Address: West End Office Park,
Building C, 3rd Floor,
246 Hall Street, Centurion,
Gauteng, South Africa
Tel: +27(0)12 644 1181
Fax: +27(0)12 644 1185

A&B GLOBAL MINING

Exhibit 96.2

Hindustan Zinc Limited – SEC - SK 1300

Technical Summary Report

Rampura Agucha Mine

Document Version:	**Rev1**
Customer Name:	**HZL Mine**
Date:	**20 July 2022**
Prepared by:	**ABGM**
Document Number:	**2022-07-25-RAM-SEC**

PREPARED AND REVIEWED BY:

TITLE	PREPARED / REVIEWED	SIGNATURE	DATE
Project Manager	Colin Little		28/07/2022
Resource Geologist	Dr Heather King		28/07/2022
Resource Geologist	Mpai Mosiuoa		28/07/2022
Mine Engineer	Hans Erasmus		28/07/2022
Geotechnical Engineer	Pieter Groenewald		28/07/2022

ACCEPTED AND APPROVED BY:

TITLE	NAME	SIGNATURE	DATE
Qualified Person – Resources	Andre van der Merwe		28/07/2022
Qualified Person - Reserves	Bill Gillespie		28/07/2022
Managing Director - ABGM	Devendra Vyas		28/07/2022

Disclaimer:

This document will serve only for the Purposes of Hindustan Zinc Limited (HZL) based on information received from the mine and its respective mining operations in India namely: Rampura Agucha Mine; Rajpura Dariba; Sindesar Khurd; Kayad and the four Zawar operations – Balaria, Baroi, Mochia and Zawarmala. The information within this technical report is aligned to the Securities Exchange Commission (SEC), SK-1300 guidelines and the information within is to be use for this purpose only.

LIST OF ABBREVIATIONS:

%	Percentage
^0C	degree Celsius
ABGM	A & B Global Mining Consultants
Ag	Silver
BOQ	Bill of Quantities
CAPEX	Capital expenditure
Coeff. Of Variation	coefficient of variation
COG	Cut-off Grade
Con	Concentrate
CSD	calc-silicate dolomites
g/t	grams per ton
GMS	graphite-mica schist
GSSA	Geological Society of South Africa
HDPE	High Density Polyethylene
Hr.	Hour
HZL	Hindustan Zinc Limited
IDW^2	Inverse Distance Weighting to the power of two
INR	Indian Rupee
kA	KiloAmpere
kL	KiloLitre
km	Kilometre
kN/m^2	KiloNewton per square metre
Koz	Kilo Ounces
Kt	Kilo tonne
kV	KiloVolt
lb	Pound
LHD	Load Haul Dumper
LOM	Life-of-Mine
LOM	Life of Mine
m	metre

m^2	Squared Metre
m^3	Cubic Metre
m^3/hr	Cubic metres per hour
mRL	Mean Relative Elevation
Mt	Million tonnes
mtpa	Million tonnes per annum
OGL	Original ground level (original surface elevation)
OK	Ordinary Kriging
OPEX	Operating expenditure
oz	Ounces
Pb	Lead
PbEQ	Lead Equivalent
PFS	Preliminary Feasibility Study
QA	Quality Assurance
QC	Quality Control
QA/QC	Quality Assurance & Quality Control
RAM	Rampura Agucha Mine
RAUG	Rampura-Agucha underground
RDM	Rajpura Dariba Mines
ROM	Run-of-Mine (ore/rock of economic value containing the target mineral(s)
ROM	Run of Mine
RPEEE	reasonable prospects of eventual economic extraction
SACNASP	South African Council for Natural Scientific professions
SEC	Securities Exchange Commission
SKM	Sindesar Khurd Mine
Std Dev	standard deviation
t	tonnes
TRS	Technical Review
USD	Us Dollar
USD/g	US Dollar per gram
USD/pb	US Dollar per pound

USD/t	US Dollar per tonne
VDS/PDS	Vehicle Detection System / Personnel Detection System
ZAW	Zawar Complex
Zn	Zinc
ZnEQ	Zinc Equivalent

Glossary of Terms

Block model: This is the cubical representation in 3 dimensions of the mineral resource. The block model data is usually constructed using industry accepted geological software packages.

Concentrating: The process of separating milled ore into a waste stream (tailings) and a valuable mineral stream (concentrate) by flotation.

Orebody: A well-defined mineralised rock mass that can be defined or modelled based upon its distinct mineral content or associated rock type/lithology.

Run of Mine (ROM): A loose term used to describe ore produced from the mine available for processing.

Tailings: That portion of the ore from which most of the valuable material has been removed by concentrating and that is therefore low in value and rejected.

Tonne: Metric ton, equal to 1 000kg, unless otherwise defined.

Total Stations: Surveying tools which comprise of an electronic theodolite and an electronic distance meter/measurement component

Finance

Capital expenditure (CAPEX): Total capital expenditure on mining and non-mining property, plant, equipment, and capital work-in-progress.

Effective tax rate: Current taxation, deferred taxation, and tax normalization as a percentage of profit before taxation.

IRR: Internal Rate of Return (the discount rate at which the project “NPV” becomes zero).

NPV: Nett Present Value (cash flow of the project discounted to current day value – includes project OPEX and CAPEX).

Operating expenditure (OPEX): Total operating expenditure for mining and non-mining functions pertaining to the project.

Definitions:

The following Definition apply to this report and are aligned to meanings ascribed in terms of internationally recognized institutions and standards namely the Canadian CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM) . The Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC). The South African Code for Reporting of Exploration Results, Mineral Resources and Mineral Reserves (SAMREC)

Mineral Resources:

A **'Mineral Resource'** is a concentration or occurrence of material of intrinsic economic interest in or on the earth's crust in such form, quality and quantity that there are reasonable prospects for eventual economic extraction. Mineral Resources are further sub-divided, in order of increasing geological confidence, into inferred, indicated and measured as categories.

Inferred Mineral Resource is the part of a mineral resource for which quantity, grade (or quality) and mineral content can be estimated with a low level of confidence. It is inferred from geological evidence and assumed but not verified geological or grade continuity. It is based on information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes which may be of limited or uncertain quality and it is also reliability.

Indicated resources are simply economic mineral occurrences that have been sampled (from locations such as outcrops, trenches, pits and drill holes) to a point where an estimate has been made, at a reasonable level of confidence, of their contained metal, grade, tonnage, shape, densities, physical characteristics.

Measured resources are indicated resources that have undergone enough further sampling that a 'competent person' (defined by the norms of the relevant mining code; usually a geologist) has declared them to be an acceptable estimate, at a high degree of confidence, of the grade (or quality), quantity, shape, densities, physical characteristics of the mineral occurrence.

<u>Mineral Reserves</u>

Mineral Reserve is the economically mineable part of a Measured Mineral Resource and/or Indicated Mineral Resource. Mineral Reserves are subdivided in order of increasing confidence into **Probable Mineral Reserves** or **Proved Mineral Reserves**.

Probable Mineral Reserve is the economically mineable part of an Indicated Mineral Resource, and in some circumstances, a **Measured Mineral Resources**. It includes diluting material and allowances for losses which may occur when the material is mined. A Probable Mineral Reserve has a lower level of confidence than a Proved Mineral Reserve but is of sufficient quality to serve as the basis for decision on the development of deposit.

Proved Mineral Reserve is the economically mineable part of a **Measured Mineral Resource**. It includes diluting materials and allowances for losses which occur when the material is mined.

Proved Mineral Reserve represents the highest confidence category of Mineral Reserve estimate. It implies a high degree of confidence in the geological factors and a high degree of confidence in the Modifying Factors. The style of mineralization or other factors could mean that Proved Mineral Reserves are not achievable in some deposits.

Generally the **conversion** of Mineral Resources into Mineral Reserves requires the application of various Modifying Factors, including, but not restricted to:

- mining factors.
- mineral processing / ore dressing related factors.

- metallurgical factors.
- infrastructure factors.
- economic factors.
- marketing factors.
- legal factors.
- ESG factors: Environmental, Social (including Health and Safety) and Governance

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1 EXECUTIVE SUMMARY

1.1 Property summary and ownership

The Rampura Agucha Mine ("RAM") zinc-lead mine is located in central Rajasthan, approximately 400 km southwest of Delhi. The mine is located in an area of flat terrain with poor agricultural land use. The Company applied for a new prospecting permit covering the surrounding area which was secured during 2010. Production in the open pit commenced in 1991 and was eventually closed in March 2018. Underground mining was initiated in 2013. The production has increased from 3.33 Mt in 2019 to 4.27 Mt in 2021. The current mining lease has been extended until 2030.

1.2 Mineral Resource Statement

The Mineral Resources described in this Item are based on appropriate geoscientific information, economic and technical parameters, and grade and tonnage estimation processes. The Mineral Resource estimates were determined using ordinary kriging (OK) geostatistical methodology and considered sample lengths, grade capping / cutting, the spatial distribution of drill holes and the quality assurance and quality control results for the analytical sample grades determined. Geological modelling and grade estimation used Datamine software

The Mineral Resources exclusive of the Mineral Reserves as at the end of the last fiscal year are summarised in Table 1, whilst Table 2 summarises the same period where the Mineral Resources are inclusive of the Mineral Reserves.

The Mineral Resources Table 16 and Table 17 are reported at a ZnEQ COG of 4.15% for the Main Zone, and a PbEQ COG of 4.41% for the Galena Zone. Depletions as of 31 March 2022 have been removed.

Table 1: Mineral Resource Statement (exclusive of Mineral Reserves) – 31 March 2022

Classification	Tonnage	Grade			Metal Content		
	(Mt)	Zn (%)	Pb (%)	Ag (g/t)	Zn (Kt)	Pb (Kt)	Ag (Koz)
Measured	10.2	14.70	2.20	64	1,498	223	21,014
Indicated	0.1	17.60	3.00	80	20	3	290
Measured + Indicated	**10.3**	**14.70**	**2.20**	**64**	**1,517**	**227**	**21,304**
Inferred	17.6	6.00	3.60	97	1,060	638	55,174
Total	**28**	**9.20**	**3.10**	**85**	**2,578**	**865**	**76,578**

The Measured Mineral Resources as a proportion of the total Inclusive Mineral Resource as of 31 March 2022 accounts for ~33% of the tonnes, the Indicated Mineral Resources is ~44%, and the Inferred at ~23%.

Table 2: Mineral Resource Statement (inclusive of Mineral Reserves) – 31 March 2022

Classification	Tonnage	Grade			Metal Content		
	(Mt)	Zn (%)	Pb (%)	Ag (g/t)	Zn (Kt)	Pb (Kt)	Ag (Koz)
Measured	25.9	14.90	2.20	72	3,851	560	60,202
Indicated	33.6	13.70	1.30	41	4,612	448	44,167
Measured + Indicated	**59.5**	**14.20**	**1.70**	**55**	**8,464**	**1,008**	**104,369**
Inferred	17.6	6.00	3.60	97	1,060	638	55,174
Total	**77.1**	**12.30**	**2.10**	**64**	**9,524**	**1,646**	**159,543**

1.3 Mineral Reserve Statement

The HZL mine operations and technical teams engage on annual industry statutory evaluation and conduct standard works that is applied across all the mine operation from the geology resource estimations to applying the mine designs and evaluating the potential Mineral Reserves. The HZL Resources and Reserves technical team has kept extensive data that is used annually to do the statutory mineral reserves statements and calculations that is audited and supervised by reputable consultant houses. Each mine undergoes individual assessments and apply the modifying factors, grade cut-off calculation and assumptions.

ABGM collated the data and reviewed the mine designs, input parameters and mine design criteria for the mine operation. RAM has in recent years gone from an open pit mine to an underground massive mining operation utilising a sub-level Long Hole Open Stope (LHOS) mining extraction method with backfill. A comprehensive analysis and relevant input parameters is applied to the Mineral Resource to develop the Mineral Reserves.

The Mineral Reserve statement (March 2022) suggests RAM has 47.0Mt at 11.8g/t Zinc, 1.3% Lead and 44 g/t Silver within the minable Ore Reserve.

Table 3: Mineral Reserves Estimates (2022)

Ore Reserve summary							
Ore Reserve	**Tonnage (Mt)**	**Grade (Zn %)**	**(Pb %)**	**(Ag g/t)**	**Metal (Zn kt)**	**(Pb kt)**	**(Ag koz)**
Rampura Agucha							
Proved	14.7	12.4	1.8	65.0	1,820	261	30,493
Probable	32.3	11.6	1.1	35.0	3,739	368	36,519
Ore Reserves (Total)	**47.0**	**11.8**	**1.3**	**44.0**	**5,559**	**629**	**67,012**

1.4 Geology and Mineralization

The deposits are hosted by middle Proterozoic Delhi Fold Belt of metasediments (2000 my-750 my). The deposit is hosted by meso-metamorphic rocks of the Precambrian Mangalwar complex. Gneisses and schists form a northeast-southwest-striking isoclinal synform, with the mineralisation in the centre. The sequence is cut by amphibolite, aplite and pegmatite dykes. The immediate host rock of the mineralisation is garnet-biotite-sillimanite gneiss.

The deposit extends over 1,550 m along strike and has now been intersected to a vertical depth of 1,100 m below surface. The horizontal thickness varies between a few metres to 80 m, averaging between 50 to 60 m in the upper 250 m of the deposit. The mineralised unit also contains some inclusions of sub-ore grade material. The average dip shallows from around 70- 80° in the upper part of the deposit to about 50° in the middle and lower portions, as well as the northeast and southwest flanks. Deep drilling has shown that the deposit thickens and steepens back to approximately 70° in the lower portions, while the strike length at these lower portions appears to remain restricted to around 500 m.

Two different main ore types are identified at RAM from the North and South areas of the deposit. The material from the North area is understood to be softer and more difficult to float, while material from the South area is harder and contains higher levels of graphite.

1.5 Metallurgical Testing

The HZL has a network of operations across India and RAM is currently an operating mine with a working CPP with many years of operations behind them. The mineral processing is well understood and there is no need to conduct any additional metallurgical test work at the current operations.

1.6 Mine Design, Optimizations and Scheduling

The current mine design implemented at RAM is to date been successfully extracted and maintained as per the original design. There is currently no indication that the mine cannot continue in the current stage. The design has made sufficient provisions for a crown pillar that will be mined out at the end of mine life. Additional stability pillars have also been included in the LOM design to ensure the overall stability of the mine operations.

The current schedule suggests that RAM will be targeting 4.9mtpa ROM and will continue so with the additional infill drilling to increase the ore reserves to keep steady state.

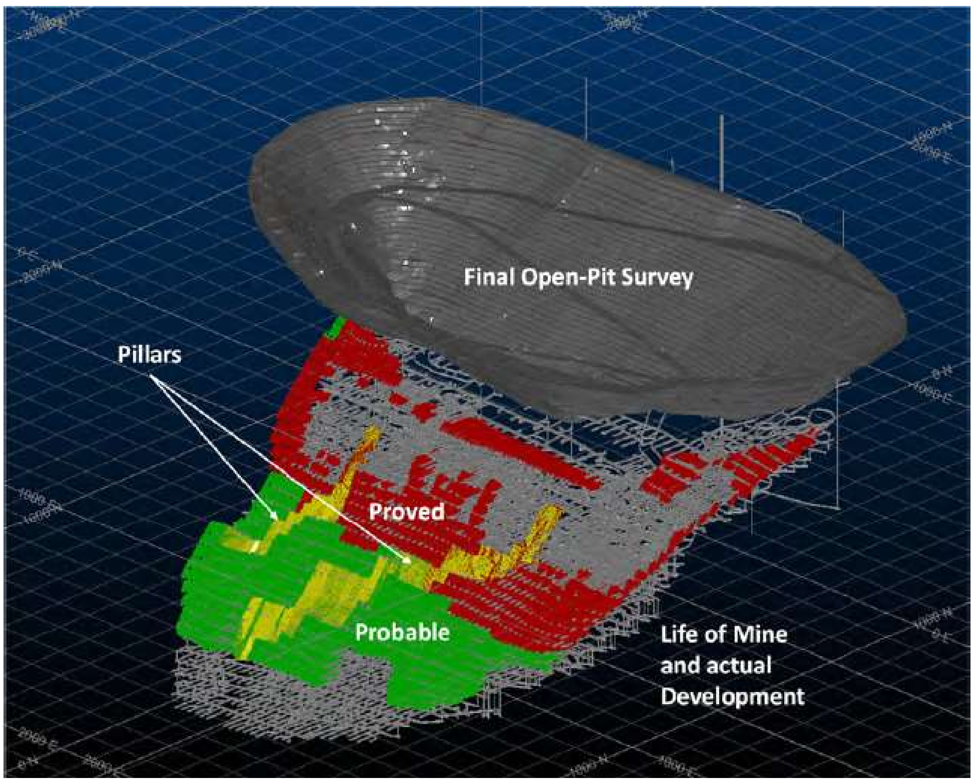


Figure 1: Overall Mine Design & Mineral Reserves Classifications

Table 4: RAM - 12 Year Schedule (4.9Mtpa - 2Mtpa)

Tonnes	Area / Block	Totals	Year1	Year2	Year3	Year4	Year5	Year6	Year7	Year8	Year9	Year10	Year11	Year12
Totals	North Block	15,106,562	2,747,274	2,801,667	2,314,257	1,757,357	2,170,710	1,330,554	1,182,000	704,004	98,740	-	-	-
	South Block	25,452,410	2,056,957	2,085,306	2,419,134	2,448,217	1,460,170	2,155,184	2,099,300	1,503,898	2,250,344	2,491,019	2,420,410	2,062,471
	North Extreme/Block 1	515,141	131,102	-	-	-	-	-	-	120,004	264,035	-	-	-
	LOM Pillar	5,970,281	-	-	-	-	-	-	-	-	-	-	-	-
	Total	47,044,394	4,935,332	4,886,973	4,733,391	4,205,573	3,630,880	3,485,738	3,281,300	2,327,906	2,613,119	2,491,019	2,420,410	2,062,471
Grade		Average	Year1	Year2	Year3	Year4	Year5	Year6	Year7	Year8	Year9	Year10	Year11	Year12
Totals	Zn%	11.82	12.19	11.83	12.95	12.14	12.54	11.39	11.96	10.94	11.61	10.10	10.03	10.01
	Pb%	1.34	1.77	1.66	1.83	1.46	1.22	0.99	1.15	1.29	1.25	1.03	0.90	0.82
	TMC%	13.16	13.96	13.49	14.77	13.60	13.77	12.38	13.11	12.24	12.86	11.13	10.93	10.83
	Ag(ppm)	45	62	63	61	47	41	31	32	37	36	34	28	29

1.7 Mineral Processing

The RAM concentrator processes two mine operations ROM namely the Rampura Agucha and Kayad Mine. The CPP at RAM has a nominal capacity of 6.5Mtpa and has sufficient capacity to process the materials from RAM and KDM. The ore received from Kayad Mine is separately treated in one of the streams. All the vital parameters are monitored through instream analysers and chemical composition of the feed, concentrates and tails are continuously sampled and analysed at site laboratory well equipped with state-of-the-art instruments. The table shown below typically depicts the assays of ore treated from Rampura-Agucha underground (RAUG) mine and Kayad mine for a day. However, it may vary depending upon the mineral composition of the respective mine.

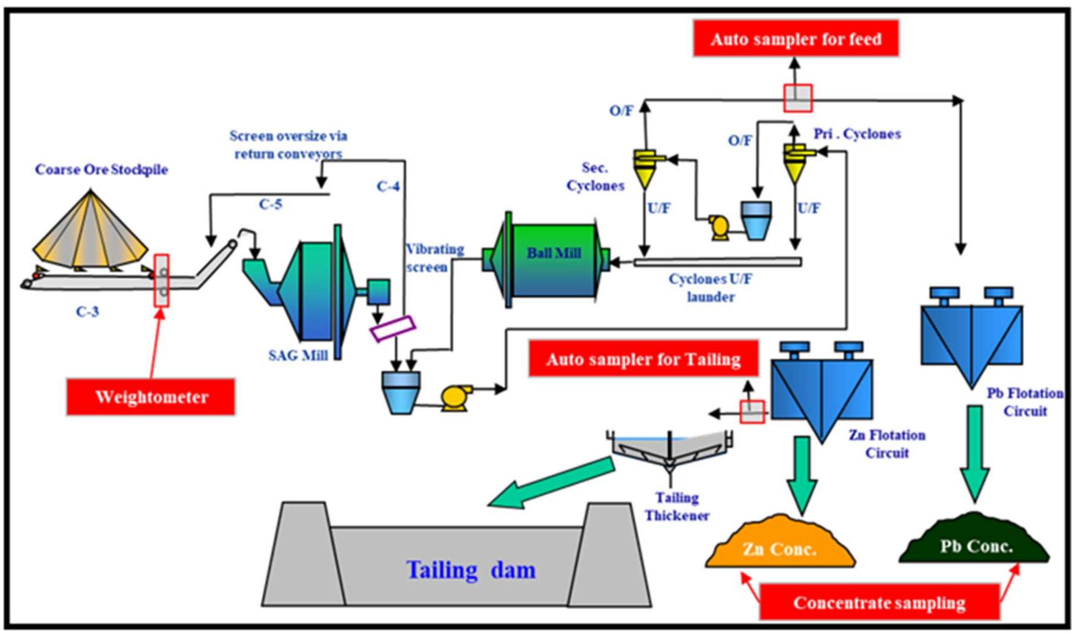


Figure 2: Simplified flowsheet - RAM Concentrator

1.8 Environmental, Permitting and Community Impact

RAM is doing sufficient work around the environmental assessments and are continuously monitoring all vital statutory aspects required. There is various sites and locations where site monitoring is conducted in regard to the following main elements:

- Land Use
- Water Quality and Management
- Air Quality

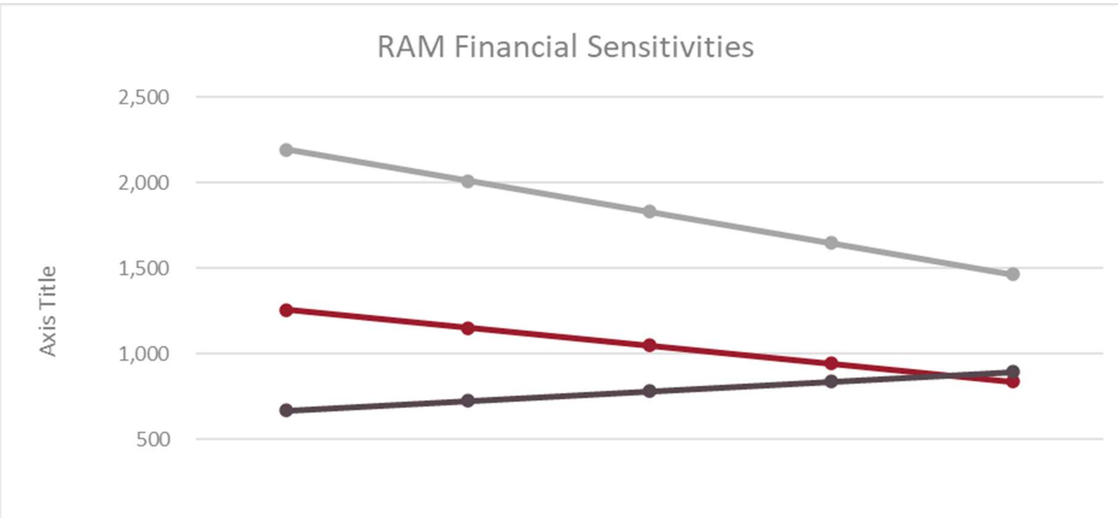
- Noise Pollution
- Soil Monitoring
- Tailing disposal

The mine is also engaging with the local communities to ensure alignment with EIA requirements overall. The mine is also planning, reviewing and executing the annual mine closure plan as required by IBM.

1.9 Capital Costs, Operating Costs and Financial Analysis

As this is an operating mine, HZL provided their operational estimates for the mine operations to develop financial sensitivities.

The operating costs for RAM is indicated in this document and financial sensitives indicate that RAM is not very sensitive to economic parameters at plus and minus 20%



	20%	10%	100%	-10%	-20%
Cashflow	1,257	1,152	1,048	943	838
Cost	668	725	782	839	895
Revenue	2,195	2,012	1,829	1,646	1,463

Figure 3: RAM Financial Sensitivity Analysis (unit: Million US $)

2 Introduction

2.1 Terms of Reference and Purpose of the Report

A&B Global Mining (ABGM) was commissioned by Hindustan Zinc Limited (HZL) to prepare a review on the Mineral Resources and Ore Reserves – 31 March 2022 Statement, for the following mine operations that are operated by HZL namely:

- Rampura Agucha (RAM)
- Kayad (KDM)
- Sindesar Khurd (SKM)
- Rajpura Dariba (RDM)
- Zawar Mines (ZAW)

This report is a Technical Report Summary (TRS) which summarizes the findings of the review in accordance with Securities Exchange Commission Part 229 Standard Instructions for Filing Forms Regulation S-K subpart 1300 (S-K 1300).

The purpose of this TRS is to report the review of Resource and Reserve Estimates as stated in their Draft document dated 31 March 2022, and to review the data & information received from HZL that will potentially be included in the 2022 Resource and reserve Statement Technical report. The effective date of this report is 29 July 2022.

The quality of information, conclusions, and estimates contained herein is based on the data and information received from HZL and is consistent with the level of effort involved in ABGM's services, based on:

i. information available at the time of preparation,
ii. data supplied by the client, and
iii. the assumptions, conditions, and qualifications set forth in this report.
iv. The time available to complete this review

Any opinions, analysis, evaluations, or recommendations issued by ABGM under this report are for the sole use and benefit of HZL. Because there are no intended third-party beneficiaries, ABGM (and its affiliates) shall have no liability whatsoever to any third parties for any defect, deficiency, error, omission in any statement contained in or in any way related to its deliverables provided under this Report.

2.2 Sources of Information

The information, opinions, conclusions, and estimates presented in this report are based on the following:

- Information and technical data provided by HZL
- Review and assessment of previous investigations
- Assumptions, conditions, and qualifications as set forth in the report
- Review and assessment of data, reports, and conclusions from other consulting organizations and previous property owners.

These sources of information are presented throughout this report and in the References section. The qualified persons are unaware of any material technical data other than that presented by HZL.

ABGM and their associates received a database of information of the HZL operations between 30 June 2022 and 25 July 2022 and reviewed the documents, datasets, and information to consolidated into the document presented as of date 29 July 2022.

2.3 List of source materials

HZL provided AGBM with access to a data room housed in EthosData. Here HZL delivered the required documentation and design data from the modelling to the string. Point and wireframes used in the development of the resources model estimates and the reserve estimates.

Figure 4: File Structure and extraction source of data

2.4 Qualified Persons and Details of Inspection

A comprehensive site visit was conducted in the week of 17 July to 25 July 2022. The objectives of the site visits was to conduct the following:

- Physical verification of the mining operations and onsite infrastructure.
- Obtain outstanding information required to complete the technical report.
- Interact with the mine technical team to obtain further clarifications.

Three consultants from ABGM attended the site visits at the various mines.

- **Devendra Vyas:** Managing Director and Principal Mining Engineer
- **Andre van der Merwe:** General Manager and Principal Consultant (Geophysics, Hydrogeology, Geology, Mineral Processing and Environmental Engineering)
- **Pieter Groenewald:** Head – Technical Services and Principal Consultant (Rock Engineering and Hydrogeology)

2.5 Previous Reports on the Project

This is the only SEC -S-K 1300 TRS, A&B Global Mining (ABGM) has submitted for the Hindustan Zinc Limited (HZL) and authors are not aware of any other TRS submitted by prior owners of the project.

3 Property Description and Locations

3.1 Property Location

Rampura Agucha Mine is situated 15 km south-east of twin towns of Gulabpura and Vijaynagar , the nearest railway station of the western railway, 75 km from Ajmer and 220 km south-west of Jaipur, in the district of Bhilwara, Rajasthan. The national highway from Delhi to Mumbai passes along the twin towns of Gulabpura and Vijaynagar. The nearest airports are at Jaipur and Udaipur almost at equidistance from the mine site.

The Mine is located on a flat plain region and local slope is towards North and South. Location co-ordinates are 25 0 50' 00" North latitude and 74 0 44' 15" East Longitude. The altitude of the site is 390 meters above Mean Sea Level. Geographically the area is covered by Survey of India (SOI) toposheet No. 45K/9 and 45 K/13. Average annual Max. and Min. temperatures are around 46.7 °C and 3.2 °C respectively. Lease area is of 1200 Ha.

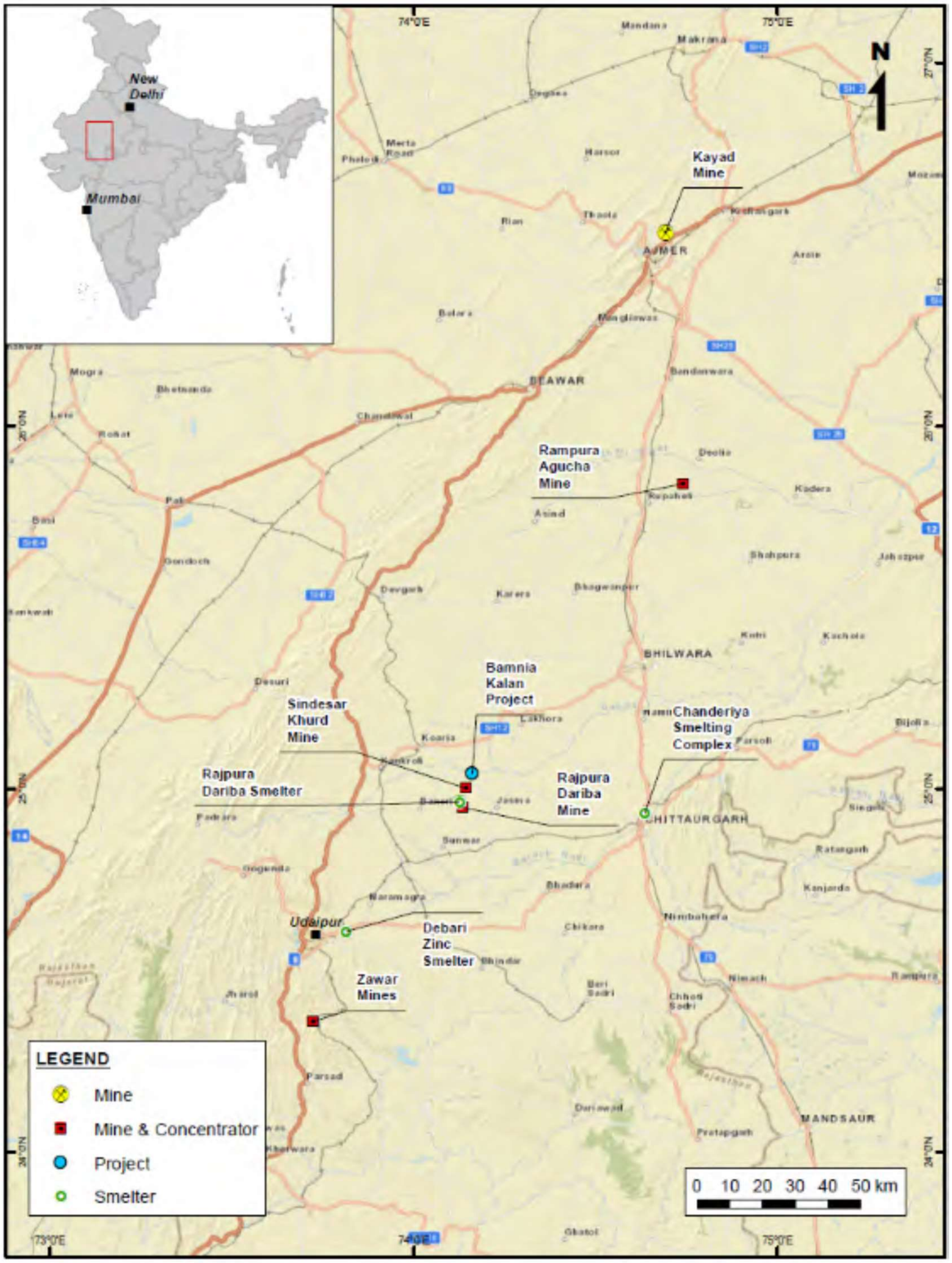


Figure 5: HZL Mine Locations

3.2 Mineral Titles, Claim Rights, Leases and Options

3.2.1 Lease agreements - Details

Table 5 : Lease Agreement - Details

Lease Details	Existing Mine
Name of the Mine	Rampura Agucha Lead Zinc Mine
Lat/long of any boundary point	Pillar A (Latitude 25°49'10.18" Longitude 74°44'15.32")
Date of Grant of Lease	Initial – 13.03.1980 The mining lease of RAM lead-zinc was sanctioned as ML No.08/99 for a period of 20 years to M/S Hindustan Zinc Limited
Period/Expiry Date	Date of expiry of lease is 12.03.2030, extended by ME, DMG, Bhilwara vide letter no.ME/BHL/CC2/Kh. P-8/1999/2406 dated 07.04.2015.

3.2.2 Lease agreements – Area

Table 6: Lease Agreement - Area

Forest Land	Nil Ha	Non-Forest	Area (Ha)
		(i) Waste Land	81.97
		(ii) Grazing land	70.0
		(iii) Agriculture Land	1048.03
		(iv) Others	
Total Forest	**Nil**	**Total Non-forest**	**1200.00 Ha**

3.2.3 Lease agreements – District

Table 7: Lease Agreement - District

District & State	Bhilwara, Rajasthan
Taluka / Tehsil (Administration Area)	Hurda & Phulia Kalan
Village	Agucha, Bherukheda, Kothiyan & Khera Palola
Whether the area falls under Coastal Regulation Zone (CRZ)	Not Applicable

3.3 Environmental Impacts, Permitting, Other Significant Factors and Risks

3.3.1 Commitments

RAM has been undergoing numerous public hearings and results of commitments from the mine is tabulated below:

Table 8: RAM Commitments from Public Hearings

SN	Issue	Action Plan	Status
1	Fodder – water supply for animals	Declaration of draught by Government of Rajasthan , HZL will open the fodder camp in vicinity of mine to provide drinking water and fodder-camp for animals.	Arrangements made for animal drinking water in nearby villages. In case of fodder requirement, same is being arranged.
2	Desired for further development in education sector.	• Mid day Meal for school children • Bal Chetna Kendra • Computer distribution to the schools • Sambal Siksha programme for Board students • Sponsoring girls for Higher Education in college	Initiatives have been implemented for improvement of education like Shiksha Sambal, Unchi Uddan, Summer education camps. Rs. 807.90 Lacs spent during 2018-19.
3	Drinking water supply	· Water supply by pipeline to Bherukhera –I · Water supply to Bherukhera –II By tankers · Water supply to Agucha Village in summer season by tankers · Water supply to Papari khera village by pipeline daily	Health, water & Sanitation is one of the focus area. An expenditure of Rs. 110.82 Lacs incurred during 2018-19.
4	Employment for local people	Preference to local people will be given. Presently 80% of contractor Employees are from the near by village and 75 % permanent manpower are from Near by village from Bhilwara District	Based on requirement and available skills, preference to local people is given and at present 80% of contractor employees are from nearby villages and 75% permanent manpower are from Bhilwara district.
5	Allocation of 1% royalty amount given to Panchayats by Government of Rajasthan. The share of Hurda Panchayat to be raised	District Collector / CEO Zila Parisad is to take action. Allocation of royalty is to be decided by Government of Rajasthan	District Mineral Fund is being allocated by state government

6	Villagers demanded for Rehabilitation and Resettlement of village Bherukhera-I	A committee of representatives of local village, District administration and HZL has been made and R&R compensation finalised.	Bherukhera-I village was rehabilitated as per plan. Spending Rs. 28.13 Crore as compensation and Rs. 3.00 Crores spent for facility development.
7	Bherukhera–II wanted to increase the water supply	The number of drinking water supply tanker will be increased for village Bherukhera –II.	Drinking water is supplied to fulfil the community requirement of the nearby villages/.

3.3.2 Security bonds

Bank Guarantee for degraded land has been submitted and reclamation plan submitted to IBM

The area that has been put to use is approx. 982.96 Ha. after part rehabilitation (64 Ha. On mature waste dump benches).

Financial assurance required for as part of PCMP is computed as 982.96 Ha x Rs. 5,00,000 per Ha = INR. 49,14,80,000.00 (i.e. Rupees Forty nine crore fourteen lakh and Eighty thousand only)

Table 9: Security Bond Summary

Total Area Proposed to be put to use in hect (Year 1 to 5)	**Amount of Bank Guarantee** (Lac INR)	**Valid till (dd/mm/yyyy)**
982.96 Hect	*4914.80 lac*	*31/03/2027*

3.4 Royalties and Agreements

3.4.1 Joint ventures

RAM is in no current joint venture with third parties or other companies.

3.4.2 Royalty agreements

At Rampura Agucha Mine Lead & Zinc Concentrates are being produced and dispatched from the lease area to the smelters which are located outside the leased area, then royalty shall be chargeable on the processed product i.e. concentrate.

The Lead and Zinc Mineral royalty is to be paid based on London Metal Exchange or London Bullion Market Association price, the royalty shall be calculated at the specified percentage of the average sale price of the metal for the month as published by the Indian Bureau of Mines, for the metal contained in the concentrate of such mineral for the month.

The Rates of royalty declared as per Mine and Mineral (Development and Regulation) Act 1957 and royalty rates are:

- The Royalty for Lead is 14.5% of London Metal Exchange lead metal price chargeable on the contained lead metal in the concentrate produced.
- The Royalty for Zinc is 10% of London Metal Exchange Zinc metal price on ad valorem basis chargeable on contained zinc metal in the concentrate produced.

4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

4.1 Topography, Elevation and Vegetation

Rampura Agucha Mine, the zinc-lead mine, is located in an area of flat terrain with poor agricultural land use. Detailed ecological studies were conducted to assess the present biological resources in and around the mine lease area of HZL's Rampura Mine. 320 plant species were recorded during the study period.

4.2 Accessibility and Transportation to the Property

Existing public infrastructure provides easy access to the mining are. Rampura Agucha Mine is situated 15 km south-east of twin towns of Gulabpura and Vijaynagar , the nearest railway station of the western railway, 75 km from Ajmer and 220 km south-west of Jaipur, in the district of Bhilwara, Rajasthan. The national highway from Delhi to Mumbai passes along the twin towns of Gulabpura and Vijaynagar. The nearest airports are at Jaipur and Udaipur almost at equidistance from the mine site..

4.3 Climate and Length of Operating Season

The climate of the district is generally dry and healthy, and the seasons are similar to nearby SKM and RDM. The summer season starts during the middle of February and - continues up to the first week of June. Summer is followed by south-west monsoon which last till the end of September. October and November are the post-monsoon months. December is the coldest month with mean daily maximum and minimum temperatures being 24 ^{0}C and 10^{0}C respectively.

During peak summer, temperature shoots up to 44.6 ^{0}C. Relative humidity varies from 25% in summer to 82% in winter (Census 2011). Due to the mild weather conditions, the mines can operate throughout the year, and should have no weather-related restrictions. The study area is largely a plain and lies in the arid region with an average rainfall of 571 mm/a in last 10 years (during the period from

2011 to 2020), with a maximum rainfall of 885 mm in 2019 and minimum of 385 mm in year 2018. The occurrence and movement of the ground water is controlled by the topography and the physical characteristics of the rocks occurring in the area.

Table 10: Annual Rainfall Records – [1987-2008] supplied HZL

Year	Rainfall (mm)	Year	Rainfall (mm)	Year	Rainfall (mm)
1989	455	2000	352	2011	1017
1990	806	2001	428	2012	747
1991	315.4	2002	153	2013	749
1992	288.4	2003	575	2014	709.5
1993	294.5	2004	604	2015	471
1994	457	2005	418	2016	814
1995	308.5	2006	792	2017	491
1996	480.6	2007	721	2018	611.2
1997	453	2008	472	2019	847.5
1998	309	2009	289	2020	504
1999	180	2010	951	2021	791
				Average	541.0

4.4 Infrastructure Availability and Sources

Existing public infrastructure provides easy access to the mining area. Production commenced in 1991 with an open pit mine which ceased production in March 2018. Mining from underground commenced in 2013. The Company applied for a new prospecting permit covering the surrounding area which was secured during 2010. This is important as the deposit is dipping towards the eastern limit of the mining lease and the location of the lease boundary makes deep exploration drilling challenging.

Power for the processing operations is supplied by the captive power plants (“CPP”) at Zawar and Chanderiya which provide around 90% of the requirements. Power requirements for processing facilities are around 38-40 MW.

Overall water usage is approximately 3.2 m^3/t, with fresh water, making 0.45 m^3/t of the required water. Fresh water is sourced from the Banas River 55 km from the RA Mine concentrator. There is also a constraint by the Rajasthan Government related to the maximum amount of water that can be extracted, which is currently limited to 11,200 m^3/day.

It is reported that plant water requirements are satisfied by internal mine and recycle water sources and that no additional water is required.

5 History

5.1 Historical Exploration and Production

The Rampura Agucha Mine has an extensive history with exploration recorded from 1977 to 1982 when 229 holes were drilled for a total meterage of 24,900 m. Historic records indicate that exploration was only resumed in 2007, possibly when the mine was nearing the end of its economic life as an open pit.

Production in the open pit commenced in 1991 and was eventually closed in March 2018. It is assumed that the drilling that commenced in 2007 was searching for extensions of the orebody but drilling declined to just 4 holes in 2011. The results must have been positive as exploration drill-rate picked up. Underground mining was initiated in 2013. The production rate is increasing, and production has increased from 3.33 Mt in 2019 to 4.27 Mt in 2021. The Mining Lease is valid until 12.03.2030.

