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Technical Report

Pegmont Mineral Resource Update and PEA

Vendetta Mining Corp

Queensland, Australia

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

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1 Summary

1.1 Introduction

AMC Mining Consultants (Canada) Ltd. (AMC) was commissioned by Vendetta Mining Corp. (VTT or the Company), of Vancouver, Canada, to prepare a Preliminary Economic Assessment (PEA) and corresponding Technical Report (Technical Report) for the Pegmont Property (Property) in Queensland, Australia. This report is an update to an earlier report titled "Technical Report on Pegmont Resource Update June 2017, Queensland, Australia for Vendetta Mining Corp," written by AMC and with an effective date of 22 June 2017 (2017 AMC Technical Report).

This report has been prepared in accordance with the requirements of National Instrument 43-101 (NI 43-101), "Standards of Disclosure for Mineral Projects", of the Canadian Securities Administrators (CSA) for lodgment on CSA's "System for Electronic Document Analysis and Retrieval" (SEDAR).

1.2 History, location, and ownership

The Property is located in north-west Queensland, Australia, approximately 175 km south-east of Mount Isa, 130 km south-southeast of Cloncurry and 700 km west-southwest of Townsville. The project is at approximately latitude 21°51′ south, longitude 140° 41′ east. It is situated in an area with a number of producing mines within 30 km of the Property. These are South32 Limited's Cannington Silver Lead Zinc Mine, Chinova Resources Pty Ltd (Chinova)'s Osborne Copper-Gold operations, and Chinova's Starra Gold-Copper Mine.

It can be accessed from Mount Isa by sealed roads and secondary dirt roads with local access by tracks created by local graziers and previous mineral explorers, which require only occasional grading. Secondary gravel roads are well-maintained by Cloncurry Shire Council. There are several river and creek crossings through causeways, which can become impassable for relatively short periods of time (days) during the peak wet season.

The Property is in the Mount Isa Mineral Province, a region recognized as a world-class mining region, with more than a quarter of the world's lead and zinc Mineral Reserves, 5% of the world's silver Mineral Resources, and 1.5% of the world's copper resources. There is a large skilled workforce currently employed on other projects in the Mount Isa Mineral Province and at the city of Townsville, 700 km to the north-east.

The Mount Isa Mineral Province is connected to Townsville and the coast via the Great Northern Rail Line and the Flinders Highway. The Great Northern Rail Line is operated by Queensland Rail, which is owned by the Queensland Government. A deep-sea port is operated in Townsville, which services the metal mining, sugar, and fertilizer industries. Korea Zinc (Sun Metals) operates a zinc and lead smelter located at Townsville.

The closest population centre to Pegmont is the town of Cloncurry (population approximately 3,500), 90 km to the north. The larger regional centre of Mount Isa (population approximately 23,000) is 150 km to the north-west of the project area. Cloncurry is serviced by commercial aircraft from Townsville and Brisbane, while Mount Isa can be reached by commercial aircraft from Townsville, Cairns, and Brisbane.

There are two existing concentrate loading sidings on the Great Northern Rail Line, one at the Phosphate Hill siding, approximately 80 km to the west of Pegmont. Phosphate Hill is the site of the Duchess Phosphate Deposit, Australians Largest Phosphate operating mine and is also the location of the Osborne Mine copper-gold concentrate rail loading siding. Lead-Silver and Zinc Concentrate from the Cannington Mine is loaded at Yurbi, about 15 km east of the town of Cloncurry.

The dominant land use in the area of the Pegmont tenements is mining and cattle grazing. A long history of mining is established in the Cloncurry Shire, and several operating and historic mines are located less than an hour's drive from the Pegmont project site, including Cannington, Osborne, Starra, and the Merlin project.

The Cannington Lateral natural gas pipeline lies about 50 km to the south of Pegmont and provides natural gas for power generation to the Osborne and Cannington Mines.

There are several historic and active weather stations within a 100 km radius of the Property. The area has a semi-arid climate with temperatures varying between 7° and 25°C in winter, and from 25° to 39°C in summer. The weather is acceptable for exploration and mining operations year-round; however, field operations may be interrupted during heavy rainfall events, which typically occur in the summer, rendering access roads impassable due to boggy ground conditions.

The Property is fully owned by Pegmont Mines Ltd (PML) and consists of both mining and exploration leases. There are three mining leases (ML); 2623, 2621, and 2620, and one Exploration Permit for Minerals (EPM); 26210 which cover a total of approximately 8,390 hectares (ha), of which 361.5 ha are covered by the three mining leases. The EPM is valid for the exploration of all minerals except coal. The EPM is currently comprised of 26 sub-blocks. This must be reduced to 16 at the end of the third year, which occurs on 21 August 2020. VTT has executed an Option Agreement and Royalty Agreement with PML to acquire 100% ownership of the project.

There is a 1.50% Net Smelter Return (NSR) royalty agreement with PML for which VTT retains the right of first refusal should PML choose to sell. Queensland State Mineral legislation imposes a royalty on the sale of minerals, which are a variable rate royalty system.

Land access agreements are in place. All work areas related to the project are located on a Pastoral Lease held by Argylla Mountains Pty (Argylla). In the case of the MLs which were in existence before the pastoral lease was granted there is no requirement for land owner compensation. As the EPMs were granted after the Pastoral Lease there is a schedule of compensation to Argylla, of A$200 per drillhole and A$500 for new access tracks, in addition to the cost of any damage.

There are no known risks to the titles that may affect access or rights to the MLs or EPMs that comprise the Pegmont project.

The Property has been held by way of EPMs and MLs which overlapped the current Property at different times. The main historical owners of the Property were Placer Prospecting (Aust.) Pty Ltd. (1970 – 1975), BHP Minerals Pty Ltd, Mount Isa Mines Ltd & Newcrest Mining Ltd. (1976 – 1996); Pegmont Mines NL, its antecedents, and the current company Pegmont Mines Ltd (1996 to present).

Prior to 1975 the MLs on the Property were tested with 100 shallow rotary and percussion and 35 diamond drillholes of which 28 fall inside the current ML area. Most holes were drilled vertical. There was a gap in exploration work being carried out from 1975 to 1996 though BHP drilled three drillholes outside the deposit, and regional aeromagnetic and airborne EM (GEOTEM™) surveys were carried out. In 1998 / 99, Initially with PML being the owner, North Ltd entered a joint venture over the twelve MLs that existed at the time. North Ltd ran an integrated program of remote sensing, geophysics sampling and drilling 15 drillholes which focused on the shallow oxide / transition areas. Subsequent to that, drilling was carried out by Cloncurry Metals Limited (CLU) by way of an option in 2007 / 08 and by PML itself.

Work on the EPMs has consisted of air photo based geological mapping, airborne geophysics with ground geophysical follow up. This led to the testing of delineated anomalies by a number of companies over time.

While there have been a number of historical Mineral Resource estimates carried out, none of which have been validated by the Qualified Person (QP) and are not NI 43-101 compliant, the latest Mineral Resource estimate was carried out by AMC and disclosed in the 2017 AMC Technical Report. There has been no production from the Property.

1.3 Geology and mineralization

The Mount Isa Inlier is a multiple deformed and metamorphosed Early to Middle Proterozoic terrain. The inlier has been subdivided into three broad north-trending Provinces - the Western, the Kalkadoon-Ewen, and the Eastern Fold Belts, based on various tectonic, structural, and paleogeographic criteria. The Eastern Fold Belt is subdivided into a further three zones, from west to east; the moderately deformed Wonga and Quamby-Maldon Subprovinces and the structurally complex Cloncurry Subprovince.

The Western Fold Belt contains the sediment hosted, stratiform world class Mount Isa, Hilton, George Fisher lead-zinc-silver deposits and the Mount Isa copper deposit. The Eastern Fold Belt, in particular the Cloncurry Sub-province contains deposits of a different character including several significant iron-oxide copper-gold deposits (Ernest Henry, Osborne, Mt Elliot, and the Starra deposits) and Broken Hill Type (BHT) lead-zinc-silver deposits including the world class Cannington and several smaller examples of which Pegmont is the largest.

The deposit scale geology at Pegmont has been the subject of a number of scientific studies including the unpublished Ph.D. thesis of Vaughan (1980) and Newbery (1990) which was completed prior to most of the drilling which occurred from 1998. There are few outcrops on the Property, therefore the geological framework of the Pegmont deposit draws not only on surface fact mapping but is also heavily reliant on drillholes and interpretations of geophysical surveys.

The informally named “Pegmont Beds” are a monotonous sequence consisting mainly of psammitic (quartz-feldspar-biotite) gneiss, feldspathic quartzite and psammo-pelitic (quartz-feldspar-biotite-muscovite) schist, with subordinate pelitic (quartz - muscovite - biotite ± graphite) schist. Narrow bands of tourmaline-bearing schist are locally developed. The schists have a well-developed foliation and lineation and, in places, a crenulation cleavage. The sequence is thought to represent a metamorphosed clastic sedimentary sequence, originally feldspathic and lithic sandstone and siltstone, with subordinate fine clastic (in part carbonaceous) sediment.

Garnet quartzite (metaarkose) and the garnet-bearing schists are present on the margins of the lead-zinc mineralization, which is contained in a banded iron formation (BIF). The BIFs consist of banded quartz-magnetite-fayalite-garnet-grunerite-hedenbergite-sulphide. Apatite, gahnite and graphite are common minor minerals. Bedding is typically on a scale of 1 to 5 mm. In fresh rocks the main sulphide minerals are galena, sphalerite, with subordinate pyrrhotite, pyrite, and chalcopyrite. Garnet quartzite can be present at the lateral extents of the BIF, where the BIF has apparently been attenuated due to folding.

The overall morphology of the stratiform mineralized horizons at Pegmont are variably folded, but with an overall shallow dip to the east. It is described in terms of seven Zones; 1, 2, 3, 4, 5, the Bridge Zone, and the Burke Hinge Zone (BHZ), each of which have differing structural styles. The current interpretation favours a folded sequence that has seen ductile re-mobilization of lead-zinc mineralization into fold hinges and other low-pressure zones with subsequent thinning and depletion of lead-zinc mineralization on the fold limbs.

The known mineralization extends approximately 2 km along strike and approximately 1 km in the down dip direction to the southwest. At present mineralization is known to extend to a depth of 350 m below surface. To date three BIF / garnet quartzite horizons have been identified; Lens A, B, and C. Lead-zinc mineralization is most developed in Lens B, around 2 to 8 m thick. Lens C is becoming increasingly thicker and with improving lead-zinc grade to the south-east in Zone 5, where it is up to 5 m thick. Lens A is present intermittently in all Zones, but lead-zinc grades are also improving to the south-east, however it remains < 2 m thick.

The sequence is cut by a later amphibolite dyke, without any off-set. The dyke and the BIFs have been deformed by the emplacement of the Squirrel Hills granite, located on the northern boundary of the lease, where it forms several low outcrops of biotite and hornblende-bearing granite with coarse K feldspar.

1.4 Exploration and data management

While exploration work is predominantly drilling, a transient electromagnetic survey was carried out in 2017 to assist in vectoring a copper intersection on EPM 14491. The anomalies found were not explained by drilling.

Drilling of different types forms the basis of the 2018 Pegmont Mineral Resource estimate and has been carried out over a number of years by several operators. VTT has carried out 14 RC holes, four diamond drillholes, and 151 holes with percussion / RC tops and diamond tails.

This has resulted in a database consisting of the results of 588 drillholes with a total of 526 drillholes contributing to the model. The database is a mix of percussion, reverse circulation (RC), and diamond drilling (DD), while the early data is of variable quality with some back up documentation not available from the early work, the VTT work is to modern industry standards. Since the 2017 AMC Technical Report, VTT has carried out validation and checking of the old data making it more assured than previously; generally, the older information which has been collected by larger companies has turned to have been carried out in a professional manner.

VTT routinely perform downhole surveys, and core orientation is attempted at the start of each drill run. Drillholes are precollared using RC methods designed to fall short of the mineralization by approximately 20 m and mineralized intercepts are cored. Even with folding, VTT has developed a high proficiency in predicting the location of the mineralized interval to minimize coring. All core is photographed wet and dry.

Drillhole data is stored in an Access database which is managed by VTT. Of the 588 drillholes in the database, 89% have intersected significant mineralization as defined by greater than 1% Zn+Pb. The mineralization outcrops in part and the deepest intersections are approximately 400 m below surface.

1.5 Sampling, analyses, and security

Prior to 2007, there is limited information on the sampling procedures used, but documented sampling procedures used since 2007 appear to meet current industry standards.

VTT sample all garnet sandstone and BIF horizons intersected, regardless of the visual grade estimate made by the geologist.

Sampling of mineralized zones begins at the hangingwall contact. Sample intervals 1 m in length are marked downhole from this point with a minimum 0.5 m sample at the footwall contact. Samples are continued 5 m into the hangingwall and footwall sequences.

The majority of drill samples have been sent to commercial mineralogical laboratories using standardized industry practice base metals analysis, with laboratory QA/QC protocols in place. The analytical methods used to assess the Zn, Pb, and Ag grades are considered suitable for this style of deposit.

QA/QC results were not routinely reviewed by the historic operators. VTT conducted a review which indicates that the results from most drilling programs are reasonable. AMC has reviewed this work and concludes that the data is suitable for this Mineral Resource Estimate with the Mineral Resource confidence classifications chosen. However, further work is required including further checks and test work to verify their validity for use in future estimates.

Sample preparation for the VTT samples has been completed at ALS Townsville. The assay methods chosen by VTT are the same as those used by the previous operator (PML). QA/QC procedures are in place in the VTT programs with blanks, Certified Reference Material (CRM), and field duplicates being used.

There are no sample security procedures at site, but the current practices and nature of mineralization minimize the chance of sample loss or interference during transport, and AMC considers it unlikely that there are any material sample security issues.

1.6 Data verification

The presence of BIF-hosted lead-zinc mineralization is supported by the QP's observation of galena and sphalerite in fresh BIF in core and in drill chips at site, the observation of BIF in oxidized core material at site, and the presence of honeycomb-textured gossans in outcrop at site.

The site visit confirmed major drilling phases, the presence of sulphide mineralization in outcrop, core and drill chips, evidence of original assay certificates, and use of the supporting information. The geological interpretation appears to be reasonable in the context of observations of outcrops and drill core at site.

Whilst all original records have not been cross-checked in detail against the database results, the checks conducted indicated good correlation, and their presence gives the QP confidence in verifying that the VTT drilling programs were completed as reported and were recorded appropriately.

Underlying limitations in the historic data, which have drawn attention before, are being systematically addressed by VTT and in the opinion of the QP, the data is adequate as a basis of estimating the current Mineral Resources.

1.7 Mineral Resources

The Mineral Resource estimate for the Pegmont deposit has been carried out by independent QP, Ms Dinara Nussipakynova, P.Geo. of AMC who takes responsibility for the estimate.

Table 1.1 shows the Mineral Resources for the lead-zinc-silver mineralization as of 31 July 2018 at a 3% lead plus zinc cut-off grade for the Mineral Resources constrained by an open pit and a 5% lead plus zinc cut-off grade for Mineral Resources potentially mineable by underground methods.

Oxide lead-zinc mineralization is not included in the current Mineral Resource, as with the sequential flotation processing flow sheet envisaged, it is considered that there is no effective method for mineral processing of oxide mineralization and hence no economic basis for its inclusion.

Table 1.1 Total Mineral Resources at 31 July 2018

	Classification	Material type	Tonnes (kt)	Pb (%)	Zn (%)	Ag (g/t)
Open pit (Zones 1, 2, 3, and BHZ)	Indicated	Transition	1,111	4.9	2.3	8
		Sulphide	4,003	6.5	2.6	11
		Total	**5,114**	**6.2**	**2.6**	**11**
	Inferred	Transition	1,829	5.2	2.0	7
		Sulphide	2,567	5.0	2.3	10
		Total	**4,396**	**5.1**	**2.2**	**8**
Underground (Zones 3, 4, 5, and Bridge)	Indicated	Sulphide	**644**	9.0	2.6	14
	Inferred	Sulphide	**3,880**	5.1	3.6	4
Total	Indicated	Transition	1,111	4.9	2.3	8
		Sulphide	4,647	6.9	2.6	12
		Sub total	**5,758**	**6.5**	**2.6**	**11**
	Inferred	Transition	1,829	5.2	2.0	7
		Sulphide	6,447	5.1	3.1	9
		Total	**8,277**	**5.1**	**2.8**	**8**

Notes:
1. CIM Definition Standards (2014) were used to report the Mineral Resources.
2. Cut-off grade applied to the open pit Mineral Resources is 3% Pb+Zn and that applied to the underground is 5% Pb+Zn.
3. Based on the following metal prices: US$0.95/lb for Pb, US$1.05/lb for Zn, and US$16.5/oz for silver.
4. Exchange rate of US$0.75:A$1.0.
5. Metallurgical recoveries vary by zone and material type as follows:
   - Lead to lead concentrate: from 80.6% to 91.3% for transition and 88.0% to 92.7% for sulphide.
   - Zinc to zinc concentrate: from 19.3% to 75.2% for transition and 61.8% to 78.5% for sulphide.
6. Using drilling results up to 15 April 2018.
7. Mineral Resource tonnages have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

1.8 Metallurgical and mineral processing

Test work previously carried out was on samples tested mainly from Zone 2 and consisted of a broad spectrum of oxidation states from oxides through transition material to primary sulphide mineralization.

A metallurgical test work program was carried out by ALS in their laboratory in Burnie, Tasmania during 2016, and focussed on samples from the BHZ and deeper material from Zone 5. This program consisted mainly of conventional differential flotation test work, in which this laboratory is well-experienced.

Pre-concentration by HMS or magnetic separation was investigated but found to be not feasible. For the primary mineralization, the flowsheet will be a straightforward Pb / Zn differential flotation.

The recently completed test work program at ALS focused on metallurgical drillhole samples from Zones 1, 2, and 3, plus some from the Bridge Zone between the main mineralization zones and the BHZ zone tested in 2016. These samples were received in late 2017 and reported in 2018, with key findings summarized below:

- Mineralogical studies have shown that galena is well liberated but sphalerite less so, especially in the transition material. Regrinding of rougher concentrate is indicated.
- This is confirmed by the flotation results where more intensive regrinding of the lead concentrate is required in order to release zinc losses to the zinc circuit.
- Although batch flotation tests exhibited some variability in zinc performance, locked cycle tests which are a better predictor of plant performance, showed more consistency. Overall the

results gave 90 – 93% lead recovery to high grade concentrates ranging from 66% - 73% Pb. As expected, zinc recoveries were lower, being 70 – 75% to concentrates in the 52 – 55% Zn range.

- In benchmark terms these are typical results from complex sulphide deposits and Pegmont can be considered amenable to a conventional differential flotation route.

1.9 Mining

The Pegmont project includes three resources consisting of the BHZ, the Main Zones (Zones 1 - 5), and the Bridge Zone. The BHZ deposit can be developed as an open pit, the Main Zone can be developed with a combination of open pit and underground mining methods; and the Bridge Zone is a potential underground operation.

1.9.1 Open pit

The open pits are proposed to be mined using a conventional truck and excavator mining method using 90 t payload trucks and 150 t – 200 t excavators. Waste will be mined on 10 m benches while mineralized material is selectively mined in 2.5 m flitches. AMC assumed a mining recovery of 95% and a mining dilution of 5% at zero grade.

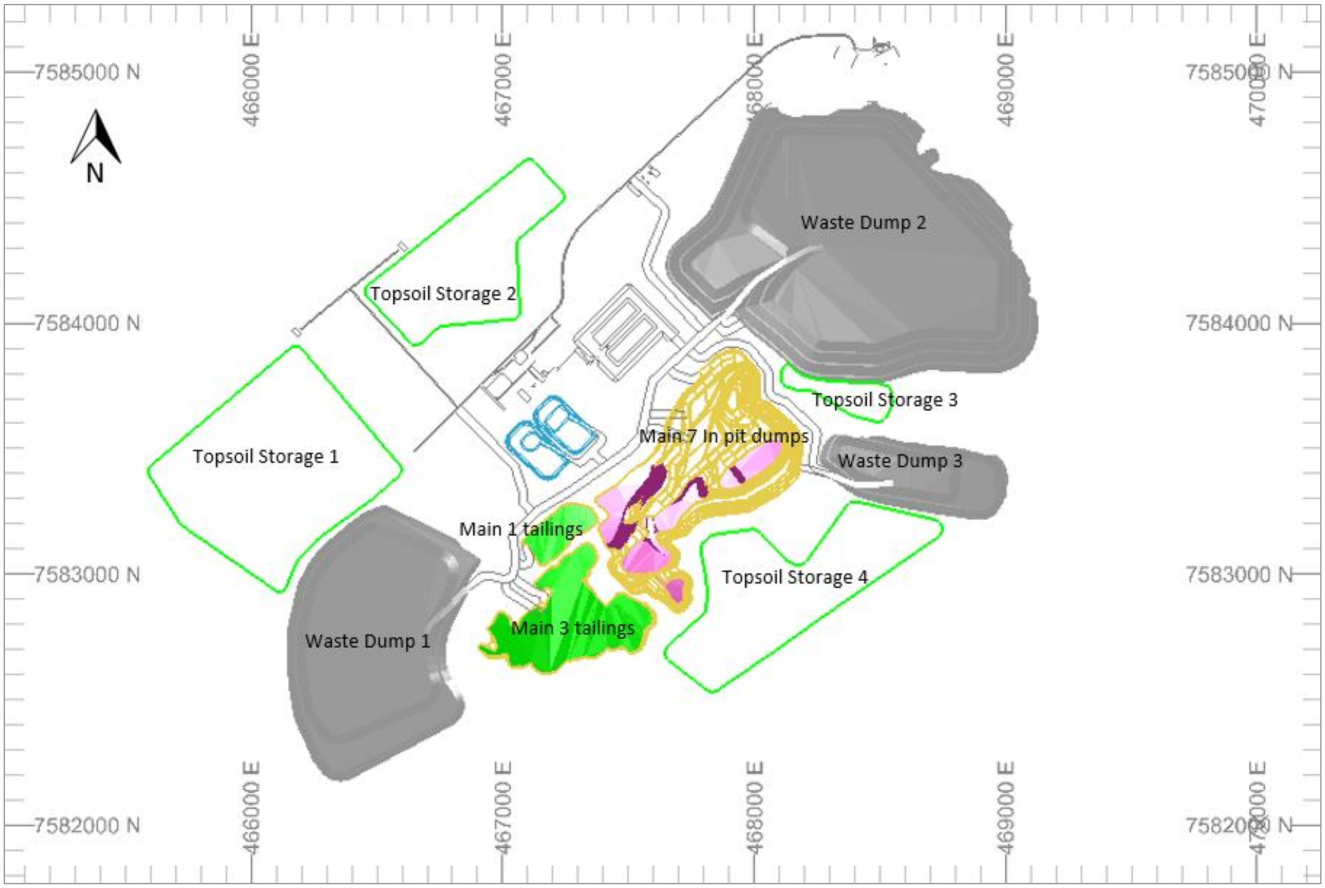
The cut-off grade to differentiate mineralized material to waste material was based on an NSR calculated for different mineralogical zones and ranges between A$30.49/t and A$31.90/t.

AMC used Whittle's pit optimization software to define the ultimate pit limits, while taking into consideration the possibility to mine some of the deposits form underground and maximize project value.

In total seven pushbacks have been designed in the Main area (Main 1 to 7) and one for the BHZ pit. Three starter pits will be used as tailings storage facilities and will be mined in order to provide adequate storage in relation to the amount of mineralized material to be processed.

Two out-of pit and three in-pit waste dumps have been designed to provide flexibility and costs savings for waste placement. Top soil areas have been designed in strategic locations to accommodate the top soil removed from the pit, waste dumps, long-term stockpiles, and site infrastructure.

Figure 1.1 Open pit site layout



Source: AMC

The open pit contains approximately 8.9 Mt of mineralized material with a grade of 5.15% Pb, 2.2% Zn, and 8.45 g/t Ag and 120 Mt of waste material, with an overall waste to mineralized material strip ratio of 12.5 to 1. The open pits are mined over a ten-year period.

AMC estimated mining costs, assuming a mining contractor with an average mineralized material mining cost of A$3.54/t and A$3.08/t for waste mining. Capital costs for mobile machinery and equipment have been incorporated into the operating costs as leasing costs using a contractor profit margin of 15%.

1.9.2 Underground

To assess the underground mining potential, a pit shell was generated that incorporates the potentially economic open pit mineralization for the Main Zone, all mineralized material outside of the pit shell was considered for underground mining with the focus on the Bridge Zone.

The geometry of the underground resources is primarily flat dipping (23° to 30°) and varying in thickness across each zone (3 m to 12 m). However, the Zone 5 mineralization and part of Zone 3 tends to be narrow in width and fairly vertical in dip. The geometry lends itself to room and pillar (R&P) mining in the flat dipping zones and longhole stoping in the steeply dipping part of Zone 3. Preliminary stope design shapes were generated to reflect the selected methods.

In general, the host rock is considered to be "Good" with highly competent hangingwall and footwall. Minimal ground support is required for the R&P operations and the longhole stopes will be filled with waste rock. The water table was assumed to lie 50 m below topographical surface.

To optimize the open pit to underground interface, AMC has generated pit shells and determined open pit value and compared this to the value generated by underground stopes on a level basis. After selection of the optimal depth of the pit relative to underground stopes, a combined value is determined to confirm that the selected depth generates the maximum value. Stopes are then clipped to the pit design to determine tonnes and grade for the potential underground mine. It was assumed that a 20 m crown pillar will be left beneath the Main pit.

After the analysis it was determined that Zone 3 and the Bridge Zone could be optimally mined from underground. Zone 3 has two distinct areas, one directly beneath the Main pit (termed Zone 3A) and one to the side of the Main pit (Zone 3B).

Based on the likely mining method selection, AMC used benchmark cost models for Australian metal mines over a range of production rates per day. Given the competent host rock and the minimal need for ground support the R&P operating cost selected for this study is A$50/t of mineralized material. A minimal amount of material is mined using longhole stoping and a benchmark cost of A$50 was selected for this mining method. This benchmark costs compare well with similar sized operating mines in AMC's database.

The cut-off value is based on an NSR value; it was calculated based on initial parameters that reflect the total operating cost of A$50/t for mining, A$28.42 for processing, and A$2.1 for General and Administration (G&A), for a total of A$80.49/t of mineralized material. Processing costs and G&A were estimated by GR Engineering Services Limited (GRES).

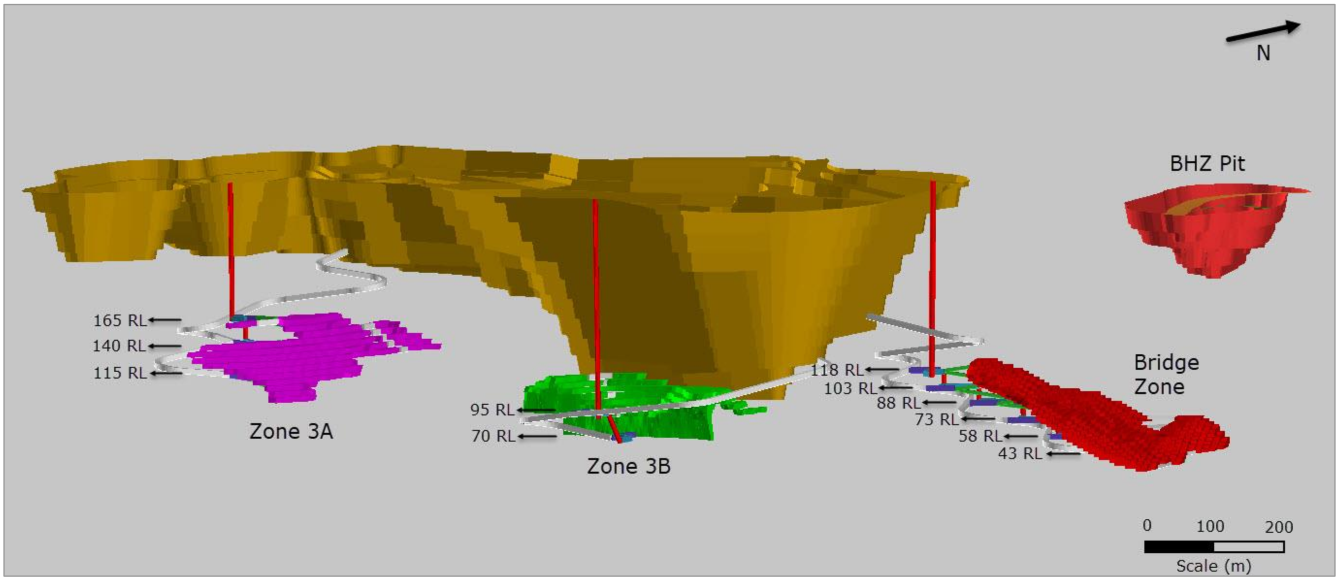
AMC used a function of the Datamine™ software, Mine Shape Optimizer (MSO) to evaluate preliminary stope wireframes for the R&P and longhole mining methods. For R&P AMC has applied a dilution factor of 10% at zero grade to the Mineral Resource and a mining recovery factor of 86% has been applied to the stopes, 100% recovery is assumed for the mineralized material from development.

For longhole mining, AMC has applied a dilution factor of 12% at zero grade to the Mineral Resource and a mining recovery factor of 95% has been applied to the stopes; 100% recovery is assumed for the mineralized material from development.

There will be no backfill in the R&P areas. AMC assumed that waste rock from development will be placed as backfill in the longhole stopes.

AMC has developed conceptual mine designs for each zone to determine mine physicals and development costs. Each zone is accessed from the bottom of the Main pit with individual declines. The three main access declines have a total length of 2,166 m, level access development has a total length of 1,786 m, and the remaining development includes return air drive access and mineralized development. Vertical development consists of return air raises with a total of 698 m. Total development for the three underground zones including vertical development is 5,547 m. The mine design for each zone is shown in Figure 1.2.

Figure 1.2 Underground mine design for all zones



Source: AMC

AMC has undertaken a preliminary estimate of the ventilation requirements in consideration of the production rate, ore handling, and mining method. It is estimated that the total mine airflow should be 261 m^3/s.

The mine will be ventilated by a "Pull" or exhausting type ventilation system. That is, the primary mine ventilation fans will be located on surface at the primary exhaust airways of the mine. Fresh air will enter the mine via the internal declines from the portals inside the pit and exhaust to the surface via dedicated return airways. Most production activities will require auxiliary fans and ducting with level airflows managed through regulators located at raise accesses.

AMC has completed an estimate of the main equipment required to meet the production rate for the combined underground mining zones. However, it is assumed that the equipment will be supplied by the contractor as the open pit production ramps down and the underground production ramps up.

AMC has provided an estimate of the manning required to meet the planned production rate from the three zones. The estimate assumes that the labour will operate on a 2 week on 1 week off roster. The maximum number of personnel hired for the underground mine operations will be 202, which includes an 8% allowance for absenteeism.

Electrical power will be generated by gas fired generator sets, and the maximum demand will be approximately 1.9 MW for the underground operations.

Potable water will be generated onsite from the raw water supply via a reverse osmosis plant before being pumped to the plant and mining amenities as well as the accommodation village.

Raw water for the Project shall be sourced from a dedicated bore field. A four-inch HDPE line will be installed in stages down the declines to provide service water for use in the mine. Every 40 m vertically, a pressure reducing valve will be installed to control the pressure in the line.

A total capacity of 10 l/s is required for the dewatering system for each mine. During development of the declines and prior to establishment of the main sumps, any water will be collected and discharged using submersible pumps.

The three mines will have a main dewatering sump installed near the intersection of the main access decline and the lowest access to the main working levels. At the main sump there will be two – 75 kW driven centrifugal multi-stage discharge pumps, one running and the other on standby.

Portable air compressors (one for each zone in operation) will be moved together with the primary mining equipment.

A leaky feeder system will provide means for communication underground. All vehicles will be fitted with radios. The underground mine personnel will also have radios and cap lamps equipped with emergency warning systems.

The workshop located on surface and surface magazine will be used to support the underground workings.

Portable 20 personnel refuge stations will be located appropriately relative to operating levels. Lunchrooms near the maintenance area will also serve as refuge stations. Return air raises will be equipped with the Safescape System and will act as the second egress for each zone.

The mine ventilation systems will be provided with an ethyl mercaptan (stench gas) system (activated manually or remotely) to warn underground personnel in the event of an emergency. Radio contact via the leaky feeder system provides an alternative method of communication.

Key capital cost areas for the underground mine include access development and related underground infrastructure. The total capital required for the underground mine is estimated to be A$40.9M. Capital cost estimates are based on mine physical take-offs from the mine design and either benchmark costs or recent vendor quotes. Development is costed at a unit rate of A$5,000/m.

1.9.3 Combined open pit and underground schedule

AMC undertook a value analysis to optimize the combined open pit and underground production schedules. Based on this analysis, it was concluded that the optimal combination is to mine the underground zones at the end of the mine life with operations starting in all three zones in Year 8 and being mined out by the end of Year 10.

The combined potential mill feed is summarized in Table 1.2. The maximum production from underground is in Year 9 and peaks at 0.8 Mtpa.

Table 1.2 Combined open pit and underground mill feed

	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Mill feed (Mt)	10.6	0.98	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	0.82
NSR (A$)	135.26	140.2	130.9	145.9	128.8	144.1	141.5	152.6	121.6	120.7	124.1
Pb (%)	5.3	5.7	4.8	5.2	4.7	5.7	5.8	6.3	4.9	5.0	5.2
Zn (%)	2.2	2.0	2.5	3.1	2.6	2.4	1.9	2.0	1.7	1.7	1.6
Ag (g/t)	8.8	6.8	7.5	7.7	7.6	8.7	10.4	11.0	10.5	8.5	8.9
OP mill feed (Mt)	8.9	1.0	1.1	1.1	1.1	1.1	1.1	1.1	0.8	0.3	0.2
NSR (A$)	132.9	140.2	130.9	145.9	128.8	144.1	141.5	152.6	113.6	47.5	47.5
Pb (%)	5.1	5.7	4.8	5.2	4.7	5.7	5.8	6.3	4.5	1.73	1.7
Zn (%)	2.2	1.9	2.5	3.1	2.6	2.4	1.9	2.0	1.6	0.9	0.9
Ag (g/t)	8.4	6.8	7.5	7.7	7.6	8.7	10.4	11.0	10.6	3.7	3.7
UG mill feed (Mt)	1.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.8	0.6
NSR (A$)	147.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	140.3	148.1	150.6
Pb (%)	6.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	5.8	6.2	6.4
Zn (%)	1.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1.9	2.0	1.9
Ag (g/t)	10.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	10.2	10.3	10.6

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

1.10 Mineral processing

The proposed process plant was based on the outcome of the metallurgical test programs and consists of a three-stage crushing circuit and primary ball mill followed by a sequential flotation process utilizing a regrind stage and cleaner stage flotation. The sequential flotation process will produce a silver enriched lead concentrate and a separate zinc concentrate.

It is envisioned that the process plant will treat 1,100,000 tonnes per annum from the open pit and underground operations.

Estimated mill recoveries and concentrate grades are summarized in Table 1.3.

Table 1.3 Mill recoveries and concentrate grades

	Lead	Zinc	Silver
Recoveries to Lead concentrate	91.0%	10.4%	75.2%
Lead concentrate grade	68.9%	3.2%	100 g/t
Recoveries to Zinc concentrate	1.8%	73.2%	2.4%
Zinc concentrate grade	3.07%	54.0%	7.6 g/t

The plant will consist of a three-stage crushing circuit to reduce the ROM ore sized material to a ball mill feed size of P80 of 10 mm. A single stage ball mill will treat 138 tonnes per hour of fresh feed in closed circuit configuration with hydro-cyclones to produce a cyclone overflow P80 of 106 µm suitable for the flotation process.

A sequential flotation process including a regrind stage of each rougher concentrate has been selected based on the lock cycle test work results. The plant will consist of a lead roughing and scavenger flotation stage with a concentrate regrind to a P80 of 20 µm, prior to a two stage cleaner flotation to final lead concentrate grade. Tailings from the lead circuit will be pumped to the zinc flotation circuit. The zinc circuit will consist of a rougher and scavenger stage with a concentrate regrind to a P80 of 20 µm, prior to a two stage cleaner flotation to final zinc concentrate grade.

Flotation concentrates will be dewatered by conventional thickeners for water recovery. Thickened slurry will then be filtered in horizontal plate and frame pressure filters to produce filter cakes with approximately 10% moisture for export.

Lead concentrates will be transported by road to the Mt Isa Smelter as bulk concentrate. Zinc concentrate will be stored into sealable ½ height sea containers for export by road and rail to Sun Metals in Townsville or to the Port of Townsville for export.

1.11 Infrastructure

1.11.1 Access

A 10.5 km access road from the Selwyn-Toolebuc road will be required crossing Sandy Creek to the operations site and accommodation village at Pegmont. A two-lane unsealed road suitable for use by heavy vehicles hauling concentrate is proposed. The creek crossing will have concrete culverts and causeway to ensure year-round access to site.

1.11.2 Power

A high-pressure natural gas pipeline is located approximately 16 km south of the project. It is envisioned to access gas from this line to site via a buried pipeline and run four natural gas fired generator sets to provide the necessary power to site and mining operations. Each set will operate at 80% producing 2100 ekW, with maximum output of 2500 ekW each when required. Power will be reticulated at 11 kV throughout the site.

1.11.3 Communications

A fibre optic line is located approximately 16 km to the south and has been run in parallel to the high-pressure gas line. It is envisioned that a new branch from a fibre node will provide for both voice and data communications for the site. There are also several mobile communications towers in the area, however these are not ideal for the project and can only provide a backup service for the area.

1.11.4 Water

Water for the project will be sourced from the artesian basin resource approximately 27 to 32 km to the south of the project and will require a remote bore field. Local bores at the project can also provide water, however these are limited and will be used for construction water requirements.

1.11.5 Tailings storage

Tailings will be pumped from the processing facility to mined out pits as mining progresses. Initial tailings will be sent to BHZ, mined during the pre-production period, followed by Main 1 and Main 3. Conceptual pillars have been left between the tailings pits and adjacent pushbacks to permit safe operations. A 5 m head board has been assumed to accommodate rain events.

1.12 Market studies and contracts

Marketing study for the concentrate products to be produced from Pegmont was contributed to by Ocean Partners.

The Pegmont Zn and Pb concentrate qualities are both reasonably clean, and as such are expected to be marketable to a wide number of smelters and other potential buyers. The Project will produce two concentrates, being a 65 – 70% lead concentrate with payable silver credits, and a 49 – 55% zinc concentrate.

Limited information was available with respect to penalties but initial tests were used to assign penalties for Cadmium, Fluoride, Iron and Lead by zone for cut-off grade calculation and potential economic evaluation.

There are no contractual arrangements for smelting at this time, nor are there any contractual arrangements for the sale of lead, zinc, or silver products, such as streaming or off-takes at this time.

The metal prices used in this PEA are based on an assessment of market forecast information (institutional consensus prices), current metal prices and rolling three-year London Metal Exchange (LME) averages and are presented in Table 1.4.

Table 1.4 Metal price and exchange rate

	Unit	Base case value	Spot price case (22 Jan 2019)
Lead price	US$/lb	1.09	1.18
Zinc price	US$/lb	0.94	0.91
Silver price	US$/oz	16.50	15.31
Exchange rate	US$:A$	0.75	0.71

1.13 Environmental

Environmental studies completed on the Project site to date have consisted of one baseline terrestrial ecology survey. Additional seasonal surveys will have to be conducted but with the exception of one Commonwealth listed marine species (Rainbow Bee-eater), no species or communities of conservation significance have been identified on the Project site, to date

Other approvals which may be required as part of the project are road use agreements for transport of product, or off lease approvals for transport / utility corridors.

The three existing MLs were granted prior to 1 January 1994, and therefore applications over these leases are not subject to a native title process.

Future application for mining licenses or variation of the terms of the existing MLs would require a statutory negotiation process to be undertaken at the time.

To minimize financial assurance commitments, rehabilitation would be undertaken progressively where permitted by the mine design and activities. Closure and remediation costs have been estimated at a high-level at approximately A$15M.

Based on the information reviewed as part of this preliminary environmental assessment, there were no known significant environmental issues or sensitive receptors/features identified that could materially influence project viability, nor affect the major design components for future mine development.

1.14 Capital and operating costs

The capital and operating costs estimate have been developed by the following contributors:

- GR Engineering Services: process plant, plant infrastructure, non-process infrastructure, and general and administration costs.
- Queensland Rail: Malbon rail siding.
- Wasco (Australia) Pty Ltd: natural gas pipeline.
- Energy Power Systems: power station.

- Commins Contracting: Selwyn Toolebuc Road to site access road development.
- AMC: underground mine infrastructure, and open pit and underground operating mining costs.
- Ocean partners: concentrate costs.
- VTT: owner's costs.

Operating and capital costs are presented in Australian dollars (A$) and are based on 2018 real dollars.

The underground mining costs have been estimated based on benchmarking information for similar mining methods and operation size in Australia. Open pit mining costs were estimated based on high level first principles calculations and benchmarked against peer Australian operations.

The process operating costs were estimated with an accuracy of +/- 30%. This includes crushing, milling, flotation, dewatering, concentrating handling and trucking, site services (power, air, and water), and administration costs.

G&A costs cover flights, camp accommodation, and site administration.

Table 1.5 Operating costs summary

Operating Cost	Unit	Value
Processing	A$/t of mineralized material	26.30
Underground mining costs	A$/t of material mined	50.00
Open pit mining costs	A$/t of material mined	3.08
G&A costs	A$/t of mineralized material	6.24

Capital costs were estimated based on information collected in Q4 2018, benchmarking information and capital costs from similar size equipment.

Capital expenditure for the project are presented in Table 1.6.

Table 1.6 Capital expenditure summary

	Life-of-mine (A$M)	Initial (A$M)	Sustaining (A$M)
Open pit mining	18.3	18.3	-
Underground mining	37.0	-	37.0
Processing	72.0	69.9	2.1
Infrastructure	40.1	39.6	1.2
Indirects	32.3	32.3	-
Closure costs	14.5	-	14.5
Contingency	14.1	10.2	3.9
Total	**228.9**	**170.3**	**58.7**

1.15 Economic analysis

All currency is in A$ unless otherwise stated. Foreign exchange rates were applied as required. Pricing submitted in US dollars (US$) were converted to A$ dollars using the exchange rate A$1:US$0.75.

The cost estimate was prepared with a base date of 2019 and does not include any escalation beyond this date. For economic analysis (Net Present Value) all costs and revenues are discounted at 8% from the base date. Metal prices were selected in discussion with VTT, and in keeping with three year LME averages.

A corporate tax rate of 30% is applied as the mining income will be earned in Australia. A 1.5% NSR royalty is paid to the vendor and the graduated Queensland state royalties for lead and zinc are paid at a rate of 5% based on the metal prices assumed, less allowable deductions for in state refining.

AMC conducted a high-level economic assessment of the proposed open pit and underground operations of the Pegmont Deposit. The project is projected to generate approximately A$201M pre-tax NPV and A$124M post-tax NPV at 8% discount rate, pre-tax IRR of 32% and post-tax IRR of 24%. A summary of the potential economic outcome of the project is presented in Table 1.7.

Table 1.7 Summary of potential economic results

	Unit	Base case
Cash cost	$/lb payable lead	0.65
AISC cost	$/t lb payable lead	0.71
Gross sales revenue	$M	1,826
Realization costs (royalties, transport smelting)	$M	398
Site operating costs (mining, processing, G&A)	$M	788
EBITDA	$M	640
Total taxes	$M	123
EBDA	$M	517
	% Lead	73.4
Revenue split by commodity	% Zinc	25.3
	% Silver	1.3
Pre-tax NPV at 8%	$M	201
Pre-tax IRR	%	31
Pre-tax payback period	Years	2.7
LOM cash flows (Undiscounted)	$M	288
After-tax NPV at 8%	$M	124
After-tax IRR	%	24
After-tax payback period	**Years**	**3.5**

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

1.16 Conclusions and recommendations

Preliminary economic analysis indicate that the Project shows an attractive potential and should be progressed to the next stage of study.

The PEA shows a pre-tax NPV of A$201M, and post-tax NPV of A$124M and associated post-tax IRR of 24% and payback of 3.5 years.

The Pegmont deposit has been explored in phases over a long period of time by a number of companies. Data collected since 1971 forms the basis of the Mineral Resource estimate, and while the early data is not supported by all required back-up, the later data is of good quality. Verification work by VTT has validated to earlier data for use in Mineral Resource estimation, and this work continues.

The geological model for Pegmont is reasonably robust and well understood. There is a total of 588 drillholes in the area of the deposit, of which 526 have been used in the Mineral Resource estimate. The data is based on a mix of RC and DD, with some percussion drilling used in the early years.

The following recommendations are drawn from the QP's observations as well as input from and discussions with VTT. The majority of these activities are designed to attain best practices and should be embedded in the way the project is run. Indicative cost for conducting some of the proposed recommendations is highlighted in the text.

1.16.1 Geology

1.16.1.1 Data collection

- Collection of core recovery and RQD data on consistent intervals (for example, the assay interval) rather than on the logged geological intervals currently used. It is difficult to compile meaningful spatial statistics for recovery and RQD across non-uniform intervals.
- Continuation of verification of historical (pre-VTT) data.

1.16.1.2 QA/QC protocol

- Generation of a matrix matched CRM to better accommodate the ratio of lead to zinc and the silver values seen at Pegmont.
- Continue with the assaying of duplicate pulps at an umpire laboratory as a further check on laboratory accuracy.
- Re-assessment of assay fail criteria to enable earlier detection of problematic results.
- Generation of a crushed and homogenized bulk sample of quartz float material for use as blank material.
- A selection of RC holes should be twinned by drilling proximal (within 5 m) DD holes to verify historic drilling program assays where QA/QC data is absent or indicates some issues of bias or uncertainty.

1.16.1.3 Modelling

- Incorporate the use of sulphur assays of the mineralized zones to refine the boundary between mineralized zone oxidation states oxide / transition and transition / sulphide. Rebuild these modelled surfaces.
- For the purpose of accurately assigning dilution, build low grade mineralization domains after investigating the selection of the grade boundaries of mineralized domains.

1.16.1.4 Drilling general

- Continue use of a mix of RC drilling and DD through the target horizon.
- Further drilling should aim to establish the controls on grade distribution in the oxide/transition boundary and improve the interpretation of sub-domains to control the distribution of high-grades.
- This could be combined with collection of samples for metallurgical testing and have laboratory density measurements performed on full core sticks. Also twinning adjacent to any poorer performing historical QA/QC results should be considered.
- Continued collection of geotechnical and hydrogeological data collection in subsequent drilling and core logging activities.

- Drill selected drillholes below the current target horizon as the presence of additional BIFs below the known occurrences should not be discounted. The use of downhole geophysical techniques that potentially could identify off-hole mineralized BIFs should be investigated.

1.16.1.5 Exploration program

Continue drilling to further evaluate the deposit. Activities are to both infill the current Mineral Resource to increase the level of confidence and upgrade the non-classified portion of the model.

- Undertake drilling to inflll data gaps between the Zone 4 and the two Zone 3 underground panels, approximately 4,400 m has been designed and costed by VTT at approximately A$910,000.
- Further follow up exploration drilling of copper intersection and TEM anomaly on EPM14491, three RC holes drilled in the opposite direction to the previous drilling should be considered, approximately A$80,000.
- Recommendations in earlier sections above in relation to data collection should be undertaken during any further drilling to maximize the value of the data collected from each drillhole, such that it can be used in any subsequent study work.
- Exploration drilling outside of the resource area for extensions and new zones; approximately A$150,000 is required to test immediate targets.
- Expand detailed outcrop geological mapping, A$40,000.

1.16.2 Processing

The following recommendations are made for consideration as the project progresses to the next phase:

- Continue metallurgical test work programs, including variability test work, estimated at approximately A$100,000.
- Filtration testing on the concentrates to support filter sizing and selection. Cost approximately A$5,000.
- Additional comminution tests are recommended including specific tests such as the integrated JK drop weight / SMC test to determine the AG / SAG mill parameters of DWi, Axb, Mia, Mib, Mic, and t_a. This is required to assess the suitability of the Pegmont ore to AG / SAG milling options (approximate cost A$30,000).
- Zone 1 Transitional program to cover the first open pit material. The program would include A Bond Rod Mill Index, a Bond Ball Mill Index and Bond Abrasion index (Ai) for this initial material. Batch rougher and cleaner flotation tests following T1092 program including some rougher kinetic tests. Lock cycle test on the ore zone. Mineralogy of the feed sample ground to a P80 of 106 µm and static settling tests on the lock cycle products (approximate cost A$77,500).
- Bond Crushing Work Index, Bond Rod Mill Work Index, Bond Ball Mill Work Index, and Bond Abrasion Index tests should also be undertaken in conjunction with the JKMRC tests to confirm the current process design and examine the amenability of a primary crush with AG / SAG mill option (approximate cost A$15,000).
- Future flotation test work is also recommended to better define rougher flotation kinetics to investigate use of flash flotation for recovery of fast floating galena, the effect of grind and regrind sizes on both lead and zinc flotation, optimization of zinc grade and recovery and to establish the effects of site water, mild steel media and sample aging (oxidation) (approximate cost A$50,000).
- Next phase of lock cycle flotation tests using site water is recommended (approximate cost A$39,000).

- Filtration test work will need to be undertaken on the produced concentrates to confirm the size of the selected filters. As no work has been done to date, a database of similar regrind size concentrates for both lead and zinc to size the current selection was used (approximate cost A$10,500).
- Thickening test work on the concentrates is also recommended to assess viscosity which may impact on both pumping and filtration rates (approximate cost A$8,000).

1.16.3 Project implementation

As the project progresses to the next phase further consideration should be given to the sourcing of used equipment that may reduce the project implementation duration or costs. In particular, the Ball mill, concentrate filters, and the accommodation village.

1.16.4 Open pit mining

AMC recommends that the following aspects are examined in the next study stage:

- AMC recommends that a dilution study is conducted in the next stage of study to ascertain the anticipated mining dilution and ore recovery in combination with the most appropriate mining fleet and associated costs (A$50,000).
- A geotechnical program should be continued to collect additional data for wall angle stability analysis (A$500,000).
- A geotechnical study should be undertaken to understand the offset distance to be left between the open pit and underground workings (crown pillar) and pillars for the tailings pit (A$20,000).
- A dump stability analysis should be undertaken for all waste dumps especially waste dump 2 which is to cover the BHZ tailings pit (A$20,000).
- AMC recommends that quotes from Australian mining contractors are collected to firm up the mining costs estimates for the open pit operations (A$10,000).
- Hydrological and hydrogeological studies should be conducted to better define dewatering requirements for the open pit and underground workings (A$100,000).

1.16.5 Underground mining

AMC makes the following recommendations for the underground mine:

- Operating costs are based on benchmarking data, AMC recommends that actual quotes be obtained from mining contractors for the next level of study (A$10,000).
- Further work is required to obtain sufficient geotechnical information to support the design criteria assumptions for mining recovery factors and mine design (A$100,000).
- Additional exploration drilling should be carried out with an aim to increasing the confidence in the underground Mineral Resources and increasing the potential throughput for the underground mine in order to fill the mill, see Section 1.16.1.5.

1.16.6 Environmental and social

AMC recommends that the following studies are conducted for the next study stage:

- Conduct additional baseline studies as per Table 20.2 (A$150,000).
- Review progressive restoration potential (A$50,000).
- Undertake waste rock, ore, and residue characterization (A$100,000).

Contents

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2 Introduction

2.1 General and terms of reference

This Technical Report (Report) on the Pegmont Property (Property) has been prepared by AMC Mining Consultants (Canada) Ltd. (AMC) of Vancouver, Canada on behalf of Vendetta Mining Corp (VTT or the Company), of Vancouver, Canada. This report is an update to an earlier report titled "Technical Report on Pegmont Resource Update June 2017, Queensland, Australia for Vendetta Mining Corp," written by AMC and with an effective date of 22 June 2017 (2017 AMC Technical Report). The report has been prepared in accordance with the requirements of National Instrument 43-101 (NI 43-101) "Standards of Disclosure for Mineral Projects" and the Canadian Securities Administrators (CSA) for lodgement on CSA's System for Electronic Document Analysis and Retrieval (SEDAR).

The issuer VTT is a limited liability public company that was incorporated under the Business Corporations Act (British Columbia) and was listed on the TSX Venture Exchange in 18 October 2013.

2.2 Qualification of authors

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

Table 2.1 Persons who prepared or contributed to this Technical Report

Qualified Persons responsible for the preparation of this Technical Report						
Qualified Person	**Position**	**Employer**	**Independent of VTT**	**Date of last site visit**	**Professional designation**	**Sections of report**
Mr J M Shannon	General Manager / Principal Geologist	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	P.Geo. (ON & BC)	3 – 9, 23, and 24, and parts of 1, 2, 25, 26, and 27
Ms M Angus	Senior Geologist	AMC Consultants Pty Ltd	Yes	16 – 17 May 2017	MAIG	10 – 12 and parts of 1, 25, 26, and 27
Ms D Nussipakynova	Principal Geologist	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	P.Geo. (ON & BC)	14 and parts of 1, 25, 26, and 27
Mr P Lebleu	Open Pit Manager / Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	13– 14 June 2018	P.Eng. (BC), M.Eng.	15, 19, 20, and 22, and parts of 1, 16, 18, 21, 25, 26, and 27
Mr G Methven	Underground Manager / Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	P.Eng. (BC)	Parts of 1, 16, 21, 25, 26, and 27
Mr B Mulvihill	Senior Process Engineer	GR Engineering Services Limited	Yes	No visit	MAusIMM (CP Met), RPEQ	13 and 17, and parts of 1, 2, 18, 21, 25, 26, and 27

Other Experts who have assisted the Qualified Persons						
Expert	**Position**	**Employer**	**Independent of VTT**	**Visited site**	**Professional designation**	**Sections of report**
Mr Peter Voulgaris	Principal	Vendetta Mining Corp	No	Yes	MAusIMM, MAIG	All
Mr Matthew Wilson	Project Manager	GR Engineering Services Limited	Yes	6 – 8 Aug 2016	B.Tech (M/E.) T.M.I.E.Aust	18 and parts of 21
Mr Sean Sara	Engineering Manager / Senor Process Engineer	GR Engineering Services Limited	Yes	6 – 8 Aug 2016	MAusIMM	13 and 17, and parts of 21

An inspection of the Property was undertaken by QP, Ms Maree Angus from the AMC Brisbane office, on 16 – 17 May 2017, accompanied by Mr David Esser of VTT. The scope of the visit covered the geology, data collection, and sampling aspects of the Property, including inspections of drill core and old drill sites. In addition, Mr Philippe Lebleu made a visit to site on 13 – 14 June 2018 to review site aspects as they may affect mining and infrastructure.

2.3 Sources of information

The sources of information are listed in Section 27 References. The text for parts of Sections 4 to 10 were supplied in draft form by Peter Voulgaris and validated and edited by the QPs. Some of the information has been carried through from or added to from the earlier AMC Technical Reports, Pegmont Resource Update 2017, February 2014, (2014 AMC Technical Report) and Pegmont Mineral Resource Update June 2017, (2017 AMC Technical Report).

The costs are generally shown in Australian dollars (A$), and the US dollar (US$) exchange rate where used was A$1 = US$0.75. The Canadian dollar (C$) exchange rate was C$1 = A$1.06.

2.4 Effective date

This report is effective as of 21 January 2019.

VTT was provided with a draft of this report to review for factual content and conformity with the brief.

3 Reliance on other experts

The QP's have relied, in respect of legal aspects, upon the work of the Experts listed below. To the extent permitted under NI 43-101, the QP's disclaim responsibility for the relevant section of the Technical Report.

The following disclosure is made in respect of this Expert:

- Stacey Venter of Hetherington, Exploration Services Pty Ltd. of Brisbane, Australia.

Report, opinion, or statement relied upon:

- Tenement Obligations Report for Period 1 December 2018 to 31 March 2019 dated 28 November 2018.

Extent of reliance:

- Full reliance.

Portion of Technical Report to which disclaimer applies:

- Section 4.3.

The following disclosure is made in respect of this Expert:

- AARC Environmental Solutions (AARC).

Report, opinion, or statement relied upon:

- Relevant legislation, approvals and permitting, summary of environmental constraints, native title and cultural heritage, closure, and remediation.

Extent of reliance:

- Full reliance following a review by the Qualified Person (QP).

Portion of Technical Report to which disclaimer applies:

- Section 20 Environmental Studies, Permitting, and Social Impact.

There are no other reports, opinions, or statements of legal or other experts on which the QP's have relied.

4 Property description and location

4.1 Property location

The Property is located in north-west Queensland, Australia, approximately 175 km south-east of Mount Isa, 130 km south-south-east of Cloncurry, and 700 km west-south-west of Townsville. The project is at approximately latitude 21° 51′ south, longitude 140° 41′ east. The Property location within Queensland is illustrated in Figure 4.1.

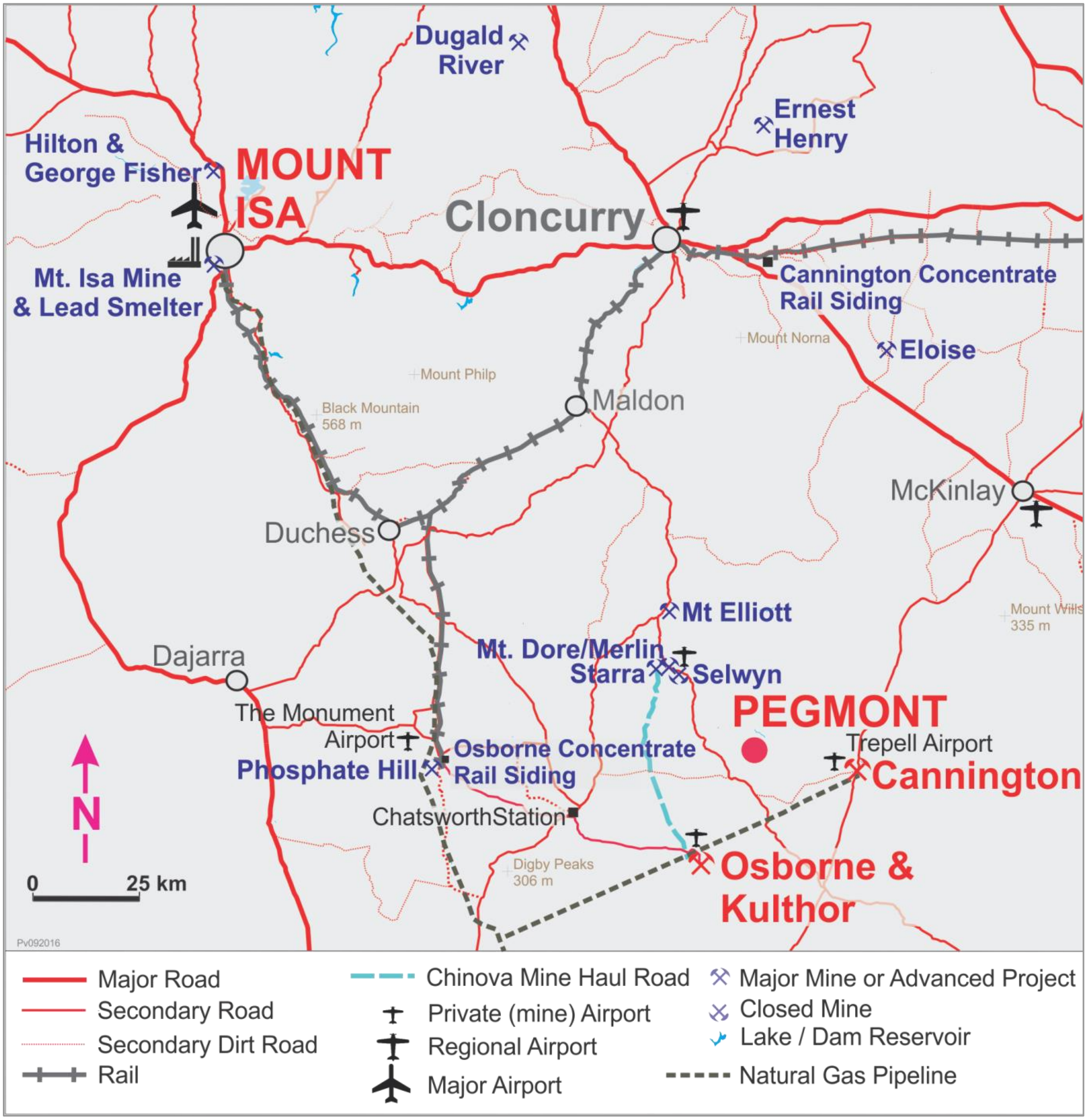
Figure 4.1 Location of Pegmont property

Source: The State of Queensland (Department of Natural Resources and Water) 2007.

It is situated 25 km west of South32 Limited's (South32) Cannington Silver Lead Zinc Mine, 28 km north-northeast of Chinova Resources Pty Ltd (Chinova)'s Osborne Copper Gold operations, and 27 km south-east of Chinova's Starra Gold-Copper Mine and advanced exploration projects at Merlin and Mount Dore.

Figure 4.2 shows the Property location with reference to infrastructure and centers in the region.

Figure 4.2 Mount Isa – Cloncurry region



Source: Provided by VTT and compiled from Gov. of Queensland sources.

4.2 Property description and ownership

The Property consists of both mining and exploration leases which are 100% owned by Pegmont Mines Ltd (PML). VTT has executed an Option Agreement and Royalty Agreement with PML to acquire 100% ownership of the project, which includes a pre-payment of future royalty payments which is discussed in Section 4.6.

The Option Agreement was originally executed in August 2014, it was amended in December 2016. As of 6 November 2018, VTT has made all cash payments and met all exploration commitments under the terms of the Option Agreement.

In November 2018, VTT and PML amended the Royalty Agreement, providing a mechanism for VTT to extend the payment of the pre-paid royalty by 12 months by making additional payments and reductions in the credit against future royalties as described below:

- In the event the Pre-Paid Royalty is paid between 1 – 30 November 2018, the Company will make an additional payment of A$50,000 and the Companies credit against the future royalties will be reduced from A$5,250,000 to $5,000,0000.
- In the event the Pre-Paid Royalty is not paid by 30 November 2018, the Company will make a payment of A$100,000 on 31 December 2018 and in doing so the Vendor has been granted an extension to 31 March 2019.
- In the event the Pre-Paid Royalty is not paid by 31 March 2019, the Company will pay an additional A$300,000 on 1 April 2019 and in doing do so the Vendor has granted an extension to 6 May 2019.
- In the event the Pre-Paid Royalty is not paid by 6 May 2019, the Company will pay an additional A$350,000 and the Companies credit against the future royalty will be reduced from A$5,000,000 to $4,500,0000 and in doing do so the Vendor has granted an extension to 6 November 2019.

The Option is only exercised once the Pre-Paid Royalty and any additional payments, as defined above are made.

4.3 Land tenure

The legislative framework for exploration, development, and mining tenure is administered by the State of Queensland through the *Mineral Resources Act, 1989*.

An Exploration Permits for Minerals (EPM) is granted under the act, and:

- Is issued for the purpose of exploration.
- Allows the holder to take action to determine the existence, quality, and quantity of minerals on, in or under land by methods which include prospecting, geophysical surveys, drilling, and sampling and testing of materials to determine mineral bearing capacity or properties of mineralization.
- May eventually lead to an application for a mineral development license or mining lease.
- Can be granted for a period of up to five years.
- Can be renewed.
- Rent is payable at a rate of A$150.50 per sub-block.
- EPMs are granted over blocks and / or sub-blocks. Each block is approximately 75 km^2 and is divided into 25 sub-blocks, each approximately 2.8 km^2, approximately one minute of latitude and one minute of longitude. Blocks are numbered in sequence and sub-blocks are identified in alphabetical order. An EPM can consist only of sub-blocks. Each sub-block is approximately 2.8 km^2 in area.

- Relinquishment of sub-blocks over time is a condition of granting. A 40% sub-block reduction is due by the end of the first three years after the permit is granted. A further 50% of the remaining sub-block is to be reduced at the end of the first 5 years upon renewal. Application for sub-block retention can be made.
- Minimum expenditure requirements are a minimum of A$1,500 per sub-block per year for minerals aggregated over the EPM area. Expenditure requirements increase after the second year of the granting of the EPM. The assessment of the work program is determined on the appropriateness of the program, on a year-by-year basis. Generally, over the permit term, the expenditure per sub-block is expected to increase as the size of the permit is reduced and the knowledge of the area and geology develops.
- It is a requirement to submit annual, relinquishment, and final reports to the Queensland Department of Natural Resources and Mines.

A Mining Lease (ML) is granted under the act for mining operations and:

- Entitles the holder to machine-mine specified minerals and carry out activities associated with mining or promoting the activity of mining.
- Is not restricted to a maximum term, this is determined in accordance with the amount of reserves identified and the projected mine life.
- Can be granted for those minerals specified in either the prospecting permit, exploration permit, or mineral development license held prior to the grant of the lease.

