102760-05-RPT-01
Revision Number 1

Bear Creek Mining
Corani Project

NI 43-101 Technical Report
17 December 2019

17 December 2019

CERTIFICATE of QUALIFIED PERSON

I, Gregory Searle Lane do hereby certify that:

1. I am currently employed as Chief Technical Officer by:

   Ausenco Services Pty Ltd, 144 Montague Road, South Brisbane, Queenland, Australia.

2. I am a graduate of University of Tasmania and received a Master of Science degree in Chemistry in 1987 and a Bachelor of Applied Science (App. Chem.) in 1981.

3. I am a FAusIMM in Australia # 203005.

4. I have practiced Process Engineering and Study Management for thirty years. I have worked for engineering companies for 25 years and for Ausenco Pty Ltd for 15 years.

5. 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.

6. I have not visited the Corani property.

7. I am responsible for the preparation of the following sections of the technical report titled Bear Creek Mining Corani Project NI43-101 Technical Report, dated December 17, 2019 (the "Technical Report"), relating to the relating to the Corani Silver-Lead-Zinc project in Peru.

SECTION	SECTION NAME
1	Summary
2	Introduction
3	Reliance on Other Experts
13.9	Interpretation
17	Recovery Methods
18 (except 18.2.6 and 18.3)	Project Infrastructure
19	Concentrate Market Studies and Contracts
21	Capital and Operating Costs
22	Economic Analysis
23	Adjacent Properties
24	Other Relevant Data and Information
25	Interpretation and Conclusions
26	Recommendations
27	References

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

9. I am Independent (as defined in NI 43-101) of Bear Creek Mining Corporation.

10. I have read NI 43-101 and Form 43-101F1 and confirm the sections of the Technical Report for which I am responsible (as listed in Item 7 above) have been prepared in compliance with that instrument and form.

11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this Seventeenth day of December, 2019

____
Signature of Qualified Person

Greg Lane
____
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Michael Meyer, do hereby certify that:

1. I am currently employed as Principal Scientist by:

Meyer EPS, Inc. and INSIDEO S.A.C.

2. I am a graduate of Davidson College and received a Bachelor of Science degree in Chemistry in 1989 and a Master of Science degree in Biochemistry and Biophysics from the University of North Carolina-Chapel Hill in 1992, and a Doctor of Philosophy degree in Wildlife Biology from Colorado State University in 1997.
3. I am a Registered Qualified Professional Member of the Mining and Metallurgy Society of America (N° 01377QP) in Environmental Permitting and Compliance.
4. I have practiced my profession of environmental management, compliance and permitting continuously for the past 24 years on mining projects in North America, Central and South America, Australia/Indonesia, Europe, and Africa. I have worked for a variety of environmental consulting firms, including the past 11 years for Meyer EPS, Inc.
5. 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.
6. I have not visited the Corani property.
7. I am responsible for the preparation of the following sections of the technical report titled Bear Creek Mining Corani Project NI43-101 Technical Report, dated 17 December, 2019 (the "Technical Report"), relating to the Corani Silver-Lead-Zinc project in Peru.

SECTION	SECTION NAME
20 (except 20.1.1, 20.1.3, 20.3.7, 20.3.8 & 20.3.10)	Environmental Studies, Permitting and Social or Community Impact

8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I am Independent (as defined in NI 43-101) of Bear Creek Mining Corporation.
10. I have read NI 43-101 and Form 43-101F1 and confirm the sections of the Technical Report for which I am responsible (as listed in Item 7 above) have been prepared in compliance with that instrument and form.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this Seventeenth day of December, 2019

____

Signature of Qualified Person

__Michael Meyer___________

Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Kevin Gunesch, do hereby certify that:

1. I am currently employed as Principal Mining Engineering by:

Global Resource Engineering Ltd. (GRE)

600 Grant Street, STE# 975

Denver, Colorado USA

2. I am a graduate of Colorado School of Mines and received a Bachelor of Science degree in Mining Engineering in 2000.
3. I am a US registered professional engineer in Alabama #27448
4. I have practiced mining engineering for 20 years. I have worked for several mining companies for 5 years, been self employed as a mining consultant for 5 years, and been employed by GRE for 10 years.
5. 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.
6. I visited the Corani property on numerous occasions for several weeks at a time with my most recent 13 day trip completed in November and December of 2011.
7. I am responsible for the preparation of the following sections of the technical report titled Bear Creek Mining Corani Project NI43-101 Technical Report, dated 17 December, 2019 (the “Technical Report”), relating to the relating to the Corani Silver-Lead-Zinc project in Peru.

SECTION	SECTION NAME
4	Property Description and Location
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography
6	History

8. I have had prior involvement with the property that is the subject of the Technical Report including participating in site geotechnical investigations, surface water management, mine planning, and cost estimation and have contributed to the previous 3 technical reports published in 2011, 2015, and 2017.
9. I am Independent (as defined in NI 43-101) of Bear Creek Mining Corporation.
10. I have read NI 43-101 and Form 43-101F1 and confirm the sections of the Technical Report for which I am responsible (as listed in Item 7 above) have been prepared in compliance with that instrument and form.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this Seventeenth day of December, 2019

_Kevin Gunesch (Signed)________

Signature of Qualified Person

_Kevin Gunesch____________

Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Terre Lane, do hereby certify that:

1. I am currently employed as Principal Mining Engineer by:

Global Resource Engineering LTD

600 Grant St

Denver CO 80203

2. I am a graduate of Michigan Technological University and received a Bachelor of Science degree in Mining Engineering in 1982.
3. I am a Qualified Professional with MMSA #01407QP and a Registered Member of SME #4053005.
4. I have practiced my profession since 1982 in capacities from mining engineer to senior management positions for engineering, mine development, exploration, and mining companies.

   My relevant experience for the purpose of this report is as the resource estimator (including geometallurgy) and mine planner, with 25 or more years of experience in each area.

   I have worked in geology, managed geologic teams, created lithological and structural models, and I have been involved in, conducted or led all aspects of the estimation of resources for several hundred projects at locations in North America, Central America, South America, Africa, Australian/New Zealand, India, China, Russia and Europe using nearly all estimation techniques.
5. 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.
6. I have not visited the Corani property.
7. I am responsible for the preparation of the following sections of the technical report titled Bear Creek Mining Corani Project NI43-101 Technical Report, dated 17 December, 2019 (the "Technical Report"), relating to the Corani Silver-Lead-Zinc project in Peru.

SECTION	SECTION NAME
13.8	Geometallurgical Modeling
14	Mineral Resource Estimates
15	Mineral Reserve Estimate
16	Mining Methods

8. I have previously contributed to the Optimized and Final Feasibility Study Corani Project, dated May 30, 2015 and the NI43-101 Technical Report, Corani Project Detailed Engineering Phase 1 (FEED)", dated effective September 13, 2017.
9. I am Independent (as defined in NI 43-101) of Bear Creek Mining Corporation.

10. I have read National Instrument 43-101 and Form 43-101F1, and confirm the sections of the Technical Report prepared under my supervision (as listed in Item 7 above) have been prepared in compliance with that instrument and form.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this Seventeenth day of December 2019

Signature of Qualified Person

Terre Lane

Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Jeffrey Todd Harvey, do hereby certify that:

1. I am currently employed as Director of Process Engineering by:

Global Resource Engineering

600 Grant St, Suite 975

Denver, Colorado,

USA 80203

2. I am a graduate of Queen's University and received a Ph.D. degree in Mineral Processing in 1994 and a M.Sc., B.Sc. also from Queen's University. I also hold an MBA from the University of New Brunswick (2000) and an BS in Metallurgical Engineering from Ryerson University.
3. I am a Registered Engineer in the United States from the Society of Mining, Metallurgy and Exploration (SME) membership number 04144120
4. I have practiced mineral processing for 30 years. I have worked for several mining companies including Barrick, Ashanti Goldfields and GeoBiotics LLC and most recently for GRE for the last 2 years.
5. 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.
6. I have not visited the Corani property.
7. I am responsible for the preparation of the following sections of the technical report titled Bear Creek Mining Corani Project NI 43-101 Technical Report, dated 17 December, 2019 (the "Technical Report"), relating to the Corani Silver-Lead-Zinc project in Peru.

SECTION	SECTION NAME
13	Mineral Processing and Metallurgical Testing

8. I have had prior involvement with the property that is the subject of the Technical Report including review of metallurgical testing and geometallurgical analysis. I have not authored any previous reports.
9. I am Independent (as defined in NI 43-101) of Bear Creek Mining Corporation.
10. I have read NI 43-101 and Form 43-101F1 and confirm the sections of the Technical Report for which I am responsible (as listed in Item 7 above) have been prepared in compliance with that instrument and form.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this Seventeenth day of December, 2019

J Todd Harvey

Signature of Qualified Person

Jeffrey Todd Harvey

Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Hamid Samari, do hereby certify that:

1. I am currently employed as Senior Geologist by:

Global Resource Engineering Ltd.,

600 Grant St., Suite 975

Denver, Colorado,

USA 80203

2. I am a graduate of Azad University, Sciences and Research Branch, Tehran and received a PhD in Geology-Tectonics in 2000 and I am a graduate of Beheshti University, Tehran and received a MS in Geology-Tectonics in 1995 MS and I am a graduate of Beheshti University, Tehran and received a BS in Geology in 1991
3. I am a Qualified Professional in the United States from the Mining and Metallurgical Society of America (MMSA) with special expertise in Geology with membership number 0151QP
4. I have practiced area of geology, mining, and civil industry for over 20 years. I have worked for Azad University, Mahallat branch as assistant professor and head of geology department for 19 years, for Tamavan consulting engineers as senior geologist for 12 years, and for Global Resource Engineering for near three years.
5. 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.
6. I visited the Corani Silver-Lead-Zinc project property from August 7, 2017 to August 11, 2017.
7. I am responsible for the preparation of the following sections of the technical report titled Bear Creek Mining Corani Project NI 43-101 Technical Report, dated 17 December, 2019 (the "Technical Report"), relating to the Corani Silver-Lead-Zinc project in Peru.

SECTION	SECTION NAME
7	Geological Setting and Mineralization
8	Deposit Types
9	Exploration
10	Drilling
11	Sample Preparation
12	Data Verification

8. I have had prior involvement with the property that is the subject of the Technical Report including review of geological setting and mineralization to data verification sections. I have been as part of the 2017 FEED 43-101.
9. I am Independent (as defined in NI 43-101) of Bear Creek Mining Corporation.

10. I have read NI 43-101 and Form 43-101F1 and confirm the sections of the Technical Report for which I am responsible (as listed in Item 7 above) have been prepared in compliance with that instrument and form.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this Seventeenth day of December, 2019

Signature of Qualified Person

Hamid Samari

Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Denys Parra, do hereby certify that:

1. I am currently employed as General Manager by:

Anddes Asociados SAC, Av. Circunvalación El Golf Los Incas 154. Piso 13, Surco, Lima, Peru

2. I am a graduate of National University of Engineering in Lima, Peru and received a Bachelor of Science degree in Civil Engineering in 1989 and the Civil Engineer Professional Title in 1991. I also received a Master in Civil Engineering degree in 1996 from Pontifical Catholic University of Rio de Janeiro, Brazil.
3. I am a Civil Engineer in Lima, Peru with professional license number 42347 and SME Registered Member number 4222036 in USA
4. I have practiced civil and geotechnical engineering for 28 years. I have worked for Vector Peru SAC company (a subsidiary of Vector Engineering Inc. of California, USA) for 13 years and for Anddes Asociados SAC for 9 years
5. 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.
6. I visited the Corani property on on September 8th, 2011
7. I am responsible for the preparation of the following sections of the technical report titled “Bear Creek Mining, Corani Project, NI 43-101 Technical Report” dated 17 December 2019 (the “Technical Report”), relating to the Corani Silver-Lead-Zinc project in Peru

SECTION	SECTION NAME
16.6	Waste Managment
18.3	Mine Waste Rock and Tailings Management Facilities
18.2.6	Water Supply and Management

8. I have had prior involvement with the property that is the subject of the Technical Report including contributions to past technical reports
9. I am Independent (as defined in NI 43-101) of Bear Creek Mining Corporation
10. I have read NI 43-101 and Form 43-101F1 and confirm the sections of the Technical Report for which I am responsible (as listed in Item 7 above) have been prepared in compliance with that instrument and form.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this Seventeenth day of December, 2019

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Signature of Qualified Person

Denys Parra__________________________

Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Eduardo Ruiz, do hereby certify that:

1. I am currently employed as General Manager: Amphos 21 Consulting Peru, S.A.C.

Amphos 21 Consulting Peru, S.A.C. – c/ Pietro Torrigiano 396, San Borja 41, Lima, Peru

I am currently member of the Board: Amphos 21 Group S.L.

Amphos 21 Group, S.L. – c/ Paseo de la Castellana 40, 8ª planta, 28046 Madrid, España

2. I am a graduate of the Barcelona School of Civil Engineering (ETSICCP), Polytechnic University of Catalonia, Barcelona, Spain, and from the Geological Faculty (University of Barcelona, Spain) and received title of Geological Engineering and Master in Geological Engineering (2002).
3. I am a Geological Engineer in Lima, Peru with professional license number CIP 234195.

   I am registered as a Professional Geologist in Spain (Reg. 7207) and as EurGeol in the European Federation of Geologists (Reg. 1234).
4. I have practiced as hydrogeologist and Hydrologist over 20 years. I have worked for the School of Civil Engineering for 3 years, Amphos 21 Consulting S.L. for 11 years and Amphos 21 Consulting S.A.C. for 8 years.
5. 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.
6. I visited the Corani property on November 28-29th, 2013.
7. I am responsible for the preparation of the following sections of the technical report titled "Bear Creek Mining, Corani Project, NI 43-101 Technical Report" dated 17 December 2019 (the "Technical Report"), relating to the Corani Silver-Lead-Zinc project in Peru

SECTION	SECTION NAME
20.1.1	Summary of air, noise, groundwater and surface water studies
20.3.7	Site wide water balance
20.3.8	Closure phase water management
20.3.10	Monitoring and maintenance

8. I have had prior involvement with the property that is the subject of the Technical Report including contributions to past technical reports
9. I am Independent (as defined in NI 43-101) of Bear Creek Mining Corporation
10. I have read NI 43-101 and Form 43-101F1 and confirm the sections of the Technical Report for which I am responsible (as listed in Item 7 above) have been prepared in compliance with that instrument and form.

11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this Seventeenth day of December, 2019

____

Signature of Qualified Person

Eduardo Ruiz

Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, David Arcos, do hereby certify that:

1. I am currently employed as Mining Geochemistry Manager by: Amphos 21 Consulting, S.L.

Amphos 21 Consulting, S.L. c. Venezuela 103, 2ª planta, 08019 Barcelona, España

2. I am a graduate of Universidad de Barcelona and received a Bachelor of Science degree in Geology in 1996. I also had a PhD in Geology by the University of Barcelona, receiving the title of Doctor in Geology in 1996.
3. I am registered as a Professional Geologist in Spain (Reg. 7163) and as EurGeol in the European Federation of Geologists (Reg. 1186).
4. I have practiced as geologist and geochemist for 29 years. I have worked for the University of Barcelona for 6 years and for Amphos 21 Consulting S.L. for 23 years.
5. 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.
6. I visited the Corani property on November 28-29th, 2013.
7. I am responsible for the preparation of the following sections of the technical report titled "Bear Creek Mining, Corani Project, NI 43-101 Technical Report" dated 17 December 2019 (the "Technical Report"), relating to the Corani Silver-Lead-Zinc project in Peru

SECTION	SECTION NAME
20.1.3	Summary of geochemical studies

8. I have had prior involvement with the property that is the subject of the Technical Report including contributions to past technical reports
9. I am Independent (as defined in NI 43-101) of Bear Creek Mining Corporation
10. I have read NI 43-101 and Form 43-101F1 and confirm the sections of the Technical Report for which I am responsible (as listed in Item 7 above) have been prepared in compliance with that instrument and form.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this Seventeenth day of December, 2019

Signature of Qualified Person

David Arcos

Print name of Qualified Person

IMPORTANT NOTICE

This National Instrument 43-101 Technical Report was prepared for Bear Creek Mining Corporation (BCM), co-ordinated by Ausenco Services Pty Ltd and with input from Bear Creek Mining S.A.C., Global Resource Engineering Ltd, Anddes Asociados S.A.C., Insideo S.A.C., and, Amphos 21 Consulting Peru S.A.C.; collectively the “Report Authors”. The information conclusions and estimates herein are based on the collective information available at the time of preparation and are subject to change. This report is intended for the use of BCM and solely for purposes legistated under the Canadian provincial and territorial securities law. Any other use of, or reliance on, this report by any third party is at that party’s sole risk.

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Tables

Figures

1 Summary

1.1 Introduction

Bear Creek Mining (BCM), whose shares are listed for trading on the Toronto Stock Exchange (TSX-V:BCM), Lima Stock Exchange (BVL:BCM), and posted for trading on the OTCQX market (OTCQX:BCEKF) contracted Ausenco to prepare this Technical Report in accordance with the Canadian National Instrument 43-101 (NI 43-101) for the Corani Project located in Puno, Peru. This Technical Report summarises the outcomes of work completed on the project to assess the technical and economic viability of a project with a throughput of 9.85 Mt/y and project execution and operational approaches.

This report follows a previous NI 43-101 Technical Report submitted on 13 September 2017 that was based on a throughput of 7.875 Mt/y.

All dollar ($) amounts in this report are US dollars.

Physical changes to the project include:

- re-routing the mine access road simplifying access to the Corani site and allowing for a redesign of the internal haul roads
- re-designing mine haul roads to reduce ore and waste haulage distance by an average of 2 km
- de-bottlenecking the process plant and increasing the filter capacity to obtain a 20% increase in daily ore production from 22,500 t/d to 27,000 t/d
- updating the water balance to match higher ore throughput
- changing the location of the concentrator reducing cut and fill earthworks
- re-designing the concentrator to reduce its footprint by 30% resulting in lower earthwork, concrete and steel costs
- preparing a new block model and mine plan for ore and waste. There was no material change to the Corani Mineral Reserves and Mineral Resources as outlined in the 2017 Technical Report.

The principal work and studies undertaken to reduce risk included:

- additional metallurgical testing
- an update of the geometallurgical model
- comminution test work to confirm mill capacities
- thickening, filtering and rheology tests to confirm handling characteristics of the tailings
- materials handling testing on crushed ore and filtered tailings for stockpile and conveyor designs
- studies of tailings stability and disposition characteristics
- nine additional geotechnical drill holes, 28 test pits, 31 Lightweight Dynamic Penetration Tests and 6 structural station evaluations were performed. These tests were in addition to 70 drill holes, 221 test pits and 68 previous tests to confirm facilities locations
- a quarry study to confirm the location, volume and quality of aggregate suitable for concrete

- developing Owner's Costs from first principles and benchmarking them against other recent projects resulting in an increase to \$65.3 million from \$32.3 million
- a legal review of the Peruvian tax regime
- an updated concentrate marketing and transportation study
- development of alternative execution approaches and associated capital costs
- refinement of the operating plans and associated operating costs
- an update of the project schedule.

In addition, work has commenced on site infrastructure including:

- detail design of main access to the project
- substantially completed an electrical substation. The connection to the high-tension electrical grid is planned for January 2020
- start of construction of the camp platform (earthworks).

1.2 Key Findings

The work completed has reduced construction, development and operating risks and identified potential improvements to the expected economic performance. The outcomes (Table 1-1) include a \$127 million (31%) increase in after-tax Net Present Value (NPV_5), a 52% increase in after-tax Internal Rate of Return (IRR) from 15.1% to 22.9%, a 1.2 year (33%) reduction in the payback period, lower All-In-Sustaining-Costs (AISC) and significantly reduced construction, development and operating risks.

Table 1-1: Financial summary

Parameter	Value
After tax NPV_5	\$531 million
After tax IRR	22.9 %
Initial capital	\$579.3 million
Capital payback	2.4 years
Ore processed per day	27,000 tonnes
AISC per oz silver (Life of Mine ("LOM"))	\$4.55
Average annual silver production (LOM)	9.6 million oz

The Proven and Probable Mineral Reserves (Table 1-2) are substantially unchanged from the 2017 Corani Report.

Table 1-2: Corani Project mineral reserves

Classification	Tonnes kt (dry)	Grade			NSR	Contained Metal		
		Silver g/t	Lead %	Zinc %	\$/t	Silver Moz	Lead Mlb	Zinc Mlb
Proven	20,330	59.7	1.00	0.60	34.02	39.0	450.0	268.5
Probable	118,253	49.9	0.88	0.55	29.48	189.6	2,290	1,426
Total Proven + Probable	138,582	51.3	0.90	0.55	30.15	228.6	2,740	1,694

1.3 Property and Location

The Project site is in the Andes Mountains of south-eastern Peru at elevations of 4,800 to 5,200 meters above sea level (masl), specifically within the Cordillera Vilcanota of the Eastern Cordillera. The site is in the Region of Puno, immediately northeast of the continental divide that separates Pacific drainages from Atlantic drainages. The site location is approximately 160 kilometers in a direct line to the southeast of the major city of Cusco, with Universal Transverse Mercator (UTM) coordinate ranges of 312,000E to 322,000E and 8,443,000N to 8,451,000N. The nearest town of significant size and infrastructure is Macusani, which is located around 30 km to the east of the Project.

The Corani Project consists of a series of thirteen (13) mineral claims or concessions. The 12 original concessions are grouped in the UEA (Administrative Economic Unit) with the 13th (recently acquired) to be integrated into the UEA in 2020. Mineral concessions in Peru are filed with the Instituto Nacional de Concesiones y Catastro Minero (INACC) which is part of the Ministerio de Energía y Minas in Peru (MINEM). Claims can vary in size from 100 to 1,000 hectares (ha). They are rectangular geometries parallel to the UTM grid system employed in the district. The Corani Project is in the district of Corani, province of Carabaya department of Puno, in Peru, and covers an aggregate extent of approximately 5,480 ha. The concessions are fully controlled by BCM and are free of any mortgage, lien, charge, third-party royalty, or encumbrance. BCM owns and controls the surface rights that cover the entire project area, including the open pit, waste dump, process plant, water ponds, camp, and ancillary facilities required for operation.

The mineral concessions comprising the Project are subject to compliance with payment of annual license fees of $3.00 per hectare (“License Fees”). In addition, they are subject to an annual maintenance requirement with one of the following alternative obligations:

- minimum required levels of annual production of at least $100 per ha in gross sales (“Minimum Production”)
- payment of an additional amount referred as the Penalty
- exploration expenditures of 10 times the Penalty.

Compliance with one of these three maintenance obligations, together with timely payment of License Fees, is required to keep them in good standing. Failure to comply with License Fee payments or Penalty payments for two consecutive years causes the forfeiture of the mineral concessions. The maintenance obligations apply equally to the one (1) new concession once it is integrated into the UEA.

In 2019, the original twelve (12) mineral concessions comprising the Project were subject to the obligations of Minimum Production, Penalties and exploration expenditures in accordance with the maintenance regime (see Section 4 for detail). BCM has made the minimum annual expenditure in 2019 and was not obliged to pay the penalty. BCM believes the same scenario would be repeated in 2020.

Control and current status were verified in October 2019 through an electronic database search of the Instituto Geológico Minero y Metalúrgico (INGEMMET). All concessions are in good standing.

1.4 Accessibility and Climate

The Project site is located in the district of Corani, also in the Province of Carabaya in the eastern Andes mountain range. The area is characterized by mountainous terrain dominated by volcanic rock, above which sits glacial gravel. The lithologic and climatic conditions have given rise to a series of cirques or bowl-shaped, steep-walled basins. Apart from the vegetation associated with wetlands, areas of “puna” or alpine tussock grassland occupy the valleys and moderate to steep slopes. The areas above 4,700 masl mostly consist of steep mountainous slopes where erosion and climatic conditions largely prevent the development of soils or vegetation. These areas are scarcely vegetated with species specially evolved to withstand the harsh conditions. The naturally occurring acidic soils related to oxidation of sulfide bearing materials, and the resulting acid rock drainage from exposed mineralized zones within the project area, have also prevented the development of vegetation where these conditions occur.

Existing access to the Project site is primarily by road from the town of Macusani (located on the paved dual lane Interoceanic Highway), which is readily accessible from the town of Juliaca, serviced by commercial airlines from Lima. This route typically takes 4.5 to 5 hours by vehicle. There are other access routes to the site from Cusco, taking approximately 6 hours by vehicle on increasingly primitive roads approaching the site. The city of Cusco is also serviced by commercial airlines from Lima.

The nearest town of significant size and with significant infrastructure is Macusani, which is the capital of the Province of Carabaya in the Region of Puno. Macusani is approximately 30 km east of the Project in a direct line. The access road from Huiquisa Bridge to the permanent camp will be improved. The length of the proposed mine access road connecting the process plant to Macusani is anticipated to be approximately 64 km of which 42 km will need to be improved.

1.5 History

Prior to the early 1950s, mineral exploration in the Corani district consisted of shallow prospect pits and adits in the northern portion of the current Project area. These prospects are of unknown age and may date back to colonial Spanish time. Antimony prospects south and east of the property reportedly were active in the early 1900s, when there was limited antimony production (Petersen, 1967).

The first modern evaluation of silver-lead mineralization began with the location of mineral concessions in 1951, and in 1956 Compañía Minera Korani was formed to develop the silver-lead mineralization previously prospected. The mines were developed and operated from 1956 to at least 1967. Total historical production is uncertain but is estimated at 100,000 tonnes of silver-lead-zinc ore. In early 1967, estimated mine production was reported at about 3,400 short tons per month, with grades of 7.0-9.0% lead, 2.3% zinc, and 8.0 to 11.0 oz/ton silver (Petersen, 1967).

The next exploration activity was by a private Peruvian company, Minsur S.A. That exploration was reported to include 40 shallow drill holes in various locations, including a number of close proximity holes in the gold zone (located south of the current resource area). Although Minsur is an active mining company in Peru, attempts by BCM to secure copies of Minsur's exploration data have been unsuccessful. None of Minsur's exploration information is available or verifiable, although, reportedly, gold mineralization was encountered in some of Minsur's drilling.

In late 2003 and early 2004, Rio Tinto Mining and Exploration began a surface exploration program for porphyry copper mineralization. That initial work by Rio Tinto defined anomalous silver and lead mineralization to the south of the Korani mines and also defined a zone of anomalous gold mineralization in rock and soils. The concession ownership by Compañía Minera Korani apparently lapsed during the 1970s. The ownership of Minsur also lapsed prior to Rio Tinto's exploration activities after 2000. Rio Tinto re-established some of the older concessions in their name beginning in 2003. BCM acquired two additional concessions in 2005, and between 2007 and 2011 acquired the Rio Tinto concessions to consolidate the project. One additional concession was added in 2019 to create the current land position described in Section 4.

Seven previous resource estimates and four previous mineral reserve estimates have been completed for the Project and are published in previous technical reports beginning in 2006. Since 2006, the Measured and Indicated Mineral Resource has grown from approximately 40 million ounces (Moz) of silver to over 300 Moz of silver.

1.6 Geological Setting and Mineralization

The Corani Project area is located within the Cordillera Oriental of the Central Andes. The Project area is underlain by tertiary volcanic rocks of the Quenamari Formation, specifically a thick series of crystal-lithic tuffs and andesite flows which overlie variably deformed Lower Paleozoic to Mesozoic metasediments of the Ambo and Tarma Groups. The primary host of mineralization is the Chacaconiza Member of the Quenamari Formation. The Chacaconiza is the youngest member of the Quenamari and is comprised of a sequence of crystal-lithic and crystal-vitric-lithic tuffs. The tuffs are widely hydrothermally altered and pervasively argillized to low-temperature clays, and are variably faulted, fractured, and brecciated.

Mineralization at the Corani Project occurs in three distinct and separate zones: Corani Main, Corani Minas, and Corani Este, each differing slightly in character with regard to both alteration and mineral assemblages. In general, mineralization in outcrops throughout the Corani Project is associated with iron and manganese oxides, barite, and silica. Silicification is both pervasive and structurally controlled along veins. In drill core, the mineralization occurs in typical low to intermediate sulfidation silver-lead-zinc (Ag-Pb-Zn) mineral assemblages. The most abundant silver-bearing mineral is fine-grained argentian tetrahedrite or freibergite.

Structurally, the Corani deposit is situated within a stacked sequence of listric normal faults striking dominantly north to north-northwest with moderate to shallow (50° to <10°) westerly dips. The hanging walls of the listric faults are extensively fractured and brecciated, providing the structural preparation for subsequent or syngenetic mineralization. The stacked listric faults are more prominent in the Corani Minas and Corani Main areas. The Corani Este area contains a single known listric fault with an extensively fractured and brecciated hanging wall. The contact with the underlying Paleozoic sediments corresponds locally to listric faults dipping shallowly to the west.

1.7 Deposit Types

The Corani deposit is best described as a low- to intermediate-sulfidation epithermal deposit with silver, lead, and zinc mineralization hosted in stock works, veins, and breccias. Mineralization is principally located in a set of listric faults with a general north-northwest strike and dipping west, with dilational segments related to subvertical structures and breccias in the hanging wall, and veinlets forming stockworks in the footwall. Structural control of the mineralization is a product of extensional tectonics that developed the series of north- to northwest-trending fractures and faults and whose movements provided the structural preparation for the influx of mineralizing hydrothermal fluids.

Mineralization at Corani is likely both laterally and vertically distal to an intrusive fluid source. Mineral textures grade from coarse crystalline quartz-pyrite-chalcopyrite in the southern portion of the Project area, to finer grained, pyrite-dominated sulfide minerals in the north, suggesting a south-to-north hydrothermal fluid flow. This spatial zonation suggests a rapidly cooled ore fluid typical of a distal setting surrounding a buried intrusion. The multiphase nature of the mineralization and zonation at Corani may be related to multiple fluid exsolution events from an evolving porphyry type system that possibly underlies the southern part of the area. Alternatively, the mineralizing solutions may be related to shallow, subvolcanic dome emplacement.

1.8 Exploration

BCM began exploring the Corani Project in early 2005. In addition to drilling, exploration activities carried out by BCM include detailed geologic mapping, trenching, and geophysical surveying.

BCM has conducted general geologic surface mapping over the entire Project area. The total mapped surface is about 4.5 km wide (east-west) and 7.5 km long (north-south). In 2015, detailed surface mapping, including lithology, alteration, and structures, was performed at a scale of 1:2,500 in the area of the proposed pits.

BCM has completed 25 trenches within the Project resource area (Corani Main, Minas, and Este) to verify the continuity of the structures covered by quaternary sediments. Spacings between the trenches were roughly 50 to 100 meters. Channel samples from these trenches have produced an associated 1,295 assay intervals for a total of 2,924 meters of trench data.

VDG del Perú S.A.C. (VDG) conducted a ground geophysical campaign at the Corani Project on behalf of BCM in the fall of 2005. A total of 44.20 line-km of induced polarization (IP) data was collected, along with 50.95 line-km of magnetic survey. The geophysical surveys were aimed at assisting in geological mapping, including lithologies and key structures and at mapping mineralization and alteration associated with a low sulfidation gold-silver system. The objective of the IP/Resistivity survey was to map the electrical response by means of high-resolution IP traverses across the favorable north-south corridor identified based on the results of both trench and drilling exploration. The final chargeability and resistivity depth sections mapped systematically clear contrasts from line to line between the sub-surface and a nominal depth of 283 meters below surface. The chargeability outlined five (5) IP anomalies, two of which correspond to the Corani Main and Corani Este areas. Those anomalies accurately mapped the known mineralization and extended the size of both mineralized zones.

1.9 Drilling

Since 2005, BCM has completed a total of 562 drill holes at the Corani Project, for a total of approximately 101,401 m. Drilling was completely by the Peruvian contractor, Bradley MDH, primarily using LD250, JKS35, and LJ44 drill rigs. All of the drilling to date has been completed using diamond core drilling methods to produce either HQ (6.35 cm dia.) or NQ (4.76 cm dia.) core. Diamond drill hole data in the Project database used to model the resource includes 476 drill holes with an associated 36,103 sample intervals over a total of 83,104 m of drilling. The Project database contains 36,103 assay values each for silver, lead, zinc, and copper. In 2019, BCM completed a total of six drill holes at the project site, with a total of 906.0 m. Although these drills were made to obtain material for metallurgical studies, the results of laboratory testing include 984 assay values each for silver, lead, zinc, and copper, which were added to the project database for updating the resource estimate.

1.10 Sample Preparation, Analyses and Security

BCM employs standard, basic procedures for both drill core and trench sample collection and analysis. Formal chain-of-custody procedures are maintained during all segments of sample transport. Samples prepared for transport to the laboratory are bagged and labelled in a manner that prevents tampering and remain in BCM control until released to private transport carrier in Cusco or Juliaca. Upon receipt by the laboratory, samples are tracked by a blind sample number assigned and recorded by BCM. The samples are prepared according to ALS-Chemex preparation code PREP-31, and silver, lead, zinc, and copper assays are carried out by three-acid digestion followed by atomic absorption spectrophotometry (AA) analysis. Multi-element inductively coupled plasma (ICP) analysis is conducted on select sample intervals to assist with mineralization classifications and to guide the interpretation of the metallurgical process response.

BCM maintains an internal Quality Assurance/Quality Control (QA/QC) program, which includes both standards and check (lab) sampling. Global Resource Engineering Ltd. (GRE) conducted a critical review of BCM's QA/QC program; toward that end, BCM provided GRE with QA/QC data in multiple Excel spreadsheet files. GRE compiled the data into a single, comprehensive QA/QC data worksheet for analysis and evaluation. Based on the results of GRE's review, in conjunction with observations and conversation with BCM personnel during the QP site visit, BCM's routine sample preparation, analytical procedures, and security measures are, in general, considered reasonable and adequate to ensure the validity and integrity of the data derived from BCM's sampling programs. GRE recommended that BCM expand the existing QA/QC program to include at least standards, blanks, and duplicates, and that QA/QC analysis be conducted on an on-going and documented basis, including consistent acceptance/rejection tests.

1.11 Data Verification

Data verification efforts included an on-site inspection of the Corani Project and core storage facility, check sampling, and manual and mechanical auditing of the Project database.

During the on-site inspection in August 2017, GRE's (QP) representative conducted general geologic field reconnaissance, including inspection of bedrock exposures and other surficial geologic features, ground-truthing of reported drill collar and trench sample locations, and superficial examination of historic mine workings. One full day of the site visit was spent at the core storage facility in Juliaca, where select intervals of whole and half core were visually inspected, and samples were selected to submit for check assay. Field observations during the site visit generally confirm previous reports on the geology of the Project area. Bedrock lithologies, alteration types, and significant structural features are all consistent with descriptions provided in existing Project reports, and the QP did not see any evidence in the field that might significantly alter or refute the current interpretation of the local geologic setting.

Specific core intervals from 35 separate drill holes were selected for visual inspection and potential check sampling based on a preliminary review of the drill hole logs and associated assay values. The core intervals were selected prior to the site visit, and the core was laid out by BCM staff and ready for inspection upon arrival. With few exceptions, the core samples accurately reflect the lithologies recorded on the logs. A total of 17 samples were selected for check assay. The samples were selected from low, moderate, and high-grade intervals based on original assay results. In all cases, the degree of visible alteration and evidence of mineralization observed was generally consistent with the grade range indicated by the original assay value. Laboratory analysis was completed by ALS Peru S.A. using the same sample preparation and analytical procedures as were used for the original samples. Standard t-Test statistical analysis was completed to look for any significant difference between the original and check assay population means. The results of the t-Test showed no statistically significant difference between the means of the two trials (original versus check assay).

GRE completed a QA/QC audit of the digital Project database by comparing a random selection of original assay certificates to the assay information contained in the Corani Project database. Results of the QA/QC audit indicate a minor and acceptable error rate. GRE also completed a mechanical audit of the Project database to evaluate the integrity of data from a data entry perspective. The mechanical audit identified a small number of data entry errors, including gaps, overlaps, and missing sample intervals. All data entry errors were easily rectified and are considered insignificant with regard to potential impact to the mineral resource and mineral reserve estimates. The database audit work completed to date indicated that occasional inconsistencies and/or erroneous entries are likely inherent or inevitable in the data entry process. GRE also completed overall view on the BCM's in-house QA/QC over all drilling in the 2019 campaign. The overall view on the QA/QC program indicates acceptable performance of blank and standard for all drilling data. GRE recommended that BCM establish a routine, internal mechanical audit procedure to check for overlaps, gaps, total drill hole length inconsistencies, non-numeric assay values, and negative numbers. The internal mechanical audit should be carried out after any significant update to the database, and the results of each audit, including any corrective actions taken, should be documented and stored for future use in database validation.

1.12 Mineral Processing and Metallurgical Testing

The Corani deposit is a silver-lead-zinc deposit with varying mineralogy associated with specific mineral zones. Review of the testing data shows that the metallurgical response of Corani samples to flotation is heavily dependent on the mineralogy. The most frequent geological ore classification, Fine Black Sulfide (FBS), exhibited a range of lead-silver and zinc flotation recoveries. The variable response was shown to be generally related to the fine texture of the mineralization and presence of non-sulfide lead minerals. However, the geological classifications do not provide any insight into the texture or quantity of non-sulfide lead minerals.

Additional test work was performed in 2018 and 2019 on 12 samples from 9 boreholes (6 of which were new) drilled in the Este, Minas, and Main pits to optimize the known flotation test conditions as well as the comminution parameters, reagents scheme, and dewatering of concentrates and tailing characteristics. The selected samples reasonably cover the entire ore deposit and included ore with some degree of oxidation and ore with low sulfide content. The information obtained validated and improved the recovery formulas, providing additional confidence in the LOM production schedule. The locked-cycle flotation tests performed on the sulfide ore composites showed that lead recoveries to the lead concentrate ranged from approximately 62% to 78% with corresponding concentrate grades of 61% to 49% lead. Total silver recoveries ranged from approximately 63% to 84%. Zinc recoveries to the zinc concentrate ranged from 39% to 75% with corresponding concentrate grades of 55% and 53% zinc.

This test work confirmed that marketable quality lead and zinc concentrates can be produced using the processing parameters selected for the process plant design.

GRE updated the geometallurgical database and performed an exploratory data analysis, including the identification of outliers and review of the mineralization styles, mineralogy, and geologic log data to see if improvements could be made to the metallurgical performance predictions. In addition, the statistical models were updated and a comparison was made to locked cycle testing (LCT) to estimate the final recoveries to the lead and zinc concentrates. The geometallurgical model was updated to include a transition indicator to discriminate between sulfide and transition zone responses.

With metallurgical response linked to block modelling parameters, the mine plan was optimized to maximize the revenue for the Project. Table 1-3 displays the estimated metal recoveries by mine schedule.

Table 1-3: Recovery predictions for mine schedule

Production Year	Tonnes (000)	Feed Grade			Recovery to Pb Con %		Recovery to Zn Con %	
		Silver g/t	Lead %	Zinc %	Silver	Lead	Silver	Zinc
Year 1	8,600	100	1.10%	0.84%	64.2%	51.3%	5.4%	78.4%
Year 2	9,882	71	1.04%	0.77%	66.3%	63.7%	6.9%	78.1%

Production Year	Tonnes (000)	Feed Grade			Recovery to Pb Con %		Recovery to Zn Con %	
		Silver g/t	Lead %	Zinc %	Silver	Lead	Silver	Zinc
Year 3	9,855	78	1.12%	0.69%	66.8%	60.6%	5.2%	72.3%
Year 4 to 5	19,710	65	1.26%	0.60%	61.4%	58.3%	7.1%	70.7%
Year 6 to 10	49,329	39	0.74%	0.51%	61.6%	58.9%	10.5%	70.5%
Year 11 to 15	41,206	32	0.79%	0.44%	64.1%	62.7%	7.5%	74.5%
LOM	**138,582**	**51**	**0.90%**	**0.55%**	**62.9%**	**57.2%**	**6.1%**	**72.3%**

1.13 Mineral Resource Estimates

GRE updated the Mineral Resources for the Corani Project with new drilling completed in 2019. This drilling added 6 holes to the database used for estimation. The drill hole database was updated with geologic logs and assays of primary recovery indicators: copper, goethite, manganese oxide, pyrite, and galena. These geometallurgical indicators were modelled along with the economic metals in the block model. The 2019 model uses the updated drill hole database, including the 6 additional drill holes drilled subsequent to the development of the previous database. An indicator field was added to the model to estimate the extent of the transition material.

The resource model has three main lithologies: a basement sediment with minor quantities of mineralization, the mineralized (pre-mineral) tuff, and a mostly unmineralized post-mineral tuff that is assumed to be barren. Mineralization has been defined by three mineralization groups: oxidized, transition, and sulfide. The Mineral Resources for the Corani Project are shown in Table 1-4. The Mineral Resources were generated within the $30.00/troy ounce silver, $1.425/pound (lb) lead, and $1.50/lb zinc price Lerchs-Grossman economic pit shell and the calculated $10.79/tonne NSR cutoff.

Table 1-5 shows the potentially leachable Mineral Resource contained within the Whittle pit shell at a 15 gram per tonne silver (g/t Ag) cutoff that is available in addition to the Mineral Resource shown in Table 1-4.

Table 1-4: Total mineral resources (includes both resources and reserves)

Category	Tonnes (000)	Silver g/t	Lead %	Zinc %	Silver Moz	Lead Mlb	Zinc Mlb
Measured	30,585	50.0	0.79	0.49	49.1	534	329
Indicated	208,050	40.9	0.64	0.43	273.5	2,933	1,985
Measured + Indicated	**238,635**	**42.1**	**0.66**	**0.44**	**322.7**	**3,466**	**2,313**
Inferred	73,185	35.5	0.40	0.30	83.5	641	484

Note: Cutoff Value: $10.79/tonne covers process and general and administrative costs.

Table 1-5: Total mineral resource of potentially leachable material (includes the mineral reserve)

Category	Tonnes (000)	Silver g/t	Silver Moz
Measured	4,302	28.9	4.0
Indicated	36,104	30.1	35.0
Measured + Indicated	40,406	30.0	39.0

Category	Tonnes (000)	Silver g/t	Silver Moz
Inferred	24,311	38.2	29.9

1.14 Mineral Reserve Estimates

GRE reviewed and verified that the phased mine design generated by BCM was prepared with sound engineering principles and is correct. The mine design was compared to Lerchs-Grossman (LG) pits estimated using the current GRE Mineral Resource block model. GRE has found the work performed by BCM to reasonably conform to those current economic pits estimated. The LG estimation used $20.00/oz silver, $1.00/lb zinc, and $0.95/lb lead for the mine design (unchanged from the 2017 Technical Report).

The Project Mineral Reserves consider only measured and indicated resource categories, which have been converted to proven and probable reserves categories, respectively. Mineral Reserves are defined as being the material to be fed to the process plant in the mine plan already described and are demonstrated to be economically viable in the Corani Project economic model. The Mineral Reserves are shown in Table 1-6.

Table 1-6: Corani Project mineral reserves

Classification	Tonnes kt (dry)	Grade			NSR	Contained Metal		
		Silver g/t	Lead %	Zinc %	$/t	Silver Moz	Lead Mlb	Zinc Mlb
Proven	20,330	59.7	1.00	0.60	34.0	39.0	450.0	268.5
Probable	118,253	49.9	0.88	0.55	29.5	189.6	2,290	1,426
Total Proven + Probable	**138,582**	**51.3**	**0.90**	**0.55**	**30.2**	**228.6**	**2,740**	**1,694**

Notes:

1. The Mineral Reserves have been estimated using the definitions of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM).
2. The Mineral Reserves have been estimated using the following metal prices: $20.00/oz silver, $1.00/lb zinc, $0.95/lb lead using a revenue factor 1.00 pit shell as a basis for the pit design.
3. Only pre-mineral tuff type of material has been considered as reserves.
4. NSR Cutoff grades used are equal or higher than: $10.79/t.
5. The effective date for these Mineral Reserves is 5 November 2019.
6. Totals / Averages may not add up due to rounding of individual tonnes and grades.
7. The tonnes and grades shown above are considered a Mineral Reserve because they have been demonstrated to be economically viable through the Corani Project financial model using the following metal prices: $18.00/oz silver, $1.10/lb zinc, $0.95/lb lead.

1.15 Mining Methods

The Corani Project will be mined using conventional open pit mining methods, with either an owner mining or a contractor mining scenario. The base case assumes contractor mining. The rock will be broken by drilling 0.156 m diameter blast holes and blasting with ammonium nitrate/ fuel oil (ANFO) and emulsion. Broken rock will then be loaded into 140 tonne trucks using a 19 cubic meter (m^3) front end loader or one of two 22 m^3 hydraulic shovels. Support equipment includes two Caterpillar D9 bulldozers, a road grader, water trucks, rubber tire dozer, compactor, excavator, fuel and lube trucks, and other miscellaneous equipment.

During a 17 month pre-production stripping, pioneering, haul road construction phase prior to plant production, 4.7 million tonnes of waste rock will be mined to generate construction material. Another 13.8 million tonnes will be mined immediately prior to production. The mine is designed to generate 9.855 million tonnes of ore per year with an average strip ratio of 2:1 during the first three years, falling to below an average of 1.1:1 the remainder of the mine life.

1.16 Recovery Methods

A cost-effective plant has been designed to process the Corani ore at a rate of 27,000 t/d. This was achieved by minimising the footprint, maximising throughput and taking advantage of the topography.

The mined ore will be crushed by a single gyratory crusher prior to two stages of grinding in a semi-autogenous (SAG) mill and ball mill. The lead and zinc bearing minerals will be recovered in a two-stage sequential flotation and regrind circuits. The design allows for 75% of the lead to be recovered in the lead flotation circuit followed by 69% recovery of zinc to the zinc flotation circuit.

To reduce the water consumption, the tailings from the flotation circuits will be thickened in high-compression thickeners prior to filtration in conventional pressure filters. The filtered tailings will be co-disposed together with the mined waste to produce a stable waste deposit.

1.17 Project Infrastructure

The Corani Project site location is remote, at high elevation, and 42 km from the interoceanic highway. The nearest urban area is Macusani with a population of approximately 12,700 people (2017 census). The infrastructure to be developed for the Project includes access to site, internal access routes, process buildings and related facilities, water supply, power supply, communications systems, and storage and warehousing.

Project components have been optimized subsequent to the 2017 Technical Report. Optimization engineering studies, geotechnical site investigation work, quarry studies and laboratory testing programs have been performed in order to reduce capital and operating costs.

The most significant changes were advanced through additional fieldwork, trade-offs and detailed engineering to support the optimization concepts presented in this study. A summary of the infrastructure related work performed subsequent to the 2017 report is presented below:

1. additional geotechnical investigations, adding 9 boreholes, 28 test pits and 31 Lightweight Dynamic Penetration Tests (LDPT)
2. improved access from the camp to the process plant
3. a quarry study, locating proper aggregates for concrete near the process plant
4. review and optimization of the project and plant footprints
5. optimised access for mine vehicles (haul roads).

1.17.1 Transportation, access, and site roads

Transportation to and around the site is by roadways that have been developed and improved to accommodate the demands of the Project.

A 42 km access road (PU 516 – PU 514) which connects to the interoceanic highway 7 km from Macusani has been evaluated and designated to be used for the construction stage (“Construction Access”). This road is part of the national road network and will require repair and maintenance during the construction stage in order to allow the access of equipment, supplies and materials for the project.

The construction access has few interferences and requires minimal CAPEX investment due to the alignment and location. There is minimal impact on local residents as there are no communities situated along the route. The access will also be available if needed during operations and can be used to receive supplies and deliver the lead and zinc concentrates to the Port of Matarani or other ports via trucks connecting to Peru's public highway system.

Another access road (a 44 km new highway design by GMI and included in the 2017 Technical Report), has a government investment budget for construction approved for 2021 and will be available for operations assuming the funds are released and construction is completed as planned (north route shown in Figure 18-1).

1.17.2 Mine service facilities

The mine service facilities will primarily be located on the Mine Infrastructure Area (MIA) adjacent to the process plant. The facilities include:

- truck workshop
- wash bay and associated repair facilities
- mine offices
- warehouse
- fleet management system (dispatch)
- explosives storage facility.

The explosives storage facility will be located in a remote area adjacent to the mine for safety and security purposes.

1.17.3 Administration facilities

The administration facilities include the following buildings:

- process plant gatehouse building
- administration building.
- warehouse building
- first aid building
- reagent storage building.

The administration facilities are located near the process plant and will contain the offices for the local management and administration personnel. The process plant gate house is located at the entry to the site near the contact and non-contact water ponds.

There will be a small administration building at the accommodation camp for camp management as well as the main medical post. Access to the plant site will be controlled from the gates in the camp.

1.17.4 Project water management

Surface water and groundwater will be used to provide the water required for the project. Surface water (runoff and streamflow) and groundwater (from pit dewatering) will both come from the watershed that hosts the project. No cross-basin abstractions will be required. Water on the project is classified as either contact water or non-contact water. Contact water is defined as water that has had contact with any area disturbed by the project where the water quality could be degraded from Acid Rock Drainage (ARD) or other water contaminants. Non-contact water is defined as water that has not had contact with the process components or any area that has been disturbed. Contact water and non-contact water will be managed and conveyed separately. They will ultimately be stored in a water storage pond which has two separate compartments, one for each circuit. The contact water that has been stored will be consumed as preferential process water (make-up) for the plant. This water cannot be discharged to the environment during operations (see Section 20). A portion of the non-contact water stored in the pond will be used to supplement the process water demand during the dry seasons. Non-contact water that is not used will be discharged, if necessary, to the Quebrada Chacaconiza. The project is required to discharge a fixed quantity of non-contact water downstream as part of the environmental impact study and ITS (see Section 20).

1.17.5 Power supply

A 138 kV power transmission line is necessary to provide power to the Corani Project. A new power substation (the Antapata Substation, currently under construction) will connect with power transmission lines L-1010 and L-1051 (San Gabán II – Azángaro) as the power source. A new 138 kV power transmission line will be built to connect the Antapata substation to the main Corani substation to be built near the Project's main process buildings. The transmission line will be 29.4 km in length. The proposed alignment for the 138 kV line was provided by Promotora de Proyectos S.A.C. company in 2019. The transmission line route uses the route already provided by the Project's access road.

Rights of way for the power line have been agreed with the local communities but have yet to be purchased from the individual land holders.

1.17.6 Waste rock and tailings management facilities

The main mine waste and filtered tailings deposit or 'deposito de desmonte mina y relaves filtrado principal' (DDMR)" serves for disposal of mine waste and filtered tailings in a common deposit, the size of which has been designed for the quantities considered in the mine plan. The height could be increased to give more capacity in the future if required. In total, 79% of the waste to be mined is classified as non acid generating (NAG). The co-disposal will use a 25 meter thick layer of NAG material on the foundation and outer shell for encapsulating the potentially acid generating (PAG) material and tailings. Initially, a base platform will be constructed using NAG mine waste from the mine pre-stripping stages. This facility and the mine pits are designed to minimize and mitigate the formation of acid rock drainage (ARD), which is a natural process which arises from the oxidation of sulfide minerals. This risk is present in Corani waste rock, tailings, and pit walls. Section 20 describes the ARD management plan.

The co-disposal during the wet season will be carried out on the upstream zone of the deposit. During the dry season co-disposal will be carried out in the downstream zone. The upstream zone will also be used for placing filtered tailings with moisture over 17% w/w or mine waste with high clay content. A detailed disposal plan has been prepared over a monthly basis for the first two years and year-on-year for the life of mine. If times occur during operations that mine waste is not available in sufficient quantities for co-disposal, tailings will be placed on the upstream zone.

Filtered tailings and mine waste will be placed in the same location for conforming layers of 2 metre maximum thickness. The disposal will be performed from upstream to downstream in order to facilitate water management.

In years 10, 11 and 12, 35.5 Mt of mine waste will be used to backfill the East, Minas and Main pits.

1.18 Market Studies and Contract

BLB Advisory prepared an analysis of market prices and market conditions for lead and zinc. This included a review of current and forecast treatment and refining charges and penalties from smelters/refineries, costs associated with concentrate handling, and shipping costs (inland and ocean) to potential customers. All information was sourced from public and subscription-based sources, quotations collected from the market and BLB´s experience. The supplied information was used as a guide to develop all associated payments and expenses associated with the sale of Corani concentrates. There are no letters of intent or concentrate sale agreements in place.

1.19 Environmental Studies, Permitting and Social or Community Impact

For the development of mining projects in Peru, the approval of the Environmental and Social Impact Assessment (ESIA) is required in order to start the project development. The original ESIA was approved by the Ministry of Energy and Mines (MINEM) in 2013, based on the Feasibility Study (FS) prepared in 2011. Two modifications to the ESIA, in the form of an Informe Tecnico Sustentatorio (ITS), have been approved. The first one was in 2016 and an additional one was completed in 2017. Currently, Bear Creek is planning to complete a third ITS modification of the ESIA that includes the latest engineering changes.

The ESIA requires the filing of a mine closure plan. The mine closure plan was approved in April, 2015. An update to the closure plan was approved in 2018.

As the primary changes to the ESIA are relatively minor and focused on optimization of the mineral processing, additional public hearings are not required. Additionally, the modifications reduce the environmental impact of the proposed Corani operation, which may result in quicker approval of the third ITS once it is submitted. The submission and approval of the ITS is not expected to impact construction.

Significant community consultation has been undertaken with the Chacaconiza and Quelcaya communities to date. Discussions have included a proposal for mining employment, which has generated widespread acceptance of the Project, mainly among younger community members, the teachers at local educational facilities, and community leaders. The current labor force is generally unskilled, mainly working on highway remediation and maintenance. A technical training program that is directed at developing the skills of community members to fulfill employment requirements of the Project has been started.

Bear Creek completed a Life of Mine (LOM) Investment Agreement in June 2013 with the District of Corani, five surrounding communities, and relevant, ancillary organizations. The agreement specifies investment commitments over the 23-year project life, which includes the pre-production construction period. Under the agreement, annual payments are to be made into a trust designed to fund community projects totaling 4 million Nuevos Soles per year (approximately $1.2 million per year). Once the Project commences development, the payments will remain constant throughout the development/construction phase and during production. Cessation or interruptions of operations will cause a pro-rata decrease in the annual disbursements. As an integral part of the LOM agreement, a trust or foundation structure is established for approval of investments and disbursement of funds. Some initial projects have already been funded.

1.20 Capital and Operating Cost

The Corani Capital cost estimate has been prepared in US dollars ($) to an accuracy of -10% / +15% and has generally been prepared in line with the Association for the Advancement of Cost Engineering (AACE) International, Recommended Practice No. 47R-11 for a Class 3 Estimate.

The concentrates are seen to be easily marketable due to the high silver grade in the lead concentrate and the overall grade of the zinc concentrates. Life of mine capital cost, initial capital and life of mine operating cost estimates are summarised in Table 1-7, Table 1-8 and Table 1-9, respectively. The capital cost estimates have been divided between the scopes estimated by Ausenco (process plant and on-site infrastructure) and BCM (mining, waste dump, and off-site infrastructure).

Table 1-7: Life of mine capital cost summary

Cost Type	Cost ($ M)
Initial CAPEX	579
Sustaining CAPEX	23.5
TOTAL	**603**

Table 1-8: Capital cost summary

WBS	Description	Ausenco Value ($M)	Bear Creek Value ($M)	TOTAL Value ($M)
1000	Mining	0.0	59.3	59.3
2000	Process plant	234	0.0	234
3000	On-site infrastructure	17.4	40.8	58.2
4000	Off-site infrastructure	0.0	25.5	25.5
5000	Field indirects	20.7	0.3	20.9
6000	Other	3.9	0.0	3.9
7000	Engineering	60.0	0.0	60.0
8000	Owner's costs	0.0	65.3	65.3
9000	Contingency	34.4	17.1	51.5
	Total	**371**	**208**	**579**

Table 1-9: Life of mine operating cost summary

Operating Cost	$/ tonne ore
Mine	4.29
Process Plant	10.04
General and Administration	1.88
Concentrate Transportation	2.48
Total Operating Cost	**18.70**

1.21 Economic Analysis

The economic analysis was performed using a Discounted Cash Flow (DCF) as per standard industry practice. The key assumptions used for the study are shown in Table 1-10 and establish a "Base Case". The table provides the life-of-project averages for grade recovery and these values vary over the life of the project depending on head grades and split between mixed sulfide ore and transitional ore.

Table 1-10: Key assumptions for the Corani Project – Base Case

Parameter	Assumption
Annual ore production – years 1 to end of life (kt)[1]	9,239
Overall process recovery – silver – into both lead and zinc con (%)	69.0
Overall process recovery – lead – into lead con (%)	57.2
Overall process recovery – zinc – into zinc con (%)	72.3
Total processed (Mt)	139
Average silver grade (g/t)	51.3
Average lead grade (%)	0.90
Average zinc grade (%)	0.55
Payable ounces of silver net of smelter payment terms (Moz)	144
Payable pounds of lead net of smelter payment terms (Mlbs)	1 480
Payable pounds of zinc net of smelter payment terms (Mlbs)	1 040
Overall stripping ratio	1.42 : 1
Life-of-Mine (years)	15

[1]excluding planned ramp up period in year 1

Project financial analysis outcomes are summarised in Table 1-1.

1.22 Adjacent Properties

There are no adjacent mineral properties which might materially affect the interpretation or evaluation of the mineralization or exploration targets of the Corani Project.

1.23 Other Relevant Data and Information

1.23.1 Project execution plan

A Project Execution Plan (PEP) has been prepared by Bear Creek and Ausenco creating a project development pathway considering location and site conditions to:

- minimize risk and uncertainty
- manage construction performance and schedule
- deliver the Project on budget.

The Project is planned to be constructed over a three year time period.

The PEP defines the overall approach that will be taken in the project and details the specific philosophies, strategies, methods of work, accountabilities, and resources that will be used in the execution of the Corani Project.

The PEP also serves to align different functions within the project team and quickly orient new team members coming into the project. The Project will be executed in accordance with the PEP to achieve the following objectives:

- achieve an unparalleled safety and environmental record
- educate and involve the local communities and stakeholders in the project
- utilize an efficient "fit for purpose" design

- be constructed on time and on cost
- ensure compliance with project quality standards.

1.23.2 Project development schedule

The project development schedule will begin with the construction of the camp, followed by road construction, then engineering in parallel with the procurement of the first equipment from the second quarter of 2021. Construction activities continue through 2021 and 2023, with the planned start-up until the first quarter of 2024.

The following list shows the estimate time duration for each main activity:

- detailed engineering – 12 months
- early works (camp, power line, access road) – 23 months
- mine construction/pre-stripping – 22 months
- plant construction – 25 months
- commissioning and initial ramp-up – 4 months.

The total time from a decision to proceed is estimated to be approximately 36 months.

1.23.3 Project delivery

The contracting strategy adopted for the project is aimed at minimising risk, by having an experienced EPC/EPCM company for the process plant and a major mining contractor for pre-stripping and mine operations. Responsibilities for scope will be split between the EPCM company and BCM with BCM managing mining related works and some general earthworks and infrastructure. The detailed engineering work will be developed by the EPC/EPCM company.

Due to the location and altitude of the site, pre-fabrication and skid-mounted packages will be considered to the greatest extent practical. Pre-assembled modules will be equipped with piping, valves, wiring and instrumentation to reduce onsite labour.

The majority of equipment and materials are expected to be sourced from USA, Canada, Europe, Chile and China. Some mechanical equipment, consumables and material will be procured from Peruvian suppliers, such as platework and steel structure.

Subcontract packages will be let on a horizontal trade basis for earthworks, concrete works, SMP and E&I. Certain trade packages may be combined should the benefits offered by the subcontractor be large enough to make it worthwhile.

A construction execution plan has been developed to facilitate construction planning with a particular emphasis on strategy, set-up, construction team and functional activities.

Commissioning includes those activities necessary for an effective transition between construction and mechanical completion when systems are turned over to the commissioning and start-up team.

The commissioning and start-up team is planned to be an integrated organization of plant operators, contractors and suppliers.

1.24 Recommendations

Recommendations for further work and associated costs are summarised in Section 26. The key recommendations relate to the concentrate marketing, main dump geotechnical stability, mine geoteotechnical studies, metallurgical test work, water balance and environmental/social/ permitting activities.

Project execution risk can be reduced through further detailed planning, materials sources, site geotech and early contractor involvement for sub-contract and major fabrication activities. However, this work should not be commenced until sufficient project finance is in place to ensure a continuous progression to construction completion.

2 Introduction

Bear Creek Mining Corporation (BCM) is a leading Peru-focused silver exploration and development company. Its common shares are listed on the TSX Venture Exchange in Canada and the Bolsa de Valores de Lima in Peru under the symbol “BCM”, and are posted for trading on the OTCQX Market in the U.S. under the symbol “BCEKF” and on the Börse Frankfurt in Germany under the symbol “OU6”. BCM is focused on developing their Corani Project located in the Department of Puno in southern Peru.

2.1 Purpose

BCM requested that Ausenco assist with updating the National Instrument 43-101 Technical Report (“Technical Report” or “Report”) previously prepared in 2017 by Sedgman, and 2015 by M3. This report was prepared in accordance with “Form 43-101F1, Technical Report” of the Canadian Securities Administrators National Instrument 43-101 (NI 43-101).

The Project includes mining, crushing, grinding and flotation of mixed sulfides, and transitional ores, for proven and probable Mineral Reserves containing 139 million tonnes of ore having a Life-of-Mine average grade of 51.3 g/t silver, 0.90% lead, and 0.55% zinc, and containing 229 million ounces of silver, 2.7 billion pounds of lead and 1.7 billion pounds of zinc over a period of 15 years of mining.

2.2 Sources of Information

This Technical Report is the product of technical contributions from a number of consultants and BCM personnel. Listed below are the “Qualified Persons” (as defined in the National Instrument 43-101) that each compiled and reviewed different sections of the report.

Table 2-1 describes the QP contributions by section.

- Greg Lane, FAusIMM, Chief Technical Officer of Ausenco Services Pty Ltd, is the co-ordinating author and Qualified Person ("QP") responsible for the Introduction, Project Description, Recovery Methods, Project Infrastructure, Market Studies, Capital and Operating Costs, Economic Analysis, Other Relevant Data and Information and Conclusions and Recommendations. While Greg Lane has taken responsibility for ensuring that the sections requiring contributions from all QPs accurately reflect those contributions, the individual QPs remain responsible for their own contributions for the Summary, Introduction, Reliance on Other Experts, Interpretation and Conclusions, Recommendations, References.
- Michael Meyer, MMSA QP, Principal Scientist of Meyer EPS Inc. and INSIDEO S.A.C., is the QP responsible for Environmental Studies, Permitting, and Social Impact.
- Kevin Gunesch, PE, Principal Mining Engineer of Global Resource Engineering Ltd (GRE), is the QP for the Property Description, Accessibility and Climate, and History.
- Terre Lane, MMSA QP, Principal Mining Engineer of GRE and Todd Harvey, SME QP, Director of Processing and President of GRE, are the QP's for Mineral Processing and Metallurgical Testing.
- Hamid Samari, MMSA QP, Senior Geologist of GRE, is the QP for Geology and Mineralization, Deposit Types, Exploration, Drilling, Sample Preparation Analyses and Security, and Data Verification.
- Terre Lane, MMSA QP, Principal Mining Engineer of GRE, is responsible for Mineral Resource and Reserve Estimates and Mining Methods.

- Denys Parra, SME Registered Member, General Manager of Anddes Asociados S.A.C.; (Anddes) is the QP responsible for Waste Rock and Tailings Management Facilities, Final Pit Limits and Designs, and the Site Water Balance.
- Eduardo Ruiz, EFG Register Member, General Manager of Amphos 21 Consulting Peru S.A.C., (Amphos 21) is the QP for Groundwater and Surface Water Studies, Closure Phase Water Balance, Monitoring, and Maintenance.
- David Arcos, EFG Register Member, Geochemistry Manager of Amphos 21, is the QP for Geochemical Studies.

Table 2-1: List of QPs responsibility by report Section

No.	Section name	Qualified Person
1	Summary	Greg Lane
2	Introduction	Greg Lane
3	Reliance on Other Experts	Greg Lane
4	Property Description and Location	Kevin Gunesch
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Kevin Gunesch
6	History	Kevin Gunesch
7	Geology Setting and Mineralization	Hamid Samari
8	Deposit Types	Hamid Samari
9	Exploration	Hamid Samari
10	Drilling	Hamid Samari
11	Sample Preparation	Hamid Samari
12	Data Verification	Hamid Samari
13.1 to 13.7	Mineral Processing and Metallurgical Testing	Todd Harvey
13.8	Continuous Predictive Metallurgical Model	Terre Lane
13.9	Interpretation	Greg Lane
14	Mineral Resource Estimates	Terre Lane
15	Mineral Reserve Estimates	Terre Lane
16.1 to 16.5	Mining Methods	Terre Lane
16.6	Waste Management	Denys Parra
16.7	Mining Equipment	Terre Lane
16.8	Work Schedule	Terre Lane
17	Recovery Methods	Greg Lane
18 (except 18.2.6 and 18.3)	Project Infrastructure	Greg Lane
18.2.6	Water supply and management	Denys Parra
18.3	Mine Waste Rock and Tailings Management Facilities	Denys Parra
19	Concentrate Market Studies and Contracts	Greg Lane
20 (except 20.1.1, 20.1.3, 20.3.7, 20.3.8 & 20.3.10)	Environmental Studies, Permitting and Social or Community Impact	Michael Meyer
20.1.1	Summary of air, noise, groudwater and surface water studies	Eduardo Ruiz
20.1.3	Summary of geochemical studies	David Arcos

No.	Section name	Qualified Person
20.3.7	Site wide water balance	Eduardo Ruiz
20.3.8	Closure phase water management	Eduardo Ruiz
20.3.10	Monitoring and maintenance	Eduardo Ruiz
21	Capital and Operating Costs	Greg Lane
22	Economic Analysis	Greg Lane
23	Adjacent Properties	Greg Lane
24	Other Relevant Data and Information	Greg Lane
25	Interpretation and Conclusions	Greg Lane
26	Recommendations	Greg Lane
27	References	Greg Lane

This report was compiled for BCM by Ausenco. The report is based on information and data supplied by the various authors and reviewed and accepted by the QPs.

Where possible, the authors have confirmed the information provided by comparison against other data sources, similar projects in Peru and South America or by field verification where checks and confirmations were not possible. Where estimates and approximations have been used, it is stated, and the assumptions made in making such estimates and approximations are also stated.

The authors have reviewed information supplied in, and related to, the previous technical reports. This report updates and supercedes the previous reports. The report retains text from the 2017 Technical Report where no significant changes have been made to the project.

This report conforms to the standards of a NI 43-101 Technical Report. It is based on the work completed to date for the Corani Project and evaluates the economics of a project that will operate at a processing rate of 27,000 tonnes per day as an annual average.

The report sets forth conclusions and recommendations, based on the authors' experience and professional opinion, which result from their analysis of work and data collected.

This report should be read in original context - all readers should refer to referenced documents for clarification of the original context.

2.3 Site Visit and Personal Inspections

The following site visits were made by QPs:

- Hamid Samari visited the site from August 7 to 10, 2017
- Eduardo Ruiz and David Arcos visited the site from November 28 to 29, 2013
- Kevin Gunesch visited the site on four separate occasions in 2011
- Denys Parra visited the site on September 8, 2011.

Other relevant site visits have been completed, notably:

- Ausenco's project team including, Jim Bryce, Project Director; Adam McCosh, Civil Engineer; and Tamara Champ, Project Manager in May 2018.

2.4 Terms of Reference

Ausenco was engaged by BCM to co-ordinate the preparation of, and to prepare substantial elements of, this NI 43-101 Technical Report for the Corani Project. The focus of the work completed by Ausenco, with input form other contributors, was optimisation of the ore processing plant and associated site infrastructure to reflect the potential value of the project based on available information. Ausenco and the other contributors produced a number of reports for BCM and this Technical Report summarises the work that was, when taken as a whole, completed to Feasibility Study level of accuracy. A separate feasibility study report was not compiled.

The units of production in this report are metric unless otherwise noted. Production is in tonnes (t). All monetary amounts are in June 2019 US dollars unless otherwise noted.

Lists of reference documents, acronyms, glossary of terms and summary of units of measure are provided in Section 27.

3 Reliance on Other Experts

The QPs for this report have relied on certain reports, opinions and statements of legal and technical experts who are not considered "Qualified Persons", as defined by NI 43-101.

Reports received from other experts have been reviewed for factual errors by the relevant QPs and determined that they conform to industry standards, are professionally sound, and are acceptable for use in this report. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statements and opinions expressed in these documents are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of these reports.

3.1 Mining Concessions

Legal review concerning the status of mineral concessions covering the Corani Project was conducted by Estudio Grau Abogados. The status report (Abogados, 2015) with respect to the mining concessions on the Project property stated that the claims are in good standing and that BCM owns the title. Control and current status were verified by GRE in October 2019 through an electronic database search of the Geologic Mining and Metallurgical Institute (INGEMMET). All concessions are in good standing.

3.2 Surface Rights

BCM provided the information on surface rights for the Project.

3.3 Taxation and Social Costs

BCM provided all estimates of Peruvian taxes, which were reviewed by a Peruvian law firm (Rodrigo, Elias & Medrano Abogados). Social costs were provided by BCM based on historical experience within the Corani region.

4 Property Description and Location

4.1 Location

The Project site is in the Andes Mountains of southeastern Peru at elevations of 4,800 to 5,200 masl, specifically within the Cordillera Vilcanota of the Eastern Cordillera. The site is in the Region of Puno, immediately northeast of the continental divide that separates Pacific drainages from Atlantic drainages. The site location is approximately 160 km in a direct line to the southeast of the major city of Cusco, with UTM coordinate ranges of 312,000E to 322,000E and 8,443,000N to 8,451,000N. Figure 4-1 illustrates the general location on the map of Peru. The nearest town of significant size and infrastructure is Macusani, which is located approximately 30 km to the east of the Project.

Figure 4-1: Corani Project location in Peru (Source: Google earth)

4.2 Mineral Tenure

4.2.1 Summary

The Project consists of a series of thirteen (13) mineral claims or concessions. The 12 original concessions are grouped in the UEA (Administrative Economic Unit) with the 13th (recently acquired) to be integrated into the UEA in 2020. Mineral concessions in Peru are filed with the Instituto Nacional de Concesiones y Catastro Minero (INACC), which is part of the Ministerio de Energía y Minas in Peru (MINEM). Claims can vary in size from 100 to 1,000 ha. They are rectangular geometries parallel to the UTM grid system employed in the district. The Corani Project is in the district of Corani, province of Carabaya, department of Puno, in Peru, and covers an aggregate extent of approximately 5,480 ha.

4.2.2 Purchase agreements

On March 15, 2007, the Company and Rio Tinto Mining and Exploration Ltd. ("Rio Tinto") executed a definitive option and shareholders' agreement (the "Option Agreement") in respect of the Corani Property. The Option Agreement formally defined and confirmed the terms as set out in the Letter of Understanding signed between the parties on January 19, 2005. On January 15, 2008, the Company made the final $3 million payment to Rio Tinto under the Option Agreement, resulting in the Company owning 70% of the Corani Mining Property.

On March 6, 2008, Bear Creek entered into an agreement (the "Purchase and Sale Agreement") with Rio Tinto, which was subsequently amended to purchase Rio Tinto's remaining 30% interest in the Corani Project and extinguish all of Bear Creek's future payment obligations, royalties, and Rio Tinto's back-in rights under the Option Agreement.

On July 17, 2008, the Company agreed to issue an additional 120,000 common shares to Rio Tinto, in consideration for which Rio Tinto extended $15 million of the $20 million cash payment that had been required to be made under the Purchase And Sale Agreement.

On February 27, 2009, the Company entered into an amendment agreement (the "Amendment Agreement") with Rio Tinto with respect to its purchase of Rio Tinto's remaining 30% interest in the Corani Project. Under the Amendment Agreement, Rio Tinto agreed to restructure the final two cash payments of $15 million and $25 million. In consideration for deferring most of these payments for several years, the purchase price increased from $75 million to $77.2 million, representing an increase of $2.2 million, of which $36.1 million had been paid in shares or cash.

On February 3, 2011, the Company entered into an additional amendment agreement whereby Rio Tinto agreed to accept a final payment of $23 million in lieu of and in full satisfaction of the remaining two cash payments of $10 million and $15 million. Accordingly, the Company acquired a 100% interest in the Corani project.

This final payment extinguished all security interests, share pledges and other encumbrances that Rio Tinto held over the Corani Project and Company's other assets. Copies of the Purchase and Sale Agreement, the Amendment Agreement, and the 2011 Amendment Agreement may be obtained under the Company's profile on the SEDAR website (www.sedar.com).

4.2.3 Property identification

The Estudio Grau Report (Abogados, 2015) indicated that BCM controls the surface property and established the validity of the original twelve (12) mineral concessions that were entitled by the company between 2011 and 2015. The legal status remains the same in 2019 in that BCM and its subsidiaries own 100% of the title of the original twelve (12) mineral concessions of the Corani Project and one (1) newly acquired mineral concession, listed in Table 4-1. Control and current status were verified by GRE in October 2019 through an electronic database search of the Geologic Mining and Metallurgical Institute (INGEMMET). All concessions are in good standing.

Table 4-1: Mineral concessions comprising the Corani Project

Identification Code	Name	Holder	Available Hectares [1]	Status	Province	District
10250805	Chaupitera	Bear Creek Mining S.A.C.	800	Current	Carabaya	Corani
10251005	Corani 100	Bear Creek Mining S.A.C.	5 [2]	Current	Carabaya	Corani
10251105	Corani 200	Bear Creek Mining S.A.C.	21.9730 [2]	Current	Carabaya	Corani
10068505	Corani 5	Bear Creek Mining Company Sucursal Del Perú	93.2601 [2]	Current	Carabaya	Corani
10289403	Corani I	Bear Creek Mining S.A.C.	300	Current	Carabaya	Corani
10289503	Corani Ii	Bear Creek Mining S.A.C.	300	Current	Carabaya	Corani
10021905	Corani Iii	Bear Creek Mining Company Sucursal Del Perú	300.0074	Current	Carabaya	Corani
10289203	Minazpata 1	Bear Creek Mining S.A.C.	1,000	Current	Carabaya	Corani
10289303	Minazpata 2	Bear Creek Mining S.A.C.	300	Current	Carabaya	Corani
10038904	Minazpata 3	Bear Creek Mining S.A.C.	1,000	Current	Carabaya /Melgar	Corani /Macusani/ Nuñoa
10357604	Minazpata 4	Bear Creek Mining S.A.C.	159.8808 [2]	Current	Carabaya	Corani
10250905	Pacusani	Bear Creek Mining S.A.C.	900	Current	Carabaya	Corani
10214917 [3]	Minazpata Oeste 1	Bear Creek Mining SAC	300	Current	Carabaya /Melgar	Corani/Nuñoa
	All	All	5,480			

Obtained from: http://www.ingemmet.gob.pe/sidemcat

1 Available area not including overlaps with prior mineral concessions

2 Overlaps with existing concession

3 The concession has recently been acquired and will be added to the UEA in 2020.

The Corani Project comprises thirteen (13) metallic mineral concessions (collectively the "Corani Project"). BCM owns the Corani Administrative Economic Unit (UEA) which gathers the original twelve (12) concessions of the Corani Project, as approved by INGEMMET Presidency Resolution

No. 4679-2013-INGEMMET / PCD / PM dated December 13, 2016. The thirteenth concession will be incorporated into the UEA in 2020.

The Corani Project is in the district of Corani, province of Carabaya, department of Puno, in Peru, and covers an aggregate available extent of approximately 5,480 ha.

The location of the Corani Project is fixed, for all legal purposes, by the UTM coordinates (UTM Zone 19S, Datum WGS 84) for each of their vertices shown on documents recorded in the Public Registry.

Figure 4-1 shows the location of the project within Peru. Figure 4-2 shows a map of the Corani mineral concessions within the area.

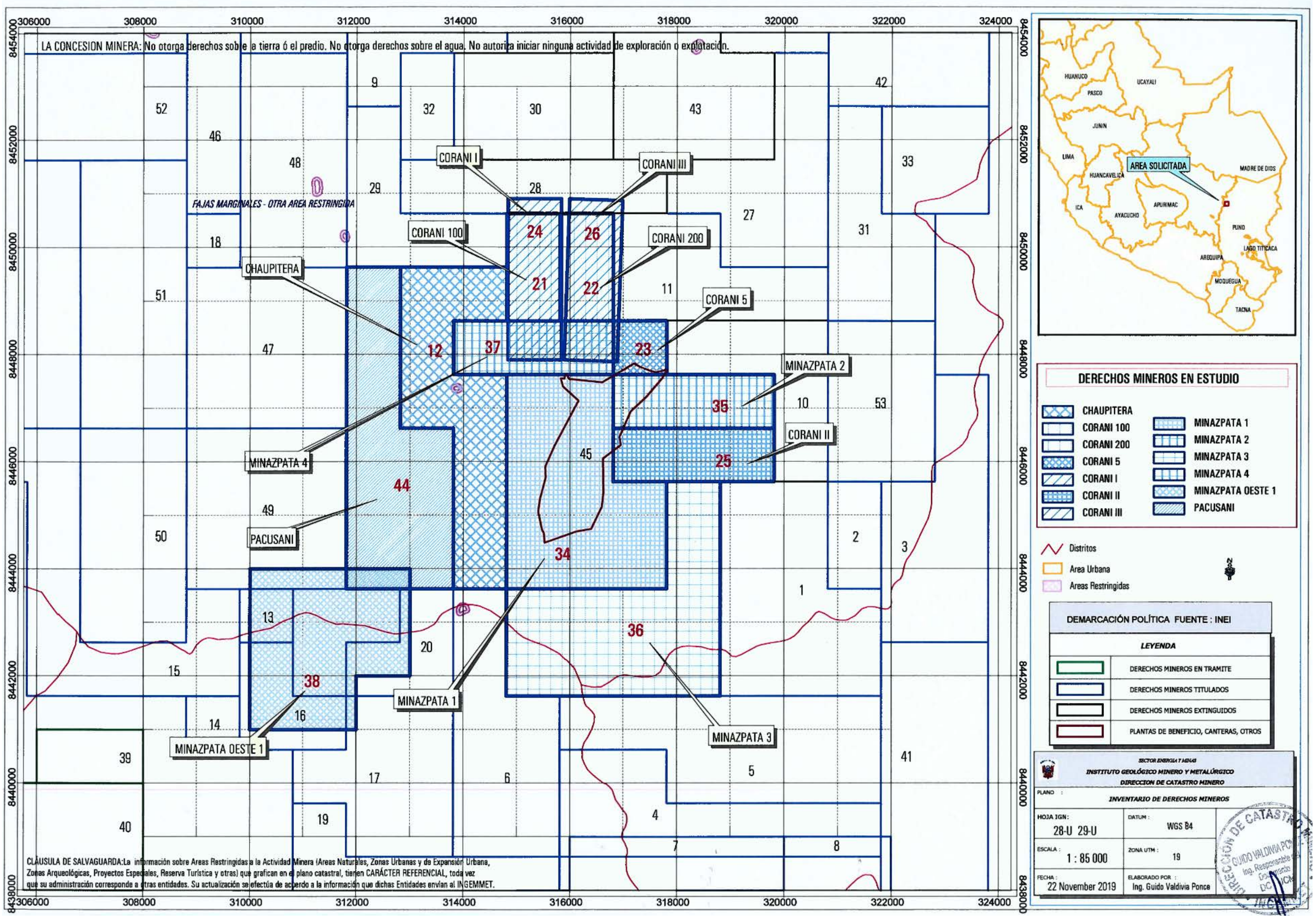


Figure 4-2: Map of Corani mineral concessions

4.2.4 Maintenance obligations

According to the Estudio Grau Report (Abogados, 2015), and applied until 2018:

The original twelve (12) mineral concessions comprising the Corani Project are subject to compliance with payment of annual license fees in the amount of $3.00 per ha ("License Fees") This applies to the one (1) new concession once it is incorporated into the UEA.

The mineral concessions comprising the Corani Project are also subject to compliance with any one of the three alternative obligations:

a) minimum required levels of annual production of at least $100 per ha in gross sales ("Minimum Production")

b) payment of an additional amount referred to as a Penalty

c) minimum exploration or investment expenditures of at least 10 times the Penalty

Compliance with any one of these three maintenance obligations, together with timely payment of License Fees, is required to maintain them in good standing. Failure to comply with License Fee payments or minimum or penalty payments for two consecutive years causes the forfeiture of the mineral concessions.

The mineral concessions comprising the Corani Project were subject to the obligations of Minimum Production, Penalties and Exploration expenditures in accordance with the maintenance regime in force as of October 2008 whereby:

- The minimum production will be equivalent to one Peruvian Tax Unit Value per year (approximately U.S. $1,295.00) per ha granted for metallic minerals, and 10% of one Tax Unit per year per ha granted for non-metallic minerals ("Minimum Production").
- Failure to attain Minimum Production will trigger the obligation to pay a penalty equivalent to 10% of the Minimum Production per year per ha, until the year in which the Minimum Production is attained.
- Year 2028 shall be the maximum deadline for the mineral concessions comprising the Corani Project to attain Minimum Production. Failure to do so will result in the forfeiture of these mineral concessions.

Starting in 2019 the Corani Project is subject to compliance with one of three alternative obligations:

a) minimum required levels of annual production of at least one UIT (Peruvian Tax Unit) per ha in gross sales ("Minimum Production")

a) payment of an additional amount referred to as a Penalty

b) minimum investment expenditures of at least 10 times the Penalty

From 2019 onwards, the calculation of the minimum investment has been divided into two parts. During years 11 to 15 following the granting the title of the mining concession, the penalty is 2% of the minimum production. For the years between year 16 and 20 following the receipt of title for the mining concession, the penalty is 5% of the minimum production. In the case of the Corani Project, the 2% penalty was applicable for 2019. If the project does not achieve the minimum production, or investment, the higher rate of 5% is applicable from 2020 onward.

BCM has made the minimum annual expenditure in 2019 and was not obliged to pay the penalty. BCM believes the same scenario would be repeated in 2020. Additionally, the Corani UEA provides the right to report exploration and investment expenditures incurred in any one or more mineral concessions to the benefit of the Corani UEA, as a whole.

Table 4-2 shows the annual amounts for each of the alternative maintenance obligations to keep the Corani Project in good condition from 2012 to 2019. Projected amounts for penalties are shown for 2020 if minimum expenditures are not made. BCM has completed all required payments for all mineral concessions in 2019.

Table 4-2: Corani mineral concessions maintenance obligations (BCM, 2019)

Alternative Annual Maintenance Obligations Years 2012-2016										
Mineral Concession	Annual License Fees Years 2012-2020	Minimum Production in Gross Sales ($)			Minimum Expenditures ($)				Penalty Payment ($)	
	($)	2012-2018	2019	2020	2012-2013	2014-2018	2019	2020	2012-2019(1)	2020(2)
Corani I	900	30,000	372,978	377,472	18,000	60,000	74,596	188,736	0	18,874
Corani II	900	30,000	372,978	377,472	18,000	60,000	74,596	188,736	0	18,874
Corani III	900	30,001	372,987	377,481	18,000	60,001	74,597	188,740	0	18,874
Corani 100	15	500	6,216	6,291	300	1,000	1,243	3,146	0	5,867
Corani 200	66	2,197	27,318	27,647	1,318	4,395	5,464	13,824	0	315
Corani 5	280	9,326	115,947	117,344	5,596	18,652	23,189	58,672	0	1,382
Minazpata 1	3,000	100,000	1,243,259	1,258,238	60,000	200,000	248,652	629,119	0	50,330
Minazpata 2	900	30,000	370,978	377,472	18,000	60,000	74,596	188,736	0	62,912
Minazpata 3	3,000	100,000	1,243,259	1,258,238	60,000	200,000	248,652	629,119	0	18,874
Minazpata 4	480	15,988	198,773	201,168	9,593	31,976	39,755	100,584	0	62,911
Chaupitera	2,400	80,000	994,608	1,006,591	48,000	160,000	198,922	503,295	0	10,058
Pacusani	2,700	90,000	1,118,933	132,415	54,000	180,000	223,787	566,207	0	56,621
Total	15,540	518,012	6,438,234	5,517,829	310,807	1,036,024	1,288,049	3,258,914	0	325,892
	2017-2020 Fees ($)	2017-2018	2019	2020		2017-2018	2019	2020	2017-2019(1)	2020(2)
Minazpata Oeste 1(3)	2,400	80,000	994,608	1,006,591		160,000	198,922	503,295	0	10,058

(1) Between 2012-2019, BCM did not have the obligation to pay penalty because the minimum annual expenditure was reached. The same scenario would be repeated in 2020.

(2) Projected penalty, if BCM does not meet the minimum annual expenditure.

(3) Applicable from the time that the concession is incorporated into the UEA

Finally, in 2018 BCM obtained the approval of the construction permit for the Processing Plant for the Corani project. In 2018, BCM had been required to pay for that right, related to its processing capacity. The fee is equivalent to $ 12,472 per annum at a processing rate of 22,500 tonnes per day. The processing rate will be amended to 27,000 tonnes per day with the filing of the third ITS. BCM is current on these payments.

Table 4-3: Corani benefit concession maintenance obligations (BCM, 2019)

Year	Capacity kt/d	Annual Licence Fees ($)
2018	22,500	12,472.10
2019	22,500	12,569.80
2020	22,500	15,869.08 (1)

(1) Projected using 2019 tax basis

4.2.5 Legal standing

According to Estudio Grau (Abogados, 2015), the legal standing of the Corani Project is as follows:

- The original twelve (12) mineral concessions comprising the Corani Project are valid and in good standing.
- They were validly applied for and granted title to concession by the competent governmental authority.
- Each of the original twelve (12) mineral concessions comprising the Corani Project is designated metallic as a mineral concession and allows its titleholder or lessee the exclusive right to explore and exploit all metallic minerals located within their internal boundaries. BCM also has the surface rights of the concessions tied to the Corani Project.
- The mineral concessions comprising the Corani Project have been granted to the titleholders for an indefinite period of time, provided that maintenance obligations, including license fee payments, minimum production, investment and/or payment of applicable penalties are attained when due. The year 2028 is the current legal absolute limit as to when production needs to occur with respect to the mineral concession comprising the Corani Project; failure to do so will cause their termination or expiry. The mineral concessions comprising the Corani Project will therefore remain valid through the maximum legal deadline to be put into production as long as the titleholder or lessee continues complying with annual license fee payments, qualified investments, and/or applicable penalties.
- Exercise of the rights derived from the original twelve (12) mineral concessions comprising the Corani Project, including the right to explore, develop and further exploit, on an exclusive basis only the designated minerals within the internal boundaries of the mineral concession, is subject to the awarding of the required permits, authorizations and approvals, including relevant surface lands.

The new concession recently acquired has the same status and rights described above as the original twelve (12) concessions.

4.3 Surface Rights

BCM controls the surface rights that cover the entire project area, including the open pit, waste dump, process plant, water ponds, camp, and ancillary facilities required for operation. Surface rights total 2,424 hectares.

4.4 Royalties

There are no royalties existing on the thirteen (13) mineral concessions comprising the Corani Project.

4.5 Environmental Liabilities

Historical mining activities have been carried out in the vicinity of the proposed mine and associated facilities. The history of the Project site, including ownership and any known mineral exploration and production, are described in Section 6 of this report.

In accordance with Peruvian Law 28271, generators of environmental liabilities are responsible for remediation activities. Therefore, if historical environmental liabilities are defined, responsibility for these lies with the original generator; the new investor is not responsible for either the consequences of such liabilities or the activities of remediation. However, BCM may assume responsibility for them in order to expedite the development of the site.

In December 2010, Walsh Environmental undertook an environmental-liabilities study in order to declare to the MINEM the existence of liabilities left from previous mining activities in the Corani project area (Walsh Environmental, 2014).

These previous mining activities have left excavations, stopes, test pits, and mine portals on the site. During the study site visit, the location of each liability (either previously known or discovered during the study) was inventoried and registered with the MINEM. A total of 141 liabilities were recorded; however, it is possible there are others. Several the environmental liabilities are located within the boundaries of proposed project components. Therefore, for ground-breaking and or development activities to occur, BCM may need to assume responsibility for these.

Environmental liabilities associated with development of the property (past and future) are managed through an Environmental Closure Plan or Plans. The Environmental Liabilities Closure Plans that were approved by the Peruvian Government in April 2015 were updated in 2018. These plans have been reviewed by the MINEM in 2018, according to Peruvian legislation. MINEM Directive Resolution No. 173-2018-MEM-DGAAM of September 14, 2018 has approved the update.

4.6 Permitting

BCM obtained the permits required for the previous field exploration activities and has identified the permits required for the construction, exploitation, and closure phases. An outline of the national, territorial, and municipal legislation, and the associated approvals and permits that apply to the Project, has been compiled and is provided in the Permitting Handbook Corani Project (Vector Perú S.A.C., 2009). There has been no material change in the permit requirements, notwithstanding some changes to the project description. Table 4-4 presents a summary of the permits received and required.

Table 4-4: Summary of permit requirements by phase

Permit	Construction	Exploitation	Closure
ESIA (modifications through the life of the mine may apply)	C	C	C
Permits related for the construction of landfills outside the area of mining concessions	F		
Closure Plan (modifications through the life of the mine may apply)	C	C	C
Certificate of Non-Existence of Archaeological Remains – CIRA	C	C	C
Archaeological Monitoring Plan - PMA	C	NA	NA
Water licence accreditation of the Corani project	C	C	NA
Authorization for the execution of surface water use works	C	NA	
Surface water use license	C	F	
Groundwater use license	NA	F	
Sanitary authorization for wastewater treatment system and discharge	F	F	
Sanitary authorization for drinking water treatment system	F	F	
Fuel Direct Consumer's favourable technical report	F	F	
Registration as a direct consumer of liquid fuels - fixed or mobile facilities	F	F	
Authorization for explosives use	F	F	
Explosives Storage Magazine operation license	F	NA	
License for explosives handlers	F	NA	
Authorization for explosives transportation	F	F	
Identification code for users of Controlled Chemicals (IQPF)	F	F	
Verifying deed for the purchase and transportation of IQPFs used by companies	F	F	
Authorization for opening Special Registry of IQPFs	F	F	
Incorporation in the Special Registry for IQPFs	F	F	
Monthly reports of IQPF Special Registry	F	F	
Annual Opinion for making, marketing, and warehousing explosives of civilian use and related goods	F	F	

Permit	Construction	Exploitation	Closure
Approval of the pre-operation study for the connection to the SEIN of the LTE in 138 kV Antapata Corani and Substations	C		
Electric Transmission Line ESIA	C		
Building license for the construction of the Electric Substation	C		
Definitive Concession for energy transmission line	F		
Individual license for radioactive facilities' handling	F	F	
Installation license for the fixed nuclear measuring equipment	F	F	
Transportation Guide of hazardous materials and wastes	F	F	
Insurance for transportation of hazardous materials and wastes	F	F	
Certification of transportation personnel	F	F	
Registry for ground transportation	F	F	
Special driver's license	F	F	
Beneficiation Concession	C	NA	
Title and Operating Authorization of the beneficiation plant		F	
Authorization to start operation		F	
Posting Financial Assurance		F	
Final Closure Plan (2 years before final closure)			F
Final Closure Certificate ("Exit ticket")			F

C=COMPLETED, these permits have been obtained for the early construction phase.

F=FUTURE, these permits will be required before of the execution of future activities or future construction.

NA=Not Applicable.

4.7 Water Supply

The approval of the Water Right for the Corani project was approved by Directive Resolution No. 354-2017-ANA / AAA-XIII and specified by Direction Resolution No. 105-2018-ANA / AAA-XIII MDD.

The approval of the Water Right for the future camp was approved by Directive Resolution No. 0222-2018-ANA-AAA.MDD.

The Authorization for water use for construction was approved by Directive Resolution No. 199-2018-ANA-AAA.MDD of July 18th, 2018 and specified by Directive Resolution No. 0340-2018-ANA-AAA.MDD of December 5th, 2018.

4.8 Environmental and Permitting

The main environmental approval required to begin mining activities is an Environmental and Social Impact Assessment (ESIA). In 2013, the Ministry of Energy and Mines approved the ESIA based on the Feasibility Study prepared in 2011. The Closure Plan was approved in April 2015 with an update approved in 2018.

Subsequently, two modifications to the ESIA have been approved, which reflect the optimization of the Corani project based on the Front End Engineering Design (FEED) Study. Included in these modifications is co-disposal of tailings with mine waste and removal of the tailing dams. These changes have reduced the project impact area by more than 40%.

In May 2018, the MEM approved the Authorization of Exploitation Activities (Corani Project Mining Plan), which authorizes the construction of the Corani Project and the construction of auxiliary and complementary mining facilities, such as access roads, the mine camp, and maintenance and storage buildings (Directorate Resolution No. 0119-2018-MEM-DGM).

Also, in June 2018, the MEM approved the Authorization for the Construction of the Beneficiation Plant, which includes the main waste and tailings storage facility, surface water management and water supply well construction for the plant, and other auxiliary facilities (Resolution No. 0570 - 2018-MEM-DGM / V).

Also, BCM has obtained the Certificate of Inexistence of Archaeological Remains (CIRA) for 100% of the Corani Project Area. Also, based on the CIRA, in July 2018, the "Archaeological Monitoring Plan for the Corani (PMA) - Puno Project" was approved by the Ministry of Culture. With this permit, all the permits and approvals linked to the start of construction of the Corani project were completed.

In September 2018, the start of the early construction works for the Corani project was notified by BCM to the MINEM. These "Early Works" are presently being executed.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

Existing access to the Project site is primarily by road from the town of Macusani (located on the paved dual lane Interoceanic Highway), which is more readily accessible from the town of Juliaca, also serviced by commercial airlines from Lima. This route typically takes 4.5 to 5 hours (Figure 5-1). From Juliaca, the route generally trends north towards the city of Azángaro on the paved Interoceanic Highway. The Interoceanic Highway extends approximately 180 km between Azángaro and Macusani. At Macusani, the route extends west and northwest for approximately 60 km to the mine site on improved gravel roads.

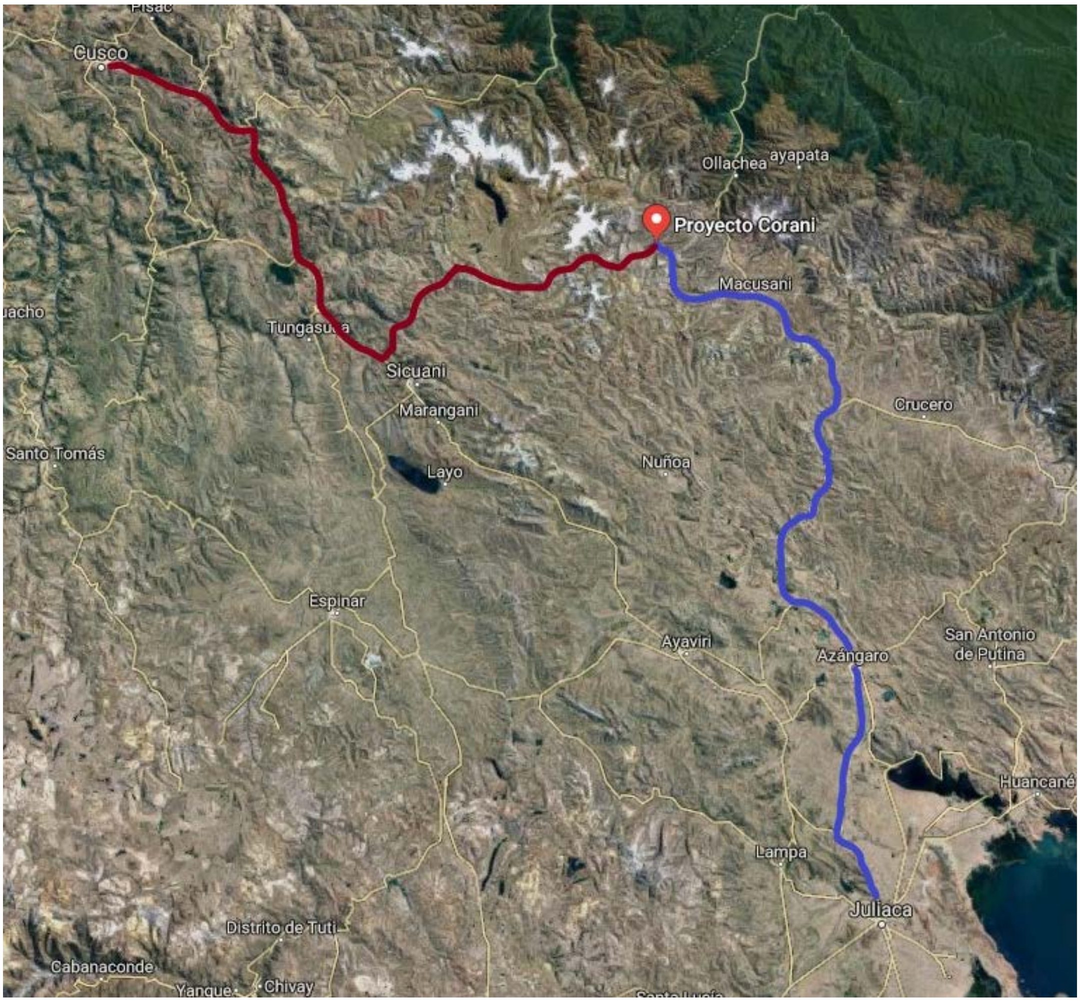


Figure 5-1: Map of existing access to the Project (source: Google earth)

The city of Cusco is serviced by commercial airlines from Lima. There are other access routes to the site from Cusco, taking approximately 6 hours by vehicle on increasingly primitive roads approaching the site. The access route is shown in Figure 5-1 and passes through Sicuani and the town of Santa Rosa, which is located approximately 208 km southeast of Cusco on a good paved road. From there, the route extends approximately 33 km northeast on improved gravel roads to the village of Nuñoa and continues northeast for approximately 27 km on a less improved gravel road to the small village of Huaycho. From Huaycho, the access route continues north on an unimproved gravel road for approximately 70 km and ascends a mountain pass to the Project site.

5.2 Local Resources and Existing Regional Infrastructure

5.2.1 General

The nearest town of significant size and with significant infrastructure is Macusani, which is the capital of the Province of Carabaya in the Region of Puno. Macusani is approximately 30 km east of the Project in a direct line. The access road from Huiquisa Bridge to the Permanent Camp will be improved. The length of the proposed Mine Access Road connecting the process Plant to Macusani is anticipated to be approximately 64 km.

Macusani has a total area of approximately 1,030 square km and a population of approximately 12,600, and its economy is based mainly on agriculture and transportation of agricultural products.

Infrastructure in the town of Macusani includes a national highway - the Interoceanic Highway - currently complete from the Peruvian Port of Matarani to the town of Macusani. Other paved and unpaved roads, trails, and footpaths allow access to most areas of the municipality.

The Project site is in the district of Corani, also in the Province of Carabaya. The closely orientated campesino communities of Chacaconiza and Quelcaya, which have a joint population of approximately 205 families (82 and 123 families, respectively), will be directly impacted by the mine development in terms of landholding, rights to water, employment, etc.

Chacaconiza and Quelcaya are communities that maintain a fragile, high altitude economy. Both communities are below the poverty line, with few resources for economic and social development. The main economic activity in these communities is the raising of alpacas. Approximately 90% of their economy is dependent on this activity, which is augmented to a very marginal degree by trading and seasonal migration.

5.2.2 Available labor force

The community consultation undertaken with the Chacaconiza and Quelcaya communities to date has included a proposal for mining employment, generating widespread acceptance, mainly among younger community members, the teachers at local educational facilities, and community leaders. The current labour force is generally unskilled, mainly working on highway remediation and maintenance. A technical training program is planned to develop the skills of community members to fulfil employment requirements of the Project, which will include agreements with universities and institutes to improve the local population’s vocational skill levels. The training program will include a system of scholarships that will allow the most successful students to occupy positions of greater responsibility on the project. The training program has been designed to be conducted over a 5 year period. After this, BCM will continue to support the training program.

Mining and services-related training will be segmented by age group, to allow older people to be trained for simpler tasks, while younger people will have access to jobs that demand more knowledge and specialization, such as the operation and maintenance of heavy machinery, woodworking, and electrical work.

The Project’s requirement for labour will exceed the labour resources available in the Chacaconiza and Quelcaya communities. A ranking system will be developed regarding geographical location of employment applicants, together with categorization and quantification of the labour force required.

5.2.3 Power

The National Interconnected Electric System (SEIN) is the source of power supply for the project. The San Gabán II Hydroelectric Power Station is located on the San Gabán River, some 260 km northwest of the city of Puno and 100 km east of Cusco. The 138-kV power transmission line that connects the Hydroelectric Power Station of San Gabán II (CH San Gabán II) with the SEIN at the Azángaro Substation (SE Azángaro), passes through the neighbouring areas of the Project, near the town of Macusani. Therefore, the Project's recommended access to power supply is from the SEIN, connected to this transmission line.

At the end of 2017, BCM undertook to build an electrical substation (the Antapata substation) near the city of Macusani, the largest and closest city to the Corani project, located on the Interoceanic Highway, approximately 30 kilometers directly east of Corani (approximately 64 kilometers by road). The construction activities of Antapata began in September 2018, and BCM issued a purchase order for the transformer in October 2018. The transformer was delivered in September 2019. All activities are carried out through its subsidiary company, Electro Antapata SAC.

This substation will be used to direct electricity to a future power line that will supply the Corani project and to provide a constant power supply to Macusani residents, who experience regular power outages.

In March 2018, the Economic Operation Committee of the National Interconnected System (COES), the official entity of the Peruvian government that regulates and approves the viability of this type of energy projects, approved the "Update of the Pre-Operational Study for the connection to the SEIN of the Transmission Line in 138 kV Antapata - Corani and Substations," which includes the conformity of Empresa de Generación Eléctrica San Gabán S.A in relation to the future interconnection.

The Ministry of Culture / Regional Puno has granted the CIRA for the construction work of the Antapata Substation and for the access to the trans-oceanic station. The approval was given with CIRA No. 124-2019-DDCPUN / MC of May 2019.

In August 2018, the ownership of one (1) ha of land for Electro Antapata SAC was registered in Public Registries. This land is where the Antapata substation is being built and equipped.

With Direction Resolution No. 175-2018-GRP-GRDE-DREM-PUNO / D of August 2018, the Semi-Detailed Environmental Impact Study of the Project "Transmission Line 138 kV Antapata substation - Corani substation and Primary Line 22.9 KV to System" was approved.

To date, the engineering is complete, and the civil works for the Antapata Substation are more than 90% completed. The commissioning of the equipment has already begun.

The main facilities that have a direct impact on the Project are described below and are represented schematically in Figure 5-2.

San Gabán II Hydroelectric Power Station

This hydroelectric power station is owned by Empresa de Generación Eléctrica San Gabán S.A., a state-owned company in charge of the operation of the plant's facilities since the end of its construction in the year 2000. The plant's characteristics are the following:

- number of units: 2
- power (each): 55 MW
- energy (annual average): 800 GWh

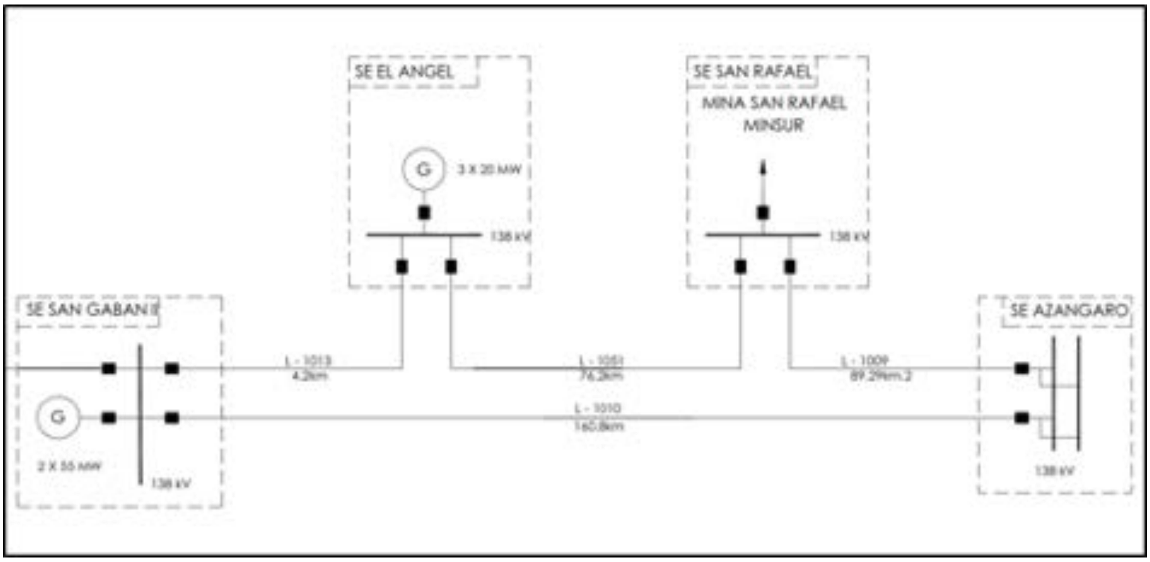


Figure 5-2: Existing facilities schematic

138 kV transmission line Substation San Gabán II – Substation Azángaro

The energy produced by the hydroelectric power station is delivered to the SEIN at the Substation Azángaro. The characteristics of this transmission line are the following:

- 2 circuits:
  - L-1010 (first circuit)
  - L-1013, L-1051 and L-1009 (second circuit)
- tension: 138 kV
- length: 160.8 km
- capacity by circuit: 120 MVA

El Angel hydroelectric power station with a capacity of 60 MW, operated by GEPSA company (Generadora de Energía del Perú S.A.) and San Rafael substation of Minsur Mining company are also connected to the second circuit.

Azángaro Substation

The Azángaro substation is a state-owned substation, granted in concession to Red Eléctrica del Perú (REP). It is part of the southern transmission ring and has a three-winding transformer with a capacity of 12/12/5 MVA and voltage of 138/60/22.9 kV. The substation San Rafael is also connected from this substation through a 60 kV transmission line, which is on service to Antauta town.

Southern Transmission Ring

The Peruvian Southern Transmission Ring interconnects the cities of Azángaro, Juliaca, Puno, Moquegua, Arequipa, Tintaya and Ayaviri. The Southern Transmission Ring is represented in Figure 5-3. At the Socabaya substation of Arequipa city, the southern ring gets interconnected with the electrical system of the center-north, integrating the National Interconnected Electric

System (SEIN). The main transmission lines between Puno, Azángaro, Tintaya and Arequipa are 138 kV and between Azángaro, Puno, Moquegua, Arequipa and Tintaya are 220 kV.

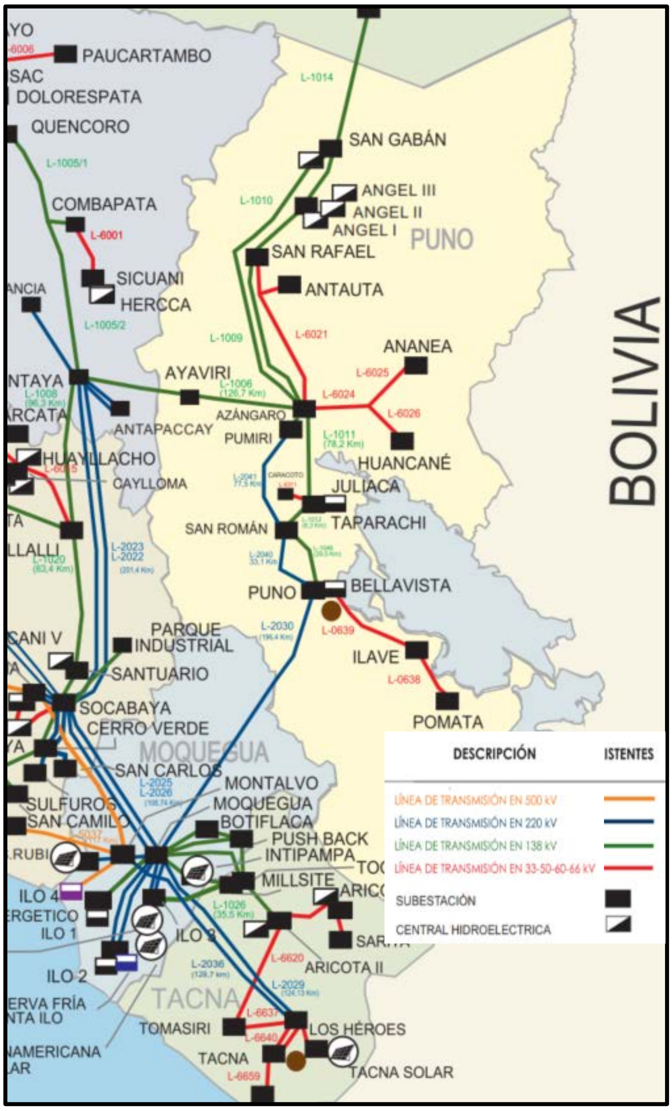


Figure 5-3: Southern electric ring schematic (COES, 2019)

5.2.4 Water

Surface water in the region is typically taken for farming and livestock watering, as well as essential human needs, such as a drinking and bathing. Surface water and water from springs and marshlands is collected in catchments in the communities of Chacaconiza and Quelcaya and is distributed to basic water supply systems that conduct water to distribution points located in public squares of each respective community.

Near the Project site, hydraulic works related to irrigation or water storage were not observed. It was noted that local inhabitants have constructed handmade canals to irrigate pastoral areas, although the structures are considered temporary and many have been abandoned depending on irrigation requirements.

5.3 Climate

The Corani Project meteorological station is in the vicinity of the proposed plant site. Almost eleven years of data are available since the station was commissioned in December 2008.

The climate at the project site is characterized by an estimated average annual precipitation of 717.1 millimetres (mm), with the highest values recorded between October and April (89% of the annual precipitation). The annual average evaporation was determined to be on the order of 810.4 mm, with the highest monthly evaporation rates occurring in August (97.1 mm) and the lowest monthly evaporation occurring in March (43.5 mm).

Monthly average temperature vary from 1.2 to 3.3°C. Monthly average minimum temperature vary from -2.1°C to 1.5°C while the monthly average maximum temperature vary from 3.3°C to 4.8°C.

The average relative humidity goes from 65 to 75%, with monthly averages ranging from a low of 51% in July to a high of 83% in February.

The annual average wind speed is estimated to be 2.2 m/s, with monthly averages ranging from 1.9 m/s in April to 2.4 m/s in July. The wind direction is generally from the southeast.

Limited comparison of the site data may be drawn to other weather stations in the region. Several regional weather stations have relatively long data records. However, all the available stations are a significant distance from the project and only general seasonal trends correlate with the project meteorology station.

Engineering designs should employ appropriate conservatism based on the limited site climate data available. The site climate will allow for year-round operations, with normal operating delays for conditions such as snow and fog. Freeze protection will be required for all hydraulic works.

5.4 Physiography and Vegetation

The Project site is in the eastern Andes mountain range, between 4,800 and 5,200 masl. The area is characterized by mountainous terrain dominated by volcanic rock, above which sits glacial gravel. The lithologic and climatic conditions have given rise to a series of cirques or bowl-shaped, steep-walled basins. During periods of rainfall, the valley floors collect precipitation, allowing the generation of small wetlands (bofedales).

Apart from the vegetation associated with the wetlands mentioned above, areas of “puna” or alpine tussock grassland occupy the valleys and moderate to steep slopes. The areas above 4,700 masl mostly consist of steep mountainous slopes, where erosion and climatic conditions largely prevent the development of soils or vegetation. These areas are scarcely vegetated, with species specially evolved to withstand the harsh conditions. The naturally occurring acidic soils related to oxidation of sulfide bearing materials and the resulting ARD from exposed mineralized zones within the project area has also prevented the development of vegetation where these conditions occur.

5.5 Existing Local Infrastructure

Existing infrastructure at the project site consists of an accommodation facility that can accommodate 80 personnel. Internet and telephone communication are provided via satellite connection. Non potable water for camp operations is sourced from the nearby creek. Potable water is delivered via truck. Water for drilling operations is sourced from surface water ponds and drainage flows throughout the property.

5.6 Planned Infrastructure

Planned infrastructure for the future mining operation is detailed in Section 18.

6 History

6.1 Prior Ownership and Production

Prior to the early 1950s, mineral exploration in the Corani district consisted of shallow prospect pits and adits in the northern portion of the current Project area. These prospects are of unknown age and may date back to colonial Spanish time. Antimony prospects south and east of the property reportedly were active in the early 1900s, when there was limited antimony production (Petersen C. , 1967).

The first modern evaluation of silver-lead mineralization began with the location of mineral concessions in 1951 by Augusto Leon y Leon. In 1953, Fernando de Las Casas visited the site and prepared a geological report titled "The Negrominas – Corani District." He mentioned that the rocks exposed in the area covered by the Negrominas Claims consist principally of a series of rhyolitic tuffs, breccias, and flows tilted to the northeast. Also, he determined that two main types of ore-bearing structures are distinguished at Negrominas. Compañía Minera Korani was formed in 1956 to develop the silver-lead mineralization previously prospected. The mines were developed and operated from 1956 to at least 1967; initially producing 80 tonnes per day (tpd) of ore. In 1967, Compañía Minera Korani was two-thirds owned by Compañía Minera Palca and one-third owned by M. Hochschild. Total historical production is uncertain but is estimated at 100,000 tonnes of silver-lead-zinc ore. In early 1967, estimated mine production was reported at about 3,400 short tons per month, with grades of 7.0-9.0% lead, 2.3% zinc, and 8.0 to 11.0 oz/ton silver (Petersen C. , 1967). Figure 6-1 shows some of the historical equipment used at the site.

Figure 6-1: Part of the old flotation circuit (BCM, 2017)

Historical maps of the underground workings show development on four levels (4820, 4843, 4860, and 4870 m levels for 50 m vertically) that extend over an area of approximately 500 m in a general north-south direction (parallel to strike) by about 150 m in an east-west direction. It is not known when operations of Compañía Minera Korani ceased, but presumably they ceased in the late 1960s or early 1970s. This mining operation left behind many mine portals, waste piles, and mine tailings that continually produce ARD. Smaller portals are located approximately 2 km to the south of the historical mill and discharge ARD into the Collpa Mayo drainage.

The next exploration activity was by a private Peruvian company, Minsur. That exploration was reported to include 40 shallow drill holes in various locations, including several close proximity holes in the gold zone (located south of the current resource area). Although Minsur is an active mining company in Peru, attempts by BCM to secure copies of Minsur's exploration data have been unsuccessful. None of Minsur's exploration information is available or verifiable, although, reportedly, gold mineralization was encountered in some of Minsur's drilling.

In late 2003 and early 2004, Rio Tinto Mining and Exploration began a surface exploration program for porphyry copper mineralization. During 2004, Rio Tinto conducted surface mapping, sampling, and ground magnetic surveys and developed access roads into the area. That initial work by Rio Tinto defined anomalous silver and lead mineralization to the south of the Korani mines and also defined a zone of anomalous gold mineralization in rock and soils.

The concession ownership by Compañía Minera Korani apparently lapsed during the 1970s. The ownership of Minsur also lapsed prior to Rio Tinto's exploration activities after 2000. Rio Tinto re-established some of the older concessions in their name beginning in 2003. BCM has added two concessions early in 2005 and one in 2019 to create the current land position described in Section 4.

6.2 Historical Exploration and Estimates

There have been seven previous mineral resource estimates for the Project and three estimates of mineral reserves, all of which have been published in previous technical reports. The previous technical reports are summarized below:

- March 31, 2006, National Instrument 43-101 Technical Report, Initial Resource Estimate for Corani Silver-Gold Exploration Project. SRK Consulting. Tucson, Arizona, United States.
- October 4, 2006, Corani Project Mineral Resource Technical Report, Independent Mining Consultants, Inc, Tucson, Arizona, United States.
- May 12, 2008, Technical Report, Corani Resource Estimate and PEA, Independent Mining Consultants, Inc. Tucson, Arizona, United States.
- October 14, 2009, NI 43-101 Technical Report, Prefeasibility Study Corani Project Puno Perú, Vector Perú S.A.C.
- December 22, 2011, NI 43-101 Technical Report, Feasibility Study. Corani Project. M3 Engineering and Technology Corporation, Tucson, Arizona, United States.
- May 30, 2015, NI 43-101 Technical Report, Optimized and Final Feasibility Study. Corani Project. M3 Engineering and Technology Corporation, Tucson, Arizona, United States.
- Sept 17, 2017, NI 43-101 Technical Report, Corani Project Detailed Engineering Phase 1 (FEED), Sedgman Chile SpA.

The mineral resource and reserve estimates from each report are summarized in Table 6-1 through Table 6-6.

