ASX Announcement

ASX: POL

Polymetals

16 October 2023

ENDEAVOR MINE RESTART STUDY DEMONSTRATES ROBUST FINANCIAL RETURNS

Further profitable life ahead for the Endeavor Silver-Zinc-Lead Mine

HIGHLIGHTS[1]

- **Updated Endeavor Mine Ore Reserve**
- **Initial 10-year mine life with significant growth potential**
- **LOM Free Cashflow of $323M, Pre-tax NPV$_8$ of $201M and IRR of 91%**
- **Low Pre-Production Capex of $23.7M, Maximum cash drawdown of $37.8M**
- **Project Revenues of $1,412M (US$2,750/t Zn, US$2,200/t Pb & US$23/oz Ag)**
- **Project EBITDA of $400M at an average margin of 28.5% p.a.**
- **First Concentrate production targeted for H2 2024**

Polymetals Resources Ltd (ASX: **POL**) (**Polymetals** or the **Company**) is pleased to announce the outcomes of its Endeavor Mine Restart Study (MRS) which demonstrates strong technical and robust economic support to recommence Silver, Zinc and Lead concentrate production at the Mine.

Polymetals Executive Chairman Dave Sproule commented:

"The delivery of the Endeavor Mine Restart Study is the culmination of an immense body of work completed to a high level of confidence that can support a positive investment decision.

Bringing silver back into the revenue stream via our reset of the historic 100% silver streaming royalty has unlocked significant value at the Endeavor Mine.

The MRS shows a mining operation that makes swift payback of capital because of high operating margins and the restart nature of activity. We are well advanced in our negotiations to replace the Rehabilitation Bond[2] which will complete Polymetals acquisition of the Project, as well as a finance facility to ensure coverage for peak negative cash drawdown.

[1] Refer Appended Mine Restart Study
[2] Refer ASX announcement – "Replacement of $28M Endeavor Rehabilitation Bond" dated 14th August 2023

I am buoyed by the growth potential inherent in the project, given the economic qualities of the mine outlined in the MRS, and the potential to expand the mineral resource of the project since the mine remains open to depth and there are many targets in the mining lease that remain untested.

In the past year, Polymetals share price appreciation ranks the Company in the top 5% of the 817 ASX listed Materials Companies during the period[3] and the Board, and Management are proud and excited to move back to our roots as producers and to advance the exploration targets identified.

To unlock Endeavor's embedded value, Polymetals is laser focussed on recommencement of mining."

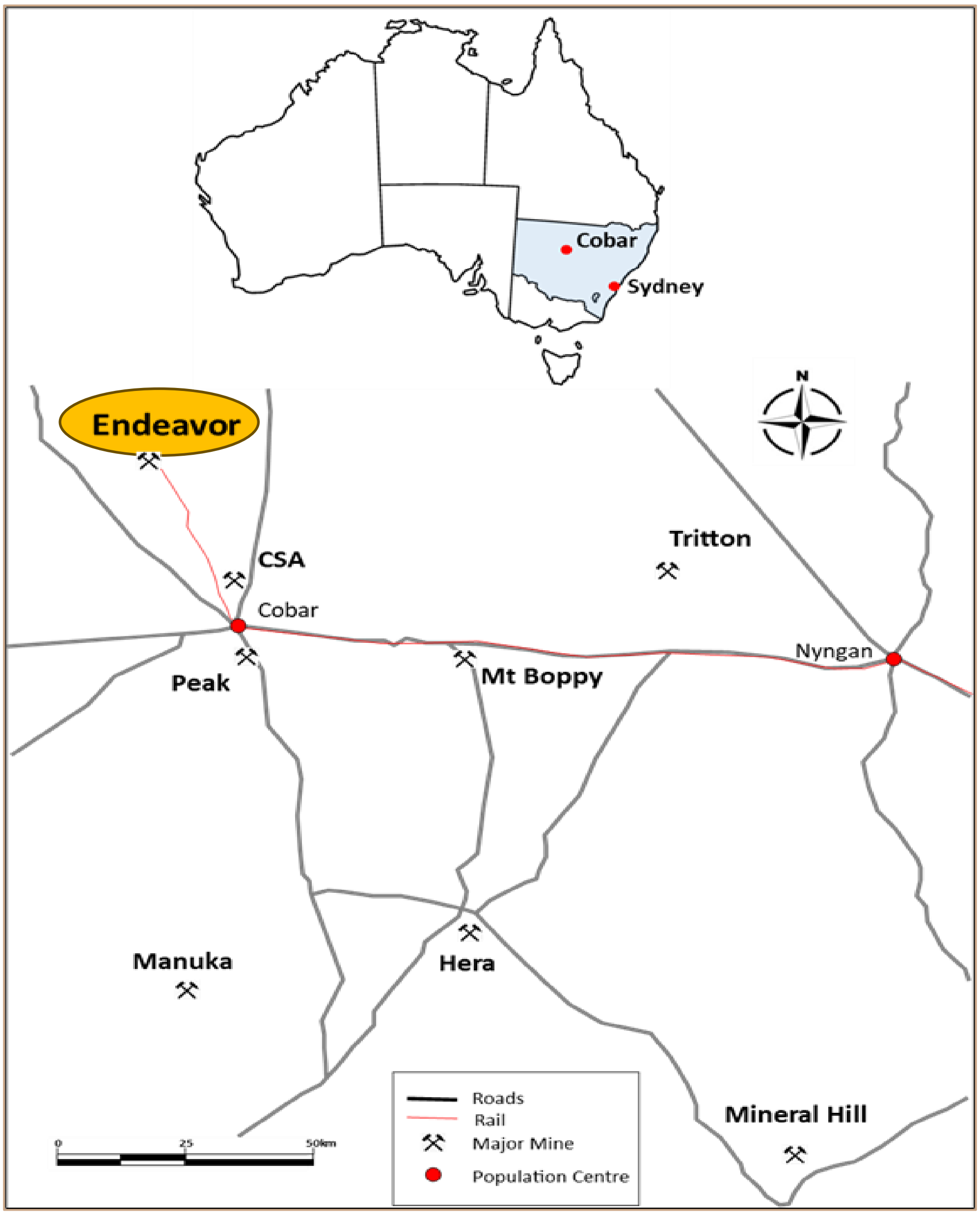


Figure 1: Endeavor Project Location and Nearby Mines

[3] Source – www.Marketindex.com.au (13th October 2023)

Mine Plan

The Mine Restart Study is based on a mine plan and optimised mining schedule that includes initial mining from three zones in the underground mine (*Figure 2*) as well as the later retreatment of high-grade Sector 1 tailings that were produced during the early years of production which commenced in 1983. The project benefits from the ability to utilise the extensive existing high-quality and well maintained underground and surface infrastructure.[4]

The MRS has estimated total ore mined and processed of 8.4 Mt over an initial mine life of 10 years. Mining of underground ore extends from Years 1 to 6 and re-treatment of Sector 1 tailings commences in Year 5 (*Figure 3*). Underground mining is scheduled to commence within 8 months of a project restart decision with concentrate production 2 months thereafter.

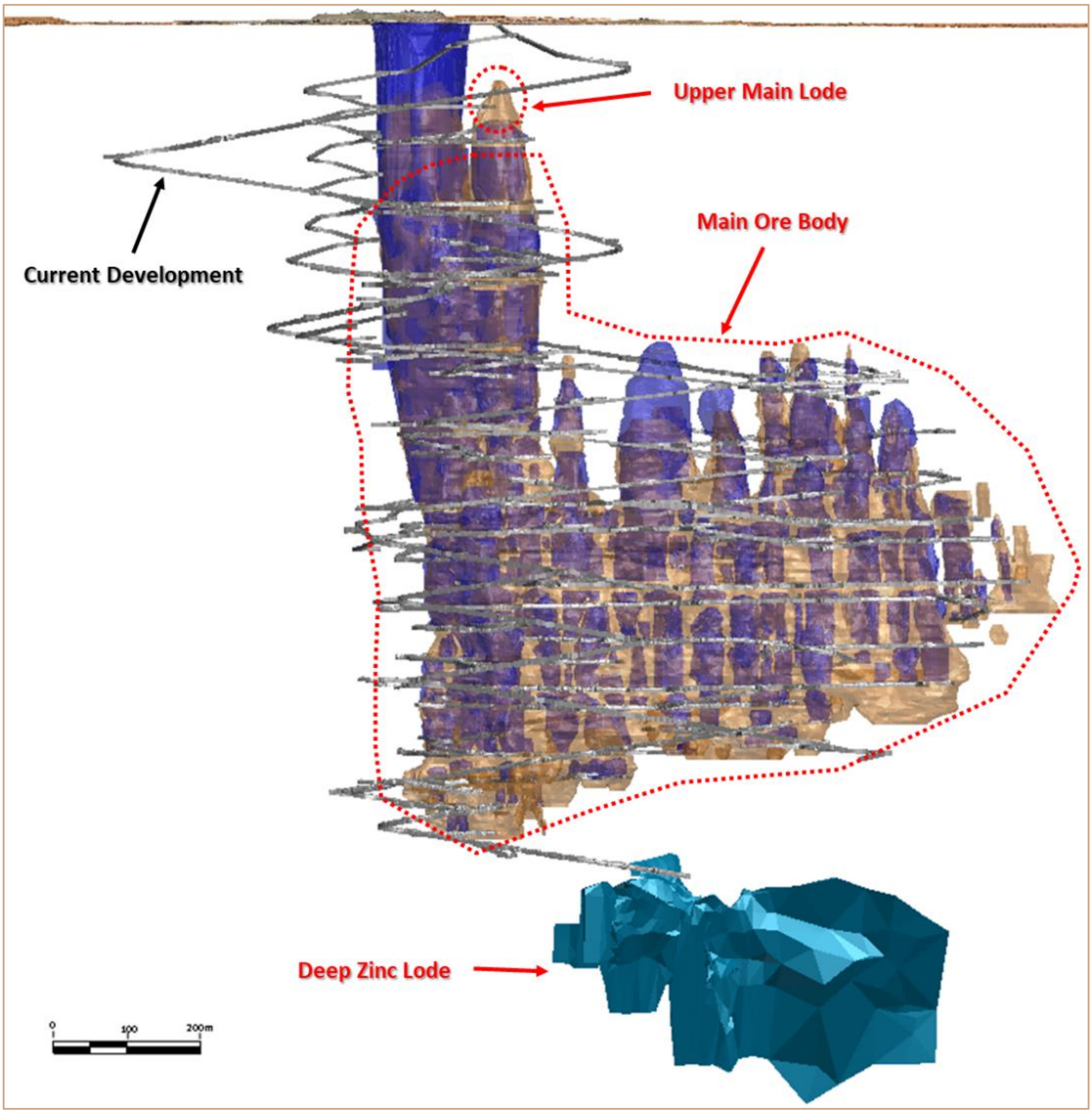


Figure 2: Endeavor Underground Mining Areas

[4] Refer ASX announcement – "Endeavor Mine Acquisition Final" dated 28th March 2023

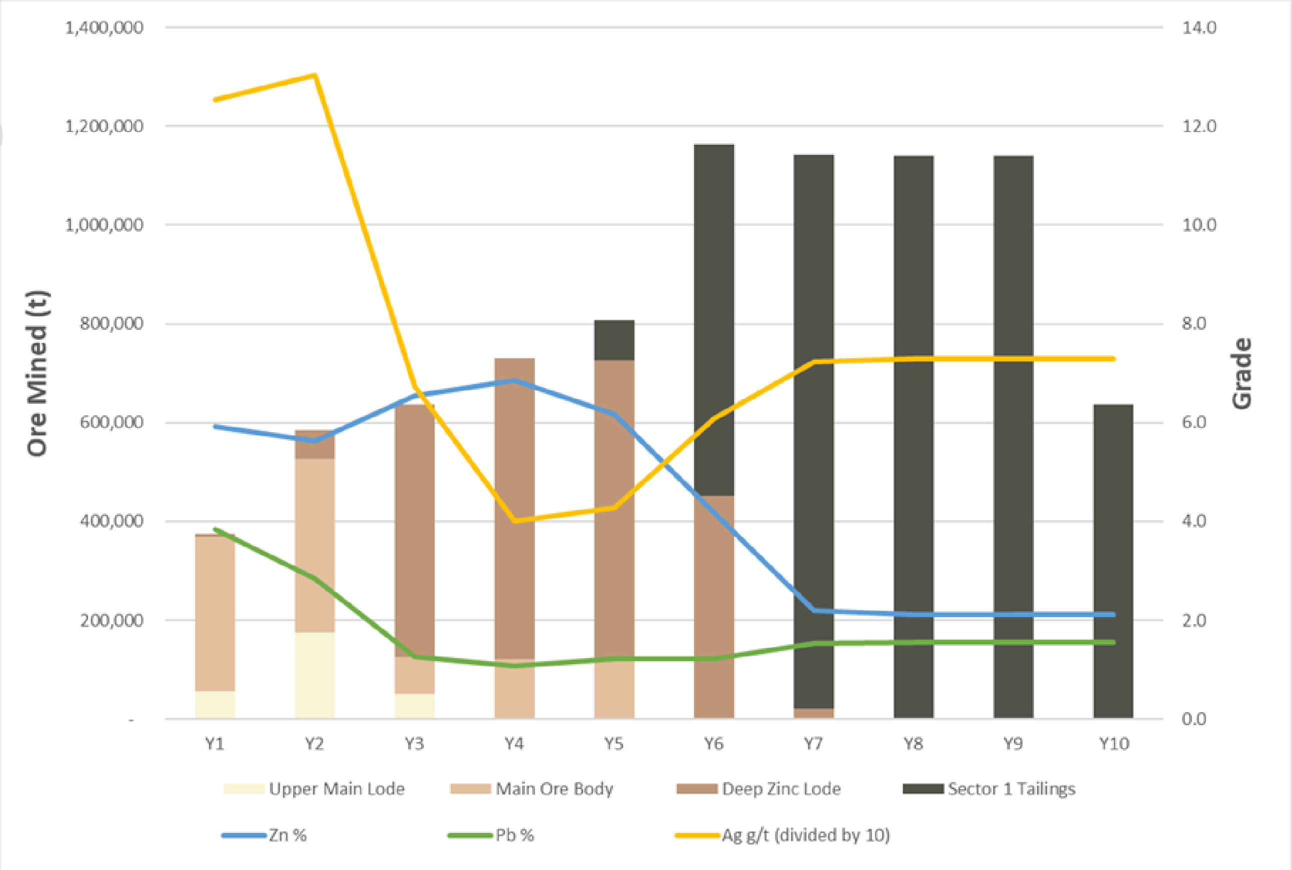


Figure 3: LOM Schedule Tonnes and Grade

Key project outcomes are summarised by Table 1 for the initial 10-year project life.

Table 1: Key Metrics

Item	Unit	Value
Physicals		
Ore Processed	Mt	8.36
Initial Project Life	Years	10
Average annual Processing Rate	tpa	840,000
Payable Zinc	t	210,000
Payable Lead	t	62,000
Payable Silver	oz	9,757,067
Financials		
Project Revenue	**A$**	**1,411,899,621**
EBITDA	A$	400,463,438
Net Present Value @ 8% discount (Pre-tax)	**A$**	**201,022,552**
Internal Rate of Return (Pre-tax)	**%**	**91%**
Pre-Production Capital	A$	23,733,607
Payback	Years	2.3

Table 2 – Endeavor Production Schedule Tonnes & Grade (Mine Plan)

Source	Ore Tonnes Mined	% Measured and Indicated	Zn %	Pb %	Ag g/t
Upper Main Lode	281,575	97%	5.63	4.40	364
Main Ore Body	975,722	85%	5.63	3.30	59
Deep Zinc Lode	2,270,271	53%	7.01	0.64	37
Tailings	4,833,413	73%	2.12	1.55	79
Total	**8,360,981**	**70%**			

The MRS has determined total material to be mined which includes Measured, Indicated and Inferred Mineral Resources. The production schedule (Table 2) contains 30% from the Inferred Mineral Resource category primarily from Tailings and Deep Zinc Lode. There is a low level of geological confidence associated with inferred mineral resources and there is no certainty that further exploration work will result in the determination of indicated mineral resources or that the production target will be realised.

The company notes that the project forecasts a positive financial performance and is therefore satisfied that the use of Inferred Resources in Production Target reporting and forecast financial information is not the determining factor in overall project viability and that it is reasonable to report the Life of Mine (LOM) Plan with Inferred Resources. It is to be noted that the Deep Zinc Lode contribution (one of 3 underground ore sources) only comes on stream in Year 3 and Tailings in Year 5. The Mine Plan includes capital for in-fill drilling with the objective of upgrading the resource confidence in these two areas.

The positive outcomes of the MRS have enabled Polymetals to generate an Ore Reserve of 5.6 Mt (Table 3), compiled from the Measured and Indicated Mineral Resources in the mine plan. The estimated Ore Reserves and Mineral Resource underpinning the Base Case Production Target have been prepared by a Competent Person in accordance with the requirements in the JORC Code.

*Table 3 – Endeavor Mine Ore Reserve Summary September 2023**

Category	Source	Mt	Zinc (%)	Lead (%)	Silver (g/t)
Proved	Underground	0.49	6.11	3.90	132
Probable	Underground	1.7	7.17	1.64	60
	Sector 1 Tailings	3.4	2.14	1.56	80
Total Proved and Probable Reserves		**5.6**	**4.04**	**1.79**	**78**

**Discrepancies may occur due to rounding. NOTE: Refer to MRS for JORC Code Compliance Statements.*

Processing

Internal and third-party reviews of the previous metallurgical performance of the Endeavor Processing Plant as well as historic and recent metallurgical test work has validated forecast estimates of metal recoveries from the different ore sources.

The existing Endeavor mill and flotation circuit has a nameplate capacity of 1.2 Mtpa. Underground ore is planned to be mined and processed at an average rate of 600,000 tpa with reprocessing of the high-grade Sector 1 tailings at the rate of 1.2 Mtpa. Zinc and silver-lead concentrates will be loaded into containers on rail wagons utilising the site's dedicated rail siding for transport to market.

Figure 4: Endeavor Mine Processing Plant

Financial Analysis

A financial analysis of the Project was carried out using outputs from the LOM scheduling process, capital and operating cost estimates, various industry standard assumptions, historic operating parameters and first principles generated costs. An owner / operator mining model was developed, with a gradual ramping up of personnel numbers over the first 12 months to match the production profile of the mine. First principles mining costs were generated using up to date quotations for consumables, and the supply and maintenance of mobile plant. These costs have been validated by an independent third party.

Table 4 summarises the input metal prices, and exchange rate assumptions which the Company has applied to the MRS financial model.

Table 4: Metal Price and Exchange Rate Inputs

Input	Unit	Value
Zinc Price	US$t	2,750
Lead Price	US$t	2,200
Silver Price	US$oz	23.00
Exchange rate	AUD:USD	0.67

Given current market uncertainty, Polymetals has taken a real metal price and exchange rate approach to its Endeavor Restart Study. *Figure 5* presents actual 10-year historic prices[5] and exchange rates with Polymetals input assumptions noted against the trend line for each key input variable (listed in Table 4). The Company believes this is a sensible and cautious approach, understanding, in particular, the cyclical nature of metal prices.

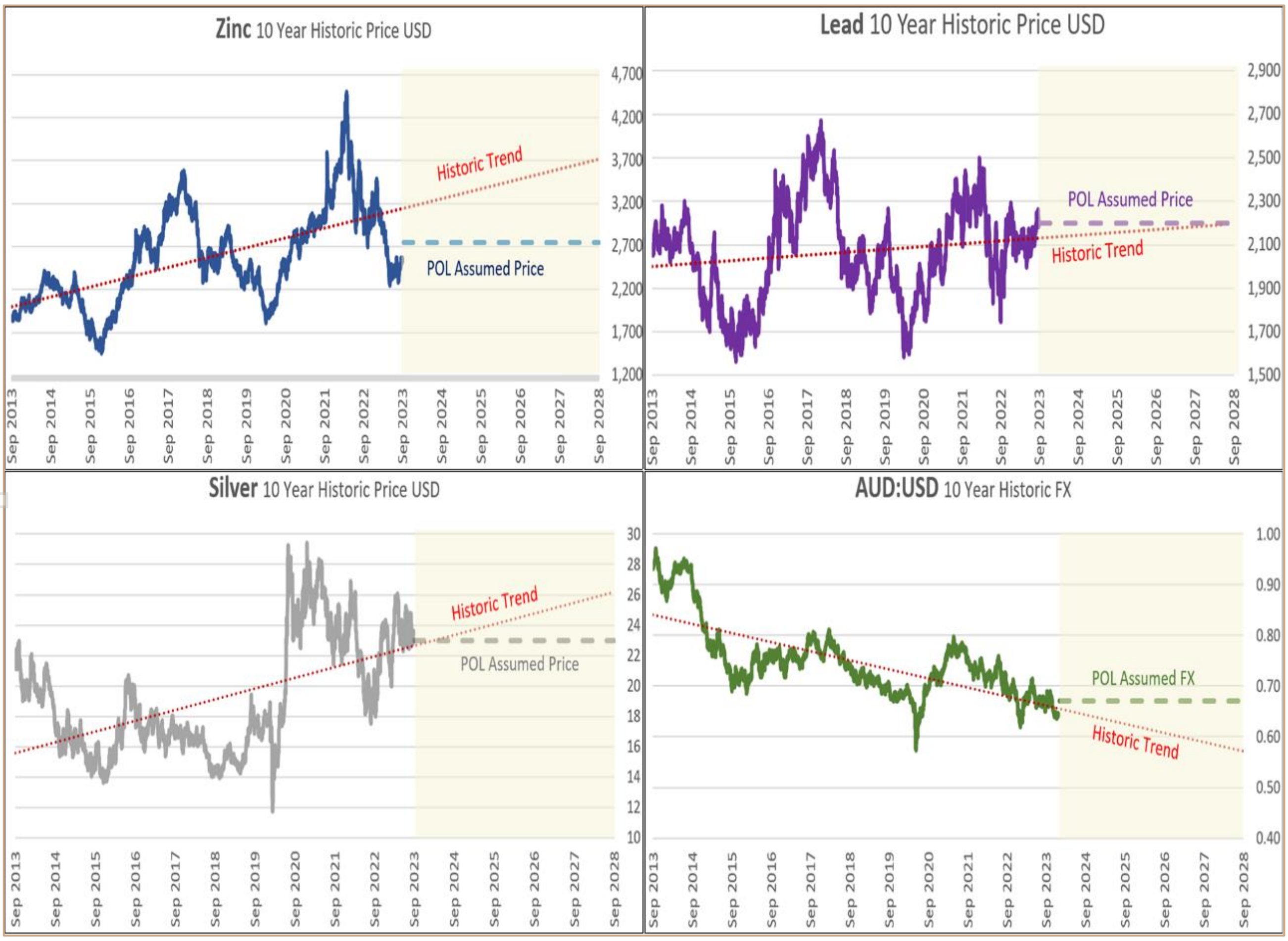


Figure 5: Historic Metal Price, AUD:USD & POL (Polymetals) Price Assumptions

[5] Source – www.Investing.com (13th October 2023)

Table 5 presents two NPV scenarios which compares current spot metal prices and exchange rate to a 10% increase in metal price. Polymetals MRS NPV_8 which applies price and exchange rate assumptions noted in Table 4 and summarised in Table 1, falls within this range, reinforcing the real approach to price and exchange rate inputs adopted by the Company.

Table 5: NPV Scenarios	**Spot Metal Price***	**Spot Metal Price +10%***
Metals Price Inputs	US$2,442/t Zinc, US$2,117/t Lead, US$22.00/oz Silver	US$2,686/t Zinc, US$2,328/t Lead, US$24.20/oz Silver
Pre-Tax NPV @ 8% discount	A$162 million	A$250 million
IRR (Pre-Tax)	73%	118%
Net Cashflow	A$268 million	A$393 million

*** Source: Market Index 10:56am 13/10/2023 and quoted AUD:USD = 0.6319**

Sensitivity Analysis

Sensitivity analysis (*Figure 6*) illustrates that the project is resilient to changes in capital and most sensitive to the AUD:USD exchange rate. It also shows the potential upside if realised zinc and silver prices exceed those assumed.

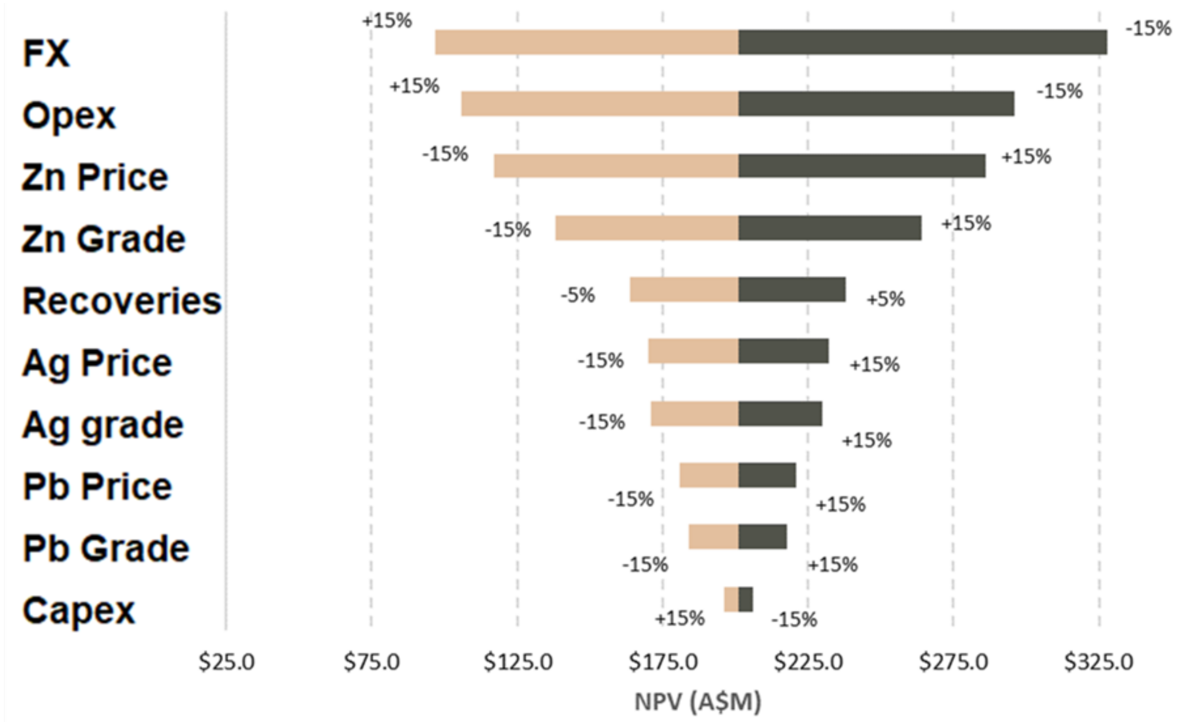
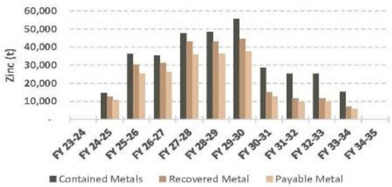
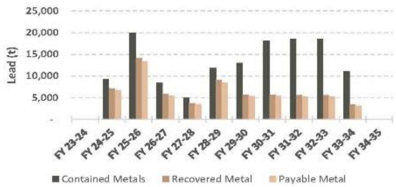


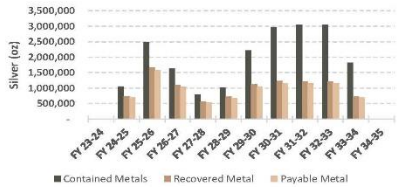
Figure 6: Project NPV Analysis

Figure 7: Financial Model Summary

Production Physicals														
Year End		**FY 23-24**	**FY 24-25**	**FY 25-26**	**FY 26-27**	**FY 27-28**	**FY 28-29**	**FY 29-30**	**FY 30-31**	**FY 31-32**	**FY 32-33**	**FY 33-34**	**FY 34-35**	**Total**
Production														
Upper Main Lode	t	-	14,282	180,117	87,176	-	-	-	-	-	-	-	-	**281,575**
Main Ore Body	t	-	222,129	443,100	63,939	24,800	221,753	-	-	-	-	-	-	**975,722**
Deep Zinc Lode	t	-	5,298	25,879	404,156	663,677	518,723	586,385	66,153	-	-	-	-	**2,270,271**
Sector 1 Tailings	t	-	-	-	-	-	-	579,759	1,135,095	1,201,248	1,201,248	716,062	-	**4,833,413**
	t	**-**	**241,710**	**649,096**	**555,271**	**688,477**	**740,476**	**1,166,144**	**1,201,248**	**1,201,248**	**1,201,248**	**716,062**	**-**	**8,360,980**
Contained Metals														
Zinc	t	-	14,535	36,206	35,378	47,785	48,287	55,602	28,462	25,466	25,466	15,254	-	**332,441**
Lead	t	-	9,330	19,949	8,411	4,913	11,841	12,991	18,113	18,619	18,619	11,153	-	**133,939**
Silver	oz	-	1,058,310	2,485,394	1,619,714	803,717	1,017,617	2,221,760	2,980,121	3,051,061	3,051,061	1,827,593	-	**20,116,347**
Concentrate														
Zinc	dmt	-	24,988	60,330	62,243	85,682	85,884	89,077	30,001	23,368	23,368	13,998	-	**498,939**
Lead	dmt	-	14,069	28,056	11,522	7,310	17,892	11,263	11,188	11,009	11,009	6,594	-	**129,911**
Recovered Metal														
Zinc	t	-	12,526	30,243	31,203	42,952	43,054	44,654	15,040	11,715	11,715	7,017	-	**250,118**
Lead	t	-	7,139	14,236	5,846	3,709	9,078	5,715	5,677	5,586	5,586	3,346	-	**65,917**
Silver	oz	-	744,011	1,663,461	1,089,361	562,951	716,327	1,115,139	1,222,163	1,220,424	1,220,424	731,037	-	**10,285,297**
Payable Metal														
Zinc	t	-	10,527	25,417	26,223	36,098	36,183	37,528	12,639	9,845	9,845	5,897	-	**210,203**
Lead	t	-	6,717	13,394	5,501	3,490	8,541	5,377	5,341	5,256	5,256	3,148	-	**62,019**
Silver	oz	-	703,001	1,572,906	1,034,893	534,803	677,736	1,059,382	1,161,054	1,159,403	1,159,403	694,485	-	**9,757,067**







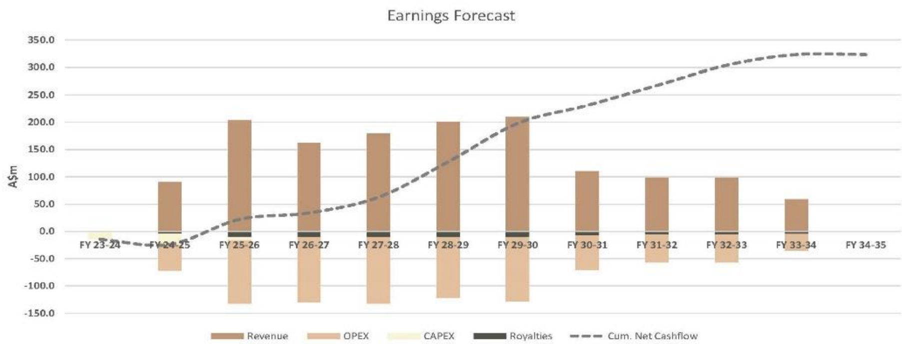
Earnings Forecast														
Year End		**FY 23-24**	**FY 24-25**	**FY 25-26**	**FY 26-27**	**FY 27-28**	**FY 28-29**	**FY 29-30**	**FY 30-31**	**FY 31-32**	**FY 32-33**	**FY 33-34**	**FY 34-35**	**Total**
Metal Revenue														
Zinc	A$m	-	43.53	105.11	108.44	149.28	149.63	155.19	52.27	40.71	40.71	24.39	-	**869.26**
Lead	A$m	-	22.22	44.31	18.20	11.54	28.26	17.79	17.67	17.39	17.39	10.41	-	**205.18**
Silver	A$m	-	24.31	54.40	35.79	18.50	23.44	36.64	40.16	40.10	40.10	24.02	-	**337.46**
Revenue	A$m	-	90.07	203.82	162.43	179.32	201.33	209.62	110.10	98.20	98.20	58.82	-	**1,411.90**
OPEX	A$m	(1.18)	(71.75)	(132.26)	(130.15)	(132.72)	(122.57)	(128.04)	(70.51)	(56.83)	(56.83)	(35.09)	-	**(937.93)**
CAPEX	A$m	(13.65)	(24.42)	(16.05)	(10.76)	(8.64)	(3.45)	(0.26)	-	-	-	-	-	**(77.23)**
Royalties	A$m	-	(3.07)	(9.66)	(9.63)	(9.11)	(10.58)	(10.74)	(6.67)	(4.95)	(4.95)	(4.16)	-	**(73.51)**
Net Cashflow	A$m	(14.82)	(9.17)	45.85	11.90	28.86	64.72	70.58	32.91	36.42	36.42	19.57	-	**323.23**
Cum. Net Cashflow	A$m	(14.82)	(24.00)	21.85	33.75	62.61	127.33	197.91	230.82	267.24	303.66	323.23	323.23	**323.23**

Summary														
Year End		**FY 23-24**	**FY 24-25**	**FY 25-26**	**FY 26-27**	**FY 27-28**	**FY 28-29**	**FY 29-30**	**FY 30-31**	**FY 31-32**	**FY 32-33**	**FY 33-34**	**FY 34-35**	**Total**
EBITDA	A$m	(1.18)	15.25	61.90	22.66	37.50	68.17	70.84	32.91	36.42	36.42	19.57	-	**400.46**
EBITDA Margin %	%	-	16.9%	30.4%	13.9%	20.9%	33.9%	33.8%	29.9%	37.1%	37.1%	33.3%	-	**28.4%**
EBIT	A$m	(1.18)	9.24	51.60	12.36	27.20	57.88	60.54	22.61	26.98	36.42	19.57	-	**323.23**
NPAT	A$m	(1.18)	9.24	36.47	8.88	19.04	40.51	42.38	15.83	18.89	25.50	13.45	-	**229.01**

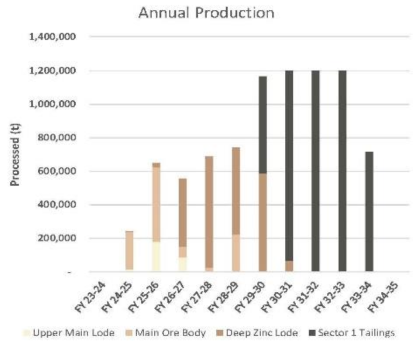
Key Outputs

Cashflow		
Revenue	A$m	1,411.90
OPEX	A$m	937.93
Capex	A$m	77.23
Royalties	A$m	73.51
Net Cashflow	A$m	323.23

Valuation		
Net Operating Cashflows (pre tax)		
Discount Rate	%	8%
NPV	A$m	201.0
IRR	%	91%
Net Operating Cashflows (post tax)		
Discount Rate	%	8%
NPV	A$m	143.6
IRR	%	78%
MAX Cash Negative	A$m	(37.79)
	Date	Jan-25
Payback	month	27
Capital Efficiency		8.5 x
Average Annual EBITDA	A$m	40.16



Upper Main Lode			
	Ore	t	281,575
	Waste	t	106,245
	Contained Metals		
	Zinc	t	15,860
	Lead	t	12,380
	Silver	g	102,411,983
		oz	3,292,622
	Grade		
	Zinc	%	5.63%
	Lead	%	4.40%
	Silver	g/t	363.71
Main Ore Body			
	Ore	t	975,722
	Waste	t	85,051
	Contained Metals		
	Zinc	t	54,896
	Lead	t	32,177
	Silver	g	57,504,000
		oz	1,848,797
	Grade		
	Zinc	%	5.63%
	Lead	%	3.30%
	Silver	g/t	58.93
Deep Zinc Lode			
	Ore	t	2,270,271
	Waste	t	733,491
	Contained Metals		
	Zinc	t	159,217
	Lead	t	14,465
	Silver	g	83,932,714
		oz	2,698,499
	Grade		
	Zinc	%	7.01%
	Lead	%	0.64%
	Silver	g/t	36.97
Tailings			
	Ore	t	4,833,413
	Waste	t	-
	Contained Metals		
	Zinc	t	102,468
	Lead	t	74,918
	Silver	g	381,839,627
		oz	12,276,429
	Grade		
	Zinc	%	2.12%
	Lead	%	1.55%
	Silver	g/t	79.00



Opportunities

Additions to Ore Reserves

There are a number of areas within the underground mine that have the potential to add significant tonnes to the currently identified Ore Reserves and extend the project life.

- **Pillar Recovery of the Upper Main Lode** – When mining commences in the Upper Main Lode, the ground conditions will be assessed, and if found to be favourable, approximately 77,000 t of high-grade ore could be recovered by pillar extraction.
- **Extension of the Deep Zinc Lode** – The Deep Zinc Lode mineralisation remains open along strike and down dip. The development of a dedicated diamond drilling platform has been included in the mine plan and capital cost estimates. This platform will allow for infill drilling of the current known ore body as well as drilling for extensions.
- **New Northwestern Pods** – The existence of further mineralisation northwest of the currently defined pods remains poorly tested. Drill intersections contain mineralisation grades similar to those in the northern pods. The development of a dedicated diamond drilling platform has been included in the mine plan as capital to test this area.

Improved Precious Metals Recovery

There is an opportunity to further investigate potential gold and silver recovery via cyanidation of supergene ore and Sector 1 tailings. An option is to store tailings from high grade silver/gold ore separately for later treatment if ongoing test work confirms viability.

Next Steps

Completion of the Endeavor Mine Restart Study has generated an initial 10-year profitable mine life. Polymetals will now proceed with the replacement of the Endeavor Rehabilitation Bond, which completes the acquisition of the project, and will also secure a suitable finance facility to restart production at the mine. It is anticipated that refurbishment and pre-production works will commence very early in 2024.

Figure 8: Endeavor Mine Shaft Headframe and Main Mine Office

Cautionary Statements

The MRS discussed herein has been undertaken to explore the technical and economic feasibility of restarting production at the Endeavor Mine. The Production Target and financial forecasts presented in the MRS are shown on a 100% Project basis. The Production Target underpinning financial forecasts included in the MRS comprises 67% Ore Reserves including 70% Measured & Indicated Resources, and 30% Inferred Resources. The estimated Ore Reserves and Mineral Resource underpinning the Base Case Production Target have been prepared by a Competent Person in accordance with the requirements in the JORC Code. There is a low level of geological confidence associated with Inferred Resources and there is no certainty that further exploration work will result in the conversion of Inferred Resources to Indicated Resources or return the same grade and tonnage distribution.

The stated Production Target is based on the Company's current expectations of the future results or event and should not be solely relied upon by investors when making investing decisions. The economic outcomes associated with the MRS are based on certain assumptions made for commodity prices, concentrate treatment and recovery charges, exchange rates and other economic variables, which are not within the Company's control and subject to change from time to time. Changes in such assumptions may have a material impact on economic outcomes. To achieve the range of outcomes indicated in the MRS, debt and equity funding will be required. Investors should note that there is no certainty that the Company will be able to raise the amount of funding when needed and/or reach a Final Investment Decision by the date proposed in the MRS. This announcement contains forward-looking statements. Polymetals has concluded it has a reasonable basis for providing the forward-looking statements included in this announcement and believes it has a reasonable basis to expect it will be able to fund the development of the project. However, several factors could cause actual results, or expectations to differ materially from the results expressed or implied in the forward-looking statements. Given the uncertainties involved, investors should not make any investment decisions based solely on the results of the MRS. This announcement has been prepared in compliance with the JORC Code (2012) and the current ASX Listing Rules.

This announcement was authorised for release by the Polymetals Resources Ltd Board.

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COMPETENT PERSON STATEMENT

The information supplied in this release regarding Mineral Resources of the Endeavor Project is based on information compiled by Mr Troy Lowien, a Competent Person who is a Member of the Australian Institute of Mining and Metallurgy. Mr Lowien is an employee of Polymetals Resources Ltd and has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves". Mr Lowien consents to the inclusion of matters based on information in the form and context in which it appears.

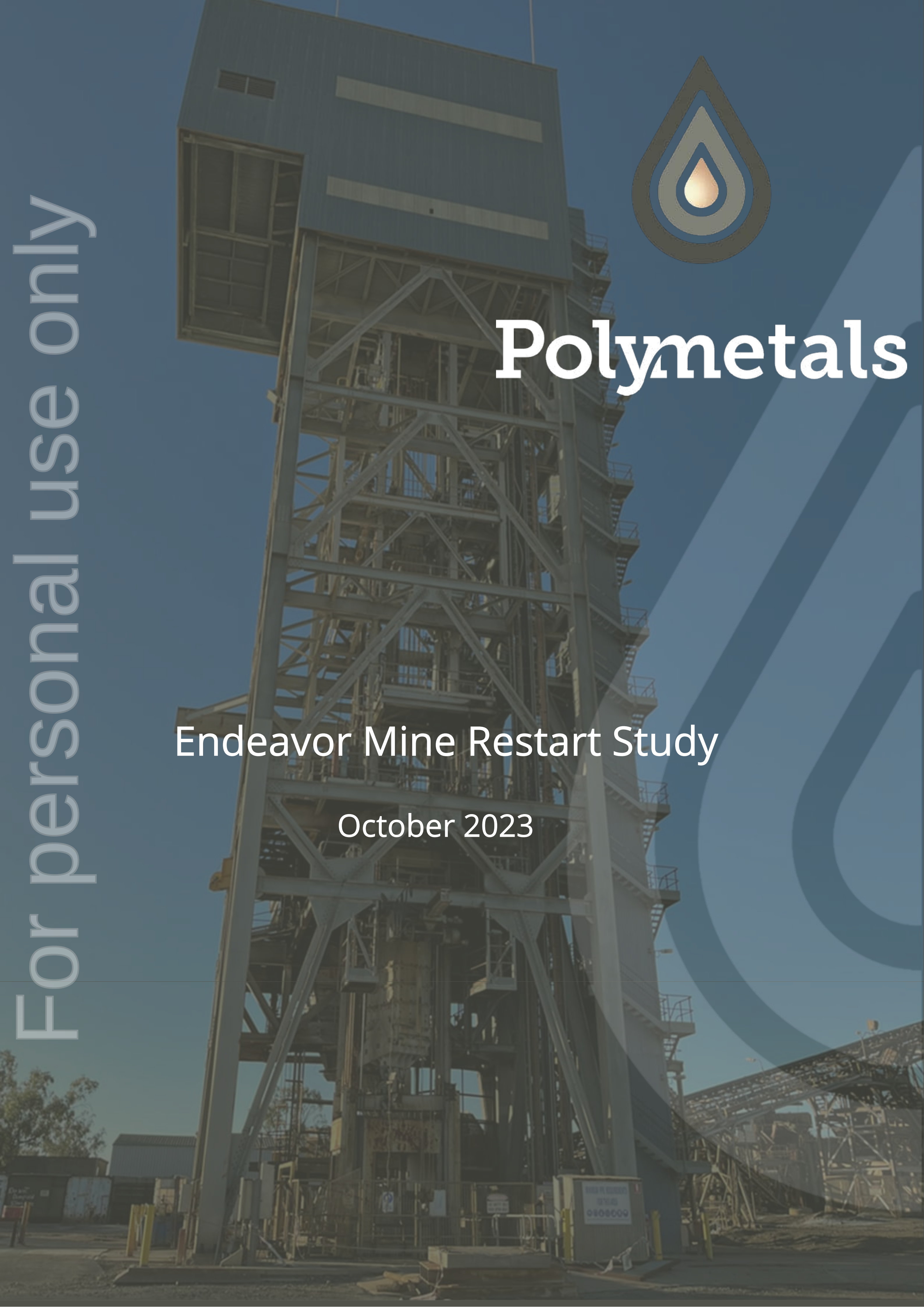
The information supplied in this release regarding Ore Reserves of the Endeavor Project is based on information compiled by Mr Matthew Gill, a Competent Person who is a Fellow of the Australian Institute of Mining and Metallurgy. Mr Gill is a Non-executive Director of Polymetals Resources Ltd and has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves". Mr Gill consents to the inclusion of matters based on information in the form and context in which it appears.

FORWARD LOOKING STATEMENT

This announcement contains "forward-looking information" that is based on POL's expectations, estimates and projections as of the date on which the statements were made. This forward-looking information includes, among other things, statements with respect to the mine restart study, POL's business strategy, plan, development, objectives, performance, outlook, growth, cashflow, projections, targets and expectations, mineral resources, ore reserves, results of exploration and related expenses. Generally, this forward-looking information can be identified by the use of forward-looking terminology such as 'outlook', 'anticipate', 'project', 'target', 'likely', 'believe', 'estimate', 'expect', 'intend', 'may', 'would', 'could', 'should', 'scheduled', 'will', 'plan', 'forecast', 'evolve' and similar expressions. Persons reading this announcement are cautioned that such statements are only predictions, and that POL's actual future results or performance may be materially different. Forward-looking information is subject to known and unknown risks, uncertainties and other factors that may cause POL's actual results, level of activity, performance, or achievements to be materially different from those expressed or implied by such forward-looking information. Forward-looking information is developed based on assumptions about such risks, uncertainties and other factors set out herein, including but not limited to general business, economic, competitive, political and social uncertainties; the actual results of current exploration activities; conclusions of economic evaluations; changes in project parameters as plans continue to be refined; future prices and demand of iron and other metals; possible variations of ore grade or recovery rates; failure of plant, equipment or processes to operate as anticipated; accident, labour disputes and other risks of the mining industry; and delays in obtaining governmental approvals or financing or in the completion of development or construction activities. This list and the further risk factors detailed in the remainder of this announcement are not exhaustive of the factors that may affect or impact forward-looking information. These and other factors should be considered carefully, and readers should not place undue reliance on such forward-looking information. POL disclaims any intent or obligations to revise any forward-looking statements whether as a result of new information, estimates, or options, future events or results or otherwise, unless required to do so by law. Statements regarding plans with respect to POL's mineral properties may contain forward-looking statements in relation to future maters that can only be made where POL has a reasonable basis for making those statements. Competent Person Statements regarding plans with respect to POL's mineral properties are forward looking statements. There can be no assurance that POL's plans for development of its mineral properties will proceed as expected. There can be no assurance that POL will be able to confirm the presence of mineral deposits, that any mineralisation will prove to be economic or that a mine will successfully be developed on any of POL's mineral properties.

ABOUT POLYMETALS

Polymetals Resources Ltd (**ASX: POL**) is an Australian mining and exploration company with a project portfolio with significant potential for the discovery and development of both precious and base metal resources. With our cornerstone asset the Endeavor Silver-Zinc-Lead Mine, Polymetals is seeking to become a long term, consistent and profitable base and precious metal producer. Polymetals holds a strong exploration portfolio for organic growth, are development driven and continually measure strategic acquisition opportunities. POL is committed to developing genuine long-lasting relationships within our community, building strong relationships with investment partners, local stakeholders and providing our shareholders with capital growth and dividends. For more information visit www.polymetals.com



Polymetals

Endeavor Mine Restart Study

October 2023

Polymetals

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Attachment 1	JORC Code Table 1
Attachment 2	TSF Sector 1 Tailings Mineral Resource Estimate Report 2023

1 Introduction

Polymetals Resources Ltd (Polymetals) is a New South Wales based mineral exploration company listed on the Australian Securities Exchange (ASX:POL). The company's board and senior management have extensive experience in exploring for minerals, developing and operating assets and producing base and precious metals from mining projects.

Polymetals has entered into a legally binding arrangement to acquire 100% interest in the Endeavor silver-zinc-lead underground mine and associated assets and is assessing the economic viability of recommencing operations at the mine which has been under a regime of care and maintenance since closure at the end of 2019.

This report summarises the outcomes of a study (to Pre-Feasibility Study level) which investigated the technical, financial, environmental, and social aspects of the Project, based on the following work program:

- Drill testing of the Upper Main Lode high-grade silver zone.
- Review and update of the Mineral Resources.
- Review of site infrastructure condition.
- Review status of licencing, permitting, and approvals.
- Review of previous site performance and technical information.
- Development of a robust mine plan and schedule based on multiple ore sources and conservative assumptions.
- Confirmation of the economic viability of restarting the mine through detailed financial modelling.
- Generation of an Ore Reserve estimate based on the study outcomes.

1.1 Study Team

The study was compiled and managed by Polymetals staff with input from external consultants and specialists in the following areas:

- Mineral Resource Estimation – Groundwork Plus.
- Metallurgical Review – AMC Consultants.
- Geotechnical Review – Ground Control Engineering.
- Stope optimisation, mine design and scheduling – Ground Control Engineering.
- Mining Costs – Ripago.
- Marketing and Logistics – Ocean Partners.

2 Property Description and Location

2.1 Property Description and Ownership

The Endeavor mine ("the **Project**") is located 47km north west of Cobar, New South Wales, Australia (**Figure 1**).

The project consists of an underground silver-zinc-lead mine, processing plant, tailings dams and rail loading facility. The mine is owned by Cobar Operations Pty Ltd (COPL), operated by Endeavor Operations Pty Ltd (EOPL), with housing infrastructure managed by Cobar Infrastructure Pty Ltd (CIPL). The three companies are currently wholly owned subsidiaries of CBH Resources Ltd (CBH). CBH is owned by Toho Zinc, a Japanese listed company.

Polymetals, through its 100% owned subsidiary company Cobar Metals Pty Ltd, has entered into a legally binding arrangement to acquire 100% of the Endeavor mine and associated assets by acquiring COPL, EOPL and CIPL from CBH. In order to complete the acquisition, Cobar Metals will be required to secure the release and replacement of the Environmental Rehabilitation Bond on or before 30 April 2024.

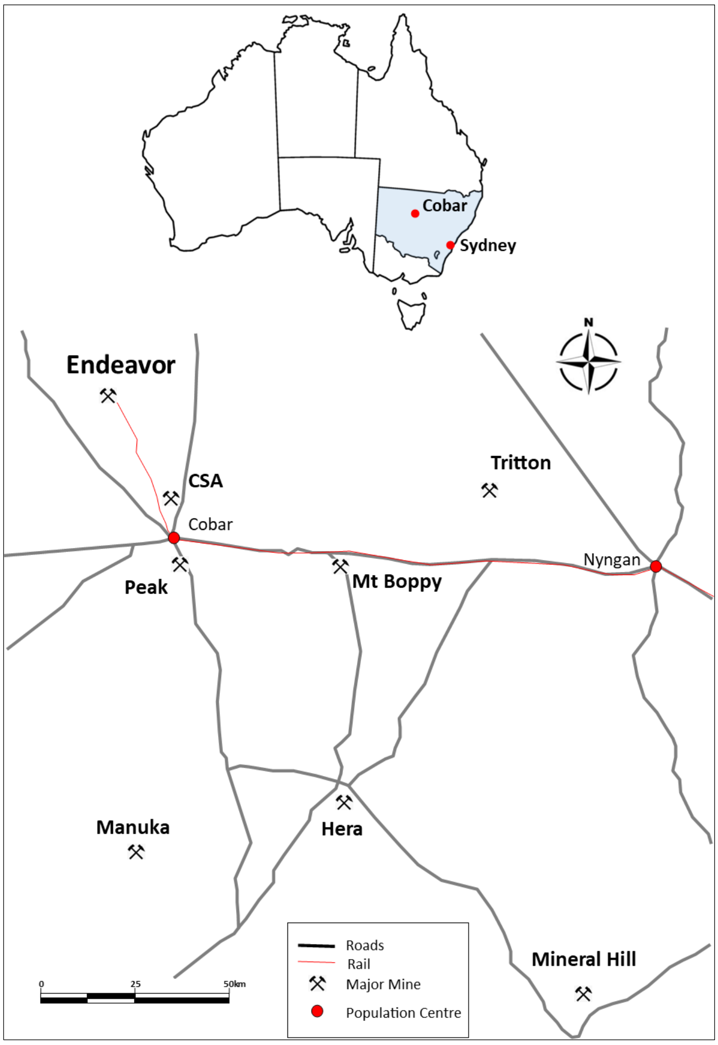


Figure 1: Project Location and Nearby Mines

2.2 Status of Mineral Titles

The Endeavor Mine is covered by five granted Mining Leases that sit within a broader package of three Exploration Licences as shown in **Table 1** and **Figure 2** and **Figure 3**.

Table 1 – Relevant Mineral Titles

Title	Holder	Expiry Date	Purpose
		Mining Leases	
ML158	Cobar Operations Pty Ltd	12/03/2028	Surface and underground mining activities for minerals.
ML159	Cobar Operations Pty Ltd	12/03/2028	
ML160	Cobar Operations Pty Ltd	12/03/2028	
ML161	Cobar Operations Pty Ltd	12/03/2028	
ML930	Cobar Operations Pty Ltd	20/05/2028	Underground mining activities for minerals (surface exclusion of 10m)
		Exploration Licences	
EL5785	Cobar Operations Pty Ltd	05/10/2027	Exploration for Group 1 Minerals
EL8583	Cobar Operations Pty Ltd	02/06/2029	Exploration for Group 1 Minerals
EL8762	Cobar Operations Pty Ltd	27/06/2024	Exploration for Group 1 Minerals

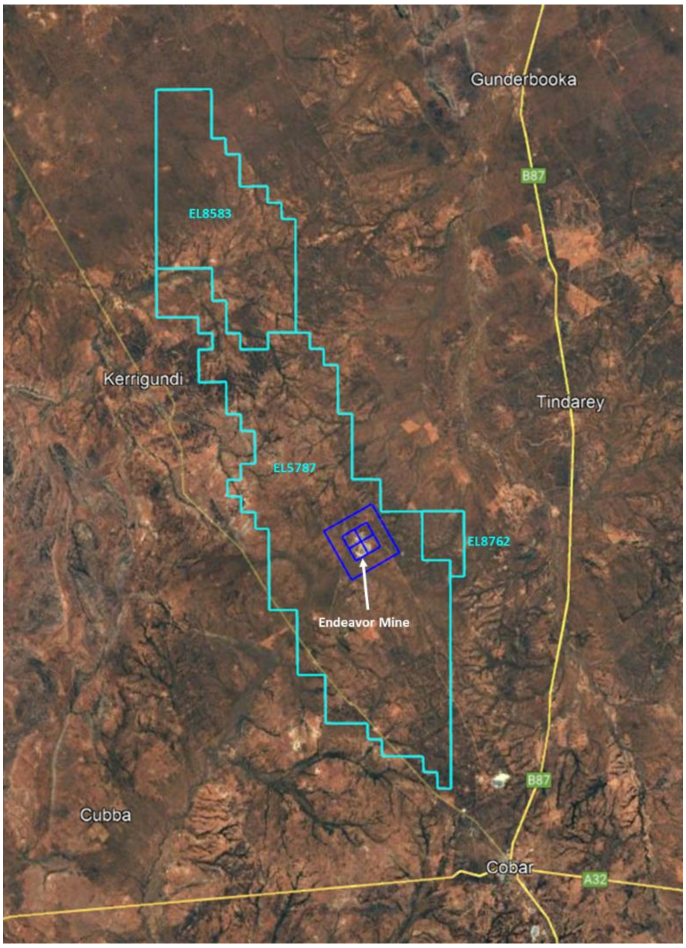


Figure 2: Exploration Licences

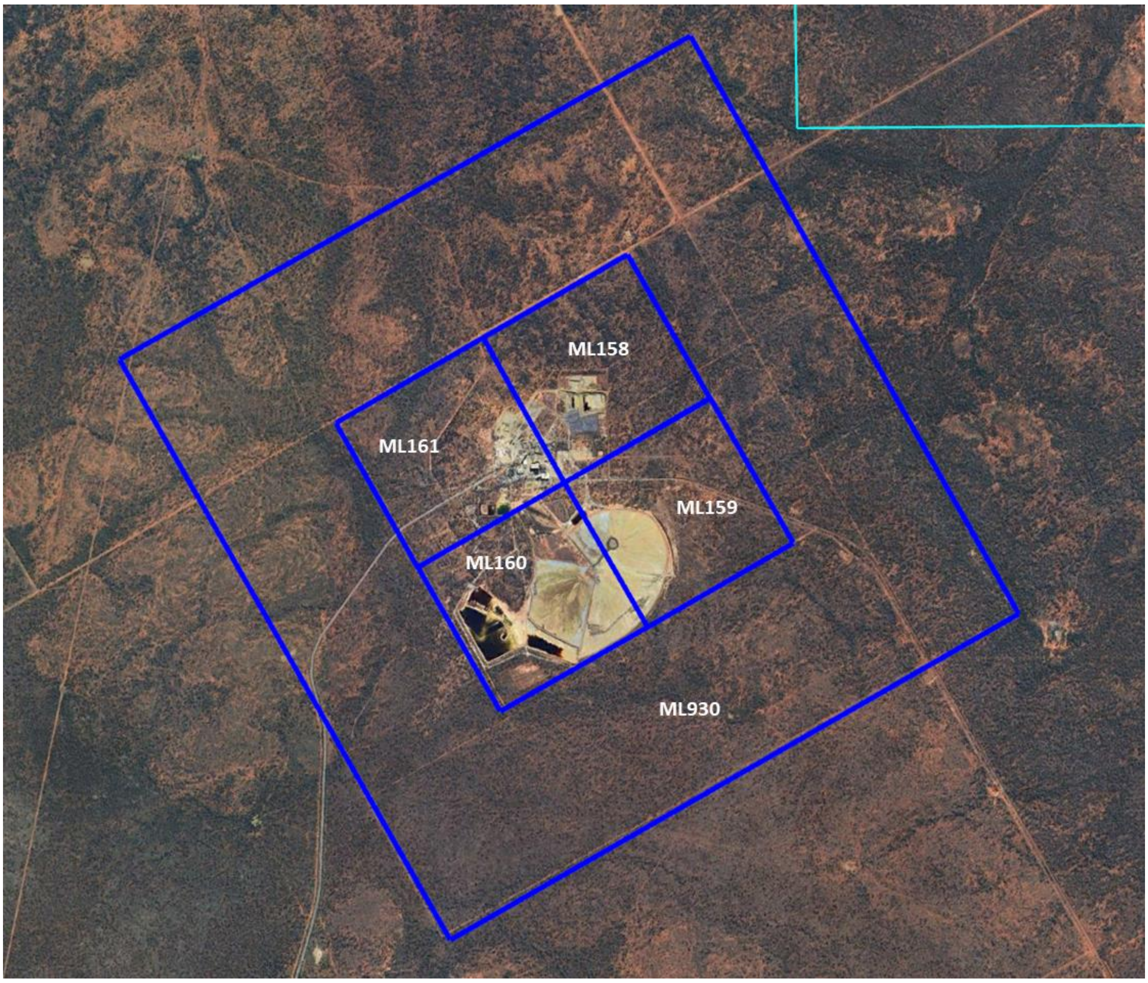


Figure 3: Mining Leases

2.3 Land Ownership and Land Use

The mine site and surrounding land falls within the Western Lands Division of New South Wales and is held under Western Land Leases as shown in **Figure 4**. The land title underlying Mining Leases 158, 159, 160, 161 and a portion of 930 is held by Cobar Operations Pty Ltd as WLL13839. The surrounding land is used for agricultural purposes, predominantly low intensity sheep and goat grazing.