Table 11: RAM Surface Exploration Drilling

Period	no. of holes	Metres Drilled
1977 - 1982	229	24,900
2007	69	38,307
2008	36	28,170
2009	33	12,681
2010	6	5,096
2011	4	3,789
2012	38	9,488
2013	40	8,436
2014	47	9,068
2015	31	14,765
2016	67	11,595
2017	30	10,443
2018	64	7,056
2019	189	27,458
2020	246	32,124
2021	216	30,234
Total	**1,345**	**273,609**

Table 12: RAM Production 2017 - 2021

Description	Unit	F2017 Actual	F2018 Actual	F2019 Actual	F2020 Actual	F2021 Actual
Open Pit	Mt	3.32	1.76			
Underground	Mt	1.43	2.08	3.33	3.94	4.27
Ore Mined	Mt	4.75	3.84	3.33	3.94	4.27
Ore Processed	**Mt**	**5.16**	**5.7**	**5.11**	**5.16**	**5.47**
Zn grades	%	12.1	10.52	10.23	10.06	9.71
Pb grade	%	1.66	1.52	1.56	1.87	1.83
Ag grade	g/t	50.8	51.7	54.9	50.3	53.6
Zn Recovery	%	92	89.8	88.2	87.9	89.3
Pb recovery	%	63.9	60.4	56.3	54.9	57.8
Ag recovery	%	68.9	72.1	64.2	60.2	61.1
ZN metal produced	**t**	**574.5**	**538.7**	**461.1**	**456.4**	**474.4**
Pb metal produced	**t**	**54.7**	**52.4**	**45**	**40.8**	**45.3**
Ag metal produced	**koz**	**5810**	**6833**	**5,791**	**5,029**	**5,758**

6 Geological Setting, Mineralization and Deposit

6.1 Regional Geology

Rampura-Agucha deposit forms a part of Pre-Cambrian Banded Gneissic Complex (BGC) group of rocks, which occupy an extensive area in Central Rajasthan peneplain consisting of gneisses, schists and intrusive of acid and basic igneous rocks, that occupy predominantly the southeast plain of Ajmer and Bhilwara.

The BGC extends from Nathdwara in South up to Sambhar in North and the eastern limit extends up to Shahpura whereas western limit ends with Delhi formation. BGC is composed of heterogeneous rock type of parametamorphics, migmatites, pegmatites, granites and basic intrusive rocks of variable composition. The principal rocks that underlie Rampura-Agucha are gneisses and schists and their variants of BGC (renamed as Mangalwar complex). The rocks have been profusely intruded by igneous rocks of acidic and basic type.

6.2 Property Geology

Rampura Agucha is a stratiform, sediment-hosted Lead Zinc deposit, occurs in Pre-Cambrian Banded Gneissic Complex and forms a part of Mangalwar complex of Bhilwara geological cycle (3.2-2.5 billion years) of Archean age and comprising of magmatites, gneisses, graphite mica schist, pegmatite and impure marble. The rocks have been subjected to polyphase deformations and high-grade metamorphism. The deposit lies in the contact zone of Mangalwar - Sandmata complex and along the major NE trending Delwara lineament and predominantly covers southeast plain of Ajmer and Bhilwara.

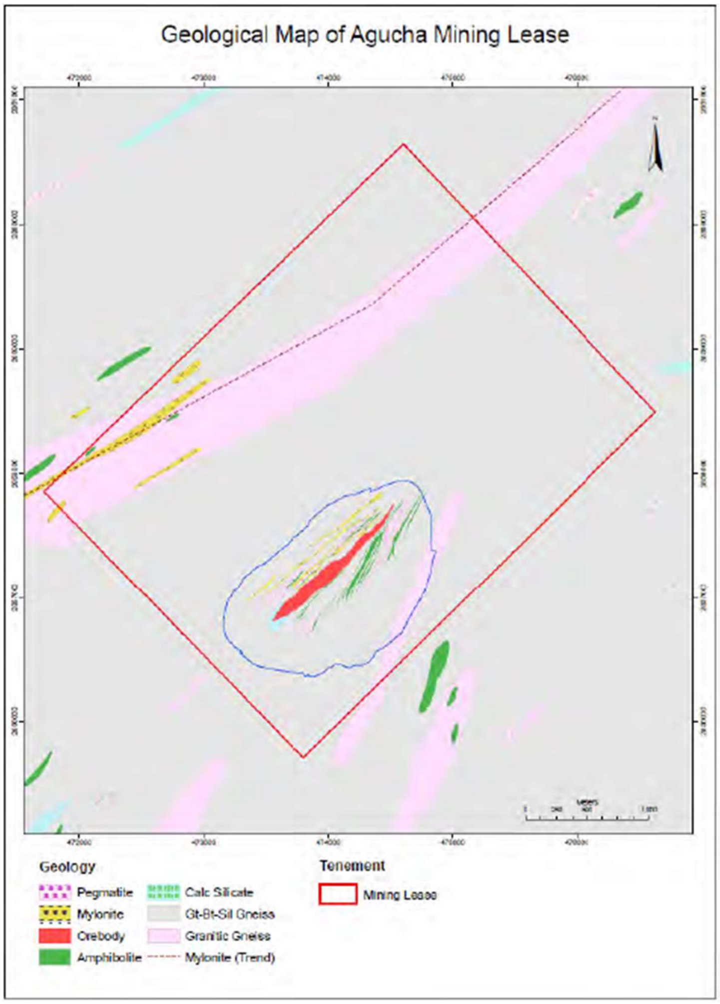


Figure 6: RAM Property Geology

6.3 Deposit Types

The deposit has been broadly domained into four discrete zones, referred to as; 'south ore' (zone 1); 'core ore' (zone 2); and 'north ore' (zones 3 and 4). The basis of the zones is the reasonably distinct variations in the zinc grades along strike. Furthermore, the relationship between lead and silver between the northern and southern zones shows a markedly different regression profile.

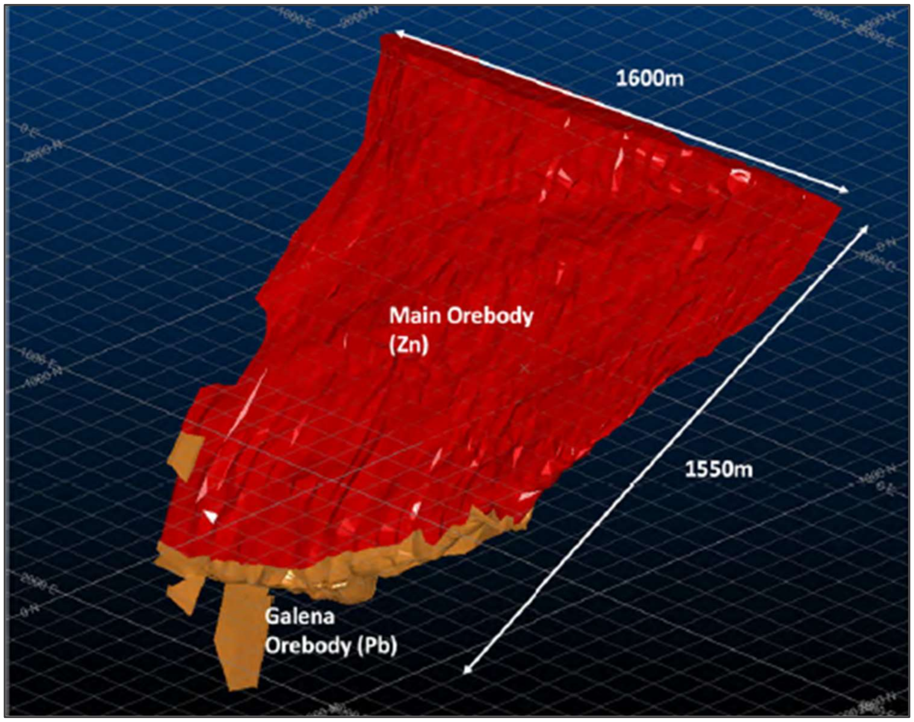


Figure 7: RAM Orebody

6.4 Mineralization

The graphite-mica-sillimanite schist hosts the economic mineralization in the Rampura Agucha deposit. The economic minerals in order of decreasing abundance along with their ranges are:

- Sphalerite (15-20%)
- Pyrite (15-18%)
- Pyrrhotite (12-14%)
- Galena (1-2%)
- Sulphosalts (0.1-0.2%)

At RAM, the main base metal minerals are sphalerite and galena with pyrite levels reported to be approximately 9% by mass. Plant feed grades are around 12-14% Zn and 2% Pb. Two different main ore types are identified at RAM from the North and South areas of the deposit. The material from the North area is understood to be softer and more difficult to float, while material from the South area is harder and contains higher levels of graphite. Levels of graphitic carbon in the plant feed are indicated to be relatively high, at around 5%. The graphite occurs as flakes and reports into the lead and zinc concentrate despite the use of selective depressants in the flotation circuit.

7 Exploration

The HZL complex has undergone extensive exploration since their discovery in the mid-1970's. In the past few years, exploration has been undertaken by means of diamond drilling underground for two purposes, namely grade control and extension of current mineral envelopes to deeper levels.

7.1 Summary of Exploration Activities

In general, the underground exploration drilling is undertaken during the underground mining activities. On-reef drilling is carried out by the mine and constitutes the majority of the meterage drilled.

Table 13: Summary of Exploration Drill for HZL Complex (2017-2021)

Mine/Deposit	F2017		F2018		F2019		F2020		F2021		F2022*
	No. Holes	(m)	No. Holes	(m)	No. Holes	(m)	No. Holes	(m)	No. Holes	(m)	(m)
Rampura Agucha	30	10,443	64	7,056	189	27,458	246	32,124	216	30,234	53,500
Kayad	100	16,154	75	46,365	90	46,891	118	37,371	199	44,950	83,000
Rajpura Dariba	37	15,598	89	10,838	168	32,435	203	34,717	98	27,639	43,200
Sindesar Khurd	288	72,912	606	112,906	643	111,824	604	106,670	446	76,657	135,675
Zawar	280	89,550	1,268	126,413	1,292	157,666	1,062	200,032	1,165	177,029	209,700
Bamnia Kalan	-	-	5	3,615	62	28,184	52	25,682			
TOTAL	**735**	**204,657**	**2,107**	**307,193**	**2,444**	**404,458**	**2,285**	**436,596**	**2,124**	**356,509**	**525,075**

*Planned meterage for the financial year

In the year F2020, the Company drilled a total 436,597m between both exploration and mining related targets, representing an increase of almost 32km (8%) from the same period in F2019. Exploration drilling activities have been planned for all of HZL mining properties in F2021, most notably Zawar and Sindesar Khurd.

In the year F2021, the Company drilled a total 356.5km between both exploration and mining related targets, representing a decrease of 80.1km (-18%) from the that achieved in F2020. Exploration drilling activities have been planned for all of HZL mining properties in F2022, most notably Zawar and Sindesar Khurd.

7.2 Exploration Work – Drilling

Initial exploration drilling at RAM was carried out from 1977 to 1982 and intersected the mineralisation to a depth of 400 m below surface. In total, 243,376 m of drilling was completed for some 1,129 holes at the end of 2020. Further drilling, some 246 holes, were completed in F2021, mainly underground infill drilling (233 holes) to assist in orebody delineation for underground mine planning as shown in table below.

Table 14: : Exploration Drilling History of Rampura Agucha Mine

Year	No. Holes	(m)
1977-1982	229	24 900
2007	69	38 307
2008	36	28 170
2009	33	12 681
2010	6	5 096
2011	4	3 789
2012	38	9 488
2013	40	8 436
2014	47	9 068
2015	31	14 765
2016	67	11 595
2017	30	10 443
2018	64	7 056
2019	189	27 458
2020	246	32 124
2021	216	30 234
Total	**1 345**	**273 610**

7.2.1 Drilling Technique, Spatial Data & Logging

The Company is employing directional drilling techniques and can drill several deflections from one master hole. Drilling is generally in NQ core diameter. During 2012-2013 drilling was outsourced to drilling partner Asian Oilfield Services Limited and was mainly conducted from hanging wall benches.

Core recovery is excellent averaging some 95%.

Gyroscopic downhole surveys are completed using a multi-shot camera (Reflex). Surface Drill Collars are initially located on ground using GPS. Subsequently, these are tied up with Local Grid by the mine survey team using Total Station. Down-hole direction arrows are marked on every core piece by the driller. Similarly run ends and meter pegs are neatly marked and placed. Meter depth is also painted on the core just before the peg. Geologists ensure that run pegs are at the correct locations. UG mine workings are surveyed using Total Station.

Standard logging practices, adequate for resource estimate purposes, were used. Almost the entire available core, particularly within 100m of the deposit extents on the hanging wall and up to the end of the holes on the footwall side, has been photographed using a digital camera.

Geological and geotechnical logging is undertaken on a systematic basis by HZL geologists. Logging is largely qualitative. Certain semi-quantitative metrics are logged: structural data, RQD, fracture frequency and model mineral percentages. All data is logged in hardcopy and entered in to excel before being input into the central database, which is securely stored and backed-up at the office in Udaipur. Core is well logged and interpreted by competent geologists onsite. Basic structural and geotechnical data are recorded and is recommended.

7.2.2 Sample Preparation

The following process was carried out during the sample preparation at RAM:

- 1m length core samples in the visible mineralised zones,
- Separate samples for notably different core recoveries in two contiguous runs,
- Separate samples for visibly significant grade variations (Zn and / or Pb).
- Longitudinal split line is marked along the marked sample so as to ensure equal division of ore portion in the two halves.

Samples are clearly labelled and there is good control of samples through the preparation and analytical process. The sample preparation procedure is in line with the industry standard. Samples are stored securely onsite at RAM.

Note: *No clear mention of security, transport and chain of responsibility and custody, but no incidents have been reported either.*

7.2.3 Sample Analysis and QAQC Protocols

Samples are analysed at Shiva Laboratory, an accredited local mineral laboratory. For 29 elements:

- Pb, Zn, Fe, Cd, Cu, Co, Sb, Bi, Ni and Mn by 4-acid digest and ICP-OES finish, and
- Ag and As by aqua regia digestion followed by AAS finish.

The analytical laboratory, introduces its own standards at 1 in 30 samples and repeats the analysis for our every 10th sample.

For QA/QC (Quality Assurance and Quality Control) purposes, the following procedures are employed:

- Insertion of blank material (quartz/pegmatite) into the numbered sequence as the first sample at the beginning of the mineralization in each sample batch,

- Systematic insertion of certified reference material ("CRM") (GESTAT and OREAS) at frequency of 1 in 25. The choice of CRM is at the discretion of the logging Geologist,
- Random insertion of duplicate pulp checks, at a frequency of 1 in 10.
- Umpire samples are carried out at HZL's laboratory at RAM

QA/QC procedures are adequate and no significant deviations have been reported since 2004.

Density and Bulk Density analyses are undertaken on site. A weight / volumetric displacement method (Archimedes' Principle) is used to asses bulk density, which typically reconciles well against production data. Bulk density for each sample is measured by volumetric method using Archimedes' Principle. Where correlations between the metal and density measurements is well established, and the adequacy of measurements is sufficient, the bulk density is estimated from regression analysis.

Note:

- *While this is a well-accepted standard method of measuring the density, and the technical staff are competent to carry this out on site, it is always good to take a few samples for independent analysis at an accredited laboratory, just as a confirmation that measurements are accurate – this is recommended for future drilling*

7.3 Opinion of adequacy

The QP believe that the procedures used in the sampling are adequate for mineral estimation purposes and reporting of mineral resources and reserves.

8 Mineral Processing and Metallurgical Testing

The HZL has a network of operations across India and RAM is currently an operating mine with a working CPP with many years of operations behind them. The mineral processing is well understood and there is no need to conduct any metallurgical test work at the current operations.

9 Mineral Resource Estimate

9.1 Introduction

The Mineral Resources described in this Item are based on appropriate geoscientific information, economic and technical parameters, and grade and tonnage estimation processes. The Mineral Resource estimates were determined using ordinary kriging (OK) geostatistical methodology and considered sample lengths, grade capping / cutting, the spatial distribution of drill holes and the quality assurance and quality control results for the analytical sample grades determined. Geological modelling and grade estimation used Datamine software.

9.2 Geological Models

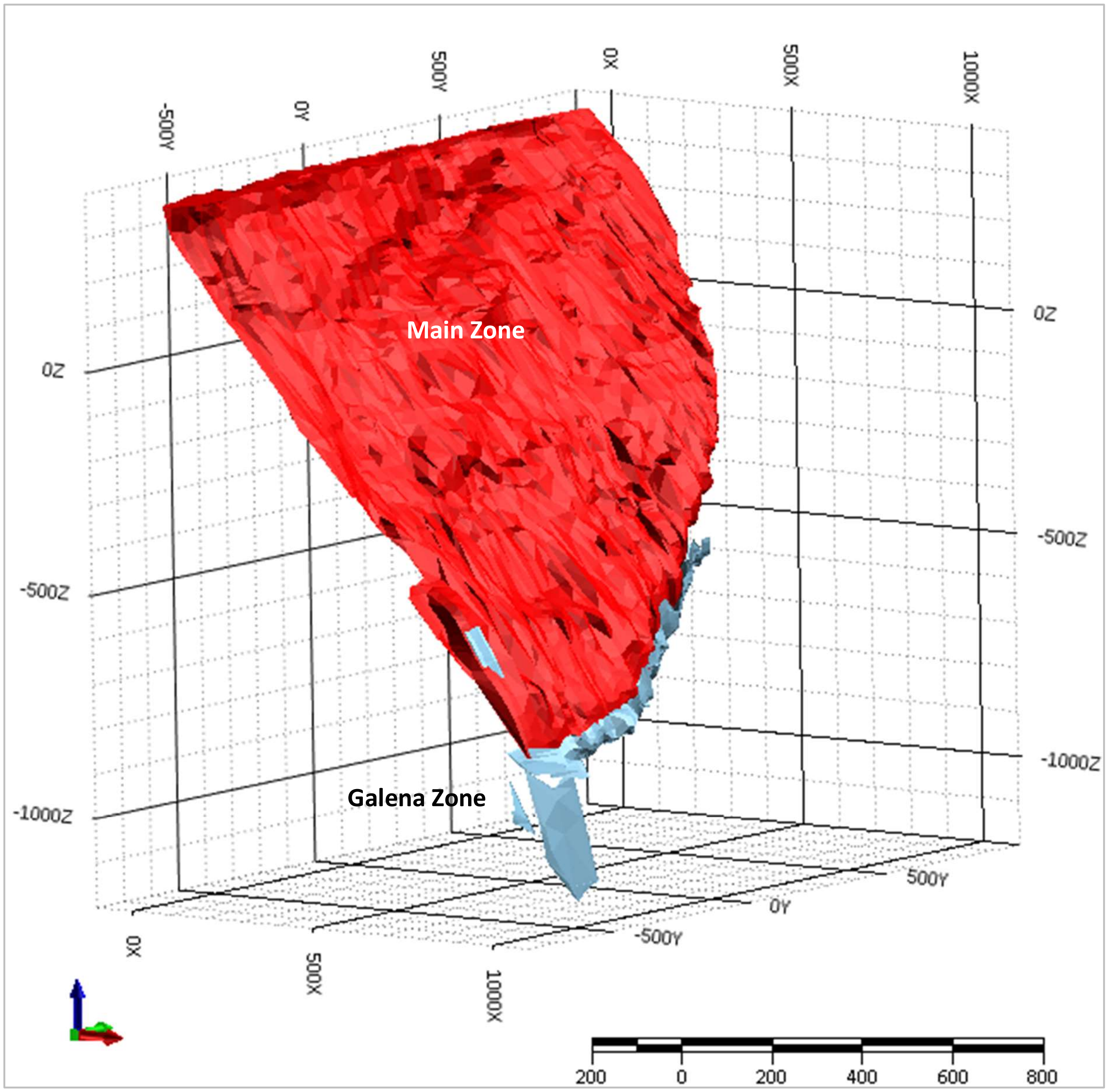
The zinc (Zn) and lead (Pb) sulphide mineralization at Rampura Agucha is a shear controlled sedimentary exhalative (SEDEX) deposit type with sharp hanging- and footwall contacts. The zinc and lead sulphides are hosted within Precambrian garnet-biotite-sillimanite gneiss and schists which is deformed by a tight northeast-southwest-striking isoclinal synform.

The mineralized zone has a~1,550 m strike length with a NE-SW direction but is restricted to ~500 m in the lower portions. The dip varies between ~40 – 70° to the SE. The thickness of the mineralized zone is 50 - 60 m but widens to ~80 m. The established depth is between ~1,100 m to 1,500 m vertical below surface however the Mineral Resource is limited to 555 m vertical depth below surface. The hanging- and footwall contacts are sharp, confirmed by mapping and statistically assessed through contact analysis.

The mineralized zone is divided by a wide low grade (internal waste) zone in the upper portions which is modelled, and the grades estimated as a separate domain. The estimated mean grade of this zone is 0.15% Pb, 0.99% Zn. The grade continuity within the larger Zn and Pb-bearing (Main) zone is variable. The primary sulphide minerals within the main ore body are sphalerite, lesser galena and minor silver-bearing minerals. Mica, chlorite, feldspar, quartz and minor graphite (~5%) constitute the gangue minerals.

The Main Zone consists of three broad zones known as the South Zone (Zone 1), the North Zone, which is comprised of two sub-zones (zones 3 and 4), and the Core Zone (Zone 2). The zones are based on distinct variations in the zinc grades along strike and the relationship between lead and silver. A further zone known as the Galena Zone which comprises five lenses runs parallel to the geometry of the lower Main Zone, and between ~500 m and 1,150 m below surface. The average vertical thickness of the Galena Zone is ~25 m.

Figure 1-1: Main and Galena zones of the Rampura Agucha deposit



9.3 Database

The database used for the grade estimates is based on data received from Hindustan Zinc Limited. A total of 1,345 drill holes were available for use in generating the wireframes, and grade and tonnage estimates. The cut-off date for the database for used in the Mineral Resource estimation process is the end of F2022.

ABGM has reviewed the drill hole database up to F2022 and concluded that the drill hole data is adequate for use in the Mineral Resource estimation process. The post-2015 drill holes were found to have adequate industry standard quality assurance and quality control (QAQC) quality assurance programmes and procedures which allowed for replication, precision and accuracy of the sample grades. Assay results prior to 2015 were also reviewed by previous consultants who concluded that the grade results are satisfactory although short-comings were noted. The reader is directed to Item 8 for sample preparation, analytical techniques and security.

9.4 Block Model Orientation and Dimensions

The wireframes for the Main and Galena zones were constructed using cross-sections taken across the ore body with a maximum extrapolation distance of 3 m beyond the last data. A geological cut-off of 2% Zn + Pb cut-off defined the limits of the mineralized zones. The wireframes were filled with a block model based on a parent-block size of 10 m x 25 m x 10 m (across-strike (X) / along-strike (Y) / vertical height (Z)).

9.5 Exploratory data Analysis

In the early years of exploration and mining silver was not consistently undertaken. To accommodate these absent values were populated in the drillhole database using a regression formula and these values were subsequently used in the estimation process.

$$Ag = (17.951 \times Pb)\ 6.0452.$$

Table below summarizes the classical statistics of the sample population distribution.

Table 15: Classical statistics of the Main and Galena zones

Element	Minimum	Maximum	No. of Points	Mean	Mode	Median	Coeff. of Variation
Zn (%)	0.00	56.20	39,015	12.13	0.50	10.70	0.76
Pb (%)	0.00	46.68	38,952	1.88	0.07	0.92	1.38
Ag (g/t)	0.00	2509.00	39,054	60.50	11.50	32.00	1.32

Normal space histograms for the drill hole samples show that Zn% has a symmetrical skewed distribution, whilst Pb (%) and Ag (g/t) represent strongly skewed distributions indicating that the mean is greater than the median value which is greater than the mode. This indicates that the mean value is influenced by higher grade values.

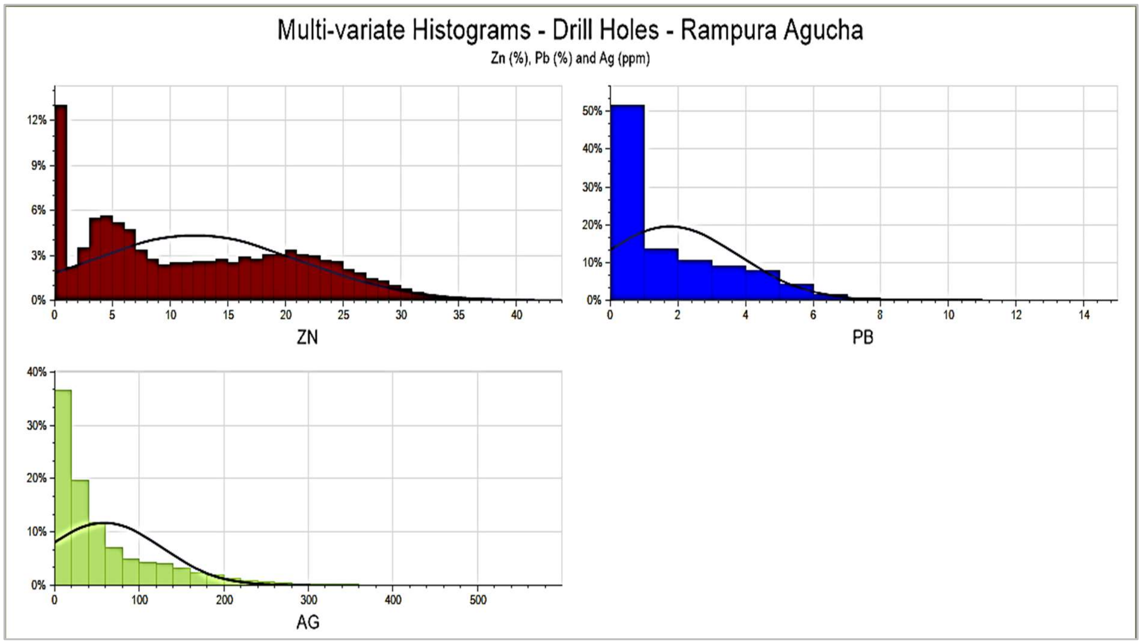


Figure 8: multi-variate histograms of the drill hole samples for Zn (%), Pb (%), and Ag (g/t)

9.6 Data Compositing

Most of the drill hole samples are 1.0 m in length and were therefore composited to 1.0 m lengths. Data compositing reduces the data population variability and contributes to a more uniform data support for estimation purposes. The drill hole and composite data populations correspond with each other for the normal space histograms drill hole samples demonstrating the data population distribution has been maintained. The variability of the low Zn grades in has been lessened by compositing.

9.7 Density Determination

Specific gravity (density) was analysed using the Archimedes' Principle methodology. The Archimedes' Principle measures the weight / volumetric displacement of a sample within air and water. The correlation between the values reconciles well against production data. The density data was used to interpolation the density values during the Mineral Resource estimation process.

9.8 Grade Capping / Cutting

Grade capping and cutting removes outlier values which will unduly influence the variability in the estimation process. Outlier values were determined via statistical analyses. Cutting removes the values above a threshold whilst capping converts the outlier values to the threshold value. Grade cutting was applied to the Main and Galena zones

Top-cuts were applied as follows:

- Main Zone: Zn - 22.5%, Pb-13 %, and Ag – 300 g/t
- Main Zone (R18): Zn - 22.5%, Pb – 13%, and Ag – 300g/t
- North Zone: Zn - 22.5%, Pb - 13.8%, and Ag – 300g/t
- South Zone: Zn – 28%, Pb - 13.5%, and Ag – 300g/t
- East Zone (E0): Zn – 28%, Pb - 9.6%, and Ag – 300g/t
- East Zone (EL): Zn – 28%, Pb - 13.5%, and Ag – 300g/t.

9.9 Estimation/Interpolation Methods

There grade and density estimation process used parent-block size of 10 x 25 x 10 m (across-strike (X), along-strike (Y) and vertical height (Z) respectively). Appropriate sub-blocking has been used at mineralisation contacts to honour the geometry and volume.

Ordinary Kriging (OK) using search ellipsoids with anisotropic weighting that encompasses the geological trends was used to both interpolate and extrapolate grades for Zn (%) , Pb (%), and Ag (g/t). Restricted searches of 40 x 30 x 10 m have also been applied to Zn (%), Pb (%), and silver (g/t), as well as low grade domain searches using 70 x 70 x 70 m. Specific search neighbourhood estimation parameters have been established for each zone, including separate domains for low and high grade domains. Restricted searches of 40 x 30 x 10 m have also been applied to zinc, lead, and silver, as well as low grade domain searches using 70 x 70 x 70 m. The first search ellipsoid was expanded by up to a factor of five to ensure that all blocks are filled with estimates. The resultant estimates form the basis for the annual Mineral Resource statement.

Variograms and specific search neighbourhood estimation parameters have been established for each zone, including separate zones for low and high grades.

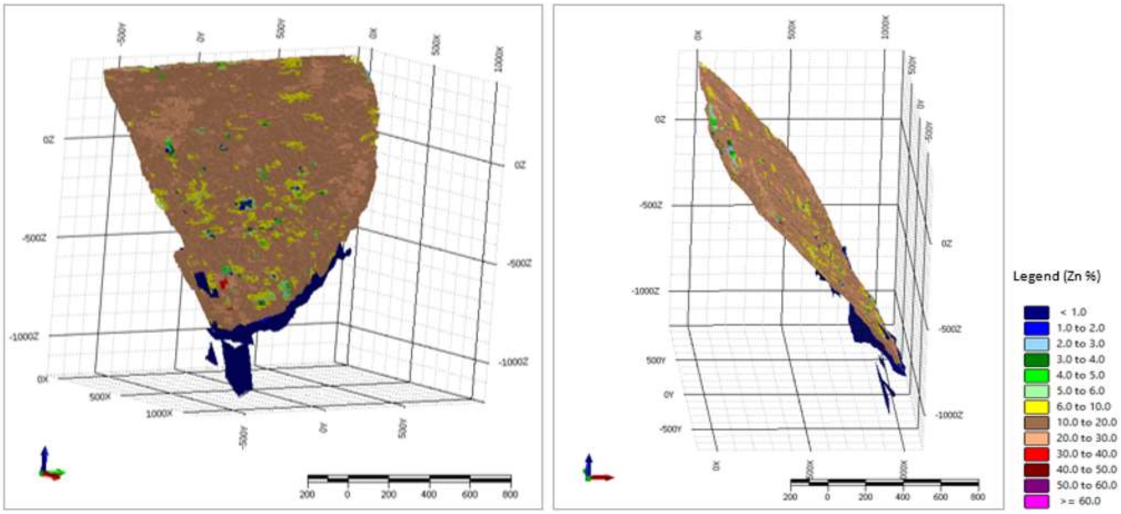


Figure 9: OK estimates for Zn (%) – no economic cut-off grade applied

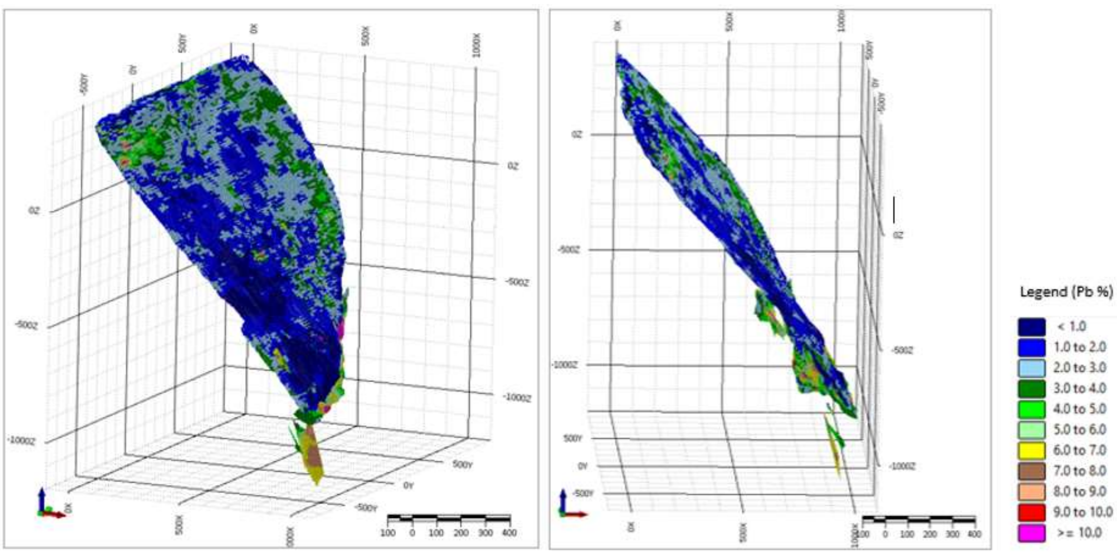


Figure 10: OK estimates for Pb (%) – no economic cut-off grade applied

9.10 Reasonable Prospect of Eventual Economic Extraction and Cut-offs

Mineral Resources must demonstrate reasonable prospects for eventual economic extraction (RPEEE). Consideration of geological, mining engineering, processing, metallurgical, legal, infrastructural, environmental, marketing, socio-political and economic assumptions used should satisfy that the project is economically viable eventually.

At Rampura Agucha confidence can be placed on the geological understanding and models, the mining methods and planning as well as the understanding of the metallurgy, processing legal, infrastructural, environmental, marketing, socio-political and economic assumptions. The reader is referred to the relevant Items within this report for detail and opinions.

The Mineral Resource for the Main Zone has been reported using a Zn Equivalent (ZnEQ) cut-off grade (COG) of 3.46% derived from:

$$ZnEQ = Zn + (0.363 \times Pb) + (0.01060 \times Ag).$$

The Mineral Resource for the Galena Zone has been reported using a Pb Equivalent (PbEQ) COG of 3.68% derived from:

$$PbEQ = Pb + (0.185 \times Zn) + (0.04886 \times Ag).$$

The COGs used for the Mineral Resource were calculated using on nett smelter return (NSR) values for the individual metals of Pb, Zn and Ag and based on the following:

- at prices of USD 2,057/t, USD 2,759/t and USD 21.24/oz, respectively
- costs based on the F2022 Business Plan
- no mining factors have been applied
- includes planned and unplanned dilution
- no metallurgical factors have been applied; however, metallurgical recoveries were based on metallurgical and smelter performance.

9.11 Classification of Mineral Resources

The level of confidence in the geology and the volume, tonnage and grade estimates determines the classification/s of a Mineral Resource. The lowest level of confidence is an Inferred Mineral Resource and with increasing levels of confidence Indicated followed by a Measured Mineral Resources can be classified. The universal requirement across the three categories is the requirement of reasonable prospects of eventual economic extraction (RPEEE) must exist. There is no uncertainty that all or any part of this Mineral Resource will be converted into Mineral Reserve.

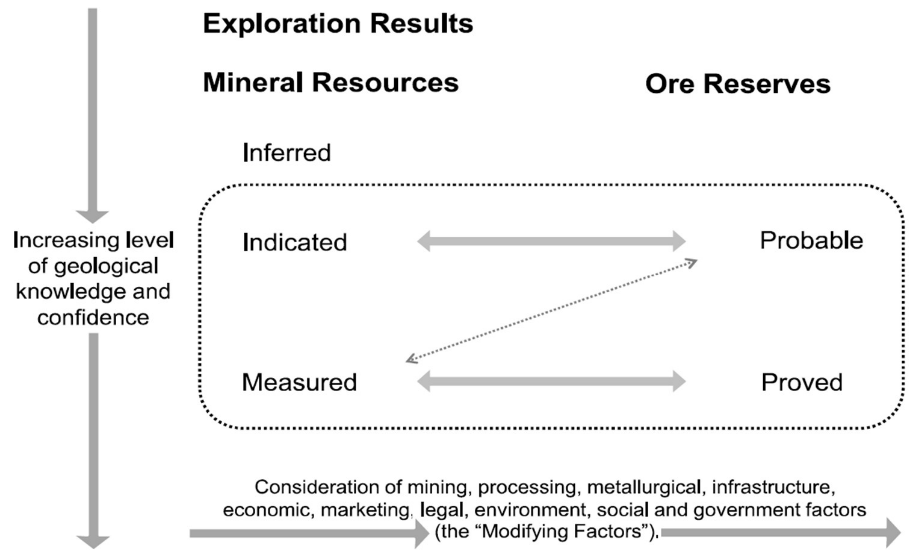


Figure 11: JORC Mineral Resource and Ore Reserve classification framework

Source: The Australasian Code for Reporting Exploration Results, Mineral Resources and Ore Reserves (the JORC Code), p9.

Mineral Resource classification is based on both technical and economic factors, namely:

- geological understanding, continuity and confidence
- grade continuity
- drill hole / sample spatial representativity and spacing
- data quality assurance and control
- appropriate geochemical analytical and density techniques applied
- application of the appropriate estimation methodologies and confidence therein
- RPEEE
- validity and ownership of the relevant government license types such as exploration, and environmental licenses
- consideration of social factors
- legal and governmental risk factors.

Inferred, indicated and Measured classification are based on increasing order of confidence. Various international mineral reporting codes define the criteria required for each confidence category.

An Inferred Mineral Resource is defined as:

- quantity and quality of the grade or quality data is based on 'limited geological evidence and sampling
- geological evidence is sufficient to imply but not verify geological and grade or quality continuity
- has demonstrated RPEEE
- it may not be considered in the assessment of the economic viability of a mining project
- it cannot be converted to Mineral Reserves
- it is expected that most of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with further exploration (Code of Federal Regulations, 229.1300, 2022).

An Indicated Mineral Resource is defined as:

- quantity and quality of the grade or quality data is sufficient to allow the application of modifying factors in sufficient details that will support mine planning and economic viability
- data has been gathered from adequately detailed and reliable exploration, sampling and testing
- geological evidence is sufficient to assume geological and grade or quality continuity between data points
- has demonstrated RPEEE
- can be converted to a probable Mineral Reserve with the application of reliable modifying factors (Code of Federal Regulations, 229.1300, 2022).

A Measured Mineral Resource is defined as:

- the confidence in the quantity and quality of the grade and scientific data is sufficient to allow the application of modifying factors to support detailed mine planning and final determination of the economic viability of the deposit
- data has been gathered from detailed and reliable exploration, sampling and testing data
- geological and grade or quality continuity between data points has been demonstrated
- has demonstrated RPEEE
- can be converted to a Probable and Proven Mineral Reserve with the application of reliable modifying factors (Code of Federal Regulations, 229.1300, 2022).

The Mineral Resources stated for the Main Zone (Figure 12) for the previous and most current completed fiscal years were based on these factors. The data spacing for the Measured classification required a minimum of 25 x 25 m spacing, a grid spacing of 50 m to 100 m for Indicated classification and a spacing of greater than 100 x 100 m for an Inferred classification. The Inferred Mineral Resource was limited to a depth from surface of 555 m.

The Galena Zone has been classified as Inferred, due to the more limited drill hole intersections and the lower geological confidence (Figure 12). The drill hole spacing is slightly irregular due to drill hole deviation from long surface collared holes.

Figure 12: Mineral Resources of the Main and Galena zones – 31 March 2022

9.12 Grade Model Validation

Grade model validation methods included visually and statistically validating the estimated block grades relative against the original sample results and the generation of swath plots.

Statistically validation indicated that the data population for the grade estimates (Figure 13) mirror those of the sample (Figure 8) and composited grades (**Error! Reference source not found.**).

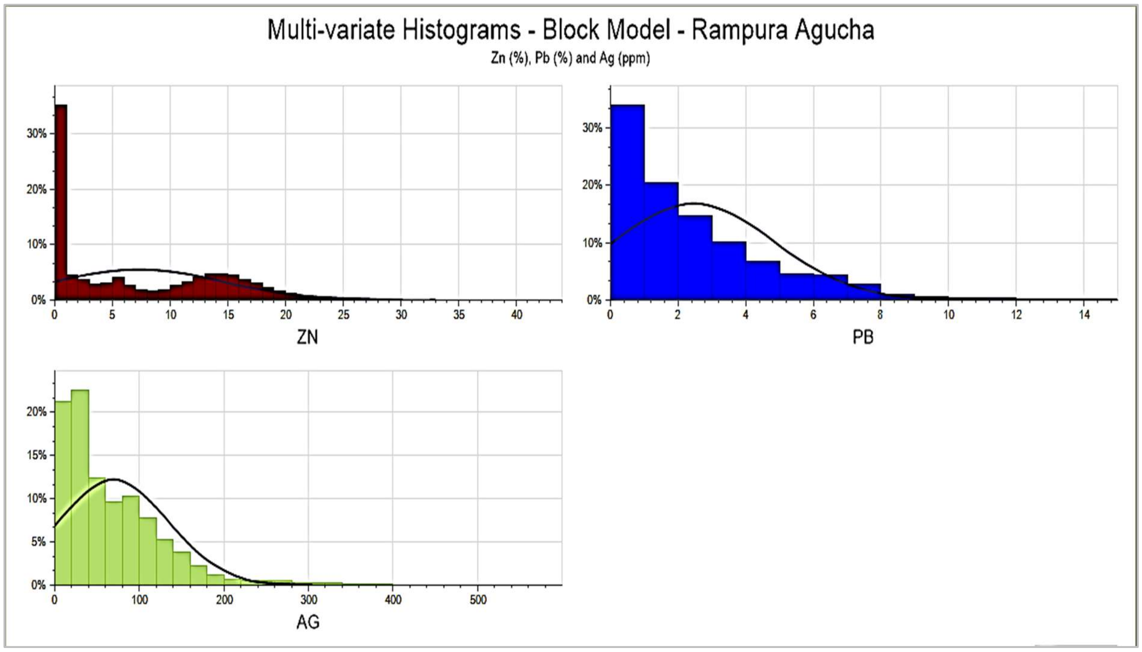


Figure 13: multi-variate histograms of the estimated grades for Zn (%), Pb (%), and Ag (g/t)

Swath analysis Figure 14 and Figure 15 of the block model estimates versus the drill hole and composite grades are relatable indicating the estimates honour the input data population.

