capital costs but increases the operating cost. The mode of operation will be analyzed in more detail in the pre-feasibility stage.
- The after-tax financial evaluation of the Project gives the following results (BISA, 2014a, pp. 2 to 4):
  - Net Present Value (NPV) at a 10% discount rate: US$86.72 million
  - Internal Rate of Return (IRR): 30%
  - Payback Period: 2 years

SLR notes that the preliminary economic assessment 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 economic forecasts on which the preliminary economic assessment is based will be realized.

On March 12, 2018, Adventus filed on SEDAR a Technical Report prepared by Roscoe Postle Associates Inc. (RPA, now part of SLR), containing an updated Mineral Resource estimate. This updated estimate included drilling from the Phase V drill program. The resource estimate had an effective date of January 19, 2018. Results are presented in Table 6.5 (Weierhäuser, 2018).

Table 6.5 – El Domo Mineral Resource Estimate – January 19, 2018

Category	Tonnes (Mt)	Grade					Contained Metal				
		Copper (%)	Lead (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Copper (kt)	Lead (kt)	Zinc (kt)	Gold (koz)	Silver (koz)
Open Pit Resources											
Indicated	6.9	1.5	0.29	2.51	2.58	52	104.2	20.3	173.5	573	11,600
Inferred	1.6	0.90	0.19	1.23	0.90	28	14.7	2.6	20.1	47	1,500
Underground Resources											
Indicated	1.8	2.06	0.18	2.09	1.47	32	37.6	3.2	38.3	86	1,900
Inferred	1.0	1.93	0.11	1.96	1.39	31	19.2	1.1	19.5	44	1,000
Total Mineral Resources											
Indicated	8.8	1.62	0.27	2.42	2.34	48	141.8	23.5	211.8	660	13,400
Inferred	2.6	1.29	0.14	1.51	1.09	29	33.9	3.7	39.6	92	2,500

Notes:

1. CIM (2014) definitions were followed for Mineral Resources.
2. A minimum mining height of two metres was applied to the Mineral Resource wireframes.
3. Bulk density assigned to mineralization ranged from 2.84 t/m³ to 3.62 t/m³.
4. Mineral Resources are reported above a cut-off NSR value of US$22/t for potential open pit Mineral Resources and US$62/t for potential underground Mineral Resources.
5. The NSR value is based on estimated metallurgical recoveries, assumed metal prices, and smelter terms, which include payable factors treatment charges, penalties, and refining charges.
6. Metal price assumptions were: US$3.25/lb Cu, US$1.10/lb Pb, US$1.30/lb Zn, US$1,500 Au, and US$23/oz Ag.
7. Metallurgical recoveries assumptions were based on three mineral types defined by the metal ratio Cu/(Pb+Zn):
   - Zinc Mineral (Cu/(Pb+Zn)<0.3): 84% Cu, 84% Pb, 95% Zn, 51% Au, and 71% Ag;
   - Mixed Cu/Zn Mineral (0.3≤Cu/(Pb+Zn)≤3.0): 83% Cu, 65% Pb, 80% Zn, 51% Au, and 63% Ag;
   - Copper Mineral (Cu/(Pb+Zn)>3.0): 90% Cu, 43% Pb, 81% Zn, 24% Au, and 40% Ag.
8. NSR factors were also based on the metal ratio Cu/(Zn+Pb):
   - Zinc Mineral (Cu/(Pb+Zn)<0.3): 31.08 US$/% Cu, 10.51 US$/% Pb, 13.71 US$/% Zn, 15.75 US$/g Au and 0.35 US$/g Ag;
   - Mixed Cu/Zn Mineral (0.3≤Cu/(Pb+Zn)≤3.0): 40.62 US$/% Cu, 7.57 US$/% Zn, 17.48 US$/g Au, and 0.32 US$/g Ag;
   - Copper Mineral (Cu/(Pb+Zn)>3.0): 49.95 US$/% Cu, 6.54 US$/g Au, and 0.12 US$/g Ag.
9. Numbers may not add due to rounding.
10. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
11. Open pit Mineral Resources have been constrained within a preliminary pit shell.

On June 14, 2019, a PEA and supporting Technical Report was completed by RPA with assistance from Knight Piésold Ltd. (KP) and filed on SEDAR (El-Rassi, 2019). The PEA was supported by an updated Mineral Resource estimate incorporating results from the Phase VI drill program. Results of the resource estimate, with an effective date of May 2, 2019, are presented in Table 6.6.

Table 6.6 – El Domo Mineral Resource Summary – May 2, 2019

Category	Tonnes (Mt)	Grade					Contained Metal				
		Copper (%)	Lead (%)	Zinc (%)	Gold (g/t)	Silver (g/t)	Copper (kt)	Lead (kt)	Zinc (kt)	Gold (koz)	Silver (koz)
Open Pit Resources											
Measured	1.4	1.92	0.37	3.52	3.75	58	27.8	5.3	50.9	174	2,704
Indicated	5.7	1.74	0.28	2.60	2.47	51	99.0	16.1	147.8	452	9,417
M+I	7.1	1.78	0.30	2.78	2.73	53	126.8	21.4	198.7	627	12,121
Inferred	0.7	0.67	0.21	1.72	1.60	46	4.6	1.5	11.9	36	1,032
Underground Resources											
Indicated	1.8	2.91	0.20	3.51	1.85	43	51.9	3.6	62.5	106	2,467
Inferred	0.6	2.46	0.19	2.82	2.09	37	15.5	1.2	17.8	42	751
Total Mineral Resources											
Measured	1.4	1.92	0.37	3.52	3.75	58	27.8	5.3	50.9	174	2,704
Indicated	7.5	2.02	0.26	2.81	2.33	49	150.9	19.7	210.3	559	11,884
M+I	8.9	2.00	0.28	2.93	2.56	51	178.7	25.0	261.3	733	14,588
Inferred	1.3	1.52	0.20	2.25	1.83	42	20.1	2.7	29.7	78	1,783

Notes:

1. CIM (2014) definitions were followed for Mineral Resources.
2. A minimum mining height of two metres was applied to the Mineral Resource wireframes.
3. Bulk density assigned on a block per block basis using the correlation between measured density values and base metal grade.
4. Mineral Resources are reported above a cut-off NSR value of US\$25/t for potential open pit Mineral Resources and US\$100/t for potential underground Mineral Resources.
5. The NSR value is based on estimated metallurgical recoveries, assumed metal prices, and smelter terms, which include payable factors treatment charges, penalties, and refining charges.
6. Metal prices are based on consensus, long term forecasts from banks, financial institutions, and other sources averaging US\$3.15/lb Cu, US\$1.00/lb Pb, US\$1.15/lb Zn, US\$1,350/oz Au, and US\$18/oz Ag.
7. Metallurgical recoveries assumptions were based on three mineral types defined by the metal ratio Cu/(Pb+Zn):
   - Zinc Mineral (Cu/(Pb+Zn)<0.33): 84% Cu, 84% Pb, 95% Zn, 51% Au, and 71% Ag;
   - Mixed Cu/Zn Mineral (0.33≤Cu/(Pb+Zn)≤3.0): 88% Cu, 85% Pb, 96% Zn, 66% Au, and 69% Ag;
   - Copper Mineral (Cu/(Pb+Zn)>3.0): 88% Cu, 69% Pb, 73% Zn, 27% Au, and 50% Ag.
8. NSR factors were also based on the metal ratio Cu/(Zn+Pb):
   - Zinc Mineral: 29.94 US\$/% Cu, 9.17 US\$/% Pb, 11.52 US\$/% Zn, 14.17 US\$/g Au, and 0.27 US\$/g Ag;
   - Mixed Cu/Zn Mineral: 44.20 US\$/% Cu, 11.34 US\$/% Zn, 22.90 US\$/g Au, and 0.27 US\$/g Ag;
   - Copper Mineral: 46.27 US\$/% Cu, 6.86 US\$/g Au, and 0.19 US\$/g Ag.
9. Numbers may not add due to rounding.
10. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
11. Open pit Mineral Resources have been constrained within a preliminary pit shell.

The conclusions and results of the PEA study include:

- 15 year mine life, with open pit for the first nine years, and combination of open pit and underground for the remainder of the mine life.
- The processing plant will process 612,500 tpa and will consist of crushing and grinding, flotation, concentrate thickening and filtration, and tailings thickening and disposal.
- The total capital cost for the Project is approximately $289 million. The pre-production capital cost is $185 million, including 25% contingency, as well as the value added tax (VAT), which will be a credit against taxes once exporting of concentrates commences, and the sustaining capital cost estimate totals $104 million.
- The LOM operating cost for the Project is estimated at $54.80/t processed.
- Considering the Project on a stand-alone basis, the undiscounted after-tax cash flow totals $565 million over the mine life, and simple payback occurs two years from start of production.
- The after-tax NPV at an 8% discount rate is $288 million, and the after-tax IRR is 40%.

SLR notes that the preliminary economic assessment 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 economic forecasts on which the preliminary economic assessment is based will be realized.

7 GEOLOGY AND MINERALISATION

The Project is located in western central Ecuador, in the province of Bolívar near the town of Ventanas. It is part of the western and lowermost foothills of the northern Cordillera Occidental of the Andes, near the boundary with the western coastal plain (Figure 7.1). Geologically, the Project area is located in the central equatorial Andes, within the Macuchi Terrane, a predominantly juvenile magmatic island arc of Paleocene–Eocene age adjacent to the Amazonian Craton.

7.1 Regional Geology

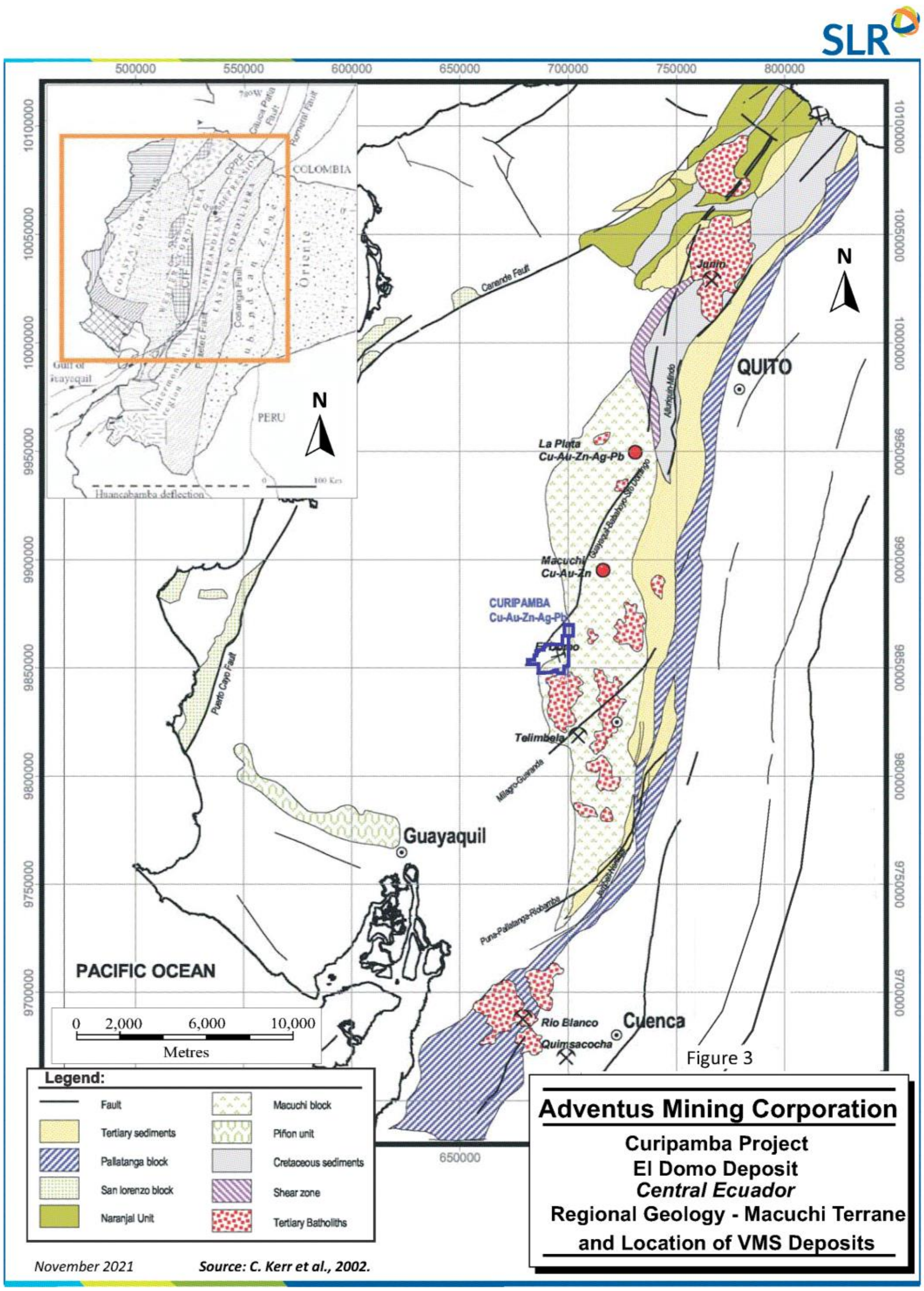
The Andes of Ecuador comprise two mountain chains, the Cordillera Central and the Cordillera Oriental, separated by a central inter-Andean basin. To the west, the Cordillera Occidental predominantly consists of fault-bounded Cretaceous to Tertiary volcanic oceanic and island-arc terranes (Litherland and Aspden, 1992; Kerr et al., 2002; Spikings et al., 2005). Here, strike-slip fault displacement along approximately north-south trending faults has resulted in a complicated assemblage of tectono-stratigraphic units that juxtaposes volcanosedimentary successions of similar lithologies but different ages. These terranes, of dominant oceanic affinity, have traditionally been interpreted as being successively accreted onto the western edge of the Amazon craton along a long-lived continental margin. Accretion occurred from the Late Jurassic to the Eocene. Superimposed on the accretionary assemblage are four magmatic arcs, which are related to the subduction of the Farallon/Nazca plate beneath the continent.

The largest, and also the youngest, accretionary units are the Pallatanga and Macuchi terranes (Figure 3), which have been interpreted to have been accreted during the Eocene. The Macuchi Terrane, which hosts the Curipamba Project, is several hundred kilometres long and tens of kilometres wide. In the past, it has been interpreted as an allochthonous terrane accreted to the Pallatanga continental margin during the Late Eocene in response to the closure of a back-arc basin (Spikings et al., 2001; Hughes and Pilatasig, 2002; Kerr et al., 2002). Recent work suggests that the Pallatanga Terrane represents the oceanic basement of the Western Cordillera and a dismembered terrane from the Caribbean plateau (Luzieux et al., 2006; Vallejo et al., 2006; Spikings et al., 2005; Vallejo et al., 2009). In addition to the new interpretation of the Pallatanga Terrane, the exact nature of the Macuchi Terrane is not yet clear. The presence of detrital zircon of Cambrian age inherited from the nearby basement suggests that the Macuchi Terrane may represent a forearc basin that formed near its present position and close to an eroding basement like that in the Eastern Cordillera.

The Macuchi Terrane crops out along the western flank of the Cordillera Occidental (between 0° and 2°30’ S) and contains an intra-oceanic island-arc volcanic sequence represented by the namesake Macuchi Group, which is interpreted as an oceanic plateau sequence. The Macuchi Group comprises predominantly submarine volcanic and volcaniclastic rocks with subordinate sedimentary rocks (BGS-CODIGEM, 1993; McCourt et al., 1997). More than 80% of the sequence is dominated by volcaniclastic and epiclastic rocks, including lithic-rich sandstone and breccia with accessory siltstone and chemical sediments. The latter are mostly cherty sediments grouped in a turbidite-like sequence. The sequence also includes domes and flows of basalt to basaltic andesite, with abundant pillow lavas/breccias and hyaloclastic textures as well as their sub-volcanic equivalents (dikes) of micro-porphyritic basalt and diabase. Most of these rocks show pervasive hydrothermal-submarine alteration with chlorite-epidote (Aguirre and Atherton, 1987). The non-volcanic sedimentary fraction is dominated by scarce calcarenite and recrystallized limestone, which are associated with what has been interpreted as a reef system (Hughes and Pilatasig, 2002). The Macuchi Group has an estimated minimum thickness of 2.0 km to 2.5 km (Aguirre and Atherton, 1987).

The Macuchi Group can be divided further into two primary sub-units (Chiaradia and Fontboté, 2001). The Basal Macuchi unit includes primitive basalt, mostly as submarine lava flows, interbedded with mudstone. The Main, or upper, Macuchi unit contains predominantly volcaniclastic, less primitive, basaltic andesite to andesite. Although most research emphasizes the predominance of mafic rocks, the Macuchi Group also includes several dacitic to rhyolitic domes that appear to be directly related to massive sulphide mineralization similar to those found at El Domo (Chiaradia and Fontboté, 2001; Vallejo, 2013), suggesting that the magmatism was bimodal.

Figure 7.1 – Regional Geology – The Macuchi Terrane and Location of VMS Deposits



7.2 Property Geology

The geology of the Project is characterized by the predominantly volcanic and volcaniclastic rocks of the Macuchi Group of Middle Paleocene–Eocene age (Hughes and Pilatasig, 2002; Vallejo, 2007; McCourt et al., 1997). In the Project area, rocks of the Macuchi Group are overlain by Late Tertiary to Holocene volcanic rocks and Holocene alluvial deposits that are the dominant outcropping rocks. In addition, the Project is covered by five metres to six metres of volcanic ash from eruptions of the Quilotoa Volcano.

Specifically, the volcanic pile in the Project area comprises a basal rhyodacite unit overlain by two interfingering volcaniclastic sequences, one mafic and the other felsic, and two coherent younger lithofacies, one andesitic and the other rhyolitic, which intruded the sequence in both the north and south of the property. The massive sulphides are located along the contact between the rhyodacite and the volcaniclastic rocks, but also within the mafic volcaniclastics. These latter rocks, locally known as grainstone, are interpreted as a marker unit in the immediate hanging wall of the massive sulphides (Franklin, 2009) that may be used to guide exploration for additional camp-wide resources. Figure 7.2 is a geological map of the Las Naves/El Domo area (Pratt, 2008). The strata are generally sub-horizontal with an overall synclinal shape. Numerous sub-vertical faults that strike northerly and east-northeasterly complicate the overall geology with vertical offsets of up to approximately 50 m.

The massive sulphides are related to a zone of abundant hydrothermal alteration, which includes extensive sericitization-silicification in the rhyodacitic footwall and widespread silicification-chloritization-argillitization in the overlying mafic volcaniclastic rocks. The rhyodacite hosts a sulphide-rich stockwork zone and abundant gypsum replacing earlier anhydrite. The known lateral extent of the massive sulphide mineralization is approximately 1,000 m by approximately 800 m. Additional mineralization increases the footprint of the known mineralization to approximately 1,300 m by 1,100 m.

The area lacks indication of regional metamorphism, such as metamorphic mineral assemblages or foliation fabric. Instead, volcanogenic textures such sulphide replacement, collapse breccias, and peperites are widespread and well preserved in core.

The formation of the El Domo deposit led to abundant stratabound hydrothermal alteration mainly controlled by the fluid-rock ratio and the composition of the protolith. Overall, the area is affected by a large hydrothermal halo related to the mineralization process. Felsic rocks in the footwall of the main mineralised body have been affected by quartz-sericite alteration, whereas polymictic breccias in the hanging wall have been altered irregularly to an assemblage including chlorite, phyllosilicates, and quartz. Other rocks in the area are virtually unaltered and only the igneous rocks show subtle seafloor hydrothermal alteration.

A characteristic feature of El Domo is large amounts of gypsum and its high-temperature precursor, anhydrite. The calcium sulphates ± pyrite form thick veins and stratabound bodies that are up to five metres thick, mostly in a semi-continuous stratabound zone beneath the massive sulphides (Schandl, 2009). Anhydrite is common in recent submarine hydrothermal systems; its occurrence in an old system is evidence of the low amount of alteration and recrystallization post mineralization. The gypsum shows frequent evidence of ductile disturbance and movement along faults.

7.3 Regional Tectonic Setting

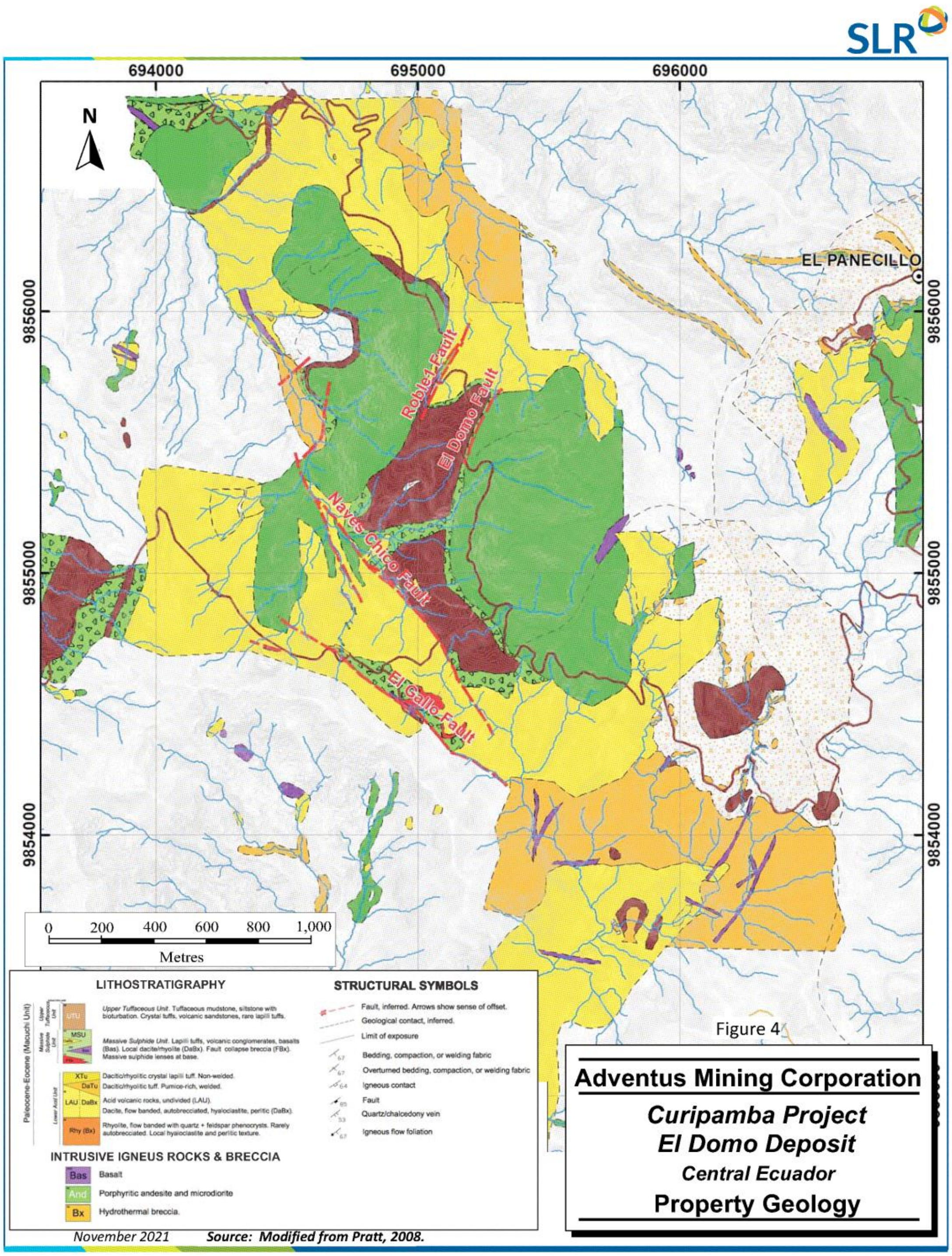
The Project area has been interpreted as a north-northeast trending graben with a minimum size of approximately six square kilometres. Structural investigations to date have identified a fault pattern consistent with dextral movement along major faults that resulted in the opening of pull-apart or transtensional basins. As interpreted by Mayor (2010), the graben is bounded by the steeply dipping Roble 1 and El Domo faults. It is truncated at the southwestern boundary against footwall rocks, by the northwesterly striking Naves Chico Fault Zone, and at the northeastern boundary by an unnamed fault. The western edge is defined by the Cade and Cade Sur anomalies and a string of unnamed and untested prospects hosted in breccia. The graben structure itself is crosscut by east-northeasterly striking extensional faults with little displacement that predominate in the area and control drainage.

Evidence for faulting is widespread in core and ranges from zones of intensely broken core indicating brittle faulting and cracking to distinct fault breccia and gouge with features of brittle-ductile shearing in the rocks with strong argillic alteration. Locally, these latter structures define major shear zones with up to 10 m of hydrothermally altered rocks (including brecciated and sheared fragments of massive sulphides) showing tectonic cleavage and, therefore, indicative of brittle-ductile behaviour. Furthermore, Pratt (2008) describes the injection of ductile sericite/illite and gypsum into the fault zones that form significant amounts of the fault gouge.

Current understanding of the timing of structural events is that the graben formed in the Eocene in a transpressional stress regime that also controlled the deposition of the volcanosedimentary rocks of the Hanging Wall Unit. The geological data suggest that the El Domo deposit formed in a third-order basin within a larger intra-magmatic arc basin. There are no geophysical or geological data supporting the presence of a caldera, however, the structure and composition of the volcaniclastic rocks are typical of caldera settings.

Late extensional faults likely led to local remobilization of sulphide mineralization and further modified north trending grabens likely related to post-collisional transform faults of post-Eocene to Quaternary age. Late dextral reactivation has also been observed in many of the faults, likely related with the modern reactivation of the Chimbo-Toachi Shear Zone.

Figure 7.2 – Property Geology



7.4 Mineralisation

Pratt (2008) was the first to document and describe a Kuroko-type VMS environment on the Project concessions. He established a lithostratigraphy for the Las Naves/El Domo area in which massive sulphide mineralization rests on a footwall sequence of rhyolite and dacitic autobreccias. He divided the sulphide mineralization into five types:

- Massive sulphides with indistinct texture. In some places, a fragmental texture can be seen within the sulphides, suggesting that they may be formed by the replacement of lapilli tuff.
- Sulphide-altered lapilli tuffs and peperites.
- Transported sulphide fragments within polymictic lapilli tuffs.
- Sulphide “pseudo”-fragments within polymictic lapilli tuffs.
- Rare, thinly laminated siliceous chert with banded sulphides.

The mineralised zone at El Domo is an intact, upright and only mildly disturbed Kuroko-type VMS deposit. As such, it displays the characteristic zoning of the model type from the underlying feeder pipe area through vertical and lateral variations upward to the abrupt termination of the massive sulphides against the characteristic hanging wall grainstone marker defined by Franklin et al. (2005). Over time, the evolution of the hydrothermal mineralizing system and the growth of the mineralised deposit account for the spectrum of mineralization types distinguished by Pratt.

Schandl (2009) conducted a petrographic study on 17 core samples from the Naves Central area and provided the first details on the mineralogy. Sphalerite, chalcopyrite, and pyrite are the principal sulphides in the mineralised rocks from the Curipamba prospect. Galena is less common, and tennantite/tetrahedrite and covellite are minor phases. Gold was identified within sphalerite + galena + barite mineralization, where it occurs as minute (5 µm to 50 µm) inclusions in sphalerite. The colloform banded sphalerite also contains an abundance of large, partly dissolved inclusions of skeletal galena. Careful microscopic examination revealed that gold was introduced to sphalerite via fractures with late chalcopyrite. Minute gold also occurs on the rim of some galena and is intergrown with chalcopyrite. The galena is partly replaced by tennantite, and it is rimmed and crosscut by chalcopyrite veinlets. Two small grains of gold were also identified in a late carbonate veinlet that crosscuts the sphalerite. The sphalerite is a pure zinc end-member with little or no iron content. In a number of samples, sphalerite is colloform banded, and just as some pyrite, often has framboidal texture. Textural evidence suggests that galena was largely contemporaneous with sphalerite, and both post-dated the pyrite. Tennantite and tetrahedrite represent a relatively minor phase and both crystallized at the expense of galena and less commonly, pyrite. Chalcopyrite was the last sulphide to crystallize in the polymetallic assemblage. In some samples, fragmented pyrite and sphalerite are “flooded” and partly replaced by massive chalcopyrite. Locally, chalcopyrite is stained to an unusual purple/blue colour. Microprobe analysis showed that in these domains the chalcopyrite has an unusual chemistry, and contains 2.2% to 3.7% bromine (by weight). Galena occurs as a skeletal inclusion in chalcopyrite and as replacement after pyrite. It contains inclusions of, and can be partly replaced by, tennantite and tetrahedrite. Covellite and chalcocyanite occur within sediments. Covellite forms a rim on detrital sphalerite and some pyrite, and chalcocyanite (anhydrous Cu-sulphate) is disseminated through the matrix. Barite is the principal gangue mineral (Schandl, 2009).

Mineralization at El Domo is broadly zoned with an upper “cap” of barite, enriched variably in silica sphalerite, galena, and gold. This cap is underlain by a massive sulphide zone with local zoning of zinc-rich mineralization along the hanging wall contact and a copper-rich base. This zonation, however, is not apparent throughout the massive sulphide zone. Zinc-rich mineralization consists of low iron sphalerite, some sulphosalts, barite, and pyrite. Copper mineralization is characterized by chalcopyrite and abundant pyrite. The base of the sulphide section is typically strongly silicified, with semi-massive pyrite and chalcopyrite as disseminations and stringer veins.

Mineralization in the grainstone shows evidence of at least two mineralization events. The sulphides include breccias which appear to have been caused by some form of collapse (possible anhydrite dissolution), while interstitial spaces were infilled by sphalerite and, in some cases, chalcopyrite. This brecciation replacement texture is common in many massive sulphide mounds as they grow by “displacement” or expansion. During this process, the core of a mound is constantly impregnated by high temperature hydrothermal fluid, displacing the lower temperature minerals outwards and leading to a constant zone refinement of the mineralization. Some samples of massive sulphide in CURI-048 display ovoid features that may be replaced tube worms that have become incorporated in a growing sulphide mound on the seafloor (Franklin, 2009).

The sulphide and precious metal compositions have numerous unusual features, usually associated with high temperature systems that have achieved boiling just prior to their expulsion on or near the seafloor. The exceptionally high gold recorded in many of the upper zones in all of the occurrences, together with the anomalous antimony, arsenic, mercury, and bromine contents of some of the minerals, can only be achieved by this process, which enables exceptionally efficient gold precipitation. Gold is conserved in the vapour phase of a hydrothermal fluid, and thus may be deposited over a much wider area than the base metals. At the Project, gold is generally associated with baritic exhalite.

8 DEPOSIT TYPES

The mineralization at El Domo shares most of the features of a VMS deposit (Franklin et al., 2005; Franklin et al., 1981; Large, 1992; Large et al., 2001; Lydon, 1996; Lydon, 1988a; Lydon, 1988b). VMS deposits are major sources of Zn, Cu, Pb, Ag, and Au, and can contain trace metals such as Co, Sn, Se, In, Bi, Te, Tl, Ga, and Ge. There are over 800 VMS deposits known worldwide, up to 56 of which are considered world class (>32 Mt). VMS deposits occur throughout geological history and typically occur in clusters, or camps, such as the Noranda and Matagami Camps in the Abitibi Greenstone Belt, the Flin Flon–Snow Lake Camp in the Flin Flon Greenstone Belt, the Bathurst Camp in New Brunswick, the Iberian Pyrite Belt in Spain, and the Mokuroko district in Japan (Large and Blundell, 2000).

Deposits of this type are spatially and chronologically related to submarine felsic and/or mafic volcanism and are characterized by an underlying stockwork or feeder zone related to major hydrothermal alteration, which is typically more prominent in the footwall than in the hanging wall, and massive or semi-massive mineralization formed on or near the seafloor.

The principal model for the genesis of VMS deposits includes a submarine rifting environment, where a sub-seafloor magma chamber drives a hydrothermal convective cell. Cold unmineralised sea water enters the oceanic crust through faults and cracks distal to the magma chamber. As the fluids circulate as part of the convective cell, fluids get progressively heated due to increasing proximity to the magma chamber and, in the process, leach metals out of the surrounding rocks. Eventually, fluids ascent back to the seafloor along focused exit pathways due to a lower specific gravity than surrounding water as heated, metal laden hydrothermal fluids. Renewed interaction with cold, low Eh and pH seawater at and near the seafloor leads to the precipitation and deposition of metals that were held in solution.

Most ancient VMS deposits still preserved in the geological record formed mainly in oceanic and continental nascent-arc, rifted-arc, and back-arc settings. The crustal composition exerts a major control on the mineral contents of VMS deposits, with Cu-Au-Zn deposits forming mainly on the primitive crust and Zn-Cu-Pb-Ag deposits, on continental crust (Barrie and Hannington, 1999).

9 EXPLORATION

Historical exploration in the Project area is discussed in Section 6 History.

In early 2020, Adventus implemented a target generation initiative (TGI) for the Curipamba Project. The goal of the TGI was to synthesize the airborne MobileMT survey from 2019 with the historic work on the property. The magnetotelluric (MT) and magnetic data were examined in conjunction with historic drilling, mapping, prospecting, and soil and stream sediment sampling data. A total of 15 priority targets were identified for further investigation as detailed in a January 21, 2020 news release (Figure 9.1).

A program of mapping and prospecting was undertaken during the 2020/2021 field program. A total of 124 rock/chip samples were collected for analysis. Most samples were collected in the area around the Agua Santa target, with scattered coverage near other targets identified by the TGI. Results from Agua Santa returned a maximum assay value of 14.7% Cu, with eight of 83 samples >1% Cu. They also returned a maximum value of 11.4 g/t Au, with eight of 83 samples >1 g/t Au. Six samples returned >25% Zn, with 19 of 83 samples >1% Zn. A maximum value of 120 g/t Ag was returned, with 12 of 83 samples >10 g/t Ag. Only one sample returned >1% Pb (1.2%). There were no strongly anomalous results from Cu, Au, Zn, Ag, or Pb from the 41 samples collected elsewhere on the property (Figure 9.2).

A soil sampling program was also completed in 2020/2021 (Figure 9.3). A total of 639 samples were collected for analysis. The majority of samples collected were in the Agua Santa and surrounding area (569 samples). Results show a strong anomalous trend extending south from Agua Santa. The trend ends to the north of Agua Santa. There are some strong isolated (>200 ppm Cu) anomalies to the east of Augua Santa. The other area of focus was the La Vaquera target and surrounding area. A, where a total of 70 samples were collected. There is a strong soil (>200 ppm Cu) anomaly that justifies following up with a tighter grid spacing.

Figure 9.1 – 2020 Target Generation Initiative

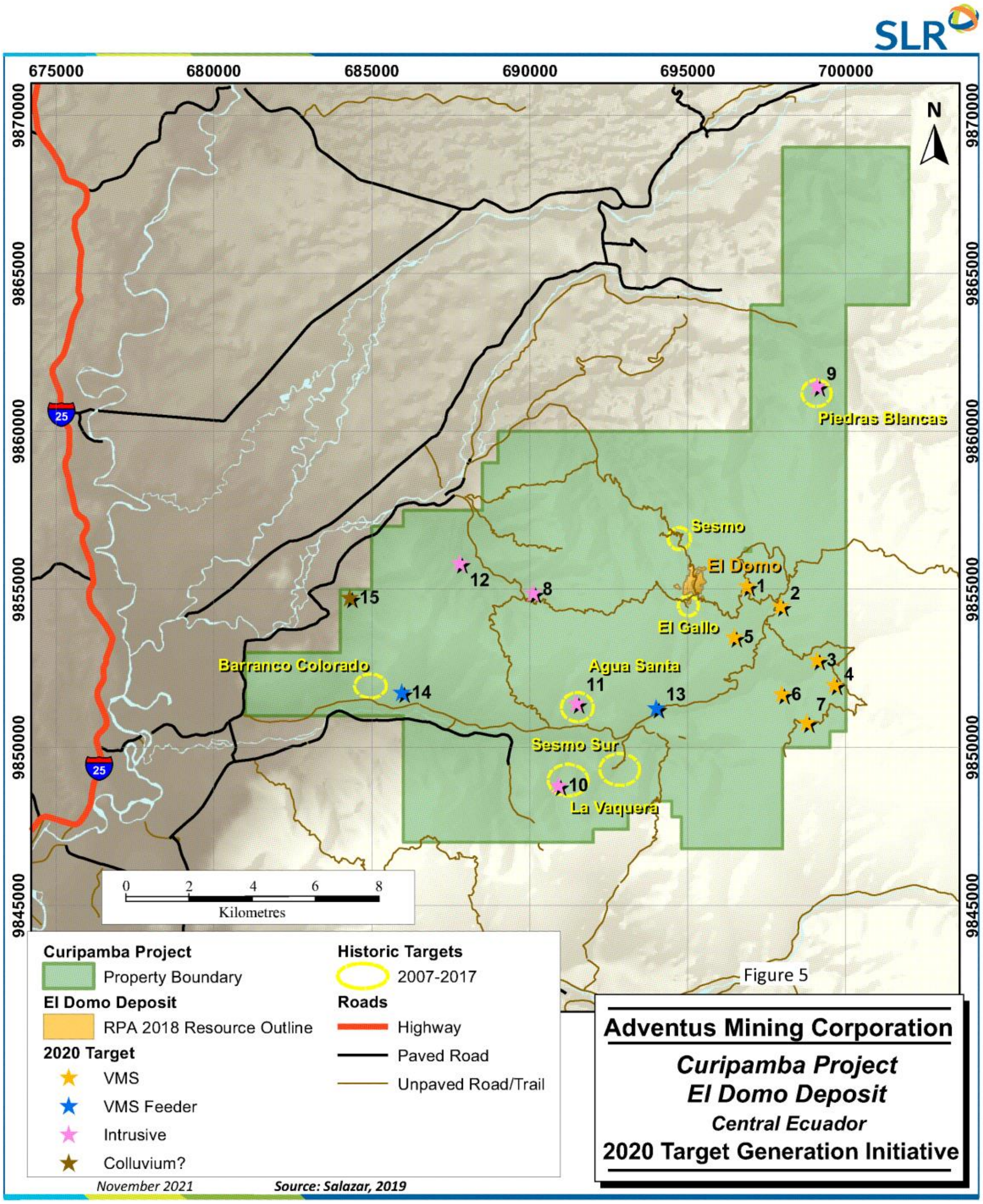


Figure 9.2 – Curipamba Regional Rock/Chip Geochemical Sampling

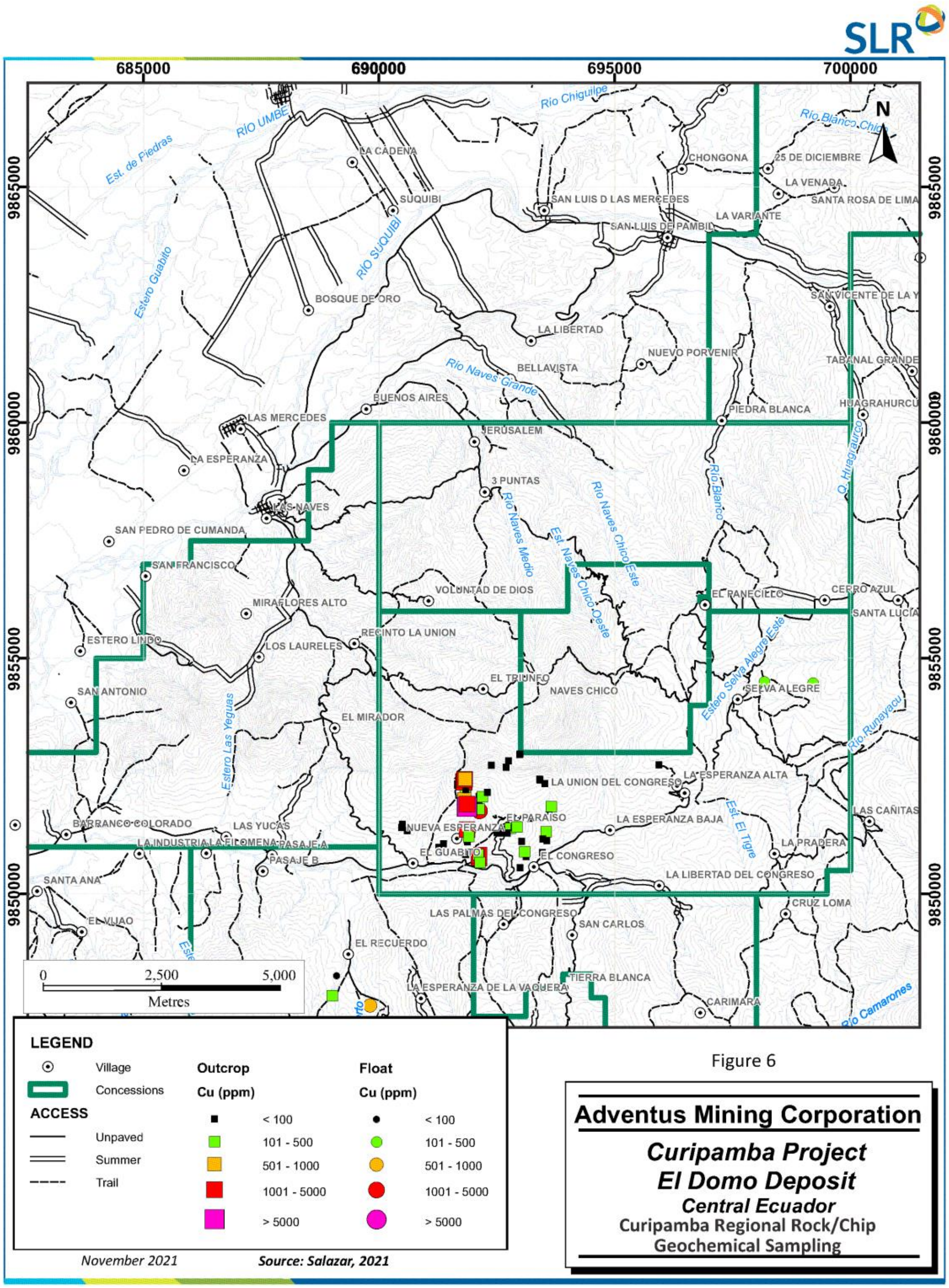
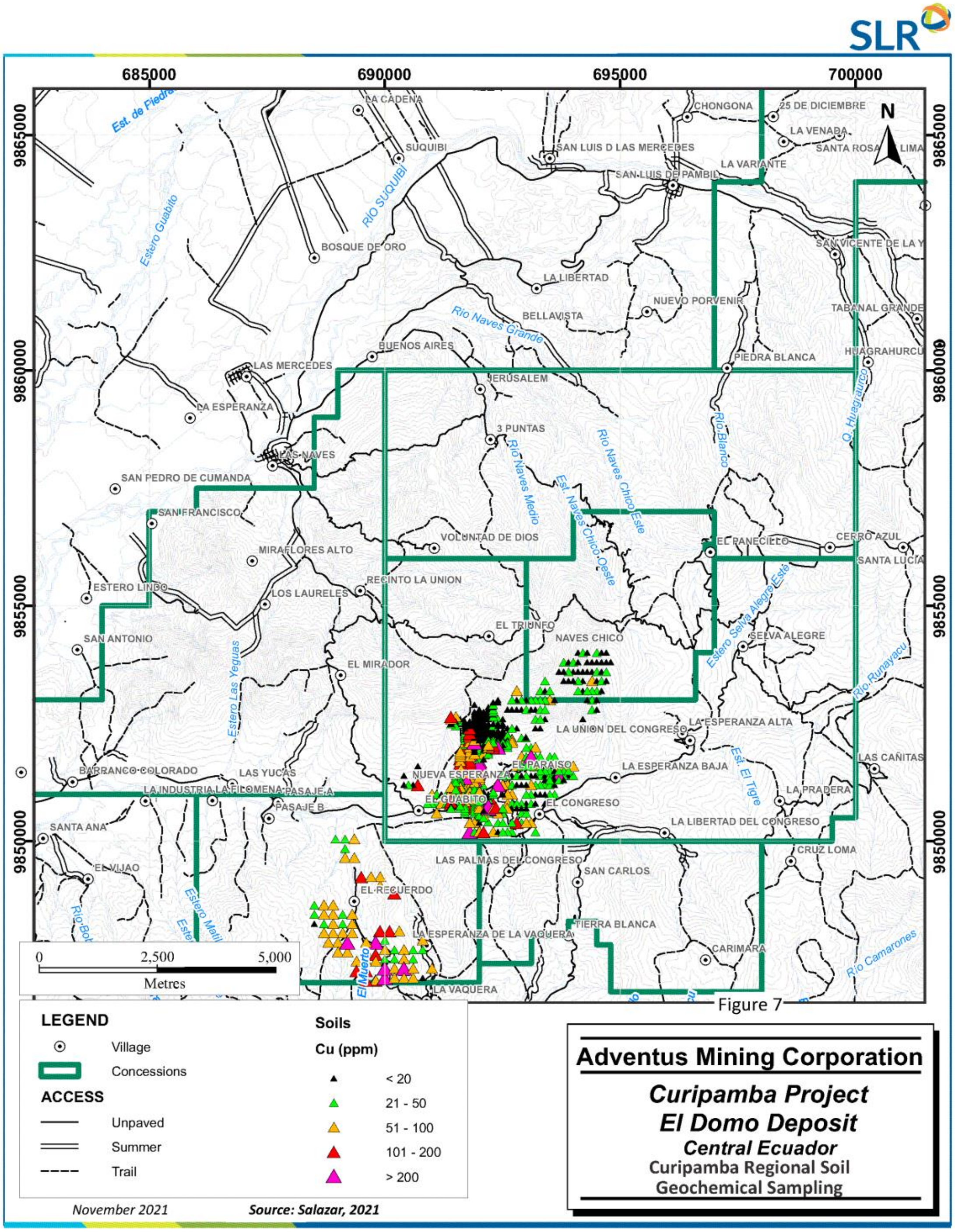


Figure 9.3 – Curipamba Regional Soil Geochemical Sampling

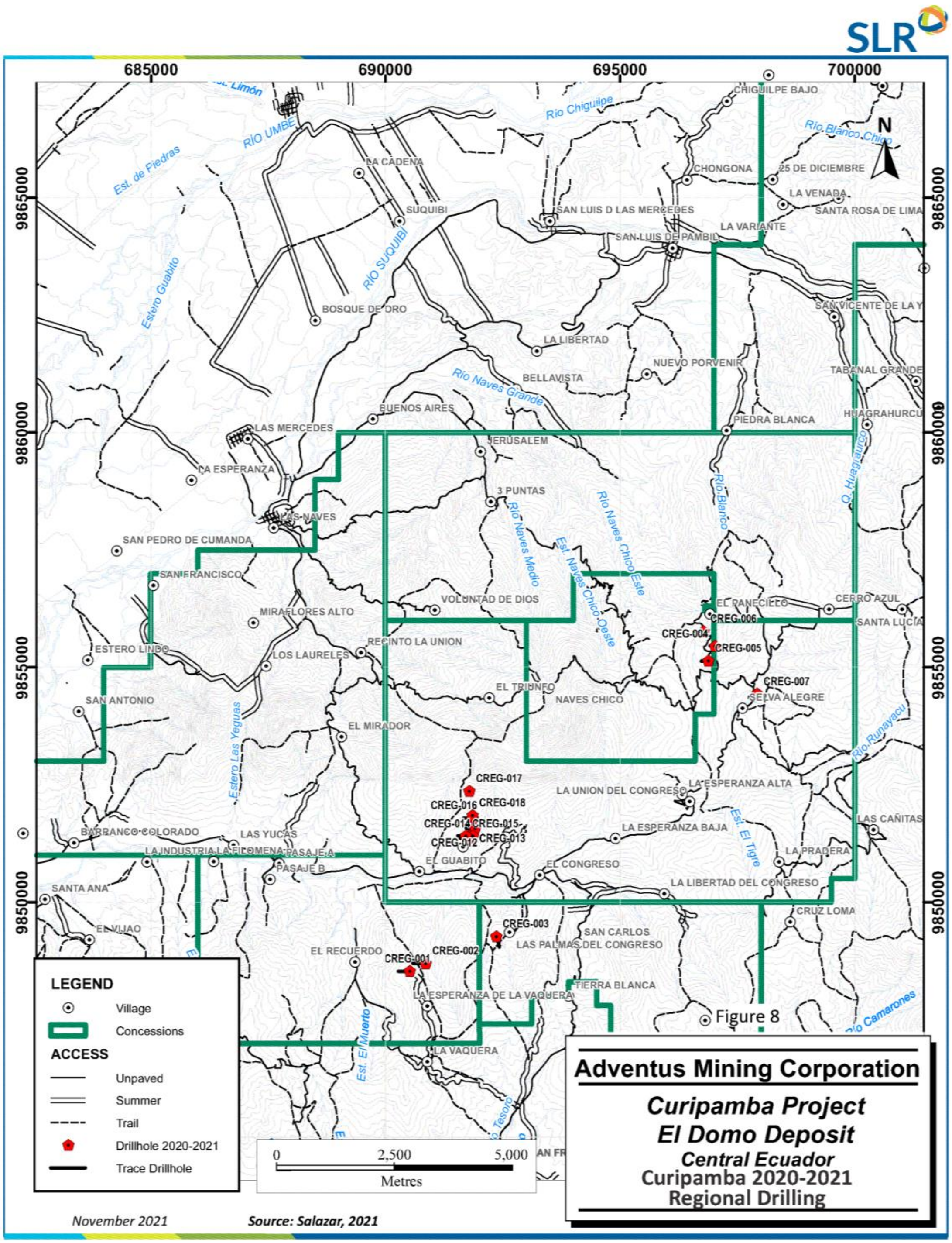


Regional drilling was undertaken between November 14, 2020 to January 10, 2021, and April 22 to September 24, 2021. A total of 18 holes were drilled on priority targets identified during the TGI; a total of 5,581 metres were drilled (Table 9.1, Figure 9.4).

Table 9.1 – Curipamba 2020/2021 Regional Drilling

Hole ID	Depth (m)	Azimuth (˚)	Dip (˚)	Target	Easting	Northing	Elevation	Start date	Finish date
CREG-001	458	270	-60	LA VAQUERA	690518	9848536	544	11/14/2020	12/09/2020
CREG-002	554	270	-60	LA VAQUERA	690855	9848700	528	12/11/2020	01/13/2021
CREG-003	429	160	-60	SESMO SUR	692355	9849275	472	12/27/2020	01/10/2021
CREG-004	398	270	-75	PANECILLO	697021	9855457	994	04/22/2021	05/02/2021
CREG-005	405	270	-70	PANECILLO	696881	9855146	954	05/03/2021	05/15/2021
CREG-006	271	270	-70	PANECILLO	696776	9855774	1025	05/16/2021	05/22/2021
CREG-007	249	270	-70	SELVA ALEGRE	697916	9854441	795	05/24/2021	05/30/2021
CREG-008	324	270	-50	AGUA SANTA	691805	9851410	448	06/14/2021	06/26/2021
CREG-009	258	270	-50	AGUA SANTA	691855	9851475	445	06/28/2021	07/03/2021
CREG-010	285	270	-50	AGUA SANTA	691778	9851372	453	07/05/2021	07/09/2021
CREG-011	242	270	-85	AGUA SANTA	691805	9851410	448	07/11/2021	07/18/2021
CREG-012	285	270	-50	AGUA SANTA	691908	9851586	460	07/20/2021	07/28/2021
CREG-013	203	270	-85	AGUA SANTA	691859	9851410	420	07/29/2021	08/03/2021
CREG-014	242	270	-50	AGUA SANTA	691705	9851410	465	08/04/2021	08/15/2021
CREG-015	209	90	-85	AGUA SANTA	691705	9851410	465	08/16/2021	08/20/2021
CREG-016	261	270	-50	AGUA SANTA	691849	9851794	510	08/24/2021	08/29/2021
CREG-017	230	105	-70	AGUA SANTA	691783.2	9852370	638.3	09/06/2021	09/12/2021
CREG-018	278	125	-62	AGUA SANTA	691851.5	9851853	509.4	09/18/2021	09/24/2021
Total	**5,581**								

Figure 9.4 – Curipamba 2020/2021 Regional Drilling



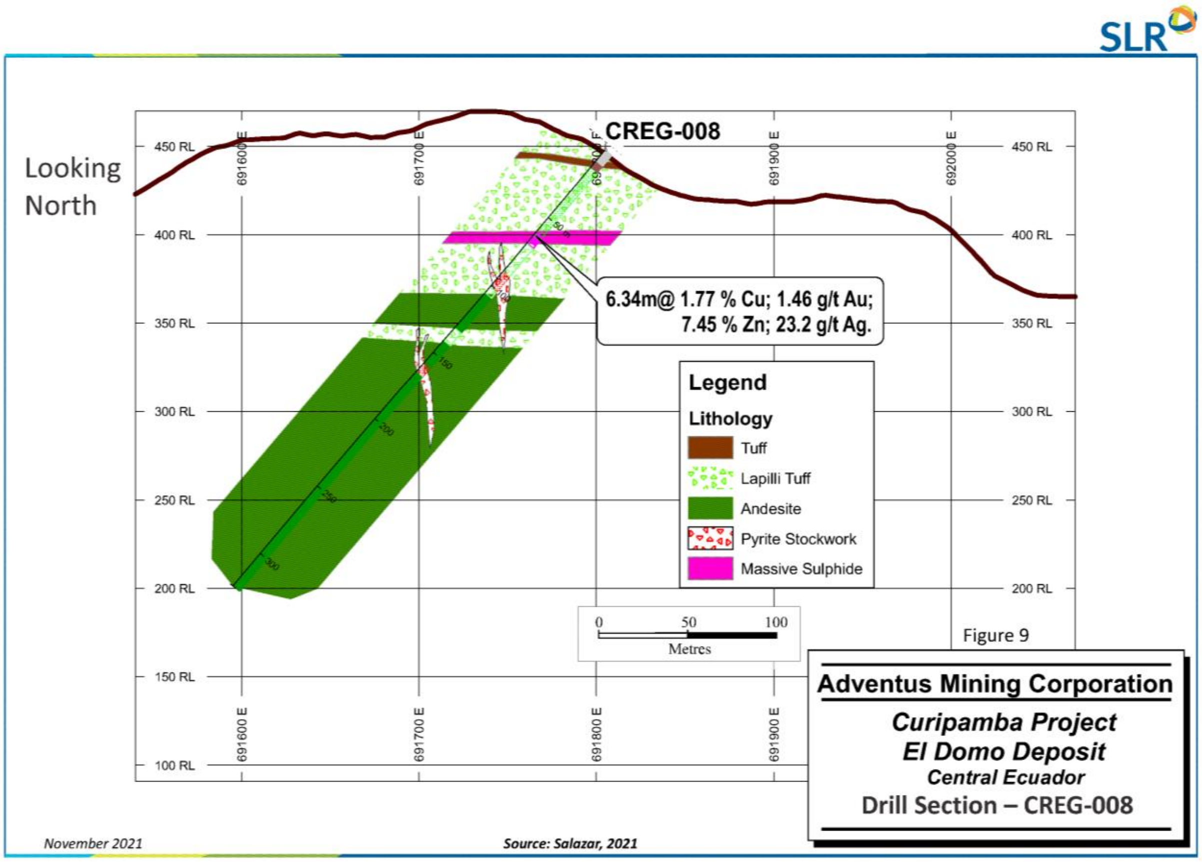
Two (2) drill holes were completed on the La Vaquera target (CREG-001 and CREG-002) and one, on the Sesmo Sur target (CREG-003). The targets were selected as possible hosts of porphyry style mineralization. Though propylitic alteration was observed in all three holes, no significant mineralization was noted. Further evaluation of the targets will be undertaken before additional drilling is completed.

Three (3) drill holes were completed on the Panecillo target (CREG-004 to CREG-006), a possible host of VMS mineralization 1.5 km east of the El Domo deposit. Results of from this target have not been publicly released.

One (1) drill hole was completed on the Selva Alegre target (CREG-007), 1.25 km southeast of the Panecillo target, a potential VMS target. Results from this drill hole have not been publicly released.

A total of 11 drill holes were completed on the Agua Santa target (CREG-008 to CREG-018). The target is located 4.5 km southwest of the El Domo deposit. The Agua Santa target was originally selected as a potential porphyry target. The first hole, CREG-008, intersected three VMS style massive sulphide intervals between 60.52 m and 66.86 m depths. The massive sulphides are located between volcaniclastic rocks, with the mineralised massive sulphide zones also appearing fragmental in nature. Assay results from this intersection returned 6.34 m of 1.77% Cu, 1.46 g/t Au, 7.45% Zn, 23.2 g/t Ag, and 0.24% Pb, including 1.40 m of 1.58% Cu, 2.67 g/t Au, 31.20% Zn, 55.0 g/t Ag, and 1.01% Pb (Figure 9.5). The volcanic stratigraphy of Agua Santa differs from that at El Domo.

Figure 9.5 – Drill Section – CREG-008



Based on mineralization observed in the initial hole at Augua Santa, follow-up drilling commenced immediately, with 10 more holes drilled in the Agua Santa region. To date only results from CREG-008 have been released publicly.

10 DRILLING

All drilling on the Project has been completed by Curimining using Salazar's subsidiary Perforaciones Andesdrill S.A.

Drilling was carried out between 2007 and 2021 in seven (7) distinct phases, during which Curimining completed 394 core boreholes for a total of 74,993 m (Table 10.1). The most recent phase of drilling, Phase VII (2020-2021) consisted of 55 core boreholes for a total of 7,094 m (Table 10.2) Curimining used HQ-sized (65.3 mm) as well as NQ-sized (47.6 mm) boring equipment. Due to the locally steep terrain, Curimining used small, man-portable drill rigs capable of reaching maximum drill depths of approximately 600 m.

The majority of drilling was focused on the El Domo deposit (Figure 10.1). Despite abundant faulting, core recovery has been good (at or exceeding 90%).

The QP did not identify any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results. Drilling established the geology of the El Domo deposit in sufficient detail to use the resulting model for mineral resource estimation purposes.

Table 10.1 – Summary of Drilling on El Domo Deposit

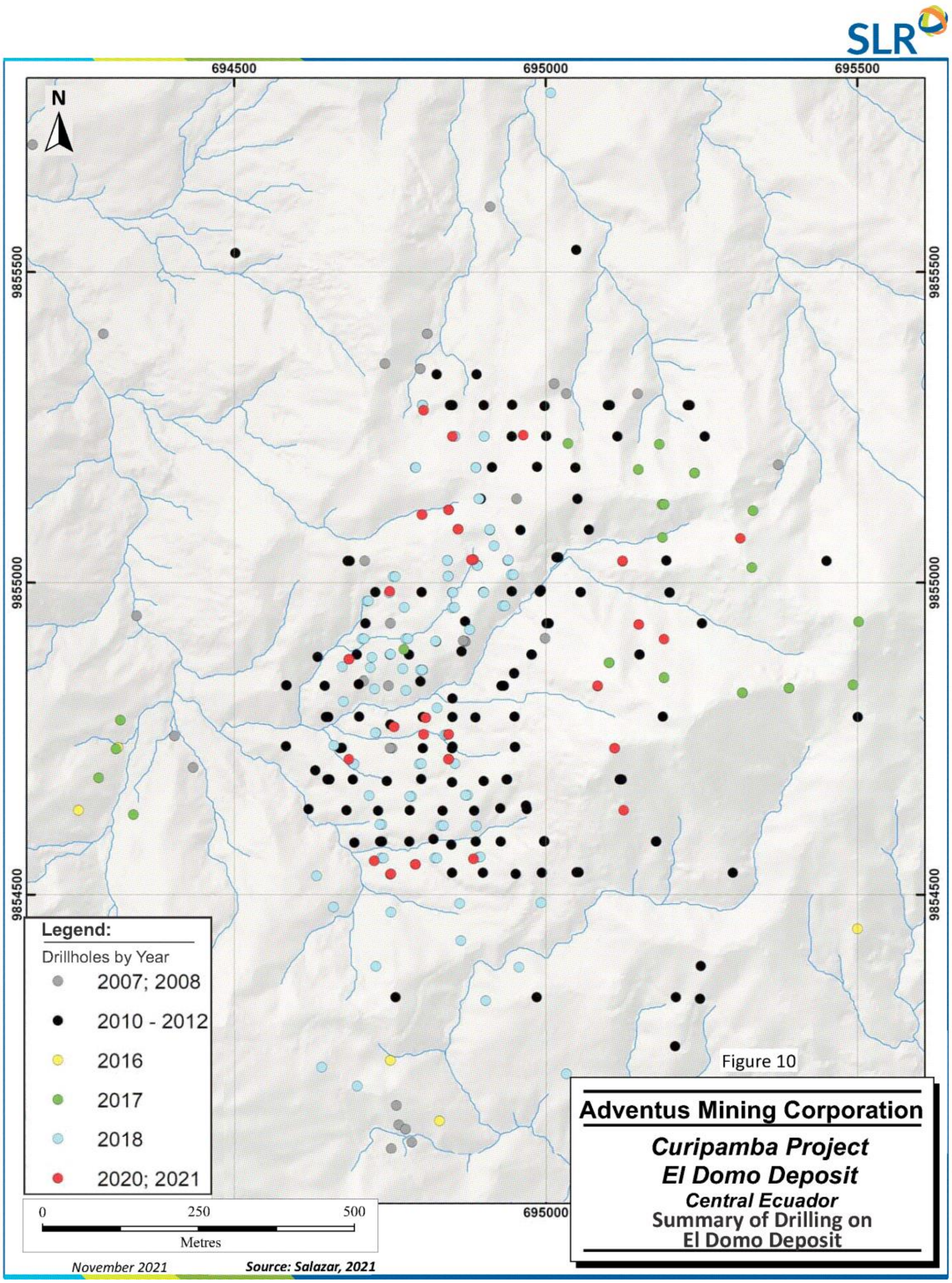
Drill Program	Year	Number of Holes	Combined Length of Holes (m)	Targets
Phase I	2007/2008	51	10,003	Sesmo Sur, El Gallo, Roble 1, El Roble, Roble Este, Cade Sur, Cade 1, Cade, Caracol 1, El Domo
Phase II	2010	20	3,242	El Domo, Sesom Sur, La Vaquera
Phase III	2010/2011	84	15,700	El Domo
Phase IV	2011/2012	51	10,249	El Domo
Phase V	2016/2017	33	9,768	El Domo
Phase VI	2018/2019	100	18,937	El Domo, Sesmo
Phase VII	2020/2021	55	7,094	El Domo, La Vaquera, Sesmo Sur, Panecillo, Selva Alegre, Agua Santa
Total		**394**	**74,993**	

Table 10.2 – Phase VII Drilling (2020-2021)

Hole ID	Depth (m)	Azimuth (˚)	Dip (˚)	Easting	Northing	Elevation	Start Date	Finish Date
CURI-338	296	270	-75	695562	9855436	1069	02/26/2020	03/10/2020
CURI-339	237	286	-81	695440	9855275	1027	03/12/2020	11/04/2020
CURI-340	149	360	-45	695134	9854922	917	11/07/2020	11/11/2020
CURI-341	117.4	360	-60	695134	9854922	917	11/11/2020	11/13/2020
CURI-342	137.1	346	-45	695041	9854913	888	11/15/2020	11/17/2020
CURI-343	142	192	-45	695094	9855082	910	11/19/2020	11/21/2020
CURI-344	84.1	360	-60	695094	9855082	910	11/23/2020	11/24/2020
CURI-345	96	142	-48	695094	9855082	910	11/26/2020	11/28/2020
CURI-346	83.4	52	-69	695094	9855082	910	11/29/2020	11/30/2020
CURI-347	91	105	-54	695094	9855082	910	12/01/2020	12/03/2020
CURI-348	135.1	276	-45	695094	9855082	910	12/04/2020	12/07/2020
CURI-349	130.2	263	-51	695094	9855122	906	12/07/2020	12/11/2020
CURI-350	85.7	180	-57.3	695094	9855122	906	12/12/2020	12/16/2020
CURI-351	77	338	-59.1	695057	9855149	894	12/13/2020	12/15/2020
CURI-352	78	306	-77	695057	9855149	894	12/17/2020	12/18/2020
CURI-353	102.4	192	-45	695094	9855122	906	12/17/2020	12/31/2020
CURI-354	81	55	-54	695057	9855149	894	12/19/2020	12/21/2020
CURI-355	86.3	36	-45	695057	9855149	894	12/22/2020	12/26/2020
CURI-356	110.6	241	-77	695006	9855134	883	12/26/2020	01/09/2021
CURI-357	105.2	229.1	-73.8	695057	9855149	894	01/03/2021	01/06/2021
CURI-358	113	255	-54	694934	9855082	869	01/07/2021	01/10/2021
CURI-359	86.5	349	-69	695006	9855134	883	01/11/2021	01/14/2021
CURI-360	83	28	-48	694934	9855242	875	01/13/2021	01/16/2021
CURI-361	173	74.9	-54	695134	9855402	946	01/14/2021	01/18/2021
CURI-362	88.9	288	-61	694975	9854919	873	01/16/2021	01/21/2021
CURI-363	95	180	-54	694934	9855242	875	01/18/2021	01/20/2021
CURI-364	130.4	108	-60	695134	9855402	946	01/20/2021	01/26/2021
CURI-364-A	121.75	108.3	-60	695131	9855402	946	01/28/2021	02/02/2021
CURI-365	69.5	151	-72	694934	9855242	875	01/21/2021	01/22/2021
CURI-366	74.5	186.1	-72.2	694975	9854919	873	01/22/2021	01/23/2021

Hole ID	Depth (m)	Azimuth (˚)	Dip (˚)	Easting	Northing	Elevation	Start Date	Finish Date
CURI-367	102.2	196	-45	695006	9855134	883	01/24/2021	01/26/2021
CURI-368	113.6	331.4	-58	695041	9854913	888	01/29/2021	01/31/2021
CURI-369	169	141.5	-48.2	695094	9855482	940	02/01/2021	02/07/2021
CURI-370	86.2	230.9	-73.8	695041	9854913	888	02/03/2021	02/04/2021
CURI-371	225.7	270	-65	695399	9855298	1016	02/05/2021	02/11/2021
CURI-372	106.3	37.9	-45.3	695041	9854913	888	02/05/2021	02/07/2021
CURI-373	197	102.8	-48.2	695094	9855482	940	02/08/2021	02/21/2021
CURI-374	120.6	296.2	-45	695006	9855134	883	02/09/2021	02/18/2021
CURI-375	250.1	270	-75	695333	9855200	1057	02/13/2021	02/17/2021
CURI-376	275.4	270	-70	695361	9855100	1087	02/18/2021	02/22/2021
CURI-377	92.3	55.4	-50.9	695054	9855122	893	02/19/2021	02/21/2021
CURI-378	113.1	180.6	-56.8	695054	9855122	893	02/22/2021	02/24/2021
CURI-379	154	167.3	-48.1	695094	9855482	940	02/23/2021	02/27/2021
CURI-380	250.1	270	-70	695375	9855000	1030	02/25/2021	03/02/2021
CURI-381	154.4	196.6	-56.9	695214	9855602	948	02/28/2021	03/03/2021
CURI-382	156.5	64.2	-48.2	695094	9855482	940	03/01/2021	03/08/2021
CURI-383	210.4	270	-65	695373	9855400	995	03/05/2021	03/16/2021
CURI-384	110.6	135.5	-54.2	695054	9855642	922	03/06/2021	03/09/2021
CURI-385	128	44.6	-54.1	695094	9855482	940	03/09/2021	03/12/2021
CURI-386	90.2	230	-65	695100	9855600	943	03/10/2021	03/21/2021
CURI-387	112.3	26	-49	695051	9855474	928	03/13/2021	03/16/2021
CURI-388	143	165	-50.6	695051	9855474	928	03/17/2021	03/20/2021
CURI-389	65	338	-70	694999	9855351	908	03/21/2021	03/23/2021
CURI-390	137.7	170	-79	695109	9855451	953	03/25/2021	03/29/2021
CURI-391	71	165	-65	695001	9854898	883	03/30/2021	03/31/2021
Total	**7,094.75**							

Figure 10.1 – Summary of Drilling on El Domo Deposit



10.1 Drilling Procedures

Boreholes were spotted by Curimining personnel initially using a global positioning system (GPS) receiver with real-time kinematic (RTK) capabilities and, since approximately 2011, with a GPS total station, prior to the drill rig being moved into position. All drill collars to date have been resurveyed using a GPS total station. Fore- and backsites were placed to aid in setting the drill to the correct azimuth. The dip was set and confirmed by Curimining using a clinometer. A drone based airborne light detection and ranging (LiDAR) survey was completed over the El Domo area in 2021. As part of the verification of the survey data, drill collar elevations were compared to the resulting topographic model. Where issues were noted, collars were verified in the field by total station.

To limit surface disturbance, a number of boreholes were drilled from the same location. Final borehole positions were surveyed using the same instrumentation used for initial hole spotting.

Drilling was conducted using a triple tube core barrel, to increase core recoveries, and Reflex oriented core drill tools. A Curimining geologist was present at the drill rig to end each hole. Once the hole was completed, casings were pulled, and the location was identified with a cement monument. Drill sites were rehabilitated.

Downhole surveys were completed using a Reflex tool at the end of each hole. For the 2020/2021 drilling program downhole surveys were completed using Stockholm Precision Tools' MagCruiser tool. No additional surveys were conducted along the borehole trace.

Core was placed into wooden core boxes at the drill and were "quick logged" prior to transport and storage. Core transport from the drill to Curimining's field office and core logging facility in Las Naves was by Curimining personnel. At the core shed, Curimining technicians determined rock quality designation (RQD) and rock mass rating (RMR), confirmed run lengths, performed density measurements and point load tests (PLT), and prepared the core for logging. Core boxes were labelled with borehole number, box number, and core "from-to" data. Curimining uses a fixed, standardized set-up to photograph two core boxes with uncut core at a time, hence, creating a full photographic record of all core prior to sampling.

Detailed logging of lithology, alteration, mineralization, oxidization, and structural (both oriented point and unoriented interval) data was completed by well trained geologists of the Exploration Department in Ecuador.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Prior Drilling Campaigns

Prior to the 2020/2021 drilling campaign, Curimining used two (2) primary analytical laboratories throughout the five core drilling campaigns: BSI Inspectorate (Inspectorate) and ALS Chemex Laboratories (ALS Chemex). Both laboratories are certified to ISO 9001:2000 and independent of Curimining and Adventus.

Sample batches for assay were initially sent with a chain of custody by pickup truck by a Salazar geologist from Ventanas to Quito for sample preparation at either Inspectorate or ALS Chemex. After preparation, sample pulps were shipped by TNT courier to Inspectorate or ALS Chemex laboratories in Lima, Peru for analysis.

At Inspectorate, the precious metal grades were determined using fire assay and atomic absorption (AA) spectroscopy (procedure Au-25). Base metals were determined by inductively coupled plasma (ICP) spectroscopy. At ALS Chemex, precious metals were determined by fire assay and AA spectroscopy and base metals, by ICP.

11.2 2020/2021 Drilling Campaign

For the 2020/2021 drilling programs, Curimining used one primary preparation laboratory Bureau Veritas in Quito, Ecuador (BV Quito), and two (2) primary analytical laboratories: Bureau Veritas in Lima, Peru (BV Lima), and in Vancouver, Canada (BV Vancouver). Both laboratories are certified to ISO 9001:2000 and independent of Curimining and Adventus. SLR was unable to determine whether either of these laboratories is certified to ISO 17025 for the analytical procedures used by Curimining.

The core was photographed, logged, marked for sampling, sawn, bagged, and sealed in rice bags for shipment by Curimining personnel at their logging facility at Las Naves. The logging facility consists of a walled and locked compound in which core is stored in a locked warehouse. Data are kept secure in a locked building within the compound with access only by selected Curimining staff.

Logging was performed by qualified Curimining personnel who also determined sample selection. Core was cut lengthwise on core saws; one half of the core remained in the core box for reference, while the other half was prepared for shipment to the analytical laboratory. Sample tags were placed in the core box, in the sample bag, and the last part of the sample tag remained as reference. Individual bagged samples were collated into sample shipments with sample numbers written on the outside of shipping bags.

Sample batches for assay were initially sent with a chain of custody by pickup truck by a Curimining geologist from Las Naves to Quito for sample preparation at BV Quito. After preparation, sample pulps were shipped by TNT courier to BV Lima or BV Vancouver for analysis.

Gold grades were analyzed using FA430 (lead collection fire assay fusion, with AA finish), and the over-limit values were analyzed using FA530 (lead collection fire assay, with a gravimetric finish). Silver and base metals were analyzed using AR301 (BV Lima) and AQ300 (BV Vancouver) which employed aqua regia digestion with ICP atomic emission spectroscopy analysis. The over-limit values for silver and base metals were analyzed using AR402 (BV Lima) and AR404 (BV Vancouver), which utilized aqua regia digestion with AA finish. Additional over-limit values for silver were analyzed using FA530 (lead collection fire assay, with a gravimetric finish), and for zinc were analyzed using GC816 (volumetric titration).

11.2.1 Density Data

Specific gravity data was collected on the in-fill drilling for every sample sent to the laboratory and the lithology units above the El Domo deposit. Specific gravity measurements were performed at BV Vancouver utilizing the SPG04 method: density by gas pycnometer. Measurements were performed on 30 g to 40 g pulp splits. The specific gravity readings were collected at core shack at regular intervals downhole as well. The analysis was completed by first taking a dry weight, wet weight, then weight submerged in water. Preparation specific gravity readings is dependent on the competence of the rock. Competent rock was soaked in water for four (4) hours before wet analysis, while highly fractured/friable rock was either wrapped in aluminum foil or weighed in water with no preparation. In these cases, wet weight was skipped. Results were visually inspected, and outlier readings were manually marked as failed. Preference was given to results that had completed the four (4) hour water bath.

11.3 Quality Assurance and Quality Control Program

Quality control measures are typically set in place to ensure the reliability and trustworthiness of exploration data. Measures include written field procedures and independent verifications of aspects such as drilling, surveying, sampling and assaying, data management, and database integrity. Appropriate documentation of quality control measures and regular analysis of quality control data are important as a safeguard for project data and form the basis for the quality assurance/quality control (QA/QC) program implemented during exploration.

Analytical control measures typically involve internal and external laboratory control measures implemented to monitor the precision and accuracy of the sampling, preparation, and assaying process. These measures are also important to prevent sample mix-up and to monitor the voluntary or inadvertent contamination of samples.

Assaying protocols typically involve regularly duplicating and replicating assays and inserting quality control samples to monitor the reliability of assaying results throughout the sampling and assaying process. Check assaying is normally performed as an additional test of the reliability of assaying results. It generally involves re-assaying a set number of sample rejects and pulps at a secondary umpire laboratory.

11.3.1 Prior Drilling Campaigns

Curimining used a range of commercial, certified as well as prepared and non-certified reference materials and blanks throughout the drilling programs. Table 11.1 to Table 11.6 summarise the types and frequency of materials used by Curimining by drill program.

Table 11.1 – Summary of Blank Samples

Drill Program	Notes	Blank Material				Total
		Blanks	BK	BL-115	BL-112	
I	No insertion between CURI 10 and CURI 38	34				34
II	No insertions	-				-
III	No insertions from CURI 72 to CURI 90 Blank BK since CURI 112		30		25	55
IV	No insertions from CURI 156 to CURI 162 Variable insertion rate		16	8		24
V	Insertion rate of 1 in 30		10		10	20
VI	No blank BK from CURI 296		30	71	59	160
	Total	**34**	**86**	**79**	**94**	**293**

Table 11.2 – Summary of Duplicates Samples

Drill Program	Notes	Total
I	No fixed insertion criteria or ratio	322
II	Mostly high-grade Au-Cu-Zn samples and every ten samples	283
III	Mostly high-grade Au-Cu-Zn samples and every ten samples, up to CURI 134	770
IV	Some samples submitted	93
V	Several samples per hole	109
VI	Several samples per hole	848
	Total	**2,414**

Table 11.3 – Summary of Standard Reference Samples

Standard Reference Certificates	Drill Program						Total
	I	II	III	IV	V	VI	
Standard Prepared Without Certification							
ESTANDAR	1						1
SR	55	23	22				100
GO1-166	1						1
GO2-109		1					1
GO2-25	3						3
MS-2(RNG1)	1						1
Standard Certificates							
PB-118	131		1				132
PB-130		19	83	21			123
PB-140			22	21		1	44
PM-1110	50						50
PM-1111	90						90
PM-1118		25	75				100
PM-1123			10		10	52	72
CU-121				8			8
CU-130	5						5
CU-145			17	21			38
CU-152			33	21		46	100
CU-155		19	86	21	11	36	173
CU-160				8	10	73	91
CU-163			18	21		5	44
CU-174				8		72	80
CU-175			15	21	12	42	90
CU-181				8			8
CU-183				8			8
Total	**337**	**87**	**383**	**187**	**43**	**328**	**1,365**

11.3.2 2020/2021 Drilling Campaign

In the 2020/2021 drilling program, Curimining continued using certified reference materials, blanks, and duplicates. Table 11.4 through Table 11.6 summarise the types and frequency of materials used by Curimining during the 2020/2021 drill program.

Table 11.4 – Summary of Blank Samples

Hole ID	Blank Material				Total
	BL-115	BL-126	BL-127	BVIP-074	
CURI-338 – CURI-345	1	5	–	3	9
CURI-346 – CURI-391	–	–	39	1	40
Total	**1**	**5**	**39**	**4**	**49**

Table 11.5 – Summary of Duplicate Samples

Hole ID	Total
CURI-338 – CURI-345	7
CURI-346 – CURI-391	41
Total	**48**

Table 11.6 – Summary of Standard Reference Samples

Hole ID	Standard Certificates				Total
	CU-121	CU-163	CU-188	CU-189	
CURI-338 – CURI-345	4	5	7	–	16
CURI-346 – CURI-391	–	30	41	11	82
Total	**4**	**35**	**48**	**11**	**98**

11.4 SLR Comments

In the QP's opinion, the sample preparation, analysis, and security procedures at the Project are adequate for use in Mineral Resource estimation. The QA/QC program as designed and implemented by Curimining is adequate and the assay results within the database are suitable for use in a Mineral Resource estimate.

The QP makes the following recommendations to improve and streamline procedures:

- Use only one standard blank material that should be available for the foreseeable future.
- Use full-length samples for the collection of density data at the laboratory in order to make these data more representative.

12 DATA VERIFICATION

12.1 Data Verification by Curimining and Adventus

The exploration work on the Project is conducted by Curimining personnel and qualified subcontractors, and field work is supervised by qualified geologists. Curimining and Adventus have implemented a series of routine verifications to ensure the collection of reliable exploration data.

Exploration data are recorded digitally to minimize data entry errors. Core logging, surveying, and sampling were monitored by qualified geologists from both Curimining and Adventus and data was verified routinely for consistency. Drilling data is entered into MX Deposit, a cloud based logging software. Surface geochemical data was captured in Microsoft Excel format.

The digital database is maintained by a senior Adventus geologist who coordinates data import and validation. Analytical results were delivered electronically by the primary and umpire laboratories to Curimining and Adventus and were examined for consistency and completeness. A comprehensive analytical QA/QC program was implemented by Adventus and Curimining. Analytical results for control samples were reviewed and compared to set tolerances. Failures were scrutinized further and affected sample batches were submitted for re-analysis when required. Back-ups of the database were performed automatically by the software provider.

12.2 Data Verification by SLR QP

12.2.1 Site Visit

Pursuant to NI 43-101, Ms. Dorota El-Rassi, P. Eng., of SLR completed two (2) site visits to the Curipamba Project. The first site visit occurred from January 8 to 10, 2019, and the second from July 26 to 28, 2021. During the site visits, Ms. El-Rassi reviewed the logging environment in the core shack and procedures for data collection and sampling, including SG, inspected core samples and outcrops in the El Domo, Sesmo Sur, and Sesmo areas, interviewed Curimining personnel, and gathered other information for the completion of this report. Additionally, Ms. El-Rassi examined drill hole core relevant to Mineral Resource estimation, including visually checking stratigraphy against interpreted drilling sections. Curimining provided full access during all parts of the site visits. Ms. El-Rassi was accompanied by Mr. Jason Dunning, Vice President Exploration for Adventus, during the 2019 site visit, and Ms. Andreina Zarabia, Technical Coordinator for Curimining, during the 2021 site visit

12.2.2 Assay Certificates Cross-Check

The QP cross-referenced assays certificates with value stored in the database provided and found no significant errors. The QP has performed assay certificate checks on significant portions of previous programs and 100% of the 2020 and 2021 assay database.

12.3 Verification of Analytical Quality Control Data

Quality assurance consists of evidence to demonstrate that assay results have both precision and accuracy within generally accepted limits for the sampling and analytical method(s) used and therefore provide confidence in the support data used in a resource estimate. Quality control consists of procedures used to ensure that an adequate level of quality is maintained in the process of collecting, preparing, and assaying the exploration drilling samples. In general, QA/QC programs are designed to prevent or detect contamination and allow analytical precision, repeatability, and accuracy to be quantified.

12.3.1 Pre-2020 QA/QC Programs

The QP has performed analysis on all QA/QC programs prior to the 2021 to 2021 drilling programs. The analysis involved the review of the blank, standards and duplicate analyses inserted into the sample stream. The QP assessed the appropriateness of the types of QA/QC material used and insertion rates and used control charts and statistical analysis to determine the accuracy and precision of the assay results.

The QP is of the opinion that the results of the QA/QC programs pre-2020 are suitable to support the estimation of Mineral Resources and Mineral Reserves.

12.3.2 Blanks (2020 and 2021)

Curimining utilized three different types of blank material: two certified materials were from WCM Minerals (WCM), in Burnaby, British Columbia, and a custom material provided by BV. The custom blank from BV was required due to shipping delays of replacement WCM blanks after exhausting the stock of the previously used WCM blank (no longer available for purchase). A failure is defined as an assay result greater than three times the method detection limit (DT). Only two failures were reported but were considered to be non-material due to the low grades. Nevertheless, the QP recommended investigating if the reason for failures is due to sample preparation in the laboratory or if a slight sample contamination issue existed. Table 12.1 is a summary of failure rates for the blank reference materials. Figure 12.1 to Figure 12.4 are graphical representations of the performances of all three blank materials for gold, silver, copper, zinc, and lead.

Table 12.1 – Summary of Blank Failures

Description	Gold	Silver	Copper	Lead	Zinc	Total
No. Assays	49	49	49	49	49	245
No. values outside 3 DT	1	1	0	0	0	2
Percent outside 3 DT	2%	2%	0%	0%	0%	1%

Figure 12.1 – Performance of Blank Reference Material BL-115


BL-115 : Au (ppm) FA430
1
Passed (1)
Max value = 0.02
Au (ppm)
12/16/2020
Completed Date
BL-115 : Ag (ppm) AR301
1
Passed (1)
Max value = 0.3
Ag (ppm)
12/16/2020
Completed Date
BL-115 : Cu (ppm) AR301
1
Passed (1)
Max value = 56
Cu (ppm)
12/16/2020
Completed Date
BL-115 : Pb (ppm) AR301
1
Passed (1)
Max value = 6
Pb (ppm)
12/16/2020
Completed Date
BL-115 : Zn (ppm) AR301
1
Passed (1)
Max value = 54
Zn (ppm)
12/16/2020
Completed Date

Figure 12.2 – Performance of Blank Reference Material BL-126

Figure 12.3 – Performance of Blank Reference Material BL-127

Figure 12.4 – Performance of Blank Reference Material BVIP-074

12.3.3 STANDARDS (2020 AND 2021)

Curimining's analytical quality control program comprised the insertion of standard reference material (SRM) into the normal sample stream. Insertion rates were considered to be in line with industry practice.

Curimining utilized four different SRMs to cover a range of expected mineralization grades of gold, silver, and copper. SRMs were sourced from WCM.

The performance of SRMs was generally acceptable; one standard (Cu-163) had a more than 10% failure rate.

Curimining considered the "warning" range to be when any results were greater than ±2SD from the certified value and these were sent for re-analysis only if other standards in the same or adjacent certificates were in the same "warning" range. Any results greater than ±3SD were immediately returned, with five samples before and after, for re-analysis. If re-analysis confirmed that the issue extended to surrounding samples, further samples would be selected for re-analysis.

Gold samples had only one failure (1%) using CU-188 SRM. The BV Lima laboratory, Peru was notified of possible bias in December 2020 and the results improved in following months. No silver failures were observed; CU-121, however, showed minor positive bias and CU-188 showed minor negative bias in the BV Lima laboratory (Figure 17). Both laboratories had one failure analyzing higher copper grades, CU-121 in BV Lima and CU-189 in BV Vancouver. Curimining re-ran all copper values greater than 5,000 ppm from BV Vancouver with the over-limit method AR404, which performed well, and no failure was noted. The over-limit method AR402 in BV Lima produced two failures, which prompted Curimining to move to BV Vancouver, BC, Canada in March 2021.

Additional analyses that returned values outside of 3SD from expected values were few, and the QP considered them to be non-material.

Table 12.2 summarises failure rates of SRMs. Figure 12.5 to Figure 12.8 are graphical representations of each SRM used during the 2020/2021 drilling campaign.

Table 12.2 – Summary of Standard Failures

Description	Gold	Silver	Copper	Lead	Zinc	Total
No. Assays	98	98	106	-	-	302
No. Values outside 3SD	1	0	4	-	-	5
Percent outside 3SD	1%	0%	4%	-	-	2%

Figure 12.5 – Performance of Standard Reference Material CU-121

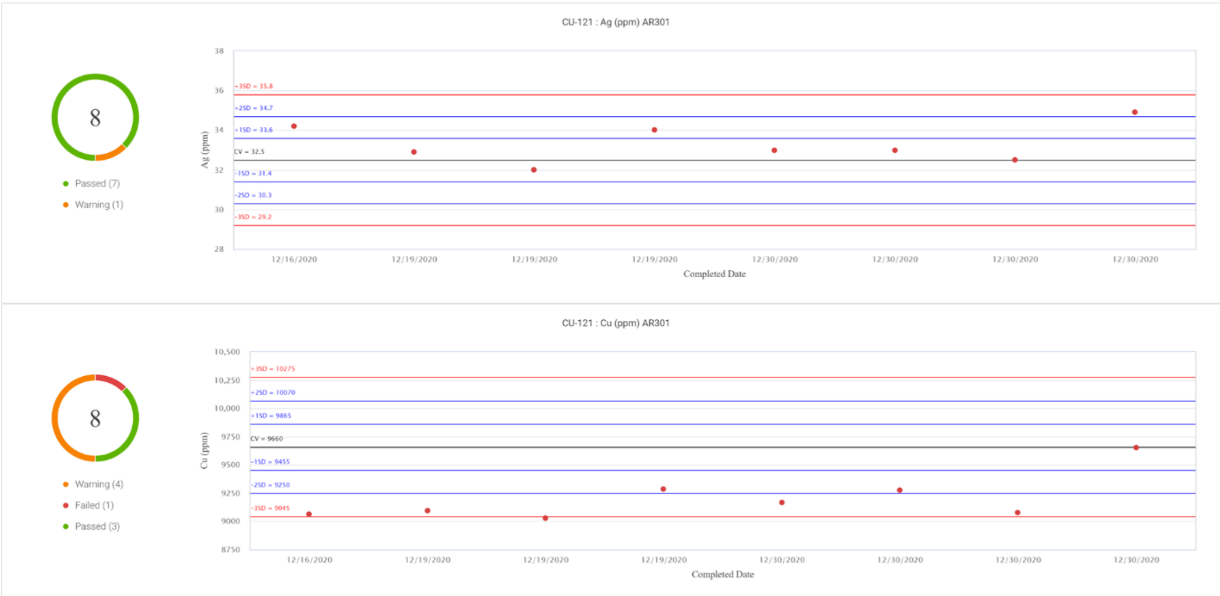


Figure 12.6 – Performance of Standard Reference Material CU-163

Figure 12.7 – Performance of Standard Reference Material CU-188

Figure 12.8 – Performance of Standard Reference Material CU-189

12.3.4 UMPIRE SAMPLES (2020 AND 2021)

Umpire samples, which consisted of pulps prepared by BV Quito and analyzed at a laboratory different than the primary laboratory, performed well. A total of 274 pulp samples (21% of the total program) were submitted to ALS Vancouver Geochemistry Analytical Laboratory (ALS) for check analysis. The QP noted that analyses of gold at ALS returned slightly lower values than those obtained from analyses at BV laboratories, however, the bias was slight and the QP considered the data to be within industry standards.

Figure 12.9 to Figure 12.13 show the original assays for gold and silver plotted against the umpire laboratory assays.

Figure 12.9 – Original versus Umpire Assay Results (Au)

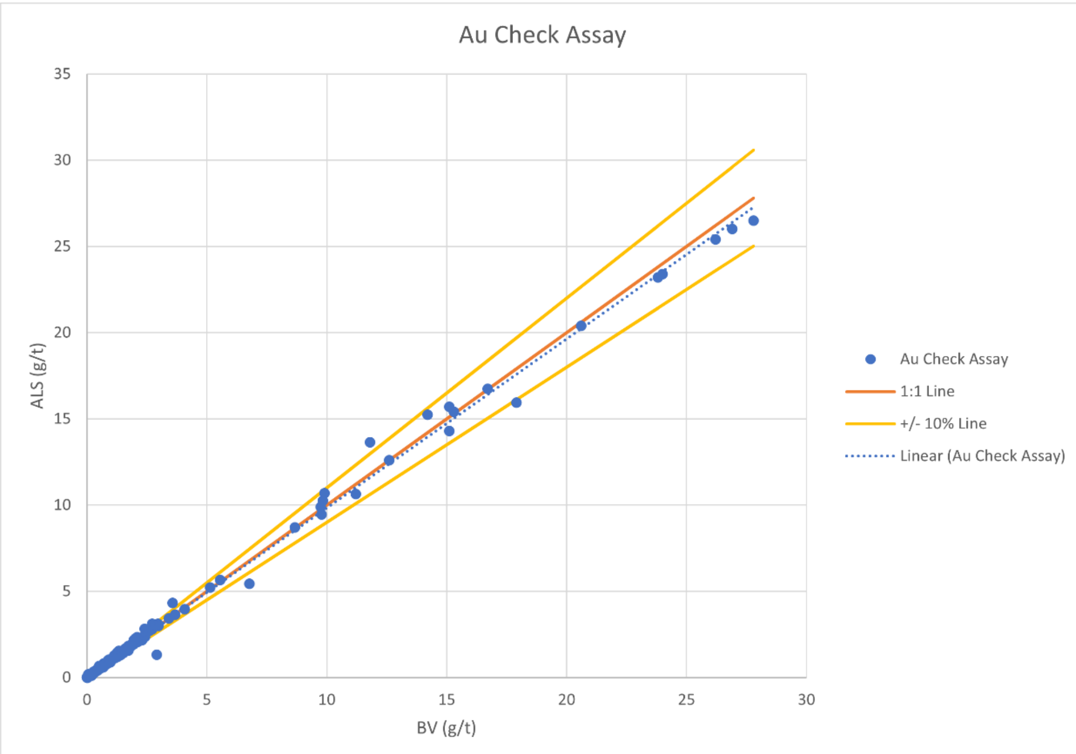


Figure 12.10 – Original versus Umpire Assay Results (Ag)

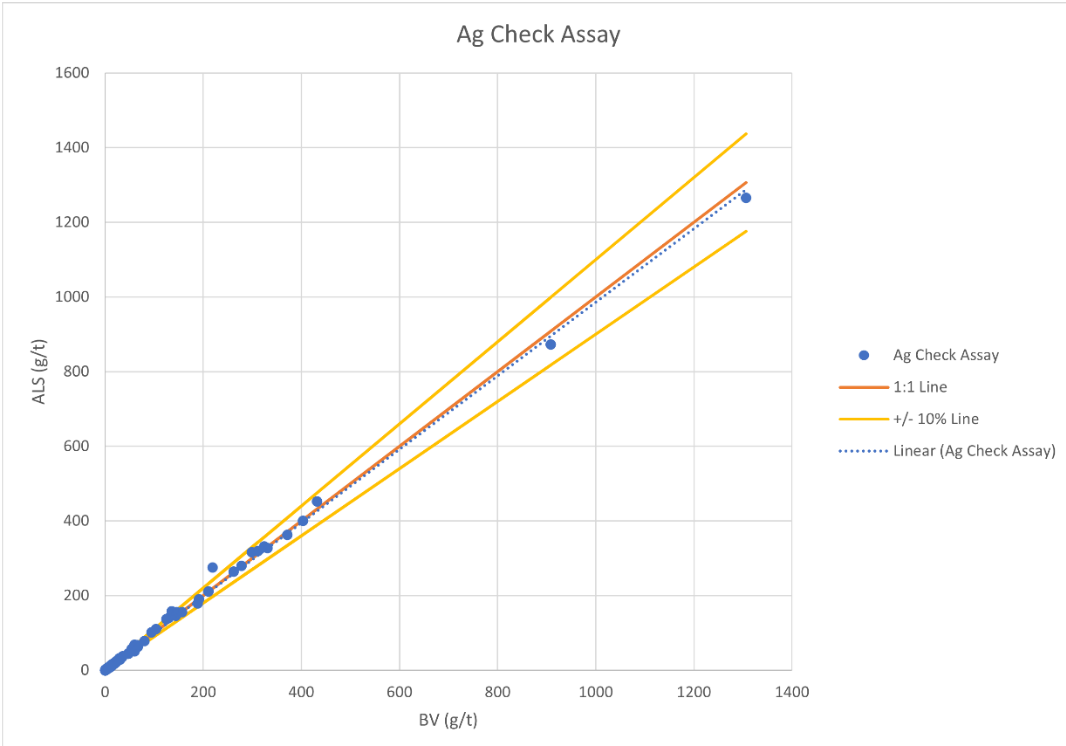


Figure 12.11 – Original versus Umpire Assay Results (Cu)

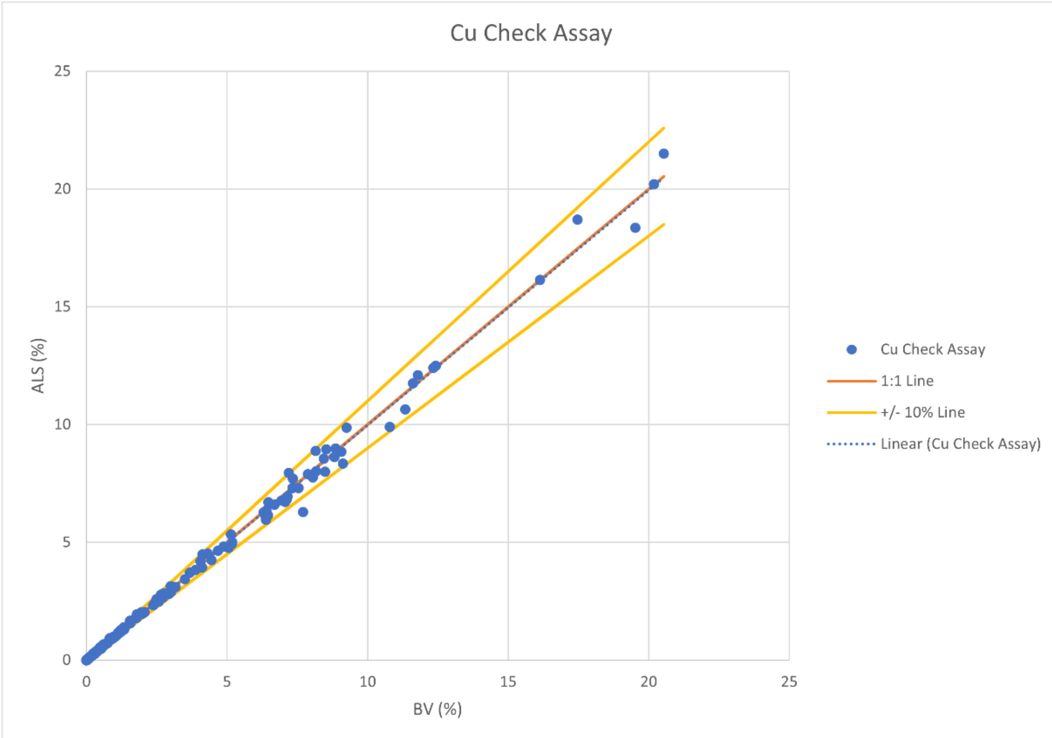


Figure 12.12 – Original versus Umpire Assay Results (Pb)

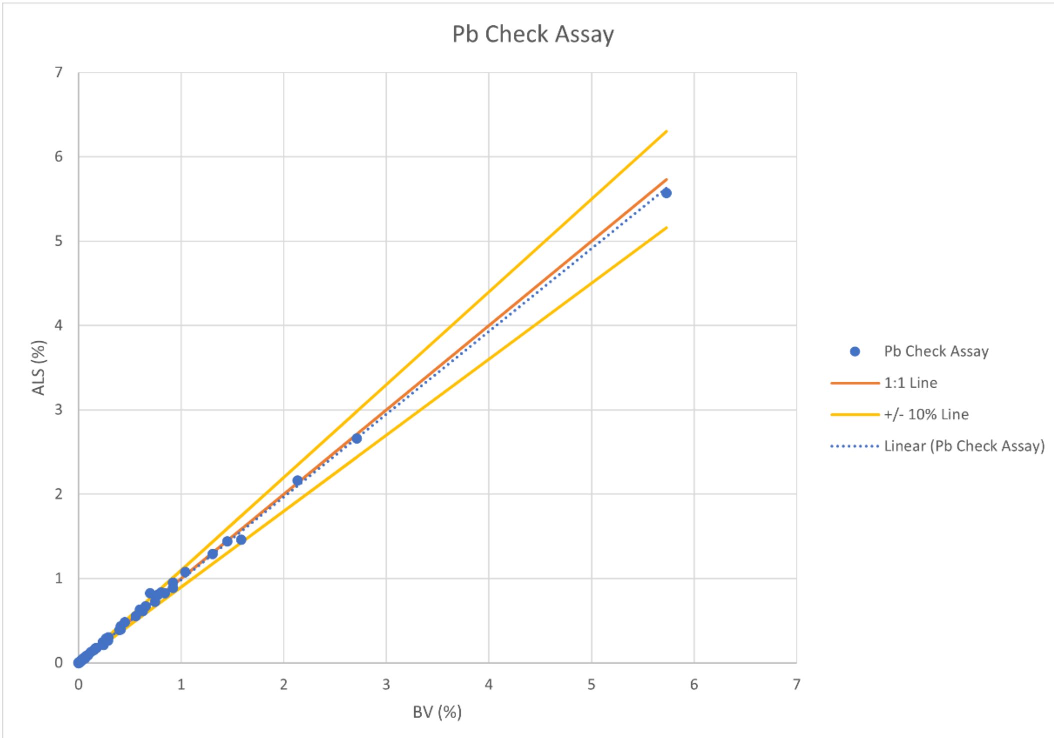
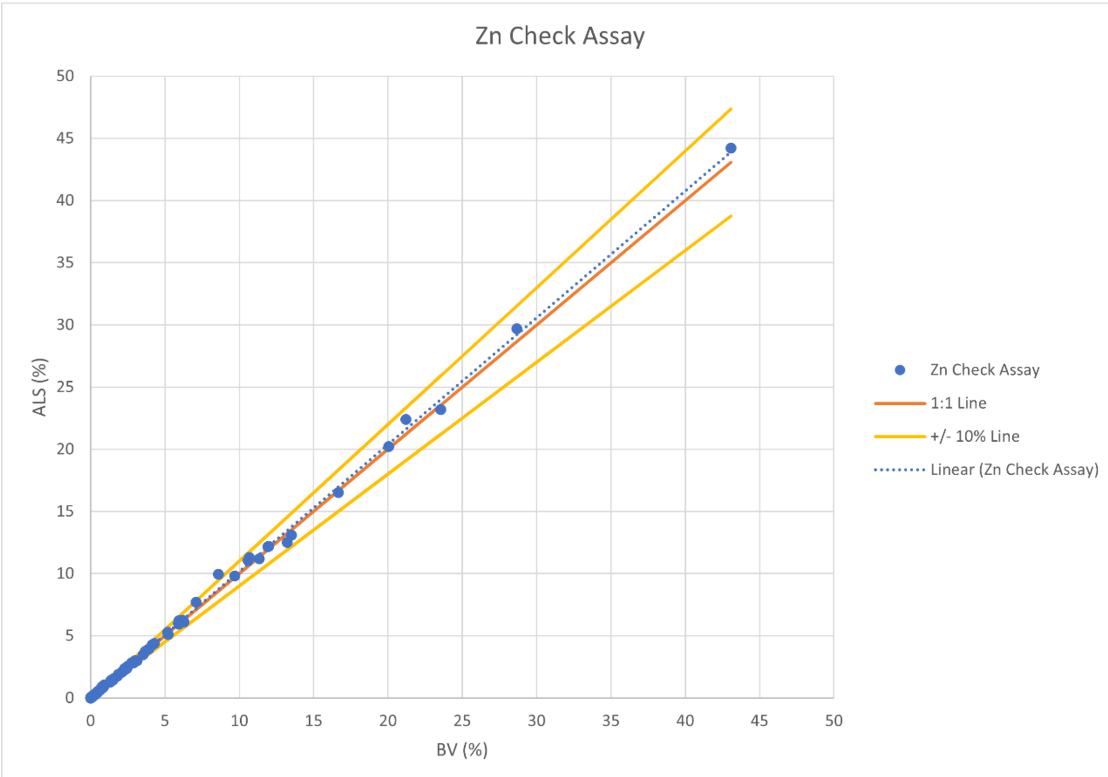


Figure 12.13 Original versus Umpire Assay Results (Zn)



12.3.5 DUPLICATES (2020 AND 2021)

The purpose of these duplicates is to test the quality of preparation and repeatability of samples along the analysis pathway. Approximately 4% of samples were submitted for a duplicate analysis. Figure 12.14 to Figure 12.18 are graphical representations of the pulp duplicates.

The QP recommends that more reject and field duplicates be implemented for future drilling campaigns. The QP considers the pulp duplicate results to be satisfactory and to support the reliability of an estimated mineral resource.

Figure 12.14 – Pulp Duplicates Results (Au)

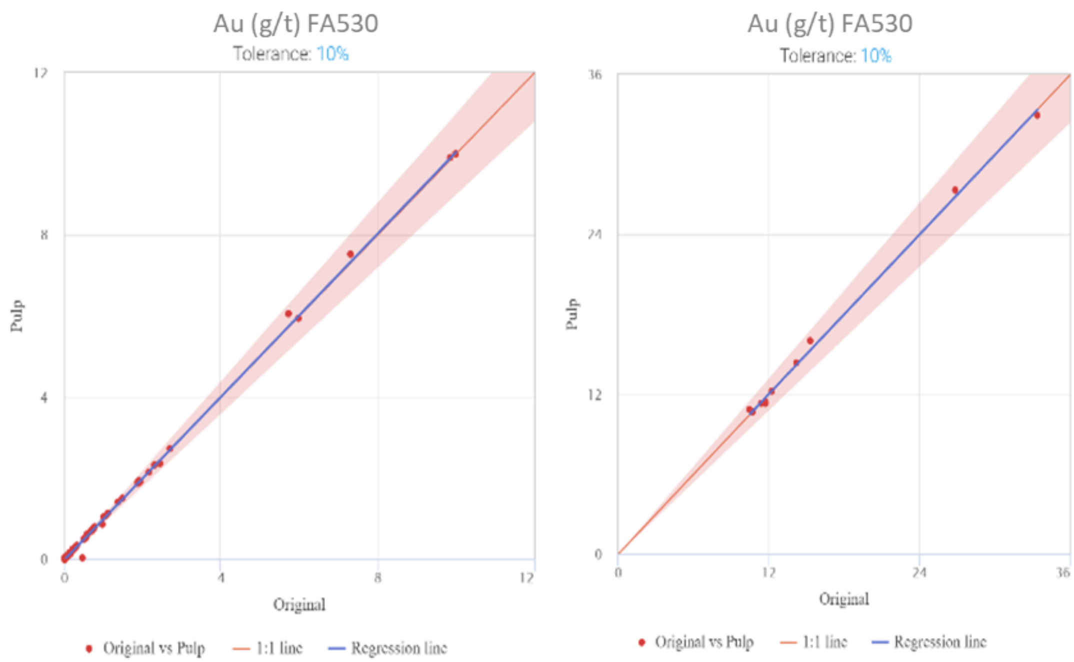


Figure 12.15 – Pulp Duplicates Results (Ag)

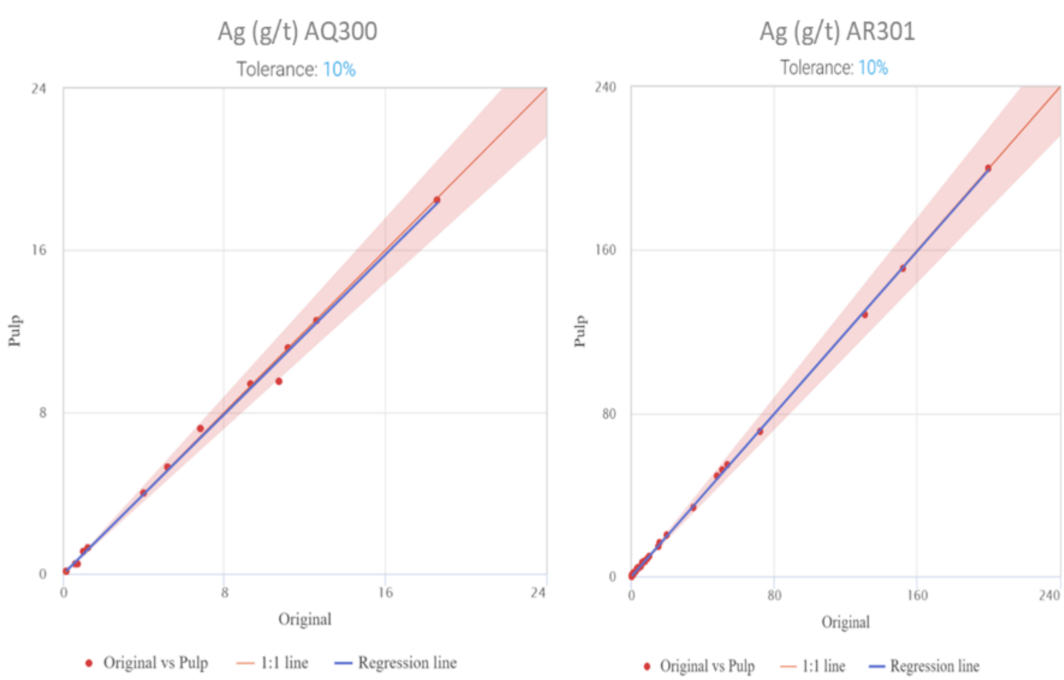


Figure 12.16 – Pulp Duplicates Results (Cu)

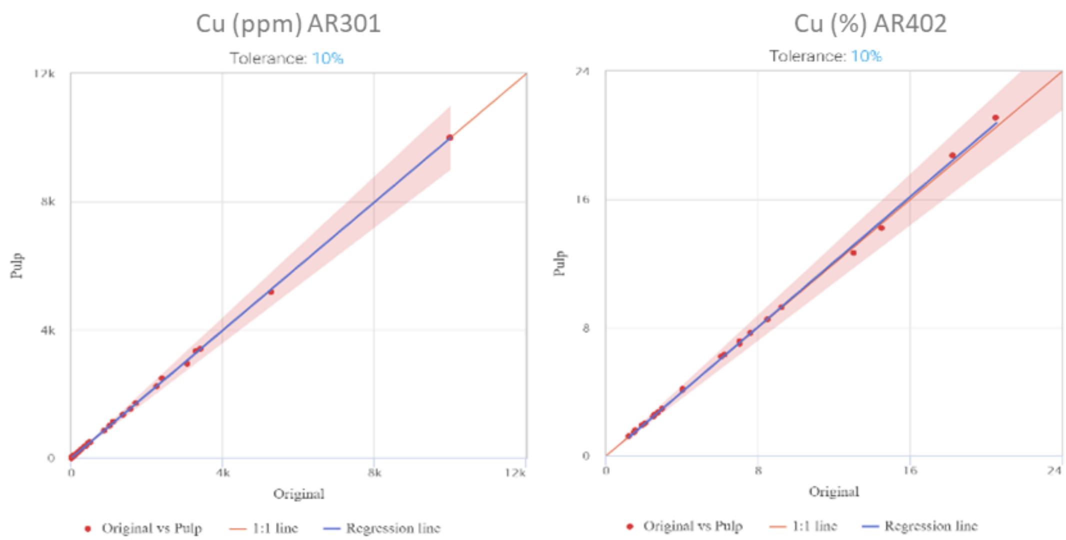


Figure 12.17 – Pulp Duplicates Results (Zn)

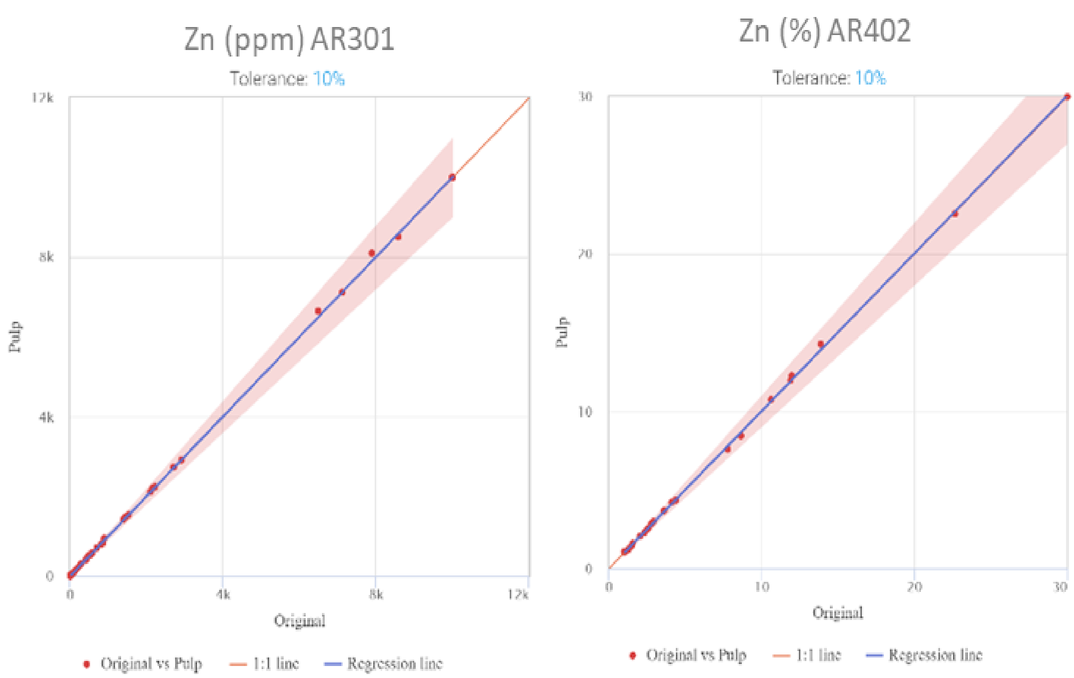
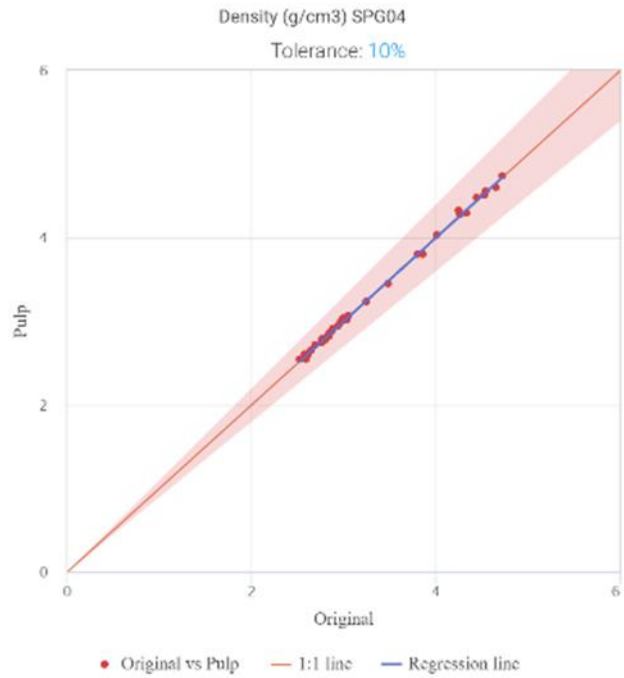


Figure 12.18 – Pulp Duplicates Results Density



The QP reviewed the data and their analysis and considers the results to be reasonable, with no significant issues identified. Hence, the QP is of the opinion that all exploration data from the 2020/2021 drilling are sufficiently reliable for mineral resource estimation and in line with industry best practice standards as defined in the CIM Exploration Best Practice Guidelines (CIM, 2018).

Based on data verification steps and a review of all of the QA/QC programs, the QP is of the opinion that the entire drill hole database is suitable for the estimation of Mineral Resources and Mineral Reserves per CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (CIM, 2019).

13 MINERAL PROCESSING AND METALLURGICAL TESTING

The test work conducted through the Curipamba (El Domo) project history was focussed on sample ore physical characterisation, mineralogy, metal grades, mineral liberation analysis, and the development of the flotation flowsheet for saleable concentrate production and assessment of the gold and silver recovery through the gravity, leach and Sulphidisation, Acidification, Recycling and Thickening (SART) processes.

A detailed summary of historical testing up until 2019 was completed by RPA (now SLR) as a part of the PEA study and is included in this Section of the Report. Work completed by Base Metallurgical Laboratories Ltd. (BML) in 2019-2020 (post PEA) and results produced during 2020 and 2021 testing campaigns are summarised by DRA. Interpretation of the tests results produced in 2019-2021 was completed by DRA.

13.1 Historical Testwork

Historically, El Domo ore samples have been tested by several laboratories with tests commencing in 2009. Testwork results have been documented in reports provided by:

- Inspectorate Services Perú S.A.C. (2009);
- G&T Metallurgical Services (2010);
- Surface Science Western (2011);
- SGS Mineral Services (2013);
- Laboratorio BISA (2013); and
- Base Metallurgical Laboratories Ltd. (BML) (2018- early 2019).

13.1.1 Inspectorate Services Peru S.A.C.- 2009

The earliest metallurgical test work on samples from the Project was completed in 2009 by Inspectorate Services Perú S.A.C. (Inspectorate Peru). In addition, mineralogical studies were completed by GeoConsult in Toronto, Canada. Samples used for the metallurgical study were sourced from sample rejects that were not adequately stored to prevent oxidation.

While the results from the Inspectorate Peru work were questionable, the metallurgical test work did suggest that recovery by differential flotation was likely the most effective process for recovering the valuable minerals, with production first of a bulk lead-copper rougher

13.1.2 G&T Metallurgical Services - 2010

In 2010, Salazar commissioned RPA to manage and supervise a metallurgical test program at G&T Metallurgical Services Ltd. (G&T) in Kamloops, British Columbia, Canada. Sample material from 12 core drill holes were provided to G&T to produce three (3) master composite samples. The final master samples consisted of samples from seven (7) core drill holes and weighed 90 kg. The samples, comprising the master samples, were selected to reasonably represent the deposit with respect to assays, mineralogy, lithology, and spatial representation.