The Property currently consists of three MLs; 2623, 2621, and 2620, and one EPM; 26210, which together cover approximately 8,380 hectares (ha), of which 388.5 ha are covered by the three mining leases. The MLs are entirely located within the exploration permit. This is shown along with surrounding EPMs in Figure 4.3. Details of the MLs and EPM are given in Table 4.1.

The three MLs are valid for the purpose of mining lead, zinc, silver, and copper. Iron is not listed as a specified mineral for the existing MLs, but it is understood that the addition of iron to the MLs should only be a formality. ML boundaries were resurveyed in 2010 to confirm their location. This was carried out by Lodewyk Pty Ltd of Mount Isa, who is an independent licensed surveyor.

The EPM is valid for the exploration of all minerals except coal. The EPM is currently comprised of 26 sub-blocks. This must be reduced to 16 at the end of the third year, which occurs on 21 August 2020.

Table 4.1 Mining leases and exploration permits

<table>
<tr><th>Tenure type</th><th>ML or EPM number</th><th>Blocks</th><th>Sub-blocks</th><th>Name</th><th>Area (ha)</th><th>Granted</th><th>Expires / renewals</th></tr>
<tr><td rowspan="3">ML</td><td>2623</td><td rowspan="3">N/A</td><td rowspan="3">N/A</td><td>Pegmont No. 4</td><td>129.5</td><td>24 Jan 1974</td><td>31 Jan 2022</td></tr>
<tr><td>2621</td><td>Pegmont No. 2</td><td>129.5</td><td>24 Jan 1974</td><td>31 Jan 2022</td></tr>
<tr><td>2620</td><td>Pegmont No. 1</td><td>129.5</td><td>24 Jan 1974</td><td>31 Jan 2022</td></tr>
<tr><td rowspan="4">EPM</td><td rowspan="4">26210</td><td>CLON1544</td><td>E, J, K, M, N, O, P, S, U, X, Y</td><td rowspan="4">Pegmont</td><td rowspan="4">8,290</td><td rowspan="4">22 Aug 2017</td><td rowspan="4">21 Aug 2022</td></tr>
<tr><td>CLON1545</td><td>A, F, L, Q</td></tr>
<tr><td>CLON1616</td><td>E</td></tr>
<tr><td>CLON1617</td><td>A, B, C, F, G, H, L, M, N, Q</td></tr>
</table>

Minimum annual exploration expenditure requirements and actual expenditures for the current EPM are detailed in Table 4.2. All previous exploration expenditure commitments have been met.

There are no annual expenditure requirements on the three MLs.

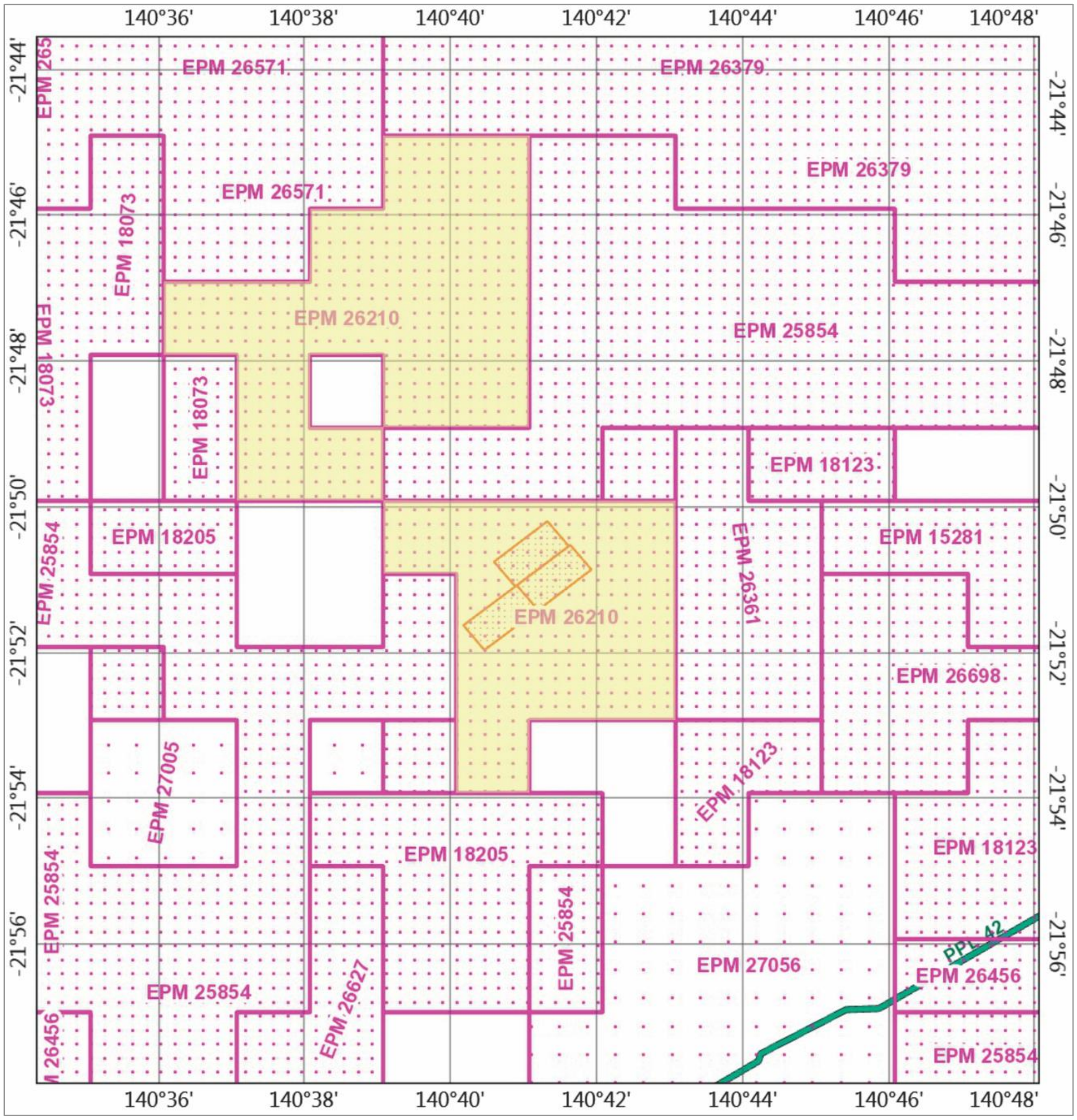
Table 4.2 EPM 26210 exploration permit minimum expenditure requirements

Year	Period	Minimum expenditure (A$)	Actual expenditure (A$)
1	22 Aug 2017 – 21 Aug 2018	500,000	873,284
2	22 Aug 2018 – 21 Aug 2019	550,000	N/A
3	22 Aug 2019 – 21 Aug 2020	600,000	N/A
4	22 Aug 2020 – 21 Aug 2021	650,000	N/A
5	22 Aug 2021 – 21 Aug 2022	680,000	N/A
Total		**2,980,000**	

Annual rental paid to the Government of Queensland for the three MLs totals about A$25,000 year and for the EPM about A$4,000.

Other government payments include the Cloncurry Shire Council rates payable on the three-granted mining leases every six months at a cost of about A$5,000, an annual Queensland Government health and safety levy is determined based on the number of workers during the preceding year, the 2018 charge was about $12,000, an annual fee of about $650 for administration of the environmental permit and a minor charge for groundwater extraction of less than $100.

Figure 4.3 Pegmont EPM 26210 area shown in yellow



Source: From Queensland Interactive Resource Tenure Map as of 15 December 2018.

4.4 Surface rights

All work areas related to the project are located on a Pastoral Lease held by Argylla Mountains Pty. (Argylla). VTT and Argylla have a land access agreement in place. In the case of the MLs which were in existence before the pastoral lease was granted there is no requirement for land owner compensation. Land access compensation relates only to the EPM which was granted after the Pastoral Lease, the schedule of compensation to Argylla is described below:

- A$500 plus GST per kilometre of access track constructed.
- A$150 plus GST per auger drillhole completed.
- A$200 plus GST per RC or rotary air blasthole (RAB) completed.
- A$250 plus GST per diamond core hole completed.

There are no known risks to the titles that may affect access or rights to the MLs or EPM that comprise the Pegmont project.

4.5 Native title

Native title processes may be required for the valid grant of tenements granted after 1 January 1994. Native title processes will not be required where native title has been 'extinguished' over the land subject of the tenement – for example by an earlier vesting of the land in freehold.

If native title has not been extinguished, it may continue to exist – triggering the need for a native title process before a tenement may be validly granted.

Searches with the National Native Title Tribunal suggest that the Yulluna People (QUD189 / 2010) have been determined as the native title holders over part of the area of the EPMs and ML 2620, conditional upon a number of requirements, including the registration of certain Indigenous Land Use Agreements.

The MLs were granted prior to 1 January 1994. If the grants were otherwise valid, no native title process was required on commencement of the Native Title Act 1993 (NTA) – and the grants were 'validated' on commencement of the NTA. This extends to any renewal of the MLs on the same terms as the original grant.

A set of conditions (known as the Native Title Protection Conditions) have been attached to the EPM. The holder of the EPMs must comply with these conditions as if they were a condition of its grant.

If VTT pursues a future mining lease application out of the existing EPMs or seeks to vary its rights under or the terms of the MLs, a statutory negotiation process will need to be undertaken at that time if native title exists over any parts of the area of the relevant ML application or ML (as applicable).

4.6 Royalties

The Royalty Agreement with PML provides for a royalty to PML on future concentrate production from the Property of 1.50% of Net Smelter Return (NSR), subject to the credit of A$5.0M in favour of VTT. In the case where ore is sold rather than concentrate, a separate royalty formula allows for a royalty of $1.05 per tonne of ore sold, indexed to lead prices, again subject to the A$5.25M credit. Where ore that is sold contains silver at concentrations above 64 ppm, an additional royalty amount is payable, starting at A$0.06 per gram, indexed to the price of silver.

VTT retains the right of first refusal should PML choose to sell the Royalty.

Queensland State Mineral legislation imposes a royalty on the sale of minerals. Mineral royalties in Queensland operate under a variable rate royalty system. For example, for lead prices at A$1,100/tonne the royalty rate is 2.5%, whereas for a lead price of A$2,500/tonne the royalty rate is 5%.

Lead and zinc are described in the act as prescribed minerals and are subject to reasonable variation in contained and payable metal. The deduction for lead concentrate is 3% and zinc concentrate attracts an 8% discount. Lead is paid at 95% of the assayed value and zinc at 85%. Silver in lead and zinc concentrate attracts a deduction of 100 g/t and is payable on 95% after the deduction for lead concentrate, and 60% for zinc concentrate.

Queensland Royalty discount applies for base metals processed within Queensland to a particular metal content, as prescribed by section 51 of the Mineral Resources Regulation 2013. Lead and zinc attract a "processing discount" of 25% and 35% respectively if processed to 95% purity in the state of Queensland.

The relevant price is the "LME Daily Official and Settlement Prices" "Cash Seller" daily price. The daily exchange rate used is the WM / Reuters Australian Fix 10:00 AM rate.

4.7 Existing environmental liabilities

Project environmental approvals are in the form of two Environmental Authorities (EA), the EPM (EPSX00957013) and the three MLs (EPSL0057813). They have been issued under the Environmental Protection Act 1994 (Qld) (the EP Act) by the Queensland Department of Resource and Environmental Management (DERM). The DERM is now defunct and this is now the responsibility of the Queensland Department of Environment and Heritage Protection (DEHP).

The Pegmont project is issued as a "Level 2" code compliant EA. The following criteria (schedule 1A of the Environmental Protection Regulation 1998) are used to determine the level of assessment required for an application for an EA for a standard exploration or mineral development project:

- The mining activities do not, or will not, cause more than 10 ha of any land to be significantly disturbed at any one time.
- No more than 5,000 m^2 are disturbed at any campsite at any one time.
- No more than 20 m^3 of any substance is extracted from each kilometre of any riverine area in any one year.
- The mining activities are not, or will not be, carried out in a category A or B Environmentally Sensitive Area.
- The mining activities do not include a Level 1 environmentally relevant activity.

These conditions restrict the activities that can be carried out on the Tenements. Any activities proposed to be carried out which do not meet the requirements of the relevant code of compliance will be unauthorized unless an amendment is made to the relevant EA.

There are separate codes for MLs and EPMs:

- Code of Environmental Compliance for Mining Lease Projects.
- Code of Environmental Compliance for Exploration and Mineral Development Projects.

As part of VTT's due diligence the DEHP provided written confirmation on 5 March 2014 that the register it maintains under the Environmental Protection Act 1994 (Qld) does not contain any record of environmental compliance issues against the EAs for the Tenements. There have been no compliance issues reported since.

The requirements for operations on the Property, what is currently in place, and the relevant legislation and authorities are discussed in Section 24.

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

5.1 Accessibility

The Property can be accessed from Mount Isa by sealed roads via Dajarra (Boulia Mount Isa Highway and Monument Dajarra Rd), and past Dajarra on secondary dirt roads (Chatsworth Phosphate Rd and Burnham Rd). From Cloncurry, the site can be accessed on a sealed road from the Flinders Highway south (Cloncurry Duchess Rd and Malbon Selwyn Rd) to Malbon, and then on secondary dirt roads. A third alternative route is via McKinlay on a sealed road to South32's Cannington Mine site (Toolebuc McKinlay Rd). For each of these three options the Property itself is then only assessable by a track created by local graziers and maintained by VTT, which require only occasional grading. Secondary gravel roads are well-maintained by Cloncurry Shire Council. See Figure 4.2 for detail.

There are several river and creek crossings through causeways, which can become impassable for relatively short periods of time (days) during the peak wet season.

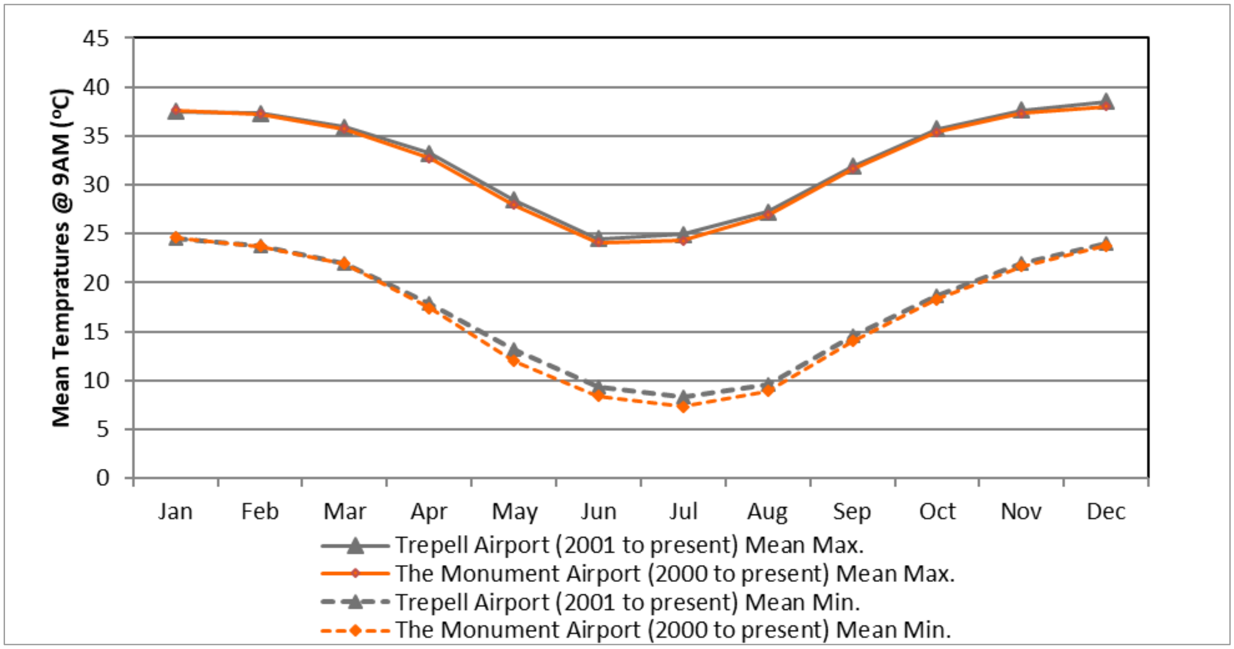
5.2 Climate

The Australian Bureau of Meteorology collects climate statistics (www.bom.gov.au/climate), as there are a number of historic and active weather stations within a 100 km radius of the Property. The closest currently operating stations are at the Trepell Airport, servicing the Cannington Mine, 21 km to the east and The Monument Airport, servicing the Phosphate Hill Mine, 79.5 km to the west. Rainfall measurements are also collected at Chatsworth Station, 43 km to the east-southeast of the Property.

Mean maximum and minimum temperatures at these two weather stations are nearly identical. The area has a semi-arid climate with temperatures varying between 7° and 25°C in winter, and from 25° to 39°C in summer. Mean temperatures at the Trepell and The Monument Airports are shown in Figure 5.1.

The weather is acceptable for exploration and mining operations year-round; however, field operations may be interrupted during heavy rainfall events, which typically occur in the summer, rendering access roads impassable due to soft ground conditions.

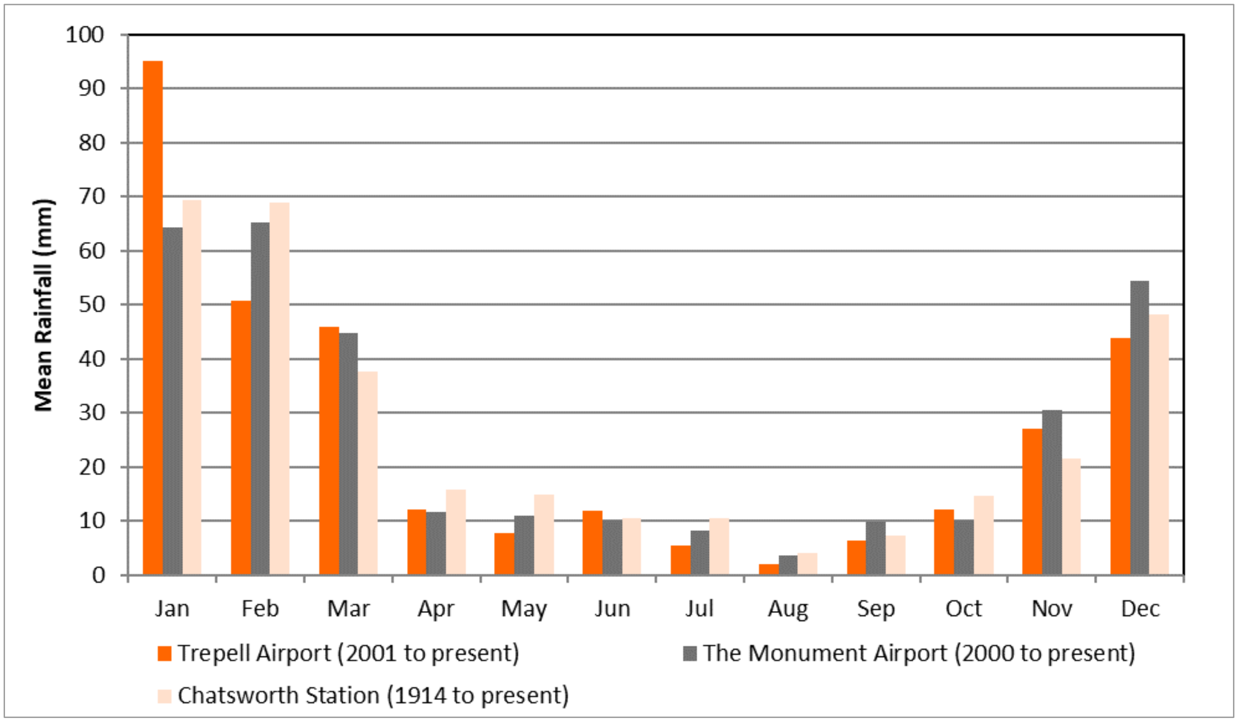
Figure 5.1 Monthly temperature statistics



Source: Australian Bureau of Meteorology (www.bom.gov.au/climate)

Average annual rainfall is around 350 mm but can be highly variable. The majority of the rainfall is during the annual monsoon season from November to March. Australian Bureau of Meteorology rainfall statistics for the weather stations at Trepell and The Monument Airports and Chatsworth Station are shown in Figure 5.2.

Figure 5.2 Monthly rainfall statistics



Source: Australian Bureau of Meteorology (www.bom.gov.au/climate)

5.3 Local resources and infrastructure

The Property is located in the Mount Isa Mineral Province, a region recognized as a world-class mining region, with more than a quarter of the world's lead and zinc Mineral Reserves, 5% of the world's silver Mineral Resources, and 1.5% of the world's copper resources. There is a large skilled workforce currently employed on other projects in the Mount Isa Mineral Province and at the city of Townsville, 700 km to the north-east. This is shown in Figure 4.2.

The Mount Isa Mineral Province is connected to Townsville and the coast via the Great Northern Rail Line and the Flinders Highway. The Great Northern Rail Line is operated by Queensland Rail, which is owned by the Queensland Government. A deep-sea port is operated in Townsville, which services the metal mining, sugar, and fertilizer industries. Korea Zinc (Sun Metals) operates a zinc and lead smelter located at Townsville. In Mount Isa, Glencore operates a lead smelter.

The closest population center to Pegmont is the town of Cloncurry (population approximately 3,500), 90 km to the north. The larger regional center of Mount Isa (population approximately 23,000) is 150 km to the north-west of the project area. Cloncurry is serviced by commercial aircraft from Townsville and Brisbane, while Mount Isa can be reached by commercial aircraft from Townsville, Cairns, and Brisbane.

There are two existing concentrate loading sidings on the Great Northern Rail Line, one at the Phosphate Hill siding, approximately 80 km to the west of Pegmont. Phosphate Hill is the site of the Duchess Phosphate Deposit, Australia's largest operating phosphate mine and is also the location of the Osborne Mine copper-gold concentrate rail loading siding. Lead-silver and zinc concentrate from the Cannington Mine are hauled on road train to the second possible rail loading facility at Yurbi, about 15 km east of the town of Cloncurry.

The dominant land use in the area of the Pegmont tenements is mining and cattle grazing. A long history of mining is established in the Cloncurry Shire, and several operating and historic mines are located less than an hour drive from the Pegmont project site, including South32's Cannington Lead-Silver-Zinc mine, Chinova's Osborne Copper-Gold operation, Chinova's Starra Gold-Copper mine, and Merlyn Moly-Rhenium development project.

The Cannington Lateral natural gas pipeline lies about 16 km to the south of Pegmont and provides natural gas for power generation to the Osborne and Cannington Mines.

Temporary infrastructure to support ongoing exploration activities exists at the project site in the form of a ten-bed accommodation block, mess facility, ablution block, office, two diesel generators, and core yard. On-site there is the gravel airstrip which is not currently maintained and an emergency helicopter landing pad.

Sealed all weather air strips are accessible; at Chinova's Osborne Mine, 28 km to the south-southwest and at the Selwyn Project about 26 km to the north-west, and at South32's Trepell Airport about 25 km to the east.

5.4 Physiography and vegetation

The project area is part of an extensive flat alluvial plain, at an average of 300 m above sea level. Low mesas, rising 10 to 20 m above the plain are capped by thin sediments or by residual laterite and silcrete. Lateritic ironstone rubble covers much of the black soil plain. A black manganese stained gossan at Mount Lucas is 10 m above the plain, other gossans are not as prominent.

The current land uses on the Project site include moderate intensity cattle grazing and exploration mining activities. An Agricultural dam exists near the eastern border of the Project site along with a network of fences throughout. A large portion of the Project site is unsuitable for cattle grazing due to the absence of appropriate fodder species and rugged terrain. These areas are characterized as jump-ups or mesas primarily dominated by Spinifex grass species.

Figure 5.3 Mt Lucas gossan outcrop

Source: VTT.

Pegmont lies midway between two ephemeral creeks; Sandy Creek and Bustard Creek, which are both draining to the south-east, eventually joining the Hamilton River. One agricultural dam is located within the EPM.

Vegetation consists of tussock grassland (spinifex and kerosene grass), sparse eucalypts, and patches of acacia and grevillea trees. Drainage features are marked by denser tree growth.

Quaternary residual and fluvial alluvial sediments are present along all significant drainages.

6 History

The Property has been held by way of EPMs and Mining Licenses (MLs) which overlapped at different times and sometimes with common ownership, sometimes not. Thus, the discussion below will deal with each separately where applicable. Note that QLD government changed the designation of exploration leases from Authority to Prospect (APT) to Exploration Permit for Minerals (EPM), thus both terms are used in the text.

6.1 Ownership

6.1.1 Exploration permits

Prospector Mr Colin Harris was employed by Placer Prospecting (Australia) Pty Ltd. (antecedent of Placer Pacific Ltd.) on a finder's reward basis to prospect for copper in the Cloncurry district. In 1970 he discovered outcropping gossanous lead-rich ironstone (Mount Lucas) at Pegmont. Placer took up Authority to Prospect (ATP) 1041 M and applied for 18 mining licenses centered on the outcropping gossans at Mount Lucas in 1972. The mining licenses were granted in 1974, however, in 1973 Placer had surrendered in the surrounding EPM 1041 M.

A joint venture made up of Newmont (operator), ICI Australia Ltd. (ICI), and Dampier Mining (BHP Minerals) applied for an exploration permit surrounding Placer's mining lease applications, ATP 1334 M was granted in December 1973. The joint venture also had a farm-in agreement with Placer over the mining leases. Placer sold their interest to Mimets Exploration Pty Ltd. (Mount Isa Mines Ltd.) in 1974. Later, ICI withdrew from the joint venture, leaving BHP Minerals, Newmont (later Newcrest Mining through the merger of Newmont Australia Ltd and BHP Gold Mines Ltd), and Mount Isa Mines as the joint venture partners.

ATP 1334 M was relinquished, however, the joint venture took up essentially the same area again in February 1976 with ATP 1603 M, as a number of geochemical and geophysical anomalies remained to be tested. The joint venture relinquished 1603 M in July 1977, whilst retaining the mining licenses.

Amoco Exploration held EPM 2326 M, which was largely centered to the east of Pegmont, and only overlapped one sub-block with the current Pegmont project area.

In the public record there is no evidence of work outside of the mining leases until Carpentaria Exploration Co. Pty Ltd. obtained ATP 3362 M in October 1982. The permit lay approximately 1.5 km to the south and east of the Pegmont deposit, partially overlapping the current exploration permits. The ground was taken up with the specific objective of investigating three magnetic anomalies, thought to be similar in character to Pegmont. The area to the east of Pegmont was relinquished in 1982 and the remainder in January 1984.

In September 1989 Perilya Mines N.L. obtained ATP 6029 M, which surrounded but excluded the Pegmont mining leases. The original 6029 M was larger than, but fully encompassed the current exploration permits. Perilya undertook work in their own right in 1989 but subsequently joint ventured the lease out to Aberfoyle Resources Ltd. (1990 to 1993), BHP Minerals (1995 to 1998) and North Ltd / Rio Tinto Exploration (1999 to 2001), before finally relinquishing the tenement in 2003.

During this time, the mining leases remained under the ownership of BHP Minerals, Mount Isa Mines, and Newcrest joint venture until April 1996 when PML, through its predecessor companies, purchased the Pegmont mining leases. In August 1998 PML entered into an earn-in agreement with North Ltd (North) whereby North could earn 75% interest in the mining licenses by completing a bankable feasibility study. North carried out geophysical and drilling programs but withdrew from the joint venture on 21 December 1999.

PML listed on the Newcastle Stock Exchange (NSX) in December 2000.

Prior to listing, PML signed a letter of intent with Billiton to provide an equity investment in PML for the exploration of deep targets (Pegmont Deeps) below the known mineralization where Billiton could elect to form a joint venture to earn up to 70% in Pegmont Deeps by spending A$4M within six years, however, Billiton were not satisfied with exploration results and they withdrew from the project in June 2001.

In August 2006, PML were granted exploration permits 15106 (Pegmont Extended) and 14491 (Pegmont Extended 2). EPM 15106 has been partially relinquished in 2008 which resulted in multiple non-contiguous areas. These outlying non-contiguous areas were relinquished in 2010, retaining ten contiguous sub-blocks centered over the Pegmont deposit. A further two contiguous sub-blocks were relinquished in 2013.

Cloncurry Metals Limited (CLU) entered into a two-year option agreement with PML on the EPMs and MLs in May 2007. The option agreement was for a total consideration of A$12M (A$9.6M cash and A$2.4M in shares), a minimum work commitment of 4,000 m of drilling and a 1.25% NSR royalty. CLU entered into a memorandum of understanding with BHP Billiton (BHPB) in September 2008. BHPB undertook due diligence which included infill drilling on the MLs, however, as a result of the global financial crisis and the reduction in budgets, BHPB withdrew from discussions with CLU and subsequently CLU also decided not to exercise its option rights over Pegmont.

The boundaries of antecedent exploration leases are never coincident with the current boundaries of the Property. Wherever possible, the summary provided in Table 6.2 of the ownership, operatorship, and description of the exploration activities relates specifically to the current two exploration permits. For details of any exploration activities of the current owners see Section 9, and for the resource development drilling history see Section 10.

6.1.2 Mining leases

The MLs have been consolidated and renamed over time. Table 6.1 gives a summary of the ownership of the PML mining leases held over the current EPMs that the Property now consists of.

Table 6.1 Mining Leases held over the current EPMs

ML number	Holder	Approximate dates	Land area (ha)
8067, 8068, 8069, 8070, 8071, 7942, 7943, 7944, 7923, 7924, 7941, 7945, 7946, 7947, 8448, 8449, 8450, & 8451	Placer Prospecting (Aust.) Pty Ltd.	1970 – 1975	2,331
2620, 2621, 2622, 2623, 2624, 2625, 2626, 2627, 2629, 2630, 2662, & 2663	BHP Minerals Pty Ltd, Mount Isa Mines Ltd, & Newcrest Mining Ltd.	1976 – 1996	1,554
2620, 2621, 2622, 2623, 2624, 2625, 2626, 2627, 2629, 2630, 2662, & 2663	Dakota Consolidated Mines NL & Pegasus Enterprises Ltd. (antecedents of Pegmont Mines NL)	1996 – 1997?	1,554
90119 (A consolidation of the 12 MLs)	Pegmont Mines NL	1997? – 2001	1,554
2623, 2620, & 2621 (Reinstatement of 3 original MLs and dropping of 9 MLs)	Pegmont Mines Ltd (name change from Pegmont Mines NL)	2001 – present	361.5

6.2 Exploration work

Prior to 1975 the Property had been tested with 100 shallow rotary and percussion drillholes and 35 diamond drillholes, of which 28 fall inside the current ML area. Most holes were drilled vertically.

The ownership history of the mining leases diverged from the exploration licenses after the Newmont joint venture sold the mining leases to the antecedents of PML in 1996.

The discussion below is in regard to work on the mining leases. The historical work on the EPMs is discussed in a tabular form in Table 6.2.

From 1975 to 1996 there was little exploration work of significance undertaken on the MLs. BHP Minerals drilled three holes to test two electromagnetic (EM) anomalies and one magnetic anomaly generated from programs of gravity, airborne and ground EM, aeromagnetic, and ground magnetic surveys. The holes were not drilled in the vicinity of the known mineralization and did not intersect any mineralization of significance. It was during the latter part of this period that the MLs and environs were covered by regional aeromagnetic and airborne EM (GEOTEM™) surveys.

In 1996 the joint venture dissolved and the MLs were sold to Pegasus Enterprises Ltd. and its wholly owned subsidiary PML. Initially PML undertook a review and reinterpretation of magnetic data and completed a program of reverse circulation (RC) drilling at the Gossan Lode.

From 1998 to 1999 North entered a joint venture with PML over the twelve MLs that existed at the time. North undertook the following work program, which focused on the shallow oxide / transition areas:

- Aerial photography at a scale of 1:10,000.
- A gravity survey on a 100 m x 100 m grid in-filled to 50 m in places.
- Three lines of Mise a la Masse (resistivity) surveying.
- Five lines of dipole-dipole induced polarization (IP) surveying over Mount Lucas and Bonanza areas.
- 520 line km of rapid sampling ground magnetic surveying.
- Down-hole EM surveying in one drillhole.
- The collection of 242 rock samples along traverses designed to pass through known mineralization.
- The drilling of nine diamond and six RC holes to test a range of geophysical targets and for down-dip extensions of mineralization at Mount Lucas and Bonanza prospects.

On the basis of its work, North concluded that the Pegmont MLs held limited potential for mineralization of interest to it and withdrew from the joint venture at the end of 1999.

During the "Pegmont Deeps" Billiton agreement, PML completed two drillholes during 2000 and PML focused its attention on metallurgical test work on composite samples from oxide and sulphide material.

In 2001, a three-dimensional IP survey over the Sharry Fault was undertaken. This showed a 1,200 m x 300 – 400 m wide moderate to strong intensity anomaly at an interpreted depth of about 275 m located about 500 m north of the outcropping Mount Lucas Lode. No field work was performed by PML until June – July 2005 when 15 drillholes were drilled for 1,072 m. The purpose of the drilling was to generate samples for metallurgical test work.

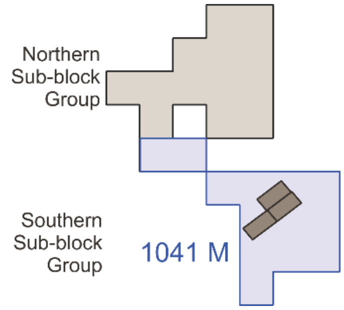
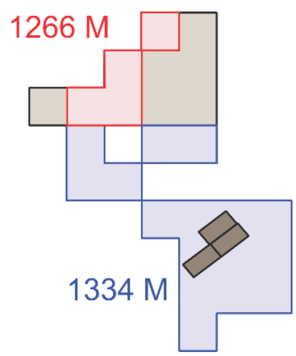
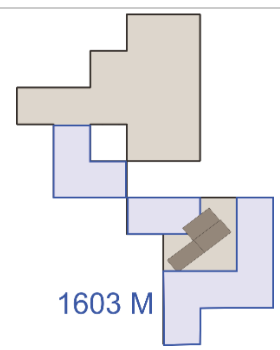
With the signing of the two-year option agreement with CLU, exploration activities increased. During 2007, 5,410 m of drilling was completed by PML and reimbursed by CLU. In 2008, with CLU as operator, drilling continued with an infill and extension program of 32 drillholes and a total of 7,319 m was completed. As discussed above, CLU withdrew having failed to attract BHPB with the onset of the financial crisis and down turn in metal prices.

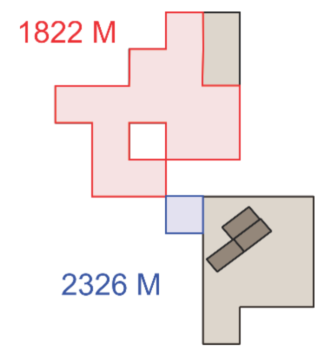
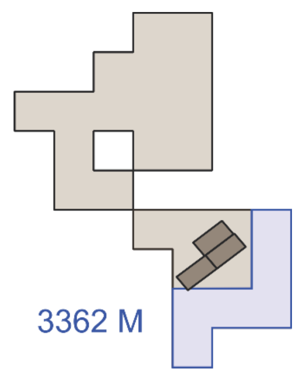
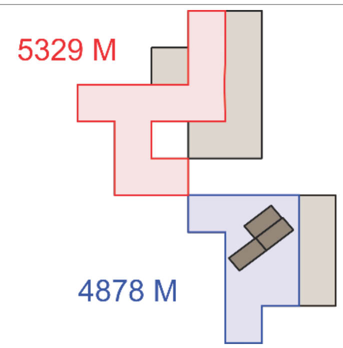
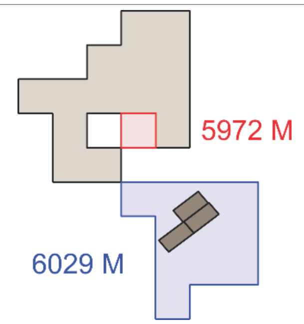
In 2009, PML completed four RC drillholes for 726 m in the south-east part of the deposit and a small ground magnetic survey was conducted. In 2010, drilling was initiated to test for extensions to the shallow oxidized lead zinc mineralization as identified in the 2007 ground magnetic survey where 10 drillholes were completed for 642 m. Further detailed ground magnetic surveys were conducted and followed up with a 47 drillhole infill program on a 50 x 50 m grid; a total of 2,937 m was completed.

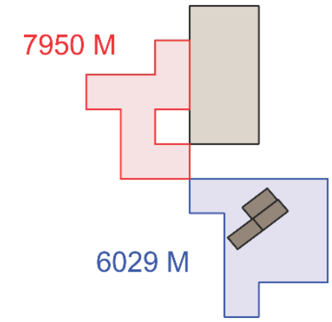
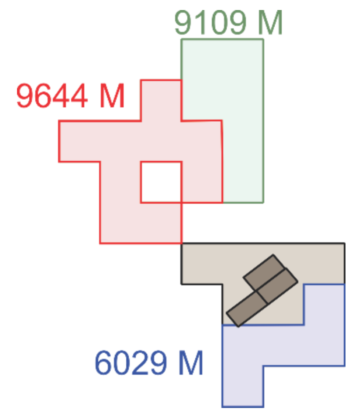
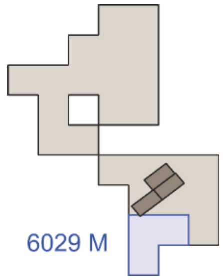
An infill program of 21 vertical RC drillholes was completed in 2013, being drilled on a 50 x 50 m grid for a total of 3,000 m.

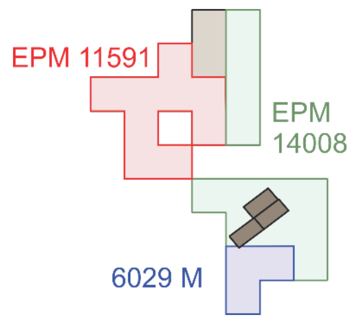
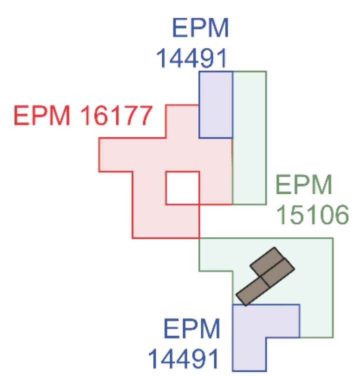
As the EPMs overlapped over time the exploration history as carried out on the Property is compiled in Table 6.2, which also shows the outline of the EPM(s) of the time relative to the Property. The information in the table is based on various open file reports and was compiled by VTT.

Table 6.2 Exploration history over the current EPMs

Historic EPM	Holder and approx. dates	Work performed	Historic EPMs Overlap with current project area (MLs dark grey and EPM light grey)
1041 M	Placer Prospecting 1972 – 1973	Northern Sub-block Group Two sub-blocks coincident with EPM 26210. Southern Sub-block Group Contains EPM 26210, Pegmont ML's excluded. • Air photo reconnaissance geological mapping 1:24,000. • Airborne magnetics, X-ray spectrometry, and very low frequency EM. • Ground follow up of selected anomalies with magnetics and auger drilling.	
1266 M	Newmont Aug 1973 – Aug 1976	Northern Sub-block Group **Four sub-blocks coincident with EPM 26210. of the current EPM 2610.** • Airborne magnetometer and scintillometer survey, quarter mile spaced lines at a height of 300 ft. • Preliminary field mapping. • No base metal mineralization identified.	
1334 M	Newmont Dec 1973 – Dec 1975	Southern Sub-block Group Contains EPM 26210, Pegmont ML's excluded. • Air photo reconnaissance geological mapping 1:25,000. • Airborne Input EM, magnetics, and gamma-ray spectrometry. • 17 residual input anomalies were selected for follow up, only anomaly "254.08" located on the current EPM 15106, "patchy gossan with up to 1600 ppm Cu, one percussion hole max value 352 ppm Cu. • Air-track Percussion drilling on traverse lines 800 m apart with a drillhole spacing of 25 m to 25 m maximum depth. One traverse is partially on the current EPM 14491. • Gossan prospecting search, including two gossans on the current EPM 15106.	
1603 M	Newmont Feb 1976 – Jul 1977	Southern & Northern Sub-block Groups Eleven sub-blocks coincident with EPM 26210, excluded the Pegmont MLs. • No fieldwork was performed on the current EPM 26210.	

Historic EPM	Holder and approx. dates	Work performed	Historic EPMs Overlap with current project area (MLs dark grey and EPM light grey)
1822 M	Amoco Minerals Australia Aug 1977 – Aug 1982	Northern Sub-block Group **Ten sub-blocks coincident with EPM 26210** • No work reported on these sub-blocks.	
2326 M	Amoco Minerals Australia Feb 1982 – Feb 1982	Southern Sub-block Group **One sub-block coincident with EPM 26210** • No work was performed on the single sub-block coincident with EPM 26210. • One aeromagnetic anomaly immediately on the southern boundary of EMP 26210, "Spartan Dan" received follow up ground magnetics and gravity surveys, vacuum auger drilling to sample bedrock geochemistry, and a single 150 m percussion hole drilled but abandoned above target depth due to poor drilling conditions.	
3362 M	Carpentaria Exploration Oct 1982 – Jan 1984	Southern Sub-block Group **Six sub-blocks coincident with EPM 26210** • EPM taken up to investigate 3 magnetic anomalies identified by Placer; "A1" (Spartan Dan), "A2" and "A3". A1 and A2 are located on the current EPM 14491, A3 is located on the current EPM 15106. • Geological reconnaissance suggested "A2" and "A3" were due to poorly outcropping granite. The eastern half of the EPM was then relinquished. • A1 was delineated with three ground magnetometer traverses. In 1983 it was drilled with a single percussion drillhole (PEX-01) drilled to a depth of 170 m. Magnetite and pyrite were intersected (8 m from 70 m, 2 m from 74 m) and with susceptibility values considered sufficient to explain the anomaly. Assays returned no significant results.	
5329 M	Placer Exploration Ltd Apr 1985 – Nov 1990	Northern Sub-block Group Nine sub-blocks coincident with EPM 26210.	
4878 M	William Edgar Mathews Aug 1987 – Dec 1988	Southern Sub-block Group Eight sub-blocks coincident with EPM 26210, **Pegmont MLs excluded** • No exploration report filed.	
5972 M	Metana Minerals N.L. Apr 1989 – Jan 1990	Northern Sub-block Group **One sub-blocks coincident with EPM 26210** • No work performed on single sub-block.	
6029 M	Perilya Mines Sep 1989 – Sep 1990	Southern Sub-block Group Contains EPM 26210, Pegmont MLs excluded. • No record of any field work being performed.	

Historic EPM	Holder and approx. dates	Work performed	Historic EPMs Overlap with current project area (MLs dark grey and EPM light grey)
7950 M	Placer Exploration Ltd Sep 1991 – 1993	Northern Sub-block Group **Overlapped 7 sub-blocks of EPM 26210** • No exploration report filed / available. • Combined airborne magnetic, electromagnetic (QUESTEM) survey.	
6029 M	Perilya Mines NL. *JV with* Aberfoyle Resources Ltd. Sep 1989 – Feb 1996	Southern Sub-block Group Contains EPM 26210, Pegmont MLs excluded. • One reconnaissance traverse using an APEX Max-Min2 EM to determine suitability of using airborne EM. • 1400 line kilometres of airborne electromagnetic (125 Hz GEOTEM) and magnetic surveys flown over the entire tenement, 200 m line spacing at a height of 120 m. Eight conductive anomalies were identified on the current EPMs. • Soil and rock chip sampling. • The western portion of the tenement was relinquished in November 1992.	
9644 M	Placer Exploration Ltd Sep 1993 – Sep 1998	Northern Sub-block Group **Nine sub-blocks coincident with EPM 26210, relinquished in 1996** • Part of an originally larger EPM, that contained several mineralized targets. • Reconnaissance rock chip samples. • Bulk Leach Extractable Gold (BLEG) stream sampling CU, Pb, Zn, Ag, Au. One anomalous Au, but poor follow up results.	
9109 M	Hunter Exploration / Mount Isa Mines Dec 1992 – Jan 2001	Northern Sub-block Group **Six sub-blocks coincident with EPM 26210** • Review of regional airborne magnetic and radiometric survey, no fieldwork was performed on the six sub-blocks, the sub-blocks were relinquished in 1994.	
6029 M	Perilya Mines *JV with* BHP Minerals Feb 1996 – Apr 1999	Southern Sub-block Group **Five sub-blocks coincident with EPM 26210** • Airborne GEOTEM$_{DEEP}$ (25Hz) was flown at a height of 105 m on east–west lines, spaced 300 m for a total of 1138 line kilometres. 4 "priority 4" and one "priority 3" conductors are located in the NE part of EPM 26210. There is also a low amplitude conductive trend on the eastern side of EPM 26210. A moderate amplitude conductive trend is present on southern part of EPM 26210, including priority 3 conductor "47". No conductive anomalies on the current EPM were followed up.	
6029 M	Perilya Mines *JV with* North Ltd. May 2000 – Jun 2001	Southern Sub-block Group **Three sub-blocks coincident with EPM 26210** • No fieldwork was performed on the three sub-blocks, work focused on other parts of the EPM.	

Historic EPM	Holder and approx. dates	Work performed	Historic EPMs Overlap with current project area (MLs dark grey and EPM light grey)
11591	BHP Billiton Minerals P/L Jan 2004 – Sep 2006	Northern Sub-block Group **Nine sub-blocks coincident with EPM 26210** • Part of an originally larger EPM. • No fieldwork was performed on these sub-blocks.	
14008	PML Jul 2003 – Aug 2006	Northern & Southern Sub-block Groups Contains EPM 26210, Pegmont MLs excluded. **The antecedent EPM to EPM 15106 and hence in part the current EPM 26210** • No fieldwork was performed.	
6029 M	Perilya Mines *JV with* Rio Tinto Expl. P/L Jun 2001 - Jun 2003	Southern Sub-block Group Three sub-blocks coincident with EPM 26210. • No fieldwork was performed on these sub-blocks, work focused on other parts of 6029 M.	
16177	Chinova Resources Cloncurry Mines Oct 2006 –	Northern Sub-block Group Nine sub-blocks coincident with EPM 26210. • Exploration reports not yet public open file.	
15106	PML Sep 2006 – Aug 2017 (VTT operator since 2014)	Northern & Southern Sub-block Groups Nine sub-blocks coincident with EPM 26210 northern sub-block group. Eight sub-blocks coincident with EPM 26210 northern sub-block group. Progressively relinquished down to the eight core sub-blocks that were then conditionally surrendered as part of the current EPM 26210. • 22 line km of ground magnetics over Newmont aeromagnetic anomalies. • 2014 airborne magnetic and radiometric survey, 50 m line spacing, 35 m flying height (also over EPM 14491). • Drilling totalled 62 RC drillholes, 61 RC pre-collared diamond drillholes and 2 diamond drillholes.	
14491	PML Aug 2006 – Aug 2017 (VTT operator since 2014)	Northern & Southern Sub-block Groups Two sub-blocks coincident with EPM 26210 northern sub-block group. Three sub-blocks coincident with EPM 26210 northern sub-block group. Progressively relinquished down to the three core sub-blocks that were then conditionally surrendered as part of the current EPM 26210. • Two individual 1 km magnetic traverses were also carried out in 2012, but the survey was abandoned due to interference. • Ground transient electromagnetic (TEM) survey on line separations of 100 - 200 metres, with the spacing between successive stations on each line of 50 metres. • Drilling totalled 7 RC drillholes and 1 RC pre-collared diamond drillhole. VTT RC hole intersected copper-gold mineralized in silica altered pegmatite.	

6.3 Historical Mineral Resource and Mineral Reserve estimates

A number of historical, non-NI 43-101 compliant Mineral Resource estimates have been prepared on the Property by previous operators. These are summarized in Table 6.3. All historic Mineral Resource estimates have no direct comparison to the current Mineral Resource estimate in Section 13.1.

Newmont (Sheppy and Verwoerd, 1975) produced a manual polygonal estimate using sections. The specific gravity applied was 3.75 t/m^3 for oxide and 4.2 t/m^3 for sulphide, with a cut-off grade of 4% Pb+Zn.

In 1996 PML commissioned consultants Behre Dolbear Australia Pty Ltd (BDA) to undertake a review of the earlier resource estimate that had been based on the work carried out by Newmont in the mid-1970s (Hancock, 1997). This earlier resource estimate had not included additional drilling undertaken since Newmont's work nor had it included the results of Pegmont Mining's work at the Gossan Lode. The BDA review was also asked to address some concerns regarding the geological interpretation adopted by Newmont and used in the early resource estimate.

BDA used cross sectional polygonal methods for both Newmont and PML drill data, with the same specific gravity and cut-off grade as used by Newmont. Each drillhole was reviewed and mineralization above the 4% lead plus zinc cut-off correlated between sections if possible. Resource intersections were extrapolated halfway between adjacent holes and / or sections. In comparison to the Newmont estimate, less continuity was assumed by BDA, and the maximum extrapolation used by BDA was 62.5 m in both dip and strike directions. Silver was not estimated.

The scope of the BDA review was strictly limited and did not include an audit of the data. The primary objective was to provide an indicative comparative estimate for internal purposes. A simple sectional estimation methodology was adopted, and BDA noted that the structure could be significantly more complex than the outlines used for the estimate.

As BDA's resource estimate relied primarily on the unaudited data from Newmont's work in the 1970s and no site visit was undertaken, BDA did not consider the estimate to be JORC-compliant.

Hellman & Schofield (2010) prepared a resource estimate for PML using ordinary kriging to inform a rotated block model (10 m x 10 m x 2 m) to a maximum depth of 200 m below surface. A total of 315 drillholes were used in the estimate. The data was composited to two metres. A grade envelope of 0.25% lead plus zinc was used to select samples. Four different search passes, using four different radiuses were used: 50 m x 10 m, 100 m x 20 m, 200 m x 40 m, and 400 m x 80 m, all with a dip of -30° to 140° azimuth. The resource was classified as Inferred using the first two passes with a minimum of 6 and maximum of 32 samples. For oxide and transition a specific gravity of 3.2 and 3.5 t/m^3 was used, and for sulphide 3.9 t/m^3 was used. Grade was estimated by Hellman & Schofield as the percentage of lead plus zinc and reported as such. Silver was not estimated.

JM Geological Consulting (2011) updated the Hellman & Schofield estimate with the addition of 57 RC drillholes. The only difference in methodology to the previous estimate was the application of different search ellipse dips to better match the changes in dip of the mineralization. The search ellipse dip was varied by domain; -10, -25, and -80° were used. Again, four search radiuses were used, however the estimate was classified as Indicated if the 50 m x 10 m search could be informed with a minimum of 10 samples and Inferred using the 100 m x 20 m search and a minimum of 6 samples. Silver was not estimated.

Table 6.3 Historic resources

Year	Author	Cut-off Pb+Zn %	Material type	Category	Tonnes (Mt)	Pb (%)	Zn (%)	Ag (g/t)
1975	Newmont	4% Pb+Zn	Oxide	Indicated & Inferred	4.5	6.4	2.3	7.7
			Sulphide	Indicated & Inferred	6.6	8.4	3.7	11.5
1997	BDA	4% Pb+Zn	Oxide	Inferred	2.6	7.72	2.84	-
				Indicated	1.7	7.04	2.50	-
			Sulphide	Inferred	2.0	8.51	3.70	-
				Indicated	2.0	7.62	4.85	-
2010	Hellman & Schofield Pty Ltd	4% Pb+Zn	Oxide / transitional	Inferred	1.5	5.8 combined		-
			Sulphide		4.3	6.1 combined		-
2011	JM Geological Consulting Pty Ltd	3% Pb+Zn	Oxide	Inferred	7.217	3.5	1.5	-
			Sulphide	Indicated	1.635	3.31	1.39	-

The reliability of the above estimates has not been validated, and other than the summary in the text, the key assumptions, parameters, and methods used to prepare the historical estimate are not known.

The QP has not done sufficient work to classify the historical estimates as current Mineral Resources and the issuer is not treating the historical estimate as current Mineral Resources.

6.4 Historical production

There has been no mining activity, hence no historical production, from the Property.

7 Geological setting and mineralization

7.1 Regional geology

The Mount Isa Inlier is a multiple deformed and metamorphosed Early to Middle Proterozoic terrain. The inlier has been subdivided into three broad north-trending Provinces – the Western, Kalkadoon-Ewen, and the Eastern Fold Belts, based on various tectonic, structural, and paleogeographic criteria. The inlier is bounded in the north-east by the Mesozoic Carpentaria Basin, in the south and south-east by the Mesozoic Eromanga Basin, in the south and west by the Paleozoic Georgina Basin, and in the north and north-east by the younger, Proterozoic South Nicholson and McArthur Basin (Blake, 1987).

The Eastern Fold Belt is subdivided into a further three zones, from west to east; the moderately deformed Wonga and Quamby-Maldon Subprovinces and the structurally complex Cloncurry Subprovince.

The Western Fold Belt contains the sediment hosted, stratiform world class Mount Isa, Hilton, George Fisher Lead, Zinc, Silver deposits and the Mount Isa copper deposit. The Eastern Fold Belt, in particular the Cloncurry Sub-province, contains deposits of a different character including several significant iron oxide copper gold deposits (Ernest Henry, Osborne, Mt Elliot, and the Starra deposits) and Broken Hill Type (BHT) lead, zinc, silver deposits including the world class Cannington and several smaller examples of which Pegmont is the largest. Dugald River is the only known sediment hosted lead, zinc, silver deposit in the Eastern Fold Belt.

Figure 7.1 shows the major tectonic elements and principle lead-zinc deposits of the McArthur-Mount Isa Proterozoic.

Blake (1987) proposed three major cover sequences of Proterozoic sediment and volcanic rocks which overlying previously deformed basement (older than 1875 Ma), each sequence being the result of a major rift-sag event. Pegmont is located in the Soldiers Cap Group, assigned to cover sequence 2 by Blake (1987) but has recently been shown by dating (Page and Sun, 1988) to be part of cover sequence 3. This is the basis of the geological map of the Eastern Fold Belt shown in Figure 7.2.

An alternative to the cover sequences Blake is proposed by Beardsmore et al (1988) following a major regional mapping exercise, who suggest the Mount Isa Inlier has evolved through three ensialic rifting cycles; rift cycle 1 (1880 – 1810 Ma), 2 (1810 – 1750 Ma), and 3 (1700 – 1660 Ma). The Cloncurry basin is formed during rift cycle 1, Mount Norna Quartzite is thought to be deposited near the end of this cycle. Rift cycle 2 saw a second basin open to the west, the Leichardt River Fault Trough. The third ensialic rift sequence was superimposed over the sediments of both the Cloncurry Basin and Leichardt River Fault Trough.

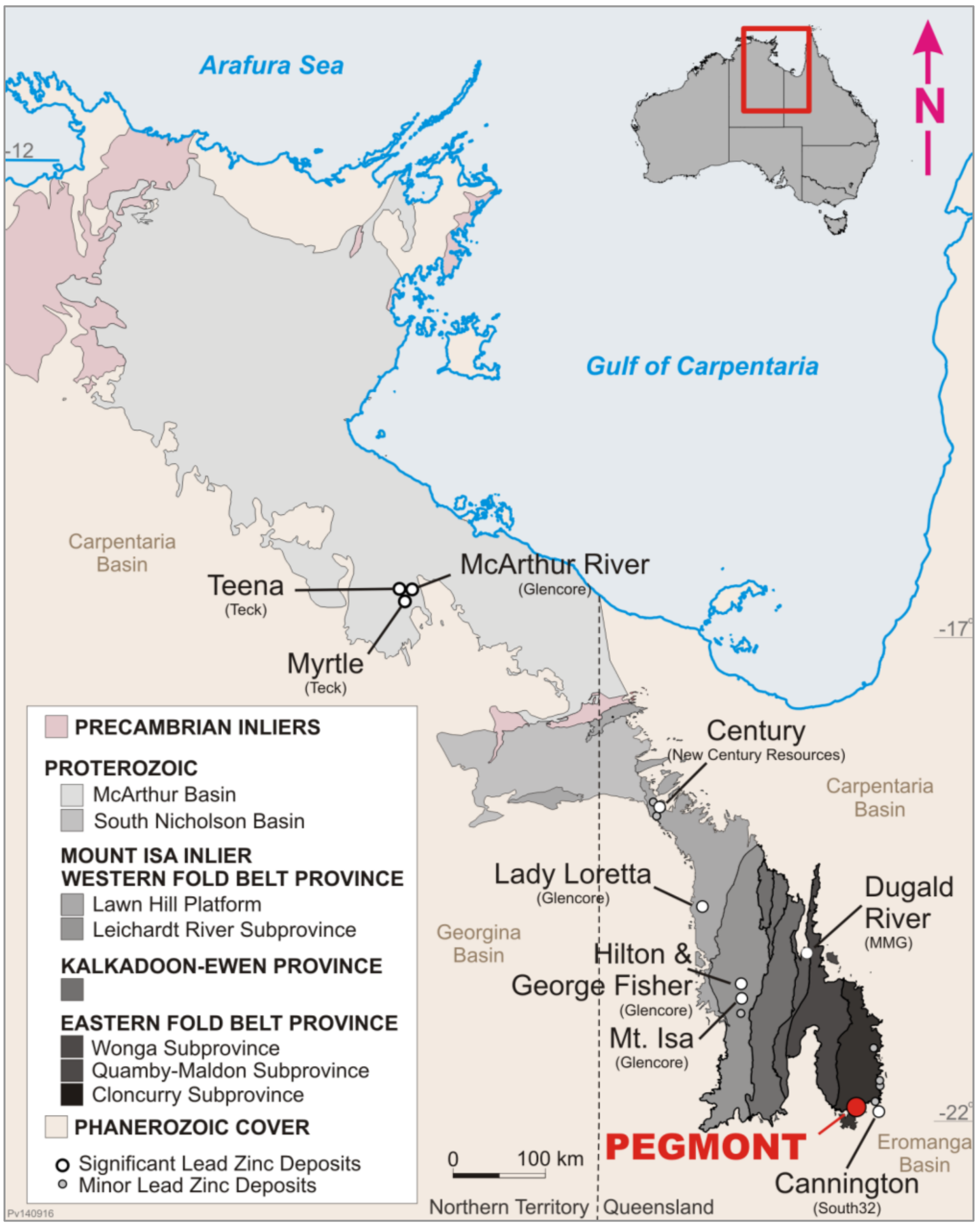
Beardsmore et al. expanded the Soldiers Cap Group to include the Kuridala Formation (rendered obsolete) and also divided the then undifferentiated Soldiers Cap Group into three individual formations (from oldest to youngest), Llewellyn Creek Formation, the Mount Norna Quartzite, and the Toole Creek Volcanics. Underlying this is the Fullarton River Group which is divided into three individual formations (from oldest to youngest), Gandry Dam Gneiss (host of the Cannington deposit), Glen Idol Schist and New Hope Arkose. The Soldiers Cap Group and Fullarton River Group combined are termed the Maronan Supergroup as Beardsmore et al. (1998) believe the sequence to be conformable and with no perceptible change across the boundary between the two groups. Total thickness of the Maronan Supergroup is up to 10 – 12 km, and comprises metamorphosed bimodal volcanic, clastic, and chemogenic sediments.

Under the stratigraphy scheme proposed by Blake (1987), Pegmont was placed in the Kuridala Formation. It is now believed that Pegmont is located near the base of the Mount Norna Quartzite (Newbury, 1990). The Mount Norna Quartzite is described as consisting predominately of weakly feldspathic psammitic, politic, and quartzitic meta sediments, with minor carbonaceous metasiltstones. Three stratiform iron formations occur at different stratigraphic levels of which Pegmont is the oldest. These are locally referred to as banded iron formation (BIF).

The generally un-deformed granites of the Williams and Naraku Batholiths intruded rocks in the Cloncurry area at 1510 – 1460 Ma towards the end of the Isan Orogeny.

Apart from uplift, erosion, and deposition of sediments during various periods in the Phanerozoic, the Mount Isa Inlier has been part of the stable Australian craton since ~ 1450 Ma ago.

Figure 7.1 Map of McArthur-Mount Isa Proterozoic



Source: Supplied by VTT, compiled from Gov. of Northern Territory and Gov. of Queensland sources.

Authors in various study areas have adopted different structural nomenclature and interpretations making correlation between the various structural histories difficult across the region. The following discussions draw on the descriptions given by Newbery (1999) and Laing (2007) and relate to the structural history of the Cloncurry Sub-province.

The rift-sag extensional phases of the Isan Orogeny described above were followed by basin closure, onset at about 1620 Ma. This period of complex polyphase thrusting and folding that usually described in terms of up to six deformational episodes.

The first deformation event (D1) is characterized by the north-south thrusting of younger Soldiers Cap Group rocks over Staveley Formation and the development of east-west oriented folds and an axial planar cleavage, both of which are poorly preserved, and in a layer-parallel foliation (S1).

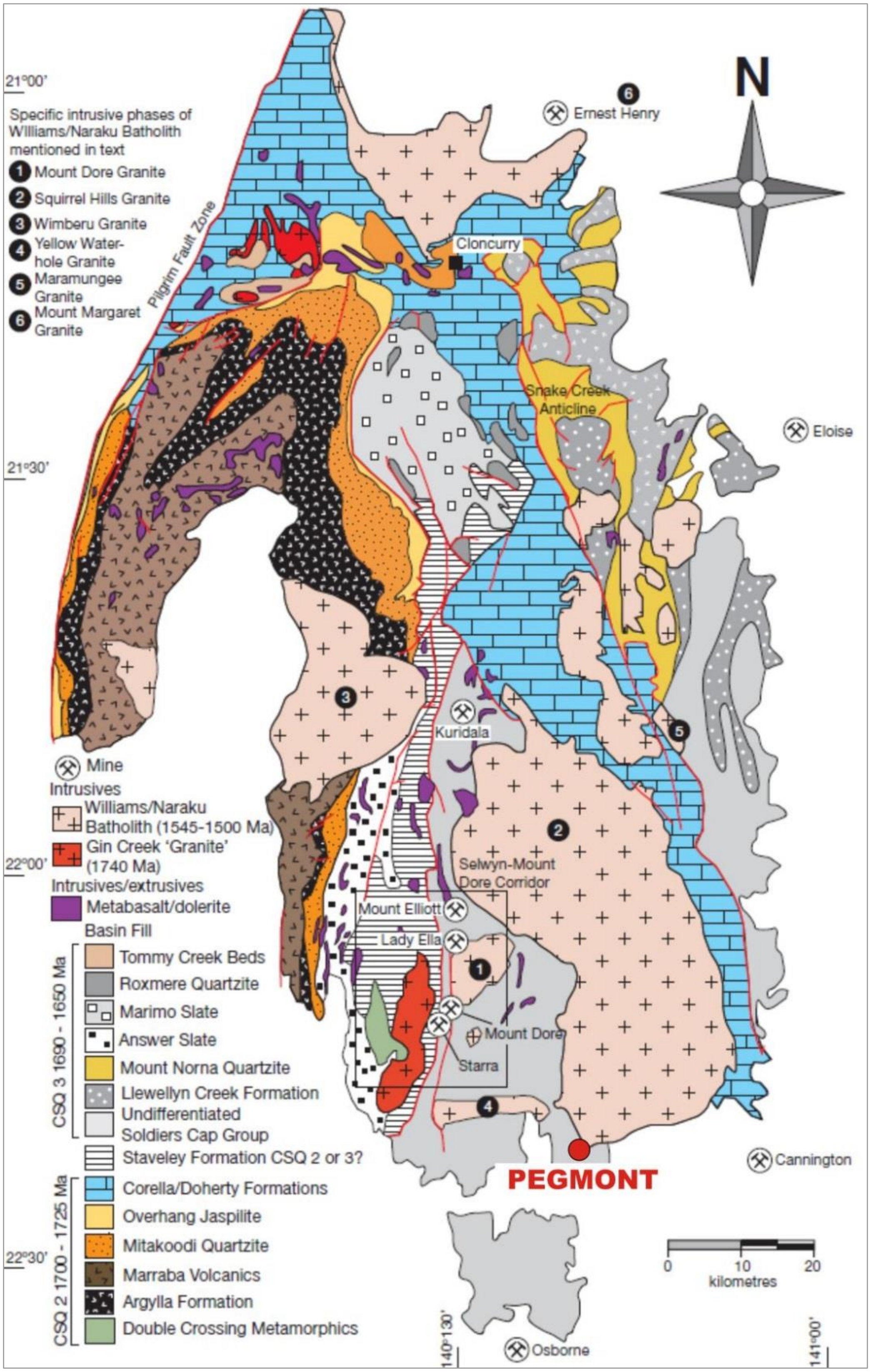
The D2 event is the principle deformational episode in the Mount Isa inlier, it is the dominant observable structural event in the Eastern Fold Belt. With east towards west movement direction D2 overprinted D1 structures and resulted in the development of upright, tight to isoclinal, folds (F2), wavelengths of between 2 km and 0.2 km are seen in the Soldiers Cap Group. They are accompanied by an intense, steeply east dipping axial planar foliation (S2) that strikes northeast-southwest. This was accompanied by the formation of steeply east dipping reverse faults. The major D2, north-south striking faults from west to east are the Pilgrim River, Mount Dore, and Cloncurry Faults.