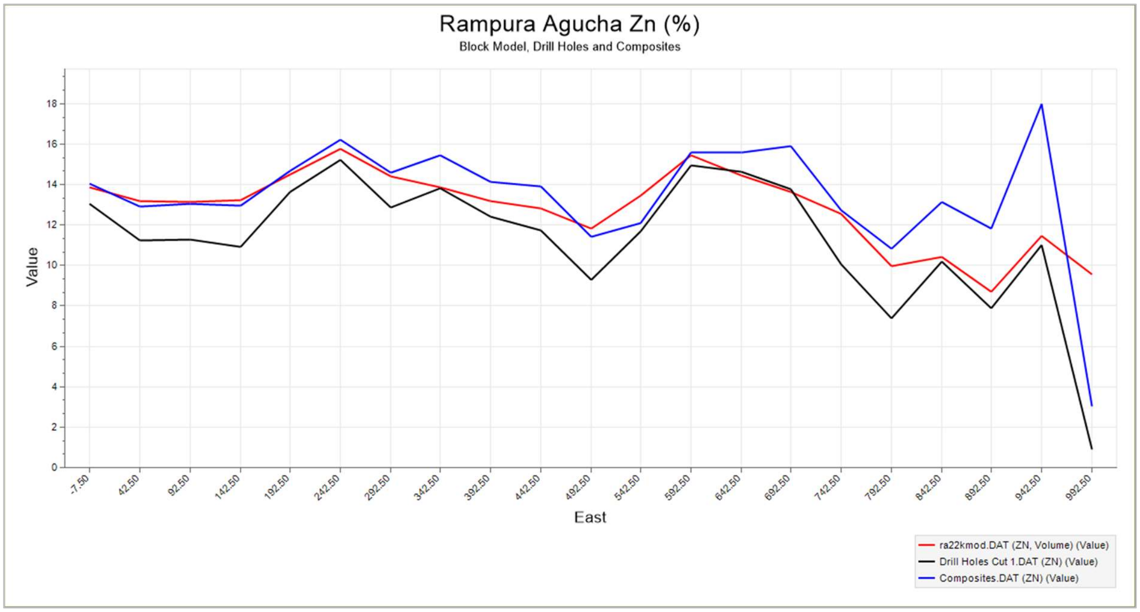


Figure 14: Swath plot (east) for the block model OK Zn % estimates versus the composite and drill hole grades

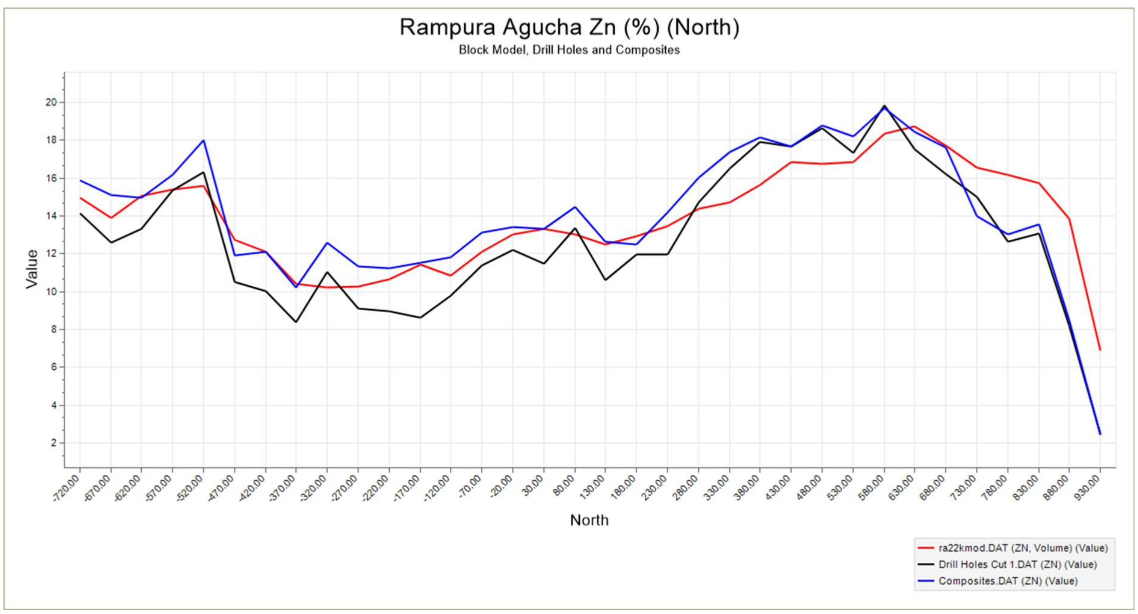


Figure 15: Swath plot (north) for the OK block model Zn % estimates versus the composite and drill hole grades

ABGM considers that sufficient check calculations have been conducted to conclude that the tonnage and grade estimates are considered valid, however there is room for improvement in terms of the estimation scheme and optimisation of the estimation parameters.

9.13 Mineral Resource Statement

The Mineral Resources exclusive of the Mineral Reserves as at the end of the last fiscal year are summarised in Table 16, whilst Table 17 summarises the same period where the Mineral Resources are inclusive of the Mineral Reserves.

The Mineral Resources Table 16 and Table 17 are reported at a ZnEQ COG of 3.46% for the Main Zone, and a PbEQ COG of 3.68% for the Galena Zone. Depletions as of 31 March 2022 have been removed.

Table 16: Mineral Resource Statement (exclusive of Mineral Reserves) – 31 March 2022

Classification	Tonnage	Grade			Metal Content		
	(Mt)	Zn (%)	Pb (%)	Ag (g/t)	Zn (Kt)	Pb (Kt)	Ag (Koz)
Measured	10.2	14.70	2.20	64	1,498	223	21,014
Indicated	0.1	17.60	3.00	80	20	3	290
Measured + Indicated	**10.3**	**14.70**	**2.20**	**64**	**1,517**	**227**	**21,304**
Inferred	17.6	6.00	3.60	97	1,060	638	55,174
Total	**28**	**9.20**	**3.10**	**85**	**2,578**	**865**	**76,578**

- Stated as exclusive of Mineral Reserves
- COGs - ZnEQ of 3.46% for the Main Zone, and a PbEQ of 3.68% for the Galena Zone
- Stated as underground Mineral Resources
- Measured classification is drilled at a 50 x 50 m spacing, between 50 m to 100 m for Indicated classification and a spacing of greater th33an 100 x 100 m for Inferred classification
- The Mineral Resource is limited to a depth from surface of -705 m
- The regional dip pillars are included in the inclusive Mineral Resource
- Mineral Resources are reported on a 100% basis
- Totals may not sum due to rounding

The Measured Mineral Resources as a proportion of the total Inclusive Mineral Resource as of 31 March 2022 accounts for ~33% of the tonnes, the Indicated Mineral Resources is ~44%, and the Inferred at ~23%.

Table 17: Mineral Resource Statement (inclusive of Mineral Reserves) – 31 March 2022

Classification	Tonnage	Grade			Metal Content		
	(Mt)	Zn (%)	Pb (%)	Ag (g/t)	Zn (Kt)	Pb (Kt)	Ag (Koz)
Measured	25.9	14.90	2.20	72	3,851	560	60,202
Indicated	33.6	13.70	1.30	41	4,612	448	44,167
Measured + Indicated	**59.5**	**14.20**	**1.70**	**55**	**8,464**	**1,008**	**104,369**
Inferred	17.6	6.00	3.60	97	1,060	638	55,174
Total	**77.1**	**12.30**	**2.10**	**64**	**9,524**	**1,646**	**159,543**

- Stated as exclusive of Mineral Reserves
- COGs - ZnEQ of 3.46% for the Main Zone, and a PbEQ of 3.68% for the Galena Zone
- Stated as underground Mineral Resources
- Measured classification is drilled at a 50 x 50 m spacing, between 50 m to 100 m for Indicated classification and a spacing of greater th33an 100 x 100 m for Inferred classification
- The Mineral Resource is limited to a depth from surface of -705 m
- The regional dip pillars are included in the inclusive Mineral Resource
- Mineral Resources are reported on a 100% basis
- Totals may not sum due to rounding

The net material difference between the Mineral Resource at the end of the 31 March 2022 fiscal year and the preceding fiscal year is demonstrated in table below. The differences relate to depletion or production, changes in commodity prices, additional resources discovered through exploration, and changes due to the methods employed.

Table 1-4: Net difference between the 31 March 2022 and 31 March 2021 exclusive Minerals Resources inclusive of Mineral Reserves

Classification	Tonnage	Grade			Metal Content		
	(Mt)	Zn (%)	Pb (%)	Ag (g/t)	Zn (Kt)	Pb (Kt)	Ag (Koz)
Measured	0.3	0.2	0.1	1	63	11	1,072
Indicated	-0.3	0	-0.2	-12	-57	-11	-1,004
Measured + Indicated	**0.0**	**0.0**	**0.0**	**0.0**	**6**	**1**	**68**
Inferred	-6.5	-2.5	0.5	9	-976	-110	-13050
Total	**-6.5**	**-1.1**	**0.3**	**4**	**-985**	**-110**	**-13,041**

The net difference between the mineral resources or reserves at the end of the last completed fiscal year and the preceding fiscal year, as a percentage of the resources or reserves at the end of the fiscal year preceding the last completed one.

An explanation of the causes of any discrepancy in mineral resources including depletion or production, changes in commodity prices, additional resources discovered through exploration, and changes due to the methods employed

9.14 Relevant Factors that may affect the Mineral Resource Estimates

It is the opinion of the Qualified Person that the likelihood of high-risk factors affecting the Mineral Resource estimates is low.

Risks that can affect Mineral Resource estimates include:

- Geological model
- Spatial representation of the drill hole data
- Sample QAQC
- Estimation methodology and assumptions
- Estimation search parameters
- Application of grade capping and/or cutting
- Further drill hole results that may impact the grade and tonnage estimates
- Changes to the parameters used to derive the COGs, such as metal prices, operating costs, metallurgical recoveries.

ABGM is of the opinion that these risks have a low probability of having a material impact on the Mineral Resource estimates.

9.15 Qualified Person's Opinion

Conclusions:

- The geology is understood well, and the geological model is sufficiently detailed to estimate reliable Zn, Pb, and Ag grades
- Historical reviews of the data and QAQC and concluded there is sufficient and spatially representative drill holes to estimate reliable grade and tonnage estimates.

- The regression method to determine the missing Ag assay results is appropriate
- The use of OK methodology is appropriate as is the related estimation parameters applied
- Sufficient estimation model validations have been undertaken and indicate the grade estimates are reliable
- The criteria used to define the Mineral Resource confidence categories are appropriate
- The parameters used to determine the COGs are appropriate
- The Mineral Resources are amenable to underground mining
- The estimation of the grades and tonnages have been performed to industry best practices and conform to the requirements of international Mineral Resource reporting codes
- Successful brown-fields exploration to replace Mineral Resources depleted by production has occurred and is on-going
- The nett differences between the most recent Mineral Resources and the previous fiscal year's Mineral Resources are well understood
- The persons undertaking the estimation and classification of the Mineral Resources are sufficiently experienced to undertake such
- The March 31, 2022, Mineral Resource estimate has been estimated in accordance with the December 26, 2018, SEC S-K1300 regulations.

Dr Heather King who reviewed the Mineral Resource estimates and statements is independent of HZL and registered as a professional with the South African Council for Natural Scientific professions (SACNASP) and the Geological Society of South Africa (GSSA).

The Mineral Resource is based on a geological model prepared by the HZL and Rampura Agucha technical teams. It has been independently reviewed by the Dr King. Furthermore, the review relies on information provided by HZL, along with technical reports by specialist consultants, and other relevant published and unpublished data, which include relevant geological data and information, and reports.

ABGM has endeavoured, by making all reasonable enquiries, to confirm the authenticity and completeness of the technical data upon which the review and Mineral Resource statement has relied. Dr Heather King is suitably qualified and experienced to act as the Qualified Person for the Mineral Resources. Dr Heather King was unable to visit the site.

Recommendations:

- The commodity prices for Zn and Ag should be reassessed based on both long-term prices and reliable forecasts
- Alternative geostatistical methodologies be applied in the validation of the OK estimates
- Interpolation and extrapolation of density estimates to facilitate a higher accuracy in the tonnage and metal content estimates

Consideration of geological and mining loss factors to facilitate a higher accuracy in the tonnage and metal content estimates.

10 Mineral Reserve Estimate

10.1 Basis, Assumptions, Parameters and Methods

The HZL mine operations and technical teams engage on a universal standard that is applied across all the mine operation from the geology resource estimations to applying the mine designs and evaluating the potential Mineral Reserves. The HZL Resources and Reserves technical team has kept extensive data that is used annually to do the annual mineral reserves statements and calculation. Each mine undergoes individual assessments and apply the modifying factors, grade cut-off calculation and assumptions.

10.1.1 Exchange Rates and Financial Assumptions

The exchange rates applied in the cut-off calculation are based on the LME forecasted indicated in the document below. The following rates were used by HZL:

Dollar to INR Conversion	Long term 5yrs
Dollar value in INR	76.65

Information received from HZL indicate the following parameters were used to calculate the cut-off grade.

Historical prices are stipulated in Table 18 below illustrating the last six years of commodity prices.

Table 18: Historic Commodity Prices

Year	LME Zinc		LME Lead		LBE Silver
(USc/lb)	(USD/t)	(USc/lb)	(USD/t)	(USD/oz)	(USD/oz)
2016	95	2,101	85	1,878	17.1
2017	131	2,892	106	2,327	17.1
2018	131	2,893	102	2,251	15.7
2019	114	2,504	91	2,006	16.2
2020	103	2,278	83	1,836	20.5
2021	120	2,646	88	1,940	23.8

LME Forecast used by HZL for the calculation of the cut-off grades are indicated below. The ten year average of the forecasted prices indicated were used to calculate the cut-off grades.

Table 19: LME Projected Commodity Prices

Particulars	UOM	FY'23	FY'24	FY'25	FY'26	FY'27	FY'28	FY'29	FY'30	FY'31	FY'32	Average
LME - Zinc	$/MT	3,183	2,911	2,684	2,621	2,658	2,706	2,706	2,706	2,706	2,706	2,759
- Lead	$/MT	2,179	2,047	1,974	1,962	1,997	2,082	2,082	2,082	2,082	2,082	2,057
- Silver	$/Troz	22.19	20.47	21.61	21.33	21.30	21.10	21.10	21.10	21.10	21.10	21.24
Ex Rate	Rs/USD	74.94	75.43	76.52	77.73	78.64	79.11	79.51	79.51	79.51	79.51	76.65

The 3 year and 10 year averages are indicated below

Table 20: Next three year average prices

Particulars	UOM	3yr Avg	10yr Avg
LME - Zinc	$/MT	2,926	2,759
- Lead	$/MT	2,067	2,057
- Silver	$/Troz	21.42	21.24
Ex Rate	Rs/USD	75.63	76.65

10.1.2 Cut-off Grade

A zinc and a lead concentrate are produced at RAM although the principal metal is zinc. Zinc grades at RAM are exceptional and there is minimal Mineral Resource within the orebody below COG. The COG and NSR assumptions used to support the F2022 Ore Reserve estimate are presented in the tables below:

Table 21: Cut-Off Grades and NSR Calculation inputs

Description	Units	Pb Con	Pb Con	Zn Con
Input Assumptions		**Pb**	**Ag**	**Zn**
Commodity Price	USD/t or USD/oz	2,057	21.24	2,759
Commodity Price	USD/t or USD/g	2,057	0.683	2,759
Exchange rate	USD:INR	76.65	76.65	76.65
Average grade	% Or g/t	5.65	146.1	0.05
Concentrator recovery	%	90	85	10
Concentrate grade	% Or g/t	70	1,710	51
Moisture content	%			
Payability/ smelter rec	%	93.5	93.5	95.6
Minimum deduction	% Or g/t			
Treatment charge	USD/dmt	305.3		231.6
Refining charge	USD/lb or USD/g		0.016	
Transport cost	USD/dmt			
Freight cost	USD/dmt	6.6		7.5
Mineral royalty	%	20.5	9.9	13.8
NSR Values		**Pb**	**Ag**	**Zn**
Gross payable	USD/dmt	1,346	1,092	1,345
Treatment charge	USD/dmt	-305		-232
Refining charge	USD/dmt		-25	
Transport cost	USD/dmt			
Freight cost	USD/dmt	-7		-8
Mineral royalty	USD/dmt	-276	-108	-186
Net payable	**USD/dmt**	**759**	**959**	**920**
Equivalent Grade Calculation				
Metal values	USD/t or USD/g	975	0.48	180
Equivalent grade factors	**no**	**1**	**0.048**	**0.185**
Equivalent grade	%Zn or %Pb	5.65	7.14	0.01
Total equivalent grade	%Zn or %Pb			**12.8**
NSR	USD/t rom	55.1	69.6	0.1
Total NSR	USD/t rom			**124.8**

10.1.3 Modifying Factors and Reconciliation

The mine has been using a Cavity Monitoring System ("CMS") for a number of years and all stope voids are regularly surveyed, and the results compared with the mine design. The results from some 520 stope CMS surveys is summarised below in terms of the pre-mining designed stope (includes planned dilution and ore loss) and the post-mined stope void. Dilution is principally associated with hanging wall and footwall waste from the moderate dip of the orebody of 55°, thinner ore zones, and poor ground conditions associated with a graphitic shear that is located within the orebody along strike. Historical results of the mine operations and the grade reconciliations are as followed:

Table 22: CMS Stope Reconciliation

Description	Unit	F2017 Actual	F2018 Actual	F2019 Actual	F2020 Actual	F2021 Actual
Planned Stopes	Mt	1.11	1.63	2.27	2.61	2.21
Zn grade	%	10.76	12.51	13.09	12.14	13.69
Pb grade	%	1.22	1.6	1.88	1.78	2
Ag grade	g/t	38.3	50.4	61.7	56.1	67.3
Mined stopes	Mt	0.99	1.51	2.22	2.65	2.43
Zn grades	%	10.18	11.83	11.91	11	11.1
Pb grade	%	1.15	1.51	1.71	1.62	1.61
Ag grade	g/t	35.9	47.8	56.2	51	54.1
Planned dilution	%	4.9	4.9	11.8	10.7	16.8
External dilution	%	6.2	6.2	11.8	12	13.8
Mining recovery	%	91.5	95	102.5	104.8	113.8

Table 23: Mined versus Processed Grades

Description	Unit	F2017 Actual	F2018 Actual	F2019 Actual	F2020 Actual	F2021 Actual
Mined	**Mt**	**1.38**	**2.08**	**3.33**	**3.94**	**4.27**
Zn grade	%	10.32	11.85	12.27	11.83	11.07
Pb grade	%	1.19	1.49	1.75	1.75	1.62
Ag grade	g/t	35.6	46.9	57.6	55.5	54.3
Processed	**Mt**	**1.07**	**2.31**	**3.4**	**3.91**	**4.29**
Zn grade	%	9.9	11.06	11.77	11.13	10.89
Pb grade	%	1.28	1.52	1.65	1.62	1.61
Ag grade	g/t	42.1	56.9	62.4	57.5	60.6
Tonnes Factor	%	0.78	1.11	1.02	0.99	1
Zn grade factor	factor	0.96	0.93	0.96	0.94	0.98
Pb grade factor	factor	1.08	1.02	0.94	0.93	0.99
Ag grade factor	factor	1.18	1.21	1.08	1.04	1.12

The zinc and lead grade reported as head feed to the plant compares reasonably to that projected and improvements to operating practices as well as the prediction assumptions are being undertaken to improve the correlation.

The modifying factors are generally well understood and based on extensive CMS reconciliation using mined stopes to date. The results of the reconciliation have been investigated and a relationship between the modifying factors and orebody thickness has been established. The mine continues with a number of initiatives to improve performance. It should be noted that regional dip pillars included in the Ore Reserve are assumed to have a 50% mining recovery.

Table 24: External Dilution and Mine Recoveries applied

Thickness	Bottom Up Stopes		Top Down Stopes	
	External Dilution (%)	Mine Recovery (%)	External Dilution (%)	Mine Recovery (%)
0-10	30.2	83.9	30.6	88.1
10-20	8.6	88.2	11.4	91.1
20-30	5.0	90.7	9.1	96.4
30-40	3.6	87.1	7.3	98.0
>40	3.0	92.9	8.2	99.0
Ore Dev	3.0	93.0	8.0	100.0
Reg Pillar	15.0	50.0		

10.2 Mineral Reserves

The classification block model reports developed before, was scrutinised and the tonnes were evaluated. This classification model is the start point of the Reserve estimation before external dilution and mine recoveries are applied. The model report suggests there is sufficient tonnes and grade in the model that is used to develop the Reserve Statement in March 2022, to determine the final Mineral Reserves. A direct comparison between the tonnes and grade was conducted as a check. The block model report indicated the tonnes and grade are higher than what is reported in the estimates.

Table 25: Reserve Model vs Reserve Estimate

	Classification Model (Output)				HZL Reserves Estimate			
Ore Reserve	Tonnage (Mt)	Grade (Zn %)	(Pb %)	(Ag g/t)	Tonnage (Mt)	Grade (Zn %)	(Pb %)	(Ag g/t)
Proved	15.7	13.2	1.9	68	14.7	12.4	1.8	65
Probable	33.0	12.4	1.2	37	32.3	11.6	1.1	35
Ore Reserves (Total)	**48.8**	**12.6**	**1.4**	**47**	**47.0**	**11.8**	**1.3**	**44**

The current Reserve statement reports there is 47.0Mt at 11.8 g/t Zinc, 1.4% Lead and 47g/t Silver.

The Ore Reserve estimate for RAM is based on a computerised stope design for the Measured and Indicated Mineral Resource above -555 mRL and excludes the regional dip pillars. It has been assumed in previous technical reviews that 50% of the regional dip pillars can be recovered however, it is likely this could be increased if supported by an appropriate mine design. The zinc orebody at RAM extends below the limit of the Indicated Mineral Resource and currently classed as Inferred Mineral Resource. The reserves at RAM would be significantly increased once the resource below the -555 mRL is upgraded by exploration drilling and sampling.

The use of CMS and reconciliation with the stope designs has led to a good understanding of the modifying factors of dilution and mining recovery which are generally dependent on orebody thickness and the influence of the graphitic shear. Total dilution of 30% was considered by HZL in previous reserve statements elevated and related to the high proportion of stopes that are located in the thinner parts of the orebody. Generally, HZL has in previous years, expect total dilution to be in the range of 10-15% which would be considered good for the RAM zinc orebody. Mining recovery has improved from 85% to 90% which is also considered reasonable for the orebody and mining methods. The orebody is generally of exceptional zinc grade and there is very little resource below the COG of 3% ZnEq (fully diluted). The current COG is based on F2022 operating costs which are probably higher than the long-term costs once full production is established, and the shaft is being used for hoisting. ABGM would expect that separate long term COGs should be calculated for ramp haulage and shaft hoisting even though the orebody is not sensitive to COG.

The mine has been in the process of building up production and has trialled a number of different variations of the sub-level open stoping with paste fill mining method. A Top-Down (Overhand) mining method is the preferred configuration for mining below -230 mRL using 25 m levels and 15 m crosscut spacings. Currently production can be obtained from a number of levels above -230 mRL including the flanks of the old open pit. Once production is limited to a single Top-Down mining front with sequencing of the stopes to optimise the best geotechnical configuration, ABGM would expect mining rates of 3.0-4.0 Mtpa from a single mining horizon. To meet the installed plant capacity at RAM of 6.5 Mtpa, Previous reviews suggests that a second mining front would need to be established at depth, for example at -505 mRL, which could be accessed relatively easily from the new shaft infrastructure.

The build-up in production has been inhibited by lower than planned development performance in turn impacted by establishment of the necessary levels, ventilation and supporting infrastructure requirements. Equipment performance in terms of availability and utilisation is also below industry norms. The mine has a number of initiatives to improve performance in these areas.

ABGM recommends, that current performance as well as allowing for reasonable improvement needs to be built into a practical LoM schedule linked to the mine design that includes sufficient buffers for infill drilling, the necessary support and paste fill requirements. The current Lom plan is sufficient for annua purposes, however having a more robust LoM plan in place would lead to more realistic targets for and allow for quicker analysis and schedule optimisation opportunities. The critical path for production performance at RAM is considered to be underground development, the geotechnical constraints in terms of stope sequencing and operating the production shaft at the design capacity.

10.3 Classification and Criteria

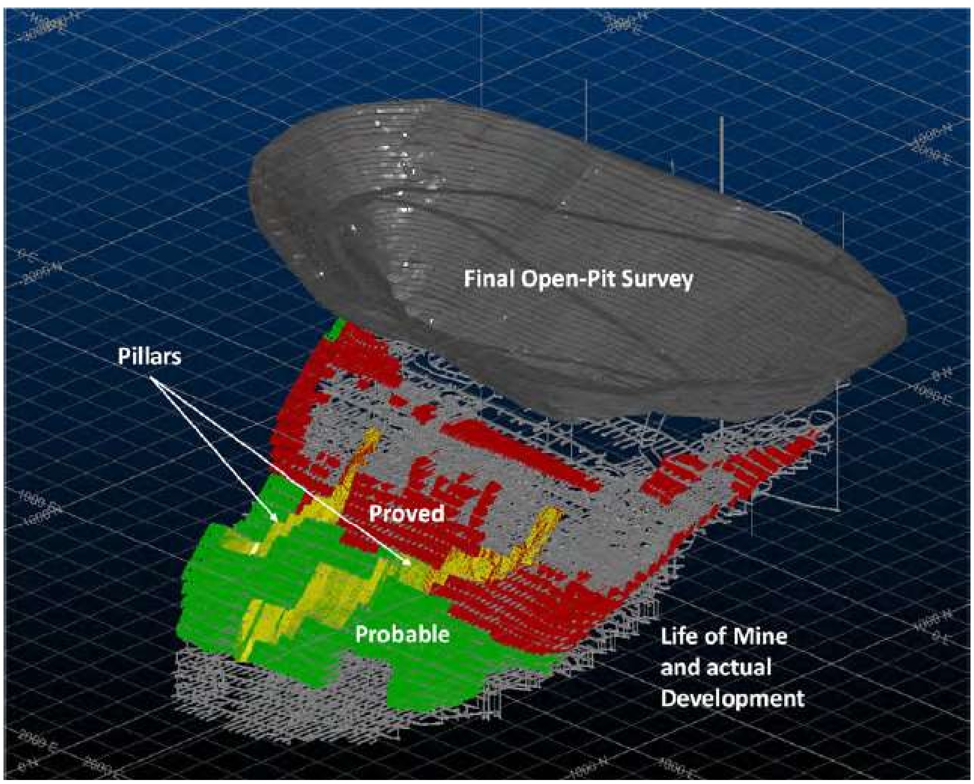


Figure 16: Overall Mine Design & Mineral Reserves Classifications

10.4 Relevant Factors

Table 26: Input and calculation parameters

No	Description	Units	RAM ZnEq			RAM PbEq		
			Zn Orebody (Res+Rev)			Pb Orebody (Res)		
			Pb Con	Pb Con	Zn Con	Pb Con	Pb Con	Zn Con
			Pb	Ag	Zn	Pb	Ag	Zn
1.00	**Input Assumptions**							
1.01	Commodity Price	USD/t or USD/oz	2,057	21.24	2,759	2,057	21.24	2,759
1.02	Commodity Price	USD/t or USD/g	2,057	0.683	2,759	2,057	0.683	2,759
1.03	Exchange rate	USD:INR	76.65	76.65	76.65	76.65	76.65	76.65
1.04	Average grade	% or g/t	1.65	66.1	11.10	5.65	146.1	0.05
1.05	Concentrator recovery	%	61.7	29.6	90.0	90.0	85.0	10.0
1.09	Concentrate grade	% or g/t	62.2	1,192	50.5	70.0	1,710	51.0
1.10	Moisture content	%						
1.11	Payability/ smelter rec	%	94.8	98.2	96.0	93.5	93.5	95.6
1.12	Minimum deduction	% or g/t						
1.13	Treatment charge	USD/dmt	363.8		229.3	305.3		231.6
1.14	Refining charge	USD/lb or USD/g		0.024			0.016	
1.15	Transport cost	USD/dmt						
1.16	Freight cost	USD/dmt	6.9		9.3	6.6		7.5
1.17	Mineral royalty	%	20.2	9.4	13.8	20.5	9.9	13.8
2.00	**NSR Values**							
2.01	Gross payable	USD/dmt	1,212	800	1,337	1,346	1,092	1,345
2.02	Treatment charge	USD/dmt	-364		-229	-305		-232
2.03	Refining charge	USD/dmt		-28			-25	
2.04	Transport cost	USD/dmt						
2.05	Freight cost	USD/dmt	-7		-9	-7		-8
2.06	Mineral royalty	USD/dmt	-245	-75	-184	-276	-108	-186
2.07	**Net payable**	**USD/dmt**	**596**	**697**	**914**	**759**	**959**	**920**
3.00	**Equivalent Grade Calculation**							
3.01	Metal values	USD/t or USD/g	592	0.17	1,629	975	0.48	180
3.02	**Equivalent grade factors**	**no**	**0.363**	**0.01060**	**1.000**	**1.000**	**0.04886**	**0.185**
3.03	Equivalent grade	%Zn or %Pb	0.60	0.70	11.10	5.65	7.14	0.01
3.04	Total equivalent grade	%Zn or %Pb			**12.40**			**12.80**
3.05	NSR	USD/t rom	9.8	11.4	180.8	55.1	69.6	0.1
3.06	Total NSR	USD/t rom			**202.0**			**124.8**
4.00	**Costs and COG Calculation**		**F2022 Act**	**F2022 BP**	**F2023 BP**	**F2022 Act**	**F2022 BP**	**F2023 BP**
4.01	Mining	INR/t rom	2,633		2,553	2,134		2,134
4.02	Processing	INR/t rom	1,041		1,088	615		615
4.03	Overhead	INR/t rom	0		0	0		0
4.04	Transport	INR/t rom						
4.05	Sub-total operating cost	INR/t rom	3,674		3,641	2,749		2,749
4.06	Corporate, royalty & others	INR/t rom						
4.07	Other costs	INR/t rom						
4.08	Sustaining capex	INR/t rom	651		69	0		0
4.09	Sub-total other costs	INR/t rom	651		69	0		0
4.10	Total cost	INR/t rom	4,325		3,710	2,749		2,749
4.11	Total cost (USD)	USD/t rom	56.4		48.4	35.9		35.9
4.12	Waste development cost	INR/t rom	1,000		1,342	1,000		1,000
4.13	Economic cut-off margin	%						
4.14	**Diluted %ZnEq Cut-off Grades**							
4.15	Operating cut-off (diluted)	%ZnEq or %PbEq	3.46		2.97	3.68		3.68
4.16	Section cut-off (diluted)	%ZnEq or %PbEq	4.26		4.05	5.01		5.01
4.17	Economic cut-off (diluted)	%ZnEq or %PbEq	3.46		2.97	3.68		3.68
4.18	**Modifying Factors**							
4.19	Planned dilution	%	9.4		9.4	9.4		9.4
4.20	External dilution	%	9.6		9.6	9.6		9.6
4.21	Mining recovery	%	90.8		90.8	90.8		90.8
4.22	**Insitu %ZnEq Cut-off Grades**							
4.23	Operating cut-off (insitu)	%ZnEq or %PbEq	4.15		3.56	4.41		4.41
4.24	Section cut-off (insitu)	%ZnEq or %PbEq	5.11		4.85	6.01		6.01
4.25	Economic cut-off (insitu)	%ZnEq or %PbEq	4.15		3.56	4.41		4.41
5.00	**Cut-off grade formulas**							
5.01			ZnEq for Zn = 1.000 x %Zn			PbEq for Zn = 0.185 x %Zn		
5.02			ZnEq for Pb = 0.363 x %Pb			PbEq for Pb = 1.000 x %Pb		
5.03			ZnEq for Cu = 0.000 x %Cu			PbEq for Cu = 0.000 x %Cu		
5.04			ZnEq for Ag = 0.01060 x g/t Ag			PbEq for Ag = 0.04886 x g/t Ag		
5.05			Ore Reserve COG = 3.46 %ZnEq (diluted)			Ore Reserve COG = 3.68 %PbEq (diluted)		
5.06			Mineral Resource COG = 4.15 %ZnEq (insitu)			Mineral Resource COG = 4.41 %PbEq (insitu)		

10.5 Recommendation and conclusions

The mine is conducting their estimation process in the correct manner, and they have ensured there is enough resource drilling, modelling and estimations to ensure the Reserve estimation are more accurate and real. The operations have undertaken many projects and studies from various mining consultancies globally and has implemented recommendations accordingly.

11 Mining methods

11.1 Introduction

The developmental activities at Rampura-Agucha (RA) Mine as an open pit mine was started in 1989, for extraction of ore body and production started from 1991 onwards. RA open pit mine had ramped up to capacity near to its planned capacity of 6.15 million ton per annum (mtpa) ore production over the period. The open pit mining operation ceased in March 2018, after attending its economic ultimate pit depth of 400m (-10mRL) below surface (+390mRL). Beyond 400m depth, underground mining was considered as the best suitable option for sustaining ore production from RA Mine.

Rampura-Agucha underground (RAUG) Mine development work commenced in year 2010 and started ore production through underground stoping operation from August 2013 onwards. As planned, Underground mine operated concurrently with the open pit mine till March 2018, and after completion of open pit operation, open pit area is being used to support underground mine operation in terms of material (ore and waste) handling, and infrastructure requirement like ventilation, dewatering, etc.

Presently, RAM (Rampura-Agucha Mine) through underground operation is producing at the rate of 6.0 Mtpa.

11.2 Underground Mining

At the conceptual stage, the RAUG mine study had been carried out by M/s SRK, UK and AMC Australia and other renowned consultants independently. During study various mining methods were initially assessed to determine the most suitable mining method for Rampura-Agucha underground Mine deposit. Geotechnical investigations for RAM concluded that a major shear zone, termed the orebody shear zone (OSZ), is continuous throughout the resource and that the shear's location within the ore zone does have consistency. It has been evaluated through geotechnical model result that with the presence of OSZ, the ground conditions shall possess will be challenging during underground mining operation.

The studies carried out in past determined that the performance of mining is constrained by the dip of ore body and poor rock mass. Thus, based on various criteria for mining method selection (like as Safety, Reliability, Stability, Recovery, Productivity and overall economics), the proposed underground mining method considered as Long Hole Open Stope (LHOS) and its variants (in bottom up and/or top down approach, as per stability requirement) with backfill.

Paste fill (PF) is most preferred and majorly used backfill method at RAM, as it fulfils stipulation of mining method for high strength backfill, to facilitate redevelopment in cemented backfill. However, depending upon requirement on type of backfill and actual ground condition apart from Paste Fill (PF), Cemented Rock Fill (CRF) or Rock Fill (RF) may also be used as method of backfill in the mined out void area.

11.2.1 Development

Lateral development (ramps, footwall drives and stope cross cut drives) activities in underground mine are being carried out through drilling holes in the rock by using development drill machines. Charging of holes is being carried out by conventional method with cartridge explosives and as well as mechanized method with bulk emulsion. Perimeter or pre-split techniques are also considered on case to case basis, in development drives to minimise over break and damage to surrounding rock. After drilling and blasting operation, the blasted rock materials are transported through mechanized means using appropriate capacity equipment like LHDs (Load Haul Dump) and LPDTs (Low Profile Dump Truck) / Mine trucks. Vertical development activities in mine are being carried out by conventional method like drop raise technique and also by mechanized means with raise borer machine.

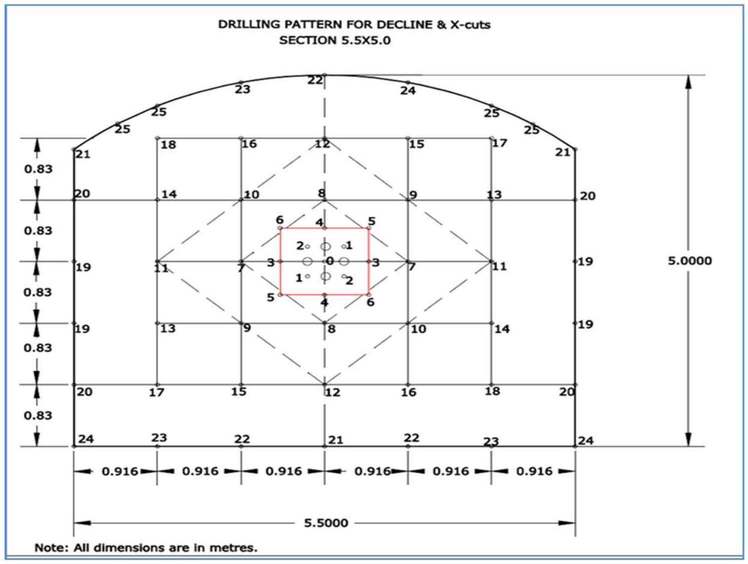
System of Drilling and Blasting for Development:	
Drilling pattern for mine lateral developmental drives	For lateral development Burn Cut drilling pattern is followed at mine. However, on case to case basis based on rock characteristics drag or other drilling may also be followed. Illustrative drawing of  one development drive for burn cut pattern is shown below.
Maximum number of holes blasted in a round	**Development**: 58-80 Nos. depending upon dimension of drive.
Charge per round (Kg)	**Development**: 200-400 Kg
Charge per hole (kg)	**Development**: 3-5 kg per hole
Type of explosive	**Development:** Cartridge / Emulsion explosive / Site mixed emulsion
Powder factor (Norms) Rock development- Ore development-	 1.07-1.40 kg/ton (Avg. 1.15 kg/ton) 1.07-1.40 kg/ton (Avg. 1.12 kg/ton)

Figure 17: Development Drill and Blast System

11.2.2 Stoping

Stope blocks within ore body formed through development between upper and lower levels shall be drilled using the production drill machine as per drill design to minimize the dilution. Charging of drilled holes shall be carried out by conventional method with cartridge explosives and as well as mechanized method with bulk emulsion. Perimeter or pre-split techniques may also be considered on case to case basis, in hanging wall and footwall contact of ore body to minimise dilution and damage to surrounding rock. After drilling and blasting operation, the blasted rock materials are transported through mechanized means using appropriate capacity equipment like LHDs (Load Haul Dump) and LPDTs (Low Profile Dump Truck) / Mine trucks.

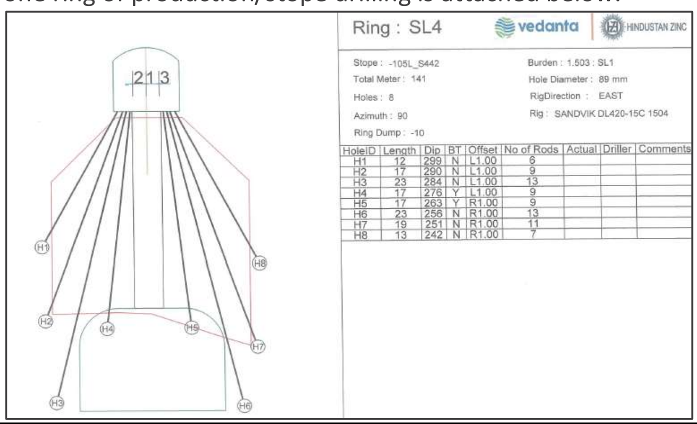
System of Drilling and Blasting for stoping:	
Drilling pattern in Stopes	For stope extraction ring drilling in fan cut pattern is followed at mine. Based on rock characteristics down hole or up hole production/stope drilling may be carried out. Illustrative drawing of one ring of production/stope drilling is attached below. 
Maximum number of holes blasted in a round	**Stoping:** 10-130 Nos. depending upon the ground conditions.
Charge per round (Kg)	**Stoping:** 0.5-9.0 ton
Charge per hole (kg)	**Stoping:** 100-250 kg per hole
Type of explosive	**Stoping:** 76-83mm diameter Cartridge / Emulsion / Site mixed emulsion with 100/250/400 g booster / Primer cartridge charge.
Powder factor (Norms) **Stope-**	0.24-0.45 kg/ton (Avg. 0.39 kg/ton)

Figure 18: Stoping Drill and Blast System

11.2.3 Mining Method

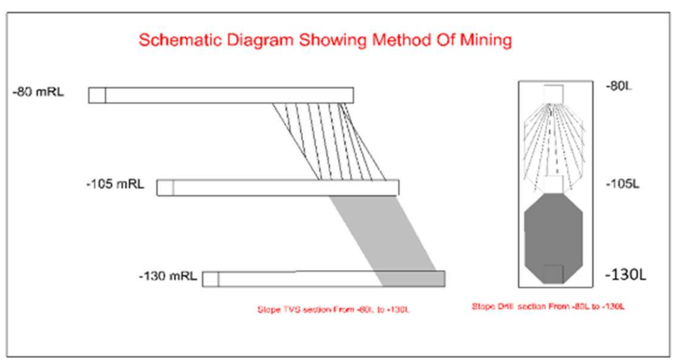
The studies carried out in past determined that the performance of mining is constrained by the dip of ore body and poor rock mass. Thus, based on various criteria for mining method selection (like as Safety, Reliability, Stability, Recovery, Productivity and overall economics), the proposed underground mining method considered as Long Hole Open Stope (LHOS) and its variants (in bottom up and/or top down approach, as per stability requirement) with backfill.

Paste fill (PF) is most preferred and majorly used backfill method at RAM, as it fulfils stipulation of mining method for high strength backfill, to facilitate redevelopment in cemented backfill. However, depending upon requirement on type of backfill and actual ground condition apart from Paste Fill (PF), Cemented Rock Fill (CRF) or Rock Fill (RF) may also be used as method of backfill in the mined out void area.

Major advantages of preferred mining method LHOS with paste fill in Top-Down/Underhand mining approach are.