The test work at G&T included:

- A mineralogical analysis of the master composite sample;
- A Bond ball mill work index (BWi) test;
- A series of eight (8) rougher flotation tests, and
- Two (2) cleaner flotation tests.

Results indicated that mineralised material is of medium hardness with a BWi of 11.6 kWh/tonne. The fragmentation characteristics of the composite, measured at 91 micron D_{80}, determined that copper sulphides and sphalerite were 37% and 40% liberated, respectively. The majority of the unliberated copper occurred either as binaries with pyrite, or in structurally complex multiphase particles. The majority of the unliberated sphalerite was locked in complex multiphase structures.

The limited test work program was unable to produce copper and zinc concentrates of saleable quality, primarily due to the presence of chalcocite and covellite in the material, which resulted in high levels of sphalerite reporting to the copper flotation concentrate. Significant interlocking of copper sulphides with sphalerite further negatively influenced selectivity.

Mineralogical analysis using Quantitative Evaluation of Materials by scanning electron microscopy (QEMSCAN) on the copper and zinc concentrate from a select cleaner test revealed that impurities of extremely fine copper sulphide particles were present within some of the sphalerite particles. This occurrence was confirmed by Surface Science Western, at the University of Western Ontario, by conducting scanning electron microscopy with energy dispersive X-ray spectroscopy (SEM/EDX) analysis on a number of sphalerite particles. The sphalerite particles were found to contain very fine copper sulphide particles embedded within them. This analysis further determined that not all sphalerite particles in the samples occurred in this fashion. The sample also contained sphalerite particles which were clean or contained no copper.

13.1.3 SGS Mineral Services - 2013

In 2013, Salazar commissioned BISA to supervise mineral processing and metallurgical testing.

BISA subcontracted certain parts of the study to Transmin Metallurgical Consultants (Transmin) in Lima, Peru. Test work was carried out by SGS Mineral Services (SGS). The test work was designed with the aim of developing an efficient flotation scheme for all mineral types found at El Domo. The investigation considered three composite samples sourced from 134 core samples stored at Curimining's core storage facility in Ventanas, Ecuador. The three composite samples considered the Cu/(Pb+Zn) ratio, the spatial distribution of the core, lithology, and distribution of grades. Final composite samples were grouped according to the Cu/(Zn + Pb) metal ratio and labelled as:

- Composite CPO-001: Zinc mineral: Cu/(Pb+Zn) <0.33;
- Composite CPO-002: Mixed copper/zinc mineral: 0.33 ≤Cu/(Pb+Zn) ≤3;
- Composite CPO-003: Copper mineral: Cu/(Pb+Zn) >3.

The study aimed to better understand the following parameters of mineralised material:

- Grind size;
- Effect of solids concentration in flotation on reducing interference from clays;
- Types of reagents for pyrite depression and reduction of insoluble entrainment to concentrate;
- Regrind effect;
- Flotation schemes.

Preliminary tests showed that flotation froths were overloaded with gangue. The use of lime to adjust the flotation pH was noted to cause the activation of gangue minerals, impairing the selective flotation of valuable minerals. Consequently, a bulk sulphide flotation stage at low pH and coarse grinding was successfully implemented resulting in the successful rejection of gangue minerals. This finding enabled the use of fine milling and lime to achieve selective flotation of a bulk concentrate. This same method was necessary for the three composites studied.

After implementing the bulk flotation system for the rougher stage, the use of the following schemes was evaluated for the separation of copper and zinc:

- Float the zinc and depress the copper; and
- Float the copper and depress the zinc.

Better results were achieved by depressing zinc.

Assays of concentrates produced during the test work found the presence of potentially penalty elements such as arsenic, antimony, bismuth, and cadmium.

Based on the results from the 2013 flotation testing program, Transmin estimated recoveries and grades of the copper and zinc concentrates that would be produced in an efficiently operated industrial plant, with mineralised material similar to the composites studied. Modelled recoveries and grades for each of the three (3) test composites are shown in Table 13.1.

Table 13.1 - Transmin Metallurgical Balance Projection – Recoveries and Grades

Composite ID	Products	Weight (%)	Assays					Distribution				
			Cu (%)	Zn (%)	Pb (%)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)	Au (%)	Ag (%)
CPO-001 Zinc Mineral	Cu Concentrate	2.46	19.7	17.0	1.31	8.18	748	58.2	8.24	7.67	6.16	22.6
	Pb Concentrate	0.61	17.7	17.0	47.3	21.8	1,995	6.47	0.92	69.0	4.11	15.1
	Zn Concentrate	9.49	1.73	45.5	0.31	14.0	286	19.7	85.4	6.90	40.8	33.4
	Final Tails	87.4	0.15	0.31	0.08	1.83	26.9	15.5	5.39	16.4	49.0	28.9
	Head Calculated	100	0.89	5.11	0.42	3.26	81.2	100	100	100	100	100
CPO-002 Mixed Mineral Cu / Zn	Cu Concentrate	7.62	21.0	8.51	0.94	8.77	230	75.0	29.8	49.6	29.3	40.8
	Zn Concentrate	2.59	6.28	42.0	0.86	19.4	371	7.62	50.0	15.4	22.0	22.4
	Final Tails	89.8	0.41	0.49	0.06	1.24	17.6	17.4	20.2	35.0	48.7	36.8
	Head Calculated	100	2.13	2.17	0.14	2.28	42.9	100	100	100	100	100
CPO-003 Copper Mineral	Cu Concentrate	13.9	24.2	2.34	0.09	3.69	53.1	89.7	80.9	43.4	24.4	39.7
	Final Tails	86.1	0.45	0.09	0.02	1.84	13.0	10.3	19.1	56.6	75.6	60.3
	Head Calculated	100	3.75	0.40	0.03	2.10	18.6	100	100	100	100	100

Source: RPA (PEA,2019)

13.2 Base Metallurgical Laboratories Inc - 2018- Early 2020

13.2.1 Assays, Mineralogy and Liberation

13.2.1.1 Head Samples

Three (3) different composite samples were selected for the program:

- Composite 1 – Representative of the deposit with respect to resource grade and spatially within the deposit;
- Composite 2 – Relatively high zinc grade; and
- Composite 3 – Relatively high copper grade and low zinc grade.

In addition to these three (3) composite samples, two continuity composite samples were identified but were not tested. Representative head samples were cut and removed from each composite and assayed using a multi-element ICP scanner, via fire assay (Au) and using infrared analysis (S and

C). A sequential copper speciation analysis was also performed on each sample. A sample assay summary is shown in Table 13.2.

Table 13.2 - Head Samples Assays

Sample	Element (% or g/t)										
	Cu	Pb	Zn	Fe	S	C	Ag	Au	As	Co Oxides	Cu Cyanide Leachable
Composite 1	1.66	0.25	2.77	12.40	15.60	0.13	35	2.54	697	0.02	0.13
Composite 2	1.58	0.54	5.40	11.40	16.30	0.11	62	2.76	727	0.02	0.13
Composite 3	2.14	0.04	0.39	20.80	26.00	0.08	14	1.23	301	0.05	0.17

Table 13.2 indicates that the composites have high copper grade and reasonable gold grade levels. Composites 1 and 2 also have high zinc, lead and silver levels. The levels of copper, zinc and lead in the composites dictated the process flowsheet developed during the test work program targeting recovery of these minerals into separate concentrates. Gold and silver recovery to separate copper and lead concentrates was evaluated during the flotation tests as these elements present additional value with respect to concentrate sale pricing.

The head ore assays identified that the composites contain detectable levels of arsenic. Further mineralogy and flotation test work has determined that copper-arsenic sulphides can be recovered to both copper and lead concentrates. Arsenopyrite or arsenic in solid solution with pyrite has been rejected to flotation tailings.

The copper speciation data indicates that there are few copper oxide mineral species in the samples. This corresponds with the relatively high copper recoveries achieved throughout the test work program. The level of cyanide soluble copper indicates the presence of secondary copper sulphide species within the composites. The presence of these mineral species, especially chalcocite and covellite, leads to sphalerite activation during flotation and therefore, aggressive chemical depressant schemes were required to reject sphalerite from the copper concentrate during the initial stages of flotation.

Sample mineralogy was determined quantitatively using QEMSCAN using the PMA protocol on 5 sized fractions per sample as presented in Table 13.3.

Table 13.3 - Head Samples Mineralogy Summary

Mineral	Mineral Content (%)		
	Composite 1	Composite 2	Composite 3
Chalcopyrite	4.67	4.10	5.97
Bornite	0.11	0.13	0.36
Chalcocite	0.11	0.10	0.09
Covellite	0.03	0.02	0.01
Enargite/Tennantite	0.16	0.24	0.06
Tetrahedrite	0.04	0.10	0.01
Galena	0.25	0.62	0.04
Sphalerite	4.55	7.99	0.65
Pyrite	23.40	20.90	43.10
Total Sulphides	33.32	34.20	50.29
Iron Oxides	0.52	0.28	0.48
Non-sulphides	66.20	65.50	49.30

The composites tested are considered to be semi massive sulphides (VMS). Composites 1 and 2 had about one third of the sample mass present as sulphide mineral. Composite 3 had considerably more sulphide mineralization, representing about half of the sample mass. The gangue minerals are predominantly quartz, muscovite/illite and chlorite. The levels of muscovite/illite, chlorite and kaolinite can also interfere with the flotation process. They are often associated with pulp viscosity effects which suggested lowering flotation solids density down to 25% w/w compared to about 35% w/w which is typical for base metals flotation at the rougher stage. All three (3) composites have these minerals totalling in the range of 38% (Composite 3) to 51% (Composite 1) of the sample mass.

Mineral liberation was studied using QEMSCAN analysis of the feed samples. The analysis was performed on samples ground to a nominal grind size P_{80} of 70 µm. The samples were separated into 5 size fractions, each fraction mounted in epoxy, polished, and analysed. A mineral liberation summary is presented in Table 13.4.

Table 13.4 - Head Samples Minerals Liberation Summary

Mineral Status	Composite 1 – 68 µm D_{80}					Composite 2 – 78 µm D_{80}					Composite 3 – 63 µm D_{80}				
	Cs	Ga	Sp	Py	Gn	Cs	Ga	Sp	Py	Gn	Cs	Ga	Sp	Py	Gn
Liberated	54	29	60	71	93	42	24	57	65	91	52	26	43	80	92
Binary – Cs	-	9	7	7	1	-	8	9	7	1	-	14	7	8	1
Binary – Ga	<0.5	-	1	<0.5	<0.5	1	-	2	<0.5	<0.5	<0.5	-	<0.5	<0.5	<0.5
Binary – Sp	5	16	-	2	1	13	13	-	2	2	1	5	-	1	<0.5
Binary – Py	23	7	9	-	4	19	2	5	-	4	32	16	26	-	6
Binary – Gn	9	9	12	17	-	8	5	12	20	-	9	13	9	10	-
Multiphase	9	30	11	3	1	17	48	15	6	2	6	26	15	2	1
Total	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**

Abbreviations: Cs-Copper Sulphide, Ga-Galena, Sp-Sphalerite, Py-Pyrite, Gn-Non-sulphide gangue

Comparing the data in Table 13.4 to similar polymetallic deposits in operation, the following should be noted:

- Copper sulphide liberation levels are low for efficient recovery to selective rougher concentrates;
- Composite 2 had noticeably lower copper sulphide liberation levels compared to the Composites 1 and 3;
- Copper sulphides for Composite 2 are interlocked with mineral species including pyrite, sphalerite, multiphase particles, and non-sulphide gangue;
- Sphalerite liberation for Composites 1 and 2 would be considered sufficient for selective rougher flotation recovery. Sphalerite interlocking is associated with non-sulphide gangue and multiphase particles. Recovery of sphalerite into a selective concentrate from Composite 3 is expected to be difficult due to the low sphalerite content and poor liberation; and
- Galena liberation for all composites was very low. Galena is interlocked with many sulphide species and gangue minerals. Production of a separate lead concentrate was not expected based upon these initial results but, the tests completed in the end of the program have produced lead concentrates with a Pb grade up to 39.4% Pb grade and included silver and gold credits.

13.2.2 Flotation Products

13.2.2.1 Mineralogy and Liberation

The mineralogy and liberation results for Test 6 which was conducted on Composite 2 included the copper flotation feed and final concentrate sample. It is important to note that the initial flotation test work focused on using a cyanide – zinc sulphate – copper sulphate scheme. The methodology employed was the same method employed for the head samples. This assessment conducted provided minerals association and therefore evaluated the potential for a regrind mill application and a reagent scheme to improve copper selectivity. A summary of these mineralogical results is shown in Table 13.5.

Table 13.5 - Copper Flotation Products Mineral Assays and Liberation Summary on Composite 2

Composite / Test	**Test 6**									
Particle Size	27 µm D_{80}									
Flotation Product	Cu Feed (Bulk Regrind Discharge)					Final (2nd Cu Cleaner) Concentrate				
Mineral	Cs	Ga	Sp	Py	Gn	Cs	Ga	Sp	Py	Gn
Mineral Assay	14%	2%	24%	48%	12%	63%	6%	9%	17%	5%
Mineral Status	**Mineral Liberation – 2 Dimensions (%)**									
Liberated	85	67	79	87	82	89	74	41	57	60
Binary – Cs	-	0	10	9	5	-	0	35	36	13
Binary – Ga	0	-	1	0	0	0	-	7	0	0
Binary – Sp	5	16	-	2	11	4	15	-	0	10
Binary – Py	4	0	3	-	0	5	0	2	-	0
Binary – Gn	0	0	3	0	-	1	9	2	1	-
Multiphase	6	16	4	1	2	2	2	13	5	16
Total	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**

Abbreviations:
Cs-(Chalcopyrite, Chalcocite/Covellite, Bornite, Tennantite/Enargite/Tetrahedrite), Ga-Galena, Sp- Sphalerite, Py-Pyrite, Gn-Non-sulphide Minerals

The results indicate the following:

- Copper mineral liberation is on the low side but not expected to impact copper recovery or concentrate levels;
- The majority of the galena is interlocked with sphalerite. The lead concentrate is expected to be reasonably clean. Elevated levels of zinc are expected;
- The sphalerite depression reagent scheme employed for Test 6 is considered adequate; and

- A large proportion of the sphalerite present is interlocked with the copper minerals and complex aggregates. This suggests that elevated zinc levels are expected in the copper concentrate. Due to the fine aggregation of the minerals, in all likelihood, a concentrate regrind step will not lead to an upgrade in concentrate grade.

13.2.2.2 Penalty Elements

Copper, lead and zinc concentrates produced from Composites 1, 2 and 3 during Locked Cycle Flotation tests (LCT) # 34, 35, and 36 respectively were assayed for the minor elements which attract the penalties to the concentrate. A summary is presented in Table 13.6. Smelter specification numbers in the table were referenced from the spelter terms analysis report prepared for Adventus (Exen, 2020), and updated smelter terms provided during the FS (Adventus ST, 2021).

Table 13.6 – Concentrate Penalty Elements and Smelter Specifications for Concentrates

Composite	Test	Concentrate	As	Sb	Hg	Cd	Bi	Pb+Zn	Mn	Fe
			%	%	ppm	%	%	%	%	%
1	BL0453-34	Cu Con	0.25	0.09	16	0.026	0.009	7.54		
2	BL0453-35	Cu Con	0.45	0.24	27	0.066	0.009	14.14		
3	BL0453-36	Cu Con	0.13	0.03	4	0.012	0.015	2.53		
Smelter terms (Exen, 2020)			0.20	0.05	10	0.030	none	4.00	none	none
Smelter terms – (Adventus ST, 2021)			0.20	0.05	10	0.030	none	4.00	none	none
1	BL0453-34	Zn Con	0.16	0.07	36	0.29	0.002		0.006	4.10
2	BL0453-35	Zn Con	0.20	0.08	101	0.3	0.002		0.008	3.20
Smelter terms (Exen, 2020)			none	none	none	0.30	none		0.50	8.00
Smelter terms – (Adventus ST, 2021)			none	none	none	0.30	none	none	none	none
1	BL0453-34	Pb Con	0.44	0.27	42	0.033	0.021			
2	BL0453-35	Pb Con	0.66	0.48	50	0.064	0.012			
Smelter terms (Exen, 2020)			0.50	0.50	50	none	0.10	none	none	none
Smelter terms – (Adventus ST, 2021)			none	none	none	none	none	none	none	none

The copper concentrates produced demonstrate penalty elements level higher than the referenced smelter terms limits.

Zinc and lead concentrates are relatively clean and no penalty element exceeds the limit established in the smelter terms referenced.

Based on DRA's in-house information and received smelter terms, the quality of all three (3) concentrates produced with in regards to inherent penalty elements with respect to marketability.

13.2.2.3 Gold Mineralogy

Gold mineralogy was conducted for the flotation tailings streams. Two of the three streams were expected to contain sufficient gold for recovery following flotation. The bulk rougher tailings and zinc rougher tailings streams were evaluated by Surface Scientific Western (SSW) who were subcontracted by BML to conduct these studies. A summary can be seen in Table 13.7.

Table 13.7 – Gold Mineralogy and Deportment Summary – Composites 1, 2, and 3

Description	Sample	Test #	Au Assay (g/t)	Au Distribution (% of head ore)	Gold Association (% of Total Au in Sample)				
					Visible	Pyrite	Chalcopyrite	Sphalerite	Fe Oxide
Bulk Rougher Tailings	Composite 1	Test 25	0.43	8.7	66.5	28.1	2.4	0	3
	Composite 2	Test 26	0.29	5.2	58.1	34.6	2.4	0	4.9
	Composite 3	Test 24	0.5	20.4	37.5	56.5	1.7	0	4.3
Zn Rougher Tailings	Composite 1	Test 25	4.2	35.6	66.8	32.8	0.4	0	0.1
	Composite 2	Test 26	6.7	60.0	63.8	20.6	0.4	15.2	0
Cu Cleaner Scavenger Tailings	Composite 3	Test 24	2.3	67.4	31.9	63.3	3.6	0.3	0

From Table 13.7, it is evident that the zinc rougher tailings (or copper cleaner scavenger tailings for Composite 3) stream appears to be potentially suitable for further extraction of gold as it contains a reasonably high gold grade and includes a considerable quantity of the gold being free and visible and therefore, potentially amenable to cyanidation. Recovery of the gold locked in pyrite may require processing which is currently considered to be cost prohibitive (except pyrite flotation for pre-concentration).

13.2.3 Comminution Tests

Bond Ball Mill Work Index (Bwi) testing was completed by Base Metallurgical Laboratories (BML). A summary of the results in comparison to content of total sulphides, quartz, muscovite / illite and clays is presented in Table 13.8.

Table 13.8 – Bond Ball Mill Work Index Results Summary

Sample	Bwi	P_{80}	Total Sulphides	Quartz	Muscovite / Illite	Kaolinite (Clay)
	kWh/t	microns	%	%	%	%
Composite 1	14.1	72	33.30	22.90	21.20	1.71
Composite 2	12.4	70	34.20	19.50	18.20	1.68
Composite 3	12.3	73	50.30	12.60	17.60	0.91

The BWI numbers suggest that from a ball milling perspective, the ore hardness can be described as medium for Composites 2 and 3 and medium hard for Composite 1.

13.2.4 Flotation Tests

The flotation of the El Domo ores have been studied over a number of years. The history of these tests has identified that flotation selectivity is key to concentrates produced during testing.

The BML flotation test work program included a sufficient number of tests which were used to define the main test work parameters for the designed flowsheet design:

- Flotation feed primary grind size;
- Depressant chemical scheme and regrind grind size conditions; and
- A conceptual flowsheet recommended for the projected concentrator.

Due to the identified selectivity issues, bulk flotation prior to the separation of Cu, Pb, and Zn was found to be the most effective and allowed for rejection of gangue and enabled the recovery of payable metals from the bulk concentrate.

13.2.4.1 Flotation Feed Grind Size

The optimal grind size was identified for Composite 1 during the ore bulk rougher tests by determining the optimal grind size versus Cu and Zn recovery and concentrate mass pull. Results are summarised in Table 13.9.

Table 13.9 - Composite 1 Bulk Flotation Performance vs Feed Grind Size

Averaged Values	D_{80} µm	Mass pull	Assay						Distribution					
		%	Cu %	Pb %	Zn %	Ag, g/t	Au, g/t	S %	Cu %	Pb %	Zn %	Ag %	Au %	S %
Test 15, 25, 28	75	28	5.8	0.8	9.2	104	8.8	43.5	93	92	91	86	87	71
Tests 1, 14	100	37	4.5	0.6	7.1	84	6.4	38.9	95	94	98	89	91	83
Tests 13, 16, 19, 20,21	150	34	4.7	0.6	7.5	87	6.8	42.2	95	91	92	86	88	84

Note: The values in the table are the weighed averages of the referenced tests

As seen from the table, coarser grinds do not have an adverse impact on the metal recoveries, and concentrate grades at coarser grinds are acceptable, however based on the better Cu grade for grind size of P_{80} of 75 µm, this grind size was considered for the PEA design and taken forward as the base case for further test work.

13.2.4.2 Depressant Scheme and Regrind Conditions

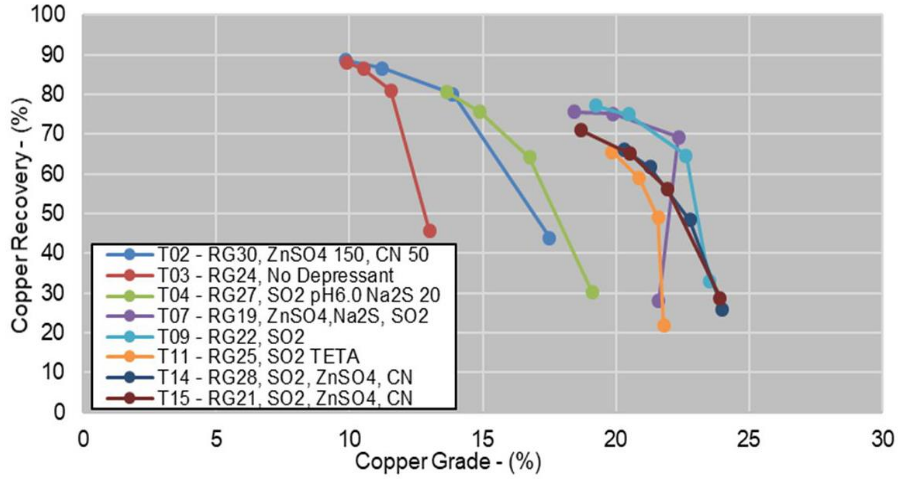
Testing of Composite 2 material was used to determine the optimal depressant chemical scheme and regrind size.

The selection of the depressant scheme significantly impacted copper flotation performance. Flotation performance was observed to be poorest when depressant was not used. The combination of cyanide and zinc sulphate produced superior results compared to the absence of depressants. However, poor selectivity against zinc was observed

The best performance achieved during the depressant evaluation tests was the test when sulphur dioxide, controlled to a pH of 4.5, was employed during conditioning. A reduction due to sulphur dioxide combined with an increase of the pH was less effective. Also, the addition of other depressants did not provide further benefit.

To simplify the depressant scheme, it was decided that testing move forward with utilising only sulphur dioxide as the depressant. Later on in the program it was discovered that SMBS could replace sulphur dioxide whilst producing similar results. Sodium cyanide was used successfully as a copper depressant during the lead separation. A summary of the copper circuit depressant optimization test results can be found in Figure 13.1.

Figure 13.1 – Effect of Depressant Type on Copper Performance

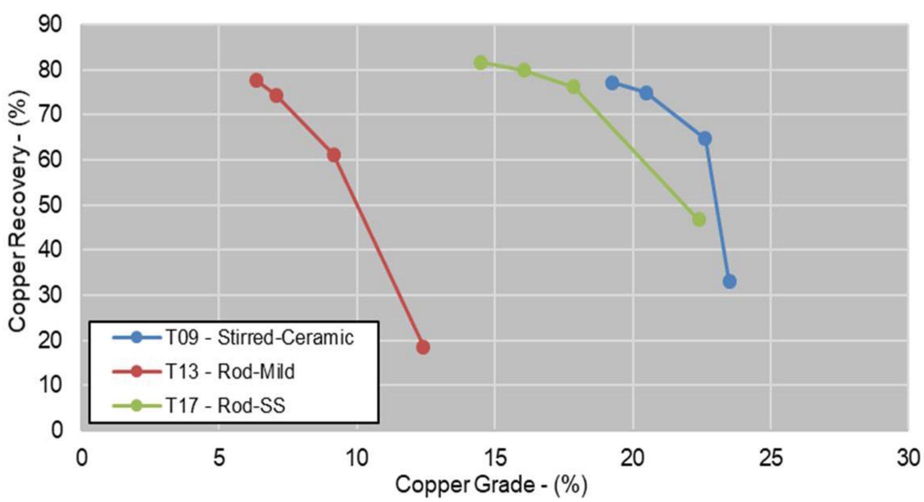


Note: RG indicates a P_{80} regrind size in microns. (Source: Base Metallurgical Laboratories Ltd.)

An effect of grinding media on flotation performance is an important ore specific, test based empirical indicator which varies based on ore type processed.

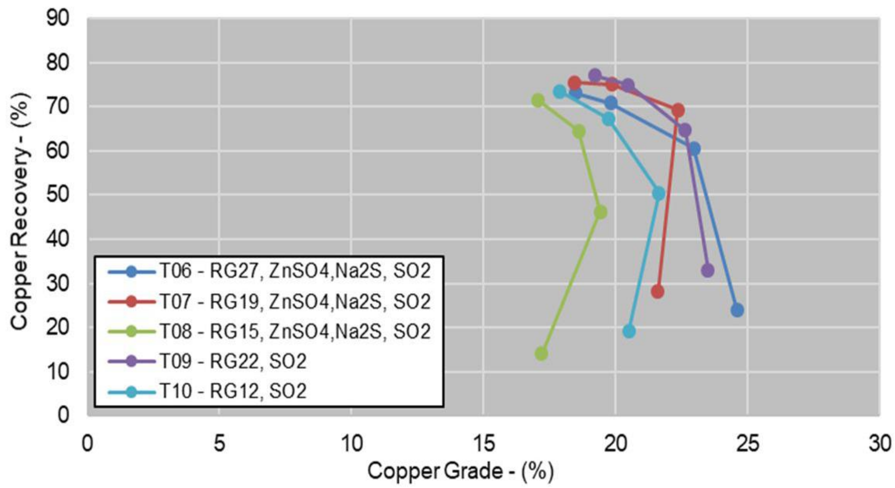
The effect of the regrind media material type comparing ceramic, stainless steel or mild steel media showed that when using mild steel media that this was detrimental to flotation performance. Using stainless steel or ceramic media was clearly beneficial to copper flotation performance. Based on DRA process design and operations experience, ceramic media is understood to be a most economical solution due to the substantially low consumption rate compared to the steel media. Ceramic media, therefore, was used for the majority of the regrind tests in the program. The impact of the regrind mill type and media type is shown below in Figure 13.2.

Figure 13.2 – Effect of Mill Type and Media Material on Copper Flotation Performance



(Source: Base Metallurgical Laboratories Ltd.)

Figure 13.3 – Copper Performance vs Regrind Size and Depressants



Note: RG indicates a D_{80} regrind size in microns.

13.2.4.3 Pyrite Flotation

An attempt to recover gold into a pyrite cleaner concentrate was made during the locked cycle testing. The concentrate produced was of unsaleable grade (3.9 g/t gold) at 20.1% gold recovery to pyrite concentrate.

13.2.4.4 Locked Cycle Tests

The Locked Cycle Test (LCT) conditions employed were based on results of the open circuit bench scale rougher and cleaner tests which set the baseline flotation flowsheet conditions and reagent scheme.

The same regrind conditions were applied for the LCT – regrind of the bulk concentrate to 25 µm P_{80}.

The depressant scheme varied slightly, and utilised H_2SO_3, sodium metabisulfite (SMBS), and a combination of SMBS with NaCN to depress copper in flotation, and a combination of $ZnSO_4$ and NaCN.

An SMBS / SMBS and NaCN regime was observed in producing the highest concentrate grades at good recovery for all circuits and all three (3) composites.

The use of the $ZnSO_4$ and NaCN scheme resulted in higher copper recovery, but concentrate grades were lower. Lead concentrate production was not tested for this scheme.

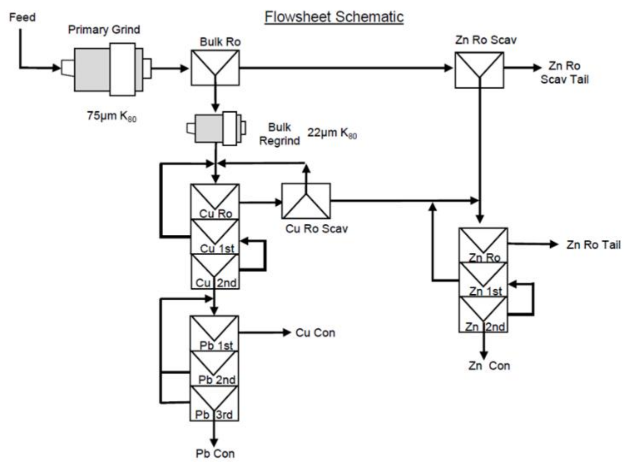
The use of H_2SO_3 resulted in a high concentrate grade and recovery within the range of the previous two schemes, however, the H_2SO_3 chemical is not commonly used in commercial applications. Lead flotation was not tested using this reagent scheme. A summary of locked cycle tests results is presented in Table 13.10 and Figure 13.4 depicts LCT flowsheet schematics.

Table 13.10 - Locked Cycle Tests Results Summary

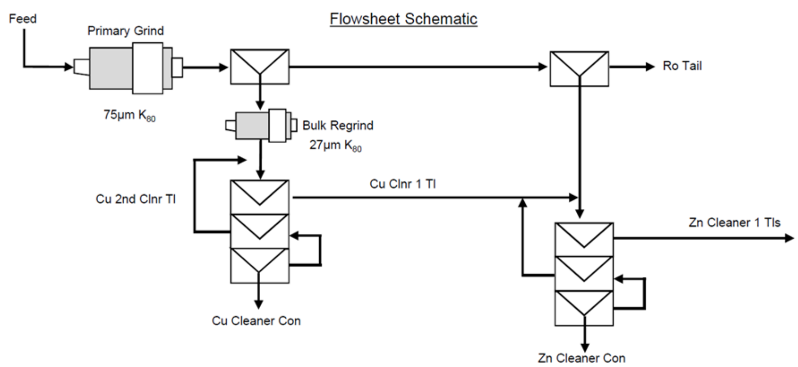
Product	Mass	Assay - % or g/t							Distribution (%)						
	%	Cu	Pb	Zn	Fe	Ag	Au	S	Cu	Pb	Zn	Fe	Ag	Au	S
SMBS, NaCN (in Pb flotation)															
BL0453 Composite 1 Locked Cycle Test 34															
Pb Conc.	0.4	9.55	39	8.42	12.6	401	38.4	26.2	2	55	1	0	4	6	1
Cu Conc.	5.3	27	0.7	6.8	26.8	244	11.2	37.1	81	15	13	11	36	24	12
Zn Conc.	4.1	1.41	0.72	56	4.1	239	15.3	34.3	3	12	81	1	27	25	8
Zn Ro Tail	19.5	0.59	0.13	0.3	33.8	37	3.91	41.7	7	10	2	52	20	31	48
Zn Ro Sc Tail	71.1	0.18	0.03	0.1	6.2	7	0.5	7.34	7	8	3	35	14	14	31
BL0453 Composite 2 Locked Cycle Test 35															
Pb Conc.	1	9.77	37	12.9	10.1		22.4	26.2	6	67	3	1	11	9	2
Cu Conc.	5.2	23	1.4	13	26.1	294	14.1	36.6	74	13	13	11	27	30	11
Zn Conc.	7.3	1.4	0.81	57	3.2	306	10	33.6	6	10	81	2	40	29	15
Zn Ro Tail	20.8	0.6	0.18	0.34	32.6	36	2.64	38.4	8	6	1	56	13	22	47
Zn Ro Sc Tail	66.7	0.15	0.03	0.11	5.5	7	0.35	6.36	6	3	1	30	8	9	25
BL0453 Composite 3 Locked Cycle Test 36															
Cu Conc.	6.1	29	0.3	2.3	29.8	73	2.94	37.5	80	37	36	9	29	14	9
Cu Cnr Scav Tail	37.2	0.67	0.06	0.55	39.5	20	2.29	48.4	11	50	54	69	49	65	69
Cu Rougher Tail	56.7	0.32	0.01	0.07	8.3	6	0.5	10	8	13	10	22	22	22	22
$ZnSO_4$, NaCN															
BL0293 Composite 1 Locked Cycle Test 49															
Cu Conc.	5.7	24	3	8.7	23.4	340	21.8	35.1	81	71	17	12	45	47	12
Zn Conc.	4.1	2.7	0.63	54	6	260	12.3	33.5	7	11	78	2	25	19	8
Zn 1st Clnr Tail	17.6	0.51	0.09	0.35	34.1	39	3.36	44	5	7	2	52	16	22	47
Zn Rougher Tail	72.5	0.16	0.04	0.08	5.4	9	0.41	7.26	7	11	2	34	15	11	32

Product	Mass	Assay - % or g/t							Distribution (%)						
	%	Cu	Pb	Zn	Fe	Ag	Au	S	Cu	Pb	Zn	Fe	Ag	Au	S
BL0293 Composite 1 Locked Cycle Test 52															
Cu Conc.	6.1	22	3	11	24.5	315	23.3	36.3	85	85	24	14	49	54	13
Zn Conc.	3.9	1.67	0.35	52	6.6	215	9.38	34.7	4	6	73	2	21	14	8
Zn 1st Clnr Tail	20.2	0.39	0.05	0.24	25.5	32	3.13	41.5	5	5	2	48	16	24	51
Zn Rougher Tail	69.9	0.13	0.01	0.08	5.6	8	0.31	6.41	6	4	2	36	14	8	27
BL0293 Composite 2 Locked Cycle Test 64															
Cu Conc.	7.4	17	6.7	21	17	495	24.6	33.9	80	83	30	12	53	59	15
Zn Conc.	5.9	1.66	0.71	58	3.3	325	8.59	33.4	6	7	65	2	28	16	12
Zn 1st Clnr Tail	16.7	0.54	0.2	0.88	28.3	42	2.79	42.7	6	5	3	44	10	15	43
Zn Rougher Tail	70	0.18	0.04	0.14	6.4	9	0.47	6.93	8	5	2	42	9	11	29
BL0293 Composite 3 Locked Cycle Test 59															
Cu Conc.	8.3	23	0.3	2.7	31.6	105	3.92	37.6	88	58	59	12	46	24	12
Cu Cnr Scav Tail	35.6	0.31	0.03	0.3	41.8	18	1.95	49.3	5	24	33	69	35	51	69
Cu Rougher Tail	56.1	0.28	0.02	0.06	7.4	7	0.59	8.62	7	19	8	19	19	25	19
H_2SO_3															
BL0453 Composite 1 Locked Cycle Test 25															
Cu Conc.	5.9	24	2.8	11	25.1	280	16.1	35.8	82	76	22	12	44	35	13
Zn Conc.	3.9	1.72	0.58	55	6.1	225	11.3	33.9	4	10	73	2	23	16	8
Zn 1st Clnr Tail	23.7	0.54	0.1	0.22	34.1	35	4.58	41.7	7	11	2	65	22	40	60
Zn Rougher Tail	66.6	0.18	0.01	0.12	4	6	0.36	4.7	7	3	3	22	11	9	19
BL0453 Composite 2 Locked Cycle Test 26															
Cu Con	5	24	7	9.3	23	328	17.5	34.9	74	65	9	10	28	26	10
Zn Con	2.3	2.85	2.62	52	4.9	307	8.75	33.8	4	12	25	1	13	6	5
Zn 1stClnr Tail	28	0.88	0.39	11.3	29.8	108	7.26	41.9	16	21	64	71	52	62	70
Zn Rougher Tail	64.7	0.14	0.03	0.1	3.3	6	0.26	3.74	6	3	1	18	7	5	15
BL0453 Composite 3 Locked Cycle Test 24															
Cu Conc.	5.8	27	0.3	2.4	31.2	64	3.19	38	71	35	36	8	25	14	8
Cu Cnr Scav Tail	39	1.2	0.07	0.51	40.8	21	2.34	50.6	21	54	53	73	55	68	74
Cu Rougher Tail	55.2	0.31	0.01	0.07	7.3	6	0.44	8.62	8	11	10	18	20	18	18

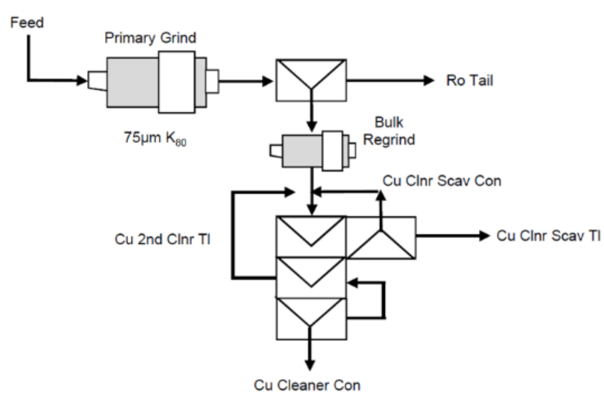
Figure 13.4 – Locked Cycle Tests Flowsheet Schematics



BL0453, Composite 1, Test 34, 35



BL0293, Composite 1, Test 49, 52, Composite 2, Test 64



BL0453, Composite 3, Test 24, 36, BL0293, Composite 3, Test 59

The optimal LCT results, the SMBS-NaCN depressant scheme, and the flowsheet utilised during Tests # 34 and 35, were suitable to form the basis of design for the commercial flotation plant flowsheet and the ability to produce three (3) separate saleable concentrates of Cu, Pb and Zn as well as to accommodate expected plant feed grade variations.

13.2.5 Cleaner Tailings Cyanide Leach Tests

Flotation tests and mineralogical studies conducted by BML have identified that cleaner flotation tailings have gold and silver potentially amenable for recovery via cyanidation. In order to evaluate the amenability, samples of cleaner tailings were split from LCT # 34, 35, and 36 and subjected to bench scale bottle roll cyanide leach tests. The concentration of cyanide in the leach of feed strength and feed P_{80} were varied for the tests. Cyanide leach kinetics and metal recoveries were determined versus the change in conditions. The results summary can be found in Table 13.11.

Leach tests determined that at a NaCN concentration of 1,000 ppm that metal recovery rates are low. By increasing NaCN concentration to 5,000 ppm and by introducing a leach feed regrind, this has resulted in extractions of around 50% of Au, 60% of Ag, 75% of Cu for each of the three (3) composites. On average, around 15 to 20% of the contained Zn appears to be leachable at the conditions tested.

Consumption of cyanide and lime to achieve these results is high averaging 20.5 kg/t, and 12.7 kg/t of leach feed solids respectively. The high cyanide consumption and extraction of copper to cyanide solution indicated that evaluating the recovery of copper and precious metals using the SART process to reduce cyanide consumption and clean the leach solution of soluble copper would be prudent.

Table 13.11 - Leach Tests Results Summary

Leach Test			37	40	43	46	38	41	44	47	39	42	45	48
Flotation Tailings			BL0453-34				BL0453-35				BL0453-36			
Composite			Composite 1				Composite 2				Composite 3			
Leach Feed D80		micron	**25**	25	15	9.3	**25**	25	14	9.2	**25**	25	16	8.6
Leach NaCN Strenght		ppm	**1000**	5000	5000	5000	**1000**	5000	5000	5000	**1000**	5000	5000	5000
NaCN Consumption		kg/t	**5.8**	13.2	15.5	24.2	**5.9**	11.4	17	21.2	**5.9**	9.5	17.5	27.3
Lime Consumption		kg/t	**6.2**	12.9	11	13.4	**4.5**	6.9	11.4	13.2	**5.8**	12	13.8	13.2
Residence Time, h		Leach Recovery, %	Au											
	2		**5.3**	25.9	30.2	36.8	**5.3**	29.5	32.1	38.9	**9.8**	23.7	25.5	40.6
	6		**8.8**	32.7	34.8	48.9	**9.1**	38.1	35.8	47.9	**12.8**	26.6	28.3	55.8
	24		**11.4**	34.6	43.0	48.6	**17.1**	40.3	50.4	48.5	**15.1**	28.1	36.4	57.9
	48		**14.0**	33.7	44.0	47.4	**23.0**	41.6	49.7	49.1	**16.9**	27.6	35.6	59.5
			Ag											
	2		**0.5**	26.0	7.1	43.9	**0.7**	28.0	3.4	40.6	**2.1**	29.2	7.8	28.6
	6		**1.7**	36.3	24.0	55.2	**1.4**	33.9	21.1	50.4	**7.1**	30.8	8.2	48.3
	24		**8.0**	44.3	53.0	59.6	**6.4**	37.8	53.5	56.0	**16.6**	35.4	54.1	58.0
	48		**12.1**	46.7	58.2	61.2	**13.6**	41.9	57.2	58.3	**24.2**	37.3	57.7	59.4
			Cu											
	2		**10.4**	42.3	45.5	54.0	**10.9**	32.6	40.4	48.1	**11.2**	24.6	27.9	45.5
	6		**19.9**	49.8	48.5	66.8	**20.8**	39.7	44.6	59.7	**20.4**	28.4	35.6	62.3
	24		**29.4**	57.7	68.2	82.5	**30.2**	47.3	69.6	71.1	**25.3**	36.4	62.6	83.1
	48		**33.0**	60.9	77.3	86.1	**36.2**	52.6	77.2	76.0	**32.4**	40.1	70.1	88.7
			Zn											
	2			9.0	0.2	13.2		16.3	1.7	14.7		5.8	1.5	5.7
	6			25.6	10.8	16.2		22.2	11.6	19.8		6.9	4.0	6.6
	24			29.3	14.5	21.4		26.9	13.0	26.5		7.7	2.0	8.7
	48			31.9	16.7	25.5		31.1	17.3	31.7		8.3	5.0	10.8

13.2.6 GRAVITY TESTS

Bench Knelson concentrator tests were conducted at 120 G with panning of the gravity concentrate performed on each ore composite produced. Due to the relative fineness of P_{80} of 1,000 µm of the feed material, these tests were performed without any further grinding. Gold recovered from the feed composites was 14% to 33% of contained gold from the feed to the Knelson concentrate at a mass recovery around 8% (1.75 to 4 upgrade ratio). Panning this concentrate to about 0.7% mass recovery reduced the gold recovery significantly – 1% to 9% for the composites. Based on these initial results, removal of coarse liberated free gold from the grinding circuit is unlikely.

13.2.7 CLEANER TAILINGS

For each of the composites, samples of cleaner tailings used in the cyanide leach work were tested with respect for the potential for gravity concentration using a Mozley table. The samples had a K_{80} of 25 µm. The maximum recovery of gold to concentrates was around 25% for all three (3) composites. These recoveries were produced at high mass pull. Composites 1 and 2 did show some upgrading, but overall, the recovery performance was poor. It should be noted that the tests conducted were preliminary in nature with minimal mass available, thereby reducing representative confidence..

13.2.8 SETTLING TESTS

Static settling tests were performed on final tailings Composite 1 and 3. The results summary is in Table 13.12.

Table 13.12 – Settling Test Results

Sample	Flocculant	Flocculant Dosage g/t	pH	Settling Rate mm/sec	Final Density % solids
Composite 1, BL0293-49, Bulk Rougher Tailings plus Zn 1st Cleaner Tailings	MF-10	20	7.9	1.06	-
	MF-155	20	7.9	0.02	-
	MF-336	20	7.9	0.46	-
	MF-351	20	7.9	0.65	-
	MF-10	20	10	0.86	-
	MF-351	20	10	0.63	-
	MF-10	10	7.9	0.02	46.4
	MF-10	30	7.9	0.05	46.1
	MF-10	40	7.9	0.13	45.8
Composite 3, BL0293-59, Rougher Tailings plus Cu 1st Cleaner Scavenger Tailings	MF-10	20	7.1	0.28	-
	MF-155	20	7.1	0.05	-
	MF-336	20	7.1	0.03	-
	MF-351	20	7.1	0.04	-
	MF-10	20	10	0.08	-
	MF-155	20	10	0.08	-
	MF-10	10	7.1	0.00	41.6
	MF-10	30	7.1	0.01	41.4
	MF-10	40	7.1	0.02	40.9

Both samples demonstrated low settling velocities (for the majority of tests) with Composite 1 material showing improved settling characteristics. Final solids densities were observed to be low for both samples. Possibly, Mica minerals are responsible for the poor settling performance. Additional settling tests were recommended for tailings dewatering.

13.3 Base Metallurgical Laboratories Inc - 2020

The program commenced at the end of August 2020 and was completed by the end of December 2020. The program used samples from the previous programs, received in 2018. Two (2) sources of samples were available.

- The leftovers from Composites 1, 2, and 3 identified as Sample Type 1 tested during 2018-2019; and
- The drill core sample identified as Sample Type 2. The Sample Type 2 composites were constructed by geological lithology, six (6) such samples were created.

Sample Type 1 was used to further optimise the bulk flotation process and generate tailings for dewatering studies. Oxidation of the sample due to the fine crush size of the material was a concern. Standard tests were performed to determine if the samples were suitable for some of the more complex metallurgical testing.

Sample Type 2 was intended for comminution testing, Dense Media Separation (DMS) tests and ore sorting evaluations to establish metallurgical performance of the developed process by lithology.

After the sample preparation on the Sample Type 2 commenced, it was apparent that the core was extremely friable and crumbled easily.

The SMC tests were limited to relatively fine fractions and the DMS tests and ore sorting evaluations were cancelled.

13.3.1 Assays, Mineralogy, Liberation

The head assays of the samples tested are shown in Table 13.13.

Table 13.13 – 2020 Program Head Assays

Sample	Element (% or g/t)										
	Cu	Pb	Zn	Fe	S	C	Ag	Au	As	CuOx	CuCN
Composite 1	1.66	0.25	2.77	12.4	15.6	0.13	35	2.54	697	0.02	0.13
Composite 2	1.58	0.54	5.40	11.4	16.3	0.11	62	2.76	727	0.02	0.13
Composite 3	2.14	0.04	0.39	20.8	26.0	0.08	14	1.23	301	0.05	0.17
VMS-1 (Volcanic Massive Sulphide)	4.00	0.32	2.95	31.1	40.5	<0.01	64	4.00	1048	0.07	0.16
SMS-2 (Semi-Massive Sulphide)	1.65	0.24	1.56	17.8	21.9	0.01	31	1.56	1343	0.05	0.14
Falla-3 - Fault Zone Rock	0.39	0.05	0.19	26.7	32.7	0.04	14	0.74	523	0.01	0.02
BX-4 - Breccia	0.29	0.38	1.69	3.33	5.08	0.05	56	2.35	631	0.00	0.01
Falla-Gr-5 - Fault Zone Grainstone	0.87	0.20	1.19	7.90	9.64	0.04	42	2.25	591	0.02	0.02
SED-6 - Sedimentary Rock	1.40	0.13	0.89	2.58	3.62	0.08	19	0.59	-	0.01	0.05

Legend:
CuOx - Copper oxides
CuCN - Copper cyanide leachable

As seen from the table, all the samples have considerable copper and gold levels, except for Falla-3. Select composites also have high zinc, lead, and silver levels. The levels of copper, zinc and lead will dictate process selection, as these minerals would ideally be recovered into separate concentrates. Gold and silver are often recovered as by-product values into the concentrates.

The arsenic levels in the feed are relatively high, the mineral form of arsenic will impact deportment to concentrate or tailings. Arsenic contained in copper-arsenic sulphides recovered to copper and lead concentrates, and arsenopyrite or arsenic in solid solution with pyrite is rejected to tailings.

13.3.1.1 Mineral Content and Fragmentation Characteristics

The content and fragmentation characteristics were measured for each composite. The mineral content was determined quantitatively by QEMSCAN using the PMA protocol.

The Sample Type 1 composites were a mix of geological lithologies and were broadly classified as semi-massive sulphide mineralization. The second set of composites (Sample Type 2) were based on geological lithology and therefore were expected to display more variation with the sulphide content varied by geological classification. The sulphide content ranged from 10% for the BX-4 (Breccia) composite to 71% in the VMS-1 (Volcanic Massive Sulphide). There were some differences in the host rock, but the non-sulphide component was mostly muscovite/illite and relatively high levels of kaolinite and chlorites. These minerals are the likely candidates responsible for the poor settling characteristics and high viscosity if lime is used in bulk flotation.

The primary sulphide minerals in the samples were pyrite, chalcopyrite, sphalerite, and galena. Pyrite was the most abundant sulphide mineral in each sample. Chalcopyrite content ranged from 0.7 to 10.2%. All samples had trace levels of bornite, chalcocite, covellite, enargite, tennantite and tetrahedrite. The levels of secondary copper sulphides bornite, covellite and chalcocite are relatively low, but can have great significance on the flotation process. These minerals are much more reactive than chalcopyrite and contribute copper ions causing activation of sphalerite during the copper flotation process. The mineral composition summary is shown in Table 13.14.

Table 13.14 – Samples Mineralogy Summary

Mineral	Mineral Content (%)							
	Comp 1	Comp 2	Comp 3	VMS-1	SMS-2	Falla-3	BX-4	Falla Gr-5
Chalcopyrite	4.67	4.10	5.97	10.20	3.83	1.44	0.71	2.55
Bornite	0.11	0.13	0.36	0.05	0.16	0.01	<0.01	<0.01
Chalcocite	0.11	0.10	0.09	0.09	0.14	0.02	0.01	0.04
Covellite	0.03	0.02	0.01	0.01	0.05	<0.01	<0.01	<0.01
Enargite/Tennantite	0.16	0.24	0.06	0.40	0.37	0.03	0.03	0.03
Tetrahedrite	0.04	0.10	0.01	0.13	0.09	0.01	0.04	0.03
Malachite/Azurite	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Galena	0.25	0.62	0.04	0.35	0.24	0.06	0.42	0.16
Sphalerite	4.55	7.99	0.65	4.35	2.51	0.38	2.64	2.09
Pyrite	23.40	20.90	43.10	55.70	35.00	52.70	5.79	14.10
Pyrrhotite	0.02	0.01	0.04	0.09	0.03	0.09	0.03	0.03
Arsenopyrite	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Total Sulphides	33.34	34.21	50.33	71.37	42.42	54.74	9.67	19.03
Iron Oxides	0.52	0.28	0.48	0.10	0.20	0.07	0.14	0.13
Quartz	22.90	19.50	12.60	9.20	18.40	9.10	24.00	11.00
Muscovite/Illite	21.20	18.20	17.60	13.70	28.10	24.50	31.30	38.10
Other Non-sulphides	22.07	27.81	19.03	5.55	10.99	11.61	34.84	31.84

Mineral liberation summary is presented in Table 13.15.

Table 13.15 – Mineral Liberation Summary

Mineral		Liberated	Binary - Cs	Binary - Ga	Binary - Sp	Binary - Py	Binary - Gn	Multi phase	Total
Comp 1 68 micron K_{80}	Copper Sulphides	54		<0.5	5	23	9	9	100
	Galena	29	9		16	7	9	30	100
	Sphalerite	60	7	1		9	12	11	100
	Pyrite	71	7	<0.5	2		17	3	100
	Gangue	93	1	<0.5	1	4		1	100
Comp 2 78 micron K_{80}	Copper Sulphides	42		1	13	19	8	17	100
	Galena	24	8		13	2	5	48	100
	Sphalerite	57	9	2		5	12	15	100
	Pyrite	65	7	<0.5	2		20	6	100
	Gangue	91	1	<0.5	2	4		2	100
Comp 3 63 micron K_{80}	Copper Sulphides	52		<0.5	1	32	9	6	100
	Galena	26	14		5	16	13	26	100
	Sphalerite	43	7	<0.5		26	9	15	100
	Pyrite	80	8	<0.5	1		10	2	100
	Gangue	92	1	<0.5	<0.5	6		1	100
VMS-1 72 micron K_{80}	Copper Sulphides	51		0	6	31	5	8	100
	Galena	21	7		21	10	12	28	100
	Sphalerite	44	14	2		17	6	17	100
	Pyrite	82	8	0	1		6	3	100
	Gangue	79	3	0	2	12		4	100
SMS-2 69 micron K_{80}	Copper Sulphides	45		1	6	27	10	11	100
	Galena	30	16		7	8	12	26	100
	Sphalerite	52	12	1		11	11	14	100
	Pyrite	69	5	0	1		23	2	100
	Gangue	87	1	0	1	10		1	100

Mineral		Liberated	Binary - Cs	Binary - Ga	Binary - Sp	Binary - Py	Binary - Gn	Multi phase	Total
Falla-3 78 micron K_{80}	Copper Sulphides	43		0	1	45	6	4	100
	Galena	39	7		6	21	3	23	100
	Sphalerite	35	3	1		36	15	10	100
	Pyrite	90	1	0	0		8	0	100
	Gangue	90	0	0	0	9		0	100
BX-4 78 micron K_{80}	Copper Sulphides	47		2	8	17	14	12	100
	Galena	43	4		17	5	13	18	100
	Sphalerite	63	3	3		4	19	7	100
	Pyrite	43	2	0	1		52	2	100
	Gangue	95	0	0	1	4		0	100
Falla GR-5 72 micron K_{80}	Copper Sulphides	41		0	7	14	23	15	100
	Galena	27	8		14	8	16	28	100
	Sphalerite	49	9	1		7	18	17	100
	Pyrite	42	2	0	1		52	2	100
	Gangue	88	1	0	0	10		1	100

As seen from the table, on average, liberation of copper sulphides, galena and sphalerite is 47%, 30%, and 50% respectively which points on the necessity of regrind post bulk flotation to produce high grade concentrates. Another observation is that up to 48% of galena is locked within the multimineral (multiphase) agglomerates, which in turn, explains the challenges of producing the high grade lead concentrates.

13.3.2 Comminution Tests

Comminution testing of the Sample Type 1 was limited to Bond Ball Mill Index determination because of the fine crush size of Sample Type 1.

The Sample Type 2 material represented by the drill core was subjected to SMC, Bond Ball Mill Index tests and Bond Abrasion tests. During the preparation of the core for the SMC test, which required large competent core pieces, it was noted that the core was extremely friable. The drill core in many cases was unrecognisable as competent rock, but more like soft rubble. This is reportedly common for drill core from this deposit.

Due to the core friability for Sample Type 2 material, the Dense Media Separation (DMS) tests and ore sorting evaluations were cancelled.

A summary of the comminution test results is provided in Table 13.16.

Table 13.16 - Comminution Test Results Summary

Sample	DWi	Mia	Mih	Mic	A x b	ta	SCSE	SG	BWi	Ai
	kWh/m^3	kWh/t	kWh/t	kWh/t			kWh/t		kWh/t	
BX-4	3.09	10.7	6.7	3.5	86	0.84	7.19	2.65	14.6	0.243
Falla-3	2.07	6.2	3.7	1.9	159	1.26	6.03	3.28	14.0	0.097
Falla-Gr-5	2.81	9.5	5.9	3.1	98	0.92	6.92	2.75	14.6	0.129
SMS-2	2.88	7.9	5	2.6	117	0.9	6.72	3.37	13.3	0.321
VMS-1	2.69	6.4	3.9	2	148	0.96	5.81	3.98	11.6	0.106
Composite 1									14.1	
Composite 2									12.4	
Composite 3									12.3	

SMC test data for the geological lithology samples indicate the material as low competence.

The Bond ball mill determinations were completed with a closing screen size of 106 microns, resulting in a P_{80} size of nominally 70 to 84 microns. The Bond ball mill work index values ranged from 11.6 to 14.6 kWh/t. Therefore, the samples were medium hard from a ball milling perspective.

The Bond Abrasion tests had a considerable range in values from 0.097 to 0.321. The average value was 0.179, which would be considered moderately abrasive.

13.3.3 Bulk Flotation Optimisation

Impact of primary grind on bulk flotation recoveries and concentrate grades was studied on the Sample Type 1 material (Composites 1, 2, 3) within a range of the flotation feed size (K_{80}) from 75 to 150 microns. A summary of the results is shown in Table 13.17.

Table 13.17 - Impact of Primary Grind on Bulk Rougher Performance

Test	Sample	K80, microns	Masspull %	Assays, % or g/t							Distribution, %						
				Cu	Pb	Zn	Fe	S	Ag	Au	Cu	Pb	Zn	Fe	S	Ag	Au
BL0597-01	Composite 1	75	35.0	4.54	0.63	7.24	27.79	39.94	97.72	7.22	94.58	94.41	94.65	81.46	83.45	89.75	95.17
BL0597-04		100	37.0	4.54	0.60	7.04	28.39	40.92	88.43	7.31	96.38	89.77	94.51	86.86	86.40	89.64	89.24
BL0597-07		125	36.5	4.51	0.56	6.99	29.48	40.72	92.51	7.26	95.93	89.01	94.81	87.76	87.24	89.85	90.38
BL0597-10		150	37.7	4.89	0.58	7.20	25.42	38.28	88.39	7.00	97.04	89.75	95.61	87.52	87.62	89.90	88.37
BL0597-02	Composite 2	75	35.2	4.17	1.45	13.50	25.20	39.92	162.06	7.69	93.40	95.17	95.19	79.34	82.61	91.66	95.69
BL0597-05		100	37.8	3.98	1.37	12.84	22.15	39.14	156.08	8.15	95.26	94.34	95.58	80.77	85.95	93.12	94.50
BL0597-08		125	35.5	4.13	1.45	13.54	24.04	41.75	151.89	8.14	94.20	93.03	94.43	81.54	84.85	90.30	92.16
BL0597-11		150	36.2	3.95	1.45	13.01	24.62	40.64	149.31	7.38	94.52	91.13	94.26	83.31	86.40	90.40	91.92
BL0597-03	Composite 3	75	50.2	4.15	0.09	0.70	38.66	49.52	27.87	2.32	95.01	90.08	94.61	89.95	90.18	87.55	94.39
BL0597-06		100	50.9	4.13	0.10	0.70	35.56	48.28	24.66	2.24	95.76	91.07	94.76	89.78	90.27	86.48	88.33
BL0597-09		125	48.5	4.11	0.09	0.74	34.75	48.11	26.23	2.29	95.08	81.35	93.29	88.61	88.97	86.05	86.68
BL0597-12		150	50.9	4.05	0.10	0.72	34.92	47.98	27.19	2.34	96.10	83.72	93.71	90.28	91.00	87.55	89.02

As seen from Table 13.17, within the studied range, the recoveries for the payable metals are not adversely impacted by the coarser primary grind.

An impact of varied SIPX collector dosage on bulk rougher performance is presented in Table 13.18.

Table 13.18 - Impact of Collector Dosage on Bulk Rougher Performance

Test	Sample	Collector Dosage, %	K80, µm	Masspull %	Assay - percent or g/t							Distribution - percent						
					Cu	Pb	Zn	Fe	S	Ag	Au	Cu	Pb	Zn	Fe	S	Ag	Au
BL0597-13	Composite 1	80%	75	30.23	5.18	0.74	7.78	29.00	43.92	98.50	7.30	91.81	88.85	85.33	74.77	76.31	82.59	83.17
BL0597-14		90%	75	30.01	5.63	0.78	8.45	27.81	44.22	105.59	7.35	90.96	84.82	83.43	73.41	77.55	81.91	83.79
BL0597-01		100%	75	34.97	4.54	0.63	7.24	27.79	39.94	97.72	7.22	94.58	94.41	94.65	81.46	83.45	89.75	95.17
BL0597-15		80%	125	31.72	5.01	0.69	8.06	28.80	42.77	95.03	7.15	91.73	88.94	91.50	78.98	80.48	81.53	84.06
BL0597-16		90%	125	31.74	4.91	0.70	8.16	29.01	43.56	98.60	6.69	91.58	89.12	91.77	78.56	80.26	85.14	83.16
BL0597-07		100%	125	36.48	4.51	0.56	6.99	29.48	40.72	92.51	7.26	95.93	89.01	94.81	87.76	87.24	89.85	90.38
BL0597-17	Composite 2	80%	125	38.51	4.11	1.39	12.88	22.84	39.90	145.44	7.05	95.98	95.61	95.96	84.02	87.50	91.93	93.84
BL0597-18		90%	125	37.05	4.13	1.37	12.86	21.97	40.06	142.07	7.98	94.96	94.17	96.31	81.99	86.09	91.27	93.62
BL0597-08		100%	125	35.54	4.13	1.45	13.54	24.04	41.75	151.89	8.14	94.20	93.03	94.43	81.54	84.85	90.30	92.16

As seen from Table 13.18, collector dosage can be reduced within the range studied without detrimental impact on bulk rougher flotation.

13.3.4 Sample Type 2 – Copper/Lead Rougher Circuit Optimization

Samples of different lithology (VMS, SMS, Breccia, and Grainstone) were tested using the developed flowsheet to determine the metallurgical response. The separation tests on the bulk concentrate produced separate copper/lead and zinc concentrates. Initially, the performance was poor, with high levels of zinc recovered to the copper/lead concentrate. A series of additional tests were performed to optimise the performance. Figure 13.5 depicts the copper/lead rougher flotation tests and the efforts to improve the circuit selectivity against zinc.

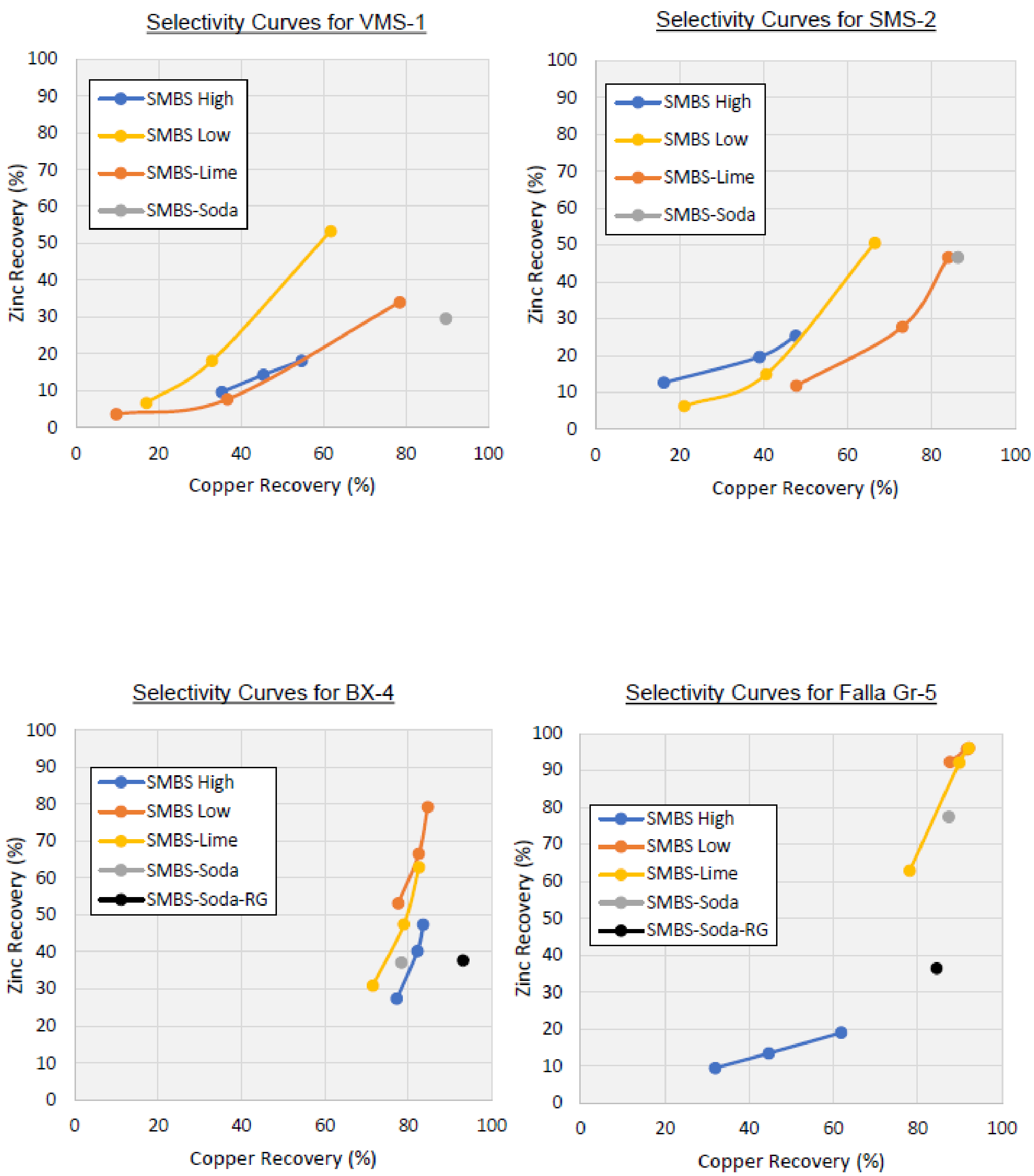
During the tests, it was noted that many of the bulk concentrates after regrinding had much lower pH (5.2 to 5.8) when compared to the tests for Composites 1 to 3 (pH 7.5 to 7.8). The flotation pulp pH was used as a control for the dosage levels of SMBS, attempting to achieve a pH level of 5.5.

Since the pH of the flotation pulp was already near the target levels, very little SMBS was added in the initial tests. As shown in Figure 13.5, the low (SMBS) demonstrated very poor selectivity. High levels of zinc were recovered to the copper/lead concentrate.

Tests with high SMBS levels caused the pH level to decrease (4.0 to 5.9). Although the selectivity was better, copper recovery was low for most samples. To rectify the low pH, the addition of lime preceded the SMBS addition for the next series of tests. Lime was added to increase the pH to about 8.0, then more SMBS was added to bring the pH back to approximately 5.5. As illustrated in Figure 13.5, better copper grades were achieved with higher copper recoveries.

For composites BX-5 (Breccia) and Falla Gr-5 (Grainstone), high recovery of zinc to the copper/lead concentrate persisted. Adding SMBS to the regrind mill was investigated along with the replacement of lime with soda ash as pH modifier. These tests (black data points) generally displayed high copper recovery with the lowest zinc recovery. This reagent approach was used in subsequent LCTs.

Figure 13.5 – Copper/Lead Rougher Optimisation



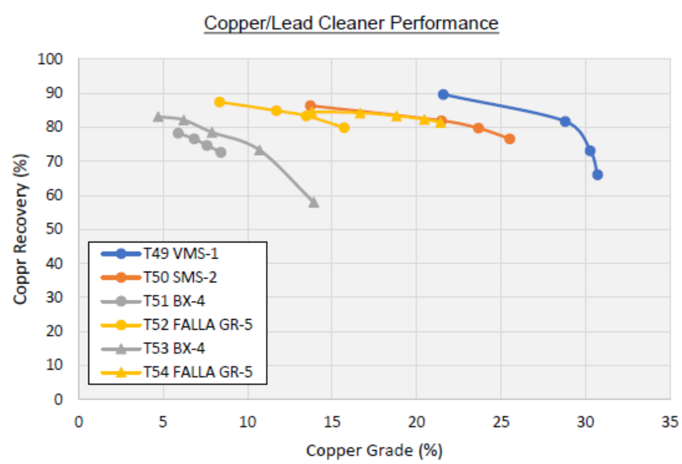
Legend: VMS-1 - VMS sample; SMS-2 - SMS sample; BX-4 - Breccia; Falla Gr-5 - Grainstone

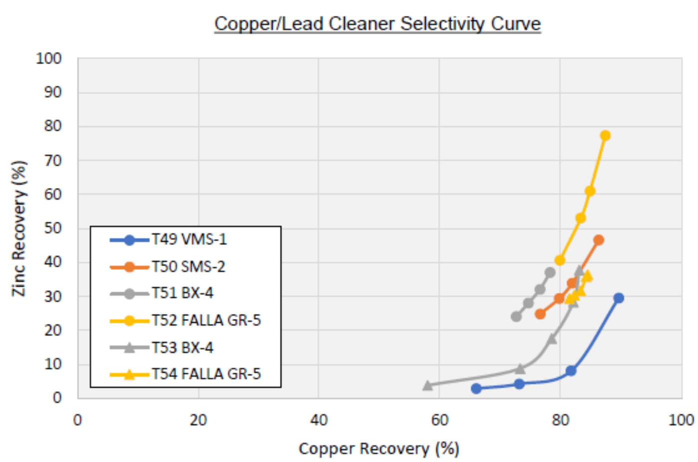
Source: BML

13.3.5 SAMPLE TYPE 2 – COPPER/LEAD CLEANER CIRCUIT PERFORMANCE

A limited number of batch cleaner tests were performed on the four (4) main lithology samples. These tests were performed to confirm the reagents in the copper/lead cleaner circuit and calibrate the regrind times in advance of locked cycle testing. **Error! Reference source not found.** displays t he copper grade recovery curves for the copper/lead concentrate. The selectivity of the copper/lead circuit against zinc in the cleaner circuit are shown in the bottom graph.

BMLFigure 13.6 – Copper/Lead Batch Cleaners Selectivity and Flotation Performance





Source: BML

As shown in Error! Reference source not found., the VMS-1, SMS-2, and Falla Gr-5 all d emonstrated copper concentrates grading better than 22% could be produced at recoveries greater than 80% in batch test. The BX-4 composite had much lower grade copper concentrate, due mainly to high lead content. A lead separation circuit will be required to increase the grade of the copper concentrate. The copper concentrate using soda ash and higher SMBS levels had good rejection of zinc from the copper/lead concentrate.

A summary of the batch cleaner tests is presented in Table 13.19.

Table 13.19 - Copper/Lead Batch Cleaner Results Summary

Product	Weight		Assay - percent or g/t							Distribution - percent						
	%	grams	Cu	Pb	Zn	Fe	S	Ag	Au	Cu	Pb	Zn	Fe	S	Ag	Au
Test BL0597-49 - VMS-1 - Regrind with Soda Ash																
Recalc. Feed	100.0	2000.0	3.7	0.26	2.66	30.6	38.5	63	4.1	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Cu/Pb Con	8.0	159.1	30.7	1.49	1.00	28.7	36.1	105	3.1	66.1	44.9	3.0	7.5	7.5	13.2	5.9
Zn Con	2.5	50.5	1.3	0.23	62.40	1.9	34.2	327	8.3	0.9	2.2	59.2	0.2	2.2	13.0	5.1
Zn Ro Tl	58.1	1162.0	0.5	0.08	0.16	42.2	51.3	40	4.7	8.0	17.6	3.5	80.1	77.4	36.5	65.4
Ro Tail	21.6	431.8	0.2	0.06	0.21	3.4	3.4	9	0.4	0.9	4.9	1.7	2.4	1.9	3.1	2.2
Test BL0597-50 - SMS-2 - Regrind with Soda Ash																
Recalc. Feed	100.0	1961.7	1.6	0.21	1.50	17.9	22.7	32	1.6	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Cu/Pb Con	4.9	96.3	25.5	2.49	7.60	24.2	37.4	186	8.9	76.6	57.0	24.9	6.6	8.1	28.5	27.9
Zn Con	1.1	22.4	2.3	0.29	58.40	3.1	34.3	238	3.0	1.6	1.5	44.6	0.2	1.7	8.5	2.2
Zn Ro Tl	33.6	659.5	0.3	0.10	0.09	37.2	45.4	22	1.6	6.4	15.7	2.0	69.7	67.3	23.1	33.8
Ro Tail	50.9	999.1	0.0	0.05	0.05	1.7	1.8	4	0.2	1.2	11.9	1.7	4.8	3.9	6.4	6.5
Test BL0597-51 - BX-4 - Breccia - Regrind with Soda Ash																
Recalc. Feed	100.0	1875.8	0.3	0.36	1.54	3.1	5.2	55	2.5	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Cu/Pb Con	2.5	46.5	8.4	11.20	15.00	20.9	38.2	1242	40.4	72.7	77.2	24.2	16.5	18.3	56.1	40.3
Zn Con	1.5	28.0	0.8	0.40	58.60	2.5	33.9	328	7.2	3.9	1.7	56.9	1.2	9.8	8.9	4.4
Zn Ro Tl	5.9	111.4	0.5	0.31	0.42	16.7	22.6	93	6.2	11.2	5.1	1.6	31.5	26.0	10.0	14.9
Ro Tail	87.5	1642.0	0.0	0.04	0.04	1.1	1.6	6	0.5	2.1	9.7	2.3	31.4	26.8	9.6	17.5
Test BL0597-53 - BX-4 - Breccia - Regrind with Soda Ash & SMBS, A3418 for Cu/Pb cleaners																
Cu/Pb Con	1.2	23.3	13.9	13.70	5.10	28.0	38.5	1590	55.6	58.0	48.3	3.9	9.7	9.3	34.8	30.5
Zn Con	1.5	28.4	0.7	0.75	62.00	2.8	35.1	324	6.3	3.5	3.2	58.3	1.2	10.3	8.6	4.2
Zn Ro Tl	10.6	201.7	0.3	0.13	0.12	10.0	11.5	38	1.9	10.5	4.0	0.8	29.8	24.0	7.2	8.8
Ro Tail	81.4	1543.3	0.0	0.04	0.04	1.0	1.3	5	0.4	1.1	9.3	2.0	23.1	21.2	7.2	14.9
Recalc. Feed	100.0	1895.2	0.3	0.35	1.59	3.6	5.1	56	2.2	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Test BL0597-52 - FALLA GR-5 - Grainstone - Regrind with Soda Ash																
Recalc. Feed	100.0	1864.5	0.8	0.16	1.14	6.8	9.8	36	2.3	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Cu/Pb Con	4.3	80.0	15.7	1.63	10.80	23.6	40.9	281	15.5	79.9	43.2	40.8	14.8	17.9	33.2	28.9
Zn Con	0.9	16.7	1.8	0.56	21.50	20.2	45.2	207	8.1	1.9	3.1	16.9	2.7	4.1	5.1	3.2
Zn Ro Tl	13.1	243.7	0.4	0.17	0.09	18.1	26.4	47	3.8	5.7	13.7	1.0	34.7	35.3	17.0	21.7
Ro Tail	75.4	1406.3	0.0	0.04	0.05	2.3	2.0	7	0.5	3.1	18.6	3.3	25.7	15.4	15.1	16.7
Test BL0597-54 - FALLA GR-5 - Grainstone - Regrind with Soda Ash & SMBS, A3418 for Cu/Pb cleaners																
Recalc. Feed	100.0	1891.5	0.9	0.18	1.20	8.2	9.7	39	2.2	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Cu/Pb Con	3.4	64.5	21.4	2.89	10.40	27.2	37.2	352	21.3	81.5	56.2	29.5	11.2	13.1	30.9	32.3
Zn Con	1.3	24.9	3.4	0.54	52.00	8.1	35.2	454	8.8	5.0	4.1	57.0	1.3	4.8	15.4	5.2
Zn Ro Tl	17.9	338.4	0.3	0.12	0.12	24.9	29.3	54	3.8	6.4	12.2	1.8	54.0	54.0	24.8	30.7
Ro Tail	73.9	1397.2	0.0	0.05	0.05	2.3	1.9	8	0.5	2.6	21.1	3.1	20.4	14.5	14.4	16.5

13.3.6 Locked Cycle Test Performance

Four (4) LCTs were performed: one test on Composite 1 (Sample Type 1), and a test each on VMS-1, SMS-2, and BX-4 lithological samples. Falla Gr-5 was not tested due to budget and timeline constraints.

Figure 13.7 displays an LCT flowsheet and Table 13.20 presents the tests conditions. A summary of the locked cycle test results can be found in Table 13.21.

Figure 13.7 – LCT Flowsheet Schematics

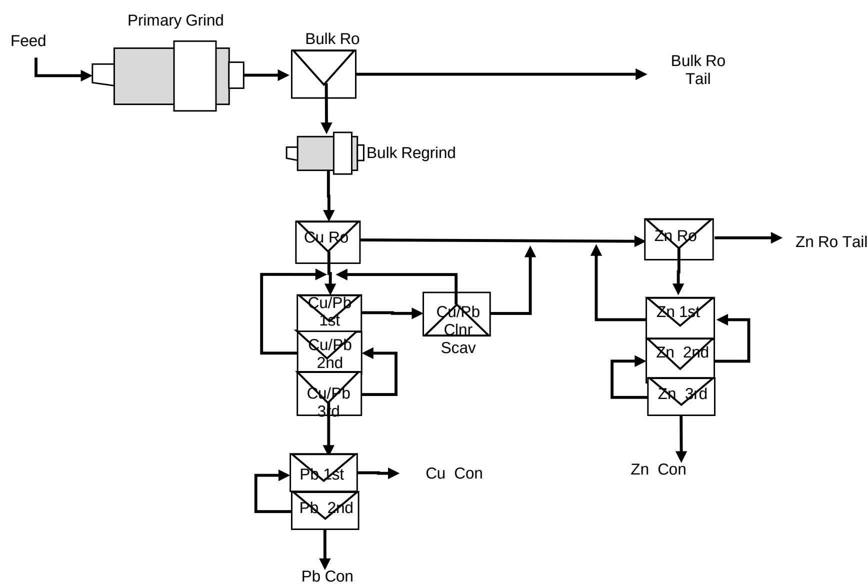


Table 13.20 – LCT Test Conditions

Stage	Reagents - g/tonne						K_{80}, µm	Float (min)	pH
	SMBS	SIPX	$CuSO_4$	NaCN	3418A	A241			
Bulk Ro		24					75-125	8	5.3-7.5
Bulk R/G	0-2000						13-16		6.1-6.8
Cu/Pb Ro	1475-2700	0-35			23	0-55		6-10	5.2-6.5
Cu/Pb Clnr	0-850	3			3-10			2-7	6.2-7.0
Pb Clnr				500	10			2	10-10.3
Zn Ro		15-20	1250-1500					5	10-10.5
Zn Clnr		0-5			15			2-4	10.5-11

Table 13.21 - LCT Results Summary

Product	Weight		Assay - percent or g/t							Distribution - percent						
	%	grams	Cu	Pb	Zn	Fe	S	Ag	Au	Cu	Pb	Zn	Fe	S	Ag	Au
Test BL0597-33 - Composite 1																
Feed	100.0	3807.4	1.58	0.23	2.61	12.1	16	34.7	2.4	100	100	100	100	100	100	100
Pb Con	0.5	19.2	22.7	15.8	5.4	21.5	32	396	16.7	7.2	34.3	1.0	0.9	1.0	5.8	3.6
Cu Con	4.1	156.0	27.4	0.9	5.0	28.4	36	212	15.3	71.0	15.0	7.8	9.6	9.0	25.0	26.5
Zn Con	3.9	150.2	1.8	0.4	54.8	5.3	35	233	11.4	4.5	6.1	82.8	1.7	8.4	26.4	19.0
Zn Ro Tail	23.3	886.8	0.72	0.22	0.5	35.8	43	38.7	3.7	10.6	22.4	4.7	68.7	61.8	26.0	36.0
Zn Ro Sc Tail	68.7	2614.4	0.16	0.08	0.14	3.39	4.7	8.51	0.52	6.7	22.2	3.7	19.2	19.8	16.8	14.9
Test BL0597-55 - VMS-1																
Feed	100.0	4047.6	4.11	0.28	2.58	32.2	39	61.7	4.4	100	100	100	100	100	100	100
Cu/Pb Con	11.4	462.4	30.2	1.0	1.9	29.2	36	140	7.7	84.0	39.7	8.5	10.3	10.5	25.9	19.9
Zn Con	4.5	182.8	5.7	1.8	48.3	8.1	35	383	12.8	6.2	28.7	84.8	1.1	4.1	28.0	13.1
Zn Ro Tail	63.4	2566.7	0.6	0.1	0.2	43.8	51	42	4.5	8.9	26.1	5.0	86.2	83.5	43.2	64.8
Bulk Ro Tail	20.6	835.7	0.16	0.07	0.2	3.6	4	8.7	0.5	0.8	5.5	1.7	2.3	1.9	2.9	2.2
Test BL0597-56 - SMS-2																
Feed	100.0	3921.7	1.61	0.20	1.51	17.8	23	32.6	1.5	100	100	100	100	100	100	100
Cu/Pb Con	8.9	349.9	15.8	1.5	7.9	30.2	42	190	6.9	87.4	65.8	46.3	15.2	16.3	51.9	42.2
Zn Con	2.2	87.4	3.7	0.5	34.2	18.6	41	210	6.8	5.1	5.1	50.4	2.3	3.9	14.3	10.3
Zn Ro Tail	38.2	1497.3	0.3	0.1	0.1	36.3	46	23	1.6	6.5	17.7	1.6	78.0	76.2	27.5	40.8
Bulk Ro Tail	50.7	1987.1	0.03	0.05	0.1	1.6	2	4.0	0.2	1.0	11.4	1.7	4.5	3.6	6.2	6.7
Test BL0597-57 - BX-4 - Breccia																
Feed	100.0	3806.1	0.28	0.34	1.51	3.3	5	50.2	2.0	85	46	97	99	98	59	68
Pb Con	0.4	16.7	9.7	41.6	10.7	9.7	25	4669	146.5	15.0	54.5	3.1	1.3	2.3	40.8	31.8
Cu Con	1.8	68.2	10.2	1.5	4.9	35.7	45	548	15.6	64.5	8.0	5.8	19.4	17.1	19.6	13.8
Zn Con	2.1	81.0	0.7	2.7	60.5	2.0	33	347	6.8	5.5	17.2	85.4	1.3	14.9	14.7	7.2
Zn Ro Tail	7.9	299.2	0.44	0.42	0.5	20.4	24	97.0	6.6	12.2	9.8	2.8	48.7	39.9	15.2	25.5
Bulk Ro Tail	88.2	3357.7	0.01	0.04	0.05	1.1	1	5.5	0.5	3	11	3	29	26	10	22

As seen from Table 13.21, for Composite 1, the overall metallurgical performance was similar to the tests performed in 2019. The notable difference was the separation of lead and copper, it was less efficient in this round of testing. There were adjustments needed to the reagent scheme to achieve this performance. These adjustments and slight change in performance were likely a result of sample aging (oxidation).

The remaining composites have varied lithologies, highly variable feed grade ratios and varied sulphide content.

Overall, the developed flowsheet responded well to the varied conditions. It should be noted that the flotation times and regrind times of the bulk circuit were altered to achieve high sulphide recovery and consistent bulk regrind discharge size.

The Type 2 samples had minimal reagent optimisation prior to conducting the LCTs. The metallurgical performance of the copper, lead and zinc was roughly proportional to the feed content of that metal in that sample. For example, the samples with higher copper feed grade (VMS-1 and SMS-1) had good metallurgical performance. The BX-4 sample, had a low copper feed grade and subsequently~~,~~ the poorest metallurgical performance. The same sample had good zinc and lead head grades and produced good grades and recoveries of lead and zinc concentrates.

Gold deportment to the concentrates was variable but followed a similar trend as the Sample Type 1 composites. Gold was 20 to 46 percent recovered to the copper/lead concentrates. An additional 7 to 13 percent of the gold was recovered to the zinc concentrate. For samples with high sulphide content, most of the remaining gold was contained in the zinc rougher tailings stream.

The BX-4 sample contains low sulphide levels (4.7 % sulphur in the feed) with gold losses that were nearly equally divided between zinc rougher and bulk rougher tailings.

13.3.7 Quality of the Concentrates

A series of assays were performed on concentrates from the locked cycle tests performed on VMS-1, SMS-2 and BX-4 samples. The concentrates from Cycle E were used for analysis. A summary of the assay data shown in Table 13.22.

Table 13.22 - Penalty Elements in 2020 LCT Concentrates

Sample	Test	Concentrate	As	Sb	Hg	Cd	Bi	Pb+Zn	Mn	Fe
			%	%	ppm	%	%	%	%	%
VMS-1	BL0597-55	Pb/Cu Con	0.09	0.03	5.37	76.00	< 2	2.88		
SMS2	BL0597-56	Cu/Pb Con	0.58	0.15	13.40	239.00	< 2	9.37		
BX-4	BL0597-57	Cu Con	0.49	0.06	16.10	146.00	13.00	6.41		
Smelter terms (Exen, 2020)			0.20	0.05	10	0.030	none	4.00	none	none
Smelter terms – (Adventus ST, 2021)			0.20	0.05	10	0.030	none	4.00	none	none
VMS-1	BL0597-55	Zn Con	0.30			1710.00			0.003	8.08
SMS-2	BL0597-56	Zn Con	0.64			1390.00			0.01	18.58
BX-4	BL0597-57	Zn Con	0.04			> 2000			0.01	2.02
Smelter terms (Exen, 2020)			none	none	none	0.30	none	none	0.50	8.00
Smelter terms – (Adventus ST, 2021)			none	none	none	0.30	none	none	none	none
BX-4	BL0597-57	Pb Con	1.20	1.14	42.70		< 2			
Smelter terms (Exen, 2020)			0.50	0.50	50	none	0.10	none	none	none
Smelter terms – (Adventus ST, 2021)			none	none	none	none	none	none	none	none

There were a number of elements that may attract smelter penalties: Hg, As for some concentrates, Sb for some concentrates, and Cd for some of the zinc concentrates. It should be noted that the concentrates represent a lithological variability material and not the concentrates produced from the composite samples which includes various rock types present in the ore.

13.3.8 CYANIDE LEACH AND SART

In attempt to improve an overall gold and silver recovery, cyanide leaching and SART processes were investigated on the zinc rougher tailings streams from the Type 2 LCTs.

Due to limited mass, a single cyanide leach test and a single SART test were performed. The conditions for the cyanide leach were selected as for the most successful leach tests completed in 2019:

- Pre oxidation;
- Leach residence time - 48 hours
- Leach solution grade of 5,000 ppm NaCN;
- pH of 10.5 to 11.

SART testing used a standard methodology where the sample is first leached for 48 hours, then the slurry undergoes 6 hours of carbon sorption, and a post sorption solution is treated through the SART conditions to precipitate copper and recover the cyanide (Figure 13.8).

Figure 13.8 - Leach Test Schematics

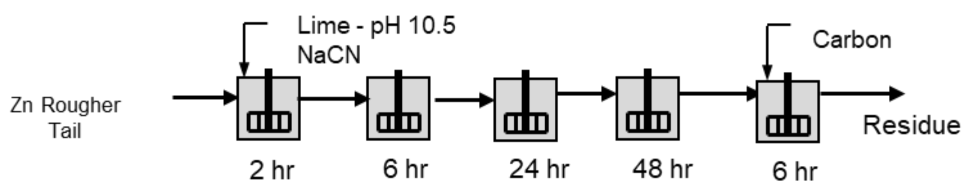


Table 13.23 summaries the Leach and SART test and Table 13.24 presents the overall cycle test performance.

Table 13.23 - Leach and SART Test Summary

Test		**T58**	**T59**	**T60**
Sample		VMS-1	SMS-2	BX-4
Cyanide Leach				
Consumption (kg/t)	NaCN	18.3	14.3	18
	Lime	11.6	12.2	4.7
Leach Recovery (%)	Au	41.6	24.1	26.4
	Ag	81.6	58.1	50.1
	Cu	84.7	86.0	75.7
	Zn	40.1	44.8	9.1
	Fe	0.03	0.2	0.1
Recovery Carbon (%)	Au	43.2	26.5	25.7
	Ag	69	48.4	40.4
	Cu	0.9	1	1.1
	Zn	1.9	3	0.4
Barren Solution Recovery (%)	Au	0.2	0.6	0.2
	Ag	2.2	4	1.5
	Cu	84.2	83.9	73.4
	Zn	39.2	39.9	8.9
SART				
Consumption (g/L)	NaSH	1.7	0.9	1.1
	H_2SO_4	9.4	7.3	7.2
	Lime	4.5	3.3	1.7
Precipitate Grade (% or g/t)	Ag	130.0	74	310
	Cu	53.0	54.6	27.2
	Zn	9.9	6.0	24.5
	Fe	2.4	3.0	3.4
Recovery (%)	Ag	99	99	96
	Cu	87	75	6
	Zn	97	85	50
	Au	0	0	0

Table 13.24 - Overall Cycle Test Performance

Stream	Distribution (%)			
	Cu	Zn	Ag	Au
VMS-1 - Test 55				
Cu/Pb Conc.	84	8.5	25.9	19.9
Zn Conc.	6.2	84.8	28	13.1
Carbon	0.1	0.1	29.8	28
SART Precip	6.6	1.9	0.9	0
SMS-2 - Test 56				
Cu/Pb Conc.	87.4	46.3	51.9	42.2
Zn Conc.	5.1	50.4	14.3	10.3
Carbon	0.1	0	13.3	10.8
SART Precip	4.1	0.6	1.1	0
BX-4 - Test 57				
Pb Conc.	15	3.1	40.8	31.8
Cu Conc.	64.5	5.8	19.6	13.8
Zn Conc.	5.5	85.4	14.7	7.2
Carbon	0.1	0	6.1	6.6
SART Precip.	0.5	0.1	0.2	0

The process had high usage of cyanide and lime. This is not unusual considering the presence of secondary copper minerals and relatively high levels of sulphide.

The SART process was performed. Standard stoichiometric ratios of sodium hydrosulphide, sulfuric acid and lime were added. There was insufficient solution to conduct optimisation of the reagents.

For the VMS-1 and SMS-2 the process demonstrated material recoveries of base and precious metals and marketable concentrates. A precipitate grading about 54% copper, 8% zinc and 3% iron was produced. The precipitate also contained about 100 g/t of silver. The SART process performed well with the recovery of copper from solution ranging between 75 and 87%. However, the SART process had limited success on the BX-4 sample where low copper precipitate recoveries.

Due to the high levels of secondary copper and subsequent cyanide consumption in the leach; the addition of a cyanide leach process could be costly. The additional recovery of copper and silver with SART combined with cyanide regeneration is required to make this process economical.

Further trade-off studies, completed by DRA in 2020, evaluated current costs and risks associated with implementation of the Leach and SART hydromet plant at the Curipamba site, have concluded that the health and safety, commissioning, and operational risks, and capital and operating expenditures of this plant likely outweight the potential benefits of the additional metals recovery. Future testing results, technology development, and/or metal price environment may change the viability of a SART hydromet plant at Curipamba.

13.3.9 Gravity Tests

Gravity tests done within 2019 BML program had been performed on tailings streams from the flotation circuits. The results were poor and in line with an exhaustive gold mineralogical studies.

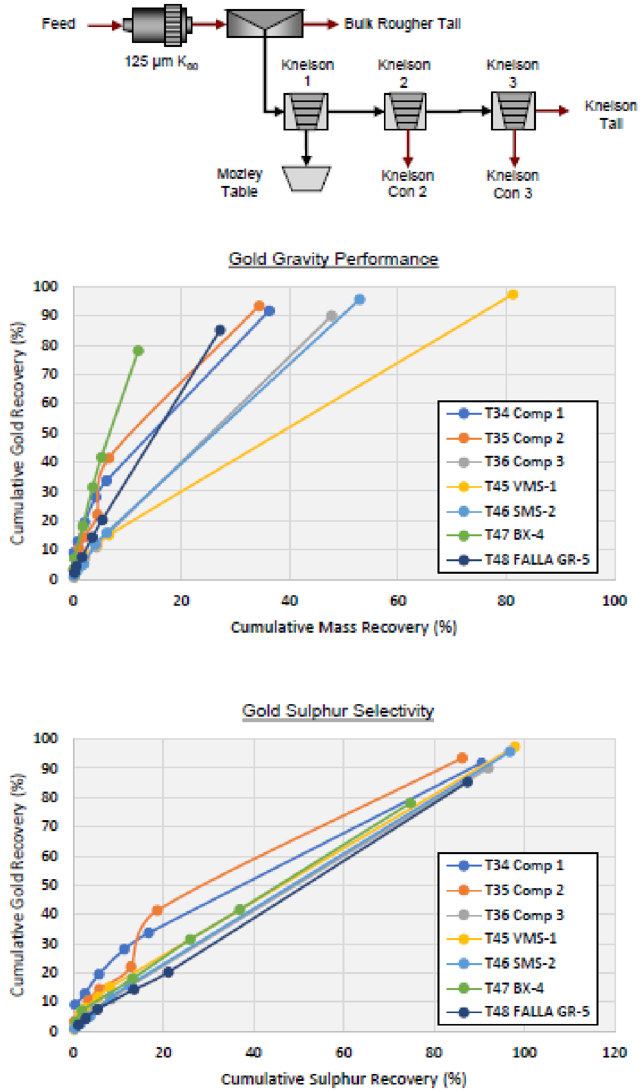
To take a fresh approach, in the 2020 program, samples were ground to a nominal 125µm K_{80} and the bulk rougher concentrate was sequentially passed three (3) times through a Knelson Concentrator. The first Knelson concentrate was further upgraded by Mozley table to determine the response when producing high grade gold concentrates.

Figure 13.9 displays the gravity response of the bulk concentrates. The top graph shows the cumulative performance, the far-left points on the curves are the Mozley concentrate, the far-right points on the curve are the bulk concentrate grade and recovery positions. The upper graph in the figure indicated that Knelson units recover 15 and 43 percent of the gold after three (3) passes. The concentrate mass was nominally 5%. Upgrading of the first Knelson concentrate, likely containing any free gold, had a low recovery level. Gold recovery to a gravity concentrate was less than 10% at relatively low gravity concentrate grade.

The second graph indicated a strong relationship between gold and sulphur recovery in the gravity concentrate. Only two samples indicated a slight deviation: Composite 1 and 2. This would indicate that some of the gold in the sample was not associated with sulphides.

The testing in this program on new samples indicated a similar trend as it was concluded in 2019: there is very little gravity recoverable gold in the samples tested.

Figure 13.9 – Bulk Rougher Gravity Performance



Source: BML

13.3.10 BML Settling Tests

A series of static cylinder settling tests were performed on the bulk rougher tailings. This regime was tested providing for possible incorporation of the hydromet facility into a concentrator and separation of the zinc rougher tailings from the total tailings stream.

One-litre cylinders were used for the tests. Cylinder raking was performed at 24 hours to determine the final densities. All samples had flocculent screening in advance to determine the best performing flocculant. The results are shown in Table 13.25.

The material, as expected, did not settle well, due to the inherent minerals present causing slow settling rates associated with high viscosity levels. Flocculant screening indicated that on average, anionic Magnafloc 336 produced the best settling and supernatant clarity.

Composite 2 and BX-4 (Breccia) demonstrated moderately high free settling rates and good supernatant clarity. The other samples settled relatively slowly.

Table 13.25 - Settling Test Results

Sample	Test	Flocculant		pH	Free Settling Rate	Final Density
		Type	Dosage		m/h	(% Solids)
Comp 1 - T34 Bulk Ro Tl	S1	336	20	7.3	0.04	30.6
	S2	336	40	7.3	0.08	35.9
	S3	336	60	7.3	0.08	36.3
	S4	336	25	10.3	0.19	37.3
	S5	336	40	10.40	0.21	37.1
Comp 2 - T35 Bulk Ro Tl	S6	336	20	7.6	0.46	37.8
	S7	336	40	7.6	4.14	39.6
	S8	336	60	7.6	3.71	39.9
	S9	336	25	10.5	0.08	33.9
	S10	336	40	10.50	0.16	35.3
Comp 3 - T36 Bulk Ro Tl	S11	336	20	7.5	0.17	34.1
	S12	336	40	7.5	0.65	36.5
	S13	336	60	7.5	1.79	36.2
	S14	336	25	10.5	0.32	34.2
	S15	336	40	10.5	0.29	35.2
SMS-2- T46 Bulk Ro Tl	S16	336	20	6.6	0.21	35.6
	S17	336	40	6.6	0.61	31.7
	S18	336	60	6.6	0.73	33.8
	S19	336	25	10.5	0.25	34.8
	S20	336	40	10.5	0.59	36.0

Sample	Test	Flocculant		pH	Free Settling Rate	Final Density
		Type	Dosage		m/h	(% Solids)
BX-04- T47 Bulk Ro Tl	S21	336	20	7.5	0.63	41.1
	S22	336	40	7.5	4.10	43.3
	S23	336	60	7.5	7.66	40.9
	S24	336	25	10.5	0.36	39.4
	S25	336	40	10.5	0.50	39.7
Falla Gr-5 - T48 Bulk Ro Tl	S26	336	20	5.9	0.36	37.0
	S27	336	40	5.9	0.35	36.6
	S28	336	60	5.9	0.56	36.3
	S29	336	25	10.5	0.62	35.5
	S30	336	40	10.5	1.32	35.6
SMS-1 - T48 Bulk Ro Tl	S31	336	40	6.9	0.04	28.0
	S32	336	60	6.9	0.05	27.5
	S33	336	60	10.5	0.27	28.7
T46 Fines	S34	336	60	7.2	0.02	18.7
	S35	336	60	10.5	0.04	18.3
T47 Fines	S36	336	60	7.4	0.01	21.1
	S37	336	60	10.5	0.02	21.4
T48 Fines	S38	336	60	5.9	0.03	22.1
	S39	336	60	10.5	0.04	21.7

The addition of the lime did not improve settling performance, and in some cases was detrimental.

The final density, after 24 hours and raking ranged from 29 to 41%.

Removing fines (desliming) and settling the slimes separately was investigated. The fines from select streams were tested and indicated very slow free settling and very low final density.

13.4 Base Metallurgical Laboratories Inc - 2021

The 2021 testing campaign, conducted by the BML, has been a continuation of the work completed during 2018-2020 and targeted optimisation and improvements of the metallurgical performance of the developed flowsheet.

The goals of this program were:

- Conduct variability assessment of the process across a wide spectrum of samples;
- Generate additional engineering data for comminution and dewatering;
- Conduct further optimization of the separation process, specifically with respect to the copper lead circuit;
- Assess the effects of stockpile aging on metallurgical performance;
- Conduct detailed mineralogical assessment of the locked cycle test products.

13.4.1 Sample Selection

The program developed for the feasibility study considered three composites reflecting the life of mine, and fifteen lithology based variability samples.

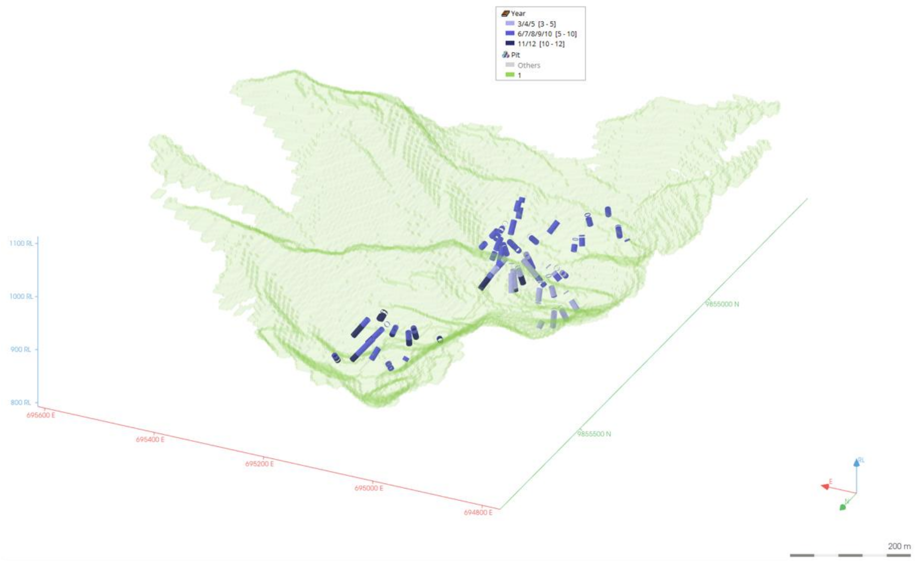
These samples were produced out of the drilling campaign specifically designed for the feasibility study work, and involved 42 drillholes across the mineable material of the deposit. A total weight of the ore subjected for metallurgical tests was more than 5,000 kg.

The composites were constructed based on the life of mine mining schedule:

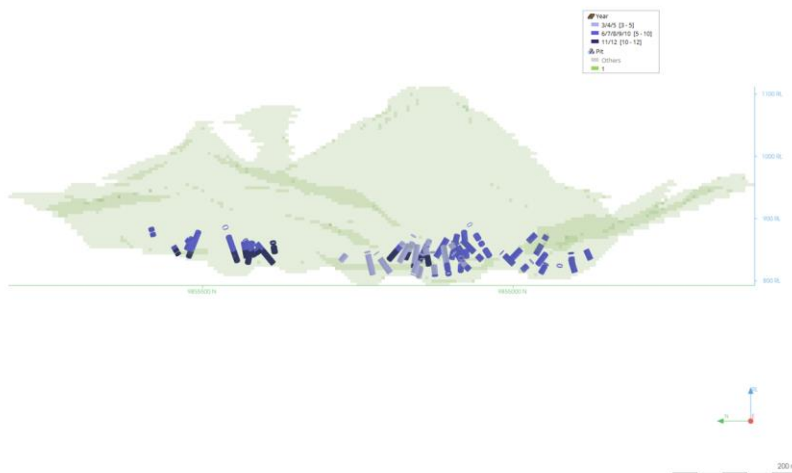
- Composite 1 - Years 3 through 5 of mine operation including ore scheduled for pre-production Year 1 and 2;
- Composite 2 - Years 6 through 10; and
- Composite 3 - Years 10 and 11.

Location of the drill holes within the pit shell and years of production depicted in Figure 13.10.

Figure 13.10 - FS Test Program Drill Holes



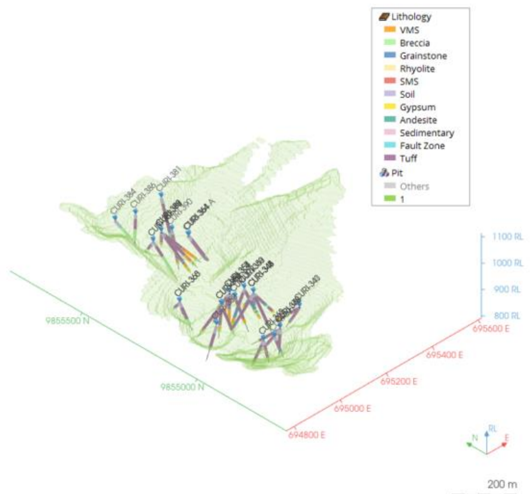
Isometric



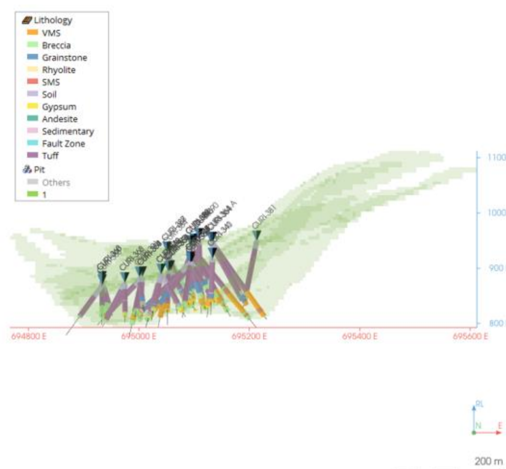
Side View

The view of drill holes with indication of the lithological rock types can be found in Figure 13.11.

Figure 13.11 - FS Test Program Drill Holes by Lithology



Isometric



Section View

Figure 13.10 and Figure 13.11 have been generated from the geometallurgical modelling software in order to validate sample representativity and assay reconciliation. The pit shell outline and drillholes coordinates, rock types, and mine production sequence has been synchronized with the mine schedule developed by the DRA mining specialists.

Information provided in Figure 13.10 and Figure 13.11 demonstrate that the material sampled for the FS testing program sufficiently covers the minable ore within the pit shell, and reflects the rock types within the deposit.

Sample representativity, and reconciliation chart can be found in Figure 13.12.

Figure 13.12 – Sample Representativity and Reconciliation Chart

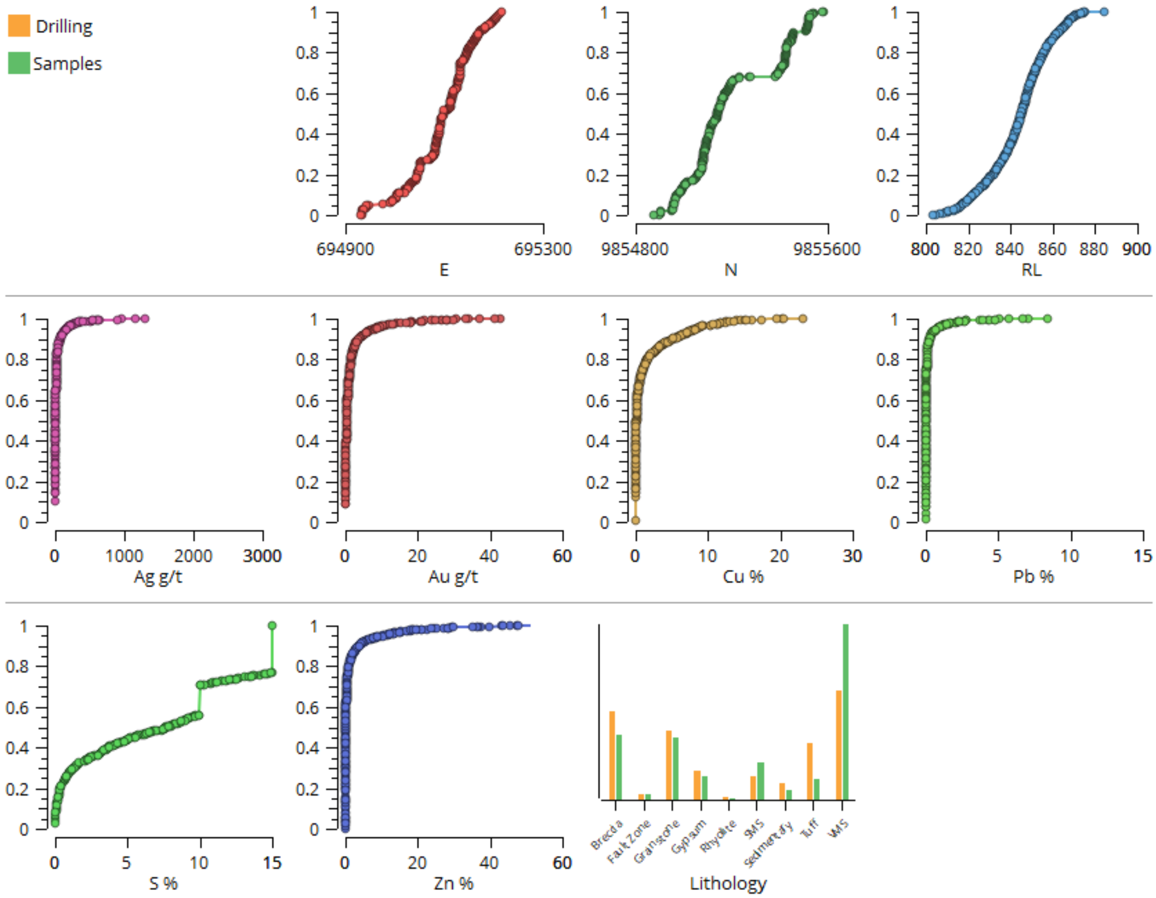


Figure 13.12 demonstrates that the metallurgical samples generated during the drilling program sufficiently cover the grade intervals for the ore principal elements.

A representativity report was created from the Cancha geometallurgy software outputs which assesses the representativity of samples selected. A focus case was applied to filter drill cores within the mine. The samples shown are composed of the main metallurgical composites for the metallurgical test work program. Principal features represented are grade (numerical) and lithology (categorical).

The below charts are cumulative probability plots of metal grades. The y-axis is the probability of each value. The x-axis is the value range (grade in %). The solid line represents drilling, solid dots represent samples. In the case of Cu, approx. 95% of drilling and samples are less than 8% Cu.

Samples selected are representative. Samples are evenly distributed through the curve. Selections have continuous and normally distributed values. Spatial (E, N, RL) coverage throughout the deposit is complete (refer to the first three plots).

The last plot is a cumulative histogram of lithologies, comparing the focus case (lithology) distribution to the sample population distribution. Selection bias is minimal. The main ore bearing lithology, volcanic massive sulphide (VMS), as major metal containing lithologies has been looked at in greater details than the low metal value lithology types.

Based on the above, the composites and variability samples generated for the FS met testing campaign should be considered as representative of the ore body.

13.4.2 Assays, Lithology, Mineral Composition

13.4.2.1 Composites

The major assays for the composites are shown in Table 13.26.

Table 13.26 - Composites Assay

Sample	Assay - percent or g/t						
	Cu	Pb	Zn	Fe	S	Ag	Au
Composite 1	1.6	0.4	4.3	8.0	12	62	3.5
Composite 2	1.9	0.3	2.6	14.3	19.8	40	2.7
Composite 3	3.8	0.2	1.9	22.1	29.3	20	1.9

Composites lithological composition can be seen in Table 13.27.

Table 13.27 - Composites Lithology

Composite		Composite 1	Composite 2	Composite 3
From, m depth		25	25	59
To, m depth		122	141	152
Lithology			%	
Code	Description			
B	Breccia	0.0	3.0	0.0
BxH	Hydrothermal Breccia	26.3	9.6	11.4
DaBx	Dacite Breccia	9.4	2.6	0.0
Gr	Grainstone	13.4	15.4	14.1

Composite		Composite 1	Composite 2	Composite 3
Gy	Gypsum	1.0	10.0	11.4
LapT	Lapilli Tuff	1.0	2.5	0.0
Rhy	Rhyolite	0.0	0.9	0.0
Sed	Sedimentary	5.3	1.8	0.0
SMS	Semi Massive Sulphide	9.9	7.6	5.2
T	Tuff	1.1	0.6	0.0
TRhy	Rhyolite Tuff	1.1	2.7	0.0
VMS	Volcanogenic Massive Sulphide	30.2	42.4	58.0
ZnFlt	Fault zone (an area within the deposit)	1.1	0.8	0.0
Total		**100**	**100**	**100**

Composites mineral content is in Table 13.28.

Table 13.28 - Composites Mineralogy Summary

Mineral (%)	Composite 1	Composite 2	Composite 3
Chalcopyrite	4.25	4.92	10.98
Bornite	0.06	0.13	0.08
Chalcocite	0.01	0.02	0.02
Enargite/Tennantite	0.14	0.11	0.07
Tetrahedrite	0.08	0.05	0.09
Malachite/Azurite	0.00002	0.00000	0.0002
Galena	0.51	0.37	0.34
Sphalerite	6.32	3.94	3.05
Pyrite	16.2	28.1	46.3
Pyrrhotite	0.05	0.03	0.09
Arsenopyrite	0.001	0.001	0.001
Other Sulphides	0.0006	0.0008	0.0026
Total Sulphides	27.60	37.70	61.02
Iron Oxides	0.28	0.43	0.41
Quartz	23.2	14.0	9.5
Muscovite/Illite	24.3	21.6	14.8
Other Non-sulphides	24.60	26.22	14.34
Total	**100.00**	**100.00**	**100.00**

Note: numbers may not add due to rounding

Composites minerals liberation summary is depicted in Table 13.29.

Table 13.29 - Composites Mineral Liberation Summary

Mineral Status (%)	Composite 1						Composite 2						Composite 3					
	Ac	Cs	Ga	Sp	Py	Gn	Ac	Cs	Ga	Sp	Py	Gn	Ac	Cs	Ga	Sp	Py	Gn
Liberated	18.8	38.2	26.3	51.6	56.9	90.5	8.4	44.6	26.3	53.0	68.5	89.0	35.8	50.8	42.2	57.0	73.0	83.6
Binary - Ac		0.0	0.0	0.0	0.0	0.0		0.0	0.1	0.0	0.0	0.0		0.0	0.3	0.0	0.0	0.0
Binary - Cs	9.0		4.0	12.1	11.9	1.2	14.6		6.4	12.1	8.4	1.1	6.5		5.3	14.0	14.7	1.9
Binary - Ga	3.4	0.5		2.3	0.2	0.1	3.8	0.4		2.2	0.1	0.1	3.7	0.1		1.5	0.0	0.0
Binary - Sp	5.7	11.5	18.6		2.0	1.5	6.9	7.1	19.8		1.0	0.8	3.0	2.9	12.7		0.4	0.8
Binary - Py	3.0	24.0	3.6	5.2		5.0	14.4	27.6	4.5	7.1		7.4	4.8	36.9	5.5	7.0		11.2
Binary - Gn	24.9	10.5	9.4	13.7	22.6		25.0	8.4	8.7	10.3	18.1		28.6	4.4	7.0	9.4	10.0	
Multiphase	35.2	15.2	38.1	15.1	6.5	1.7	26.8	12.0	34.2	15.2	3.9	1.6	17.6	4.9	26.9	11.2	1.9	2.4
Total	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**

Legend:
Ac-Acanthite/Argentite including Gold, Hessite, Polybasite and Freibergite;
Cs-Copper Sulphides including Chalcopyrite, Bornite, Chalcocite/Covellite, Tennantite/Enargite and Tetrahedrite/;
Ga-Galena;
Sp-Sphalerite;
Py-Pyrite/Marcasite including Pyrrhotite and Arsenopyrite;
Gn - Non-Sulphide Gangue including Iron Oxides.

As seen from Table 13.29, the sphalerite has a best liberation followed by the copper sulphides. Galena liberation is low, and substantial quantity of it locked within the multimineral aggregates. This explains the challenges in the production of high grade lead concentrates and corresponds with the previously completed mineralogy studies

13.4.2.2 Variability Samples

Variability samples assays can be found in Table 13.30.