Table 2 – Nearby Land Tenure

Land Holding	Holding Type	Lessee
WLL13839	Western Land Lease	Cobar Operations Pty Ltd
"Bundella South"	Western Land Lease	Meredith Cantwell
"Bundella"	Western Land Lease	John & Carol Abeni
"Darling Downs" & "Tindary"	Western Land Lease	Keith & Ruth Francisco
"Poon Boon"	Western Land Lease	Rod & Kaylene Boal
"Omrah Downs"	Western Land Lease	Roger Anderson & Katrina Virgoe

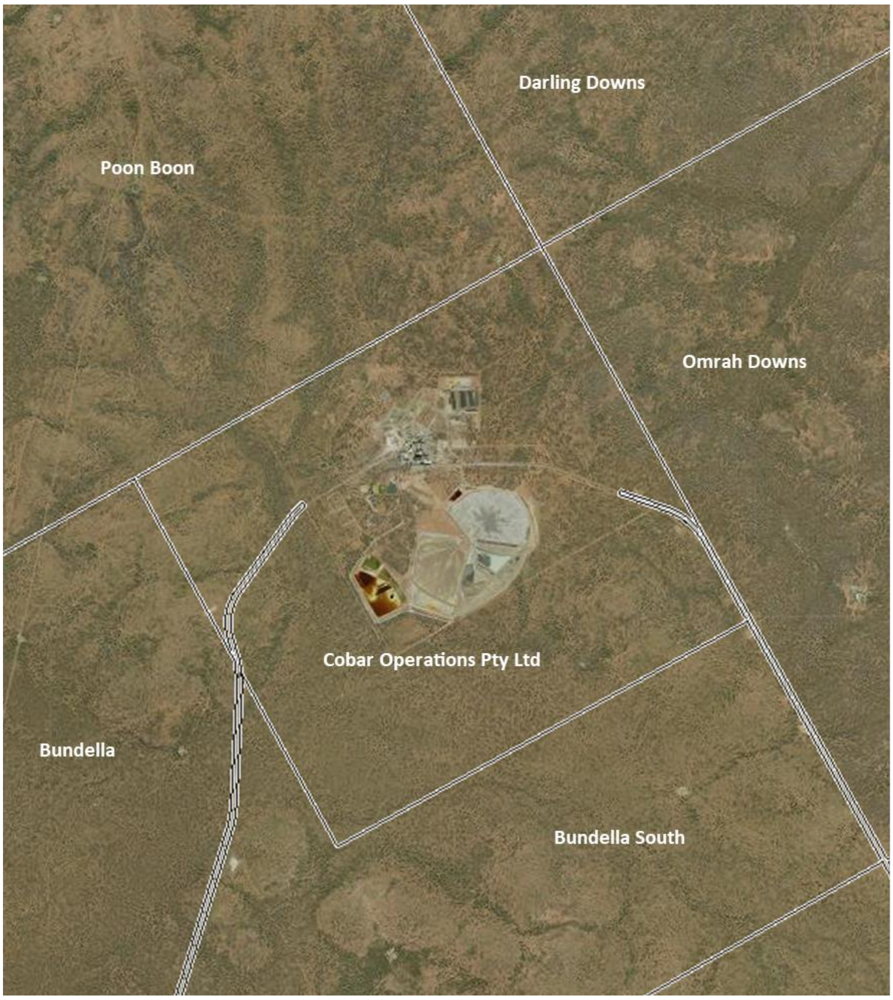


Figure 4: Cadastre Boundaries

2.4 Royalties

In New South Wales, mineral royalties are payable to the state under the Mining Act (1992) and, for high value to volume ratio minerals, is calculated Ad valorem. The royalty is calculated as 4 percent of the value of production, less allowable deductions such as the direct costs incurred in upgrading the material and bringing it to market after the first stockpile.

A third-party royalty is also payable to Metalla Royalty & Streaming Ltd. This royalty agreement was renegotiated by Polymetals Resources Ltd in January 2023 to replace the original 100% royalty on Silver with a 4% royalty based on the Net Smelter Return for lead, zinc and silver.

2.5 Environmental Liabilities

2.5.1 Environment Protection Licence

The NSW Environment Protection Authority (EPA) is the primary environmental regulator in New South Wales. Its role is to protect and enhance the environment and human health by implementing and enforcing environmental legislation and policies. The main pieces of legislation relevant to the project are the Protection of the Environment Operations (POEO) Act 1997, and the Protection of the Environment Operations (General) Regulation 2022. Under the POEO Act, the EPA issues environment protection licences to the owners and operators of industrial premises. The Endeavor Mine holds an environment protect licence, and therefore must:

- Comply with the conditions of their licence.
- Prepare pollution incident response management plans.
- Publish and/or make pollution monitoring data available.
- Pay annual administrative fees.
- Submit annual returns.

2.5.2 Rehabilitation Management Plan / Rehabilitation Bond

The NSW Resources Regulator, a division of the Mining Exploration and Geoscience group within the Department Regional NSW, regulates rehabilitation activities against the conditions of the mining lease (issued under the Mining Act 1992), to ensure rehabilitation commitments outlined in the development consent are met. Title holders are required to prepare a Rehabilitation Management Plan (RMP), which provides specific and measurable criteria regarding the rehabilitation implementation strategy for the project. The conditions of a mining lease also require a titleholder to report against agreed rehabilitation objectives and completion criteria. The NSW Resources Regulator undertakes an auditing, compliance, and enforcement program to ensure titleholders meet their rehabilitation obligations.

In NSW, a mining rehabilitation security bond must be provided before exploration and mining activities are undertaken. The security bond covers the full cost of all rehabilitation and mine closure activities required if a mining company defaults on their rehabilitation obligations. Before a security bond is returned the mining company must provide evidence to demonstrate to the Regulator that they:

- Have met the rehabilitation objectives.
- Have achieved the rehabilitation completion criteria.
- Have implemented the final landform and final land use.

The Environmental Rehabilitation Bond for the Endeavor Mine is currently $27,956,000.

Prior to recommencement of mining at Endeavor, the current RMP (dated July 2022) will be amended to reflect the change from care and maintenance to the resumption of operations.

The NSW Resources regulator also requires an Annual Rehabilitation Report and Forward Program to be submitted.

2.5.3 Tailings Dam

Dams in NSW are regulated under the Dams Safety Act 2015 and the Dams Safety Regulation 2019 by Dams Safety NSW, which 'declares' dams that can potentially endanger life downstream, cause major damage or loss to infrastructure, the environment or have major health and social impacts. Each dam is given a consequence category that reflects this potential. The Endeavor Operation Central Thickened Discharge Tailings Storage Facility (CTD TSF) has a dam consequence category rating of "significant". Owners and operators of dams with a consequence rating of "significant" are required to regularly review the documents as shown in **Table 3**. Documents with an asterisk must be submitted to Dams Safety NSW every time they are reviewed.

Table 3 – Tailings Dam Document Review List

Frequency of Review	Document
Annually	Annual Dams Safety Standards Report*
	Dam Safety Management System
	Emergency Plan – update contact details
Every 5 years	Emergency Plan – full review*
	Operations and Maintenance Plan
	Risk Report
Every 15 years	Consequence Category Assessment*
	Safety Review

2.6 Consents, Authorisations, Permits and Licences

Development consents pertaining to the project are listed in **Table 4**.

Table 4 – Development Consents

Consent Number	Date Issued	Purpose
Ref:SW:KT T3-1	26/01/1979	Develop land to establish a mining operation
2004/LDA-00033	13/01/2004	Construct a paste fill plant
2004/LDA-00044	18/02/2005	Develop Tailings Storage Facility (Sector 5)
2006/LDA-00030	10/10/2006	Construct a concrete batching plant
2007/LDA-00016	24/04/2007	Construct a decline bypass (additional access – U/G)
2007/LDA-00059	05/12/2007	Construct a tailings dam wall raise
2007/LDA-00084	16/12/2007	Alternate Backfill Project
2018/LDA-00030	17/07/2019	Waste rock storage facility
2019/LDA-00019	17/07/2019	Installation of a bulk air refrigeration plant (not commenced)

The project holds a number of licences authorising a variety of activities as listed in **Table 5**.

Table 5 – Licences

Licence	Licence No.	Issued By	Date of Expiry or Renewal	Purpose
Environment Protection Licence	1301	NSW Environment Protection Agency	Upon surrender, suspension, or revocation	Authorises scheduled activities: Crushing, grinding, or separating. Chemical Production. Chemical Storage. Extractive Activities. Mining for Minerals.
Radiation Licence	5061132-RML28863	NSW Environment Protection Agency	Expired – To Renew prior to commencement of operations	Licence to Sell/Possess radiation apparatus and /or radioactive substances or items containing radioactive substances.
Western Land Lease	WLL13839	NSW Department of Lands	In perpetuity	Granted for “Business Purposes” under Western Lands Act 1901. Subject to lease conditions.
Dangerous Goods Notification Acknowledgement	NDG019577	NSW Work Cover	Upon surrender, suspension, or revocation	Authorises the storage and use of dangerous goods
Explosives storage and manufacture	XSTR100161	NSW Work Cover	Expired – To Renew prior to commencement of operations	Authorises the possession and storage of explosives
Refrigerant Trading Authorisation	AU03561	Australian Refrigeration Council	Expired – To Renew prior to commencement of operations	Authorises the handling of refrigerants

Licence	Licence No.	Issued By	Date of Expiry or Renewal	Purpose
Water Use Approval	85WA752582	NSW Department of Industry	05/05/2025	Authorises the extraction of groundwater from bores: 85BL256033, 85BL256034, 85BL256035, 85BL256036, 85BL256037, 85BL256038
Water Licence – Groundwater	85AL752581	NSW Department of Industry	In perpetuity	Extraction of 790ML per year from bores.
Water Licence – Surface Water	80AL716062	NSW Department of Industry	In perpetuity	Usage of 1,605ML per year of water from Lake Burrendong.
Exploration Licence	5785	NSW Department of Trade & Investment	05/10/2027	Provide surface rights for exploration on ML930.

3 Access, Climate, Local Resources, and Physiography

3.1 Accessibility

The Endeavor mine Project is located approximately 47 km NW of Cobar, New South Wales, Australia. Access to the mine is by sealed roads via the Cobar-Louth road (Mulya Rd). A dedicated railway branch line of the Nyngan to Cobar railway provides transport for concentrates.

3.2 Climate

Cobar experiences a hot desert climate characterised by extremely high temperatures, low annual rainfall, and an average of 300 days of sunshine throughout the year.

During Summer, the average maximum temperature is around 34°C, and can occasionally exceed 45°C. The nights are generally warm, with temperatures rarely dropping below 20°C. Heatwaves are common during summer, with consecutive days of extreme heat.

Winters are mild to cool with the average daytime temperature around 17°C, while night-time temperatures can drop to around 5°C or lower. Frost can occur during winter nights, especially in the colder months of June and July.

Rainfall in Cobar is limited, and the region is known for its aridity. The annual precipitation averages around 300 to 400 mm, with most of the rainfall occurring in sporadic and often unpredictable events. The wettest months tend to be during the summer thunderstorm season, which is typically from December to March.

Mining and transport operations operate year-round with little to no interruptions from weather related events.

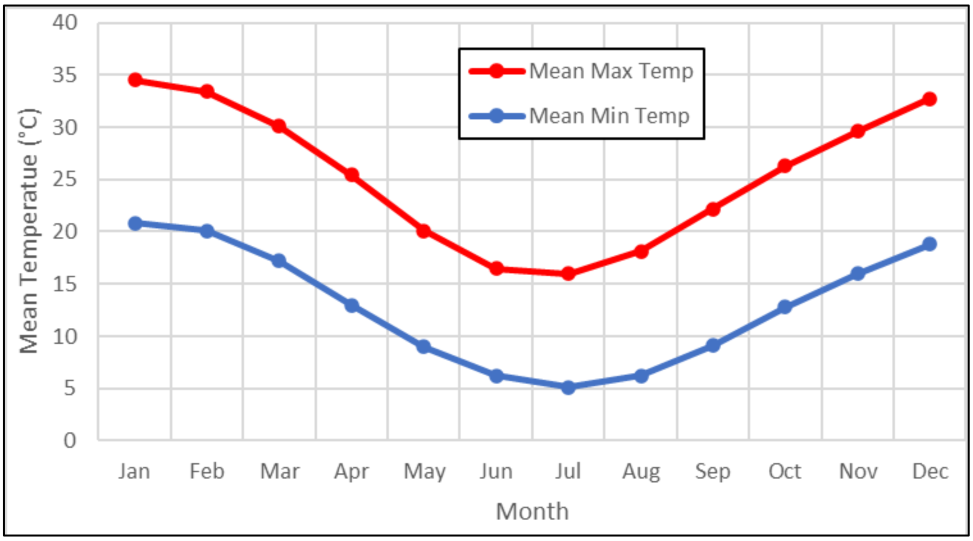


Figure 5: Mean Monthly Temperatues – Cobar (1962-2023)

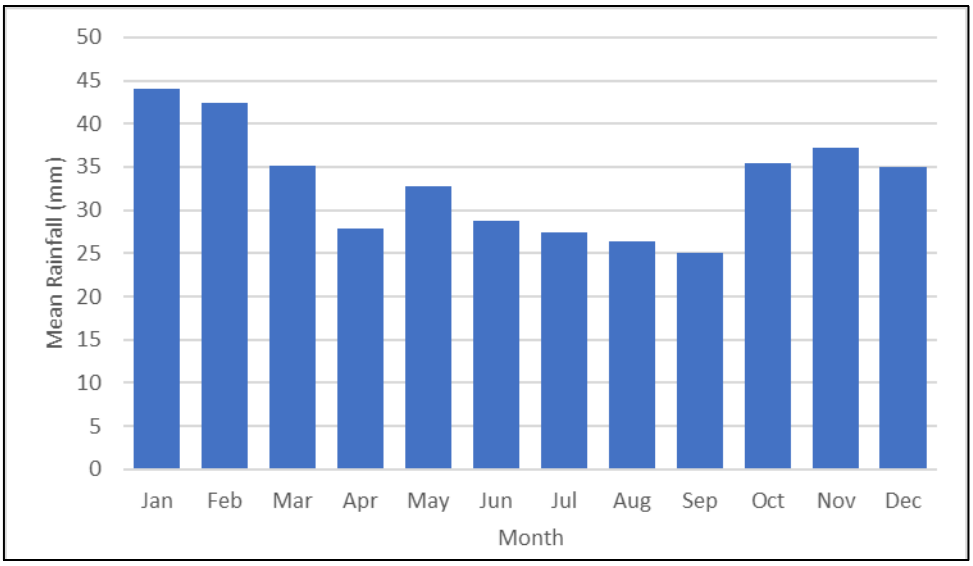


Figure 6: Mean monthly Rainfall – Cobar (1962-2023)

3.3 Local Resources

The Cobar region has a long history of mining activity, with several large underground mines currently operating in the Cobar area (e.g. the CSA Mine and Peak Mine), and with numerous mining suppliers and contractors located locally in Cobar. Operations and technical personnel are likely to be sourced from, and reside within, the surrounding Cobar area.

The mine is connected directly to the state electricity grid and a water supply pipeline managed by the Cobar Water Board. There are sufficient areas of suitable land surrounding the mine to accommodate any future need for expanded operating areas.

3.4 Physiography

Relief at the Endeavor Mine is flat or gently undulating with no outstanding surface features, with elevations ranging between 220m (Australian Height Datum) AHD to the north of the mine site to 204 m AHD to the south. The geologically old landscape is comprised of low erosional mounds between, broad sediment-filled watercourses, which in the area are not distinct and not clearly defined due to the nature of rainfall events.

Soils in and around the Endeavor Mine Leases are predominately red earths, non-calcareous loams. They have a weak profile differentiation, except for the darker surface horizon and neutral pH trend. On the ridges and slopes, the upper 5cm are red sandy clay loams with a reasonable content of organic matter. Below this, the soil is massive red sandy clay with little to no organic matter.

4 History

4.1 Previous Owners, Historic Exploration & Mining

The Elura silver-zinc-lead deposit was first discovered in 1973 by the Electrolytic Zinc (EZ) Company of Australia using aeromagnetic surveys followed up by auger and diamond drilling. This drilling enabled the reporting of an initial mineral resource of 27 Mt @ 5.6% Lead (Pb), 8.6% Zinc (Zn) and 135 g/t Silver (Ag).

Further exploration was carried out in 1976 via the excavation of a 165m deep shaft and crosscut to access the deposit and extract material for metallurgical test work.

Following a positive feasibility study in 1977 construction began on the Elura Mine project (as it was referred to then) in 1980, with the first ore milled in November 1982. A total of 0.7 Mt of ore was milled during the first year of production.

The mine was acquired by North Broken Hill Holdings Ltd in 1985, after the latter took over EZ Industries Ltd in 1984. Subsequently it became part of Pasminco Ltd Holdings in 1988. Production increased to around 1.2 Mt per year until the early 90's when the rate was reduced back to around 0.7 Mt per year due to a fall in metal prices, before increasing back to around 1 Mt per year in 1995.

Pasminco was placed into voluntary administration in 2001 and the mine was acquired by CBH Resources in 2003, changing the name of the project to the Endeavor Mine. From 2009 the mine operated again on a reduced production rate of around 0.6 Mt per year due to lower metal prices before being placed on care and maintenance in 2019.

During the life of the mine around 32 million tonnes of ore has been extracted.

In March 2023 Polymetals announced it had executed a Share Sale and Purchase Agreement to acquire 100% interest in the Endeavor Mine via the acquisition of Cobar Metals Pty Ltd, a company that had separately entered into an arrangement to acquire the project. A drilling program was completed by Polymetals in March 2023 to evaluate the unmined portion of the upper Main Lode mineralisatio, known for its high silver grade.

5 Geology

Mineralisation at the Elura deposit is hosted by a fine grained turbidite sequence of the Cobar Basin (**Figure 7**) and comprises multiple sub-vertical elliptical shaped pipe-like pods that occur within the axial plane of an anticline and are surrounded by an envelope of sulphide stringer mineralisation, in turn surrounded by an envelope of siderite alteration extending for tens of metres away from the sulphide mineralisation (**Figure 8**). Around 150m below the base of the main mineralised pods/lodes, mineralisation is hosted within the western limb of a folded limestone unit, occurring in veins and fractures. A zone of supergene enrichment occurs at the top of the Main Lode. Recent reviews favour a syngenetic formation model of an original stratiform deposit that was later emplaced by tectonic force into a favourable structural site during deformation.

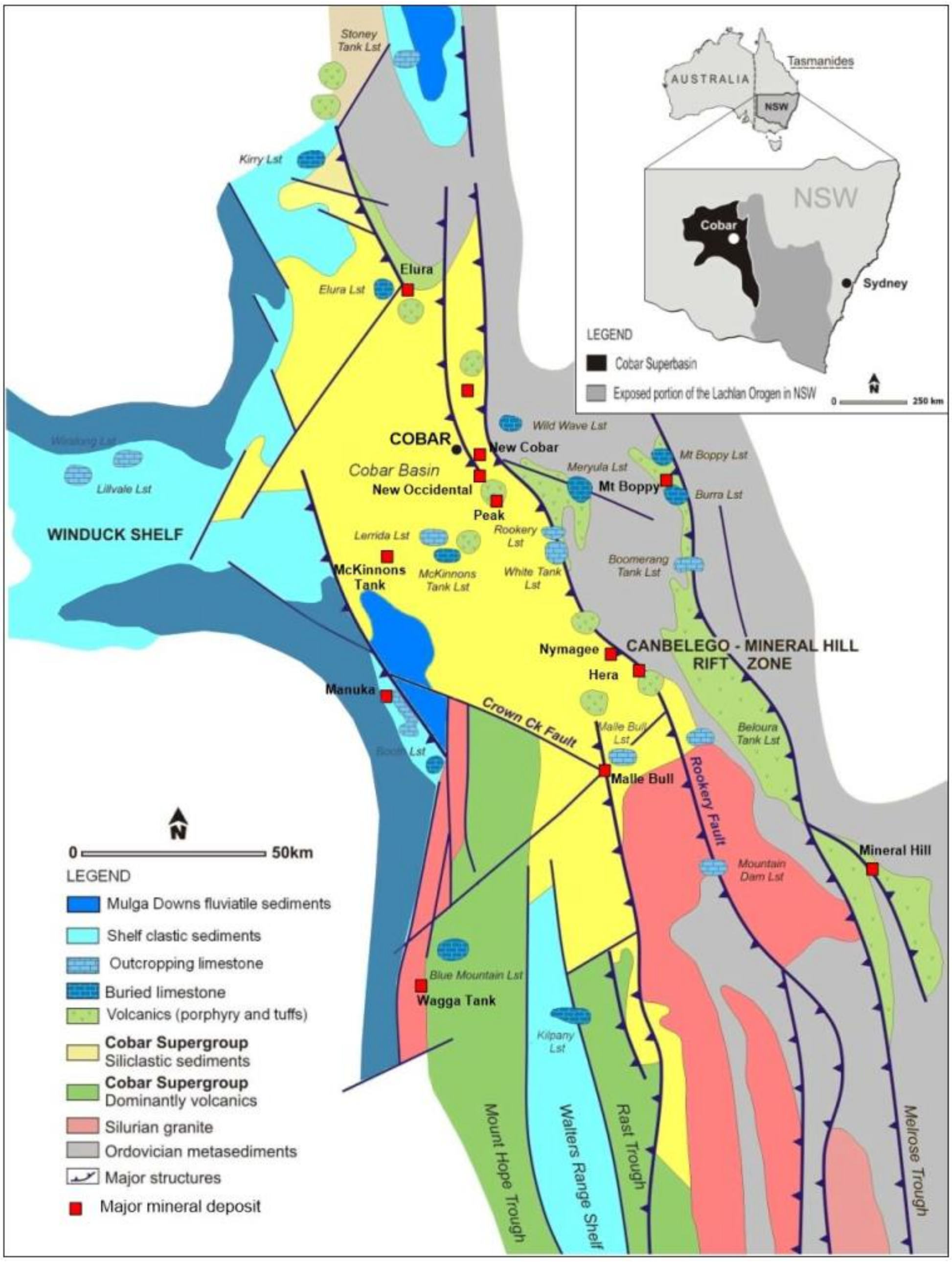


Figure 7: Regional Geology – Cobar Basin (David 2018)

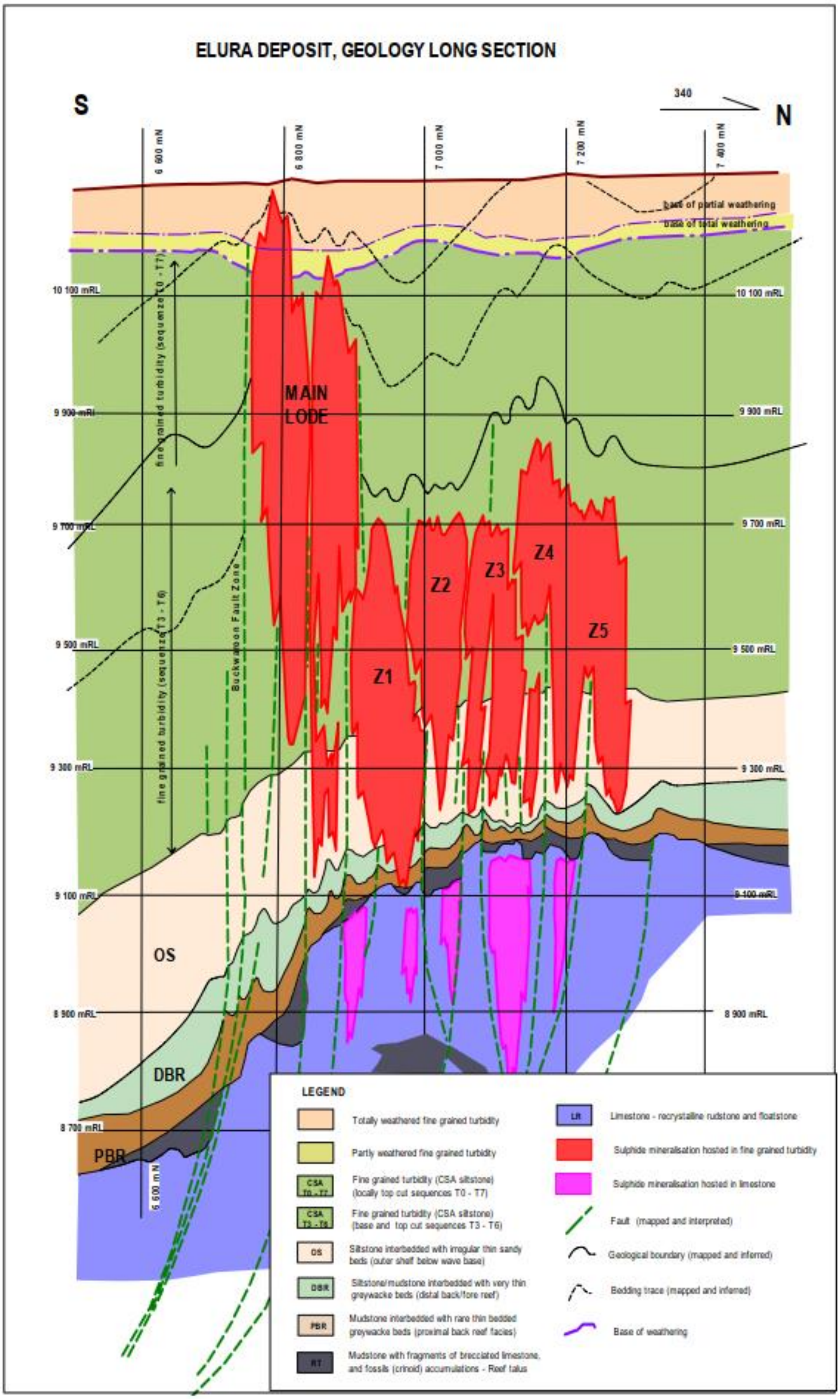


Figure 8: Long Section Elura Deposit (Reed 2004)

6 Mineral Resource Estimates

There are two Mineral Resource estimates which formed the basis of this study into the viability of restarting the Project. These are the Endeavor Mine in situ Mineral Resource and the TSF Sector 1 Tailings Mineral Resource. Summaries of the in situ and tailings Mineral Resource Estimates are provided in **Table 6** and **Table 7**.

Table 6 – Endeavor Mine In Situ Mineral Resource May 2023[1]

Category	Mt	Zinc (%)	Lead (%)	Silver (g/t)
Measured	4.4	8.3	5.1	93
Indicated	8.8	7.9	4.6	82
Inferred	3.1	7.7	3.7	78
Total[2]	**16.3**	**8.0**	**4.5**	**84**

1. Reported using NSR cut-off values of $190/t for mineralisation above 10,080mRL, and $150/t for mineralisation below 10,080mRL
2. Discrepancies may occur due to rounding

Table 7 – Endeavor Mine TSF Sector1 Tailings Mineral Resource October 2023[1]

Category	Mt	Zinc (%)	Lead (%)	Silver (g/t)
Indicated	3.6	2.14	1.56	80
Inferred	1.6	2.07	1.53	77
Total[2]	**5.2**	**2.12**	**1.55**	**79**

1. Reported without use of cut off grade
2. Discrepancies may occur due to rounding

The Mineral Resources on which the Production Target and Ore Reserves for the Endeavor Mine are based were compiled by Competent Persons in accordance with guidelines set out in the 2012 edition of the JORC Code.

The Endeavor Mine in situ Mineral Resource Estimate was first published by the Company in the ASX release *Endeavor Mine Acquisition Final* (28 March 2023) and an update Mineral Resource Estimate was subsequently published in the ASX release *Endeavor Near Surface Resource 94% Measured & Indicated* (23 May 2023).

The Endeavor Mine TSF Sector 1 Tailings Mineral Resource Estimate is being published for the first time in this report. A full copy of the Mineral Resource Estimate Report for the TSF Sector 1 Tailings is provided in **Attachment 1** of this report. A summary of the information material to understanding the Sector 1 Tailings Mineral Resource Estimate is provided in the following sections.

6.1 TSF Sector 1 Tailings Mineral Resource Estimate Summary

6.1.1 Location

The CTD TSF is a raised "turkey's nest" type dam, with Sector 1 measuring approximately 550m by 850m and an average depth of around 7m (**Figure 9**). The tailings material was deposited in Sector 1 from the beginning of operations in 1983 until 1989.

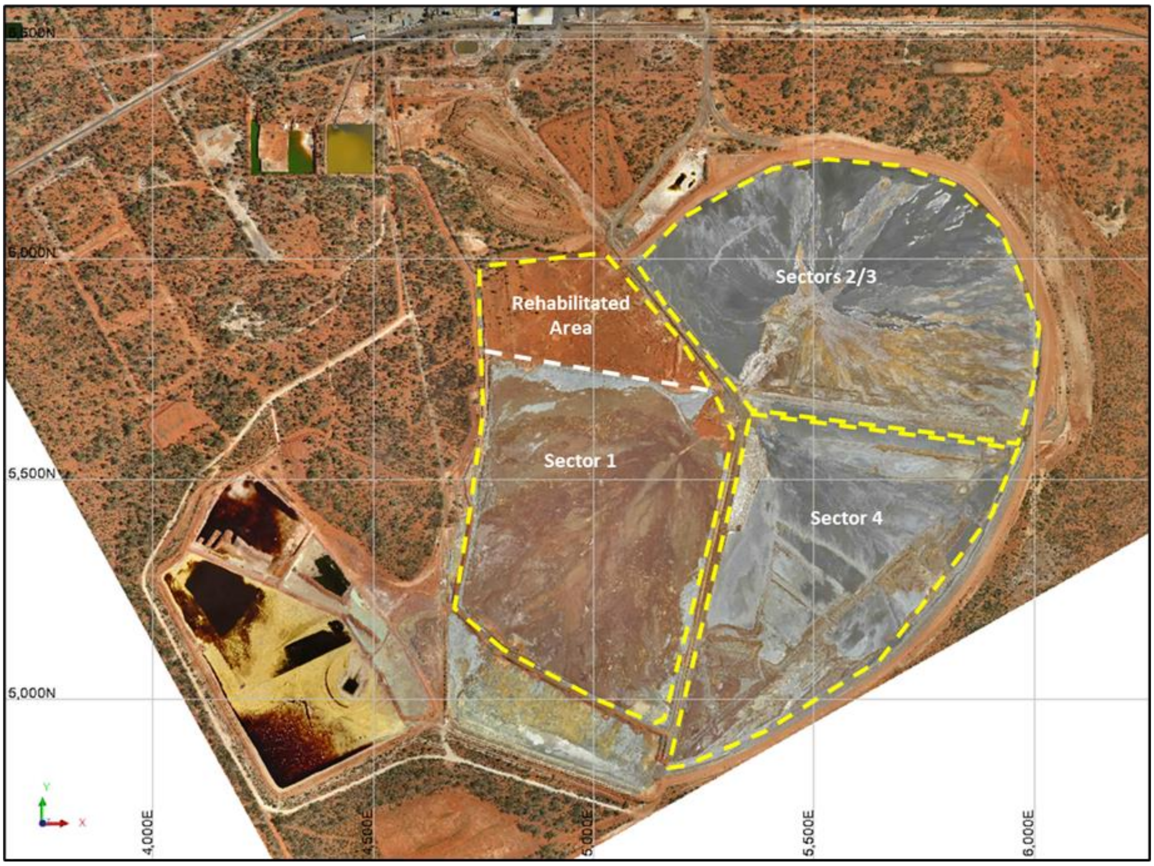


Figure 9: Endeavor Mine TSF Layout

6.1.2 Geology and Mineralisation

Mineralised material in the tailings storage facility consists of clay to fine sand sized particles deposited in sub-horizontal layers from centrally located outflow sites. The particles contain remnant sulphides that were not captured during processing of the Endeavor Mine silver-zinc-lead ore. The primary lead and zinc bearing minerals from all orebodies processed are galena (~13%wt) and sphalerite (~14%wt). Pyrite and pyrrhotite (~60 to 70%wt in total) are the main floatable gangue in the ore. Tetrahedrite is the major host of silver, apart from galena and chalcopyrite.

6.1.3 Drilling Sampling and Analysis

The tailings contained within Sector 1 of the TSF have been investigated by drilling programs in 2014, 2015 and 2017 (CBH Resources). Overall, 204 holes were drilled, totalling 1,135 m of drilling, of which 34% was completed using push tube methods, and 66% by air core methods. Drilling in the rehabilitated area of Sector 1 was not carried out due to directive from the Environmental Protection Authority (**Figure 10**).

During the 2014 air core drilling, 2m composites were taken from 1m samples intervals by spear method. During the 2015 and 2017 push tube drilling, samples were split laterally with average sample lengths of 1.2 m (2015) and 1.0m (2017). The 2017 drilling was completed for metallurgical test work only, so samples were combined into various composites to represent global and local areas.

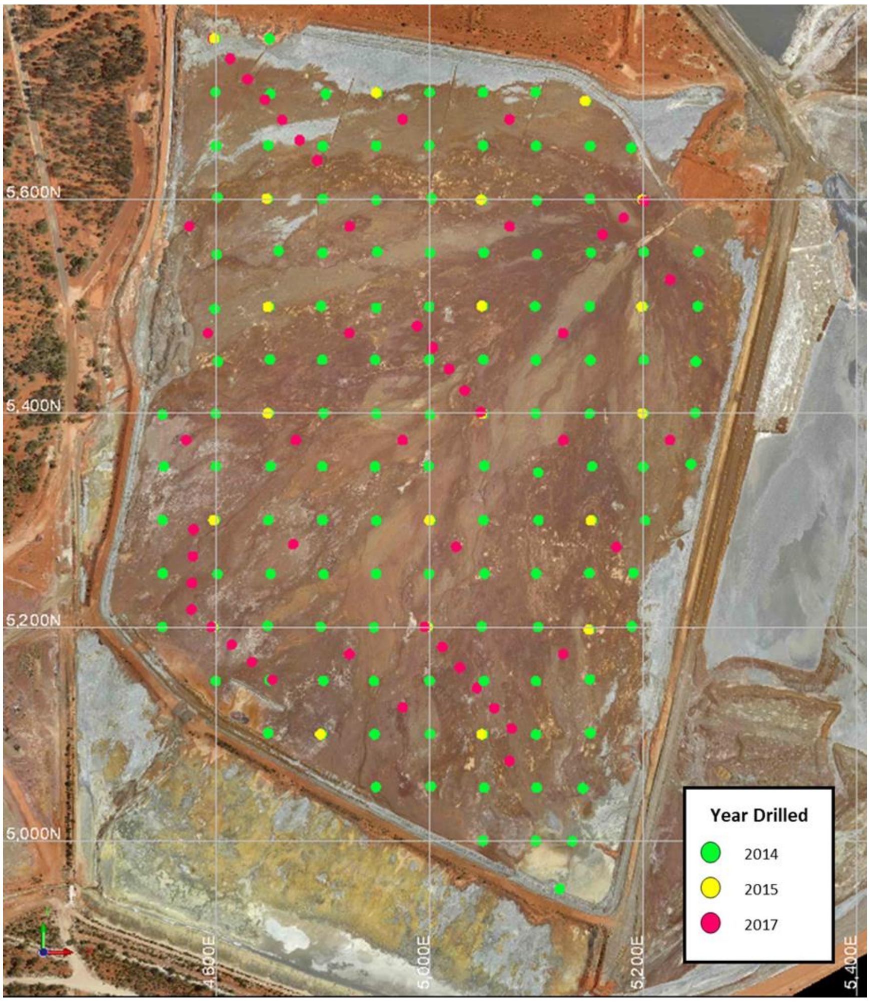


Figure 10: TSF Sector 1 Drill Hole Locations Coloured by Year Drilled

Samples collected in 2014 were prepared and assayed at the Endeavor laboratory using an Aqua Regia digest with atomic absorption spectrometry (AAS) for lead, zinc, silver, iron and copper analyses. Samples collected in 2015 were sent to ALS-Orange and assayed by an Aqua Regia digestion using AAS (ICP-AES) analysis for lead, zinc, silver, iron and copper. A composite comprised of each hole was also sent to the ALS Metallurgy laboratory in Burnie, Tasmania and an assay of the head sample was carried out by XRF on a pulverised sample. Samples collected in 2017 were sent ALS Metallurgy laboratory in Burnie, Tasmania and an analysis of the head samples were carried out by unknown method.

Quality control samples submitted during the 2014 program identified an issue with the lead grade results reported by the site laboratory with the lead grade of the certified reference material biased low by about 20%. Subsequent re-assays of selected samples conducted at ALS Orange were on average 13% higher than the original assays, and drill hole twinning also returned higher grades on average. Due to these issues, assay results from the 2014 drilling program were chosen not to be used in the Mineral Resource Estimate. Assessment of the accuracy and precision of assays from the 2015 drilling quality control program indicate these results are robust.

6.1.4 Bulk Density

During the 2014 drilling program, 551 samples for density analysis were taken from each 1m interval by firmly compressing the material into a grout sampling and levelling the top off. Each sample was stored in zip-lock plastic bags and taken to the site laboratory for wet weight and dry weight measurements. The average dry density value was 1.74 t/m^3.

6.1.5 Geological Interpretation and Modelling

The main features of TSF Sector 1 were modelled using drill hole data and detailed surface surveys. A surface representing the bottom of the tailings deposit was modelled based on 10 drillholes from that penetrated the TSF floor. The tailings surface was surveyed using aerial photogrammetry from which a surface DTM model was created.

A lateral boundary of the tailings deposit was created to constrain the estimation and reporting process. The boundary was kept within the bunded walls of the TSF and away from the area in the north where sludge from the Cockle Ck smelter Primary Electrostatic Mist Precipitator (PEMP) is stored (**Figure 11**).

Figure 11: TSF Sector 1 Tailings Resource Boundary

6.1.6 Metallurgy and Mineral Processing

Metallurgical test work has indicated saleable Zn and Pb/Ag concentrates can be obtained from processing the tailings through the existing flotation process on site. Refer to Section 10.5 of this report for detailed information.

6.1.7 Estimation Methodology

Grade estimation was undertaken using Ordinary Kriging (OK) interpolation method with block discretisation into a block model with parent block size of 50m x 50m 2m. Search ellipse orientations and distances were determined based on variogram orientation, variogram model anisotropy and ranges, horizon geometry and data distribution. A multiple search strategy was undertaken with the estimation carried out in two passes, increasing the search radius (isotropic) from 270m in the first pass to 500m in the second pass.

6.1.8 Classification Criteria

The Resource estimate has been classified as Indicated and Inferred Mineral Resources in accordance with guidelines as set out in the Joint Ore Reserves Committee (JORC) Code (2012). Resource categories have been defined using definitive criteria determined during the validation of the grade estimates, with detailed consideration of the JORC Code categorisation guidelines.

The exploration data used for the TSF Sector 1 Resource estimate is robust and appropriate for resource estimation purposes, with the current data spacing sufficient to generate robust grade estimates. Confidence in the estimate is increased by favourable comparisons to historical tailings deposition records and head grades from global metallurgical composite samples.

There are reasonable prospects for the eventual economic extraction of the resources because of proximity to an existing floatation processing plant and metallurgical test work indicates economic recoveries for Zn, Pb and Ag.

Based on the consideration of items listed above, and review of the resource block model estimate quality, classification criteria were determined as summarised in the following: -

- **Indicated**
    - Blocks in the tailings domain that occur between drill holes or no more than 50m from a drill hole.
- **Inferred**
    - All remaining blocks in tailings domain no assigned Indicated.

A plan displaying the areas of Indicated and Inferred Resources is displayed in **Figure 12**.

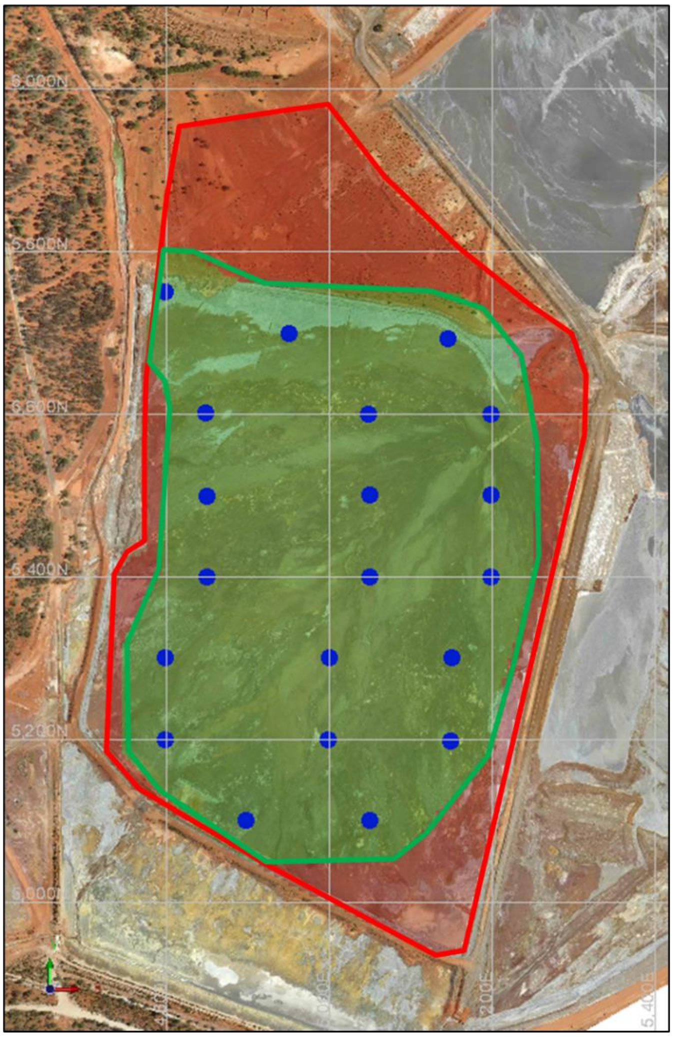


Figure 12: TSF Sector 1 Tailings Resource Categories – Indicated (green) and Inferred (red)

6.1.9 Cut Off Grade

The Mineral Resource has been reported without the use of a cut off grade as the proposed mining method (hydro mining) would not allow for efficient selective mining to be carried out..

7 Mining Methods

7.1 Past Production

Production from the Endeavor Mine commenced in 1982 with a total of 32Mt at a grade of 8.0% Zn, 5.0% Pb and 89.2 g/t Ag being mined and processed up to the end of 2019. Total metal in concentrate over this period was 2.0Mt Zn, 1.2Mt Pb and 41.6Moz Ag. The production rate varied over the life of the mine, mainly due to fluctuating metal prices, as shown by the chart of yearly metal produced in **Figure 13**. A peak production rate of 1.25Mtpa was achieved, with an average of 874ktpa over the mine life.

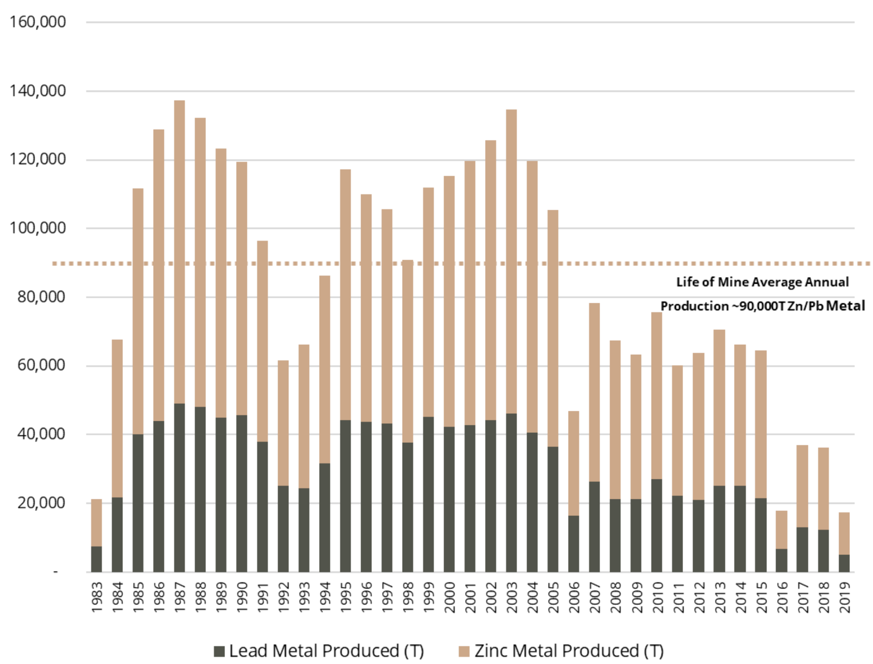


Figure 13: Yearly Metal Produced

7.2 Mine Development

Mine access for personnel and equipment is via a portal and decline. Prior to 2008, the upper portion of the decline (surface to the 800 level) was 4.5 m wide and 3.3 m high at a grade of 1:7, to a depth of 410 m below the surface. A new decline was commissioned in 2008, to upgrade the top part of the mine access to 5.5 m wide × 5.5 m high at a grade of 1:7. The bottom portion of the mine, from the 800 level down to the 190 level is accessed by a 5.5 m wide × 5.5 m high decline at a 1:7 grade. Below the 190 level the decline has been continued down to access the Deep Zinc Lode. The decline face is currently at the 140 level, some 1,060 m vertically below the surface and which is the top level of the Deep Zinc Lode production area.

A haulage shaft extends from surface to just below the 900 level (407m vertical height). The shaft is circular and 6m in diameter.

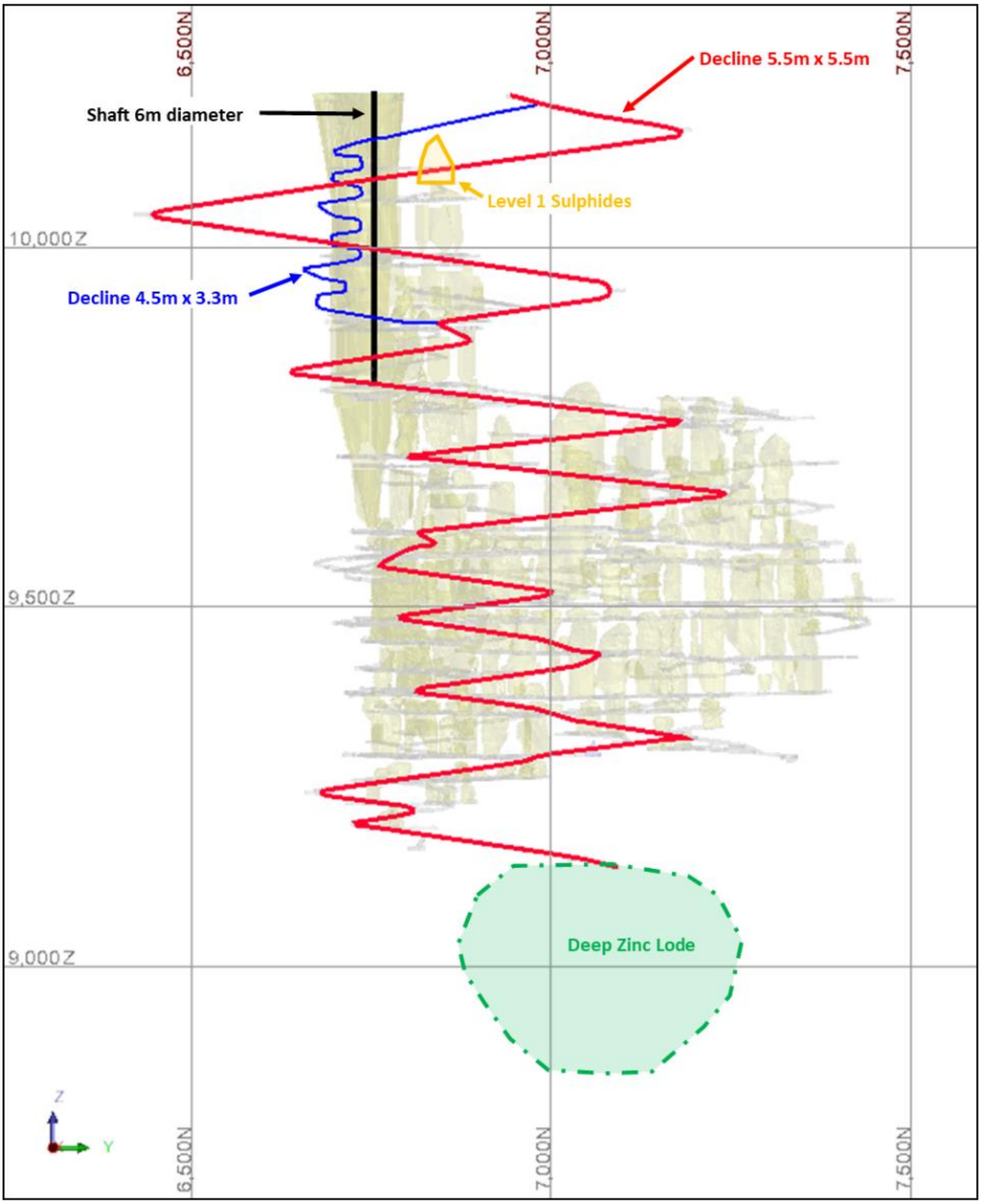


Figure 14: Mine Access Development

7.3 Mining Methods

The Endeavor Mine utilised a long hole open stoping mining method to extract high grade ore. This method varied when mining primary or secondary stopes and when mining remnants (rib pillars, crown pillars and halos). An initial slot rise was created by either drilling a raise bore hole or by mining a long hole slot (uphole or downhole) into the orebody. Slot rings were then fired into this rise to fully open up the slot. After the slot was opened up, production rings (uphole or downhole) were then drilled and fired into the slot. Stope firings used bulk emulsion explosives initiated by electronic detonators. Stopes were fired remotely from the surface.

Rib pillars and crown pillars were often left when mining remnant stopes to avoid dilution from previously mined surrounding stopes. Pillar extraction between levels and between existing filled stopes was undertaken routinely. Cemented paste fill was utilised to fill stope voids where it was deemed necessary for maintaining access to other stoping areas. Loose waste rock was utilised where paste fill was not considered necessary.

Sub-Level stoping will be utilised as the mining method in the Deep Zinc Lode, using cemented rock fill to reduce the number of pillars and enable the maximum recovery of ore. The Level 1 Sulphide ore is planned to be mined by cut and fill methods at this stage because of potentially poor ground conditions. The mining method for this area will be re-assessed once further geotechnical information is gathered from drilling and initial development.

Prior to the mine being placed on care and maintenance in 2019, ore was broken using 102 mm or 115 mm blastholes drilled by Tamrock Solo 1520 drills. Bogging was carried out conventionally or tele-remotely from the stopes by Elphinstone R2900 loaders. The ore was then trucked up to the 900 level by AD45B Cat trucks where it was tipped into a crusher and hoisted to the surface. It is planned to use similar specification equipment when mining resumes in the main ore body and Deep Zinc Lode. Smaller jumbos, loaders, and trucks will be used in the Level 1 Sulphide area due to the reduced size of the openings to access this area.

7.4 Geotechnical

7.4.1 Introduction

The current understanding of geotechnical conditions at the Endeavor Mine is formed from a combination of sources including geotechnical data collected from core logging and underground mapping and inspections, actual mine performance, geomechanical testing and various geotechnical assessments undertaken by external consultants.

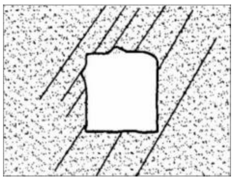
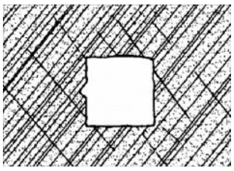
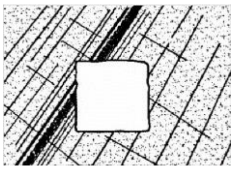
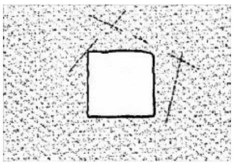
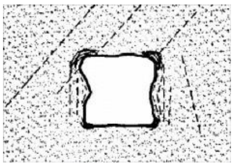
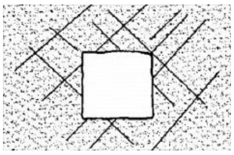
Volumes of rock mass with similar geotechnical properties have been grouped into three rock mass, or geotechnical, domains:

1. CSA Siltstone.
2. Ore Zone (incorporating massive sulphides (MS).
3. Brecciated vein (BV).

Smaller, sub-domains have also been defined in the different mineralisation zones.

A summary schematic of the various ground conditions that may be encountered at Endeavor is shown in **Table 8**.

Table 8 – Summary of Ground Condition Types

Ground Conditions	Sketch	Description / Example
Slabby		•Continuous bedding planes •CSA siltstone and bedded ore
Closely Bedded		•Continuous and closely spaced bedding planes •Potentially very blocky ground •CSA siltstone and BV. Also, a chloritic foliation with variable orientation
Fault / Shear Zone		•Crushed rock and clay gouge •Strongly bedded and blocky ground, found in fault zone
Massive		•Discontinuous jointing •Very good ground conditions •Areas within MV orebody
Highly Stressed		•Variable ground conditions •Spalling from sidewalls and crushing in corners may occur •Sidewalls may start to buckle in weaker or strongly bedded ground •Massive and bedded ore in orebody pillars
Blocky		•Continuous joints forming large blocks •CSA siltstone and BV units

7.4.2 Intact Rock Properties

Limited laboratory testing was undertaken by AMC and is summarised in **Table 9** below.

Table 9 – Rock Property Test Results (mean)

Domain	Rock Type	Dry Density (t/m^3)	P-Wave Velocity (m/s)	S-Wave Velocity (m/s)	Young's Modulus (GPa)	Poisson's Ratio	UTS50 (MPa)	UCS (MPa)	Fracture Toughness (MPa√m)
Ore Zone	MS Pyrrhotitic (PO)	4.43	5,011	2,965	83	0.3	12	181	1.03
	MS Pyritic (PY)	4.52	6,408	3,851	153	0.28	17	198	1.36
	Siliceous (SIPY, SIPO)	4.02	6,987	4,140	118	0.22	16	263	1.56
BV	BV Contact Zone	3.67	6,842	4,142	73	0.22	10	146	0.93
CSA Siltstone	CSA Siltstone sandy	2.74	7,686	4,577	73	0.26	20	139	1.39
	CSA Siltstone silty	2.76	6,241	3,395	58	0.32	-	100	1.23

The following observations can be made from the results of the laboratory testing:

Ore zone

- Pyritic rocks (PY) are significantly stiffer than pyrrhotitic rocks (PO).
- The siliceous massive sulphides (SIPY, SIPO) have relatively lower Poisson's ratios.
- PO and PY have similar compressive strengths, but the pyrrhotitic rocks are weaker in tension.
- SIPY and SIPO have similar tensile strengths to PY, but are significantly stronger than both PY and PO.

BV contact zone

- Compressive strength and Young's Modulus for BV samples are lower compared with ore zone samples.
- Tensile strength and fracture toughness are lower in BV samples compared with ore zone and CSA siltstone samples.

CSA siltstone

- Both sandy and silty samples are softer and weaker than ore zone rocks. This may be a function of pervasive chloritic alteration and strong cleavage in this unit.
- Fracture toughness is similar to ore zone samples.

7.4.3 Structural Model

The nature of geological structures within the massive sulphides (MS) and CSA siltstones at Endeavor is summarised below:

Massive sulphides (MS)

Main Lode structures are dominated by flat NNE dipping joints (~15E/025E). They will form thin wedges in development and stope backs and will need to be supported. Steep north dipping joints (~85E/005E) are also present in the Main Lode, along with a wide range of randomly oriented, steep and shallow dipping joints.

Flat dipping structures have been observed in variable frequencies throughout the entire orebody. MS Pods are truncated and off-set by steeply dipping faults, striking NNW-SSE and NE-SW respectively. The immediate boundary between MS and altered CSA siltstone is characterised by strong quartz/carbonate veining and brecciation.

CSA siltstones

The orientation of bedding in the CSA siltstones is dominated by the steep east and west dipping limbs of Endeavor's D1 (first phase of deformation) folds (~60E/245° and 50E/065E). A wide scatter of bedding orientations occurs due to D2 (second phase of deformation) cross folding. Two steeply dipping cleavages (~85E/245E and 90E/125E) are associated with the D1 and D2 folds respectively.

Jointing is widely distributed and complex in the CSA siltstones. Flat and steeply dipping sets (~05E/025E and 80E/320E) are perpendicular to the D1 cleavage. Other joints dip moderately to the northwest and southeast (~45E/305E and 40E/130E) and are sub-perpendicular to the bedding.

Large, high wedges are possible in development backs, particularly in drives oriented NNW-SSE (sub-parallel to D1 fold-axes), due to combinations of steeply dipping cleavage/joints, moderately dipping bedding and shallow dipping joints. Arching development backs will significantly reduce support requirements in CSA siltstones.

Steeply dipping cleavages and joints, and numerous moderate to shallowly dipping cross-joints imply that all development walls in CSA Siltstone will be potentially slabby.

Brecciated vein (BV)

The brittle nature of the siliceous, quartz and siderite veined (BV) contact rocks implies that it will be prone to deterioration due to:

- Stope blasting; and
- Time dependent displacement of the blockier and more deformable adjacent CSA siltstones.

In general, remnant bedding (various dip and dip directions) with three poorly developed low-continuity (random) joint sets are found within the BV contact rocks.

Faults and shears

Endeavor's faults and shears dip steeply from roughly WSW to E (~85E/240E to ~85E/100E) and are likely to be related to conjugate/anastomosing shearing of D1 (first phase of deformation) cleavage around the massive sulphide pods. These shears and faults are sub-perpendicular to Endeavor's sub-horizontal major east-west principal stress direction.

7.4.4 Seismology

The Endeavor mine uses an IMS seismic monitoring system to record mining-induced micro seismicity. The system was commissioned in September 2006 with the following objectives:

- Assess trends between micro seismic activity and production or development.
- Identify regions where events are clustered or becoming more frequent or larger in magnitude.
- Identify trends in activity to validate exclusion or re-entry protocols.
- Identify areas where non-standard ground support regimes may be required to account for seismic loading.

Previous mining activity at the Endeavor mine has left most of the Main Lode and Northern Pods in a "de-stressed" or "shadowed" state and little seismicity associated with mining has been noted in recent years. Primary stoping in the 260 and 225 levels has also resulted in little increase in activity.

7.4.5 Principal Stress Orientations and Magnitudes

Stress measurements carried out using HI cells indicate that the major principal stress (σ_1) generally dips gently to the east, however two different directions could be present:

- Towards 080E, opposite the Northern Zone pods; and
- Towards 110E near the Main and Crusher Lodes.

The major principal stress is sub-parallel to Endeavor's flat northeast dipping structures. The Northern Zone pod orientation also makes a high angle with the D1 shears and steep northeast and southeast dipping faults.

The intermediate (σ_2) and minor (σ_3) principal stresses define a plane normal to σ_1;

- σ_2 generally dips 40E to the north; and
- σ_3 generally dips steeply 50E to the south.

As their magnitudes are similar it should be expected that their orientations "swap" between sites. It is likely that they are, "on average" equal and can be adequately represented as being horizontal and sub-vertical, respectively.