- The re-developed drifts provide a safe working environment that is fully supported using friction bolts / split sets / rock bolts and shotcrete material
- The underhand sequence avoids multiple encounters with the orebody shear zone (OSZ)
- Stoping activities are planned to occur beneath safe exposures of the cemented paste fill
- Allows larger LHOS stopes to be mined under stable engineered cemented paste fill

LHOS with backfill in Bottom-Up / Overhand Mining approach



LHOS with backfill in Top-Down / Underhand Mining approach

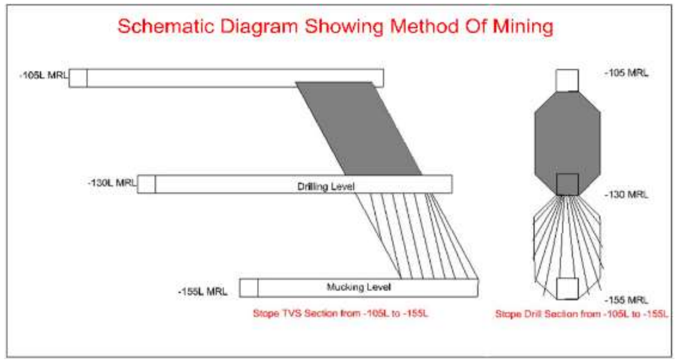


Figure 19: Long-hole Open Stope Mining Method Schematics

The mining fronts advance either in continuous or primary-secondary sequence towards the resource limits with stopes planned to be mined. The preferred extraction approach and sequence preliminary supports, to best manage the OSZ under stable engineered cemented backfill. Also, suggested approach and sequence avoids multiple encounters with the OSZ, with only one initial exposure for the stopes at the top of each mining block and exposure in stope walls thereafter, as the stoping activities are planned to occur beneath exposures of the cemented backfill.

In consideration to apparent width of ore body, transverse manner of stoping is preferred. However, longitudinal manner of stoping may also propose, wherever the ore body apparent width is favourable with respect to ground stability. It is proposed that each of these mining methods utilizes a high level of mechanization and proposed variations on these methods shall be blind up hole stoping depending on actual ground conditions.

Production plan of achieving 6.15 mtpa has been conceptualized based on the selection criteria of safety, deployment of highly mechanised and productive equipment, efficiency (optimized recovery and dilution), economics, ease of execution and working environment.

Multilevel mining has been proposed for extraction of stopes based on current mining method. Further based on requirement of ground stability of mine, mining method shall be optimized, to maximise recovery from the stopes while maintaining overall stability of the stopping fronts. Working methodology of the mining method is presented in the table below for reference purpose. However, the present methodology shall be refined as per the behaviour of the ground, success of methodology and the safety feature.

After completion of mining from all blocks/levels, the crown, sill, rib and stope pillars (wherever placed as per stability requirement) are then proposed for extraction using centre out continuous retreat method at the end of the life of mine.

11.2.4 Backfill

The open stope after complete withdrawal of broken muck, shall be back filled with paste fill/RF/CRF from upper level. During filling, it will be ensured that complete filling of stoping voids have been done. The most preferred backfill methodology practiced is mine is Paste fill.

Paste fill is defined as dewatered tailings that are non-segregating in nature and bleed little or no water. Paste does not segregate during low velocity transport, making its conveyance through pipes practical.

Paste fill is typically placed with a cement binder at a slump of 150 to 200mm to minimize pressure losses in the pipes and maximize the final paste strength.

The paste fill plant process circuit consist of dewatering of the tails slurry in a conventional thickener to 50 to 60% by weight. The product is further dewatered in a disc filtration plant to produce a wet filter cake comprising of 80-85% solids. Batches of this filter cake are then mixed in a high intensity shear mixture with water, cement and fly ash in a required proportion to make a consistent paste product.

The paste is conveyed in the mine through directly dropping into paste holes or pumping to destination through pipeline laid in underground roadways.

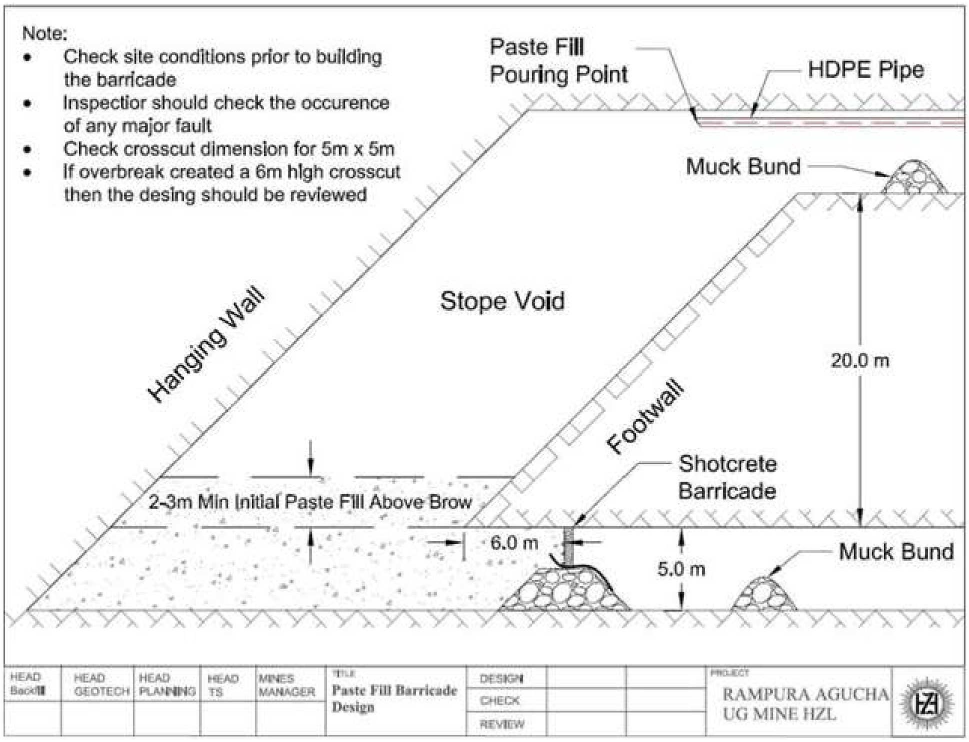


Figure 20: Illustration of Backfill

11.2.5 Mode of Entry

The picture shown below depict the Mine access system cross section. In the picture as build development area and proposed development area of the mine are pictorially described.

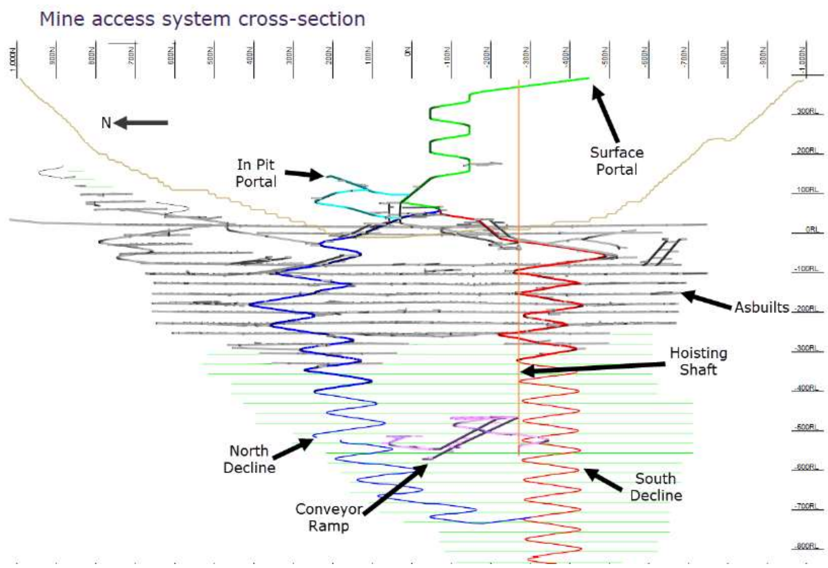


Figure 21: Mode of Entry - RAM

The Mine access approach to the deposit is presently by twin ramps at down gradient of 1 in 7. Out of two ramp, first ramp alias main ramp (E -485.38, N -449.44, +391.2mRL) is from surface (Surface Portal), which branches into north and south ramps at about +55 mRL, and second ramp alias open pit ramp (E +100.8, N +216.6, +140mRL) is from open pit (In pit Portal) at +140mRL, which is connected to the main ramp at +95mRL and connecting with north ramp at +36mRL. Both ramps have an independent access and named as Open Pit Ramp alias North Ramp and Main Ramp alias South Ramp, respectively. The primary equipment and manpower access will be provided by the Main Ramp and material movements by Open Pit Ramp. Open Pit Ramp also provide separate access to manpower under emergency.

A vertical main shaft alias hoisting or production shaft (E -600, N -270, +391.6mRL) of 7.5 m diameter, 950m depth is also commissioned. The vertical main shaft is equipped with two distinct compartments, one to facilitate the skip winding arrangement and another for man winding arrangement. Presently main shaft is well connected with north ramp of mine and serves as a secondary egress and provide accesses to the lower working levels of mine. Developmental activities are in progress to establish connection with south ramp as well.

Secondary egress from the mine is provided by twin ramps and main shaft system, also in connecting footwall drives between various working levels. Presently two working levels are well equipped with ladderway arrangement, further ladder way connecting the far north and south ends of the footwall drives are also conceptualized through 1.5m -3.5m sized drop raise or raise bore, to provide a secondary means of egress from extreme working areas.

Two ventilation shafts (North ventilation shaft E -414.8, N +337.7, +394.5mRL and South ventilation shaft E -520, N -665, +391.2mRL) of 7.5m diameter of approx. 450m depth have been completed and connected with mine ventilation network. Both ventilation shafts connection is at -80mRL in underground. Both shafts equipped with adequate capacity ventilation fans to facilitate working in south and north zones of ore body.

Apart from above, the life of mine layout has considered the followings regarding interaction between the open-pit and the underground mining operations:

- Major underground infrastructures located at surface and in open pit with safe standoff distances.
- The twin decline system having finished dimension of 5.5 m width and 5.0 m height, provides:
    - Flexibility and capacity for large material movements along the extensive 1.5 km strike length of ore body, and
    - Flexibility for footwall drive and stope cross cut development more rapidly with opening the headings from both decline sides.
    - Faster access and reduced level haulage distance.

11.2.6 Stoping Sequence

11.2.6.1 Stoping

For stoping, the stope cross cuts in waste and ore shall be developed across the strike of ore body at an interval of 15-45 m from the footwall drives. Stoping activity at RAM is majorly in primary-secondary sequence. The primary stopes shall be extracted in sequential manner. Secondary stopes cross cuts in ore will be developed and stopes will be extracted after completion of two adjacent primary stopes. However, depending upon ground condition of rock continuous sequence of stoping shall also be followed. Production drilling shall be done in downward/upward direction depending upon the requirement of Mining Method, extraction sequence and operational safety.

11.2.6.2 Stoping Geometry

The strike length of the ore body in this block will be divided into stoping panels. The stoping shall be carried out depending on the width of the ore body. Stoping panels have been divided into primary and secondary or continuous manner as per requirement of ground. A rectangular prism shaped transverse stope has been proposed for stoping having 0.5m ring collar spacing and 2-4m ring toe spacing with 2-3.5m ring burden. The 15m stope strike length is determined from geotechnical analysis. Further, depending upon the ground condition, width and length of stope may vary. It also suits single draw point flat bottom transverse stope layout. The transverse ore drive and crosscuts will be 4.5-5.0m wide and 4.5-5.0m high.

11.2.6.3 Stoping Panels and Their Sequence Of Extraction (Bottom- Up)

The stope panel shall be 15m wide (along strike), 25m high (Level interval) and up to 10-50m Long (along width of Ore body). Above process is shown in the schematic diagram below

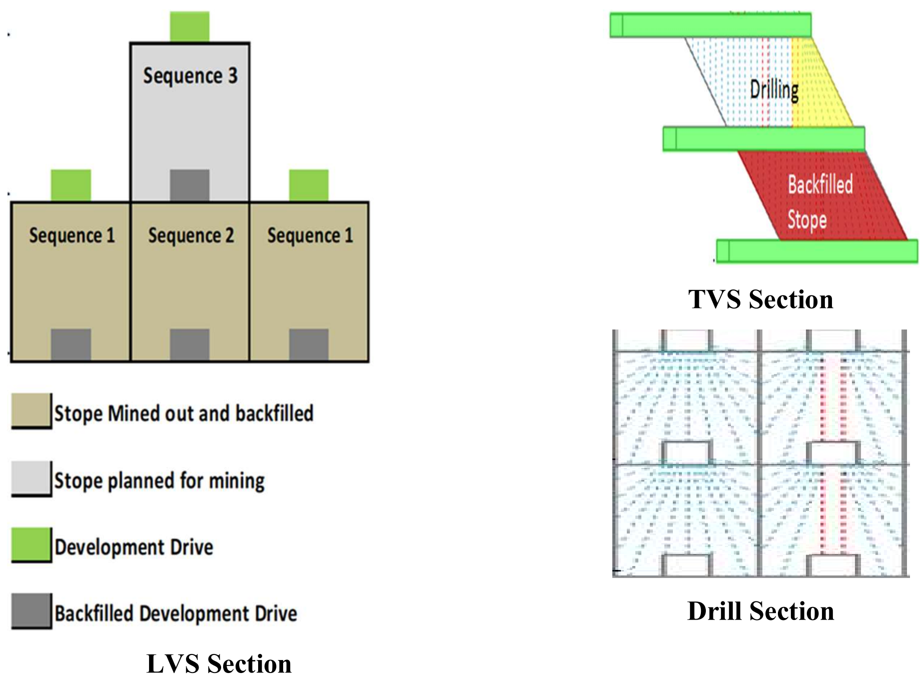


Figure 22: Stoping Panel Sequence – Bottom up

11.2.6.4 Stoping Panels and Their Sequence Of Extraction (Top Down)

The stope panel shall be 15m wide (along strike), 25m high (Level interval) and up to 10-50m Long (along width of Ore body). The stope will be sequenced below a completely paste filled stope. Extraction shall be conducted with two scenarios.

- Through Up hole drilling from Bottom Level: In some stopes, the production blast hole will be drilled from extraction Level in upward direction.
- Through Down hole drilling from Top Level: In some stopes, the production blast hole will be drilled from Drill Level in downward direction. In this scenario, before drilling operation to commence, the drive at Top Level will be developed in paste filled stopes.

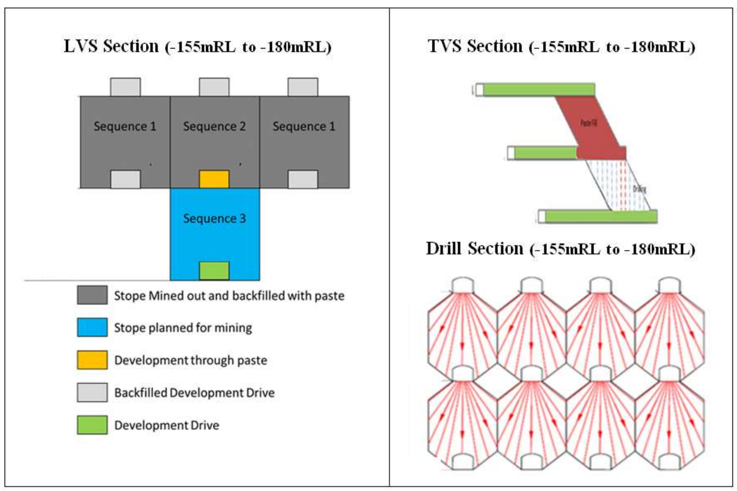


Figure 23: Stoping Panel Sequence – Top Down

11.2.6.5 Ground Support Requirement

All drives and crosscuts shall be systematically supported using rock bolts, split sets, cable bolts, wire mesh and Shotcreting etc. as per recommended systematic support rules. In the case of brow support, the support consists of rings of cable bolts located at the specified distance from the brow. The fully extracted empty stope voids shall be backfilled with PF/RF/CRF as per applicability and requirement depending upon the ground condition.

11.2.6.6 Drilling

- Development headings of 4.5-5.0mW x 4.5-5.0mH is being drilled with jumbo drill having a hole diameter of 45 mm. The length of each hole is approx. 4.0-6.0m, having 50 to 80 holes per round. Wet drilling system is being followed.
- Production drilling shall be done by using production drills to make 89-152 mm diameter holes. These holes shall be charged with either emulsion or cartridge explosives with a charge per delay shall be such that ground vibration is within the safe limit.
- Production drilling shall be of 89-152mm diameter, up to 30-40 m long with the ring burden 2-3.5 m and toe spacing will be up to 4m depending on the rock characteristics and fragmentation obtained. The slot will be widened to full width and stripped. Rings will be then blasted against the free face using delay detonators. The slot will be either at footwall or hang wall contact of ore body depending upon the ground condition and requirement and free face will retreat towards other stope end.

11.2.6.7 Slot Drilling

For opening of the 15m wide slot to full ore body width and drilling shall be carried out by Simba/ Solo drill rigs from the drill levels of the stoping blocks. Drill holes of 89-152mm diameter shall be drilled from drill drives.

11.2.6.8 Stope Drilling

The blast holes in the stope shall be drilled from the respective drill levels of the stoping blocks with 89-152mm diameter holes in downward direction with production drill machine. For drill holes the ring spacing and toe burden will be 2.3-3.5m and 2.3-4.0 m respectively. However, depending upon fragmentation the spacing and burden shall be modified so as to achieve optimum results. The blast hole rings shall be drilled in vertical plane across strike along the drill drives.

11.2.6.9 Blasting

After completion of production drilling, the holes will be charged with either emulsion or cartridge. For blasting, the delays will be so arranged so as to blast the slot raise, widen it and then fire the blast rings. Rings shall be blasted in multi delay so that the blast vibrations are kept within safe limits. Numbers of ring for blasting will be dependent upon stope dimensions. However, care shall be taken to maintain a sufficiently uniform blasting profile from hang wall to footwall in each ring through proper hole measurement, charge length and adequate delay in holes.

11.2.6.10 Broken Ore Removal at Extraction Level

As blasting progresses, the stope panel will be emptied of broken ore using diesel LHD. Remote control operation of the LHD shall be used to recover ore from the hanging wall side of the slot area at the final clean up stage. For rest of the stope area complete ore is recovered from the trough drives and cross cuts.

The ore will be directly loaded into LPDT through LHDs. Ore will be hauled out through the ramp to surface stockpile and then beneficiation plant for treatment. The waste from underground will be either transported to surface or disposed-off in void stopes as per requirement.

11.3 Geotechnical Parameters

This mine is a geotechnically most critical mine of Hindustan Zinc, as the complete working is on-going below an exhausted huge Open Pit of 400m deep. Here the rock conditions are a bit challenging as the average RMR of country rock is 50 to 55 which falls under the Poor category. As the rock is highly foliated, the rate of deformation while mining in this kind of rock mass is too high, which leads to excess rehabilitation or re-supporting of our excavation drives.

11.3.1 Historic LOM Geotechnical Study Results

Detailed life of mine geotechnical study was done by Beck Engineering. The extraction strategy and study summary are as follows,

- With increasing depth, the stress in work areas is increasing, leading to higher strain and deeper wall damage. The strain around tunnels matures faster, and the zone of influence of the stoping extends deeper into the footwall. Over time, the occurrence of seismic events and rapid increments of strain will increase.
- Recommended a modified plan with an additional stabilisation pillar, so the mine strike length would now be divided into 4 blocks. Previous mine plans simulated by BE included 2 major pillars dividing the mine into 3 blocks. The 3x pillars were sited in the recommended plan to minimise loss of the best value ore as far as possible. A sill in the north block can also be designed, to add a 5th active mining block. Generally, this sill should be around 8 levels below the current active stoping front in the north

- Simulation of the recommended plan shows an improvement to expected performance compared to past analysis even with the 5th block and suggests that the plan is a good fit to the conditions, though a number of residual challenges must be managed:
    - The mine already plans a modified stand-off with depth and an escalation of ground support. The model suggests the proposed increase with depth is a good compromise and that the previously recommended escalation in support with depth (2019) is a good fit to the projected changes.
    - We modelled both continuous and primary-secondary extraction for the same global extraction sequence. If the mine can follow the plan, the primary secondary stoping can continue for longer. The actual depth at which continuous stoping should be implemented fully is hard to estimate, as it offers some benefits to stability in some areas while being less reliable for recovery in others and the best compromise depends on many operational factors.
    - The best approach is to strictly limit the lag between the primary and secondary front to less than or equal to 3 levels to extend the life of the primary secondary sequence and to consider the local experience to appreciate where secondary stopes perform well and where they do not and extrapolate an area by area forecast. In general, the mine should be able to extend the utility of pri-sec stoping across most of the front for several years if the secondary stopes are not left to lag excessively.
    - Eventually, the Galena orebodies need to be integrated into the mine plan. We have previously recommended how this should occur.

Considering the normal escalating hazards with depth, even with an improved extraction strategy the mine will need to employ enhanced control measures as mining progresses. Recommended an approach to managing the various control measures and hazard identification process focusing on an observational approach supported by analysis. The approach builds on the current knowledge and skills of the mines own engineers as they are best placed to manage the situation.

A key input will be the measurement and orebody knowledge program. The current plan involves expanding the seismic system and improving on the processes for measuring residual support capacity.

11.3.2 Stress and Strain

To minimise the increments of strain through the mine resulting from larger than currently experienced seismic events and to minimise the potential for events, it is important that the mine continue to progress to the regularised mining fronts and managed lead-lags recommended by the geotechnical team. The mine should further prepare for the increased hazard by continuing the escalation of support standards (BE 2020) and integrating improved orebody knowledge into decision making and planning.

- The progressive increased stand-off built into the design, according to past recommendations is a good compromise.
- Intermittent cable bolting of the footwall drives in the GBSG rock mass will likely be required on and below -280mL or -305mL. This includes cable bolting of the backs and eastern shoulder of the drive due to progressive deformation associated with stoping on the level.
- Systemic cable bolting of the footwall drives will be required below -330mL due to deformation associated with nearby stoping. The pattern cable bottling is not forecast to be required during development and may be undertaken on a campaign basis to improve efficiency.
- Model forecasts show a step changing the cable bolting requirements will be necessary to maintain long term access on the footwall drives, particularly in the GBSG below the -305mL to -330mL levels. We note the onset of deformation will begin slowly during the initial stages of stoping on the level and increase rapidly with progressive nearby stoping on the level.
- Cable bolting of pillar noses in the waste cross cuts should be undertaken below -305 level due to forecast damage. Osro or mesh straps and cable bolts are recommended to be installed in the pillars below -405 level due to increasing deformation in the forecasts.
- The length of resin bolts in the decline should be increased to 3m for decline development below approximately -380mL. Cable bolting is required below -705mL in the decline based on current forecasts. However, we note the limited geological information available at this depth. We recommend the support requirements be reviewed based on local geological conditions on an ongoing basis with increasing depth.
- High stress and deformation are forecast in proximity (within ~50m) to the regional pillar. This results in a higher closure and depth of fracturing, requiring higher levels of ground support and potential requirements for stripping and rehabilitation of drives to maintain long term access. Level access should not be planned within proximity of the regional pillars.

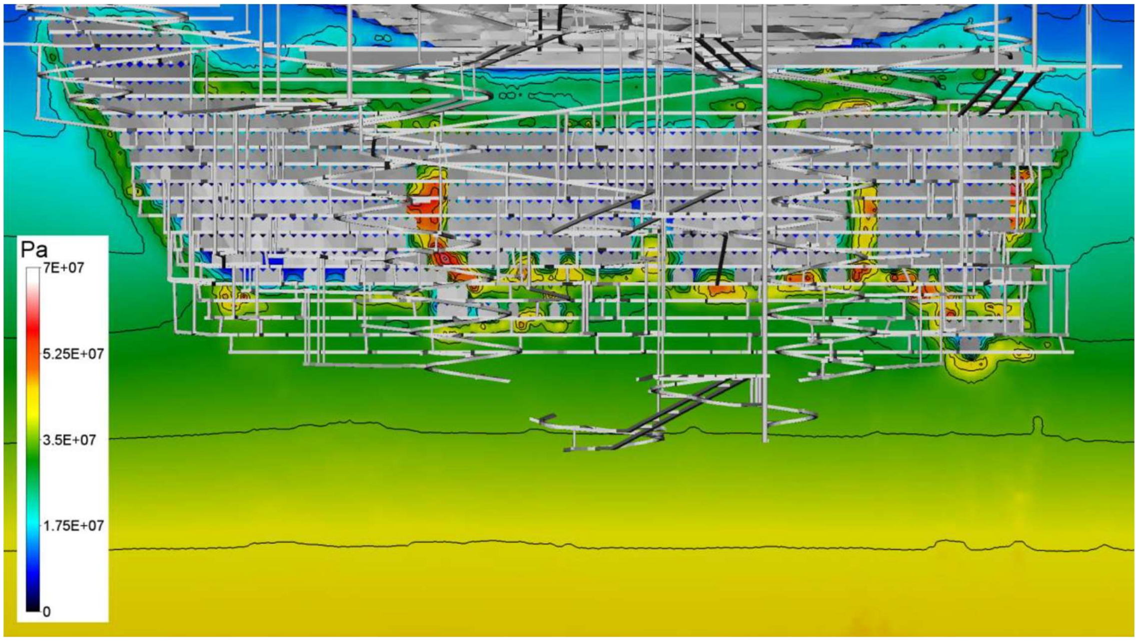


Figure 24: Stress on a mid-plane through the orebody. 2022

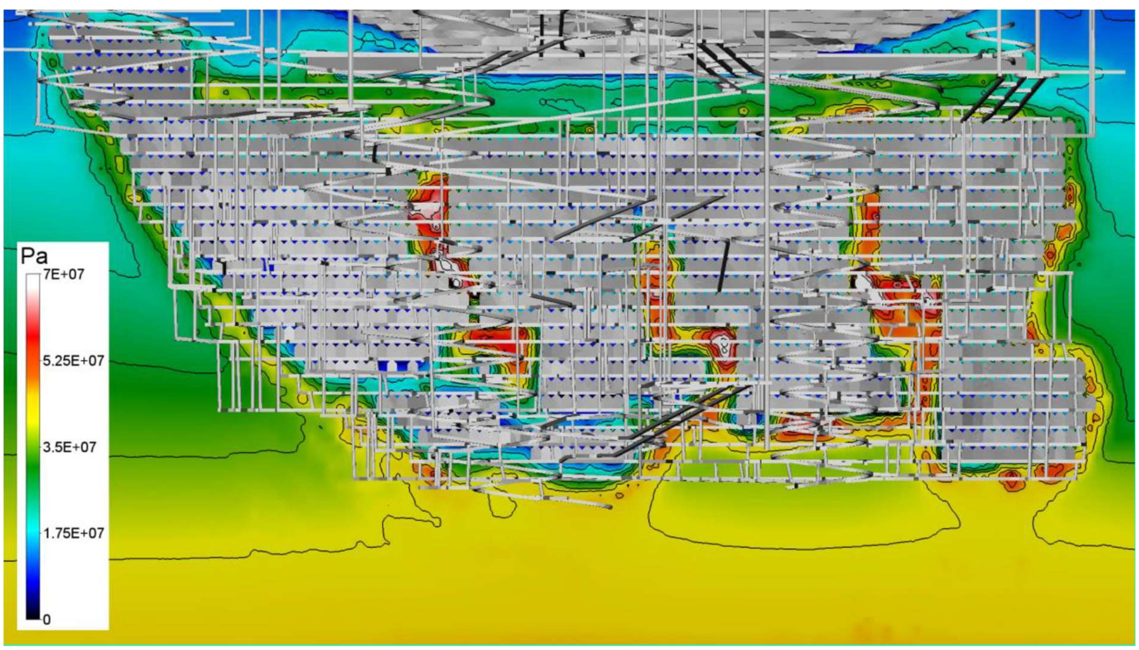


Figure 25: Stress on a mid-plane through the orebody. 2026

11.3.3 Rock Mass Quality

Table 27: Rock Mass Quality Summary

Domain	Sub Domain	RQD				Joint Sets	Jn	Jr	Ja	Q'		
		Avg	25 %	50 %	75 %					25%	50%	75%
Foliated Schist / Gneiss	GBSS / GBSG	82	76	88	92	3	9	3	3	8.5	9.7	10.2
	GBG	74	69	79	93	3	9	3	2	11.5	13.1	15.6
Psammitic Bands and Intrusions	QTZ/QBG	91	90	97	100	2	4	3	3	22.5	24.3	25
	PEG	64	54	70	87	3	9	3	3	6	7.8	9.7
	AMP	80	73	83	91	3	9	3	1	24.4	27.7	30.4
Ore Zone	Ore	57	30	65	83	2+R	6	2	4	2.6	5.4	6.9
	OSZ	22	26	29	32	3+R	12	1.5	4	0.8	0.9	1

11.3.4 Stope Level intervals

The following table summarises the Sub level intervals and the Geotech parameters for primary support.

Table 28: Sub level intervals and the Geotech parameters for primary support

S. No.	Block (mRL)	Name of Working Stope	Size of the Stope (m x m)	Thickness of Crown Pillar (m)	Thickness of Sill Pillar (m)	Thickness of Rib Pillar (m)	Size / Shape of Man Way (m x m)	Size / Shape of Ore Pass (m x m)	Method of Stowing / Back Filling	Proposed Stope Design Sequence of Stoping	Sub Level Interval (m)
1	Main Block (-80mRL to -655mRL)	Main Block Stopes	15m width along strike length of ore body X 5-40m length across strike length of ore body depending upon the width of ore body	60 Meters	NA	NA	2-4m*2-4m depending upon the requirement	3.5 Dia	Paste Fill/Rock Fill/Cement Rock Fill	Diamond Shape stope with stable shoulder and trough/Primary-secondary and contineous	25m (Floor to Floor)
2	North Block (173mRL to -80mRL)	Main Block Stopes		60 Meters	NA	NA	2-4m*2-4m depending upon the requirement	3.5 Dia	Paste Fill/Rock Fill/Cement Rock Fill	Diamond Shape stope with stable shoulder and trough/Primary-secondary and contineous	Varies between 18m and 25m (Floor to Floor) depending upon the dip of the ore body
3	South Block (73mRL to -80mRL)	Main Block Stopes		60 Meters	NA	NA	2-4m*2-4m depending upon the requirement	3.5 Dia	Paste Fill/Rock Fill/Cement Rock Fill	Diamond Shape stope with stable shoulder and trough/Primary-secondary and contineous	

11.3.5 Hydrogeological Parameters

The local hydrogeological environment is well described in two hydrogeological reports prepared by Hydro-Geo Consultants Private Limited (HCPL) in 2006 and 2015. Recommendation into water control have been suggested and implemented by the RAM mine operation successfully over the past years. Some of the main groundwater sources and means to control them is indicated below.

11.3.5.1 Main Aquifer – Fractured Basement Rocks

The main local (and regional aquifer) is the basement complex, regionally referred to as the Banded Gneissic Complex (BGC) of the Bilwara Super Group. The following mine nomenclature is used:

- Ore host sequence – Graphite Mica Schist (GMS)
- Footwall and hanging wall sequence – Granite, Biotite, Sericite, Gneiss (GBSG).

The basement rocks themselves are largely impermeable, with bulk (average) aquifer properties being controlled by the distribution of secondary permeability and porosity associated with shears, faults and joints. In general, comparatively higher permeability is associated with shallow (<100m depth below surface) GBSC rocks and with the ore host shear (in the GMS rocks).

Based on regional data, and limited site test pumping, HCPL (2006 and 2015) concluded that bulk average basement rock aquifer permeability was in the range 0.02 to 0.5m/d (~2 x 10-7 to 5 x 10-6 m/s). These are considered to be reasonable "undisturbed" values for aquifers of this nature.

However, it is possible that the permeability of some shear/fault zones could be higher (up to 1m/d) and it is likely that permeability will increase in some fracture zones as a result of ground deformation above underground mine development (Beck Engineering, 2016).

Groundwater salinity (of inflows reporting to the current underground mine) is reported to be generally, in the order of 4,000 to 5,000mg/L TDS, but with some isolated "pockets" of groundwater which exceed 40,000mg/L TDS. Inflows to the main shaft during construction had a salinity of around 45,000mg/L TDS.

11.3.5.2 Secondary Aquifer – Shallow Alluvium/Eluvium

The basement rock aquifer is overlain by a thin (generally <10m thick) veneer of alluvial sediments and decomposed basement. These surface sediments are generally above the regional water table but form a shallow intermittent (and perched) aquifer during and following the wet season.

Groundwater salinity is reported to be around 1,500mg/L TDS, with some higher values (2,500mg/L TDS) reported at local bogs/swamps, likely due to evaporative concentration.

11.3.6 Ground Water

11.3.6.1 Groundwater Recharge

Groundwater recharge is by infiltration of rainfall (and runoff) into the shallow sediments and then by

vertical leakage into the basement rock aquifer where shears, faults and joints subcrop (or via old exploration drill-holes). Recharge will be focussed along the main surface water flow pathways, especially the overflow drainage line downstream of the Agucha Pond.

In place (e.g. at the southern margin of the open pit), recharge to the deeper basement rock aquifer is likely to also be enhanced by flow through old underground mine workings. It is also likely that leakage from the TSF, the waste rock dumps and the Agucha Pond will contribute to recharge to the shallow (and deep) aquifers.

11.3.6.2 Groundwater Discharge

The main groundwater discharge processes are:

- Baseflow from the shallow aquifer to local surface water drains during and after each wet season.
- Evapotranspiration from shallow, boggy areas.
- Pumping from bores and wells, including the numerous irrigation tube wells and dug wells, and the open pit dewatering borefield.
- Inflows to the mine.

11.3.7 Mine inflow mechanisms

The main mechanism for groundwater inflow to the open pit and underground mine workings is flow through open fractures. Key characteristics of this mine inflow mechanism are:

- There are (and will be) only limited inflows through the general basement rock itself.
- Inflows will be focussed along open shear/fault zones.
- The brackish to saline nature of most inflows to the current underground mine workings suggests limited hydraulic connection with surface water and that the inflows (at least to date) have been derived from groundwater storage.
- However, some reported variations in groundwater salinity indicate that there is some mixing with shallow recharge water.

- Some reported variations in mine inflows also indicate some direct recharge pathways to shallow sections of the underground mine, particularly within the footprint of the open pit.

11.3.8 Impacts of mine dewatering

As a result of the low bulk permeability of the basement rock aquifer, the drawdown impacts of mine dewatering (open pit and underground) will generally be constrained to the immediate area of the mine.

That is, there will be a steep "cone of depression" in the water table around the mine. Monitoring bore/piezometer data (available during the site visit) are confined to the immediate pit area and show water table drawdowns ranging from <10m up to 80m. Reported drawdowns in more distant bores are generally less than 10m.

Notwithstanding the available data, it is possible that there could be some drawdowns in excess of 10m extending along discrete fracture pathways away from the pit.

11.3.9 Underground mine inflow risks

A number of potential mine inflow risks were identified during the site visit and during pre-reading and review of background documents (particularly reports by Beck Engineering). The major potential risk to underground mining is clearly the potential for inflow from the pit sump following wet season rainfall runoff.

However, the risks associated with inflows from the pit sump and the lesser risks associated with other inflow mechanisms are not considered to be fatal flaws to underground mining and can all be managed by

relatively straightforward and commonly used mine water management strategies. The specific risks and

management strategies are outlined below.

It should be noted that the definition of risks (volumes/rates of potential inflows) are at an order of magnitude level of reliability and the strategies presented are at a conceptual level. Further, more detailed

work will be required as follows:

- Hydrogeological (and hydrological) analysis to provide more reliable quantitative estimates of potential inflows (if required). This will most likely require external technical input.
- Confirmation of favoured concepts (where there are options) and detailed design of risk management measures. This should largely be undertaken by HZL mine planning and mine operations personnel, with external technical support where required.

11.3.10 Inflow from Pit Base

11.3.10.1 Background

At present, wet season runoff to the pit is managed by an in-pit sump pumping system at the pit base. However, following the completion of mining and progressive deformation of the basement rocks above the underground mine development, it is expected that there will be some bench scale pit wall failures (mainly sloughing) especially on the hanging wall. This will constrain access to the pit base and the ability to pump water from the pit.

The underground mine plan includes a 70m crown pillar in the ore host unit (from the pit base at -10mRL to -80mRL). However, geo-mechanical modelling (Beck Engineering 2016) predicts that progressive deformation of the crown pillar will occur as the underground mine develops which will lead to increasing vertical permeability within the crown pillar and the FW and HW contacts at the margins of the crown pillar.

Vertical permeability could increase by several orders of magnitude. Beck Engineering predicted leakage rates through the crown pillar (assuming such an increase in vertical permeability) range up to around 60L/s. However, it is recognised that flows of such magnitudes (and consequent flow velocities) could result in physical erosion of the flow pathways and lead to “piping failures”. If this were to occur, then potential leakage rates could increase significantly.

11.3.10.2 Inflow Risk

As outlined above, the inflow risk relates to the potential for the development of a pit lake during the wet season and the leakage of water (at potentially high rates) from the pit lake to the underground mine through the crown pillar. Preliminary estimates of the potential mine inflows from pit leakage have been made based on the following assumptions:

- Maximum wet season rainfall – 1017mm (based on maximum on site record – recorded in 2011)
- Peak 3 day storm event – 530mm (conservatively based on twice the maximum daily rainfall on site record - 265mm in 2012).
- Runoff coefficient of 0.5 for overall wet season.
- Runoff coefficient of 0.8 for the peak 3 days storm.

Based on the above, the following preliminary estimates were derived:

- Peak wet season:
  - Total volume of runoff to pit – 660,000m3 (this would fill the pit to 22mRL if no leakage).
  - Average leakage rate over 4 month wet season – 64L/s.
- Peak storm:
  - Total volume of runoff to pit – 550,000m3 (this would fill the pit to 20mRL if no leakage).
  - Time for pit base to drain at various leakage rates:
    - at 50L/s – 128 days
    - at 100L/s - 64 days
    - at 200L/s – 32 days
    - at 300L/s – 21 days-

There are additional risks associated with the peak storm event. Depending on the actual initial rates of leakage from the pit, the pit lake will rise for a period until the pit finally drains to the underground mine.

Initial leakage rates could also be less if mud and silt (which will accumulate in the base of the pit) cause some interim blockage of the flow paths, resulting in even higher pit lake levels (and higher hydraulic gradients) following the storm. The additional risk relates to the potential for the higher hydraulic gradients to suddenly overcome the blockage of flow paths which could result in an uncontrolled “mud rush” or “water rush”.

11.3.10.3 Risk Management Strategies

The key management strategy to reduce the risk of high and/or uncontrolled water (or mud) inflows to the underground mine, is to reduce the build-up of runoff water at the pit base. That is, to install and operate a pit drainage system (that does not require ongoing access to the pit base). There two broad approaches to reducing the volume and head of water in the pit that could be easily adopted, as follows:

- Intercept (some) runoff from within the pit, above the pit base.
- Drain water directly from the pit base to underground pumping stations via engineered structures and bypass the crown pillar.

The first option will only be able to intercept some runoff and the second option would be required in any event, but at less capacity than if it were the sole option adopted. For either option, or combination of options, there will still be some residual leakage through the crown pillar and interception of (some of) any such residual leakage can be achieved using inclined drain-holes.

11.3.11 Interception within the Pit

It may be possible to install interception trenches/sumps on the pit haul ramps with pipe work back to the pit crest along the ramps. Such a system is in place at the Ernest Henry Mine in North Queensland (Australia) and has been successfully used to reduce runoff volumes to the pit base and subsequent leakage into the subsidence zone above sub-level cave underground mine workings.

The degree to which a similar approach could be adopted at RAM will depend on how long various sections of the haul remain in the pit remain accessible. For example, Figures below shows potential catchment areas that could be intercepted by haul ramps sumps that extend down to the southern pit switchback (FW and HW – Catchment A), and then down to the northern pit switchback (FW only – Catchment B).

Haul ramps sumps shown for Catchment A could intercept up to 25% of the runoff to the pit base. Haul road sumps shown for Catchments A and B could intercept up to 50% total pit catchment.

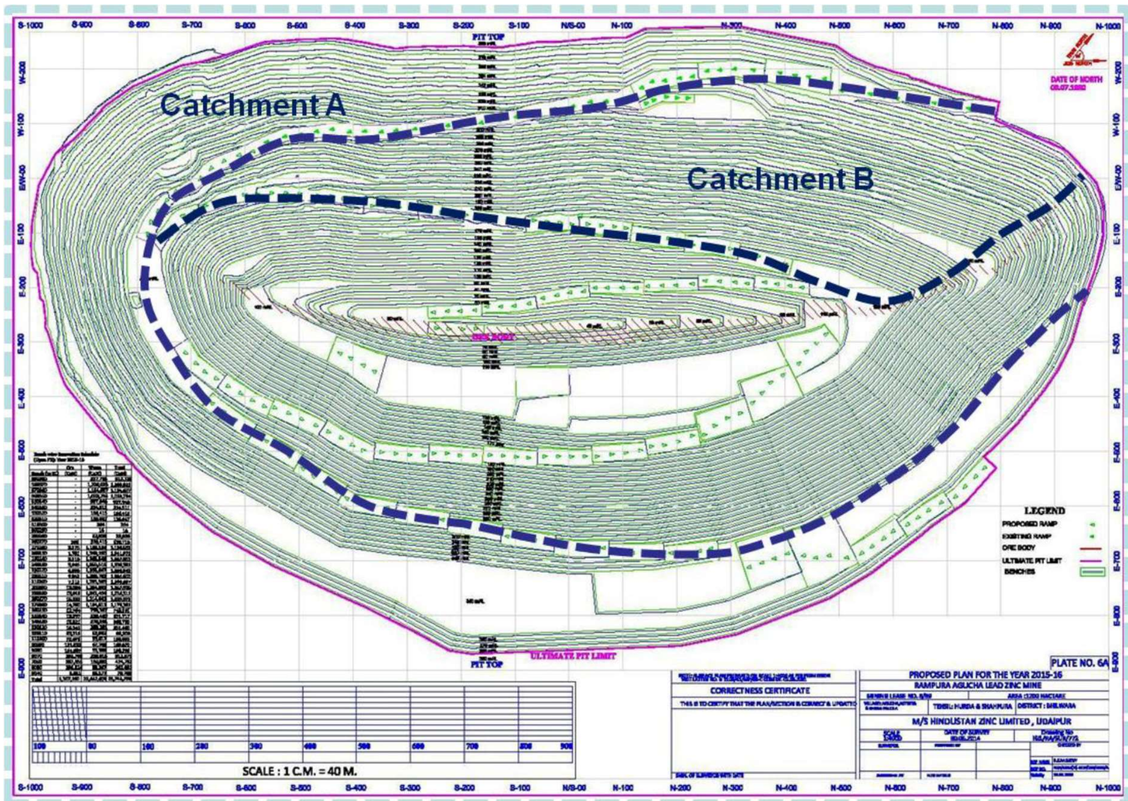


Figure 26: Examples of Catchment Areas Intercepted by Haul Ramp Sumps

11.3.12 Pit Drainage Systems

These involve using existing/modified mine workings and/or sump bores to drain any pit lake directly to the underground dewatering system. Figure 27 and Figure 28 schematically show both options. Please note that both figures also show interception drain-holes (refer next section). Key features and potential variations on concept design for each option are as follows:

- **Using existing/modified mine workings**
- The concept system as shown makes use of the existing access drives to the CAP stope at the +13mRL (although some of the access drives are as low as 0mRL).
- At least two access drives should be used (spaced along strike) to provide 100% back-up.
- Water from the pit lake would decant to the access drives and be directed to a pumping station. It is noted that the current mine plan includes the pump station at the -80mRL.
- Sediment traps should be installed along the access drives and remotely operated flow control systems (barricade doors or valved conduit pipes) should be installed to allow for control of inflows when required for pump station maintenance.