This event was accompanied by low-pressure, high-temperature upper greenschist to amphibolite grade metamorphism at ~1585 Ma (Page and Sun, 1998).

Continued east–west crustal shortening during D3 generated open, rounded folds with NNE-striking, subvertical axial planes and gently plunging fold axes, and produced a shallow, east / west dipping crenulation cleavage.

The D4 event marked the change from ductile to brittle deformation and led to the formation of a fault network that was associated with widespread breccia formation. The dominant brittle structure is the Cloncurry Fault which flanks the western margin of the belt.

Figure 7.2 Simplified geologic map of the eastern fold belt



Source: After Duncan et al, 2011.

7.2 Property geology

The deposit scale geology at Pegmont has been the subject of a number of scientific studies including the unpublished Ph.D. thesis of Vaughan (1980) and Newbery (1990), unpublished honours thesis of Pendergast (1993), and is the subject of a number of scientific journal articles principally; Locsei (1977), Stanton and Vaughan (1979 &1997), Vaughan and Stanton (1984, 1986a, 1986b), Scott and Taylor (1987), and Williams et al (1998). It should be noted that all of this scientific work occurred prior to the larger drilling campaigns, which occurred from 1998 on.

There are few outcrops on the Property, mapping is available from Newmont in 1978, North in 1999 and in 2018 VTT mapped in detail the immediate area around the deposit (Figure 7.3).

The geological framework of the Pegmont relies predominately on interpretation of drillhole data, complimented by the fact mapping.

7.2.1 Stratigraphy

The host sequence is informally named the "Pegmont Beds". These are a monotonous sequence consisting mainly of feldspathic psammites (quartz+biotite±microcline±plagioclase±garnet) and pelitic schists (quartz+biotite+muscovite±microcline±plagioclase±garnet±andalusite). There are several Ca, Fe, Mn and P rich BIF which host the lead and zinc mineralization. Because of the monotonous nature of the host sequence and the gradational boundaries between psammites and pelites it is the principle BIF (Lens B) is the only distinctive and reliable marker unit in the Pegmont Beds.

The BIFs are enveloped by a garnet rich metaarkose (GASS), consisting of varying proportions of quartz, Fe-Mn-Ca garnet and biotite with accessory albite, gahnite, zircon, apatite, sphene, muscovite, magnetite, ilmenite and sulphides. The garnet-bearing rocks are therefore a good indicator of proximity to the mineralized BIF. The garnet in these wall rocks is manganese poor almandine, 1 – 2 mm in diameter in the GASS and up to 1 cm in a micaceous garnet rich schist. The hangingwall GASS is typically thinner than the footwall GASS at any one location however both can reach thicknesses of up to 4 m. The GASS frequently grades into a garnet-bearing schist, consisting of garnet, quartz, biotite and K-feldspar with subordinate muscovite, plagioclase, cordierite, tourmaline and apatite.

Sun *et al* (1994) calculated a lead isotope age for the Pegmont mineralization at about 1670 Ma, this is within the age ranges published for the host sequence.

Unconformably overlying the Pegmont Beds is the Jurassic Gilbert River Formation comprising siltstones, sandstones and conglomerates. Around the deposit the Gilbert River Formation is found capping many of the low-lying hills.

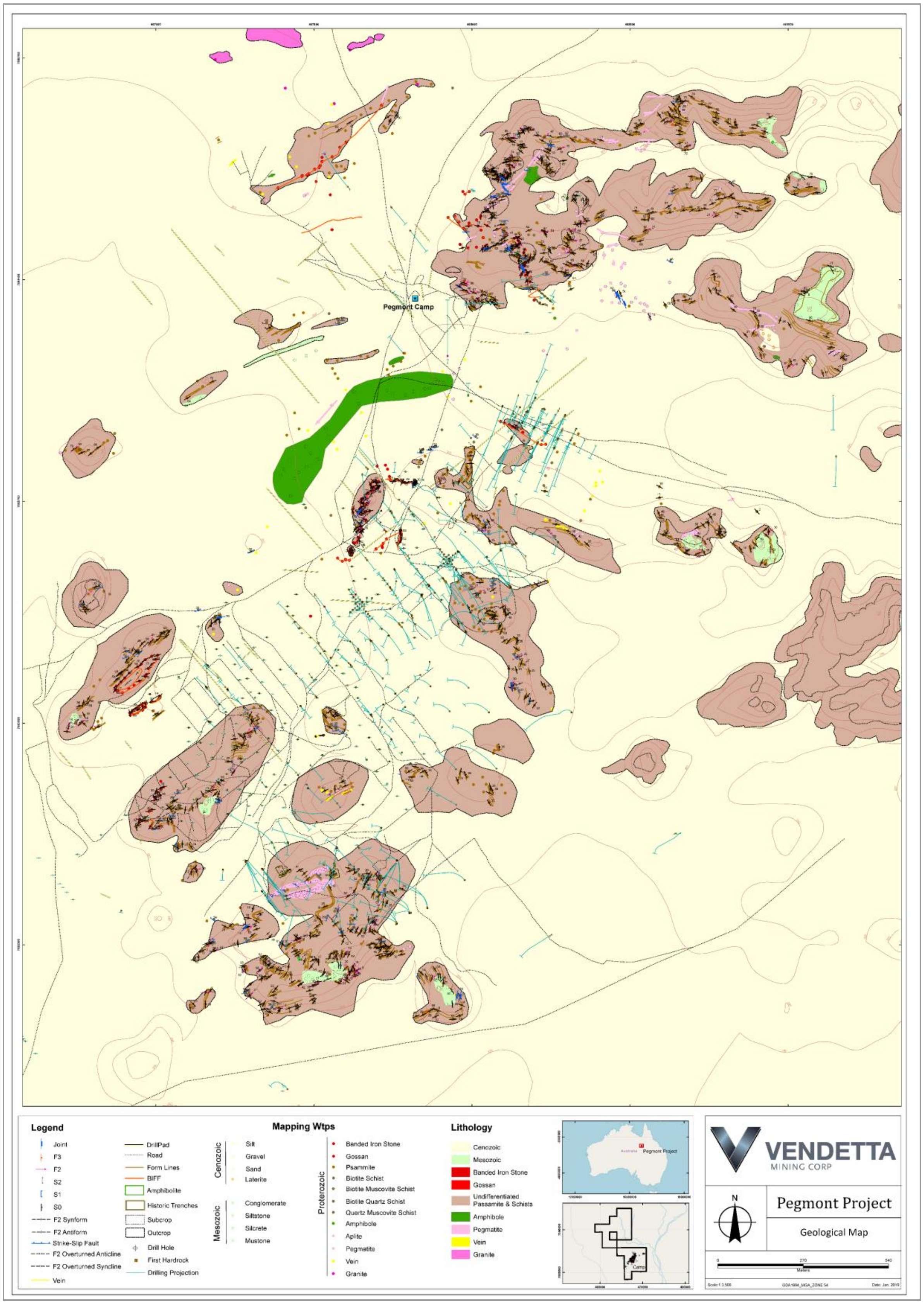
A summary of the local stratigraphy in the Pegmont project area is given in Table 7.1.

A surface geological map is shown in Figure 7.3, which is a compilation of "1:25,000 Scale Surface Fact Mapping" by Newmont in 1975.

Table 7.1 Generalized stratigraphic column for the Pegmont project area

Age	Formation	Thickness (m)	Lithology description
Mesozoic	Gilbert River	Max 25 m	Medium to coarse-grained, clayey quartzose sandstone and pebbly sandstone; minor quartz pebble conglomerate, siltstone, mudstone.
Unconformity			
Proterozoic	Mount Norna Quartzite	+100’s	Metaarkose (PSAM), ≈30% Biotite Quartz Schist (BQSH), minor Biotite Schist (BISH).
		≈20-40	Metaarkose (PSAM), minor Biotite Schist (BISH) or Biotite Quartz Schist (BQSH).
		0 - 1	Lens A Garnet-rich metaarkose (GASS) with occasional Banded iron formation, low level of lead-zinc mineralization.
		2 – 7	Metaarkose (PSAM), minor Biotite Schist (BISH) or Biotite Quartz Schist (BQSH).
		0 – 4	Garnet-rich metaarkose (GASS) weak lead & zinc mineralization.
		1 – 6	Lens B Banded iron formation (BIFF) and moderate to strong lead-zinc mineralization.
		0 – 4	Garnet-rich metaarkose (GASS) weak lead & zinc mineralization, usually more strongly developed than Lens B hangingwall GASS.
		10 - 40	Metaarkose (PSAM), minor Biotite Schist (BISH) or Biotite Quartz Schist (BQSH).
		0 – 1	Garnet-rich metaarkose (GASS), weak lead-zinc mineralization. Average 08 m thick when present.
		0 – 4	Lens C Banded iron formation (BIFF), low to strong lead-zinc mineralization. Usually present in Bridge Zone, most strongly mineralised in Zone 5.
		0 – 1	Garnet-rich metaarkose (GASS), weak lead & zinc mineralization, usually more strongly developed than Lens C hangingwall GASS. Average 08 m thick when present.
		20 – 60	Metaarkose (PSAM), minor Biotite Schist (BISH) or Biotite Quartz Schist (BQSH).
		+100’s	Metaarkose (PSAM), ≈30% Biotite Quartz Schist (BQSH), minor Biotite Schist (BISH).

Figure 7.3 Vendetta Pegmont deposit area fact mapping



7.2.2 Metamorphism

The amphibolite facies metamorphic grade is indicated by the development of hornblende, cordierite, garnet, muscovite, and biotite. Sillimanite has not been observed.

7.2.3 Alteration

Pegmont is enveloped by a variable but large volume of late or postorogenic fracture/vein-controlled alteration. Williams *et al* (1998) identified two stages of alteration. Stage 1 veins typically are composed of quartz ± K feldspar ± tourmaline ± biotite ± rutile ± ilmenite associated with K feldspar ± muscovite ± tourmaline, or biotite ± muscovite ± garnet alteration. Stage 2 quartz ± chlorite ± calcite ± ferroan dolomite ± hematite ± sulphide veins are associated with fine-grained muscovite (illite-phengite) + chlorite ± carbonate alteration.

The psammites are more altered than adjacent pelites. This is interpreted to be due to a combination of their mechanical susceptibility to fracturing and the chemical reactivity of the feldspars in the psammites. Veins in psammites are strongly dis-cordant to this fabric and associated with ≈5 mm thick selvages of pervasive K feldspar replacement of the metamorphic assemblage.

The relative abundance of sulphides in the Stage 2 veins is Cu > Zn > Pb, which is opposite to metal ratios exhibited by the BIF.

Williams *et al* concluded that the alteration occurred during the later stages of the regional deformation and metamorphism, and on the basis of these structural relationships, could have occurred at the time of emplacement of the nearby granitoids.

7.2.4 Intrusions

The oldest intrusive are Amphibolites, these bodies outcrop poorly, but are intersected in drillholes. They have a mineral assemblage of hornblende, plagioclase, biotite, quartz, and ilmenite, overprinted by retrograde chlorite and sericite. Minor minerals include titanite and apatite, and iron sulphides are locally present. The amphibolites are typically foliated. Amphibolite horizons may originally have been andesitic to basaltic lavas.

A largest amphibolite (hornblende-plagioclase) body is poorly exposed in sub-crop on the edge of the mining lease, called the "Lease Amphibolite", it is a flay lying, highly continuous body that transgresses the BIF without any apparent displacement. The Lease Amphibolite displays textural zoning, which may reflect chilled margins and compositional layering. The southeast edge of Zone 3 is defined where the amphibolite intersects the Lens B BIF, to the southeast of this the Lease Amphibolite is below the BIF's and to the northwest it is above the BIF's. Drill data density in the Lease Amphibolite is less than the mineralization.

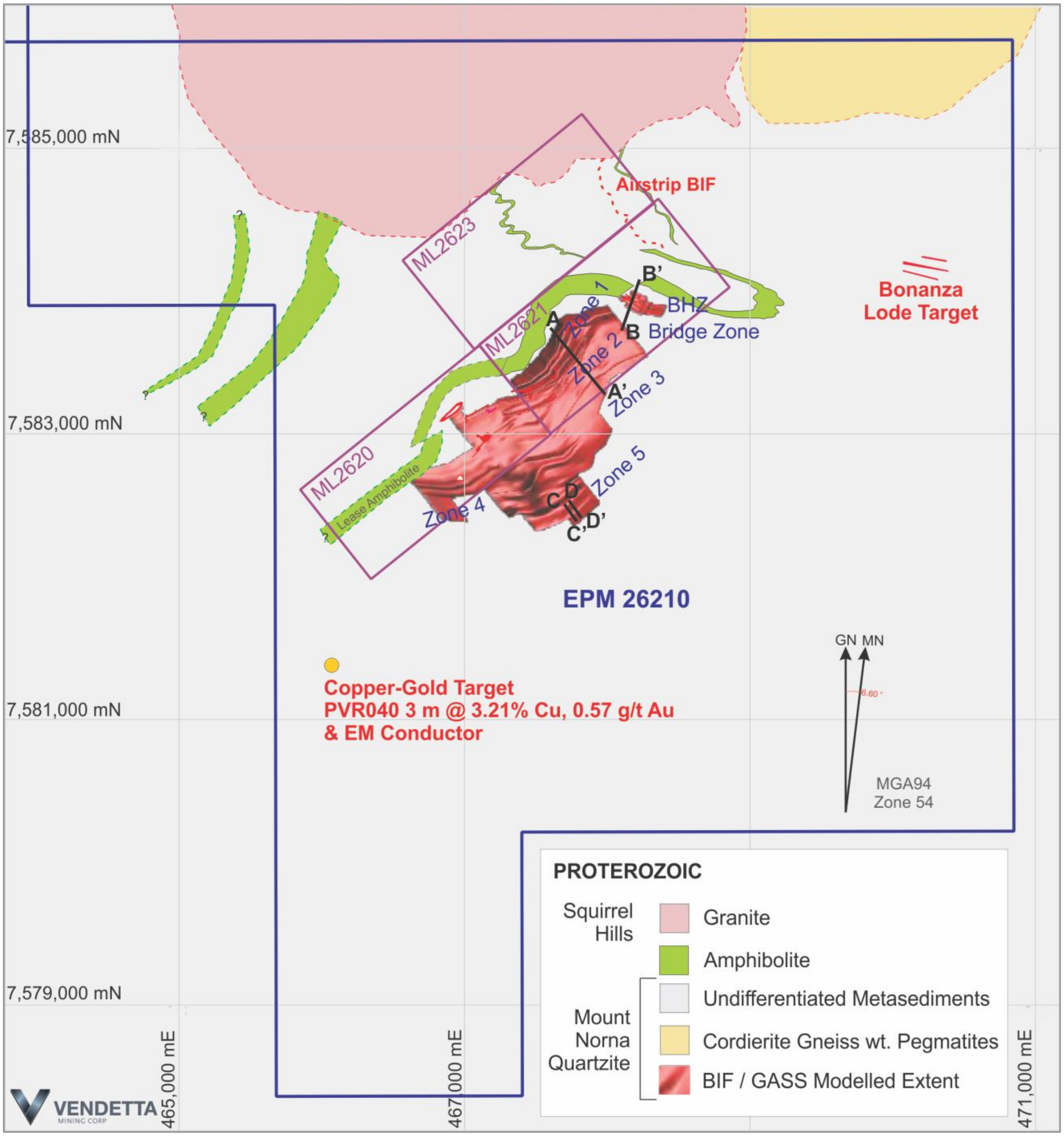
The Squirrel Hills Granite, part of the Williams Batholith, is over 100 km north-south and up to 25 km wide. It has intruded the project sequence in the northern boundary of the Pegmont lease. Present as several low outcrops of tors exhibiting spheroidal and onion skin weathering. The southern contact is inferred from airborne magnetics. The Squirrel Hills Granite is a non-foliated, porphyritic granite, composed of biotite and hornblende with coarse K feldspar. The Yellow Water Hole Granite, also part of the Williams Batholith, approximately 18 km from Pegmont, is dated at 1493±8 Ma (Page and Sun, 2015), postdating main tectonism that affected the sequence.

There are several outcrops of a distinctive cordierite-bearing schist, interpreted to be a contact metamorphic rock related to the emplacement of the Squirrel Hills granite hornfels. The rocks consist of dominant quartz and K feldspar, with significant amounts of cordierite, biotite, muscovite, plagioclase, magnetite, sillimanite, and andalusite.

Quartz-feldspar Pegmatites fall into two types. Pegmatite sills, concordant with original banding and later foliations formed during progressive metamorphism and folding, typically 5 to 20 cm thick. In many cases the pegmatites are garnetiferous near their contacts, poor in mica, and rich in tourmaline, often exhibiting feldspar altered selvages. There are quartz-tourmaline veins with coarse microcline selvages.

Small local transgressive pegmatites are present, they are of limited extent and can not be reliably interpreted between drill holes, they are not expected to disrupt the mineralized bed.

Figure 7.4 Subsurface interpretation map



SOURCE: VTT

7.3 Mineralization

Lead-zinc mineralization at Pegmont is contained within BIF's. The BIF’s consists of banded quartz-magnetite-fayalite-garnet-grunerite-hedenbergite-sulphide. Apatite, gahnite, and graphite are common minor minerals. Bedding is typically on a scale of 1 to 5 mm. In fresh rocks, the main sulphide minerals are galena, sphalerite, with subordinate pyrrhotite, pyrite and chalcopyrite. In contrast to the almandine garnet in the hangingwall and footwall, the garnet in the BIF is manganiferous spessartite, reflecting the higher whole rock MnO content of the BIF.

The overall morphology of the stratiform mineralized banded iron formation at Pegmont is a flat, gently easterly dipping sheet. The known mineralization extends approximately 2 km along strike and approximately 1 km in the down dip direction to the southwest. Mineralization is known to extend to a depth of 350 m below surface but remains open down dip.

The principal lead and zinc mineralized BIF is termed Lens B, it is around 2 to 8 m thick. At surface, the Lens B BIF is present in banded ferruginous, jaspery, manganiferous gossans. Outcrops occur in two areas termed the Mount Lucas Load and Burke Hinge Zone (BHZ). Additional oxide mineralization has been intersected at the Bonanza Lode, which is about 2 km to the northeast of Mount Lucas. Work by Scott and Jones (1987) shows that there has been considerable depletion of Mg, Ca, Na, K, S, Ag, Cd, and Zn during weathering. The least mobile elements have undergone residual concentration up to the profile, whereas concentrations of other elements are either unaffected by weathering or vary irregularly, in many cases reflecting local variations within the ore horizon. On oxidation, galena changes either through a pyromorphite ± cerussite assemblage to plumbogummite / corkite or directly to coronadite.

Oxidation reaches depths of 25 m below surface. The surface gossans display delicate boxworks after fayalitic olivine and are composed of goethite, clay, and secondary lead minerals such as pyromorphite, plumbogummite, plumbojarosite, and / or beaverite, together with isomorphous hydrated lead, iron, and copper sulphates. No secondary zinc minerals have been observed. Between 0.5 and 2% graphite has been noted. A yellow-green fibrous vein mineral contained in the gossans is thought to be nontronite.

The lead content of outcropping gossan is the same as, or slightly higher than that of un-weathered ore. Although zinc is strongly depleted at surface (×10), its greater abundance (×2 depletion) only 15 m below the surface reflects the truncated profile and immaturity of the gossan (Scott and Jones, 1987).

Figure 7.5 Photograph of oxidized BIF in core

Source: VTT

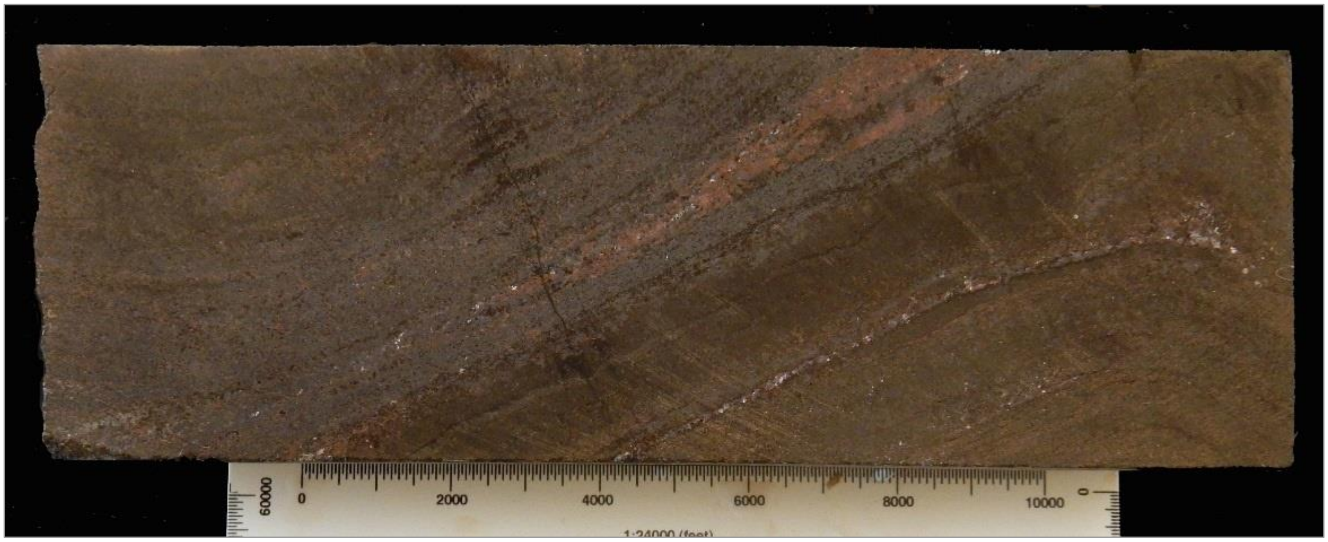
In the partially oxidized transition zone, between 25 – 40 m below the surface, galena, sphalerite, magnetite (invariably showing alteration to hematite and goethite), manganese minerals including pyrolysite, and small amounts of pyrite and graphite have been described along with montmorillonite and kaolin, garnet, mica and quartz, and minor amounts of siderite, amphibole, chlorite, and talc.

The un-weathered Lens B BIF consist mainly of galena and sphalerite associated with a finely laminated assemblage consisting of dominant magnetite and spessartite with subordinate iron-magnesium-manganese silicates and apatite. Gangue minerals are apatite, olivine (fayalite), garnet (spessartine – almandine), amphibole (hornblende and grunerite), clinopyrozene, biotite, and greenalite.

Approximately 10 to 40 m below Lens B is a second BIF horizon, Lens C, generally only weakly mineralized and when present is around 1 m thick and is separated from the Main BIF by garnet-bearing quartzite and schist. Lens C becomes more important in Zone 5 where it is present at thickness of around 4 m.

In Zone 5 there is indications of a further four mineralized BIFs below Lens C, to date they have been intersected on the south western most drill section, in an interpreted anticline position.

Figure 7.6 Photograph of sulphide BIF in core with observable lead and zinc sulphides



Source: VTT

7.4 Local structural geology

The 3D geological interpretation of the BIF and GASS beds relies heavily on the structural data obtained from the orientated drill core recorded by VTT during each of their programs. A major re-interpretation was made by VTT at the end of the 2016 program. The orientated core data now forms the largest structural dataset available at Pegmont, with over 13,500 records. On the basis of the 3D interpretation the mineralized BIF and GASS beds have been broadly grouped into zones based on different structural settings. The structural zones also form the basis of grouping metallurgical composites on the basis that there are likely metallurgical differences based on the structural setting, i.e. remobilized semi to massive sulphide with mineralized veins seen in and near folds versus laminated sulphides seen on the limbs.

No faults that displace the BIFs have been recognized. In the position of the tight folding some layer parallel shearing is noted in the footwall schists.

The current folded interpretation of the BIF's is shown in Figure 7.4 for a plan and in cross sections in Figure 7.5 and Figure 7.6, all of which were provided by VTT.

7.4.1 Zone 1

In the area around the Mount Lucas gossan the BIF's are tightly (recumbent) folded with an associated foliation. This resulted in structural repetition of the BIF in drillholes in this area. Ductile deformation of metal sulphides produced thickened and upgraded zones of lead-zinc mineralization in fold hinges and pressure shadows and conversely on the limbs of these folds at the point of maximum strain the BIF's have been thinned or pinched, often with a subsequent reduction in lead-zinc mineralization. The portion of the BIF's that display this recumbent folding is referred to as Zone 1.

One interpretation of the tight folding is that it occurred during peak metamorphism (1540 - 1520 Ma). VTT theorize that the recumbent folding is due to the emplacement of the Squirrel Hills Granite, i.e. Zone 1 lies within the deformation aureole of the Squirrel Hills Granite. This theory is supported by the reduction in the degree of overturned folding towards the SW away from the granite, the apparent curve in the recumbent fold axial planes which mirrors the granite contact and the overturned folding in the Amphibolite dyke.

7.4.2 Zone 2

Down dip to the east-southeast, the recumbent folding is not seen or is of a lower amplitude that it only forms gentle dipping rolls in the BIF, this area is termed Zone 2. The Bridge Zone, discovered by VTT in June 2017 is now understood to be the strike extension of Zone 2, separated by an attenuated area.

7.4.3 Zone 3

An apparent upright drag fold, with the same sense of movement as the recumbent Zone 1 folds (NW towards SE) drops the BIF's about 80 to 100 m lower, this is the boundary between Zones 2 and 3. The upright fold is referred to as the "Z" fold and, the lower hinge is the site of ductile remobilization and upgraded lead-zinc mineralization. Zone 3 is not well drilled at this stage but appears to have an overall flat dip to the southeast. A second upright drag fold was identified during 2017 but has only limited drilling to date.

7.4.4 Zone 4

The intersection of the Lease Amphibolite and the BIF is used as the boundary for Zone 3 and 4. This area has not been drilled by VTT, more work is required to gain a better understand the structure of this zone.

7.4.5 Zone 5

Zone 5 is characterized by open folding into a series of anticline and syncline pairs. With additional metal sulphide remobilization and upgrading into fold hinges, limbs can be, but are not always attenuated and depleted.

7.4.6 Burke Hinge Zone

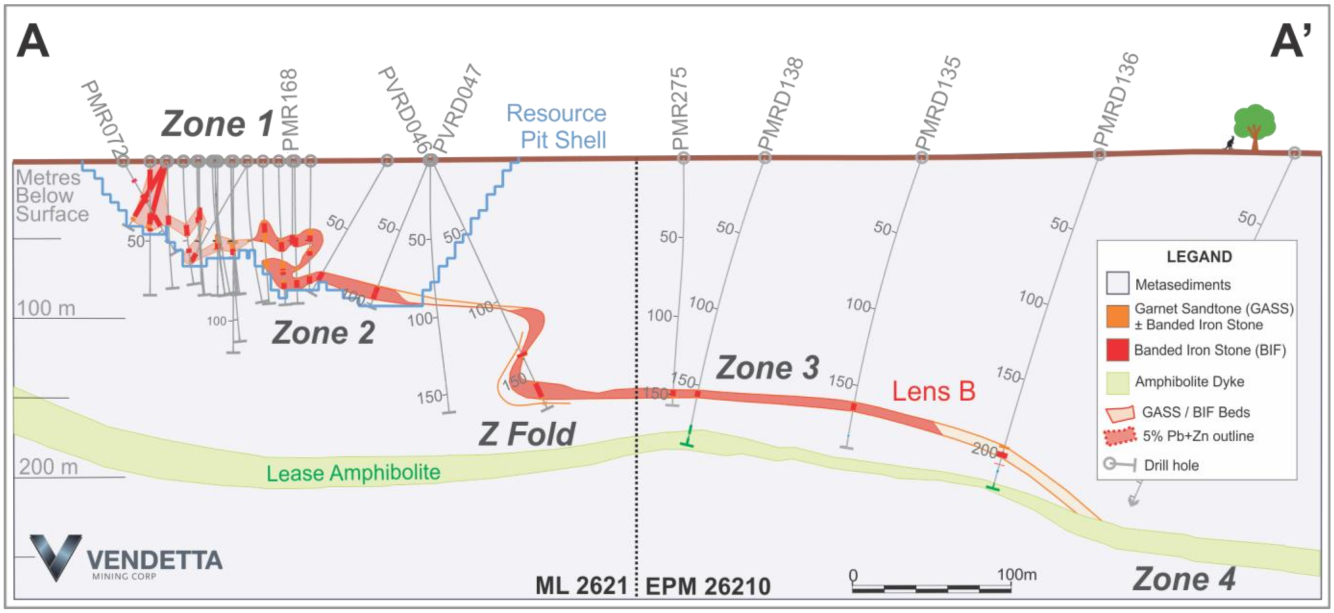
The BHZ and the Bonanza Lode lie on opposite sides of an amphibolite body, likely the Lease Amphibolite. Based on surface work by Newmont and drilling by North (1999) appears to trace out a steeply southeast dipping tight fold.

Two sub-parallel BIFs at BHZ are interpreted by VTT as limbs of a tight recumbent isoclinal fold with an axial plane parallel to the interpreted fold in the amphibolite and mapped folds.

Within the BHZ there is a flat plunging kink fold that is seen on both upper and lower limbs and possibly in the underlying Bride Zone, this is a location of upgrading of the lead-zinc mineralization at BHZ. At depth (around 120 m) both BHZ limbs appears to pinch out, with an accompanying decrease in grade.

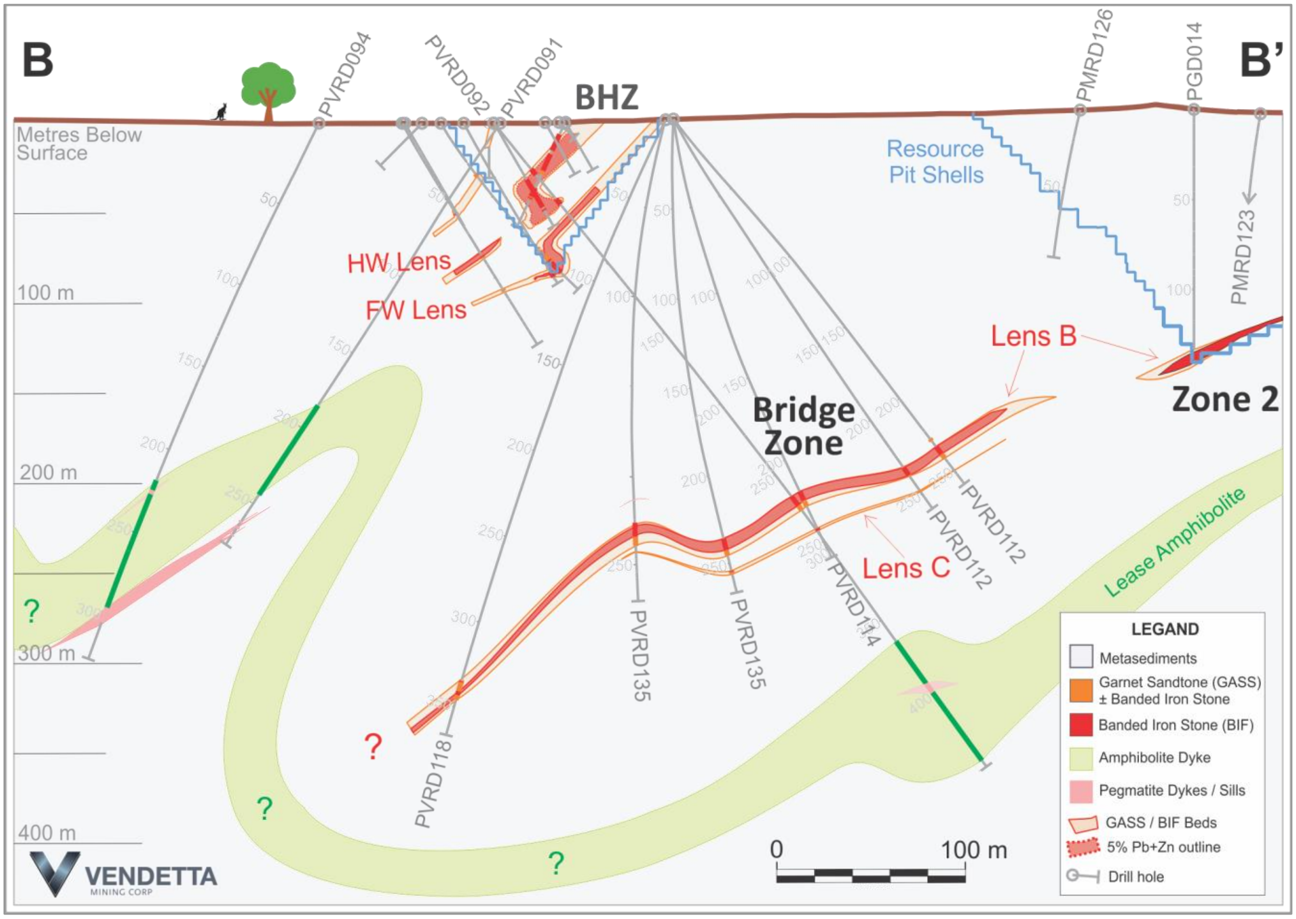
The Bonanza Lode is believed to be in a similar structural setting as BHZ but is yet to be tested by VTT.

Figure 7.7 Cross section interpretation Zones 1 to 3



Source: VTT

Figure 7.8 Cross section interpretation of the BHZ – Bridge Zone Area



Source: VTT

Figure 7.9 Zone 5, Syncline-Anticline "C"

Source: VTT

8 Deposit types

8.1 Deposit and analogues

The Pegmont lead-zinc deposit is a body of stratiform lead-zinc sulphides contained within a metamorphosed silicate facies BIF. It is considered to be an example of a BHT deposit.

The Broken Hill lead-zinc-silver deposit, located in New South Wales, is the archetype for this style of lead-zinc-silver deposit. Broken Hill is classed as a "super giant" and is believed to have originally contained in excess of 280 Mt and is the largest zinc-lead-silver accumulation on Earth (Huston et al., 2006). The world class Cannington silver-lead-zinc deposit, located 25 km to the east of Pegmont, is another important example of a BHT deposit. Other examples of BHT type deposits include; Zinkgruvan (Sweden) and Aggeneys-Gamsberg district deposits (South Africa). BHT deposits occur in dominantly sedimentary basins that have been overprinted by high-grade (upper amphibolite to granulite) metamorphism. These basins, which include the Broken Hill block and the Eastern succession of the Mount Isa inlier, are dominated by siliciclastic sedimentary rocks, with minor but genetically important felsic volcanic rocks, granites, and tholeiitic mafic sills.

Both the Broken Hill and Cannington deposits consist of a series of stacked lenses with Ag, Sb, and F abundances increasing stratigraphically upward, and with Au, Bi, and Cu abundances decreasing. Stacking of ore lenses suggests that these deposits were the sites of long-lived and episodic hydrothermal vents. At Pegmont, there is an indication of two stacked lenses.

BHT deposits exhibit post depositional modifications. High-grade metamorphism (amphibolite facies), metasomatism, and deformation have all affected the morphology and architecture of Broken Hill-type deposits, coarsening the ores, converting original alteration assemblages to metamorphic mineral assemblages, partially melting ore minerals, and remobilizing and upgrading ore into high-grade zones.

Within the Proterozoic Mount Isa-McArthur basin there are six world class lead-zinc-silver deposits (Mount Isa, Hilton, George Fisher, Century, MacArthur River and Cannington). On a regional scale they are spatially related, however, the BHT deposits (Pegmont and Cannington) differ from the other Mount Isa type sedimentary deposits in the following ways:

- They occur in high-grade metamorphic rocks.
- They lack a finely laminated, bedding parallel sulphide mineral textures.
- They lack organic-rich siltstones and a more oxidized siliciclastic sedimentary rock host package.
- They have a very high concentration of manganese.
- They have a magnetite association, either within or along strike from ore.
- They have the presence of semi-regional garnet-bearing, alteration halo.

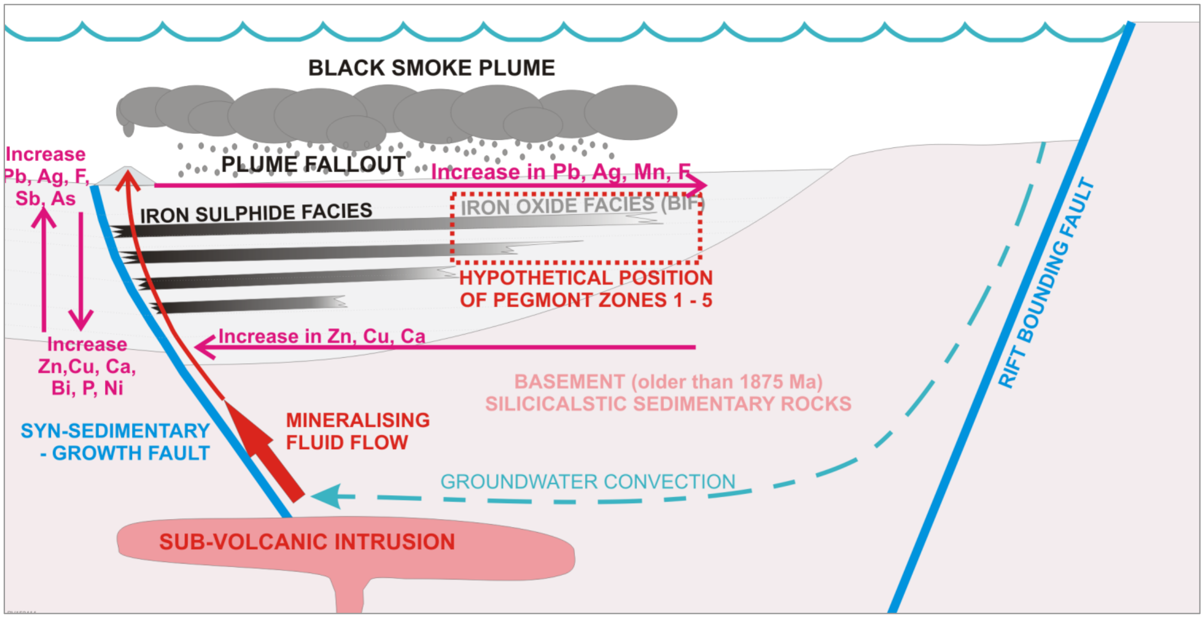
Stanton and Vaughan (1979) discuss the differences and similarities between the host iron formation at Pegmont and Broken Hill. It is noted that at Pegmont fayalitic olivine is an important non-sulphide constituent of the mineralization, that well bedded apatite is present in significant quantities and that the lead-zinc mineralization is characterized by notable amounts of manganese and phosphate, which are all features found to exist in the Broken Hill ore bodies and their associated BIF. At Broken Hill, however, the BIF is present both stratigraphically above and below, but not at, the Broken Hill ore horizon.

8.2 Genetic models

Due to extensive and complex metamorphic and structural overprints, there has been considerable debate on genetic models applied to the formation of BHT deposits. There are presently two competing genetic models; the modified sedimentary model, i.e. derived from sedimentary deposits, and the distal skarn model, i.e. a replacement of a favourable horizon.

Figure 8.1 is a schematic illustration of the BHT ore-forming system, showing the hypothetical position of the Pegmont deposit. Key illustrated components include the host basin (ensialic rift), the heat source (subvolcanic intrusion), the plumbing system (extensional syn-sedimentary faults), the source of fluid components, the nature of fluid flow (convection), the site of metal deposition, and the outflow zone (water column). After Huston et al., (2006), mineral zonation seen in BHT deposits is also shown.

Figure 8.1 Schematic illustration of the Broken Hill Type ore-forming system



Source: After Huston et al., 2006.

Genetic concepts fall in and out of favour over time, however, at present the bulk of the evidence supports a modified sedimentary model with either primary synsedimentary or syndiagenetic origin.

8.2.1 Modified synsedimentary model

This model suggests that the deposit formed on or below the sea floor during sedimentation (synsedimentary) from deeply sourced sedimentary brines. Metals were remobilized and zone refined during subsequent metamorphic and metasomatic events, but without significant external metal addition.

Vaughan and Stanton (1979, 1984, 1986) and Vaughan (1980, 1986) provide evidence that the Pegmont ironstones are a silicate facies of a phosphatic, manganiferous iron formation. Stanton and Vaughan divide the Pegmont silicate facies ironstone into three microfacies; a fayalite, a hornblende-clinopyroxene, and a garnet microfacies, with the significant lead-zinc concentrations being confined almost entirely to the fayalite microfacies. This is potentially an important exploration tool that could be used to discriminate between iron formations of ore-bearing terrains and “barren” iron formations. The metamorphic mineral assemblage and variation to the hangingwall and footwall of the ironstone is regarded as reflecting the primary composition of the sediment stemming from the interplay between hydrothermal and marine regimes on the original basin floor. The lateral mineralogical changes in the mineralized BIF across Pegmont Basin are summarized into a generalized scheme in Table 8.1, however, there has been significantly more drilling since this scheme was formulated and the mapping of the microfacies has not been updated.

Sun et al (1994) has shown that the results of lead isotope work at Pegmont and Cannington indicate that the primary galena introduction was roughly synchronous with sedimentation or diagenesis at -1670 Ma. This is a similar age to the host rocks, and significantly older than the Mount Isa orogen peak metamorphism and metasomatism (75 – 150 Ma later) and is consistent with a synsedimentary or syndiagenetic origin for these deposits, although with a mantle or deep-crustal lead source. Huston et al. (2006) suggests that the source of primary lead in BHT deposits may be intermediate between the evolved crustal source seen in Mount Isa type deposits and the primitive signature observed in some volcanic-hosted massive sulphide (VHMS) deposits, and consistent with tectonic models invoking an ensialic-rift setting.

Features of Pegmont and Cannington which are not easily explained by the synsedimentary model are the lack of barite, even though the ore assemblage suggests an oxidized sedimentary environment, and the extreme manganese enrichment in some ore lenses, which is in an order of magnitude greater than commonly found in seafloor lead-zinc-silver deposits.

Table 8.1 Generalized scheme of lateral mineralogical variation

Basin centre	**Intermediate**	**Basin edge**
High grade mineralization; galena > sphalerite	Lower grade mineralization; sphalerite > galena	Insignificant mineralization; sphalerite ≈ chalcopyrite
Magnetite (Iron about 30%)	Magnetite, pyrite, and pyrrhotite	Pyrrhotite (>> pyrite) (Iron about 10%)
Quartz absent	Some quartz	Abundant quartz
Garnet present but subordinate	Garnet present, becoming more abundant	Abundant garnet
Abundant fayalite	Minor fayalite, sometimes present	Fayalite is absent
Green hornblende, clinopyroxene and grunerite weakly developed	Green hornblende and pyroxene dominate; hornblende > pyroxene	Traces of green hornblende sometimes persist
Constant apatite (≈7% by weight)	Constant apatite (≈7% by weight)	Constant apatite (≈7% by weight)
Minor biotite	Minor biotite	Abundant biotite

Source: After Stanton and Vaughan (1979).

The interpretation discussed above forms the geological model and concept that has been applied to exploration and Mineral Resource estimation in this Report.

8.2.2 Distal skarn model

Williams et al (1998) have shown that at Pegmont there is a large, fracture controlled hydrothermal alteration event that extends out from the 1 to 6 m thick ironstone for up to several hundred metres. Alteration of the wall rocks occurred after the amphibolites grade peak regional metamorphism, with the intensity generally increasing towards the ironstone. Early alteration produced quartz ± tourmaline ± K-feldspar ± biotite veins and K-feldspar-muscovite-biotite alteration together with bed selective garnet (intermediate almandine-grossular-spessarite) – biotite alteration in some units close to the ironstone. This involved the infiltration of hot (>500°C), very saline, and Na-K-Fe-Ca-Mn-Cl rich fluids. Microprobe analysis of paragenetically late, high-salinity and highly concentrated lead and zinc fluid inclusions suggests lead and zinc were mobile beyond the boundaries of the mineralized ironstone, which is consistent with the possibility that the lead was transported, or even concentrated, from more than one host rock source during the late orogenic metasomatic event.

9 Exploration

9.1 2017 TEM survey and follow up

Since 2014 VTT has carried out a limited regional exploration program that has comprised drilling and a transient electromagnetic (TEM) survey in an area to the south of the Pegmont deposit. To put the exploration work into context the background and drilling is also discussed in this section.

In 2015 RC / diamond drillhole PVRD022 was drilled targeting a major interpreted break in the 2014 airborne magnetics, a feature considered important to the understanding the potential strike and dip continuation of the Pegmont deposit. This was followed up in 2017 with two RC drillholes PVR097 and 098, to determine the attitude of the structure. Both drillholes intersected a typical Pegmont sequence, with PVR097 intersecting the structure.

In 2016 three exploration RC drillholes, PVR038, 39, and 040 were drilled to test low level magnetic highs. PVR038 was also located in close proximity to a low-level EM conductor identified by Aberfoyle in the 1990 GEOTEM survey. PVR038 or 039 intersected similar lithologies seen at Pegmont but no significant mineralized was intersected.

PVR040 intersected 3 m of visible coarse chalcopyrite mineralization centred in a 10 m wide intersection of silica, k-spar and pyrite altered pegmatite. The high-grade centre of the zone assayed 3.0 m at 3.21% Cu and 0.57 g/t Au from 113 m downhole. The hangingwall alteration, intersected over 5 m, assayed 0.09% Cu, 0.02 g/t Au, and the footwall alteration zone, intersected over 5 m, assayed 0.24% Cu, 0.10 g/t Au. Additional drilling is required to estimate the true thickness.

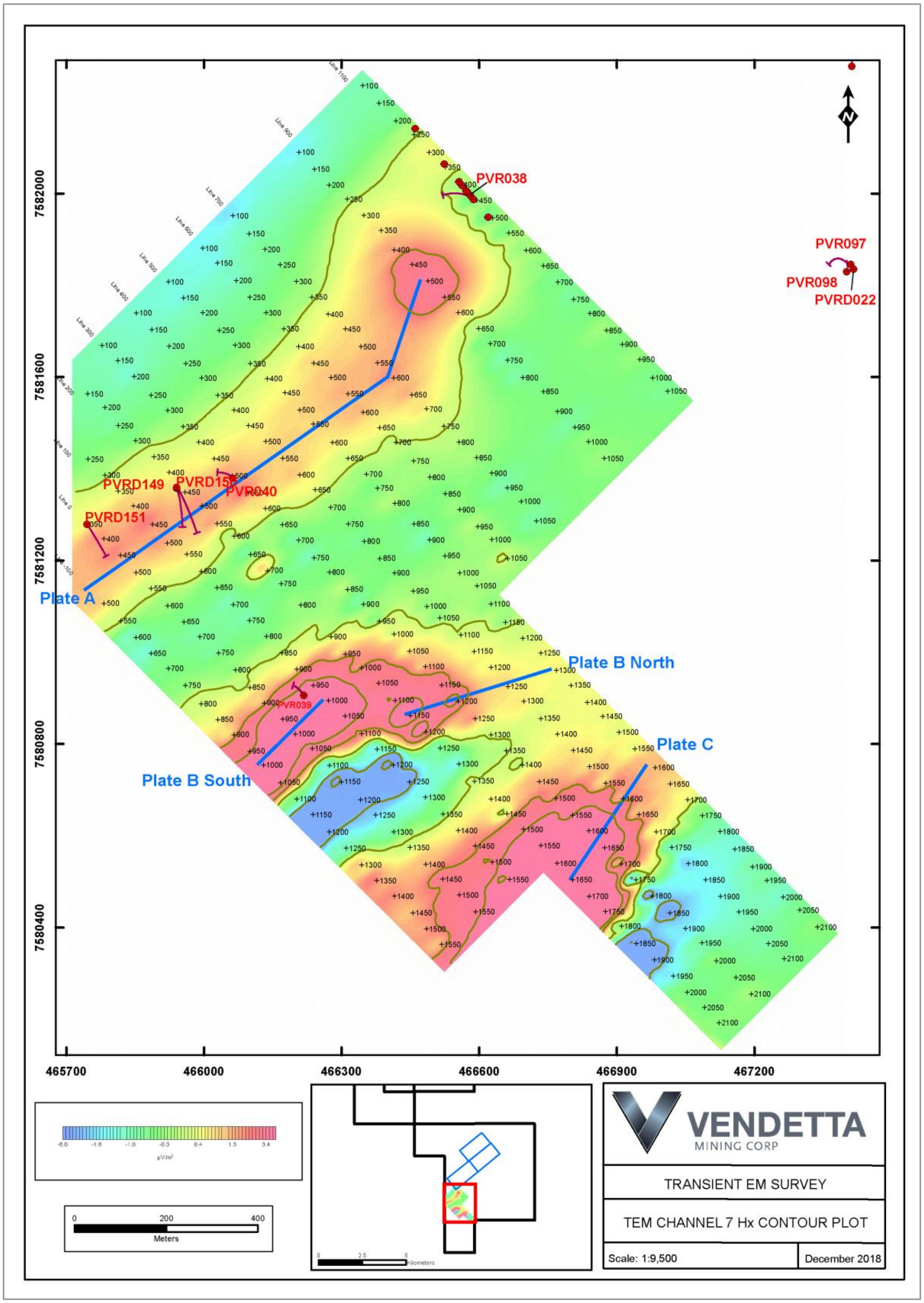
To determine a possible strike and dip and extent of the copper mineralization seen in PVR040 a Transient Electro-Magnetic (TEM) survey was conducted in May 2017 by Zonge Engineering & Research Organization (Aust) Pty Ltd. The survey employed an EMIT SMARTem24 receiver to record the EM data. The SMARTem24 is a 24-bit receiver with programmable gains on each channel. Three channels of data were recorded at each station; Hz (vertical), Hx (horizontal inline), and Hy (horizontal cross-line). The primary field was supplied by a Zonge transmitter providing a TEM waveform of approximately 30 – 50 Amps in a moving 100 m by 100 m single turn loop. The data was gathered at line separations of 100 – 200 m, with the spacing between successive stations on each line of 50 m. The grid was truncated to the west due to the lease boundary.

The TEM survey data was processed by Frontier Geoscience Inc. Three areas of anomalously high EM response are present. Anomaly A is the primary target in the north-west of the survey area. Anomaly B is located in the central section of the survey area, and Anomaly C is in the south-east. All three anomalies are open to the south-west and display a reduction in amplitude to the north-east.

In 2017 three RC / diamond drillholes were drilled to test the TEM conductor, two holes (PVRD149 and 150) were completed without intersecting sufficient sulphide mineralization to explain the anomaly. PVRD149 intersected a zone of haematite dusted carbonate-quartz-pyrite with trace chalcopyrite. The third hole, PVRD151 was suspended after only about 20 m of core was drilled due to the onset of the wet season.

Figure 9.1 shows a contour plot of channel 7 Hx from the Transient EM survey and the VTT exploration drillholes.

Figure 9.1 Transient EM survey contour plot of channel 7 Hx



Source: VTT

10 Drilling

10.1 Introduction

Since the commencement of the 2014 drilling program VTT has invested considerable time scrutinizing historical reports for drillhole related information to arrive at an acceptable level of confidence in both the location and quality of the data used in the 2018 Mineral Resource estimation. Details of the validation of historical data completed by VTT are described in Section 12.

Table 10.1 includes all holes in the database, although some of the drillholes were completed off the Property. This table has been updated by AMC since the 2017 NI 43-101 Technical Report to include details of the drillholes recently completed by VTT. The table includes all holes drilled up to the end of the 2017 drilling campaign. The database was closed-off for the Mineral Resource estimate on 15 April 2018, once final assay results were received.

Table 10.1 Summary of drilling on the Property

Year	Operating company [owner]	Number of drillholes				Metres drilled			
		Rotary / percussion	RC*	Diamond	Percussion / RC / diamond tail	Rotary / percussion	RC*	Diamond	Total
1971	Placer	53	-	4	11	3,482.2		1,283.6	4,765.8
1975	Newmont [Newmont JV]*	46	-	14	-	1,098.8		4,709.8	5,808.5
1994	BHP Minerals [Newmont JV]	-	2	-	1		397.0	113.0	510.0
1996	PML	-	27	-	-		2,442.0		2,442.0
1998 / 99	North Ltd [PML]	-	6	8	1		1,251.0	1,772.7	3,023.7
2000	PML on behalf of Billiton Agreement [PML]	-	-	-	2		343.0	189.0	532.0
2005	PML	-	1	7	7		360.0	710.0	1,070.0
2008	Cloncurry Metals Ltd Option [PML]	-	52	-	17		7,537.3	1,905.8	9,443.1
2008	BHP Billiton Due Diligence for Cloncurry Metals Ltd Option [PML]		44				4,600.0		4,600.0
2009					3		203.6	91.1	294.7
2009	Cloncurry Metals Ltd Option [PML]				12		1,730.0	1,640.5	3,370.5
2009	PML	-		-	5			834.0	834.0
2010	PML	-	56	-	-		3,482.0		3,482.0
2012	PML	-	3	-	3		654.0	581.0	1,235.0
2013	PML	-	21	-	-		3,000.0		3,000.0
2014	PML (reimbursed by VTT)	-	13	-	-		2,977.0		2,977.0
2014	VTT	-	6	2	10		1,935.0	1,216.2	3,151.2
2015	VTT	-	-	-	1		146.0	553.9	699.9
2016	VTT	-	5	-	31		4,349.8	2,893.4	7,243.2
2017	VTT		3	2	109		11,866.2	11,926.7	23,792.9
Sub total		**99**	**239**	**37**	**213**	**4,580.9**	**47,273.9**	**30,420.7**	**82,275.6**

Note: Meterages rounded to 1 decimal place.
*Includes abandoned RC pre-collars.

Table 10.2 shows the percentage of drilling by drilling type. Much of the drilling was undertaken by RC drilling. A significant number of meters were also drilled using diamond drilling (DD), either from surface or as tails to RC drillholes.

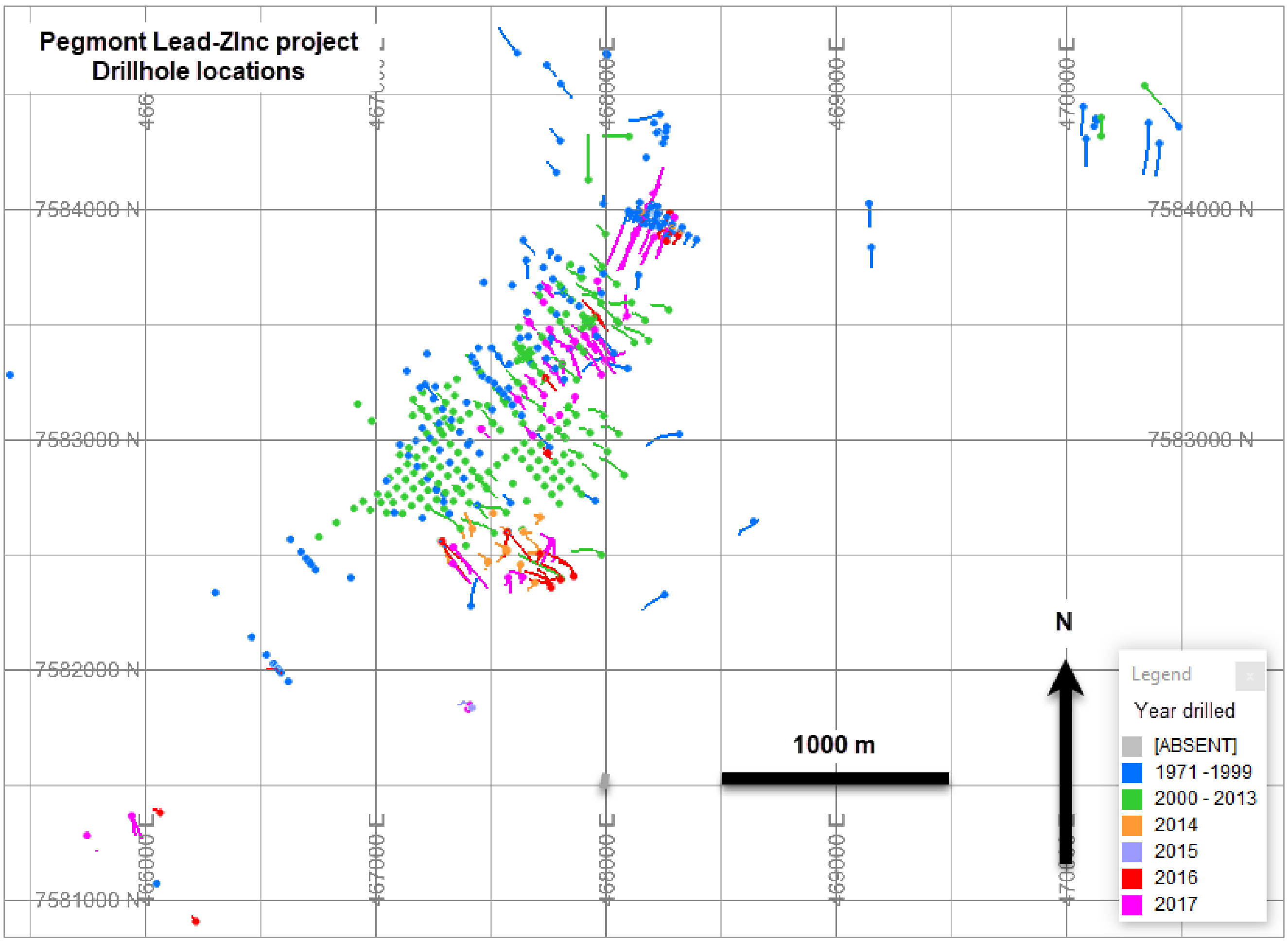
Table 10.2 Percentage of drilling type

	Rotary / percussion	RC	DD
Number of holes	17%	41%	42%
Number of metres	6%	57%	37%
Number of meters > 1% Pb + Zn	10%	42%	49%

Note: Number of DD holes includes holes with RC precollars and DD tails.

The locations of the Pegmont drillholes are shown in Figure 10.1.

Figure 10.1 Drillhole collar locations by year drilled



Source: AMC

Drillhole data is stored in an Access database which is managed by VTT. Of the 588 drillholes in the database, 89% have intersected significant mineralization as defined by greater than 1% Zn+Pb. The mineralization outcrops in part and the deepest intersections are approximately 400 m below surface (-100 mRL).

Representative examples of drill sections through the deposit are shown in Section 14.

10.1 Collar surveys

Prior to 2010, a local grid established by Newmont was used to locate drillholes at Pegmont. In 2010 a local licensed surveyor, M. Lodewyck from Mount Isa, (Lodewyks), converted the locations of the drillhole collars to the current standard datum and grid used in Australia (Map Grid of Australia, 1994 (MGA94) Zone 54, a UTM-based grid using GRS80 ellipsoid).

Since 2010, Lodewyks have re-surveyed the locations of many historical drillholes and those of more recent drilling programs using differential GPS. Surveyed collar locations are received digitally and loaded into the database by VTT.

During 2017, Lodewyks advised VTT of an error in the locations of the 2016 drillhole collar surveys (6 m displacement in easting) and the error has been corrected for this Mineral Resource update. In addition, VTT continued the re-surveying of historical drillhole collars during 2017, with a further 96 historical holes being checked. At the completion of the 2017 program, 418 collars in total had been located by Lodewyks, (and its predecessor company RC Todd and Associates), leaving 159 historic drillholes to be checked. Note that 46 of the 159 drillholes are percussion holes whose collars are no longer visible on the ground.

Aerial photography was flown in October 2010 and a new digital terrain model (DTM) was produced with an accuracy of +/-0.5 m. This DTM was used to update elevations for all collars not picked up by the surveyor M. Lodewyck. This is considered to be sufficiently accurate in a predominantly flat landscape for the purposes of this Mineral Resource estimate.

The topography is flat and drillhole collars are all in the range of 288 to 310 m above sea level.

10.2 Downhole surveys

Drillhole depth, dip and direction are highly variable at Pegmont and this is a reflection of the complex nature of the stratigraphy. A number of shallower angled drillholes have been completed in oxide to help with the interpretation of the recumbent folding.

Most drillholes at Pegmont have at least one downhole survey (200 drillholes are limited to a collar dip and azimuth with no recorded downhole surveys). The method used for downhole surveying is not known for many of the historic drillholes. Where the methodology is recorded, pre-2013, it is generally by Eastman single shot camera. During 2017, VTT has re-surveyed 12 historic drillholes using a true north finding gyro to confirm drillhole orientations and resolve apparent locational discrepancies. This work takes the total number of re-surveyed drillholes to 31.

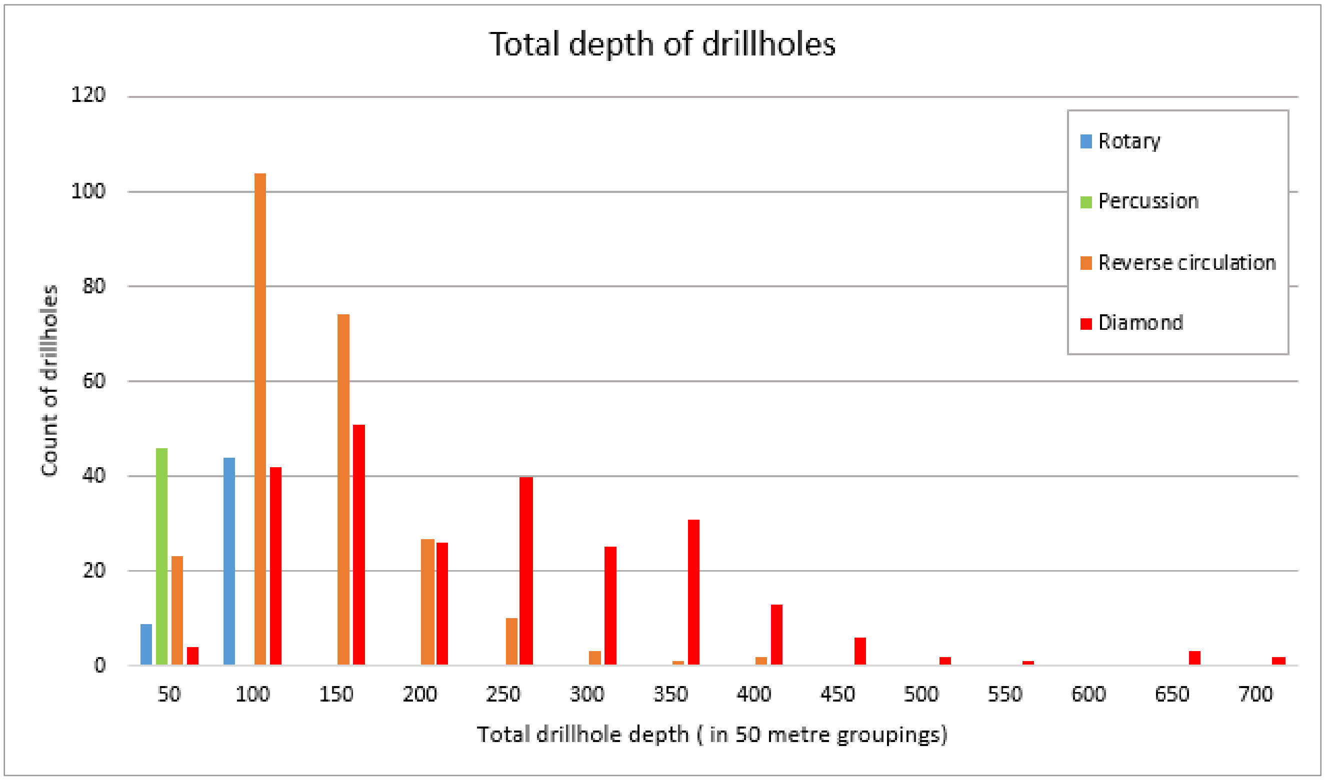
Drillholes completed 2014 – 2015 were surveyed using a Reflex multi-shot camera at 5 m intervals on completion of the drillhole. For the 2016 and 2017 drilling a 'Champ Navigator' true north finding gyro has been used at the end of each 6 m drill rod and the data is transmitted electronically (Bluetooth), to surface in real time for checking by the drilling supervisor.

Figure 10.2 shows the drillhole length by drilling method, which indicates that 54% of drillholes have a final depth in excess of 100 m.

A summary of drillhole dips and orientation is given in Figure 10.3. As the majority of the mineralization is interpreted to be essentially shallow dipping in Zones 1 to 4, the vertical drilling provides a reasonable test of much of the mineralization envelope. If the interpretation of this shallow dip is correct, the dips would not be expected to be at a high angle to most drillholes. Close to true width intersections are present on a number of sections, where the interpreted bedding dip is perpendicular to the drillholes. Other sections with dips at an angle to drillholes are not true width.