Table 6-1: Mineral resources - March 2006 (SRK Consulting, 2006)

Category	Tonnes (000)	Silver g/t	Lead %	Zinc %
Measured	7,759	65.1	1.08	0.16
Indicated	20,123	43.6	0.68	0.25
Measured + Indicated	27,882	49.6	0.79	0.23
Inferred	87,627	72.9	1.03	0.58

Table 6-2: Mineral resource - October 2006 (Independent Mining Consultants, Inc., 2006)

Category	Tonnes (000)	Silver g/t	Lead %	Zinc %	Silver Moz	Lead Mlb	Zinc Mlb
Main							
Measured	7,899	52.5	0.93	0.29	13.3	162.0	50.5
Indicated	44,196	40.7	0.70	0.39	57.8	682.0	380.0
Measured + Indicated	52,095	42.5	0.73	0.37	71.1	844.0	430.5
Inferred	11,898	49.7	0.64	0.26	19.0	167.9	68.2
Minas							
Measured	2,487	77.1	1.41	0.53	6.2	77.3	29.1
Indicated	39,405	52.2	1.03	0.40	66.1	894.8	347.5
Measured + Indicated	41,892	53.7	1.05	0.41	72.3	972.1	376.6
Inferred	20,713	47.3	0.74	0.30	31.5	337.9	137.0
Este							
Measured	14,558	82.7	1.07	0.76	38.7	343.4	243.9
Indicated	31,856	72.6	0.91	0.75	74.4	639.1	526.7
Measured + Indicated	46,414	75.8	0.96	0.75	113.2	982.5	770.6
Inferred	5,326	55.9	0.41	0.25	9.6	48.1	29.4
All Deposits							
Measured	24,944	72.6	1.06	0.59	58.2	582.7	323.5
Indicated	115,457	53.4	0.87	0.49	198.3	2,215.9	1,254.2
Measured + Indicated	140,401	56.9	0.90	0.51	256.5	2,798.6	1,577.7
Inferred	37,937	49.3	0.66	0.28	60.1	553.9	234.6

Source: (Published by Independent Mining Consultants, Inc. Based on 16 grams per tonne (g/t) silver cutoff grade contained within an approximate open pit)

Table 6-3: Historic mineral resource - May 2008 (Independent Mining Consultants, Inc. , 2008)

Category	Tonnes (000)	Silver g/t	Lead %	Zinc %	Silver Moz	Lead Mlb	Zinc Mlb
Main							
Measured	10,025	42.3	0.80	0.37	13.6	176.8	81.8
Indicated	64,250	30.0	0.57	0.43	62.0	807.4	609.1
Measured + Indicated	74,275	31.7	0.60	0.42	75.6	984.2	690.9
Inferred	11,928	33.1	0.57	0.36	12.7	149.9	94.7
Minas							
Measured	6,168	53.4	1.05	0.44	10.6	142.8	59.8
Indicated	106,970	38.2	0.75	0.38	131.4	1,768.7	896.1
Measured + Indicated	113,138	39.0	0.77	0.38	142.0	1,911.5	955.9
Inferred	19,698	32.5	0.54	0.39	20.6	234.5	169.4

Category	Tonnes (000)	Silver g/t	Lead %	Zinc %	Silver Moz	Lead Mlb	Zinc Mlb
Este							
Measured	20,523	63.3	0.91	0.69	41.8	411.7	312.2
Indicated	40,485	52.0	0.75	0.57	67.7	669.4	508.7
Measured + Indicated	61,008	55.8	0.80	0.61	109.5	1,081.1	820.9
Inferred	1,526	30.4	0.41	0.21	1.5	13.8	7.1
All Deposits							
Measured	36,716	55.9	0.90	0.56	66.0	731.3	453.8
Indicated	211,705	38.4	0.70	0.43	261.1	3,245.5	2,013.9
Measured + Indicated	248,421	40.9	0.73	0.45	327.1	3,976.8	2,467.7
Inferred	33,152	32.6	0.54	0.37	34.8	398.2	271.2

Source: (Published by Independent Mining Consultants, Inc. Based on $9.35/t NSR Cutoff Grade Contained Within an Approximate Open Pit)
oz = troy ounces
Lbs = pounds

Table 6-4: Mineral reserve and resource - August 2009 (Vector Perú S.A.C., 2009)

Mineral Reserves, $9.10 NSR Cutoff					Contained Metal			Equivalent oz	
Category	Tonnes (000)	Silver g/t	Lead %	Zinc %	Silver Moz	Lead Mlb	Zinc Mlb	Silver Moz	Silver g/t
Proven	27,957	70.2	1.08	0.59	63.1	665.7	363.6	115.0	127.9
Probable	111,666	54.3	0.90	0.43	194.9	2,215.6	1,058.6	360.3	100.4
Proven + Probable	139,623	57.5	0.94	0.46	258.0	2,881.3	1,422.2	475.3	105.9

Mineral Resources in Addition to Reserves $7.85 NSR Cutoff					Contained Metal			Equivalent oz	
Category	Tonnes (000)	Silver g/t	Lead %	Zinc %	Silver Moz	Lead Mlb	Zinc Mlb	Silver Moz	Silver g/t
Measured	10,791	16.7	0.43	0.45	5.8	102.3	107.1	16.2	46.8
Indicated	99,626	20.6	0.45	0.39	66.0	988.4	856.6	158.2	49.4
Measured + Indicated	110,417	20.2	0.45	0.40	71.8	1,090.7	963.7	174.4	49.1
Inferred	34,215	32.4	0.54	0.34	35.6	407.3	256.5	69.0	62.7

Note: for this reserve resource statement silver equivalency calculation represents the contained equivalent silver ounces sent to concentrate and is based on the resource metal prices assumptions of $13.00/oz silver, 0.70/lb lead and 0.65/lb zinc and recoveries to concentrate of 74.5% for silver and 71.7% for lead and 71.3% for zinc. The calculation does not take into account the net smelter return.

Table 6-5: Mineral reserve and resource - October 2011 (M3, 2011)

Mineral Reserves, $10.54/tonne NSR Cutoff					Contained Metal		
Category	Tonnes (000)	Silver g/t	Lead %	Zinc %	Silver Moz	Lead Mlb	Zinc Mlb
Proven	30,083	66.60	1.041	0.603	64.4	690.4	399.9
Probable	126,047	50.73	0.872	0.467	205.6	2,423	1,298
Proven + Probable	156,130	53.79	0.904	0.493	270.0	3,113	1,698

Mineral Resources in Addition to Reserves, $9.20/tonne NSR Cutoff					Contained Metal		
Category	Tonnes (000)	Silver g/t	Lead %	Zinc %	Silver Moz	Lead Mlb	Zinc Mlb
Measured	10,878	17.50	0.380	0.330	6.1	91.1	79.1
Indicated	123,583	20.80	0.380	0.290	82.6	1,035	790.1
Measured + Indicated	134,461	20.5	0.380	0.290	88.7	1,126	869.2
Inferred	49,793	30.00	0.464	0.278	48.0	509.4	305.2

Metal Prices: for Mineral Reserve - $18.00/oz silver, $0.85/lb lead, $0.85/lb zinc
for Mineral Resource - $30.00/oz silver, $1.00/lb lead, $1.00/lb zinc

Table 6-6: Mineral reserve and resource – May 2015 (M3, 2015)

Mineral Reserves, Variable $23.00-11.00 NSR Cutoff							
Category	Tonnes (000)	Silver g/t	Lead %	Zinc %	Silver Moz	Lead Mlb	Zinc Mlb
Proven	19,855	69.1	1.09	0.72	44.1	478.7	313.4
Probable	117,843	48.6	0.88	0.57	184.3	2289.2	1471
Proven & Probable	137,698	51.6	0.91	0.59	228	2,768	1,784

Mineral Resources in Addition to Reserves, $11.00 NSR cutoff, 15 g/tonne Silver Cutoff (oxide)							
Category	Tonnes (000)	Silver g/t	Lead %	Zinc %	Silver Moz	Lead Mlb	Zinc Mlb
Measured	14,360	32.01	0.34	0.19	14.8	108.4	61.6
Indicated	83,749	25.37	0.37	0.28	68.3	682.2	512.8
Measured + Indicated	98,109	26.34	0.37	0.27	83.1	790.6	574.4
Inferred	39,953	37.20	0.58	0.40	47.8	510.6	352.4

Notes: The Mineral Reserve is within the 20 $/oz designed pit and utilizes variable NSR cutoff values to maximize early cash flows. This is the tonnage processed in the economic model. The Mineral Resource is the tonnage contained within the 30$/oz silver, 1.425 $/lb lead, and 1.50 $/lb zinc prices Whittle pit using a 20 $/oz silver, 0.95 $/lb lead, and 1.00 $/lb zinc prices at a cutoff of 11 $/tonne NSR plus potentially leachable oxide at a 15g/t silver cutoff ($4.80/tonne using 50% recovery in addition to ore already categorized within the Mineral Reserve.

Table 6-7: Mineral reserve and resource – Sept 2017

Mineral Reserves, $11.00 NSR cutoff							
Category	**Tonnes (Mt)**	**Silver g/t**	**Lead %**	**Zinc %**	**Silver Moz**	**Lead Mlb**	**Zinc Mlb**
Proven	20.8	65.8	1.03	0.71	44	472	323
Probable	118.3	47.5	0.87	0.57	181	2,274	1,486
Proven & Probable	139.1	50.3	0.90	0.59	225	2,746	1,809

Notes:
1) The Mineral Reserves have been estimated using the definitions of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM).
2) The Mineral Reserves have been estimated using the following metal prices: $20.00/oz Ag, $1.00/lb Zn, $0.95/lb Pb using a revenue factor 1.00 pit shell as a basis for the pit design.
3) Only pre-mineral tuff type of material has been considered as reserves.
4) NSR Cut-off grades used are equal or higher than: $11.11/t for the East Pit, and $11.26/t for Minas and Main pits.
5) The effective date for these Mineral Reserves is 1 May 2017.
6) Totals / Averages may not add up due to rounding of individual tonnes and grades.
7) The tonnes and grades shown above are considered a Mineral Reserve because they have been demonstrated to be economically viable through the FEED study financial model using the following metal prices: $18.00/oz Ag, $1.10/lb Zn, $0.95/lb Pb

Mineral Resources in Addition to Reserves, $11.00 NSR cutoff							
Category	**Tonnes (000)**	**Silver g/t**	**Lead %**	**Zinc %**	**Silver Moz**	**Lead Mlb**	**Zinc Mlb**
Measured	29,209	56.2	0.912	0.582	52.8	587	375
Indicated	181,902	40.7	0.741	0.495	238	2971.3	1983.5
Measured + Indicated	211,111	42.8	0.765	0.507	291	3,558	2,359
Inferred	31,231	40.6	0.742	0.512	40.8	510.6	352.4

Note: Cut-off Value : $11.00/tonne covers process and general and administrative costs. The Mineral Resources were generated within the $30.00/troy ounce silver, $1.425/lb lead, and $1.50/lb zinc price Whittle pit shell and the calculated $11/tonne NSR cut-off.

7 Geology Setting and Mineralization

7.1 Regional Geology

The Corani Project area is located in the northern part of Puno Department, southern Peru, within the Cordillera Oriental of the Central Andes (Figure 7-1). The Cordillera Oriental in this region is represented by northwest-trending, glaciated peaks ranging in elevation from 5,800 to 6,400 masl. Including its western foothills, the Cordillera Oriental forms an approximately 125 km wide mountainous zone eroded into variably deformed Lower Paleozoic to Mesozoic metasediments (Laubacher, 1978) intruded by granitoid plutons of Triassic to Early Jurassic age (Kontak, 1991; Kontak, 1990).

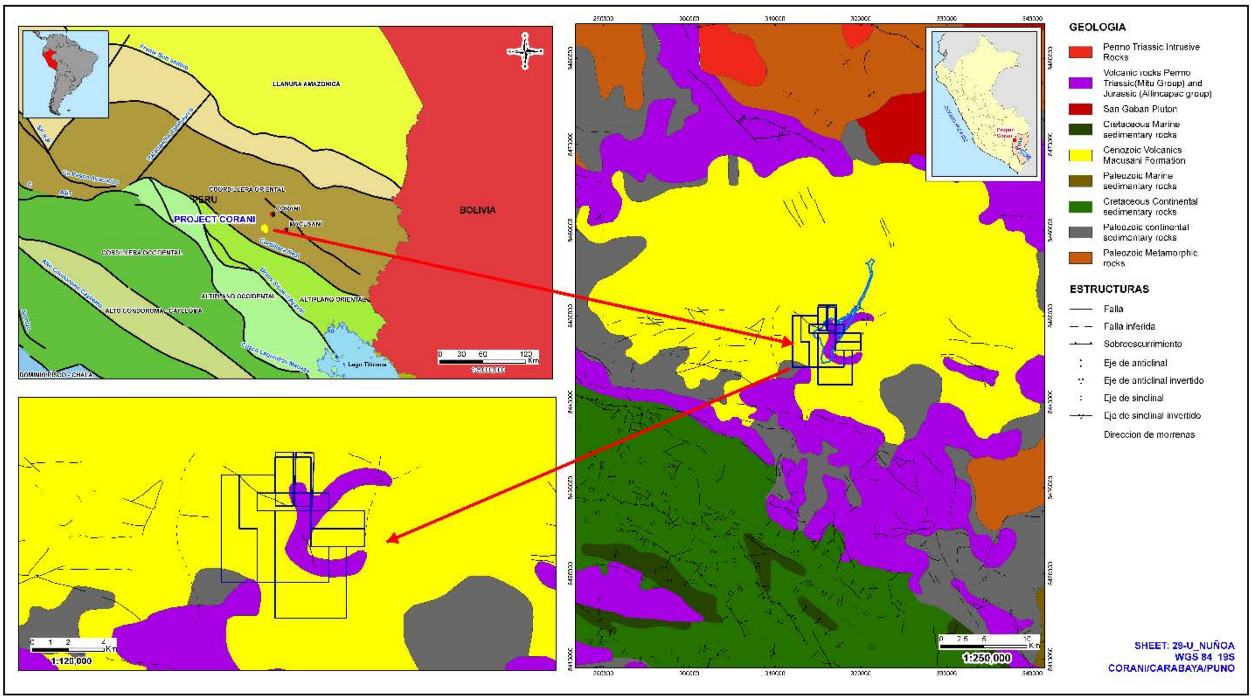


Figure 7-1: Regional geologic map, Corani Project vicinity (BCM, 2015)

Throughout the region, the Lower Paleozoic is represented by the Ananea Group, a thick (up to 10,000 m) sequence of predominantly turbiditic sediments. These rocks were subsequently deformed during the early phase of the Hercynian Orogeny in Early Carboniferous time (340 Ma). The Upper Paleozoic is characterized by the accumulation of the Ambo Group, a thinner but lithologically more variable sequence of sedimentary rocks. The Ambo Group is comprised of sandstone, conglomerate, and minor carbonaceous layers of mostly continental derivation and was deposited unconformably upon the Lower Paleozoic strata in a post-Hercynian basin.

During the Lower Carboniferous, carbonates, shales, and sandstones of the Tarma Group were deposited in isolated basins; during the Upper Carboniferous, Copacabana Group carbonates were deposited over an extensive epicontinental area. The Mitu Group, comprised of continental redbed sandstone and conglomerate with volcanic intercalations, was deposited during the Permo-Triassic period, and the interval from the Triassic to the Upper Cretaceous is characterized by basin carbonate deposition and volcanism.

Magmatism occurred in several widely separated episodes during the Mesozoic and Cenozoic (Clark, 1990). Intrusive activity was most active during Late Cretaceous and Early Tertiary, and volcanism began to dominate after Middle Tertiary. Tertiary volcanic activity is largely represented by the Oligocene to lowermost Miocene Picotani Group and the Lower to Upper Miocene Quenamari Group. The Picotani Group is comprised of rhyodacites intercalated with K-rich basalts and ultrapotassic, lamproitic, and lamprophyric flows, whereas the Quenamari Group is dominated by rhyolites and two-mica syenogranites (Li, 2016).

7.2 Local and Property Geology

The following description of the geology, structure, alteration, and mineralization specific to the Corani Project is largely modified from, and in some cases is excerpted directly from Swarthout (2010). GRE has reviewed this information and available associated supporting documentation in detail and finds the discussion and interpretations presented herein to be reasonable and suitable for use in this report.

7.2.1 Lithology

The oldest strata encountered in the vicinity of the Project belong to the Ambo and Tarma Groups, which consist primarily of shales with minor quartzites, sandstones, and local carbonate lenses. Within the Project area, the sedimentary units are dominantly red to grey shales, which commonly contain syngenetic or diagenetic pyrite. These rocks are moderately folded and faulted, striking northwest with 10° to 50° dips east and west along the flanks of a modest anticlinal fold. Though outcrops of these rocks within the Project area are relatively few, resistant quartzite units often form ridges throughout the greater regional area, whereas weathered shales generally form slopes. The local lithology is illustrated in Figure 7-2.

The overlying Chacaconiza Member of the Quenamari Formation is the primary host of mineralization at the Corani Project. The Chacacozina is the youngest member, ca. 23.94 ± 0.15 Ma (Ullrich, 2006), of the Quenamari and is comprised of a sequence of crystal-lithic and crystal-vitric-lithic tuffs. Within the Project area, these rocks strike northwest and dip sub-horizontally to 40° northeast. Bedding ranges from well-bedded to massive, locally showing poorly developed columnar jointing. The tuffs are widely hydrothermally altered and pervasively argillized to low-temperature clays and are variably faulted, fractured, and brecciated. Individual units of the tuff sequence are pumiceous, and in general, the more lithic-rich units occur near the base, while the finest grained crystal tuffs are located within the upper part of the overall sequence. The upper portion of the tuff sequence includes a generally well-bedded andesite flow (or flows) of variable thickness. The andesite hosts mineralization in similar character to the surrounding tuffs, with no apparent constraints attributable to the change in lithology.

The Chacaconiza is unconformably overlain by the upper members (undifferentiated) of the Quenamari Formation, the Sapanuta and Yapamayo Members, ca. 10.375 ± 0.080 Ma (Ullrich, 2016). These rocks are nearly identical to the Chacaconiza in lithology, composed primarily of crystal lithic tuffs, but are notably different in both alteration and mineralization. They are generally flat lying to shallowly dipping and are separated from the underlying tuff sequence by an angular unconformity with substantial topographic relief. The upper (Sapanuta/Yapamayo) tuff sequence is commonly collectively referred to as the “post-mineral” tuff, while the tuffs of the Chacaconiza are referred to as “pre-mineral.” The post-mineral tuff occupies much of the volcanic section in the northern portion of the Project area and is largely unaltered and considered effectively barren.

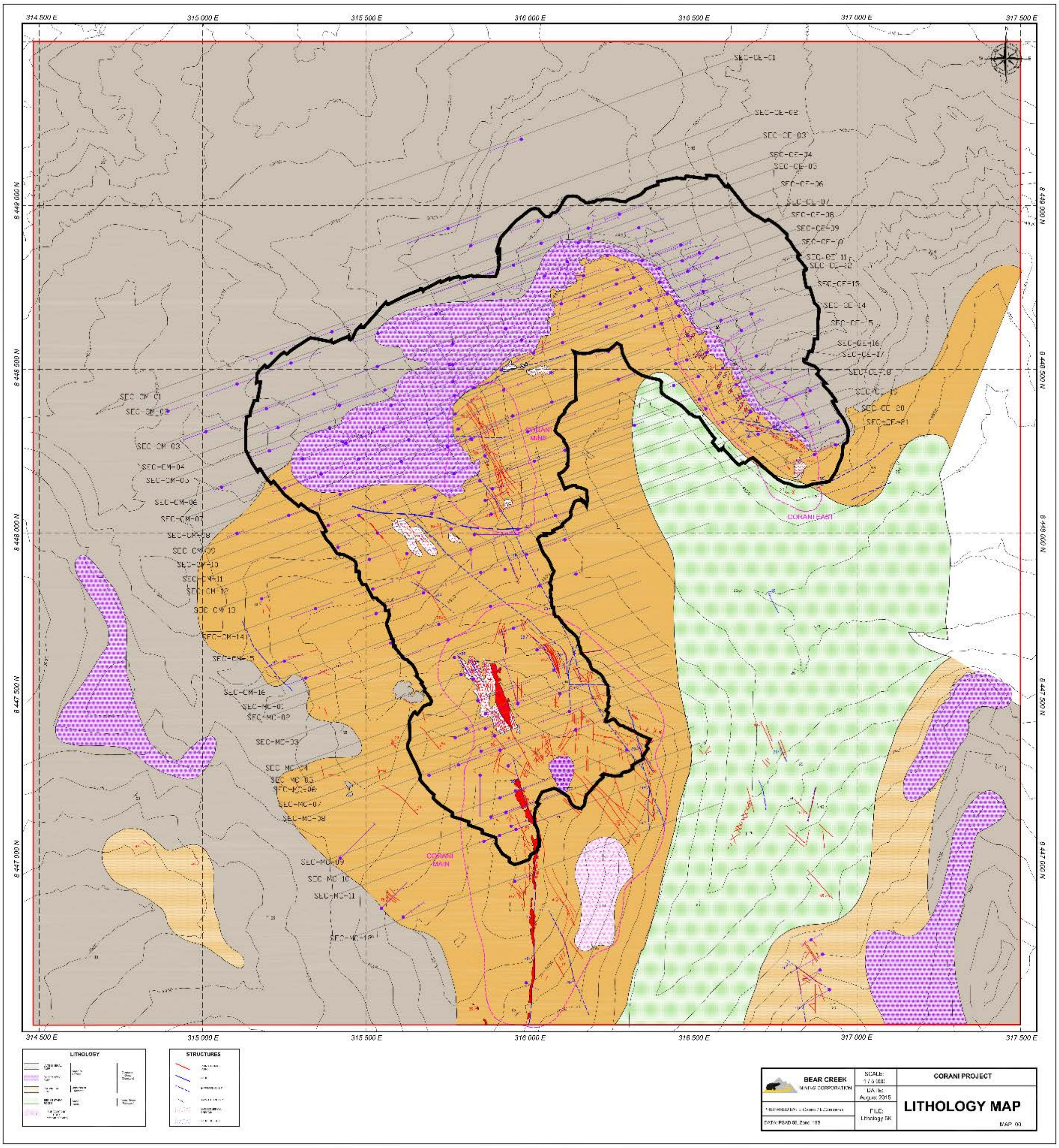


Figure 7-2: Geologic map of the Corani Project area (BCM, 2015)

7.2.2 Structure

The Corani Project area has been affected by brittle deformation at both local and regional scales. The geometry and kinematics of known structures suggest the occurrence of two tectonic events: a) an extensive tectonic event contemporaneous with mineralization within and to the south of Corani Main, locally expressed by listric faults that have generated a light tilting and rotation of the blocks; and b) a second extensional tectonic event resulting in post-mineralization structures such as those identified within and to the south of Corani Este (Prado, 2008)

The Corani deposit is hosted within a stacked sequence of listric normal faults striking dominantly north to north-northwest with moderate to shallow (50° to <10°) westerly dips. The hanging walls of the listric faults are extensively fractured and brecciated, providing the structural preparation for subsequent or syngenetic mineralization. The stacked listric faults are more prominent in the Corani Minas and Corani Main areas. The Corani Este area contains a single known listric fault with an extensively fractured and brecciated hanging wall. The contact with the underlying Paleozoic sediments corresponds locally to listric faults dipping shallowly to the west.

Desrochers (Desrochers, 2005) described the structural geology of Corani as being dominated by a series of north- and north-northwest−trending faults and veins with silver, gold, and antimony mineralization. Field observations indicate that the mineralized veins formed during normal movement along the faults, although the extension was accompanied by a minor amount of sinistral strike slip. The faulting moved the western, or hanging wall, block down and away from the main structure with a southwest-trending transport direction. Maximum extension took place at the intersection of the north- and north-northwest−striking structures and in northwest-trending bends along the main north-south structure. These bends also coincide with steeper parts of the main north-south structure.

7.2.3 Alteration

Generally, illite-kaolinite alteration of the pre-mineral tuffs is present over a 5 by 2 km area, approximately occupying the entire window exposed beneath the post-mineral tuffs. Most common alteration phases associated with the mineralization are illite, kaolinite, and smectite/chlorite/celadonite. Distribution of dominant alteration types is shown in plan view on Figure 7-3.

Gangue minerals include massive to banded quartz, barite, chalcedony, and iron and manganese oxides. The mineralization and alteration assemblages are well documented in thin and polished sections (Espinoza, 2006) and by quantitative evaluation of minerals by scanning electron microscopy (QEMSCAN) (Gunning, 2007). Each of the mineralized areas exhibits differences in alteration and gangue phases, which are described in further detail in Section 13 of this report.

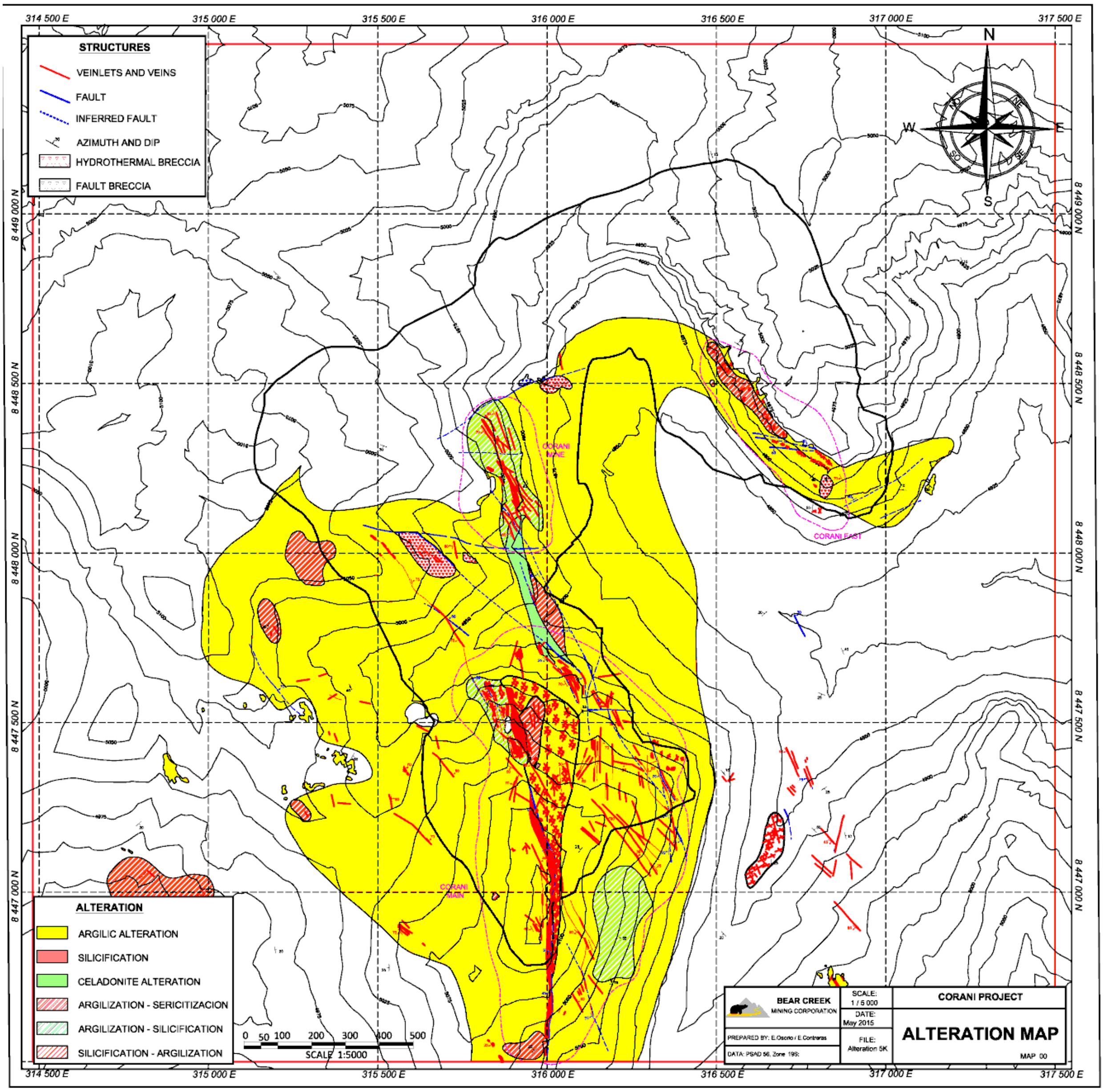


Figure 7-3: Alteration map (BCM, 2015)

7.2.4 Mineralization

Mineralization at the Corani Project occurs in three distinct and separate zones: Corani Main, Corani Minas, and Corani Este, each differing slightly in character with regard to both alteration and mineral assemblages. In general, mineralization in outcrop throughout the Corani Project is associated with iron and manganese oxides, barite, and silica. Silicification is both pervasive and structurally controlled along veins. In drill core, the mineralization occurs in typical low to intermediate sulfidation Ag-Pb-Zn mineral assemblages.

The most abundant silver-bearing mineral is fine-grained argentian tetrahedrite or freibergite (Hazen Research Inc., 2006; Gunning, 2007). Other minor silver minerals present include acanthite and the lead-silver sulfosalts, adorite and diaphorite. Other sulfide minerals include pyrite-marcasite, boulangerite, sphalerite, and galena. Boulangerite and galena do not appear to be significant hosts for the silver. Sphalerite, mainly high Fe type, overlaps the silver mineralization but can be more areally extensive, particularly at Corani Minas, where sphalerite may extend 10 to 100 m beneath and lateral to the silver-bearing minerals. Lead also occurs as plumbogummite, a lead-aluminum phosphate. Lead mineral speciation is dependent on pH, and the plumbogummite is believed to be secondary in origin, forming as a result of the remobilization of lead in the presence of phosphate in a very acidic environment with abundant aluminum.

BCM geologists have defined nine specific mineral domains within the Corani deposit, based in part on metallurgical properties, as described below:

- CSC – Coarse-grained silica-sulfide-celadonite characterized by readily discernible sulfides (galena-sphalerite-chalcopyrite ± tetrahedrite) with celadonite in crystalline to locally opaline quartz with good Ag-Pb-Zn recoveries.
- CS - A subset of CSC that contains coarse galena-sphalerite-chalcopyrite ± tetrahedrite without green celadonite clay.
- TET – silver-bearing tetrahedrite characterized by recognizable late-stage, coarse-grained tetrahedrite cutting earlier sulfides and displaying the highest silver contents and best silver recoveries by flotation or leach: typically with low Pb-Zn content.
- PM – Pyrite-marcasite ± quartz typical of low temperature early-stage mineralization with little polymetallic mineralization, mainly Zn.
- FBS – Fine-grained black silica-sulfides characterized by very fine-grained mineralogy deposited from quenched ore fluids with highly variable metal content and generally poor leach recoveries and good flotation recovery with some challenges in separation.
- QSB - Crystalline quartz-sulfide-barite interpreted as early fault fill or late-stage breccia fill.
- PG – Plumbogummite, identified as a pale-green, waxy, Pb-phosphate mineral that in metallurgical test results shows diminished lead flotation and difficulties in separation of base metals.
- FeO – Iron-oxide mineralization with locally elevated silver and generally low Pb-Zn. This is a gradational zone with mixtures of FeO and FBS, and the most strongly oxidized areas show high silver leach recovery results.
- MnO – Manganese-oxide mineralization hosting mainly silver with lesser Pb-Zn with very poor response to flotation and leach tests.

Mineralization is largely structurally controlled in each of the three areas along a general north-northwest strike. Figure 7-4 illustrates the distribution of silver mineralization based on thickness at an approximate cutoff grade of 15 g/t silver. Strike length of silver mineralization is roughly 2 km for Corani Main and Corani Minas combined and 1.5 km for Corani Este.

The cross sections presented in Figure 7-5 illustrate the general thickness and dip of the mineralized zone(s).

All mineralization, with the exception of pyrite, dies out at depth, typically ending as much as 50 m above the contact with underlying Paleozoic basement. In drill core, the contact with the Paleozoic sedimentary rocks is locally sheared with gouge and slickensides. The sedimentary rocks are locally brecciated, but the breccias do not contain a hydrothermal matrix. The strata locally contain pyrite veins, with bleached (Fe-reducing) halos, and Fe carbonate veinlets (Nelson, 2006). Veining within the Paleozoic sedimentary rocks is completely barren of economic mineralization.

Corani Main

The primary mineralized vein breccia in the Corani Main area can be traced for 800 m and undulates with strikes and dips varying between S50°E, 55°W and S20°W, 40°W, with steepening dips to the north. Vein breccias locally attain widths of greater than 10 m. To the north, strike and dip change, which indicates a plunging, dilatant structure that attains widths of 80 m, with near-vertical quartz veins surrounded by stockwork systems in adjacent wall rock. The change in strike suggests that an overall sinistral (left-lateral) strike-slip component affects the veining, causing the mineralization to blow-out, or widen and intensify, to the north (Nelson, 2006). Breccias and stockworks are characterized by chalcedonic, cockscomb, crystalline, hyaline, and amethystine quartz, with hematite-jarosite-goethite and barite stringers resulting in a highly banded texture. Manganese oxides are generally sparse but locally abundant. Pyrite, dark sphalerite, and freibergite are present (Petersen, 2005; Espinoza, 2006). A post-mineral transverse fault, striking N70°E and showing dextral movement, separates the Corani Main and Corani Minas mineralized areas. The fault deforms the veins on either side; however, post-mineral displacement appears minimal, despite the fault's appearance as a major lineament. Figure 7-6 shows a representative cross-section of Corani Main.

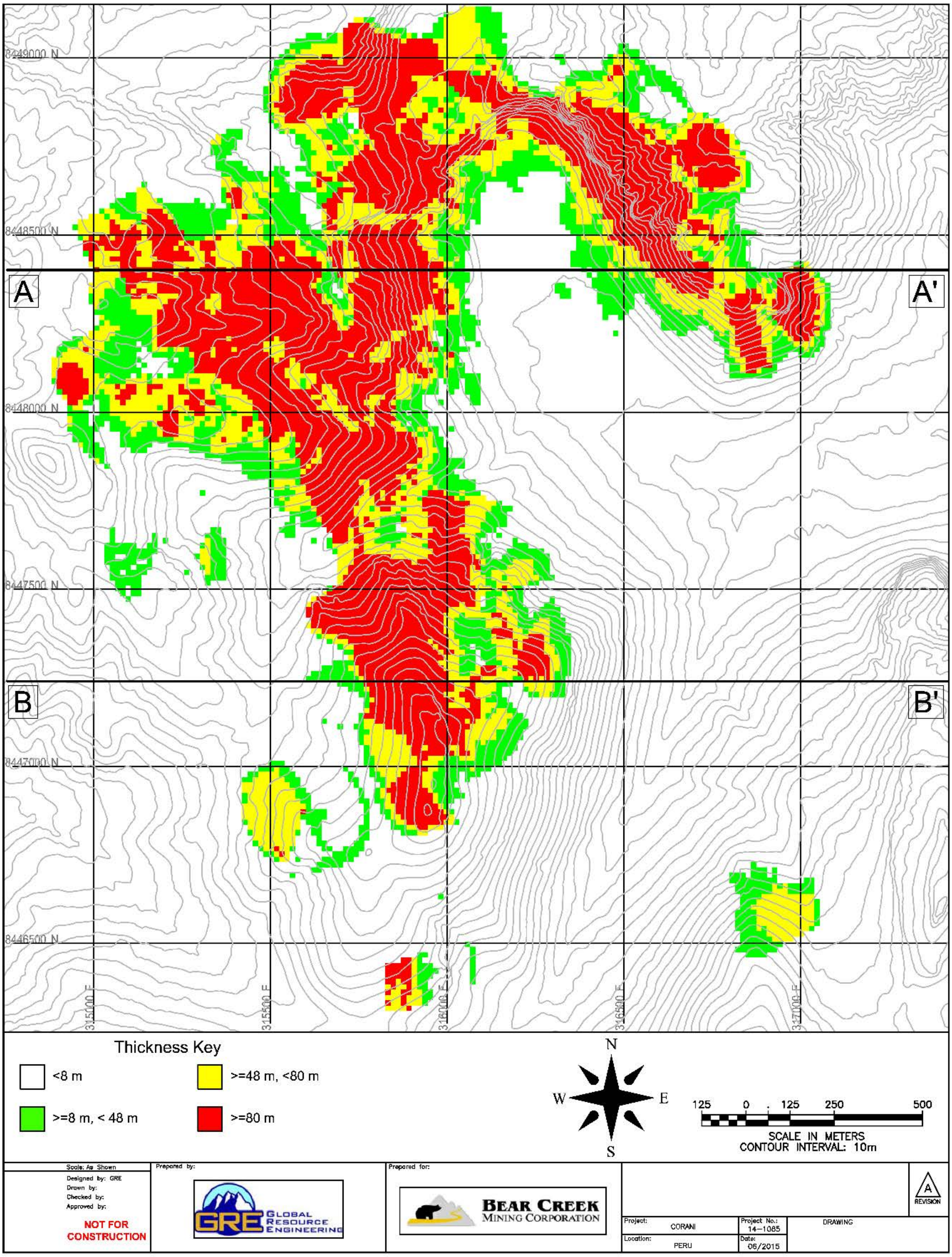


Figure 7-4 Silver mineralization thickness at 15 g/t cutoff grade (BCM, 2015)

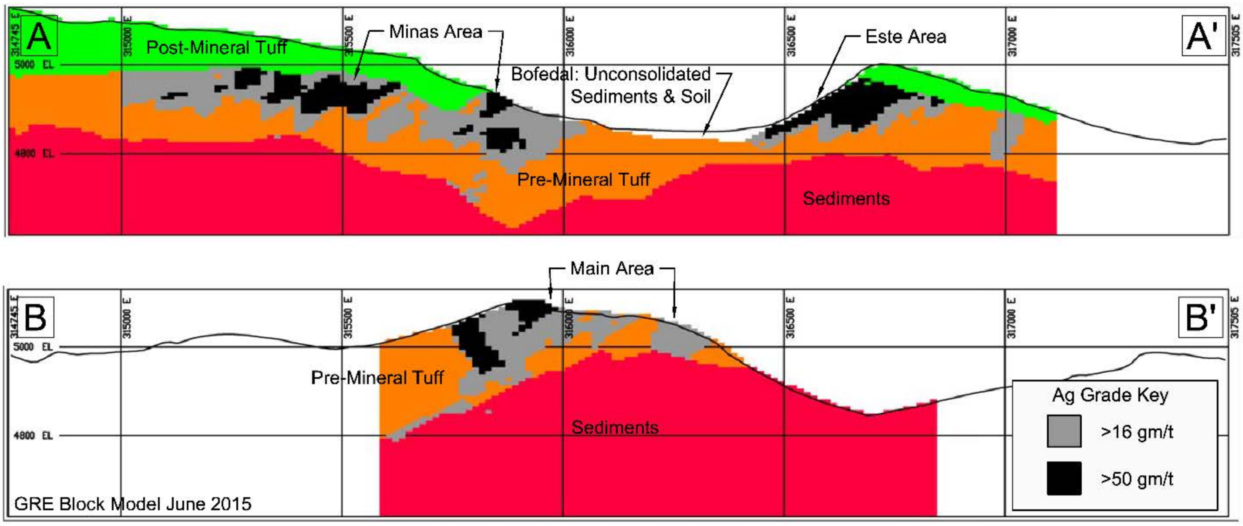


Figure 7-5 Silver mineralization distribution (BCM, 2015)

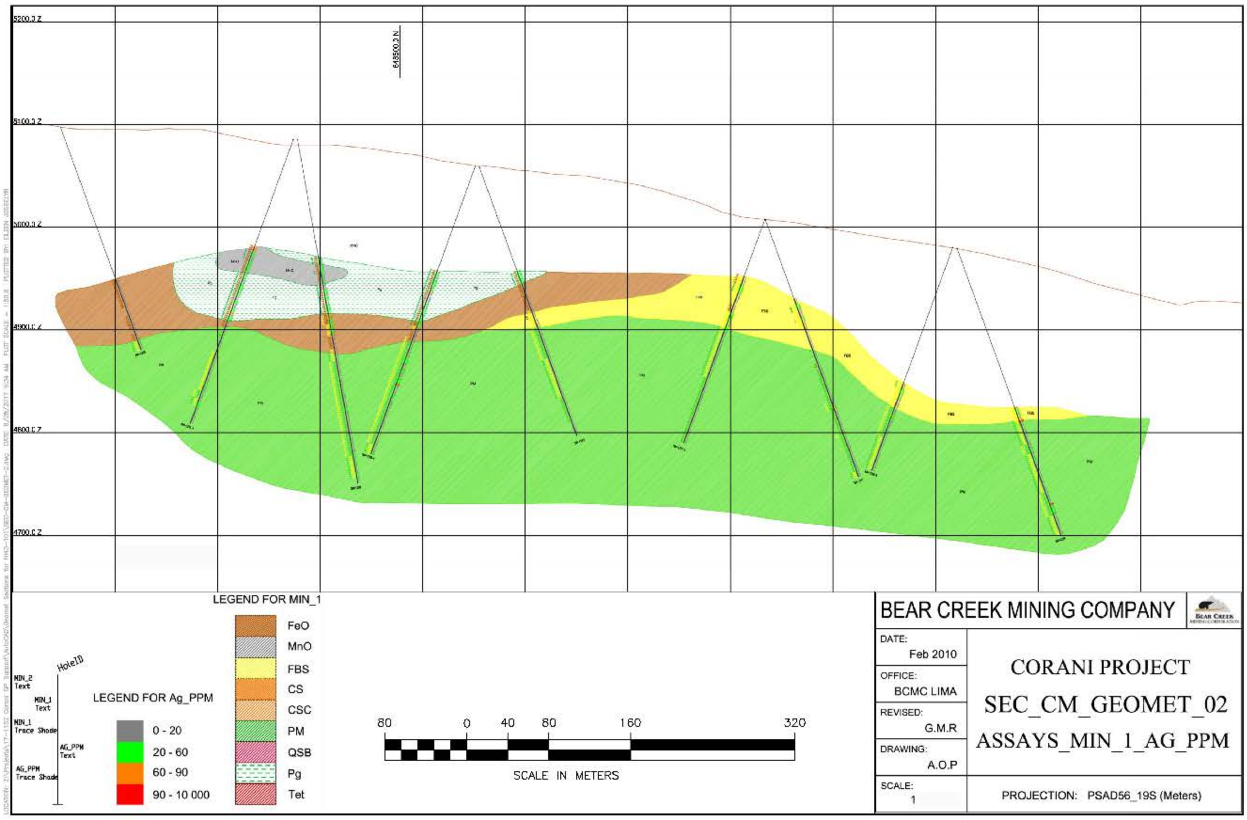


Figure 7-6: Corani main representative cross-section (BCM, 2015)

Corani Minas

Corani Minas is structurally complex and characterized by a large area of small, crested ridges formed by breccias, silicification, and quartz vein ribs. Veins, 0.1 to 2.0 m wide, are composed of banded, chalcedonic, and hyaline quartz, barite, hematite, jarosite, goethite, pyrite, and proustite-pyrargerite. The veins generally strike north 20°−60° west with 50° to 80° dips to both the west and east (Desrochers, 2005; Nelson, 2006; Prado, 2008), although almost any dip angle can be observed (Nelson, 2006). Hanging-wall breccias are composed mainly of subrounded to angular clasts with void-filling barite crystals in a siliceous matrix, whereas stockwork veining is mainly within the footwall. According to Nelson, orientations and textures, such as the absence of slickensides, indicate that veining was extensional and that block rotations tended to occur surrounding an east-west axis. Figure 7-7 shows a representative cross-section of Corani Minas.

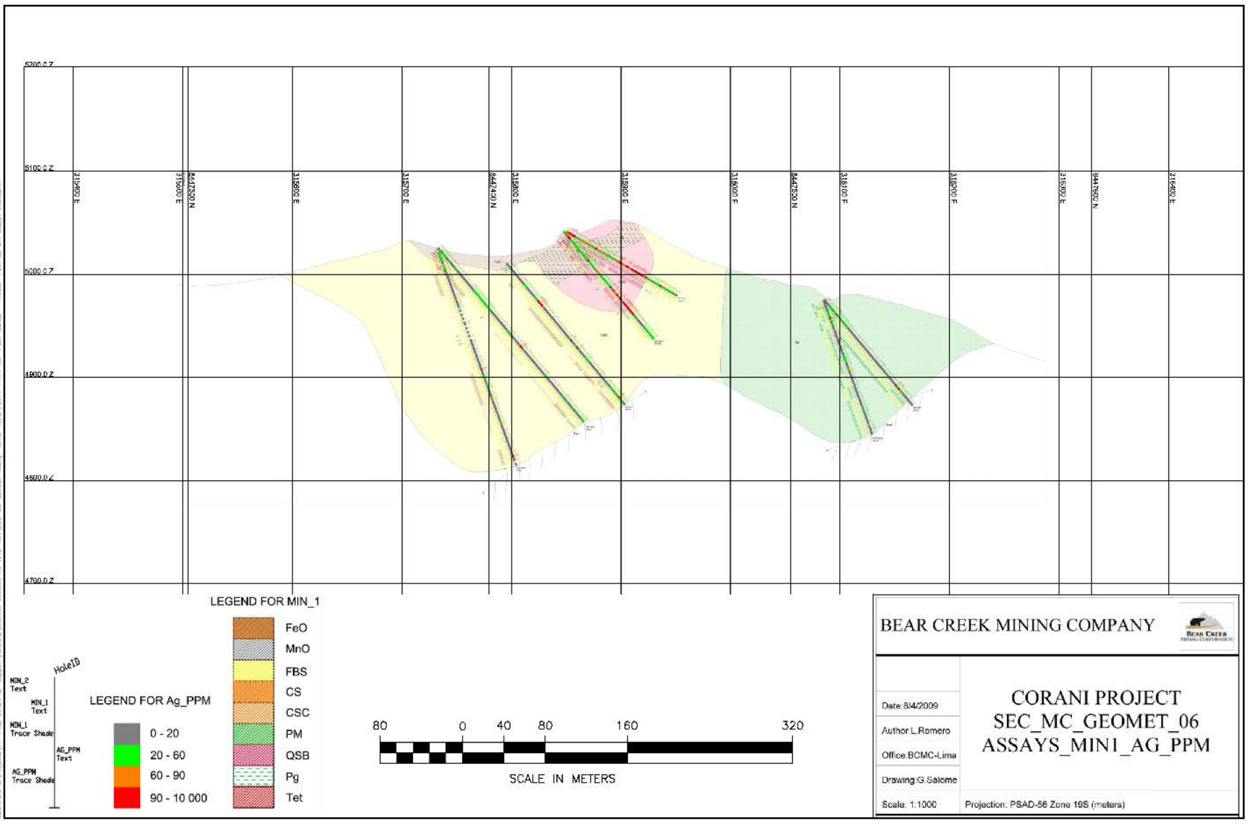


Figure 7-7: Corani Minas representative cross-section (BCM, 2015)

Corani Este

Corani Este is distinct in that mineralization is controlled by a single listric fault that does not crop out due to post-mineral tuff cover. Silicified breccias and stockwork veining are formed in the hanging wall of the main vein and crop out as silica-rich ribs along a north-south strike. Dips are difficult to determine in outcrop and drill core due to the structural complexity and hydrothermal alteration; however, conjugate vein sets, occasionally north-south striking and dipping both east and west, are observed. Importantly, a small breccia pipe occurs in Corani Este containing high-grade silver values (≥300 g/t silver). Mineralization within the pipe occurs in hydrothermal breccias with a dark-gray, sulfide matrix and dark-purple, jasperoid vein breccias. Figure 7-8 shows a representative cross-section of Corani Este.

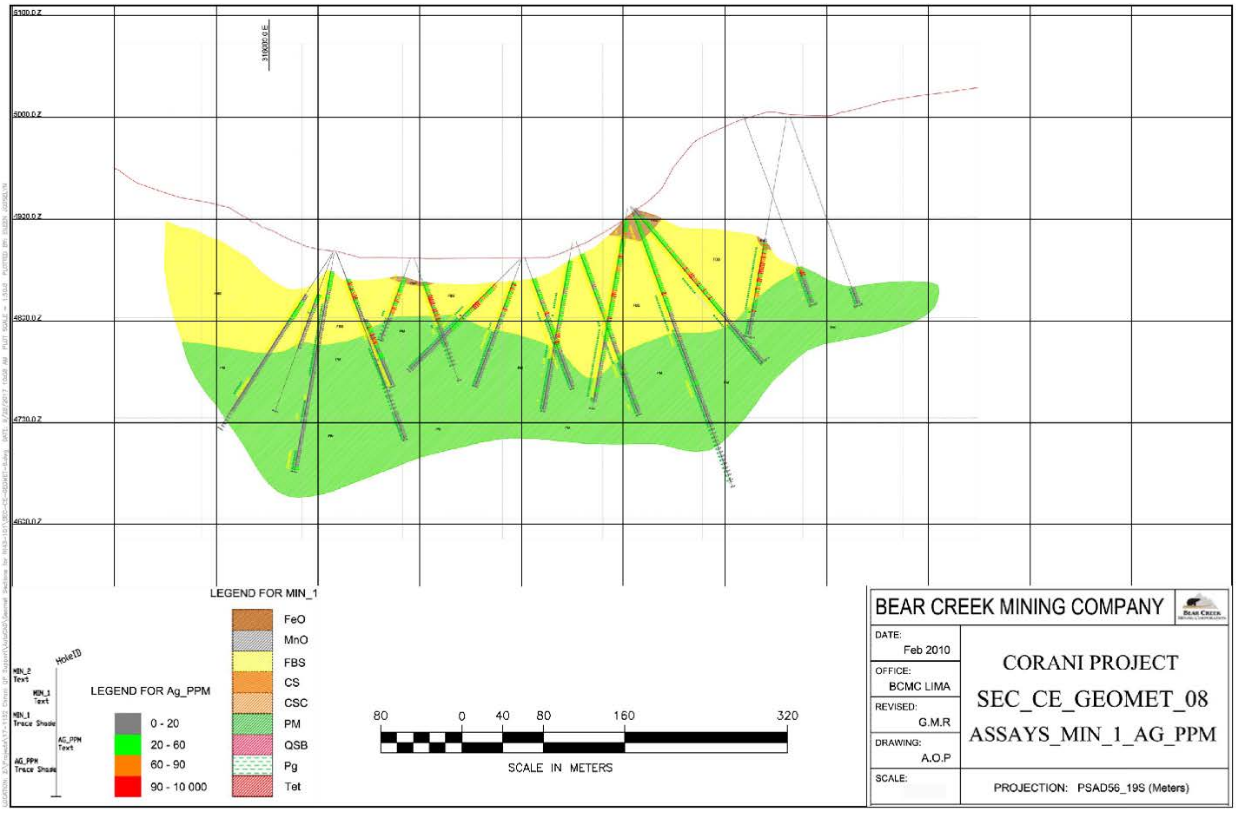


Figure 7-8: Corani Este representative cross-section (BCM, 2015)

8 Deposit Types

The Project's mineralization is a low to intermediate sulfidation epithermal system with silver, lead, and zinc deposits hosted in stock works, breccia veins, and fractures. The gold zone to the south is a low sulfidation epithermal gold occurrence in association with silica. The antimony zone is comprised of stibnite-pyrite veins with silica. There is also sulfide mineralization in the sediments that are essentially barren of silver and lead.

The above combinations are indicative of the epithermal mineralization that is sometimes associated with distal zoning around a porphyritic intrusion.

The Project resource within the Main, Minas, and Este zones is comprised of the low to intermediate sulfidation, silver, lead, and zinc mineralization. The Main and Minas areas are more associated with vein structures, and the Este zone appears to be a broader zone of veinlets and stock works.

The polymetallic Ag-Pb-Zn mineralization is typical of that developed at an elevated crustal setting by rapid cooling of a hot hydrothermal fluid, derived dominantly from an intrusion source that has contacted cool wall rocks and remains at an unknown depth and uncertain metal grade. The important aspect of Corani is that the dilational listric faults (discussed in Section 7) focused substantial intrusion-derived sulfide ore fluids, which were rapidly cooled to provide economic silver - polymetallic grades.

9 Exploration

BCM began exploring the Corani Project in early 2005. In addition to drilling, which is discussed in detail in Chapter 10 of this report, exploration activities carried out by BCM include detailed geologic mapping, trenching, and geophysical surveying.

9.1 Geophysical Exploration

VDG del Perú S.A.C. (VDG) conducted a ground geophysical campaign at the Corani Project on behalf of BCM in the fall of 2005. A total of 44.20 line-km of induced polarization (IP) data was collected, along with 50.95 line-km of magnetic survey. The line layout was established by means of a Real-Time global positioning system (GPS) prior to the geophysical surveys, for a total of 51.65 line-km.

The geophysical surveys were aimed at assisting in geological mapping, including lithologies and key structures, and at mapping mineralization and alteration associated with a low sulfidation gold-silver system. The objective of the IP/Res survey was to map the electrical response by means of high-resolution IP traverses across the favourable north-south corridor identified based on the results of both trench and drilling exploration. The chargeability is instrumental in defining disseminated sulfides associated with economic mineralization.

The field results of both methods were of good quality and were meaningful. The final chargeability and resistivity depth sections mapped systematically clear contrasts from line to line between the sub-surface and a nominal depth of 283 m below surface. The chargeability outlined five (5) IP anomalies (Figure 9-1), two of which, IP1 and IP3, correspond to the Corani Main and Corani Este areas, respectively. Those anomalies accurately mapped the known mineralization and extended the size of both mineralized zones. A separate anomaly (IP4), located to the south and east of the existing resource area, exhibited the same type of geophysical response as the Corani Este anomaly. This anomaly extends for 1,600 m by 400 m and remains open to the south. Another anomaly (IP5) extends to the south of Corani Main and appears to be the extension of the Main anomaly. This anomaly is a wider zone, some 600 by 600 m, and is also open to the south. A reconnaissance survey completed in the southern part of the property along three lines spaced every 500 m outlined encouraging chargeability anomalies that allowed extending the favourable prospective corridor to the south (Figure 9-2).

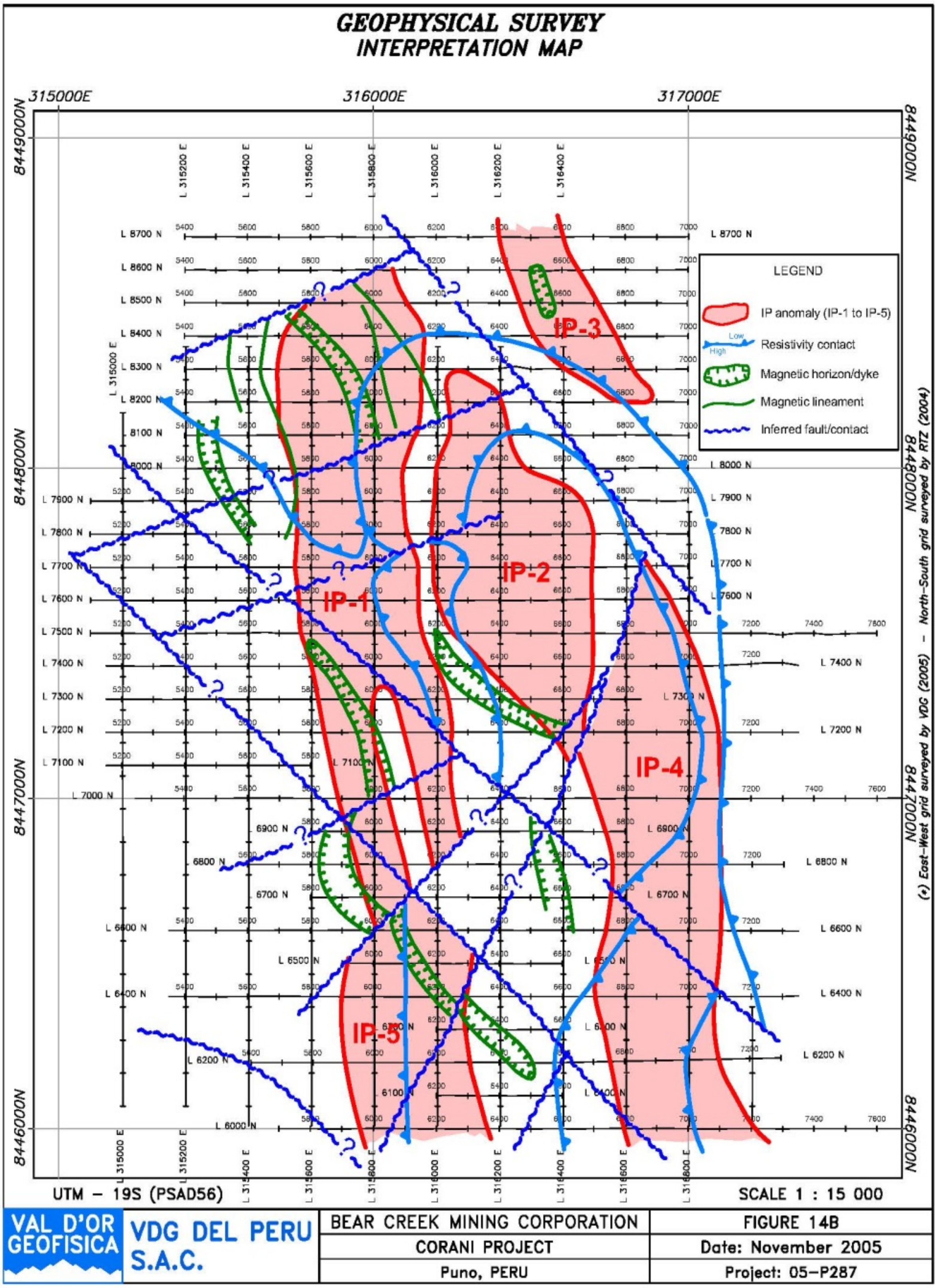


Figure 9-1 Geophysical interpretation map (BCM, 2015)

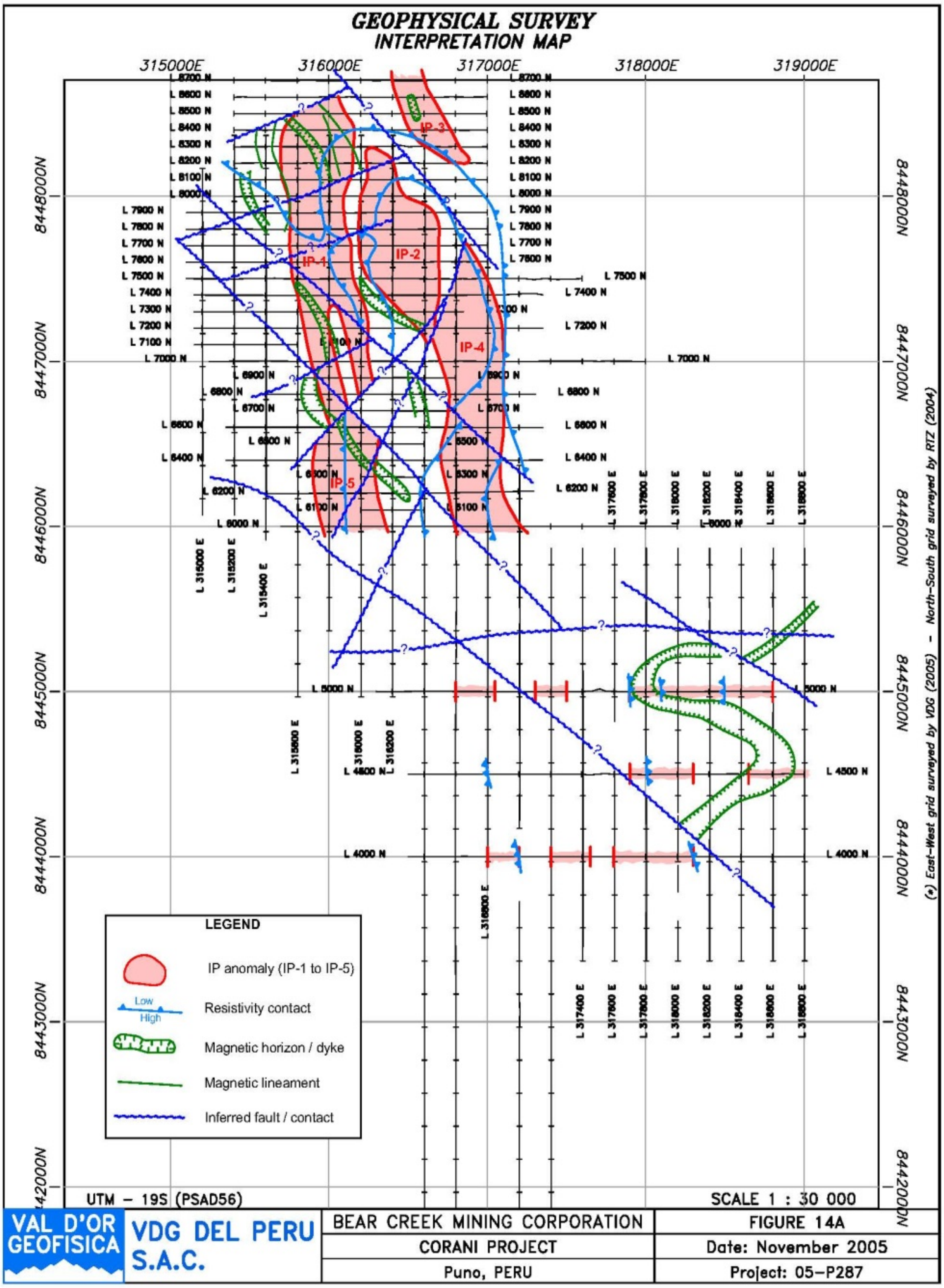


Figure 9-2 Geophysical interpretation map with reconnaissance area (BCM, 2015)

9.2 Trenching Exploration

BCM has completed 25 trenches within the Project resource area (Corani Main, Minas, and Este) to verify the continuity of the structures covered by Quaternary sediments. Spacing between the trenches is roughly 50 to 100 m. Channel samples from these trenches have produced an associated 1,295 assay intervals for a total of 2,924 m of trench data. Another 16 trenches have been completed within the Project area south of the current resource area. All trenching and channel sampling was completed by hand, and samples are collected on 2-m horizontal intervals based on GPS survey of the sample start and end points. Actual sample length was established by digitally draping the 2-m sample intervals onto the topographic surface and adjusting the horizontal sample length to account for the recorded plunge or dip of the trench. In general, the results of trenching exploration indicated near surface mineralization both within and to the south of the current resource area.

9.3 Mapping

BCM has conducted general geologic surface mapping over the entire Project area. The total mapped surface is about 4.5 km wide (east-west) and 7.5 km long (north-south). In 2015, detailed surface mapping, including lithology, alteration, and structures, was performed at a scale of 1:2,500 in the area of the proposed pits.

10 Drilling

10.1 BCM Drilling Exploration

Since 2005, BCM has completed a total of 562 drill holes at the Corani Project for a total of approximately 101,401 m. Drilling was completely by the Peruvian contractor, Bradley MDH, primarily using LD250, JKS35, and LJ44 drill rigs. All the drilling to date has been completed using diamond core drilling methods to produce either HQ (6.35 cm dia.) or NQ (4.76 cm dia.) core.

Figure 10-1 shows the drill hole collar map that covers the area of the project with the estimated resource. Representative sections of the deposit are shown in Section 7.

The typical drill pattern employed by BCM consists of a series of drill fans on section lines spaced 50 m apart. The fans are arranged perpendicular to the strike of specific targets. Angled holes are used to drill perpendicular to the orientation of the mineralization. Multiple holes are often drilled from one site in order to reduce surface impact and obtain the necessary drill coverage (sample spacing) at depth. Some areas have been in-filled to 25 m drill spacing and in other, largely lower-grade target areas, drill spacing remains at 100 m.

Drill hole collars were originally surveyed by handheld GPS with a reported accuracy of ±3.0 meters. Since 2008, all the drill hole collars have been resurveyed by conventional survey with substantially higher precision. Comparison of the resurveyed collar elevations to the surface topography map indicated that about 15% of the collar elevations differed from the topographic elevation by more than 5 m. About two-thirds of these occur in steep topography where the collar elevation is below topography as a result of the cut required to establish the drill pad.

Down hole surveys were not performed for the majority of drill holes. In late 2007, a series of 12 relatively deep holes that targeted deep, high-grade zones within the pit were drilled and down hole surveyed to confirm the presence and location of the grade intercepts. The average depth of the 12 surveyed holes was 220 m. If the down hole survey location of the last interval in the drill hole is compared with the location using only the collar survey, the average error for all 12 holes was 4.89 m. The maximum of all errors was 11.13 m. In the worst case, the drill hole location without survey would have been within one model block.

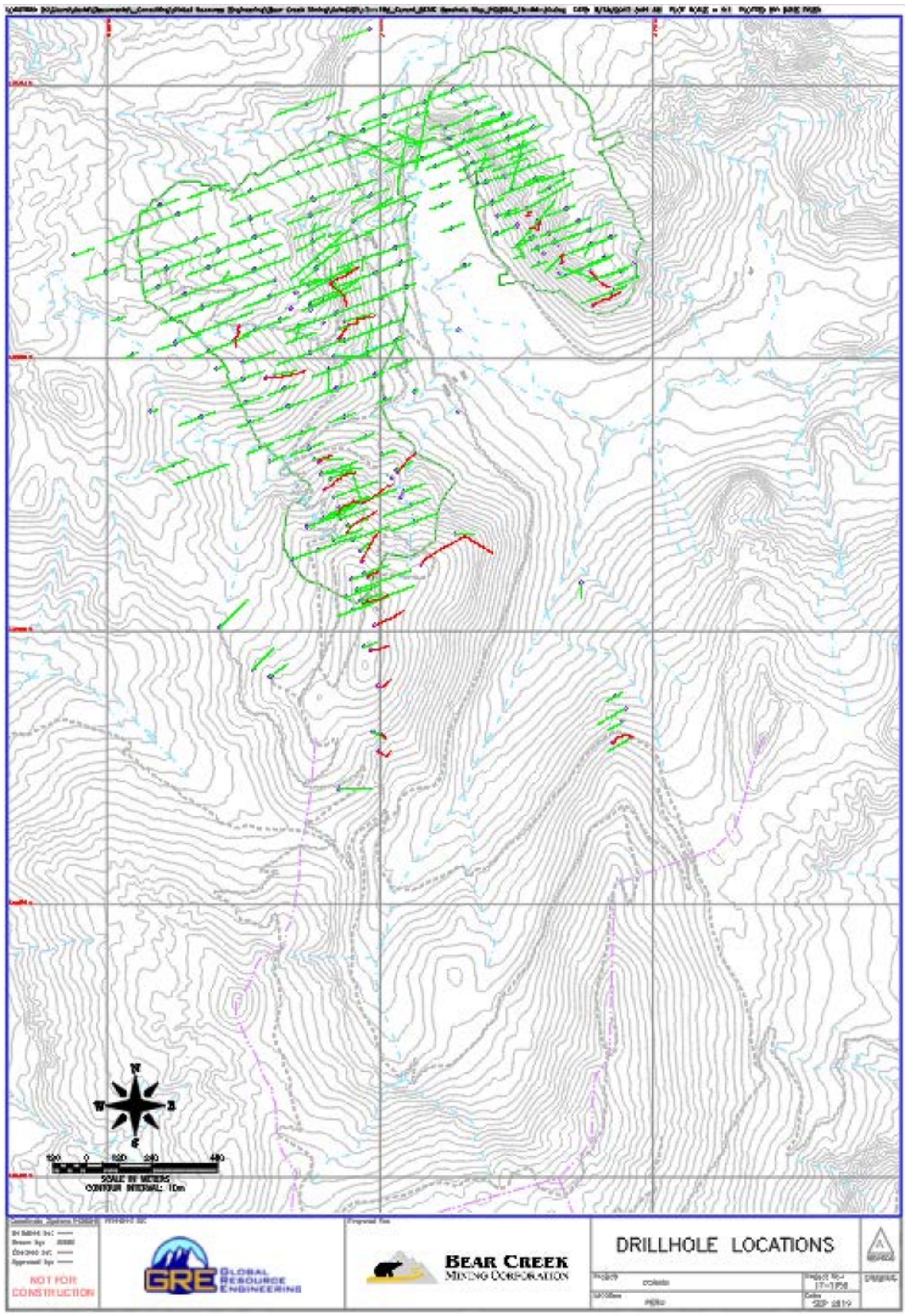


Figure 10-1: BCM drill hole collar locations (GRE, 2019)

The indication from the 12 surveyed holes is that the lack of down hole surveys for holes of 200 m or less would have no major impact on the development of the block model. However, as additional drilling is completed, precision surface surveys and down hole surveys will be implemented for all future drilling. In summary, 12 drill holes have multiple down hole surveys, and the remaining holes are not surveyed. The QP does not believe the potential deflection error in the drilling has a material impact on the resource estimate.

Core recoveries are generally excellent, with no discernible variation in rate of recovery between the two core sizes (HQ and NQ). While on site, the QP carefully reviewed the drilling and sampling procedures employed by BCM with BCM staff. Based on that review, the QP finds no drilling, sampling, or recovery factors that might materially impact the accuracy or reliability of the drilling results.

Considering the recommendations made by the QP for the 2017 Technical Report, BCM implemented a drilling master report. This report contains detail concerning the drill rig, drilling contractor, number of holes, total meters, recovery rates, drill targets, and rationale for drill hole distribution, etc., to ensure that all pertinent information is captured and catalogued in a practical and efficient manner for ease of future use. The 2019 drilling campaigns have been included in this report.

10.2 Drill Hole Data and General Drilling Results

Diamond drill hole data contained in the Project database to date includes 562 drill holes. Not all the 562 drill holes included in the database are in the immediate area of the Project resource. The current block model relies on data from 476 drill holes, with an associated 36,103 sample intervals over a total of 83,104 m of drilling. The block model relies on 36,103 assay values each for silver, lead, zinc, and copper.

In general, drilling exploration has identified and further defined the distribution of mineralization within the three primary resource areas, Corani Main, Corani Este, and Corani Minas. Drilling results indicate that significant mineralization occurs in two basic forms: in large veins associated with the principal listric fault structures and in stockwork veins found in the surrounding rocks. The larger veins are generally rich in quartz / silica and barite and contain typical higher lead-zinc-silver (Pb-Zn-Ag) mineralization and can be several meters in width and quite continuous along strike and down dip. The stockwork veins occur in the wall rock between the faults and contain significant Pb-Zn-Ag mineralization. The basement sediments at Corani are largely barren, though local, discontinuous occurrences of mineralization have been encountered within these units.

Since 2006, BCM has undertaken to in-fill the known areas of mineralization in order to increase confidence in the resource estimate. This has been accomplished by in-fill drilling to produce a nominal 50-m drill hole spacing in previously more widely spaced drilling areas, with a focus on areas of higher-grade mineralization. In late 2018 BCM, drilled 3 new holes in the Minas deposit area, and 3 new holes in the Main deposit area primarily to obtain samples for metallurgical testing. These holes were sampled and prepared for metallurgical testing in 2019, we refer to this as the 2019 drill program.

Drilling exploration since 2005 is summarized by year in Table 10-1.

Table 10-1: Exploration drilling summary by year

Year	No. Holes	Resource Area	Comments/Results
2005	20	Este	Central and southern portion of deposit, directly above outcrop of mineralized structures
	20	Minas	In area of historic workings, intended to test the depth of known veins and breccias
	31	Main	Holes distributed along a wide, westerly dipping quartz-barite-sulfide structure
	3	Other	“Gold zone,” south of Corani Main
2006	136	Este	Defined the form and continuity of Ag-Pb-Zn mineralization beneath post-mineral tuff
	115	Minas	Delineated the projection of mineralization in the north and east portions of the deposit
	43	Main	Holes drilled along the projection of the quartz-barite-sulfide structure and in areas of stockwork with silver sulfides
	39	Other	“Gold zone,” south of Corani Main, defined the depth of the principal quartz-barite-oxide structure with a 14 m width and Au content ranging from trace to 11 g/t with 44 g/t silver
	2	Other	Sedimentary units
2007	18	Este	Holes located in the north portion of the deposit and within the bofedal
	68	Minas	Drilling to confirm distribution of mineralization in the north and west portions of the deposit
	12	Main	Drilling targeted mineralization west of the principal structures
	17	Other	“Gold zone,” south of Corani Main, drilling defined extent of mineralization to the north and at depth
2008	1	Este	
	2	Minas	
	3	Other	Sedimentary units, targeting potential mineralization to the south
2010	10	--	Metallurgical test holes
	10	Other	“Corani South,” holes explored structures with silver and antimony
2011	5	--	Geotechnical and condemnation test holes
	1	Other	“Corani South” exploration holes
2019	3	Minas	Metallurgical test holes
	3	Main	Metallurgical test holes

11 Sample Preparation

The following description of BCM sampling and analytical procedures is based largely on details presented in the 2015 NI 43-101 Technical Report issued by BCM, and in part on observations and conversation with BCM personnel during the QP site visit conducted in August 2017 (M3, 2015).

11.1 Sample Preparation

Diamond core samples are collected and placed into plastic and weatherproof cardboard boxes at the drill rig by the drill crew and are transported by vehicle to the Project camp, where the core preparation facilities are located. BCM geologists photograph the core as it is received from the drill rig and collect geotechnical (rock quality designation [RQD]) and core recovery information prior to selecting sample intervals for splitting. Assay samples, generally 2 m in length, are selected by the on-site BCM geologist and are split using a manual core splitter. One half of the sampled core is returned to the box for geologic logging, and the other half is bagged and tagged with a blind sample number assigned by BCM (Figure 11-1).

Figure 11-1 Bagged core samples prepared for shipment (GRE, 2017)

Channel samples are collected by BCM geologists from hand-dug trenches using a hammer and a moil point chisel. The trenches are excavated by hand to remove the overburden and expose a clean bedrock surface on the trench floor.

Bagged trench and core samples are transported by BCM staff to Cusco or Juliaca, where they are transferred to a bus for shipment to (ISL-certified) ALS-Chemex labs in Arequipa, Peru. The samples are prepped in Arequipa and are subsequently shipped to the ALS-Chemex lab in Lima for analysis. Chain of custody is documented throughout the entire transportation process.

The samples are prepped according to ALS-Chemex preparation code PREP-31, which entails the following:

- the sample is dried at 110° to120°C and crushed by jaw and roll crusher to 70% passing 2 mm (about #10mesh)
- a 250 g subsample is obtained using a riffle splitter
- the split is pulverized using a ring-and-puck pulverizer to 85% passing 75 microns (µm)
- coarse rejects are returned to BCM

11.2 Analytical Procedures

Silver, lead, zinc, and copper assays are carried out by three-acid digestion followed by atomic absorption spectrophotometry (AA) analysis according to ALS-Chemex method AA62, as outlined below:

- a sample of the pulp is digested with three acids: hydrofluoric, nitric, and perchloric, to produce a cake
- the cake is leached with hydrochloric acid
- the hydrochloric acid solution is subjected to AA to determine the concentration of dissolved silver, lead, zinc, and/or copper

The procedure described above is reported to be robust over the reported range of 1 to 1,500 g/t silver.

Multi-element inductively coupled plasma (ICP) analysis is conducted on select sample intervals to assist with mineralization classifications and to guide the interpretation of the metallurgical process response.

11.2.1 2007 – 2012 quality assurance/quality control program

BCM maintains an internal Quality Assurance/Quality Control (QA/QC) program that includes both standard and check sampling. The QA/QC program was initially limited to silver assay data but was expanded to include check assays for Pb and Zn in 2011. BCM's QA/QC efforts currently do not include routine insertion of duplicate or blank samples into the sample stream, except for a few gold blanks.

11.2.2 2017 – 2019 quality assurance/quality control program

Commercially prepared standard samples are inserted into the sample stream at a rate of one standard for every 20 samples. Eight separate standard samples, each with a unique and specific certified assay value, are used. The standards are in pulp form, in contrast to the half-core samples from the Project, so are readily identifiable to the lab; however, the lab is not able to discern which specific standard has been inserted at any given time.

BCM personnel periodically review the standard sample analytical results. If the laboratory analytical result differs from the certified assay value by more than 10%, the entire associated assay run (set of 20 samples) is submitted for re-assay.

2008 Standard Review

In 2008, Mr. Christian Rios, CPG, was reported to have conducted a critical review of BCM's QA/QC program. The work was previously reported in the 2015 NI 43-101 Technical Report issued by BCM (M3, 2015) and included the following discussion:

As of March 2008, 20 sample batches had been submitted for re-assay due to out of tolerance response of the associated standard. The author used the standards to check on assay bias and sample handling procedures.

The statistical results of the ALS-Chemex silver assays indicated that the ALS-Chemex lab tended to undervalue the low-grade silver standards. Several standards in the 1.2 to 1.9 g/t range were reported back as trace by ALS. This is of no impact on reserves as the cutoff grades are significantly above these values.

Several points were outside of the cluster of the data for that standard. These points are typically indications of sample swaps. There are probably about 0.5% sample swapping in the standards database. It is not known if this is a function of improper sample insertion (the likely cause) or assay and database reporting errors elsewhere in the system.

2017 Standard Review

In 2017, GRE conducted a critical review of BCM's QA/QC program. Toward that end, BCM provided GRE with QA/QC data in multiple Excel spreadsheet files. GRE compiled the data into a single, comprehensive QA/QC data worksheet for analysis and evaluation. GRE prepared plots of the certified value of each standard against analytical values returned by the lab. This evaluation is used to check for calibration errors at the lab level and assay bias. Figure 11-2, Figure 11-3, and Figure 11-4 show the results of the standards check for silver, lead, and zinc, respectively. The line on the graph represents a 45 degree line, or the ideal result from all values. The silver analytical results generally correlate well with the standard values with a few outliers possibly indicating that the wrong standard was identified for the sample. The lead and zinc analytical results, however, correlate less well, with two sets of results showing over-valuation.

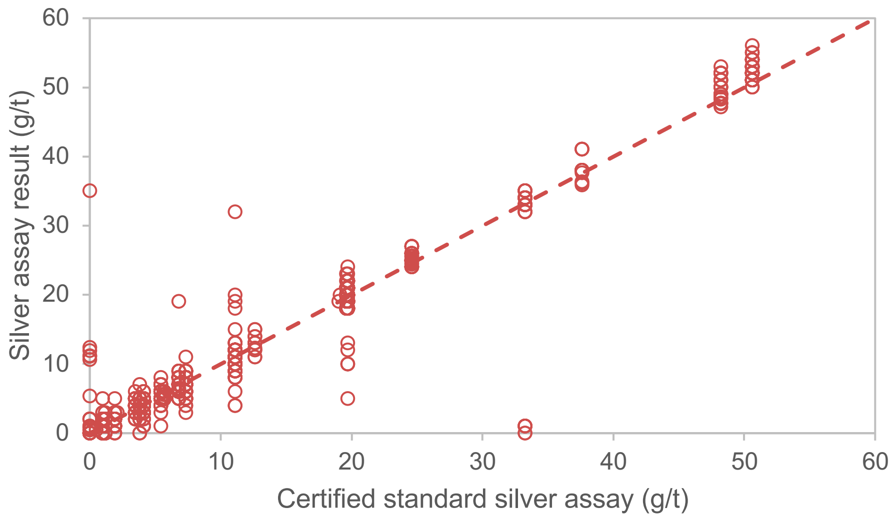


Figure 11-2 Corani Project standards results for silver (GRE, 2017)

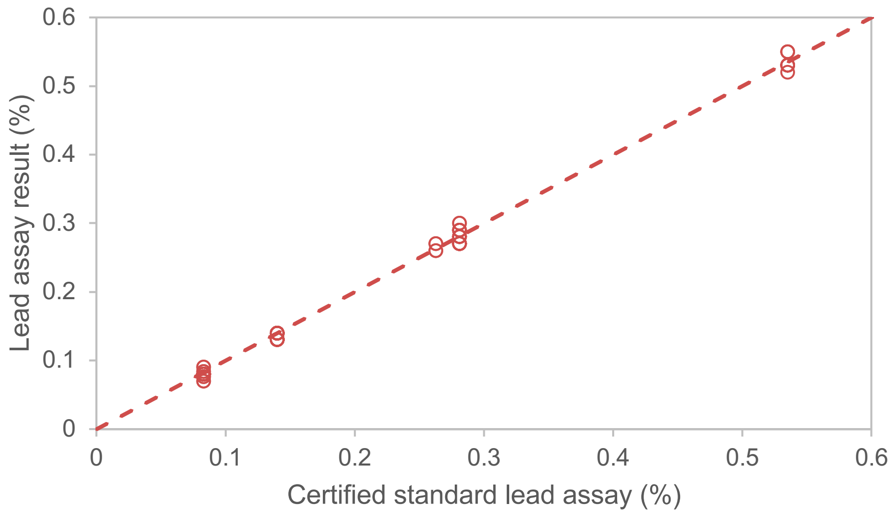


Figure 11-3: Corani Project standards results for lead (GRE, 2017)

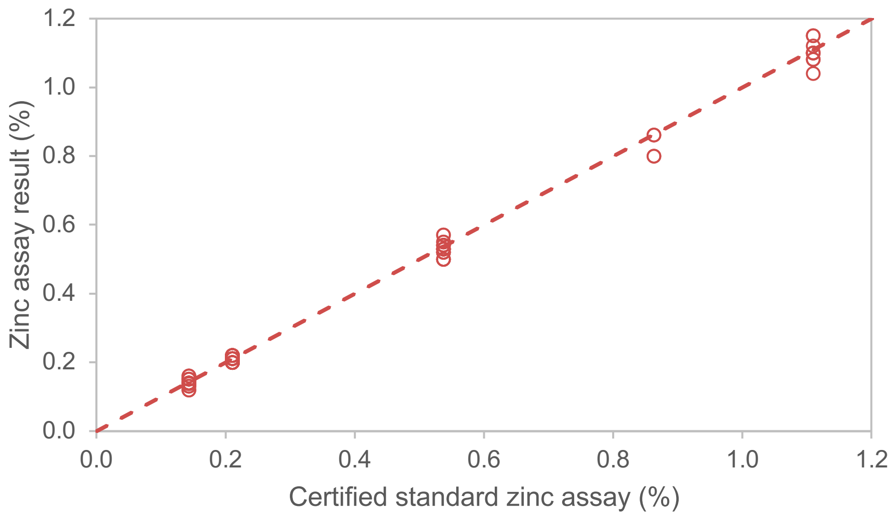


Figure 11-4: Corani Project standards results for zinc (GRE, 2017)

2019 Standard Review

Figure 11-5 through Figure 11-8 show a scatter plot of the certified value for each assay standard compared to the value obtained from the laboratory for 2019 campaign.

A total of 24 standard samples were inserted into the sample stream of 2019, including three different grade of high grade (165 ± 10 ppm Ag, 0.556% ± 0.02% Cu, 1.19% ± 0.07% Pb, and 4.74% ± 0.15% Zn), medium grade (48.2 ppm Ag, 0.535 % Pb, and 1.11% Zn), and low grade (24.6 ppm Ag, 0.281 % Pb ,and 0.537 % Zn).

A 45 degree line represents an excellent correlation between the standard assay certified value and actual assay results. This line passes through all of the sample sets for silver, copper, lead, and zinc with the majority of the points directly adjacent to the line, indicating acceptable accuracy performance for the standards.

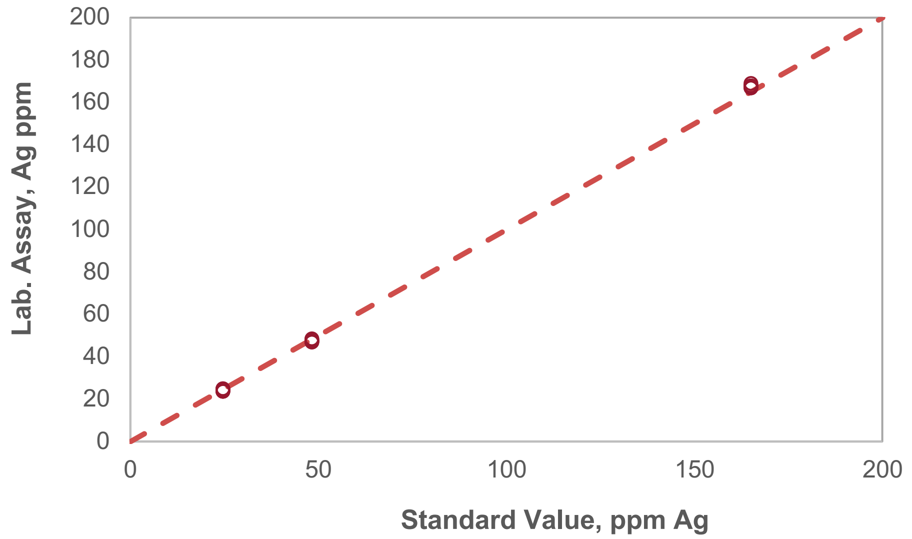


Figure 11-5: Assay standard results, campaign standards (2019) for silver

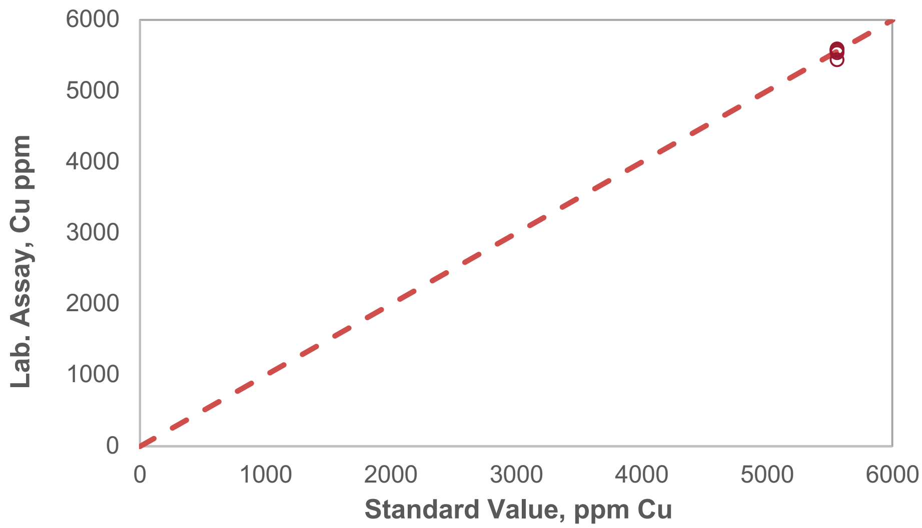


Figure 11-6: Assay standard results, campaign standards (2019) for copper

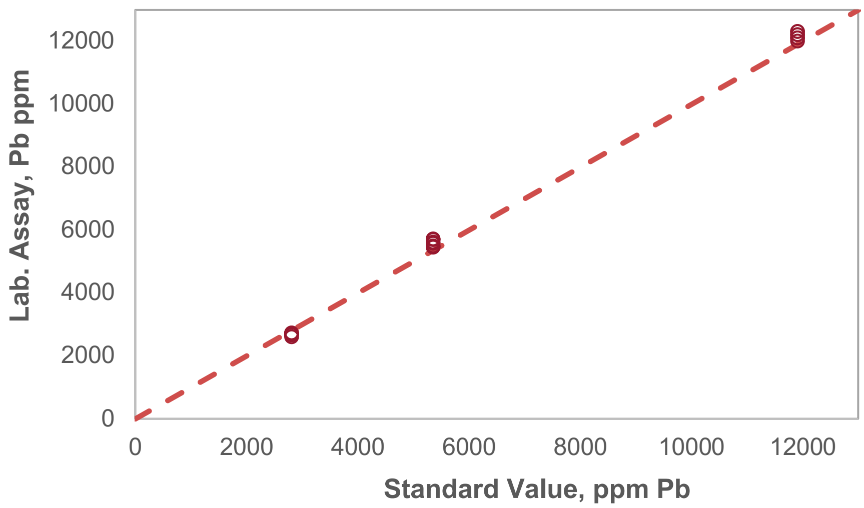


Figure 11-7: Assay standard results, campaign standards (2019) for lead

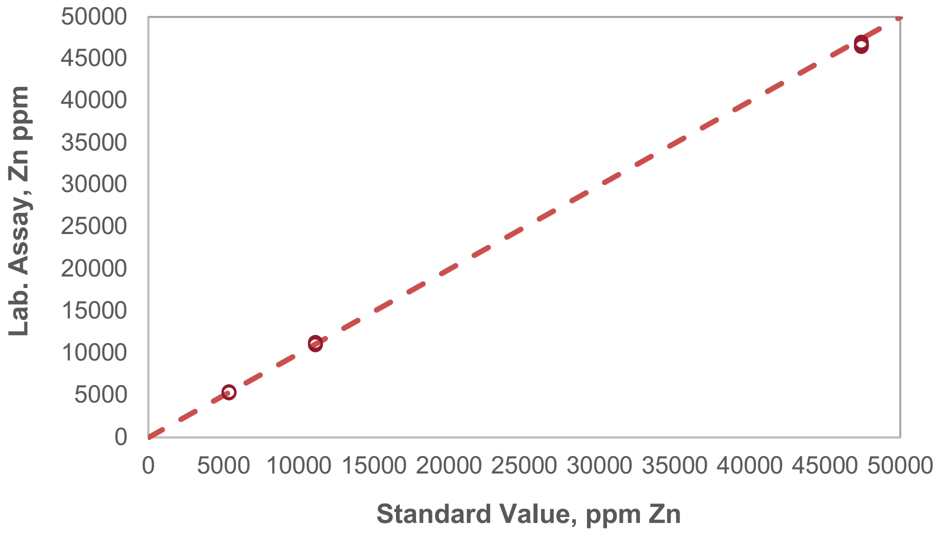


Figure 11-8: Assay standard results, campaign standards (2019) for zinc

2008 Check Assay Review

The following discussion was also prepared by Mr. Rios, and is presented here as previously reported in the 2015 NI 43-101 Technical Report issued by BCM (M3, 2015):

Check assay pulps are submitted to a second laboratory on a roughly 1 in 25 basis. The initial check assay protocols were established during 2005 with silver assay checks only. The procedures were amended in late 2005 to include check assays of silver, lead, and zinc.

The initial checks during 2005 are summarized on Figure 11-3. During this period, check assay pulps were submitted to SGS Mineral Services labs (SGS) in Lima. There is substantial variability in the check assay results during this period. There is no indication of bias in the data set, but there is substantial variability between the original and the check assays. The cause of the variability is not known. There is some potential that the check pulps have been swapped or mislabelled when shipped to the check lab.

The same trend is apparent regarding the high degree of scatter with the Inspectorate checks as with the SGS checks before them. The variability occurs in all three metals: silver, lead, and zinc. Although a degree of scatter is typical for the precious metal assays, the variability of lead and zinc are unusual for base metal check assays on pulps.

There is effectively no scatter in the plots for the most recent checks. It is not certain if the issue has been corrected or that it is not apparent with only 115 samples in the most recent check set.

The variability in check assays for the period of 2005 through 2007 can be summarized by a quick scan of the percentage of checks that were more than 25% different than the original assay.

Silver - 1,978 checks - 18.2% are more than 25% different

Lead - 1,983 checks - 4.2% are more than 25% different

Zinc - 1,984 checks - 7.3% are more than 25% different

Hypothesis tests for each set of check assays do indicate that they can be accepted with 95% confidence and there is effectively no bias in the check assay result. However, the variability issue should be understood. It could simply be a function of misassignment of batch results to the working spreadsheet, or potential miss-labelling of pulps prior to shipment for check assay.

Many of the scattered silver results are the same samples with scattered lead and zinc results. The implication is that the entire check assay has been mis-labelled, or mis-located when inserted into the master spreadsheet.

2017 Check Assay Review

In 2017, GRE conducted a critical review of the check assay results. GRE prepared quantile-quantile (or Q-Q) plots of the check assay data to determine if the two data sets come from populations with a common distribution. The Q-Q plot plots the quantiles (i.e., the fraction of points below a given value) of the first data set against the quantile of the second data set. A 45 degree line is also plotted. If the two sets come from a population with the same distribution, the points should fall approximately along the 45 degree reference line. A Q-Q plot can also test for shifts in location, shifts in scale, changes in symmetry, and the presence of outliers. Figure 11-9 through Figure 11-14 provide the check assay Q-Q plots prepared by GRE. Each plot also shows an R^2 value. The R^2 value is a statistical measure of how close the data are to the 45 degree (1:1) regression line and can vary from 0 to 1. In general, the higher the R^2 value (i.e., the closer to 1), the better the model fits the data.

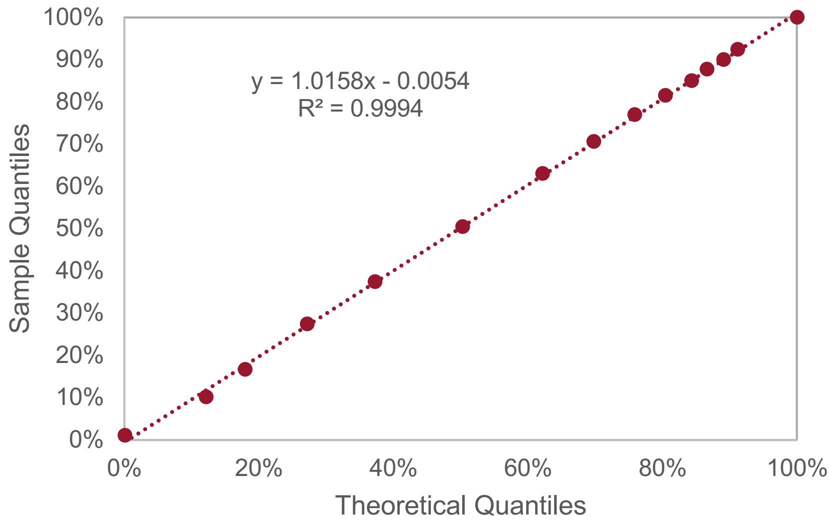


Figure 11-9: Inspectorate check assays (silver) Q-Q Plot (GRE, 2017)

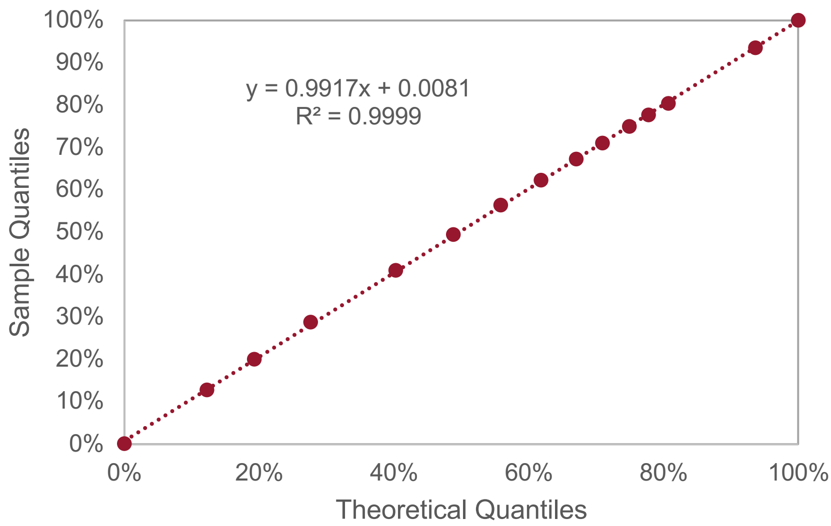


Figure 11-10: Inspectorate check assays (lead) Q-Q plot (GRE, 2017)

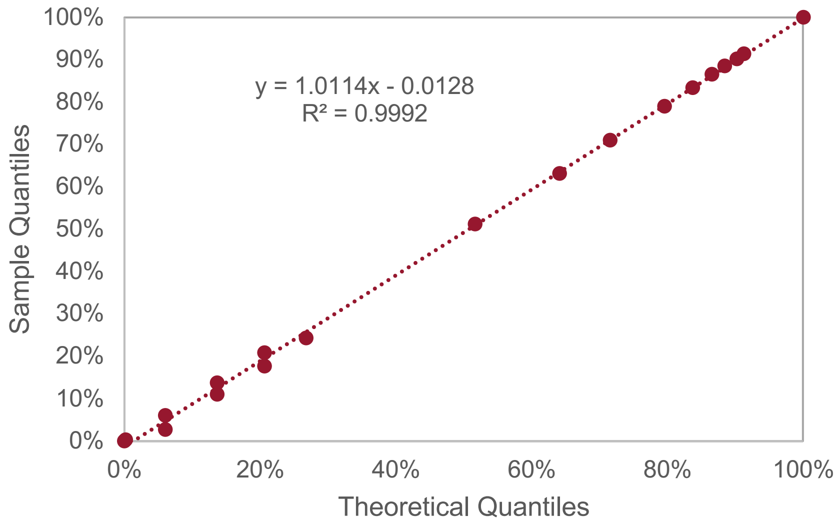


Figure 11-11: Inspectorate check assays (zinc) Q-Q plot (GRE, 2017)

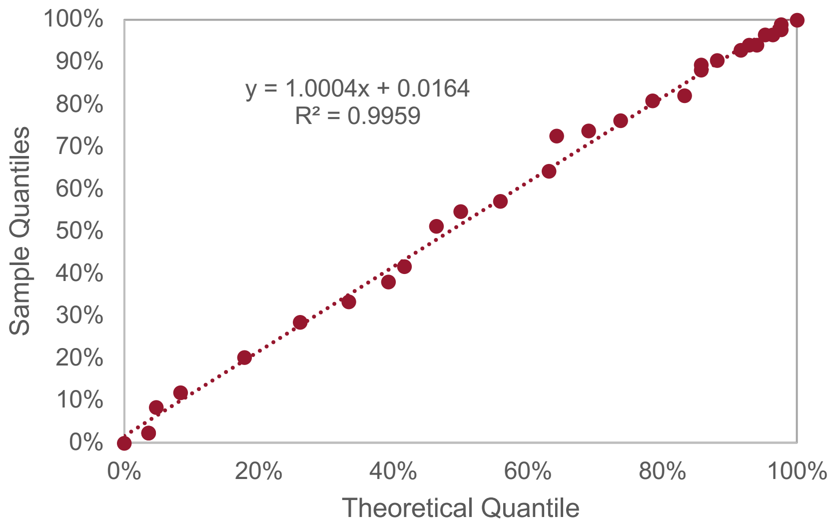


Figure 11-12: CIMM check assays (silver) Q-Q plot (GRE, 2017)

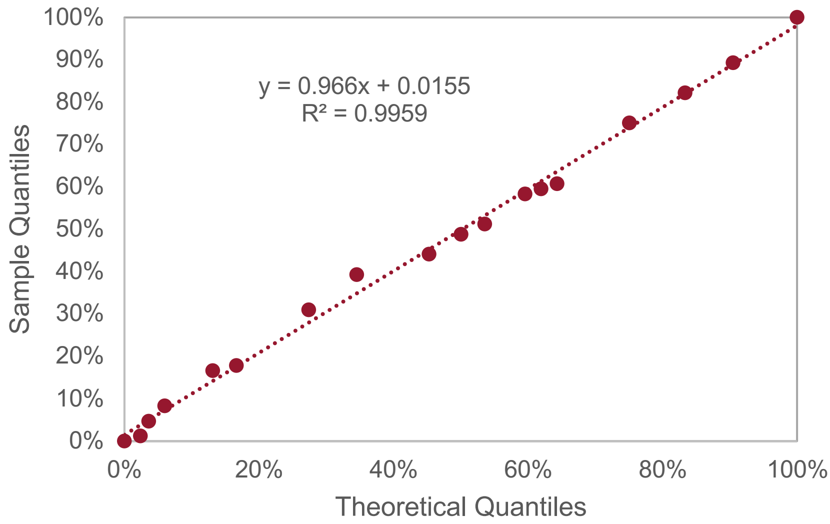


Figure 11-13: CIMM check assays (lead) Q-Q plot (GRE, 2017)

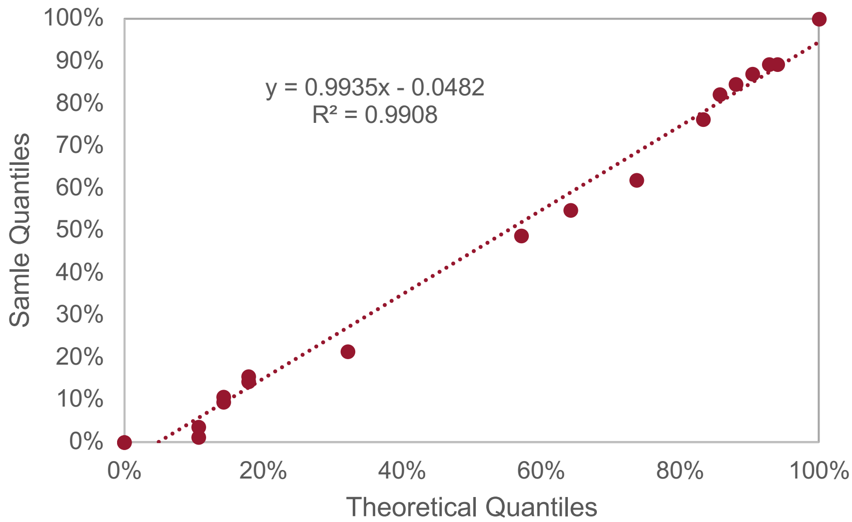


Figure 11-14: CIMM check assays (zinc) Q-Q plot (GRE, 2017)

The Q-Q plots indicate effectively no scatter in the data, with R^2 values ranging from a low of 0.9908 to a high of 0.9999. The ALS inspectorate Q-Q plots, in general, show less scatter than the ALS CIMM Q-Q plots.

Blanks

Gold blanks were included for 16 samples in 2011. All gold analytical results for these 16 samples were below detection, indicating no artificially introduced contamination in the field for these samples.

Although BCM has not routinely inserted blank samples into the sample stream, the laboratories do periodic blank checks internally to check for trace sources of artificially introduced contamination within the lab. A cursory review of the lab blank results indicated no apparent bias introduced by the labs.

2019 Blank Review

A total of 17 blank samples were inserted into the sample stream of 2019 for six holes of DDH-C84-MET, DDH-C152-MET, DDH-CTJ30-MET, DDH-CM13-MET, DDH-CM14-MET, and DDH-C144-MET. Figure 11-15 through Figure 11-18 show the assay results of the blanks used in the QA/QC 2019 campaigns program for silver, copper, lead, and zinc.

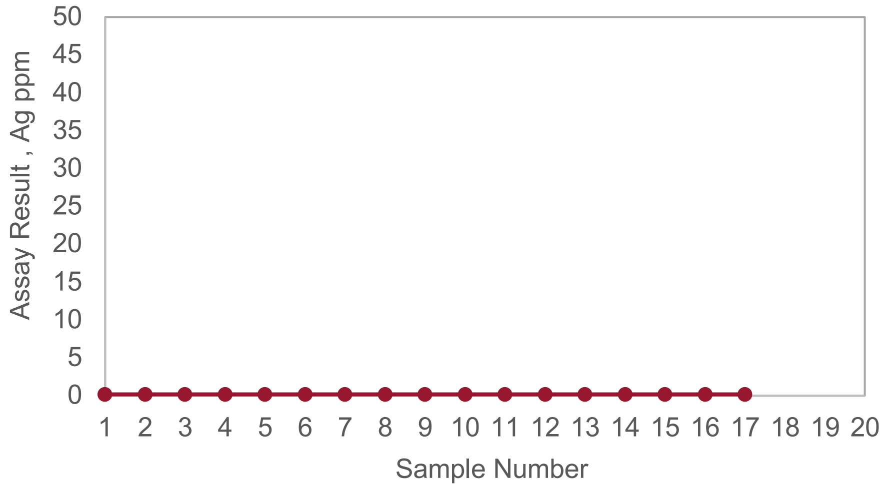


Figure 11-15: Assay results blank samples (2019) for silver

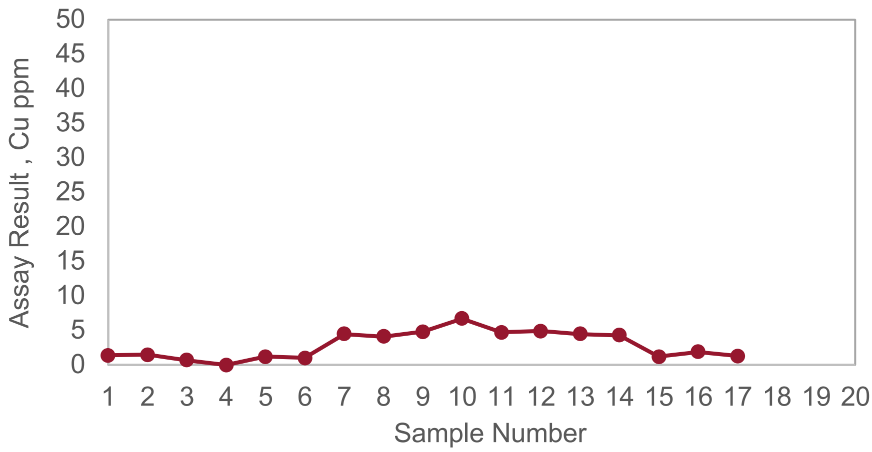


Figure 11-16: Assay results blank samples (2019) for copper

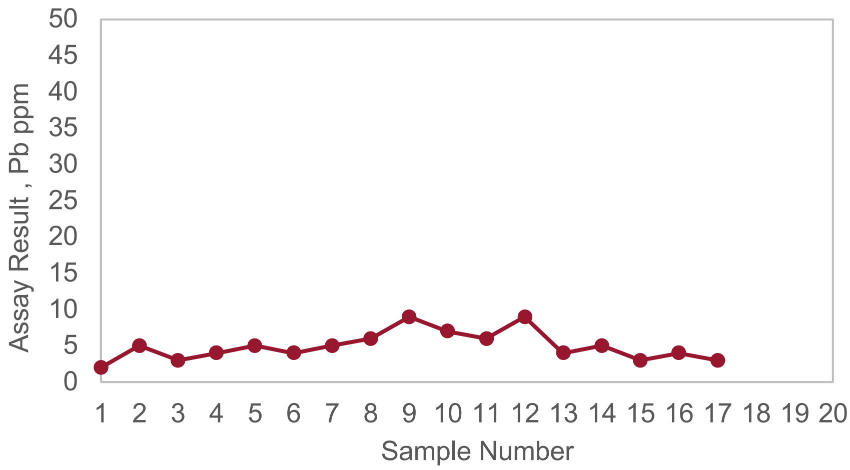


Figure 11-17: Assay results blank samples (2019) for lead

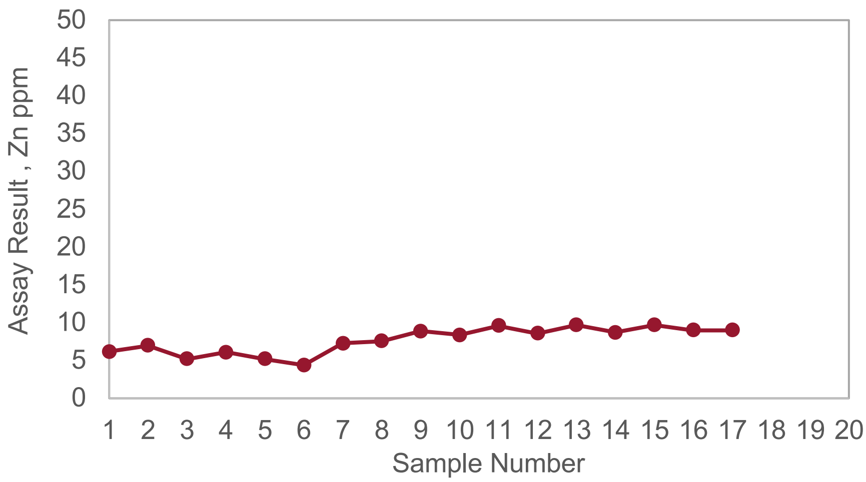


Figure 11-18: Assay results blank samples (2019) for zinc

11.3 Sample Security

During exploration campaigns, BCM employed standard chain of custody procedures during all segments of drill core and trench sample transport. Samples prepared for transport to the laboratory were bagged and labelled in a manner that prevents tampering and remained in BCM control until released to private transport carrier in Cusco or Juliaca. Upon receipt by the laboratory, samples were tracked by a blind sample number assigned and recorded by BCM.

All whole and retained half core samples are stored in a fenced compound located in Juliaca (Figure 11-20 and Figure 11-20). The core is neatly stored in labelled core boxes, which are arranged according to drill hole on sturdy, covered shelving units. Access to the core storage facility by unauthorized personnel is prohibited by a locked gate. Coarse reject and pulp samples are stored in BCM's Lima warehouse facility, which is accessible only to authorized BCM personnel.

Figure 11-19: Juliaca core storage facility (1) (GRE, 2017)

Figure 11-20: Juliaca core storage facility (2) (GRE, 2017)

11.4 QP Opinion on Adequacy

The QP finds the sample preparation, analytical procedures, and security measures described herein to be reasonable and adequate to ensure the validity and integrity of the data derived from BCM's sampling programs, with some room for improvement. Based on observations and conversation with BCM personnel during the QP site visit, in conjunction with the results of GRE's review and evaluation of BCM's QA/QC program, the QP made the following observations and recommendations:

- In 2017 GRE recommended BCM document and implement a chain of custody program for future drill programs. BCM followed those recommendations for the 2019 drill program. GRE was provided the document describing the chain of custody program and the chain of custody reports. The 2018-2019 drill program was overseen by a certified Geological Engineer.
- BCM current QA/QC program with standards, blanks, and duplicates meet international mining industry standards.
- More coarse duplicates are recommended to quantify the variances introduced into the assay grade by errors at different sample preparation stages. Coarse duplicates are inserted into the primary sample stream to provide an estimate of the sum of the assay variance plus the sample preparation variance, up to the primary crushing stage. An alternative to the coarse duplicate is the field duplicate, which, in the case of core samples, is a duplicate from the core box (i.e., a quarter core or the other half core). Because sample preparation is currently carried out by the laboratory (and not by BCM), if coarse duplicates are preferred (in order to preserve drill core), the coarse duplicates should be sent for preparation and assaying by the second laboratory.
- QA/QC analysis should be conducted by BCM on an on-going basis and should include consistent acceptance/rejection tests. Each round of QA/QC analysis should be documented, and reports should include a discussion of the results and any corrective actions taken.

12 Data Verification

12.1 Site Inspection

GRE representative and QP, Hamid Samari, conducted an on-site inspection of the Corani Project and the Juliaca core storage facility on August 7 through 10, 2017. GRE staff spent two full days at the Project site accompanied by BCM geology staff Enrique Osorio Cornejo and Jorge Ganoza. While on site, he conducted general geologic field reconnaissance, including inspection of bedrock exposures and other surficial geologic features, ground-truthing of reported drill collar and trench sample locations, and superficial examination of historic mine workings. One full day of the site visit was spent at the core storage facility in Juliaca, where select intervals of whole and half core were visually inspected, and samples were selected to submit for check assay.

Field observations during the site visit generally confirm previous reports on the geology of the Project area. Bedrock lithologies, alteration types, and significant structural features are all consistent with descriptions provided in existing Project reports, and the author did not see any evidence in the field that might significantly alter or refute the current interpretation of the local geologic setting.

A total of 96 collar locations (approximately 20% of the total used to develop the block model) were verified in the field using a handheld GPS unit. The average variance between field collar coordinates and collar coordinates contained in the Project database is roughly 18 m, well within the expected margin of error accounting for the difference between the methods of survey (handheld unit versus professional ground survey).

Specific core intervals from 35 separate drill holes were selected for visual inspection and potential check sampling based on a preliminary review of the drill hole logs and associated assay values. The core intervals were selected prior to the site visit, and the core was laid out by BCM staff and ready for inspection upon arrival. With few exceptions, the core samples accurately reflect the lithologies recorded on the logs. Given the similarity in composition of the pre- and post-mineral tuff, it is often difficult to distinguish between the two without greater context. In some cases, (unsampled) core intervals logged as post-mineral tuff were variably altered and oxidized, similar in character to most of the core logged as pre-mineral tuff. Three such sample intervals were selected from a single drill hole, DDH-C59B, for assay. The sample intervals selected were exhibited gradational visible evidence of alteration/oxidation (i.e., one very altered, one moderately altered, and one barren in appearance). Assay results (Table 12-1) indicate that the samples are in fact mineralized.

Table 12-1: Previously unsampled post-mineral tuff assay results

DHID	Interval		Assay Results		
	From m	To m	Ag ppm	Pb %	Zn %
DDH-C59B	11.67	12.81	66	0.25	0.03
DDH-C59B	26.01	27.33	141	2.19	0.07
DDH-C59B	32.67	33.87	35	0.14	0.01

A total of 17 samples were selected for check assay. The samples were selected from low, moderate, and high-grade intervals based on original assay results. In all cases, the degree of visible alteration and evidence of mineralization observed was generally consistent with the grade range indicated by the original assay value. Laboratory analysis was completed by ALS Peru using the same sample preparation and analytical procedures as were used for the original samples. Standard t-Test statistical analysis was completed to look for any significant difference between the original and check assay population means. A single sample was removed from the total sample population based on an erroneous original assay value. The results of the t-Test showed no statistically significant difference between the means of the two trials (original versus check assay).

12.2 IMC Audit

In 2011, BCM contracted Independent Mining Consultants (IMC) to perform an audit of the digital Project database. The following discussion was previously presented in the 2015 NI 43-101 Technical Report issued by BCM (M3, 2015):

IMC compared a random selection of original assay certificates to the assay information contained in the Corani Project database. Assay data from the following drill holes was used during the audit process:

DDH-C3-A	DDH-C7	DDH-C12-A	DDH-C16-B	DDH-C18-A	DDH-C20-A
DDH-C29-B	DDH-C32-A	DDH-C34-A	DDH-C41	DDH-C42-A	DDH-C43-B
DDH-C46-A	DDH-C58-B	DDH-C66-A	DDH-C70-A	DDH-C74-B	DDH-C79-A
DDH-C84-A	DDH-C86-A	DDH-C92			

The assay certificate data was entered into an excel spreadsheet and then added to the IMC database containing the Corani data.

The 21 drill holes evaluated had 1,524 associated silver, copper, lead, and zinc assay intervals. Assay certificate data was received for 1,310 assay intervals; the intervals for which original assay data was not provided are summarized in Table 12-2.

Table 12-2: Missing original assay certificates

Hole Number	Interval	Number of Intervals Missing Assay Certificates
DDH-C3-A	1,988 to 2,052	62 intervals
DDH-C12-A	2,210 to 2,287	74 intervals
DDH-C16-B	9,388 to 9,447	57 intervals
DDH-C20-A	4,669 to 4,690	21 intervals

Thirty-one silver assays with a value less than 1.0 g/t were entered into the database with a value of 1.0. Otherwise, there were 2 assay intervals with silver assay values that did not match the assay certificates (Table 12-3).

Table 12-3: Certificate check errors for silver

Hole Number	From m	To m	Sample Number	Database Ag grade g/t	Certificate Ag grade g/t
DDH-C41	98	100	8842	8	108
DDH-C70-A	104	106	15521	29	11

Eight lead assays with a value less than 0.01 were entered into the database with a value of 0.01. There were 2 assay intervals with lead assay values that did not match the assay certificates (Table 12-4).

Table 12-4: Certificate check errors for lead

Hole Number	From m	To m	Sample Number	Database Ag grade g/t	Certificate Ag grade g/t
DDH-C41	98	100	8842	0.04	5.90
DDH-C70-A	104	106	15521	0.02	0.01

Two zinc assays with a value less than 0.01 were entered into the database with a value of 0.01. There were 2 assay intervals with zinc assay values that did not match the assay certificates (Table 12-5).

Table 12-5: Certificate check errors for zinc

Hole Number	From m	To m	Sample Number	Database Zn grade %	Certificate Zn grade %
DDH-C41	98	100	8842	0.01	1.59
DDH-C70-A	104	106	15521	0.04	0.01

A total of 383 copper assays with a value of less than 0.01 were entered into the database with a value of 0.01. There was 1 assay interval with a copper assay value that did not match the assay certificate (Table 12-6).

Table 12-6: Certificate check errors for copper

Hole Number	From m	To m	Sample Number	Database Cu grade %	Certificate Cu grade %
DDH-C41	98	100	8842	0.01	0.13

The results for the individual metals all appear on the same record, indicating that two records out of 1,310 records were entered in error. This sampling shows an error rate of 0.15%, which is an acceptable error rate, and the data is considered reasonably accurate and suitable for use in estimating mineral resources and mineral reserves.

The observed discrepancy between trace assay entries and certificate values is likely a function of continuity between data entry personnel. This issue is minor and has no material impact on the determination of mineral resources or reserves but should be addressed for consistency. The stated procedure by BCM personnel is to enter the less than trace results at half of the value of the trace assay. For example, <1 g/t silver should be entered into the database as 0.5 g/t silver according to BCM protocol.