- This system will result in a pool of “stagnant water” below the lowest inlet level of the access drives and could result in a stagnant pool some 10 to 23m deep.
- Such a system is likely to produce relatively “clean” and fresh water to the pump station.

A potential modification would be to decline from a higher level; (say +30mRL or around 10m above the maximum pit lake level) down to a breakthrough to the pit at around +10mRL (or lower) with floating pumps to pump water over the “hump” at +30mRL. This would provide for a natural P-trap to control water flow to the pump station.

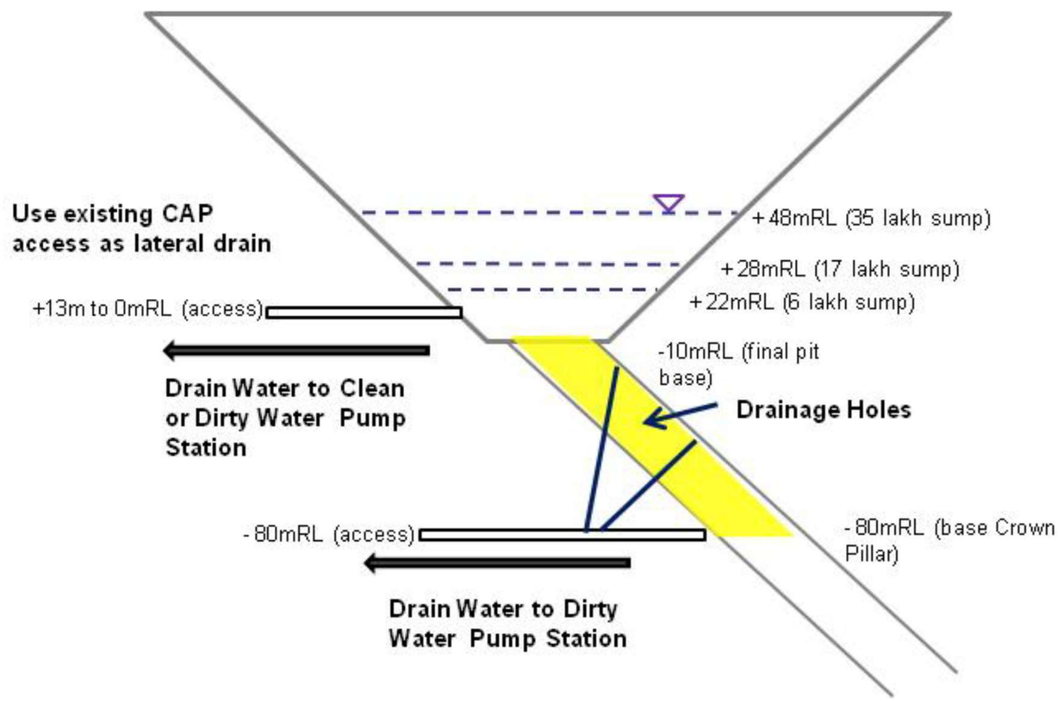


Figure 27: Schematic of Pit Drainage System Using Existing Mine Workings

- **Using sump bores**
- The concept system as shown involves the drilling and construction of a number of large diameter bores from the pit floor (or one or two benches up from the pit floor) into an underlying access drive. Figure below shows the bores connecting to the -80mRL (same level as the planned pump station) but the sump bores could connect with higher levels if required.
- The bores should be lined with large diameter (at least 300mm) ABS or PVC casing and pressure grouted into place (using a high strength but “plastic” grout mix) to provide a long term seal against flow around the casing annulus and to minimise corrosion.

- The top of the casing will need to be extended above the pit base (by at least 10m) and protected against slumping damage by a protective sleeve (e.g. concrete rings) and coarse rock rip-rap.
- At least four such sump bores should be installed (spaced along strike) to provide for high rate drainage and to provide 100% back-up.
- Sediment traps should be installed along the receiving drive (e.g. -80m access drive) and control valves should be installed at the base of the sump bore casing to allow for control of inflows when required for pump station maintenance.
- This system will result in a pool of "stagnant water" below the top of the sump bore casing and could result in a stagnant pool at least 10m deep.
- Such a system is likely to produce comparatively "dirty" but fresh water.

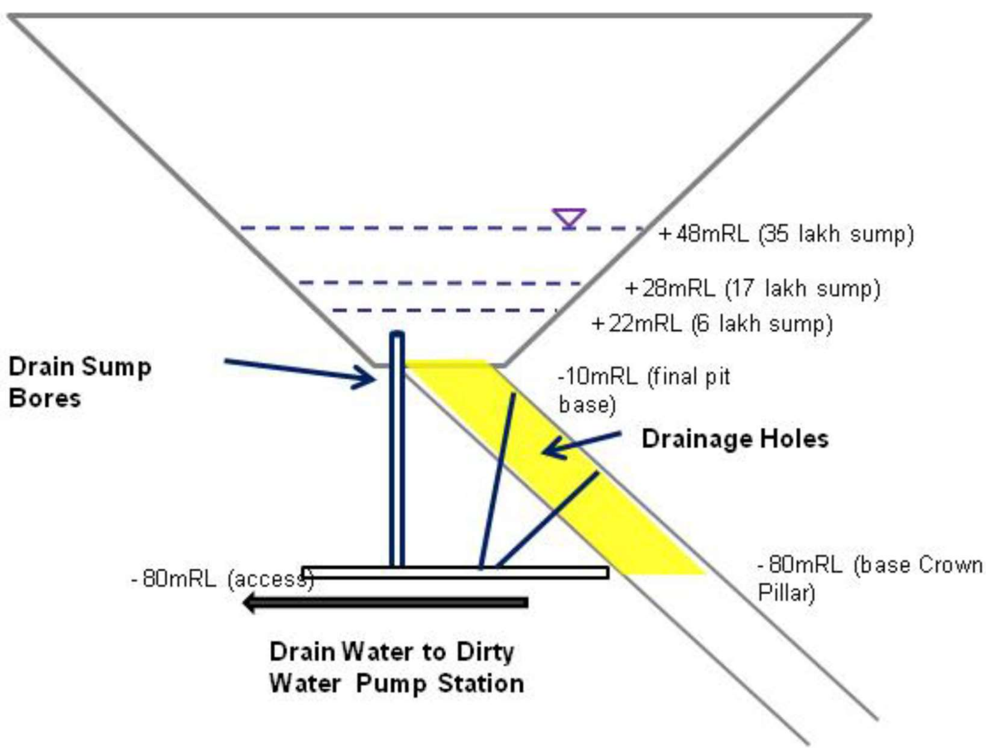


Figure 28: Schematic of Pit Drainage System Using Sump Bores

The former system provides more flexibility and security in the long term. There will always be access to the drive away from the pit and new access to the pit can be developed if required. That is, if the existing access gets blocked by material from pit wall slumping/sloughing. With the latter system, it would not be possible to install new sump bores (if required) once access to the pit base is lost.

11.3.13 Residual Leakage Interception

As outlined above, regardless of which pit drainage system is adopted, there will likely be some residual leakage from the stagnant pool (below the drainage system inlet level) through the crown pillar. To reduce the volumes of any such residual seepage, it is recommended that inclined interception bores should be installed. Key features of these bores are as follows:

- The drain-holes can be drilled from any access drive below the top of the crown pillar.
- Drain-holes should be drilled at regular locations (say 50m) along strike. At least two drain-holes should be fan drilled from each location to provide broad coverage of the width of the crown pillar.
- The drain-holes should be pre-collared (to at least 6m depth) and the collars grouted into place to prevent uncontrolled leakage around the collars.
- The drain-holes should then be drilled through the crown pillar and in to the HW contact zone. The drain-holes do not need to be lined.
- The drain-hole collars should be equipped with flow control valves and pressure gauges.

11.3.14 Inflow from basement rock aquifer

11.3.14.1 Background

Groundwater flow through permeable fractures. As outlined in Section 1.2, most of the groundwater inflows that have occurred to date have been sourced from groundwater storage, with only minor influence of recharge from surface waters observed at shallow depths. The implication of this for future mine inflows is that the longer term rates if inflow will be largely controlled by the bulk permeability of the basement aquifer as a whole.

The predicted deformation of the rock mass above the mine workings (Beck Engineering 2016) may influence short term mine inflows as more permeable conditions will be created close to the mine workings. However, the main impact of this will be to release water from storage (close to the mine) more quickly that from un-deformed basement rocks. Once this near mine groundwater storage has been released, mine inflows will be controlled by the background permeability.

Unless deformation breaks through to the surface, there will be no potential for increased hydraulic connection to shallow aquifer or surface water sources. Longer term mine inflow rates will be influenced by two competing processes:

- Potential for increasing inflows as the depth of mining increases (and the hydraulic gradient towards the mine increases).
- Potential for decreasing flows with time as the radius of influence of dewatering expands (and the hydraulic gradient decreases).

In cases where bulk permeability remains relatively constant with depth, once the open pit has been developed to near final plan, these two processes largely cancel each other out and mine inflows remain relatively consistent over the remaining life of mine. Where bulk permeability declines with depth, then inflows tend to decline with time.

11.3.14.2 Inflow Risk

Groundwater modelling undertaken as part of a previous hydrogeological study (HCPL 2015) predicted that overall mine inflows would increase from around 40L/s in 2014-15, to around 50L/s at the end of open pit mining, and then to around 55L/s at the end of underground mining. The modelling adopted constant bulk permeabilities of 0.2m/d (horizontal) and 0.02m/d (vertical). We have some concerns about the way the model was set up, but we believe that the relative differences in predicted inflows are of the right magnitude.

It is noted that total predicted inflows include any pumping from pit perimeter bores.

Current total mine inflows (based on recorded pumping from the perimeter dewatering bores and from the underground mine sumps – corrected for recirculation of drilling water) are approximately 90L/s, as follows:

- Approximately 60L/s – from the surface bores.
- Approximately 30L/s – from underground sumps.

As part of this review, we undertook some check modelling, using a simple analytical mine inflow model. This model is a lumped parameter 2D model based on the Theim and Dupuit-Forcheimer equations for flow to large diameter wells. For this exercise, the current pit and underground was represented by a well of 270m radius, a bulk specific yield of 1% was adopted and then calibrated against current total mine inflows to derive bulk aquifer permeability. The derived bulk aquifer permeability was around 0.01m/d. This was then used to predict future inflows to the underground mine, with the full underground mine workings and pit being represented by a well of smaller radius (average of pit and underground of around 200m) and progressively greater depth. The model predicts inflows of around 80 to 90L/s over the next 20 years.

The groundwater inflow risk is not considered a major risk as the magnitude in inflows is largely known and/or can be relatively easily predicted, and underground pumping capacity can be sized accordingly (with appropriate contingency). However, specific inflows zones (fractures) may have some impact on the stability of the backs (roof of mine workings) due to physical erosion or may cause some corrosion of meshing and bolting.

In terms of unknown risks, there is the potential for "burst inflows" if a new underground heading intersects a "pocket" of water under high head/pressure (e.g. hydraulically isolated fault zone). The risk relates to the head of water and the size of the initial breakthrough (i.e. jumbo hole or full drive), which will determine the force of the burst, and to the volume of water in isolated storage, which will determine how long the flow will continue. However, given the nature of the basement rocks around the mine workings and the potential for deformation above the mine workings, the risk of intersecting such a hydraulically isolated pocket of water is considered to be very low.

11.3.14.3 Risk Management Strategies

There are some simple management strategies that can reduce the risk and impact of inflows from the basement aquifer and/or isolated fracture zones. These include:

- Cover drilling ahead of any new development into "unknown" areas.
- Target any known major fracture zones with drain-holes.
- Any water flows from such holes is likely to be "clean" but brackish.

The most important design features, in terms of ensuring controlled flows, is the installation of appropriately grouted pre-collars.

However, as the risk of uncontrolled flow from structures is not considered to be high, and alternative strategy would be to wait until the structures are intersected and then decide on whether to install drainholes based on the magnitude (and nuisance value) of any inflows.

11.3.15 Seasonal inflow from shallow aquifers

11.3.15.1 Background

It is reported that there are some locations where mine inflows either increase, or only flow, during the wet season. One such area is the incline from DSP12 to the pit portal. During, my recent site visit, there was some inflow through pegmatites, FW foliations and some bolt holes over a <5m interval. It is reported that, during the west season, this inflow zones extends all the way to the portal (over 50m). The source of this water is most likely to be ponded rainfall on berms above the incline. Similar inflows could be expected in other shallow development or access workings. We also suspect that seasonal inflows could be experienced in shallow mine workings in the southern part of the mine where there may be some enhanced hydraulic connection to the surface (or near surface) beneath the Agucha Pond overflow channel via old underground mine workings.

11.3.15.2 Inflow Risk

The reported inflows to the exiting incline (and potential future inflows to other drives) are not expected to be major and have any significant impact on total mine inflows. Similarly, to general inflows from the basement rocks, the main risks relate to the potential impact on the stability of the backs (roof of mine workings) due to physical erosion or may cause some corrosion of meshing and bolting.

11.3.15.3 Risk Management Strategies

There are some simple management strategies that can reduce the risk and impact of seasonal inflows through fracture zones (or old underground workings). These include:

- Target any known major fracture zones with drain-holes. Note that any water flows from such holes are likely to be “clean” but brackish.
- Minimise the potential for vertical leakage from the Agucha Pond overflow channel, by either/or:
    - Re-surfacing the channel base so that there is a constant fall from upstream to downstream and no undulations that can “trap” water into ponds.
    - Line the channel base (concrete, clay etc) to minimise infiltration.

11.3.16 Inflow to main shaft and link access

11.3.16.1 Background

During development of the main shaft, a major fault zone was intersected at 590 to 605m depth (-200mRL to –215mRL). This was not unexpected, as a cored investigation hole was drilled own the centre-line of the shaft prior to shaft sinking. However, significant inflows were encountered when the shaft intersected the fault. Total inflows were estimated (visually) at around 20L/s.

During grouting operations to seal the shaft, individual cover holes flowed at up to 7L/s. More importantly, it was found that the hydraulic pressure that needs to be overcome to allow for grout pumping was around 5MPa (which is equivalent to 500m water head). This indicates that the potentiometric surface (essentially the groundwater level providing the inflow driving head) was around 90m below ground surface. This is somewhat below the pre-mining water table, but similar to the lowest recorded water levels in some pit perimeter bores.

Once the fault zone was grouted off, residual inflows behind the shaft liner (as indicated by shaft sump pumping requirements) were around 2 to 3 L/s in early 2016. These have since declined to around 1L/s. This decline in residual inflows is as a result of dewatering/depressurization of the fault zone. The salinity of the initial inflow was measured at around 45,000mg/L TDS. This has remained relatively constant since. This indicates that the inflowing groundwater is “old” and most likely to be associated with an isolated pocket of groundwater with little to no hydraulic connection to the shallow water or the broader basement aquifer. This suggests that, once the fault zone is dewatered/depressurised, it is unlikely to be actively recharged and any residual inflows should remain small.

A second cored investigation hole was drilled to confirm the orientation of the fault zone and the location where it may be intersected again by the link access drive (at the -470mRL) from the shaft base to the main mine workings. It should be noted that there remains some uncertainty as to where the main fault zone intersects the link drive. Two fault zones were intersected in the second cored drill-hole, the main interpreted fault some 250m from the shaft and a secondary (possibly splay) fault located 170m from the shaft. The current access development plan includes the drilling of two inclined (around 45° up) cover holes to confirm the location of the fault before link drive development gets close to the fault and to allow for some dewatering/depressurisation. It is also planned to construct a liner around the link drive to keep water out and strengthen the floor/walls/backs.

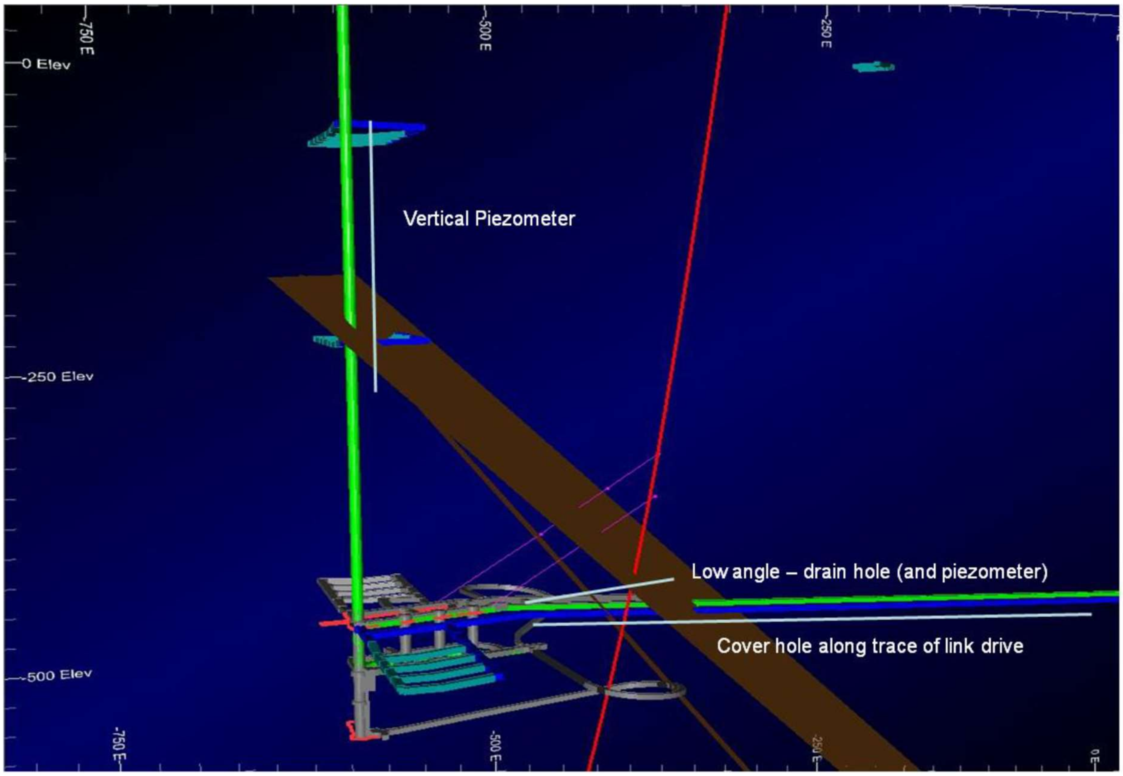


Figure 29: Main Shaft and Schematic of Monitoring and Depressurisation Strategy

11.3.16.2 Inflow Risk

The main risks associated with the fault zone (as had already been identified by the shaft team (HZL and contractor) relate to the potential for very high hydraulic heads where the link drive intersects the fault zone, with consequent high burst inflows and impact on the strength/stability of the drive. A preliminary assessment of potential burst inflows and pressures has been made as part of this review, using a simple analytical model for flows to a tunnel (commonly known as the Goodman Equation).

If the fault zone has only been marginally dewatered by inflows to the shaft sump (say, the potentiometric surface is round 150m below ground surface), then the burst pressure at the link drive (at 860m depth or - 470mRL) would be around 8MPa. This could present a significant safety risk to jumbo drilling and drive development. For this case (and assuming a fault permeability of 1m/d and a fault intersection of 5m length), the burst inflow rate could be as high as 50L/s, although this would decline to less than 10L/s after several days.

In this model, inflow is linearly proportional to head, permeability and fault width. That is, if the head is half that assumed, then inflows will be half that predicted. Equally, if fault permeability is twice that adopted, then inflow will be twice that predicted.

There is also the potential risk that there is another (unknown) fault zone (with similar properties) between the currently identified fault and the mine workings.

11.4 Mine Design Parameters

Table 29: Mine Design Parameters

SN	Particulars	Description
1	Number of working stopes for extraction	20-35 stopes per month
2	Size of the panel (width of stope)	15-30m (along the strike length of ore body)
3	Level interval	18-25m (Vertical height-floor to floor)
4	Thickness of crown pillar	60m (Barrier / separation between Open pit and underground working)
5	Thickness of Sill pillar	Proposed Mining Method does not require sill pillar, thus not planned. However, sill pillar of 25-75m may be considered depending upon strata condition and requirement of stoping sequence.
6	Thickness of Rib pillar	Proposed Mining Method does not require rib pillar, thus not planned. However, rib pillar may be considered depending upon strata condition and requirement of stoping sequence
7	Size and interval of Stope pillar	Proposed Mining Method does not require stope pillar, thus not planned. However, stope pillar may be considered depending upon the strata condition and requirement of stoping sequence.
8	Size/shape of man way	2-3m Width x 2-3m breadth x 25m vertical height

9	Size/shape of ore pass and interlevel raises	(2-3.5) m Width x (2-3.5) m breadth x (25-100) m vertical height. However, circular raises of 2-3.5m diameter are also being conceptualized.
10	Method of stowing/back filling	Paste filling (PF), cemented rock fill (CRF) or rock fill (RF) is proposed for the backfilling of the stoping area depending upon requirement on type of backfill and actual ground condition.
11	Method of drainage of stowed water	The above filling methodology incorporates very less water only in case of PF, and no water in CRF or RF. Water from each level shall be drained through series of boreholes/raises and collected at Main sumps. As the mine goes deeper, additional sump shall be constructed and multi stage pumping proposed. All mine water so pumped shall be collected into a tank/sump at surface and recycled.
12	Material (Ore / Waste) from face/stope to loading point	By LHD
13	Material (Ore / Waste) from loading point to surface	Transportation by LPDT / mine trucks directly to surface yard or dumping at underground crusher level and hoisting through shaft system
14	Material (Ore / Waste) to end use plant	Transportation by trucks from surface yard to crusher at processing plant or direct feed to processing plant system through shaft hoisting arrangement.
15	Safety features provided on conveyor/ haulage track/ roadway	Traffic light signals through Leaky feeder system and Voice communication system through hand held devices.

11.5 Mine Schedule

Figure 30: RAM Mine Design and LOM Plan schematics

Table 30: RAM - 12 Year Mine Schedule Summary [supplied by HZL]

Tonnes	Area / Block	Totals	Year1	Year2	Year3	Year4	Year5	Year6	Year7	Year8	Year9	Year10	Year11	Year12
Totals	North Block	15,106,562	2,747,274	2,801,667	2,314,257	1,757,357	2,170,710	1,330,554	1,182,000	704,004	98,740	-	-	-
	South Block	25,452,410	2,056,957	2,085,306	2,419,134	2,448,217	1,460,170	2,155,184	2,099,300	1,503,898	2,250,344	2,491,019	2,420,410	2,062,471
	North Extreme/Block 1	515,141	131,102	-	-	-	-	-	-	120,004	264,035	-	-	-
	LOM Pillar	5,970,281	-	-	-	-	-	-	-	-	-	-	-	-
	Total	47,044,394	4,935,332	4,886,973	4,733,391	4,205,573	3,630,880	3,485,738	3,281,300	2,327,906	2,613,119	2,491,019	2,420,410	2,062,471
Grade		Average	Year1	Year2	Year3	Year4	Year5	Year6	Year7	Year8	Year9	Year10	Year11	Year12
Totals	Zn%	11.82	12.19	11.83	12.95	12.14	12.54	11.39	11.96	10.94	11.61	10.10	10.03	10.01
	Pb%	1.34	1.77	1.66	1.83	1.46	1.22	0.99	1.15	1.29	1.25	1.03	0.90	0.82
	TMC%	13.16	13.96	13.49	14.77	13.60	13.77	12.38	13.11	12.24	12.86	11.13	10.93	10.83
	Ag(ppm)	45	62	63	61	47	41	31	32	37	36	34	28	29

Table 31: RAM - 12 Year Mine Schedule detailed [supplied by HZL]

Tonnes	Area / Block	Totals	Year1	Year2	Year3	Year4	Year5	Year6	Year7	Year8	Year9	Year10	Year11	Year12
Stoping	North Block	15,106,562	2,747,274	2,801,667	2,314,257	1,757,357	2,170,710	1,330,554	1,182,000	704,004	98,740	-	-	-
	Zn%	12.32	14.58	13.07	13.47	11.90	11.38	10.28	10.53	8.08	8.43	0.00	0.00	12.61
	Pb%	1.77	2.46	2.27	2.09	1.43	1.25	0.93	1.09	1.19	1.34	0.00	0.00	1.89
	TMC%	14.09	17.05	15.34	15.56	13.33	12.63	11.21	11.61	9.27	9.77	0.00	0.00	14.50
	Ag(ppm)	65	89	89	82	54	44	31	33	32	33	0	0	62
	South Block	25,452,410	2,056,957	2,085,306	2,419,134	2,448,217	1,460,170	2,155,184	2,099,300	1,503,898	2,250,344	2,491,019	2,420,410	2,062,471
	Zn%	11.38	9.44	10.16	12.44	12.32	14.27	12.08	12.77	12.44	11.73	10.10	10.03	10.01
	Pb%	1.12	0.90	0.85	1.57	1.49	1.19	1.02	1.18	1.33	1.14	1.03	0.90	0.82
	TMC%	12.50	10.34	11.01	14.01	13.80	15.46	13.10	13.95	13.76	12.88	11.13	10.93	10.83
	Ag(ppm)	33	29	27	41	42	36	31	32	39	32	34	28	29
	North Extreme/Block 1	515,141	131,102	-	-	-	-	-	-	120,004	264,035	-	-	-
	Zn%	9.43	5.09	0.00	0.00	0.00	0.00	0.00	0.00	9.04	11.76	0.00	0.00	0.00
	Pb%	1.64	0.89	0.00	0.00	0.00	0.00	0.00	0.00	1.46	2.09	0.00	0.00	0.00
	TMC%	11.07	5.99	0.00	0.00	0.00	0.00	0.00	0.00	10.50	13.86	0.00	0.00	0.00
	Ag(ppm)	57	23	0	0	0	0	0	0	51	76	0	0	0
	LOM Pillar	5,970,281												
	Zn%	12.64												
	Pb%	1.14												
	TMC%	13.77												
	Ag(ppm)	40												

Table 32: RAM - 12 Year Mine Schedule detailed backfill [supplied by HZL]

cu.m	Area / Block	Totals	Year1	Year2	Year3	Year4	Year5	Year6	Year7	Year8	Year9	Year10	Year11	Year12
Paste filling / Backfill	North Block	4,924,276	876,770	902,064	751,911	580,068	720,889	440,501	389,965	229,350	32,758	-	-	-
	South Block	8,357,822	686,031	690,939	797,342	804,008	477,753	709,420	690,709	493,291	738,594	811,985	791,841	665,909
	North Extreme/Block 1	178,248	46,906	-	-	-	-	-	-	41,497	89,845	-	-	-
	LOM Pillar	1,899,773												
	Total	15,360,119	1,609,707	1,593,003	1,549,253	1,384,076	1,198,642	1,149,921	1,080,674	764,138	861,197	811,985	791,841	665,909

Recovery	Average	Year1	Year2	Year3	Year4	Year5	Year6	Year7	Year8	Year9	Year10	Year11	Year12
Recovery_Zn	89.97	89.97	89.97	89.97	89.97	89.97	89.97	89.97	89.97	89.97	89.97	89.97	89.97
Recovery_Pb	61.74	61.74	61.74	61.74	61.74	61.74	61.74	61.74	61.74	61.74	61.74	61.74	61.74

	Totals	Year1	Year2	Year3	Year4	Year5	Year6	Year7	Year8	Year9	Year10	Year11	Year12
Zn_MIC	4,324,692	541,154	520,057	551,293	459,425	409,807	357,172	353,082	229,205	273,008	226,391	218,367	185,732
Pb_MIC	345,984	53,984	50,181	53,344	37,964	27,419	21,279	23,249	18,587	20,122	15,890	13,510	10,455
Total MIC	4,670,676	595,138	570,238	604,637	497,389	437,226	378,451	376,330	247,792	293,131	242,282	231,877	196,186

11.6 Mining Fleet Requirements

11.6.1 Loading & Hauling Equipment

S. No.	Type	Function	Make	Capacity	Unit	Motive Power	Number of Equipment
1	Loading	Loading	Sandvik	17	Tons	369 HP	13
2	Loading	Loading	Sandvik	21	Tons	472 HP	8
3	Loading	Loading	CAT	17	Tons	409 HP	6
4	Loading	Loading	CAT	20	Tons	409 HP	6
5	Hauling	Hauling	Sandvik	30	Tons	394 HP	2
6	Hauling	Hauling	Epiroc	60	Tons	567 KW	2
7	Hauling	Hauling	Epiroc	30	Tons	298 KW	4
8	Hauling	Hauling	Epiroc	65	Tons	567 KW	10
9	Hauling	Hauling	CAT	60	Tons	579 KW	28

11.6.2 Drilling Equipment

S.N.	Type	Make	Diameter of Hole (mm)
1	Development Drill	Jumbo Drill Sandvik DD321	45mm
2	Development Drill	Jumbo Drill Sandvik DD421	45mm
3	Development Drill	Jumbo Drill Sandvik DD422i	48mm
4	Development Drill	Atlas Copco Boomer - B282	45mm
5	Development Drill	Epiroc Boomer - M2C	48mm
6	Production Drill	Production Drill Sandvik DL421-15C	89mm-204mm

11.6.3 Equipment Productivity and Usage

High level data from the R&R presentations is illustrated below.

Table 33: Major Equipment Productivity Summary

S.No	Particulars	UOM	Plan FY 17-18	Actual FY 17-18	Plan FY 18-19	Actual FY 18-19	Plan FY 19-20	Actual FY 19-20	Plan FY 20-21	Actual FY 20-21	Plan FY 21-22	Actual FY 21-22	Plan* FY 22-23
	LHD (17 / 21 MT)												
	Availability	%	80%	73%	80%	74%	80%	72%	75%	73%	75%	72%	75%
1	Utilization	%	65%	60%	75%	61%	65%	57%	70%	53%	61%	55%	61%
	Productivity	TPH	81	68	70	76	86	74	80	80	80	69	80
	OEE	%	52%	37%	60%	49%	52%	35%	53%	38%	46%	35%	46%
	LPDT (60 / 65 MT)												
	Availability	%	82%	72%	85%	71%	82%	73%	75%	77%	78%	76%	78%
2	Utilization	%	80%	78%	90%	79%	80%	75%	80%	73%	75%	73%	74%
	Productivity	TKPH	158	154	188	139	170	154	173	151	163	150	156
	OEE	%	66%	55%	77%	41%	66%	49%	60%	49%	59%	51%	58%
	Production Drilling												
	Availability	%	80%	74%	80%	75%	80%	75%	78%	75%	78%	76%	78%
3	Utilization	%	25%	20%	20%	16%	20%	17%	20%	15%	19%	12%	20%
	Productivity	Mtr/Hr	58	58	60	55	58	62	60	64	65	69	68
	OEE	%	20%	15%	16%	11%	16%	14%	16%	12%	15%	10%	15%
	Jumbo												
	Availability	%	80%	77%	80%	72%	80%	70%	75%	74%	75%	72%	77%
4	Utilization	%	45%	19%	41%	24%	34%	21%	30%	17%	26%	17%	26%
	Productivity	Mtr/Hr	89	85	90	74	90	87	90	81	90	84	90
	OEE	%	36%	14%	33%	14%	27%	15%	23%	11%	20%	12%	20%
	* Draft BP, may change after finalisation												

11.6.4 Mine Personnel Requirements

No data was received and reviewed, however considering the other operations and the overall HZL group of mines, there does not seem that there will be any major issues with labour and skills to operate the equipment.

11.7 Ventilation

RAM has undergone numerous extensive ventilation studies into the overall system and requirements to ensure there is the required ventilation and cooling power to ensure a safe operational environment underground. The recommendation for the studies has been implemented successfully over the last years and is running effectively. Below are extract from previous studies to determine the ventilation requirements at the mine.

11.7.1 Ventilation System

The primary ventilation circuit has been designed as a conventional negative pressure system. Primary fans will be located on surface and will be connected to a series of raises and drives, allowing exhaust air to be drawn from the mine at multiple locations.

Negative pressure created by the exhaust fans will cause fresh air to flow into the mine via the hoisting shaft, main decline and a system of dedicated intake shafts and lateral development.

The ventilation circuit consist of two main surface exhaust shafts of 7.5 m diameter. The two surface exhaust shafts connect to four internal (underground) exhaust shafts systems. Each of the four internal exhaust shaft systems is comprised of three 3.5 m diameter raises. Two of the internal shaft systems will ventilate the northern half of the mine and two will ventilate the southern half of the mine. A supplementary surface exhaust shaft of 3.5 m diameter is required to meet peak airflow demands. All surface exhaust shafts will be equipped with a fan installation. This will create two semi-discreet ventilation districts (north and south), with some limited interaction between them. Exhaust airflow will be distributed evenly between the four internal shaft systems.

The surface exhaust shafts will connect to the internal exhaust systems via a north and south lateral development connection from the bottom of each surface shaft. The cross sectional area of the connecting drives will need to be sufficient as to not exceed maximum allowable velocity limits. AMC have sized the connecting drives at 6.0 m wide by 7.5 m high in order to safely develop and support these drives. AMC suggests these drives to be initially developed at 6.0 m wide and 5.5 m high, then following installation of the ground support, undertake 2 m of floor stripping.

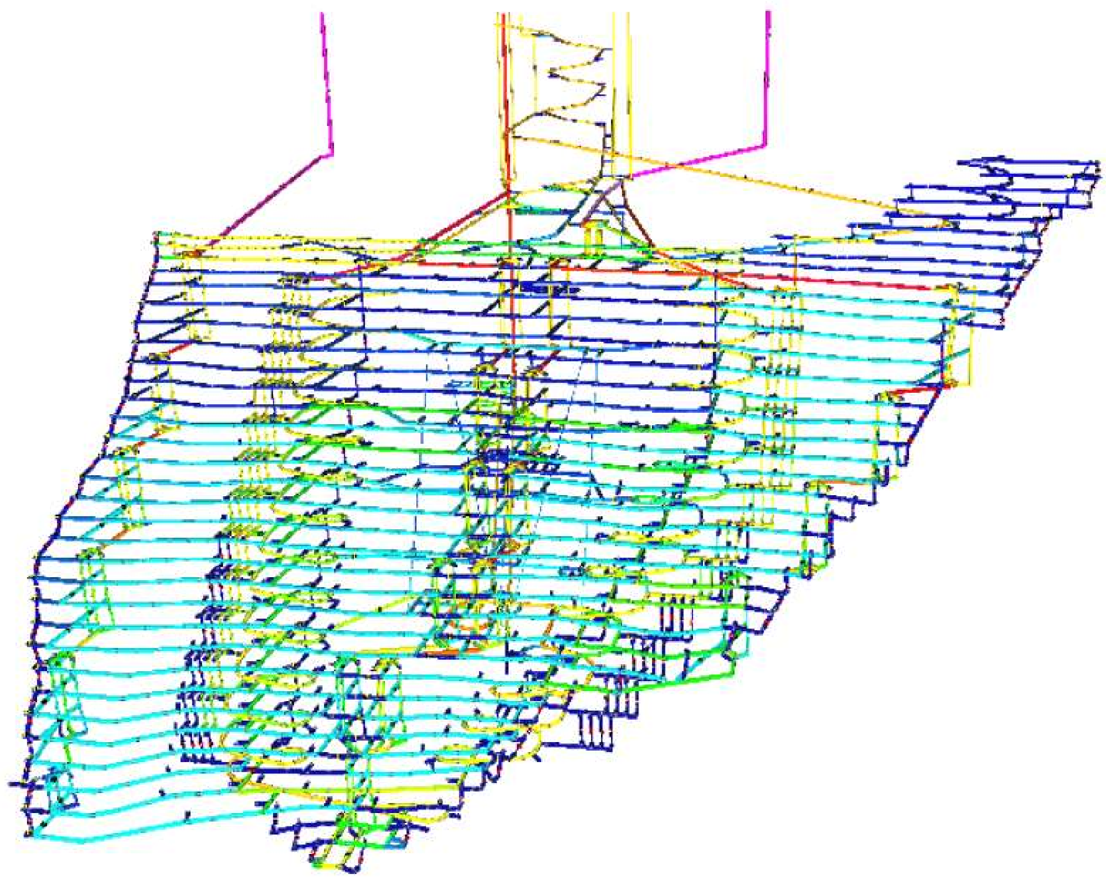


Figure 31: VentSim Visual Model of Rampura Agucha Mine (2013)

The north and south ventilation districts will be subdivided into smaller districts corresponding to the active production panels and development areas. The internal exhaust airways will be located at the ends and centre of the FWD. Additional exhaust is developed with the decline. It is planned to turn this exhaust into a potential intake airway.

The crusher, workshop and tipping level infrastructure are connected to the exhaust. This is regulated as required. All other major infrastructure (for example magazine and fuel bay) is connected to exhaust to ensure fresh air to these locations.

The internal fresh airways will be the declines and intake shaft connected to the declines. Regular connections are established between fresh airways and the decline to allow fresh air into the decline as required. All connections, between fresh airways and the decline are regulated.

11.7.2 Development Ventilation

Numerous auxiliary ventilation systems are implemented where diesel equipment is used and required air velocities in work places.

Peak diesel exhaust emission (DEE) dilution demand occurs during loading and hauling operations from developing faces. Sufficient airflow must consequently be supplied to satisfy the requirements of one LHD (17 t) and one truck (60 t), equating to approximately 45 m3/s. Mining industry best standard for DEE dilution factor as of 0.05 m3/s/kW of installed diesel power has been applied to all diesel-powered equipment operating underground, with the exception of drill jumbos and other equipment operated intermittently.

Fans should be located in the fresh air rise (FAR) as close to the working heading as possible. This will reduce the distance of ventilation ducting required and reduce pressure on the fan, which will improve efficiency. A stopping will be required to segregate the FAR from the decline to ensure short-circuiting does not occur.

The current design has one FAR system following each decline, additional shafts/rises will be required to maintain ventilation standards and increase efficiency. The development auxiliary ventilation needs 45 m3/s. A suitable auxiliary ventilation setup would consist of two, 110 kW twin stage, co-rotating, 1400 mm diameter axial fan with 1400 mm low leakage, low resistance, sealed, flexible ducting.

11.7.3 Development Ventilation

The mining method selected will result in a large number of small stopes and associated development that will be active simultaneously Each production level will be split up into four separate ventilation circuits. This will allow greater control of the ventilation in production areas.

Each production level will have two intakes and four returns, that is two returns in the centre of the FWD and one return at either end of the FWD. The intakes are the access drives, and these are located between the end of the FWD and the central return.

The FWD is ventilated with air drawn in from the access drive along the FWD to the exhaust shafts at the end and middle of the FWD. The ore drives will use secondary ventilation with fans located in the access drive or the decline forcing air into the ore drive. The air from the ore drive will then return to the FWD and to the closest exhaust.

The current design has flood ventilation along the FWD. This will require regulators on all the exhaust locations along the FWD. The ore drives will need to be ventilated by ducted, auxiliary ventilation systems that can be turn on and off. The simplest setup of this would be to install 55 kW single stage, co-rotating, 1,200 mm diameter axial fan with 1,200 mm low leakage, low resistance, sealed, flexible ducting. The ducting will have multiplied t-pieces from the main duct. This can then be regulated as required.