Table 13.30 - Variability Samples Assays

Sample	Lithology	Element, % or g/t						
		Cu	Pb	Zn	Fe	S	Ag	Au
V1	VMS	2.70	0.90	4.35	20.5	29.4	123	4.49
V5		2.58	0.08	0.75	28.2	37.5	14.5	1.74
V13		2.00	0.16	3.35	27.0	36.7	28.4	2.92
V15		4.24	0.53	5.40	24.7	33.2	114.5	5.27
V9	SMS	2.54	0.24	6.60	11.1	20.2	51.6	2.23
V12		0.95	0.10	0.35	13.4	17.5	12.0	0.95
V14		2.36	0.53	3.50	14.4	20.3	104.1	4.02
V2	Grainstone	0.87	0.16	1.14	5.80	3.51	34	1.89
V7		0.45	0.10	0.96	6.05	3.68	19.6	0.85
V4	Breccia	0.16	0.04	0.51	7.15	7.82	12	0.22
V8		0.12	0.06	0.28	3.54	4.65	6.3	0.23
V3	Tuff	0.14	0.11	0.49	4.20	1.75	59	0.36
V6		0.022	0.04	0.23	3.61	0.90	3.2	0.27
V11	Gypsum	0.23	<0.01	0.25	8.25	19.3	3.2	0.29
V10	Sedimentary	0.29	0.10	0.72	5.65	4.64	15.8	0.84

Variability samples mineral content can be seen in Table 13.31

Table 13.31 - Variability Samples Mineralogy Summary

Lithology	**VMS**				**SMS**			**Grainstone**		**Tuff**		**Breccia**		**Sedimentary**	**Gypsum**
Sample	**V1**	**V5**	**V13**	**V15**	**V9**	**V12**	**V14**	**V2**	**V7**	**V3**	**V6**	**V4**	**V8**	**V10**	**V11**
Chalcopyrite	5.82	6.61	5.45	10.10	6.76	2.19	6.15	2.34	1.22	0.34	0.08	0.45	0.36	0.72	1.08
Bornite	0.88	0.35	0.00	0.00	0.35	0.00	0.00	0.11	0.00	0.00	0.00	0.04	0.00	0.00	0.00
Tetrahedrite	0.54	0.17	0.40	0.38	0.23	0.22	0.59	0.08	0.04	0.00	0.00	0.06	0.03	0.08	0.04
Galena	1.31	0.21	0.27	0.75	0.39	0.18	0.72	0.17	0.17	0.10	0.03	0.05	0.04	0.12	0.09
Sphalerite	6.95	1.50	5.50	8.65	11.60	0.51	5.71	2.01	1.86	0.66	0.33	0.94	0.43	1.28	0.62
Pyrite	48.9	65.5	60.4	52.3	23.3	32.2	33.0	3.8	5.3	2.8	1.2	14.3	10.6	9.0	26.8
Total Sulphides	**64.40**	**74.34**	**72.02**	**72.18**	**42.63**	**35.30**	**46.17**	**8.51**	**8.54**	**3.94**	**1.63**	**15.84**	**11.46**	**11.19**	**28.63**
Iron Oxides	0.08	0.07	0.06	0.06	0.08	0.05	0.03	1.17	0.91	1.51	1.36	0.17	0.09	0.24	0.08
Quartz	7.83	2.30	5.52	9.03	17.10	15.40	19.50	12.10	12.90	16.90	23.80	42.70	43.30	12.80	16.10
Muscovite	7.38	14.80	14.40	7.91	15.60	31.60	8.06	3.23	3.08	5.59	3.42	16.00	13.30	11.70	17.20
Clays	4.28	1.66	1.59	3.42	5.94	6.29	10.30	9.81	11.10	12.50	11.50	9.08	13.30	23.10	7.21
Gypsum	0.03	0.03	0.11	0.01	1.97	0.16	0.00	0.00	0.00	0.00	0.00	0.00	0.18	0.00	18.50
Other Non-sulphides	15.99	6.82	6.34	7.31	16.72	11.10	15.96	65.22	63.35	59.50	58.30	16.17	18.41	41.04	12.35
Total	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100

Note: numbers may not add due to rounding

As seen from Table 13.31, VMS, SMS and Grainstone material bear most of the metal value. Pyrite is a major sulphur deportment. Secondary copper minerals can represent from 3 to 9% in average of total copper in VMS, SMS, and Grainstone samples with one VMS sample (V1) that has 20% of total copper as bornite and tetrahedrite. This copper deportment explains challenges in copper-zinc selectivity during the copper flotation. Tuff, Breccia, Sedimentary, and Gypsum rock types are of low metal value with the Gypsum sample being the lowest. Metals and sulphur distribution in these samples generally follows the pattern for SMS, VMS, and Grainstone material.

13.4.2.3 Flotation Products Mineralogy

Mineralogy and liberation studies of the flotation products (concentrates and tailings) returned valuable observations and are summarised below. It should be noted that concentrates represent the material reground to 13-16 microns (D_{80}).

Mineral content of copper and zinc concentrates produced from LCTs at optimised conditions can be found in Table 13.32. Lead concentrates were not studied due to the very low quantity available.

Table 13.32 - Copper and Zinc Concentrates Mineral Content Summary

Test		**LCT BL743-70**	**LCT BL743-71**	**LCT BL743-70**	**LCT BL743-71**
Feed		Composite 2	Composite 1	Composite 2	Composite 1
Concentrate		Copper	Copper	Zinc	Zinc
Cu	Chalcopyrite	79.1	60.1	7.39	8.43
	Bornite	1.42	0.48	0.08	0.04
	Chalcocite/Covellite	0.07	0.15	0.04	0.04
	Tennantite/Enargite	0.34	2.25	0.17	0.22
	Tetrahedrite	0.32	1.39	0.15	0.12
	Malachite/Azurite	0.00	0.00	0.00	0.00
Pb	Galena	2.70	4.64	0.93	0.79
Zn	Sphalerite	2.81	14.6	84.3	83.8
Fe	Pyrite	10.0	12.7	3.23	2.01
	Pyrrhotite	0.02	0.01	0.01	0.00
	Arsenopyrite	0.01	0.03	0.01	0.00
Gangue		3.15	3.57	3.69	4.57
Total		**100.0**	**100.0**	**100.0**	**100.0**

As seen from Table 13.32, copper secondary minerals are present in copper concentrates which explains the sphalerite presence. Similarly, chalcopyrite and other copper minerals present in zinc concentrate. Pyrite is present in copper concentrates in considerable quantity and this way carries the precious metals locked within it.

Results of the concentrates mineral liberation studies can be found in Table 13.33.

Table 13.33 - LCT Concentrates Mineral Liberation Summary

Copper Concentrate												
Mineral Status	**LCT BL743-70, Composite 2**						**LCT BL743-71, Composite 1**					
	Ac	**Cs**	**Ga**	**Sp**	**Py**	**Gn**	**Ac**	**Cs**	**Ga**	**Sp**	**Py**	**Gn**
Liberated	38.8	93.8	38.9	49.3	55.0	67.3	29.3	87.6	55.1	63.1	57.0	62.2
Binary - Ac		0.0	0.0	0.0	0.0	0.0		0.0	0.0	0.0	0.0	0.0
Binary - Cs	43.8		41.0	40.8	39.4	22.3	39.5		16.5	23.7	31.2	14.8
Binary - Ga	0.0	0.3		0.7	0.3	0.4	2.5	1.0		4.0	1.3	3.0
Binary - Sp	0.8	1.3	1.3		0.7	0.6	2.1	4.9	11.6		3.7	4.7
Binary - Py	2.0	3.6	2.8	1.8		3.1	3.7	4.6	2.5	2.3		4.1
Binary - Gn	0.0	0.5	1.3	0.6	1.2		1.7	0.4	1.2	0.6	0.8	
Multiphase	14.6	0.4	14.8	6.9	3.4	6.4	21.3	1.4	13.1	6.3	5.9	11.2
Total	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**
Zinc Concentrate												
Mineral Status	**LCT BL743-70, Composite 2**						**LCT BL743-71, Composite 1**					
	Ac	**Cs**	**Ga**	**Sp**	**Py**	**Gn**	**Ac**	**Cs**	**Ga**	**Sp**	**Py**	**Gn**
Liberated	40.8	57.9	35.2	92.2	47.8	58.8	59.9	67.0	52.0	93.9	60.7	55.4
Binary - Ac		0.0	0.0	0.0	0.0	0.0		0.0	0.1	0.0	0.0	0.2
Binary - Cs	3.6		2.8	3.2	5.6	3.5	0.0		2.4	2.9	6.0	1.9
Binary - Ga	0.0	0.2		0.3	0.3	0.4	0.7	0.2		0.2	0.2	0.3
Binary - Sp	29.5	31.7	41.3		36.1	26.2	14.3	26.9	28.3		26.1	33.5
Binary - Py	0.6	3.2	1.6	2.7		2.5	0.2	2.1	1.4	1.5		1.7
Binary - Gn	0.3	1.2	2.2	0.9	1.2		2.9	0.9	3.5	1.0	1.3	
Multiphase	25.1	5.9	16.8	0.7	9.1	8.6	22.0	2.9	12.3	0.4	5.6	7.0
Total	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**

Legend:
Ac-Acanthite/Argentite including Gold, Hessite, Polybasite and Freibergite;
Cs-Copper Sulphides including Chalcopyrite, Bornite, Chalcocite/Covellite, Tennantite/Enargite and Tetrahedrite/;
Ga-Galena;
Sp-Sphalerite;
Py-Pyrite/Marcasite including Pyrrhotite and Arsenopyrite;
Gn - Non-Sulphide Gangue including Iron Oxides.

As seen from Table 13.33, copper concentrates have well liberated copper sulphides which explains that high Cu grades are achievable, at the same time the concentres have sphalerite interlocked with copper minerals. This sphalerite is responsible for elevated Zn grades in copper concentrate. Similarly, galena interlocked with copper sulphides can be observed in the copper concentrates.

Zinc concentrates demonstrate similar picture: well liberated sphalerite (in excess of 90%), and copper sulphides interlocked with the sphalerite.

Galena interlocked with copper sulphides can be well noted in both copper and zinc concentrates. This explains the challenges in lead separation during the lead cleaning stage. Tailings mineral composition summary can be found in Table 13.34.

Table 13.34 - Tailings Mineral Content Summary

Test	BL743-70	BL743-71	BL743-70	BL743-71
Feed	Composite 2	Composite 1	Composite 2	Composite 1
Product	Bulk Rougher Tail		Zinc Rougher Tail	
Chalcopyrite	0.32	0.16	0.71	1.26
Bornite	0.003	0.003	0.017	0.034
Chalcocite/Covellite	0.002	0.003	0.036	0.091
Tennantite/Enargite	0.004	0.003	0.270	0.317
Tetrahedrite	0.002	0.002	0.121	0.131
Malachite/Azurite	0.0002	0.0001	0.0000	0.0000
Galena	0.069	0.058	0.175	0.270
Sphalerite	0.18	0.15	0.34	1.21
Pyrite	2.71	1.41	73.90	60.28
Pyrrhotite	0.006	0.006	0.20	0.05
Arsenopyrite	0.0004	0.0005	0.007	0.004
Iron Oxides	0.49	0.28	0.59	0.46
Quartz	18.8	30.1	10.7	14.8
Muscovite/Illite	37.3	35.3	6.37	9.16
K-Feldspars	6.11	10.6	0.37	0.85
Chlorite	14.5	6.80	1.64	1.33
Plagioclase Feldspar	8.90	4.77	0.38	0.70
Barite	3.77	5.33	2.85	7.46
'Kaolinite' (clay)	2.97	2.17	0.35	0.65
Other Sulphides	0.001	0.0001	0.004	0.003
Other Gangue Minerals	3.93	2.85	0.93	0.93
Total	**100.00**	**100.00**	**100.00**	**100.00**

Note: numbers may not add due to rounding

As seen from Table 13.34, high content of fine grained aluminosilicates (Muscovite/Illite, Chlorite) explains poor settling properties of the bulk rougher tailings observed during the 2020 campaign. The zinc rougher tailings mostly represented with the pyrite, and when mixed with the bulk rougher tailings would improve settling properties of the combined final tailings of the concentrator.

13.4.2.4 Gold and Silver Mineralogy

Gold and Silver mineralogy of zinc rougher tailings from the VMS and SMS variability samples were studied to evaluate the form and quantity of the precious metals. A summary of the gold deportment can be found in Table 13.35.

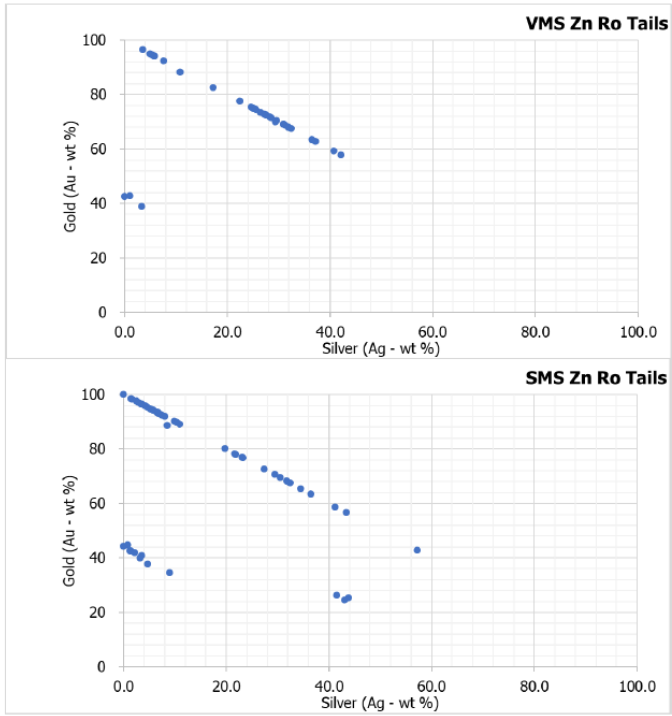
Table 13.35 – Au Deportment in Flotation Tailings Summary

Sample	VMS Zinc Rougher Tails		SMS Zinc Rougher Tails	
Gold Assayed in Sample (g/t)	1.91		1.80	
Forms and Carriers of Gold	Au /(g/t)	% in Sample Assay	Au /(g/t)	% in Sample Assay
Visible Gold	0.18	9.4	0.21	11.7
Sub-Microscopic (refractory) gold				
Pyrite	1.73	90.6	1.59	88.3
Subtotal sub-microscopic gold	1.73	90.6	1.59	88.3
Total	**1.91**	**100**	**1.8**	**100**

The majority of gold in both the VMS and SMS zinc rougher tails is in the form of refractory sub-microscopic gold contained in pyrite with 9.4% and 11.7% of the gold as free visible gold in the VMS and SMS samples respectively.

Both samples also contain a significant amount of both gold, silver and tellurium in the form of electrum. The gold and silver compositional ratios are shown Figure 13.13 and indicate that the majority of gold present is in the form of Au-Ag electrum. There is also quantities of electrum containing tellurium (Te).

Figure 13.13 – Gold and Silver Compositional Ratios for VMS and SMS Rougher Tails



13.4.3 Comminution Tests

Comminution testing included SMC tests on three (3) composites and variability samples covering all lithologies, and Bond testing (BWi, RWi, Ai) on two composites, and variability samples. Bond low energy impact work index testing (CWi) has not been performed as this test typically has low accuracy, instead, the Mic values were determined during the SMC tests. The CWi value can be calculated from the Mic and then used for the purpose of the crushing equipment sizing.

For SMC tests samples were screened and representative rocks ranging between 19 mm, and 22.4 mm were selected for the SMC tests.

To keep consistency with previous testing (2019 and 2020 test programs), the Bond ball mill index (BWi) determinations were completed with a closing screen size of 106 micron, resulting in a P_{80} size of nominally 70 to 84 microns.

Bond rod mill work index (RWi) values were determined as per the standard methodology.

Weighted average values were calculated based on the percentage of the rock types within the deposit determined during the drilling campaign and aligned with the life of mine mining schedule.

Comminution tests summary can be found in Table 13.36.

Table 13.36 – Comminution Tests Summary

Sample	Lithology	DWi	Mia	Mih	Mic	A x b	Ta	SCSE	Ai	RWi	BWi
		kWh/m^3	kWh/t	kWh/t	kWh/t			kWh/t		kWh/t	kWh/t
V1	VMS	2.89	7.4	4.6	2.4	126	0.9	6.39		10.2	13.5
V2	Grainstone	5.48	18	12.8	6.6	46	0.47	9.11		16.1	21.4
V3	Tuff	5.21	17.1	12	6.2	48	0.5	8.91		14.8	20.7
V4	Breccia	4.81	15.7	10.9	5.6	53	0.54	8.57		14.4	17
V5	VMS	2.08	5.6	3.3	1.7	176	1.24	5.65	0.078	5.5	10.4
V6	Tuff	4.38	15.4	10.4	5.4	56	0.59	8.42	0.042	15.4	18.7
V7	Grainstone	6.41	19.2	14.1	7.3	41	0.4	9.67	0.052	17.3	20.3
V8	Breccia	5.19	16.3	11.4	5.9	51	0.5	8.79	0.252	13.5	17.3
V9	SMS	4.1	10.9	7.3	3.8	79	0.63	7.91	0.043	10.8	11.2
V10	Sedimentary	6.58	20	14.7	7.6	39	0.39	9.79		17.1	18.8
V11	Gypsum	1.99	7.8	4.5	2.4	128	1.3	6.35	0.003	8.4	8.8
V12	SMS	2.85	8.9	5.5	2.9	105	0.91	6.92	0.124	11.6	13.7
V13	VMS	1.72	5	2.8	1.4	208	1.51	5.41		6.3	10.1
Weighted average		3.86	12.01	8.13	4.21	102	0.83	7.55	0.093	11.68	15.04
Composite 1		2.8	8	5	2.6	117	0.92	6.74	0.083	11.3	14.9
Composite 2		1.58	5.8	3.2	1.7	181	1.64	5.74	0.061	9.6	14.4
Composite 3		2.03	6	3.5	1.8	166	1.28	5.92			

Note:
Important to note that variability samples comminution response represents the rock types within the deposit and not the entire ore body. Comminution properties of the ore would be a weighted average of the lithological ore types within the deposit.

As seen from the table, competency and hardness of the variability samples vary considerably from low/soft to high/hard. Gypsum, VMS, and SMS rock types demonstrate the low values compared to other rock types tested. Weighted average estimated from variability samples, and results of the composite tests suggest that the ore overall is of low competency with A x B value in range from 102 to 181, and medium hard with BWi ranging from 14.4 to 15.04 kWh/t. Abrasion index (Ai) value suggests that the ore is of low abrasion.

13.4.4 BULK FLOTATION TESTS

Bulk rougher flotation tests were performed on all fifteen variability samples using a primary grind size of 125 micron K_{80}. The results are displayed in Table 13.37. The tests used the developed process: natural pH and sodium isopropyl xanthate (SIPX 35 g/t). Copper sulphate was added for the final rougher flotation stage to ensure all sphalerite was recovered to the bulk concentrate.

Table 13.37 - Variability Bulk Rougher Performance Summary

Test	Sample	Lithology	K80	Weight	Assay - percent or g/t							Distribution - percent						
				%	Cu	Pb	Zn	Fe	S	Ag	Au	Cu	Pb	Zn	Fe	S	Ag	Au
BL0743-01	V1		75	67	4.22	1.45	6.19	30.04	41.67	170.8	6.8	98.95	97.35	99.21	96.82	95.14	96.38	96.24
BL0743-02	V1		125	69	4.05	1.36	5.93	29.03	41.27	177.7	6.6	99.13	97.44	99.25	97.96	95.98	97.08	96.34
BL0743-31	V13	VMS	125	73	2.66	0.21	4.46	38.14	47.95	38.8	4.2	99.50	96.54	99.67	97.41	97.59	96.47	97.41
BL0743-21	V5		75	70	3.76	0.10	1.08	37.77	51.11	31.9	2.5	96.97	88.79	96.62	85.04	94.17	92.67	91.38
BL0743-49	V15		125	72	5.52	0.77	7.28	32.07	43.56	220.3	6.1	99.04	97.11	99.58	97.19	97.30	99.09	97.15
BL0743-23	V9		75	35	6.95	0.62	19.01	20.97	36.01	141.3	6.7	94.87	89.12	94.40	60.66	60.92	84.32	80.57
BL0743-30	V12	SMS	125	39	2.26	15.02	0.84	35.22	41.00	23.5	2.2	97.57	99.95	96.41	99.25	93.48	88.79	89.28
BL0743-48	V14		125	50	4.56	1.00	7.20	27.63	39.65	178.1	7.3	98.03	94.32	98.49	93.60	92.32	95.32	92.71
BL0743-03	V2		75	28	3.01	0.53	4.08	10.22	11.04	114.0	6.5	97.56	91.38	94.21	50.08	89.88	93.80	96.28
BL0743-04	V2	Grainstone	125	20	4.42	0.75	6.33	13.18	16.24	139.5	9.2	97.64	90.09	96.85	42.38	90.47	82.90	94.92
BL0743-22	V7		75	10	4.54	0.79	9.73	21.12	30.44	208.1	8.7	91.30	80.71	91.99	32.80	80.06	90.95	87.71
BL0743-05	V3	Tuff	75	17	0.77	0.59	2.77	8.99	9.14	57.0	1.9	93.37	85.72	94.91	34.86	88.60	69.78	88.61
BL0743-06	V3		125	16	0.82	0.63	3.07	9.39	10.11	63.6	2.1	92.65	92.14	94.98	34.06	90.07	85.46	88.61
BL0743-07	V4	Breccia	75	42	0.41	0.11	1.18	14.52	17.03	13.5	0.5	96.50	88.86	97.75	87.24	95.47	90.86	97.29
BL0743-08	V4		125	33	0.52	0.12	1.51	18.51	22.46	16.8	0.6	96.62	85.30	98.68	78.25	95.69	89.32	94.10
BL0743-20	V10	Sedimentary	125	11	2.17	0.60	6.32	18.70	25.06	119.4	4.4	81.80	79.27	92.05	29.11	59.70	68.33	65.37

Overall, the samples responded well to bulk flotation process. Surprisingly, even samples with very low feed grade performed reasonably well. There was a proportional relationship between sulphur recovery and gold recovery to the bulk rougher concentrate. This relationship was observed in the previous (2020) testing program. High sulphur recoveries typically resulted in high gold recovery to the bulk rougher concentrate. On a feed grade weighted basis, the economical metal recoveries were in excess of 92% to the bulk rougher concentrate.

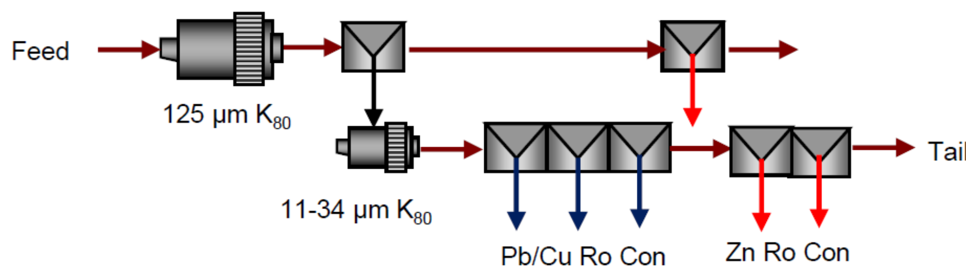
It is important to note that the rougher mass recovery to achieve the stated results varied considerably, from 4% to 74% following the total sulphur and iron grades.

The primary grind size of 125 micron K_{80} was selected after work completed during 2020 program had demonstrated bulk flotation circuit was insensitive to grind size. To confirm this result, select samples were chosen for repeat tests at a finer primary grind size. When comparing the pairs of tests, there was no statistically significant difference between the results.

13.4.5 Variability Sequential Rougher Tests

To determine if the variability samples response to the developed sequential flotation conditions, a series of tests were performed using the flowsheet shown if Figure 13.14.

Figure 13.14 - Sequential Rougher Test Flowsheet



After bulk flotation, the concentrate was reground to nominally 15 microns K_{80}. Regrind times were adjusted to accommodate variable bulk rougher mass and hold the regrind size. The pH was increased to 8.5 with soda ash, then lowered with sodium metabisulphite (SMBS) to a controlled pH set-point of 6.5. Copper and lead specific collectors are then used to recover the copper, lead, and free gold and silver to this bulk concentrate.

The copper/lead rougher flotation stage was followed by zinc rougher flotation. The zinc rougher scavenger concentrate was added to the copper/lead rougher tailings and conditioned with lime and copper sulphate. The summary of the tests results can be found in Table 13.38.

Table 13.38 - Sequential Rougher Performance Summary

Test	Sample	Lithology	Product	Weight	Assay - percent or g/t							Distribution - percent						
				%	Cu	Pb	Zn	Fe	S	Ag	Au	Cu	Pb	Zn	Fe	S	Ag	Au
BL0743-09	V1	VMS	Cu Con	18	13.94	4.34	8.23	26.66	37.92	407.6	13.1	85.20	82.77	35.24	22.38	23.47	57.55	53.59
			Zn Con	8	2.10	0.59	33.05	17.09	38.25	348.5	7.8	5.62	4.90	61.89	6.27	10.35	21.51	13.90
BL0743-33	V13		Cu Con	25	8.03	0.53	5.08	36.57	43.58	82.0	9.9	94.09	73.24	41.08	31.27	32.39	68.22	74.00
			Zn Con	9	0.74	0.21	21.26	28.53	41.14	83.0	3.8	2.92	9.95	58.15	8.25	10.34	23.35	9.51
BL0743-53	V15		Cu Con	20	19.77	2.05	6.89	28.74	39.39	195.6	11.9	93.43	83.43	28.46	24.30	23.97	59.86	60.33
			Zn Con	7	1.03	0.24	48.57	9.31	36.36	200.2	3.9	1.64	3.33	67.58	2.65	7.45	20.63	6.59
BL0743-32	V12	SMS	Cu Con	16	5.35	0.40	0.74	35.19	39.78	34.9	3.1	88.07	65.24	29.10	35.95	38.29	56.25	49.17
			Zn Con	3	1.28	0.16	9.04	33.61	42.78	38.7	4.2	3.93	4.88	65.82	6.39	7.67	11.63	12.46
BL0743-52	V14		Cu Con	15	15.89	3.08	9.44	25.77	36.09	457.8	17.8	93.66	83.52	39.43	25.66	25.79	70.92	66.67
			Zn Con	5	0.81	0.25	39.60	11.95	33.39	218.9	5.1	1.63	2.36	56.65	4.07	8.17	11.61	6.59
BL0743-10	V2	Grainstone	Cu Con	8	9.39	1.55	10.57	22.49	31.84	308.9	21.2	90.34	75.96	72.93	30.46	73.35	83.55	85.90
			Zn Con	2	1.34	0.57	15.18	9.31	18.03	158.8	7.9	2.72	5.90	22.08	2.66	8.75	9.05	6.75
BL0743-11	V3	Tuff	Cu Con	3	2.98	2.28	8.91	21.34	32.34	226.7	8.4	69.10	72.12	63.12	17.15	63.85	67.74	60.85
			Zn Con	1	0.48	0.36	8.31	9.54	12.81	104.1	2.6	4.17	4.23	22.08	2.87	9.48	11.66	7.15
BL0743-12	V4	Breccia	Cu Con	6	2.77	0.49	4.15	23.17	31.36	64.8	2.3	77.71	64.60	47.59	19.60	23.77	62.07	52.19
			Zn Con	5	0.32	0.08	4.81	30.30	41.12	23.5	1.0	7.92	9.35	48.89	22.68	27.57	19.90	19.97

On average, the copper/lead rougher circuit recovered 85% of the copper in the feed to the concentrate. Similarly, lead and gold were also 70 and 59% recovered to the rougher concentrate respectively. These values are in line with previous (2020) testing. Of critical importance is the recovery of zinc to copper/lead concentrate. High levels of zinc recovery indicate either poor liberation or unoptimized depressant levels to control the flotation of zinc. Target levels of zinc to this concentrate should be 35% or less. Many high-grade samples achieved this level of performance. Perhaps the Grainstone (Gr) mineralization might be the exception as both samples had poor copper-zinc selectivity. Samples V3 and V4 did not perform particularly well, but the feed grades were well below the cut grade so they may not be representative of the mineralization.

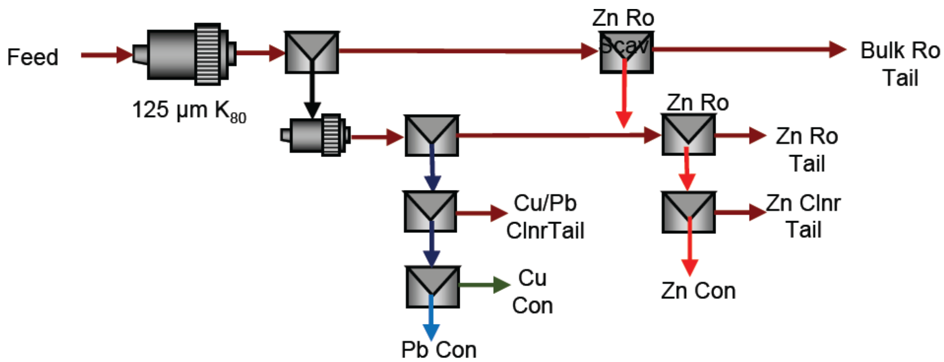
For the zinc rougher circuit, about 52% of the zinc in the feed was recovered to this concentrate. The combined copper/lead and zinc rougher recovery indicated that available zinc was recovered from the bulk concentrate. As noted in earlier programs, significant levels of gold and silver were also recovered to the zinc circuit. The average for the variability program was 18% silver and 12% gold recovery.

Overall, the developed process achieved expected rougher performance on a wide range of geological lithologies.

13.4.6 Variability Cleaner Tests

A series of open circuit batch cleaner tests were performed on selected variability samples. Low grade samples were not examined. A simplified schematic of the flowsheet and metallurgical summaries are displayed in Figure 13.15.

Figure 13.15 - Variability Cleaner Test Flowsheet



The copper-lead separation was performed on the bulk copper/lead concentrate. Due to the ratio of copper to lead in the feed, many of the bulk copper/lead concentrates had relatively low lead levels and would not affect the marketing of the copper concentrate. These samples may not need a lead separation. The summary of the tests can be seen in Table 13.39.

Table 13.39 - Variability Cleaner Results Summary

Test	Sample	Lithology	Product	Weight	Assay - percent or g/t							Distribution - percent						
				%	Cu	Pb	Zn	Fe	S	Ag	Au	Cu	Pb	Zn	Fe	S	Ag	Au
BL0743-34	V1	VMS	Pb Con	1.3	22.70	20.80	3.90	21.10	29.70	456.0	12.2	10.42	27.71	1.22	1.30	1.36	4.51	3.55
			Cu Con	2.2	22.30	2.19	8.20	26.10	38.90	568.0	14.8	17.18	4.90	4.29	2.71	2.98	9.42	7.22
			Zn Con	4.6	1.84	0.82	57.20	4.00	33.90	439.0	9.4	2.99	3.87	63.26	0.88	5.49	15.38	9.68
BL0743-36	V5		Pb Con	1.1	29.70	1.60	0.81	30.50	36.50	48.0	4.1	12.33	16.71	1.10	0.98	1.04	3.04	2.48
			Cu Con	5.5	28.70	0.56	0.87	30.00	36.70	70.0	2.3	60.40	29.65	6.00	4.91	5.30	22.47	6.86
			Zn Con	0.9	1.40	0.21	58.80	2.46	34.90	279.2	6.0	0.47	1.76	64.09	0.06	0.80	14.16	2.91
BL0743-39	V13		Pb Con	0.5	22.40	7.90	4.60	25.30	33.50	164.0	26.6	5.64	25.35	0.78	0.51	0.53	3.05	4.14
			Cu Con	6.1	22.50	0.33	3.40	28.80	39.10	107.0	21.1	66.51	12.44	6.73	6.77	7.23	23.40	38.67
			Zn Con	3.4	0.74	0.27	62.00	0.93	32.20	223.0	3.7	1.24	5.75	69.34	0.12	3.37	27.56	3.79
BL0743-56	V15		Cu Con	6.7	32.50	0.48	1.51	31.80	35.00	655.0	11.9	48.97	5.57	1.95	8.70	7.18	24.05	16.52
			Zn Con	4.0	1.71	0.31	60.80	1.98	32.90	295.0	3.6	1.54	2.15	46.95	0.32	4.04	6.48	3.00
BL0743-37	V9	SMS	Pb Con	1.1	25.50	6.90	6.90	26.00	33.80	264.0	10.6	10.84	28.01	1.06	2.23	1.82	4.98	4.49
			Cu Con	5.5	27.20	1.28	6.90	28.80	37.20	253.0	17.6	59.57	26.76	5.46	12.71	10.32	24.58	38.28
			Zn Con	6.5	0.73	0.16	60.80	1.19	34.10	194.0	3.2	1.87	3.92	56.36	0.62	11.08	22.08	8.16
BL0743-38	V12		Pb Con	0.1	23.60	1.48	0.94	28.10	36.40	66.0	4.0	2.24	1.90	0.22	0.20	0.19	0.83	0.40
			Cu Con	1.8	26.10	0.10	0.48	28.30	37.20	42.0	3.7	54.54	2.82	2.46	4.37	4.38	11.65	7.96
			Zn Con	0.4	2.08	0.34	58.40	3.40	32.30	138.0	6.0	0.84	1.85	57.70	0.10	0.73	7.37	2.51
BL0743-55	V14		Cu Con	4.7	30.90	0.64	1.40	28.60	37.50	344.0	20.0	62.34	5.10	1.83	9.56	8.63	15.01	24.36
			Zn Con	3.3	0.96	0.24	61.60	1.42	32.20	366.0	6.9	1.34	1.33	55.98	0.33	5.14	11.09	5.86
BL0743-35	V2	Grainstone	Pb Con	0.6	11.70	10.10	13.50	21.60	35.80	547.0	38.9	8.71	41.28	7.65	2.30	6.51	11.36	12.09
			Cu Con	3.3	14.50	0.80	14.80	26.40	39.10	399.0	29.1	55.87	16.92	43.40	14.56	36.83	42.87	46.75
			Zn Con	0.6	2.04	0.69	25.40	14.60	30.40	271.0	18.0	1.40	2.59	13.24	1.43	5.09	5.18	5.14

As shown in the table, copper was recovered into a concentrate grading 25 percent copper. Average levels of lead and zinc were 2.4 and 4.6%. Most of the variability samples performed well, except for V2 (Grainstone), which had high levels of zinc. Gold and silver levels in the concentrate were 15 and 309 g/tonne, which will contribute significantly to the revenue. The open circuit gold recoveries to the bulk copper/lead concentrate were highly variable ranging from 10 to 74%. It is understood that LCTs would be better representation of the actual distribution between the concentrates and tailings.

The lead separation was performed by adding cyanide to depress copper. Conditions from the previous program were used. In general, the separations were not efficient. The concentrates tended to have high copper levels. It was concluded that that additional optimization required to produce lead concentrates that will be marketable.

The zinc circuit demonstrated that high grade concentrates could be produced from all of the lithologies, except for V2 (Grainstone). Locked cycle tests required to accurately predict the zinc concentrate grades and recoveries.

13.4.7 Master Composite Cleaner Test Optimisation

Batch cleaner tests were completed on the three main composites. These tests were performed to optimize the copper/lead separation circuit as well as to optimise zinc rejection from the copper/lead bulk circuit. Figure 13.16 displays the flowsheet schematic utilised for these tests. The tests conditions can be found in Table 13.40. The metallurgical results obtained during the tests are in Table 13.41.

Figure 13.16 - Composite Batch Cleaner Flowsheet Schematics

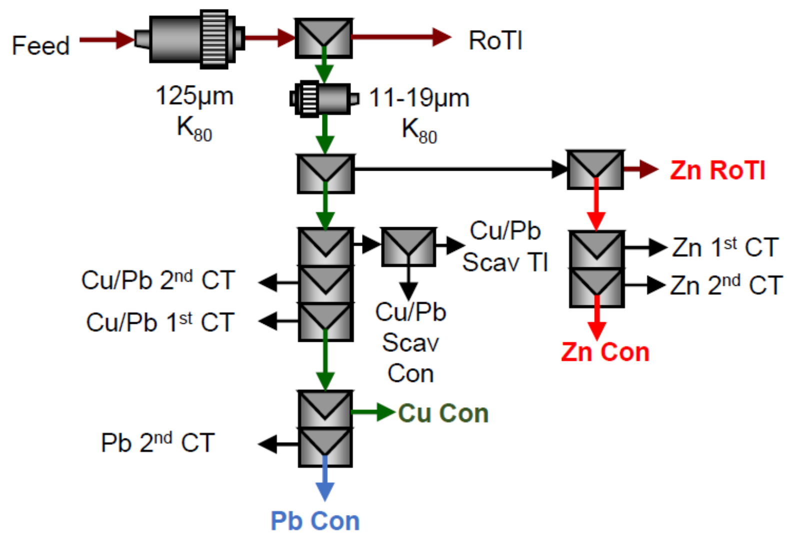


Table 13.40 - Composite Batch Cleaner Test Conditions

Sample	Test	Cu/Pb Cleaner pH	Cu/Pb Cleaner Reagents, g/t				Pb Cleaner pH	Pb Cleaner Circuit Reagents, g/t			
			SMBS	SIPX	3418A	A241		NaCN	FO	LGN	CMC
Composite 1	BL743-62	6.5	750	5	9	-	10.4	600	-	-	-
	BL743-64	7.9	-	5	5	-	10.0	600	-	-	-
	BL743-65	8.1	-	5	5	-	10.0	600	-	-	-
	BL743-66A	6.0	720	7	-	7	10.3	600	10	-	-
	BL743-66B	6.0	720	7	-	7	10.3	600	-	100	-
	BL743-66C	6.0	720	7	-	7	10.3	900	-	-	125
Composite 2	BL743-63	6.5	605	5	9	-	10.0	600	-	-	-
	BL743-67A	6.0	470	7	-	7	11.0	600	15	-	-
	BL743-67B	6.0	470	7	-	7	11.0	600	-	100	-
	BL743-67C	6.0	470	7	-	7	11.0	900	-	-	125
Composite 3	BL743-60	6.5	1325	6	-	6	-	-	-	-	-

Table 13.41 - Composite Batch Cleaner Test Results Summary

Sample	Test	Mass	Assay - percent or g/t							Distribution - percent						
		%	Cu	Pb	Zn	Fe	S	Ag	Au	Cu	Pb	Zn	Fe	S	Ag	Au
Copper/Lead Concentrate																
Composite 1	BL743-62	5.30	23.80	5.73	8.63	24.40	35.50	642.00	30.00	76.60	71.50	10.90	17.30	16.40	43.70	54.80
	BL743-64	5.31	24.80	5.80	8.01	25.00	34.80	525.00	29.7	78.2	68.3	10.1	16.9	16.2	36.0	55.7
	BL743-65	5.16	27.2	3.97	5.33	27.1	36.3	315	17.3	83.3	70.6	11.7	9.9	9.8	33.8	41.3
	BL743-66	4.26	28.00	5.25	3.60	26.80	35.00	520.00	28.10	72.40	50.60	3.50	14.70	12.40	26.60	41.10
Composite 2	BL743-63	4.84	28.50	3.97	2.97	27.90	36.50	248.00	15.70	80.80	65.30	6.10	9.50	9.20	28.40	33.20
	BL743-67	4.74	28.10	4.10	3.29	26.90	33.80	264.00	18.20	83.30	64.70	6.80	8.70	8.30	29.90	38.80
Composite 3	BL743-60	10.22	31.40	0.26	0.59	30.30	36.60	83.00	3.70	83.50	18.10	3.20	13.80	11.90	25.10	18.00
Lead Concentrate																
Composite 1	BL743-62	0.82	18.20	25.40	8.40	16.90	29.20	585.00	19.50	9.10	49.10	1.60	1.90	2.10	6.20	5.50
	BL743-64	1.45	22.10	16.20	8.20	20.50	31.40	590.00	26.40	19.10	52.10	2.80	3.80	4.00	11.00	13.50
	BL743-65	0.43	22.5	18.7	5.10	21.7	31.9	390.00	12.8	5.80	27.7	0.90	0.70	0.70	3.50	2.50
	BL743-66A	0.12	20.0	26.6	4.30	17.0	28.2	757.00	42.9	1.40	7.00	0.10	0.30	0.30	1.10	1.70
	BL743-66B	0.04	19.00	27.70	3.90	16.70	27.80	779.00	40.00	0.50	2.50	0.00	0.10	0.10	0.40	0.60
	BL743-66C	0.04	15.70	34.60	5.50	13.20	26.00	1380.00	14.10	0.40	3.40	0.10	0.10	0.10	0.70	0.20
Composite 2	BL743-63	0.44	19.70	26.80	4.10	19.10	29.70	323.00	12.90	5.10	40.20	0.80	0.60	0.70	3.40	2.50
	BL743-67A	0.41	24.5	6.50	3.40	23.1	37.5	310.00	23.70	6.20	8.80	0.60	0.60	0.80	3.00	4.40
	BL743-67B	0.35	28.3	7.80	2.90	25.0	36.6	331	27.6	6.2	9.1	0.4	0.6	0.7	2.8	4.4
	BL743-67C	0.10	19.00	26.30	5.30	17.90	28.40	310.00	23.70	1.10	8.40	0.20	0.10	0.10	0.70	1.00
Zinc Concentrate																
Composite 1	BL743-62	3.47	0.90	0.42	59.20	1.20	32.10	263.00	6.20	1.90	3.40	49.10	0.60	9.70	11.70	7.40
	BL743-64	3.83	0.90	0.36	63.20	1.10	32.60	398.00	6.10	1.90	3.10	57.60	0.60	10.90	19.70	8.20
	BL743-65	1.92	0.70	0.21	66.80	1.20	33.10	306.00	4.10	0.80	1.40	54.50	0.20	3.30	12.20	3.60
Composite 2	BL743-63	2.56	0.90	0.29	62.40	1.90	32.70	295.00	4.60	1.40	2.50	67.10	0.30	4.30	17.80	5.10
Composite 3	BL743-60	2.14	1.00	0.22	62.40	1.60	34.00	273.00	6.60	0.60	3.20	70.80	0.10	2.30	17.20	6.60

A number of parameters were investigated in order to improve metallurgical performance and reduce copper losses to the lead concentrate. These consisted of coarser regrind sizing, higher pH in the copper/lead cleaner circuit, as well as variations of the reagent scheme. The majority of the optimization was performed on Composite 1. Initially, tests were performed using 3418A in the copper/lead cleaner circuit at a pH of 6.5 and regrind sizing of 13 micron K_{80}. This test resulted in a copper lead bulk concentrate copper recovery of 76.6% feed copper and 71.5% of the feed lead, assaying 23.8% Cu and 5.73% percent Pb. Upon conducting a copper/lead separation, 9% of the feed copper was lost to the lead concentrate. Increasing the bulk cleaner pH, removing SMBS and coarsening the regrind size did not significantly improve metallurgical performance or zinc rejection from the bulk circuit.

A different reagent scheme, consisting of replacing 3418A with A241 was performed. The bulk concentrate from this test was split into three concentrates, followed by three separate copper/lead separations using either fuel oil (FO), Lignosulfonate (LGN) or carboxyl methylcellulose (CMC) in the separation circuit, along with sodium cyanide. These tests were successful in reducing copper reporting to the lead concentrate, while maintaining a high lead grade in the lead concentrate. This reagent scheme was conducted on Composite 2 with a similar outcome. Using CMC resulted in the lowest copper losses.

Composite 3 batch cleaner performed well. The one test conducted for this sample. The test did not include a copper/lead separation circuit due to the low lead head grade in the Composite 3.

13.4.8 Master Composite Locked Cycle Testing

Composites 1 and 2 were subject to two (2) LCTs each, using the base reagent scheme of SMBS in the bulk copper/lead cleaner circuit and using only NaCN in the lead cleaner (reverse) circuit and the optimized schematic of removing SMBS from the bulk cleaner circuit and adding CMC in the copper/lead reverse separation circuit.

Composite 3 locked cycle test produced good results without an optimization.

The flowsheet used for the LCTs can be found in Figure 13.17, with a summary of metallurgical results are displayed in table.

Figure 13.17 - Composites Locked Cycle Test Flowsheet Schematics

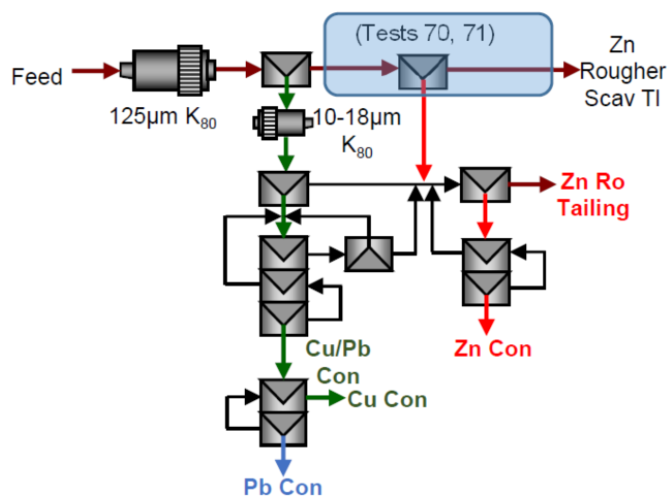


Table 13.42 - Locked Cycle Test Results Summary

Description	Mass %	Assay –% or g/t							Distribution - %						
		Cu	Pb	Zn	Fe	S	Ag	Au	Cu	Pb	Zn	Fe	S	Ag	Au
Composite 1 - Test BL743-58															
Feed	100	1.52	0.44	3.99	7.0	11.8	71	2.3	100	100	100	100	100	100	100
Pb Concentrate	0.9	15.8	28.9	11.7	11.6	27.2	1310	18.2	9	59	3	1	2	16	7
Cu Concentrate	5.1	21.6	1.93	12.7	22.4	36.9	411	19.0	73	22	16	16	16	29	42
Zn Concentrate	5.1	1.40	0.42	58.9	1.8	33.2	397	7.11	5	5	75	1	14	28	16
Zn Rougher Tail	20.9	0.71	0.22	0.96	22.1	33.1	71	3.24	10	11	5	66	59	21	29
Bulk Rougher Tail	68.9	0.07	0.02	0.08	1.5	1.57	5	0.17	3	3	1	15	9	5	5
Composite 1 - Test BL743-71															
Feed	100	1.61	0.39	3.70	8.7	12.6	73	2.16	100	100	100	100	100	100	100
Pb Concentrate	0.1	14.9	34.9	5.67	11.4	26.0	1115	3.35	1	12	0.2	0.2	0.3	2	0.2
Cu Concentrate	5.7	23.3	4.42	7.43	24.6	35.8	572	18.6	83	65	12	16	16	45	49
Zn Concentrate	5.1	1.45	0.47	58.7	2.3	34.1	372	6.01	5	6	82	1	14	26	14
Zn Rougher Tail	20.2	0.69	0.23	0.95	30.4	37.2	76	3.03	9	12	5	70	60	21	28
Bulk Rougher Tail	68.9	0.05	0.03	0.07	1.6	1.77	6	0.24	2	5	1	12	10	6	8
Composite 2 - Test BL743-59															
Feed	100	2.10	0.32	2.49	13.1	18.9	46	2.39	100	100	100	100	100	100	100
Pb Concentrate	0.8	25.2	16.0	4.51	21.6	30.1	656	7.35	9	39	1	1	1	11	2
Cu Concentrate	5.8	28.3	2.11	4.45	27.8	34.0	260	14.4	78	39	10	12	10	32	35
Zn Concentrate	3.6	2.03	0.92	56.2	4.0	32.8	337	10.2	4	11	82	1	6	26	15
Zn Rougher Tail	29.8	0.49	0.10	0.41	30.7	44.7	38	3.11	7	10	5	70	70	24	39
Bulk Rougher Tail	60.8	0.07	0.01	0.07	3.4	3.60	5	0.34	2	2	2	16	12	6	9
Composite 2 - Test BL743-70															
Feed	100	1.56	0.28	2.24	14.7	19.4	47	1.93	100	100	100	100	100	100	100
Pb Concentrate	0.2	13.3	42.4	3.54	12.6	23.7	394	4.08	2	28	0.3	0.2	0.2	2	0.4
Cu Concentrate	4.3	30.4	2.34	1.78	28.1	36.2	235	13.7	83	35	3	8	8	21	30
Zn Concentrate	3.5	1.59	0.84	58.0	3.3	34.2	322	7.22	4	10	90	1	6	24	13
Zn Rougher Tail	31.3	0.45	0.18	0.31	36.6	45.8	42	2.87	9	20	4	78	74	28	46
Zn Rougher Scavenger Tail	61.0	0.06	0.03	0.07	3.2	3.70	19	0.32	2	6	2	13	12	25	10
Composite 3 - Test BL743-61															
Feed	100	3.74	0.21	1.83	24.2	32.7	33	2.02	100	100	100	100	100	100	100
Cu Concentrate	11.7	28.7	0.74	0.65	28.2	36.8	94	4.47	89	41	4	14	13	33	26
Zn Concentrate	3.3	4.48	1.94	51.4	5.8	34.2	326	9.04	4	31	92	1	3	32	15
Zn Rougher Tail	49.6	0.43	0.09	0.09	39.4	51.8	20	2.26	6	20	2	81	79	30	55
Zn Rougher Scavenger Tail	35.4	0.10	0.04	0.08	3.3	4.47	4	0.24	1	8	1	5	5	4	4

Initially for Composite 1, copper was 73% recovered into a copper concentrate, assaying 21.6% percent Cu, 1.9% Pb and 12.7% Zn. The lead product for this test has 59% lead, and 9% Cu recovery respectively with 28.9% Pb grade. The zinc concentrate grade was 58.9% at 75% zinc recovery.

By modifying the reagent scheme as stated above, metallurgical performance improved substantially for all payable metals. Copper recovery increased to 83% in the final copper concentrate, assaying 23.3% Cu and at lower lead and zinc concentrations of 4.4% and 7.4%, respectively. Lead recovery in the lead concentrate decreased, however Pb grade increased to 34.9% percent. Zinc performance was also higher for this test at 82% zinc recovery to the zinc final concentrate, at 58.7% Zn grade.

Similarly for Composite 2, modifying the reagent scheme in the copper/lead cleaner circuit and copper/lead separation circuit resulted in superior metallurgical results, as well as reaching the target lead plus zinc concentration of less than 10% in the copper concentrate. For this test, copper recovery was 83% percent at 30.4% Cu grade, 2.3% Pb and 1.8% Zn. Lead recovery to the lead concentrate was 28% at concentrate grade assaying 42.4% Pb. Copper recovery to lead concentrate was only 2%. Zinc also performed very well with 90% percent zinc recovery at 58.0% zinc concentrate grade.

Composite 3, using only base condition testing, also performed well with 89% copper recovery, at 28.7% Cu concentrate grade. Lead and zinc grades were very low in this concentrate. Zinc recovery to zinc concentrate 92% concentrate, assaying 51.4% percent zinc grade.

13.4.9 Ore Aging Tests

Ore aging tests have been completed on a Composites 1 and 2 in order to evaluate any impact of the stockpiling on the ore flotation response. This phenomenon had been observed on some of the sulphide ores resulting in reduced flotation performance.

The Curipamba ore samples were tested using a standard methodology where ore material is split in two subsamples. First subsample is being exposed to the ambient at particle size 100% passing 3.35 mm, and held at 5% moisture during up to sixteen weeks. A second sample of same ore is preserved following the standard practices as sealed and frozen. Both samples are floated at same conditions after the time, and the flotation response of the sample exposed to ambient is being compared to the flotation response of the sample which was sealed and frozen.

Evaluation of the results is indicative in nature, and if significant difference in flotation response has not been observed then the ore can be stockpiled, and if the flotation response differs significantly, the ore cannot be stockpiled.

Ore aging test flowsheet can be found in Figure 13.18. Summary of the results is in Table 13.43.

Figure 13.18 - Ore Aging Test Flowsheet Schematics

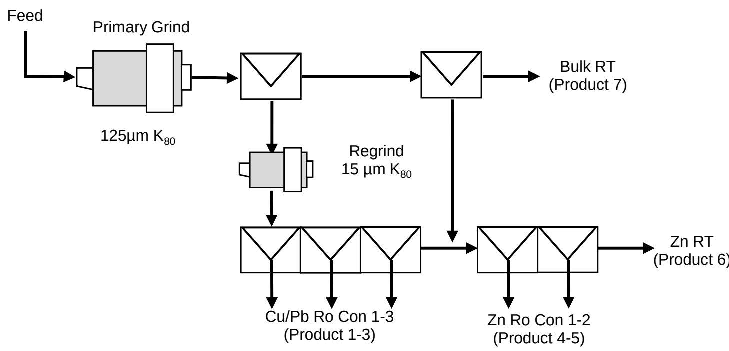


Table 13.43 - Ore Aging Tests Results Summary

Sample	Test	Product	Weight	Assay - percent or g/t							Distribution - percent						
			%	Cu	Pb	Zn	Fe	S	Ag	Au	Cu	Pb	Zn	Fe	S	Ag	Au
COMPOSITE 1																	
aged	BL0743-72	Cu/Pb Ro Con 1-3 (Product 1-3)	9.4	14.05	3.40	16.14	21.3	36.08	454.2	19.87	85.5	77.8	31.7	24.0	25.5	55.2	67.7
	BL0743-72	Zn Ro Con 1-2 (Product 4-5)	8.0	1.50	0.39	39.26	11.8	35.55	282.6	4.83	7.8	7.6	65.9	11.4	21.5	29.4	14.1
	BL0743-72	Zn Ro Tail	14.3	0.44	0.23	0.43	28.6	39.30	52.0	2.71	4.1	8.0	1.3	49.1	42.2	9.6	14.0
	BL0743-72	Bulk Ro Tail	68.3	0.06	0.04	0.08	1.9	2.10	6.6	0.17	2.7	6.7	1.1	15.5	10.8	5.8	4.2
non aged	BL0743-50	Cu/Pb Ro Con 1-3 (Product 1-3)	9.7	14.23	3.59	13.39	22.9	37.28	421.5	21.79	87.6	78.8	33.9	27.7	28.0	59.7	66.9
	BL0743-50	Zn Ro Con 1-2 (Product 4-5)	5.0	1.37	0.42	45.69	8.3	34.58	287.7	6.06	4.4	4.8	59.8	5.2	13.4	21.0	9.6
	BL0743-50	Zn Ro Tail	16.6	0.57	0.27	1.21	28.1	39.00	58.4	3.55	6.0	10.2	5.2	58.1	50.2	14.2	18.7
	BL0743-50	Bulk Ro Tail	68.8	0.05	0.04	0.06	1.0	1.56	5.1	0.22	2.0	6.2	1.1	9.0	8.3	5.1	4.8
COMPOSITE 2																	
aged	BL0743-73	Cu/Pb Ro Con 1-3 (Product 1-3)	10.2	15.90	2.54	9.37	24.8	37.98	271.7	13.14	91.2	79.0	38.3	18.4	19.8	58.3	60.4
	BL0743-73	Zn Ro Con 1-2 (Product 4-5)	4.8	0.88	0.23	30.48	18.3	39.54	179.3	3.62	2.4	3.3	58.8	6.4	9.7	18.1	7.8
	BL0743-73	Zn Ro Tail	24.2	0.27	0.14	0.15	33.6	46.20	32.0	2.01	3.7	10.3	1.5	59.4	57.2	16.3	21.9
	BL0743-73	Bulk Ro Tail	60.8	0.08	0.04	0.06	3.6	4.30	5.7	0.36	2.7	7.4	1.5	15.8	13.3	7.3	9.9
non aged	BL0743-51	Cu/Pb Ro Con 1-3 (Product 1-3)	9.3	17.99	2.64	7.10	27.2	38.40	250.3	15.89	89.8	78.4	25.9	18.1	17.6	52.2	54.2
	BL0743-51	Zn Ro Con 1-2 (Product 4-5)	4.3	1.28	0.27	41.86	13.1	39.54	232.7	6.98	3.0	3.8	70.9	4.0	8.4	22.6	11.1
	BL0743-51	Zn Ro Tail	27.1	0.35	0.14	0.17	33.4	47.50	31.2	2.79	5.1	12.1	1.8	64.7	63.5	19.0	27.7
	BL0743-51	Bulk Ro Tail	59.2	0.07	0.03	0.06	3.1	3.58	4.7	0.32	2.1	5.7	1.4	13.2	10.4	6.2	6.9

As seen from Table 13.43, there is no material difference in flotation response between the aged and non aged samples of Composites 1 and 2. Composite 3 was not tested due to insufficient quantity of the ore. Based on the results of the ore aging tests, the Curipamba ore can be stockpiled without loss of metallurgical performance.

13.4.10 MASTER COMPOSITE DEWATERING TESTS

13.4.10.1 Tailings Settling Tests

A series of dewatering tests were conducted on the combined bulk rougher and zinc rougher tailings from two bulk cleaner tests, performed on Composites 1 and 2. Tests were performed to specifically to generate tailings for this work. The tailings were kept as a slurry and not dried. Flocculant scoping, static settling, pressure filtration, and sheared viscosity tests were conducted.

Flocculant scoping tests were performed on the rougher tailings from generated from Composite 1 (Test 68) and Composite 2 (Test 69). The tests evaluated four various flocculants with Magnafloc 336 selected, providing best overflow clarity and fastest free settling rate.

Once the flocculant was selected, static settling tests were performance to assess flocculant dosages between 30 and 60 g/t. Results are shown in Table 13.44.

Table 13.44 - Settling Tests Results Summary

Sample	Floc, g/t	Free Settling Velocity, m/h	Final solids density, %
Composite 1			
T68 Final Tail	30	0.94	46
	40	0.89	44
	50	3.06	45
	60	6.09	46
Composite 2			
T69 Final Tail	30	0.51	43
	40	0.95	42
	50	0.96	43
	60	1.48	41

As shown in the table, high dosages of flocculant were required produce adequate settling rates. These results correspond reasonably to the data produced in 2019 testing campaign.

13.4.10.2 Tailings Filtration Tests

A series of the tailings pressure filtration tests were conducted in order to evaluate a potential for the tailings dry stacking. The tailings were fed to a lab filter press at a nominal 45 percent solids by weight. Longer blow times and thinner cake thicknesses resulted in the lowest cake moistures, but at slower cake production rates. Results are summarised in Table 13.45.

Table 13.45 - Pressure Filtration Results Summary

Sample	Test	Filtration Rate kg/m^2/h	Cake moisture, %
Composite 1			
T68 Final Tail	F1	243	24
	F2	210	25
	F3	211	23
	F4	241	29
	F5	228	26
	F6	213	25
Composite 2			
T69 Final Tail	F1	184	24
	F2	194	22
	F3	199	22
	F4	203	25
	F5	207	26
	F6	198	27

As seen from the table, the filtration rate is relatively slow compared to the typical filtration rates for the concentrates of same particle size being greater than 300 kg/m^2/h. Low filtration rate results to the increased filter area required and long dry times for the cake.

Cake moisture observed during the tests is high when compared to the cake moisture typically produced through the pressure filtration having 20% moisture as a median value observed in the mineral processing operations.

Based on the above data it is understood that tailings filtering is expected to have a considerable capital expenditure add significantly to the facility operating cost.

13.5 Recovery Estimates

13.5.1 Copper (Cu) Recovery

After analysing all of the flotation results, a Cu recovery relationship could not be determined and therefore a fixed value of 84.3% was used. This was the average of all the copper recovery results from the LCTs conducted.

This fixed value was estimated from the average between the master composites and variability samples. For the variability samples the average was calculated by using the masses of samples based on the master composite mass splits. The fixed Cu recovery value is shown as follows:

$\mathrm{Cu}\ Recovery\ \% = 84.3\%;\ Fixed\ Value$

The same methodology was applied to the Au recovery in the Cu concentrate. The fixed Au recovery value is shown as follows:

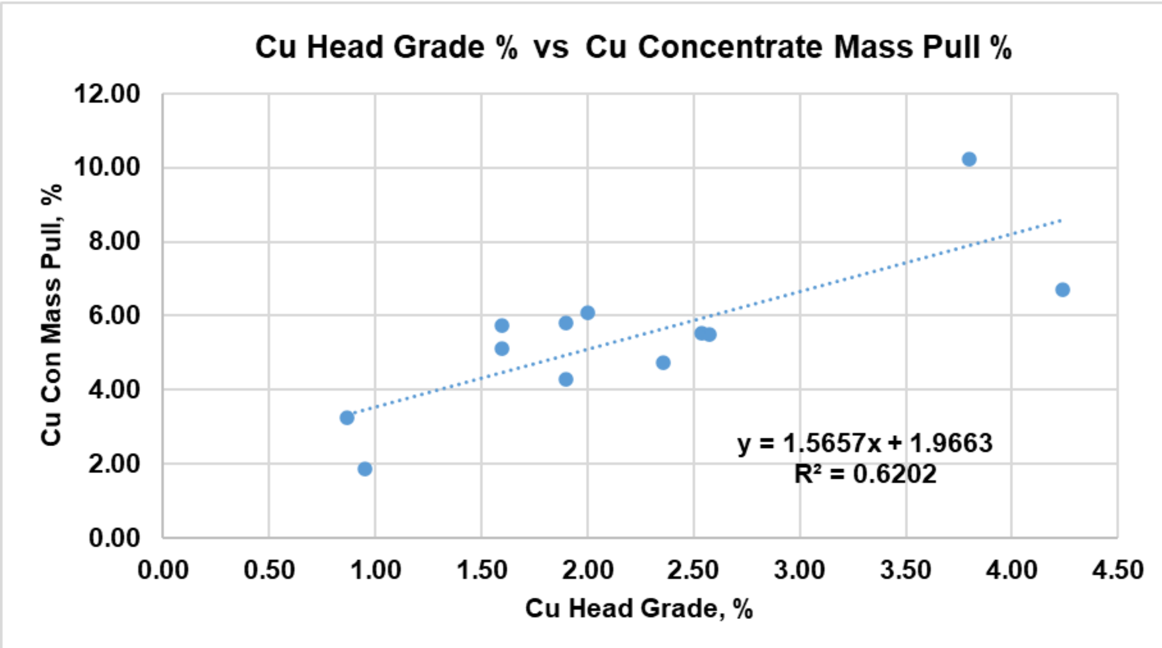
$\mathrm{Cu}\ Conc\ Gold\ Rec\ \% = 36.3\%;\ Fixed\ Value$

$\mathrm{Cu}\ Conc\ \mathrm{Silver}\ Rec\ \% = 33.2\%;\ Fixed\ Value$

The Cu mass pull % to the copper concentrate is based on the relationship shown below in Figure 13.19. This is a reasonably weak relationship (R^2 – 0.6202) but is sensible based on the copper head grade, copper minerals present (predominantly Chalcopyrite), copper recovery to copper concentrate and the level of contamination in the copper concentrate with other sulphides and gangue minerals.

$\mathrm{Cu}\ Mass\ Pull\ \% = 1.5657\ x\ \mathrm{Cu}\ Feed\ Grade + 1.9663$

Figure 13.19 Copper Head Grade versus Mass Pull %



13.5.2 Lead (Pb) Recovery

Regarding lead recoveries and concentrate levels, average values between the variability and master composite samples were used.

These are shown as follows:

Pb $Recovery$ % = 29.0%; $Fixed\ Value$

Pb $Mass\ Pull$ % = 0.22%, $Fixed\ Value$

$Au\ Recovery\ in$ Pb $Concentrate$ % = 1.2%; $Fixed\ Value$

$Ag\ Recovery\ in$ Pb $Concentrate$ % = 5.6%; $Fixed\ Value$.

13.5.3 Zinc (Zn) Recovery

After analysing all of the flotation results, a Zn recovery relationship could not be determined and therefore a fixed value of 81.9% was used. This was the weighed average of all the zinc recovery results from the LCTs conducted.

This fixed value was estimated from the average between the master composites and variability samples. For the variability samples the average was calculated by using the masses of samples based on the master composite mass splits. The fixed Zn recovery value is shown as follows:

$\mathrm{Zn}\ Recovery\ \% = 81.9\%;\ Fixed\ Value$

The same methodology was applied to the Au recovery in the Cu concentrate. The fixed Au and Ag recovery values are shown as follows:

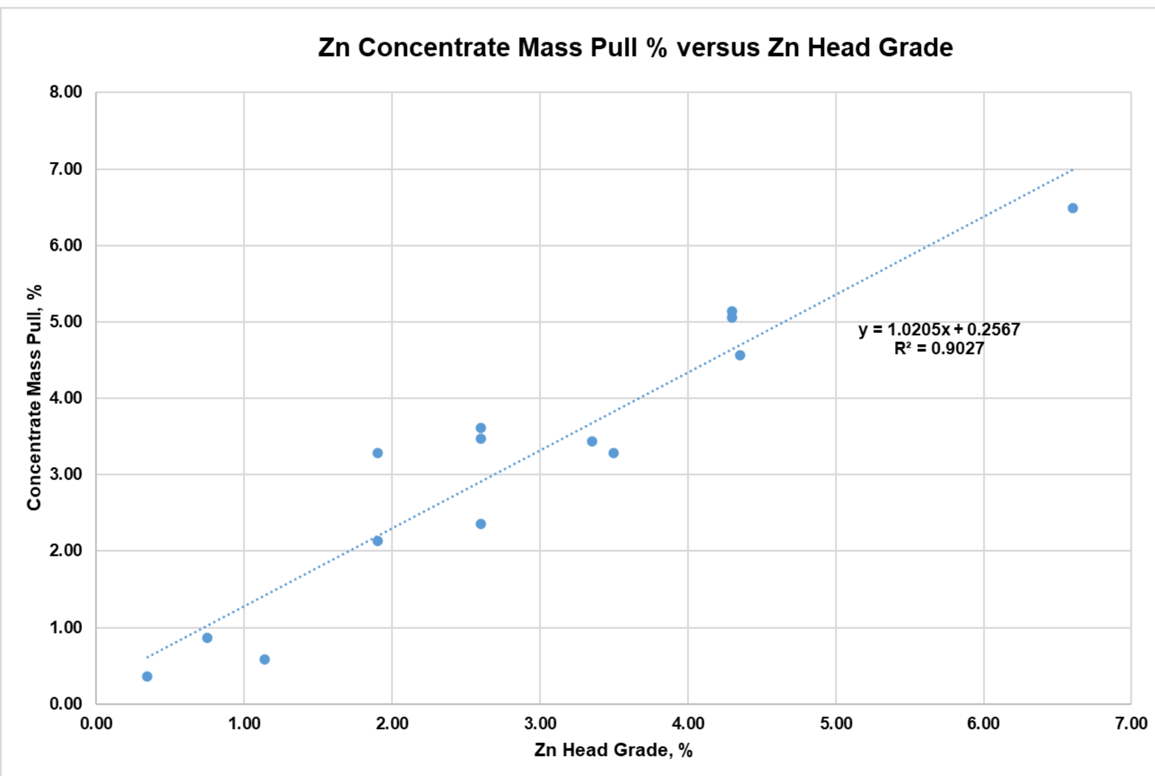
$\mathrm{Zn}\ Conc\ Gold\ Rec\ \% = 24.1\%;\ Fixed\ Value$

$\mathrm{Zn}\ Conc\ \mathrm{Silver}\ Rec\ \% = 15.4\%;\ Fixed\ Value$

The Zn mass pull % to the copper concentrate is based on the relationship shown below in Figure 13.20Figure 13.19. This is a strong relationship (R^2 – 0.9027) but is sensible based on the zinc head grade and zinc minerals present, zinc recovery to zinc concentrate and low levels of contamination in the zinc concentrate from other sulphide and gangue minerals.

$\mathrm{Zn}\ Mass\ Pull\ \% = 1.0205\ x\ \mathrm{Zn}\ Feed\ Grade + 0.2567$

Figure 13.20 – Zinc Head Grade versus Zinc Concentrate Mass Pull %



13.6 Metallurgical Variability

The metallurgical testwork completed at BML in the two programs are based on samples which adequately represent the variability of the El Domo deposit.

Mineralogical analysis of the various composite and variability samples has shown that the deposit is reasonably consistent with respect to mineralogy in the various lithologies established including, VMS, SMS and grainstone ore types. Tuff and breccia lithology samples demonstrated lower metal recoveries compared to the VMS, SMS, and grainstone.

13.7 Deleterious Elements

Copper, lead, and zinc concentrates from LCTs completed during the BML test work programs were assayed for the deleterious (penalty) elements in order to evaluate concentrate quality against smelter terms but were not available at the time of this report.

Minor element assays completed for selected samples provided substantial results with respect to the expected level of minor penalty elements. The copper concentrates produced demonstrate penalty elements level higher than the referenced smelter terms limits. Zinc and lead concentrates are relatively clean and no penalty element exceeds the limit established in the smelter terms referenced. Based upon DRA in house information the quality of all three concentrates (copper, lead, and zinc) with regards to the penalty elements is seen as acceptable.

It is understood that the deleterious elements will be similar to those produced in the concentrates from historical test work. There are not expected to be any issues with concentrate saleability resulting from issues around deleterious elements.

14 MINERAL RESOURCE ESTIMATE

The Mineral Resource estimate presented in this Report is a third estimate prepared by SLR and is a reasonable representation of the Mineral Resources of the Project at the current level of sampling. The Mineral Resources conform to CIM (2014) definitions and are reported in accordance with the NI 43-101. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Open pit Mineral Resources have been constrained within a Whittle pit shell. A summary of the Mineral Resources is presented in Table 14.1.

Table 14.1 – Mineral Resource Estimate – October 26, 2021

Resource Category	Tonnes (Mt)	Grade					Contained Metal				
		Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu (kt)	Pb (kt)	Zn (kt)	Au (koz)	Ag (koz)
Open Pit Resources											
Measured	3.2	2.61	0.2	2.50	3.03	45	84.9	7.7	81.1	316	4,704
Indicated	3.8	1.38	0.3	2.77	2.29	52	52.6	11.3	105.2	280	6,370
M+I	**7.1**	**1.95**	**0.3**	**2.64**	**2.63**	**49**	**137.5**	**19.0**	**186.3**	**596**	**11,074**
Inferred	0.3	0.34	0.2	1.01	1.34	39	1.2	0.7	3.5	15	430
Underground Resources											
Indicated	1.9	2.72	0.14	2.38	1.37	31	51.9	2.6	45.4	84	1,895
Inferred	0.8	2.31	0.11	2.68	1.74	29	17.3	0.8	20.1	42	688
Total Mineral Resources											
Measured	3.2	2.61	0.2	2.50	3.03	45	84.9	7.7	81.1	316	4,704
Indicated	5.7	1.83	0.24	2.64	1.98	45	104.5	13.9	150.6	364	8,265
M+I	**9.0**	**2.11**	**0.24**	**2.59**	**2.36**	**45**	**189.4**	**21.6**	**231.7**	**680**	**12,969**
Inferred	1.1	1.72	0.14	2.18	1.62	32	18.5	1.5	23.6	57	1,118

Notes:

1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are reported above an NSR cut-off value of US$29/t for potential open pit Mineral Resources and the underground portion of the 2021 Mineral Resources are reported with mining shapes which were generated using an NSR cut-off value of US$105/t NSR.
3. The NSR value is based on estimated metallurgical recoveries, assumed metal prices, and smelter terms, which include payable factors treatment charges, penalties, and refining charges.
4. Mineral Resources are estimated using the metal price assumptions: US$4.00/lb Cu, US$1.05/lb Pb, US$1.30/lb Zn, US$1,800/oz Au, and US$24/oz Ag.
5. Metallurgical recovery assumptions were based on three mineral types defined by the metal ratio Cu/(Pb+Zn):
   a. Zinc Mineral (Cu/(Pb+Zn) <0.33): 86% Cu, 90% Pb, 97% Zn, 68% Au, and 78% Ag;
   b. Mixed Cu/Zn Mineral (0.33≤ Cu/(Pb+Zn) ≤3.0): 86% Cu, 82% Pb, 95% Zn, 55% Au, and 67% Ag;
   c. Copper Mineral (Cu/(Pb+Zn) >3.0): 80% Cu, 37% Pb, 36% Zn, 14% Au, and 29% Ag;
6. NSR factors were also based on the metal ratio Cu/(Pb+Zn):
   a. Zinc Mineral (Cu/(Pb+Zn) <0.33): 53.41 US$/% Cu, 7.99 US$/% Pb, 13.47 US$/% Zn, 30.91 US$/g Au, and 0.39 US$/g Ag,
   b. Mixed Cu/Zn Mineral (0.33≤ Cu/(Pb+Zn) ≤3.0): 58.99 US$/% Cu, 7.05 US$/% Pb ,13.41 US$/% Zn, 25.12 US$/g Au, and 0.34 US$/g Ag;
   c. Copper Mineral (Cu/(Pb+Zn) >3.0): 57.83 US$/% Cu, 6.84 US$/g Au, and 0.19 US$/g Ag.
7. Bulk density interpolated on a block per block basis using assayed value, the correlation between measured density values and iron content, and base metal grade.
8. Mineral Resources are inclusive of Mineral Reserves.
9. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
10. The underground portion of the Mineral Resources are reported within underground reporting shapes and include low grade blocks falling within the shapes.
11. Numbers may not add due to rounding.

Leapfrog Geo software (version 2021.1) was used to construct the geological solids, prepare assay data for geostatistical analysis, construct the block model and estimate grades in all of the domains, and tabulate Mineral Resources.

The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

14.1 Mineral Resource Estimation Methodology

The evaluation of Mineral Resources involved the following procedures:

- Database compilation and verification;
- Generation of the lithological model;
- Definition of geostatistical resource domains;
- Data conditioning (capping and compositing) for geostatistical analysis and variography;
- Selection of the estimation strategy and estimation parameters;
- Block modelling and grade interpolation;
- Validation, classification, and tabulation;
- Assessment of “reasonable prospects for eventual economic extraction” and selection of reporting assumptions;
- Preparation of the Mineral Resource statement.

14.2 Resource Database

The database used to estimate resources includes 391 (not including abandoned and redrilled holes) core boreholes (74,993 m) completed between 2007 and 2021, including 55 boreholes (7,094 m) drilled in the 2020/2021 campaign. There were 46 boreholes (10,611 m) located outside of the resource area or not assayed, and these were not considered for the Mineral Resource model.

Table 14.2 summarises the records in the borehole database used to model Mineral Resources.

Table 14.2 – Summary of Database Records

Table	Number of Records
Collars	345
Survey	5,219
Assay	14,089
Lithology	4,744
2 m Composites within Resource Domains	2,915
Density	8,848 (9,477 All data)
Solid Intervals of Resource Domains	670

Data verification of the drill hole database included manual verification against original digital sources, a series of digital queries, and a review of Adventus' QA/QC procedures and results. No discrepancies were identified, and the QP considers the database fit for purpose, and in the QP's opinion, the geological data provided by Adventus for the purposes of Mineral Resource estimation were collected in line with industry best practice as defined in the CIM Exploration Best Practice Guidelines (CIM, 2018) and the CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (CIM, 2019). As such, the data are suitable for use in the estimation of Mineral Resources for the El Domo deposit.

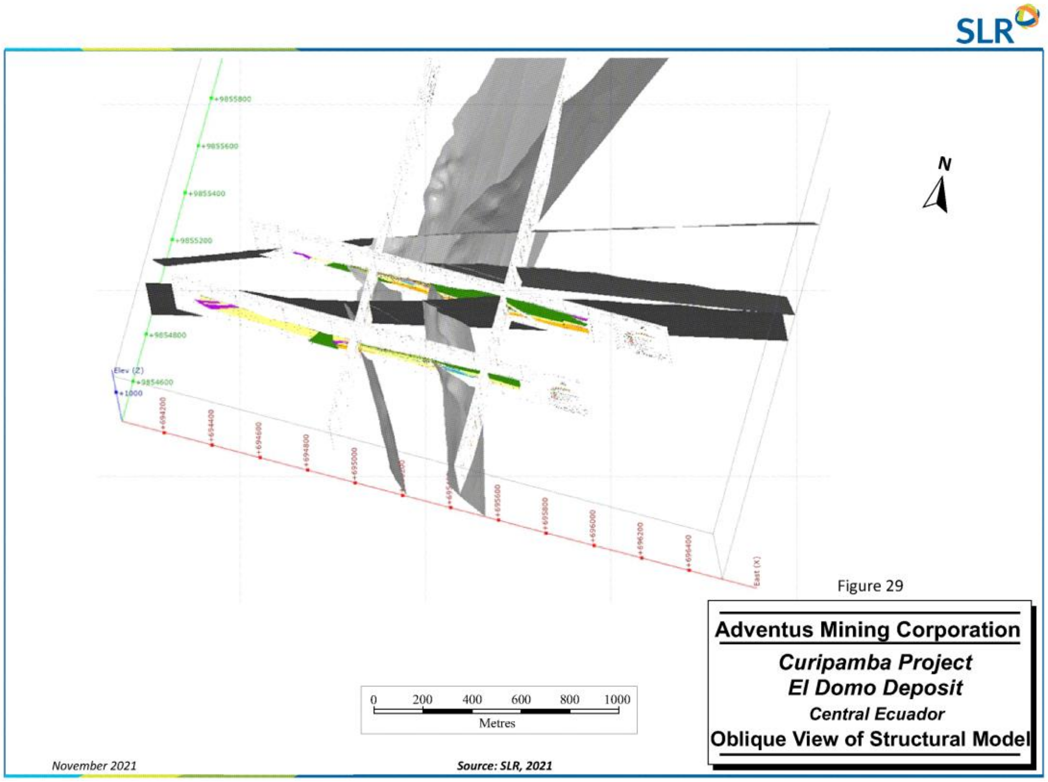
14.3 Geological Model

SLR used Leapfrog software (version 2012.1.2) to generate structural, geological, and domain models.

14.3.1 Structural Model

Using geological logs, cross and long sections, and surface maps, a preliminary structural model was created, composed of eleven faults. This model was the basis for modelling of all the lithological units and mineralized domains. Figure 14.1 shows an oblique view of the structural model with a long and cross section that were supplied by Adventus for reference. Faults were modelled from interpreted drilling sections but had to be modified locally to take into consideration and honour borehole information.

Figure 14.1 – Oblique View of Structural Model



14.3.2 LITHOLOGICAL MODEL AND GRADE DOMAINS

SLR used a combination of geological logs, assays, and sectional information to construct the geological model in each fault block generated in Leapfrog by activating the structural model. The main rock types include rhyolite, andesite, dacite, tuffs, and overburden/sediments. Dacite unit encompasses smaller lithological bodies such as gypsum, hydrothermal breccia, and basalt dikes.

The base and precious metal mineralization at the Project occurs primarily in a tabular zone comprising semi-massive to massive sulphides. Secondary loci of mineralization are in a breccia zone in the immediate hanging wall of the massive sulphide zone (the grainstone horizon), and in smaller lenses throughout the footwall of the massive sulphide zone.

SLR modelled the massive sulphide horizon based primarily on lithological logging information, however, where high-grade borehole interceptions had been recorded that were in otherwise barren lithology, the model was adjusted to include these high-grade intersections in the massive sulphide domain.

SLR calculated an NSR value for all assay intervals that takes into account all five economic metals (gold, silver, copper, lead, and zinc). This approach was used because no uniform zoning was detected where one element would provide the majority of the value rather than all elements contributing rather evenly to the value of the intersection.

In addition to lithological logging, modeling of the grainstone unit took into account the calculated NSR values. An indicator grade shell based on an NSR cut-off value of US $20/t was generated to better define mineralization within the grainstone domain and prevent grade smearing into unmineralised part of the grainstone. Where the grainstone domain coalesces into the massive sulphide domain, precedence was given to the massive sulphide domain.

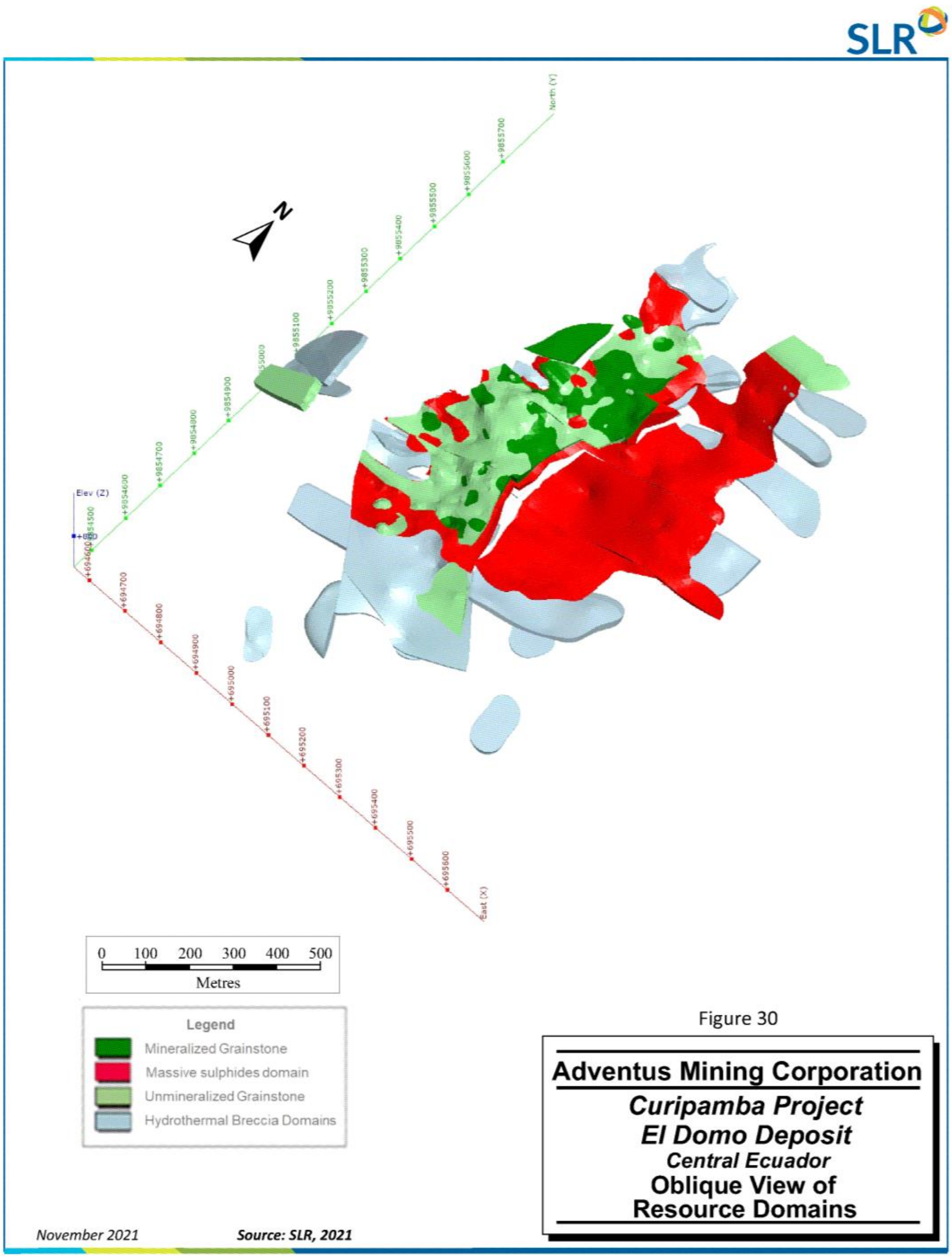
Other, typically small lenses of mineralization were modelled using the calculated NSR value alone. These lenses are generally associated with the hydrothermal breccia units occurring in the footwall of the massive sulphide domain.

All the modelled domains included in the current model are listed below:

- Lithological Domains:
  - Andesite
  - Basalt
  - Dacite
  - Gypsum
  - Hydrothermal Breccia
  - Rhyolite
  - Lapilli Tuff
  - T-Rhyolite
- Resource Domains:
  - VMS
  - Grainstone inside Indicator shell
  - Grainstone outside Indicator shell
  - LG Breccia

Figure 14.2 shows an oblique view of the resource domains, with the offset along faults apparent (faults not shown).

Figure 14.2 – Oblique View of Resource Domains



14.4 Density

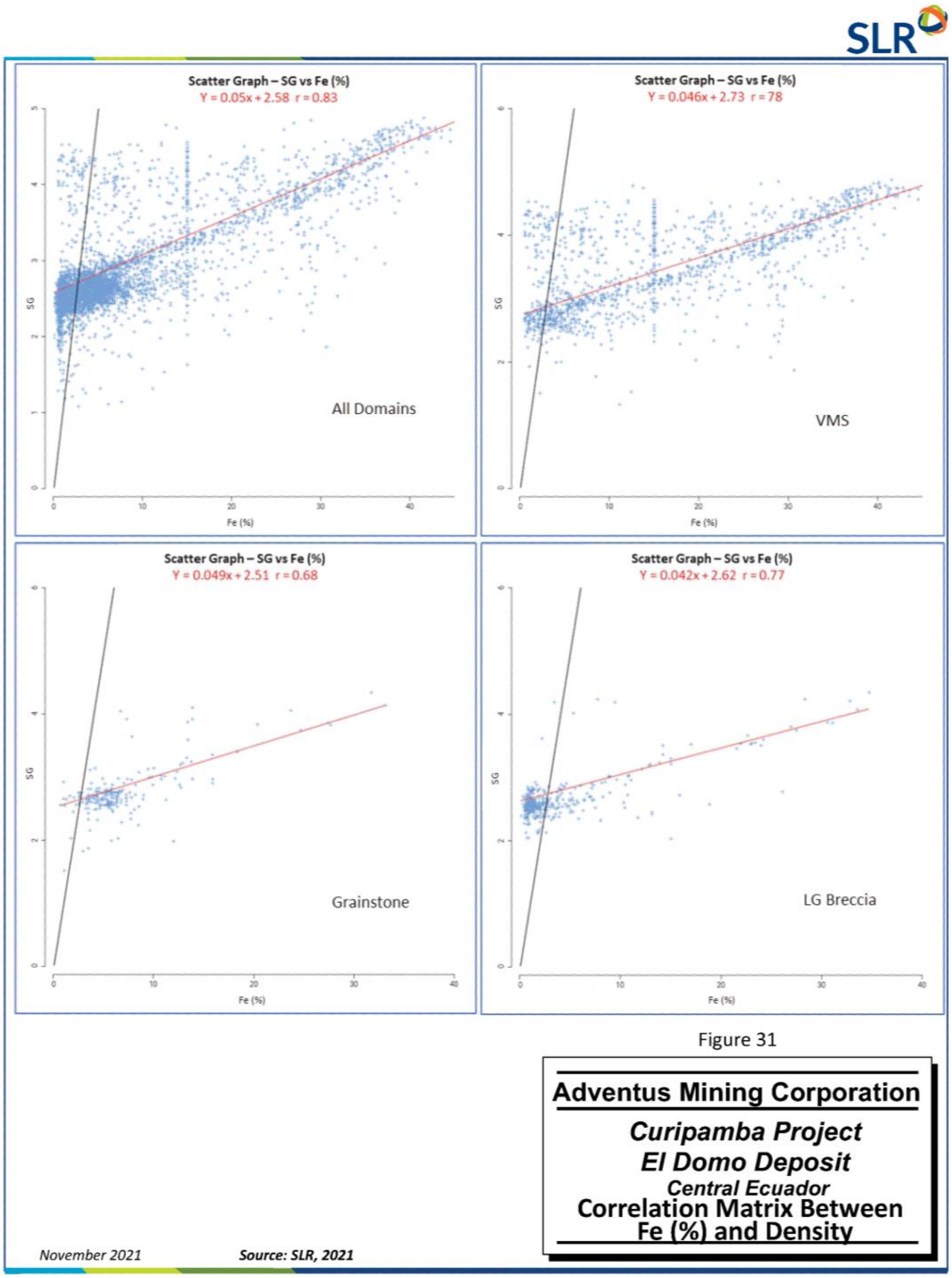
In previous years, density data were collected by Curimining from core samples using approximately 10 cm long sub-samples. In 2020/2021, all assayed intervals had their density values tested. In total, 9,477 density records were provided to SLR. The density data was coded by the geological domains and the summary descriptive statistics are presented in Table 14.3. For the lithological domains, SLR assigned averaged values (Table 14.3). For the resource domains, the density was estimated using the inverse distance squared (ID2) interpolation method. For the assay intervals missing density data, SLR used a correlation between iron values and density (Figure 14.3). For the intervals missing iron assays or when iron assays equalled 15%, however, SLR used correlation between the three base metals and density.

Table 14.3 – Summary of Specific Gravity

Domain	Count	Minimum	Maximum	Mean	Variance	SD [4]	CV [5]
Andesite	1,156	1.20	4.20	2.52	0.02	0.13	0.05
Basalt	127	2.10	3.51	2.69	0.01	0.10	0.04
BxHy [1]	778	1.07	5.21	2.88	0.09	0.30	0.10
Dacite	1,666	1.14	3.88	2.58	0.04	0.20	0.08
Gr Unmin [2]	281	1.37	3.13	2.69	0.01	0.11	0.04
Gr Min [3]	212	1.51	4.34	2.81	0.05	0.22	0.08
Gypsum	424	1.62	4.41	2.80	0.08	0.28	0.10
Lapilli Tuff	1,108	1.09	4.63	2.61	0.03	0.16	0.06
LG Breccia	378	2.02	4.35	2.94	0.16	0.40	0.14
Rhyolite	412	1.28	3.37	2.49	0.02	0.14	0.06
Sediments	19	2.34	2.63	2.51	0.01	0.08	0.03
T-Rhyolite	185	1.65	4.42	2.66	0.03	0.17	0.06
Tuff	774	1.46	3.77	2.44	0.02	0.15	0.06
VMS	1,328	1.31	6.05	3.51	0.45	0.67	0.19

Notes:
1. BxHy – Hydrothermal Breccia
2. Gr Unmin - Grainstone outside Indicator shell
3. Gr Min - Grainstone inside Indicator shell
4. SD – Standard Deviation
5. CV – Coefficient of Variation

Figure 14.3 – Correlation Matrix Between Fe (%) and Density



14.5 Exploratory Data Analysis – Assays

14.5.1 Mineral Resource Metals

The first step in developing a block model estimate after completing the 3D solid models is to assess the assay data contained inside the solid models and to determine whether any additional domaining is required prior to the compositing. Typically, raw assay data are extracted from each domain and assessed using histograms and cumulative probability plots. Table 14.4 and Table 14.5 show descriptive statistics for length weighted uncapped and capped gold, silver, copper, lead, and zinc assays, respectively, within the various domains.

Table 14.4 – Summary Uncapped Assay Statistics

	Domain	Count	Min	Max	Mean	Variance	SD	CV
Gold (g/t)	VMS	2,477	0.00	94.00	2.84	35.12	5.93	2.08
	Gr Min[1]	433	0.00	42.70	1.57	6.68	2.59	1.65
	Gr UnMin[2]	483	0.00	4.09	0.08	0.02	0.15	1.94
	LG Breccia	1,347	0.00	25.90	0.50	1.83	1.35	2.71
Silver (g/t)	VMS	2,477	0.00	2,168.00	54.53	17,269.37	131.41	2.41
	Gr Min[1]	433	0.00	675.00	30.65	4,880.76	69.86	2.28
	Gr UnMin[2]	483	0.00	102.90	1.57	16.97	4.12	2.62
	LG Breccia	1,347	0.00	763.00	15.96	1,939.56	44.04	2.76
Copper (%)	VMS	2,477	0.00	23.30	2.30	12.77	3.57	1.56
	Gr Min[1]	433	0.00	23.63	0.84	4.66	2.16	2.58
	Gr UnMin[2]	483	0.00	1.46	0.02	0.00	0.05	2.36
	LG Breccia	1,347	0.00	4.49	0.17	0.14	0.38	2.31
Lead (%)	VMS	2,477	0.00	17.20	0.31	1.09	1.04	3.34
	Gr Min [1]	433	0.00	2.07	0.12	0.06	0.25	2.05
	Gr UnMin[2]	483	0.00	0.71	0.01	0.00	0.03	2.93
	LG Breccia	1,347	0.00	9.32	0.09	0.06	0.25	2.89
Zinc (%)	VMS	2,477	0.00	52.75	3.08	50.99	7.14	2.32
	Gr Min[1]	433	0.00	34.42	0.97	6.71	2.59	2.68
	Gr UnMin[2]	483	0.00	1.59	0.05	0.01	0.11	2.34
	LG Breccia	1,347	0.00	44.23	0.74	2.79	1.67	2.25

Notes:
1. Gr Min - Grainstone inside Indicator shell
2. Gr Unmin - Grainstone outside Indicator shell

Table 14.5 – Summary Capped Assay Statistics

	Domain	Count	Min	Max	Mean	Variance	SD	CV
Gold (g/t)	VMS	2,477	0.00	35.00	2.76	27.44	5.24	1.90
	Gr Min[1]	433	0.00	11.00	1.48	4.15	2.04	1.38
	Gr UnMin[2]	483	0.00	0.80	0.08	0.02	0.14	1.85
	LG Breccia	1,347	0.00	7.00	0.46	0.85	0.92	2.02
Silver (g/t)	VMS	2,477	0.00	650.00	51.37	11,304.44	106.32	2.07
	Gr Min[1]	433	0.00	260.00	27.52	2,524.74	50.25	1.83
	Gr UnMin[2]	483	0.00	23.00	1.49	11.74	3.43	2.29
	LG Breccia	1,347	0.00	230.00	14.88	1,120.72	33.48	2.25
Copper (%)	VMS	2,477	0.00	15.00	2.26	11.70	3.42	1.51
	Gr Min[1]	433	0.00	7.00	0.73	1.99	1.41	1.94
	Gr UnMin[2]	483	0.00	1.46	0.02	0.00	0.05	2.36
	LG Breccia	1,347	0.00	3.00	0.16	0.12	0.35	2.16
Lead (%)	VMS	2,477	0.00	7.00	0.30	0.79	0.89	2.99
	Gr Min [1]	433	0.00	1.20	0.11	0.04	0.20	1.78
	Gr UnMin[2]	483	0.00	0.71	0.01	0.00	0.03	2.93
	LG Breccia	1,347	0.00	1.40	0.08	0.03	0.17	2.02
Zinc (%)	VMS	2,477	0.00	35.00	2.97	42.84	6.54	2.20
	Gr Min[1]	433	0.00	10.00	0.85	2.62	1.62	1.90
	Gr UnMin[2]	483	0.00	0.50	0.04	0.01	0.08	1.87
	LG Breccia	1,347	0.00	6.00	0.69	0.84	0.92	1.34

Notes:
1. Gr Min - Grainstone inside Indicator shell
2. Gr Unmin - Grainstone outside Indicator shell

Extreme high-grade values, commonly called “outliers”, can lead to overestimation of grade in the block model. Histograms and probability plots were generated for each population; a review determined that grade capping was required for most metals and domains.

Table 14.6 shows the assay grade capping statistics, the selected capping values, and the theoretical metal loss. Typically, a metal loss of greater than 20% indicates that the capping level may be too severe as a significant amount of contained metal would be lost.

Table 14.6 – Assay Grade Capping and Metal Loss

	Domain	Number of Assays	Raw Assay Mean	Raw Assay Cap	Capped Assay Mean	Number of Assays Capped	Metal Loss (%)
Gold (g/t)	VMS	2,482	2.81	35.0	2.75	16	3
	Gr Min[1]	433	1.57	11.0	1.48	9	6
	Gr UnMin[2]	483	0.08	0.8	0.08	11	0
	LG Breccia	1,344	0.50	7.0	0.46	10	8
Silver (g/t)	VMS	2,482	53.87	650	51.06	23	5
	Gr Min[1]	433	30.65	260	27.52	8	10
	Gr UnMin[2]	483	1.57	23	1.49	8	5
	LG Breccia	1,344	15.96	230	14.88	13	7
Copper (%)	VMS	2,482	2.30	15.0	2.26	35	1
	Gr Min[1]	433	0.84	7.0	0.73	11	13
	Gr UnMin[2]	483	0.02	NA	0.02	0	0
	LG Breccia	1,344	0.17	3.0	0.16	6	0
Lead (%)	VMS	2,482	0.30	7.0	0.29	13	3
	Gr Min [1]	433	0.12	1.2	0.11	5	8
	Gr UnMin[2]	483	0.01	NA	0.01	0	0
	LG Breccia	1,344	0.09	1.4	0.09	8	11
Zinc (%)	VMS	2,482	3.07	35.0	2.96	38	3
	Gr Min[1]	433	0.97	10.0	0.85	8	12
	Gr UnMin[2]	483	0.05	0.5	0.04	10	20
	LG Breccia	1,344	0.74	6.0	0.69	11	7

Notes:
1. Gr Min - Grainstone inside Indicator shell
2. Gr Unmin - Grainstone outside Indicator shell

14.5.2 Deleterious Elements

As part of the FS, DRA requested that SLR update the Mineral Resource block model with estimates for seven (7) deleterious elements: arsenic, bismuth, cadmium, iron, mercury, sulphur, and antimony.

14.5.2.1 Metallurgical Zonation

El Domo metallurgical test work considered three (3) types of concentrates for the flotation recoveries based on the metal ratio of copper to the sum of lead plus zinc (metal ratio = Cu/(Pb + Zn)). SLR used the metal zonation as additional domains for spatial analysis since the three (3) types of concentrates have different penalizable levels of all deleterious elements.

The blocks within the block model were flagged as one of the three (3) types of mineralisation, based on the metal ratio of copper to the sum of lead and zinc. Blocks with a metal ratio less than 0.33 were assigned Zn-type mineralization – Type 1; blocks with a metal ratio between 0.33 and 3.0 were assigned Mixed Cu-Zn mineralization – Type 2; and blocks with a metal ratio greater than 3.0 were assigned Cu-type mineralization – Type 3.

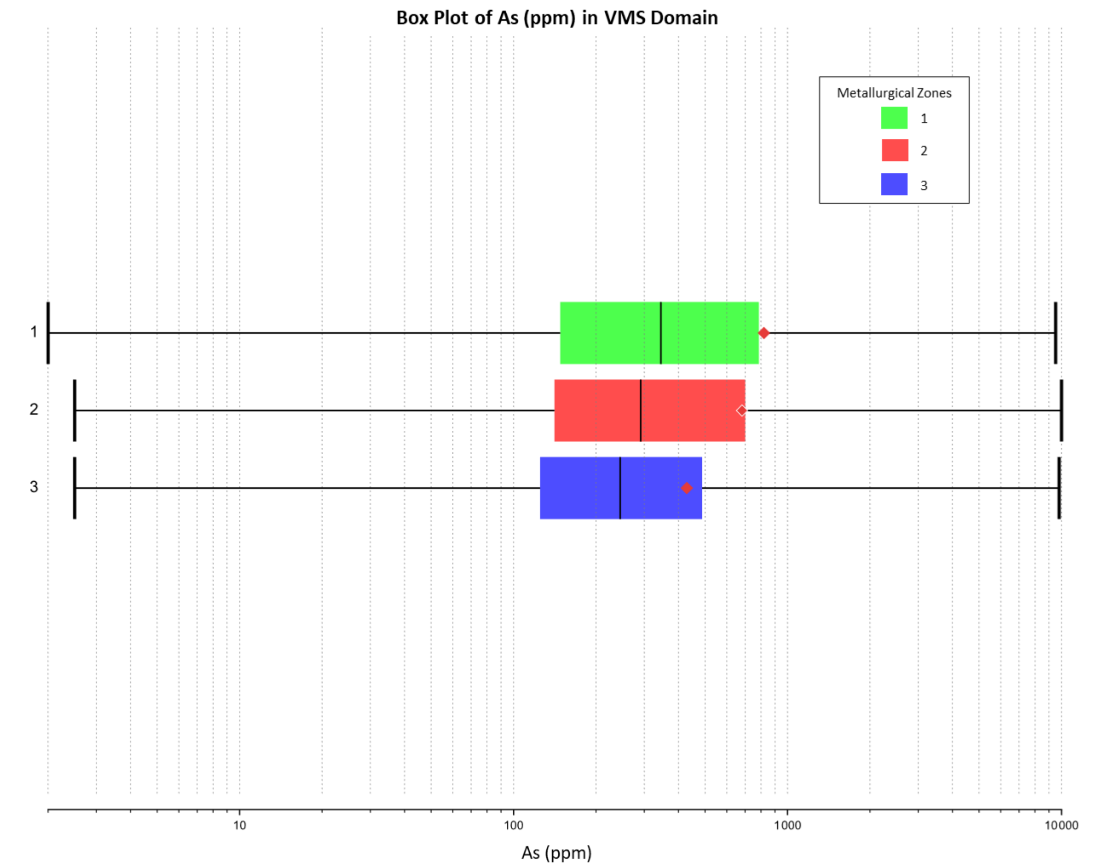
Assay values located inside the mineralised wireframes were tagged with domain identifiers. Additionally, assays were assigned ore type flagging and exported for statistical analysis. Histograms, probability, and box plots were generated for each population to assist in verifying the modelling process.

Figure 14.4 to Figure 14.10**Error! Reference source not found.** show box plots of all the deleterious elements for the three (3) metallurgical zones within the VMS domain.

While capping or cutting of high grades is usually used as a risk mitigation strategy for elements of economic interest, higher concentrations of deleterious elements will result in negative impacts on project economics. For the purpose of deleterious element estimation, it is common to leave the input data uncapped, resulting in a more conservative outcome, assuming that the outliers are not a result of analytical error. As a result, SLR did not apply capping in the estimation of the deleterious elements.

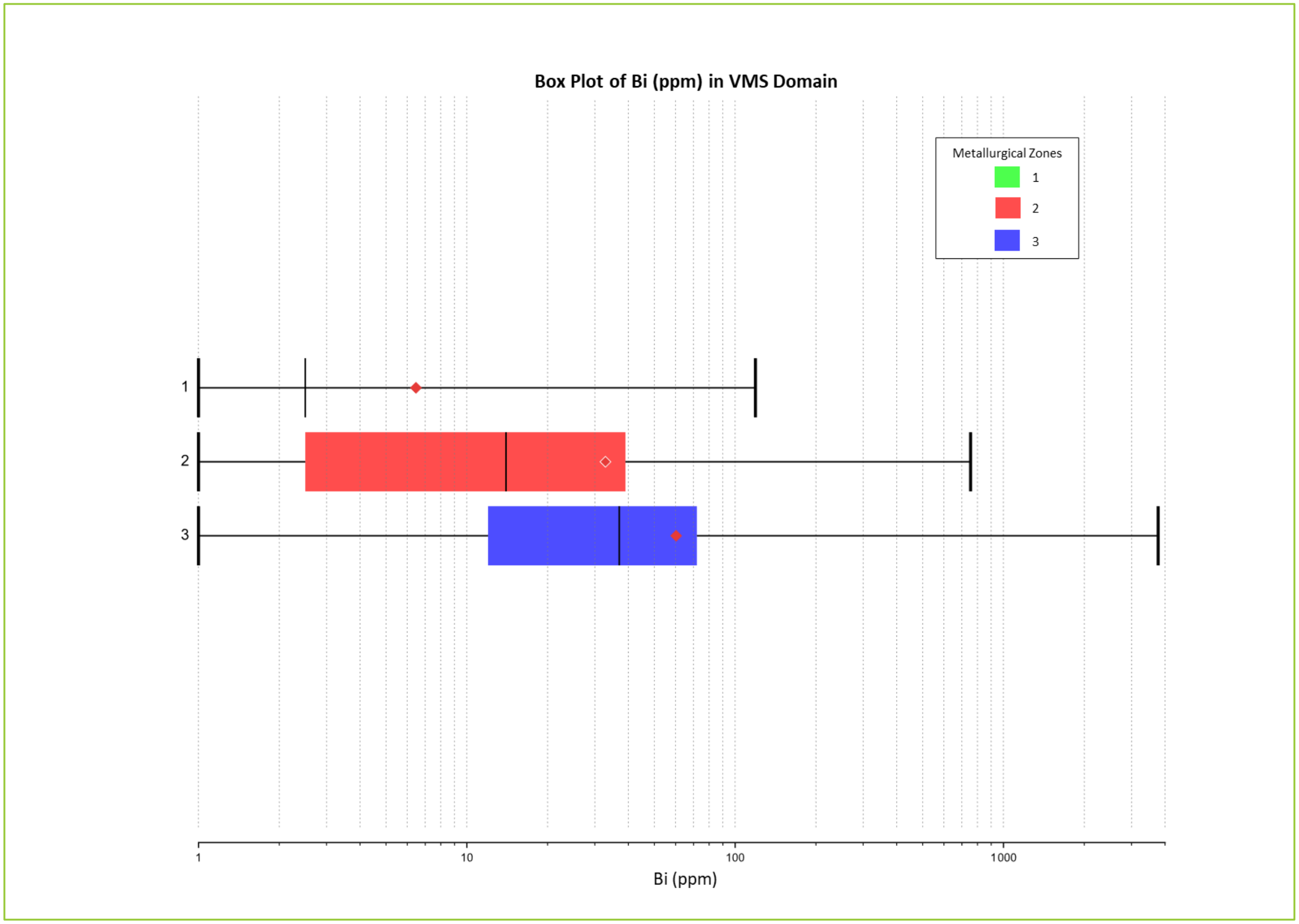
Table 14.7 to Table 14.10 present assay statistics for arsenic, bismuth, cadmium, iron, mercury, sulphur, and antimony for the four (4) grade domains and three (3) metallurgical zones.