The following generalised relationships apply to the principal stresses and depth within the CSA siltstone at Endeavor (**Figure 15**):

- σ_1 = 0.060d,(080/00)
- σ_2 = 0.037d, (350/40)
- σ_3 = 0.022d, (170/50) where d = depth below surface (m)

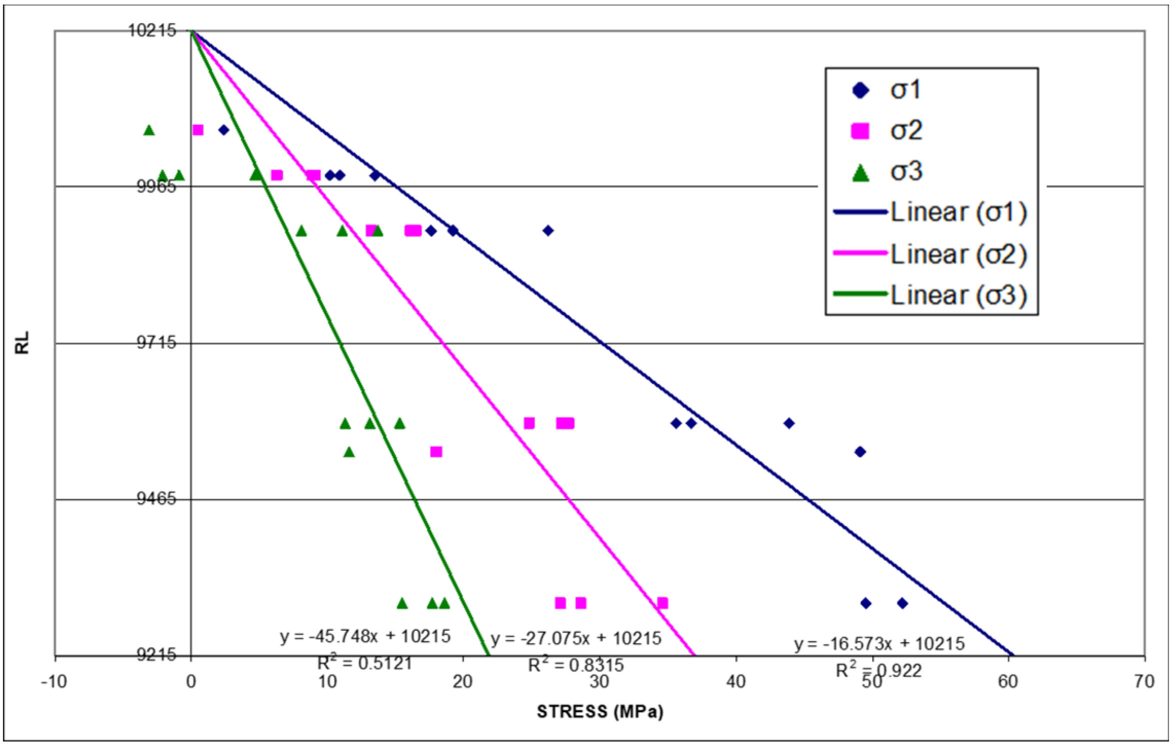


Figure 15: Principal Stress Magnitudes – CSA Siltstone

7.4.6 Rock Mass Classification

Rock mass classification methods are used to categorise the ground conditions in each of the geotechnical domains and guide ground support requirements. The two main rock mass classification systems that have been used at the Endeavor Mine are:

- Barton et al.'s Tunnelling Quality Index (Q); and
- Laubscher's Mining Rock Mass Rating (MRMR).

Following from the Q-system, the Modified Matthew's, or stability graph, method is used to provide indicative stable stope spans and associated support requirements.

It is important to note that the Q-system and subsequent empirical methods do not take into account major structures such as faults and shears, and therefore it is critical that these structures are identified during mapping and drilling programs and incorporated into the stope design process.

Table 10 is a summary of Q values obtained for the different rock domains at Endeavor

Table 10 – Summary of Q Values

Domain	Statistic	Q Back	Q Wall	ESR*	Drive Size	Span / ESR	Comment
Ore Zone (MS)	Mean	3.6	9.0	1.6	5.2m x 5.2m arched	3.25	Moderately stressed crowns and ribs
	Mininum	1.2	3.0				
BV	Mean	4.0	10.0	1.6	5.2m x 5.2m arched	3.25	Blast damaged (single structure assumed)
	Mininum	1.2	3.0				
CSA Siltstone	Mean	4.3	10.8	1.3	5.2m x 5.2m arched	4.0	Moderate stress
	Stressed	1.4	3.5				High stressed or blast damaged
	Blocky	0.2	0.5				Assumed shear zone

* ESR = Excavation Support Ratio

7.5 Excavation Design

7.5.1 Infrastructure, access and production development

Underground infrastructure and development excavations are designed to reduce (as far as reasonably possible) the exposure of the excavations to deterioration potential over their effective life. Factors influencing deterioration potential will vary for each design; however, a set of key design principles has been established as a guide to optimising design performance. These design principles include:

- Arched profiles should be mined unless local slab or wedge potential may be better managed using specifically modified geometries (e.g. mining to a lithology contact).
- Turn-out spans should be kept to a minimum, particularly in ore, to maximise stability conditions which will in turn minimise cable bolting requirements. Four-way intersections are avoided where possible to minimise spans.
- Stripping or turn-out fillets must be mined to an approved design.
- Parallel drives less than 5 m apart (pillar width) may require increased ground support to maintain stability and should be avoided where poorer ground conditions are encountered, or where significant stress redistribution associated with production may occur.
- The creation of brows in turn-outs or wide spans should be avoided.
- The location of ore drives must take into consideration stope reinforcement opportunities both for individual stopes and the overall production geometry of the completely mined ore block.
- Ground support standards for the routinely used drive profiles used at the Endeavor Mine have been engineered and used for a number of years.

7.5.2 Production

Historically, the sequence of production was considered in terms of:

- Intra-stope sequence;
- Inter-ore body sequence; and
- The sequence between discrete stoping areas.

Consideration of these factors will influence development access methodologies and backfilling strategies.

Due to the diversity of the geometry, continuity and orientation of the deposit, that primary and secondary stoping has largely been completed in both the Main Lode and Northern pods, and the proposed mining operation will move into a phase of pillar removal and remnant stoping while the Deep Zinc Lode is developed, the approach to managing production excavation risk needs to take into consideration the following key parameters where applicable.

Intra-stope sequence: with consideration of the following factors:

- Limiting the need for charging up-holes at open brows.
- Limiting the production risks associated with blind up-hole slot firings.
- Managing loss of containment at stope brows by the pre-placement of cablebolts at interim and final stope brows.
- Stress redistribution including destressed “cantilever” type failures from unconfined pillars.
- Slot winze caps should not have a vertical thickness to undercut-span ratio less than 1:1.
- Ring firing sequences must consider temporary or undercut spans over which access is needed for subsequent production.
- Undercut spans with a vertical-thickness to undercut-span ratio less than 1:1 must be avoided and may need to be increased where regional structures or poor ground are encountered.
- Remnant operations often involve utilisation of drives for drilling and charging platforms that have been mined in the past with no consideration of future stoping. Blast sequencing should consider this and avoid re-entry into wide spans where possible.
- If drilling “just in time” strategies are being employed due to poor ground conditions or for scheduling constraints, a minimum of a 3-ring drilled buffer should be maintained where possible.

Intra-ore body sequence:

- Secondary stoping blocks must be maintained with sufficient dimensions to prevent difficulty establishing production drilling or mucking access and minimise the requirement to mine development within back-filled areas.
- Production from secondary ore blocks will result in combined stope back spans where confinement from fill is likely to be poor. Cable-bolting of both primary and secondary stope spans will reduce the likelihood of both individual stope dilution and late-stage production dilution or instability.
- In tertiary or remnant stopes, access for stope crown cable-bolts may be difficult. Consideration should be given to rib pillars to break spans under these circumstances. Site experience is that these rib pillars should be a minimum of 5m thick.

- The Endeavor Mine has used a variety of fill methods in its history. Site experience is that these should be assumed to lack cohesion.
  - Where the fill is paste, a small crown pillar should be left when mining under or adjacent to fill. Again, this should be a minimum of 5m thick.
  - Where the fill is loose rock it is proposed to stabilise the fill adjacent to the planned stope with injected grout for a minimum 3m thick stabilised zone.
- Significant overbreak and caving was encountered in portions of the primary and secondary stoping. Survey methods utilised in some older stopes did not include CMS void measurement, or this was not feasible due to stope failure. Extensive probe hole drilling campaigns were conducted prior to development and tertiary stoping in these areas to minimise risk associated with proximity to historic voids.

Sequence between ore lenses:

- Where the recovery of a stope or stopes may initiate a regional or widespread stress response, most likely associated with regional structures, the impact of this on other active production or backfilling areas must be considered.

7.6 Ground Support

7.6.1 Minimum Ground Support Standards

Support systems are designed and selected for the expected life and serviceability of excavations. All support systems for permanent openings are designed with a life expectancy of greater than three years or as a permanent mine opening. Support for temporary openings (i.e. ore drives) is designed for a minimum life expectancy of three years.

The Endeavor Mine has twelve primary minimum ground support standards based on profile size and application as listed in **Table 11**.

Ground support standards are to be reviewed and updated as required to better reflect the actual ground conditions and underground development. Important considerations when assessing the suitability of ground support and application of support standards include:

- Galvanised support used to minimise corrosion.
- All development is to be surface screen supported to within 2.5 m of the floor.
- Cable bolts to provide deep ground support for spans over 6m where normal rock bolt lengths are insufficient.

Table 11 – Minimum Ground Support Standards

Profile Code	Minimum Ground Support			
	Backs & Shoulders	Walls	Row Spacing	Surface Support
DGSS_1 5m x 5m ARCHED	7 x 2.4m Split Sets	3 x 2.4m Split Sets per side	1.4m	Mesh to 0.5m above floor
DGSS_2 5m x 5m ARCHED	7 x 2.4m Split Sets	2 x 2.4m Split Sets per side	1.4m	Mesh to 1.5m above floor
DGSS_3 5m x 5m ARCHED	7 x 2.4m Split Sets	2 x 2.4m Split Sets per side	1.4m	Mesh to 2.5m above floor
DGSS_4 5m x 5m ARCHED	7 x 2.4m Split Sets	3 x 2.4m Split Sets per side	1.4m	FC to 0.5m above floor 50mm thick
DGSS_5 5m x 5m ARCHED	7 x 2.4m Split Sets	2 x 2.4m Split Sets per side	1.4m	FC to 1.5m above floor 50mm thick
DGSS_6 5m x 5m ARCHED	7 x 2.4m Split Sets	2 x 2.4m Split Sets per side	1.4m	FC to 2.5m above floor 50mm thick
DGSS_7 REHAB REHAB PROFILE	2.4m Split Sets as required	2.4m Split Sets as required	1.4m	Mesh as required FC as required
DGSS_8 DECLINE 5m x 5m ARCHED	7 x 2.4m Split Sets	2 x 2.4m Split Sets per side	1.4m	Mesh to 1.5m above floor
DGSS_9 5m x 5m SQUARE	5 x 2.4m Split Sets	3 x 2.4m Split Sets per side	1.3m	Mesh to 2.0m above floor
DGSS_10 5.7m x 5.5m ARCHED	5 x 2.4m Split Sets	4 x 2.4m Split Sets per side	1.4m	Mesh to 1.7m above floor
DGSS_11 6m x 5.5m ARCHED	5 x 2.4m Split Sets	3 x 2.4m Split Sets per side	1.4m	Mesh to 2.3m above floor
DGSS_12 6m x 5.5m ARCHED	5 x 2.4m MD Bolts	2 x MD Bolt & 1 x Split Set per side	1.4m	Mesh to 2.3m above floor

7.6.2 Wide Spans - Intersections

Wide spans are defined as excavations having a span greater than 6 m. Spans greater than 6 m require deeper reinforcement than routine development profiles to manage increased exposure to structures in the rock mass. The cable bolt reinforcement design method used at the mine follows typical Australian practice. Ground support at wide spans is managed using plated and grouted cable bolts. The cable-bolting requirement calculation defines the minimum number of cables required for spans of a nominated size. The positioning of cable bolts is determined by engineering personnel after assessment of the span geometry. The following are key requirements in the management of all wide spans:

- First-pass turn-out cables are installed prior to excavating wide spans.
- Second-pass cable installation is completed as soon as practical following the development of wide spans. Second-pass cable installation (and plating) is to occur and no later than 1 full cut from a drag or within 1 week of developing the wide span.
- All cables are plated (1 strand on twin installation). First-pass cables may be plated in conjunction with second pass cables.
- Where continuous wide spans are to be mined, the Geotechnical Engineer is to assess the requirement to split the planned strip into multiple discrete advances of firing and support.

7.6.3 Wide Spans – Continuous

In other wide spans, such as decline passing bays, the volume of rock to be supported is assumed to be a parabolic prism along the length of the wide span. Using the following method:

Prism mass = 2phS/3x; *where x is length and S is cross section span*

This mass of rock can be used to calculate the number of cable bolts required in the excavation on a linear basis.

With due consideration of the rock mass conditions and abutment off-sets, the cable bolt installation pattern should be defined as a square or staggered pattern of uniformly spaced rings of cable bolts as determined by the Geotechnical Engineer.

7.6.4 Production Spans

Stopes are designed to be self-supporting only to a standard sufficient for production within acceptable dilution limits, and not to permit personnel access. Unlike development spans, production spans are mined with partial or limited access to final stope spans and reinforcement opportunities are limited in extent.

Stope reinforcement design should take into consideration the local stope span size and geometry, and the size and geometry of the combined spans of primary and secondary stopes.

The function of cable bolt reinforcement is to limit dilution associated primarily with rock mass structures and stress redistribution at the excavation surface. Practical limitations on cable bolt installation densities and spatial distribution over the stope spans exist and will vary for stoping at Endeavor.

As a guide, the assumed dead weight to be reinforced by cable bolts is arbitrarily limited to a maximum depth of one fifth of the production span at the centre of the span with the following additional considerations:

- Ore drives on cable bolting horizons should be designed to optimise the central location of cable bolt arrays for discrete stopes and take into consideration access location and reinforcement opportunities in existing or planned adjacent stopes.
- Stope design should avoid unfavourable pendant or convex exposures which cannot be practically reinforced using cable bolts.
- The spatial restrictions on cable bolt installation will result in locally reinforced areas of the final stope span being separated from unsupported regions. Cable bolting stopes will not completely eliminate the risk of dilution or prevent local failure if stope spans are excessive.
- All cables used in support of production spans are either single or double strand 15.6 mm diameter cable.
- Ore drives in all areas of the mine are assessed for cable bolting requirements taking into account the expected geotechnical conditions during stoping.
- Drawpoints and stope brows are cable bolted where geotechnical conditions are considered to be detrimental to brow stability.
- Depending on the stoping method and ground conditions, the ore drives are cable bolted with 6 m long, twin strand cable bolts.

7.7 Stope Backfill

In determining the appropriate backfill system (cemented or uncemented Waste Rock Fill or Paste Fill), the following are to be taken into account:

- Stope planning.
- Fill delivery.
- Stope filling and drainage.
- Fill mass exposure.
- Testing procedures.
- Documentation (including stope notes, barricade notes, and quality assurance testing).
- Communication (meetings and reports) and auditing requirements.

Where backfill is used in the larger stopes, the type and strength will depend on the dimensions of the stope and the future exposure of the fill. If a fill exposure is designed, the strength of the cemented fill will depend on the exposed height and width. The final strength requirements must be determined for each individual exposure through risk assessment and design. Where no post-filling exposure will occur, unconsolidated waste rock is utilised.

Any exposure of cemented fill requires geotechnical assessment, whether as an unsupported production span or a supported development excavation. The following guidelines have been adopted to minimise geotechnical risk associated with fill exposure in development:

- Arched profiles are to be mined.
- A row of closely spaced uncharged perimeter holes should be drilled to protect the fill-mass from blast damage and maintain the design profile.
- Turnouts and wide spans have specific ground support requirements.
- Access to established fill should be restricted where there is likely to be a change in the state of stress in the fill-mass without appropriate monitoring controls.
- No backfill tipping point may be established at the edge of an unconfined fill mass.

The quality of filling is crucial for subsidence control, to minimise the potential for long term caving of stope crowns.

7.7.1 Waste Fill

Crushed CSA siltstone was used as rock fill prior to 1989, then mixed with 4 to 6 percent cement until 1993. The use of CSA siltstone as fill was discontinued in September 1995 due to its poor drainage properties.

Waste fill (mostly calcrete rock from a nearby quarry) was used during and after 1996 as thickened hydraulic fill to fill several of the upper-level stopes. Weathered siltstone excavated from the existing tailings storage facilities has been placed in the stockpile over the caved zone, which is referred to on site as Mt Elura. Experience with the uncemented hydraulically placed material is that it did not drain and remained as low strength slurry.

Run of mine development waste (including any waste material from over-break, fall of ground, etc.) has replaced the quarried siltstone and was most recently the main source of waste fill material used for stope filling. The suitable waste material needs to be hard enough to remain in lump form during transport from source to stope – i.e. not be reduced to fines. It must have free draining properties to prevent water build up in transfer passes and stopes.

7.7.2 Paste Fill

Stopes to be paste filled are designed and selected to allow the extraction of ore adjacent to the mined-out stopes for the expected life and serviceability of the excavations. Paste fill delivery to the stope is made by a reticulation system with pipes and connections rated to handle the maximum pressure for a required condition.

Vertical sections of the reticulation system are drilled through the rockmass and were generally steel cased. Regular inspections of the reticulation lines were incorporated in the general maintenance to ensure the safe delivery of the paste fill.

Mined out stopes are barricaded at the draw points to contain the paste fill.

The paste filled stopes provides the required support to the adjacent stope to allow the mining of secondary stopes.

7.8 Ventilation

7.8.1 Existing Ventilation System

The current primary ventilation system comprises 3 primary extractive ventilation fan installations servicing the underground workings at the Endeavor Mine.

- Three upcast shafts
  - WVR # 1 Surface – 5 Haul (3.0m raise bore)
  - WVR # 2 Surface – 4 Haul (3.75m raise bore)
    - NEVR 6z4 to 290 (3.0m raise bore)
  - Allied Shaft (EVR) Surface – 3 Haul (1.8m raise bore)
- Two intake shafts
  - Main Shaft Surface to 4 Haul (6.0m raise bore)
  - Exploration Shaft Surface to 4 Drill (3.0m raise bore)
- Main decline (intake 4.5m x 5.0m)
  - Old Decline to 4 Haul 4.5m x 3.5m
  - New Decline to 3 Haul 5.0m x 5.0m

The Main Decline is the chief source of fresh air until the 260 level where the bottom of the Exploration Shaft is providing fresh, cool air flows, almost uninterrupted from the surface.

7.8.2 Deep Zinc Lode Ventilation

A preliminary report on heat modelling in the mine (2018) recommended the installation of an intake air cooling facility to keep temperatures in the Deep Zinc Lode development areas at acceptable levels during summer. The report indicated the mine would require 1.5MW of cooling at the Exploration Shaft collar during initial development, increasing to 4.5 MW when the Deep Zinc Lode is in full production.

A 4.5MW Water Cooled Bulk Air Cooling (BAC) Plant has been designed, with plans for it to be situated at the collar of the Exploration Shaft. The installation of the BAC plant has been approved by the Cobar Shire Council and costs included in the mine restart capital estimate.

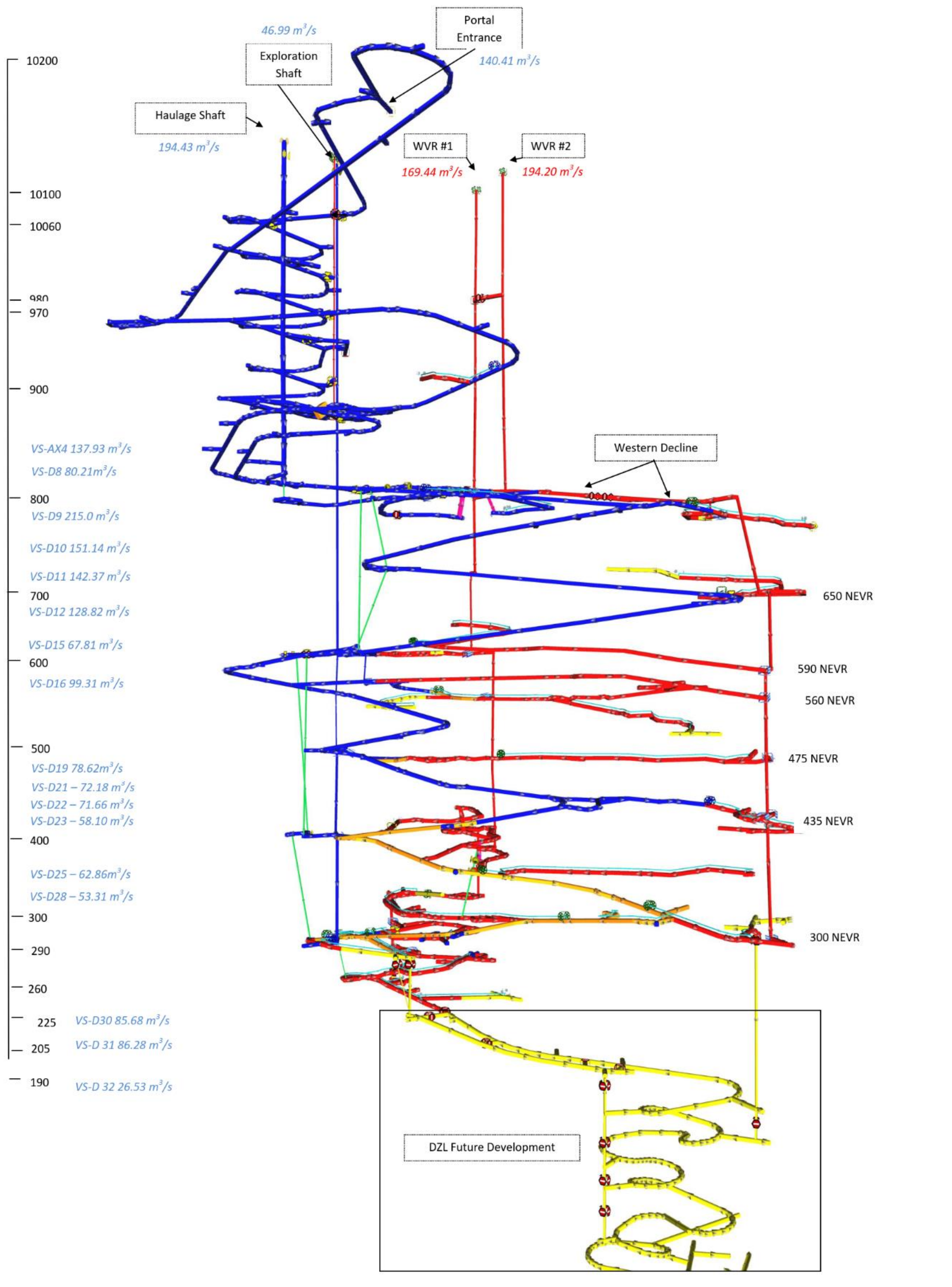


Figure 16: Primary Intake and Exhaust Circuits

7.9 Dewatering

The water table in the area around the Endeavor mine is 60 to 80 m below ground level with groundwater occurring in fracture zones associated with the folding of siltstones around the orebody. Groundwater yields from the fracture zones are very low, occurring as seepage only.

Regionally where the siltstones have not been folded creating axial plane fractures, groundwater occurrence is probably even more sporadic. There are no known landholder bores in the area due to the depth of the water table, the very low yields and the poor water quality making it unsuitable for stock use.

It is estimated that the total water ingress from the water table is a maximum of 3-5 L/s, accumulated across many small seeps in the decline, predominately from the sandstones and siltstone rock units. Pumping volumes average around 9 L/s when process water (primarily from drilling operations and paste fill operations) are considered.

As a result, the risk of water inrush from aquifers is considered as extremely low.

Surface catchments are separated from the mining operation and are generally situated at a lower elevation than the mine infrastructure. As a result, the risk of inrush from tailings dams etc. has been assessed as low.

The rock mass at the Endeavor Mine is generally very competent. Deterioration of the rock mass due to water is considered as low risk.

Water quality is periodically monitored. Results are generally poor, with high EC and pH's generally recording around 3. This requires that longer life ground support utilises galvanised ground support elements to minimise corrosion.

8 Mine Plan

Ground Control Engineering (GCE) were commissioned by Polymetals to undertake a review of underground geotechnical conditions, generate mine designs, and produce a Life Of Mine (LOM) schedule incorporating the Upper Main Lode, Main Ore Body and Deep Zinc Lode areas of the Endeavor Mine (**Figure 17**). Mining costs were generated from first principles based on an owner-operator model.

Mining of tailings from the TSF Sector 1 is based on a 2015 study by CBH Resources which concluded that hydromining of the tailings is the preferred mining method.

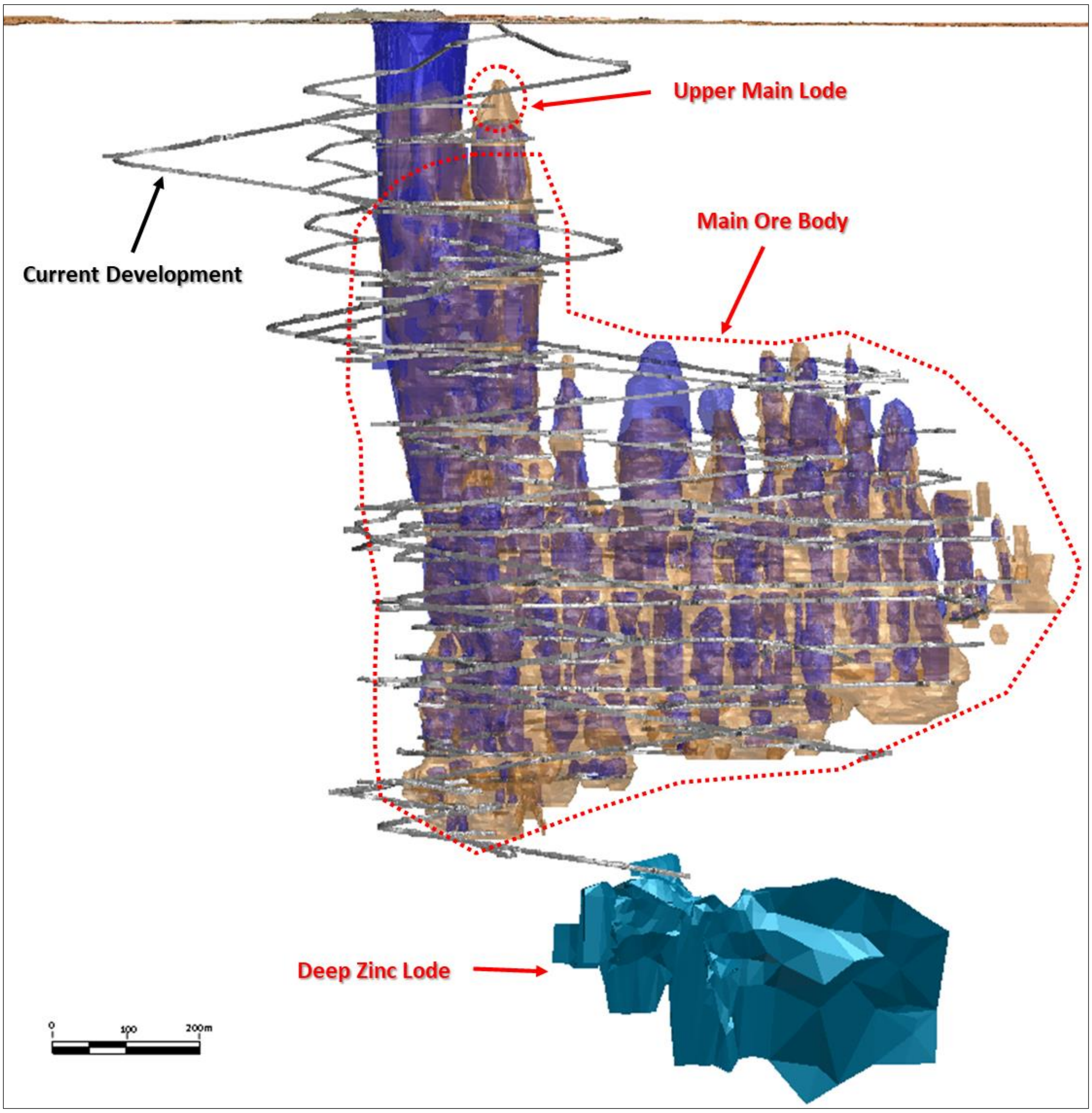


Figure 17: Underground Mining Areas

8.1 Underground Mining

Underground mining at the Endeavor Mine is proposed to be undertaken by industry standard methods, most of which were previously utilised for production at the mine. As mentioned previously in Section 7.3, extraction of ore will be by a combination of Long Hole Open Stoping (LHOS) in the Main Ore Body, Sub Level Stoping (SLS) in the Deep Zinc Lode and Cut & Fill method in the Upper Main Lode.

8.1.1 Stope Optimisations

Stope optimisations were based on Net Smelter Return (NSR) values that have been assigned to each block in the resource block model based on calculations using the assumptions shown in **Table 12**.

Table 12 –NSR Calculation Assumptions

Metal	Metal Price	Exchange Rate	Flotation Recovery			Smelting Recovery	Smelting and Freight costs per tonne
			Below 10080mRL	Above 10080mRL	DZL		
Pb	US$2,076/t	AU$1= US$0.70	75%	77%	-	95%	$523
Zn	US$2,915/t		84%	76%	90%	85%	
Ag	US$22.4/oz		52%	57%	52%	95%	

The formula for calculating NSR value of each tonne of material is:

$$\mathbf{NSR(x_1, x_2, x_3) = x_1r_1p_1(V_1) + x_2r_2p_2(V_2) + x_3r_3p_3(V_3) - (C_s + C_t)/K}$$

Where:

x_1, etc = Grade of metal 1, etc

r_1, etc = Floatation Recovery of metal 1, etc

p_1, etc = Smelting Recovery of metal 1, etc

V_1, etc = Value of metal 1, etc

Cs + Ct = Smelting and freight costs per tonne of concentrate

K = Tonnes of ore required to make one tonne of concentrate

An NSR value of $150/t, based on historic mining and processing costs on site, was utilised for the stope optimisation process. The stope shapes were generated using Deswik Stope Optimiser (SO) using a minimum strike of 5m and attempting to align the height of the stopes with the existing level intervals. Post processing was completed to eliminate shapes with a volume below 500 m^3 and any part of the stope shape within 5m of a previously mined stope.

8.1.2 Mine Design

8.1.2.1 Upper Main Lode

Mine planning for extracting ore above the existing 2 Drill Level in the Northern Pod of the Main Lode is anticipated to be challenging due to variable ground conditions, partial failures of previous development, and the presence of voids. A number of mining studies have been completed in the past which have put forward conceptual plans (Keasehagen 2002, Coffey Mining 2006, Mining One 2007, AMC 2011).

Prior to mining, filling and stabilisation of voids will be required to provide a safe platform for mining activities (**Figure 18**). Broken ground is planned to be stabilised by low viscosity grout, while voids are planned to be filled with cement aggregate and/or expanding phenolic foam.

Mining of the Upper Main Lode is proposed to be undertaken using a combination of Sub Level Stoping and Cut & Fill methods. A Cut & Fill method with pillars between drives, is to be used above 10090mRL due to potentially poor ground conditions compared to the deeper mining areas. This conservative approach to mining of the very high value material in the Upper Main Lode is intended to mitigate the risk of any ground failure sterilising the resource in this area. If ground conditions are found to be favourable during initial mining, scope remains to adjust the Cut & Fill mining method to a lower cost option (i.e. Sub Level Modified Avoca Stoping) and which may also include pillar recovery.

Filling of voids and Cut & Fill drives is proposed to be carried out by a combination of cemented rock fill (CRF) and cemented aggregate fill (CAF), depending on accessibility and fill strength requirements.

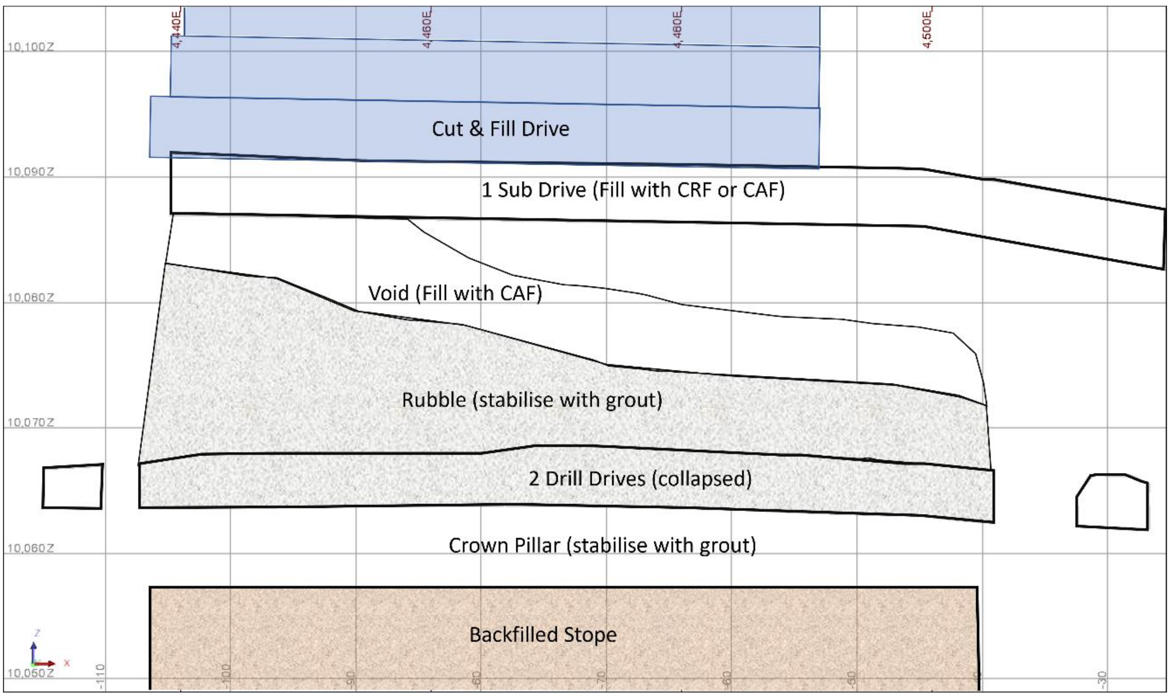


Figure 18: Cross Section Through Upper Main Lode Void Showing Stabilisation Plan

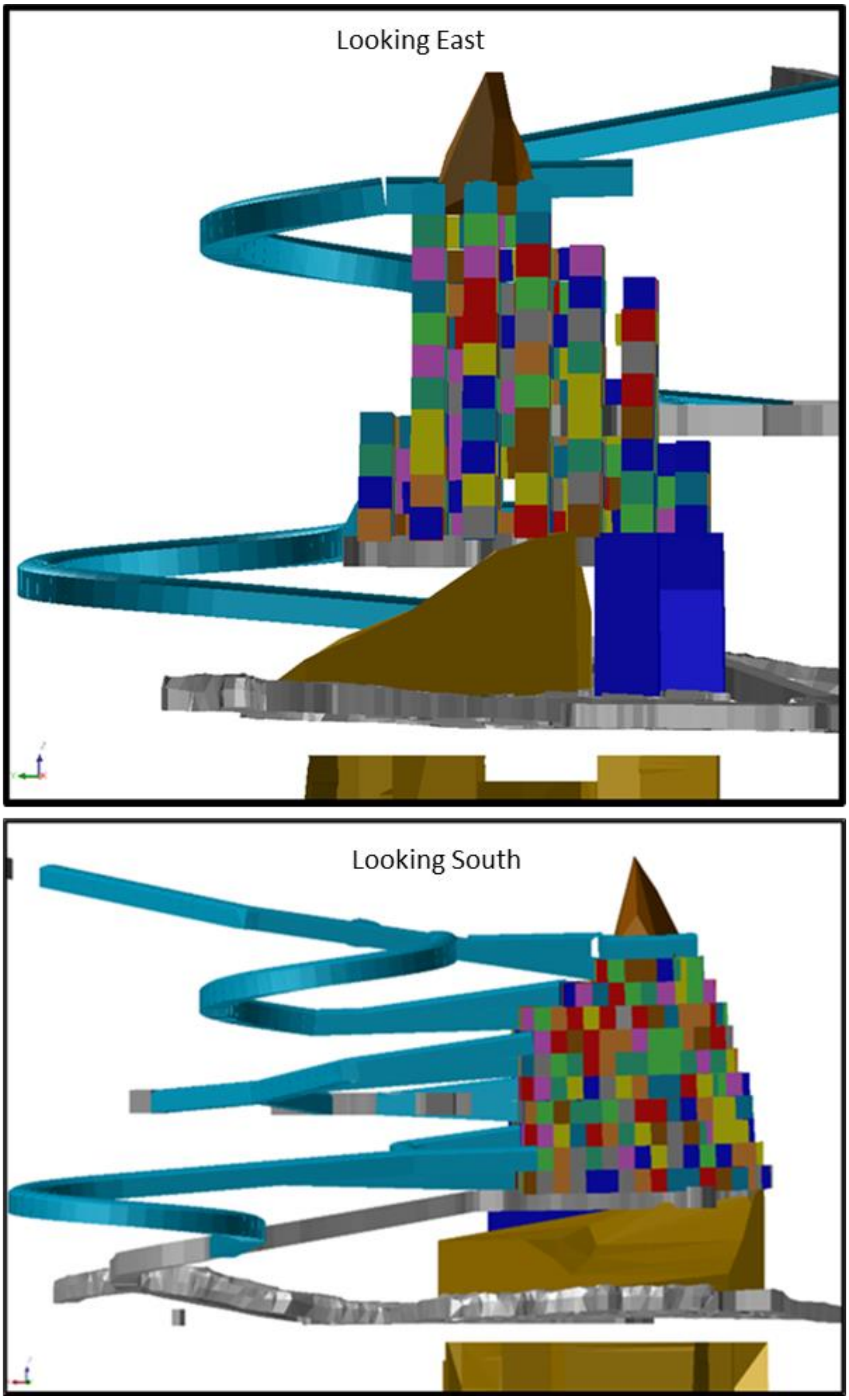


Figure 19: Stope and Development Designs – Upper Main Lode

8.1.2.2 Main Ore Body

Remaining Main Ore Body material is proposed to be mined using Long Hole Open Stoping methods, with minor amounts of unconsolidated rock fill to facilitate double-lift stoping. Where stopes have been designed adjacent to previously mined stopes containing loose rock fill the 5m “stope skin” has been included in the mine plan. It is proposed to stabilise the rock fill by consolidating a 3-5m thick pillar of fill using injected grout to enable full recovery of the stope skin (**Figure 21**).

Pillar recovery through injection of low viscosity grout into loose rock fill to form artificial Stabilised Rock Fill (SRF) has been successfully carried out in the past by a number of mines such as Crusader (Sainsbury *et al*, 2003), Cracow (Potts *et al* , 2012), Ballarat (Sainsbury *et al*, 2014), Toguraci (Proudman *et al*, 2017) and Pajingo (McTyer and Carswell, 2018),

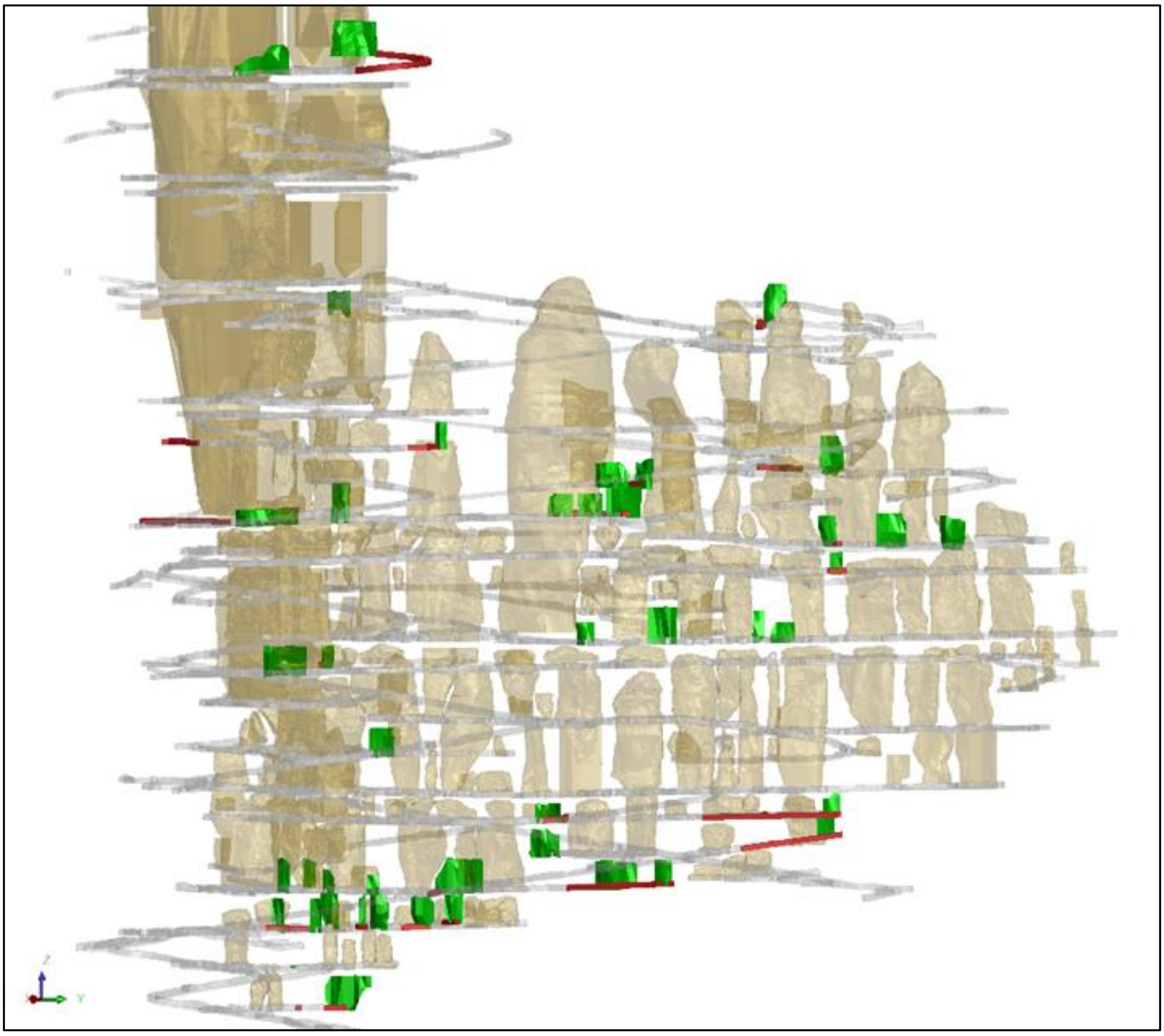


Figure 20: Stope (green) and Development (red) Designs – Main Ore Body

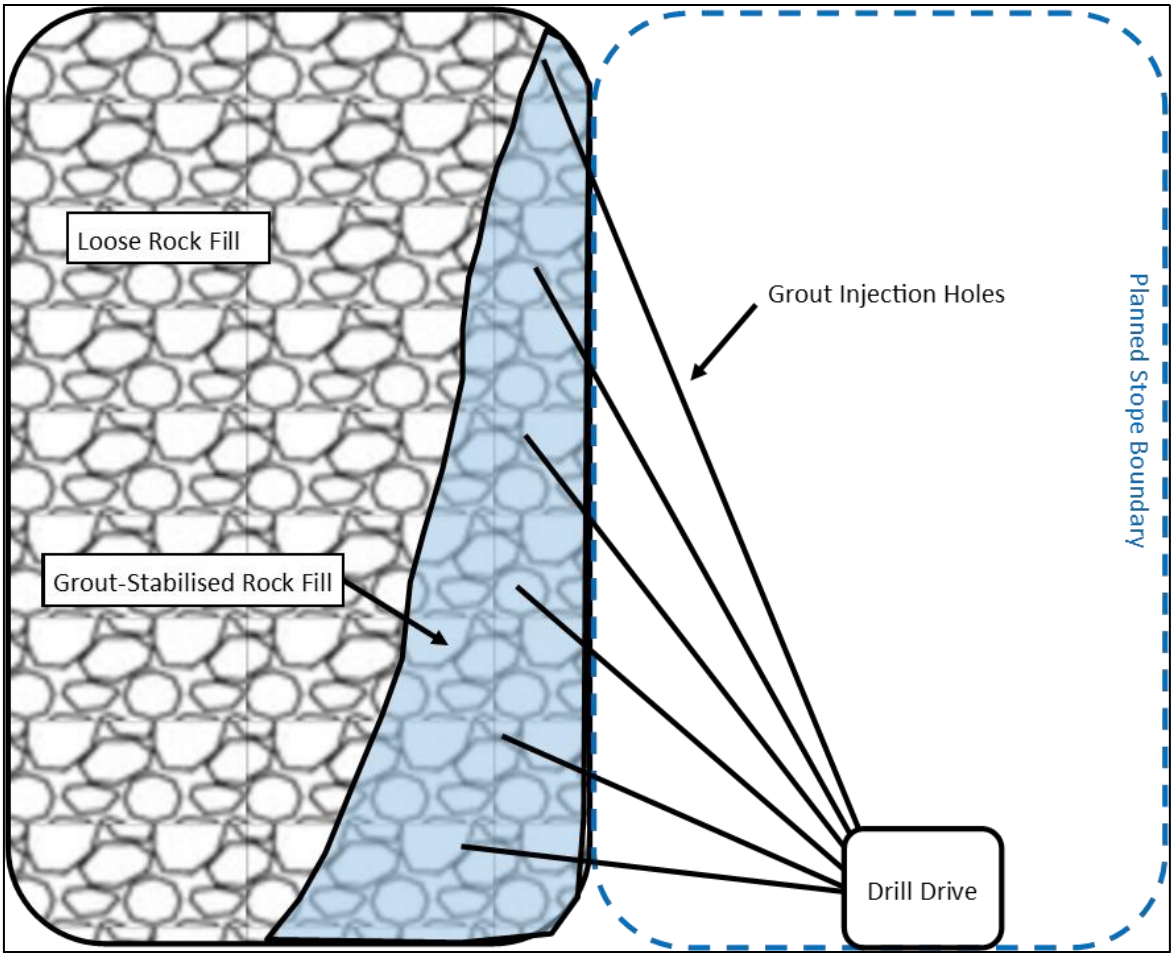


Figure 21: Example of Stope Fill Stabilisation by Injection Grouting

6/6 Stope

In 1999 a mass firing of 450,000 t of ore was carried out in the 6/6 Stope. Ore extraction was halted after approximately 150,000 t had been removed from the drawpoints on 6 Haul after a void developed within the stope, raising fears of an air blast like that which had occurred at North Parkes the same year. Subsequent investigations into the size of the void by drilling and using conventional cavity surveying methods as well as seismic tomography have provided more precise information on the size and location of the void. This information, combined with the drawpoints having become choked with material, has mitigated the potential for a sudden inrush of air from the stope.

A mine plan and design has been prepared to attempt to retrieve approximately 300,000 t of broken ore remaining in the 6/6 stope. The proposed plan involves the establishment of a new access drive to connect to the three draw points on 6 Haul and allow enough distance for bogging activities. Rings are to be drilled from the 3 draw points to fire the A pillar into the trough and recover the material hung up in the stope. A drill horizon will be established on the 665 Level from which shaker holes can be drilled if hung up material does not dislodge from the A pillar firing.

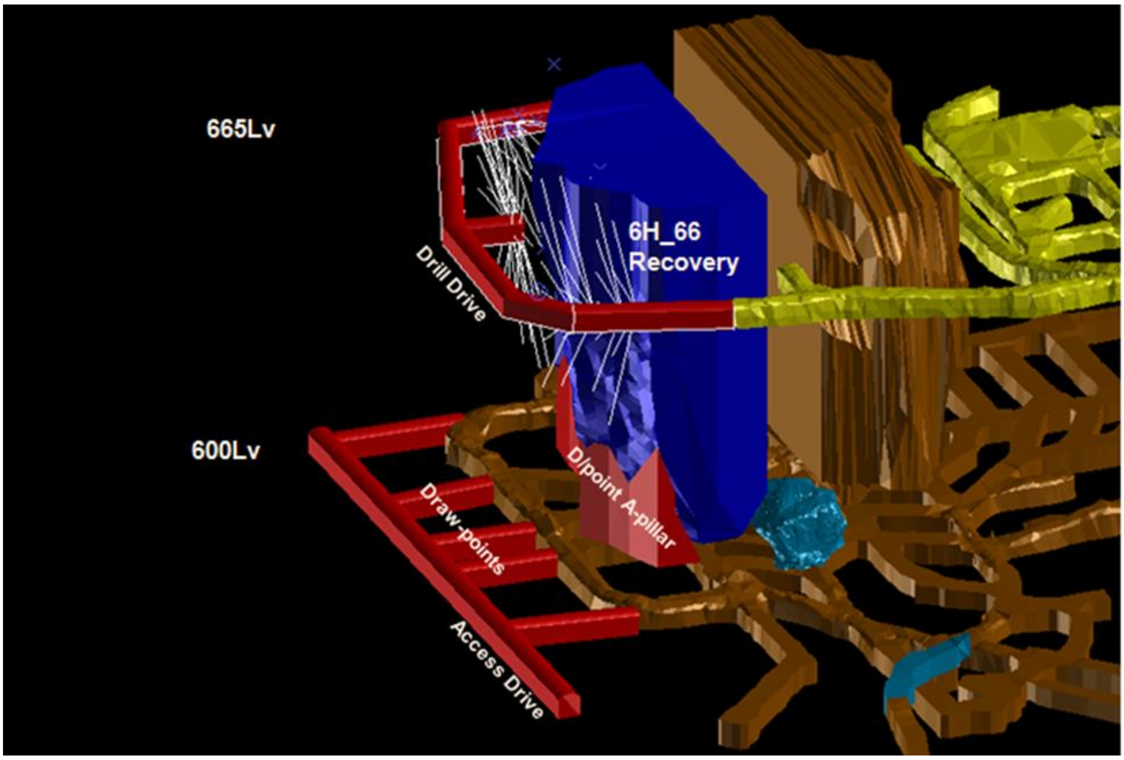


Figure 22: 6/6 Stope Recovery Plan

8.1.2.3 Deep Zinc Lode

The Deep Zinc Lode (DZL) is proposed to be mined by Sub Level Open Stoping, mined in two lifts, separated by a crown pillar approximately halfway up the mining block. A combination of loose and cemented rock fill will be used to maximise pillar recovery. As each panel will be mined from the bottom up, a form of Avoca mining may be used to minimise the requirement for cemented fill.

Waste material required for stope back fill will be sourced from lateral and vertical development headings, as well as waste material currently stored in various stockpiles and disused drives throughout the mine. Cement for the reinforced backfill will be delivered to the required area by mobile agitator or reticulated lines.

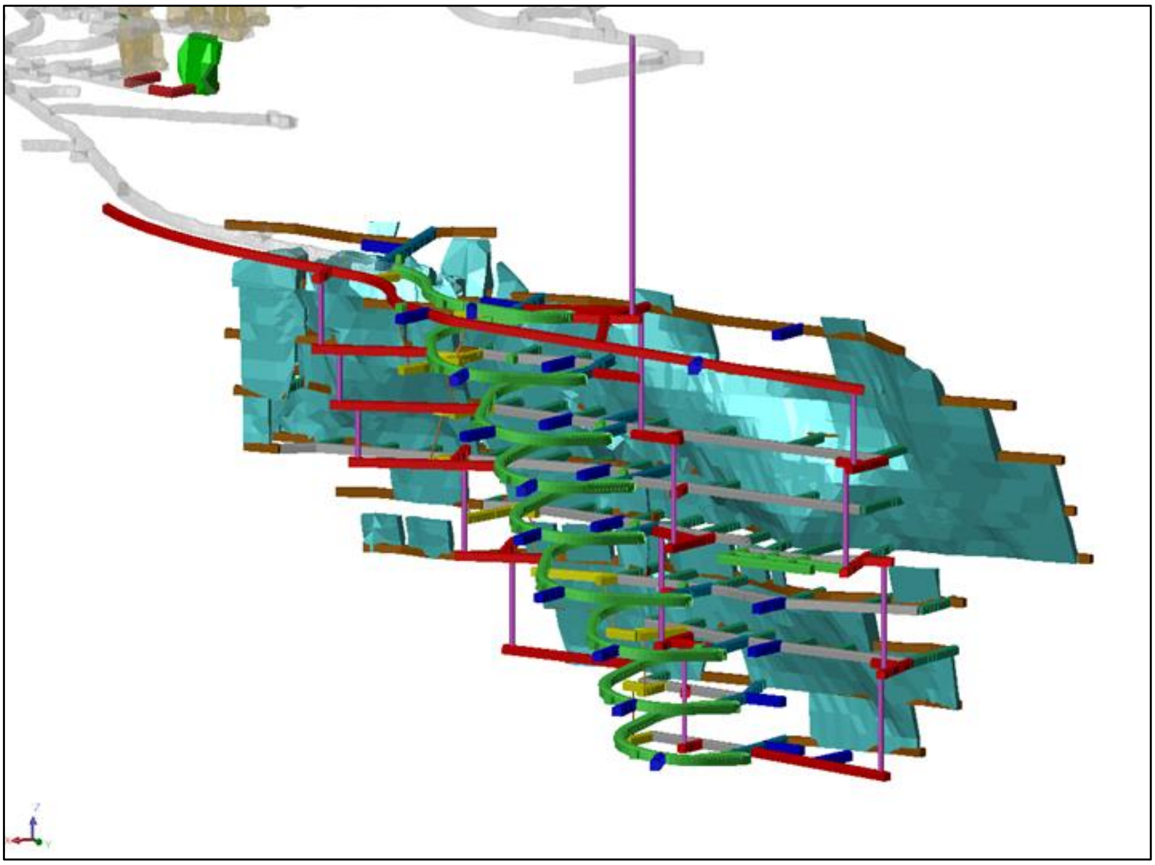


Figure 23: Stope and Development Designs – Deep Zinc Lode

8.2 Tailings Mining

The Tailings Storage Facility (TSF) at the Endeavor Mine is divided into 4 sectors. From the commencement of production in 1983, through to 1989, tailings from the flotation process were deposited into Sector 1 of the TSF. It is proposed to mine and re-process Sector 1 tailings towards the end of production from underground mining.

8.2.1 Mining Method

A tailings reclamation report prepared by CBH (Kastanas 2015) investigated the optimal method for mining the tailings and recommended the use of high-pressure water monitors (Hydromining). This method uses a top down approach utilising a monitor to sluice the tailings into a channel which connects to a sump/pond for slurry pumping back to the Mill, and another monitor for cleaning up stubborn tailings which hang up along the tailings channel catchment (**Figure 24** and **Figure 25**).

Recovery of tailings will be affected by the need to retain berm batters to provide TSF stability and support for critical access roadways peripheral to Sector 1, as well as a central pillar/containment barrier between the northern and southern mining zones. Mining recovery is estimated to be around 95%.

Dilution is regarded as less of an operational problem, as it is assumed that Hydromining, if properly managed, should attempt to sluice the majority of the tailings inventory without overly scouring the TSF bedrock profile.

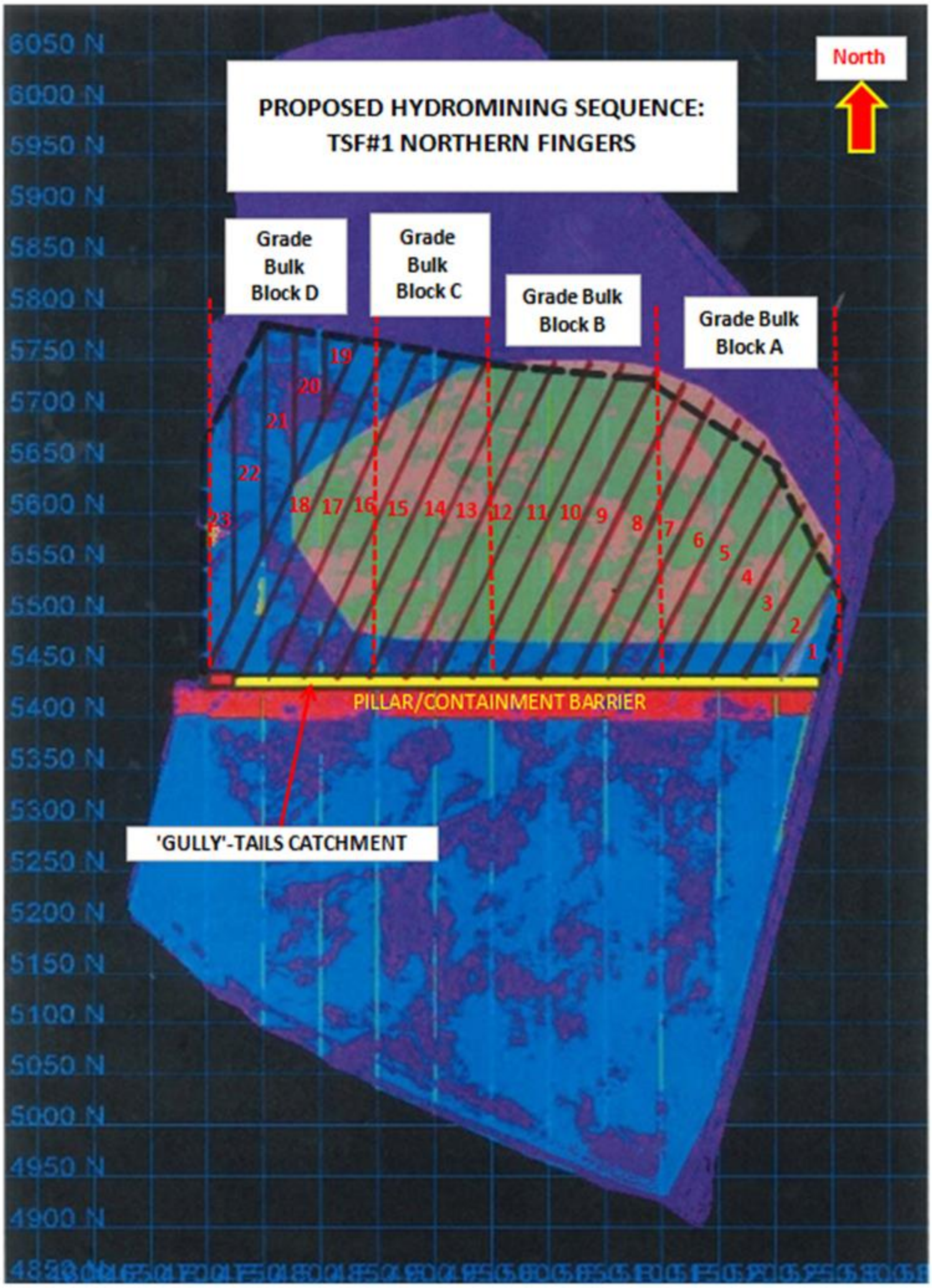


Figure 24: Example Layout for Hydromining TSF Sector 1 Tailings Nth Zone

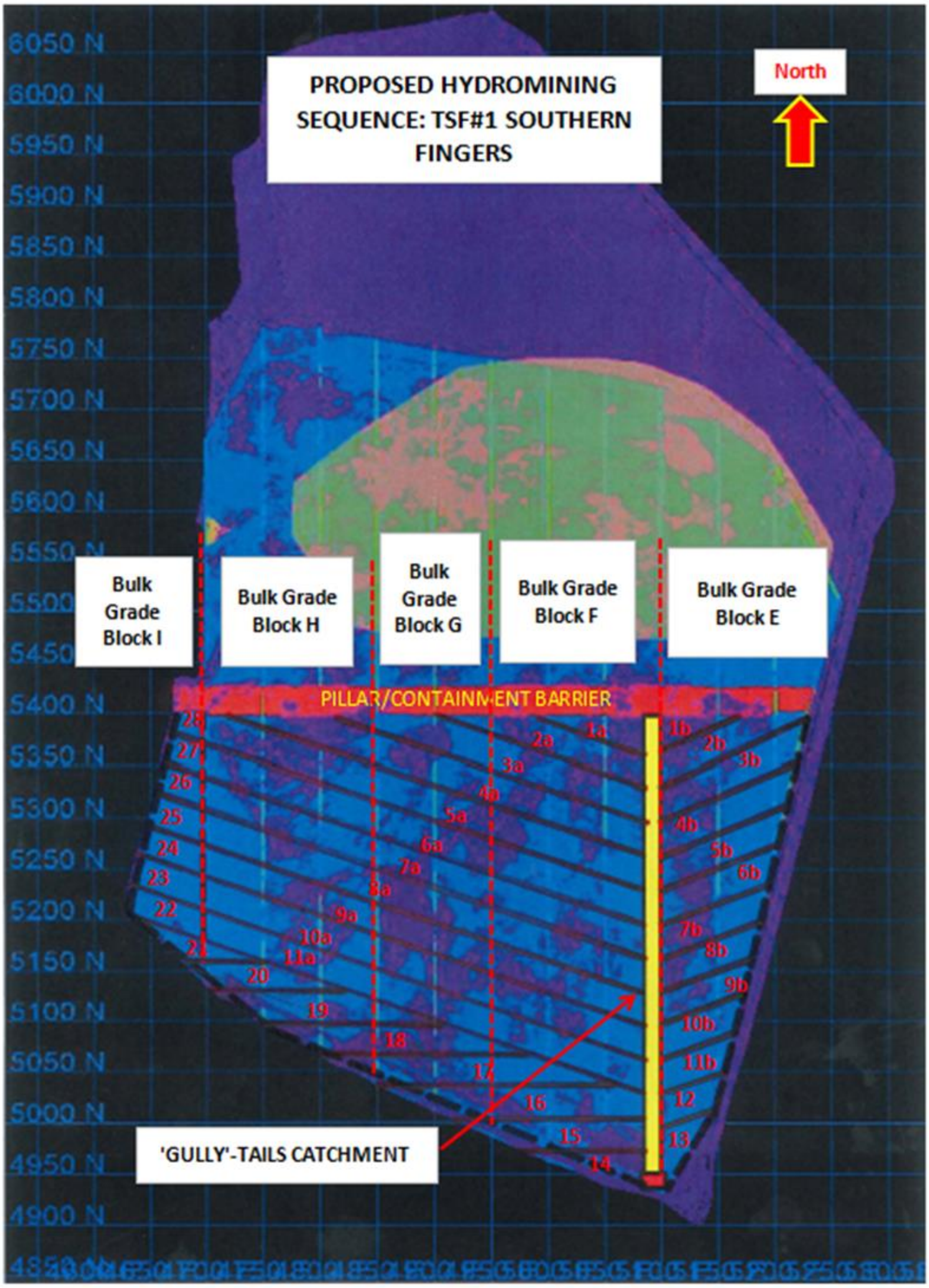


Figure 25: Example Layout for Hydromining TSF Sector 1 Tailings Sth Zone

8.3 Life of Mine (LOM) Production Schedule

Underground mine production schedules were generated on a monthly increment by Ground Control Engineering using Deswik.Sched underground scheduling and mine planning software. A number of scenarios were run to find the optimal production sequencing and mining rate for maximum project NPV.