IMC also compared trench sample assay values with nearby core sample assay values. IMC composited the data into 8 m down hole (or down trench) length composites. The composites were then paired on a nearest neighbour basis.

The nearest neighbour procedure finds pairs of trench and diamond drilling composites that are within a specified distance to each other, so that a statistical comparison of the two data sets can be completed. For this test, IMC used 8 m, 16 m, and 24 m spacing between data pairs, which corresponds to the unit size of the 8 m length composites. There were only 20 to 25 pairs at the 8-m spacing, but there were over 80 pairs at the 16 m spacing, which is a sufficient quantity to provide for a robust statistical estimate.

Table 12-7 summarizes the results of the nearest neighbour comparison. The comparison was applied for silver and lead only, as trench samples were not assayed for zinc. Statistical hypothesis tests were completed on the two closely located sample sets. The pass/fail determinations presented in Table 12-7 are based on the application of a 95% confidence interval. The T-test is a comparison of the population means. The Paired T calculates the differences between individual pairs and confirms that the differences are sufficiently small. The binomial test is a check of how many times one population is greater than the other, and the KS (Komologorov-Smirnoff) test is a comparison of the overall shape of each distribution.

In all cases, the test results indicate that the trench and core sample assay data can be commingled for the purposes of developing a block model and calculating the mineral resource estimate.

Table 12-7: Nearest neighbor comparison – trench vs. diamond drill samples

Metal	Maximum Spacing Between Composites	Number of Pairs	Diamond		Trench		T Test on Means	Paired T on Pairs	Binomial Test	KS Test
			Mean	Var	Mean	Var				
Ag (g/t)	16 m	84	82.10	3535	82.94	3622	Pass	Pass	Pass	Pass
Pb (%)	16 m	87	1.00	0.439	1.09	0.621	Pass	Pass	Pass	Pass

12.3 GRE Audit

12.3.1 Digital project database

In 2017, GRE completed a QA/QC audit of the digital Project database. GRE compared a random selection of original assay certificates to the assay information contained in the Corani Project database. Assay data from the following certificates was used during the audit process:

LI05099746	LI06037967	LI07052871	LI07003983	LI08034674	LI05029718
LI05074420	LI07032682	LI05023340	LI06040456	LI06115686	LI06007458
LI07026517	LI06036875	LI06080182	LI07041835	LI06115686	LI06047343
LI07026518	LI06037966	LI06058508	LI07063684	LI07038585	

The results of the audit are summarized below:

- All assay results viewed that were less than the reporting limit were entered into the database with a value of 0.
- Samples from six of the checked certificates were not located in the Project database.
- One certificate included sample IDs that were in the database twice, with one set of results consistent with the certificate and one set different from the certificate.
- All other results were consistent between the database and the certificates.

The observed discrepancy between trace assay entries and certificate values is likely a function of continuity between data entry personnel. This issue is minor and has no material impact on the determination of mineral resources or reserves but should be addressed for consistency. The stated procedure by BCM personnel is to enter the less than trace results at half of the value of the trace assay. For example, <1 g/t silver should be entered into the database as 0.5 g/t silver according to BCM protocol.

The other QA/QC checks show minimal to no error, which GRE believes represents an acceptable error rate, and the data is considered reasonably accurate and suitable for use in estimating mineral resources and mineral reserves.

GRE also completed a mechanical audit of the Project database in order to evaluate the integrity of data from a data entry perspective. The mechanical audit identified a small number of data entry errors, including gaps, overlaps, and missing sample intervals. All data entry errors were easily rectified and are considered insignificant with regard to potential impact to the mineral resource and mineral reserve estimates.

12.3.2 Density data

BCM has performed 1,100 density determinations using a waxed core method. Samples were chosen out of every 5th core box to provide a sample spacing of approximately 15 m.

In 2015, GRE analysed the density data to determine if a relationship exists between density and a variety of parameters, including assay grade, mineralization type, and deposit area. Silver assay values in g/t were converted to percent to normalize the grade units. Statistical analysis indicates that the best predictor of density is a combined silver, lead, and zinc grade of 0.94%. The average density of samples with a combined grade less than 0.94% is 2.31 t/m^3, and the average density of samples with a combined grade of greater than or equal to 0.9381% is 2.43 t/m^3. The average density of post-mineral tuff with no associated assay data is 2.3 t/m^3, and the average density of non-tuff material is 2.53 t/m^3. The densities (in t/m^3) applied for each lithologic unit during mineral resource estimation are presented in Table 12-8.

Table 12-8: 2015 Updated densities

Rock Type	Grade Pb+Zn %	Density t/m^3
Pre-Mineral Tuff	< 0.9381	2.31
Pre-Mineral Tuff	>= 0.9381	2.43
Post-Mineral Tuff	Not Applicable	2.3
Other Materials	Not Applicable	2.53

12.4 QP Opinion on Adequacy

Based on the results of the QP's check sampling effort, verification of drill hole collars in the field, visual examination of selected core intervals, and the results of both manual and mechanical database audit efforts, the QP considers the collar, lithology, and assay data contained in the Project database to be reasonably accurate and suitable for use in estimating mineral resources and reserves.

The database audit work completed to date indicates that occasional inconsistencies and/or erroneous entries are likely inherent or inevitable in the data entry process. The QP recommends that BCM establish a routine, internal mechanical audit procedure to check for overlaps, gaps, total drill hole length inconsistencies, non-numeric assay values, and negative numbers. The internal mechanical audit should be carried out after any significant update to the database, and the results of each audit, including any corrective actions taken, should be documented and stored for future use in database validation.

Based on the positive assay results of the selected intervals of previously unsampled post-mineral tuff, the QP recommends that BCM sample at least one 2 m interval for every 20 m drilled and logged as post-mineral tuff. If positive assay results are returned, additional intervals should be selected accordingly to ensure that all mineralized material is analysed.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

Mineral processing investigations for the Corani Project have evolved over the past 13 years. Initially, a complex geological classification was used that included 9 classifications, of which FBS (Fine Black Sulfide) was the predominant classification. These nine classifications were categorized into four ore types based upon general amenability to processing (see Section 13.2). Type I and Type II ores generally exhibited good, but highly varied metallurgical results. Type III ores demonstrated relatively poorer results but were still regarded as treatable. Type IV ores were regarded as untreatable. During the investigations it became apparent that metallurgical performance was primarily influenced by mineralization and not the general geological classifications.

The current approach simplifies the classification into Sulfide (Type I / II), Transition (Type III), and Oxide (Type IV). Oxide ores are not included in the resources / reserves. The following terms can be used interchangeably in the reading of the text: Sulfide - Type I/II, Transitional - Type III. A rigorous and robust statistical evaluation of the test work has been conducted and validated to yield the recovery formulas incorporated into the block model. The lead and silver recovery formulas (Section 13.8) have been separated for Sulfide and Transitional ores in order to improve the model predictions. Zinc maintains a single recovery formula across Sulfide and Transitional ore types.

This section describes the evolution of the test-work, model development, and changes to the mineral processing and metallurgical testing, and as such, uses the terminology represented in the underlying reports. Section 13 (Mineral Processing and Metallurgical Testing) should be read as a whole as individual portions taken out of context can be misleading.

The Corani deposit is a lead-zinc deposit with significant associated silver. The deposit is fairly typical of a complex base metal sulfide, composed of fine grained intergrown lead, zinc and iron minerals. The mineralogy also exhibits varying degrees of oxidation resulting in oxide and transitional zones with subsequently lower sulfide concentrations. The oxide and transition zones do not make up an appreciable amount of the deposit.

A wide array of testing has been conducted on the Corani deposit including comminution, cyanide leaching, flotation both bulk and selective, thickening and filtration, materials handling as well as environmental analysis. The metal recovery investigations indicate that selective flotation to produce rougher concentrates followed by cleaner flotation provided the best overall results for the sulfide mineral types. Locked-cycle flotation results indicate that salable concentrates can be made for both lead-silver and zinc with reasonable metal recoveries. The locked-cycle flotation tests performed on the sulfide ore composites showed that lead recoveries ranged from approximately 62% to 78% with corresponding concentrate grades of 61% to 49% lead. Total silver recoveries ranged from approximately 63% to 84%. Zinc recoveries ranged from 39% to 75% with corresponding concentrate grades of 55% and 53% zinc.

Comminution tests on the deposit indicated that the material appears to be of medium hardness with respect to SAG and ball milling with an average Bond rod mill work index (WiRM) of 10.4 kWh/t and a Bond ball mill work index (WiBM) of 14.9 kWh/t. The deposit shows a range of variability that is dependent on the mineral zone.

Cyanide leaching was also investigated as a primary metal recovery method. Unfortunately, the recovery of silver by cyanide did not produce consistently high recoveries across all deposit types. Silver recoveries ranged from 26% to 89% across 12 composites. Therefore cyanide leaching was rejected as an option.

A chronology of the metallurgical tests that have been conducted on the Corani deposit is described in Table 13-1.

Table 13-1: Corani metallurgical testing chronology

Year	Testing Lab	Samples Used	Description
2006	Dawson Metallurgical Laboratories Inc.	13 composites: coarse assay reject and whole core; sulfide, and mixed ore type	Bulk sulfide flotation tests lead rougher and scavenger flotation; selective flotation tests performed on six composites with subsequent zinc flotation on four composites with high zinc content. Combined flotation and leaching tests performed on two composites, including whole-ore cyanidation and flotation tails leaching.
2006	G&T Metallurgical Services Ltd.	1 composite prepared from 12 core samples from one mineral zone	Rougher tests producing bulk (lead) and zinc concentrates; cleaner flotation tests to produce selective concentrates.
2007	G&T Metallurgical Services Ltd.	71 drill core samples from the three pit zones (27 Este, 19 Main, 25 Minas)	Rougher flotation and bulk cleaner flotation tests performed on all samples; sequential flotation tests performed on selected samples containing adequate sphalerite and galena content. Bottle roll cyanidation leach tests conducted on selected samples and silver head assays used to verify results
2007	SGS Lakefield / SGS Vancouver	7 composites created from drill core samples: A, B, B2, C, D, E, F	Bulk sulfide flotation and sequential flotation; selective lead flotation and bulk lead/zinc flotation to produce a high-grade bulk concentrate with further lead/zinc separation tests on the concentrates produced from bulk flotation tests; bulk sulfide flotation
2008	SGS Lakefield / SGS Vancouver	27 composites created from drill core samples: 15 alphabetical (G – U) and 10 according to source area (Main, Minas, and Este)	Numerous batch flotation tests, lead rougher tests, lead cleaner tests, locked cycle tests, zinc cleaner tests, and sequential flotation tests performed on samples listed. Extensive testing performed on selected composites
2008	SGS Lakefield / SGS Vancouver	Composites created from drill core samples: L1, L2, L3, M1, G1, G2, K, U1, U2, L, OT I/II	Preliminary grindability tests (using samples not used in flotation testing).
2008	Pocock Industrial Inc. / SGS Vancouver	Flotation concentrates (Type I/II Pb Con, Type I/II Zn Con, Type III Pb Con) and tailings (Type I/II and Type III); products taken from 2007/2008 SGS testing	Flocculant screening and conventional static thickening tests and environmental tests including aqua-regia digestion, whole-rock analysis, Synthetic Precipitation Leaching Procedure (SPLP) tests,
2009	SGS Vancouver	2 composites: type I/II Pb-Ag-Zn material, type III Pb-Ag	86 batch flotation tests and four locked cycle tests: 47 rougher tests on Type I/II material and 38 tests with Type III material; 18 cleaner tests on Type I/II and 20 tests with Type III, with some tests including a regrind step; two batch flotation tests and one locked-cycle test conducted on a blended sample of material.
2009	G&T Metallurgical Services Ltd.	2 composites created from 20 drill core samples representing two drill holes: Hole 26 from the Main zone, and Hole 181 from the Minas zone	Thirty-two (32) 96-hour Bottle Roll Cyanidation leach tests to assess effect of various leach parameters; silver head assays also conducted to validate results

Year	Testing Lab	Samples Used	Description
2010	SGS Chile	6 composites: 3 transitional and 3 mixed sulfide composites from the Este, Minas, and Main pits	Numerous grind tests on the composites including SAG Mill Comminution (SMC) tests, SAG Mill Power Index (SPI), Bond Abrasion tests, Low Energy Impact Tests (LEITs), Bond Rod Mill Work Index, Bond Ball Mill Work Index, and specific gravity (SG).
2011	University of British Columbia / SGS Vancouver	2 tailings samples – Zn tails, Pyrite tails	Rheology tests at various pup densities to obtain shear stress and shear rate data
2012	SGS Vancouver	9 composites: 3 Pb-Ag-Zn, 3 Pb-Ag, prepared from 14 variability samples out of 25 received; 2 master composites created from 12 received variability samples; one Pb-Ag-Zn composite created for flotation and third-party testing on flotation products	Batch flotation tests performed on variability samples and all composites created; lead rougher, lead cleaner, zinc rougher, zinc cleaner tests; locked-cycle tests performed on master composites and bulk flotation composites, including sequential rougher flotation for lead/zinc separation, concentrate regrinding, and cleaning; bulk flotation performed on Pb-Ag-Zn composite created for external testing;
2012	SGS Chile	20 composites made from drill core samples; composites separated by ore type (transitional, high sulfide, low sulfide)	Grind tests on drill core composites included: Unconfined Compressive Stress (UCS), Point Load Tests (PLT), Low Energy Impact Test (LEIT), SAG Power Index (SPI), SAG Mill Comminution (SMC), Bond Abrasion Index, Bond Rod Mill Work Index, and Bond Ball Mill Work Index
2018/ 2019	Base Metallurgical Laboratories Ltd.	6 composites created for test-work done in 2018: 1, 2, 3, 4, 5, and 6; 6 more composites created for 2019 test-work	Extensive flotation testing; lead rougher and cleaner tests performed on Pb-Ag concentrates, lead rougher, lead cleaner, zinc rougher, and zinc cleaner tests performed on Pb-Ag-Zn composites; locked-cycle tests performed on all concentrates, with Pb-Ag-Zn composites producing zinc concentrates.
2019	Base Metallurgical Laboratories Ltd. / SGS Vancouver	Composites previously used for 2019 test-work performed by Base Met Labs (flotation)	Tests performed by Base Met Labs included: Unconfined Compressive Stress (UCS), Point Load Tests (PLT), SAG Mill Comminution (SMC), Crusher Work Index (CWi), Bond Ball Mill Work Index and Bond Abrasion Index; Tests performed by SGS Vancouver included: LEIT, SPI, Bond Rod Mill Work Index, and specific gravity.
2019	Jenike and Johanson Chile S.A.	3 representative samples of crushed ore	Ore characterization and material flowability tests: moisture tests, cohesive strength, compressibility, bulk density, wall friction, minimum outlet size, chute size, and material drawdown tests

Year	Testing Lab	Samples Used	Description
2019	Jenike and Johanson Chile S.A.	1 representative sample of filtered tailings	Ore characterization and material flowability tests: saturation (moisture) tests; tests at different moisture levels include instantaneous and time flow function tests, wall friction tests, chute tests, compressibility tests, bulk density, minimum outlet size, chute size, material drawdown tests, and conveyability tests
2019	Outotec (Lima)	1 sample of tailings slurry taken from Corani tails (presumed use of the same sample for all Outotec 2019 test-work, provided by SGS Perú)	Thickening tests using a lab-scale dynamic sedimentation process; Lab-scale filtration test; Semi-pilot-scale thickening tests using a various thickener types and reagent dosages
2019	Golder Associates Perú, S.A.	2 pails of tailings slurry (Relave) from the Corani project	Health and Safety tests including HCN, H2S, and Volatile Organic Compound (VOC) analysis, Particle Size analysis, specific gravity, Rheology (static and dynamic yield stresses and viscosity)

Several processing options have been explored for this deposit including direct cyanide leaching for silver recovery and both, bulk and selective flotation systems. Test work on the deposit dates back to 2006 and continues through 2019. The performance of the ore is predominantly dependent on the mineralogy and grade of the material. Oxide and transitional materials showed poorer flotation performance when tested in a sulfide flotation regime. Further, the intergrown fine grained nature of the minerals impacts the ability to selectively recovery lead and zinc into their respective concentrates. This tends to influence the ultimate concentrate grade as well.

A significant quantity of comminution test work has been conducted on a wide distribution of samples. The ore ranges from soft to hard depending on the mineralized zone examined. Similarly, the ore ranges from non-abrasive to moderately abrasive.

Despite the mineralogy complications, the overall performance of the selective flotation system can be viewed as reasonably good and typical for this type of deposit. Most recently additional test work has been undertaken to validate the performance of the proposed flotation flowsheet and geometallurgical predictions as well as to further investigate concentrate dewatering and tailings treatment.

The flotation recovery has been estimated for each of the final concentrates using formulas that examine the grade, mineralogy, elevation, metal tenor and other variables. The formula developed for lead and silver recovery into the lead concentrate is mainly a function of the mineralogy; either transition or non-transition zones (sulfide) and other measurable attributes. A single formula for zinc recovery into the zinc concentrate was developed that is independent of the deposit zone. The deportment of the silver to the zinc concentrate is a function of the recovery of the silver to the lead concentrate.

The geometallurgical factors considered in these formulae are related to the sulfide, transition, and oxide mineralization. These zones are not discrete but are gradational. The sulfide indicator minerals include galena (PbS), pyrite (FeS_2), chalcopyrite ($CuFeS_2$) and zinc mineralization (predominantly sphalerite (ZnS)). Indicators of oxidation include manganese oxide (MnO), goethite (FeO(OH)) and elevation. The indicators for transition mineralization are areas near surface with low zinc and higher silver grades.

The zinc flotation recovery follows a traditional grade recovery curve shown in Figure 13-1.

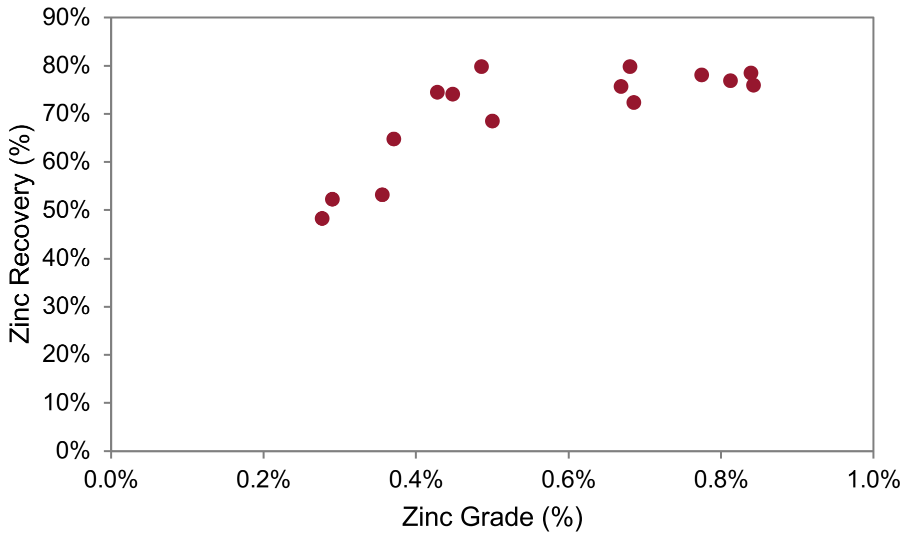


Figure 13-1 Zinc flotation recovery versus zinc feed grade based on geomet modelling

The lead flotation recovery does not produce a traditional grade recovery curve because of the influence of oxide lead minerals that don't respond well to sulfide flotation as shown in Figure 13-2. The geomet equations for both lead and zinc recovery consider the oxidation state of the minerals in the calculation of expected recovery. The lead minerals in the Corani deposit appear to have been subjected to a greater degree of oxidation and thus the flotation response is less predictable and requires more than a simple grade-recovery model.

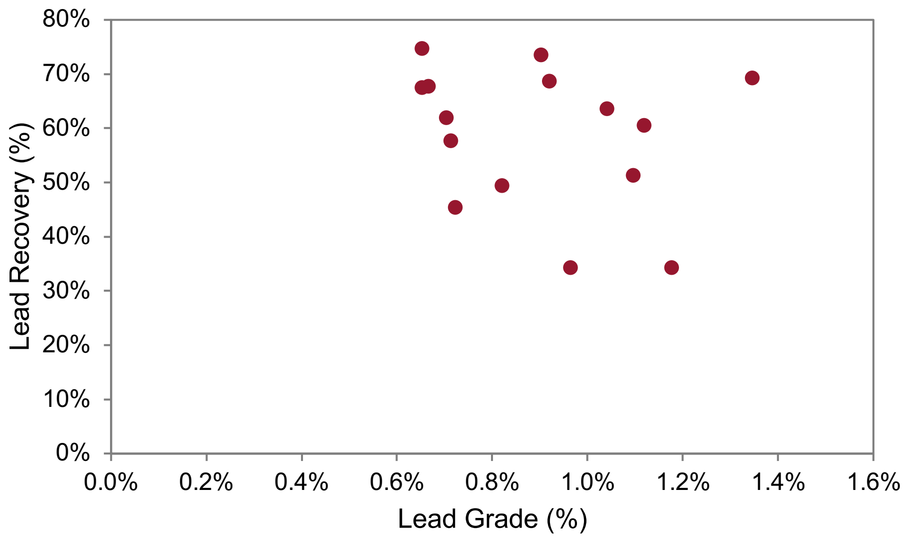


Figure 13-2 Lead flotation recovery versus lead feed grade based on geomet modelling

Silver behaves in a similar fashion to lead as it is mainly associated with the lead sulfide minerals and does not produce a traditional grade-recovery flotation curve. However, the lead-silver association is not absolute as the 2018 / 2019 test-work demonstrated good silver recovery even when lead performance was relatively poor.

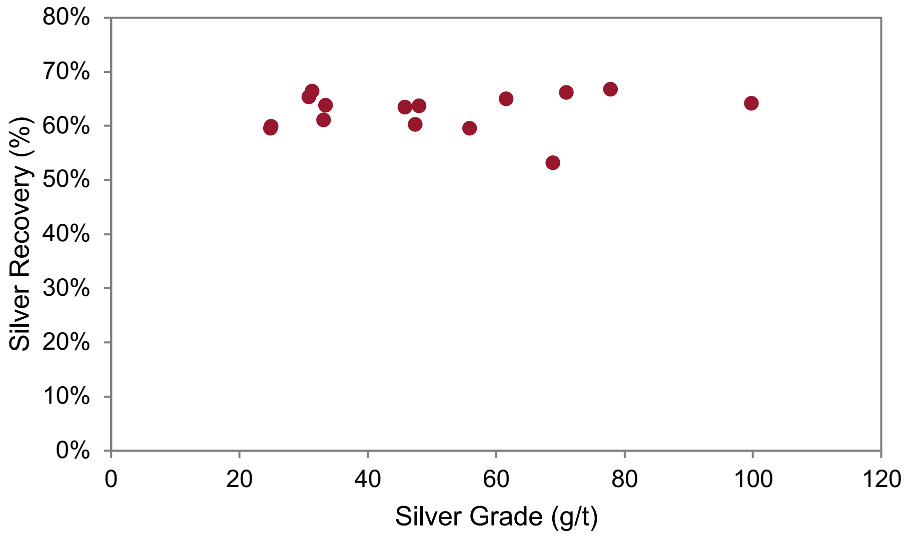


Figure 13-3 Silver flotation recovery versus silver feed grade based on geometallurgical modelling

The details of the metallurgical testing and geometallurgical modelling are found in the balance of this section.

13.2 Samples

A wide range of metallurgical samples have been taken for the Corani project. These samples, mainly from drill core, have been grouped and classified based on their geological domains. In the final analysis it was found that recovery was best predicted by mineralogy, physical parameters (grade, elevation etc). The grouping proved very useful in the early evaluation of the metallurgical response of the various materials.

- Type I
  - 1 – CS – Coarse sulfide
  - 2 – CSC – Coarse sulfide and celadonite
  - 3 – FBS – Fine black sulfide
  - 8 – PM – Pyrite marcasite
  - 10 – TET – Tetrahedrite.
- Type II
  - 4 – FeOX – Iron oxide
  - 6 – MnO – Manganese oxide
  - 7 – PG – Plumbogummite
- Type III
  - 9 – QSB – Crystalline quartz sulfide barite.

These groups were created based on the relative locations of the mineral domains to each other. This created 9 composite groups for the 8 grade labels. The relative proportion of each ore type is shown in Figure 13-4. Originally, the Corani deposit was classified into Type I and II Ag-Pb-Zn minerals. Type III representing a transition Ag-Pb zone. Type I materials responded well to flotation, Type II being slightly more difficult to recover and separate by flotation and Type III as a transitional material had mixed performance but still potentially treatable. The view of the Corani deposit has changed moderately with Type I/II being considered a mixed sulfide Ag-Pb-Zn and Type III still considered a transitional Ag-Pb material. A summary of the new ore type classification is shown below:

Type I: Good Pb/Zn selectivity by conventional flotation, high Pb, Zn, Ag recoveries.

Type II: More challenging Pb/Zn selectivity by conventional flotation, high Zn and Ag recoveries, and moderately high Pb recoveries.

Type III: Moderately high Ag recoveries, low Pb recoveries. Zinc grades for the most part too low to warrant a separate zinc float.

Type IV: Poor flotation of all metals.

GRE has attempted to present the metallurgical data by ore type classifications to provide a better understanding of how the test results relate to the overall project performance. As is typical with many metallurgical evaluations, composites were used extensively for test work. Although composites can provide the required performance metrics, their formation is often done at a time when the required information about how the ore will be mined and processed is not fully validated. The net result is the development of composites that may or may not represent the actual processing sequence. The preferred approach is to test individual ore types. GRE has attempted to extract the ore type information from all testing wherever possible.

As shown from Figure 13-4 the two main ore types are FCB and PM, comprising approximately 79% of the measured, indicated and inferred reserve tonnage and metal content.

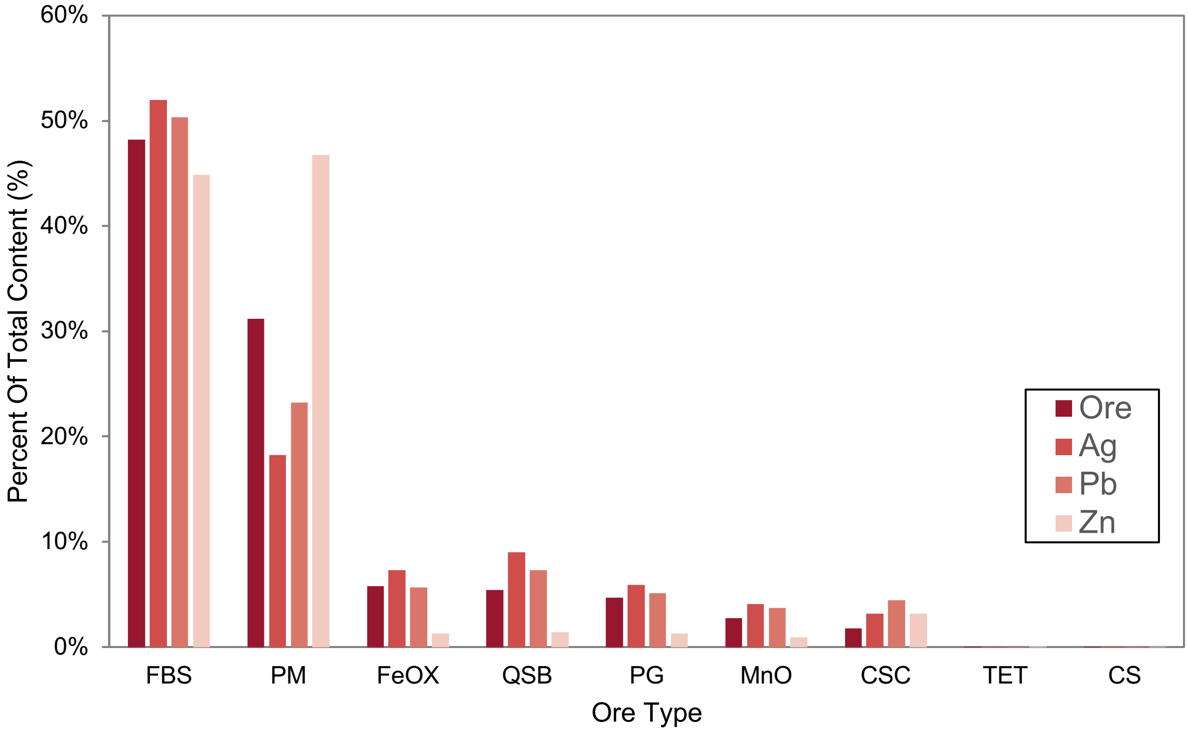


Figure 13-4 Distribution by ore type – tonnage and metal

A description of the geological classification used by BCM to delineate the samples is as follows:

- FBS – Fine-grained black silica-sulfides characterized by very fine-grained mineralogy deposited from quenched ore fluids with highly variable metal content
- PM – Pyrite-marcasite +- quartz typical of early-stage mineralization with little polymetallic mineralization
- FeOx – Iron-oxide mineralization with locally elevated Ag and generally low Pb-Zn. This is a gradation zone with mixtures of FeOx and FBS
- QSB - Crystalline quartz-sulfide-barite interpreted as early fault fill or late-stage breccia fill
- PG – Plumbogummite, identified as a pale-green, waxy, Pb-phosphate mineral that in metallurgical test results showed diminished lead flotation and difficulties in separation of base metals
- MnO – Manganese-oxide mineralization hosting mainly Ag with lesser Pb-Zn
- CSC - Coarse-grained silica-sulfide-celadonite characterized by readily discernible sulfides (galena-sphalerite-chalcopyrite+-tetrahedrite) with celadonite in crystalline to locally opaline quartz
- TET – Ag-bearing tetrahedrite characterized by recognizable late-stage, coarse-grained tetrahedrite cutting earlier sulfides and displaying the highest Ag contents: normally ores with low Pb-Zn contents
- CS - A subset of CSC that contains coarse galena-sphalerite-chalcopyrite +- tetrahedrite without green celadonite clay.

A total of 184 samples of various ore types have been tested and a summary of the geological classification is provided in Table 13-2. FBS and PM are the main geological lithologies by tonnage and metal content, and thus, these domains dominated the testing.

Table 13-2 Number of samples tested by ore type

Ore Type	Dawson 2006	G&T 2007	SGS 2007/2008	SGS 2009	SGS 2011	Base Metal 2018/2019	Total
FBS	2	34	17	-	11	5	69
PM	-	18	6	-	4	1	29
QSB	-	10	2	-	6	-	18
MnO	-	12	2	-	1	-	15
PG	-	14	-	-	1	-	15
Mixed	5	-	-	2	8	-	15
FeOx	1	9	1	-	2	-	13
CSC	2	6	2	-	-	-	10

Notes: a) Samples that contained two ore types were categorized into the dominant ore type for that sample.
b) Samples from multiple holes that were combined are referred to as ‘mixed’.

During the 2018 program a total of six composites were tested from the Este pit representing the first 3 years of production. Of those, BCM identified six tests which were incorporated into the geometallurgical model. Those tests included four primary sulfide materials, (composites 1, 2, 5 and 6) and two transitional materials, (composites 3 and 4). The 2019 program included six additional tests on samples representing ore from the Minas and Main pits which were tested to optimize lead rougher recovery and zinc rougher recovery in lead zinc locked cycle testing (LCT). In addition, two transitional composites were also used for locked-cycle testing to an ultimate concentrate.

Prior testing conducted across the Corani deposit included 72 discrete sections of drill core that were batch tested using a sequential lead-zinc flotation flowsheet. Closed circuit metallurgical performance was determined via completion of locked-cycle tests on a total of 38 samples. Bulk flotation response, evaluated via batch flotation tests, was determined for an additional ± 173 samples. Notably, for a portion of these samples, zinc content was low and only a lead concentrate was produced. The response of 46 of the samples to cyanidation leaching was tested on whole ore. In a single test, cyanidation of a flotation tailing was evaluated. Most of the samples tested in the flotation evaluations, bulk and sequential, were subjected to mineralogical assessment, primarily using QEMSCAN. A summary of tests conducted by year and testing facility is summarized in Table 13-3 .

Table 13-3 Number of discrete samples for metallurgical testing

Test Type	Dawson 2006	G&T 2007	SGS 2007/2008	SGS 2009	SGS 2010	SGS 2011	SGS 2012	BML 2018/2019	Total
Flotation - Selective	5	6	31	-	-	25	103	103	170
Flotation - Locked Cycle	1	-	12	2	-	2	22	22	39
Flotation - Bulk	8	71	-	-	-	-	-	-	79
Whole Ore Leaching	12	32	2	-	-	-	-	-	46
Comminution	-	-	10	-	6	-	20	6	42

In total, 42 samples were subjected to a variety of comminution tests. A summary of the number test conducted is shown in Table 13-4.

Table 13-4: Number of comminution tests conducted

Test	SGS 2008	SGS 2010	SGS 2012	BML 2018/2019	Total
SPI	10	6	20	6	42
Bond ball	10	6	20	6	42
Bond abrasion	2	6	20	6	34
SMC test	-	6	20	6	32
Bond rod	-	6	20	6	32
LEIT	-	6	20	5	31
PLT	-	-	17	6	23
UCS	-	-	13	6	19

The 42 comminution samples consist of 15 originating from the Este deposit, 18 from the Minas deposit and 9 originating from the Main deposit area.

The location of the metallurgical samples is shown in Figure 13-5.

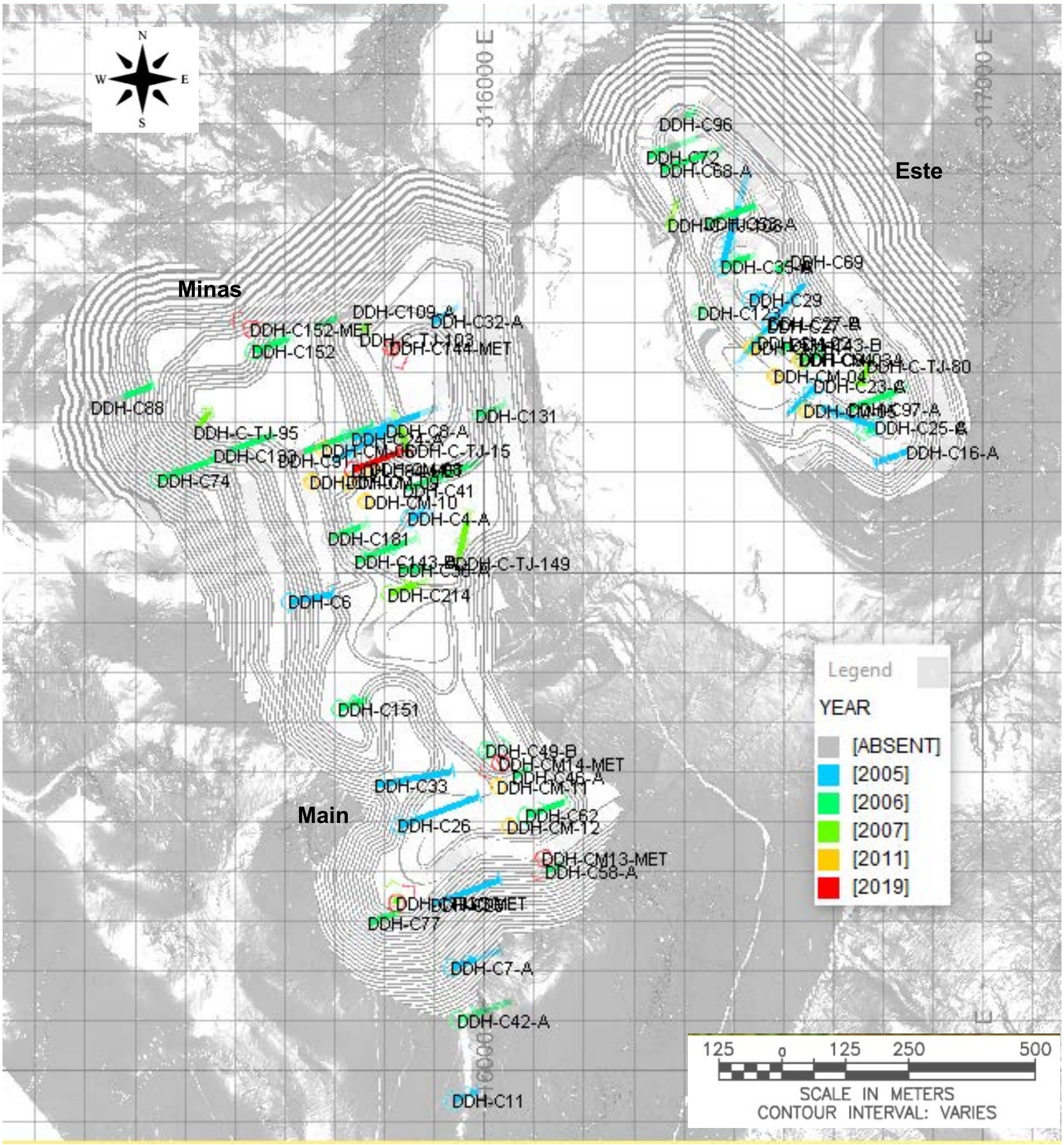


Figure 13-5: Location of drill holes from which metallurgical test samples were sourced (GRE, 2019)

13.3 Mineralogy

A variety of mineralogical investigations have been conducted on the Corani deposit. The results of the investigations are very similar and only the highlights have been presented in this report.

SGS Vancouver Metallurgy (2009) conducted detailed mineralogy on several composites; one representing Type I/II mixed sulfide mineralogy and one representing the Type III transitional material. The modal mineralogical composition of the Type I/II composite shows very little oxide or complex sulphate minerals with the lead and zinc primarily occurring as galena and sphalerite. Conversely, the Type III mineralogical composition showed a significant increase in the presence of oxide and sulphate minerals such as plumbogummite. The primary sulfide impurity was pyrite in both cases.

The work also provided information on the liberation size of the various sulfide minerals. Both galena and sphalerite in the Type I/II composite achieve approximately 70% liberation at 10 µm as shown in Figure 13-6. The Type III ore exhibited similar characteristics with a slightly lower sulfide liberation percentage for a given size when compared to the Type I/II materials. The gangue minerals are primarily quartz, muscovite/clays, kaolinites, k-feldspar and a series of oxides of iron, titanium and manganese. Oxidation products of lead sulfide are also present.

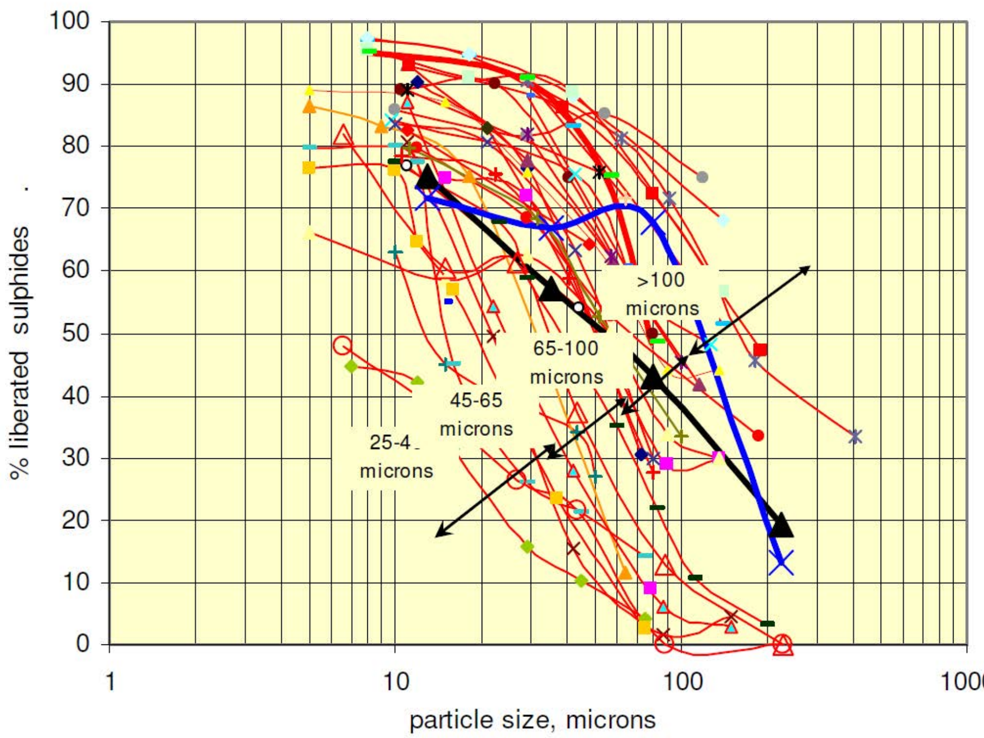


Figure 13-6 Typical ore type i/ii mineral liberation. notes: blue = sphalerite, black = galena, red = SGS database

The relationship between mineralogical characteristics and metallurgical response is reinforced in Figure 13-8 through Figure 13-9. These figures, from the 2008 SGS Report, correlate mineralogical parameters to bulk flotation response. SGS related QEMSCAN mineralogical data against the flotation performance data generated by G&T in 2007.

The flotation test work had indicated that lead speciation is a key driver in the lead recovery and the comparison work conducted by SGS reinforces this observation. Figure 13-7 displays the correlation between lead as galena to the lead recovery to the bulk concentrate. The relationship is reasonably predictable. The flotation response was also compared to the galena grains size. The finer the grain size the more difficult liberation becomes as shown in Figure 13-8. Note the fine grain size of the galena, measuring less than 17 µm in all tests reviewed.

These results indicate that lead recovery can be reasonably well predicted by QEMSCAN modal data through the estimations of the proportion of lead minerals as galena and the grain size as a function of lead mineralogy. A methodology similar to this has been employed in the geometallurgical modelling of the Corani deposit discussed below.

A similar relationship exists for zinc recovery to the bulk concentrate. Mineral grain size is the primary governing property since the zinc mineral in the Corani deposit is primarily sphalerite as shown in Figure 13-9. The sphalerite grain sizes were similarly fine to that of galena. This fine grain size will require a fine grind size to achieve liberation through either primary grinding or a combination of primary and secondary size reduction.

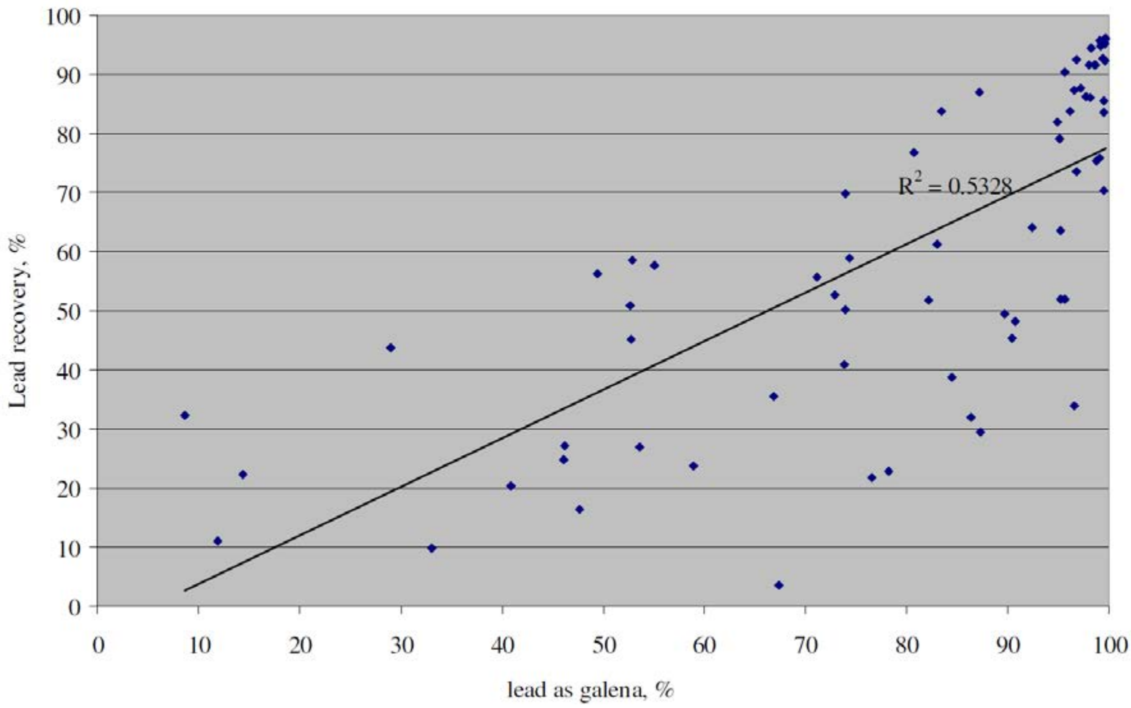


Figure 13-7: Relationship between lead recovery and deportment of lead to galena (SGS Vancouver Metallurgy, 2008a)

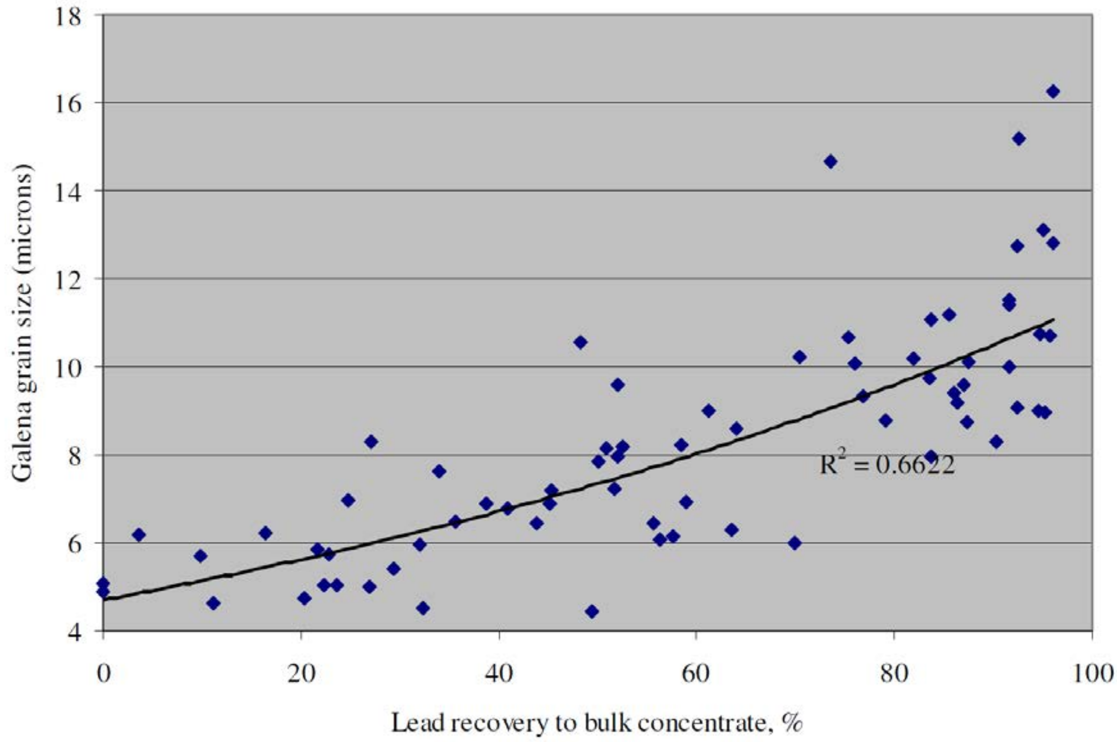


Figure 13-8: Lead recovery and galena grain size (SGS Vancouver Metallurgy, 2008a)

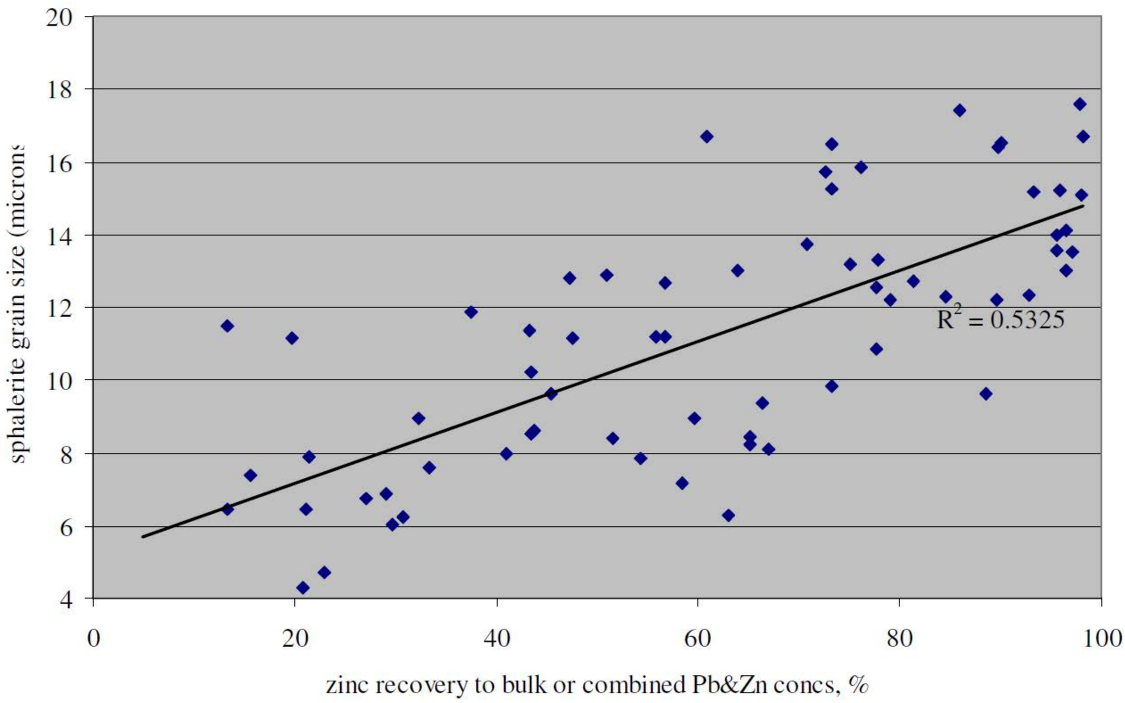
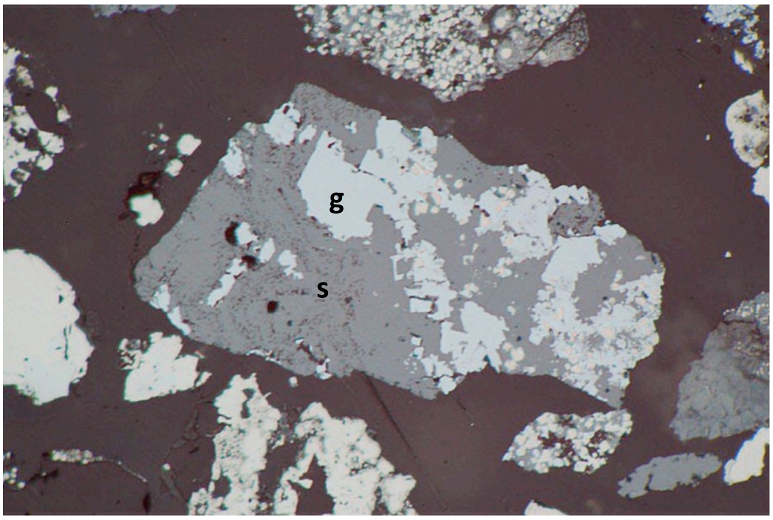


Figure 13-9: Relationship between zinc recovery and sphalerite grain size (SGS Vancouver Metallurgy, 2008a)

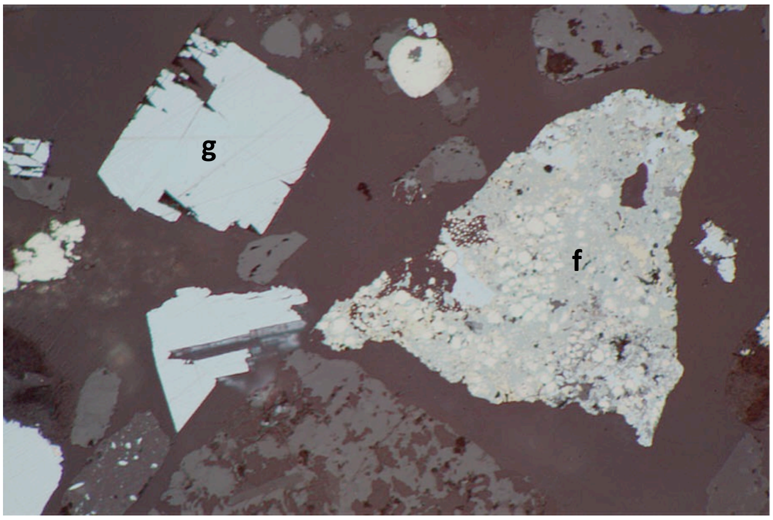
Silver mineralogy was investigated by Hazen Research (2006) on samples tested by Dawson in 2006. In 2008 SGS also examined the silver mineralogy. The chief sulfides in the samples were pyrite–marcasite (grouped together due to frequent close association), boulangerite (a lead antimony sulfosalt), which is particularly abundant, sphalerite, and galena. Present in minor amounts is argentian tetrahedrite, referred to as freibergite, which appears to be the main source of silver. Other, probably minor, sources of silver are acanthite and one or more members of the lead–silver sulfosalt group of minerals, such as andorite, diaphorite, etc. Boulangerite and galena do not appear to be a source of silver. The freibergite is frequently associated with poorly crystallized pyrite–marcasite and varies from less than 5 to about 65 µm, with an average size of 30 µm or less. The acanthite detected during microprobe analysis is very fine.

Photomicrographs and back-scatter images from a Scanning Electron Microscope (SEM) were taken by Hazen Research and confirmed this complex, fine-grained mineralogy. However, coarse galena grains were also noted in some samples. An example showing both complex and coarse liberated particles from Hazen Research on the Dawson composites is shown in Figure 13-10, a photomicrograph showing a relatively simple intergrowth of sphalerite (s) and galena (g).



= 100 µm 300 ×

Figure 13-10: Dawson composite B, minus 35-mesh sink (Hazen 2006)



= 100 µm 300 ×

Figure 13-11: Dawson composite D, minus 35-mesh sink (Hazen 2006)

Figure 13-11 is a photomicrograph of a composite from Dawson showing freibergite (f) intergrown with pyrite–marcasite (yellow) and coarse, liberated galena (g) cleavage fragments.

13.4 Grindability Tests

A summary of the grindability results is provided in Table 13-5. The number of samples tested per zone is also provided. The following tests were conducted on the Corani samples:

- Unconfined Compressive Strength, (UCS)
- Point Load Test, (PLT)
- Low Energy Impact Test, (LEIT)
- Drop Weight Index (DWI)
- SAG Mill Comminution, (SMC)
- SAG Power Index, (SPI)
- Crusher Work Index (Wi_C)
- Bond Rod Mill Work Index, WiRM
- Bond Ball Mill Work Index, WiBM P80 75 microns
- Abrasion Index (Ai)

Table 13-5: Comminution test results summary

	UCS, MPa	PLT, MPa	LEIT, kWh/t	Wi_D, kWh/m³	Axb	SPI, min	Wi_C, kWh/t	WiRM, kWh/t	WiBM, kWh/t	Ai
No. Tests Total	19	22	31	32	32	42	42	32	42	34
No. Este	6	8	13	13	13	15	15	13	15	14
No. Minas	10	9	11	12	12	18	18	12	18	12
No. Main	3	5	7	7	7	9	9	7	9	8
Mean	32.4	1.5	6.2	2.4	112	36.0	24.6	10.4	14.9	0.16
Max	58.6	2.9	10.2	4.0	221	90.6	49.8	13.4	19.2	0.49
Min	16.8	0.7	2.7	1.1	60.8	15.3	6.5	6.5	10.1	0.024

Note: The number after BBWI refers to the closing screen size used for the test, in µm.

The results shown above are categorized by deposit area, and the work was conducted on composite samples from these areas. Ideally this test work would have been conducted on individual ore types. It is difficult to comment on the behaviour of individual ore types given this data set. Efforts have been taken to try to delineate the results based on mineralogy and mineral group class, but in most cases, this was not possible based on the reported data.

A wide range of results were observed, particularly within the UCS, LEIT, and SPI-related indices. A statistical analysis of the data was performed, and outliers in the data were identified. A result was deemed an "outlier" if it fell outside the range of 2.58 standard deviations of the mean. In all cases, the results were skewed higher with the inclusion of outliers. For this report, all data have been included.

Based on the above information, the ore appears to be of medium hardness with respect to SAG and ball milling with an average Bond rod mill work index (Wi_{RM}) of 10.4 kWh/t and a Bond ball-mill work index (Wi_{BM}) of 14.9 kWh/t.

Given below in Table 13-6 are the results of grindability studies for each deposit area.

Table 13-6: Comminution Test Results Summary by Pit Area

	UCS, MPa	PLT, MPa	LEIT, kWh/t	Wi_D, kWh/m^3	Axb	SPI, min	Wi_C, kWh/t	WiBM, kWh/t	WiBM, kWh/t	Ai
Main										
Mean	36.9	1.5	6.6	2.3	116	32.7	23.5	9.4	14.6	0.15
Max	52.7	2.5	10.2	4.0	221	62.2	48.3	13.4	18.6	0.43
Min	26.4	0.7	4.1	1.1	60.8	19.1	14.1	6.5	12.1	0.031
Minas										
Mean	31.3	1.6	6.2	2.5	98.9	40.9	25.6	11.6	15.1	0.14
Max	58.6	2.9	8.4	3.4	160	90.6	49.8	13.0	19.2	0.34
Min	16.8	0.8	3.8	1.5	66.9	21.1	10.5	9.3	10.1	0.024
Este										
Mean	27.1	1.5	5.2	2.1	126	31.7	24.4	10.1	14.9	0.17
Max	29.1	2.7	9.1	3.7	185	55.6	43.4	11.7	16.8	0.49
Min	23.1	0.8	2.7	1.3	67.2	15.3	6.5	9.0	13.2	0.03

Note: The number after BBWI refers to the closing screen size used for the test, in µm.

The results above show that the Este and Minas deposits have the widest range of material hardness. The data from these deposits have both the global maximum and minimum values for the complete data set. For example, the Este deposit has the most extreme values for DWI, Axb, and Wi_{RM}, whereas the Minas deposit has the most extreme data for UCS and Wi_{BM}.

Examining the mean values for the deposits shows that the Minas has the hardest material, whereas Main has the softest material. However, the main deposit has the highest average abrasion index. The Minas deposit has the lowest average abrasion index. The Bond rod mill and ball mill work indices are in fairly close agreement with each other throughout all zones.

Box and whisker plots have been developed for the SPI and Wi_{BM} for the three main deposits. These plots show the maximum and minimum values, as well as the median (the average is also often shown with an x), first and third quartile boundaries (the box) and outliers.

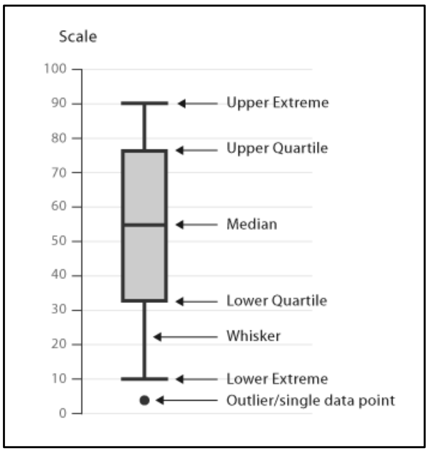


Figure 13-12: Box and whisker plot explanation

Figure 13-13 and Figure 13-14 show the variation across the three main deposits for both the SPI and WiBM, respectively.

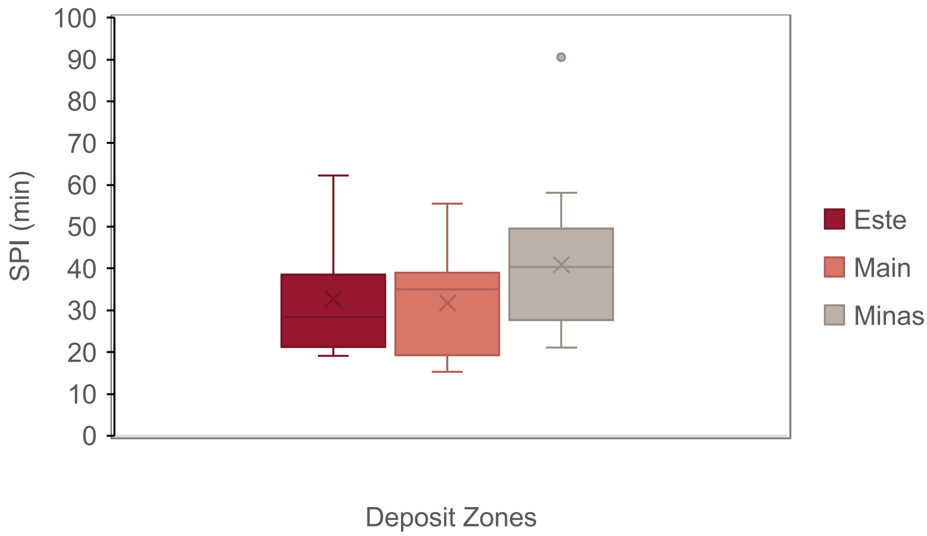


Figure 13-13: Box and whisker plot - SPI by deposit

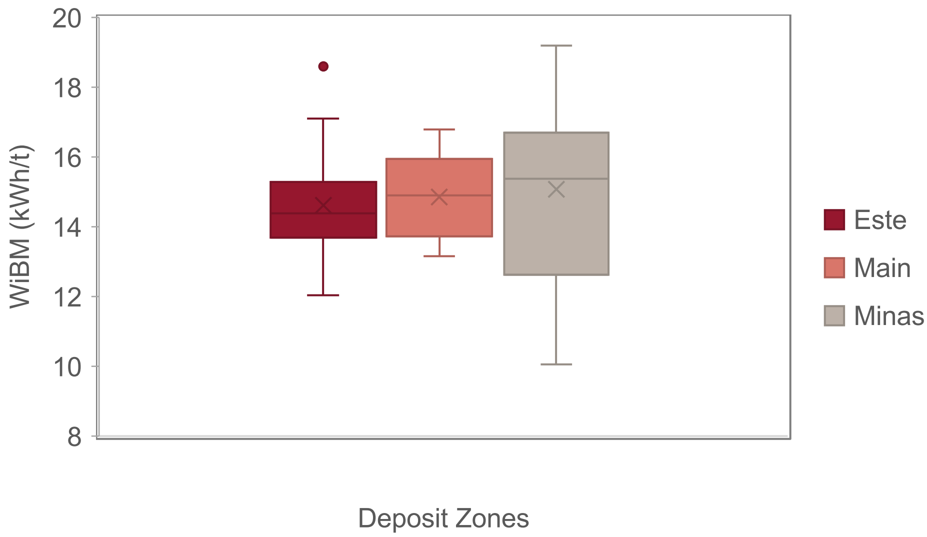


Figure 13-14: Box and whisker plot – WiBM by deposit

Similar variation was found within the other comminution metrics evaluated.

In 2014 Alex G Doll Consulting Ltd. (AGD) was contacted by Bear Creek Mining Corp. to review grindability test work, grinding circuit design, primary crusher selection and to perform grinding circuit modelling of Bear Creek's Corani project in Peru. AGD reviewed two data sets produced by SGS in 2010 and 2012.

SGS 2010 program conducted at SGS Lakefield Research Chile Ltda. in Santiago, Chile. This program consisted of six samples that represent lithology and zone composites. Testing conducted on the samples includes the three Bond work index tests (Wi_{BM}, Wi_{RM}, Wi_{C}), the SMC DWI tests, SPI testing and the Bond abrasion index test. In comparison to AGD's global database of test results, this program is characterized as “soft”. Whole diameter HQ diameter core was used in the grindability work.

SGS 2012 program conducted at SGS Lakefield Research Chile Ltda. in Santiago, Chile. This program consisted of twenty samples that represent discrete intervals of whole-diameter PQ core ranging from 12 m to 70 m in length. Testing conducted on the samples includes the point load index (PLI) geotechnical test, the unconfined compressive strength (UCS) geotechnical test, the three Bond work index tests (Wi_{BM}, Wi_{RM}, Wi_{C}), the SMC drop weight test, SPI testing and the Bond abrasion index test. The 2012 program is generally harder than the 2010 program, but in comparison to AGD's global database of test results, this program is still characterized as “soft” due to the very low Wi_{RM} and Wi_{C} values.

13.5 Flotation

Metallurgical testing has been conducted on numerous composites from the Corani deposit since 2005 by several laboratories. There is a significant amount of flotation test work and the resulting data set is large. Blue Coast Metallurgy was contracted to summarize the testing from 2007 through to 2011. Early work in 2006 by Dawson Metallurgical Laboratories focused on bulk flotation and silver cyanide leaching and is discussed in separate sections.

In total, approximately 130 different composites have been tested in approximately 725 flotation tests. The metallurgical drivers of the flotation response of the Corani minerals and the flowsheets required to achieve acceptable performance took time to develop. As a result of this some of the test work, in particular, early stage work, had little bearing on the ultimate flotation regime selected.

Early flotation test work produced inconsistent results and indicated that the Corani deposit exhibited significant variability, ranging from ores with good results to others yielding poor flotation response. The test programs evolved into two general forms; variability programs where a number of composite samples were subjected to a standard, although non-optimized, flowsheet to assess this variability, and flowsheet optimization conducted on a single composite typically over the course of about 50 tests. The two types of program often ran consecutively with flowsheet developments made on a given composite applied in the following round of variability work. The flowsheets employed in early work have evolved to the more optimal current flotation regimes which generally produce good flotation results on Type I/II minerals. The early test work revealed the nature of the mineralogy, liberation sizes and the flotation responses of each mineral assemblage. Table 13-7 shows a summary of the flotation test work conducted.

Table 13-7: Flotation test work summary

Year Published	Testing Lab	Samples Used	Description
2006	Dawson	13 composites: coarse assay reject and whole core; oxide, sulfide, and mixed ore type	Bulk sulfide flotation tests with various reagents following lead rougher and scavenger flotation; selective flotation tests performed on six composites with subsequent zinc flotation on four of the tested composites with high zinc content. Combined flotation and leaching tests performed on two composites, including whole-ore cyanidation and flotation tails leaching.
2006	G&T	1 composite prepared from 12 core samples from one mineral zone	Rougher tests producing bulk (lead) and zinc concentrates using various reagents; cleaner flotation tests to produce selective concentrates.
2007	G&T	71 drill core samples from the three pit zones (27 Este, 19 Main, 25 Minas)	Rougher flotation and bulk cleaner flotation tests performed on all samples; sequential flotation tests performed on selected samples containing sufficient sphalerite and galena content; cyanide leaching tests performed on selected samples.
2007	SGS Lakefield / Vancouver	7 composites created from drill core samples: A, B, B2, C, D, E, F	Bulk sulfide flotation and sequential flotation performed on composite D; selective lead flotation and bulk lead/zinc flotation to produce a high-grade bulk concentrate from composite B, with further lead/zinc separation tests on the bulk concentrates produced from bulk flotation tests; bulk sulfide flotation on composite A, C, E, and F.
2008	SGS Lakefield / Vancouver	27 composites created from drill core samples: 15 alphabetical (G – U) and 10 according to source area (Main, Minas, and Este)	Numerous batch flotation tests, lead rougher tests, lead cleaner tests, locked cycle tests, zinc cleaner tests, and sequential flotation tests performed on samples listed. Extensive testing performed on composites G, H, and K. Analysis of results focused on metallurgical domains and ore type, as well as individual composite results.

Year Published	Testing Lab	Samples Used	Description
2009	SGS Vancouver	2 composites: type I/II Pb-Ag-Zn material, type III Pb-Ag	86 batch flotation tests and four locked cycle tests: 47 rougher tests with various reagents conducted on Type I/II material and 38 tests with Type III material; 18 cleaner tests on Type I/II and 20 tests with Type III, with some tests including a regrind step; two batch flotation tests and one locked-cycle test was conducted on a blended sample of material.
2012	SGS Vancouver	6 composites: 3 Pb-Ag-Zn, 3 Pb-Ag, prepared from 14 variability samples out of 25 received; 2 master composites created from 12 received variability samples; one Pb-Ag-Zn composite created for flotation and third-party testing on flotation products	Batch flotation tests performed on variability samples and composites created; lead rougher, lead cleaner, zinc rougher, zinc cleaner tests; locked-cycle tests performed on master composites and bulk flotation composites, including sequential rougher flotation for lead/zinc separation, concentrate regrinding, and cleaning; bulk flotation performed on Pb-Ag-Zn composite created for external testing;
2018 / 2019	Base Met Labs	6 composites created for test-work done in 2018: 1, 2, 3, 4, 5, and 6; up to 6 more composites created for 2019 test-work, but only 6 reported on in the latest report	Extensive flotation testing, split between two reports so far; lead rougher and cleaner tests performed on Pb-Ag concentrates 3 and 4, lead rougher, lead cleaner, zinc rougher, and zinc cleaner tests performed on Pb-Ag-Zn composites 1, 2, 5, and 6; locked-cycle tests performed on all concentrates, with Pb-Ag-Zn composites 1, 2, 5, and 6 producing zinc concentrates.

The metallurgical testing on the Corani deposit has covered the majority of the deposit and features all ore zones and geological ore types. While early test work was conducted mainly on Este composites, the three most recent programmes provide good representation of the deposit as a whole, and hence, provide confidence in the flowsheet development exercise under taken (Figure 13-5).

13.5.1 Selective flotation tests

In the G&T Variability 2007 program many of the composites yielded good silver and bulk sulfide recovery but the use of selective lead flotation from zinc (using the standard flowsheet at the time), was not successful. The SGS mineralogical investigation 2007 was successful in identifying ores susceptible to this problem and those where selective flotation could be achieved by typical reagent regimes. Ores classed as Type II, exhibiting lower sulfide contents, more alteration and the presence of plumbogummite and gorceixite, were the ones where Pb-Zn selectivity could not be achieved consistently across the G&T variability testing.

SGS 2007 applied a bulk sulfide float approach on several samples that exhibited poor selectivity with the aim to maximise silver recovery in the event the selectivity problem could not be resolved. A wide range of reagent systems were examined to improve both selective flotation and bulk floatation responses. Ultimately a selective flotation flowsheet was developed that allowed the production of saleable Pb-Ag and Zn concentrates. This flowsheet was applied in the most recent locked-cycle testing conducted in 2018 and 2019.

13.5.2 Primary grind size

The mineralogical investigations conducted on the Corani samples had provided an indication of the required primary grind and also insight into the required regrind size required to achieve liberation. Early analysis dictated that a fine primary grind would be required (2007 and 2008 test work) but test work in 2009 indicated that a primary grind P80 in the 100 µm region would achieve adequate initial liberation (SGS Vancouver Metallurgy, 2009).

SGS Vancouver Metallurgy (2009) analysed the impact of grind on lead and silver recovery and zinc selectivity. Figure 13-15 shows the effect of grind size on Pb and Ag flotation, by means of a rougher mass pull vs. recovery plot. This test work was conducted on a zone 3 Pb-Ag-Zn master composite.

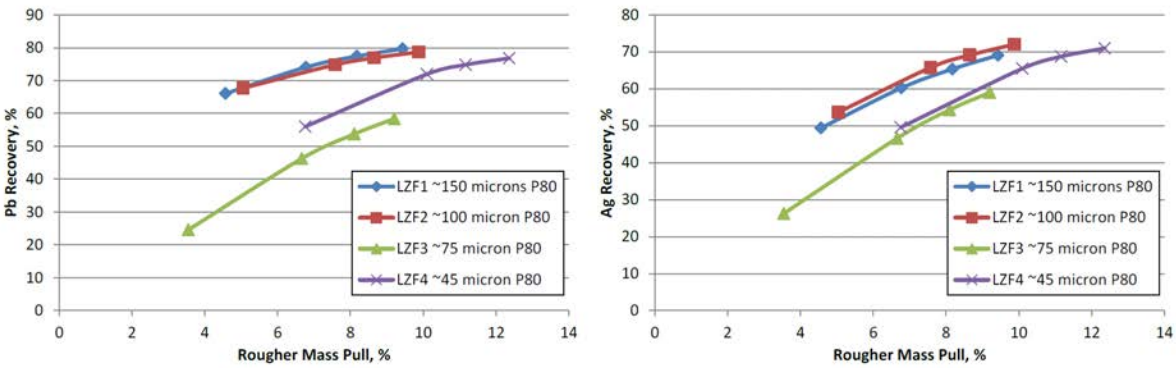


Figure 13-15: Effect of grind size on lead rougher recovery - 2009 (Blue Coast Metallurgy, 2011)

The highest recoveries achieved at the lowest mass pulls were found at the coarser grinds. The excellent result achieved at a P80 of 150 microns suggests that galena liberates somewhat preferentially in grinding. Using the knowledge that previous samples exhibited finer grain sizes than this composite, a grind P80 of ~100 microns was selected for the basis of the subsequent testing on this composite. Other similar type I/II composites exhibited similar results indicating that a grind size coarser than the liberation size identified mineralogically could be utilized. Transitional type ores also showed a similar trend but at a slightly finer upper grind size (P80 74 µm) (Blue Coast Metallurgy, 2011).

13.5.3 Concentrate regrind

There has not been a lot of focus on optimizing the regrind sizes. The quantitative mineralogy suggested that some of the mineralization (FBS) would require very fine primary and rougher regrind targets to achieve acceptable concentrate grades. Most of the testing conducted in 2008-2009 by SGS did only semi-quantitative sizing analysis. The target regrind P80 size for both the lead and zinc regrinds appeared to be between 20 and 30-micron K80 in these programs.

SGS Vancouver Metallurgy (2009) evaluated three grinds – a coarse grind with a P80 size of 50 microns, a medium grind with a P80 size of 35 microns and a fine grind with a P80 size of 20 microns. The silver minerals appeared to produce slimes with finer regrinding (Figure 13-15) while the lead recovery showed a definite improvement with finer regrind P80 sizes (Figure 13-17). Galena is a well know sliming mineral and care must be taken not to overgrind the concentrate making cleaning more difficult.

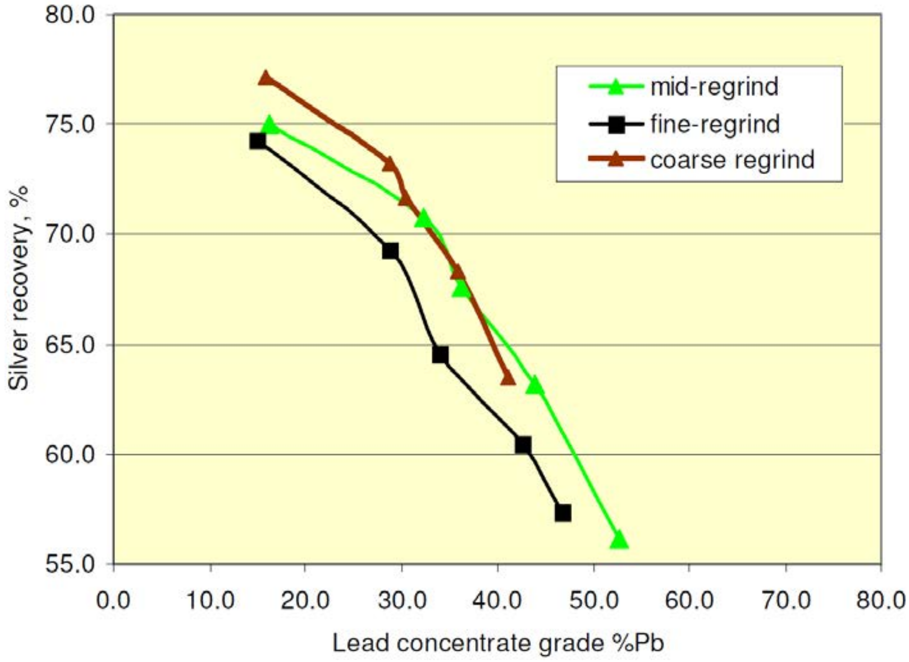


Figure 13-16: Effect of regrind on Ag recovery in lead cleaning (Ag-Pb-Zn composite) (SGS Vancouver Metallurgy, 2009)

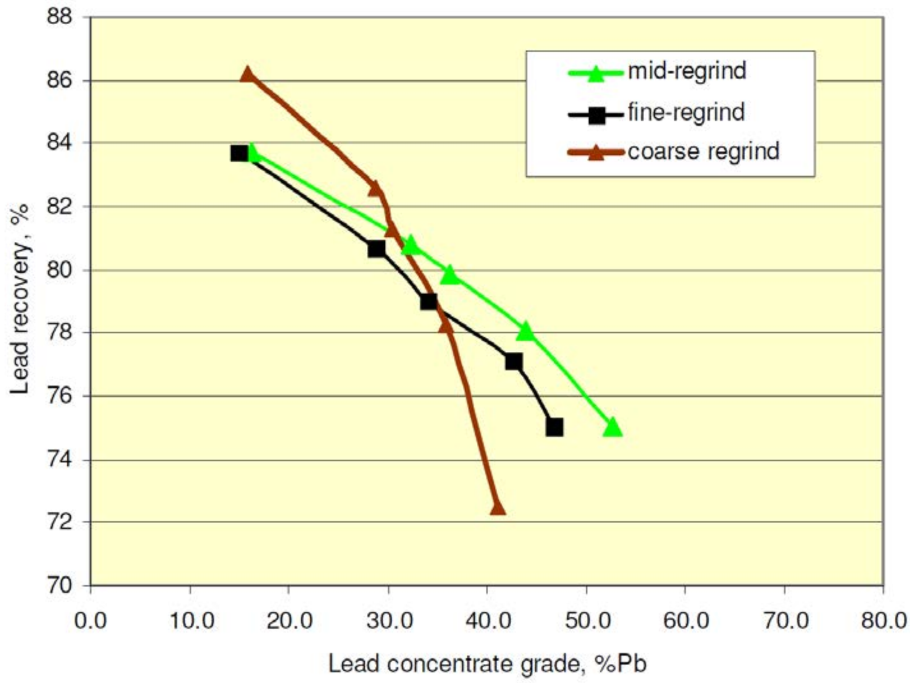


Figure 13-17: Effect of regrind on Pb recovery in lead cleaning (Ag-Pb-Zn composite) (SGS Vancouver Metallurgy, 2009)

The zinc concentrates also showed the optimal regrind to be in the range of a P80 of 25 µm.

13.5.4 Zinc depressants

A challenge with many of the Corani deposit samples was the poor selectivity between lead and zinc in the lead flotation circuit. Most of the testing conducted was devoted to improving the selectivity of the lead flotation circuit against the flotation of sphalerite (zinc).

To tackle the separation of sphalerite, several depression schemes were employed. The depression schemes could be classified into general categories as follows:

- zinc sulphate ($ZnSO_4$) – used in combination with selective collectors
- zinc sulphate/cyanide (NaCN) or zinc oxide/cyanide (ZnO / NaCN) – most common depressant scheme for zinc depression during lead flotation. Very effective to combat copper ion activation of sphalerite and pyrite.
- sulfite/sulfide includes reagents Na_2SO_3, Na_2S, SO_2 – less common, but effective for sphalerite, pyrite depression when cyanide is not permissible. High dosages of these reagents will also depress galena (lead).

The development of depressants used for most of the locked cycle tests occurred while testing of composites A, B, B2, and G during the SGS program in 2008-2009. The most complete matrix of testing was completed on sample G. Results of zinc depression are displayed in Figure 13-18. Unfortunately, there were no comparable tests without depressant but the results would likely be predictably poor.

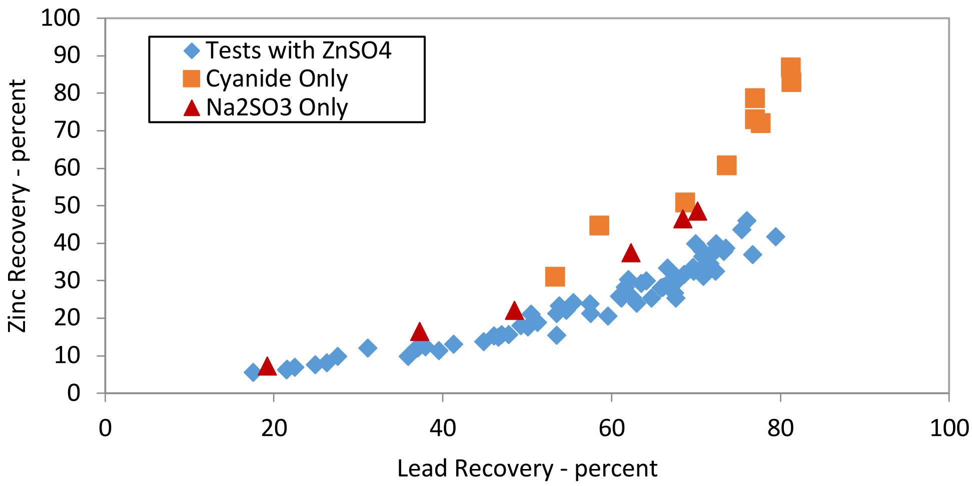


Figure 13-18: Effect of depressants on lead rougher flotation – Composite G (FBS) (SGS Vancouver Metallurgy, 2008a)

The best selectivity was achieved when zinc sulphate ($ZnSO_4$) was added. When cyanide was added without zinc sulphate the activity of zinc was increased. It should be noted that one test was conducted with both zinc sulphate and cyanide and the results mirrored the zinc sulphate only conditions. Sodium sulphite (Na_2SO_3) alone did not appear to be effective for depressing zinc in this sample.

To further investigate the effects of zinc sulphate, there were a number of tests that utilized variable dosages of zinc sulphate (g/t) with the other variables remaining constant. The results of these tests are displayed in Figure 13-19.

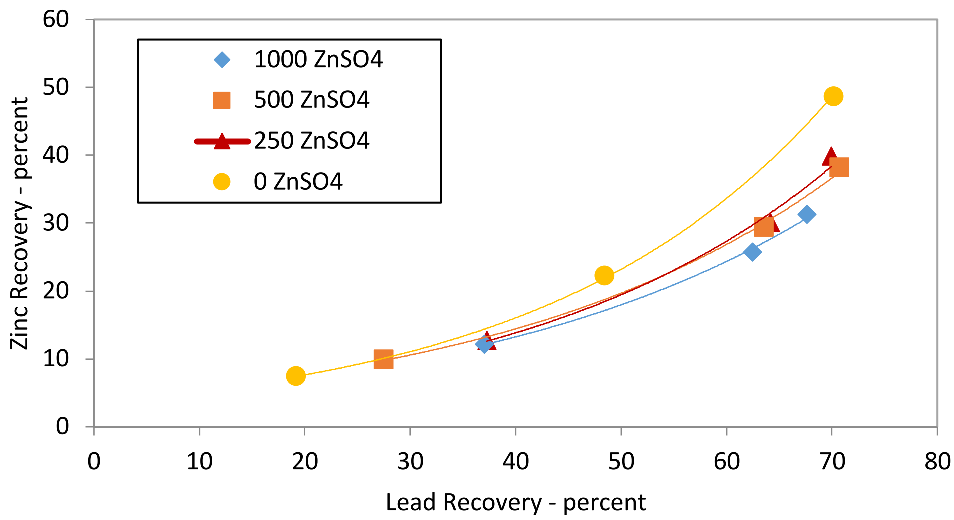


Figure 13-19: The effect of zinc sulphate dosage (g/t) on zinc depression from the lead rougher flotation, Composite G (FBS) (SGS Vancouver Metallurgy, 2008a)

The results indicate that the addition of zinc sulphate improved lead flotation selectivity against sphalerite and there was a slight improvement with increased dosages. The evolution of the flowsheet produced a depression regime using a combination of zinc sulphate, cyanide and sodium sulphite.

The application of emulsified diesel oil (EDO) was also considered (Table 13-8). A breakthrough occurred with the use of EDO in conjunction with ZnSO4/CN depressant dosage. SGS described a process of EDO acting to agglomerate galena fines and promote galena flotation. The effect of EDO is shown in a series of tests evaluating EDO in combination with depressants (Figure 13-20 and Figure 13-21).

Table 13-8: Test conditions for depressant and EDO series –Composite B2 (mixed sulfide) (SGS Vancouver Metallurgy, 2009)

Test No.	47	48	49	50	51	52	53	54
$ZnSO_4$ g/t	0	0	0	0	500	0	1000	0
$ZnSO_4$/NaCN, g/t	500	500	500	500	0	500	0	500
Na_2SO_3, g/t	500	500	500	500	500	500	0	500
EDO, g/t	300	250	0	300	300	300	300	300
Regrind, minutes	0	0	0	5	0	10	0	20

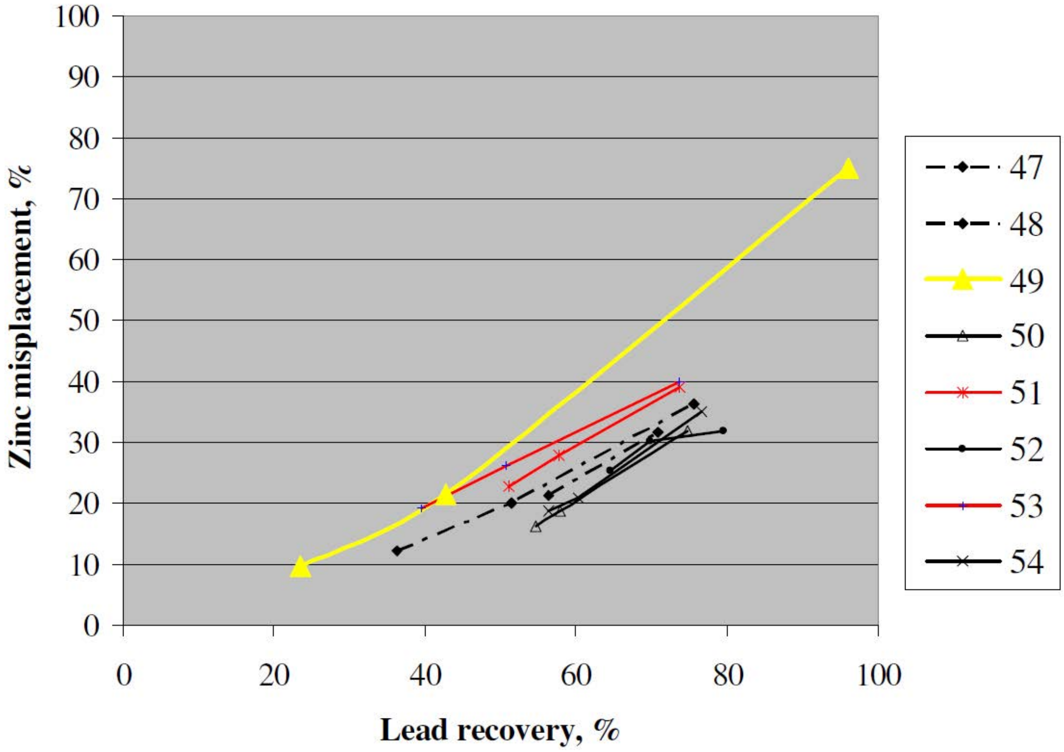


Figure 13-20: Effect of EDO on zinc misplacement in lead flotation – Composite B2 (mixed sulfide) (SGS Vancouver Metallurgy, 2007)

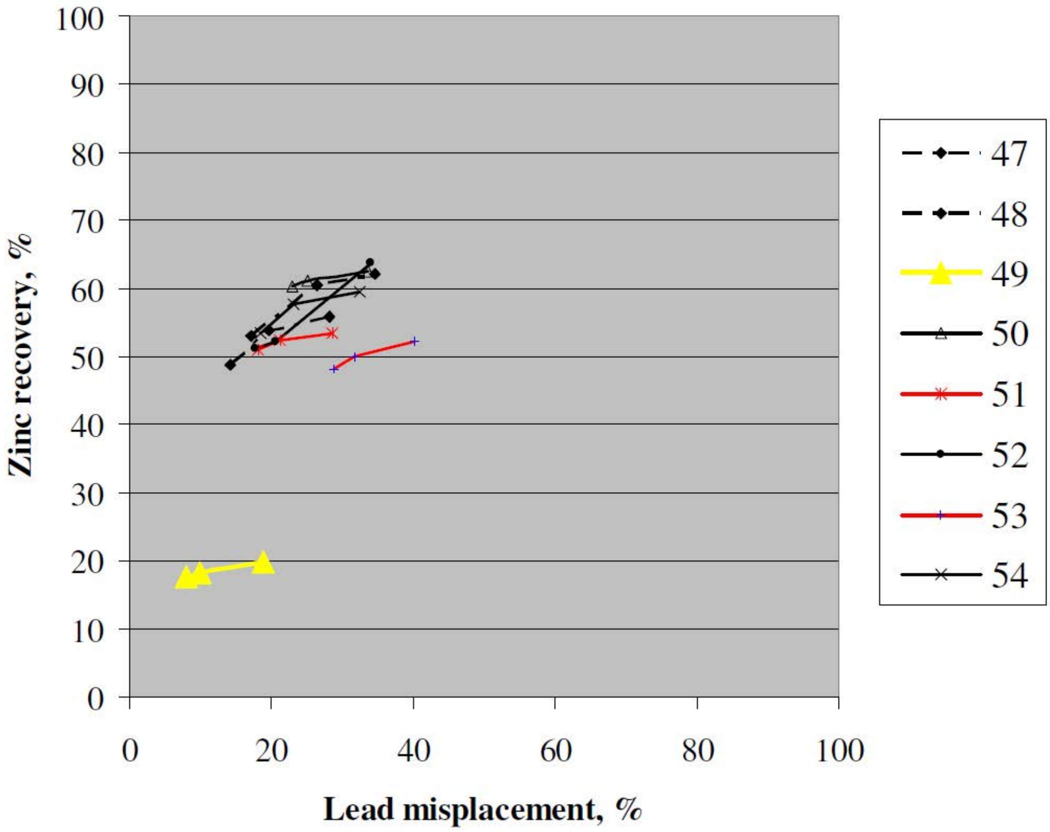


Figure 13-21: Effect of EDO on lead misplacement in zinc flotation – Composite B2 (mixed sulfide) (SGS Vancouver Metallurgy, 2007)

The results of these tests are shown in the following plots (Figure 13-22), showing lead recovery versus zinc misplacement in the Pb circuit, and vice versa in the zinc circuit. The effect of EDO, not present in Test 49 (in yellow) is apparent. In a later series of rougher tests (SGS 2011), the effects of EDO were further evaluated. In this testing the EDO did not appear to impart any gains in selectivity, as shown in Figure 13-22.

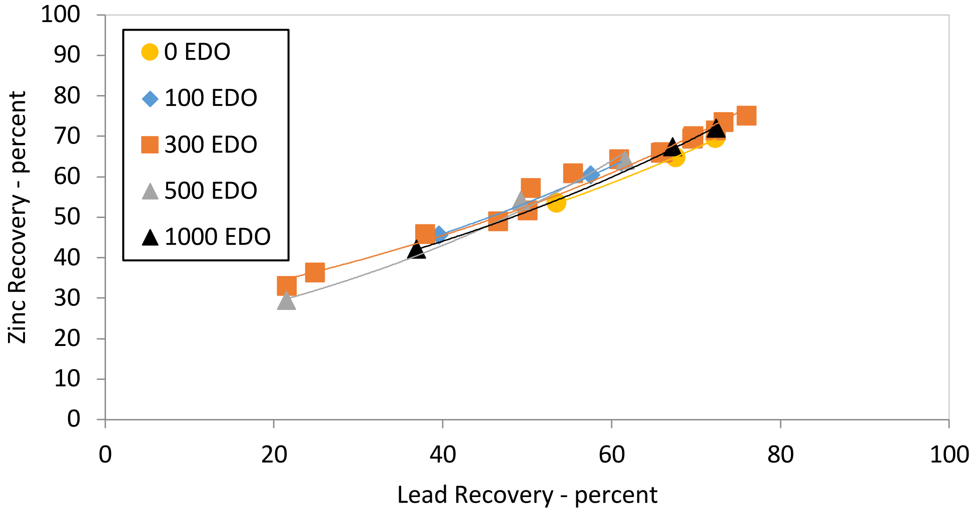


Figure 13-22: The effect of EDO (g/t) on lead rougher flotation selectivity against Zinc, Composite G (FBS) (SGS Vancouver Metallurgy, 2008a)

The role of EDO in the lead flotation circuit is still not fully defined but its use appears to have merit at least for certain ore types.

13.5.5 Collectors

A significant focus of the flowsheet development program was dedicated to promoting selectivity between lead and zinc minerals in the lead flotation circuit. Several collectors were tested including; Cytec products 3418A, 242, and 404. Xanthates were also included in combination with Cytec collectors. Potassium xanthate (strong) and sodium isopropyl xanthate (weaker) were primary xanthate collectors used. Xanthates were also used in the zinc circuit. Figure 13-23 shows the selectivity impacts of different collectors.

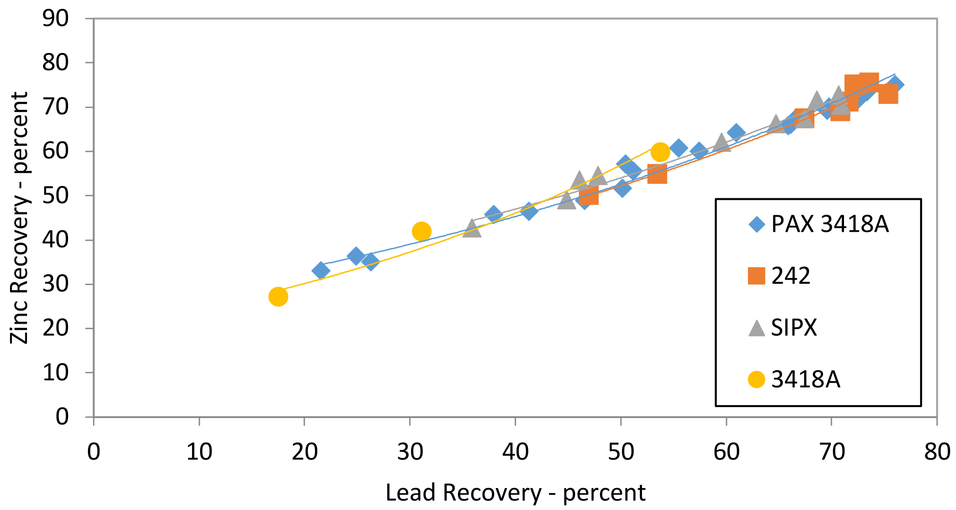


Figure 13-23: The effect of collector type on lead rougher flotation selectivity against zinc, Composite G (FBS) (SGS Vancouver Metallurgy, 2008a)

The test work results indicate the various collectors tested did not have a significant impact on the selectivity between lead and zinc in the lead rougher flotation. Collector dosage did shift the results up the curve (higher dosages) or down the curve (lower dosages).

In other test work there was a strong indication that the collectors played an important role in maintaining selectivity (Figure 13-24). These tests essentially examined SIPX vs. PAX as the primary collector along with the effect of various secondary collectors. Interestingly a step change in performance was found among most of the tests using SIPX compared with those using PAX, therefore, SIPX was selected ahead of PAX as the primary collector. The importance of the secondary collector was somewhat inconclusive, although a marginal benefit was inferred. As the various secondary collectors all gave similar performances, the mercaptonbenzothiozole (MBT) Aero-404 was selected for its known ability to collect silver minerals as a cost-effective alternative to Aerophine 3418A.

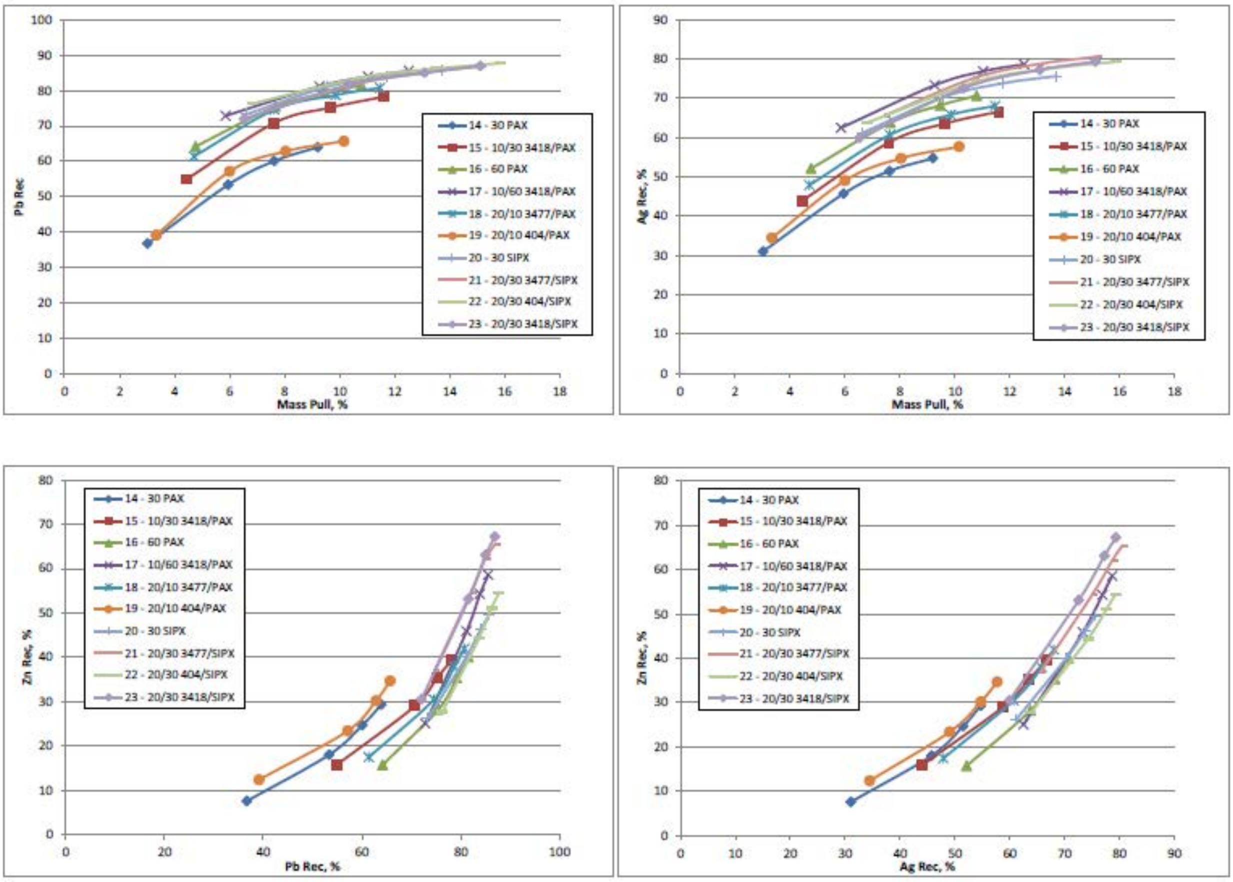


Figure 13-24: Collector optimization results in Pb roughers – 3 Zone Pb-Zn-Ag Composite (Blue Coast Metallurgy, 2011)

13.5.6 Activators

In the lead circuit, activators such as sodium hydrosulfide (NaHS) and hydroximate collectors were briefly tested as a means of activating and recovering non-sulfide lead. The test results were discouraging, indicating that the plumbogummite, the dominant non-sulfide lead mineral, did not respond to these reagents.

In the zinc circuit, activation of zinc was accomplished with copper sulphate ($CuSO_4$). This is standard practice in industry. Dosages of copper sulphate required ranged between 100 and 350 g/t.

13.5.7 pH regulators

The use of pH regulators can be effective in controlling pyrite when used in combination with selective collectors and other depressants. In the lead circuit, controlling the pH to 8 to 9 is common to help depress pyrite and other iron sulfides. Above pH 9, lead depression occurs. The natural pH of many of the samples tested was well below 7; most samples averaged 5.5. Increasing the pH with lime in the lead circuit was ineffective and often resulted in increased activity of sphalerite flotation. Decreasing the pH with sulphuric acid was also investigated but showed no advantage.

In 2011, a global composite (Ag-Pb-Zn) used for detailed flowsheet development indicated issues with pyrite activation as well as zinc activation in the lead flotation circuit. To combat issues of pyrite activation, sodium carbonate ($NaCO_3$) was used effectively to reject pyrite and increase the grade of lead in the concentrate for this composite. Many other composites did not require the use of sodium carbonate to produce high-grade lead concentrates.

For the zinc flotation circuit, the addition of lime to pH 11 or 11.5 was used for most of the composites tested. Base Metallurgical Laboratories (2019) tested the use of soda ash with no real benefit.

13.5.8 Surface modifiers

There were many surface modifiers tested, mostly to improve the quality of the lead concentrates. These included sodium silicate (silicate mineral depressant and dispersant) and starches (cellulose gum and guar gum for silicate depressants). None of these reagents were useful in improving the metallurgical response of the flotation circuits.

13.5.9 Summary of sequential locked-cycle test results

Locked cycle testing has taken place on multiple ore samples with multiple conditions applied. A summary of the results is shown in Table 13-9.

Table 13-9: Summary of sequential locked-cycle test data

Zone	No. Samples	Pb Concentrate Grade		Pb Concentrate Recovery		Zn Concentrate Grade		Zn Concentrate Recovery	
		Pb %	Ag g/t	Pb %	Ag %	Zn %	Ag g/t	Zn %	Ag %
CS	1	66	1235	75	66	55	258	61	5
CSC	2	68	1620	79	70	52	729	72	12
FBS	21	49	2541	52	49	52	496	75	22
PM	6	50	1269	62	46	56	251	59	12
QSB	2	47	3386	64	64	-	-	-	-
FeOx	1	waste							
MnO	2	waste							

No LCT was conducted on the PG (plumbogummite) zone. Given the poor metallurgical response of PG samples to bulk sulfide flotation, it is likely that this mineralization would respond well to a sequential flotation system.

The results of the SGS (2008) locked-cycle tests are shown in Table 13-10 and Table 13-11. Details of the flotation conditions can be found in the report.

Table 13-10: Locked cycle flotation results (SGS Vancouver Metallurgy, 2008a)

		Composite Type I/II							
	Units	D LCT1	G LCT2	K LCT1	M LCT1	R LCT1	U LCT1	Minas 3 LCT1	Average
GRADES									
Feed	Ag, g/t	57.8	35.4	19.9	68.5	23.3	41.8	153.5	57.2
	Pb, %	1.6	1.1	0.7	2.2	2.1	0.8	5.1	1.9
	Zn, %	1.9	1.0	1.4	1.3	0.6	1.4	1.9	1.3

	Units	Composite Type I/II							
		D LCT1	G LCT2	K LCT1	M LCT1	R LCT1	U LCT1	Minas 3 LCT1	Average
Lead concentrate	Ag, g/t	1680	1646	904	1671	441	2392	2199	1562
	Pb, %	65.0	49.7	50.2	53.5	57.4	54.9	79.7	58.6
	Zn, %	7.2	9.5	3.3	4.2	2.5	7.9	2.9	5.4
Zinc concentrate	Ag, g/t	651	374	192	295	297	283	729	403
	Pb, %	6.7	5.2	1.9	6.6	9.1	1.2	13.7	6.3
	Zn, %	49.3	51.8	58.1	54.0	46.6	55.2	52.0	52.4
Zinc cleaner scav tails	Ag, g/t	33.2	17.7	9.0	20.2	20.6	10.6	117.9	32.7
	Pb, %	0.6	0.5	0.2	0.6	0.8	0.2	3.5	0.9
	Zn, %	0.7	0.3	0.4	0.4	0.4	0.4	1.0	0.5
Zinc rougher tails	Ag, g/t	4.0	10.4	5.0	7.0	4.3	4.7	12.0	6.8
	Pb, %	0.2	0.4	0.1	0.3	0.2	0.1	0.4	0.2
	Zn, %	0.2	0.2	0.4	0.2	0.1	0.2	0.3	0.2
DISTRIBUTION									
Lead concentrate	Ag, %	50.9	55.8	54.3	79.7	56.3	73.3	69.0	62.8
	Pb, %	72.9	55.8	80.5	80.3	81.5	87.7	75.3	76.3
	Zn, %	6.5	11.9	2.7	10.6	13.4	7.5	7.7	8.6
Zinc concentrate	Ag, %	36.3	12.8	17.5	7.2	9.1	13.1	12.2	15.4
	Pb, %	13.9	5.9	4.5	5.0	3.1	2.9	6.9	6.0
	Zn, %	82.4	65.3	73.1	69.4	60.2	78.4	71.7	71.5
Zinc cleaner scav tails	Ag, %	7.1	6.3	8.7	5.2	21.1	4.8	12.9	9.5
	Pb, %	5.1	6.4	5.0	5.1	9.2	4.2	11.6	6.6
	Zn, %	4.6	3.4	4.8	5.4	16.1	5.8	8.9	7.0
Zinc rougher tails	Ag, %	5.7	25.0	19.6	7.9	13.5	8.8	5.9	12.3
	Pb, %	8.2	31.8	10.0	9.6	6.2	5.2	6.3	11.0
	Zn, %	6.4	19.4	19.4	14.6	10.3	8.3	11.6	12.9

Table 13-11: Type III locked cycle flotation results (SGS Vancouver Metallurgy, 2008a)

	Units	Composite Type III				
		H LCT1	Minas 1 LCT1	Q LCT2	T LCT	Average
GRADES						
Feed	Ag, g/t	52.9	94	43.3	139.9	82.5
	Pb, %	0.8	3.6	1.3	1.9	1.9
Lead concentrate	Ag, g/t	3034	1409	1625	3708	2444
	Pb, %	35.8	72.8	55.7	51.1	53.8

	Units	Composite Type III H LCT1	Minas 1 LCT1	Q LCT2	T LCT	Average
Lead clnr scav tails	Ag, g/t	47.9	46	92.7	153	84.9
	Pb, %	1.1	1.4	2.5	2.4	1.8
Lead rougher tails	Ag, g/t	23.2	39.8	14.5	35	28.1
	Pb, %	0.4	0.7	0.3	0.5	0.5
DISTRIBUTION						
Lead concentrate	Ag, %	53.7	58.7	57.7	73.2	60.8
	Pb, %	42.8	79.3	66.5	73.6	65.6
Lead clnr scav tails	Ag, %	5.4	4.1	11.1	3.3	6
	Pb, %	8.4	3.3	10	3.7	6.3
Lead rougher tails	Ag, %	40.9	37.2	31.2	23.6	33.2
	Pb, %	48.8	17.4	23.5	22.7	28.1

Further locked-cycle testing was conducted by SGS Vancouver Metallurgy (2009) on the Master composite Ag-Pb-Zn, the results are shown in Table 13-12. A further test on an Ag-Pb Composite was conducted and the results are shown in Table 13-13. Flotation condition details can be found in the report.

Table 13-12: Locked cycle test results on Ag-Pb-Zn Composite (SGS Vancouver Metallurgy, 2009)

Product	Weight	Assays			Distribution		
	%	Ag g/t	Pb %	Zn %	Ag %	Pb %	Zn %
Pb Cln 3 Conc	3.2	1686	59.6	7.1	62	76	9.3
Zn Cln3 Conc	3.5	390	3.7	53.1	15.5	5.1	75.4
Zn Cln1 Scav Tails	11.1	87	2.1	1.6	11	9.4	7.4
Zn Ro Tails	82.1	12	0.3	0.24	11.5	9.6	7.9
Feed	100	88	2.5	2.5	100	100	100

Table 13-13: Locked cycle test results on Ag-Pb Composite (SGS Vancouver Metallurgy, 2009)

Product	Weight	Assays		Distribution	
	%	Ag g/t	Pb %	Ag %	Pb %
Pb Cln 3 Con	2.6	2404	41.8	55.6	52.3
Pb Scav Tails	9.7	77	3.2	6.7	14.8
Pb Ro Tails	87.7	48	0.78	37.8	32.9
Feed	100	112	2.1	100	100

A LCT was conducted on a blend of 5:1 Ag-Pb-Zn and Ag-Pb composites. The Pb concentrate assayed 56% Pb and 1777 g/t silver, at 73% and 61% recovery respectively, while the zinc concentrate assayed 50% Zn and 410 g/t Ag, at 74% Zn recovery while raising the total silver recovery to 76%.

SGS (2012) conducted further locked cycle testing on a new master composite Ag-Pb-Zn and Ag-Pb (Table 13-14).

Table 13-14: Locked cycle test results (1) on Ag-Pb-Zn Composite (SGS Canada, 2012)

Product	Weight	Assays				Distribution			
	%	Ag g/t	Pb %	Zn %	Fe %	Ag %	Pb %	Zn %	Fe %
Pb Cln 3 Conc	0.2	2377	53.2	7.8	5.9	41.5	53.1	5.4	1.8
Zn Cln3 Conc	0.4	552	4.1	45.6	7.8	17.8	7.5	58.2	4.3
Zn Cln1 Tail	5.7	19	0.4	0.3	2.9	8.3	9.4	5.5	21.9
Zn Ro Tail	18.8	22	0.4	0.5	2.8	32.4	30	30.9	72
Feed	25.2	51	0.89	1.28	2.96	100	100	100	100
(direct)		54	0.95	1.3	3.21				

A second locked cycle test was performed based on the information obtained from LCT1. EDO was completely removed from both lead and zinc circuits to improve concentrate grade. Collector SIPX dosage was increased in zinc roughing and cleaning. The results are shown in Table 13-15.

Table 13-15: Locked cycle test results (2) on Ag-Pb-Zn Composite – No EDO (SGS Canada, 2012)

Product	Weight	Assays				Distribution			
	%	Ag g/t	Pb %	Zn %	Fe %	Ag %	Pb %	Zn %	Fe %
Pb Cln 3 Conc	0.95	2122	50.9	8.1	6.8	40.1	54.1	5.8	2.1
Zn Cln3 Conc	1.61	584	4.82	52.7	6.2	18.9	8.7	64.6	3.2
Zn Cln1 Tail	9.27	23.7	0.48	0.31	3.6	4.4	5	2.2	10.6
Zn Ro Tail	88.17	20.8	0.33	0.41	3	36.6	32.2	27.4	84.1
Feed	100	50	0.89	1.32	3.14	100	100	100	100
(direct)		54	0.95	1.3	3.21				

Base Metallurgical Laboratories (2019) undertook six new additional LCT on Ag-Pb-Zn composites (Type I/II) and Ag-Pb composites (Type III). Six samples were constructed from three drill holes from the Corani Deposit. Two of the samples were low grade lead and zinc, these samples were Composite 3 and 4. The other samples had higher levels of lead and zinc. The results are shown in the following tables. Details on the composites and the flotation conditions can be found in the original report.

Table 13-16: Locked cycle test results (Composite 3) on Ag-Pb Transitional Composite (Base Metallurgical Laboratories, 2019)

Product	Weight	Assay			Distribution		
	%	Pb %	Zn %	Ag g/t	Pb %	Zn %	Ag %
Ag-Pb Con	0.9	6.6	2.97	8250	10	26.4	69.9
Ag-Pb Clnr Tail	11.5	0.75	0.13	64	14.2	14.4	6.7
Ag-Pb Rougher Tl	87.6	0.525	0.07	29	75.8	59.2	23.4
Feed	100	0.61	0.1	109	100	100	100

Table 13-17: Locked cycle test results (Composite 4) on Ag-Pb Transitional Composite (Base Metallurgical Laboratories, 2019)

Product	Weight	Assay			Distribution		
	%	Pb %	Zn %	Ag g/t	Pb %	Zn %	Ag %
Ag-Pb Con	1.1	17.3	7.65	5100	24.9	44.6	59.1
Ag-Pb Clnr Tail	7.3	0.94	0.57	100	9.3	22.9	7.9
Ag-Pb Rougher Tl	91.6	0.534	0.06	33	65.9	32.5	33
Feed	100	0.74	0.18	92	100	100	100

Table 13-18: Locked cycle test results (Composite 1) on Ag-Pb-Zn Sulfide Composite (Base Metallurgical Laboratories, 2019)

Product	Weight	Assay				Distribution			
	%	Pb %	Zn %	Fe %	Ag g/t	Pb %	Zn %	Fe %	Ag %
Pb Con	2.9	50.3	15.9	3.75	1796	69.7	21.1	9.3	50.7
Zn Con	2	4.67	50	4.4	815	4.4	45.6	7.5	15.8
Zn Clnr –Scav Tail	5	2.42	10.7	4.68	301	5.7	24.2	19.8	14.5
Zn Rougher Tl	90	0.48	0.23	0.83	22	20.2	9.2	63.4	19
Feed	100	2.12	2.21	1.18	104	100	100	100	100

Table 13-19: locked cycle test results (Composite 2) on Ag-Pb-Zn Sulfide Composite (Base Metallurgical Laboratories, 2019)

Product	Weight	Assay				Distribution			
	%	Pb %	Zn %	Fe %	Ag g/t	Pb %	Zn %	Fe %	Ag %
Pb Con	2.5	52.6	13.1	4.66	1311	74.1	11.2	9.4	54.4
Zn Con	4.1	3.35	53	2.52	378	7.9	75.1	8.4	26.1
Zn Clnr –Scav Tail	5.2	1.19	2.91	5.66	89	3.5	5.2	23.9	7.8
Zn Rougher Tl	88.2	0.29	0.28	0.81	8	14.5	8.5	58.2	11.8
Feed	100	1.76	2.91	1.23	60	100	100	100	100

Table 13-20: Locked cycle test results (Composite 5) on Ag-Pb-Zn Sulfide Composite (Base Metallurgical Laboratories, 2019)

Product	Weight	Assay				Distribution			
	%	Pb %	Zn %	Fe %	Ag g/t	Pb %	Zn %	Fe %	Ag %
Pb Con	1.6	43.2	13.7	7.56	1900	66.9	8.8	5.9	42.8
Zn Con	3.3	1.71	53.2	7	567	5.4	69.8	11.1	26.1
Zn Clnr –Scav Tail	4.7	0.86	3.61	10.7	149	3.9	6.8	24.3	9.8
Zn Rougher Tl	90.5	0.27	0.4	1.33	17	23.8	14.6	58.7	21.3
Feed	100	1.03	2.49	2.06	71	100	100	100	100

Table 13-21: Locked cycle test results (Composite 6) on Ag-Pb-Zn Sulfide Composite (Base Metallurgical Laboratories, 2019)

Product	Weight	Assay				Distribution			
	%	Pb %	Zn %	Fe %	Ag g/t	Pb %	Zn %	Fe %	Ag %
Pb Con	0.6	34.5	13.9	5.86	4690	28	12.2	1.4	42.9
Zn Con	1	2.67	48.6	6.28	807	3.4	67	2.3	11.6
Zn Clnr –Scav Tail	6.2	1.02	0.82	6.3	108	8.1	7.1	14.6	9.7
Zn Rougher Tl	92.2	0.51	0.11	2.34	26	60.5	13.6	81.7	35.8
Feed	100	0.77	0.71	2.65	68	100	100	100	100

In early 2019 Base Metallurgical Laboratories conducted a further series of LCT work. Testing was conducted on both sulfide and transition ore types. The following tables show the results of the LCT work.