Figure 14.4 – Box Plot of Arsenic (ppm) Assay Data



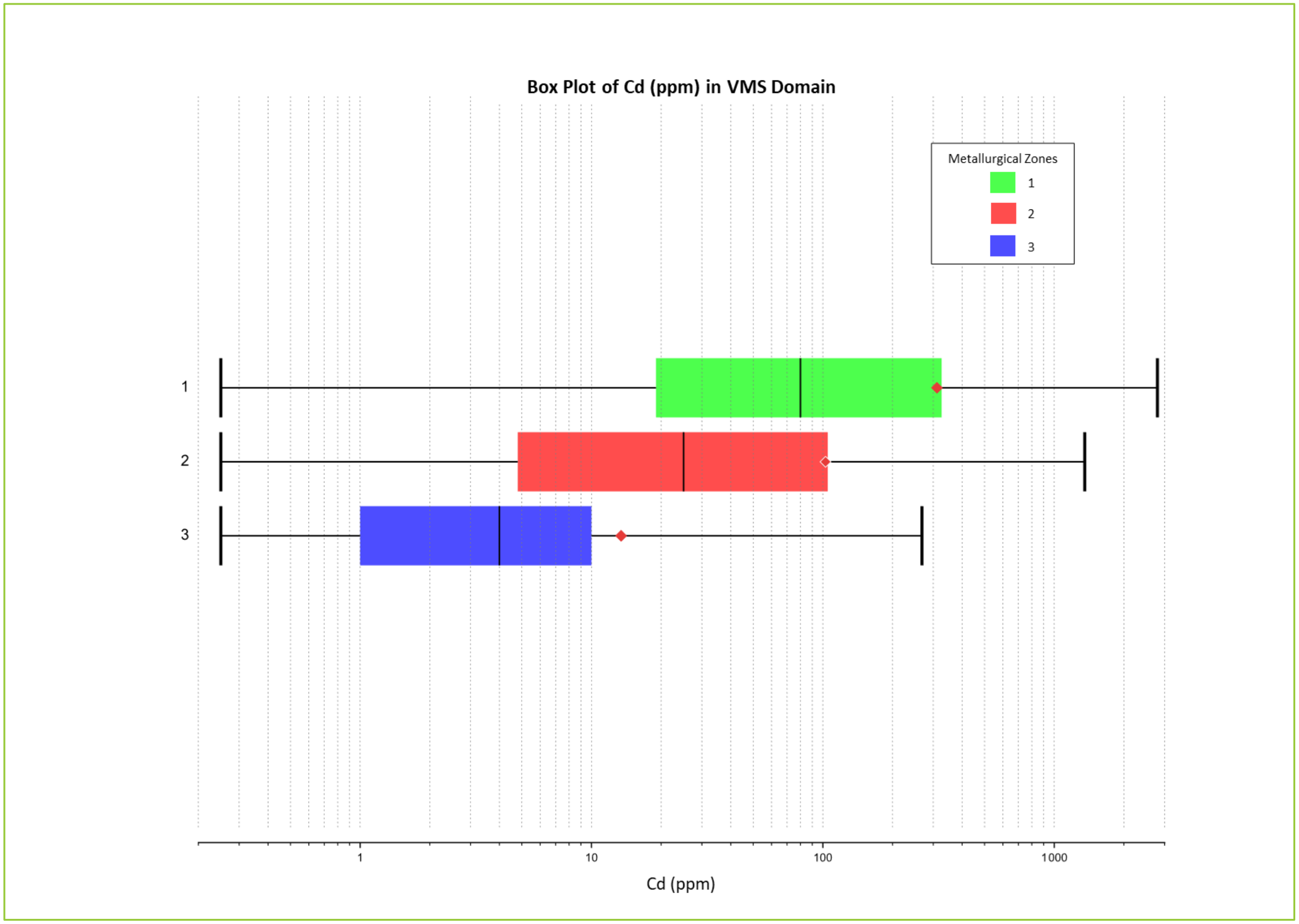
Source: SLR, 2021

Figure 14.5 – Box Plot of Bismuth (ppm) Assay Data



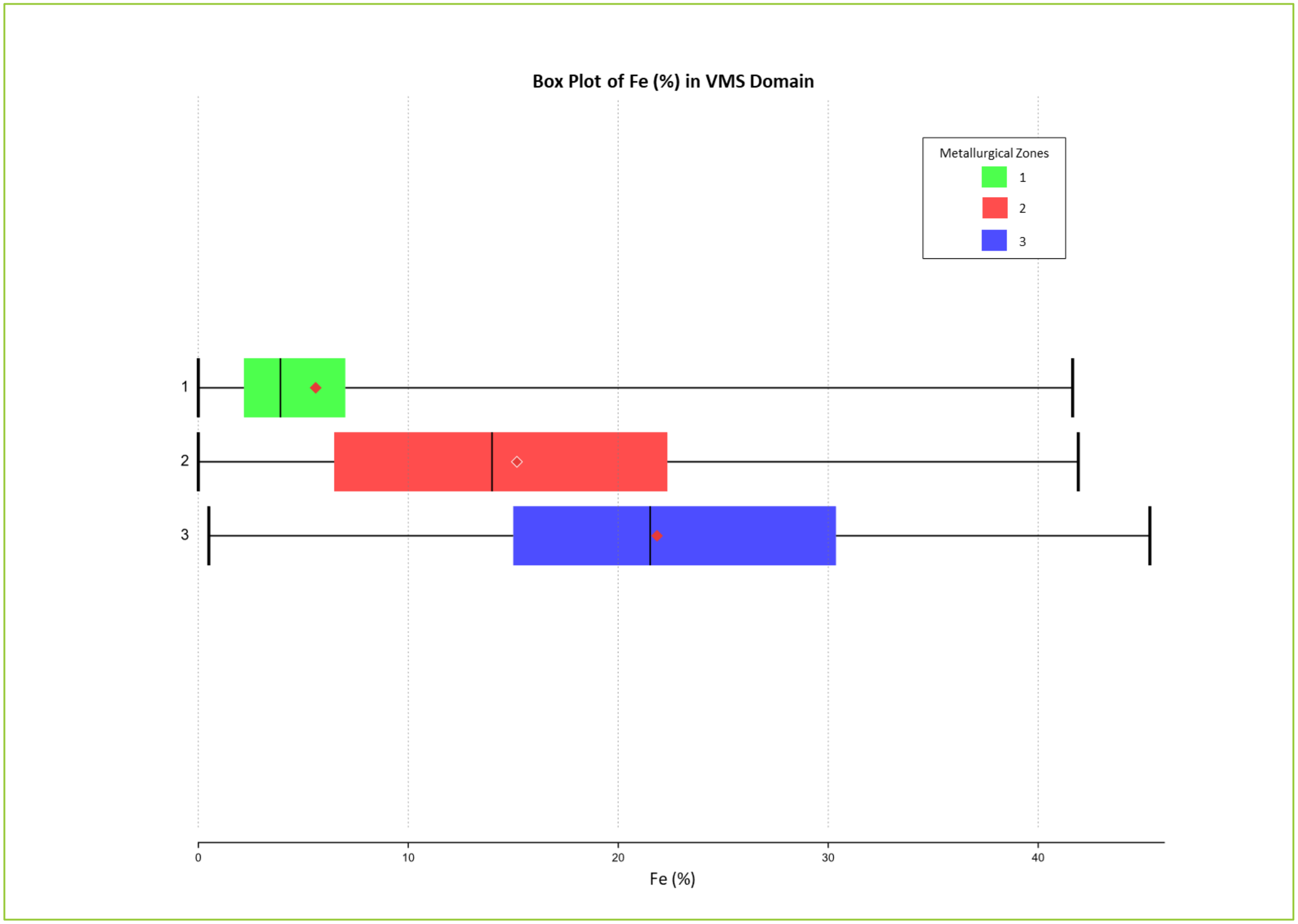
Source: SLR, 2021

Figure 14.6 – Box Plot of Cadmium (ppm) Assay Data



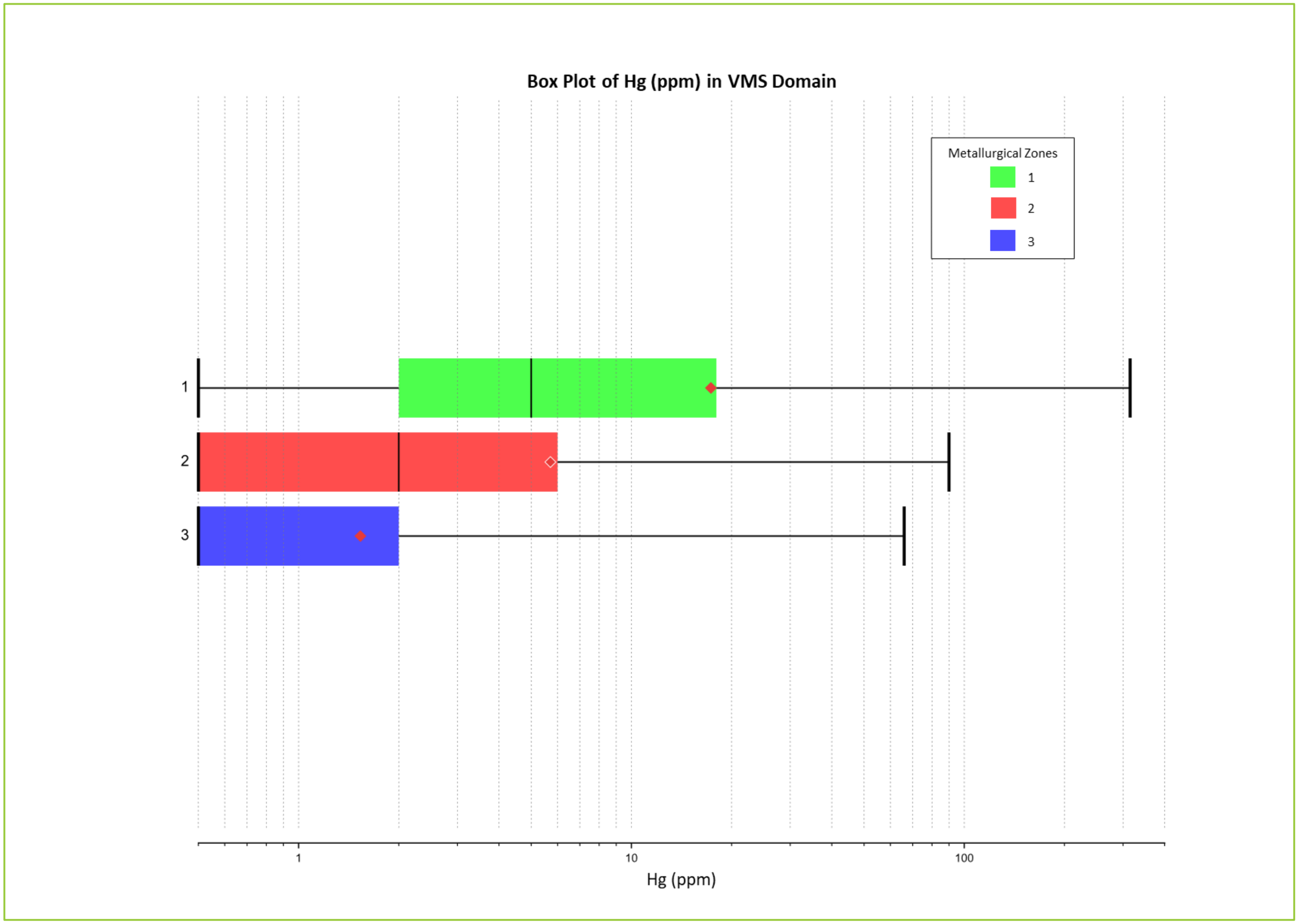
Source: SLR, 2021

Figure 14.7 – Box Plot of Iron (%) Assay Data



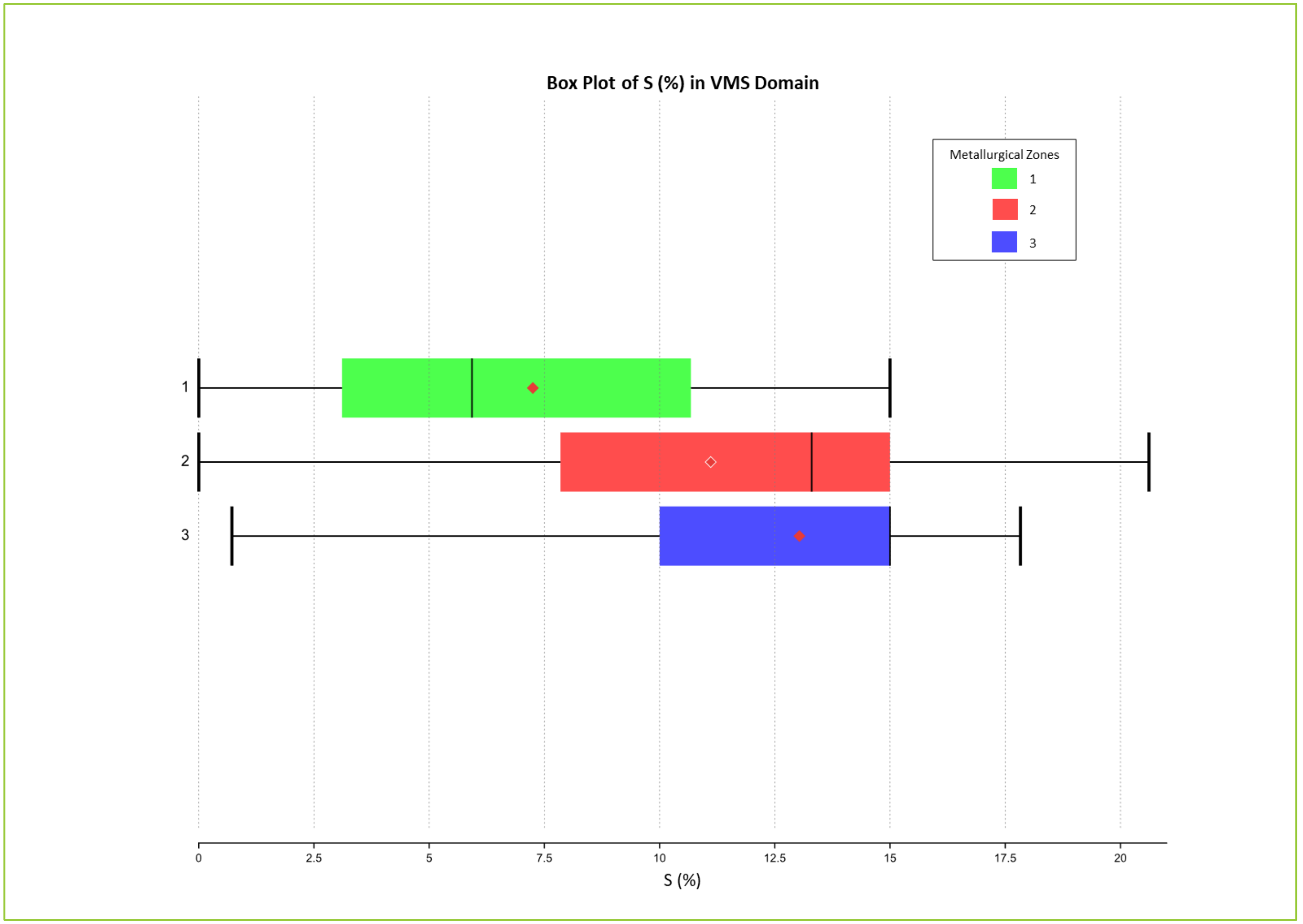
Source: SLR, 2021

Figure 14.8 – Box Plot of Mercury (ppm) Assay Data



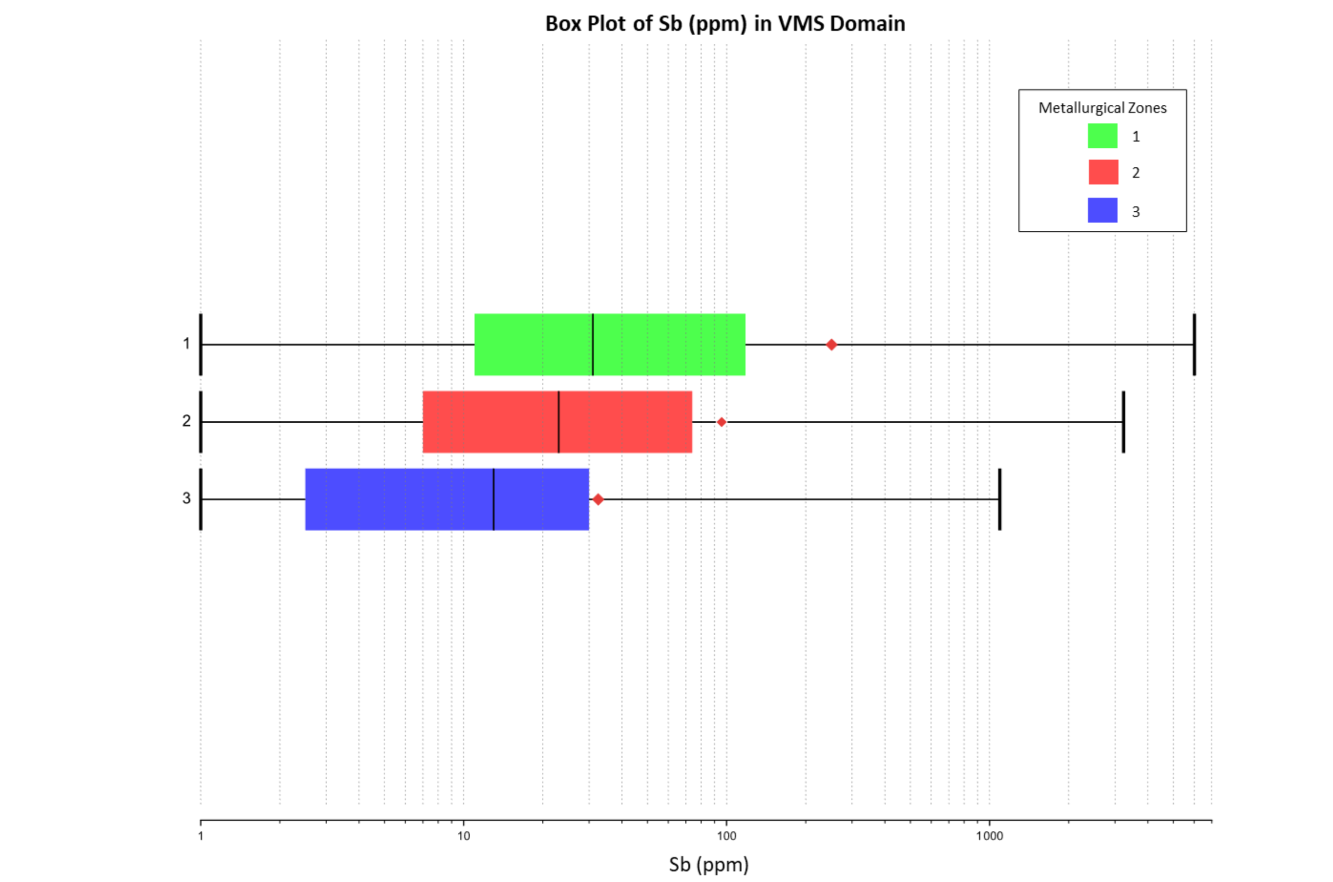
Source: SLR, 2021

Figure 14.9 – Box Plot of Sulphur (%) Assay Data



Source: SLR, 2021

Figure 14.10 – Box Plot of Antimony (ppm) Assay Data



Source: SLR, 2021

Table 14.7 – Summary Deleterious Elements Assays Statistics – VMS Domain

Zone	Statistics	As (ppm)	Bi (ppm)	Cd (ppm)	Fe (%)	Hg (ppm)	S (%)	Sb (ppm)
All	Count	2,472	2,472	2,471	2,471	2,471	2,471	2,472
	Min	2.00	1.00	0.25	0.00	0.50	0.00	1.00
	Max	10,000.00	3,770.00	2,791.00	45.32	315.00	20.62	6,015.00
	Mean	643.63	33.47	141.13	14.29	8.14	10.50	126.97
	Variance	1,149,776.33	10,153.29	114,042.56	125.40	401.47	23.48	166,824.85
	SD	1,072.28	100.76	337.70	11.20	20.04	4.85	408.44
	CV	1.67	3.01	2.39	0.78	2.46	0.46	3.22
1 – Zinc Zone	Count	801	801	801	801	801	801	801
	Min	2.00	1.00	0.25	0.00	0.50	0.00	1.00
	Max	9,519.00	119.00	2,791.00	41.64	315.00	15.00	6,015.00
	Mean	825.19	6.45	312.05	5.58	17.43	7.25	256.24
	Variance	1,757,463.06	158.04	264,839.48	31.19	964.32	23.94	415,526.02
	SD	1,325.69	12.57	514.63	5.58	31.05	4.89	644.61
	CV	1.61	1.95	1.65	1.00	1.78	0.67	2.52
2 – Mixed Zone	Count	821	821	821	821	821	821	821
	Min	2.50	1.00	0.25	0.00	0.50	0.00	1.00
	Max	10,000.00	754.00	1,355.00	41.91	90.00	20.62	3,234.00
	Mean	682.99	32.84	102.83	15.16	5.72	11.11	95.88
	Variance	1,230,904.26	3,064.33	34,429.50	99.14	112.78	21.08	59,748.20
	SD	1,109.46	55.36	185.55	9.96	10.62	4.59	244.43
	CV	1.62	1.69	1.80	0.66	1.86	0.41	2.55
3 – Copper Zone	Count	850	850	849	849	849	849	850
	Min	2.50	1.00	0.25	0.50	0.50	0.72	1.00
	Max	9,801.00	3,770.00	268.00	45.32	66.00	17.83	1,094.00
	Mean	428.80	60.28	13.43	21.85	1.53	13.03	32.49
	Variance	404,378.67	25,446.86	908.71	111.50	9.61	8.43	6,185.17
	SD	635.91	159.52	30.14	10.56	3.10	2.90	78.65
	CV	1.48	2.65	2.24	0.48	2.02	0.22	2.42

Table 14.8 – Summary Deleterious Elements Assays Statistics – Mineralized Grainstone Domain

Zone	Statistics	As (ppm)	Bi (ppm)	Cd (ppm)	Fe (%)	Hg (ppm)	S (%)	Sb (ppm)
All	Count	431	431	430	430	430	430	431
	Min	2.50	1.00	0.25	0.61	0.50	0.01	1.50
	Max	10,000.00	238.00	1,466.00	33.17	105.00	15.00	2,834.00
	Mean	462.55	10.34	44.76	6.22	4.39	4.53	57.40
	Variance	837,902.63	639.61	14,394.62	14.70	73.42	15.41	32,783.61
	SD	915.37	25.29	119.98	3.83	8.57	3.93	181.06
	CV	1.98	2.44	2.68	0.62	1.95	0.87	3.15
1 – Zinc Zone	Count	173	173	173	173	173	173	173
	Min	2.50	1.00	0.25	0.61	0.50	0.02	1.50
	Max	7,050.00	22.00	1,466.00	19.50	105.00	15.00	2,694.00
	Mean	631.36	3.75	39.96	4.95	5.15	4.64	40.96
	Variance	903,337.14	12.49	17,795.29	8.88	78.20	11.19	31,451.61
	SD	950.44	3.53	133.40	2.98	8.84	3.35	177.35
	CV	1.51	0.94	3.34	0.60	1.72	0.72	4.33
2 – Mixed Zone	Count	220	220	220	220	220	220	220
	Min	2.50	1.00	0.25	1.21	0.50	0.01	1.50
	Max	10,000.00	238.00	1,355.00	24.68	90.00	15.00	2,834.00
	Mean	358.73	12.46	50.80	6.50	4.17	4.23	61.63
	Variance	874,629.92	858.75	13,727.36	8.37	80.05	16.46	31,772.57
	SD	935.22	29.30	117.16	2.89	8.95	4.06	178.25
	CV	2.61	2.35	2.31	0.45	2.14	0.96	2.89
3 – Copper Zone	Count	38	38	37	37	37	37	38
	Min	25.00	2.50	0.50	1.65	0.50	0.25	2.50
	Max	2,696.00	203.00	266.00	33.17	14.00	15.00	1,091.00
	Mean	325.80	27.06	30.51	10.29	2.25	5.83	105.15
	Variance	162,021.05	1,686.45	3,392.46	55.44	8.86	26.87	43,349.52
	SD	402.52	41.07	58.24	7.45	2.98	5.18	208.21
	CV	1.24	1.52	1.91	0.72	1.32	0.89	1.98

Table 14.9 – Summary Deleterious Elements Assays Statistics – Unmineralised Grainstone Domain

Zone	Statistics	As (ppm)	Bi (ppm)	Cd (ppm)	Fe (%)	Hg (ppm)	S (%)	Sb (ppm)
All	Count	434	434	434	434	434	434	434
	Min	1.00	1.00	0.25	0.49	0.50	0.01	1.00
	Max	2,620.00	54.00	79.00	33.74	25.00	15.00	107.00
	Mean	143.94	2.69	2.78	3.89	1.05	1.44	5.95
	Variance	65,117.16	2.95	48.28	2.44	4.12	4.02	78.62
	SD	255.18	1.72	6.95	1.56	2.03	2.01	8.87
	CV	1.77	0.64	2.50	0.40	1.94	1.39	1.49
1 – Zinc Zone	Count	218	218	218	218	218	218	218
	Min	1.00	1.00	0.25	0.49	0.50	0.01	1.00
	Max	2,620.00	9.00	79.00	7.41	25.00	15.00	80.00
	Mean	203.05	2.50	3.59	3.39	1.30	1.71	6.61
	Variance	97,892.29	1.52	85.00	2.16	6.70	4.14	92.79
	SD	312.88	1.23	9.22	1.47	2.59	2.03	9.63
	CV	1.54	0.49	2.57	0.43	1.99	1.19	1.46
2 – Mixed Zone	Count	207	207	207	207	207	207	207
	Min	1.00	1.00	0.25	0.76	0.50	0.01	1.00
	Max	1,114.00	54.00	25.00	30.02	14.00	15.00	107.00
	Mean	78.99	2.87	1.93	4.40	0.75	1.11	5.11
	Variance	22,524.01	4.24	7.74	2.08	1.22	3.76	61.50
	SD	150.08	2.06	2.78	1.44	1.10	1.94	7.84
	CV	1.90	0.72	1.45	0.33	1.46	1.75	1.53
3 – Copper Zone	Count	9	9	9	9	9	9	9
	Min	25.00	2.50	0.50	3.47	0.50	0.25	2.50
	Max	397.00	48.00	5.00	33.74	3.00	15.00	26.00
	Mean	120.59	3.65	1.36	5.52	1.26	2.16	9.40
	Variance	19,491.37	11.96	2.11	5.71	1.18	3.31	107.42
	SD	139.61	3.46	1.45	2.39	1.09	1.82	10.36
	CV	1.16	0.95	1.07	0.43	0.86	0.84	1.10

Table 14.10 – Summary Deleterious Elements Assays Statistics – LG Breccia Domain

Zone	Statistics	As (ppm)	Bi (ppm)	Cd (ppm)	Fe (%)	Hg (ppm)	S (%)	Sb (ppm)
All	Count	1,335	1,335	1,335	1,335	1,335	1,335	1,335
	Min	2.00	1.00	0.25	0.00	0.50	0.01	1.00
	Max	5,728.00	85.00	2,176.00	34.74	55.00	15.00	1,941.00
	Mean	177.13	3.51	32.24	2.92	1.89	4.08	27.07
	Variance	63,287.35	32.30	6,536.17	17.97	14.65	17.09	7,071.55
	SD	251.57	5.68	80.85	4.24	3.83	4.13	84.09
	CV	1.42	1.62	2.51	1.45	2.02	1.01	3.11
1 – Zinc Zone	Count	1,016	1,016	1,016	1,016	1,016	1,016	1,016
	Min	2.50	1.00	0.25	0.00	0.50	0.01	1.00
	Max	3,589.00	50.00	2,176.00	21.53	47.00	15.00	1,941.00
	Mean	185.52	2.65	38.94	1.77	1.94	2.99	26.74
	Variance	61,460.75	2.89	8,240.09	3.68	11.97	9.53	6,631.01
	SD	247.91	1.70	90.77	1.92	3.46	3.09	81.43
	CV	1.34	0.64	2.33	1.09	1.79	1.03	3.04
2 – Mixed Zone	Count	226	226	226	226	226	226	226
	Min	2.00	1.00	0.25	0.00	0.50	0.01	1.00
	Max	2,010.00	85.00	150.00	34.74	55.00	15.00	1,024.00
	Mean	166.78	5.48	13.72	6.07	2.17	7.13	32.66
	Variance	49,534.18	96.67	465.28	45.41	31.97	26.11	10,028.42
	SD	222.56	9.83	21.57	6.74	5.65	5.11	100.14
	CV	1.33	1.79	1.57	1.11	2.61	0.72	3.07
3 – Copper Zone	Count	93	93	93	93	93	93	93
	Min	8.00	1.00	0.25	1.04	0.50	0.29	1.00
	Max	5,728.00	75.00	13.00	34.74	10.00	15.00	717.00
	Mean	108.16	8.37	1.94	8.12	0.74	8.84	17.00
	Variance	113,663.64	166.89	6.55	46.08	1.15	21.67	4,778.49
	SD	337.14	12.92	2.56	6.79	1.07	4.66	69.13
	CV	3.12	1.54	1.32	0.84	1.46	0.53	4.07

14.6 Exploratory Data Analysis – Composites

14.6.1 Mineral Resource Metals

Prior to grade interpolation, the assay data within each of the individual mineralised domains were combined into two metre downhole composites. Table 14.11 shows composite statistics for gold, silver, copper, lead, and zinc for the four grade domains.

Table 14.11 – Summary Composite Statistics

	Domain	Count	Min	Max	Mean	Variance	SD	CV
Gold (g/t)	VMS	1,363	0.00	33.70	2.76	20.18	4.49	1.63
	Gr Min[1]	258	0.00	8.92	1.48	2.47	1.57	1.06
	Gr UnMin[2]	377	0.00	0.70	0.08	0.01	0.12	1.56
	LG Breccia	917	0.00	7.00	0.46	0.65	0.81	1.77
Silver (g/t)	VMS	1,363	0.00	650.00	51.38	8,254.15	90.85	1.77
	Gr Min[1]	258	0.30	257.42	27.54	1,695.96	41.18	1.50
	Gr UnMin[2]	377	0.00	23.00	1.50	8.95	2.99	2.00
	LG Breccia	917	0.00	230.00	14.89	896.53	29.94	2.01
Copper (%)	VMS	1,363	0.00	15.00	2.26	9.18	3.03	1.34
	Gr Min[1]	258	0.00	6.24	0.73	1.17	1.08	1.49
	Gr UnMin[2]	377	0.00	0.41	0.02	0.00	0.04	1.79
	LG Breccia	917	0.00	3.00	0.16	0.09	0.29	1.81
Lead (%)	VMS	1,363	0.00	6.81	0.30	0.55	0.74	2.50
	Gr Min [1]	258	0.00	1.13	0.11	0.03	0.17	1.47
	Gr UnMin[2]	377	0.00	0.22	0.01	0.00	0.02	2.64
	LG Breccia	917	0.00	1.40	0.08	0.02	0.14	1.70
Zinc (%)	VMS	1,363	0.00	35.00	2.97	31.89	5.65	1.90
	Gr Min[1]	258	0.01	7.63	0.85	1.51	1.23	1.44
	Gr UnMin[2]	377	0.00	0.50	0.04	0.01	0.08	1.73
	LG Breccia	917	0.00	6.00	0.69	0.63	0.79	1.15

Notes:
1. Gr Min - Grainstone inside Indicator shell
2. Gr Unmin - Grainstone outside Indicator shell

14.6.2 Deleterious Elements

As for the Mineral Resource metals, the assay data within each of the individual mineralised domains were combined into two metre downhole composites. Table 14.12 to Table 14.15 present composite statistics for arsenic, bismuth, cadmium, iron, mercury, sulphur, and antimony for the four (4) grade domains and three (3) metallurgical zones.

Table 14.12 – Summary Composite Statistics – VMS Domain

Zone	Statistics	As (ppm)	Bi (ppm)	Cd (ppm)	Fe (%)	Hg (ppm)	S (%)	Sb (ppm)
All	Count	1,360	1,360	1,360	1,360	1,360	1,360	1,360
	Min	2.50	1.00	0.25	0.00	0.50	0.06	1.00
	Max	8,818.06	3,567.32	2,372.61	43.75	219.37	19.12	4,127.67
	Mean	643.76	33.49	141.13	14.29	8.13	10.49	126.95
	Variance	802,585.02	8,945.19	86,890.76	114.88	293.42	20.34	109,218.03
	SD	895.87	94.58	294.77	10.72	17.13	4.51	330.48
	CV	1.39	2.82	2.09	0.75	2.11	0.43	2.60
1 – Zinc Zone	Count	440	440	440	440	440	440	440
	Min	5.67	1.00	0.41	0.00	0.50	0.06	1.00
	Max	7,400.87	103.17	2,372.61	39.96	219.37	15.00	4,127.67
	Mean	865.43	6.87	302.75	5.73	17.39	7.30	257.33
	Variance	1,343,624.99	144.57	194,604.26	27.68	682.58	19.23	266,986.72
	SD	1,159.15	12.02	441.14	5.26	26.13	4.38	516.71
	CV	1.34	1.75	1.46	0.92	1.50	0.60	2.01
2 – Mixed Zone	Count	470	470	470	470	470	470	470
	Min	2.50	1.00	0.32	0.00	0.50	0.16	1.50
	Max	8,818.06	303.91	1,323.40	40.74	77.05	15.00	1,667.40
	Mean	667.41	32.16	111.47	14.92	5.88	11.08	96.81
	Variance	791,218.35	1,823.50	27,737.90	85.60	77.76	17.80	37,032.76
	SD	889.50	42.70	166.55	9.25	8.82	4.22	192.44
	CV	1.33	1.33	1.49	0.62	1.50	0.38	1.99
3 – Copper Zone	Count	450	450	450	450	450	450	450
	Min	5.11	1.50	0.25	0.82	0.50	0.86	1.00
	Max	3,363.17	3,567.32	274.67	43.75	25.02	19.12	716.12
	Mean	401.95	60.96	13.82	22.00	1.42	13.01	30.74
	Variance	180,820.75	23,589.27	772.05	99.51	4.37	7.50	3,682.60
	SD	425.23	153.59	27.79	9.98	2.09	2.74	60.68
	CV	1.06	2.52	2.01	0.45	1.47	0.21	1.97

Table 14.13 – Summary Composite Statistics - Mineralized Grainstone Domain

Zone	Statistics	As (ppm)	Bi (ppm)	Cd (ppm)	Fe (%)	Hg (ppm)	S (%)	Sb (ppm)
All	Count	270	270	270	270	270	270	270
	Min	11.82	1.00	0.37	0.67	0.50	0.24	1.50
	Max	6,702.35	139.33	730.08	23.52	47.02	15.00	845.93
	Mean	468.04	9.94	43.26	6.13	4.37	4.53	56.78
	Variance	539,965.42	322.74	6,704.83	9.29	41.15	10.90	12,998.54
	SD	734.82	17.97	81.88	3.05	6.41	3.30	114.01
	CV	1.57	1.81	1.89	0.50	1.47	0.73	2.01
1 – Zinc Zone	Count	106	106	106	106	106	106	106
	Min	13.00	1.00	0.38	0.67	0.50	0.25	1.50
	Max	6,702.35	58.78	730.08	18.85	36.00	14.65	845.93
	Mean	640.79	4.41	38.43	4.83	5.11	4.67	45.20
	Variance	663,984.48	42.65	8,255.13	7.11	37.58	8.59	13,506.53
	SD	814.85	6.53	90.86	2.67	6.13	2.93	116.22
	CV	1.27	1.48	2.36	0.55	1.20	0.63	2.57
2 – Mixed Zone	Count	143	143	143	143	143	143	143
	Min	11.82	1.00	0.37	2.73	0.50	0.24	1.50
	Max	5,492.05	139.33	484.00	15.57	47.02	14.76	719.00
	Mean	361.72	11.90	48.69	6.63	4.15	4.32	61.55
	Variance	479,568.31	410.90	6,305.52	5.98	48.13	11.80	12,426.35
	SD	692.51	20.27	79.41	2.45	6.94	3.44	111.47
	CV	1.91	1.70	1.63	0.37	1.67	0.79	1.81
3 – Copper Zone	Count	21	21	21	21	21	21	21
	Min	26.36	2.50	0.65	4.58	0.50	0.38	2.50
	Max	1,272.67	109.46	141.39	23.52	8.65	15.00	387.12
	Mean	324.06	24.73	29.99	9.33	2.10	5.25	83.03
	Variance	106,207.78	793.49	1,478.73	24.63	4.53	16.94	14,262.73
	SD	325.90	28.17	38.45	4.96	2.13	4.12	119.43
	CV	1.01	1.14	1.28	0.53	1.01	0.78	1.44

Table 14.14 – Summary Composite Statistics - Unmineralised Grainstone Domain

Zone	Statistics	As (ppm)	Bi (ppm)	Cd (ppm)	Fe (%)	Hg (ppm)	S (%)	Sb (ppm)
All	Count	290	290	290	290	290	290	290
	Min	1.17	1.00	0.25	0.63	0.50	0.01	1.00
	Max	2,296.44	16.00	79.00	7.75	24.00	13.99	62.48
	Mean	143.63	2.68	2.79	3.88	1.04	1.44	5.91
	Variance	55,143.19	2.01	45.22	1.92	3.42	3.51	57.81
	SD	234.83	1.42	6.72	1.39	1.85	1.87	7.60
	CV	1.63	0.53	2.41	0.36	1.78	1.30	1.29
1 – Zinc Zone	Count	147	147	147	147	147	147	147
	Min	1.17	1.00	0.25	0.63	0.50	0.02	1.00
	Max	2,296.44	8.00	79.00	6.60	24.00	7.29	58.93
	Mean	212.50	2.50	3.58	3.41	1.31	1.72	6.70
	Variance	87,759.30	1.39	81.25	1.87	5.91	3.24	71.50
	SD	296.24	1.18	9.01	1.37	2.43	1.80	8.46
	CV	1.39	0.47	2.52	0.40	1.85	1.05	1.26
2 – Mixed Zone	Count	138	138	138	138	138	138	138
	Min	1.75	1.00	0.25	1.12	0.50	0.01	1.50
	Max	641.36	16.00	17.35	7.75	8.03	13.99	62.48
	Mean	69.01	2.86	1.98	4.36	0.74	1.11	4.94
	Variance	10,677.39	2.62	6.19	1.48	0.64	3.72	41.77
	SD	103.33	1.62	2.49	1.22	0.80	1.93	6.46
	CV	1.50	0.57	1.26	0.28	1.08	1.73	1.31
3 – Copper Zone	Count	5	5	5	5	5	5	5
	Min	27.00	2.24	0.50	3.38	0.50	0.17	2.50
	Max	261.53	6.00	1.78	6.43	2.24	3.29	17.45
	Mean	114.44	3.18	0.88	5.02	1.13	1.88	8.31
	Variance	8,551.56	2.65	0.31	1.84	0.72	1.95	58.03
	SD	92.47	1.63	0.55	1.36	0.85	1.40	7.62
	CV	0.81	0.51	0.63	0.27	0.75	0.74	0.92

Table 14.15 – Summary Composite Statistics – LG Breccia Domain

Zone	Statistics	As (ppm)	Bi (ppm)	Cd (ppm)	Fe (%)	Hg (ppm)	S (%)	Sb (ppm)
All	Count	900	900	900	900	900	900	900
	Min	2.50	1.00	0.25	0.00	0.50	0.02	1.00
	Max	1,906.03	68.21	2,176.00	33.40	44.50	15.00	1,098.15
	Mean	177.32	3.52	32.26	2.91	1.90	4.08	27.10
	Variance	42,176.22	27.38	5,082.01	16.21	11.18	15.84	4,877.47
	SD	205.37	5.23	71.29	4.03	3.34	3.98	69.84
	CV	1.16	1.49	2.21	1.38	1.76	0.98	2.58
1 – Zinc Zone	Count	681	681	681	681	681	681	681
	Min	2.50	1.00	0.41	0.00	0.50	0.03	1.00
	Max	1,906.03	25.00	2,176.00	15.64	30.48	15.00	1,098.15
	Mean	188.02	2.64	39.03	1.78	1.92	3.03	26.69
	Variance	45,028.65	1.53	6,378.06	3.14	8.81	8.68	4,440.88
	SD	212.20	1.24	79.86	1.77	2.97	2.95	66.64
	CV	1.13	0.47	2.05	1.00	1.55	0.97	2.50
2 – Mixed Zone	Count	149	149	149	149	149	149	149
	Min	2.50	1.00	0.25	0.00	0.50	0.02	1.00
	Max	1,201.97	68.21	134.79	33.40	44.50	15.00	576.47
	Mean	169.86	5.55	14.35	5.91	2.29	6.83	33.68
	Variance	41,208.03	86.79	390.32	40.07	26.23	25.18	7,695.63
	SD	203.00	9.32	19.76	6.33	5.12	5.02	87.72
	CV	1.20	1.68	1.38	1.07	2.24	0.73	2.60
3 – Copper Zone	Count	70	70	70	70	70	70	70
	Min	8.84	1.00	0.25	1.03	0.50	0.27	1.00
	Max	365.03	54.55	13.00	29.86	9.82	15.00	414.55
	Mean	84.68	8.02	2.15	8.00	0.82	8.80	17.02
	Variance	6,169.90	127.53	7.12	40.07	1.66	19.01	3,185.89
	SD	78.55	11.29	2.67	6.33	1.29	4.36	56.44
	CV	0.93	1.41	1.24	0.79	1.56	0.50	3.32

14.7 Block Model Parameters

An unrotated block model was created using Leapfrog Edge. The block model coordinates are based on the local UTM grid (Zone 17 S, PSAD 1956). Table 14.16 summarises the block model definition.

Table 14.16 – Block Model Definition

Description	X	Y	Z
Base Point	694,450	9,854,370	570
Parent Block Size (m)	5	5	2.5
Min. Sub-block Size (m)	1.25	1.25	0.625
Number of Parent Blocks	275	311	224
Boundary Size	1,375	1,555	560

14.7.1 Variography and Grade Estimation

Variogram analyses were completed for each domain. The composite data do not yield robust, interpretable experimental grade variograms, which the QP attributes to the thin, but folded and faulted domain shapes for the massive sulphide domain and the small, isolated grade domains in the footwall of the massive sulphide. As a result, the QP elected to apply an ID2 estimator with universal search radii, similar to that used for the 2019 Mineral Resource estimate. SLR did however use the calculated variograms in the validation process which compares ordinary kriging (OK) to ID2.

In order to reproduce the direction of the thin, folded and faulted domains, SLR employed a variable orientation tool in Leapfrog Edge. The tool allows the search to be locally adjusted to the orientation of the mineralization, which results in improved local grade estimates. SLR used the hanging wall and footwall of each domain to guide the variable direction search. Hard boundaries were used to limit the use of composites between different mineralization domains. SLR separated the VMS domain defined in each fault block and treated them as individual units using hard and restrictive semi-soft boundaries during the estimation process. SLR carried out boundary analysis using contact plots for each metal for adjacent fault blocks that contained VMS mineralization. A one metre semi-soft boundary was used between the units where the fault line was arbitrary, composites were straddling faults, and contact plots suggested that the set of composites could belong to either of the units (Figure 14.11).

Table 14.17 summarises search parameters used in estimation of the 2021 Mineral Resource update.

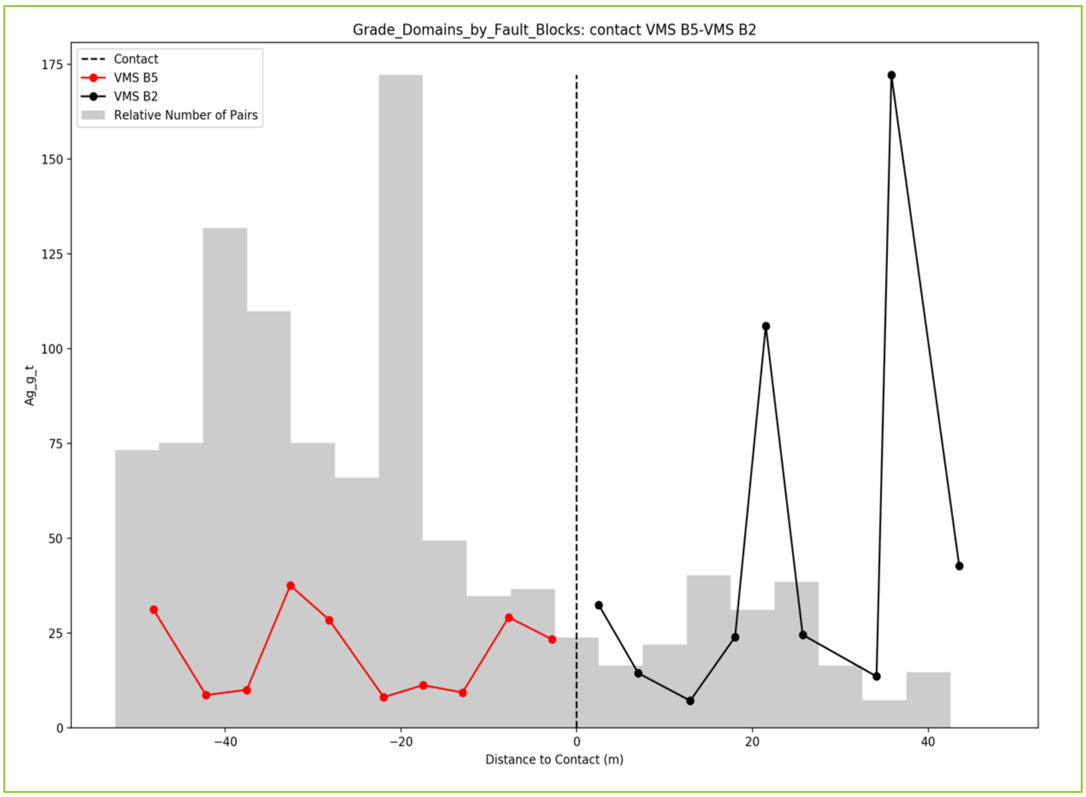
Table 14.17 – Summary of Search Parameters

Search Domain	Parameter	Search Range (m)			High Grade Restriction[3]		
		Pass 1	Pass 2	Pass 3	% of Pass	Limit Au (g/t)/Cu (%)	Range 1/2/3 Pass (m)
Mineral Resource Metals and SG							
VMS	Principal Azimuth	70	140	160	40	20/10	28/28/28
	Principal Dip	70	140	160	20	20/10	28/28/28
	Intermediate Azimuth	5	5	20	18	20/10	2/1/3.6
	Min Comps	3	3	1			
	Max Comps	8	8	10			
	Max Comps/Hole	2	2				
Gr Min[1]	Principal Azimuth	70	140	160	40	5/4	28/28/28
	Principal Dip	70	140	160	20	5/4	28/28/28
	Intermediate Azimuth	5	5	20	18	5/4	2/1/3.6
	Min Comps	3	3	1			
	Max Comps	8	8	10			
	Max Comps/Hole	2	2				
Gr UnMin[1]	Principal Azimuth	70	140	160	40		
	Principal Dip	70	140	160	20		
	Intermediate Azimuth	5	5	20	18		
	Min Comps	3	3	1			
	Max Comps	8	8	10			
	Max Comps/Hole	2	2				
LG Breccia	Principal Azimuth	70	140	160	40	3/-	28/28/28
	Principal Dip	70	140	160	20	3/-	28/28/28
	Intermediate Azimuth	5	5	20	18	3/-	2/1/3.6
	Min Comps	3	3	1			
	Max Comps	8	8	10			
	Max Comps/Hole	2	2				

Search Domain	Parameter	Search Range (m)			High Grade Restriction[3]		
		Pass 1	Pass 2	Pass 3	% of Pass	Limit Au (g/t)/Cu (%)	Range 1/2/3 Pass (m)
Deleterious Elements							
All Domains	Principal Azimuth	80	80	5			
	Principal Dip	160	160	10			
	Intermediate Azimuth	300	300	100			
	Min Comps	3	2	1			
	Max Comps	8	10	10			
	Max Comps/Hole	2					

Notes:
1. Gr Min - Grainstone inside Indicator shell
2. Gr Unmin - Grainstone outside Indicator shell
3. Restriction applied to Au and Cu only

Figure 14.11 – Contact Plots of Silver between the Fault Domain 2 and 5



Source: ??

14.8 Block Model Validation

Validation of the block grade estimates was conducted using the following processes:

- Visual comparison of block grades versus the informing two metre composites on sections and level plans;
- Global and local mean grade comparison between the primary ID2 grade estimates, OK, the nearest neighbour (NN), and the informing two metre composite grades;
- Swath plots along the three axes of composite grades versus ID2, OK, and NN.

A thorough visual section-by-section comparison was completed between informing data and block estimates. Sample sections are shown in Figure 14.12 to Figure 14.14 for gold, copper, and zinc. In addition, swath plots were used to compare the informing data with the estimated grades using both ID2, OK, and NN methods (Figure 14.15 to Figure 14.17). Block grade estimates compared well with the informing data, indicating that the estimation parameters used in the interpolation of grades at El Domo were appropriate for the estimation (Table 14.18 and Table 14.19).

Figure 14.12 – Section – Au (g/t)

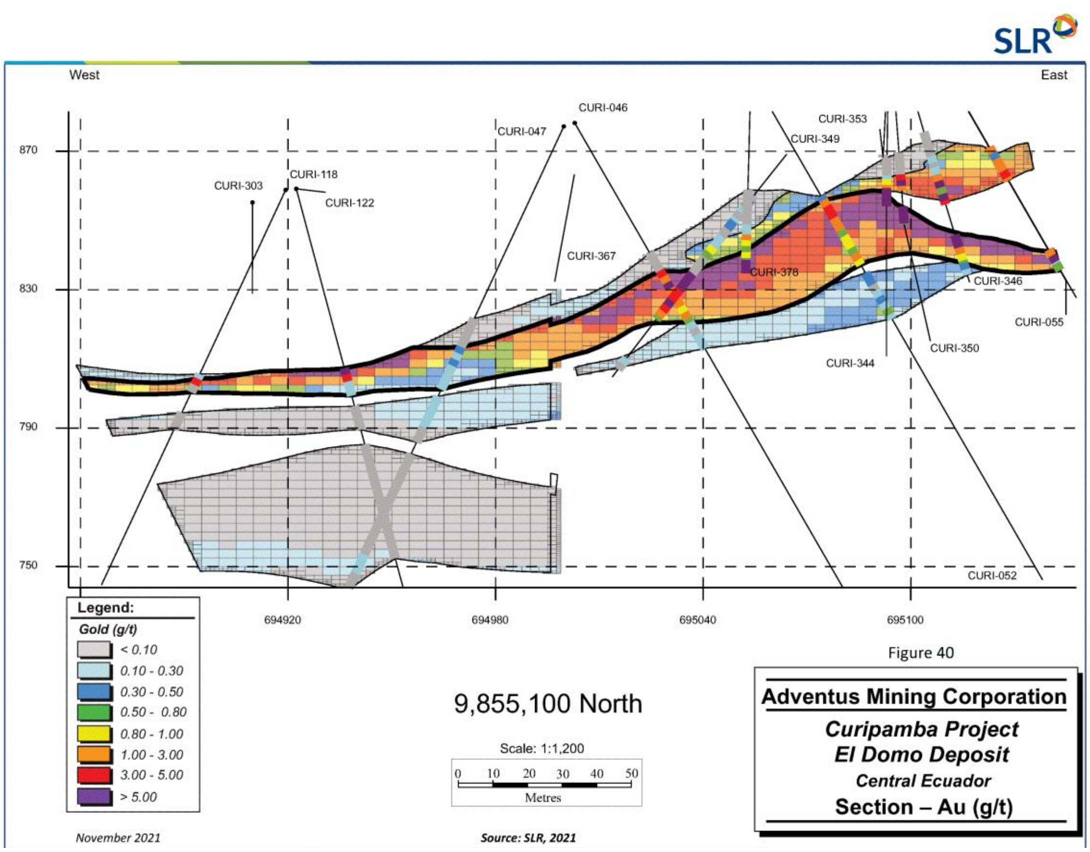


Figure 14.13 – Section – Cu (%)

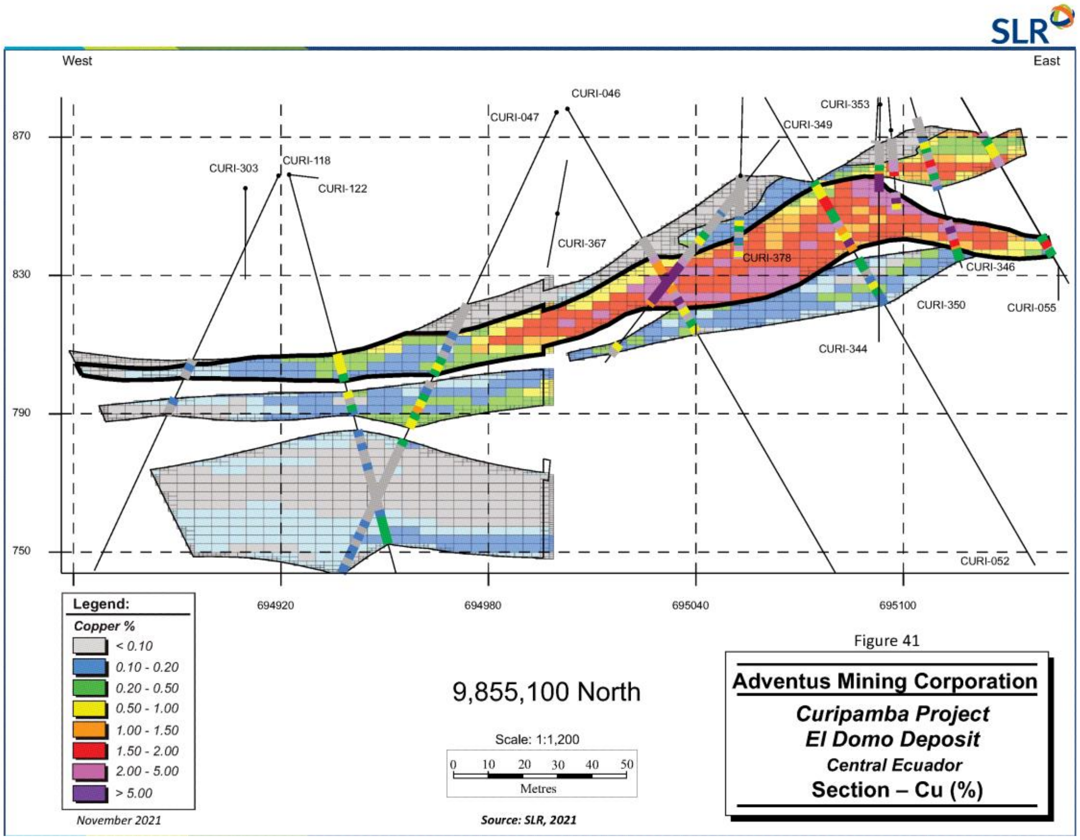


Figure 14.14 – Section – Zn (%)

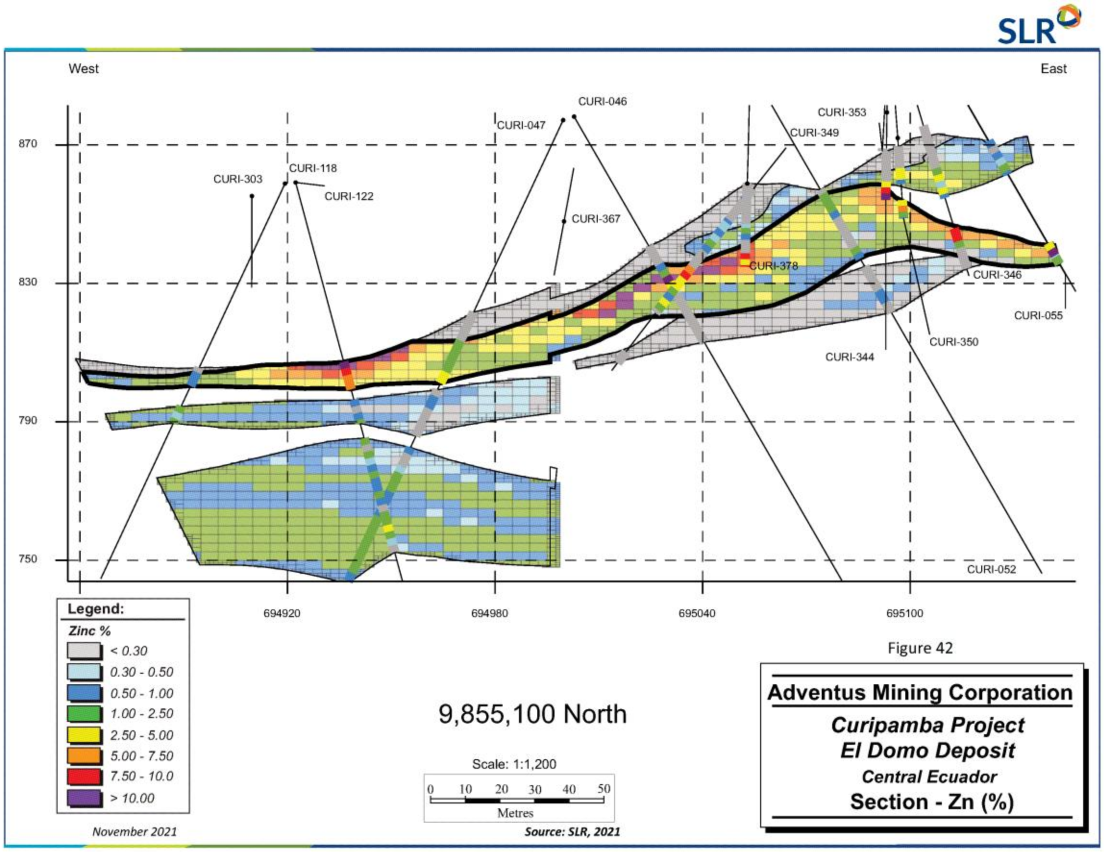


Table 14.18 – Composite versus Block Data – Mineral Resource Metals

Metal	Domain	Capped Composites		Blocks – ID2		Blocks – OK		Blocks - NN	
		Max	Mean	Max	Mean	Max	Mean	Max	Mean
Gold (g/t)	VMS	33.70	2.76	31.93	2.10	20.63	2.11	33.70	2.12
	Gr Min[1]	8.92	1.48	6.77	1.32	5.55	1.31	8.92	1.29
	Gr UnMin[2]	0.70	0.08	0.59	0.07	0.46	0.07	0.70	0.08
	LG Breccia	7.00	0.46	6.91	0.44	6.16	0.43	7.00	0.48
Silver (g/t)	VMS	650.00	51.38	640.00	46.46	527.01	47.09	650.00	46.31
	Gr Min[1]	257.42	27.54	253.72	23.86	202.60	23.97	257.42	23.96
	Gr UnMin[2]	23.00	1.50	22.89	1.32	18.05	1.35	23.00	1.38
	LG Breccia	230.00	14.89	229.02	15.27	216.70	14.99	230.00	15.20
Copper (%)	VMS	15.00	2.26	14.29	1.75	11.80	1.77	15.00	1.61
	Gr Min[1]	6.24	0.73	5.13	0.59	3.99	0.60	6.24	0.55
	Gr UnMin[2]	0.41	0.02	0.29	0.02	0.22	0.02	0.41	0.02
	LG Breccia	3.00	0.16	2.47	0.16	1.64	0.16	3.00	0.16
Lead (%)	VMS	6.81	0.30	5.92	0.27	4.85	0.28	6.81	0.28
	Gr Min [1]	1.13	0.11	1.11	0.09	0.72	0.09	1.13	0.09
	Gr UnMin[2]	0.22	0.01	0.21	0.01	0.16	0.01	0.22	0.01
	LG Breccia	1.40	0.08	1.17	0.08	0.87	0.08	1.40	0.09
Zinc (%)	VMS	35.00	2.97	33.17	2.82	29.23	2.83	35.00	2.68
	Gr Min[1]	7.63	0.85	6.68	0.74	5.01	0.74	7.63	0.68
	Gr UnMin[2]	0.50	0.04	0.46	0.04	0.41	0.04	0.50	0.05
	LG Breccia	6.00	0.69	5.75	0.70	4.38	0.69	6.00	0.70

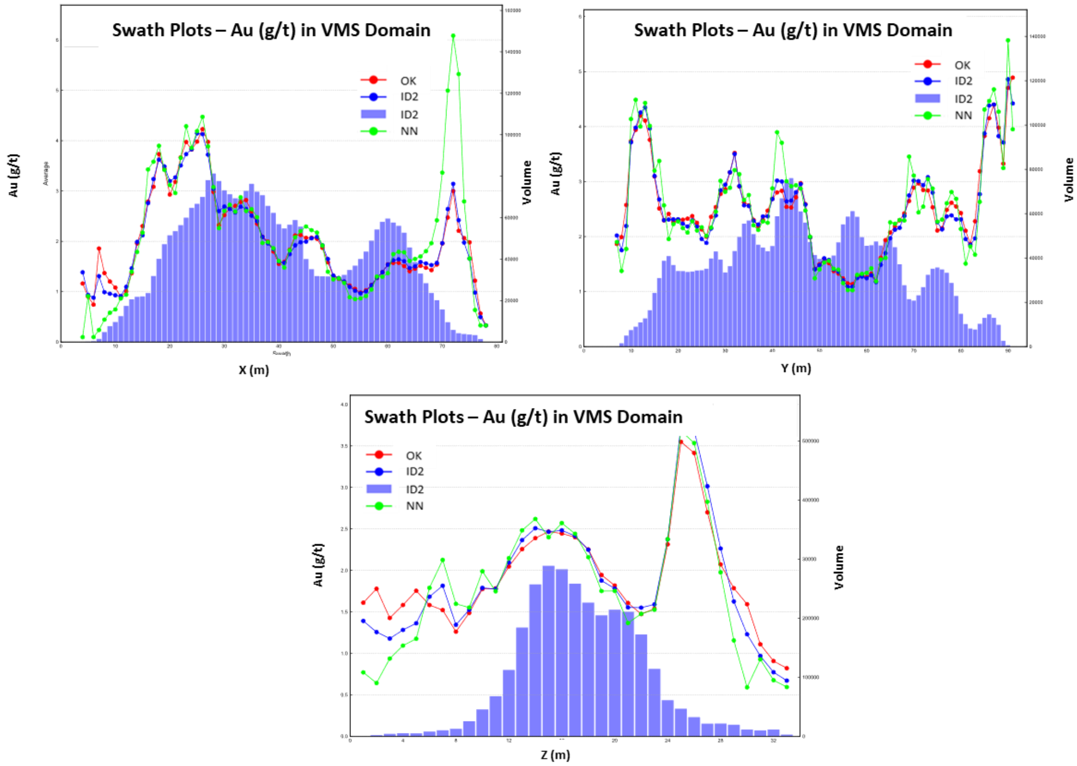
Notes:
1. Gr Min - Grainstone inside Indicator shell
2. Gr Unmin - Grainstone outside Indicator shell

Table 14.19 – Composite versus Block Data – Deleterious Elements

Metal	Domain	Capped Composites		Blocks – ID2		Blocks – OK		Blocks - NN	
		Max	Mean	Max	Mean	Max	Mean	Max	Mean
Arsenic (ppm)	VMS	8,818.06	643.76	7,947.12	608.18	6,022.06	601.44	8,818.06	604.54
	Gr Min¹	6,702.35	468.04	5,081.59	423.87	4,258.70	407.12	5,492.05	402.75
	Gr UnMin²	2,296.44	143.63	1,918.37	99.74	573.46	74.59	2,296.44	96.35
	LG Breccia	1,906.03	177.32	1,850.52	187.35	1,417.44	186.50	1,906.03	194.08
Bismuth (ppm)	VMS	3,567.32	33.49	2,705.22	20.60	2,548.66	20.00	3,567.32	18.44
	Gr Min¹	139.33	9.94	114.81	9.02	65.75	8.54	139.33	8.16
	Gr UnMin²	16.00	2.68	14.33	2.62	11.97	2.61	16.00	2.62
	LG Breccia	68.21	3.52	66.96	3.40	49.53	3.42	68.21	3.43
Cadmium (ppm)	VMS	2,372.61	141.13	2,282.96	163.00	2,065.75	155.49	2,372.61	152.17
	Gr Min¹	730.08	43.26	640.86	37.57	536.74	36.49	730.08	35.16
	Gr UnMin²	79.00	2.79	69.56	2.88	64.00	2.88	79.00	2.90
	LG Breccia	2,176.00	32.26	1,195.02	35.16	1,249.76	34.59	2,176.00	35.25
Iron (%)	VMS	43.75	14.29	42.95	10.68	41.21	10.66	43.75	10.43
	Gr Min¹	23.52	6.13	21.46	5.86	18.34	5.83	23.52	5.75
	Gr UnMin²	7.75	3.88	7.56	3.62	7.14	3.56	7.75	3.65
	LG Breccia	33.40	2.91	32.56	2.78	27.86	2.83	33.40	2.84
Mercury (ppm)	VMS	219.37	8.13	159.62	9.13	135.92	8.93	219.37	9.01
	Gr Min¹	47.02	4.37	37.57	3.85	37.53	3.81	47.02	3.72
	Gr UnMin²	24.00	1.04	18.94	0.93	17.48	0.97	24.00	0.98
	LG Breccia	44.50	1.90	30.58	2.04	27.66	2.03	44.50	2.06
Sulphur (%)	VMS	19.12	10.49	18.18	9.78	17.01	9.70	19.12	9.42
	Gr Min ¹	15.00	4.53	14.05	4.26	13.70	4.16	15.00	4.09
	Gr UnMin²	13.99	1.44	13.68	1.01	10.67	1.01	13.99	0.97
	LG Breccia	15.00	4.08	15.00	4.14	15.00	4.18	15.00	4.21
Antimony (ppm)	VMS	4,127.67	126.95	3,318.37	149.48	3,151.68	142.80	3,318.37	149.41
	Gr Min¹	845.93	56.78	738.95	52.49	598.44	52.18	845.93	52.33
	Gr UnMin²	62.48	5.91	52.28	4.96	45.98	5.10	62.48	5.28
	LG Breccia	1,098.15	27.10	1,000.81	27.62	756.89	26.68	1,098.15	28.30

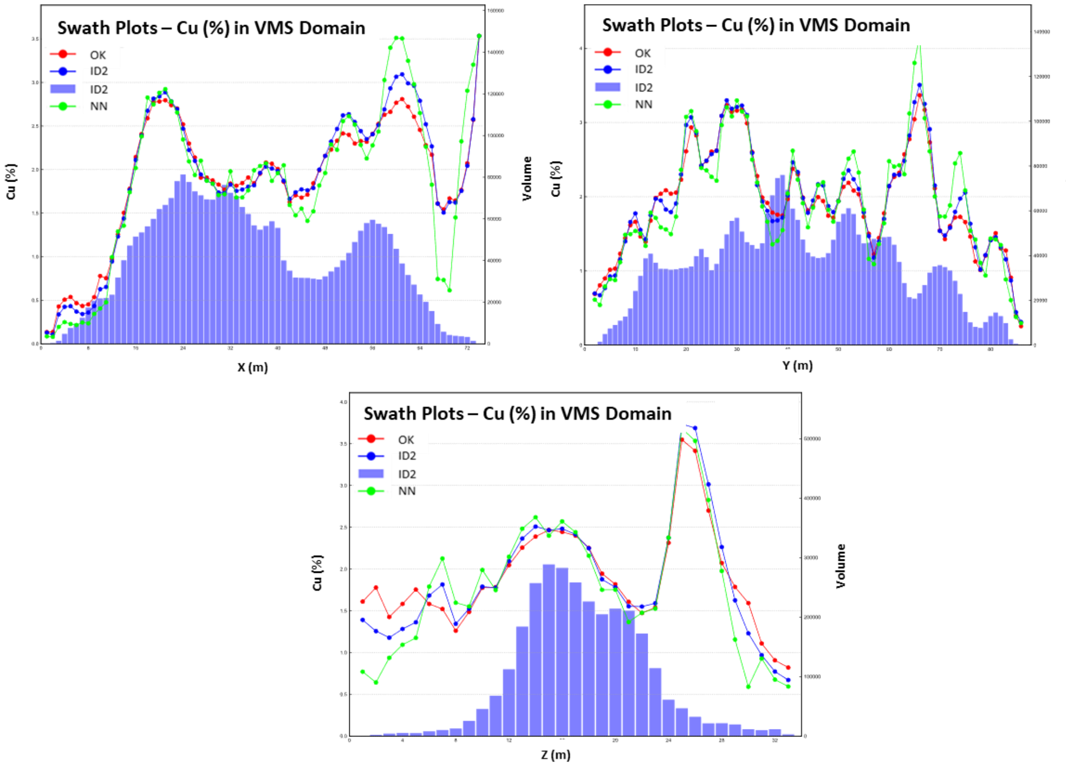
Notes:
1. Gr Min - Grainstone inside Indicator shell
2. Gr Unmin - Grainstone outside Indicator shell

Figure 14.15 – Swath Plots – Au (%) in VMS Domain



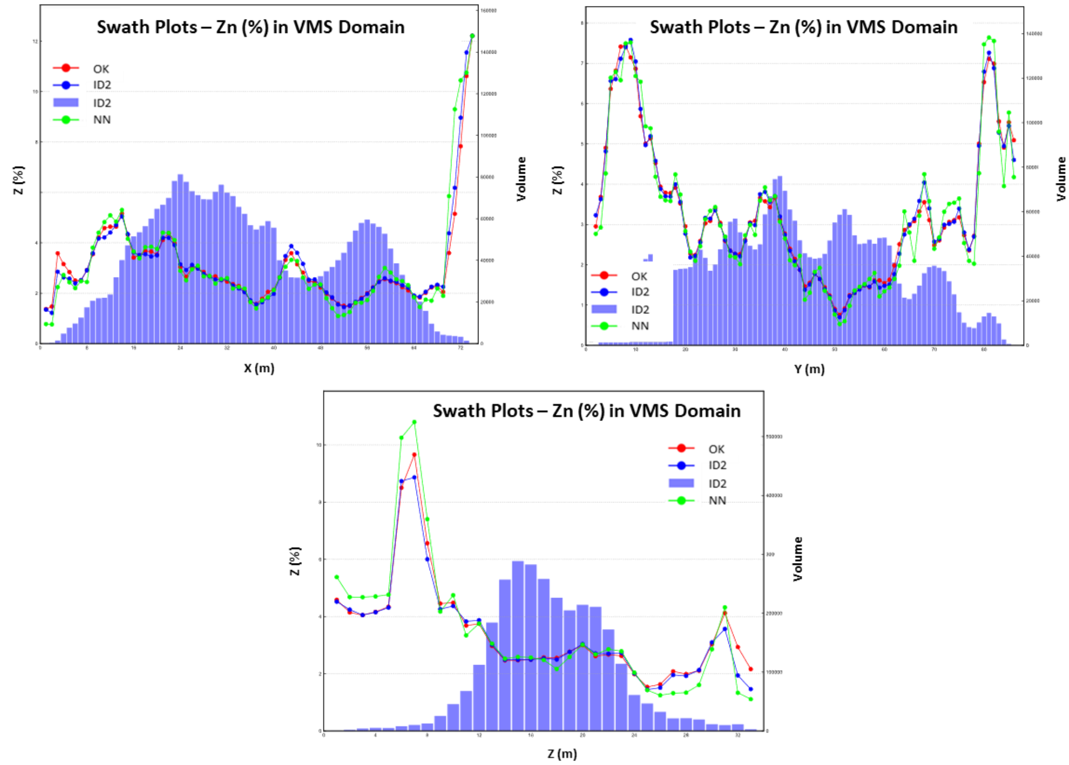
Source: SLR, 2021

Figure 14.16 – Swath Plots – Cu (%) in VMS Domain



Source: SLR, 2021

Figure 14.17 – Swath Plots – Zn (%) in VMS Domain



Source: SLR, 2021

14.9 NSR Cut-Off Value

NSR factors were developed by SLR for the purposes of geological interpretation and resource reporting. NSR is the estimated value per tonne of mineralised material after allowance for metallurgical recovery and consideration of smelter terms, including payables, treatment charges, refining charges, price participation, penalties, smelter losses, transportation, and sales charges.

The calculation of the NSR value of each block was a two-step process. Initially, each block was flagged as one of three types of mineralization, based on the metal ratio of copper to the sum of lead plus zinc (metal ratio = Cu/(Pb + Zn)).

Input parameters used to develop the NSR factors have been derived from recent metallurgical test work on the El Domo deposit and smelter terms from comparable projects. These assumptions are dependent on the processing scenario and will be sensitive to changes in inputs from further metallurgical test work. The net revenue from each metal was calculated and then divided by grade to generate an NSR factor. These NSR factors represent revenue (US$) per metal unit (per g/t Au, for example), and are independent of resource grade. Table 14.20 presents key assumptions and factors used by SLR to calculate NSR.

Table 14.20 – Key Assumptions for the NSR Calculation

	Gold	Silver	Copper	Lead	Zinc
Metal Prices	US$1,800/oz	US$24/oz	US$4.00/lb	US$1.05/lb	US$1.30/lb
	US$/g	US$/g	US$/%	US$/%	US$/%
Value Factor Zn-Type	30.91	0.39	53.41	7.99	13.47
Value Factor Mixed Cu/Zn-Type	25.12	0.35	58.99	7.05	13.41
Value Factor Cu-Type	6.84	0.19	57.83	0.00	0.00

Note:
Type 1: Zinc mineral: Cu/(Pb+Zn) <0.33
Type 2: Mixed Cu/Zn mineral: 0.33≤ Cu/(Pb+Zn) ≤3
Type 3: Copper mineral: Cu/(Pb+Zn) >3

SLR performed pit optimization analyses on the Mineral Resource to determine the economics of extraction by open pit methods. The parameters used in the pit optimization runs, using Whittle software, are presented in Table 14.21.

Table 14.21 – El Domo Whittle Parameters

Parameter	Unit	Input
Block Size (Parent)	m	5 x 5 x 2.5
Whittle Block Size	m	5 x 5 x 5
Overall Slope Angle – Overburden	Degrees (°)	21
Overall Slope Angle – Rock	Degrees (°)	44 to 51
Au Price	US$/oz	1,800
Ag Price	US$/oz	24.00
Cu Price	US$/lb	4.00
Pb Price	US$/lb	1.05
Zn Price	US$/lb	1.30
Process Recoveries per Metallurgical Zone	%	Zn /Mixed/Cu
Au	%	68/55/14
Ag	%	78/67/29
Cu	%	86/86/80
Pb	%	90/82/37
Zn	%	97/95/36

Parameter	Unit	Input
Mining Extraction	%	95
Mining Dilution	%	5
Open Pit Mining Cost - Overburden	US$/t mined	2.33
Open Pit Mining Cost - Waste	US$/t mined	3.54
Open Pit Mining Cost - Ore	US$/t mined	4.53
Process Cost	US$/t milled	18.58
G&A Cost	US$/t milled	7.91
Tailings Cost	US$/t milled	1.96
TSF Dewatering Treatment Cost	US$/t milled	1.13
Underground Alternative Mining Cost	US$/t mined	100
Resource Classification	Measured, Indicated & Inferred	All

Whittle calculates a final break-even pit shell based on all operating costs (mining, processing, and general and administration (G&A)) required to mine a given block of material. Since all blocks within the break-even pit shell must be mined, any block that has sufficient revenue to cover the costs of processing and G&A is sent to the processing plant. The open pit Mineral Resources were reported from within the pit shell using an NSR cut-off value of US$29/t based on US$18.58/t processing costs and US$7.91/t G&A costs.

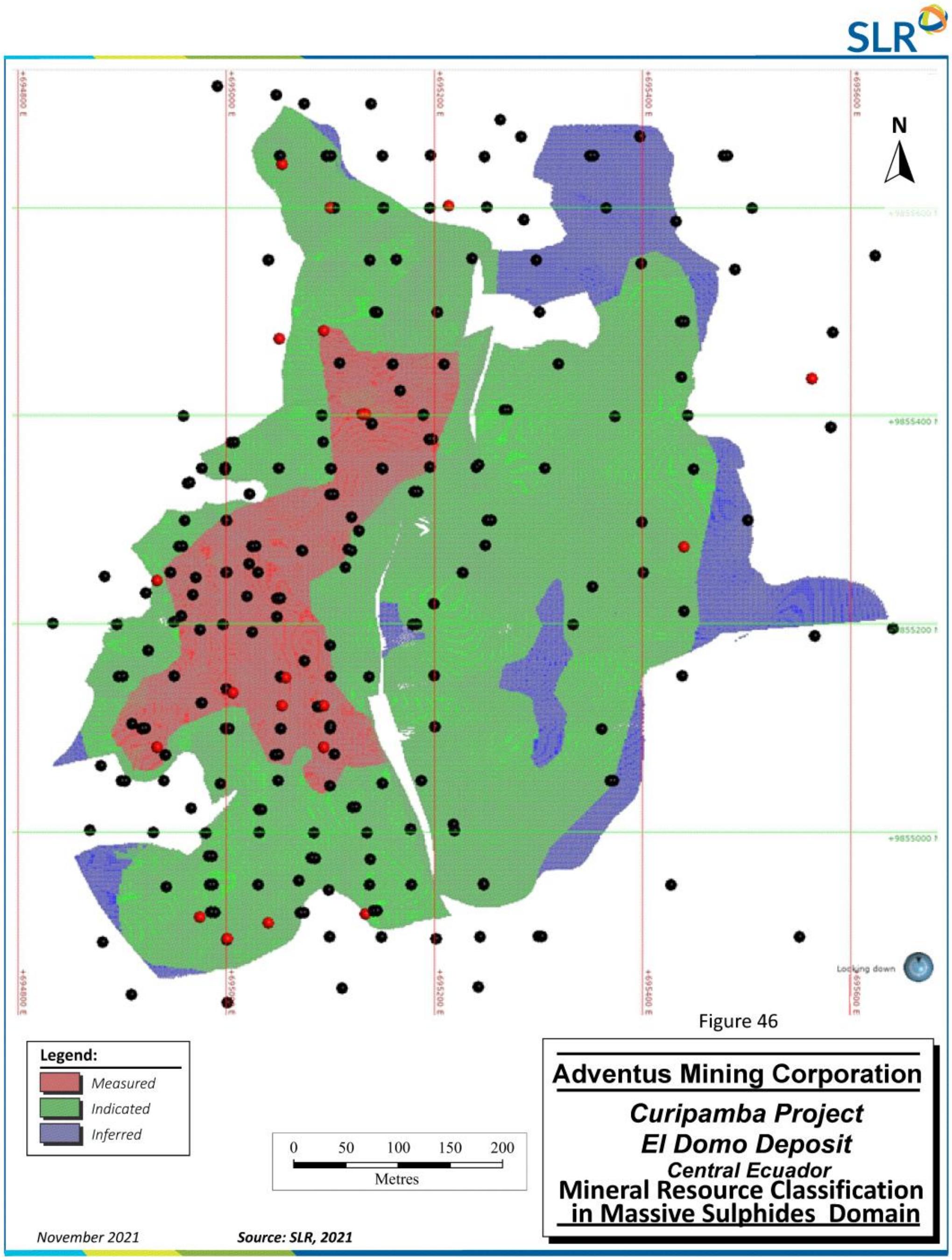
The Whittle parameters for Mineral Resources located outside the pit shell are assumed to be reported on the basis of a potential underground mining scenario at an NSR cut-off value of US$100/t. This cut-off value was based on mining costs of US$100/t and the same processing and G&A assumptions as listed above.

14.10 Classification

Definitions for resource categories used in this report are consistent with CIM (2014) definitions incorporated by reference into NI 43-101. In the CIM classification, a Mineral Resource is defined as "a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity, and other geological characteristics of a Mineral Resource are known, estimated, or interpreted from specific geological evidence and knowledge, including sampling." Mineral Resources are classified into Measured, Indicated, and Inferred categories. A Mineral Reserve is defined as the "economically mineable part of a Measured and/or Indicated Mineral Resource" demonstrated by studies at pre-feasibility or feasibility level as appropriate. Mineral Reserves are classified into Proven and Probable categories.

The classification of the El Domo deposit is based on borehole spacing, confidence in the available data, and the apparent continuity of mineralization. With significant increase in lithological understanding and the infill drilling carried out in 2020/2021, the QP was able to increase the portion of the Mineral Resources in the Measured and Indicated categories. The Mineral Resources assigned to the Measured category have borehole spacing of approximately 25 m, located within the massive sulphide horizon (Figure 14.18), and exhibit good grade continuity that was assessed visually. The areas with borehole spacing of 50 m, located within the massive sulphides and grainstone domain, with good mineralization continuity, were classified as Indicated Mineral Resources. Additionally, parts of the hydrothermal breccia that are positioned in proximity to the massive sulphide domain, with boreholes spacing of 50 m or less, were assigned an Indicated category. All other areas of the Mineral Resources were assigned to the Inferred category.

Figure 14.18 – Mineral Resource Classification in VMS Domain



14.11 Mineral Resource Statement

The Mineral Resource estimate has an effective date of October 26, 2021. The Mineral Resources are reported in Table 14.22.

Table 14.22 – Mineral Resource Estimate – October 26, 2021

Resource Category	Tonnes (Mt)	Grade					Contained Metal				
		Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu (kt)	Pb (kt)	Zn (kt)	Au (koz)	Ag (koz)
Open Pit Resources											
Measured	3.2	2.61	0.2	2.50	3.03	45	84.9	7.7	81.1	316	4,704
Indicated	3.8	1.38	0.3	2.77	2.29	52	52.6	11.3	105.2	280	6,370
M+I	**7.1**	**1.95**	**0.3**	**2.64**	**2.63**	**49**	**137.5**	**19.0**	**186.3**	**596**	**11,074**
Inferred	0.3	0.34	0.2	1.01	1.34	39	1.2	0.7	3.5	15	430
Underground Resources											
Indicated	1.9	2.72	0.14	2.38	1.37	31	51.9	2.6	45.4	84	1,895
Inferred	0.8	2.31	0.11	2.68	1.74	29	17.3	0.8	20.1	42	688
Total Mineral Resources											
Measured	3.2	2.61	0.2	2.50	3.03	45	84.9	7.7	81.1	316	4,704
Indicated	5.7	1.83	0.24	2.64	1.98	45	104.5	13.9	150.6	364	8,265
M+I	**9.0**	**2.11**	**0.24**	**2.59**	**2.36**	**45**	**189.4**	**21.6**	**231.7**	**680**	**12,969**
Inferred	1.1	1.72	0.14	2.18	1.62	32	18.5	1.5	23.6	57	1,118

Notes:

1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are reported above an NSR cut-off value of US$29/t for potential open pit Mineral Resources and the underground portion of the 2021 Mineral Resources are reported with mining shapes which were generated using an NSR cut-off value of US$105/t NSR.
3. The NSR value is based on estimated metallurgical recoveries, assumed metal prices, and smelter terms, which include payable factors treatment charges, penalties, and refining charges.
4. Mineral Resources are estimated using the metal price assumptions: US$4.00/lb Cu, US$1.05/lb Pb, US$1.30/lb Zn, US$1,800/oz Au, and US$24/oz Ag.
5. Metallurgical recovery assumptions were based on three mineral types defined by the metal ratio Cu/(Pb+Zn):
   a. Zinc Mineral (Cu/(Pb+Zn) <0.33): 86% Cu, 90% Pb, 97% Zn, 68% Au, and 78% Ag;
   b. Mixed Cu/Zn Mineral (0.33≤ Cu/(Pb+Zn) ≤3.0): 86% Cu, 82% Pb, 95% Zn, 55% Au, and 67% Ag;
   c. Copper Mineral (Cu/(Pb+Zn) >3.0): 80% Cu, 37% Pb, 36% Zn, 14% Au, and 29% Ag;
6. NSR factors were also based on the metal ratio Cu/(Pb+Zn):
   a. Zinc Mineral (Cu/(Pb+Zn) <0.33): 53.41 US$/% Cu, 7.99 US$/% Pb, 13.47 US$/% Zn, 30.91 US$/g Au, and 0.39 US$/g Ag,
   b. Mixed Cu/Zn Mineral (0.33≤ Cu/(Pb+Zn) ≤3.0): 58.99 US$/% Cu, 7.05 US$/% Pb ,13.41 US$/% Zn, 25.12 US$/g Au, and 0.34 US$/g Ag;
   c. Copper Mineral (Cu/(Pb+Zn) >3.0): 57.83 US$/% Cu, 6.84 US$/g Au, and 0.19 US$/g Ag.
7. Bulk density interpolated on a block per block basis using assayed value, the correlation between measured density values and iron content, and base metal grade.
8. Mineral Resources are inclusive of Mineral Reserves.
9. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
10. The underground portion of the Mineral Resources are reported within underground reporting shapes and include low grade blocks falling within the shapes.
11. Numbers may not add due to rounding.

The QP is not aware of any environmental, permitting, legal, title, taxation, socio- economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

14.12 Comparison to Previous Mineral Resource Estimates

The current Mineral Resource estimate supersedes the Mineral Resource estimate effective May 2, 2019. The current Mineral Resource model incorporates additional drilling completed in 2021. Table 14.23 shows a direct comparison between the May 2, 2019, and October 26, 2021 Mineral Resource estimates. Although the overall tonnage has increased, the overall metal content has decreased.

Table 14.23 – Comparison Between 2019 and 2021 Mineral Resources

2019 Model prepared by SLR

Class	Tonnage (Mt)	Grade					Contained Metal				
		Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu (kt)	Pb (kt)	Zn (kt)	Au (koz)	Ag (koz)
M+I	8.9	2.00	0.3	2.93	2.56	51	178.7	25.0	261.3	733	14,588
Inferred	1.3	1.52	0.2	2.25	1.83	42	20.1	2.7	29.7	78	1,783

2021 Model prepared by SLR

Class	Tonnage (Mt)	Grade					Contained Metal				
		Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu (kt)	Pb (kt)	Zn (kt)	Au (koz)	Ag (koz)
M+I	9.0	2.11	0.24	2.59	2.36	45	189.4	21.6	231.7	680	12,969
Inferred	1.1	1.72	0.14	2.18	1.62	32	18.5	1.5	23.6	57	1,118

Difference

Class	Tonnage	Grade					Contained Metal				
		Cu	Pb	Zn	Au	Ag	Cu	Pb	Zn	Au	Ag
M+I	1%	5%	-20%	-12%	-8%	-12%	6%	-14%	-11%	-7%	-11%
Inferred	-15%	13%	-30%	-3%	-11%	-24%	-8%	-44%	-21%	-27%	-37%

Notes:
2019 numbers are reported as per press release dated May 2, 2019.
2021 numbers are reported as per press release dated October 26, 2021

The current geological model benefited from additional 55 boreholes (7,094 m) completed between 2020 and 2021, and new LIDAR topography. All the drill collars, from previous years and from the 2020/2021 campaign, were resurveyed using GPS total stations and audited with LIDAR data.

The new topography surface, collar location resurveys, and new drilling prompted adjustments to the faults, lithological, and mineralization modelling which resulted in minor changes to the Mineral Resources. While the majority of the infill drilling confirmed VMS mineralization, some of the boreholes indicated that the mineralised zones terminate earlier in the eastern part of the deposit than previously expected. The volumetric comparison of all the mineralised zones shows that the 2019 and 2021 volumes are comparable despite local increase or decrease in the unit thickness.

Assay data were analyzed for grade continuity, and high-grade assays were capped where deemed necessary. In most cases, capping values are comparable to those used during the 2019 estimation. The infill drilling allows for the increase of the Mineral Resource in the Measured category.

The differences in the 2019 and 2021 Mineral Resource inventory are mostly due to the following:

- Higher processing and mining costs resulting in a smaller reporting shell, hence an open pit tonnage decrease;
- Transfer of the model from Gems Percent to Leapfrog Sub-block;
- Increase in the Measured category due to new drilling;
- Hard boundaries applied between VMS fault domains;
- Higher NSR cut-off values for the open pit mining method ($29/t versus $25/t) in the current 2021 resource;
- A drop in the metal grades due to higher NSR factors;
- 2021 underground Mineral Resources reported within the mining shapes;
- Improved density determination.

15 MINERAL RESERVE ESTIMATE

15.1 Introduction

This section discusses the conversion of the Project's Mineral Resources to Mineral Reserves.

15.2 Mineral Resource Model Import and Preparation

The FS was based on Mineral Resources with an effective date of October, 26, 2021, which is detailed in Section 14 of this Report.

The Datamine block model was imported into HxGN Mine Plan to perform pit optimisation activities.

15.3 Pit Optimisation

The Project's economic pit limits were determined using HxGN MinePlan's MSOPit module. The optimiser operates on a net value calculation for all blocks in the model. The net value (Equation 15.3) of each block is equal to the revenue (Equation 15.1) generated by the block's material minus the costs (Equation 15.2) associated with extracting and processing it. The NSR is calculated (Equation 15.4) using the metal grades and the revenue generated per metal (NSR Factor), which is in turn determined using the process recoveries for each metal and material type as well as the metal selling prices.

$$Revenue\ Generated = NSR \times Tonnage\ Processed$$ Equation 15.1

$$Cost\ Incurred = Mining\ Cost + Processing\ Cost + G\&A\ Cost + Rehab\ Cost + Royalty$$ Equation 15.2

$$Net\ Value = Revenue\ Generated - Cost\ Incurred$$ Equation 15.3

$$NSR_{block} = \sum_{metals} NSR\ Factor_{metal} \times Grade_{metal}$$ Equation 15.4

The parameters used in the NSR calculation are presented in Table 15.1.

Table 15.1 – NSR Calculation Parameters

Material Type				
	Unit	**High Zn**	**Mixed Cu/Zn**	**High Cu**
Recoveries (1)				
Au	%	68.0	55.0	14.0
Ag	%	78.0	67.0	29.0
Cu	%	86.0	86.0	80.0
Pb	%	90.0	82.0	37.0
Zn	%	97.0	95.0	36.0
Metal Prices				
Au	$/oz	1,630.00		
Ag	$/oz	21.00		
Cu	$/lb	3.31		
Pb	$/lb	0.92		
Zn	$/lb	1.16		
Revenue per Metal (NSR Factor)				
Au	$/ g Au	25.74	20.93	5.69
Ag	$/ g Ag	0.30	0.26	0.14
Cu	$/ % Cu	37.76	42.51	41.84
Pb	$/ % Pb	6.08	5.47	0.00
Zn	$/ % Zn	9.91	9.84	0.00

(1) Indicated metal recoveries for the initial reserve pit shell development were based on available recovery test work at the time and individual ore's recovery responses. These recoveries will not reflect annualized, ore blended, weighted averaged recoveries depicted in Sections 17 and 22 of this Report.

Only Measured and Indicated Mineral Resources from the 3D model provided by SLR have been considered for the pit optimisation and mine planning. The optimisation was carried out using the parameters listed in Table 15.2. The parameters were developed assuming a standard open pit truck and shovel operation, and a 0.67 Mt/year throughput at the mill. Process recoveries and all production costs were defined or developed by DRA based on metallurgical testing and detail cost calculations. The Mineral Reserves are reported at the metal prices listed in Table 15.1.

Table 15.2 – Mineral Reserves Pit Optimisation Parameters

Work Schedule	Unit	Value	Source
Slope angles	Variable per material type. Refer to Section 16 for details		
Mining Recovery	%	100	See Section 15.3.3
Mining Dilution	%	0	
Mining Costs			
Overburden	$/t mined overburden mined	2.10	DRA-estimated contracting costs
Saprolite	$/t mined overburden mined	2.10	
Waste	$/t mined waste mined	3.31	
Ore	$/t ore mined	4.20	
Stockpile Rehandle	$/t moved	0.33	
Mine Dewatering	$/t mined	0.23	DRA-estimated
Processing Costs			
Processing	$/t milled	18.58	DRA-estimated
Tailings	$/t milled	1.96	
G&A	$/t milled	7.91	
TSF Dewatering Treatment	$/t milled	1.13	DRA-estimated
Royalty	%	6	See Section 22

15.3.1 Reserve Metal Prices

The selected metal prices, as listed in Table 15.1 and Table 15.3, are based on a combination of industry consensus prices for long-term metal prices and historical 3-year price average. Details on metal price selection can be found in Section 19 of the Report.

Table 15.3 – Reserve Metal Prices

Element	Unit	Metal Price
Au	US $/oz	1,630.00
Ag	US $/oz	21.00
Cu	US $/lb	3.31
Pb	US $/lb	0.92
Zn	US $/lb	1.16

15.3.2 Overall Slope Angles

The slope angles used to optimise the pit are detailed in Section 16. A summary of the slope angles can be found in Table 15.4.

Table 15.4 – Curipamba Pit Slope Criteria

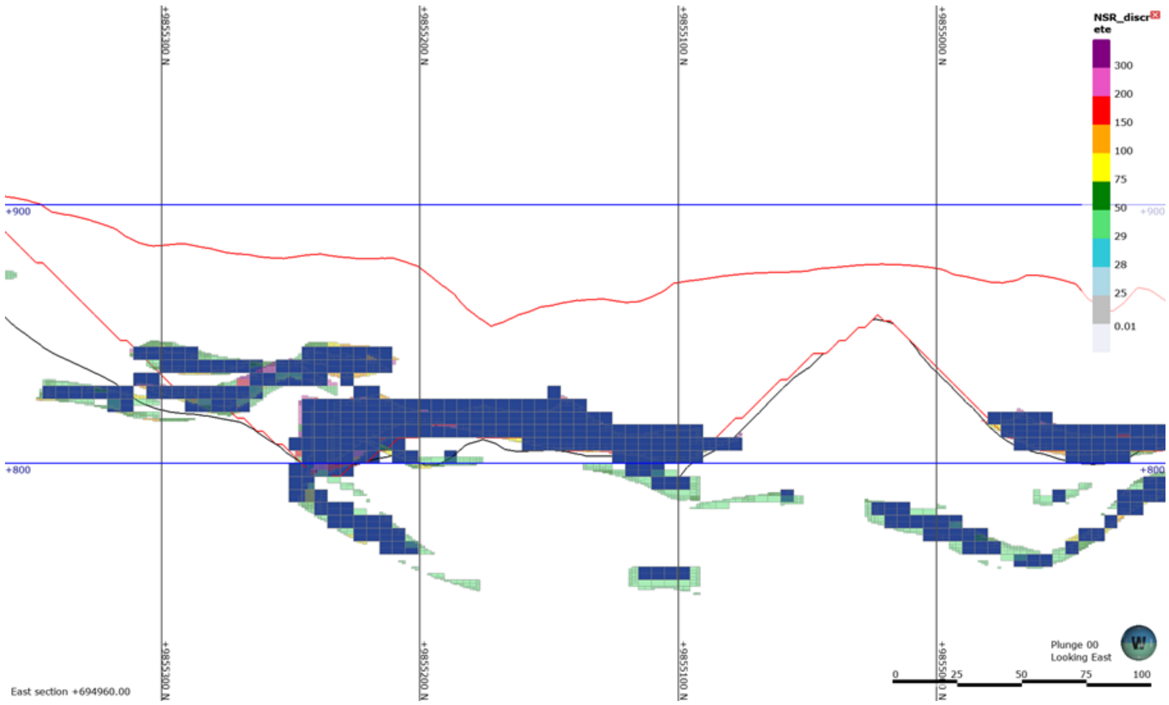
Domain	Lithology	Wall Facing Direction (°)	BFA (°)	Bench Height (m)	Planned Berm Width (m)	Design IRA (°)	Stack Height (m)	Geotechnical Berm Width (m)	Overall Slope Angle (°)
East Wall	Andesite	230-320	80	10	5.5	54	70	14	50.7
	Tuff	270-310	80	10	6	52.2	70	14	49.2
	Lapilli Tuff								
All other Walls	Andesite	080-100	70	10	5.5	47.6	70	12	45.5
	All other lithologies	320-180	70	10	6	46.1	70	12	44.2
Pit's Edge	Overburden and Saprolite		75	5	12	20.5		20 offset from edge of pit	

Notes:
1. The slope angles are for fully drained slopes
2. Pit wall directions are based on the following directions: North, 0°; West, 90°; South, 180°; East, 270°
3. The East wall was used to evaluate the maximum wall height, which was used to determine the stack height
4. The design IRA is a toe-to-toe angle
5. Pre-splitting of the final pit wall to maximize effective berm width and limit breakback, as well as careful blasting, are required to minimize damage to the final pit wall slopes
6. The 20 m offset for the overburden and saprolite is intended to allow equipment access to remove any sloughed material

15.3.3 Mine Dilution and Recovery

The Curipamba deposit is a massive multi-element deposit. SLR developed its Mineral Resource Estimate based on a sub-blocked model, which was then converted to a standard 5 m x 5 m x 5 m SMU block size for the Mineral Reserve Estimate and subsequent mine planning. By doing so, ore and waste sub-blocks were combined to created regularized blocks. Throughout the deposit, this resulted in waste material being included in ore blocks, diluting the ore by approximately 8%. Due to this included dilution, additional dilution or ore loss was not required. At the edges of the deposit, as shown in Figure 15.1, the re-blocking resulted in waste beyond the limits of the ore being include in the ore blocks.

Figure 15.1 – Results of Re-Blocking



Source : DRA 2021

15.3.4 Mining Costs

The mining costs used in the pit optimisation assume contract mining and were based on the mine plan developed for an internal study. DRA applied equipment productivities, equipment operating costs and manpower requirements to derive estimated individual contractor operating costs for the overburden, saprolite, ore and waste mining. These costs were later confirmed by requesting local contractor costs based on the FS mine plan, presented in Section 16. Further details on contractor costs are presented in Section 21.

15.3.5 NSR

An NSR was calculated to assign a revenue value to each block, which takes into consideration the process recoveries for each metal and material type, as shown in Table 15.1, as well as the metal prices, as listed in Table 15.3.

15.3.6 Process Costs

Process unit costs were estimated by DRA process engineers based on the process flowsheet from a mill producing three (3) concentrates: copper, lead and zinc. The mill has a 0.67 Mt/year throughput rate. Further details are presented in Section 21.

15.3.7 G&A Costs

G&A costs were estimated by DRA based on an annual production of 0.67 Mt. G&A costs are presented in Section 21.

15.3.8 Cut-Off Grade

The cut-off grade (COG) is used to determine whether the material mined and processed will generate profit after paying the mining, processing, G&A, and other relevant costs, and is the basis for defining an economic pit limit. Material below the COG will be sent to a waste storage facility. The (COG) has been calculated on the NSR according to Equation 15.5, resulting in a $32.99 NSR COG.

$$COG = Processing\ Cost + Tailings + G\&A + Ore\ Mining\ Cost + Ore\ Stockpile\ Rehandle\ Cost \quad \text{Equation 15.5}$$

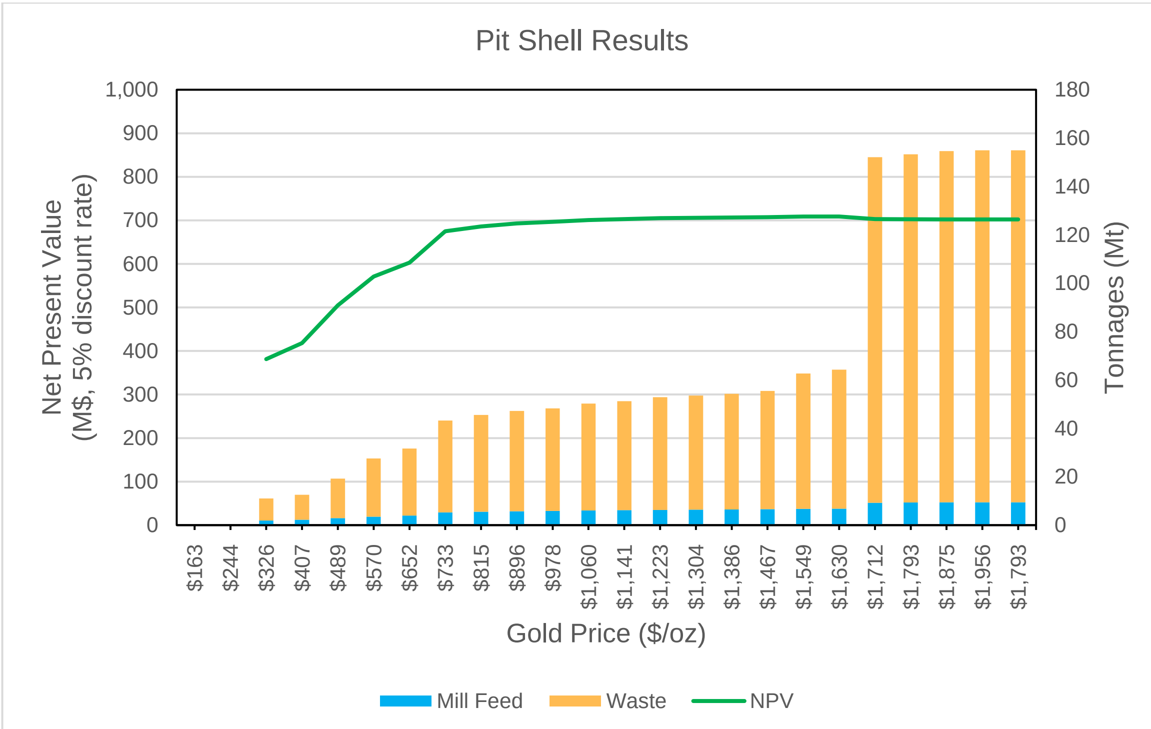
15.3.9 Pit Optimisation Results

Optimal open pit mining limits were established using HxGN MinePlan's MSOPit module, using the Pseudoflow algorithm. The Pseudoflow algorithm obtains the same results as the Lersch-Grossman algorithm but is much more computationally efficient and therefore results in faster solution times.

Net present value and life-of-mine were the main criteria used to select the ultimate pit. The NSR is varied using revenue factors between 0.1 and 1.2, with 0.05 increments, as shown in Figure 15.2 and represented by gold prices. The NPV, discounted at 5%, increases gradually until reaching a maximum at revenue factor 1 (gold price of $1,630/oz); after this inflexion point the NPV decreases as the costs exceed the revenues.

The pit shell corresponding to revenue factor 0.9 (gold price of $1,467/oz) was chosen as it provided an NPV of $707.37 M and life-of-mine of approximately 10 years of production.

Figure 15.2 – MSO Pit Optimisation Results



Source : DRA 2021

15.4 Pit Design

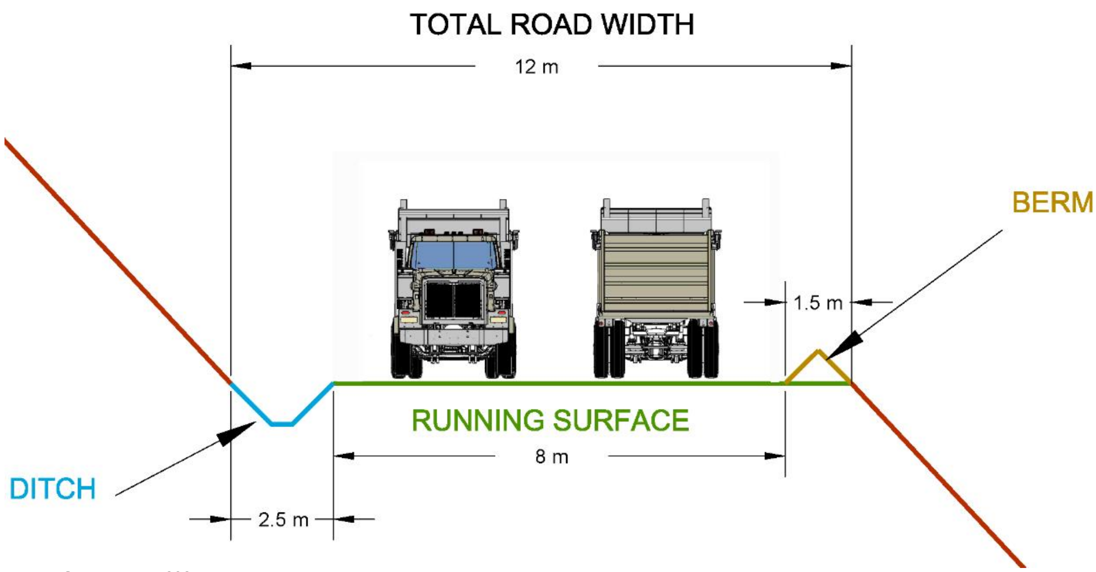
The next step in the Mineral Reserve estimation process is to design an operational pit that will form the basis of the production plan. The pit design uses the selected economic pit shell as a guideline and includes smoothing the pit walls, adding ramps to access the pit bottom and ensures the equipment's minimum mining width is respected. The pit design was created in HxGn MinePlan based on the selected pit shell and the 3D block model provided by SLR. The following section provides the parameters used in the open pit design and presents results.

15.4.1 Haul Road Design

The ramps and haul roads were designed for an overall width of 12 m, to accommodate double lane traffic using 8x4 trucks. When assuming double lane traffic, industry standards indicate that the running surface be 3 times the width of the largest equipment travelling on it; in this case the largest equipment is the haul truck. Allowances for ditches and berms are also made. A typical road cross-section is presented in Figure 15.3. The bottom 1 to 3 benches of the pit were designed for single-lane traffic, with an overall road width of 8 m, due to the zones being low-traffic areas.

A maximum ramp grade of 10% was used for this FS. Due to the rugged terrain, short sections with a grade between 10% and 12% were acceptable. These sections never exceeded 200 m in length and were only used for temporary haul roads.

Figure 15.3 – Haul Road Design



Source : DRA 2021

15.4.2 Pit Slopes

The pit design followed the recommended geotechnical slopes as described in Section 16 of the Report and summarised in Section 15.3.2.

15.4.3 Open Pit Design Results

Figure 15.4 presents the open pit design which was used to estimate the Mineral Reserve. The final pit design delineates 6.48 Mt of ore and 55.34 Mt of waste, representing an increase of 1.12% in ore, 11.25% in waste and 10.09% in overall tonnage compared to the ultimate pit shell. These increases are caused by the inclusion of catch benches, roads, and the minimum mining width of 24 m to ensure adequate room for equipment operation.

Figure 15.4 – Curipamba Final Pit Design

Source: DRA 2021

15.5 Mineral Reserves

The Mineral Reserves for the Project are estimated at 3.14 Mt of Proven Reserves at an average grade of 2.83 g/t Au, 41.42 g/t Ag, 0.21% Pb, 2.30% An and 2.50% Cu (for an average NSR of $207.94) and 3.34 Mt of Probable Reserves at an average grade of 2.23 g/t Au, 49.70 g/t Ag, 0.29% Pb, 2.68% Zn and 1.39% Cu (for an average NSR of $164.29). To access these reserves, 55.34 Mt of waste will need to be removed, for an overall strip ratio of 8.54 to 1 (tonnes of waste to tonnes of ore). Table 15.5 presents the open pit Mineral Reserves for the Project.

Table 15.5 – Mineral Reserves Estimate – Effective October 26, 2021

Classification	Ore Type	Tonnage (kt)	NSR ($)	Grades					Metal				
				Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Cu (%)	Au (koz)	Ag (koz)	Pb (kt)	Zn (kt)	Cu (kt)
Proven Reserves	High Zn	370.8	222.5	3.56	67.42	0.60	4.96	0.92	42.4	803.7	2.2	18.4	3.4
	Mixed Zn/Cu	1,676.9	227.3	3.15	48.77	0.23	2.66	2.30	169.8	2,629.4	3.9	44.6	38.6
	High Cu	1,087.9	173.2	2.09	21.21	0.06	0.83	3.35	73.1	741.8	0.7	9.0	36.4
	Subtotal	*3,135.5*	*207.9*	*2.83*	*41.42*	*0.21*	*2.30*	*2.50*	*285.4*	*4,174.9*	*6.7*	*72.0*	*78.4*
Probable Reserves	High Zn	999.3	197.6	2.92	72.66	0.51	4.47	0.86	93.8	2,334.5	5.1	44.7	8.6
	Mixed Zn/Cu	2,068.0	149.7	1.94	41.98	0.20	2.04	1.44	129.0	2,791.1	4.1	42.2	29.8
	High Cu	275.5	152.9	1.89	24.41	0.07	0.91	2.93	16.7	216.2	0.2	2.5	8.1
	Subtotal	*3,342.8*	*164.3*	*2.23*	*49.70*	*0.29*	*2.68*	*1.39*	*239.5*	*5,341.9*	*9.4*	*89.4*	*46.4*
Total Reserves	**High Zn**	**1,370.1**	**204.3**	**3.09**	**71.24**	**0.53**	**4.61**	**0.88**	**136.1**	**3,138.1**	**7.3**	**63.2**	**12.1**
	Mixed Zn/Cu	**3,744.9**	**184.5**	**2.48**	**45.02**	**0.22**	**2.32**	**1.82**	**298.6**	**5,420.4**	**8.2**	**86.9**	**68.2**
	High Cu	**1,363.4**	**169.1**	**2.05**	**21.86**	**0.06**	**0.84**	**3.27**	**89.9**	**958.2**	**0.8**	**11.5**	**44.6**
	Total	**6,478.4**	**185.4**	**2.52**	**45.69**	**0.25**	**2.49**	**1.93**	**524.6**	**9,516.7**	**16.2**	**161.4**	**124.9**

Notes:
1. The effective date of the Mineral Reserve Estimate is October 26, 2021.
2. Mineral Reserves are reported in accordance with CIM guidelines.
3. An NSR cut-off grade of 32.99$ was used for all material.
4. Mineral reserves were estimated at a gold price of 1,630 $/oz, a silver price of 21.00 $/oz, a lead price of 0.92 $/lb., a zinc price of 1.16 $/lb., and a copper price of 3.31 $/lb.; they include modifying factors related to mining cost, dilution, mine recovery, process recoveries and costs, G&A, royalties, and rehabilitation costs.
5. Figures have been rounded to an appropriate level of precision for the reporting of Mineral Reserves.
6. Due to rounding, some columns or rows may not compute exactly as shown.
7. The Mineral Reserves are stated as dry tonnes processed at the crusher.
8. Tonnages are presented in metric tonnes.
9. Ounces are presented in troy ounces.

16 MINING METHODS

16.1 Introduction

This section describes the development of the mine design and schedule for the open pit.

16.2 Mining Operation

The Project is planned as a conventional open pit operation with haul trucks, hydraulic excavators and loaders. The ore will be transported from the pit to its destination by truck; ore material will be loaded preferentially by wheel loaders and waste material will be loaded preferentially by hydraulic excavators. Ore material will either be sent directly to the plant ROM pad or to the ore stockpile which then would to be rehandled and sent to the mill at a later time. At the stockpile, the different ore material types (High Zn, Mixed Zn/Cu, and High Cu) will be stockpiled separately to enable blending. The mill has a capacity of 666 kt/y, of which a maximum of 15% High Cu material is accepted, until there is no longer sufficient supply of the other ore materials for blending.

Waste material will be sent to one of four destinations: the overburden stockpile, the saprolite stockpile, the tailings storage facility, or the waste rock facility. Overburden will be sent to the overburden stockpile, saprolite will be sent to the saprolite stockpile, and other waste material will be divided as required to stabilise the overburden and saprolite stockpiles, or to build the tailings storage facility. Additional waste material will be sent to the waste storage facility.

Figure 18.1 shows the location of the pit as well as the different material destinations.

DRA completed a Trade-Off Study between an owner-operated mining fleet and a contractor-operated fleet. The Trade-Off Study outcome recommended the contractor-operated case which was agreed on by DRA and Adventus. DRA received quotes from two (2) contractors, and the average of the two (2) quotes was used to determine the mining Capex and Opex which is further outlined in Section 21 of the Report.

The mine will operate year-round, seven (7) days per week, twenty-four (24) hours a day with two (2) 12-hour shifts a day. Fifteen (15) days of weather delays are considered in the mine plan.

16.3 Mining Design

Mineral Reserves were estimated for the Project based on the economic pit limit and pit design parameters detailed in Section 14. Total tonnage to be mined from this pit is 61.82 Mt, which will be mined over a 10-year mine life with an additional 1.5 years of pre-production.

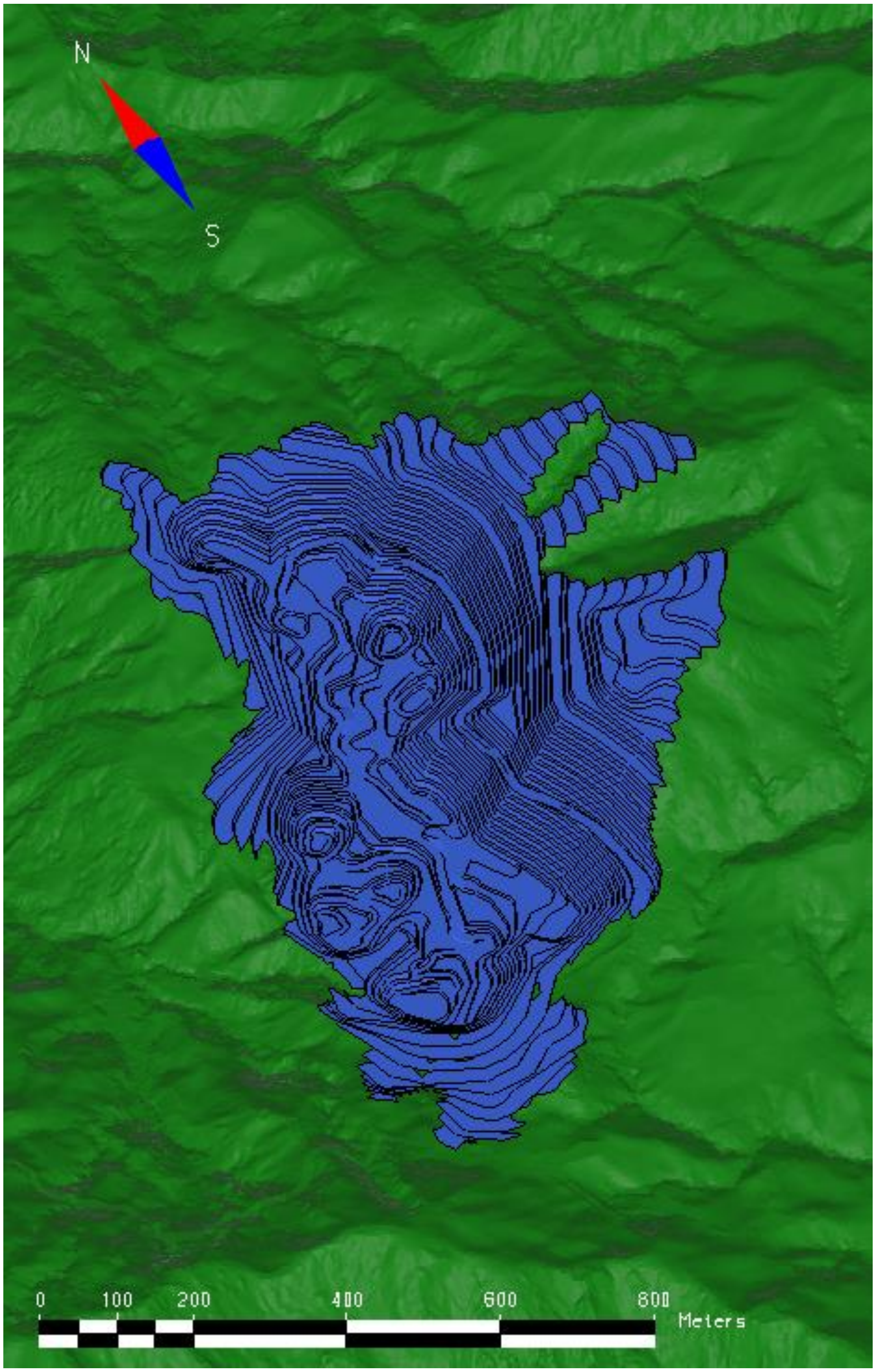
16.3.1 Pit Designs

The final pit design for the Project, as shown in Figure 16.1, follows the recommended geotechnical parameters and domains as defined in Section 16.4. A summary of the Reserve tonnages is presented in Table 16.1 and detailed tonnages are presented in Table 16.5.

Table 16.1 – Curipamba Reserve ($ US)

Material Type	Unit	Value
Ore	kt	6,478.4
	NSR ($/t)	185.4
	Au (g/t)	2.52
	Ag (g/t)	45.69
	Pb (%)	0.25
	Zn (%)	2.49
	Cu (%)	1.93
Waste	kt	55,343.6
Stripping Ratio	w/o	8.54

Figure 16.1 – Curipamba Pit



Source: DRA, 2021

16.3.2 Waste Stockpile Designs

The waste stockpiles have been designed to accommodate the required mine waste volumes. Waste stockpiles are further described in Section 20.8.

16.3.3 Ore Stockpile Design

There will be three (3) ore stockpiles, one for each of the ore types, as shown in Figure 16.2. The maximum size for each stockpile is presented in Table 16.2. The stockpiles will be located near the entrance of the pit along the road to the process plant, near or on top of the saprolite stockpile, depending on the year. The parameters used to design the stockpiles are presented in Table 16.3. Water drainage and run-off will be collected and enter the Saprolite dump water containment area for treatment in the TSF.

Table 16.2 – Ore Stockpile Requirements

Ore Type	Maximum Tonnage (t)	Maximum Volume (m^3)	Year Maximum is Reached
High Zn	126,000	53,000	1
Mixed Zn/Cu	178,700	69,000	1
High Cu	170,000	57,000	9

Table 16.3 – Ore Stockpile Design Parameters

Parameter	Unit	Value
Bench Height	m	5
Bench Width	m	5
Face Angle	°	60
Slope Angle	°	30
Ramp Width	m	8

Figure 16.2 – Ore Stockpile Locations

Source: DRA, 2021

16.4 Geotechnical

This section summarises the geotechnical investigation performed by Curimining at the El Domo deposit site and any additional work completed at DRA's request.

Claude Bisaillon, DRA QP, visited the Project site between the 25th and 29th of July, 2021. There was no ongoing drilling at the Project site but a review of some of the most recent drill core was performed as well as discussions with geotechnical core loggers from Curimining.

16.4.1 Pit Slope Angles

The open pit slope design parameters for the Project are based on geotechnical site investigations (completed by DRA, Adventus and Curimining), available local and regional geological data, drilling data and well-established geotechnical design methods.

Curimining personnel conducted all the fieldwork from 2007 to 2021. During this period, a total of five (5) HQ-sized triple-tube geotechnical drill holes for a total of 1,211.7 m were drilled, logged and sampled to collect geotechnical information from the walls of the planned ultimate pit. The five geotechnical drill holes are concentrated in the East wall portion of the pit as it is expected to be the highest wall. Over the years, geotechnical data (RMR and RQD) was also collected by Curimining from the vast majority of the 394 exploration drillholes. RMR data was collected from 343 holes, representing over 68,000 m of data, and RQD data was collected from 350 holes, representing over 66,000 m of data. Core photos of selected resource drill holes were reviewed for areas where insufficient geotechnical data had been collected.

DRA has identified 10 geotechnical units associated with the DRA has identified 10 geotechnical units out of the 14 primary rocks as identified by Curimining. Rock types with similar geotechnical properties and alteration types were combined (i.e Dacite and Dacite Breccia, and the various grainstones), while some other rock types were not considered as a geotechnical units (i.e. VMS, and the Sediments) which are dealt with as surficial rocks/soils. This was based on the core logging, and the results of the geotechnical investigation by Curimining.. Major geological structures (faults and foliation) have been included in the geotechnical slope stability analyses for the pit. Slope stability analyses were conducted using industry standard limit-equilibrium software Slide2 by RocScience, and in-house proprietary DRA tools. Various failure modes were looked at including bench scale, inter-ramp and overall slopes with groundwater conditions from dry to fully saturated slopes. Four sections were looked at, the most critical East wall then the North, South and West walls.

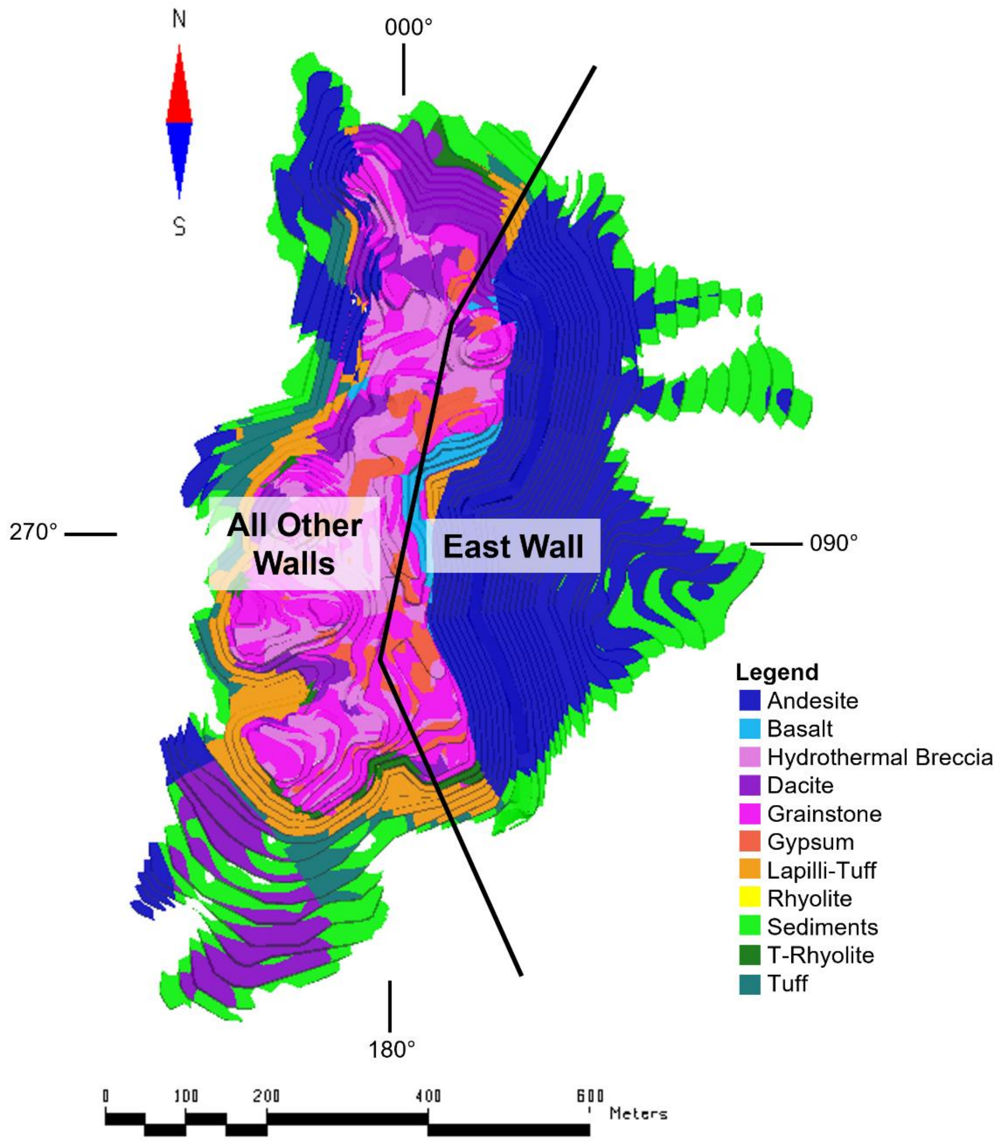
A point load testing program, set up early in the drilling program by Curimining, collected information from over 17,097 tests. A multi-component geomechanical laboratory testing program was designed by DRA but due to time constraints, only UCS testing was performed to correlate the point load testing. The testing was conducted at Construladesa Suelos y Hermigones S.A. in Guayaquil. The testing consisted of 102 samples from the 10 geotechnical units at El Domo. Care was taken by Curimining to select samples to spatially cover the entire El Domo deposit.

A field program to characterize the hydrogeological conditions was conducted as part of the investigations by KCB and is discussed in Section 16.5.

DRA used site observations, data, and statistical analyses to define geotechnical domains and select representative geomechanical properties. An appropriate quantity of quality data was collected to characterise the geological units of the study area and support FS-level slope designs. Domains were grouped by lithology and modelled in HxGN MinePlan.

The Curipamba pit has been divided into different geotechnical domains based on lithologies, the different structural components present in the area, and the anticipated main orientations of the proposed pit walls. These anticipated orientations are based on previous studies. The geotechnical domains are presented in Figure 16.3.

Figure 16.3 – Plan View of the Ultimate Pit Showing the Geotechnical Domains and Rock Types



Source: DRA, 2021

Recommended inter-ramp angles vary between 46.1° and 54°, based on wall orientation, overall wall height, geotechnical domain, and controls on slope stability. Inter-ramp slope heights are limited to 70 m, after which a geotechnical berm (or ramp) with a minimum width of 12 m or 14 m, depending on the area, is required. The inter-ramp height limits and geotechnical berms provide flexibility in the mine plan to mitigate potential slope instability, access slope monitoring installations, working space for drainholes and other water management infrastructure. All final pit slopes are assumed to be excavated using controlled blasting. Full depressurization of the pit slopes is required to meet the slope design criteria and should be achievable using drainholes.

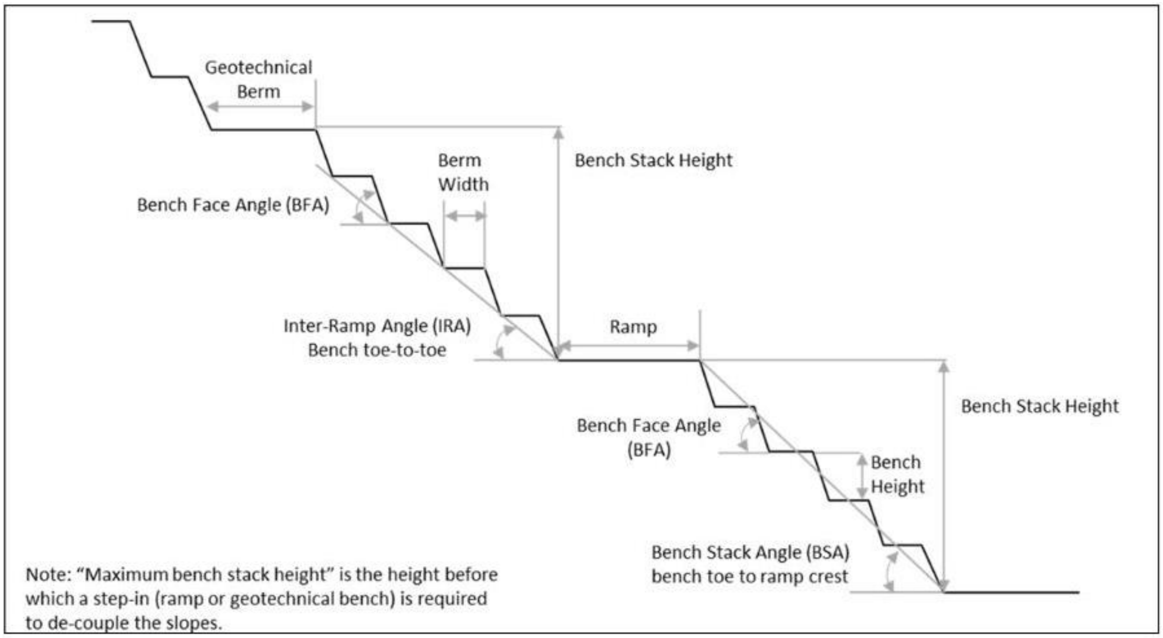
The Curipamba pit slope designs are presented in Table 16.4 and Figure 16.3.