8.3.1 Resource Models

The Resource models provided to Ground Control Engineering and used in the mine design and scheduling activities for the underground mine were the updated Main Ore Body model (2023) and the Deep Zinc Lode model (2019). The Resource model used for scheduling of the tailings retreatment was produced in 2023.

8.3.2 Input Parameters

The parameters used for the mine scheduling process are shown in **Table 13** to **Table 16**. Mining dilution and ore loss assumptions are based on historical development and stope reconciliations at the Endeavor Mine. Dilution has been assumed to have zero grade and provides a conservative estimate of production grades. Actual dilution grade will vary depending on location.

Table 13 – LOM Schedule Parameters - Resources

Resource	Number	Rate
Bogger - Backfill	1	800 t/d
Boggers - Stope	2	1,850 t/d
Cabletec	1	90 m/d
Jumbo	3	200 m/month
Paste Plant	1	1,000 t/d
Production Drill	2	200 m/d
Raisebore	1	3 m/d

Table 14 – LOM Schedule Parameters – Schedule Constraints

Schedule Constraint	Rate
Fill Tonnes	32,000 t/month
Lateral Development Metres	400 m/month
Mined Tonnes – Ore	65,000 t/month
Production Drill Metres	6,000 m/month
Rehab Metres	200 m/month

Table 15 – LOM Schedule Parameters – Recovery and Dilution

Stope Type	Recovery	Total Dilution
Primary Stopes	95%	5%
Secondary Stopes	90%	5%
Tertiary Stopes	90%	5%
Remnant Stopes	90%	5%
6/6 Stope Recovery	70%	5%
Development	98%	12%

Table 16 – LOM Schedule Parameters – Miscellaneous Factors

Cut Off Values	
Development	66 $/t NSR
Stoping	150 $/t NSR
Schedule Factors	
Backfill winze drilling factor	10
Intersection Cable Metres	66 m
Probe Metres – DZL	200 m
Probe Metres – Main Ore Body	500 m
Stope Fill Prep Duration	3 days
Teleremoting Factor	0.8
TKM per Truck/mth	112,000 tkm
Tonnes Per Loader/mth	54,000 t
Tonnes/Drill Metre	10 t
Typical Stope Cable Metres	126 m
Jumbo Drilling GS/mth	24
Physical Factors	
Fill Density - CRF	2.2 t/m^3
Fill Density - Pastefill	2.2 t/m^3
Fill Density - Rockfill	2.2 t/m^3
Fill Density - CRF	2.2 t/m^3
Overbreak Grade Ag	0 g/t
Overbreak Grade Zn	0 %
Overbreak Grade Pb	0 %
Overbreak NSR	0 $/t NSR

The scheduling exercise assumed that 10% of the main decline and 100% of the old decline and most level accesses requires ground support rehabilitation works.

The production schedule has generated a projected underground mine life of approximately 6 years. It is proposed to commence mining and reprocessing of the Sector 1 tailings towards the end of production from underground mining in year 5. **Table 17** and **Figure 26** display the LOM schedule outputs.

Table 17 – LOM Production Schedule Tonnes and Grade

Source	Ore Tonnes Mined	% Measured and Indicated	Zn %	Pb %	Ag g/t
Upper Main Lode	281,575	97%	5.63	4.40	364
Main Ore Body	975,722	85%	5.63	3.30	59
Deep Zinc Lode	2,270,271	53%	7.01	0.64	37
Tailings	4,833,413	73%	2.12	1.55	79
Total	**8,360,981**	**70%**	**3.98**	**1.60**	**75**

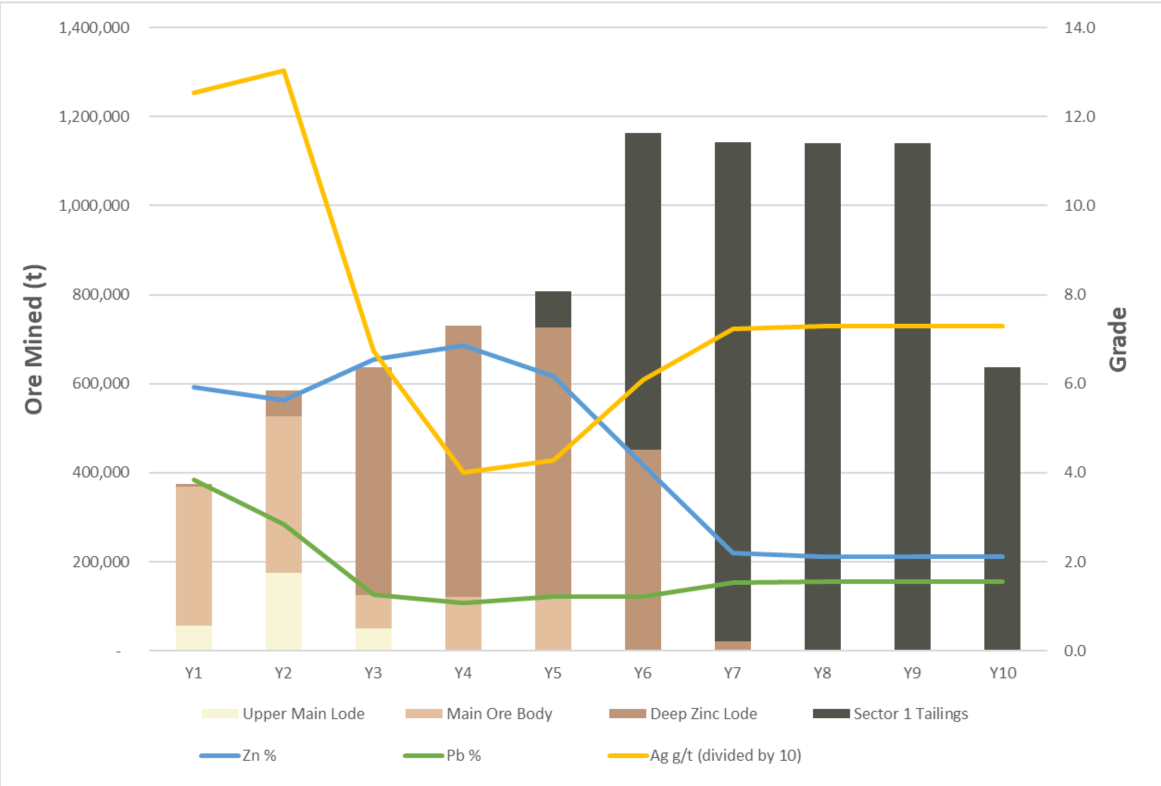


Figure 26: LOM Schedule Tonnes and Grade

The total material to be mined over the LOM schedule of 8.4 Mt includes Measured, Indicated and Inferred Mineral Resources, with 2.5 Mt (30%) of ore from the Inferred Mineral Resource category. There is a low level of confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources, or that the Production Target itself will be realised.

Inferred Resources contribute approximately 20% of the mined ore tonnes and 15% of the contained metal value in the first 2.5 years of production and is not considered material to the viability of the Project.

9 Ore Reserve Estimates

An Ore Reserve estimate has been compiled from the Measured and Indicated Mineral Resources in the mine plan using the methods outlined in Section 8 and is shown in **Table 18**. Modifying Factors were considered in the conversion of the Measured and Indicated Resources to Reserves. These factors included, but were not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors. The Ore Reserves were compiled in accordance with the guidelines defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' (2012 JORC Code). The Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce Ore Reserves.

Table 18 – Endeavor Mine Ore Reserve Summary September 2023*

Category	Source	Mt	Zinc (%)	Lead (%)	Silver (g/t)
Proved	Underground	0.49	6.11	3.90	132
Probable	Underground	1.7	7.17	1.64	60
	S1 Tailings	3.4	2.14	1.56	80
Total Proved and Probable Reserves		**5.6**	**4.04**	**1.79**	**78**

**Discrepancies may occur due to rounding*

10 Metallurgy

10.1 Introduction

A review of the historic metallurgical studies and processing plant performance was undertaken by AMC Consultants Pty Ltd (Greenhill 2023) on behalf of Polymetals Resources. The review was intended to assess the estimates for recoveries used in this study from the following areas of the ore body:

- The historically mined areas which are predominantly siltstone hosted Pb/Zn/Ag ore. The focus is on pillar recovery and remnant ores.
- Ores close to surface in the Upper Main Lode. This is a silver-rich zone with lower lead and zinc grades.
- Unmined limestone hosted Zn/Pb/Ag ore from the Deep Zinc Lode.
- The existing tailings storage facility.

10.2 Historic Mines Areas

The historic mined areas at Endeavor include the Main Ore Body and the Northern Pods. Scatter graphs compiled by AMC (**Figure 27** to **Figure 29**) show actual monthly zinc, lead, and silver recoveries verses head grade for the years 2013 through 2019. Even with reducing head grade, recoveries of all metals improved over 2018 and 2019, which was in part (aside from physical process optimisation) related to improved mineralogy, with lesser pyrite presenting in the ore. Pyrrhotite rich ores tended to allow better recoveries and concentrate grades to be achieved.

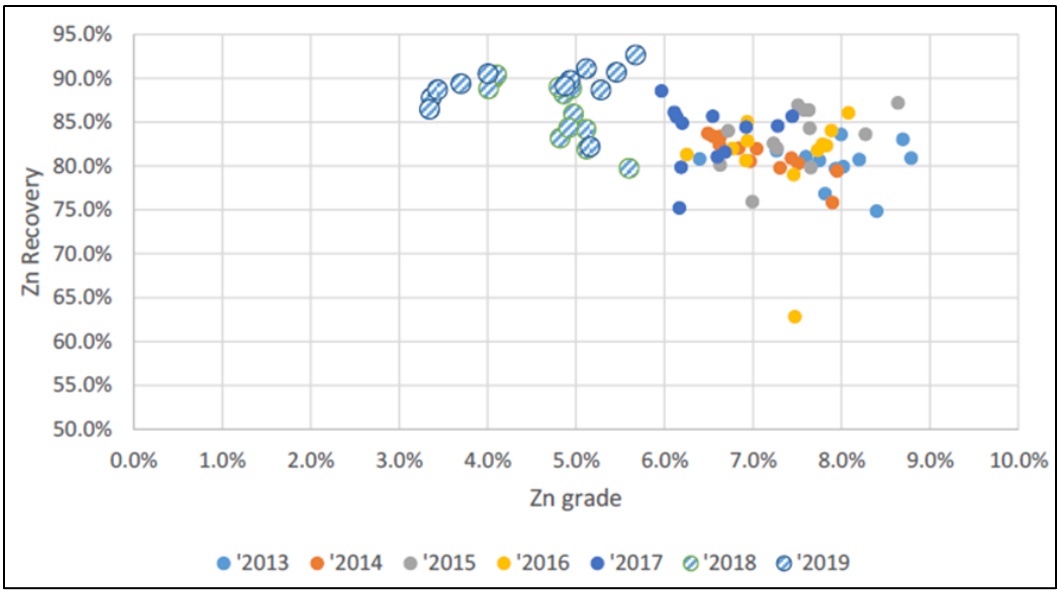


Figure 27: Monthly Zn Feed Grade v Recovery 2013-2019

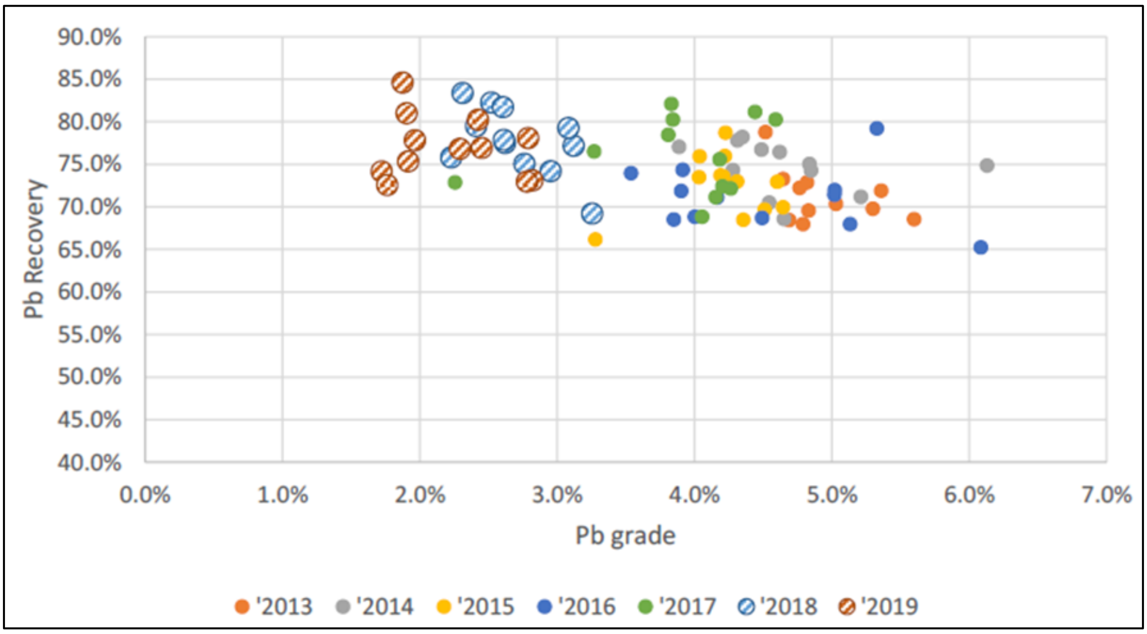


Figure 28: Monthly Pb Feed Grade v Recovery 2013-2019

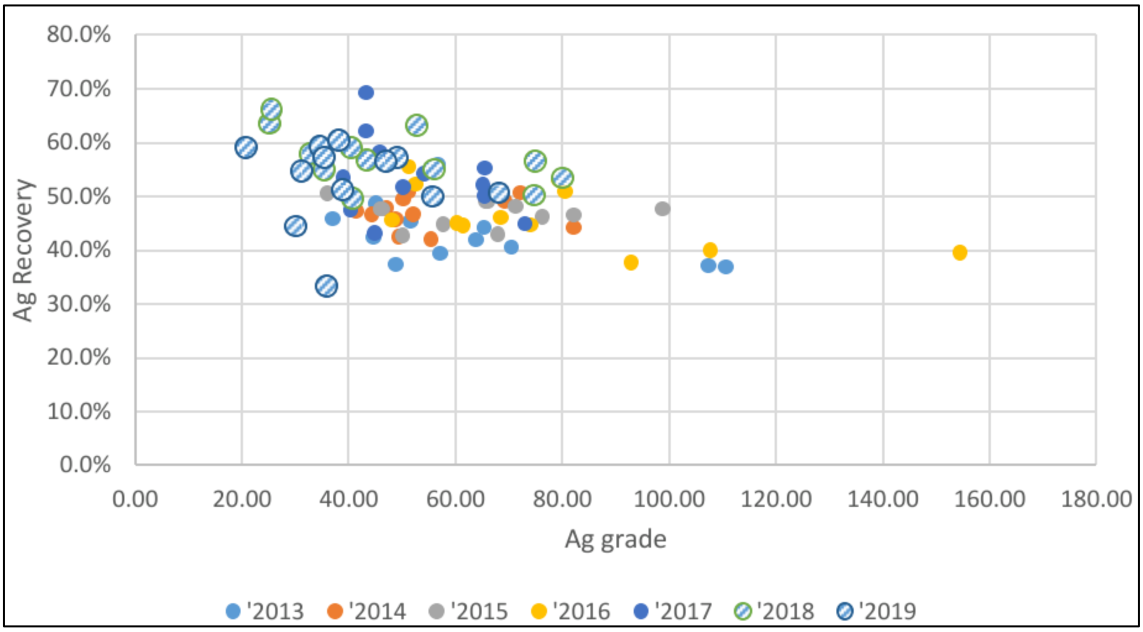


Figure 29: Monthly Ag Feed Grade v Recovery to Pb Concentrate 2013-2019

Monthly concentrate grades from 2012 to 2019 are shown in **Figure 30**.

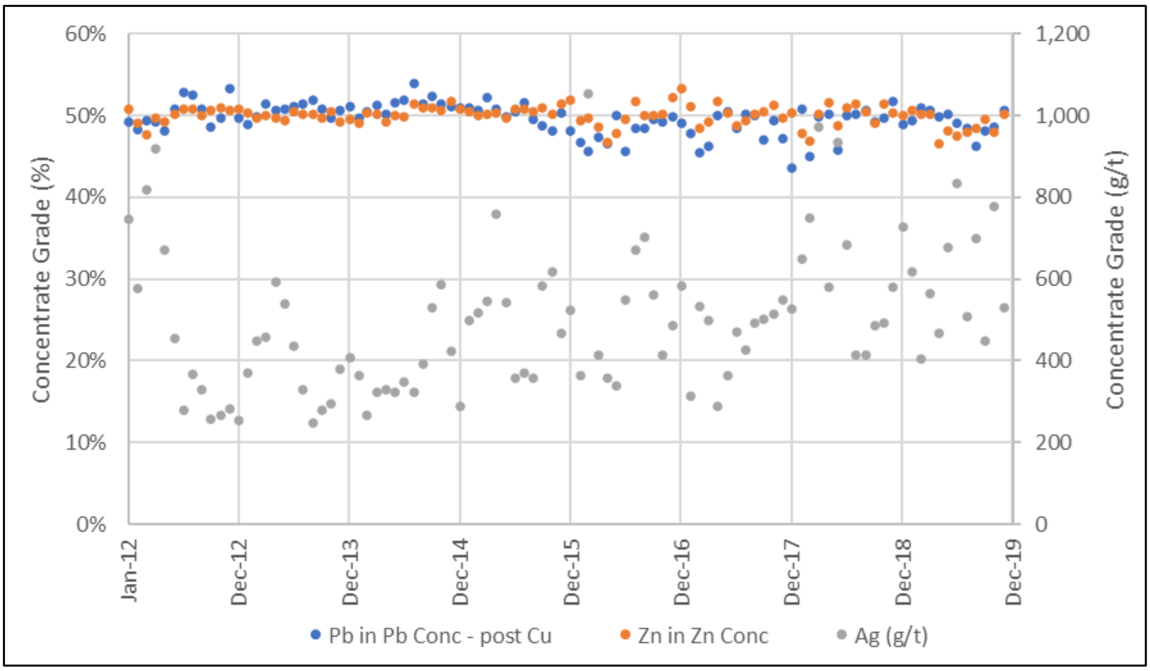


Figure 30: Monthly Concentrate Grades 2012-2019

Daily operational data was used to estimate recommended grade and recovery parameters for processing of future ore from the historically mined areas. This data did not display any significant grade/recovery dependency, so production weighted averages were calculated from the data and are shown in **Table 19**.

Table 19 – Pb and Zn Concentrate Grades and Recoveries 2018-2019

	Lead Concentrate		Zinc Concentrate	
	Pb (%)	**Recovery (%)**	**Zn (%)**	**Recovery (%)**
2019	49.6	77.0	49.4	88.6
2018	49.5	77.6	50.0	85.0
Average	**49.5**	**77.4**	**49.8**	**86.8**

AMC recommends the average grade and recovery parameters for ore to be mined from historical zones at Endeavor (Main Lode, Northern Pods) to be:

- Lead concentrate: 77.4% Pb recovery @ 49.5% grade, 55% Ag recovery @ 625 g/t
- Zinc concentrate: 86.8% Zn recovery @ 49.8% grade, 16% Ag recovery @ 94 g/t

The AMC report notes that a potential opportunity exists to increase silver recovery to the zinc concentrate which is to be further investigated. This trend is particularly relevant to the Upper Main Lode ores which have significant silver grades.

10.3 Upper Main Lode

The Upper Main Lode (UML) has significant silver grades and represents an early focus for the Endeavor mining strategy. Resource grades for the UML are 5.63% Zn, 4.40% Pb and 363.7g/t Ag.

The UML can be divided into upper and lower sulphide zones, with the upper zone being more oxidised. Mineralogy is dominated by pyrite with lesser galena and sphalerite and enriched silver values. Previous test work has been completed on the UML over previous years with variable recoveries achieved which is believed to be related to ageing of the ore (oxidation). Careful attention to treatment of the high silver grade UML ores must be maintained once mining in this area has commenced.

AMC recommends the following metallurgical performance for processing of UML ores.

- Lead concentrate: 62% Pb recovery @ 48% grade, 45% Ag recovery @ 1500 g/t
- Zinc concentrate: 76% Zn recovery @ 48% grade, 21% Ag recovery @ 200 g/t

10.4 Deep Zinc Lode

The Deep Zinc Lode (DZL) makes up a significant portion of the Ore Reserves to be exploited by the planned mining campaign at Endeavor. Zinc (as sphalerite) at 7.7% is the dominant metal within the limestone unit, with minor silver and lead. Previous metallurgical test work has shown a relatively coarse primary grind of 75um produces high zinc recoveries of +90% Zn to a concentrate grade of 50%.

AMC recommends the following metallurgical performance for processing of DZL ores.

- Zinc Concentrate: 90% Zn recovery @ 50% grade

Polymetals believes that processing of DZL ores in isolation, could result in higher grade concentrate than forecast.

Although the lead and silver grades hosted by the DZL are modest at 0.8% Pb and 36 g/t Ag, encouraging metallurgical test work results suggest that a saleable high silver-grade lead concentrate can be produced. Recent test work generated lead and silver recoveries of 83% and 81% respectively to a concentrate grading 1810 g/t Ag. Production of a concentrate from the DZL has been assumed with recoveries of 75% Pb and 70% Ag to a 48% lead concentrate.

10.5 TSF Sector 1 Tailings

Historical records indicate that around 5.4Mt of tailings, grading 2.1% Zn, 1.8% Pb and 83g/t Ag were deposited in Sector 1 of the TSF from commencement of operations in 1983 to 1989. Recent drilling and Resource modelling has estimated 5.2 Mt at 2.1% Zn, 1.6% Pb and 79 g/t Ag of tailings in Sector 1.

Several metallurgical test work programs have been conducted on Sector 1 tailings at Endeavor in previous years. Oxidation tests have determined that 33% of the lead and 1.6% of the zinc is oxidised with this portion not recoverable by flotation.

Locked cycle flotation test work by ALS Burnie in 2017 has determined that 50% of the zinc can be recovered to a 50% concentrate. AMC recommends achievable reprocessing performance as:

- Zinc Concentrate: 46% Zn recovery @ 50% grade

Recent test work on the Sector 1 tailings at ALS Burnie resulted in a 62.1% to 64.7% silver recovery to concentrate. With these promising results from the preliminary tests, a future test work programme will be completed to confirm final lead and silver recovery expectations.

Recoveries of lead and silver from Sector 1 Tailings have been conservatively estimated at:

- Lead/silver Concentrate: 30% Pb and 40% Ag recovery to a 50% Pb grade.

10.6 Metal Recovery Summary

The various ore sources at the Endeavor Mine have slightly differing metallurgical characteristics which have a bearing on historic and forecast metal recoveries and concentrate grades. **Table 20** provides a summary of the recommended achievable process recoveries. Several metallurgical recoveries and concentrate grades have been estimated for the Deep Zinc Lode and Tailings, which are the subject of ongoing or planned flotation test work.

Table 20 – Summary Metal Recoveries and Concentrate Grades

Ore Source	Metallurgical Recovery			Pb Concentrate Grade		Zn Concentrate Grade	
	Pb (%)	Zn (%)	Ag (%)	Pb (%)	Ag (g/t)	Zn (%)	Ag (g/t)
Historic Areas	77.4	86.8	71	50	625	50	94
Deep Zinc Lode	75*	90	70*	48*	1,800*	50	100*
Upper Main Lode	62	76	66	48	1,500	48	200
Tailings	30*	46	40*	50*	1,500*	50	-

*Estimated recoveries and grades

10.7 Opportunities for Increased Recovery

10.7.1 Leaching

Metallurgical test work has been completed in the past for increased recovery of silver and gold from the ore stream. This is particularly relevant with respect to potential supergene ore and flotation tailings. The only historic commercial leaching of Endeavor mineralisation was conducted by Polymetals Chairman, David Sproule, during 1993 – 1995. The tailings processed had resulted from treatment by Pasminco of 100,000 t of supergene ore mine from the "South Lode", which contained approximately 16 Moz of silver and 29,000 oz of gold. Then owner Pasminco, produced a flotation concentrate which typically assayed 15,000 g/t silver, 15 g/t gold, 6% copper and 15% lead.

The tailings retreatment campaign by Mr Sproule (as Polymetals Australia Pty Ltd) involved purchase and off-site treatment of the tailings to produce silver/gold doré.

A total of 84,000 t of flotation tailings grading 550g/t silver and 3.5g/t gold was treated via cyanidation and Merill Crowe to recover 81% (1.2 Moz) silver and 70% (6 koz) gold. Elevated cyanide soluble copper required that aluminium be used as the cementation (reducing) agent.

There is an opportunity to progress with investigation of potential gold and silver recovery via cyanidation of UML supergene ore and Sector 1 tailings. This possibility is considered as a potential Stage 2 revenue stream, following further test work and re-establishment of steady state commercial operations at Endeavor. It may be that high silver/gold grade tailings generated from the restart operation will be stored separately for later treatment.

11 Mineral Processing

11.1 Introduction

The Endeavor Mine processing plant was designed as a standard, differential lead-zinc flotation circuit, and commissioned in 1982. It was engineered and constructed by global engineering firm Fluor Daniel.

Nameplate capacity of the Endeavor mill is 1.2 Mtpa, although throughput has been largely mine constrained. A total of 32 million tonnes of ore has been processed over 38 years of operations, with an average annual throughput of 850,000 tpa. The mill remains in excellent condition with a number of process item modifications from the original design made over the years to enhance efficiency. Notable changes have been the replacement of concentrate regrind mills with Svedala Sand Detritors to enhance concentrate grades.

Figure 31: Endeavor Mine Processing Plant

Primary Endeavor ore historically mined and processed consists of galena (~13 %wt) and sphalerite (~14 %wt) with Pyrite and pyrrhotite being the main floatable gangue. Tetrahedrite is the major host of silver, with minor silver within galena and chalcopyrite. The average grain size of galena and sphalerite ranges from 10 - 40 μm.

Six main ore types have been identified within the Endeavor Mine orebody:

- Pyritic mineralisation (PY),
- Pyrrhotitic mineralisation (PO),
- Silicious pyrrhotitic ore (SIPO),
- Silicious Pyritic ore (SIPY),
- Vein ore, and
- Mineralised altered CSA siltstone (MINA).

Whilst the ore is considered complex, metallurgical recoveries of lead, zinc, and silver continued to improve with time, which was likely a combination of improved metallurgy, attention to optimal grind size, improved reagents, better process control and more experienced floatation operators.

11.2 Process Flowsheet

The processing of ore at the Endeavor Mine follows a conventional single stage crushing, two stage grinding, and differential flotation, including concentrate regrind, to produce separate lead/silver and zinc concentrates. Concentrates are dewatered using thickeners and filters and loaded directly into concentrate containers for rail transport to a shipping port or smelter. Tailings are deposited in a Central Discharge Tailings Storage Facility (CTD TSF). Historically tailings were utilised in the underground mine as consolidated backfill.

Table 21 represents the major process items included in the process flowsheet.

Table 21 – Major Processing Equipment

Category		**Equipment Detail**
Crushing	Underground	60" x 48" Jacques single toggle jaw crusher
	Surface	48" x 42" Terex Jacques single toggle jaw crusher
Grinding (Flotation feed)	Primary	1.68MW Hardinge overflow SAG mill 7.3m x 2.44m
	Secondary	1.68MW Clyde Marcy overflow mill, 4.1m x 6.7m
	Tertiary	1.7MW Sala overflow mill 3.9m x 6.9m
Classification		4 x 20" Warman Cavex Cyclones
		7 x 10" Cavex cyclones
		6m2 Delkor linear trash screen
Re-grinding (Concentrate)	Lead	2 x Svedala 185kW sand detritors
	Zinc	2 x Svedala 355kW sand detritors
Flotation		
Roughers / Scavengers	Lead	9 x 8.5m3 Agitair cells
	Zinc	9 x 8.5m3 Agitair cells
Cleaners	Lead	20 x 5.7m3 Agitair cells
	Zinc	28 x 5.7m3 Agitair cells
Cleaner scavengers	Zinc	20 x 2.8m3 Agitair cells
Concentrate Dewatering		
Thickening	Lead	1 x 25m diameter
	Zinc	1 x 25m diameter
	Tailings	1 x 50m diameter
Filtration	Lead	1 × Larox PF543
	Zinc	2 × Larox PF84/96

11.2.1 Crushing Circuit

Crushing is conducted underground using a 60" x 48" Jacques single toggle jaw crusher. The nominal ROM crushed size (F80) is reported as 150mm, however typically the ore is delivered at a F80 of 125mm. The ore is hoisted to surface and discharged onto an open stockpile. This stockpile has been designed at about 10,000 t live capacity, with total capacity of about 25,000t. Because of concerns regarding ore oxidation, the stockpile is generally managed at a lower level, unless there is a need to increase the stockpile to cover mine shutdowns.

11.2.2 Grinding Circuit

The grinding circuit is made up of two stages of grinding, with the second stage in closed circuit with hydrocyclones for size classification.

The primary grinding circuit uses a high aspect (diameter greater than length) Semi-Autogenous Grinding mill (SAG). The mill is a Hardinge 7.2m diameter by 2.4m long mill, powered by a 1,850kW motor and lined with polymet lifters which are essentially a steel cap on a normal rubber liner/lifter. The mill operates largely as an Autogenous Grinding mill (AG) or a hybrid SAG, with less than 3-4% ball charge. Balls are added dependent upon the ore hardness. The primary mill operates in open circuit. The grate discharge has rubber grates with 12 x 25mm slots.

The secondary mill is a standard 4.1m diameter by 6.7m long ball mill powered by a 1,400kW motor re-rated to 1,680kW, like that connected to the AG mill. The ball mill is also rubber lined and uses 40mm balls to produce, after classification, a flotation feed product at P80 = 45µm. The ball mill discharges into a pump sump common with the SAG mill. This pump feeds the combined mill discharge to a bank of Cavex-type Warman 350mm hydrocyclones for classification.

In 2012 a Sala overflow mill, 3.9 m × 6.9 m, 1700 kW tertiary mill was installed into the grinding circuit. The mill has been removed from the process flow sheet but remains in place if future projects call for a third stage of grinding.

Table 22 – Endeavor Comminution Circuit

Parameter	Units	Existing Plant
General		
Throughput	Mtpa	1.20
Throughput	tph	148
Utilisation	%	92.8
Operating Time	Hours/annum	8,129
Feed Size	F80 = mm	125
Primary Grinding		
Mill Type		AG/SAG
Number of Mills		1
Mill Size	m	7.3 x 2.4
Connected Drive	kW	1,850
Circuit Type		Open
Liners		Rubber/Polymet
Ball Charge	%	4
Product	P80 = µm	200
SAG Mill Factor		0.8
Mill Speed	% of Critical Nc	72
Power Required at Pinion	kWh	1,453
Power Draw on Mill	kWh	1,456
Secondary Grinding		
Mill Type		Ball
Number of Mills		1
Mill Size	m	4.1 x 6.7
Connected Drive	kW	1,400
Circuit Type		Closed
Liners		Rubber
Ball Charge	%	35
Ball Size	mm	40
Product	P80 = µm	45
Mill Speed	% of Critical Nc	70
Power Required at Pinion	kWh	1365
Power Draw on Mill	kWh	1,467

11.2.3 Lead Flotation

Figure 33 illustrates the Endeavor lead flotation circuit as configured and operated by CBH Resources until the project was placed on care and maintenance in December 2019.

Liberation sizes of galena range between 10 – 20µm with a target flotation feed grind size of 80 – 85% passing 45µm. Cyclone overflow from the grinding circuit is screened for trash removal and fed at pH 8.5 – 9 to two conditioning tanks each with 8 mins residence time. Aerophine 3418A is added to the second conditioning tank (B) after which the slurry is introduced to two parallel banks of Agitair cells.

Liberation sizes of lead sulphide ore (galena) range between 10 – 20µm with a target flotation feed grind size of 80 – 85% passing 45µm. Cyclone overflow from the grinding circuit is screened for trash removal and fed at pH 8.5 – 9 to two conditioning tanks each with 8 mins residence time. Aerophine 3418A is added to the second conditioning tank (B) after which the slurry is introduced to two parallel banks of Agitair cells.

A significant improvement in lead float circuit performance was achieved following the installation in 1998 by then owners Pasminco of four sand regrinders (Svedala detritors). Ultimately, only one of the 185kW detritors fed with lead rougher and scavenger concentrate was required to achieve the necessary size reduction to P80 of 15 – 20μm.

Following regrinding, the combined concentrate is fed to three stages of cleaning with the final concentrate pumped to the 25m diameter lead thickener.

It should be noted that production of saleable copper concentrates was undertaken when pockets of lead and zinc mineralisation contained copper grades of +0.3%.

11.2.4 Zinc Flotation

Figure 34 illustrates the zinc flotation flow sheet. The lead float tailing reports to Conditioning Tank (A) as Zinc flotation feed. Here, pH is adjusted (via addition of lime) to 9.5 followed by copper sulphate addition (to activate the zinc) at Conditioner Tank (B) as well as a sulfonate to suppress iron (pyrite). The collector, sodium isopropyl xanthate (SIPX) is added to the slurry as it exits Conditioning Tank (B) and enters the zinc rougher flotation cells. pH is maintained between 10.8 – 11.2 throughout the zinc circuit with rougher and scavenger concentrates reporting to regrind (Svedala detritors) to achieve a P80 of 35um prior to the cleaning stages.

Figure 32: Endeavor Flotation Floor

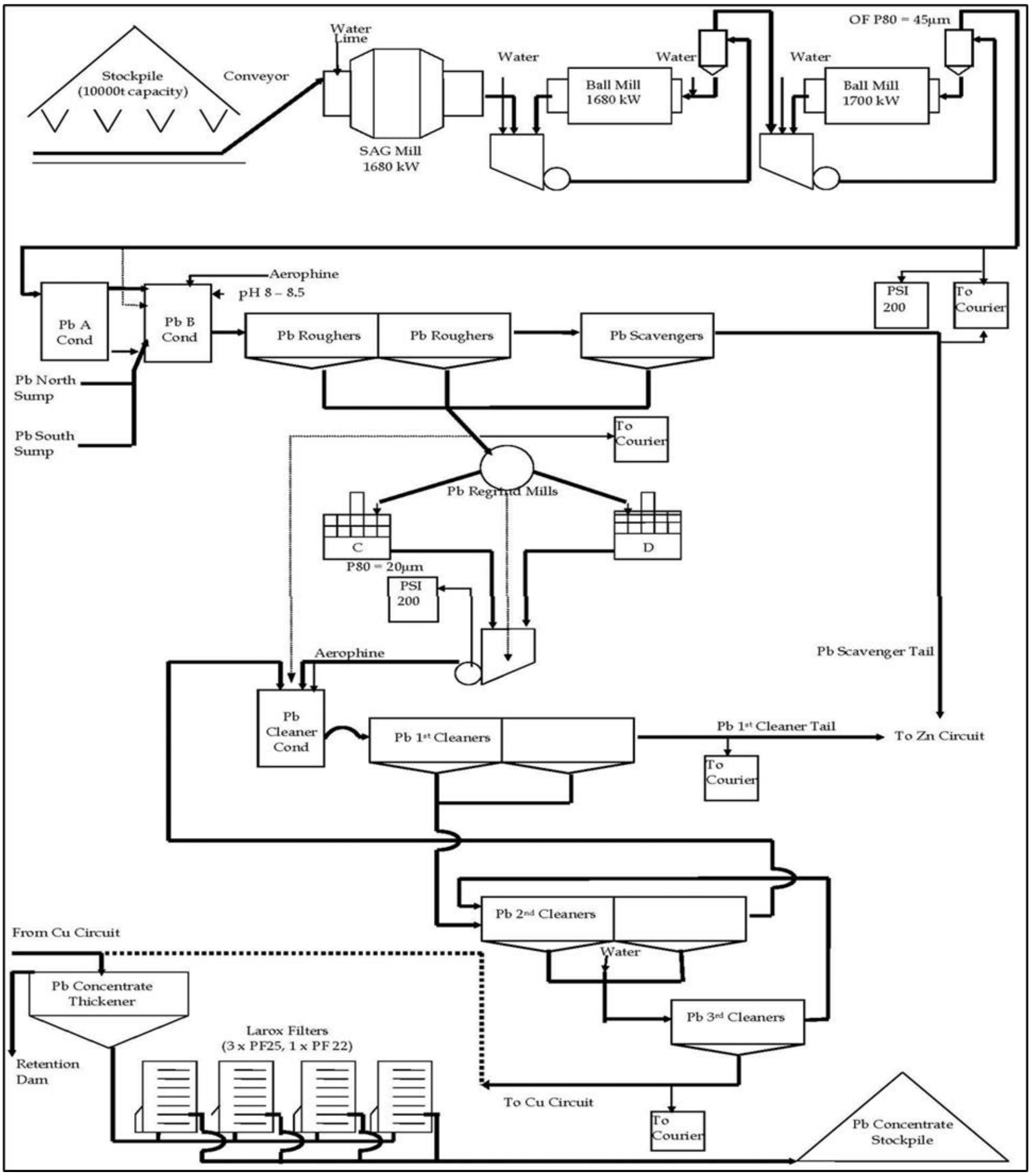


Figure 33: Endeavor Lead Flotation Flowsheet

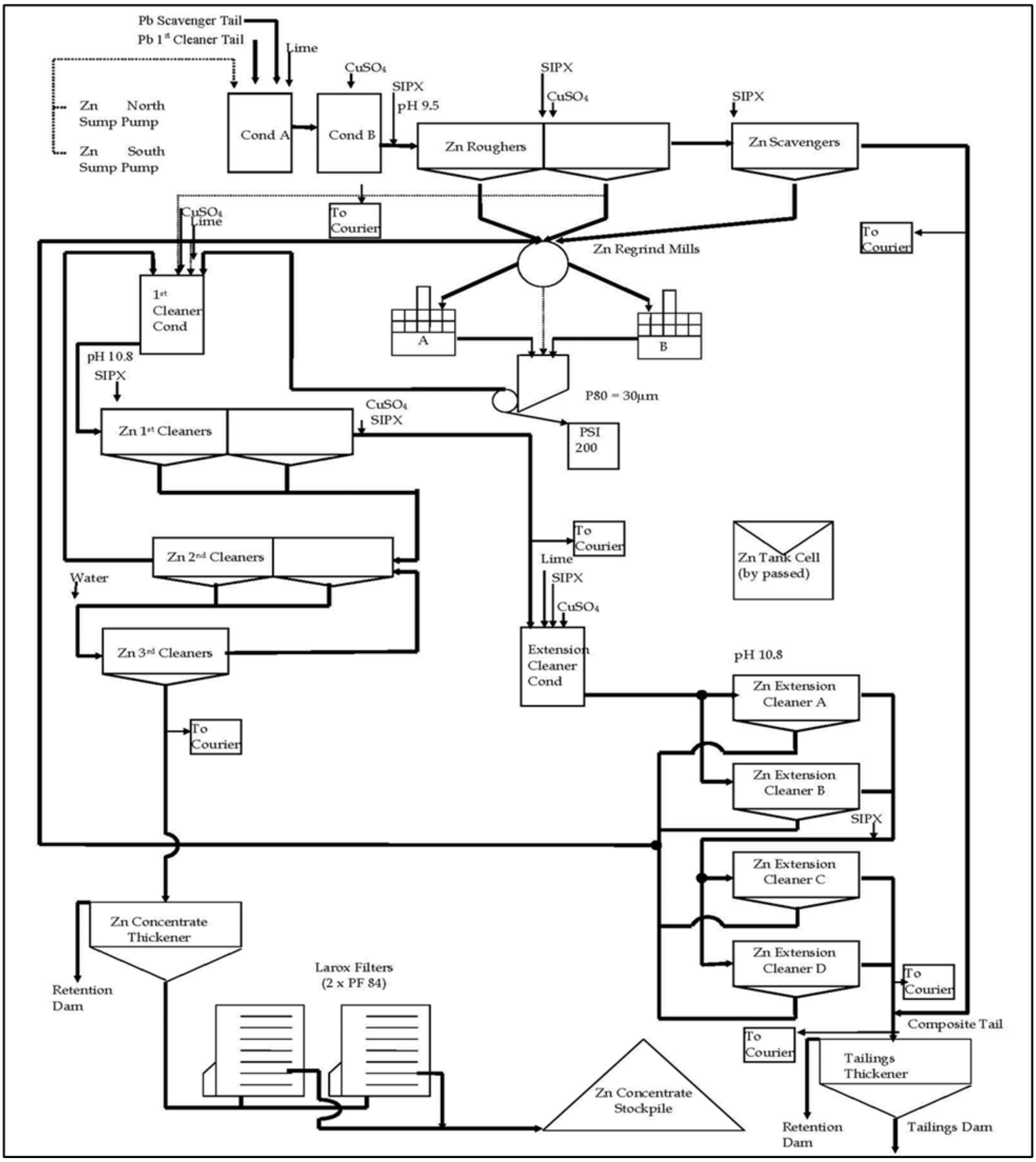


Figure 34: Endeavor Zinc Flotation Flowsheet

11.2.5 Concentrate Thickening

The lead concentrate is delivered to a 25m diameter thickener. Underflow from the thickener is pumped to a holding tank prior to filtering. Thickener overflow is sent to the process water system.

The zinc concentrates are initially dewatered in a 25m diameter conventional thickener. The underflow is stored in a holding tank prior to filtration, with the overflow pumped to the process water system.

11.2.6 Concentrate Filtering

Lead thickener underflow is filtered using a Larox filter. The Endeavor operation was the first in Australia to use this type of pressure filter, which has performed satisfactorily in terms of producing a concentrate for transport at or below the Transportable Moisture Limit ("TML") of 8%.

Zinc thickener underflow is filtered using two $84m^2$ Larox pressure filters, similar to the lead concentrate filters, but much larger. The nominal TML for the zinc concentrates is 10%.

11.2.7 Concentrate Transport

The lead concentrate is discharged from the filters to a conveyor that discharges to a stockpile within the filter building. The concentrate is loaded by a Front-End Loader (FEL) into half-height containers on rail wagons, which in the past were transported to the Port Pirie smelter in South Australia.

The zinc concentrate is discharged from the filters to a conveyor that discharges to a stockpile within the filter building. The concentrate is loaded by a FEL into half height containers which in the past were transported to the Newcastle ship loader facility for shipment overseas.

11.2.8 Tailings Deposition and Storage

Flotation tailings slurry is thickened in a 50m diameter thickener and pumped to the Tailings Storage Facility (TSF). Run off process water is reclaimed and recycled to the concentrator along with water generated by mine de-watering activities. Make-up water is drawn from the Cobar pipeline.

The Endeavor Mine has remaining tailings storage capacity of approximately 900,000 t. A conceptual design and costing have been completed for a TSF wall lift (Stage 3) resulting in an additional storage capacity of 3.6 Mt. The design is an upstream construction method with commencement scheduled during Year 3.

11.3 Reagents and Consumables

Table 23 provides detail of reagents and consumables utilised by CBH Resources to produce lead and zinc concentrate through the mill. Whilst metallurgical optimisation will be a keen focus for Polymetals operational management, there is little to be gained with testing of alternate reagents and/or consumables prior to achieving steady state production from a restart.

Table 23 – Reagents, Consumables & Nominal Usage

Process Consumables	Details	Approximate Usage (g/t)
	Grinding Media	
SAG	125mm	270
Primary mill	64mm	1270
Concentrate re-grind	Silica sand	153
	Flotation	
	Lime	3,100
	Frother	40
Lead	Aerophine 3418A	95 - 105
Zinc	SIPX	185 - 190
	CuSO4	1,120 – 1,150
	Iron depressant	160 - 165
Flocculant	Optimer 83376	35 - 40

12 Project Infrastructure

12.1 Site Layout

The Endeavor site was constructed by Fluor Daniel and commissioned in 1982. The layout of the site remains largely unchanged. **Figure 35** presents the key locations of site infrastructure.

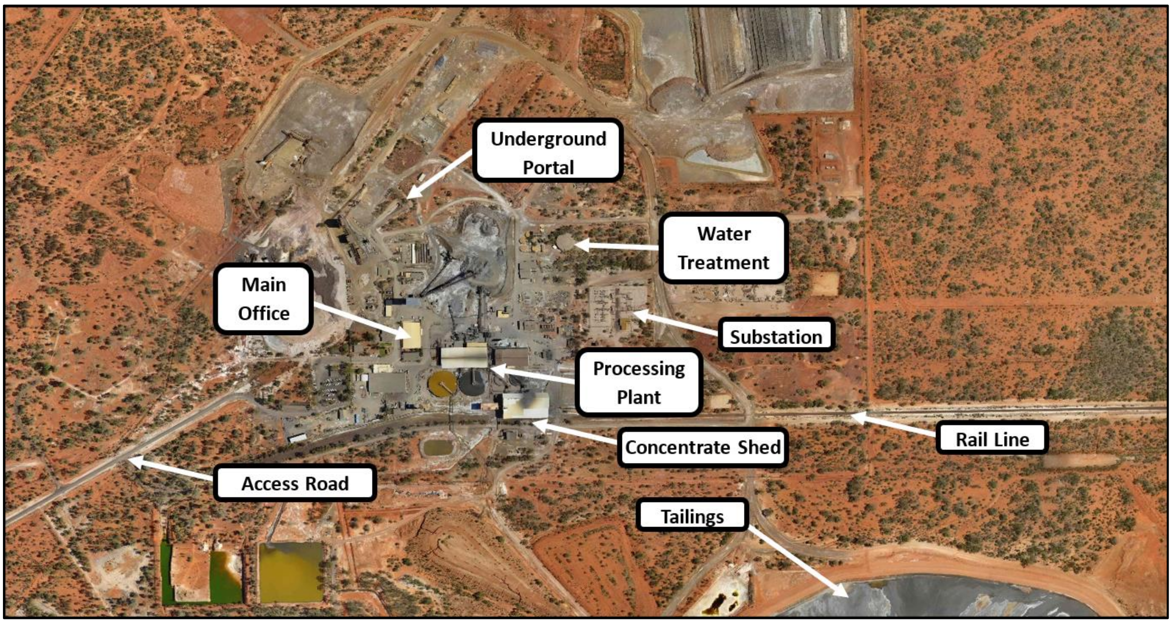


Figure 35: Site Layout – Surface Infrastructure

12.2 Roads

Road access to the Project is via the sealed Cobar to Louth road (Mulya Rd). The mine access road is fully sealed to the mine gate. Internal roads within the project area are a mixture of sealed and gravel roads. During operation gravel roads will be subjected to dust suppression via a water cart.

12.3 Rail

All concentrates are transported from the mine by rail. A spur railway line runs from the mine to Cobar and links to the national rail network enabling the transport of concentrate from the mine to smelters at Port Pirie or to a ship loading terminal at Newcastle.

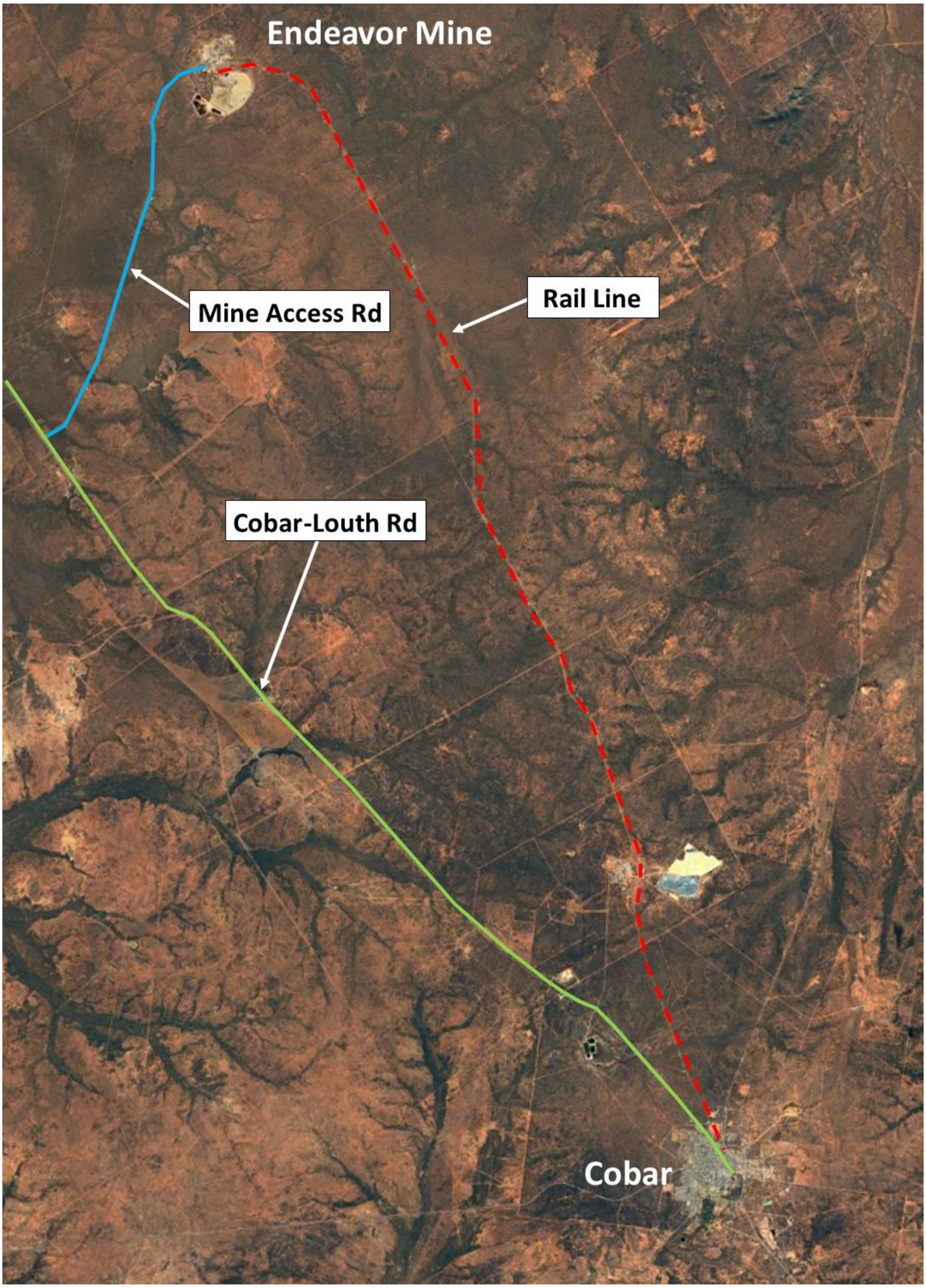


Figure 36: Site Access Infrastructure

12.4 Water

12.4.1 Raw Water

Raw water supply is managed by the Cobar Water Board (part of the Cobar Shire Council). The principal source of the water is the Bogan River at Nyngan, where water is stored in a series of pools known as the Bogan Storages. From Nyngan, the water is pumped through parallel pipelines some 130 km to a 1.14 ML reservoir at Fort Bourke Hill, Cobar. Raw water is distributed from the Fort Bourke Hill Reservoir to terminal storages located 4 km Northwest of Cobar. A pumping station at the storages supplies raw water to the Endeavor Mine via a 250mm buried pipeline which runs parallel to the rail line to site.

12.4.2 Potable Water

Potable water for the offices, future camp and change rooms is provided by the water treatment plant located adjacent to the main raw water storage tank on site.

12.4.3 Process Water

Process water will be predominantly supplied by raw water and recycling return process water from thickeners and TSF decant. Decant water will be collected from the TSF decant dam and returned to the process water tank, via the tailing's thickener.

Figure 37: Water Infrastructure on Site

12.5 Electrical Power

The Project is supplied with State Grid power via a 132 kV, 15 MW sub-station adjacent to the mine. The main substation is owned and maintained by Essential Energy. Transformers are used at the plant site to step down the voltages to the levels required by the various electric drives.

Power is reticulated around the site via a mixture of underground and overhead cables to supply the following main load centres:

- Underground mining substations.
- Winder.
- Surface fans.
- Processing plant.
- Concentrate shed.
- Workshops.
- Change rooms.
- Offices.

Historical data has been used to forecast power consumption. From the forecast model the maximum demand for the combined underground operations, Process Plant and associated equipment at the 720,000 tpa rate has been calculated as 7,707 kWh.

Figure 38: Electrical Sub Station on Site

12.6 Tailings Storage Facilities

There are two tailings storage areas on site: a Centrally Thickened Discharge Tailings Storage Facility (CTD TSF, Sectors 1 to 4) and a quarry void tailings storage area (Sector 5). There is approximately 900,000 t of storage capacity remaining in Sectors 2 to 5. Sector 1 will be subject to mining in year 5 of the mine plan and therefore will not be used for deposition. When Sector 5 is at capacity it will be capped and rehabilitated. Tailings deposition will re-commence in the CTD TSF, following a wall height increase (Stage 3 Embankment Raise and Buttress Extents) to increase capacity by a further 3.6.Mt. The Stage 3 Embankment Raise has in-principle approval by the NSW Dam Safety Committee but requires detailed designs to be completed.

The CTD Tailings Dam is made up of 4 sections subdivided over an arc of 360° covering 136 hectares. The segmented design also permits sequential and progressive rehabilitation of the tailings dam over the life of the mine. Approximately 12.5 hectares of Sector 1 was successfully rehabilitated in 1997.

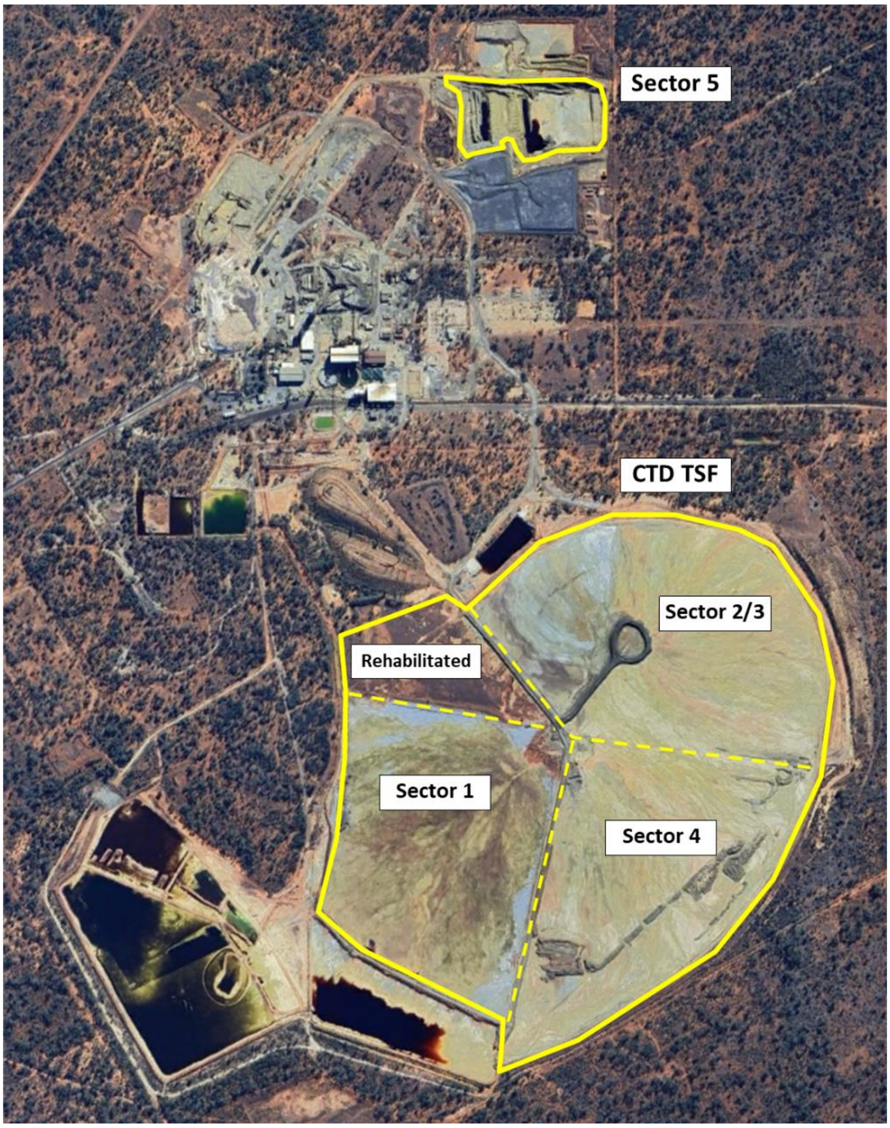


Figure 39: Tailings Storage Facilities

12.7 Surface Workshop and Stores

The site is equipped with both surface and underground workshops. The surface workshop is equipped with overhead cranes, vehicle hoist and a service pit. The surface workshop is located beside the stores at the front of the mine entry. The site also has a consumable logistics shed equipped with operational stores. The store currently remains ~30% stocked with consumables, equipment, and parts.

12.8 Offices

The office complex consists of three main areas:

- Main office / change rooms space 1,350m^2
- Front office space 300m^2
- Laundry and secondary change rooms 435m^2

The office complexes have cabled networks and land line phones for external communication. A WIFI network operates throughout the office buildings for access to internal and external networks and internet.

Figure 40: Workshop and Stores (foreground), Offices and Headframe

12.9 Laboratory

The Endeavor Mine is fortunate to have a substantial sample preparation, metallurgical test work and analytical facility. The facility requires minimal effort to bring it back into operation. The facility provides metallurgical and geological staff with access to testing equipment and analytical capability and will support both grade control and exploration analyses.