Table 13-22: Locked cycle test results (Composite 7) on Ag-Pb Transitional Composite (Base Metallurgical Laboratories, 2019)

Product	Weight	Assay			Distribution		
	%	Pb %	Zn %	Ag g/t	Pb %	Zn %	Ag %
Ag-Pb Con	0.8	33.6	4.60	9377	21.1	13.6	42.1
Ag-Pb Clnr Tail	3.9	4.05	0.64	628	13.1	9.8	14.5
Ag-Pb Rougher Tl	95.4	0.825	0.20	76	65.8	76.7	43.3
Feed	100.0	1.20	0.25	167	100	100	100

Table 13-23: Locked cycle test results (Composite 9) on Ag-Pb Transitional Composite (Base Metallurgical Laboratories, 2019)

Product	Weight	Assay			Distribution		
	%	Pb %	Zn %	Ag g/t	Pb %	Zn %	Ag %
Ag-Pb Con	0.7	26.4	3.99	836	23.4	13.8	35.1
Ag-Pb Clnr Tail	5.3	2.13	0.78	71	13.8	19.6	21.6
Ag-Pb Rougher Tl	93.9	0.550	0.15	8	62.8	66.6	43.2
Feed	100.0	0.82	0.21	17	100	100	100

Table 13-24: Locked cycle test results (Composite 8) on Ag-Pb-Zn Sulfide Composite (Base Metallurgical Laboratories, 2019)

Product	Weight	Assay				Distribution			
	%	Pb %	Zn %	Fe %	Ag g/t	Pb %	Zn %	Fe %	Ag %
Pb Con	2.7	65.6	3.3	4.92	1235	75.2	8.3	1.8	65.8
Zn Con	1.0	5.24	55.0	6.07	258	2.3	52.3	0.9	5.3
Zn Clnr –Scav Tail	4.3	1.14	2.42	10.42	38	2.1	9.6	6.1	3.2
Zn Rougher Tl	92.0	0.51	0.35	7.25	14	20.4	29.8	91.3	25.7
Feed	100	2.33	0.92	7.31	50	100	100	100	100

Table 13-25: Locked cycle test results (Composite 14) on Ag-Pb-Zn Sulfide Composite (Base Metallurgical Laboratories, 2019)

Product	Weight	Assay				Distribution			
	%	Pb %	Zn %	Fe %	Ag g/t	Pb %	Zn %	Fe %	Ag %
Pb Con	0.6	61.2	5.4	5.70	895	62.0	3.8	1.0	42.1
Zn Con	1.3	6.51	45.6	8.27	330	14.4	70.8	3.1	33.8
Zn Clnr –Scav Tail	6.6	0.38	0.82	15.30	17	4.4	6.7	30.2	9.2
Zn Rougher Tail	91.5	0.12	0.17	2.41	2	19.3	18.7	65.7	14.9
Feed	100	0.57	0.81	3.35	12	100	100	100	100

Table 13-26: Locked cycle test results (Composite 15) on Ag-Pb-Zn Sulfide Composite (Base Metallurgical Laboratories, 2019)

Product	Weight	Assay				Distribution			
	%	Pb %	Zn %	Fe %	Ag g/t	Pb %	Zn %	Fe %	Ag %
Pb Con	0.7	58.8	3.6	6.07	1010	75.0	3.9	1.3	48.7
Zn Con	0.7	2.57	46.0	9.85	254	3.7	55.2	2.3	13.7
Zn Clnr –Scav Tail	8.9	0.50	1.90	14.23	38	8.5	27.2	39.0	24.6
Zn Rougher Tl	89.7	0.07	0.09	2.08	2	12.8	13.7	57.4	12.9
Feed	100	0.52	0.62	3.24	14	100	100	100	100

Table 13-27: Locked cycle test results (Composite 16) on Ag-Pb-Zn Sulfide Composite (Base Metallurgical Laboratories, 2019)

Product	Weight	Assay				Distribution			
	%	Pb %	Zn %	Fe %	Ag g/t	Pb %	Zn %	Fe %	Ag %
Pb Con	0.9	48.6	4.6	11.90	3356	78.4	6.9	4.5	76.9
Zn Con	0.4	7.02	55.1	2.87	629	5.3	38.6	0.5	6.7
Zn Clnr –Scav Tail	8.4	0.43	2.14	18.20	60	6.6	30.2	64.7	12.9
Zn Rougher Tl	90.3	0.06	0.16	0.79	2	9.7	24.2	30.3	3.5
Feed	100	0.56	0.60	2.36	39	100	100	100	100

13.5.10 Concentrate quality

A suite of the lead and zinc concentrates produced by SGS from locked-cycle testing was assayed for a series of minor elements. Results are summarized for the lead and zinc concentrates, respectively, in Table 13-28 and Table 13-29.

Table 13-28: Minor elements in lead concentrates

Element	Units	Lead Concentrate from Locked Cycle Test on Designated Composites																				
		U	M	G	R	Main 1	3-zone AgPbZn Ave	1-5 yr	Q	T	Minas 1	Minas 3	Comp 1	Comp 2	Comp 3	Comp 4	Comp 5	Comp 6	Comp 7	Comp 8	Comp 9	Average
Ag	g/t	2448	1776	1678	465	810	1576	2100	1752	3696	1493	2173	1766	1317	8382	5970	1904	4768	9377	1235	836	**2776**
Pb	%	55.5	55.5	51.2	58.9	53.6	54.8	51	62.6	51.1	73	81.1	52.5	53.2	6.4	19.7	43.1	34.7	33.6	65.6	26.4	**49.2**
Zn	%	7.18	4.11	9.03	2.6	6.1	8.15	8.1	0.98	5.4	0.82	2.9	14.3	12.6	2.8	7.6	13.3	13.5	4.6	3.3	4	**6.6**
Cu	%	4.5	1.7	1.9	0.38	1.9	1.25	1.62	0.52	2	1.5	0.35	1.43	1.02	1.1	1.99	0.633	2.26	1.78	0.87	0.62	**1.47**
Au	g/t	-	-	-	-	-	0.29	0.37	-	-	-	-	0.036	<2	0.082	0.299	0.3	1.47	-	-	-	**0.41**
S	%	-	-	-	-	-	20.3	21.3	-	-	-	-	-	-	-	-	-	-	-	-	-	**20.8**
C(t)	%	-	-	-	-	-	0.27	1.77	-	-	-	-	-	-	-	-	-	-	-	-	-	**1.02**
Cl	g/t	-	-	-	-	-	15	<10	-	-	-	-	0.01	0.01	0.02	<0.01	0.01	<0.01	0.01	<0.01	0.06	**2.16**
F	%	-	-	-	-	-	0.02	0.014	-	-	-	-	<0.01	<0.01	<0.01	0.04	<0.01	0.02	0.02	<0.01	0.03	**0.02**
Hg	g/t	6.2	4.4	23.9	4.9	30.2	22.1	16.8	47	15.6	10	1.6	22	13	274	283	73	188	96	8	63	**60.1**
Al	g/t	7366	4156	8122	14545	4911	3950	0.61	11333	3400	8878	1889	3600	2200	13600	13000	1600	9000	2000	300	2400	**5813**
As	g/t	510	1500	3000	2500	440	525	2750	740	910	470	<40	1800	1200	4500	2600	2500	3000	1600	900	7700	**2016**
Ba	g/t	2800	360	360	34	200	225	396	23	2800	3400	190	<1	6	<1	4	<1	1	<10	<10	<10	**771**
Bi	g/t	<200	<30	<30	<20	<20	62	74	<20	130	120	<20	0.69	6.1	17	11	1.37	14.7	1.72	13.5	28	**36.9**
Ca	g/t	190	420	400	200	<40	240	232	2100	480	540	240	0.06	0.04	0.04	0.05	0.09	0.13	0.14	0.05	0.06	**265**
Cd	g/t	450	620	2800	2000	310	720	1390	5000	450	46	190	>2000	974	106	>2000	1090	>2000	397	211	509	**1015**
Cr[1]	g/t	-	750	71	230	190	320	-	520	140	97	97	18	2770	562	12600	22	153	797	213	1900	**1218**
Fe	%	5.6	11	8.6	7.6	10	9.45	8.1	5.1	13	3	1.3	3.7	6.6	30.3	19.1	7.7	6	16.1	4.9	17.7	**9.74**
Mg	g/t	600	190	79	190	78	115	117	220	940	480	150	< 0.01	0.01	0.02	0.02	0.01	0.02	0.11	0.05	0.09	**166**
Mn	g/t	370	410	160	130	320	540	278	170	1000	1400	240	< 0.01	0.05	0.09	0.14	0.02	0.01	1.84	0.04	0.1	**264**
Na	g/t	23	<30	<30	280	72	58.5	72	170	62	150	130	0.014	0.014	0.013	0.017	0.015	0.017	0.02	<0.01	0.03	**60**
P	g/t	370	360	570	<150	<200	<200	266	<150	890	1600	<200	300	110	530	2560	250	1110	1360	250	1130	**777**

[1] Cr assays elevated due to contamination by stainless steel

Element	Units	Lead Concentrate from Locked Cycle Test on Designated Composites																				
		U	M	G	R	Main 1	3-zone AgPbZn Ave	1-5 yr	Q	T	Minas 1	Minas 3	Comp 1	Comp 2	Comp 3	Comp 4	Comp 5	Comp 6	Comp 7	Comp 8	Comp 9	Average
Sb	%	1.7	0.62	1.9	0.38	0.7	0.58	1.21	0.2	0.9	0.48	0.34	0.69	6.1	1.99	3.19	1.37	14.7	0.85	0.42	0.99	**1.97**
Se	g/t	<30	<30	30	30	30	30	30	30	<40	30	<30	4.2	3.4	33.2	16.9	4.2	13.5	6.5	3.2	8.3	**19**
Sn	g/t	<25	<30	<30	<20	<40	<30	<20	110	<50	<20	<20	9	13	29	46	7	18	5	3	20	**26**
Si	%	-	-	-	-	-	2.95	4.15	-	-	-	-	3.35	2.91	5.09	5.89	2.23	5.36	2.12	0.81	5.24	**3.65**

Note: The 3-zone mixed sulfide (Ag-Pb-Zn) Composite ("3 zone AgPbZn Ave" in the table above) result is an average of two suites of assays from two locked cycle tests conducted on this composite.

Table 13-29: Minor elements in zinc concentrates

Element	Units	Zinc Concentrate from Locked Cycle Test on Composite												
		U	M	G	R	Main 1	3-zone AgPbZn Ave	1-5 yr	Comp 1	Comp 2	Comp 5	Comp 6	Comp 8	Average
Ag	g/t	286	272	371	288	250	385	463	813	367	578	788	258	427
Pb	%	1.27	5.76	4.86	6.63	2.16	4.04	4.8	4.43	3.16	1.85	2.59	5.2	3.90
Zn	%	56.4	55.8	53	49.3	58	51.9	52.3	50.2	52.7	51.8	48.2	55	52.9
Cu	%	0.5	0.3	0.37	0.52	0.28	0.29	0.63	0.52	0.26	0.2	0.28	0.32	0
Au	g/t	-	-	-	-	-	0.1	0.39	<2	0.03	0.02	0.098	-	0.13
S	%	-	-	-	-	-	28.6	31.7	-	-	-	-	-	30.2
C(t)	%	-	-	-	-	-	0.2	0.16	-	-	-	-	-	0.18
Cl	g/t	-	-	-	-	-	62.5	<10	200	200	100	100	100	127
F	%	-	-	-	-	-	0.021	0.008	<0.01	<0.01	<0.01	<0.01	<0.01	0.015
Hg	g/t	23.3	38.8	67.3	47	55.1	89.1	69.4	51	49	205	206	52	79.4
Al	g/t	4700	2078	4345	11334	8122	13750	2500	3000	3000	2500	3000	300	4886
As	g/t	600	890	170	740	230	285	722	0.1	0.07	0.18	0.19	0.1	303
Ba	g/t	970	140	200	23	390	465	179	6	11	2	2	<10	217
Bi	g/t	<200	<30	<30	<20	<20	<50	<20	0.24	1.26	0.31	1.62	2.2	1
Ca	g/t	1400	800	920	2100	1100	1835	1090	0.13	0.1	0.2	0.23	0.09	770

Element	Units	Zinc Concentrate from Locked Cycle Test on Composite												
		U	M	G	R	Main 1	3-zone AgPbZn Ave	1-5 yr	Comp 1	Comp 2	Comp 5	Comp 6	Comp 8	Average
Cd	g/t	1800	3300	5200	5000	1600	2650	4940	>2000	>2000	>2000	>2000	>1000	3499
Cr[2]	g/t	380	140	140	220	190	-	-	280	273	144	260	168	220
Fe	%	4.6	3.8	3.1	5.1	5.3	4.6	5.2	4.40	4.3	7.02	6.28	6.10	4.98
Mg	g/t	960	1100	70	220	140	305	71	<0.01	<0.01	<0.01	0.01	0.03	318
Mn	g/t	540	370	130	170	190	555	287	0.02	0.02	0.02	0.02	0.04	187
Na	g/t	<10	<30	<30	170	100	190	49	150	130	140	140	<0.01	134
P	g/t	240	230	370	<150	<200	370	<200	230	130	360	390	130	272
Sb	%	0.36	0.08	0.16	0.2	0.36	0.145	0.21	0.24	0.11	0.14	0.23	0.09	0
Se	g/t	<30	<30	<30	<30	<30	<30	<30	11.7	10.5	10.3	10.4	24.9	13.6
Sn	g/t	62	<30	110	110	<40	<30	57	30	27	20	24	19	51
Si	%	-	-	-	-	-	7.65	2.66	3.21	6.65	3	3.33	0.53	3.86

Note: The 3-zone mixed sulfide (Ag-Pb-Zn) Composite ("3 zone AgPbZn Ave" in the table above) result is an average of two suites of assays from two locked cycle tests conducted on this composite.

[2] Possible contamination by stainless steel has potentially caused elevated Cr levels in Zn concentrate similar to those observed in Pb concentrate

Early work by SGS (SGS Vancouver Metallurgy, 2009; SGS Vancouver Metallurgy, 2007) produced lead concentrates that assayed, on average, 59% lead and 1.8 kg/t silver. Those concentrates, on average, graded about 0.8 % antimony, which would be expected to result in smelting penalties. Arsenic, mercury, and cadmium were also elevated, and there may also be penalties applied for these elements.

The zinc concentrates produced by SGS during those programs, averaged 53.8% zinc and 331 g/t silver. Mercury was elevated in these concentrates, grading on average 56 g/t in the zinc concentrates. At this level, penalties would be anticipated. Antimony and cadmium were also elevated in these concentrates, and potential penalties may be applicable. Elevated Chromium assays in some Pb concentrate samples were determined to be from shards of stainless steel in the concentrate, potentially from laboratory sample preparation.

Recent work groups concentrate grades by mineral zones and uses averages. Grades of other contained metals are calculated based on the concentrate tonnages at the lead and zinc grades. For the economic analysis, the lead concentrates were assumed to contain an average of 9% Zinc. Concentrate grades by metal and zone are shown in Table 13-30.

Table 13-30: Concentrate grades by metal and zone

Lead Concentrate Grades	
Mineral Groups	**Concentrate Lead Grade**
CS and CSC	68%
QSB	47%
Other	50%
Zinc Concentrate Grades	
Mineral Groups	**Concentrate Zinc Grade**
PM	56%
Other	52%

13.5.11 Bulk flotation

In the program of testing by G&T Metallurgical Services, Ltd (2007), a test was conducted on each of 71 samples using the same reagent regime. In these tests, the objective was to recover all the sulfides into a bulk concentrate. A nominal primary grind size P80 of 75 µm was used. Only collectors, SIPX, and 3418A were added to the bulk roughing stage, with lime also added in the regrinding stage. Since no depressants were added beyond the elevated pH in the cleaner, higher recoveries but poorer concentrate grades were obtained as compared to the sequential flowsheet.

Upon analysis of the G&T results, it is clear that the different ore types designated by BCM were useful to delineate metallurgical performance based on these results. A summary of average batch cleaner test performance, obtained from each ore-type, is provided in Table 13-31.

Table 13-31: Average bulk circuit performance by ore-type (G&T, 2007)

Ore Type	No. Samples	Ave Head Grade % or g/t for Ag				Con Grade % or g/t for Ag			Recovery %		
		Pb	Zn	Ag	S	Pb	Zn	Ag	Pb	Zn	Ag
CSC	6	3.70	0.94	137	3.55	27.7	7.3	1110	82	68	78
FBS	26	1.83	1.59	96	3.32	15.8	12.5	1200	64	57	71
PM	15	1.37	2.32	62	5.87	10.4	11.8	444	76	49	75
QSB	7	1.02	0.18	75	1.67	13.9	0.9	12723	11	3	46
FeOx	4	0.52	0.09	71	1.20	9.8	2.9	2665	34	36	55
MnO	6	1.75	0.23	64	3.04	10.6	1.5	3564	5	5	31
PG	7	1.23	0.11	27	0.69	9.9	2.1	1277	5	10	25

Note: Ag grades are denoted in g/tonne, all other assays are in percent.

13.5.12 Whole ore cyanidation tests

Dawson Metallurgical Laboratories, Inc. (2006) utilized a nominal primary grind sizing P80 of 75 µm to conduct the cyanidation bottle roll tests. The 96-hour silver extraction and cyanide consumption data for the tests conducted by Dawson are summarized in Table 13-32.

Table 13-32: Cyanidation leaching results (Dawson, 2006)

Composite	Ag Extraction 96 hours %	Ag Grade g/t		NaCN kg/t
		Residue	Calculated Head	
1	56.9	99	229	4.5
2	58.8	110	266	4.6
3	54.4	105	230	3.4
4	70.5	41	139	3.0
5	64.6	53	150	4.6
6	79.1	30	143	5.7
A	48.8	63	122	4.0
B	39.9	64	100	7.4
C	56.9	36	93	4.9
D	50.1	60	110	4.3
E	45.1	52	94	2.8
F	25.5	140	188	2.9
Wt. Ave.	**54.1**			

On average, just over half the silver was extracted using cyanidation bottle roll tests. Silver extraction kinetics tended to be slow after 8 hours.

Cyanidation testing was only conducted on selected samples, generally of poor flotation response, in the G&T and SGS testing programs. However, results showed that under conditions almost the same as used by Dawson, similar silver extractions were obtained by G&T and SGS on these samples. Using a weighted average, about 51 percent of the silver was extracted from the samples tested by G&T and SGS. Average performance for each deposit and ore type is summarized in Table 13-33. Results show that samples that have poor bulk flotation response (FeOx, QSB, MnO, and PG) tended to respond more favorably to cyanidation. It may be possible to take advantage of this by pursuing a flowsheet involving cyanidation of flotation tailings.

Table 13-33: Average cyanidation extraction by deposit and classification (G&T, 2007)

Zone	No. Samples	Wt. Ave. Ag Extraction %
Deposit		
Este	9	38
Main	11	73
Minas	12	54
Ore Type		
FBS	8	35
PM	3	26
FeOx	3	70
QSB	5	81
MnO	6	59
PG	7	79

Note: Silver extraction averages were weighted using silver head grades.

13.5.13 Cyanidation leaching of flotation tailings

The opportunity to process Corani mineralization using a combination of flotation and cyanidation was only evaluated by Dawson Metallurgical Laboratories (2006). In that program, Dawson tested only two samples using this methodology. The silver recovery was superior to that obtained by the use of either flotation or cyanidation alone. Further testing would be required to determine if this flowsheet option has merit across a broad range of samples.

13.6 Thickening and Filtration

A wide variety of testing has been conducted by BCM on the tailings and concentrates including thickening and filtration tests as well as rheology investigations. Early work was conducted by SGS and University of British Columbia and later more recent work by Golder and Outotec. As summary of the relevant work is described below.

SGS (2018) conducted two flotation tests to produce tailings samples for thickening testing. The samples represented ore types I/II and III. The type I/II sample was produced from two existing composites developed from DDH-C-TJ-30 (composite U) and DDH-C-TJ-15 (composite M). The type III sample was produced from DDH-C-TJ-80 (composite N). Typical flotation parameters were employed including regrind and concentrate cleaning to produce a tailings sample that would represent a production scenario. The flocculant screening and testing was conducted on:

- Type I/II Flotation Tailing
- Type III Flotation Tailing
- Type I/II Pb Concentrate
- Type I/II Zn Concentrate
- Type III Pb Concentrate

Static thickening tests were conducted to examine flocculation requirement, hydraulic loading rate, unit area requirements, and feed solids concentration sensitivity and conducted only in the tailings samples.

Type I/II Pb and Zn Concentrate and Type III Pb Concentrate

Results of flocculant screening tests conducted on Type I/II Pb and Zn Concentrate at 20% solids and ambient temperature (20 °C) indicated that a minimum dose of 10 – 15 g/t was required to produce good settling rates and supernatant clarity. Results of flocculant screening tests conducted on Type III Pb Concentrate at 20% solids and ambient temperature (20 °C) indicated that a minimum dose of 6 – 10 g/t was required to produce good settling rates and supernatant clarity. The flocculant selected for best overall performance for all Concentrate materials was Hychem AF 307, a medium to high molecular weight 40% charge density anionic polyacrylamide. Products meeting this same description may also serve.

Type I/II and Type III Flotation Tailing

Results of flocculant screening tests conducted at 10% solids on the Type I/II and Type III Flotation Tailing samples at ambient temperature, indicated that a minimum dose of 30 - 40 g/t was required to produce good settling rates and reasonable supernatant clarity. The flocculant selected for best overall performance, and used in SLS testing was Hychem AF 307, a medium to high molecular weight 40% charge density anionic polyacrylamide. Products meeting this same description may also serve.

Conventional (static) thickening tests were conducted on the Corani Type I/II and Type III Flotation Tailing samples at various flocculant doses, and feed solids concentrations to develop and optimize parameters for conventional thickener design. A general summary of static test results for the samples tested is shown inTable 13-34, followed by a summary of recommended optimum conventional thicker design parameters based on the data obtained.

Table 13-34 Summary of recommended conventional thickening design parameters (SGS Vancouver Metallurgy, 2008a)

Sample Material	Material P80	Feed Solids Concentration	Flocculant Dose[1]	Underflow Concentration[2]	Unit Area[3,4]
	(um)	(%)	(g/t)	(%)	(m^2/t/d)
Type I/II Tailing	ND	10 – 15	35 - 45 (Hychem 307)	50 – 55	0.35 – 0.51
Type III Tailing	ND	10 – 15	30 - 40 (Hychem 307)	50 – 55	0.25 – 0.35

Notes:

1. Flocculant used for flotation tail sample was Hychem AF 307, a 40% charge density anionic polyacrylamide. Other products meeting the same description may also serve. Flocculant solution concentration of 0.1 to 0.2 grams per liter is recommended prior to contact with fresh feed slurry at the concentration noted.
2. Note: This range is a best guess from test observations, the actual range that can be achieved for thickener design should be based on rheology testing data.
3. Recommended Unit Area for conventional thickener design (includes a 1.25 scale-up factor).
4. Unit Area is based on minimum recommendations for full-scale conventional thickeners. High rate thickener testing could result in reduced thickener diameter recommendations.

Based on static thickening test data, estimated high rate type thickener design parameters would be 2.5 m^3/m^2 h for the Type I/II tailing, and 3.0 to 4.0 m^3/m^2 h for the Type III tailing material (in feed solids concentration, and flocculant dose ranges noted above).

The University of British Columbia conducted a rheology study for SGS in 2011. A rheological study was conducted on tailing samples from SGS for the Corani Project. The objective was to measure the rheological properties of the samples as a function of pulp density. All the samples showed non-Newtonian viscosity behavior and the Bingham model was fitted to the experimental data and the main results are summarized in Table 13-35.

Table 13-35 Rheogram values (Sakuhuni and Holuszko, 2011)

Sample I.D.	% Solids	Plastic viscosity	Bingham Yield stress
	(%)	(mPa.s)	(Pa)
Zinc Tailing	30	3.5	0.8
	35	4.7	1.5
	40	5.9	1.5
	45	7.9	2.6
	50	9.9	7.9
	55	18.0	15.6
	60	67.0	58.0
Pyrite Tailing	30	3.5	0.8
	35	3.5	1.7
	40	3.5	3.7
	45	3.5	7.0
	50	9.3	11.9
	55	16.2	14.4
	60	70.8	58.0

Golder (2019) was contracted to examine the physical properties of a tailings sample. Two buckets, each containing 20 L of slurry, were received. It was confirmed in the laboratory that the samples were in optimal condition for testing. The sample was classified as BCM HE1050 Relave. The sample was analyzed for health and safety test, particle size distribution, specific gravity, and rheology.

No health and safety concerns were noted, the pH of the sample was 9.8. The size distribution was D_{50} 37.5 µm and D_{85} 101 µm. It was classified as 26% sand, 74% fines and 17% clay.

Rheological tests were carried out in order to evaluate the fluid behavior and transport properties of the sample. These tests provide information on how the tailings will behave during pumping, transport and while at rest. The rheology of the tailings sample was measured for different solids concentrations using a HAAKE RheoWin rotational viscometer considering two types of sensors: a vane and a Bob and Cup. These tests measure the static yield stress, the dynamic yield stress and the dynamic viscosity of the sample. The static yield stress is determined using a vane type sensor. Static yield stress is defined as the minimum force required to initiate fluid movement. It also indicates the consistency of the material. Dynamic stress and dynamic viscosity are determined using a Bob and Cup type sensor. The samples tested were analyzed as a plastic slurry in accordance with the Bingham model. From the modeling of the measured rheograms and the application of the Bingham model, the rheological parameters of yield stress and the viscosity were obtained for the range of concentrations tested.

The values of static yield stress, dynamic yield stress and viscosity are shown in Table 13-36 and shows the impact of solids concentration on the static yields stress of the tailings.

Table 13-36 Rheogram values (Golder, 2019)

Slump	Solids	Yield Stress	Rheogram Values			
			Ramp Up		Ramp Down	
			Yield Stress	Viscosity	Yield Stress	Viscosity
(in)	(%)	(Pa)	(Pa)	(Pa.s)	(Pa)	(Pa.s)
7.5	69.9	376	314	1.36	343	1.31
9	67.8	207	249	0.543	270	0.535
10.5	64.0	76.7	121	0.164	136	0.128
	62.4	58.7	77.5	0.101	86.3	0.080
	61.2	44.6	55.1	0.077	63.0	0.053
	60.0	33.3	45.1	0.051	47.8	0.046
	57.7	23.7	30.1	0.041	32.6	0.035
	56.2	17.5	23.2	0.033	25.4	0.027
	54.4	13.5	18.0	0.026	18.8	0.025
	51.1	8.9	11.3	0.019	12.0	0.017
Cake	75.0					

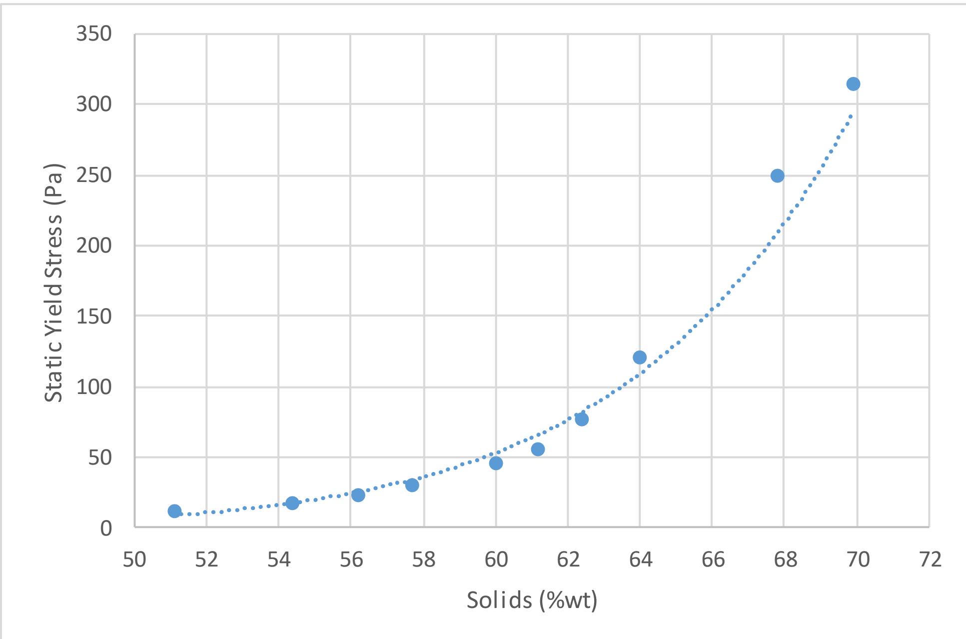


Figure 13-25 Impact of solids concentration on static yield stress (Golder, 2019)

Jenike and Johanson (2019) conducted additional test on materials handling of filtered tailings. A single filtered tailing sample was delivered for testing (18% moisture). The sample was stated by BCM to be representative of typical tailings materials. The wet screened material had a P80 of approximately 75 µm (Figure 13-26). The dry screened material represents agglomerates formed by filtering. The sample was not broken down before screening in order to provide representative details for materials handling characterization of the filtered tailings.

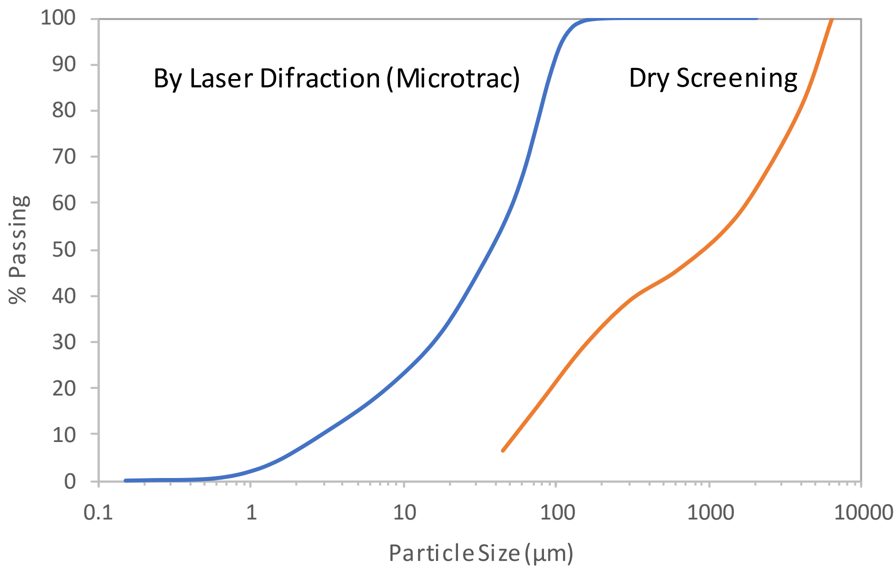


Figure 13-26: Particle size distribution of the sample of tailings recieved for testing using Tyler screens (dry method) and by laser diffraction (wet technique) (Janike and Johanson, 2019b).

Basic flow property testing was conducted and the following conclusions reached. It was not possible to test the sample of tailings at 20% and higher moisture levels because the material exhibited a plastic behavior and did not fail in the shear cell at all. In general, based on Jenike's classification ("Storage and Flow of Solids", Bulletin 123, University of Utah, 1964) and given the consolidating pressure to be encountered in the plant, the tailings received could be considered a very cohesive material, mainly because of the high moisture content levels and fine particles.

Tests results also indicate that the sample of tailings tested is a compressible materials, i.e. the bulk density does vary significantly with consolidating pressure (height level of a stockpile, silo and/or hopper), and also with moisture content. The material tested exhibit a strong tendency to form cohesive arches when handled continuously (instantaneous flow) but this arching tendency increases significantly with moisture content and storage time at rest and under pressure, especially if handled in funnel flow bins and hoppers.

In June of 2019, Outotec Peru conducted thickening and filtration test work based on final tails samples provided by SGS Peru. The objective was to evaluate the behavior of a sample of tailings from the Corani project to the process of filtration using a pressure filter vertical plates semi-pilot and using as filtration stages the feeding step to the filter, the pressing stage and the blowing stage. In addition, to evaluate the filtration process parameters as cycle time and filtration rate for sizing equipment at industrial level.

The tests were performed with laboratory equipment MFP 0.3 which corresponds to a vertical filter plates and the clothes chosen corresponds to the ASKO T54, which has a permeability of 50 m^3/m^2min. The thickness of the chamber used was 45 mm, and the solids concentration of the slurry feed to the filter was 55% solids. The results of the testing are shown in Table 13-37.

Table 13-37 Filtration test summary (Outotec, 2019a)

Variable	Unit	T-1	T-2	T-3	T-4	T-5	T-6	T-7
Solids concentration	%	55	55	55	55	55	55	55
Feed time	min	7	2	3	4	4	4	4
Pressing time	min	0	2	2	2	6	1	2

Variable	Unit	T-1	T-2	T-3	T-4	T-5	T-6	T-7
Blowing time	min	0	0	0	0	0	2	2
Technical time	min	4.5	4.5	4.5	4.5	4.5	4.5	4.5
Cycle time	min	11.5	8.5	9.5	10.5	14.5	11.5	12.5
Cake weight (dry)	kg	8.9	6.9	7.3	7.8	6.9	7.3	7.6
Cake thickness	mm	47.4	32	37	40.4	34.1	37.8	39.3
Cake moisture	%	20.1	17.6	17.8	18.5	18	17.7	17
Filtration rate	kg/m²h	173	182	171	166	106	140	135
Suspended solids	ppm	210	530	480	350	460	320	230

With a feed of 55% solids, a feeding time of 4 minutes, 2 minutes of pressing and 2 minutes of blowing, it is possible to obtain a moisture of the cake of 17% and a filtration rate of 135 kg/m^{2}h. For the cycle times above, a good cake quality, with a thickness of 38.3 mm on average was observed (85% of the thickness of the chamber used).

Outotec (2019b and 2019c) also conducted thickening tests with two continuous thickener sizes, 99 mm and 190 mm. Static tests were conducted to evaluate the performance of different flocculants and to determine the optimal dilution of slurry feed. Once the optimal flocculant and dilution were obtained, dynamic tests were undertaken on the two dynamic thickeners. Figure 13-27 shows the results of the flocculant selection tests.

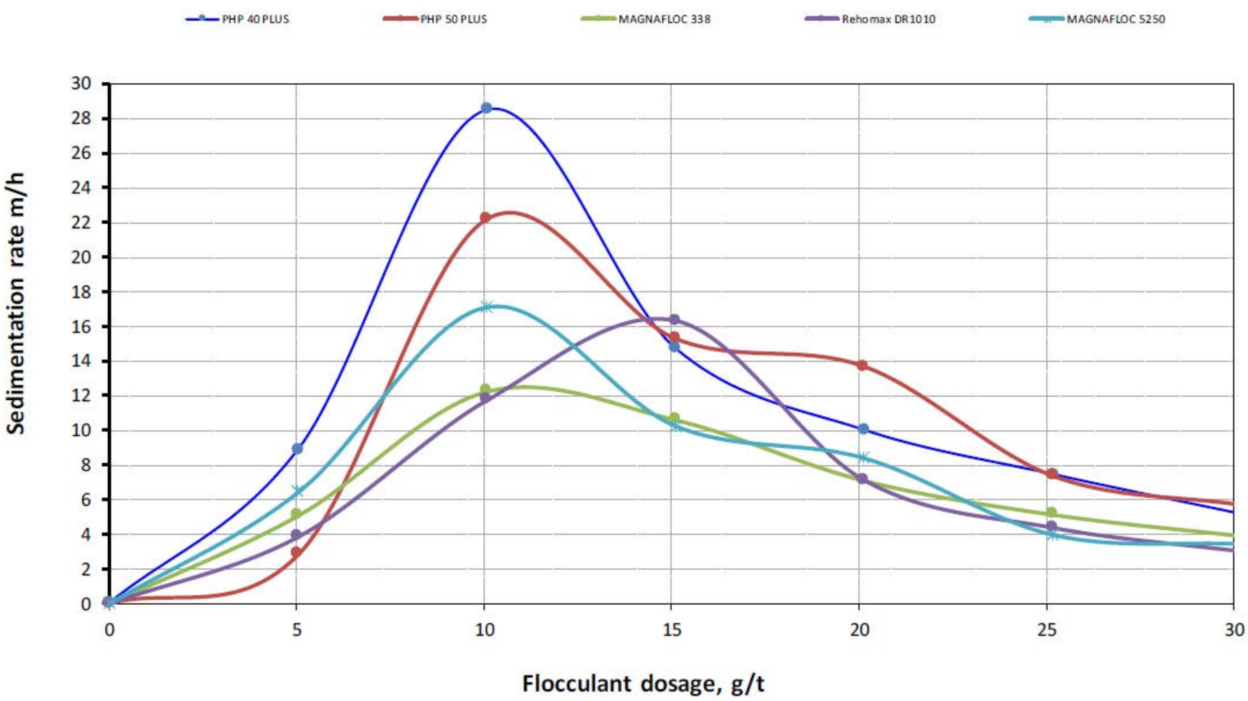


Figure 13-27 Flocculant selection tests (Outotec, 2019b)

The results of the flocculent selection and dilution were employed for the dynamic thickener testing. Dynamic thickener testing was conducted using PHP40PLUS at dosages of 15,20 and 25 g/t at a feed dilution of 12% solids. Figure 13-28 shows the underflow solids concentration for various solids loadings. At low flocculent dosages there was a high degree of turbidity making interface identification difficult. Overflow turbidity also increases with increased loading rate and can be combated by higher flocculent dosages.

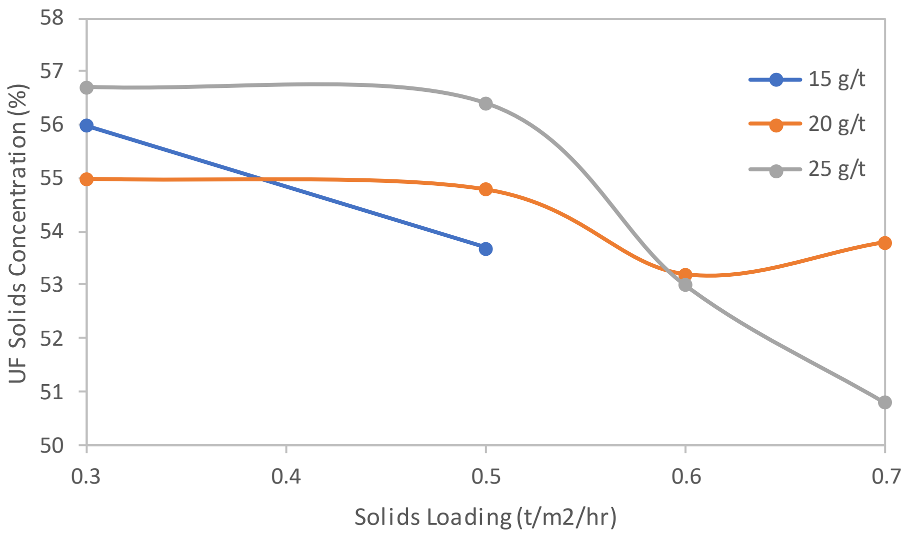


Figure 13-28 Underflow solids based on solid loading (Outotec, 2019b)

The yield stress of the thickened tailings samples was also measured at different solids concentrations (Figure 13-29).

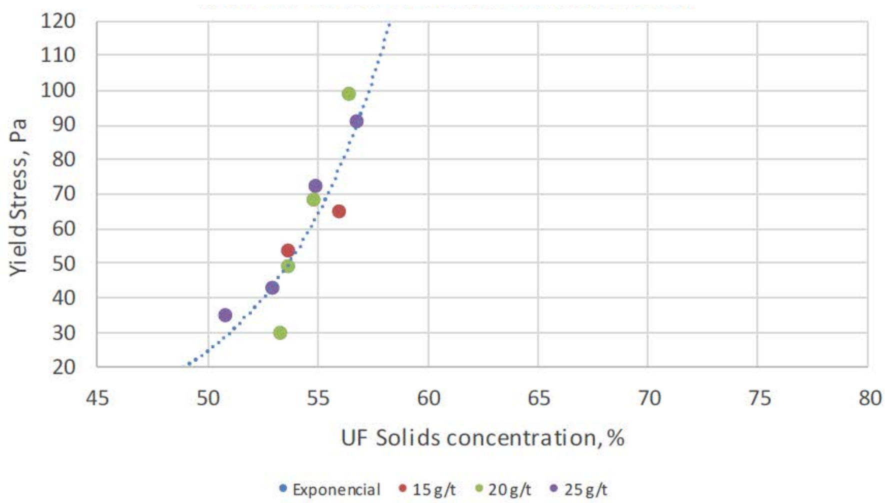


Figure 13-29 Yield stress based on underflow density (Outotec, 2019b)

The following conclusions were drawn from this thickening test work.

- The optimal dilution of pulp in the feed was 12%.
- Dynamic tests on using 99 mm diameter thickener shows a maximum of 56% solids in the underflow.
- At high solids loading and low flocculent dosages the flocs were very small and did not have sufficient weight to decant causing it to be maintained in suspension and control interface in the test was lost.
- It was evidenced that, by decreasing the pH of the slurry feed to the thickener, reduces the solids density obtained in underflow.

- The overnight settling produced 63% solids, indicating the potential for compaction of the sample, a solids loading of 0.50 t/m^{2}h and 25 g/t of flocculant PHP 40 Plus and yield stress gave a value of 159 Pa was used for this test.

The aim of the second study by Outotec on the 190 mm diameter thickener was to determine the behavior of the final tailings sample in different thickener configurations; high rate thickener (HRT), high compression thickener (HCT) and a paste thickener (PT). This testing was conducted to evaluate a tonnage feed to the thickener of 988 t/h adding dosages of flocculant PHP40 Plus flocculent at 15, 20 and 25 g/t with a solids loading of 0.5 t/m^{2}h. The results indicate that: 48-55% solids in the underflow can be achieved with an HRT of Ø50 m, 51-58% solids in the underflow can be achieved with an HCT of Ø50 m or 55-62% solids in the underflow can be achieved with two Paste thickeners of Ø35 m. The yield stress for each of the thickener underflow materials is shown in Figure 13-30.

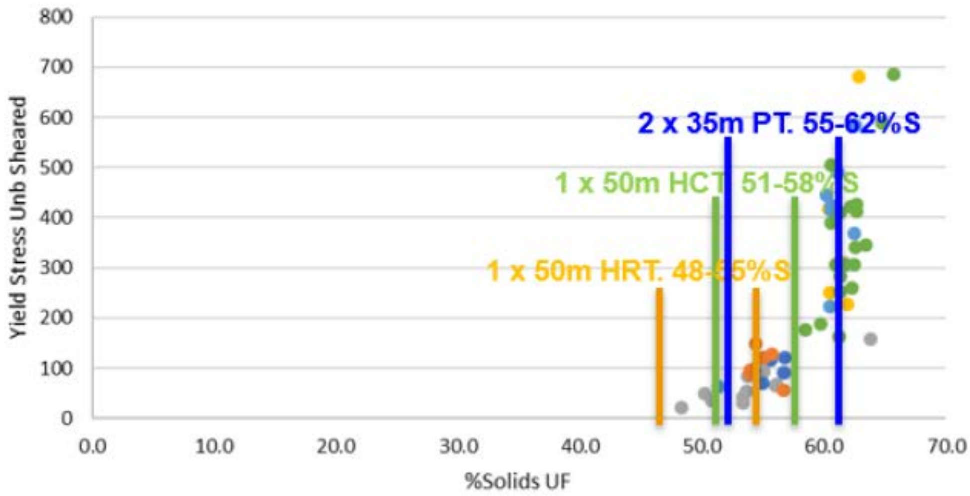


Figure 13-30 Thickener configuration yield stress (Outotec, 2019b)

13.7 Environmental Testing (Acid Base Accounting)

Testing by included acid-base-accounting (ABA), net acid generation (NAG), aqua regia digest ICP, and whole rock analysis, and the standard synthetic precipitation leaching procedure (SLPL). All test work was conducted by or under the supervision of SGS CEMI in Burnaby, British Columbia (SGS Vancouver Metallurgy, 2008a). The results are as shown in Table 13-38.

Table 13-38 Net acid generation testing of Corani tails (SGS Vancouver Metallurgy, 2008a)

Sample ID	NAG pH after reaction	Vol. of 0.1 N NaOH		NAG (kg H2SO4/tonne)	
		to pH 4.5	to pH 7.0	to pH 4.5	to pH 7.0
50000-003 VFNS1 Ro	3.64	1	2.95	1.96	5.78
50000-003 VFUM1 Ro	3.3	1.6	7.5	3.14	14.7
Blank	4.66	-	-	-	-

Table 13-39 Aqua-regia digest metals analysis (SGS Vancouver Metallurgy, 2008a)

Sample ID	Ag ppm	Al %	As ppm	Ba ppm	Be ppm	Bi ppm	Ca %
50000-003 VFNS1 Ro	84.4	1.17	628	836	9	3.3	0.26
50000-003 VFUM1 Ro	13	0.37	338	164	5	0.7	0.35
Sample ID	**Cd ppm**	**Ce ppm**	**Co ppm**	**Cr ppm**	**Cs ppm**	**Cu ppm**	**Fe %**
50000-003 VFNS1 Ro	2.5	23	7.5	38	28.1	162	>10.00
50000-003 VFUM1 Ro	19.8	39	4.4	46	56.8	259	5.08
Sample ID	**Ga ppm**	**Ge ppm**	**Hf ppm**	**Hg ppm**	**In ppm**	**K %**	**La ppm**
50000-003 VFNS1 Ro	8	0.3	0.1	9.8	0.17	0.08	11
50000-003 VFUM1 Ro	4	0.3	0.2	0.6	1.03	0.22	16
Sample ID	**Li ppm**	**Mg %**	**Mn ppm**	**Mo ppm**	**Na %**	**Nb ppm**	**Ni ppm**
50000-003 VFNS1 Ro	21.8	0.04	3805	10	0.01	0.3	6.6
50000-003 VFUM1 Ro	21.4	0.22	1534	4.4	0.02	0.3	14
Sample ID	**P %**	**Pb ppm**	**Rb ppm**	**Re ppb**	**S %**	**Sb ppm**	**Sc ppm**
50000-003 VFNS1 Ro	0.316	>10000	6.1	<5	0.82	2249	1.8
50000-003 VFUM1 Ro	0.11	3530.6	45.7	<5	1.47	105.7	2.1
Sample ID	**Se ppm**	**Sn ppm**	**Sr ppm**	**Ta ppm**	**Te ppm**	**Th ppm**	**Ti %**
50000-003 VFNS1 Ro	<0.5	9.2	1343	<0.1	<0.1	4.4	<0.005
50000-003 VFUM1 Ro	<0.5	3	28	<0.1	<0.1	9.8	0.02
Sample ID	**Tl ppm**	**U ppm**	**V ppm**	**W ppm**	**Y ppm**	**Zn ppm**	**Zr ppm**
50000-003 VFNS1 Ro	50.7	9.2	4	11.4	5.4	338	2.7
50000-003 VFUM1 Ro	17.1	5.1	11	7.4	5.6	5619	5.2

Table 13-40 Whole rock analysis (SGS Vancouver Metallurgy, 2008a)

Sample ID	SiO_2 %	TiO_2 %	Al_2O_3 %	Fe_2O_3 %	MnO %	MgO %	CaO %
50000-003 VFNS1 Ro	55.0	0.15	8.74	16.97	0.57	0.12	0.51
003 VFUM1 Ro	62.1	0.36	10.4	9.22	0.22	0.43	0.70
Sample ID	**Na_2O %**	**K2O %**	**P_2O_5 %**	**Ba(F) %**	**LOI %**		**Total %**
50000-003 VFNS1 Ro	0.25	2.13	1.12	0.41	11.5		97.5
50000-003 VFUM1 Ro	0.37	6.59	0.31	1.16	5.93		97.8

Table 13-41 Synthetic precipitation leaching procedure (SPLP) (SGS Vancouver Metallurgy, 2008)

Parameter	Method	Units	VFSN 1	VFUM 2
			RO Tails	RO Tails
Sample Weight		g	100	100
Extractant Volume		mL	2000	2000
Extractant pH	pH	m	4.22	4.22
Leachate pH (18 h)	meter		6.34	7.68
Redox Potential		mV	325	294
Conductivity	meter	uS/cm	107	241
Acidity (ph 4.5)	titration	mg $CaCO_3$/L	0	0
Total Acidity	titration	mg $CaCO_3$/L	4.9	2.7

Parameter	Method	Units	VFSN 1	VFUM 2
			RO Tails	RO Tails
Total Alkalinity		mg $CaCO_3$/L	2.2	21.6
Sulphate Turbidity		mg/L	51	97
Chloride	IC	mg/L	1.4	7.9
Fluoride	IC	mg/L	0.05	0.19
Ion Balance				
Major Anions	Calc	meq/L	1.1	2.7
Major Cations	Calc	meq/L	1.1	2.8
Difference	Calc	meq/L	-0.1	0.1
Balance	(%)	Calc (%)	-3%	1%
Hardness ($CaCO_3$)	Calc	mg/L	34.1	118
Aluminum	ICP-MS	mg/L	0.009	0.0111
Antimony	ICP-MS	mg/L	0.0083	0.0281
Arsenic	ICP-MS	mg/L	0.002	0.00108
Barium	ICP-MS	mg/L	0.0796	0.0472
Beryllium	ICP-MS	mg/L	0.0013	<0.00001
Bismuth	ICP-MS	mg/L	<0.00005	<0.000005
Boron	ICP-MS	mg/L	<0.5	<0.05
Cadmium	ICP-MS	mg/L	0.00475	0.000248
Calcium	ICP-MS	mg/L	11	42.8
Chromium	ICP-MS	mg/L	<0.001	0.0003
Cobalt	ICP-MS	mg/L	0.0291	0.0042
Copper	ICP-MS	mg/L	0.0069	0.00046
Iron	ICP-MS	mg/L	0.099	0.011
Lead	ICP-MS	mg/L	0.0404	0.00732
Lithium	ICP-MS	mg/L	0.052	0.0655
Magnesium	ICP-MS	mg/L	1.64	2.68
Manganese	ICP-MS	mg/L	8.08	0.0733
Mercury	ICP-MS	mg/L	<0.1	0.04
Molybdenum	CVAA	ug/L	<0.0005	0.00686
Nickel	ICP-MS	mg/L	0.0357	0.00174
Phosphorus	ICP-MS	mg/L	<0.02	0.009
Potassium	ICP-MS	mg/L	2.79	8.81
Selenium	ICP-MS	mg/L	<0.0004	0.00009
Silicon	ICP-MS	mg/L	<1	1
Silver	ICP-MS	mg/L	<0.00005	0.000152
Sodium	ICP-MS	mg/L	0.66	3.87
Strontium	ICP-MS	mg/L	0.105	0.0405
Sulphur	ICP-MS	mg/L	22	45
Thallium	ICP-MS	mg/L	0.151	0.0109
Tin	ICP-MS	mg/L	<0.0001	<0.00001
Titanium	ICP-MS	mg/L	0.007	<0.0005
Uranium	ICP-MS	mg/L	0.00008	0.000115

Parameter	Method	Units	VFSN 1	VFUM 2
			RO Tails	RO Tails
Vanadium	ICP-MS	mg/L	<0.002	<0.0002
Zinc	ICP-MS	mg/L	0.162	0.0246
Zirconium	ICP-MS	mg/L	<0.001	<0.0001

13.8 Continuous Predictive Metallurgical Model

During the 2018 program a total of 6 composites were tested. Of those BCM identified 6 tests which were incorporated into the geometallurgical model. Those tests included 4 primary sulfide materials, (composites 1, 2, 5 and 6) and two transitional materials, (composites 3 and 4). The 2019 program included 6 additional tests which were tested to optimize lead rougher recovery, zinc rougher recovery and lead zinc locked cycle testing. In addition, two transitional composites were also used for locked-cycle testing to an ultimate concentrate.

The geometallurgical modelling developed for the 2015 and 2017 Technical Reports was updated using new test work and a similar approach as was done with the 2015 and 2017 geometallurgical models. The new effort however identified a difference in transition mineralization from the sulfide and oxide mineralization. Separating the transition samples from oxide and sulfide produced a better prediction of the oxide and sulfide, lead and silver recovery, to concentrates. Analysis showed the inclusion of tests of transition material with sulfide and oxide material showed a significant reduction in the recovery of sulfide and oxide when transition material was included reducing the overall recovery. Since transition mineralization only represents about 3% of the deposit, GRE worked to separate the transition material in the block model and create separate recovery formulae for that material.

Various advanced statistical methods were employed to analyze the data and better understand the various drivers that affect recovery. These methods included k-means cluster analysis, classification and regression trees (CART), and multivariate adaptive regression splines (MARS). Using these methods, zinc grade, elevation, oxide minerals, pyrite, and the form of lead (galena vs. phosphate) were identified as the best predictors of lead recovery; zinc grade, elevation, copper grade, and pyrite were identified as the best predictors of zinc recovery; and lead recovery was identified as the best predictor of silver recovery in the lead concentrate. To model the complex relationship between these parameters and recovery, multivariate adaptive regression splines were used to develop a model capable of predicting recoveries based on this data. Multivariate adaptive regression splines are a form of multiple regressions that has the flexibility to model non-linear relationships between variables. The method also uses data partitioning, which allows the model to identify characteristics within the dataset that potentially lead to different outcomes.

Though this method appears complex, it combines the benefits of a model based on ore type characterization and a model using regression. It partitioned data based on similarities, similar to ore-type characterization and develops a regression model for predicting a continuous recovery result, similar to regression modelling. The result is a single model equation that can be used to predict recovery across all ore types.

To create an overall balance for lead, silver, and zinc, the concentrates were assumed to be constant grade. The grades of the concentrates and the adjustments to the rougher data were determined by the averages of locked-cycle testing. The sections below give a brief summary of each model used for recovery predictions. Additional details regarding the complete statistical analysis were reported by GRE in a separate document.

The same basic forms of model for the lead and zinc models were used from the 2017 geometallurgical modelling, however the intercept correction was adjusted using the locked-cycle testing in the updated dataset. The silver model for recovery to the lead concentrate was changed to include additional factors to better estimate the recovery in the transition and non-transition areas. The silver model for recovery to the zinc concentrate remains of the same form but was updated using the updated dataset.

13.8.1 Geometallurgical database

The geometallugical models were created based on a compiled dataset of metallurgical test work that includes 125 metallurgical samples from drill holes and a total of 519 lead rougher flotation tests on those metallurgical samples. Nine of the metallurgical samples were composites from multiple drillholes. QEMSCAN data was available for most samples and sample composites.

An effort was made to eliminate outlier tests where an extreme condition led to an extremely low recovery. These outliers were tests where pH was greater than 10 (3 tests) or lime addition in excess of 7000 g/t (3 tests). Three additional tests were eliminated as invalid based on notes on the test sheet or extremely low recovery not consistent with known sample mineralogy. Of the original tests, 9 were eliminated on this basis. Test work completed during 2018 and 2019 included an additional 12 samples.

In addition, the composites where multiple drill holes were used to create the composite were not used in the analysis due to difficulty in accurately constraining the sample mineralogy. However, these samples were later used for model validation. For the final analysis, 110 of the 125 samples were used (9 multi-hole composites were eliminated and 7 were eliminated due to invalid tests/conditions).

13.8.2 Recovery of lead to lead concentrate

Using the QEMSCAN data, the most statistically significant predictor of high and low lead recovery continues to be sphalerite. In the best performing group, high and low recovery is best defined by plumbogummite/gorceixite. Amongst the lowest recoveries, recovery is best predicted by chalcopyrite. Pyrite and plumbogummite/gorceixite are both close surrogates for chalcopyrite in this group.

Since the FBS material comprises the largest portion of the deposit and showed the most variability in lead recoveries, an analysis of the FBS samples was performed to determine what factors define high and low performing FBS material with respect to lead recovery. The updated analysis shows that the lead recovery in the FBS mineralization style continues to be best predicted by using oxide minerals, sphalerite, pyrite, and lead types as good indicators to predict lead recovery.

The relationship between lead recovery and parameters from the geologic logs were revisited. The zinc grade, elevation, manganese oxide, and goethite remain excellent predictors for recovery. Except for elevation, the relationships appear to be nonlinear. In the case of elevation, the relationship with recovery is strongest at elevations above roughly 4900 masl.

Within the high recovery group, manganese oxide continues to distinguish the best recoveries, while in the low recovery group, goethite, elevation and manganese oxide are the best indicators to predict those recoveries.

Transition versus Non-Transition Ores

Near-surface ores with low zinc and higher silver grades in the Este zone had lower recoveries and were separated using an indicator. Composites were coded based on the samples being less than or equal to 0.3 percent zinc, greater than or equal to 15 g/t silver, within 90 meters of the hole collar. Once coded, the indicator was estimated as a majority code in the block model using an OK model. This indicator was used to segregate the data for calculation of the lead and silver to lead concentrate recoveries.

13.8.3 Model selection and validation

The multivariate adaptive regression splines were updated for modelling lead recovery based on block model parameters. Zinc grade, elevation, galena, goethite, manganese oxide, and pyrite were selected for inclusion in the model based upon the data analysis.

Based on the results from the above analyses, multivariate adaptive regression splines (MARS) models were updated for lead rougher recovery. The performance of the model for the validation dataset and across all data is shown in Figure 13-31. The colors in Figure 13-31 represent the mineralization types assigned to samples using the prior geometallurgical approach.

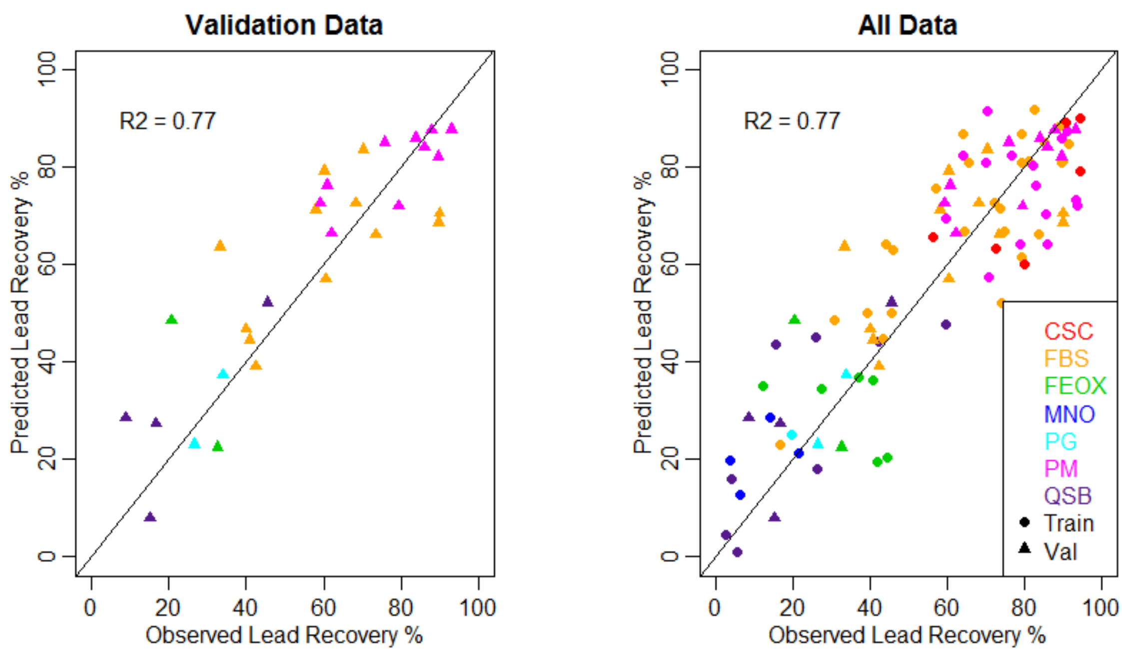


Figure 13-31: Selected model with training and validation data

The estimated lead rougher recoveries were compared to the final lead concentrate recoveries observed during locked-cycle tests for the 12 samples where both locked-cycle test and batch test results were available. To adjust the rougher to the estimated final cleaner concentrate, the LCT test recovery was subtracted from the predicted rougher recovery. This average adjustment was applied to the rougher recovery formula to estimate the predicted final recovery to concentrate. The intercept correction for transition ores is 26.3% and for non-transition ores is 0.4%.

The estimated lead rougher recoveries were compared to the final lead concentrate recoveries observed during locked cycle tests for the 12 samples where both lock cycle test and batch test results were available. The results indicate that the model developed using lead rougher results from batch testing is a good predictor of final lead concentrate recovery. A comparison of observed lead rougher recoveries from batch testing, predicted lead rougher recoveries from batch testing, and observed final lead concentrate recoveries for non-transition ores are shown in Figure 13-32 and for transition ores are shown in Figure 13-33.

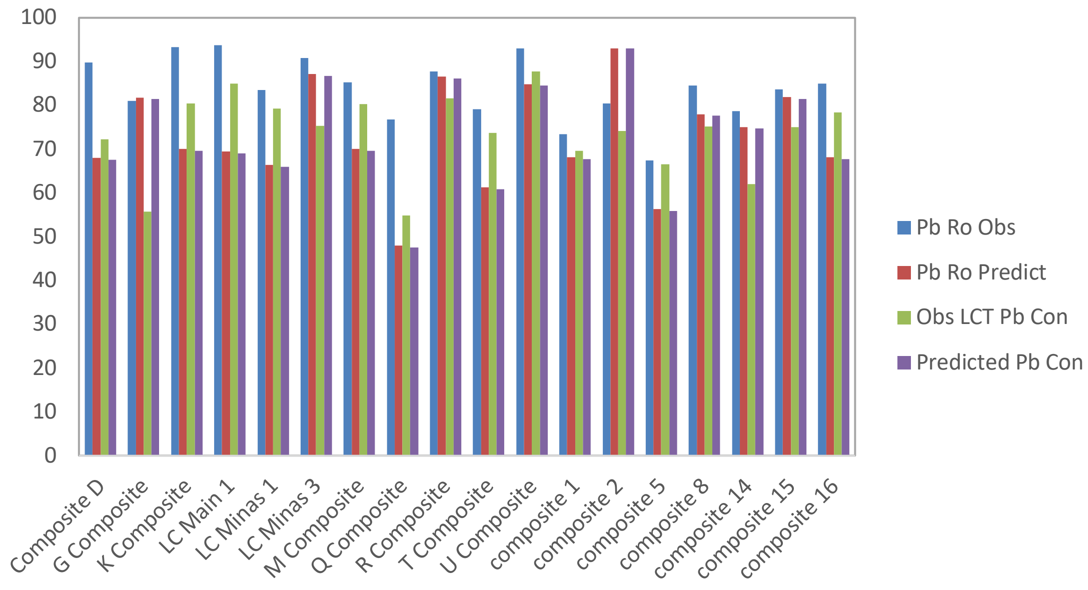


Figure 13-32 Predicted vs. observed lead recovery from LCT non-transition ores

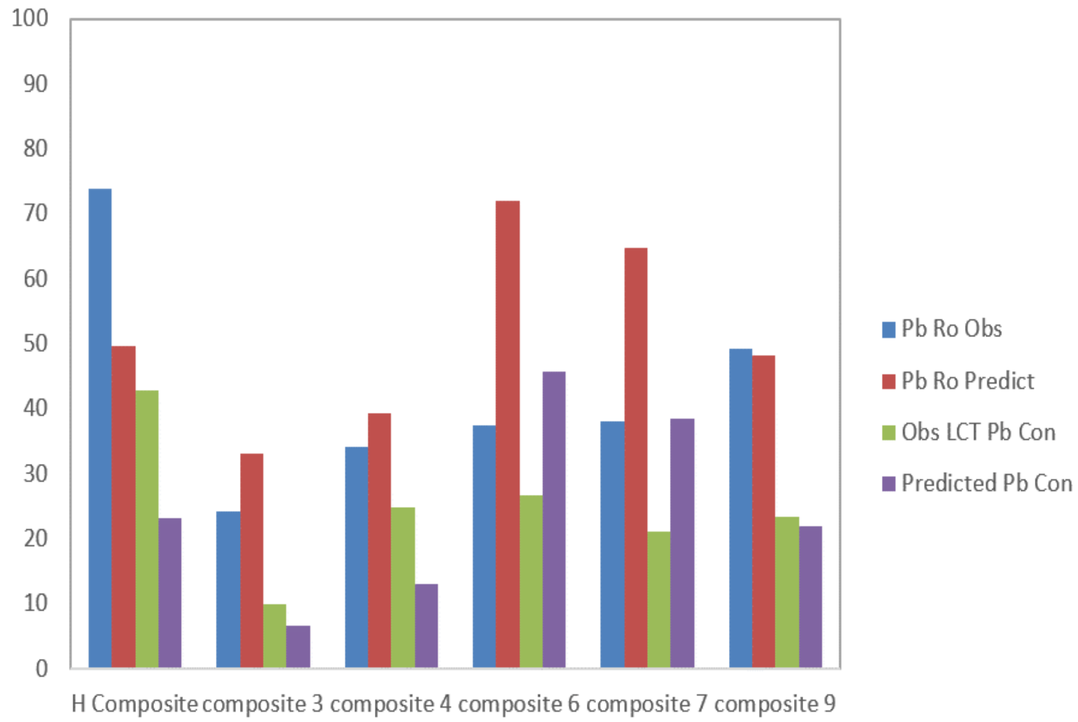


Figure 13-33 Predicted and observed lead recovery from lct transition ores

Based on the LCT analysis, the final lead recovery to the lead/silver concentrate for non-transition ores was calculated using the following equation:

$$Pb\ Recovery\ to\ Final\ Lead/Silver\ Concentrate = 61.5 - 40.9 * max(0,0.57 - zinc) + 7.7 * max(0, galena - 0.38) + 45.4 * max(0,0.37 - goethite) - 0.12 * max(0, elevation - 4891) + 32.9 * max(0,0.27 - MnOxi) - 6.21 * max(0, Pyrite - 1.07) - 16.4 * max(0,1.07 - Pyrite)$$

Also based on the LCT analysis, the final lead recovery to the lead/silver concentrate for transition ores was calculated using the following equation:

$$Pb\ Recovery\ to\ Final\ Lead/Silver\ Concentrate = 35.6 - 40.9 * max(0,0.57 - zinc) + 7.7 * max(0, galena - 0.38) + 45.4 * max(0,0.37 - goethite) - 0.12 * max(0, elevation - 4891) + 32.9 * max(0,0.27 - MnOxi) - 6.21 * max(0, Pyrite - 1.07) - 16.4 * max(0,1.07 - Pyrite)$$

These equations were used to assign final lead recoveries to the Corani block model.

13.8.4 Silver recovery

A cursory analysis of silver revealed that the strongest predictor of silver recovery was lead recovery. The relationship between silver recovery to the lead rougher and lead recovery to the lead rougher is shown below, in Figure 13-34. However, once the samples were separated by transition and non-transition samples, the recovery was shown to be more closely related to silver and zinc grade and lead recovery.

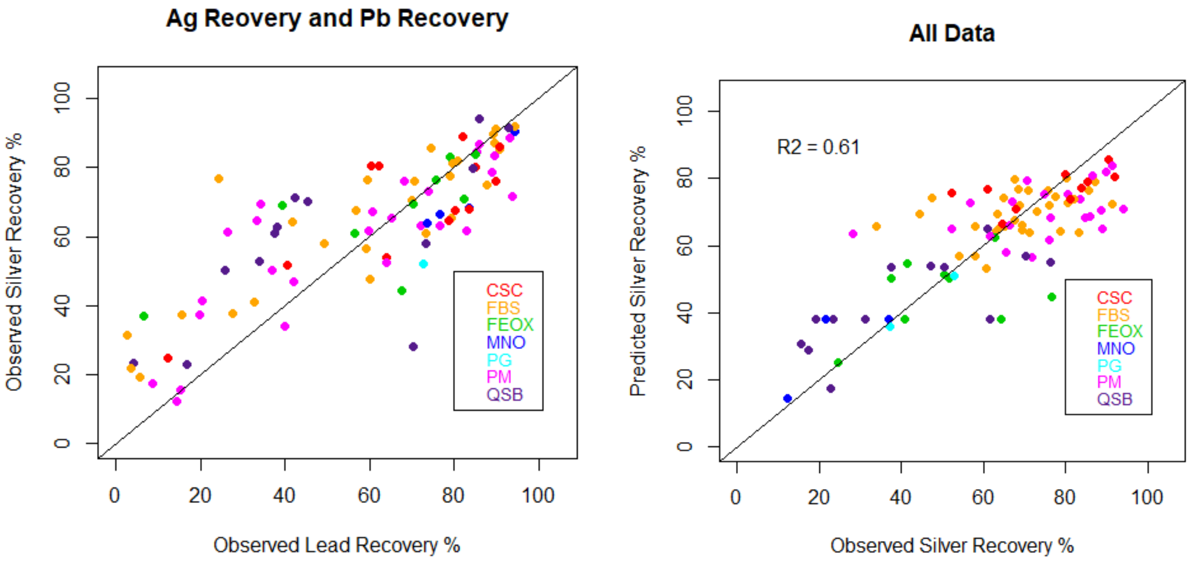


Figure 13-34 Silver recovery vs lead recovery and modelled silver recovery

The predicted silver recoveries were compared to LCT test results for samples where batch results and LCT results were available. Silver recovery to the lead rougher was estimated using *predicted* lead recovery to the lead rougher. The results show that, on average, the difference between the predicted silver recovery to the lead rougher and the observed silver recovery to the final silver/lead concentrate is on average, for non-transition ores 13.8% less and for transition ores, 11.9% less. This difference is assumed to represent the loss of silver during the lead cleaning stage. This yields an estimate of final silver recovery that closely matches the observed LCT test results for non-transition (Figure 13-35 and Figure 13-36).

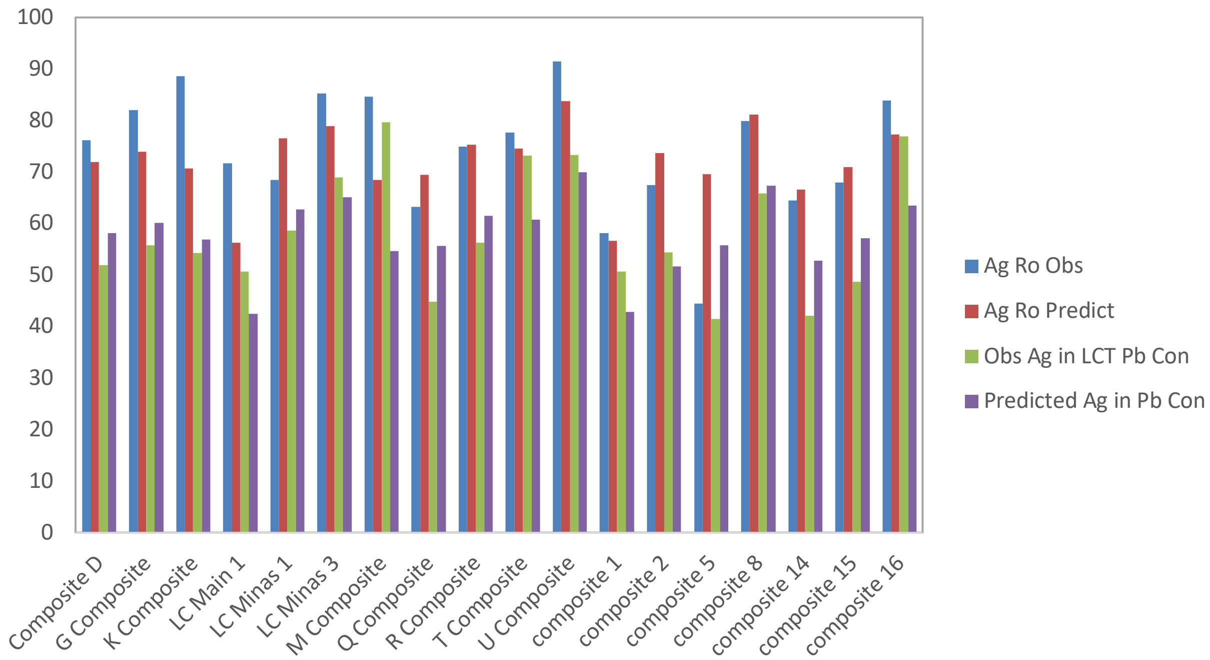


Figure 13-35 Predicted and observed silver recovery to lead concentrate for non-transition ores

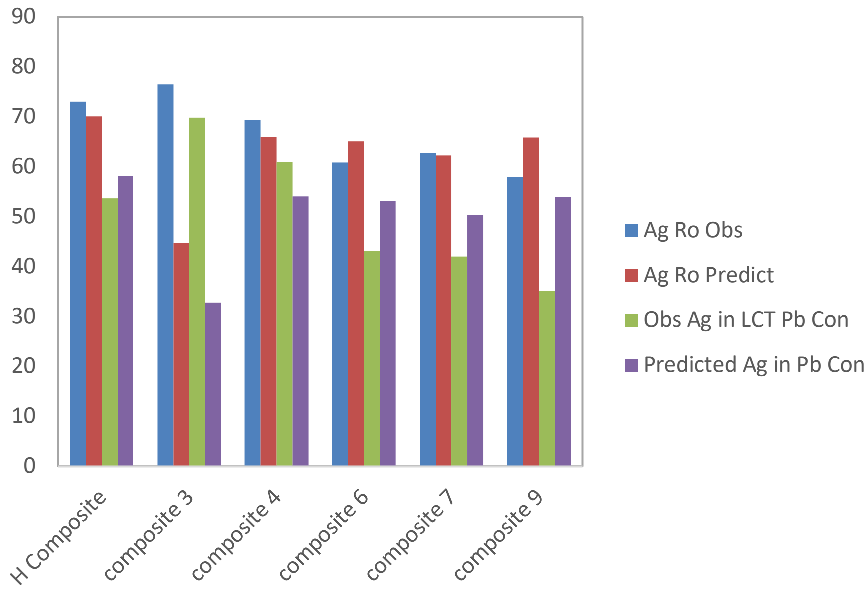


Figure 13-36 Predicted and observed silver recovery to lead concentrate for transition ores

13.8.5 Silver recovery to final lead/silver concentrate

Based on the LCT analysis, the final silver recovery to the lead/silver concentrate for the non-transition ores was calculated using the following equation:

$$\begin{aligned}&Silver\ Recovery\ in\ Lead\ Concentrate\\&\quad = 24.24 - 0.49 * max(0, 42.5 - silver - 13.68 * max(0, zinc - 1.52)\\&\quad + 17.06 * max(0, zinc - 3.16) + 13.67 * max(0, 2 - Rock) - 2.69\\&\quad * max(0, Predicted\ Lead\ Recovery - 21.54) + 13.72\\&\quad * max(0, Predicted\ Lead\ Recovery - 30.33) - 10.64\\&\quad * max(0, Predicted\ Lead\ Recovery - 33.57)\end{aligned}$$

Again, based on the LCT analysis, the final silver recovery to the lead/silver concentrate for the transition ores was calculated using the following equation:

$$\begin{aligned}&Silver\ Recovery\ in\ Lead\ Concentrate\\&\quad = 26.14 - 0.49 * max(0, 42.5 - silver - 13.68 * max(0, zinc - 1.52)\\&\quad + 17.06 * max(0, zinc - 3.16) + 13.67 * max(0, 2 - Rock) - 2.69\\&\quad * max(0, Predicted\ Lead\ Recovery - 21.54) + 13.72\\&\quad * max(0, Predicted\ Lead\ Recovery - 30.33) - 10.64\\&\quad * max(0, Predicted\ Lead\ Recovery - 33.57)\end{aligned}$$

These equations were used to assign final silver recoveries for the lead/silver concentrate within the Corani block model.

13.8.6 Zinc recovery

A total of 228 flotation tests representing 85 samples report zinc recovery to both the lead and zinc rougher. From this test database, 4 tests were eliminated as invalid based on notes on the laboratory test sheets or extremely low recovery not consistent with known sample mineralogy. These tests were also removed from the lead recovery analysis. Tests from two additional samples (LC Minas 2 and Minas 10) were removed based on inconsistencies between the zinc head grade measured prior to flotation testing and zinc grade from composited assay sample results.

Included in the dataset were four LCT tests which used soda ash as a surfactant. The tests showed a potential to improve circuit performance. More work needs to be done to evaluate the value of the addition of soda ash or an alternative pH modifier to the reagent scheme.

From the remaining tests, the test with the highest total zinc recovery (assumed to be optimized reagent scheme) for each sample was selected. The resulting dataset represents 79 samples.

From this test database, the test representing the best total zinc recovery result was selected for each sample. Also, as was done for the lead analysis, samples representing composites from multiple drillholes were removed from the potential training dataset. The resulting dataset included 58 of the original 72 samples considered for analysis.

Within the independent variable combination that was selected the most times, the most robust model (highest validation and training R^2) was selected. The final model remains similar to the 2017 model, and includes zinc grade, elevation, copper, and pyrite in the following form:

$$\begin{aligned}&Total\ Flotable\ Zinc\\&\quad = 93.9 - 50.6 * max(0{,}1.02 - zinc) - 0.15 * max(0, elevation - 4901) - 5.4\\&\quad * max(0, pyrite - 1.9) - 11.2 * max(0{,}1.9 - Pyrite) + 104.1\\&\quad * max(0, copper - 0.03) + 1620.2 * max(0{,}0.03 - copper)\end{aligned}$$

The R^2 of the training dataset was 0.92, and the R^2 of the validation dataset was 0.96. The R^2 considering all the data was 0.92. Once the model was limited to predicted zinc recoveries between 0 and 100%, the training R^2 improved to 0.93; the overall R^2 remained the same.

The performance of the model for the validation dataset and across all data is shown in Figure 13-37.

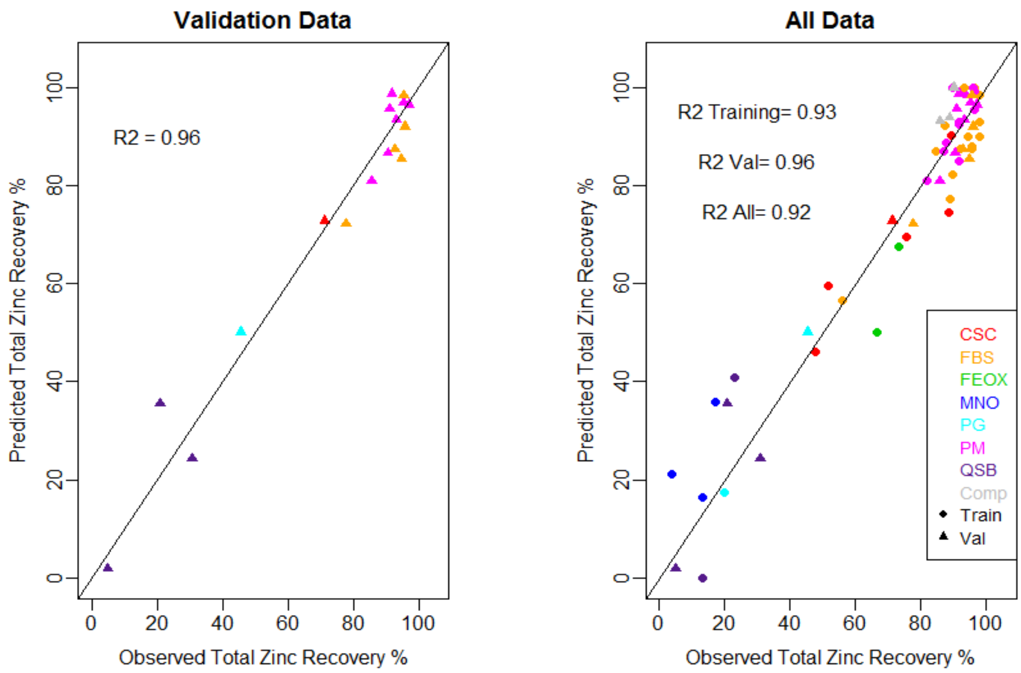


Figure 13-37: Selected model with training data and validation data

The results from the model were compared to the locked-cycle test results for the 15 samples where zinc recovery was reported. The average difference between the locked-cycle tests and the predicted total floatable zinc was 22.5%. This difference is anticipated to represent the zinc that reports to the final lead concentrate and the zinc that is lost during the zinc cleaning stage.

Three LCT test results were notable for lower-than-expected zinc recovery. The difference between the predicted total zinc recovery and the observed LCT recovery for samples K, R, and LC Main 1 were on average 34.5%, while the difference between the LCT zinc recovery and predicted total zinc recovery for the remaining samples was only 15.2%.

A comparison between the calculated final zinc concentrate recoveries from batch tests is shown in Figure 13-38. The LCT result for these samples is much lower than what was achieved in batch testing. The remaining samples show a good match between the LCT result and batch testing. Considering this, the adjustment of 22.5% to the predicted total floatable zinc may be an over-estimate of the zinc lost to the lead rougher and the zinc lost to the zinc cleaner tail.

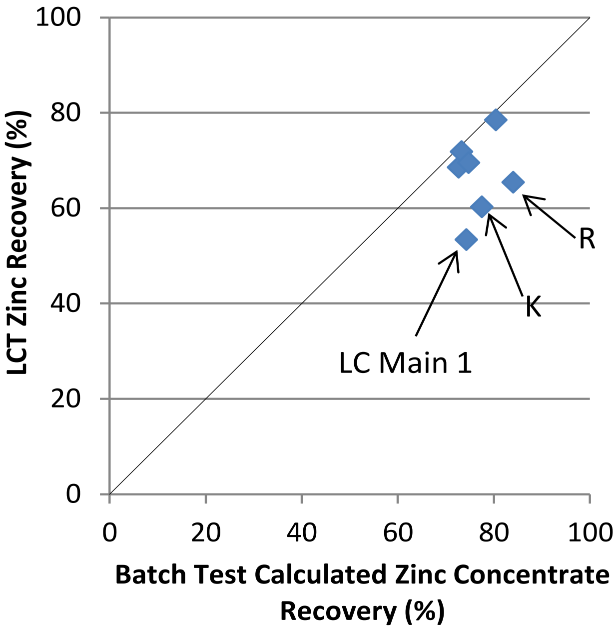


Figure 13-38: Comparison between batch and LCT results

A final zinc concentrate recovery was predicted based on the total predicted zinc recovery (Zn Ro+ Pb Ro) minus 9.5%, which is the average difference between the predicted recovery and the LCT zinc concentrate recovery result (not including samples R, K, and LC Main 1). The LCT zinc concentrate recovery, predicted total recovery, and predicted recovery to zinc concentrate are shown in Figure 13-39.

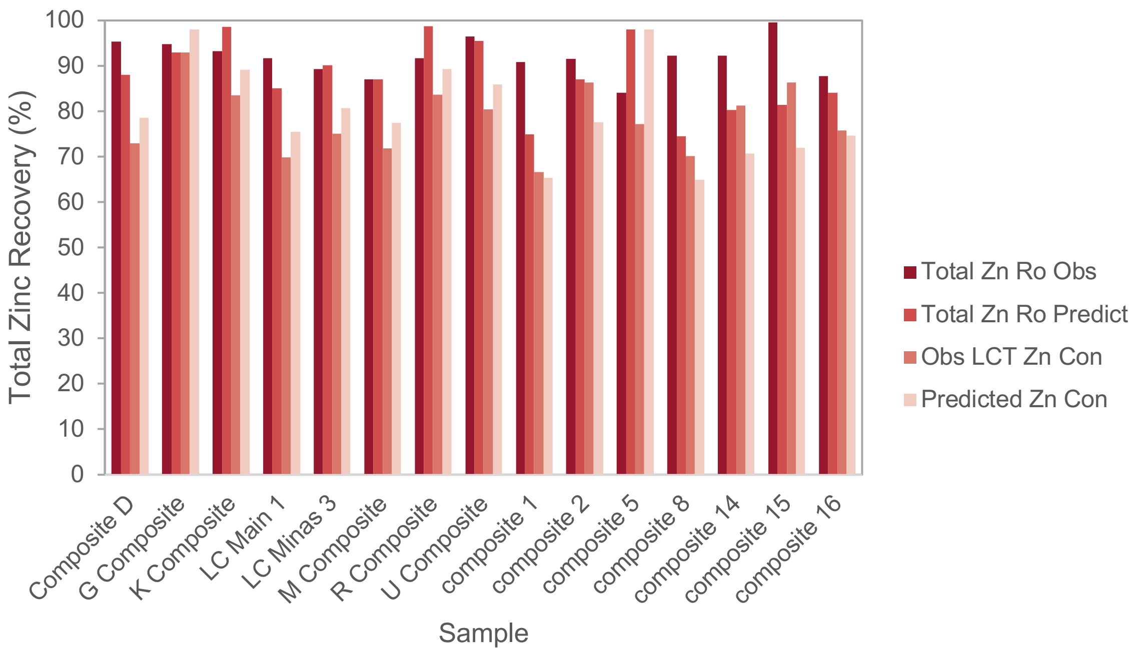


Figure 13-39: Zinc concentrate LCT v's predicted zinc grade and recovery comparison

13.8.7 Zinc recovery to final zinc concentrate

Based on the LCT analysis, the final zinc recovery to the zinc concentrate was calculated using the following equation:

$$\begin{aligned} &Zinc\ Recovery\ to\ Final\ Zinc\ Concentrate \\ &\quad = 84.4 - 50.6 * max(0{,}1.02 - zinc) - 0.15 * max(0, elevation - 4901) - 5.4 \\ &\quad * max(0, pyrite - 1.9) - 11.2 * max(0{,}1.9 - Pyrite) + 104.1 \\ &\quad * max(0, copper - 0.03) + 1620.2 * max(0{,}0.03 - copper) \end{aligned}$$

This equation was used to assign final zinc recoveries for the zinc concentrate to the Corani block model.

13.8.8 Zinc displacement to lead/silver concentrate

Based on the LCT results, the average displacement of zinc in the final lead/silver concentrate is 9%. Batch test results revealed that, in general, the lead cleaning stage was relatively effective in removing zinc from the final lead concentrate. Of the 97 batch tests used to create the lead rougher recovery model, 21 reported displacements for zinc in the lead concentrate following the cleaning stage. The average zinc displacement among these tests was 9.3%. Except for 2 outliers, most samples have zinc recoveries to lead concentrate below 15%. Over 80% of the samples have zinc recoveries to the lead concentrate of less than 10%.

From these results, a recovery of zinc to the final lead concentrate was estimated to be approximately 9%. This average recovery was used for economic analysis of the lead concentrate value.

13.8.9 Silver recovery to zinc concentrate

Based on the tests used in the total flotable zinc model analysis, there does not appear to be a strong relationship between silver recovery to the zinc concentrate and zinc grade as was previously assumed. Further analysis revealed that the strongest indicator of silver recovery to the zinc concentrate is silver recovery to the lead/silver concentrate. Regressions for silver in the lead concentrate and silver in the zinc concentrate for lock cycle tests are shown in Figure 13-40.

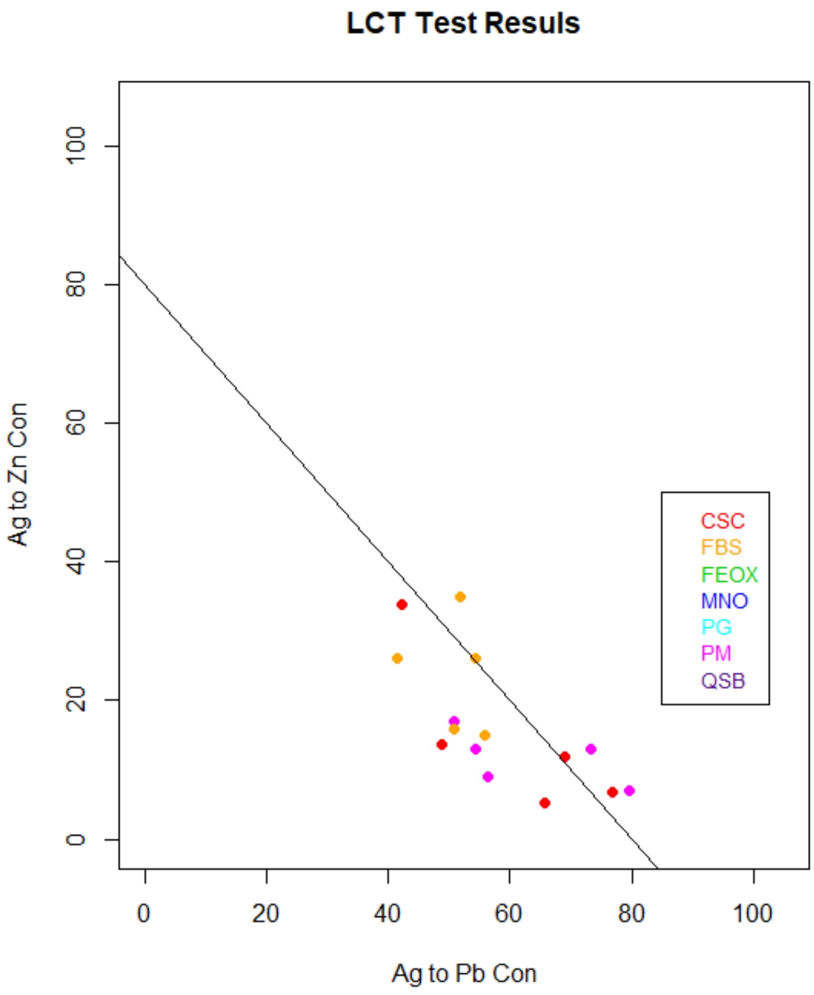


Figure 13-40: Silver to lead and zinc concentrates

Based on these results, the following equation was used to predict silver recovery in the final zinc concentrate:

$$Silver\ Recovery\ to\ Final\ Zinc\ Concentrate = 48.156 - 0.544 * Silver\ Recovery\ to\ Lead/Silver\ Con$$

Using the above relationship, the predicted silver recoveries to the zinc concentrate range from approximately 10% to about 35% and total silver recovery (Pb concentrate + Zn concentrate) is estimated to be between ~45% and 90% which is consistent with the ranges observed in batch testing.

13.8.10 Model output results

The models for recovery, along with the concentrate grade estimates, were input into the block modelling process to optimize the mining process. As discussed above, concentrate grades were set by mineral zones and uses average grades. Grades of other contained metals are calculated based on the concentration ratio. For the economic analysis, the lead concentrates were assumed to contain an average of 9% zinc. Zinc concentrates were assumed to contain an average of contained 430 g/tonne silver.Using the mine plan with revenue factor of 100, the estimates of metal recoveries were determined by mine operating year. These values are displayed for the lead concentrate and zinc concentrate in Table 13-42 and, respectively. Compared to the previous study (M3, 2011), the average lead recovery was reduced by 8% while the silver and zinc recovery both increased by 8%.

Table 13-42: Lead concentrate grades and recoveries by mine schedule

					Lead Concentrate					Zinc Concentrate				
	Mill Feed	Feed Grade				Grade		Recovery			Grade		Recovery	
Production Year	Tonnes (000)	Ag (g/t)	Pb (%)	Zn (%)	Tonnes (000)	Ag (g/t)	Pb (%)	Ag (%)	Pb (%)	Tonnes (000)	Ag (g/t)	Zn (%)	Ag (%)	Zn (%)
Year 1	8,600	99.8	1.10	0.84	97	5672	49.8	64.2	51.3	107	430	52.6	5.4	78.4
Year 2	9,882	70.9	1.04	0.77	131	3545	50.0	66.3	63.7	112	430	53.1	6.9	78.1
Year 3	9,855	77.8	1.12	0.69	129	3975	51.8	66.8	60.6	93	430	52.4	5.2	72.3
Year 4-5	19,710	65.2	1.26	0.60	243	3114	54.2	58.9	53.0	156	430	52.2	5.2	69.2
Year 6-10	49,329	38.6	0.74	0.51	395	2970	50.5	61.6	54.8	326	382	53.2	6.5	69.0
Year 11-15	41,206	38.7	0.79	0.44	399	2539	50.0	63.5	61.6	251	427	53.9	6.7	74.2
LOM	138,582	51.3	0.90	0.55	1,394	3207	51.0	62.9	57.2	1,047	414	53.1	6.1	72.3

13.8.11 Comparison of 2015 and 2019 recovery formulae

In general, improvements in predicted versus the observed recoveries for metals were seen in all areas with the evaluation of the existing and new data. The following figures demonstrate the improvement in recovery for the lead recovery (Figure 13-41), the zinc recovery (Figure 13-42) and for the silver recovery (Figure 13-43). All show an improvement in the goodness of fit (R^2 value) and show a rotation of the slope of the linear regression towards the 1:1 bisector. An R^2 value of 1 and a line laying along the 1:1 bisector would demonstrate a perfect fit.

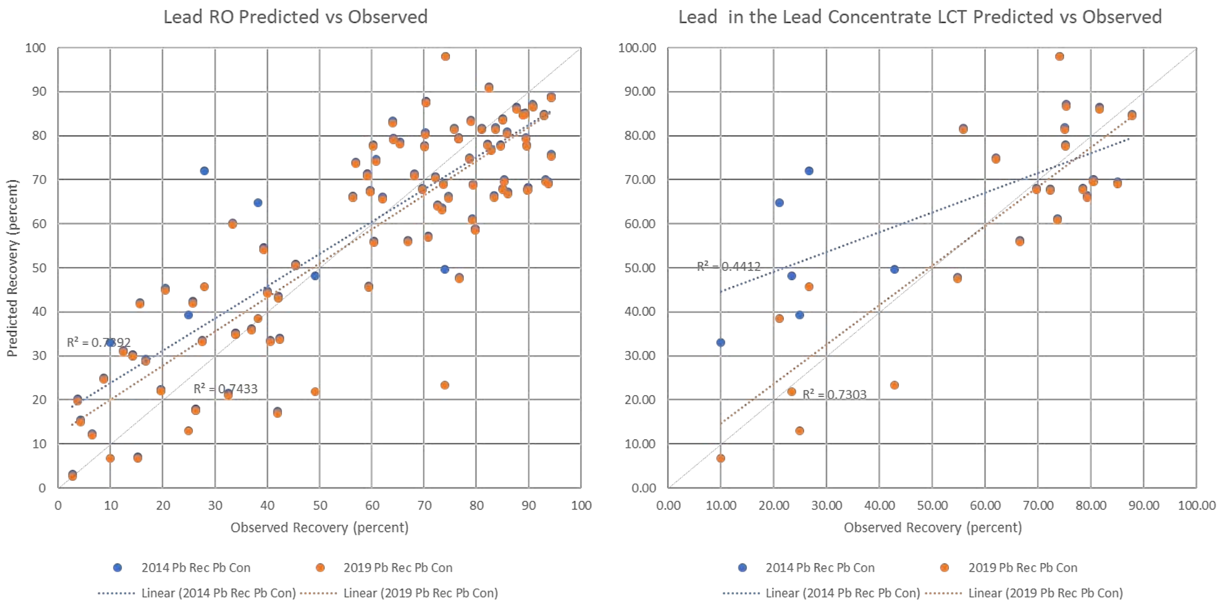


Figure 13-41: Lead recovery to the lead rougher and locked-cycle test concentrates

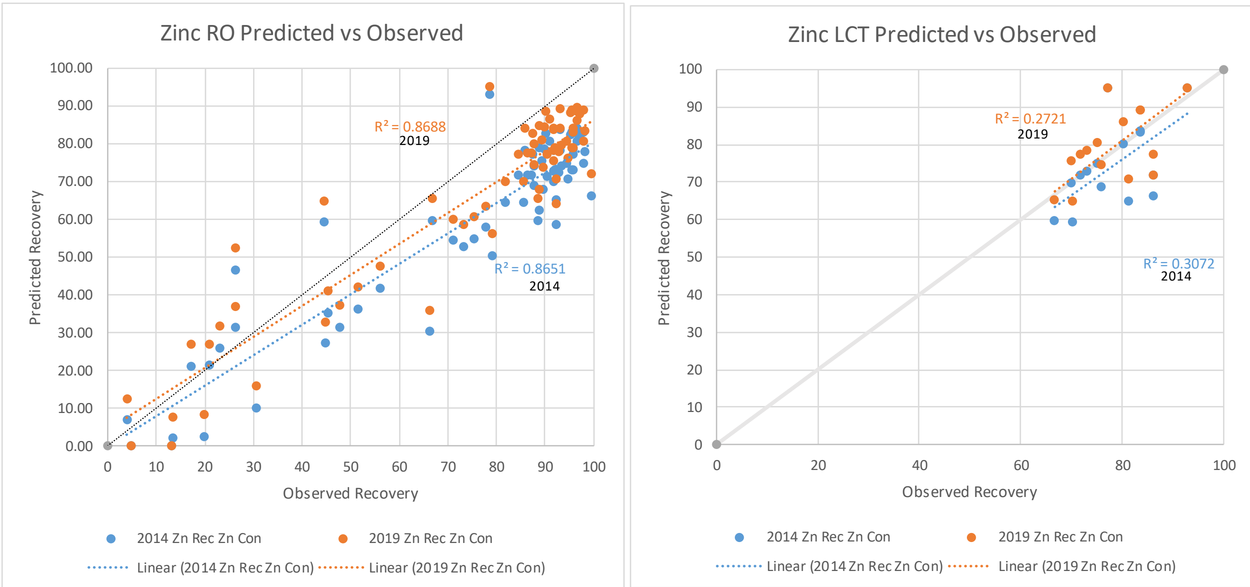


Figure 13-42: Zinc recovery to the zinc rougher and locked-cycle test concentrates

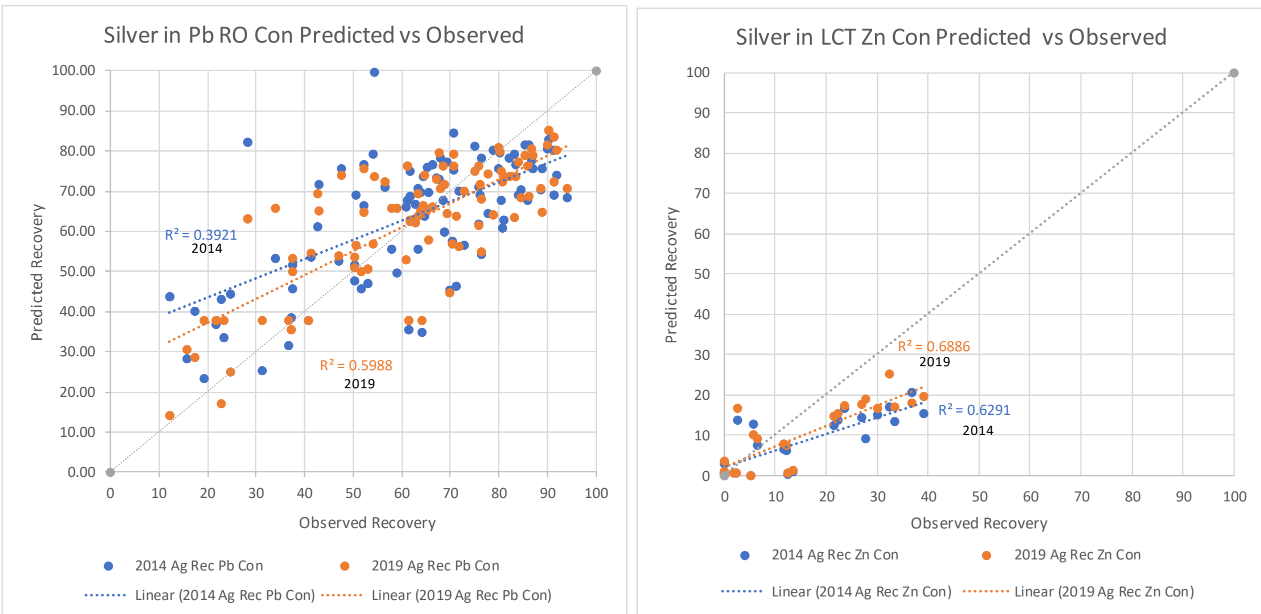


Figure 13-43: Silver recovery to the lead rougher and zinc locked-cycle test concentrates

13.8.12 Geometallurgical modelling summary

This geometallurgical study uses test work for all metallurgical conditions. Future test work done under optimal conditions may improve the metallurgical performance.

Zinc grade and elevation are the strongest indicators available in the block model for lead rougher recovery and total zinc recovery.

Parameters recorded in the geologic logs (pyrite, goethite, MnOx, and galena) improve the ability to predict lead rougher recovery and total zinc recovery.

Lead recovery is the strongest predictor of silver recovery to the lead/silver concentrate.

Improvements to the method of predicting transition resource material, such as optical or near-infrared and infrared hyperspectral core scanning, may help improve the understanding of the full extent of the transition zones and may help develop an improved recovery model similar to the results seen using QEMSCAN data, rather than geologic log data.

13.9 Interpretation

Since the 2017 Technical Report, metallurgical tests have been conducted to assess key aspects of materials handling as well as tails thickening and filtration.

13.9.1 Crushed ore bulk flow characteristics

Jenike and Johanson flow property test results for crushed ore received in May 2019 indicated a strong tendency to form stable ratholes. Based on the test results, the live capacity of the stockpile reduced from the nominal target of 12 hours to 2.6 hours which is not desirable from a reclaim perspective. These ratholes will require breaking down mechanically to improve the live capacity of the stockpile once processing of ore with poor flow characteristics is completed.

13.9.2 Tailings thickening

The tailings thickeners were selected to optimize downstream filtration rates and produce an underflow pumpable by conventional centrifugal pumps (without shear thinning systems).

Thickening testwork was completed by Outotec and included dynamic testwork on six samples in 2014, dynamic testwork on a single composite sample in May 2019, and 190 mm semi-pilot testwork on a composite sample in June 2019.

Static flocculant screening tests in 2014 determined that Orifloc 1021 was the preferred flocculant. Results from the dynamic thickening results are presented in Table 13-43.

Table 13-43: 2014 dynamic thickening results

Sample Name	pH	Solids Loading Rate (t/m^2h)	Rise Rate (m/h)	Flocculant Dosage (g/t)	Achievable Underflow Density (%w/w solids)	Achievable Overflow Clarity (ppm TSS)	Maximum Unsheared Underflow Yield Stress (Pa)
CM-03A	11	0.20-0.50	1.4 – 3.5	10 – 25	55 – 63	<100	153
CM-04	11	0.20-0.50	1.8 – 4.4	10 – 35	52 – 61	<110	128
CM-06	11	0.20-0.50	1.5 – 3.8	10 – 30	57 – 65	<100	186
CM-08	11	0.20-0.50	1.8 – 4.5	15 – 25	47 – 57	<100	174
CM-02	11	0.20-0.50	0.9 – 2.2	5 – 15	63 – 70	<100	148
CM-12	11	0.10-0.50	0.1 – 3.6	10 – 25	46 – 59	<100	172

The testwork indicates that sample CM-02 has the best settling characteristics and CM-08 has the worst settling characteristics with a high yield stress at a low underflow density.

The optimum flocculant for the 2019 composite sample was PHP 40 PLUS, dosed at 0.025% w/v into diluted feed at a density of 12% w/w. Dynamic testwork was completed at flocculant dosages between 15 g/t and 25 g/t, and achieved underflow densities between 50 and 57% w/w solids at yield stresses between 30 and 100 Pa. An overnight compaction test was completed and indicated that underflow densities as high as 64% w/w solids could be achieved at a yield stress of 160 Pa. Dynamic testwork also indicated a relationship between increasing feed slurry pH and underflow density.

Semi-pilot tailings thickening tests were conducted by Outotec in a larger 190 mm test unit to confirm the thickening performance and underflow rheology for high rate thickener (HRT), high compression thickener (HCT) and paste thickener (PT) options. Testwork indicated that at a solids loading rate of 0.50 t/m^2.h, the HRT could achieve 48-55% w/w solids, the HCT could achieve 51 – 58% w/w solids, and a paste thickener could achieve between 55 and 62% w/w solids. The semi-pilot test results are broadly consistent with the dynamic thickener results on the same sample. The testwork also revealed a rapid increase in yield stress beyond 60% w/w solids. All thickening testwork was completed in the original process water.

To achieve the design duty, two 42 m diameter high compression thickeners were selected.

13.9.3 Tailings filtration

Standard Proctor compaction tests were completed by MEG Geotechnical Laboratory in 2014. These results are plotted in Figure 13-44.

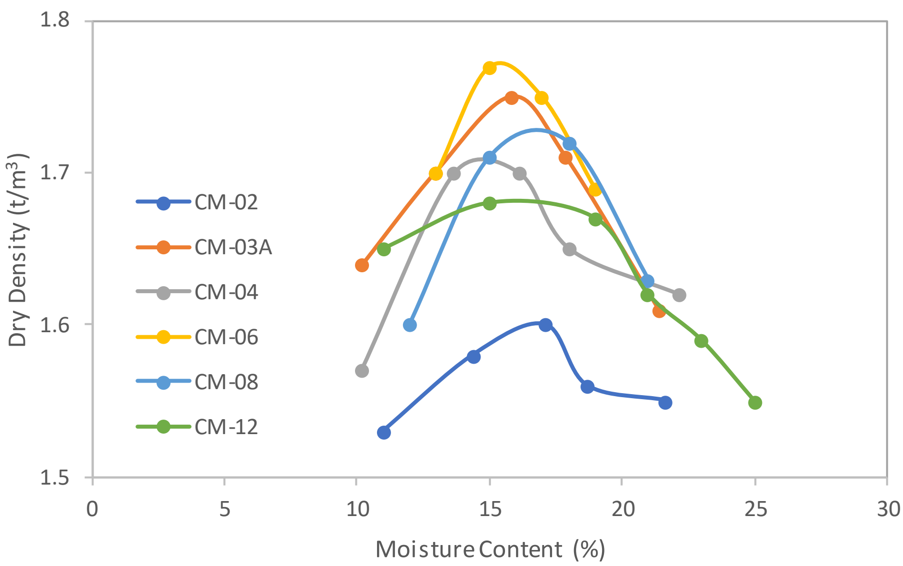


Figure 13-44: Standard proctor compaction tests

The target filtration moisture targets were selected based on 95% maximum dry density (MDD), as presented in Table 13-44.

Table 13-44: Filtration moisture targets

Sample	Optimum Proctor Moisture (%)	Proctor MDD (t/m^3)	95% Proctor MDD (t/m^3)	Moisture @ 95% Proctor MDD (%)	Design Moisture (% w/w)
CM-02	16.5	1.6	1.52	22	18.0
CM-03A	16	1.74	1.65	20	16.7
CM-04	15	1.72	1.63	20	16.7

Sample	Optimum Proctor Moisture (%)	Proctor MDD (t/m^3)	95% Proctor MDD (t/m^3)	Moisture @ 95% Proctor MDD (%)	Design Moisture (% w/w)
CM-06	16	1.78	1.69	19	16.0
CM-08	16.6	1.73	1.64	21	17.4
CM-12	16	1.68	1.60	23	18.7

Tailings filtration testwork was completed in 2014 in a Labox 100 test unit with 50 mm chambers on the same six thickened samples. Additional filtration testwork was completed in 2019 also in a Labox 100 test unit on a thickened composite sample. The filtration testwork parameters were averaged and adjusted to cater for expected feed densities from the selected thickener and target moistures based on 85% saturation of the filter cake. The key filtration parameters and filter throughputs (for the nominated Outotec FFP-3512 74/74 filter) are presented in Table 13-45 and

Table 13-46.

Table 13-45: Filtration testwork summary (parameters)

Year	Sample	Feed Density (% w/w)[1]	Cake Thickness (mm)	Feed Time (min)	Pressing Time (min)	Air Blow Time (min)	Total Cycle Time (min)
2014	CM-02	62	40	2	0.5	1	8
	CM-03A	56	34	3	1.5	2.3	11.3
	CM-04	54	35	4	1.5	2	12
	CM-06	57	38	3	1.5	2	11
	CM-08	49	33	3	1.5	2	11
	CM-12	54.5	34	4	1.5	1	11
2019	Composite	55	37	3	1.5	1	10

Table 13-46: Filtration testwork summary (throughput)

Year	Sample	Dry Cake Density (kg/m^3)	Expected Moisture (% w/w)	Filter Cake Saturation (%)	Filter Throughput (t/h)[2]
2014	CM-02	1,568	14.5	66	208
	CM-03A	1,758	16.7	95	140
	CM-04	1,714	16.5	97	133
	CM-06	1,710	15.8	80	157
	CM-08	1,606	16	74	128
	CM-12	1,588	18.4	85	130
2019	Composite	1,461	17	70	144

1. Feed densities estimated based on a thickener underflow yield stress of 50 Pa, representing robust and continuous operation for a high compression thickener.
2. Throughputs calculated based on an Outotec FFP-3512 74/74

The thickening results were scaled up to a nominated full size filter (FFP-3512) and the outcomes are presented in Figure 13-45. This graph indicates that samples that settled to high underflow densities also filtered at higher throughputs, which is typical. Based on these results, the 2019 sample represents the average of the 2014 samples well.

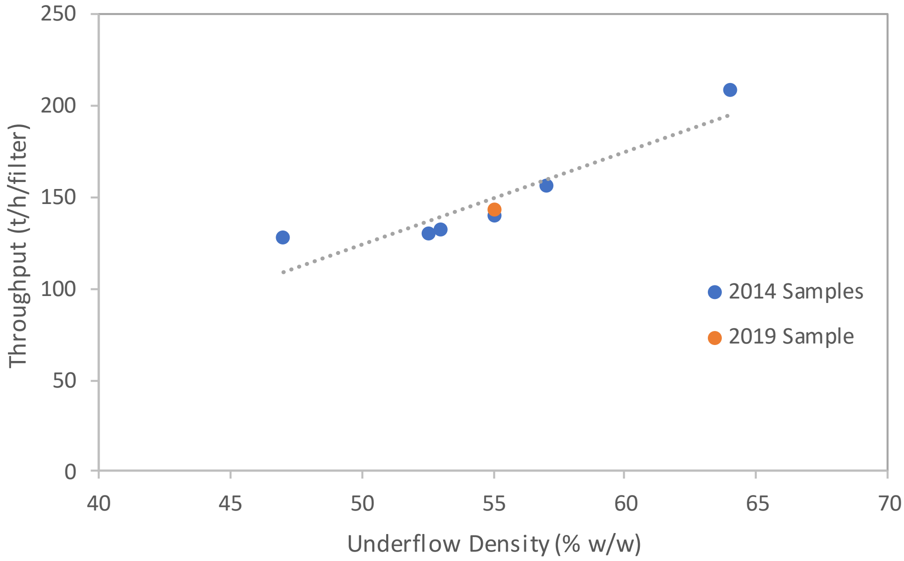


Figure 13-45: Filter throughput

To achieve the design duty, 10 Outotec Larox FFP 3512 automated vertical plate filters were selected. Approximately 8.4 filters will be required to operate when the plant is running, leaving 1.6 filters on average offline for maintenance under typical operation. Planned maintenance will be scheduled to occur weekly as well as during site shutdowns. The filters are arranged in two trains of five filters each.

14 Mineral Resource Estimates

GRE updated the Mineral Resources for the Corani Project with new drilling completed in 2019. This drilling added six holes to the database used for estimation.

A block model of the Corani deposit was developed to be the basis for determination of the Mineral Reserves and Mineral Resources. This section summarizes the development of the block model as well as the development of the mineral resource. BCM's block model was updated by GRE to include drill hole data gathered since the 2015 model was created. The drill hole database was updated with geologic logs and assays of primary recovery indicators: copper, goethite, manganese oxide, pyrite, and galena. These new geometallurgical indicators were modelled along with the economic metals in the block model.

14.1 Block Model

The Corani block model produced for the 2011 Feasibility Study (FS) (M3, 2011) and the 2015 FS (M3, 2015) was used as the basis for the updated block model created for the 2019 update (2019 block model). Since 2011, several block model parameters have been re-evaluated and updated based on a more detailed analysis of grade, mineralization types, geometallurgy, and acid rock drainage. The 2019 model uses the updated drill hole database including the six additional drill holes drilled subsequent to the development of the previous database. An indicator field was added to the model to estimate the extent of the transition material.

The following sections describe the development of the current block model. Where there are no major differences in assumptions or estimation between the previous and updated block model, a brief summary of the previous work is given. Where there are significant differences between the models, a more fulsome discussion is presented. This is intended to help the reader understand the construction of the updated block model.

The model area is divided into three areas shown in Figure 14-1: Main, Minas, and Este. The model is large enough to contain all reasonable open pit configurations for the three resource areas. The total model size and block size are summarized in Table 14-1.

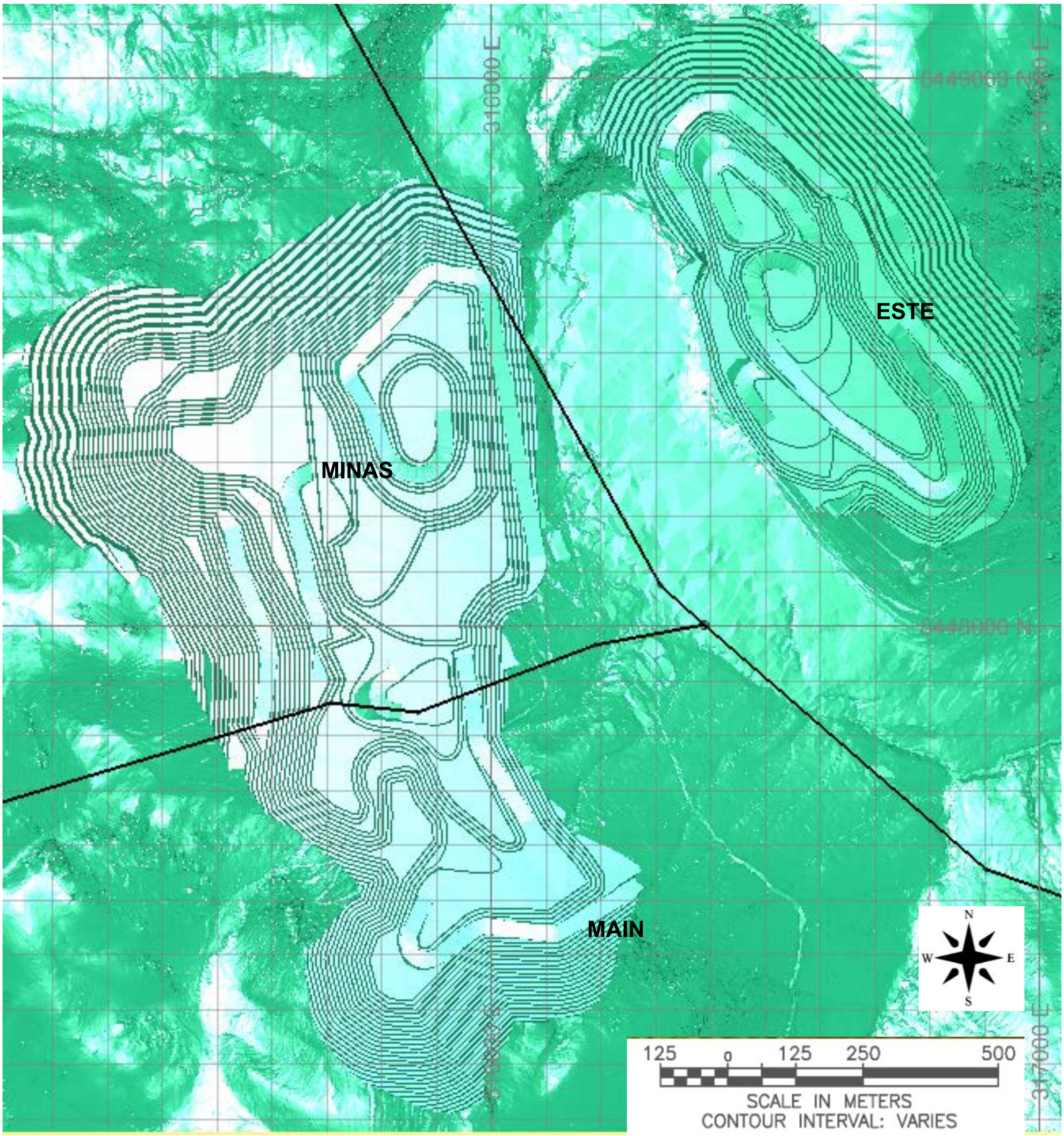


Figure 14-1: Resource areas with final pit configuration

Table 14-1: Block Model Information

Parameter	X	Y	Z
Block Size (m)	15	15	8
Number of Blocks (m)	184	214	64
Model Limits	314,745	8,446,250	4,618
	317,505	8,449,460	5,130

The model was assembled in the UTM (Zone 19S, Datum PSAD56) coordinate system and is parallel to the UTM grid. Topographic information was assigned to the model based on topographic maps provided by BCM. The topography was consistent with field observations and the elevations of the drill holes collars.

14.1.1 Lithology boundaries

The lithology solids define the general geology observed for the Corani deposit, including mineralized tuff, post mineral tuff, basement sediments, and cover. GRE has continued to use the lithology assignments from the 2011 block model in each resource estimate, including the current estimate.

Cross sections of the drill hole data in each of the zones (Este, Minas, and Main) were developed which confirmed the observations regarding the distribution of mineralization within the Corani rock units. Post-mineral volcanic tuffs were thought to be barren; however, check samples by GRE in 2017 showed potentially economic silver mineralization. GRE recommended that BCM undertake an assaying program of existing core for the post-mineral tuffs. The basement metasediments are generally barren at Corani, although there are also local occurrences of mineralization within those units. Potential exists for additional mineralization to be found in the sediments. The mineral reserves and resources are completely contained within the pre-mineral tuffs.

The drill hole geologic logs provided by BCM personnel to develop wire frame surfaces for: 1) top of sediments (bottom of pre-mineral) and 2) top of pre-mineral volcanics. These surfaces were extrapolated beyond the outside drill holes a sufficient distance so as not to limit the ability to assign rock type or grade when grade estimation was completed.

The wire frame interpretation was checked by plotting detailed cross sections through each of the three deposits. The rock type interpretation was then checked against the logged drill hole data by GRE staff and BCM geology and engineering staff. In 2019, only minor corrections were made and the check repeated.

The wire frame surfaces were used to assign a code to the block model on a nearest whole block basis. Blocks were coded as: 1 = cover (including bofedal), 11 = post-mineral volcanics, 20 = pre-mineral volcanics, and 31 = sediments. Unassigned blocks do exist outside of the interpreted area. The blocks that did not receive a rock type assignment were assigned a default average density in case they were mined as waste within the mine plan.

Figure 14-2 is an example of the geologic assignment of the mineral domains to the model blocks on cross-section.