Table 16.4 – Curipamba Pit Slope Design

<table>
<tr><th rowspan="2">Geotechnical Domain</th><th rowspan="2">Lithology</th><th>Wall Facing Direction</th><th>BFA</th><th>Max. Bench Height</th><th>Planned Berm Width</th><th>Design IRA</th><th>Stack Height</th><th>Geotechnical Berm Width</th><th>Overall Slope Angle</th></tr>
<tr><th>(°)</th><th>(°)</th><th>(m)</th><th>(m)</th><th>(°)</th><th>(m)</th><th>(m)</th><th>(°)</th></tr>
<tr><td rowspan="3">East Wall</td><td>Andesite</td><td>50-140</td><td>80</td><td>10</td><td>5.5</td><td>54</td><td>70</td><td>14</td><td>50.7</td></tr>
<tr><td>Tuff</td><td rowspan="2">090-130</td><td rowspan="2">80</td><td rowspan="2">10</td><td rowspan="2">6</td><td rowspan="2">52.2</td><td rowspan="2">70</td><td rowspan="2">14</td><td rowspan="2">49.2</td></tr>
<tr><td>Lapilli Tuff</td></tr>
<tr><td rowspan="2">All other Walls</td><td>Andesite</td><td>260-280</td><td>70</td><td>10</td><td>5.5</td><td>47.6</td><td>70</td><td>12</td><td>45.5</td></tr>
<tr><td>All other lithologies</td><td>180-320</td><td>70</td><td>10</td><td>6</td><td>46.1</td><td>70</td><td>12</td><td>44.2</td></tr>
<tr><td>Pit's Edge</td><td>Overburden and Saprolite</td><td></td><td>75</td><td>5</td><td>12</td><td>20.5</td><td></td><td>20 offset from edge of pit</td><td></td></tr>
</table>

Notes:
1. The slope angles are for fully drained slopes
2. Pit wall directions are based on the following directions: North, 0°; West, 90°; South, 180°; East, 270°
3. The East wall was used to evaluate the maximum wall height, which was used to determine the stack height
4. The design IRA is a toe-to-toe angle
5. Pre-splitting of the final pit wall to maximize effective berm width and limit breakback, as well as careful blasting, are required to minimize damage to the final pit wall slopes
6. The 20 m offset for the overburden and saprolite is intended to allow equipment access to remove any sloughed material

Figure 16.4 – Typical Open-Pit Wall Design



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16.4.2 Slope Design Implementation

Achieving the proposed design criteria will require the pit walls to be fully depressurized using drainholes because geological structures such as faults may affect bench and inter-ramp-scale slope stability.

Pit slope displacements will require monitoring at various scales. Inter-ramp and overall-scale slopes should be monitored for deformations. A state-of-the art monitoring system is recommended. This system could include a combination of systems, including multiple robotic theodolites and survey prisms, mobile slope stability radar units, slope inclinometers, piezometers, and extensometers. The system should also be computerized and use radio telemetry to provide real-time data to on-site geotechnical mining staff. This system has not been included in the Project's Capex.

16.5 Pit Dewatering

16.5.1 Pit Inflows

Initial groundwater inflow (KCB) and rainfall modelling (SLR) studies were undertaken during the FS. Groundwater inflows into the pit are expected to predominantly take place within the Weathered Bedrock hydrostratigraphic unit, as well as via preferential flow conduits along specific fault zones.

Groundwater inflows into the pit are estimated to gradually increase from year 1 to 4, and then varies between 50 L/s and 60 L/s until the end of mining in year 12.

The water-bearing capacity of the fault zones that transgress El Domo pit will condition the water transport, and their study represents an opportunity of optimization of the predicted flows and dewatering methods.

Annual rainfall predictions were collated with the phased pit expansion that resulted in expanding in flows and treatment requirements of over the life of mine.

16.5.2 Dewatering

Current in-pit dewatering will be undertaken by the mining contractor. It is envisaged to have a common lower pit area at the bottom of phase 1 where water will be pumped / directed for initial accumulation and temporary flood buffering. The mining contractor will be responsible for moving in-pit water from other portions of the open pit to the central sump.

From the phase 1 sump, the water will be pumped to a water treatment plant and discharged to a mine water pond for monitoring and discharge. The water treatment plant and mine water pond will be located to the west of the open pit.

The related water infrastructure is further described in Section 18 of this report.

16.6 Mine Planning

A mine plan (or schedule) was prepared to estimate a probable production scenario for the Project and assess the mine equipment fleet requirements as well as mine Capex and Opex for the financial model. The mine plan was based on feeding the mill 666 kt ore per year. High Copper material was limited to 15% of the total material sent to the mill until the end of the mine life.

Mine planning was performed using MinePlan's Schedule Optimizer (MPSO) based on the final pit design, the intermediate phases described in the following sections and the block model received from SLR.

The mine plan was estimated on a monthly basis for the pre-production period and the first year of production. The remaining nine (9) years of production were generated on a yearly basis.

16.6.1 Phase Design

DRA defined five (5) phases in order to mine the Curipamba deposit and ensure access to the mining faces throughout the life of mine. Other considerations were also taken into account, such as ensuring enough andesite could be extracted for the tailings dam construction. In addition, the main ramp was designed to act as both a ramp and a geotechnical berm, to reduce waste.

Benches were designed to be either 5 m or 10 m in ore, depending on the area, to maximise ore extraction, while they were designed to be 10 m in waste to reduce the amount of waste mined.

The phases are described in Table 16.5 and shown in Figures 16.5 to 16.11 A N-S cross-section showing each phase is presented in Figure 16.6.

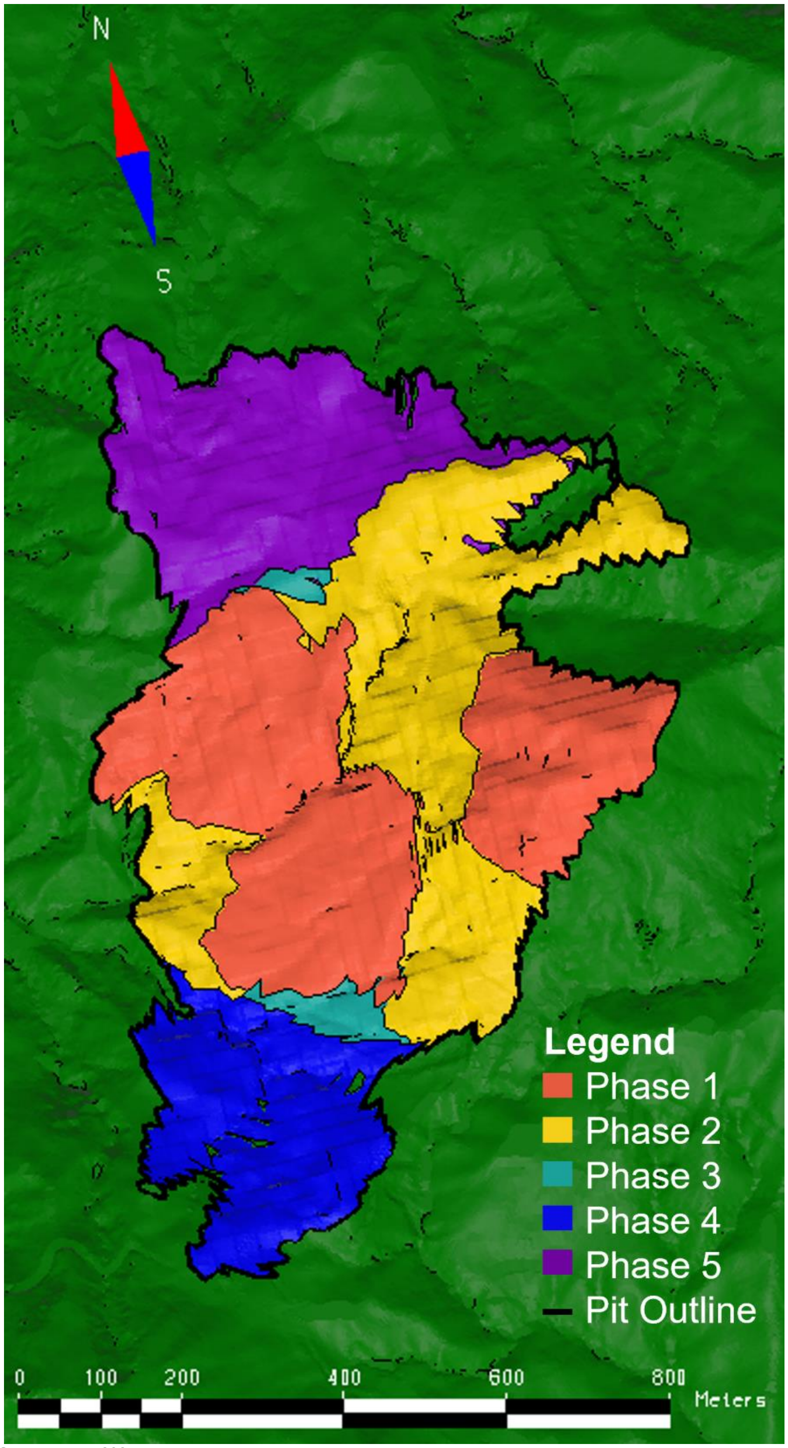
Table 16.5 – Material Mined by Phase

Phase	Ore Type	Tonnage (kt)	NSR ($/t)	Grade					Metal				
				Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Cu (%)	Au (koz)	Ag (koz)	Pb (kt)	Zn (kt)	Cu (kt)
Phase 1	High Zn	-	-	-	-	-	-	-	-	-	-	-	-
	Mixed Zn/Cu	5.53	56.9	0.63	11.96	0.08	0.37	0.72	0.1	2.1	0.0	0.0	0.0
	High Cu	-	-	-	-	-	-	-	-	-	-	-	-
	Ore Subtotal	*5.53*	*56.9*	*0.63*	*11.96*	*0.08*	*0.37*	*0.72*	*0.1*	*2.1*	*0.0*	*0.0*	*0.0*
	Waste	8,108.9	-	-	-	-	-	-	-	-	-	-	-
Phase 2	High Zn	573.4	185.2	3.01	59.25	0.54	4.06	0.73	55.5	1,092.3	3.1	23.3	4.2
	Mixed Zn/Cu	1,345.2	233.9	3.29	53.95	0.26	2.82	2.330	142.3	2,333.3	3.5	37.9	31.3
	High Cu	582.8	207.1	2.53	25.01	0.06	1.03	4.01	47.4	468.6	0.3	6.0	23.4
	Ore Subtotal	*2,501.5*	*216.5*	*3.05*	*48.42*	*0.28*	*2.69*	*2.33*	*245.3*	*3,894.2*	*7.0*	*67.3*	*58.3*
	Waste	14,376.5	-	-	-	-	-	-	-	-	-	-	-
Phase 3	High Zn	46.9	259.6	3.40	89.91	0.79	6.43	1.29	5.1	135.6	0.4	3.0	0.6
	Mixed Zn/Cu	154.3	146.2	1.88	47.11	0.22	2.23	1.32	9.3	233.7	0.3	3.4	2.0
	High Cu	1.8	173.3	2.14	24.91	0.04	0.89	3.33	0.1	1.4	0.0	0.0	0.1
	Ore Subtotal	*203.1*	*172.6*	*2.23*	*56.80*	*0.35*	*3.19*	*1.33*	*14.6*	*370.9*	*0.7*	*6.5*	*2.7*
	Waste	4,945.0	-	-	-	-	-	-	-	-	-	-	-
Phase 4	High Zn	333.6	244.9	3.44	94.88	0.66	5.67	1.11	36.9	1,017.6	2.2	18.9	3.7
	Mixed Zn/Cu	511.3	187.6	2.02	51.83	0.30	2.73	1.96	33.2	852.0	1.5	14.0	10.0
	High Cu	20.9	145.1	2.08	30.65	0.16	1.59	2.72	1.4	20.6	0.0	0.3	0.6
	Ore Subtotal	*865.8*	*208.7*	*2.57*	*67.91*	*0.44*	*3.84*	*1.65*	*71.5*	*1,890.4*	*3.8*	*33.2*	*14.3*
	Waste	4,423.7	-	-	-	-	-	-	-	-	-	-	-
Phase 5	High Zn	416.1	191.9	2.90	66.70	0.40	4.30	0.85	38.8	892.3	1.7	17.9	3.5
	Mixed Zn/Cu	1,728.7	148.8	2.05	35.98	0.16	1.82	1.46	113.9	1,999.7	2.8	31.5	25.2

Phase	Ore Type	Tonnage (kt)	NSR ($/t)	Grade					Metal				
				Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Cu (%)	Au (koz)	Ag (koz)	Pb (kt)	Zn (kt)	Cu (kt)
	High Cu	757.9	140.5	1.68	19.18	0.05	0.68	2.71	40.9	467.4	0.4	5.2	20.5
	Ore Subtotal	*2,902.7*	*152.8*	*2.08*	*36.00*	*0.17*	*1.88*	*1.70*	*194.1*	*3,359.7*	*4.9*	*54.6*	*49.3*
	Waste	23,270.2	-	-	-	-	-	-	-	-	-	-	-
Total Material	**High Zn**	**1,370.1**	**204.3**	**3.09**	**71.24**	**0.53**	**4.61**	**0.88**	**136.1**	**3,138.1**	**7.3**	**63.2**	**12.1**
	Mixed Zn/Cu	**3,744.9**	**184.5**	**2.48**	**45.02**	**0.22**	**2.32**	**1.82**	**298.6**	**5,420.4**	**8.2**	**86.9**	**68.2**
	High Cu	**1,363.4**	**169.1**	**2.05**	**21.86**	**0.06**	**0.84**	**3.27**	**89.9**	**958.2**	**0.8**	**11.5**	**44.6**
	Ore Total	***6,478.4***	***185.4***	***2.52***	***45.69***	***0.25***	***2.49***	***1.93***	***524.6***	***9,516.7***	***16.3***	***161.5***	***124.8***
	Waste	**55,343.6**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**

Notes:
1. The effective date of the Mineral Reserve Estimate is October 22, 2021.
2. Mineral Reserves are reported in accordance with CIM guidelines.
3. An NSR cut-off grade of 32.99$/t was used for all material.
4. Mineral reserves were estimated at a gold price of 1,630 $/oz, a silver price of 21.00 $/oz, a lead price of 0.92 $/lb., a zinc price of 1.16 $/lb., and a copper price of 3.31 $/lb.; they include modifying factors related to mining cost, dilution, mine recovery, process recoveries and costs, G&A, royalties, and rehabilitation costs.
5. Figures have been rounded to an appropriate level of precision for the reporting of Mineral Reserves.
6. Due to rounding, some columns or rows may not compute exactly as shown.
7. Tonnages are stated as dry tonnes
8. Tonnages are presented in metric tonnes.
9. Ounces are presented in troy ounces.

Figure 16.5 – Curipamba 5 Phases at beginning of pre-production

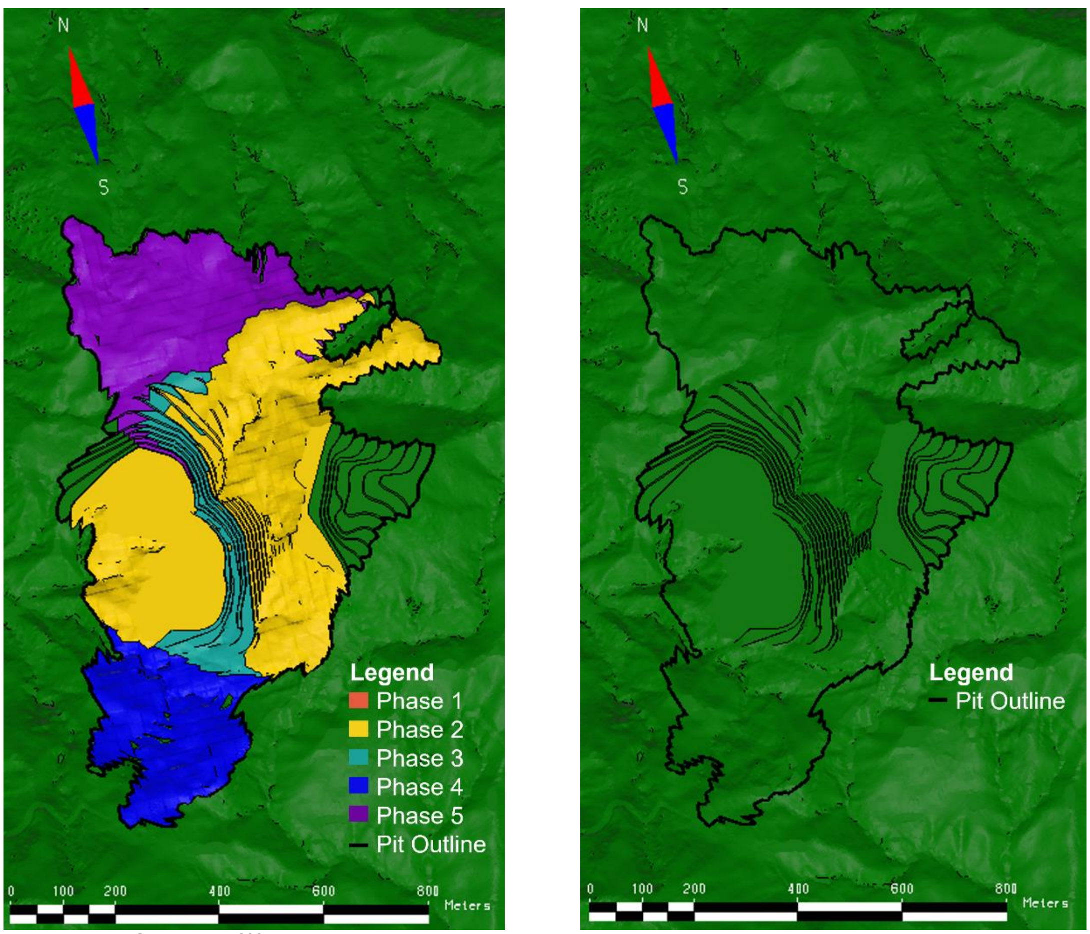


Source: DRA, 2021

Figure 16.6 – 5 Phases Cross-Section at Beginning of Pre-Production (N-S)

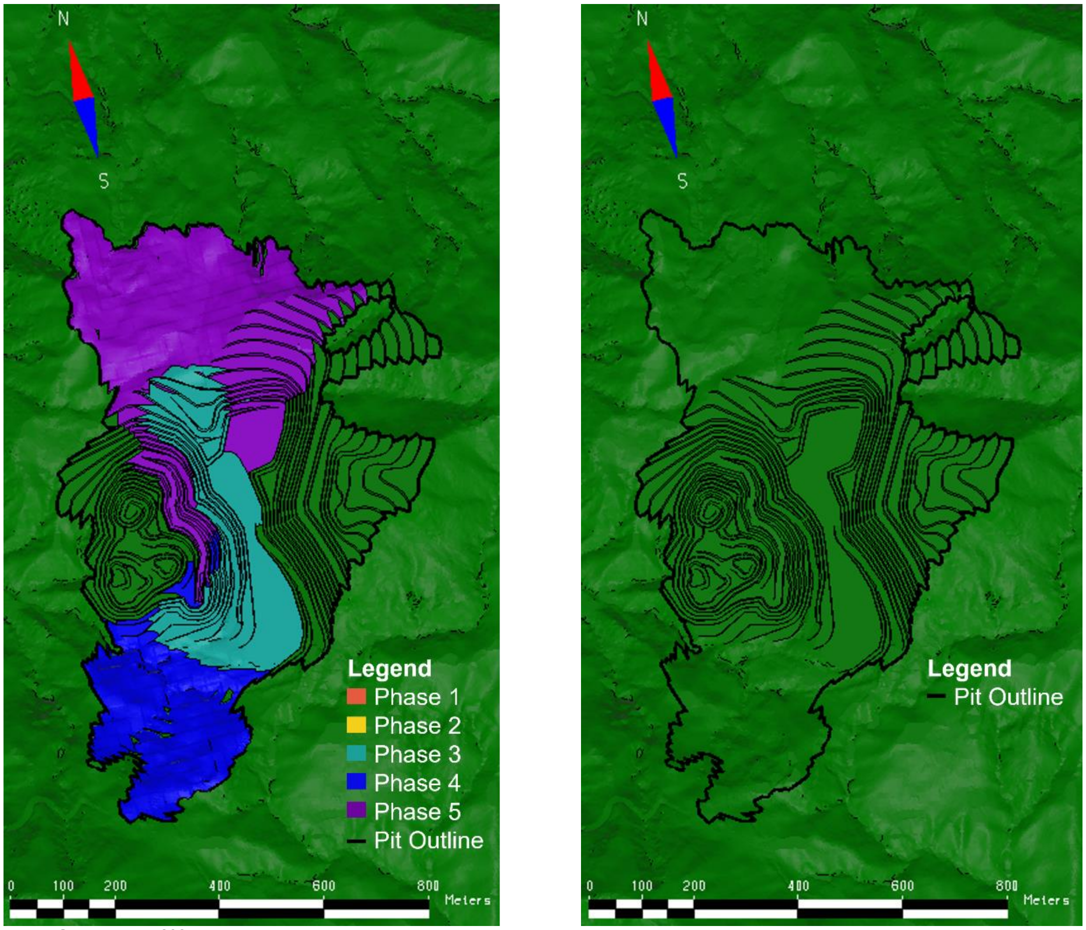
Source: DRA, 2021

Figure 16.7 – Pit at End of Phase 1



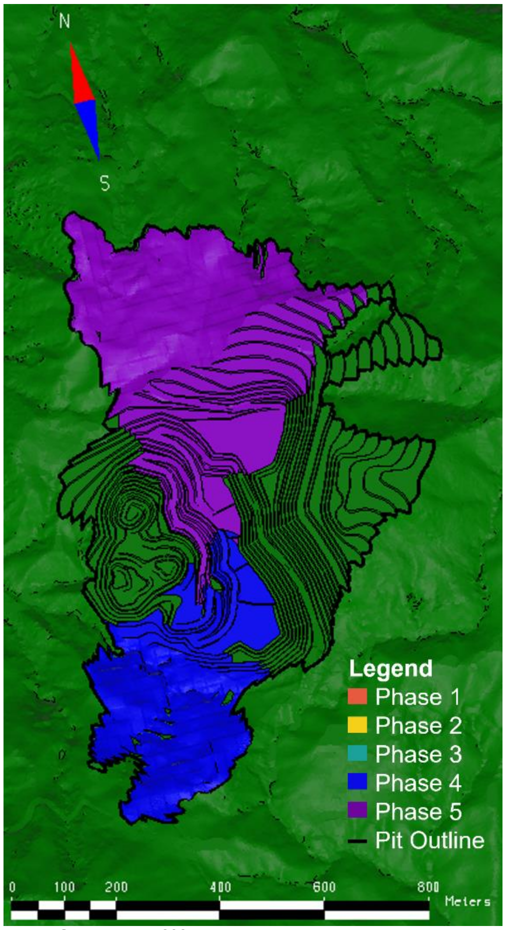
Source: DRA, 2021

Figure 16.8 – Pit at End of Phase 2



Source: DRA, 2021

Figure 16.9 – Pit at End of Phase 3



Source: DRA, 2021

Figure 16.10 – Pit at End of Phase 4

Source: DRA, 2021

Figure 16.11 – Pit at End of Phase 5

Source: DRA, 2021

16.6.2 Mine Production Schedule

The mine production schedule was established using the HxGN MinePlan Schedule Optimizer (MPSO), maximizing NPV, respecting blending constraints at the mill and minimizing material movement.

During pre-production, waste material is required to build the tailings storage facility prior to the start of production. As such, the pre-production period has a higher mining rate than the production period. During the pre-production period, the total material movement was capped at 1 Mt per month while it was capped at 6.5 Mtpy during production to ensure a feasible schedule considering the difficulty of mining an area with steep terrain. Additionally, the vertical advance rate was limited during pre-production to ensure the schedule is feasible.

The proportion of High Cu material sent to the mill was limited to 15% of the total mill feed in any given year to optimise the mill productivity and metal recoveries, as determined through process design. To ensure this constraint was met, the different ore types were stockpiled and rehandled as required. At the end of the open pit mine life, when there is no longer sufficient High Zn and Mixed Zn/Cu material to supply the mill at 0.67 Mtpy, a larger proportion of High Cu material was allowed.

A detailed mine schedule by material and destination is presented in Tables 16.6 and 16.7 and Figures 16.12 to 16.14 End of period maps are presented in Figures 16.15 to 16.26 In these figures, the final pit outline is represented by a thick line, and the areas being mined in each period is represented in purple. The contours are drawn on 5 m intervals.

16.6.3 Mine Closure

Mine closure is addressed in Section 20 of the Report.

Table 16.6 – Mine Production Schedule (Ore)

Year	Month	Pit to Mill				Pit to Stockpile				Total Extracted				Stockpile to Mill				Total to Mill			
		High Zn	Mixed Zn/Cu	High Cu	Total	High Zn	Mixed Zn/Cu	High Cu	Total	High Zn	Mixed Zn/Cu	High Cu	Total	High Zn	Mixed Zn/Cu	High Cu	Total	High Zn	Mixed Zn/Cu	High Cu	Total
		kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt
-1	11	-	-	-	-	-	5.5	1	5.5	-	5.5	-	5.5	-	-	-	-	-	-	-	-
	12	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1	1	-	6.0	-	6.0	0.3	-	0.1	0.4	0.3	6.0	0.1	6.4	-	-	-	-	-	6.0	-	6.0
	2	-	22.1	0.3	22.4	-	-	-	-	-	22.1	0.3	22.4	0.3	5.5	0.1	5.9	0.3	27.7	0.3	28.3
	3	-	55.5	-	55.5	-	1.1	-	1.1	-	56.6	-	56.6	-	-	-	-	-	55.5	-	55.5
	4	8.4	47.1	-	55.5	-	21.4	-	21.3	8.4	68.5	-	76.9	-	-	-	-	8.4	47.1	-	55.5
	5	-	55.5	-	55.5	19.7	68.7	-	88.3	19.7	124.2	-	143.9	-	-	-	-	-	55.5	-	55.5
	6	0.1	54.9	0.5	55.5	8.5	40.2	17.7	66.4	8.6	95.1	18.2	121.9	-	-	-	-	0.1	54.9	0.5	55.5
	7	-	55.5	-	55.5	7.0	18.2	29.4	54.7	7.0	73.8	29.4	110.2	-	-	-	-	-	55.5	-	55.5
	8	-	5501	0.4	55.5	22.5	10.9	15.4	48.8	22.5	66.1	15.7	104.3	-	-	-	-	-	55.1	0.4	55.5
	9	-	55.5	-	55.5	48.2	18.1	7.1	73.4	48.2	73.6	7.1	128.9	-	-	-	-	-	55.5	-	55.5
	10	0.3	6.2	-	6.5	1.9	10.3	7.8	20.0	2.2	16.5	7.8	26.5	-	39.0	10.0	49.0	0.3	45.2	10.0	55.5
	11	8.5	35.9	1.2	45.5	4.6	10.0	33.8	48.4	13.1	45.9	35.0	93.9	-	10.0	-	10.0	8.5	45.9	1.2	55.5
	12	2.6	22.8	0.2	25.5	14.1	16.7	30.7	61.5	16.7	39.4	30.9	87.0	-	30.0	-	30.0	2.6	52.6	0.2	55.5
2		63.2	239.6	59.4	362.2	37.8	3.0	58.7	99.5	101.0	242.5	118.1	461.7	126.5	136.7	40.6	303.8	189.7	376.3	100.0	666.0
3		116.0	346.0	83.3	545.4	-	-	89.8	89.8	116.0	346.0	173.1	635.2	37.8	3.0	79.8	120.6	153.8	349.0	163.2	666.0
4		270.2	266.4	0.5	537.1	-	-	138.9	138.9	270.2	266.4	139.4	676.0	-	-	128.9	128.9	270.2	266.4	129.4	666.0
5		254.3	317.3	3.1	574.8	39.3	120.0	27.0	186.3	293.6	437.3	30.1	761.1	-	-	91.2	91.2	254.3	317.3	94.3	666.0
6		181.2	225.4	5.5	412.2	20.1	90.0	1.8	111.9	201.3	315.5	7.3	524.1	39.3	120.0	94.5	253.8	220.6	345.4	100.0	666.0
7		110.2	377.5	86.9	574.7	34.8	108.2	4.7	147.7	145.0	485.7	91.6	722.4	-	78.2	13.1	91.3	110.2	455.8	100.0	666.0
8		12.6	443.1	100.0	555.7	49.5	110.3	158.9	318.7	62.155	553.4	258.9	874.4	-	110.3	-	110.3	12.6	553.4	100.0	666.0
9		26.5	207.2	289.3	523.0	7.5	97.2	49.4	154.1	34.0	304.4	338.7	677.1	-	100.0	43.0	143.0	26.5	307.2	332.3	666.0
10		-	100.6	61.6	162.2	-	-	-	-	-	100.6	61.6	162.2	111.9	117.2	170.0	399.1	111.9	217.8	231.6	561.3
Total		***1,054.1***	***2,995.2***	***692.3***	***4,741.6***	***316.0***	***749.9***	***671.1***	***1,737.1***	***1,370.1***	***3,744.9***	***1,363.4***	***6,478.4***	***316.0***	***749.9***	***671.1***	***1,737.1***	***1,370.1***	***3,745.1***	***1,363.4***	***6,478.7***

Notes:
1. Figures have been rounded to an appropriate level of precision.
2. Due to rounding, some columns or rows may not compute exactly as shown.
3. The quantities are stated as dry tonnes processed at the crusher.
4. Tonnages are presented in metric tonnes.

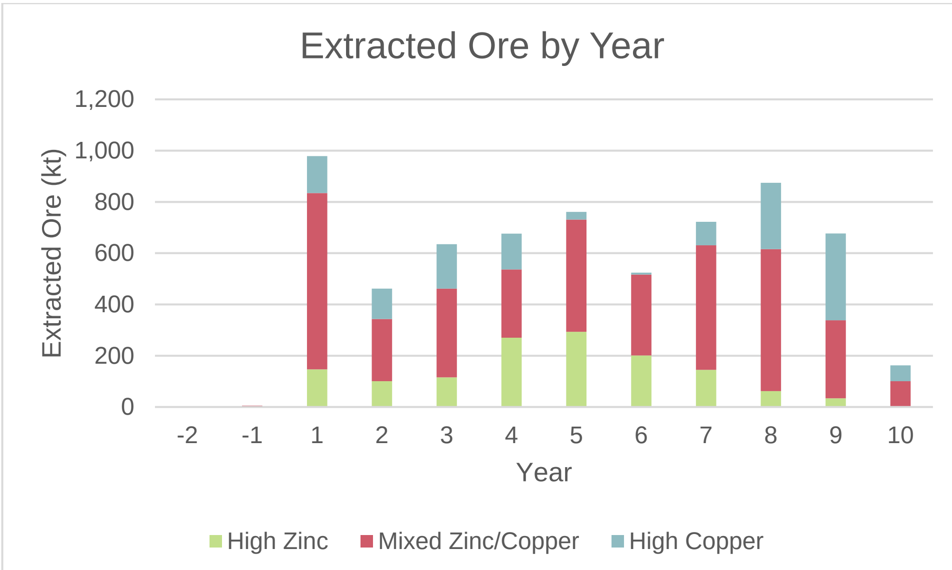
Table 16.7 – Mine Production Schedule (Waste)

Year	Month	Pit to Overburden Stockpile	Pit to Saprolite Stockpile					Pit to Tailings Storage Facility				Pit to Waste Rock Facility				Total to Waste Facilities					
		Overburden	Saprolite	Andesite	Tuff	General Waste	Total	Andesite	Tuff	General Waste	Total	Andesite	Tuff	General Waste	Total	Overburden	Saprolite	Andesite	Tuff	General Waste	Total
		kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt
-2	7	25.6	486.9	20.1	-	0.0	507.1	54.1	-	0.1	54.2	13.1	-	0.0	13.1	25.6	486.9	87.3	-	0.1	600.0
	8	15.1	2874	55.8	-	0.0	343.3	150.5	-	0.1	150.6	36.4	-	0.0	36.4	15.1	287.4	242.7	-	0.1	545.4
	9	13.1	249.8	60.2	-	0.0	310.1	162.3	-	0.0	162.3	39.3	-	0.0	39.3	13.1	249.8	261.8	-	0.0	524.8
	10	23.0	437.8	121.4	-	0.0	559.3	327.4	-	0.0	327.4	79.2	-	0.0	79.2	23.0	437.8	528.0	-	0.0	988.9
	11	16.0	304.2	101.9	0.0	0.0	406.1	274.7	0.0	0.0	274.7	66.4	0.0	0.0	66.4	16.0	304.2	443.0	0.0	0.0	763.2
	12	17.8	338.9	134.0	9.1	0.1	482.1	361.1	24.2	0.2	385.5	87.4	27.2	0.0	114.6	17.8	338.9	582.5	60.4	0.4	1,000.0
-1	1	16.7	317.2	92.4	39.6	0.5	449.3	249.2	105.6	0.1	354.9	60.3	118.8	0.0	179.12	16.7	317.1	402.0	264.0	0.1	1,000.0
	2	12.4	235.2	137.5	23.1	0.1	395.9	370.8	61.6	0.2	432.6	89.7	69.3	0.0	159.1	12.4	235.2	598.0	154.1	0.3	1,000.0
	3	15.4	291.8	147.1	7.9	0.2	447.0	396.6	21.0	0.3	418.0	96.0	23.6	0.0	119.6	15.4	291.8	639.8	52.5	0.5	1,000.0
	4	11.9	227.0	92.2	53.4	1.5	374.1	248.5	142.3	2.6	393.5	60.1	160.1	0.2	220.4	11.9	227.0	400.8	355.7	4.4	1,000.0
	5	8.9	169.1	70.4	62.8	34.0	336.3	189.8	167.5	58.2	415.6	45.9	188.4	4.8	239.2	8.9	169.1	306.2	418.8	97.0	1,000.0
	6	12.7	242.2	69.0	66.4	0.9	378.5	186.0	177.0	1.6	364.6	45.0	199.1	0.1	244.2	12.7	242.2	300.0	442.4	2.6	1,000.0
	7	8.3	157.5	69.0	63.9	37.8	328.3	186.0	170.4	64.8	421.3	45.0	191.7	5.4	242.2	8.3	157.5	300.0	426.1	108.1	1,000.0
	8	9.9	187.8	121.8	37.2	8.65	355.4	328.4	99.2	14.7	442.4	79.5	111.6	1.2	192.3	9.9	187.8	529.7	248.0	24.6	1,000.0
	9	8.1	153.8	130.9	20.5	46.3	351.5	352.8	54.7	79.4	486.9	85.4	61.5	6.6	153.5	8.1	153.8	569.1	136.7	132.3	1,000.0
	10	4.5	85.4	132.1	37.2	30.8	285.5	356.0	99.1	52.9	508.0	86.1	111.5	4.4	202.0	4.5	85.4	574.2	247.7	88.1	1,000.0
	11	6.9	131.8	57.4	34.2	132.3	355.7	154.7	91.3	226.7	472.7	37.4	102.7	18.9	159.0	6.9	131.8	249.5	228.3	377.9	994.5
	12	15.4	293.4	87.8	27.5	26.6	435.3	236.7	73.4	45.5	355.7	57.3	82.6	3.8	143.6	15.4	293.4	381.8	183.5	75.9	950.0
1	1	8.3	157.1	20.5	29.5	100.9	308.1	8.8	78.6	173.0	260.5	263.9	88.5	14.4	366.8	8.3	157.1	293.2	196.6	288.4	943.6
	2	0.8	15.7	38.9	18.6	78.9	152.1	16.7	49.5	135.3	201.4	500.3	55.7	11.3	567.2	0.8	15.7	555.9	123.8	225.4	921.6
	3	5.6	106.2	0.6	19.1	120.8	246.8	0.3	51.0	207.1	258.3	8.1	57.3	17.3	82.7	5.6	106.2	9.0	127.4	345.1	593.4
	4	4.3	82.2	0.0	17.3	129.8	229.3	0.0	46.1	222.5	268.5	0.6	51.8	18.5	71.0	4.3	82.2	0.7	115.1	370.8	573.1
	5	4.0	75.8	-	23.0	95.5	194.3	-	61.4	163.7	225,1	-	69.1	13.6	82.7	4.0	75.8	-	153.5	272.8	506.1
	6	3.7	70.5	-	31.7	85.0	187.2	-	84.4	145.7	230.1	-	95.0	12.1	107.1	3.7	70.5	-	211.1	242.8	528.1
	7	6.5	123.4	1,3	34.3	39.5	198.5	0.5	91.5	67.7	159.8	16.5	102.9	5.6	125.0	6.5	123.4	18.3	228.7	112.9	489.8
	8	12.5	237.2	1.3	13.2	48.6	300.4	0.6	35.3	83.2	119.1	17.1	39.7	6.9	63.7	12.5	237.2	19.0	88.2	138.7	495.6
	9	9.5	181.3	10.6	1.6	41.0	234.6	4.6	4.3	70.4	79.2	136.9	4.8	5.9	147.6	9.5	181.3	152.1	10.8	117.3	471.0

Year	Month	Pit to Overburden Stockpile	Pit to Saprolite Stockpile					Pit to Tailings Storage Facility				Pit to Waste Rock Facility				Total to Waste Facilities					
		Overburden	Saprolite	Andesite	Tuff	General Waste	Total	Andesite	Tuff	General Waste	Total	Andesite	Tuff	General Waste	Total	Overburden	Saprolite	Andesite	Tuff	General Waste	Total
		kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt
	10	3.6	68.1	14.3	2.6	32.1	138.0	6.1	62.9	55.0	124.0	183.5	70.8	4.6	258.9	3.6	68.1	203.9	157.2	91.6	524.4
	11	3.7	70.6	18.1	9.2	35.5	133.4	7.8	24.4	60.8	93.0	233.4	27.5	5.1	265.9	3.7	70.6	259.3	61.1	101.3	496.1
	12	3.3	63.5	16.5	9.4	58.7	148.1	7.1	25.1	100.6	132.8	212.2	28.2	8.4	248.8	3.3	63.5	235.8	62.7	167.7	533.0
2		42.7	812.3	110.8	130.0	435.3	1,488.2	27.7	345.6	746.3	1,119.7	2,632.8	388.9	62.2	3,083.9	42.7	812.4	2,771.4	864.1	1,243.8	5,734.5
3		55.9	1,061.8	468.6	209.2	523.6	2,263.2	121.5	558.0	897.6	1,577.1	1,145.4	627.8	74.8	1,848.0	55.9	1,061.8	1,735.5	1,395.0	1,496.0	5,744.2
4		68.2	1,296.3	114.1	320.8	404.2	2,135.4	72.6	855.4	693.0	1,621.0	850.3	962.4	57.8	1,870.4	68.2	1,296.3	1,037.0	2,138.6	1,155.0	5,695.1
5		3.4	65.1	1,152.1	267.3	837.1	2,321.7	1826	712.9	1,435.0	2,330.6	70.2	802.0	119.6	991.9	3.4	65.1	1,405.0	1,782.3	2,391.8	5,647.6
6		-	-	-	238.5	911.1	1,149.7	184.8	636.1	1,562.0	2,382.9	1,135.6	715.6	130.2	1,981.3	-	-	1,320.4	1,590.1	2,603.3	5,513.8
7		-	-	-	0.4	569.2	569.5	128.6	1.0	975.7	1,105.4	238.9	1.1	81.3	321.4	-	-	367.5	205	1,62601	1,996.2
8		-	-	-	-	215.4	215.4	130.3	-	369.3	499.6	-	-	30.8	30.8	-	-	130.4	-	615.4	745.8
9		-	-	-	-	240.5	240.5	64.4	-	412.3	476.7	-	-	34.4	34.4	-	-	64.4	-	687.2	751.6
10		-	-	-	-	24.2	24.2	2.7	-	41.4	44.1	-	-	3.4	3.4	-	-	2.7	-	69.1	71.8
Total		***478.1***	***9,084.7***	***3,669.1***	***1,879.1***	***5,346.3***	***19,979.3***	***5,553.6***	***5,011.0***	***9,123.6***	***19,729.8***	***8,755.1***	***5,637.4***	***763.8***	***15,156.3***	***478.1***	***9,084.7***	***17,977.9***	***12,527.5***	***15,275.3***	***55,343.6***

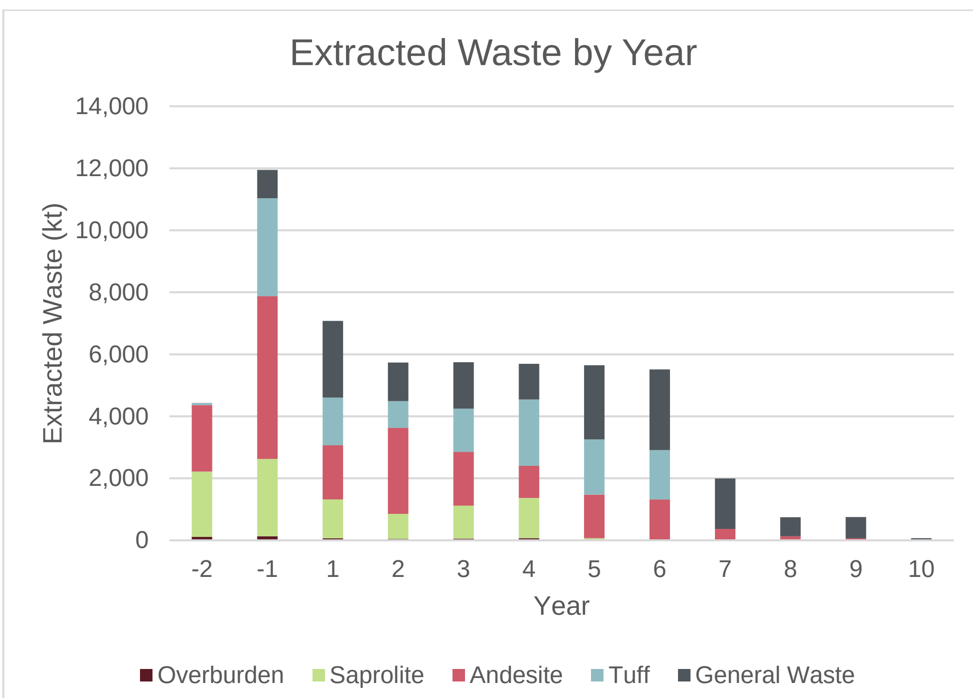
Notes:
1. Figures have been rounded to an appropriate level of precision.
2. Due to rounding, some columns or rows may not compute exactly as shown.
3. The quantities are stated as dry tonnes placed in a waste facility.
4. Tonnages are presented in metric tonnes.

Figure 16.12 – Mine Production Schedule (Ore)



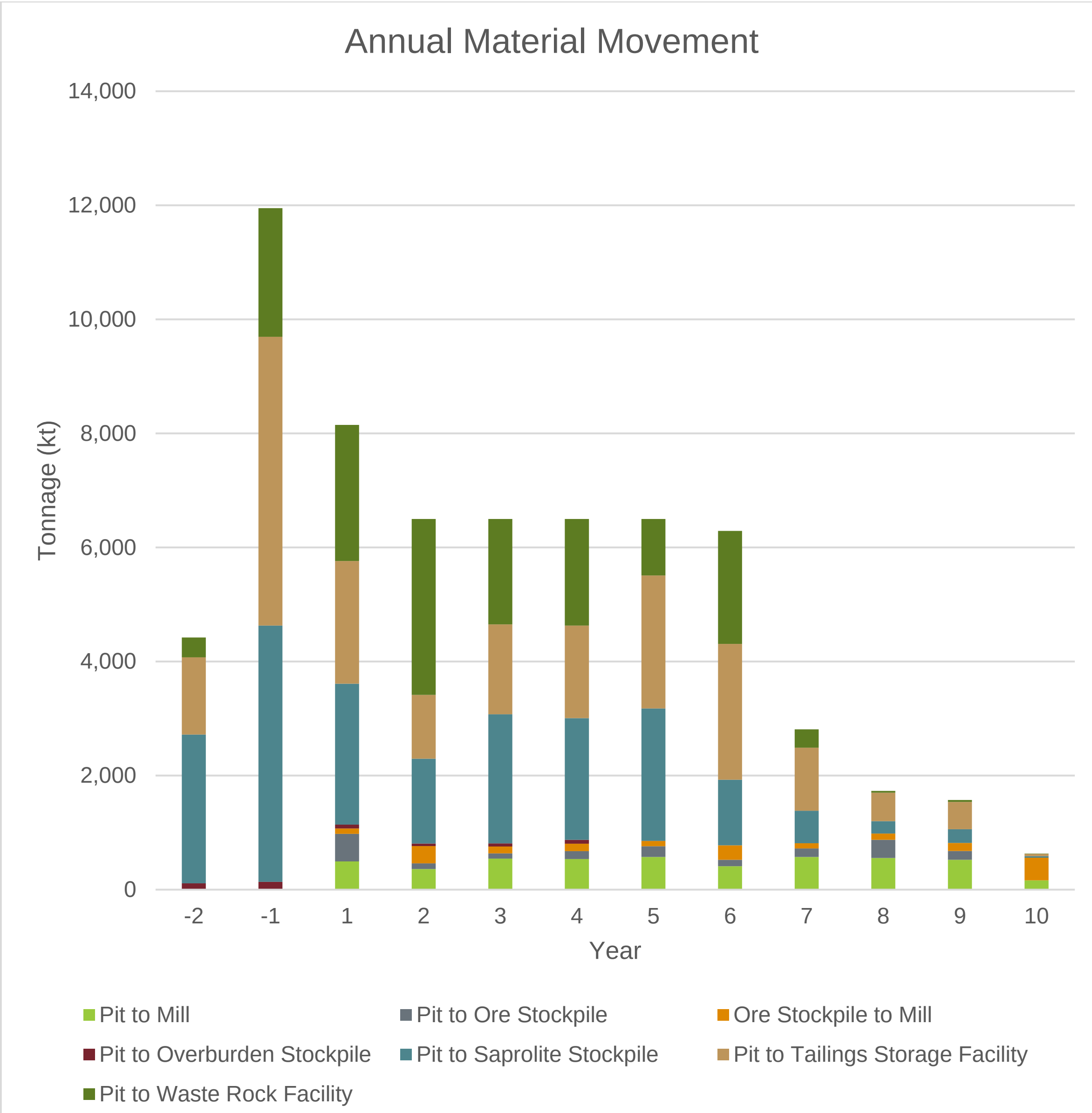
Source: DRA, 2021

Figure 16.13 – Mine Production Schedule (Waste)



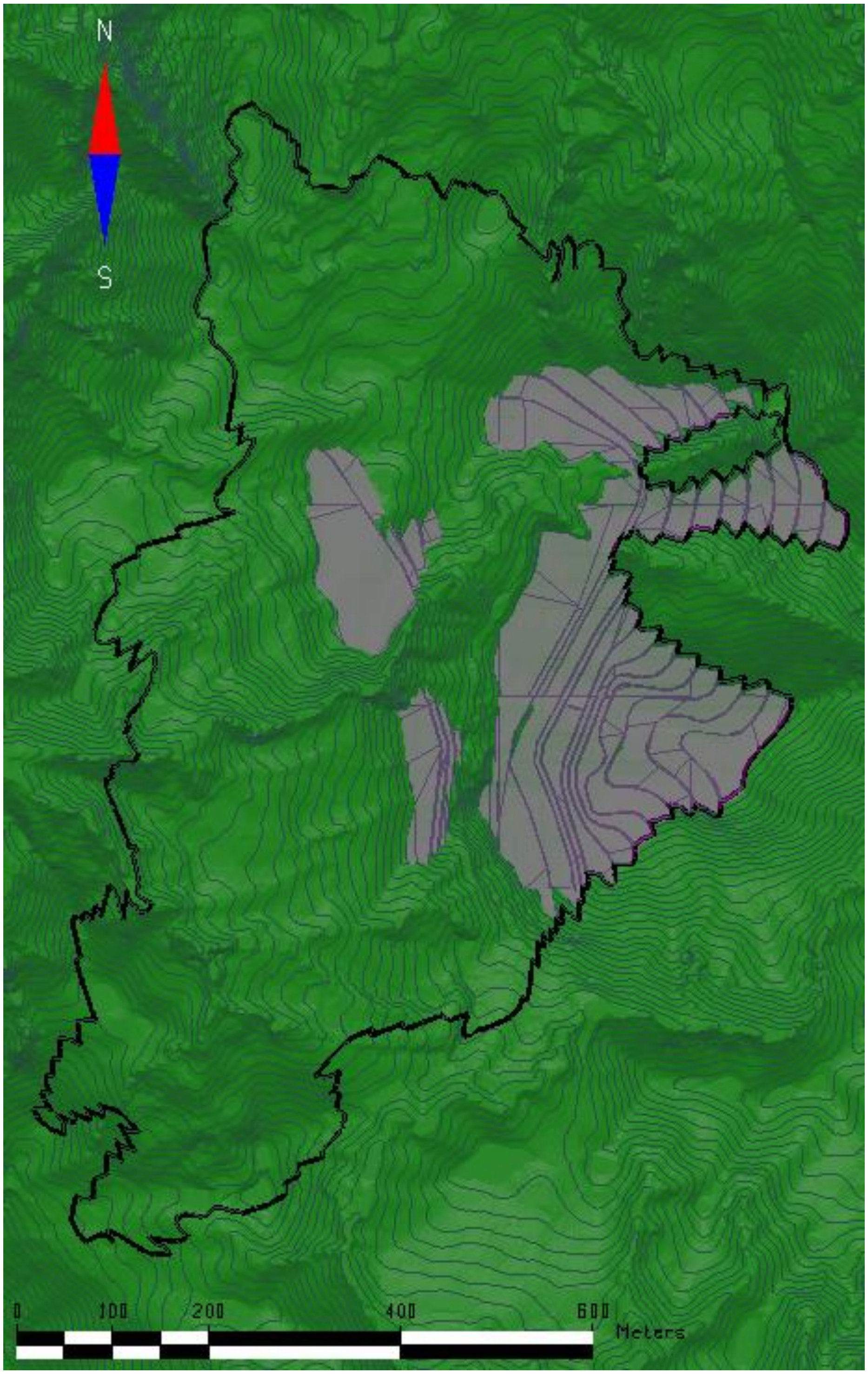
Source: DRA, 2021

Figure 16.14 – Total Material Movement



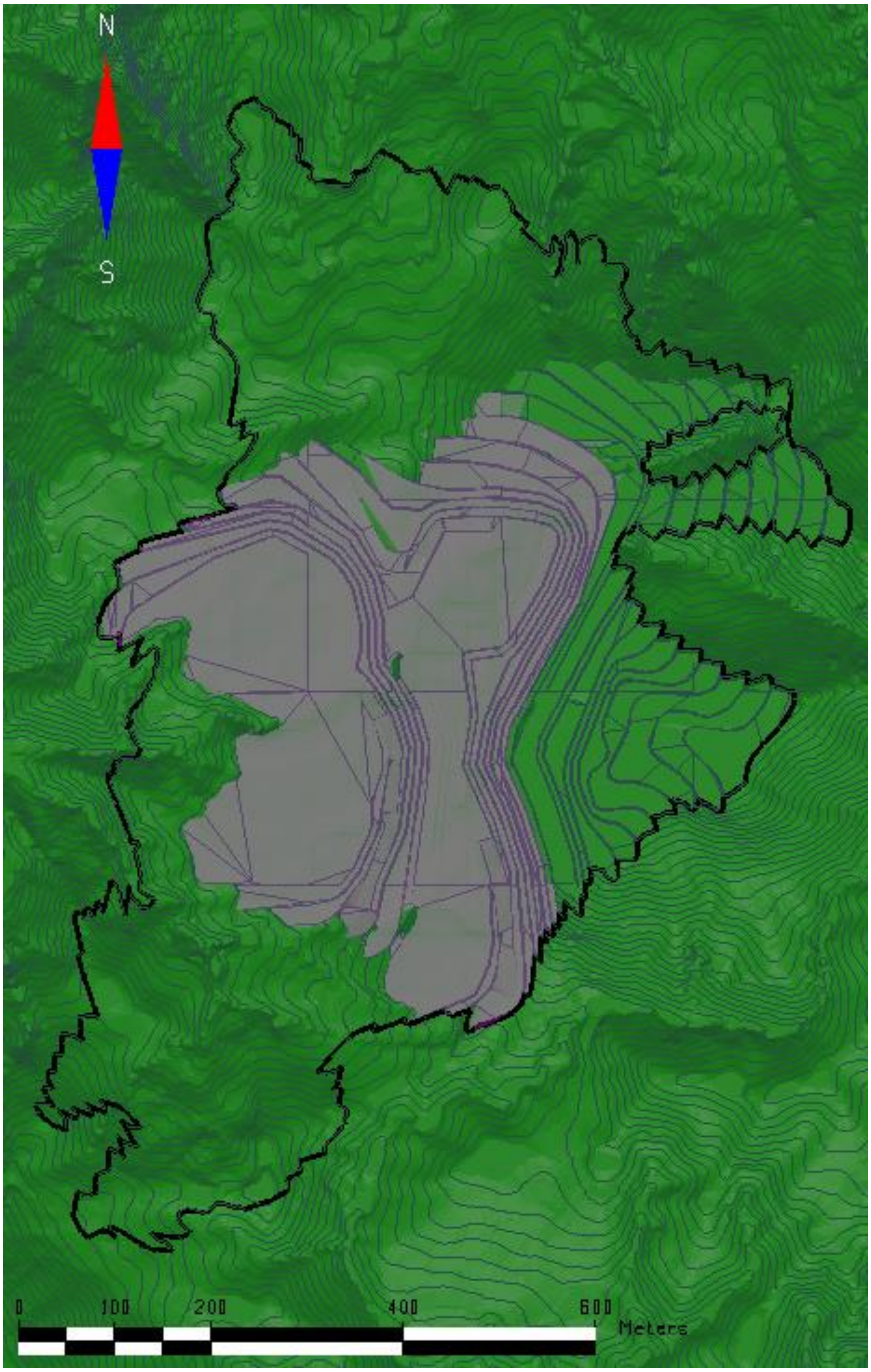
Source: DRA, 2021

Figure 16.15 – End of Period Map – End of First Year of Pre-Production



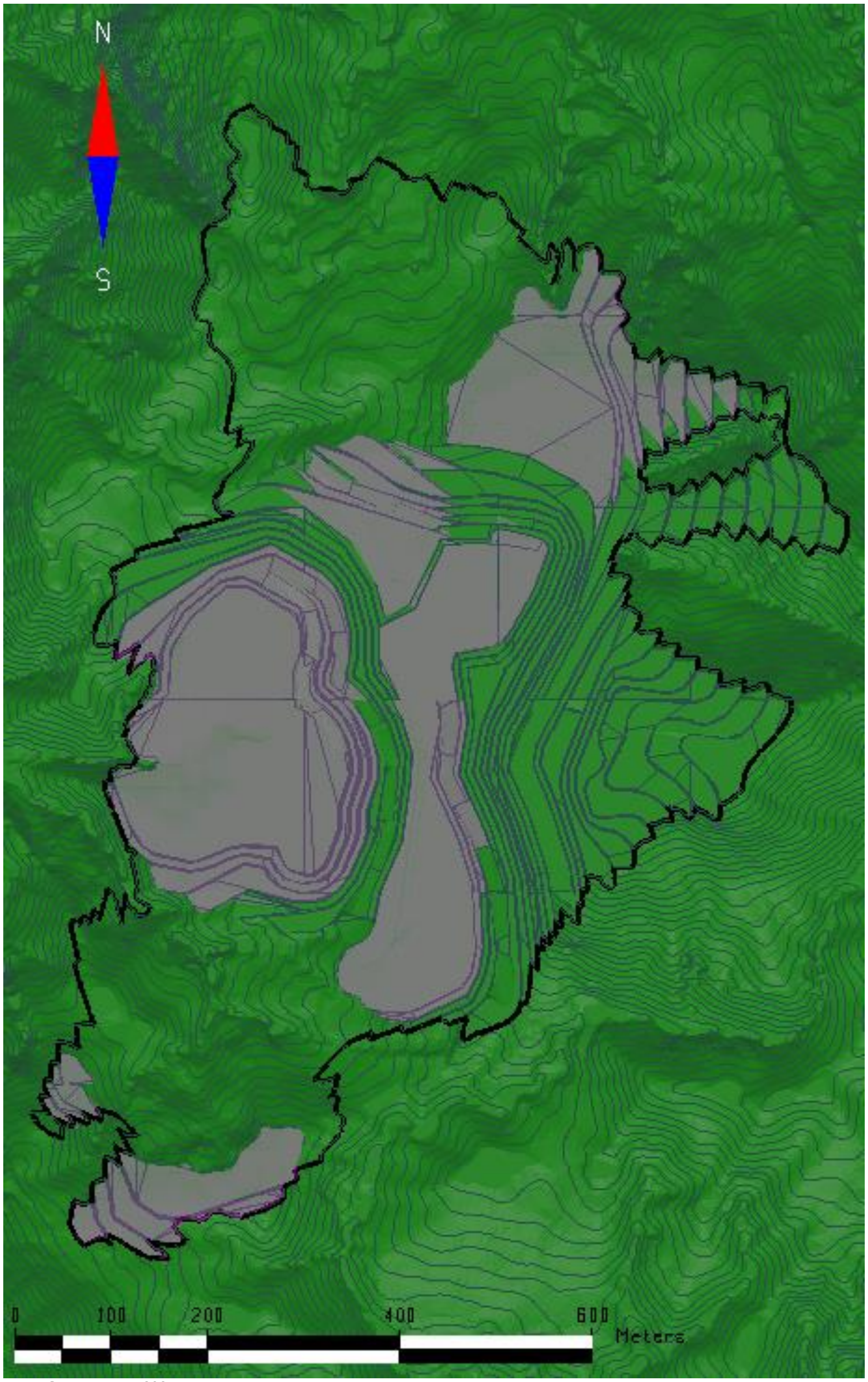
Source: DRA, 2021

Figure 16.16 – End of Period Map – End of Second Year of Pre-Production



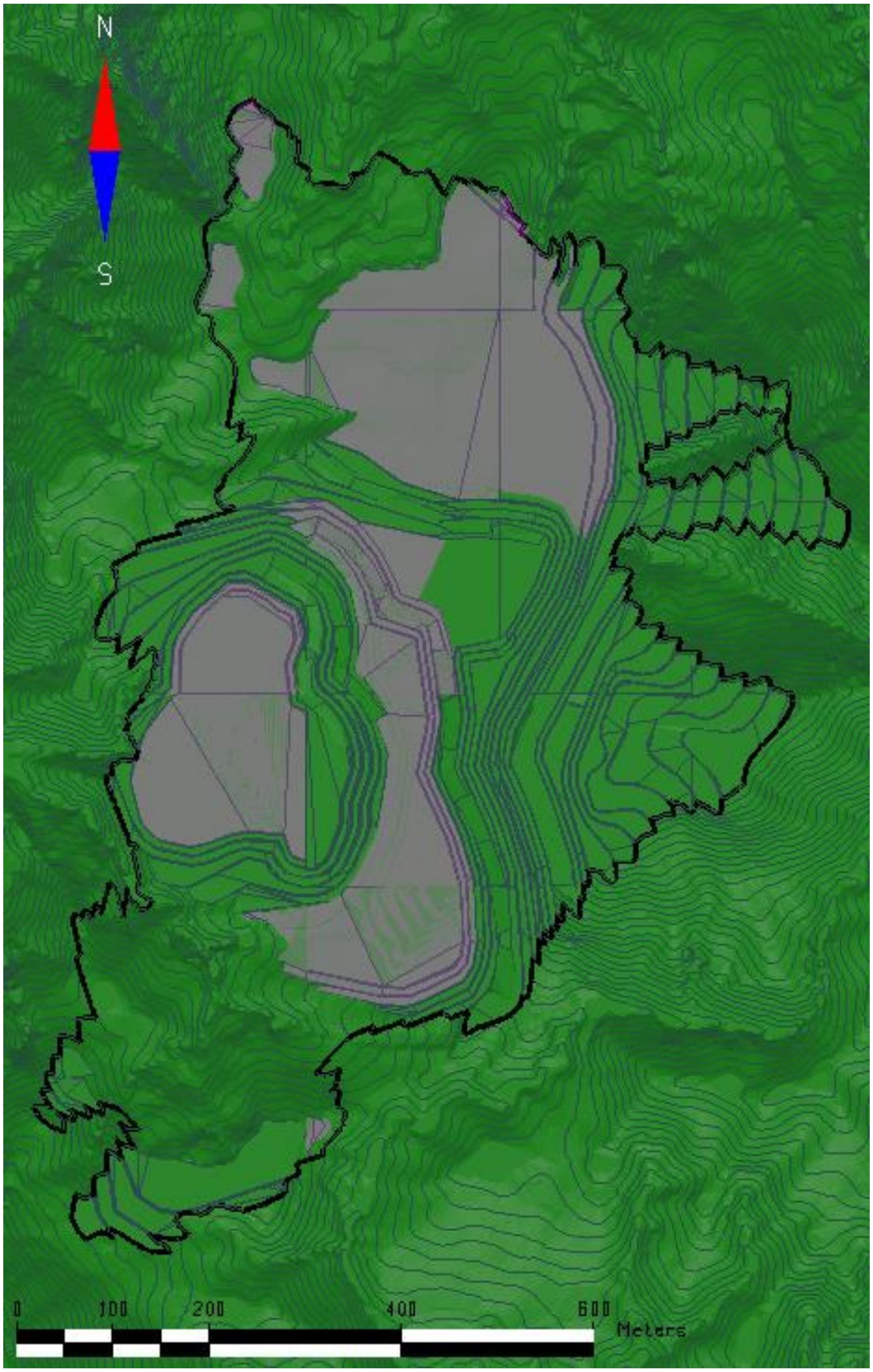
Source: DRA, 2021

Figure 16.17 – End of Period Map – End of Production Year 1



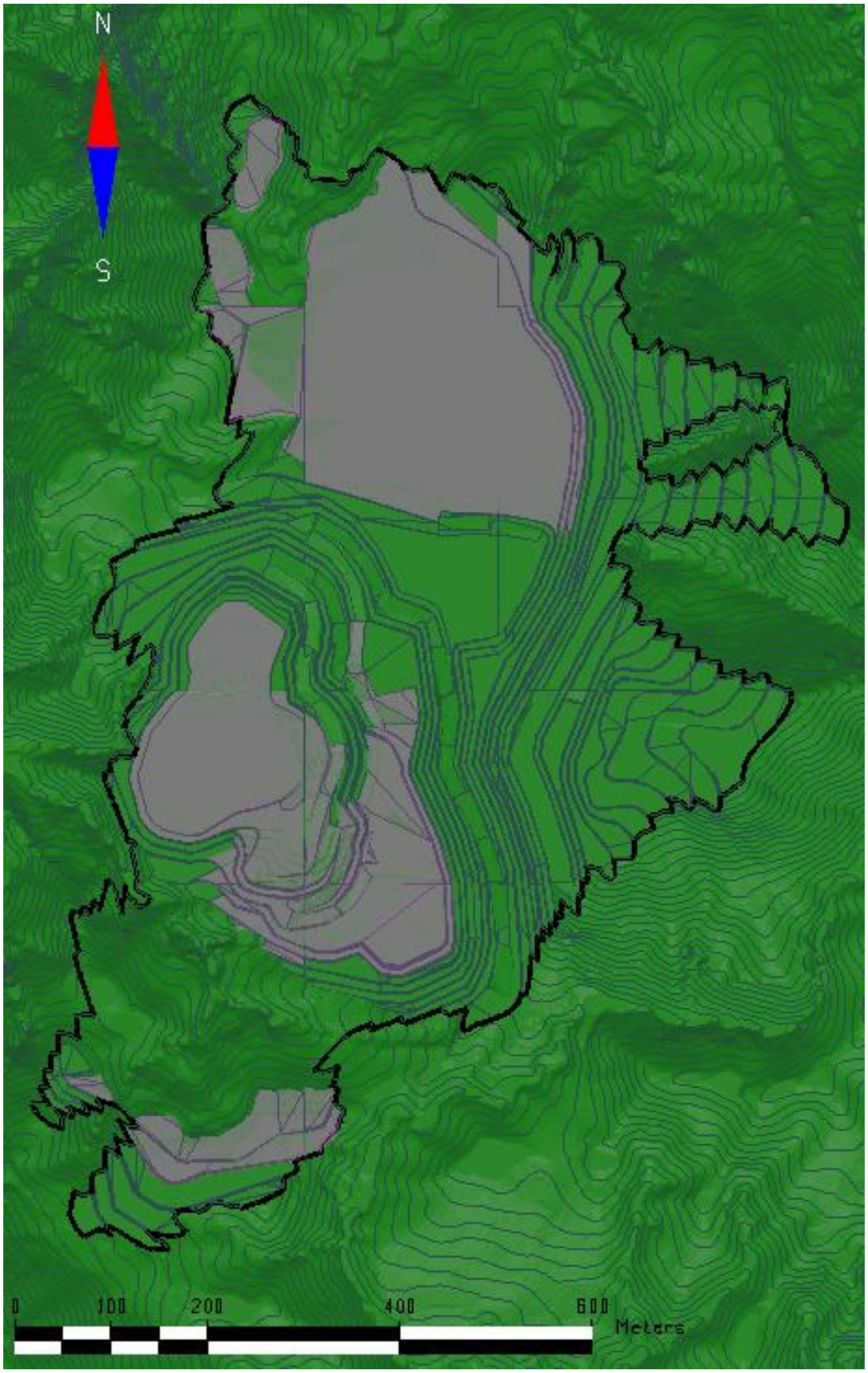
Source: DRA, 2021

Figure 16.18 – End of Period Map – End of Production Year 2



Source: DRA, 2021

Figure 16.19 – End of Period Map – End of Production Year 3

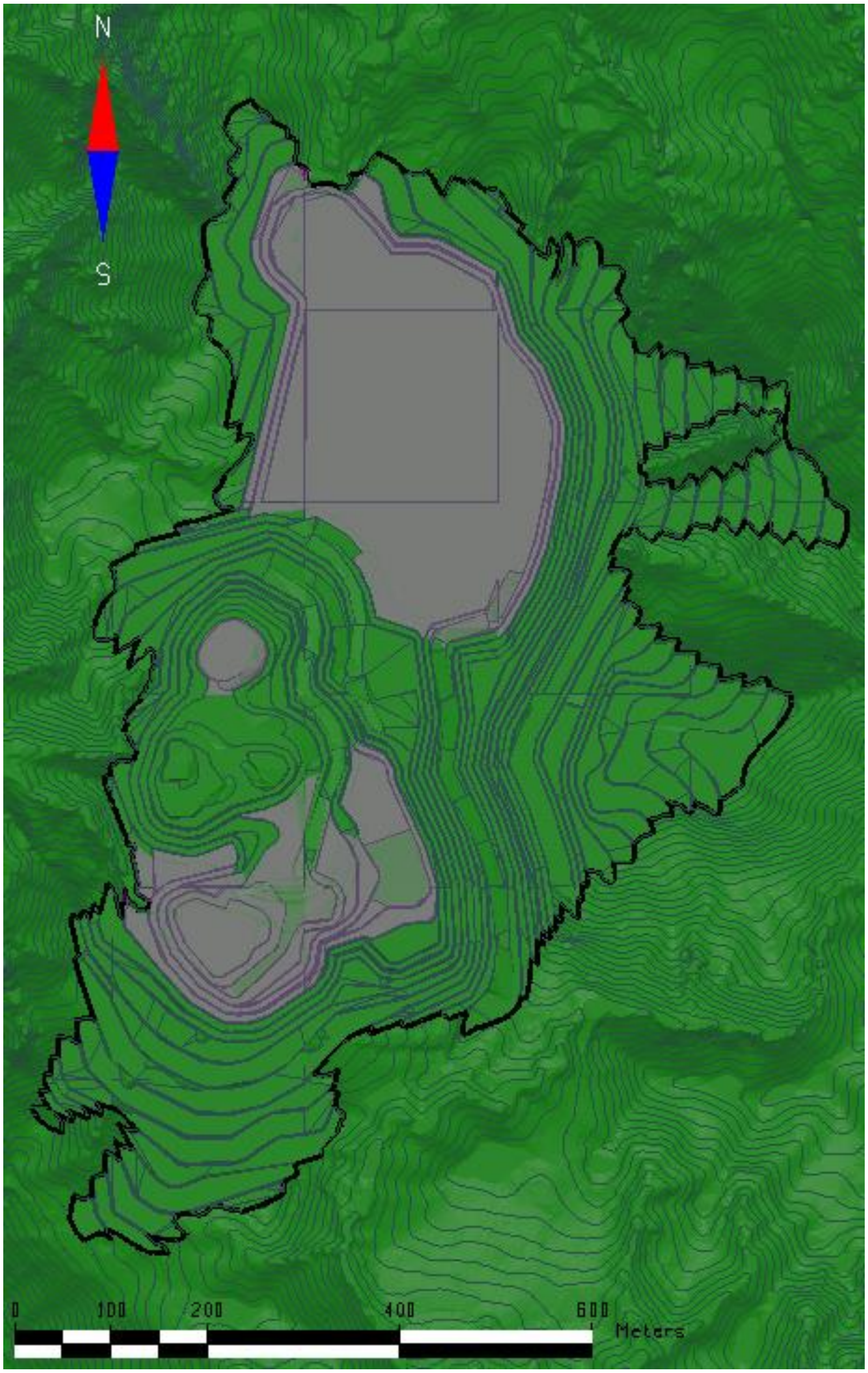


Source: DRA, 2021

Figure 16.20 – End of Period Map – End of Production Year 4

Source: DRA, 2021

Figure 16.21 – End of Period Map – End of Production Year 5



Source: DRA, 2021

Figure 16.22 – End of Period Map – End of Production Year 6

Source: DRA, 2021

Figure 16.23 – End of Period Map – End of Production Year 7

Source: DRA, 2021

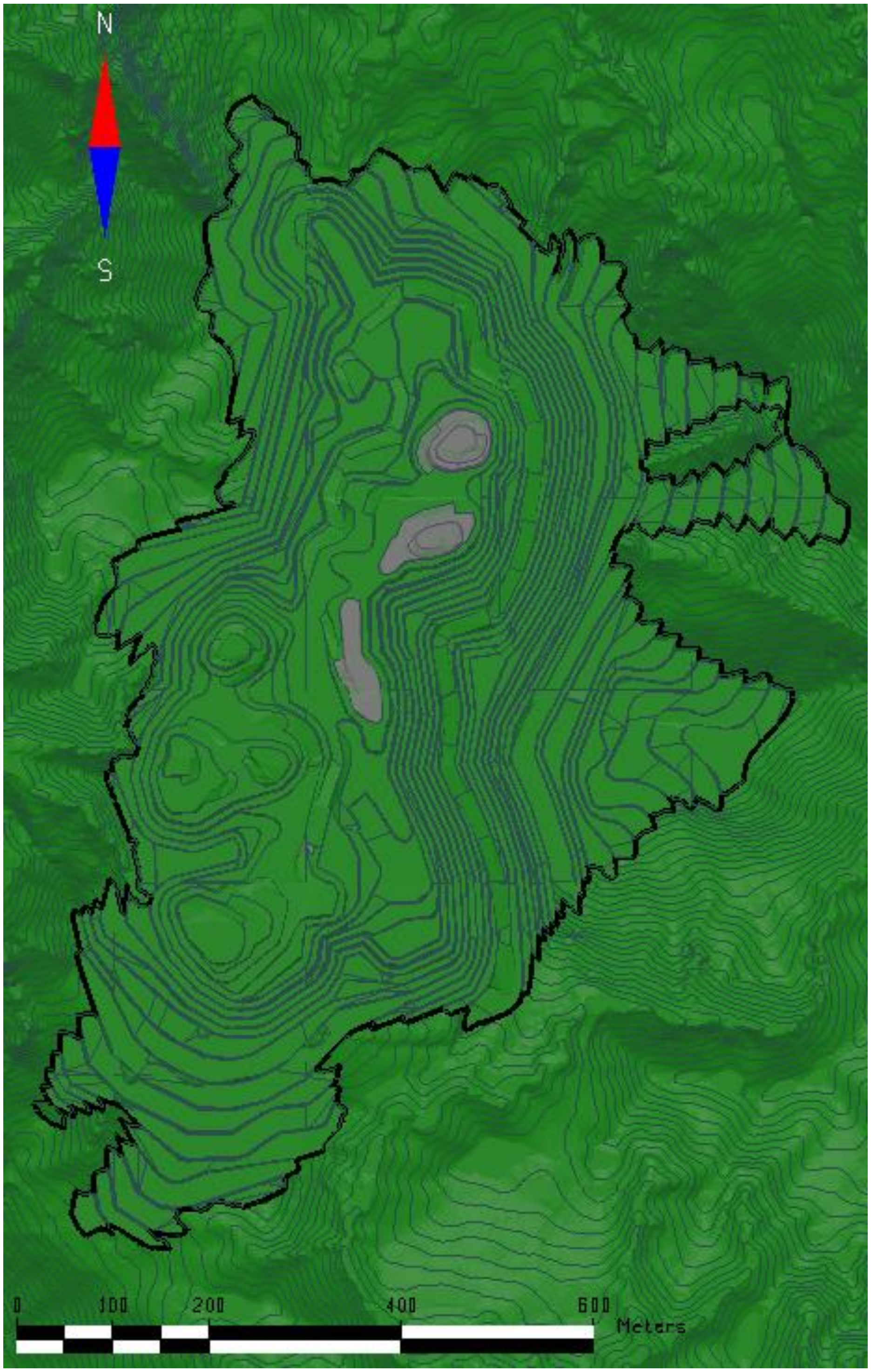
Figure 16.24 – End of Period Map – End of Production Year 8

Source: DRA, 2021

Figure 16.25 – End of Period Map – End of Production Year 9

Source: DRA, 2021

Figure 16.26 – End of Period Map – End of Production Year 10



Source: DRA, 2021

16.7 Mine Equipment

The following section discusses the fleet requirements to carry out the proposed mine plan. The mine will be operated by a contractor, who will be supplying their own fleet. The mine will operate on a two (2) shifts a day, twelve (12) hours per shift, 350 days per year basis, including weather delays.

16.7.1 Haulage

The contractor estimated their haul truck requirements based on the mine plan detailed in Section 16.6, the haulage distances listed in Table 16.8 and the following parameters:

- Mechanical availability: 87%;
- Utilisation: 80%;
- Operational delays: 85 min/shift, including lunch breaks, fueling, inspections and shift changes;
- Rolling resistance: 3%.

The contractor has assumed a fleet of 10 x 6 trucks (45 t payload) based on their experience with similar operations. Ann example of a 10 x 6 truck is presented in Figure 16.27. The expected number of trucks will be higher during pre-production and gradually decrease over the years following the gradual reduction in waste stripping, as shown in **Error! Reference source not found.**. Some trucks w ill be kept on-site as back-up equipment.

Figure 16.27 – Typical 10 x 6 Haul Truck

Source: 10x6 tipper - BAS Mining Trucks, 2021

Table 16.8 – Estimated Haulage Distances (km; round-trip)

Year	Pit to Mill	Pit to Stockpile	Stockpile to Mill	Pit to Overburden Stockpile	Pit to Saprolite Stockpile	Pit to Tailings Storage Facility	Pit to Waste Storage Facility
-2	-	2.3	1.5	6.4	8.2	8.1	6.0
-1	-	1.8	1.5	6.0	7.8	7.6	5.5
2	4.4	2.9	1.5	7.0	7.3	8.3	6.6
3	5.7	4.3	1.5	8.4	8.7	9.7	7.9
4	5.9	4.5	1.5	8.6	4.9	9.9	7.0
5	5.6	4.1	1.5	8.2	4.6	9.5	6.7
6	4.2	2.8	1.5	6.9	3.2	8.2	5.3
7	4.4	2.9	1.5	7.0	3.4	8.3	5.5
8	4.5	3.0	1.5	7.2	3.5	8.4	5.6
9	4.8	3.4	1.5	7.5	3.8	8.8	7.0
10	5.3	3.9	1.5	8.0	4.3	9.3	6.4
Average	***4.4***	***2.5***	***1.5***	***6.6***	***6.9***	***8.1***	***6.0***

NOTES
1. The haulage distances for Years -2, -1 and 1 were determined on a monthly basis and are averaged in this table
2. The average haulage distances were calculated using the monthly breakdown rather than the yearly average. As such, the average may not compute as shown.

Table 16.9 – Expected Fleet Requirements

Year	Number of Ore Trucks	Number of Waste Trucks	Total
-2	-	20	20
-1	-	22	22
1	1	18	19
2	1	11	12
3	1	11	12
4	1	10	11
5	1	10	11
6	1	8	9
7	1	3	4
8	1	2	3
9	1	2	3
10	1	1	2
Maximum	***1***	***22***	***22***

16.7.2 LOADING EQUIPMENT

The waste will be loaded onto trucks using hydraulic excavators, and the ore will be loaded using front-end loaders. Excavators will be used for the waste material, which will be mined in 10 m benches. The excavators will have a capacity of 3.3 m^3. The ore will be mined in 5 m benches to allow for more selectivity and a loader will be used to allow more flexibility in terms of areas to be mined. The loaders will have a capacity of 3.9 m^3. The following parameters were used to determine the required loading equipment:

- Mechanical availability: 73%;
- Utilisation: 67%;
- Operational delays: 85 min/shift, including lunch breaks, fueling, inspections and shift changes.

In addition to the loader in the ore, a second loader will be used for general clean up and stockpile rehandling. Based on the parameters above, the loaders will have a productivity of 931 t/h when matched with the trucks in ore and the excavators will have a productivity between 672 t/h and 900 t/h when paired with the trucks in waste.

The estimated number of required loading equipment is presented in Table 16.10.

Table 16.10 – Loading Equipment Requirements

Year	Number of Loaders	Number of Excavators	Total
-2	1	5	6
-1	1	5	6
1	2	5	7
2	2	3	5
3	2	3	5
4	2	3	5
5	2	3	5
6	2	3	5
7	2	1	3
8	2	1	3
9	2	1	3
10	1	1	2
Maximum	**2**	**5**	**7**

16.7.3 Drilling and Blasting

Production drilling will be carried out using down-the-hole drills. The overburden and saprolite will not require drilling or blasting, but the ore and remaining waste will require it. The drill pattern specifications are listed in Table 16.11, and the estimated number of required drills is listed in Table 16.12. An overall powder factor of 0.346 kg/t was estimated.

Pre-splitting of the final pit walls will be required to maximize effective berm width and limit breakback. Pre-split design will be completed in the detailed engineering stage.

Drilling and blasting activities as well as facilities maintenance will be carried out by the mining contractor. Explocen is the only explosives supplier in Ecuador, with a monopoly in the market. They will be responsible for the delivery of explosives materials to the site.

Bulk explosives are not available in the country, therefore packaged explosives materials will be used. For the ore, an average of 0.238 kg/t of emulsion and 0.064 kg/t of ANFO will be required, while waste will require an average of 0.151 kg/t of emulsion and 0.195 kg/t of ANFO.

Table 16.11 – Drill Pattern Specifications

Specification	Unit	Ore			Waste		
		High Zn	Mixed Zn/Cu	High Cu	Andesite	Tuff	Other Waste
Bench height	m	5			10		
Subdrilling	m	1			2		
Blasthole diameter	mm	114			114		
Blasthole effective diameter	mm	103			103		
Spacing	m	3.3			3.3		
Burden	m	3.3			3.3		
Hole length	m	6			12		
Redrill	%	5			5		
Effective hole length	m	6.3			12.6		
Productivity	t/h	446	483	555	421	409	442

Table 16.12 – Drill Requirements

Year	Number of Ore Drills	Number of Waste Drills	Total
-2	-	4	4
-1	-	5	5
1	1	4	5
2	1	3	4
3	1	3	4
4	1	3	4
5	1	3	4
6	1	3	4
7	1	1	2
8	1	1	2
9	1	1	2
10	1	1	2
Maximum	***1***	***5***	***5***

16.7.4 Support Equipment

The contractor will supply their own equipment fleet, including graders, dozers, water trucks, fuel & lube truck, maintenance truck, pumps, lights, etc. This equipment will perform tasks such as road maintenance, stockpile maintenance, loading face clean up, etc. The owner will have its own fleet of pick-up trucks for its personnel. The required support equipment is listed in Table 16.13.

Table 16.13 – Required Support Equipment

Description	Capacity	Year											
		-2	-1	1	2	3	4	5	6	7	8	9	10
Bulldozer	9 m^3	2	2	2	2	2	2	2	2	1	1	1	1
Grader	2.58 m	2	2	2	2	2	2	2	2	1	1	1	1
Water Tank	16,000 L	2	2	1	1	1	1	1	1	1	1	1	1
Fuel & Lube Truck		1	1	1	1	1	1	1	1	1	1	1	1
Service Truck		1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Truck	13.5 t	1	1	1	1	1	1	1	1	1	1	1	1
Bobcat	1.7 m^3	1	1	1	1	1	1	1	1	1	1	1	1
Owner Pick Up Trucks	4 x 4	5	5	5	5	5	5	5	5	5	5	5	5

Contractor Pick Up Trucks	4 x 4	5	5	5	5	5	5	5	5	5	5	5	5
Personnel Bus		2	2	2	2	2	2	2	2	1	1	1	1
Pumps		2	2	2	2	2	2	2	2	2	2	2	2
Lights		6	6	6	6	6	5	4	4	4	4	4	4

16.8 Manpower Requirements

The contractor will supply its own personnel for operations and maintenance. The owner's personnel, listed in Table 16.14, is based on management and supervision as well as geology and engineering requirements throughout the life-of-mine. The operations and maintenance staff will be supplied by the contractor. The contractor manpower requirements are listed in Table 16.15. Operations and maintenance staff are assumed to be working on a 7 days on/7 days off shift schedule while the owner staff as well as supervisory and administrative roles are assumed to work on a 5 days on/2 days off shift schedule.

Table 16.14 – Owner Manpower Requirements

Position	Year											
	-2	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
Mine General Foreman	1	1	1	1	1	1	1	1	1	1	1	1
Chief Engineer	0	1	1	1	1	1	1	1	1	1	0	0
Senior Mining Engineer	0	1	1	1	1	1	1	1	1	1	1	0
Open Pit Planning Engineer	1	1	2	2	2	2	2	2	2	2	1	1
Geotech Engineer	1	1	1	1	1	1	1	1	1	1	1	1
Surveyor/Mine Technician	2	2	4	4	4	4	4	4	4	4	2	1
Surveyor Helper/Mine Technician Helper	2	2	4	4	4	4	4	4	4	4	2	1
Clerk/Secretary	2	2	2	2	2	2	2	2	2	2	2	2
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	0
Senior Geologist	0	1	1	1	1	1	1	1	1	1	1	1
Grade Control/Modeller Geologist	0	1	2	2	2	2	2	2	2	2	2	0
Sampling/Geology Technician	1	1	1	1	1	1	1	1	1	1	1	1
Total	**9**	**14**	**22**	**23**	**24**	**25**	**26**	**27**	**28**	**29**	**24**	**19**

Table 16.15 – Contractor Manpower Requirements

Position	Year											
	-2	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
Mine Operations												
Technical Superintendent	1	1	1	1	1	1	1	1	1	1	1	1
Mine Shift Foreman	4	4	4	4	4	4	4	4	4	4	4	4
Road Crew/Service Foreman	1	1	1	1	1	1	1	1	1	1	1	1
Clerk/Secretary	1	1	1	1	1	1	1	1	1	1	1	1
Truck Operator	75	83	72	45	45	42	42	64	15	12	12	8
Excavator Operator	22	22	22	13	13	13	13	13	5	5	5	5
Loader Operator	5	5	9	9	9	9	9	9	9	9	9	5
Drill Operator	18	22	22	18	18	18	18	18	9	9	9	9
Support Equipment Operator	33	33	29	29	29	29	29	29	21	21	21	19
Blaster	4	4	4	4	4	4	4	4	4	4	4	2
General Mine Labourer	6	6	6	6	6	6	6	6	6	6	6	3
Road/Pump Crew	2	2	2	2	2	2	2	2	2	2	2	1
Trainee	1	1	1	1	1	1	1	1	1	1	1	0
Subtotal Operations	**173**	**185**	**174**	**134**	**134**	**131**	**131**	**153**	**79**	**76**	**76**	**59**
Maintenance												
Maintenance Shift Foreman	4	4	4	4	4	4	4	4	4	4	4	4
Maintenance Planner/Contract Admin	1	1	1	1	1	1	1	1	1	1	1	1
Clerk/Secretary	1	1	1	1	1	1	1	1	1	1	1	1
Lube Truck Driver	2	2	2	2	2	2	2	2	2	2	2	2
Tyre Man	3	3	2	2	2	2	2	2	2	2	2	2
Mechanic	3	3	2	2	2	2	2	2	1	1	1	1
Welder	3	2	2	2	2	2	2	2	1	1	1	1
Electrician	3	2	2	2	2	2	2	2	1	1	1	1
Apprentice	3	2	2	2	2	2	2	2	0	0	0	0
Subtotal – Maintenance	**23**	**20**	**18**	**18**	**18**	**18**	**18**	**18**	**13**	**13**	**13**	**13**
Total	**196**	**205**	**192**	**152**	**152**	**149**	**149**	**171**	**92**	**89**	**89**	**72**

17 RECOVERY METHODS

The process plant design is derived from the data and design criteria provided by Adventus, DRA, equipment vendor data, testwork results from Base Metallurgical Laboratories (BML) (2018 to 2021), Inspectorate Services Perú S.A.C. (2011), G&T Metallurgical Services (2011), SGS Mineral Services (2013), and Laboratorio BISA (2013). These results are provided in Section 13 of the Report and constitute the basis of process design detailed in the process design criteria, process mass balance, and process area flowsheets.

The comminution circuit design is based upon the design throughput requirements, ore competency and hardness, and power unit costs. Two-stage crushing followed by the Run of Mine (ROM) grate discharge variable speed ball mill circuit selection is based on DRA's experience on similar projects. The proposed grinding circuit is simple, produces stable flotation feed size, and is based on the numerous comminution studies completed in the past by DRA. This design for Curipamba offers the lowest net present cost compared to three-stage crushing and ball milling, two-stage milling SAG Mill and Ball Mill (SAB), and SAG Mill and Ball Mill and Pebble Crusher (SABC) circuits. These alternative circuits were evaluated by DRA in the numerous past studies for similar projects and all proved more complex and costly than the configuration selected for this study.

The ball mill size requirement is based on achieving the primary grind size required to obtain optimal flotation performance for the ore to be processed based on the outcomes of the metallurgical testwork.

Sizing calculations have been completed using first principle, power-based modelling methods.

The design of the flotation circuit is based upon the testwork results from 2018 to 2021. The results of the locked cycle testwork provided the basis for recovery and grade calculations as well as residence times.

Final flotation tailings and concentrates are thickened using conventional high-rate thickener technology. Concentrates are subject to filtering following thickening while thickened tailings are pumped to the Tailings Storage Facility (TSF).

Concentrates are dewatered using conventional plate and frame horizontal pressure filtration technology. The filtration circuits designs are based on pressure filtration testwork for the tailings stream and considers conventional design practices for flotation concentrates.

For other major equipment items generally determined by volumetric flow, the sizing requirement is based on 110% to 120% of the nominal flow.

Electrical power is provided by the local utility via an overland power line. The Ecuadorian government will build a 30 km power line to the site, with Adventus building a 7 km tie-in. Approximately 80 to 90% of the electrical power grid in Ecuador is supplied by hydro-electric power generation.

Emergency backup power is provided by onsite diesel generators to sustain critical electric loads.

17.1 Process Design Criteria

The Curipamba concentrator operation was designed to process 666,000 tonnes per year of ore into copper, lead and zinc concentrates. The design criteria was developed based upon: recent test work results summarised in Section 13, the Life of Mine (LOM) plan, process design calculations, and vendor budget quotations.

The process definition has been completed based on results of the of techno-economical trade-off studies which resulted in the following plant setup based on the financial assessment and risks evaluation:

- Flotation concentrator producing copper, lead and zinc concentrates with precious metal premiums;
- No ore pre concentration through ore sorting or dense media separation;
- Crushing area - two stage crushing with skid built direct powered crushing and screening modules and modular belt conveyors;
- Use of the front end loaders to feed the grinding area instead of the crushed ore stockpile tunnel or crushed ore silo;
- Use of conventional flotation cells for the flotation circuits.

Grinding circuit configuration of a single stage grate discharge ball mill in closed circuit with hydrocyclones was chosen based on the ore comminution properties, plant throughput required, and DRA's previous comminution design experience.

A summary of the process design criteria (PDC) is depicted in Table 17.1.

Table 17.1 – Key Process Design Criteria

Description	Unit	Value
Nominal Throughput – Daily	t/d	1,850
Nominal Throughput – Annual	t/y	666,000
Mill Feed Grades (LOM average)		
Cu	%	1.93
Zn	%	2.49
Pb	%	0.25
Au	g/t	2.52
Ag	g/t	45.69
S	%	9.01
Concentrate Production		
Copper Concentrate	t/y LOM average	39,751
Zinc Concentrate	t/y LOM average	24,139
Lead Concentrate	t/y LOM average	1,433
Concentrate Grades (LOM average)		
Copper Concentrate		
Cu	%	27.20
Ag	g/t	268.39
Au	g/t	12.55
Zinc Concentrate		
Zn	%	56.32
Ag	g/t	340.11
Au	g/t	8.14
Lead Concentrate		
Pb	%	33.84
Cu	%	16.33
Ag	g/t	680.17
Au	g/t	6.31
Recoveries (LOM average)		

Description	Unit	Value
Copper Concentrate		
Cu	%	84.3
Ag	%	33.2
Au	%	36.3
Zinc Concentrate		
Zn	%	81.9
Ag	%	24.1
Au	%	15.4
Lead Concentrate		
Pb	%	29.0
Cu	%	2.8
Ag	%	5.6
Au	%	1.2
Run Time		
Crushing circuit	%	70
Concentrator	%	92
Ore Hardness		
Crushing Work Index (CWi)	kWh/t	7.9
Abrasion Index (Ai)	g	0.16
Bond Ball Mill Work Index (BWi)	kWh/t	14.8
Bond Rod Mill Work Index (RWi)	kWh/t	11.7
SMC Values		
Mia	kWh/t	11.4
Mih	kWh/t	7.7
Mic	kWh/t	4.0
A x b		117

A key process equipment summary can be found in Table 17.2.

Table 17.2 – Key Process Equipment

Description	Unit	Value
Primary Crusher		
Type		single toggle jaw crusher, skid built
Installed Power	kW	110
Feed Size F_{80}	mm	433
Closed Size Setting	mm	125
Secondary Crusher		
Type		cone crusher, skid built
Installed Power	kW	220
Feed Size F_{80}	mm	103
Crusher Product P_{80}	mm	18
Crushing Screen		
Type		dry, double deck, skid built
Size		4.9 m L x 1.8 m W
Aperture	mm	50 top deck, 25 bottom deck
Installed Power	kW	11
Grinding		
Crushed Ore Stockpile (Live)	tonnes	880
Ball Mill		
Ball Mill Power Installed	kW	1550
Ball Mill Type and Size		13' x 19.5' ROM ball mill, grate discharge
Ball Mill Drive		Variable Speed
Ball Mill Liners		Steel
Ball Mill Circulating Load	%	250
Grinding Product P_{80}	µm	125
Bulk Flotation		
Cell Volume / Number of Cells	number of cells, m^3 per cell	8 x 19.8
Cell Type		tank cell

Description	Unit	Value
Bulk Regrind Mill		
Bulk Concentrate Regrind P_{80}	µm	15
Regrind Mill Type and Size		3.07 m H x 1.2 m Ø, vertical stirred mill
Regrind Mill Power Installed	kW	699
Regrind Mill Drive		Variable Speed
Regrind Mill Lining		Urethane
Mill Media Type		Ceramic
Copper Rougher		
Cell Volume / Number of Cells	number of cells, m^3 per cell	5 x 19.8
Cell Type		tank cell
Copper 1st Cleaner		
Cell Volume / Number of Cells	number of cells, m^3 per cell	6 x 2.8
Cell Type		mechanical
Copper 1st Cleaner Scavenger		
Cell Volume / Number of Cells	number of cells, m^3 per cell	3 x 2.8
Cell Type		mechanical
Copper 2nd Cleaner		
Cell Volume / Number of Cells	number of cells, m^3 per cell	3 x 2.8
Cell Type		mechanical
Lead 1st Cleaner		
Cell Volume / Number of Cells	number of cells, m^3 per cell	4 x 1.4
Cell Type		mechanical
Lead 2nd Cleaner		
Cell Volume / Number of Cells	number of cells, m^3 per cell	2 x 1.4
Cell Type		mechanical
Lead 3rd Cleaner		
Cell Volume / Number of Cells	number of cells, m^3 per cell	2 x 1.4
Cell Type		mechanical
Zinc Rougher		
Cell Volume / Number of Cells	number of cells, m^3 per cell	2 x 19.8
Cell Type		tank cell

Description	Unit	Value
Zn 1st Cleaner		
Cell Volume / Number of Cells	number of cells, m^3 per cell	4 x 2.8
Cell Type		mechanical
Zn 2nd Cleaner		
Cell Volume / Number of Cells	number of cells, m^3 per cell	3 x 2.8
Cell Type		mechanical
Zn 3rd Cleaner		
Cell Volume / Number of Cells	number of cells, m^3 per cell	2 x 2.8
Cell Type		mechanical
Concentrate and Tailings Thickeners		
Copper Concentrate Thickener	size, type, design	8 m Ø x 2.4 m sidewall, high-rate, bolted
Copper Concentrate Underflow Solids	%wt	60%
Zinc Concentrate Thickener	size, type, design	8 m Ø x 2.4 m sidewall, high-rate, bolted
Zinc Concentrate Thickener Underflow Solids	%wt	60%
Lead Concentrate Thickener	size, type, design	4 m Ø x 1.8 m sidewall, high-rate, bolted
Lead Concentrate Thickener Underflow Solids	%wt	65%
Final Tailings Thickener	size, type, design	30 m Ø x 3 m sidewall, high-rate, bolted
Final Tailings Thickener Underflow Solids	%wt	55%
Concentrate Filters		
Copper Concentrate Filter Press	area, type	29.2 m^2 plate and frame filter press
Copper Concentrate Filter Cake Moisture	%wt	10%
Zinc Concentrate Filter Press	area, type	29.2 m^2 plate and frame filter press
Zinc Concentrate Filter Press Cake Moisture	%wt	10%
Lead Concentrate Filter Press	area, type	8.8 m^2 plate and frame filter press
Lead Concentrate Filter Press Cake Moisture	%wt	10%

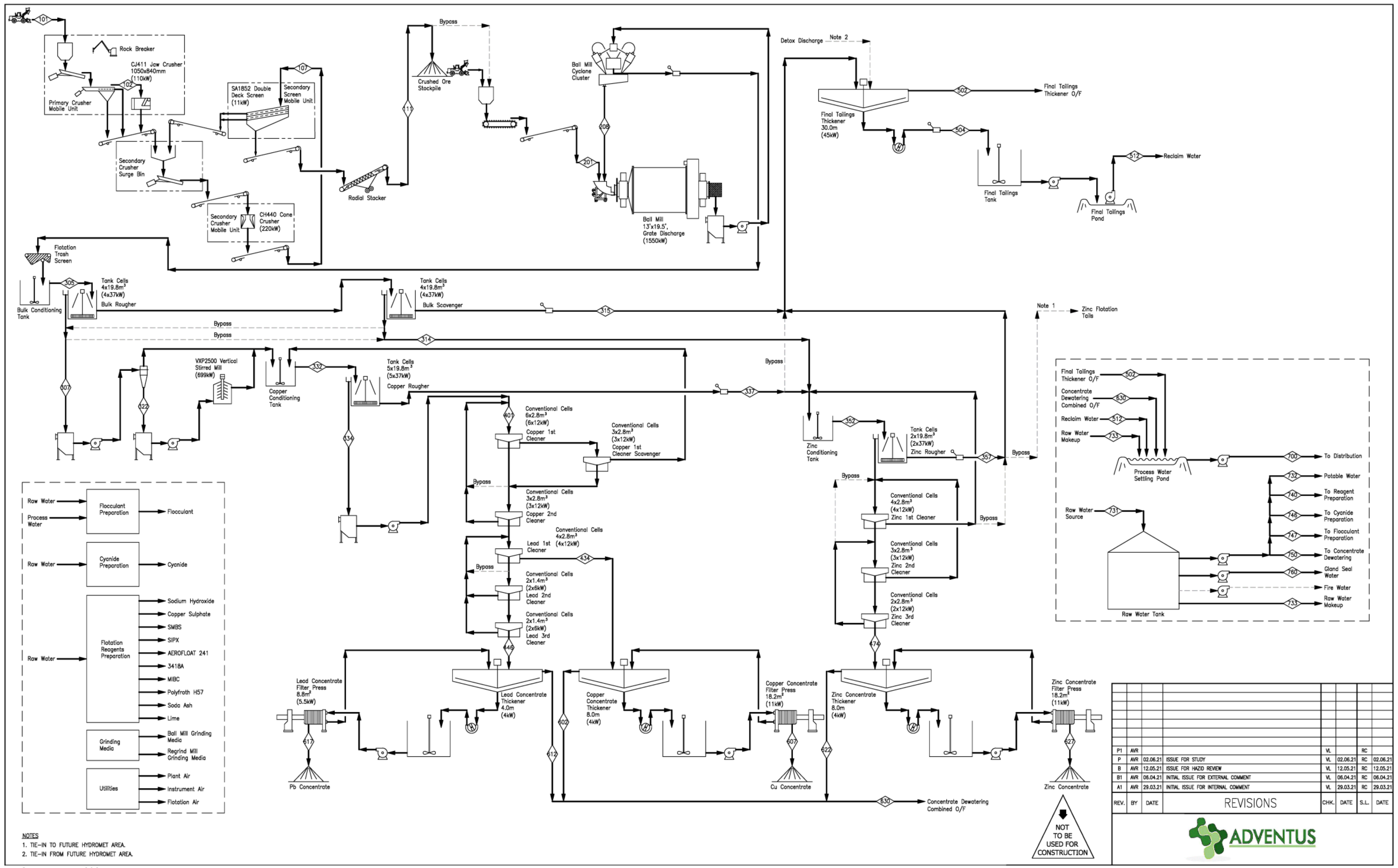
17.2 Process Description

The process consists of the following unit operations:

- Primary and Secondary Crushing;
- Grinding and Classification;
- Bulk Flotation and Regrind;
- Copper and Lead Flotation;
- Zinc Flotation;
- Copper Concentrate Dewatering;
- Lead Concentrate Dewatering;
- Zinc Concentrate Dewatering;
- Concentrate Storage and Loadout;
- Reagent Preparation and Distribution;
- Tailings Thickening and Handling;
- Water Services;
- Air Services; and
- Process Consumables.

The concentrator process flowsheet is depicted in Figure 17.1, and a process description is described in this Section 17.2.

Figure 17.1 – Process Flowsheet



Source: DRA, 2021

17.2.1 Primary and Secondary Crushing

Run-of-mine (ROM) ore is dumped into a feed bin by the front-end loader (FEL). The mining operation should target minimising the quantity of oversize material that cannot pass through the 600 mm vibrating grizzly.

The ore crushing circuit uses skid built crushing equipment and skid-built screen and conveyors as a cost-efficient option for this size of the concentrator plant.

Primary crushing is completed using a mobile jaw crushing skid. A vibrating pan feeder delivers ore to the vibrating grizzly. Oversize material gravity feeds to the primary 110 kW 1050 x 800 mm CJ411 or equivalent jaw crusher, assisted with a rock breaker used for the larger rocks. The jaw crusher product discharges onto the belt conveyor where it is combined with grizzly undersize material. The material is then conveyed to secondary crushing which operates in a closed circuit.

Primary crushed ore combines with screen oversize in a surge bin allowing for choke feed to the cone crusher. Feed to the 220 kW CH440 or equivalent cone crusher is controlled via a vibrating pan feeder which discharges material onto a feed conveyor. Secondary crushed material is conveyed to a double deck vibrating screen for classification.

Over-size material passes over the screen and is conveyed to the secondary crusher surge bin. Screen undersize of 18 mm D_{80} is the final product conveyed to a radial stacker which stockpiles material within the crushing area. The stockpile provides a buffer for at least 8 hours of the concentrator plant operation.

Crushed ore is reclaimed from the stockpile by means of a FEL used to load the crushed ore hopper. Crushed ore is reclaimed by an apron feeder and fed to the grinding circuit using a ball mill feed conveyor. The crushed ore can also be directly fed onto the ball mill feed conveyor through the bypass hopper by means of the FEL. The radial stacker has the capability to feed into the crushed ore hopper directly.

17.2.2 Grinding and Classification

Grinding is performed using a 1.55 MW VFD driven 13' x 19.5' grate discharge ball mill operating in closed-circuit with a cluster of hydrocyclones. Fresh ball mill feed (crushed ore from the stockpile) is combined with cyclone underflow in a mill feed chute. The grinding circuit reduces feed material to a product size P_{80} of 125 µm which is the target grind size for the bulk flotation process. Process water is added to the circuit to maintain solids density in the mill. Mill discharge is pumped to the cyclone cluster for classification. Cyclone underflow material returns to the mill feed chute for further grinding. Cyclone overflow is fed to a linear trash screen prior to bulk flotation conditioning.

17.2.3 Bulk Flotation and Regrind

The objective of the bulk flotation circuit is to obtain maximum recovery of metal sulphides and reject gangue material prior to separation in the subsequent copper, lead , and zinc flotation circuits.

Cyclone overflow (post trash screen) from the grinding and classification circuit is gravity fed to the bulk flotation conditioning tank where flotation reagents are added. The slurry is then conditioned and fed to the bulk rougher flotation cells. The circuit consists of eight (8) tank cells sized to provide sufficient flotation residence time.

The bulk rougher flotation circuit cells operate in a rougher/scavenger mode with the concentrate produced from the first cells fed to the copper flotation circuit. The bulk rougher flotation concentrate is pumped to the bulk regrind circuit to liberate the base metal sulphide minerals present which allows for cleaning and separation. The bulk flotation concentrate is pumped to a scalping cyclone cluster. Cyclone overflow material at a P_{80} of 15 µm combines with bulk regrind mill discharge in an open circuit and is sent to the copper rougher flotation. Cyclone underflow reports to the feed of the bulk regrind mill for size reduction. A vertical stirred mill, using a ceramic grinding media, is used for this duty.

Based on the feed conditions, the number of cells producing either copper or zinc feed can be varied by means of an interchangeable concentrate launder system which can direct the concentrates to either copper or zinc flotation feed. Bulk scavenger flotation tailings are sent to the final tailings thickener and the scavenger concentrate is sent to zinc flotation.

17.2.4 Copper and Lead Flotation

The purpose of the copper and lead flotation circuit is to recover copper, lead and associated silver and gold into their respective saleable copper and lead concentrates.

The bulk regrind product is fed to the copper conditioning tank where it is mixed with additional flotation reagents. The conditioned slurry flows to the copper circuit rougher tank cells bank which provide sufficient residence time within the flotation circuit.

Copper and lead cleaner flotation uses mechanical cells for the concentrate production.

Copper rougher flotation concentrate produced is fed to first cleaner. First cleaner flotation tailings are scavenged in the first copper cleaner scavenger cells. First cleaner concentrate is pumped to the second cleaner circuit.

The first cleaner scavenger flotation tailings are recirculated back to the copper conditioning tank whereas copper rougher flotation tailings and bulk scavenger concentrate and is fed to the zinc flotation circuit. The first cleaner scavenger concentrate is fed to the copper second cleaner.

The second cleaner flotation concentrate is directed to the lead cleaners. The cleaner tailings stream is recirculated back to the first copper cleaner cells.

In the lead flotation cleaner cells, copper minerals are depressed using cyanide and the final copper concentrate is produced through the reverse flotation process. The lead is floated into the lead first cleaner concentrate subject for further recleaning. Lead first cleaner tailings are the final copper concentrate.

The lead cleaning process considers three (3) sequential stages of lead cleaning with the third lead cleaning concentrate being a final product.

The lead cleaner tailings are combined and recirculated back to the first lead cleaner feed to prevent a build-up of any circulating load.

The produced copper and lead concentrates produced are sent to their respective dewatering circuits for thickening and pressure filtration.

17.2.5 Zinc Flotation

Fresh feed to the zinc conditioning tank consists of the copper rougher tailings and may include a bulk scavenger concentrate from the last cells of the bulk circuit, if and when, directed by means of interchangeable froth launder. Conditioned slurry is fed to the zinc rougher, which consists of two tank cells which provide a sufficient residence time for flotation. The tailings stream is sent to the final tailings thickener.

Zinc concentrate cleaning is performed in the mechanical cells.

Zinc rougher concentrate is fed to the first zinc cleaners. The cleaner concentrate is fed to the second cleaner, and the tailings stream is recirculated back to the zinc conditioning tank.

Second cleaner concentrate is fed to the third cleaner flotation bank. The tailings stream is recirculated back to the first cleaner. Third cleaner or final zinc concentrate is pumped to zinc concentrate dewatering and the flotation tailings stream is fed back to the second cleaning stage.

When the grades of zinc are low (processing of high copper ore type) in the mill feed stream, zinc flotation can be bypassed and the zinc feed stream (copper rougher tailings) is directed to final tailings.

17.2.6 Copper Concentrate Dewatering

Copper concentrate is pumped to the 8 m diameter high-rate copper concentrate thickener. Thickener overflow gravitates to the process water settling pond. Thickener underflow is pumped to a holding tank prior to pressure filtration. A dewatered concentrate filter cake is bagged and weighed via an automated bulk bagging system. Filtrate is returned to the copper concentrate thickener.

17.2.7 Lead Concentrate Dewatering

Similar to copper concentrate dewatering, lead concentrate is pumped to the dedicated a 4 m diameter high-rate lead concentrate thickener. Thickener overflow is sent to the process water settling tank. Thickener underflow is pumped to a holding tank prior to pressure filtration. Lead concentrate filter cake is packaged and weighed via an automated bulk bagging system, and the filtrate is returned to the lead concentrate thickener.

17.2.8 Zinc Concentrate Dewatering

Zinc concentrate slurry is fed to the dedicated zinc concentrate thickener. Thickener overflow is recycled to the process water settling pond. Thickener underflow is pumped to a holding tank prior to pressure filtration. Zinc concentrate filter cake is packaged and weighed via an automated bulk bagging system, and the filtrate is returned to the zinc concentrate thickener.

17.2.9 Concentrate Storage and Loadout

Forklifts transport bulk concentrate bags to storage prior to loading to trucks for transportation to destination. Concentrate bags shall be segregated and clearly identified to avoid mixing of concentrates.

17.2.10 Tailings Thickening and Handling

The tailings thickener receives the following feed streams:

- Bulk rougher tailings; and
- Zn rougher tailings.

These streams are combined in the thickener feed well where flocculant is added to facilitate solids settling. Thickener overflow gravitates to the process water settling pond. Thickener underflow is pumped to the final tailings tank and is then pumped to the TSF. The water from the TSF is reclaimed back to the process water settling pond to minimise fresh water make-up usage.

Any solids carryover from the tailings thickener will be captured within the process water settling pond with intermittent purge of the collected solids into the final tailings tank by means of a dedicated pump.

17.2.11 Water Services

There are six (6) different types of water defined in the process plant:

- Raw Water;
- Reclaim water returned from the TSF;
- Gland Seal Water;
- Process Water;
- Potable Water; and
- Fire Water.

Process water for flotation circuits is sourced from the process water settling pond and distributed through the process areas through the process water piping network.

The site has a positive water balance with sufficient volumes produced from mining as well as a atmospheric precipitation. This water is considered a raw water and will be collected to the tailings storage facility and returned to the process water circuit.

Raw (well) water, used to supply the raw water tank, will be used for process water makeup, for gland seal water, for reagents preparation, potable water treatment, and fire protection purposes.

Potable water will be supplied from the raw water tank and will utilise a UV sterilisation unit for potable water requirements. Potable water is used in the safety showers and eyewash stations, and for potable water reticulation to the non-process infrastructure.

Raw water demand for the process is minimised with reticulation of process water to meet most process water demands with the exception of raw water solely used for reagent preparation and filter press wash water.

In addition, a gland water circuit has been incorporated, supplying gland water to all slurry pumps within the process. This is fed from the raw water tank with particle filtration on the suction side of the gland water distribution pumps.

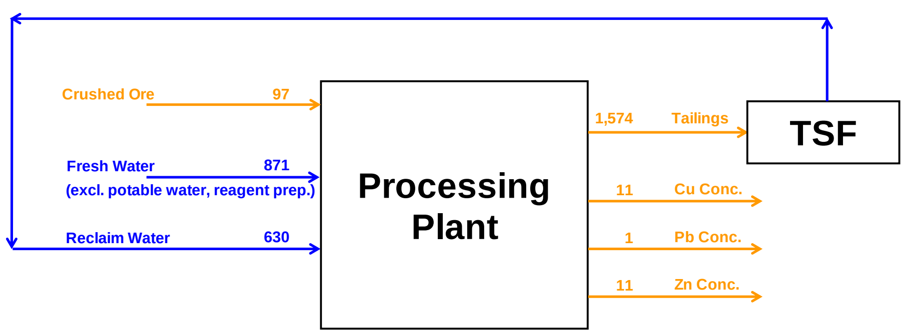
A dedicated fire water volume has been allocated within the raw water tank. The fire water circuit includes a standard fire water skid consisting of an electric fire water pump, and a pressure maintaining jockey pump, plus an emergency diesel powered fire water pump to ensure there is still fire water available in the event of loss of power at the process plant.

The process water settling pond collects the various thickener overflow streams, reclaim water, and fresh makeup water to enclose the water balance. The process water is then distributed within the concentrator plant through the process water pumps and the delivery piping. Settled solids are intermittently pumped to the final tailings tank.

The concentrator water balance summary is shown in Figure 17.2.

As illustrated in the figure below, fresh water usage has been minimised to less then 0.5 m^3 of fresh water per tonne of mill feed ore.

Figure 17.2 – Concentrator Water Balance Summary



General Design Criteria

Item	Units	Value
Operating days per year	days	360
Annual feed (dry)	t/y	666,000
Ore moisture	% w/w	5.0
Copper concentrate filter cake mositure	% w/w	10.0
Lead concentrate filter cake mositure	% w/w	10.0
Zinc concentrate filter cake mositure	% w/w	10.0
Reclaim water recycling rate	%	40.0

871	m^3/day fresh water
26,117	m^3/month fresh water
313,401	m^3/year fresh water
666,000	t/year feed process plant
0.47	ratio (m^3 fresh water per tonne of dry ore)

Fresh Water (incl. potable water, reagent prep.)

Item	Units	Value
Potable water	m^3/d	68
Makeup water	m^3/d	223
Gland seal water	m^3/d	521
Raw water to concentrate de-watering	m^3/d	105
Raw water to flocculant preparation	m^3/d	22
Raw water to reagent preparation	m^3/d	69
		1007

LEGEND
Average water flow (m^3/d)
Water in slurry stream
Water stream

Source: DRA, 2021

17.2.12 Air Services

Flotation air is supplied from the centralised blower station to the consumers within the flotation circuits. Compressed air of the required quality, flow, and pressure will be produced in the central compressor station and delivered to the plant users through a piping network. Individual air receivers are in service for concentrate pressure filters.

17.2.13 Reagent Preparation and Distribution

The function of the reagent preparation facility is to supply the reagents of required quality and concentration to the concentrator plant.

The summary of the facility is as follows:

- Sodium Isopropyl Xanthate (SIPX) collector, is used in solution for all three flotation circuits;
- AEROFLOAT 241 is a selective galena collector used in the for copper and lead flotation circuit to recover the lead. It is delivered as a liquid and distributed directly to flotation;
- Frothers - (MIBC and Polyfroth H57) delivered as liquids, is distributed directly to the flotation circuits.
- Copper sulphate; delivered as prills, flakes, or powder; is used in solution for zinc flotation (activator);
- Sodium Metabisulfite (SMBS), delivered as a powder, is used in solution for copper flotation circuit;
- Lime, delivered as a hydrated lime powder, is used as a lime slurry to maintain the pH level in flotation;
- Soda Ash, delivered as a powder, is used in solution as a modifier;
- Sodium cyanide (NaCN), delivered as pellets (or via solid-to-liquid system), is mixed with high pH water produce the cyanide solution required for lead flotation depression;
- Carboxymethyl cellulose (CMC) dispersant is used in the lead cleaning to facilitate the separation of galena into a concentrate through reducing the viscosity of the flotation froth. CMC is delivered as a powder and used as a solution added to the lead cleaners.

Flocculants, delivered as powders, are used in solution to facilitate the settling in the following thickeners:

- Copper concentrate thickener;
- Lead concentrate thickener;
- Zinc concentrate thickener;

- Final tailings thickener.

17.2.14 Process Consumables

Process consumables such as: mill grinding media (steel and ceramic) and liners, filter cloth, screen panels, and any lubricants; are delivered from the plant warehouses as per the operational demand.

17.3 Process Control, Sampling

Field instruments will provide inputs to a set of programmable logic controllers (PLCs).

Process control cubicles are located in the motor control centres (MCC) and contain the PLC hardware, power supplies, and input/output cards for the instrument monitoring and loop control.

The PLCs perform the control functions by:

- Collecting status information of drives, instruments, and equipment;
- Providing drive control and process interlocking;
- Providing proportional-integral-derivative control for process control loops;

Standard personal computers will be located in the main control room. The computers are networked to the PLCs and operate a supervisory control and data acquisition (SCADA) system that provides an interface to the PLCs for control and monitoring the concentrator. The SCADA system is configured to provide outputs to alarms, control the function of the process equipment, and provide logging and trending facilities to assist in analysis of the concentrator operations.

Uninterrupted power supplies provide operating control stations with 20 minutes of standby power.

The general approach to the concentrator control system isas follows:

- Integrated control via process control system (PCS) for the circuits where equipment requires sequencing and process interlocking;
- Hardwired interlocks for safety of personnel;
- Motor controls for starting and stopping of drives at local control stations, via the PCS or hardwired depending on the drive classification;
- All drives can be stopped from the local control station at all times. Local and remote starting is dependent on the drive class and the control mode;
- Control loops via the PCS except where exceptional circumstances apply;
- Monitoring of all relevant operating conditions on the PCS and recording select information for data logging or trending; and

- Trip and alarm inputs to the PCS will be fail-safe in operation (i.e., the signal reverts to the de-energized state when a fault occurs).