12.10 Fuel Storage and Distribution

Diesel fuel will be delivered to site via road trains from Dubbo. The trucks will unload fuel in a new 68,000 litre self-bunded tank supplied by the fuel distributor. A fuel management system will be installed to control the dispensing of diesel fuel to the mobile fleet.

The underground refuelling station is fed via a 25mm plastic line located inside a 50mm steel pipe which is installed in a dedicated hole linked to a surface 5,300L batch tank which is filled automatically from the main surface storage tank.

12.11 Accommodation

Employees will be accommodated at either the proposed site camp or the Project-owned houses in Cobar. The Project currently owns 42 houses and 46 units in Cobar. A 100-person camp will be built, fully catered, and managed by the operation.

12.12 Underground Mine Facilities

The Endeavor Mine is fully reticulated for power, water and compressed air requirements. The Main Shaft is equipped with a tower mounted Koepe friction winder directly coupled to a 2,000 kW DC thyristor drive motor. The two conveyances consist of a 9.25 t skip over a man cage and a 13.75 t solo skip. The hoisting system is fully automatic and has a capacity of 460 t/h. The conveyances operate on rope guides, engaging into fixed guides at the loading and tipping positions. A list of underground facilities is displayed in **Table 24**.

Table 24 – Underground Facilities

Level	Description
900 Level	Jaw Crusher – Terex Jacques, 60 x 48-inch, double toggle, 185 kW.
	Refuelling Station – 11,300 L storage direct fed from surface via bore hole.
850 Level	Skip Loading Station with Feeder
600 Level	Workshop, Cribroom and Office
230 Level	Pump Station – 3 x WT737AMS Crown Triplex plunger pumps, nominal discharge 8.5 L/s per pump, provision for 4th pump if required. Pumped directly to surface via steel rising main.
Various	Refuge Chambers (8) – 1x20 person, 1x15 person, 5x10 person, 1x4 person
Various	Leaky Feeder and Telephone Communications

Figure 41: Underground Pump Station 230 Level

13 Market Studies and Logistics

13.1 Introduction

Polymetals has engaged with Ocean Partners, a global base & precious metal trading firm, to assess the marketability of the concentrates which will be produced from the Endeavor Mine.

13.2 Commodity Price & Foreign Exchange Outlook

13.2.1 Zinc

The International Lead and Zinc Study Group ('ILZSG') indicate that world refined zinc metal supply exceeded demand by 370,000 t during the first half of 2023, with total reported inventories increasing by 85,000 kt. World zinc mine production fell in Burkina Faso, Canada, Sweden and Australia, where mining activity was negatively impacted by heavy rains in the first quarter. These declines were partially balanced by rises in Brazil, Kazakhstan, Peru and Portugal, resulting in an overall mine production reduction globally of 0.7%.

A significant rise in Chinese refined metal production was the main driver behind an overall increase in global metal production of 2.7%. Output also rose in Australia, benefiting from the commissioning of additional capacity at the Sun Metals Zinc Refinery, and in Mexico. However, in Europe, Canada and Japan production was lower than the corresponding period of 2022.

Increases in the usage of refined zinc metal in China, India and the United States were partially offset by reductions in Europe, Brazil, the Republic of Korea, Taiwan, Thailand and Türkiye, resulting in an overall global rise of 0.9%. Chinese imports of zinc contained in zinc concentrates rose by 25.5% to 1,087,000 t. Net imports of refined zinc metal totalled 94,000 t compared to net exports of 11,000 t in the first half of 2022.

Table 25 – World Refined Zinc Supply & Usage 2018 - 2023

Tonnes (kt)						Jan – Jun		2023			
	2018	2019	2020	2021	2022	2022	2023	Mar	Apr	May	Jun
Mine Production	12,723	12,799	12,252	12,801	12,460	6,081	6,041	1,012	1,019	1,057	1,075
Metal Production	13,142	13,546	13,780	13,873	13,342	6,753	6,937	1,213	1,173	1,173	1,172
Metal Usage	13,730	13,801	13,285	14,076	13,503	6,512	6,567	1,132	1,106	1,106	1,096

Source: ILZSG

The Zinc price has moved downwards from its 3-year peak of US$4,498 in April 2022, and the 3-year mean of US$3,042/t, and is currently sitting around US$2,500/t (**Figure 42**). Zinc appears to have recovered from its 3-year low and has been moving upwards over the past 3 months. Consensus Economics long term price forecasts vary between US$2,491 and US$3,328.

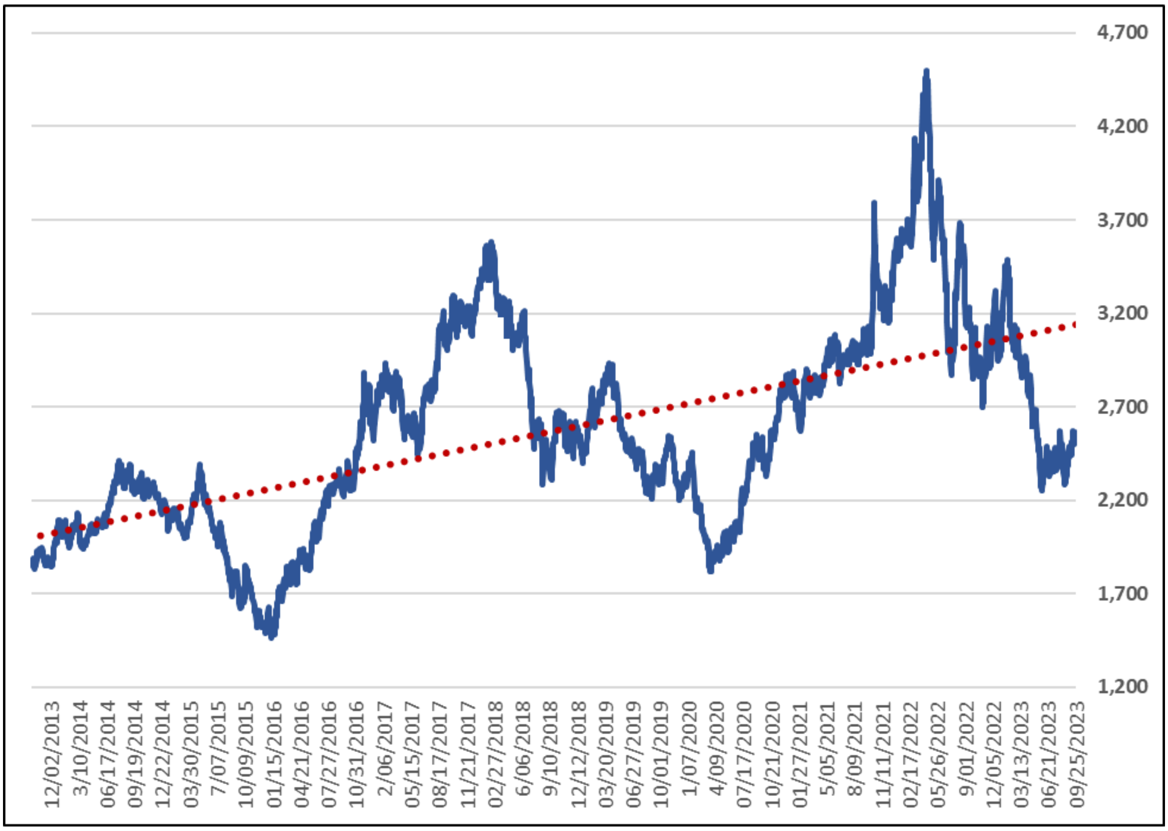


Figure 42: Zinc 10 Year Historic Price

13.2.2 Lead

According to data by the ILZSG, the global market for refined lead metal was in surplus by 25,000 t over the first half of 2023, with total reported inventories remaining more or less unchanged. Global lead mine production rose by 3.3%. This was primarily a consequence of increases in Kazakhstan, South Africa and Australia, where Galena Mining (ASX:G1A) successfully commissioned their 95,000 tpa Abra mine in January.

A 1.9% rise in global lead metal production was mainly a result of higher output in Australia, China, India and the Republic of Korea. In Europe however, output fell by 4.8%. This was mainly a consequence of reductions in Bulgaria, Italy and the United Kingdom that were partially balanced by an increase in Germany. Rises in refined lead metal usage in Mexico and India were more than balanced by falls in Europe, the Republic of Korea, Türkiye and the United States, resulting in an overall lead metal usage decrease globally of 0.9%. Chinese imports of lead contained in lead concentrates increased by 30.8% to 316,000 t. Net exports of refined lead metal totalled 76,000 t compared to net imports of 89,000 t over the same period of 2022.

Table 26 – World Refined Lead Supply & Usage 2018 - 2023

Tonnes (kt)	2018	2019	2020	2021	2022	Jan – Jun 2022	Jan – Jun 2023	2023 Mar	2023 Apr	2023 May	2023 Jun
Mine Production	12,723	12,799	12,252	12,801	12,460	6,081	6,041	1,012	1,019	1,057	1,075
Metal Production	13,142	13,546	13,780	13,873	13,342	6,753	6,937	1,213	1,173	1,173	1,172
Metal Usage	13,730	13,801	13,285	14,076	13,503	6,512	6,567	1,132	1,106	1,106	1,096

Source: ILZSG

The Lead price has moved upwards from its 3-year mean of US$2,135/t, currently sitting at US$2,219/t, and relatively rangebound between US$2,300 and US$2,200/t over the past 3 months (**Figure 43**). Consensus Economics long term price forecasts vary between US$1,874 and US$2,476/t.

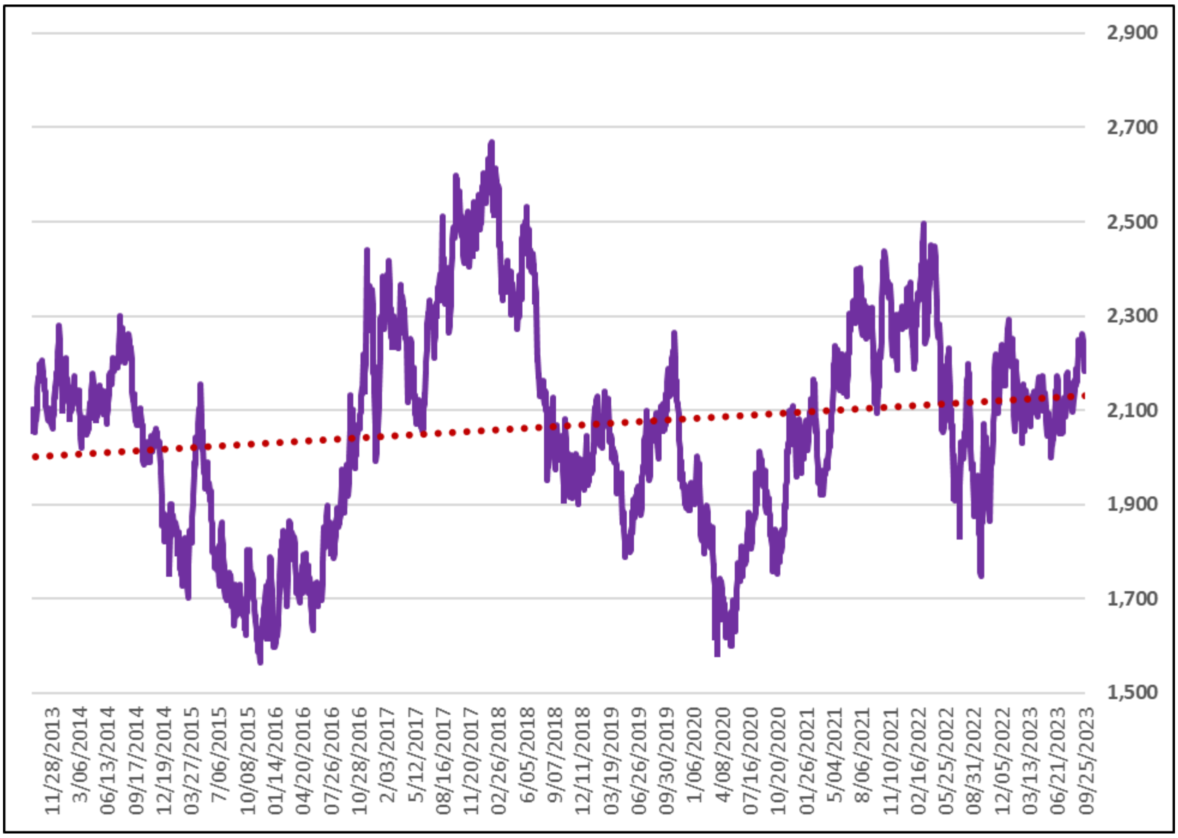


Figure 43: Lead 10 Year Historic Price

13.2.3 Silver

It is estimated that approximately 60% of today's silver is used for industrial purposes such as electronics, solar cells, automotive, and soldering, with the remaining 40% available for investment. Data from the Silver Institute shows that global silver demand has increased by 38% since 2020. In 2022 demand for silver increased by 18% to a record high of 1.24 billion ounces. According to the 2023 World Silver Survey, the global silver market was undersupplied by 237.7 million ounces in 2022. The most significant deficit on record. Looking into the next decade, and taking into consideration the green-energy transition, industrial consumption presents strong market conditions for Silver.

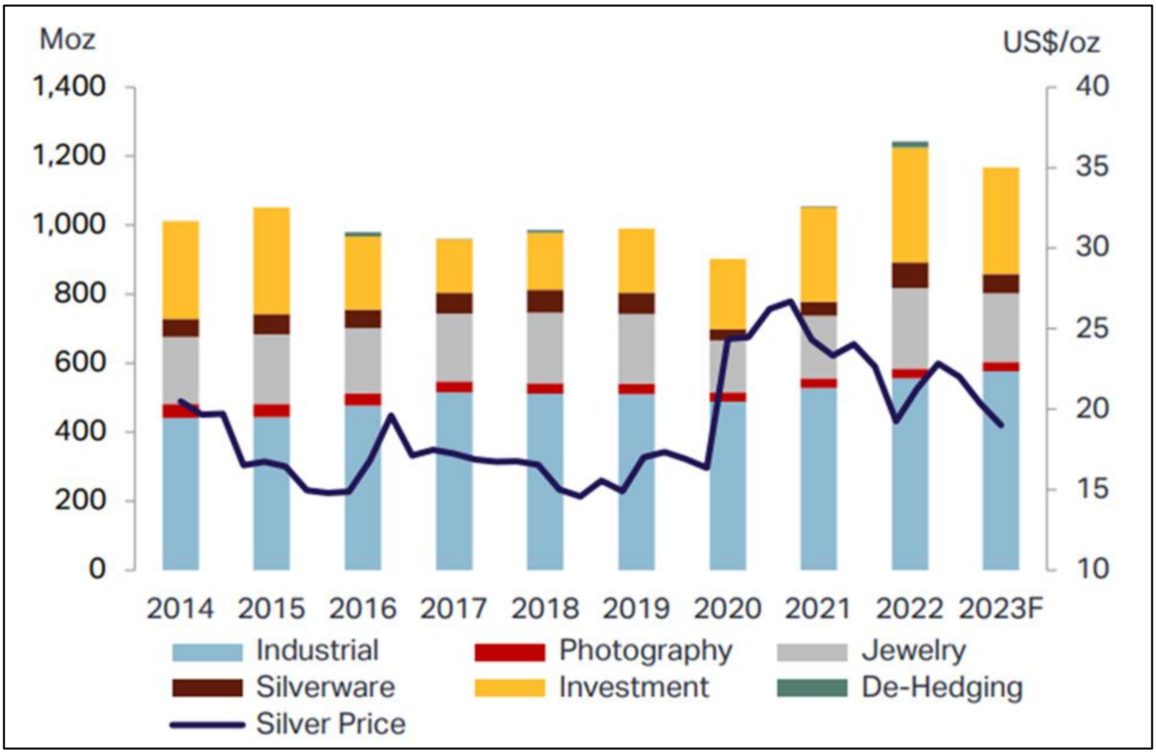


Figure 44: Silver Global Demand 2014-2023 ***(Source: Metals Focus, Bloomberg)***

Table 27 – Silver Supply & Demand

											Year on Year	
Million ounces	**2014**	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023F**	**2022**	**2023F**
Supply												
Mine Production	882.0	896.8	899.8	863.6	850.3	836.6	782.2	827.6	822.4	842.1	-1%	2%
Recycling	160.4	146.9	145.6	147.0	148.5	148.0	166.0	175.3	180.6	181.1	3%	0%
Net Hedging Supply	10.7	2.2	-	-	-	13.9	8.5	-	-	-	na	na
Net Official Sector Sales	1.2	1.1	1.1	1.0	1.2	1.0	1.2	1.5	1.7	1.7	13%	-1%
Total Supply	**1,054.2**	**1,046.9**	**1,046.4**	**1,011.7**	**1,000.0**	**999.5**	**957.9**	**1,004.5**	**1,004.7**	**1,024.9**	**0%**	**2%**
Demand												
Industrial (total)	440.9	443.4	477.4	515.3	511.2	509.7	488.7	528.2	556.5	576.4	5%	4%
Electrical & Electronics	269.8	272.3	308.9	339.7	331.0	327.3	321.8	351.0	371.5	382.3	6%	3%
...of which photovoltaics	48.4	54.1	93.7	101.8	92.5	97.8	100.0	110.0	140.3	161.1	28%	15%
Brazing Alloys & Solders	53.3	51.0	49.0	50.8	51.9	52.3	47.4	50.4	49.0	49.8	-3%	2%
Other Industrial	117.8	120.1	119.5	124.8	128.3	130.1	119.4	126.8	136.0	144.4	7%	6%
Photography	41.0	38.2	34.7	32.4	31.4	30.7	26.9	27.7	27.5	26.4	-1%	-4%
Jewelry	193.5	202.5	189.1	196.2	203.1	201.4	150.5	181.5	234.1	199.5	29%	-15%
Silverware	53.5	58.3	53.5	59.4	67.1	61.3	31.2	40.7	73.5	55.7	80%	-24%
Net Physical Investment	283.0	309.3	212.9	155.8	165.5	187.0	204.8	274.0	332.9	309.0	22%	-7%
Net Hedging Demand	-	-	12.0	1.1	7.4	-	-	3.5	17.9	-	409%	na
Total Demand	**1,011.9**	**1,051.7**	**979.7**	**960.2**	**985.7**	**990.0**	**901.9**	**1,055.6**	**1,242.4**	**1,167.0**	**18%**	**-6%**

Source: The Silver Institute

The Silver price has remained close to its 3-year mean of US$23.54/oz, currently sitting at US$22/oz, and relatively rangebound between US$23 and 24/oz over the past 3 months. Consensus Economics long term price forecasts vary between US$21.5 and US$27.4/oz.

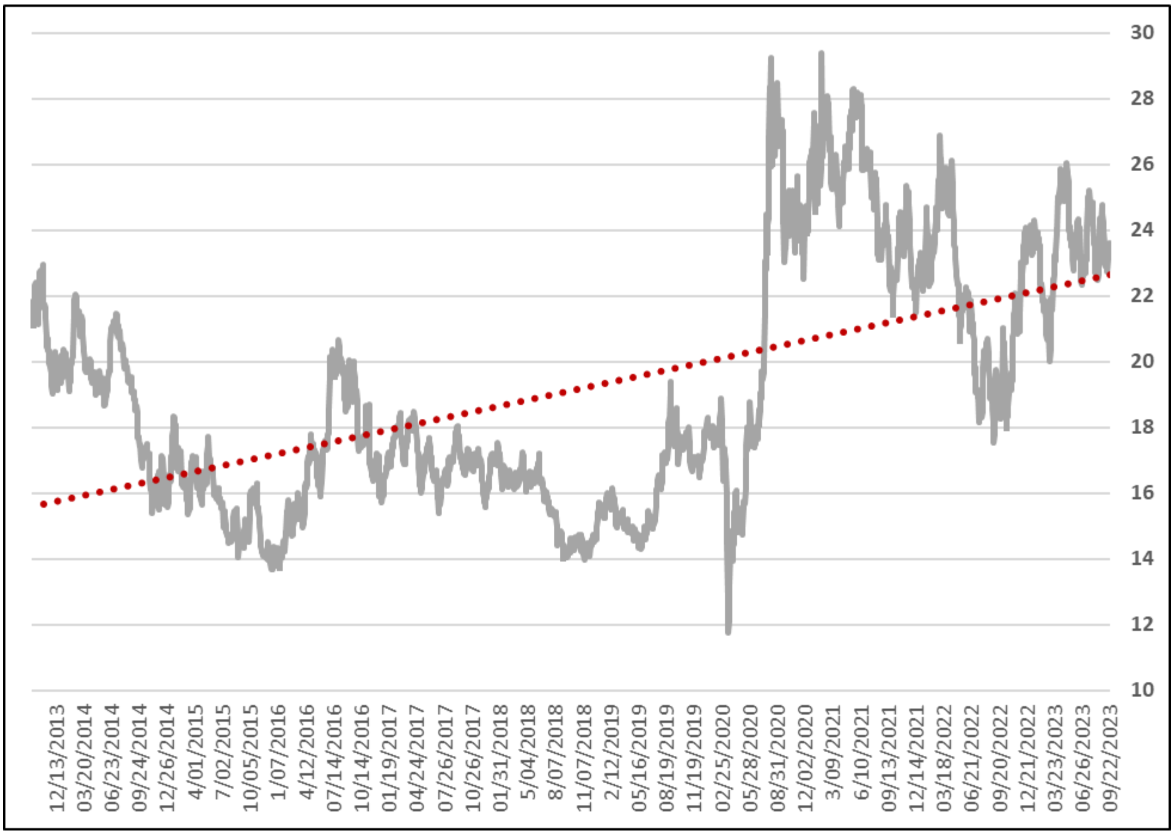


Figure 45: Silver 10 Year Historic Price

13.2.4 Foreign Exchange - AUD:USD

The AUD:USD currency exchange rate has been trending downwards over the past 10 years. The 5-year historic average of 0.70 sits well above the current spot of 0.635. Long term FX forecasts range between 0.65 and 0.71

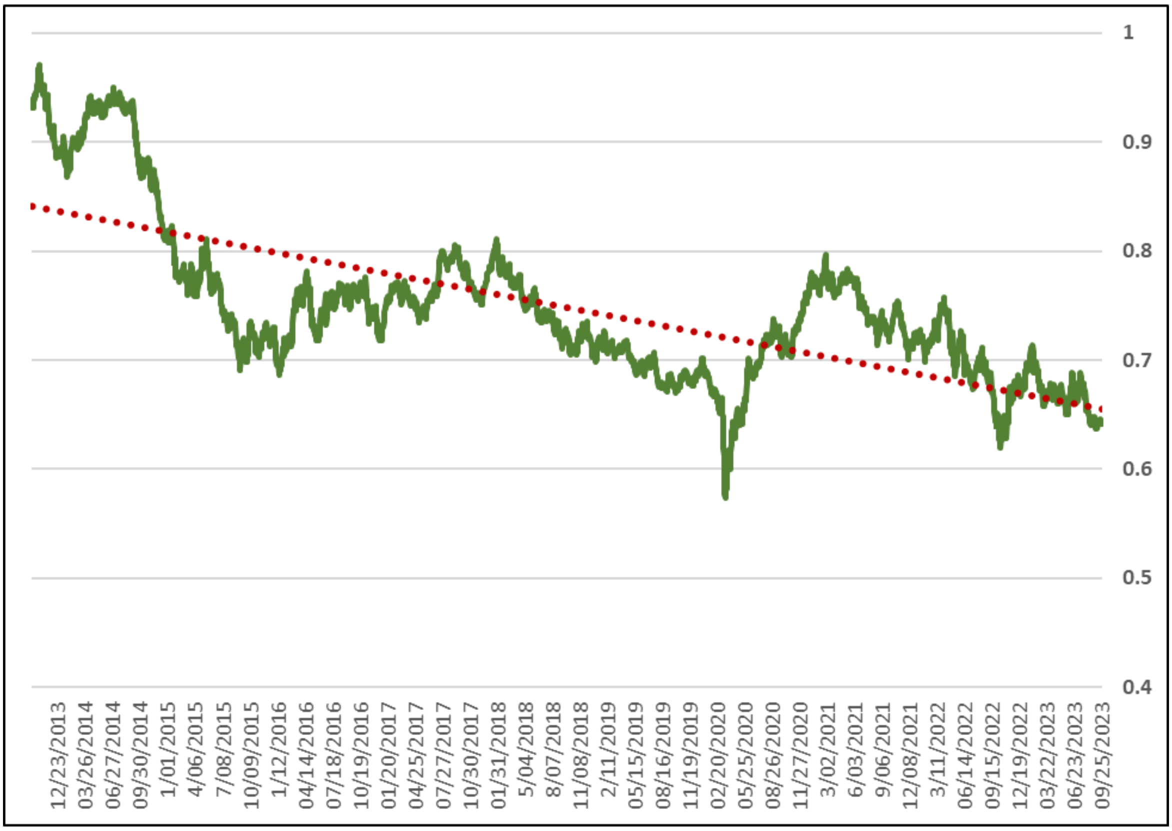


Figure 46: AUD:USD 10 Year Historic FX

13.3 Product Characterisation

The operation will initially produce two products from the flotation process, namely zinc and silver-lead concentrates. Each of these products can be competitively marketed, and Ocean Partners has identified buyers in Australia and Asia interested in purchasing both concentrates at terms consistent with the current market for concentrates with similar attributes.

Modelled LOM concentrate production and contained metal are shown in **Table 28**.

Table 28 – LOM Concentrate Production

Attributes	Zinc Concentrate	Silver-Lead Concentrate
Total Concentrate Produced (dmt)	498.939	129,911
Payable Metal in Concentrate	250,118 t Zn	65,917 t Pb 10,285,297 oz Ag

The zinc concentrate is classified as a mid-grade zinc concentrate and can be readily sold into smelters throughout Asia including China, Japan and South Korea. There are no price penalties historically on Zinc concentrate produced at Endeavor.

The lead concentrate is classified as a mid-grade lead / high-grade silver concentrate and can be sold throughout Asia, but likely to be sold and delivered by rail to Nyrstar's Port Pirie smelter in South Australia.

Overall payabilities were calculated individually for each of the Project ore sources based on the concentrate specifications, minimum deductions and payability thresholds provided by Ocean Partners. Average payabilities from concentrates produced over the LOM are:

- 84.04% Zinc.
- 94.09% Lead.
- 94.86% Silver.

13.4 Logistics

Ocean Partners were engaged by Polymetals to complete a review of the supply chain from mine to market for Endeavor concentrates. Lead concentrate was loaded into half-height bulk concentrate containers on rail wagons at the mine, in the concentrate storage and rail load out facility, and then railed to the Port Pirie smelter in South Australia. Zinc concentrate was loaded into half height bulk concentrate containers and railed to the Newcastle Port, where it was loaded onto bulk vessels in 5,000t or 10,000t parcels. The study confirms historic logistics is currently the best method for Endeavor's mine to market supply chain.

13.5 Economic Assumptions

Polymetals has taken a real price outlook for the economic assessment of the LOM production schedule. Using historic trends, consensus outlooks, spot prices and peer assumptions, Polymetals has formed a view on metal prices and foreign exchange rates. The financial model assumes flat metal prices across all years of the LOM schedule. A summary of the economic inputs are shown in **Table 29**.

Table 29 – Economic Assumptions

Metric	Unit	LOM
Zinc	US$/t	2,750.00
Lead	US$/t	2,200.00
Silver	US$/oz	23.00
Exchange Rates	AUD:USD	0.67

The assumed metal prices and exchange rate, projected over the next five years, compared to historic trends are shown in **Figure 47**.

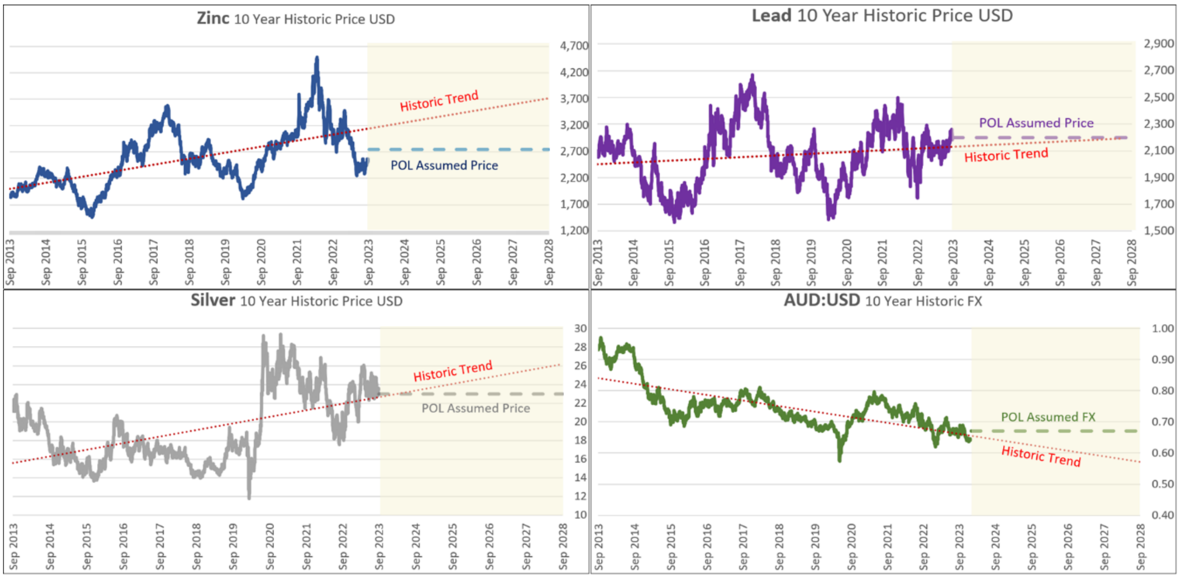


Figure 47: Metal Price and FX Assumptions v Historic Trends

14 Environmental, Social or Community Impact

14.1 Social or Community Impacts

The Endeavor Mine has had a long history in the Cobar region, having operated continuously for almost 40 years. In that time the mine has made a significant contribution to the local community in the following ways:

Employment Opportunities

The mine has been a significant source of employment for residents of Cobar and the surrounding areas. It has created jobs for a diverse workforce, including miners, engineers, technicians, and administrative staff, thereby reducing unemployment rates and providing a stable income for many local families.

Economic Growth

The mine has injected a substantial amount of capital into the local economy. Through various business contracts, procurement, and support services, it has stimulated economic growth, benefiting local businesses, suppliers, and service providers.

Infrastructure Development

The Endeavor Mine's presence has necessitated infrastructure development in Cobar. This includes improved road networks, housing facilities, and other essential amenities. The mine has often contributed to community projects that enhance the quality of life for local residents.

Community Investment

The mine has a strong tradition of investing in the local community. This involves supporting community initiatives, sponsoring events, and contributing to local charities and educational programs. These investments have helped enhance the overall quality of life in Cobar.

Environmental Responsibility

The Endeavor Mine has also demonstrated a commitment to environmental sustainability. It has employed modern, environmentally friendly mining practices and works towards minimizing its ecological footprint. This responsible approach ensures that the local environment remains healthy and viable for future generations.

Skills Development

The mine has provided opportunities for skill development and training for local workers. Through various training programs, employees have gained valuable skills that can be applied both within and outside the mining industry, enhancing their employability.

Social Integration

The Endeavor Mine fostered a sense of community by encouraging employees to engage in local events and activities. This helped to integrate the mine's workforce with the broader community and promotes a sense of belonging.

14.2 ESG Considerations

14.2.1 Company ESG Strategy

Developing a comprehensive ESG (Environmental, Social, and Governance) strategy is essential for organizations committed to sustainability, ethical practices, and long-term value creation. Below is a step-by-step ESG strategy framework that Polymetals will be considering and developing, as appropriate and in a fit-for-purpose way for a small Junior Resources company, as it prepares to re-start the Endeavor silver-zinc-lead mine:

Assessment and Benchmarking:

- Conduct an in-depth assessment of the company's current ESG performance and practices.
- Benchmark the company's ESG performance against industry peers and global standards like the Global Reporting Initiative (GRI) or Sustainability Accounting Standards Board (SASB) to identify gaps and opportunities for improvement. From the ASX (June 2023):-
- "*Investing with a consideration for environmental, social and governance (ESG) impacts and outcomes in mind has become increasingly mainstream. This year is likely to be a watershed for the ESG movement, with the International Sustainability Standards Board's (ISSB) project to devise a single global framework for disclosing sustainability information through the International Financial Reporting Standards (IFRS) Foundation gaining momentum.*"

Stakeholder Engagement:

- Identify and prioritise key stakeholders, including local communities, investors, regulatory bodies, and NGOs.
- Establish regular channels of communication to understand their concerns and expectations related to the mine's ESG performance.

ESG Policy and Commitments:

- Develop a clear and concise ESG policy that outlines the company's commitment to environmental stewardship, social responsibility, and strong governance.
- Ensure alignment with international standards and frameworks such as the United Nations Sustainable Development Goals (SDG's).

Risk Assessment and Management:

- Identify ESG-related risks that could impact the mine's operations, reputation, and financial performance.
- Develop risk mitigation strategies and integrate them into your overall risk management framework.

Environmental Responsibility:

- Implement initiatives to minimize the mine's environmental footprint, including energy efficiency, waste reduction, water management, and biodiversity conservation.
- Invest in responsible sourcing of materials and sustainable resource management.

Social Responsibility:

- Foster positive relationships with local communities by creating jobs, supporting education, and engaging in community development projects.
- Ensure safety and health standards for mine workers and promote diversity and inclusion within the workforce.

Governance and Ethics:

- Strengthen corporate governance practices, including board diversity, transparency, and accountability.
- Implement anti-corruption measures and adhere to ethical business practices.

Reporting and Transparency:

- Develop a robust ESG reporting framework that includes key performance indicators (KPI's) related to ESG goals and targets.
- Publish regular ESG reports, following recognized reporting frameworks like GRI, SASB, or the Task Force on Climate-related Financial Disclosures (TCFD).

Investor Relations:

- Engage with investors who prioritize ESG factors and communicate your ESG efforts and progress.
- Explore opportunities for sustainable financing, such as green bonds or sustainability-linked loans.

Continuous Improvement:

- Establish a culture of continuous improvement, where ESG goals are regularly reviewed and adjusted.
- Encourage innovation to find new ways to reduce environmental impact and improve social outcomes.

Regulatory Compliance:

- Stay up-to-date with evolving ESG regulations and ensure compliance with relevant laws and standards.
- Proactively engage with regulators to provide input on ESG policies and regulations.

Monitoring and Verification:

- Implement a robust monitoring and verification system to track progress toward ESG goals.
- Consider third-party audits or certifications to enhance credibility.

Supply Chain Responsibility:

- Extend the company's ESG efforts to suppliers and contractors by setting expectations for responsible practices throughout the supply chain.

Long-Term Vision:

- Develop a long-term ESG vision that aligns with the company's overall strategic objectives, creating value for all stakeholders.

Education and Training:

- Provide training and awareness programs to ensure that all employees understand and contribute to the company's ESG goals.

Public Relations and Reputation Management:

- Actively engage with the media and the public to communicate the company's commitment to ESG principles and demonstrate progress.

By following this comprehensive ESG strategy, it is believed that Polymetals can not only mitigate risks but also build a sustainable and socially responsible business that contributes positively to the environment and society while maintaining strong governance standards.

Real Action

Examples of the company's commitment to the environment and minimising the impact of mining on the environment are:-

Reducing CO_2 emissions through renewable energy – Solar Power

Investigations are underway into the establishment and operation of a solar power facilty to generate renewable energy for the mining operation. Example:- 10.6MW solar farm, Degrussa Mine, Sandfire Resources.

Figure 48: Example Solar Power Facility – Degrussa Mine

Energy from Waste – Post Mining Land Use

Energy from waste involves the thermal treatment of waste or waste-derived materials in order to recover energy. In an Energy from Waste (EfW) facility, non-recyclable waste is burned to heat water, resulting in steam that powers a turbine to generate electricity. Waste that would have otherwise been disposed of at landfill instead goes to an EfW facility. The electricity produced is then sold to the power company directly by linking the plant to the grid. Example:- Woodlawn Mine NSW, 30MW pa.

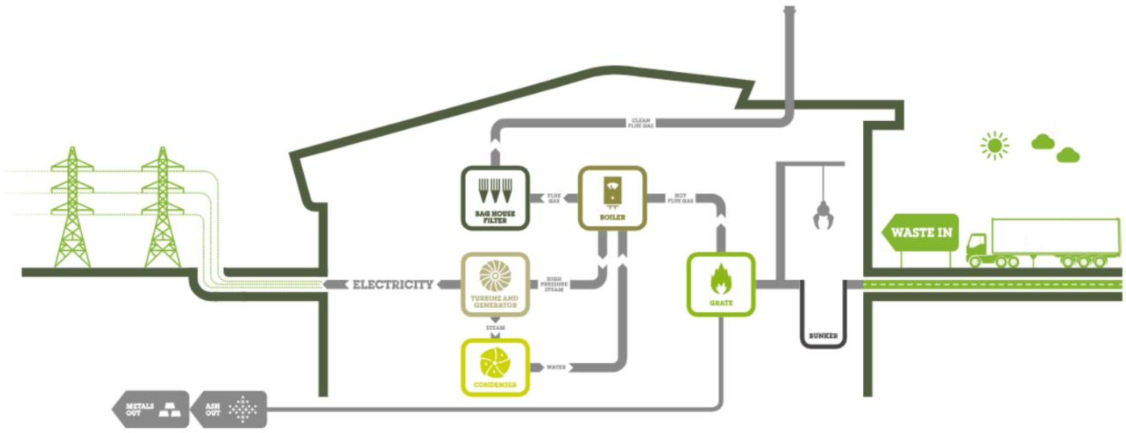


Figure 49: Example Energy from Waste Power Facility

15 Capital and Operating Costs

The Endeavor Mine Restart Study has been compiled during a period of high inflationary pressure, due to global factors, that has impacted most aspects of the industry. Where possible, up to date quotes have been requested from suppliers and used in the estimates.

Capital and operating costs have been estimated to accuracies of +/- 15% to +/- 25%.

15.1 Capital Cost Estimate

The estimates of capital expenditure (Capex) were compiled by Polymetals, where possible using rates and quotes received from contractors and suppliers and are quoted in Australian dollars (AUD).

The total pre-production capital estimated to be required for the recommencement of operations at the Endeavor Mine is **$23.8 M**, including a 20% contingency. The Capex estimate has been divided into four categories as detailed in **Table 30**.

Table 30 – Endeavor Mine Restart Pre-Production Capital Estimate

Cost Area	Cost (A$M)
Processing Fixed Plant	5.1
Mining Fixed Plant	3.6
Mobile Plant	3.1
Site Establishment	8.0
Contingency (20%)	4.0
Total Pre-Production Capital	**23.8**

Operating capital over the life of the operating mine (an initial 10 years) has been estimated to be **$53.6 M** and can be broken down into five categories as follows:

Tailings Storage Capacity Increase

Golder Associates provided a cost estimate for the Central Thickened Discharge (CTD) Tailings TSF stage 3 raise in 2019 from which Polymetals has developed an estimate of **$4.2 M**. Expenditure on the raise has been estimated to commence in Month 20 of the project.

Upper Main Lode

A provision of **$1.6 M** has been made for extra ventilation and mine services when the Upper Main Lode is mined. To ensure maximum ore recovery an allowance has been made for ground stabilisation (grouting and void filling).

Deep Zinc Lode

Capex of **$9.5 M** has been allocated for bulk air cooling, booster fans, access, compliance, dewatering, electrical and engineering.

Sustaining Capital

Sustaining capital provisions of **$35.7 M** were made for underground mining development to access production areas over the LOM schedule, with some 6.5km of capitalised underground development scheduled to be completed over its initial 6-year mine life.

Tailings Mining

Capex of **$2.6 M** has been allocated for the provision of hydraulic mining equipment (monitors), pipe work, discharge hopper & screens, electrical and engineering.

In summary, the life of mine Capex costs total **$77.4M**, including pre-production Capex of **$23.8 M** and operating Capex of **$53.6 M** as shown in **Table 31**.

Table 31 – Endeavor Mine Restart Total Capital Estimate

Cost Area	Cost (A$M)
Pre-Production	
Processing Fixed Plant	5.1
Mining Fixed Plant	3.6
Mobile Plant	3.1
Site Establishment	8.0
Contingency (20%)	4.0
Total Pre-Production Capital	**23.8**
Operating	
Tailings Storage Capacity Increase	4.2
Upper Main Lode	1.6
Deep Zinc Lode	9.5
Sustaining Capital	35.7
Tailings Mining	2.6
Total Operating Capital	**53.6**
Total LOM Capital	**77.4**

15.2 Operating Costs

The Operating Expenditure (OPEX) for the Project, summarised in **Table 32**, has been estimated from first principles for an operating cost model that incorporates input costs for mining, processing, maintenance, administration / commercial, HSETS (Health, Safety, Environment, Training & Stores), and housing costs. The mining component was validated using a third party mining cost estimate.

LOM OPEX, which includes all costs of mining, processing, site administration, royalties, selling and transportation costs, but excludes corporate costs of the Company are calculated at **$990 M**.

Table 32 – Endeavor Mine Restart Operating Cost Estimates

Cost Area	Ore Source	Cost (A$M)	Cost per tonne ore (A$/t)
Mining	Underground	305.5	86.6
	Tailings	18.8	3.9
Processing	Underground	98.5	27.9
	Tailings	94.6	19.6
Maintenance	Underground	72.4	20.5
	Tailings	34.7	7.2
General Admin	Underground	62.8	6.0
	Tailings	23.9	1.6
HSETS	Underground	21.9	6.2
	Tailings	8.0	1.7
Housing	Underground	16.1	4.6
	Tailings	5.9	1.2
TC/RC, Transport, Shipping	Underground	212.5	60.2
	Tailings	72.3	15
Totals	**Underground**	**748.0**	**212.0**
	Tailings	**242.1**	**50.1**
	Combined	**990.1**	**262.1**

Indicative pricing for concentrate transport, port fees, and shipping. Realisation costs are summarized in **Table 33**.

Table 33 – Realisation Costs

	Zinc Concentrate	Silver-Lead Concentrate
Moisture	9.5%	7%
Rail & Loading	A$72/wmt	A$72/wmt
Assay	A$1/wmt	A$3.03/wmt
Shipping	US$35/wmt	-

15.2.1 Treatment & Refining Charges (TC/RC's)

Benchmark treatment charges and refining charges (TC/RC's) have been used for the study. For Zinc, the Teck/KZ Red Dog Benchmark TC's are applied, and for Lead-Silver the Cannington/KZ Benchmark TC/RC's.

Historically, concentrates from Endeavor have never exceeded contained metal above upper threshold of 54% with LOM historic grades being 50.13% Zn & 50.74% Pb respectively. Therefore, payabilities for metals in concentrate have been modelled as shown in **Table 34**.

Table 34 – Metal Payabilities

Payability	Zinc	Silver-Lead
Minimum Deduction	8%	3% Pb, 50 g/t Ag
Payability	100%	100%
Example Calculation	*1,000t parcel @ 50% Zinc grade* 1. 50% - 8% = 42% 2. 1,000t x 42% = 420t Zinc payable 3. 420t x LME Zinc Price = Revenue	*1,000t parcel @ 50% Lead grade, 1,000 g/t Silver* 1. 50% - 3% = 47% Lead payable 2. 1,000 g/t – 50 g/t = 950 g/t Ag payable 3. Payable metal units x spot price = Revenue

15.2.2 Workforce Modelling

Personnel requirements were modelled on a ramp-up and ramp-down scenario based on total tonnes of material moved in the underground mining schedule. Key positions such as senior management (General Manager, Safety & Training, Administration and Human Resources) are the first to be recruited to manage pre-production activities. The remaining roles are systematically filled in response to need, with peak personnel numbers of 230 being reached in month 33. The total maximum number of personnel required for the LOM plan, subdivided by department is shown in **Table 35**.

Table 35 – Personnel Numbers

Department	Maximum Personnel
Administration / Commercial	8
Mine Technical	16
Mine Management / Supervision	8
Mining	91
Processing	36
Maintenance	54
HSETS	14
Housing	5
Total	**232**

16 Economic Analysis

A financial analysis of the Project was carried out by a cashflow model using outputs of the LOM scheduling process, CAPEX and OPEX estimates, and economic assumptions as outlined in Section 13.5. The analysis is based on a mine life of 10 years, with mining of underground ore from Years 1 to 6 and re-treatment of Sector 1 tailings from Years 5 to 10. Mining is scheduled to commence 8 months after site establishment begins, with processing to commence 2 months after mining starts.

The financial model estimates monthly pre-financing cashflows for the LOM in Australian dollars, with the evaluation reported on a pre-tax basis. Net present Valus (NPV) is calculated using a pre and post-tax discount rate of **8%**.

A summary of the key economic outcomes from the financial analysis of the re-commencement of mining and processing at the Endeavor Mine are shown in **Table 36** while the key physical outputs are shown in **Table 37**. A chart of the yearly cashflow profile is provided in **Figure 51**.

Table 36 – Key Economic Outcomes

Output Metric	Unit	Outcome
Project Revenue	A$M	1,411.9
Free Cashflow	A$M	323
Pre-Production Capital	A$M	23.8
NPV	A$M	201
IRR	%	91
Payback	Months	27
Maximum Cash Drawdown	A$M	37.8

Table 37 – Key Physical Outcomes

Output Metric	Unit	Outcome
Mined Ore Tonnes	Mt	8.4
Nominal Throughput Underground Ore	Mtpa	0.6
Nominal Throughput Tailings	Mtpa	1.1
Life of Mine	Years	10
Processed Tonnes	Mt	8.4
Avg. Zn Grade	%	3.98
Avg. Pb Grade	%	1.60
Avg. Ag Grade	g/t	75
Payable Zinc Metal	Kt	210.2
Payable Lead Metal	Kt	62
Payable Silver Metal	Moz	9.8

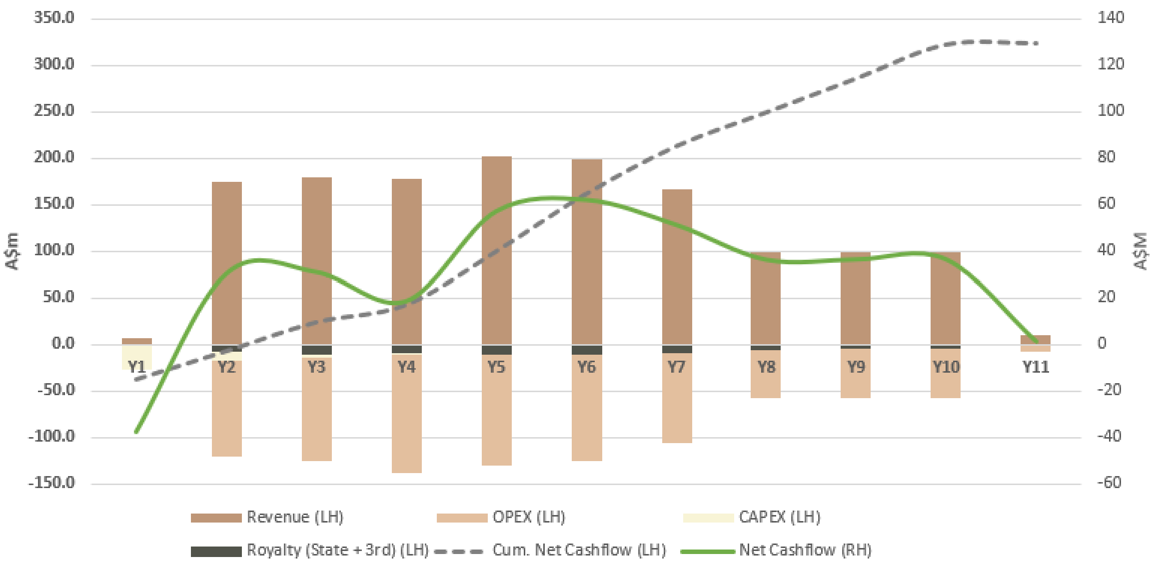


Figure 50: Project Cashflow Profile

16.1 Sensitivity Analysis

The sensitivity of the Project NPV_8 to variations in metal grades, metal prices, metal recoveries, foreign exchange rate, CAPEX and OPEX have been modelled with the results shown in **Figure 51**. This analysis shows the project is most resilient to Capex, Pb/Ag grades, and metal recoveries, with significant upside if Zn prices are above $2,750 US$/t. The project is most sensitive to changes in the AUD:USD exchange rate.

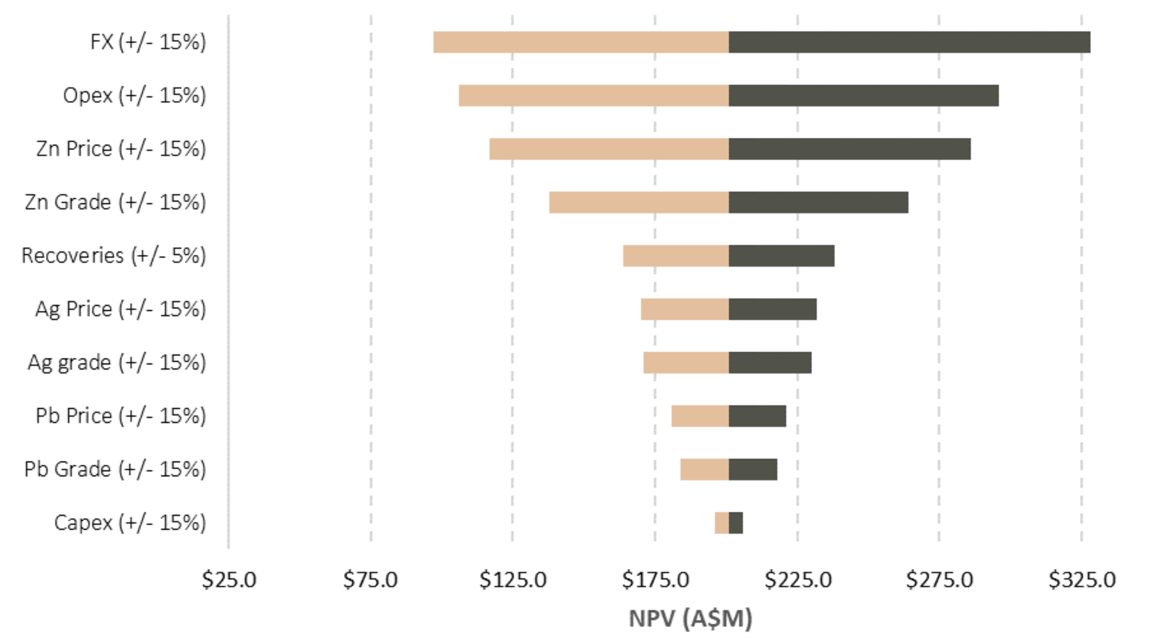


Figure 51: Project NPV Analysis

17 Risks and Opportunities

The identification, assessment, and management of risks and opportunities are fundamental to the process of evaluating the feasibility of any project. A thorough understanding of these factors allows for informed decisions to be made. The company has identified and reviewed the main risks and opportunities that could influence the short-term and long-term success of the Project and are outlined as follows.

17.1 Key Risks

Recruitment

Being able to recruit the required number of people with the suitable skills set in a timely manner will be a major challenge for the project given the current skill shortage situation in the industry. This is somewhat offset by the location and history of the mine. A number of suitably qualified personnel reside in Cobar, with anecdotal evidence that many would like to return to the Endeavor Mine if it were to reopen. The Project's proximity to Cobar provides employees with a choice of residential or drive-in-drive-out arrangements, possibly increasing the pool of potential employees with differing lifestyle preferences.

Procurement of Mining Fleet

Advice from mining equipment suppliers indicates the possibility of extended lead times for delivery of mobile equipment for mining. For those items for which delivery times would hinder the planned production schedule the company will look to temporarily hire equipment or retain the services of a mining contractor.

Project Funding

The mine restart will be dependent on the Company's ability to secure funding for the re-development, ensuring sufficient coverage for the peak cash negative drawdown of A$37.8 million. The company has secured a binding US$10 million unsecured pre-payment facility with concentrate trading partner Ocean Partners UK Limited. Additionally, the Company is advanced in its process of securing a project financing facility to support the mine restart. The robust project economics along with existing US$10 million unsecured pre-payment facility are key mitigations to the risk of not securing project finance.

Operational

- **Resource tonnage and grade** – Approximately 34% of the material in the LOM schedule is derived from Inferred Resources. The mine plan includes capital for grade control drilling to increase the confidence in stope tonnes and grade, as well as the mining of dedicated development drives to act as platforms for drilling previously inaccessible areas of the Deep Zinc Lode currently hosting Inferred Mineral Resources.
- **Mining Production Rate –** The current mine plan relies on the supply of ore originating from areas spread throughout the entire vertical profile of the mine. Whilst this results in logistical pressures, the existence of multiple production sources will also result in increased flexibility in case unforeseen circumstances were to delay production in a particular area. To account for any issues that may arise that impact on production a conservative mining rate has also been built into the model.

17.2 Key Opportunities

Additions to Ore Reserves

There are a number of areas within the mine that have the potential to add significant tonnes to the currently identified Ore Reserves and extend the life of the underground operations.

- **Stope Skins** – Stope skins are the 5m buffer zone around previously mined primary stopes. The stopes included in the LOM schedule were optimised based on the exclusion of these skins. Final stope designs included these skins where a planned stope is adjacent to loose rock filled void and which will be stabilised by grout injection. When operations commence the remaining stope skins will be assessed for inclusion in Ore Reserve updates.
- **Pillar Recovery of the Upper Main Lode** – When mining commences in the Upper Main Lode, the ground conditions will be assessed, and if found to be favourable, the mining method will be reviewed to assess the risks of lower cost mining methods and/or recovery of planned pillars. Approximately 77,000 t of high-grade ore could be recovered by pillar extraction.
- **Extension of the Deep Zinc Lode** – The Deep Zinc Lode mineralisation remains open along strike and down dip due to the inability to drill test areas because of unfavourable drilling angles from the currently available drilling platforms. The development of a dedicated diamond drilling platform, in the form of a 280 m long drive to be mined in the hangingwall of the ore body from the current level of the decline, has been included in the mine plan. This platform will allow for infill drilling of the current known ore body as well as drilling for extensions.
- **New Northwestern Pods** – The existence of more mineralisation northwest of currently defined pods remains poorly tested. Drill intersections at depth below this zone contain mineralisation grades similar to those below the northern pods. The development of a dedicated diamond drilling platform, in the form of a 150m long northern extension drive to be mined on the 500 Level, has been included in the mine plan to test this area.

Improved Precious Metals Recovery

As mentioned in Section 10.7.1 there is an opportunity to investigate potential gold and silver recovery via cyanidation of supergene ore and Sector 1 tailings. An option is to store tailings from high grade silver/gold ore separately for later treatment if test work confirms viability.

Long Term Site Options

The Project site contains a broad range of high-quality infrastructure items such as paved road access, rail siding, water and power supply, that provide increased optionality for concurrent or post mining land use options such as:

- Toll treatment of ore from the region.
- Establishing a renewable energy hub.
- Construction of a sulphide roasting plant and production of sulphuric acid.
- Zinc metal or chemical production using renewable energy.

18 Work Program

The positive results from the Mine Restart Study have encouraged the Company to pursue funding arrangements to raise the required pre-production capital to return the Endeavor Mine to an operating state. The Company is able to immediately commence preparations for the resumption of mining and processing once funding is secured. The pre-production activities will be planned and executed by the Company's senior management, with the first tasks being the recruitment of personnel in positions critical for the timely execution of the re-start plan and the ordering of long lead-time items of equipment. A broad outline of the execution schedule is provided in **Figure 52**.

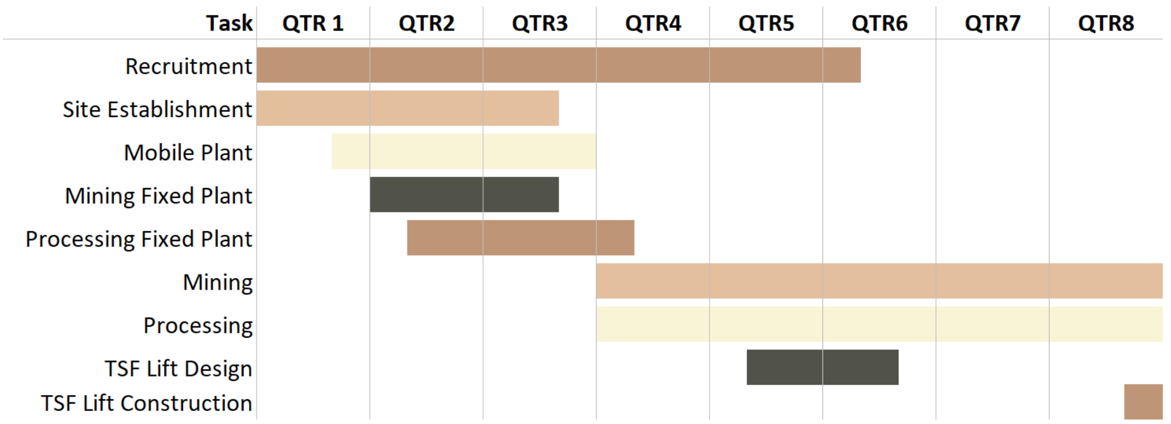


Figure 52: Project Execution Schedule

19 Competent Persons Statements

The information in this report that relates to Exploration Results and Mineral Resources of the Endeavor Project is based on information compiled by Troy Lowien, a Competent Person who is a Member of The Australasian Institute of Mining and Metallurgy. Mr Lowien is a full time employee of Polymetals Resources Ltd. Mr Lowien has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'. Mr Lowien consents to the inclusion in the report of matters based on his information in the form and context in which it appears.

The information in this report that relates to Ore Reserves of the Endeavor Project is based on information compiled by Matthew Gill, a Competent Person who is a Fellow of The Australasian Institute of Mining and Metallurgy. Mr Gill is a non-executive Director of Polymetals Resources Ltd. Mr Gill has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'. Mr Gill consents to the inclusion in the report of matters based on his information in the form and context in which it appears.