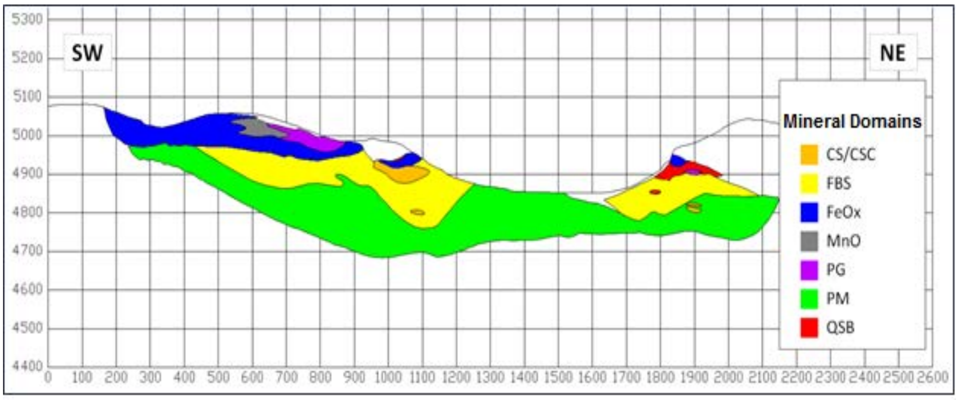


Figure 14-2: Geologic assignment of mineral domains to model blocks

14.1.2 Density assignment

Density was based on the combined grade of silver, lead, and zinc. The silver grade was converted to a percent assay to have common units with lead and zinc. The combined grade and density showed a break at 0.9381%. Blocks with a combined grade under 0.9381% were assigned a density of 2.31 tonnes per cubic meter (t/m^3), and blocks with a combined grade greater than or equal to 0.9381% were assigned a density of 2.43 t/m^3. Post-mineral tuff with no metal grades to analyse was assigned an average density of 2.3 t/m^3, and non-tuff material was assigned an average density of 2.53 t/m^3.

Table 14-2: Assigned block densities

Rock Type	Grade (Pb + Zn %)	Density t/m^3
Pre-Mineral Tuff	< 0.9381	2.31
Pre-Mineral Tuff	>= 0.9381	2.43
Post-Mineral Tuff	Not Applicable	2.3
Other Materials	Not Applicable	2.53

14.1.3 Block grade estimation

The block model grades were updated for this study incorporating the six new holes.

For the updated current estimate, the mineralization types were divided into a sulfide group, an oxide group, and a sulfide group with quartz and barite. This resulted in Mineral Groups 1, 2, and 3. Mineral Group 1 consisted of the following mineral types:

- CS – Coarse sulfide
- CSC – Coarse sulfide and celadonite
- FBS – Fine black sulfide
- PM – Pyrite marcasite
- TET – Tetrahedrite.

Mineral Group 2 consisted of the following mineral types:

- FeOx – Iron oxide
- MnO – Manganese oxide
- PG – Plumbogummite

Mineral Group 3 consisted of mineral type QSB – crystalline quartz sulfide barite. Although QSB had similarities to the first group, it was geographically distinct from the sulfide-rich zones.

The Mineral Groups were further sub-divided by zone (Corani Este, Corani Minas, and Corani Main), creating a total of 12 grade domains. The grade domains were determined through a process of iteratively testing different combinations of zone and mineral groups; this included considering groups individually and all groups lumped together. During this process, some continuity within the Minas and Main mineralization was apparent. In the future, some of the upper mineralization zones like QSB could be modelled as a continuous body through Minas and Main. Since the QSB is likely a later mineralization, it may have formed as a continuous body.

14.1.4 Compositing

Prior to compositing, individual assay values were cut to limit the influence of high-grade outliners on the block grade distribution. The cumulative probability distribution (CPD) of silver has a break at 1,410 parts per million (ppm), and 12 silver samples were capped at 1,410 ppm (Figure 14-3). Although there was a definite break on the copper CPD, it was only used as a recovery indicator, and the break was well above the indicator threshold. Thus, there was no need to cap copper assays. There was no cumulative frequency analysis on the goethite, MnOx, pyrite, and galena labels since there was no assay on those labels.

The cumulative frequency plot of lead manifests itself as a continuous population with no discernible break. For this reason, lead was not capped before compositing. The CPD of zinc shows a continuous population like lead; therefore, it was not capped before compositing.

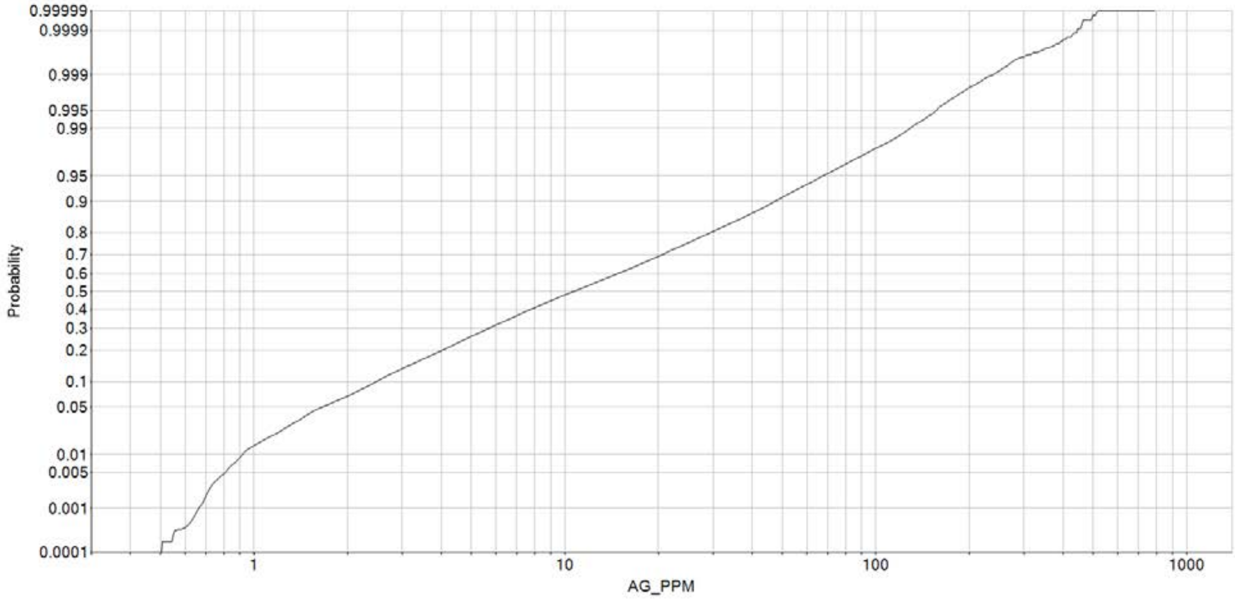


Figure 14-3: Cumulative frequency plot of silver grade (g/tonne) with break and capping

Within each drill hole, assays were composited within mineralization types. Those composites were created as equal-length composites of approximately eight meters, so that all composites within each continuous mineralization interval had the same length. The length may vary from eight m in order to define composites of equal length within a mineralization interval. This process eliminated the existence of a short or partial composite at the mineralization boundary.

The eight-m value was selected based on a grade-versus-count evaluation of alternative composite lengths, to determine whether there was an improvement in ore selectivity with smaller composites versus the cost of production, compared with more cost-efficient use of longer composites and corresponding higher bench heights. The final selection of eight m was also guided by the expected production rates and mine loading equipment that might be employed for production.

14.1.5 Variography

Composite samples for silver, lead, zinc, copper, goethite, manganese oxide, pyrite, and galena were loaded into Leapfrog Geo to verify the prior variogram analysis. The composites were divided into groups for each resource area (Main, Minas, and Este) and subdivided into mineral groups.

These groups were created based on the relative locations of the mineral domains to each other. This created nine composite groups for the eight grade labels, resulting in 72 total variograms to model. No major changes were seen in the variography.

It was determined that a correlogram created the best fit for this set of data. Downhole correlograms were used to determine the nugget for each composite sample group. Experimental correlograms had search parameters set for 30 degree increments for azimuth and 15 degree increments for dip. Omnidirectional correlograms were created for grade zones that did not have a distinct directional trend. These grade zones were modelled with isotropic parameters. The rest were modelled using anisotropic parameters for block modelling.

Typically, the general trend of all the estimation zones followed the trend of the contact between pre-mineral tuff and post-mineral tuff, which has an azimuth of 300° to 330° and a dip of 15° to 30°. This is in agreement with BCM geologists' conceptual understanding of the deposit.

Figure 14-4 through Figure 14-24 present the correlograms for the three economically viable metals in each deposit zone and mineral domain (where applicable). Table 14-3 through Table 14-5 show the parameters for input to the modelling software for all eight grade labels in the block model. These parameters were taken from the variography of the composite data. Notably, the search range was increased from the previous study. The range from the variography analysis showed that the model has a range of 180 m in two zones and 170 m in another zone for the silver models. Since silver is the primary metal in this Project, a search range of 180 m was used for all grade labels to preserve uniformity of the number of blocks modelled.

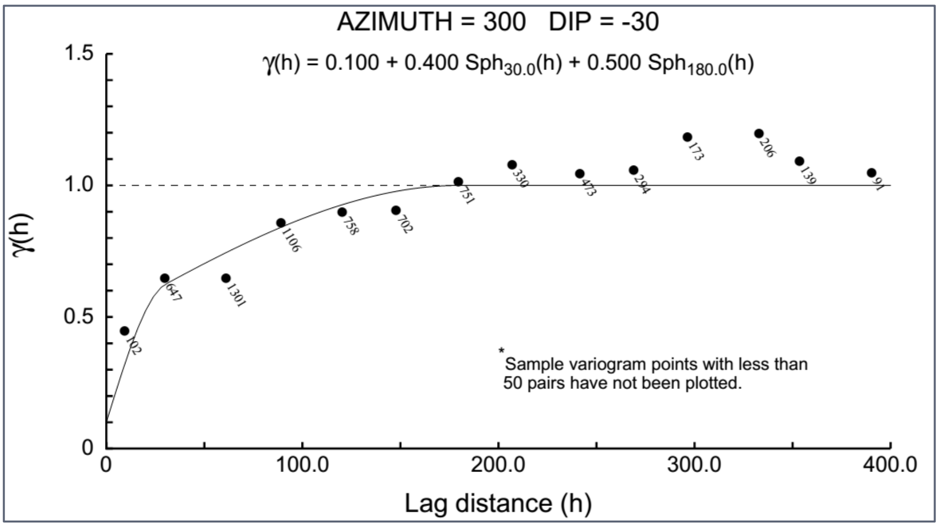


Figure 14-4: Silver correlogram – Este

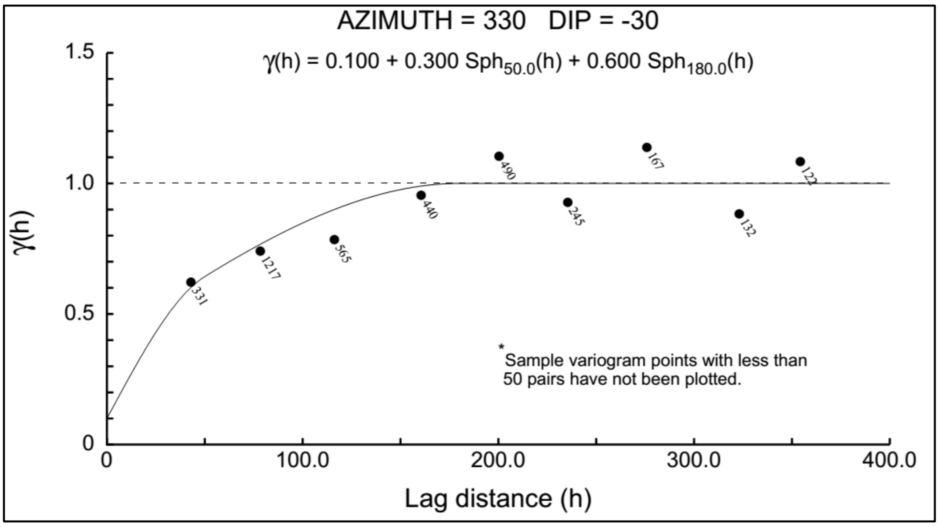


Figure 14-5: Silver correlogram – Main

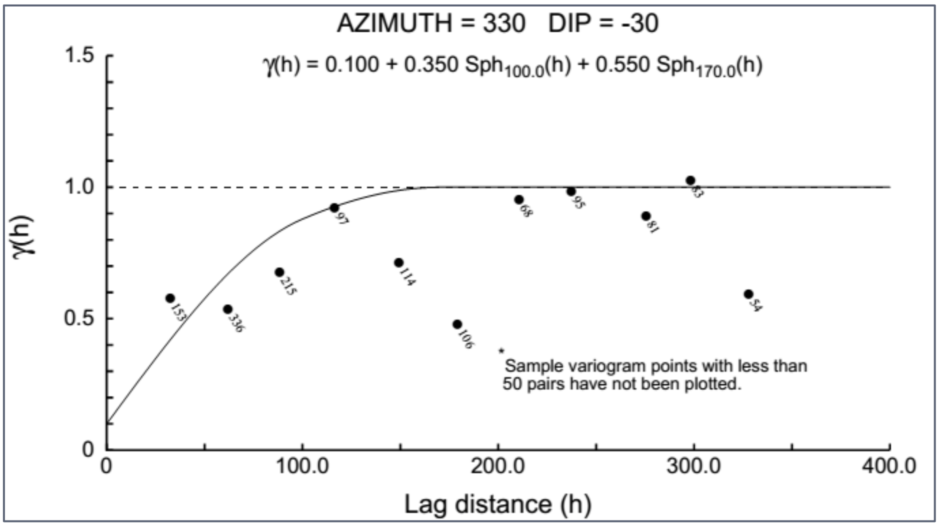


Figure 14-6: Silver correlogram – Minas

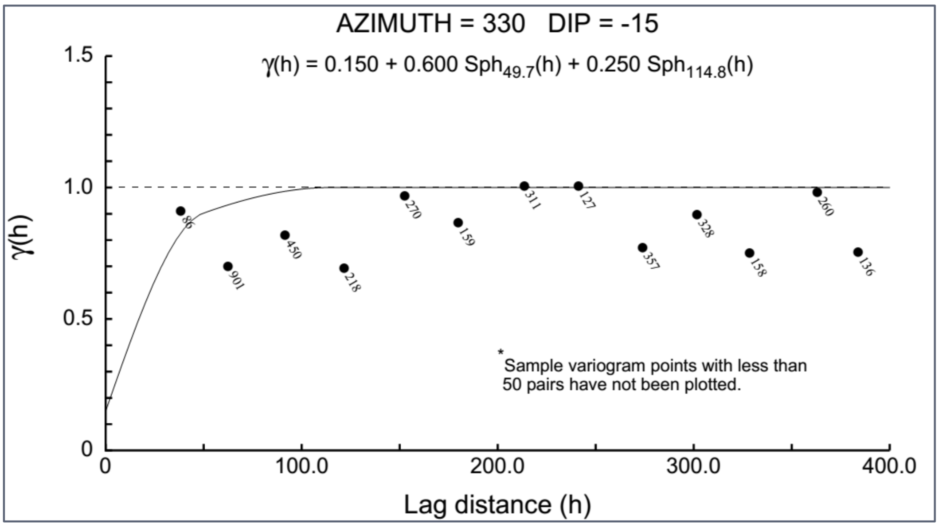


Figure 14-7: Lead correlogram – Main, Mineral Group 1

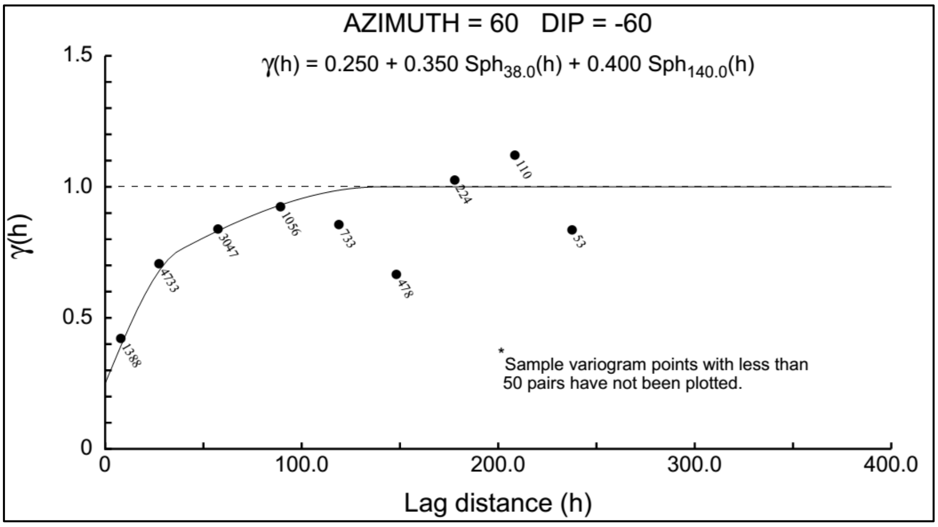


Figure 14-8: Lead correlogram – Main, Mineral Group 2

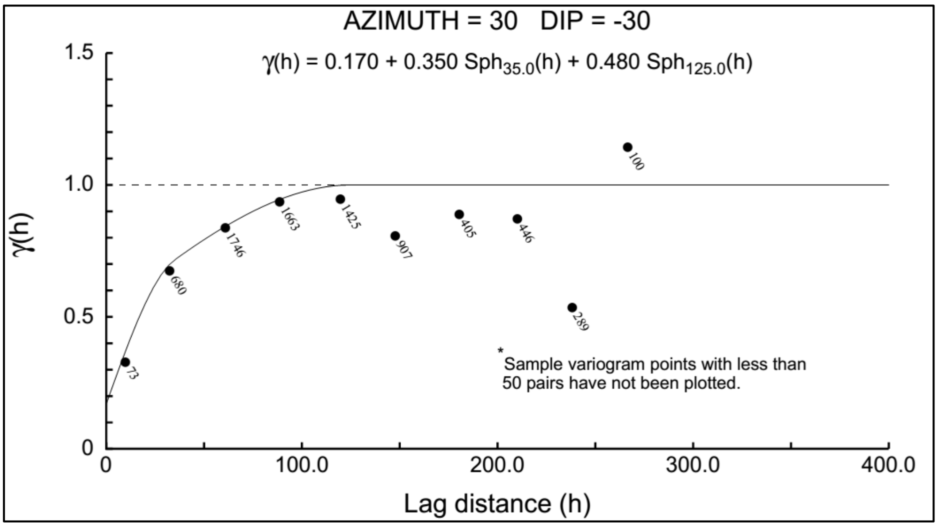


Figure 14-9: Lead correlogram – Main, Mineral Group 3

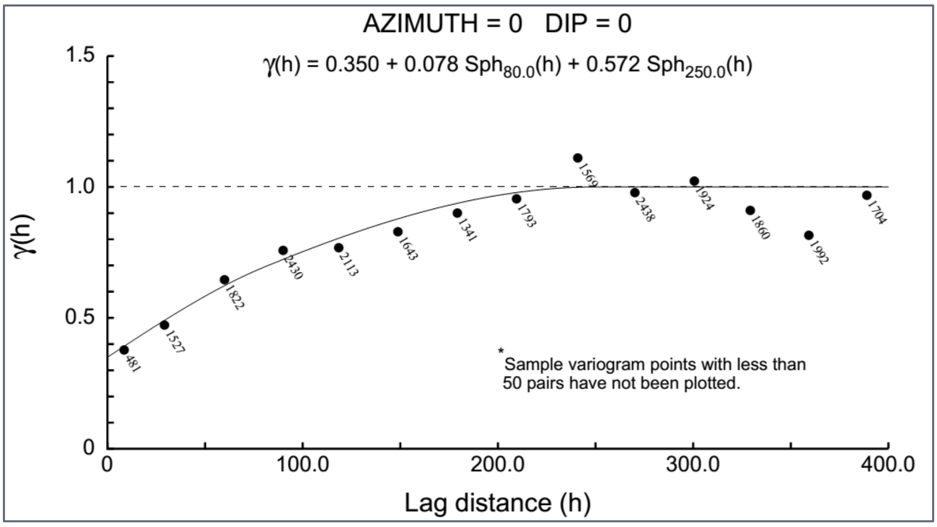


Figure 14-10: Lead correlogram – Minas, Mineral Group 1

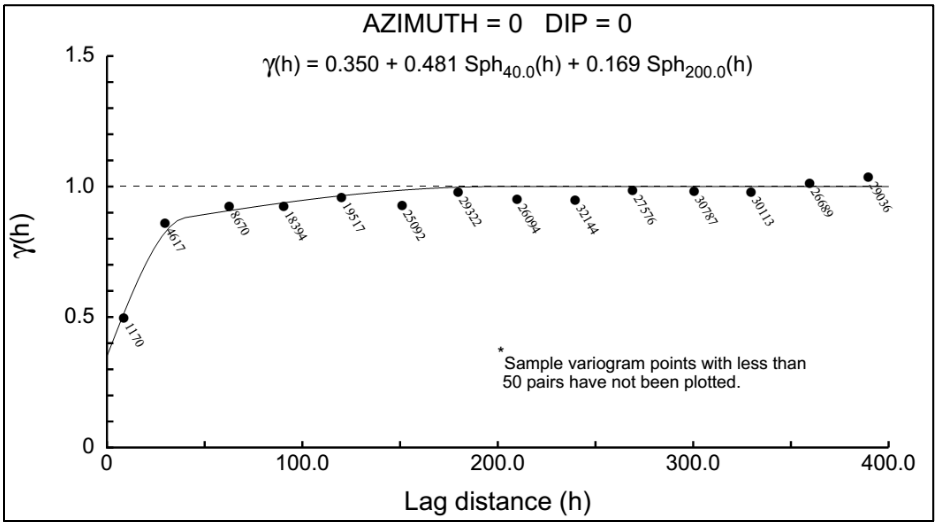


Figure 14-11: Lead correlogram – Minas, Mineral Group 2

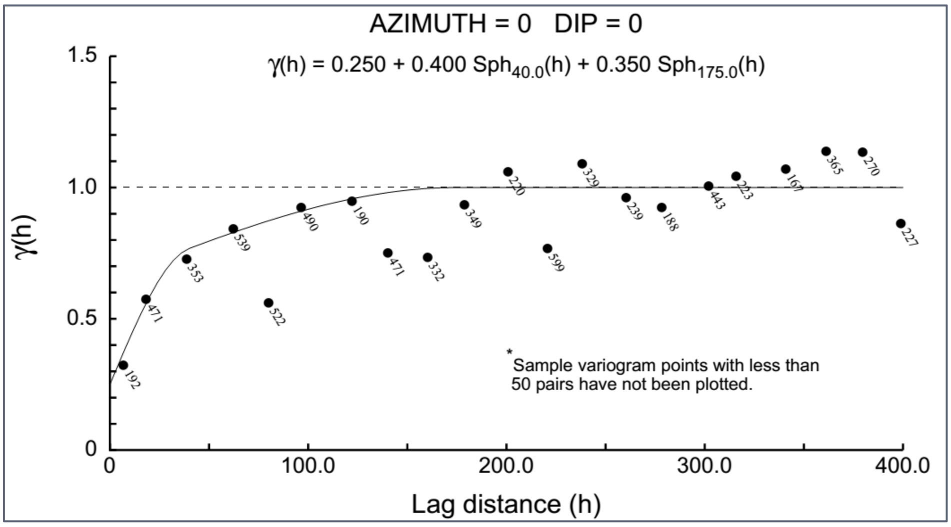


Figure 14-12: Lead correlogram – Minas, Mineral Group 3

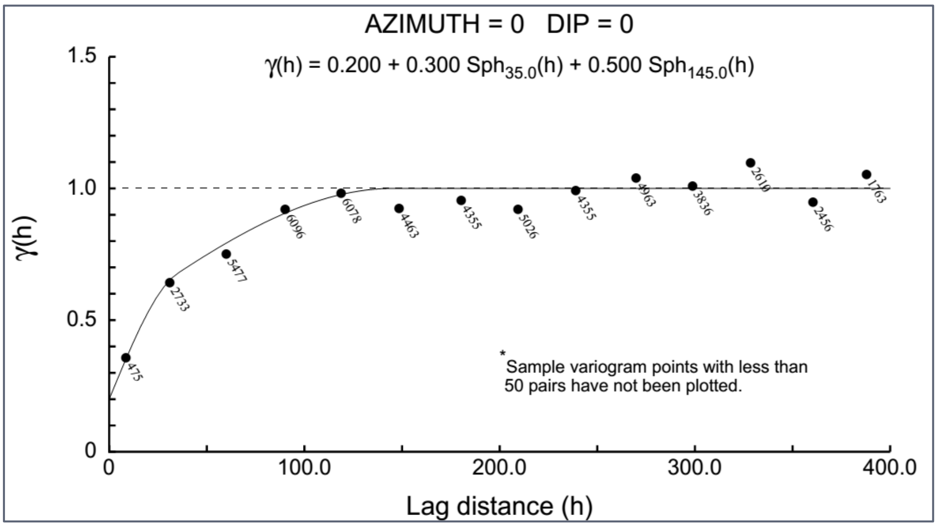


Figure 14-13: Lead correlogram – Este, Mineral Group 1

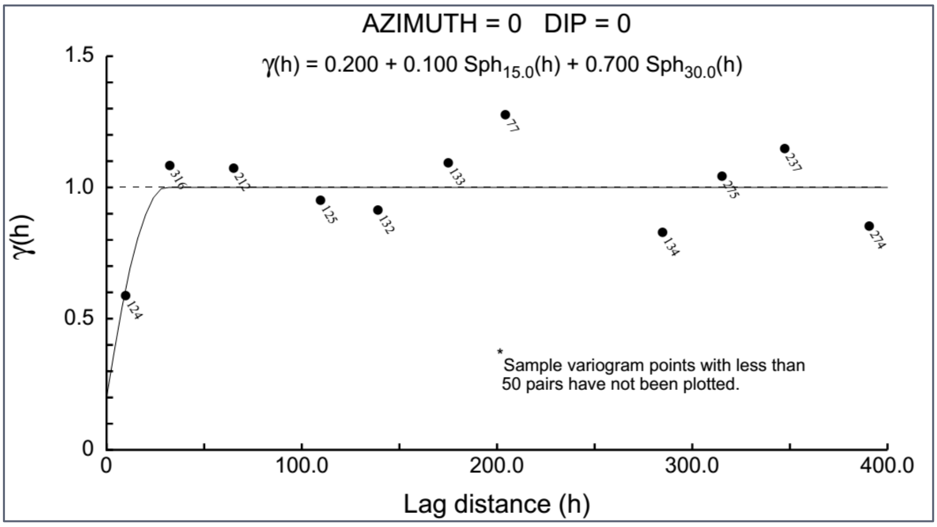


Figure 14-14: Lead correlogram – Este, Mineral Group 2

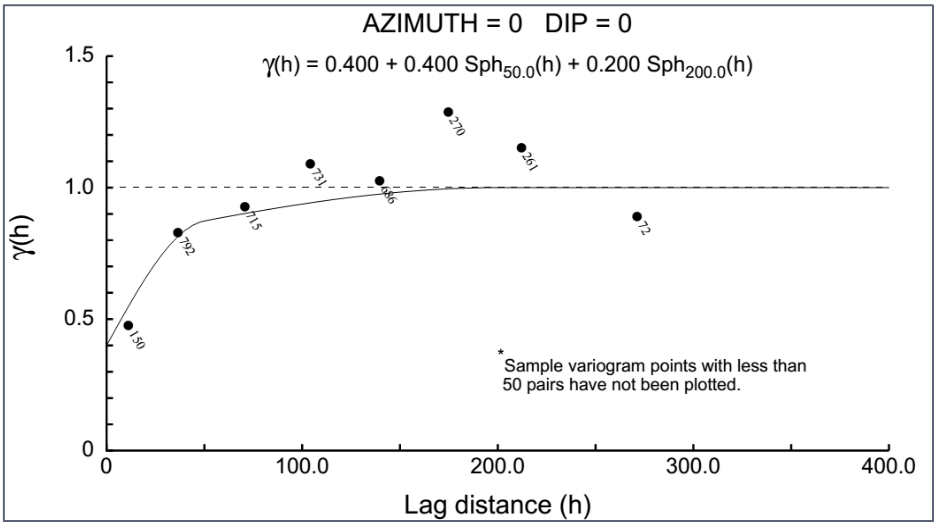


Figure 14-15: Lead correlogram – Este, Mineral Group 3

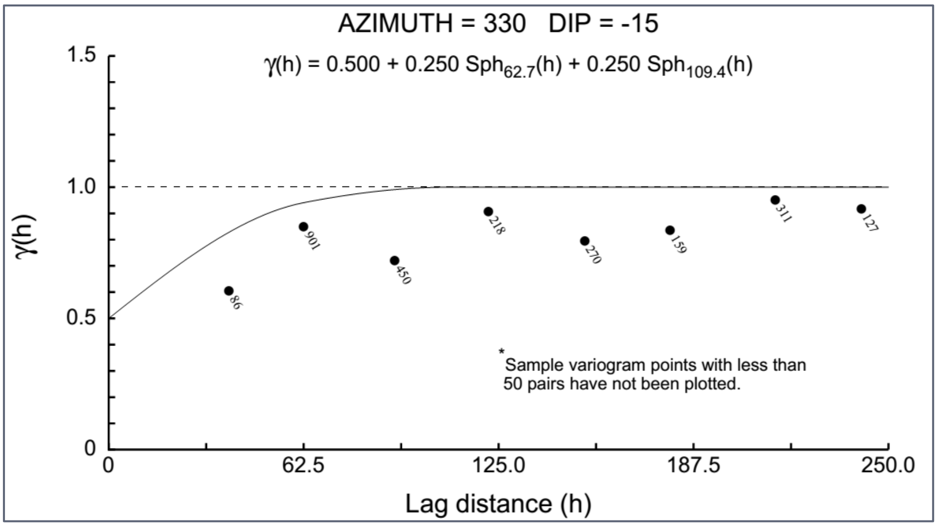


Figure 14-16: Zinc correlogram – Main, Mineral Group 1

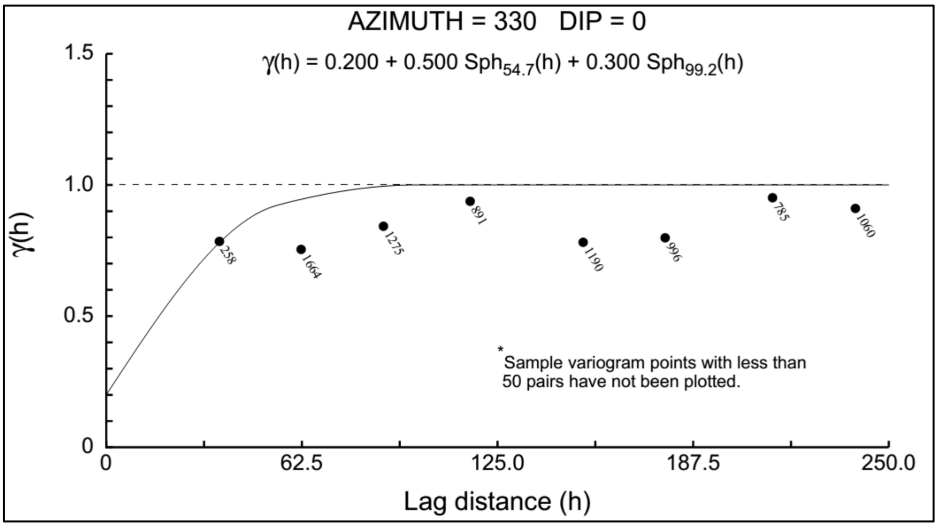


Figure 14-17: Zinc correlogram – Main, Mineral Group 2

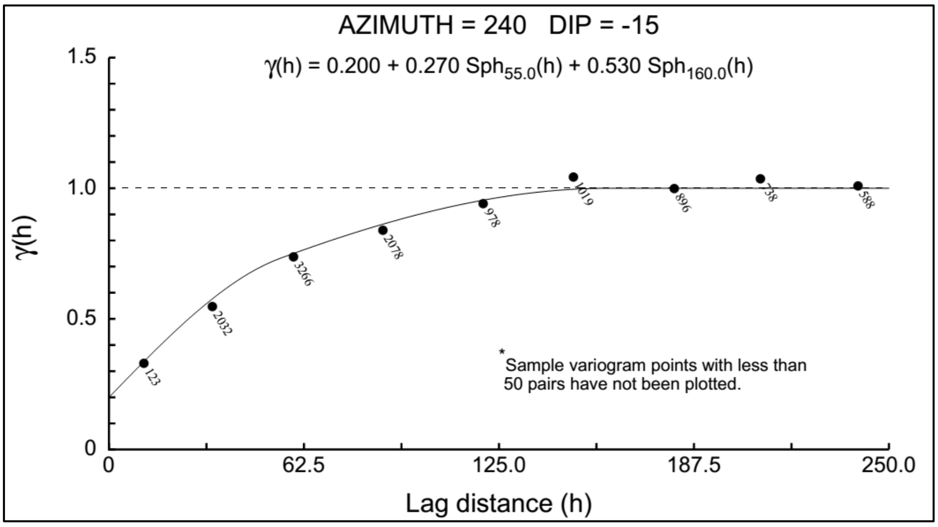


Figure 14-18: Zinc correlogram – Main, Mineral Group 3

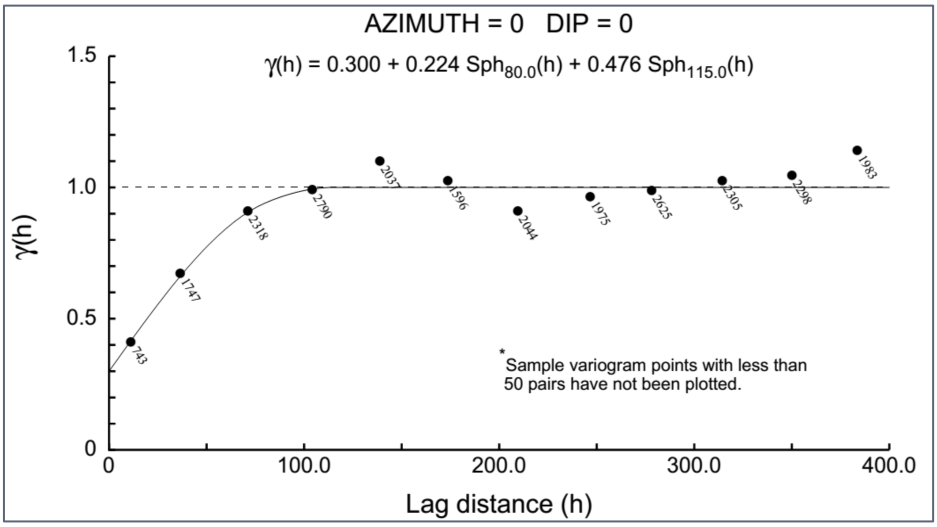


Figure 14-19: Zinc correlogram – Minas, Mineral Group 1

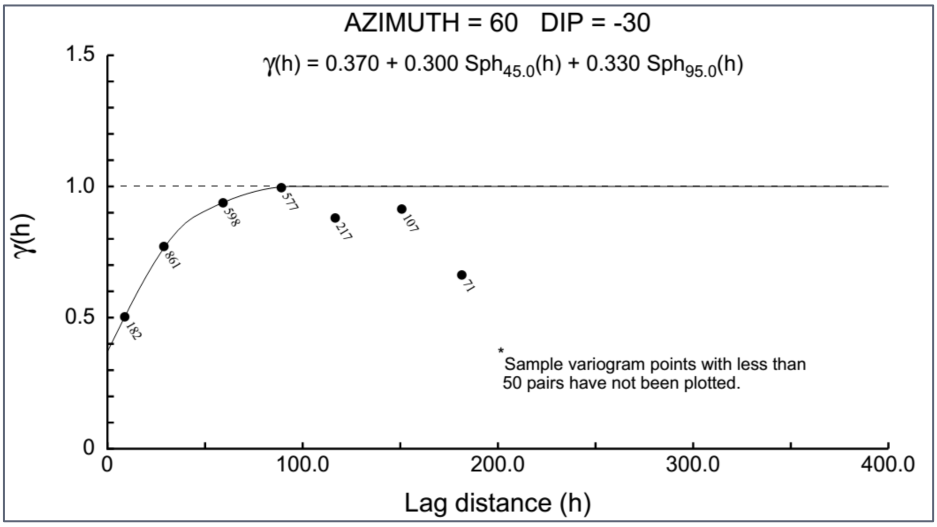


Figure 14-20: Zinc correlogram – Minas, Mineral Group 2

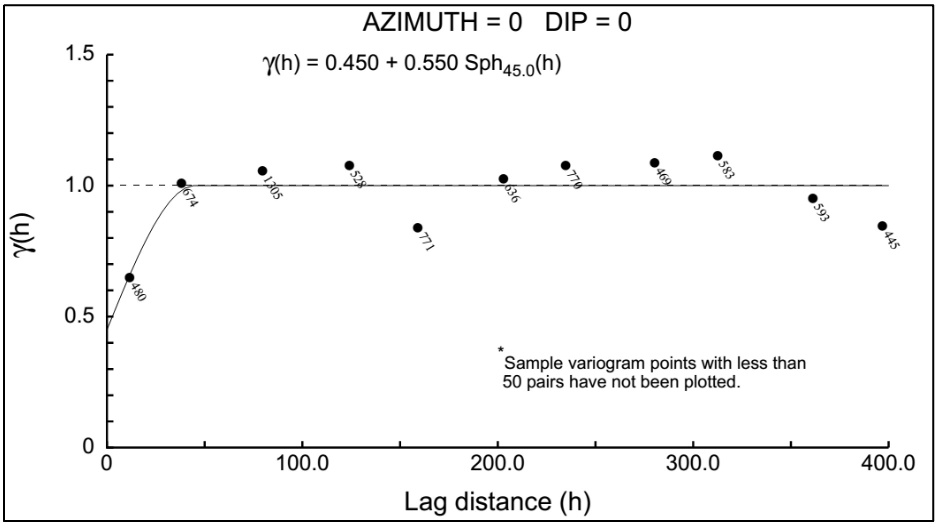


Figure 14-21: Zinc correlogram – Minas, Mineral Group 3

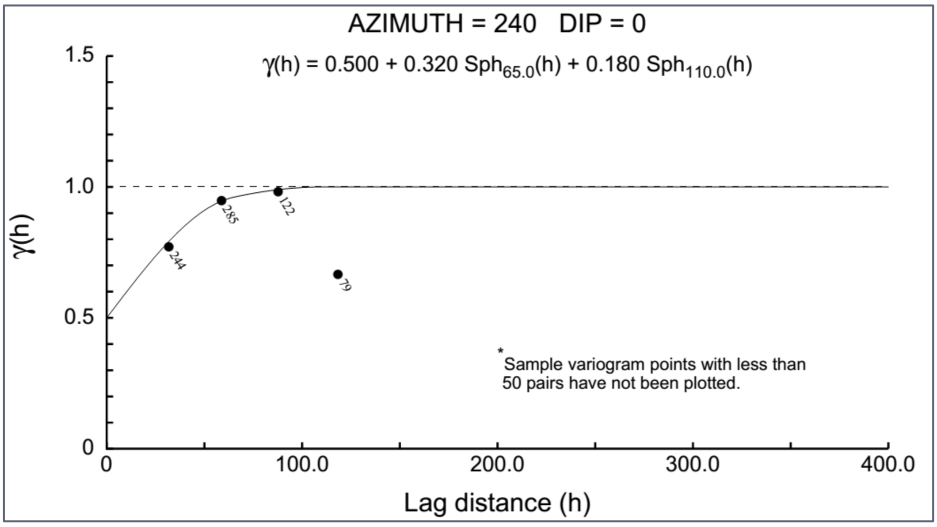


Figure 14-22: Zinc correlogram – Este, Mineral Group 1

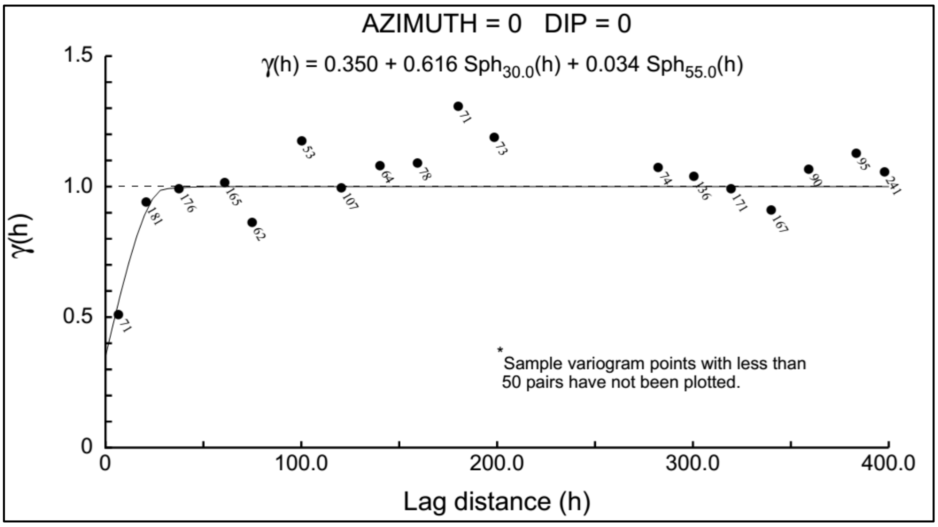


Figure 14-23: Zinc correlogram – Este, Mineral Group 2

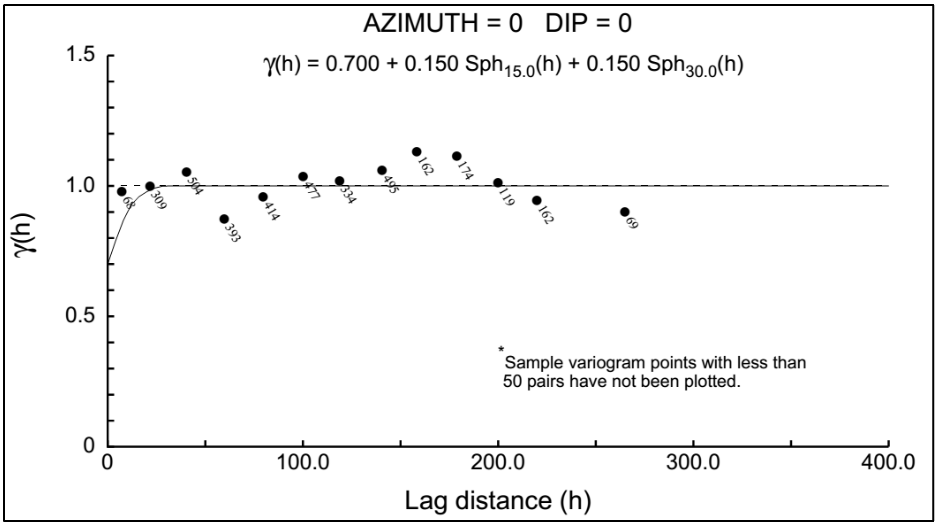


Figure 14-24: Zinc correlogram – Este, Mineral Group 3

Based on the above analysis, the parameters reported in Table 14-3 were used to model the three economic metals (silver, lead, and zinc) in the block model (labeled Ag, Pb, and Zn).

Table 14-3: Modeling parameters for silver, lead, and zinc

Grade	Zone	Mineral Group	C0	C1	C2	h1	h2	Azimuth	Dip	Tilt (LH)	Primary Axis	Secondary Axis	Tertiary Axis	Search Range
Silver	Main	ALL	0.1	0.35	0.55	100	170	330	-30	15	170	140	100	180
Silver	Minas	ALL	0.1	0.3	0.6	50	180	330	-30	15	180	150	90	180
Silver	Este	ALL	0.1	0.4	0.5	30	180	330	-30	15	180	150	95	180
Lead	Main	1 2 3 8 10	0.15	0.6	0.25	50	125	338	-14	30	125	120	55	180
Lead	Main	4 6 7	0.25	0.35	0.4	38	145	330	0	-60	145	140	100	180
Lead	Main	9	0.17	0.35	0.48	35	130	334	-14	15	130	125	70	180
Lead	Minas	1 2 3 8 10	0.35	0.08	0.57	80	250	0	0	0	250	250	250	180
Lead	Minas	4 6 7	0.35	0.48	0.17	40	200	0	0	0	200	200	200	180

Grade	Zone	Mineral Group	C0	C1	C2	h1	h2	Azimuth	Dip	Tilt (LH)	Primary Axis	Secondary Axis	Tertiary Axis	Search Range
Lead	Minas	9	0.25	0.35	0.4	40	175	0	0	0	175	175	175	180
Lead	Este	1 2 3 8 10	0.2	0.3	0.5	35	145	0	0	0	145	145	145	180
Lead	Este	4 6 7	0.2	0.1	0.7	15	30	0	0	0	30	30	30	180
Lead	Este	9	0.4	0.4	0.2	50	200	0	0	0	200	200	200	180
Zinc	Main	1 2 3 8 10	0.5	0.25	0.25	50	140	338	-14	30	140	135	100	180
Zinc	Main	4 6 7	0.2	0.5	0.3	45	125	330	0	-60	125	120	90	180
Zinc	Main	9	0.2	0.27	0.53	55	165	334	-14	15	165	160	60	180
Zinc	Minas	1 2 3 8 10	0.3	0.22	0.48	80	115	0	0	0	115	115	115	180
Zinc	Minas	4 6 7	0.37	0.3	0.33	45	95	60	-30	0	95	80	70	180
Zinc	Minas	9	0.45	0.55	0	45	45	0	0	0	45	45	45	180
Zinc	Este	1 2 3 8 10	0.5	0.32	0.18	65	110	240	0	0	110	60	60	180
Zinc	Este	4 6 7	0.35	0.62	0.03	30	55	0	0	0	55	55	55	180
Zinc	Este	9	0.7	0.15	0.15	15	30	0	0	0	30	30	30	180

Table 14-4: Modeling parameters for copper, geothite, and MnOx

Grade	Zone	Mineral Group	C0	C1	C2	h1	h2	Azimuth	Dip	Tilt (LH)	Primary Axis	Secondary Axis	Tertiary Axis	Search Range
Copper	Main	1 2 3 8 10	0.6	0.05	0.35	100	180	330	-15	15	180	150	45	180
Copper	Main	4 6 7	0.4	0.45	0.15	75	150	330	-30	30	150	125	75	180
Copper	Main	9	0.5	0.1	0.4	75	220	330	-15	15	220	215	60	180
Copper	Minas	1 2 3 8 10	0.4	0.32	0.28	30	275	0	0	0	275	275	275	180
Copper	Minas	4 6 7	0.4	0.1	0.5	75	200	330	-15	15	200	90	50	180
Copper	Minas	9	0.3	0.24	0.46	40	250	0	0	0	250	250	250	180
Copper	Este	1 2 3 8 10	0.5	0.2	0.3	70	220	330	-15	30	220	125	40	180
Copper	Este	4 6 7	0.3	0.42	0.28	30	140	0	0	0	140	140	140	180
Copper	Este	9	0.3	0.38	0.32	35	135	0	0	0	135	135	135	180
Goethite	Main	1 2 3 8 10	0.45	0.25	0.3	55	120	0	0	0	120	120	120	180
Goethite	Main	4 6 7	0.35	0.55	0.1	50	125	240	-45	-85	125	120	80	180
Goethite	Main	9	0.45	0.25	0.3	50	110	270	-30	-10	110	100	60	180
Goethite	Minas	1 2 3 8 10	0.25	0.6	0.15	40	100	270	-60	30	100	50	90	180
Goethite	Minas	4 6 7	0.1	0.6	0.3	60	130	240	-30	0	130	55	120	180
Goethite	Minas	9	0.65	0.16	0.19	80	160	0	0	0	160	160	160	180
Goethite	Este	1 2 3 8 10	0.25	0.37	0.38	30	125	90	-15	0	125	90	75	180
Goethite	Este	4 6 7	0.4	0.37	0.23	10	65	0	0	0	65	65	65	180
Goethite	Este	9	0.15	0.85	0	40	40	0	0	0	40	40	40	180
MnOx	Main	1 2 3 8 10	0.03	0.17	0.8	40	250	30	-60	-70	250	165	120	180
MnOx	Main	4 6 7	0.2	0.7	0.1	40	125	60	0	0	125	85	110	180
MnOx	Main	9	0.2	0.3	0.5	110	210	240	-15	0	210	60	200	180
MnOx	Minas	1 2 3 8 10	0.1	0.55	0.35	60	175	0	0	0	175	175	175	180
MnOx	Minas	4 6 7	0.1	0.8	0.1	40	125	240	-15	0	125	100	100	180
MnOx	Minas	9	0.35	0.05	0.6	20	225	0	0	0	225	225	225	180
MnOx	Este	1 2 3 8 10	0.1	0.2	0.7	75	135	300	-60	30	135	75	125	180
MnOx	Este	4 6 7	0.35	0.65	0	66	66	0	0	0	66	66	66	180
MnOx	Este	9	0.6	0.4	0	50	50	0	0	0	50	50	50	180

Table 14-5: Modeling parameters for pyrite and galena

Grade	Zone	Mineral Group	C0	C1	C2	h1	h2	Azimuth	Dip	Tilt (LH)	Primary Axis	Secondary Axis	Tertiary Axis	Search Range
Pyrite	Main	1 2 3 8 10	0.3	0.15	0.55	30	150	240	-45	0	150	55	120	180
Pyrite	Main	4 6 7	0.3	0.3	0.4	20	95	240	-45	-25	95	95	75	180
Pyrite	Main	9	0.2	0.45	0.35	45	90	240	-45	-65	90	90	75	180
Pyrite	Minas	1 2 3 8 10	0.1	0.32	0.58	30	350	0	0	0	350	350	350	180
Pyrite	Minas	4 6 7	0.65	0.24	0.11	31	150	0	0	0	150	150	150	180
Pyrite	Minas	9	0.72	0.12	0.16	98	100	0	0	0	100	100	100	180
Pyrite	Este	1 2 3 8 10	0.15	0.64	0.21	42	125	0	0	0	125	125	125	180
Pyrite	Este	4 6 7	0.4	0.6	0	40	40	0	0	0	40	40	40	180
Pyrite	Este	9	0.05	0.75	0.2	35	40	0	0	0	40	40	40	180
Galena	Main	1 2 3 8 10	0.3	0.5	0.2	33	125	240	-30	-15	125	100	110	180
Galena	Main	4 6 7	0.3	0.42	0.28	45	170	60	-45	-60	170	75	50	180
Galena	Main	9	0.2	0.45	0.35	55	135	150	-15	-75	135	65	105	180
Galena	Minas	1 2 3 8 10	0.75	0.14	0.11	35	80	0	0	0	80	80	80	180
Galena	Minas	4 6 7	0.25	0.58	0.17	25	100	0	0	0	100	100	100	180
Galena	Minas	9	0.7	0.25	0.05	8	80	0	0	0	80	80	80	180
Galena	Este	1 2 3 8 10	0.4	0.6	0	60	60	0	0	0	60	60	60	180
Galena	Este	4 6 7	0.8	0.14	0.06	5	50	0	0	0	50	50	50	180
Galena	Este	9	0.2	0.25	0.55	20	50	0	0	0	50	50	50	180

14.1.6 Estimation

The model chosen for all grades was inverse distance power (IDP), specifically inverse distance cubed (ID3) for silver and inverse distance to the 2.5 power (IDP2.5) for all others, with the exception of the transition indicator (discussed below). This model was picked to model the high local variability of the drill hole data. By adjusting the power, a satisfactory level of smoothness was applied to the grade model. The IDP model was compared to a kriged model, and IDP was chosen due to the kriged model's failure to reflect the highest and lowest grades in the population.

Comparative statistics for the drill holes, composites, and block model are tabulated in Table 14-6 through

Table 14-12.

Table 14-6: Borehole statistics for silver, lead, and zinc

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Ag	1	1	4	70.0	187.0	125.5	2,140.9	46.3	0.4
Ag	1	2	376	0.3	1,410.0	86.9	23,167.0	152.2	1.8
Ag	1	3	11,791	0.2	2,580.0	42.1	6,599.1	81.2	1.9
Ag	1	8	11,452	0.2	5,840.0	13.8	4,276.0	65.4	4.7
Ag	1	10	5	80.0	364.0	149.4	13,122.0	114.6	0.8
Ag	1	ALL	23,628	0.2	5,840.0	29.1	5,991.7	77.4	2.7
Ag	2	4	3,341	0.3	1,240.0	36.3	4,945.9	70.3	1.9
Ag	2	6	814	0.5	243.0	44.5	1,777.2	42.2	0.9
Ag	2	7	1,154	1.0	420.0	40.4	1,286.8	35.9	0.9
Ag	2	ALL	5,309	0.3	1,240.0	38.5	3,680.2	60.7	1.6
Ag	3	9	2,019	0.5	1,750.0	63.2	5,810.8	76.2	1.2
Ag	0	ALL	34,649	0.2	5,840.0	33.2	5,660.6	75.2	2.3

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Pb	1	1	4	1.72	3.54	2.87	0.54	0.73	0.26
Pb	1	2	376	0.01	16.65	2.20	4.93	2.22	1.01
Pb	1	3	11,722	0.00	15.55	0.71	1.00	1.00	1.40
Pb	1	8	11,278	0.00	15.30	0.31	0.47	0.68	2.22
Pb	1	10	5	1.25	2.17	1.75	0.11	0.33	0.19
Pb	1	ALL	23,385	0.00	16.65	0.54	0.89	0.94	1.74
Pb	2	4	3,324	0.00	5.93	0.42	0.38	0.62	1.46
Pb	2	6	814	0.01	10.85	0.77	1.03	1.02	1.32
Pb	2	7	1,154	0.01	5.92	0.67	0.55	0.74	1.11
Pb	2	ALL	5,292	0.00	10.85	0.53	0.54	0.73	1.38
Pb	3	9	2,019	0.01	8.92	0.94	0.93	0.97	1.02
Pb	0	ALL	34,190	0.00	16.65	0.56	0.82	0.90	1.62
Zn	1	1	4	0.78	2.17	1.80	0.40	0.63	0.35
Zn	1	2	376	0.04	9.00	1.03	1.91	1.38	1.34
Zn	1	3	11,677	0.00	16.15	0.42	1.01	1.00	2.38
Zn	1	8	11,430	0.00	23.91	0.48	0.73	0.86	1.80
Zn	1	10	5	0.11	0.20	0.16	0.00	0.04	0.24
Zn	1	ALL	23,492	0.00	23.91	0.46	0.89	0.95	2.07
Zn	2	4	3,211	0.01	3.45	0.10	0.02	0.15	1.49
Zn	2	6	808	0.01	5.03	0.18	0.06	0.24	1.32
Zn	2	7	1,137	0.01	1.14	0.13	0.01	0.11	0.82
Zn	2	ALL	5,156	0.01	5.03	0.12	0.03	0.17	1.35
Zn	3	9	1,904	0.01	3.24	0.15	0.05	0.23	1.49
Zn	0	ALL	33,313	0.00	23.91	0.37	0.67	0.82	2.24

Table 14-7: Composite statistics for silver, lead, and zinc

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Ag	1	1	16	5.51	532	84.8	14,964	122.3	1.44
Ag	1	2	122	2.15	462	71.6	6,604	81.3	1.14
Ag	1	3	3,016	0.20	731	42.8	3,315	57.6	1.35
Ag	1	8	3,001	0.20	704	13.4	796	28.2	2.11
Ag	1	10	3	114	181	141	854	29.2	0.21
Ag	1	ALL	6,158	0.20	731	29.2	2,444	49.4	1.69
Ag	2	4	889	0.50	534	38.0	3,627	60.2	1.58
Ag	2	6	224	0.96	241	47.3	1,620	40.2	0.85
Ag	2	7	301	2.00	268	41.8	996	31.6	0.75
Ag	2	ALL	1,436	0.50	534	40.2	2,735	52.3	1.30
Ag	3	9	539	1.24	614	66.1	3,711	60.9	0.92
Ag	ALL	ALL	8,961	0.10	731	33.4	2,623	51.2	1.53
Pb	1	1	16	0.35	4.58	1.89	1.16	1.08	0.57
Pb	1	2	122	0.09	8.22	1.83	2.15	1.47	0.80
Pb	1	3	3,016	0.00	8.51	0.72	0.64	0.80	1.11
Pb	1	8	2,992	0.00	6.97	0.30	0.26	0.51	1.69
Pb	1	10	3	2.17	3.23	2.83	0.22	0.47	0.17

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Pb	1	ALL	6,149	0.00	8.51	0.54	0.57	0.76	1.39
Pb	2	4	889	0.00	3.44	0.41	0.26	0.51	1.23
Pb	2	6	224	0.01	5.75	0.81	0.73	0.86	1.06
Pb	2	7	301	0.01	4.26	0.69	0.41	0.64	0.93
Pb	2	ALL	1,436	0.00	5.75	0.54	0.39	0.63	1.16
Pb	3	9	539	0.02	7.05	0.94	0.59	0.77	0.82
Pb	ALL	ALL	8,926	0.00	8.51	0.56	0.54	0.73	1.31
Zn	1	1	16	0.09	2.41	0.91	0.44	0.67	0.73
Zn	1	2	122	0.06	3.62	0.97	0.78	0.89	0.91
Zn	1	3	2,975	0.00	7.89	0.42	0.66	0.81	1.94
Zn	1	8	2,997	0.01	9.75	0.47	0.43	0.65	1.39
Zn	1	10	3	0.15	3.91	2.35	2.56	1.60	0.68
Zn	1	ALL	6,113	0.00	9.75	0.46	0.56	0.75	1.63
Zn	2	4	836	0.00	1.48	0.10	0.01	0.11	1.08
Zn	2	6	223	0.02	3.20	0.23	0.13	0.36	1.57
Zn	2	7	296	0.01	0.80	0.13	0.01	0.09	0.68
Zn	2	ALL	1,377	0.00	3.20	0.13	0.03	0.18	1.40
Zn	3	9	500	0.01	1.82	0.15	0.03	0.17	1.12
Zn	ALL	ALL	8,596	0.00	9.75	0.37	0.43	0.66	1.80

Table 14-8: Composite statistics for copper, MnOx, and goethite

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Cu	1	1	16	0.01	0.20	0.043	0.002	0.042	0.98
Cu	1	2	121	0.00	0.27	0.051	0.002	0.049	0.96
Cu	1	3	2,723	0.00	1.05	0.029	0.002	0.046	1.56
Cu	1	8	1,790	0.00	0.54	0.018	0.002	0.040	2.24
Cu	1	10	3	0.06	0.09	0.079	0.000	0.014	0.17
Cu	1	ALL	4,653	0.00	1.05	0.024	0.002	0.044	1.80
Cu	2	4	709	0.00	0.59	0.015	0.001	0.028	1.80
Cu	2	6	223	0.00	0.31	0.032	0.001	0.032	1.01
Cu	2	7	293	0.00	0.08	0.020	0.000	0.014	0.72
Cu	2	ALL	1,247	0.00	0.59	0.019	0.001	0.027	1.41
Cu	3	9	494	0.00	0.22	0.045	0.001	0.031	0.70
Cu	ALL	ALL	6,834	0.00	1.05	0.024	0.002	0.040	1.69
MnOx	1	1	7	0.00	1.00	0.404	0.216	0.465	1.15
MnOx	1	2	31	0.00	2.00	0.190	0.202	0.449	2.36
MnOx	1	3	831	0.00	5.48	0.165	0.251	0.501	3.04
MnOx	1	8	390	0.00	6.00	0.073	0.107	0.327	4.48
MnOx	1	10	1	0.00	0.01	0.005	0.000	0.007	1.41
MnOx	1	ALL	1,260	0.00	6.00	0.123	0.123	0.123	1.00
MnOx	2	4	283	0.00	3.25	0.164	0.195	0.442	2.70
MnOx	2	6	124	0.00	5.25	0.466	0.704	0.839	1.80
MnOx	2	7	114	0.00	2.67	0.177	0.214	0.463	2.61
MnOx	2	ALL	531	0.00	5.25	0.213	0.213	0.213	1.00
MnOx	3	9	314	0.00	5.39	0.748	0.853	0.924	1.24

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
MnOx	ALL	ALL	2,237	0.00	6.00	0.159	0.241	0.491	3.09
Goeth	1	1	517	0.00	6.00	0.285	0.693	0.833	2.93
Goeth	1	2	6	0.00	1.00	0.313	0.179	0.423	1.35
Goeth	1	3	20	0.00	1.00	0.053	0.033	0.182	3.40
Goeth	1	8	591	0.00	3.00	0.057	0.055	0.234	4.11
Goeth	1	10	136	0.00	5.15	0.019	0.034	0.184	9.69
Goeth	1	ALL	753	0.00	5.15	0.040	0.040	0.040	1.00
Goeth	2	4	599	0.00	5.39	0.441	0.557	0.746	1.69
Goeth	2	6	183	0.00	4.37	0.969	0.823	0.907	0.94
Goeth	2	7	255	0.00	5.75	1.160	1.389	1.178	1.02
Goeth	2	ALL	1,040	0.00	5.75	0.653	0.848	0.921	1.41
Goeth	3	9	343	0.00	7.39	0.796	1.319	1.148	1.44
Goeth	ALL	ALL	2,653	0.00	7.39	0.200	0.378	0.615	3.08

Table 14-9: Composite statistics for pyrite and galena

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
FeS	1	1	14	0.00	5.22	2.75	3.40	1.84	0.67
FeS	1	2	104	0.00	8.29	1.36	2.11	1.45	1.07
FeS	1	3	2,805	0.00	10.00	0.93	1.31	1.14	1.23
FeS	1	8	2,906	0.00	12.68	1.16	1.17	1.08	0.94
FeS	1	10	3	2.15	4.43	3.62	1.09	1.04	0.29
FeS	1	ALL	5,832	0.00	12.68	1.05	1.29	1.13	1.08
FeS	2	4	431	0.00	10.21	0.20	0.30	0.54	2.66
FeS	2	6	55	0.00	5.12	0.29	0.69	0.83	2.82
FeS	2	7	73	0.00	3.26	0.12	0.17	0.42	3.43
FeS	2	ALL	579	0.00	10.21	0.22	0.36	0.60	2.74
FeS	3	9	200	0.00	17.27	0.29	1.00	1.00	3.49
FeS	ALL	ALL	7,154	0.00	17.27	0.78	0.78	0.78	1.00
PbS	1	1	16	0.35	4.58	1.89	1.16	1.08	0.57
PbS	1	2	122	0.09	8.22	1.83	2.15	1.47	0.80
PbS	1	3	3,016	0.00	8.51	0.72	0.64	0.80	1.11
PbS	1	8	2,992	0.00	6.97	0.30	0.26	0.51	1.69
PbS	1	10	3	2.17	3.23	2.83	0.22	0.47	0.17
PbS	1	ALL	3,758	0.00	12.27	0.54	0.57	0.76	1.39
PbS	2	4	889	0.00	3.44	0.41	0.26	0.51	1.23
PbS	2	6	224	0.01	5.75	0.81	0.73	0.86	1.06
PbS	2	7	301	0.01	4.26	0.69	0.41	0.64	0.93
PbS	2	ALL	235	0.00	7.97	0.04	0.12	0.35	8.14
PbS	3	9	539	0.02	7.05	0.94	0.59	0.77	0.82
PbS	ALL	ALL	4,392	0.00	12.27	0.11	0.19	0.44	4.15

Table 14-10: Block statistics for silver, lead, and zinc

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Ag	1	1	4	59.06	96	81.9	195.8	14.0	0.17
Ag	1	2	822	5.17	295	85.1	3,320.3	57.6	0.68
Ag	1	3	53,290	0.59	516	29.1	981.3	31.3	1.08
Ag	1	8	102,884	0.50	793	11.1	302.3	17.4	1.56
Ag	1	10	13	78.95	165	111.2	596.6	24.4	0.22
Ag	1	ALL	157,013	0.50	793	17.6	17.6	17.6	1.00
Ag	2	4	21,550	0.57	308	21.1	627.8	25.1	1.19
Ag	2	6	3,517	0.50	147	35.3	590.5	24.3	0.69
Ag	2	7	6,861	0.50	147	32.4	424.7	20.6	0.64
Ag	2	ALL	31,928	0.50	308	25.1	25.1	25.1	1.00
Ag	3	9	5,920	2.42	348	49.9	1,479.6	38.5	0.77
Ag	ALL	ALL	194,861	0.50	793	19.8	701.5	26.5	1.34
Pb	1	1	4	0.25	0.38	0.30	0.00	0.05	0.15
Pb	1	2	822	0.03	6.41	1.05	1.20	1.10	1.04
Pb	1	3	47,778	0.00	8.86	0.13	0.09	0.30	2.36
Pb	1	8	89,444	0.00	7.70	0.10	0.08	0.28	2.89
Pb	1	10	13	0.13	0.48	0.29	0.01	0.11	0.39
Pb	1	ALL	138,061	0.00	8.86	0.35	0.14	0.38	1.07
Pb	2	4	11,878	0.00	0.79	0.01	0.00	0.03	3.60
Pb	2	6	1,860	0.00	0.33	0.01	0.00	0.02	3.23
Pb	2	7	3,031	0.00	0.58	0.00	0.00	0.02	4.40
Pb	2	ALL	32,023	0.01	2.62	0.38	0.12	0.34	0.91
Pb	3	9	4,816	0.00	2.22	0.05	0.01	0.10	1.87
Pb	ALL	ALL	194,770	0.00	4.58	0.37	0.15	0.38	1.03
Zn	1	1	4	0.55	1.09	0.90	0.04	0.21	0.23
Zn	1	2	822	0.14	4.02	0.90	0.27	0.52	0.58
Zn	1	3	53,306	0.01	4.57	0.32	0.18	0.42	1.30
Zn	1	8	102,846	0.01	4.28	0.34	0.09	0.30	0.90
Zn	1	10	13	0.15	2.76	1.46	0.93	0.97	0.66
Zn	1	ALL	156,991	0.01	4.57	0.34	0.12	0.35	1.04
Zn	2	4	21,551	0.01	0.86	0.11	0.00	0.07	0.63
Zn	2	6	3,512	0.01	2.12	0.15	0.01	0.10	0.67
Zn	2	7	6,796	0.03	0.44	0.12	0.00	0.05	0.39
Zn	2	ALL	31,859	0.01	2.12	0.12	0.01	0.07	0.61
Zn	3	9	5,920	0.02	0.98	0.15	0.01	0.09	0.57

Table 14-11: Block statistics for copper, goethite, and MnOx

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Cu	1	1	4	0.02	0.05	0.03	0.00	0.01	0.27
Cu	1	2	822	0.01	0.17	0.04	0.00	0.03	0.61
Cu	1	3	52,169	0.00	0.84	0.03	0.00	0.03	1.15
Cu	1	8	89,217	0.00	0.33	0.01	0.00	0.02	1.53

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Cu	1	10	13	0.02	0.07	0.04	0.00	0.01	0.28
Cu	1	ALL	142,225	0.00	0.84	0.02	0.00	0.03	1.38
Cu	2	4	20,584	0.00	0.32	0.02	0.00	0.02	1.34
Cu	2	6	3,445	0.00	0.12	0.02	0.00	0.02	0.73
Cu	2	7	6,472	0.00	0.09	0.02	0.00	0.01	0.70
Cu	2	ALL	30,501	0.00	0.32	0.02	0.00	0.02	1.15
Cu	3	9	5,919	0.00	0.16	0.05	0.00	0.02	0.50
Cu	ALL	ALL	297,455	0.00	0.84	0.02	0.02	0.02	1.00
Goethite	1	1	3	0.00	0.03	0.02	0.00	0.01	0.69
Goethite	1	2	575	0.00	0.83	0.08	0.02	0.13	1.67
Goethite	1	3	29,887	0.00	2.93	0.06	0.03	0.16	2.71
Goethite	1	8	24,381	0.00	3.74	0.02	0.01	0.09	5.70
Goethite	1	10	6	0.00	0.01	0.00	0.00	0.00	1.68
Goethite	1	ALL	54,852	0.00	3.74	0.03	0.01	0.12	3.91
Goethite	2	4	21,034	0.00	4.36	0.47	0.19	0.44	0.94
Goethite	2	6	3,422	0.00	4.06	0.87	0.38	0.61	0.71
Goethite	2	7	6,617	0.00	4.73	1.01	0.51	0.71	0.70
Goethite	2	ALL	31,073	0.00	4.73	0.63	0.34	0.58	0.92
Goethite	3	9	5,502	0.00	4.63	0.67	0.44	0.66	0.99
Goethite	ALL	ALL	166,422	0.00	4.81	0.13	0.11	0.32	2.41
MnOx	1	1	2	0.00	0.31	0.09	0.02	0.13	1.38
MnOx	1	2	760	0.00	2.54	0.32	0.16	0.40	1.26
MnOx	1	3	31,350	0.00	4.89	0.18	0.16	0.41	2.30
MnOx	1	8	36,218	0.00	2.77	0.07	0.06	0.24	3.50
MnOx	1	10	9	0.00	0.51	0.19	0.03	0.17	0.93
MnOx	1	ALL	68,339	0.00	4.89	0.11	0.10	0.31	2.93
MnOx	2	4	15,164	0.00	3.53	0.14	0.09	0.30	2.05
MnOx	2	6	2,845	0.00	3.61	0.34	0.23	0.48	1.40
MnOx	2	7	5,200	0.00	2.47	0.15	0.08	0.27	1.84
MnOx	2	ALL	23,209	0.00	3.61	0.17	0.10	0.32	1.93
MnOx	3	9	5,575	0.00	3.56	0.74	0.52	0.72	0.97
MnOx	ALL	ALL	150,703	0.00	4.89	0.12	0.11	0.33	2.65

Table 14-12: Block statistics for pyrite and galena

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Pyrite	1	1	4	0.95	1.09	1.00	0.00	0.06	0.06
Pyrite	1	2	822	0.15	4.84	1.25	0.56	0.75	0.60
Pyrite	1	3	52,187	0.00	9.00	0.97	0.66	0.81	0.83
Pyrite	1	8	101,544	0.00	10.75	1.08	0.56	0.75	0.69
Pyrite	1	10	13	1.29	3.90	2.79	0.60	0.78	0.28
Pyrite	1	ALL	154,570	0.00	10.75	1.05	0.60	0.77	0.74

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Pyrite	2	4	17,604	0.00	5.13	0.19	0.10	0.32	1.68
Pyrite	2	6	2,612	0.00	2.21	0.15	0.09	0.29	2.02
Pyrite	2	7	3,994	0.00	6.49	0.09	0.06	0.23	2.76
Pyrite	2	ALL	24,210	0.00	6.49	0.16	0.09	0.31	1.86
Pyrite	3	9	5,113	0.00	10.03	0.23	0.23	0.48	2.12
Pyrite	ALL	ALL	295,690	0.00	10.75	0.74	0.53	0.72	0.98
Galena	1	1	4	0.25	0.38	0.30	0.00	0.05	0.15
Galena	1	2	822	0.03	6.41	1.05	1.20	1.10	1.04
Galena	1	3	47,778	0.00	8.86	0.13	0.09	0.30	2.36
Galena	1	8	89,444	0.00	7.70	0.10	0.08	0.28	2.89
Galena	1	10	13	0.13	0.48	0.29	0.01	0.11	0.39
Galena	1	ALL	138,061	0.00	8.86	0.11	0.09	0.30	2.71
Galena	2	4	11,878	0.00	0.79	0.01	0.00	0.03	3.60
Galena	2	6	1,860	0.00	0.33	0.01	0.00	0.02	3.23
Galena	2	7	3,031	0.00	0.58	0.00	0.00	0.02	4.40
Galena	2	ALL	16,769	0.00	0.79	0.01	0.00	0.03	3.73
Galena	3	9	4,816	0.00	2.22	0.05	0.01	0.10	1.87
Galena	ALL	ALL	269,954	0.00	8.86	0.07	0.05	0.22	3.19

14.1.7 Classification

Blocks were coded as measured, indicated, or inferred based on the silver grade estimate. The number of samples and the distance to the nearest sample were the discriminating factors. Silver grades were estimated using a minimum of one composite and maximum of ten composites, with a maximum of three composites from one drill hole. If the number of composite samples is at least six and the distance to the nearest composite is less than 12 meters, the block is classified as measured. If the number of composite samples is at least two, the distance to the nearest composite is less than 50 meters, and the block has not been classified as measured, the block is classified as indicated. All other blocks with an estimated silver grade are classified as inferred.

14.1.8 Transition material

An indicator variable, TransIND, was created to classify the material which has problematic flotation recovery for lead. The indicator was set by looking at composite intervals within Este pit zone based on the following formula. The transition zone indicator only applies to the lead and silver recoveries to the lead concentrate.

$$IF(to < 90 \text{ AND } silver > 15\ AND\ zinc < 0.3)\ Transition\ Indicator \\ = 1\ ELSE\ Transition\ Indicator = 0$$

14.1.9 Metallurgical recovery estimate

Following a detailed geometallurgical analysis, the recovery models were updated with the most recent metallurgical test results to use a statistical model with zinc grade, elevation, and geologic log data (i.e., pyrite %, galena %, MnO, etc.) to create a continuous recovery model/formula for each of the metals. Application of this method resulted in generation of the following formulas for the prediction of metals recoveries.

$$
\begin{aligned}
&\mathit{Lead\ Recovery\ to\ Final\ Lead/Silver\ Concentrate}\ (\boldsymbol{non-Transition}) \\
&\quad = 61.5 - 40.9 * max(0{,}0.57 - zinc) + 7.7 * max(0, galena - 0.38) + 45.4 \\
&\quad * max(0{,}0.37 - goethite) - 0.12 * max(0, elevation - 4891) + 32.9 \\
&\quad * max(0{,}0.27 - MnOxi) - 6.21 * max(0, pyrite - 1.07) - 16.4 \\
&\quad * max(0{,}1.07 - pyrite)
\end{aligned}
$$

$$
\begin{aligned}
&\mathit{Lead\ Recovery\ to\ Final\ Lead/Silver\ Concentrate}\ (\boldsymbol{Transition}) \\
&\quad = 35.6 - 40.9 * max(0{,}0.57 - zinc) + 7.7 * max(0, galena - 0.38) + 45.4 \\
&\quad * max(0{,}0.37 - goethite) - 0.12 * max(0, elevation - 4891) + 32.9 \\
&\quad * max(0{,}0.27 - MnOxi) - 6.21 * max(0, pyrite - 1.07) - 16.4 \\
&\quad * max(0{,}1.07 - pyrite)
\end{aligned}
$$

$$
\begin{aligned}
&\mathit{Zinc\ Recovery\ to\ Final\ Zinc\ Concentrate} \\
&\quad = 84.4 - 50.6 * max(0{,}1.02 - zinc) - 0.15 * max(0, elevation - 4901) - 5.4 \\
&\quad * max(0, pyrite - 1.9) - 11.2 * max(0{,}1.9 - pyrite) + 104.1 \\
&\quad * max(0, copper - 0.03) + 1620.2 * max(0{,}0.03 - copper)
\end{aligned}
$$

$$
\begin{aligned}
&\mathit{Silver\ Recovery\ to\ Final\ Lead/Silver\ Concentrate}\ (\boldsymbol{non-Transition}) \\
&\quad = 24.25 - 0.49 * \max(0{,}42.5 - silver) - 13.68 * \max(0, zinc - 1.52) + 17.06 \\
&\quad * \max(0, zinc - 3.16) + 13.67 * \max(0{,}2 - Rock) - 2.69 \\
&\quad * \max(0, \mathit{Predicted\ Lead\ Recovery} - 21.54) + 13.72 \\
&\quad * \max(0, \mathit{Predicted\ Lead\ Recovery} - 30.33) - 10.64 \\
&\quad * \max\ (0, \mathit{Predicted\ Lead\ Recovery} - 33.57)
\end{aligned}
$$

$$
\begin{aligned}
&\mathit{Silver\ Recovery\ to\ Final\ Lead/Silver\ Concentrate}\ (\boldsymbol{Transition}) \\
&\quad = 26.14 - 0.49 * \max(0{,}42.5 - silver) - 13.68 * \max(0, zinc - 1.52) + 17.06 \\
&\quad * \max(0, zinc - 3.16) + 13.67 * \max(0{,}2 - Rock) - 2.69 \\
&\quad * \max(0, \mathit{Predicted\ Lead\ Recovery} - 21.54) + 13.72 \\
&\quad * \max(0, \mathit{Predicted\ Lead\ Recovery} - 30.33) - 10.64 \\
&\quad * \max\ (0, \mathit{Predicted\ Lead\ Recovery} - 33.57)
\end{aligned}
$$

$$
\begin{aligned}
&\mathit{Silver\ Recovery\ in\ Zinc\ Concentrate} \\
&\quad = 48.156\ -\ 0.544\ \times (\mathit{Silver\ Recovery\ in\ Lead\ Concentrate})
\end{aligned}
$$

14.1.10 Capping and limitations on recoveries

Limits were placed on the recoveries calculated from the formulae for each metal, based on actual results seen in the locked cycle testing. A minimum and maximum recovery were used to ensure that the recoveries are within rational limits. Calculated recoveries less or greater than the limits were set to the limit. The limits were used when calculating metal recoveries stored in the block model. The limits are shown in Table 14-13.

Table 14-13 Minimum and maximum recovery limits

Metal	Minimum Recovery	Maximum Recovery
Zinc	0%	95%

Metal	Minimum Recovery	Maximum Recovery
Lead	2%	98%
Silver $_{total}$	0%	95%
Silver $_{Pb\ conc}$	0%	95%
Silver $_{Zn\ conc}$	0%	95%

Silver grades in the zinc concentrate were also capped in the production schedule to a maximum of 430 grams per dry metric tonne (g/dmt) based on the maximum expected silver grades seen in the zinc concentrates from the locked cycle testing. The remainder of the silver recovered was shown as reporting to the lead concentrate for the purposes of scheduling.

14.2 Acid Rock Drainage

No additional data was captured in the 2019 campaign, so the work in 2015 was not updated in this model. In 2015, several parameters related to the acid rock drainage characteristics of the waste rock were selected for inclusion in the block model. These parameters include Acid (generating) Potential (AP)/total sulphur, Neutralization Potential (NP), and metals of concern, including arsenic, cadmium, mercury, and nickel. The following is verbatim from the 2015 report, for completeness.

To use waste rock for construction of the dump, backfill cap, embankment dams, etc., there needed to be a way to distinguish non-acid-generating rock (NAG) from potentially acid-generating rock (PAG). For this purpose, GRE developed an acid rock drainage (ARD) block model using available geochemical sample data collected by Vector in 2009 (Vector Perú S.A.C., 2009), GRE in 2011, and AMEC in 2012 (AMEC, 2012).

Various analyses were performed on each sample group, including full Acid-Base Accounting (ABA) analysis, whole rock analysis, metals by aqua regia digestion, and total carbon/total sulphur assay by LECO furnace. A summary of each sample group and analysis is given in Table 14-14.

Table 14-14: Geochemical sample summary

Collected By	Date	No. of Samples	Analyses			
			ABA	LECO	WRA	Metals
Vector	2009	23	23	23	7	7
GRE	2011	224	23**	201	23**	23**
AMEC	2012	397*	397	397	397	397
Total		**644**	**443**	**621**	**427**	**427**

*2 samples are reported as the same location and interval. The results for these samples were averaged

**23 Samples from onsite kinetic cell program

The parameters selected for inclusion in the model included AP/total sulphur, NP, pH, and metals of concern, which are described in the Geochemical Characterization Report (GRE, 2012) and include arsenic, cadmium, copper, lead, mercury, nickel, and zinc.

For the parameters considered, comparable analyses from different labs (i.e., total sulphur/total sulphur/ S-%) were matched, and results were converted to consistent units. GRE developed linear models relating to ABA, AP, and total sulphur by LECO furnace by rock type for samples collected by Vector in 2009 and GRE in 2011. These relationships were used to predict AP for samples where only LECO furnace total sulphur/carbon was analysed. AP was predicted only for samples from rock types PM, FeO, and FBS, for which the relationships between total sulphur and AP had an R^2 greater than 0.97. If the model predicted an AP less than 0, an AP of 0 was assumed. This analysis is described in the Geochemical Characterization Report (GRE, 2012).

Table 14-15 and Table 14-16 contain summary statistics for the raw data.

Table 14-15: Raw data summary statistics

Parameter	AP	NP	As	Cd	Cu
Number	610	442	426	426	426
Mean	22.4	3.9	248.6	13.0	94.5
Std Dev	32.7	7.0	518.2	44.1	149.2
Variance	1066.4	48.5	268529.5	1942.5	22265.4
Maximum	206.0	97.0	5560.0	569.0	1200.0
Minimum	0.0	0.00	10.1	0.02	1.5
Range	206.0	97.0	5549.9	569.0	1198.5
Coef Var	145.7	178.0	208.4	339.8	157.9
Std Err	1.3	0.3	25.1	2.1	7.2

Table 14-16: Raw data summary statistics

Parameter	Pb	Ni	Hg	Zn	pH
Number	426	426	426	426	442
Mean	2231.5	8.7	0.9	1518.3	6.2
Std Dev	4412.9	11	1.7	2596.2	1.5
Variance	19473957.0	120.7	3	6740476.8	2.3
Maximum	61900.0	74.8	18.8	32300.0	9.5
Minimum	10.0	0.6	0.01	59.0	2.3
Range	61890.0	74.2	18.8	32241.0	7.2
Coef Var	197.8	126.1	189.7	171.0	24.3
Std Err	213.8	0.5	0.08	125.8	0.07

The geochemical samples were loaded into TECHBASE software and composited to eight-meter intervals along lithology boundaries. Inverse distance cubed weighting was used to estimate constituent concentrations in all model blocks contained within a wireframe boundary based on geologic boundaries of the Project mineralized bodies. For each block in the block model, two to eight samples were considered within a search ellipsoid, and a weighted average of the parameter of concern was calculated.

Using this model, the spatial variation of acid-generating waste rock within the pit and in the pit walls was determined. Material with an estimated net neutralizing potential less than -20 tonne/kilotonne was designated as potentially acid generating; material with an estimated net neutralizing potential greater than -20 tonne/kilotonne was designated as non-acid generating.

14.3 Mineral Resources

Lerchs-Grossman (LG) analyses were compared to the prior Whittle™ shells and to the BCM designed pits to validate those designs used in the 2019 study. This analysis concluded that the BCM-designed pits and the ultimate pit design is valid and can serve as a basis for 2019 FS. GRE recommends additional phase design and scheduling work to optimize cash flow and minimize stockpiling and blending requirements.

Input assumptions for the LG economic pit shells were based on updated costs, including contract mining estimates. GRE tested the sensitivity of model inputs and found the design to be insensitive to the minor level of variation between the 2015 Whittle assumptions and the final economic model results.

The economic input for mineral resource determination was identical to that applied to the mineral reserve, with the following exceptions:

- The resource Whittle pit shell did not receive economic credit for inferred-class material. Inferred was treated as waste for the mineral reserve.
- The Mineral Resources were generated within the $30.00 silver, $1.425 lead, and $1.50 zinc price pit shell and the calculated $10.79/t Net Smelter Return (NSR) cutoff (see Table 14-17).
- The Mineral Resource contains potentially leachable material processed at $7.23/tonne and above a 15 g/t silver cutoff (Table 14-18). This Resource is contained within the economic pit shell but is not included in Table 14-17. The Mineral Reserve does not include any potentially leachable material.

The resource statement on Table 14-17 includes the mineral reserves that are presented in Chapter 15.

The qualified person responsible for the estimation of the Mineral Resource was Terre Lane. Metal price changes could materially change the estimated mineral resources in either a positive or negative way.

At this time, there are no unique situations relative to environmental, permitting, legal, title, taxation, political, or socioeconomic conditions that would put the Corani mineral resource at a higher level of risk than any other Peruvian developing resource.

Table 14-17: Total mineral resources (includes both resources and reserves)

Category	Tonnes (000)	Silver g/t	Lead %	Zinc %	Silver Moz	Lead Mlb	Zinc Mlb
Measured	30,585	50.0	0.79	0.49	49.1	534	329
Indicated	208,050	40.9	0.64	0.43	273.5	2,933	1,985
Measured + Indicated	**238,635**	**42.1**	**0.66**	**0.44**	**322.7**	**3,466**	**2,313**
Inferred	73,185	35.5	0.40	0.30	83.5	641	484

Note: Cutoff Value: $10.79/tonne covers process and general and administrative costs.

Table 14-18: Total mineral resource of potentially leachable material (includes the mineral reserve)

Category	Tonnes (000)	Silver g/t	Silver Moz
Measured	4,302	28.9	4.0
Indicated	36,104	30.1	35.0
Measured + Indicated	40,406	30.0	39.0
Inferred	24,311	38.2	29.9

Figure 14-25, Figure 14-26, and Figure 14-27 compare cumulative frequencies of silver, lead, and zinc. Each plot shows the cumulative frequency of composite grades, nearest-neighbour block grades, inverse distance block grades, and kriged block grades. The plots show how a nearest-neighbour model estimate follows the trends of the composite assay data very closely in all the metal models. The inverse distance and kriged models show how, once spatial variability is applied, the frequency plot of the population takes on the smoothed distribution that is expected of a grade model. Figure 14-28 and Figure 14-29 are examples of a plan and cross-section of the block model with drill hole intercepts plotted with composite assay data and the block grade model for silver. The visual inspection shows how the model agrees with the assay data and smooths the grade among input data points.

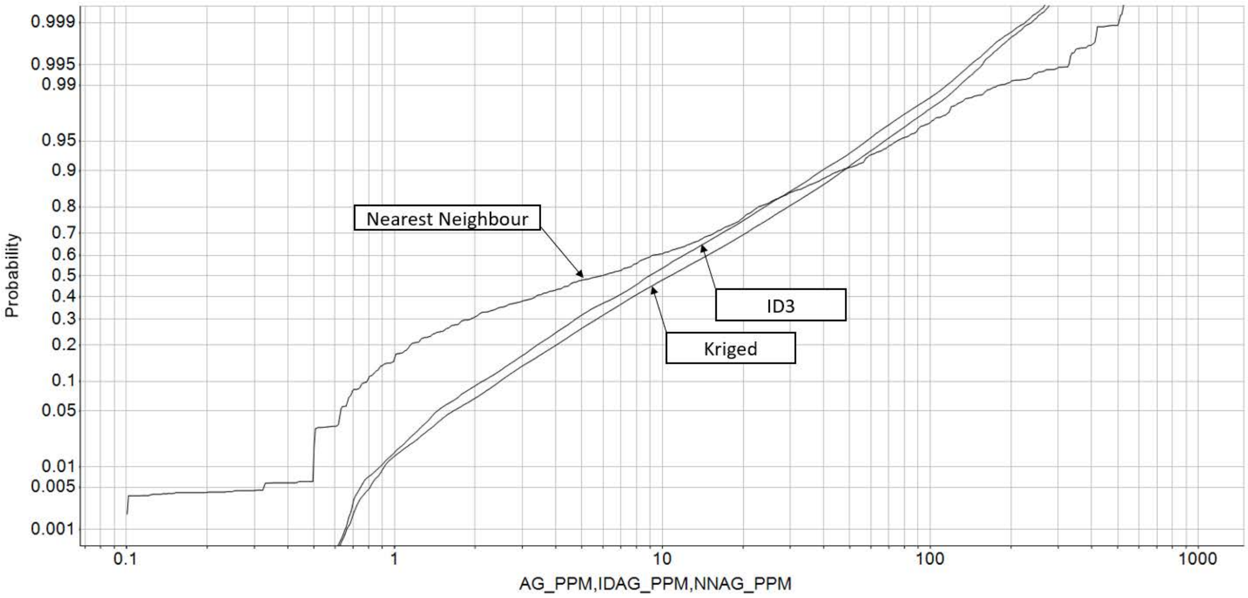


Figure 14-25: Cumulative frequency of silver grades – kriged, ID3, and nearest neighbour

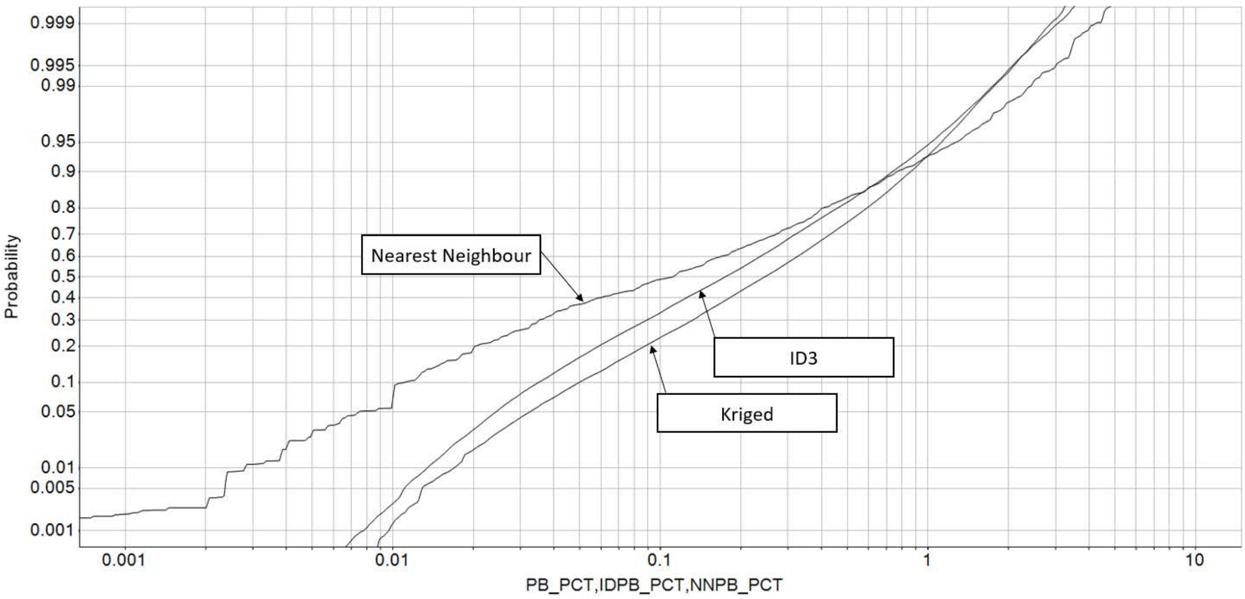


Figure 14-26: Cumulative frequency of lead grades –kriged, id2.5, and nearest neighbour

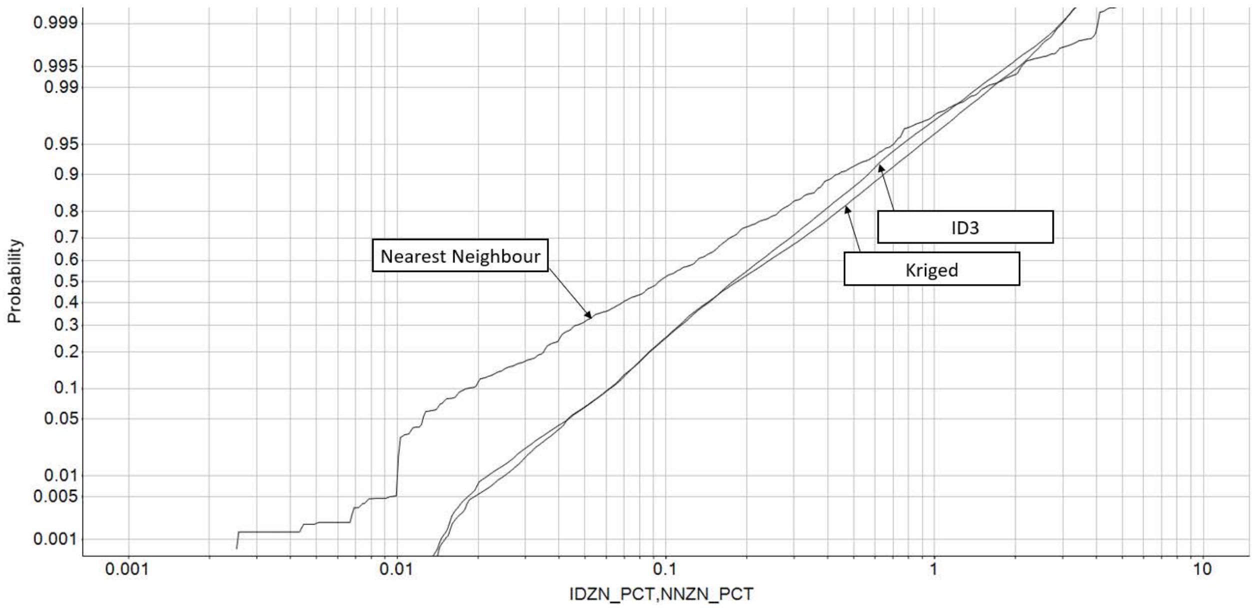


Figure 14-27: Cumulative frequency of zinc grades –kriged, ID2.5, and nearest neighbour

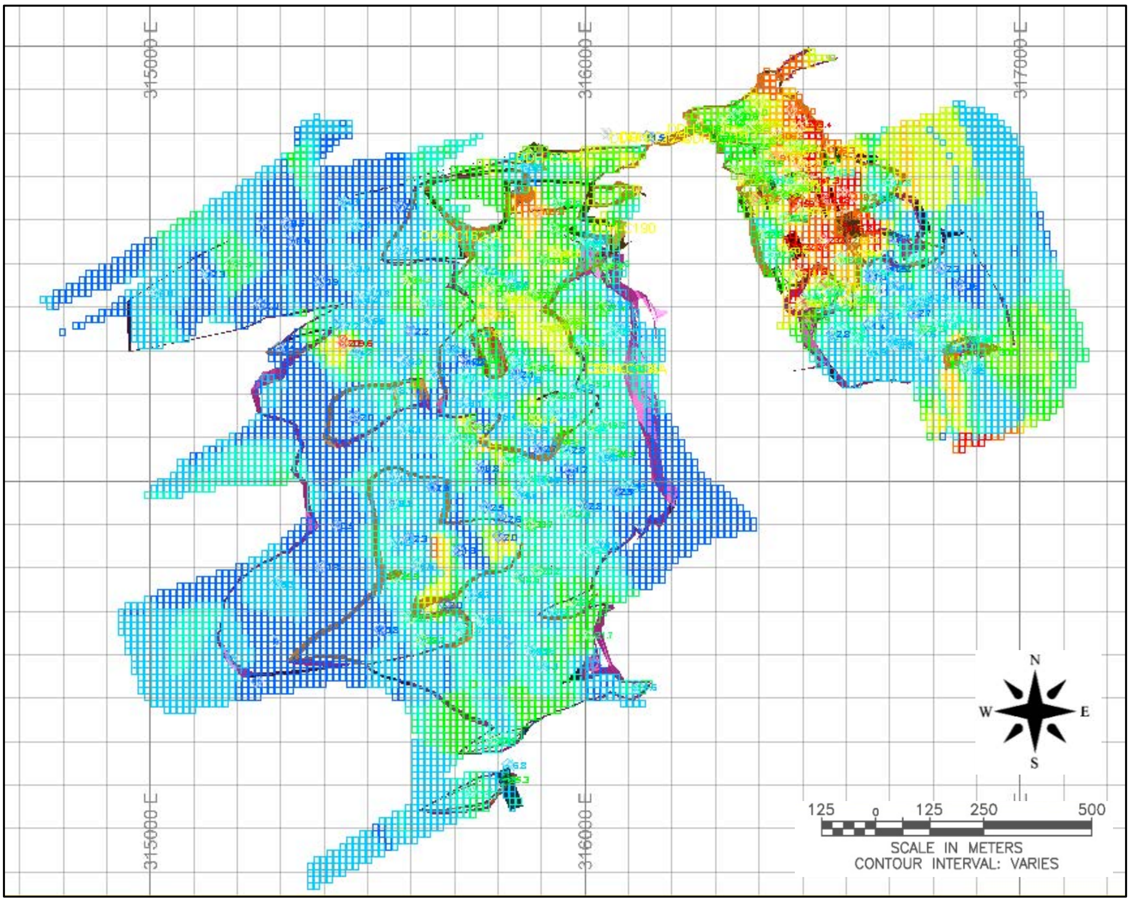


Figure 14-28: Silver composite assays and ID3 block grades on bench 4854

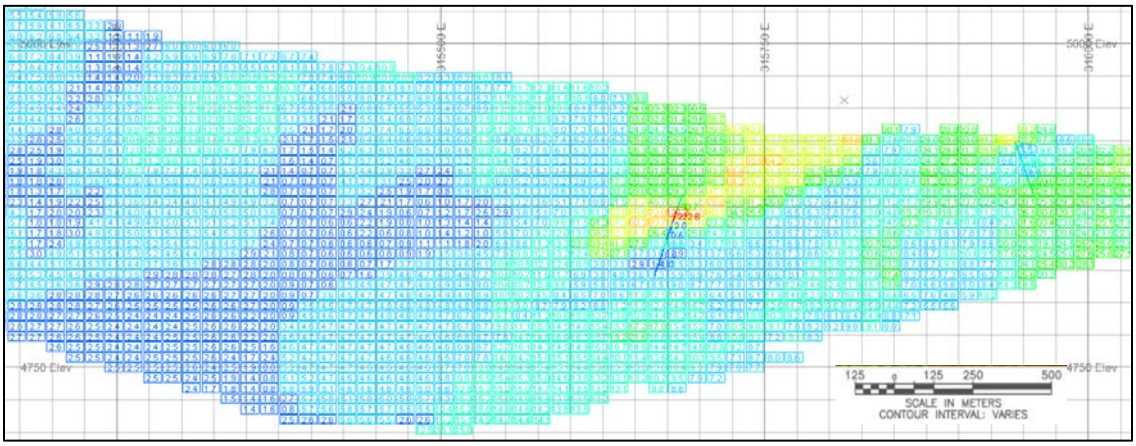


Figure 14-29: Silver composite assays and ID3 block grades Section 8,447,712.5 N, looking north

15 Mineral Reserve Estimates

GRE reviewed and verified that the phased mine design generated by BCM has been prepared with sound engineering principles and is correct. The mine design was compared to LG pits estimated using the current GRE Mineral Resource block model. GRE has found the work performed by BCM to reasonably conform to those current economic pits estimated. The LG estimation used \$20.00/oz silver, \$1.00/lb zinc, and \$0.95/lb lead for the mine design. The latest BCM phase designs are designed to mine outside of the Bofedal units due to geotechnical risks, mining costs, and operational challenges.

The mine plan also uses a variable cutoff value higher than the economic cutoff value (these cutoffs would be equivalent to the economic cutoff calculated at significantly higher metal prices). This combined approach using higher metal prices for the pit design and lower metal prices in the economic model is excellent for projects nearing production in a potentially rising metal price environment, as it is can be difficult or impossible to make small changes to the pit design and size after mining begins because of bench width and equipment maneuvering requirements. The elevated cutoff increases cash flow, reduces risk, and improves the economic performance of the Project.

The mineral reserve is the total of all proven- and probable-category ore that is planned for production. The mine design and mining schedule presented in Chapter 16 details the production of that reserve. The mineral reserve was established by tabulating the contained tonnes of measured and indicated material (proven and probable) within the designed final pit geometry that are fed to the process plant in the economic model. The final pit design and the internal phase designs were guided by the results of Whittle pit optimization analyses.

15.1 Pit Optimization

The LG analysis was performed in Maptek's Vulcan Envisage software, version 11.0.4. The software creates an optimum shell by using the LG algorithm. It uses metal prices, average cost inputs, and a block-by-block recovery model with an approximate slope angle to produce a theoretical maximum pit containing the highest net economic value possible. Only Measured and Indicated blocks were considered in the economic analysis.

15.1.1 Optimisation parameters

Pit optimisations were performed using the Net Smelter Return and block values calculated into the Datamine block model. The NSR includes only the payable revenue less the selling costs, including the smelter treatment charges and refining charges. The block values consider the mining, processing, and general and administrative costs.

Revenue parameters

Metal prices selling costs and other costs associated with the selling, freight, and penalties used for the economic pit analysis are shown in Table 15-1. Selling costs have been taken from the prior Technical Report (Sedgman, 2017).

Table 15-1: Lerchs-Grossman estimation parameters

Parameter	Units	Main	Minas	Este
Metal Prices				
Lead Price	$/lb	0.95	0.95	0.95
Zinc Price	$/lb	1.00	1.00	1.00
Silver Price	$/tr. oz.	20.00	20.00	20.00
Discount Rate	%	5.0	5.0	5.0
Unit Operating Cost				
Mining Cost - Ore	$/t mined	1.502	1.502	1.502
Mining Cost - Waste	$/t mined	1.840	1.840	1.840
Processing	$/t processed	9.251	9.251	9.251
G&A Cost	$/t processed	1.875	1.875	1.875
Unit Sustaining Capital Cost				
Mining	$/t mined	0.17	0.17	0.17
Processing	$/t processed	0.11	0.11	0.11
Process Recoveries				
Maximum Lead Recovery	%	98	98	98
Minimum Lead Recovery	%	2	2	2
Maximum Zinc Recovery	%	95	95	95
Minimum Zinc Recovery	%	0	0	0
Maximum Silver Recovery	%	95	95	95
Minimum Silver Recovery	%	0	0	0
Zinc in Lead Concentrate	%	9	9	9
Zinc Penalty Threshold	%	8	8	8
Penalty - applied per % above threshold	$/DMT	1.00	1.00	1.00
Zinc Penalty	%	1	1	1
Treatment Charges and Refining Charges (TCs/RCs)				
Pre-Mineral Tuff (default)	degrees	42	42	42
Post-Mineral Tuff	degrees	46	46	46
Bofedal	degrees	15	15	15
Mining Recovery Fraction	%	100	100	100
Mining Dilution Fraction	%	0	0	0
Production Rate				

Parameter	Units	Main	Minas	Este
Process Plant	Mt/y	9.855	9.855	9.855

DMT = dry metric tonne

Processing parameters

The processing plant throughput used was 9.855 Mt (dry) per year. The Project is considered to be “process limited,” rather than being limited by mining or selling capacity.

For optimization work, blocks coded as within mineral zones were considered for processing, while all other blocks were treated as waste.

The metal recovery equations applied to the economic pit analysis (detailed in Section 13.8) were used for calculating recoveries to the Pb-Ag and Zn concentrates.

Lead Recovery to Lead Concentrate – non-Transition zone material

$$\begin{aligned} &Lead\ Recovery\ to\ Final\ Lead/Silver\ Concentrate \\ &= 61.5 - 40.9 * max(0,0.57 - zinc) + 7.7 * max(0, galena - 0.38) + 45.4 \\ &* max(0,0.37 - goethite) - 0.12 * max(0, elevation - 4891) + 32.9 \\ &* max(0,0.27 - MnOxi) - 6.21 * max(0, pyrite - 1.07) - 16.4 \\ &* max(0,1.07 - pyrite) \end{aligned}$$

Lead Recovery to Lead Concentrate – Transition zone material

$$\begin{aligned} &Lead\ Recovery\ to\ Final\ Lead/Silver\ Concentrate \\ &= 35.6 - 40.9 * max(0,0.57 - zinc) + 7.7 * max(0, galena - 0.38) + 45.4 \\ &* max(0,0.37 - goethite) - 0.12 * max(0, elevation - 4891) + 32.9 \\ &* max(0,0.27 - MnOxi) - 6.21 * max(0, pyrite - 1.07) - 16.4 \\ &* max(0,1.07 - pyrite) \end{aligned}$$

Silver Recovery to Lead Concentrate – non-Transition zone material

$$\begin{aligned} &Silver\ Recovery\ to\ Final\ Lead/Silver\ Concentrate \\ &= 24.25 - 0.49 * \max(0,42.5 - silver) - 13.68 * \max(0, zinc - 1.52) + 17.06 \\ &* \max(0, zinc - 3.16) + 13.67 * \max(0,2 - Rock) - 2.69 \\ &* \max(0, Predicted\ Lead\ Recovery - 21.54) + 13.72 \\ &* \max(0, Predicted\ Lead\ Recovery - 30.33) - 10.64 \\ &* \max(0, Predicted\ Lead\ Recovery - 33.57) \end{aligned}$$

Silver Recovery to Lead Concentrate – Transition zone material

$$\begin{aligned} &Silver\ Recovery\ to\ Final\ Lead/Silver\ Concentrate \\ &= 26.14 - 0.49 * \max(0,42.5 - silver) - 13.68 * \max(0, zinc - 1.52) + 17.06 \\ &* \max(0, zinc - 3.16) + 13.67 * \max(0,2 - Rock) - 2.69 \\ &* \max(0, Predicted\ Lead\ Recovery - 21.54) + 13.72 \\ &* \max(0, Predicted\ Lead\ Recovery - 30.33) - 10.64 \\ &* \max(0, Predicted\ Lead\ Recovery - 33.57) \end{aligned}$$

Zinc Recovery to Zinc Concentrate

$$\begin{aligned} &Zinc\ Recovery\ to\ Final\ Zinc\ Concentrate \\ &= 84.4 - 50.6 * max(0,1.02 - zinc) - 0.15 * max(0, elevation - 4901) - 5.4 \\ &* max(0, pyrite - 1.9) - 11.2 * max(0,1.9 - pyrite) + 104.1 \\ &* max(0, copper - 0.03) + 1620.2 * max(0,0.03 - copper) \end{aligned}$$

Silver Displacement to Zinc Concentrate

$$\begin{aligned}&Silver\ Recovery\ in\ Zinc\ Concentrate\\&\qquad = 48.156\ -\ 0.544\ \times (Silver\ Recovery\ in\ Lead\ Concentrate)\end{aligned}$$

To match the metallurgical testing results, lead concentrate grades were fixed at 56.6% lead, and zinc concentrate grades were fixed at 52.9%. Silver grade in the zinc concentrate was limited to a maximum value of 430 g/t in the production schedule.

Economic pit vs design comparison

The result from the Vulcan optimization routine (Figure 15-1) was used to compare to the BCM ultimate pit designs (Figure 15-4). The phased pits are then scheduled to create the production schedule, which in turn forms the basis from which the feasibility study capital and operating costs are estimated. The estimated capital and operating costs are then used in the financial model.

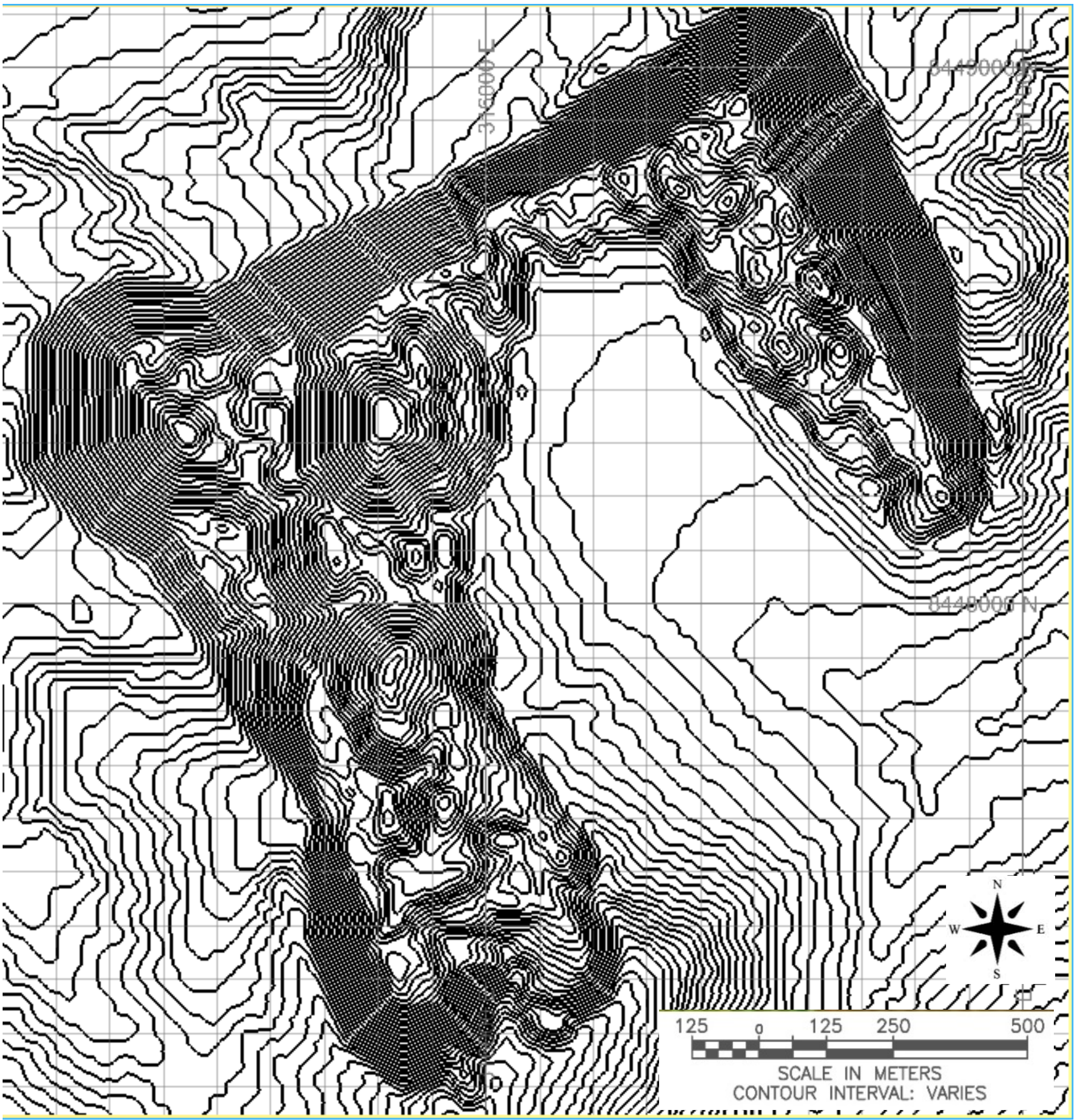


Figure 15-1: LG ultimate pit

15.2 Mine Design

15.2.1 Final pit design

The final pit was designed based on the use of 135-140 tonne capacity haul trucks, with ramp widths of 29 m in accordance with width requirements of the Peruvian mining regulations. Pit ramp grades are 10%. Ramp width is narrower, allowing for single lane traffic, for the final 3 to 4 benches in each pit. Figure 15-2 shows the haul road dimensions for two-way traffic.

Description	Dimension	CAT 785Da
Tyre Model		37.00 R51
Tyre Height (m)	A	3.06
Truck Width (m)	B	7.05
Windrow Height (m)	C= 3/4*A	2.30
Windrow Angle		37
Windrow Width (m)	E=C/tan(D)*2	6.09
Transitable Width (m)	F=B*3.0	21.15
Drains (m)	G	0.60
Ramp Width - Theoretical (m)	H=F+E+2*G	28.44
Ramp Width - Design (m)	I=round(H)	29.00

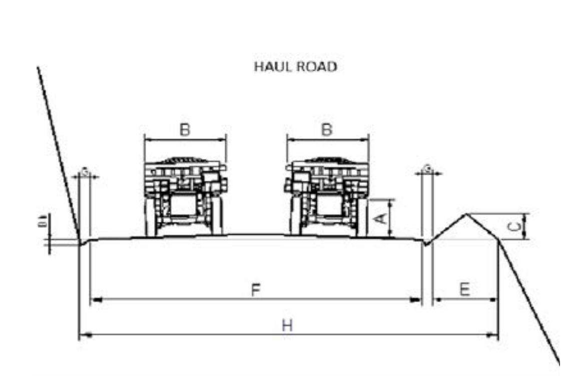
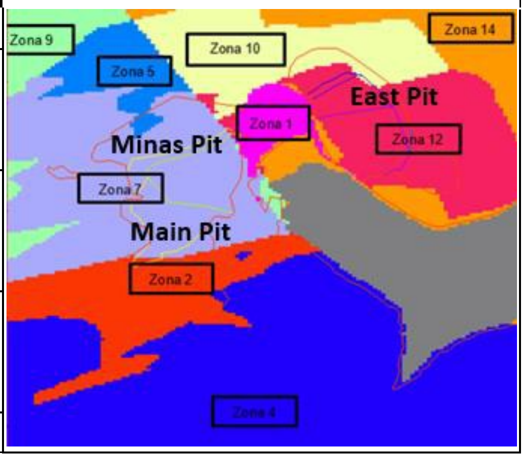


Figure 15-2:Haul road dimensions

Wall angles used in the design are from the McDonald 2012 and Anddes 2017 geotechnical studies and are shown in Figure 15-3.

Pit - Lithology	Zone	IRA Degrees	Face Angle Degrees	Bench Height m	Berm Width m
All – Bofedal	1	15	37	8.0	19.2
Main - Post Mineralization	15	45	70	16.0	10.2
Main - Pre Mineralisation	2	45	70	8.0	5.1
Main - Other	4	41	63	8.0	5.1
Minas - Post Mineralisation	5	38	58	16.0	10.3
Minas - Pre Mineralisation	7	45	70	8.0	5.2
Minas - Other	9	42	65	8.0	5.2
East - Post Mineralisation	10	46	70	16.0	9.6
East - Pre Mineralisation	12	46	70	8.0	4.8
East – Other	14	46	70	8.0	4.8
Other		46	70	8.0	4.8



Note: IRA = Inter Ramp Angle.

Figure 15-3: Wall angles

Figure 15-4 shows the final pit design.

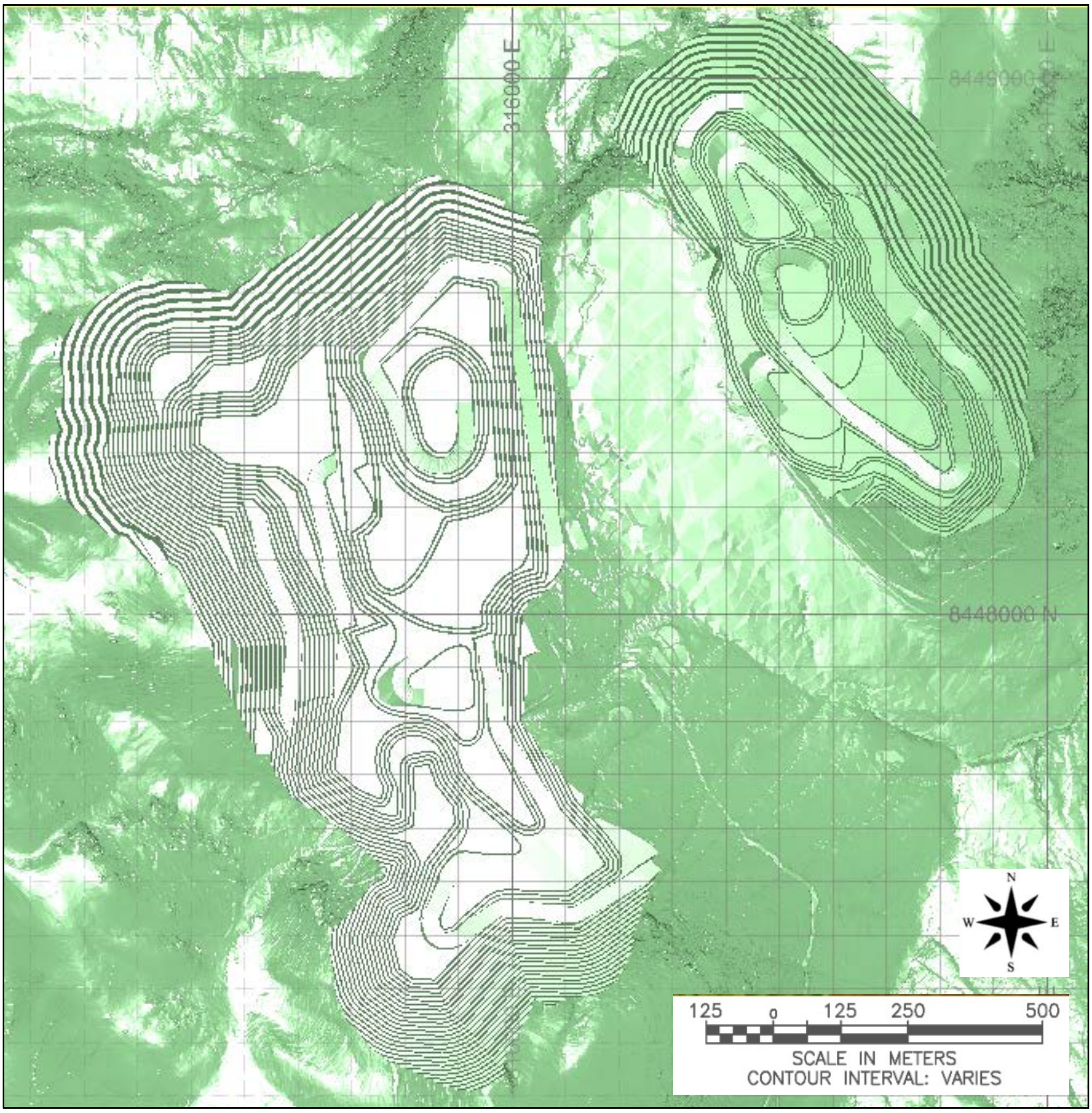


Figure 15-4: Final pit design plan

Table 15-2 shows the main dimensions of the final pit.