Sampling of the ore, slurries, water, reagents, and chemicals will be performed as per the approved process control scheme continuously or discretely as necessary.

Conveyor belt scales will be used to control the ore tonnages fed to the concentrator, and manual sampling of the ore will be organized for the grade control and metal accounting.

Slurry and water flow within the plant will be controlled by means of the flowmeters appropriate for the duty.

Slurry sampling within the grinding and flotation will be performed by means of the vezin samplers.

Environmental sampling will be organised as per the local regulations and requirements of the relevant international regulatory organisations.

18 PROJECT INFRASTRUCTURE

This section describes the infrastructure required to deliver the mining and mineral concentrate production process described in this FS Report. This section describes the main project elements related to process, followed by supporting infrastructure. Waste facilities and water management are described in Section 20.

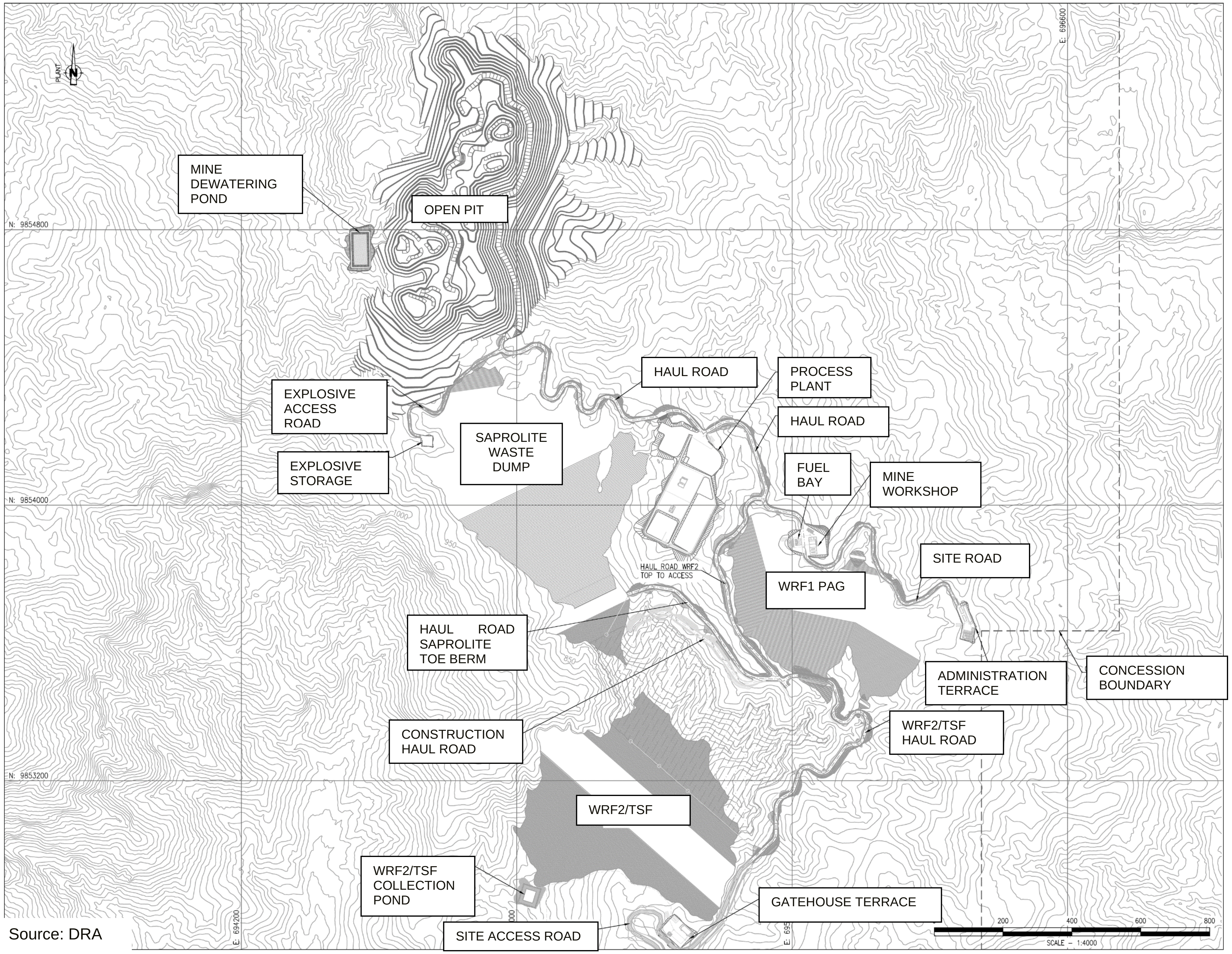
This section commences with a description of the process plant around which other sections of the mine site is located. Plant related services, support building infrastructure and terrace locations are also described. Electrical automation, control and instrumentation are also discussed.

The haul road network interconnects the various site elements with access and material flow optimisation being the main priority. The positioning of the haul roads provide opportunity for the development of the electrical power reticulation network infrastructure that will maximise access through positioning along the haul roads.

Following the on-site infrastructure discussion, site access and off-site infrastructure is discussed.

Figure 18.1 depicts the overall site layout.

Figure 18.1: Overall Site Layout



Source: DRA

18.1 Site Logistics

The Project site will be continuously consuming and producing materials and products that require transport to-and-from site in addition to daily personnel movements. Overarching site logistical flows are summarised in Table 18.1.

Table 18.1 – Major Site Material Flows and Logistics

Material Flow	Description
Personnel	Operational and daily shift personnel will be bussed to site toward limiting individual vehicle traffic to the site.
Mine Waste and Ore Haulage	The main activity continuing throughout the LOM is the haulage of waste and ore material from the Open Pit Mine along established haul roads to the dedicated locations. This will involve a continuous flow mine trucks on the site. It is not envisaged to haul mine ore or waste off site at this.
Fuel Deliveries	To support the mining operations, periodic fuel deliveries to site will be delivered and stored in the fuel storage area. Fuel deliveries will arrive from Guayaquil.
Plant Consumables and reagent deliveries	The Process and water treatment plants will require steady deliveries of consumable reagents to the site daily. Reagent deliveries have not been confirmed yet. It is assumed that shipment from Guayaquil port area will be most likely.
Concentrate production and transport.	The produced concentrates require bagging and shipping from site. The FS assumed that containers will be loaded on site and collected for delivery to ports in the Guayaquil area' which is located 3 hours away by road. The next project phase will explore the option for an off-site consolidation area to allow for less varied truck traffic to the site.

18.2 Roads

The Project requires roads to connect the Project infrastructure elements and form an integrated transport network that can be traversed by all required traffic. The steep terrain of the site inhibits the ability to construct wide roads without incurring significant cost and material movements. DRA has optimized the current road designs by remaining within established road corridors as far as practically possible, and the development of a suitable road design criteria to ensure roads are optimally sized for the required traffic. The site is aided by three existing local roads that connect the El Domo deposit to nearby towns and villages.

DRA is responsible for the haul roads, internal site roads, and temporary construction roads. Adventus retained a local Ecuadorian civil engineering company, *AOC Ingenieria* (AOC), to design and estimate the site access road and public road upgrades.

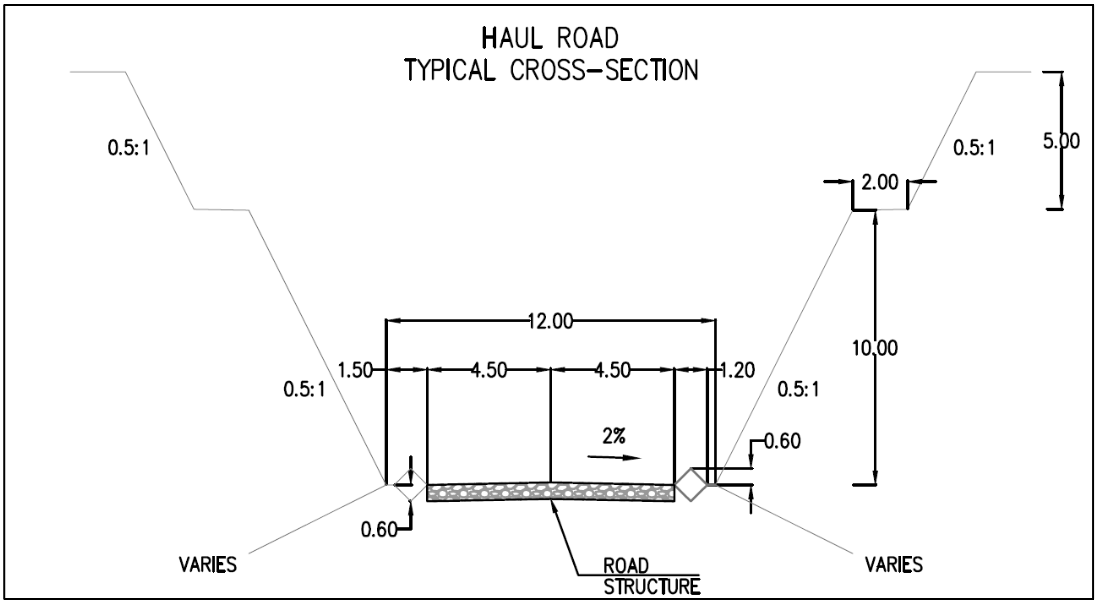
Preliminary designs for all roads are completed to FS level, including major stability works estimation.

18.2.1 Road Design Criteria

A general road design criteria was established to ensure design conformity throughout the Project. The Design criteria is based on DRA internal designs and local geotechnical conditions that determined stability work assessment. Traffic analysis and truck selection (mine material and concentrate haulage) has been considered to optimise the road design.

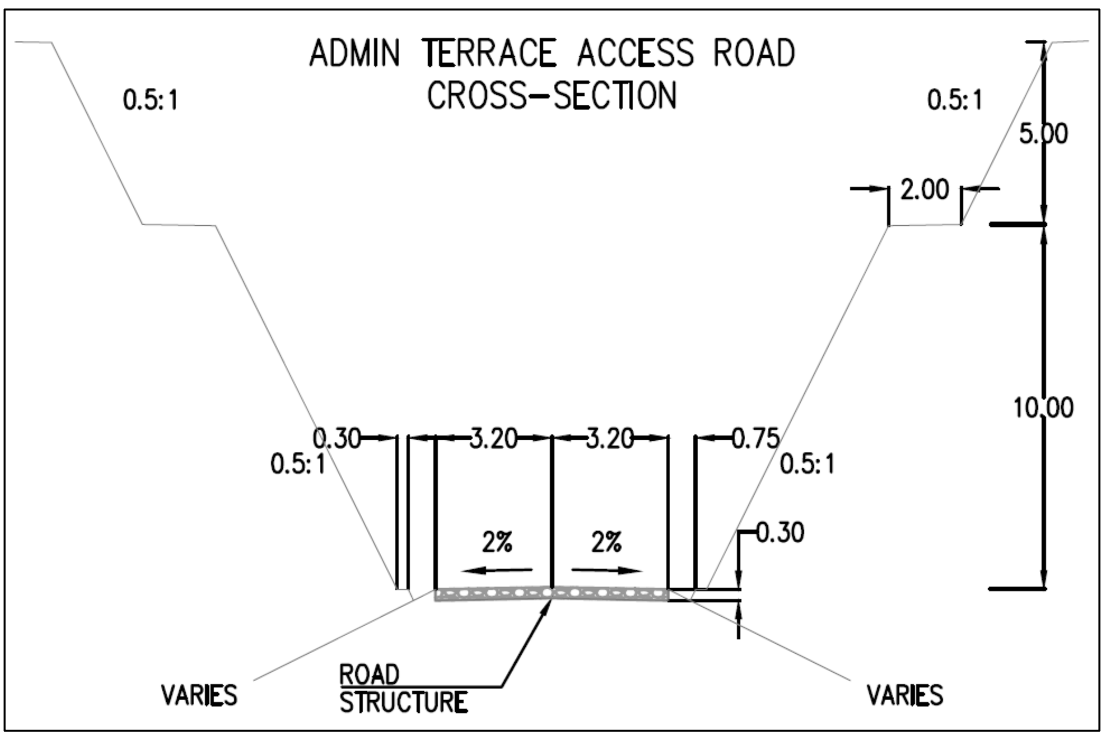
Figure 18.3 and Figure 18.4 present the typical road and stability parameters applied to the Project road designs.

Figure 18.2 – Typical Haul Road Cross Section



Source: DRA or AOC, 2021

Figure 18.3 – Access and Site Road Cross Section



Source: DRA or AOC, 2021

18.2.2 Site Access Roads

AOC evaluated and established roads that lead to site to determine the optimal access route. Multiple route combinations that involved new, upgraded, and combined options were evaluated against an evaluation criteria that included for the operational and construction phase requirements by indicating required traffic frequencies, truck sizes, minimal turning radiuses, and maximum gradients. Further non-technical selection criteria included the social and environmental impact comparison and long-term community upliftment potential.

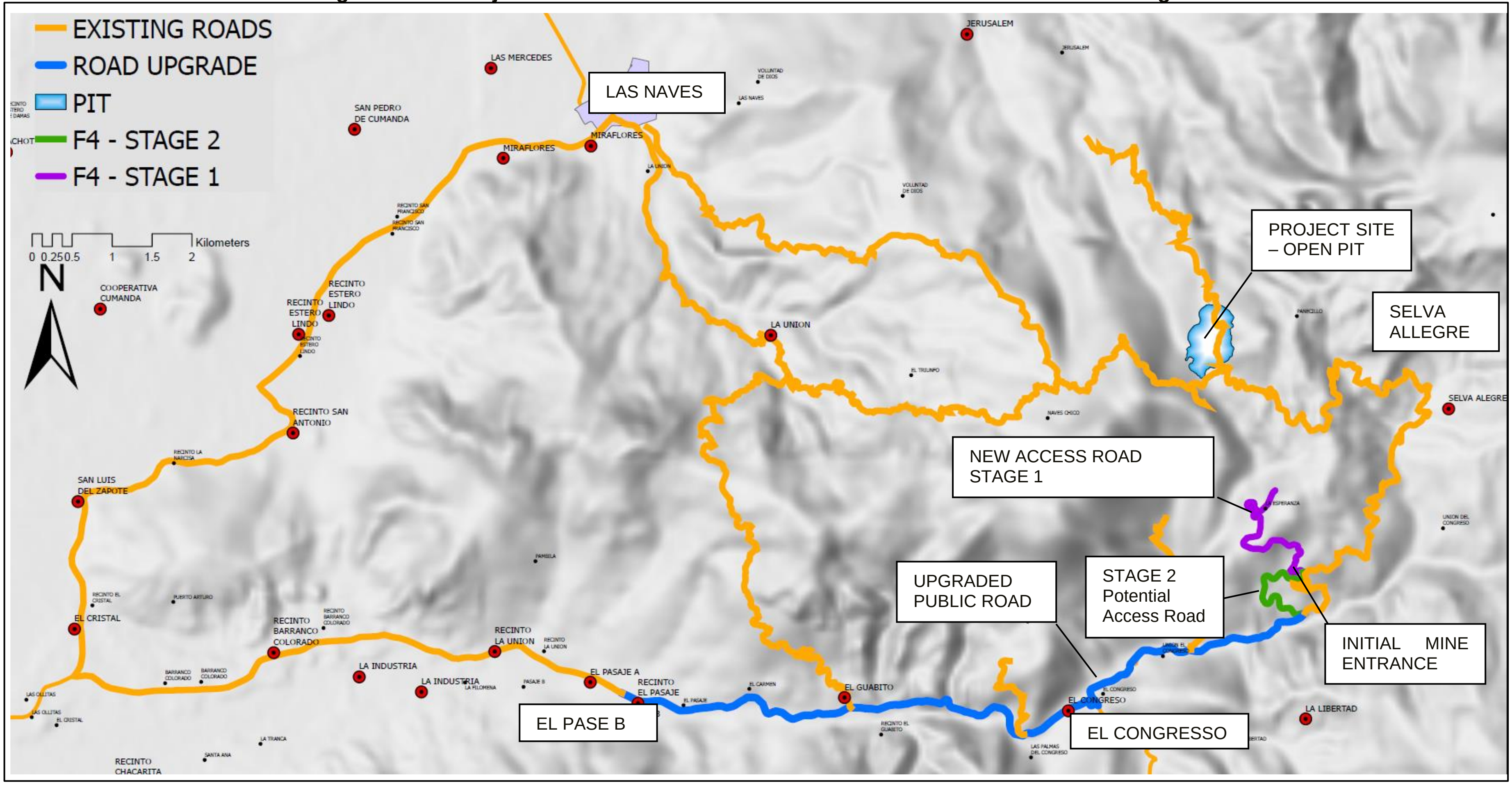
The chosen site access routing is a combination of established road upgrades and new road construction as presented in Figure 18.4. A section of road between El Pasaje B, past El Congreso to the mine access point is planned to be upgraded as part of the early works package which also involves the upgrading of bridges and culverts to ensure heavy mine equipment can be safely delivered to site. Refer to Figure 18.4

Figure 18.5depicts a new site access road section that will be constructed in two (2) Stages: Stage 1 during the early works and construction phase, potentially followed by Stage 2 during early the production phase, if required. The planned upgrades and new roads have been selected with the aim to maximise community benefit and minimise new road impact. The selected routing provides technical advantages in terms of gradients and turning radius over other evaluated options. The access road ends at the Project gatehouse terrace located near the Tailings Storage Facility (TSF)/Waste Rock Facility 2 (WRF2) from where the site network of haul roads and site roads connect the Project elements.

Refer to Figure 18.6.The site road network has been designed to provide fit-for-purpose access to the various site elements. The steep terrain inhibits the ability to establish wide and parallel roads. Mine and operational vehicles will share the road infrastructure. Considering the terrain constraints, mine truck size as defined in Section 16 of the Report, and mine haulage frequency during the operational phase, the risk associated with shared road usage is deemed manageable with sufficient mitigations that can involve training, speed limits, traffic signals, and potential anti-collision technologies.

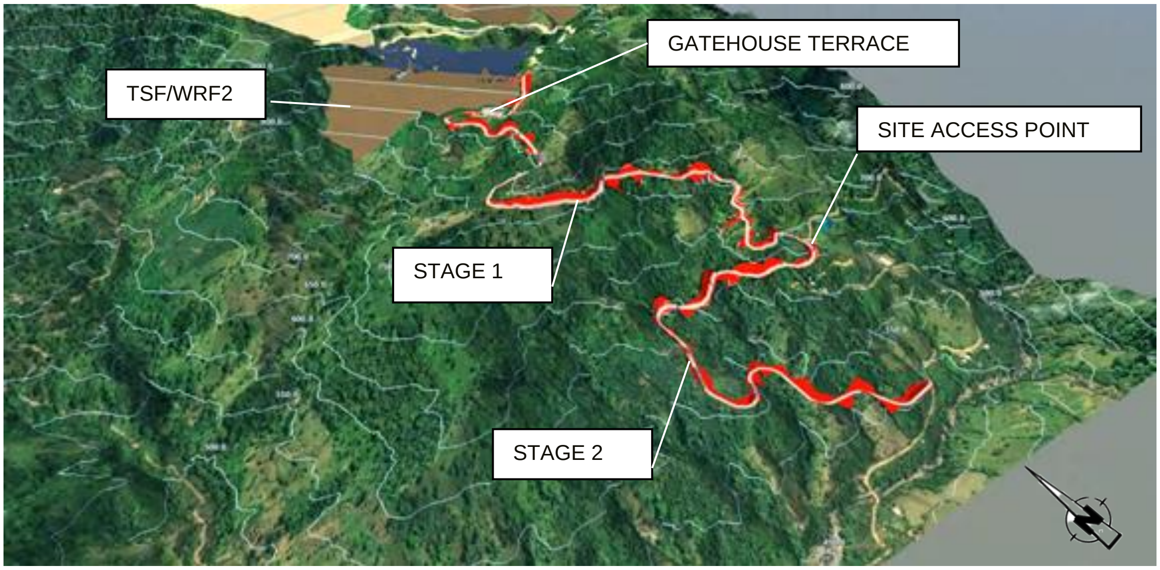
Figure 18.6

Figure 18.4 – Project Site Access Road Network and Selected Site Access Routing



Source: AOC, 2021

Figure 18.5 – Site Access Road Highlighted Topographical Presentation



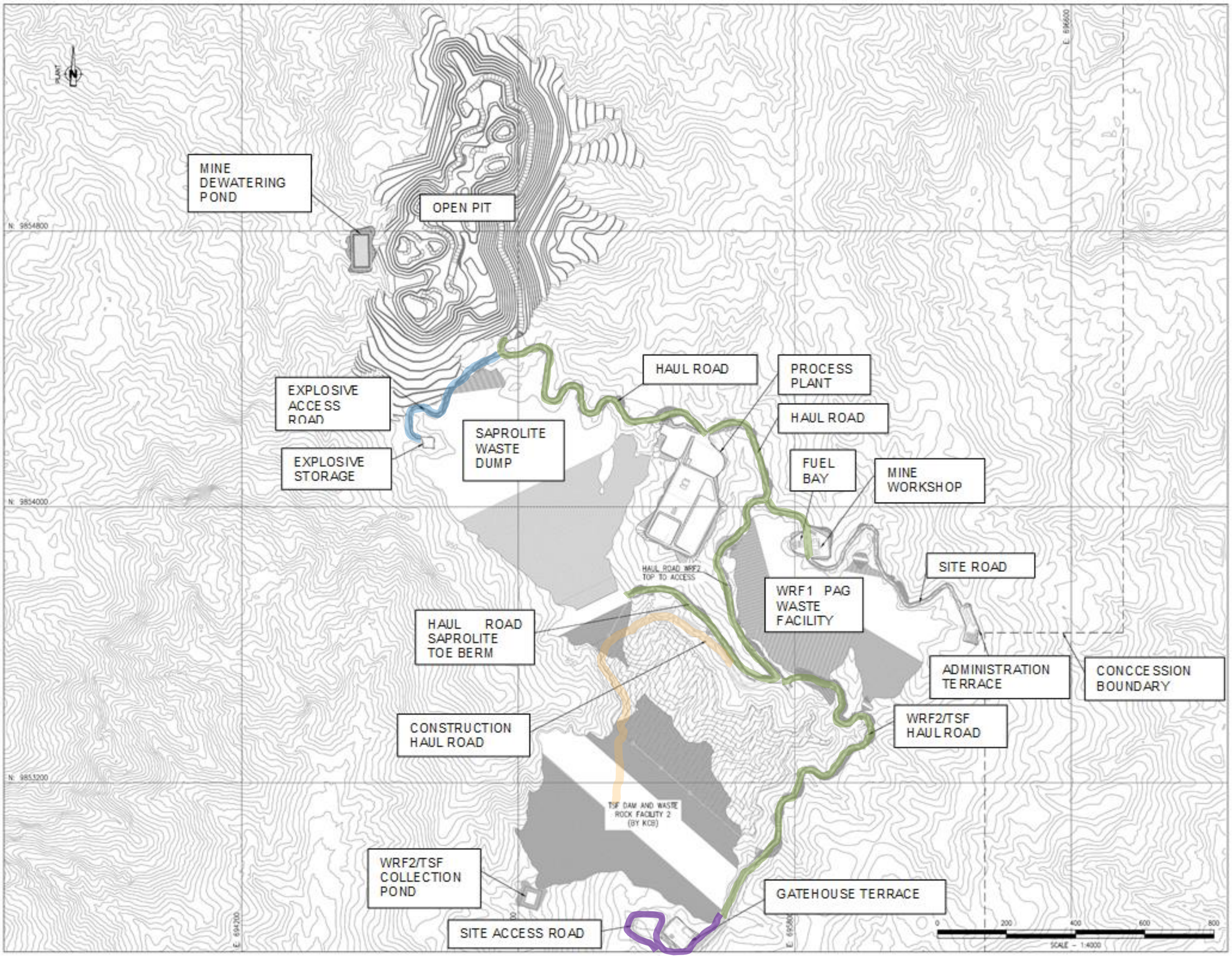
Source: DRA, adapted from AOC,2021 designs

Further detailed road design work is planned during the next Project phase, following detailed route and corridor surveying and a geotechnical drilling and test pit program.

18.2.3 Site Roads

The site road network has been designed to provide fit-for-purpose access to the various site elements. The steep terrain inhibits the ability to establish wide and parallel roads. Mine and operational vehicles will share the road infrastructure. Considering the terrain constraints, mine truck size as defined in Section 16 of the Report, and mine haulage frequency during the operational phase, the risk associated with shared road usage is deemed manageable with sufficient mitigations that can involve training, speed limits, traffic signals, and potential anti-collision technologies.

Figure 18.6 – Site Road Type Indication

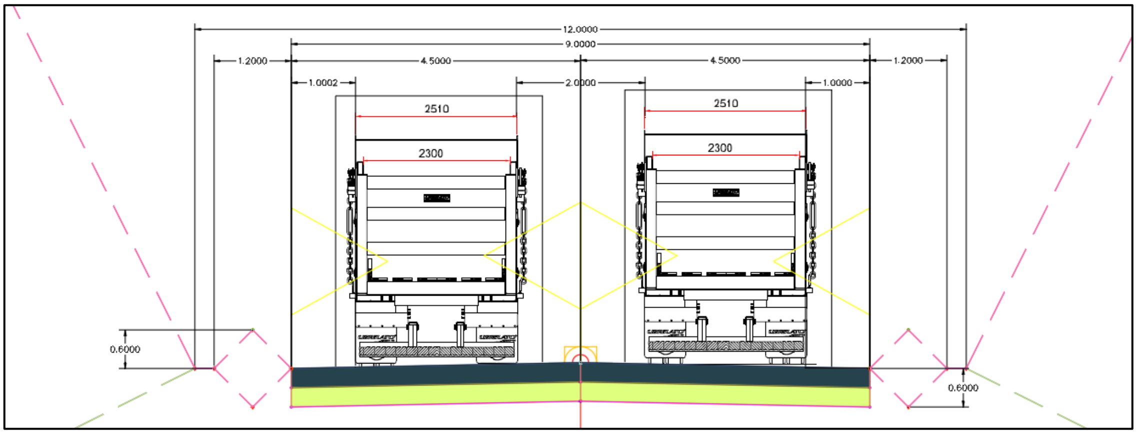


Source: DRA, 2021

18.2.4 Site Haul Roads

A network of gravel haulage roads connects the open pit to the process plant, waste facilities and mine workshops with an approximate length of 4.8 km. Haul roads have been designed with a compacted gravel (assumed 400 mm) layer and width of 12 m including berms on local fill conditions and drainage ditches in local cut conditions. Culverts are included according to drainage requirements. The haul road width has been specifically designed to accommodate the selected mining trucks and optimized to limit road with and construction impact. Presented a typical haul road cross section that includes the selected mining trucks.

Figure 18.7 – Haul Roads - Typical Cross Section



Source: DRA, 2021

18.2.5 Site Roads

The Project will have some internal roads to connect multiple locations with independent access to explosive and Administration areas. All internal roads will be constructed in compacted gravel layers with a width of 8.0 m and will include ditches and culverts according to drainage requirements.

Sufficient allowances have been included in the estimate for all site road construction.

18.2.1 Temporary Haulage Roads

The construction phase of the Project requires access to the El Silencio basin valley floor for initial earthwork and foundation construction associated with the waste rock and tailings facilities. The roads have been designed and costed as temporary but with sufficient integrity to last for the intensive mine pre-production haulage period. The roads will be reclaimed as the TSF embankment level rises and can be accessed from the Southern haul road.

18.3 Terraces and Buildings

The Project infrastructure is housed on a series of terraces spread throughout the Project site. The location of the terraces was highly dependent on limited gradually sloped terrain where terraces could be established without major earthwork impact. Furthermore. locations of the plant and infrastructure facilities were considered with safety proximities from open pit mine blasting. Each terrace and its function will be briefly described in this section.

All terraces have been designed to a suitable degree for FS quantity estimation. Terraces were designed with grading and drainage systems that will allow the runoff water to reach the collection ponds and natural drains in accordance with the site water management requirements.

Buildings located on each terrace are described within each selected terrace.

18.3.1 Camps, Accommodation and Catering

The construction phase of the Project will see the peak requirement for personnel and contractor accommodation. Adventus will be providing a 100-man construction camp, but have requested each contractor to provide suitable temporary accommodation and catering as part of their construction quotations for peak periods of construction, where overflow capacity is required. Most contractors have agreed that off-site accommodation will be preferred due to the site terrain limitations. Contractors aim to establish temporary accommodation rented properties for the duration of their scope. The Owner's teams will be housed in locally available accommodation during the construction phase that Adventus will rent and maintain. The current Curimining canteen facilities may be expanded to accommodate the construction workforce.

As part of early works preparation, Adventus purchased a one hundred (100) man prefabricated accommodation camp with sanitary facilities in Q2, 2021 (Adventus PR, 2021). It is envisaged to establish this camp to serve as limited site accommodation during the early works and construction phase for essential personnel. At the commencement of operations, the camp buildings will either be converted to office and administration buildings, or remain as an accommodation camp for the production period. The camp will require augmentation with a canteen and catering facility.

18.3.2 Process Terrace

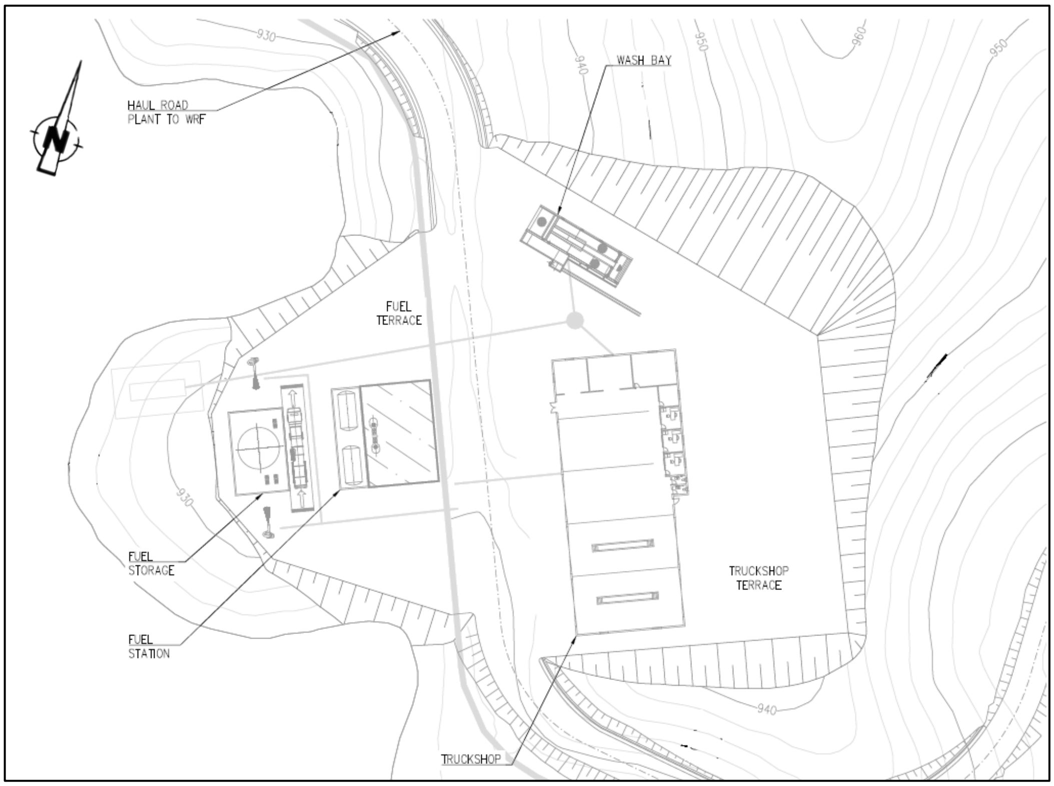
The process plant terrace supports the process equipment required to produce saleable concentrates from the Run-of-Mine (ROM) ore. The process terrace and process infrastructure are discussed in Section 18.4.

18.3.3 Mine Workshop Terrace

The mine workshop terrace is located in a relatively flat terrain area that suits the service and logistics requirement of the mine haul fleet. The terrace is located back from the main waste haulage route to improve traffic safety for personnel and vehicles. The terrace area houses the main mine workshop, a wash bay and a fuel storage and fuel bay area. The mine workshop is a covered steel structure building with an overhead crane and a reinforced floor containing service pits. The wash bay consists of open elevated walkway platforms centred around a containment bund.

The fuel storage area consists of bunded containment and fuelling areas that is open. Fuel tanks will be free-standing within the containment area and be provided by the contracted fuel supplier.

Figure 18.8 – Mine Workshop and Fuel Storage Terrace



Source: DRA, 2021

18.3.4 Administration Terrace

The Project requires an administration terrace to house non-production staff that is still required on site. The administration terrace will support an administration and training building, as well as an camp that will be erected to provide temporary accommodation during the construction phase and potentially converted to offices prior to the start of operations.

Should the camp facilities remain at site after construction is complete, the initial terrace will be expanded as needed to construct an additional administration building to house relevant staff. The Administration building will also serve as the training centre. The current Capex assumes the camp conversion scenario and does not include costs for an additional administration building. Refer to Figure 18.1 for an indication of the Administration Terrace.

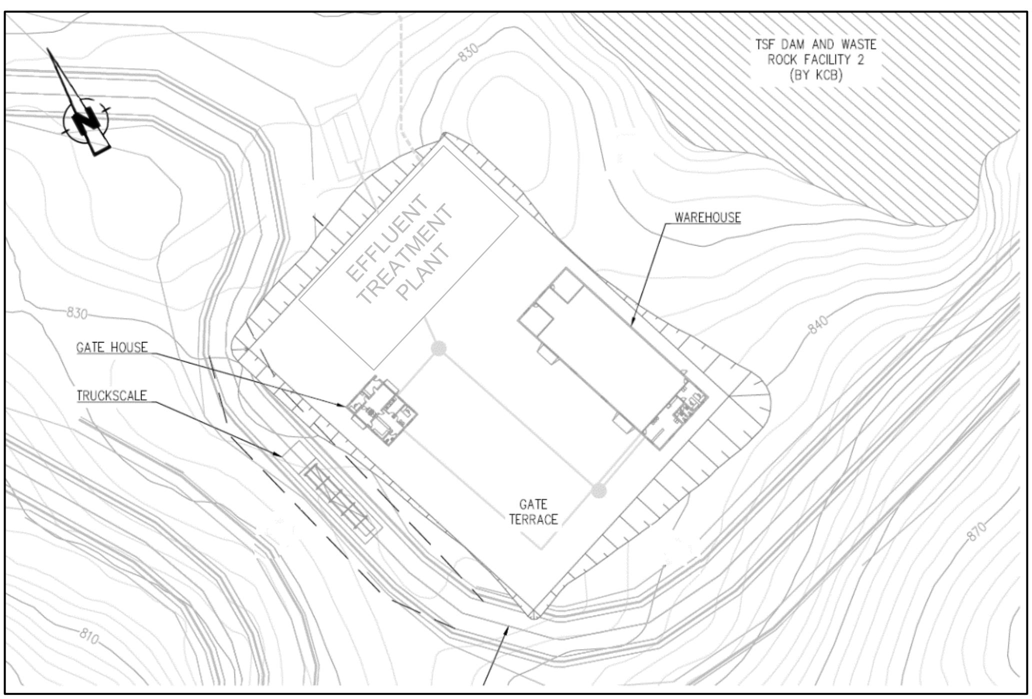
18.3.5 Gatehouse Terrace

The Gatehouse terrace is located near the TSF where the access road reaches the Project site (Figure 18.9). The location is influenced by available terrain in the area that will not require significant earthworks. The Gatehouse terrace will serve as the main access point to the Project site. The terrace will house the gatehouse and security offices, as well as a delivery warehouse and the tailings facility water treatment plant. The concentrate truck weigh bridge will also be positioned as an extension to the access road. The Gatehouse and security office buildings will be constructed with local brick material and sheeted roofing. The warehouse will be a sheeted steel building. The water treatment plant will be housed under a sheeted roof cover, open on the sides.

18.3.6 Explosives Terrace

An explosive storage terrace has been located near the open pit mine site. The location was chosen to keep it remote from inhabited structures and in close proximity to the open pit mine where the explosives will be used. The terrace will house suitably designed explosive storage buildings that will be surrounded by protection berms. The explosives area will be also protected by security fences and gates and will be monitored and controlled by cameras. Refer to Figure 18.1 for an indicative location of the explosive terrace.

Figure 18.9 – Gate House Terrace Layout



Source: DRA, 2021

18.4 Process Plant

The process plant houses the process equipment required to produce concentrate from ROM. The recovery methods are discussed in Section 17 of the Report.

- The process plant layout was determined through an iterative process that involved location selection, topographical terrain and external considerations:
  - Process equipment sizing from the process design and material flow through plant.
  - Maintenance corridors to ensure easy access to equipment for repair and maintenance activities.
  - Compact layout to accommodate steep terrain.
  - Location relative to haul roads and mining pit (remain out of blast radius).
  - Environmental, social and property impacts (access, traffic, noise, dust and light).
  - Optimised earthwork quantities to reduce economic cost of establishing a terrace.
  - Discharge location of tailings and concentrate, and delivery of ROM by haul trucks.

After preparation of an initial site layout to determine sizing (200 m x 150 m), a location trade-off was undertaken to determine the ideal location. The current location was chosen based on position from pit, suitable terrain that will allow terrace establishment, limited line of sight and noise levels to local communities..

18.4.1 Plant Terrace Design

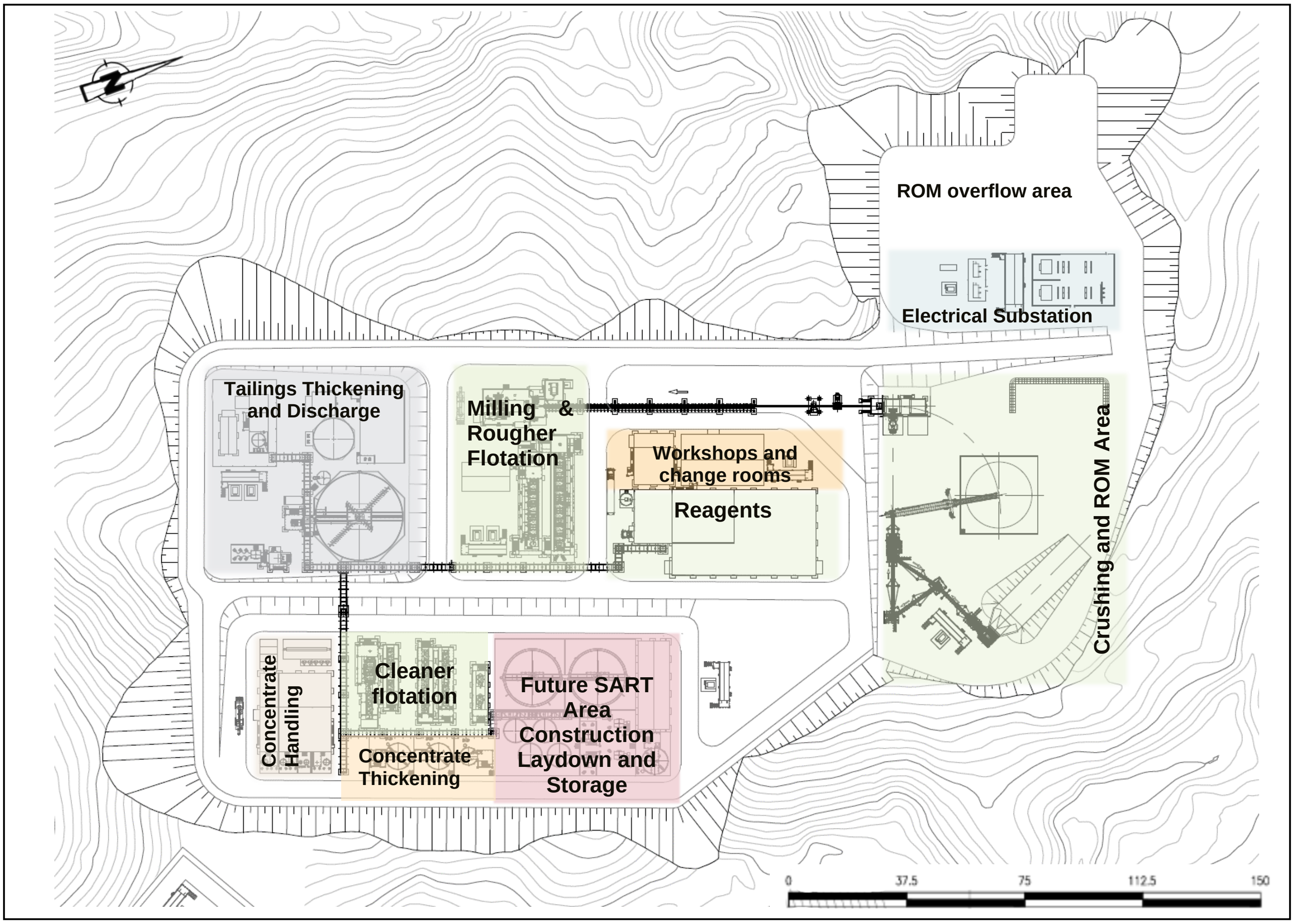
Following the site selection, terrace design was undertaken to optimise earthworks quantities. Further geotechnical drilling was performed indicating that foundations will be sited mostly on compacted saprolite requiring local support earthwork to prevent excessive infrastructure settling. The process plant layout further considered movement of required materials around the platform and have room available for local roads.

The process equipment will be serviced on the platform utilising mobile equipment such as cranes, forklifts, scissor jacks and telescopic baskets. Sufficient room has been left around the structures to accommodate for maintenance

The process terrace will be constructed through the removal of mostly saprolite material to establish the initial terrace. Engineering fills have been avoided where possible due to their complexity in steep terrain. Limited space is available on the terrace for construction and operations. The process terrace also includes for a future expansion area for an optional Sulfidization, Acidification, Recycling and Thickening (SART) circuit. This area will also serve as the laydown and storage areas during construction.

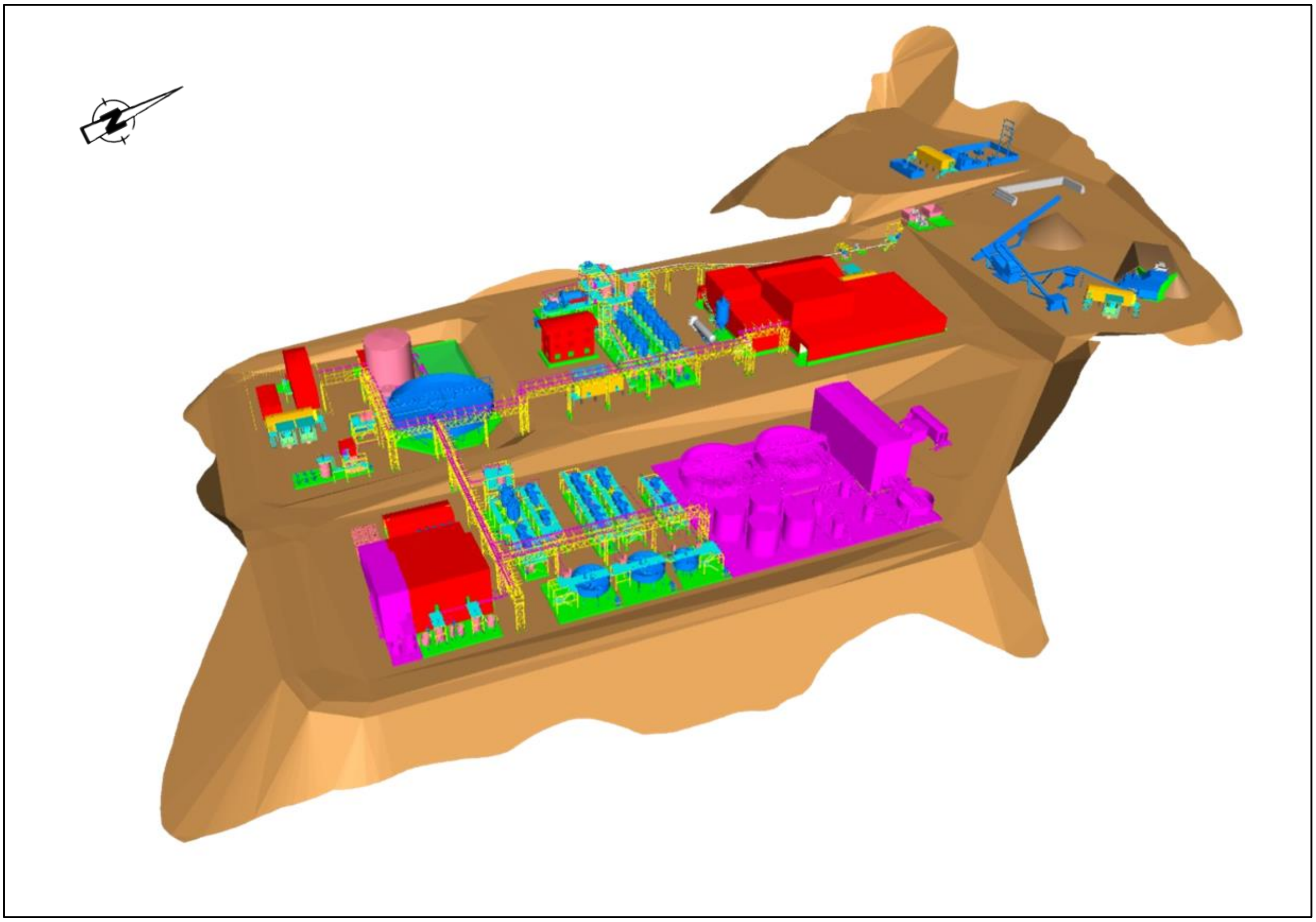
Figure 18.10 presents the process terrace layout along with a descriptive process area overlay. Figure 18.11 presents a 3D layout of the process terrace.

Figure 18.10 – Process Terrace Layout and Process Area Description



Source: DRA, 2021

Figure 18.11 – 3D Representation of the Process Terrace



Source: DRA, 2021

18.4.2 ROM Crushing and Handling

The ROM pad and crushing arrangement allows for the storage of ROM and crushed ore to accommodate crusher downtime. The crushers and mill will be fed by Front-End-Loader machinery. The crushed ore stockpile conveyor is slewing so that it can increase crushed or stockpile size and also feed the mill directly once the crushed ore stockpiles are filled. The ore will remain reactive on the ROM pad where run-off and seepage water will be collected and stored. The saprolite base of the ROM pad will be compacted with gravel following a sealing saprolite layer to avoid seepage water ingress.

18.4.3 Process Plant Operations Support

The process plant is supported by maintenance workshops, reagent dosing, air compression and laboratories located on the process platform. These support services are deemed critical to operate the process plant and is collocated. Sanitary facilities and change rooms are available on the platform for the process personnel.

18.4.4 Services

The process terrace will incorporate buried and rack mounted services:

- A fire water ring main system will be buried on the platform terrace along with potable water and sewage reticulation.
- The fire water reservoir will be combined with the process water reservoir in the process water tank, located in the tailings area, and will have a dedicated fire water reservoir. Fire protection is further discussed in 18.8.2.
- A potable water treatment plant will be present near the process terrace. Potable water will be distributed to the remote terraces via pipeline to ensure local reservoirs remain topped up as a source of fire water.
- A centralised packaged sewage treatment plant has been included at the process plant terrace. The sewage treatment plant will process sewage and grey water run-off from all sanitary facilities. A septic tank system with trucking for remote terraces was evaluated and deemed feasible.
- A domestic waste landfill site is planned as part of Waste Rock Facility 1. A completed area will be prepared to accept landfill. A detailed design will be developed in the next project phase.
- Hazardous materials (including compressed gases, reagents and chemicals, biological waste) storage will be specifically designed for hazard containment. The plant terrace accommodates for reagents and typical temporary hazardous waste containment. Specialised services providers will be contracted to stabilize, transport and dispose of the required hazardous materials.

18.4.1 Plant Structures and Buildings

The process plant structures have been designed to house the process equipment utilising gravity flow where practical. Most process structures are elevated to allow a bunded containment area below to capture any process related spillage or contact rainwater. The structures are of open air design to allow for maintenance access utilising mobile equipment. Local roofing and covering for sensitive process or electrical equipment has been implemented as needed. **Error! Reference source not f**

ound. described the buildings present on the process platform. The majority of the process equipment itself is located in open-air steel structures due to the favourable climatic conditions which allow this approach at El Domo.

Table 18.2 – Plant Buildings Description

Building	Description
Mechanical, Electrical and Instrumentation workshops	Steel structured buildings covered in sheeting. Includes for overhead cranes for maintenance purposes.
Plant stores	Steel structured building covered in sheeting. Includes for overhead cranes spares lifting
Control room, offices and laboratory	3 storey steel building with prefabricated room elements erected within the structure.
Reagents mixing and storage	Sheet covered steel buildings that includes for overhead cranes for lifting reagents and a bunded offloading zone to capture reagent spills.
General covered storage	Steel structures covered with sheeting to protect spares against water damage.
Concentrate storage building	Steel structure covered in sheeting to keep maintain filtered concentrate moisture levels. Includes for a contained bagging and container loading area to capture spills.
Motor Control Centres	Elevated traditional brick buildings with sheeted roofing. Sealed units for environment control.

18.5 Power Supply and Distribution

The main Ecuadorian electrical system is designed as a high-quality electricity service matrix. The distribution system is called the *Sistema Nacional de Distribución* (SND, National Distribution System). The SND is controlled by *CELEC EP Transelectric*, a government institution in charge of power transmission and distribution. Approximately 80% to 90% of the Ecuadorian national power grid is supplied by hydroelectric power sources.

Adventus has progressed preliminary applications with the relevant electric utility authorities to secure the power allocation required for the Project.

18.5.1.1 Site Power Demand

Electrical power demand was determined from the process design and equipment requirements, typical support services and infrastructure power draws, dewatering pumping and treatment plants. Additional capacity was also considered for future expansions that may include underground mine expansions, further infrastructure or plant expansion and increase dewatering or treatment demands. Power demand is presented in Table 18.3.

Table 18.3 – Site Electrical Power Demand

WBS	Area Description	Total Installed Load (kW)	Total Absorbed Power (kW)
	Process Plant		
3100	Crushing & Stockpiling	755	642
3200	Grinding	1,527	1,204
3300	Bulk/Rougher Flotation & Regrind	1,103	867
3400	Cleaner Flotation	411	336
3500	Tailings Thickening & Handling	531	253
3600	Concentrate Handling	188	131
3700	Process Plant Services & Utilities	1,361	765
3800	Reagent Preparation and Grinding Media	120	96
3900	Other Services	-	-
3000	*Total Process Plant*	*5,995*	*4,294*
	Other Areas		
1000	Future Mine Expansion	2,094	1,500
5000	Infrastructure Expansion	2,094	1,500

WBS	Area Description	Total Installed Load (kW)	Total Absorbed Power (kW)
6000	Additional dewatering or water treatment	2,094	1,500
	Total other allowances	*6,282*	*4,500*
	Sub-Total	12,277	8,794
	Contingency (10%)	*1,228*	*879*
	Total Site Demand*	**13,504**	**9,674**
	* Figures may not add due to rounding		

18.5.2 Power Supply

Adventus has opted for grid power supply from the main Ecuadorian public utility. Preliminary negotiations with the public utility have progressed and a suitable tie-in point has been identified. Approximately 80% to 90% of Ecuador's power grid is supplied from hydroelectric supply sources.

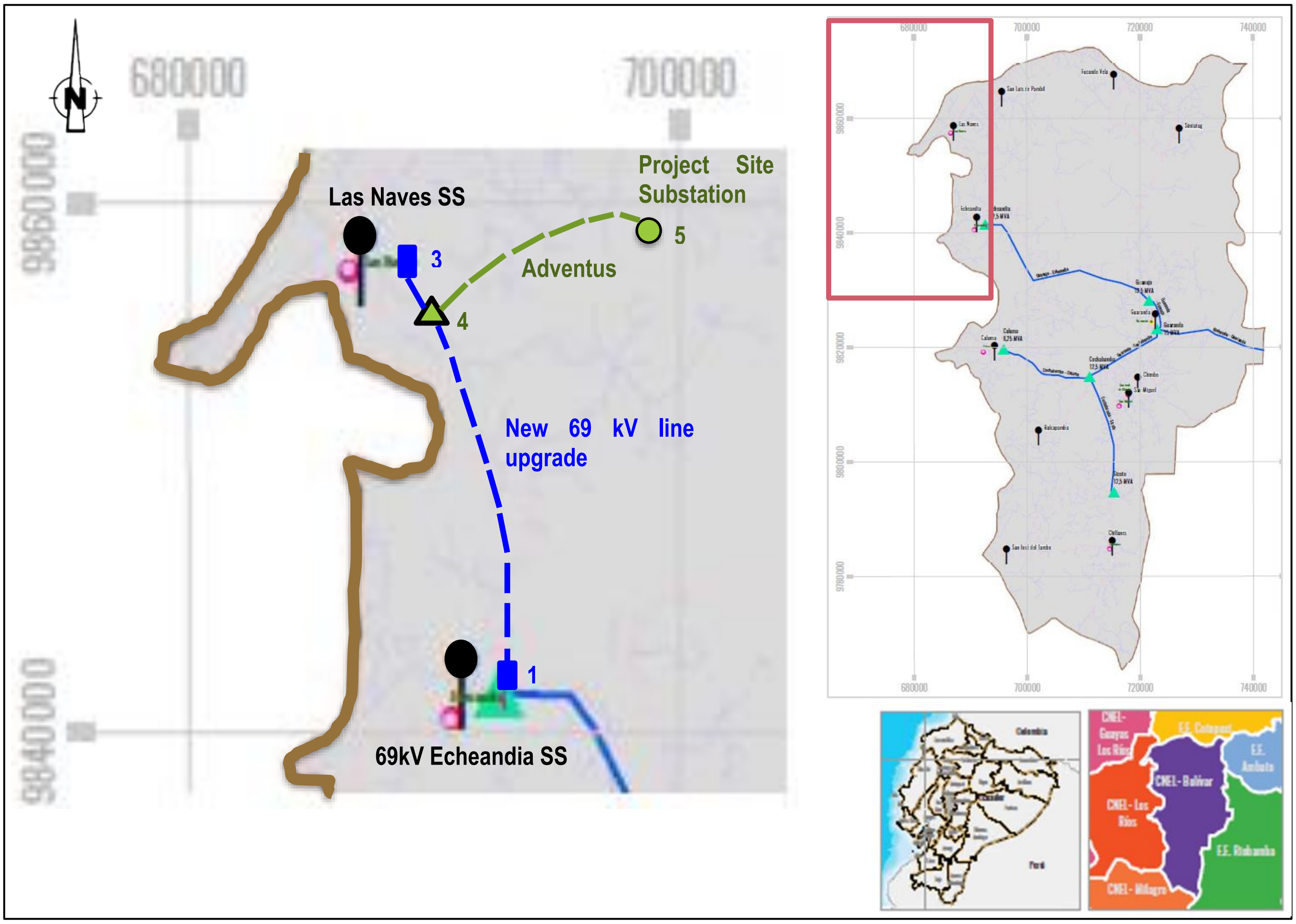
The nearest suitable transmission line connection point is located seven (7) km to the West of the Project site. Adventus will be required to construct a suitable 69 kV overhead line to the Project site. Additional to the new site power line, the local grid power line requires upgrading to manage the increased load demand.

The power grid is shown in Figure 18.12. Currently, the 69 kV power line ends south of the Project site in Echeandia where it is stepped down to 13.8 kV for distribution to the Las Naves Substation (SS). The Echeandia- Las Naves line will require upgrading to 69 kV to allow Adventus to connect to the 69 kV infrastructure. The Las Naves SS will also require upgrading to allow for the step-down from 69 kV even though this will not have any direct benefit for the Project.

Adventus retained Engywork to design and estimate the cost of the 7 km Project site power line for which an allowance has been included in the Capex discussed in Section 21 of the Report. Initial indications were that the Ecuadorian government will invest to upgrade the Echeandia-Las Naves power line to 69 kV. A potential option may exist for Adventus to fund the construction of the required powerline against a tax incentive or other refunding arrangement to be agreed with the relevant Ecuadorian authorities. There is no cost allowance included for such in the FS Capex.

Adventus has retained Engywork to progress with the detailed design of the site Powerline. Construction of the new overhead line infrastructure is estimated at 10 months following approval and receipt of relevant permits for which applications preparations are also under way.

Figure 18.12 – CNEL-Bolivar Province 69 kV Grid Indicating Adventus Line Extensions



Source: DRA, 2021

18.5.2.1 Project Site Reticulation

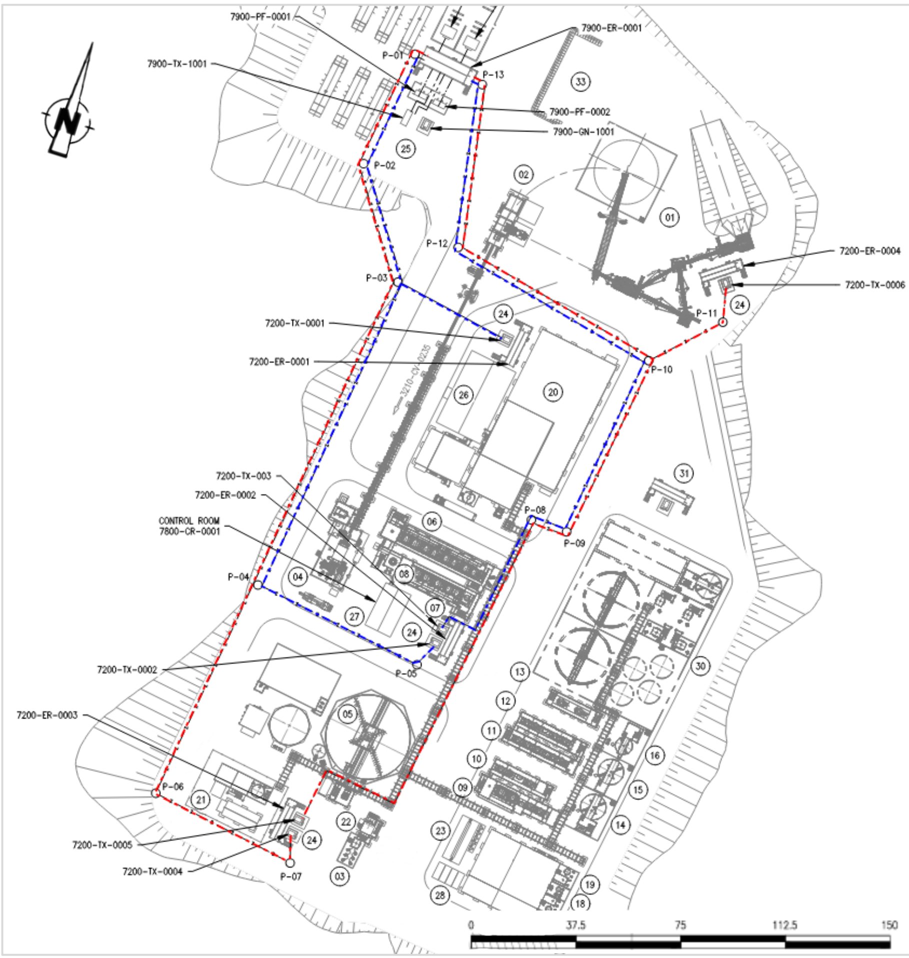
The 69 kV national grid power line will tie-into the main substation located at the Project site from where it will be stepped down to 13.8 kV for main power distribution to local substations located on the process and remote terraces. Remote terraces will be fed via overhead lines established along the haul road routes.

The process plant will have two 13.8 kV cabled ring mains that will be buried, or rack mounted as presented in Figure 18.13. Local transformers will step down to either 4.16 kV for large power consumers or more commonly 480 V for the majority of power consumers.

Suitable protections and redundancies have been allowed for to ensure planned and unplanned maintenance can continue during plant operations.

Backup generators will allow critical functions and processes to remain online in the event of a disruption to grid power.

Figure 18.13 – Process Plant 14.8kV Power Distribution Ring Mains



Source: DRA, 2021

18.5.2.2 Electrical Design Criteria

The Project has established a design criteria suitable for a FS level of design. An extract on site distribution voltage requirements is presented in Table 18.4. The electrical infrastructure for the site has been designed with the stated voltages in mind.

Table 18.4 – Electrical Design Criteria Extract for the Project Site

Description	Voltage levels to be used for distribution
Site Distribution	13.8 kV and 4.16 kV, 60 Hz, 3 phase
Utilization (Medium Voltage)	13.8 kV and 4.16 kV, 60 Hz, 3 phase
Utilization (Low Voltage)	480 V, 60 Hz, 3- phase, 3-wire
Lighting and Power Outlets	480 V, 60 Hz, 3-phase, 3-wire
General control and instrumentation:	120 V, 60 Hz, single phase
Control for 69 kV and 13.8 kV switchgear:	110 or 125 VDC
Control for 4.16 kV and 480 V motor controllers:	120 V, 60 Hz, single phase

18.5.2.3 Motor Control Centres (MCCs)

Motor Control Centres are designed to meet the Project equipment operating objectives with additional spare capacity buckets for later expansions. The Project has opted for locally constructed brick buildings housing MCCs and substations within which electrical switchgear panels will be installed.

MCCs will be installed adjacent to process areas. The panels will be supplied complete with medium voltage switchgear, low voltage MCC's, VFD's, DCS panels and COM panels. The buildings will be fully serviced with fire-alarm panel, lighting, and air conditioning. Fire-alarm panels will report a fire trouble or alarm to the main control room via the communications network. E-Rooms will be supplied with handheld fire extinguishers. E-Rooms will be elevated above grade for bottom-entry cables.

Power transformers are located adjacent to the electrical rooms, separated by concrete block walls Oil filled transformers will be installed on reinforced concrete oil containment pads. Dry type transformers will be installed indoors within ventilated transformer vaults.

18.6 Automation

DRA and Adventus conducted an automation review to determine the level of automation to be included in the plant. The outcome of this review indicated the need for a level of automation suitable to the operational risk, available skills, and remote monitoring requirements. The current process design and control estimation reflects the agreed level of automation for the plant.

The process plant operational control network will be a separate ring-style fibre-optic and ethernet based network to ensure operational redundancy.

18.7 Communications

The process plant and remote terraces will be connected through a fibre optic network for local telephone, internet and data services during the operational phase.

On-site communications for operational personnel will be predominantly by wireless hand-held radios and cell phones using a local wireless cell phone service provider.

A local wireless provider connection will be engaged during the early works phase to provide additional tower infrastructure near the mine site, providing internet and VPN network connectivity required for construction works.

In addition, as soon as on-site Project network hardware and internet connections are commissioned, a local wireless network will establish Wi-Fi internet connection across most of the site terrace platforms.

With the installation of the overhead power grid infrastructure, a fibre optic line internet line is being investigated as part of the infrastructure investment. Adventus is further considering a satellite internet connection. Communication redundancy will be critical to ensure continuous communication.

18.8 Site Security and Emergencies

18.8.1 Site Security

The site access road will serve as the main access control point to the site. The gatehouse terrace will provide access control, induction facilities, and house the security control room and offices.

Other established roads that lead to site will be barricaded with security fencing and locked. To serve local community labour access, an access arrangement at these points will only be granted under site security supervision and availability of mine transport from the point. For safety and security purposes, unescorted, or unsupervised civilian access to site is prohibited. .

The steep terrain proves challenging to install and maintain physical security barriers across the entire site. Public and personnel safety remains of top priority. The main Project terraces will be fenced, along with safety related waste elements such as the TSF and water collection ponds. A physical barrier fence will be erected above the open pit wall to prevent ingress to the pit.

Adventus has opted for an active security management approach with an increased number of security personnel being employed.

18.8.2 Fire Emergency Management

- The process plant will have a dedicated pressurised fire ring piping and hydrant system. The plant fire water reservoir will be combined with the process water reservoir in the process water tank, located in the tailings area, but will have a dedicated fire water reservoir.
- Remote terraces will have a local potable water system with a sufficiently sized water tank that will also serve as fire water. Tanks will be continuously fed via pipelines from the process terrace potable water treatment plant to ensure tanks are kept above minimal levels. A pump pressurised potable water system will have fire hydrants connected to the main lines.
- All Project elements will be fitted with fire extinguishers as a first response.
- The Project site will also have trailer mounted fire water bousers that can be used in an emergency on and off the process terraces.

18.8.3 Medical Emergency Management

The site will have a first-aid station at each terrace supported by a site clinic based at the administration area. The site medical response is focussed upon first response and stabilisation for medical treatment off-site.

Adventus and Curimining have established relationships with local related authorities and facilities with which response plans will be formalised in the next project phase.

18.9 Site Geotechnical Investigations

18.9.1 Terrace Geotechnical Work

Geotechnical investigations on the main process terrace platform were undertaken to establish founding conditions for the process plant equipment. Borehole samples were taken near the planned major process equipment positions as the heavy structures are more prone to settling concerns. Local terrace improvements were recommended and allowed for as part of the estimate. Further optimisation of the terrace is recommended during the next Project phase and includes review of the current terrace elevation. Further lowering of the terrace can improve foundation conditions and possible reduction of large quantities of engineering fill material.

18.9.2 Road Geotechnical Investigation

Geotechnical investigations for the site roads were not undertaken during the FS. The Project site roads follow established road corridors, but also require new road construction. The El Domo deposit is connected to nearby towns by three (3) existing local roads.

The proposed site roads follow established road corridors, but also require new road construction. A site visit was conducted to inspect and understand local geotechnical conditions. Further regionally typical stability works parameters were agreed between AOC and DRA as input for the FS road design and estimate basis. As part of the next Project phase, AOC will undertake detailed corridor surveying and geotechnical drilling to determine site specific stability parameters as part of the detailed design phase.

18.9.3 Geotechnical Investigation for Waste facility design

A site investigation (SI) program was conducted to: (i) characterise the soils and bedrock profile and, (ii) obtain geotechnical information of the foundation soils for the design of the mine waste management facilities and the geotechnical foundation assessment of the process plant.

The foundation conditions and geotechnical assessments are based on:

- The texture, index testing, density, compressibility and shear strength of the soils and the characteristics of the transition/bedrock, and depth of the soil/bedrock contact.
- The in-situ density and penetration resistance of the soils and rock.
- The measured and inferred hydraulic conductivity of the soil horizons and bedrock.
- The measured and inferred location of the groundwater level.
- The SI program started in March 2020, but due to the COVID-19 pandemic, field activities were stopped, resumed in October 2020 and completed in July 2021.

The site investigation program was prepared by Klohn Crippen Berger S.A. (KCB) using preliminary layouts of the mine waste and plant facilities and reviewed/modified as the design and investigation evolved. Investigations were constrained in some zones where access to areas within in the tailings and waste dump were not available at the time of the program. Additional investigations are required to assess areas added to the mine waste areas and process plant facilities at the Project.

The scope of work for the investigation comprised of:

- Visual reconnaissance and surficial geological mapping at the mine waste area for the future tailings and waste rock storage facilities.
- Manual test pits to observe near surface soils and to recover disturbed and undisturbed (block) samples for laboratory testing.
- Diamond drilling to characterise the soil profile, obtain disturbed and undisturbed (Shelby) samples for laboratory testing, conduct in-situ testing, and install shallow and deep stand-pipe piezometers. Three drilling rigs worked in parallel during the program.

- Groundwater level measurements and execution of in-situ permeability tests in new piezometers and selected existing piezometers installed during previous programs.
- Seismic Cone Penetration Test with Pore-pressure measurements (SCPTU) to review the soil behaviour and obtain in-situ geotechnical parameters. One SCPTU rig was used.
- Laboratory testing on recovered samples, including index testing in a local soil testing laboratory in Ecuador and advanced testing in KCB's soil testing laboratory in Vancouver, BC, Canada is underway, and results will be incorporated into the next project phase.
- Soils were logged per the Modified Unified Soil Classification system and KCB's "Field Practices Manual", while the weathering profile was described as per Deere and Patton (1971), and the Scale of weathering grades of rock mass (Fooke's, 1997.)

Table 18.5 presents a summary of the site investigation program and results from the drillholes and test pits.

Table 18.5 – Geotechnical Field Program Summary

Unit name	Total
Shallow Test Holes Investigations	
Test pits	38
Deep investigations	
Drill holes	29
Manual SPT	172
SPT Analyzer	127
Electrical Vane Shear Test	50
Lugeon Permeability Test	32
LeFranc Permeability Test	41
SCPTu	11
Instrumentation	
Deep Piezometers	25
Shallow Piezometers	21
Testing in installed stand-pipe piezometers	
Well development and Slug Tests	46

The residual soils and saprolite are the main foundation materials. These materials are formed by complete weathering of the parent rock and both horizons have been found throughout the site and they vary in colour, texture, moisture, density and shear strength depending on the main composition of the parent rock. It was observed that residual soils and saprolite are thicker (up to 35 m) on high ground and shallow (5 m approx.) near creeks.

18.10 Saprolite Waste Dump Facility

Refer to Section 20.7.1

18.11 Waste Rock Facility 2 and Tailings Storage Facility

Refer to Section 20.7.2

18.12 Waste Rock Facility 1

Refer to Section 20.7.3

18.13 Ore Stockpiles 1

Refer to Section 20.7.4

18.14 Water Management Infrastructure

Refer to Section 20.8.5 and 20.8.8

18.15 Off Site Infrastructure

Curimining SA maintains an established presence in the town of Las Naves where several properties are owned and rented. The current in-town infrastructure is focussed toward supporting exploration and drilling of the Project site.

Curimining currently maintains a core shed and geological offices, additional administration and security offices and a messing facility in town.

Adventus plans to continue with a presence in Las Naves during the project operational phase where non-production related personnel and services can be accommodated off site, alleviating congestion and infrastructure requirements on the site.

The area of Las Naves is connected with road, power, water and sewage infrastructure, allowing for the Project to support and utilise the local infrastructure.

A detailed off site operational phase plan will be developed in the next phase following on completed operational readiness planning during the FS, described in Section 24.

Additional to the in-town presence, Adventus will evaluate the establishment of an off-site consolidation area for deliveries to and from the Project site. The potential benefit involves the standardisation and limitation of non-company vehicles up to the Project site that may increase traffic and vehicle related risks.

19 MARKET STUDIES AND CONTRACTS

19.1 Introduction

The Curipamba process facility will produce three (3) polymetallic concentrates, namely copper, zinc and lead concentrates. Each concentrate contains a mixture of Copper, Zinc, Lead, Silver and Gold metals in varying saleable proportions. Refer to Section 17 for a detailed breakdown of the concentrate recoveries and metal deportments. Based on the initial flow sheet and concentrate grades, a preliminary market study and further preliminary refiner negotiations have progressed to provide sales parameters for the FS and input to the financial model. The sales terms are described for each concentrate. Concentrates will be shipped on a regular schedule to refiners via the deepwater port at Posorja (southwest of Guayaquil) that is co-owned by DP World and Consorico Nobis. Transportation costs were also sourced and updated during the FS for inclusion in the financial model and further discussed in Section 18 of the FS.

19.2 Market Studies

Adventus retained Exen Consulting Service (Exen) to prepare an initial independent market study (Exen, 2020) for the planned sales of concentrates. Adventus requested the confirmation if concentrates were marketable and the expected preliminary terms from off takers. Exen conducted a preliminary investigation as to best suited markets and potential smelter purchasing terms for each concentrate.

Contained concentrate metals are well known and traded freely on metals exchanges globally. Multiple smelters in Asian and European markets have been identified that can accept the three (3) concentrates for refinement (Exen, 2020).

The initially sourced terms were used early on in the FS to evaluate and optimize the process plant flowsheet and concentrate metal deportment to maximise Net Smelter Returns (NSR) for the ore body.

Following the initial market study (Exen, 2020), further process design and recovery test work results, Adventus conducted further term negotiations with specific off-takers to refine the market study terms. Preliminary terms were received from specific off-takers (Adventus ST, 2021) and evaluated for optimal project revenue. Although DRA did receive supporting documentation and verified that the terms are accurately reflected, the terms were not independently validated.

The terms and relevant reference for each concentrate is depicted in Tables 19.1 and 19.2.

19.2.1 Terms and Delivery

Delivery terms have been assumed as Cost, Insurance, Freight (CIF) to likely Asian discharge ports for transportation of all concentrates. Under CIF terms, the seller is responsible for all costs up to point of discharge, including inland transport to port, warehousing, container stuffing or stevedoring for bulk loads, ocean freight and insurance. For the FS, container stuffing has been assumed and accounted for in the transport costs. Preliminary transportation, shipping and port charges were obtained from local companies and off-takers (Adventus ST, 2021). Although DRA did receive supporting documentation and verified that the transport costs are accurately reflected, the transport costs were not independently validated.

19.2.2 Concentrate Payables

Table 19.1 – Concentrate Payables

Concentrate Payable %	Payable	Minimum Deductions
Cu Concentrate Payable (Adventus ST, 2021)		
Payable Au	96.0% ≥ 20 g/dmt 95.0% < 20 g/dmt	
Payable Ag	90.0% ≥ 30 g/dmt 0% < 30 g/dmt	
Payable Cu	96.5% ≥ 20% dmt 96.5% < 20% dmt	1.0% when grade above 20% dmt 1.2% when grade below 20% dmt
Pb Concentrate Payable (Exen, 2020)		
Payable Au	95.0%	1.0 g/dmt deduction
Payable Ag	95.0%	50 g/dmt deduction
Payable Pb	95.0%	3 unit deduction
Payable Cu	100%	2 unit deduction
Zn Concentrate Payable (Adventus ST, 2021)		
Payable Au	75.0% ≥ 1 g/dmt 0% < 1 g/dmt	1 g/dmt
Payable Ag	75.0% ≥ 3 oz/dmt 0% < 3 oz/dmt	3 oz/dmt
Payable Zn	85%	8 unit minimum deduction

19.2.3 Expenses and Penalties

Transport costs (Adventus ST, 2021), refining and penalty costs for copper and zinc concentrates (Adventus ST, 2021) and lead concentrate (Exen, 2020) are presented in Table 19.2.

Table 19.2 – Sales Terms for Financial Analysis

Category	Terms
Transport (Inland, Shipping, Port Charges, Insurance) (Adventus ST, 2021)	
Cu Concentrate	$US 71.74 / wmt conc
Pb Concentrate	
Zn Concentrate	
Treatment Charges	
Cu Concentrate	$US 80.00 / dmt conc
Pb Concentrate	$US 180.00 / dmt conc
Zn Concentrate	$US 220.00 / dmt conc
Refining cost	
Au	$US 5.00 /oz in copper concentrate $US 15.00 / oz in lead concentrate
Ag	$US 0.50 /oz in copper concentrate $US 1.50 / oz in lead concentrate
Cu	$US 0.08 /lb in copper concentrate $US 0.41 /lb in lead concentrate
Pb	$US 0.00
Cu Concentrate (Adventus ST, 2021)	Pb + Zn $3.00/dmt for each 1.0% Pb+Zn > 4.0% As $3.00/dmt for each 0.10% As ≥ 0.20% Sb $5.00/dmt for each 0.10% Sb ≥ 0.05% Hg $2.00/dmt for each 10 ppm Hg ≥ 10 ppm Cd $3.00/dmt for each 0.01% Cd ≥ 0.03%
Pb Concentrate (Exen, 2021)	$2.00/dmt for each 1.0% Zn > 5.0% As $1.50/dmt for each 0.10% As > 0.50% Sb $1.50/dmt for each 0.10% Sb > 0.50% Bi $1.50/dmt for each 0.01% Bi > 0.10% Hg $2,00/dmt for each 10 ppm Hg > 50 ppm
Zn Concentrate (Adventus ST, 2021)	Cd $1.50/dmt for each 0.1% Cd ≥ 0.3%

The indicated terms were utilised to prepare the Net Smelter Return (NSR) as input to the financial model.

19.3 Commodity Pricing

Adventus has adopted the following price projections for the FS financial model base case as depicted in Table 19.3 based on internal long-term projections.

Table 19.3 – Base Case Metal Pricing

Element	Unit	Financial Model
Au	$US/oz	1,700.00
Ag	$US/oz	23.00
Zn	$US/lb	3.50
Cu	$US/lb	0.95
Pb	$US/lb	1.20

19.4 Contracts

There are no material contracts or agreements in place as of the effective date of this Report. Adventus has not hedged, nor committed any of its production pursuant to an off-take agreement.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Regulatory Overview

20.1.1 LEGAL FRAMEWORK

Mining activities in Ecuador are mainly regulated by *the Ministerio de Energía y Recursos Naturales No Renovables* [Ministry of Energy and Non-Renewable Natural Resources] (MERNNR), the *Agencía de Regulación y Control Minero* [Mining Control and Regulatory Agency] (ARCOM), and the *Ministerio del Ambiente, Agua y Transición Ecológica* [Ministry of Environment, Water and Ecological Transition] (MAATE). The principal environmental laws that apply to the mining industry are the Constitution, *the Ley de Minería* [Mining Law], the *Reglamento Ambiental para Actividades Mineras*, or RAAM [Environmental Regulation for Mining Activities], the *Texto Unificado de la Legislación Secundaria del Ministerio del Ambiente*, or TULSMA [Unified Text of Secondary Environmental Legislation], the *Ley Orgánica de Recursos Hídricos, Usos y Aprovechamiento Del Agua* [Water Use and Exploitation Act, and the *Código Orgánico del Ambiente*, or COA [Environmental Code], which entered into force in April 2018 and encompasses all the environmental legislation in one single body of law.

The MAATE issues an environmental licence and water authorization for mining following approval of an Environmental and Social Impact Assessment (ESIA) and Environmental Management Plan (EMP). The Ministry of Mining is responsible for mine planning in Ecuador, including the negotiation of contracts for the exploitation of minerals. ARCOM is responsible for supervising mining activities. Other permits required for mining activities include those for explosives use from the *Ministerio de Defensa Nacional* [Ministry of National Defense], special labour shifts from the *Ministerio del Trabajo* [Ministry of Labour], fire department, and construction permits (from ARCOM and the municipalities). A summary of the major permits required for construction and operation of the project is provided in Table 20.1.

Table 20.1 – Summary of Major Permits for Construction and Operation

Permit Name	Permitting Authority
Certificate of Validity of Mining Rights	Mining Regulation and Control Agency
Federal Environmental License (EA) - Exploitation	Ministry of Environment, Water and Ecological Transition (Quito)
Federal Environmental License (EA) - Processing	Ministry of Environment, Water and Ecological Transition (Quito)
Water License - Industrial Use	Ministry of Environment, Water and Ecological Transition (Quito)
Authorization to Install a Process Plant	Ministry of Energy and Non-Renewable Natural Resources
Authorization to Build a Tailings Storage Facility	Ministry of Energy and Non-Renewable Natural Resources
Explosive Transportation, Storage, and Use Permits	Armed Forces Department of Arms Control
Quarry / Borrow Permits	Local Government
Fuel Purchase Permit	Regional Hydrocarbon Regulation and Control Administration
Approval of Camp Specifications	Mining Regulation and Control Agency
Mining Operations Contract	Ministry of Energy and Non-Renewable Natural Resources
Prior to Operations	
Registration as Hazardous Waste Generator	Ministry of Environment, Water and Ecological Transition (Quito)
Possession and Use of Controlled Substances	Technical Secretariat of the National System of Professional Qualifications
Registration of Hazardous Chemical Storage and Use	Ministry of Environment, Water and Ecological Transition (Quito)

20.1.2 Requirements For Environmental and Social Impact Assessment

The following sections from Chapter II and Chapter III of the *Ley De Gestion Ambiental, Codificación* [Law of Environmental Management, Codification] Coding 19, Official Register Supplement 418 of September 10, 2004, are relevant to baseline studies and impact assessments:

- Art. 21: Requires that environmental management systems include baseline studies, environmental impact evaluation, risk assessment, management plans, risk management plans, monitoring; contingency and mitigation plans, environmental audits, and abandonment plans.
- Art. 22: Environmental management systems may be evaluated at any time, at the request of the Ministry of the branch or of the people affected. The evaluation of compliance with approved environmental management plans will be carried out through the environmental audit, practiced by consultants previously qualified by the Ministry of the branch, in order to establish the corrections that must be made.
- Art. 23: The environmental impact assessment will include:
  a) The estimation of the effects caused to the human population, biodiversity, soil, air, water, the landscape and the structure and function of the ecosystems present in the affected area.
  b) Public tranquillity conditions, such as: noise, vibrations, and any other induced environmental changes.
  c) Assessment of historical, scenic, and cultural heritage impacts.
- Art. 28: Outlines the right of people to participate in environmental management, through the mechanisms established for this purpose by the Regulation, including consultations, public hearings, and other initiatives.

In order to comply with the ESIA submission necessary to obtain mining permits, Adventus has prepared an ESIA with the following components:

- A description of the Legal Framework;
- Detailed description of the project, including an alternatives analysis;
- Determination of the Area of Influence for all planned project infrastructure on the environmental and social landscape;
- Characterization of the physical and biological baseline condition;
- Characterization of the socio-economic baseline condition;
- Characterisation of the archaeological baseline condition;
- Identification, prediction, and evaluation of environmental impacts;

- Risk assessment;
- Forest inventory and economic evaluation;
- Citizen Participation Process;

20.1.3 Environmental Management Plan Existing Approvals

Curimining holds valid approvals to conduct advanced exploration granted from the Ministry of Environment (now MAATE). Resolution 509, issued on May 3, 2011, provides authorization for advanced exploration on the Curipamba Sur 1, Las Naves, Las Naves 3, and Jordan 2 blocks. The authorized advanced exploration activities include:

- Geophysical sampling (e.g. electromagnetic, induced polarisation, etc.);
- Soil sampling;
- Trenching;
- Exploration drilling (e.g. diamond drilling).

20.2 Biophysical Baseline Environment

20.2.1 Soils and Land Capability

A soil survey and edaphology analysis was conducted from 96 samples collected at 39 locations across the Project area in 2021 (Cardno, 2021). The dominant soil type was classified as Andisol, have high organic matter content, resulting in light soils with very high water-holding capacity. The soils are derived from coarse-grained volcanic sandstones, breccias, tuffs, hyaloclastites, volcanic siltstones, sub-porphyritic basalts, lavas, andesites, and dacites.

Morphological conditions are characterized by mountainous relief, with slopes ranging between 25 to 40%. Chemical analysis identified higher concentrations of sulphur, barium, copper, cobalt, lead, vanadium, selenium, and zinc. These elements are related to the nature of the parent material and are associated with volcanogenic massive sulphides.

The majority of the Project area is characterized by agricultural lands, followed by shrubland and then secondary forest. Small patches of primary forest remain, largely in steep-sided gullies and other inaccessible areas.

20.2.2 Surface Water Quality

The Project area drains from dendritic catchments to the Runayacu River, which becomes the Oncebí River, in turn discharging to the Zapotal River.

Curimining has been collecting discharge measurements on a number of creeks and sites in the project area on a monthly basis since 2011. Higher flows correspond with rainy season between December and May, where maxima of over 25 m^3/s were observed in the Runayacu River. Low flows, occurring between July and November, are typically less than 1 m^3/s.

Monthly sampling of surface water quality was initiated in 2019 and are ongoing at 27 locations. Stations were selected to provide upstream and downstream representation to compare between baseline and operational data, and to provide control sites. Samples were collected by trained technicians and provided to a certified laboratory for analysis of the following parameters:

- Physical attributes;
- Nutrients;
- Hydrocarbons;
- Metals.

Across all sampling events, most parameters were within Ecuadorian limits for the protection of water resources for the baseline condition. Exceptions to this included incidents of elevated aluminium, and isolated occurrences of copper, iron, and zinc. In general, the baseline surface water quality is slightly alkaline, with low metals concentrations, but high organics content.

20.2.3 Flora

Field studies to characterize the flora of the project area were conducted in 2019, 2020, and 2021. The vegetative cover is predominantly secondary forest, natural pastures, and moderately disturbed forests on the banks of rivers, streams, and estuaries, along with some small, isolated remnants of natural vegetation in steep and high elevation areas.

The dominant flora of the Project area is comprised of grass species within the family Poaceae, found throughout the pasture and disturbed polygons. Within forest remnants, including secondary forest, a total of 12 vegetation species were recorded that are listed as either Endangered or Vulnerable according to the International Union for the Conservation of Nature (IUCN). These sensitive species exhibited very low abundance, often detected by a single representative, and include the following species:

- *Blakea involvens;*
- *Browneopsis disepala;*
- *Brugmansia versicolor;*
- *Drymonia ecuadorensis;*
- *Matisia coloradorum;*

- *Miconia explicita;*
- *Paradrymonia hypocyrta;*
- *Philodendron ventricosum;*
- *Siparuna palenquensis;*
- *Siparuna eggersii;*
- *Unonopsis magnifolia;*
- *Wettinia aequatorialis.*

20.2.4 FAUNA

Comprehensive field studies were conducted in 2019, 2020, and 2021 for birds, mammals, reptiles, amphibians, and insects. Studies included directed netting, trapping, as well as observational studies throughout each of the dominant vegetation and habitat types of the Project area.

20.2.4.1 Avifauna

Ecuador is known for its bird life and a total of 137 species were recorded in the Project area. The greatest diversity of birds was found in forest remnant areas, wherein the dominant species were the Orange-bellied Euphonia (*Euphonia xanthogaster*) a small, finch-like bird, the Golden Tanager (*Tangara arthus*) the Crimson-rumped Toucanet (*Aulacorhynchus haematopygus*) and the Scarlet-rumped cacique (*Cacicus uropygialis*).

Of the species recorded, two (2) are considered either Vulnerable or Threatened by the IUCN, including the Long-wattled Umbrella Bird (*Cephalopterus penduliger*) and the Ruddy Pigeon (*Patagioenas subvinacea*).

20.2.4.2 Mammals

A total of 40 mammal species were recorded within the Project area. The majority of these were bats, followed by rodents. Of the mammal species identified, the Mantled Howler Monkey (*Alouatta palliata*) and the Greater Long-tailed Bat (*Choeroniscus periosus*) were the only two that are considered either Endangered or Vulnerable. The majority of the mammals found in the study area are associated with disturbed or agricultural areas.

20.2.4.3 Reptiles and Amphibians

A total of 14 species of reptiles and 19 species of amphibians were identified in the Project area. Of these, 2 reptile and 10 amphibian species are tracked as either Vulnerable or Endangered, including:

- *Lepidoblepharis conolepis;*

- *Alopoglossus festae;*
- *Hyloxalus infraguttatus;*
- *Epipedobtes tricolor;*
- *Pristimantis crenunguis;*
- *Pristimantis parvillus;*
- *Pristimantis labiosus;*
- *Pristimantis w-nigrum;*
- *Pristimantis crucifer;*
- *Pristimantis muricatus;*
- *Pristimantis nyctophylax;*
- *Pristimantis sobetes.*

Herpetiles are sensitive organisms that are closely associated with their habitat. The majority of the listed species identified are associated with forest remnants.

20.2.5 Aquatic Ecosystems

Surveys for fish and aquatic invertebrates were conducted in the drainages throughout the project area in 2019, 2020, and 2021. Twenty-seven (27) fish species were recorded, dominated by Astroblepus, a species of small catfish endemic to the Andean region. They typically inhabit clean, well-oxygenated water, and as such it is useful to compare distribution and abundance over time as a bioindicator.

Similarly, sampling of aquatic invertebrates was undertaken to establish a baseline for comparison during future monitoring efforts.

20.2.6 Air Quality

Baseline measurements for air quality included the following parameters at 15 sites:

- Carbon monoxide;
- Nitrogen monoxide;
- Nitrogen dioxide;
- Ozone;
- Sulphur dioxide;
- PM_{10} (Particulate Matter less than 10 μm in diameter);

- $PM_{2.5}$.(Particulate Matter less than 2.5 µm in diameter).

Each monitoring site was selected to represent communities as potential receptors. Measurements were taken once during the rainy season and once during the drier season in each of 2019, 2020 and 2021.

All measurements were within national limits specified for each parameter.

20.2.7 Environmental Noise and Vibration

Ambient noise was measured at 13 locations that were identified as sensitive receptors in the Project area, including communities and forest patches. Measurements were conducted during the day and during the night in wet and drier seasons of 2019, 2020, and 2021 using a calibrated sound monitor. All day time noise measurements in the baseline condition were within Ecuadorian standards, however, there were exceedances of the night time limit of 45 dB at some locations attributed to insects, farm animals, or strong flowing water.

Similarly, baseline vibration measurements were acquired at 9 locations close to surrounding community infrastructure. Measurements were conducted during the day and during the night in wet and drier seasons of 2019, 2020, and 2021 using a calibrated vibration monitor. All baseline vibration levels were below Ecuadorian limits.

20.3 Socio-Economic Baseline Environment

To support the environmental and social assessment of the Project, a directed socio-economic study was completed in 2021. Local communities identified as being potentially affected by the exploration activities were all located within the cantons and parish of Las Naves, in the Bolivar Province. Demographic characteristics for the province of Bolivar presented based on results of the 2010 census indicate a total population of approximately 184,000, with approximately 6,000 living in the canton of Las Naves. The population growth is relatively stable, with little expansion over time.

Food security in the area is high, with most inhabitants consuming three (3) meals per day. Food and consumer goods are traded locally and produced on individual properties. The majority of dwellings are serviced by public water infrastructure, with the remainder reliant on a private well or surface water source. Regardless of its source, many families boil water for potable use, as treatment facilities are lacking. Electricity is provided by the grid for more than 90% of the houses.

There are four (4) health services clinics in the area, of which one is private, two are operated by the Ministry of Public Health, and one by the Ecuadorian Institute of Social Security. Traditional medicinal practices, including the use of locally collected herbs, is commonly practiced.

There are 14 schools in Las Naves. Approximately 40% of the adults in Las Naves have completed secondary school, with approximately 35% having completed only a primary or basic education level. On average, women have a lower level of education than men, which is reflected in the lower literacy rates for women also. However, of those from the area who went beyond secondary school to study at university, all are women.

The community housing in the area is mainly constructed of brick or cement blocks, with a smaller proportion made of wood or adobe. Most houses have corrugated zinc roofs.

Agriculture, forestry, hunting, and fishing were noted as the main economic drivers in the parish.

20.4 Archaeology

The Project area would likely have historically been used by various cultures as a transportation corridor between the Andes mountains and the coast (Cardno, 2021). A focused archaeological study in 2021 encountered ceramic and lithic remnants, estimated to have originated with Chimbos-Tomabelas culture, and possibly influenced by the Puruhá and Milagro-Quevedo. Dating of the recovered artefacts was not completed, but the cultures that they are associated with were predominant in the area between approximately the years 500 and 1500.

20.5 Stakeholder Consultation and Disclosure

20.5.1 Consultation Activities

Curimining conducted interviews with representatives of organisations, local governments, community leaders and members of the public in affected communities to gauge the perception of communities on the presence of the concessions. With some local exceptions, there is general support for the project at the exploration stage, as the community benefits from local employment. Further local infrastructure improvement and employment were the top comments regarding the advancement of mine development.

20.5.2 Corporate Social Responsibility and Community Development Projects

Curimining has located their offices in the town of Las Naves, and, in lieu of having a camp kitchen, provides staff with a per diem to take meals in local restaurants. This has allowed for good integration of the company to the community and demonstrated real benefits to the local economy. Curimining has selectively sponsored local groups and activities, based on community feedback to determine where there was greatest interest. Sponsorships include:

- Education and capacity building for local businesses;
- Traditional dance group "*Qhuya Kawsay*";

- Development of youth soccer leagues;
- Support to the "*Los Mineros*" local soccer club;
- Scholarships for local schools;
- An agro-forestry plant nursery;
- Support for cattle farmers;
- Support of local festivals; and
- In-kind and financial contribution to local road maintenance.

Curimining continues to operate a robust community engagement program, with an office available to the public and as a base of outreach activities.

20.6 Environmental and Social Risks and Impacts

As per Ecuadorian requirements, a Risk Assessment was conducted and included as part of the Project ESIA. It considered risks to the Project from the biophysical environment, including:

- Vegetation overgrowth;
- Toxic plants / venomous fauna;
- Seismicity;
- Volcanic activity;
- Floods; and
- Extreme climate events.

With appropriate design, management, and monitoring, each risk in this category was assessed as moderate or lower.

Additional risks in the ESIA consider the potential socio-economic effects on the Project, such as anti-mining sentiment, which can have higher short-term effects and will be an important aspect to monitor.

The ESIA also included an assessment of social and environmental impacts from the Project. The majority of impacts were identified in the construction phase, and these primarily revolved around the clearing and removal of vegetation. Local impacts to air quality, especially from fugitive dust were also noted during construction and early operations. Increased noise and vibration are expected.

The change to surface water flow and quality was assessed and found to have potentially moderate impacts, given the design of the water management structures which minimize contact water.

Potential Social Impacts were identified and assessed, including immigration of workers, increased pressure on health and municipal services, increased traffic, and changes to local market conditions. Positive social impacts, including increased access to employment, education, and healthcare were also considered. The majority of assessed impacts were deemed moderate or not significant, in consideration of mitigation applied.

20.7 Environmental Management and Monitoring Plans

A detailed Environmental Management Plan (EMP) was developed for the ESIA in order to establish the actions that the Project will implement to prevent, avoid, mitigate, control, correct, compensate, restore and repair any negative impacts and enhance the positive impacts foreseen in the environmental impact assessment. The EMP is composed of the following components following:

- Impact Prevention and Mitigation Plan;
- Contingency Plan;
- Communication and Training Plan;
- Waste Management Plan;
- Community Relations Plan;
- Rehabilitation Plan;
- Closure Plan;
- Monitoring Plan.

Each component of the EMP provides planning and commitments by Curimining to avoid, mitigate, or compensate for any identified impact. It addresses the physical, biological, and socio-economic environments.

20.8 Waste Management

Curipamba will have two (2) main waste dump facilities and a tailings storage facility:

- A waste facility for storage of overburden and saprolite waste (Saprolite Waste Dump (SWD));
- Waste Rock Facility 1 (WRF1) for storage of PAG and unsuitable waste rock for construction; and
- TSF/WRF2's embankment dump for the storage of suitable NAG material and storage of process tailings in a lined facility formed by the waste embankment.

The Site layout is presented in Figure 20.1.

20.8.1 Saprolite Waste Dump

The SWD will store saprolite and overburden waste from mine pre-stripping and mine infrastructure construction. The natural conditions of the saprolite at Curipamba show high in-situ water content, which will make it difficult to traffic and compact as engineered fill to achieve both high density and shear strength. To improve the workability of the saprolite and increase the stability of the dump the outer zone of the dump is designed to allow co-mingling of saprolite and waste rock. Waste rock placed in the zone will improve trafficability, density of the mix and shear strength as follows:

- The co-mingled zone will be 150 m wide and will be built in cells bounded with rockfill berms, for safety of the equipment and workers.
- The saprolite waste will be placed in the cells in horizontal layers. Traffic of equipment will help compact this material to a consistency adequate for placement of waste rock on top. Caution must be taken to avoid overworking of the saprolite and destruction of its internal structure, which could be make this soil to behave like a fluid.
- The waste rock for mixing or co-mingling, will be placed on top of the saprolite layer placed previously. Waste rock will be placed over the saprolite layer to provide a trafficable surface for future placement. The thickness of the waste rock layer will vary and depend on the moisture content of the saprolite, the placement procedures, the mining equipment used and the compaction efforts during construction. There are no special quality requirements for the waste rock for co-mingling, provided the waste rock can support access and operation. Preference is given to tuff and PAG rocks which will be encapsulated in the saprolite.
- Construction of the co-mingling zone is preferred during the dry season, where it is expected that reduced volumes of waste rock will be required.
- The general zone is for placement of saprolite waste without mixing with rockfill. Saprolite in this area will be unloaded from the trucks, then pushed and accommodated by the equipment. Nominal compaction will be achieved by traffic of heavy haulers.

Sequencing for disposal of excavation saprolite in the general zone should be reviewed with more detail in the next phase of design. Construction of the co-mingled and general zones should be done in parallel, but as construction advances into the wet season, it is recommended that the co-mingled zone be built to a given elevation to leave adequate capacity for placement of general saprolite fill during the wet season.

The SWD will be founded mostly on dacite saprolite. Removal of this material down to the transition zone will occur only at bottom of the creek. The geotechnical parameters for the foundation soils were reviewed from the results of the field investigations. The foundation materials underneath the SWD showed similar density and strength conditions to the ones observed in the foundation of TSF/WRF 2 embankment dump.

The co-mingled zone is assumed to behave in a drained manner and founded on the transition zone in the lower portions of the valley. The general saprolite waste placed behind the co-mingled zone is assumed to: (i) be loose to compact and near saturation, (ii) have an undrained shear response and, (iii) be susceptible to liquefaction, resulting in a minimum post-earthquake strength ratio of 0.08. The assumed shear strength parameters should be confirmed with field trial compaction tests before construction.

The SWD is not designed to store water and an underdrain is proposed at the bottom to lower any potential phreatic surface within the body of the dump

The estimated factors of safety meet the stability criteria and confirm the recommended 4H:1V slopes and 150 m wide co-mingling zone.

The SWD will have two key seepage management components described next:

- Underdrains placed below the waste material are designed following a similar approach to the one discussed for the TSF/WRF2. The underdrain will consist of rockfill protected with a heavy-duty geotextile and a filter material above to prevent migration of fines into the fill.
- A temporary water collection pond downstream. The pond will provide storage capacity to manage sediments during construction and monitor seepage water quality after the end of construction. The pond will be eventually flooded by the tailings beach and become inoperative. Seepage from the SWD will flow into the TSF pond from where it will be reclaimed for process or sent to the WTP.

20.8.2 Tailings Storage Facility

The TSF will contain the process tailings in a lined dam created by the construction of a TSF embankment. The TSF embankment will be further supported through the establishment of a downstream Waste Rock Facility (WRF) 2. For purposes of this report, the embankment section of TSF/WRF2 will be referred as the "TSF/WRF2 embankment dump" or "embankment dump", and the tailings impoundment will be referred as the "TSF/WRF2 impoundment".

The TSF/WRF2 is located in El Silencio creek, next to the Process Plant. Tailings is discharged and stored behind the 150 m high and 600 m wide embankment dump, which will be fully lined and built following the downstream construction method.

The TSF/WRF2 is designed to provide storage of tailings, temporary excess water storage (before water treatment and discharge) and the design flooding containment.

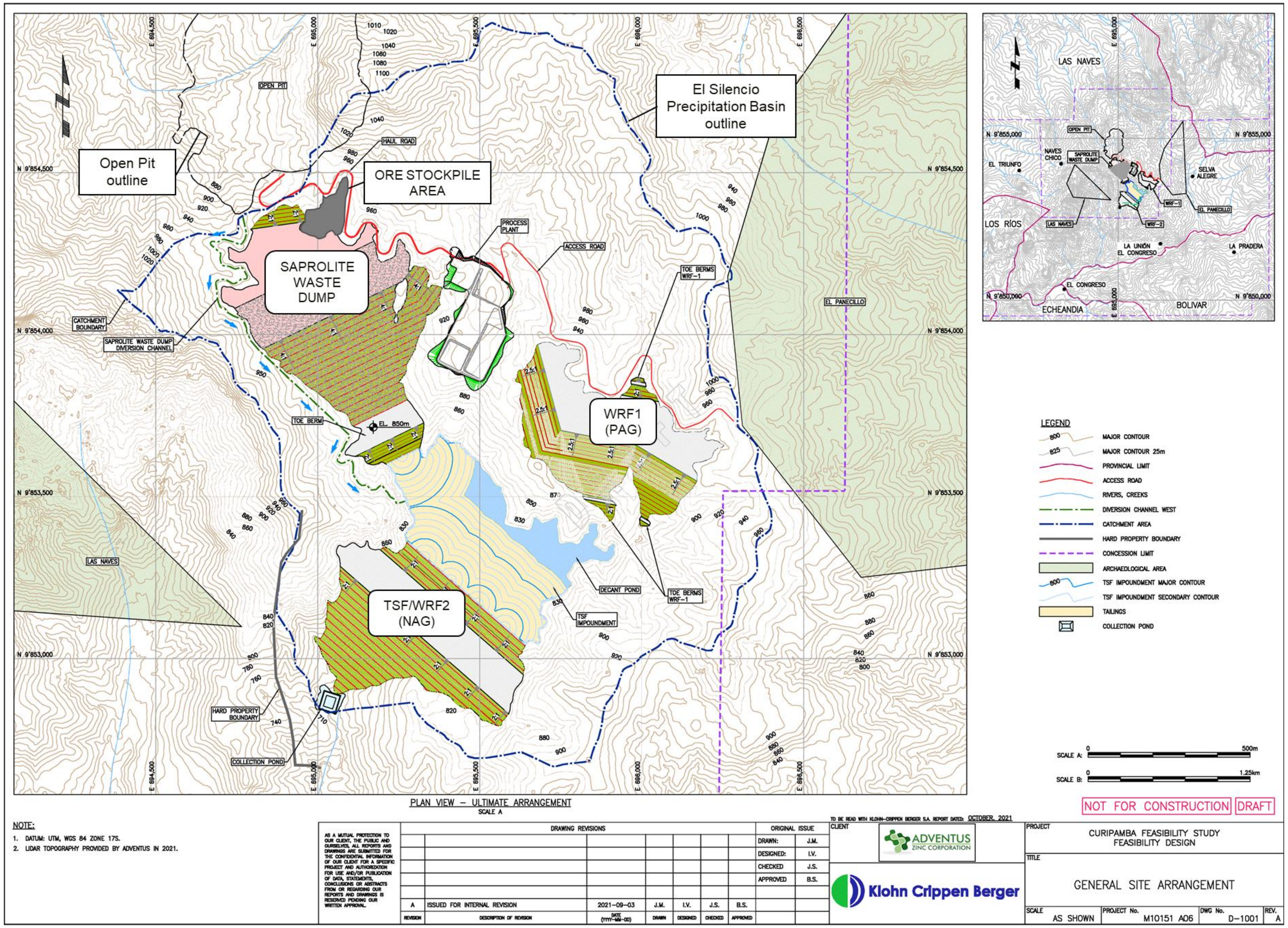
20.8.2.1 TSF Design Criteria

The TSF has been designed following the Ecuadorian and International design guidelines as follows:

- The "Ecuadorian Instructive for Mine Plants and Tailings Facilities" (2020).
- The "Global Industry Standard on Tailings Management" (GISTM) (ICMM, UN and PRI, 2020).
- The Canadian Dam Association (CDA) guidelines (2019).

A review of the potential impacts under hypothetical failure scenarios of this facility was used for its classification including the results of a dam break assessment conducted by KCB for the TSF.

Figure 20.1– General Layout of Waste Facilities at End of Mine Life



Source: KCB, 2021

Based on the results of this assessment, the Curipamba tailings facility is classified as an "Extreme" consequence dam following the GISTM and CDA Guidelines. This implies using the maximum probable precipitation and maximum credible earthquake for engineering design.

This design approach meets two (2) key objectives:

- It provides adequate storage for tailings and mine waste within the mine property.
- It makes material unwanted events such as overtopping, slope failure and internal erosion, less credible for a catastrophic failure, given the size and width of the waste dump embankment.

20.8.2.2 TSF Tailings and Waste Rock Management

The waste rock dump embankment will act as the retention structure and tailings will be discharged and stored using the embankments formed. The tailings and waste rock disposal strategy implies that:

- The waste dump will become the dam and will be designed, constructed and managed according to national and international design guidelines (20.8.2.1) for an "Extreme" consequence dam.
- The upstream side of the embankment dump includes a geomembrane liner and filters to mitigate seepage. The TSF/WRF2 impoundment will be fully lined with a geomembrane.
- The embankment dump is designed to have global and internal stability (against internal erosion) with a compacted embankment fill designed to limit erosion, settlement and deformation while allowing management of drainage and seepage.
- The dump embankment will be built using available mine waste materials which will be placed strategically to maintain the physical stability and mitigate geochemical ARD) potential.
- The tailings process includes crushing, milling grinding and flotation for production of copper (Bulk Rougher Tailings, Bulk Ro TL) and zinc (Zinc Rougher Tailings, Zn Ro TL) tailings. Approximately 77% of the flotation tailings production will rougher copper (Bulk Ro TL) and the rest will be zinc rougher tailings (Zn Ro Tl). Both streams will be thickened to a solids density of 55% and pumped to the TSF.

Pilot plant scale samples were received and tested in KCB's soils laboratory in Vancouver, Canada. Tests included index and compressibility testing to understand the texture and consolidation behaviour of the tailings. A conservative density of 1.35 t/m^3 has been selected to account for potential adverse initial operational conditions, where mill and water management upsets could result in more water stored in the tailings impoundment starter dam. This is a credible condition for Curipamba where the rainfall is high and positive water balance conditions are expected.

Tailings will be discharged from the crest of the embankment and from the toe of the SWD to promote the formation of a tailings beach while maintaining the decant pond away from the TSF embankment.

Tailings are PAG and their ARD potential ARD will be managed in the TSF/WRF2 impoundment, which will be lined to limit seepage. Minimising length of exposure during operations by continually discharging new tailings and alkaline supernatant over older tailings will minimise the potential for sulphide oxidation in the tailings beaches during operations.

The quality and durability of the waste rock materials was reviewed using in-situ information from exploration and pit geotechnical drillholes summarised in a rock mechanics database provided by DRA. The in-situ conditions of the rock pit slopes were used as indicative for the future conditions of the waste rock after blasting, handling, transport and placement for dam/dump construction. Using this information, a qualitative classification system was established to screen and select rock for dam construction.

Based on the above, the andesite and basalts were screened as the preferred materials, while more work is required to confirm the quality and durability of the tuff. The potential for deterioration of Curipamba's tuff was reviewed with a simple test where two (2) tuff cores were exposed to air and water (submerged in a bucket full of water) and their conditions observed over a period of one week. No changes or deterioration were observed. The results of this test could indicate Curipamba's tuff may not degrade in the short-term, but more rigorous testing is recommended on additional samples to confirm this initial observation.

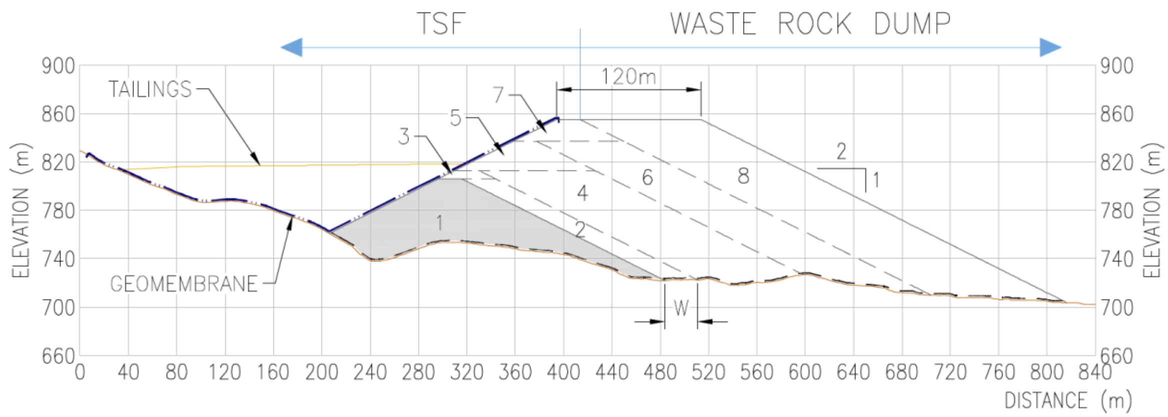
From these preliminary assessments and from the database information, the feasibility design has made a conservative allowance that 35% of the tuff in Curipamba may be suitable for use in dam construction, along with andesite and basalt for dam construction. This value may be increased following further testing and may provide significant reductions in pre-mining overburden stripping should this be confirmed. This additional testing is planned for future studies for the Project.

20.8.2.3 TSF Embankment Dump Design

The TSF/WRF2 embankment dump includes a starter dam, subsequent raises using competent rockfill on the upstream slope and general mine waste rock on the downstream slope. The rockfill and waste material are assumed to be "free-draining". This distribution of materials is intended to limit deformations and damage to the liner and filters on the upstream zone, while the general waste rock material provides the mass required to maintain the facility stable.

The typical cross section of the TSF/WRF2, including the proposed particle size distribution (PSD) of the materials Figure 20.2. At the time of preparation of this design, the PSD for the ROM was not available; it is assumed that blast designs can be adjusted to meet the design PSD envelope required.

Figure 20.2 – TSF/WRF2 Cross Section



Source: KCB, 2021

Slope stability analysis were completed using geotechnical models including the topography, design configuration, foundation conditions, and geotechnical parameters assigned to the foundation and construction material. The geotechnical parameters for the foundation materials were estimated from the 2020/2021 field investigations completed for the FS. The geotechnical parameters for the construction materials were defined based on available information and KCB's experience in similar geology. Further testing on these construction materials is recommended in future stages of the Project.

The stability analysis was carried out following two-dimensional limit equilibrium method proposed by Morgerstern and Price (1966), under static conditions and post-earthquake conditions, using drained and undrained strength parameters, as applicable. The slope stability analysis was executed using the computer program SLOPE/ W (Geostudio 2021). The TSF/WRF2 will be a lined facility built following the downstream construction method. The estimated factors meet the stability static stability criteria.

The performance of the dam under seismic conditions was assessed using pseudo static analysis used as a preliminary screening tool. A horizontal acceleration component equal to 50% of the design PGA was included in the models as recommended by Hynes-Griffin (1982) and the estimated factors of safety were below 1, suggesting seismic displacements could be large. Following this, seismic displacements were estimated using simplified methods as proposed by Hynes-Griffin (1982), Newmark (1965) and Bray, Travassarou and Macedo (2007 and 2018). From the results of these analysis, a maximum seismic displacement of 1.5 m was allowed for in the design. Following this, the filters and drains in the embankment dump have been sized with thicknesses that double the estimated seismic displacement.

20.8.2.4 TSF Seepage Control

The TSF/WRF2 will have three main seepage management components described next:

- A geomembrane liner system is included on the upstream slope and basin. The nominated geomembrane is Linear Low-Density Polyethylene (LLDPE) placed on the upstream embankment slope extending into the full impoundment of the TSF. The LLDPE geomembrane was selected as the preferred geomembrane material due to its strain properties and puncture resistance, which is compatible with potential irregular surfaces and reduces the risk of puncturing or tearing due to settlement-induced strains in the foundation (differential settlement) and seismic-induced strains at the dam slope. The TSF is founded on saprolite of varying thickness with increasing coarseness with depth. Hydraulic conductivity testing showed values up of 1x10-5m/s for the natural saprolitic soils, which are low to medium hydraulic conductivity soils which prompted the decision to line of the full impoundment.
- Underdrains will be placed below the embankment dump and designed with a flow capacity 10 times larger than the base average flows estimated for the El Silencio creek and its small catchments. The flow capacity of the drains is also larger than the estimated infiltration assuming the maximum daily precipitation during normal and El Niño years.

A collection pond will be located downstream of the TSF/WRF2 embankment dump. The TSF/WRF2 impoundment will be a fully lined facility and seepage flows will mainly come from infiltration into the waste rock dump captured in the underdrains. These flows will be delivered to a collection pond sized to provide storage capacity to monitor water quality before it is released, pumped back to the TSF or sent to the WTP, as required.

20.8.2.5 TSF Water Management

The TSF/WRF2 impoundment will be designed to manage and release floods as an emergency measure during initial operations and closure. This means the TSF will have an emergency structure designed to convey the peak flow of the inflow design flood (IDF) including an appropriate freeboard allowance to prevent overtopping of the dam. According to CDA guidelines, when a spillway is considered, the TSF should be designed with a storage allowance (without release) for a contingency inflow defined as the Environmental Design Flood (EDF) (CDA, 2019). Later in the operations of the TSF, the facility will be designed as a “zero-release” dam. This means the IDF will be stored without release through a spillway and the dam will be built with adequate freeboard to contain the flood.

During operations, the TSF/WRF2 impoundment will be the main repository of surface runoff and precipitation water from the site’s catchment. The TSF/WRF2 will collect the water, recycle the required volumes for process and provide temporary storage of excess water to maintain a monthly peak water rate of 470 m/h (130 L/sec) to the WTP. This peak water rate was determined by DRA and used to size the WTP.

The waste rock used to build the TSF/WRF2 embankment/dump will be "free-draining". Most of the water from this area will be from infiltration through the waste material. Seepage and runoff from this area will be captured in a collection pond at the toe, from where water will be sent to the WTP.

20.8.3 Waste Rock Facility 1

Curipamba will produce both PAG and NAG mine waste materials that require separate storage. NAG waste rock will preferably be used for construction of the TSF/WRF2 embankment, particularly the later outer stages of the facility. NAG waste rock will also be used to encapsulate any good quality PAG rockfill considered for TSF/WRF2 embankment dump construction. The objective is to avoid using any PAG rock in the construction of the TSF/WRF2 if conditions allow it.

WRF1 is proposed for storage of poor quality and PAG waste rock not sent to construction of TSF/WRF2 and not used for co-mingling in the SWD.

WRF1 has no waste rock zones in its design. As a general recommendation, the waste dump can be built in "ascending platforms" built with an "end -dumping" method (Method IV) (Hawley and Cunning, 2017), contained behind toe berms which will initially act as an "impact berms" to reduce risks of boulders rolling downstream.

The dump has been designed to a maximum capacity of 3.5 Mm^3. This volume can be reduced if more waste rock is sent and stored in the SWD, either used for co-mingling or as part of the external rockfill zone discussed as an opportunity.

The estimated factors of safety meet the stability criteria and confirm the recommended 2.5H:1V downstream slope.

The seepage management components for this facility are similar to the ones proposed for the TSF/WRF2 and SWD. The main underdrain is designed with a capacity larger than the base flows in the main creek that runs from north to south in the WRF1 area. Similar to the SWD, the collection pond is proposed for seepage and contact runoff water storage in the short-term and it will be eventually flooded by the tailings. In the long-term, seepage from WRF1 will flow into the TSF for reclaim for operations or will be sent to the WTF.

20.8.4 Ore Stockpiles

The FS mine plan requires the temporary stockpiling of the three (3) ore types during the LOM. The mine plan is presented in Section 16 of the Report. Three (3) long-term stockpiles are required to be built for the LOM. A suitable area has been identified that is located between the open pit and the process plant along the main haulage route.

Although the stockpiles are designed to be temporary, the storage area will be utilized throughout the life of mine. The stored material is expected to be acid generating and will require suitable containment to avoid groundwater contamination. The stockpile floor area will be suitably sealed with a layered saprolite design to capture and channel seepage water and run-off water toward the Saprolite waste facility for temporary water storage and treatment in the TSF water treatment plant.

The maximum combined stockpile size is 400 kt in Year 9 of production.

20.8.5 OTHER WASTES

Chemical waste produced by laboratories and other reagent processes, including emergency spills and accidents will be locally contained for passivation and removal. Water soluble chemical waste will be treated and chemically passivated before being discharged into the TSF for tailings entrainment. Non-soluble waste will be captured and suitably bagged or contained for transport and disposal off-site at a registered disposal facility. Any affected equipment or material that is required on-site will be suitably cleaned and cleaning materials also disposed of.

Organic waste, from local flora and wooden sources will be suitably treated and sized for disposal to a designed area of the topsoil stockpile toward developing sustainable composting practices on site.