It was observed at site in a number of selected drill cores that the bedding dips were at moderate angle to the drillhole traces. In these cases, the drilling intercepts are not true width. AMC reviewed these selected holes in the context of the geological interpretation and considers the overall interpretation on the sections is reasonable for the bedding dips observed. Some core is at a higher angle and indicates the structure may be more complex in detail than shown in some parts of the geological interpretation. Further structural geology measurements and more detailed interpretation would assist to better understand this detail, but the shallow dipping interpretation used is considered to be reasonable overall and likely to support a conservative Mineral Resource estimate.

Figure 10.2 Drillhole length by drilling type



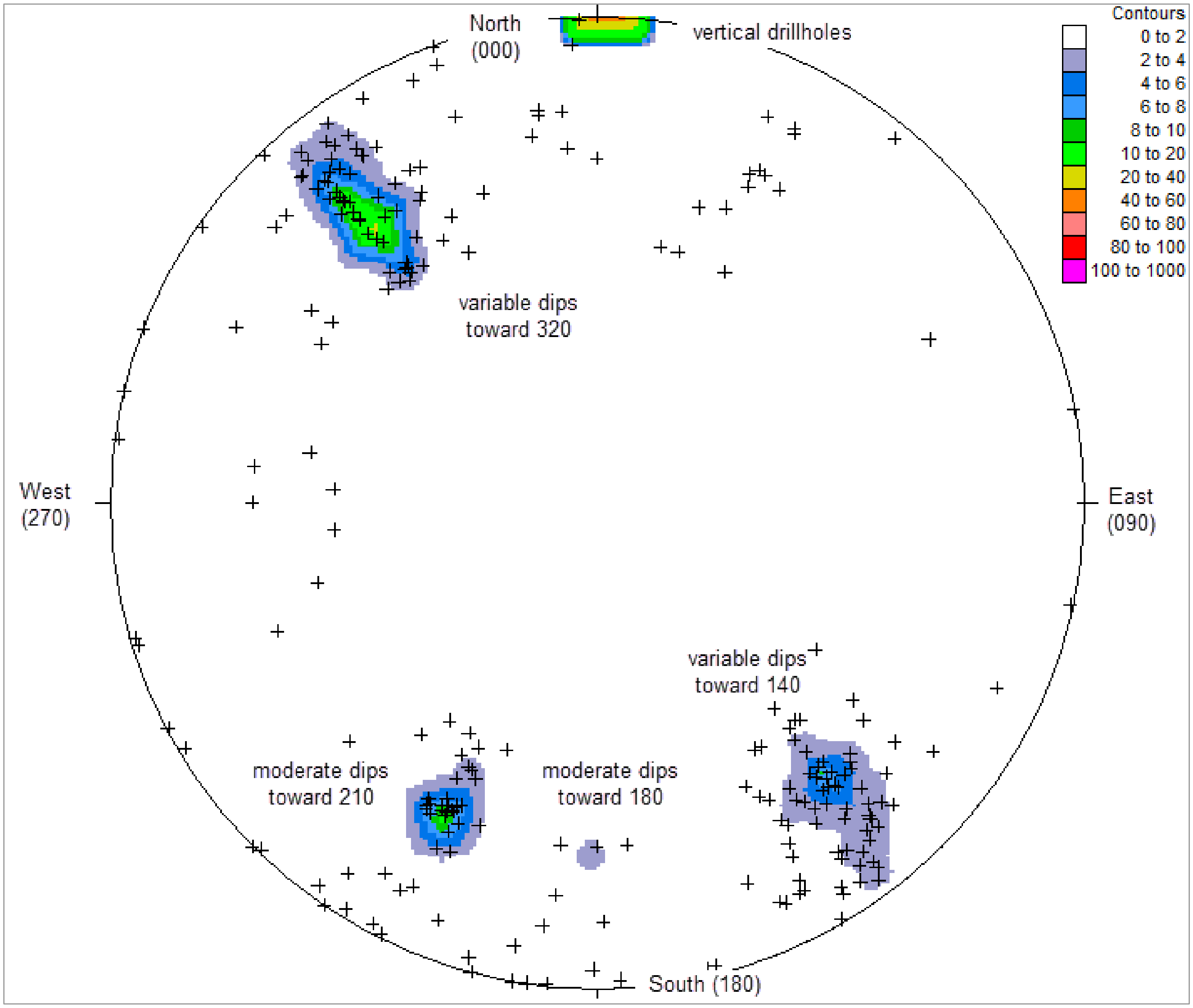
Source: VTT

To illustrate the range of dips and orientations of the drillholes a stereonet was prepared and is shown in Figure 10.3.

Clusters of data can be seen at several positions on the stereonet and the clusters indicating the most common drillhole orientations have been annotated. No single zone within the deposit is restricted to one cluster of orientations and this is consistent with understanding of local variability in the orientation of the mineralized zone that has been developed by VTT.

Note that these are collar setups, and there is a dip and strike allowance built in to the setup to ensure that the target is intersected at the planned point.

Figure 10.3 Stereonet of drillhole orientations at Pegmont



Source: VTT

10.3 Drillcore markup and orientation

Drillcore is laid out in a covered core logging area for pre-logging markup by the project geologist. All core trays are marked with drillhole identifier and tray number.

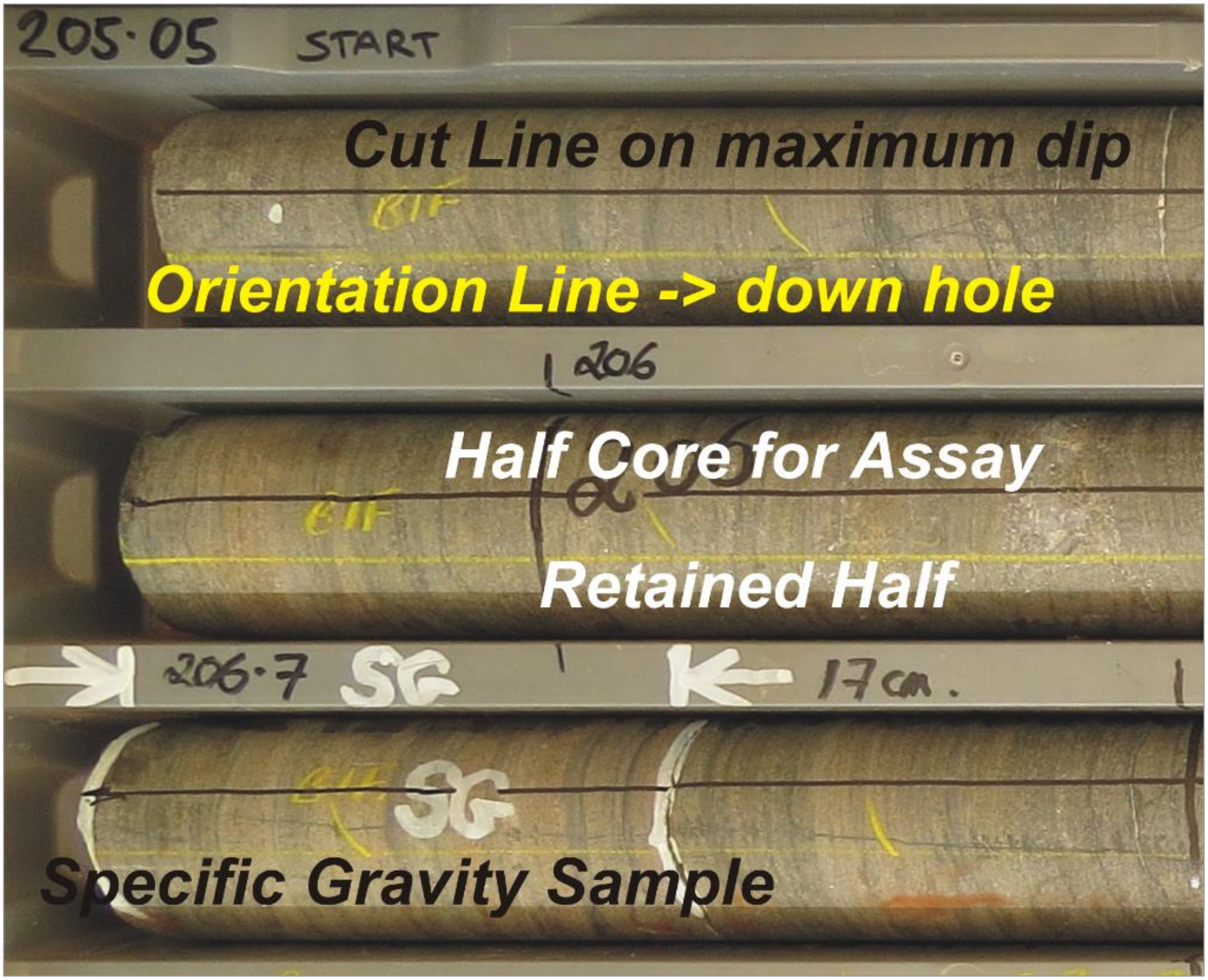
All drillcore is assembled on an angle iron to facilitate correct alignment of core orientation marks and recovery and RQD measurements. Depth downhole is marked in 1 m intervals and cross referenced with the drillers' end of run blocks.

VTT routinely have the drillers attempt core orientation at the start of each drill run. A Reflex ACT II (2014-2015) or Orifinder_DS1 (2016) tool is used (gravity controlled) to mark the bottom of the hole. The driller marks the core and the appropriate core block when a successful orientation mark has been made.

A bottom of hole orientation line is marked on the core as is a one-sided arrow indicating the downhole direction. The colour of the bottom of hole orientation line is determined by the number of consecutive drill runs with orientation marks falling within an 8 mm tolerance in the offset of consecutive orientation marks. Alpha and beta angle measurements are made on veins, discontinuities, bedding, and foliation. The measurements are later translated to their true dip and dip direction orientations using a software program (Dips).

A cutting line is marked, along an approximation of the maximum dip angle through the intersection, to facilitate consistency of core sampling down the drillhole as shown in Figure 10.4.

Figure 10.4 Example of drill core markup



Source: VTT

10.4 Logging

There have been several major companies involved with the drilling at Pegmont and each had a different system for collecting the data.

In March 1999, North (Collins, 2000) undertook a re-logging of 6,400 m of available diamond core from 27 diamond core drillholes drilled to 1975, which represented 97% of all diamond core drilled to that date.

Selected historic drillholes (32 in total) were relogged by VTT during the 2014, 2015, and 2016 field seasons to aid the interpretation of amphibolite units and expand on the summary logs previously available.

VTT has a Standard Operating Procedure in place for the collection of geological data on the Property. No material changes were made to the procedure for the 2017 drilling program.

RC chips are collected for each 1 m interval and stored in plastic chip trays. The RC chips are logged on geologic intervals and stored in a sealed sea container on site for later reference if required.

Drillcore is logged in geological intervals. Observations are recorded on paper sheets for later entry to the Access database when they are validated.

The current geological logging of diamond core and RC comprises recording the dominant lithology, alteration, veining, and mineralization. VTT uses a different set of fields to those used by previous operators, however, the historical lithology, alteration, texture, and mineralization fields used by previous operators have been amalgamated, where appropriate, with those used by VTT in consolidation of the database. The fields currently recorded in logging by VTT include:

- Lithology: main and secondary lithologies (lith and lith2), qualifier (lith_qual).
- Weathering and oxidation (weath and oxid).
- Dominant and ancillary minerals (lith_min1, lith_min2, lith_min3).
- Colour (colour) and a qualifier (color_qual).
- Alteration: alteration type (alt_1 or alt_2), intensity (alt 1 Intensity or alt 2 Intensity), and style (alt 1 qual or alt 2 qual).
- Veins: vein proportion of interval (vein_%), type (vein) style (vein qualifier) and dominant, and ancillary minerals (vein_min1, vein_min2).
- Mineralization: mineralization proportion (Gn%, Sp%), mineralization type (Gn minz type, Sp minz type), mineralization grain size (Gn grain size, Sp grain size).
- Estimates of significant minerals magnetite (mag%), chalcopyrite (ccp %), pyrite (py%), and pyrrhotite (po%).
- General comments (comments).

Three historic drillholes from Zone 3 were relogged by VTT during the 2017 program as only summary logs were previously available. No additional historic detailed drill logs were located. 47% of the original drill logs have now been located by VTT and it is not anticipated that this number will increase.

10.5 Core recovery

A total of 6,775 core recovery and 6,206 rock quality designation (RQD) measurements exist in the database.

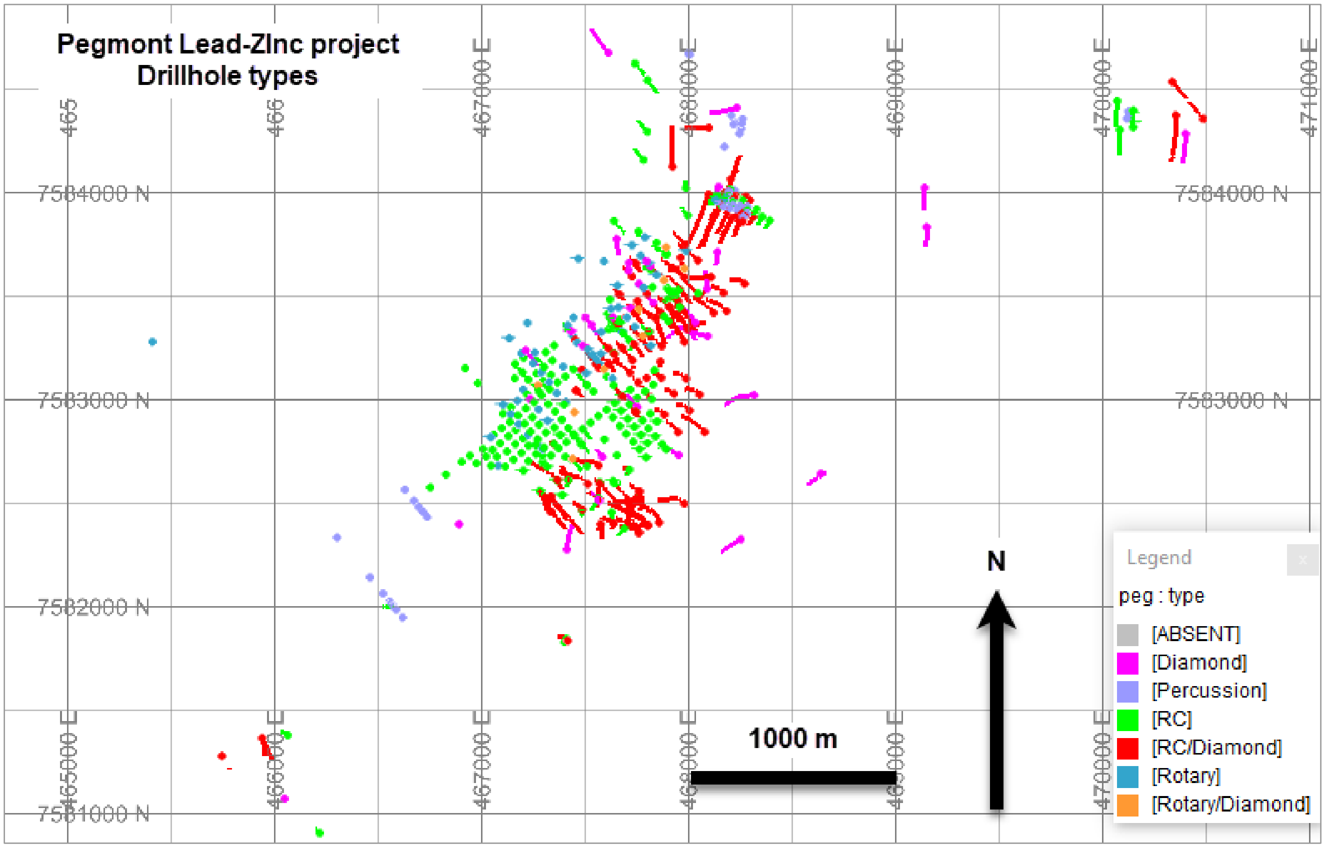
Core recovery information for the historical drilling is restricted to measurements, taken per drilling run, by PML during the 2008 DD program. Overall, recovery was very good, ranging between 97% to 100%. PML also recorded qualitative RC recoveries (approximate percentage of normal sample volume recovered) and moisture contents during their 2007 – 2008 drilling. All but three RC holes were calculated to be within the "good" PML RC sample recovery. It is noted that once recoveries reduce, for example to less than 90%, the confidence in sampling representivity becomes progressively less.

Qualitative RC sample moisture is recorded in historic RC drilling and whilst there are a low number of wet or moist samples recorded the majority are noted as dry. It is noted that moist, wet, and saturated drillholes are likely to become progressively less confident in sampling representivity.

Core recovery and RQD are routinely recorded by VTT. The measurements indicate generally very high core recovery (>99%) and RQD (90%). Core recovery and RQD are reduced through some intersections of mica or biotite rich schist, but elsewhere the drillcore is generally very competent. VTT measure RQD and recovery across variable interval lengths that correspond to the logged geological intervals (rather than the more usual practice of measurements over uniform intervals of 1 or 2 m). This method of measurement may result in smoothing of the recovery and RQD data and loss of definition of narrow zones of low recovery or RQD.

The use of combined RC and DD samples in the resource estimate introduces a mixing of datasets with different confidence levels, (Figure 10.5). Aside from a single comparison of a twinned RC drillhole against a diamond drillhole during the 2005 PML program, the QP is not aware of any studies by VTT comparing the quality of the different types of drilling samples. This mixing of datasets has the potential to introduce a lower level of confidence in the resulting Mineral Resource estimate. Whilst VTT use DD though the mineralization zone, further twinned drilling adjacent to selected historical RC holes is recommended.

Figure 10.5 Drillhole locations, coloured by drillhole type



Source: VTT

10.6 Bulk density

Historic resource estimates by Newmont and Behre Dolbear used bulk densities of 3.75 t/m^3 and 4.2 t/m^3 for oxide and sulphide mineralization respectively. No original data from the Newmont bulk density tests are available.

There have been two more recent campaigns of obtaining bulk density measurements. PML collected bulk density measurements in 2007, which included 507 measurements from 37 drillholes, as well as material from earlier programs. A second bulk density measurement program by PML in 2008 comprised 435 measurements from 16 drillholes drilled during the 2008 program.

These two data sets were merged by VTT into the current database and validated. The wax coated immersion method is thought to have been used for the 2007 and 2008 measurements.

The current bulk density database contains 1,871 density measurements. VTT routinely undertake calibration measurements and these are not included in the total tally. The calibration measurements are taken at the start of each batch of measurements using a piece of competent pegmatite from the Project (PVRD006 127.8 – 180.0 m).

VTT has a Standard Operating Procedure in place for the measurement of bulk density. A representative sample of approximately 0.2 m length is chosen by the geologist for each assayed interval in mineralized lithologies (garnet sandstone and banded iron formation) and 1 sample per 5 m through the hangingwall and footwall sequence. The position of the sample is marked on the core prior to core photography as a further record of its location. Bulk density is calculated using the immersion method on a set of digital scales which are set up over a large bucket full of water. A wire cage is used to hold the sample during the immersion. The apparatus is set up in a sheltered position and weights are recorded to the nearest 1 gram. The equation used by VTT to calculate the bulk density is:

$$bulk\ density = \frac{weight\ in\ air\ (g)}{weight\ in\ air\ (g) - weight\ in\ water\ (g)}$$

Table 10.3 shows the average bulk densities of logged lithologies at Pegmont.

Table 10.3 Bulk density data summary

Weathering	Major lithology	Rock codes	Number of samples	Average bulk density (t/m³)	Standard deviation (t/m³)
Highly weathered	Banded Iron Formation	BIFF	33	3.25	0.39
	Schists – various types	BISH	8	2.42	0.11
		SCHS			
Moderately weathered	Banded Iron Formation	BIFF	87	3.24	0.43
	Garnet Sandstone	GASS	16	2.95	0.38
	Psammite	PSAM	11	2.67	0.15
	Undifferentiated Metasediment	MTSD	11	2.74	0.31
	Schists – various types	BISH	9	2.66	0.21
		BQSH			
		QMSH			
		SCHS			
Slightly weathered	Banded Iron Formation	BIFF	53	3.61	0.4
	Garnet Sandstone	GASS	19	3.05	0.48
	Psammite	PSAM	28	2.77	0.22
	Undifferentiated Metasediment	MTSD	7	2.75	0.09
	Schists – various types	BMSH	17	2.73	0.11
		BQSH			
		QBSH			
		QMSH			
		SCHS			
Unweathered	Banded Iron Formation	BIFF	645	4	0.43
		SXX			
	Banded Iron Formation + Garnet Sandstone	BIFF + GASS	112	3.87	0.54
	Garnet Sandstone	GASS	278	3.26	0.34
		GAQZ			
	Psammite	PSAM	262	2.81	0.27
		PSCH			
	Undifferentiated Metasediment	MTSD	14	2.78	0.07
	Schists – various types	B1SH	125	2.84	0.24
		BQSH			
		QBSH			
		QMSH			
		SCHS			
	Schists + Garnet Sandstone	B1SH	8	3.21	0.38
		BCSH			
		BQSH			
		QBGN			
		QBSH			
		SCHS + GASS			
	Amphibolite	AMPH	53	3.03	0.15
	Pegmatite	PEGM	16	3.25	0.4

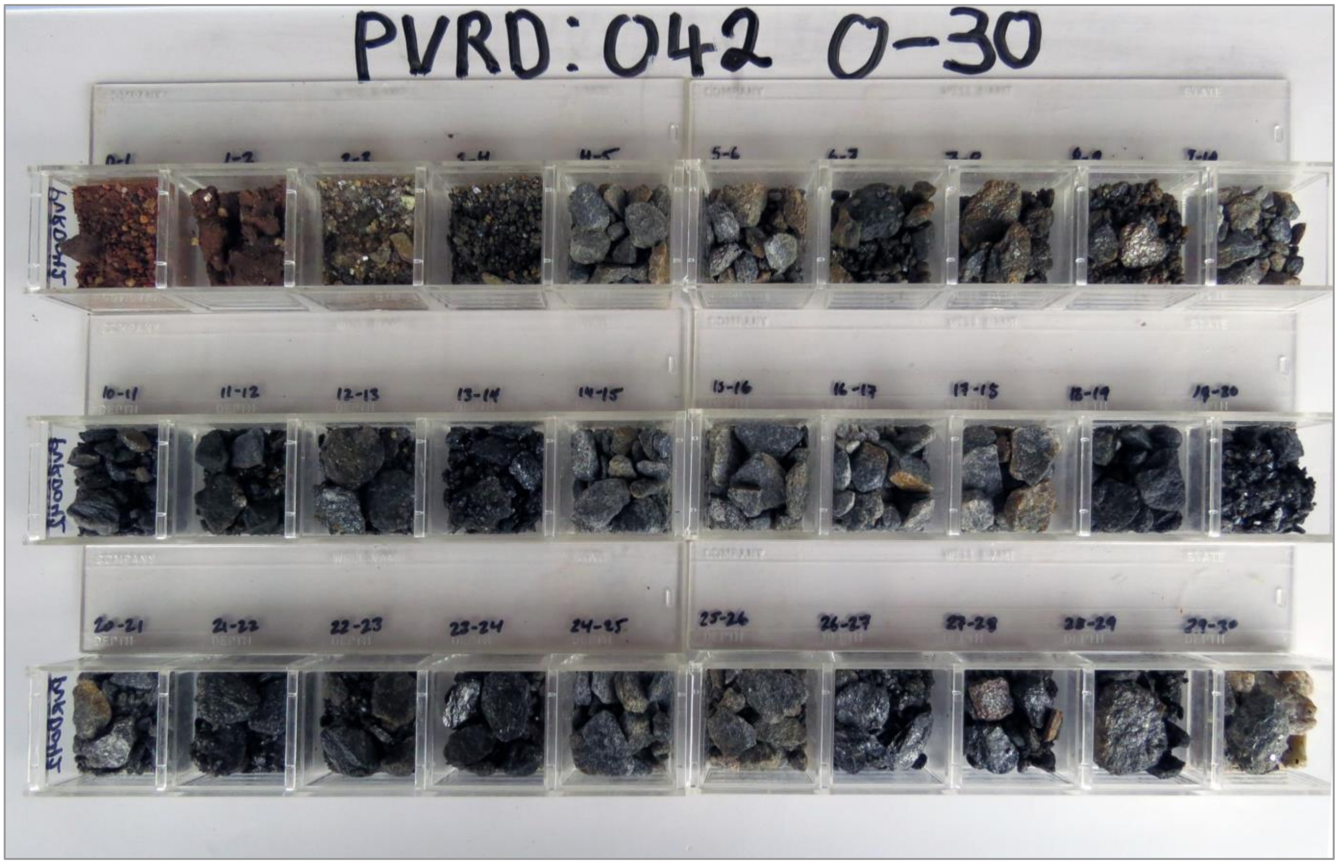
10.7 Core and RC chip photography

VTT photograph all RC chip trays and drill core. RC chips are sprayed with water before photography.

Drill core is photographed after all sampling and bulk density sampling marks have been made. The orientation line is shown in the photograph and major lithologies are noted on the core. Drill core is photographed both wet and dry and includes a colour match chart and meter rule for scale.

Examples of the quality of photographs of both RC chips and core are shown in Figure 10.6 and Figure 10.7.

Figure 10.6 RC chip tray photography



Source: VTT

In Figure 10.7 an example of dry and wet drill core photography is shown for a tray which averages 9% Pb + Zn.

Figure 10.7 Example of dry and wet drill core photography

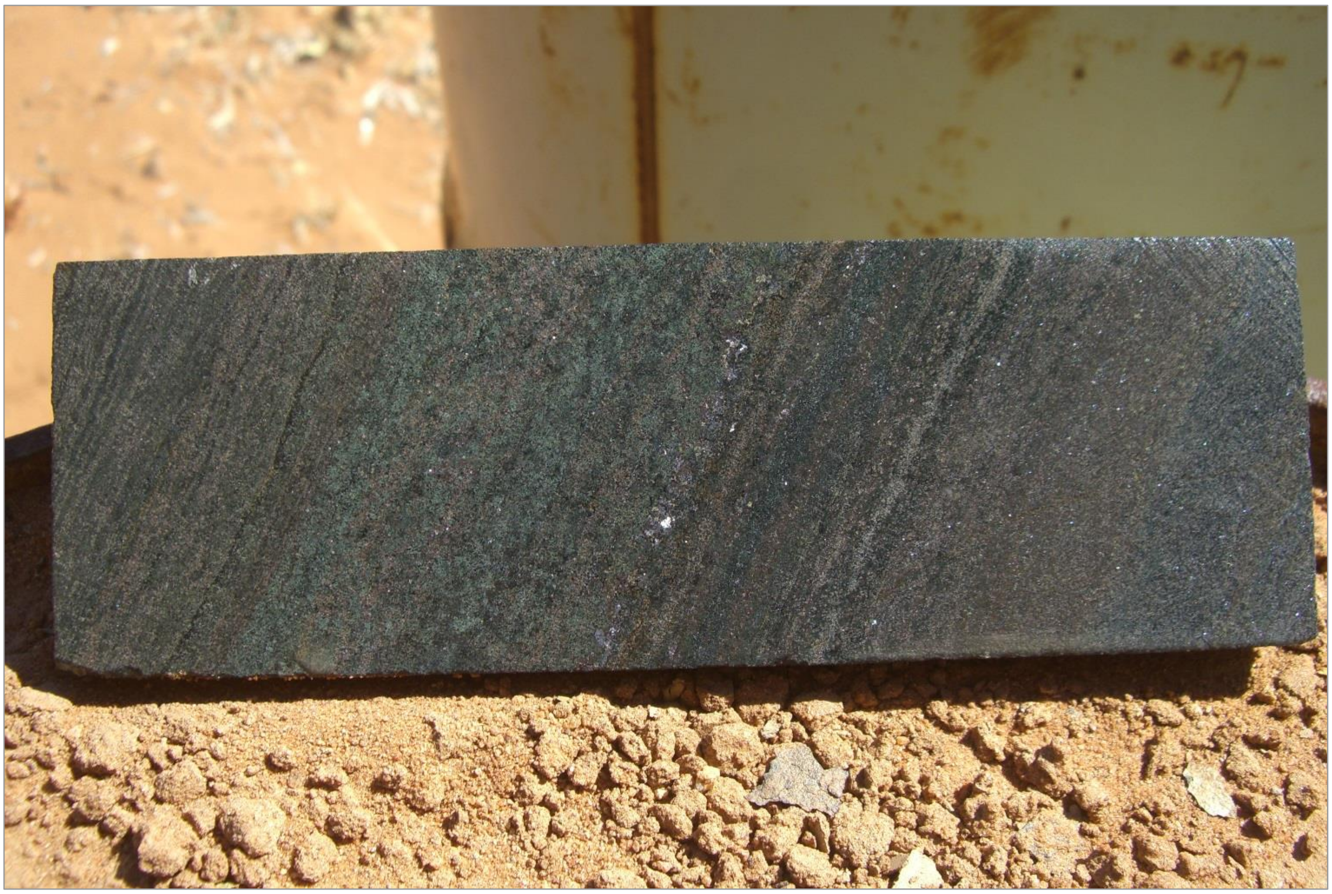
Source: VTT

10.8 Observations on drilling from site visit

A site visit was undertaken by the QP in May 2017. Drilling procedures were provided to the QP prior to the commencement of the drilling program and these were used as a cross reference during the QP's review.

A drill core inventory and layout plan were provided, and the QP reviewed the mineralized intersections in 16 drillholes and found no issue with their location, which was as logged, in each drillhole. Figure 10.8 is an example of mineralized core.

Figure 10.8 BIF hosted galena and sphalerite



Note: From drillhole PVRD001 126.4 – 126.6 m. Core diameter 63.5 mm.
Source: VTT

In fresh BIF in core, observable textures, grain size, and sulphide mineral species appeared to be similar across drillholes. Evidence was found of multiple deformation events in the metasediments as shown in Figure 10.9.

Figure 10.9 Folding in psammite

Note: Drill intercept shown is from drillhole PVRD056 between 98.2 -98.5 m. Core diameter 50.5 mm.
Source: VTT

Mineralized intersections of historical drill core, and all available RC chips are stored undercover by VTT to reduce deterioration. The drill chips observed were clean, well labelled and in good condition. Drill core from VTT programs is stacked in the open with lids on the upper core trays to minimize weathering and core deterioration. Most of the historical core trays observed under cover are well labelled. Unmineralized historic core trays are stored neatly in the open and are in poorer condition with some labelling illegible.

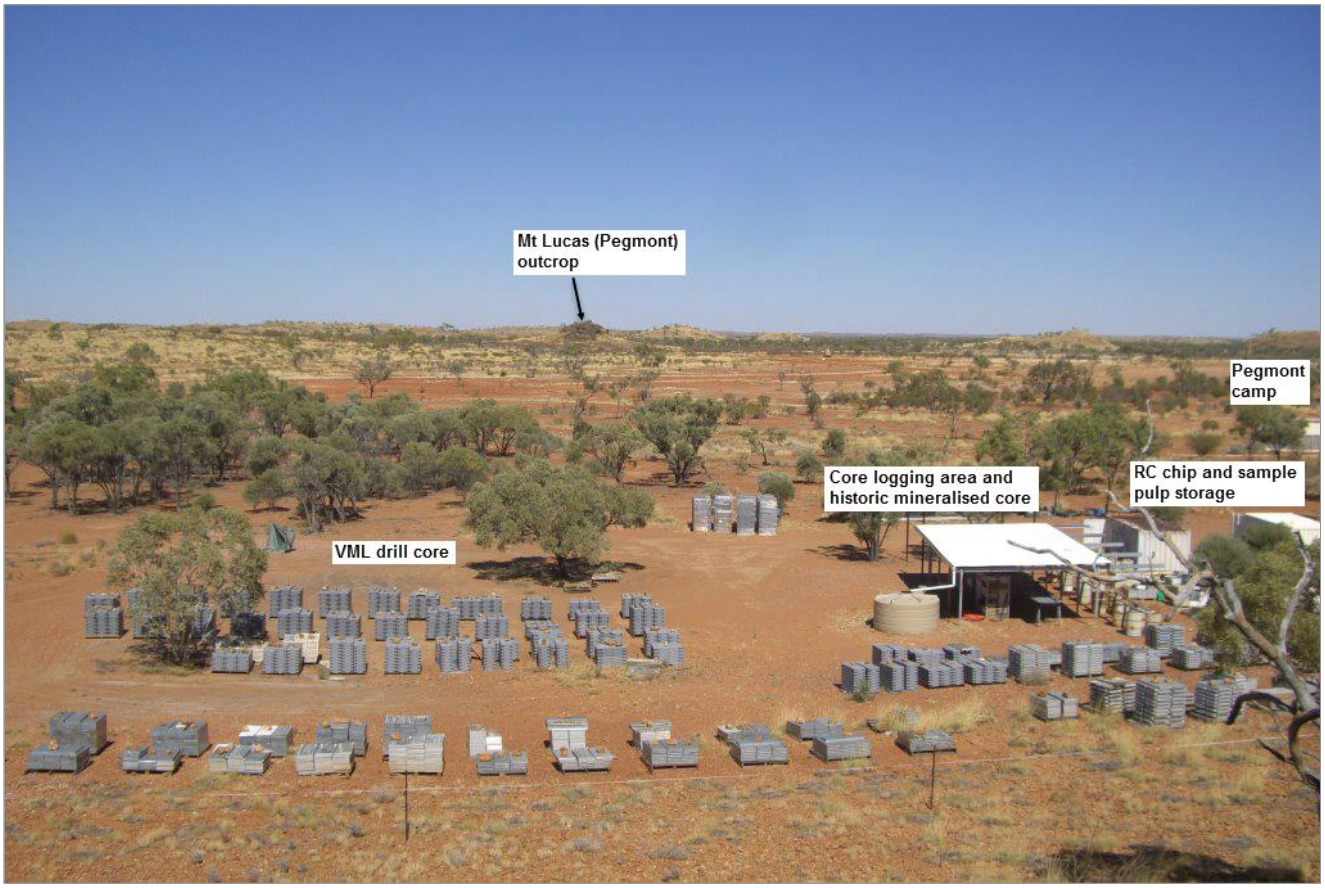
Most of the core observed was HQ in diameter and all observed intervals logged as mineralized BIF had been sampled. Remaining core is predominantly half core or quarter core. Quarter core remains where metallurgical has been completed.

Core recovery in the observed drill core was very good, reflecting the high core recovery calculations described in Section 10.6. The QP queried the measurement method used for core recovery and RQD and advised the use of a uniform interval for the calculations.

In addition, the QP reviewed the core markup and logging procedure, and the core and chip storage facility (Figure 10.10). Core orientation mark-up downhole was observed in diamond drill core and the method of core markup though oriented zones demonstrated by VTT geologist David Esser.

As drilling had only recently commenced the bulk density measurement apparatus had not been set up. However, the components were sighted, and their general operation demonstrated.

Figure 10.10 View looking west across core storage and logging facility to Pegmont gossan



Source: VTT

Large plastic bags containing reject RC sample after riffle splitting were observed at recent drillhole sites from the 2017 drilling program. The drill rig was diamond coring at the time of the site visit, so it was not possible for the QP to observe RC sub-sampling procedures.

The QP visually inspected several drillhole collars and cross-checked labelling and orientations against those in the database. Outcropping mineralization at the main Pegmont gossan was observed (Figure 10.11) as was the gossan material from the BHZ.

Figure 10.11 Pegmont gossan, looking east-north-east

Source: VTT

The presence of BIF-hosted lead-zinc mineralization is supported by the QP’s observation of galena and sphalerite in fresh BIF in core and the presence of honeycomb-textured gossans in outcrop at site.

11 Sample preparation, analyses, and security

11.1 Introduction

There have been many different drilling programs on the Property spanning four decades, using different laboratories for sample preparation and analysis. This section only includes commentary on the work carried out by VTT, with some tables summarizing historical information to give continuity. The historical work carried out by PML is reported in the 2014 AMC Technical Report. In summary, approximately two thirds to three quarters of the Certified Reference Materials (CRMs) submitted pre-2014 returned assays within an acceptable range above or below the certified value. The QP considers the data since 2007 provides a reasonable representation of the location of mineralized and unmineralized BIF horizons and is appropriate for use in geological interpretation and Mineral Resource estimation. There is limited information on the sampling procedures used and no quality assurance and quality control (QA/QC) data prior to 2007. The level of uncertainty in the early data is reflected in the Mineral Resource classification.

A summary of the current understanding of the status of the assay certificates and QA/QC for the various drilling programs at Pegmont since discovery is given in Table 11.1.

Table 11.1 Summary of Pegmont data status

Year	Operating company	Metres and % of total	Assay certificates and QA/QC
1971	Placer	4,765.8	No digital assay certificates available. No record of QA/QC.
		6%	
1975	Newmont [Newmont JV]	5,808.5	No digital assay certificates available. No record of QA/QC.
		7%	Copper data from 34 of 60 holes manually entered from Sheppy and Verwoerd (1975). Existing lead, zinc, and silver assays in the database cross-checked against Newmont reports.
1994	BHP Minerals [Newmont JV]	510.0	No digital assay certificates available, paper copies thought to be at Pegmont camp. No record of QA/QC.
		1%	
1996	PML	2,442.0	No digital assay certificates available, paper copies located at Pegmont camp. ALS Chemex Townsville hard copies seen at site. No record of QA/QC.
		3%	Copper, lead, cadmium, phosphorus and antimony assays manually entered from 3 hard copy assay certificates. Existing lead, zinc, and silver assays in the database cross-checked against Newmont reports.
1998 & 1999	North Ltd [PML]	3,023.7	No digital assay certificates available, assays and sample numbers available as transcribed data in Collins (2000) have been manually entered. Standards were used, one per drillhole, results not in database. No other record of QA/QC.
		4%	
2000	PML on behalf of Billiton Agreement [PML]	532.0	No digital assay certificates available, paper copies from ALS Chemex located at Pegmont camp. No record of QA/QC.
		1%	
2005	PML	1,070.0	ALS Townsville digital assay certificates available, paper copies located at Pegmont camp. No record of QA/QC.
		1%	
2008	Cloncurry Metals Ltd Option [PML]	9,443.1	ALS Townsville assay certificates in PDF file formats available.
		11%	
2008	BHP Billiton Due Diligence for Cloncurry Metals Ltd Option [PML]	4,600.0	SGS Townsville assay certificates in PDF and SIF file formats available.
		6%	
2009		294.7	
		0.4%	
2009	Cloncurry Metals Ltd Option [PML]	3,370.5	ALS Townsville assay certificates in PDF file formats available.
		4%	

Year	Operating company	Metres and % of total	Assay certificates and QA/QC
2009	PML	834.0	ALS Townsville assay certificates in PDF and / or CSV file formats available.
		1%	
2010	PML	3,482.0	ALS Townsville assay certificates in PDF and / or CSV file formats available.
		4%	
2012	PML	1,235.0	No significant results, no analysis.
		2%	
2013	PML	3,000.0	ALS Townsville assay certificates in PDF and file formats available.
		4%	
2014	PML	3,000.0	ALS Minerals Townsville assay certificates in PDF and CSV file formats available.
	(Reimbursed by VTT)	4%	QA/QC program of CRMs, blank and field duplicates
2014	VTT	3,151.2	ALS Minerals Townsville assay certificates in PDF and CSV file formats available.
		4%	QA/QC program of CRMs, blank and field duplicates
2015	VTT	699.9	No significant results, no analysis.
		1%	
2016	VTT	7,243.2	ALS Minerals Townsville assay certificates in PDF and CSV file formats available.
		9%	QA/QC program of CRMs, blank and field duplicates.
2017	VTT	23,792.9	ALS Minerals Townsville assay certificates in PDF and CSV file formats available.
		29%	QA/QC program of CRMs, blanks, field duplicates and umpire laboratory checks (SGS Townsville)

Note: CRM=Certified Reference Material.

Although all drilling within the model area was used in the Mineral Resource estimate, areas where the block model was populated only by older drilling was classified as Inferred to reflect the lower degree of confidence in data with no QA/QC data.

11.2 Sampling method and approach

11.2.1 Reverse circulation drillhole samples

Drillholes completed by VTT generally have RC precollars that are designed to fall short of the mineralization by approximately 20 m. Whole 1 m intervals of chips are collected at the drill rig and stored in large plastic bags until the chips have been logged and intervals to be sampled have been decided. If logging indicates the need for sampling the bagged chips are passed multiple times through a 50:50 riffle splitter to produce a 2 – 3 kg sample for despatch to the assay laboratory.

11.2.2 Diamond drill core samples

VTT sample all garnet sandstone and BIF horizons intersected, regardless of the visual grade estimate made by the geologist.

Sampling of mineralized zones begins at the hangingwall contact. Sample intervals, 1 m in length, are marked downhole from this point with a minimum 0.5 m sample at the footwall contact. Samples are continued 5 m into the hangingwall and footwall sequences.

Diamond core is prepared for sampling as described in Section 10.4. Drill core is cut in half using a diamond saw on site and 1 m sample intervals are transferred to labelled calico bags. CRM and blank samples are inserted into each drillhole inside separate calico bags.

The calico bags are packed into large polyweave sacks and transported to the NQX transport depot in Cloncurry for transport to the ALS laboratory in Townsville.

11.3 Laboratories and analysis

Sample preparation for the VTT samples has been completed at ALS Townsville using the following sample procedures. The assay methods chosen by VTT are the same as those used by the previous operator (PML).

The ALS Townsville laboratory has been ISO9001 certified since February 2011 and the Brisbane laboratory is ISO/IES 17025 certified. During the 2008 BHP program the SGS laboratory in Townsville was used. SGS operate in accordance with ISO/IEC 17025.

Both ALS and SGS are independent of the issuer.

11.3.1 Sample preparation

Sample preparation entailed the following steps at ALS:

- Sorting and checking of samples.
- Recording of received sample weight (ALS code WEI-21).
- Crushing of entire samples to >70% passing 6 mm (ALS code CRU-21).
- Splitting of samples and pulverizing to 85% passing 75 microns (ALS code PUL-23).
- RC, and if needed, diamond core samples, are riffle split to sub-sample (SPL-21).

Sample preparation entailed the following steps at SGS:

- Sorting and checking of samples (SGS code MSC80).
- Drying and pulverizing of RC samples to 75% passing 75 microns and split to <3.5 kg (SGS code PRP86) or in the case of diamond core.
- Drying and crushing of entire diamond core sample (-6 mm); pulverizing to 75% passing 75 microns and split to <3.5 kg (SGS code PRP88).

11.3.2 Sample analysis

Analysis by ALS or SGS is initially by a multi-element geochemical suite with over-range geochemical samples re-analyzed using assay methods. During the 2017 program, samples were analysed for lead, zinc, silver, cadmium, iron, manganese, and total sulphur.

The multi-element geochemical methods have changed over time; two different ICP methods have been used, at ALS: ME-ICP61s (up to 27 elements) and ME-ICP61 (up to 32 elements), and at SGS: ICP40Q. All multi-element geochemical methods are 4 acid near total digestions and read using an ICP-AES (Inductively Coupled Plasma - Atomic Emission Spectroscopy) instrument.

Over range assay methods at ALS are based on two digests, code ME-OG46 (and its Pb, Zn, and Ag equivalents) used an aqua regia digest and code MR-OG62 (and its Pb, Zn, and Ag equivalents) used a four-acid digest (at higher concentrations than the geochemical digest). Default finish is by ICP-AES, but atomic absorption (AA) can be used at the discretion of the laboratory.

At SGS over range assay methods included codes ICP43Q and AAS43Q, the distinction being the instrument used to read the analysis; ICP or AA.

Detection limits are given in Table 11.2 and a summary of the method used by VTT is shown in Table 11.3.

Table 11.2 Assay method detection limits

Method (laboratory code)	Lead		Zinc		Silver	
	Lower	Upper	Lower	Upper	Lower	Upper
ME-ICP61s	2 ppm	10,000 ppm	2 ppm	10,000 ppm	0.5 ppm	100 ppm
ME-ICP61	2 ppm	10,000 ppm	2 ppm	10,000 ppm	0.5 ppm	100 ppm
ICP40Q	5 ppm	5,000 ppm	5 ppm	10,000 ppm	2 ppm	100 ppm
[Pb, Zn, Ag]-OG46	0.01%	20%	0.01%	60%	1 ppm	1,500 ppm
[Pb, Zn, Ag]-OG62	0.001%	20%	0.001%	30%	1 ppm	1,500 ppm
ICP43Q	50 ppm	200,000 ppm	25 ppm	200,000 ppm	20 ppm	10,000 ppm
AAS43Q	0.02%	25%	0.02%	50%	-	-

Table 11.3 Sample analysis methods used between 2014 and 2017

Program	Laboratory	Geochemical level ICP	Over range as required			
			Over range Pb	Over range Zn	Over range Ag	Over range other elements
2014 – 2017 VTT	ALS Townsville	ME-ICP61	Pb-OG62	Zn-OG62	Ag-OG62	ME-OG62

11.4 Quality assurance and quality control procedures

This section details QA/QC procedures employed at Pegmont for drilling completed since the 2017 Technical Report. Graphs and details relating to earlier QA/QC programs are detailed in the relevant Technical Reports. However, summary statistics of VTT blank and CRM results are provided.

11.4.1 Field blanks

Blanks provide information on contamination from all phases of sampling and analysis and are inserted by VTT at the start of each mineralized interval. Blanks were inserted at a rate of 1 in 14 during the 2017 drilling program.

Table 11.4 summarizes the use of field blanks during the programs at the project from 2014 to 2017.

Table 11.4 Summary of total number of field blanks used per program

Program and operator	Total number of drillholes	Total number of samples (total excluding QA/QC samples)	Total number of field blanks
2014 VTT (including PML option)	16	524 (393)	30
2015 VTT	1	No assays – no significant results	
2016 VTT	30	912 (781)	51
2017 VTT	114	1,840	130

Field blanks used by VTT are sourced from either metallurgical grade quartz (2014) or local quartz float material (2016 and 2017). VTT favour the use of quartz as a blank as it also acts as a flush of the crushing and pulverizing steps of sample preparation. The metallurgical grade quartz used in 2014 is no longer available.

The quartz material used as blank in 2017 was prepared in the same way as the material used in the 2016 drilling program:

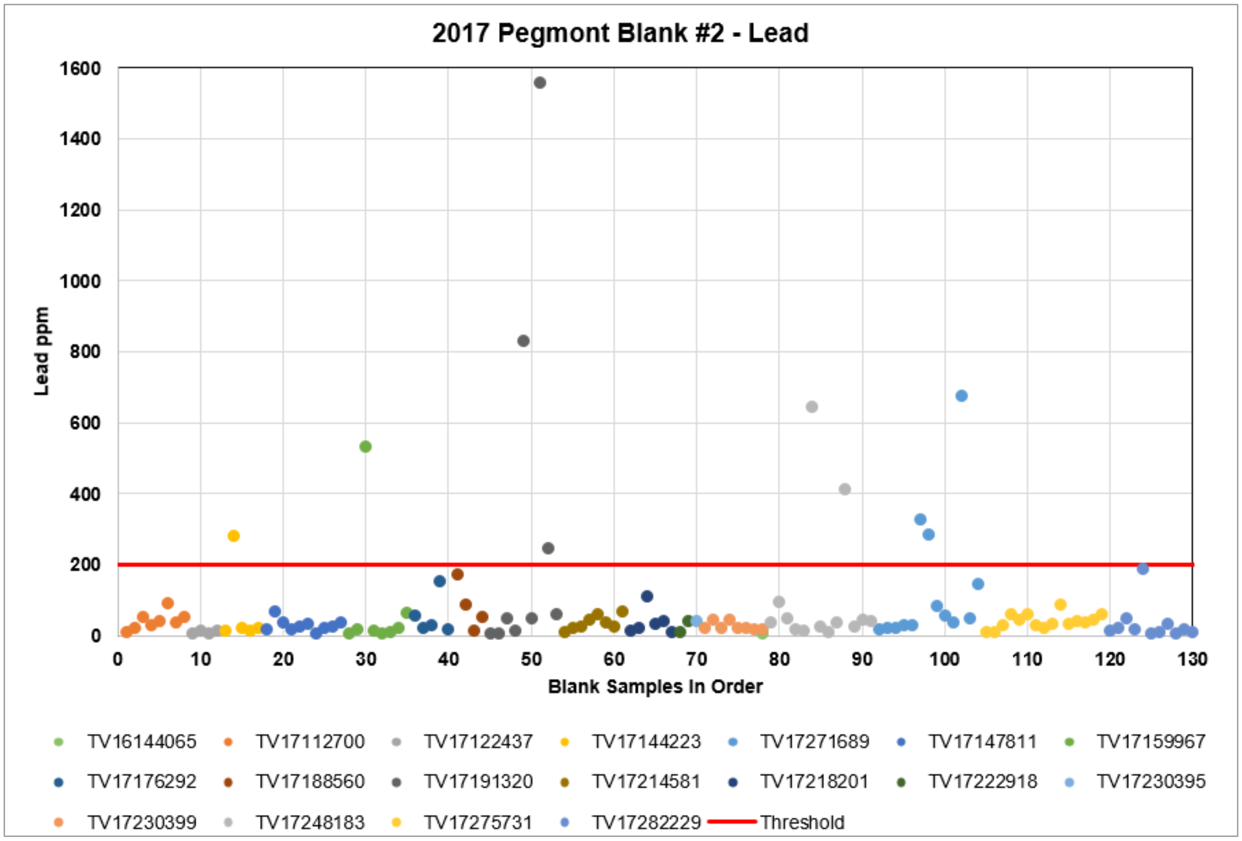
- A 44-gallon drum of quartz float material (without visible sulphides or their oxidized equivalents) is collected from the area surrounding the Project (approximately 300 kg).
- The contents of the drum are spread on a tarpaulin and samples are taken in a grid pattern.

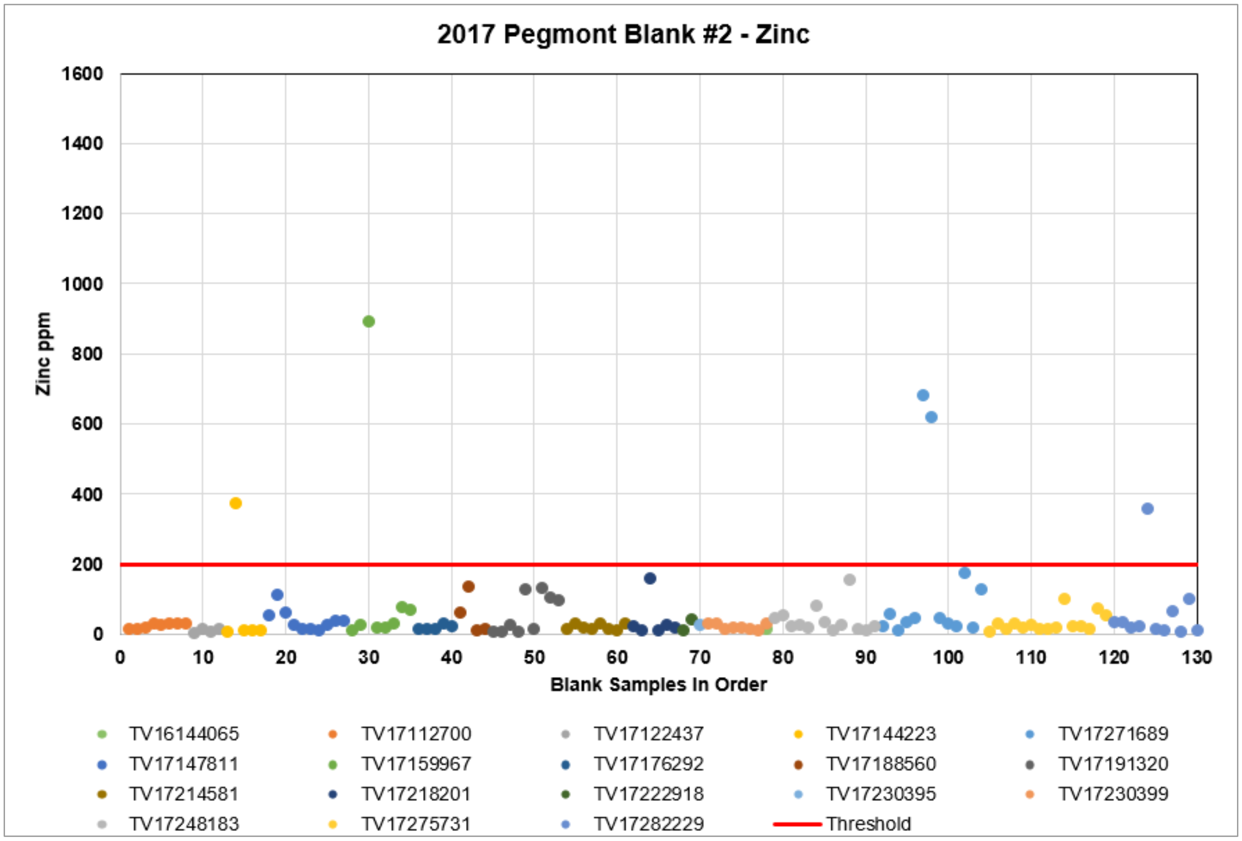
- The samples are sent to ALS Townsville, ALS Brisbane, and SGS Brisbane for analysis and the average of the returned assays is used as the reference value.
- The unsampled quartz float is returned to the drum and inserted (as lump material) into the sample stream for laboratory analysis.

Mean values for the quartz float used during the 2017 drilling program area 2.7 ppb lead, 3.1 ppm zinc, and 0.3 ppm silver.

Field monitoring charts for lead and zinc from the 2017 programs are presented in Figure 11.1. Elevated values of lead and zinc in the blanks may be attributed to isolated incidents of poor cleaning of the crushing and pulverizing sample preparation equipment, however, the sample selection and preparation method used in generation of the blank material is unlikely to result in a homogenous sample and repeatable results. Similar results were seen in the 2014 – 2016 drilling programs. The QP considers the blank material has generally acted as an effective flush of the laboratory system but is not a reliable check of a null value sample and recommends crushing, pulverizing and round robin style assaying of enough quartz to supply blanks for entire drill programs to achieve repeatability at low detection levels.

Figure 11.1 Field blank monitoring charts by laboratory batch for 2017





Silver was below detection in all analyses.

11.4.2 Standards

Four CRMs were used by VTT as a check on laboratory accuracy during 2017 and are listed in Table 11.5. As in the 2014 – 2016 drilling programs, CRMs are generally inserted a rate of 2 per drillhole. One low grade CRM is inserted at the start of the high-grade intersection as well as either a moderate or high-grade CRM at the end of the high-grade portion of the intersection, depending on the estimate grade of the mineralized zone. VTT keep a tally sheet of the position of the CRMs inserted in each drillhole.

Table 11.5 CRMs used by VTT in 2017 program

Provider	Standard	Number of determinations	Four-acid certified value					
			Lead		Zinc		Silver	
			Grade (%)	SD	Grade (%)	SD	Grade (g/t)	SD
Ore Research & Exploration Pty Ltd.	OREAS 131A*	3	1.72	0.049	2.83	0.08	30.9	1.29
	OREAS 131B	108	1.88	0.086	3.04	0.119	33.3	1.21
	OREAS 132B	57	3.86	0.066	5.25	0.195	60.7	2.19
	OREAS 133A	50	4.9	0.162	10.87	0.354	99.9	2.42

Note: *OREAS 131A is discontinued by manufacturer and replaced with OREAS 131B; SD denotes standard deviation.

The average insertion rate by VTT for the 2017 program was one CRM per 9 core samples. A CRM 'fail' is recorded where:

- A single assay returns a value outside 3 standard deviations for both the lead and zinc certified assay.
- Two consecutive assays return values outside 2 standard deviations for both the lead and zinc certified assay.

Single values outside two standard deviations were monitored (as 'warnings') and anomalous results were checked for sample mis-identification. Silver assays were monitored but as the lead and zinc CRMs in use do not generally have silver values matched to the relatively low silver grades seen at Pegmont, the results from the silver CRM assays are not considered a representative check on the quality of the silver assays. Generation of a matrix matched CRM that better reflects the proportions of lead, zinc, and silver seen at Pegmont has been recommended and is planned for future drilling programs, particularly given the reporting of silver in the Mineral Resource tabulation. In addition, VTT plan to implement a comparison trial of silver assaying by fire assay with a gravimetric finish to improve the reliability of the silver assaying.

Several results outside 2 or 3 standard deviations were recorded during the 2017 drilling program. Many of the anomalous CRM results were from silver CRM's, however the high-grade lead and zinc CRM results also included less accurate results. None of the lower accuracy CRM results met the criteria of having concurrent poor results in the other element and so no 'fails' were recorded. The QP recommends VTT reconsider their 'fail' criteria such that a fail is recorded if either the lead or zinc assays do not meet the criteria set out above, rather than if both the lead and zinc fail to meet the criteria. This would allow for earlier detection of problematic results.

Table 11.6 summarizes the results of the CRMs for the 2017 program.

Table 11.6 Assay results of Certified Reference Material

CRM		OREAS 131A	OREAS 131B	OREAS 132B	OREAS 133A
No of assays		3 (35)	104* (32)	58 (13)	50** (45)
Lead 2017 (2014 – 16)	Warnings	0 (1)	1 (0)	1 (1)	19 (8)
	Fails	0 (0)	0 (0)	0 (0)	0 (0)
	Average difference to CRM	-1.2% (0.0%)	-1.6% (1.1%)	-0.8% (1.8%)	-2.0% (0.6%)
Zinc 2017 (2014 – 16)	Warnings	0 (1)	1 (1)	1 (0)	21 (8)
	Fails	0 (0)	0 (0)	0 (0)	0 (0)
	Average difference to CRM	0.1% (-0.07%)	1.0% (3.3%)	0.6% (1.3%)	-1.9% (-0.4%)
Silver 2017 (2014 – 16)	Warnings	1 (0)	14 (10)	7 (1)	12 (12)
	Fails	0 (0)	0 (0)	0 (0)	10 + (0)
	Average difference to CRM	6.1% (1.1%)	3.1% (5.6%)	3.3% (5.4%)	-0.8% (1.2%)

Notes:
Results from 2014 – 16 programs in brackets
* 3 recorded removed from dataset and one transferred to an alternative CRM due to obvious error in labelling or laboratory handling.
** 1 record removed from dataset as obvious labelling error.
+ High grade silver CRM OREAS133A recorded here for completion, but not considered relevant to silver grades seen at Pegmont.

Figure 11.2 shows the results of the CRM analyses. Whilst few of the lead and zinc analyses meet the 'fail' criteria, there are trends in the data that indicate a degree of bias in the results, as shown in Table 11.6. Of particular note is the consistency of the lower than expected results for the lead CRMs generally, and the low results for the high-grade zinc CRM.

Figure 11.2 Control chart for lead, zinc, and silver, CRM OREAS 131B, 132B, and 131A, coloured by laboratory batch number

Note: Std.Dev. denotes standard deviation.

11.4.3 Duplicates

11.4.3.1 Quarter core duplicates

VTT submit selected mineralized intervals as 2 x quarter core samples rather than 1 x half core sample. This is done as a check on the short-range variability of the mineralization and, during the 2017 drilling program, occurred at the rate of 1 intersection per visually estimated interval >1% Pb+Zn over 2 m in each drillhole. Table 11.7 summarizes the analysis of duplicates by year.

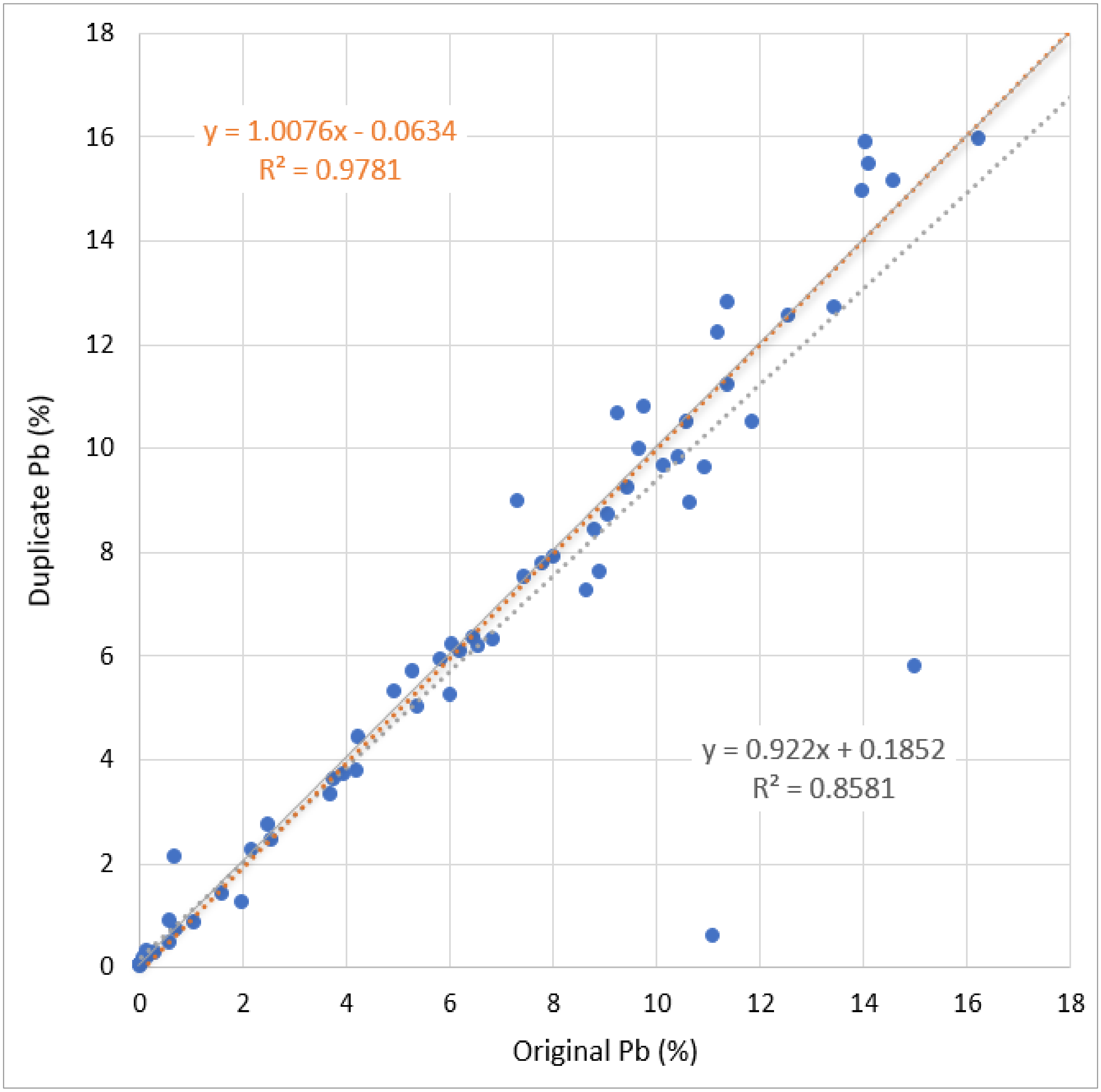
Table 11.7 Number of field duplicates by program

Program	Total number of drillholes	Total number of samples	Total number of field duplicates
2014 VTT	16	524	41
2015 VTT	1	No assays – no significant results	
2016 VTT	30	912	17
2017 VTT	115	1,840	65 (1 in 28)

There is generally reasonable correlation between duplicate lead and zinc pairs (Figure 11.3 and Figure 11.4). Some outliers are likely to be mislabelled samples. At higher levels, above 8% for lead and 10 ppm for silver (Figure 11.5), there is greater scatter, although there are not enough high-grade results for this observation to be conclusive.