Table 15-2: Dimensions of final pit

Description	Unit	Pit		
		Este	Minas + Main	Total
Length (max)	m	1,050	1,630	n/a
Width (max)	m	550	1,000	n/a
Area	ha	45.2	105.1	150.3
Highest Point (Crest)	masl	5,050	5,130	5,130
Lowest Point	masl	4,786	4,754	4,754

Description	Unit	Pit		
		Este	Minas + Main	Total
Height of Highest Wall	m	262	348	348

15.2.2 Internal phase designs

To minimize initial pre-stripping requirements and target higher value ore in the early years, a series of internal pit phases has been developed. In total, there are eight phases (four at Este pit, three at Minas pit, one at Main pit). The different phases are shown in Figure 15-5, along with the distribution of NSR ($/t) for the bench 4,894 (note some phases are not mined on this particular bench).

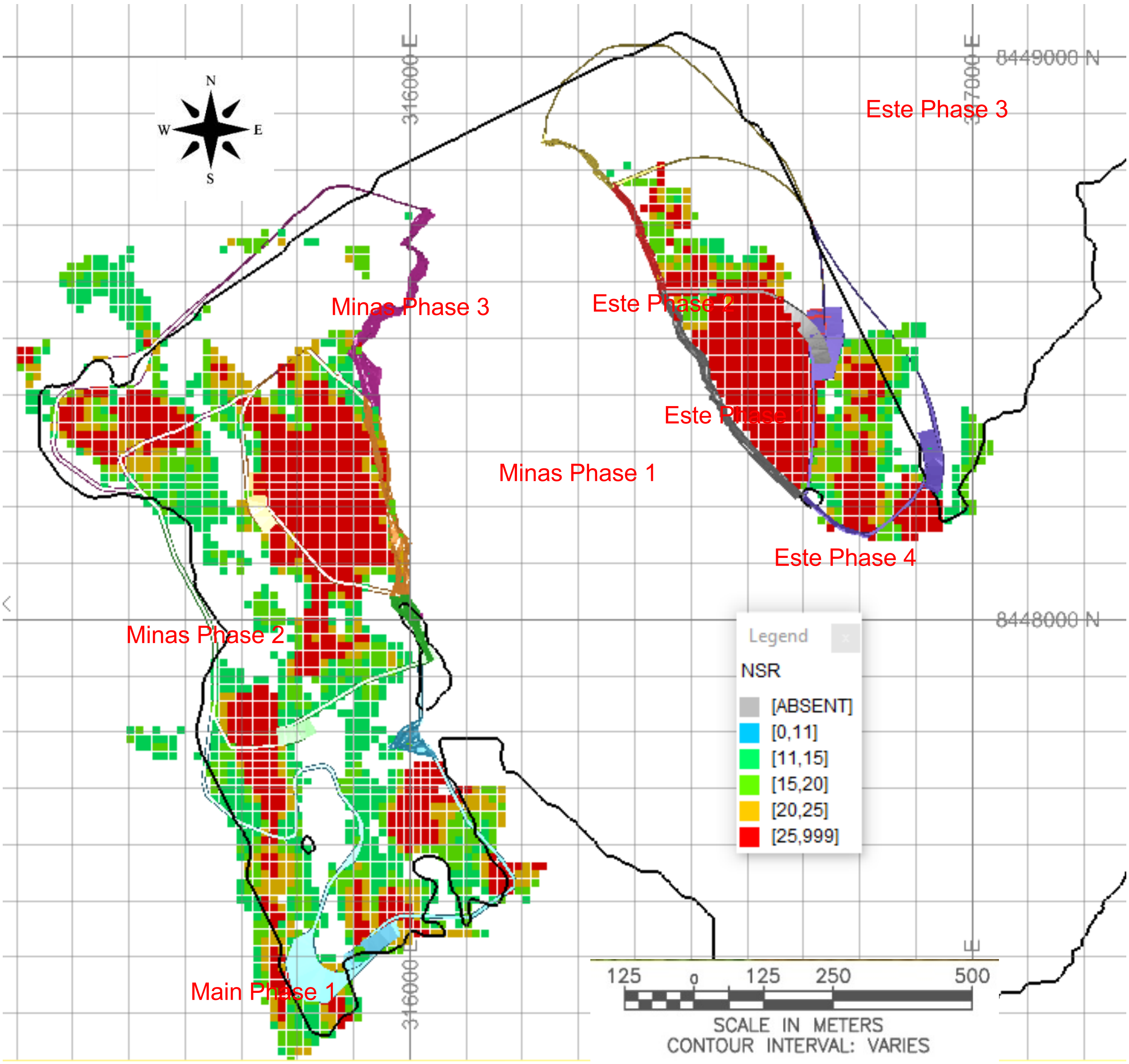


Figure 15-5: Pit phases

15.2.3 Haul road design

A system of haul roads is required to link the open pits to the primary crusher, short term ore stockpile, and the Main Mine Waste and Filtered Tailings Deposit. Haul roads 1, 2, 3, 5, 7, 11, 12 and 13 (a total of 7.56 km) must be constructed during the pre-strip period. This will involve 3.321 Mt of cut and 6.861 Mt of fill. The excess fill (i.e., the portion not sourced directly from cut) will be sourced from non-acid generating (NAG) waste from the Este pit during the pre-strip period. Haul roads will be finished with 1 m of base where required (estimated at 25% of haul road length, as most haul roads are constructed from mine fill, which will not require a layer of base), and 0.4 m of wearing course. Culverts to direct the water from one side of the haul road to the other will be installed at approximately 200-m spacings depending on gradient and catchment area.

Five further haul-roads (4, 6, 8, 9, and 10, totalling 2.12 km) are required to be constructed during the operational life of the mine, in years two, five, and eight.

The haul roads are shown on Figure 15-6.

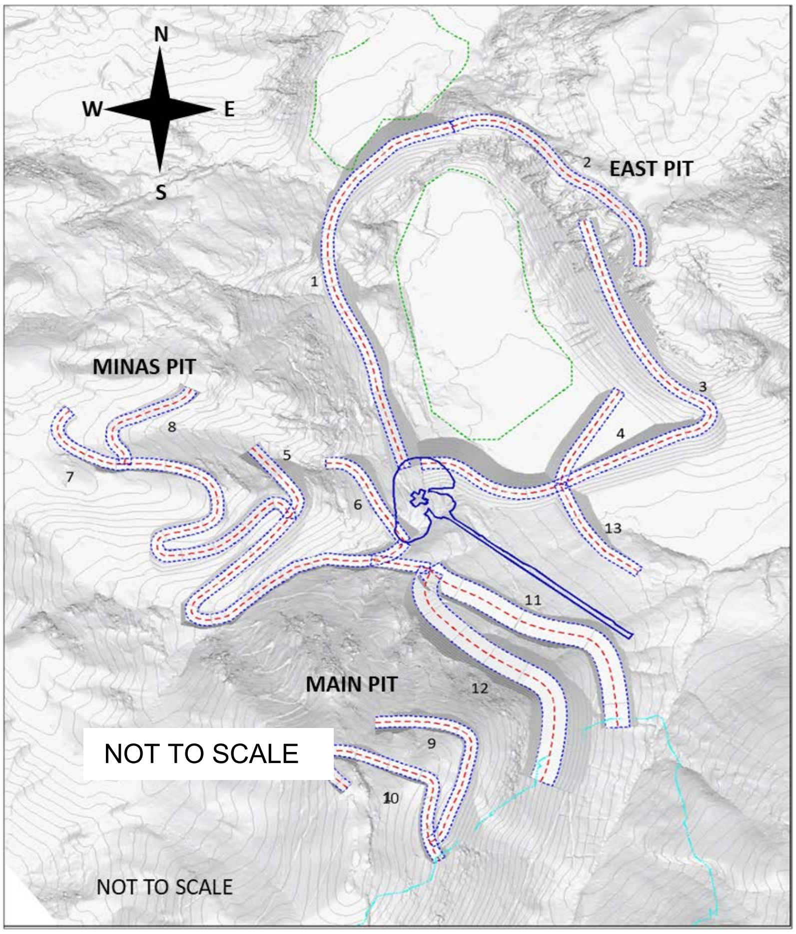


Figure 15-6: Haul roads

15.2.4 Short term stockpile design

A short-term stockpile will be used during the pre-strip stage and until the end of production. The stockpile is located within the East pit limit (as shown in Figure 15-7) and is primarily used to maintain the Zn/Pb ratio above 0.4. The maximum capacity is 3.0 Mt.

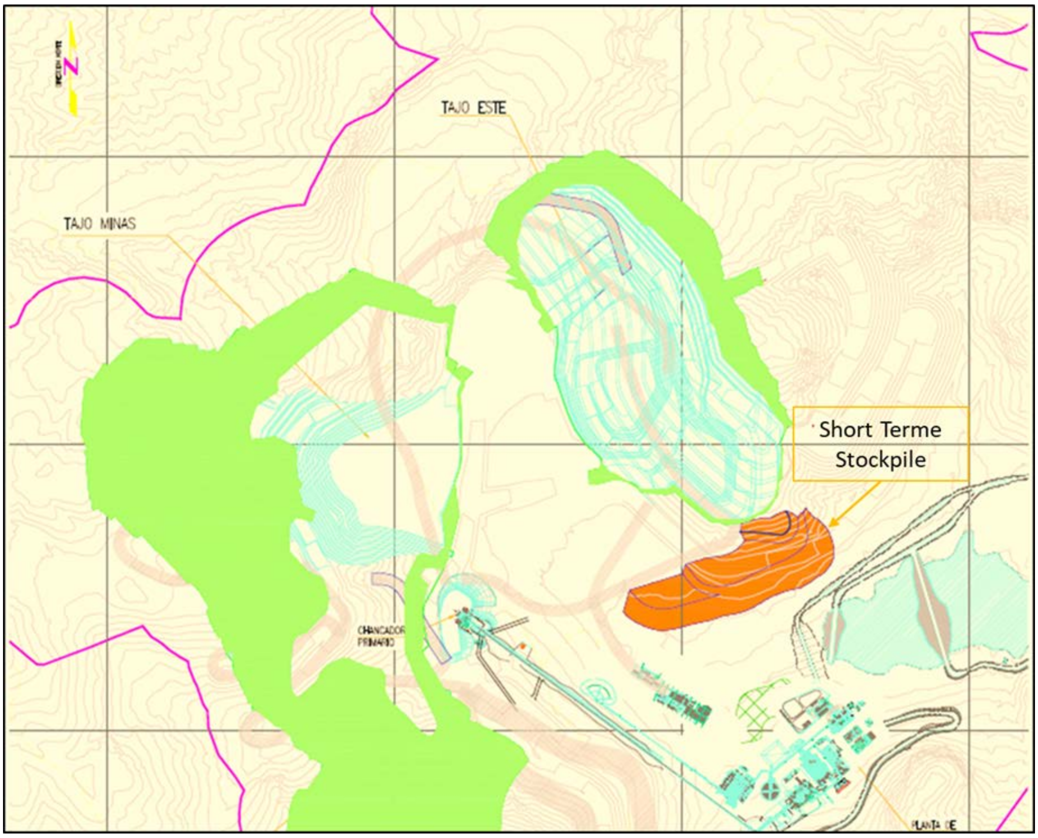


Figure 15-7: Short-term stockpile location

15.3 Mineral Reserves

The Project Mineral Reserves consider only measured and indicated resource categories, which have been converted to proven and probable reserves categories, respectively. Mineral Reserves are defined as being the material to be fed to the process plant in the mine plan already described and are demonstrated to be economically viable in this report's economic model. The Mineral Reserves are shown in Table 15-3.

Table 15-3: Corani Project mineral reserves

Classification	Tonnes Kt (dry)	Grade			NSR $/t	Contained Metal		
		Silver g/t	Lead %	Zinc %		Silver Moz	Lead Mlb	Zinc Mlb
Proven	20,330	59.7	1.00	0.60	34.02	39.0	450.0	268.5
Probable	118,253	49.9	0.88	0.55	29.48	189.6	2,291.8	1,425.9
Total Proven + Probable	**138,582**	**51.3**	**0.90**	**0.55**	**30.15**	**228.6**	**2,741.7**	**1,694.4**

Notes:

1. The Mineral Reserves have been estimated using the definitions of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM).
2. The Mineral Reserves have been estimated using the following metal prices: \$20.00/oz Ag, \$1.00/lb Zn, \$0.95/lb Pb using a revenue factor 1.00 pit shell as a basis for the pit design.
3. Only pre-mineral tuff type of material has been considered as reserves.
4. NSR Cutoff grades used are equal or higher than: \$10.79/t.
5. The effective date for these Mineral Reserves is 5 November 2019.
6. Totals / Averages may not add up due to rounding of individual tonnes and grades.
7. The tonnes and grades shown above are considered a Mineral Reserve because they have been demonstrated to be economically viable through this report's study financial model using the following metal prices: \$18.00/oz Ag, \$1.10/lb Zn, \$0.95/lb Pb.

The Mineral Reserves from this report differ only very slightly when compared to the Mineral Reserves detailed in the 2017 NI 43-101 report (Sedgman, 2017). The current study reserve tonnes are 99.6% of the NI 43-101 tonnage, while the contained silver is approximately 2% more, the lead metal is approximately the same, and the contained zinc metal is approximately 6% less. The Mineral Reserves from the NI 43-101 report are shown in Table 15-4.

Table 15-4: 2017 NI 43-101 Mineral Reserves

Classification	Tonnes Mt (dry)	Grade		Contained Metal			
		Silver g/t	Lead %	Zinc %	Silver Moz	Lead Mlb	Zinc Mlb
Proven	20.8	65.8	1.03	0.71	44	472	323
Probable	118.3	47.5	0.87	0.57	181	2274	1486
Total Proven + Probable	**139.1**	**50.3**	**0.90**	**0.59**	**225**	**2,746**	**1,809**

Mineral Reserves: from the Sedgman NI 43-101 report (2017)

16 Mining Methods

16.1 Summary

The Corani deposit is planned to be mined using conventional open pit mining methods. The mine design and planning are based on the resource model and reserve estimate as indicated in the previous sections.

The mine plan considers extraction of the proven and probable ore material included in the mineral reserve presented in Section 15. The mine plan has been developed to deliver 9,855 kt of ore per year (27,000 t/d x 365 processing days) to the crusher for processing by flotation to produce two concentrates: lead-silver, and zinc-silver.

The mine plan development included the following:

- ultimate pit design including benches, ramps, and haul roads (Section 15)
- pit phase design based on the incremental Whittle Shells (Section 15)
- detailed pit phase designs with benches, ramps, and haul roads (Section 15)
- phase design
- mine production scheduling
- waste storage design and material allocation
- time sequence mine plan drawing development
- equipment and manpower requirement calculations.

The following sections detail the development of the mine plan.

16.2 Phase Design and Pre-stripping

Pre-striping of 18.54 Mt, over a period of 17 months, is required to allow sustainable mine production to commence in Jan 2024, which coincides with the date when the process plant will be ready to start its ramp-up. Pre-stripping will be conducted in two distinct phases:

- Phase 1, with construction equipment (10 months, 4.74 Mt). August 2022 to May 2023. The primary purpose is to utilize mine waste to construct access to the Este pit, platform of the Main Mine Waste and Filtered Tailings Deposit and process plant site within the project area.
- Phase 2, with the same mining equipment to be employed during the Production period (7 months, 13.81 Mt). June 2023 to December 2023.

16.3 Pre-strip Phase 1, Initial Haul Road and Access Road Construction

Excavation of 4.74 Mt (2.00 Mbcm) of material from the East pit, and a further cut-to-fill volume in the haul roads, is required for the construction of haul roads, filling of the start platform of the Main Mine waste and filter tailings deposit and the ore stockpile. This work will be carried out over a 10 month period and includes:

- topsoil stripping at East Pit Phase 1, haul roads and ore stockpile Construction of haul road bulk fills using mine waste and cut-to-fill
- crushing and screening and placement of haul road base (1.0 m) in selected areas and wearing course (0.4 m) on haul roads.

- lining of an estimated 50% of haul road water tables with high density polyethylene (HDPE) and installation of culverts.
- provision and installation of reflective guideposts.

This work is considered as a separate contract, and the resources identified in Table 16-1 will be required.

Table 16-1: Pre-strip phase 1 and construction of haul roads: equipment

Equipment	Aug-22	Sept-22	Oct-22	Nov-22	Dec-22	Jan-23	Feb-23	Mar-23	Apr-23	May-23
Bulldozer CAT D8		3	9	9	9	9	9	9	6	3
Bulldozer CAT D6		1	1	1	1	1	1	1	1	0
Rock Drill 3.5"-5"		0	5	5	5	5	3	3	1	0
Excavator CAT 374		0	4	4	4	3	2	2	1	0
Excavator CAT 336		2	3	3	4	4	3	3	2	1
Excavator CAT 320		1	2	2	2	2	2	1	1	0
Front-end Loader CAT 966		2	2	2	2	2	0	0	0	0
Grader CAT 140		0	0	0	0	2	2	2	2	2
Tip-truck 8x4		15	30	30	35	40	40	35	20	10
Water Truck 5000 gal		2	4	5	5	5	5	5	3	2
Compactor 19t		0	1	3	4	4	4	4	4	2
Compactor 10t		0	0	0	1	1	2	2	2	2
Crusher and Screening		1	1	1	1	1	1	0	0	0
Backhoe CAT 420		1	1	1	1	2	2	2	1	1
Lighting Tower	2	6	12	24	24	30	30	24	24	12
Total	**2**	**34**	**75**	**90**	**98**	**111**	**106**	**93**	**68**	**35**

Personnel requirements range between 43 and 280 during the construction period.

16.4 Mining Contractor

The mine will be operated by a mining contractor. General mine coordination functions will be carried out by the mining contractor, which is typical practice in the industry. BCM will perform the mine planning (including geotechnical and survey), and geology functions.

16.5 Mine Production and Process Plant Schedules

Mine production schedules were developed to meet mining and processing constraints. The mine production schedule is focused on maximizing project Net Present Value (NPV) using an elevated cutoff grade in early mine life and a stockpiling strategy to keep the process plant fully supplied and minimizing the periods which exceed the concentrate handling capacities. Similar to the 2017 scheduling effort, the strategies evaluated were:

- Variable cutoff grade (NSR $/t), while mining the highest-grade phases to maximize overall NPV. This effort was to establish the relative value of increasing the mill throughput versus the capital required to facilitate that expansion.

- Variable cutoff grade (NSR $/t), while mining the highest-grade phases and stockpile to attempt to respect the limits on mass flows of concentrate. The goals for concentrate mass flows were 20 tonnes per hour for lead concentrates and 14 tonnes per hour for zinc concentrates.

The initial NSR cutoffs were set to $12 /t for the Este phases 1 and 2 and the Minas Phase 1 pits. The remaining phases were scheduled using the $10.79 /t cutoff. The second strategy used to increase the NSR was to schedule phases without concluding mining within the phase. This suggests that there are some optimizations to be made in phase designs.

The mine schedule includes a pre-strip phase in the Este Phase 2 pit, in which 12,400 kilotonnes (kt) of waste will be mined. In addition, 5,300 kt of ore will be mined and 90 kt stockpiled during the same period. The total pre-strip requirement is 31,000 kt. The pre-strip period is broken into a 10-month Phase 1 period and a 7-month Phase 2 period. During the Phase 1 period, construction equipment will be used. In the Phase 2 period (June to December 2023), mining equipment will be used to mine the remaining 12,400 kt.

Following completion of the Phase 2 pre-stripping, phases 1 through 4 of the Este pit will be mined continuously. Phase 4 of the Este pit follows in 2032 through 2034. Backfilling of the pit with mine waste and filtered tailings will commence in 2034 and take 4 years to complete.

Mining of Minas pit will commence 2024 and will continue almost continuously until the end of the pit in 2037, except during 2032 when mining in this pit is not required.

Mining at the Main pit commences in 2027 and will continue almost continuously until 2032.

The mine production schedule is shown in Table 16-2 and Table 16-3.

16.5.1 Mine production schedule

To provide an appropriate blend of feedstock to the process facilities, total material movements are highest from 2024 through 2027 (average 27.1 Mt mined per year), and then reduces over the next five years to an average of 18.5 Mt per year. The production requirements then increase to 27.5 Mt per year over the years from 2033 to 2035. From 2035 onwards, mining requirements reduce to 15.0 Mt average in the final three years, with mining being completed in early 2038.

Table 16-2: Mine production schedule – by Phase (Mt)

Pit	Phase	2023 Pre-strip	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	Total (Mt)
Este Phase 1	1	7.0	11.3	6.2														24.5
Este Phase 2	2	11.5	16.6	5.7	3.1													36.8
Minas Phase 1	3			11.3	8.9	7.7	0.5											28.4
Este Phase 3	4			4.4	17.1	5.3												26.9
Minas Phase 2	5					11.0	8.0	7.8	11.1	14.9	14.9							67.6
Main Phase 1	6						9.9	9.3	8.3	5.1	3.0	17.1						52.7
Este Phase 4	7											5.1	11.6	6.1				22.8
Minas Phase 3	8											2.5	18.7	21.3	16.2	13.9	2.8	75.4
Total		18.5	27.9	27.5	29.1	24.0	18.4	17.0	19.4	20.0	17.9	24.7	30.3	27.4	16.2	13.9	2.8	335.0

Table 16-3: Mine production schedule – material movement by type

Period	Ore Mined to Plant (kt)	Ore Stockpiled (kt)	Ore Mined total (kt)	Waste Mined NAG (kt)	Waste Mined PAG (kt)	Waste Mined Total (kt)	Total Mined (kt)	Strip Ratio	Ore Rehandled to Plant (kt)	Total Material Moved (kt)	Ore Feed to Plant (kt)
2023 Q2	0	0	0	1,579	0	1,579	1,579	-	0	1,579	0
2023 Q3	0	0	0	7,025	0	7,025	7,025	-	0	7,025	0
2023 Q4	0	0	0	9,915	21	9,936	9,936	-	0	9,936	0
2024 Q1	1,164	23	1,186	7,636	106	7,742	8,929	6.53	23	8,951	1,186
2024 Q2	2,446	11	2,456	6,030	314	6,344	8,800	2.58	0	8,800	2,446
2024 Q3	2,484	0	2,484	3,931	268	4,199	6,683	1.69	0	6,683	2,484
2024 Q4	2,484	0	2,484	613	349	961	3,445	0.39	0	3,445	2,484
2025 Q1	2,457	0	2,457	282	317	598	3,055	0.24	0	3,055	2,457
2025 Q2	2,457	0	2,457	902	508	1,410	3,867	0.57	0	3,867	2,457
2025 Q3	2,484	0	2,484	5,074	615	5,689	8,173	2.29	0	8,173	2,484
2025 Q4	2,484	430	2,914	8,832	677	9,509	12,423	3.26	0	12,423	2,484
2026	9,824	0	9,824	18,042	1,247	19,289	29,114	1.96	31	29,144	9,855
2027	9,855	2,519	12,374	10,510	1,145	11,654	24,028	0.94	0	24,028	9,855
2028	7,492	0	7,492	10,263	627	10,890	18,382	1.45	2,363	20,745	9,855
2029	9,374	0	9,374	6,524	1,124	7,648	17,022	0.82	508	17,530	9,882
2030	9,855	0	9,855	6,636	2,869	9,505	19,360	0.96	0	19,360	9,855
2031	9,803	321	10,124	4,674	5,157	9,831	19,955	0.97	52	20,007	9,855
2032	9,531	0	9,531	1,733	6,648	8,382	17,913	0.88	324	18,237	9,855
2033	9,882	153	10,035	8,341	6,374	14,714	24,749	1.47	0	24,749	9,882
2034	9,708	0	9,708	18,506	2,038	20,544	30,251	2.12	147	30,399	9,855
2035	9,802	98	9,900	14,649	2,869	17,517	27,418	1.77	8	27,426	9,811
2036	9,855	0	9,855	3,281	3,057	6,338	16,193	0.64	0	16,193	9,855
2037	9,882	0	9,882	391	3,581	3,972	13,854	0.40	0	13,854	9,882
2038	1,705	0	1,705	31	1,112	1,143	2,849	0.67	98	2,946	1,803
Total	135,028	3,554	138,582	155,399	41,022	196,421	335,003	1.42	3,554	338,557	138,582

16.5.2 Process plant feed schedule

Ore to be fed to the primary crusher of the process plant will be sourced directly from the mine (and also from the short-term stockpile until October 2024). The tonnes and grades to be fed per period are shown in Table 16-4.

Table 16-4: Process plant feed schedule

Period	Ore Sent to Process Plant (kt)	NSR ($/t)	Grade Ag (g/t)	Grade Pb (%)	Grade Zn (%)
2023	0	0	0.00	0.00	0.00
2024	8,600	55.24	99.77	1.10	0.84
2025	9,882	47.02	70.88	1.04	0.77
2026	9,855	48.21	77.83	1.12	0.69
2027	9,855	49.20	61.51	1.34	0.84
2028	9,855	29.77	68.85	1.18	0.36
2029	9,882	25.60	55.89	0.96	0.28
2030	9,855	23.79	45.80	0.72	0.29
2031	9,855	25.08	33.39	0.70	0.50
2032	9,855	27.37	24.95	0.65	0.81
2033	9,882	27.67	33.13	0.65	0.67
2034	9,855	26.49	47.33	0.82	0.37
2035	9,811	30.38	47.93	0.71	0.49
2036	9,855	24.38	30.83	0.67	0.43
2037	9,882	28.00	31.33	0.92	0.45
2038	1,803	29.25	24.86	0.90	0.68
Total	138,582	33.19	51.31	0.90	0.55

16.6 Waste Management

Mine waste is classified as either non-acid generating (NAG) or potentially acid generating (PAG) based on the Net Neutralizing Potential (NNP) field in the resource block model supplied by BCM, blocks with NNP values less than 20 are considered to be PAG and blocks with more than 20 are considered to be NAG. In total, 79% of the waste to be mined is classified as NAG. During the pre-strip period, 99.9% of the waste to be mined is NAG, and this will be used to construct the initial haul roads, primary crusher platform, the starter platform of the Main Mine Waste and Filtered Tailings Deposit (DDMR) and a dyke in the upstream zone of the deposit.

During the production period, waste rock will be co-disposed with filtered tailings into the Main Mine Waste and Filtered Tailings Deposit. A 25 m wide layer of NAG material will be placed on the outer shell of the deposit to encapsulate the PAG material and tailings. In years 10, 11 and 12 part of the waste material will be deposited as backfill in the Este, Main and Minas pits.

The Main Mine Waste and Filtered Tailings Deposit was designed by Anddes and is discussed in more detail in Section 18.3 of this report. Generally, the downstream toe will be at approximately 4,850 masl and the top will be at 5,175 masl, giving an overall height of approximately 325 m. A starter platform will be constructed by using NAG mine waste, downstream face angles of 24 degrees will be used, with 7.5 m wide berms spaced every 10 m vertical, giving an overall angle of 18 degrees. The downstream face will be profiled and rehabilitated progressively during the mine life. Figure 16-1 and Figure 16-2 show the layout and section of the Main Mine Waste and Filtered Tailings Deposit. The unsuitable natural material consisting of non-saturated, loose and soft soils obtained during the foundation stripping will be placed as part of the starter platform as shown in Figure 16-2.

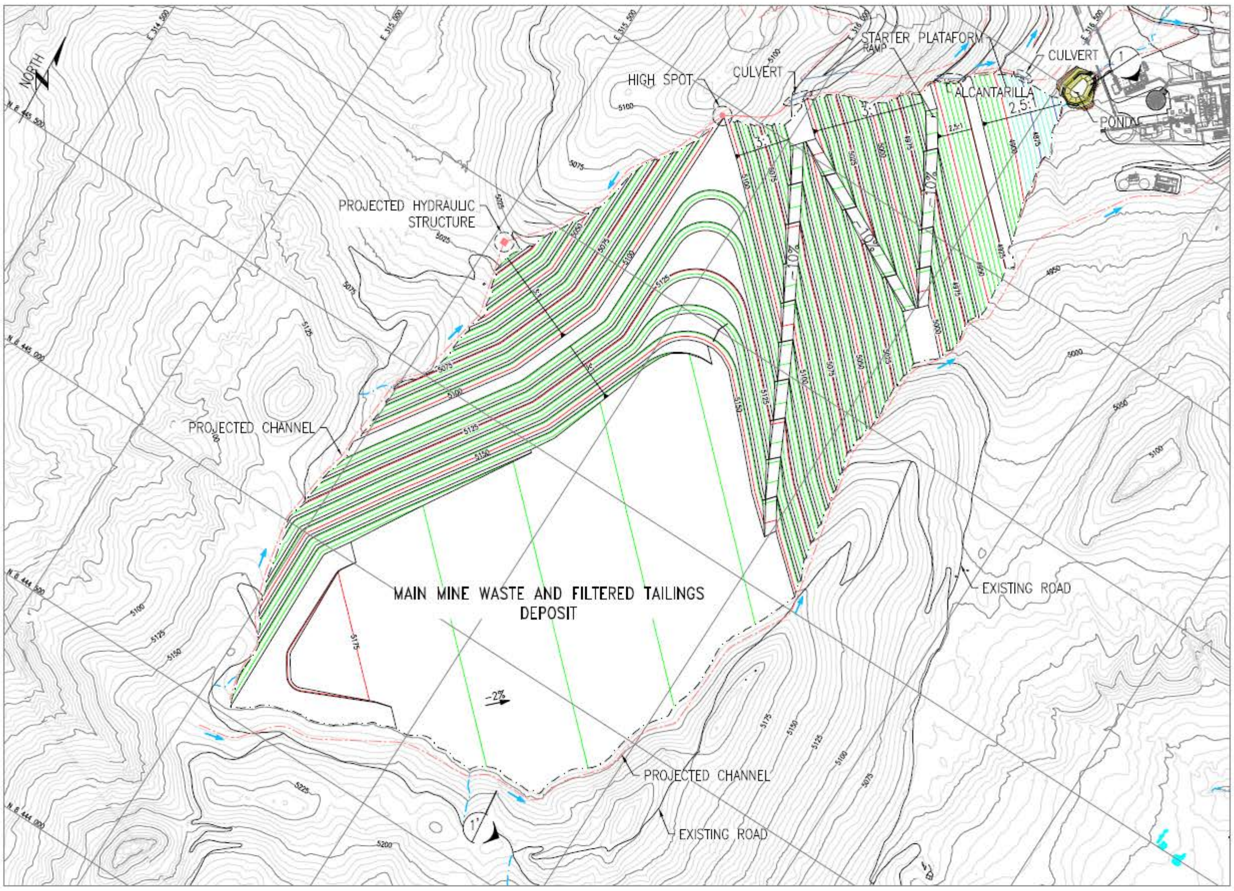


Figure 16-1: Layout of main mine waste and filtered tailings deposit

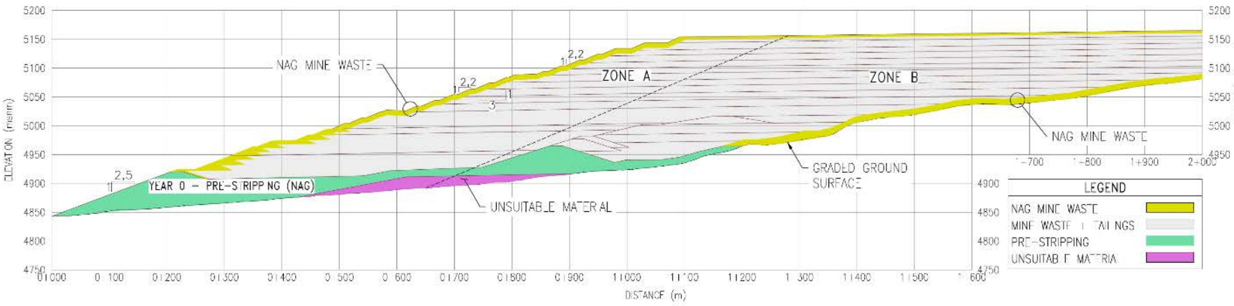


Figure 16-2: Section of main mine waste and filtered tailings deposit

Figure 16-3 and Figure 16-4 show the details of the waste-tailings co-disposal plan which is based on the mine plan provided by BCM. The co-disposal is shown on a monthly basis for the first two years and then annually until the end of the life of mine. The height of each layer depends on the material availability by month or year. The following criteria have been considered for the co-disposal plan:

- Filter press must provide tailings with an average (metallurgical, w/w) moisture of 17% and maximum of 19% to be hauled by trucks to the deposit.
- The starter platform will be conformed with 2% slope in the downstream direction to facilitate the water runoff and management. All the subsequent co-disposal platforms will have the same slope, assuming a uniform thickness.

- The deposit has been divided in two zones: Zone A located in the downstream side which controls geotechnical stability (see Figure 16-3) and Zone B in the upstream side (see Figure 16-4). Those zones have been divided by an imaginary line representing approximately the most critical failure surface based on the stability analysis. A dyke will be constructed between those zones with NAG mine waste for providing containment to the tailings of the year 01 disposal.
- During the dry season co-disposal will be carried out on Zone A, where maintaining shear strength of the materials is required. In order to generate a zone with sufficient geotechnical characteristics, just compliant materials must be placed in this area, i.e., filtered tailings with a moisture equal or lower than 17% and good quality mine waste. Compaction of these compliant filtered tailings is not required.
- The co-disposal during the wet season will be carried out in Zone B, where the reduced shear strength of the wet filtered tailings can be accepted.
- Zone B will also be used for disposing non-compliant tailings with metallurgical moistures over 17% w/w as well as compliant tailings during periods when insufficient waste material is mined. A monitoring of the filtered tailings moisture and mine waste quality is required permanently, this must be described in detail in the operations manual.
- Mine trucks will be used to haul both mine waste and filtered tailings from the pits and the tailings stockpile, respectively.

The following aspects have been considered for placing mine waste and tailings:

- Mine operation will start on January 2024 (year 01, month 01) under rainy season conditions. Therefore, co-disposal will start in Zone B.
- As shown in Figure 16-4, in year 01 month 01, mine waste and tailings will be co-disposed, but in year 01 months 02 and 03, mine waste availability is not sufficient, therefore filtered tailings will be disposed in isolation during those months.
- Starting the dry season in year 01 month 04 and up to month 07, mine waste and tailings will be co-disposed on Zone A. However, between months 08, 09 and 10, still in the dry season, and months 11 and 12 already in the wet season, mine waste is not available from the pits or insufficient, therefore, only tailings will be disposed in Zone B on those months (year 01, months 08 to 12).
- Similar disposal layout for wet and dry season will be followed in year 02 starting in Zone B and continuing in Zone A. Also, because mine waste is not available all the time or insufficient, just filtered tailings will be disposed in months 01, 02 and 03 of year 02.
- Similar strategies will be employed from year 3 onwards, and this is planned on an annual basis.
- In general, when mine waste is not available from the pits or insufficient for co-disposal and only tailings must be disposed in the deposit, the process must be done in Zone B. Zone A must be used just for mine waste and compliant filtered tailings co-disposal.
- In all cases when mine trucks must travel on a surface of tailings materials; a minimum 1 m layer of mine waste will be placed for surface reinforcement to facilitate traffic.

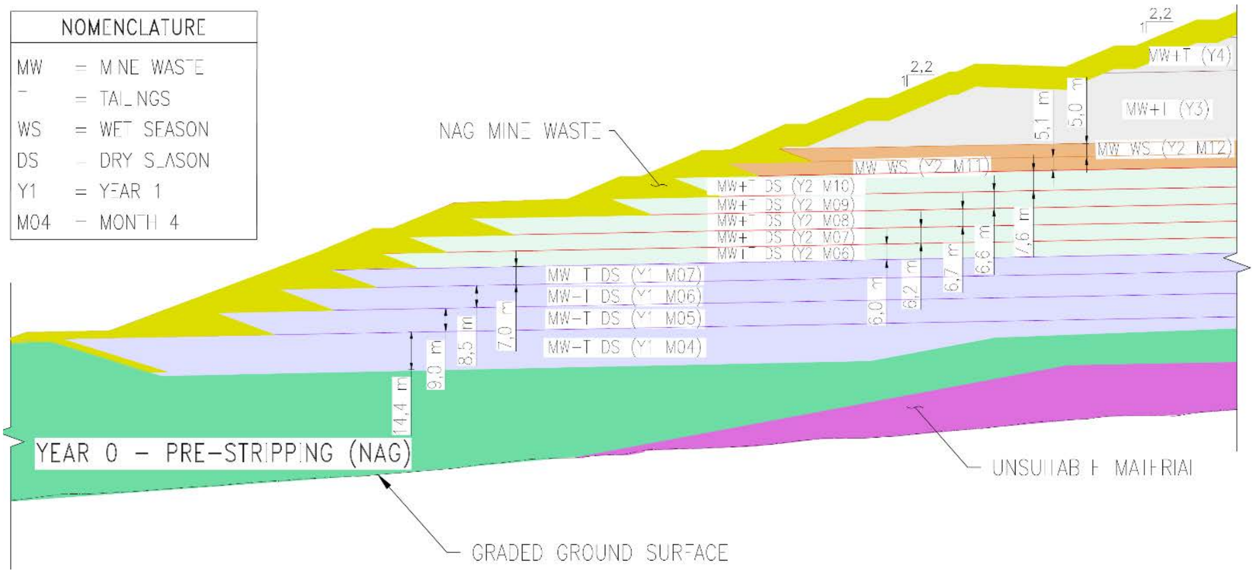


Figure 16-3: Mine waste-filtered tailings co-disposal - Zone A

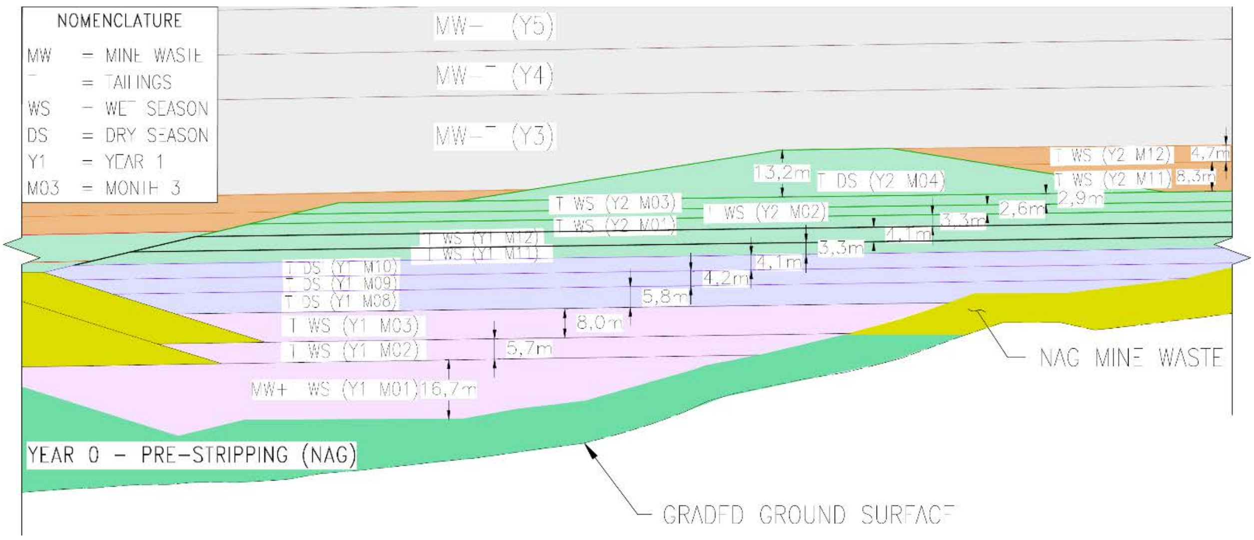


Figure 16-4: Mine waste-filtered tailings co-disposal - Zone B

Two alternatives methods may will be used for the co-disposal:

Method A - Independent modules

- In Zone A, and when mine waste is available for Zone B, the filtered tailings and mine waste will be placed in separate adjacent modules parallel to the storage facility slope (perpendicular to the axis of the gorge) as shown in the Figure 16-5. The width and height of each module will depend on the tailings/waste ratio in the operational period.
- The co-disposal must be performed by generating a module (platform) of mine waste on which trucks can travel and unload the material. Filtered tailings will be discharged at the side end of the mine waste module and the conformation of tailings module will be done with a dozer pushing the materials laterally.

- Trucks may not travel directly on a tailings surface, therefore, for the conformation of an upper module of mine waste on the surface of the lower tailings module, a 1 m layer of mine waste will be placed on the tailings surface to guarantee a safe operation of trucks.
- The disposal will be performed from upstream to downstream all the times, in order to facilitate water collection and management.

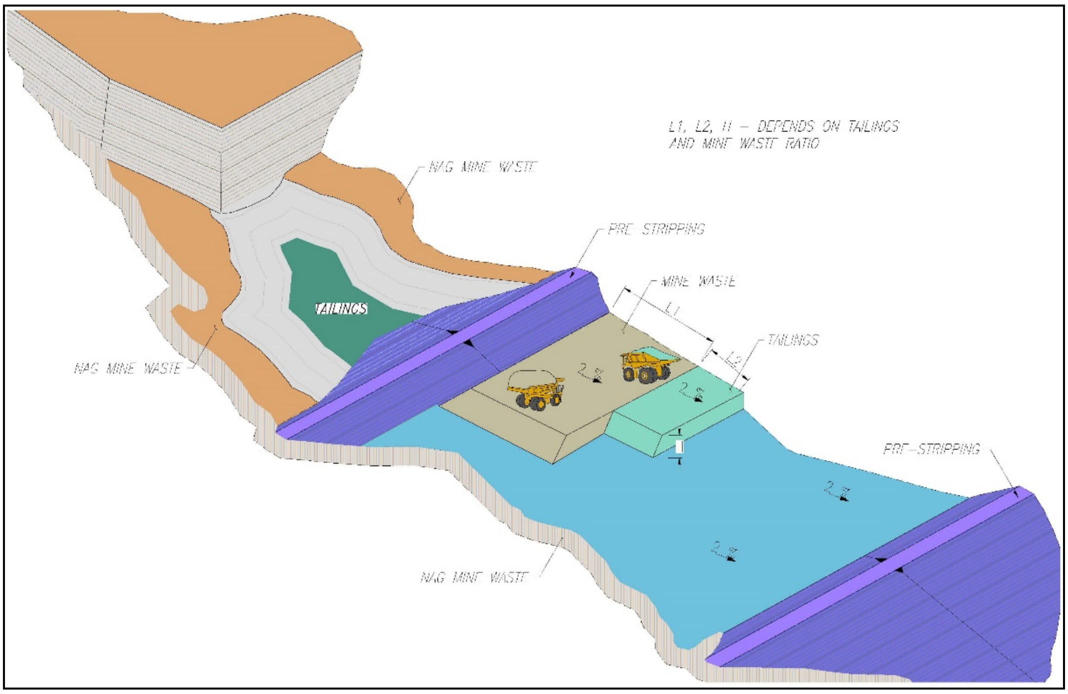


Figure 16-5: Mine waste-filtered tailings co-disposal in independent modules

Method B - Single layer:

- In Zone A, and in general when mine waste is available for the whole area, mine waste and filtered tailings will be discharged in the same location and a dozer will conform a single layer of 2 meters maximum height, as shown in Figure 16-6.
- In Zone B, during years 01 and 02 mine waste is not available or insufficient, then just tailings will be placed mainly in the wet season of those years.
- The dozer operation should generate a "blended" material that will improve the tailings strength. Also, traffic of the trucks will promote layer compaction improving the geotechnical characteristics of the co-disposed set.
- Trucks may travel directly on a surface of blended material, however, in case blending is not efficient and in some location tailings areas are exposed, then a 1 m of mine waste layer must be placed on the tailings surface to guarantee a safe operation of trucks.
- The disposal will be performed from upstream to downstream all the time, in order to facilitate water collection and management.

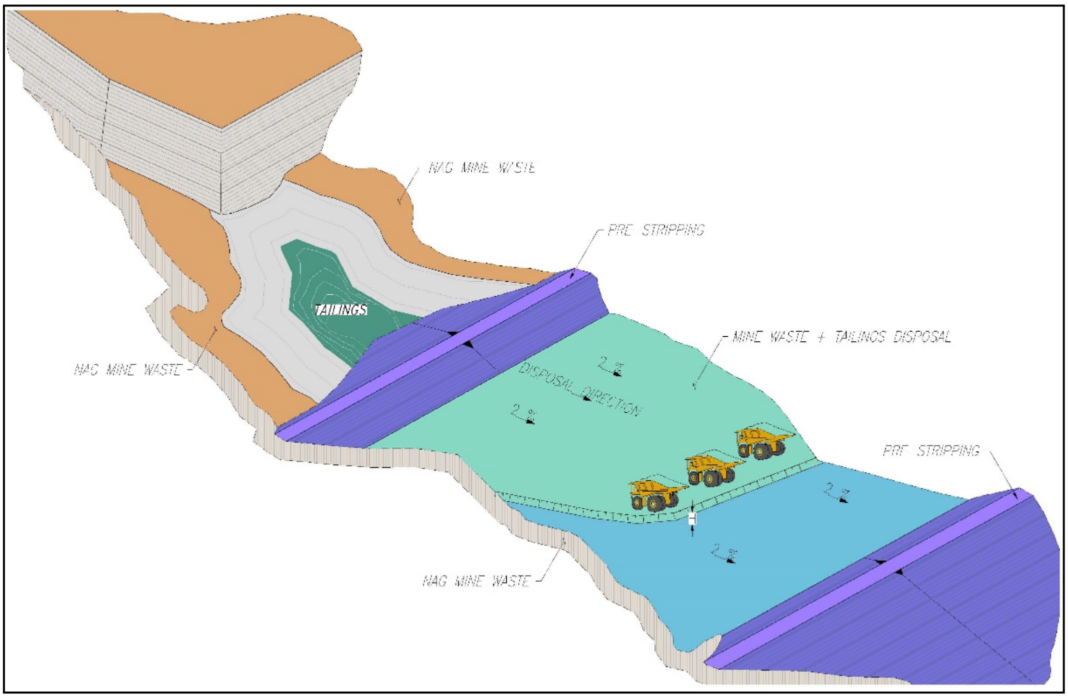


Figure 16-6: Mine waste-filtered tailings co-disposal in single layer

Method B is the preferred and planned method. However, the co-disposal method should be reviewed once the mine operations begin, and adjusted to the actual weather conditions and availability of mine waste and tailings. The objective will be that the materials be placed in such a way that the best geotechnical conditions are obtained in Zone A and, in general, the work is carried out in appropriate safety conditions for personnel and equipment at all times.

Impermeable covers or raincoats may be used during rainy season for managing runoff water, avoiding tailings saturation and reducing contact water and sediment transportation. Raincoats will be paced in the inactive areas of the deposit and the collected water will be diverted to a low spot for pumping out to the no contact water stream. This cost has been included in the financial model as a sustaining capital expenditure.

Between 2033 and 2035, NAG and PAG mine waste will be deposited as backfill in the Este, Minas and Main pits. 14 Mt of mine waste will be used to backfill East pit to 4,786 masl, while 21.5 Mt are required to backfill Minas and Main pits to 5,039 masl. In total 35.5 Mt of mine waste will be used to backfill the pits for three years. A small quantity of mine waste will continue to be sent to the Main Mine Waste and Filtered Tailings Deposit during this time, and from 2034 until the end of mining in March 2038 this will be the only destination for mine waste and filtered tailings.

Figure 16-7 shows the backfill of the both pits. The design basis of the mine waste dump into the pits considers local slopes of 38 degrees and global 22 degrees, in accordance with stability analysis. Local lifts will have 10 m height and setbacks of 7 m. The stacking ramps will have a total width of approximately 29 m and a maximum slope of 10%, based on the mine trucks operating parameters.

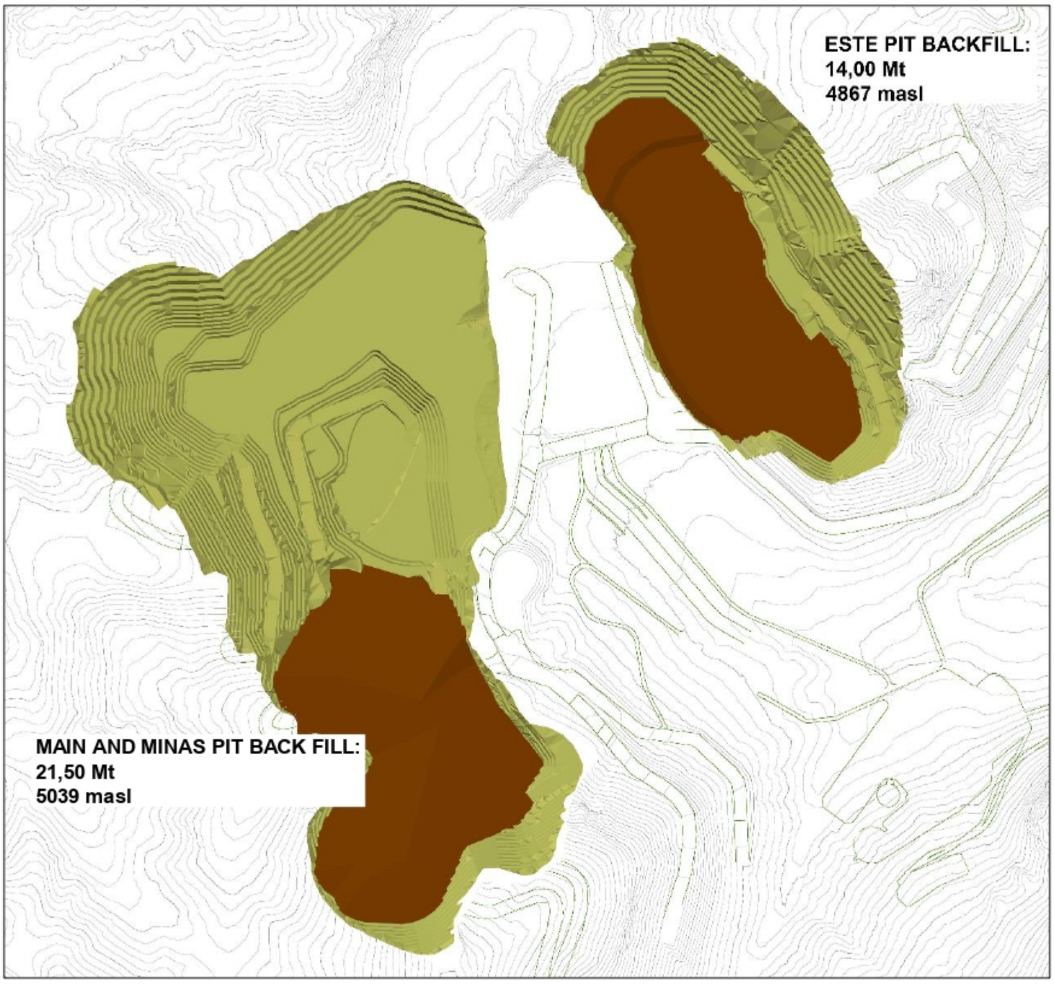


Figure 16-7: Mine waste backfill of pits

16.7 Mining Equipment

16.7.1 Drilling

Drilling will be carried out using drill rigs equipped with down-the-hole hammers. Based on the rock hardness documented in the report "Corani Blast Fragmentation Project, Topex" (2015), the rock is of low to medium hardness with an unconfined compressive strength ranging between 27.5 MPa and 36.9 MPa. The drill pattern selected allows for a powder factor of 0.26 kg/t, consistent with the requirement documented by Topex.

Drilling will be conducted using two Atlas Copco DM45 drills, with 156 mm diameter holes, using an average pattern as shown in the following table. These drills will work 169,464 hours during the life of the mine. The drill and blast parameters are summarized in Table 16-5.

Table 16-5: Drill and blast parameters

Parameter	Unit	Ore	Waste
Density (average)	t(dry)/bcm	2.4	2.31
Burden	m	4.4	4.4

Parameter	Unit	Ore	Waste
Spacing	m	5.06	5.06
Bench height	m	8	8
Subdrill	m	0.8	0.8
Total meters drilled per hole	m	8.8	8.8
Hole diameter	inch	6.125	6.125
Hole diameter	m	0.156	0.156
Stemming	m	3.11	3.11
Volume per hole	bcm/hole	178	178
Buffer	%	4.00%	4.00%
Re-drills	%	4.00%	4.00%
Penetration rate	m/h(SMU)	45	45
Meters drilled per hole (excl buffer, re-drills)	m/hole	8.8	8.8
Meters drilled per hole (incl buffer, re-drills)	m/hole	9.5	9.5
% ANFO	%	50%	50%
% HANFO 60/40	%	50%	50%
Density ANFO	g/cm^3	0.8	
Density HANFO 60/40	g/cm^3	1.25	
ANFO per meter charged	kg/m	7.6	7.6
HANFO 60EM/40AN per meter charged	kg/m	11.9	11.9
Explosives per hole ANFO	kg/hole	43.3	43.3
Explosives per hole HANFO 60/40	kg/hole	67.6	67.6
Explosives per hole total	kg/hole	110.8	110.8
Powder factor	kg/bcm	0.62	0.62
Powder factor	kg/t(dry)	0.26	0.27

A third drill will be required during the mine life. The third drill selected is an Atlas Copco ROC L8 and will drill between 76.2 mm to 127 mm diameter holes. This drill has been selected as it can also be utilized for drilling of wall control holes, secondary basting, and probe drilling to confirm locations and safe cover of underground workings.

Presplit drilling is envisaged for formation of stable bank walls and slopes, creation of the shovel blade penetration limit, and maintenance of scheduled bench profiles.

16.7.2 Blasting

Blasting activates will be undertaken by the mining contractor with either their own specialist team in blasting or utilizing a sub-contractor. In either scenario the contractor will provide the explosives and a “down-the-hole” service, whereby the contractor is responsible for designing drill patterns (with approval by the client), priming, loading, and firing of each blast. As some holes will be wet (rainfall and ground water), it is estimated that 50% of blasting will utilize ANFO (for dry holes) and 50% will utilize a waterproof blend of emulsion and ANFO. Explosives will be loaded into drill holes using a Mobile Mixing Unit Truck provided by the contractor. Blasting is expected to be conducted on average three times per week. All personnel within a radius of 500 m of the blast must be evacuated during firing time, as required by the Peruvian mining regulations.

Blasting accessories to be used include boosters (1 lb), downhole non-electric detonators, surface delay non-electric detonators, detonating cord, and non-electric lead-in line. To ensure acceptable blasting performance, crushed stemming will be used.

16.7.3 Loading, hauling, and ore rehandling

In addition to the mining requirements, some ore will need to be re-handled to the primary crusher from the short-term stockpiles (located within the East Pit limit), during the mine life.

Production loading will be undertaken using two Caterpillar 6040FS face shovels, loading Caterpillar 785D dump trucks. A Caterpillar 994F front-end loader will also be required for loading of stockpiled ore into the dump trucks and to allow loading activities to start up again quickly after blasting.

The characteristics of the loading and hauling equipment are shown in Table 16-6 and Table 16-7.

Table 16-6: Loading equipment

Parameter	Unit	Equipment Make/Model		
		Face Shovel Caterpillar	Face Shovel Caterpillar	Front-end Loader Caterpillar
		6040	6040	994F
		Material		
		Ore 2.40	Waste	Ore Rehandle
Density (average) moisture content	t(dry)/bcm)	2.4	2.31	-
Moisture content	%	5	5	5
Loose density (average)	t(wet)/bcm)	1.938	1.866	1.938
Operating weight	t	405	405	240
Net power	kW	1,516	1,516	1,297
Bucket capacity (heaped 2:1)	m^3	22	22	19
Bucket capacity (max payload)	t(wet)/pass	44	44	33.2
Bucket fill factor	%	90%	90%	90%
Bucket capacity (at bucket fill factor)	t(wet)/pass	39.6	39.6	29.88
Number of passes to fill Cat 785D truck	#	4	4	5
Loading cycle time	min	2.3	2.3	3.5
Physical availability	%	85.0%	85.0%	85.0%
Utilisation of available time (max possible)	%	85.4%	85.4%	85.4%
Loading time per day (max possible)	h	20.5	20.5	20.5
Average productivity (LOM)	t(wet)/h(SMU)	2,700	2,700	1,979

Table 16-7: Hauling equipment

Parameter	Unit	Equipment Make/Model		
		Dump Truck - Cat785D	Dump Truck - Cat785D	Dump Truck - Cat785D
		Loader - Caterpillar 6040FS	Loader - Caterpillar 6040FS	Loader - Caterpillar 994F
Gross vehicle weight	t	249	249	249
Net power	kW	1,005	1,005	1005
Tray capacity (nominal)	t(wet)	138.6	138.6	138.6
Payload average (matched to loader passes)	t(wet)	138.6	138.6	138.6
Physical availability	%	85.0%	85.0%	85.0%
Utilisation of available time (max possible)	%	85.4%	85.4%	85.4%
Productivity - maximum - LOM	t(wet)/h(SMU)	1225	1312	610
Productivity - minimum - LOM	t(wet)/h(SMU)	293	308	610
Productivity - average - LOM	t(wet)/h(SMU)	588	405	610

The shovel requirements are illustrated in Figure 16-8.

One caterpillar 994F front-end loader will be required for re-handle of ore. The front-end loader will work over the life of the mine.

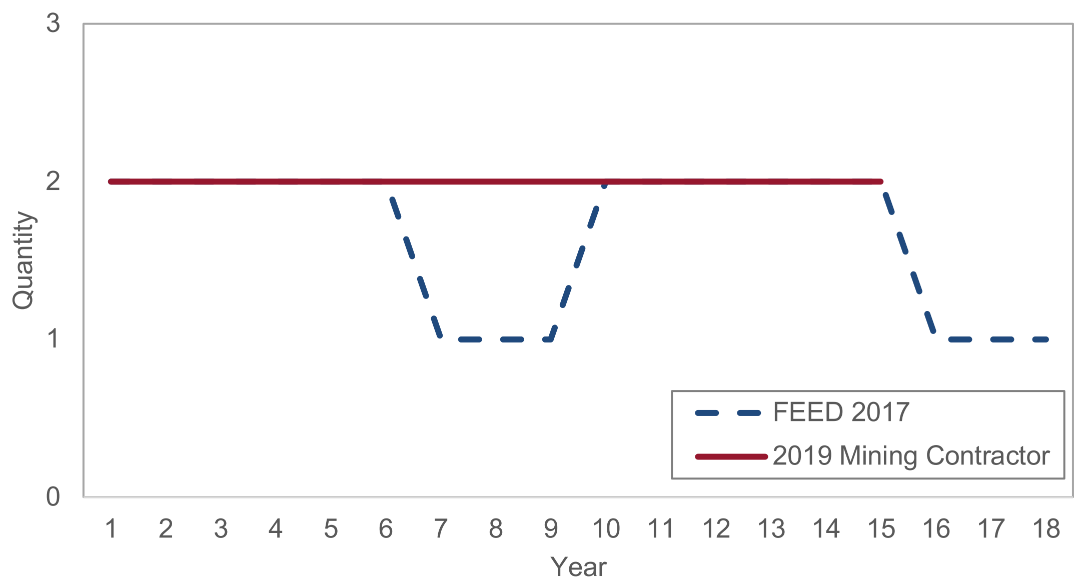


Figure 16-8: Quantity of shovels – Caterpillar 6040 (2017 FEED vs 2019 Contractor)

The quantity of dump trucks required and used is shown in Figure 16-9.

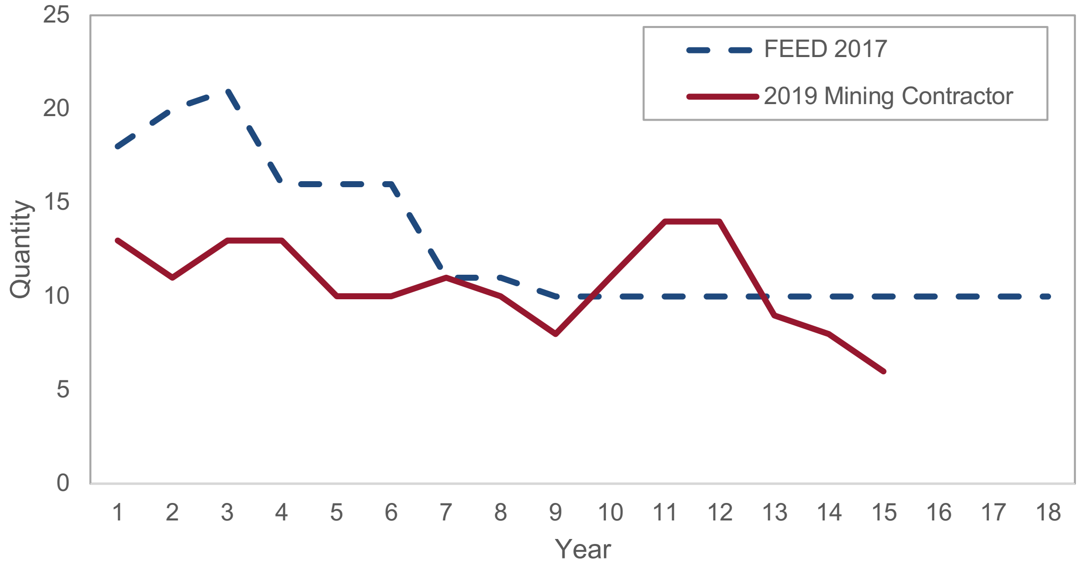


Figure 16-9: Quantity of trucks – Caterpillar 785D (2017 FEED vs 2019 Contractor)

16.7.4 Ancillary work and equipment

Typical ancillary activities will be required to support drilling, loading and hauling activities. These include maintaining floors in the loading and unloading areas, scaling pit walls, construction of temporary ramps and accesses, maintaining haul roads, spreading waste, profiling the external faces of the DDMR and maintenance of short-term ore stockpiles. Ancillary equipment requirements are indicated in Table 16-8.

Table 16-8: Ancillary equipment requirements

Equipment	Make	Model	Quantity	Hours Each	Hours Total	Useful Life
Bulldozer	Caterpillar	D9T	2	64,054	128,107	60,000
Grader	Caterpillar	16M3	1	42,542	42,542	36,000
Wheeldozer	Caterpillar	834K	1	25,810	25,810	36,000
Compactor	Bomag	BW211-D40	1	8,900	8,900	15,000
Water Truck	Volvo FMX	5,000 gl	2	44,500	89,000	15,000
Backhoe	Caterpillar	420F	1	21,360	21,360	15,000
Excavator	Caterpillar	374FL	1	36,312	36,312	18,000
Ligthing Tower	Terex	RL 4000	8	53,400	427,200	
Fuel Truck	Volvo FMX	5,000 gl	2	35,600	71,200	15,000
Lube Truck	Volvo FMX		2	53,400	53,400	15,000

16.7.5 Dewatering

Groundwater inflow to the pit is anticipated to be on the order of 2,000 to 3,000 m^3/day. Inflowing groundwater and water from direct precipitation in the pits will be collected in sumps. Sump locations will change as the pits develop, moving to the deepest portions of the pits as they develop. Water from the pits will be pumped to the contact- water cell of the water supply pond for subsequent consumption during operations.

Pumps will be trailer-mounted, diesel units, and will pump through HDPE pipes to the process plant contact water pond. A range of pump sizes will be required, including some small portable pump and “layflat” hose for transferring water accumulated in surface puddles to a central sump.

16.7.6 Mine supervision

Mine supervision includes the following areas, under BCM responsibility:

- Mining Operations
- Mine Planning
- Geology

Table 16-9 identifies the quantity of mine supervision personnel needed and compares the numbers from FEED 2017 for the Owner Operator and Mining Contractor scenario to the current estimates for 2019 Mining Contractor scenario.

Table 16-9: Comparison of mining supervision requirements (phase 2 pre-strip and operations) mining contractor (2017) vs 2019 mining contractor

Position	Area	FEED (Mining Contractor)	2019 (Mining Contractor)
Mine Manager	Mine Operations	1	1
Mine Superintendent	Mine Operations	1	1
Mine Foreman	Mine Operations	-	-
Foreman	Mine Operations	-	-
Engineer	Mine Operations	1	1
D&B Engineer	Mine Operations	-	-
Heavy Equipment Trainer	Mine Operations	-	-
Heavy Equipment Evaluator	Mine Operations	-	-
Heavy Equipment Training Assistant	Mine Operations	-	-
Chief Mine Engineer	Planning	1	-
Geotechnical Engineer	Planning	1	1
Surveyor	Planning	2	4
Mine Planning Engineer ST	Planning	2	2
Mine Planning Engineer LT	Planning	1	2
Chief Geologist	Geology	1	0
Geologist	Geology	-	2
Geologist - Field	Geology	2	2
Geologist - Grade Control Modeling	Geology	2	2
Senior Geologist - Resources	Geology	1	0
Database Administrator	Geology	1	0
Database Assistant	Geology	2	0
Maintenance Superintendent	Maintenance	-	0
Maintenance Chief	Maintenance	-	0

Position	Area	FEED (Mining Contractor)	2019 (Mining Contractor)
Maintenance Foreman	Maintenance	-	0
Maintenance Shift Supervisor	Maintenance	-	0
Supervisor - Tires	Maintenance	-	0
Chief Maintenance Planner	Maintenance	-	0
Supervisor - Monitoring	Maintenance	-	0
Maintenance Analyst	Maintenance	-	0
Maintenance Programmer	Maintenance	-	0
Tailings Transport Supervisor	Tailings Transport	3	0
Subtotal - Owner		22	18
Project Manager	Mining Contractor	1	1
Mine Superintendent	Mining Contractor	1	1
Shift Supervisor	Mining Contractor	3	3
Shift Supervisor - Assistant	Mining Contractor	3	-
D&B Engineer	Mining Contractor	1	3
Heavy Equipment Trainer	Mining Contractor	1	6
Heavy Equipment Evaluator	Mining Contractor	1	-
Heavy Equipment Training Assistant	Mining Contractor	1	-
Chief - Technical Office	Mining Contractor	1	1
Contract Administrator	Mining Contractor	1	-
Assistant	Mining Contractor	2	1
Engineer - Planning & Costs	Mining Contractor	1	2
Surveyor	Mining Contractor	1	2
SSOMA: Jefe de SSOMA	Mining Contractor	1	1
SSOMA: Ing SSOMA	Mining Contractor	3	7
Project Administrator	Mining Contractor	1	1
Chief - Logistics and Warehouse	Mining Contractor	1	1
Warehouse Assistant	Mining Contractor	1	6
Social Assistant	Mining Contractor	3	2
Payroll Assistant	Mining Contractor	2	2
Chief - Human Resources	Mining Contractor	1	1
Community Relations Assistant	Mining Contractor	1	1
General Services Coordinator	Mining Contractor	1	1
IT Coordinator	Mining Contractor	1	-
Driver	Mining Contractor	3	6
Maintenance Superintendent	Mining Contractor	1	1
Maintenance Chief: Equipment	Mining Contractor	1	-
Maintenance Shift Supervisor	Mining Contractor	4	3
Supervisor - Tires	Mining Contractor	1	1
Maintenance Chief: Planning and Reliability	Mining Contractor	1	-

Position	Area	FEED (Mining Contractor)	2019 (Mining Contractor)
Maintenance Reliability	Mining Contractor	3	-
Maintenance Planner	Mining Contractor	2	3
Maintenance Programmer	Mining Contractor	2	2
Tailings Transport Supervisor	Tailings Transport	-	3
Subtotal - Mining Contractor	All	52	59
Total - Owner + Mining Contractor		**74**	**77**

16.7.7 Principal and ancillary equipment

Table 16-10 list of principal and ancillary equipment for the construction (Pre-strip Phase 2) and operations phases

Note: All of Year -1 is Construction. Year 1 onwards is operations.

Table 16-10: List of principal and ancillary equipment - mining contractor

Start date		**Jun-23**	**Jan-24**	**Jan-25**	**Jan-26**	**Jan-27**	**Jan-28**	**Jan-29**	**Jan-30**	**Jan-31**	**Jan-32**	**Jan-33**	**Jan-34**	**Jan-35**	**Jan-36**	**Jan-37**	**Jan-38**
End date		**Dec-23**	**Dec-24**	**Dec-25**	**Dec-26**	**Dec-27**	**Dec-28**	**Dec-29**	**Dec-30**	**Dec-31**	**Dec-32**	**Dec-33**	**Dec-34**	**Dec-35**	**Dec-36**	**Dec-37**	**Mar-38**
Months		**7**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**3**
Period		**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Year 12**	**Year 13**	**Year 14**	**Year 15**
Equipment Make	**Model**																
Drill - Atlas Copco	DM45	3	3	3	3	3	2	2	2	2	2	3	3	3	2	2	2
Drill - Atlas Copco	ROC L8	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shovel - Caterpillar	6040FS	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Front-end Loader – Caterpillar	994F	0	1	0	1	1	1	0	1	1	0	1	1	0	0	0	1
Dump Truck - Caterpillar	785D	13	13	11	13	13	10	10	11	10	8	11	14	14	9	8	6
Bulldozer – Caterpillar	D9T	3	2	2	3	2	2	1	2	2	2	2	3	2	1	1	1
Grader - Caterpillar	16M	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Wheel Dozer - Caterpillar	834K	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Compactor - Bomag	BW211-D40	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water Truck - Volvo	FMX	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Backhoe - Caterpillar	420F	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel Truck - Volvo	FMX	2	2	2	2	2	2	1	2	2	1	2	2	2	1	1	1
Lube Truck - Volvo	FMX	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Explosives Mixing Truck - Kenworth	T-800	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Excavator - Caterpillar	374DL	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Ligthing Tower - Terex	RL4000	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Total		42	42	39	43	42	38	35	39	38	34	40	44	42	34	32	31

16.7.8 Mine operations personnel

For Mine operations personnel, Bear Creek Mining, would require a total of 18 staff for the four areas (mine operations, geology, planning and tailings transport), with the mining contractor the responsible for providing the necessary personnel for mine operations (loading, hauling and dumping), drilling, blasting, equipment maintenance facilities and services. Table 16-11 shows the mine operations personnel requirements and the comparison between the 2017 FEED and the current mining contractor.

Table 16-11: Mine operations personnel requirements

Area	Quantity (Max)	
	FEED (Mining Contractor)	2019 (Mining Contractor)
Mine operations	3	3
Geology	7	6
Planning	9	9
Maintenance	0	0
Tailing transport	3	0
Subtotal - owner	22	18
Mining contractor	52	59
Total	74	77

16.7.9 Filtered tailings transport

Transport of filtered tailings from the filtered tailings stockpile to the main mine waste and filtered tailings deposit (DDMR) will be undertaken by a contractor.

The contractor proposes to transport the filtered tailings from the stockpile to the deposit (TSF) using one Caterpillar 994F front-end loader to load six CAT 785 dump trucks for the first 2 years of filter tailings production.

From year 3 until the end of the filter tailings production time, the quantity of dump trucks will decrease from 6 in the first 2 years to 3 in the next years as haulage cycle times decrease with the construction of the conveying system. The conveying system will shift the location of the stockpile from the main process plant area up to the front face of the TSF. The conveying system will be extended to maintain the filtered tailings stockpile at the front face of the TSF as the height increases.

Two Caterpillar D6T bulldozers will be used to spread the tailings in layers approximately 1 m thick. As the same contractor responsible for mining will be transporting the filtered tailings, common resources for management of the TSF will be employed.

16.7.10 Fuel

For the Phase 1 pre-strip (construction of haul roads), diesel fuel will be supplied by a major Peruvian fuel company, who will construct temporary fuel storage facilities (capacity 40,000 gallons) and provide a refueling service in the field using fuel trucks (two on day shift and 1 on night shift).

For the Phase 2 pre-strip and the rest of the mine life, a fuel facility will be constructed near the main entrance gate. It will have 100,000 gallons of storage capacity (equivalent to 9 days of storage during the peak demand) and be equipped with two fuel bowsers for light equipment. The mining contractor will be responsible for refueling and lubrication of all mining equipment in the field, using two mixed fuel/lubrication trucks operating 24 h/day. Equipment from other areas will refuel at the fuel station also.

16.7.11 Explosives

For the Phase 1 pre-strip (construction of haul roads), Phase 2 pre-strip, and production years, a Peruvian explosives supplier will be responsible for constructing explosives storage facilities, supplying explosives, and providing a "down-hole" blasting service. Facilities to be provided are:

- Storage magazine for detonator
- Storage magazine for high explosives
- Roofed area for ammonium nitrate storage and loading silo
- Silos for emulsion storage

The storage magazine for detonators will be approximately 18 m^2 and will have enough capacity for 500 kg. The storage magazine for high explosives will be approximately 14 m^2 and have enough capacity for 1,800 kg. Boosters and detonator cord will be stored in this magazine. Both magazines will be separated by an earth wall, with a minimum distance of 19 m between the magazines.

The bulk emulsion storage silos will have storage for 160 tonnes distributed in two silos, which will be filled by the explosive's supplier´s trucks which will bring the emulsion to the mine. These silos will be used to fill the mobile mixing unit trucks for use in the mine.

The ammonium nitrate storage shed will be a closed sided, steel-clad shed with an area of 375 m^2. It will have capacity for 300 tonnes of material in big bags of 1.00 or 1.25 tonnes.

All storage facilities will be surrounded by security fences and provided with external security lighting and lightning protection as well as manned secuiry per Peruvian regulations.

16.7.12 Truck shop and facilities

The truck shop will have an area of approximately 17,500 m^2 and will be located in the northwest part of the industrial area knows as the Mine Infrastructure Area (MIA). The MIA will have access from the mine via a haul road connecting near the primary crusher (ROM Pad), and with access for light vehicles via the processing plant. The truck shop will include:

- mine operations and maintenance offices, and meeting room
- toilets and changing rooms
- satellite dining room
- warehouse for consumables and spare parts
- area for lubricants and additives
- area for warehouse
- area for hazardous materials and gases
- a total of 4 maintenance bays: (2) assigned to preventative maintenance, (1) assigned to corrective maintenance, (1) multipurpose, additionally 1 welding bay.
- parking area for light vehicle and operations equipment.

A tire changing area, will be located next to the truck shop and includes a concrete slab for jacking trucks up and changing tires. In addition, there are two areas of approximately 500 m^2 for storage of tires.

A truck and equipment wash area of 800 m^2 will be in front of the truck shop. It includes a concrete slab with steel protection rails, enclosures, and elevated platforms equipped with water cannons and hose reels.

16.7.13 Fleet management system

As requested by Bear Creek Mining, a fleet management system has been included. The mining contractor will be responsible for providing all hardware and software required and for operating the system for the commencement of the phase 2 pre-strip (start of operations with mining equipment) and to the end of the mine life.

16.8 Work Schedule

Mining will be carried out using 8-m high benches, consistent with the NI43-101 report (M3 Engineering, 2015) and will intersect some historic underground workings. Mine operations will work 2 x 12-hour shifts per day, 365 days per year. In total, three crews of operations personnel will be required, one working in day shift, one working in night shift, and one on rest days. Crews will work 14 days shift (days and nights) and 7 days off for the roster (14 by 7). Maximum equipment working hours per day are estimated to be 20.50 resulting in a maximum possible utilization of 85.4%. Time lost on average per day is shown in Table 16-12.

Table 16-12: Calculation of utilization

Description	Unit	Time
Hours per Day	h/day	24
Shift Change	h/day	1
Meals	h/day	2
Blasting	h/day	0.50
Training	h/day	0
Weather	h/day	0
Total Non-Utilizable Time	h/day	3.5
Total Utilizable Time	h/day	20.5
Utilization (Max Possible)	%	85.4%

Personnel requirements for the construction (Phase 1 pre-strip and construction of haul roads) are shown in Table 16-13.

Table 16-13: Pre-strip phase 1 and construction of haul roads: personnel

Personnel	Aug-22	Sep-22	Oct-22	Nov-22	Dec-22	Jan-23	Feb-23	Mar-23	Apr-23	May-23
Foreman		1	2	2	10	10	10	9	8	4
Operator		3	3	9	9	9	2	1	1	1
Official		0	0	1	12	12	12	3	1	0
Labourer / Helper / D&B		9	10	26	80	80	80	65	43	21

Personnel	Aug-22	Sep-22	Oct-22	Nov-22	Dec-22	Jan-23	Feb-23	Mar-23	Apr-23	May-23
Operator - Heavy Equipment		2	2	10	27	27	27	30	13	13
Operator - Medium Equipment		1	1	14	38	39	39	16	16	10
Maintenance	24	47	51	49	49	48	49	49	49	40
Subtotal - Direct Labour	24	63	69	111	225	225	219	173	131	89
Staff	8	13	15	15	22	22	20	20	19	14
Assistant / Helper	12	25	28	30	32	33	31	31	28	25
Subtotal - Indirect Labor	20	38	43	45	54	55	51	51	47	39
Total Personnel	44	101	112	156	279	280	270	224	178	128

17 Recovery Methods

17.1 Site Layout Considerations

The Project site is located on steep sloping, high-altitude terrain that has limited flat space. These considerations required particular care in identifying an acceptable site for the facilities.

The process plant layout was developed to reduce footprint and, therefore, capital cost. The layout works well with the constraints imposed by the topography, being the mine to the north, a steep mountain range to the west and a swamp to the east. The plant has been laid out as a narrow facility, with cost-effective design elements, such as, the primary crusher product conveyor utilizing the existing hill to feed the stockpile with a shallow incline.

17.2 Process Description

The Corani process plant is a two-stage sequential flotation concentrator generating separate lead (with silver) and zinc concentrates.

The design treatment rate for the process plant is 27,000 t/d based on operating 365 days per year. The operating basis for design is 70% availability for the crushing circuit, 92% availability for the grinding and flotation circuits, and 82.8% availability for concentrate and tailings filtration. The process plant includes the following unit processes and facilities:

- primary crushing
- stockpile and reclaim
- grinding and classification
- flotation and regrind
- concentrate thickening and filtration
- tailings thickening
- tailings filtration and stockpile
- reagents
- air and water services.

Figure 17-1 shows the overall process flow diagram for the Corani process plant. Ore will be crushed using a gyratory crusher. Crushed ore will be discharged to a single conical coarse ore stockpile via conveyors. The grinding circuit will consist of a semi-autogenous grinding (SAG) mill and ball mill operating in closed circuit with a cyclone cluster (SAB).

Flotation will consist of a two-stage sequential circuit with lead flotation generating a lead concentrate enriched with silver followed by zinc flotation generating a zinc concentrate with a lower silver content. The primary cyclone overflow will feed lead rougher flotation. The lead rougher concentrate will feed a lead regrind circuit followed by a cleaner scalper (Jameson cell) and three stages of cleaning (including two stages of column flotation) to produce a final lead concentrate. The lead rougher tailings and lead cleaner scavenger tails feed the zinc rougher cells. The zinc rougher concentrate will feed the zinc regrind circuit followed by a cleaner scalper (Jameson cell) and three stages of cleaning (including two stages of column flotation) to produce a final zinc concentrate. The zinc rougher tails and zinc cleaner scavenger tails will be combined to form the final tailings from the process. Two high compression thickeners will dewater the tailings prior to filtration in conventional pressure filters. Filtered tailings will be transferred to a tailings stacker via conveyor belts and stockpiled before being transported to DDMR. The final

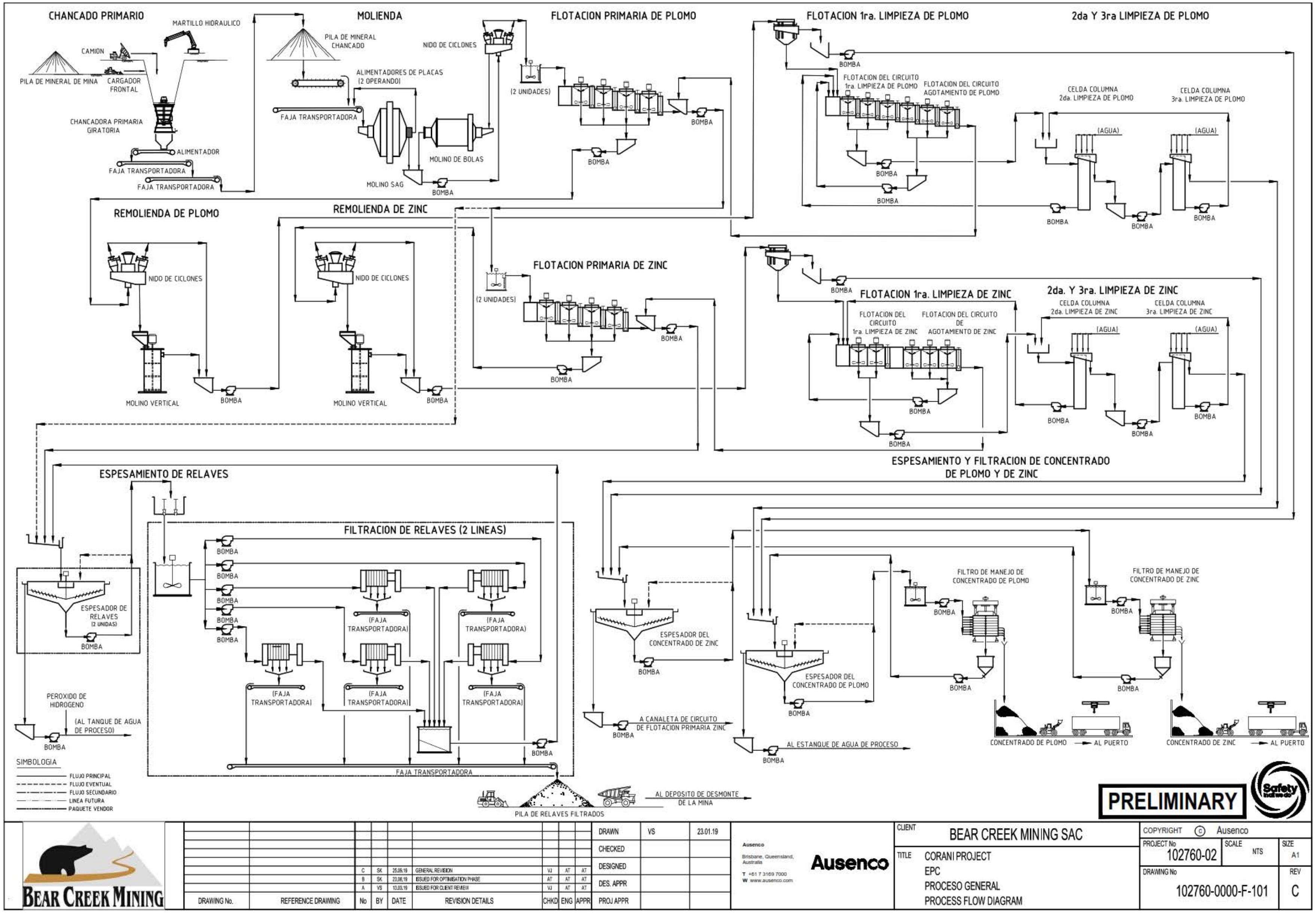


Figure 17-1: Overall process flow diagram for Corani Project

lead and zinc concentrates will be thickened using high rate thickeners prior to filtration in conventional pressure filters and transport to the port.

17.2.1 Primary crushing

Run-of-mine (ROM) ore (F_{100} of 1000 mm) will be delivered using 135-140 t trucks (CAT 785) and fed into the crushing circuit via a single dump hopper with 240 t live capacity. The dump pocket is designed to allow truck dumping from one side. Using a single truck dump design reduces the ROM area and associated earthworks, improves maintenance accessibility and reduces operating complexity.

Ore will be crushed using a 50’ x 65’ gyratory crusher, with a closed side setting (CSS) of 120 mm, and discharged via a belt feeder onto the crushed ore sacrificial conveyor (38 m) and transferred onto the overland conveyor (650 m). The sacrificial conveyor is equipped with a metal detector to detect tramp metal. The conveyor will stop if tramp metal is detected and an operator will use an overhead hoist to remove the tramp metal. The overland conveyor will be fitted with a weightometer to monitor crusher production/stockpile feed rates.

Auxiliary crusher equipment includes lubrication, hydraulic and cooling systems, dust seal blower and grease barrels and dosing equipment as well as the rock breaker and its hydraulic pack.

17.2.2 Coarse ore stockpile and reclaim

The overland conveyor will feed the crushed ore to a single conical coarse ore stockpile. The feed to the stockpile is launched from a ridge to reduce concrete, steel and bulk earthworks (Figure 17-2). The stockpile will have a total storage of approximately 49 000 t. The live capacity was estimated to be only 7% of the total stockpile capacity, with the remaining capacity able to be recovered with earthmoving equipment. The Jenike and Johanson flow property test results for crushed ore (received in May 2019) indicated a strong tendency for the ore to form stable ‘ratholes’ that reduces the likely live capacity. This will be managed by use of a dozer or excavator on the stockpile when ore is not be crushed to the stockpile. The stockpile will provide surge capacity between the primary crusher and the process plant for ROM ore tonnage fluctuations and maintenance activities.

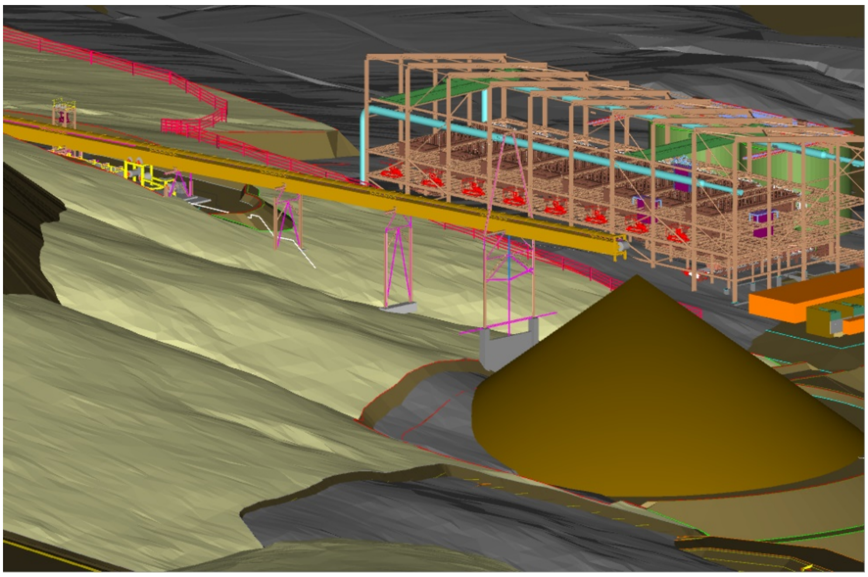


Figure 17-2: Screenshot of conveyor and stockpile (foreground) from 3D model

Coarse ore will be reclaimed from the stockpile by two variable-speed drive apron feeders. Each feeder will have capacity to provide 100% of the full base case tonnage rate to the SAG mill, but will generally operate at 50% capacity for even draw down of the stockpile.

The apron feeders will discharge onto the SAG mill feed conveyor. Coarse ore, combined with pebble recycle, will be transported to the SAG mill feed chute.

The SAG mill feed conveyor will be fitted with a weightometer, prior to pebble recycle conveyor discharge, to measure new feed rate for process control and metallurgical accounting.

17.2.3 Grinding and classification circuit description

The grinding circuit was designed using ore characterization tests that determined Bond crushing, rod and ball mill work indices of 9.0, 11.8 and 14.7 kWh/t, respectively, and a JK SMC test (Axb) result of 80 (Table 17-4). The grinding circuit will consist of a SAG mill, and a ball mill operating in closed circuit with a cyclone cluster. The product from the grinding circuit (cyclone overflow) will have a nominal density of 32% w/w solids. The P_{80} will vary as a function of ore hardness, say between 70 µm and 120 µm based on the life of mine (LOM) ore hardness data available from test work (Figure 17-3).

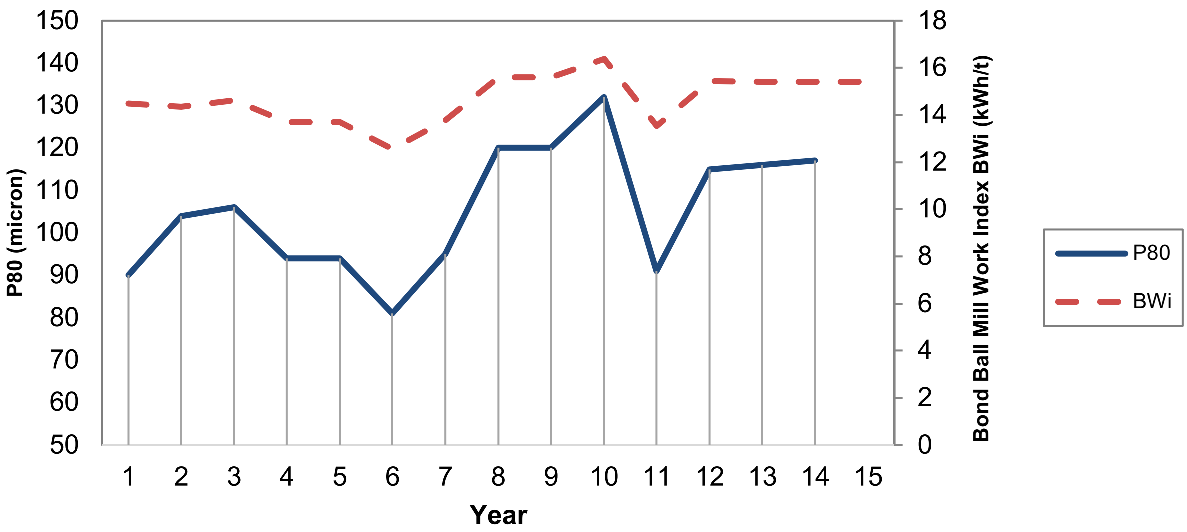


Figure 17-3: Cyclone overflow P_{80} by year for a throughput of 27,000 t/d.

The SAG mill feed conveyor will transfer reclaimed ore at a design rate of 1223 t/h (dry) to the SAG mill feed chute where it will be combined with dilution water, grinding media and lime slurry. The 7 MW SAG mill (7.92 m diameter x 7.32 m EGL) is provided with a slip energy recovery (SER) hyper-synchronous drive through a single pinion.

The SAG mill is fitted with a trommel screen equipped with water sprays to separate slurry and coarse unbroken rock or pebbles. SAG mill trommel undersize will gravitate to the cyclone feed hopper where it will be combined with the ball mill trommel undersize. The trommel oversize (pebbles) will be transported by a pebble recycle conveyor which transfers the pebbles to the

SAG mill feed conveyor. Pebbles can be discharged into a bypass bunker by a diverter gate as required.

Slurry from the cyclone feed hopper will be pumped to the primary cyclone cluster (24 hydrocyclones each 500 mm diameter). The cyclone underflow stream will report to the ball mill feed chute. The ball mill will operate in closed circuit with the primary cyclone cluster. The 14 MW ball mill (7.32 m diameter x 12.8 m EGL) will be fitted with a fixed speed twin pinion drive arrangement. The ball mill product will overflow onto a trommel screen, oversized scats will feed an adjacent ball mill scats bunker and undersize slurry will gravitate into the cyclone feed hopper.

SAG and ball mill grinding media will be transferred from their individual storage bunkers into kibbles via a ball loading chute. A ball loading hoist will be used to lift the kibble and discharge the grinding media into each mill's ball loading hopper. A ball feeder will transfer media at a controlled rate to the mill feed chute.

17.2.4 Lead flotation

The lead flotation circuit is fed from the primary cyclone overflow via a metallurgical sampler. The lead rougher flotation train will consist of two rougher conditioning tanks (Ø6.5 m × 7.0 m H and ~4.5 minutes residence time each) and four 500 m^3 forced air mechanical flotation tank cells. Zinc cyanide and sodium sulphite will be added in the first conditioning tank to depress sphalerite and pyrite. Collector (sodium isopropyl xanthate—SIPX), promoter (AP404) and frother (methyl isobutyl carbinol—MIBC) are added in the second conditioning tank. The conditioned slurry gravitates to the lead rougher flotation cells.

The lead rougher flotation cells will produce a concentrate that requires further liberation and upgrading. The concentrate slurry will be pumped to the lead regrind circuit. The lead regrind circuit will consist of a cyclone cluster (eight hydrocyclones each 250 mm diameter) operating in open circuit with a vertical regrind mill (2,000 kW HIG mill). Cyclone underflow will feed the regrind mill and the overflow will bypass the mill, combining with the regrind product, before being pumped to the lead cleaner scalper feed hopper.

A lead cleaner scalper flotation cell will be employed following regrind to produce a high-grade concentrate and reduce the load on the remaining cleaner circuit. The lead cleaner scalper will consist of a single B5000/16 Jameson cell with an external recycle mechanism. Regrind circuit products will be combined with the cleaner scalper cell tailings recycle stream and pumped into the cell via a set of 16 downcomers. The cleaner scalper concentrate will be pumped to the lead final concentrate sampler, while the tailings will gravitate to the first cleaner stage.

Lead cleaner 1 will consist of three 70 m^3 forced air mechanical tank cells. Cleaner scalper tailings, cleaner scavenger concentrate and cleaner 2 tailings will be combined in the lead cleaner 1 feed box. Concentrate recovered from lead cleaner 1 will be pumped to lead cleaner 2 for further upgrading, with the tailings gravitating to the lead cleaner scavenger flotation cells.

Lead cleaner scavenger will consist of a further three 70 m^3 forced air mechanical tank cells. The lead cleaner scavenger cells will recover a low-grade concentrate that will be pumped back to lead cleaner 1 feed. Cleaner scavenger tailings will be combined with the rougher tailings in the lead rougher flotation tailings hopper and pumped to the zinc flotation circuit.

Lead cleaner 2 and cleaner 3 will both be column cells. Cleaner 1 concentrate and cleaner 3 tailings will be combined in the lead cleaner 2 feed box. Concentrate from lead cleaner 2 will be pumped to lead cleaner 3 for further upgrading, whilst cleaner 2 tailings will be pumped into lead cleaner 1 and cleaner 3 tailings will be pumped into lead cleaner 2.

Concentrate from the lead cleaner scalper and lead cleaner 3 will be combined and gravitate to the lead concentrate thickening circuit via a metallurgical sampler.

Lead flotation and regrind equipment selections are shown in Table 17-1.

Table 17-1: Summary of major equipment in the lead flotation and regrind circuit

Duty	Equipment	Specification
Pb rougher	Tank cells	4 x 500 m³
Pb regrind mill	HIG mill	2,000 kW
Pb cleaner scalper	Jameson cell	B5000/16 (Ø5.0 m)
Pb cleaner 1	Tank cells	3 x 70 m³
Pb cleaner scavenger	Tank cells	3 x 70 m³
Pb cleaner 2	Column cell	1 x Ø4.5 m × 11.0 m H
Pb cleaner 3	Column cell	1 x Ø3.0 m × 12.0 m H

When required, lead flotation circuit tailings can be directed to the tailings thickener via the zinc rougher tailings hopper, bypassing the zinc flotation circuit.

17.2.5 Zinc flotation

Zinc flotation feed, from the lead rougher tailings hopper, will be pumped to the zinc rougher flotation train. The zinc rougher flotation train will consist of two rougher conditioning tanks (Ø6.5 m × 7.0 m H and ~4.1 minutes residence time each) and four 500 m^3 forced air mechanical flotation tank cells. Copper sulphate will be added in the first conditioning tank to activate sphalerite. Lime, collector (SIPX) and frother (MIBC) will be added in the second conditioning tank. The conditioned slurry gravitates to the zinc rougher flotation cells.

The zinc rougher flotation cells will produce a concentrate that requires further liberation and upgrading. The concentrate slurry will be pumped to the zinc regrind circuit. The zinc regrind circuit will consist of a cyclone cluster (eight hydrocyclones each 250 mm diameter) operating in open circuit with a vertical regrind mill (3,000 kW HIG mill). The zinc regrind mill is larger than the lead regrind mill due to the higher rougher mass recovery in the zinc circuit and higher specific energy requirement to achieve the target grind size. Cyclone underflow will feed the regrind mill and the overflow will bypass the mill, combining with the regrind product, before being pumped to the zinc cleaner scalper feed hopper.

In a similar way to the lead circuit, the zinc cleaner circuit will begin with a cleaner scalper flotation cell to produce a high-grade concentrate and reduce the load on the remaining cleaner circuit. The zinc cleaner scalper will consist of a single B5000/16 Jameson cell with an external recycle mechanism. Regrind circuit products will be combined with the cleaner scalper cell tailings recycle stream and pumped into the cell via a set of 16 downcomers. The cleaner scalper concentrate will be pumped to the zinc final concentrate sampler, while the tailings will gravitate to the first cleaner stage.

Zinc cleaner 1 will consist of two 70 m^3 forced air mechanical tank cells. Cleaner scalper tailings, cleaner scavenger concentrate and cleaner 2 tailings will be combined in the zinc cleaner 1 feed box. Concentrate recovered from zinc cleaner 1 will be pumped to zinc cleaner 2 for further upgrading, with the tailings gravitating to the zinc cleaner scavenger flotation cells.

Zinc cleaner scavenger will consist of three 70 m^3 forced air mechanical tank cells. The zinc cleaner scavenger cells will recover a low-grade concentrate that will be pumped back to zinc cleaner 1 feed. Cleaner scavenger tailings will be combined with the rougher tailings in the zinc rougher flotation tailings hopper and pumped to the tailings thickening circuit via a metallurgical sampler.

Zinc cleaner 2 and cleaner 3 will both be column cells. Cleaner 1 concentrate and cleaner 3 tailings will be combined in the zinc cleaner 2 feed box. Concentrate from zinc cleaner 2 will be pumped to zinc cleaner 3 for further upgrading, while cleaner 2 tailings will be pumped into zinc cleaner 1 and cleaner 3 tailings will be pumped into zinc cleaner 2.

Concentrate from the zinc cleaner scalper and zinc cleaner 3 will be combined and gravitate to the zinc concentrate thickening circuit via a metallurgical sampler.

Zinc flotation and regrind equipment selections are shown in Table 17-2.

Table 17-2: Summary of major equipment in the zinc flotation and regrind circuit

Duty	Equipment	Specification
Zn rougher	Tank cell	4 x 500 m³
Zn regrind mill	HIG mill	3,000 kW
Zn cleaner scalper	Jameson cell	B5000/16 (Ø5.0 m)
Zn cleaner 1	Tank cell	2 x 70 m³
Zn cleaner scavenger	Tank cell	3 x 70 m³
Zn cleaner 2	Column	1 x Ø3.5 m × 12.0 m H
Zn cleaner 3	Column	1 x Ø3.0 m × 9.0 m H

17.2.6 Sampling and on-stream analysis

The on-stream analyser (OSA) will be used to monitor metal contents and solids concentrations in the flotation feed stream, major concentrate streams, rougher and cleaner scavenger tailings streams, and the final tailings stream to allow operators to optimise reagent additions and flotation performance. The particle size indicator (PSI) will also be used to provide on-line size analysis for flotation feed, lead regrind circuit discharge and zinc regrind circuit discharge in order to monitor grinding and regrinding performance.

In addition to the four metallurgical samplers outlined previously (flotation feed, flotation tailings, lead concentrate and zinc concentrate) the following slurry samplers will provide samples for monitoring and control purposes:

- lead rougher concentrate – pressure sampler
- lead regrind circuit discharge (sizing only) – pressure sampler
- lead cleaner 1 concentrate – pressure sampler
- lead rougher tailings – peristaltic pump
- lead cleaner scavenger tailings – peristaltic pump
- zinc rougher concentrate – pressure sampler
- zinc regrind circuit discharge (sizing only) – pressure sampler
- zinc cleaner 1 concentrate – pressure sampler
- zinc rougher tailings – peristaltic pump
- zinc cleaner scavenger tailings – peristaltic pump

Samples will be collected using metallurgical samplers, pressure samplers or peristaltic pumps (tails pumped from within the flotation cell) and transferred to the OSA using either gravity lines or peristaltic pumps to preserve the quality of the sample. The samples will then be sorted by two multiplexers (lead and zinc) prior to the OSA. Shift composite samples will also be taken by the multiplexers and transferred into buckets.

Once the samples have been analysed, they will be discarded through a 4-way de-multiplexer. Sample return will consist of low grade, medium grade and concentrate return pumps for both lead and zinc samples with returns directed to various points in the circuit.

17.2.7 Lead concentrate thickening and filtration

Lead concentrate from the lead flotation circuit will be dewatered in a 13 m diameter high-rate thickener. The concentrate thickener overflow will report to the lead concentrate thickener overflow tank and on to the process water tank. Lead concentrate solids will settle for collection at the underflow cone at a density of 65% w/w solids. The thickener underflow stream will be pumped to a static screen prior to an agitated lead filter feed tank.

The lead filter feed tank will provide 12 hours surge capacity at design production rates, allowing filter maintenance to be conducted without affecting mill throughput. Filter feed will be pumped to the lead concentrate filter (Outotec Larox PF 15 series automated horizontal plate filter including 22 plates of 1,050 mm x 2,400 mm each) to produce a filter cake of 8% w/w moisture. The lead filter cake will be discharged by gravity to the lead concentrate filter bunker.

Periodically the lead concentrate will be loaded by a 30 t front-end loader (CAT980 or similar) onto a retractable conveyor system which will fill concentrate containers for transport from the mine site to the storage terminal at the port facility.

Fresh water will be used for cloth washing, and process water and plant air will be used to flush the filter manifold. Filtrate, cloth wash and manifold flushing water will be returned from the filtrate tank to the lead concentrate thickener via the thickener feed box.

A semi-automatic truck washing system and weighbridge will be provided at the site entrance gate. These facilities will be used for both lead and zinc concentrate dispatch. The water used for truck washing will be recovered through a sump pump and directed back to the plant.

17.2.8 Zinc concentrate thickening and filtration

Zinc concentrate from the zinc flotation circuit will be dewatered in a 12 m diameter high-rate thickener. The concentrate thickener overflow will report to the zinc concentrate thickener overflow tank and will be recirculated back to the zinc flotation circuit, primarily for use in flotation cell launders and froth wash water. Zinc concentrate solids will settle for collection at the underflow cone at a density of 65% w/w solids. The thickener underflow stream will be pumped to a static screen prior to an agitated zinc filter feed tank.

The zinc filter feed tank will provide 12 hours surge capacity at design production rates, allowing filter maintenance to be conducted without affecting mill throughput. Filter feed will be pumped to the zinc concentrate filter (Outotec Larox PF 15 series automated horizontal plate filter including 16 plates of 1,050 mm x 2,400 mm each) to produce a filter cake of 8% w/w moisture. The zinc filter cake will be discharged by gravity to the zinc concentrate filter bunker.

Periodically the zinc concentrate will be loaded by a 30 t front-end loader (CAT980 or similar) onto a retractable conveyor system which will fill concentrate containers for transport from the mine site to the storage terminal at the port facility. The ability to direct load into an open top truck will also be included for zinc only.

Fresh water will be used for cloth washing, and process water and plant air will be used to flush the filter manifold. Filtrate, cloth wash and manifold flushing water will be returned from the filtrate tank to the zinc concentrate thickener via the thickener feed box.

17.2.9 Tailings plant thickening and filtering

Tailings from the zinc flotation circuit will be dewatered in two 42 m diameter high-compression thickeners. The tailings thickener supernatant will overflow to the process water tank. Tailings solids will settle for collection at the underflow cone at a density of 52 – 64% w/w solids, depending on the ore type. Thickener underflow will be pumped to the tailings filtration circuit.

Thickened flotation tailings will be filtered by 10 filters arranged in two trains of five filters each. The filter plant is designed to achieve a moisture that enables compaction to 95% of the maximum dry density as per a standard proctor test. This target gives achieved filter cake moistures in the range of 14.5 to 18.4% w/w. Each train will consist of a scalping screen, agitated feed tank and five vertical plate filters, each with a tailings filter conveyor for filter cake discharge.

The filter feed tanks will provide a total of 4.5 hours surge capacity at nominal production rates, allowing filter maintenance to be conducted with minimal impact on the upstream plant. For each train, filter feed will be pumped from the filter feed tank to the tailings filters (Outotec Larox FFP 3512 automated vertical plate filters) to produce filter cake for disposal.

Filtered process water will be used for cloth washing and flushing the filter manifold. Plant air will be used for filter cake air drying and flushing the filter manifold. Filtrate, cloth wash and manifold flushing water will be returned from the filtrate tanks to the tailings thickener via the thickener feed box.

17.2.10 Tailings disposal

The filtered tailings will discharge by gravity onto its own dedicated tailings filter conveyor before being combined onto the tailings conveyor. The tailings conveyor is equipped with a weightometer to monitor the filtered tailings production rate. The conveyor will discharge to a radial telescopic stacker to create the tailings stockpile with 24 hours storage capacity. This stockpile will be reclaimed using front-end loaders (CAT994 or similar) and loaded onto 785 CAT haul trucks to then be transferred to the main mine waste and filtered tailings deposit (DDMR) where it will be encapsulated with coarse non-acid forming waste. A conveyor system has been investigated as an alternative to trucking filtered tailings to the DDMR. The conveyor system would include three fixed conveyors up to the lower bench of the DDMR on the east side with an additional two conveyors required to reach the year 3 pad. A radial stacker on the DDMR will provide 12 hours storage.

17.2.11 Reagents

The reagents area includes the reagent handling, mixing and distribution systems for lime, flocculant, collector (SIPX), frother (MIBC), promoter (AP404), copper sulphate, sodium sulphite and zinc cyanide.

17.3 Process Design Criteria

The main process design criteria used for the project are described below.

17.3.1 Process design criteria

The design feed grades used as the basis for the process mass balance were 1.38% Pb, 0.95% Zn and 83.6 g/t Ag. Table 17-3 summarises the overall recoveries used for the process design including lead, zinc and silver in each concentrate (lead and zinc concentrates).

Table 17-3: Metal recoveries used for process mass balance

Concentrate	Lead Recovery, %	Zinc Recovery, %	Silver Recovery, %
Lead concentrate	74.6	6.2	61.0
Zinc concentrate	5.3	68.7	13.0

These recoveries were used only to generate the mass balance and not for calculation of metal recoveries in the financial model.

Table 17-4: Process design criteria

Area	Description	Units	Value
Ore characteristics	Ore specific gravity	-	2.44
	Ore moisture	% water (w/w)	3
	Bond crushing work index	kWh/t	9.0
	JK SMC test parameters, A x b	-	80
	Bond rod mill work index	kWh/t	11.8
	Bond ball mill work index	kWh/t	14.7
	Bond abrasion index	-	0.22
Primary crushing	Crushing plant capacity, nominal (dry)	t/h	1607
	Crusher feed size F_{100}	mm	1000
	Crusher feed size F_{80}	mm	220
	Closed Side Setting, CSS	mm	120
Stockpile and reclaim	Capacity, total	t	49 000
	Total residence time, nominal	d	1.7
Grinding and classification	Grinding circuit capacity, design (dry)	t/d	27 000
	Grinding circuit capacity, design (dry)	t/h	1223
	Grinding circuit configuration	-	SAB
	Grinding circuit product size, P_{80}	µm	70 - 120
	SAG mill motor power, installed	kW	7000
	SAG mill diameter, inside shell	m	7.92
	SAG mill effective grinding length, EGL	m	7.32
	SAG mill speed, % of critical, typical	%	75
	SAG mill ball charge, nominal	%	12
	SAG mill total charge, nominal	%	25
	Ball mill motor power, installed	kW	14 000
	Ball mill diameter, inside shell	m	7.32
	Ball mill effective grinding length, EGL	m	12.8
	Ball mill speed, % of critical	%	78
	Ball mill ball charge, nominal	%	30
	Circulating load, maximum for design	%	350
	Cyclone overflow density	% solids (w/w)	32
Lead flotation and regrind	Rougher residence time, design	min	30
	Rougher froth carrying capacity, design	$t/m^2/h$	1.5
	Cleaner 1 residence time, design	min	12.5
	Cleaner 1 froth carrying capacity, design	$t/m^2/h$	2.0

Area	Description	Units	Value
	Cleaner scavenger residence time, design	min	15
	Cleaner scavenger froth carrying capacity, design	t/m^2/h	1.0
	Cleaner 2 froth carrying capacity, design	t/m^2/h	2.25
	Cleaner 3 froth carrying capacity, design	t/m^2/h	2.25
	Regrind circuit product, P_{80}	µm	25
	Concentrate regrind specific energy, design	kWh/t	12.4
Zinc flotation and regrind	Rougher residence time, design	min	26.4
	Rougher froth carrying capacity, design	t/m^2/h	1.5
	Cleaner 1 residence time, design	min	12
	Cleaner 1 froth carrying capacity, design	t/m^2/h	2.0
	Cleaner scavenger residence time, design	min	10
	Cleaner scavenger froth carrying capacity, design	t/m^2/h	0.8
	Cleaner 2 froth carrying capacity, design	t/m^2/h	2.25
	Cleaner 3 froth carrying capacity, design	t/m^2/h	2.25
	Regrind circuit product, P_{80}	µm	30
	Concentrate regrind specific energy, design	kWh/t	15.2
Lead concentrate thickening	Unit area thickening rate, design	t/m^2/h	0.20
	Underflow pulp density	% solids (w/w)	65
	Thickener diameter, design	m	13
Lead concentrate filtration	Filter cake moisture	% (w/w)	8
	Specific filtration rate	t/m^2/h	0.566
Zinc concentrate thickening	Unit area thickening rate, design	t/m^2/h	0.18
	Underflow pulp density	% solids (w/w)	65
	Thickener diameter, design	m	12
Zinc concentrate filtration	Filter cake moisture	% (w/w)	8
	Specific filtration rate	t/m^2/h	0.456
Tailings thickening	Unit area thickening rate, design	t/m^2/h	0.50
	Number of units	-	2
	Thickener diameter, design	m	42
	Underflow pulp density, nominal	% solids (w/w)	51 - 58
Tailings filtration	Number of units	-	10
	Average filter cake moisture	% (w/w)	17
	Specific filtration rate	t/m^2/h	0.145
Reagents	Lime	g/t	3500
	Sodium isopropyl xanthate (SIPX)	g/t	40
	A404	g/t	15
	Sodium hydroxide	g/t	10
	Zinc sulphate	g/t	620
	Sodium cyanide	g/t	210
	Copper sulphate	g/t	290
	Methyl isobutyl carbinol (MIBC)	g/t	50
	Sodium sulphite	g/t	505

Area	Description	Units	Value
	Flocculant - concentrate	g/t (thickener feed)	20
	Flocculant - tailings	g/t (thickener feed)	20
Consumables	SAG mill grinding media	kg/kWh	0.04
	Ball mill grinding media	kg/kWh	0.055
	Regrind mill grinding media	kg/kWh	0.01

17.4 Water Requirement

17.4.1 Fresh water consumption

A water balance for the process plant was developed based on the outputs of the overall process mass balance. Fresh water makeup in conjunction with water added through feed ore moisture balances water losses in lead concentrate, zinc concentrate and filtered tails, and requirements for potable water and dust suppression. Water recovered from concentrate and tailings dewatering processes is recycled to provide process water throughout the circuit. Process water will also be topped from the contact water pond as needed.

The Corani process plant requires approximately 238 m^3/h of fresh water makeup. In addition, an average of 22 m^3/h of fresh water is estimated for mining dust suppression.

The fresh water requirements are summarised in Table 17-5.

Table 17-5: Fresh water consumption

Area	m^3/h	m^3/t ore
Mine (dust suppression)	22	0.018
Crushing and coarse ore stockpile (dust suppression)	22.1	0.018
Grinding	2.4	0.002
Flotation, regrinding and concentrates thickening and filtration	123	0.097
Tailings thickening and filtration	-	-
Reagents	33.3	0.027
Potable water	4.1	0.003
Gland water	48.2	0.039
Other	4.6	0.004
TOTAL	260	0.209

17.4.2 Fresh water

The process water requirements are summarised in Table 17-6. Zinc concentrate thickener overflow (~78 m^3/h) is recirculated directly to the zinc flotation circuit and additional process water required in the zinc circuit (~100 m^3/h) is sourced from the process water tank. The recirculated water will contain residual copper ions that could activate sphalerite surfaces, therefore the preference is to reuse this water in the zinc flotation circuit.

Table 17-6: Process water consumption

Area	m^3/h	m^3/t ore
Grinding	2,530	2.069

Area	m^3/h	m^3/t ore
Lead flotation and regrind	276	0.226
Zinc flotation and regrind	178	0.146
Concentrate thickening and filtration	33	0.027
Tailings thickening and filtration	545	0.446
Reagents (flocculant dilution)	73	0.060
Miscellaneous	2.4	0.002
TOTAL	3,637	2.975

17.4.3 Water storage and distribution

Fresh water will come from the non-contact water conveyance system (see Section 18.2.6) and be stored in the non-contact water pond. Fresh water for the process plant will be sourced from the non-contact water pond and pumped using submersible pumps on rafts to a head tank. The head tank will hold the raw and fire water, with the fire water storage being at the bottom of the tank and the raw water on top with an elevated dedicated discharge line to the plant.

Process water will be mainly made up of the overflow stream from the lead concentrate and tailings thickeners. The lead concentrate overflow will be pumped to the process water tank (also acts as the tailings thickener overflow tank) and there it will be mixed with the tailings thickener overflow and pumped around the plant. Process water will be topped from the contact water pond using a submersible pump mounted on a floating raft that is permanently located in the pond. Contact water may have elevated salt, dissolved metals and acidity concentrations, however it is considered suitable for process water requirements.

Zinc concentrate thickener overflow will report to the zinc concentrate thickener overflow tank and will be recirculated back to the zinc flotation circuit (primarily in flotation cell launders). Any additional process water requirements for the zinc circuit will be sourced from the process water tank.

The overall process plant water consumption and distribution is summarised in Table 17-7.

Table 17-7: Process plant water input and outputs

Source	m^3/h	m^3/t ore
Ore moisture	37.8	0.031
Make up fresh water	209	0.17
Water losses in concentrates	3.3	0.003
Water losses in tailings	244	0.20

100% of the water recovered from the process is reused. The following reuse criteria were applied:

- The process plant make-up water will be preferentially sourced from the contact water pond. This water may have elevated salt, dissolved metals, and acidity concentrations. Despite these concerns, it is considered suitable.
- When necessary (when the contact pond is dry, or for applications requiring better water quality), water will be collected from the non-contact water pond. The drainage basin near the mine has sufficient water to supply operations (see Section 18.2.6).
- tailings from the lead concentrate thickener may be pre-treated with Hydrogen Peroxide and Activated Carbon in solution and pass through in-line filter units in order to allow the extraction of any colloidal fine particles and residual reagents.

17.5 Power Consumption

The process plant power consumption in summarized in Table 17-8. The average annual energy consumption is 370 GWh, which corresponds to 37.5 kWh/t of ore processed.

Table 17-8: Process plant power consumption

Area	Average Operating (kW)	Power Consumed (kWh/t)
Primary crushing	760	0.47
Stockpile and reclaim	485	0.37
Grinding and classification	21,428	17.5
Flotation and regrind	11,591	9.48
Concentrate thickening and filtration	1,263	0.94
Tailings thickening	647	0.53
Tailings filtration and stockpile	8,615	6.34
Reagents	388	0.32
Air and water services	1,373	1.12
Auxiliary	893	0.36
Total	**47,539**	**37.5**

As is shown in Figure 17-4, the areas of highest power consumption will be Grinding and Classification (46.7%), followed by Flotation and Regrind (25.1%) and Tailings Filtration and Stockpile (16.9%), while the remaining areas of the plant account for 11.3% of power consumed.

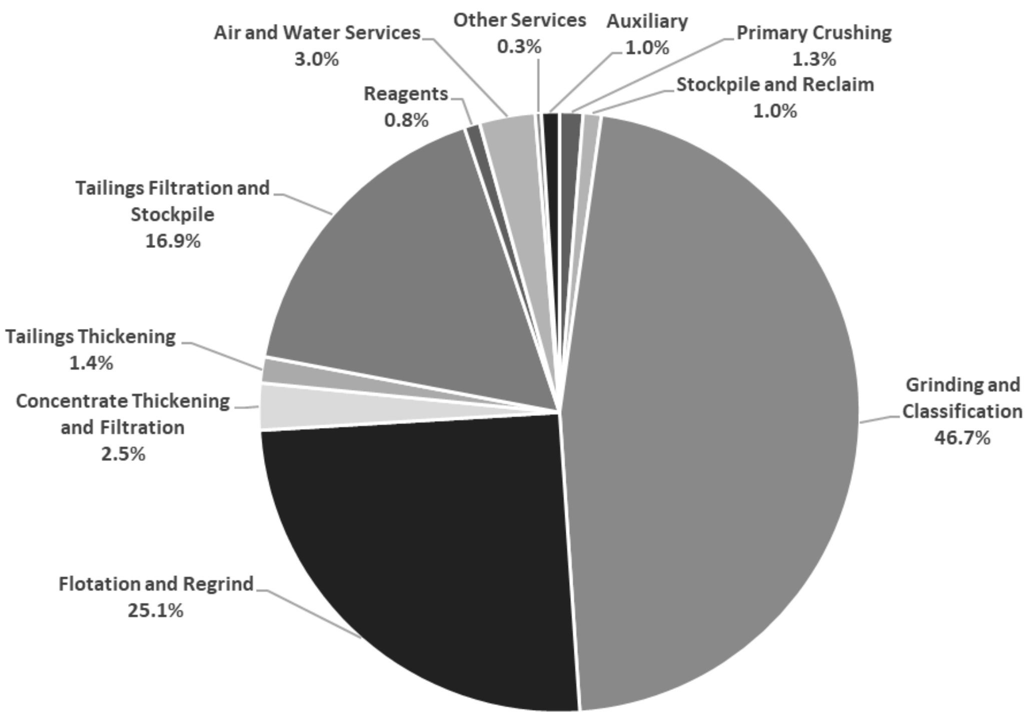


Figure 17-4: Process plant power consumption by area

17.6 Process Control System

The control philosophy proposed for the Corani process plant is typical of control systems employed in recent mineral processing plants.