20 References

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ATTACHMENTS

Attachment 1

JORC Code (2012) Table 1

JORC Code, 2012 Edition – Table 1

Section 1 Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.* • *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* • *Aspects of the determination of mineralisation that are Material to the Public Report.* • *In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.*	**Underground Resource** • Diamond drilling was carried out to define the mineralisation from which variable length samples (predominantly 1 or 2m) were obtained which were crushed, pulverized and split to 200 – 300 ml aliquots for assay by Aqua Regia digest followed by AAS. • Sludge samples were taken during underground percussion drilling to determine mineralized extents. These samples were used as a guide only for interpretation and not used in grade estimation. • During Feb-March 2023 reverse circulation percussion drilling was carried from the surface to target the upper Main Lode. Samples were all collected by qualified geologists or under geological supervision. Representative samples of the material drilled were collected for every metre drilled. 2 x 2-4kg samples (one for assay and a duplicate) and a bulk sample of the remainder of each metre was collected directly from the rig cyclone. **Tailings Resource** • 2014 Drilling – Air core drilling was used to obtain 1m samples from which 2m composite samples were created for assay by acid digest. • 2015 Drilling – Push tube drilling was used to obtain an average sample length of 1.2m from which sub samples were collected for assay by acid digest. • 2017 Drilling - Push tube drilling was used to obtain an average sample length of 1m from which various composites were created for metallurgical test work.
Drilling techniques	• *Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	**Underground Resource** • Diamond Drilling has been carried out from surface and underground locations, with the majority having been drilled from underground development. • Overall, there are 2,538 diamond drill holes in the database, totaling 402,359m

Criteria	JORC Code explanation	Commentary
		of drilling. Of those, a total of 2,459 holes totaling 389,697m of drilling were used in the Mineral Resource estimation • Holes drilled prior to 2011 (1,648 holes for 297,896m) were predominantly BQ in size with some AQ size core. Holes drilled post 2011 varied in size from BQ up to HQ, with the majority LTK60. • No core orientation has been recorded. • Reverse circulation drilling was carried out in Feb-March 2023 and consisted of 21 drill holes, using a Schramm 1200 with an onboard 350 psi/900 cfm compressor. An auxiliary air booster was used on all holes. The drill string utilised standard 6m rods and a 5 ½ inch face sampling hammer. **Tailings Resource** • 2014 Drilling - Aircore methods on where a 100mm cutting bit with a hollow centre is pushed through unconsolidated material using rotation. Air is pumped through an annulus between the inner and outer tubes of the drill string and out through orifices in the cutting head. Sample is returned up the centre of the drill string and collected in a cyclone. • 2015 and 2017 Drilling - Push tube methods where casing is advanced down the hole and a solid “core” of unconsolidated material is extracted from within the casing encased in a rigid plastic sleeve
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.* • *Measures taken to maximise sample recovery and ensure representative nature of the samples.* • *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	**Underground Resource** • The core trays were laid out along racking systems, washed down and metre marked by the field technician using a chinagraph pencil and/or permanent marker and then measured for recovery and RQD information. • Diamond Drilling - Core recovery (total core recovery) averaged >98% and the average RQD was 61%. • Recovery in the March 2023 reverse circulation percussion holes was visually estimated and was generally close to 100% apart from voids encountered due to underground development and vughs in the supergene zone. The average recovery of samples in the supergene zone was 83%. • There is no apparent relationship between sample recovery and grade. The ore is competent with no apparent loss of fine or coarse material that would introduce bias.

Criteria	JORC Code explanation	Commentary
		Tailings Resource • No recovery information is available. • During the 2014 air core drilling program the sample collection cyclone was vigorously cleaned after each 1m interval to ensure complete sample recovery.
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.* • *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.* • *The total length and percentage of the relevant intersections logged.*	**Underground Resource** • All diamond drill core was delivered to the core yard compound on surface at the end of each shift by the drilling contractor where it was then prepared for logging and sampled by the geologist and field technician. The core trays were laid out along racking systems under cover that provided adequate working conditions in all weather. The core was washed down and metre marked by the field technician using a chinagraph pencil and/or permanent marker and then measured for recovery and RQD information. The geologist then followed by logging the core using coloured chinagraph pencils to mark-up structures, mineralised domains and sampling intervals. • Core was routinely photographed and stored in racking systems or on pallets in a core farm. • A recent review of the core storage by the CP has revealed a high degree of oxidation and destruction of core that has been exposed to the elements. • Reverse circulation percussion drill chips were logged for lithology, mineralisation, weathering, alteration, colour, and any other relevant characteristics. Geological logging conformed to the standardised system adopted by the previous operators of the project. • Logging was both qualitative of quantitative depending on the characteristic being recorded. Small representative samples of chips are stored in chip trays for reference. The whole length of each hole was logged. **Tailings Resource** • Detailed logging of the tailings is considered impractical and unnecessary as the tailings have been homogenised from processing and deposition. Material changes were noted when drill holes intersected the base of the tailings dam

Criteria	JORC Code explanation	Commentary
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.* • *If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.* • *For all sample types, the nature, quality and appropriateness of the sample preparation technique.* • *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.* • *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.* • *Whether sample sizes are appropriate to the grain size of the material being sampled.*	**Underground Resource** • Diamond Drilling - Core was cut down the structural long axis using a fully automated Almonte Core Saw. Core samples were half cut or alternatively, quarter cut if the sample is submitted as a duplicate. • Historically, most sample preparation was carried out at the onsite laboratory with overload sent to ALS Orange. • Samples were crushed in a small jaw crusher and a split was placed into the pulveriser. • Samples were then pulverized to pass 38 micron and split to usually a 200-300ml aliquot. • Sample sizes are appropriate for the grain size of the material being sampled. • No systematic collection of field duplicate or second half sampling was recorded. • RC Drilling - The top 12m of each hole were not sampled as this interval was predominantly fill material. Due to the closely spaced nature of the drill holes, only selected holes were sampled above the mineralised domains (above 72mRL). These samples were composed of 4m composites, collected from each 1m interval using spear methods. Below 72m samples were collected on an individual 1 metre basis directly from the on-rig cone splitter. Samples were all collected by qualified geologists or under geological supervision. Representative samples of the material drilled were collected for every metre drilled. 2 x 2-4kg samples (one for assay and a duplicate) and a bulk sample of the remainder of each metre was collected directly from the rig cyclone. **Tailings Resource** • During the 2014 air core drilling, 2m composites were taken from 1m samples intervals by spear method., as the material was too puggy for a riffle splitter. • Push tube samples were split laterally down the hole with one side used to create metallurgical sample composites and the other side for assay. • Sample preparation was carried out at the onsite laboratory for the 2014 program and ALS Orange for the 2015 program. Sample preparation of the metallurgical composites was carried out at ALS Burnie. • Field duplicate sampling results indicate no issues with the methods used for collection of sub samples. • Sample sizes are appropriate for the grain size of the material being sampled.

Criteria	JORC Code explanation	Commentary
Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.* • *For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.* • *Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	**Underground Resource** • Samples were assayed at the Endeavor laboratory using an Aqua Regia digest with atomic absorption spectrometry (AAS) for lead, zinc, silver, iron and copper analyses. • Sample sent to ALS-Orange were assayed by an Aqua Regia digestion using AAS (ICP-AES) analysis for lead, zinc, silver, iron and copper. The prepared sample is digested in 75% aqua regia for 120 minutes and after cooling, the resulting solution is diluted to volume (100mL) with de-ionised water, mixed and then analysed for inductively coupled plasma-atomic emission spectrometry or by atomic absorption spectrometry. • Assay techniques are considered total and appropriate for the mineralisation style. • There is no documentation of the systematic collection of field duplicates • Quality Control procedures appear to have been implemented at the Endeavor Mine in 2005 with the accuracy of the assay data and the potential for cross contamination of samples during sample preparation assessed based on the assay results for the field standards and blanks. Standards (including blanks) have been inserted at the rate of approximately one in 20 samples • During 2018-2019 all four of the standards used during the year performed better than the previous 12 month although Ag continued to produce some variability (with 4 outliers from 93 samples) in the low grade OREAS 131B as shown in Figure 6. A total of 367 CRM samples were assayed throughout 2018-2019 with 277 going to the mine lab and the remaining 90 going to ALS/Orange. Of the 11 outliers greater than 10% above or below the expected value, three were analysed at ALS and eight analysed at the mine lab. The 11 outliers comprised six Ag (1.6% of total CRM analyses), two Pb (0.5%) and three Zn (0.8%) assays. • A total of 364 blanks were added to the sample stream during the 2018-2019 drilling programs. A small percentage of samples reported Pb and Zn grades above the level of detection (BLD), but these were considered to be well within acceptable limits given the low grades being reported • Previous reporting on internal laboratory accuracy and precision has not raised any significant issues. • Samples from the March 2023 drilling program were sent to North Australian

Criteria	JORC Code explanation	Commentary
		Laboratories in Pine Creek NT. Base metals including Pb, Zn, Cu and Ag were determined by a four-acid digest procedure. Initial charge weight is 0.5g with metal concentrations determined by ICP analysis of final diluted solutions. If Cu, Pb or Zn exceed 10,000ppm then an Ore Grade procedure is used reducing charge size to 0.3g. If Ag exceeds 100ppm the analysis is repeated as an Ore Grade digest with excess HCL added to maintain Ag in solution for ICP analysis. • Gold grades were determined using fire assay method, a fusion technique which breaks down the mineral content of the sample completely. The PbO flux is reduced to Pb metal during the fusion process, and precious metals are accumulated within the resultant Pb prill. Dissolution of the prill, and measurement of the abundance in the resultant solution provides a precise and accurate measure of the total Au abundance in the sample. • During the March 2023 drilling program field duplicate samples were collected at a rate of 1in 20 samples. Certified reference material (standards) were inserted in to the sample stream at a rate of 1 in 20 samples. • Acceptable levels of precision and accuracy have been established. **Tailings Resource** • 2014 Drilling - Samples were assayed at the Endeavor laboratory using an Aqua Regia digest with atomic absorption spectrometry (AAS) for lead, zinc, silver, iron and copper analyses. • 2015 Drilling – Samples were sent to ALS-Orange were assayed by an Aqua Regia digestion using AAS (ICP-AES) analysis for lead, zinc, silver, iron and copper. The prepared sample is digested in 75% aqua regia for 120 minutes and after cooling, the resulting solution is diluted to volume (100mL) with de-ionised water, mixed and then analysed for inductively coupled plasma-atomic emission spectrometry or by atomic absorption spectrometry. • Assay techniques are considered total and appropriate for the mineralisation style. • The quality control regime used in the 2014 drilling program consisted of Certified Reference Material (CRM) and Blanks inserted into the sample stream, field duplicate samples, and re-assays of laboratory pulp samples. The insertion rate of QC samples into the submission stream was 1 in 6

Criteria	JORC Code explanation	Commentary
		samples. • The quality control regime used in the 2015 drilling program consisted of CRM and Blanks inserted into the sample stream at a rate of about 1 in 10 samples. However, these samples were not assayed at the laboratory due to insufficient sample quantities according to the results certificate. Instead, assay accuracy and precision were assessed based on CRM and pulp duplicates inserted in the sample stream by the laboratory. • No recorded quality control samples were included in the submission of the 2017 samples to the metallurgical laboratory. • Assessment of the QC data from the 2014 drilling indicate acceptable levels of precision but an issue with the accuracy of Pb assays, showing a significant bias to lower grades. • Acceptable levels of precision and accuracy have been established for the 2015 assays.
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.* • *The use of twinned holes.* • *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.* • *Discuss any adjustment to assay data.*	**Underground Resource** • The Competent Person inspected mineralised intervals in core and underground exposures during site visits. A selection of original laboratory certificates were also located and verified against database entries. No errors were found. • No twinned holes were assessed. There are a number of drill holes that have intercepted mineralisation within relatively close proximity to each other and these drill holes have been investigated. Holes located less than 10m apart were assessed and found to have satisfactory levels of similarity and acceptable to be used in Resource estimation. • The geology department kept written procedures for data collection and storage. A user manual was written for the use of the Drilling Management system (MS Access Database). • The Competent Person is not aware of any adjustment to assay data. **Tailings Resource** • There are no records of independent or alternative verification of significant intersections.

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>
The 2015 drill holes were drilled as twins of selected holes from the 2014 program. The results show overall increase in grades for Zn, Pb and Ag, up 112%. Further investigation has ascertained that the magnitude of the differences for each element do not corelate with any particular holes or areas of the TSF. This indicates an issue with the 2014 sample representivity and therefore have been rejected for use in resource estimation.
The geology department kept written procedures for data collection and storage. A user manual was written for the use of the Drilling Management system (MS Access Database).
The Competent Person is not aware of any adjustment to assay data.
</td></tr>
<tr><td>Location of data points</td><td>
Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.
Specification of the grid system used.
Quality and adequacy of topographic control.
</td><td>
The Endeavor Mine is situated within Zone 55 of the MGA94 grid coordinate system. A local mine grid was established for the site. All drill hole and undergound development survey data was collected using this local grid.
The MRE estimate uses the local mine grid, which relates to MGA94 using the following transform:

<table>
<tr><th></th><th></th><th>MGA94</th><th>Local Mine Grid</th></tr>
<tr><td rowspan="2">Point 1</td><td>Northing</td><td>6551419.471</td><td>6451.175</td></tr>
<tr><td>Easting</td><td>372517.808</td><td>5231.564</td></tr>
<tr><td rowspan="2">Point 2</td><td>Northing</td><td>6551409.739</td><td>6452.863</td></tr>
<tr><td>Easting</td><td>371884.310</td><td>4597.827</td></tr>
<tr><td colspan="2">Elevation Correction</td><td colspan="2">+10,000</td></tr>
</table>
Underground Resource

Drill holes were surveyed using total station methods or RTK GPS on surface..
Holes paths were surveyed using a downhole gyro or an Eastman single shot down-hole camera at least every 30 metres downhole.
The level of accuracy for drill hole locations is considered appropriate for Resource estimation purposes.
</td></tr>
</table>

Criteria	JORC Code explanation	Commentary
		• A reasonably detailed surface topographic survey was supplied. This Resource estimate is not impacted by surface topography as the uppermost extents of the mineralised domains occur approximately 100m below the surface. **Tailings Resource** • Drill hole collars were surveyed by the mine surveyor by unknown methods. • There were no downhole surveys undertaken on the drill holes. All holes were drilled vertically and were relatively short (<15m depth), and therefore any downhole deviation would have negligible effects on the location of datapoints. • An aerial photogrammetry survey was carried out over the site in December 2015 by Arvista Pty Ltd at a ground resolution of 5cm per pixel. A Digital Terrain Model (DTM) in Surpac format was supplied and used in this study.
Data spacing and distribution	• *Data spacing for reporting of Exploration Results.* • *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.* • *Whether sample compositing has been applied.*	**Underground Resource** • Drill hole intercept spacing averages around 10m to 15m along strike and in the dip direction. Underground drill fans have resulted in closely spaced intercepts. Down hole sampling intervals were predominantly (80%) 1 to 2m in length. • The data spacing and distribution is sufficient to establish grade continuity appropriate for the Mineral Resource estimation procedures and classifications applied. • Sample composites of 2m were predominantly used in the MRE. 1m composites were used in one domain where the majority of sampling was over intervals of 1m or less. **Tailings Resource** • Drilling density is on a notional 50m x 50m grid with those holes used in the resource estimate on 100m x 200m grid. Down hole sampling intervals were on average around 1m in length. • The data spacing and distribution is sufficient to establish grade continuity appropriate for the Indicated Resource estimation category after all other confidence factors are applied. • Sample composites of 2m were used in the MRE.

Criteria	JORC Code explanation	Commentary
Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.* • *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	**Underground Resource** • The mineralization occurs as sub-vertical pipe-like structures with concentric grade zoning. Drill holes have been collared from the surface and multiple underground drill platforms resulting in a wide range of intercept angles from opposite sides. The majority of intercepts are at a high angle (orthogonal) to principal direction of mineralisation. This reduces the likelihood of biased sampling. **Tailings Resource** • Tailings were deposited sub-aerially forming beaches with a slight slope towards the perimeter of the storage facility. Therefore, any grade variations over time will be represented by sub-horizontal layering. Drilling of vertical drill holes ensures sampling is undertaken as close as possible orthogonal to the direction of maximum grade continuity.
Sample security	• *The measures taken to ensure sample security.*	• All samples were collected and sub-sampled on site by company staff. Samples were either submitted to an internal on site laboratory or off site laboratory. • Samples were collected and placed in numbered and ticketed calico bags that were securely fastened. Sample intervals were marked on the preserved core. Samples batches were kept to approximately 30 submitted samples at any one time to avoid overloading the lab. • A dedicated geologist and field assistant were in attendance at all stages of drilling of the tailings.
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	**Underground Resource** • Previous reporting on internal laboratory accuracy and precision has not raised any significant issues. • In the twenty years of the mine's history mining reconciliation and metallurgical balances have not identified any serious systematic problems with the prediction of ore grade. This reflects the fact that the Elura ore has low internal grade variability. The massive ore has an average grade of composite assays of around 10% zinc with a standard deviation of around 2. At the current very close drill spacing there is very little risk that assay error will significantly over value the Resource and historically no bias has been detected

Criteria	JORC Code explanation	Commentary
		Tailings Resource • There are no records of any audits or reviews of the sampling techniques or data.

Section 2 Reporting of Exploration Results

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.* • *The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	• The project is located within granted Exploration Licence EL5785 Mining leases ML158, ML159, ML160, ML316, ML161, and ML930 with the earliest expiry date of 12 March 2028. The leases are held by Cobar Operations Pty Ltd. • Metalla Royalty and Streaming Ltd have a royalty based a flat rate of 4% on payable Pb, Zn and Ag.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	**Underground Resource** • Exploration of the Elura deposit has been carried out by various companies since the early 1970's using surface and underground mapping and sampling, geophysical investigations, diamond and reverse circulation drilling. Previous exploration appears to have been performed to industry standards. **Tailings Resource** • The tailings in Sector 1 were drilled in 2014, 2015 and 2017 by CBH Resources. The drilling was undertaken by standard methods and the results used to generate an approximate tonnage and grade • Exploration appears to have been performed to industry standards.
Geology	• *Deposit type, geological setting and style of mineralisation.*	**Underground Resource** • Mineralisation at the Elura deposit is hosted by fine grained turbidite sequence of the Cobar Basin and comprises multiple sub-vertical elliptical shaped pipe-like pods that occur within the axial plane of an anticline and are surrounded by an envelope of sulphide stringer mineralisation, in turn surrounded by an envelope of siderite alteration extending for tens of metres away from the sulphide mineralisation. • Around 150m below the base of the main mineralised pods/lodes, mineralisation is hosted within the western limb of a folded limestone unit, occurring in veins and fractures. • Recent reviews favour a syngenetic formation model of an original stratiform deposit that was later emplaced by tectonic force into a favourable structural site during

Criteria	JORC Code explanation	Commentary
		deformation. • The zonation of mineralisation types has been categorised with abbreviations as follows: ○ PO – massive pyrrhotite-pyrite-galena-sphalerite ore, with pyrrhotite predominant, forming the central core of all zones, typically averaging about 9% Zn and 6% Pb. ○ PY – massive pyrite-pyrrhotite-galena-sphalerite ore, with pyrite predominant, commonly surrounding the pyrrhotitic core or at the outer margin of massive mineralisation, again typically averaging about 9% Zn and 6% Pb. ○ SIPO – siliceous pyrrhotite-pyrite-galena-sphalerite ore, with inclusions of silicified country rock and some quartz veining; pyrrhotite is the predominant sulphide; occurs at the margin of PO and PT mineralisation; typical ore grade averages around 12% combined Pb+Zn. ○ SIPY – siliceous pyrite-pyrrhotite-galena-sphalerite ore, with inclusions of silicified country rock and some quartz veining; similar to SIPO but pyrite is the predominant sulphide. ○ VEIN – lower grade mineralisation comprising a stockwork of quartz and sulphide veins within silicified siltstone, around the edges of mineralised pods. ○ MINA – mineralised altered siltstone. ○ SG – Supergene enriched zone at the top of the Main Lode. **Tailings Resource** • Mineralised material in the tailings storage facility consists of clay to fine sand sized particles deposited in sub-horizontal layers from centrally located outflow sites. The particles contain remnant sulphides that were not captured during processing of the Endeavor Mine silver-zinc-lead ore. • The primary lead and zinc bearing minerals from all orebodies processed are galena (~13%wt) and sphalerite (~14%wt). Pyrite and pyrrhotite (~60 to 70%wt in total) are the main floatable gangue in the ore. Tetrahedrite is the major host of silver, apart from galena and chalcopyrite.

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr>
<td>Drill hole Information</td>
<td>

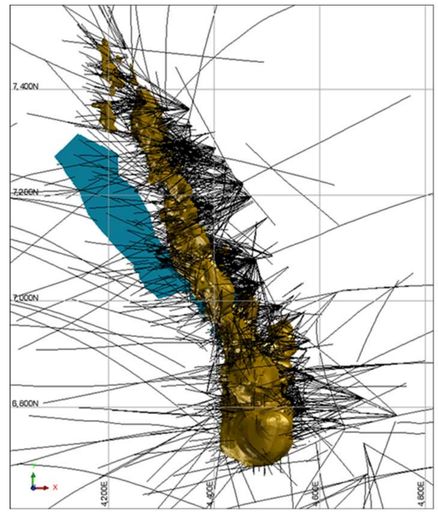
A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:

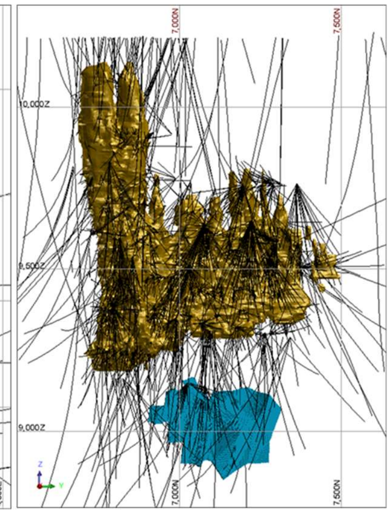
easting and northing of the drill hole collar
elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar
dip and azimuth of the hole
down hole length and interception depth
hole length.

If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.

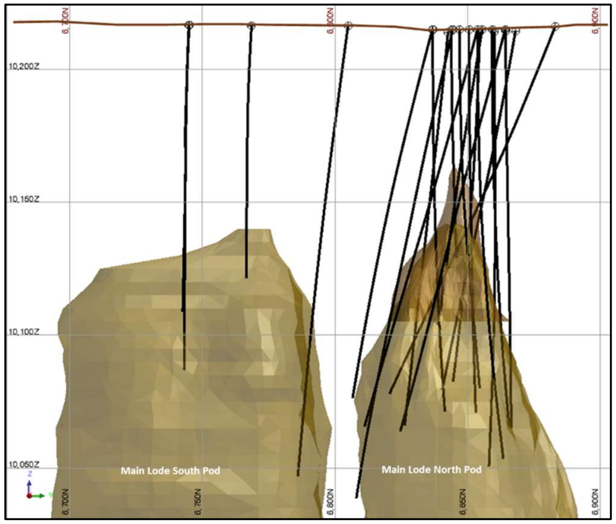
</td>
<td>
Underground Resource

There are 2,538 diamond drill holes and 21 RC holes in the database, totaling over 400,000m of drilling. Plan and long section views of the drill hole traces are shown below.





</td>
</tr>
</table>

Criteria	JORC Code explanation	Commentary
		 • • **Tailings Resource** • A table of drill hole data is included as an attachment to this report.
Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.* • *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* • *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	**Underground Resource** • A net smelter return (NSR) value was applied to the MRE for reporting purposes. A detailed description of the NSR calculation is provided in the report and in Section 3 of this table. **Tailings Resource** • No data aggregation methods have been used.
Relationship between	• *These relationships are particularly important in the reporting of Exploration Results.*	**Underground Resource** • The geometry of the mineralisation (vertical pods and tabular, steeply dipping

Criteria	JORC Code explanation	Commentary
mineralisation widths and intercept lengths	• *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* • *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').*	limestone-hosted) has been well defined from diamond drilling and underground development. Drill hole intercepts are predominantly at a high angle (orthogonal) to main mineralisation directions. **Tailings Resource** • Holes were drilled vertical, intersecting the direction of main grade continuity at approximate right angles.
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	• Exploration results are not the subject of this report..
Balanced reporting	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	• Exploration results are not the subject of this report.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	• The project is a mature stage development with the bulk of drilling undertaken for grade control purposes. • Bulk density measurements and metallurgical test results are discussed in Section 3 of this table. • The CP considers there is no other meaningful and material exploration data in relation to this report.
Further work	• *The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).* • *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	**Underground Resource** • Further exploration work planned includes drilling remaining upper Main Lode southern pod, drilling for potential economic gold and copper mineralisation, and investigation of potential nearby (<5km) mineralisation using drilling and geophysical methods. **Tailings Resource** • No further work planned

Section 3 Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary

Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.*
	• *Data validation procedures used.*
		• The following database validation activities have been carried out:
		  • Ensure compatibility of total hole depth data in the collar and assay drill hole database files.
		  • Check for overlapping sample intervals.
		  • Checking of drill hole locations against the surface topography.
		  • Visual validation in Surpac software.
		• A selection of laboratory assay certificates were checked against database entries.
		• No issues were found with the database.
		
		Underground Resource
		• The data used in this Mineral Resource estimate was provided in a Microsoft Access database and was originally managed using a Drilling Management System (DMS) that utilised. Microsoft Access to enter and store data. The system was set up with data security protocols that restricted access and ability to edit based on security levels.

Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	• The Competent Person has visited the Endeavor Mine on two occasions. • The first visit was in 2010 to undertake a review of the Mineral Resources. During this visit inspections were carried out on mineralised intercepts in drill core and underground exposures. Observations were made of drilling, logging, sampling, QAQC, data handling procedures. • The second visit was in February 2023 whilst the mine was in care and maintenance to collect data and observe drilling, logging, sampling and QAQC procedures for the drilling program that was underway targeting supergene mineralisation. • The Competent Person regards the procedures and protocols observed during the site visits to be of a good standard.
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.*	**Underground Resource** • Confidence in the geological interpretation is high as the deposit has been the subject of nearly 50 years of investigations and mining.

Criteria	JORC Code explanation	Commentary

	• *The effect, if any, of alternative interpretations on Mineral Resource estimation.*
	• *The use of geology in guiding and controlling Mineral Resource estimation.*
	• *The factors affecting continuity both of grade and geology.*
		• Data from sampling of diamond drill holes and underground exposures has been used in the interpretation and modelling of geological and grade domains.
		• There are currently no alternative geological interpretations as the current interpretation is the result of many years of geological investigations. Any changes to the interpretation would not significantly change the MRE due to the density of data.
		• The Elura deposit comprises multiple zones of mineralisation styles based on mineralogy, grade, veining etc. that typically transition from a massive sulphide core to an altered siltstone and veined outer halo. These zones were, from high to low grade:
		  ◦ Supergene Enrichment (SG)
		  ◦ Pyrrhotitic (PO)
		  ◦ Pyritic (PY)
		  ◦ Siliceous Pyritic (SIPY)
		  ◦ Siliceous Pyrrhotitic (SIPO)
		  ◦ Vein (VEIN)
		  ◦ Mineralised Altered Siltstone (MINA
		• Another style of mineralisation is located about 150m beneath the siltstone-hosted mineralisation which is hosted in limestone.
		• Domain boundaries of the siltstone-hosted mineralisation were interpreted on 5m elevation intervals for the entire deposit using drill-hole data, geological interpretation and back mapping from all the underground levels. The grade domains were further divided into lode domains for estimation
		• The contact of the limestone and the surrounding sediments was modelled on ~10 m sections using all the available drillholes. This wireframe was not used for the grade estimation however was used to help define the mineralised domains within the Limestone domain
		• The mineralised domain for the limestone-hosted mineralisation was interpreted using a combination of cross-sections and level plans.
		
		Tailings Resource
		
		• There is no geological interpretation of the tailings deposits, and it is assumed the tailings were deposited in sub-horizontal layers.
		• The volume of tailings is constrained by surveys of the topography prior and subsequent to the deposition of the tailings.

Criteria	JORC Code explanation	Commentary
		• The style of deposit (tailings) does not allow for alternative interpretations. • The mineralisation within the TSF is considered highly continuous with low variability.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	**Underground Resource** • The sub vertical high grade pods occur in the axial plane of an anticline and progressively decrease in size towards the north west. The Main Lode occurs at the southern end of mineralisation, extending from near-surface to approximately 1,000m depth, with lateral extents of between 50m and 120m. The Northern Lodes extend north west from the Main Lode, generally occur only below a depth of 400 – 500m and have lateral extents typically between 30 – 50m. • The top of the limestone-hosted mineralisation occurs approximately 1,050m below the surface. The mineralised zone is broadly tabular in form and currently measures 300m long by 250m high with widths ranging between 10m and 30m, dipping around 70° towards the south west. **Tailings Resource** • The Resource estimate entails the bulk of Sector 1 of the CTD TSF, which measures approximately 550m by 850m and an average depth of 7m
Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.* • *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* • *The assumptions made regarding recovery of by-products.* • *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).*	**Underground Resource** • Vulcan and Surpac software was used for data validation, analysis, geological and mineralized domain modelling, sample compositing, and grade interpolation. • Grade domains for constraining Resource estimation were interpreted and modelled based on geological logging and assay results. Six grade domains and five lode domains were modelled. • The resource model is based on statistical and geostatistical investigations generated using 1m (Main Lode Deeps) and 2m (all other domains) composited sample intervals. Assessment of the data suggested requirement for high grade cutting for the input datasets to be used for resource estimation of Ag in some domains. The estimate search distance for Au in the supergene zone was controlled by grade restriction. Otherwise the composite data sets for other metals displayed low coefficients of variation. • The modelled variography for Pb, Zn and Ag in all domains display low relative nugget values. The variograms have short range structures that account for between 30% (Zn-MLDeeps) and 80% (Ag-DZL) of the total variance including

Criteria	JORC Code explanation	Commentary

	• *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.*
	• *Any assumptions behind modelling of selective mining units.*
	• *Any assumptions about correlation between variables.*
	• *Description of how the geological interpretation was used to control the resource estimates.*
	• *Discussion of basis for using or not using grade cutting or capping.*
	• *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*
		nugget effect, with ranges of between 10m (Zn-MLDeeps) and 55m (Ag-ML). Overall ranges range from 15m (Pb, Zn-WM) to 500m (Ag-ML).
		• Rotated, sub-celled block models were constructed using parent block dimensions of 5m East by 5m North by 10mRL in the upper siltstone-hosted model and 5m East by 10m North by 5mRL in the limestone-hosted model, with sub-blocking for the purpose of providing appropriate definition of the grade domain boundaries. Data spacing ranged from 10-15m in densely drilled areas to 80m in parts of the deep zinc lode...
		• Resource estimation was carried out for lead, zinc, silver and gold (upper main lode only) on the basis of analytical results available up to May 2023. Ordinary Kriging (OK) was selected as an appropriate estimation method based on the quantity and spacing of available data and style of deposit under review. A three-pass strategy was employed to generate the grade estimates. Restrictions of the maximum number of samples per drillhole were applied to the first and second search passes. The search axes were aligned with the average orientation of the mineralised domains while search distances were derived from variographic analyses of the data sets. Search axes utilised a Locally Varying Anisotropy in the deep zinc lode due to it's narrow, tabular nature.
		• Combinations of modelled grade and lode domains were used to constrain sample selection and grade interpolation using both soft and hard boundaries.
		• The maximum extrapolation distance from known data points was around 80m.
		• Comparison of the estimated grades and mill production for the calendar year 2019 revealed a reconciliation of 102% of expected Pb+Zn% grade.
		• No assumptions of byproduct recovery have been made.
		• Iron content was estimated using the same process as the other metals.
		• No assumptions have been made regarding underground mining selective units.
		• No assumptions about correlation between variables has been made.
		• Validation of the estimate was completed and included both interactive and statistical review. The validation methods included: -
		  ◦ Visual comparison of the input data against the block model grade in plan and cross section.
		  ◦ Comparison of global statistics.
		  ◦ Swath plots, comparing the composite grade and the estimated grade grouped by intervals in plan and section
		• The model was found to be robust.

Criteria	JORC Code explanation	Commentary

		Tailings Resource
		• The resource model is based on statistical and geostatistical investigations generated using 2m composited sample intervals of the holes drilled in 2015. Assessment of the data suggested no requirement for high grade cutting. The composite data sets displayed low coefficients of variation.
		• A sub-celled block model was constructed using parent block dimensions of 50m East by 50m North by 2mRL. Block sizes were based on average drill hole spacing of 100m.
		• Resource estimation was carried out by Ordinary Kriging (OK) method using multi-pass-pass strategy, with the first pass set at a distance less than the total range of the variogram. The number of composites for a successful estimate was restricted to a minimum of 3 and a maximum of 12 for the first pass and a minimum of 2 and a maximum of 10 for the second pass. The search axes were aligned with directions of maximum continuity derived from variographic analyses of the data set. Surpac mining software was used carry out the estimation.
		• The estimated tonnes and grade have been compared to historical tailings deposition records and are within 4% of the tonnes and 0.5% of the Zn grade. The grades also compare well with global metallurgical composite head grades.
		• The tailings are contained within a licensed facility and will be re-processed and deposited into another facility that is licensed to handle potential acid forming material.
		• The maximum extrapolation distance from known data points was around 150m.
		• No assumptions of byproduct recovery have been made.
		• No assumptions about correlation between variables has been made.
		• The search radii were aligned to reflect the sub-horizontal nature of tailings deposition with blocks and composite selection confined to within the Sector 1 boundary and modelled top and base of tailings.
		• Validation of the estimate was completed and included both interactive and statistical review. The validation methods included: -
		    • Visual comparison of the input data against the block model grade in plan and cross section.
		    • Comparison of global statistics.
		    • Swath plots, comparing the composite grade and the estimated grade grouped by intervals in plan and section

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>The model was found to be robust.</td></tr>
<tr><td>Moisture</td><td>Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.</td><td>The tonnages were estimated on a dry basis.</td></tr>
<tr><td>Cut-off parameters</td><td>The basis of the adopted cut-off grade(s) or quality parameters applied.</td><td>
Underground Resource

The MRE has been reported using a net smelter return (NSR) value cut-off determined from mining, processing, and overhead costs per tonne of material milled.
The NSR is defined as the return from sales of concentrates, expressed in dollars per tonne of ore, excluding mining and processing costs.
An NSR value was calculated for each block in the model using the following parameters:

<table>
<tr><th rowspan="2">Metal</th><th rowspan="2">Metal Price</th><th rowspan="2">Exchange Rate</th><th colspan="2">Flotation Recovery</th><th rowspan="2">Smelting Recovery</th><th rowspan="2">Smelting and Freight costs per tonne</th><th colspan="2">Tonnes ore / Tonnes concentrate</th></tr>
<tr><th>Below 10080m RL</th><th>Above 10080m RL</th><th>Below 10080mR L</th><th>Above 10080mR L</th></tr>
<tr><td>Pb</td><td>US$2,050/t</td><td rowspan="3">AU$1= US$0.69</td><td>74%</td><td>62%</td><td>95%</td><td rowspan="3">$523</td><td rowspan="3">5.15</td><td rowspan="3">5.36</td></tr>
<tr><td>Zn</td><td>US$3,000/t</td><td>83%</td><td>75%</td><td>85%</td></tr>
<tr><td>Ag</td><td>US$22.50/oz</td><td>51%</td><td>66%</td><td>95%</td></tr>
</table>

An NSR value of $150/t was chosen as the cut-off value for reporting material below 10080mRL and represents a 25% increase to mining, processing and general overhead costs since the cessation of mining in 2019. An NSR value of $190/t was chosen as the cut-off value for reporting material above 10080mRL (Level 1 Sulphides) and is based on higher processing costs to achieve acceptable recoveries and higher mining costs to account for increased ground support required for softer material.

Tailings Resource

Little to no selectivity is assumed from the prosed mining method (hydromining) therefore no cut off grade has been applied to the estimate for reporting purposes.

</td></tr>
</table>

Criteria	JORC Code explanation	Commentary
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	**Underground Resource** • It is understood similar scale mechanised mining to what was used previously would be carried out once operations recommenced on site. • The Elura deposit is extensively developed by underground openings and the base of the main decline has reached a depth equal to the top of the deep zinc lode. • No mining dilution has been applied to the MRE. • The Mineral Resource Statement also includes 5m skins surrounding existing stoped areas. The mine has a history of using paste fill to backfill stope voids, allowing the recovery of pillars and other remnant material. Some of this material may be excluded from Ore Reserve estimations if assessed as being non-recoverable. Information is not available at this stage of Mineral Resource estimation to determine the extent of recovery of remnant material. However, there is a reasonable prospect for eventual extraction of remnant material. **Tailings Resource** • The tailings is proposed to be mined by hydromining methods, where water cannons liquify and push the tailings into a collection drain which runs to a sump where a pump delivers the slurry to the processing plant.
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	**Underground Resource** • The ore from the Endeavor Mine is processed through a conventional Pb/Zn/Ag flotation plant with a demonstrated capacity of 1.2 Mtpa. • The mill has demonstrated recoveries of 74% for Pb, 83% for Zn and 51% for Ag which have been factored in to the calculation of NSR values. • Adjusted flotation recoveries have been applied to reporting material in the marcasite-rich Level 1 Sulphides (>10080mRL). **Tailings Resource** • Metallurgical test work has indicated saleable Zn and Pb/Ag concentrates can be obtained from processing the tailings through the existing flotation process on site.
Environmental factors or assumptions	• *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider*	• There is a fully permitted Tailings Storage Facility on site with adequate storage capacity as well as approved plans for capacity increase through a perimeter wall raise..

Criteria	JORC Code explanation	Commentary
	the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	
Bulk density	• *Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.* • *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.* • *Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	**Underground Resource** • Historically, Bulk Density had been assigned to the block model on a domain by domain basis. Work completed by H&S Consulting in 2015 recommended that a calculated density value be used. Since calculated bulk densities have been used, stopes tonnes have generally reconciled well, which has been attributed to the change to the use of calculated densities. • The formula used to derive the calculated densities involves a number of steps: 1. gn = Pb x 100/86.6 where Pb > 0.0 2. sp = Zn x 100/67.1 where Zn > 0.0 3. po_pct = Fe x 2 4. fe_gangue = (30-Fe)/60, with a minimum of 5% (0.05) 5. py = fe x 100/46.5 x (100 – po_pct) x (1- fe_gangue)/100 6. po = fe x 100/60.4 x po_pct x (1- fe_gangue)/100 7. total_sulph_1 = gn + sp + py + po 8. if total_sulph_1 > 95%, total_sulp_2 = 95%, otherwise total_sulph_2 = total_sulp_1 9. py_final = py x (total_sulp_2 – gn – sp)/(total_sulp_1 – gn –sp) 10. po_final = po x (total_sulp_2 – gn – sp)/(total_sulp_1 – gn –sp) 11. gangue_pct = (100 - total_sulp_2) 12. density_calc = (gn x 7.5 + sp x 4.0 + po x 4.6 + py x 5.02 + gangue_pct x 2.5)/100 **Tailings Resource** • During the 2014 drilling program, 551 samples for density analysis were taken from

Criteria	JORC Code explanation	Commentary
		each 1m interval by firmly compressing the material into a grout sampling and levelling the top off. Each sample was stored in zip-lock plastic bags and taken to the site laboratory for wet weight and dry weight measurements. The average dry density value was 1.74 t/m3.

Classification	• *The basis for the classification of the Mineral Resources into varying confidence categories.*
	• *Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).*
	• *Whether the result appropriately reflects the Competent Person's view of the deposit.*
		Underground Resource
		• The Resource has been classified as Measured, Indicated and Inferred with the key parameters considered during the resource classification being:
		  • Geological knowledge and interpretation.
		  • Deposit style.
		  • Confidence in the sampling and assay data.
		  • The spacing of the exploration drill holes.
		  • Variogram model ranges in relation to the local data spacing and the estimation variance.
		  • Prospects for eventual economic extraction.
		• The exploration data used for the MRE is robust and appropriate for resource estimation purposes, with the current data spacing sufficient to generate robust mineralisation interpretations. The geology of the project area has been studied in detail over numerous years, providing confidence in the interpretation of mineralisation style. Historical mining records give further confidence in the existence of economic mineralisation.
		• Prospects for eventual economic extraction are high as the deposit is highly developed, metals are beneficiated using standard methods and there is an existing processing plant on site.
		• Based on the consideration of items listed above, and review of the resource block model estimate quality, classification criteria were determined as summarised in the following: -
		  • **Measured**
		    ○ Blocks that were estimated in the first pass (except for SG and VEIN domains and DZL).
		  • **Indicated**
		    ○ Blocks that were estimated in the second pass (or first and second pass in the SG domain and first pass in the VEIN domain).
		    ○ Blocks in DZL domain estimated in first or second pass and a slope of regression greater than 0.3.

Criteria	JORC Code explanation	Commentary

		• **Inferred**
		  ○ Blocks that were estimated in the third pass (or second pass in the VEIN domain).
		  ○ Blocks in DZL domain estimated in first or second pass and a slope of regression less than 0.3, or estimated in the third pass.
		• The classification reflects the Competent Person's view of the deposit.
		
		• **Tailings Resource**
		• The Resource has been classified as Indicated and Inferred with the key parameters considered during the resource classification being:
		  ○ Geological knowledge and interpretation.
		  ○ Deposit style.
		  ○ Confidence in the sampling and assay data.
		  ○ The spacing of the exploration drill holes.
		  ○ Variogram model ranges in relation to the local data spacing and the estimation variance.
		  ○ Prospects for eventual economic extraction.
		• The exploration data used for the TSF Sector 1 Resource estimate is robust and appropriate for resource estimation purposes, with the current data spacing sufficient to generate robust grade estimates. Confidence in the estimate is increased by good comparisons to historical tailings deposition records and head grades from global metallurgical composite samples.
		• There are reasonable prospects for the eventual economic extraction of the resources because of proximity to an existing floatation processing plant and metallurgical test work indicates economic recoveries for Zn, Pb and Ag.
		• Based on the consideration of items listed above, and review of the resource block model estimate quality, classification criteria were determined as summarised in the following: -
		  • **Indicated**
		    ○ Blocks in the tailings domain that occur between drill holes or no more than 50m from a drill hole.
		  • **Inferred**
		    ○ Blocks that were estimated in the third pass (or second pass in the VEIN domain).
		    ○ All remaining blocks in tailings domain no assigned Indicated.

Criteria	JORC Code explanation	Commentary
		• The classification reflects the Competent Person's view of the deposit.
Audits or reviews	• *The results of any audits or reviews of Mineral Resource estimates.*	**Underground Resource** • .Numerous audits of data collection, geological interpretation and domaining, data quality assurance, and MRE methodology have been undertaken in the past by internal company personnel and external consultants. No major issues were identified. **Tailings Resource** • There have been no audits or reviews of the estimate.
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	• There has been no attempt to apply geostatistical methods to quantify the relative accuracy of the Mineral Resources to within a set of confidence limits. • The Competent Person believes the Mineral Resource estimates provide a good estimate of global tonnes and grade. • Higher local variances in tonnes and grade can be expected in areas classified as Inferred due to lower data density. • No change of support adjustment has been made to the block estimates. • The accuracy and confidence of this Mineral Resource estimates are considered suitable for public reporting by the Competent Person. • Previous Mineral Resource estimates of underground material have reconciled well with mill production. .

Section 4 Estimation and Reporting of Ore Reserves

(Criteria listed in section 1, and where relevant in sections 2 and 3, also apply to this section.)

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr>
<td>Mineral Resource estimate for conversion to Ore Reserves</td>
<td>

Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve.
Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.

</td>
<td>

The Mineral Resource estimates used as a basis for the conversion to an Ore Reserve are the Main Ore Body and Deep Zinc Lode estimates, last reported by Polymetals on 23 May 2023, and the TSF Sector 1 Tailings estimate, first reported by Polymetals in this announcement.

<table>
<tr><th colspan="5">Endeavor Mine In Situ Mineral Resource May 2023</th></tr>
<tr><th>Category</th><th>Mt</th><th>Zinc (%)</th><th>Lead (%)</th><th>Silver (g/t)</th></tr>
<tr><td>Measured</td><td>4.4</td><td>8.3</td><td>5.1</td><td>93</td></tr>
<tr><td>Indicated</td><td>8.8</td><td>7.9</td><td>4.6</td><td>82</td></tr>
<tr><td>Inferred</td><td>3.1</td><td>7.7</td><td>3.7</td><td>78</td></tr>
<tr><td>Total[2]</td><td>16.3</td><td>8.0</td><td>4.5</td><td>84</td></tr>
</table>
1. Reported using NSR cut-off values of $190/t for mineralisation above 10,080mRL, and $150/t for mineralisation below 10,080mRL
2. Discrepancies may occur due to rounding
<table>
<tr><th colspan="5">Endeavor Mine TSF Sector 1 Mineral Resource October 2023</th></tr>
<tr><th>Category</th><th>Mt</th><th>Zinc (%)</th><th>Lead (%)</th><th>Silver (g/t)</th></tr>
<tr><td>Indicated</td><td>3.6</td><td>2.14</td><td>1.56</td><td>80</td></tr>
<tr><td>Inferred</td><td>1.6</td><td>2.07</td><td>1.53</td><td>77</td></tr>
<tr><td>Total[2]</td><td>5.2</td><td>2.12</td><td>1.55</td><td>79</td></tr>
</table>
1. Reported without use of cut off grade
2. Discrepancies may occur due to rounding

All estimates were based on tonnes and grade reported from block models with block grades interpolated using Ordinary Kriging methods.
Mineral Resources are reported inclusive of Ore Reserves.

</td>
</tr>
<tr>
<td>Site visits</td>
<td>

Comment on any site visits undertaken by the Competent Person and the outcome of those visits.

</td>
<td>

Mr Matthew Gill, the competent Person for the Ore Reserves in this announcement, is a non-executive director of Polymetals Resources Ltd. Mr Gill

</td>
</tr>
</table>

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td></td><td>If no site visits have been undertaken indicate why this is the case.</td><td>has visited the site on two occasions during the preparation of the Endeavor Mine Restart Study and compilation of the Ore Reserves. Mr Gill inspected surface and underground infrastructure which were found to be in good order and suitable for use in the recommencement of operations.</td></tr>
<tr><td>Study status</td><td>The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves.
The Code requires that a study to at least Pre-Feasibility Study level has been undertaken to convert Mineral Resources to Ore Reserves. Such studies will have been carried out and will have determined a mine plan that is technically achievable and economically viable, and that material Modifying Factors have been considered.</td><td>The Ore Reserves reported in this announcement are supported by a study undertaken to a Pre-Feasibility level of accuracy. The study included economic analyses based on a mine schedule incorporating suitable mine designs, modifying factors and up to date costings.</td></tr>
<tr><td>Cut-off parameters</td><td>The basis of the cut-off grade(s) or quality parameters applied.</td><td>The mine schedule and Ore Reserve estimate for in situ material use a Net Smelter Return calculation as a cut-off for reporting purposes. NSR values were assigned to each block in the resource block model based on calculations using the assumptions shown below:
<table>
<tr><th rowspan="2">Metal</th><th rowspan="2">Metal Price</th><th rowspan="2">Exchange Rate</th><th colspan="3">Flotation Recovery</th><th rowspan="2">Smelting Recovery</th><th rowspan="2">Smelting and Freight costs per tonne</th></tr>
<tr><th>Below 10080mRL</th><th>Above 10080mRL</th><th>DZL</th></tr>
<tr><td>Pb</td><td>US$2,076/t</td><td rowspan="3">AU$1= US$0.70</td><td>75%</td><td>77%</td><td>-</td><td>95%</td><td rowspan="3">$523</td></tr>
<tr><td>Zn</td><td>US$2,915/t</td><td>84%</td><td>76%</td><td>90%</td><td>85%</td></tr>
<tr><td>Ag</td><td>US$22.4/oz</td><td>52%</td><td>57%</td><td>52%</td><td>95%</td></tr>
</table>
The formula for calculating NSR value of each tonne of material is:

$$\mathbf{NSR(x_1, x_2, x_3) = x_1r_1p_1(V_1) + x_2r_2p_2(V_2) + x_3r_3p_3(V_3) - (C_s + C_t)/K}$$

Where:
x_1, Grade of metal 1, etc
r_1, Floatation Recovery of metal 1, etc
p_1, Smelting Recovery of metal 1, etc
V_1, Value of metal 1, etc
Cs Smelting and freight costs per tonne of concentrate
K Tonnes of ore required to make one tonne of concentrate</td></tr>
</table>

Criteria	JORC Code explanation	Commentary
		• An NSR value of $150/t was used for in situ material, based on a combination of historic mining and processing costs on site, as well as updated mining and processing costs calculated during the study process. • Ore Reserves for the TSF Sector 1 Tailings are reported with no cut-off due to lack of selectivity in the proposed mining method.

Mining factors or assumptions	• *The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore Reserve (i.e. either by application of appropriate factors by optimisation or by preliminary or detailed design).*
	• *The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc.*
	• *The assumptions made regarding geotechnical parameters (eg pit slopes, stope sizes, etc), grade control and pre-production drilling.*
	• *The major assumptions made and Mineral Resource model used for pit and stope optimisation (if appropriate).*
	• *The mining dilution factors used.*
	• *The mining recovery factors used.*
	• *Any minimum mining widths used.*
	• *The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion.*
	• *The infrastructure requirements of the selected mining methods.*
		• Underground stope optimisation was carried out using Deswik Stope Optimiser (SO). Preliminary detailed stope designs were generated from the optimised stope shapes along with designs for development to access stoping areas. Underground mine production schedules were generated using Deswik.Sched underground scheduling and mine planning software. A number of scenarios were run to find the optimal production sequencing and mining rate for maximum project NPV.
		Tailings retreatment mine designs were based on a hydromining method with allowances for berm batters, a central containment pillar, catchment gullies and mining sequence.
		• The underground mining methods are:
		  ○ Long hole open stoping with minor amounts of unconsolidated rock fill for the Main Ore Body
		  ○ Sub Level Open Stoping, with a combination of loose and cemented rock fill, mined from the bottom up in the Deep Zinc Lode.
		  ○ Cut & Fill method with pillars between drives, is to be used above 10090mRL (Upper Main Lode) due to potentially poor ground conditions.
		• The mining method for the Sector 1 tailings is hydromining, a monitor based hydraulic mining method. This type of mining was chosen after comparison to a dredging method.
		• All stope designs have been guided by geotechnical advice and considerations with parameters defined by the rock strength characteristics within the immediate area of the planned void. Grade control drilling to increase the confidence in stope grades will commence immediately on recommencement of operations.
		• Stope optimisation was carried out using a minimum strike of 5m and attempting to align the height of the stopes with the existing level intervals. Post processing was completed to eliminate shapes with a volume below 500 m^3 and any part of the stope shape within 5m of a previously mined stope. The Mineral Resource models used for the optimisation process were the Main Ore Body and Deep Zinc Lode block models.
		• Mining dilution and ore loss assumptions are based on historical development

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>
and stope reconciliations at the Endeavor Mine. Dilution has been assumed to have zero grade and provides a conservative estimate of production grades. Actual dilution grade will vary depending on location as shown below:
<table>
<tr><th>Stope Type</th><th>Recovery</th><th>Total Dilution</th></tr>
<tr><td>Primary Stopes</td><td>95%</td><td>5%</td></tr>
<tr><td>Secondary Stopes</td><td>90%</td><td>5%</td></tr>
<tr><td>Tertiary Stopes</td><td>90%</td><td>5%</td></tr>
<tr><td>Remnant Stopes</td><td>90%</td><td>5%</td></tr>
<tr><td>6/6 Stope Recovery</td><td>70%</td><td>5%</td></tr>
<tr><td>Development</td><td>98%</td><td>12%</td></tr>
</table>

The mine design and schedule include approximately 30% Inferred Mineral Resources. The majority of the Inferred material (94%) in the mine plan occurs in the Deep Zinc Lode and Sector 1 Tailings and is scheduled to be mined at the back end of the mine plan.
All major surface and underground infrastructure is already in place and has been kept in good order since the mine ceased operations at the end of 2019.

</td></tr>
<tr><td>Metallurgical factors or assumptions</td><td>

The metallurgical process proposed and the appropriateness of that process to the style of mineralisation.
Whether the metallurgical process is well-tested technology or novel in nature.
The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied.
Any assumptions or allowances made for deleterious elements.
The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole.
For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?

</td><td>

Past production (~32Mt) over the last 40 years from the Endeavor Mine has been processed through a conventional Pb/Zn/Ag flotation plant with a demonstrated capacity of 1.2 Mtpa. The proposed mine plan will utilise this process.
The metallurgical process is a common one for base metal sulphide mineralisation. It has been used successfully on the site for almost 40 years.
There has been a vast amount of metallurgical test work that has been carried out on the mineralisation at the Endeavor Mine over its long history. This test work, along with historic records of mill performance have enabled the assessment of recommended metal recoveries and concentrate grades for each metallurgical domain as shown below. Several metallurgical recoveries and concentrate grades have been estimated for the Deep Zinc Lode and Tailings, which are the subject of ongoing or planned flotation test work.

</td></tr>
</table>

Criteria	JORC Code explanation	Commentary

Ore Source	Metallurgical Recovery			Pb Concentrate Grade		Zn Concentrate Grade	
	Pb (%)	Zn (%)	Ag (%)	Pb (%)	Ag (g/t)	Zn (%)	Ag (g/t)
Historic Areas	77.4	86.8	71	50	625	50	94
Deep Zinc Lode	75*	90	70*	48*	1,800*	50	100*
Upper Main Lode	62	76	66	48	1,500	48	200
Tailings	30*	46	40*	50*	1,500*	50	-

*Estimated recoveries and grades

- Historically there have been no price penalties on concentrate produced at the Endeavor Mine due to deleterious elements.
- Metallurgical factors and assumptions benefit from a long history of actual mill performance records.

Criteria	JORC Code explanation	Commentary
Environmental	• *The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.*	• The Endeavor Mine has operated for nearly 40 years with minimal impact on the local environment. Numerous environmental studies have been undertaken over the years to support mining approvals and regulatory compliance. • Waste rock could be regarded as predominantly potentially acid forming (PAF) due the presence of sulphide minerals. All PAF waste rock will be re-used in the underground mine as loose or cemented rock fill of voids. • There is a fully permitted Tailings Storage Facility on site with adequate storage capacity as well as approved plans for capacity increase through a perimeter wall raise.
Infrastructure	• *The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can be provided or accessed.*	• The Endeavor Mine project is endowed with extensive infrastructure, in good condition, ready to be utilised once mining recommences. Surface infrastructure includes a 1.2Mtpa processing plant, rail loading facility for concentrate, raw water and electricity connections to local grids, workshops, partly stocked stores warehouse, offices, and a laboratory. The Project also owns 42 houses and 46 units in the Cobar township, 47km away. A 100-person camp will be built, fully catered, and managed by the operation. Underground infrastructure includes a tower mounted friction winder for hauling ore to the surface, a truck haulage decline from surface to 9135mRL (1,065m

Criteria	JORC Code explanation	Commentary
		below surface), underground crushing station, workshops, refuelling station, dewatering pump station, refuge chambers and reticulated water and air.
Costs	• *The derivation of, or assumptions made, regarding projected capital costs in the study.* • *The methodology used to estimate operating costs.* • *Allowances made for the content of deleterious elements.* • *The source of exchange rates used in the study.* • *Derivation of transportation charges.* • *The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc.* • *The allowances made for royalties payable, both Government and private.*	• The estimates of capital expenditure were compiled by Polymetals, where possible using rates and quotes received from contractors and suppliers and using recommendations for repairs and refurbishment made by independent inspections. • Operating costs have been estimated from first principles for a model that incorporates input costs for mining, processing, maintenance, administration / commercial, HSETS (Health, Safety, Environment, Training & Stores), and housing costs. The mining component was validated using a third-party mining cost estimate. • No allowances have been made for deleterious elements as this has not been an issue historically at the mine. • Exchange rates used in the study were derived from analysis of historic trends, consensus outlooks, spot rate and peer assumptions. • Transportation rates were derived from previous costs and provider quotes. • Benchmark treatment charges and refining charges (TC/RC's) have been used for the study. For Zinc, the Teck/KZ Red Dog Benchmark TC's are applied, and for Lead-Silver the Cannington/KZ Benchmark TC/RC's. Historically, concentrates from Endeavor have never exceeded contained metal above upper threshold of 54% with LOM historic grades being 50.13% Zn & 50.74% Pb respectively. • Allowances have been made in the study to account for State Royalties (4%) as well as the third-party royalty payable to Metalla Royalty and Streaming (4% Net Smelter Return).
Revenue factors	• *The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc.* • *The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.*	• Assumptions of head grade were made directly from the monthly mining schedule ouput. Assumptions of metal prices and exchange rates were made using historic trends, consensus outlooks, spot prices and peer assumptions to form a view. Assumptions of transportation, treatment and refining charges were made using benchmark costs. The study assumes flat metal prices and exchange rates across all years of the LOM schedule as shown below. **Metric** / **Unit** / **LOM** Zinc / US$/t / 2,750.00

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>
<table>
<tr><td>Lead</td><td>US$/t</td><td>2,200.00</td></tr>
<tr><td>Silver</td><td>US$/oz</td><td>23.00</td></tr>
<tr><td>Exchange Rates</td><td>AUD:USD</td><td>0.67</td></tr>
</table>
Overall payabilities were calculated individually for each of the Project ore sources based on the concentrate specifications, minimum deductions and payability thresholds provided by Ocean Partners. Average payabilities from concentrates produced over the LOM are:

84.04% Zinc.
94.09% Lead.
94.86% Silver.

Realisation costs used in the study were:
<table>
<tr><th></th><th>Zinc Concentrate</th><th>Silver-Lead Concentrate</th></tr>
<tr><td>Rail & Loading</td><td>A$72/wmt</td><td>A$72/wmt</td></tr>
<tr><td>Assay</td><td>A$1/wmt</td><td>A$3.03/wmt</td></tr>
<tr><td>Shipping</td><td>US$35/wmt</td><td>-</td></tr>
</table>
Treatment and refining charges used in the study remain commercial in confidence.
</td></tr>
<tr><td>Market assessment</td><td>

The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future.
A customer and competitor analysis along with the identification of likely market windows for the product.
Price and volume forecasts and the basis for these forecasts.
For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.

</td><td>

Polymetals engaged with Ocean Partners, a global base & precious metal trading firm, to assess the marketability of the concentrates which will be produced from the Endeavor Mine.
The Zinc price has moved downwards from its 3-year peak of US$4,498 in April 2022, and the 3-year mean of US$3,042/t. Zinc appears to have recovered from its 3-year low and has been moving upwards over the past 3 months. Consensus Economics long term price forecasts vary between US$2,491 and US$3,328.
The Lead price has moved upwards from its 3-year mean of US$2,135/t, and relatively rangebound between US$2,300 and US$2,200/t over the past 3 months. Consensus Economics long term price forecasts vary between US$1,874 and US$2,476/t.
The Silver price has remained close to its 3-year mean of US$23.54/oz, and

</td></tr>
</table>

Criteria	JORC Code explanation	Commentary
		relatively rangebound between US$23 and 24/oz over the past 3 months. Consensus Economics long term price forecasts vary between US$21.5 and US$27.4/oz.
Economic	• *The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc.* • *NPV ranges and sensitivity to variations in the significant assumptions and inputs.*	• A financial analysis of the Project was carried out by a cashflow model using outputs of the LOM scheduling process, CAPEX and OPEX estimates, and economic assumptions. The analysis is based on a mine life of 10 years, with mining of underground ore from Years 1 to 6 and re-treatment of Sector 1 tailings from Years 5 to 10. Mining is scheduled to commence 8 months after site establishment begins, with processing to commence 2 months after mining starts. The financial model estimates monthly pre-financing cashflows for the LOM in Australian dollars, with the evaluation reported on a pre-tax basis with no account for inflation. Net present Valus (NPV) is calculated using a pre and post-tax discount rate of 8%. • The sensitivity of the Project NPV_8 to variations in metal grades, metal prices, metal recoveries, foreign exchange rate, CAPEX and OPEX have been modelled with the NPV most sensitive to exchange rate giving a range of NPV's between A$97M and A$328M for a +/-15% variation in the rate.
Social	• *The status of agreements with key stakeholders and matters leading to social licence to operate.*	• The Endeavor Mine has had a long history in the Cobar region, having operated continuously for almost 40 years. In that time the mine has made a significant contribution to the local community in the form of employment opportunities, economic growth, and community investment. • Polymetals has presented the plan for resumption of operations at the Endeavor Mine to the local Cobar Shire Council which stated it's ongoing support for the Project.
Other	• *To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore Reserves:* • *Any identified material naturally occurring risks.* • *The status of material legal agreements and marketing arrangements.* • *The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory*	• Polymetals has not identified any naturally occurring risks to the Project. • Polymetals, through its 100% owned subsidiary company Cobar Metals Pty Ltd, has entered into a legally binding arrangement to acquire 100% of the Endeavor mine and associated assets by acquiring the operating entities from CBH. In order to complete the acquisition, Cobar Metals will be required to secure the release and replacement of the Environmental Rehabilitation Bond on or before 30 April 2024. The bond amount is A$27,956,000. Ocean Partners and Polymetals has entered a Memorandum of Understanding (MOU) for the purpose of agreeing commercial terms to replace the Endeavor Mine Environmental Rehabilitation

Criteria	JORC Code explanation	Commentary
	approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.	Bond subject to a positive study outcome. • A $15 million Concentrate Pre-Payment Facility has also been secured with Ocean Partners. • All mining leases are current, with no outstanding government approvals required to restart mining operations.
Classification	• *The basis for the classification of the Ore Reserves into varying confidence categories.* • *Whether the result appropriately reflects the Competent Person's view of the deposit.* • *The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).*	• The classification of the Endeavor Mine Ore Reserves have been carried out in accordance with the guidelines contained within the JORC Code (2012). Classifications are based on data density, geological knowledge, historical mine performance and proposed mining methods. Measured Mineral Resources were converted in Proven Ore Reserves while Indicated Mineral Resources were converted to Probable Ore Reserves. • The results of the Ore Reserve estimate appropriately reflect Competent Person's view of the deposit. • All of the Probable Ore Reserves have been derived from Indicated Mineral resources.
Audits or reviews	• *The results of any audits or reviews of Ore Reserve estimates.*	• Ther have been no audits of the Ore Reserve estimate.
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *Accuracy and confidence discussions should extend to*	• The Mineral Resource Estimate and hence the Ore Reserve Estimate relate to global estimates. • The Ore Reserve Estimate is derived from the Mine Restart Study which was prepared to a Pre-Feasibility level of accuracy. Capital and operating costs have been estimated to accuracies of +/- 15% to +/- 25%. Modifying factors for mining are based on actual historical site performance. • There has been an appropriate level of consideration given to all modifying factors to support the declaration and classification of Ore Reserves.