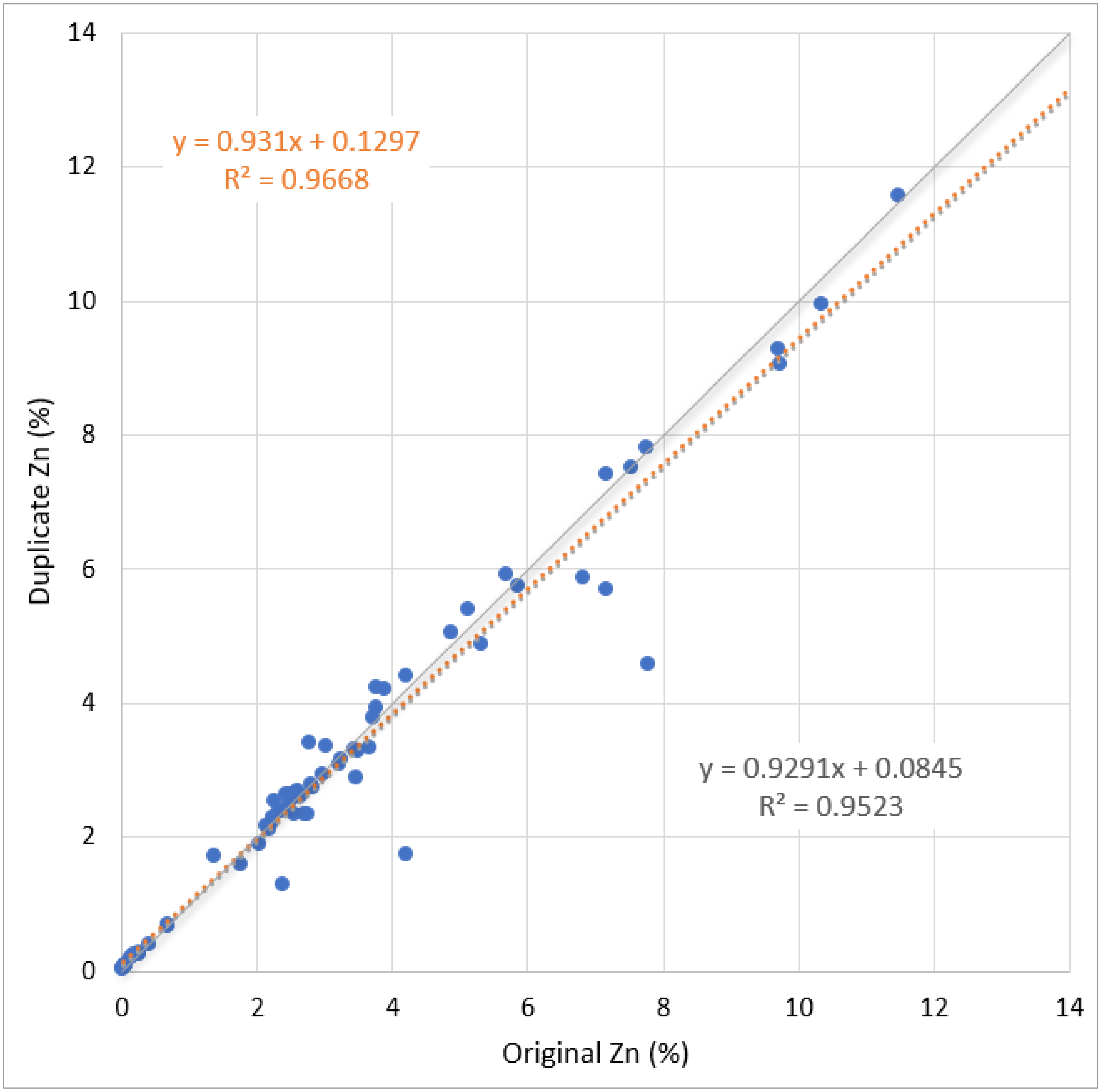
Lead is observed to have more scatter than zinc, and scatter increases with increasing lead grade. This is attributed to a higher proportion of millimetre wide veins of galena in the core, thought to be recrystallized from the layer parallel galena. This is thought to contribute to a variability on the scale of the quarter core. Sphalerite (zinc) mineralization is more homogeneous, being constrained to mineralized bands.

Figure 11.3 Lead duplicate pairs 2017



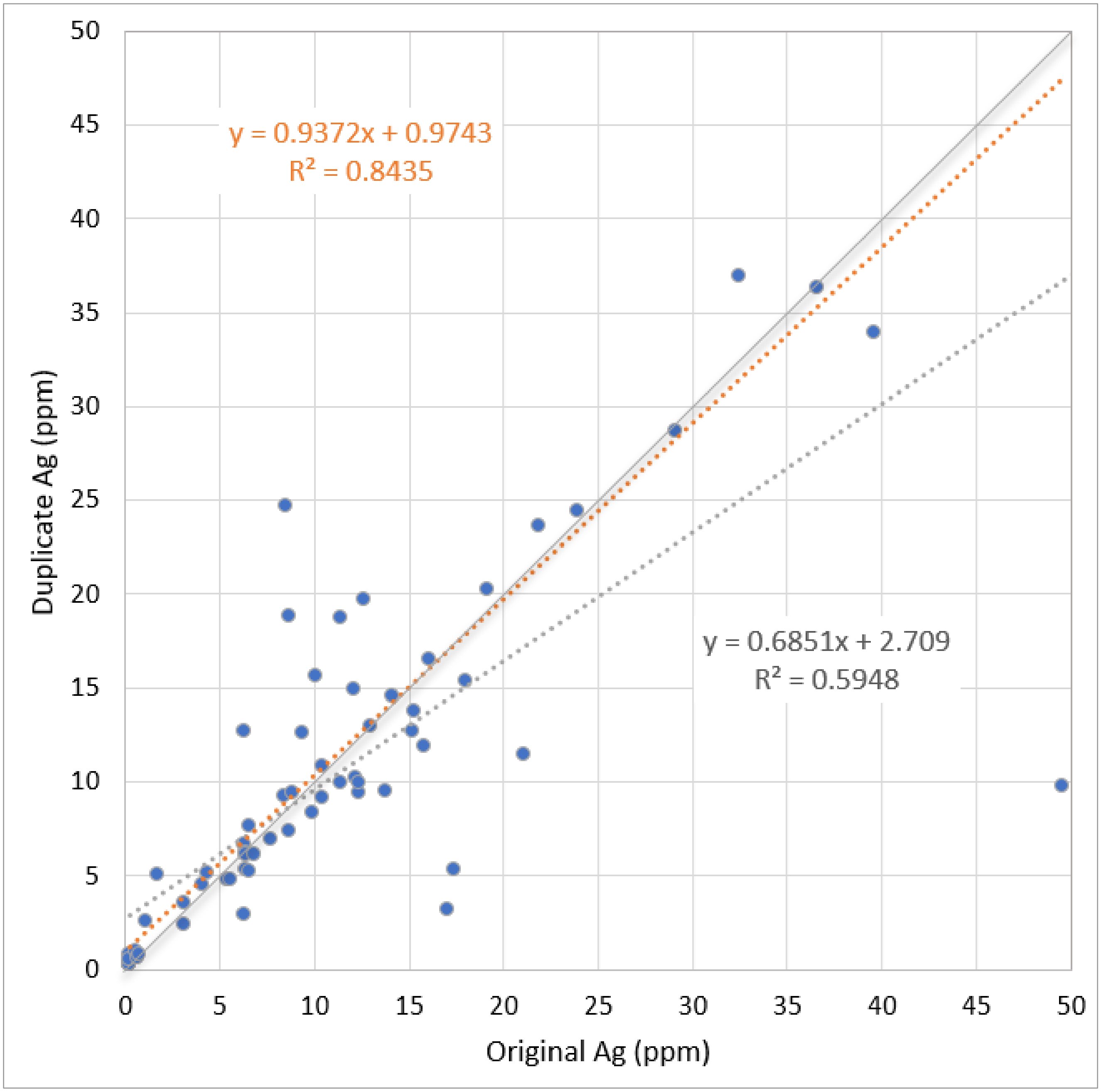
Note: grey dashed regression line includes outlier samples.
Source: AMC

Figure 11.4 Zinc duplicate pairs 2017



Note: grey dashed regression line includes outlier samples.
Source: AMC

Figure 11.5 Silver duplicate pairs 2017



Note: grey dashed regression line includes outlier samples.
Source: AMC

11.4.3.2 Pulp duplicates

VTT has not performed any pulp duplicate assaying.

11.4.4 External check assays

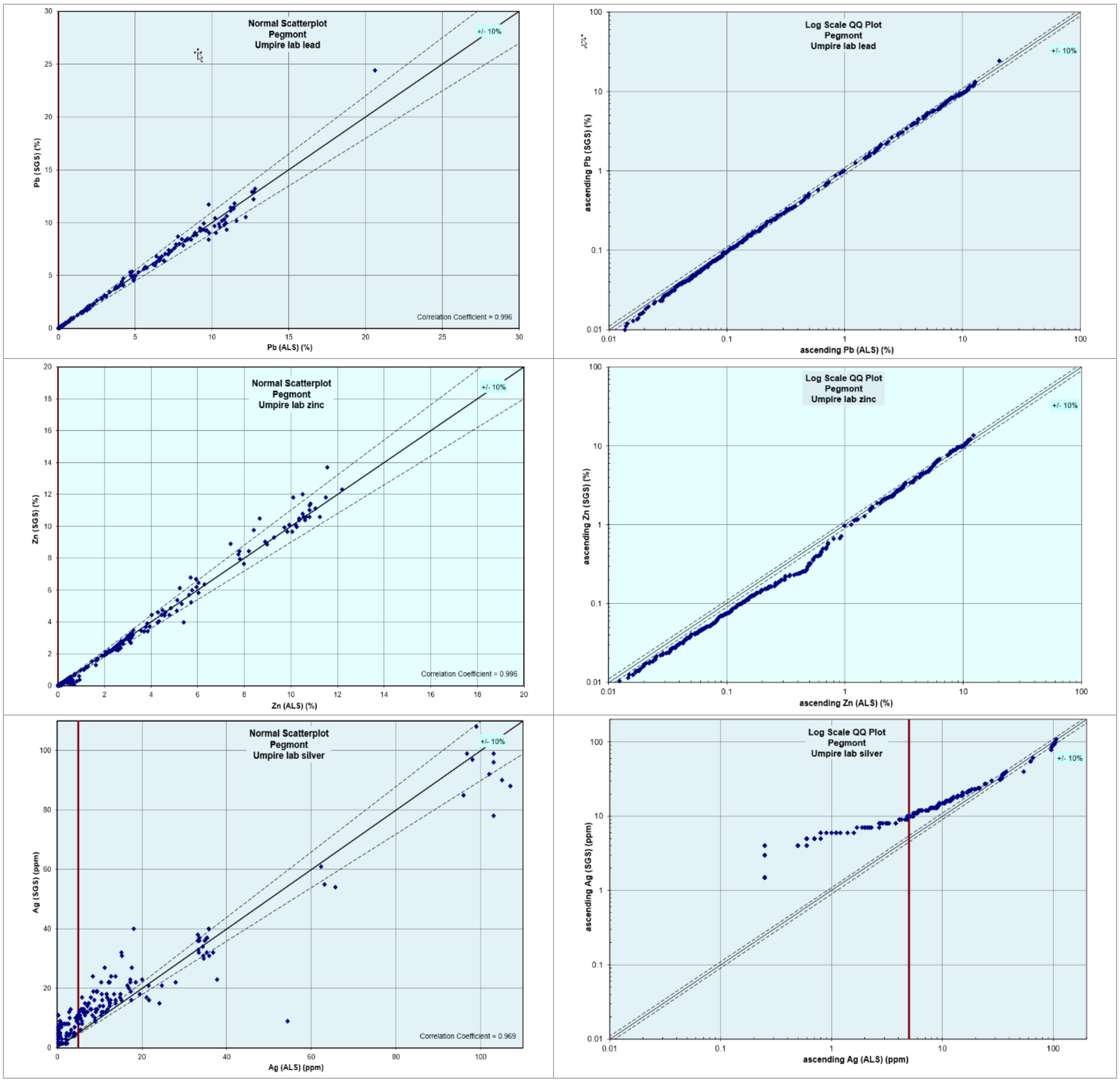
VTT submitted the samples from three batches to SGS Townsville as a check on the precision of the original assay results (Table 11.8).

Table 11.8 Number of external check assays

Program	Total number of drillholes	Total number of samples	Total number of field duplicates
2017 VTT	115	1,840	399 (1 in 5 from 3 batches)

There is generally good correlation between duplicate pairs for lead and zinc. Lead and zinc are observed to have a similar degree of scatter, which increases with increasing grade (Figure 11.6). There is an apparent bias at zinc grades <1% whereby ALS results are higher than the SGS results. The same effect is less pronounced in the lead results. The silver umpire laboratory results show a bias toward the SGS silver results which compounds the high silver CRM results seen from ALS.

Figure 11.6 Umpire laboratory check assays 2017



Note: red vertical line is at 2x detection limit.
Source: VTT

11.5 Database management

VTT, as part of its initial project acquisition due diligence, undertook to load all available drillhole data for Pegmont in MS Excel format into a password secured MS Assess database, with field validation and look up table libraries. This database remains in effect. VTT has continued to improve the integrity of the historic data.

Previously the database captured only lead, zinc, and silver; multi-element analyses were not recorded. Since the 2014 Mineral Resource, VTT has merged all available original assays (including multi-element data) into the database. Generally, original assay certificates are available for the 1996, and 2005 to 2017 programs. In some cases, the assays have been transcribed from historic company reports into the database.

As described in Section 12, VTT has undertaken a significant amount of data validation, all of which has been updated in the database. VTT has re-logged a total of 32 drillholes during 2017, either partially or entirely. VTT has also performed an additional 12 downhole surveys using north seeking gyroscopes on historic holes. Collar surveys of 96 historic drillholes were surveyed by licensed surveyors.

11.6 Sample security

Sample security practices were changed for the 2017 drilling program whereby samples remained under the care of VTT personnel until delivery to the ALS Mt Isa laboratory (previously a third party had transported the samples from Cloncurry to Mt Isa). In addition to the security benefits this also has the advantage of reduced the turnaround time for laboratory results.

The practice of double bagging and plastic wrapping of pallets was retained and reduces the chance of sample loss or interference during transport from site to the laboratory. The practice of cross-checking received sample numbers with the laboratory provides a rudimentary check to reduce the likelihood of samples being lost. Given the nature of mineralization, consistency of Zn and Pb results, and verification of database grades against original assay certificates, the QP considers it unlikely that the security procedures have introduced any material issues.

11.7 Data collection protocols and database verification update

Several recommendations were made in the 2013 Technical Report and 2017 Technical Report regarding data collection protocols and database validation. VTT has prioritized the verification of data relating to the historic drilling and is systematically working through the dataset. Table 11.9 summarizes the verification work completed by VTT in preparation for this Mineral Resource estimation.

Table 11.9 Summary of data verification completed by VTT to end of 2017 drilling program

Dataset	Verification completed
Collar surveys	418 collars surveyed by Lodewyks (71% of database)
	113 historic drillhole remain to be re-surveyed.
Downhole surveys	31 historic drillholes resurveyed to resolve apparent contradictions in the database and / or confirm the location of significant intersections.
Drillhole logs	Re-logging of 33 historic drillholes to expand on information in historic summary logging and refine the interpretation of the amphibolite dykes for modelling.
	203 historic drill logs scanned.
Photography	35 historic drillholes, including each of the re-logged drillholes were digitally photographed (RC chips and drillcore) and relabeled.
Assay certificates	The database has been updated with all available multi-element data from the 1975 Newmont, 1996 PML, 2000 North, and all post-2005 drilling campaigns. Lead and zinc values in the database have been checked against reports or original assay certificates where available.

11.8 Comment on sample preparation, analyses, security, and historic data verification

Prior to 2007, there is limited information on the sampling procedures used. Documented procedures used since 2007 meet industry standards. Whilst current sampling practices could not be observed by the QP whilst at site, documented procedures used by VTT also meet current industry standards.

Most assays are suitable to be used to generate a Mineral Resource estimate although some data prior to 2007 is of uncertain quality. The Mineral Resource classification takes into account the reduced uncertainty regarding the location and tenor of mineralization that is the result of the ongoing data verification program undertaken by VTT since the 2017 AMC Technical Report.

The analytical methods used to assess the Zn, Pb, and Ag grades are considered suitable for this style of deposit. QA/QC checks have been carried out since 2007 and analysis of QA/QC work undertaken indicates the lead-zinc dataset provides a reasonable representation of the location of mineralized BIF horizons. The continued erratic performance of the silver QA/QC program results in a lower degree of confidence in the silver assaying and this is recognized by VTT. Trials of alternative assay methods are to be undertaken in conjunction with the next drilling program and the sourcing or production of a more appropriate silver standard is a priority.

Work should continue on historic data verification including reanalysis of sample pulps and twinned holes to verify historic drilling program assays where QA/QC data is absent or indicates some issues of bias or uncertainty.

VTT has invested considerable effort to verify historic Pegmont data and has found no material issues in the data provided by the previous operator (PML). In the QP's opinion, the data available is adequate to support the 2018 Mineral Resource update.

12 Data verification

12.1 Introduction

The commentary in this section refers to observations based on a site visit by the QP in May 2017.

The QP site visit during 2017 verified the following aspects of the Pegmont project:

- Evidence of major historical drilling programs.
- Confirmation of a portion of drillhole collar locations across several domains in the deposit.
- Confirmation of those drillhole collar locations against the drillhole database used in the resource estimate.
- Confirmation that the (then) current geological interpretation was reasonable and not likely to over-state the Mineral Resource estimate.
- Evidence of the presence of mineralization in core.
- Evidence of original assay certificates which can confirm the current VTT database assay values in representative mineralized intercepts, and confirmation of a portion of those assay certificates against the drillcore.
- Evidence of original drilling logs, structural orientation data sheets, and bulk density measurements, and validation of portions of these against the VTT database and drill core intersections.
- Documentation of current sampling and analytical procedures, and bulk density measurement procedures.
- Evidence of original collar location surveys and downhole surveys, and validation of a portion of same against the VTT database.

12.2 Verification of logging

The original VTT logging sheets are stored on site in a well-organized filing cabinet. The logging data is entered into Excel on site then forwarded to VTT head office for validation on entry to the Access database. On examining the data, verification was made between the paper input and the database and no errors were seen. The QP cross referenced the paper logs with the database entries through the mineralized zones for five of the VTT drillholes and performed spot checks on several other drillholes during the course of the site visit. No transcription errors were found and the QP is satisfied that the level of detail in the logs is adequate to support the current geological interpretation.

12.3 Verification of original assays

Copies of assay certificates from the drilling carried out by VTT from 2014 – 2017 were available to the QP as electronic copies. No errors were found in cross checks of each of the assay certificates for the 2014 – 2017 VTT drilling against the assay database.

VTT maintains sampling tally sheets to monitor the insertion of standards and blanks and these were sighted, and a selection verified by the QP.

12.4 Verification of drilling

The QP's site visit confirmed the presence of capped drillhole collars over the spatial extent of the deposit. An example of the capping method employed by VTT is shown in Figure 12.1. The QP toured the area with a VTT representative and sighted many of the historic and VTT drillholes as well as permanent survey marks.

Figure 12.1 Example of completed drillhole collar (PVRD045)

Source: AMC

The drillholes were observed to be generally well capped and clearly labelled. The original email documentation containing collar locations surveyed by Lodewyks was sighted and checked against the database with no discrepancies found. In addition, the QP sighted two permanent survey marks located on the Property.

The QP sighted examples of downhole surveyed files generated at the drill rig and received digitally by VTT on site. No discrepancies were found in the selected cross checks of these files against the downhole survey database.

12.5 Verification of mineralization

Two outcrops were visited on the Property which exhibited honeycomb-textured gossans in BIF. These were BIF outcrops at Mount Lucas (Pegmont) and the BHZ outcrop. In the QP's opinion, the geology observed and the observations from core confirm the presence of amphibolite grade metasediments including BIF-hosted sulphide mineralization within key drillholes in the material resource domains at the Pegmont deposit.

Figure 12.2 The BHZ outcrop and gossanous textures

Source: AMC

12.6 Geological interpretation

The site visit confirmed the presence of outcropping structurally complex, folded BIF. At site, it was noted there are no paper geological sections or geological interpretation of the drilling. The geological interpretation undertaken for use in the Mineral Resource estimate is relatively conservative, as it assumes minimal folding between sections.

The QP understands VTT has invested a significant amount of time refining the geological interpretation and that the current drilling program targets interpreted high grade zones associated with fold hinges. The QP sighted outcrops of metasediment that indicate multiple phases of deformation and the Pegmont gossan outcrop is folded on the scale of tens of metres. The QP also observed folding of metasediments in drill core (tens of centimetres scale).

The Mineral Resource estimate classification takes into account drillhole spacing and structural complexity by placing a limit on the distance from drillholes for each reported domain. Whilst the interpretation can be improved, given the conservative approach to geological interpretation and domaining, the risk of underestimating is considered to be low.

A geological interpretation of the host BIF and garnet rich metasediment horizon was built by VTT using the geological logging, structure logging (only available on VTT holes) and a background nominal 0.2% lead plus zinc assay as a proxy for the mineralized horizon in the absence of detailed logging in RC holes. An interpretation of the flat laying amphibolite dyke was also built for the first time which was the result of relogging many holes as the amphibolite had in many instances been mis-logged as meta-sediments. Previous interpretations had placed inferred steeply dipping faults to explain apparent off sets in the vertical position of the host horizon. The current interpretation has removed these faults in favour of a ductile folded model. VTT drilling across these Inferred structures found no presence of faulting and structure logging of orientated core has confirmed the presence of folding.

Figure 12.3 Mt Lucas outcrop and gossanous textures

Source: AMC

12.7 Verification of critical information

Paperwork on site is well organized and inventories and up to date procedures are in place. In the QP's opinion, the risk of misplaced information or errors being introduced is low, with the exception of typographical errors on original data entry.

A significant portion of the original data is present and has been transferred to the VTT database. Original drill logs and assay certificates were sighted from several drilling programs. The geological logging detail was reasonable in the selected records sighted by the QP, and indicated the presence of metasediments, pegmatite, and BIF intersections downhole, and the presence of Pb and Zn intersections, confirming the nature and style of the Broken-Hill style mineralization.

Whilst all original records have not been cross-checked in detail against the database results, the checks conducted indicated good correlation, and their presence gives the QP confidence in verifying that the VTT drilling programs were completed as reported and were recorded appropriately.

Underlying limitations in the historic data have been systematically addressed by VTT and in the opinion of the QP, the data is adequate as a basis for estimating the 2018 updated Mineral Resources.

13 Mineral processing and metallurgical test work

13.1 Introduction

The Pegmont deposit can be broadly described as a multiple lens, Broken Hill type lead, zinc, and silver deposit and has been categorized by oxidation state. Both open pit and underground mining will be undertaken to extract material for processing.

This section will cover the current and previous test work programs referred to in the 2017 AMC Technical Report.

Two metallurgical test work programs have been conducted on samples from the Project, between 2016 to 2018. An initial metallurgical test work program was completed by ALS Metallurgy Burnie Laboratory, Tasmania in May 2017 (ALS Report T1072) using four metallurgical composite samples.

A follow up program in April 2018 (ALS Report T1092) expanded on the initial test work program. Five additional composite samples which better reflect the high tonnage mineralized zones were examined based on previous test work program. Results from the initial batch flotation tests on the samples formed the basis of the proposed lock cycle flotation test conditions and stages.

The locked cycle test results from the 2018 test program (ALS Report T1092) have been used to develop the process design criteria for the PEA. Results of previous test work programs were however, also considered in the interpretation and process design.

13.1.1 Historical test work

This section summarizes the main results of the historical test work, previously published in, Section 13 of the 2017 AMC Technical Report. The report covers the historical test work for conventional flotation and hydrometallurgical processes and the then current metallurgical test work program conducted at ALS Burnie Laboratory, Tasmania from 2016 to 2017 (ALS Report T1027), refer to Section 13.2.

The historical metallurgical test work was carried out in two phases:

- Phase 1 (2006 – 2008), initially focussed on flotation processing routes and subsequently included preliminary work on a hydrometallurgical route.
- Phase 2 (2011 – 2012), largely pursued the hydrometallurgical treatment of oxide and sulphide composites through pilot plant testing.

The conclusions from this previous work were:

- Although conventional differential flotation to generate saleable lead and zinc concentrate grades was possible, recoveries were not economically viable, and the oxide material was not amenable to sulphide flotation.
- A bulk lead / zinc concentrate however was produced at acceptable recoveries from sulphide and mixed feed and was considered to be suitable feedstock for downstream processing by hydrometallurgical routes.
- The EcoZinc® and EcoLead® hydrometallurgical processes, essentially chloridization followed by acid leaching, were tested at the lab scale and found to be very effective in handling a wide range of feedstocks and a degree of lead:zinc selectivity was achieved at the chloridization stage.
- With acid leaching of chloridization fume and calcine, and hot caustic leaching of the acid leach residue, overall zinc and lead extractions of >90% and >80% respectively were achieved.

- The chloridization stage in a fluid bed reactor was piloted; laboratory extractions were largely confirmed, however significant difficulties were encountered with sticky zinc chloride fume due to its hygroscopic properties, resulting in operational problems with pipe blockages, etc.
- No test work was carried out on leach solution purification and final metal recovery. Reliance was placed on standard metallurgical practices of sequential pH adjustment and metal precipitation followed by final zinc recovery by electrowinning or also by precipitation in order to develop a conceptual overall flowsheet.
- Although the chloridization and leaching chemistry appeared to have been proven at the laboratory stage, only the fluid bed reactor was piloted and there was no integrated flowsheet pilot plant to test the entire feed-to-final product process route.

13.2 Test work program to May 2016 – 2017 (T1027)

The scope of the recent metallurgical test work is substantially different from the approach taken in the early historical work noted above.

13.2.1 Previous test work program (T1027) samples

Polymetallic core intersections from the newly discovered BHZ and primary sulphide extensions of Zone 5 of the Pegmont lead and zinc resource were received by ALS Burnie laboratories for composition into four metallurgical testing composites. The four metallurgical composites are listed as:

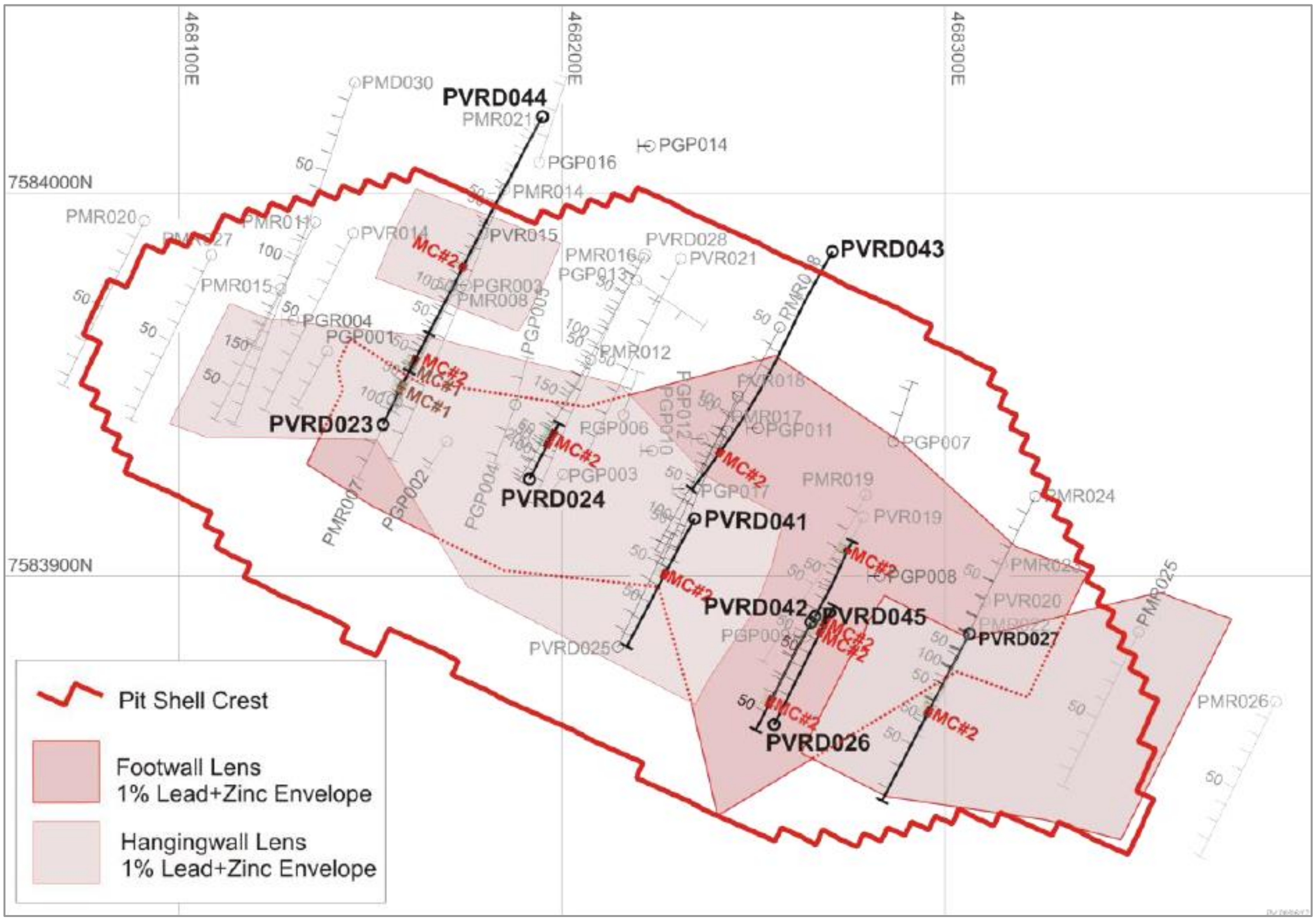
- BHZ Met Comp 1 – Shallow transition material from BHZ
- BHZ Met Comp 2 – Primary sulphide mineralization from BHZ
- Z5 Met Comp3 – Primary sulphide mineralization Lens B Zone 5
- Z5 Met Comp4 – Primary sulphide mineralization Lens C Zone 5

All samples used quarter core BQ, except for drillhole BHZ#1 which was half core. The intersections included downhole dilution of waste on either side of the mineralized zone to represent probable mining dilution. The transition zone material has been categorized to be sulphide mineralization hosted in fresh to partially weathered rock.

The location of the drillholes from which the core was taken for this metallurgical test work program is well documented and reported in “Pegmont Mineral Resource Update June 2017”. The drillholes used to form the metallurgical composites were considered to adequately represent the primary sulphide zones, however the BHZ metallurgical composite 1 was comprised of intersections from only one drillhole and therefore, additional sampling and testing will be required on the transitional material.

The drillholes are shown in Figure 13.1.

Figure 13.1 BHZ metallurgical sample locations



Source: VTT

The head grades of the prepared composites have been summarized in Table 13.1.

Table 13.1 Summary of test composites

Composite	%Cu	%Pb	%Zn	%Fe	%S	F ppm	g/t Ag
BHZ Met Comp 1	0.03	3.05	2.81	33.4	2.29	1380	5.1
BHZ Met Comp 2	0.02	4.71	2.94	30.0	2.96	2080	8.6
Z5 Met Comp 3	0.02	5.20	4.54	28.2	3.82	1180	6.3
Z5 Met Comp 4	0.02	4.00	5.22	27.3	4.06	1340	5.6

The BHZ and Zone 5 (Z5) represent a smaller proportion of the Pegmont resource on a weight basis compared to other mineralized zones. Additional composites representing the higher tonnage mineralized zones were subsequently prepared for the follow up program in April 2018 (ALS Report T1092) which included the locked cycle flotation test work.

13.2.2 Comminution tests

The initial comminution test work focused on the Bond Ball Mill Work Index (BW_i). The quarter core from BQ holes had dimensions too small to carry out a full suite of comminution tests. The BW_i tests were conducted using a 106 µm closing screen generating a finer product size compared to the target P_{80} of 106 µm for the primary grind size. It should be noted that the grind calibration test conducted to produce samples for subsequent flotation test work was done to achieve the actual target P_{80} of 106 µm. The BW_i indices indicate the material is moderately hard to hard and will require a high energy input for grinding. The primary sulphide samples exhibited a higher BW_i indices as expected.

Table 13.2 Bond ball mill work index

Sample	Value
BHZ Met Composite 2	16.6 kWh/t
Z5 Met Composite 3	19.4 kWh/t

13.2.3 Mineralogy

A quantitative optical microscopy assessment on the four metallurgical composites was undertaken by McArthur Ore Deposit Assessments Pty Ltd (MODA) on behalf of ALS in December 2016.

The main objective was to examine the extent of liberation of the main sulphide minerals at the target grind P80 size of 106 µm.

The samples were prepared into size fractions, mainly +150 µm, +106 µm, +75 µm, +38 µm and +20 µm.

The mineralization of the four samples have been summarized in the ALS McArthur Ore Deposit Assessment (MODA) report December 2016 which forms part of the ALS report T1027 May 2017. The report covers the following generalization for the transition and the sulphide composites.

- Burke Hinge Zone Met Comp 1, the sample was noted as being rich in goethite, as part of the transition ore zone. Other associated minerals include pyrite, pyrrhotite, sphalerite, galena, cerussite, magnetite, and gangue (quartz, silicates).
- Sulphide samples (BHZ Met Comp 2, Z5 Met Comp 3 and Z5 Met Comp 4) were goethite poor – primary sulphide ore. Other associated minerals include pyrite, pyrrhotite, sphalerite, galena, chalcopyrite, cerussite, magnetite-goethite, and gangue (quartz, silicates).

The identified minerals are summarized in the Table 13.3.

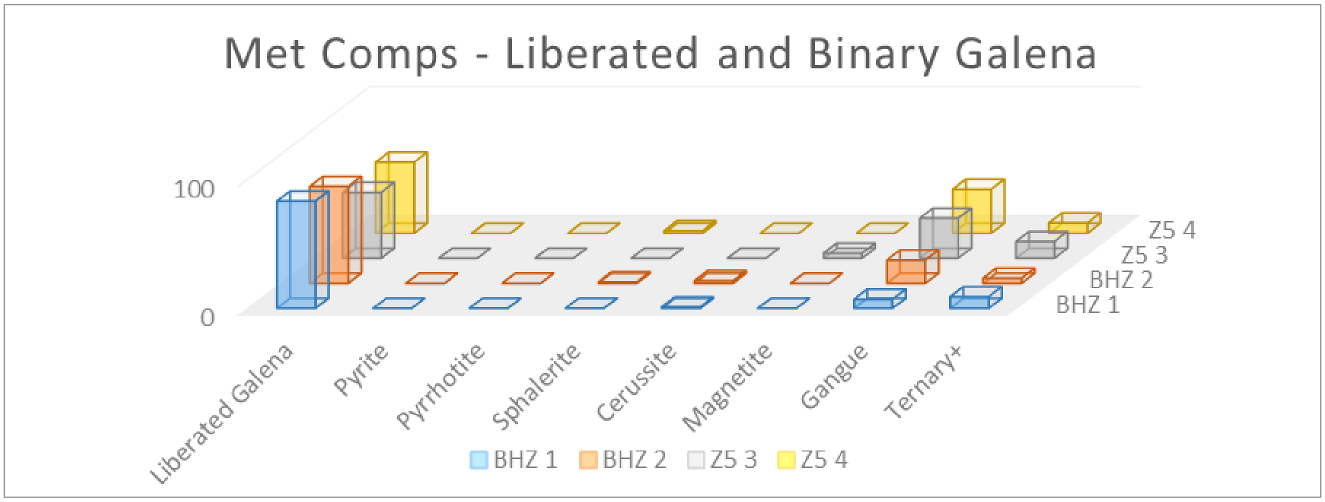
Table 13.3 Mineral observations

Mineral	Observations
Pyrite	Mainly as porous melinikovite, with very minor crystallized pyrite.
Pyrrhotite	Occurs as ultrafine inclusions in sphalerite and magnetite
Sphalerite	Mainly as coarse high iron grains, occasionally dusted by ultrafine “chalcopyrite-disease”. Associations are mostly with non-sulphide gangue but was also observed with magnetite.
Galena	Mainly as coarse grains, only rarely oxidized to cerussite. Significant amounts as witnessed as ragged, fine grained inclusions in gangue (mostly chlorite) and often accompanying magnetite. Some rare flakes of graphite are noted.
Magnetite	Mainly well liberated but also occurs as ragged grains with galena in gangue. Occasionally hosts fine grained blebs of sphalerite and galena.
Goethite	Only occurring in the transition ore sample Z1 Comp 8. Has widely varying textures. It occasionally hosts fine grained sphalerite and galena.
Gangue	Is comprised of quartz, garnet, chlorite, graphite, and other minor silicates.

13.2.3.1 Galena liberation

Galena liberation was categorized as being mainly liberated in the BHZ, with BHZ 1 and 2 reporting liberated quantitative analysis values of 83% to 75% respectively. The liberation of the Zone 5 metallurgical composites 3 and 4 reported a lower level of galena liberation at 51% and 55% respectively. The liberation characteristics are displayed in Figure 13.2.

Figure 13.2 Met composites – liberated and binary galena



Source: GRES

On both the Z5 composites (composites 3 and 4) the non-liberated galena is mainly binary and associated with non-sulphide gangue in the range of 30% to 35%. Minor inclusions of galena associated with ternary products were also identified in the range of 8% to 13%.

Both the BHZ composites 1 and 2 display a high degree of mineral liberation below 38 µm, moderate to high at 75 µm, with binary and ternary mineral locking above 106 µm.

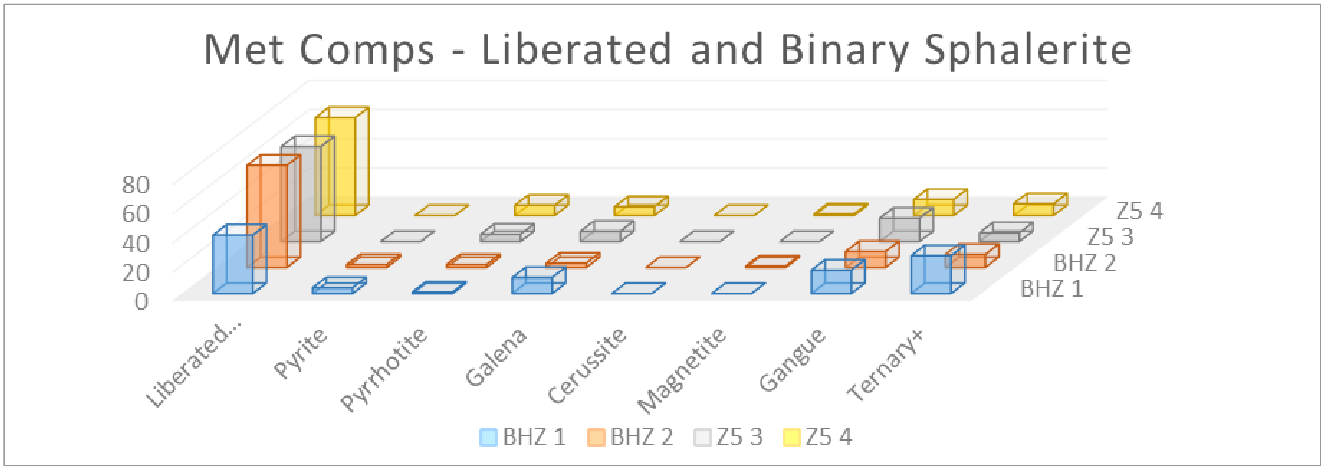
The Zone 5 composites show a high degree of competency with lower percentages of liberated galena in the +20 µm to 38 µm size range. This is also displayed in the binary gangue and ternary figures with higher residual figures indicating poorer liberation compared to the BHZ samples.

For both the BHZ and Zone 5 composites a regrind of the flotation concentrate at the target size of 20 – 38 µm is warranted to liberate galena from the binary and ternary products.

13.2.3.2 Sphalerite liberation

Sphalerite liberation was poor in the transitional BHZ composite 1, but moderate in all remaining three sulphide composites, with liberation in the range of 65 to 70%. Binary and ternary associations represent the bulk of the non-liberated sphalerite with minor binary association of sphalerite with galena, pyrite, and pyrrhotite. The liberation characteristics are displayed in Figure 13.3.

Figure 13.3 Met composites – liberated and binary sphalerite



Source: GRES

Apart from the BHZ Comp 1, sphalerite liberation in the finer +38 µm size fraction was > 60% for the sulphide composites. A greater degree of mineral locking identified above the 75 µm size fraction with binary gangue and ternary associations with sphalerite.

The high degree of binary sphalerite / gangue particles suggests the zinc rougher flotation concentrate would require a regrind stage to achieve the mineral liberation necessary for effective mineral selectivity. A target regrind size less than 38 µm would achieve the degree of mineral liberation required.

13.2.4 Flotation tests

Flotation tests were conducted at ALS Burnie Laboratories, Tasmania as part of the T1027 test work program. The composites described in Section 13.2.1 above were used.

The test work program focused on sequential lead and zinc roughing and cleaning flotation tests to develop the process flowsheet that would produce separate saleable lead and zinc concentrate and the reagent regime for the lock cycle tests that were conducted during the T1092 test work program.

The initial reagent regime for the sequential flotation program is noted below.

13.2.4.1 Primary grinding and lead flotation

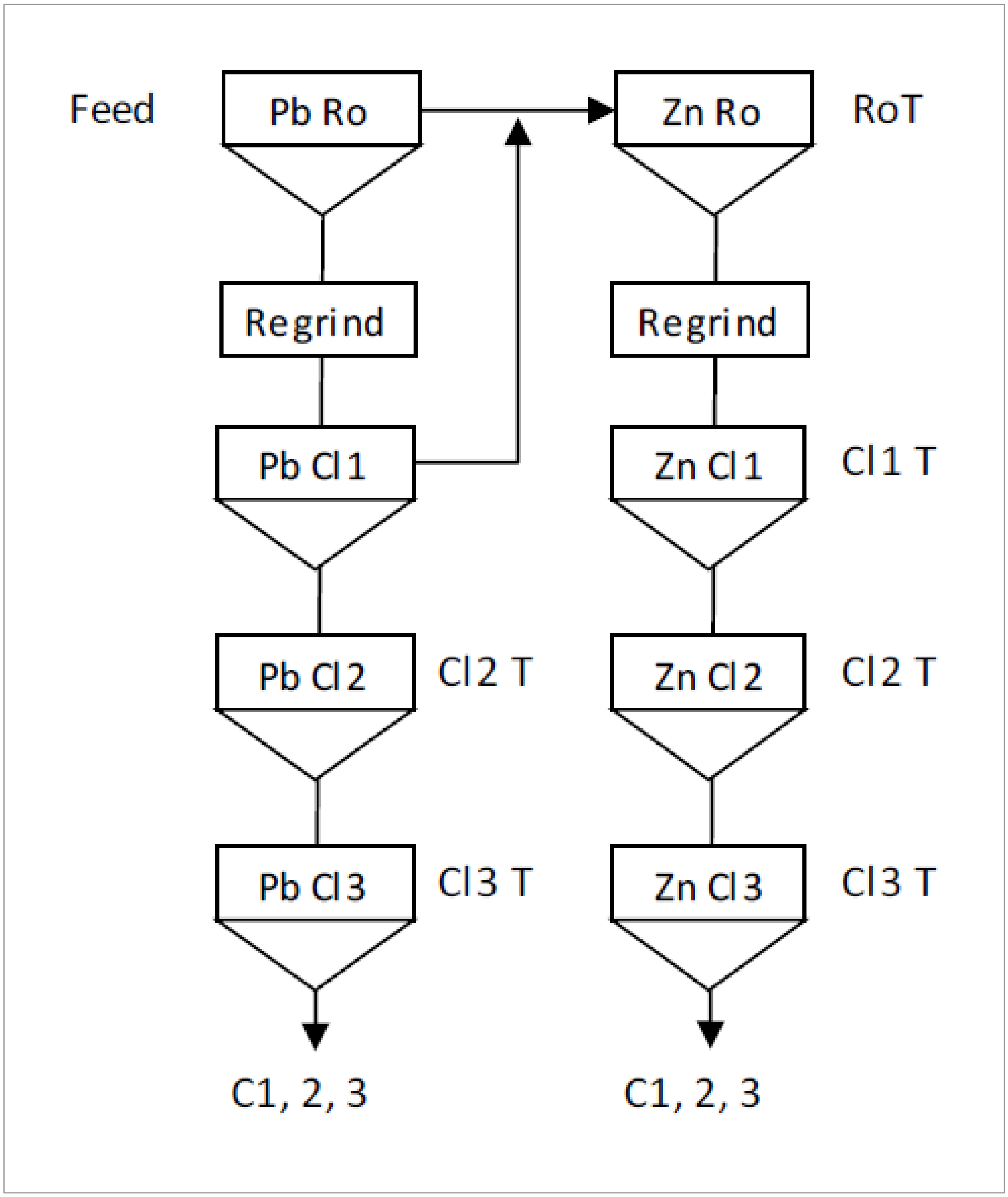
- Lime – pH modifier.
- Sodium Cyanide (NaCN) – Activator for galena and pyrite depressant.
- Sodium Metabisulphite (SMBS) – Depressant for pyrite and sphalerite.
- Cytec 3418A – Specialty lead sulphide collector.
- Methyl Isobutyl Carbinol (MIBC) – Frother.

13.2.4.2 Zinc flotation

- Lime – pH Modifier.
- Copper Sulphate ($CuSO_4$) – Activator for sphalerite.
- Sodium Isobutyl Xanthate (SIBX) – Collector for sulphide minerals.
- Methyl Isobutyl Carbinol (MIBC) – Frother.

The flotation flowsheet applied is detailed in Figure 13.4 and includes a regrind stage for the lead and zinc rougher concentrates prior to the cleaner flotation stage.

Figure 13.4 Flotation flowsheet



Source: GRES

Key test results are summarized in Table 13.4 for lead rougher and cleaner flotation stages and Table 13.5 for zinc rougher and cleaner flotation stages.

Table 13.4 Lead Cleaner 3

			Pb		Zn		Fe		S	
Comp	**Product**	**Wt%**	**%**	**Rec%**	**%**	**Rec%**	**%**	**Rec%**	**%**	**Rec%**
T13 – BHZ Met Comp 1	Pb Cl 3 (c1,2,3)	4.3%	57.0	80.6	14.0	22.3	5.3	0.7	16.3	32.3
T09 – BHZ Met Comp 1	Pb Cl 3 (c1,2,3)	6.8%	56.5	87.2	12.1	25.9	6.0	1.3	15.0	39.4
T11 – BHZ Met Comp 2	Pb Cl 3 (c1,2,3)	6.3%	70.6	91.5	3.7	8.2	4.2	0.9	12.8	29.2
T07 – BHZ Met Comp 2	Pb Cl 3 (c1,2,3)	6.0%	72.0	89.7	3.1	6.1	3.3	0.7	12.8	26.4
T12 – Z5 Met Comp 3	Pb Cl 3 (c1,2,3)	7.9%	61.2	90.5	4.0	6.9	6.8	2.0	11.6	24.0
T08 – Z5 Met Comp 3	Pb Cl 3 (c1,2,3)	7.5%	68.0	88.5	3.1	4.9	5.2	1.4	11.3	23.3
T14 – Z5 Met Comp 4	Pb Cl 3 (c1,2)	5.0%	66.1	83.1	3.0	2.9	5.1	1.0	12.2	15.5
T10 – Z5 Met Comp 4	Pb Cl 3 (c1,2)	5.9%	57.9	86.6	4.0	4.5	6.8	1.5	10.6	16.1

13.2.4.3 BHZ composites – lead flotation

Flotation test results for the BHZ composite 1 (Transition) were impacted by high pulp viscosity which were amplified by the addition of lime. As a consequence, the BHZ Met Comp 1 flotation was performed at a lower pulp density. The rougher flotation tests consumed more lime, SMBS and collector when compared to the other sulphide composites samples. Given these difficulties it was still possible to produce a final concentrate at 56.5% Pb with a high recovery of 87.5%. Zinc deportment to the lead concentrate was also high for the BHZ composite in the range of 22 to 26% for tests T09 and T13 respectively.

Flotation tests on the BHZ Met Comp 2 (Sulphide) using SMBS and 3418A achieve excellent results in test T07 and T11 with concentrate grades above 70% Pb and recoveries above 89%. Deportment of zinc to the lead concentrate was 6 to 8% of the distribution, and below that reporting the lead concentrate for the transition composite sample.

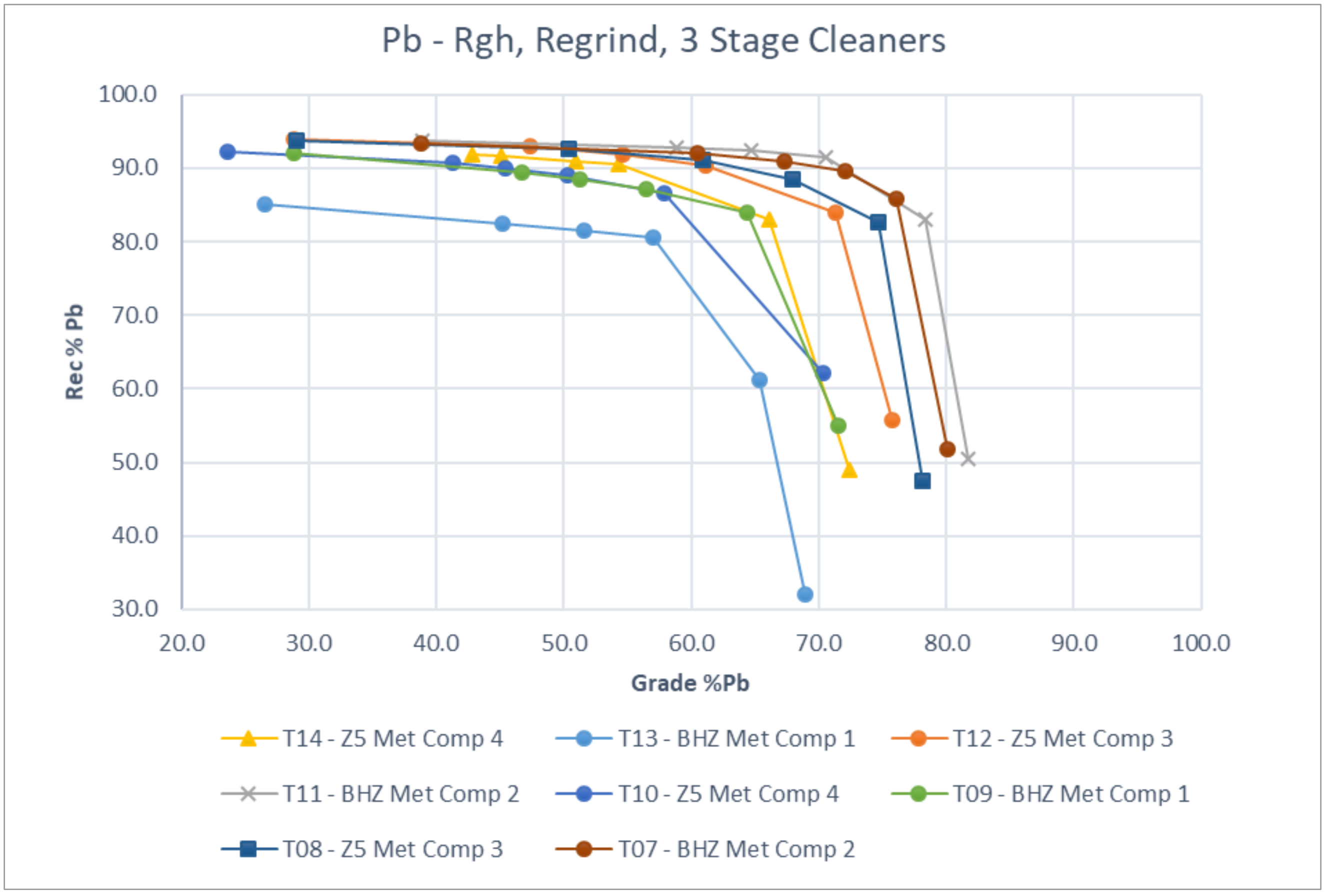
13.2.4.4 Z5 composites – lead flotation

The Z5 Met Comp 3 and 4 (Sulphide) samples performed well in the batch flotation tests using the same reagent regime as in the BHZ flotation tests. The best results were achieved in T08 (P_{80} = 27 µm) and T12 (P80 = 20 µm) and included a regrind stage using a specific energy of 10 kWh/t. Test T08 produced the best result with a 68% Pb concentrate at 88.5% Pb recovery.

Results for the Z5 Met Comp 4 sample were slightly lower while using the same reagent regime and regrind conditions with the best test being T14 with a 66.1% Pb concentrate at 83.1% Pb recovery.

Test results are shown in the grade vs recovery graph for lead where a regrind stage has been utilized prior to cleaner flotation.

Figure 13.5 Grade vs recovery for lead (T1027)



Source: GRES

Table 13.5 Zinc rougher and Cleaner 3 (stages)

			Pb		Zn		Fe		S	
Comp	**Product**	**Wt%**	**%**	**Rec%**	**%**	**Rec%**	**%**	**Rec%**	**%**	**Rec%**
T13 – BHZ Met Comp 1	Zn Cl 3 (c1)	1.1%	3.4	1.2	48.9	19.3	11.4	0.4	30.4	15.0
T09 – BHZ Met Comp 1	Zn Cl 3 (c1,2)	1.8%	2.9	1.2	50.7	28.7	10.5	0.6	30.0	20.8
T11 – BHZ Met Comp 2	Zn Cl 3 (c1,2)	2.6%	1.2	0.6	53.6	49.0	10.1	0.9	31.4	29.5
T07 – BHZ Met Comp 2	Zn Cl 3 (c1)	2.0%	3.1	1.3	50.4	33.0	11.1	0.8	32.2	22.3
T12 – Z5 Met Comp 3	Zn Cl 3 (c1,2,3)	7.5%	1.9	2.6	49.2	79.8	11.2	3.1	29.5	57.8
T08 – Z5 Met Comp 3	Zn Cl 3 (c1,2,3)	7.1%	1.7	2.1	50.1	75.6	11.0	2.9	27.0	52.7
T14 – Z5 Met Comp 4	Zn Cl 3 (c1,2)	6.5%	1.8	3.0	50.8	62.6	11.1	2.7	29.9	49.1
T10 – Z5 Met Comp 4	Zn Cl 3 (c1,2)	8.0%	1.5	3.0	50.3	76.7	11.2	3.4	29.8	61.7

13.2.4.5 BHZ composites – zinc flotation

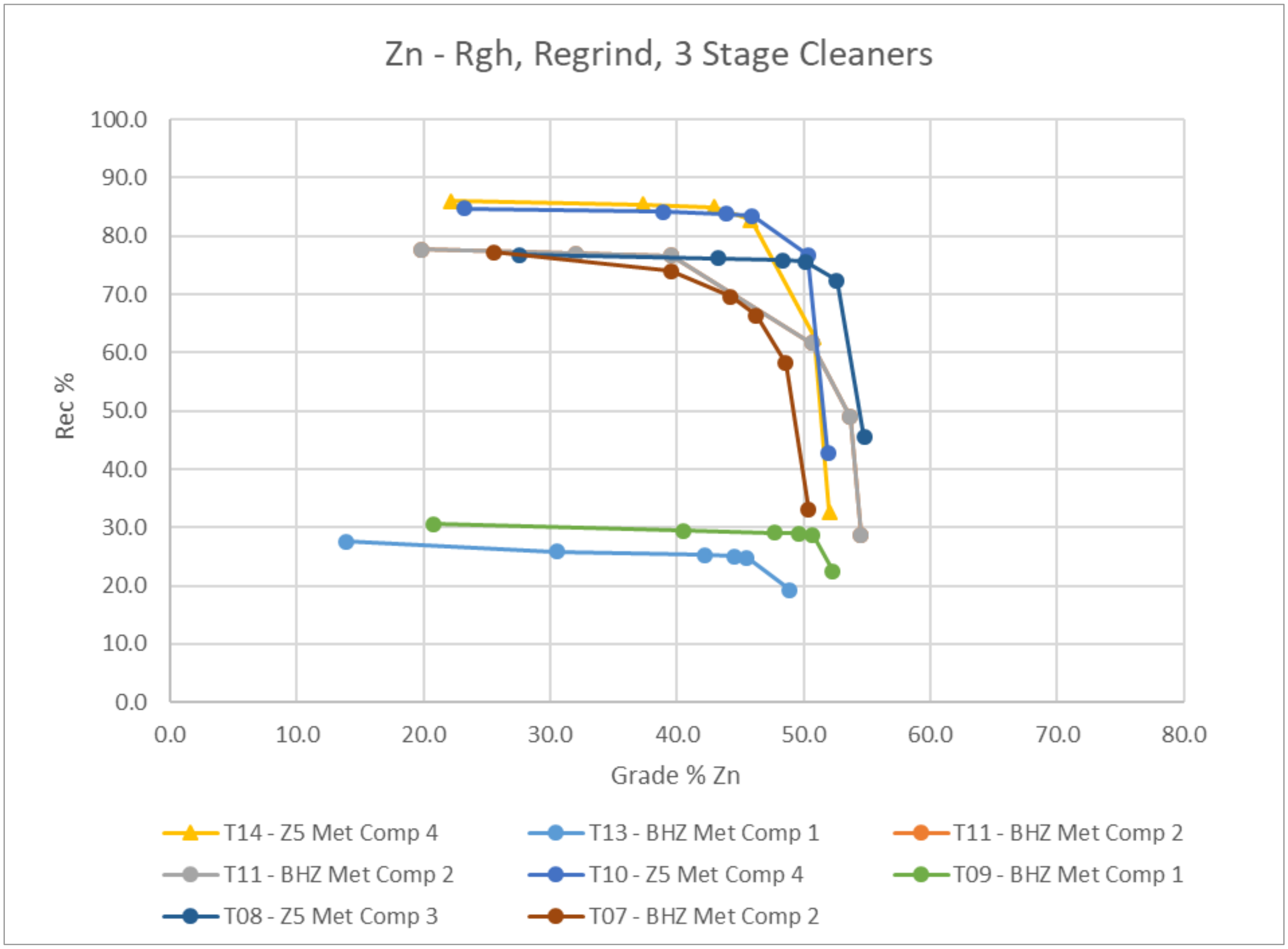
BHZ Comp 1 (Transition) produced poor zinc flotation results with the best result of 42% recovery to the rougher stage at a concentrate grade of 6.6% Zn. Mineralogical assessment indicated the sphalerite liberation was poor in the coarser grind size fractions above 38 µm, with sphalerite mainly locked in binary particles with non-sulphide gangue and also in ternary particles. Regrinding the zinc rougher concentrate improved the cleaner concentrate grade significantly, generating 50.7% zinc but at extremely poor recovery of 28%.

BHZ Comp 2 (Sulphide) had slightly better sphalerite liberation but still showed a poor roughing flotation performance. Subsequent regrinding of the zinc rougher concentrate at a specific power of 15 kWh/t (achieving a P80 size of 20 µm) improved the liberation of the sphalerite minerals and achieved an increase in the cleaner concentrate grade of 53.6% Zn at a recovery of 49% Zn.

13.2.4.6 Z5 composites – zinc flotation

Both Z5 Comps 3 and 4 were impacted by the limited sphalerite liberation in the roughing stages however still produced 13% to 20% Zn grades with zinc recovery ranging from 76 to 86%. Subsequent regrind of the concentrates at a specific power of 15 kWh/t (achieving a grind size P_{80} in the range of 20 to 30 µm), resulted in an improvement in concentrate grade at the expense of zinc recovery. The test outcomes have been summarized in Table 13.5 and graphically in Figure 13.6.

Figure 13.6 Grade vs recovery for zinc (T1027)



Source: GRES

13.2.5 Pre-concentration test work

Heavy media separation, gravity separation and Low Intensity Magnetic Separation (LIMs) magnetic enrichment were examined as part of the earlier program. Heavy media separation indicated only a limited potential for effective pre-concentration by HMS. Mozley gravity separation yielded average recoveries, and LIMs magnetite separation failed to produce a clean magnetite concentrate with high deportment of both lead and zinc.

13.2.5.1 Heavy media separation

Heavy media tests were conducted on two sulphide composites (BHZ Met Comp 2 and Z5 Met Comp 3). The results indicated a 10% mass pull to the float reject containing < 2.5% Pb and < 4.7% Zn distribution. The moderate upgrade to the sink product has only slightly increased the sample head grade with 90% of the mass reporting to the sinks.

13.2.5.2 Mozley gravity separation

Mozley gravity separation tests (T15 to T18) were conducted on each composite to examine the separation of a lead concentrate via gravity methods. Gravity concentrate grades of less than 42% Pb where achieved, with recovery values and ranged from 31% to 50% Pb.

13.2.5.3 LIMS magnetite separation

Each composite was assessed for amenability to produce a magnetite concentrate. The composites were ground to a P80 of 106 µm and separated by LIMS. A high mass recovery of magnetics to the magnetic concentrate was achieved with mass recovery of 34 to 54%. Department of both lead and zinc were high to the magnetic concentrate.

13.2.6 Thickening test work

Preliminary settling tests were conducted on the combined rougher and cleaner tails sample for each of the composites. Settled densities > 50% w/w where achieved after three minutes for the sulphide ore composite, as detailed in Table 13.6. The BHZ 1 (Transitional) ore recorded lower densities of 43 – 45% w/w and is expected to have rheology impacts following the observations witnessed during the flotation test work on this material. No settling tests were conducted on the concentrate samples.

Table 13.6 Cylinder settling tests

Batch cylinder settling tests – summary									
Test	Zone / comp	Material	Floc used	Dose g/t	Underflow % solids (at time, minutes)				
					0.5	1	3	10	60
T15	BHZ 2	Zn RoT + Zn Cl1T (T11)	910SH	80	45.4	48.2	51.3	52.6	54.2
T16	BHZ 2	Zn RoT + Zn Cl1T (T11)	910SH	120	47.1	49.5	52.8	53.5	54.1
T17	BHZ 2	Zn RoT + Zn Cl1T (T11)	910SH	40	45.7	48.7	52.6	52.9	54.2
T18	BHZ 1	Zn RoT + Zn Cl1T (T13)	910SH	80	37.4	40.1	43.8	46.1	46.6
T19	BHZ 1	Zn RoT + Zn Cl1T (T13)	945VHM	40	35	40.1	45.6	47.3	48.1
T20	Z5 3	Zn RoT + Zn Cl1T (T12)	910SH	80	47.1	50.2	53.6	54.6	56
T21	Z5 3	Zn RoT + Zn Cl1T (T12)	945VHM	120	47.1	50.4	54.1	55.5	56.3
T22	Z5 4	Zn RoT + Zn Cl1T (T14)	910SH	80	45.2	48.5	52.3	54.2	55.6
T23	Z5 4	Zn RoT + Zn Cl1T (T14)	945VHM	120	47.1	50.4	53.8	55.2	56.6
T24	Z5 3	Zn RoT + Zn Cl1T (T12)	910SH	40	47.1	50.4	54.2	56.4	57.9
T25	Z5 4	Zn RoT + Zn Cl1T (T14)	945VHM	40	46.5	50.1	54.1	55.9	57
T26	Z5 3	Zn RoT + Zn Cl1T (T12)	910SH	20	52	55.3	58.7	59.5	60.7
T27	Z5 4	Zn RoT + Zn Cl1T (T14)	910SH	20	48.1	51.6	55.5	57.8	58.6

13.3 Test work program to April 2018 (T1092)

13.3.1 Current study test work program (T1092) samples

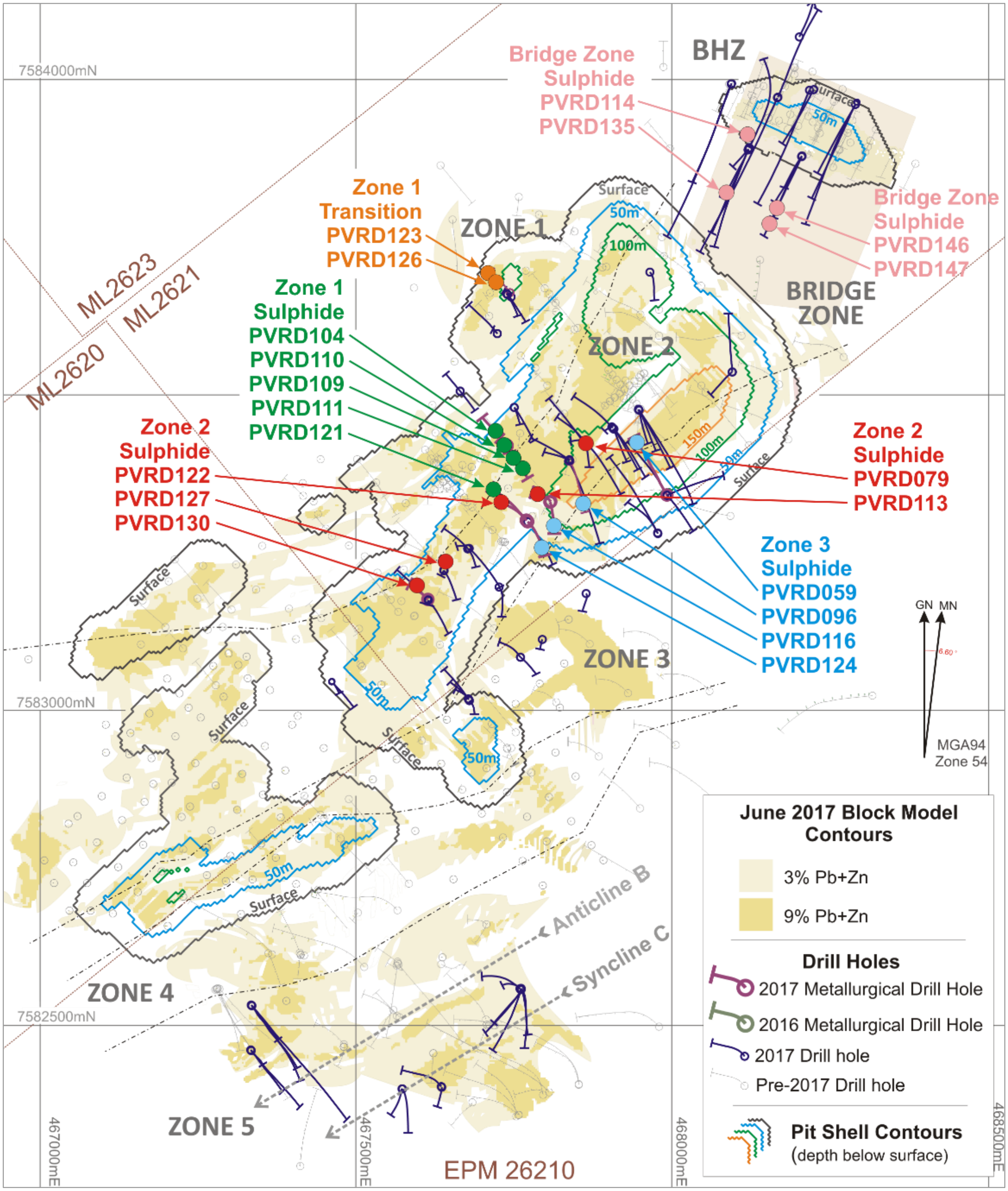
Polymetallic core intersections of the Pegmont lead and zinc resource were received by ALS Metallurgy Burnie Laboratories, Tasmania for composition into five additional metallurgical testing composites. The five metallurgical composites are listed as:

- Z1 Met Comp 5 – Primary sulphide mineralization from Zone 1.
- Z2 Met Comp 6 – Primary sulphide mineralization from Zone 2.
- Z3 Met Comp 7 – Primary sulphide mineralization from Zone 3.
- Z1 Met Comp 8 – Transitional material from Zone 1.
- BZ Met Comp 9 – Primary sulphide mineralization from Bridge Zone.