The proposed plant process control system (PCS) will be a Programmable Logic Controller (PLC) based Supervisory, Control and Data Acquisition System (SCADA) with PC-based operator control stations.

The PCS will be an integrated plant wide design, enabling the start-up, monitoring, control, and shutdown of equipment from the plant control rooms. PC-based human machine interface (HMI) stations will provide dynamic graphical representation of plant operation, alarm notification, trending, data collection and reporting to assist in the control of the plant.

The following information will be displayed via the HMI stations:

- graphic (mimic) displays of plant areas, showing equipment status (ready, not-ready, running, etc.) and analogue values for process variables
- alarm display and logs
- graphic trend displays
- PID loop controllers/displays.

Field instrumentation will provide inputs to, and be controlled by, PCS controllers. The controllers will perform the following control functions:

- collect status information for drives, instrumentation and packaged equipment
- provide control for drives, valves and process interlocking
- provide PID (proportional-integral-derivative) control for process control loops.

Standard process control equipment such as flow meters, level sensors, and density gauges will provide inputs to the PCS.

PCS controller software will be configured in a structured form using a defined library of ‘function blocks’. The function block library and control actions will be described in the PCS programming specification, which will be developed during the next phase of engineering.

In general, plant equipment will be controlled directly from the PCS. However, where specific equipment forms part of an approved vendor package (e.g. filters, OSA) instrumentation and drives are controlled from a vendor control panel. Where required, a communications interface will be made available to enable remote control and monitoring from the PCS including digital and analogue signals for alarms, faults, instrumentation and monitoring, motor and valve control, process variables and interlocking controls.

17.6.1 Process control philosophy

The strategy for control of the primary crushing circuit involves:

- maximizing feed rate to the grinding circuit at the required SAG mill feed size distribution
- maintaining the crushed ore stockpile at a suitable level to ensure the downstream processes are not unduly affected by minor fluctuations in ROM ore feed rate.

The control strategy in the stockpile and reclaim area involves:

- monitoring and controlling the stockpile feed to maintain a reasonable inventory for delivery at a controlled rate to the grinding circuit.

The control strategy for the grinding circuit is to maximise mill throughput whilst optimising SAG mill load and power draw. To prevent over-loading the SAG mill, the power draw and mill load controllers will be configured in parallel and have their respective outputs cascaded to the mill feed rate controller.

The primary control strategy in the grinding circuit involves maintaining the SAG mill weight within a required band to optimise grinding performance and mill throughput as feed conditions change. SAG mill power draw can also be controlled by varying the speed of the motor. The process has a significant dead time component in seconds (dead time in this instance is the time for a change in feeder speed setting to be noticed by the weightometer). The output of the mill load controller or power controller is paused if the SAG mill or SAG mill feed conveyor belt stops.

Other key aspects of the control strategy will include:

- optimizing product size distribution (flotation feed size)
- stable volumetric flow rate of slurry to the flotation circuit
- controlling cyclone feed density and pressure to optimise cyclone cut point.

Cyclone feed density and pressure will be automatically controlled to optimise cyclone performance. Cyclone feed pump speed will be automatically controlled to maintain the cyclone feed flow rate and/or cyclone feed hopper level. Cyclone feed density will be automatically controlled with the addition of water to the cyclone feed hopper. Cyclone feed pressure will be adjusted manually by opening or closing cyclones from the PCS or controlled automatically via an on / off controller.

The strategy for control of the flotation circuits involves:

- stabilising flotation cell pulp levels
- optimising intermediate flow rates within the flotation circuit (stabilising mass pull and circulating load)
- use of on-stream analysis to assist in selection of set points for reagents, air addition and cell level
- Lead Circuit: maximise recovery of lead to the lead concentrate whilst producing a concentrate grade to a nominated target and minimising the recovery of zinc to the lead concentrate
- Zinc Circuit: maximise recovery of zinc to the zinc concentrate whilst producing a concentrate grade to a nominated target and minimising the recovery of pyrite to the zinc concentrate.

Separate controllers will be provided for cell level control; air addition control; pump hopper level control and select water addition flow rates. Flotation feed densities are checked manually, and water flow rates adjusted to provide selective recovery and dilution cleaning downstream of the regrind mill.

The overall strategy for control of the regrind circuits involves:

- providing a stable feed rate to the regrinding circuit (cyclones and regrind mill) whilst producing a cleaner flotation feed size distribution as close as possible to a nominated target

- controlling the specific energy by varying mill speed and media addition
- controlling cyclone feed density and pressure to optimise cyclone performance
- use of a particle size analyser to monitor product size and adjust regrind mill parameters.

The strategy for control of the concentrate and tailings thickeners involves:

- controlling flocculant addition to produce a clarified overflow suitable for use as process water and a stable underflow density
- controlling thickener inventory and bed pressure to produce a consistent underflow at the required density, while maintaining rake torque within the limits specified by the vendor.

The concentrate and tailings filters will be each controlled from a vendor supplied PLC which performs the following functions:

- operation of the filters
- starting and stopping of the filter feed, filter pressing water, filter cloth wash and manifold wash pumps
- opening and closing of sequence control valves for filter air, water and slurry feed requirements
- control of filter feed pump speed
- monitoring of filter feed tank levels.

18 Project Infrastructure

18.1 Transportation

Transportation to and around the site is by roads that will be developed and improved to accommodate the demands of the project. An access road has been designed to link the project site to the Interoceanic Highway that provides access to the town of Macusani and to the rest of the country for receiving supplies and delivering products. The lead and zinc concentrate produced by the mining and mineral processing operations will be delivered to the Port of Matarani or other destination via trucks using the access road and Peru's public highway system.

18.1.1 Access road

The Construction Access Road to the Corani Project, an improvement of an existing access (PU 516 – PU 514), will be a 42 km access road connecting to the Interoceanic Highway (PU 55) which in turn is connected through the Peruvian highway system to the Port of Matarani (632 km from the Project site). The Port has facilities for concentrate shipment. Figure 18-1 shows the Plant access and site roads. The Interoceanic Highway is a two-lane, paved highway that connects the Peruvian port cities of Matarani and Ilo.

The construction access road intersects the Interoceanic Highway at km 7+000 from Macusani just after the Macusani toe unit (from intersur road concession), about 7 km north of Macusani.

The construction access has few interferences and requires minimal CAPEX investment due the alignment and location. The road will follow the natural undulations of the highlands terrain without compromising sightlines and safety features. There is minimal impact on local residents as there are no communities along the route. The access will also be available if needed during operations and can be used to receive supplies and deliver the lead and zinc concentrates to the Port of Matarani or other ports via over the road trucks connecting to Peru's public highway system.

The 44 km new highway design by GMI and included in the 2017 technical report, has a government investment budget for construction approved for 2021 and will be available for operations assuming funds are released and construction advances as planned.

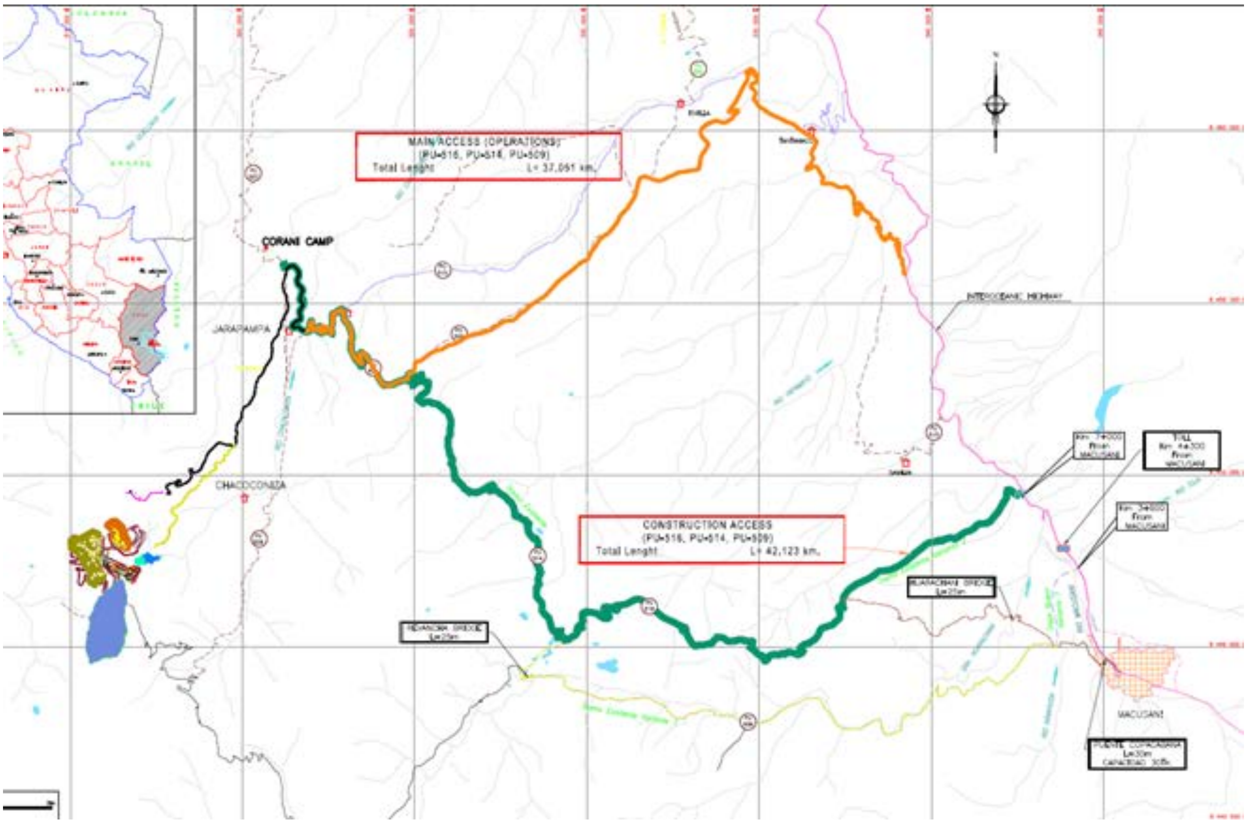


Figure 18-1: Map of main access to the operation

The access from the camp to the plant site, has been re-routed to add the power line maintenance access, which will enter the process plant through the area adjacent to the contact and non-contact water ponds. compared to the previous alignment, this avoids requiring all incoming traffic to be crossing haul roads, reduces the frequency and risk of interactions between light vehicles and mine equipment, and shortens the route by 2.4 km. Figure 18-2 shows the access.

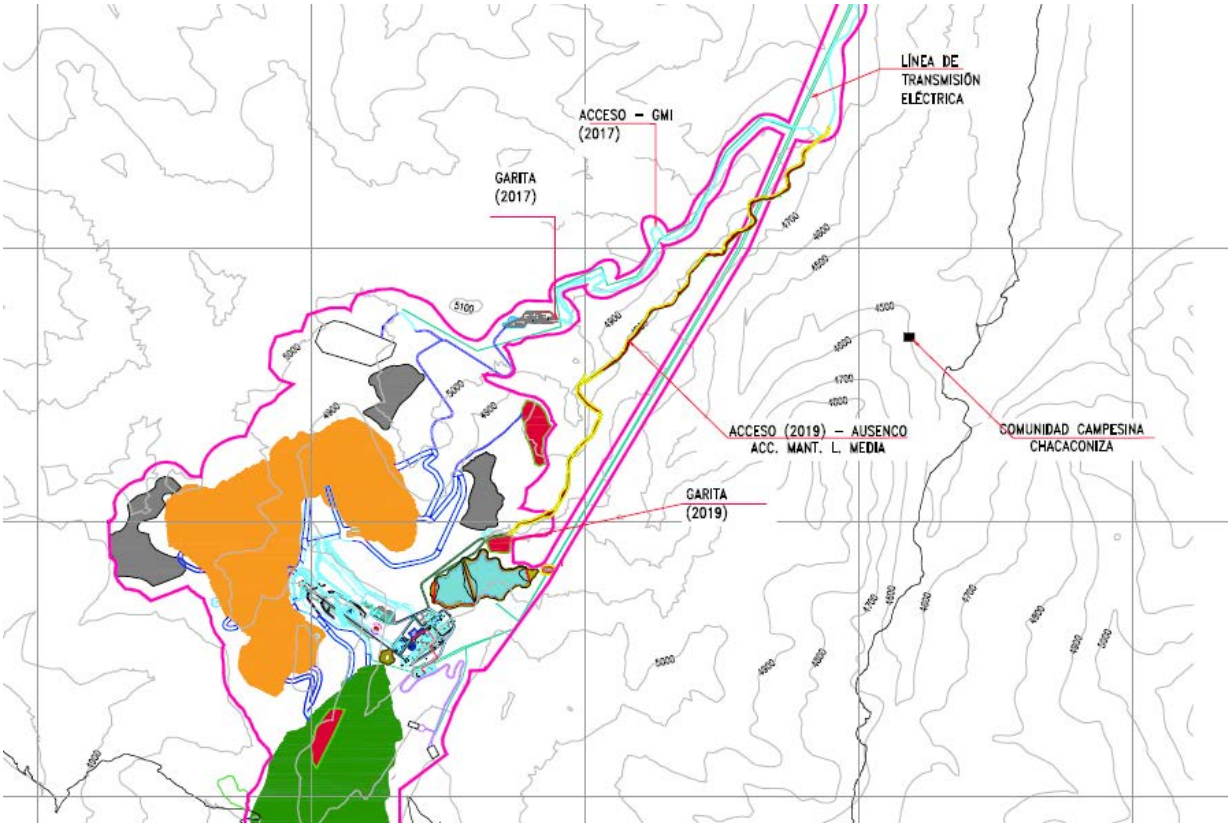


Figure 18-2: Map of access from camp to site

A network of roads has been designed to connect the facilities within the mine and plant site (Figure 18-3). The internal access 1, which includes the power line maintenance access, will connect the process plant and its facilities to the camp with a 12 km access road.

The process plant entrance (gatehouse) will be located at 4,750 masl. Additional roads for access to the various components of the process plant, the plant water pond, and other project facilities have been laid out and included in the initial capital cost (Figure 18-3).

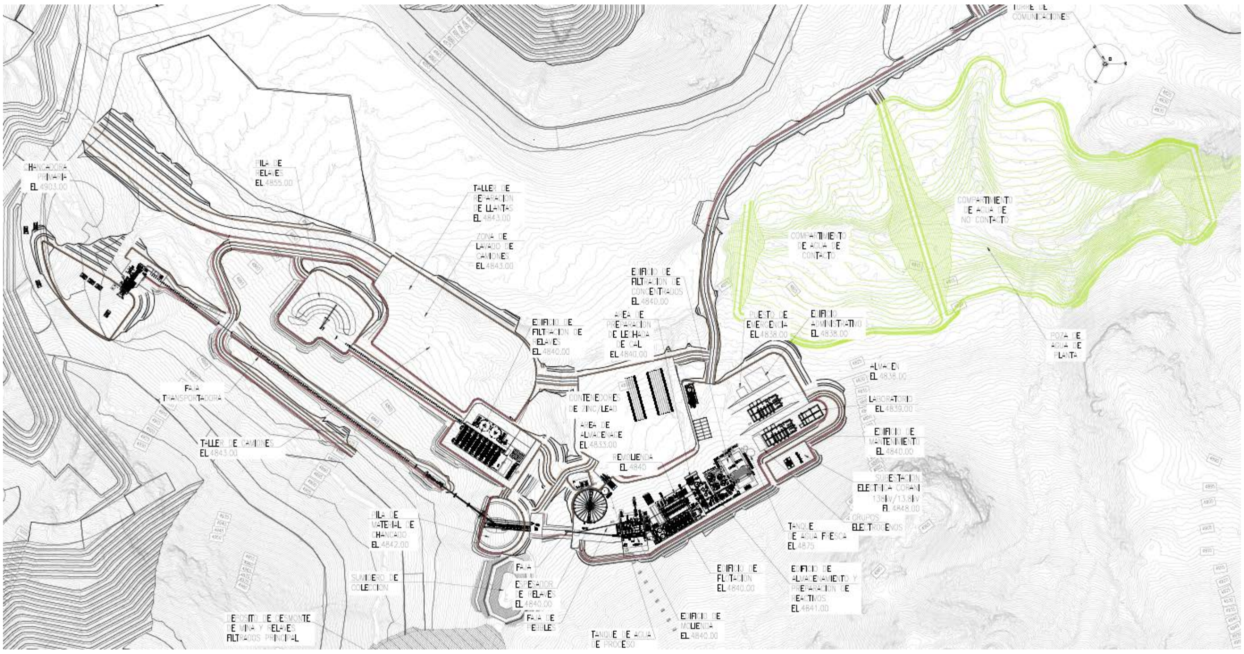


Figure 18-3: Map of access to site

18.2 Site Buildings and Facilities

Corani Project buildings and facilities are divided into four functional areas: mine service faciliites, administration, process facilities, and residential facilities.

18.2.1 Mine service facilities

Truck and facility workshop

The truck workshop will be in the north-western part of the industrial area and will have access from the mine via the primary crusher and will also have access for light vehicles from the process plant.

The truck workshop (approximately 17,500 m^2) is intended for the preventive maintenance of heavy and light vehicles. It has 3 bays for heavy trucks and 1 bay for special equipment, and 1 welding bay.

A truck wash will be in front of the truck workshop building and will have 1 bay for heavy trucks and 1 bay for special equipment.

The buildings will be structural steel with metal cladding.

The truck workshop will house the following:

- reception
- dining room
- mine operations and maintenance offices
- meeting room
- washrooms

- tire changing area and tires storage
- heavy equipment spare parts storage
- welding equipment
- electric equipment
- mechanical equipment
- light equipment maintenance workshop
- light vehicle parts storage
- truck washing bays
- maintenance bay for heavy equipment
- maintenance bay for special equipment.
- lubricants and additives warehouse
- electrical rooms
- IT room
- training rooms
- parking area.

Fleet management system

The mining contractor will provide a fleet management system including all necessary hardware and software and operate the system from the commencement of the phase 2 pre-strip until the end of the mine life. The system will be integrated with the overall site operational data management system.

Powder and detonator magazine

The powder and detonator magazine cover an area approximately of 2,000 m^2, intended to store accessories and explosives (ammonium nitrate and emulsion).

18.2.2 Administration facilities

The administration offices and warehousing are located near the process plant. The administration facilities include the main gate and guard house, an administration office building, and a warehouse to receive parts and supplies necessary for operation and maintenance.

The administration and warehouse are located adjacent to the process plant along with the laboratory, first aid building, plant maintenance building and reagent storage.

Gatehouse, truck wash and weighbridge

The gatehouse is located near the ponds at the end of the power line maintenance access.

The main gatehouse and plant entry at the entrance of the industrial area, will be composed of a checkpoint for the control of personnel and an area for the control and registration of vehicles. The guard house will contain security offices, restrooms, and a small reception area. A security firm will be contracted to provide on-site security services to the Project starting at the time of preproduction. The area also includes a weighbridge (scale) to weigh incoming trucks of supplies and outgoing concentrate and a wheel wash for concentrate trucks.

Warehouse building

The warehouse (820 m^2) is a steel structure building with an interior module measuring 29 m^2 (equivalent of a 40 ft container), a truck manoeuvring area, and the warehouse yard used to store materials that are not affected by climatic conditions. The warehouse is on a raised levelled area at 1.20 m above the natural terrain, in order to facilitate the unloading of trucks.

The building has different accesses for plant personnel and deliveries. Inside the main building there will be a separate area enclosed by wire mesh for small high value small articles.

Within the main building (roofed area) are the main storage area, small item storage, offices, restrooms, and a small reception. Outside areas include a material receiving area with a truck reception platform, manoeuvring area and storage yard.

Administration building

The administration building (1500 m^2) is located close to the process plant and includes workstations, offices, a reception area, a conference room, a training room, and female/male restrooms. The offices will be used for senior personnel, records and archives, accounting, and engineering. Other areas of the building contain conference rooms, restrooms, and a reception area. This facility will be a heated/air-conditioned pre-engineered metal modular building with insulated roofing and siding installed on a concrete mat foundation. A gravel surfaced visitors and employee parking area is also provided.

There will be also an administration building and access control at the Camp facilities, which will be the first point of entrance control to the Corani Project.

18.2.3 Process facilities

The process plant facilities include a primary crushing facility located close to the mine, crushed ore stockpile area located close to the concentrator area, a grinding building, a flotation area, a concentrate dewatering and load-out facility, a tailing filtering facility, a plant maintenance building, reagent storage facility and laboratory building.

Primary crusher

The primary gyratory crusher structure will be of reinforced concrete and structural steel construction. ROM ore will be delivered using 140 t trucks and fed into the crushing circuit via a single dump hopper with 240 t live capacity.

Ore will be crushed using a gyratory crusher and discharged via an apron feeder onto the crushed ore sacrificial conveyor and transferred onto the overland conveyor. The primary crusher will have the following components:

- ROM dump pocket
- gyratory crusher and ancillary equipment
- rock breaker and power pack
- crusher discharge vault
- feeder, sacrificial conveyor and overland conveyor and associated chute work

- hoists for maintenance
- dust extraction.

Stockpile and reclaim

The overland conveyor will feed the crushed ore to a single conical coarse ore stockpile. The stockpile will have a total storage of approximately 49,000 t, which is equivalent to approximately 1.7 days at nominal throughput. The stockpile will provide adequate surge capacity between the primary crusher and the process plant for ROM ore tonnage fluctuations and maintenance activities.

The crushed ore storage facility will be an open stockpile with no superstructure. A concrete tunnel will be provided beneath the stockpile to house the reclaim feeders. The feeders discharge onto the SAG mill feed conveyor that connects the stockpile to the mill building.

Coarse ore will be reclaimed from the stockpile by two apron feeders. Each apron feeder will be fitted with a variable-speed drive. Each feeder will have the capacity to provide 100% of the full design tonnage rate to the SAG mill but will generally operate at 50% capacity for even draw down of the stockpile.

The apron feeders will discharge ore onto the SAG mill feed conveyor which will transport it to the SAG mill feed chute.

Grinding and classification area

The grinding circuit will consist of a SAG mill, and a ball mill operating in closed circuit with a cyclone cluster. The mills are located almost perpendicular to the feed conveyor to optimise footprint. A platform is located adjacent to the grinding mills to provide sufficient space for mill relining activities. The grinding cyclone, ball charge hoppers and metallurgical sampler are located above the mill maintenance floor.

The grinding area includes overhead cranes for maintenance and ball charging lifts. The grinding area includes a drive on ramp to facilitate spare parts and liner movements as well as giving access to relining machinery and vehicles.

The grinding area can be accessed from the SAG mill conveyor walkways and stair towers. There is also an elevated walkthrough between the flotation and grinding areas where the control room will be located. The platforms around the mills include sufficient spacing in between mills and on the outside to perform reline activities such as removing and tightening bolts, and general inspections. A dedicated platform has been included in the cyclone feed line for operators to reach the density meter and flow meter. Access has also been provided to the cyclone top level and all around the sampler and ball charging equipment for maintenance.

The grinding building will house a SAG mill, ball mill, primary cyclones, mill discharge screen and primary cyclone feed pumps in an engineered structure with metal roofing and siding. Internal structural steel platforms that provide maintenance access are supported independent of the main building structure. The building foundations will be concrete spread footing, piers and grade beams. Concrete floor slabs will be provided over the entire area of the building with curbed containment for spillage control and thicknesses suitable for maintenance traffic where appropriate. Overhead travelling cranes are provided for maintenance of the grinding equipment. Controls rooms and offices within the structure will be air conditioned/heated, with high-bay lighting provided throughout.

Flotation area

The flotation area is located east and adjacent to the grinding area taking advantage of the gently sloping ground. The cells have been arranged in parallel halves with one half comprising the rougher cells for both lead and zinc, and the other half housing the cleaner and cleaner scavenger trains for both lead and zinc.

The cells are organized in a cascading arrangement with each cell corresponding to a step. The rougher cells are located to the south of the plant whilst the cleaner cells are on the north side. The flotation cells are mounted on ground on concrete foundations with the exception of the 1st cleaner and some of the cleaner scavenger cells which are mounted on steel frames.

The flotation circuit has an engineered structure with metal roofing and siding. Concrete floor slabs with curbed containment are provided to maintain separation between the lead and zinc circuits. Flotation tanks will be elevated on concrete and structural steel supports and cell covers and structural steel access platforms will be included. Maintenance on the flotation circuit will be facilitated by two cranes to serve the lead and zinc sections separately. High-bay area lighting will be provided to illuminate the area.

Concentrate thickening and filtration

The concentrate area is located east of the flotation building across the internal access road. The area consists of a Ø12 m and a Ø11 m thickener for lead and zinc respectively, filter feed tanks, and the concentrate filtration plant including two horizontal plate filters and ancillary equipment.

The thickeners arefed by gravity from the 3rd cleaner column concentrate launder after it has passed through the metallurgical sampler and the feed box. The metallurgical sampler is located in the column cell structure that also houses the concentrate sample transfer pump and the access stairs. The sampler has access around the feed box to adjust the feed sluice gate and the cutter for maintenance.

The filters are housed inside an elevated building. Filtered concentrates will be stored in segregated piles underneath the filters. The filter floor is made of concrete to prevent spillages reaching the final concentrate and facilitate clean-up. The filters are both fitted with plastic enclosures to control spillages in case of hose breaks or cloth failures. The filter cake is passed though the floor to the storage area where a front-end loader will transport concentrate into the loading system with the final product being loaded into shipping containers.

Dust is captured using separate and dedicated lead and zinc cartridge dust collectors. To avoid fugitive dust in the loading area, the containers will seal against a rubber seal installed in the building's flashing.

The filter building is an engineered structure with metal roofing and siding and is fitted with an overhead travelling crane to help with the removal of plates and filter components. There is an opening to the side of the building to drop these items onto a truck loading/unloading zone

Tailings area

Tailings from the Zn flotation circuit are passed to two 42 m diameter high compression thickeners where the solids are increased to 60% before being filtered. The location of the thickeners is between the grinding and tailings filtration areas is ideal as it will allow shorter pipe routes to the tailings filtration area and also shorter process water piping routes from the overflow tank to the grinding area which is the heaviest user.

The thickeners are on-ground steel structure high rate thickeners with a concrete slab and steel walls. The rakes are driven by a centre column mechanism which can be accessed via bridge from the tank's edge or from the centre column ladder. The bridge structure includes a monorail to remove drive components to an accessible location for a mobile crane to transfer to a truck. The thickener mechanism is a planetary gear driven by individual pinions.

Tailings filtration and stockpile

The tailings filtration plant consists of 10 vertical plate filters and ancillary equipment. The building has been located to the north of the thickener to take advantage of the terrain and minimise the number of conveyors transporting the final tails material to the stacking system. The slurry feed tanks are located on the side of the building and provide the best location to minimise piping runs to the head of the filters. The filters are mounted high in the building to allow space underneath for the bomb bay style doors and a belt conveyor to take the cake to the transfer conveyor. The space above the filters is reserved for access platforms to the top of the filters for cloth changeouts and to house the hydraulic power packs.

The filter building includes cranes which will be used to remove cloths, plates, hydraulic cylinders or other items as required for maintenance.

The transfer conveyor runs on the east side of the building on the outside alongside the tanks. All items outside of the building are intended to be maintained using a mobile crane.

The transfer conveyor will have a road adjacent to it so that maintenance activities can be carried out with mobile equipment. The conveyor will be supported on ground modules with the drive mounted to the head pulley shaft.

The conveyor discharges to a radial and telescoping stacker to create the tailings stockpile which will be reclaimed using FELs and trucks to then be transferred to the main mine waste and filtered tailings deposit. The tailings stockpile has capacity for 24 hours of production.

Reagent storage

The reagents area includes the lime, flocculant, SIPX, MIBC, promoter, copper sulphate, sodium sulfide and zinc cyanide reagent areas as well as the reagent storage building.

The reagent building has access through a roller door. The building includes an overhead travelling crane for general maintenance and loading of bags or IBCs to the respective reagent mixing areas. Furthermore, two hoists will be provided for the mixing of sodium sulfide and zinc sulphate due to the large amount of bags expected to be mixed per day for these specific reagents.

The reagent storage is a separate building with capacity for 14 days storage. The 20 m x 60 m covered building is located on the junction between the main entrance road and the main plant road east of the concentrate building. The storage is based on double stacking and all movements will be completed by a forklift. The building is a simple concrete apron with full coverage and enough space to manoeuvre a forklift to tram the bags and IBCs in and out of the storage zones. This location is ideal as it is outside the main plant area and also in close proximity to the reagents building. The cyanide storage area will be fenced to improve safety around this hazardous reagent.

Plant maintenance building

The plant maintenance building will be a pre-engineered structure with insulated roofing, siding, and a reinforced concrete mat foundation. Is a 624 m^2 building of mixed modular construction and a steel structure.

The building includes the following:

- Storage module – 124 m^2: Electric and Instrumentation storage, tools storage, electric Room, machine room and Compressors room
- Office module – 2 floors, 248 m^2 each: female / male locker room, female / male restrooms, reception area, canteen, airlock, 5 offices, 12 workstations, print area, meeting room, data center, female / male restrooms and kitchenette.

Laboratory

The laboratory building will be a pre-engineered structure with insulated roofing and siding, and a reinforced concrete foundation. This building shall have 120 m^2 of light steel structure (average height 3 m) and 444 m^2 of modular building (Total of 564 m^2), The building will be heated/air conditioned, and fume extraction and dust collection equipment is provided.

The building includes the following:

- samples reception and drying
- sample preparation
- mine samples storage
- fire assay lab
- radioactive storage
- instrument washing and deposit area
- wet lab
- metallurgical lab
- technical lab
- weigh area
- office
- chemical supervisor
- airlock
- electrical / mechanical room
- female / male restroom
- female / male locker room
- janitor’s area
- storage gas tanks (02).

First-aid

The first-aid building shall have 156 m^2 of light steel structure (average height 3 m) and 58 m^2 of modular building, both types of construction are part of the scope.

The light steel structure will be use as a parking lot, therefore, it can be designed as an “arcotecho” or similar efficient structure. The modules will be 40 ft containers or similar and can support the metallic structure.

- module 1: office and storage
- module 2: medical treatment room (02) and restrooms (02).

18.2.4 Residential facilities

The distance of the Project site from any significant urban area that could provide lodging and services to mine personnel requires that the Project includes a residential camp. The camp site is located at an elevation of approximately 4,400 masl, about 10 km to the northeast of the mine facilities. The UTM, Zone 19S, Datum PSAD56 coordinates of the site are: 8,456,510 m North; 321,312 m East.

Figure 18-4: Camp location

The camp has been organized in such a way that the facilities are located in a central core. The core/circle design is to offer higher protection against cold wind and other inclement weather.

Parking areas are centrally located close to the dormitories. Vehicle traffic is restricted in dormitory areas and only design for pedestrians is considered. The inner circle distributes traffic for facilities centrally located for delivery of supplies.

The external circle is for ease of pick-up of workers to transport them to the mine and return to the camp.

Camp capacity was estimated based on the projected occupancy curve resulting from the construction program and the estimation of permanent staff in the operation stage.

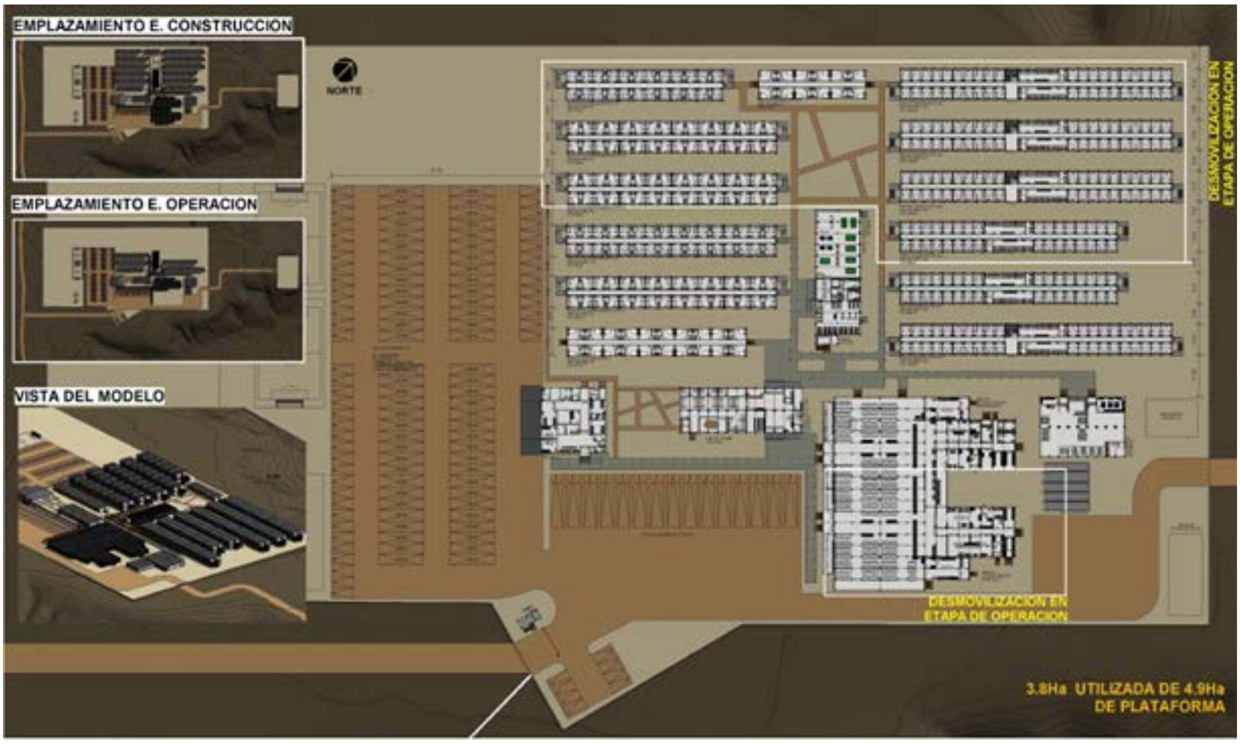


Figure 18-5: Camp layout

The accomodation village include the following areas:

- design, supply and installation construction accommodation village
- camp management services
- camp services: electrical power including substation, fire water, communications and fire detection
- camp stand-by power generators
- water collection system
- dining and food preparation
- recreational facilities
- water and wastewater management
- security services
- first aid medical post
- the camp includes four types of accommodation:
  - type 1 – managers, 1 person per room each with own bathroom
  - type 2 – superintendents 1 person per room shared bathroom between every 2 rooms
  - type 3 – supervisors 2 persons per room shared bathroom between every 2 rooms
  - type 4 – workers 2 persons per room common bathroom (96 to common bathroom).

18.2.5 Communications

Radio communication system

The project will have a digital radio communication system at site that will have the ability to communicate with radios located in all areas of the mine, plant and administration areas, across the project footprint. It will include a base station and the repeater stations that are necessary for its operation, and the provider will procure all permits and licenses for the use of the frequencies that are needed.

At least 5 radio channels (surveying, operations, supervision, maintenance and emergency) will be required for the Project

Voice and data communication system

The voice and data communication system will be installed in the different administrative offices, hospital, dining rooms, recreation, accommodation, control rooms and checkpoints, all buildings will include conduits and trays for cables (voice and data) and for the power supply for the telecommunications cabinets.

The telecommunications cabinets will be distributed in different areas of the plant. The cabinets will contain the voice and data network, equipment and accessories, the network of the fire detection system and the controls network. Transmission is via interconnecting fibre optic cables.

Cable TV distribution system (CATV)

Buildings will include conduit required for CATV system. These will be independent of the conduit for other systems, but they will be able to share cable tray, provided metal separators are installed.

The CATV system supplier will supply all the equipment, licenses and accessories for the reception and distribution of the TV signal in the housing modules of the camp.

Storm detection system

A storm detection system will be installed that is capable of detecting lightening discharge and generating at least 3 alarms (15 km, 5-10 km and less than 5 km distance).

Two detection sensors will be included, one is located in the camp and another in the plant. Each sensor must have the capacity of detecting lightening at 20 km distance as a minimum. Each sensor will be connected to a control panel from which alarms will be sent to strobe lights, sirens and to an operating station located in the camp security gates.

Red strobes will be installed, which will activate when the lightening is less than 5km away. In the camp, a minimum of 10 strobe lights will be connected to the storm sensor controller. For the process plant, 06 strobes with siren will be required.

All alarms and sensors will be powered by 120 VAC supplied from the instrumentation panels located in the electric rooms.

The alarms will be located at front of the doors of the buildings, in pedestrian zones and in parking lots.

18.2.6 Water supply and management

Surface water and groundwater will be used to supply the water needs for the Corani Project. Water is classified in two ways: either as contact water or non-contact water. Contact water is water that comes in contact with mine facilities or rocks capable of generating ARD. Contact water includes pit dewatering water, runoff from the open pit, and seepage from the mine waste and filtered tailings deposit. Non-contact water is generated from runoff from unimpacted ground surfaces. Both contact and non-contact water are stored until required in a dual-cell water pond located near the plant. The contact water cell can hold 257,000 m^3, and the non-contract water cell can hold 572,000 m^3. At the east side of the water pond it´s located a water table pond with a capacity of 11,000 m^3 storage.

Section 20 describes the site-wide water balance used to verify that the plant has enough water in extreme dry conditions. The plant water requirements are drawn from either the contact or non-contact water pond depending on the water quality requirements of the plant. The issues of surface water management, water supply, fire protection, sanitary waste management, and are presented in the following sections.

Surface water management plan

The site surface water management plan was formulated to provide a template for management of storm water runoff and stream flows in the project area. The effective management of surface water resources at the project site is critical to the protection of water resources downstream of the project area. In general, surface water will be managed to separate contact water and non-contact water. Contact water is stored and consumed. Non-contact water will generally be diverted around the project facilities, except when mineral processing needs dictate the use of some or all the non-contact water.

From the site water balance (see Section 20), the sources of contact water are:

- pit dewatering water
- drainage from mine haul roads
- water due to precipitation in process plant area and other facilities
- precipitation over the contact water compartment in the pond
- seepage from the mine waste and filtered tailings deposit (this is primarily water extracted from the consolidation of tailings and percolation from precipitation over the deposit area)
- water collected in the organic material storage, the excess material deposit, and the solid waste storage areas.

Non-contact water sources are:

- water from creeks and valleys that have not had any contact with areas disturbed by the project
- precipitation above the non-contact water reservoir in the pond
- water runoff from precipitation in all undisturbed areas of the project. the consumption and losses in the water balance are as follows:
    - o evaporation
    - o the water balance was completed on a replenishment rate for the process plant of 201 m^3/h. However, the current process water requirement is 238 m^3/h (section 17.4.1). The higher water demand will improve the results of the water

balance, so the impact on the water balance update is expected to be positive. The water balance is currently being revisited based on the updates to the design; however, this is not expected to materially impact the outcome.

- water for suppression and dust control on roads and in the mine
- permanent discharge of non-contact water to maintain flow in the Chacaconiza creek 2.6 L/sec

Major surface water conveyance and storage structures have been designed using hydrology and hydraulics predictive models. Data used in the models was derived from previous and ongoing site investigations. Conservative assumptions have been applied where possible.

Development of the mine pits will be completed sequentially, with site conditions changing frequently. To effectively manage evolving site conditions, surface water management features and designs have been presented in "snapshots" for various years during the LOM, including the pre-production and closure periods. In practice, the surface water management features will be re-evaluated and expanded as necessary each year during the dry season in order to accommodate the expanding project footprint.

During construction, contact water will be directed to settling ponds for recycling and consumption (as dust suppression water), as necessary. At closure, contact water will be minimized by reclamation activities; however, portions of the mineral processing plant will be converted to a water treatment plant to treat any ARD impacted leachate or runoff.

Diversion ditches, culvert systems, ponds, sumps and pipelines, have been designed to address the majority of surface water flow at the project site. In addition, best management practices (BMPs) will be employed to minimize erosion and sediment transport as well as deposition of sediment at the project site. Erosion control BMPs will include the use of temporary diversion ditches, check dams, rock-containment berms, straw wattles/coir logs, silt fencing, terraced and bermed slopes, ditch linings, riprap, erosion matting/blankets, and rock or geotextile covers.

Contact and non-contact water pond design

As mentioned above, the non-contact and contact water pond are located northeast of the plant complex (Figure 18-6). The ponds have approximately 830,000 m^3 of storage, equivalent to approximately 6 months of makeup demand.

The non-contact water pond and contact water pond are adjacent to one another (Figure 18-2). The contact water pond is upstream and can store approximately 257,000 m^3 of contact water. The freshwater pond is downstream and can store approximately 572,000 m^3 of non-contact water. Zoned embankment dams will partition the basin into two ponds to manage contact and non-contact water separately. The two-pond system is intended to minimize the amount of contact water produced at the project while still ensuring a dependable water source is available during the dry season. The dams will have a low permeability core, compacted rockfill shells, a seepage cut-off, and internal drainage and filter control. The dam separating the contact water and freshwater ponds will be 15 m high, with a 1 m freeboard allowance. The non-contact water pond embankment dam will be 35 m high, with a 1 m freeboard allowance. A spillway will allow non-contact water to be released downstream. Water will be pumped from the contact water pond to the plant for use in the process circuit.

The pond size has been determined by the site wide water balance (see Section 20). It is designed from the results of a probabilistic water balance in accordance with best-industry practice. It accommodates enough water to contain (for consumption) the 95 percentile wet conditions, and enough water to supply the mine in the 5% dry conditions.

The pond system will be constructed prior to other facilities to be used as a construction-phase sediment pond and for water storage for the start of operations. The non-contact water pond includes a spillway to release water during operations.

Site-wide water conveyance and management

Precipitation runoff internal to the plant facilities is a minor percentage of the total water supply requirement. Runoff in plant areas will be collected in small sumps and recycled to the Process Water Tank.

Pit water will be collected in sumps excavated into the pit floor. Sumps are configured as a simple box-cut into the un-mined bench below the active pit floor. Water will be pumped from the sumps to the plant via skid-mounted pumps and high-density polyethylene (HDPE) pipelines.

Seepage and runoff from the mine waste and filtered tailings deposit will report to a sump that will be located at the toe of the facility, and water will be pumped from there to the plant or the contact water pond via skid-mounted pumps and HDPE pipelines.

Water may be released from the non-contact water pond during operations when necessary. The pond system includes a bypass to allow stream flows in excess of water supply demands to be routed around the pond and released to the natural drainage downstream.

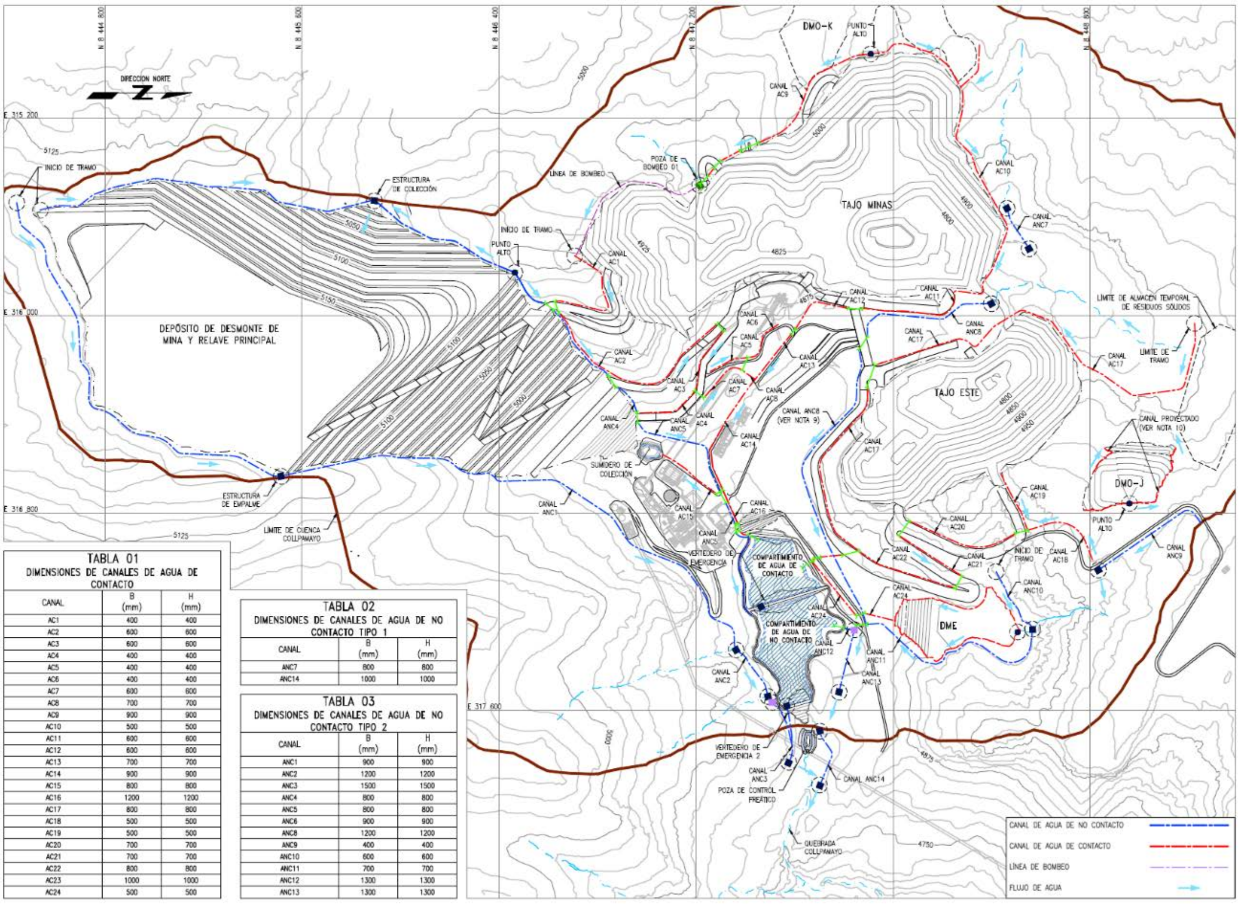


TABLA 01 DIMENSIONES DE CANALES DE AGUA DE CONTACTO		
CANAL	B (mm)	H (mm)
AC1	400	400
AC2	600	600
AC3	600	600
AC4	400	400
AC5	400	400
AC6	400	400
AC7	600	600
AC8	700	700
AC9	900	900
AC10	500	500
AC11	600	600
AC12	600	600
AC13	700	700
AC14	900	900
AC15	800	800
AC16	1200	1200
AC17	800	800
AC18	500	500
AC19	500	500
AC20	700	700
AC21	700	700
AC22	800	800
AC23	1000	1000
AC24	500	500

TABLA 02 DIMENSIONES DE CANALES DE AGUA DE NO CONTACTO TIPO 1		
CANAL	B (mm)	H (mm)
ANC7	800	800
ANC14	1000	1000

TABLA 03 DIMENSIONES DE CANALES DE AGUA DE NO CONTACTO TIPO 2		
CANAL	B (mm)	H (mm)
ANC1	900	900
ANC2	1200	1200
ANC3	1500	1500
ANC4	800	800
ANC5	800	800
ANC6	900	900
ANC8	1200	1200
ANC9	400	400
ANC10	600	600
ANC11	700	700
ANC12	1300	1300
ANC13	1300	1300

Figure 18-6: Management of contact and non-contact water

The project will utilize a fresh/fire water tank at an elevation of approximately 4,900 masl. Water for this tank will be drawn from the non-contact water pond. Fresh water for plant use would be drawn from the upper nozzle on the tank to ensure an adequate reserve for a fire water supply. Fresh water will also be drawn from the upper nozzle for the drinking water treatment plant to supply potable water to sinks, eye-wash stations, and drinking fountains throughout the facilities. A nozzle at the bottom of the tank provides water for fire suppression (Section 18.3).

Fresh water for the Administration area and Camp area (near the main entrance) will be taken directly from the Imagina Mayu River at coordinates 320,600 East, 8,456,155 North, through a catchment and pumping system located on the shore. The water will then be pumped to a Fresh Water Tank located in the Camp. The fresh water will be treated for sanitary use and cleaning. Additional treatment will be provided for the kitchen. The water for human consumption will be provided from the fresh water tank at the camp. Water from this tank will provide freshwater and fire suppression for the Administration area. Water for camp use will be drawn from the upper nozzle on the storage tank, while water for fire suppression will be withdrawn from the bottom nozzle to maintain sufficient capacity in the lower part for fire suppression at the camp.

All wastewater from the Camp will be collected through a drainage system and taken to a Wastewater Treatment Plant (WWTP) also located inside the camp. Wastewater will be treated to comply with the corresponding LMPs to be discharged to the Chacoconiza river at the coordinates 322,079 East, 8,456,150 North.

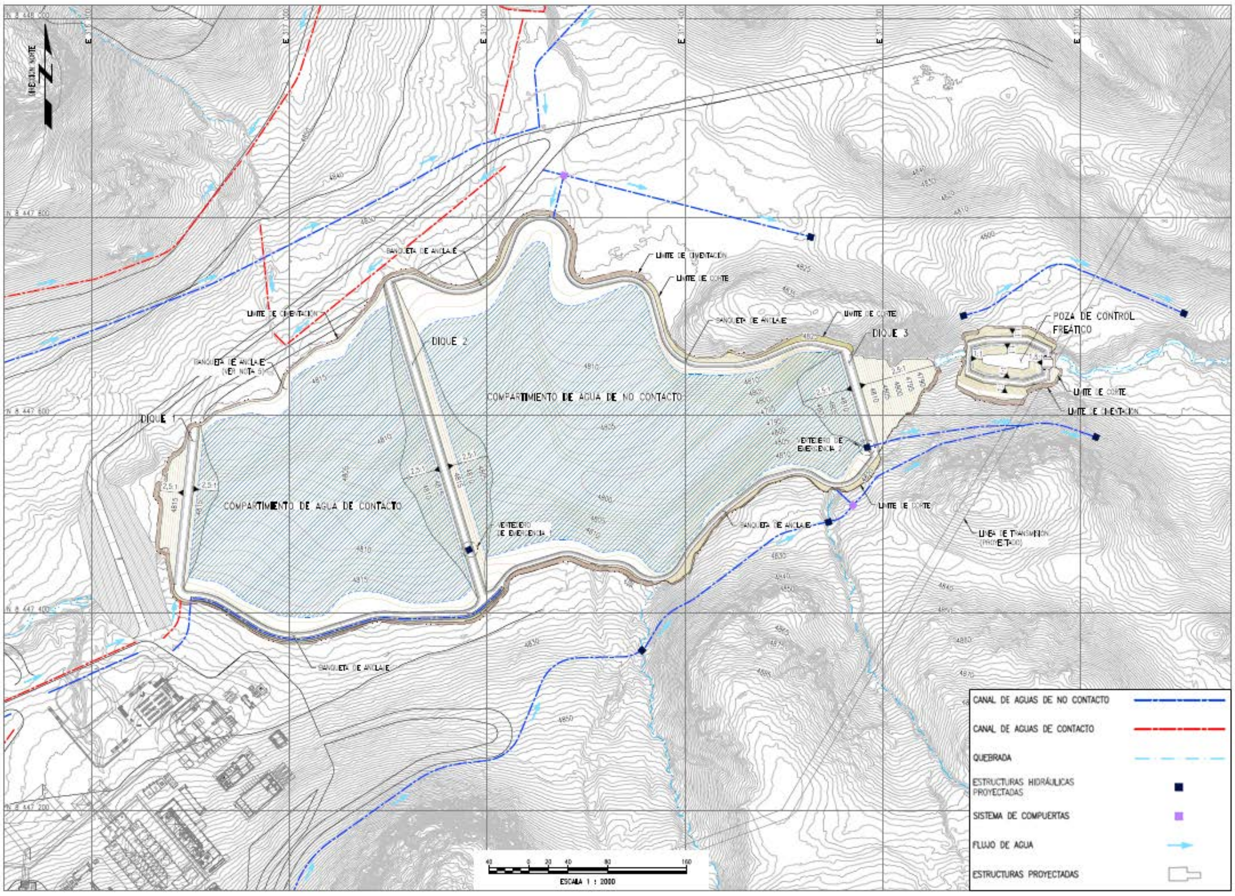


Figure 18-7: Contact water pond and freshwater pond

18.2.7 Power supply and power distribution

A new 138 kV power transmission line is necessary to provide power to the Corani project. Antapata new power substation will connect with power transmission lines L-1013 and L-1051 (San Gabán II – Azángaro) as the power source. A new 138 kV power transmission line will be built to connect the Antapata substation to the main Corani substation to be built near the Project's main process buildings. The proposed alignment for the 138kV line (Figure 18-1) was provided by Promotora de Proyectos Company (2019). The transmission line route was selected based on using the route already provided by the Project's mine access road.

Power will be distributed from the main corani substation from a transformer to the distribution voltage of 13.8 kV. Both overhead and underground power lines will be used to distribute power throughout the plant site and administration area. A 13.8 kV transmission line with a length of approximately 13 km will connect the mine site to the residential camp site.

New antapata substation

This substation will be connected to the San Gabán II – Azángaro 138 kV transmission line at approximately 35 km from the San Gabán II Substation. The new Antapata substation will include the installation of five (5) switch yards at 138 kV and a transformer that will be used as energy supply for communities, 138 ±10x1%/22.9/10 kV – 6/8 MVA (ONAN/ONAF).

Transmission line

The link between the Antapata and Corani substations is made through a 138 kV – 60 Hz transmission line, approximately 29.4 km in length. The line includes:

- a metallic self-supporting lattice towers
- aluminum alloy conductors (AAAC) with a 600 mm^2 nominal section
- two ground wires (one steel galvanized wire and another OPGW-type wire with 24 optical fibers)
- glass insulators with a 146 mm spacing and a 280 mm diameter
- 445 mm leakage distance, and
- a 120 kN electromechanical rupture load.

The projected transmission line has the following technical characteristics:

- rated voltage: 138 kV
- maximum voltage: 145 kV
- system frequency: 60 Hz
- length: 29.4 km
- design power: 50 MW
- number of circuits: 01
- number of conductors / phase: 01
- circuit layout: triangular
- guard cable: 02, type OPGW and AoGo

- conductors: type AAAC
- structures: self-supporting towers of A ° G °
- insulators: tempered glass

The insulator chains in the suspension are made up of fifteen (14) insulators and the insulator chains in the grounding have sixteen (14) insulators and a reinforced concrete foundation, forming a set of four separate columns. Each column will be made of corrugated steel-reinforced concrete with a formulation of f'c=210 kg/cm^2.

Rights of way for the power line have been agreed with the local communities but have yet to be purchased from the individual land holders.

New Corani substation

This substation will require the installation of a 138/13.8 kV transformer. The 138kV side will have a power switch and the power transformer, be suitable for an installation altitude of 5000 masl, and will have an automatic regulation load of: 138 ±10x1%/13.8/4.16 kV - 50/50/17 MVA (ONAN) - 60/60//20 MVA (ONAF).

It will also have a YNyn0(d1) connection (or vector) group as well as a regulation board and include eight cells at 13.8 kV with an automatic capacitive reactive compensation bank of 12 MVAR.

Power will be distributed from the receiving substation at the Project site through underground duct banks to nearby major loads via local substations. Power distribution to all other areas, such as contact and non-contact water, camp, crushing station and truck shop, and to the mine, will be via overhead 13.8 kV power lines.

18.2.8 Communications system

The project off-site telecommunications will be served by a fiber optic telecommunications link to a data centre in Lima that will be carried on the 138 kV transmission line from the Antapata substation. Telephone and data communications including voice, data and internet communications will be provided for the mine site. The communications system will connect to a central communications centre, which will include a telephone/fax PBX and network servers for email, internet and data services. Other network servers to manage site operations and for data storage will also be located in the central communications centre, with the exception of the process servers which will be located at the processing facility. The mine site telephone system will link all essential areas of the site together, and through the satellite system, to outside of the project site.

The mine radio system will include one base station and a control-tower station at the mine from which all mining equipment and haul trucks will be dispatched and controlled, and a number of repeater stations will be installed. One station will provide coverage to the tailing area, and others are required to extend coverage throughout the mine site.

All vehicles will be equipped with radios and essential personnel will have hand-held radios.

18.3 Mine Waste Rock and Tailings Management Facilities

Detailed engineering of the main mine waste and filtered tailings deposit has been carried out by Anddes based on mine waste rock and filtered tailings co-disposal approach. The basic design criteria prepared by Anddes have been used as a reference.

The design of the deposit includes the foundation grading, removal of organic material, construction of the underdrain system, collection pond and the access ramps for hauling of waste rock and filtered tailings.

The design is based on local slopes of 2.2H:1V and global 3H:1V, in accordance with stability analyses and to design criteria. Local lifts will have 10 m height and setbacks of 7.5 m. NAG waste rock coming from stripping operations during 2022 will be used for conforming the base platform. The stacking ramps were designed with a total width of approximately 29 m and a maximum slope of 10%, based on the mine trucks operating parameters.

The mine waste and filtered tailings deposit is a large structure that will store mine waste and filtered tailings, reaching a maximum height of 325 m on a complex topography. The arrangement during the stacking of the deposit considers the construction of the starter platform with NAG mine stripping material, corresponding to the production of 2022 and 2023.

A description of criteria and considerations for mine waste and filtered tailings co-disposal plan is presented in Section 16.3. Co-disposal will start on January 2024 through to March 2038 in accordance with the mine plan for life of the project provided by BCM in October 2019 are foreseen. Co-disposal plan has been elaborated over a monthly basis up to year 2 and annually after year 2. Mine waste and filtered tailings co-disposal in horizontal modules and vertical raise will allow drainage of excess water from the tailings, if any, through the waste rock. If a single layer co-disposal is selected the blended material will provide also drainage for any excess water.

In addition to the NAG mine waste of the base platform, a layer of NAG mine waste with a minimum thickness of 25 m will placed around the whole deposit, including the foundation and the outer shell to provide an encapsulation of the PAG mine waste and tailings.

Geotechnical instrumentation must be installed in certain areas in order to monitor displacements, groundwater into the foundation, pore-water pressure into the co-disposed materials and seismic behavior of the facility through accelerographs, allowing the monitoring of geotechnical stability. Also, alert levels and operation and contingency manual, should be developed for the whole mine facilities.

19 Concentrate Market Studies and Contracts

19.1 Markets

The Corani project will produce two types of concentrates: a lead concentrate with high silver content and zinc concentrate with silver credits. Due to its high silver grade, the lead concentrate will be attractive to smelters with recovery capabilities, while the zinc concentrate will be more marketable given its characteristics. Most of the concentrates are likely to be sold under medium and long-term agreements mainly to Asian refineries/smelters. Sales to the local and European markets are also an option.

19.2 Contracts

There are no established contracts for the sale of concentrates currently in place for this project.

19.3 Concentrate Transport Logistics

Lead concentrate is considered a hazardous material (IMO) therefore it will be shipped in standard 20 ft shipping containers lined with plastic films from the mine site to the Peruvian port of Matarani in Arequipa at an estimated inland cost of $ $70.96/t (wet), the containers will then be exported as is to final destination. Zinc concentrate will be bulk shipped from the mine site to the port of Matarani at an estimated inland cost of $59.00/t (wet). Zinc concentrates will then be exported in bulk to final destination.

Both estimated costs were quoted by BLB Advisory as part of the “Corani Marketing and Sale Logistics Study”, dated September 2019

19.3.1 Concentrate transport insurance

Insurance will be applied to the provisional invoice value of the concentrate to cover transport from the mine site to the smelter. The ratio considered for this report is 0.10% of the provisional invoice value, with costs of $2.49/t (wet) for lead and $0.99/t (wet) for zinc concentrates.

19.3.2 Owner’s representation

Most concentrate sales are finalized with weight, moisture content and assays at final destination at the discharge port, where normally a third-party surveyor is appointed. The cost of the surveyor can vary from port, country and client. For the purpose of this report the cost of the surveyor is $1.00/t (dry) for each of the lead and zinc concentrates.

19.3.3 Transportation options

Transportation of lead concentrates will be by container trucks to the Port of Matarani. The container trucks will meet all required environmental regulations for hazardous materials. The first five years of lead concentrate production will be approximately 130,000 t/y (wet) dropping to an average of 85,000 t/y (wet) for the remaining 10 years of the mine life.

Matarani port does not have permits in place for handling bulk lead concentrates. Therefore, the lead concentrate will be containerized at mine site and receive customs approval before shipment to a final destination. For this report it is assumed that the concentrate will be sold to the Asian market, the total freight cost including overland freight, ocean freight, insurance and handling fees for lead concentrates shipped from the Corani plant to the Asian port of Shanghai is estimated to be $ $137.71/t (wet).

Transportation of zinc concentrates will be in bulk by enclosed trucks to the Port of Matarani. The trucks will meet all required environmental authorizations. The first five years of zinc concentrate production will be approximately 100,000 t/y (wet), dropping to an average of 63,000 t/y (wet) for the remaining 10 years of the mine life.

For this report, it is assumed that the concentrate will be sold to the Asian market, to Shanghai Port (other alternatives include Peru and/or Europe).

The total freight cost including overland freight, ocean freight, insurance and handling fees for zinc concentrates shipped from the Corani plant to the smelter is $122.28/t (wet). Shipping rates for container and bulk concentrates are currently very similar due to market and port conditions, so there is no differentiation made in the alternatives.

19.3.4 Smelter terms

Smelter terms and penalties were supported by an independent market analysis (BLB Advisory, 2019). The data from this study were used to evaluate the revenue, charges, premiums, and penalties that are presented in detail below.

19.3.5 Sale of lead and zinc concentrates

Every smelter has different rates for impurities depending on the normal feed. Higher levels of impurities will decrease the value of concentrates delivered to the smelter.

19.3.6 Zinc treatment cost and premiums

The consensus is that the oversupply in the zinc concentrate market will remain for the foreseeable future. Despite lower mine production growth due to current market prices and high Treatment Charges (TCs), mine supply growth will outstrip smelter output growth. Therefore, a sizeable build up in concentrate stocks is anticipated over the period to 2023. This will potentially lead to higher TCs in the near term. The afore mentioned market surpluses will pressure prices lower until 2022 and should bring about the mine cuts necessary to restore the market to balance and deplete existing stocks. Therefore, the outlook beyond 2022 should improve for prices and potentially for a reduction in TCs. BLB Advisory recommended using zinc TCs range of $230 - $270.

19.3.7 Lead treatment cost and premiums

Overall the lead concentrate market is in relative balance and, subject to normality on the smelter side, shouldn’t see big shifts of the Treatment Charges (TCs). However, we have not found forecast periods that get us close to Corani´s projected 2023 production start-up, and thus do not have a clear market view of the global market for that time. Considering the yearly global mine production of more than 5 Mt, Corani´s circa 120,000 t should not have a profound impact in the overall market. However, Corani’s production should be very attractive to some strategic buyers because the new volume represents a secure source of long-term supply with high silver content. BLB Advisory recommended using a lead TCs range of $ 100 - $130/t (dry) and Silver Refining Charges (RCs) range of $0.8 - $1.10/oz payable.

The price of lead has been fairly constant over the last several years and appears to be stable for the next few years. Table 19-1 shows the assays for materials found in each type of concentrate for this project.

Table 19-1: Concentrate characteristics

Element	Unit	Zinc Concentrate	Lead Concentrate
Zn	%	52.8	6.62
Ag	g/t	423	2733
As	%	0.08	0.21
Sb	%	0.19	1.9
Bi	%	0.0001	0.004
Cd	%	0.3393	0.1277
Hg	ppm	80.2	61
SiO_2	%	4.9	-
Fe	%	5.0	9.9
Cu	%	0.37	1.29
Pb	%	3.91	49.1
Al_2O_3	g/t	5135	-
Ba	g/t	304	1430
Ca	g/t	1400	546
Mg	g/t	400	300
Mn	g/t	200	1500
Na	g/t	110	-
P	g/t	284	-
Se	g/t	<30	-
Sn	g/t	58	-

19.4 Chromium Assays

The chromium assays in concentrates were elevated and as high as 1.2% in some concentrates. These levels were obviously anomalous, ranging from 20 to 12 600 ppm in Pb concentrate and from 140 to 380 ppm in Zn concentrate.

BCM checked the Cr assays from chemical analysis using QEMSCAN. The Cr assays from the stainless steel content correlated with the Cr chemical assays of the lead concentrate.

Table 19-2 Chromium assays in lead concentrate

Mineral Name	Assays (%)	
	Cr (QEMSCAN)	Cr (Chemical)
BL 439 BL250-84 Pb Con C/D/E	0.07	0.06
BL 439 BL250-85 Pb Con C/D/E	1.30	1.26
BL 439 BL250-109 Pb Con C/D/E	0.28	0.28

Chromium contamination related to stainless steel content is likely to be sourced from the laboratory equipment, either in laboratory testwork, or more likely during sample preparation.

The desired level of Cr in zinc concentrate is approximately 10 ppm with up to 50 ppm Cr considered “acceptable” and some smelters having no stated limit.

19.5 Sale of Concentrates

19.5.1 Zinc concentrates

Typical Terms for the sale of zinc concentrates in the Far East as reported by BLB Advisory are as follows:

- Payable Metals:
  - Zinc: 85 percent (subject to a minimum deduction of 8.0 units).
  - Silver: Deduct 3.0 ounces per dry metric tonne (dmt) and pay for 70% of the balance.
- Treatment Charge:
  - $ 231.3/DMT Cost Insurance and Freight Free Out (CIF FO) Main Asian Ports.
- Escalator/De-escalator: Mechanism of price-participation typical in zinc contracts. When the LME price rises, this compensates the buyer with an increase in the TCs. When the LME price falls, it compensates the seller with a reduction in the TCs.
- Zinc base price of $2,400 /t and shall be increased/decreased for each $1.00/t of variance above or below the base per tonne as follows:
  - TC: $231.3 @ base $2,400 +10% / -3% per dollar above/below base
- Penalties:
  - Arsenic $2.00/dmt per 0.1% over 0.25%
  - Cadmium $1.00/dmt per 0.1% over 0.4%
  - Lead $1.00/dmt per 1% over 4%
  - Mercury $1.00/dmt per 10 ppm over 40 ppm
  - Antimony $2.00/dmt per 0.1% over 0.25%
  - Bismuth $1.50/dmt per 0.01% over 0.15%
  - Silica $3.00/dmt per 0.1% over 3%

19.5.2 Lead concentrates

Typical Terms for the sale of Lead concentrates as reported by BLB Advisory are as follows:

- Payable Metals
  - Lead: 95 percent (subject to a minimum deduction of 3 units)
  - Silver: 95 percent (minimum deduction 50 grams per DMT)
- Treatment Charge:
  - $150 per DMT
- Refining Charge:
  - Silver, $1.00 per payable oz
- Penalties
  - Arsenic $1.50/dmt per 0.10% over 0.25%
  - Bismuth $2.00/dmt per 0.01% over 0.15%
  - Mercury $2.00/dmt per 10 ppm over 60 ppm
  - Antimony $2.00/dmt per 0.10% over 0.25%

19.5.3 Metal prices for study

The metals prices used for this report were as listed in Table 19-3

Table 19-3: Metals prices used for study

Metals Prices	
Zinc	$1.10/lb
Lead	$0.95/lb
Silver	$18.00/oz

Concerns about slowing economic growth and its medium to longer term implications have been the central theme in current forecasts and do not help price scenarios across the board on the base metals. Moreover, the long-lead nature of Corani´s construction and production profile make it difficult to forecast prices.

In zinc, from the highs of around $ 3,500/t in 2017-2018, it has seen a dramatic fall in prices in the last few months. Decreasing prices are expected to continue as metal supplies improve and surplus markets become the norm, additionally the scale and speed of the concentrates and refined metal stocks build will probably continue to pressure prices lower during 2020 and 2021. This stock build will put prices under pressure to reach lows of $ 1,850-1875/t in 2022 and bring about the necessary mine cuts to restore the market to balance, before finding some price support in 2023.

After having cyclical highs above $ 2600/t in the 2017-18 period, lead prices fell below $ 2200/t and haven't recovered. Historically low levels of stocks of refined metal are likely to cause spikes on the upside, but over the medium term the forecast for prices is to continue lower given the potential oversupply in the market and potential slowdown in the world's largest economies.

After a long period of depressed prices, Silver has begun to take on a life of its own fueled by investors looking for safe haven in precious metals. Uncertainty in the global economy, as well as socio-political unrest, will continue to attract investors to gold and silver as a hedge against currency volatility. More importantly, the silver market seems to have also turned a corner on the industrial demand side. Industrial demand for silver has accounted for more than half of the metal's consumption over the most recent five years. These factors support near term forecast prices above $19/oz with higher prices expected in the longer term.

BLB Advisory recommended using a forward price range for zinc of $ 2400 - 2500/t, lead of $2050 - $2150/t, and silver of $17.50 - $19.00/oz.

19.5.4 Smelter and refinery return factors

The lead and zinc concentrates will be sold to smelters/refineries for final processing. Smelter treatment and refining charges will be negotiated at the time of the finalization of the sales agreements. The smelter charges used in the financial model are presented in Table 22-2 and Table 22-3 for lead and zinc concentrates respectively

20 Environmental Studies, Permitting and Social or Community Impact

This section presents the following:

- a summary of existing studies
- known permitting considerations
- a brief description of the strategies for environmental management during operations, reclamation and closure
- socioeconomic and community considerations

Principal environmental risks associated with this type of project fall into the following categories:

- potential risks to air quality from dust and gas emissions
- potential degradation of surface and groundwater quality
- potential changes to surface and groundwater quantity
- visual impacts due to the creation of pits, mine waste disposal facilities, roads, and other mine workings
- permanent changes to land use resulting from mining activities

The approved Environmental and Social Impact Assessment (ESIA) quantifies the magnitude, extent, and mitigation of risks and potential impacts related to an older project configuration that utilized a conventional wet tailings facility and its associated water supply infrastructure. Modifications to the project design have been incorporated into two ITS modifications, though the final optimizations will be permitted in a third ITS modification of the ESIA that is currently being planned. Project optimizations have resulted in: a) a reduced project footprint, b) a reduction in water consumption, and c) other changes that further reduce environmental impacts associated with project development relative to the original design. In a number of cases, the development of the Project is anticipated to improve existing environmental conditions, in part due to historical environmental liabilities from historic mining activities.

Peruvian Law 28090 regulates the obligations and procedures mine owners must follow in relation to mine closure, and requires that a mine closure plan be approved and financial guarantee for the cost of implementation be established. The plan must describe the rehabilitation methods and their costs for the operation, closure, and post-closure phases. The plan must allow for progressive closure, with mine owners reporting semi-annually to the Ministry of Energy and Mines on progress with the implementation of the approved plan. As required under Peruvian regulations, the ESIA included a conceptual closure plan, which was updated by Walsh in 2014 and subsequently approved. An updated closure plan was completed and approved in 2018 (Amphos 21, 2018).

he development of closure concepts for this study has considered International Finance (IFC) guidelines and industry standards in addition to Peruvian regulatory requirements. The general approach to mine reclamation and closure developed at this time is described below. The estimated cost (based on the Preliminary Closure Plan) has been incorporated into the most recent cost estimate for the Project.

20.1 Environmental Baseline Studies

Characterization of baseline conditions has been ongoing since July 2009. Extensive site characterization was conducted in 2011 and 2012 as part of the development of the original ESIA (AMEC, 2012), though environmental monitoring has continued since then and data through 2019 have been reviewed.

20.1.1 Summary of air, noise, groundwater and surface water studies

Below is a summary of the results from the currently-published baseline studies for air, noise, and water:

- Air and noise monitoring has been completed at 12 locations to support the ESIA. Monitoring continues at two locations for air quality. The air quality results reviewed, including for 2017 through 2019, were below the environmental limits (ECAs) set by the national environmental standard for air quality (D.S. No. 003-2017-MINAM), reflecting the absence of significant air quality concerns in the baseline. This includes for PM2.5, PM10, NO_2, CO, H_2S, As, and Pb.
- Noise measurements have been evaluated at two locations near populated areas. The results reviewed were below the noise ECA specified in the Peruvian environmental standards for residential areas.
- Groundwater appears to be located in shallow aquifers comprised of alluvial, glacial, and aeolian deposits that lie over low-permeability basement rocks. Results from the site investigation indicated that the shallow aquifers have high storage but moderate to low hydraulic conductivity. Little evidence exists for a conductive and extensive hard-rock aquifer. In general, it appears that groundwater resources in the Project area are not sufficiently large to be useful for agricultural, domestic, or industrial use. However, the protection of the groundwater resources from impact is part of the focus of the ESIA and future environmental planning.
- Surface water samples exhibited highly to slightly acidic characteristics during sampling events conducted throughout the year. These acidic conditions are related to naturally-occurring oxidation of mineralized rocks exposed at the site and from areas previously disturbed by historic mining activities. Several metal concentrations exceeded the national environmental water standards. Similarly, some metal concentrations measured in sediments exceeded Canadian Environmental Quality Guidelines for sediments (no Peruvian guidelines exist). As a result, the major drainages leaving the Project area do not currently meet national environmental water standards.

20.1.2 Summary of biological studies

The Project has completed biological baseline studies that describe the ecosystem of the site and the species abundance, richness, biodiversity, and species endemism. Initial characterization was completed for both aquatic and terrestrial species and monitoring has continued until the present date for aquatic plankton, periphyton, benthic macroinvertebrates, and for terrestrial flora, birds, mammals, and herpetofauna. The specific flora and fauna species that have been documented to occur on the site have been cross- referenced with threatened species lists (international and national). A brief summary of the studies include:

- A total of 401 flora species have been registered between the ESIA and subsequent monitoring report. Fourteen species are listed as Peruvian endemics, with three listed as endemic to the Department of Puno. Three species, *Stangea wandae, Ephedra rupestris,* and *Nototriche longituba* are listed as critically endangered (CR) on the national threatened species list but no species are listed on the international IUCN list.
- A total of 82 bird species have been identified, with only one, the Andean condor (*Vultur gryphus*) listed as endangered (EN) on both the national list of conservation species and the IUCN redlist.
- Fourteen mammal species, two species of amphibian and a single species of reptile have also been identified.

20.1.3 Summary of geochemical studies

A geochemical waste characterization program was developed to assess the acid rock drainage (ARD) behavior of, and potential leaching of contained metals from, all mine wastes associated with the Project. This program included static tests, LECO Furnace total sulfur and total carbon assays, and onsite and laboratory kinetic cell tests, as presented in the ESIA (AMEC, 2012).

These were combined with geologic and metallurgical characterization of lithologies and material types. The conclusions from this work included the following:

- The geochemistry of the waste rock will be dominated by certain mineralization types; in particular, mineralized lithic tuff with fine black sulfides (FBS) and mineralized tuff with pyrite and marcasite (PM).
- Whole rock analysis of waste rock samples indicated that several metals of environmental and processing concern exist at high levels; synthetic precipitation leaching procedure testing suggested that many of these metals are readily leachable.
- The kinetic tests showed that many waste types were acid generating, though the behavior among certain mineralization types was mixed.
- The ABA indicator testing of the tailings suggested that the tailings would be potentially acid generating (PAG) material due to the presence of residual sulfides.

Additional geochemical humidity cell kinetic tests were initiated in 2014 and 2017 by Amphos 21 (Amphos 21, 2014 and 2017). Initial results from these tests confirmed previous kinetic cell testing by demonstrating that the PMT is non-acid generating due to its very low (<0.1 wt.% S) sulfide content, and that sulfidized waste and tailings often produces acidic leachate with high sulphate and metal concentrations, specially Mn, Fe, Pb, Zn, Cd, Cu and As.

20.2 Permitting Considerations

Refer to Section 4.4 for an explanation of the permits required to execute the Project.

20.3 Environmental Management During Operations, Reclamation, and Closure

The Project has been designed to incorporate the implementation of best environmental management practices to minimize impacts during construction, operations and at closure. This includes minimizing the disturbance footprint, maximizing the reuse of water, and conducting progressive reclamation/closure during the operational period. More detail on the potential impacts associated with the project development can be found in the ESIA and ITS documents, as well as the 2018 Actualization of the site Closure Plan.

20.3.1 Environmental management objectives

The operational environmental management objectives are to identify potential and viable measures to mitigate environmental impacts that can be implemented during the construction, operation, closure, and post-closure periods. The management plans include mitigation measures that are directed at eliminating or reducing the potential long-term impacts from mining operations and to reduce long-term environmental liabilities. The baseline conditions at the Project site show a degree of degradation of water quality in the general vicinity of the mineralized zones, largely due to naturally-occurring oxidation of sulfide-bearing materials that are associated with the mineralized orebody. Baseline conditions show acidic pH conditions and an increase in dissolved metals and salts in surface waters leaving the Project area. Furthermore, due to the historical mining activity within the main Corani basin, there are documented existing environmental liabilities that are associated with the presence of underground mine workings, areas of waste rock, and past mine tailings; all of which are a source of acid rock drainage (ARD). The Project design includes reclamation of many of these existing liabilities as part of the development of the Project.

The objectives for reclamation and closure of the Project include returning the environment to a condition that is consistent with the baseline conditions, and where possible, to an improved environmental state. The closure measures are not intended to change the naturally occurring conditions at the site, nor are they intended to mitigate the effects of historic mining within the project area, apart from where future mining activity can be used to implement mitigation measures in the course of the proposed mining, processing, and mine waste management defined for the Project. Historic mine areas will be closed/mined out in the pit areas and areas directly in the footprint of the waste and tailings co-disposal facility. Historic mine workings that are not directly affected by mining activity, but are still within the catchment area of the mine's water management facilities and are contributing ARD to the facilities will also be closed. Moreover, it should be noted that although a large number of the historic mining facilities contributing to ARD will be removed during Corani mine operations and closure, a significant portion of the undisturbed bedrock at the site is naturally acid-generating, and will continue to produce ARD during operations and closure.

The following activities performed during operations, reclamation and closure are intended to mitigate and minimize generation of ARD:

- encapsulation of potentially acid generating (PAG) materials within the co-disposal facility for tailings and mine waste
- extraction of legacy mine waste and disposal within the PAG encapsulation cells inside the co-disposal facility
- use of non-acid generating (NAG) waste materials to cover the backfilled waste and tailing in the pits to support the water saturation and encapsulation of PAG materials within the closed pits
- concurrent placement, as possible, of reclamation covers over facilities during operations
- segregation of contact waters from non-contact waters and the use of contact water, and a portion of naturally-acidified runoff from the watershed, as plant make-up water
- consumption of all contact water by the processing plant.

In the following sections, the overall site conditions that affect the selection and implementation of the reclamation and closure measures are identified. The reclamation and closure considerations for individual project components and facilities are then presented. The general items considered in estimation of costs for the reclamation, closure, and post-closure periods are also identified. A large proportion of the reclamation and closure measures will be completed as part of progressive closure during the operational period of the mine. Additional information regarding the cost estimation for specific items is provided in the relevant sections of this document.

20.3.2 General site conditions relevant to environmental management

Some areas of the project experience naturally-occurring ARD. Consequently, in these areas, naturally occurring growth media are limited or absent. Key areas without soils capable of sustaining vegetation include:

- the valley that will contain the co-disposal facility and portions of the plant complex
- large portions of the mountain slopes in and around orebody
- higher elevations of the project site and areas with exposed bedrock.

The areas not capable of sustaining vegetation are readily apparent from the color aerial photo shown in Figure 20-1. It is important to note that many of these areas will not sustain vegetation post-closure.

The reclamation and closure measures identified in the following subsections have been developed to minimize post-closure management requirements for the areas of disturbance related to mining activities. However, as described further below, it is anticipated that some long--term post-closure management and monitoring will be required for a five year period following the completion of mine reclamation and closure.

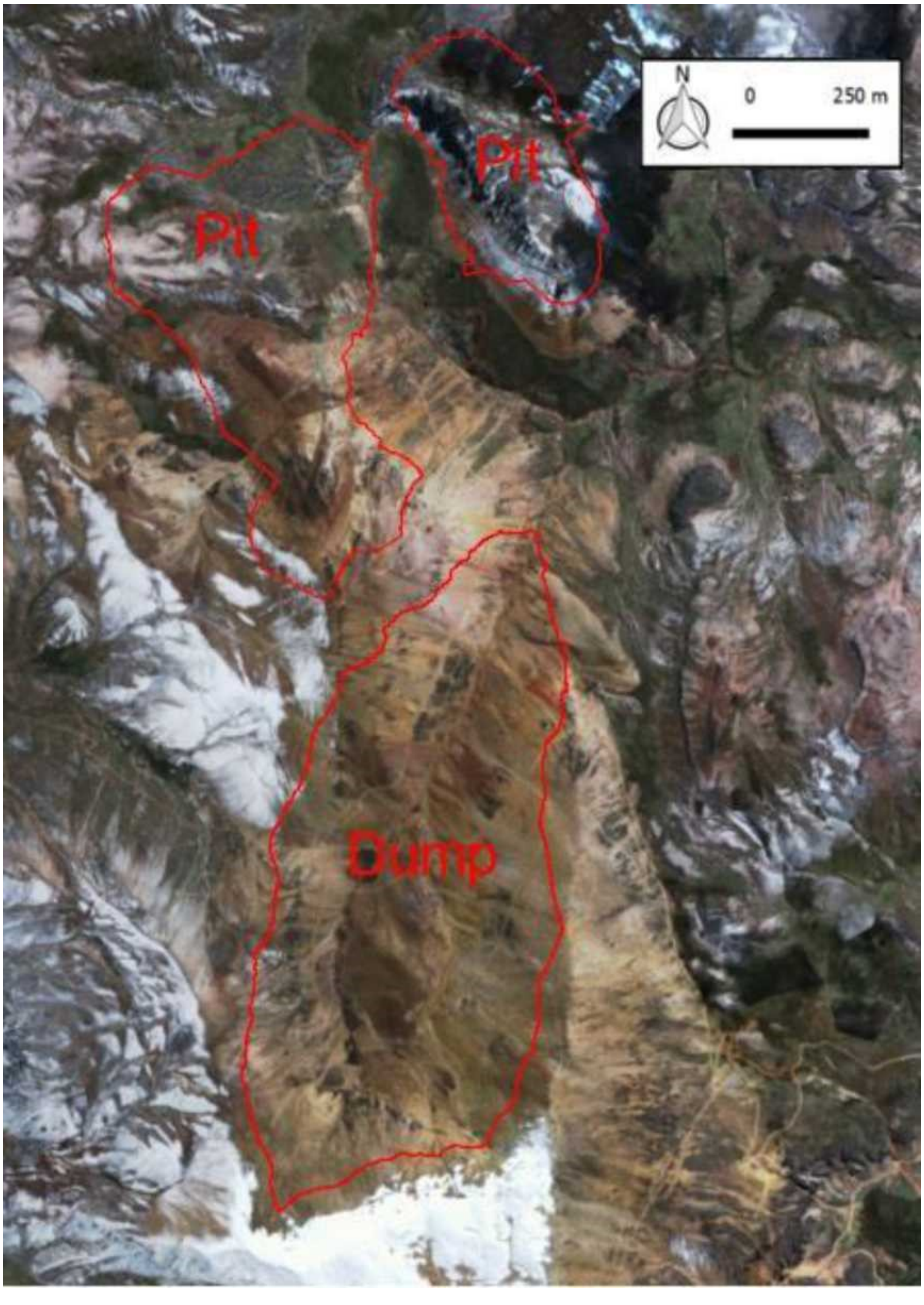


Figure 20-1: Aerial photograph of the Corani site, current conditions

20.3.3 Project components and facilities

The following sections describe the environmental management plans for specific project components and facilities. Additional design details can be found in Section 18.3. The main project components and facilities with environmental management and/or reclamation considerations include the following:

- the Este, Minas and Main Corani open pit areas
- waste rock and tailings co-disposal facility
- backfill of mine pits with tailing and waste rock
- the processing plant, crushing facilities, tailings dewatering plant, and related infrastructure
- the surface water management infrastructure and plant contact water pond, and the plant non-contact water pond
- soil stockpiles areas.

For the purpose of this review, not all aspects of environmental management, reclamation and closure measures are presented, but the focus is on the major aspects that drive environmental cost considerations.

The project configuration at the end of mining, just prior to closure, is shown in Figure 20-2, which also shows the preliminary closure plan concepts described in the following sections.

20.3.4 Open pits

The most current mine plan has 15 years of active mining. Upon completion of mining, the Este, Minas, and Main pit areas will form a discontinuous horseshoe-shaped excavation around the east, north and west sides of the Corani valley.

Bofedal soils

Since the 2015 FS, certain facilities have been redesigned, which has resulted in a reduction of impacts to bofedal soils. However, localized disturbance may occur.

Localized stockpiles of bofedal material will be established for use in reclamation and closure activities. Segregation of the materials will be performed to separate organic material from the rest of the excavated soils and unconsolidated materials. Collection and storage of these materials will be maximized to the extent practical.

Pit water management

Several natural drainages direct surface water runoff to the fully developed pit areas. During operations, this water will be diverted around the pit or routed into the channel that is retained between the Este and Minas pits. This water will be classified as non-contact and can be discharged to the environment or utilized for needed make-up water. Surface water runoff generated within the footprint of the pits and groundwater inflow to the pit will be collected in sumps and pumped to the Contact Water Pond for use as process make-up water.

The most recent closure plan provides details on which of the operational water management channels will be removed as part of closure, and which will remain. Some additional channels will also need to be developed as part of closure. During closure, a combination of backfill placement and covers will be utilized to prevent the development of ARD and water that will require long-term management. This is discussed further below.

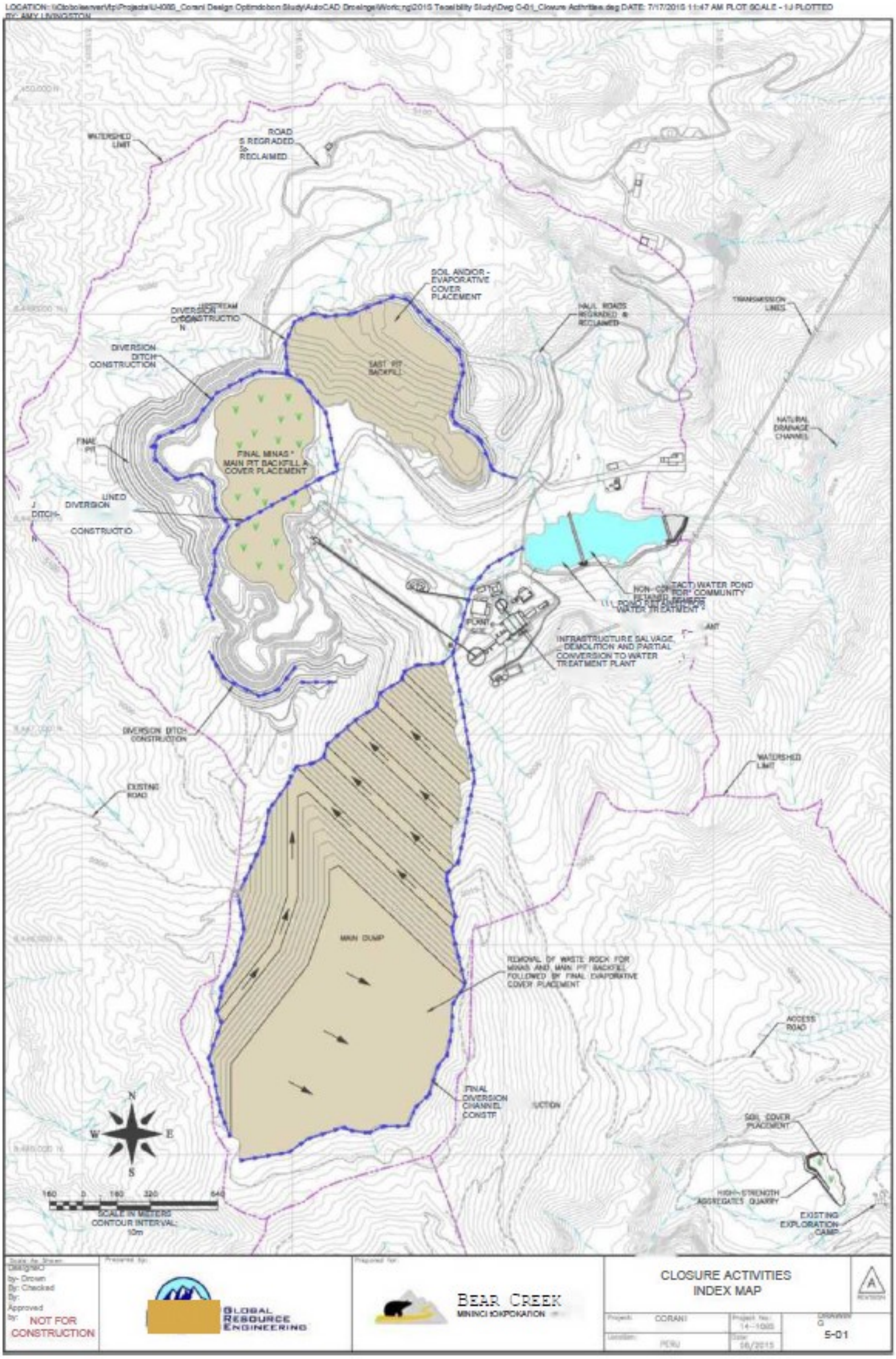


Figure 20-2: Closure activities index map

Pit water management

Several natural drainages direct surface water runoff to the fully developed pit areas. During operations, this water will be diverted around the pit or routed into the channel that is retained between the Este and Minas pits. This water will be classified as non-contact and can be discharged to the environment or utilized for needed make-up water. Surface water runoff generated within the footprint of the pits and groundwater inflow to the pit will be collected in sumps and pumped to the Contact Water Pond for use as process make-up water.

The most recent closure plan provides details on which of the operational water management channels will be removed as part of closure, and which will remain. Some additional channels will also need to be developed as part of closure. During closure, a combination of backfill placement and covers will be utilized to prevent the development of ARD and water that will require long-term management. This is discussed further below.

Backfill

The Corani pits will be backfilled for the purpose of storing waste rock and tailings, and also to support the long term management of pit waters, including maintaining PAG materials under saturated conditions and the use of a NAG cover to prevent ARD generation and water treatment. The process starts during progressive closure and is completed as part of final closure activities.

Backfilling of the Este pit will take place during mine operations using tailings and waste rock generated by ongoing mining of the Minas and Main pits during that time. Tailings and waste rock will be placed in designated backfill zones. After backfilling has been completed, an engineered soil cover will be installed over the backfilled area. The cover will consist of a layer of NAG material overlain by topsoil, which will limit the amount of precipitation water that can enter the backfill.

Mining in both Minas and Main pits prevents backfilling of this pit area until the end of the mine life. Backfilling of the Minas/Main pit area will occur over the course of several months directly following completion of mining. The area will be backfilled by importing waste rock and tailings temporarily placed in the co-disposal facility. Backfill will be placed to a level high enough to ensure that the final backfilled surface remains above the water table after hydrologic stabilization occurs during the post-closure period.

After this material has been placed, available water will be diverted to artificially recharge the backfill, saturating the PAG material as quickly as possible, which limits the time during which the backfill and the pit walls are exposed to atmospheric oxygen. Artificial recharge of the Este backfill is not required due to the low hydraulic conductivity of the tailings backfill.

The final Minas/Main backfill surface will be graded to resemble the topography of the adjoining lower bofedal area, and to merge into that topography. Stockpiled topsoil will be used to cover the backfill, with the intention of promoting gradual natural establishment of a vegetated environment that is an extension of the lower bofedal. A preliminary model of the cover was developed using a numerical model to evaluate the performance of various engineered cover configurations. Based upon the model results, direct infiltration through the cover is expected to be reduced to a minimal level following cover installation.

The developed hydrogeological model for this closure system demonstrates that the projected water chemistry will meet all legal criteria and will only require post-closure monitoring.

20.3.5 Tailing and mine waste co-disposal facility

The tailing and waste rock facility is described in Section 18; this section covers the reclamation of this facility.

During operations, PAG materials will be encapsulated into the interior portion of the facility to reduce the availability of water and air. The slopes will be progressively developed with an engineered cover that utilizes NAG materials and then reclaimed with the placement of previously stockpiled natural soils starting from the base of the dump and continuedly upstream until the final design platform. Runoff from these areas will be routed around the active dump areas and released downstream to natural drainages. Following the placement of the reclamation cover, runoff from the reclaimed surface will be directed to toe ditches and routed beyond the facility.

The current condition of the ground in Quebrada Muerta beneath the co-disposal facility is un-vegetated and sterile. However, for closure, this facility will be re-vegetated.

20.3.6 Plant facilities and related infrastructure

During operations, runoff generated in the area of the plant footprint is considered contact water and will be diverted to a sump and pumped to the contact water pond for use as makeup water. At the end of the operating life of the plant, it will be decommissioned and demolished. Where viable, equipment will be salvaged, sold, or transported offsite as scrap material. A small amount of infrastructure may be left in place for post-closure use. All other structures will be demolished, and the demolition debris buried in an on-site landfill. This would include the majority of the plant buildings, truck shop, explosives magazine, crushing system, warehouses, reagent storage areas, etc. All major pipelines will be either salvaged or, if buried, left in place.

Concrete foundations for the plant will be broken up and buried in place in most areas. The degree to which they are broken up will be determined by the ability to create a natural- appearing post-mining topography with a natural drainage pattern. Leaving foundations in place may be considered where this can be accomplished without breakage. A suitable soil cover will be placed over these areas to facilitate the function and appearance of baseline conditions.

Once the off-site operational camp is no longer required, the camp will be converted to an alternative beneficial use by a future custodian and/or local community representative. The main access road and mine roads that are required to provide access to the site for maintenance and monitoring during post-closure, will be reduced in size to that of similar local roads in the project region. A portion of the internal roads will be removed by ripping and reclaimed to conditions similar to the surrounding area; the remainder of the roads will be converted to community use.

20.3.7 Site wide water balance

A project-wide water balance was developed for the operational mine life. The water balance was set to achieve the following requirements:

- to ascertain that sufficient water exists to run the process plant, even during extreme dry conditions
- to check that contact water will be consumed by the demands of operation, thus alleviating the need of water treatment and discharge
- to ensure that sufficient non-contact water exists to maintain environmental baseflow conditions in the chacaconiza drainage basin downstream the project site.

The water balance considers the inflows and outflows for each project sector over the operational mine life span. In the water balance these flows vary over the time on a monthly basis to determine flows of contact and non-contact water coming to ponds. Variation in flows are due to growth of facilities over time. For example, as the pit increases in size, the amount of contact water increases. In contrast, following the ramp-up period, the water demand to process plant remains steady over the LOM.

In accordance with industry best-practice, a probabilistic water balance has been developed. The model replicate range of probable conditions that may exist during operations. The model varies climate input parameters (as rates of precipitation, snowfall and evaporation) based on a statistical analysis of historic data. The 95% wet and the 5% dry conditions were used to set the main criteria for the mine water management.

The database used for this analysis comes from a site weather station that has been recording hourly data since December 2008, as well as flume stream gauging stations that have been recording hourly data at the site since 2012. Site-specific runoff coefficients were calculated using the combined data record available from 2012 to 2018. Incorporation of the site-specific data measurements offers a significant improvement over previous studies, which relied on literature based on assumptions for runoff coefficients. Groundwater inflows to the mine pits were estimated based on a transient FEFLOW groundwater model reflecting the mine plan and pit backfilling schedule.

During project development and operations, a surface water management system will be developed (see Section 18) to route runoff from areas of disturbance, groundwater collected in the pit, and seepage from the mine waste and filtered tailings deposit to the contact water pond and then to the process plant for use as makeup water. The capacity of the collection pond is approximately 830,000 m^3, and will consist of two separate cells: a non-contact water cell (in the downstream portion, with a volume of 572,000 m^3) and a contact water cell (in the upstream portion, with a volume of 257,000 m^3). Both contact water and non-contact water stored in the ponds will be used to supply the plant during the dry season, as required.Additionally to the contact and non-contact cells of the water pond, at the east side of the water pond it´s located a water table pond with a capacity of 11,000 m^3 for storage the flows from the under drainage system for subsequent monitoring and evaluation thereof.

Environmental baseflow in the Chacaconiza drainage will be maintained during project development subjected to the natural availability of flow in the catchment area. Concepts related to the surface water management plan are presented in more detail in Section 18.2.6.

The total process water demand is discussed in Section 18.2.6. The process plant makeup water demand is satisfied from water available in the watershed, with the following water supply priority order: contact water (runoff and baseflow from disturbed areas and pit inflow), non-contact water (runoff and baseflow from undisturbed areas in the catchment), water from the upstream cell of the Plant Water Pond (contact water and stored non-contact water), and finally, water from the downstream cell of the Plant Water Pond (stored non-contact water). Once the plant demand is met, the excess non-contact water from the remaining area(s) is either stored in the Plant Water Pond, stored in pit or released downstream. Under the 95% wet condition (see above), all contact water is consumed by the plant or temporarily stored in the upstream cell of the Plant Water Pond and is not discharged to the environment. Conversely, the combined pond contains sufficient water to supply the plant under the 5% dry condition. In Figure 20-3 and Figure 20-4 storage over time of the Contact Water Pond and Non-Contact Water Pond is presented.

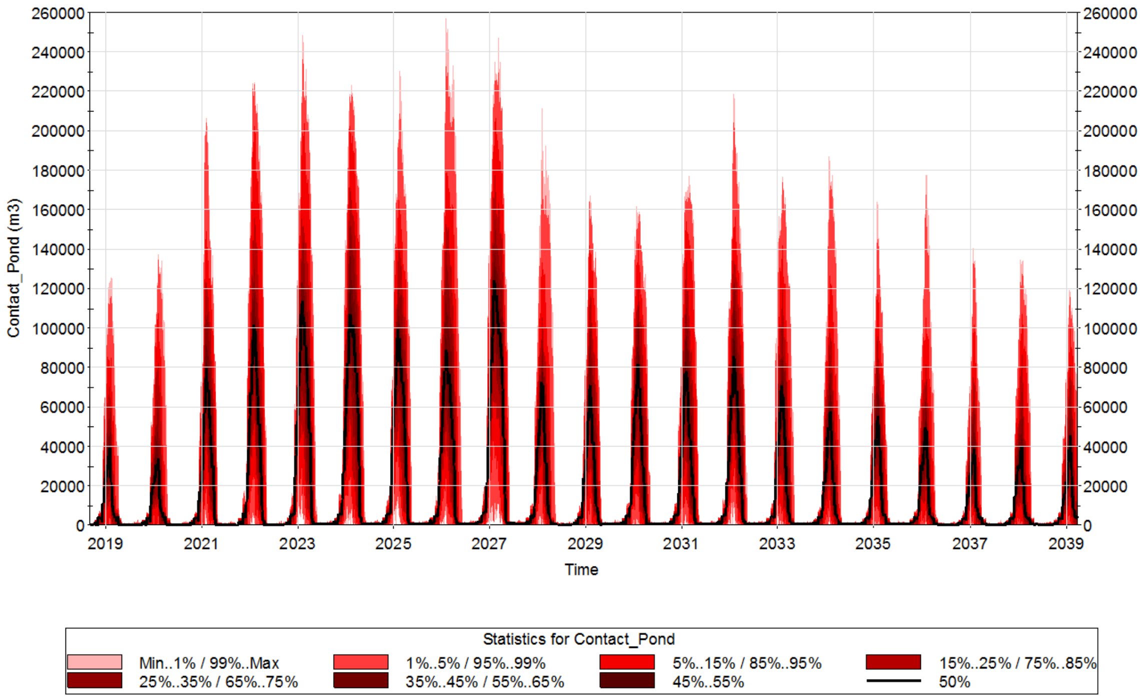


Figure 20-3: Contact water pond storage

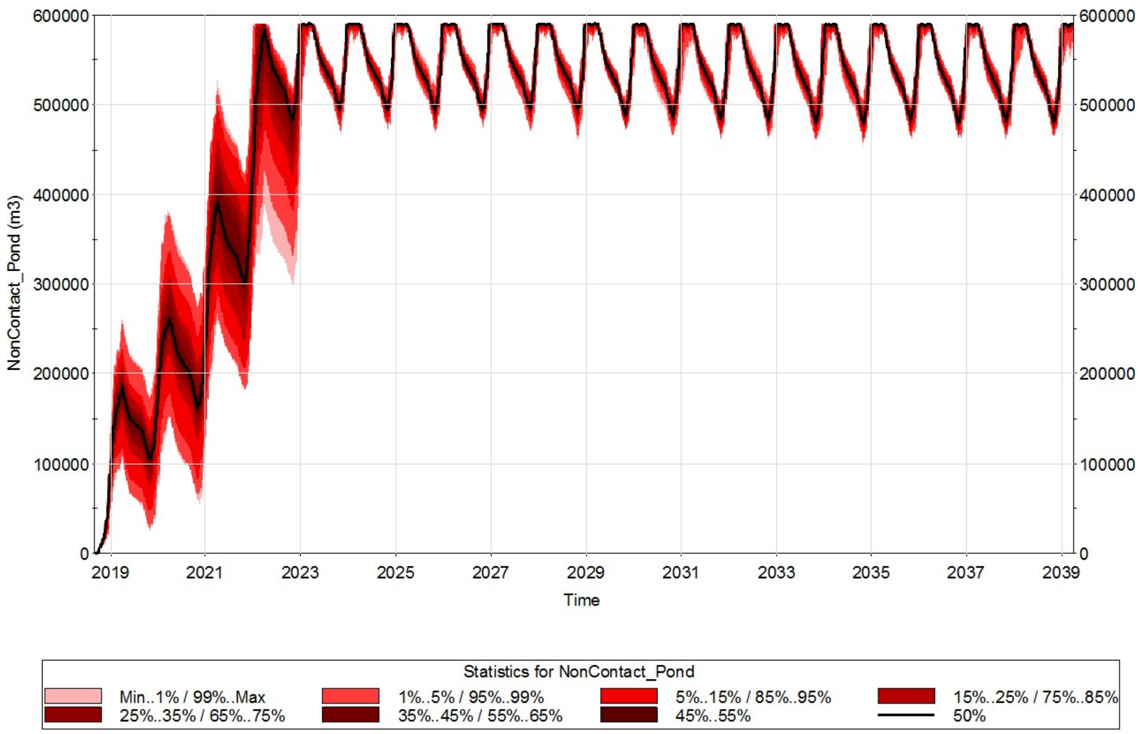


Figure 20-4: Non-contact water pond storage

In summary, the site water balance states that the water available in the watershed greatly exceeds what is required for operations. This ensures that the non-contact water pond is never dry, and that environmental baseflow can be maintained, even under extreme conditions. Conversely, the project has sufficient consumptive requirements and storage capacity to consume all contact water.

20.3.8 Closure Phase Water Management

The contact water cell of the plant water pond will be used during closure as a flow equalization pond for the water treatment plant. Use of the pond will be used in the first year of closure to store contact water while the process plant is converted into an ARD treatment plant if necessary.

At closure, the non-contact pond could constitute a potential resource for the downstream communities. Transfer of ownership and operation of the pond to a group of the local communities or government organizations would be performed following the implementation of reclamation and closure measures.

Upon closure, permanent water conveyance structures will be retrofitted as required for minimum maintenance. This may include the installation of new riprap channels and passive sediment control structures that will be planned according to engineering studies.

20.3.9 Regrading Plan

The general plan for final re-grading and re-vegetation of facilities includes the following:

- all visibly contaminated soils will be excavated and treated in an on-site soil land farm
- all compacted soils will be ripped and/or tilled to reduce soil compaction and improve drainage
- impacted areas will receive a soil cover, where cover existed prior to development
- impacted areas will be re-vegetated with native species, where capable of supporting vegetation
- erosion-control BMPs will be applied to minimize erosion until closure vegetation is established.

If required, impacted soil will be disposed of in a hazardous waste facility, but it is expected that the quantity of soil that cannot be land-farmed on-site will be limited.

20.3.10 Monitoring and maintenance

A program of monitoring and facility maintenance during the mining operations will ensure that the control measures used to prevent environmental impacts are effective. The program will include climate monitoring, water quality sampling, groundwater and surface water quality and quantity monitoring, geotechnical and hydrogeochemical monitoring of ARD conditions on the co-disposal area, pits, and pit backfill. The monitoring program for operations is described in more detail in the ESIA and approved ITS documents.

Monitoring and maintenance of the mine reclamation and closure activities will begin while these activities are underway and will continue during the 5-year post-closure period. Monitoring and maintenance will be completed primarily by a team of on-site personnel. Equipment will be made available for the monitoring of reclaimed areas of the project and for the performance of regular maintenance. The monitoring will include water quality sampling and analysis, in addition to regular inspection of the co-disposal facility, process area, and the reclaimed pits.

Monitoring activities for closure, include:

- monitoring of physical stability
- monitoring of groundwater and surface water
- monitoring of effluents

- monitoring of re-vegetation success
- monitoring of terrestrial fauna
- social monitoring.

Maintenance will also focus on the surface water channels around the pit, over the pit backfill, and the co-disposal facility. Sediment removal and repair of any damage to channel lining will be specific areas of focus for the maintenance program.

20.3.11 Closure schedule

The schedule for performing reclamation and closure activities is summarized as follows:

- Este pit backfill - concurrent reclamation during operations
- Minas/Main pit backfill - completion of backfill reclamation three years after closure
- co-disposal facility - concurrent reclamation of completed surfaces as they become available, beginning in Year 2; completion of activities in tandem with the Minas/Main pit backfill
- surface water management system - as soon as possible after closure
- post closure maintenance and monitoring for five years after final closure activities are completed.

20.4 Socioeconomics and Community

The Project site is located within the jurisdiction of two rural communities (comunidades campesinas), Chacaconiza and Quelcaya. These communities, situated at over 4,000 masl, are small isolated rural settlements, with a population of between 300-450 individuals each. The majority of the population speaks Quechua as their first language.

The geographical isolation and lack of infrastructure in the zone result in reduced economic and social development opportunities, and high levels of poverty. However, both communities have access to water, electricity, and rudimentary wastewater services. The main housing construction material is adobe, with thatched roofs and earth floors. The most common illness is respiratory infections, which predominately impacts children and the elderly during periods of snow and cold weather.

Bear Creek has undertaken community relations activities as part of previous permitting activities related to exploration campaigns. Further activities were undertaken as part of the original ESIA that was approved by the Peruvian Government in 2013. The ESIA process included a thorough description of the current social and economic status of the communities, and the analysis of possible and anticipated, positive and negative impacts on these communities. Following analysis of the impacts, methods were developed to avoid, remedy or mitigate the identified negative community impacts, including the development of social programs aimed at providing enhanced economic and social development opportunities. These are presented in the ESIA.

Further community engagement occurred concurrent with the impact assessment process. This included two initial workshops with each of the two communities (Chacaconiza and Quelcaya) affected by the project. The first workshops were undertaken before the ESIA studies began, and the ESIA process was explained to the community. Additional workshops were undertaken during the elaboration of the ESIA. Finally, a public meeting was held. At this meeting, the ESIA and Community Participation Plan were presented to the authorities for approval. The Project has been optimized since the original ESIA, which results in fewer impacts due to a reduced footprint and less water consumption. Since the optimizations, in part permitted in the ITS documents, reduce environmental and social impacts, additional public hearings have not been required.

Additionally, Bear Creek completed a Life of Mine ("LOM") Investment Agreement in June 2013. This agreement was entered into with the District of Carabaya, five surrounding communities, and relevant, ancillary organizations specifying investment commitments over the 23 year project life, including the pre-production period. Under the agreement, annual payments are to be made into a trust designed to fund community projects totaling 4 million nuevos soles per year (approximately $1.2 million $ per year). Once a development decision is made, payments will remain constant throughout the pre-development phase and during production. Cessation or interruptions of operations will cause a pro-rata decrease in the annual disbursements. As an integral part of the LOM agreement, a trust or foundation structure is established for approval of investments and disbursement of funds. Each of the five communities (Corani (Aconsaya), Chacaconiza, Quelcaya, Isivilla, and Aymaña) has agreed to the formation of committees which have considered and approved investment projects for the benefit of the communities, such as schools, medical facilities, roads, or other infrastructure. The amounts of the total annual investment to be directed towards each community is agreed to and defined in the agreement. Bear Creek is an oversight member of the trust and helps implement the community projects but does not have voting powers. In this structure, Bear Creek's intent is to appoint independent members with community social responsibility experience and credibility in order to provide oversight of the foundation's functions in meeting its commitments to the communities and all of its members.

21 Capital and Operating Costs

21.1 Capital Cost Summary

The Corani capital cost estimate has been prepared to a estimate accuracy that falls between -10% to +15% cost range and has generally been prepared in line with the AACE Guidelines for a Class 3 Estimate.

Table 21-1 lists the level and status of detail of the documents and deliverables provided for the assembly of the feasibility estimate.

Table 21-1: Estimate requirements

Discipline	Document / Deliverable	Yes / N/A	Level of Detail
Site	Geographical location	Yes	
	Maps and surveys	Yes	
	Soil and foundation tests	Yes	
Environmental	Field data collection	Yes	
	Impact assessment/EIS report	Yes	Final study level
	Impact management plan	No	
Mining	Mine planning criteria	N/A	Feasibility level
	Mine model	N/A	Feasibility level
	Strategic planning and optimisation	N/A	Feasibility level
	Mine design and scheduling	N/A	Feasibility level
	Mine equipment and facilities	N/A	Equiry to ≧ 3 contractors
	Mine infrastructure	N/A	Equiry to ≧ 3 contractors
	Mine contractor pricing	N/A	Equiry to ≧ 3 contractors
	Cost modelling	N/A	Equiry to ≧ 3 contractors
Process Engineering	Process description	Yes	Feasibility level
	Design criteria	Yes	Feasibility level
	Process flow diagrams	Yes	Feasibility level
	Piping and instrument diagrams	No	
	Bench scale tests	Yes	Suit plant optimization and accommodate ore variability
	Laboratory pilot plant tests	No	
	Energy and material balances	Yes	Feasibility level
	Water balance	Yes	Feasibility level
	Control philosophy	Yes	Basic engineered
Mechanical Engineering	Design criteria	Yes	Feasibility level
	Equipment list (Process and Utility)	Yes	Feasibility level
	General arrangement drawings	Yes	Feasibility level
	Specification and data sheets for mechanical equipment	Yes	Feasibility level
	Budget quotes for a min of 80% of total mech. equipment cost	Yes	Enquiry to ≥ 2 vendors
	Mechanical installation cost estimate	Yes	Recent local project
	First fill consumables list	Yes	Basic engineered

Discipline	Document / Deliverable	Yes / N/A	Level of Detail
	Spares list	Yes	Factored
Piping	Specifications	N/A	Factored
	Design criteria	N/A	Factored
	Piping modelling and material take-off	N/A	Factored
	Pipe, valves and fittings purchase and spool fabrication cost	N/A	Factored
	Pipe, valves and fittings installation cost	N/A	Factored
	Piping layouts and isometrics	N/A	Factored
Civil Engineering	Design criteria	Yes	Basic engineered
	Geotechnical and hydro-geological study	Yes	Test pits required
	Site plot plan and topographical map	Yes	Basic engineered
	Site drainage and sewer systems layout	Yes	Basic engineered
	Building dimensions and type of construction	Yes	Basic engineered
	Bulk earthworks MTO	Yes	Basic engineered
	Bulk earthworks cost and schedule of rates	Yes	Enquiry to ≥ 3 contractors
	Concrete design and calculations	Yes	Basic engineered
	Concrete MTO	Yes	Basic engineered
	Concrete cost and rates schedule	Yes	Enquiry to ≥ 3 contractors
Structural Engineering	Design criteria	Yes	Basic engineered
	Structural steel design and calculations	Yes	Basic engineered
	Structural steel MTO	Yes	Basic engineered
	Structural steel fabrication cost and schedule of rate	Yes	Enquiry to ≥ 3 fabricators
	Structural steel erection cost and schedule of rates	Yes	Enquiry to ≥ 3 contractors
	Platework design and calculations	Yes	Basic engineered
	Platework MTO	Yes	Basic engineered
	Platework steel fabrication rates	Yes	Enquiry to ≥ 3 fabricators
	Platework erection rates	Yes	Enquiry to ≥ 3 contractors
Electrical and Instrumentation Engineering	Design criteria	Yes	Basic engineered
	Electrical equipment datasheets	Yes	Basic engineered
	Electrical equipment list	Yes	Basic engineered
	Single line diagrams	Yes	HV, MV and examples of LV
	Detailed drawings	No	No
	Electrical equipment and bulks supply costs	N/A	Equipment budget quotes, Bulks database pricing
	Overhead Powerlines	Yes	Basic engineered
	Overhead Powerlines supply and installation	Yes	Enquiry to ≥ 3 contractors
	Electrical installation rates	Yes	Enquiry to ≥ 3 contractors
	Process control determination (PLC, DCS)	Yes	Basic engineered
	PCS quotation	Yes	Budget Enquiry

Discipline	Document / Deliverable	Yes / N/A	Level of Detail
	Major Tagged process instruments supply costs	Yes	Database pricing
	Instrument bulks supply costs	Yes	Database pricing
	Instrument installation rates	Yes	Enquiry to ≥ 3 contractors
	Instrument list	Yes	Basic engineered
	Detailed drawings	No	Factored
Scope and General Requirements	Define scope of work c/w battery limits	Yes	Final
	Define study scope of services	Yes	Final
	Work Breakdown Structure	Yes	Basic engineered
	Location specific craft labour rates	Yes	Enquiry to ≥ 3 contractors
	Location specific craft labour productivity factors	Yes	Enquiry to ≥ 3 contractors
	Location specific taxes, tariffs and duty type costs	Yes	By enquiry
	Location specific freight costs	Yes	Recent local project
	Temporary construction facilities and services	Yes	Allowances and factored from recent Peruvian project
	Project EPCM estimate for project scope of services	Yes	Final study level
	Project schedule	Yes	Level 3
	Owner's costs	Yes	As provided by Owner
	Estimate Contingency	Yes	Final study level
	Compiled and formatted estimate	Yes	Final study level
Estimating	Collated and referenced support file structure (Audit trail)	Yes	Final study level
	Estimate Plan / Basis of Estimate	Yes	Final

The capital cost estimate is summarised at WBS level 1 in Table 21-2.

Table 21-2: WBS Level 1 summary capital cost estimate by scope responsibility

WBS	Description	Ausenco Value ($M)	Bear Creek Value ($ M)	TOTAL ($ M)
1000	Mining	0.0	59.3	59.3
2000	Process plant	234.3	0.0	234.3
3000	On-site infrastructure	17.4	40.8	58.2
4000	Off-site infrastructure	0.0	25.5	25.5
5000	Field indirects	20.7	0.3	20.9
6000	Other	3.9	0.0	3.9
7000	Engineering	60.0	0.0	60.0
8000	Owner's costs	0.0	65.3	65.3
9000	Provisions	34.4	17.1	51.5
	Total	**371**	**208**	**579**

21.1.1 Currency

The capital cost estimate base date is 1st Quarter 2019 and the estimate is expressed in United States dollars ($ or $).

Table 21-3 provides a breakdown of currency basis and Table 21-4 provides the exchange rates.

Table 21-3: Foreign currency component

Code	Currency	Value (Native Currency, M)	Value ($M)
AUD	Australian Dollar	13.7	10.1
CAD	Canadian Dollar	1.8	1.3
EUR	Euro	9.0	10.5
$	United States Dollar	557	557

Table 21-4: Foreign currency exchange rates

Code	Currency	Exchange Rate
AUD	Australian Dollar	0.7407
CAD	Canadian Dollar	0.7613
EUR	Euro	1.1600
$	United States Dollar	1.0000

All items in the estimate have been entered in the native currency of the quote except for Owner's Costs which were entered into the estimate in $ as provided.

21.1.2 Estimate exclusions

The following items are excluded from the estimate:

- mining costs, other than those specifically included
- sunk costs
- refurbishing of temporary facilities upon completion of construction (if required)
- demolition and salvage of any existing on-site structures
- ongoing exploration costs
- operation and maintenance manuals other than those supplied by equipment vendors
- force majeure issues
- future scope changes
- special incentives (schedule, safety or others)
- no allowance has been made for loss of productivity and/or disruption due to religious, social and/or cultural activities
- foreign currency fluctuations and escalation
- all owner-payable taxes except as noted
- any operational insurance such as business interruption insurance and machinery breakdown etc

- license and royalty fees
- contract incentives
- project interest.