General waste on-site will be sorted into allocated bins for potential recycling off-site at the process plant terrace, for domestic landfill disposal, if applicable. The Project will have its own domestic waste landfill facility located within WRF1 area from where seepage water will drain into the TSF. The land fill facility will be designed as part of the detailed design phase.

Equipment waste will be consolidated in a designated area located near the mine workshops, or on top of WRF1. Equipment being retired will be suitably drained and cleaned of all hydrocarbon or other contaminant sources prior to be placed in the designated area. An active scrap and recovery program will be evaluated during the operational phase of the Project.

Biological waste, other than sewage treatment effluent and organic waste, will be suitable isolated and stored for treatment off site at a registered disposal facility.

20.9 Water Management

20.9.1 Site Hydrology

The site is located in the tropics, with a hot and humid climate throughout the year. The natural water balance is positive, with precipitation about seven times larger than evaporation potential.

KCB (2021) conducted the hydrology assessment for the Project, using information from regional stations operated by the Ecuadorian Hydrology Service (INHAMI) and a local station installed in the camp.

Average annual rainfall in the Project area is estimated as 2,300 mm. According to the regional records reviewed, most of the stations presented precipitation records above the average in 1998, when the highest recent El Niño phenomenon occurred, affecting the South American Pacific, especially Ecuador and Peru. The rainfall increase was more than 50% compared to average years.

The hydrology assessment results show that Curipamba could be impacted by the El Niño phenomenon and this event has been taken into consideration in the precipitation and storm estimates. The estimated monthly average precipitation and evaporation are shown in Figure 20.5.

Table 20.2 – Monthly Average / Evaporation Precipitation

Condition	Average (mm)												
	Jan	**Feb**	**Mar**	**Apr**	**May**	**Jun**	**Jul**	**Aug**	**Sep**	**Oct**	**Nov**	**Dec**	**Total**
Precipitation Normal year	383.5	423.7	478.1	387.8	191.2	57.8	25.5	15.1	27.6	47.6	51.0	175.5	2 264.3
Precipitation El Niño year	293.5	352.8	556.5	249.7	198.8	187.4	112.0	99.9	239.2	224.6	350.5	550.5	3 415.4
Pan Evaporation	28.6	20.4	26.3	34.5	48.8	64.6	80.0	95.0	90.6	80.8	66.2	45.4	681.1

The design storms were estimated using probabilistic statistical methods and are based on the records of maximum daily precipitation from the regional stations “El Corazón” and “Ventanas”. Figure 20.6 shows the probabilistic analysis results.

Table 20.3 – Storms for Different Return Periods

Return period	Daily Pmax	
(Years)	El Corazón Station Design values (mm)	Ventanas Station Comparative values (mm)
2	97	*143*
5	123	*167*
20	151	*193*
50	167	*207*
100	179	*216*
200	189	*225*
500	203	*237*
1000	213	*245*
2000	222	*253*
10000	244	*270*

Additionally, a deterministic analysis was conducted to estimate the Probable Maximum Precipitation (PMP), using the Hershfield methodology (1961) and different durations as shown in Figure 20.7.

Table 20.4 – Probable Maximum Precipitation

Description	Duration hours	Design Value El Corazón Station	*Reference Value Ventanas Station*
PMP (24)	24	544	*540*
PMP (48)	48	756	*750*
PMP (72)	72	917	*910*

The hydrological assessment will need to be revised during the Project operation as the information record from local stations near the site improves.

20.9.2 Ground Water

20.9.2.1 Hydrogeology – Naves Chico

Based on site-wide groundwater elevations recorded, there is a groundwater flow divide at approximately the same location as the topographic divide. Since, the pit is situated on the west-facing slope of the Naves Chico Valley, piezometric heads southeast (up-gradient) of the El Domo pit are expected to be affected to the greatest extent by mine dewatering, whereas piezometric heads northwest (down-gradient) are expected to be least affected. Conceptually, piezometric heads east, west and north of the pit will be affected by dewatering as far as the site boundary. As piezometric heads are gradually lowered with dewatering of the El Domo pit, the groundwater flow divide (south of the pit) is expected to shift further toward the south to some point beneath the TSF-Cell 1.

Based on the feasibility level hydraulic conductivity estimates, groundwater flow in the Naves Chico Valley is inferred to occur primarily at shallow depths within the weathered bedrock. Groundwater is expected to flow in the weathered bedrock and base of the saprolite following the topographic gradient and discharge into creeks or following preferential flow paths in transmissive faults with the underlying bedrock lithologies. Vertical pressure gradients along these faults may be locally upward or downward depending on the continuity of fractures and hydraulic conductivity in the fault zones.

The chemistry of the groundwater in the Naves Chico Valley shows three (3) distinct hydrochemical facies: calcium-sulphate, sodium-bicarbonate, and calcium-bicarbonate.

Generally, the calcium-bicarbonate facies groundwater is interpreted to be most closely aligned with the chemistry of precipitation and runoff (i.e., active recharge zone). The sodium-bicarbonate facies groundwater is inferred to represent groundwater that has flowed through the subsurface following a longer/deeper flow path.

Using the sulphate composition of water samples an indicator of groundwater seepage pathways, conceptually, precipitation infiltrates the subsurface and seeps downwards into the weathered bedrock. The groundwater flows laterally through the weathered bedrock increasing its sulphate composition following the horizontal hydraulic gradient of between 0.2 - 0.3 m/m. Furthermore, a downward vertical hydraulic gradient indicates the potential for groundwater to infiltrate deeper into the bedrock below the weathered bedrock zone and into the lapilli rock.

The groundwater flow conditions are inferred to be representative of flow conditions across the lower elevation of the future pit footprint. As groundwater continues to follow the horizontal gradient downstream of the pit footprint, it is conceptualized as increasing its sulphate concentration. Based on currently available data, it is not clear if the sulphate laden groundwater discharges into Quebrada Naves Chico or, if it remains in the subsurface flowing northwest beneath the Quebrada Naves Chico.

20.9.2.2 Hydrogeology – El Silencio

The TSF, Process Plant Site and WRF1 are all situated in the headwaters of El Silencio watershed between elevation 960 m amsl and 860 m amsl.

Based on the feasibility level hydraulic conductivity estimates, shallow groundwater flow throughout the El Silencio Valley occurs predominantly within the weathered bedrock HSU. Groundwater flow in the weathered bedrock is driven by the topographic gradient and is expected to discharge into the Estero El Silencio at low elevations. Based on one year of streamflow monitoring (between September 2019 and November 2020), runoff from the upper elevations of Estero El Silencio is inferred to recharge the weathered bedrock.

Precipitation falling in the headwaters of the watershed is inferred to infiltrate downward into the underlying weathered bedrock. Most of the flow originating from the weathered bedrock at higher elevations will flow under unconfined conditions within the weathered bedrock, while a lesser component of flow is expected to infiltrate into the deep-seated competent bedrock. Following pressure gradients, some the groundwater in weathered bedrock is expected to recharge faults in the bedrock. Recharged groundwater flows downgradient within the weathered bedrock towards the Estero El Silencio. In general, the Estero El Silencio is anticipated to recharge the adjacent shallow groundwater system in the higher zones, groundwater in the weathered bedrock is expected to discharge directly into the Estero El Silencio.

The groundwater in the weathered bedrock at most locations is conceptualized to have a chemical character representative of river water (calcium-bicarbonate facies). However, locally at the southern uplands and east-facing valley slope, groundwater is of a sodium-bicarbonate type. Since the total dissolved solids (TDS) concentrations of these samples are low relative to some calcium-bicarbonate facies groundwater, the sodium-bicarbonate facies groundwater may be a consequence of the different geological unit through which it passes and not necessarily an indication of its travel time in the subsurface.

Since the competent bedrock in the El Silencio Valley is comprised primarily of dacite (geometric mean hydraulic conductivity of 9×10^{-7} m/s) with some andesite (3×10^{-7} m/s), groundwater flow in the competent bedrock is expected to be limited and driven by pressure gradients and is expected to be predominantly sodium-bicarbonate facies groundwater. Calcium-bicarbonate facies groundwater may infiltrate directly into the competent bedrock at higher elevations where the bedrock outcrops at ground surface or is at a shallow depth. Although grainstone has been shown to have a relatively high permeability (1×10^{-6} m/s), it has been mapped locally in the upper El Silencio Valley and is generally inferred to be limited in extent.

The only hydraulic conductivity test conducted on a fault in the El Silencio Valley reported a hydraulic conductivity of 6×10^{-6} m/s. The strike and dip of this and its southern extent is unknown. Also, the capacity of El Gallo, F3-C3, and F4C faults to transmit groundwater is unknown. There may be other unmapped faults in the El Silencio Valley.

20.9.3 GEOCHEMISTRY

Mined material lithologies were identified and the geochemical characteristics of each rock type were determined. Adventus retained pHase Geochemistry Inc. (pHase) to characterise waste rock and tailings and determine source terms for the mined material. The source terms were used as inputs to the water quality models for both the El Silencio and Naves Chico basin mine facilities.

Chemical load predictions for the open pit, tailings beaches and waste facilities were determined (Phase, 2021). Prediction results indicate the SWD and WRF1 (PAG) will generate acidic conditions in the pH 2 to 3 range should the full volume of each of these facilities be available for oxidation. This prediction reflects the limited neutralization potential of the NAG rock to provide any meaningful alkalinity to buffer acidity produced from the PAG rock despite the volumetrics being dominantly NAG. At these low pH values, contact water is predicted to have associated high sulphate and metal concentrations for nearly all parameters including aluminum, arsenic, cadmium, copper, iron, lead, manganese, nickel and zinc, requiring containment and water treatment.

Source term predictions for the TSF/WRF2 facility assumed rock scheduled for placement on the outer layer of the dump (i.e. that assumed to be exposed to oxidation) would be NAG. With that assumption, source term predictions indicated that seepage should remain neutral (pH 7.2 to 7.7) but have elevated sulphate and zinc. Should any PAG rock be used in the later stages of construction, because of the limited neutralization potential of the NAG rock, contact water would be predicted to become acidic (pH 2.9).

Overall, the results suggest that the seepage quality associated with the TSF/WRF2 are very sensitive to small volumes of PAG material. This suggests that waste management will be important such that any PAG that ends up in the TSF/WRF2 should be isolated from oxygen and placed deep within during early stages of construction and not placed on the outer later stages of construction most susceptible to oxidation.

It is recommended that operational waste segregation and sorting protocols be developed for waste going to the stages of construction for this facility (e.g. sulphide analyses and cut-off criteria for units such as basalt and tuff).

A more comprehensive test work program is recommended during the next project phase to increase accuracy and acid potential determination of the various lithologies.

20.9.4 WATER TYPES

Different water types will be present on site depending upon source, geographical location and surface contact. Four (4) main types of water has been defined for the FS:

1. Contact Water – Water sourced from either rainfall, groundwater or seepage that has come in contact with, or penetrated disturbed surfaces or project related infrastructure that may be subject to rock type oxidation potential, contamination, and fines accumulation. Examples include waste rock facilities run-off, ore stockpile facilities, workshops and tailings pond.

2. Non-Contact Water – Water that is captured or diverted away from any disturbed surfaces or project infrastructure that have minimal potential to be chemically or turbidly impacted by project activities or infrastructure.

3. Domestic waste water – Water that has been used for domestic purposes and include sewage, grey wash water, laundry, kitchen, and bathroom drainage.

4. Potable water – Water that has been sourced and treated for human consumption.

20.9.5 Water Management Infrastructure

Water management infrastructure is a function of the site layout and topography. Natural water accumulation and run-off channels are impacted by the project infrastructure facilities and require either redirection, or containment, conveyance, and potentially treatment.

The main water management criteria is presented below:

- Precipitation run-off and underground seepage water entering the project infrastructure will be collected, contained, treated and discharged within the respective natural catchments basins of El Silencio (TSF/WRF2, WRF1, Plant, Infrastructure) or Naves Chico (Open Pit Mine).

- All contact water will be contained and treated prior to environmental discharge.

- All contact water containment facilities must be suitably sized for containment of agreed storm event volumes.

- Domestic water sources will be isolated and collected via a piped or tank-and-truck method and discharged to a singular point for domestic water treatment.

20.9.5.1 Naves Chico Basin

The Naves Chico basin contains the open pit mine and related mining infrastructure. Contact water will enter the pit through precipitation and ground water inflows and collect in the bottom of the phase one pit sump. From here the water will be pumped directly to the water treatment plant and then discharged to a mine water pond for normalisation from where it will naturally overflow into the natural valley drainage system.

20.9.5.2 El Silencio Basin

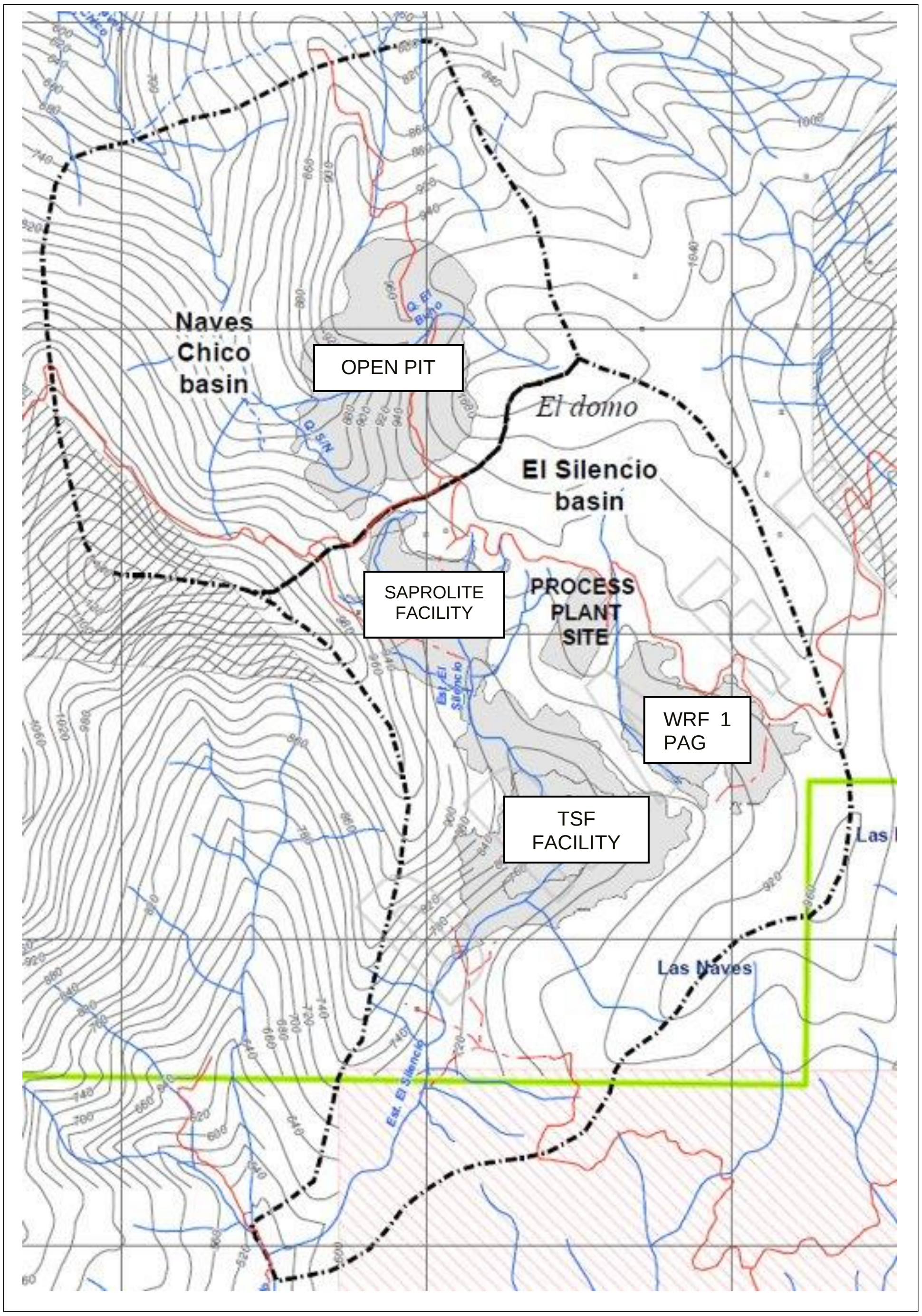
The El Silencio basin contains the majority of the project infrastructure namely: waste rock facilities, process plant, site support infrastructure and tailings facilities. The tailings facility will be constructed in the natural drainage path of the basin and will collect all precipitation water, contact and non-contact water. Diversion of non-contact water with berms and channels were evaluated, but was found to be technically challenging to construct and to maintain in the steep valley terrain, when compared to water capturing and treatment.

Contact water from the process plant, site infrastructure, WRF1 and Saprolite Dump will naturally drain into the tailings facility via constructed and suitable channels to avoid ground seepage.

Contact water run-off from WRF2 will be collected via drainage ditches in a collection pond at the downstream toe berm. This water can be monitored and directly discharged if suitable or pumped to the tailings pond for treatment.

Water will be pumped from the TSF through the water treatment plant and discharged via the WRF2 collection berms to the toe containment pond for monitoring, prior to naturally overflowing into the natural valley drainage system.

Figure 20.3 – Naves Chico and El Silencio Water Catchment Layout



Source: KCB,2021 adapted by DRA

20.9.6 Site Wide Water Balance

DRA retained SLR Consulting Canada Ltd. (SLR) and Minnow Environmental Inc (Minnow) to conduct initial water balance (SLR, 2021) and water quality predictions (Minnow, 2021) respectively for the FS.

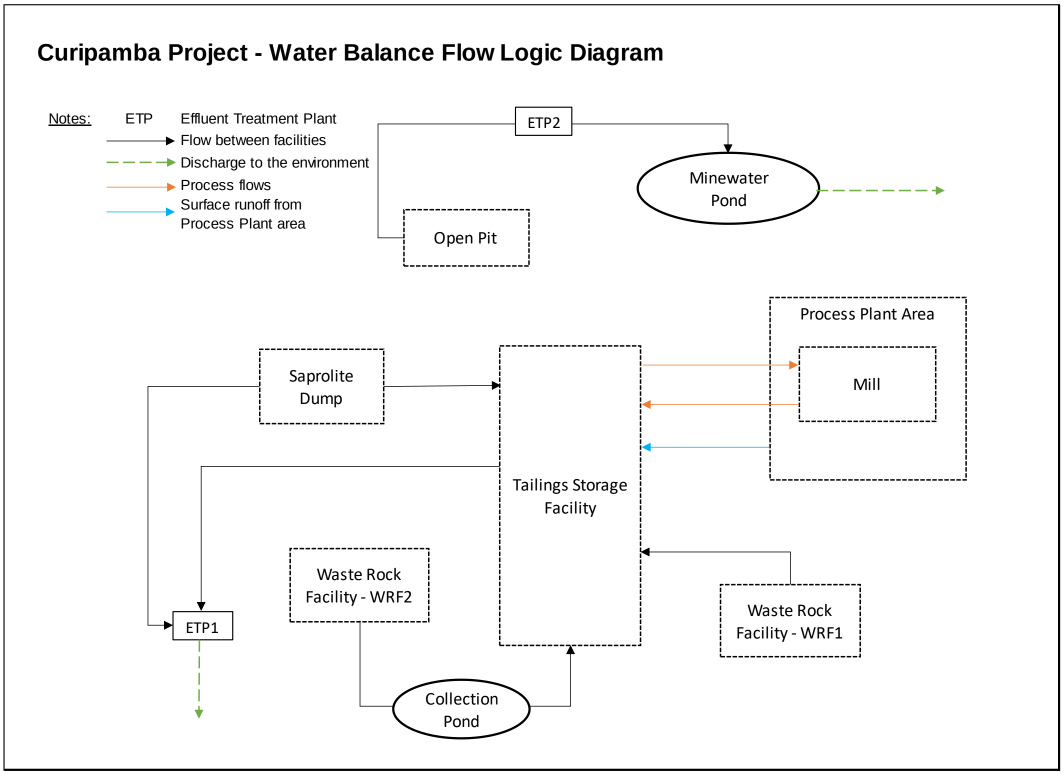
Probabilistic site-wide water balance modelling was conducted to calculate water discharge rates to the environment under variable rainfall conditions, and calculate flows associated with the open pit and waste management facilities to support the water quality modelling. Accordingly, the water balance model was set to:

- Estimate surface runoff from the Curipamba Project (the Project) facility footprints taking into account changing land cover and footprint areas within the Project site;
- Simulate water transfer between Project facilities; and
- Simulate effluent discharges to the environment.

The GoldSim modelling software was used to develop the Curipamba site-wide water balance model. GoldSim is widely used in the mining industry and has the capability to perform both deterministic and probabilistic (stochastic) simulations. The probabilistic simulations consider uncertainty in model inputs and support the prediction of model outputs in probabilistic terms.

A mine site water management schematic for the Project is shown on Figure 20.4.

Figure 20.4 – Site Wide Water Balance Schematic



Source: SLR, 2021

The inflows and outflows/losses considered in the model are as follows:

a. Inflows:

- Surface runoff from rainfall;
- Infiltration to waste management facilities from rainfall;
- Process flows;
- Open Pit groundwater inflows.

b. Outflows:

- Evaporation losses;
- Water volume retained in deposited waste;
- Flow discharge to other Project facilities or to the environment.

The following inputs were defined for the Curipamba site-wide water balance model in GoldSim:

- Rainfall long-term data from El Corazón meteorological station operated by the National Institute of Meteorology and Hydrology from Ecuador (INAMHI for its acronym in Spanish);
- Average monthly evaporation data derived from pan evaporation recorded at El Corazón meteorological station;
- Catchment areas;
- Runoff coefficients (variable according to type of surface);
- Water volume retained in voids of deposited waste;
- Open Pit seepage (pit groundwater inflows);
- Process flows:
    - Water discharged from the mill with the tailings;
    - Water reclaimed from the TSF pond to the mill;
- Maximum treatment capacity of 470 m^3/hr for surplus water to be treated and discharged from the waste storage facilities (TSF, WRF1, WRF2, and Saprolite Dump).

The 12-year mine operation period was simulated in GoldSim. Probabilistic results were obtained through the stochastic simulator performing 1,000 simulations (i.e. realizations) generated with GoldSim to capture the variability in rainfall conditions. In other words, a total of 1,000 different sequences of rainfall over the 12-year period were randomly generated by the model.

The water discharge rates to the environment under variable rainfall conditions resulting from the probabilistic modelling are presented below for the 10^{th}, 50^{th} and 90^{th} percentiles. A percentile is a measure used in statistics indicating the value below which a given percentage of observations in a group of observations falls. For example, in this case the 90^{th} percentile is the value at which 10% of the discharge rates lie above that value and 90% of the discharge rates lie below that value. The 50^{th} percentile corresponds to the median of the 1,000 realizations, representative of a normal year, whereas the 90^{th} percentile is considered representative of wet rainfall conditions.

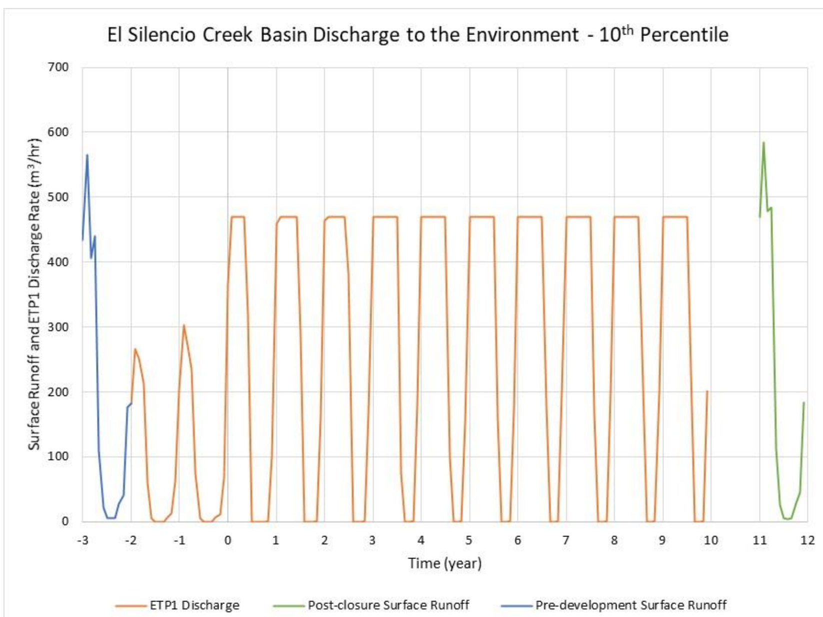
The site-wide water balance modelling outputs in Figure 20.5, Figure 20.6 and Figure 20.7 illustrated for El Silencio Creek basin (i.e. waste management facilities and plant site) show the following:

- Year -3 corresponds to the surface runoff rates for El Silencio Creek basin for pre-development (i.e. existing conditions prior to land disturbance and construction of mine infrastructure);
- Years -2 to 10 correspond to the water discharge rates to the environment from the waste management facilities effluent treatment plant (ETP1 on Figure 20.4) during the with construction/mine pre-stripping phase (Years -2 and -1) and operations phase (Years 1 to 10); and

- Year 12 corresponds to the surface runoff rates once operation activities and active water treatment have ceased, disturbed land surface has been re-vegetated, and the closure spillway at the Tailings Storage Facility is in place. For illustrative purposes a one-year gap (Year 11) was left between end-of-operations (Year 10) and post-closure (commencing in Year 12). The gap represents the closure period, which could last more than one year depending on: i) duration of active water treatment requirements before water quality discharge criteria can be met without treatment; and ii) how long it would take for the vegetation to become well established on the revegetated surfaces.

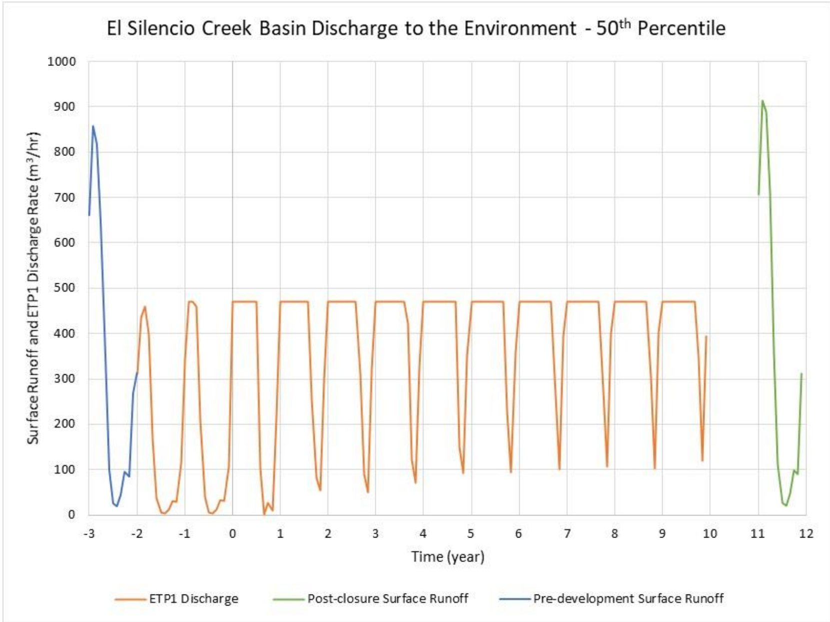
All rates depicted in Figure 20.5,Figure 20.6 and Figure 20.7 correspond to the location immediately downstream of the Waste Rock Facility 2 (WRF2) collection pond.

Figure 20.5 – Predicted El Silencio Creek Basin Discharge to Environment Flows - 10th Percentile



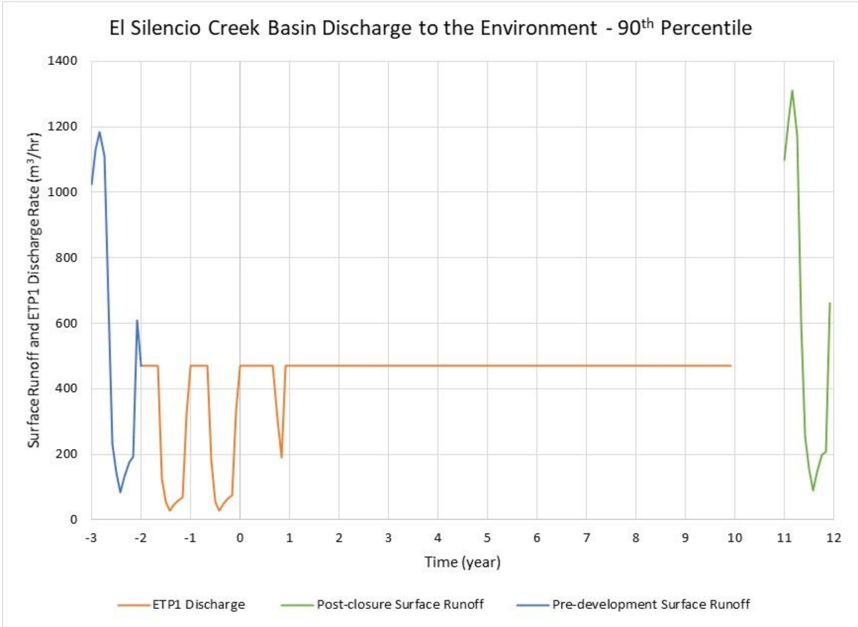
Source: SLR, 2021

Figure 20.6 – Predicted El Silencio Creek Basin Discharge to Environment Flows - 50th Percentile



Source: SLR, 2021

Figure 20.7 – Predicted El Silencio Creek Basin Discharge to Environment Flows - 90th Percentile



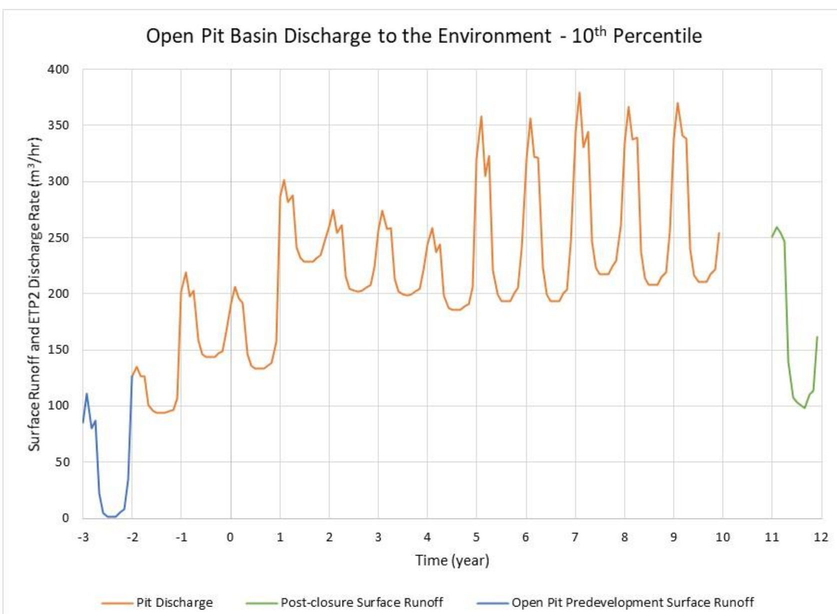
Source: SLR, 2021

The site-wide water balance modelling outputs for the open pit basin in Figure 20.8, Figure 20.9 and Figure 20.10 show the following:

- Year -3 corresponds to the surface runoff rates for pre-development (i.e. existing conditions prior to open pit development) for the basin where the open pit is located;
- Years -2 to 10 correspond to the water discharge rates to the environment from the open pit effluent treatment plant (ETP2 Figure 20.4) during the mine pre-stripping (Years -2 and -1) and operations phase (Years 1 to 10); and
- Year 12 corresponds to the surface runoff rates once open pit mining and active water treatment have ceased. As discussed above, for illustrative purposes a one-year gap (Year 11) was left between end-of-operations (Year 10) and post-closure (commencing in Year 12) representing the closure period.

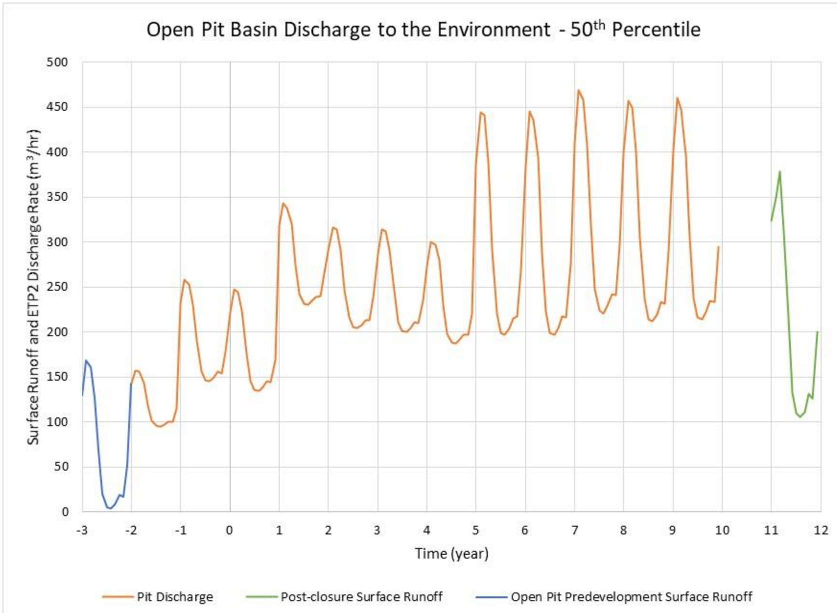
All rates shown on Figure 20.8, Figure 20.9, and Figure 20.10 account for the catchment area corresponding to the ultimate open pit footprint.

Figure 20.8 – Predicted Naves Chico Basin Discharge to Environment Flows - 10th Percentile



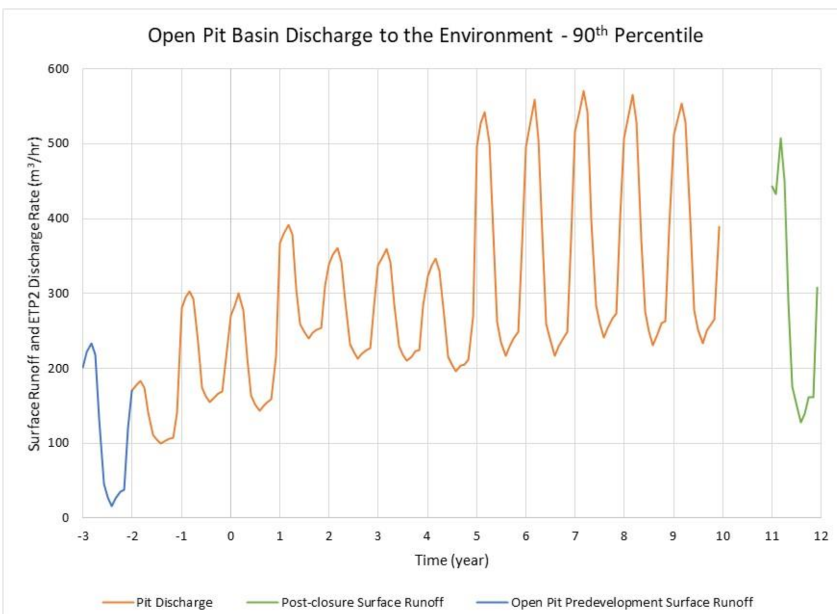
Source: SLR, 2021

Figure 20.9 – Predicted Naves Chico Basin Discharge to Environment Flows - 50th Percentile



Source: SLR, 2021

Figure 20.10 – Predicted Naves Chico Basin Discharge to Environment Flows - 90th Percentile



Source: SLR, 2021

20.9.7 Surface Water Quality

20.9.7.1 Base Line Assessments

Base line water quality assessment has been conducted and is described in Section 20.2.2.

20.9.7.2 Facility Water Quality and Types

Initial water quality predictions were prepared on the assumption that all contact water will be contained, treated and discharged in the respective catchment basins of either Naves Chico, or El Silencio (Minnow, 2021).

The effluent quality predictions were simulated for specific water management facilities during the operations phase as follows: open pit sump and TSF pond. The TSF pond is assumed to receive a mixture of inflows as follows: tailings beach runoff, process tailing and water outflow from the plant, Saprolite Dump drainage, WRF1 drainage, and WRF2 drainage. The effluent quality predictions for the Saprolite Dump drainage have also been completed, but these were taken directly from the geochemical source terms provided by pHase. In addition, water quality predictions of the pit lake were completed for the post closure period.

The results of the water quality modelling are compared to International Finance Corporation (IFC) effluent guidelines and Ecuador maximum permissible effluent limits. The water quality predictions suggest that the untreated effluent that is pumped from the open pit will be acidic (pH 2.60 to 4.86) with elevated concentrations of sulphate and metals. Parameters that are predicted to be greater than the maximum effluent discharge limits include: sulphate, aluminum, arsenic, cadmium, cobalt, iron, lead, manganese, and zinc.

During the post-closure period, the long-term average chemistry in the pit lake will be improved relative to the pit sump chemistry but will require post-closure mitigations to seal and passivate certain exposed rock types to prevent further acid generation and metal leaching.

Similar to the pit sump chemistry during operations, the water quality predictions suggest that the untreated effluent from the Saprolite Dump and TSF pond will be acidic (pH 2.30 to 3.13) with elevated concentrations of sulphate and metals. Parameters that are predicted to be greater than the maximum effluent discharge limits include: sulphate, aluminum, arsenic, boron, cadmium, copper, iron, lead, manganese, nickel, and zinc.

The low pH and elevated concentrations of sulphate and metals reflect the reactive nature of the the PAG rock and limited neutralizing nature of the NAG rock types to offset acid generation. Careful management of NAG rock type placement unto the final outer layers of the waste facilities will be critical to prevent long term active closure requirement.

The water quality models are limited in their scope and highly dependent on input data received from other sources. It is noted that the geochemical characterization and source term data was derived off a limited lithological sample set for some lithological units. Process water chemistry was not considered as it is preliminary in nature with further design to follow. It is recommended that further geochemical test work be undertaken in the next project phase to improve the understanding of PAG and NAG lithologies and subsequently improve water quality model predications. Process water quality must be further defined as part of detailed design to further improve the water quality: additional lime dosing of the process water may be beneficial to the overall TSF water quality.

Further detailed design development and characterized waste rock placement sequencing can improve the outer layer composition of the facilities and subsequently improve water quality. A balanced surface water quality model must be established during the next project phase that will improve input in water treatment designs. Base line water quality data must continue to be collected to provide longitudinal baseline information for discharge comparisons.

20.9.8 WATER TREATMENT INFRASTRUCTURE

The Project will require four two (2) contact water (effluent) treatment plants, as well as small treatment plants to provide potable water for human consumption and for domestic waste water.

20.9.8.1 Contact Water Treatment

The contact water flows and quality predictions described in this section has been evaluated and two (2) suitable water treatment plants have been designed and estimated by DRA. These plants will be located at TSF embankment area (on the administrative terrace) and near the mine dewatering pond to the west of the open pit mine.

Both plants are configured with a pre-treatment step comprising of gypsum reactors with lime dosing followed by clarifiers to precipitate salts and heavy metals in preparation for Ultra Fine Reverse Osmosis (RO) filtration to remove fine particles. The RO discharge product is clean water that will be sampled, tested and gravity discharged to the environment in compliance with the discharge specification. The RO reject stream will be passed through a second stage reactor for further sulphate removal, combined with other reject streams and returned to the TSF for tailings entrainment and further treatment. The combined stream pumped to the TSF is at 150% saturation and will assist in diluting the predicted sulphate concentration in the TSF.

20.9.8.2 Potable Water Treatment

A suitably sized potable water treatment plant will be established on the process plant terrace to treat borehole water to potable quality for human consumption.

20.9.8.3 *Domestic Water Treatment*

A domestic and sewage water treatment plant has been allowed for on the process terrace with a buffer tank to accommodate end-of-shift surges in domestic water usage. Treated water will be discharged to the TSF. Solid discharge sludge waste will be naturally dried in drying beds and disposed of in the topsoil stockpile.

20.10 Mine Closure and Reclamation

The limited operational lifespan of the Project requires consideration of closure early in the mine life cycle. The Project's conceptual closure approach is to rehabilitate the mine site so that it is physically and chemically stable and compatible with the intended future land use. The current closure aim is to restore accessible land to pre-mining conditions and to minimise or eliminate long term active aftercare activities such as water treatment requirements.

The closure activities for the mine site elements are discussed in this section. The current conceptual closure plan described here is at a FS level. A formal mine closure plan will be prepared as part of the next project phase and updated periodically throughput the mine life. This report will list the currently developed closure concepts that was utilised in the FS. It is recommended to develop a detailed and phased closure plan during the next project phase to improve early rehabilitation of the mine site and reduce contact water surfaces throughout the life of mine.

20.10.1 Open Pit Closure

The mine design and layout is described in Section 16 of the Report.

Initial water quality predictions indicate that the post closure water quality of the open pit will not meet discharge standards without de-oxygenating closure mitigation measures being undertaken. Mitigation measures will involve the establishment of a permanent pit lake that will passivate remaining exposed reactive rock type surfaces over an active closure period currently estimated at 2 years. The open pit is cut into a hillside that does not allow the pit lake to fully cover all exposed surfaces. The natural pit lake level will be elevated through the construction of a dam wall from suitable mine waste material toward maximizing the pit lake closure coverage.

The open pit is developed in four (4) main phases over the LOM as described in Section 16. Exposed pit areas that will not be passivated by the final pit lake will be suitably covered, sealed against oxygen ingress with designed closure covers, and rehabilitated after the completion of phased mining activities, and when it is operationally safe to do so.

The operational phase water treatment plant will remain active on site to treat the pit lake water over an estimated active closure period of 2 years while the pit water quality improves due to passivation. Ongoing water sampling and discharge monitoring will continue during the active closure period.

Upon completion of the active closure activities, successful passivation and proven normalised discharge water quality, the water treatment plant and control pond will be decommissioned and removed. The pit lake dam wall will have an overflow chute design that will be opened and allow natural overflow of the pit lake to commence.

20.10.2 Tailings and Waste Rock Facility 2

The design and development of the TSF and WRF2 is described in Section 18 of this Report with related closure activities noted here.

The preferred closure strategy for the TSF includes a dry closure cover over the exposed tailings and construction of a closure spillway. This closure plan is intended to reduce the water volumes ponding in the tailings impoundment (i.e., storm water will be discharged through the spillway), and reduce rainfall infiltration into the exposed tailings.

The closure spillway channel will be constructed in the left abutment of the dump and designed to safely pass the PMP-24 hours flood. The invert of this spillway will be set at El. 817 m, to leave a 200 m minimum width of tailings beach upstream of the tailings dam to augment the long-term stability of the dam. The closure cover will be mostly built using the materials excavated for construction of the spillway, which are expected to mostly consist of saprolite.

Tailings on the beach could remain unconsolidated and un-trafficable for years after cessation of deposition. This could limit use of mechanical equipment for construction of the closure cover. Alternatively, it is proposed to reclaim the beach by placement of reinforcement geogrids followed by placement of closure fill and revegetation by hydroseeding. Field trials for the reclamation method will be conducted during the operating mine life. Further engineering is required to develop the closure sequence.

The seepage collection ponds will be breached to prevent water storage.

The exterior rockfill slope of the dam is considered acceptable for permanent closure. As part of the waste facility construction, PAG waste rock will be encapsulated within NAG waste rock, in strategic zones within the dump to reduce the potential for ARD in the long-term.

20.10.3 Saprolite Waste Facility and Waste Rock Facility 1

The design and development of the SWD and WRF1 is described in Section 18 of this Report with related closure activities noted here.

The SWD can be progressively closed at the end of construction. Progressive closure activities include revegetation of the exposed downstream slope and top surface. The top surface of the SWD is sloped towards the downstream slope of the dump which promotes runoff towards the TSF. The downstream slope will be erosion resistant as the external face will expose the waste rock construction berms that confine the comingled saprolite/waste-rock layers of the dump.

The long-term stability of the dump should be monitored and reviewed during operations. Further stabilization could require rising the downstream toe berm to a higher elevation. The waste rock mass balances should be reviewed and optimised accordingly.

WRF1 will be covered and revegetated to limit infiltration of rainwater and reduce the potential for ARD in the long-term.

20.10.4 Process Plant and Infrastructure Closure

Process related equipment and infrastructure will be dismantled and sold for remaining value and re-use. Non-saleable equipment and materials will be classified and correctly disposed of. All surface concrete structures will be suitably cleaned of any residual chemicals, demolished up to a suitable depth and disposed of within the WRF1 prior to final dump closure. Compacted soils will be ripped, topsoil introduced and rehabilitated through revegetation.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The Capital Cost Estimate ("Capex") is based on the scope of work as presented in earlier sections of this Report.

The Capex consists of direct and indirect capital costs as well as contingency. Provisions for sustaining capital are also presented.

The Capex is reported in United States Dollars ($).

The Capex includes the material, equipment, labour and freight required for the mine, process facilities, infrastructure and services necessary to support the operation.

The Capex also includes for the estimates developed and provided by external sources.

21.2 Capital Estimate

21.2.1 Estimate Scope, Accuracy and Reference

The Capex includes the material, equipment, labour and freight required for the mine, process facilities, infrastructure and services necessary to support the operation and includes for the estimates developed and provided by external consultants: KCB for tailings storage and waste rock facilities, AOC for access roads, and EngyWorks for offsite power line estimates.

The Capex prepared for this FS is based on a Class 3 type estimate as per the Association for the Advancement of Cost Engineering (AACE) Recommended Practice 47R-11 with a target accuracy of ±15%. Although some individual elements of the Capex may not achieve the target level of accuracy, the overall estimate falls within the parameters of the intended accuracy.

The estimate reference date is August 31, 2021

21.2.2 Capital Estimate Work Breakdown Structure

The Capital Estimate has been developed on utilising a Work Breakdown Structure that is defined in Section 24 of this Report.

21.2.3 Estimate Assumptions

The following are assumptions on which the Capex is based:

- The Capex reflects an Engineering, Procurement and Construction Management (EPCM) type execution wherein an EPCM contractor will provide the design, procurement and construction activities for all aspects of the Project. All sub-contracts would be managed by the EPCM contractor;
- The EPCM contractor will be supplemented with local personnel in the employ of Adventus to bolster the construction execution team and facilitate knowledge and skills transfer toward bolstering local employment for the operational phase.
- Where deemed possible, the Capex estimate is aimed toward maximizing spending on Ecuadorian contracts, equipment, and materials, however no minimal requirement or limitation with respect to local content in terms of labour, materials, equipment and economic impact has been adhered to.
- No restriction to site at any time during execution of the Project is accounted for in the execution schedule.
- The Capex is based on a continuous execution from time of EPCM contractor contract award. Delays in terms of owner's financing charges or owner's permitting delays may result in extended schedules and additional costs.
- No delays as a consequence of labour disputes, social unrest or other community related matters has been allowed for.
- The Capex assumes no sub-surface obstructions for excavation activities to be performed during the construction. The Capex assumes 95% soft and medium material excavation and 5% of hard rock (blasting) excavation based on available geotechnical data and site visit observations.
- Efforts have been made to source suitable construction and building materials from within the open pit limits. Where there is a requirement for select fill it can be produced with a minimum of screening and washing on site. The process modular crushing plant will be procured early on along with additional screening and washing equipment to support the construction phase with aggregate production using excavated material from the mine pit.
- The Capex is based on the milestone schedule presented in the Report involving an 18- month construction period that excludes early works development.
- Adventus has purchased a 80-100 man construction camp that is intended to accommodate contractor personnel during construction. Overflow capacity will be provided by construction contractors who confirmed that they can find or erect local accommodation and have included such costs in their unit pricing. Allowances have been made for internationally based EPCM, vendor representatives and owners team's travelling and accommodation costs in locally available hotels and guesthouses, supplemented with in-town rented accommodations.

21.2.4 ESTIMATE EXCLUSIONS

Unless specifically included in the Owner's Cost (21.2.12), the Capex excludes allowances for the following:

- Escalation prior to and during construction;
- Interest during construction;
- Schedule delays exceeding two (2) weeks and associated costs;
- Scope changes;
- Unidentified ground conditions;
- Extraordinary climatic events;
- Force majeure;
- Labour disputes;
- Construction Insurance, bonding, permits and legal costs;
- Receipt of information beyond the control of EPCM contractors;
- Schedule recovery or acceleration;
- Cost of financing;
- All duties and taxes, expenditures to date (sunk costs), and research and exploration drilling were excluded from the capital cost. Applicable in country VAT payments during the construction period have been considered as part of the Capex.
- Adventus aims to commence an early works package to progress critical project items toward preparation for construction commencement and project risk reduction. The early works package has been removed from the Initial Capex and included as part of Development Capital phase which will become capitalised for the production phase.

21.2.5 ESTIMATE STRUCTURE

The Capex has been structured into the following major categories:

- Direct Costs;
- Indirect Costs;
- Contingencies;
- Owner's Costs.

21.2.5.1 *Direct Costs*

Direct costs are those expenditures that include supply of equipment and materials and construction labour at site.

21.2.5.2 *Indirect Costs*

Indirect costs are those expenditures covering temporary construction facilities, transport of equipment and materials to site plus engineering, procurement, and construction management (EPCM) services together with the supervision of and commissioning of the works.

21.2.6 Estimation Methodology

The Capex has been assembled on an electronic spreadsheet using the following general methods of calculation.

21.2.6.1 *Quantity Development*

Quantities were developed based on the design quantity takeoffs and supplemented by estimates or allowances.

The following engineering documents were produced to derive estimate material quantities:

- Site Plot Plans;
- Mechanical Equipment List;
- Electrical Equipment List;
- Process Flow Diagrams;
- Preliminary P&IDs;
- Plant and Infrastructure Layout Drawings;
- Electrical Single Line Diagrams;
- Sketches.

21.2.7 Productivity and Construction Man-Hours

The base unit man-hours are those for standard North American construction (Gulf units). Productivity factors were applied to the base unit man-hour for the mechanical equipment, Electrical, Instrumentation and controls (EC&I) commodities. The productivity factors were based on the quality of workers, work hours per week, and complexity of the work. Other factors considered were weather, travel and the turnover of workers. The productivity factors are based on DRA database benchmarking for similar projects.

For this Project, a productivity factor of 100% has been applied to these norms to reflect the estimated hours considered applicable in this region of South America. In other words, the estimated Gulf unit hours are multiplied by a factor of 2 to estimate actual construction hours.

Please note that any productivity losses due to labour market, schedule requirements (fast track), boom period or industrial climate have not been taken into consideration.

21.2.8 Direct Costs, Estimate and Scope Basis

The following sub-sections summarise the estimation derivation and basis adopted for the Capex.

21.2.8.1 Bulk Earthworks

Earthwork rates were selected following adjudication of solicited tenders and clarifications with potential contractors. In-country opinions were obtained to support the selection of applied rates for the estimate.

The project requires a source of suitable construction material within the project site. The open pit area is deemed the most suitable area for construction material sourcing. The mine design (Section 16) includes for the mining of required construction material. Large waste fill volumes related to facility underdrains and dam wall construction will be bulk hauled from the mine. Aggregate production is envisaged to be performed by the contractor utilising the process plant's mobile crusher configuration that can be augmented with suitable aggregate screening and washing facilities for the construction duration. The early procurement of the crushing plant is important to the project economics.

The early establishment of mine haul roads and waste dump areas will be critical to ensure mine pre-stripping can continue unabated for the duration of the construction period as the critical path item.

21.2.8.2 Detail Earthworks and Concrete

Detail earthworks and concrete rates were provided by local Ecuadorian contractors following a formal tendering process. Concrete rates were selected following an evaluation of the tenders and clarifications with the contractors.

The rates include the cost of operating a concrete batch plant, established at a nearby location.

21.2.8.3 Structural Steel

Structural steel fabrication, detailing and erection rates were received from local contractors. Unit rates were priced per tonne of steel, including shop fabrication, with separate rates for erection on site. Mechanical Equipment

Generally, offshore supplied equipment was quoted FOB port of loading. A separate allowance for freight, transport insurance and import duties have been allowed for. A flat rate of 5% import duties were applied across imported equipment cost. It is assumed that Adventus will be able to negotiate a project importation tax agreement with the Ecuadorian government that reflects the indicated taxation percentage.

The majority of mechanical equipment was quoted by reputable suppliers. Equipment selection and estimate inclusion was jointly agreed between Adventus and DRA.

21.2.8.4 Mechanical (Platework and Tanks)

Quoted prices were obtained for a range of carbon steel, alloy steel and lined tanks and platework items to establish unit prices per tonne.

Prices for mechanical bulks including chutes, ducting and insulation were based on locally provided and available plate purchasing and available steel standards, detailing, fabrication and installation rates.

21.2.8.5 Piping

The cost for process plant piping was factored as a percentage of the mechanical equipment cost based on DRA's historical data and experience.

Overland piping was quantified with material take-offs and priced using rates from international suppliers and fabricators of piping systems.

21.2.8.6 Electrical

The electrical equipment list and the single line diagrams were used to form the basis for the electrical equipment estimate.

Power and control cables were measured and priced using adjudicated rates obtained from in-country contractors.

21.2.8.7 Instrumentation and Controls

Quotations for process plant Instrumentation & Controls were obtained from suppliers or based on DRA historical data. Cabling was measured and obtained in-country quotation rates applied.

21.2.8.8 Plant Bulk Earthworks, Drainage, Storage Ponds and Access Roads

Plant bulk earthworks are based on necessary cut-to-fill and embankment construction suitable for local conditions. Drainage consists of earthen side drains and under road concrete culverts. Water storage ponds include for HDPE liners. Roads are constructed to regional standards and not paved. The infrastructure earthworks involve the removal of Saprolite overburden that will be placed into the Saprolite waste facility utilised for mine waste. The waste facility has been suitability sized for both mining and earthwork waste requirements.

21.2.9 INDIRECT ESTIMATE

21.2.9.1 Contractor Indirect Costs

Contractor's indirect costs include all contractors' overheads such as contractual requirements (safety, sureties, insurance, etc.), the site establishment and the removal thereof, and company and head office overheads.

The contractor field indirect costs also include:

- Construction temporary facilities which include:
  - Offices, mess halls, lunchrooms, bathrooms, first aid, showers, laundry;
  - Warehouses & yards, shelters, etc.
  - Power generation
  - Aggregate & concrete batch plants;
  - Water systems;
  - Temporary power;
  - Maintenance & clean up;
  - Personnel transportation.
- Temporary services:
  - Survey;
  - Inspection;
  - Quality Controls;

  - Medical Services;
  - Additional Security;
  - Heating;
  - Fuel supply;
  - Fuel stations;
  - Water;
  - Sewage and Waste Disposal;
  - Third Party Consultants;
  - Warehousing.
- Construction Equipment
  - Cranes;
  - Vehicles;
  - Mobile Equipment;
  - Specialty Equipment.

The contractor indirects were broken down into Fixed Charge, Time Related and Other categories to estimate the indirect costs. Percentage ratios were obtained across averaged quotations to best reflect average indirects. These percentages were applied to the final direct capital cost to derive at contract indirect costs for the Estimate.

Unit prices submitted by contractors are "all-in" rates, which include contractor's construction equipment, operators, insurance, overhead and profit.

21.2.10 Capital Spares and Inventory

Capital Spares and Inventory include:

a. Capital Spares

Capital Spares for major mechanical equipment were quoted by vendors/suppliers. For minor mechanical equipment, an allowance for spare parts is based on 5% of equipment purchase price.

b. Operational Spares

One-year Operational Spares for major mechanical equipment was quoted by vendors/suppliers. For minor mechanical equipment, an allowance for operational spare parts is based on 6% of equipment purchase price.

c. Critical Spares

Critical Spares for major mechanical equipment was quoted by vendors/suppliers. For minor mechanical equipment, an allowance for critical spare parts is based on 3% of equipment purchase price.

d. Commissioning Spares

Commissioning Spares for major mechanical equipment was quoted by vendors/suppliers. For minor mechanical equipment, an allowance for critical spare parts is based on 2 % of equipment purchase price.

e. Initial Fills

An allowance for one month of initial first fills was based on quantities developed by the Process Group with quoted unit rates applied.

The estimate includes the cost of the required material quantities for the first start-up fill of the following:

- Mill liners, ball charges;
- Chemical reagents;
- Quicklime;
- Fuels;
- Lubricants;
- Oils;
- Other minor items

21.2.11 Freight and Logistics

The cost for sea transport including insurance and demurrage cost was factored as a percentage of the overseas equipment cost based on DRA's historical data and experience. In this case, a factor of between 2% and 4% was applied qualitatively following and initial assessment of equipment size and weight compared to quoted equipment cost. This review was performed to ensure high value equipment (e.g. regrind mills) transport cost is not overstated.

The cost for in-land transport was factored as a percentage of the equipment cost based on DRA's historical data and experience. In this case, a factor of 5% was applied to the offshore equipment and bulk materials cost depending on the country of origin.

The freight and logistics costs include for brokerage and agent fees, warehouse services, and import. It is assumed that there will be minimal requirement for air freighted items to site.

The site is located 180 km from the major port city of Guayquill with tarred roads up to El Congreso. The planned early works road upgrades and access roads will further ease transportation to site. No major transport limitations are expected, and allowance for specialised transport has not been included in the FS estimate.

21.2.12 Owner's Costs

The Owner's Costs was compiled by DRA with assistance from Adventus.

The Owner's Costs include, but are not limited to:

- Definition drilling, assaying and related reports and models;
- Owner's Project administration team (allowance);
- Owner's pre-production and mine development;
- Health safety and security;
- Process rights, royalties, license fees, technology fees and the like (addressed in the financial model);
- Training and recruiting of plant operating personnel (Operational Readiness);
- Working capital (addressed in the economic evaluation);
- Start-up and commissioning costs (Adventus commissioning team);
- Community relations;
- Emergency response;
- Human resources;
- Information technology;
- Insurances;
- Legal;
- Site security;
- Site maintenance and mobile equipment;
- Import duties;
- Taxes (addressed as part of economic evaluation);
- VAT (addressed as part of economic evaluation);
- Site Clinic.

21.2.13 CONTINGENCY

Contingency was included in this Capex to cover items which are included in the scope of work, but which cannot be adequately defined at this time due to lack of accurate detailed design information.

Contingency covers uncertainty in the estimated quantities and unit prices for labour, equipment and materials contained within the scope of work.

Contingency, as defined herein, is not intended to cover items such items as labour disputes, change in scope, or price escalation.

A qualitative contingency of 10% was applied to the Initial Capital cost.

21.2.14 DESIGN GROWTH ALLOWANCES

Design growth allowances have not been applied to the estimated quantities.

21.2.15 EXCHANGE RATES

The base currency for this Report is the US Dollar (USD). For information purposes, the following is a list of currencies that could potentially be used, along with the conversion exchange rates based on an average of the previous 90 days prior to finalisation of the Capex.

The currency exchange rates shown in Table 21.1 will form the basis for any currency exchanges used in the Capex.

Table 21.1 – Currency Conversion Rates

Currency*	Code Name	US Dollar equivalent to 1.00 Currency	Currency Equivalent to 1.00 USD
USD	US Dollar	1.00000	1.0000
CAD	Canadian Dollar	0.76923	1.3000
AUD	Australian Dollar	0.76150	1.3132
EUR	Euro	1.18130	0.8465
GBP	British Pound	1.31579	0.7600
CNY	Chinese Yuan	0.14725	6.7912
ZAR	South African Rand	0.06049	16.5306

* Not all presented currency conversions are necessarily used in the Capex estimate

21.2.16 Summary of the Capital Estimate

21.2.16.1 Project Capital Estimate

The overall project Capex estimate is indicated in Table 21.4. The scope of the FS Capex includes open pit mine, waste facilities, ore handling, processing plant and on-site support infrastructure. Capex expenditure in different phases of the project development (Exploration), Initial Capital and production (Sustaining and Closure) is indicated with further explanations provided.

The Initial Capital phase is indicated as the main project construction expenditure phase, commencing upon receipt of all relevant construction and environmental permits and was the main focus of the FS estimate. The initial expenditure is planned as an 18-month period prior to commencement of production. Capital expenditure prior to the Initial Capital phase is considered exploration and development capital which will be capitalized; this cost includes for a planned early works package planned prior to commencement of the Initial Capital phase. Sustaining capex commences upon production of concentrate and continues throughout the mine life. Closure capital is planned toward the end of mine life in the final production year with capital cost for closure operations past the mine life. Table 21.2 presents a summary of the Project Capex cost and phasing.

Table 21.2 – Project Capex Summary by Major Area and Phase (US $M)

Description	Pre-Construction Capital	Initial Capital	Sustaining	Closure	LOM Total Capex
Direct					
Pre-construction Early Works Program	22				22
Mining - Open Pit		52			52
Processing Plant		67	6	15	89
Mining Site - Infrastructure		2			2
Tailings / Waste Rock		34	12	9	54
Surface Infrastructure - Buildings, Roads, Water Treatment		12	8	2	22
Subtotal Direct	**22**	**167**	**26**	**26**	**241**
Indirect		-			
Construction Indirect		18			18
Freight & Logistics, Plant Equipment		7			7
Owner's Costs, EPCM, Taxes		10			10
Project Contingency		22	3	8	32
Refundable VAT on Initial Capital		25			25
Salvage Value		-		(10)	(10)
Subtotal Indirect	**0**	**82**	**3**	**-2**	**82**
Total	**22**	**248**	**29**	**24**	**323**

21.2.16.2 *Exploration and development Capital*

Adventus has indicated the sunk exploration expenditure up to the effective date of this report. Further expenditure is planned as part of an early works package that will commence prior to the main project construction planned as the Initial Capital phase.

The early works package is planned to progress technical design and further recommended test work in anticipation of construction commencement. The early works package further plans to commence with detailed engineering design, including purchasing of certified information, to progress certain engineering designs. The aim of the early works package is to prepare the project site, progress current level of design, and reduce project schedule and cost risks through further definition.

Table 21.3 presents the sunk and planned exploration and development capital expenditure.

Table 21.3 – Exploration and Development Capex

Major Area	Exploration Cost (US $M)
Early Works – Engineering Plant	9.2
Early Works – Engineering TSF	2.0
Early Works - Geotechnical surveying	0.4
Early Works - Long Lead Equipment & Certified Information	1.0
Early Works - Access Road Upgrades	2.4
Early Works - Existing Road – Bridges Upgrade	3.4
Early Works - Earthworks on Site – Roads & Platforms	4.0
Total	**22.3**

21.2.16.3 *Initial Capital*

The Initial Capital depicts the main expenditure of the project over an 18-month period prior to commencement of concentrate production. Mine pre-stripping, major earthworks, infrastructure development and equipment purchasing is planned during this period.

Table 21.4 presents the Initial Capital breakdown.

Table 21.4 – Initial Capex Summary by Major Area (USD)

Major Area	Initial Capex (US $M)
Mining - Open Pit	51.8
Processing Plant	56.0
Mining Site - Surface Infrastructure	1.5
Tailings, Reclaim, Water Treatment	29.2
Process Site - Surface Infrastructure	21.1
Sub-Total Direct Costs	**159.7**
Indirect	53.0
Owner's Costs	13.1
Project Contingency	21.8
Sub-Total Indirect Costs	**87.9**
Grand Total	**247.6**

21.2.17 INITIAL CAPITAL PHASE: DIRECT CAPITAL ESTIMATE

All direct capital costs have been estimated using a cost structure developed for labour and materials.

21.2.17.1 Mining Direct Capex

Table 21.5 depicts the initial capital costs for the open pit mine development area. The open pit mining operation will be started during the pre-production phase and then continue into the initial operation phase. The pre-production period involves the establishment of the mine haul roads and other infrastructure to allow mining to commence. The pricing is based on obtained in-country and international contractor quotes.

Table 21.5 – Capex Summary: Open Pit Mining by Minor Area

Minor Area	Total Cost (US $M)
Mine Pre-Stripping	46.0
Access Road 1st stage	1.9
Mining Temporary Haul Roads	2.5
Temporary Haul Roads	1.0
Explosive Access Road	0.1
Services – Temporary Signs	0.0
Mine Facilities & Services (power, offices, first aid)	0.2
Total Open Pit Mine Development	**51.8**

21.2.17.2 Mine Surface Infrastructure Direct Capex

Table 21.5 depicts the initial capital costs for the mine site's surface infrastructure. The workshop and warehouse costs include for a full complement of the necessary equipment and office furniture necessary for the operation of the facilities. All administration facilities are constructed to local standards. Each facility includes office equipment, lockers, shelves and other items to support the particular activity within each facility. The mine administration will be co-located at the central administration terrace. As part of early works preparation, Adventus has purchased portable buildings that will be erected for use as the administration building.

Table 21.6 – Capex Summary: Mine Surface Infrastructure

Description	Total Cost (US $M)
Truck Shop	0.02
Earthworks Design Growth	0.91
Storm Water/Wash Bay/Pollution Control	0.58
Total Surface Infrastructure	**1.52**

21.2.17.3 Process Direct Capex

Table 21.7 depicts the initial capital costs for the process plant facilities. This area covers the process plant feed circuit, crushed ore stockpile, the process plant including grinding, flotation, dewatering, reagents and utilities.

Table 21.7 – Capex Summary: Process Plant Facilities

Description	Total Cost (US $M)
Crushing & Stockpiling	3.4
Grinding	2.4
Cleaner Flotation	0.7
Tailings Thickening & Handling	2.2
Concentrate Handling & Process Plant Services & Utilities	7.1
Reagent Preparation & Grinding Media	8.9
3Other Services	4.9
Total Process Plant	56.0

21.2.17.4 Process Surface Infrastructure Direct Capex

Table 21.8 details the summary for the surface infrastructure at the process site.

Table 21.8 – Capex Summary: Process Site Surface Infrastructure

Description	Total Cost (US $M)
Site Development and Roads	4.4
Site Power Distribution	4.0
Admin Facilities	1.1
Maintenance Facilities	4.0
On-Site Non-Process Facilities	0.8
On-Site Bulk Storage	0.2
On-Site Communications	1.3
Site Power Supply & Transmission (69kV)	5.5
Total Process Site – Surface Infrastructure	**21.1**

21.2.17.5 Waste Management Direct Capex

Table 21.9 presents the summary for waste management Capex.

Table 21.9 – Capex Summary: Waste Management

Description	Total Cost (US $M)
Tailings and return pipelines for TSF	1.2
Tailings Storage Facility and WRF2	8.6
Saprolite Facility Construction (SWD)	14.2
Waste Rock Facility 1 (WRF1)	1.2
Industrial Effluent Treatment	4.0
Total Process Site – Waste Management	**29.2**

The waste facilities will be constructed and expanded during the Initial Capital phase to allow for pre-stripping to continue as well as prepare for process plant tailings entrainment.

21.2.18 Initial Capital Phase: Indirect Capital Estimate

The indirect costs include engineering design, procurement and construction management activities (EPCM), vendor representatives, spare parts and first fills, contractor's indirect costs as indicated in Table 21.10.

Table 21.10 – Capex Summary: Indirect Costs

Description	Total Cost (US $M)
Contractor Indirect	22.9
Spares, Fills & Inventory	3.8
Commissioning	1.3
Freight / Traffic Warehouse Services & Logistics	4.0
Total Indirect Costs	**30.7**

21.2.19 Initial Capital Phase: Owner's Costs

The owner's costs were compiled by DRA with assistance from Adventus. The owner's costs include, but are not limited to:

- Preproduction labour costs including process, maintenance and administration labour. Mining & geology labour is estimated in the mine Capex;
- Construction insurance;
- Legal costs for contracting;
- Land acquisition
- Taxes – Import and VAT
- Early Works package

Table 21.11 – Capex Summary: Owner's Costs

WBS - Major Area	Total Cost (US $M)
Initial Capital – Owners Team Cost	5.0
Initial Capital - Operational Readiness	3.0
Initial Capital - Taxes & Duties	2.6
Initial Capital - Refundable VAT	24.8
Early Works - EP	9.2
Early Works - EP TSF	2.0

WBS - Major Area	Total Cost (US $M)
Early Works - Geotechnical	0.4
Early Works - Long Lead Equipment & Certified Information	1.0
Early Works - Access Road Upgrades	2.4
Early Works - Existing Road - Bridges	3.4
Early Works - Earthworks on Site	4.0
Total Owner's Costs	**57.8**

21.2.20 PRODUCTION PHASE: SUSTAINING COSTS

For the mining production phase, sustaining capital for equipment replacement and other relevant costs is included in the mining contractor scope and hence forms part of the mining Opex cost.

The sustaining capital requirements for the process plant includes for the purchase of spare parts for equipment, and for replacement of equipment when required and was derived as a percentage of the initial equipment cost.

The tailings and waste facility sustaining costs include the expansion of the tailings storage facility and waste facilities over the life of mine. The costs are based on estimated quantities and contractor rates equivalent to those obtained for the initial Capex. It is envisaged that tailings and facility expansion will be completed by the mining contractor for hauling and dumping, and a specialised civil contractor for detailed earthworks and liner expansions. The estimate includes for direct and indirect costs.

Access roads to site include public and private road portions that may require continuous maintenance or repairs during operations due to traffic usage and expected annual heavy rainfall deteriorations. An amount for $250k per year has been allowed to support this ongoing road maintenance work.

The mine pit will gradually expand in phases during the life of mine. The increased rainfall contact footprint will require additional water treatment capacity to be added during year 5 of operation. A $2m allowance has been made for this cost based on initial water treatment quotations.

The open pit operations will hinder current public road access through the mine property. Advents has planned for a new bypass road to be constructed during the first operational year of the mine.

Contingency on sustaining capital was applied on a qualitative basis at 10% of Sustaining capex.

The sustaining capital costs are tabulated in Table 21.12 and are distributed over the LOM as depicted in Section 22.

Table 21.12 – Sustaining Capital Cost Estimate over LOM

Description	Total Sustaining Cost (US $M)
Process Equipment	6.4
Tailings Facility expansion	5.7
Saprolite Facility expansion	4.0
Waste Rock Facility expansion	2.1
Access Road – annual maintenance	2.3
Water Treatment Plant 2 - Pit	2.0
Bypass Road Allowance	4.0
Contingency	2.6
Grand Total	**29.1**

21.2.21 CLOSURE PHASE: CLOSURE CAPITAL

Closure is planned to commence during the final year of operation and continue for the following 2-year period after the cession of commercial mining operations. Major works include the capping and closure of the tailings and waste rock facilities, including the construction of a drainage chute to control the TSF dam level.

It is assumed that the process equipment will be sold for at the end of the Project. The relatively short mine life indicates potential residual value in the larger process equipment. The anticipated gain from the sale of the equipment is estimated at $10m.

The closure of the process plant, site infrastructure and terraces has been quantified and rates applied to determine the associated estimated closure costs.

TSF and pit closure requires water treatment for a period of up to 2 years post-closure to stabilise and neutralise the TSF dam and pit lake water quality. The cost for such water treatment and operations has been allowed for. This process will require the main electrical supply to remain intact on site for the duration of closure.

A further contingency cost has been included for additional unanticipated closure costs equivalent to 30%.

The provisional cost estimate for the rehabilitation and closure of the mine, based on the rehabilitation and closure plan framework is provided in Table 21.13.

Table 21.13 – Summary Estimate of Financial Provision Required for Closure

Description	**Total** (US $M)
TSF & WRF	10.9
Process & Roads	4.8
Mine Pit	4.0
Water Treatment Post Closure	6.5
Contingency	7.9
Salvage	-10.0
Total	**24.1**

21.2.22 Initial Capital Phase: Contingency

Contingency was included in the Capex to allow for items which are included in the scope of work as described in this Report, but which cannot be adequately defined at this time due to lack of accurate detailed design information. Contingency covers uncertainty in the estimated quantities and unit prices for labour, equipment and materials contained within the scope of work. Contingency, as defined herein, is not intended to cover items such items as labour disputes, change in scope, or price escalation.

Quotations were obtained for various equipment and services, the following contingency allowances were applied to the Capex, as outlined in Table 21.14.

Table 21.14 – Contingency Values

Description	**Total** (US $M)
Contingency for Mining	4.6
Contingency for Bulk Earthworks	3.9
Contingency for Civils & Concrete	0.7
Contingency for SMPP	4.2
Contingency for Architectural Building	0.7
Contingency for EC&I	1.8
Contingency for Contractor Indirect	1.6
Contingency for Spares	0.5
Contingency for External Consultants	3.8
Total Contingency	**21.8**

21.3 Operating Cost Estimate

This Section describes the basis of estimate and approach taken in calculating the operating costs for the Project..

The Operating Cost Estimate (Opex) is presented in United States Dollars (USD). DRA developed these operating costs in conjunction with Adventus. The estimate includes mining, processing, and general and administration (G&A). The estimate has an accuracy in line with the stated study class (Section 21.2.1)

21.3.1 Basis of Opex Estimate

The Opex estimate was developed to support the operating term of the project with required personnel, equipment, infrastructure and services. The Opex estimate was developed following the completion of mine, process and infrastructure designs that provided the necessary consumable, personnel and service requirements for the project.

The Opex estimate is based on designed consumable rates, derived personnel estimates, and received contractor and supplier quotations for both mining and process operations. Both process and mining personnel and salary requirements were estimated based on DRA's experience on similar projects.

Adventus and DRA jointly developed the G&A cost estimate for the project indicating allowances for expected annual expenditures and determining required support personnel complement. Labour rates were confirmed by Adventus utilising their partner company (Curimining, a subsidiary of Salazar Resources) that currently employs approximately 150 full-time equivalent personnel in Ecuador for the exploration phase of the project.

The following are examples of cost items specifically excluded from the Opex Estimate:

- Refundable Value Added Tax (VAT);
- Project financing and interest charges.
- Capital costs prior commencement of production is excluded from the Opex calculations. This includes operations related costs such as personnel acquisition, training and other operational readiness costs. An operational readiness allowance has been made as part of the initial Capex estimate to cover these costs.

21.3.2 Opex Estimate Summary

The Opex is estimated at $364m over the life of mine or $56.21/t of ore processed, during ten years of operation. Table 21.15 summarises the Opex by area over the LOM of the project.

Table 21.15 – Opex Summary by Area

Description by Area	Average Annual Costs (US $M)	Total Cost LOM (US $M)	Cost / t moved (US $/t)	Cost / t ore processed (US $/t)	Copper Equivalent cost $/lb Copper Eq.[3]
Mining[1]	16	$157	3.35	24.53	0.34
Process	**15**	**150**		**22.74**	**0.32**
Process Plant	13	125		19.08	0.27
Water Treatment	2	24		3.66	0.05
G&A	6	56		8.95	0.12
Total[2]	**36**	**364**		**56.21**	**0.77**

1 A portion of owners team mining is captured under G&A totalling $3.2M over LOM, equivalent to $0.06/t moved ($0.50/t milled) The mining cost is presented in US $/t moved.
2 Figures may not total due to rounding
3 *CuEq is calculated as follows:*
(Payable Metals NSR Ag,Zn,Pb,Au, Ag)/(Payable Metals NSR Cu) (2.205 lb/kg)*(Payable Copper/lb)*

21.3.3 Open Pit Mine Operating Costs

The mine Opex was developed based on quotes received from contractors, who developed their quotes based on the tonnage and haulage distance requirements as defined by the mine plan in Section 16 of the Report.

21.3.3.1 Mine Cost Summary

Table 21.16 shows the total mining Opex incurred over the LOM as well as a cost per tonne mined.

Table 21.16 – Open Pit Mine Opex

Description	Production		
	Total Cost (US $)	**$/t Moved**	**Total Cost (%)**
Owner Equipment	829,000	0.02	0.52
Owner Personnel	9,560,000	0.20	7.94
Contractor Mining & Haulage	64,041,000	1.36	39.85
Contractor Drill & Blast	46,031,000	0.98	28.65
Contractor Dewatering	944,000	0.02	0.59
Contractor Fixed Costs	17,284,000	0.37	10.76
Fuel	14,116,000	0.30	8.78
Technical Services Equipment	1,229,000	0.03	0.76
Road Maintenance	3,449,000	0.07	2.15
Total	**157,484,000**	**3.35**	**100.00**

1. Figures may not add due to rounding
2. Contractor costs include mobilization, demobilization, facilities, clearing and grubbing
3. A portion of owners mining team is captured under G&A totaling $3.2M over LOM, equivalent to $0.06/t moved ($0.50/t milled)

21.3.3.2 Open Pit Mine Owner Costs

Table 21.17 presents a summary of the costs incurred by Adventus over the life of mine. A manpower schedule based on the mine requirements is presented in Section 16.

Table 21.17 – Open Pit Mine Summary of Owner Costs

Description	Production		
	Total Cost (US $)	$/t Moved	Total Cost (%)
Owner Equipment	829,000	0.02	3.22
Owner Personnel	9,560,000	0.20	37.15
Fuel	14,116,000	0.30	54.85
Technical Services Equipment	1,229,000	0.03	4.78
Total	**25,735,000**	**0.55**	**100.00**

1. Figures may not add due to rounding
2. A portion of owners mining team is captured under G&A totaling $3.2M over LOM, equivalent to $0.06/t moved (0.50/t milled).

21.3.3.3 Open Pit Mine Costs by Material Type

Table 21.18 presents a summary of the mining Opex by material type.

Table 21.18 – Mining Costs by Material Type

Description	Production		
	Total Cost (US $)	$/t Moved	Total Cost (%)
Ore Mining	26,840,000	4.15	17.04
Ore Rehandling	1,811,000	1.04	1.15
Overburden Mining	535,000	2.27	0.34
Saprolite Mining	10,160,000	2.26	6.45
Andesite Mining	36,985,000	3.50	23.49
General Waste Mining	81,152,000	2.13	51.53
Total Waste Mining	*128,832,000*	*3.31*	*81.81*
Total	**157,484,000**	**3.35**	**100.00**

1. Figures may not add due to rounding.
2. A portion of owners mining team is captured under G&A totaling $3.2M over LOM, equivalent to $0.06/t moved (0.50/t milled).

21.3.4 Process Plant Operating Costs

The breakdown of process plant operating costs is broken down into appropriate categories and these costs are summarised in Table 21.19 and described in the following sections.

Table 21.19 – Summary of Estimated Annual Process Plant Opex

Operating Cost	Average Annual Costs (US$M)	Total Cost LOM (US$ M)	Cost / Ore Processed (US$/t)	Total (%)
Manpower	1.0	9.7	$1.49	7.81
Power	3.3	33.1	$5.12	26.83
Reagent Consumption	4.6	45.7	$7.06	37.00
Grinding Media and Liners	1.1	11.0	$1.70	8.91
Maintenance	1.4	13.7	$2.12	11.11
Laboratory Services	0.3	3.2	$0.50	2.62
Mobile Equipment	0.7	7.1	$1.09	5.71
Total Operating Costs[1]	**12.4**	**123.6**	**$19.08**	**100.00**

1. Figures may not add due to rounding.

21.3.4.1 Process Plant Manpower Costs

The total operational manpower averages 54 employees for the project with 10 salaried employees and 44 hourly employees. Personnel incorporates requirements for plant operation, management, laboratory, and maintenance. Table 21.20 depicts the manpower for the process facility. The total annual cost for process plant manpower is estimated at $ 0.9M.

Table 21.20 – Plant Manpower Opex

Description	Number of Employees	Cost (M USD/a)
Salaried	10	0.78
Operators	30	0.35
Technicians	2	0.18
Maintenance	12	0.18
Total	**54**	**1.49**

21.3.4.2 Process Plant Power Costs

Power is required to operate equipment in the processing plant such as conveyors, crushers, mills, screens, pumps, agitators, plant services (compressed air and water), etc. The unit cost of grid-based electricity was established at US$ 0.11/kWh. The unit cost was derived from pervious in country experience and was not independently verified.

Power consumption was determined based on the total installed power (excluding standby equipment) derived from the Mechanical Equipment List. The power draw was based on the average power utilisation of each motor. The estimated electrical operating costs is based on the plant operating 24 hours per day, 7 days per week, with a run time of 70% for the crushing and stockpiling area, and 92% for the grinding and flotation areas as an operating percentage. The total average annual cost for process plant electrical power is estimated at $3.3M.

21.3.4.3 Process Plant Reagent Consumption Costs

Reagent consumption rates were estimated based on metallurgical test work results and adjusted to commercial plant usage rates with the correction (reduction) factors based on operations experience. Reagent unit costs were obtained from vendors and through benchmarking for similar projects performed by DRA. Details about reagents consumptions are explained in Section 13. The total average annual cost for process plant reagent consumption is estimated at $4.6M.

21.3.4.4 Process Plant Grinding Media and Liners

The Ball mill will need the addition of steel balls to replace the worn media to maintain the steel load in the mills and to perform proper size reduction on the material. The Regrind mill will require the addition of media for replacement. Consumption of the grinding media is based on the abrasion index, power consumption and experience from similar projects.

The consumption of crusher liners, grinding mill liners, regrind mill liners was obtained from the equipment suppliers and from experience with similar operations. The average annual cost for process plant grinding media and liners is estimated at $1.1M.

21.3.4.5 Process Plant Maintenance Costs

Annual maintenance costs were factored based on the total installed mechanical equipment capital cost using an approximate factor of 7% The total annual cost for process plant maintenance cost is estimated at $1.3M.

21.3.4.6 *Process Plant Laboratory Costs*

The process plant requires a metallurgical lab for sample analysis and metal accounting on site. Adventus has opted for a lab services provider to provide, operate, and maintain the laboratory on site. An allowance for such services has been included in the Opex at a cost of $0.50/t milled totalling $3.2M over Life of Mine. Formal quotations will be solicited during the next project phase.

21.3.4.7 *Process Plant Mobile Equipment*

Two (2) front end loaders will be used to feed ROM ore and Crushed ore to the mill, these costs include labour, fuel and maintenance. The total annual cost for front end loaders is estimated at $0.7M.

21.3.5 *Site Wide Water Treatment Costs*

Water treatment costs include for the operation and maintenance of the water treatment plants for both the open pit and TSF facilities as well as sedimentation ponds, diversion systems and pumps and pipelines. The operating costs excludes closure water treatment costs which is captured under sustaining capital.

A preliminary water treatment plant design was completed upon completion of the initial water quality predictions. The plant design provided the required Opex costs for reagents, power and man power.

An average estimated water treatment cost of $1.13/t milled for the open pit, and $2.54/t milled for the TSF water treatment costs was applied totalling $24.1M over the life of mine.

21.3.6 Site wide General and Administration Costs

The G&A cost for the project was derived in collaboration with Adventus and is presented in Table 21.21. Adventus provided initial annual budgetary estimates related to in-country operational expenditure. DRA developed the G&A personnel complement register and confirmed labour rates with Adventus' partner company Curimining (a subsidiary of Salazar Resources).

The G&A costs account for business operations and sustainability costs associated with support and managerial functions such as insurances, legal services, property and mine leases, local taxes, systems licenses, security, catering, environmental monitoring, and employee relations. The plant location has been taken into account with G&A allowances for employee bus transport and some meals on site.

Corporate costs such as mining leases, insurances, overheads and travel allowances have been allowed for.

Table 21.21: Annual G&A Cost Estimate

Description	Average Annual Costs (US $'000)
Labour	3,412
Corporate Costs	463
Site Costs	1,816
Support Costs	270
Total G&A	**5,961**
Total G&A $/t milled	**8.95**

22 ECONOMIC ANALYSIS

22.1 Basis of Evaluation

DRA has prepared the assessment of the Project on the basis of a discounted cash-flow model, from which net present value (NPV), internal rate of return (IRR), payback, and other measures of the Project's economic viability can be determined. Assessments of NPV are generally accepted within the mining industry as representing the economic value of a project after allowing for the cost of capital invested.

The objective of the FS was to determine the potential economic viability of developing the Project, which consists of an open-pit deposit, ore processing and support facilities to produce polymetallic concentrates.

22.2 Macro Economic Assumptions

22.2.1 METAL PRICES

Adventus has selected the base case metal price projections in Table 19.3, for input to the financial model. This pricing is only utilised for the financial model economic analysis.

Table 22.1 – Base Case Metal Pricing

Element	Unit	Financial Model
Cu	$US/lb	3.50
Zn	$US/lb	1.20
Au	$US/oz	1,700.00
Ag	$US/oz	23.00
Pb	$US/lb	0.95

22.2.2 Net Returns

Three (3) concentrates, namely Copper, Zinc and Lead concentrates will be produced with payables which are detailed in Section 19 of the Report.

22.2.3 Royalties

Applicable royalties for the Project are described in Table 22.2.

Table 22.2 – Applicable Royalties

Royalty	Terms	Comments
Government of Ecuador	4%	Applicable royalty bracket is calculated on gross sales of minerals. Royalties are applied on net sales revenue (Net Smelter Returns).
		Advance Royalty to be paid at the signature of the Exploitation Mining Contract is not applicable for medium sized mining and has not been considered in the FS cashflow. .
Altius Mineral Corporation	2%	Royalties are applied on net sales revenue (Net Smelter Returns) and are deductible for income tax purposes.

22.2.4 Taxes

Adventus retained an in-country tax consultant to the applicable taxes and application throughout the construction and operational phase.

Taxes that are applicable to mining companies under the Ecuadorian tax regime and are described in Table 22.3, such as:

- Standard Corporate Tax;
- Profit Sharing Tax;
- Sovereign Adjustment Tax;
- Value-Added Tax (VAT);
- Conservation Mining Patent;
- Import of Goods;
- Capital Outflow Tax (“COT”).

Table 22.3 – Applicable Taxes

Tax	Terms	Comments
Standard Corporate Tax	25% on taxable income	Companies domiciled in Ecuador are subject to tax on their worldwide income. The income tax is paid annually in April of the following calendar year. The income tax basis is determined by the total taxable income less allowable deductions according to the tax law. Intercompany transactions are subject to certain deductibility limits depending on the nature and concept of the service/charge. Tax losses can be amortised (carry-forward) during the following 5 years, considering an annual amortization cap/limit equivalent to the 25% of each year's taxable profit. The general income tax rate (25%) can be increased up to 28% in case the obligation to disclose the shareholding/ownership structure before the Tax Authority up to the ultimate level. Adventus to progress an investment protection agreement with the Ecuadorian Government with tax stability benefits..
Profit Sharing Tax	15% on taxable income	Based on Curipamba' s medium scale of mining, the worker's portion will be 3% and the State's portion 12%. Tax is levied on the taxable income of the project is deductible for tax purposes.
Sovereign Adjustment Tax	50% cumulative benefit to on a cumulative annual basis	Under the "Sovereign Adjustment" tax, the Ecuadorian constitution requires that the government must receive at least 50% of benefits from non-renewable resource projects. Sovereign Adjustment is only payable when the present value of cumulative company benefits exceeds the present value of cumulative government benefits. The benefits will be calculated annually as the net present value of the cumulative free cash flows of Curipamba since the concession was granted. The benefits to the Government of Ecuador will be calculated as the net present value of the cumulative sum of corporate income tax, royalties, state profit sharing portion (12%), non-recoverable VAT, and previous Sovereign Adjustment payments, if any, over the same period. Based on the base case metal prices used in the financial model presented in this report, the Project will not trigger the Sovereign Adjustment payment.
VAT	0% and 12%	VAT is levied at the rates of either 12% or 0% on the transfer of goods, import of goods, and the rendering of services, as well as on services rendered within the country or imported. Royalties and intangible property, imported or locally paid, are also levied with a 12% VAT. VAT was applied to both capital investment and operating costs items. In Ecuador, exportation of goods and services are levied with 0% VAT, as well as other goods and services specifically included in the tax law. Mining concessionaires are entitled to the refund of VAT paid since January 1, 2018. Once

Tax	Terms	Comments
		exportation of concentrate commences, a maximum of 12% of the value of exports are subject to reimbursement, on a monthly basis. Any amounts not refunded will be available for carry forward for a maximum period of five years. The VAT paid in the acquisition of goods and services in pre-production stages constitutes a tax credit, if the activities that generate future income, meaning as from the date the production starts.
Conservation Mining Patent	10% of UBR times the number of hectares	The concessionaire has to pay an annual fee per mining hectare each March, as follows: ▪ During the initial exploration phase, an amount equivalent to 2.5% of a unified basic remuneration (UBR) (the UBR is $US400 for 2021). ▪ During the period of advanced exploration and economic evaluation an amount equivalent to 5% of the UBR. During the operations (exploitation) phase, an amount equivalent to 10% of the UBR times the number of hectares.
Importation of goods	5% of import value	Payments for the import of raw materials, supplies, and capital goods contained on a list issued by the Tax Policy Committee generally pay duty on importation at a rate of 5%.
	0.5% of import CIF cost	Fodinfa: This is an additional contribution on the import of goods for the Development Fund for Children. The taxable basis is the cost, insurance and freight (CIF) of the importation, and the rate is 0.5%
Capital Outflow Tax (COT)	5% of the amount of currency/capital transferred to abroad.	Payments and transfers to abroad generally incur a 5% COT on most transfers of funds (remittances), including payments for importations of goods and services. 5% COT is also levied over importations of goods and services. Exemptions are available for payments corresponding to dividends distribution and foreign loans/credits, under certain conditions-requirements. COT paid due to the importation of certain capital goods and raw materials constitute income tax credit, which can be offset against the payable income tax or claimed back during the following 5 years. COT has not been considered in the FS cashflow.

22.2.5 DEPRECIATION

Depreciation and amortisation are applied to property, plant and equipment and pre-production and sustaining Capex, respectively commencing in the year after occurrence on the following basis:

a. Depreciation of fixed assets not associated with reserves (Property, Plant, and Equipment):

- In accordance with the nature of the goods, its lifetime and accounting technique;
- For income tax deductibility purposes, depreciation expense can not exceed the following annual percentages:
  - Immovables (except land) – 5%;
  - Mechanical equipment and Process Plant – 10%;
  - Vehicles and mobile equipment 20%;
  - Electronics and computers – 33%.

b. Amortisation:

- Exploration expenses –over at least 5 years;
- Copper Equivalent Units of Production (UOP) over life of mine;
- Preparation and construction of the mine - Development Expenditures – Life-of-Mine under UOP over the total recoverable reserves, or in 20 years;
- Sustaining Capex depreciated each following year over remaining UOP to mine closure;

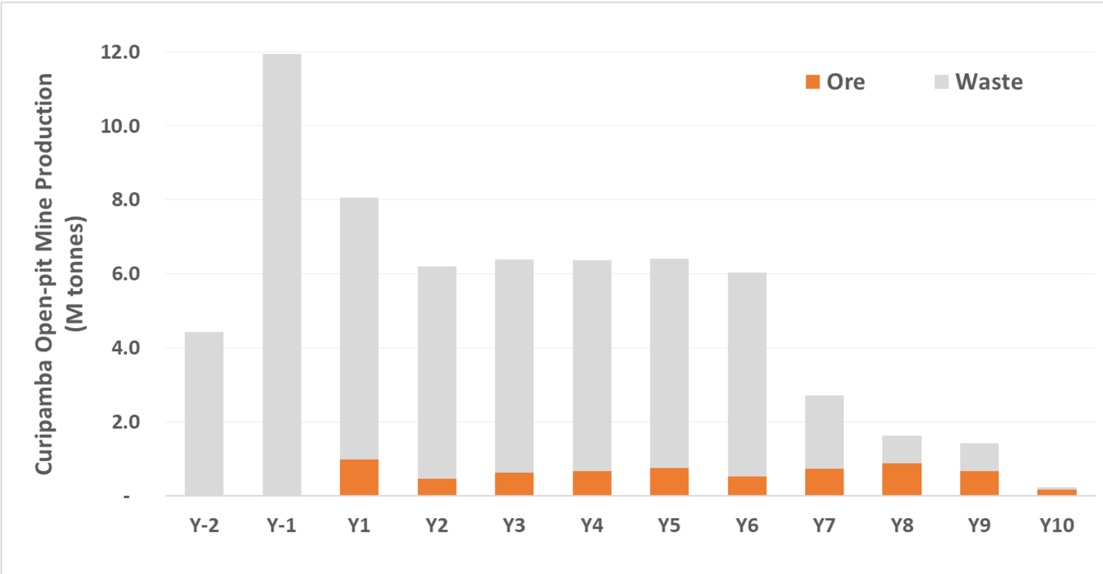
22.3 Technical Economic Assumptions

The technical parameters, production forecasts, and estimates described elsewhere in this Report for the open pit are reflected in the cash-flow model. These inputs to the model are summarised below. The measures used throughout the Report are metric tonnes and troy ounces.

22.3.1 MINE PRODUCTION SCHEDULE

The mine production schedule from the single pit have been aggregated, along with the weighted average costs resulting in the mine schedule depicted in Table 22.4 and Figure 22.1.

Figure 22.1 – Annual Mined Tonnage



Source: DRA,2021

The mining schedule differs from the processing schedule due to stockpiling required for blending and intermittent storage. Ore feed characteristics and average grades reporting to the mill is presented in the Metal production schedule Table 22.4.

Table 22.4 – Metal Production Schedule

Description		Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Total by Ore Type LOM			Total LOM
Metal	Unit	Total	Total	Total	Total	Total	Total	Total	Total	Total	Total	Cu/Zn	High ZN	High Cu	Ores
Au	oz	53,739	54,207	40,500	34,308	37,639	34,218	31,668	26,774	21,646	599	179,517	149,857	5,844	**335,298**
Ag	oz	974,231	1,032,310	1,487,867	1,114,459	1,084,473	993,810	861,708	579,738	473,973	12,842	3,805,646	4,685,600	121,836	**8,615,411**
Cu	tonnes	15,345	12,859	8,775	9,459	15,473	9,978	10,110	8,696	7,697	1,881	65,550	16,731	17,989	**100,274**
Pb	tonnes	2,078	2,969	3,113	2,103	2,640	1,595	1,244	915	677	11	6,421	10,808	106	**17,344**
Zn	tonnes	21,429	26,137	31,875	17,930	24,660	17,124	14,892	13,151	8,852	116	78,542	96,222	1,337	**176,166**

22.3.2 RECOVERIES

Adopted gold recovery rates associated with the ore types are depicted in Table 22.5. The recoveries are applied to the plant feed ore grade to determine recoverable metals and does not include for NSR terms to determine net payable recoveries.

Table 22.5 – Metal Recoveries by Concentrate for Each Ore Type

Recoveries (%)	By Ore Type			Weighted Average LOM
	Cu/Zn	High Zn	High Cu	
Cu Concentrate				
Au	31.4	51.8	29.2	36.3
Ag	27.0	43.3	35.2	33.2
Cu	81.5	82.2	89.2	84.3
Pb	40.0	49.7	52.7	45.0
Zn	6.9	17.8	4.8	11.0
Pb Concentrate				
Au	1.2	1.9	0.0	1.2
Ag	6.0	6.5	0.0	5.6
Cu	4.6	3.1	0.0	2.8
Pb	30.1	30.9	0.0	29.0
Zn	0.8	1.0	0.0	0.8
Zn Concentrate				
Au	14.4	19.1	13.3	15.4
Ag	25.2	20.0	31.2	24.1
Cu	3.6	3.9	3.8	3.7
Pb	10.0	4.8	21.2	8.2
Zn	85.9	74.8	91.4	81.9
Net Metal Recovery to Concentrate(s)				
Au	26.8	18.9	7.26	52.9
Ag	33.1	3.0	6.68	62.8
Cu	49.1	8.58	33.2	90.8
Pb	40.1	38.4	3.62	82.2
Zn	50.4	36.6	6.84	93.8

22.3.3 Operating Costs

The related operating costs (Opex) is discussed in Section 21 with summary data presented in Table 22.6.

Table 22.6 –Operational Costs ($US '000)

Description	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Total
Mining	26,631	20,671	22,814	22,537	22,319	21,131	9,860	6,272	5,507	1,332	**159,073**
Processing	11,245	12,707	12,707	12,707	12,707	12,707	12,707	12,707	12,707	12,545	**125,448**
G&A	4,980	5,628	5,628	5,628	5,628	5,628	5,628	5,628	5,628	5,556	**55,557**
Water Treatment	666	753	753	753	753	753	753	753	753	743	**7,430**
TSF, WRF, Infrastructure	1,491	1,685	1,685	1,685	1,685	1,685	1,685	1,685	1,685	1,663	**16,634**
	45,012	**41,444**	**43,587**	**43,310**	**43,091**	**41,904**	**30,633**	**27,044**	**26,279**	**21,839**	**364,143**

22.3.4 Capital Cost

The capital cost (Capex) is discussed in Section 21 of this Report with a summary included in Table 22.7. The sequencing of the Capex is in accordance with the project execution schedule and detailed as part of the financial model appended to this Report.

Table 22.7 – Capital Costs ($US '000)

Description	Pre-Construction Capital	Initial Capital	Sustaining	Closure	LOM Total Capex
Direct					
Pre-construction Early Works Program	22				22
Mining - Open Pit		52			52
Processing Plant		67	6	15	89
Mining Site - Infrastructure		2			2
Tailings / Waste Rock		34	12	9	54
Surface Infrastructure - Buildings, Roads, Water Treatment		12	8	2	22
Subtotal Direct	**22**	**167**	**26**	**26**	**241**

Description	Pre-Construction Capital	Initial Capital	Sustaining	Closure	LOM Total Capex
Indirect		-			
Construction Indirect		18			18
Freight & Logistics, Plant Equipment		7			7
Owner's Costs, EPCM, Taxes		10			10
Project Contingency		22	3	8	32
Refundable VAT on Initial Capital		25			25
Salvage Value		-		(10)	(10)
Subtotal Indirect	**0**	**82**	**3**	**-2**	**82**
Total	**22**	**248**	**29**	**24**	**323**

22.3.5 Closure and Reclamation

Provision is made in the Project cash flow for post-closure costs to be incurred in the final years of mining with costs as indicated in Table 22.7. Pit, tailings, and waste rock facility closure will occur throughout the mine life as applicable. It is anticipated that closure costs are expended during the life of the facility and as such have been evenly spread over a four-year period prior to anticipated closure date. No costs for closure performance bond financing have been allowed for.

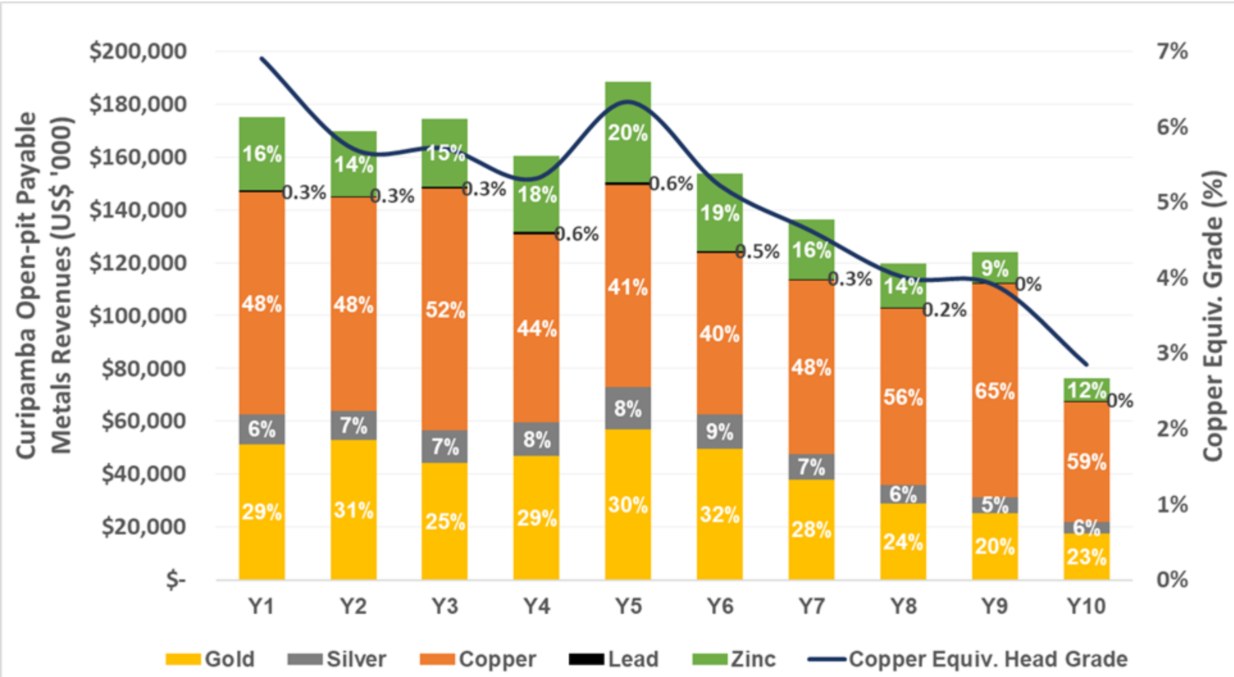
22.4 Base Case Cash Flow Analysis and Economic Results

The FS economic result for the base case is presented in this Section.

22.4.1 Revenue

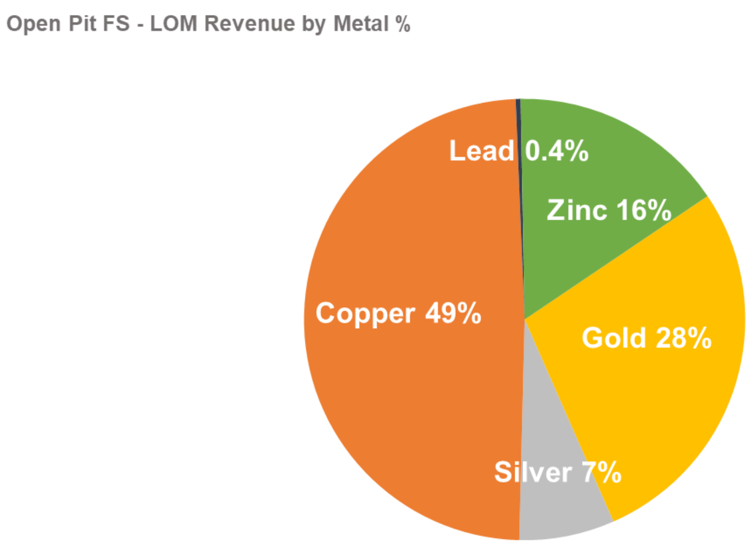
The poly metallic ore body is mined in a manner to maximise Net Smelter Return (NSR) and Net Present Value (NPV) with annual revenue by metal depicted in Figure 22.2 and LOM revenue by metal is depicted in Figure 22.3. Copper equivalent head grade declines over the LOM as expected.

Figure 22.2 – Annual Revenue by Metal



Source: DRA, 2021

Figure 22.3 – LOM Revenue by Metal – Open Pit



Source: DRA,2021

22.4.2 BASE CASE COSTS

Table 22.8 presents the All-In-Sustaining-Cost (AISC) and All-In-Cost (AIC) for the Project. AISC costs are based on recoverable metals ore mined.

Table 22.8 – AISC and AIC

Description	LOM ($'000)	Cu Eq. ($/t)
Treatment, Refining, Penalties, Transport Charges	85,909	0.18
Operating Costs	364,164	0.77
State Royalty NSR	59,134	0.13
Royalty NSR	29,567	0.06
AIC	538,774	1.14
Sustaining Capital & Closure	53,144	0.11
AISC	591,917	1.26

22.4.3 RESULTS SUMMARY

A summary of the Cashflow output is depicted in Table 22.9.

Table 22.9 – Economic Analysis Results Summary

Description	Unit	Value
Metal Price Predictions for Economic Base Case		
Metal Prices – Cu	$US/lb	3.50
Metal Prices – Zn	$US/lb	1.20
Metal Prices - Au	$US/oz	1,700.00
Metal Prices - Ag	$US/oz	23.00
Metal Prices – Pb	$US/lb	0.95
Production Profile - Open Pit Only		
Total tonnes of mineraliseore mined and processed	Million tonnes	6.48
Total capitalised pre-stripping tonnes	Million tonnes	15.25
Total LOM tonnes waste mined	Million tonnes	57.08
Operating strip ratio (following capitalised pre-stripping)	waste:ore	6.46
Overall Strip ratio	waste:ore	8.81

Description	Unit	Value
Average tonnes mined per year (waste and ore)	Million tonnes	5.15
Peak tonnes mined per year (waste and ore)	Million tonnes	11.95
Peak tonnes mineralise ore mined per year	Million tonnes	0.98
Mine life	years	10
Payable Metal Recoveries (2)		
Net average payable metal recovery to concentrates - Au	%	51.8
Net average payable metal recovery to concentrates - Ag	%	63.6
Net average payable metal recovery to concentrates - Cu	%	87.5
Net average payable metal recovery to concentrates - Pb	%	30.3
Net average payable metal recovery to concentrates - Zn	%	84.7
Unit Operating Costs		
LOM AISC(1)	$/lb Cu Eq.	1.26
Project Economics		
Royalties - Ecuadorian Government	%	4
Royalties - Altius	%	2
Average annual EBITDA	$M	103
Pre-tax NPV 8% / After-tax NPV 8%	$M	426 / 259
Pre-tax IRR / After-tax IRR	%	45 / 32
Undiscounted operating pre-tax cash flow / after-tax cash flow	$M	749 / 497

1) This Report contains certain non-GAAP (Generally Accepted Accounting Principles) measures such as cash cost and ASIC. All-in sustaining cost copper equivalent for the Project represents mining, processing, site general and administrative costs (“G&A”), water treatment costs, royalties, treatment and refining charges and sustaining capital, divided by equivalent copper tons, and excludes corporate G&A. Such measures have non-standardized meaning under GAAP and may not be comparable to similar measures used by other issuers

(2) Net recoveries only include metals payable in their respective concentrates

22.4.4 CASHFLOW

The scheduled cashflow is depicted in Table 22.10 and a cashflow graph in Figure 22.4. The peak funding requirement is anticipated in Y-1 prior to commencement of commercial production.

The cashflow is followed by NPV and IRR calculations based in the Y-2 on an 18-month pre-production major capital expenditure period.

22.4.4.1 Exploration Capital and Sunk Cost

Adventus has spent approximately $26 M in pre-production project exploration and development to date. Planned early works expenditure will be capitalised and depreciated during the LOM. Depreciation during the construction period, prior to commencement of operations will be capitalised and depreciated during the operational years.

22.4.4.2 Initial Capital

Development capital expenditure occurs over an 18-month period prior to commencement of production. The discounted cash flow date is set at 18 months prior to production. Annual cashflows are discounted to half year, or mid-point for each period.

22.4.4.3 Ramp-Up Period

A ramp up period of 3 months has been allowed for in the production schedule, offset with increased production during the second half of the first production year reducing back to normal operational throughput from Y2.

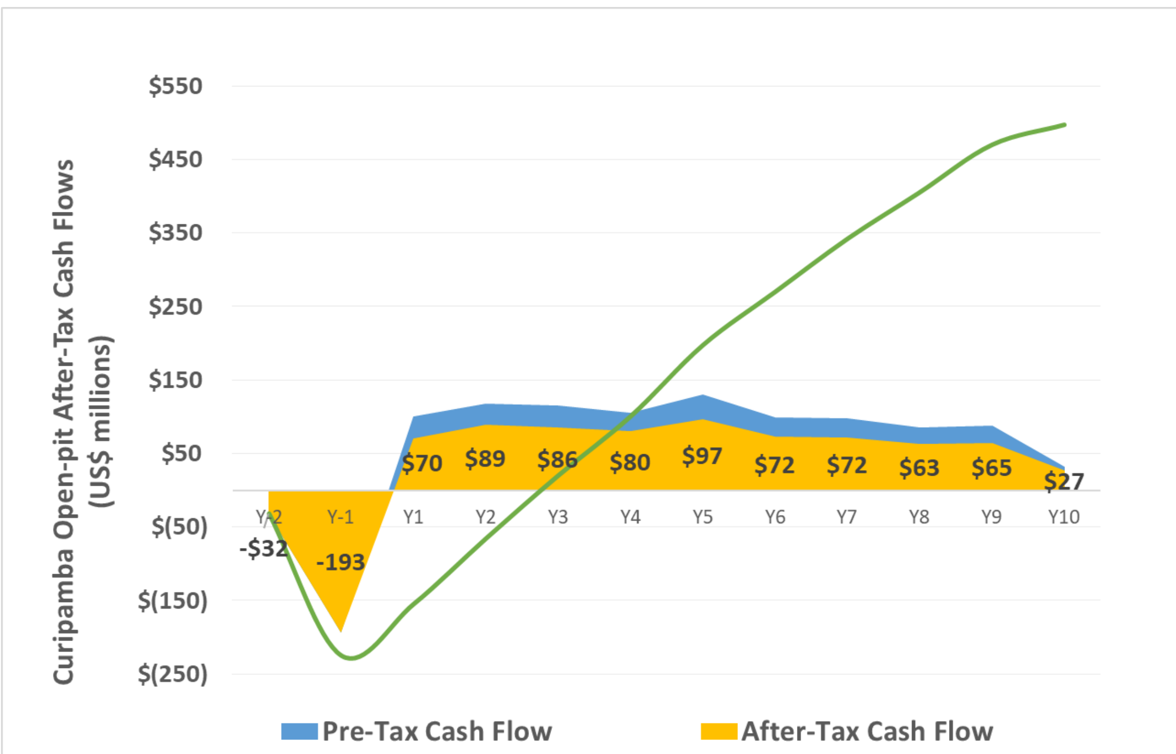
Table 22.10 – Annual Cash Flow Schedule

YEAR				FY-2	FY-1	FY1	FY2	FY3	FY4	FY5	FY6	FY7	FY8	FY9	FY10
REVENUE SCHEDULE			*(Tot. / Avg.)*												
Total Net Revenue (Open Pit)	*($'000s)*		$1,478,340			$175,104	$169,685	$174,537	$160,335	$188,614	$153,861	$136,261	$119,614	$123,929	$76,401
Total Net Revenue (Open Pit)	*($'000s)*		**$1,478,340**			**$175,104**	**$169,685**	**$174,537**	**$160,335**	**$188,614**	**$153,861**	**$136,261**	**$119,614**	**$123,929**	**$76,401**
PRE-TAX CASH FLOW															
Total Net Revenue	*($'000s)*		**$1,478,340**	-	-	$175,104	$169,685	$174,537	$160,335	$188,614	$153,861	$136,261	$119,614	$123,929	$76,401
Less: Production Royalties	*($'000s)*		**($88,700)**	-	-	($10,506)	($10,181)	($10,472)	($9,620)	($11,317)	($9,232)	($8,176)	($7,177)	($7,436)	($4,584)
Gross Income	*($'000s)*		**$1,389,639**	-	-	$164,598	$159,504	$164,065	$150,715	$177,297	$144,630	$128,085	$112,437	$116,493	$71,817
Less: Total Operating Costs	*($'000s)*		**($364,143)**	-	-	($45,012)	($41,444)	($43,587)	($43,310)	($43,091)	($41,904)	($30,633)	($27,044)	($26,279)	($21,839)
EBITDA	*($'000s)*		**$1,025,497**	-	-	$119,586	$118,060	$120,478	$107,405	$134,205	$102,726	$97,452	$85,392	$90,214	$49,978
Net VAT Refund	*($'000s)*		**$27,856**	-	-	$16,421	$4,581	$252	$596	$384	$301	$1,833	$487	$199	$2,802
Adjusted EBITDA	*($'000s)*		**$1,053,353**	-	-	$136,007	$122,641	$120,730	$108,001	$134,589	$103,027	$99,285	$85,880	$90,414	$52,780
Capital Expenditures															
Initial Capital	*($'000s)*		**$247,598**	$32,042	$193,245	$22,311	-	-	-	-	-	-	-	-	-
Sustaining Capital	*($'000s)*		**$29,070**	-	-	$4,199	$4,455	$5,375	$3,424	$1,700	$6,469	$1,150	$1,150	$1,150	-
Closure Capital	*($'000s)*		**$24,073**	-	-	-	-	-	-	-	-	-	-	-	$24,073
Total Capital Expenditures	*($'000s)*		**$300,742**	$32,042	$193,245	$26,510	$4,455	$5,375	$3,424	$1,700	$6,469	$1,150	$1,150	$1,150	$24,073
Changes in Working Capital	*($'000s)*		**($3,637)**	-	-	($8,965)	$6	($136)	$1,149	($2,384)	$2,748	$58	$939	($455)	$3,406
Pre-Tax Cash Flow	*($'000s)*		**$748,974**	**($32,042)**	**($193,245)**	**$100,531**	**$118,191**	**$115,219**	**$105,726**	**$130,506**	**$99,305**	**$98,193**	**$85,669**	**$88,809**	**$32,112**
Adj. Cumulative Pre-Tax Cash Flow	*($'000s)*			*($32,042)*	*($225,287)*	*($124,756)*	*($6,566)*	*$108,654*	*$214,379*	*$344,885*	*$444,191*	*$542,384*	*$628,053*	*$716,862*	*$748,974*
Discounted Payback Calculation	*(years)*		**2.36**	*n/a*	*n/a*	*n/a*	*n/a*	*2.1*	*2.0*	*2.4*	*1.5*	*1.5*	*0.7*	*0.9*	*-13.3*
Pre-production mid-point adjustment	*(x)*	**0.75**		*0*	*0.75*	*1.75*	*2.75*	*3.75*	*4.75*	*5.75*	*6.75*	*7.75*	*8.75*	*9.75*	*10.75*
Discount Factor	*(x)*	**8%**		*1.000*	*0.944*	*0.874*	*0.809*	*0.749*	*0.694*	*0.642*	*0.595*	*0.551*	*0.510*	*0.472*	*0.437*
Discounted Pre-Tax Cash Flow	*($'000s)*		**$425,401**	**($32,042)**	**($182,407)**	**$87,863**	**$95,646**	**$86,335**	**$73,353**	**$83,838**	**$59,069**	**$54,081**	**$43,688**	**$41,935**	**$14,040**
Pre-Tax IRR	*(%)*		**44.6%**												
AFTER-TAX CASH FLOW															
Pre-Tax Cash Flow	*($'000s)*		**$748,974**	($32,042)	($193,245)	$100,531	$118,191	$115,219	$105,726	$130,506	$99,305	$98,193	$85,669	$88,809	$32,112
Less: Advance Royalty Paid	*($'000s)*		-	-	-	-	-	-	-	-	-	-	-	-	-
Less: State & Employment Tax Paid	*($'000s)*		**($105,134)**	-	-	($12,742)	($12,171)	($12,324)	($10,620)	($13,998)	($11,204)	($10,785)	($9,291)	($9,969)	($2,028)
Less: Income Tax Paid	*($'000s)*		**($147,077)**	-	-	($17,865)	($17,056)	($17,273)	($14,859)	($19,645)	($15,687)	($15,092)	($12,976)	($13,936)	($2,687)
Less: COGEMA NPI Paid	*($'000s)*		-	-	-	-	-	-	-	-	-	-	-	-	-
Less: Sovereign Adjustment	*($'000s)*		-	-	-	-	-	-	-	-	-	-	-	-	-
After-Tax Cash Flow	*($'000s)*		**$496,763**	**($32,042)**	**($193,245)**	**$69,924**	**$88,963**	**$85,623**	**$80,246**	**$96,862**	**$72,414**	**$72,316**	**$63,401**	**$64,904**	**$27,397**
Cumulative After-Tax Cash Flow	*($'000s)*			*($32,042)*	*($225,287)*	*($155,363)*	*($66,400)*	*$19,223*	*$99,469*	*$196,331*	*$268,745*	*$341,061*	*$404,463*	*$469,367*	*$496,763*
Discounted Payback Calculation	*(years)*		**2.97**	*n/a*	*n/a*	*n/a*	*n/a*	*2.8*	*2.8*	*3.0*	*2.3*	*2.3*	*1.6*	*1.8*	*-8.1*
Discount Factor	*(x)*	8%		1.000	0.944	0.874	0.809	0.749	0.694	0.642	0.595	0.551	0.510	0.472	0.437
Discounted After-Tax Cash Flow	*($'000s)*		**$258,578**	**($32,042)**	**($182,407)**	**$61,113**	**$71,994**	**$64,158**	**$55,675**	**$62,226**	**$43,074**	**$39,829**	**$32,333**	**$30,647**	**$11,978**
After-Tax IRR	*(%)*		**31.5%**												

Table 22.11 – Economic Results at Varying Discount Rates

Description	Unit	Pre-Tax Results	After-Tax Results
IRR	%	45%	32%
Discounting ($M)			
NPV	0%	$748,974	$496,763
NPV	5%	$524,515	$331,333
NPV	8%	$425,401	$258,578
NPV	10%	$370,239	$218,195
Payback period, from start of production	Year	2.4	2.6

Figure 22.4 – Cash Flow Graph



Source: DRA, 2021

22.5 Sensitivity Analysis

The sensitivity of the Project returns in key value drivers was tested over a range of ±20% for metal prices, operating costs, capital costs, and recoveries. For this Project, sensitivity to grade is the same as for metal price. The sensitivity results are presented in Figure 22.5 and Table 22.12.