The transitional zone material is categorized as sulphide mineralization hosted in fresh to partially weathered rock within partial to moderately weathered country host rock.

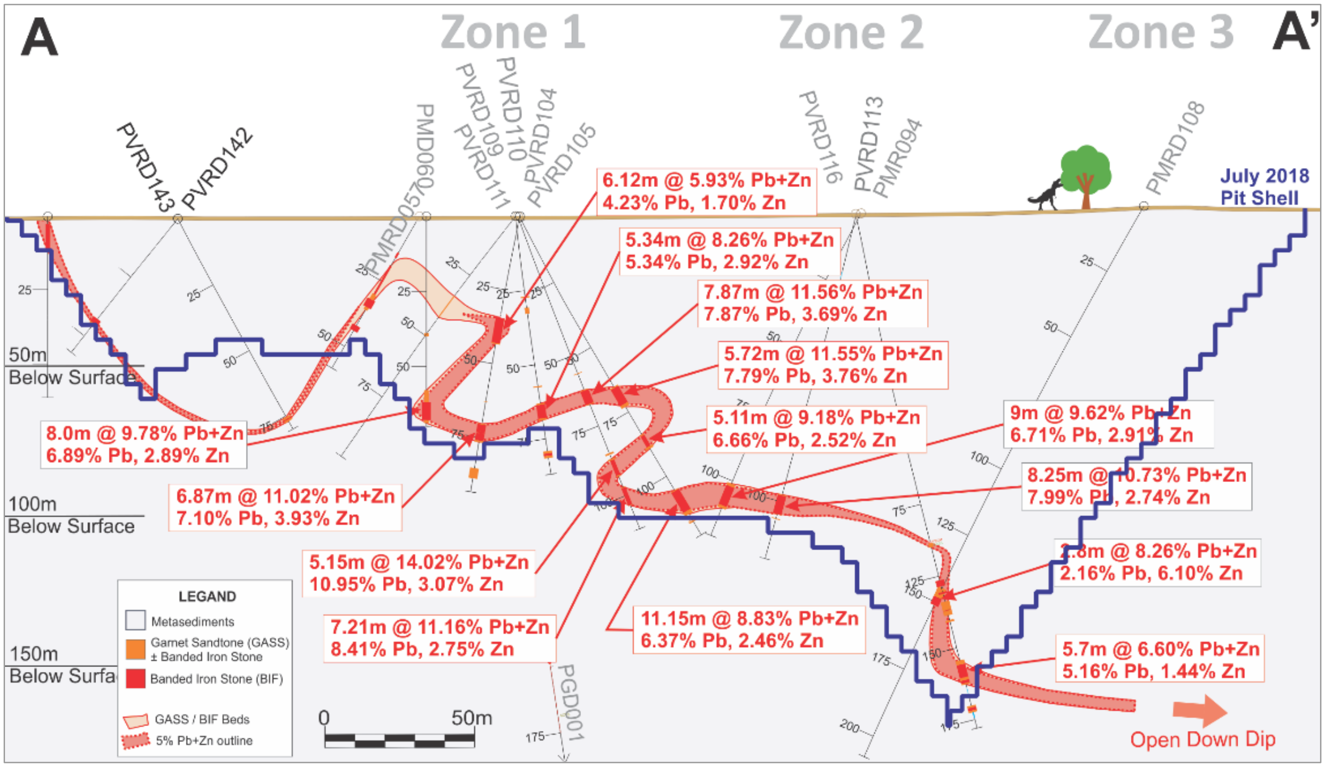
Samples received were half or quarter HQ and NQ core. For each Zone 1, 2, and 3 intersection 0.5 m downhole dilution of waste on either side of the mineralized zone was added to represent probable mining dilution. For the Bridge Zone intersections 1.0 m of downhole waste was added on the hangingwall side of the mineralized zone because of the likely mining method.

Figure 13.7 Metallurgical core locations



Source: VTT

Figure 13.8 Cross section – metallurgical cores



Source: VTT

The head grades of the five composites prepared in this test work are set out in Table 13.7.

Table 13.7 Summary of main test work composites

	%Cu	%Pb	%Zn	%Fe	%S	%F	g/t Ag
Z1 Met Comp 5	0.031	7.95	3.37	29.5	3.30	0.1	14
Z2 Met Comp 6	0.005	7.45	3.28	28.5	2.74	0.1	15
Z3 Met Comp 7	0.005	7.35	3.05	31.0	2.75	0.1	13
Z1 Met Comp 8	0.005	9.00	2.85	28.3	2.84	0.1	19
BZ Met Comp 9	0.007	8.78	2.53	28.0	2.74	0.1	18

The metallurgical test work composites exhibited head grades for lead and zinc that were mostly above the average resource grades. The 2018 resource estimate grades are 6.9% Pb and 2.6% Zn for the primary sulphide mineralization and 4.9% Pb and 2.3% Zn for transitional material. The grade of test work composites 6 and 7 are closest to the average resource grades.

13.3.2 Comminution tests

Bond Ball Mill Work Index (BW_i) tests were conducted at ALS Burnie. Given that most intercepts were supplied as quarter-core it would not have been possible to carry out anything other than BWi determinations. The BW_i tests were conducted using a 106 µm closing screen generating a finer product size compared to the target P_{80} of 106 µm for the primary grind size. Additional BWi test work, using a coarser closing screen, will be required in any subsequent study phase to better reflect target primary grind size. It should be noted that the grind calibration test conducted to produce samples for the subsequent flotation test work was done to achieve the actual target P_{80} of 106 µm. The generated ball mill work indices indicate that the primary material is moderately hard to hard and high grinding power will be required. The results align with the test results generated in the earlier test work program T1027 and reported in Section 13.2.1 which had the BWi indexes ranging from 16.6 kWh/t for transitional material to 19.4 kWh/t for primary samples.

Table 13.8 Bond ball mill work index

Sample	Value
Z1 Met Composite 5	18.4 kWh/t
Z2 Met Composite 6	20.9 kWh/t
Z3 Met Composite 7	20.1 kWh/t
BZ Met Composite 9	19.3 kWh/t

Unconfined Compressive Strength (UCS) tests were conducted on borehole sample PVRD191 to supplement the BWi test data with the following results. To date this is the only sample tested for UCS that has been completed and future work should include additional UCS determinations on samples to adequately represent the mineralized zones.

Table 13.9 Unconfined compressive strength results

Sample	Depth (m)	UCS (MPa)
1	251.5 - 251.7	216
2	252.7 - 252.9	228
3	253.4 -253.6	281
Average		241

The results of the UCS tests done to date were also indicative of competent and hard material.

The decision to proceed with the three-stage crushing and single stage ball mill comminution circuit for this study was based on:

- The measured hardness of the samples tested to date and lack of test work data for AG / SAG milling – additional samples and test work would be required to adequately design a circuit containing an AG / SAG mill.
- Ball mill circuits are typically more stable in operation than AG / SAG based circuits, a significant factor when the comminution circuit is immediately followed by a flotation circuit.

The three-stage crushing circuit adopted for this study was modelled using the Metso BRUNO software. A crushability value of 30% was used in the modelling to reflect the hard material indicated in the test work completed to date. The feed size distribution for feed to the crushing plant was typical for hard rock mining using drill and blast open pit mining techniques.

The grinding circuit design utilized the measured BWi test work values from composite tests. Calculations using the conventional Bond – Rowland method were performed to determine the power required to grind the plant feed material from a feed size of F80 of 9.9 mm to the target product size P80 of 106 µm at a rate of 127 t/h.

13.3.3 Mineralogy

A quantitative optical microscopy assessment on the five metallurgical composites was undertaken by McArthur Ore Deposit Assessments Pty Ltd (MODA) on behalf of ALS in December 2017.

The main objective was to examine the extent of liberation of the main sulphide minerals at the target grind P80 size of 106 µm.

The samples were prepared into size fractions mainly, +150 µm, +106 µm, +75 µm, +38 µm and +20 µm.

The mineralization of the five samples has been summarized in the ALS McArthur Ore Deposit Assessments (MODA) report December 2017 which forms part of the ALS report T1092 May 2018. The report covers the following generalization for the transition and the sulphide composites.

- Z1 Met Comp 8, the sample was noted as being rich in goethite, as part of the transition ore zone. Other minerals include pyrite, pyrrhotite, sphalerite, galena, cerussite, magnetite+goethite (FeOx), and gangue (quartz, silicates).
- Sulphide samples (Z1 Met Comp 5, Z2 Met Comp 6, Z3 Met Comp 7, and BZ Met Comp 9) were goethite poor – primary sulphide ore. Other minerals include pyrite, pyrrhotite, sphalerite, galena, chalcopyrite, cerussite, magnetite, and gangue (quartz, silicates).

The identified minerals are summarized in the Table 13.10.

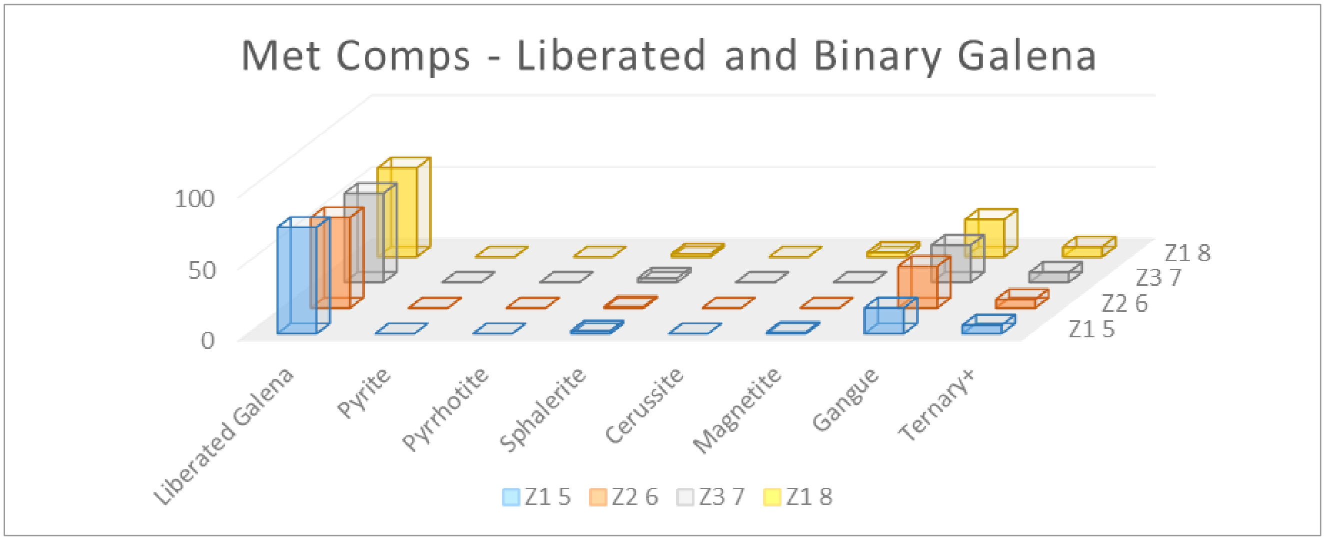
Table 13.10 Mineral observations

Mineral	Observations
Pyrite	Mainly as crystallized pyrite with lesser melnikovite pyrite.
Pyrrhotite	Occurs as ultrafine inclusions in sphalerite and magnetite, with rare coarse liberated grains.
Sphalerite	Mainly as coarse high iron grains, occasionally dusted by ultrafine "chalcopyrite-disease". Associations are mostly with non-sulphide gangue but was also observed with magnetite and rarely with graphic.
Galena	Mainly as coarse grains, only rarely oxidized to cerussite. Significant amounts as witnessed as ragged, fine grained inclusions in gangue (mostly chlorite) and often accompanying magnetite. Some rare flakes of graphite are noted.
Magnetite	Mainly well liberated but also occurs as ragged grains with galena in gangue. Occasionally hosts fine grained blebs of sphalerite and galena.
Goethite	Mainly occurring in the transition ore sample BHZ Comp 1. It occasionally hosts fine grained sphalerite and galena.
Gangue	Is comprised of quartz, garnet, chlorite, graphite and other minor silicates.

13.3.3.1 Galena liberation

Galena liberation was categorized as being 61 to 74% liberated in all samples at the target grind size of 106 µm. The liberation characteristics are displayed in Figure 13.9.

Figure 13.9 Met composites – liberated and binary galena



Source: GRES

Non-liberated galena is mainly binary associated with non-sulphide gangue in the range of 18 to 29%. Minor inclusion of galena is also associated with ternary products in the range of 5 to 7%.

13.3.3.2 Galena size by size liberation

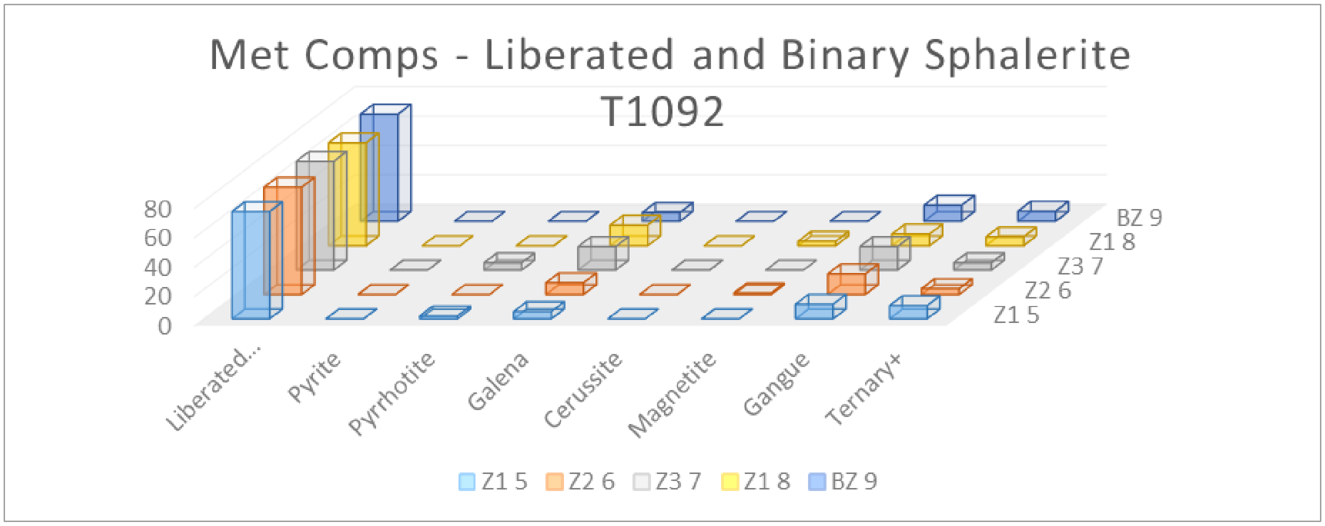
All samples show a high degree of mineral liberation below 38 µm with moderate liberation at the 75 µm size fraction. Above 106 µm, liberation drop below 30% with galena locked with binary and ternary minerals. The main binary locking mechanism is with non-sulphide gangue minerals in the binary phases and some minor locking with sphalerite in the finer size fractions. This implies the minor binary sphalerite will float with the galena concentrate, with regrinding of the concentrate required for further liberation.

For the composites a regrind of the flotation concentrate at the target size of 20 – 38 µm is warranted to liberate galena from the binary and ternary products.

13.3.3.3 Sphalerite liberation

Sphalerite liberation is very good exceeding > 70% in all composites. Binary and ternary associations represent the bulk of the non-liberated sphalerite with minor binary association of sphalerite with galena, magnetite and pyrrhotite. The liberation characteristics are displayed in Figure 13.10.

Figure 13.10 Met composites – liberated and binary sphalerite



Source: GRES

13.3.3.4 Sphalerite size by size liberation

Sphalerite liberation at -75 µm + 38 µm size fraction is > 66% and above 90% in the -38 µm + 20 µm size fraction. The exception being the Z1 Comp 8 (Transitional-Sulphide), where liberation decreases to 68%, with 19% of the binary particles associated with Galena.

Targeting a rougher concentrate regrind target size less than 38 µm to 20 µm would allow for adequate mineral liberation.

13.3.4 Flotation tests T0192

Flotation test work was conducted from 2017 to 2018 by ALS Metallurgy Burnie Laboratory, Tasmania. The samples used are described in Section 13.2.1 above.

The test work program focused on sequential roughing and cleaning tests and repeating the previous test work program (ALS Report T1027) for the 5 new composites. The development of the program followed a roughing stage for each new composite and a rougher with concentrate regrind followed by a 3 stage cleaner float.

The reagent regime for the additional batch flotation tests included sodium cyanide for the depression of pyrite and sphalerite in lead flotation instead of sodium metabisulphite (SMBS) which was used in the majority of the previous batch flotation test program (ALS Report T1072).

13.3.4.1 Primary grinding and lead flotation

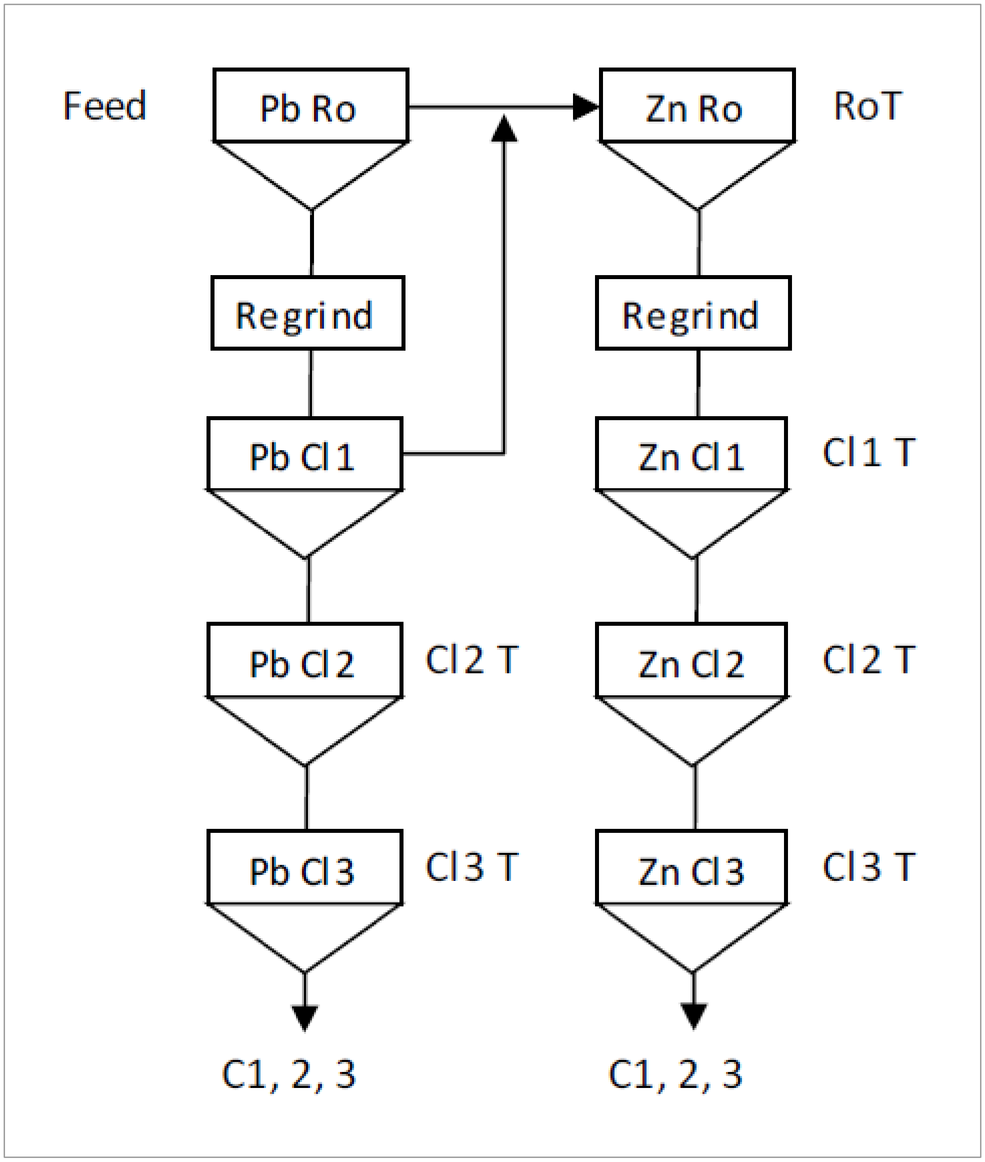
- Lime – pH modifier.
- Sodium Cyanide (NaCN) – Depressant for pyrite and sphalerite.
- Cytec 3418A – Specialty lead sulphide collector.
- Methyl Isobutyl Carbinol (MIBC) – Frother.

13.3.4.2 Zinc flotation

- Lime – pH Modifier.
- Copper Sulphate ($CuSO_4$) – Activator for sphalerite.
- Sodium Isobutyl Xanthate (SIBX) – Collector for sulphide minerals.
- Methyl Isobutyl Carbinol (MIBC) – Frother.

The flotation flowsheet used is detailed in Figure 13.11 and includes a regrind of the rougher concentrates prior to cleaner flotation.

Figure 13.11 Batch flotation flowsheet



Source: GRES

13.3.5 Lead batch flotation T1092

Test results are summarized in Table 13.11 for lead cleaner.

Table 13.11 Lead batch flotation results (T1092)

Comp	Product	Wt%	Pb		Zn		Fe		S	
			%	Rec%	%	Rec%	%	Rec%	%	Rec%
T11 – Zone 1 Sulphide Comp 5	Pb Cl 3 (c1,2,3)	10.8%	68.3	89.1	3.4	10.5	4.2	1.5	13.0	40.8
T12 – Zone 1 Sulphide Comp 6	Pb Cl 3 (c1,2,3)	10.5%	63.9	88.5	4.4	13.6	5.2	1.9	12.6	46.5
T13 – Zone 1 Sulphide Comp 7	Pb Cl 3 (c1,2,3)	10.0%	67.1	86.5	4.2	13.4	5.7	1.8	12.9	45.0
T14 – Zone 1 Transition Comp 8	Pb Cl 3 (c1,2,3)	11.8%	71.2	89.4	4.3	17.8	3.5	1.4	13.3	54.6
T16 – Zone 1 Transition Comp 8	Pb Cl 3 (c1,2,3)	10.3%	76.0	83.8	3.2	11.7	2.2	0.8	13.7	48.8
T17 – Zone 1 Transition Comp 8	Pb Cl 3 (c1,2,3)	9.7%	74.1	83.9	2.6	9.1	3.1	1.0	12.4	45.3
T15 – Bridge Zone Comp 9	Pb Cl 3 (c1,2,3)	10.9%	70.6	82.9	3.7	15.6	3.9	1.5	12.6	48.8

13.3.5.1 Z1 transition composites – lead flotation

Cleaner tests conducted on the transition Zone 1 composite 8 with the exception of T17 applied the same rougher / regrind and staged cleaner test procedure. For T17 the rougher concentrate underwent a first stage cleaner float prior to the regrind stage at 20 kWh/t, followed by 2 stages of cleaning. T14 used the standard 10 kWh/t specific energy for the regrind stage, while T16 used a higher specific energy target of 15 kWh/t.

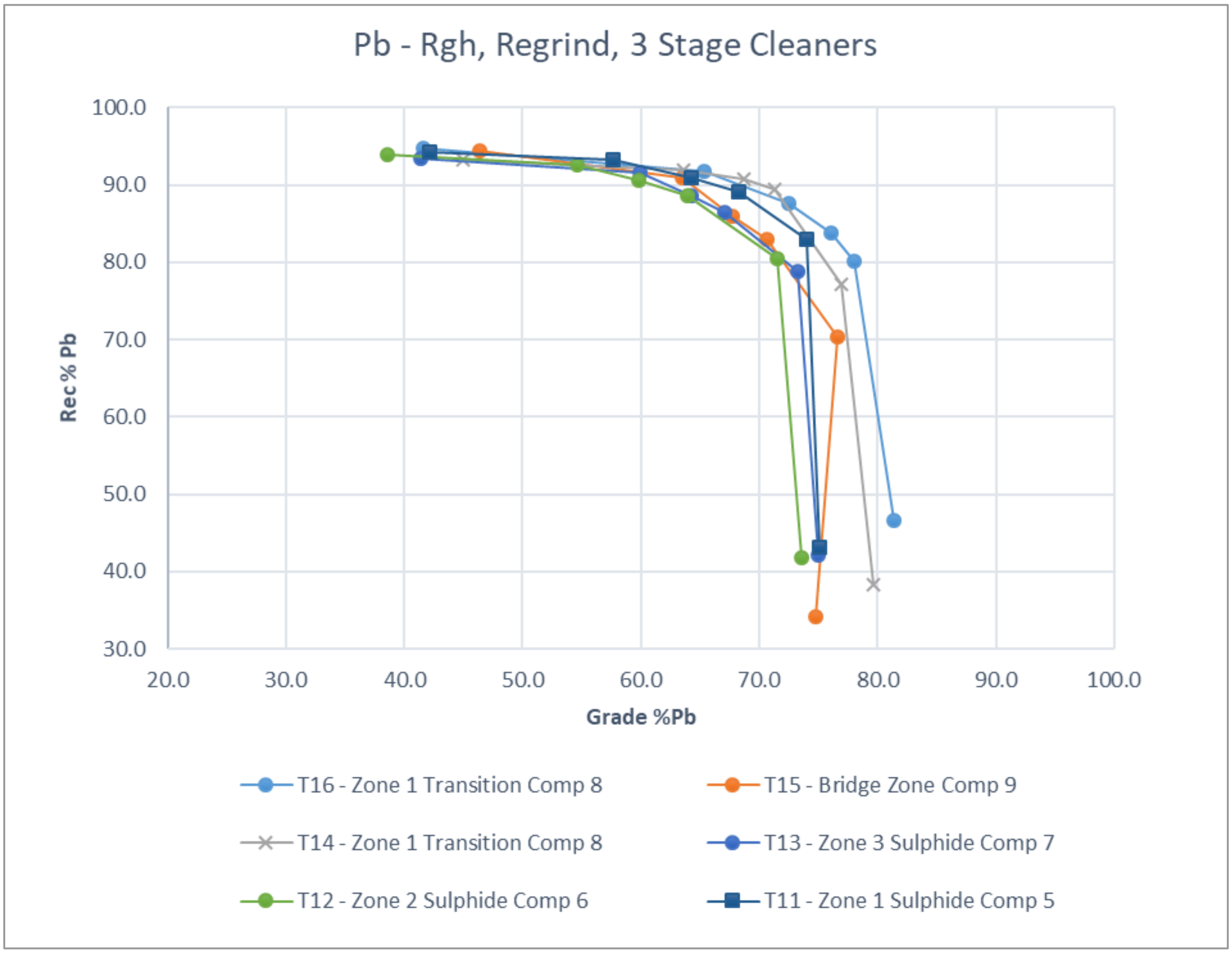
Results from all tests show a lead concentrate grade of > 70% Pb with recoveries > 83% Pb are possible on the transitional sulphide zone. A reduction in deportment of zinc to the final concentrate was improved with the increase in regrind specific energy as noted for tests T14 to T16 to T17.

It is important to note that T17 used sodium monophosphate (SMP) in the primary grinding stage as an alternative reagent to sodium cyanide and the sodium metabisulphite (SMBS) option. The use of SMP was also adopted in the subsequent locked cycle tests.

13.3.5.2 Z1 sulphide composites – lead flotation

Cleaner flotation tests have indicated excellent recoveries and grades can be generated with the sulphide ore. These are detailed in Table 13.11.

Figure 13.12 Lead grade vs recovery curves



Source: GRES

13.3.5.3 Z2 composites – lead flotation

Flotation tests for Zone 2 composite 6 included 2 roughing tests with slight variations to the reagent scheme and one cleaning test. Results of the tests are shown above in Table 13.11 and reported in detail in the "ALS Metallurgical Program Report T1092". Zinc was similarly high during the rougher tests with the deportment mainly being associated with binary and ternary particles.

Cleaner flotation test T12 reduced the deportment of zinc to the lead concentrate following a 10 kWh/t regrind stage on the rougher concentrate and three stages of cleaning. A lead cleaner concentrate grade of 63.9% Pb at 88.5% recovery was achieved in the third cleaner.

13.3.5.4 Z3 composites – lead flotation

Flotation tests for Zone 3 composite 7 included 2 roughing tests with an identical reagent regime as applied to the Z2 tests and one cleaning test to repeat the previous program tests for the new zone ore.

Cleaner flotation test T12 reduced the deportment of zinc to the lead concentrate following a 10 kWh/t regrind stage on the rougher concentrate and three stages of cleaning. A cleaner concentrate grade of 67.1% Pb at 86.5% recovery was achieved in the third cleaner.

13.3.5.5 BZ composites – lead flotation

Flotation tests for Zone BZ composite 9 included 2 roughing tests an identical reagent regime as applied to the Z2 tests and one cleaning test to repeat the previous program tests for the new zone ore.

Cleaner flotation test T12 reduced the deportment of zinc to the lead concentrate following a 10 kWh/t regrind stage on the rougher concentrate and three stages of cleaning. A cleaner concentrate grade of 70.6% Pb at 82.9% recovery was achieved in the third cleaner.

13.3.6 Zinc batch flotation T1092

Test results summarized Table 13.12 for zinc batch cleaner stages.

Table 13.12 Zinc flotation results (T0192)

Comp	Product	Wt%	Pb		Zn		Fe		S	
			%	Rec%	%	Rec%	%	Rec%	%	Rec%
T11 – Zone 1 Sulphide Comp 5	Zn Cl 3 (c1,2,3)	3.7%	1.8	0.8	58.8	61.8	5.9	0.7	32.4	34.7
T12 – Zone 2 Sulphide Comp 6	Zn Cl 3 (c1,2,3)	3.4%	2.0	0.9	60.6	61.7	4.0	0.5	32.3	39.0
T13 – Zone 3 Sulphide Comp 7	Zn Cl 3 (c1,2,3)	3.5%	3.9	1.7	56.8	62.8	5.7	0.6	31.0	37.3
T14 – Zone 1 Transition Comp 8	Zn Cl 3 (c1,2,3)	3.0%	5.3	1.7	55.2	57.2	6.0	0.6	31.2	32.2
T16 – Zone 1 Transition Comp 8	Zn Cl 3 (c1,2,3)	2.8%	2.6	0.8	59.0	58.5	5.2	0.5	31.1	27.9
T17 – Zone 1 Transition Comp 8	Zn Cl 3 (c1,2,3)	2.8%	3.6	1.2	57.6	57.5	5.2	0.5	31.5	33.0
T15 – Bridge Zone Comp 9	Zn Cl 3 (c1,2,3)	2.6%	9.0	2.5	50.1	50.7	7.7	0.7	30.2	27.9

13.3.6.1 Z1 transition composites – zinc flotation

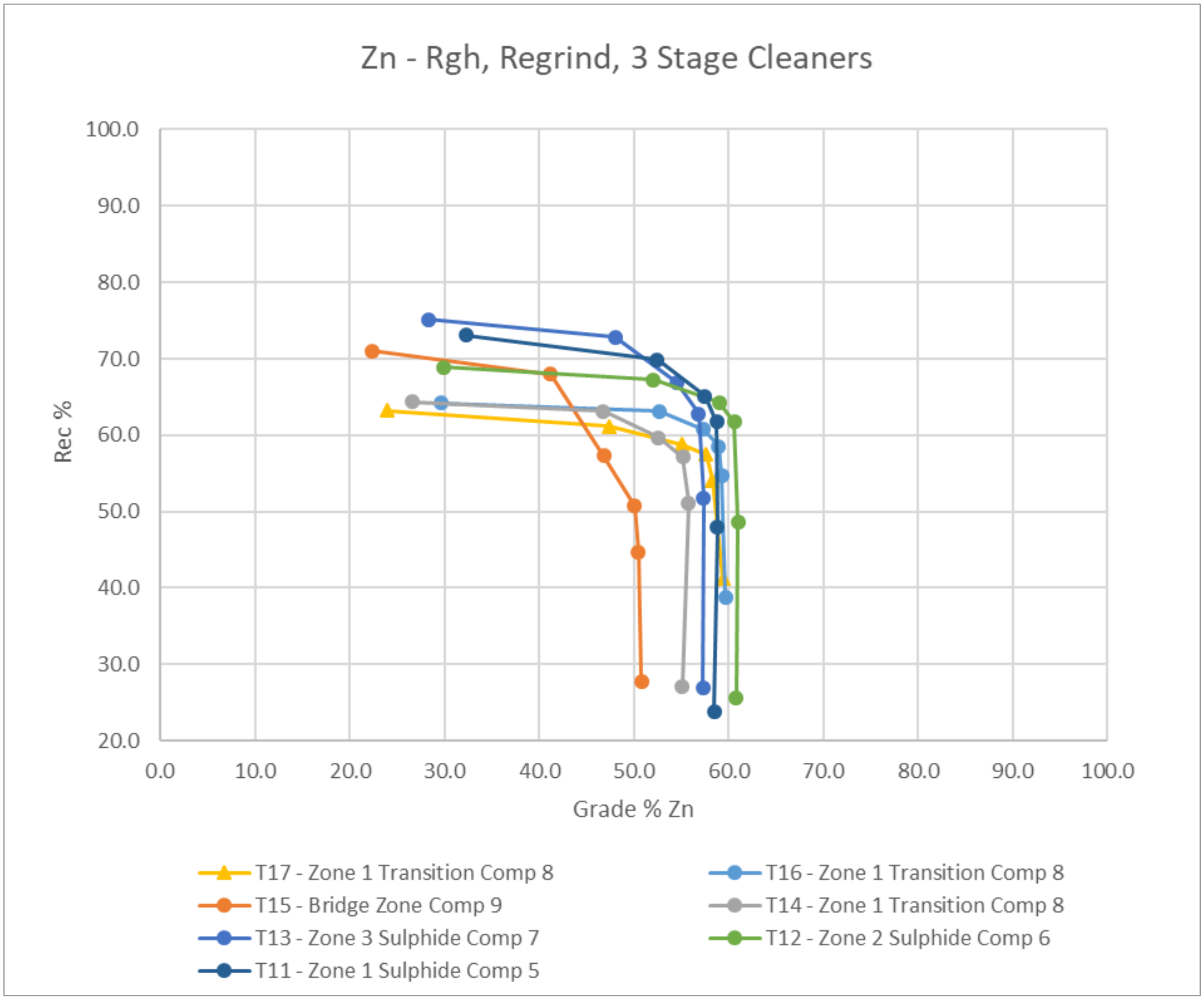
Cleaner tests conducted on Z1 composite 8 (Transition) used the same target regrind specific power of 10 kWh/t for all tests.

Results indicate that a zinc concentrate grade > 55% Zn with recoveries > 51% Zn are possible on the transitional sulphide zone.

13.3.6.2 Z1 sulphide composites – zinc flotation

Cleaner tests on the Z1 composite 8 (Sulphide), included 2 rougher tests and one cleaner test, following on from the previous flotation program. The cleaning flotation tests returned 58.8% zinc at 61.8% recovery. Lead rejection was better in the sulphide composite compared to the transitional ore results.

Figure 13.13 Zinc grade vs recovery curves



Source: GRES

13.3.6.3 Z2 composites – zinc flotation

Flotation tests for Zone 2 composite 6 included 2 roughing tests and one cleaning test. These are shown in Figure 13.13 for the zinc grade and recovery relationship.

Cleaner flotation test T12 produced a zinc grade of 60.6% zinc at 61.7% recovery.

13.3.6.4 Z3 composites – zinc flotation

Flotation tests for Zone 3 composite 7 included 2 roughing tests with and one cleaning test. Refer to Figure 13.13.

Cleaner flotation test T13 produced a concentrate grade of 56.8% zinc at 62.8%.

13.3.6.5 BZ composites – zinc flotation

Flotation tests for Zone BZ composite 9 included 2 roughing tests with and one cleaning test. Refer to Figure 13.13.

Cleaner flotation test T15 produced a concentrate grade of 50.1% zinc at 50.7%.

13.3.7 Locked cycle flotation tests

Locked cycle flotation test work was conducted on the sample described in Section 13.2.1 above.

The locked cycle tests were performed using Sodium Monophoshate (SMP) following batch Test T17. SMP was selected to provide an alternative reagent scheme to the conventional use of Sodium Cyanide and or Sodium Metabisulphite in conjunction with Zinc Sulphate for the depression of pyrite and activated sphalerite. Experience at Myra Falls operations in Canada (T Yeomans 2008) indicated the use of SMP may be beneficial to the copper or lead and zinc selectivity.

13.3.7.1 Primary grinding and lead flotation

- Lime – pH modifier.
- Sodium Monophosphate – modifying agent.
- Cytec 3418A – Specialty lead sulphide collector.
- Methyl Isobutyl Carbinol (MIBC) – Frother.

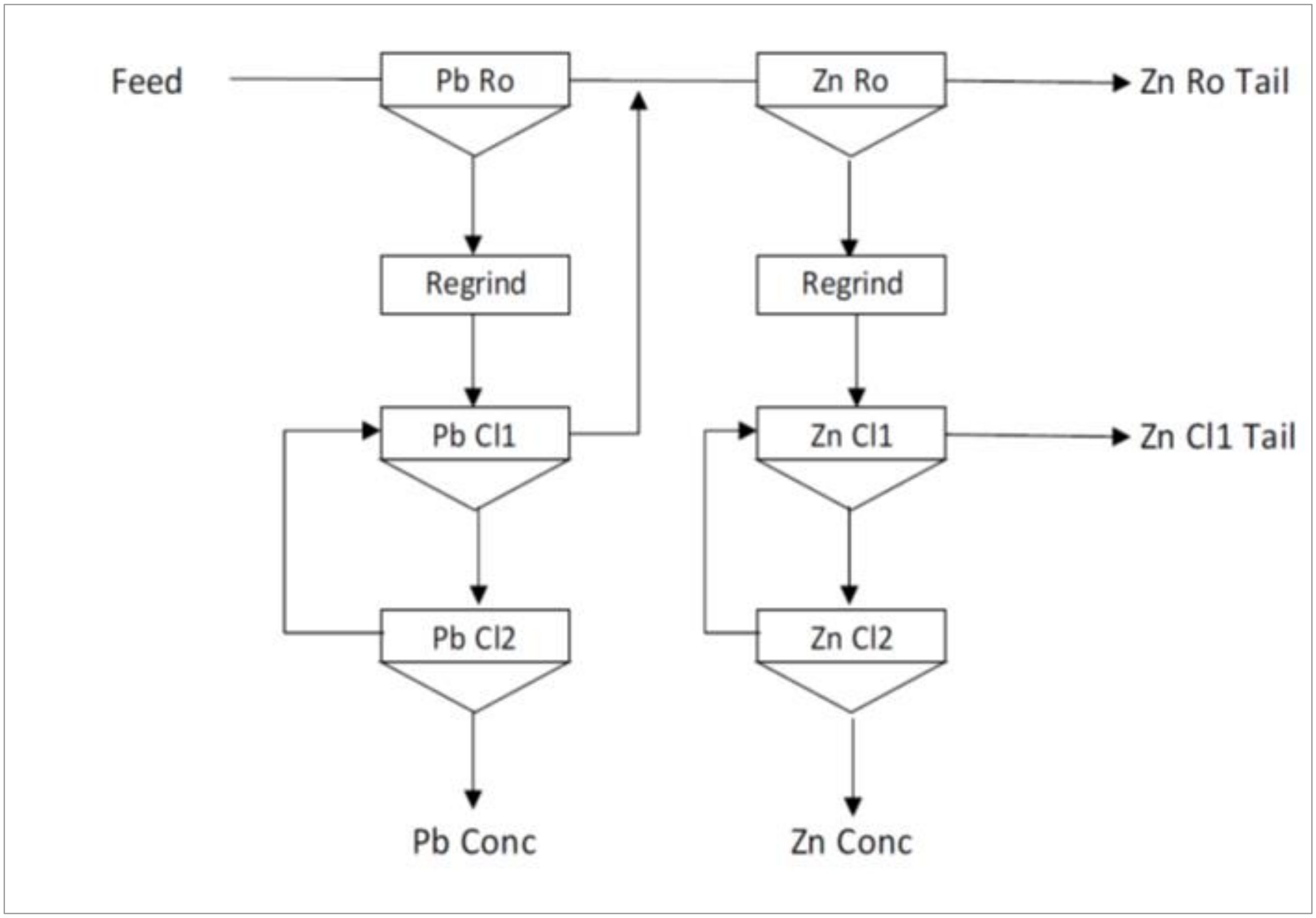
13.3.7.2 Zinc flotation

- Lime – pH Modifier.
- Copper Sulphate ($CuSO_4$) – Activator for Sphalerite.
- Sodium Isobutyl Xanthate (SIBX) – Collector for sulphide minerals.
- Methyl Isobutyl Carbinol (MIBC) – Frother.

The locked cycle flotation flowsheet is detailed in Figure 13.14 and includes a regrind of the lead and zinc rougher concentrates prior to two stages of cleaner flotation.

Lead rougher concentrate regrind was conducted using specific power input of 15 kWh/t for a target regrind P80 size of 20 to 23 µm. The zinc rougher concentrate regrind was conducted using a specific power input of 10 kWh/t for a target P80 size of 20 to 23 µm.

Figure 13.14 Lock cycle test flowsheet



Source: GRES

The locked cycle results show consistent and excellent lead response for the six composites with 66 to 72% Pb concentrate at 89 to 93% Pb recovery. Zinc flotation concentrates yielded 52 to 54% Zn at recoveries of 70 to 75%.

Results of the lock cycle tests are detailed in Figure 13.14.

Table 13.13 Locked cycle tests

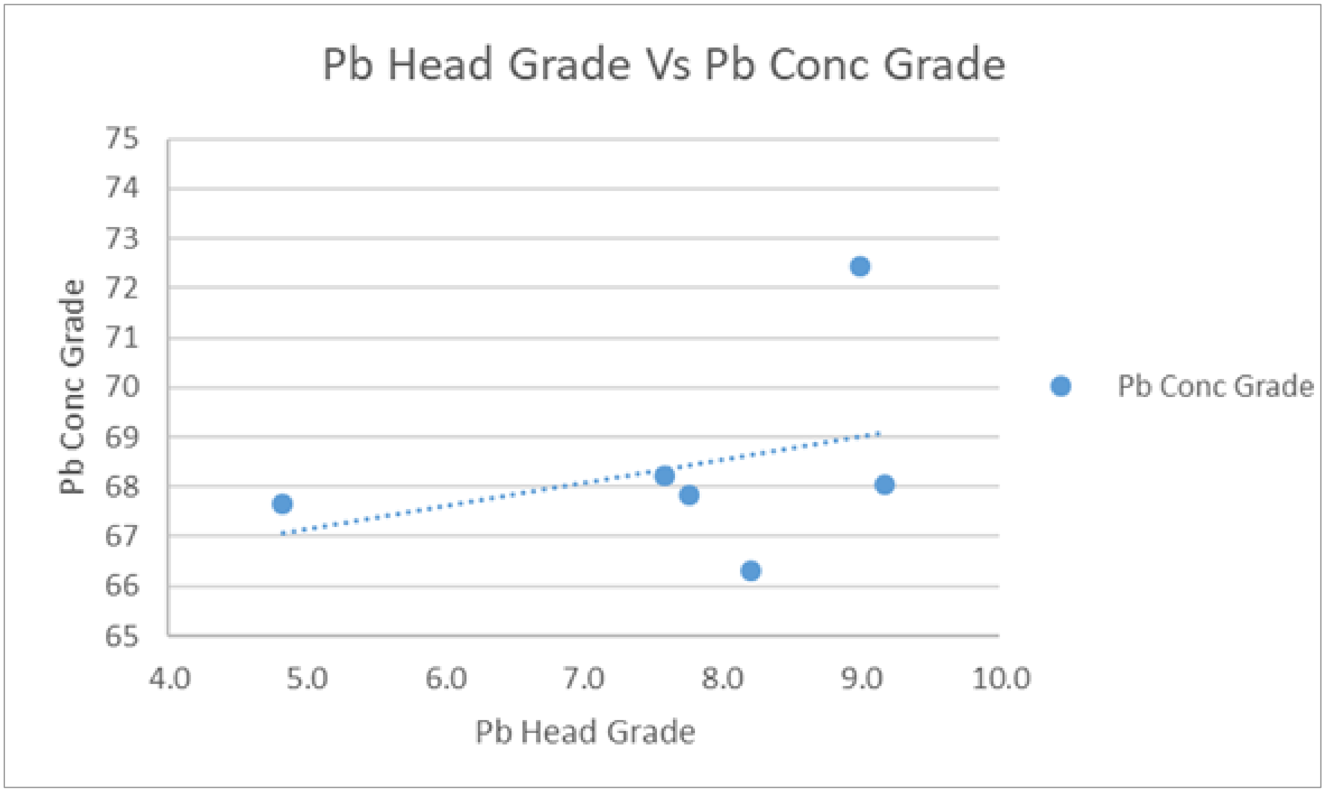
Laboratory	ALS	ALS	ALS	ALS	ALS	ALS
Sample	Comp 7	Comp 8	Comp 9	Comp 5	Comp 6	Comp 2
Zone / type	3Primary	1 Transition	BZ Primary	1 Primary	2 Primary	BHZ Primary
Test ID	LC001	LC002	LC003	LC004	LC005	LC006
Water	Burnie	Burnie	Burnie	Burnie	Burnie	Burnie
Head grade						
-% Pb calculated	7.58	9.00	9.17	8.20	7.76	4.83
-% Zn calculated	3.11	2.77	2.38	3.39	3.34	2.65
-% Fe	30.7	28.3	27.7	29.5	28.3	29.7
SMP (primary grind)	150 g/t	150 g/t	149 g/t	150 g/t	150 g/t	150 g/t
Grind P80 µm	106	106	106	106	106	106
Pb flotation						
Collector -3418A	30 g/t	26 g/t	30 g/t	30 g/t	30 g/t	27 g/t
Frother - MIBC	50 g/t	50 g/t	53 g/t	57 g/t	53 g/t	52 g/t
Rougher						
- pH	9	8	9.1	9.1	9.1	8.5
- Time min	10	11.5	11	11	11	11
Regrind P80 µm	23	22	24	24	23	NTD
Power (kwh/t)	15	15	15	15	15	15
Cleaner						
- pH	10.1	10.2	10.1	10.1	10.0	10.2
- Time min	6.5	14	13.5	14	12.5	16
Re-cleaner						
- pH	10.5	10.5	10.5	10.5	10.5	10.53
- Time min	13	9	10.5	6.5	6.5	6.5
- Mass pull %	9.9	11.4	12.5	11.3	10.4	6.3
-% Pb	68.2	72.4	68.0	66.3	67.8	67.7
- Pb recovery %	89.7	91.3	92.7	91.8	90.8	88.0
-% Zn	2.9	2.5	3.4	3.6	3.6	3.1
- Zn recovery %	9.2	10.2	17.7	12.1	11.3	7.4
Zn flotation						
Collector SIBX	19 g/t	30 g/t	40 g/t	31 g/t	26 g/t	34 g/t
Frother MIBC	67 g/t	72 g/t	63 g/t	71 g/t	64 g/t	47 g/t
Copper sulphate	140 g/t	89 g/t	139 g/t	90 g/t	90 g/t	90 g/t
Rougher						
- pH	8.2	8.5	8.4	8.5	8.7	8.55
- Time min	12.2	10.5	9.6	11	9.7	9
Regrind P80 µm	21	20	25	21	20	NTD
Power (kwh/t)	10	10	10	10	10	12.5
Cleaner						
- pH	10.5	10.5	10.5	10.5	10.5	10.8
- Time min	8.2	8.6	7.6	6.5	7.0	10
Re-cleaner						
- pH	11	11	11	11.1	11.1	11.3
- Time min	3.2	3.5	2.8	3.5	5	3

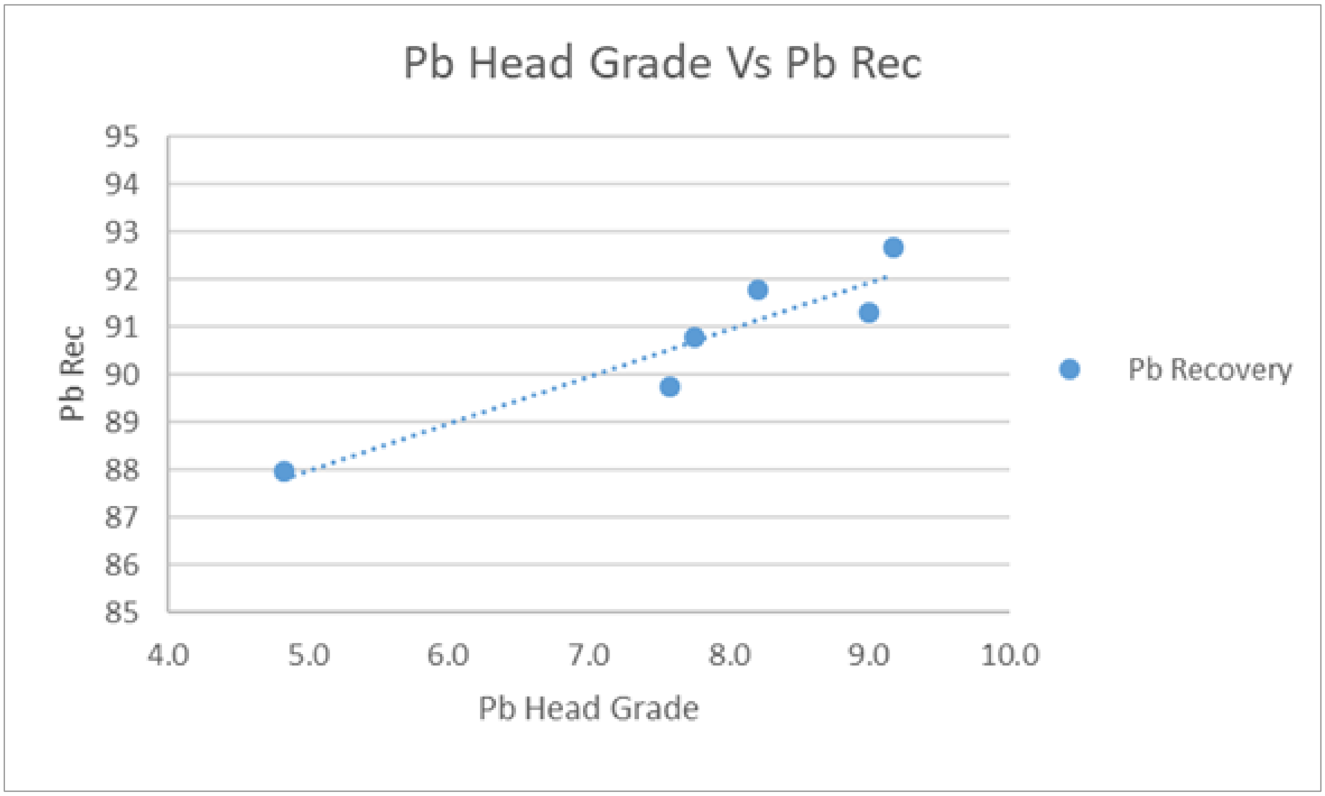
Laboratory	ALS	ALS	ALS	ALS	ALS	ALS
- Mass pull %	4.18	3.90	3.21	4.69	4.34	4.06
-% Pb	4.51	5.11	5.47	3.24	3.61	3.54
- Pb recovery %	2.49	2.22	1.91	1.85	2.02	2.98
-% Zn	54.8	53.3	52.3	54.6	54.9	51.2
- Zn recovery %	73.7	75.2	70.4	75.5	71.3	78.5
Final tail						
- Mass pull %	85.8	84.8	84.3	83.9	85.3	89.7
- Tail assay %Pb	0.69	0.69	0.59	0.63	0.66	0.52
- Tail assay %Zn	0.62	0.48	0.33	0.50	0.68	0.44

Lead deportment to the zinc concentrate was less than 3%. Zinc deportment to the lead concentrate continued to be high, with a median value of 10.7% Zn reporting to the Pb recleaner concentrate.

The relationship for lead recovery based on the lead head grade is illustrated in Figure 13.14 Lock cycle tests which have shown a trend of increasing recovery for increasing head grade

Figure 13.15 Pb head grade to recovery relationship





The lead and zinc flotation circuit design have been based on the flowsheet tested at laboratory scale. Laboratory flotation residence times for design used the locked cycle test times, with industry standard scale up factors applied and checks to ensure area and lip length capacities were not exceeded.

13.3.8 Thickening test work

Cylinder only settling tests were conducted on a combined rougher and cleaner tails sample of each of the five composites. Very good settling rates were observed. No settling tests were conducted on the lead and zinc concentrate samples.

Table 13.14 Cylinder settling tests

Batch cylinder settling tests – summary									
Test	Test	Material	Floc used	Dose g/t	Underflow % solids (at time, minutes)				
					0.25	1	5	10	60
T18	LC01	Zn RoT + Zn Cl1T (T18)	910VHM	8	39.8	53.9	60.3	61.2	61.6
T19	LC01	Zn RoT + Zn Cl1T (T19)	910VHM	2	28.5	61.2	67.9	67.9	69
T20	LC01	Zn RoT + Zn Cl1T (T20)	910VHM	12	47.7	56.7	60.7	61.6	62
T21	LC01	Zn RoT + Zn Cl1T (T21)	910VHM	8	24.4	51	58.7	58.7	59.1
T22	LC02	Zn RoT + Zn Cl1T (T22)	910VHM	2	16	45.4	58.2	61.1	63.8
T23	LC02	Zn RoT + Zn Cl1T (T23)	910VHM	8	19.6	44.7	54.9	56.3	58.2
T24	LC02	Zn RoT + Zn Cl1T (T24)	910VHM	12	25.2	44.9	54.5	55.6	57
T25	LC02	Zn RoT + Zn Cl1T (T25)	945SH	8	20.1	43.6	53.5	54.8	55.5
T26	LC03	Zn RoT + Zn Cl1T (T26)	910VHM	2	17.9	51.4	63.5	64.9	65.4
T27	LC03	Zn RoT + Zn Cl1T (T27)	910VHM	8	34.2	52	59.1	60	61.2
T28	LC03	Zn RoT + Zn Cl1T (T28)	910VHM	12	38.2	51.1	57.6	58.7	59.1
T29	LC03	Zn RoT + Zn Cl1T (T29)	945SH	8	35.3	49.8	57.6	58.4	58.8
T30	LC04	Zn RoT + Zn Cl1T (T30)	910VHM	2	17.8	50.5	63.6	65.6	66.1
T31	LC04	Zn RoT + Zn Cl1T (T31)	910VHM	8	36.2	51.2	58.5	59.3	59.7
T32	LC04	Zn RoT + Zn Cl1T (T32)	910VHM	12	28.2	50.3	57.7	59.3	59.7
T33	LC04	Zn RoT + Zn Cl1T (T33)	945SH	8	20.9	37.3	53	58.5	58.9
T34	LC05	Zn RoT + Zn Cl1T (T34)	910VHM	2	19.5	50.8	63	64	64.4
T35	LC05	Zn RoT + Zn Cl1T (T35)	910VHM	8	37.1	50.2	57.6	58.3	58.7
T36	LC05	Zn RoT + Zn Cl1T (T36)	910VHM	12	40	50.2	56.8	57.6	58.3
T37	LC05	Zn RoT + Zn Cl1T (T37)	945SH	8	336.	46.8	53.9	54.6	55

The tailings thickener has been designed on the basis of similar installations using a feed flux rate of 1 t/m^{2}h. Both concentrate thickeners have been designed on a flux rate of 0.25 t/m^{2}h.

13.3.9 Penalty elements in concentrate

Analysis of the composites for BHZ and Z5 were conducted specifically for fluorine on both the head samples and the lead and zinc concentrates from the batch flotation test work. The assays were for third cleaner concentrate. Results are tabled in Table 13.15.

Table 13.15 Fluorine analysis – T1027 batch flotation tests

Sample ID	Sample description	F ppm	Sample ID	Sample description	F ppm
1027003	BHZ Met Comp 1 Head	1380	1027286	T11 Pb Cln3 Con 1-3	147
1027004	BHZ Met Comp 2 Head	2080	1027287	T11 Zn Cln3 Con 1-3	49.9
1027005	Z5 Met Comp 4 Head	1340	1027288	T13 Pb Cln3 Con 1-2	< 25
1027006	Z5 Met Comp 3 Head	1180	1027290	T14 Pb Cln3 Con 1-2	245
1027284	T08 Pb Cln3 Con 1-3	240	1027291	T14 Zn Cln3 Con 1-2	78.9
1027285	T08 Zn Cln3 Con 1-3	81.2	1027292	T09 Zn Cln3 Con 1-2	26.8

Fluorine levels in feed were significant ranging from 1,180 ppm to 2,080 ppm. However, despite relatively high levels of fluorine in the feed samples there was excellent rejection of fluorine in the concentrates in the T1027 program.

ALS undertook an analysis of the locked cycle test lead and zinc final concentrates for a wider spectrum analysis. The assays are for second cleaner concentrate. Key penalty elements are listed in Table 13.16, with the exception of Mercury (Hg) which was not analyzed.

Table 13.16 Penalty elements in lock cycle test products

Sample description	ME-XRF15b F %	ME-MS61 Ag ppm	ME-MS61 As ppm	ME-MS61 Bi ppm	ME-MS61 Cd ppm	ME-MS61 Fe %	ME-MS61 Mg %	ME-MS61 Sb ppm
LC01 Zn Cl2 Con Cyc4-Cyc6	<0.1	7.29	12.5	1.92	3,490	5.53	0.05	13
LC02 Zn Cl2 Con Cyc4-Cyc6	<0.1	27.7	3.90	2.28	3,250	5.85	0.03	3.56
LC03 Zn Cl2 Con Cyc4-Cyc6	<0.1	10.4	7.8	2.82	2,980	6.75	0.04	16.95
LC04 Zn Cl2 Con Cyc4-Cyc6	<0.1	7.86	22.5	6.29	3,510	6.71	0.05	9.25
LC05 Zn Cl2 Con Cyc4-Cyc6	<0.1	5.01	21	0.65	3,640	5.71	0.06	10.9
LC01 Pb Cl2 Con Cyc4-Cyc6	<0.1	112	24.7	11.3	204	5	0	40
LC02 Pb Cl2 Con Cyc4-Cyc6	<0.1	92.8	8.8	16.65	187.5	3.58	0.06	10.85
LC03 Pb Cl2 Con Cyc4-Cyc6	<0.1	119	17.2	23.4	209	4.22	0.11	79.3
LC04 Pb Cl2 Con Cyc4-Cyc6	<0.1	117	42.7	99.8	248	4.73	0.11	31.8
LC05 Pb Cl2 Con Cyc4-Cyc6	<0.1	94.7	29.7	6.75	261	4.47	0.11	35.2

The assays indicate few penalty elements of significance. All the key penalty elements reported in Table 13.16 are below the typical penalty limits with the exception of cadmium (3,000 ppm to 3,600 ppm reporting to the zinc concentrate). Fluorine was measured only to less than 1,000 ppm or the detection limit of the assay method applied to each of the concentrates generated from the locked cycle tests. From the early batch flotation tests (T1027) the fluorine concentration in the lead concentrate was measured to be under 300 ppm and below 100 ppm for the zinc concentrate. Given a similar flotation performance, the fluorine deportment in the locked cycle tests is expected to be similar.

13.3.10 Concentrate grades and recoveries

A summary of the recoveries and concentrate grades applied to the plant design criteria are reported in Table 13.17. High lead recoveries of 91% were achievable at lead grades of 68.9% in the locked cycle tests following changes to the flotation reagent scheme. The zinc concentrates developed are at a commercial grade, with slightly higher than desired lead recovery to the concentrate from binary associations with sphalerite.

Table 13.17 Concentrate grades and recoveries

	Lead	Zinc	Silver
Recoveries to lead concentrate	91.0%	10.4%	75.2%
Lead concentrate grade	68.9%	3.2%	100 g/t
Recoveries to zinc concentrate	1.8%	73.2%	2.4%
Zinc concentrate grade	3.1%	54.0%	7.6 g/t

13.4 Proposed future test work

The flowsheet adopted in the flotation test work and lock cycle tests has demonstrated that the ore is amenable to the conventional sequential flotation process, inclusive of a rougher concentrate regrind stage. Due to the limited comminution test work, which has solely focused on the Bond Ball Mill Work Index (BW_i) determination, the use of a three-stage crushing circuit followed by single stage ball milling has been adopted for the study. The three-stage crush / ball mill circuit was also adopted due to the high BW_i values and high UCS figures.

- Future comminution test work is recommended, and to include both SMC and JKTech tests to assess the ore for amenability to AG / SAG / Ball grinding circuit options and further flowsheet optimization. In addition, Bond crushing, rod and ball mill work index determinations and Bond abrasion index tests will be required for verification of the crushing and grinding circuit designs and operating costs.
- Future flotation test work is also recommended to better define rougher flotation kinetics to investigate use of flash flotation for recovery of fast floating galena, the effect of grind and regrind sizes on both lead and zinc flotation, optimization of zinc grade and recovery and to establish the effects of site water, mild steel media, and sample aging (oxidation).
- Comminution variability tests on samples from different zones and at different depths are recommended. Additional flotation variability test work at different depths, grades and spatial distribution within each of the mineralized zones.
- Additional locked cycle testing to examine the variability across the ore deposit using the current regent scheme with site water is recommended.
- Dynamic thickening test work of flotation products and filtration test work on flotation concentrates.

14 Mineral Resource estimates

14.1 Introduction

The Mineral Resource estimate for the Pegmont deposit has been carried out by independent QP, Ms Dinara Nussipakynova, P.Geo. of AMC, who takes responsibility for the estimate. Table 14.1 shows the Mineral Resources for the lead-zinc-silver mineralization as of 31 July 2018, at a 3% lead plus zinc cut-off grade for the open pit Mineral Resources and a 5% lead plus zinc cut-off grade for the underground Mineral Resources.

In this update of the Mineral Resources, separate block models were built for the BHZ and the Pegmont Zones 1 – 5, also referred to as Main Zone. A Mineral Resource was also estimated for the first time on the Bridge Zone. This was done as a separate block model. The database and treatment of the data is common to all models; therefore Section 14.2 of the report applies all the models. The inputs to cut-offs applied are referenced in the footnotes to the tables and are also discussed later in the text. The open pit Mineral Resources are constrained within an optimized pit which has 55º walls.

Oxide lead-zinc mineralization is not included in the current Mineral Resource, as with the sequential flotation processing flow sheet envisaged, it is considered that there is no effective method for economic upgrading of the oxide mineralization and hence no basis for its inclusion. Table 14.1 shows the total Mineral Resources for BHZ, Bridge Zone, and Pegmont Zones 1 – 5 with open pit and underground combined. The breakdown for each zone is shown in Table 14.19, Table 14.24, and Table 14.32. The majority of the tonnes are located in the Pegmont Zones which make up 84% of the Indicated and 95% of the Inferred categories.