Table 34: Peak Ventilation Demands

Location	Number	Airflow Requirement (m3/s)	Total Airflow Requirements (m3/s)
Access Level	1	50	50
Back Fill	1	15	15
Decline	1	50	50
Definition drilling	1	15	15
Fill Prep	5	30	150
FWD	3	50	150
LDRL	1	15	15
ODRL	2	15	30
ODRT	10	15	150
Production drilling	4	15	60
RAW	1	50	50
Slot raise	10	30	300
SPL	1	50	50
Stoping	7	15	105
Stope blast	1	30	30
WDR	16	15	240
Inactive levels	5	20	100
Mined out level	9	5	45
Infrastructure		-	300
Sub Total			1905
Leakage & Balancing Inefficiencies		5%	95
TOTAL (rounded)			**2000**

11.8 Mine Map

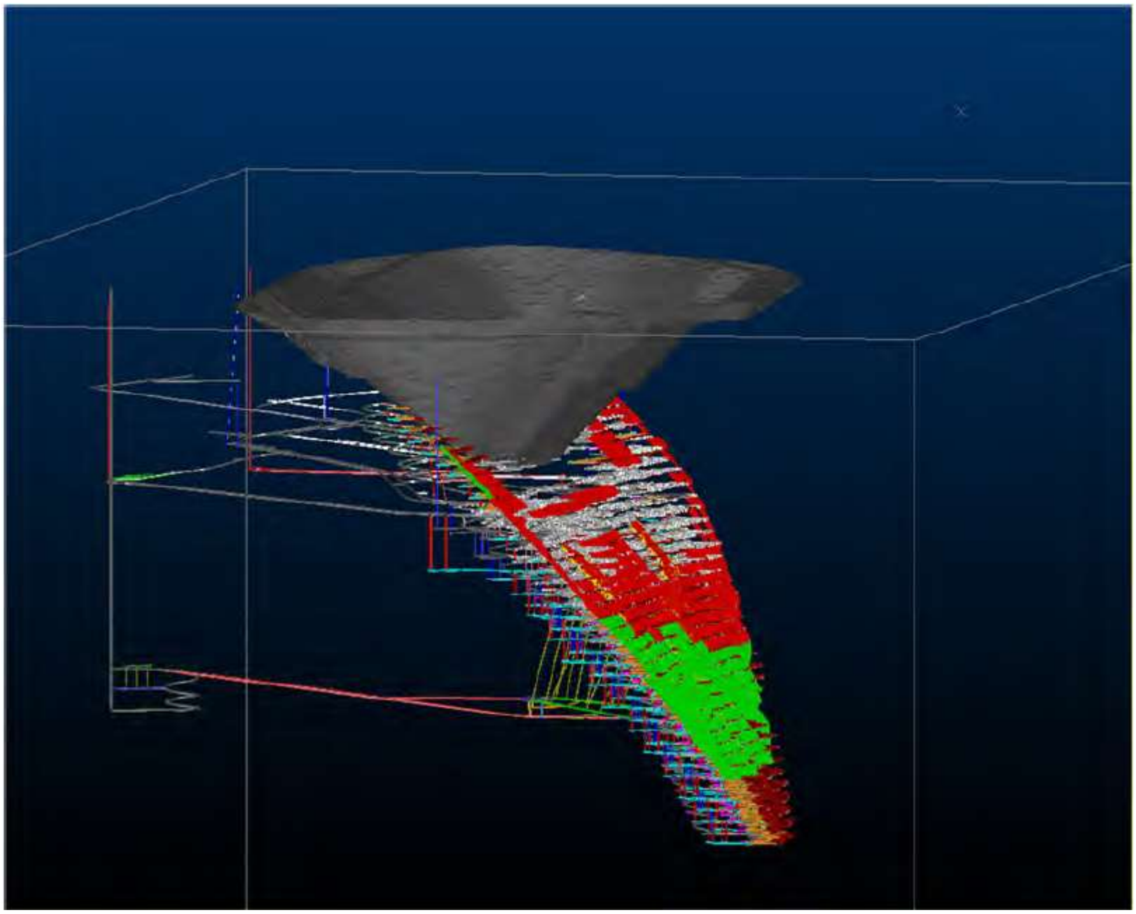


Figure 32: Overall LOM Design for RAM

12 Processing and Recovery Methods

12.1 Introduction

The RAM concentrator processes two mine operations ROM namely the Rampura Agucha and Kayad Mine. The ore received from Kayad Mine is separately treated in one of the streams. All the vital parameters are monitored through instream analysers and chemical composition of the feed, concentrates and tails are continuously sampled and analysed at site laboratory well equipped with state-of-the-art instruments. The table shown below typically depicts the assays of ore treated from Rampura-Agucha underground (RAUG) mine and Kayad mine for a day. However, it may vary depending upon the mineral composition of the respective mine

Table 35: Summary of the RAM Plant

Processing method	Mineral Beneficiation (Grinding-Froth Flotation)
Products	Pb & Zn concentrate
Nominal production capacity	6.5 MTPA
Date plant built:	
1991	Stream 1
2004	Stream 2
2008	Stream 3
2010	Stream 4
Significant Modifications post design	
High level results for last 3 years	
head feed tonnage	Increased
recoveries	Improved
concentrate grades	Improved
metals/ concentrates produced	Increased
Sampling methods	
Weightometers	Mill feed conveyors
Grade Sample Points	Automated
Frequency of samples & composition of assays	Day composites samples
Lab assay method	AAS (For feed & tailing)
	Classical titration (for concentrates)
Tailing deposition method	Wet Dam

12.2 Process Flow maps

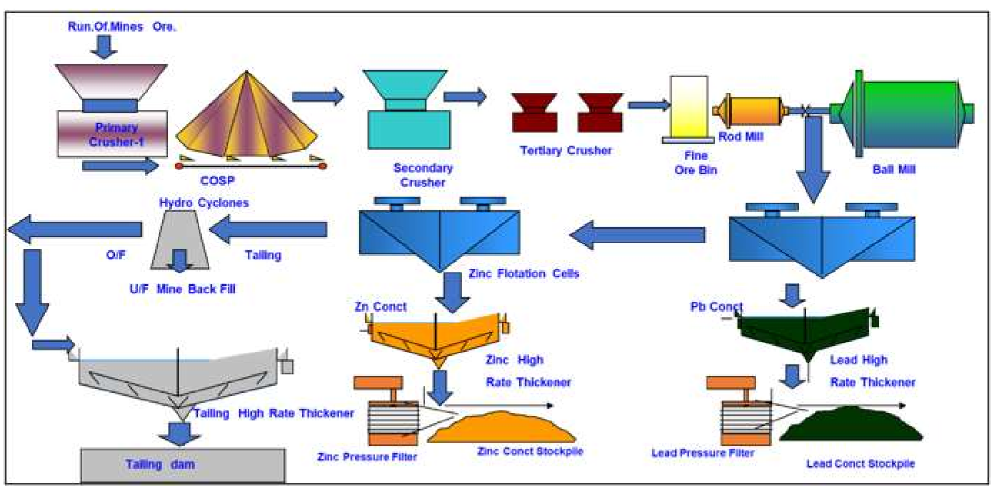


Figure 33: Process overview for mineral processing- Rod Mill/Ball Mill circuit

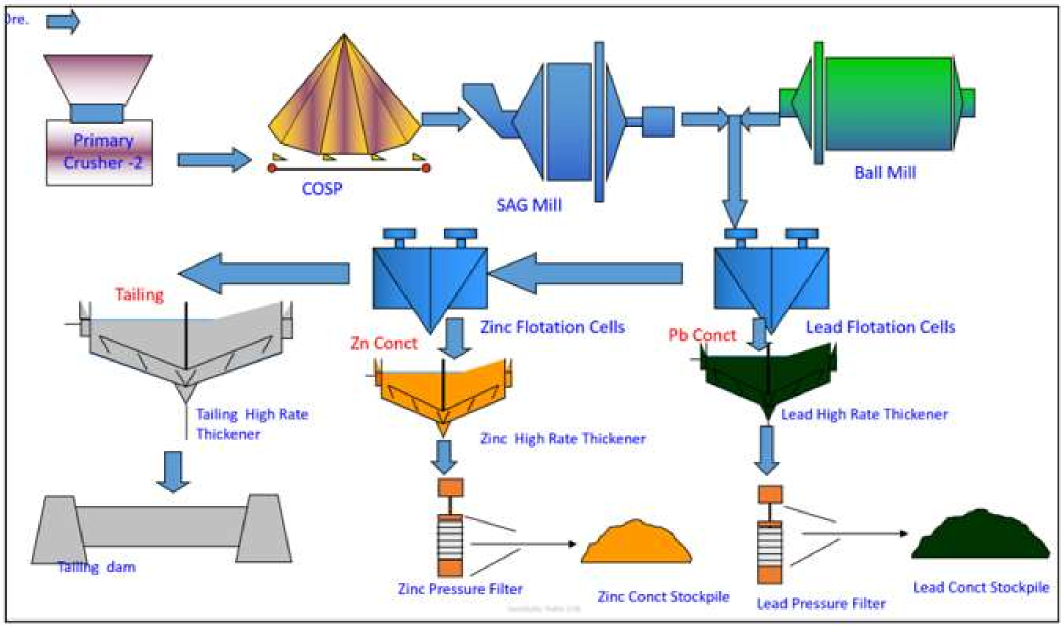


Figure 34: Process overview for mineral processing- SAG Mill/Ball Mill circuit

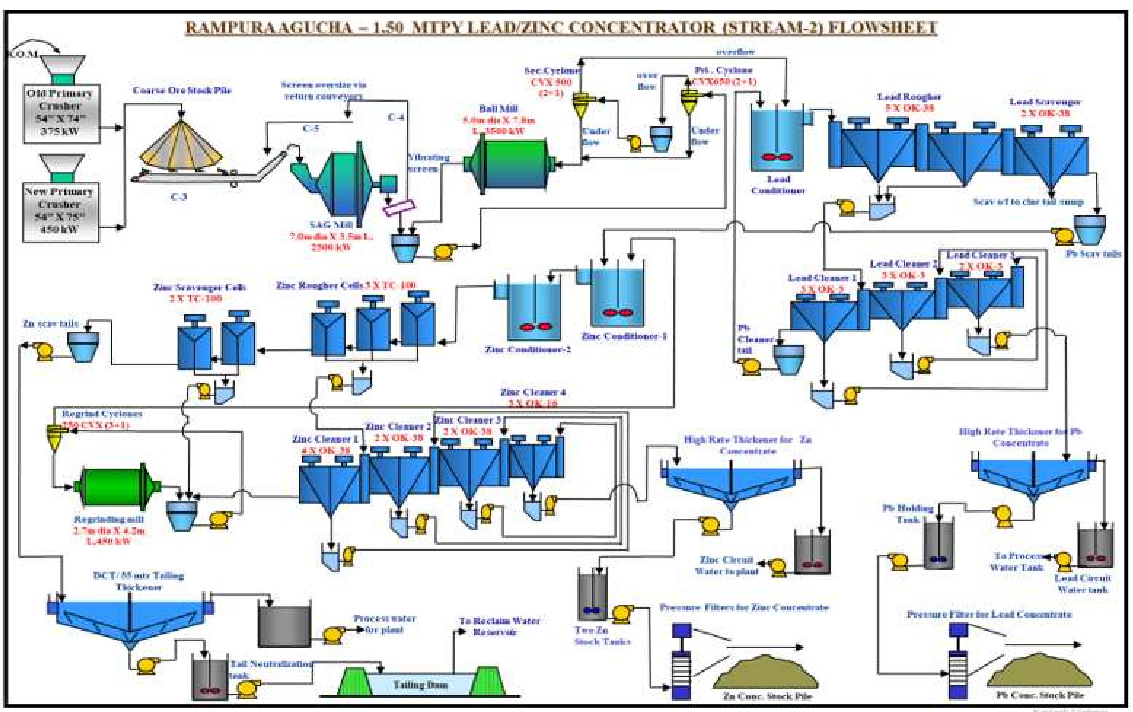


Figure 35: Schematic - RAM 1.5 Mtpa Concentrator - Stream 2 - Flowsheet

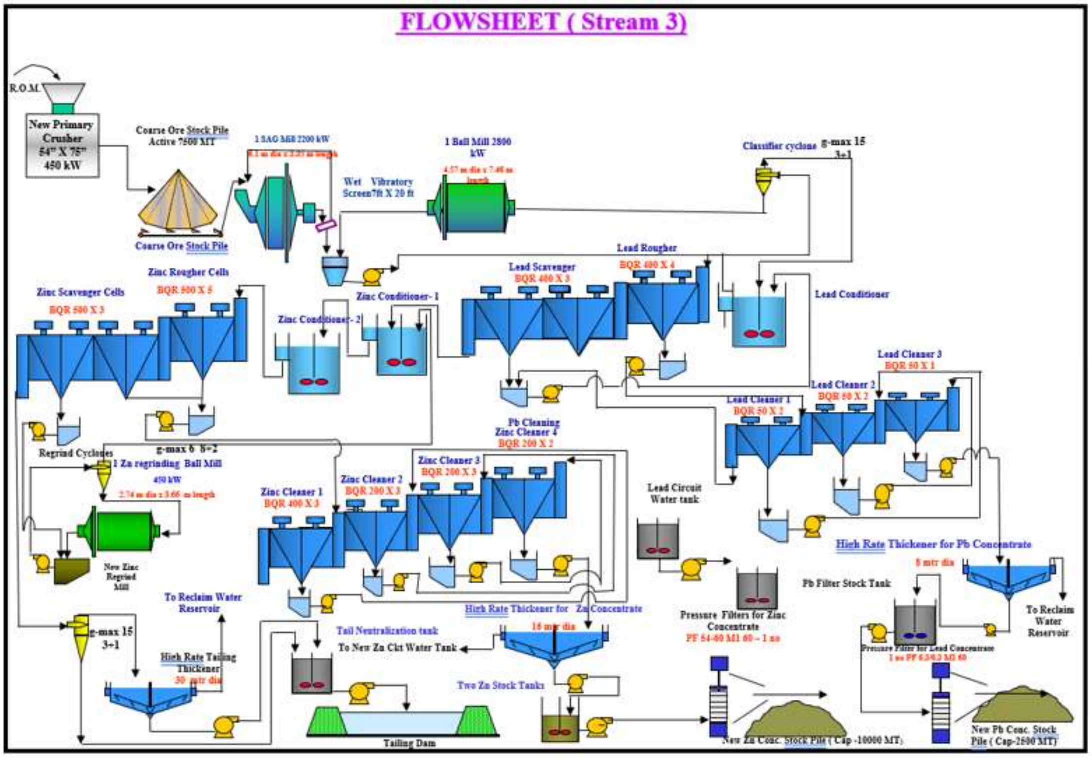


Figure 36:Schematic - RAM 1.5 Mtpa Concentrator - Stream 3 - Flowsheet

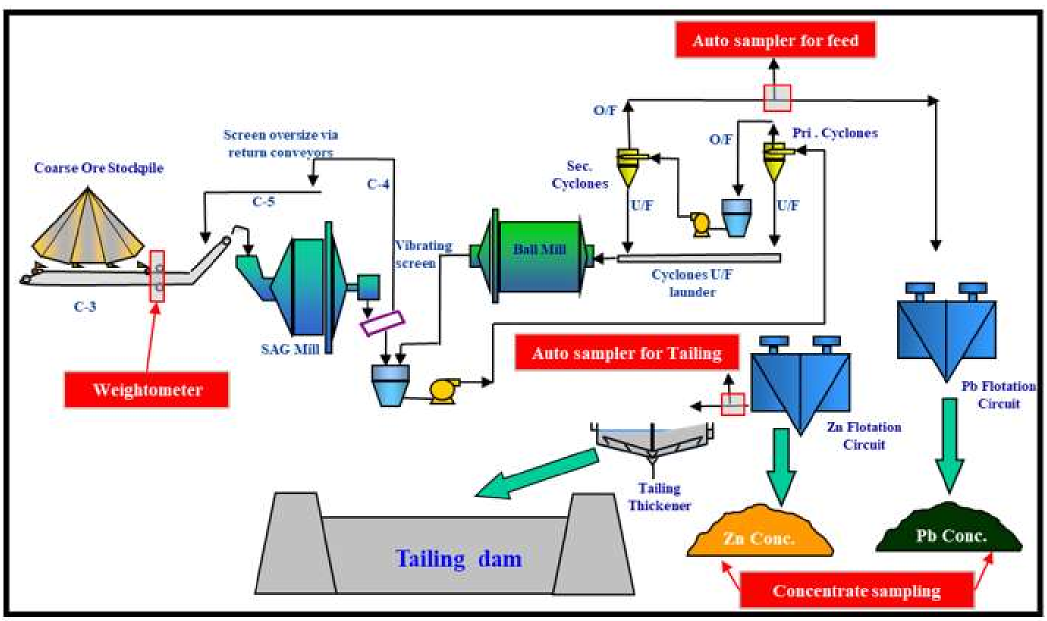


Figure 37: Simplified flowsheet - RAM Concentrator

12.3 Crushing, Screening, Ore flow

12.3.1 Crushing Section:

Beneficiation plant comprises of crushing, grinding & classification, differential floatation, concentrate dewatering & tailing disposal. The total installed capacity of beneficiation plant is 6.50 million tons per annum (mtpa).

12.3.2 Crushing

The runoff mine ore is crushed in 2 no's independent primary gyratory crusher on surface and 2 nos. jaw crusher in UG mines from 1000 mm to 90% minus 150 mm and is transferred to coarse ore stockpile. There are four coarse ore stockpiles. Ore from old coarse ore stockpile is crushed in standard cone crusher (secondary crushing) to 100% minus 50mm. Further the ore is crushed in short head cone crushers (tertiary crushing) to minus 19 mm. The fine product (minus 19 mm) is stored in fine ore bins. Ore from 3 new coarse ore stockpiles is directly fed to SAG Mills in stream-2, stream-3 and stream-4. Stream -4 is the new Stream and was commissioned on 16th Feb 2010 which has the same capacity as Stream - 2 i.e. 4650TPD. Stream -4 has the same SAG Mill, Ball Mill combination for grinding.

Table 36: Summary of CPP Primary Crushers

S. No.	Type of Crusher	Make	Capacity of Crusher (tph)	Feed Size (mm)	Product Size (mm)
1	Gyratory Crusher	Metso	700	1000	-150
2	Jaw crusher	Sandvik	700	1000	-150

Table 37: Summary of CPP Secondary Crushers

S. No.	Type of Crusher	Make	Capacity of Crusher (tph)	Feed Size (mm)	Product Size (mm)
1	Cone Crusher	Metso	350	-150	-50

Table 38: Summary of CPP Tertiary Crushers

S. No.	Type of Crusher	Make	Capacity of Crusher (tph)	Feed Size (mm)	Product Size (mm)
1	Cone Crusher	Metso	250	-50	-19

12.3.3 Dry Grinding

Dry Grinding is not used at Processing plant, thus not Applicable.

12.3.4 Wet Grinding

Grinding & Classification:

There are four streams. In Stream-1 closed circuit wet grinding is arranged in three parallel circuits of rod and ball mill combination of 2250 TPD each for flexible in operation and maintenance. The mill discharge goes to hydro cyclones for 2-stage classification. The pulp of 80% minus 63 microns goes to the lead conditioner. In Stream 2 and Stream-3 SAG Mill replaces the entire fine crushing circuit and rod mill. Stream 2 and 4 has a capacity of 4650 TPD each and Stream 3 has a capacity of 3250 TPD. In Stream- 2, Stream -3 and Stream- 4, the SAG Mill and Ball Mill discharge goes to hydro cyclone for 2-stage classification. Final product of classification has fines of 80% passing 63 microns.

Table 39: Grinding and Crushing Parameters - Stream 1

S. No.	Type of Mill	Stages	Nos.	Make of the mill	Feed Flow Rate (tph)	Feed Size (mm)	Product Size (mm)	Type of screen /Classifier	Aperture Size of Screen /Classifier (mm), if applicable	Classifier/ Screen undersize (tph)	Classifier / Screen oversize (tph)	Water Requirement (l/h)	Fresh Water Requirement (l/h)	Recirculated Water (l/h)
1.	Rod Mill	Single	3	MBE	94	19 mm	1.4 mm	Hydrocyclones	1st Stage- 142 mm	190	154	90 m3/hr	NA	90 m3/hr
2.	Ball Mill	Single	3	MBE	250	1.4 mm	63 microns	Hydrocyclones	2nd Stage- 75 mm	60	94	60 m3/hr	NA	60 m3/hr

Table 40: Grinding and Crushing Parameters - Stream 2

S. No.	Type of Mill	Stages	Nos.	Make of the mill	Feed Flow Rate (tph)	Feed Size (mm)	Product Size (mm)	Type of screen /Classifier	Aperture Size of Screen /Classifier (mm), if applicable	Classifier/ Screen undersize (tph)	Classifier / Screen oversize (tph)	Water Requirement (l/h)	Fresh Water Requirement (l/h)	Recirculated Water (l/h)
1.	SAG Mill	Single	1	Outotec	195	150 mm	1 mm	Hydro cyclones	1st Stage- 140 mm	460	335	180 m3/hr	NA	180 m3/hr
2.	Ball Mill	Single	1	Outotec	600	1 mm	63 microns	Hydro cyclones	2nd Stage- 110 mm	140	195	120 m3/hr	NA	120 m3/hr

Table 41: Grinding and Crushing Parameters - Stream 3

S. No.	Type of Mill	Stages	Nos.	Make of the mill	Feed Flow Rate (tph)	Feed Size (mm)	Product Size (mm)	Type of screen /Classifier	Aperture Size of Screen /Classifier (mm), if applicable	Classifier/ Screen undersize (tph)	Classifier / Screen oversize (tph)	Water Requirement (l/h)	Fresh Water Requirement (l/h)	Recirculated Water (l/h)
1.	SAG Mill	Single	1	Citic	145	150 mm	1 mm	Hydrocyclones	NA	NA	NA	120 m3/hr	NA	120 m3/hr
2.	Ball Mill	Single	1	Citic	340	1 mm	110 microns	Hydrocyclones	2nd Stage- 75 mm	195	145	80 m3/hr	NA	80 m3/hr

Table 42: Grinding and Crushing Parameters - Stream 4

S. No.	Type of Mill	Stages	Nos.	Make of the mill	Feed Flow Rate (tph)	Feed Size (mm)	Product Size (mm)	Type of screen /Classifier	Aperture Size of Screen /Classifier (mm), if applicable	Classifier/ Screen undersize (tph)	Classifier / Screen oversize (tph)	Water Requirement (l/h)	Fresh Water Requirement (l/h)	Recirculated Water (l/h)
1.	SAG Mill	Single	1	Citic	195	150 mm	1 mm	Hydro cyclones	1st Stage- 140 mm	467	328	180 m3/hr	NA	180 m3/hr
2.	Ball Mill	Single	1	Citic	600	1 mm	63 microns	Hydro cyclones	2nd Stage- 110 mm	133	195	120 m3/hr	NA	120 m3/hr

12.4 Flotation and Concentrate

Differential floatation of lead & zinc: Streams 1, 2, 3 & 4 have individual flotation circuits. Each lead flotation circuit comprises of lead conditioner where the flotation feed containing 1.5-1.8% lead and 12-13.5% Zinc gets conditioned with the flotation reagents. The conditioned slurry gets into the lead rougher/scavenger & cleaner floatation cells, where 59-62% of lead in the feed will be recovered with a concentrate grade of 60-62 % of lead and 5-7% of misplaced zinc in it; the produced concentrate of 1.5-1.8 (tons of lead concentrate for every 100 tons of treated ore) goes to lead thickener. The thickened underflow is taken to pressure filters for dewatering and then to concentrate stockpile and further to storage yard as well as dispatches to smelters.

The lead tailing goes to zinc conditioners for reagent conditioning and then to rougher/ scavenger & cleaner floatation cells; where 89-91% of Zinc in the feed will be recovered with a concentrate grade of 50-52 % of Zinc and 1.0-1.4% of misplaced Lead in it; Froth of scavenger & cleaner cells rejects are reground and re circulated to zinc conditioner. The produced zinc concentrates of 21.6-23.9 (tons of Zinc concentrate for every 100 tons of treated ore) is fed to thickeners & filters for dewatering. The thickened underflow is taken to pressure filters for dewatering and then to concentrate stockpile and further to storage yard as well as dispatches to smelters.

The Final tailings of 74.4-77 (tons of tailings generated for every 100 tons of ore treatment) from the Zn scavenger circuit gets pumped to the tailing thickener where further water will be recovered and gets recirculated back into the beneficiation plant. The thickened under flow from the tailings thickener will be sent to the tailings pond. In dewatering section, we have a Deep Cone Thickener which can take the tailings of any three streams. The Deep Cone Thickener is installed to produce the tailings with higher pulp density for facilitating the paste fill plant with the required higher pulp density slurry.

The ore received from Kayad Mine is separately treated in one of the streams. All the vital parameters are monitored through instream analysers and chemical composition of the feed, concentrates and tails are continuously sampled and analysed at site laboratory well equipped with state-of-the-art instruments. The table shown below typically depicts the assays of ore treated from Rampura-Agucha underground (RAUG) mine and Kayad mine for a day. However, it may vary depending upon the mineral composition of the respective mine.

12.5 Tailing, Thickening and Filtration

The tailing is stored in a specially constructed Tailing Impoundment Area. The tailing dam is lined with top impervious soil at the bottom and the inside of the walls with sand and top impervious soil on all the sides to avoid percolation of water to the underground. The material for construction of the tailing dam consists of waste materials generated from mine. The pipeline for tailing disposal is on three sides of the dam.

Thickened tailing is pumped through a tailing line and discharged into the tailing dam, after neutralization with lime.

Part of tailings generated during the life of mine shall be used in paste formation to back fill underground mined out voids and remaining shall be accommodated in the existing tailing dam. As of now the present height of tailing dam is 60 m and further to accommodate the tailings generated from processing plant during the period would be accommodated in the tailing dam. The height of tailing dam will be raised further as per requirement, maximum up to 74 m as per latest EC approval dated 28th Feb-2020 attached in annexure-3A

In tailing dam for height raising activity up to 74 m requires approx. 25 million tons of waste rock. The required waste material shall be availed preliminary from waste generated from underground mining activity and remaining quantity shall be taken from existing waste dump as per below mentioned table below.

Table 43: Source of Waste

Source of Waste	2022-23	2023-24	2024-25	2025-26	2026-27	Total
From UG Development (Tons)	2,788,468	2,193,977	2,248,840	1,888,843	1,666,548	**10,786,676**
From Waste Dump (Tons)	2,850,000	2,850,000	2,850,000	2,850,000	2,820,000	**14,220,000**
Total	**5,788,468**	**5,193,977**	**5,248,840**	**4,738,843**	**4,366,548**	**25,006,676**

A pumping station is installed in the Northwest corner of the dam to the reclaim water for re-use in the process. No water is allowed to escape outside the dam. The quality of the water in the piezometer wells around the impoundment area is regularly monitored.

Further other tailing disposal methodology like dry staking of tailing, conversion of tailing into paste or conversion of tailing into cakes over the tailing dam, are being explored to restrict the height of tailing dam below approved limit of 74m. In case of technological or financial viability prevails, either of the above method could be implemented in future. In case of the tailing dam height not raised up to 74m during the proposed period, the requirement of waste may also reduce, thus the table mentioned above are tentative.

The tailing dam has been constructed as per the approved design by IISc Banglore. However, the following protective measures shall be taken for tailing dam to ensure the stability of tailing dam.

- Reclaim Drain shall be provided along toe of Tailing dam
- Regular maintenance and monitoring shall be carried out using the instrument mentioned below.

Table 44: Types of Monitoring

Instrument	2022-23	2023-24	2024-25	2025-26	2026-27
Monitoring by Prism	√	√	√	√	√
Monitoring by Inclinometer	√	√	√	√	√
Monitoring Piezeometer	√	√	√	√	√

As of now, Rampura-Agucha Mine tailing dam height raising is completed up to 60 m height. Up to 31st Dec 2021, tailing stacked in the tailing dam is 74.5 million tons since inception.

The "tailings/rejects generated from beneficiation plant is categorized as of high volume and low effect waste and has been excluded from category of hazardous wastes by the Government of India, Ministry of Environment, Forest and Climate change vide notification G.S.R No 395(E) dated 4th April 2016".

Attached is the approx. tailings generations and chemical /mineralogical composition of tailing sample done by Jk Tech in 2010. Extensive work was carried out in past (latest was taken up in 2015 with Xstrata) to evaluate economical means of recovering valuable minerals from tailings. Further, research and development work are continuing for economical extraction for valuable minerals from tailings.

Table 45: Final Tailings Chemical Assay

Final Tailings Chemical Assay											
Element	**Ag**	**Al**	**As**	**Be**	**Bi**	**Ca**	**Cd**	**Co**	**Cr**	**Cu**	**Fe**
% element	0.0014	1.96	0.01448	<0.0001	<0.0001	0.3626	0.00292	<0.0001	0.0055	0.00632	4.56
Element	**Li**	**Mg**	**Mn**	**Na**	**Ni**	**Pb**	**Sb**	**Se**	**Sr**	**Tl**	**Zn**
% element	0.00192	0.79	0.0489	0.00368	0.00396	0.318	<0.0001	<0.0001	0.00152	0.00764	1.148

As described under above study, the most abundant silver minerals identified in the tailing are Pyrargyrite, argyrodite and Freibergite. It should be noted that pyrargyrite contains antimony (also present in freibergite), and globally this has a history of causing poor flotation performance resulting into poor recovery of silver.

Total area covered under the tailing dam as of now is 190.95 Ha. The tailing dam height has to be raised from current approved height of 60 m to 74m height to accommodate all the tailings generated during the life of the mine.

The thickened tailing is neutralized with lime before discharging into tailing pond and does not possess any toxic contents in it.

12.6 Reagents and Water

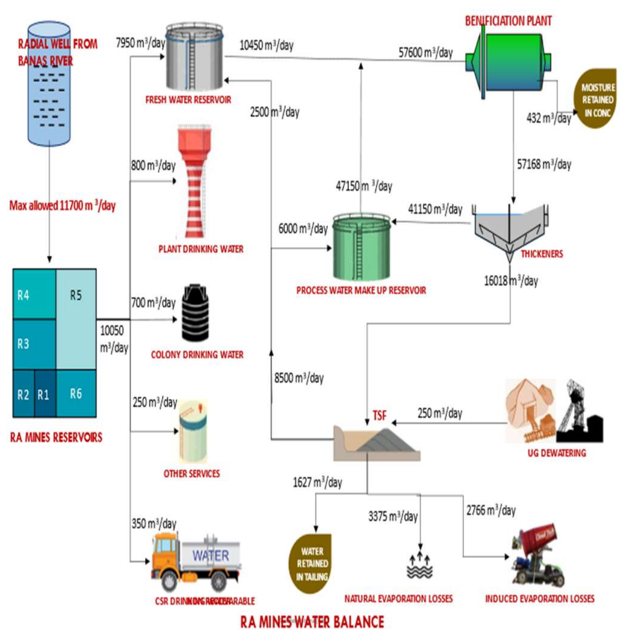


Figure 38: Water Requirement for mining and processing plants at RAM

Table 46: Flotation Stream 1

S. No.	Type of flotation equipment (froth/ column)	Stages (rougher / cleaner, etc.), if applicable	Make	Capacity (tph)	Feed Size (mm)
1	Conditioner tank	Conditioner	MBE	NA	0.063
2	Conventional flotation cell	Roughers	Outotec	NA	0.063
3	Conventional flotation cell	Scavengers	Outotec	NA	0.063
4	Conventional flotation cell	Cleaners	Outotec	NA	0.063

Table 47: Flotation Stream 2

S. No.	Type of flotation equipment (froth/ column)	Stages (rougher / cleaner, etc.), if applicable	Make	Capacity (tph)	Feed Size (mm)
1	Conditioner tank	Conditioner	Outotec	NA	0.063
2	Conventional flotation cell	Roughers	Outotec	NA	0.063
3	Conventional flotation cell	Scavengers	Outotec	NA	0.063
4	Conventional flotation cell	Cleaners	Outotec	NA	0.063

Table 48: Flotation Stream 3

S. No.	Type of flotation equipment (froth/ column)	Stages (rougher / cleaner, etc.), if applicable	Make	Capacity (tph)	Feed Size (mm)
1	Conditioner tank	Conditioner	Bateman	NA	0.110
2	Conventional flotation cell	Roughers	Bateman	NA	0.110
3	Conventional flotation cell	Scavengers	Bateman	NA	0.110
4	Conventional flotation cell	Cleaners	Bateman	NA	0.110

Table 49: Flotation Stream 4

S. No.	Type of flotation equipment (froth/ column)	Stages (rougher / cleaner, etc.), if applicable	Make	Capacity (tph)	Feed Size (mm)
1	Conditioner tank	Conditioner	MBE	NA	0.063
2	Conventional flotation cell	Roughers	Outotec	NA	0.063
3	Conventional flotation cell	Scavengers	Outotec	NA	0.063
4	Conventional flotation cell	Cleaners	Outotec	NA	0.063

12.7 Process Control Philosophy

A highly automated and instrumented process control has been envisaged in the beneficiation plant.

- On-line Stream Analysis System for measurement of elements concentration in slurries to control metal losses.
- Advanced Process Control operating system is designed to optimize, stabilize and control individual unit operations as well as the entire plant for optimum metal recovery.
- Froth Camera System makes use of machine vision technologies to measure the speed of the froth.
- Particle Size Analyzer is a sizing system installed in grinding circuit for mineral slurries. It takes automatic samples from streams and measures their particle size distribution for liberation of minerals.
- Magnetic Proflot system for fine particle recovery in zinc flotation.
- Any drive will be in running condition if all the start permissive conditions are simultaneously fulfilled.

12.8 Conclusions and Recommendations

From above details we can conclude that SK Mine has state of the art processing plants to treat run of mine ore and produce Lead and Zinc concentrates. Plants have all process controls required for optimum plant operations. Further, it can be recommended that the mine could consider stockpiling and blending of different ore types to have optimum feed grade designed for the plants.

12.9 Risks and Opportunities

The Mine has state of the art processing facilities hence there is no major risk. The process water seems to be bit heavy, could lead to plant deteriorations. There is substantial debottlenecking opportunity to further expand processing capacities. Further a study can be conducted to treat tailings to extract metals.

13 Primary Surface Infrastructure

13.1 Roads

Existing public infrastructure provides easy access to the mining are. Production commenced in 1991 with an open pit mine which ceased production in March 2018. Mining from underground commenced in 2013. The Company applied for a new prospecting permit covering the surrounding area which was secured during 2010. This is important as the deposit is dipping towards the eastern limit of the mining lease and the location of the lease boundary makes deep exploration drilling challenging.

Figure 39: Mine is located in close proximity to villages and towns

13.2 Stockpile and Storage Facilities

13.2.1 Rock Waste Management

The present waste dump has reached its planned height of 140 m from ground, the dump is having seven lift each of 20 m height, and the berm width up to 80 m lift is 20m, on 100m and 120m lifts the berm width is 24 m. The dump is so prepared to form an individual face angle of 370 and an overall slope angle of 270.

During the proposed mining period, waste generated from underground mining operation shall be dumped in and around outer periphery of tailing dam, for strengthening of tailing dam and also for making safety bund/wall around pit and tailing dam. Density of in situ waste is 2.73 t/m3 whereas the density of waste after compaction becomes 2.201 t/m3.

Details for utilization of year wise generation waste for tailing dam is mentioned under para 3.8 (a) along with table. In addition to above, dumping of part of underground waste in the any mined out stope voids in underground may also considered, whenever / wherever opportunities prevail.

These sites are sufficient to take care of waste disposal during the plan period. Due care is taken in all respect for its confinement like the provision of dump stabilization by vegetation on ultimate non-moving ends.

13.2.1.1 Measures taken for dump stabilization and precautions for preventing slope failure:

All around the periphery of waste dump, a collector drain/ bund shall be formed to divert the rainwater away from the dump.

The dump top shall be properly levelled with a slope to avoid water retention on dump top/ dump benches and to prevent the rainwater flowing along slope. Installation of renewable source of energy as solar power plant have been made at waste dump top (535m RL and 515m RL eastern part) covering approximate 50 Ha. However, the drains shall be kept clear of soil debris and effective for the free flow of water well before the onset of monsoon.

Besides doing continuous inspection of the dump, regular monitoring in the form of determining the movements of fixed points along the stabilized benches relative to the datum shall be carried out with the help of survey instruments.

The non-active area/matured area of the waste dump shall be covered with plantation for its stabilization. Further, Rampura-Agucha mine has successfully developed innovative method of overburden stabilization and rehabilitation, for the first time in Lead-Zinc mine by use of Geo-Textile mats or Geo soil savers. Geotextiles are made from 100% organic material a naturally occurring fibre derived from a renewable resource namely, coconut (cocosnucifera) husk. Coir fibers resemble the wood fibers in terms of physical properties and chemical composition. Coir fiber are spun into yarn by mechanism of friction spinning& the yarn is woven into different forms of mesh matting (netting).

Geo-textiles mats were used for stabilizing and rehabilitating with vegetation over overburden slopes. The method adopted included stabilization of slopes by levelling, laying of Geo-textile mats, supply of small quantities of conditioners, sowing of vetiver grass seeds and some native species with periodic watering.

13.2.1.2 Existing Dump

Table 50: Existing Dump

S. No.	Year	Dump Id	Type of Dump	Proposed Area (ha)	Height (m)	Total Dump Quantity (t)	Existing Dump Location
1	NA	Waste Dump	Waste Dump	NA	140	NA	North of the Lease Area

13.3 Tailings Disposal

Tailings from all four flotation plants are mixed and thickened to 65 to 68% by mass using a deep cone thickener (“DCT”) and used as cemented paste backfill for filling underground stopes or are pumped to the tailings storage facility (“TSF”).

Cemented paste backfill is produced in a new paste plant with a nameplate capacity of 2.5 Mtpa of solids or 800,000 m3 of paste fill. A portion of the DCT thickener underflow is further dewatered to around 74% solids by mass in a new paste plant. Approximately 70% of the DCT underflow is dewatered to nominally 18% moisture using four disc filters. The filter cake is mixed under controlled conditions in a twin paddle mixer with the balance of the DCT underflow and up to 12% cement to produce a 7-inch slump paste which is then pumped, by high pressure piston pumps, around 2km to four boreholes to feed the underground stopes. Typically, a 10,000 m3 stope can be filled in around 70 hours. The paste plant and paste delivery systems incorporate a high degree of standby equipment and piping to maintain a high operational availability.

The balance of the DCT underflow are neutralised with lime prior to pumping to the TSF. Alternatively, the tailings can be thickened in the older, conventional tailings thickeners, to 45% solids by mass, for disposal in the TSF. Water is recovered in to ponds at the DCT and the conventional thickeners and is recycled as required to the plant. Further water is also recovered from the TSF and recycled to the water ponds. The existing TSF covers around 110 km2.

The current tailings storage facility is constructed using waste rock with the base and walls lined with clay. The dam wall has been raised to its current level of 39m to increase the storage capacity. It is reported that the dam will hold sufficient material for the envisaged LoM at the reported maximum height of 51m. The TSF is currently at its 7th phase of raising, upon completion of which the dam height will be 45 m. A Phase 8, has been proposed, which will raise the TSF to 51 m have capacity for approximately 14Mt of tailings.

HZL is currently investigating re-processing of the tailings to use available capacity at the plant and recover additional metal, as well as investigating back-filling the open-pit void with drystacked tailings.

13.4 Power and Water

Power for the processing operations is supplied by the captive power plants ("CPP") at Zawar and Chanderiya which provide around 90% of the requirements. Power requirements for processing facilities are around 38-40 MW.

Overall water usage is approximately 3.2 m3/t, with fresh water, making 0.45 m3/t of the required water. Fresh water is sourced from the Banas River 55 km from the RAM concentrator. There is also a constraint by the Rajasthan Government related to the maximum amount of water that can be extracted, which is currently limited to 11,200 m3/day. It is reported that plant water requirements are satisfied by internal mine and recycle water sources and that no additional water is required.

14 Market Studies

14.1 Introduction

Hindustan Zinc (HZL) is India's largest and world's second largest zinc-lead miner. With more than 50 years of operational experience, they have a reserve base of 161.2 million MT and an average zinc-lead grade of 5.9% and mineral resources of 286.7 million MT, our mine life is over 25 years. Their fully integrated zinc operations hold 78% market share in India's primary zinc industry, and they are the 6th largest silver producers globally with an annual production of 913,000t.

The market was negatively impacted by COVID-19 in 2020 and 2021. Considering the pandemic scenario, the construction activities were stopped temporarily during the lockdown to curb the spread of new COVID-19 cases, thereby decreasing the demand for zinc and lead-based products such as galvanised metal, lead sheets, and others from the construction industry. Furthermore, the demand for lead-acid batteries decreased due to the temporary pause of the automotive manufacturing units during the lockdown. However, the demand for lead-acid batteries, especially valve-regulated lead-acid (VRLA) batteries, from the electronics and telecommunication industry increased during this period, as people opted to work online from their residence, which enhanced the demand in the market studied. With the lifting of restrictions, companies are keen to see a return to pre-2020 levels of activity.

14.2 Zinc

14.2.1 Application Of Zinc

Approximately 14% is used in the production of zinc die casting alloys. Nearly 9% of the zinc is utilized for oxides and chemicals and approximately 10% is used in alloys and castings. Some of the most common applications of zinc are listed below:

- **Galvanising**: Zinc offers one of the best forms of protection for steel against corrosion. It is used extensively in building & construction, infrastructure, household appliances, automobiles, steel furniture and other applications where lasting steel products are required.
- **Zinc Oxide**: The most widely used zinc compound, zinc oxide is used in the vulcanisation of rubber, as well as in ceramics, paints, animal feed, pharmaceuticals and several other products and processes. A special grade of zinc oxide has long been used in photocopiers.
- **Die Castings**: Zinc is an ideal material for die casting and is extensively used in hardware, electrical equipment, automotive and electronic components. Zinc die cast alloys are used in production of highly durable and visually appealing hardware fittings.
- **Alloys**: Zinc is extensively used in making alloys, especially brass, which is an alloy of copper and zinc.

14.2.2 Supply and Demand

The price of zinc is driven mostly by these five factors:

- Chinese Demand

- Chinese Supply
- Global Stocks
- US Demand

As with most industrial commodities, China plays a pivotal role in determining zinc prices. China is the top consumer of refined zinc used in galvanized steel. Therefore, a key indicator of zinc demand in China and elsewhere is steel demand. Decisions about whether to undertake or hold off on infrastructure projects can create huge fluctuations in steel demand. Ultimately, these decisions can flow through to the zinc market. A key factor impacting zinc output in China is the country's increasing environmental awareness. Poor air quality has forced the government to take a harder look at the mining industry as a contributor to pollution. If China curbs the production of zinc to deal with this problem, then the country will be more reliant on imports. This could drive prices higher. The London Metals Exchange (LME) keeps track of global stock levels for zinc and other industrial metals.
Current world production is approximately 13 million tonnes. HZL produces approximately 913,000t of zinc per year and is well established in the market.

Table 51: : HZL Product Range

Product	Form	Weight
Special High Grade (SHG)	Standard Ingot	25 kgs
	Jumbo Ingots	1000kgs
Continuous Galvanising Grade (CGG)	Jumbo Ingots	1000kgs
High Grade (HG)	Standard Ingot	25 kgs
	Jumbo Ingots	600 kgs
Prime Western (PW)	Standard Ingot	25 kgs
Electro-Plating SHG (EPG SHG)	Standard Ingot	25 kgs
Hindustan Zinc Die-Cast Alloy (AZDA)	Standard Ingot	9 kgs

14.2.3 Prices

As with all commodities, prices fluctuate. Prices in general are Zinc price predictions from the leading international agencies for the next few years are as follows:

- **The World Bank** in its commodity forecast report estimated that the average spot price for zinc will fall to $2,400 per metric ton (t) in 2022, down from $2,700/t at the end of 2021. After that, a slow growth period will start.