Criteria	JORC Code explanation	Commentary
	specific discussions of any applied Modifying Factors that may have a material impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the current study stage. • *It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	

Attachment 2

TSF Sector 1 Mineral Resource Estimate Report 2023

Endeavor Mine

TSF Sector 1

Mineral Resource Estimate

Date: October 2023

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TABLES

FIGURES

ATTACHMENTS

Attachment 1	JORC Code Table 1
Attachment 3	Drill Hole Details

Executive Summary

This report is a record of the assessment of the contained Mineral Resources within tailings stored within the Tailings Storage Facility (TSF) Sector 1 at the Endeavor Mine, NSW, Australia. The site is currently under care and maintenance with Polymetals Resources Ltd intending to take control of the project and restart operations.

Over the life of the mine tailings, from the processing of sulphide ores by floatation methods, were deposited into a Tailings Storage Facility (TSF) directly to the south of the processing plant. Tailings were deposited into Sector 1 of the TSF between 1983 and 1989.

The tailings in Sector 1 have been investigated by drilling programs in 2014, 2015 and 2017 and have been drilled by air core and push tube drilling methods.

Analysis of the data quality has revealed that data from the 2014 air core drilling program is unsuitable for use in the estimation of Mineral Resources do to issues with both the drilling method and assay results. Data from the 2015 push tube drilling program was considered robust and suitable for use. Sampling in 2017 was undertaken for Metallurgical test work only.

The resource model is based on statistical and geostatistical investigations generated using 2m composited sample intervals.

A sub-celled block model was constructed using parent block dimensions of 50m East by 50m North by 2mRL for, with sub-blocking for the purpose of providing appropriate definition of the topographic surface, and domain boundaries.

Grade estimation was carried out for zinc, lead and silver based on analytical results available up to August 2023. Ordinary Kriging (OK) was selected as an appropriate estimation method based on the quantity and spacing of available data and style of deposit under review. A multi-pass strategy was employed to generate the grade estimates. The number of composites for a successful estimate was restricted to a minimum of 3 and a maximum of 12. The search critera were aligned with results derived from variographic analyses of the data sets.

Grade and tonnes contained within TSF Sector 1 reported with no cut off grade is presented in **Table 1**.

Table 1 – Endeavor Mine TSF Sector 1 Mineral Resource September 2023

Category	Mt	Zinc (%)	Lead (%)	Silver (g/t)
Indicated	3.6	2.14	1.56	80
Inferred	1.6	2.07	1.53	77
Total[1]	**5.2**	**2.12**	**1.55**	**79**

1. Discrepancies may occur due to rounding

This report complies with disclosure and reporting requirements set forth in the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' of December 2012 (the Code) as prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Mineral Council of Australia (JORC).

1 Introduction

1.1 Background

Polymetals Resources Ltd has undertaken a Resource estimation study of the material contained within Sector 1 of the main Tailings Storage Facility of the Endeavor Mine, NSW.

This study follows drilling programs undertaken in 2014, 2015 and 2017.

This report provides details of the work activities and results of the resource estimation study based on the following: -

- Review drill hole data and investigate the integrity of the captured data.
- Review wireframe models that represent the mineralised domains for Sector 1.
- Complete statistical analyses of drill hole data.
- Produce grade estimates based on an appropriate method, applying suitable and appropriately optimised estimation parameters.
- Visual and statistical validation of grade estimates.
- Report contained Resource estimates in accordance with JORC Code 2012 guidelines.

The personnel involved in the Resource estimation study, including their principal areas of responsibility, are:

- Troy Lowien, General Manager – Geology, Competent Person under JORC Code 2012
    - Review of the three-dimensional model, statistical analysis, grade estimation and report preparation.

1.2 Principal Sources of Information

Digital data used in this study has been sourced from the CBH server based on site at Endeavor.

The following key data relevant to the Resource estimate was located:

- Drill hole data including, collar, survey, assay and geological information from the 2014, 2015 and 2017 drilling programs, that the CP accepts in good faith as an accurate, reliable and complete representation of available data.
- Reports on Sector 1 drilling results.
- Topographic survey of the area.
- Three dimensional wireframe model of the original; and surface.
- Metallurgical test results.
- Historical tailings deposition records.

1.3 Project Location and Tenure

The Endeavor mine project is located 47km north west of Cobar, New South Wales, Australia (**Figure 1**).

Latitude -31.160 S

Longitude 145.653 E

The project consists of an underground zinc-lead-silver mine, processing plant, tailings dams and rail loading facility. The mine is owned by Cobar Operations Pty Ltd (COPL) and operated by Endeavor Operations Pty Ltd (EOPL). Both companies are currently wholly owned subsidiaries of CBH Resources Ltd (CBH).

Polymetals, through its subsidiary company Cobar Metals Pty Ltd, has entered into a legally binding arrangement to acquire 100% interest in the Endeavor mine and associated assets by acquiring COPL, EOPL and Endeavor Infrastructure Pty Ltd (EIPL) from CBH. In order to complete the acquisition Cobar Metals will be required to procure the release and replacement of the Environmental Rehabilitation Bond on or before 30 April 2024.

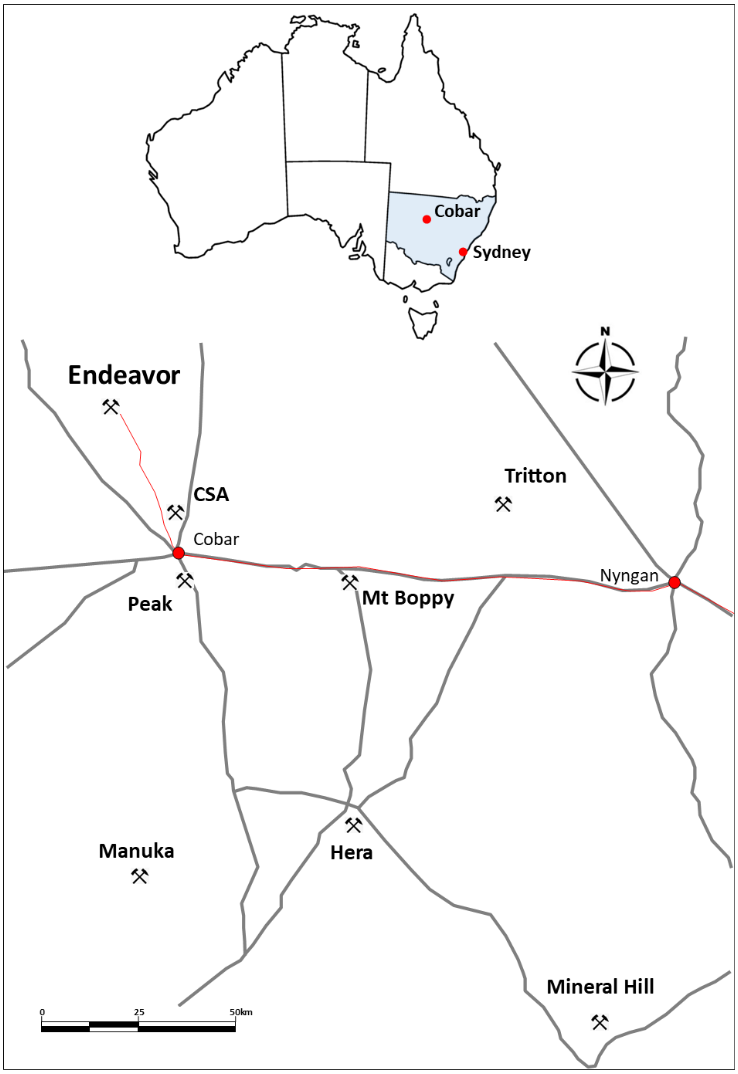

Figure 1: Project Location and Nearby Mines

The Endeavor Mine is covered by five granted Mining Leases as shown in **Table 2** and **Figure 2**.

Table 2 – Relevant Mining Leases

Title	Holder	Expiry Date	Purpose
ML158	Cobar Operations Pty Ltd	12/03/2028	Surface and underground mining activities for minerals.
ML159	Cobar Operations Pty Ltd	12/03/2028	
ML160	Cobar Operations Pty Ltd	12/03/2028	
ML161	Cobar Operations Pty Ltd	12/03/2028	
ML930	Cobar Operations Pty Ltd	20/05/2028	Underground mining activities for minerals (surface exclusion of 10m)

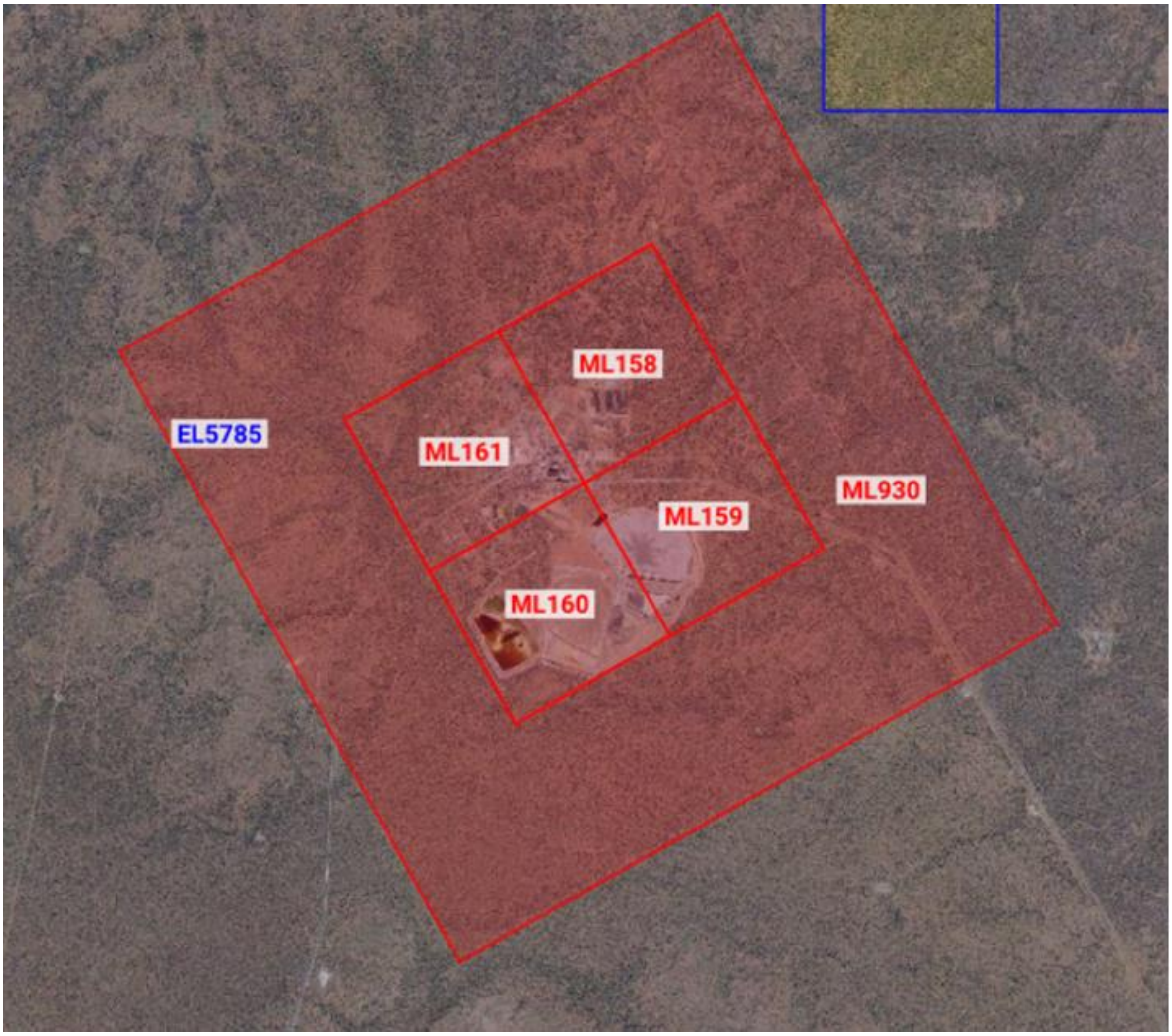

Figure 2: Mining Leases

2 Project Background

2.1 Project History

The Elura Pb-Zn-Ag deposit was first discovered in 1973 by the Electrolytic Zinc (EZ) Company of Australia using aeromagnetic surveys followed up by auger and diamond drilling. This drilling enabled the reporting of an initial resource of 27 Mt @ 5.6% Pb, 8.6% Zn and 135 g/t Ag.

Further exploration was carried out in 1976 via the excavation of a 165m deep shaft and cross-cut to access the deposit and extract material for metallurgical test work.

Following a positive feasibility study in 1977 construction began on the Elura Mine project in 1980, with the first ore milled in November 1982. A total of 0.7 Mt of ore was milled during the first year of production.

The mine was acquired by North Broken Hill Holdings Ltd in 1985, after the latter took over EZ Industries Ltd in 1984. Subsequently it became part of Pasminco Ltd Holdings in 1988. Production increased to around 1.2 Mt per year until the early 90's when the rate was reduced back to around 0.7 Mt per year due to a fall in metal prices, then increasing back to around 1 Mt per year in 1995.

Pasminco was placed into voluntary administration in 2001 and the mine was acquired by CBH Resources in 2003, changing the name of the project to Endeavor Mine. From 2009 the mine operated again on a reduced production rate of around 0.6 Mt per year due to lower metal prices before being placed on care and maintenance in 2019.

During the life of the mine around 32 million tonnes of ore has been extracted.

In March 2023 Polymetals announced it had executed a Share Sale and Purchase Agreement to acquire 100% interest in the Endeavor Mine via acquisition of Cobar Metals Pty Ltd, a company that had separately entered into an arrangement to acquire the project.

3 Geological Setting

NOTE – Geological descriptions below are not directly related the subject of this Resource estimation study (i.e. tailings) but help give an understanding of the mineralisation that was processed and ultimately deposited as tailings.

3.1 Regional Setting

The Elura Pb-Zn-Ag deposit is located in the north western region of the Cobar Basin in the Lachlan Fold Belt, central western NSW. The Cobar Basin lies on a basement of Ordovician sediments and Silurian granitic rocks and formed during the Silurian/Devonian as a series of deep-water, half graben troughs/basins and shallow water shelfs, containing predominantly siliciclastic sediments with minor volcanic and carbonate rocks (Figure 3). The basin formed by NE-SW transtension and was closed by NW transpression in the Carboniferous. Basin inversion is characterised by NW-SE folding, overprinted by NE-SW, and NNW-trending eastwards oblique left-lateral reverse faulting (David, 2018)

Mineralisation within the Cobar Basin is controlled by basement architecture, overprinted and modified with secondary controlling factors of inversion tectonics. Types of mineral deposits within the basin include massive sulphides (VMS), clastic hosted Pb-Zn and epithermal gold. These deposits were formed during the early rift-phase on the eastern margin, during later basin inversion, or a combination of early formation and later remobilisation (Figure 4).

3.2 Local Geology and Mineralisation

The Elura deposit is hosted by a limestone breccia overlain by a turbidite sequence of interbedded shale and sandstone/siltstone. The carbonate rocks have been interpreted as belonging to the Brookong Formation of the Kopyje Group and the turbidites are thought to be lithologically equivalent of the CSA Siltstone.

The main orebody is massive sulphide hosted by the fine grained turbidite sequence and comprises multiple sub-vertical elliptical shaped pipe-like pods with an envelope of sulphide stringer mineralisation, in turn surrounded by an envelope of siderite alteration extending for tens of metres away from the sulphide mineralisation. Above about 900m depth, the sulphide stringer mineralisation occurs as a large continuous 15 - 120m wide sheet within the axial plane of an anticline and extends over a strike length of at least 800m. Below 900m depth the stringer zone breaks up and occurs as grossly concordant zones paralleling the limbs of the anticline.

The orebody is generally divided up into the main lode, which consists of two elliptical pods that merge together around 10 000 m RL and the northern Pods which consist of five smaller crudely elliptical orebodies that trend towards the NNW and dip approximately 80° - 85° towards the west. The crusher pod, which is located on the eastern side of the main lode, is a small apophysis that merges into the Main Lode at the 9500 m RL.

Mineralisation in the orebody is complex. The primary lead and zinc bearing minerals from all orebodies processed are galena (~13 per cent wt) and sphalerite (~14 per cent wt). Pyrite and pyrrhotite (~60 - 70 per cent wt in total) are the main floatable gangue in the ore. Tetrahedrite is the major host of silver, apart from galena and chalcopyrite. The average grain size of galena and sphalerite are relatively very fine, ranging from 10 - 40 µm.

Polymetals

3.2.1 Tailings

Mineralised material in the tailings storage facility consists of clay to fine sand sized particles deposited in sub-horizontal layers from centrally located outflow sites. The particles contain remnant sulphides that were not captured during the beneficiation process.

The mine commenced operations in 1983 and tailings were pumped exclusively into Sector 1 (see **Figure 3**) from March 1983 until April 1989 (with the exception of tailings from concurrent Ag supergene mining which were pumped to a separate, smaller TSF southwest of the mine) when Sector 2 of the TSF was commissioned. No further tailings have been added to Sector 1 since 1989.

The northern portion of Sector 1 was rehabilitated in 1991 when the surface was re-contoured, covered with plastic, clay capped, and topsoil emplaced.

The TSF is a raised "turkey's nest" type dam, with Sector 1 measuring approximately 550m by 850m and an average depth of around 7m.

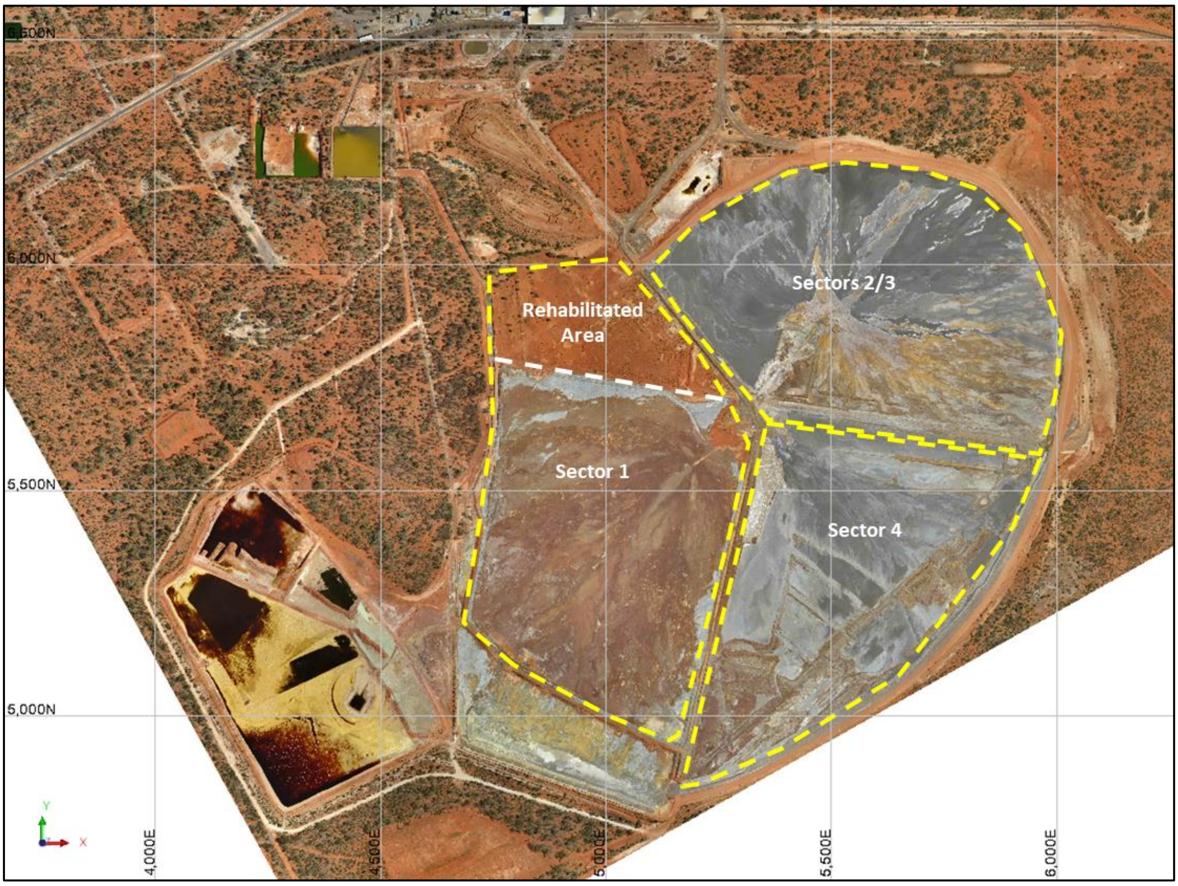

Figure 3: Endeavor Mine TSF Layout

4 Data Collection

4.1 Drilling

The tailings contained within Sector 1 of the TSF have been investigated by drilling programs in 2014, 2015 and 2017 (CBH Resources). Overall, 204 holes were drilled, totalling 1,135 m of drilling, of which 34% was completed using push tube methods, and 66% by air core methods. Drilling in the rehabilitated area of Sector 1 was not carried out due to directive from the Environmental Protection Authority.

4.1.1 2014 Drilling Campaign

Drilling was undertaken in December 2014 using an Air Core Rig by Colling Exploration. A total of 134 holes, for a total of 751m were drilled with depths between 3m and 11m and an average depth of 5.6m. A number of holes (10) spread across Sector 1were designed to breach the base of the TSF. Drilling was carried out on a 50m by 50m grid pattern.

Figure 4: Air Core Rig (2014)

4.1.2 2015 Drilling Campaign

Assayed grades from the 2014 drilling campaign were used to estimate contained tonnes and grade, which resulted in metal content significantly lower than the mine production and processing records indicated. Consistently low Pb grades were also received for standards used in the QAQC program. This cast doubt over the drilling method and/or the assaying results. As a result 21 pulps were submitted to an external laboratory for re-assay which resulted in Pb grades on average 13.4% higher. A series of trench (20) samples were also taken adjacent to air core drill collars from across Sector 1 to provide comparative data. On average, the trench samples reported Pb grades 45% higher and Zn grades 32% higher than the air core samples and generally reflected historical production records.

A 20-hole push drilling program was undertaken by Numac Drilling in April 2015 to twin both the air core holes and the trench sampling with a more reliable sampling method. The Geoprobe 77200T rig is powered by a high speed pneumatic hammer that drives the core barrel down without rotation to produce an essentially undisturbed, in-situ sample.

The rig was able to set up within 0.5m of the air core collars and drilled to almost the same depth as the air core holes while ensuring that they did not breach the TSF floor. A total of 97.8m was drilled for the 20 holes.

Figure 5: Push Tube Rig (2015)

4.1.3 2017 Drilling Campaign

A 50-hole push drilling program was undertaken by Numac Drilling in March 2017 to collect sample for metallurgical test work. Twenty one holes were spread across Sector 1 on an approximate 100m by 100m grid to collect samples for a global composite, while 5 groups of 3 to 8 closely spaced holes (29 total) were drilled to collect composites to evaluate variability.

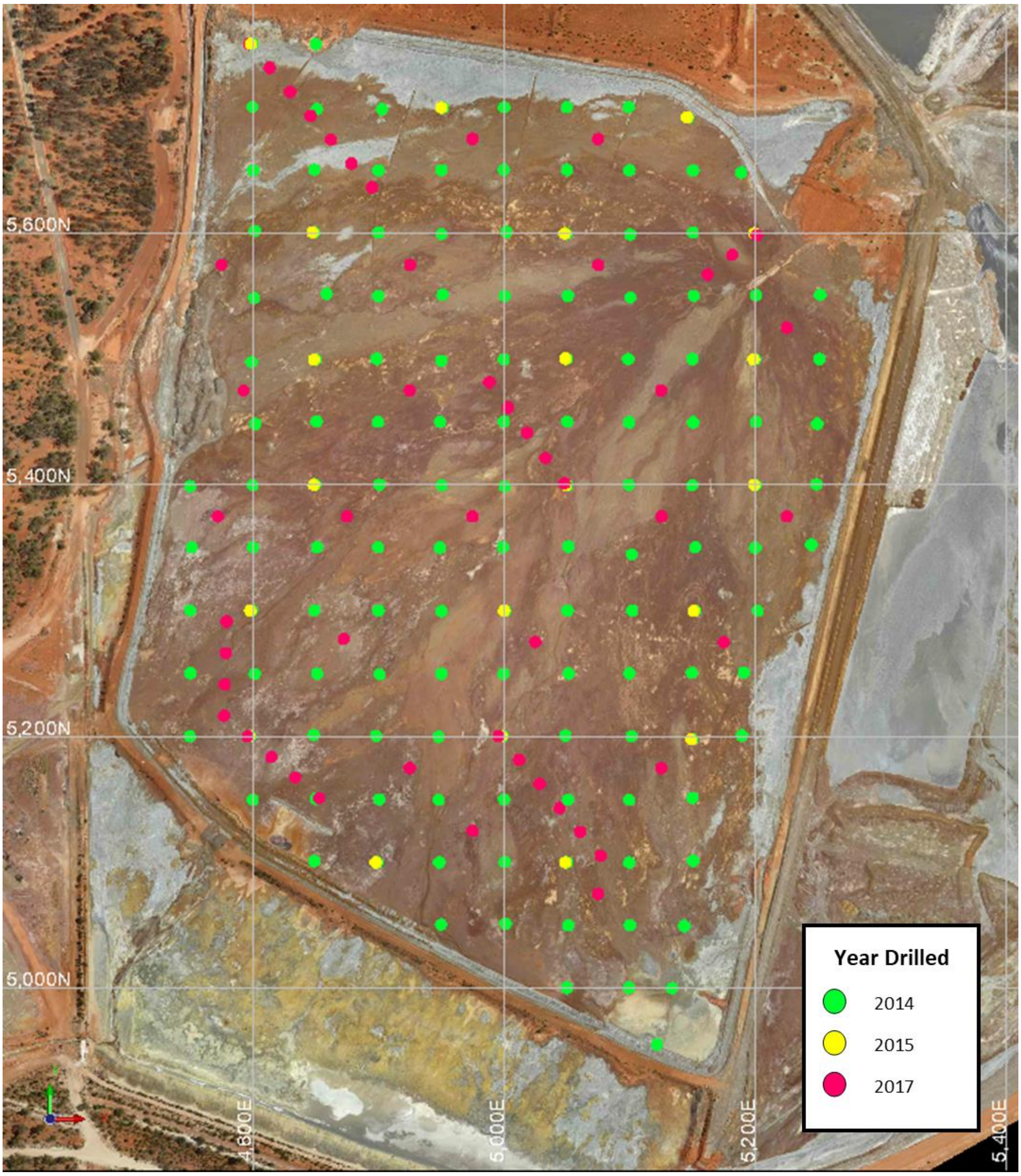

Figure 6: Drill Hole Locations Coloured by Year Drilled

Polymetals

4.2 Surveying

4.2.1 Introduction

The Endeavor Mine is located in Zone 55 of the Map Grid of Australia (MGA) 94 coordinate system. All surveying at the Endeavor Mine has been recorded in a local mine grid which is related to the MGA94 grid by the parameters as shown in **Table 3**.

Table 3 – Transform Parameters MGA94 to Local Mine Grid

		MGA94	Local Mine Grid
Point 1	Northing	6551419.471	6451.175
	Easting	372517.808	5231.564
Point 2	Northing	6551409.739	6452.863
	Easting	371884.310	4597.827
Elevation Correction		+10,000	

4.2.2 Drill Hole Collars

Drill hole collars were surveyed by the mine surveyor by unknown methods.

4.2.3 Topography

An aerial photogrammetry survey was carried out over the site in December 2015 by Arvista Pty Ltd at a ground resolution of 5cm per pixel. A Digital Terrain Model (DTM) in Surpac format was supplied and used in this study.

A DTM of the surface topography prior to tailings deposition is not available. To determine the depth of the Sector 1 floor below the surface, 10 (the maximum permitted by the EPA) of the 134 holes drilled were designed to breach the TSF floor. From these 10 points, a simple Sector 1 floor DTM was produced and the remaining 124 holes were designed to stop 0.5 to 1.0m above that surface.

4.2.4 Down Hole Surveying

There were no downhole surveys undertaken on the drill holes. All holes were drilled vertically and were relatively short (<15m depth), and therefore any downhole deviation would have negligible effects on the location of datapoints.

4.3 Logging

Detailed logging of the tailings is considered impractical and unnecessary as the tailings have been homogenised from processing and deposition. Material changes were noted when drill holes intersected the base of the tailings dam.

4.4 Sampling

4.4.1 2014 Air Core Program

To reduce the number of samples being sent to the site laboratory, 2m composites from 1m intervals were produced and 4 different sample types were taken:

- Assays with Chip Trays
- Bulk Density
- Regional Metallurgy
- Global Metallurgy

A rotary and riffle splitter were tried on the first hole but the puggy nature of the material was unsuitable, so all sampling was done by the spear method. The puggy material meant there was minimal loss of sample from dust escaping the cyclone, but did mean the cyclone usually needed vigorous cleaning after each 1m interval to ensure complete sample recovery. Samples were then taken by spearing to the bottom of the 1m sample bag.

Assay samples were taken from the 2m composites and stored in small, pre-labelled and ticketed calico bags then taken to the site laboratory on a daily basis. Rock chip tray samples were also taken but were not washed or sieved due to the fine grained and clayey nature of the material. A total of 375 samples were submitted for assays while 18 intervals had insufficient sample to submit an assay.

A dedicated geologist and field assistant were in attendance at all stages of drilling.

4.4.2 2015 Push Tube Program

Samples were split laterally with sample lengths between 0.2m to 1.6m, with an average of 1.2m.

4.4.3 2017 Push Tube Sampling

Samples were split laterally with sample lengths between 0.4m to 1.2m, with an average of 1.0m. Half core samples from the 20 “global sample” holes were combined into one composite for submission to an external metallurgical laboratory. Half core samples from the 29 variability holes were combined into 5 composites (A to E) for submission to an external metallurgical laboratory.

4.5 Recovery

No recovery information is available.

Figure 7: Push Tube Sampling

4.6 Sample Preparation and Analysis

4.6.1 2014 Samples

Samples were assayed at the Endeavor laboratory using an Aqua Regia digest with atomic absorption spectrometry (AAS) for lead, zinc, silver, iron and copper analyses. The samples were prepared at the Endeavor laboratory and were subjected to the following preparation methodology:

- A scoop sample was placed into the pulveriser.
- Samples were then pulverized to pass 38 micron and split to usually a 200-300ml aliquot.
- The pulps were prepared in an Aqua Regia digest and analysed using flame absorption spectrometry for lead, zinc, copper, iron and silver.

4.6.2 2015 Samples

Samples were sent to ALS-Orange and assayed by an Aqua Regia digestion using AAS (ICP-AES) analysis for lead, zinc, silver, iron and copper. A prepared sample (0.4 g) is digested with concentrated nitric acid for 90 minutes in a graphite heating block. The resulting solution is diluted with concentrated hydrochloric acid before cooling to room temperature. The samples are diluted in a volumetric flask (100 or 250) mL with demineralized water and analysed using atomic absorption spectrometry or atomic absorption spectrometry.

A composite sample of the 20 holes was also sent to the ALS Metallurgy laboratory in Burnie, Tasmania. An assay of the head sample was carried out by XRF on a pulverised sample.

4.6.3 2017 Samples

The global and variability composite samples were sent to the ALS Metallurgy laboratory in Burnie, Tasmania. Analysis of the head samples were carried out by unknown method.

4.7 Assay Quality Control Procedures

4.7.1 2014 Drilling Program

The quality control regime used in the 2014 drilling program consisted of Certified Reference Material (CRM or Standards) and Blanks inserted into the sample stream, field duplicate samples, and re-assays of laboratory pulp samples. The insertion rate of QC samples into the submission stream was 1 in 6 samples.

4.7.1.1 Certified Reference Material

At the time of drilling, the focus was on Zn recovery, therefore three standards in the low, medium and high range of the expected Sector 1 Zn grades were selected for inclusion the assay sample stream (**Table 4**), with Pb and Ag grades of lesser priority due to the oxidised and refractory nature of the Pb particles.

Standards were inserted in the assay stream at a rate of 1:17 samples, alternating though the 3 standards. Early assay results indicated an issue with the Pb grades so to provide more data on the Pb grades, OREAS 131B was increased to every second standard (the 2 Geostats standards having Pb grades below the level of interest) with the 2 Geostats standards alternating in between.

Table 4 – CRM List

ID	Standard Material	Certificate Issued by:	Elements
131b	SEDEX Zn-Pb_Ag deposit (carbonate siltstone)	Ore Research & Exploration Pty Ltd	Zn, Pb, Ag
GBM908-11	Cu Concentrate Ex Pilbara	Geostats Pty Ltd	Zn Ag
GBM906-14	Cu Zn Sulphide Ore	Geostats Pty Ltd	Zn

4.7.1.2 Blanks

A total of 25 blanks were included in the assay sample stream at a rate of 1 in 16 samples.

4.7.1.3 Field Duplicates

A total of 22 field duplicates were included in the sample submission at a rate of 1 in 16 samples.

4.7.1.4 Re-Assays

21 pulps (including a OREAS 131B standard) were submitted to ALS for re-assaying. The re-assay samples were selected from all 16 batches processed by the site laboratory and were chosen from 3 approximate grade ranges: 0.6, 1.2 and 1.8% Pb.

4.7.2 2015 Drilling Program

The quality control regime used in the 2015 drilling program consisted of Certified Reference Material (CRM or Standards) and Blanks inserted into the sample stream at a rate of about 1 in 10 samples. However, these samples were not assayed at the laboratory due to insufficient sample quantities according to the results certificate.

4.7.3 2017 Drilling Program

No recorded quality control samples were included in the submission of the 2017 samples to the metallurgical laboratory.

4.8 Density Measurements

Density measurements were carried out during the 2014 drilling program. 551 density samples were taken from each 1m interval by firmly compressing (manually by the field assistant) the material into a 'grout' sampling container (109mm length x 52mm diameter) and levelling the top off. Each sample was stored in zip-lock plastic bags and taken to the site laboratory for later wet weight and dry weight measurements.

5 Data Verification

5.1 Assessment of Quality Control Data

The accuracy and precision of the assay data for the 2014 drilling program was assessed based on assays of certified reference material (CRM's or Standards) including blank material, field duplicate samples inserted into the sample stream, and re-assays of sample pulp material, as part of the quality control procedures for the drilling program. Results of quality control samples inserted during sample submission are not available for the 2015 or 2017 drilling programs.

5.1.1 Assay Accuracy

The accuracy of the assay data and the potential for cross contamination of samples during sample preparation has been assessed based on the assay results for the field standards and blanks for the 2014 drilling program and laboratory standards, blanks and pulp duplicates for the 2015 drilling program.

5.1.1.1 2014 Drilling Program

The results of the statistical analysis of the standards, including blanks, as analysed by the Endeavor Operations Laboratory can be summarise as follows:

Blanks

One sample reported 0.25% Pb. Otherwise, all samples reported grades close to or below the level of detection (BLD) for Pb, Zn, and Ag. (0.01% Pb, 0.01% Zn, 2ppm Ag). The anomalous result followed samples of approximately 1.4% Pb, 1.8% Zn, and 76ppm Ag. Grades for other elements in the sample were all BLD, eliminating contamination as a cause.

Standards

Summary statistics of the assay data for each of the standards, along with the standard names and expected values are displayed in **Table 5**. Standard control plots are provided in **Figure 8** to **Figure 10**

Table 5 – Standard Statistics 2014 (EOPL Lab)

ID	Expected Value (EV)	± 2 Std Dev	No. of Analyses	Min	Max	Mean	% within ± 2 Std Dev	% RSD	% Bias (from EV)
Zn (%)									
OREAS 131b	3.04	2.80-3.28	13	2.74	3.41	3.12	69	6.02	2.48
GBM906-14	1.59	1.50-1.68	6	1.47	1.7	1.57	67	4.75	-1.26
GBM908-11	2.36	2.14-2.58	6	2.07	2.55	2.31	83	7.5	-2.12
Pb (%)									
OREAS 131b	1.88	1.71-2.05	13	0.45	1.87	1.47	31	27.01	-21.81
Ag (ppm)									
OREAS 131b	33	31 – 35	13	29	34	31.9	77	4.94	-3.50
GBM908-11	11.4	8-14.8	6	8.6	13	10.02	100	17.4	-12.1

Note: RSD = Relative Standard Deviation

Figure 8: Standard Control Plots - Zn

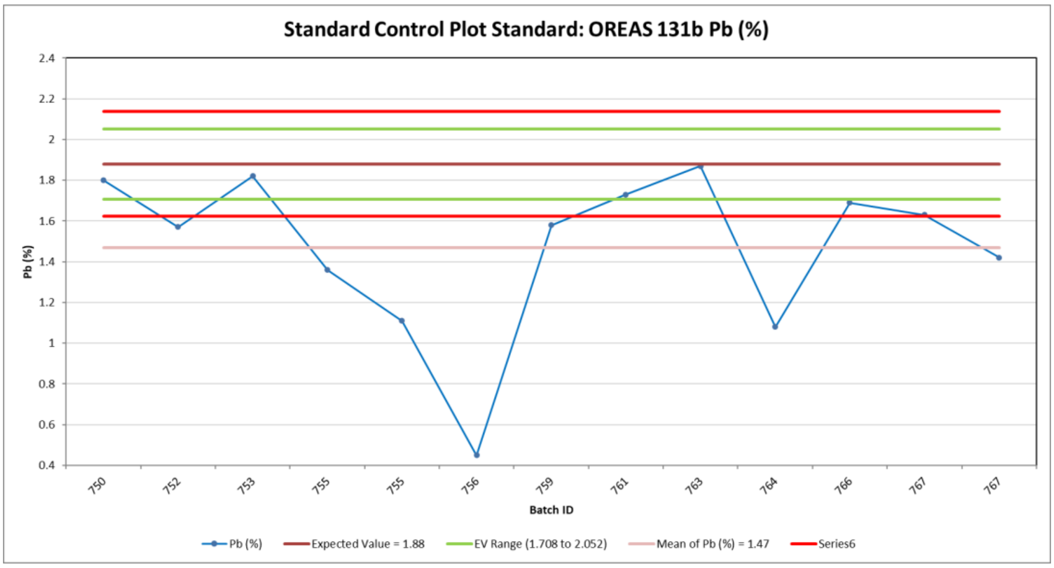

Figure 9: Standard Control Plot - Pb

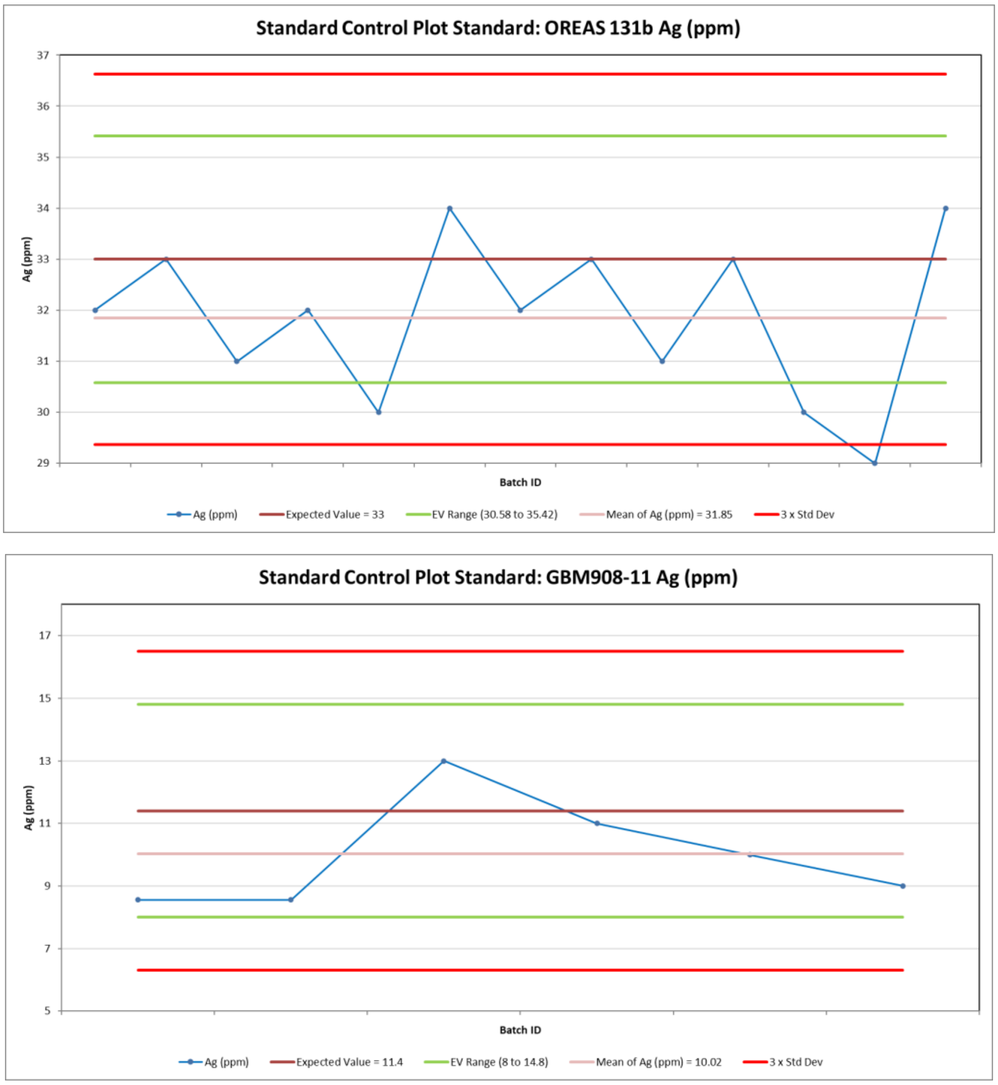

Figure 10: Standard Control Plots - Ag

The majority of results for Zn and Ag are within 2 standard deviations of the expected CRM value, with a slight negative bias for Ag. The results for Pb appear to be problematic with a large negative bias. All results are lower than the expected value with only 30% within 2 standard deviations of the expected value.

Re-Assay of Pb samples

The negative bias of Pb grades identified by the CRM analysis led to 21 pulps (including a OREAS 131B standard) being submitted to ALS for re-assaying in January 2015. The re-assay samples were selected from all 16 batches processed by the site laboratory and were chosen from 3 approximate grade ranges: 0.6, 1.2 and 1.8% Pb.

The results are shown in **Table 6** below and show 13 of the 21 samples reporting Pb grades more than 5% higher than the original assay. Re-assay results are up to 55% higher (average 13.4%) than the original assays. This is consistent with the trend shown in the CRM analysis for Pb assays from the Endeavor lab. The lower grades are most affected (**Figure 11**) but it was analysis of the 'higher' grade OREAS 131B CRM (1.86% Pb) which initially identified the trend. This is probably due to the fact that none of the high grade re-assays were sourced from the most problematic batches.

Table 6 – Pb Original and Re-assays

Sample	Pb% Orig	Pb % Re-assay	% Change
C220583	0.58	0.90	54.8
C220460	0.58	0.87	49.7
C220536	0.65	0.96	48.3
C220603	1.15	1.55	34.3
C220387	0.64	0.80	24.7
C220447	0.60	0.73	21.7
C220262	1.16	1.33	14.5
C220515	1.18	1.32	11.4
C220444	1.16	1.24	6.9
C220702	1.25	1.33	6.0
C220559	1.84	1.95	6.0
C220373	1.19	1.26	5.9
C220632	0.57	0.60	5.3
C220303	1.18	1.23	4.2
C220292 (STD)	1.80	1.85	2.5
C220331	1.17	1.19	1.7
C220606	1.75	1.76	0.6
C220706	1.81	1.80	-0.8
C220271	0.61	0.59	-3.3
C220334	0.64	0.61	-5.0
C220296	1.76	1.61	-8.8
		Avg % Change	**13.4**

Analysis of the results of both the Pb CRM and the re-assays has identified a bias in the Pb assays that could be broadly rectified by applying a factor to Pb grades based on the regression line shown in **Figure 11** as follows:

For Original Pb% < 1.73, Adjusted Pb% = Original Pb% x (1+((-21.273xOriginal Pb%)+36.931)/100)

This is likely to be a conservative adjustment based on the high grade samples in **Figure 11** being sourced from the batches with better CRM results in the original assay submissions.

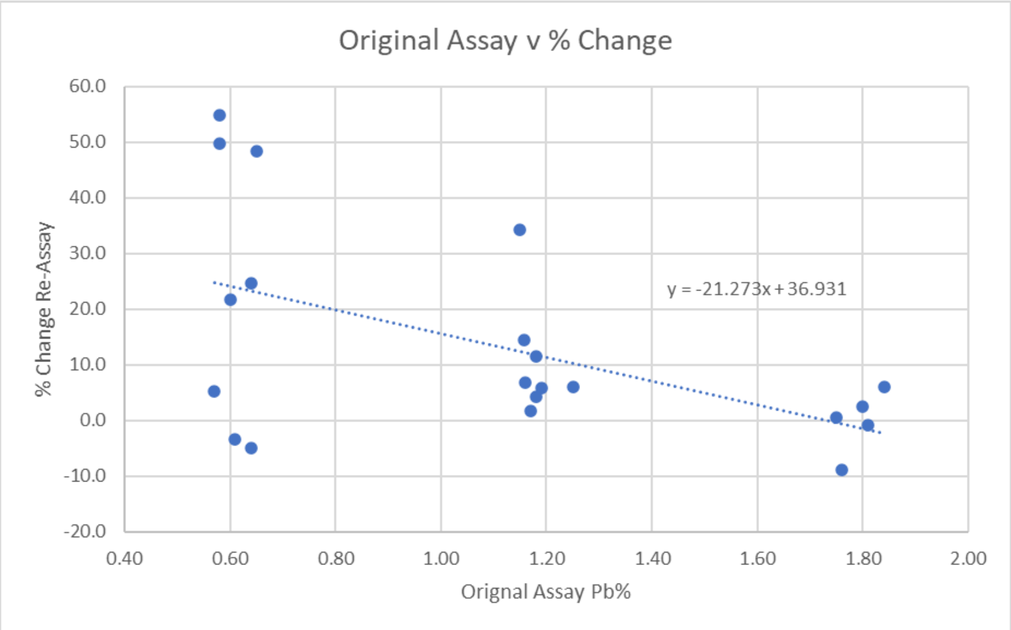

Figure 11: Original Assys v Re-Assay Change%

5.1.1.2 2015 Drilling Program

The results of the statistical analysis of the standards, including blanks, as analysed by the ALS Laboratory in Orange can be summarise as follows:

Blanks

All results (5) of the blank material assayed with the submitted samples were at or below detection levels for Zn (0.001%), Pb (0.001%) and Ag (1ppm).

Standards

Summary statistics of the assay data for each of the standards, along with the standard names and expected values are displayed in **Table 7**.

Table 7 – Standard Statistics 2015 (ALS)

ID	Expected Value (EV)	± 2 Std Dev	No. of Analyses	Min	Max	Mean	% within ± 2 Std Dev	% RSD	% Bias (from EV)
Zn (%)									
OREAS 134b	18.03	16.52-19.54	2	18	18.3	18.15	100	1.17	0.67
GBM306-12	2.06	1.93-2.20	3	2.01	2.04	2.03	100	0.75	-1.78
Pb (%)									
OREAS 134b	13.4	11.9-14.9	2	13.35	13.55	13.45	100	1.05	0.67
GBM306-12	2.71	2.52-2.90	3	2.61	2.68	2.65	100	1.36	-2.20
Ag (ppm)									
OREAS 99b	78.6	75-82.2	2	79	81	80	100	1.77	1.78
OREAS 134b	209	191-227	2	211	212	211.5	100	0.33	1.20

All results are within two standard deviations of the expected value with no appreciable bias detected.

5.1.2 Assay Precision

5.1.2.1 2014 Drilling Program

The precision of the assay data has been assessed based on the assay results for the field duplicates. Field duplicates allow assessment of total precision, reflecting sample collection, preparation, and analytical errors at the lab.

Details of the available datasets and results of the statistical analyses are summarised below. Statistical analysis of each dataset has considered only those assays data greater than or equal to 10 times the analytical detection limit.

Generally expected values are in the order of 15 to 20 Mean % HARD for field duplicate samples (i.e. the most sampling error.) In the assessment of data using Rank HARD plots, generally acceptable limits for field duplicate data are 80% within 30% Rank HARD.

The results of the statistical analysis of the comparative QAQC assay data are displayed in **Table 8** and **Figure 12** and can be summarised as follows:

- The field duplicate datasets are well within acceptable limits and display no obvious bias.
- Industry accepted levels of precision are reported for the sampling.

Table 8 – Summary of Data Precision 2014

Data Comparison	No. of Data Pairs	Mean %HARD	Median %HARD
		Ag	
Duplicate Field Samples	20	4.71	3.37
		Pb	
Duplicate Field Samples	21	4.11	3.18
		Zn	
Duplicate Field Samples	22	4.64	2.50

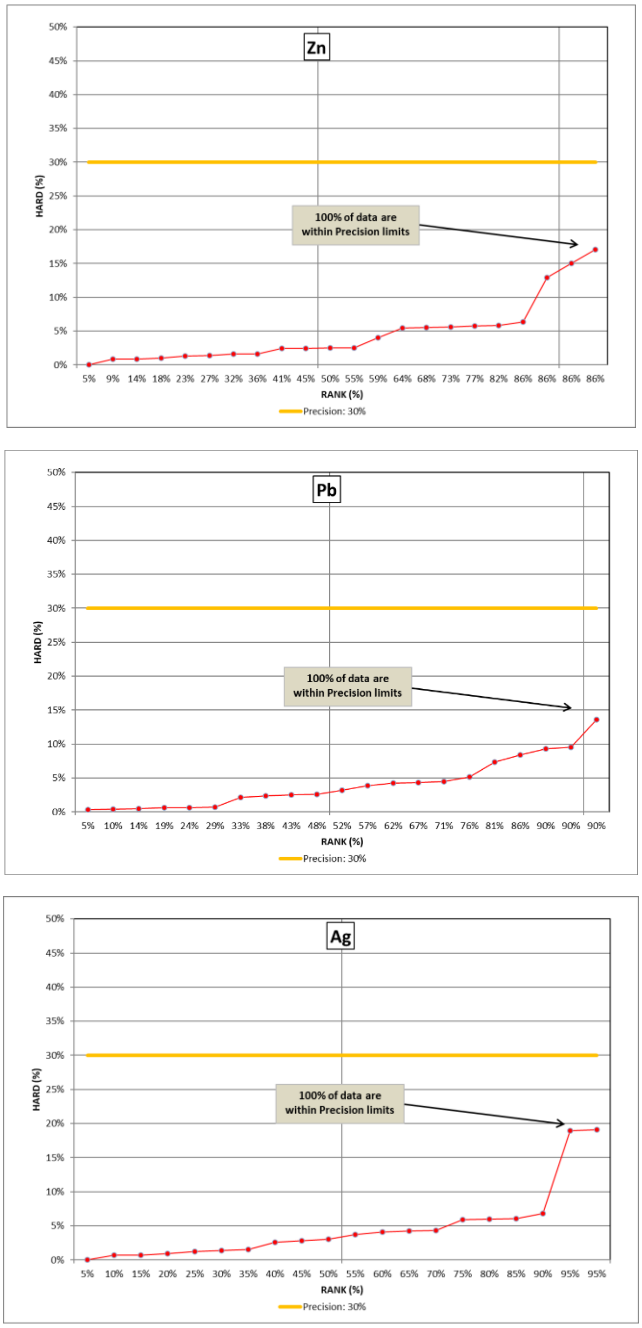

Figure 12: Rank % HARD Plots

5.1.2.2 2015 Drilling Program

The precision of the assay data has been assessed based on the assay results for the pulp duplicate analyses. Pulp duplicate analyses allow assessment of precision, reflecting sample preparation and laboratory analytical errors.

Details of the available datasets and results of the statistical analyses are summarised below. Statistical analysis of each dataset has considered only those assays data greater than or equal to 10 times the analytical detection limit.

Generally expected values are in the order of 5 to 10 Mean % HARD for pulp duplicate samples (i.e. the least sampling error.) In the assessment of data using Rank HARD plots, generally acceptable limits for pulp duplicate data are 80% within 10% Rank HARD.

The results of the statistical analysis of the comparative QAQC assay data are displayed in **Table 8** and can be summarised as follows:

- The field duplicate datasets are well within acceptable limits and display no obvious bias.
- Industry accepted levels of precision are reported for the sampling stages for the purpose of resource estimation.

Table 9 – Summary of Data Precision 2015

Data Comparison	No. of Data Pairs	Mean %HARD	Median %HARD
Ag			
Duplicate Pulp Samples	4	0.86	0.97
Pb			
Duplicate Pulp Samples	4	1.23	1.09
Zn			
Duplicate Pulp Samples	4	1.75	1.11

5.2 Assessment of Project Database

The data used in this Mineral Resource estimate was provided in Microsoft Excel spreadsheet format and combined into a single Microsoft Access database for loading into Surpac mining software.

5.2.1 Validation of Database

The following database validation activities have been carried out with no issues encountered:

- Ensure compatibility of total hole depth data in the collar, survey, assay, and geology drill hole database files.
- Check for overlapping sample intervals.
- Checking of drill hole locations against the surface topography.
- Visual validation.

5.3 Drill Hole Twinning

As mentioned in Section 4.1.2, a program of trench sampling was initiated due to the low grade assay results from the 2014 drilling program compared to historic tailings deposition records. Twenty trenches were excavated by backhoe in March 2015 to a depth of 2m adjacent to drill sites from the air core drilling program (**Figure 13**). The trenches were designed to provide 2m samples from the entire Sector 1 area to obtain an approximate determination as to whether the air core samples were understating the Pb and Zn grades.

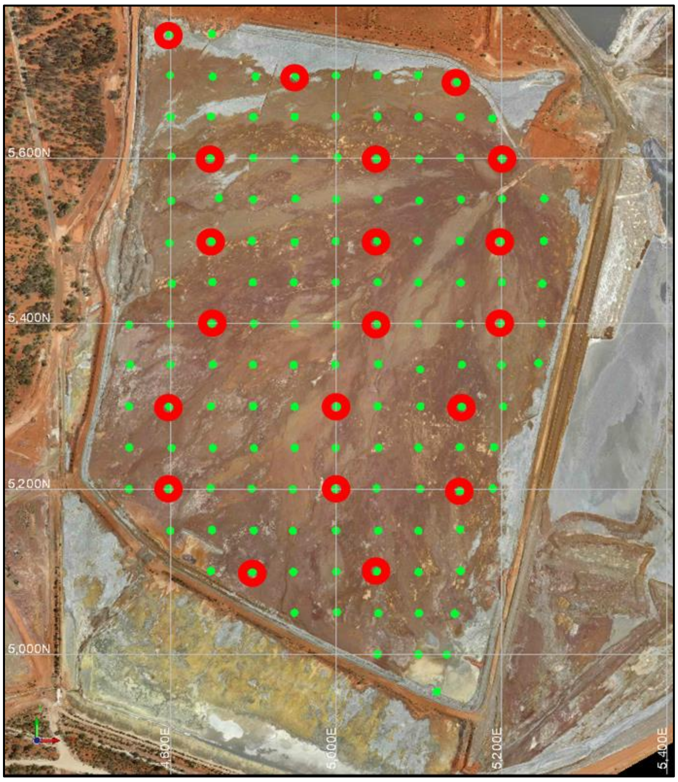

Figure 13: Air Core Holes (green) & Trench Locations (red)

The backhoe excavated each trench to a depth of 2m then moved around to the side of the trench to drag the bucket up the side to take a sample roughly corresponding to the 2m air core composites. The samples were spear sampled and sent to ALS Orange for assaying. Results are shown in **Table 10** and **Figure 14** and indicate there is a potential problem with the Air Core grades for Pb, Zn and Ag. All Pb and Ag trench assays, and the majority of Zn assays, were higher than the corresponding air core assays, with Ag showing the largest increase in grade.