Table 14.1 Mineral Resources at 31 July 2018

Classification	Material type	Tonnes (kt)	Pb (%)	Zn (%)	Ag (g/t)
Indicated	Transition	1,111	4.9	2.3	8
	Sulphide	4,647	6.9	2.6	12
	Total	**5,758**	**6.5**	**2.6**	**11**
Inferred	Transition	1,829	5.2	2.0	7
	Sulphide	6,447	5.1	3.1	9
	Total	**8,277**	**5.1**	**2.8**	**8**

Notes:
1. CIM Definition Standards (2014) were used to report the Mineral Resources.
2. Cut-off grade applied to the open pit Mineral Resources is 3% Pb+Zn and that applied to the underground is 5% Pb+Zn.
3. Based on the following metal prices: US$0.95/lb for Pb, US$1.05/lb for Zn, and US$16.5/oz for silver.
4. Exchange rate of US$0.75:A$1.0.
5. Metallurgical recoveries vary by zone and material type as follows:
    - Lead to lead concentrate: from 80.6% to 91.3% for transition and 88.0% to 92.7% for sulphide.
    - Zinc to zinc concentrate: from 19.3% to 75.2% for transition and 61.8% to 78.5% for sulphide.
6. Using drilling results up to 15 April 2018.
7. Mineral Resource tonnages have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

14.2 Data used

14.2.1 Drillholes

The data was provided as an MS Access Database with a cut-off date of 15 April 2018. The tables comprise collar coordinates, surveys, assays, SG, and lithological and structural logging, and were imported and verified using Datamine software.

The entire database contains a total of 588 drillholes, including diamond, reverse circulation, and percussion drilling. There are 65 drillholes for which the assays were missing. The number of total drillholes that have assays are 523 holes. With the exception of the Bridge Zone, the drillholes are generally spaced 100 m apart with infill drilling on some sections to 50 m spacing. Bridge Zone drillhole spacing is between 25 and 60 m where the drilling intersects the mineralization.

The total number of drillholes in the database is 588. The resource area was based on 526 drillholes shown in Figure 14.1. The mineralization envelope is based on a 1% Pb+Zn threshold. This encompasses 357 drillholes as show in Table 14.2. The grade estimation is based on the composites of the remaining 305 drillholes for Pegmont Zones 1 - 5, 46 drillholes for BHZ, and 15 drillholes for the Bridge Zone. Note some drillholes intersect mineralization in different zones.

Table 14.2 is a summary of the drillholes used in the Mineral Resource estimation, with the year drilled as currently registered in the database as of 15 April 2018.

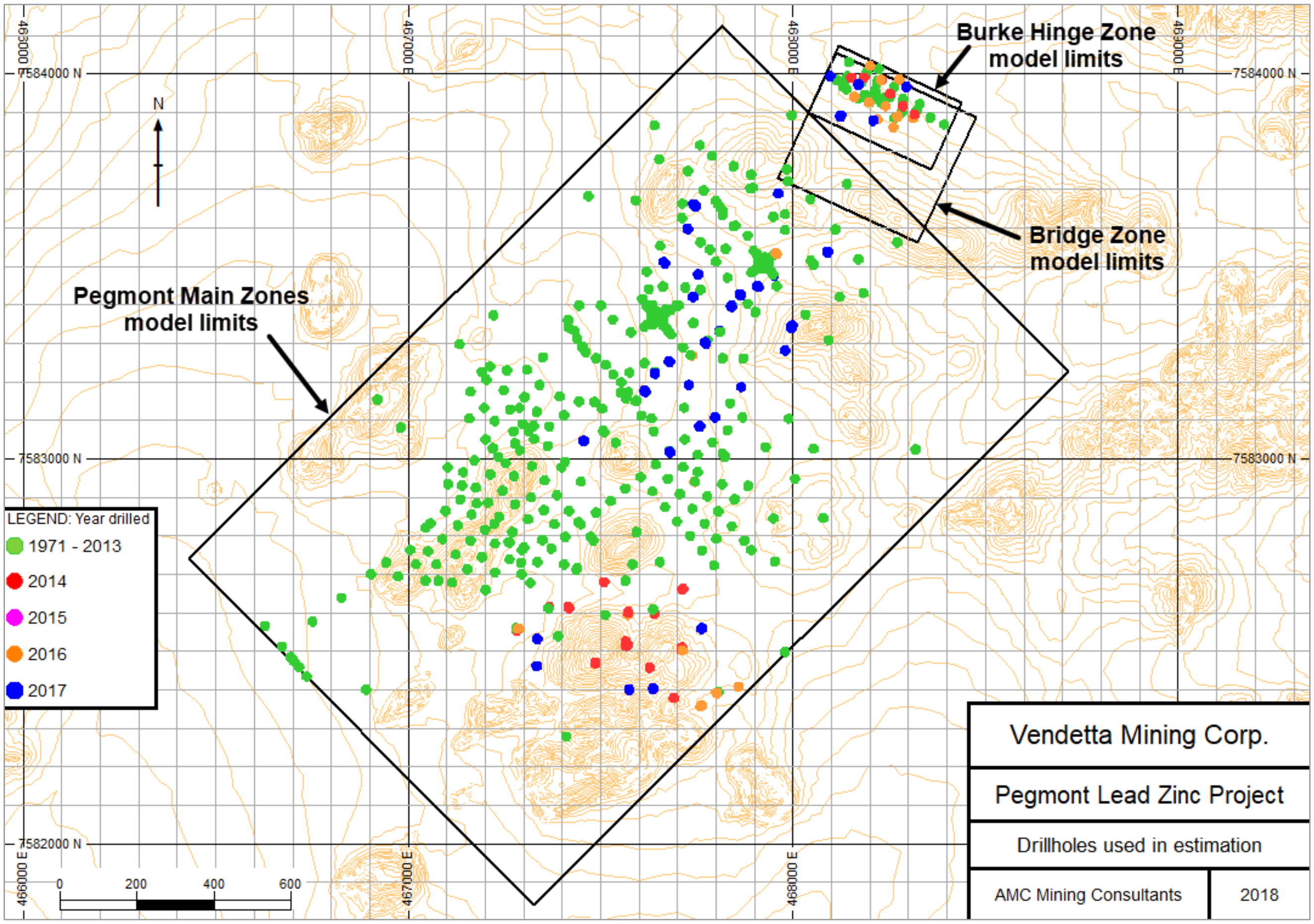
Table 14.2 Summary of drillholes used in modelling

Year	Number of drillholes used in estimation	Drilling meters
1971	39	2,836.36
1975	5	661.90
1996	26	2,473.30
1998	2	294.20
2000	2	532.00
2005	11	719.00
2008	80	10,443.70
2009	14	3,309.10
2010	39	2,383.00
2013	11	1,431.00
2014	21	4,478.02
2016	27	5,551.00
2017	80	16,424.65
Total	**357**	**51,537.23**

Note: The type of drilling per year is shown in Table 10.1. The 1971 and 1975 drilling were by rotary and percussion methods and associated Mineral Resources were classified as Inferred.

Figure 14.1 shows the location of the collars of drillholes used in the resource modelling.

Figure 14.1 Plan view of the Pegmont drillholes used in interpretation



Source: AMC

14.2.2 Bulk density

Blocks are assigned bulk density values according to oxidation state and grades for different zones. The bulk density values are derived from measurements on drill core using the wax coated immersion method.

The mean bulk density values for sulphides, transitions, and oxides types of mineralized material measurement are given in Table 14.3 to Table 14.7. Densities vary based on zone, rock type, percentage of mineralization, and oxidation states (Vendetta, 2018[1]). The bulk density for the cross-cutting amphibolite dyke has been assigned as 2.99 t/m^3.

[1] Voulgaris, P, (2018). Bulk Density Determinations and Summary Update, 6 May 2018. Unpublished report from Vendetta Mining Corp.

Table 14.3 Bulk density BHZ hangingwall and footwall lens

Zn + Pb (%)	Material type	SG (t/m³)
<1%		2.44*
1-3%	Hangingwall / oxide	2.50
>=3%		3.21
<1%		3.08
1-3%	MW / transition	3.19
>=3%		3.60
1%		3.12
1-3%	SW & Sulphide	3.47
>=3%		3.99

Notes:
*No samples so used Zone 1 Oxide <1%.
MW=moderate weathering, SW=strong weathering.

Table 14.4 Bulk density Bridge Zone – Lens B

Zn + Pb (%)	Material type	SG (t/m³)
1%		3.08
1-5%	Sulphide	3.53
>=5%		4.19

Table 14.5 Bulk density Pegmont Zone 1 – Lens B

Zn + Pb (%)	Material type	SG (t/m³)
<1%		2.44
1-3%*	Oxide	2.73
>=3%		
<1%		2.84
1-3%	Transition	3.14
>=3%		3.22
<1%		3.00
1-3%	SW & Sulphide	3.11
>=3%		3.92

Notes:
*1 sample only, 1-3% and >3% combined.
SW= strong weathering.

Table 14.6 Bulk density Pegmont Zone 2 and Zone 3 (Lens B)

Zn + Pb (%)	Material type	Zone 2 (Lens B)	Zone 3 (Lens B)
		SG (t/m³)	SG (t/m³)
<1%		3.01	2.67
1-3%	Transition	3.04	3.43**
>=3%		3.55	3.43
<1%		3.08	3.05
1-3%	Sulphide	3.27	3.45
>=3%		3.99	4.03

Notes:
*Statistically insignificant populations; used Zone 1 Transition.
**No samples; used >3% range.

Table 14.7 Bulk density Zones 4 and 5

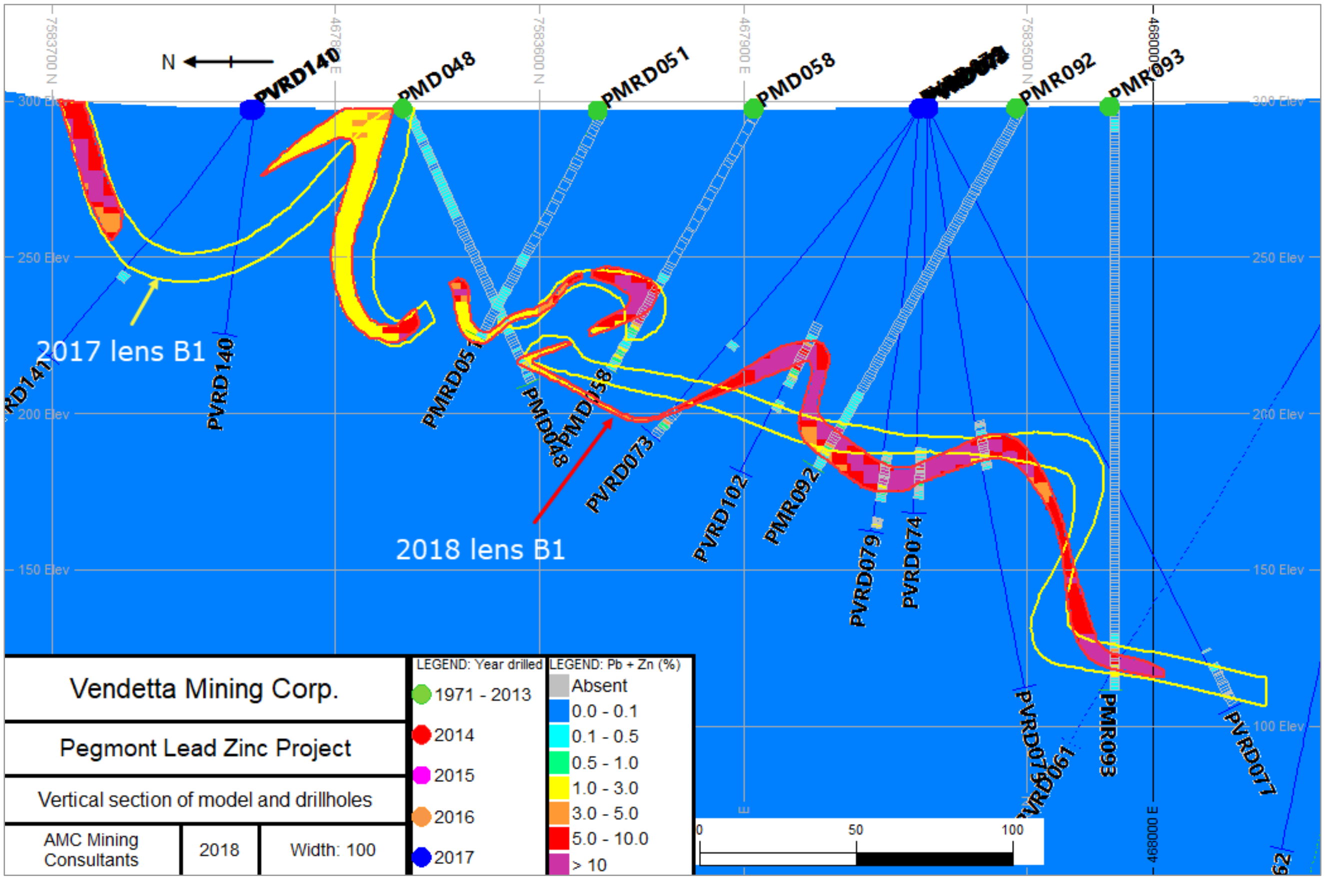
Zn + Pb (%)	Material type	Zone 4 (Lens B)	Zone 5 (all lenses)
		SG (t/m³)	SG (t/m³)
<1%	Sulphide	3.00	3.01
1-5%		3.28	3.32
>=5%		4.08	3.91

14.3 Geology and mineralization interpretation

Lead-zinc-silver mineralization on the Property is stratiform and is interpreted to have formed initially as a syngenetic deposit in a fault-bounded marine sub-basin. Galena and sphalerite mineralization are closely associated with an iron-rich sedimentary unit referred to as "iron formation", interbedded with micaceous schist and amphibolite. The host sequence comprises a 2,000 m thick unit of thickly bedded feldspathic psammites, quartzites, poorly-bedded pelites, and amphibolite, with minor marble, calc-silicates, and BIFs.

The mineralization extends approximately 2 km north-east and 1 km south-west. It is known to extend to approximately 350 m below surface, and the higher-grade mineralization is around 4 to 6 m thick. The mineralization comprises the Main Zone and the smaller BHZ area, which is a gossan area in the north-east part of the Property, as well as the high-grade Bridge Zone which is located between Main Zone and BHZ. The Main Zone features recumbent folds overprinted by upright folds. The mineralization interpretation has been slightly changed based on new drilling results. Figure 14.2 is a NW-SE vertical section of the Main Zone illustrating the new block model, drillholes, and modified folded morphology of the mineralization. Note the new interpretation is shown in solid colour and the 2017 interpretation in a yellow outline.

Figure 14.2 NW-SE Vertical Section of the Pegmont mineralization (Lens B1)



Notes: Previous interpretation in yellow outline. New interpretation coloured by PB + Zn grade in the block model.
Source: AMC

The interpretation of the mineralized domains and the oxidation surface was provided by VTT and was reviewed and approved by AMC. The mineralized domains were based on a threshold of 1% Pb+Zn. For BHZ and Bridge Zone a background envelope of nominally 0.2% Pb+Zn ("low grade envelope") was also provided. A 3D model of the cross-cutting, post-mineral amphibolite dyke was superimposed on the model and assigned a zero grade. The five Pegmont Zones numbered 1 – 5 were delineated in the previous estimate and the naming has been kept intact for reporting and comparison purposes. The Zones are shown in Figure 14.3.

There are ten mineralized domains in BHZ. This includes six background domains and four 1% envelope domains. The Bridge Zone has one background domain and one 1% envelope domain.

There are 14 mineralized domains in Pegmont Zones 1 – 5. The largest domains are domain 11 (Lens B1) and domain 14 (Lens B4). These mineralization domains are shown in Figure 14.3. Note that the red wireframes are the mineralized domains and the coloured underlay shows the location of the 5 zones in plan.

Table 14.8 BHZ estimation domain coding

BHZ mineralization	Domain	
	High grade	Low grade
Hangingwall	101, 103	111
Hangingwall	102	112
Hangingwall	NA	211, 213, 214
Footwall	301	311

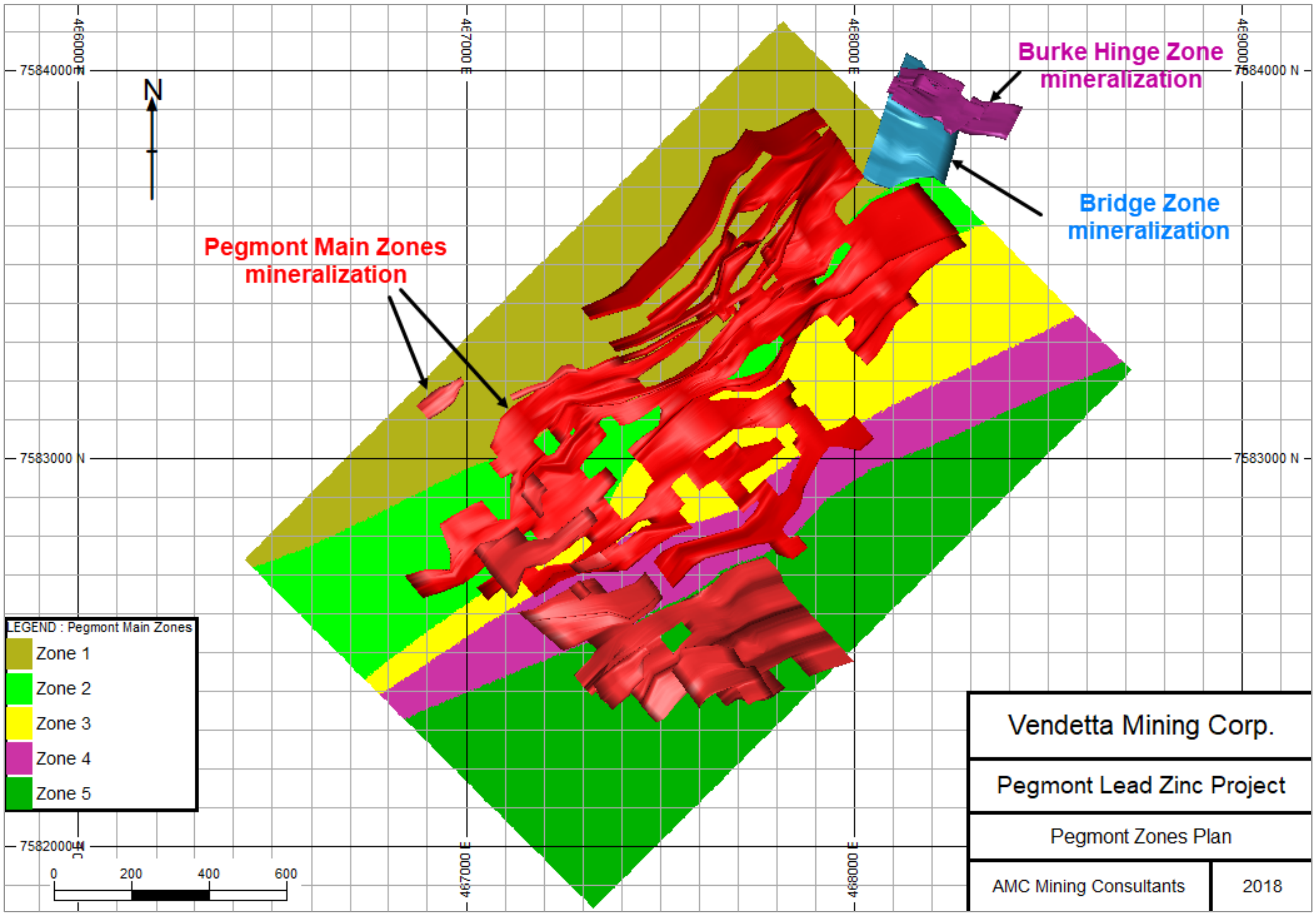
Table 14.9 Pegmont Zones 1 – 5 estimation domain coding

Pegmont Zones 1 – 5 mineralization	Domain
Lens B1	11
Lens B2	12
Lens B3	13
Lens B4	14
Lens C1	21
Lens C2	22
Lens C3	23
Lens C4	24
Lens C5	25
Lens C6	31
Lens C7	32
Lens C8	33
Lens C9	34
Lens C10	35

Table 14.10 Bridge Zones estimation domain coding

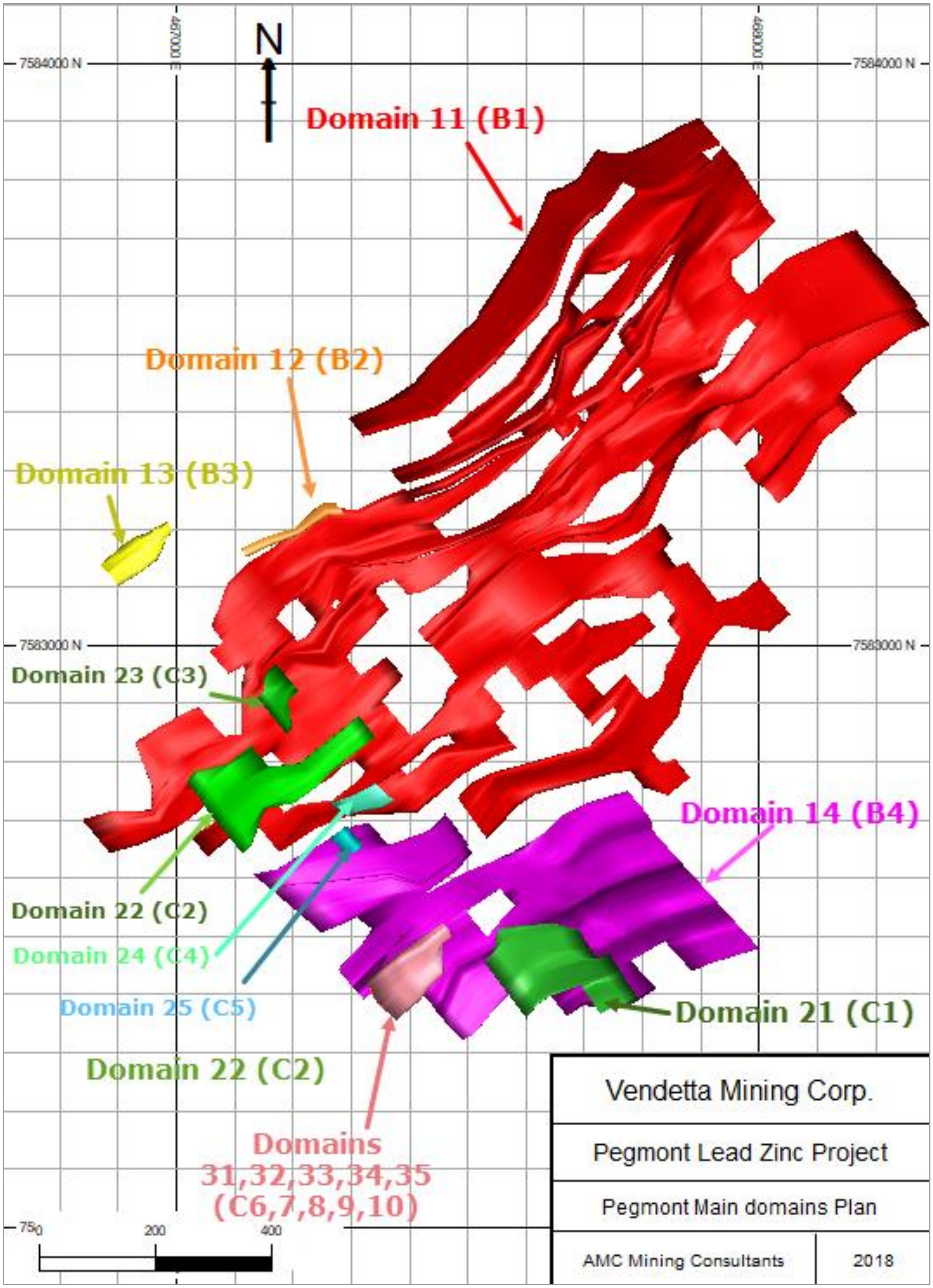
Bridge Zones	Domain
Low grade envelope	511
Mineralization domain	501

Figure 14.3 Plan view of the Pegmont mineralization zones



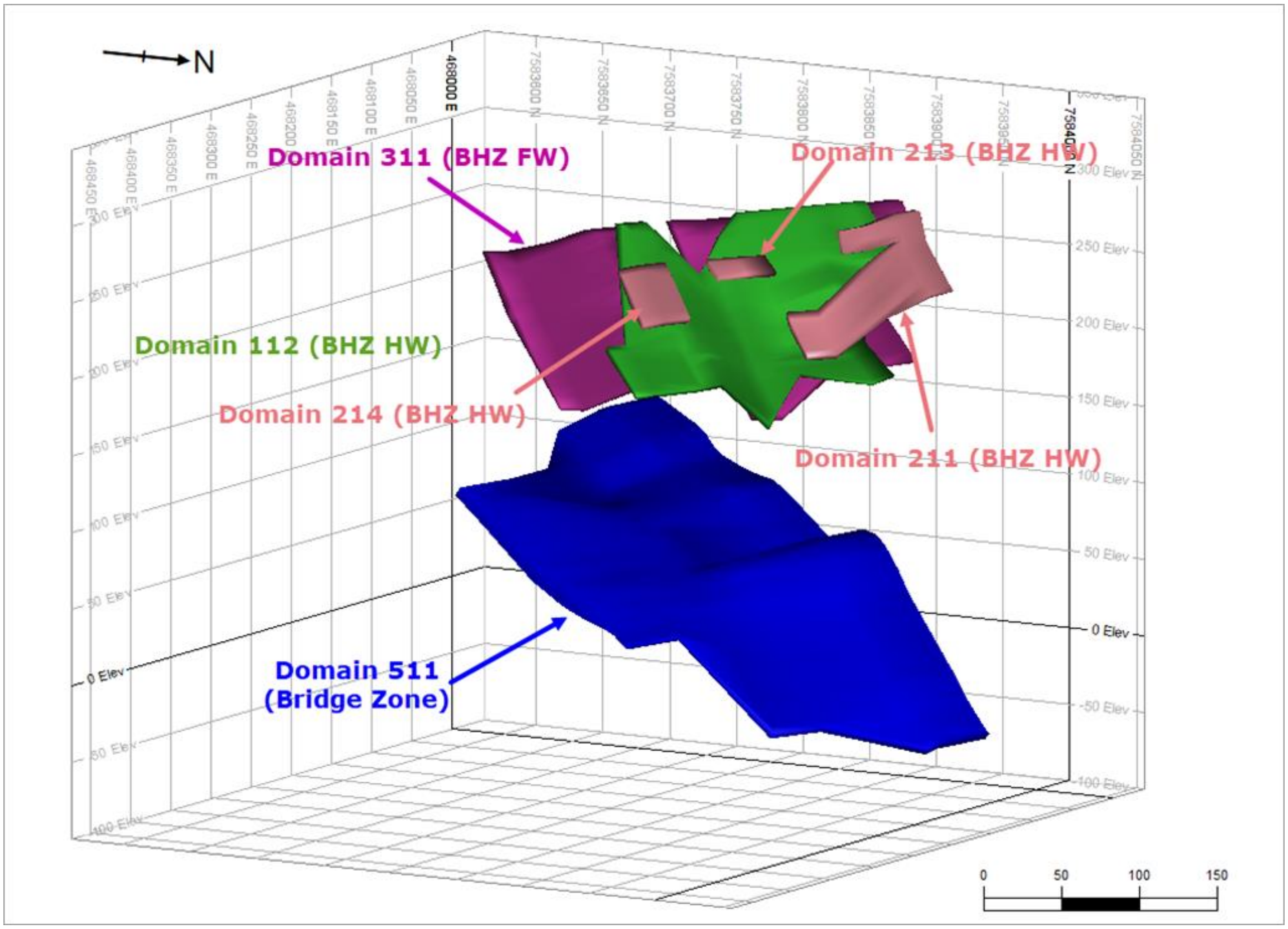
Source: AMC

Figure 14.4 Plan view of the Pegmont Zones 1 – 5 domains



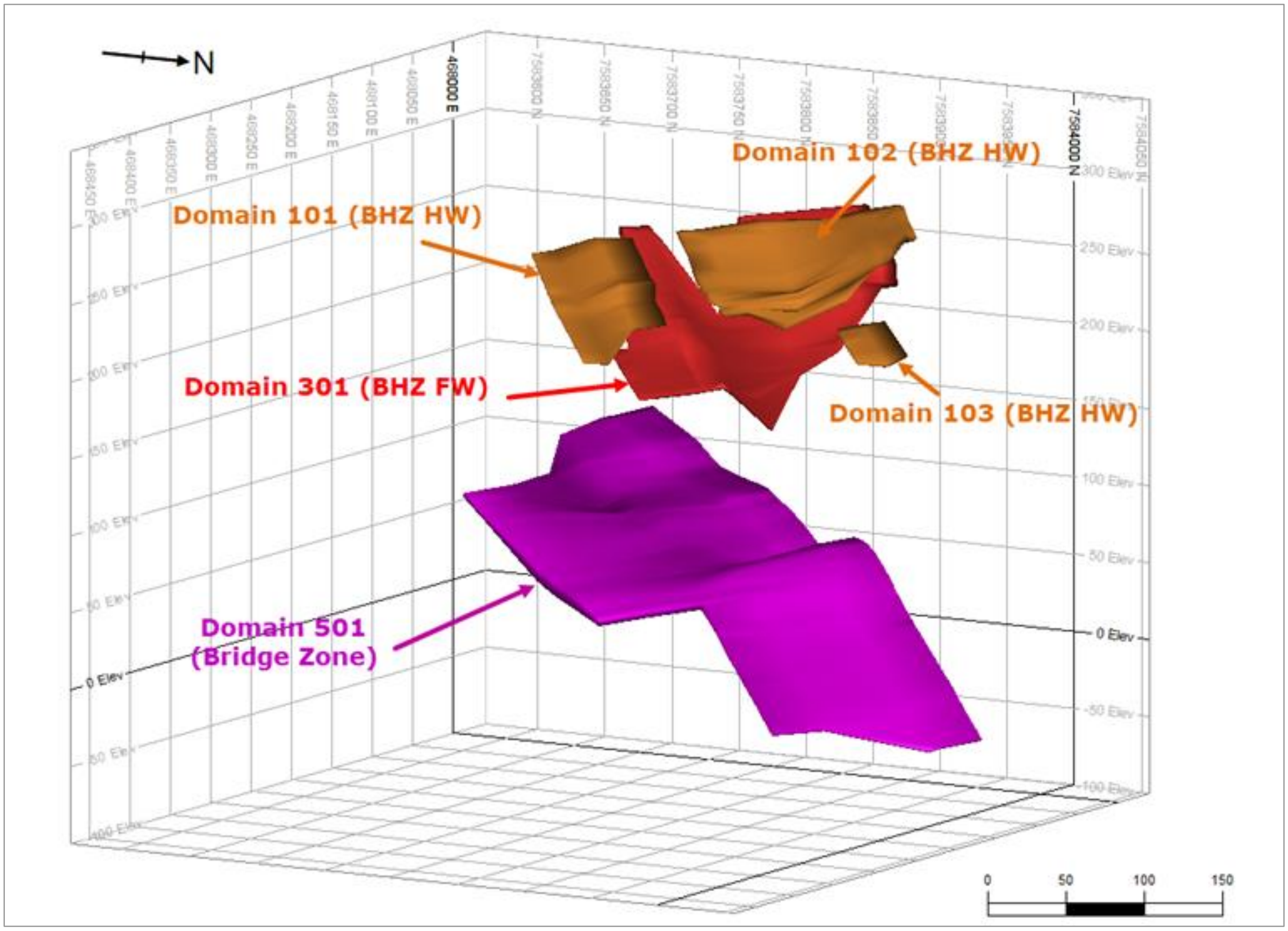
Note: The letter and number combinations (example "C2") are the original names provided by VTT. For Resource estimation purposes, AMC converted them to numeric names, and these are referred to in the rest of the text.
Source: AMC

Figure 14.5 3D view of BHZ and Bridge Zone geological domains



Note: Diagram on the right shows low-grade domains.
Source: AMC

Figure 14.6 3D view of BHZ and Bridge Zone mineralization domains



Note: Diagram on the left shows 1% envelope domains.
Source: AMC

14.4 Burke Hinge Zone

14.4.1 Statistics

14.4.1.1 Statistics of raw data

Statistics of the samples selected within the BHZ domains are presented in Table 14.11.

Table 14.11 Statistics for raw BHZ samples

Field	Pb (%)									
Domain	**101**	**102**	**103**	**111**	**112**	**211**	**213**	**214**	**301**	**311**
Nsamples	193	23	6	239	51	58	5	9	106	91
Minimum	0.03	0.05	0.22	0.01	0.01	0.01	0.01	0.03	0.01	0.01
Maximum	18.00	14.70	6.93	5.88	3.58	4.57	0.42	0.45	25.10	3.68
Mean	**4.02**	**2.01**	**2.19**	**0.26**	**0.27**	**0.50**	**0.15**	**0.16**	**4.88**	**0.33**
Standdev	3.76	3.25	2.91	0.56	0.61	1.00	0.15	0.13	4.86	0.48
Coeff. of Var.	0.94	1.62	1.33	2.18	2.23	2.01	1.00	0.82	0.99	1.46
Field	**Zn (%)**									
Domain	**101**	**102**	**103**	**111**	**112**	**211**	**213**	**214**	**301**	**311**
Nsamples	193	23	6	239	51	58	5	9	106	91
Minimum	0.09	0.06	0.86	0.02	0.02	0.07	0.07	0.11	0.07	0.03
Maximum	12.90	7.86	4.43	4.85	1.88	3.76	1.12	0.72	16.95	6.99
Mean	**2.11**	**2.83**	**2.50**	**0.30**	**0.24**	**0.61**	**0.44**	**0.32**	**3.06**	**0.32**
Standdev	1.87	2.30	1.42	0.49	0.30	0.79	0.36	0.23	2.62	0.90
Coeff. of Var.	0.88	0.81	0.57	1.63	1.26	1.29	0.82	0.70	0.86	2.81
Field	**Ag (g/t)**									
Domain	**101**	**102**	**103**	**111**	**112**	**211**	**213**	**214**	**301**	**311**
Nsamples	193	23	6	239	51	58	5	9	106	91
Minimum	0.25	0.50	1.30	0.00	0.25	0.00	1.00	0.50	0.25	0.25
Maximum	52.40	43.10	35.70	25.30	12.00	25.90	1.00	3.00	57.00	32.10
Mean	**7.24**	**8.20**	**11.12**	**1.08**	**1.09**	**1.94**	**1.00**	**1.17**	**8.27**	**1.05**
Standdev	7.24	11.94	15.06	2.53	2.08	4.54	0.00	0.78	9.32	3.46
Coeff. of Var	1.00	1.46	1.35	2.33	1.90	2.35	0.00	0.67	1.13	3.30

14.4.1.2 Statistics of composite data

Sample lengths average 1 m inside the wireframes. Assays within the wireframe domains were composited as close to 1 m intervals as possible which approximates the mean sample length. Compositing started at the first mineralized wireframe boundary from the collar and was reset at each new wireframe boundary. No capping was considered necessary.

Composite statistics for lead, zinc, and silver are shown in Table 14.12.

Table 14.12 Statistics of BHZ composites

Field	Pb (%)									
Domain	**101**	**102**	**103**	**111**	**112**	**211**	**213**	**214**	**301**	**311**
Nsamples	172	22	3	240	45	63	5	10	83	74
Minimum	0.03	0.05	0.22	0.00	0.00	0.00	0.01	0.00	0.01	0.00
Maximum	18.00	14.70	5.78	3.41	3.58	3.40	0.42	0.45	25.10	2.15
Mean	**4.01**	**2.01**	**2.19**	**0.22**	**0.26**	**0.39**	**0.15**	**0.14**	**4.88**	**0.28**
Standdev	3.74	3.25	2.54	0.42	0.59	0.77	0.15	0.13	4.62	0.39
Coeff. of Var.	0.93	1.62	1.16	1.96	2.32	1.97	1.00	0.93	0.95	1.39
Field	**Zn (%)**									
Domain	**101**	**102**	**103**	**111**	**112**	**211**	**213**	**214**	**301**	**311**
Nsamples	172	22	3	240	45	63	5	10	83	74
Minimum	0.09	0.06	0.86	0.00	0.00	0.00	0.07	0.00	0.07	0.00
Maximum	12.90	7.86	4.12	4.85	1.88	3.76	1.12	0.72	10.21	6.99
Mean	**2.10**	**2.83**	**2.50**	**0.25**	**0.22**	**0.48**	**0.44**	**0.29**	**3.06**	**0.27**
Standdev	1.86	2.30	1.33	0.42	0.29	0.70	0.36	0.24	2.36	0.82
Coeff. of Var.	0.88	0.81	0.53	1.68	1.33	1.45	0.82	0.81	0.77	2.99
Field	**Ag (g/t)**									
Domain	**101**	**102**	**103**	**111**	**112**	**211**	**213**	**214**	**301**	**311**
Nsamples	172	22	3	240	45	63	5	10	83	74
Minimum	0.25	0.50	1.30	0.00	0.00	0.00	1.00	0.00	0.25	0.00
Maximum	52.40	43.10	29.66	15.30	12.00	17.00	1.00	3.00	42.90	18.63
Mean	**7.21**	**8.20**	**11.12**	**0.90**	**1.02**	**1.51**	**1.00**	**1.05**	**8.27**	**0.90**
Standdev	7.20	11.94	13.12	1.77	2.03	3.29	0.00	0.82	8.50	2.47
Coeff. of Var	1.00	1.46	1.18	1.97	1.99	2.17	0.00	0.78	1.03	2.74

14.4.2 Block model

14.4.2.1 Block model parameters

Parent cell dimensions were chosen that were able to adequately represent the spatial distribution of grade in the folded mineralization. Parent block size was 2.5 m by 2 m by 2.5 m with sub-blocking employed. Sub-blocking resulted in minimum cell dimensions of 0.5 m x 0.4 m x 0.003 m. The block model was rotated 25° around the Z axis towards the west.

The block model parameters are shown in Table 14.13.

Table 14.13 BHZ block model parameters

Parameter	X	Y	Z
Origin	468,040	7,583,900	100
Rotation angle	0	0	25
Maximum block size	2.5	2	2.5
Minimum block size	0.5	0.4	0.003
Number of blocks	140	95	90

The block model fields are shown in Table 14.14.

Table 14.14 Block model fields

Field	Description
XC	Block's centroid X coordinate
YC	Block's centroid Y coordinate
ZC	Block's centroid Z coordinate
XINC	Cell dimension in X
YINC	Cell dimension in Y
ZINC	Cell dimension in Z
IJK	Identification number for each parent cell
DOMAIN	Individual name for mineralized domains
DENSITY	Density
NSAMP	Number of samples estimated for each block grade
PB_ID2	Estimated lead grade
ZN_ID2	Estimated zinc grade
AG_ID2	Estimated silver grade
ZN_EQV	Sum of PB_ID2 and ZN_ID2
OXIDN	Oxidation (OXIDE, PARTIAL, SULPHIDE)
RESCAT	Classification (2 - Indicated, 3- Inferred, 4- Potential)
XMORIG	X coordinate of model origin in world coordinate system
YMORIG	Y coordinate of model origin in world coordinate system
ZMORIG	Z coordinate of model origin in world coordinate system
NX	Number of cells in X direction
NY	Number of cells in Y direction
NZ	Number of cells in Z direction
X0	X coordinate of model origin in local coordinate system
Y0	Y coordinate of model origin in local coordinate system
Z0	Z coordinate of model origin in local coordinate system
ANGLE1	First rotation angle, in degrees
ANGLE2	Second rotation angle, in degrees
ANGLE3	Third rotation angle, in degrees
ROTAXIS1	Rotation axis 3 means rotation around Z axis
ROTAXIS2	Rotation axis 0 means no rotation
ROTAXIS3	Rotation axis 0 means no rotation

14.4.2.2 Grade estimation

To account for the folded nature of the deposit, the dynamic anisotropy option in Datamine was used for grade estimation for the 1% envelope domains and their corresponding lower grades envelope domains. This allowed the orientation of the search ellipsoid to be defined individually for each block in the model. Weathered surfaces were not used to constrain estimation process. Lead, zinc, and silver grades were interpolated using inverse distance to the power of two. AMC conducted high-level variographic analysis to give some guidance to the size of the search ellipsoid.

The low-grade domains in BHZ (domains 211, 213, and 214), which do not have corresponding 1% envelope counterparts, were estimated using fixed search ellipsoids. The grade estimation was completed in three passes with each successive pass using increasing search distances as shown in Table 14.15. The third pass was to fill the entire wireframes. No constraint on the maximum number of samples per drillhole was considered necessary as there were limited numbers of samples available for selection in the across-strike direction.

Table 14.15 BHZ search parameters

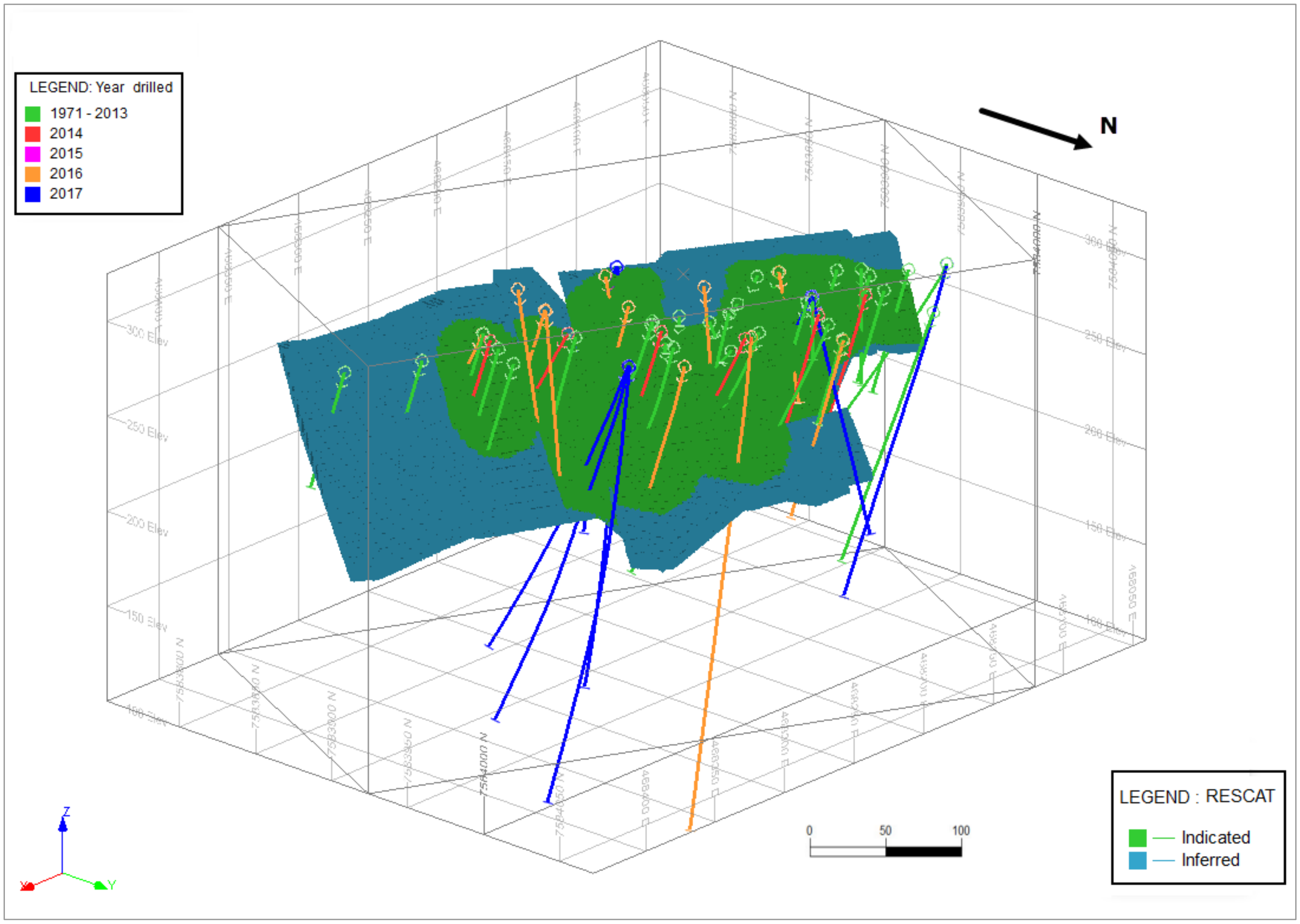
Pass	1					2					3					Dynamic anisotropy
BHZ domains	X	Y	Z	Min No. samples	Max No. samples	X	Y	Z	Min No. samples	Max No. samples	X	Y	Z	Min No. samples	Max No. samples	
101	45	45	10	3	10	90	90	20	3	10	180	180	40	1	10	yes
102	45	45	10	3	10	90	90	20	3	10	180	180	40	1	10	yes
103	50	10	30	3	10	100	20	60	3	10	200	40	120	1	10	no
111	45	45	10	3	10	90	90	20	3	10	180	180	40	1	10	yes
112	45	45	10	3	10	90	90	20	3	10	225	225	50	1	10	yes
211	50	10	30	3	10	100	20	60	3	10	200	40	120	1	10	no
213	50	10	30	3	10	100	20	60	3	10	200	40	120	1	10	no
214	50	10	30	3	10	100	20	60	3	10	200	40	120	1	10	no
301	45	45	10	3	10	90	90	20	3	10	180	180	40	1	10	yes
311	45	50	10	3	10	90	100	20	3	10	270	300	60	1	10	yes

14.4.2.3 Mineral Resource classification

Because of drilling density and geological mapping, the level of confidence in the geological and mineralization continuity, data quality, and data density allows classification of both Indicated and Inferred Mineral Resources. Wireframes were generated manually for defining the Indicated and Inferred Resource categories.

The Mineral Resource classification is presented in Figure 14.7 where green is Indicated Mineral Resources and blue is Inferred Mineral Resources.

Figure 14.7 BHZ Mineral Resource classification

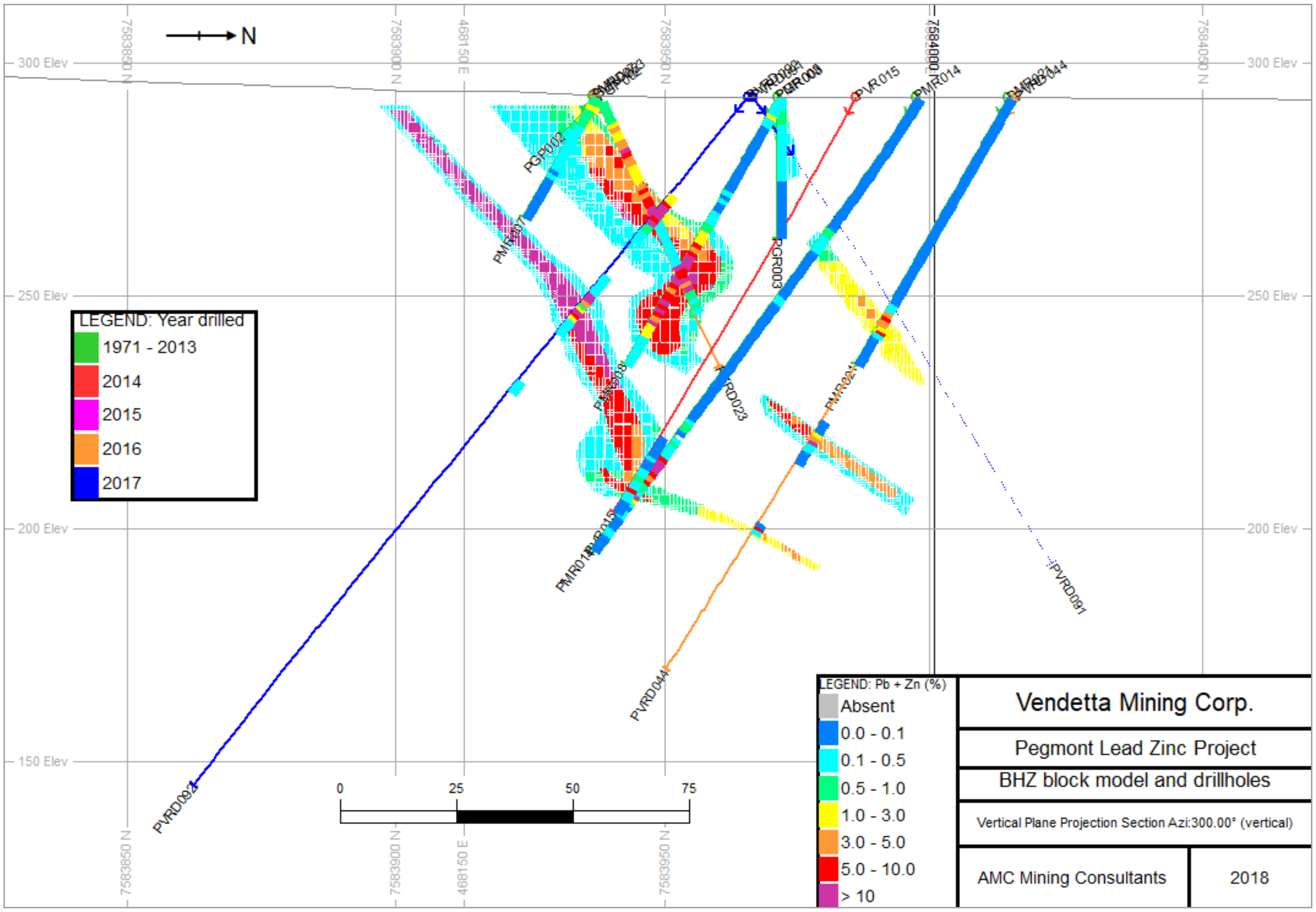


Source: AMC

14.4.2.4 BHZ block model validation

The block model was validated by visual checks, statistics, and swath plots. Visual checks were carried out to ensure that the estimated grades were consistent with the drillhole grades and to check that the estimated grade distribution was consistent with the style of mineralization. Figure 14.8 shows a cross-section through the BHZ block model.

Figure 14.8 SW-NE cross-section with drillhole and BHZ block model grades



Note: The section is constructed using a clipping distance of 12.5 m each side of the section line.
Source: AMC

Table 14.16 to Table 14.18 show the lead, zinc, and silver statistical comparison of the composite grades versus block model grades by domain. Generally, the agreement is reasonable.

Table 14.16 BHZ block model versus composite Pb (%) statistics

Data	Domain	N samples	Minimum	Maximum	Mean	Coeff. of Var
Model	101	176060	0.10	11.55	3.81	0.51
Composite		172	0.03	18.00	4.01	0.93
Model	102	85821	0.22	8.33	2.46	0.42
Composite		22	0.05	14.70	2.01	1.62
Model	103	15861	0.71	3.99	2.18	0.19
Composite		3	0.22	5.78	2.19	1.16
Model	111	410014	0.00	2.13	0.33	1.19
Composite		240	0.00	3.41	0.22	1.96
Model	112	286213	0.01	1.56	0.26	0.81
Composite		45	0.00	3.58	0.26	2.32
Model	211	92870	0.00	1.80	0.56	0.84
Composite		63	0.00	3.40	0.39	1.97
Model	213	11084	0.04	0.25	0.15	0.24
Composite		5	0.01	0.42	0.15	1.00
Model	214	26240	0.05	0.31	0.12	0.43
Composite		10	0.00	0.45	0.14	0.93
Model	301	296072	0.50	18.42	5.25	0.53
Composite		83	0.01	25.10	4.88	0.95
Model	311	370073	0.00	1.96	0.26	0.96
Composite		74	0.00	2.15	0.28	1.39

Table 14.17 BHZ block model versus composite Zn (%) statistics

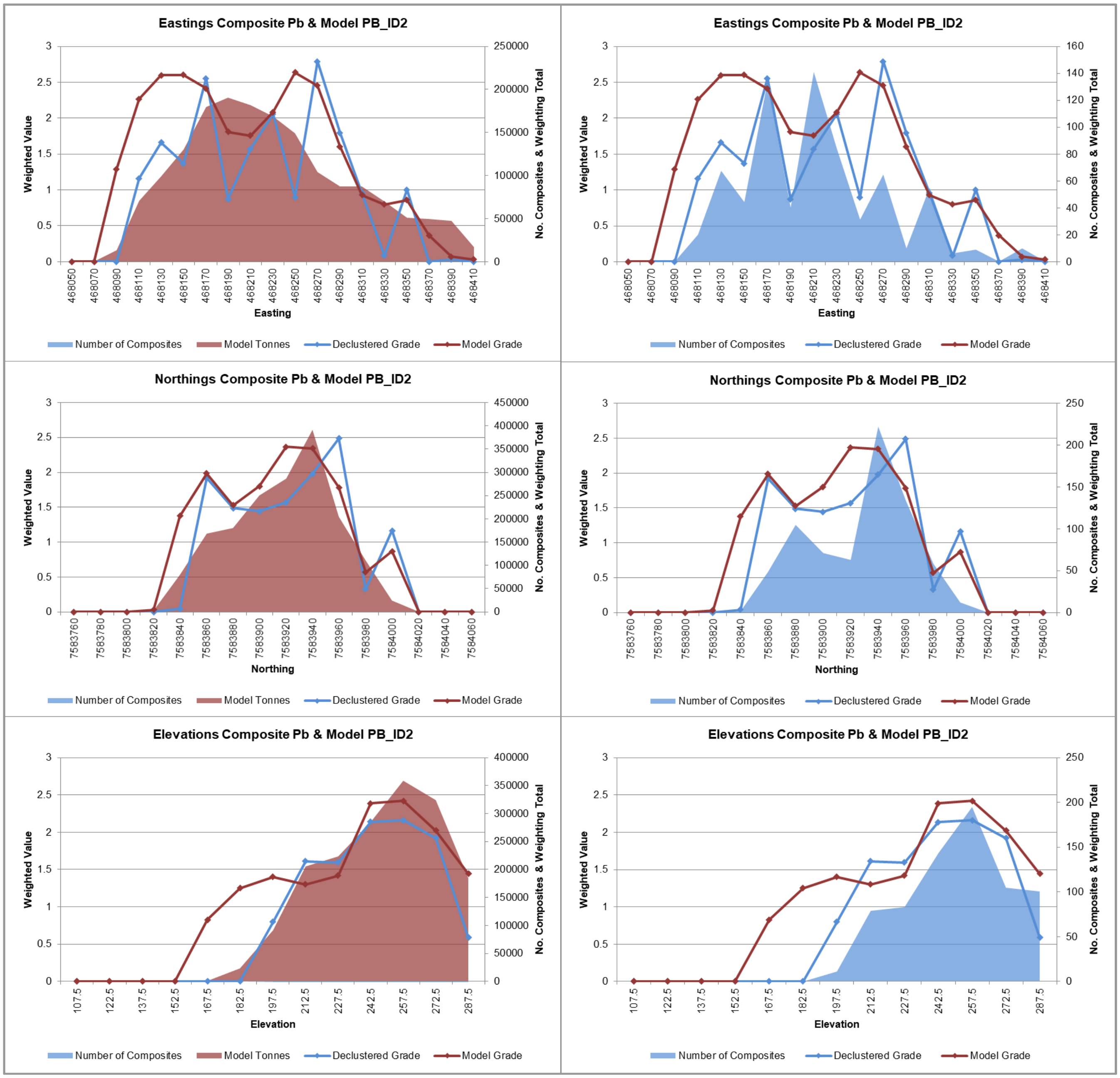
Data	Domain	N samples	Minimum	Maximum	Mean	Coeff. of Var
Model	101	176060	0.36	7.55	2.09	0.47
Composite		172	0.09	12.90	2.10	0.88
Model	102	85821	0.81	4.70	2.62	0.21
Composite		22	0.06	7.86	2.83	0.81
Model	103	15861	1.52	3.40	2.50	0.10
Composite		3	0.86	4.12	2.50	0.53
Model	111	410014	0.00	1.43	0.34	0.89
Composite		240	0.00	4.85	0.25	1.68
Model	112	286213	0.01	0.88	0.21	0.63
Composite		45	0.00	1.88	0.22	1.33
Model	211	92870	0.00	7.97	2.54	0.83
Composite		63	0.00	3.76	3.76	1.45
Model	213	11084	0.20	0.61	0.44	0.12
Composite		5	0.07	0.72	0.44	0.82
Model	214	26240	0.08	0.58	0.23	0.46
Composite		10	0.00	0.72	0.24	0.81
Model	301	296072	0.37	6.63	3.19	0.26
Composite		83	0.07	10.21	3.06	0.77
Model	311	370073	0.00	6.19	0.30	1.90
Composite		74	0.00	6.99	0.27	2.99

Table 14.18 BHZ block model versus composite Ag (g/t) statistics

Data	Domain	N samples	Minimum	Maximum	Mean	Coeff. of Var
Model	101	176060	0.81	25.16	6.48	0.48
Composite		172	0.25	52.40	7.21	1.00
Model	102	85821	1.15	36.20	10.61	0.41
Composite		22	0.50	43.10	8.20	1.46
Model	103	15861	3.65	20.39	10.61	0.19
Composite		3	1.30	29.66	11.12	1.18
Model	111	410014	0.00	9.22	1.52	1.23
Composite		240	0.00	15.30	0.90	1.97
Model	112	286213	0.17	5.48	1.06	0.69
Composite		45	0.00	12.00	1.02	1.99
Model	211	92870	0.00	7.97	1.06	0.69
Composite		63	0.00	17.00	1.51	2.17
Model	213	11084	1.00	1.00	1.00	0.00
Composite		5	1.00	1.00	1.00	0.00
Model	214	26240	0.47	2.37	0.91	0.35
Composite		10	0.00	3.00	1.05	0.78
Model	301	296072	0.76	31.22	9.21	0.51
Composite		83	0.25	42.90	8.27	1.03
Model	311	370073	0.00	16.93	0.91	1.94
Composite		74	0.00	18.63	0.90	2.74

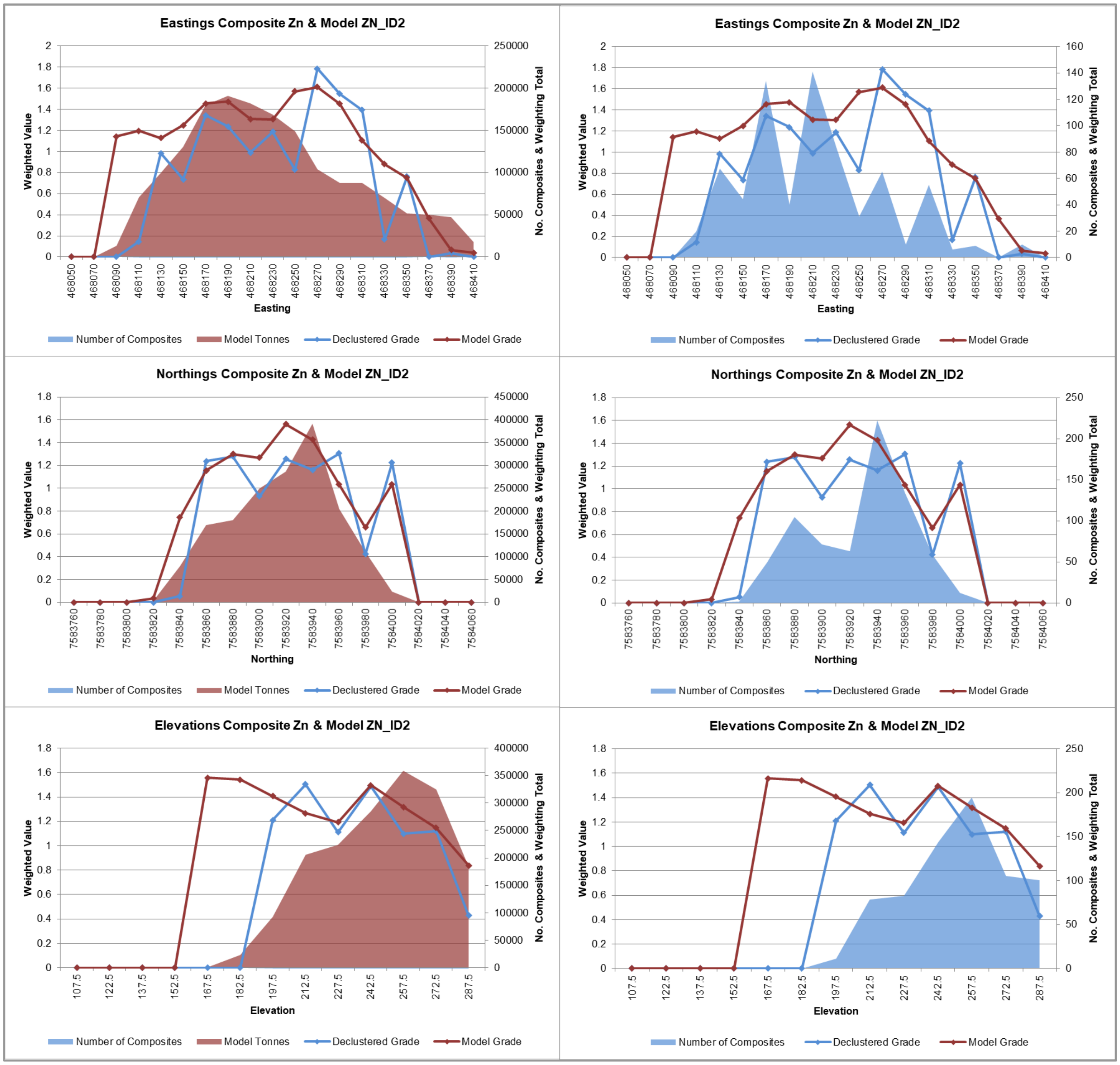
Swath plots were generated to visually compare the composites and block model grade statistics. Block model grades for lead and zinc were weighted by tonnes and are shown Figure 14.9 and Figure 14.10 respectively. The swath plots show good agreement between the composite and estimated block model grades although it should be noted that these average results may mask variations on a domain by domain basis.

Figure 14.9 Swath plots for lead



Note: Weighted value = grade; No. weighting total = tonnes.
Source: AMC

Figure 14.10 Swath plots for zinc



Note: Weighted value = grade; No. weighting total = tonnes.
Source: AMC

14.4.3 Mineral Resource estimate

The Mineral Resource estimate for the BHZ deposit as of 31 July 2018 at a 3% lead plus zinc cut-off grade is stated in Table 14.19.

This cut-off grade was provided by VTT and is based on reported operating costs and metallurgical recoveries at proximal operating mines with similar styles of mineralization. AMC has reviewed the work and considers that the cut-off grades are reasonable.

Table 14.19 BHZ Mineral Resources at 31 July 2018

Zone	Classification	Material type	Tonnes (kt)	Pb (%)	Zn (%)	Ag (g/t)
Open pit	Indicated	Transition	72	4.2	2.2	7
		Sulphide	286	4.7	3.0	9
		Sub total	**357**	**4.6**	**2.9**	**8**
	Inferred	Transition	5	3.9	2.0	6
		Sulphide	125	6.3	3.1	11
		Sub total	**131**	**6.2**	**3.1**	**11**

Notes:
1. CIM Definition Standards (2014) were used to report the Mineral Resources.
2. Cut-off grade applied to the open pit Mineral Resources is 3% Pb+Zn.
3. Based on the following metal prices: US$0.95/lb for Pb, US$1.05/lb for Zn, and US$16.5/oz for silver.
4. Exchange rate of US$0.75:A$1.0.
5. Metallurgical recoveries for the BHZ Zone as follows:
   - Lead to lead concentrate: 80.6% for transition and 88.0% for sulphide.
   - Zinc to zinc concentrate: 19.3% for transition and 78.5% for sulphide.
6. Using drilling results up to 15 April 2018.
7. Mineral Resource tonnages have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

AMC is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other similar factors that could materially affect the stated Mineral Resource estimates.

An explanation of the economic considerations for the deposits is provided in Section 14.7.

14.5 Bridge Zone

14.5.1 Statistics

14.5.1.1 Statistics of raw data

Statistics of the samples selected within the Bridge Zone are presented in Table 14.20.

14.5.1.2 Statistics of composite data

Sample lengths average 1 m inside the wireframes. Assays within the wireframe domains were composited as close to 1 m intervals as possible which approximates the mean sample length. Compositing started at the first mineralized wireframe boundary from the collar and was reset at each new wireframe boundary. No capping was considered necessary.

Composite statistics for lead, zinc, and silver are shown in Table 14.20.

Table 14.20 Statistics for raw and composites of mineralized Bridge Zone domain 511

Domain	Field	Pb (%)		Zn (%)		Ag (g/t)	
	Data	Samples	Composites	Samples	Composites	Samples	Composites
501	Nsamples	135	82	135	82	135	82
	Minimum	0.20	0.44	0.11	0.11	0.25	0.27
	Maximum	17.60	17.43	9.25	8.22	49.60	37.96
	Mean	**9.13**	**9.13**	**2.63**	**2.63**	**14.25**	**14.25**
	Standdev	4.68	4.30	1.84	1.71	8.67	7.72
	Coeff. of Var.	0.51	0.47	0.70	0.65	0.61	0.54
511	Nsamples	115	68	115	68	115	68
	Minimum	0.03	0.03	0.05	0.05	0.25	0.25
	Maximum	6.72	6.72	4.62	1.27	12.10	12.10
	Mean	**0.72**	**0.72**	**0.21**	**0.21**	**1.62**	**1.62**
	Standdev	1.21	1.17	0.20	0.19	2.63	2.58
	Coeff. of Var.	1.68	1.63	0.94	0.88	1.63	1.59

14.5.2 Block model

14.5.2.1 Block model parameters

Parent cell dimensions were chosen that were able to adequately represent the spatial distribution of grade in the folded mineralization. Parent block size was 2.5 m by 2 m by 2.5 m with sub-blocking employed. Sub-blocking resulted in minimum cell dimensions of 0.5 m x 0.4 m x 0.003 m. The block model was rotated 25° around the Z axis towards the west.