Figure 22.5 – Sensitivities

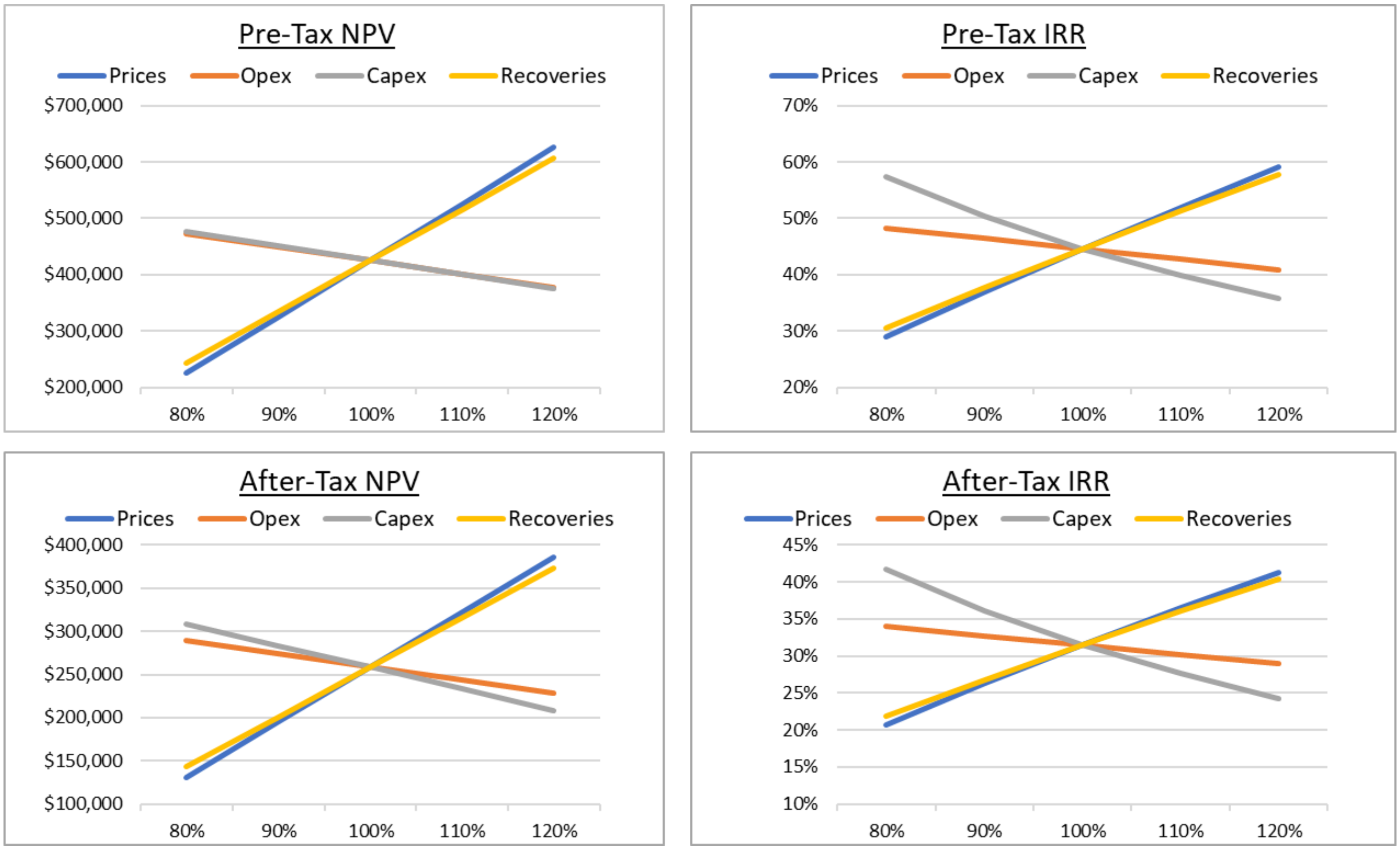


Table 22.12 – Cashflow Sensitivity Table

CHANGE	After-Tax NPV @ 8% (USD$ '000)			
	Capex	Opex	Metal Pricing	Recoveries
-20%	308,934	288,844	130,873	143,092
-10%	283,698	273,653	194,884	200,884
0%	258,462	258,462	258,462	258,462
10%	233,225	243,270	322,000	316,014
20%	207,989	228,079	385,427	373,455

23 ADJACENT PROPERTIES

There are several active properties in proximity to the Curipamba project (Figure 23.1).

The Pegasus Project, a joint venture project between Anglo American PLC (Anglo) and Luminex Resources Corp. (Luminex), is the largest project in proximity to the Curipamba Project. The Pegasus Project consists of two (2) contiguous properties totalling 676.4 km²; Pegasus A, which is 5.5 km north of the Curipamba property, and Pegasus B, three kilometres to the east. In 2018, Anglo signed an earn-in agreement with Luminex (the property owner) giving Anglo the right to earn a 60% ownership interest in the property through expenditures of US$57.3 M between 2018 and 2025 (US$50 M to be invested in exploration on the project and US$7.3 M of staged cash payments). Anglo has the right to earn an additional 10% ownership by funding of the required work to advance the project to a mining decision (Luminex Resources Corp., 2018). In September 2021, Anglo had met the 25% ownership threshold (Luminex Resources Corp, 2021). In 2019, an airborne Z-Tipper Axis Electromagnetic (ZTEM) survey was flown primarily over Pegasus A, totalling approximately 2,000 line kilometres. There was also a program of field mapping and geochemical sampling. Several copper porphyry targets were defined from these programs. Anglo plans to follow up with drilling in 2022 (Luminex Resources Corp., 2020).

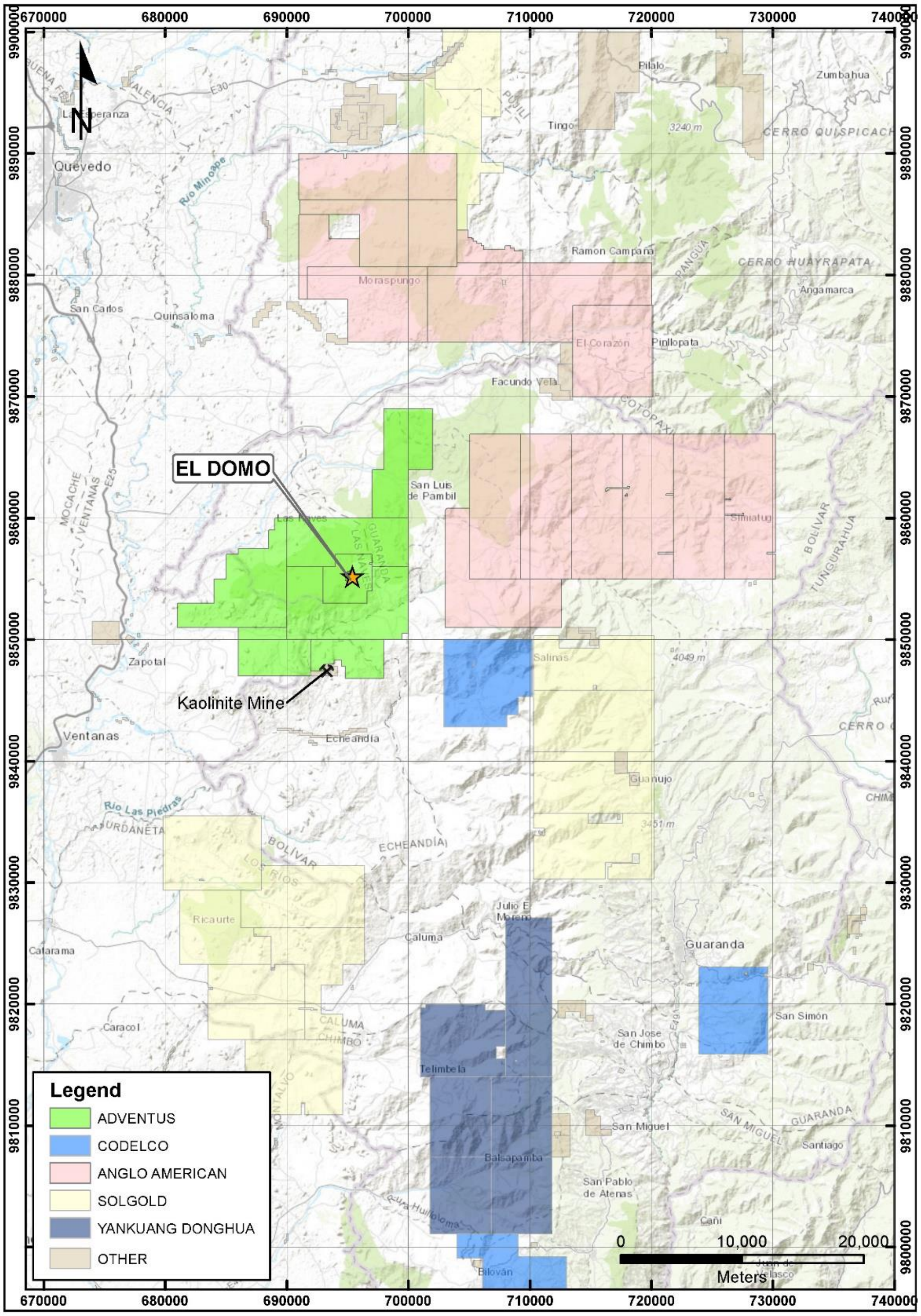
10 km east of the Curipamba project is Sol Gold's Salinas project. The project consists of four (4) concessions, totalling 188 km². The project is 100% owned by Sol Gold, through the Valle Rico Resources S.A. subsidiary. The project is considered prospective for gold-silver-copper epithermal and copper-gold porphyry mineralization. The property was previously drilled by Rio Tinto, returning 74.5 m at 2.0 g/t Au and 137 g/t Ag, including 39.5 m at 3.3 g/t Au and 168 g/t Ag. Mineralisation is hosted in structurally controlled hydrothermal volcanic breccias. A hypogene covellite-enargite chalcocite arsenopyrite paragenesis of phases suggests a nearby larger Cu-Au porphyry system (Sol Gold PLC, 2021). The property is currently considered a priority target for Sol Gold's regional exploration program.

Immediately south of the Sesmo Sur target is a kaolinite mine operated by Edesa S.A., a ceramics company based in Quito (Edesa S.A., 2021). The mine extracts kaolinite and feldspar. The mine is located within an extensive hydrothermally altered rhyolite porphyry (Beate, 2007).

A large property block 22 km south of Curipamba is owned by the Yankuang Group. Early exploration including geophysics, drilling, mapping and geochemical is listed on the company website, though no details are provided (Yankuang Group, 2021).

There are also several smaller properties owned by Codelco (*Corporación Nacional del Cobre, Chile*) in proximity to the Curipamba project, however details on these properties are not provided (Codelco, 2021).

Figure 23.1 - Adjacent Properties



24 OTHER RELEVANT DATA AND INFORMATION

This section describes additional data relevant to the Project. The project schedule, project execution plan, risk assessment, and operational readiness plan will be discussed as part of this section.

24.1 Project Execution Schedule

24.1.1 Critical Construction Activities and Key Dates

The Project has developed a conceptual execution schedule suitable for the FS stage to identify critical and near critical path activities.

The construction of the tailings storage facility embankment requires a significant volume of competent waste rock material from mining pre-production operations. Based on geotechnical drilling and testing as part of the FS, the optimum pre-stripping mine plan to access the open pit does not produce sufficient competent waste rock; therefore, the mining schedule was augmented to increase pre-production stripping. The combination of mine pre-stripping and TSF embankment construction activities form the critical path of the Project. Prior to the mine pre-stripping activities ensuing, a mining contract must be awarded and the contractor must then secure a suitable mine fleet , while the main haul roads must be constructed from the mine pit area to the saprolite dump and tailings storage embankment toe areas, followed by completion of construction excavations for waste backfilling.

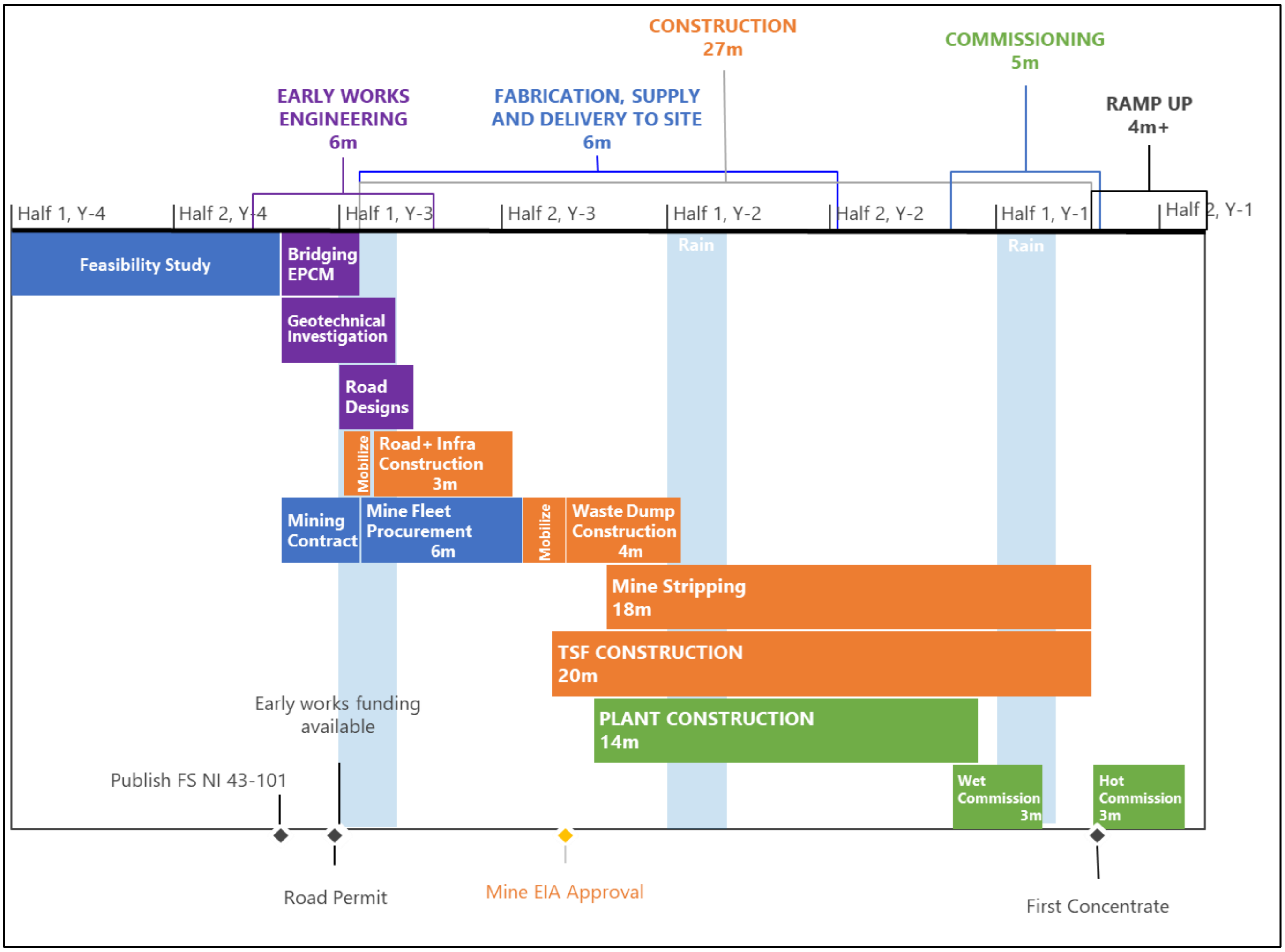
The current schedule has not considered potential delays in earthworks excavation construction during the rain season (Dec-May), assuming that excavations will be completed in the dry season and be backfilled above ground level prior to the rain season. A final decision on the commencement date of construction may impact on the timeline of the earth-works construction.

Near critical path items associated with the process plant identified the mill procurement and installation as a potential concern. Worldwide supply chain disruptions associated with the COVID-19 pandemic is extending fabrication and delivery times. Accounting for such variances, by increasing typical supply chain durations by eight (8) weeks, brings the mill and process plant on the near critical path.

The commencement of construction activities remains dependent on the receipt of required permits from local authorities. The anticipated receipt dates of the relevant permits have been indicated as key start dates for construction activities.

Figure 24.1 presents the critical activities and dates for the Project.

Figure 24.1 – Critical Patch Activities and Dates



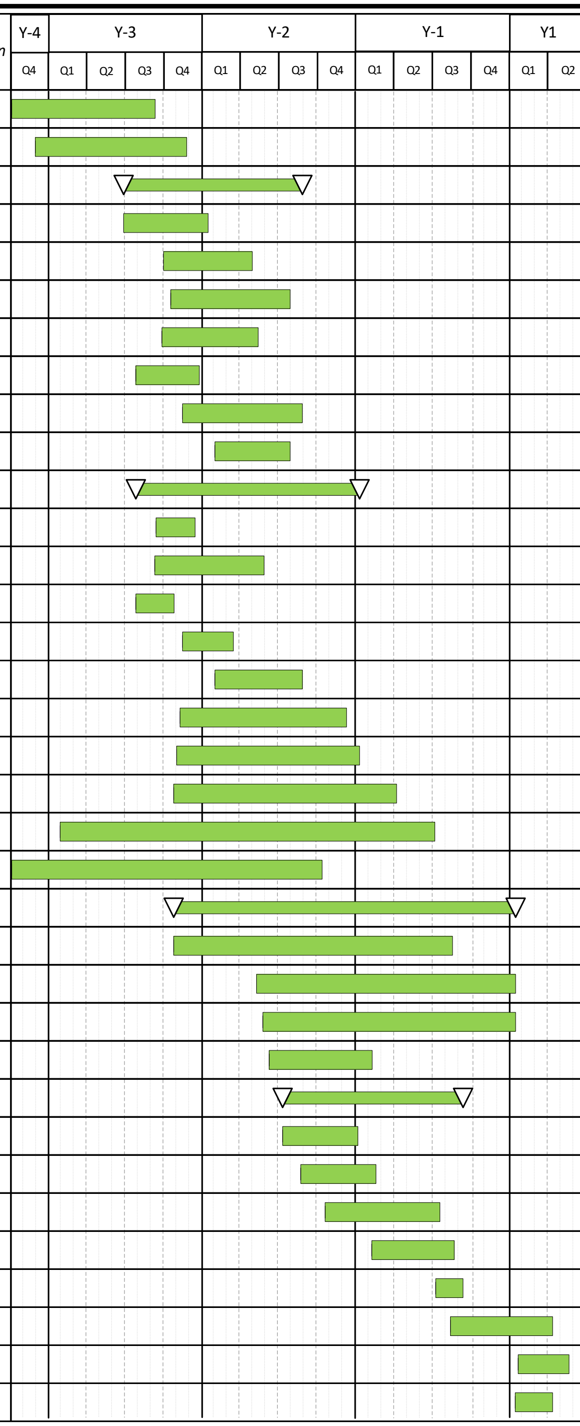
Source: DRA, 2021

24.1.2 PROJECT SCHEDULE

A more detailed breakdown of the Project FS schedule is presented in Figure 24.2.

Figure 24.2 – Curipamba Project FS Schedule

ID	Task Name	Duration
1	FEASIBILITY STUDY	69 w
2	PROJECT FINANCING	52 w
3	**DETAILED ENGINEERING**	**61 w**
4	Early Works Engineering	29 w
5	Process Engineering	30 w
6	Mechanical DO	41 w
7	Civil & Infrastructure - Engineering and Design	33 w
8	Tailings Storage Facility	22 w
9	Electrical Engineering	41 w
10	C&I engineering	26 w
11	**PROCUREMENT**	**76 w**
12	Early Works Packages	14 w
13	Process Equipment	38 w
14	Mining	13 w
15	Power Supply	18 w
16	Electrical, C&I Equipment	30 w
17	Fabrication Packages	57 w
18	Construction Packages	63 w
19	FABRICATION, SUPPLY AND DELIVERY TO SITE	76 w
20	OPERATIONAL READINESS	128 w
21	APPLICATIONS, PERMITTING AND ESIA	106 w
22	**CONSTRUCTION**	**116 w**
23	EARLY WORKS CONSTRUCTION	95 w
24	MINE PRE-STRIP	88 w
25	TSF CONSTRUCTION	86 w
26	MOBILIZATION	35 w
27	**PLANT CONSTRUCTION**	**62 w**
28	Earthworks Plant & Infrastrucure	26 w
29	Civil	26 w
30	SMPP	39 w
31	EC&I	28 w
32	Pre-Operational Verification (POV) and Pre-commissioning	10 w
33	COMMISSIONING	35 w
34	Ramp up to full production	18 w
35	PROJECT CLOSE-OUT	13 w



24.2 Project Execution Plan

24.2.1 Overview

The Project Execution Plan (PEP) details the schedule, personnel structures, and methodologies that will be employed to construct the Project scope of facilities. The PEP components noted in this Report is not intended to be encompassing, only to provide context to the execution planning and economics of the FS.

24.2.2 Project Implementation

The FS estimate considers the appointment of an Engineering, Procurement and Construction Management (EPCM) as the main contractor to provide the necessary technical and engineering design details for the project. The EPCM will appoint specialist experts for specific Project element designs such as the tailings and waste facilities, fire protection, water treatment and surveying. The procurement of equipment and construction contracts will be integrated with the design and fabrication planning to ensure an executable construction supply chain is established. A detailed contracting strategy and procurement operating plan will be developed as part of the next project phase.

The EPCM contractor will oversee the construction activities and be responsible for the supply of construction services for the project. Adventus plans to bolster an experienced EPCM construction team with locally employed personnel that can support the construction phase, and potentially progress onto the operational phase of the Project.

A phased project commissioning plan has been considered for the FS through which operational personnel take gradual ownership of the project facilities as they are completed and commissioned. The Operational Readiness Plan, discussed in Section 24.3, will prepare the operational personnel and site support services for the ramp-up and production phase. Operational personnel from the EPCM contract will the support the Project into the ramp-up and production phase to ensure continued support of plant design into operations.

The Ecuadorian mining industry is broadly observed as developing with established mines base expanding and continuous mine development in country. The Project will support the development of local Ecuadorian skills development through transference from skilled inter-provincial and limited expatriate expertise. Adventus is engaging with local institutions to promote and establish mining related skills development in anticipation of the Project execution. The FS has allowed for the involvement of expatriate construction and limited operational personnel to support skills transfer and develop local Ecuadorian expertise during the execution phase. The early detailed development and implementation of the operational readiness plan will benefit the Project and reduce human resource risk.

24.2.3 Construction Phase Quantities

Construction quantities were estimated for the major Project components as presented in Table 24.1.

Earthworks encompasses the establishment of the terraces and associated roads. The steep terrain requires the removal of mostly saprolite material. The earthworks quantities are significant and will require a dedicated earthmoving fleet to complete the construction timeously. The process plant contains most of the concrete and detailed earthworks for the Project. The saprolite dump requires the comingling of saprolite with more competent waste rock to increase long term stability. Phasing of the earthworks and mining excavations require consideration to ensure that sufficient competent waste rock is produced for co-mingling of both the mine over burden and site infrastructure saprolite waste.

Structural steel and platework is concentrated in the process plant area. The mine workshop is also noted as a large steel structure. Tonnages are reasonable for the size of plant.

Mechanical equipment is concentrated in the process plant with mostly pumps, water and sewage treatment and containment infrastructure distributed across other terraces. Electrical components are related to the mechanical equipment power and control requirements and reasonable for the size of plant.

Table 24.1 – Construction Phase Quantities

Discipline	Quantity Description	Estimation Method	Unit	Quantity
Mining	Mining Pre-Production Stripping	Estimated from mine designs and specific waste densities	t	16,370,000
Earthworks	Excavation - Terraces	3D civil designs with reasonably accurate topographical mapping suitable for FS	m^3	536,000
	Excavation - Site Roads		m^3	492,000
	Engineering Fill - Terraces		m^3	125,000
	Engineering Fill - Site Roads		m^3	122,000
	Excavations - Saprolite Facility		m^3	878,000

Discipline	Quantity Description	Estimation Method	Unit	Quantity
Earthworks (cont'd)	Excavations - Tailings Facility		m^3	372,000
	Excavations - Waste Rock Facilities		m^3	125,000
	Fills - Mine Waste - Tailings Embankment, Foundations, Drains, Toe Berms		m^3	2,388,000
	Fills - Mine Waste - Saprolite dump drains and toe berms		m^3	647,000
	Fills - Mine Waste - Waste Rock dump drains and toe berms		m^3	60,000
Concrete	Concrete	Preliminary FS layout of structures with foundations designs suitable for geotechnical conditions	m^3	9,000
Steel	Structural Steel	Preliminary layout of structures and equipment, with steel BOQs estimated from dimensioned FS drawings and compared to previous project database	t	900
	Platework	Preliminary layout of piping and platework items within plant structures with BOQs estimated off dimensioned drawings and compared to previous project database	t: unit	400: 105
Piping	Plant Piping	Factored from project database for similar sized plant.	m	7,000
	Overland Piping	Estimated from layout drawings.	m	5,000
Mechanical	Mechanical equipment items	Mechanical Equipment list	Unit	316
Electrical	Electrical Cabling - Power & Control	Estimated from layout drawings and preliminary routings.	m	66,000

24.2.4 Early Works

Adventus is in the process of obtaining the required ESIA approval and applying for required permits. The application period produces an opportunity to progress certain Project items.

An early works package is planned to address critical construction phase support project components. The implementation of the early works package will assist the Project execution schedule by alleviating construction pressure and execution economic risk.

Progression of detailed geotechnical and survey information will provide the detailed engineering phase with required information to improve design accuracy and costing.

Early progression of the detailed engineering, including securing of certified equipment information, and soliciting firm construction contractor rates, will improve the Capex accuracy and reduce construction uncertainty risk.

Planned road upgrades and access road construction will reduce logistics risks and ensure larger equipment can be delivered to site without special transportation requirements. Early development of the site construction roads to the valley floor will ensure commencement of mine pre-stripping and constructions excavations. This work remains subject to obtaining relevant permits for which applications have been filed.

24.2.5 Construction Phase Water supply and management

Adventus (via Curimining) holds current exploration permits that allows for the extraction of water from surface water which is deemed adequate for planned early works. Application for further borehole extraction permits will be filed to serve as the main source of construction water.

The construction phase will require contact water containment and treatment capabilities. Construction of temporary earth berm containment dams will capture contact run-off water for sediment control purposes and allow monitoring opportunities. The completion of the waste dump toe berms and run-off containment ponds will serve as fixed infrastructure for water containment and treatment.

Temporary containerised water treatment plants will be rented for the construction period to treat mine contact and waste dump run-off water that may contain suspended solids or become acidic due to exposed PAG rock. The initial construction water volumes are still to be determined but are deemed reasonable during the early phases of construction.

The TSF facility will capture and contain the El-Silencio catchment area water once the embankment is completed and deemed stable for water containment. Detailed embankment construction phasing, that will include early water bypass designs, will commence in the next project phase. The El-Silencio basin water quality is expected to remain stable during the construction period as no reactive ore or tailings is being produced. Construction area contact water and waste dump run-offs will be captured, contained and treated. It is expected to comingle reactive waste rock types in the saprolite facility.

The Project will expedite the water treatment plants construction as a priority to ensure sufficient treatment capacity exists when the tailings facility is closed for water containment. Although the water quality is estimated to remain good as no tailings material has been introduced, the water volumes remain significant should any form of treatment be required. The TSF starter dam design includes for an emergency overflow spillway as part of its design.

24.2.6 Construction Phase Power Supply

Adventus will be relying on grid power as far as possible. Basic 7.69 kV power line infrastructure is present on site and is deemed sufficient for basic power supply during early works, but is not deemed adequate for the construction phase of the project. Construction power will require the upgrading of local power infrastructure, or supplementation with generator power until the main 69 kV power supply is commissioned on site. It is recommended to progress a detailed study of local infrastructure and evaluate the potential to upgrade and utilise it for construction power. Back-up generation is still recommended to support the construction effort.

24.3 Operations Readiness Plan

The Curipamba Project requires an Operational Readiness Plan (ORP) to assist with transition from project construction, through ramp-up, into operations.

The aim of the ORP is to ensure:

- New assets are effectively handed over to a trained and experienced work force;
- Infrastructure can be safely and efficiently operated toward meeting business value targets;
- Stakeholders are provided with a high level of satisfaction pertaining the new assets;
- Business risk associated with new assets and destruction of value is reduced; and
- Compliance with statutory environmental and legislative requirements.

24.3.1 Operational Strategy

Adventus plans to operate the Project:

- With a combined owner/contractor model:
  - Mining operations will be contracted to an experienced mining contractor.

- Non-essential, or non-core support services will be subcontracted to local service providers.
- Core process plant and site support infrastructure and services through the development of local personnel directly employed.

The Owner's operated team will be supported periodically by expert consultants to provide operational experience and training.

24.3.2 ORP Gap Analysis and Planning

Adventus retained DRA to perform an FS level ORP gap analysis and commence the ORP process in anticipation of Project execution. Following the gap analysis assessment, DRA prepared an initial ORP plan to develop Adventus toward owner operation.

The ORP focussed on the following main areas of development:

- Asset Maintenance and Management;
- Commissioning and Ramp-up;
- Engineering & Design;
- Human Resource Management;
- Information Technology, Communications and Systems;
- Change Management;
- Metallurgical Accounting;
- Processing;
- Procurement and Inventory;
- Safety, Health, Environment and Quality Management;
- Training and Development.

The ORP indicates the necessary actions to prepare Adventus for operations and have been scheduled to coincide with the Project execution schedule. The ORP does not currently include for Owner's mining actions.

The current state of Operational Readiness is acceptable for the FS phase of the Project.

24.4 Project Risk Review

Adventus maintains a Project risk register which is regularly updated. A risk review process was undertaken as part of the FS following the technical design definition. Key risk descriptions, reduction or mitigation recommendations, as per the risk register, are summarised in Table 24.2. Recommendations are made toward addressing remaining uncertainty/risk during the next Project phase.

Table 24.2 – Project Risk Identification and Recommendations

Risk	Impact	Recommendations for Next Project Phase
Permitting	Obtaining required permits for early works and the main construction phase is critical for project execution planning.	EISA and road widening permits have been filed. Further develop compliance and permitting plan. Execute required permit applications timeously.
Geotechnical and Topographical Design Risk	Significant quantities of bulk earthworks and mine waste stripping are required for the project. The current project economics are dependant on accuracy of geotechnical data and detailed surveying to progress he bulk earthworks definition.	A geotechnical drilling program associated with current infrastructure locations have been drafted and is planned early on during the next project phase.
Site Power Supply Infrastructure	Initial interactions with electrical utilities have commenced. Based on experience on other projects in country, risk of not having grid power connection in timeframe to meet project schedule.	Formal applications for power allocations and commitments from local authorities have commenced. Meeting held with Ministry of Infrastructure secured support from highest levels of government to ensure timeframe is met.
Waste Rock Characterisation and Closure Planning	Acid Generating potential and heavy metal leaching characterisation requires additional test work to determine waste facility closure final design requirements (lining, treatment, co-disposal). The current facility designs and closure concepts relies on the Saprolite lithology as a stable sealing layer material.	Complete additional waste characterisation test work prior to detailed design phase. Complete regulatory compliance framework for various waste types to ensure efficient waste management design and operation on site.
Construction/Execution Risk Local Contractors	Initial contractor responses were well received from local and international contractors. The volume of earthworks and mining material movement may not be possible for singular contractors to undertake. Risk remains in the contractor's ability to execute parts of the project scope.	Perform further supplier vetting and tendering processes during next project phase, including plant and equipment audits, document and regulatory compliance audits. Review early works opportunities. Review owner involvement in equipment or service supply.
Global Pandemic Management	The impact of COVID-19 during the FS was prevalent in the cancelling of site visits and local contractor verifications.	Ensure a travel plan and lock-down mitigation plan is maintained to allow Detailed Engineering and site related activity personnel to travel toward ensuring required site and supplier definition is obtained. Identify key activities and develop mitigating actions.

24.5 Project Opportunity Review

Additional to the risk review process, a project opportunity review process was undertaken upon completion of the FS definition. The main project opportunities and further recommended actions are presented in Table 24.5.

Table 24.3 – Project Opportunities and Actions

Opportunity	Potential Impact	Recommendations for next project phase
Underground Mine Expansion Opportunity	Opportunity to extend mine life and gain more value from the available metal resources.	Continue with infill drilling program. Continue with rock mechanic interpretations and design. Progress the mine design into PFS or FS phase. Improve scheduling and integration with the open pit mining plan.
Waste Management Optimisation	Reduced waste facilities overall footprints and improve long term passive closure concepts.	Review of waste management strategies to optimize embankment construction and reduce mine pre-production construction material requirement. Review potential of filtered tailings to optimize waste management facilities construction and reduce upfront mine pre-stripping. Optimise the PAG/NAG waste lithology placements as part of waste management detailed design and introduce oxygen impregnable layers to reduce acid rock drainage. Review in-pit waste storage during later phases of the mine life.
Water Treatment Optimisation	Reduce amount of water captured, contained, and treated throughout the life of mine. Reduce water contamination risks.	Review alternatives to capturing, mixing and treatment of TSF water in the El-Silencio basin. Further explore upstream capturing, diverting or pumping non-contact water away from site as part of detailed design. Develop a numerical underground water model to review upstream water extractions from faults around the open reduce ingress and treatment of ground water.
Phased Closure Planning	Reduce mine footprint over life of mine. Improve early environmental regeneration. Reduce acid water generation risk and improve closure planning through early testing of closure designs.	Review early or partial pit flooding to reduce acid generating potential. Review potential for early closure of exhausted pit phases to reduce exposed rock surfaces and reduce contact water and acid generation potential.

24.6 Further Work Programs

Additional to the early works packages described, Adventus has identified the following work packages that will be progressed during the next project phase, prior to a final Initial Capital expenditure stage gate decision.

Table 24.4 – Further Work Packages

Item	Total (US $M)
Labs/test work - tailings material, geochemistry, and metallurgical	0.5
Additional Field Assessments & Geotech	1.5
TSF value engineering program (cost reduction)	1.5
Mine design development to contract award	0.25
Final land acquisition for site infrastructure	3.1
ESIA Public Consultations	0.05
TOTAL	**6.8**

24.7 Underground Mining Opportunity

This section presents the results of a Preliminary Economic Assessment (PEA) level study (±50% accuracy) that was conducted to evaluate the merits of mine life extension through further underground mining following end of the open pit mine life. The underground mining was assumed to commence upon the exhaustion of the open pit mine; there is no overlap in production or resource. Only the resources located outside of the ultimate open pit limit are considered in the underground study. The economic analysis in this section is preliminary in nature and is based, in part, on Inferred Mineral Resources, which are considered too speculative to be converted to Mineral Reserves. There is no certainty that the results of the PEA will be realised.

24.7.1 Geotechnical Parameters

There is insufficient geotechnical drilling information available for a geotechnical analysis of the underground deposit. It is recommended that geotechnical work is undertaken prior to the next phase of underground mine planning to adequately support assumptions.

24.7.2 Mining Assumptions

Stope wireframes were developed to convert the Mineral Resource to an inventory of mineralised material for underground mining. These stope wireframes include any internal dilution necessary to general a practical mining shape (planned dilution). In addition, unplanned dilution was assumed to be 5%, and a 95% mining recovery was used.

24.7.3 Mining Method and Design

The mine design is based on a single decline with a 15% gradient excavated in the final open pit east wall at an elevation of 850 m at the bottom of the pit. Stopes will be accessed from the footwall of the deposit. There will be access drives every 5 m, to match the stope height.

All mineralised material and waste developments are mined with a 5.0 m by 5.0 m end profile.

The deposit will be mined using drift and fill with cemented rock fill at a production rate of approximately 0.67 Mt/year. The mine will be operated by a contractor. The mine design is illustrated in Figure 24.3 to Figure 24.5.

Figure 24.3– Open Pit and Underground Mine Design (Plan View)

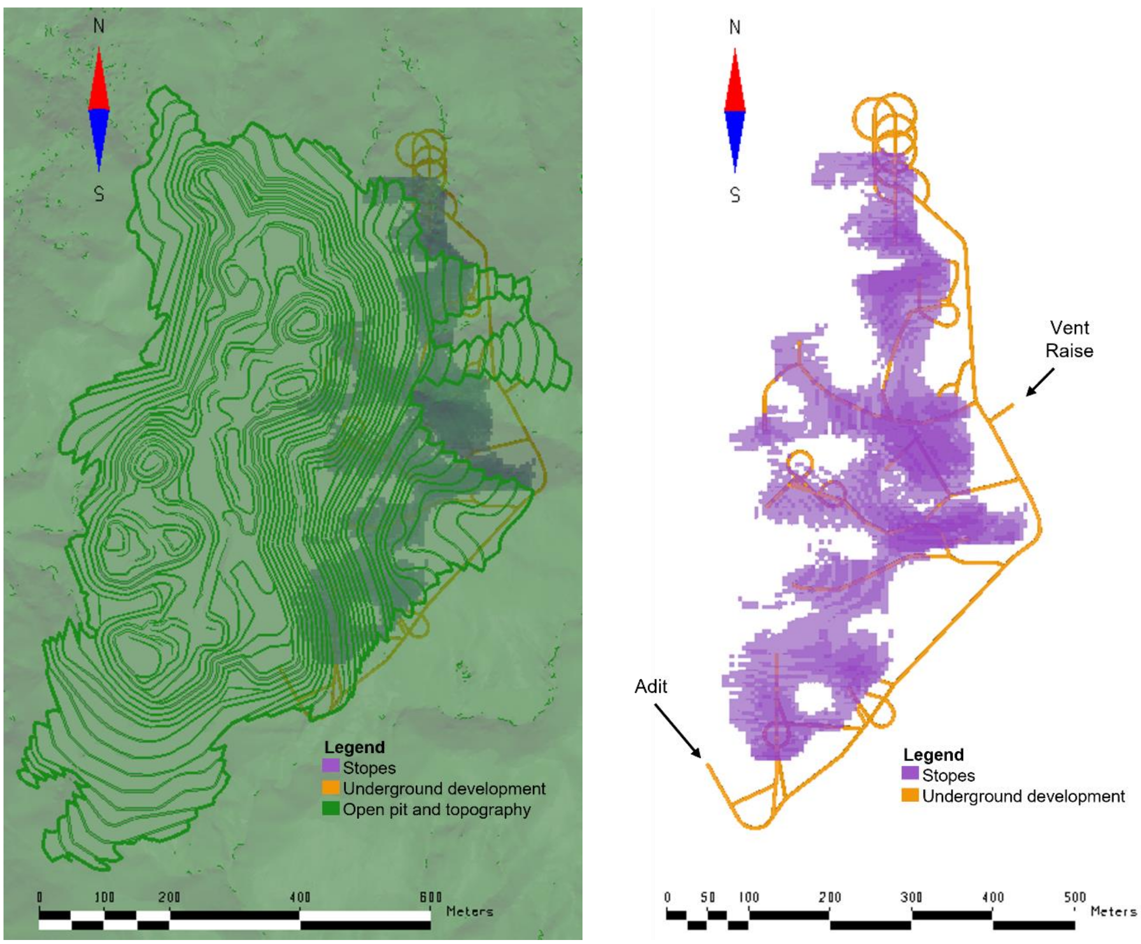
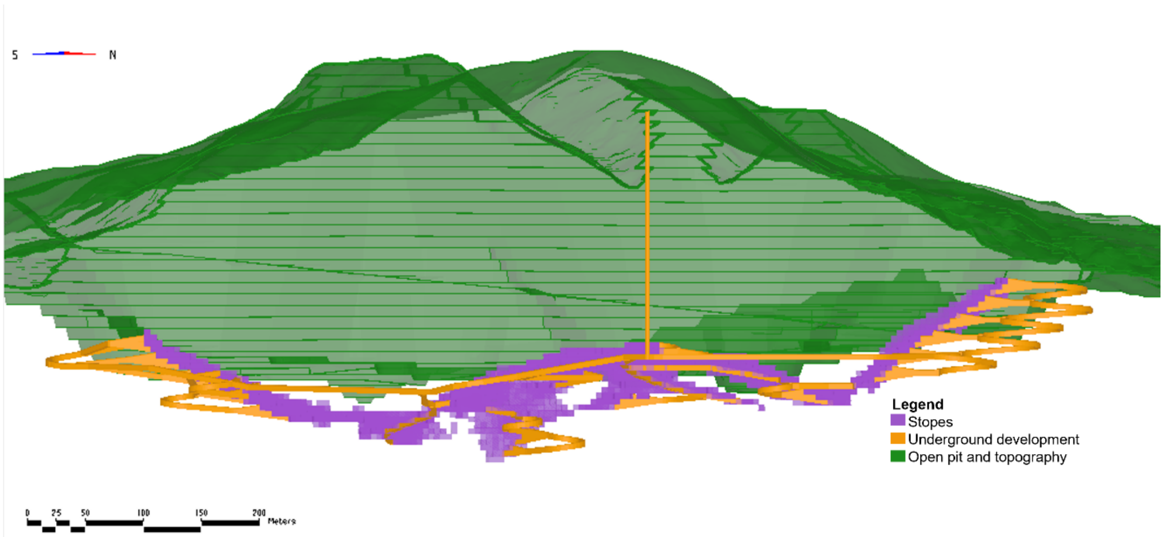


Figure 24.4 – Open Pit and Underground Mine Design (Long Section View)



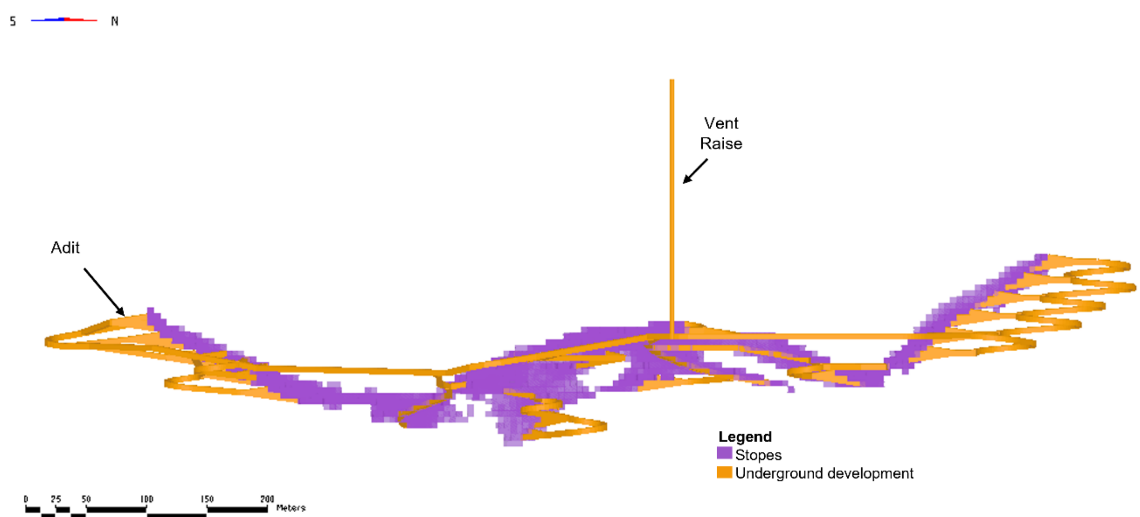
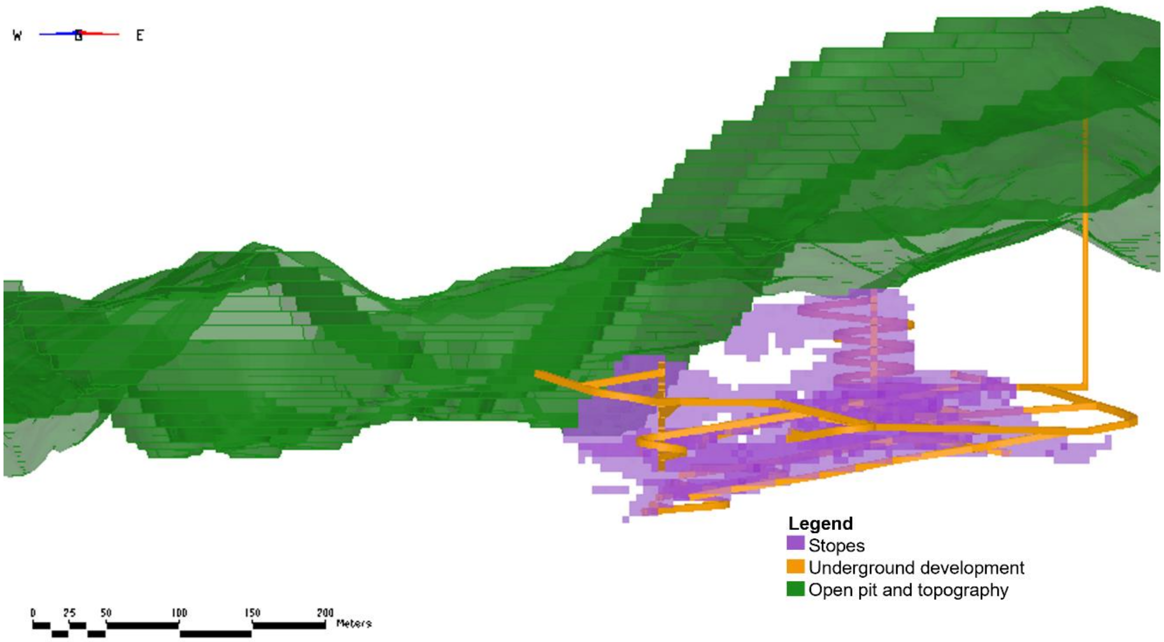
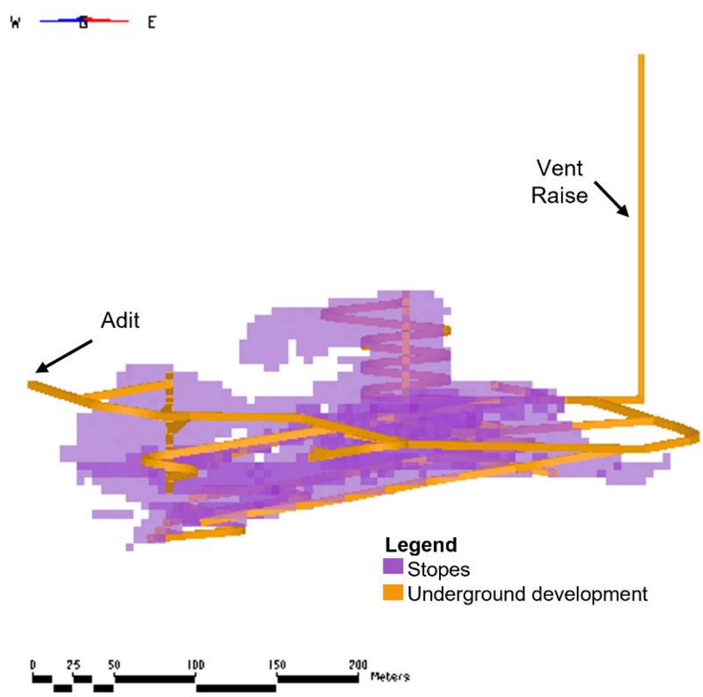


Figure 24.5 – Open Pit and Underground Mine Design (Short Section View)





24.7.4 Stope Optimisation

Stope wireframes were generated using Deswik.SO, a stope optimisation software. The parameters used in the optimisation are listed in Table 24.5. The block NSR value used is the same that was used to calculate the open pit resources (see Section 14); an NSR cut-off of US$ 105.2 /t of mineralised material was used for the stopes. Outlying or isolated stopes were removed when it could not be economically mined, including for access development cost.

Table 24.5 – Underground Opportunity Stope Optimisation Parameters

	Unit	Value			Source
		High Zn	Mixed Zn/Cu	High Cu	
Metal Prices					
Au	$/oz	1,800.00			See Section 14 – remained consistent
Ag	$/oz	24.00			
Cu	$/lb	4.00			
Pb	$/lb	1.05			
Zn	$/lb	1.30			
Process Recoveries					
Au	%	68	55	14	See Section 14 – remained consistent
Ag	%	78	67	29	
Cu	%	86	86	80	
Pb	%	90	82	37	
Zn	%	97	95	36	
Mining Costs					
Mining	$/t mined	70.00			Based on a similar project
Cemented Rock Fill	$/t ore	5.00			DRA estimate
Development	$/m	5,205.00			Based on a similar project
Stockpile Rehandle	$/t ore	0.33			DRA estimate
Mine Dewatering	$/t mined	0.25			DRA estimate, specific to Underground
Processing Costs					
Processing	$/t milled	18.58			DRA estimate, specific to Underground
Tailings	$/t milled	1.96			

G&A	$/t milled	7.91	
TSF Dewatering Treatment	$/t milled	1.13	
Royalty	%	6	See Section 22 for explanation on royalties – same applied.

24.7.5 Production Rate

The mineralised material is accessible enough to have six (6) stopes in drilling, loading, blasting, mucking and ground support at once, for a production of 300 t/d per stope, and twelve (12) stopes in backfilling and curing at once, for a production of 100 t/d per stope. Therefore, the 1,850 t/d (0.67 Mt/year) production target will be achieved.

24.7.6 Development Schedule

The development schedule is summarised in Table 24.6. Ramp development was scheduled at a rate of 150 m/month, and when multiple headings are available, it was scheduled at a maximum of 250 m/month. Development quantities are reported directly from the mine design.

Table 24.6 – Development Schedule

Description	Unit	Underground Production Year					
		1	2	3	4	5	Total
Main decline / Level	m	2,370	1,240	-	-	-	3,610
Access	m	250	250	-	-	-	510
Drift and Fill Access	m	210	450	450	450	290	1,840
Vent Access	m	30	-	-	-	-	30
Total Horizontal Development	*m*	*2,870*	*1,940*	*450*	*450*	*290*	*5,990*
Vertical Development	m	-	210	-	-	-	210
Total Development	***m***	***2,870***	***2,150***	***450***	***450***	***290***	***6,200***

Notes:
1. Figures have been rounded to an appropriate level of precision
2. Due to rounding, some columns or rows may not compute as shown.

24.7.7 Production Schedule

The production schedule was development using Deswik.Sched software. Approximately 2.8 Mt of mineralised material is planned to be mined to meet the 0.66 Mt/y production target at the mill. Any additional ore material can be stockpiled. A summary of the production schedule is presented in Table 24.7. The mining sequence, which will be on a primary/secondary, bottom up basis, is presented in Figures 24.6 to Figure 24.10.

Table 24.7 – Production Schedule

Description	Unit	Underground Mine Year					
		1	2	3	4	5	Total
Ore	kt	326.1	689.9	689.9	689.8	449.2	**2,844.9**
NSR	$/t	219.1	196.1	197.3	207.4	206.2	**203.3**
Au	g/t	1.22	1.33	1.42	1.24	1.44	**1.33**
Ag	g/t	37.51	27.58	28.47	26.75	18.90	**27.36**
Cu	%	2.54	2.21	2.18	2.52	2.47	**2.36**
Pb	%	0.16	0.13	0.12	0.11	0.06	**0.12**
Zn	%	2.64	2.14	2.38	2.02	2.18	2.23

Notes:
1. Figures have been rounded to an appropriate level of precision
2. Due to rounding, some columns or rows may not compute as shown.

Figure 24.6 – Production Year 1

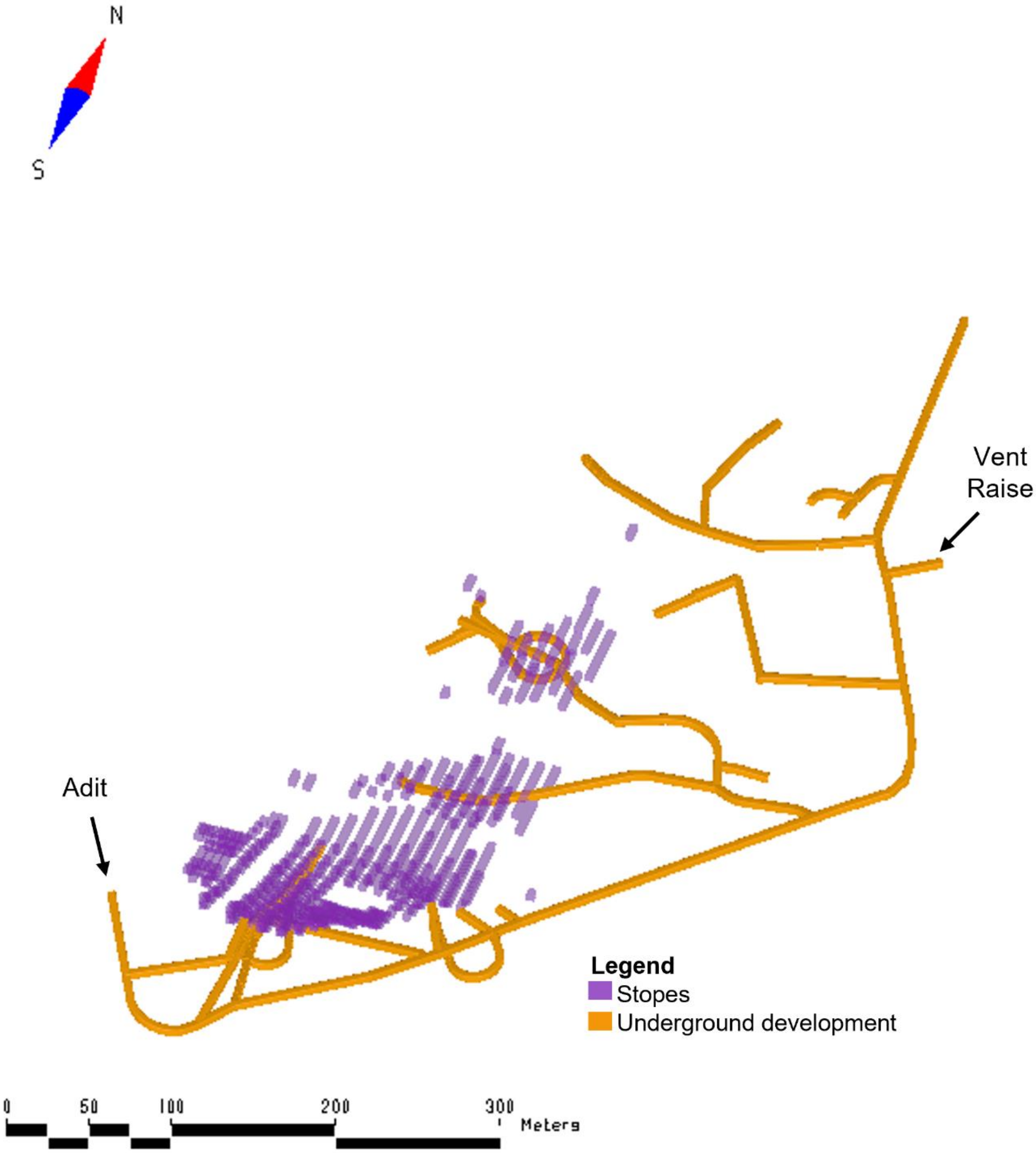


Figure 24.7 – Production Year 2 (Cumulative)

Figure 24.8 – Production Year 3 (Cumulative)

Figure 24.9 – Production Year 4 (Cumulative)

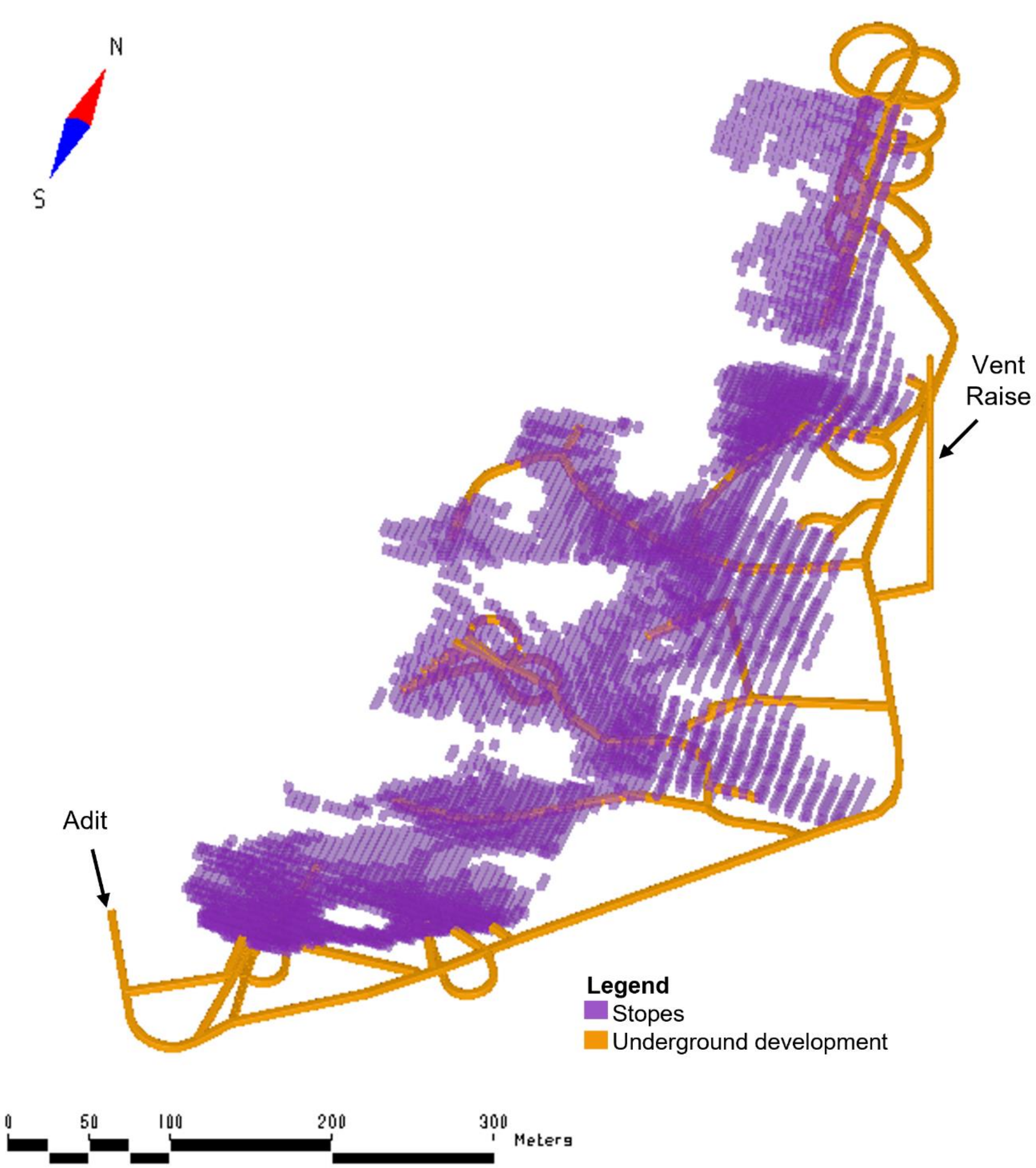
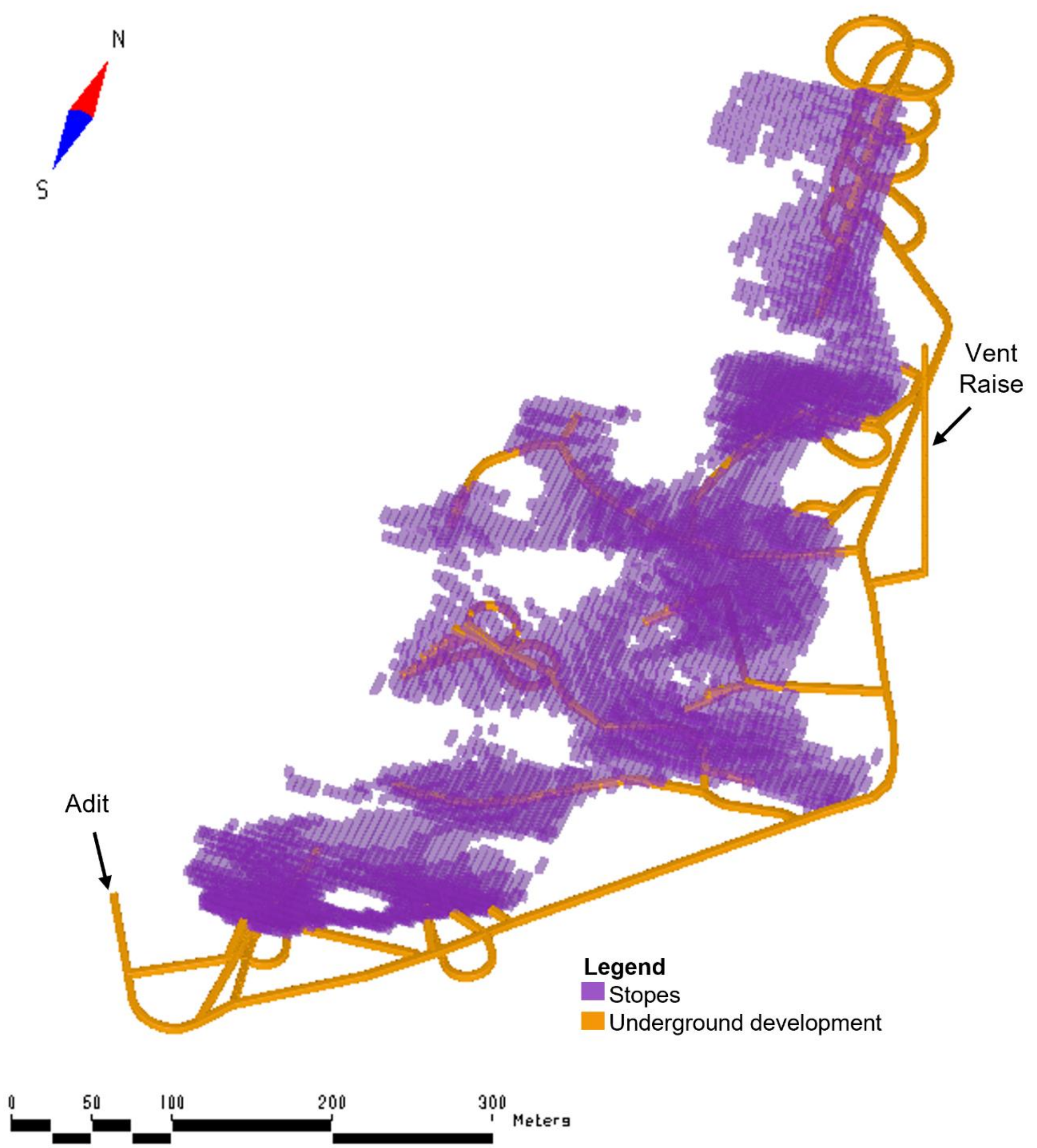


Figure 24.10 – Production Year 5 (Cumulative)



24.7.8 Mobile Equipment

Table 24.8 presents the estimated equipment requirements to support the mine plan. Some redundancy is included to accommodate breakdowns and ramp congestion.

Table 24.8 – Mobile Equipment Requirements

Primary Equipment	Number Required
LHD	10
Trucks (40 t)	5
Development Drilling Rig	6
Rock support Drilling Rig	6
Total Fleet	*27*

24.7.9 Manpower

The underground manpower requirements are outlined in Table 24.9. The mine will operate on a 2 shifts per day, 12 hours per shift basis, 7 days a week.

Table 24.9 – Manpower per Shift

Description	Number Required
Equipment Operators	27
Maintenance Team	7
Services Team	6
Supervision	3
Manpower per Shift	*43*
Mining Manager	1
Engineering Manager	1
Technical Services	6
Supervision	*7*
Total	*51*

24.7.10 BACKFILL

Cemented Rock Fill (CRF) is recommended to reduce the width of the exposed roof span and thus guarantee the stability of the stopes. CRF is a mixture of run-of-mine waste rock mixed with cement slurry. Sump mixing with a loader offers the lowest cost form of exposable backfill available for small stoping operations. The backfill delivery requirements have been estimated to be 663 m^3 per day. To minimise any backfilling delays, a stockpile of loose rockfill will be maintained underground near each stope.

24.7.11 VENTILATION

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

The main fresh air intake is the decline, while air will be returned to surface (return air) via a 'pull' system with primary exhaust fans located at the surface, at the top of the primary exhaust raise. The exhaust raise will have a diameter of 3.5 m. The airflow required for each working area will be distributed via 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.

24.7.12 EMERGENCY EGRESS

The main decline will be the escape route to surface, and the main ventilation raise will be equipped to serve as a secondary egress.

24.7.13 UNDERGROUND OPPORTUNITY ECONOMICS

24.7.13.1 Basis of Economic Estimate

The underground economics were modelled as an extension of the open pit model. Commencement of underground production was adjusted into production Year ten (Y10), a partial open pit production and closing year, to present continuous mill feed transition from open pit to underground. The models were kept separate and the Y10 underground tonnages are not reflected in any of the open pit economics presented in Section 22.

The underground cashflow was discounted to the start of the open pit economics for comparison purposes.

The estimated operating cost for the Curipamba underground mine is $75.58/t of mill feed exclusive of processing and G&A costs. DRA has estimated the operating cost based on in-country contractor and supplier quotations, industry benchmarking, proprietary information, and its professional experience. Open pit phase operating costs for process and support infrastructure was maintained for the U/G portion to present a conservative Opex scenario. Opportunity exists to refine the UG Opex process and G&A costs as partial open pit mine and waste facility closures will be completed during production year 11. (Y11). For the initial economic analysis, closure costs were moved to the end of the underground mine life period, assuming the requirement to continue utilising the waste facilities during the underground period. Open pit mine dewatering and treatment was also included as the open pit will only be flooded upon completion of the underground operations.

The presented economic numbers were determined through a difference method: The additional difference in created value by the underground extension past the initial open pit phase.

24.7.13.2 Caution Statement

The preliminary economic assessment is preliminary in nature and 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 preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

24.7.13.3 Economic Results

Summary economic results are presented in Table 24.10, Figure 24.11, Figure 24.12 and Figure 24.13.

Table 24.10 – Underground Opportunity Economic Results

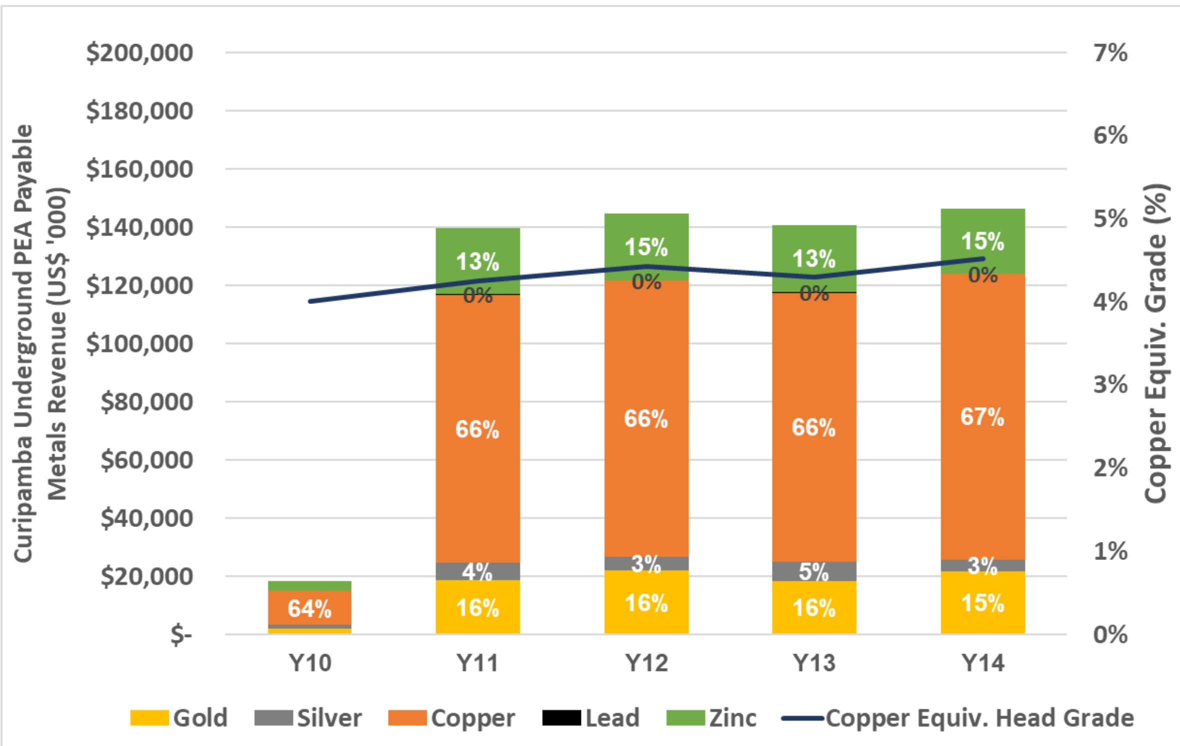
	PEA Base Case	-15% Price Deck	Spot Prices as at Oct. 19, 2021
After Tax Undiscounted Cashflow ($M)	$138	$77	$260
After-Tax NPV ($M, 8% discount rate)(1)(2)	$49	$27	$93
Total development capital for underground ($M)(3)	$42		
Nominal processing capacity (tpd)	1,850		
Average annual payable production (Years 11-14)(4)(5)	Cu= 14 kt		
	CuEq= ~20 kt	CuEq= ~21 kt	CuEq= ~19 kt
Metal prices assumed	$1,700.00 /oz Au	$1,445.00 /oz Au	$1,766.00 /oz Au

	$23.00 /oz Ag	$19.55 /oz Ag	$23.29 /oz Ag
	$3.50 /lb Cu	$2.98 /lb Cu	$4.72 /lb Cu
	$0.95 /lb Pb	$0.81 /lb Pb	$1.10 /lb Pb
	$1.20 /lb Zn	$0.98 /lb Zn	$1.70 /lb Zn

Notes:

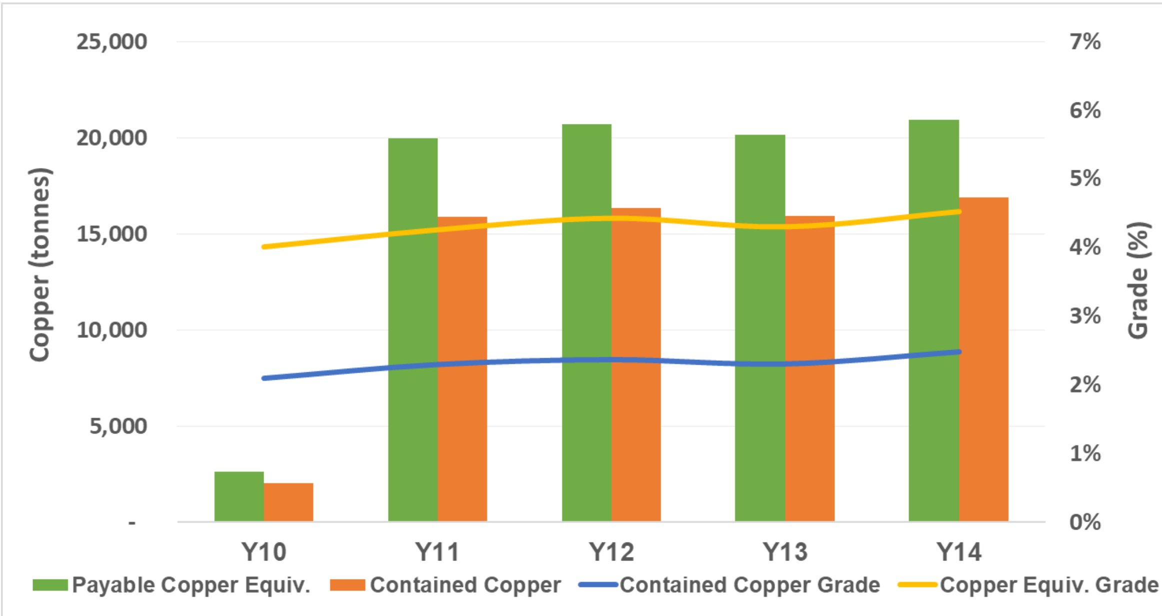
1. *All currencies are reported in US dollars on a 100% project basis.*
2. *Underground net present value ("NPV") calculations are discounted back to Y-2 of the open pit LOM for consistency*
3. *Capital cost estimate is based on DRA in-house estimates and benchmarking, inclusive of 12% VAT (~$4.5M)*
4. *CuEq is calculated as follows: (Payable Metals NSR Ag,Zn,Pb,Au, Ag)/(Payable Metals NSR Cu)* (Payable Copper t)*
5. *Underground production in Year10 (Y10) is excluded from the average as it is a partial year.*

Figure 24.11 – Underground Opportunity Revenue by Metal



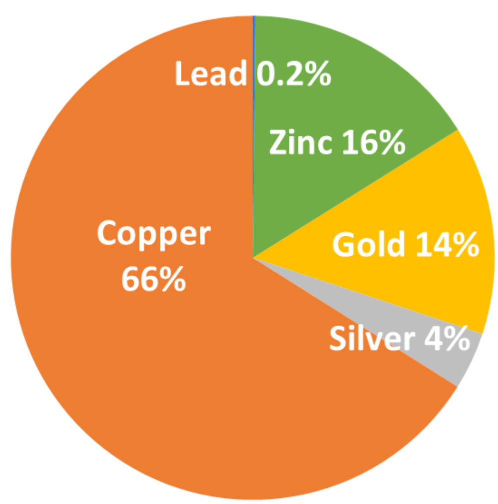
Source: DRA, 2021

Figure 24.12 – Underground Opportunity Annual Copper and Copper Equivalent Production



Source: DRA, 2021

Figure 24.13 – Underground Opportunity Life-of-Mine Revenue by Payable Metal



Source: DRA, 2021

24.7.14 Underground Opportunity Conclusions and Recommendations

Initial evaluation of the underground opportunity indicates a positive economic contribution to the project and extended use of fixed capital invested in the open pit phase. More value is also gained from the available metal resources toward optimising the resource potential. Further technical work is required to determine the feasibility of the underground mine. It is recommended to continue with planned infill drilling programs. Continue with rock mechanic drilling, interpretations and design to confirm mine stability and study potential geological faults present in the rock. Further progression of the mine design into PFS or FS phase will improve definition and cost estimating. Improving mine scheduling and integration with the open pit mining plan may produce opportunities to commence earlier with underground mining and provide blended material to the mill toward optimising metallurgical recoveries.

25 INTERPRETATION AND CONCLUSIONS

25.1 Geology

- Adventus was successful in securing surface access rights from affected landowners:
  - Adventus was able to pursue drilling on all exploration targets without encumbrances.
  - Curimining actively engaged in good community relations in order to carry out exploration effectively.
- Drilling has identified a stratiform and largely stratabound horizon of semi-massive to massive sulphide mineralization with an overlying zone of brecciated/fragmented sulphide fragments. Additional mineralization occurs in smaller lenses primarily in the footwall of the massive sulphide mineralization.
- The qualified person (QP) has reviewed procedures for drilling, sampling, sample preparation, and analysis and has concluded that those procedures are appropriate for the type of deposit and mineralization.
- The QP has reviewed and analyzed the analytical quality control data collected by Curimining and has not found any material issues with the data. As a result, the QP has concluded that the exploration data are of sufficient quality for Mineral Resource estimation.
- Open pit Mineral Resources were estimated and reported using a US$29/t NSR cut-off value, and the underground portion of the Mineral Resources is reported with mining shapes which were generated using a $105/t NSR cut-off value.
  - Mineral Resources in the Measured category contain 3.2 Mt averaging 3.03 g/t Au, 45 g/t Ag, 2.61% Cu, 0.20% Pb, and 2.50% Zn.
  - Mineral Resources in the Indicated category contain 9.0 Mt averaging 2.36 g/t Au, 45 g/t Ag, 2.11% Cu, 0.24% Pb, and 2.59% Zn.
  - Mineral Resources in the Inferred category contain 1.1 Mt averaging 1.62 g/t Au, 32 g/t Ag, 172% Cu, 0.14% Pb, and 1.71% Zn.
- The differences in the 2019 and 2021 Mineral Resource inventory are mostly due to the following:
  - Higher processing and mining costs resulting in a smaller reporting shell in 2021
  - Underground Mineral Resources reported within mining shapes resulting in smaller underground inventory in 2021
  - Transfer of the model from Gems Percent to Leapfrog Sub-block
  - Increase in the Measured category due to new drilling

- ⁄ Hard boundaries applied between volcanogenic massive sulphide (VMS) fault domains
- ⁄ Higher NSR cut-off values for the open pit mining method ($29/t versus $25/t) in the current resource
- ⁄ A drop in the metal grades due to higher NSR factors
- ⁄ Improved density determination

- Several mineralized lenses are supported by limited drilling. Additionally, a few mineral targets currently outside of the resource area of the El Domo deposit are supported by limited drilling.

25.2 Mineral Processing and Metallurgy

The test work conducted sufficiently supports the feasibility study process design and identified the physical and metallurgical properties of the orebody ore and ore variability within the deposit. The drill core material used for the test work is representative of the deposit and is well documented with detailed records appended in the relevant test work reports.

The physical and mineralogical characteristics of the and chemical composition ore, flotation feed, final concentrate and tailings samples have been studied with recognized standard techniques which produced results required to develop an approach to the metallurgical process design.

Extensive comminution work has been completed during 2020 and 2021 and included SMC, and Bond testing (BWi, RWi, and Ai) for the composites and variability samples.

The detailed flotation test work program has been completed during 2019-2021. The reported results were thoroughly analysed during the test work reporting period and present a substantial volume of data to justify flotation process for the ore within the deposit and expected variability.

The flotation test work completed during 2020 and 2021 corresponds well with the results of the previously completed 2019 program, and earlier programs, and supports the production of copper concentrate with a grade of 27% Cu, a lead concentrate grade around 34% Pb, and a zinc concentrate grade higher greater than 56% Zn at recoveries of copper, lead and zinc of 84.3%, 29.0%, and 81.9% respectively.

Optimization of the lead cleaner flotation by addition of the CMC dispersant allowed to generate a 45% Pb grade lead concentrates which brings substantial upside to the concentrate sales.

Copper concentrate grade higher than 30% is achievable at recovery exceeding 80%, and copper recovery of 89% demonstrated at copper concentrate grade of 27%.

Zinc flotation produced high grade concentrates during the LCTs with demonstrated above 59% Zn grade at 79% recovery, and zinc recoveries of 91% at 52% Zn concentrate grade, and 90% recovery at 57.8% concentrate grade.

The copper, lead, and zinc concentrates produced have contained gold and silver premiums which contribute to the saleable concentrate price. Approximately 32% the silver and 35% of the gold is anticipated to be recovered into the copper concentrate whilst 6% of silver and to 1.4% of gold recovered report to the lead concentrate with the remaining 15% of the gold and 25% silver recovered distributed to the zinc concentrate.

A gold deportment study identified that 30% to 70% of gold, depending on lithology, is present as fine free gold whilst the rest of the gold is locked with pyrite in “solid solution”. The free or visible gold is cyanide leachable based on the results of the tests. Currently, the pyrite locked gold would require flotation pre-concentration followed by oxidation of the sulphides present to release the gold for leaching (roasting, bioleach, pressure oxidation) to recover the remainder of the gold. Pyrite flotation tests conducted in 2019 demonstrated low precious metal recovery performance levels. Processes for the pyrite flotation concentrate oxidation processes are expected to be cost prohibitive for the project but should be investigated in the future.

Presence of the cyanide soluble copper in a potential hydrometallurgical plant feed (zinc rougher tailings) complicates hydrometallurgical extraction of the gold due to the very high cyanide and lime demand.

SART process applicability was tested and demonstrated some potential for copper recovery and cyanide recycle, however, evaluation of the foreseen economic benefits and associated health and safety, and technical risks have concluded that the health and safety, commissioning, and operational risks, and capital and operating expenditures of this plant currently outweigh the foreseen benefits of the additional metals recovery. Future testing results, technology development, and/or metal price environment may change the viability of a SART hydrometallurgical plant at Curipamba.

Gravity test work conducted on the samples of ore, and later, on samples of the bulk flotation concentrate using bench scale Knelson concentrator followed by panning for cleaning, and with the cleaner tailings tested using the Mozley table. In both cases results were poor with a gold recovery not exceeding 25% due to the large quantity of gold being either very fine or in a solid solution within the pyrite, and, therefore, in both cases, not amenable for gravity concentration.

Ore aging tests completed in 2021 on composite samples in order to evaluate any impact of the stockpiling on the ore flotation response have concluded that there is no material difference in flotation response between the aged and non aged samples and the ore can be stockpiled without damage to metallurgical performance.

Final tailings settling tests corresponded well to the previously produced results indicating challenging settling properties of the tailings attributed to the presence of the fine grained aluminosilicates. Settling rate improves with increased dosage of the flocculant.

Tailings pressure filtration tests demonstrated a low filtration rate and elevated (25% and higher) moisture of the filter cake. These results point on a considerable capital expenditure and operating cost of the final tailings dry stacking.

25.3 Mineral Reserves

The Mineral Reserves for the Project are estimated at 3.14 of Proven Reserves at an average grade of 2.83 g/t Au, 41.42 g/t Ag, 0.21% Pb, 2.30% An and 2.50% Cu (for an average NSR of $207.94) and 3.34 Mt of Probable Reserves at an average grade of 2.23 g/t Au, 49.70 g/t Ag, 0.29% Pb, 2.68% Zn and 1.39% Cu (for an average NSR of $164.29). To access these reserves, 55.34 Mt of waste will need to be removed, for an overall strip ratio of 8.54 to 1 (tonnes of waste to tonnes of ore).

25.4 Mining Methods

The mine plan for the Curipamba Project is based on the Minerals Reserves Estimate of 6.48 Mt of ore and 55.34 Mt of waste, for an overall strip ratio of 8.54. The ore is split into three types: 21% is high zinc, 58% is mixed zinc/copper, and 21% is high copper. The mine will operate year-round, seven (7) days a week, 24 hours a day with two (2) shifts of twelve (12) hours. Fifteen days of weather delays are considered.

Based on a 666 kt a year production target at the mill, an average of 4.55 Mt/year of material will be mined during the 10-year life of mine, in addition to a 1.5 year pre-production period where 16.37 Mt of material will be extracted.

The mine will be operated by a contractor using a fleet of 10 x 6 trucks (45 t), paired with excavators (3.3 m^3) and loaders (3.9 m^3).

25.5 Recovery Methods

A concentrator plant capable of handling all three types of Curipamba ore has been designed during the feasibility study, and based on extensive test work and pilot plant data.

A conventional sequential flotation circuit has been selected to produce separate coper, lead and zinc concentrates using a robust and flexible process flowsheet. The test work has been optimised to produce saleable concentrates.

Process design has had sufficient information sourced from the test work summarised in Section 13, equipment vendor data, information provided by Adventus, and DRA in-house design data.

A skid-built crushing circuit has been selected rather thana stationary crushing circuit to reduce capital expenditure. These units will also be used for aggregates production during the plant construction.

The grinding circuit uses a single stage grate discharge ball mill in closed circuit with the classification and cyclones.

The 2019-2021 test work results suggested an opportunity to increase a bulk rougher flotation feed size (P_{80}) from 75 to 125 microns.

Vertical stirred mill with ceramic media is suggested for the bulk concentrate regrind.

Tank cell technology was recommended for the bulk, copper, and zinc rougher flotation circuits to ensure a high unit equipment throughput, metals recovery, and control over the process flow.

Control over the cleaner feed grade and tonnage was offered by introducing a system of concentrate re-direction either to the copper circuit of to the zinc circuit by means of the launders and mining hoses in the bulk rougher flotation.

Conventional forced air mechanical cells were recommended for the cleaner circuits and represent a proven low-cost technical solution.

High-rate steel tank bolted thickeners were chosen for final tailings and concentrates settling and water recovery to provide an optimal capital cost and reduce equipment installation time. Any solids carryover from the tailings thickener will be captured within the process water settling pond with intermittent purge of the collected solids into the final tailings tank by means of the dedicated pump.

Vertical plate and frame filter presses were recommended for the concentrates dewatering as a conventional cost efficient equipment solution.

Water reticulation of the plant has been designed to minimize a raw water usage by means of the water reclaiming from the tailings dam. The process design resulted in freshwater usage which minimized the need for fresh water to less then 0.5 m^3 per tonne of mill feed ore.

25.6 Project Infrastructure

The project infrastructure designs have been sufficiently progressed for FS level estimation. The noted infrastructure is of suitable design for the site and size of operations.

Geotechnical and founding condition designs have been sufficiently progressed for an FS and relevant assumptions noted.

Sequencing of construction material production, site access, and mining activities remains important for successful execution.

Early construction of the site power infrastructure will be required to support the construction phase of the project. Temporary construction power generation on site may be needed.

Adventus may have the option to fund the 69 kV Echeandia-Las Naves upgrade on an incentive basis agreement with relevant authorities.

Adventus has secured an early works accommodation camp that will be established on the project site for early works preparation. Contractor accommodations will mostly likely be off-site.

Off-site infrastructure is suitable to support the Project. Local infrastructure and authorities may require additional support.

25.7 Market Studies

Preliminary market studies and potential off taker terms were obtained for all three concentrates that indicate saleable concentrates with multiple off-taker potential. Payables, penalties and refinement charges have been reviewed. Transportation and freight costs were evaluated and included for all three concentrates.

25.8 Environmental Studies, Permitting and Social or Community Impact

25.8.1 Environmental

An Environmental and Social Impact Assessment has been completed for the Project that will be submitted to the Ecuadorian government for review. It represents the first step in acquisition of the necessary permits to construct and operate the Project. The ESIA complies with Ecuadorian law, and details the baseline condition for biophysical and socio-economic factors. The ESIA also includes a substantial commitment to avoidance and mitigation of negative impacts.

The Project location is in an area that has largely been altered by previous agricultural and other human activities. Only small patches of original forest remain. Despite this, a number of important floral and faunal species have been identified which have been incorporated to monitoring and management planning. Most measured biophysical parameters, including air quality, noise, and vibration, fall below Ecuadorian limits.

Curimining has completed several years of community engagement activities to ensure that exploration activities and eventually mine development is understood locally. With a few exceptions, local communities accept the project, and recognize it as a source of employment and improvement on local infrastructure.

25.8.2 Waste Management

The designed facilities meet the mine waste storage requirements for the FS mine plan. Saprolite waste will be comingled with waste rock for improved stability in a dedicated Saprolite waste dump. A PAG waste rock facility (WRF1) is established above the TSF facility. Natural drainage from the Saprolite and WRF1 facilities will be contained in the lined TSF pond. The TSF will be established through the development of a downstream waste rock embankment that will contain mainly NAG material with some encapsulated PAG material within.

Mine Ore stockpiles will be required for the life of mine with a maximum size of 400 kt in year 9. The ore stockpiles will be located near the haul road between the open pit and processing plant. The ore remains reactive and acid generating. A suitably sealed area will be established to capture and channel run-off and seepage water toward the saprolite facility for collection and treatment in the TSF pond.

The disposal of other waste such as domestic, hazardous and biological waste will be properly stored and disposed of onsite in locations suitably designed to meet Ecuadorian requirements and offsite at registered disposal facilities.

25.8.3 Water Management

Water management infrastructure for each water catchment area consists of two main containment ponds namely: the TSF for the El Silencio basin and the Open Pit sump for the Naves Chico basin. Other related facilities and contact water drain into these facilities from where the water will be pumped to treatments plants prior to environmental discharge.

Mine material geochemistry was modelled and used to determine chemical characterisation and source terms for the various project elements. The geochemistry results indicated that the mine water is expected to be acidic and contain heavy metals that will require treatment prior to discharge. The source terms were also utilised to determine water quality predictions for the main contact water containment facilities from where water treatment will commence. Suitable water treatment plants were designed and estimated for the FS.

Additional water types such as potable water treatment and domestic and sewage water treatment was also addressed in terms of collection, containment as well as treatment and discharge. Potable water will be sourced from a borehole and treated for human consumption. Domestic and sewage water will be treated in a dedicated sewage treatment plant with clear liquid discharge to the TSF and solid waste to drying beds and tops oils stockpiles.

25.8.4 Mine Closure

A conceptual mine closure plan was developed that aimed to return accessible land to pre-mining condition and minimise long term active closure requirements. Active closure operations will commence in the final year of production and continue for an additional 2 year period. Water treatment may continue beyond this point depending on water quality improvements and stability.

The open pit will be closed with a pit lake that is supported by an embankment construction and operational phase water treatment plant. Other exposed areas of the mine pit will be suitably covered, sealed and rehabilitated to avoid long term acid generation. Water treatment will continue until the pit lake water quality improves and stabilizes below required discharge limits.

The waste facilities will be closed with final NAG outer layers of mine waste. The TSF beach will be reclaimed and revegetated, and a spill way introduced to control the TSF level and exposed beach head. Water treatment will continue until the TSF pond water quality is stabilized below discharge limitations.

25.9 Capital and Operating Cost Estimates

The Capital estimates was developed for each phase of the project. The initial capital expenditure is $247M which will be spent in the 18 month construction period prior to production. During production Sustaining Capex of $29M will be spent over the 10 year life of mine, following by a closure phase with estimated capex of $24M Prior to the commencement of the initial capital phase Adventus has planned an early works package of $22M that will prepare the site for construction, progress detailed design and lower project execution risk.

25.10 Economic Analysis

The cashflow analysis section has concluded a range of NPV and IRR results for the open pit Project. The model is based on input data from referenced sources within this Report, especially Section 21. Capex and Opex summaries were re-stated to provide clarity as to model inputs utilised.

The cash flow analysis output will be utilised for decision making and identification of opportunities to progress the Project into the next phase.

25.11 Other Relevant Data and Information

25.11.1 Project Schedule and Execution

25.11.1.1 Critical construction activities and Key Dates

The Project has developed a conceptual execution schedule suitable for the FS stage. Worldwide supply chain disruptions associated with the COVID-19 pandemic is extending fabrication and delivery times and must be noted as a risk. The commencement of construction activities remains dependent on the receipt of required permits from local authorities.

25.11.1.2 Project Implementation

The FS estimate considers the appointment of an Engineering, Procurement and Construction Management (EPCM) as the main contractor to provide the necessary technical, engineering design and construction experience to the Project. The FS planning and cost estimate reflects the EPCM execution model.

25.11.1.3 Construction Phase Quantities

The earthworks quantities are significant and will require a dedicated earthmoving fleet to complete the construction timeously. Phasing of the earthworks and mining excavations were considered for the FS to ensure sufficient competent waste rock production for co-mingling for the construction phase

25.11.1.4 Early Works

An early works package is planned to address critical construction phase support project components. The early works package has been costed and considered in the FS execution planning.

25.11.1.5 Construction Phase Water Supply and Management

Adventus hold current surface water extraction permits which is deemed adequate for the planned early works. Application for further borehole extraction permits required for construction phase. The construction phase will require contact water containment and treatment capabilities.

25.11.1.6 Construction Phase Power Supply

Adventus will be relying on grid power as far as possible. Basic 7.69kV power line infrastructure is present on site and is deemed sufficient for basic power supply during early works but is not deemed adequate for the construction phase of the project. Construction power will require the upgrading of local power infrastructure, or supplementation with generator power until the main 69kV power supply is commissioned on site.

25.11.2 ORP Gap Analysis and Planning

An FS level ORP gap analysis was performed to commence ORP for the project. An initial ORP plan was developed for implementation during the next project phase. The ORP is deemed sufficient for the level of study.

25.11.3 Project Risks and Opportunities

Major risk and opportunities were noted for the FS along with recommendations and actions for the next phase.

25.11.4 Underground Expansion Opportunity

A Preliminary Economic Assessment (PEA) level study (±50% accuracy) that was conducted to evaluate the merits of mine life extension through further underground mining following end of the open pit mine life

Initial evaluation of the underground opportunity indicates a positive economic contribution to the project at PEA level that requires further definition.

26 RECOMMENDATIONS

Considering the positive outcome of this Report, it is recommended to pursue the next phase of the Project though various aspects need to be monitored or done and are listed below.

26.1 Mining

For the next phase of the Project, DRA recommends further drilling of the saprolite and overburden layers to improve understanding of the material proportions, depth and thickness. The results of this drilling may help reduce the size of the related stockpiles and waste dumps. In addition, it could help reduce the extents of the mine footprint by improving current shallow saprolite slopes.

DRA also recommends studying the possibility of in-pit ore and waste stockpiling in the mined-out portions of the mine to minimize the haulage distances, ex-pit stockpile and waste dump sizes, and optimize the environmental footprint of the operation. In additional, further study of the saprolite stockpile stability and its suitability as a base for the ore stockpiles is required.

Different ore mining schedules, such as mining ore only during the day shift or only during weekdays, can be evaluated. A trade-off study between these ore mining schedules and their stockpile size requirements at the mill compared to the current proposal will need to be completed.

Additionally, DRA recommends a further review of contractor capabilities and the early on-boarding and involvement of the chosen contractor in the next phase of the Project. Collaboration with the contractor will allow for equipment selection optimization (for example, evaluating a mixed truck fleet, different drills for ore and waste, etc.), drill pattern optimization for the different material types, scheduling optimization, etc.

26.2 Mineral Processing and Metallurgy

Variability comminution tests (Bond suite and SMC) recommended on the newly produced ore samples to increase supplier confidence for ball mill sizing and contribute to the vendor performance guarantee.

Fine grinding tests are recommended to provide additional information to the regrind mill suppliers. A sample of the bulk flotation concentrate should be produced by the Base Metals lab to satisfy the quantity of material required for the tests. The fine grinding test work should be coordinated with the equipment suppliers. The sizing of the regrind mills will most likely be a combination of the test results, supplier equipment knowledge, and previous experience with similar ore types and lithology.

It is recommended to continue on batch cleaner and LCTs on the newly sampled material - variability and composites to produce concentrates for marketing purposes, and increase flotation supplier confidence for the sizing and contribute to the vendor performance guarantee.

Settling and rheology and filtration tests recommended on samples of tailings and concentrates from the flotation work. The tests would preferably be conducted by equipment suppliers.

Material handling tests and material property tests (cohesive strength; wall friction; compressibility; chute angles; angle of repose, bulk density (loose/packed)) recommended to generate sufficient data for detailed design of conveyors, chutes, bins, and hoppers. A specialized testing contractor such as Jenike and Johanson (or equivalent) should be contracted for the work.

26.3 Recovery Methods

During the next phase of the Project, it is recommended to update the process mass and water balances with the new test work data produced from the testing campaign executed post feasibility study, any new information with regard to the mining plan, and any new (post FS) data provided by the equipment suppliers.

When evaluating the purchase of the mechanical and electrical equipment, emphasis should be placed on the possible minimising of equipment suppliers in order to obtain the lowest pricing, the interchangeability of parts and components and service plans.

26.4 Site Infrastructure

The site infrastructure was developed and sized for the FS level estimation. Further detailed design development may reduce sizing and improve terrace sizing during detailed design.

The infrastructure requires further geotechnical and survey information to progress and optimize designs and estimates.

Site accommodation strategy during the construction and operations will benefit from further development and involvement of suitable catering service providers in evaluating further opportunities.

Engagement with local authorities on power supply will progress in the next stage and clarity gained on funding requirements. Telecommunication and fibre optic internet service providers will be engaged as part of the electrical infrastructure design.

Off-site infrastructure uses can be evaluated to reduce risk and create opportunities for the Project.

26.5 Environmental Studies, Permitting and Social or Community Impact

It is recommended to complete the permitting process initiated with the submission of the ESIA in order to obtain the approvals necessary to initiate construction. Concomitant with this, ongoing environmental and community relations monitoring and activities should be maintained.

26.5.1 Waste Management

The waste facility designs must progress from the FS designs with further assessments that include: updated dam break analysis, updated hydrogeology, further field investigations, further waste rock testing, tailings testing, saprolite material testing for closure purposes.

Further facility designs for domestic waste and top soil stockpiles will be progressed.

26.5.2 Hydrogeology

For improved understanding of site hydrogeology, groundwater flow patterns and water quality, as well as filling the identified data gaps, in both the Naves Chico and El Silencio Valleys:

- Future drilling to characterize groundwater flow patterns in the pit area. Estimation of hydraulic gradients and hydraulic conductivities of the faults will serve to refine the groundwater inflow estimate to the future pit and underground workings.
- Additional geological mapping is recommended to determine if there is presence of additional faults, which could change the water transport conditions.
- To characterize the hydraulic conductivities and continuity of the fault network, pumping wells should be installed at the intersection of two or more faults and pumping test developed to refine the pit inflows estimate and facilitate development and calibration of a numerical groundwater model.
- Installation of additional piezometers is recommended to characterize groundwater quality adjacent the pit for purposes of understanding groundwater inflows treatment requirements and complete a detailed anions and cations balance.
- Continue and update flow rates measurements on the streams.
- Further vertical hydraulic conductivity tests is recommended in the residual soils and saprolite in the Naves Chico and El Silencio valleys.

26.5.3 Geochemistry

The current geochemical test work results were based on a relatively small test sample of available material. for some lithological units. A more comprehensive test work program is recommended during the next project phase to increase accuracy and acid potential determination of the various lithologies.

26.5.4 Water Quality and Treatment

It is recommended that further geochemical test work be undertaken in the next project phase to improve the understanding of PAG and NAG lithologies and subsequently improve water quality model predictions. Further detailed design development and characterized waste rock placement sequencing can improve the outer layer composition of the facilities and subsequently improve water quality. Tailings acid generation potential indicates that cleaner tailings are highly acid generating compared to rougher tailings; it is recommended to review if cleaner tailings can be deposited separately into permanently submerged TSF areas to prevent oxidation. Further testing and determination of process water quality will improve water quality predictions. Inclusion of pH correction dosing in the process plant must be explored as part of a circulatory process/TSF water system. A balanced surface water quality model must be established during the next project phase that will improve input into water treatment designs. Base line water quality data must continue to be collected to provide longitudinal baseline information for discharge comparisons.

26.5.5 Mine Closure

The current conceptual closure plan described here is at a FS level. A formal mine closure plan will be prepared as part of the next project phase and updated periodically throughput the mine life. This report will list the currently developed closure concepts that was utilised in the FS. It is recommended to develop a detailed and phased closure plan during the next project phase to improve early rehabilitation of the mine site and reduce contact water surfaces throughout the life of mine.

26.6 Other Relevant Data and Information

26.6.1 Operational Readiness Plan

Skilled construction and operational personnel may be difficult to source in country, and specifically local to the mainly agriculturally based project location. The early detailed development and implementation of the operational readiness plan will benefit the Project and reduce human resource risk.

26.6.2 Construction Power Supply

Adventus will be relying on grid power as far as possible. It is recommended to progress a detailed study of local infrastructure and evaluate the potential to upgrade and utilise it for construction power. Back-up generation is still recommended to support the construction effort.

26.6.3 Project Risk And Opportunities

Risk and opportunity registers were maintained along recommendations for the next phase with main risks and opportunities noted.

26.6.4 Underground Opportunity Recommendations

The underground opportunity indicates positive preliminary economic assessment results. It is recommended to continue with planned infill drilling programs, and rock mechanic drilling, interpretations, and design to confirm mine stability and study potential geological faults present in the rock. Completing a PFS- or FS-level study will improve definition and cost estimating. With further infill drilling and geotechnical data, the mine design and schedule can be improved. In addition, trade-off studies regarding the underground mine commencement (i.e. gradual transition from open-pit to underground versus both operating independently) can be completed.

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28 CERTIFICATE OF QUALIFIED PERSON

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CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*"NI43-101 Technical Report - Feasibility Study – Curipamba, El Domo Project, Central Ecuador"* which is effective as of October 26, 2021 and issued on December 10, 2021 (the Technical Report) prepared for Adventus Mining Corporation (Adventus or the Company).

I, *Philippus (Philip) de Weerdt, Pr.Eng, PMP, MBA.*, do hereby certify:

1. I am a Project Manager, with DRA Global Limited with an office at 20 Queen Street West, 29th Floor, Toronto, Ontario, Canada.
2. I am a graduate from University of Johannesburg, Johannesburg, South Africa in 2006 with a B. Eng Mechanical, and graduate of the University of Pretoria, Pretoria in 2016 with a Masters in Business Administration.
3. I am a registered member of the Engineering Council of South Africa (#20150058). I am a register Project Management Professional (PMP).
4. I have worked continuously as a Mechanical Engineer and Project Manager since my graduation.
5. My relevant work experience includes:
   - Design and construction execution of mining related processing facilities and infrastructure in Western and Central Africa.
   - Project management and oversight of design, estimating and execution of mining related processing facilities and infrastructure in Western Africa.
   - Participation and author of NI 43-101 Technical Reports.
6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
7. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.

8. I am responsible for the preparation and author of Sections 2 to 3, 16.4, 18, 19, 20.9.4 to 20.9.8, 20.10.1, 20.10.4, 21 to 24, with the exception of Sections 18.14, 18.15, 21.3.3, 21.3.4, and 24.7. I am also responsible for the relevant portions of Sections 1, 25 to 27 of the Technical Report.

9. I personally did visit the property that is the subject to the Technical Report on July 28 to 30, 2021.

10. I have had no prior involvement with the property that is the subject of the Technical Report.

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

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

Dated this 10th day of December 2021

"Original Signed and sealed"
Philip de Weerdt, Pr.Eng, PMP, MBA
Project Manager
DRA Global Limited

SLR Consulting (Canada) Ltd
55 University Avenue, Suite 501,Toronto, ON M5J 2H7

CERTIFICATE OF QUALIFIED PERSON

I, *Dorota El Rassi, M.Sc., P.Eng.*, as an author of this Report entitled "*NI43-101 Technical Report - Feasibility Study – Curipamba, El Domo Project, Central Ecuador"* which is effective as of October 26, 2021 and issued on December 10, 2021 (the Technical Report) prepared for Adventus Mining Corporation (Adventus or the Company), do hereby certify that:

1. I am a Consultant Geologist with SLR Consulting (Canada) Ltd, of Suite 501, 55 University Ave., Toronto, ON, M5J 2H7.
2. I am a graduate of the University of Toronto in 1997 with a B.A.Sc. (Hons.) degree in Geological and Mining Engineering and in 2000 with a M.Sc. degree in Geology and Mechanical Engineering.
3. I am registered as a Professional Geological Engineer in the Province of Ontario (Reg.# 100012348).
4. I have worked as a geologist for a total of 21 years since my graduation.
5. My relevant experience for the purpose of the Technical Report is:
   - Review and report as a professional engineer on numerous mining and exploration projects for due diligence and regulatory requirements;
   - Mineral Resource estimates on a variety of commodities including gold, silver, copper, nickel, zinc, PGE, and industrial mineral deposits;
   - Experienced user of Gemcom, Leapfrog, Phinar's x10-Geo, and Gslib software.
6. 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.
7. I visited the property that is the subject to the Technical Report from January 8 to 10, 2019 and from July 28 to 30, 2021.
8. I am responsible for the preparation of Sections 4 to 12, 14, and 23. I am also responsible for the relevant portions of Sections 1 and 25 to 27 of the Technical Report.
9. I am independent of the Company applying the test set out in Section 1.5 of NI 43-101.
10. I have had prior involvement with the property that is the subject of the Technical Report:
    - "*NI 43-101 Technical Report on the Preliminary Economic Assessment for the Curipamba Project – El Domo Deposit, Central Ecuador*" with an effective date of May

2, 2019 (issued Report Date: June 14, 2019), prepared for Adventus Mining Corporation.

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

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

Dated this 10th day of December 2021

(Signed and Sealed) ***Dorota El-Rassi***

Dorota El-Rassi, M.Sc., P.Eng.
Consultant Geologist
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I, *Daniel M. Gagnon, P. Eng.,* do hereby certify:

1. I am Vice President Mining, Geology and Met-Chem Operations, with DRA Global Limited located at 555 René Lévesque West, 6th Floor, Montreal, Quebec Canada H2Z 1B1.
2. I am a graduate of École Polytechnique de Montréal, Montreal, Quebec, Canada in 1995 with a bachelor degree in Mining Engineering.
3. I am registered as a Professional Engineer in the Province of Quebec (Reg. #118521).
4. I have worked as a Mining Engineer for a total of 26 years continuously since my graduation.
5. My relevant work experience includes:
   - Design, scheduling, cost estimation and Mineral Reserve estimation for several open pit studies similar to Curipamba in Canada, the USA, South America, West Africa, and Morocco.
   - Technical assistance in mine design and scheduling for mine operations in Canada, the USS, and Morocco.
   - Participation and author of several NI 43-101 Technical Reports.
6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
7. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
8. I am responsible for the preparation of Sections 15 and 16, with the exception for Sections

16.3 and 16.4. I am also responsible for the relevant portions of Sections 1, 21.3, and 25 to 27 of the Technical Report.

9. I personally did visit the property that is the subject to the Technical Report on July 28 to 30, 2021.

10. I have had no prior involvement with the property that is the subject of the Technical Report.

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

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

Dated this 10th day of December 2021

"Original Signed and sealed"

Daniel M. Gagnon, P. Eng.
VP Mining, Geology and Met-Chem Operations
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I, *Claude Bisaillon, P. Eng.,* do hereby certify:

1. I am Senior Geotechnical Engineer with DRA Global Limited located at 555 René Lévesque West, 6th Floor, Montreal, Quebec Canada H2Z 1B1.
2. I am a graduate from Concordia University in Montreal in 1991 with a B.Sc. in geology and from the Université Laval in Quebec City in 1996 with a B.Ing. in geological engineering.
3. I am a registered member of "*Ordre des Ingénieurs du Québec*" (#116407). I am a Member of the Canadian Institute of Mining, Metallurgy and Petroleum.
4. I have worked as an engineer continuously since graduation from University in 1996.
5. My relevant work experience includes:
   - Over 23 years of consulting in the field of Mineral Resource estimation, orebody modelling, mineral resource auditing and geotechnical engineering similar to Curipamba in Canada, the USA, Asia, and South America.
   - Participation and author of several NI 43-101 Technical Reports.
   - QP Review, audits, due diligence, interpretation of geoscientific data for several projects.
6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
7. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
8. I am responsible for the preparation of Section 16.3. I am also responsible for the relevant portions of Sections 1, and 25 to 27 of the Technical Report.

9. I personally did visit the property that is the subject to the Technical Report on July 28 to 30, 2021.

10. I have had no prior involvement with the property that is the subject of the Technical Report.

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

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

Dated this 10th day of December 2021

"Original Signed and sealed"
Claude Bisaillon, P. Eng.
Senior Geotechnical Engineer
DRA Global Limited

20 Queen Street West / 29th Floor / Toronto / Ontario / Canada / M5H 3R3
T +1 416 800 8797 / **E** info@draglobal.com / https://www.draglobal.com/

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI43-101 Technical Report - Feasibility Study – Curipamba, El Domo Project, Central Ecuador*" which is effective as of October 26, 2021 and issued on December 10, 2021 (the Technical Report) prepared for Adventus Mining Corporation Inc (Adventus or the Company).

I, *Volodymyr Liskovych, Ph.D., P. Eng.*, do hereby certify:

1. I am a Principal Process Engineer, with DRA Global Limited with an office at 20 Queen Street West, 29th Floor, Toronto, Ontario, Canada.
2. I am a graduate from Zaporizhzhia State Engineering Academy, Zaporizhzhia, Ukraine in 1996 with a Metallurgical Engineer Degree, and a graduate from National Metallurgical Academy of Ukraine, Dnipro, Ukraine with the PhD degree in Metallurgical Engineering in 2001.
3. I am a registered member of the Professional Engineers of Ontario (#100157409).
4. I have worked continuously as a Metallurgical Engineer for more than *24* years since my graduation from Zaporizhzhia State Engineering Academy.
5. My relevant work experience includes:
    - Review and report on mineral processing and metallurgical operations and projects around the world for due diligence and regulatory requirements;
    - Engineering study (PEA, PFS, FS, and Detailed Engineering) project work on many minerals processing and metallurgical and hydrometallurgical projects around the world, and in North America;
    - Operational experience in operations management and operational support positions in metallurgical and hydrometallurgical operations in Ukraine, Canada, and Brazil.
    - Participation and author of several NI 43-101 Technical Reports.
6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
7. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.

8. I am responsible for the preparation of Sections 13 and 17. I am also responsible for the relevant portions of Sections 1, 21.3, and 25 to 27 of the Technical Report.
9. I did not visit the property that is the subject to the Technical Report.
10. I have had no prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 10th day of December 2021

"Original Signed and sealed"

Volodymyr Liskovych, P.Eng.
Principal Process Engineer
DRA Global Limited

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "NI43-101 Technical Report - Feasibility Study – Curipamba, El Domo Project, Central Ecuador" which is effective as of October 26, 2021 and issued on December 10, 2021 (the Technical Report) prepared for Adventus Mining Corporation (Adventus or the Company).

I, Brett Stephens, RPEQ, P.Eng., P.E., do hereby certify:

1. I am a Principal, Senior Geotechnical Engineer with Klohn Crippen Berger located at Level 1, 154 Melbourne Street, Brisbane QLD 4101, AUSTRALIA.
2. I am a graduate of the University of New South Wales in Sydney, Australia, from which I hold three engineering degrees:
   - Bachelor of Engineering (1990);
   - Master of Engineering Science (Geotechnical Engineering) (1994); and
   - Master of Environmental Engineering Science (2001).
3. I am a registered professional engineer in in Queensland, Australia (#6112), a professional engineer in Alberta, Canada (#80841), and in Minnesota, USA (#56768). I am a Fellow of the Australasian Institute of Mining and Metallurgy and a Member of the Institution of Engineers Australia (CPEng 948740).
4. I have worked continuously since my graduation as a Geotechnical Engineer for more than 30 years.
5. My relevant work experience includes:
   - Over twenty years of direct experience in the design, construction, and operation of tailings dams worldwide.
   - Designer of Record for more than 10 constructed tailings dams, and presently the Engineer of Record (EoR) for three operating tailings dams.
   - Direct involvement in preparation of engineering designs, investigations and analyses for tailings dam pre-feasibility and feasibility studies in Australia, and North and South America.
   - Participated in the external dam safety review, audit and third-party assessment of over 100 tailings storge facilities. I also participate in independent design review boards for various tailings dams in operation.
   - Participation and author of materials to be included in NI 43-101 Technical Reports.
6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education,

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affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.

7. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
8. I am responsible for the preparation of Sections 20.7, 20.8.1, 20.8.2. I am also responsible for the relevant portions of Sections 1, 25 to 27 of the Technical Report.
9. I personally visited the property that is the subject to the Technical Report (on October 21 to 22, 2021).
10. I have had no prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 10th day of December 2021

"Original Signed and sealed"
Brett Stephens, RPEQ, P.Eng, P.E.
Principal, Senior Geotechnical Engineer
Klohn Crippen Berger

pHase GEOCHEMISTRY

1032 Keith Rd. W., North Vancouver, BC V7P 1Y5
604.764.0854, www.phase-geochemistry.com

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI43-101 Technical Report - Feasibility Study – Curipamba, El Domo Project, Central Ecuador*" which is effective as of October 26, 2021 and issued on December 10, 2021 (the Technical Report) prepared for Adventus Mining Corporation (Adventus or the Company).

I, *Shannon Shaw, B.Sc., M.Sc., P.Geo (BC, NWT)*, do hereby certify:

1. I am President and Senior Geochemist with pHase Geochemistry Inc., located at 1032 Keith Rd. W, North Vancouver, BC, Canada V7P 1Y5.
2. I am a graduate from Queen's University (B.Sc.) in Kingston, Ontario Canada with a degree in Honour's Geological and Chemical Sciences (1994) and from the University of British Columbia (M.Sc) in Vancouver, British Columbia Canada with a degree in Geological Sciences (1996).
3. I am a registered member of Professional Geoscientist in British Columbia (# 32044) and the Northwest Territories in Canada (# L2929).
4. I have worked continuously since my graduation as a Geochemist for more than 25 years.
5. My relevant work experience includes:
    - Principal geochemical consultant on numerous mining projects in North and South America as well as Australasia.
    - Technical reviewer, for a specific issue or report as well as in the capacity as a member of a review panel for several mining projects
6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.

7. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
8. I am responsible for the preparation of Section 20.9.3 Geochemistry, with the exception of all other sections. I am also responsible for the relevant portions of Sections 1, 25 to 27 of the Technical Report.
9. I personally did not visit the property that is the subject to the Technical Report.
10. I have had no prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 10th day of December 2021

"Original Signed and sealed"
Shannon Shaw, B. Sc., M.Sc., P.Geo (BC, NWT)
President and Senior Geochemist
pHase Geochemistry Inc.

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CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*"NI43-101 Technical Report - Feasibility Study – Curipamba, El Domo Project, Central Ecuador"* which is effective as of October 26, 2021 and issued on December 10, 2021 (the Technical Report) prepared for Adventus Mining Corporation (Adventus or the Company).

I, *André-François Gravel, P. Eng., PMP.,* do hereby certify:

1. I am a Senior Mining Engineer with DRA Global. Limited, with an office at 555 René-Lévesque Blvd West, Montreal, Quebec, Canada.
2. I am a graduate of École Polytechnique de Montréal, Montreal, Quebec, Canada in 2000 with a bachelor degree in Mining Engineering.
3. I am registered as a Professional Engineer in the Province of Quebec (Reg. #123135).
4. I have worked as an Engineer in the Mining & Metals industry continuously since my graduation from university.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("N 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. My relevant work experience includes:
   - Design, scheduling, cost estimation and Mineral Reserve estimation for several underground and open pit studies similar to Curipamba in Canada, the USA, South America, Asia and West Africa.
   - Technical assistance in mine design and scheduling for mine operations in Canada, the US and Morocco
   - Participation and author of several NI 43-101 Technical reports
7. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
8. I have participated in the preparation of this Technical Report and am responsible for Section 24.7 and parts of Sections 1, and 25 to 27 of the Technical Report;

9. I have not personally visited the property that is the subject to the Technical.
10. I have had no prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 10th day of December 2021

"Original Signed and sealed"
Andre-Francois Gravel, P. Eng, PMP
Senior Mining Engineer
DRA Global Limited