- **The IMF**'s report indicated a completely different expectation: a rise from $2,828/t in the end of 2021 to $2,859 in 2022. For the following period, IMF experts expect a smooth, gradual decline. They predict the price will drop to $2,818/t by 2026.
- **The Industry Innovation and Science Australia**'s prediction is like the World Bank's predictions: they expect a decrease in the zinc spot price from $2,686 at the end of 2021 to $2,362 in 2022, with further slow increase through 2026.
- HZL is using prices as projected by LME as listed below:

Table 52: HZL Financial Model Prices

Particulars	UOM	FY'23	FY'24	FY'25	FY'26	FY'27	FY'28	FY'29	FY'30	FY'31	FY'32	Average
LME - Zinc	$/MT	3,183	2,911	2,684	2,621	2,658	2,706	2,706	2,706	2,706	2,706	2,759
- Lead	$/MT	2,179	2,047	1,974	1,962	1,997	2,082	2,082	2,082	2,082	2,082	2,057
- Silver	$/Troz	22.19	20.47	21.61	21.33	21.30	21.10	21.10	21.10	21.10	21.10	21.24
Ex Rate	Rs/USD	74.94	75.43	76.52	77.73	78.64	79.11	79.51	79.51	79.51	79.51	76.65

The figure below displays the price for zinc over the last five years. The price projections above agree in general with the historic average prices although it has been as low as US$1,500/t in January 2016 and as high as US$ 4,000/t in April 2022. In our opinion, the recent high price may be because of stock shortages created by the pandemic. These high levels are not sustainable, and we already see a substantial drop in prices which should stabilise at the levels projected by the LME, World Bank and others.

Figure 40: 5-Year Zinc Price in US Dollars

Source: Trading Economics

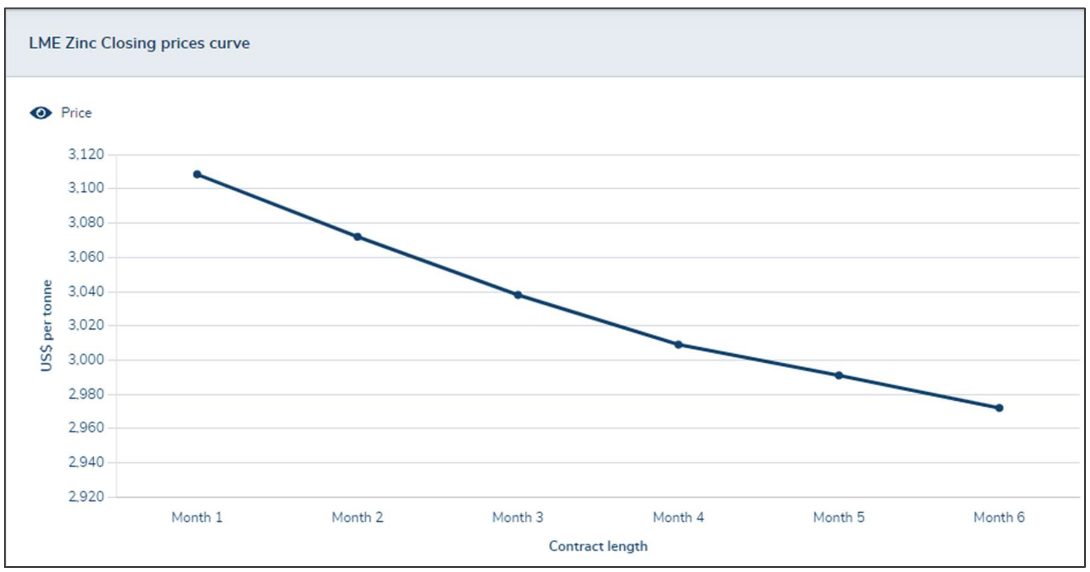


Figure 41: LME Zinc Contract Prices

Source: LME Website 12 July 2012.

Chemical Composition

Component	Guaranteed Content	HZL Typical
Zinc Zn)	99.9950%Min	99.9960% Min
Lead (Pb)	0.0030% Max	0.0020% Max
Copper (Cu)	0.0010% Max	0.0003% Max
Cadmium (Cd)	0.0030% Max	0.0002% Max
Iron(Fe)	0.0020% Max	0.0010% Max
Aluminium (Al)	0.0010% Max	0.0001% Max
Tin (Sn)	0.0010% Max	0.0001% Max

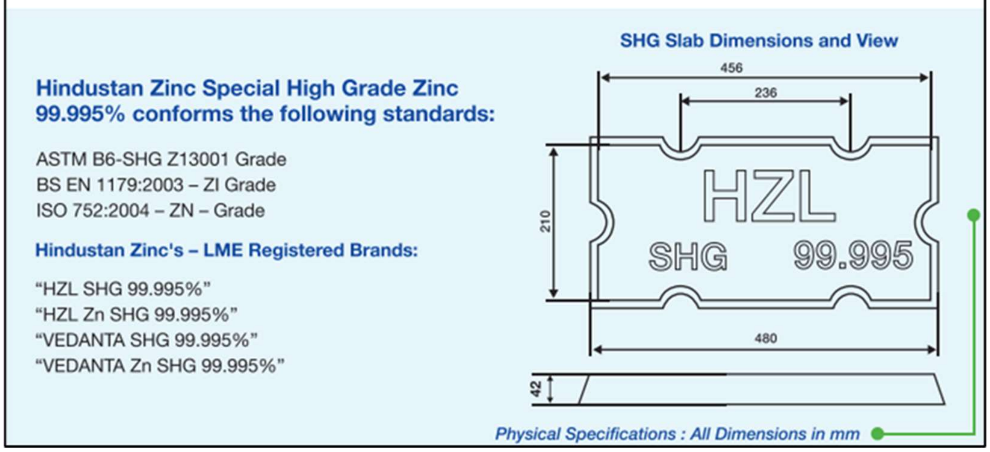


Figure 42: HZL Zinc Specifications

14.3 Lead

14.3.1 Applications for Lead

The battery sector is the single largest consumer of lead, accounting for around three-quarters of the demand. It can be sub-divided into the following groups:

- **SLI (Starting-Lighting-Ignition) batteries**, which currently accounts for over half of the total lead demand. These are mainly used in cars and light vehicles but are also found in other applications such as golf carts and boats. SLI battery demand in turn can be split into original equipment and replacement, with replacement demand outstripping original equipment demand by about 4:1 in mature markets.
- **Industrial batteries**, which currently consumes around a quarter of the total lead produced. This sector can be split roughly 50:50 into stationery and traction batteries. Stationary batteries are principally used in back up power supply systems; traction batteries are used for motive power in equipment such as forklift trucks and motorised wheelchairs.

- The remainder is used in non-battery applications. The second largest current end use of lead for non-battery applications, accounting for around 20% of lead consumption, is the alloys and chemical industry. Principal markets are for cathode ray tubes used in television screens and computer monitors, for Poly Vinyl Chloride (PVC) stabilisers and for making pigments for industrial use. Cable and other industries account for the remaining 5% of lead demand.

14.3.2 Supply and Demand

Growth in the construction industries was driving the overall market growth for a long time. High demand from renovation in the construction sector, including gutter and gutter joints and metal for roofing materials were propelling the market demand. Now, high demand for the electrical vehicle is influencing lead acid batteries demand emerging as the key driving factor for the market growth. Additionally, vigorous investment in improving telecom networks along with significant development in data centres are expected to enhance the industry position.

However, high production cost with stringent challenging processes is inhibiting the growth of lead market globally.

By the application segment, batteries segment is expected to dominate the market during the forecast period. As lead-acid battery is utilised in the form of stationary batteries, SLI batteries, portable batteries which includes electronics, consumers, telecom, energy storage system and others. SLI batteries have vast application in automobile designing and installation specifically with the automobile's charging system, that allow continuous cycle of charge and discharge in the battery each time the vehicle is in use. Furthermore, development in construction, machineries, and other battery dependent end-product is helping the market to grow.

Regionally, Asia Pacific is expected to dominate the lead market and is expected to grow during the forecast period. Developing countries like Japan, China, and India have boosted the market growth due to an increase in manufactures for machinery and tools across the world. This is expected to drive the market demand in this region.

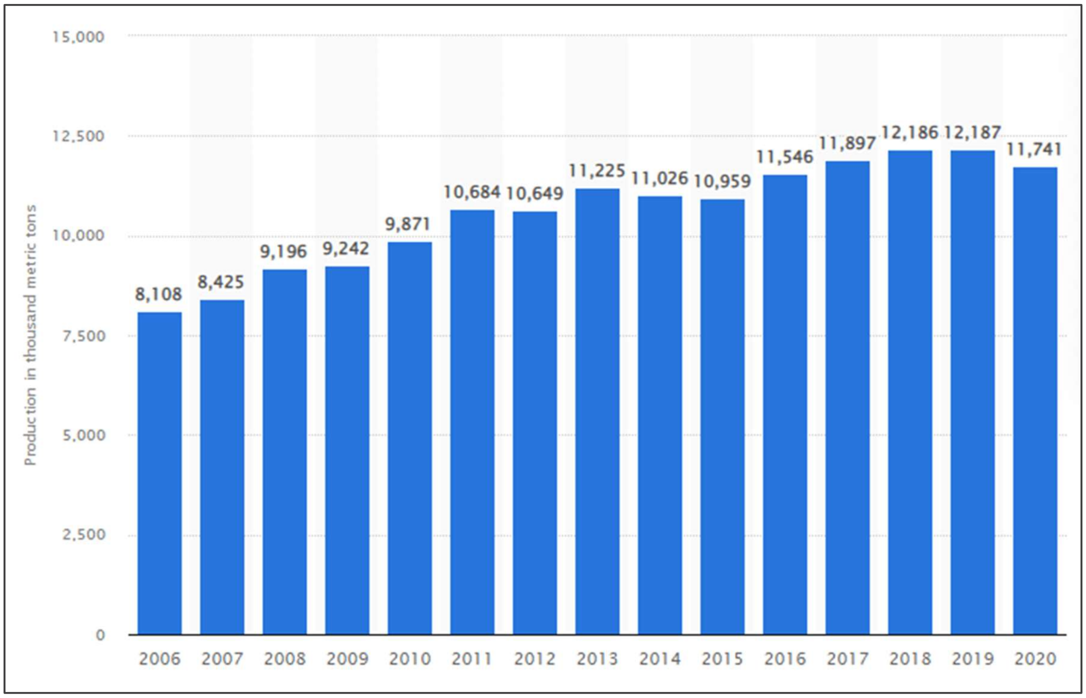


Figure 43: World Production of Lead

Figures above indicates a slow growth over the last 5 years with a slight dip in 2020, possibly due to COVID 19 restrictions. In our opinion, the production will recover and continue the slow growth of pre-pandemic years.

HZL produces lead ingots with a minimum of 99.99% purity which are registered with LME at a level of 210,000 t/a. This is expected to continue for the forecast period.

14.3.3 Prices

As with almost all commodities, prices are cyclical. Based on the 12-month price chart, there appears to be a severe reduction in price from around US$2,400/t to US$1,971 in July 2022. The 10-year price curve indicates that this is probably only a market correction. It is too early to guess at what level the price would stabilise.

Figure 44: 12-Month Price of Lead

Source – Trading Economics

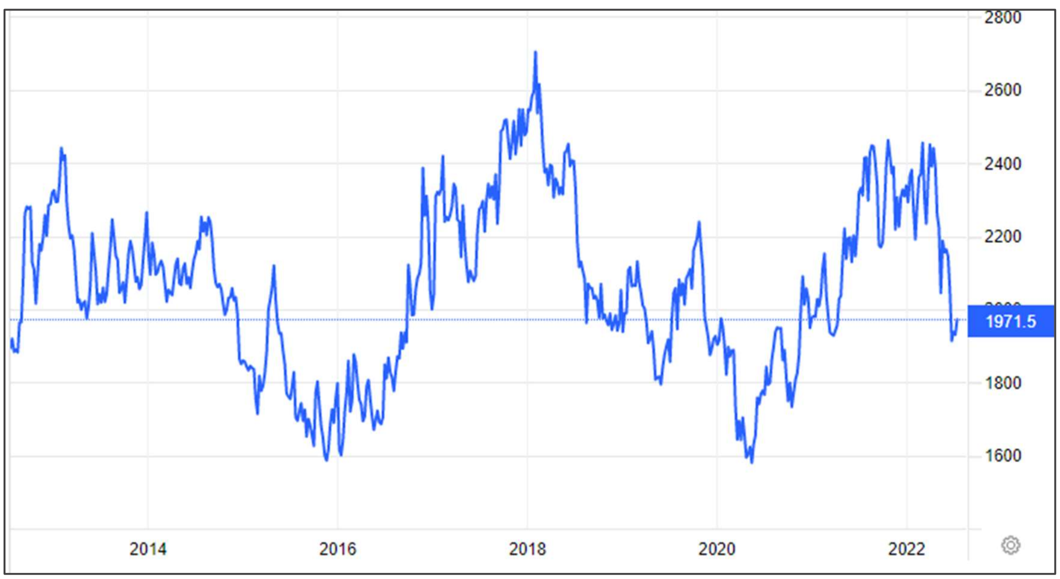


Figure 45: 5-Year Price of Lead

Source – Trading Economics

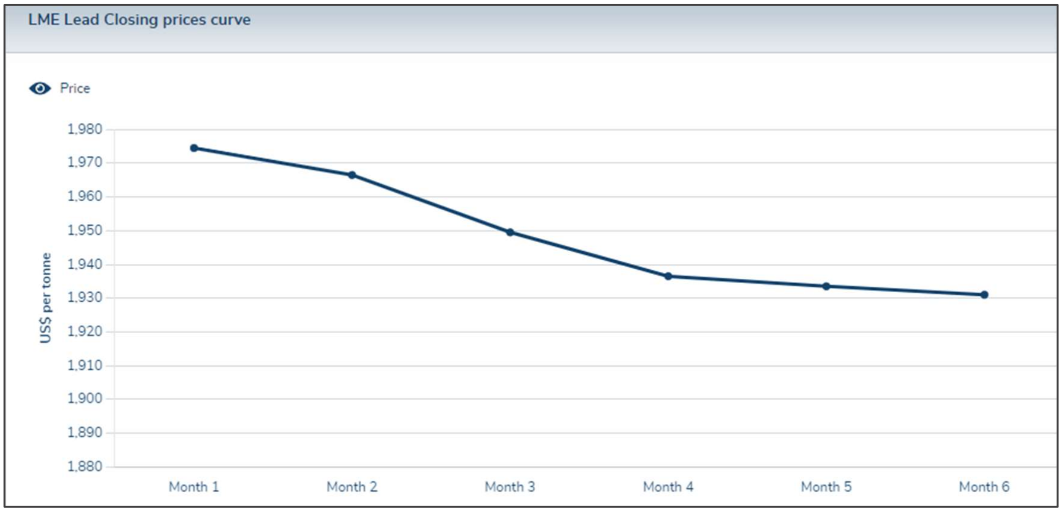


Figure 46: LME Contract Price on July 2022.

Source: LME July 2022

Chemical Composition — **LME Brand : Vedanta 99.99**

Component	Composition- BS EN 12659 : 1999 PB990R	HZL Typical
Lead(Pb)	99.99% Min	99.993%
Silver(Ag)	0.0015% Max	<0.0005%
Arsenic(As)	0.0005% Max	<0.0003%
Bismuth(Bi)	0.0100% Max	<0.0050%
Cadmium(Cd)	0.0002% Max	<0.0002%
Copper (Cu)	0.0005% Max	<0.0005%
Nickel (Ni)	0.0002% Max	<0.0002%
Antimony(Sb)	0.0005% Max	<0.0005%
Tin (Sn)	0.0005% Max	<0.0005%
Zinc(Zn)	0.0002% Max	<0.0002%

Vedanta 99.99% (Bundle Specification)

Ingot Weight	25 Kg (+/- 1.2 Kg)
Ingot Dimensions	85 (+/-2)mm Width | 535(+/-2)mm Length | 75(+/-2) mm Height
Bundle Weight	1050 Kg (+/-50 Kg)
Bundle Configuration	6 ingots/layer x 7 layers | 42 ingots
Bundle Dimension	510(+/-12)mm Width | 535(+/-2) mm Length | 520(+/-14) mm Heig
Strapping	32 mm Tenax strap | 2*2 strapping

Vedanta 99.99 & Vedanta Pb 99.99- Hindustan Zinc Lead Ingots are LME Registere

Figure 47: HZL Lead Specifications

14.4 Silver

The extraordinary events of 2020 have had a profound effect on virtually all markets around the globe and silver has been no exception. The metal's supply/demand fundamentals, investment, prices, trade-flows and inventories have all experienced sensational fluctuations over the past 12 months or so. The effect of the pandemic is set to remain relevant to silver for some time to come. Several key silver mining countries were hit hard by lockdown restrictions, and global silver supply declined. This was more than offset, however, by losses across most of silver's physical demand segments, which suffered as a result of restrictions to economic activity as well as depressed consumer sentiment and/or income loss. This resulted in a large silver market surplus. One notable exception was physical investment. A growing appetite for safe haven assets and, initially, the strength of the gold price all boosted investors' appetite for silver bars and coins last year, culminating in an 8% rise overall.

14.4.1 Applications for Silver

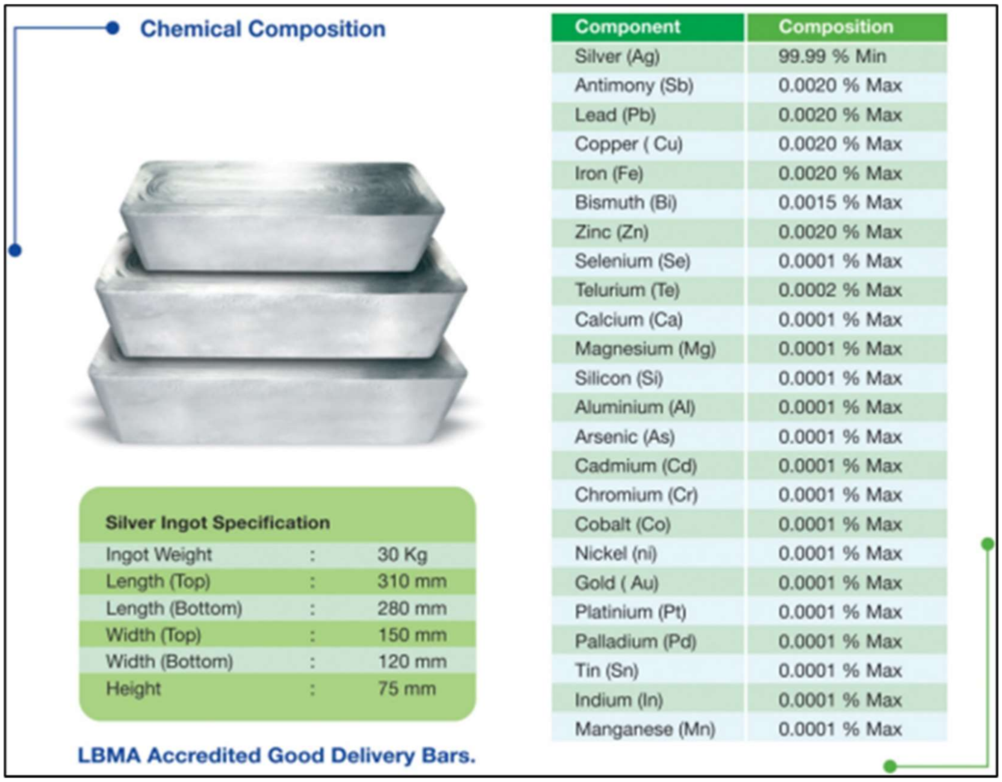
Silver metal has been known since ancient times for its brilliant white metallic lustre with high ductility and malleability properties. The precious metal has varied uses backed by its excellent heat and electrical conductivity levels.

In India, the highest usage of silver is in jewellery, followed by coins & bars silverware and industrial fabrication. With growing Indian economy, silver demand especially in the industrial sector is expected to follow a healthy growth in the coming years with an increased off take especially in electrical and electronics as well as brazing alloys and solders.

14.4.2 Supply and Demand

According to the Silver Institutes report "World Silver Survey 2021", In 2020, global mine production suffered its biggest decline of the last decade, falling by 5.9% y/y to 784.4Moz (24,399t). This was caused by temporary mine closures in several major silver producing countries in the first half of the year as a direct result of the COVID-19 pandemic. Output from primary silver mines declined by 11.9% y/y to 209.4Moz (6,513t). This exceeded the drop that silver by-product output from lead-zinc and gold mines suffered, which fell by 7.4% to 248.3Moz (7,724t) and by 5.7% to 123.3Moz (3,834t), respectively. Countering this trend, silver production from copper mines increased by 3.5% y/y to 198.3Moz (6,169t).

At the county level, the largest declines were in nations which implemented COVID-19 lockdowns that required mines to temporarily halt operations. This led to substantially lower silver production in Peru (-26.1Moz, 810t), Argentina (-10.0Moz, 311t), Mexico (-9.6Moz, 299t) and Bolivia (-7.2Moz, 223t). Despite the disruption caused by the pandemic, mines in other countries were able to continue operating at full capacity throughout the year and output increased in Chile (+9.1Moz, 284t), India (+1.2Moz, 38t) and Australia (+1.2Moz, 37t).



Component	Composition
Silver (Ag)	99.99 % Min
Antimony (Sb)	0.0020 % Max
Lead (Pb)	0.0020 % Max
Copper (Cu)	0.0020 % Max
Iron (Fe)	0.0020 % Max
Bismuth (Bi)	0.0015 % Max
Zinc (Zn)	0.0020 % Max
Selenium (Se)	0.0001 % Max
Telurium (Te)	0.0002 % Max
Calcium (Ca)	0.0001 % Max
Magnesium (Mg)	0.0001 % Max
Silicon (Si)	0.0001 % Max
Aluminium (Al)	0.0001 % Max
Arsenic (As)	0.0001 % Max
Cadmium (Cd)	0.0001 % Max
Chromium (Cr)	0.0001 % Max
Cobalt (Co)	0.0001 % Max
Nickel (ni)	0.0001 % Max
Gold (Au)	0.0001 % Max
Platinium (Pt)	0.0001 % Max
Palladium (Pd)	0.0001 % Max
Tin (Sn)	0.0001 % Max
Indium (In)	0.0001 % Max
Manganese (Mn)	0.0001 % Max

Silver Ingot Specification		
Ingot Weight	:	30 Kg
Length (Top)	:	310 mm
Length (Bottom)	:	280 mm
Width (Top)	:	150 mm
Width (Bottom)	:	120 mm
Height	:	75 mm

Figure 48: HZL Silver Specifications

After rising for two years, global silver demand weakened by 10% in 2020 to 896.1Moz (27,872t) as the impressive gains in physical investment were more than offset by heavy losses in jewellery and silverware. After falling just short of record levels in 2019, industrial fabrication fell 5% last year to a five-year low of 486.8Moz (15,142t). Unsurprisingly, this was overwhelmingly due to the impact of the COVID-19 pandemic on economic activity and, in turn, many silver end-users. Regional performances diverged, with Europe suffering a notable 8% decline, while North America rose by 2%, chiefly through higher demand in such areas as silver powder for photovoltaic (PV) ends. Demand in East Asia also fell overall, although performances were very mixed at the country-level; losses were seen in China, but Japan and Taiwan enjoyed gains.

On a sectoral basis, electronics & electrical demand fell a modest 4% as gains for PV offset losses elsewhere. Other industrial offtake in turn fell 7% as a strong showing for EO catalysts could only partially counter India's heavy losses in this segment. In general, thrifting and substitution had a limited impact on silver use as the price was insufficiently high for long enough to trigger interest and as many areas present little room for further savings.

Strong growth in mine production this year is expected to be followed by continued growth in the medium term. This will be driven by increased output from a number of major operating mines alongside new projects, with a significant contribution coming from primary silver operations in Mexico.

14.4.3 Price

Figure 49: 5-Year Silver Price

(Source – Trading Economics)

In the period 2015 to 2020, the price varied around an average of approximately US$16.5/oz. During the world pandemic, silver offered a haven for storing wealth which is reflected in the average price shooting up to an average above US$24/oz. With the acceptance that the pandemic was over in mid-July 2022, the price is starting to return to earlier levels and may stabilise near to the pre-2020 levels.

15 Environmental Studies, Permitting and Social or Community Impact

15.1 Introduction

The knowledge of present environment of the core and buffer zone of the existing mining area is important to assess the impact of various project activities on environment. The knowledge of present-day environment is also helpful in planning management of environment and planning of mitigation measures. To assess the composite baseline of mine and processing facilities related to the environmental quality of the area, field assessment has been conducted considering following components of the environment, viz. land, meteorology, air, noise, water, soil, biological and socio-economic. The relevant information and data (both primary and secondary) were collected in core as well as buffer zone (10 km distance from the Mine Lease boundary) in accordance with the guidelines of MoEF&CC for undertaking EIA Studies and preparation of Draft EIA/EMP reports.

The Rajasthan State falls in a region of low Seismic hazard zone with the exception being moderate hazard in areas along west state border. It mainly lies in Zones II and III. Several faults have been identified in this region out of which many show evidence of movement during the Holocene epoch

15.2 Environmental Studies

The projects should not cause any significant impact on the environment of the area, as adequate preventive measures have been adopted to contain various pollutants generated due to the proposed current and proposed expansion projects within permissible limits. Development of Greenbelt / Plantation around the mining lease will minimize the environment pollution and improve the overall aesthetic beauty.

Environmental Monitoring Programme has been and will be continued for various environmental components as per conditions stipulated in Environmental Clearance Letters issued by MoEFCC & Consent to Operate issued by SPCB. Six monthly compliance reports will be submitted every year to Regional Office by 1st of June & 1st of December. Quarterly compliance Report for conditions stipulated in Consent to Operate will be submitted to SPCB on regular basis.

15.2.1 Land Use

Land use studies, delineating forest area, agricultural land, grazing land, wildlife sanctuary, national park, migratory routes of fauna, water bodies, human settlements and other ecological features were completed. Land use plans of mine lease areas were prepared to encompass preoperational, operational and post operational phases.

The land use is classified into four types – forests, area under cultivation, culturable waste and areas not available for cultivation. Forest occupies a small portion of the study area. The 226 ha forest areas are only scattered bushes of thorny plants. Altogether 25,227 ha of irrigated and un-irrigated lands, is used for cultivation in the study area. This is approximately 50% of the area.

Culturable waste land is land suitable for cultivation, which have not been brought under cultivation at any time. The 13,988 ha Culturable Waste is 27.73% of the area surrounding the mine. A total of 10,973 ha in the study area is not available for cultivation, comprising the mining complex, two river systems with the reservoirs, HZL Township and other human settlements.

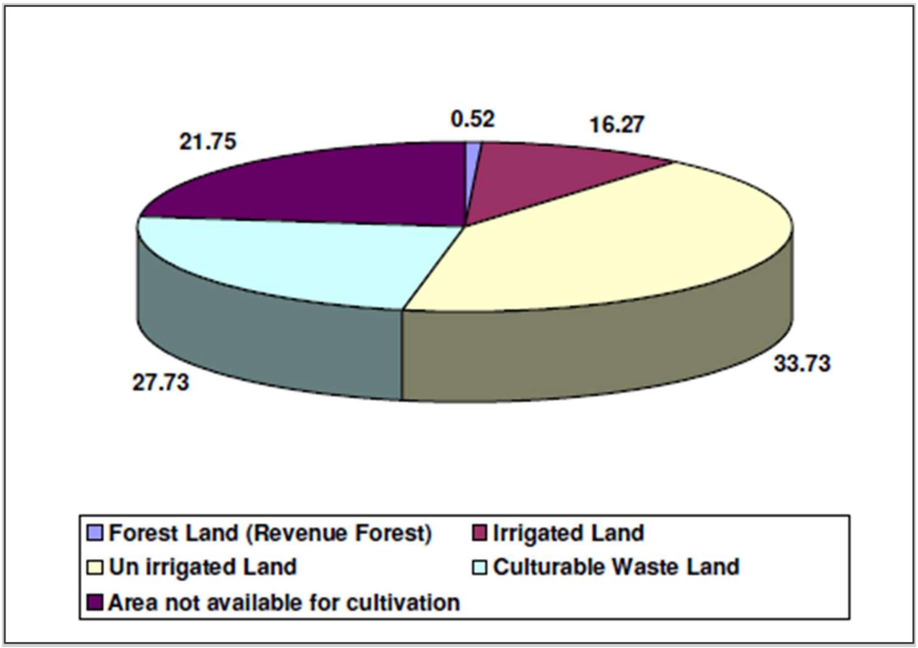


Figure 50: Breakdown - use of land

15.2.2 Climatology

The climate of the district is generally dry and healthy, and the seasons are similar to nearby SKM and RDM. The summer season starts during the middle of February and - continues up to the first week of June. Summer is followed by south-west monsoon which last till the end of September. October and November are the post-monsoon months. December is the coldest month with mean daily maximum and minimum temperatures being 24 ^{0}C and 10^{0}C respectively. During peak summer, temperature shoots up to 44.6 ^{0}C. Relative humidity varies from 25% in summer to 82% in winter (Census 2011). Due to the mild weather conditions, the mines can operate throughout the year, and should have no weather-related restrictions.

Table 53: Annual Rainfall Records [1987-2008] supplied HZL

Year	Rainfall (mm)	Year	Rainfall (mm)	Year	Rainfall (mm)
1989	455	2000	352	2011	1017
1990	806	2001	428	2012	747
1991	315.4	2002	153	2013	749
1992	288.4	2003	575	2014	709.5
1993	294.5	2004	604	2015	471
1994	457	2005	418	2016	814
1995	308.5	2006	792	2017	491
1996	480.6	2007	721	2018	611.2
1997	453	2008	472	2019	847.5
1998	309	2009	289	2020	504
1999	180	2010	951	2021	791
				Average	541.0

15.2.3 Air Quality

The ambient air quality with respect to the study zone of 10 km radius around the mine lease boundary forms the baseline information. The various sources of air pollution in th3e region are mining, industrial, transportation and residential activities. The prime objective of th1987-2008] supplied baseline air quality study was to assess the existing air quality of the area. This will also be useful for assessing the conformity to standards of the ambient air quality after proposed expansion is completed. The study area represents mostly rural environment with intermittent mixed land uses such as HZL mining and industrial activities.

HZL is also monitoring ambient air quality in and around the mine lease area at six locations as part of post project monitoring program. The results of this monitoring have also been included to understand impacts of existing activities, if any, and trends.

This section describes the selection of sampling locations, methodology adopted for sampling, analytical techniques and frequency of sampling. The monitoring was carried out during winter season (1st December 2008 to 28th February 2009).

HZL is also monitoring ambient air quality in and around the mine lease area at five locations comprising of three locations in core zone and two locations in buffer zone as part of post project monitoring program. HZL is monitoring at these locations with frequency of weekly once for 24 hourly durations. The parameters monitored are SPM, RPM, SO2 and NO x. The dustfall rates are monitored monthly. The results of past two years for three common locations monitored by both HZL and VIMTA are presented graphically to understand trends.

It is observed that the results monitored by HZL are within the stipulated standards. The data indicates that the concentrations are comparable with the concentrations monitored by VIMTA.

The noise monitoring has been conducted for determination of noise levels at 10 including 4 locations within the mining complex, and 9 locations outside the mine within the study area. The noise levels at each of the locations were recorded for 24 hours during the winter season. The noise levels at all the locations, residential, commercial and industrial, were found to be below the required minimum levels for both the day time and night time.

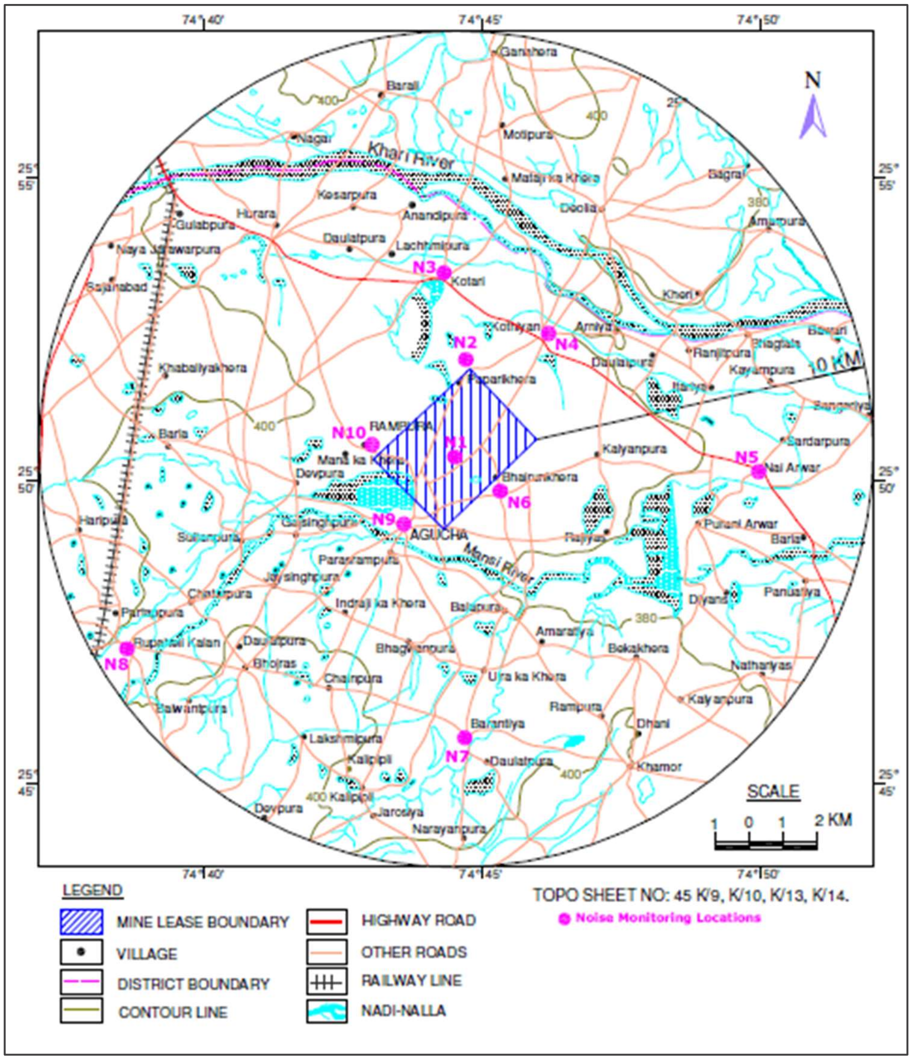


Figure 51: Location of noise monitoring stations

15.2.4 Surface Water and Wetlands

A total of 10 ground water and 3 surface water sources, consisting of bore wells, dug wells and the RA mine pit, covering a 10 km radius around the mine were collected and were examined for physico-chemical, heavy metals and bacteriological parameters in order to assess the effect of mining, industrial and other activities on surface and ground water quality. The ground water sources include two monitoring wells of HZL in the vicinity of the tailings dam of RAM. Three surface water sampling locations were selected to assess the surface water quality.

The presence of heavy metals is observed, but the concentrations are well within the permissible limits. The overall quality considerations as far as water quality in the study area indicate the absence of any external polluting sources and represent uncontaminated conditions.

The results for the parameters analysed for surface water samples and are compared with Class 'C' water quality (fit for drinking after conventional treatment) as per IS:2296-1982 "Tolerance Limits for Inland Surface Waters subject to Pollution" for surface water.

The pH of the surface water samples collected ranges in between 7.3 to 7.5. The conductivity recorded in the range of 273 to 365 gs/cm.

The sodium and potassium concentrations varied between 13.5 to 38.6 mg/l and 5.9 to 8.2 mg/l respectively. The Higher concentration of sodium and potassium was observed in - Khari River (SW1). Total hardness expressed as $CaCO_3$ ranges between 86 to 96 mg/l. The concentration of nitrate fluctuates between 0.5 to 0.9 mg/l with higher concentration of nitrate observed in Khari River (SW1).

HZL is monitoring water quality in and around the mine lease area. HZL is collecting water samples from nearby wells once in a month and analysing for physico-chemical parameters and heavy metals like Lead, Zinc, Iron and Cadmium.

The results indicate that the concentrations of lead, zinc, iron and cadmium are well within the permissible limits applicable for ground water.

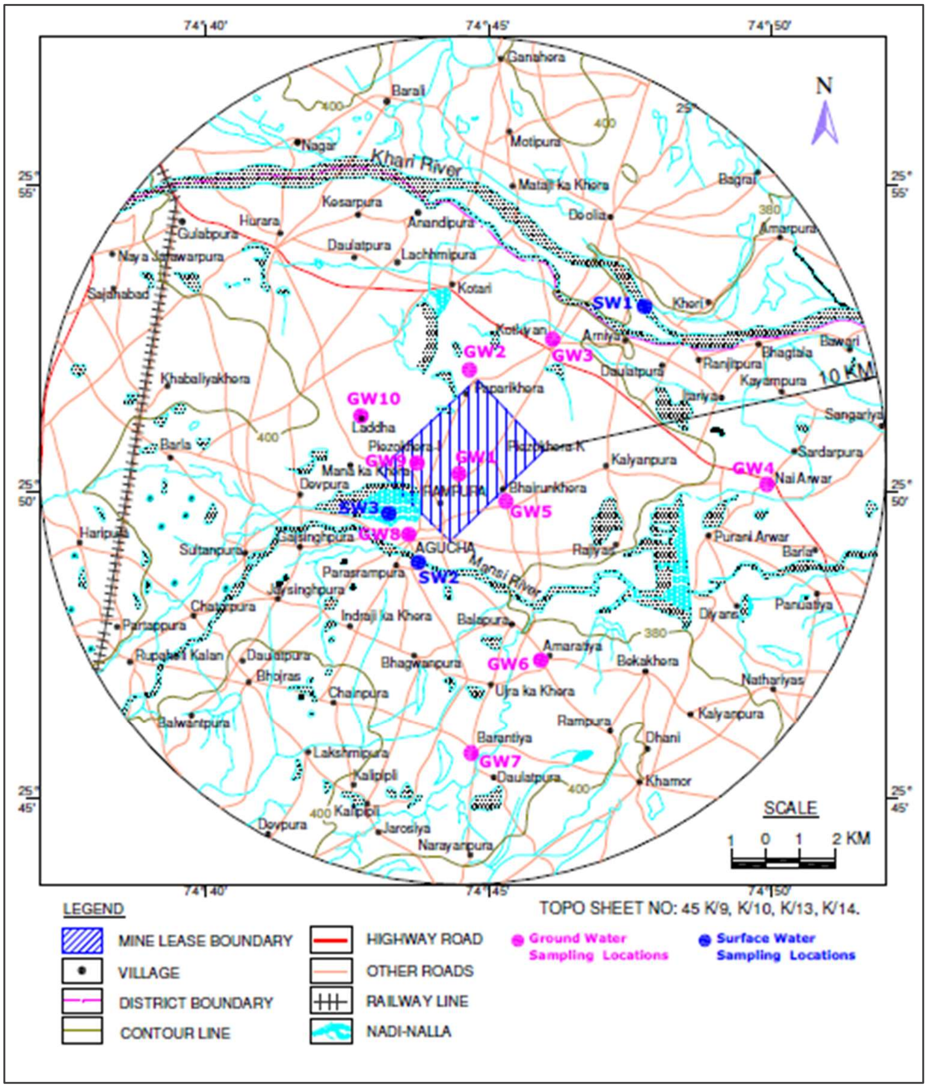


Figure 52: Location of water monitoring stations

15.2.5 Groundwater

Most of the villages in the project area have bore well and tube well facilities, as most of the residents of these villages make use of this water for drinking, agricultural, and other domestic uses. Therefore, bore well samples have been considered for sampling.

The analysis results indicate a pH range of 6.5 to 7.9, which is well within the desirable limit of 6.5 to 8.5. The TDS was observed to range between 378 to 2096 all samples are within the permissible limit of 2000 mg/l except for samples collected at one location. Total hardness was observed to range between 212 to 620 mg/l, within the permissible limit of 600 mg/l except for sample collected at three samples.

Chlorides are found to range between 38.5 to 574.3 mg/l. The chloride concentrations were observed to be within the permissible limit of 1000 mg/l. Fluorides are found to range between 0.2 to 1.9 mg/l and within the permissible limit of 1.5 mg/l except for samples collected at three locations. Nitrates are found to range between 0.7 to 60.4 mg/l and exceeding the prescribed limit of 45.0 mg/l at two locations. Presence of heavy metals is observed but the concentrations are within the permissible limits.

15.2.6 Soil

The baseline information on soils in the area is essential for assessing the impacts of current activities on the soil quality and the anticipated impacts in future after enhancement of the mining capacity. Accordingly, the assessment of the soil quality has been carried out.

Six locations within a 10 km radius of the mine lease boundary were selected for soil sampling during the winter season. At each location, soil samples were collected from 3 depths – 30 cm, 60 cm, and 90 cm. The homogenised samples were analysed for physical and chemical characteristics.

The results were compared with standard classification parameters. It has been observed that the texture of the soil is mostly clayey in the study area. The pH of the soil ranged from 7.8 – 9.1 indicating that the soils are usually moderately to very strong alkaline in nature.