21.2 Mining Capital Costs

Mining will be performed using contract mining. The initial mining capital costs are shown in Table 21-5. These mining capital costs are associated with the construction of the haul roads, processing of materials, surface drainage and the ROM Pad, the Phase 2 Construction costs are associated with pre-stripping. The pre-production costs are based on a contract mining quotation.

Table 21-5: Mining capital costs

WBS	Area Description	Ausenco Value ($M)	Bear Creek Value ($ M)	TOTAL ($ M)
1100	Mining	0.0	30.5	30.5
1300	Mining facilities	0.0	0.9	0.9
1400	Mining phase 2	0.0	27.9	27.9
	Total $	**0.0**	**59.3**	**59.3**

21.3 Process Plant Capital Costs

The Process Plant direct capital costs are summarised in Table 21-6.

Table 21-6: Processing plant capital costs

WBS	Area Description	Ausenco Value ($M)	Bear Creek Value ($ M)	TOTAL ($ M)
2100	Primary crushing	9.3	0.0	9.3
2200	Stockpile and reclaim	8.2	0.0	8.2
2300	Grinding and classification	46.9	0.0	46.9
2400	Flotation and regrind	59.6	0.0	59.6
2500	Concentrate thickening and filtration	17.3	0.0	17.3
2600	Tailings thickening	9.1	0.0	9.1
2700	Tailings filtration and stockpile	66.3	0.0	66.3
2800	Reagents	8.8	0.0	8.8
2900	Utilities, services and plant common	8.8	0.0	8.8
	Total $	**234**	**0.0**	**234**

Mechanical, electrical and instrumentation equipment lists have been prepared where applicable for each WBS area and budget marked pricing sourced for supply. Bulk material quantities for each discipline required to complete the scope of work for the process plant have been produced.

Budget rates from local contractors have been used for equipment installation costs and bulk commodity supply and installation, which includes installation productivity factor to suit the Bear Creek altitude and location.

21.4 On Site Infrastructure Capital Costs

On Site Infrastructure capital costs are summarised Table 21-7. The bulk earthworks includes the process plant and mine infrastructure access (MIA), contact and non-contact water ponds and access from the gate to site. The Tailings Waste Dump also includes drainage and seepage ponds and the temporary waste dump. It should be noted that the main transmission line to the Project site will be contracted on a Build-Own-Operate-Transfer basis (BOOT) and therefore the associated costs have been captured in the Operating Cost Estimate.

Table 21-7: On site infrastructure

WBS	Area Description	Ausenco Value ($M)	Bear Creek Value ($ M)	TOTAL ($ M)
3100	Bulk earthworks	0.0	22.4	22.4
3200	Infrastructure buildings	8.7	0.0	8.7
3300	HV substation and distribution	1.6	7.7	9.3
3400	Control system and communications	6.7	0.0	6.7
3500	Sewage	0.4	0.0	0.4
3700	Tailings and waste dump	0.0	10.7	10.7
	Total $	**17.4**	**40.8**	**58.2**

Mechanical, electrical and instrumentation equipment lists have been prepared where applicable for each WBS area and budget marked pricing sourced for supply. Bulk material quantities for each discipline required to complete the scope of work for on site infrastructure have been produced.

Budget rates from local contractors have been used for equipment installation costs and bulk commodity supply and installation, which includes installation productivity factor to suit the Bear Creek altitude and location.

21.5 Off Site Infrastructure Capital Costs

Off Site Infrastructure capital costs are summarised in Table 21-8.

Table 21-8: Off site infrastructure areas

WBS	Area Description	Ausenco Value ($M)	Bear Creek Value ($ M)	TOTAL ($ M)
4100	Main access road	0.0	18.4	18.4
4300	Accommodation village	0.0	7.1	7.1
	Total	**0.0**	**25.5**	**25.5**

Quantities were developed and budget rates from local contractors have been applied for the Access Road and Accommodation Village earthworks pad.

The accommodation village costs were taken form a contractor quotation for rental over life of mine. Rental costs during construction were captured within the capital cost estimate.

21.6 Field Indirects Capital Costs

Field Indirects are summarised in Table 21-9.

Table 21-9: Fields indirect costs

WBS	Area Description	Ausenco Value ($M)	Bear Creek Value ($ M)	TOTAL ($ M)
5100	Temporary construction facilities and utilities	18.5	0.0	18.5
5200	Construction support	1.5	0.0	1.5

WBS	Area Description	Ausenco Value ($M)	Bear Creek Value ($ M)	TOTAL ($ M)
5300	Contractor commissioning assistance	0.6	0.0	0.6
5600	Quarry and aggregate production	0.0	0.3	0.3
	Total	**20.6**	**0.3**	**20.9**

Temporary construction facilities costs are the field costs during the construction phase of the project that cannot be directly identified or attributed to specific construction activities of the permanent facilities.

These costs have been estimated based on a recent similar project in the region. These costs cover, but are not limited to temporary buildings, temporary utilities, temporary services, an additional cost of water cartage required when additional water is to be trucked in for construction.

Construction support costs cover Vendor representatives' costs during commissioning and construction includes vendor representative support during the installation of the packaged purchased equipment.

Vendor representatives' requirements have been estimated based on the mechanical equipment list. There are allowances for vendor installation supervision for the mills and crusher along with vendor commissioning allowances for major equipment. The estimate includes allowances for travel, airfares and time on site.

WBS area 7000 contains the cost of the EPCM contractor commissioning personnel. Contractor commissioning assistance on a limited level have been included in the estimate as the balance will be provided by the Owner from their operation and maintenance team as part of the commissioning process.

Cost for supervision and management of the borrow pits for aggregate production have been included. Costs for operation of the quarry are included in the concrete and earthworks contractor's costs.

21.7 Other Capital Costs

Other capital costs are summarised in Table 21-10.

Table 21-10: Other capital costs

WBS	Area Description	Ausenco Value ($M)	Bear Creek Value ($ M)	TOTAL ($ M)
6100	First Fills	3.2	0.0	3.2
6200	Spares	0.6	0.0	0.6
	Total	**3.9**	**0.0**	**3.9**

An allowance for commissioning construction first fills e.g. oils and flushing fluids, has been included in the estimate. This allowance is based on estimated costs for a similar sized plant.

Reagent and mill media first fills have been estimated by way of a detailed estimate build up from current budget market costs and included in the estimate accordingly.

Commissioning, spares have been factored from the total direct cost. The factor used have been established from a current Ausenco project costs in the region.

Insurance spares and initial operating spares have been included by the Owner elsewhere in the Owner's Start up and Operational Readiness costs.

21.8 Engineering, Project Management and Supervision Capital Costs

Engineering, project management and supervision indirect capital costs are valued at $60.0 M to Ausenco. Bear Creek commissioning services are included in the Owner's Cost.

The project was estimated based on an EPCM delivery strategy. The estimate is based on a detailed build-up of manpower schedules based on project execution plan, project schedule, location of design office, assignment conditions, nominated personnel including expatriates, man-hours, labour and burdens, overhead, expenses and fee.

The EPCM estimate and costs includes the following:

- home office manpower schedule, by positions, nominations, durations and actual spent hours, actual cost of labour, burdens and overheads
- home office expenses including printing/reproduction, postage, computers, travel, hotel accommodation, assignment conditions, etc.
- site (Construction) office manpower schedules by positions, nominations, durations, actual spent man-hours, actual cost of labour burdens and overheads
- site office (Construction) expenses, including printing/reproduction, postage, computers, travel, hotel accommodation, assignment conditions, etc.
- home office engineering, procurement, contracts, project controls, etc. production rates (manhours / deliverable for all disciplines)
- site (Commissioning) office manpower schedules by positions, nominations, durations, actual spent man-hours, actual cost of labour burdens and overheads
- site office (Commissioning) expenses, including printing/reproduction, postage, computers, travel, hotel accommodation, assignment conditions, etc.
- QA/QC and 3rd party inspections.

21.9 Owner's Capital Costs

The capital cost includes an estimate of $65.3M for Owner's Costs (by BCM) these include estimates for Owner's staffing to the completion of pre-production, camp leasing and operations costs for the Owner's team and their contractors, camp/ corani expenses, Lima office expenses, community relations and Annual Trust, insurance / capital spares and operational readiness. These costs are summarised in Table 21-11.

Table 21-11: Owner's costs

WBS	Area Description	TOTAL ($ M)
8100	Salaries and benefits	21.2
8200	Camp/Corani	15.5
8300	Lima office	13.3
8400	Environmental, health and safety	1.7
8500	Community relations	4.7
8700	Start-up and commissioning	3.0
8800	Operational readiness	5.8
	Total	**65.3**

21.10 Contingent Capital Costs

Contingent capital costs are summarised in Table 21-12.

Table 21-12: Provisions costs

WBS	Area Description	Ausenco Value ($M)	Bear Creek Value ($ M)	TOTAL ($ M)
9200	Contingency	34.4	17.1	51.5
9300	FOREX (excluded)	0.0	0.0	0.0
9400	Taxes and duties (excluded)	0.0	0.0	0.0
9500	Escalation (excluded)	0.0	0.0	0.0
	Total	**34.4**	**17.1**	**51.5**

No costs have been included for changes in foreign currency exchange as stated above or escalation beyond the base date of the estimate 1Q2019.

Taxes and duties are excluded from the estimate.

21.11 Operating Cost Summary

The estimated operating costs incurred over the life of mine are presented in Table 21-13.

Table 21-13: Operating cost summary

Cost Item	LOM Cost ($ M)
Mining	594
Process Plant	1 391
General and Administrative	261
TOTAL	2 246
LOM ROM (Mt)	139
Average LOM Operating Cost	$ 16.20 /t

21.11.1 Mine operating cost summary

The mine will be operated by a contractor and technical oversight will be provided by the Owner. The costs associated with the contract mining and Bear Creek Technical Services are presented in Table 21-14.

Table 21-14: Mine operating cost estimate

Cost Item	LOM Cost ($ M)	LOM Average Cost ($/t)
Direct Costs	443.1	3.20
Indirect Costs	123.7	0.89
Technical Services [Bear Creek]	27.6	0.20
Total	**594.4**	**4.29**

21.11.2 Process plant operating and maintenance costs

The process plant operating, and maintenance costs are summarised in Table 21-15 and include; labour, power, reagents, grinding media and wear parts and maintenance supplies and services.

Table 21-15: Process plant operating cost estimate

Cost Item	LOM Cost ($M)	LOM Average Cost ($ / tonne Ore)
Operating and maintenance costs	89.2	0.65
Overhead transmission line BOOT contract and power costs	344.5	2.49
Reagents	340.5	2.46
Wear parts and consumables	238.7	1.72
Tailings disposal	224.6	1.62
Maintenance parts and services	100.5	0.73
Mobile equipment	37.0	0.27
Laboratory	13.3	0.10
Water pumping	2.8	0.02
Total	**1,391**	**10.04**

21.11.3 Process plant labour

The Process Plant labour costs were derived from a detailed staffing plan and are based on benchmarked annual labour rates provided by Bear Creek. The labour salaries and benefits include all applicable social security benefits, payroll taxes and burdens.

A total of 124 employees will be employed at the Process Plant with 62 in Mill Operations and 62 in Mill Maintenance. A summary of the gross annual labour costs and the staffing plan are shown in Table 21-16 and Table 21-17.

Table 21-16: Process plant labour

Department	Number of Personnel	Total Labour ($M/y)
Mill operations	62	2.555
Mill maintenance	62	3.387
Total	**124**	**5.942**

Table 21-17: Process plant staffing plan

Department	Job Title	Number of Personnel
Mill Operations	Plant Manager	1
	Process Plant Supervisor	5
	Crusher	6
	Grinding	6
	Flotation	6
	Concentrate Filter	6
	Tailings Filter	6
	Reagent Preparation	3
	Utility	12
	Metallurgy Superintendent	1
	Metallurgy Supervisor	2

Department	Job Title	Number of Personnel
	Sample Preparation	4
	Metallurgist	4
Mill Maintenance	Plant Maintenance Superintendent	1
	Senior Supervisor	7
	Supervisor	6
	Planner	4
	Reliability Engineer	2
	PdM Technician	2
	Mechanic	20
	Welder	3
	Crane Operator	3
	Electrician	9
	Instrument Technician	5
Total		**124**

21.11.4 Power

The estimated cost of power for Corani was based on both the estimated electricity consumption and estimated generation-level unit prices.

- Consumption - The estimated power consumption, excluding the Camp, is 48.9 MW (based on the equipment list presented by Ausenco) it is estimated that the annual power requirement will be 588 MW and the energy consumed will be 426,043 MWH.
- Unit Prices - Since Bear Creek Mining are considered as a “free customer” with the option of direct price negotiation with a generator, the offer from “KALLPA Generación S.A.” has been used as a reference. KALLPA Generación is recognised as a prestigious company within the Power Generation Industry in Perú. The price of energy considered is $28 /MWH in HP and HFP and the power generation is $6.20 /KW.

The estimated annual power costs are $21.5M (excluding IGV) and have been calculated as follows:

- Power – This is the result of the price offered combined with the power that is recorded in HP which “matches” the same period recorded by the COES-SINAC. A factor of 0.75 is used, which is the probability of matching the maximum demand of the mine with the maximum national demand. (a factor resulting from the experience of the power generator) The annual cost is estimated at $ 2.7M.
- Energy – This is the result of the electrical energy consumed by the mine equipment for the price offered in HP and HFP. The annual cost is estimated at $ 11.9M.
- Tariffs – The tariffs are for the principal and secondary transmission tolls. The prices are regulated by OSINERGMIN and the annual cost is estimated at $ 5.0M.
- Regulated Charges – This is the payment of the state regulated charges for Rural Electrification (RE) and the Social Energy Inclusion Fund (FISE). The prices are determined by OSINERGMIN and the annual cost is estimated at $ 1.8M.

Table 21-18: Estimated energy cost reference table.

Uses	Concept	Units	Value
	Power	MW	588
	Energy	MWh	426,043
Prices	Concept		
Open	Power	$ / kW-month	6.20
	Energy	$ / MWh	28.0
Regulated	Principal Tariff	$ / kW - month	10.79
	Secondary Tariff	$ / MWh	0.65
	Rural Electrification (RE)	$ / MWh	2.55
	Social Inclusion Energy Fund (FISE)	$ / MWh	1.95
Billing	Concept		
Open	Power	$M	2.7
	Energy	$M	11.9
	Sub-total	$M	**14.7**
Regulated	Principal Tariff	$M	4.75
	Secondary Tariff	$M	0.275
	Rural Electrification (RE)	$M	1.09
	Social Inclusion Energy Fund (FISE)	$M	0.75
	Subtotal	$M	**6.86**
Total $M (not including IGV)			**21.5**

Power costs were based on purchasing power from “KALLPA Generación” as described above and the associated rates were applied. Power consumption was based on the connected kW derived from the equipment list (provided by Ausenco) and discounted for the operating time per day and the anticipated operating load level. The overall power cost is estimated at $0.0505/kWh with a consumption of 43.6 kWh per ore tonne. A summary of the power consumption is shown in Table 21-20.

Table 21-19: Summary of electric power consumption

WBS	WBS Description	Power Consumption (kW)	Average 24-Hour Demand (kW)	Average Power (MWh/y)
0000	Non-process plant	2,409	2,216	17,140
2100	Primary crushing	760	532	6,675
2200	Stockpile and reclaim	485	436	4,259
2300	Grinding and classification	21,429	19,714	188,225
2400	Flotation and regrind	11,591	11,128	101,819
2500	Concentrate thickening and filtration	1,263	1,045	11,092
2600	Tailings thickening	647	621	5,680
2700	Tailings filtration and stockpile	8,616	7,132	75,679
2800	Reagents	389	358	3,415
2900	Utilities, services and plant common	1,373	1,318	12,060
Total		**48,960**	**44,502**	**426,043**
$/kWh				**0.051**

WBS	WBS Description	Power Consumption (kW)	Average 24-Hour Demand (kW)	Average Power (MWh/y)
M $/y				**21.53**
$/t ore				**2.20**

In addition to the power costs, the BOOT Contract Financing cost and the Operation and Maintenance of the Transmission line were also estimated and are shown in Table 21-20.

Table 21-20: BOOT financing and transmission line operation and maintenance cost estimate

Cost Item	LOM Cost ($M)	LOM Average Cost $/ t
BOOT financing cost	33.5	0.240
Transmission line operating and maintenance	5.90	0.040

21.11.5 Reagents

Consumption rates were determined from the metallurgical test data or derived or estimated from industry practice. Reagents prices were supplied by Bear Creek Mining from quotes received in the local area with an allowance for freight to site. A summary of process reagent consumption and costs is shown in Table 21-21.

Table 21-21: Reagent consumption and cost estimate

Reagent	Consumption		Unit Rate	
	Kg / t	Average kg / y	$ / kg	Average $ M/y
Sodium isopropyl xanthate	0.040	369,553	2.037	1.16
Lime (calcium oxide)	3.500	32,335,871	0.127	6.31
Methyl isobutyl carbinol	0.050	461,941	2.867	2.03
Pb promoter A-404	0.015	138,582	3.507	0.746
Sodium cyanide	0.020	184,776	2.267	0.643
Copper sulphate	0.300	2,771,646	1.917	8.15
Sodium hydroxide	0.010	92,388	0.657	0.093
Sodium sulphite	0.150	1,385,823	0.615	1.31
Zinc sulphate	0.620	5,728,069	0.817	7.18
Flocculant	0.020	184,776	3.317	0.94
Antiscalent	0.005	46,194	2.387	0.169
Water treatment reagent				6.10
Total				**34.8**

21.11.6 Wear parts, consumables and maintenance

The SAG mill and ball mill ball consumption was based on the process design criteria and the remaining grinding media and wear items (liners) consumption were based on industry practice for the crusher and grinding operations. These consumption rates and unit prices are shown Table 21-22.

Table 21-22: Grinding media and wear parts consumption and cost estimate

Item	Consumption		Unit Rate	
	kg / t	Average kg / y	$ / kg	Average $M / y
Primary crusher liners	0.008	73,911	4.600	0.340
SAG mill liners	0.050	461,941	3.600	1.663
SAG mill balls	0.238	2,198,839	1.346	2.959
Ball mill liners	0.030	277,165	3.600	0.998
Ball mill balls	0.690	6,374,786	1.128	7.191
Lead regrind mill liners	0.005	46,194	5.600	0.259
Zinc regrind mill liners	0.005	46,194	5.600	0.259
Lead regrind mill balls	0.010	92,388	1.137	0.105
Zinc regrind mill balls	0.010	92,388	1.137	0.105
Filter cloths				2.033
Total				**15.9**

The cost of maintenance parts and services of $100,467M that were not specifically identified were calculated as percentage of the CAPEX for each area, these are included in Table 21-15.

21.11.7 Process supplies and services

Allowances have been included as part of the G&A cost for both outside consultants and contractors, vehicle maintenance and miscellaneous supplies for the Process Plant. These are presented as part of the G&A cost shown in Table 21-23.

Laboratory

Laboratory cost estimates are based on labour and fringe benefits, power, reagents, assay consumables, and supplies and services. The laboratory and associated labour costs are included in the process plant operating cost calculation in Table 21-15.

21.12 General Services and Administration (G&A) Operating Costs

The operating costs for the G&A areas were determined by the client and summarized by cost element shown on Table 21-23. The cost elements include salaries and benefits, camp operations, camp leasing, insurance, travel, community development, and other expenses such as software licenses.

Table 21-23: G&A operating cost estimate

Cost Item	LOM Cost ($M)	LOM Cost ($/t)
Salaries and benefits	96.8	0.70
Camp operations	58.1	0.42
Camp leasing/rental	33.8	0.24
Insurance	15.2	0.11
Travel	8.99	0.065
Community development	21.2	0.15
Other	26.7	0.19
Total	261	1.88

21.13 Concentrate Handling, Transportation and Storage Costs

In September 2019, BCM engaged BLB Advisory EIRL (BLB”), a company with recent experience in the sale and delivery of Peruvian sourced metals in concentrate, to prepare a Marketing and Sales Logistics Study. BLB indicate that lead concentrate will be loaded into standard lined containers at the plant site and transported by truck to the port of Matarani, which is able to adequately store and handle the expected volume of containers for shipment. Zinc concentrate will be loaded into semi-trailer end dump trucks and transported to the port of Matarani, for export as bulk cargo. Costs provided by BLB for concentrate handling, transportation, smelting and refining are presented in Table 21-24.

Table 21-24: Concentrate handling, treatment, and refining costs

Cost Item	LOM Cost ($M)
Concentrate treatment charges	431
Silver refining charges	109
Transportation	344
Total	883

21.14 Reclamation and Closure Cost

Costs associated with the closure and reclamation were provided by the client and reflect the closure plan approved by the Peruvian Ministry of Energy and Mines on September 12, 2018. Costs include progressive closure costs that occur during the life of the mine, final closure costs that occur after production has ceased and post closure costs that are associated with ongoing monitoring activities. These costs are expected to be partially offset by salvaging some items. Table 21-25 summarizes the closure costs.

Table 21-25: Closure cost estimate

Item	Cost ($M)
Progressive closure (years 1-15)	24.91
Final closure (years 16-18)	21.97
Post-closure (years 19-23)	0.96
Total	**47.83**

22 Economic Analysis

Ausenco and Estudio Rodrigo, Elias & Medrano Abogados have reviewed and verified that the economic model generated by BCM was prepared based on sound engineering and financial principles and is correct. The capital and operating cost inputs were sourced from the data summarised in Section 21, prepared by Ausenco, BCM and Anddes. The financial indicators stated herein are significantly improved over the 2017 Feasibility Study Estimate.

22.1 Introduction

The financial evaluation presents the net present value (NPV), payback period (time in years to recapture the initial capital investment), and the internal rate of return (IRR) for the project. Annual cash flow projections were estimated over the life of the mine based on the estimates of capital expenditures, production cost, and sales revenue. Revenues are based on the production of a lead-silver and a zinc concentrate. The estimates of capital expenditures and site operating/production costs have been developed specifically for this project and have been presented in earlier sections of this report.

22.2 Mine Production Statistics

Mine production is reported as ore and waste from the mining operation. The annual production figures were obtained from the mine plan as reported earlier in this report. A total of 138.6 million tonnes of ore are mined at an average grade of 51.3 g/t silver, 0.90% lead, and 0.56% zinc. A total of 196.4 million tonnes of waste are mined for a stripping ratio of 1.42:1, including 18 million tonnes included in the pre-stripping phase as part of the capital estimate. The strip ratio during operation is ~1.3:1.

22.3 Plant Production Statistics

The design basis for the process plant is 27,000 tonnes per day at 92% availability. The metal recoveries, which are variable by ore characteristics, are projected to average 67.1% for zinc, 61.1% for lead and 69.9% for silver.

The estimated life of mine recovered and payable metal production is presented in Table 22-1.

Table 22-1: Metal recovered to concentrate and payable metal production

Life of Mine	Metal Recovered t	Payable Metal t
Zinc (Mlb)	1,225	1040
Lead (Mlb)	1,568	1476
Silver (Moz)	157.7	144.1

22.4 Smelter and Refinery Return Factors

The lead and zinc concentrates will be transported to a holding facility at the port of Matarani and consolidated for shipment to smelters for final processing. Smelter treatment and refining charges will be negotiated. The smelter charges used in the financial model are presented in Table 22-2 and Table 22-3.

Table 22-2: Smelter treatment factors (lead concentrate)

Smelter treatment factors	Value
Lead concentrate	
Payable lead (%)	95.0
Minimum deduction (%)	3.0
Payable silver (%)	95.0
Ag minimum deduction (oz/dmt)	1.61
Treatment charge ($/dmt)	111.50
Refining charge – Ag ($/payable oz.)	0.80
Lead concentrate transportation	
Concentrate trucking and port ($/wmt)	70.96
Concentrate shipping ($/wmt)	66.75
Moisture (%)	7.0
Penalties	
Arsenic ($/dmt per 0.1% over 0.25%)	2.00
Antimony ($/dmt per 0.1% over 0.25%)	2.00
Bismuth ($/dmt per 0.01% over 0.15%)	2.00
Mercury ($/dmt per 10 ppm over 60 ppm)	1.50

Table 22-3: Smelter treatment factors (zinc concentrate)

Smelter treatment factors	Value
Zinc Concentrate	
Payable zinc (%)	85.0
Minimum deduction (%)	8.0
Price Participation Basis ($/tonne metal)	
Price participation above $2,400 ($/dmt)	0.10
Price participation below $2,400 ($/dmt)	(0.03)
Payable silver (% of balance)	70.0
Silver minimum deduction (oz/dmt)	3.0
Treatment charge ($/dmt)	231.30
Refining charge – Ag (% of metal price)	0.0
Zinc Concentrate Transportation	
Concentrate trucking and port ($/wmt)	59.00
Concentrate shipping ($/wmt)	63.28
Moisture (%)	8.0
Penalties	
Arsenic ($/dmt per 0.1% over 0.25%)	2.00
Cadmium ($/dmt per 0.1% over 0.4%)	1.00
Bismouth ($/dmt per 0.01% over 0.15%)	1.50
Mercury ($/dmt per 10 ppm over 40 ppm)	1.00
Antimony ($/dmt per 0.1% over 0.25%)	2.00
Copper + lead ($/dmt per 1% over 4%)	1.00
Silica ($/dmt per 1% over 3%)	3.00

22.5 Capital Expenditure

22.5.1 Initial capital

The base case initial capital costs are estimated to be $579.3 million, which includes pre-production waste stripping and mine development, plant construction, on and off-site infrastructure, field indirects, engineering, owners' costs, contingency, and other costs. Further detail of the capital costs by area is provided in Section 21. Approximately 60% of these expenditures will be incurred over a two-year period.

22.5.2 Sustaining capital

Sustaining capital consists of $22.5 million for the procurement and installation of a tailings conveyor and $1 million for water management. Approximately 45% of the conveyor capital expenditure will take place during the second year of operation, with additional conveyor expansions scheduled during the remaining years of operation.

22.5.3 Working capital

90% of sales are expected to be paid during the month where concentrates are sold, with the remaining 10% paid 60 days thereafter. 50% of purchase invoices are expected to be paid in cash with the balance paid after 30 days. It is assumed that the plant will hold one month's production costs as work in process inventory, estimated at $12.2 million (the average monthly cost for the first 36 months of production). The inventory component of working capital is added to uses of cash during the first year of operation and returns as a source of cash during the last year of the mine's life.

BCM has an agreement with the Peruvian authorities for the early return of Peruvian value added taxes ("IGV"). IGV is an 18% refundable tax on purchased goods and services. The financial model assumes that BCM will receive IGV during development and construction. The financial model reflects a 30-day delay between initial payment and subsequent recovery of IGV. Once in production, BCM will be able to recover prior periods' IGV payments of $14.96 million.

22.5.4 Revenue

Revenue is determined by applying estimated metal prices to the annual payable metal estimated for each operating year. Sales prices have been applied to all life of mine production without escalation. Net smelter revenue is the value of payable metals sold net of smelter charges. The metal price assumptions used in the economic model are:

- Silver: $18 per ounce
- Zinc: $1.10 per pound
- Lead: $0.95 per pound

Net Smelter Revenue for the life of mine is summarized in Table 22-4.

Table 22-4: Net smelter revenue summary

Net Smelter Revenue LOM	$M
Revenues	
Lead concentrates - Pb	1,402
Lead concentrates – Ag	2,458
Zinc concentrates - Zn	1,144
Zinc concentrates – Ag	136.0

Net Smelter Revenue LOM	$M
Total gross revenue	**5,141**
Treatment and refining charges	540
Lead Concentrates	
Treatment charges	156
Price participation	-
Penalty	21.3
Silver refining charges	109
Zinc Concentrates	
Treatment charges	242
Price participation	2.6
Penalty	9.0
Total treatment and refining charges	**540**
Total net smelter revenues	**4,601**

22.5.5 Total operating cost

The average life of mine total operating cost is estimated to be $18.69 per tonne of processed ore, excluding the capitalised cost of mine pre-stripping. Total operating cost includes mine operations, process plant operations, general and administrative cost, including social costs, and concentrate transportation cost. Table 22-5 shows the estimated operating cost by area per tonne of processed ore.

Table 22-5: Life of mine operating cost

Operating Cost	$/ore tonne
Mine	4.29
Process plant	10.04
General and administration	1.88
Concentrate transportation	2.48
Total operating cost	18.69

22.6 Total Cash Cost

The average Cash Cost of production net of by-product credits over the life of mine is estimated to be $4.39 per payable silver ounce. The average total cash cost of production net of by-product credits over the life of mine on an all-in sustaining capital basis is estimated to be $4.54 per payable silver ounce. Total cash cost for the project is summarized in Table 22-6.

Table 22-6: Life of mine cash cost per ounce of silver

Silver Cash Cost Net of By-Product Credits	LOM ($M)
Production costs	2,590
Reclamation and closure	47.8
Subtotal	2,638
Net lead payable revenue	(1,116)

Silver Cash Cost Net of By-Product Credits	LOM ($M)
Net zinc payable revenue	(890)
Operating costs, net of by-product credits	631
Payable silver ounces	144
Silver cash cost net of by-product credits ($/lb Ag)	**4.38**
Sustaining capital expenditures	22.5
All in sustaining cash cost net of by-product credits ($/lb Ag)	**4.54**

22.6.1 Salvage value

A $6.9 million allowance for salvage value received at the end of the mine life has been included in the cash flow analysis

22.6.2 Reclamation and closure

An allowance of $47.8 million for the cost of reclamation and closure of the property in accordance with the closure plan approved by the Peruvian Ministry of Energy and Mines on September 12, 2018 has been included in the cash flow projection

22.7 Depreciation

Depreciation was calculated using following assumptions for both initial and sustaining capital.

- Mine development costs amortized over 3 years
- Process plant costs depreciated over 5 years
- Infrastructure costs depreciated over 10 years
- Other initial costs depreciated over 10 years
- Remaining depreciation at the end of the mine life will be taken in the last year of operation.

22.8 Taxation

22.8.1 Contribution to Organismo Supervisor de la Inversion en Energia y Mineria (“OSINERGMIN”)

The contribution to OSINERGMIN is applied to Net Smelter Revenue at a rate of 0.13%. It is estimated that $6.0 million of contribution to OSINERGMIN will be paid during the life of the mine.

22.8.2 Contribution to Organismo de Evaluacion y Fiscalizacion Ambiental (“OEFA”)

The contribution to OEFA is applied to Net Smelter Revenue at a rate of 0.11%. It is estimated that $5.1 million of contribution to OEFA will be paid during the life of the mine.

22.8.3 Royalty tax

The royalty tax is applied to operating profit at progressive rates from 1% to 12% based on operating margin (operating profit divided by sales), subject to a minimum tax of 1% of Net Smelter Revenue. It is estimated that $46.0 million of royalty tax will be paid during the life of the mine.

22.8.4 Special Tax Impuesto Especial a la Mineria ("IEM")

A special tax (IEM) is applied to operating profit at progressive rates from 2% to 8.4% based on operating margin (operating profit divided by sales). It is estimated that $12.7 million of special tax will be paid during the life of the mine.

22.8.5 Worker's participation tax

A labor profit sharing tax is based on pre-tax profits, after deduction for the royalty tax and special tax (IEM), and is assessed at an 8% rate. It is estimated that $42.2 million of labor profit sharing tax will be paid during the life of the mine.

A mandatory contribution paid to the Peruvian Mining Retirement Fund based on pre-tax profits, after deduction for the royalty tax and special tax (IEM), and is assessed at an 0.05% rate. It is estimated that $0.2 million will be contributed to the during the life of the mine.

22.8.6 Income tax

Income taxes are assessed on pre-tax profits at a rate of 29.5%, after deduction for the special tax (IEM), royalty tax, and worker's participation tax. It is estimated that $318 million of income taxes will be paid during the life of the mine.

22.9 Net Income After Tax

Net income after taxes for the project amounts to approximately $912 million. Taxes paid by the company are summarized in Table 22-7.

Table 22-7: Breakdown of taxes paid

Taxation	$ M
OSINERGMIN	6.0
OEFA	5.1
Royalty tax	46.0
Special mining tax (IEM)	12.7
Worker participation tax	42.2
Peruvian mining retirement fund tax	0.2
Income tax	318
Total tax	**430**

22.10 Project Financing

The financial model has been prepared on the assumption that the project will be financed 100% with equity.

22.11 Net Present Value, Internal Rate of Return, Payback

The economic analyses for the project are summarized in Table 22-8.

Table 22-8: Financial analysis results

Parameter	Pre-Tax	After Tax
NPV @ 0% ($000)	$1,383	$953
NPV @ 5% ($000)	$795	$532
NPV @ 8% ($000)	$571	$369

Parameter	Pre-Tax	After Tax
IRR (%)	28.1	22.9
Payback (years)	2.15	2.4

22.12 Sensitivity Analysis

The results of the sensitivity analysis for both before taxes and after taxes are shown in Table 22-9 to Table 22-14 and Figure 22-1 to Figure 22-6.

Table 22-9: NPV sensitivity analysis @ 5% - before taxes ($M)

D	Metal Prices	Operating Cost	Initial Capital	Recovery
+20%	$ 1,415	$ 521	$ 691	$ 1,337
+10%	$ 1,105	$ 658	$ 743	$ 1,066
0%	$ 795	$ 794	$ 795	$ 795
-10%	$ 475	$ 932	$ 847	$ 524
-20%	$ 153	$ 1,069	$ 899	$ 253

Table 22-10: NPV sensitivity analysis @ 8% - before taxes($M)

D	Metal Prices	Operating Cost	Initial Capital	Recovery
+20%	$ 1,057	$ 361	$ 473	$ 996
+10%	$ 814	$ 466	$ 522	$ 783
0%	$ 571	$ 571	$ 571	$ 571
-10%	$ 320	$ 676	$ 620	$ 358
-20%	$ 68	$ 781	$ 669	$ 146

Table 22-11: IRR% sensitivity analysis - before taxes

D	Metal Prices	Operating Cost	Initial Capital	Recovery
+20%	39.8%	22.6%	22.4%	38.5%
+10%	34.3%	25.5%	25.1%	33.5%
0%	28.1%	28.1%	28.1%	28.1%
-10%	20.7%	30.5%	31.7%	21.9%
-20%	11.3%	32.8%	35.9%	14.6%

Table 22-12: NPV sensitivity analysis @ 5% - after taxes ($M)

D	Metal Prices	Operating Cost	Initial Capital	Recovery
+20%	$ 946	$ 342	$ 456	$ 893
+10%	$ 739	$ 438	$ 494	$ 713
0%	$ 532	$ 532	$ 532	$ 532
-10%	$ 314	$ 625	$ 570	$ 348
-20%	$ 73	$ 717	$ 607	$ 151

Table 22-13: NPV sensitivity analysis @ 8% - after taxes ($M)

D	Metal Prices	Operating Cost	Initial Capital	Recovery
+20%	$ 698	$ 221	$ 294	$ 656
+10%	$ 534	$ 296	$ 332	$ 513
0%	$ 369	$ 369	$ 369	$ 369
-10%	$ 196	$ 441	$ 407	$ 222
-20%	$ 5.6	$ 513	$ 444	$ 67

Table 22-14: IRR% sensitivity analysis - after taxes

D	Metal Prices	Operating Cost	Initial Capital	Recovery
+20%	32.9%	18.0%	18.2%	31.8%
+10%	28.2%	20.6%	20.4%	27.5%
0%	22.9%	22.9%	22.9%	22.9%
-10%	16.7%	25.0%	26.0%	17.7%
-20%	8.3%	27.0%	29.6%	11.3%

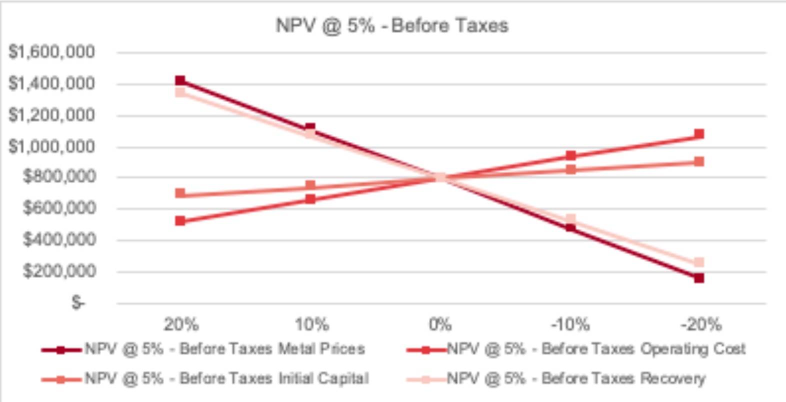


Figure 22-1: NPV sensitivity analysis @ 5% - before taxes

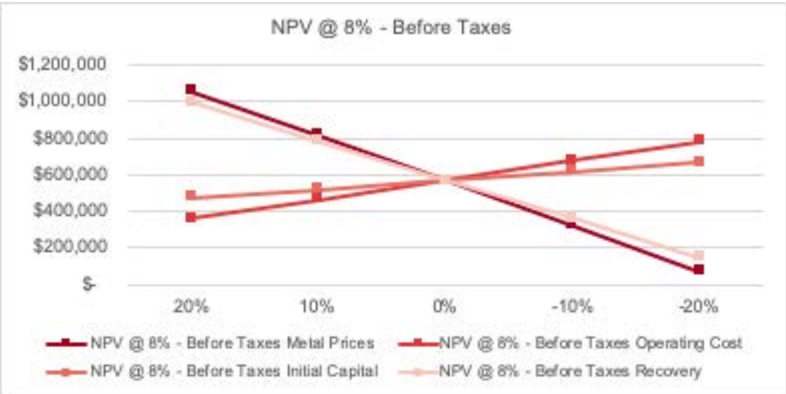


Figure 22-2: NPV sensitivity analysis @ 8% - before taxes

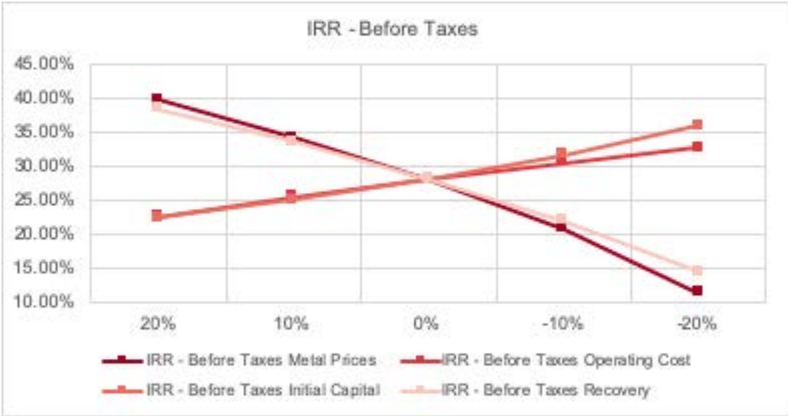


Figure 22-3: IRR% sensitivity analysis - before taxes

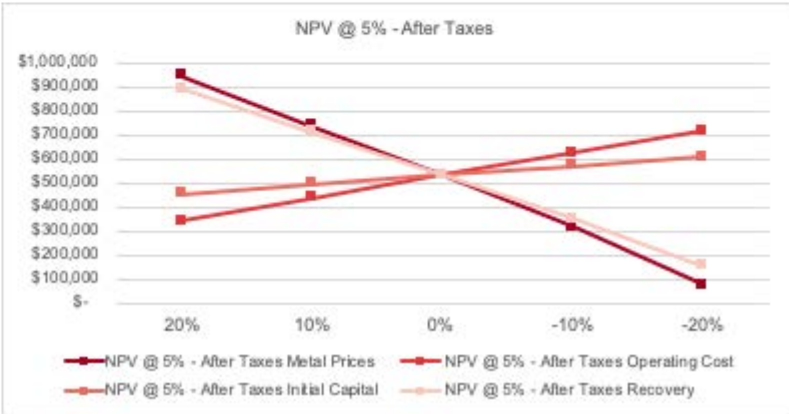


Figure 22-4: NPV sensitivity analysis @ 5% - after taxes

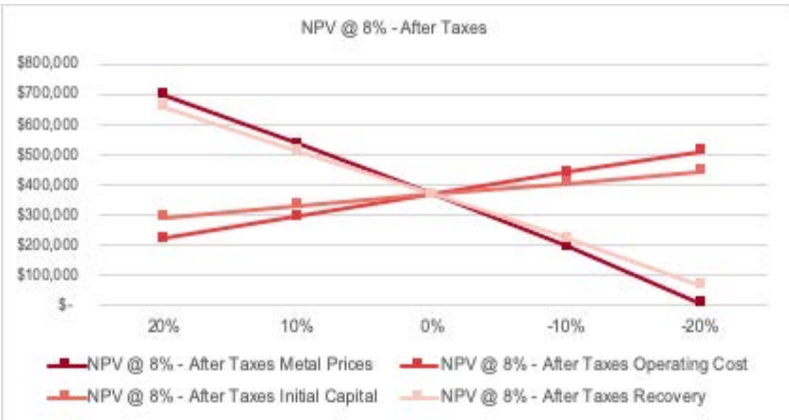


Figure 22-5: NPV sensitivity analysis @ 8% - after taxes

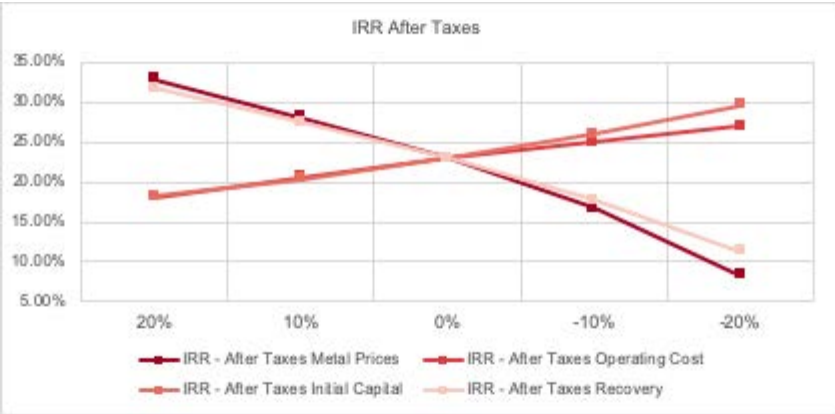


Figure 22-6: IRR% sensitivity analysis - after taxes

Table 22-15: Summary financial model

Parameter	Units	Total/Average	Prior Yrs	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
Price Assumptions																						
Zinc	$/Lb	$ 1.10						$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10	$ 1.10
Lead	$/Lb	$ 0.95						$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95
Silver	$/Oz	$ 18.00						$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00
Daily Production Rate Assumption	27,000	tdmt/day																				
Mining		Totals																				
Ore Mined	000's dmt	138,582		-	-	-	-	8,611	10,312	9,824	12,374	7,492	9,374	9,855	10,124	9,531	10,035	9,708	9,900	9,855	9,882	1,705
Waste Mined	000's dmt	177,881		-	-	-	-	19,246	17,206	19,289	11,654	10,890	7,648	9,505	9,831	8,382	14,714	20,544	17,517	6,338	3,972	1,143
Total Tonnes Mined	000's dmt	316,463		-	-	-	-	27,857	27,518	29,114	24,028	18,382	17,022	19,360	19,955	17,913	24,749	30,251	27,418	16,193	13,854	2,849
Strip Ratio		1.28		-	-	-	-	2.24	1.67	1.96	0.94	1.45	0.82	0.96	0.97	0.88	1.47	2.12	1.77	0.64	0.40	0.67
Silver Grade (g/t)	g/dmt	51.31						99.78	70.59	77.87	60.30	72.62	55.83	45.80	33.25	24.66	33.31	47.38	47.97	30.83	31.33	23.36
Zinc Grade (%)	% dmt	0.55%						0.84%	0.77%	0.69%	0.85%	0.20%	0.25%	0.29%	0.50%	0.82%	0.67%	0.37%	0.48%	0.43%	0.45%	0.70%
Lead Grade (%)	% dmt	0.90%						1.10%	1.03%	1.12%	1.34%	1.14%	0.95%	0.72%	0.70%	0.65%	0.65%	0.82%	0.71%	0.67%	0.92%	0.91%
Contained Silver	000's Oz	228,629						27,624	23,404	24,597	23,989	17,493	16,826	14,512	10,824	7,558	10,746	14,787	15,268	9,768	9,953	1,281
Contained Zinc	000's Lbs	1,694,361						159,171	174,064	148,566	232,328	33,273	50,879	63,004	110,924	173,153	148,127	78,358	105,742	93,015	97,574	26,183
Contained Lead	000's Lbs	2,741,745						207,966	235,190	242,212	366,625	188,682	195,741	156,898	156,447	136,463	144,375	176,280	155,799	144,678	200,247	34,141
Processing																						
Ore to Processing Plant	000's dmt	138,582						8,600	9,882	9,855	9,855	9,855	9,882	9,855	9,855	9,855	9,882	9,855	9,811	9,855	9,882	1,803
Ore to Tailings	000's dmt	136,141						8,396	9,639	9,633	9,568	9,743	9,791	9,762	9,705	9,647	9,702	9,731	9,659	9,708	9,697	1,762
Silver Grade (g/t)	g/dmt	51.31						99.77	70.88	77.83	61.51	68.85	55.89	45.80	33.39	24.95	33.13	47.33	47.93	30.83	31.33	24.86
Zinc Grade (%)	% dmt	0.55%						0.84%	0.77%	0.69%	0.84%	0.36%	0.28%	0.29%	0.50%	0.81%	0.67%	0.37%	0.49%	0.43%	0.45%	0.68%
Lead Grade (%)	% dmt	0.90%						1.10%	1.04%	1.12%	1.34%	1.18%	0.96%	0.72%	0.70%	0.65%	0.65%	0.82%	0.71%	0.67%	0.92%	0.90%
Contained Silver	000's Oz	228,629						27,587	22,520	24,661	19,491	21,813	17,755	14,512	10,579	7,905	10,527	14,996	15,119	9,768	9,953	1,441
Contained Zinc	000's Lbs	1,694,361						159,063	168,638	148,949	182,982	77,238	60,340	63,004	108,733	176,385	145,772	80,613	104,990	93,015	97,574	27,064
Contained Lead	000's Lbs	2,741,745						207,659	226,576	242,830	292,196	255,426	210,104	156,898	152,747	141,739	142,246	178,340	154,187	144,678	200,247	35,870
Payable Metals Produced																						
Zinc Concentrate																						
Zinc Concentrate Volume (DMT)	000's mt	1,047						107	112	93	121	36	25	28	64	115	93	44	71	58	60	18
Zinc Concentrate Volume (WMT)	000's mt	1,131						116	121	101	130	39	27	31	69	124	100	48	76	63	65	19
Concentrate Grade	%	53.09%						52.63%	53.09%	52.39%	52.16%	52.23%	52.48%	52.81%	52.69%	53.29%	53.78%	53.39%	53.62%	53.85%	54.35%	55.56%
Payable Silver (kozs)	000's Oz	7,558						814	852	706	915	270	191	214	447	563	700	336	537	442	457	115
Payable Zinc (klbs)	000's Lbs	1,039,874						105,701	111,805	91,218	117,551	34,805	24,689	27,949	63,120	115,111	93,724	44,393	71,160	58,879	61,432	18,336
Payable Zinc (kmt)	000's Mt	471,679						47,945	50,714	41,376	53,320	15,787	11,199	12,678	28,631	52,214	42,513	20,136	32,278	26,707	27,865	8,317
Lead Concentrates																						
Lead Concentrate Volume (DMT)	000's mt	1,394						97	131	129	166	77	65	65	85	93	87	80	81	89	125	24
Lead Concentrate Volume (WMT)	000's mt	1,492						104	140	138	178	82	70	70	91	99	93	86	87	95	134	26
Concentrate Grade	%	51.02%						49.79%	49.97%	51.78%	55.25%	52.01%	50.07%	49.80%	50.28%	51.90%	50.04%	50.01%	49.83%	50.04%	50.04%	50.04%
Payable Silver (kozs)	000's Oz	136,574						16,827	14,187	15,656	12,057	11,047	10,066	8,752	6,422	4,510	6,120	8,594	9,160	6,077	6,283	816
Payable Lead (klbs)	000's Lbs	1,476,041						100,185	135,664	138,687	191,663	82,667	67,796	67,080	89,070	99,797	90,281	82,950	83,754	92,151	129,486	24,810
Payable Lead (kmt)	000's Mt	669,522						45,443	61,536	62,907	86,937	37,497	30,752	30,427	40,401	45,267	40,951	37,626	37,990	41,799	58,734	11,254
Income Statement																						
Revenues																						
Zinc Concentrates - Zn		$ 1,143,861						$ 116,271	$ 122,985	$ 100,340	$ 129,306	$ 38,286	$ 27,158	$ 30,744	$ 69,431	$ 126,622	$ 103,097	$ 48,833	$ 78,276	$ 64,767	$ 67,575	$ 20,169
Zinc Concentrates - Ag		$ 136,040						$ 14,651	$ 15,341	$ 12,712	$ 16,469	$ 4,868	$ 3,434	$ 3,859	$ 8,037	$ 10,127	$ 12,593	$ 6,044	$ 9,658	$ 7,957	$ 8,227	$ 2,062
Lead Concentrates - Pb		$ 1,402,239						$ 95,176	$ 128,880	$ 131,753	$ 182,080	$ 78,534	$ 64,406	$ 63,726	$ 84,616	$ 94,807	$ 85,767	$ 78,803	$ 79,567	$ 87,543	$ 123,012	$ 23,570
Lead Concentrates - Ag		$ 2,458,334						$ 302,884	$ 255,358	$ 281,811	$ 217,021	$ 198,840	$ 181,181	$ 157,536	$ 115,592	$ 81,186	$ 110,164	$ 154,699	$ 164,885	$ 109,392	$ 113,094	$ 14,691
Treatment & Refining Charges		$ 540,866						$ 53,888	$ 57,944	$ 51,468	$ 60,013	$ 27,155	$ 23,219	$ 21,395	$ 30,555	$ 43,018	$ 38,232	$ 27,302	$ 34,317	$ 29,369	$ 35,244	$ 7,747
Total Net Smelter Revenues		$ 4,599,608						$ 475,094	$ 464,621	$ 475,148	$ 484,863	$ 293,373	$ 252,959	$ 234,470	$ 247,122	$ 269,725	$ 273,389	$ 261,077	$ 298,068	$ 240,290	$ 276,664	$ 52,745
Operating Cost																						
Mining Cost		$ 594,380						$ 53,292	$ 47,468	$ 52,984	$ 46,003	$ 38,589	$ 33,741	$ 37,711	$ 37,060	$ 33,711	$ 43,275	$ 57,016	$ 52,175	$ 29,839	$ 26,206	$ 5,311
Processing Cost		$ 1,391,030						$ 86,858	$ 97,061	$ 100,699	$ 94,933	$ 98,166	$ 96,890	$ 114,055	$ 97,107	$ 97,403	$ 97,833	$ 97,120	$ 98,104	$ 98,573	$ 98,264	$ 17,964
General Administration		$ 260,899						$ 18,042	$ 18,878	$ 18,539	$ 20,320	$ 16,910	$ 18,221	$ 18,562	$ 18,745	$ 18,331	$ 18,682	$ 18,454	$ 18,595	$ 18,562	$ 17,343	$ 2,714
Concentrate Transportation Cost		$ 343,681						$ 28,498	$ 34,158	$ 31,311	$ 40,464	$ 15,986	$ 12,950	$ 13,317	$ 21,051	$ 28,862	$ 25,109	$ 17,653	$ 21,305	$ 20,799	$ 26,365	$ 5,851
Total Operating Cost		$ 2,589,991						$ 186,690	$ 197,566	$ 203,533	$ 201,720	$ 169,652	$ 161,802	$ 183,645	$ 173,962	$ 178,306	$ 184,900	$ 190,244	$ 190,179	$ 167,773	$ 168,179	$ 31,840
Salvage Value		$ -6,903						$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (6,903)
Reclamation & Closure (Closure during LOM))		$ 24,906						$ 60	$ 48	$ -	$ -	$ 607	$ 607	$ 859	$ 1,553	$ 921	$ 607	$ 607	$ 783	$ 894	$ 5,419	$ 11,942
Reclamation & Closure (Post Mining Closure)		$ 22,927						$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Production Cost		$ 2,630,922						$ 186,750	$ 197,614	$ 203,533	$ 201,720	$ 170,259	$ 162,410	$ 184,504	$ 175,515	$ 179,228	$ 185,507	$ 190,851	$ 190,962	$ 168,667	$ 173,598	$ 36,879
Operating Income		$ 1,968,686						$ 288,344	$ 267,008	$ 271,615	$ 283,144	$ 123,114	$ 90,550	$ 49,966	$ 71,607	$ 90,497	$ 87,882	$ 70,226	$ 107,107	$ 71,623	$ 103,066	$ 15,866
Corani Amortization		$ 85,318						$ 22,368	$ 22,368	$ 22,368	$ 2,602	$ 2,602	$ 2,602	$ 2,602	$ 2,602	$ 2,602	$ 2,602	$ -	$ -	$ -	$ -	$ -
Corani Depreciation		$ 542,475						$ 73,483	$ 73,483	$ 74,451	$ 74,551	$ 74,945	$ 28,085	$ 28,085	$ 28,578	$ 28,578	$ 28,578	$ 5,843	$ 6,828	$ 5,860	$ 5,760	$ 5,366
Total Depreciation & Amortization		$ 627,793						$ 95,851	$ 95,851	$ 96,819	$ 77,153	$ 77,547	$ 30,687	$ 30,687	$ 31,180	$ 31,180	$ 31,180	$ 5,842.77	$ 6,827.77	$ 5,859.77	$ 5,759.77	$ 5,365.77
Net Income After Depreciation & Amortization		$ 1,340,893						$ 192,492	$ 171,156	$ 174,796	$ 205,990	$ 45,566	$ 59,862	$ 19,279	$ 40,427	$ 59,317	$ 56,702	$ 64,384	$ 100,279	$ 65,764	$ 97,307	$ 10,500
Interest Expense		$ -						$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Net Income After Interest		$ 1,340,893						$ 192,492	$ 171,156	$ 174,796	$ 205,990	$ 45,566	$ 59,862	$ 19,279	$ 40,427	$ 59,317	$ 56,702	$ 64,384	$ 100,279	$ 65,764	$ 97,307	$ 10,500
Taxes		$ 429,932						$ 8,812	$ 48,302	$ 34,820	$ 36,709	$ 36,091	$ (5,402)	$ 110,414	$ 6,269	$ 12,309	$ 20,600	$ 17,049	$ 27,407	$ 34,279	$ 21,077	$ 18,274
Net Income After Taxes		$ 910,962						$ 183,681	$ 122,854	$ 139,975	$ 169,282	$ 9,475	$ 65,264	$ (91,135)	$ 34,158	$ 47,008	$ 36,102	$ 47,335	$ 72,872	$ 31,484	$ 76,229	$ (7,774)
Cash Flow Statement																						
EBT		$ 1,340,893						$ 192,492	$ 171,156	$ 174,796	$ 205,990	$ 45,566	$ 59,862	$ 19,279	$ 40,427	$ 59,317	$ 56,702	$ 64,384	$ 100,279	$ 65,764	$ 97,307	$ 10,500
Plus Depreciation & Amortization		$ 627,793.05						$ 95,851	$ 95,851	$ 96,819	$ 77,153	$ 77,547	$ 30,687	$ 30,687	$ 31,180	$ 31,180	$ 31,180	$ 5,843	$ 6,828	$ 5,860	$ 5,760	$ 5,366
Change in Working Capital																						
Account Recievable (90% Cash and 10% 60 days)		$ -						$ (25,382)	$ 559	$ (562)	$ (519)	$ 10,230	$ 2,159	$ 988	$ (676)	$ (1,208)	$ (196)	$ 658	$ (1,976)	$ 3,087	$ (1,943)	$ 14,781
Accounts Payable (50% Cash and 50% 30 days)		$ -						$ 14,582	$ 848	$ 462	$ (142)	$ (2,457)	$ (613)	$ 1,725	$ (702)	$ 290	$ 490	$ 417	$ 9	$ (1,741)	$ 385	$ (10,675)
Change in Inventory		$ -						$ (12,177)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 12,177
Change in IGV		$ 14,957		$ 4,108	$ (2,020)	$ (1,173)	$ (93)	$ 14,136	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Change in Working Capital		$ 14,957		$ 4,108	$ (2,020)	$ (1,173)	$ (93)	$ (8,840)	$ 1,408	$ (100)	$ (661)	$ 7,774	$ 1,546	$ 2,713	$ (1,378)	$ (918)	$ 295	$ 1,075	$ (1,968)	$ 1,346	$ (1,558)	$ 16,282
Debt Financing		$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Less Initial Capital Expenditures		$ 579,272		$ -	$ 134,667	$ 212,900	$ 219,093	$ 12,612	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Less Sustaining Capital Expenditures		$ 22,500		$ -	$ -	$ -	$ -	$ -	$ 9,680	$ 1,000	$ 3,940	$ -	$ -	$ 3,940	$ -	$ -	$ -	$ 3,940	$ -	$ -	$ -	$ -
Cash Flow Before Taxes		$ 1,381,871		$ 4,108	$ (136,687)	$ (214,073)	$ (219,186)	$ 266,891	$ 258,735	$ 270,515	$ 278,543	$ 130,887	$ 92,096	$ 48,739	$ 70,229	$ 89,579	$ 88,177	$ 67,361	$ 105,139	$ 72,969	$ 101,508	$ 32,148
Less Taxes		$ 429,932		$ -	$ -	$ -	$ -	$ 8,812	$ 48,302	$ 34,820	$ 36,709	$ 36,091	$ (5,402)	$ 110,414	$ 6,269	$ 12,309	$ 20,600	$ 17,049	$ 27,407	$ 34,279	$ 21,077	$ 18,274
Cash Flow after Taxes		$ 951,939.67		$ 4,108	$ (136,687)	$ (214,073)	$ (219,186)	$ 258,079	$ 210,433	$ 235,694	$ 241,835	$ 94,796	$ 97,497	$ (61,675)	$ 63,960	$ 77,270	$ 67,577	$ 50,313	$ 77,732	$ 38,690	$ 80,431	$ 13,874

23 Adjacent Properties

There are no adjacent mineral properties which might materially affect the interpretation or evaluation of the mineralization or exploration targets of the Corani Project.

24 Other Relevant Data and Information

24.1 Project Execution

Ausenco has developed a Project Execution Plan (PEP) for the Corani project based on the concepts and scope outlined in this Report. BCM and Ausenco also reviewed the project Hazard and Operability study (HAZOP) presented in the 2017 Technical Report. Summaries of both documents are provided in this Section. These documents and the associated plans form the basis for the forward plan and developing the project on time and on budget, in a safe and efficient manner.

BCM is committed to carrying out its development activities of the Corani Project in a professional manner, in accordance with internationally agreed guidelines and principles and to be recognized for sustainable development and corporate social responsibility. The participation of the community, environmental stewardship and the promotion of a safe and healthy work environment are the cornerstones of this commitment. BCM also has the responsibility to promote its projects for the benefit of shareholders, to generate and sustain future growth, and to contribute to economic prosperity and social progress of Peru.

The PEP focuses on strategic drivers and organizational arrangements including roles, team alignment and general administration. The PEP will be revised as appropriate to reflect any significant changes in scope, schedule and execution strategy as the project matures.

The Corani Project scope has been divided into several distinct parts based upon factors including core competencies, social involvement, contractor interfaces, risk and cost. The overarching strategy is to manage the time and expenditure risks within acceptable tolerances for BCM.

The largest portion of the project is the combined construction of the process plant, accommodation camp, water ponds and associated infrastructure and earthworks. This scope may be executed as an EPC (Engineer, Procure, Construct) contract potentially with Ausenco (main EPC Contractor).

BCM will manage the construction contracts for the access roads, contact and non-contact water ponds, power delivery to the project, mine ancillary facilities, mine development and pre-stripping, and the preparation of the waste dump (DDMR).

- The access roads are divided into two parts. The Construction Access Road which runs from the town of Macusani to the accommodation camp at Jarapampa is a 42 km rural public road that will be pepared, maintened and improved for construction and further use during operations. The access road from the accommodation camp to the process plant is known as Internal Access 1 and will be constructed under an fixed price design and construct contract.
- The power delivery to the project is covered under a separate ESIA and provides a significant improvement in the reliability of the rural electrical supply alongside the construction of the power supply to the Project. This will be executed under a Build Own Operate Transfer (BOOT) contract where the contractor will finance the construction of the infrastructure and charge back the capital as part of a long-term operation and maintenance contract for the electrical supply.
- The mine will be operated by an experienced mine contractor on a unit rate basis. As a part of that contract, the contractor will supply the mining equipment and construct the mine facilities (Truck Shop, Explosives Storage, Offices, etc.). The waste dump, mine development, and pre-stripping are the major capitalised BCM scope.

24.1.1 Objectives

The Project goals are to achieve specific quality and performance standards while maintaining schedule and budgeted costs. A brief summary of the main objectives follows:

- Safety objectives: Zero harm to all personnel involved with the construction and commissioning of the Project.
- Environmental objectives: Zero avoidable environmental impact and zero environmental incidents.
- Community relations objectives: To achieve a harmonious social, economic, and cultural environment with neighboring communities. No negative impact on community relations. No community blockades or stoppages.
- Engineering and technology objectives: Ensure that contractors and their subcontractors use only proven technology and designs. Ensure contractor designs and equipment are consistent with all project standards.
- Quality work objectives: Ensure that adequate quality assurance and quality control procedures are effectively used and monitored.
- Cost and schedule objectives: Completing the project on time and below budget.
- Change management objectives: Create a BCM Project Group to establish and manage a system to document and manage variations in volumes and quantities, changes in prices, new unit prices, new scope for the project in accordance with the BCM's Capital Expenditure Policy.
- Procurement and purchase objectives: Ensure that the contracting and purchasing strategy is established and implemented to the maximum practical extent, restricting contracts and packages to a manageable number without increasing the total costs of the project.
- Risk management objectives: Assess all risks for the project and develop appropriate mitigation strategies for those risks.
- Capitalization objectives: BCM and the EPC/EPCM Contractor will develop a cost control strategy and a cost control system where each item to be capitalized shall contain the distributed indirect costs of the contractor. The system will be informed by all project costs, invoices, contracts, purchase orders, "as built" plans, photographic reports, and lists of all expenses incurred, so the BCM finance group can manage the BCM asset books and initiate the depreciation of the delivered facilities.

24.1.2 Scope of project execution plan

The PEP includes the following project execution documentation:

- early works program requirements
- project schedule
- project control and reporting plan
- engineering plan
- construction plan
- contracts and procurement plan
- document control plan
- health and safety plan

- environmental plan
- community relations plan
- quality plan
- site services plan
- permitting plan
- human resources plan
- operational readiness plan
- logistics plan
- commissioning plan
- risk management plan
- change management plan.

Each element describes the required planning, personnel, companies, systems, controls, work break down structures, procedures, permits, and schedules that need to be developed to efficiently and cost effectively develop the Corani Project.

24.2 Project Development Schedule

Project development continues, subject to financing, with engineering, early works, procurement on long lead items and permitting through the year 2020, construction activities continue through 2021, 2022 and 2023, with planned commissioning and startup the first quarter of 2024. The target schedule shown in Figure 24-1.

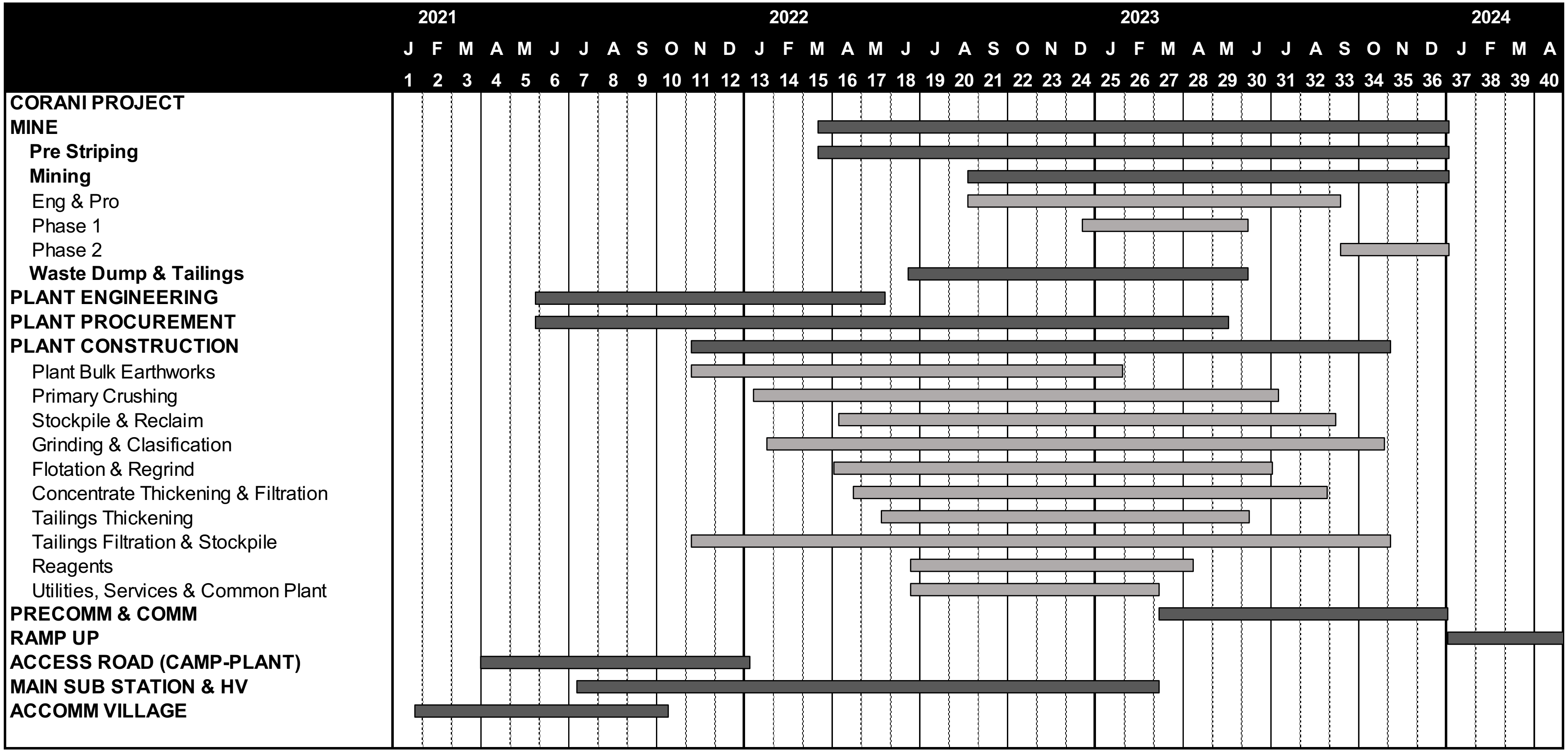


Figure 24-1: Target Project schedule

To date, the Project has obtained an approved ESIA for the Project from MINEM in 2013. Two ITS modifications of the ESIA were approved in 2016 and 2017. Bear Creek is planning to complete a third ITS modification to incorporate all of the final design optimizations. As required, the Project has completed an actualization of its final closure plan in 2018, which has been approved by the authorities.

Additional permits to support Project development have also been completed and approved, including:

- mining permit (2018)
- beneficiation concession (2018)
- all required archaeological clearances (CIRA; 2014-2018)
- water use permits (2017-2018)

24.3 Hazard and Operability Study

BCM's project team performed a project Hazard and Operability Study or HAZOP in 2017 and documented that work in the project HAZOP report. Ausenco reviewed that study and conducted further hazard identification studies as part of the updated engineering work reported herein. All hazards identified by Ausenco were mitigated with engineering solutions and reduced to moderate or low risk. Most of these hazards were associated with managing the effects of lightning strikes during construction and operation.

The 2017 HAZOP study was designed to identify operational risks as well as potential operational reliability issues, and then develop mitigation plans to guarantee these will not affect or influence the Safety and Environment of the Corani Project and subsequent operations. The goal was to have a risk-controlled facility able to meet international safety, security, and environmental standards.

The study had three basic steps: (1) identification of potential operational risk and reliability issues or aspects, (2) determining the consequence of that issue or aspect, including identifing all entities effected, and (3) determining the probability of that issue or aspect occurring and ranking the magnitude of such an occurrence. The HAZOP study included a method of combining the ranking the consequence and issues or aspects in order probability, to rank the most significant redesign issues or to form mitigation plans for them. Further HAZOP studies are required during the engineering, execution and operational readiness phases of the project.

The 2017 study identified 313 risk points, with 20 of the points being high risk, 105 moderate risk, and 188 low risk. The study had 53 recommendations, as shown below. Ausenco reviewed and classified these recommendations as either to be addressed in operational readiness process or to be addressed in design. The majority of the recommendations will be addressed by good design and project planning.

Table 24-1: 2017 HAZOP recommendations

Classification	2017 Recommendation	Project Action
Operational procedures and operations	Generate a common procedure of operations for the Mine and Plant to control haul truck and vehicle traffic and their destinations.	Address as part of operational readiness program

Classification	2017 Recommendation	Project Action
readiness processes	The maintenance manual should indicate the need for maintenance and cleaning of the area. The parameter of analysis was the frequency of arrival of trucks from the mine, in which the risk is that the trucks at the point of unloading result in an accident, loss of truck, loss of production, etc. In general cases, berms should be used at the platform perimeters and in the unloading point. Safe procedures will always be utilized, and training should be included in the operations manual.	Address as part of the operational readiness program and mine haul road design
	Review spare parts and corrective maintenance activities for belt conveyors, as well as for pan feeders; the operations manual shall contain the spare parts recommendations for corrective and preventive maintenance.	Address in operational readiness program
	Perform plant start-up according to operations	Address in commissioning and start-up plan
	Generate protocols and maintenance manuals for plant equipment, in this specific case of the trommel and SAG mill. The operations manual should contain recommendations for the maintenance of the plant equipment. Based on this information, the operator will prepare its maintenance plan and procedures, instructions, inspection records to be used in maintenance activities.	Address in VDDR process and operational readiness planning
	Review with supplier the need for lubrication system protection.	Address in operational readiness
	Include a review of the operation of the agitator in the commissioning and maintenance procedures.	Address in operational readiness
Design considerations	Review the free surface for the stockpile according to the mining plan, in addition to the ore management philosophy; the design includes an area to stockpile ore.	Review as part of mining execution
	Maintenance of ditches and rainwater collection system to ensure proper function throughout the mine life.	Review in site detailed design
	The plan will confirm the need for a roof over the hopper during normal precipitation, and, in case of torrential rains, the crushing operation must be stopped, having as backup load live in the stockpile	Ignore – not necessary
	Review the seepage design; the design contemplates waterproofing of the platform.	Manage by design
	Review the dust mitigation design to avoid the lack of visibility in the area; the design should contemplate installation of nebulizers to mitigate dust in the environment. Based on a specialist evaluation, the project has contemplated the installation of a nebulizer in the area.	Manage by design in plant Review water requirement for dust suppression in the mine area – consider alternative approaches.
	The available spaces must be secured for adequate maintainability; the mechanical arrangement of crushing has been designed considering the spaces for the mentioned activities, as well as facilities for lifting.	Normal design
	To avoid accumulation of ore, the discharge chute has been dimensioned at 100%.	Manage by design

Classification	2017 Recommendation	Project Action
	For power outages, the belts and pan feeders must be designed to start with load. The plan will review the need for an estimated time needed for an alternate source of in case of a blackout.	Normal design
	Perform preventive maintenance of water pump under pressure (water jet).	Engineer out requirement
	For the SAG mill, review recommendations by suppliers on spaces and services required; the design contemplated the requirement and the recommendations of the suppliers.	Normal design
	The plan will review the location and protection of the lubrication system. It will ensure that that the indications and recommendations of the seller of the SAG mill with respect to the location and protection of the lubrication are properly considered	Normal design
	Illumination is being considered with level of required luxes indicated along with the project design criteria; the design should contemplate localized lighting.	Normal design
	The risk for damage to the bearings due to lack of lubrication in the case of an electrical failure. An available control to consider is an oil lubrication design which does not loose pressure. The recommendation is to review the maintenance and cleaning plan and check the backup system for powering the equipment auxiliary / mill services.	Manage by design
	Design to prevent spills and allow access for cleaning from accidental events	Normal design
	For the pump boxes for the mills and in general for all pump boxes, the design should consider level controls to prevent spills.	Normal design
	In general, for pump boxes, evaluate the use of radar sensors.	Manage by design
	Drainage lines and cleaning lines have been considered for pump suction and discharge pipes, as well as gutters for the concrete slabs.	Manage by design
	In general, for the whole plant, consider maintenance access; the design has contemplated the spaces and access for maintenance.	Manage by design and maintainability reviews
	For the feeding of cyclone, the operation conditions and instrumentation for the measurement of solids should be considered.	Normal design
	Evaluate in the next stage of engineering the use of backup air or consider the use of a motorized valve (with padlock)	Manage by design for each circumstance
	Engineering should consider the distribution boxes of the flotation cells, as well as for the entire floatation area, nearby water points for cleaning.	Normal design
	The engineering shall review the structural design of the pulp distribution box to the flotation cells to confirm that they have considered the effect of vibrations and other forces.	Normal design
	The detailed engineering design should consider installing protective grating on top of the pump boxes	Generally not included

Classification	2017 Recommendation	Project Action
	The design should consider control options for the presence of foam inside the feed box.	Normal design
	For thickener rakes, as well as for the agitators of feed tanks to the filters, check the power supply back up system.	Normal design
	Engineering contemplates emergency sumps for the thickeners.	Excluded other than ability to capture thickener contents within bunded area under thickeners
	For areas of thickeners in the event of a spill, engineering includes containment such as gutters.	See above

The 2017 HAZOP also raised the impact of ore blending and loss of power supply as high risks. Further work has been completed by BCM to address ore supply management.

Project execution risks were not addressed by the 2017 HAZOP or the Ausenco hazard analysis. However, the PEP has considered the major issues impacting on project execution and early operation, namely:

- Project elevation and people's productivity and wellness. The high altitude and adverse weather of the site may have a greater-than-expected negative impact on worker productivity that would delay the construction.
- Although local communities have generally supported the Project development, there is a risk that sentiments could change, or that special interest groups from outside the community could mobilize opposition to the Project.
- The high altitude of the site may result in greater-than-expected impacts on the function and capacity of diesel-powered equipment and electrical components.
- Potential for unrecognised geotechnical issues.
- There are other mining companies close to Corani, the risk is that they want to connect to the new 138 kV line for their electricity supply.
- Water storage and supply for project start-up.
- The potential impacts of build up of dissolved solids in the recycle water on process performance.
- A currency exchange risk exists. Metal sales receipts would be priced in United States dollars ($). To the extent that the Peruvian Nuevo Sol (PEN) appreciated relative to the $ locally sourced supplies and services, labour and government payments denominated in PEN would be more expensive in $ terms. Conversely, if the PEN depreciated relative to the $ such PEN denominated expenditures would be less expensive in $ terms.

The magnitude of some of these risk elements is difficult to quantify. Mitigations for site related risks (elevation and lightning) include Ausenco's and other contractors experience in constructing projects at altitude in Peru. Initial water availability may vary depending on rainfall during the construction period and BCM will continue to assess the impact of changes in water quality on plant performance.

25 Interpretation and Conclusions

25.1 Geological Setting and Mineralization

The Corani Project area is located in the northern part of Puno Department, southern Peru, within the Cordillera Oriental of the Central Andes. The Project area is underlain by Tertiary volcanic rocks of the Quenamari Formation, specifically a thick series of crystal-lithic tuffs and andesite flows, which overlie variably deformed Lower Paleozoic to Mesozoic metasediments of the Ambo and Tarma Groups. The primary host of mineralization is the Chacaconiza Member of the Quenamari Formation. The Chacaconiza is the youngest member of the Quenamari and is comprised of a sequence of crystal-lithic and crystal-vitric-lithic tuffs. The tuffs are widely hydrothermally altered and pervasively argillized to low-temperature clays, and are variably faulted, fractured, and brecciated.

Mineralization at the Corani Project occurs in three distinct and separate zones: Corani Main, Corani Minas, and Corani Este, each differing slightly in character with regard to both alteration and mineral assemblages. In general, mineralization in outcrops throughout the Corani Project is associated with iron and manganese oxides, barite, and silica. Silicification is both pervasive and structurally controlled along veins. In drill core, the mineralization occurs in typical low to intermediate sulfidation Ag-Pb-Zn mineral assemblages. The most abundant silver-bearing mineral is fine-grained argentian tetrahedrite or freibergite.

Structurally, the Corani deposit is situated within a stacked sequence of listric normal faults striking dominantly north to north-northwest with moderate to shallow (50° to <10°) westerly dips. The hanging walls of the listric faults are extensively fractured and brecciated, providing the structural preparation for subsequent or syngenetic mineralization. The stacked listric faults are more prominent in the Corani Minas and Corani Main areas. The Corani Este area contains a single known listric fault with an extensively fractured and brecciated hanging wall. The contact with the underlying Paleozoic sediments corresponds locally to listric faults dipping shallowly to the west.

25.2 Deposit Types

The Corani deposit is best described as a low- to intermediate-sulfidation epithermal deposit with silver, lead, and zinc mineralization hosted in stock works, veins, and breccias. Mineralization is principally located in a set of lístric faults dipping west, with dilational segments related to subvertical structures and breccias in the hanging wall, and veinlets forming stockworks in the footwall. Structural control of the mineralization is a product of extensional tectonics that developed the series of north- to northwest-trending fractures and faults, and whose movements provided the structural preparation for the influx of mineralizing hydrothermal fluids.

25.3 Exploration

BCM began exploring the Corani Project in early 2005. In addition to drilling, exploration activities carried out by BCM include detailed geologic mapping, trenching, and geophysical surveying.

BCM has conducted general geologic surface mapping over the entire Project area. The total mapped surface is about 4.5 km wide (east-west) and 7.5 km long (north-south). In 2015, detailed surface mapping, including lithology, alteration, and structures, was performed at a scale of 1:2500 in the area of the proposed pits.

BCM has completed 25 trenches within the Project resource area (Corani Main, Minas, and Este) to verify the continuity of the structures covered by Quaternary sediments. Spacing between the trenches is roughly 50 to 100 meters. Channel samples from these trenches have produced an associated 1,295 assay intervals for a total of 2,924 meters of trench data.

VDG del Perú S.A.C. (VDG) conducted a ground geophysical campaign at the Corani Project on behalf of BCM in the fall of 2005. A total of 44.20 line-km of induced polarization (IP) data was collected, along with 50.95 line-km of magnetic survey. The geophysical surveys were aimed at assisting in geological mapping, including lithologies and key structures and at mapping mineralization and alteration associated with a low sulfidation gold-silver system. The objective of the IP/Res survey was to map the electrical response by means of high-resolution IP traverses across the favorable north-south corridor identified based on the results of both trench and drilling exploration. The field results of both methods were of good quality and were meaningful. The final chargeability and resistivity depth sections mapped systematically clear contrasts from line to line between the sub-surface and a nominal depth of 283 meters below surface. The chargeability outlined five (5) IP anomalies, two of which correspond to the Corani Main and Corani Este areas, respectively. Those anomalies accurately mapped the known mineralization and extended the size of both mineralized zones.

25.4 Drilling

Since 2005, BCM has completed a total of 562 drillholes at the Corani Project for a total of approximately 101,400.57 m. Drilling was completely by the Peruvian contractor, Bradly MDH primarily using LD250, JKS35, and LJ44 drill rigs. All of the drilling to date has been completed using diamond core drilling methods to produce either HQ (6.35 cm dia.) or NQ (4.76 cm dia.) core. Core recoveries are generally excellent, with no discernible variation in rate of recovery between the two core sizes (HQ and NQ). While on site, the QP carefully reviewed the drilling and sampling procedures employed by BCM with BCM staff. Based on that review, the QP finds no drilling, sampling, or recovery factors that might materially impact the accuracy or reliability of the drilling results.

Between 2006 and 2011, BCM undertook an in-fill drill program of the known areas of mineralization in order to increase confidence in the resource estimate. This was accomplished by in-fill drilling to produce a nominal 50-m drillhole spacing in previously more widely spaced drilling areas, with a focus on areas of higher grade mineralization. In general, drilling exploration has identified and further defined the distribution of mineralization within the three primary resource areas, Corani Main, Corani Este, and Corani Minas. Drilling results indicate that significant mineralization occurs in two basic forms; in large veins associated with the principal listric fault structures, and in stockwork veins found in the surrounding rocks.

In 2018, BCM completed a total of six drillholes of DDH-C84-MET, DDH-C152-MET, DDH-CTJ30-MET, DDH-CM13-MET, DDH-CM14-MET, and DDH-C144-MET at the Corani Project for a total of 906 m including three holes in Minas and three holes in Main areas. These holes were drilled primarily for metallurgical testing.

25.5 Sample Preparation

BCM employs standard, basic procedures for both drill core and trench sample collection and analysis. Formal chain of custody procedures are maintained during all segments of sample transport. Samples prepared for transport to the laboratory are bagged and labelled in a manner which prevents tampering, and remain in BCM control until released to private transport carrier in Cusco or Juliaca. Upon receipt by the laboratory, samples are tracked by a blind sample number assigned and recorded by BCM. The samples are prepped according to ALS-Chemex preparation code PREP-31, and silver, lead, zinc, and copper assays are carried out by three-acid digestion followed by atomic absorption spectrophotometry (AA) analysis. Multi-element inductively coupled plasma (ICP) analysis is conducted on select sample intervals to assist with mineralization classifications and to guide the interpretation of the metallurgical process response.

BCM maintains an internal Quality Assurance/Quality Control (QA/QC) program which includes both standard and check (lab) sampling. GRE conducted a critical review of BCM's QA/QC program; toward that end, BCM provided GRE with QA/QC data in multiple Excel spreadsheet files. GRE compiled the data into a single, comprehensive QA/QC data worksheet for analysis and evaluation. Based on the results of GRE's review, in conjunction with observations and conversation with BCM personnel during the QP site visit, BCM's routine sample preparation, analytical procedures, and security measures are, in general, considered reasonable and adequate to ensure the validity and integrity of the data derived from BCM's sampling programs. GRE recommends that BCM expand the existing QA/QC program to include at least standards, blanks, and duplicates, and that QA/QC analysis be conducted on an on-going basis, including consistent acceptance/rejection tests. Each round of QA/QC analysis should be documented, and reports should include a discussion of the results and any corrective actions taken.

25.6 Data Verification

Data verification efforts included an on-site inspection of the Corani Project and core storage facility, check sampling, and manual and mechanical auditing of the Project database. Field observations during the site visit generally confirm previous reports on the geology of the Project area. Bedrock lithologies, alteration types, and significant structural features are all consistent with descriptions provided in existing Project reports, and the author did not see any evidence in the field that might significantly alter or refute the current interpretation of the local geologic setting.

Specific core intervals from 35 separate drill holes were selected for visual inspection and potential check sampling based on a preliminary review of the drill hole logs and associated assay values. In all cases, the degree of visible alteration and evidence of mineralization observed was generally consistent with the grade range indicated by the original assay value. Laboratory analysis was completed by ALS Peru using the same sample preparation and analytical procedures as were used for the original samples. Standard t-Test statistical analysis was completed to look for any significant difference between the original and check assay population means. The results of the t-Test showed no statistically significant difference between the means of the two trials (original versus check assay).

GRE completed a QA/QC audit of the digital Project database by comparing a random selection of original assay certificates to the assay information contained in the Corani Project database. Results of the QA/QC audit indicate a minor and acceptable error rate. GRE completed a mechanical audit of the Project database in order to evaluate the integrity of data from a data entry perspective. GRE also conducted a review of BCM's QA/QC 2019 program for the six metallurgical drill holes. BCM provided GRE with QA/QC data multiple excel spreadsheet files. GRE analysed the data and prepared related plots of the standards and blank samples for Ag, Cu, Pb, and Zn.

Based on the results of the QP's check sampling effort, verification of drill hole collars in the field, visual examination of selected core intervals, the results of both manual and mechanical database audit efforts, and standard and blank review, the QP considers the collar, lithology, and assay data contained in the Project data base to be reasonably accurate and suitable for use in estimating mineral resources and reserves.Mineral Processing and Metallurgical Testing

25.7 Environmental Studies, Permitting and Social or Community Impact

The Project is well advanced on the environmental and social aspects needed to support the future development. Required permits to support the overall project and construction are in place, and those to support operations are defined and included in a permitting plan. Project optimizations have reduced the disturbance footprint and water use, which have reduced the potential environmental and social impacts. In support of permitting, there has been a robust geochemical testing program, development of a stochastic water balance, and supporting baseline data collection. Air and noise quality are good and there are no problematic biodiversity concerns, though development of a biodiversity action plan is recommended as the Project proceeds. Baseline data indicates the occurrence of acidic surface waters and that several metals exceed the Peruvian environmental standards (ECA). This is a result of natural geological conditions and historic mining impacts. The Project has authorizations for all required water and the use of filtered tailing technology will reduce water demands. All recovered process water will be re-used for process make-up water and there will be no discharge of mine-contact water to the environment. Collected non-contact water will be utilized to maintain environmental flows in the Chacaconiza drainage. A detailed closure plan has been developed, including closure costs, and has been approved by the Peruvian authorities. The Project has completed extensive community engagement and has developed good community relations with the two small rural communities (Chacaconiza and Quelcaya) close to the planned facilities. Positive relations are also supported by a signed Life of Mine ("LOM") Investment Agreement with the District of Carabaya, five surrounding communities, and relevant, ancillary organizations specifying investment commitments over the 23 year project life.

25.8 Mineral Resource Estimates

GRE updated the mineral resources for the Corani project incorporating six new drill holes, modelling of the transition material separate from oxide and sulfide mineralization, and an updated geometallurgical model.

Like previous estimates, the resource model has three main lithologies: basement sediment with minor quantities of mineralization, the mineralized (pre-mineral) tuff, and a mostly unmineralized post mineral tuff which is assumed to be barren. Mineralization has been defined by 9 mineralization types, which were later grouped into, oxidized, transition, and sulfide groups. The mineral resources for the Corani project are shown in Table 25-1. The Mineral Resources were generated within the $30.00/troy ounce silver, $1.425/lb lead, and $1.50/lb zinc price Whittle pit shell and the calculated $10.79/tonne NSR cut-off. Table 25-2 shows the potentially leachable mineral resource contained within the Whittle pit shell at a 15 g/t cut-off and $7.23/tonne leach cost.

Table 25-1: Total mineral resources (includes both resources and reserves)

Category	Tonnes (000)	Silver g/t	Lead %	Zinc %	Silver Moz	Lead Mlb	Zinc Mlb
Measured	30,585	50.0	0.79	0.49	49.1	534	329
Indicated	208,050	40.9	0.64	0.43	273.5	2,933	1,985
Measured + Indicated	**238,635**	**42.1**	**0.66**	**0.44**	**322.7**	**3,466**	**2,313**
Inferred	73,185	35.5	0.40	0.30	83.5	641	484

Note: Cutoff Value: $10.79/tonne covers process and general and administrative costs.

Table 25-2: Total mineral resource of potentially leachable material (includes the mineral reserve)

Category	Tonnes (000)	Silver g/t	Silver Moz
Measured	4,302	28.9	4.0
Indicated	36,104	30.1	35.0
Measured + Indicated	40,406	30.0	39.0
Inferred	24,311	38.2	29.9

25.9 Mineral Reserve Estimates

The Mineral Reserve Estimate is based on the updated 2019 geometallurgical and resource block models, and an ultimate pit and phase designs by BCM.

The project Mineral Reserves consider only measured and indicated resource categories, which have been converted to proven and probable reserves categories, respectively. Mineral Reserves are defined as being the material to be fed to the process plant and are demonstrated to be economically viable this study. The 2019 Corani Mineral Reserves are shown in Table 25-3.

Table 25-3: Corani Project mineral reserves

Classification	Tonnes Kt (dry)	Grade			NSR $/t	Contained Metal		
		Silver g/t	Lead %	Zinc %		Silver Moz	Lead Mlb	Zinc Mlb
Proven	20,330	59.7	1.00	0.60	34.02	39.0	450.0	268.5
Probable	118,253	49.9	0.88	0.55	29.48	189.6	2,292	1,426
Total Proven + Probable	**138,582**	**51.3**	**0.90**	**0.55**	**30.15**	**228.6**	**2,742**	**1,694**

Notes:

1. The Mineral Reserves have been estimated using the definitions of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM).
2. The Mineral Reserves have been estimated using the following metal prices: $20.00/oz Ag, $1.00/lb Zn, $0.95/lb Pb using a revenue factor 1.00 pit shell as a basis for the pit design.
3. Only pre-mineral tuff type of material has been considered as reserves.
4. NSR Cutoff grades used are equal or higher than: $10.79/t.
5. The effective date for these Mineral Reserves is 5 November 2019.
6. Totals / Averages may not add up due to rounding of individual tonnes and grades.
7. The tonnes and grades shown above are considered a Mineral Reserve because they have been demonstrated to be economically viable through this report's study financial model using the following metal prices: $18.00/oz Ag, $1.10/lb Zn, $0.95/lb Pb.

25.10 Mining Methods

The Corani Project will be mined using conventional open pit mining methods using contractor mining. San Martín Contratistas Generales S.A. has prepared a bid to conduct the mining over the 16 year project life. The rock will be broken by drilling 0.17-m diameter blast holes and blasting with ANFO and emulsion. Broken rock will then be loaded into 140 tonne (Cat 785) trucks using two 22 m3 Cat 6040 hydraulic shovels. A 994 front end loader is provided for backup. Support equipment includes two D9 bulldozers, a road grader, water trucks, rubber tire dozer, compactor, excavator, fuel and lube trucks, and other miscellaneous equipment.

During a 10-month pre-production stripping, pioneering, haul road construction phase prior to plant construction, 4.7 million tonnes of waste rock will be mined to generate construction material. Another 13.81 million tonnes will be mined in the 7 months immediately prior to production. Table 16-3 shows the Life of Mine production schedule.

25.11 Market Studies

There are no established contracts for the sale of concentrates currently in place for this project.

BLB Advisory completed a "Corani Marketing and Sale Logistics Study", dated September 2019. Due to its high silver grade, the lead concentrate will be attractive to smelters with silver recovery capabilities, while the zinc concentrate will be more marketable. Transportation methods for lead and zinc concentrate have been developed and integrate with the plant design and port capabilities.

The high chromium level in lead concentrate samples was demonstrated to be related to contamination by stainless steel from the test work and/or sample preparation process.

25.12 Economic Analysis

The economic analysis, summarised in Table 25-4, resulted in a material increase in project value when compared with the 2017 NI 43-101 report report. This included:

- a $127 million (31%) increase in after-tax Net Present Value5 ("NPV5")
- a 52% increase in after-tax Internal Rate of Return ("IRR") from 15.1% to 22.9%
- a 1.2 year (33%) reduction in the payback period
- lower All-In-Sustaining-Costs ("AISC")
- significantly reduced construction, development and operating risks.

Table 25-4: Financial summary

Parameter	2019 Report*
After tax NPV_5	$531 million
After tax IRR	22.9 %
Initial capital	$579 million
Capital payback	2.4 years
Ore processed per day	27,000 tonnes
AISC per oz silver (Life of Mine ("LOM"))	$4.55
Average annual silver production (LOM)	9.6 million oz

25.13 Other Relevant Data and Information

The project scope will be managed by a main contractor and BCM, with BCM managing the mine prestripping, bulk earthworks and some infrastructure.

Ausenco prepared two execution plans for the process plant and associated site infrastructure, one based on a reimbursable EPCM approach and the other based on a fixed price EPC approach reflecting the additional risk in the cost estimate.

BCM has commenced some site early works in preparation for project execution.

26 Recommendations

26.1 Ausenco Recommendations

Geotechnical data

Ausenco recommends that additional work be done to ensure that the currently planned site layout is optimal from a geotechnical standpoint. Some of the assumptions made in designing project facilities require field verification. Specific areas requiring additional field evaluation include:

- building foundations
- primary crusher structure, conveyor supports
- project support facilities foundation requirement review
- roadways
- DDMR foundation
- pit slopes.

Several facilities were relocated during the project optimization, and foundation characterization has yet to be performed for some of the new locations. Standard geotechnical drilling and foundation characterization needs to be performed to allow detailed design of the facilities. These include buildings such as the new truck shop, the new primary crusher location, and supports for the new conveyor alignment. Foundation requirements for several support facilities also need to be evaluated.

Early works program

Early works programs need to be progressed in order to achieve and maintain the target project schedule.

Engineering and design

Engineering and design has progressed to a sufficient level for the cost estimates in this report. Further detailing of the engineering and design approach based on the contracting and procurement strategy is required to optimise the project schedule and detail vendor strategies, including potential sole sourcing and effective vendor data management.

Concentrate quality

Further work is required to check chromium content in lead and zinc concentrates and confirm Cr grade in zinc concentrate in particular. This should be included in future metallurgical programs.

Execution planning

Detailed definition of the execution and contracting plan is required.

26.2 GRE Recommendations

Drilling and exploration

GRE recommends that BCM produce annual (or seasonal) exploration reports to describe the drilling and sampling carried out during each given year or drilling campaign. The exploration report should contain adequate detail concerning the drill rig, drilling contractor, number of holes, total meters, recovery rates, drill targets, and rationale for drill hole distribution, etc., to ensure that all pertinent information is captured and catalogued in a practical and efficient manner for ease of future use.

As additional drilling is completed, precision surface surveys and down hole surveys should be implemented for all future drilling.

<u>Data verification, sample preparation, analysis, and security</u>

The issues identified during the 2017 Technical Report have largely been addressed. However, the insertion of blanks, standards, and duplicates is still not being performed consistently. GRE recommends that the procedure be standardized to ensure that sufficient QA/QC samples are inserted into the sample stream.

<u>Mineral processing and metallurgical testing</u>

The following recommendations have been made:

- The test work should focus on the maximization of silver recovery to the lead concentrate. This will help to maximize the value of the recovered silver.
- Additional comminution test work should be conducted focusing on individual mineral types instead of pit zones or composites.
- Additional thickening and filtration testing should be undertaken for the main mine waste and filtered tailings deposit design examining individual mineral types. Composites representing the projected mining sequences may also be utilized, providing they are reasonably representative.
- Additional work needs to be conducted on the oxide and transitional portion of the deposit.
- The geometallurgical model was developed using all metallurgical testing data, including tests representing non-optimized conditions. Once additional testing has been performed, samples should be selected that represent optimized test conditions. Using those sample data, the statistical model should be re-evaluated to ensure estimated recoveries represent optimal conditions.

<u>Mineral resource estimates</u>

Improvements in the estimation of the transition zone material could improve the estimate of the amount of material that has lower recovery. This work should be done to help with the creation and implementation of a practical ore control procedure to use once operations begins.

26.3 Anddes Recommendations

<u>Main mine waste and filtered tailings deposit</u>

Limited geotechnical tests on filtered tailings were completed as part of the 2017 Feasibility Study and recently some shear strength and consolidation characteristics has been carried out. However, additional testing is required as soon as additional tailings samples are available in order to verify shear strength, saturated and unsaturated hydraulic conductivity, dynamic properties among others.

Updated geotechnical modelling of the main mine waste and filtered tailings deposit must be performed based on new laboratory data, including seepage and seismic response analysis. The consolidation of tailings under loading must be updated to determine if excess pore pressure conditions are occurring and to determine the seepage behaviour of tailings under geostatic loading.

Once deposit begins operations and during the life of the project, it is recommended that MASW and MAM geophysical tests be completed to obtain the shear wave velocities profile in the field. Although conservative parameters have been used in the 1D seismic response analysis, actual dynamic parameters will be needed to determine for avoiding potential problems during the operation.

A geotechnical testing program should be prepared as part of the operation manual to determine geotechnical properties of the main waste and filtered tailings once the disposal in the deposit start, with the objective of verifying the assumptions taken as part of the design and ensure proper stability conditions.

Geotechnical monitoring instrumentation must be installed in certain areas in order to monitor displacements, groundwater into the foundation, pore-water pressure into the co-disposed materials and seismic behavior of the facility through accelerographs, allowing the monitoring of geotechnical stability. Also, alert levels and operation and contingency manual, should be developed for the whole mine facilities. A best industry practice should be applied in the development of those documents, mainly for the critical components. Those documents and manuals will support the routine operation and will produce a safe environment.

A group of specialists, including geotechnical, hydrologist, hydrogeochemists and mechanical engineers, should carry out Annual Safety Reviews of the whole facilities, taking into consideration the Canadian Dam Association guidelines. The Safety Review Report should update the monitoring plan, operation and contingency manual, stability condition, water management, hydrogeochemical behavior, critical equipment operation of each mine facility and provide action lists for improving the functioning of them. Regardless the Annual Safety Review, all the facilities will have to be inspected after the occurrence of an extreme storm or earthquake event.

Plant water pond

Due to the geotechnical characteristics of the Plant Water Pond foundation, a good quality control and quality assurance program should be implemented during construction, including geoelectrical surveying for verifying potential geomembrane defects which produce leaks.

Water balance

The probabilistic water balance employs updated models from 2019 to predict pit dewatering and mine waste and filtered tailings deposit seepage quantities. These models are expected to be sufficiently accurate for this level of study. However, they must be updated once operations start to reflect the revised mine plan and the planned modifications of the deposit operation methods. Data gathering at piezometric network, surface water controls and a geochemical investigation plan will contribute to generate robust data to update quantity and quality models.

Once these models are revised and new results are obtained, the water balance must be remodeled to ensure the project can meet the required water commitments in the ESIA. The surface water management plan will likely require updates to meet the new water balance.

Pit geotechnics

Additional geotechnical drilling should be completed within the planned pit for confirming the current pit slope design basis and potentially allow an increase in the pit slope angles. Boreholes should be completed as monitoring wells, and multiple-well aquifer testing should be performed to better assess the dewatering requirements for the material. Also, additional geochemical tests should be performed to assess and ensure closure activities. Detailed pit slope design and soil mining plans must then be developed.

26.4 BCM Recommendations

Market studies and contract

The smelters selected for the study should be contacted to verify their capacity and ability to accept the proposed quantity and quality of produced lead and zinc concentrates. As part of the program, additional concentrate analysis should be completed to further define the concentrate qualities. Letters of intent would be desirable for future project financing and associated due diligence along with firm quotes from the smelters. In addition, firm quotes should be obtained to confirm the transportation costs for the concentrate to the Matarani port as well as port handling costs.

Materials sources and balance

The earthworks mass balance should be developed integrating the scopes of the Main Access 1, process plant earthworks and mine pre-stripping. This could optimize earthworks haulage, reduce possible double handling or waste material.

26.5 Amphos Recommendations

The development of the Corani Project should consider continued monitoring of the environmental parameters (water, air, noise, biology, etc.) approved by the authorities in Peru, according to the original ESIA (AMEC, 2012). Collected data will allow predictive hydrological and hydrogeochemical models to be updated.

26.6 Estimated Cost for Recommended Work

During the course of the work on this Project, the contributors developed several recommendations for future consideration and execution by BCM. The estimated costs to complete the recommended work that are not directly related to project execution are shown in Table 26-1.

Table 26-1: Estimated costs for recommended work

Item	Estimated Cost ($)
Main dump and topsoil geotechnical	30,000
Mine geotechnical	150,000
Metallurgy	250,000
Water balance	15,000
Environmental / social / permitting	25,000
Total	**450,000**

Project execution risk can be reduced through further detailed planning, site geotech and early contractor involvement for sub-contract and major fabrication activities. However, this work should not be commenced until sufficient project finance is in place to ensure a continuous progression to construction completion.

27 References and Reference Information

Alex Doll, P. M. (2014). Comminution Modelling Report. Logan Lake, BC: Alex G. Doll Consulting Ltd.

AMEC. (2012). Estudio de Impacto Ambiental del Proyecto Corani. Lima, Peru.

Amphos 21 (2014). Proyecto Corani: Integración y Adecuación de Estudios Geoquímicos. 228_14. Abril 2014, 91pp.

Amphos 21 (2017). Actualización Hidrológica, Hidrogeológica y Geoquímica. Proyecto Corani. Octubre 2017 1105.10.11-5-200-12-ITE-001. 418pp.

Amphos 21 (2018). Primera Actualización del Plan de Cierre de Minas del Proyecto Corani. 438_18-ITE-001 R0. Abril 2018. 1926pp.

Base Metallurgical Laboratories Ltd. (2019). Metallurgical Assessment of the Corani Lead-Zinc-Silver Deposit. Kamloops, BC.

Blue Coast Metallurgy Ltd. (2011). Bear Creek Mining Corporation Summary of Metallurgical Testing of Corani 2007-2011. Parksville, BC.

Clark, A. F. (1990). Geologic and geochronologic constraints on the metallogenetic evolution of the Andes of southeastern Peru. ECONOMIC GEOLOGY, v. 85, 1520−1528, 1549.

Dawson Metallurgical Laboratories Inc. (2006). Bear Creek Mining Corp. Final Report Describing Laboratory Test Work on Samples from the Corani Project in Peru. Salt Lake City, UT. Project No. P-2894. September, 2006.

Desrochers, J. (2005). Structural analysis of the Corani prospect, Peru: Unpublished report to Bear Creek Mining.

Estudio Grau Abogados. (2015). Bear Creek Mining Corporation – Corani Project. Legal opinion on the status of claims for the Corani Project. GA - 327/15. June, 2015. Lima, Peru.

G&T Metallurgical Services Ltd. (2006). Preliminary Metallurgical Assessment LL Composite Corani Silver Gold Prospect - Peru Bear Creek Mining Corporation KM1880. Kamloops, BC.

G&T Metallurgical Services Ltd. (2007). Establishing a Geometallurgical Response Profile Bear Creek Mining Corporation Corani Deposits Peru. Kamloops, BC.

G&T Metallurgical Services Ltd. (2009). Advanced Development Studies Investigating Silver Leaching Bear Creek Mining Corporation Corani Deposits KM2307. Kamloops, BC.

Gagliuffi Espinoza, P. (2006). Estudio Microscópico de once (11) muestras de rocas correspondientes a testigos de perforación procedentes del Proyecto Corani. unpublished.

Ganoza, J. (2019). Final Testing Program 2019 Final Report. Bear Creek Mining Company.

Global Resource Engineering. (2019, November 1). Copy of Corani Sequence Schedule 20190904 - v9-tal. Denver, CO, USA.

Golder Associates Peru S.A. (2019). Tailings Laboratory Test Work. Lima, Peru.

Gunning, G. M. (2007). A QEMSCAN (Quantitative Evaluation of Minerals Scan—registered trademark of PEI Company) investigation into metallurgical test products from the Corani deposit:. Unpublished report to Bear Creek Mining.

Hazen Research Inc. (2006). Mineralogy of Bear Creek Silver Ore Samples Hazen Project 10404. Golden, CO.

Hazen Research Inc. (2006). Mineralogy of Bear Creek Silver Ore Samples. Hazen Project 10404, July 2006. Hazen Research Inc.

Independent Mining Consultants, Inc. (2006). October 4, 2006, Corani Project Mineral Resource Technical Report. Tucson, Arizona, United States.

Independent Mining Consultants, Inc. (2008). May 12, 2008, Technical Report, Corani Resource Estimate and PEA. Tucson, Arizona, United States.

Jenike and Johanson Chile S.A. (2019a). Flow Property Test Results and General Recommendations for Handling Several Crushed Ores. Vina del Mar, Chile.

Jenike and Johanson Chile S.A. (2019b). Flow Property Test Results and General Recommendations to Handle and Store the Filtered Tailings Corani Project Report #68513-2. Vina del Mar, Chile.

Kontak, D. C. (1990). Late Paleozoic-early Mesozoic magmatism in the Cordillera de Carabaya, Puno, southeastern Peru: Geochronology and Petrochemistry. ournal of South American Earth Sciences, v. 3, 213-230.

Kontak, D. C. (1991). Geochronology and petrology of the San Gabán igneous complex of southern Peru: Implications for Triassic-Jurassic felsic magmatism and Sn-W metallogeny in the Central Andes. Geological Association of Canada-Mineralogical Association of Canada with the Society of Economic Geologists, Program with Abstracts, 16, A67.

Laubacher, G. (1978). Géologie de la Cordillere Orientale et de l'Altiplano au nord et au nord-ouest du Lac Titicaca (Pérou). ORSTOM, Trav. Doc., v. 95, 217.

Li, V. (2016). The Uranium Mineralization of the Macusani District, Southeast Peru: Mineralogy, Geochemistry, Geochronology and Ore-Genetic Model. Kingston, Ontario, Canada: PhD Thesis, Queens University.

M3. (2011). Form NI 43-101F1 Technical Report Feasibility Study. Tuscon, Arizona: M3 Engineering and Technology Corporation. December, 2011.

M3. (2015). Optimized and Final Feasibility Study, Corani Project. Tuscon, Az. July 2015: M3 Engineering and Technology Corporation.

MEG Technical Services. (2014a). Laboratory Compaction Characteristics of Soil Using Standard Effort (ASTM D 698) CM-02. MEG Consulting Limited.

MEG Technical Services. (2014b). Laboratory Compaction Characteristics of Soil Using Standard Effort (ASTM D 698) CM-03A. MEG Consulting Limited.

MEG Technical Services. (2014c). Laboratory Compaction Characteristics of Soil Using Standard Effort (ASTM D 698) CM-04. MEG Consulting Limited.

MEG Technical Services. (2014d). Laboratory Compaction Characteristics of Soil Using Standard Effort (ASTM D 698) CM-06. MEG Consulting Limited.

MEG Technical Services. (2014e). Laboratory Compaction Characteristics of Soil Using Standard Effort (ASTM D 698) CM-08. MEG Consulting Limited.

MEG Technical Services. (2014f). Laboratory Compaction Characteristics of Soil Using Standard Effort (ASTM D 698) CM-12. MEG Consulting Limited.

Nelson, R. (2006). Field geological analysis of the Corani project, Peru. Unpublished report to Bear Creek Mining.

Outotec. (2014). Filtration Test Report 109981T1.

Outotec. (2019a). Filtration Test Report PEF 19-02-R0. Lima, Peru.

Outotec. (2019b). Thickening Test Report PET - 19-04 - R1. Lima, Peru.

Outotec. (2019c). Thickening Test Report PET - 19-05 - R0. Lima, Peru.

Petersen, C. (1967). Korani Mining District: Unpublished Cerro De Pasco Corporation internal memorandum. (May 22, 1967), p. 17.

Petersen, E. (2005). Corani project progress report. Unpublished report to Bear Creek Mining.

Pocock Industrial, Inc. (2008). Flocculant Screening and Gravity Sedimentation Testing Conducted for SGS Canada Inc. (Vancouver Metallurgy) Corani Project. Salt Lake City, UT.

Prado, A. (2008). Estudio estructural del yacimiento epitermal Ag-Pb- Zn-Au de Corani: Tesis de Maestría en aprovechamiento sostenible de los recursos minerales. Lima, Perú: Unpublished M.S. thesis, Universidad Mayor de San Marcos.

Sakuhuni, G. and Holuszko, M. (2011). Rheological Analysis Coroni Tailings. The University of British Columbia, Norman B. Keevil Institute of Mining. Vancouver, BC: SGS.

Sedgman. (2017). NI 43-101 Technical Report Corani Project Detailed Engineering Phase 1 (FEED). Santiago, Chile. Sept 2017: Sedgman Chile SpA.

SGS Canada Inc. (2012). A Report on 2010-2011 Testing of the Recovery of Silver, Lead and Zinc from Corani Samples prepared for Bear Creek Mining Corporation Project 50000-007 - Final Report. Vancouver, BC: SGS Group (SGS SA).

SGS Lakefield Research Limited. (2008a). Grindability Characteristics of Samples from the Corani Project submitted by Bear Creek Mining Corporation Project CAVM-50000-003 - Final Report. Lakefield, ON: SGS Group (SGS SA).

SGS Lakefield Research Limited. (2008b). Ore Characterization and Preliminary Grinding Circuit Design Using CEET2(R) Technology Based on Samples from the Corani Deposit Prepared for Bear Creek Mining Project Project CAVM-50000-005 - report for review. Vancouver, BC: SGS Group (SGS SA).

SGS Minerals Services. (2010). Grinding Testwork on Samples from Corani Project Prepared for Bear Creek Mining Company Project OL - 4406. Santiago, Chile: SGS Group (SGS SA).

SGS Minerals Services. (2012). Grinding Testwork on Samples from Corani Project Prepared for Bear Creek Mining Company Final Report Project OL-4709. Santiago, Chile: SGS Group (SGS SA).

SGS Vancouver Metallurgy. (2007). An update Report on the Recovery of Silver, Lead and Zinc from Corani Samples prepared for Bear Creek Mines Project MV 50000-001. Vancouver, BC: SGS Group (SGS SA).

SGS Vancouver Metallurgy. (2008a). A Report on 2008 Testing of the Recovery of Silver, Lead and Zinc from Corani Samples Prepared for Bear Creek Mines Project VM 50000-004. Vancouver, BC: SGS Group (SGS SA).

SGS Vancouver Metallurgy. (2008b). An update Report on the Recovery of Silver, Lead and Zinc from Corani Samples prepared for Bear Creek Mines Project VM 50000-001. Vancouver, BC: SGS Group (SGS SA).

SGS Vancouver Metallurgy. (2009). An Investigration into the Mineralogy and Flotation of Samples from the Corani Deposit Prepared for Bear Creek Mining Corporation - Project 50000-006. Vancouver, BC: SGS Group (SGS SA).

SRK Consulting. (2006). March 31, 2006, National Instrument 43-101 Technical Report, Initial Resource Estimate for Corani Silver-Gold Exploration Project. Tucson, Arizona, United States.

Swarthout, A. L. (2010). The Discovery History and Geology of Corani: A Significant New Ag-Pb-Zn Epithermal Deposit, Puno Department, Peru. Littleton, Colorado, USA: Society of Economic Geologists Special Publication No. 15.

Ullrich, T. (2016). Report on Ar40/Ar39 geochronological analysis: Unpublished report to Bear Creek Mining.

Vector Perú S.A.C. (2009). October 14, 2009, NI43-101 Technical Report, Prefeasibility Study Corani Project Puno Perú, . Lima, Perú.

Walsh Environmental. (2014). Preliminary Closure Plan, September 2014.

27.1 List of acronyms

Acronym	Definition
AA	Atomic absorption
ABA	Acid-Base Accounting
Ag	Chemical symbol for silver
Amphos 21	Environmental consultant
ANA	Water National Authority
AP	Acid (generating) Potential

Acronym	Definition
ARD	Acid Rock Drainage
As	Chemical symbol for arsenic
BCM	Bear Creek Mining Corporation
bcm	Bank cubic meter
Bi	Chemical symbol for bismuth
BLB	BLB Advisory – concentrate marketing and sales consultant
CAPEX	Capital expenditure
CATV	Cable TV distribution system
CFP	Cumulative frequency plot
CIF FO	Cost insurance and freight free out
CM	Construction manager
Co	Chemical symbol for cobalt
CS	Coarse-grained silica-sulfide
CSA	Canadian Securities Administrators
CSC	Coarse-grained silica-sulfide-celadonite
Cu	Chemical symbol for copper
DCS	Distributed Control System
DDMR	Main Waste Rock and Filtered Tailings Deposit (Deposito de Desmonte de la Mina y Relaves Principal)
EDO	Emulsified Diesel Oil
EPC	Engineering Procurement and Construction
EPCM	Engineering Procurement and Construction Management
ESIA	Environmental and Social Impact Assessment
F	Chemical symbol for fluorine
FBS	Fine-grained black silica-sulfides
Fe	Chemical symbol for iron
FEED	Front End Engineering Design
FeO	Iron oxide
FS	Feasibility Study
G&T	G&T Metallurgical Services
GA	General arrangement

Acronym	Definition
GFA	General facilities arrangement
GMI	GMI S.A. Ingenieros Consultores
GPS	Global positioning system
GRE	Global Resource Engineering Ltd.
HAZOP	Hazard and operability study
HDPE	High density polyethylene
Hg	Chemical symbol for mercury
ICP	Inductively-coupled plasma
ID2.5	Inverse distance to the 2.5 power
ID3	Inverse distance to the 3rd power
IDP	Inverse distance to a power
IEM	Impuesto Especial a la Minería (Special Mining Tax)
IFC	International Finance Corporation
IGV	Impuesto General a las Ventas (Peruvian value added tax)
ITS	Informe Tecnico Sustentatorio
IMC	Independent Mining Consultants
INACC	Instituto Nacional de Concesiones y Catastro Minera
INGEMMET	Geologic Mining and Metallurgical Institute (Instituto Geológico Minero y Metalúrgico)
IP	Induced polarization
IRA	Inter-ramp angles
IRR	Internal Rate of Return
LCT	Locked cycle test
LOM	Life of Mine
M3	M3 Engineering and Technology Corporation
MARC	Maintenance and repair contract
MgO	Chemical symbol for magnesium oxide
MINEM	Ministerio de Energía y Minas
Mn	Chemical symbol for manganese
MnO	Chemical symbol for manganese oxide
NAG	Non Acid Generating

Acronym	Definition
NI 43-101	Technical Report of the Canadian Securities Administrators National Instrument 43-101
NNP	Net Neutralization Potential
NP	Neutralization Potential
NPV	Net Present Value
NSR	Net Smelter Return
OEFA	Organismo de Evaluación y Fiscalización (Agency for Environmental Assessment and Enforcement)
OPEX	Operating expenses
OSINERGMIN	Organismo Supervisor de la Inversión en Energía y Minería (Supervisory Agency for Investment in Energy and Mining)
PAG	Potentially Acid Generating
Pb	Chemical symbol for lead
PDS	Power distribution center
PE	Plan of execution
PEA	Preliminary Economic Assessment
PEN	Peruvian New Sol (currency)
PEP	Project Execution Plan
PFS	Prefeasibility Study
PG	Plumbogummite
PM	Pyrite marcasite + quartz
PMT	Post mineral tuff
QA/QC	Quality analysis / quality control
QEMSCAN	Quantitative evaluation of minerals by scanning electron microscopy.
QP	Qualified Persons
QSB	Crystalline quartz sulfide-barite
R_2	Coefficient of determination
RF	Revenue factor (Whittle)
ROI	Return on Investment
Sb	Chemical symbol for antimony
SEDGMAN	Sedgman Chile SpA, Engineering Company
SGS	SGS Mineral Services Laboratory

Acronym	Definition
SOW	Scope of work
Sph	Spherical
GyM	GyM S.A. Mine Engineering Design Company
TC	Treatment charge
TET	Silver-bearing tetrahedrite
TS	TS Technical Services (Tom Shouldice)
TSF	Tailings storage facility
UEA	Corani Administrative Economic Unit
UTM	Universal Transverse Mercator
VDG	VDG del Perú S.A.C.
Zn	Chemical symbol for zinc

27.2 Glossary

Term	Definition
Bofedal	Organic soil found in the wet areas in the central parts of the valley
Campesino	A term in Spanish meaning farmer
Quebrada	A term in Spanish, meaning gorge, valley or draw.
Tailing	Finely ground materials from which the desired mineral values have been largely extracted. Typically, approximately 98 per cent of the material mined for processing is discharged as tailings
Waste rock	Material such as soils, barren or uneconomic mineralized rock that surrounds a mineral ore body and must be removed in order to mine the ore. This is generally referred to as waste rock in metalliferous mines.

27.3 Units of measure

Unit Abbreviation	Definition
cm	centimetre
d	day
ft	foot
g	gram
g/t	gram per metric tonne (metric), equivalent to Parts Per Million

Unit Abbreviation	Definition
h	hour
ha	hectare
hp	Horsepower
kg	kilogram
kg/t	kilogram per tonne (metric)
km	kilometer
km^2	Square kilometers
kt	kilotonne
kW	kilowatt
kWh	kilowatt hour
kWh/t	kilowatt hour per tonne (metric)
lb	pound
m	meter
m^2	square meter
m^3	cubic meter
Ma	million years
masl	meters above sea level
Mbcm	million bank cubic meters
min	minutes
mm	millimeters
Moz	million troy ounces
MPa	million Pascals
Mt	million tonnes
oz	Troy ounce
oz/t	ounces per dry metric tonne
ppm	parts per million
t	tonne (metric)
t/d	tonnes (metric) per day
t/h	tonnes (metric) per hour
t/m^3	Tonnes per cubic meter (density)

Unit Abbreviation	Definition
t/y	tonnes (metric) per year
t (wet)	wet metric tonne
μm	micrometer (microns)
$/t	dollars per dry tonne
$/oz	dollars per payable ounce
$/t (wet)	dollars per wet tonne
%	percent