Table 10 – Air Core & Trench Assay Comparisons

DHID	Pb %			Zn %			Ag ppm		
	AirCore	Trench	% Diff	AirCore	Trench	% Diff	AirCore	Trench	% Diff
S1_004	1.04	1.52	46%	1.52	2.21	45%	53	61	15%
S1_005	1.18	1.48	25%	2.85	3.83	34%	52	68	31%
S1_009	1.37	1.68	22%	1.63	2.26	39%	55	86	56%
S1_022	1.40	1.82	30%	1.55	1.20	-23%	50	70	40%
S1_025	1.17	1.68	43%	1.54	2.20	43%	43	88	105%
S1_029	1.56	1.68	8%	2.31	2.28	-1%	62	74	19%
S1_042	1.04	1.35	29%	1.35	1.29	-5%	34	39	15%
S1_045	1.14	1.42	24%	1.52	2.08	37%	41	60	46%
S1_049	1.35	1.50	11%	1.92	2.69	40%	62	78	26%
S1_062	1.32	1.90	44%	1.60	2.91	82%	47	110	134%
S1_065	0.93	2.24	141%	1.62	0.98	-40%	42	87	107%
S1_069	1.02	1.98	94%	1.50	1.61	7%	49	74	51%
S1_084	1.26	1.71	36%	1.87	2.08	11%	59	85	44%
S1_087	0.77	1.77	130%	1.25	2.24	79%	30	99	230%
S1_091	1.13	1.77	57%	1.48	1.97	33%	40	77	93%
S1_104	1.14	2.13	87%	1.75	2.06	18%	55	103	87%
S1_107	1.35	1.92	42%	1.62	2.24	38%	42	119	183%
S1_111	1.36	1.51	11%	1.66	2.56	54%	55	56	2%
S1_125	1.55	1.72	11%	1.60	2.87	79%	55	111	102%
S1_128	1.01	2.06	104%	1.40	2.90	107%	35	89	154%
Average	**1.20**	**1.74**	**44%**	**1.68**	**2.22**	**33%**	**48.05**	**81.70**	**70%**

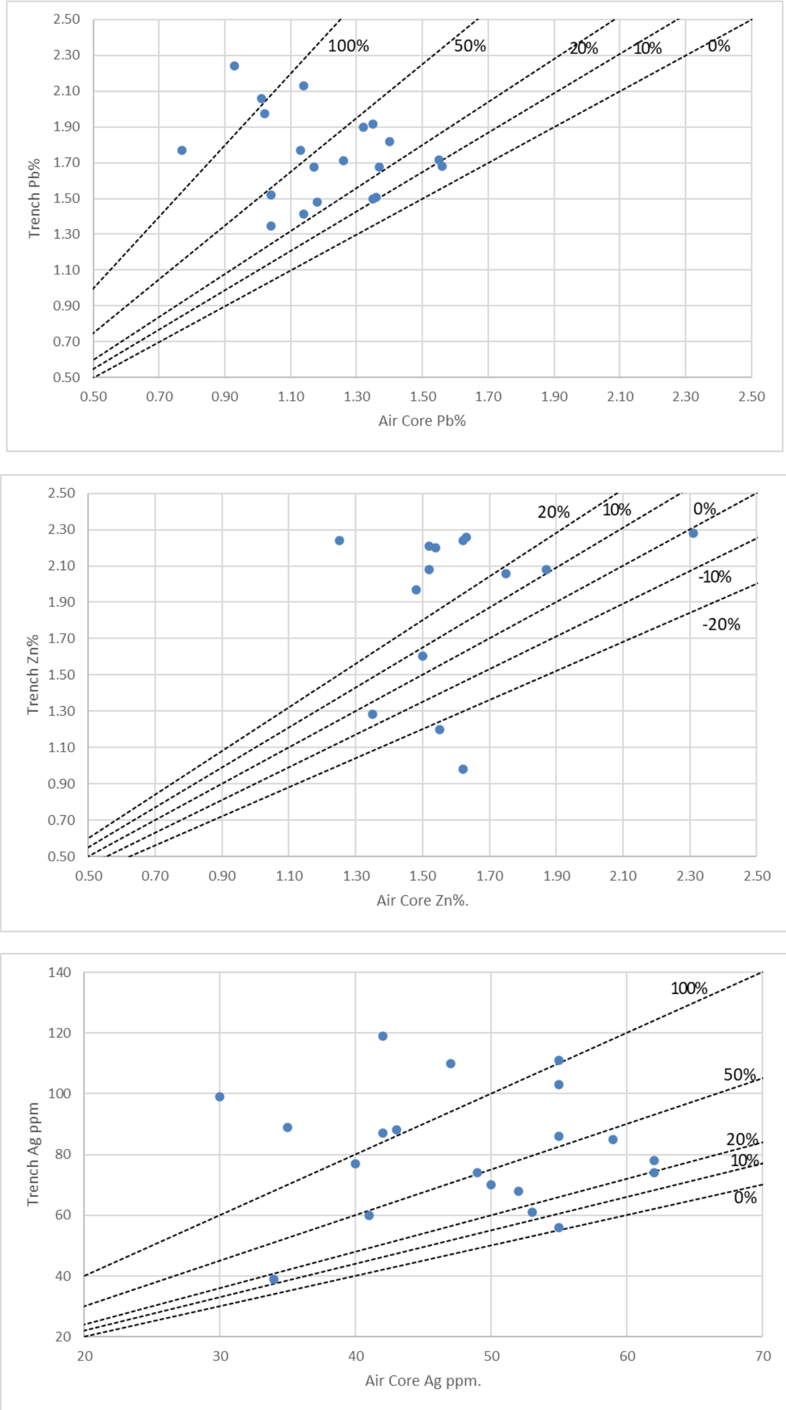

Figure 14: Air Core v Trench Assays

A program of 20 push tube holes were drilled at the same locations as the trenches, within 0.5m of the original air core collars. Samples were sent to ALS Orange for assay. Results of the push tube sample assays, composited to the same intervals as the air core assays, are shown in **Table 11** and **Figure 15**. The results show overall increase in grades for Zn, Pb and Ag, with differences up to 78% for Pb, 90% for Zn and 112% for Ag. Further investigation has ascertained that the magnitude of the differences for each element do not corelate with any particular holes or areas of the TSF.

Table 11 – Air Core & Push Tube Assay Comparisons

DHID		Composite Interval Depth (m)	Pb %			Zn %			Ag ppm		
AirCore	Push Tube		AirCore	Push Tube	% Diff	AirCore	Push Tube	% Diff	AirCore	Push Tube	% Diff
S1_004	S1_PD01	0-2	1.52	1.69	11%	1.04	1.23	19%	53	51	-3%
S1_029	S1_PD02	0-2	2.31	2.92	26%	1.56	1.70	9%	62	83	34%
S1_049	S1_PD03	0-2	1.92	2.13	11%	1.35	1.65	22%	62	84	35%
S1_069	S1_PD04	0-2	1.50	1.91	28%	1.02	1.39	37%	49	59	21%
		2-4	1.71	2.33	36%	0.95	1.79	89%	59	97	65%
S1_091	S1_PD05	0-2	1.48	1.75	18%	1.13	1.36	20%	40	58	44%
S1_111	S1_PD06	0-2	1.66	2.27	37%	1.36	1.64	20%	55	77	40%
S1_128	S1_PD07	0-2	1.40	2.20	57%	1.01	1.85	83%	35	74	112%
S1_125	S1_PD08	0-2	1.60	1.93	20%	1.55	1.60	3%	55	78	41%
		2-4	1.25	1.54	23%	0.81	1.22	50%	37	68	82%
S1_107	S1_PD09	0-2	1.62	2.01	24%	1.35	1.97	46%	42	78	85%
		2-4	2.05	2.38	16%	1.52	1.59	4%	60	85	42%
S1_087	S1_PD10	0-2	1.25	1.70	36%	0.77	1.46	90%	30	57	90%
		2-4	2.28	2.61	14%	1.04	1.66	59%	61	91	49%
S1_065	S1_PD11	0-2	1.62	1.64	1%	0.93	1.56	67%	42	58	38%
		2-4	2.02	1.95	-3%	1.23	1.60	30%	69	91	31%
		4-6	1.71	3.05	78%	0.98	1.63	66%	66	102	54%
S1_045	S1_PD12	0-2	1.52	1.87	23%	1.14	1.29	13%	41	49	20%
		2-4	2.45	2.37	-3%	1.61	1.63	1%	72	90	25%
		4-6	2.20	2.72	24%	1.36	1.79	32%	72	95	32%
S1_025	S1_PD13	0-2	1.54	1.81	18%	1.17	1.48	27%	43	74	72%
		2-4	1.77	1.79	1%	1.64	1.55	-5%	75	91	21%
		4-6	1.99	2.80	41%	1.87	1.94	4%	83	102	23%
S1_009	S1_PD14	0-2	1.63	2.32	42%	1.37	1.72	26%	55	76	38%
		2-4	1.96	2.06	5%	1.60	1.58	-2%	70	86	23%
S1_005	S1_PD15	0-2	2.85	2.39	-16%	1.18	1.68	42%	52	73	41%
		2-4	2.20	1.84	-16%	1.60	1.42	-11%	72	85	18%
S1_022	S1_PD16	0-2	1.55	1.62	4%	1.40	1.40	0%	50	57	15%
		2-4	2.25	2.48	10%	1.51	1.65	9%	69	83	20%
		4-6	2.07	2.07	0%	1.70	1.64	-4%	73	102	39%
S1_042	S1_PD17	6-8	2.47	2.57	4%	1.58	1.83	16%	82	100	22%
		0-2	1.35	1.57	16%	1.04	1.22	17%	34	39	15%
		2-4	2.14	2.02	-6%	1.42	1.68	18%	63	85	35%
S1_062	S1_PD18	4-6	1.93	2.05	6%	1.38	1.58	14%	67	112	67%
S1_084	S1_PD19	6-8	1.95	2.76	41%	1.52	1.95	29%	74	105	42%
		0-2	1.60	1.92	20%	1.32	1.47	11%	47	64	36%
S1_104	S1_PD20	0-2	1.87	2.08	11%	1.26	1.78	41%	59	75	26%
Average			**1.85**	**2.13**	**17%**	**1.31**	**1.60**	**26%**	**58**	**79**	**40%**

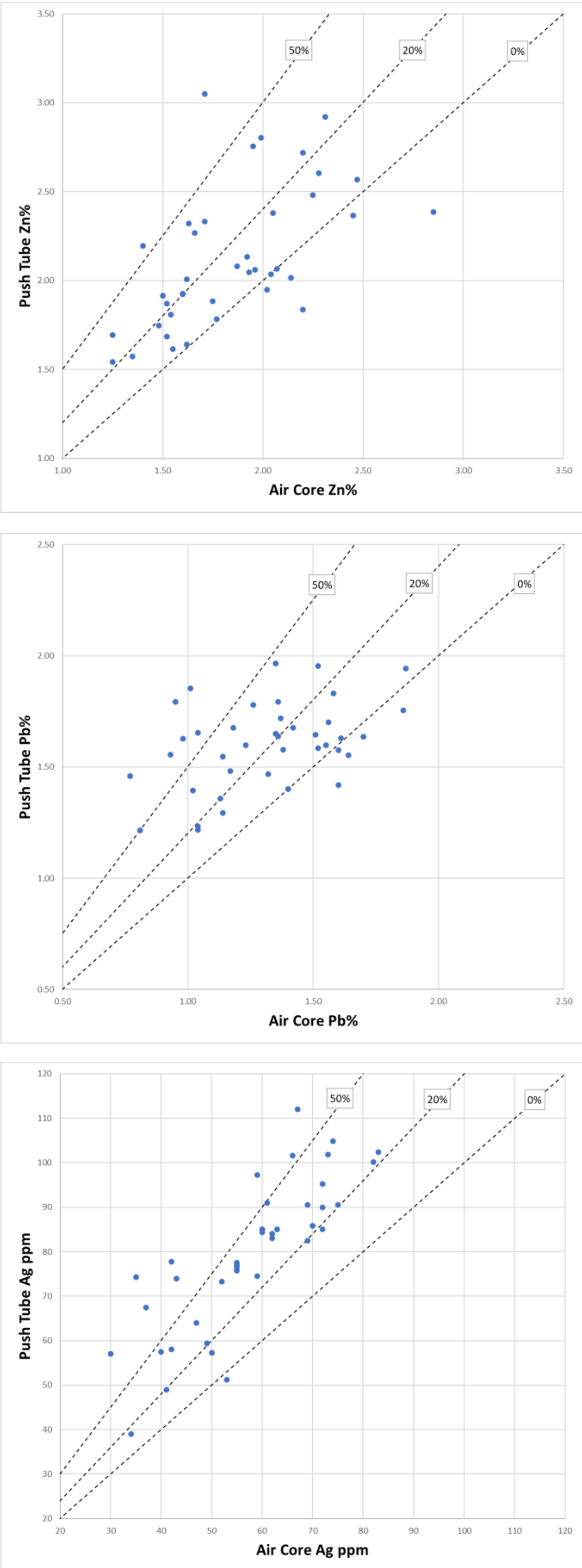

Figure 15: Air Core v Push Tube Assays

5.4 Data Quality Summary

Review of the database veracity, including data quality, has identified issues with the 2014 air core drilling program results. Out of range results of CRM Pb assays undertaken at the EOPL laboratory on site, and discrepancies in check assays, have reduced the confidence in Pb grades reported by the on site laboratory from the 2014 drilling. Subsequent trenching and push tube drilling in 2015 highlighted further issues with a possible under calling of Pb, Zn and Ag grades.

These problems with grades from the 2014 drilling program appear to stem from a combination of laboratory performance and drilling method. When air core drilling saturated tailings, there is a likelihood of tailings material sticking to the inside of the drill string and air hoses leading to the cyclone. The cyclone can be cleaned out after every metre, but there still exists potential for sample contamination from the hoses etc.

Therefore the results from the 2014 drilling program are not considered suitable for use in Resource estimates of the tailings, and only results from the 2015 drilling program will be utilised.

6 Geological Interpretation and Modelling

6.1 Introduction

Based on all available geological and grade information, suitable material boundaries have been interpreted and wireframes constructed to constrain grade estimation for the material in TSF Sector 1. Interpretation and digitising of constraining boundaries have been undertaken on cross sections as well as horizontal plans. The resultant digitised boundaries have been used to construct wireframe surfaces or solids defining the 3-D geometry of TSF Sector 1.

Construction of the physical domains were carried out using the interactive modelling facilities in the Surpac mining software package. All modelling work was completed in local mine coordinates.

6.2 Material Boundaries

The main features of TSF Sector 1 were modelled using drill hole data and detailed surface surveys. A surface representing the bottom of the tailings deposit was modelled based on 10 drillholes from the 2014 drilling program that were designed to penetrate the TSF floor (**Figure 16**). The tailings surface was surveyed using aerial photogrammetry from which a surface DTM model was created.

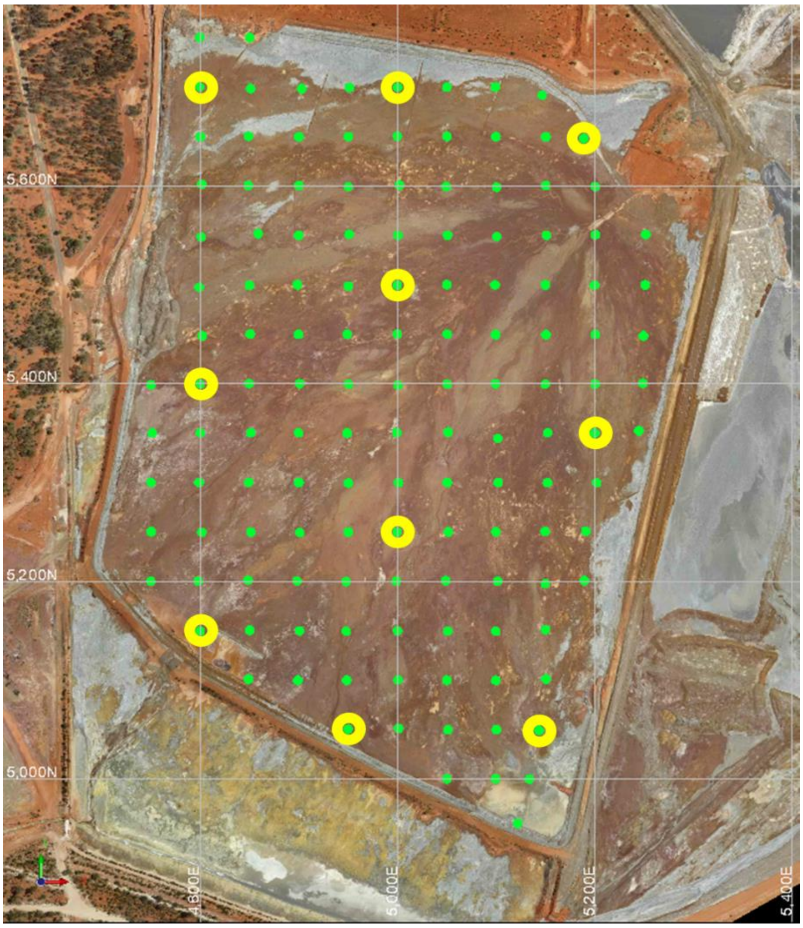

Figure 16: 2014 Drilling – TSF Floor Intercept Holes

A lateral boundary of the tailings deposit was created to constrain the estimation and reporting process as shown in **Figure 17**. The boundary was kept within the bunded walls of the TSF and away from the area in the north where sludge from the Cockle Ck smelter Primary Electrostatic Mist Precipitator (PEMP) is stored.

Figure 17: TSF Sector 1 Tailings Resource Boundary

7 Metallurgical Review

A number of metallurgical test work programs have been conducted on TSF Sector 1 tailings at the Endeavor Mine based on samples collected during the 2015 and 2017 drilling programs.

In 2015 a 200kg sample was subject to mineralogical analysis and flotation testing to recover a zinc concentrate. The composite was assayed with the result shown in **Table 12**.

Table 12 – 2015 Composite Head Assays.

	Cu(%)	Pb(%)	Zn(%)	Fe(%)	S(%)	Ag (g/t)	Au (g/t)
Value	0.14	1.6	2.05	33.9	31.2	86.0	0.6

The sample was noted as being 54% pyrite and 4.5% sphalerite. Galena was oxidised to sulphate forms. Oxidation tests determined that 33% of the lead and 1.6% of the zinc is oxidised with this portion not recoverable by flotation.

The best locked cycle test result was 47.7% Zn concentrate at 56% overall recovery.

Six composites were collected in 2017, one global and five regional, with assay results as shown in **Table 13**.

Table 13 – 2017 Composite Head Assays.

Composite	Cu(%)	Pb(%)	Zn(%)	Fe(%)	S(%)	Ag (g/t)	Au (g/t)
Global	0.14	1.6	2.16	34	30	80	0.62
GV_A	0.14	1.2	1.72	28	24	59	0.48
GV_B	0.15	1.67	2.4	34	29	81	0.36
GV_C	0.14	1.55	2.06	34	31	80	0.58
GV_D	0.16	1.48	1.94	34	30	66	0.57
GV_E	0.14	1.62	2.03	35	39	77	0.62

Locked cycle flotation test work determined that 50% of the contained zinc can be recovered to a 50% Zinc concentrate.

Recent 2023 test work at ALS Burnie on a limited number of samples collected by hand auger, has replicated this result and AMC (2023) recommends achievable reprocessing performance as 46% Zn recovery at a 50% Zn grade

Recent test work on the Sector 1 tailings at ALS Burnie has also resulted in a 62.1% to 64.7% silver recovery to concentrate. With these promising results from the preliminary tests, a future test work programme will be completed to confirm final lead and silver recovery expectations. Recoveries of lead and silver from Sector 1 Tailings have been conservatively estimated at 30% Pb and 40% Ag recovery at a 50% Pb grade.

8 Statistical Analysis

8.1 Introduction

Statistical analysis was undertaken based on composited datasets of the lead, zinc and silver assays. The activities completed in this phase of the study were as follows: -

- Determination of a suitable composite length.
- Compositing of the drill hole data to lengths within the coded domain intervals.
- Compilation of descriptive statistics and histogram plots of the composite data sets.

8.1.1 Sample Length Analysis and Compositing

In compositing to an appropriate regular downhole length, the aim is to: -

- Achieve uniform sample support.
- Reduce the impact of random variability; and
- Minimise the effect of averaging samples of a skewed distribution.

Note, however, that equalising sample length is not the only criteria for standardising sample support. Factors such as angle of intersection of the sampling to mineralisation, sample type and diameters, drilling conditions, recovery, sampling/sub-sampling practices and laboratory practices all effect the 'support' of a sample. Composites are generated downhole at the nominated interval within domain boundaries with length used to weight each contributing sample in calculating the composite grade.

As the vast majority of raw sample lengths from the 2015 drilling program were 1.5 metre or less, composites were generated using a nominal length of 2 metres using the best fit method in Surpac.

8.1.2 Statistical analysis of Composite Data

Detailed statistical analysis of the composite assay data was conducted. Descriptive statistics for the composites are presented in **Table 14**.

Table 14 – Summary Composite Statistics

	Pb(%)	Zn(%)	Ag (g/t)
Count	49	49	49
Minimum	1.01	1.47	44
Maximum	1.95	3.04	115
Mean	1.56	2.10	78
Median	1.60	2.05	78
Standard Deviation	0.21	0.38	15
Coefficient of Variation	0.13	0.18	0.20

8.2 Bulk Density Analysis

As mentioned in Section 4.8 dry bulk density measurements were undertaken on 551 air core samples. Statistics of the results are shown in **Table 15**.

Table 15 – Summary Density Statistics

Number	Minimum	Maximum	Mean	Median	Std Deviation
551	1.08	2.82	**1.74**	1.70	0.31

A tailings dry density value of 1.74 t/m^3 has been adopted for this Resource estimate.

A report by Golder (2018) stated the average dry density of tailings deposited between May 2017 and April 2018 was 2.0 t/m^3. This value was estimated using tailings beach surveys and the dry weight of tailings deposited during this period. Therefore, a value of 1.74 t/m^3 could be considered conservative.

8.3 Spatial Analysis

8.3.1 Introduction

Variography is used to describe the spatial variability or correlation of an attribute. The spatial variability is traditionally measured by means of a variogram, which is generated by determining the averaged squared difference of data points at a nominated distance (h), or lag. The averaged squared difference (variogram or γ(h)) for each lag distance is plotted on a bivariate plot where the X-axis is the lag distance and the Y-axis represents the average squared differences (γ(h)) for the nominated lag distance.

Fitted to the determined experimental variography is a series of mathematical models which, when used in the kriging algorithm, will recreate the spatial continuity observed in the variography.

Surpac software has been employed to generate and model the variography. The rotations are reported as input for grade estimation, with X (rotation around Z axis), Y (rotation around Y`) and Z (rotation around X``) also being referred to as the major, semi-major and minor axes.

8.3.2 Grade Variography

Variography was completed for the TSF Sector 1 composites.

The variogram model has direction of maximum continuity which is horizontal and isotropic, with a short vertical range for the minor direction.

The modelled variography for Zn displays variability that is comprised of a moderate (15%) relative nugget and an overall range of 270m. The models for Pb and Ag were very similar to that of Zn so the Zn model was adopted for all metals.

The fitted variogram model is presented in **Table 16** and the variogram plot displayed in **Figure 18**.

Table 16 – Summary Variogram Model

Domain	Metal	Nugget	Structure	Sill	Azm°	Plunge	Dip°	Major	Semi	Minor
TSF Sector1	Zn, Pb, Ag	0.15	Spherical	0.85	0	0	0	270	270	5

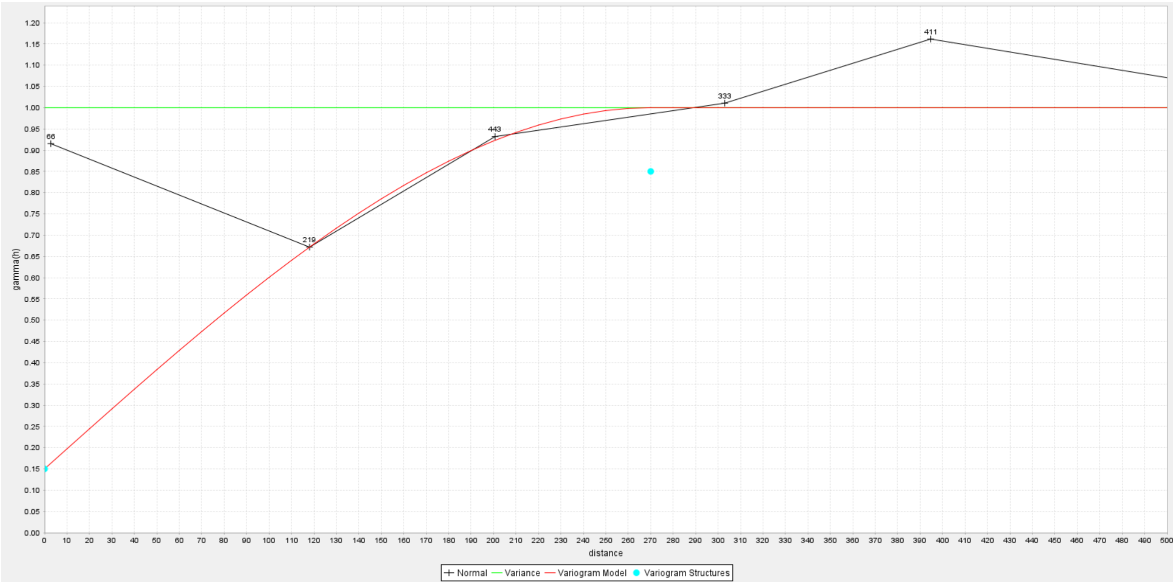

Figure 18: Variogram Model

9 Block Model Development

9.1 Introduction

A three-dimensional block model was constructed for TSF Sector 1 using Surpac mining software, in preparation for undertaking resource estimation. The block models contain sufficient variables to record the results of grade estimates and other required parameters.

9.2 Block Model Construction Parameters

Table 17 summarises the extents of the block model. The block model was developed using block dimensions that took into consideration data spacing and mining constraints. The block model was also sub-blocked to provide accurate reproduction of the domain wireframe volumes.

Table 17 – Block Model Parameters

	Y	X	Z
Minimum Coordinates	4925	4725	10198
Maximum Coordinates	6025	5325	10218
Parent Block Size	50	50	2
Sub Block Size	6.25	6.25	0.25

9.2.1 Block Model Attributes

A series of attributes were incorporated into the block model for recording variables assigned and calculated throughout development of the block model and during grade estimation. A list of the attributes contained within the final block models are displayed in **Table 18**. Intermediate variables utilised for validation and classification such as number of samples, distance to samples, kriging variance, slope of regression etc. were removed in the final block model to reduce the size of the file.

Table 18 – Block Model Attributes

Attribute	Default	Description
Ag	-99	Estimated Ag grade (ppm) Ordinary Kriging
Pb	-99	Estimated Pb grade (ppm) Ordinary Kriging
Zn	-99	Estimated Zn grade (ppm) Ordinary Kriging
Domain	waste	Air, tails, waste
Density	1.74	Average dry bulk density from 2014 drilling

9.2.2 Block Model Validation

The block model was extensively validated against the domain model wireframes. The model has been validated by viewing in multiple orientations using the 3-D viewing tools in Surpac. Based on the visual review, and reproduction of the wireframe volumes, the block model was considered a robust representation of the interpreted mineralised domains.

10 Grade Estimation

10.1 Introduction

Grade estimation was undertaken using Ordinary Kriging (OK) as the estimation methodology for, Pb, Zn, and Ag within the tailings domain of TSF Sector 1.

OK is one of the more common geostatistical methods for estimating the block grade. In this interpolation technique, contributing composite samples are identified using a search volume applied from the centre of each block. Weights are determined so as to minimise the error variance considering both the spatial location of the selected composites and the modelled variogram. Variography describes the correlation between composite samples as a function of distance and direction. The weighted composite sample grades are then combined to generate a block estimate and variance.

10.2 Search Neighbourhood and Grade Estimation

Search ellipse orientations and distances were determined based on variogram orientation, variogram model anisotropy and ranges, horizon geometry and data distribution.

A multiple search strategy was undertaken in obtaining the estimates using the results of a search neighbourhood analysis. **Table 19** provides the sample search parameters applied for each estimation pass.

Block discretisation was carried out on a 3 x 3 x3 basis, for a total of 27 discretisation points per whole block estimate.

The estimates were completed using Surpac mining software. In estimating grade, the standard fields relating to the search neighbourhood used, number of composites selected, the distance to the nearest composite, the average distance of composites, the number of drill holes from which the selected composites were derived, kriging variance, and slope of regression were recorded.

The resultant grade estimates are held in the model file ***tsf_s1_2023.mdl***.

Table 19 – Grade Interpolation Search Parameters – Ordinary Kriging

Metal	Search Ellipse (deg)			Est Run	Search Ellipse (m)	Max Vertical Search (m)	Samples Accessed	
	Bearing	Plunge	Dip				Min	Max
Pb, Zn, Ag	0	0	0	1	270	4	3	12
				2	500	4	2	10

10.2.1 Validation

Validation of the estimate was completed and included both interactive and statistical review. The validation methods included: -

- A visual comparison of the input data against the block model grade in plan and cross section.
- Comparison of global statistics.
- Swath plots, comparing the composite grade and the estimated grade grouped by intervals in plan and section.

The visual assessment of block model grades compared to drill hole grades (**Figure 19**) did not highlight any issues. Block grades display good correlation with nearby composite grades and acceptable representation of interpreted grade continuity.

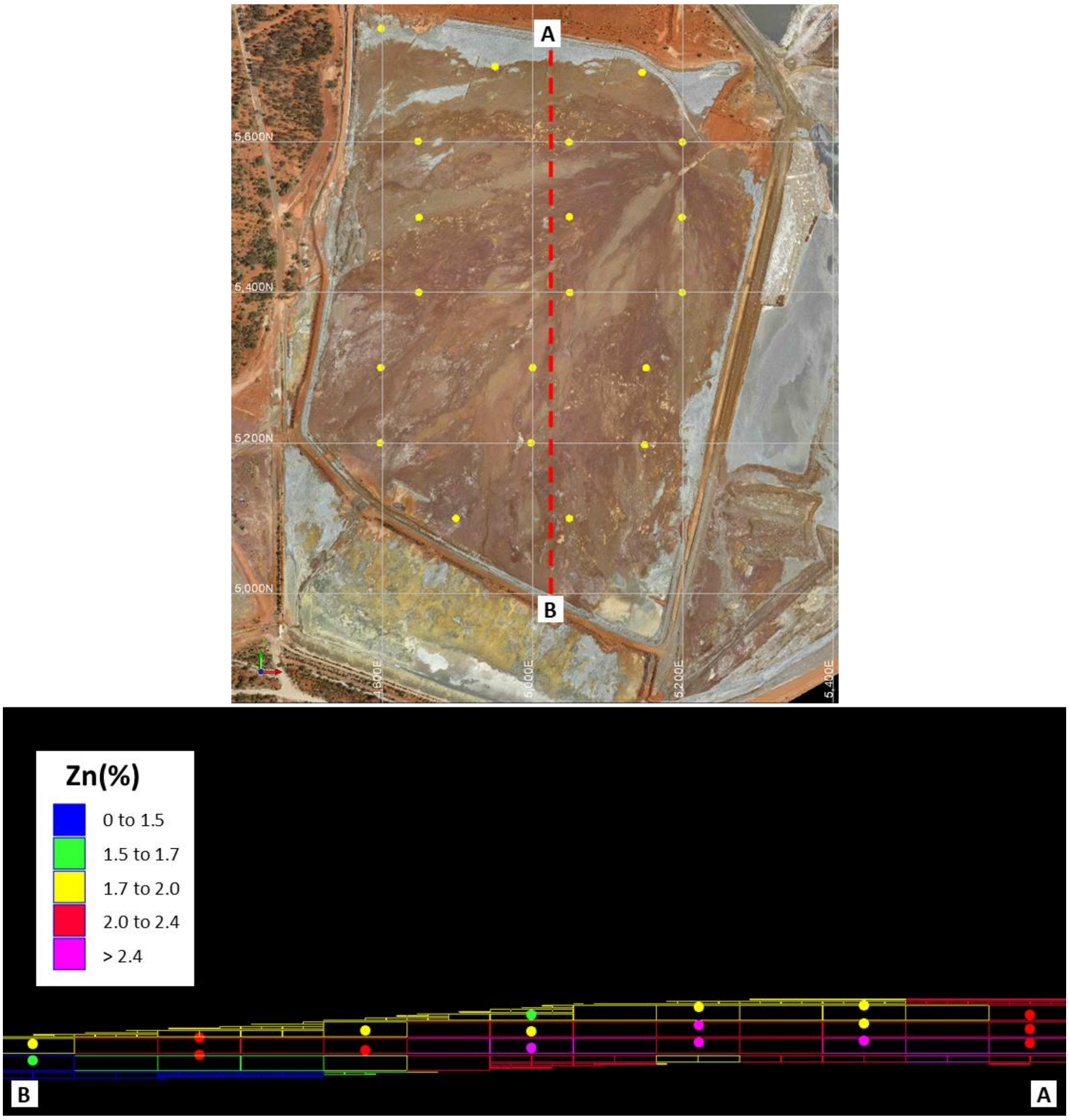

Figure 19: Block Model Validation – Cross Section (5025mE) – Blocks and Composites (5 x vertical exaggeration).

A comparison between the raw composite, composite and volume weighted block model grades are shown in **Table 20**. The table shows the block model grades are comparable to the composite data.

Table 20 – Composite v Block Model Mean Grades

Metal	Raw Composite Mean	Block Model Weighted Mean	% Difference
Zn	2.10	2.12	1.1%
Pb	1.56	1.55	-0.5%
Ag	78	79	1.2%

The local estimates were reviewed by graphing summary statistics of composite and block grades on 100m spaced Northing vertical slices (swath plots). The analysis of swath plots (**Figure 20**) demonstrates that the grade variability in composites (blue lines) is greater than that of grade estimates (green lines) which is the smoothing effect of the OK estimate. The directional trends observed in composites are reproduced within the block estimates. Acceptable levels of reproducibility are noted between the input composites data and the block estimates based on visual review.

On this basis and the other validation checks, the whole block estimates are appropriate and robust.

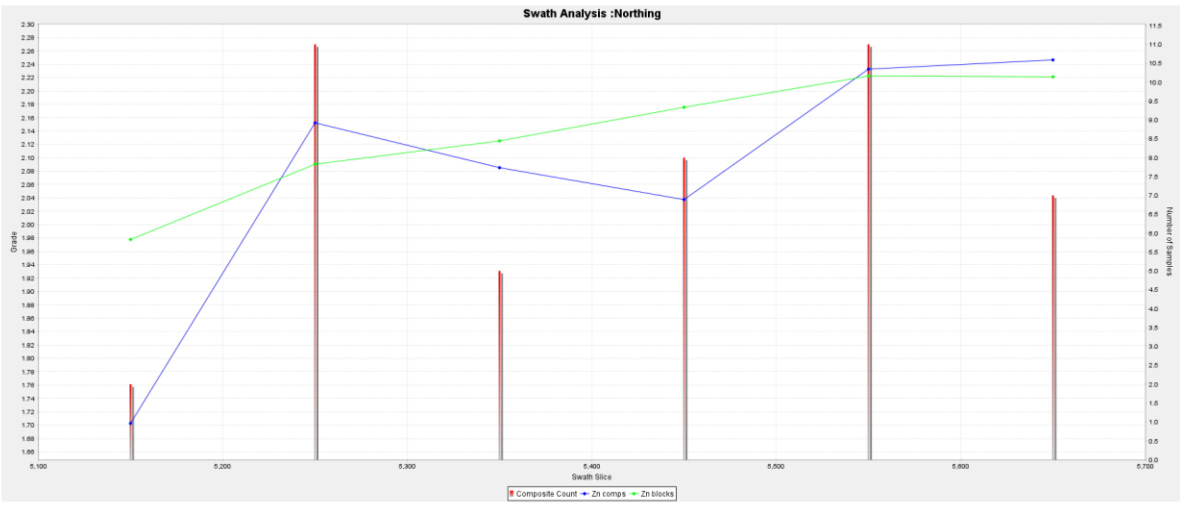

Figure 20: Block Model Validation – Swath Plots – Blocks and Composites 100m Slices – Northing.

11 Mineral Resource Reporting

11.1 Introduction

The Resource estimate has been classified as Indicated and Inferred Mineral Resources in accordance with guidelines as set out in the Joint Ore Reserves Committee (JORC) Code (2012). Resource categories have been defined using definitive criteria determined during the validation of the grade estimates, with detailed consideration of the JORC Code categorisation guidelines.

11.2 Resource Categorisation

The key parameters considered during the resource categorisation are as follows: -

- Geological knowledge and interpretation.
- Deposit style.
- Confidence in the sampling and assay data.
- Spacing of the exploration data.
- Variogram model ranges in relation to the local data spacing and the estimation variance.
- Prospects for eventual economic extraction.

The exploration data used for the TSF Sector 1 Resource estimate is robust and appropriate for resource estimation purposes, with the current data spacing sufficient to generate robust grade estimates.

The categorisation of tailings grade and tonnage estimates benefits from knowledge of the source, amount and properties of the tailings that have been deposited, which are obtained from processing records. This data can be used for reconciliation and increase the confidence in the estimates. **Table 21** displays the estimated tonnes and grade of tailings in TSF Sector 1 from processing records compared to the global tonnes and grade from the block model estimate. This table shows the two groups of data compare favourably. These comparisons help to improve confidence in the block model estimates.

Table 21 – Comparison of Processing Data v Block Model Estimates

Data Source	Dry Tonnes	Grades		
		Zn (%)	Pb (%)	Ag (ppm)
Processing Data	5.4 Mt	2.11	1.76	83
Block Model Estimate	5.2 Mt	2.12	1.55	79

Another check of the block model estimates can be achieved from comparing block grades to assays of global metallurgical samples collected from the same area. A comparison is shown in **Table 22** which shows the block grades compare favourably to the grades of the global composites.

Table 22 – Comparison of Global Metallurgical Sample Grades v Block Model Estimates

Data Source	Grades		
	Zn (%)	Pb (%)	Ag (ppm)
Block Model Estimate	2.12	1.55	79
2015 Sampling	2.05	1.60	86
2017 Sampling	2.16	1.60	80

The tailings deposited in TSF Sector 1 have reasonable prospects for eventual economic extraction for the following reasons:

- Proximity to an existing flotation plant (currently under care and maintenance) for processing.
- Metallurgical test work indicates economic recoveries for Zn, Pb and Ag.

Based on the consideration of items listed above, and review of the resource block model estimate quality, classification criteria were determined as summarised in the following: -

- **Indicated**
    - Blocks in the tailings domain that occur between drill holes or no more than 50m from a drill hole.
- **Inferred**
    - All remaining blocks in tailings domain not assigned as Indicated.

A plan displaying the areas of Indicated and Inferred Resources is displayed in **Figure 21**.

The key criteria that were considered during resource classification are presented in JORC Table1 in **Attachment 1.**

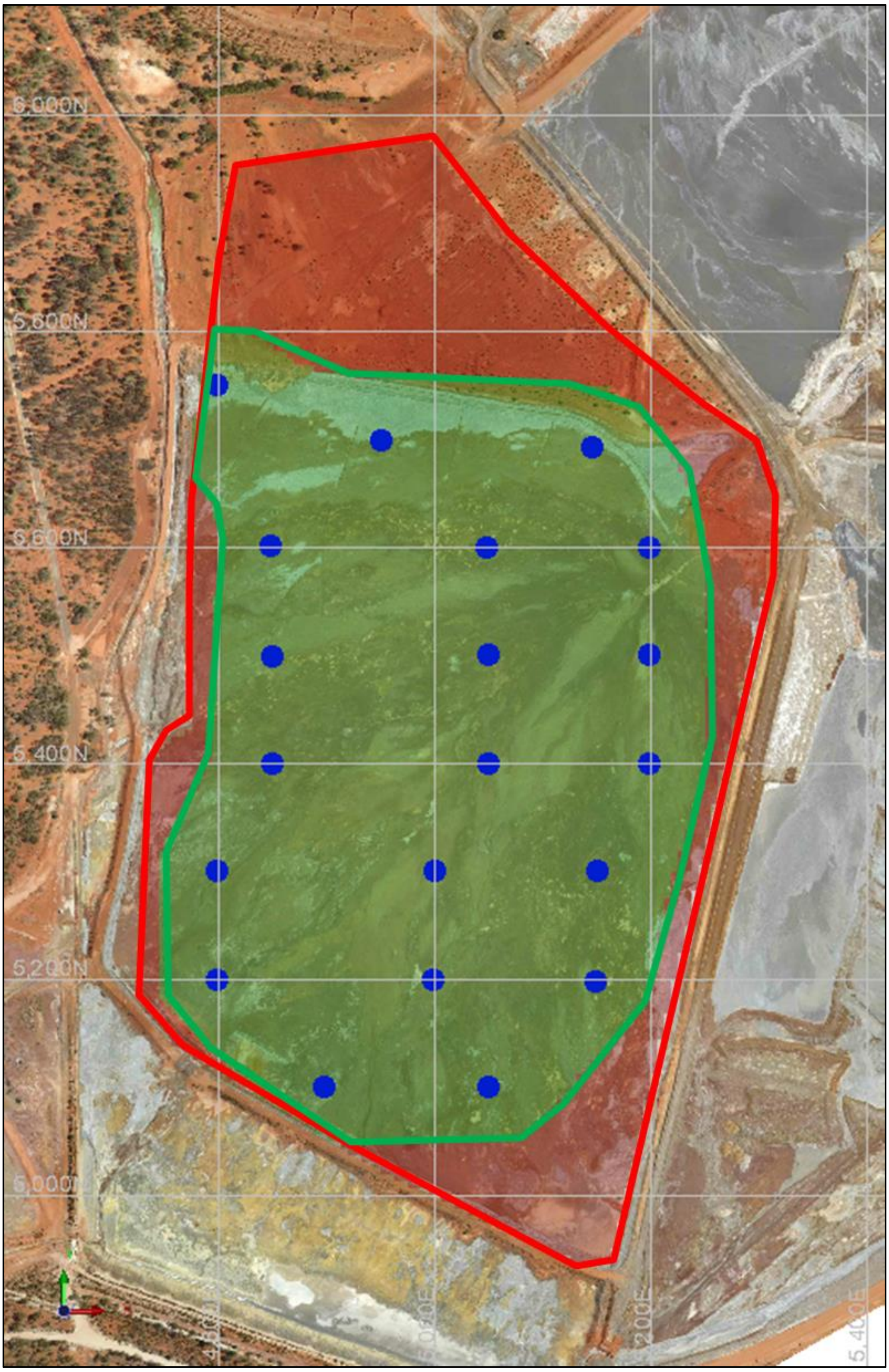

Figure 21: Plan showing Indicated (green) and Inferred (red) Resources..

11.3 Mineral Resource Statement

The Mineral Resource Statement for the Endeavor Mine TSF Sector 1 Mineral Resource Estimate, based on information available as at October 2023, reported with no cut off grade, is presented in **Table 23**.

Table 23 – Endeavor Mine TSF Sector 1 Mineral Resource October 2023

Category	Mt	Zinc (%)	Lead (%)	Silver (g/t)
Indicated	3.6	2.14	1.56	80
Inferred	1.6	2.07	1.53	77
Total[1]	**5.2**	**2.12**	**1.55**	**79**

1. Discrepancies may occur due to rounding

The Mineral Resource has been reported without the use of a cut off grade as the proposed mining method (hydro mining) would not allow for efficient selective mining, requiring the entire tailings domain to be mined.

12 Competent Persons Statement

The Mineral Resources Estimate Report for the Endeavor Mine TSF Sector 1 has been compiled in accordance with the guidelines defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' (2012 JORC Code).

The information in this report that relates to Exploration Results and Mineral Resources is based on information compiled by Troy Lowien, a Competent Person who is a Member of The Australasian Institute of Mining and Metallurgy. Troy Lowien is employed by Polymetals Resources Ltd.

Troy Lowien has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'. Troy Lowien consents to the inclusion in the report of matters based on his information in the form and context in which it appears.

Troy Lowien has visited the Endeavor Mine on numerous occasions since 2010.

13 References

AMC Consultants. 2023. Endeavor PFS Metallurgy Support. *Report prepared for Polymetals Resources Ltd Ltd.*

David V. 2008. Structural-geological setting of the Elura Zn-Pb-Ag massive sulphide deposit, Australia. *Ore Geology reviews, 34, 428-444*

Golder Associates. 2018. Life of Mine tailings storage assessment. *Report prepared for Endeavor Operations Pty Ltd.*

ATTACHMENTS

Attachment 1

JORC Code (2012) Table 1

JORC Code, 2012 Edition – Table 1

Section 1 Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.* • *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* • *Aspects of the determination of mineralisation that are Material to the Public Report.* • *In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.*	• 2014 Drilling – Air core drilling was used to obtain 1m samples from which 2m composite samples were created for assay by acid digest. • 2015 Drilling – Push tube drilling was used to obtain an average sample length of 1.2m from which sub samples were collected for assay by acid digest. • 2017 Drilling - Push tube drilling was used to obtain an average sample length of 1m from which various composites were created for metallurgical test work.
Drilling techniques	• *Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	• 2014 Drilling - Aircore methods on where a 100mm cutting bit with a hollow centre is pushed through unconsolidated material using rotation. Air is pumped through an annulus between the inner and outer tubes of the drill string and out through orifices in the cutting head. Sample is returned up the centre of the drill string and collected in a cyclone. • 2015 and 2017 Drilling - Push tube methods where casing is advanced down the hole and a solid "core" of unconsolidated material is extracted from within the casing encased in a rigid plastic sleeve.
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.* • *Measures taken to maximise sample recovery and ensure*	• No recovery information is available. • During the 2014 air core drilling program the sample collection cyclone was vigorously cleaned after each 1m interval to ensure

Criteria	JORC Code explanation	Commentary
	representative nature of the samples. • *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	complete sample recovery.
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.* • *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.* • *The total length and percentage of the relevant intersections logged.*	• Detailed logging of the tailings is considered impractical and unnecessary as the tailings have been homogenised from processing and deposition. Material changes were noted when drill holes intersected the base of the tailings dam
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.* • *If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.* • *For all sample types, the nature, quality and appropriateness of the sample preparation technique.* • *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.* • *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.* • *Whether sample sizes are appropriate to the grain size of the material being sampled.*	• During the 2014 air core drilling, 2m composites were taken from 1m samples intervals by spear method., as the material was too puggy for a riffle splitter. • Push tube samples were split laterally down the hole with one side used to create metallurgical sample composites and the other side for assay. • Sample preparation was carried out at the onsite laboratory for the 2014 program and ALS Orange for the 2015 program. Sample preparation of the metallurgical composites was carried out at ALS Burnie. • Field duplicate sampling results indicate no issues with the methods used for collection of sub samples. • Sample sizes are appropriate for the grain size of the material being sampled.
Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.* • *For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.* • *Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	• 2014 Drilling - Samples were assayed at the Endeavor laboratory using an Aqua Regia digest with atomic absorption spectrometry (AAS) for lead, zinc, silver, iron and copper analyses. • 2015 Drilling – Samples were sent to ALS-Orange were assayed by an Aqua Regia digestion using AAS (ICP-AES) analysis for lead, zinc, silver, iron and copper. The prepared sample is digested in 75% aqua regia for 120 minutes and after cooling, the resulting solution is diluted to volume (100mL) with de-ionised water, mixed and then analysed for inductively coupled plasma-atomic emission spectrometry or by atomic absorption spectrometry.

Criteria	JORC Code explanation	Commentary
		• Assay techniques are considered total and appropriate for the mineralisation style. • The quality control regime used in the 2014 drilling program consisted of Certified Reference Material (CRM) and Blanks inserted into the sample stream, field duplicate samples, and re-assays of laboratory pulp samples. The insertion rate of QC samples into the submission stream was 1 in 6 samples. • The quality control regime used in the 2015 drilling program consisted of CRM and Blanks inserted into the sample stream at a rate of about 1 in 10 samples. However, these samples were not assayed at the laboratory due to insufficient sample quantities according to the results certificate. Instead, assay accuracy and precision were assessed based on CRM and pulp duplicates inserted in the sample stream by the laboratory. • No recorded quality control samples were included in the submission of the 2017 samples to the metallurgical laboratory. • Assessment of the QC data from the 2014 drilling indicate acceptable levels of precision but an issue with the accuracy of Pb assays, showing a significant bias to lower grades. • Acceptable levels of precision and accuracy have been established for the 2015 assays.
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.* • *The use of twinned holes.* • *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.* • *Discuss any adjustment to assay data.*	• There are no records of independent or alternative verification of significant intersections. • The 2015 drill holes were drilled as twins of selected holes from the 2014 program. The results show overall increase in grades for Zn, Pb and Ag, up 112%. Further investigation has ascertained that the magnitude of the differences for each element do not corelate with any particular holes or areas of the TSF. This indicates an issue with the 2014 sample representivity and therefore have been rejected for use in resource estimation. • The geology department kept written procedures for data collection and storage. A user manual was written for the use of the Drilling Management system (MS Access Database). • The Competent Person is not aware of any adjustment to assay data.

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td>Location of data points</td><td>Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.
Specification of the grid system used.
Quality and adequacy of topographic control.</td><td>Drill hole collars were surveyed by the mine surveyor by unknown methods.
There were no downhole surveys undertaken on the drill holes. All holes were drilled vertically and were relatively short (<15m depth), and therefore any downhole deviation would have negligible effects on the location of datapoints.
The Endeavor Mine is situated within Zone 55 of the MGA94 grid coordinate system. A local mine grid was established for the site. All drill hole and undergound development survey data was collected using this local grid.
The MRE estimate uses the local mine grid, which relates to MGA94 using the following transform:
<table>
<tr><th></th><th></th><th>MGA94</th><th>Local Mine Grid</th></tr>
<tr><td rowspan="2">Point 1</td><td>Northing</td><td>6551419.471</td><td>6451.175</td></tr>
<tr><td>Easting</td><td>372517.808</td><td>5231.564</td></tr>
<tr><td rowspan="2">Point 2</td><td>Northing</td><td>6551409.739</td><td>6452.863</td></tr>
<tr><td>Easting</td><td>371884.310</td><td>4597.827</td></tr>
<tr><td colspan="2">Elevation Correction</td><td colspan="2">+10,000</td></tr>
</table>
An aerial photogrammetry survey was carried out over the site in December 2015 by Arvista Pty Ltd at a ground resolution of 5cm per pixel. A Digital Terrain Model (DTM) in Surpac format was supplied and used in this study.</td></tr>
<tr><td>Data spacing and distribution</td><td>Data spacing for reporting of Exploration Results.
Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.
Whether sample compositing has been applied.</td><td>Drilling density is on a notional 50m x 50m grid with those holes used in the resource estimate on 100m x 200m grid. Down hole sampling intervals were on average around 1m in length.
The data spacing and distribution is sufficient to establish grade continuity appropriate for the Indicated Resource estimation category after all other confidence factors are applied.
Sample composites of 2m were used in the MRE.</td></tr>
</table>

Criteria	JORC Code explanation	Commentary
Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.* • *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	• Tailings were deposited sub-aerially forming beaches with a slight slope towards the perimeter of the storage facility. Therefore, any grade variations over time will be represented by sub-horizontal layering. Drilling of vertical drill holes ensures sampling is undertaken as close as possible orthogonal to the direction of maximum grade continuity.
Sample security	• *The measures taken to ensure sample security.*	• All samples were collected and sub-sampled on site by company staff. Samples were either submitted to an internal on site laboratory or off site laboratory. • Samples were collected and placed in numbered and ticketed calico bags that were securely fastened. A dedicated geologist and field assistant were in attendance at all stages of drilling.
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	• There are no records of any audits or reviews of the sampling techniques or data.

Section 2 Reporting of Exploration Results

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.* • *The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	• The project is located within granted Exploration Licence EL5785 Mining leases ML158, ML159, ML160, ML316, ML161, and ML930 with the earliest expiry date of 12 March 2028. The leases are held by Cobar Operations Pty Ltd. • Metalla Royalty and Streaming Ltd have a royalty based a flat rate of 4% on payable Pb, Zn and Ag.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	• The tailings in Sector 1 were drilled in 2014, 2015 and 2017 by CBH Resources. The drilling was undertaken by standard methods and the results used to generate an approximate tonnage and grade • Exploration appears to have been performed to industry standards.
Geology	• *Deposit type, geological setting and style of mineralisation.*	• Mineralised material in the tailings storage facility consists of clay to fine sand sized particles deposited in sub-horizontal layers from centrally located outflow sites. The particles contain remnant sulphides that were not captured during processing of the Endeavor Mine silver-zinc-lead ore. • The primary lead and zinc bearing minerals from all orebodies processed are galena (~13%wt) and sphalerite (~14%wt). Pyrite and pyrrhotite (~60 to 70%wt in total) are the main floatable gangue in the ore. Tetrahedrite is the major host of silver, apart from galena and chalcopyrite.
Drill hole Information	• *A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:* ○ *easting and northing of the drill hole collar* ○ *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* ○ *dip and azimuth of the hole* ○ *down hole length and interception depth* ○ *hole length.* • *If the exclusion of this information is justified on the basis that the*	• A table of drill hole data is included as an attachment to this report.

Criteria	JORC Code explanation	Commentary
	information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.	
Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.* • *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* • *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	• No data aggregation methods have been used in the table of drill hole data.
Relationship between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.* • *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* • *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').*	• Holes were drilled vertical, intersecting the direction of main grade continuity at approximate right angles.
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	• Maps and sections of the drill hole locations, mineralised intercepts and domain interpretations are included in the body of the report.
Balanced reporting	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	• Exploration results are not the subject of this report.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	• Bulk density measurements and metallurgical test results are discussed in the report. • The CP considers there is no other meaningful and material exploration data in relation to this MRE.
Further work	• *The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).*	• No further exploration work is planned for tailings.

Criteria	JORC Code explanation	Commentary
	• *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	

Section 3 Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary

Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.*
	• *Data validation procedures used.*
		• The following database validation activities have been carried out:
		  ◦ Ensure compatibility of total hole depth data in the collar and assay drill hole database files.
		  ◦ Check for overlapping sample intervals.
		  ◦ Checking of drill hole locations against the surface topography.
		  ◦ Visual validation in Surpac software.
		• A selection of laboratory assay certificates were checked against database entries.
		• No issues were found with the database.

Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	• The Competent Person has visited the Endeavor Mine on several occasions since 2010.
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	• There is no geological interpretation of the tailings deposits, and it is assumed the tailings were deposited in sub-horizontal layers. • The volume of tailings is constrained by surveys of the topography prior and subsequent to the deposition of the tailings. • The style of deposit (tailings) does not allow for alternative interpretations. • The mineralisation within the TSF is considered highly continuous with low variability.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	• The Resource estimate entails the bulk of Sector 1 of the CTD TSF, which measures approximately 550m by 850m and an average depth of 7m
Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.*	• The resource model is based on statistical and geostatistical investigations generated using 2m composited sample intervals of the holes drilled in 2015. Assessment of the data suggested no requirement for high grade cutting. The composite data sets displayed low coefficients of variation. • A sub-celled block model was constructed using parent block

Criteria	JORC Code explanation	Commentary

	• *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.*
	• *The assumptions made regarding recovery of by-products.*
	• *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).*
	• *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.*
	• *Any assumptions behind modelling of selective mining units.*
	• *Any assumptions about correlation between variables.*
	• *Description of how the geological interpretation was used to control the resource estimates.*
	• *Discussion of basis for using or not using grade cutting or capping.*
	• *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*
		dimensions of 50m East by 50m North by 2mRL. Block sizes were based on average drill hole spacing of 100m.
		• Resource estimation was carried out by Ordinary Kriging (OK) method using multi-pass-pass strategy, with the first pass set at a distance less than the total range of the variogram. The number of composites for a successful estimate was restricted to a minimum of 3 and a maximum of 12 for the first pass and a minimum of 2 and a maximum of 10 for the second pass. The search axes were aligned with directions of maximum continuity derived from variographic analyses of the data set. Surpac mining software was used carry out the estimation.
		• The estimated tonnes and grade have been compared to historical tailings deposition records and are within 4% of the tonnes and 0.5% of the Zn grade. The grades also compare well with global metallurgical composite head grades.
		• The tailings are contained within a licensed facility and will be re-processed and deposited into another facility that is licensed to handle potential acid forming material.
		• The maximum extrapolation distance from known data points was around 150m.
		• No assumptions of byproduct recovery have been made.
		• No assumptions about correlation between variables has been made.
		• The search radii were aligned to reflect the sub-horizontal nature of tailings deposition with blocks and composite selection confined to within the Sector 1 boundary and modelled top and base of tailings.
		• Validation of the estimate was completed and included both interactive and statistical review. The validation methods included: -
		  • Visual comparison of the input data against the block model grade in plan and cross section.
		  • Comparison of global statistics.
		  • Swath plots, comparing the composite grade and the estimated grade grouped by intervals in plan and section
		The model was found to be robust.

Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	• The tonnages were estimated on a dry basis.

Criteria	JORC Code explanation	Commentary
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	• Little to no selectivity is assumed from the prosed mining method (hydromining) therefore no cut off grade has been applied to the estimate for reporting purposes.
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	• The tailings is proposed to be mined by hydromining methods, where water cannons liquify and push the tailings into a collection drain which runs to a sump where a pump delivers the slurry to the processing plant.
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	• The Endeavor Mine has a conventional Pb/Zn/Ag flotation plant with a demonstrated capacity of 1.2 Mtpa. • Metallurgical test work has indicated saleable Zn and Pb/Ag concentrates can be obtained from processing the tailings through the existing flotation process on site.
Environmental factors or assumptions	• *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	• There is a fully permitted Tailings Storage Facility on site with adequate storage capacity as well as approved plans for capacity increase through a perimeter wall raise.
Bulk density	• *Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.* • *The bulk density for bulk material must have been measured by*	• During the 2014 drilling program, 551 samples for density analysis were taken from each 1m interval by firmly compressing the material into a grout sampling and levelling the top off. Each sample was stored in zip-lock plastic bags and taken to the site laboratory for wet weight and dry weight measurements. The average dry density value

Criteria	JORC Code explanation	Commentary
	methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit. • *Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	was 1.74 t/m3.

Classification	• *The basis for the classification of the Mineral Resources into varying confidence categories.*
	• *Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).*
	• *Whether the result appropriately reflects the Competent Person's view of the deposit.*
		• The Resource has been classified as Indicated and Inferred with the key parameters considered during the resource classification being:
		  ○ Geological knowledge and interpretation.
		  ○ Deposit style.
		  ○ Confidence in the sampling and assay data.
		  ○ The spacing of the exploration drill holes.
		  ○ Variogram model ranges in relation to the local data spacing and the estimation variance.
		  ○ Prospects for eventual economic extraction.
		• The exploration data used for the TSF Sector 1 Resource estimate is robust and appropriate for resource estimation purposes, with the current data spacing sufficient to generate robust grade estimates. Confidence in the estimate is increased by good comparisons to historical tailings deposition records and head grades from global metallurgical composite samples.
		• There are reasonable prospects for the eventual economic extraction of the resources because of proximity to an existing floatation processing plant and metallurgical test work indicates economic recoveries for Zn, Pb and Ag.
		• Based on the consideration of items listed above, and review of the resource block model estimate quality, classification criteria were determined as summarised in the following: -
		  • **Indicated**
		    ○ Blocks in the tailings domain that occur between drill holes or no more than 50m from a drill hole.
		  • **Inferred**
		    ○ Blocks that were estimated in the third pass (or second pass in the VEIN domain).
		    ○ All remaining blocks in tailings domain no assigned Indicated.
		• The classification reflects the Competent Person's view of the deposit.

Criteria	JORC Code explanation	Commentary
Audits or reviews	• *The results of any audits or reviews of Mineral Resource estimates.*	• There have been no audits or reviews of the estimate.
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	• There has been no attempt to apply geostatistical methods to quantify the relative accuracy of the Mineral Resource to within a set of confidence limits. • The Competent Person believes the Mineral Resource estimate provides a good estimate of global tonnes and grade. • Higher local variances in tonnes and grade can be expected in areas classified as Inferred due to lower data density. • No change of support adjustment has been made to the block estimates. • The accuracy and confidence of this Mineral Resource estimate is considered suitable for public reporting by the Competent Person.

Attachment 2

Drill Hole Information (Holes used in MRE)

Drill Hole Collar Information

Hole ID	East	North	RL	depth	Type
S1_PD01	4798.74	5750.49	10210.7	2.3	Push Drill
S1_PD02	4847.93	5600.59	10210.44	3.5	Push Drill
S1_PD03	4849.08	5499.55	10210.12	4.2	Push Drill
S1_PD04	4848.83	5400.15	10209.8	4.5	Push Drill
S1_PD05	4798.24	5299.98	10208.79	3	Push Drill
S1_PD06	4797.3	5200.05	10208.27	3	Push Drill
S1_PD07	4898.11	5099.8	10208.22	4.5	Push Drill
S1_PD08	5048.92	5099.94	10208.32	4	Push Drill
S1_PD09	4997.98	5200.16	10209.12	4.2	Push Drill
S1_PD10	5000.11	5299.86	10210.16	4.9	Push Drill
S1_PD11	5049.24	5399.83	10211.76	5.9	Push Drill
S1_PD12	5048.98	5500.31	10212.76	6.4	Push Drill
S1_PD13	5048.34	5599.75	10212.97	6.4	Push Drill
S1_PD14	4950.37	5699.79	10211.68	5.2	Push Drill
S1_PD15	5145.64	5692	10214.35	4	Push Drill
S1_PD16	5199.08	5599.78	10215.58	9.5	Push Drill
S1_PD17	5198.62	5499.9	10214.59	8.8	Push Drill
S1_PD18	5199.11	5399.97	10212.49	3	Push Drill
S1_PD19	5150.94	5299.79	10210.72	5.7	Push Drill
S1_PD20	5148.85	5197.85	10209.26	4.8	Push Drill