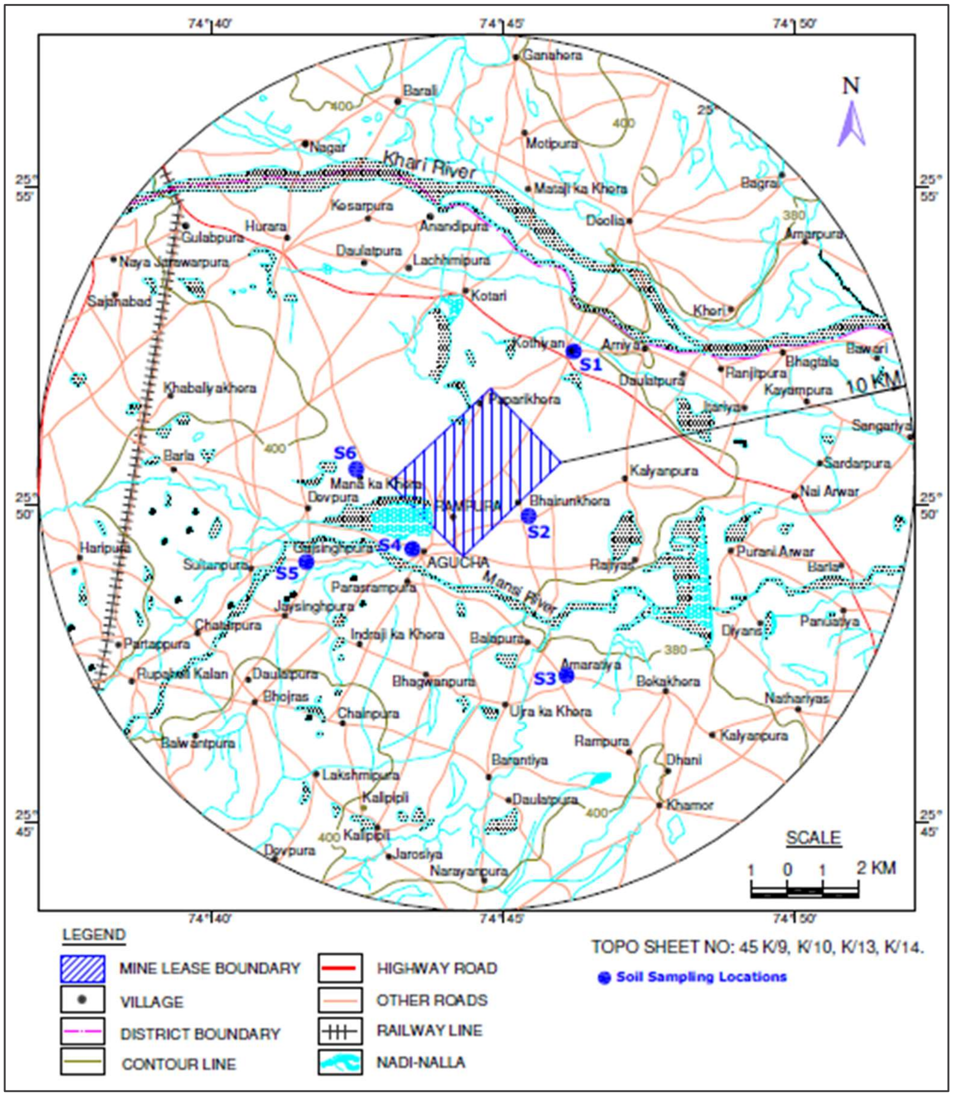


Figure 53: Location of soil monitoring stations

15.2.6.1 Data Generation

For studying soil quality in the region, sampling locations were selected to assess the existing soil conditions in and around the project area representing various land use conditions. The physical, chemical and heavy metal concentrations were determined. The samples were collected by ramming a core-cutter into the soil up to a depth of 90 cm.

The sampling locations have been identified with the following objectives:

- To determine the baseline soil characteristics of the study area;

- To determine the impact of current activities on soil characteristics; and
- To determine the impact on soils more importantly from agricultural productivity point of view.

Six locations within 10-km radius of the mine lease boundary were selected for soil sampling in winter season. At each location, soil samples were collected from three different depths viz. 30 cm, 60 cm and 90 cm below the surface and homogenized. The homogenized samples were analysed for physical and chemical characteristics. Samples were taken twice during the study period. The samples have been analysed as per the established scientific methods for physicochemical parameters. The heavy metals have been analysed by using Inductive Coupled Plasma Analyzer.

15.2.6.2 Baseline Soil Status

It has been observed that the texture of soil is mostly clayey in the study area. It has been observed that the pH of the soil ranged from 7.8 to 9.1 indicating that the soils are usually moderately to very strongly alkaline in nature.

The electrical conductivity was observed to be in the range of 150 to 1206 pS/cm, with the maximum (1206 pS/cm) observed in Bhairunkhera (S2) and with the minimum (150 pS/cm) observed in Kothiyan (S1).

The nitrogen values ranged between 17.7 to 69.8 Kg/ha. The maximum value (69.8 Kg/ha) was observed in village Agucha (S4), the minimum value (17.7 Kg/ha) was observed in Mana ka Khera village (S6) indicating that the soils have very less to less quantity of nitrogen.

The phosphorus values ranged between 116 to 222.9 Kg/ha. The maximum value (222.9 Kg/ha) was observed in Bhairunkhera village (S2), the minimum value (116 Kg/ha) was observed in Agucha (S4) indicating that the soils have more than sufficient quantity of phosphorus.

The potassium values range between 136.3 to 191.7 Kg/ha. The maximum value (191.7 Kg/ha) was observed in Agucha (S4), and the minimum value (136.3 Kg/ha) was observed in the village Mana ka Khera village (S6) indicating that the soils have less to medium quantity of potassium.

NPK values are less to more than sufficient in most of the locations. Occurrence of heavy metals is observed from all the locations.

Table 54: Soil Quality Monitored by HZL for the year 2008 (Pre Monsoon)

Sr. No.	Location	Pb (ppm)	Zn (ppm)	Fe (ppm)	Cd (ppm)
1	Jai Singh Pura	24.87	59.33	52.63	0.84
2	Daulat Pura	26	52	44.5	0.92
3	Bhillon Ka Khera	28.58	85.25	118.34	0.96
4	Kalyan Pura	19.8	82.9	61.7	1.15
5	Muna Ka Khera	18.99	42.44	48.25	1.05
6	Gaj Singh pura	19.1	55.4	73.2	0.97
7	Agucha Plantation	18.45	53.67	52.8	1.14
8	Bherukhera	17.72	54.6	62.49	1.13
9	Magzine	20.47	48.37	47.63	0.89

Values are expressed in ppm

Table 55: Soil Quality Monitored by HZL for the year 2008 (Pre Monsoon)

Sr. No.	Location	Pb (ppm)	Zn (ppm)	Fe (ppm)	Cd (ppm)
1	Jai Singh Pura	24	51.55	45.9	0.8
2	Daulat Pura	24.2	52.1	45.5	0.86
3	Bhillon Ka Khera	28.95	72.75	106.75	0.92
4	Kalyan Pura	17.75	82.25	64.56	1.1
5	Muna Ka Khera	18.95	35.4	48.82	0.92
6	Gaj Singh pura	17.2	48.65	72.75	0.8
7	Agucha Plantation	17.35	56.25	48.95	1.1
8	Bherukhera	16.65	48.5	62.58	1.2
9	Magzine	19.85	47	47.65	1.12

Values are expressed in ppm

15.2.7 Biological Environment

A detailed ecological survey of the study area was conducted, particularly with reference to the listing of species and the assessment of the existing baseline ecological conditions in the study area. The presence of no critically threatened species was recorded. There are no wildlife sanctuaries or national parks in 10 km or 25 km from the mine lease boundaries.

Among the identified birds, the Indian myna, common myna are the local migratory birds which were observed.

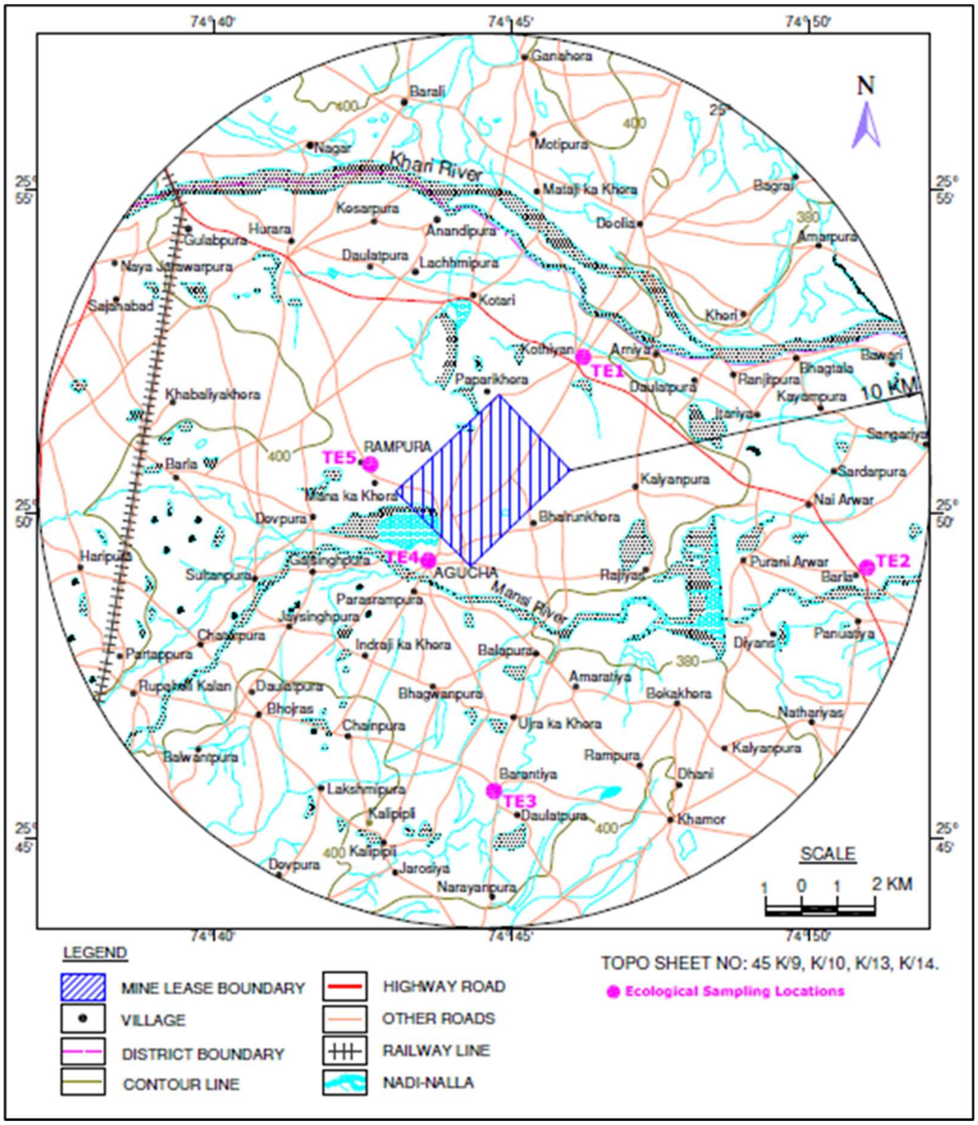


Figure 54: Ecological sampling locations

15.2.7.1 Agriculture

Agriculture is the main occupation of a majority of the population of the Bhilwara District. The major crops in district of Bhilwara are Maize, Wheat, jo, til, Urad, Moong, Jeera, Gram, Ground Nut, Raee, Mustard & Cotton are the major crops of the Bhilwara District. Kharif crops constitute the bulk of the food production in the district since the agricultural activities are mostly dependent on Monsoons. Kharif crops include urad, maize, moong, groundnut etc... Rabi crops are usually sown in November whereas Kharif crops are sown with the beginning of the first rains in July.

15.2.7.2 Forest

The forest crop is mainly xerophytic species common to the more arid tracts of India. The district is completely outside the timberline i.e. Teak, Sal and Sissoo zone of the tropical and sub-tropical India. The main species found in the district are Dhokra (Anogeissus pendula), Kumpta (Acacia rupestris), Salar (Boswellia serrata), Khejri (Prosopis spicigera), Khair(Acacia catechu), Ber (Zizyphus jujube), JinjaI(Bauhinia racemosa), Koulassi (Dichrostachys Cineria), Aranja (Acacia leucophloe), Go (Lannea grandis), Tambolia (Ehretia laevis), Sainjora (Moringa concanensis), Thor (Euphorbia nivula), Grangan (Grewia populifolia), Jharbor (Zizyphus numalarica), Dassen (Phus mysorensis), Firangan (Grewia pillosa), Salepan (Securenega oblota), Arni (Clerodendron phlomoides), Neem (Azadirachta indica), Semal (Salmalia malabarica), Pipal (Ficus relisiosa), Golia dhau (Anogeissus latifolia), Lambaba (Bridelia retusa), Timbru (Disospyros melanoxylon), Umbia (Saccopetalum tomentosum), Ghatolon (Randia dumetorum), Kalia (Albizia odaratissima), Kar(Sterculia urens), Khirni (Wrightia tomentosa), Bahera (Terminalia bellerica), Amaltas (Cassia fistula) and Bijasar (Soymida fabrifuga) etc. The most common grasses found in the district are Cenchurus ciliaris, Schima marvosus, Dichanthium annulatus, Chloris barbata, Chrysopogon maintanus and Eremonopogon foevealatus etc.

15.2.7.3 Reserve Forest

In study area, as per Forest department of Bhilwrara, there are no reserved or protected forests in study area. Six designated village forest areas belong to panchayats along the banks of Khari and Mansi Rivers

15.2.7.4 Plantations

There are a number of small plantations within the area, some augment forested area and some form part of the social forestry programmes, particularly evident at roadsides. Trees commonly used for planting include Acacia nilotica, Eucalyptus, Dalbergia sissoo, Cassia siamea and Prosopis julifera. Plantations and fields are often hedged with cactus-Euphorbia nerifolia. Hindusthan Zinc Limited area is having a good number of plants. Dalbergia sisso, Leucena leucophloe, Cassia fistula, Cassia siamea, Parkinsonia sp, Pithocolobium dulce, Pongamia glabra, Casuarina sp, Albizzia procera, Albizzia amara. As corporate policy of Hindusthan Zinc Limited, extensive green belt programme is under implementation to make greenery in and round mine lease.

15.3 Requirements and Plans for Waste and Tailings Disposal, Site Monitoring, and Water Management

15.3.1 Waste and Tailings Disposal

Tailings from all four flotation plants are mixed and thickened to 65 to 68% by mass using a deep cone thickener (“DCT”) and used as cemented paste backfill for filling underground stopes
or are pumped to the tailings storage facility (“TSF”).

Cemented paste backfill is produced in a new paste plant with a nameplate capacity of 2.5 Mtpa of solids or 800,000 m^3 of paste fill. A portion of the DCT thickener underflow is further dewatered to around 74% solids by mass in a new paste plant. Approximately 70% of the DCT underflow is dewatered to nominally 18% moisture using four disc-filters. The filter cake is mixed under controlled conditions in a twin paddle mixer with the balance of the DCT underflow and up to 12% cement to produce a 7-inch slump paste which is then pumped, by high pressure piston pumps, around 2km to four boreholes to feed the underground stopes. Typically, a 10,000 m^3 stope can be filled in around 70 hours. The paste plant and paste delivery systems incorporate a high degree of standby equipment and piping to maintain a high operational availability.

The balance of the DCT underflow is neutralised with lime prior to pumping to the TSF. Alternatively, the tailings can be thickened in the older, conventional tailings thickeners, to 45%
solids by mass, for disposal in the TSF.

Water is recovered into ponds at the DCT and the conventional thickeners and is recycled as required to the plant. Further water is also recovered from the TSF and recycled to the water ponds. The existing TSF covers around 110 km^2.

The current tailings storage facility is constructed using waste rock with the base and walls lined with clay. The dam wall has been raised to its current level of 39 m to increase the storage capacity. It is reported that the dam will hold sufficient material for the envisaged LoM at the reported maximum height of 51 m.

The TSF is currently at its 7th phase of raising, upon completion of which the dam height will be 45 m. A Phase 8, has been proposed, which will raise the TSF to 51 m have capacity for approximately 14 Mt of tailings.

HZL is currently investigating re-processing of the tailings to use available capacity at the plant and recover additional metal, as well as investigating back-filling the open-pit void with dry stacked tailings.

15.3.2 Site Monitoring

HZL is monitoring water quality in and around the mine lease area, HZL is collecting water samples from nearby wells once a month and analysing for physio-chemical parameters and heavy metals like lead, zinc iron and cadmium. Soil samples from all the selected areas are collected regularly for assay

15.3.3 Water Management

Overall water usage is approximately 3.2 m^3/t, with fresh water about 0.45 m^3/t of the required water. Fresh water is sourced from the Banas River 55 km from the RAM concentrator. There is also a constraint by the Rajasthan Government related to the maximum amount of water that can be extracted, which is currently limited to 11,200 m^3/day.

It is reported that plant water requirements are satisfied by internal mine and recycle water sources and that no additional water is required.

Process water will be supplied from the surface through the main accesses. Potable water could be supplied in plastic containers or in bulk containers delivered to a Normet carrier or integrated tool carrier. Process water is primarily required for drilling operations and dust control. See figure below for an estimate on underground mining water consumption.

A significant proportion of the water introduced for drilling and dust suppression will not require pumping because it will be removed along with the ore and waste rock as contained moisture via the materials handling system.

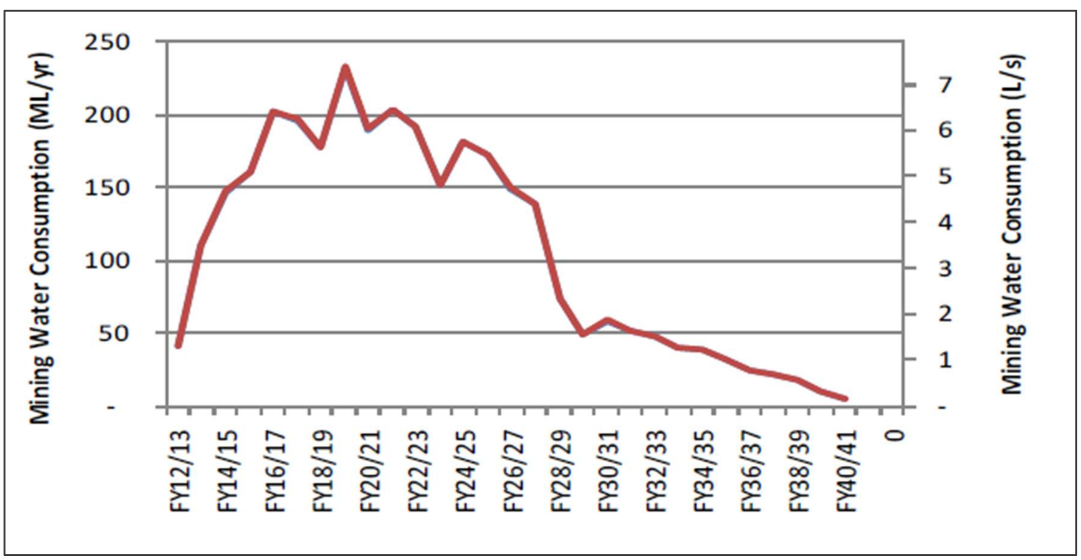


Figure 55: : RAM Mining Operations water Consumption

Rampura Agucha underground is considered to be a relatively dry mine with little ground water observed to date. By early 2013 HZL was dewatering at a rate of 7 litres per second, equivalent to approximately 600 m3/day from the +13 m level and surrounding decline development.

15.4 Required Permits and Status

15.5 Community Engagement

It is necessary to study the existing socio-economic status of the local population, which will be helpful in making efforts to further improve the quality of life in area under study. For assessing the prevailing socio-economic aspects of people in the study area around the mine, the data has been collected from various secondary sources and analysed.

Focus Group discussions were held in the villages for eliciting the general information of the study area, to support or supplement the information collected through secondary and primary sources.

According to the 2001 census, the study area has 106,424 inhabitants in 58 villages.

Different types of health facilities including one hospital, 8 primary health centres, 12 dispensaries.

The study area is served by rail and road transport facilities. All villages have paved road connections.

As a whole the study area has a moderate level communication network.

15.6 Mine Closure

The proposed mining will be carried out within the existing mine lease area, and it is already categorized under industrial area. The land use pattern of leasehold area (1200 Ha) is as under. Out of 1200 Ha only 1021.51 Ha land has been acquired, the details of acquired land is furnished below in the table:

Table 56: Mine Closure summary

Total Area Degraded					Total mined out area Reclaimed and Rehabilitated			Other Areas Reclaimed and Rehabilitated	
Total area under excavation in the lease		Area under Dumps (in hect)	Area under utility services (in hect)	Area under Stack yards (in hect)	Mined out Area Reclaimed but not rehabilitated (in hect)	Mined out Area fully Rehabilitated from Reclaimed area (in hect)	Area under Water Reservoir considered Rehabilitated (in hect)	Stabililized Waste dump Rehabilitated (in hect)	Virgin area under Green Belt (in hect)
Area under mining operation	Mined Out area in the lease								
0	180.13	314.39	526.99#	0	0	0	0	64*	-

15.6.1.1 Backfilling

No progressive mine pit reclamation is possible as steeply dipping ore body at Rampura Agucha Mine does not allow the concurrent filling of the pit and hence other suitable alternatives like fencing of the pit is done during the last plan period. To ensure the safety of the human beings and animals, stone masonry wall constructed around the perimeter of the ultimate pit limit

15.6.1.2 Water Reservoir

The water requirement for the proposed facility will be met from the existing radial well at Banas river bed located at a distance of 60-km from the mine site. There will be no additional water requirement, thus proposal for water reservoir during the planned period.

At present, the mining activity is confined to the mining lease area of 1,200 ha and there is no stream flowing in the mining area requiring any diversion. With the future mining operation being confined to the existing mining lease area, surface water resources are not affected, as it does not disturb the drainage pattern.

The surface water is not pumped or utilized anywhere in the lease area for any purpose. The rainfall collected in the mine pit during the monsoon season will be pumped out and used for green belt and the balance if any will be routed to the water reclamation pond for subsequent use in the plant.

15.6.1.3 Ground Water Recharging

The ground water assessment studies carried out in core zone (lease area) indicates that there is surplus exploitable ground water potential of 0.389 mcm as there is hardly any ground water withdrawal in the lease area. The status of ground water development in core and buffer zones being within the safe limits, i.e. the long term ground water recharge being more than the present ground water withdrawal, the static ground water reserves are safely preserved and are not depleted either due to mining activity in the core zone or due to well irrigation by the cultivators in the buffer zone.

Garland drain of 8.6 KM length already constructed. Repair and cleaning of same will be made on regular basis before onset of monsoon along the waste dump.

15.6.1.4 Topsoil Management

The soil cover in the mining area varies from 0.3 to 0.8 m with an average of 0.55 depths. The fresh breakage of ground takes place only at a very slow rate since strike length of ore body is limited and majority of which has already opened. Development / production of ore body mostly take place in depths by underground method. The topsoil, which generated in initially development phase during open pit mining operation, was used for green belt development particularly on waste dump, and also in tailing dam embankment whenever raising of tailing dam was done. All the pre-topsoil generated till date has been used for the lining of tailing dam and vegetation on matured ends of waste dump.

15.6.1.5 Tailings Dam Management

The tailing is stored in a specially constructed Tailing Impoundment Area. The tailing dam is lined with top impervious soil at the bottom and the inside of the walls with sand and top impervious soil on all the sides to avoid percolation of water to the underground. The material for construction of the tailing dam consists of waste materials generated from mine. The pipeline for tailing disposal is on three sides of the dam. Thickened tailing is pumped through a tailing line and discharged into the tailing dam, after neutralization with lime.

Part of tailings generated during the life of mine shall be used in paste formation to back fill underground mined out voids and remaining shall be accommodated in the existing tailing dam. As of now the present height of tailing dam is 60 m and further to accommodate the tailings generated from processing plant during the period would be accommodated in the tailing dam. The height of tailing dam will be raised further as per requirement, maximum up to 74 m as per latest EC approval dated 28th Feb-2020.

The method and manner of tailing disposal is same as indicated above, which is constructed for zero discharge bases. Capacity of the existing tailing dam is proposed to be increased by raising its height up to 74m to take care of the proposed treatment capacity. The height raising activity of the tailing dam will continue year on year basis and reach 74m height in 2026-27 and the capacity of the tailing dam capacity increased from 65,391,864 m3 to 73,791,864 m^3.

No progressive mine pit reclamation is possible as steeply dipping ore body at Rampura Agucha Mine does not allow the concurrent filling of the pit and hence other suitable alternatives like fencing of the pit is done during the last plan period.

To ensure the safety of the human beings and animals, stone masonry wall constructed around the perimeter of the ultimate pit limit

15.7 Adequacy of Plans

The mine operations have undergone numerous PFS studies in the last decade and has ensure they remain within the industry standard of operations for the orebodies and have a good hand on the design and production. The actuals versus the LOM plan and budget plans were not evaluated in detail to determine and reconciliate the planned schedules and plans to the actual achieved.

16 Capital and Operating Costs

16.1 Operating Cost Estimate

ABGM did not undertake any techno-economic assessments on the mining costs, processing costs and fixed royalties. The following operating cost estimates was received from HZL that is used in their financial and budget planning for FY2022 – 2033.

Table 57: Summary of Operating Costs (source HZL)

Particulars	UoM	Yr 2022	Yr2023	Yr 2024	Yr 2025	Yr 2026	Yr 2027	Yr 2028	Yr 2029	Yr 2030	Yr 2031	Yr 2032	Yr 2033
Mining Cost	$/MT Ore	34	36	38	40	42	44	46	48	51	53	56	59
Processing Costs	$/MT Ore	14	14	15	16	17	17	18	19	20	21	22	23
Development	$/MT Ore	13	14	14	15	16	17	17	18	19	20	21	22
Smelter Cost													
Zn Metal	$/MT Metal	454	454	454	454	454	454	454	454	454	454	454	454
Lead Metal	$/MT Metal	585	585	585	585	585	585	585	585	585	585	585	585
Silver Metal	$/MT Metal	24	24	24	24	24	24	24	24	24	24	24	24
Overall Cost													
Mining Cost	$ Mn	301	313	318	297	269	271	268	200	235	236	240	215
Smelter Cost	$ Mn	299	286	304	247	215	185	185	123	145	119	114	96
Royalty Cost	$ Mn	234	224	237	194	169	146	146	97	114	94	90	76

16.2 Capital Cost Estimate

No capital cost estimates was evaluated in the limited timeframe of this technical review.

17 Economic Analysis

ABGM did not undertake any techno-economic assessments on the reserves and only evaluated the mining cost and applied the high level mining cost, processing costs and fixed royalties to determine a nominal cashflow with some limited sensitivities.

17.1 Model Parameters

Table 58: Costing and Financial Inputs per annum (supplied by HZL)

Assumption	Yr 2022	Yr2023	Yr 2024	Yr 2025	Yr 2026	Yr 2027	Yr 2028	Yr 2029	Yr 2030	Yr 2031	Yr 2032	Yr 2033
Grade												
Zn (g/t)	12.19	11.83	12.95	12.14	12.54	11.39	11.96	10.94	11.61	10.10	10.03	10.01
Pb (g/t)	1.77	1.66	1.83	1.46	1.22	0.99	1.15	1.29	1.25	1.03	0.90	0.82
Silver (g/t)	62.41	62.80	60.67	47.10	40.54	31.03	32.25	37.12	36.25	34.11	28.29	28.94
Recovery-Mill												
Zn (%)	95.96	95.96	95.96	95.96	95.96	95.96	95.96	95.96	95.96	95.96	95.96	95.96
Pb (%)	94.75	94.75	94.75	94.75	94.75	94.75	94.75	94.75	94.75	94.75	94.75	94.75
Silver (%)	98.25	98.25	98.25	98.25	98.25	98.25	98.25	98.25	98.25	98.25	98.25	98.25
Recovery-Smelter												
Zn (%)	96.16	96.16	96.16	96.16	96.16	96.16	96.16	96.16	96.16	96.16	96.16	96.16
Pb (%)	97.57	97.57	97.57	97.57	97.57	97.57	97.57	97.57	97.57	97.57	97.57	97.57
Silver (%)	99.81	99.81	99.81	99.81	99.81	99.81	99.81	99.81	99.81	99.81	99.81	99.81
Impact on Cost												
Mining	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%
Zn Premium	180	180	180	180	180	180	180	180	180	180	180	180
Pb Premium	150	150	150	150	150	150	150	150	150	150	150	150
Ag Premium	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4

17.2 Taxes, Royalties, Depreciation

The normal government taxes have been applied to the models indicated below. The exact percentages were not disclosed by HZL. The royalties applied in the models are indicated in the start of document but can be seen below.

- The Royalty for Lead is 14.5% of London Metal Exchange lead metal price chargeable on the contained lead metal in the concentrate produced.
- The Royalty for Zinc is 10% of London Metal Exchange Zinc metal price on ad valorem basis chargeable on contained zinc metal in the concentrate produced.

17.3 Cashflow Forecasts and Annual Production Forecasts

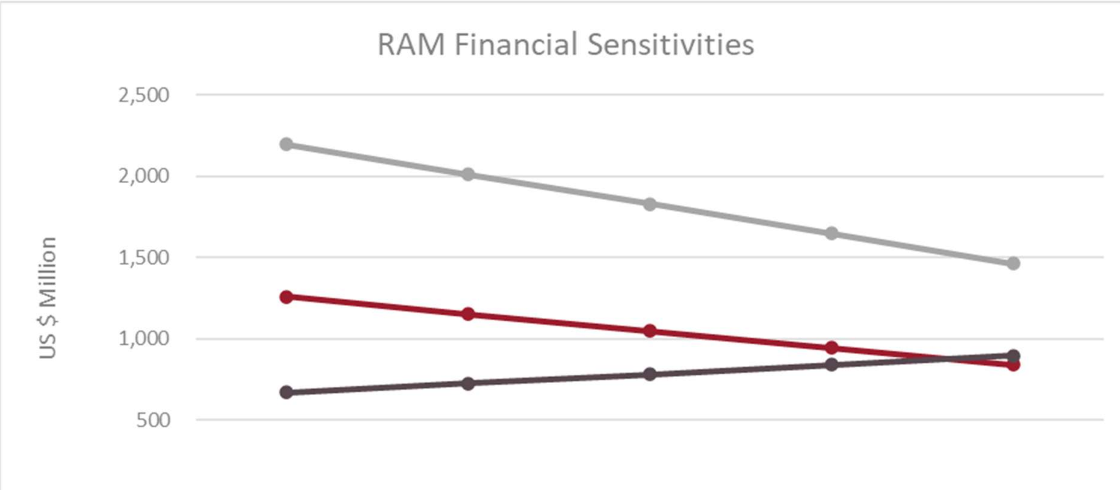
Table 59: Financial model outcome (2023 - 2033)

Particulars	UoM	Yr 2022	Yr2023	Yr 2024	Yr 2025	Yr 2026	Yr 2027	Yr 2028	Yr 2029	Yr 2030	Yr 2031	Yr 2032	Yr 2033
Ore production	MT	4,935,332	4,886,973	4,733,391	4,205,573	3,630,880	3,485,738	3,281,300	2,327,906	2,613,119	2,491,019	2,420,410	2,062,471
Zn Metal in Concentrate	MT	577,181	554,679	587,995	490,010	437,089	380,950	376,588	244,464	291,184	241,463	232,905	198,097
Lead Metal in Concentrate	MT	82,849	77,013	81,867	58,263	42,080	32,657	35,679	28,525	30,882	24,387	20,734	16,045
Silver Metal in Concentrate	KG	303	302	282	195	145	106	104	85	93	83	67	59
Zn Metal	MT	555,008	533,371	565,406	471,186	420,298	366,316	362,121	235,073	279,998	232,187	223,957	190,487
Lead Metal	MT	80,838	75,144	79,880	56,849	41,058	31,865	34,814	27,833	30,132	23,795	20,230	15,656
Silver Metal	KG	302	301	282	194	144	106	104	85	93	83	67	59
Mining Cost	$/MT Ore	34	36	38	40	42	44	46	48	51	53	56	59
Processing Costs	$/MT Ore	14	14	15	16	17	17	18	19	20	21	22	23
Development	$/MT Ore	13	14	14	15	16	17	17	18	19	20	21	22
Smelter Cost													
Zn Metal	$/MT Metal	454	454	454	454	454	454	454	454	454	454	454	454
Lead Metal	$/MT Metal	585	585	585	585	585	585	585	585	585	585	585	585
Silver Metal	$/MT Metal	24	24	24	24	24	24	24	24	24	24	24	24
Royalty													
Zinc	$/MT Metal	364	364	364	364	364	364	364	364	364	364	364	364
Lead	$/MT Metal	394	394	394	394	394	394	394	394	394	394	394	394
Silver	$/MT Metal	60	60	60	60	60	60	60	60	60	60	60	60
Zn LME	$/MT	2,759	2,759	2,759	2,759	2,759	2,759	2,759	2,759	2,759	2,759	2,759	2,759
Pb LME	$/MT	2,057	2,057	2,057	2,057	2,057	2,057	2,057	2,057	2,057	2,057	2,057	2,057
Ag LBMA	$/Troz	21.24	21.24	21.24	21.24	21.24	21.24	21.24	21.24	21.24	21.24	21.24	21.24
Ex Rate	Rs/USD	76.65	76.65	76.65	76.65	76.65	76.65	76.65	76.65	76.65	76.65	76.65	76.65
Revenues													
Zn	$ Mn	1,631	1,567	1,662	1,385	1,235	1,076	1,064	691	823	682	658	560
Pb	$ Mn	178	166	176	125	91	70	77	61	66	53	45	35
Ag	$ Mn	230	229	214	148	110	81	79	64	71	63	51	44
Mining Cost	$ Mn	301	313	318	297	269	271	268	200	235	236	240	215
Smelter Cost	$ Mn	299	286	304	247	215	185	185	123	145	119	114	96
Royalty Cost	$ Mn	234	224	237	194	169	146	146	97	114	94	90	76
Net Cashflow	**$ Mn**	**1,205**	**1,139**	**1,193**	**920**	**782**	**625**	**621**	**397**	**466**	**349**	**310**	**253**

17.4 Sensitivity Analysis

A high level sensitivity analysis is illustrated below that indicates the RAM mining operations is in good position with up to 20% variation illustrate and overall positive financial outcome.

Figure 56: RAM Financial Sensitivity Analysis (unit: Million US $)



	20%	10%	100%	-10%	-20%
Cashflow	1,257	1,152	1,048	943	838
Cost	668	725	782	839	895
Revenue	2,195	2,012	1,829	1,646	1,463

18 Adjacent Properties

There are no other mine operation located adjacent to the RAM operations.

19 Other Relevant Data and Information

There is an abundance of information and other data that was not fully reviewed by ABGM during this review process as there was not enough time to conduct thorough investigation in all the data sets and documents. However, the key aspects were addressed as far as possible to ensure the relevant data and information required for this technical review was addressed.

20 Interpretation and Conclusions

20.1 Results

The results of the technical review indicate that the property contains a Mineral Resource, and a significant portion of that Mineral Resource converts to a Mineral Reserve. The project / operation has a positive economic outcome given the data, parameters and estimates outlined in the TRS. Due to the short time allocated to conduct this review.

ABGM is satisfied that the mine operation is conducting the required annual statutory work and reviews and submits to the IBM (Indian Bureau of Mines). The mine is compliant in order to develop the data, scrutinize and report on information for the development of Mineral Resources and Reserves in conjunction with their current consultancy professionals.

20.2 Significant Risks

No significant risks was observed during the site visits and neither were any picked up in the data and information provided by HZL.

20.3 Significant Opportunities

No significant opportunities was observed during the site visits and neither were any picked up in the data and information provided by HZL. However some suggestions will be listed in the consolidated report.

21 Recommendations

21.1.1 Pricing assumptions

Comparing the market research and the current trend of commodity pricing. The assumptions used by HZL seem to be on the higher side to establish the cut-off grades. However, running the sensitivities, all the operations seem to be well established and not sensitive to volatile changes. HZL used the 10 year pricing average in their estimates, the three year LME forecast average seems to be higher than the last 3 year average.

21.1.2 Environmental, Social Studies

Where information is available, environmental studies are done to an acceptable level for the Indian authorities. In some cases exemplary work is being done, beyond what is strictly necessary. Sites are monitored on strict schedules and the regular submission of updated reports are sufficient for the permits to be renewed again. Standardisation of programmes and procedures is recommended for all operations as this will prevent some areas of not receiving the correct attention. Interaction with and contributions to the local staff and communities appears to be positive with the establishing of medical centres and schools.

21.1.3 Closure Plans

Mine closure plans are not always easy to see in the provided documents, but the plans appear to be well thought out and implemented. As the mining activities are mostly underground, the effect on the environment is limited to the dust and noise generated due to ore transport and the storage of tailings, a large portion of which is used for backfilling mined stopes.

21.1.4 Geotechnical Drill Core Logging

The mines need to continue its practise with respect to core logging. Recommend more detailed geotechnical logging and recording of structures.

21.1.5 Sampling Transportation

Even though no information was provided around this practise, the site inspections proved that core, as well as samples from core, are transported in solid, closed and locked metal containers. Logging, tagging and security of the custody chain for samples are tracked in acQuire database, with unique barcodes and numbers. However it is suggested that the procedures followed need to be documented and recorded better.

21.1.6 Independent Lab Analysis

It is recommended that regular pulp duplicates are submitted an independent laboratory to be analysed as umpire samples.

21.1.7 Bulk Density Data

While bulk density data has been reconciled with production data, independent bulk density analyses should be undertaken at an independent laboratory during the exploration phase to avoid errors.

22 References

- 31161 HZL Audit 2021_Report_Finalr
- HZL_RR Audit presentation RAMine_FY22-V2
- Business Plan FY23_RAM
- RAM LOM Plan Summary
- RAM summary - tonnes check

EIA details.zip	2022/07/22 14:36	WinZip File	4,140 KB
RAM-General Information.docx	2022/07/21 11:38	Microsoft Word D...	60 KB
ME_to_HZL_Lease_Validity_extn_upto_12032030.pdf	2022/07/15 11:03	Adobe Acrobat D...	235 KB
RAM-PMCP.docx	2022/07/15 11:03	Microsoft Word D...	62 KB
RAM_-_LOM_geotechnical_study_-_Extract.pdf	2022/07/14 13:58	Adobe Acrobat D...	1,203 KB
RAM-Geotechnical_Overview.pdf	2022/07/14 13:58	Adobe Acrobat D...	463 KB
Geology_of_rampura_agucha.docx	2022/07/07 11:09	Microsoft Word D...	36 KB
RAUG LOMP Prim Sec - REPORT.pdf	2022/07/05 09:59	Adobe Acrobat D...	8,655 KB
RAM-Mining_Method.docx	2022/06/29 12:45	Microsoft Word D...	1,001 KB
Mine_Meetings_and_Questions_-RA_Mine_Update.xlsx	2022/06/29 12:45	Microsoft Excel W...	98 KB
RAUG_Hydrogeological_Report_RPS_Feb2017_FINAL.pdf	2022/06/29 12:44	Adobe Acrobat D...	757 KB
RAM-_Mineral_Beneficiation_Process.docx	2022/06/29 12:44	Microsoft Word D...	564 KB
EIA details	2022/07/24 21:04	File folder	

Figure 57: Files & Folders List - HZL Data and Information Pack – RAM

CROWN_3_pt.dm	2022/04/08 21:50	DM File	12 KB
CROWN_3_TR.dm	2022/04/08 21:50	DM File	88 KB
DEV15pt.dm	2022/04/08 21:50	DM File	38,916 KB
DEV15TR.dm	2022/04/08 21:50	DM File	156,844 KB
dev22pt.dm	2022/04/08 21:52	DM File	10,056 KB
dev22tr.dm	2022/04/08 21:52	DM File	88,712 KB
dmstusub.dat	2022/04/01 11:52	DAT	0 KB
ENDPIT_pt.dm	2022/04/08 21:50	DM File	1,132 KB
ENDPIT_tr.dm	2022/04/08 21:50	DM File	6,220 KB
LOMPILLAR_22v2pt.dm	2022/04/08 22:01	DM File	48 KB
LOMPILLAR_22v2tr.dm	2022/04/08 22:01	DM File	252 KB
meas22pt.dm	2022/04/08 22:02	DM File	8 KB
meas22tr.dm	2022/04/08 22:02	DM File	16 KB
RA_22_CLASS_SRK_V5.mac	2022/04/08 17:28	MAC File	19 KB
RA22FKMOD_2.dm	2022/03/12 17:57	DM File	127,784 KB
ra22fkmod_2b.dm	2022/04/08 21:50	DM File	98,984 KB
ra22fkmod_2cl.dm	2022/04/08 22:03	DM File	231,484 KB
ra22fkmod_2cl_r.csv	2022/04/08 22:03	Microsoft Excel C...	5 KB
ra22fkmod_2cl_r.dm	2022/04/08 22:03	DM File	12 KB
ra22fkmod_2cl_report.xlsx	2022/07/07 16:31	Microsoft Excel W...	17 KB
RAM_ABGM-Reblock.mdl	2022/06/30 14:11	MDL File	13,034 KB
RAM22.rmproj	2022/04/08 22:04	RMPROJ File	119 KB
STP15Bpt.dm	2022/04/08 21:52	DM File	8 KB
STP15Btr.dm	2022/04/08 21:52	DM File	8 KB
stp22pt.dm	2022/04/08 21:52	DM File	16 KB
stp22tr.dm	2022/04/08 21:52	DM File	72 KB
TongradLog.txt	2022/04/08 22:03	Text Document	77 KB
WF_BU_22V2pt.dm	2022/03/19 13:53	DM File	168 KB
WF_BU_22V2tr.dm	2022/03/21 13:51	DM File	540 KB
wf_bu_22v22pt.dm	2022/04/08 21:52	DM File	168 KB
wf_bu_22v22tr.dm	2022/04/08 21:52	DM File	540 KB
WF_TD_22V4pt.dm	2022/03/21 13:56	DM File	3,460 KB
WF_TD_22V4tr.dm	2022/03/21 13:56	DM File	10,752 KB
wf_td_22v42pt.dm	2022/04/08 21:52	DM File	3,460 KB
wf_td_22v42tr.dm	2022/04/08 21:52	DM File	10,752 KB

Figure 58: HZL Data Pack - Resource to Reserve Estimation (DataMine Files)