The block model parameters are shown in Table 14.21.

Table 14.21 Bridge Zone block model parameters

Parameter	X	Y	Z
Origin	467,960	7,583,730	-100
Rotation angle	0	0	25
Maximum block size	2.5	2	2.5
Minimum block size	0.5	0.4	0.003
Number of blocks	160	180	180

The block model fields are shown in Table 14.14.

14.5.2.2 Grade estimation

To account for the folded nature of the deposit, the dynamic anisotropy option in Datamine was used for grade estimation for the 1% envelope domain and its corresponding lower grades envelope domain. This allowed the orientation of the search ellipsoid to be defined individually for each block in the model. Lead, zinc, and silver grades were interpolated using inverse distance to the power of two. AMC conducted high-level variographic analysis to give some guidance to the size of the search ellipsoid.

The grade estimation was completed in three passes with each successive pass using increasing search distances as shown in Table 14.22. The third pass was to fill the entire wireframes. A constraint on the maximum number of samples per drillhole was set to two samples per drillhole.

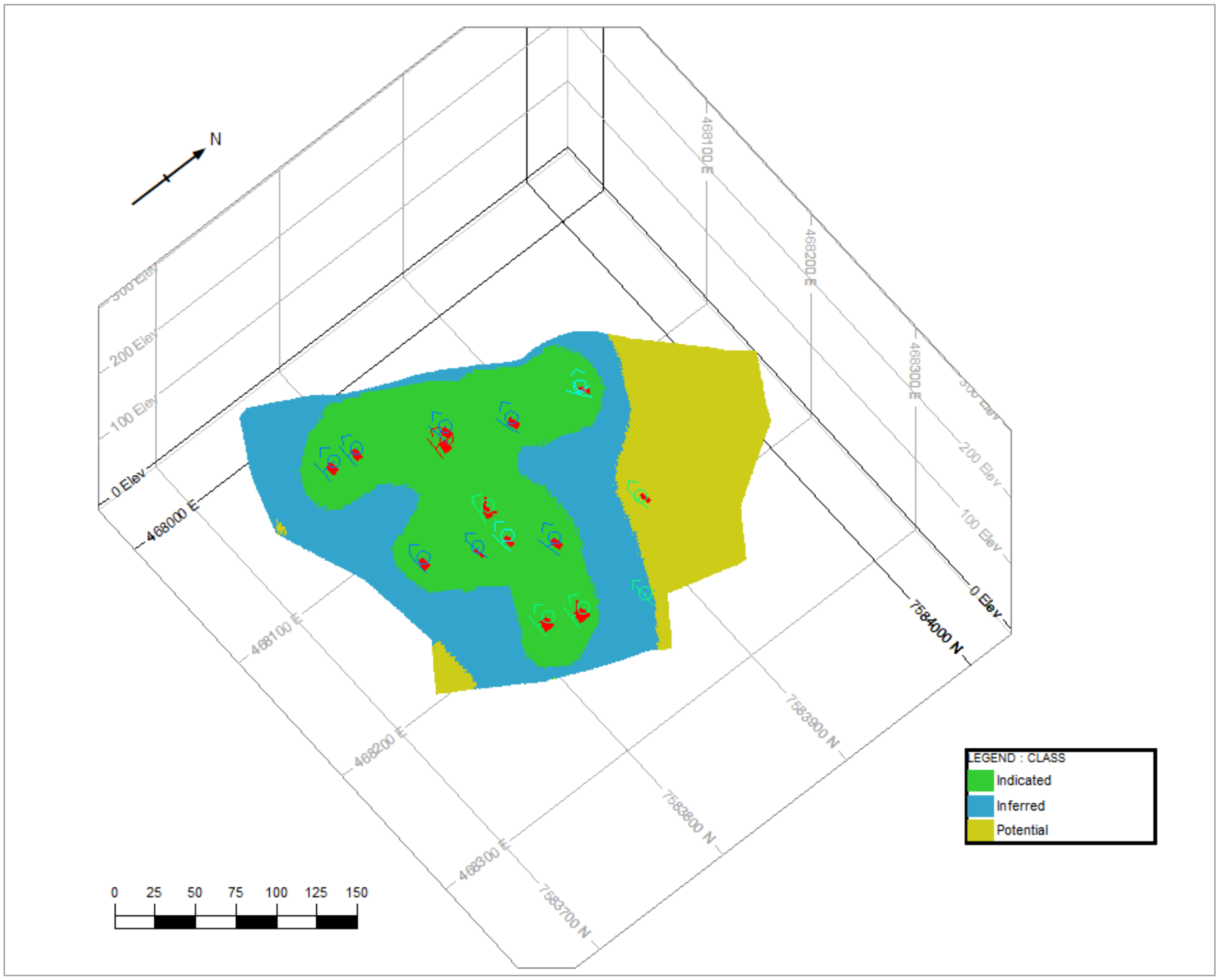
Table 14.22 Bridge Zone search parameters

Bridge domains	X	Y	Z	Min no. samples	Max no. samples	Dynamic anisotropy
Pass				**1**		
501	100	125	10	6	10	yes
511	100	125	10	6	10	yes
Pass				**2**		
501	200	250	20	6	10	yes
511	200	250	20	6	10	yes
Pass				**3**		
501	400	500	40	4	10	yes
511	400	500	40	1	10	yes

14.5.2.3 Mineral Resource classification

The Mineral Resource classification for the Bridge Zone is presented in Figure 14.11 where green is Indicated Mineral Resources and blue is Inferred Mineral Resources. Wireframes were generated manually for defining the Indicated and Inferred Resource categories.

Figure 14.11 Bridge Zone Mineral Resource classification



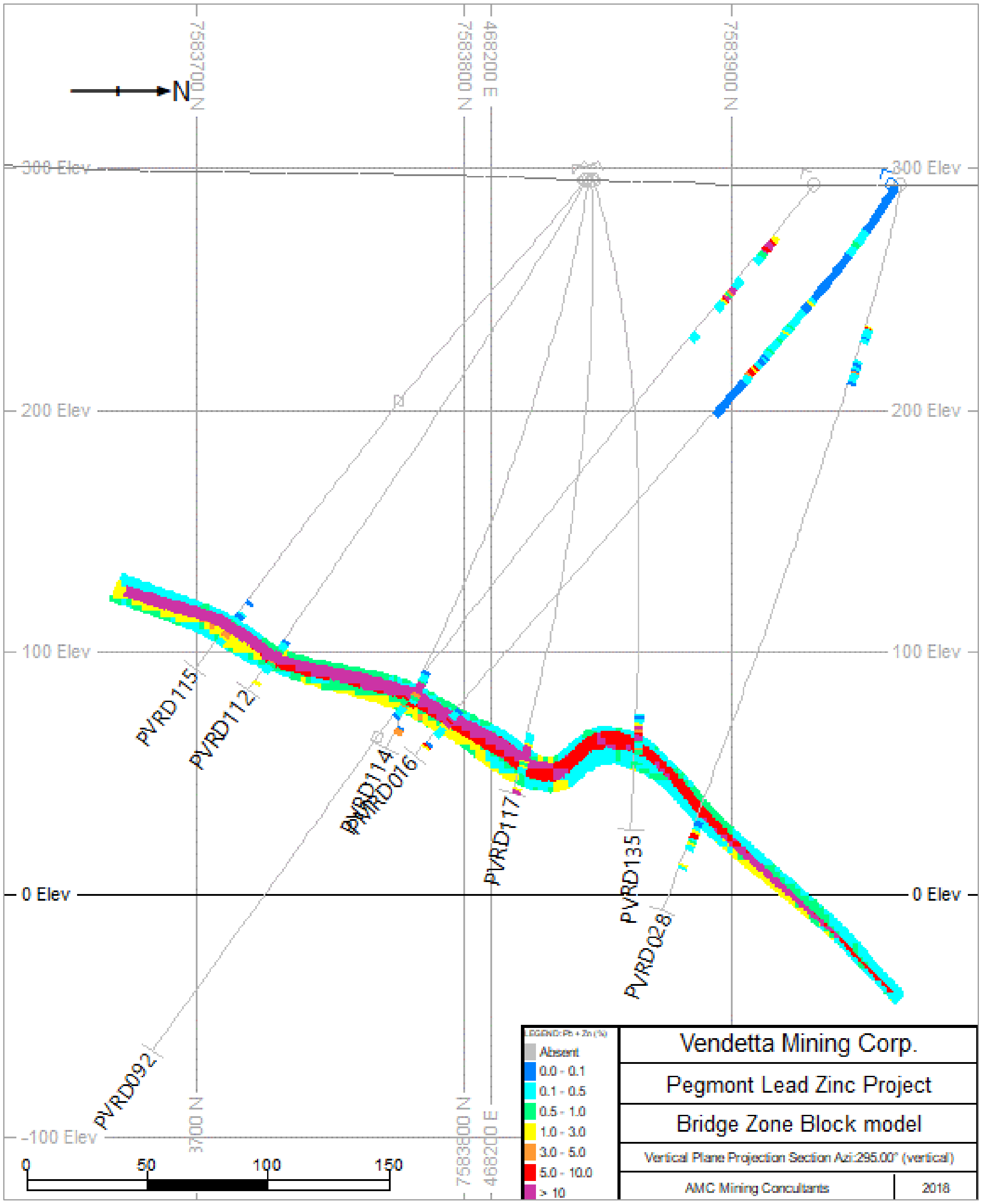
Source: AMC

14.5.2.4 Bridge Zone block model validation

The block model was validated by visual checks, statistics, and swath plots. Visual checks were carried out to ensure that the estimated grades were consistent with the drillhole grades and to check that the estimated grade distribution was consistent with the style of mineralization.

Figure 14.12 shows a cross-section through the Bridge Zone block model.

Figure 14.12 SW-NE cross-section with drillhole and Bridge Zone block model grades



Note: The section is constructed using a clipping distance of 50 m each side of the section line. Mineralization near surface is the BHZ Zone.
Source: AMC

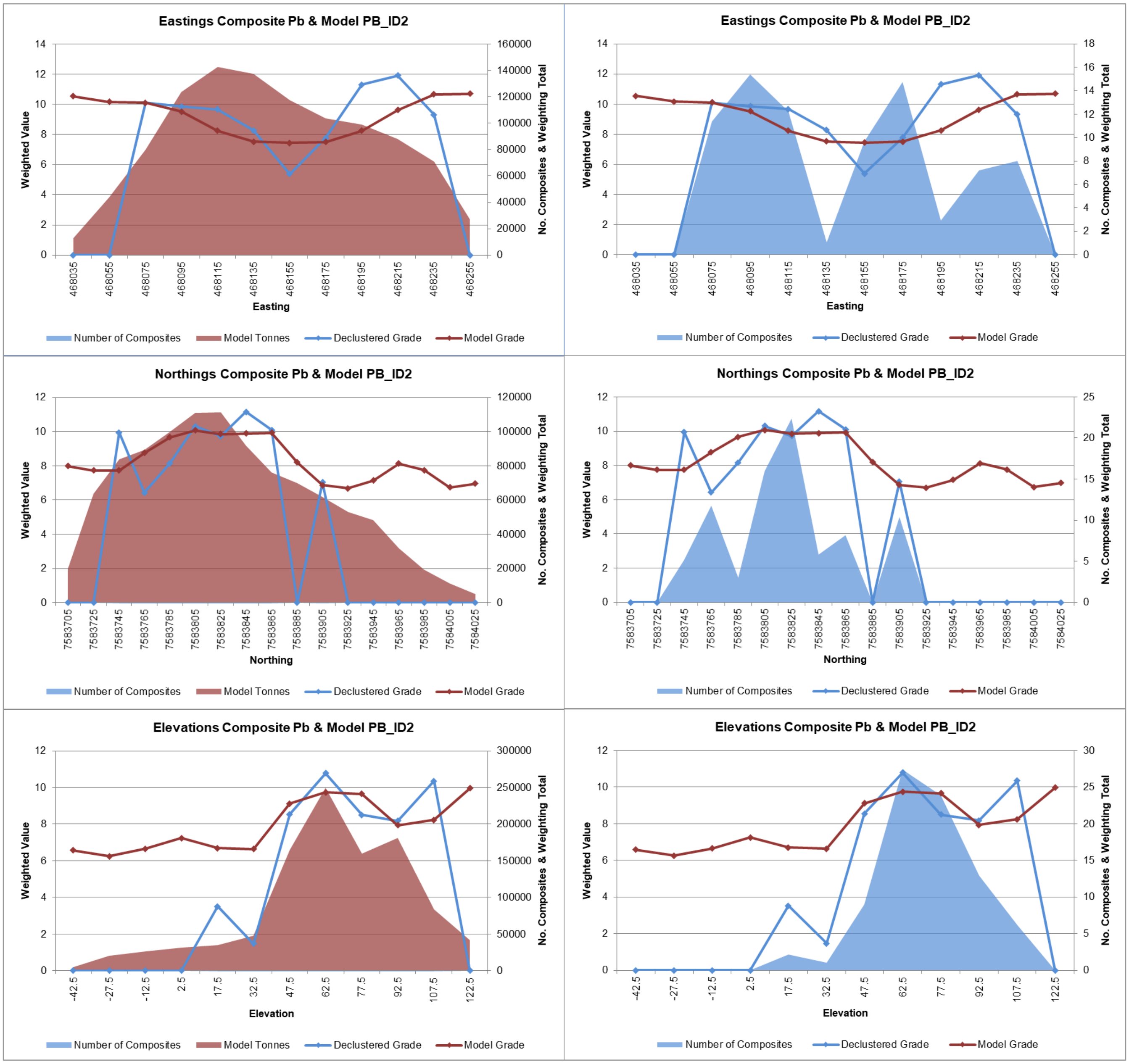
Table 14.23 shows the lead, zinc, and silver statistical comparison of the composite grades versus block model grades by domain. Generally, the agreement is reasonable.

Table 14.23 Bridge Zone block model versus composite statistics

Domain	Field	Pb (%)		Zn (%)		Ag (g/t)	
	Data	Composites	Model	Composites	Model	Composites	Model
501	Nsamples	82	704,102	82	704,102	82	704,102
	Minimum	0.44	0.61	0.11	0.63	0.27	2.49
	Maximum	17.43	15.44	8.22	6.53	37.96	37.06
	Mean	**9.13**	**8.74**	**2.63**	**2.61**	**14.25**	**13.74**
	Standdev	4.30	2.53	1.71	0.93	7.72	3.65
	Coeff. of Var.	0.47	0.29	0.65	0.36	0.54	0.27
511	Nsamples	68	706,504	68	706,504	68	706,504
	Minimum	0.03	0.04	0.05	0.07	0.25	0.25
	Maximum	6.72	3.91	1.27	0.80	12.10	7.46
	Mean	**0.72**	**0.64**	**0.21**	**0.20**	**1.62**	**1.41**
	Standdev	1.17	0.53	0.19	0.08	2.58	1.27
	Coeff. of Var.	1.63	0.84	0.88	0.38	1.59	0.90

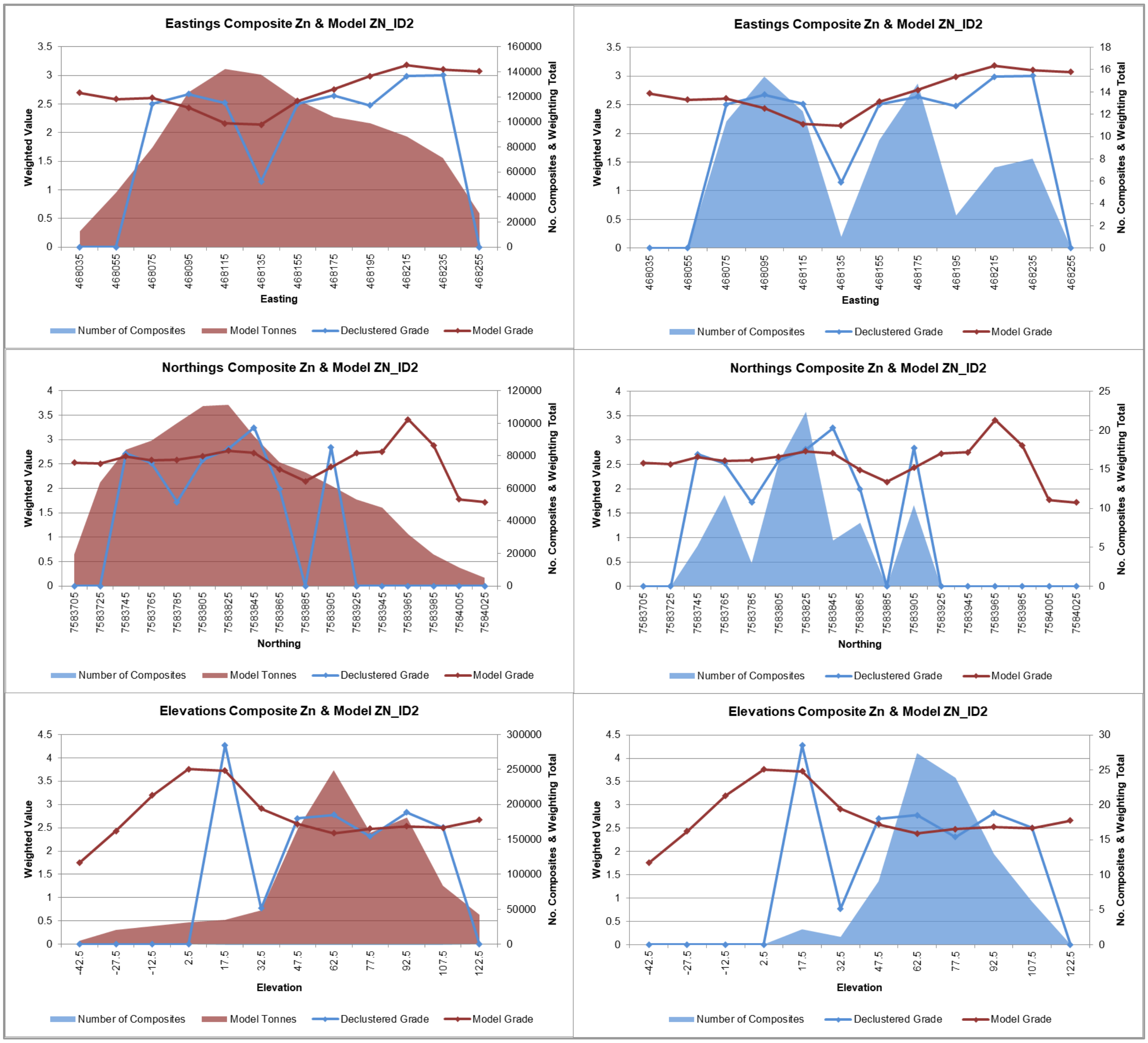
Swath plots were generated to visually compare the composites and block model grade statistics. Block model grades for lead and zinc were weighted by tonnes and are shown Figure 14.13 and Figure 14.14 respectively. The swath plots show good agreement between the composite and estimated block model grades although it should be noted that these average results may mask variations on a domain by domain basis.

Figure 14.13 Swath plots for lead



Note: Weighted value = grade; No. weighting total = tonnes.
Source: AMC

Figure 14.14 Swath plots for zinc



Note: Weighted value = grade; No. weighting total = tonnes.
Source: AMC

14.5.3 Mineral Resource estimate

The Mineral Resource estimate for the Bridge Zone deposit as of 31 July 2018 at a 5% lead plus zinc cut-off grade is stated in Table 14.24.

This cut-off grade was provided by VTT and is based on reported operating costs and metallurgical recoveries at proximal operating mines with similar styles of mineralization. AMC has reviewed the work and considers that the cut-off grades are reasonable.

Table 14.24 Bridge Zone Mineral Resources at 31 July 2018

Classification	Material type	Tonnes (kt)	Pb (%)	Zn (%)	Ag (g/t)
Indicated	Sulphide	560	9.5	2.5	15
Inferred	Sulphide	309	8.7	2.5	14

Notes:
1. CIM Definition Standards (2014) were used to report the Mineral Resources.
2. Cut-off grade applied to the underground Mineral Resources is 5% Pb+Zn.
4. Exchange rate of US$0.75:A$1.0.
5. Metallurgical recoveries are not applied for the Bridge Zone
6. Using drilling results up to 15 April 2018.
7. Mineral Resource tonnages have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

AMC is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other similar factors that could materially affect the stated Mineral Resource estimates.

An explanation of the economic considerations for the deposits is provided in Section 14.7.

14.6 Pegmont

14.6.1 Statistics

14.6.1.1 Statistics of raw data

Statistics of the samples selected within the Pegmont Zones 1 – 5 domains are presented in Table 14.25.

Table 14.25 Statistics for Pegmont Zones 1 - 5 raw samples

Field	Domain	Nsamples	Minimum	Maximum	Mean	Standdev	Coeff. of Var
Pb (%)	11	2079	0.005	27.8	**5.89**	4.44	0.75
	12	10	0.965	9.73	**5.97**	3.59	0.60
	13	6	0.910	12.9	**5.45**	4.11	0.75
	14	341	0.001	20.6	**3.82**	3.54	0.93
	21	63	0.000	12.6	**3.21**	2.95	0.92
	22	13	0.091	1.83	**0.82**	0.52	0.63
	23	2	0.481	0.5	**0.49**	0.01	0.02
	24	2	1.220	1.28	**1.25**	0.03	0.02
	25	3	3.730	7.4	**6.00**	1.44	0.24
	31	8	0.037	6.09	**2.88**	2.12	0.74
	32	11	0.032	11.9	**3.63**	4.71	1.30
	33	8	1.235	6.57	**3.92**	1.90	0.48
	34	21	0.014	8.27	**2.22**	2.60	1.17
	35	12	0.036	1.335	**0.81**	0.29	0.36
Zn (%)	11	2079	0.010	27.4	**2.51**	2.55	1.02
	12	10	0.572	5.87	**2.71**	1.93	0.71
	13	6	0.062	0.661	**0.27**	0.20	0.75
	14	341	0.005	11.5	**3.57**	2.99	0.84
	21	64	0.063	11.75	**4.18**	2.91	0.70
	22	13	0.274	2.55	**1.28**	0.72	0.56
	23	2	0.592	0.89	**0.77**	0.15	0.19

Field	Domain	Nsamples	Minimum	Maximum	Mean	Standdev	Coeff. of Var
	24	2	0.429	2	**1.21**	0.79	0.65
	25	3	5.600	7.12	**6.46**	0.69	0.11
	31	8	0.383	10.05	**6.04**	3.57	0.59
	32	11	0.123	9.73	**5.78**	2.60	0.45
	33	8	0.585	10.35	**6.98**	3.95	0.57
	34	21	0.193	15.45	**7.28**	4.41	0.61
	35	12	0.052	3.08	**1.96**	0.98	0.50
Ag (g/t)	11	2048	0.000	64.3	**9.63**	8.36	0.87
	12	10	0.250	16.8	**8.15**	5.97	0.73
	13	6	0.250	2.1	**0.80**	0.62	0.78
	14	340	0.000	36.6	**5.42**	5.13	0.95
	21	64	0.250	21.6	**4.51**	3.96	0.88
	22	13	0.250	7.9	**3.92**	2.27	0.58
	23	2	0.500	2.02	**1.42**	0.74	0.52
	24	2	2.000	4.9	**3.45**	1.45	0.42
	25	3	5.700	9.8	**8.36**	1.62	0.19
	31	8	0.250	10	**4.92**	3.47	0.71
	32	11	0.250	16.8	**5.28**	6.58	1.25
	33	8	0.900	8.7	**4.92**	2.79	0.57
	34	21	0.250	9.2	**3.06**	2.82	0.92
	35	12	0.250	6.5	**4.02**	1.39	0.35

14.6.1.2 Statistics of composite data

Sample lengths averaged 1 m inside the wireframes. Assays within the wireframe domains were composited as close to 1 m intervals as possible which approximates the mean sample length. Compositing started at the first mineralized wireframe boundary from the collar and was reset at each new wireframe boundary. No capping was considered necessary.

Composite statistics for lead, zinc, and silver are shown in Table 14.26. The compositing increased the total number of samples from 2,559 to 2,244 because of splitting of large samples.

Table 14.26 Statistics of Pegmont Zones 1 – 5 composites

Field	Domain	Nsamples	Minimum	Maximum	Mean	Standdev	Coeff. of Var
Pb (%)	11	1875	0.000	25.44	**5.72**	4.37	0.77
	12	12	0.965	9.73	**5.97**	3.40	0.57
	13	6	0.910	12.90	**5.45**	4.11	0.75
	14	259	0.001	19.92	**3.82**	3.45	0.90
	21	35	0.000	11.63	**3.21**	2.75	0.86
	22	13	0.091	1.83	**0.82**	0.52	0.63
	23	3	0.481	0.50	**0.49**	0.01	0.02
	24	2	1.220	1.28	**1.25**	0.03	0.02
	25	3	4.594	7.32	**6.00**	1.12	0.19
	31	6	0.037	5.91	**2.88**	2.12	0.73
	32	6	0.036	11.90	**3.63**	4.68	1.29
	33	4	1.235	6.44	**3.92**	1.88	0.48
	34	15	0.014	7.16	**2.23**	2.44	1.09
	35	5	0.521	1.20	**0.76**	0.22	0.29
Zn (%)	11	1875	0.000	23.28	**2.44**	2.41	0.99
	12	12	0.572	5.87	**2.71**	1.79	0.66
	13	6	0.062	0.66	**0.27**	0.20	0.75
	14	259	0.005	11.10	**3.57**	2.92	0.82
	21	36	0.128	11.58	**4.17**	2.69	0.64
	22	13	0.274	2.55	**1.28**	0.72	0.56
	23	3	0.592	0.89	**0.77**	0.15	0.19
	24	2	0.429	2.00	**1.21**	0.79	0.65
	25	3	5.678	6.93	**6.46**	0.55	0.09
	31	6	0.435	10.05	**6.04**	3.57	0.59
	32	6	1.573	8.14	**5.78**	2.27	0.39
	33	4	0.585	10.27	**6.98**	3.95	0.57
	34	15	0.526	14.83	**7.32**	3.97	0.54
	35	5	0.263	3.08	**1.94**	0.96	0.50
Ag (g/t)	11	1844	0.000	64.30	**9.35**	8.16	0.87
	12	12	0.250	16.80	**8.15**	5.48	0.67
	13	6	0.250	2.10	**0.80**	0.62	0.78
	14	255	0.000	36.60	**5.42**	5.01	0.92
	21	36	0.250	20.18	**4.51**	3.73	0.83
	22	13	0.250	7.90	**3.92**	2.27	0.58
	23	3	0.500	2.02	**1.42**	0.74	0.52
	24	2	2.000	4.90	**3.45**	1.45	0.42
	25	3	6.429	9.80	**8.36**	1.42	0.17
	31	6	0.250	9.66	**4.92**	3.43	0.70
	32	6	0.250	16.80	**5.28**	6.54	1.24
	33	4	0.900	8.50	**4.92**	2.76	0.56
	34	15	0.250	8.10	**3.08**	2.62	0.85
	35	5	2.700	6.28	**3.90**	1.17	0.30

14.6.2 Block model

14.6.2.1 Block model parameters

Parent cell dimensions were chosen that were able to adequately represent the spatial distribution of grade in the folded mineralization. For grade estimation, the parent block size was 5 m by 5 m by 2 m. In order to minimize the total records after creating the waste material blocks the block size was changed to 5 m by 5 m by 5 m as shown in Table 14.27.

The block model was rotated 45° around the z axis towards the east.

Table 14.27 Pegmont Zones 1 - 5 block model parameters

Parameter	X	Y	Z
Origin	466,430	7,582,740	-190
Rotation angle	0	0	45
Maximum block size	5	5	5
Minimum block size	1	1	0.5
Number of blocks	254	392	104

The block model fields are shown in Table 14.14.

14.6.2.2 Grade estimation

To account for the folded nature of the deposit, the dynamic anisotropy option in Datamine was used for grade estimation for the 1% envelope domains. This allowed the orientation of the search ellipsoid to be defined individually for each block in the model. Weathered surfaces were not used to constrain estimation process. Lead, zinc, and silver grades were interpolated using inverse distance to the power of two. AMC conducted high-level variographic analysis to give some guidance to the size of the search ellipsoid.

The grade estimation was completed in three passes with each successive pass using increasing search distances as shown in Table 14.28. The third pass was to fill the entire wireframes. The maximum number of samples per drillhole was constrained on a domain by domain basis. Details for each domain are shown in Table 14.28.

Table 14.28 Pegmont Zones 1 - 5 search parameters

Pass	1					2					3					MaxKEY
Pegmont domains	X	Y	Z	Min No. samples	Max No. samples	X	Y	Z	Min No. samples	Max No. samples	X	Y	Z	Min No. samples	Max No. samples	
11	60	60	30	6	16	120	120	60	4	12	360	360	180	2	12	2
12	60	60	30	6	12	120	120	60	4	12	300	300	150	2	12	2
13	50	50	20	6	12	100	100	40	4	12	250	250	100	2	10	0
14	50	50	20	6	12	100	100	40	4	12	250	250	100	2	10	0
21	50	50	20	4	12	100	100	40	4	12	200	200	80	2	10	3
22	50	50	20	3	12	100	100	40	2	12	200	200	80	1	12	2
23	50	50	20	3	12	100	100	40	2	12	250	250	100	1	12	2
24	50	50	20	1	12	100	100	40	1	12	250	250	100	1	12	1
25	60	60	30	4	12	120	120	60	4	12	300	300	150	1	12	3
31	60	60	30	4	12	120	120	60	4	12	300	300	150	1	12	3
32	60	60	30	4	12	120	120	60	4	12	300	300	150	1	12	3
33	60	60	30	4	12	120	120	60	4	12	300	300	150	1	12	3
34	60	60	30	4	12	120	120	60	4	12	300	300	150	1	12	3
35	60	60	30	4	12	120	120	60	4	12	300	300	150	1	12	3

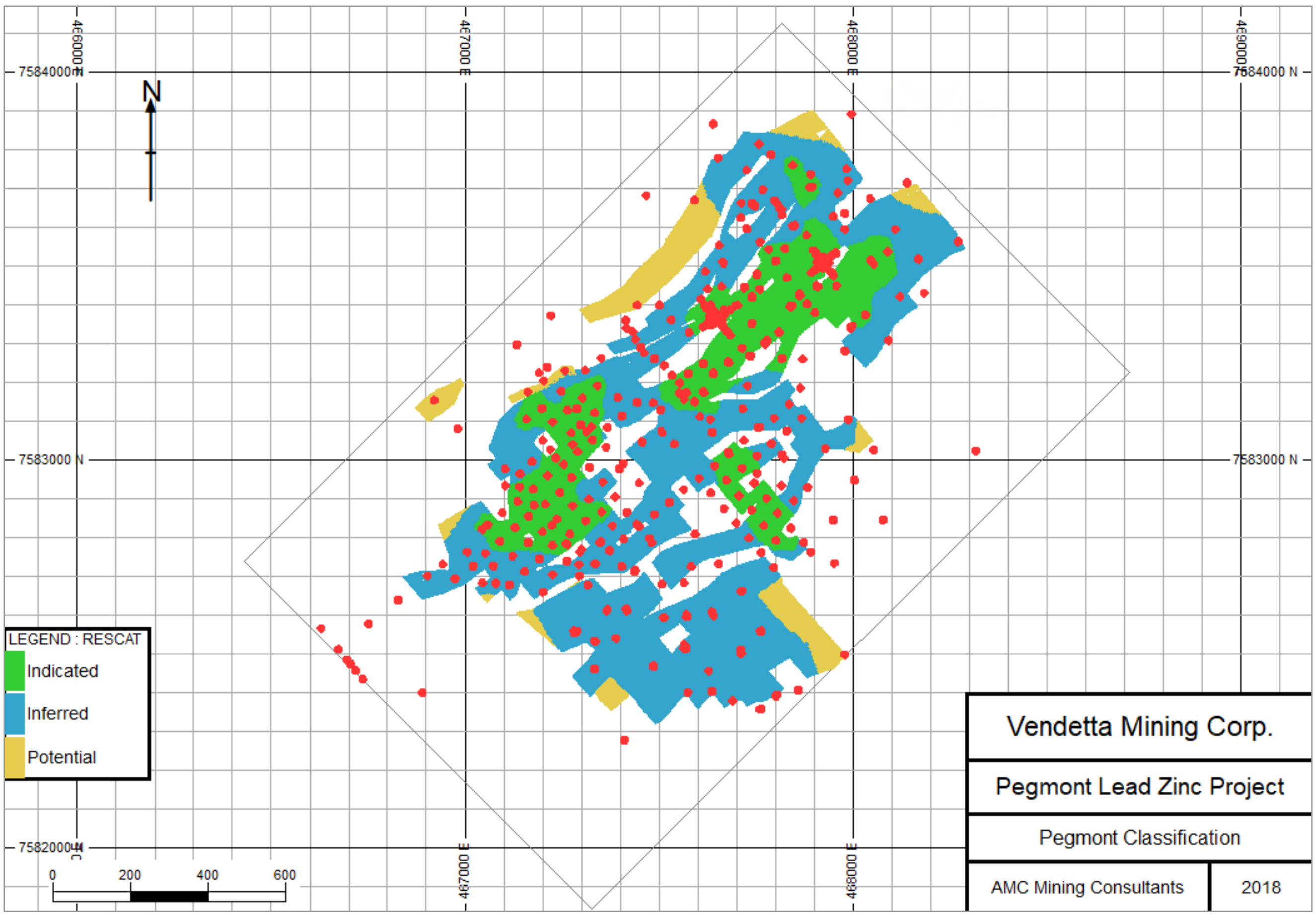
Note: MaxKEY controls the maximum number of samples per drillhole.

14.6.2.3 Mineral Resource classification

Classification was carried out using data support as the main criteria. Distance to the nearest sample was estimated into each block, which was then used to manually generate contiguous 3D wireframes defining the Indicated and Inferred Resource categories. These wireframes were then used to code the block model. The drillhole spacing and the confidence in the quality of the data is not sufficient at this stage to classify any part of the Mineral Resource as Measured.

The Mineral Resource classification is presented in Figure 14.15 where green is Indicated Mineral Resources and blue is Inferred Mineral Resources. Unclassified material (potential) is yellow.

Figure 14.15 Pegmont Zones 1 – 5 Mineral Resource classification



14.6.2.4 Block model validation

The block model was validated by visual checks, statistics, and swath plots. Visual checks were carried out to ensure that the estimated grades were consistent with the drillhole grades and to check that the estimated grade distribution was consistent with the style of mineralization. Figure 14.15 shows a cross-section through the Pegmont Zones 1 – 5 block model.

Table 14.29 shows the lead, zinc, and silver statistical comparison of the composite grades versus block model grades by domain. Generally, the agreement is reasonable. Overall the grades in the model are lower than the grades in the composites.

Table 14.29 Pegmont Zones 1 – 5 block model versus composite Pb (%) statistics

Data	Domain	N samples	Minimum	Maximum	Mean	Standdev	Coeff. of Var
Model	11	2,065,322	0.00	21.23	**4.62**	2.70	0.58
Composite		1,875	0.00	25.44	**5.72**	4.37	0.77
Model	12	10,099	0.99	9.71	**4.26**	2.82	0.66
Composite*		12	0.97	9.73	**5.97**	3.40	0.57
Model	13	11,581	4.21	6.76	**5.46**	0.20	0.04
Composite		6	0.91	12.90	**5.45**	4.11	0.75
Model	14	619,569	0.00	9.42	**3.29**	1.91	0.58
Composite		259	0.00	19.92	**3.82**	3.45	0.90
Model	21	51,661	0.26	9.08	**3.52**	1.73	0.49
Composite		35	0.00	11.63	**3.21**	2.75	0.86
Model	22	42,203	0.20	1.82	**0.81**	0.31	0.38
Composite		13	0.09	1.83	**0.82**	0.52	0.63
Model	23	6,004	0.48	0.50	**0.49**	0.01	0.01
Composite		3	0.48	0.50	**0.49**	0.01	0.02
Model	24	3,921	1.22	1.28	**1.26**	0.02	0.02
Composite		2	1.22	1.28	**1.25**	0.03	0.02
Model	25	2,757	5.69	6.16	**6.00**	0.05	0.01
Composite		3	4.59	7.32	**6.00**	1.12	0.19
Model	31	2,777	0.90	5.02	**2.36**	0.92	0.39
Composite		6	0.04	5.91	**2.88**	2.12	0.73
Model	32	2,346	0.23	7.86	**3.03**	1.72	0.57
Composite		6	0.04	11.90	**3.63**	4.68	1.29
Model	33	2,043	1.68	5.15	**4.33**	0.62	0.14
Composite		4	1.24	6.44	**3.92**	1.88	0.48
Model	34	14,577	0.13	5.44	**2.03**	1.38	0.68
Composite		15	0.01	7.16	**2.23**	2.44	1.09
Model	35	14,038	0.61	1.20	**0.78**	0.12	0.16
Composite		5	0.52	1.20	**0.76**	0.22	0.29

Table 14.30 Pegmont Zones 1 – 5 block model versus composite Zn (%) statistics

Data	Domain	N samples	Minimum	Maximum	Mean	Standdev	Coeff. of Var
Model	11	2,065,322	0.00	19.82	**1.94**	1.28	0.66
Composite		1,875	0.00	23.28	**2.44**	2.41	0.99
Model	12	10,099	0.63	4.93	**2.09**	1.30	0.62
Composite*		12	0.57	5.87	**2.71**	1.79	0.66
Model	13	11,581	0.17	0.37	**0.27**	0.02	0.07
Composite		6	0.06	0.66	**0.27**	0.20	0.75
Model	14	619,569	0.00	9.86	**3.04**	1.61	0.53
Composite		259	0.00	11.10	**3.57**	2.92	0.82
Model	21	51,661	0.90	7.59	**4.44**	1.45	0.33
Composite		36	0.13	11.58	**4.17**	2.69	0.64
Model	22	42,203	0.45	1.92	**1.28**	0.26	0.20
Composite		13	0.27	2.55	**1.28**	0.72	0.56
Model	23	6,004	0.59	0.89	**0.76**	0.10	0.14
Composite		3	0.59	0.89	**0.77**	0.15	0.19
Model	24	3,921	0.43	2.00	**1.37**	0.65	0.48
Composite		2	0.43	2.00	**1.21**	0.79	0.65
Model	25	2,757	6.43	6.50	**6.46**	0.01	0.00
Composite		3	5.68	6.93	**6.46**	0.55	0.09
Model	31	2,777	1.49	8.17	**6.78**	0.81	0.12
Composite		6	0.44	10.05	**6.04**	3.57	0.59
Model	32	2,346	4.60	6.96	**5.63**	0.49	0.09
Composite		6	1.57	8.14	**5.78**	2.27	0.39
Model	33	2,043	1.65	9.42	**7.94**	1.43	0.18
Composite		4	0.59	10.27	**6.98**	3.95	0.57
Model	34	14,577	1.28	12.15	**7.46**	2.10	0.28
Composite		15	0.53	14.83	**7.32**	3.97	0.54
Model	35	14,038	0.34	3.06	**1.98**	0.52	0.26
Composite		5	0.26	3.08	**1.94**	0.96	0.50

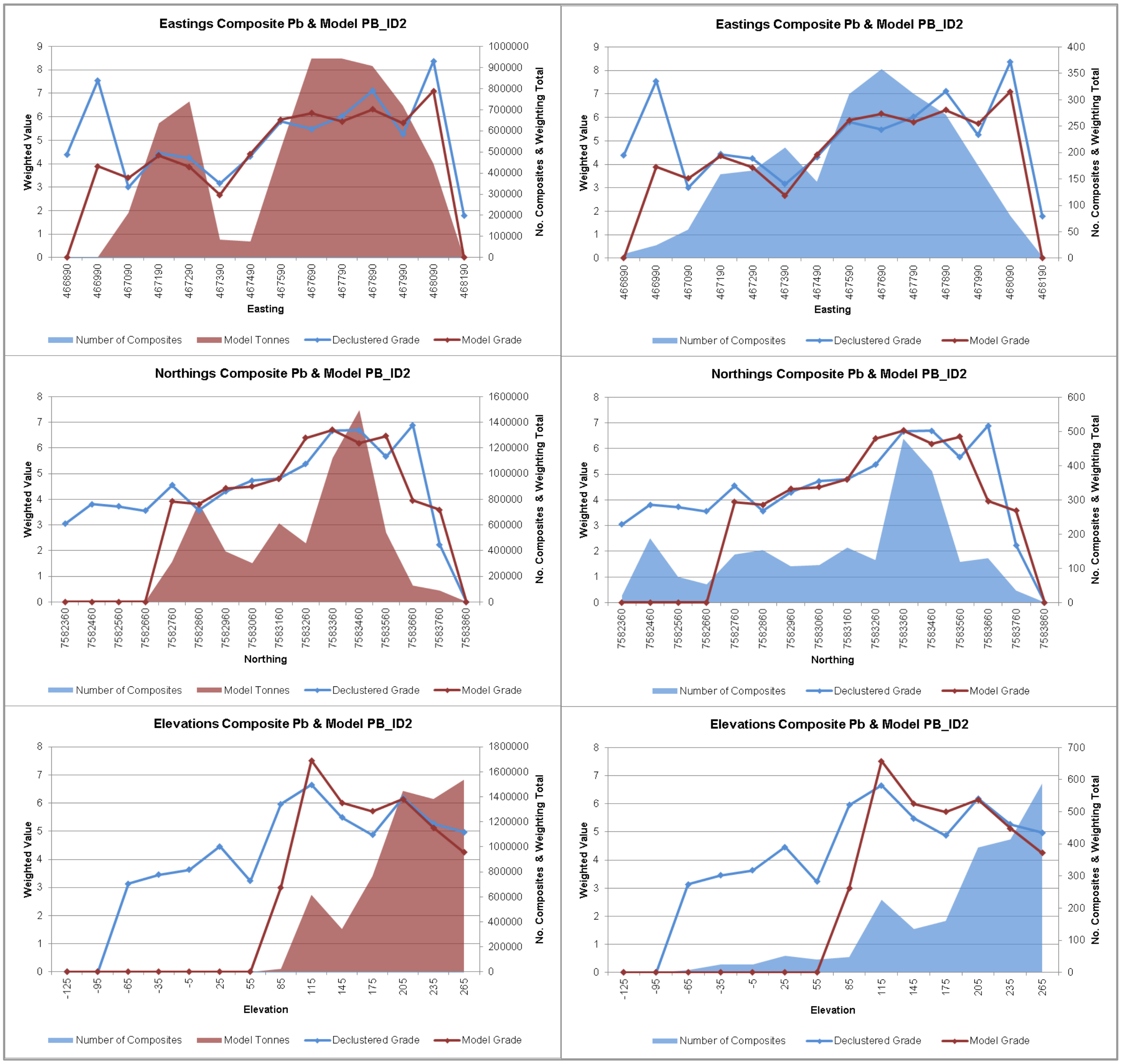
Table 14.31 Pegmont Zones 1 – 5 block model versus composite Ag (g/t) statistics

Data	Domain	N samples	Minimum	Maximum	Mean	Standdev	Coeff. of Var
Model	11	2,065,322	0.00	46.54	**8.06**	4.68	0.58
Composite		1,844	0.00	64.30	**9.35**	8.16	0.87
Model	12	10,099	0.30	14.86	**5.60**	4.23	0.75
Composite*		12	0.25	16.80	**8.15**	5.48	0.67
Model	13	11,581	0.66	0.94	**0.80**	0.02	0.03
Composite		6	0.25	2.10	**0.80**	0.62	0.78
Model	14	612,001	0.00	19.00	**4.87**	2.35	0.48
Composite		255	0.00	36.60	**5.42**	5.01	0.92
Model	21	51,661	1.01	15.27	**4.70**	1.86	0.40
Composite		36	0.25	20.18	**4.51**	3.73	0.83
Model	22	42,203	1.18	6.54	**3.89**	1.23	0.32
Composite		13	0.25	7.90	**3.92**	2.27	0.58
Model	23	6,004	0.51	2.02	**1.35**	0.53	0.39
Composite		3	0.50	2.02	**1.42**	0.74	0.52
Model	24	3,921	2.00	4.90	**3.73**	1.20	0.32
Composite		2	2.00	4.90	**3.45**	1.45	0.42
Model	25	2,757	7.78	8.78	**8.38**	0.10	0.01
Composite		3	6.43	9.80	**8.36**	1.42	0.17
Model	31	2,777	1.31	8.21	**3.91**	1.60	0.41
Composite		6	0.25	9.66	**4.92**	3.43	0.70
Model	32	2,346	0.52	11.21	**4.44**	2.41	0.54
Composite		6	0.25	16.80	**5.28**	6.54	1.24
Model	33	2,043	1.57	6.69	**5.52**	0.92	0.17
Composite		4	0.90	8.50	**4.92**	2.76	0.56
Model	34	14,577	0.39	5.94	**2.85**	1.30	0.45
Composite		15	0.25	8.10	**3.08**	2.62	0.85
Model	35	14,038	2.76	6.25	**3.87**	0.55	0.14
Composite		5	2.70	6.28	**3.90**	1.17	0.30

Swath plots were generated to visually compare the composites and block model grade statistics. Block model grades for lead and zinc were weighted by tonnes. Generally, the swath plots show good agreement between the composite and estimated block model grades.

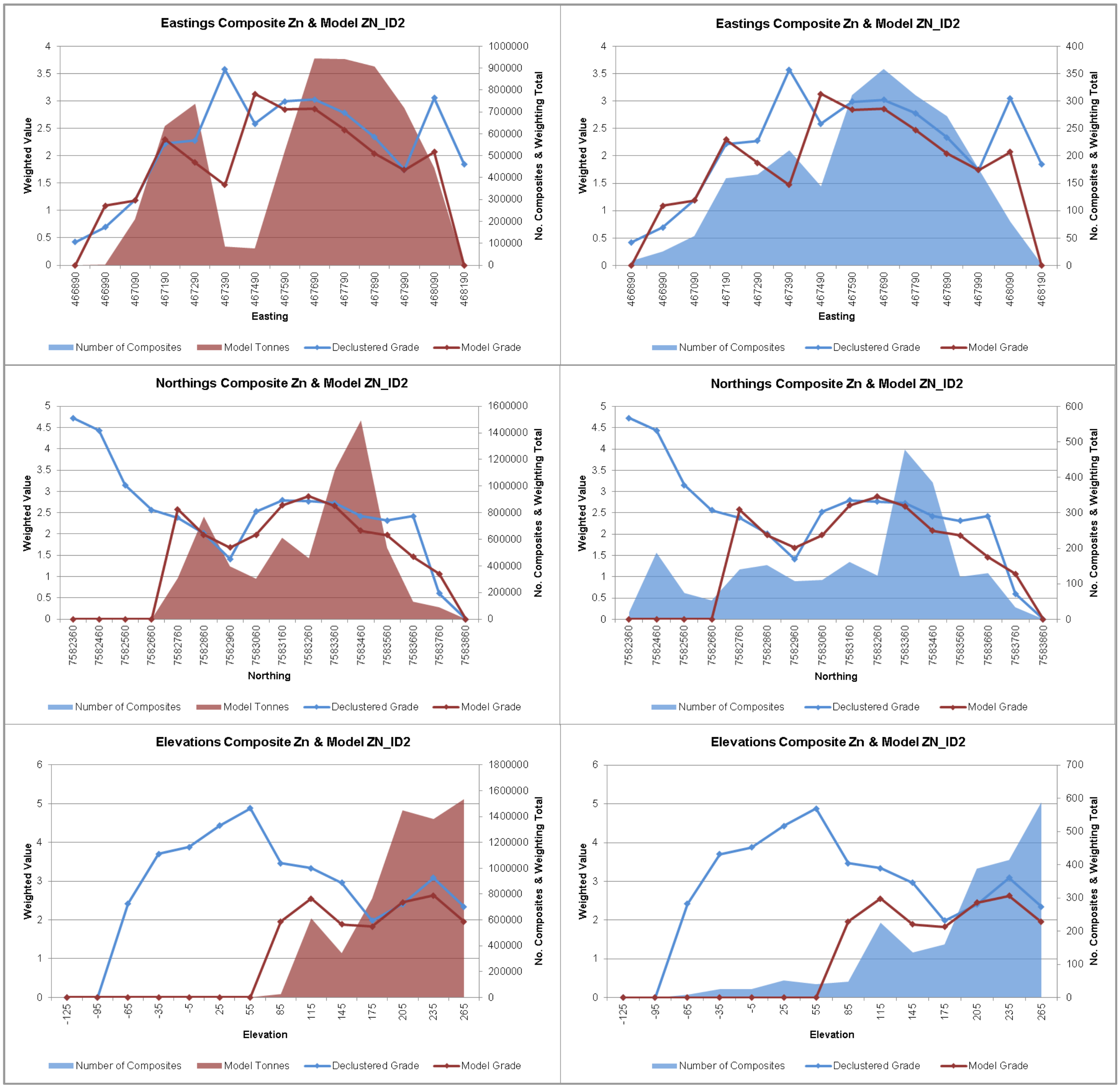
Figure 14.16 to Figure 14.18 show the swath plots for Indicated material.

Figure 14.16 Swath plots of Pegmont domains for Indicated categories – lead



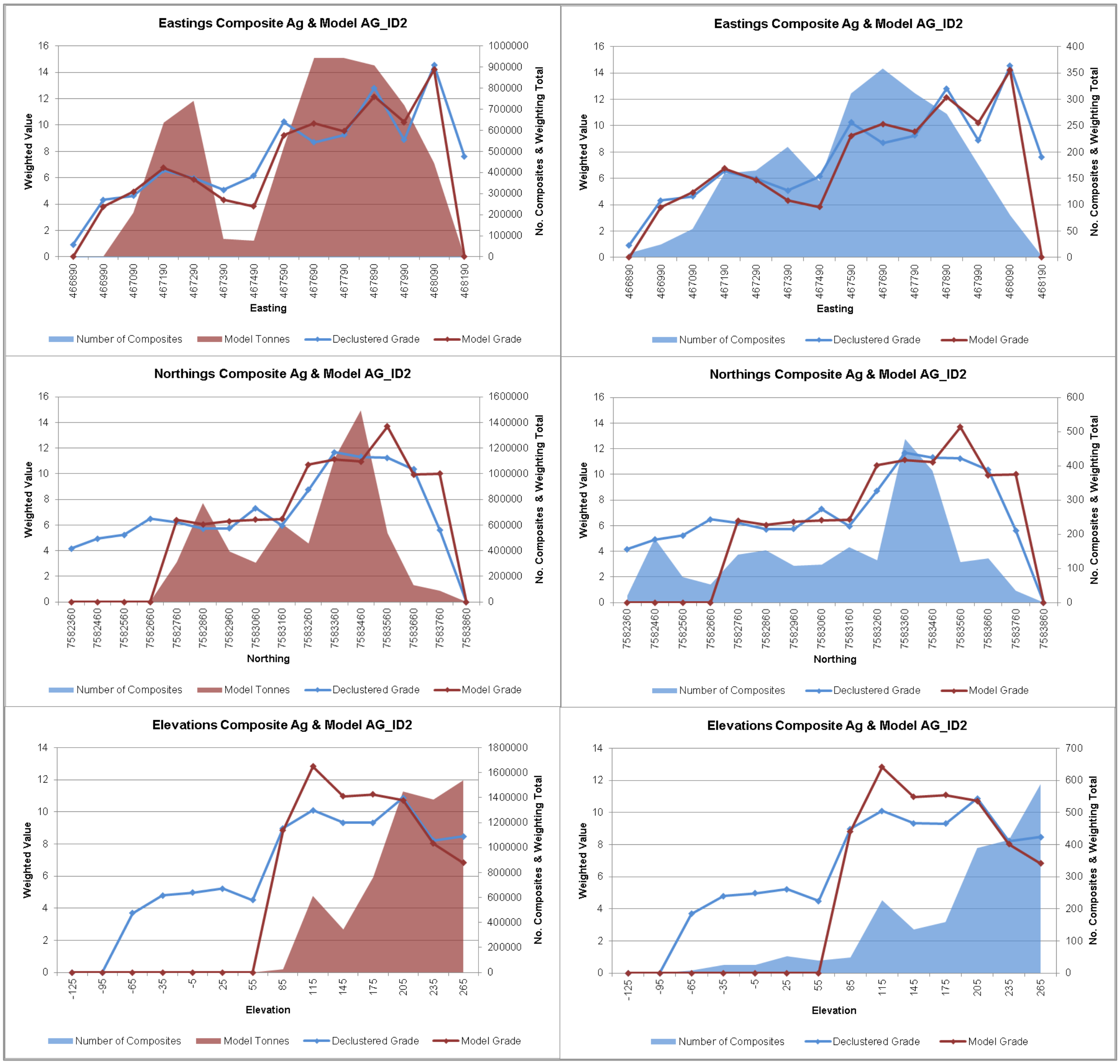
Note: Weighted value = grade; No. weighting total = tonnes.
Source: AMC

Figure 14.17 Swath plots of Pegmont domains for Indicated categories – zinc



Note: Weighted value = grade; No. weighting total = tonnes.
Source: AMC

Figure 14.18 Swath plots of Pegmont domains for Indicated categories – silver



Note: Weighted value = grade; No. weighting total = tonnes.
Source: AMC

14.6.3 Mineral Resource estimate

The Mineral Resource estimate for the Pegmont Zones 1 - 5 as of 31 July 2018 at a 3% lead plus zinc cut-off grade for open pit Mineral Resources and a 5% lead plus zinc cut-off grade for the underground is stated in Table 14.32.

These cut-off grades were provided by VTT and are based on reported operating costs and metallurgical recoveries at proximal operating mines with similar styles of mineralization. AMC has reviewed the work and considers that the cut-off grades are reasonable.

Table 14.32 Pegmont Zones 1 - 5 Mineral Resources at 31 July 2018

Zone	Classification	Material type	Tonnes (kt)	Pb (%)	Zn (%)	Ag (g/t)
Total open pit	Indicated	Transition	1,039	5.0	2.3	8
		Sulphide	3,718	6.7	2.6	12
		Sub total	**4,757**	**6.3**	**2.5**	**11**
	Inferred	Transition	1,823	5.2	2.0	7
		Sulphide	2,442	5.0	2.3	9
		Sub total	**4,265**	**5.1**	**2.2**	**8**
Total underground	Indicated	Sulphide	**84**	**5.4**	**3.0**	**9**
	Inferred		**3,571**	**4.8**	**3.6**	**7**
Total	Indicated	Transition	1,039	5.0	2.3	8
		Sulphide	3,801	6.7	2.6	11
		Total	**4,840**	**6.3**	**2.5**	**11**
	Inferred	Transition	1,829	5.2	2.0	7
		Sulphide	6,012	4.9	3.1	8
		Total	**7,842**	**4.9**	**2.8**	**8**

Notes:
1. CIM Definition Standards (2014) were used to report the Mineral Resources.
2. Cut-off grade applied to the open pit Mineral Resources is 3% Pb+Zn and that applied to the underground is 5% Pb+Zn.
3. Based on the following metal prices: US$0.95/lb for Pb, US$1.05/lb for Zn, and US$16.5/oz for silver.
4. Exchange rate of US$0.75:A$1.0.
5. Metallurgical recoveries for the Main Zones are as follows:
   - Lead to lead concentrate: 91.3% for transition and from 89.7% to 91.8% for sulphide.
   - Zinc to zinc concentrate: 75.2% for transition and from 61.8% to 75.5% for sulphide.
6. Using drilling results up to 15 April 2018.
7. Mineral Resource tonnages have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

AMC is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other similar factors that could materially affect the stated Mineral Resource estimates.

14.7 Economic considerations

The parameters that were used for the open pit optimization and for the calculation of cut-off grades are common to all zones.

14.7.1 Open pit constrained Mineral Resource

AMC performed the open pit optimization using the Lerch-Grossman algorithm coded into the Whittle software. The open pit shell used to constrain the Mineral Resource was based on an NSR cut-off of A$32.42/t determined using the parameters defined above and the following assumptions:

- 55° pit slopes were used, based on experience with similar rocks and conditions within the region.
- Zones 1, 2, 3, and BHZ sulphide and Zone 1 transition metallurgical recoveries and concentrate grades (locked cycle) are as per VTT News Release dated 5 March 2018.
- BHZ transition metallurgical recoveries and concentrate grades (open cycle) are as per Vendetta News Release dated 6 March 2017.
- An 8% discount rate is applied.
- Open pit mining costs of A$3.50/t ore and A$2.50/t waste, A$1/t ore ROM rehandle, A$4.50/t ore mine supervision and technical services, A$2/t ore surface General and Administration (G&A) overheads, and processing costs of A$23.92/t ore.

Within the open pit shell, the Mineral Resource is stated at a 3% lead + zinc cut-off, based on a comprehensive cut-off approach, approximately equal to the A$32.42/t NSR cut-off used to generate the pit shells.

Oxide lead-zinc mineralization is not included in the current Mineral Resource as with the sequential flotation processing flow sheet envisaged it is considered that there is no effective method for mineral processing of oxide mineralization and hence no economic basis for its inclusion.

14.7.2 Underground Mineral Resource

The underground mineralization is defined as the areas outside of the optimization pits. The cut-off grade of 5% Pb + Zn is based on a comprehensive cut-off approach and the following assumptions were used:

- Zone 3 and Bridge Zone metallurgical recoveries and concentrate grades (locked cycle) are as per VTT News Release dated 5 March 2018.
- Zone 5 metallurgical recoveries and concentrate grades (open cycle) are as per VTT News Release dated 6 March 2017.
- Underground mining costs of A$45.00/t ore and G&A of A$5.00/t ore and processing costs of A$23.92/t ore.

14.8 Total Mineral Resources

The total Mineral Resources are shown in Table 14.33 which tabulates the open pit and underground contributions by class and material type.

Table 14.33 Total Mineral Resources by location

	Classification	Material type	Tonnes (kt)	Pb (%)	Zn (%)	Ag (g/t)
Open pit (Zones 1, 2, 3, and BHZ)	Indicated	Transition	1,111	4.9	2.3	8
		Sulphide	4,003	6.5	2.6	11
		Total	**5,114**	**6.2**	**2.6**	**11**
	Inferred	Transition	1,829	5.2	2.0	7
		Sulphide	2,567	5.0	2.3	10
		Total	**4,396**	**5.1**	**2.2**	**8**
Underground (Zones 3, 4, 5, and Bridge)	Indicated	Sulphide	**644**	9.0	2.6	14
	Inferred	Sulphide	**3,880**	5.1	3.6	4
Total	Indicated	Transition	1,111	4.9	2.3	8
		Sulphide	4,647	6.9	2.6	12
		Sub total	**5,758**	**6.5**	**2.6**	**11**
	Inferred	Transition	1,829	5.2	2.0	7
		Sulphide	6,447	5.1	3.1	9
		Total	**8,277**	**5.1**	**2.8**	**8**

14.9 Overall comparison with 2017 Mineral Resource

The differences between the 2018 and 2017 Mineral Resources are due to the discovery of the Bridge zone, new drilling and different economic parameters applied to the optimization pits.

Table 14.34 is a comparison between the two estimates to show the net effects due to drilling, new pits, and the inclusion of the Bridge Zone. Table 14.35 shows the percentage difference between the two years.

Table 14.34 Comparison of the 2017 and 2018 Mineral Resources (as reported)

Classification	Material type	2017				2018			
		Tonnes (kt)	Pb (%)	Zn (%)	Ag (g/t)	Tonnes (kt)	Pb (%)	Zn (%)	Ag (g/t)
Indicated	Transition	685	5.2	2.5	9	1,111	4.9	2.3	8
	Sulphide	1,560	5.7	2.7	11	4,647	6.9	2.6	12
	Sub total	**2,245**	**5.6**	**2.6**	**10**	**5,758**	**6.5**	**2.6**	**11**
Inferred	Transition	1,035	5.3	2.6	8	1,829	5.2	2.0	7
	Sulphide	8,612	4.9	2.9	8	6,447	5.1	3.1	9
	Total	**9,647**	**5.0**	**2.9**	**8**	**8,277**	**5.1**	**2.8**	**8**

Table 14.35 Differences between the 2018 and 2017 Mineral Resources

Classification	Material type	Difference 2018 – 2017			
		Tonnes (kt)	Pb (%)	Zn (%)	Ag (g/t)
Indicated	Transition	62%	-6%	-8%	-11%
	Sulphide	198%	21%	-4%	9%
	Total	156%	16%	0%	10%
Inferred	Transition	77%	-2%	-23%	-13%
	Sulphide	-25%	4%	7%	13%
	Total	-14%	2%	-3%	0%

Notes for 2017 Mineral Resource Estimate
1. CIM definitions were used for the Mineral Resources.
2. The cut-off grade applied to the open pit Mineral Resources is 3% Pb + Zn and the underground cut-off grade is 5% Pb + Zn.
3. Based on metal prices of US$0.90/lb for Pb, US$0.95/lb for Zn and US$15/oz for silver.
4. An exchange rate of $0.73 US$:A$ exchange rate.
5. Metallurgical recoveries as follows:
   - BHZ: vary from 80.6% for transition to 91.5% for sulphide for lead to the lead concentrate and 19.3% for transition to 61.8% for sulphide for zinc to the zinc concentrate.
   - U/G: vary from 83.0% to 88.5% for lead to the lead concentrate and 75.6% to 76.7% for zinc to the zinc concentrate.
6. Using drilling results up to 8 May 2017.
7. The numbers may not add precisely due to rounding.

Notes for 2018 Mineral Resource Estimate are in the footnotes under Table 14.1.

The following observations have been made by the QP from the table comparing the 2018 estimate with the 2017 estimate:

- Total Indicated tonnes have increased by 156%, while the Inferred tonnes have decreased by 14%.
- The grades of lead, zinc, and silver have increased overall in the Indicated category and decreased in the Inferred category.
- There are additions due to new drilling and discovery of the Bridge Zone.
- The 2018 optimization pit used different economic parameters and different recoveries resulting in a larger open pit shell constraining the Mineral Resources.

15 Mineral Reserve estimates

There are no Mineral Reserves on the Property.

16 Mining methods

16.1 General description

The Pegmont project includes three resources consisting of the BHZ, the Main Zones (Zones 1 - 5), and the Bridge Zone. The BHZ can be developed as an open pit, the Main Zone can be developed as a combination of open pit and underground, and the Bridge Zone is an underground resource.

16.2 Hydrogeological parameters

A network of 26 drillholes was set up by VTT to monitor phreatic water levels. Data collected to date shows the hydraulic gradient to run from North to South, from elevations of 250 to 241 mRL, generally 50 m below, and parallel to the topographical surface.

AMC assumed that water table lies 50 m below the topographical surface for this study.

16.3 Open pit mining

16.3.1 Resource model for open pit mining

AMC developed the Mineral Resource block models for the Main Zone and BHZ (reference: peg_bm_jul18_optim.dm and bhz_2018_for_optim_f.dm, respectively) for evaluation of the open pit mining potential. The block models are proportional models, containing a field for ore percent and waste percent, with block dimensions of 5 m in the X (east) direction by 5 m in the Y (north) direction by 5 m in the Z (vertical) direction for the Main Zone, and 2.5 m in X by 2 m in Y by 2.5 m in Z direction for the BHZ.

16.3.2 Open pit geotechnical considerations

Elysium Mining Ltd developed scoping-level slope design configurations to be used in the PEA, as summarized in a report entitled “EML_Slope_Design_Criteria_PEA_2018.pdf”, 2018.

Three geotechnical core holes were drilled into the deeper parts of Zone 3. RQD information was also collected and used in the slope geotechnical analysis.

Drillhole information was used to identify the depth of the weathering profile, which was set at 235 m RL.

The geotechnical holes were analyzed to estimate quantitative Geological Strength Index (GSI) and empirical method to derive modifying parameters to estimate the slope stability rating (SSR). Design charts were then used to describe the relationship between rock slope height and SSR versus stable slope angles.

Bench scale design was performed using wedge and planar failure kinematic analysis.

The design was developed assuming double benching of 10 m high benches in waste rock with 5 m catch berms placed every 20 m. Bench face angles (BFA) of 80 degrees and a limiting bench stack height of 60 m were recommended.

Bench stack angles (BSA) and limiting overall slope angles (OSA) were assessed for each geotechnical sector shown in Figure 16.1. The weathered domain boundary lies at approximately 235 mRL. A ramp or geotechnical berm of minimum width of twice the catch berm must be placed at 235 mRL. The BSA recommended for weathered material are generally shallower than the BSA for fresh rock. AMC modified the catch berm width for weathered material in order to comply with the recommended BSA by sector.

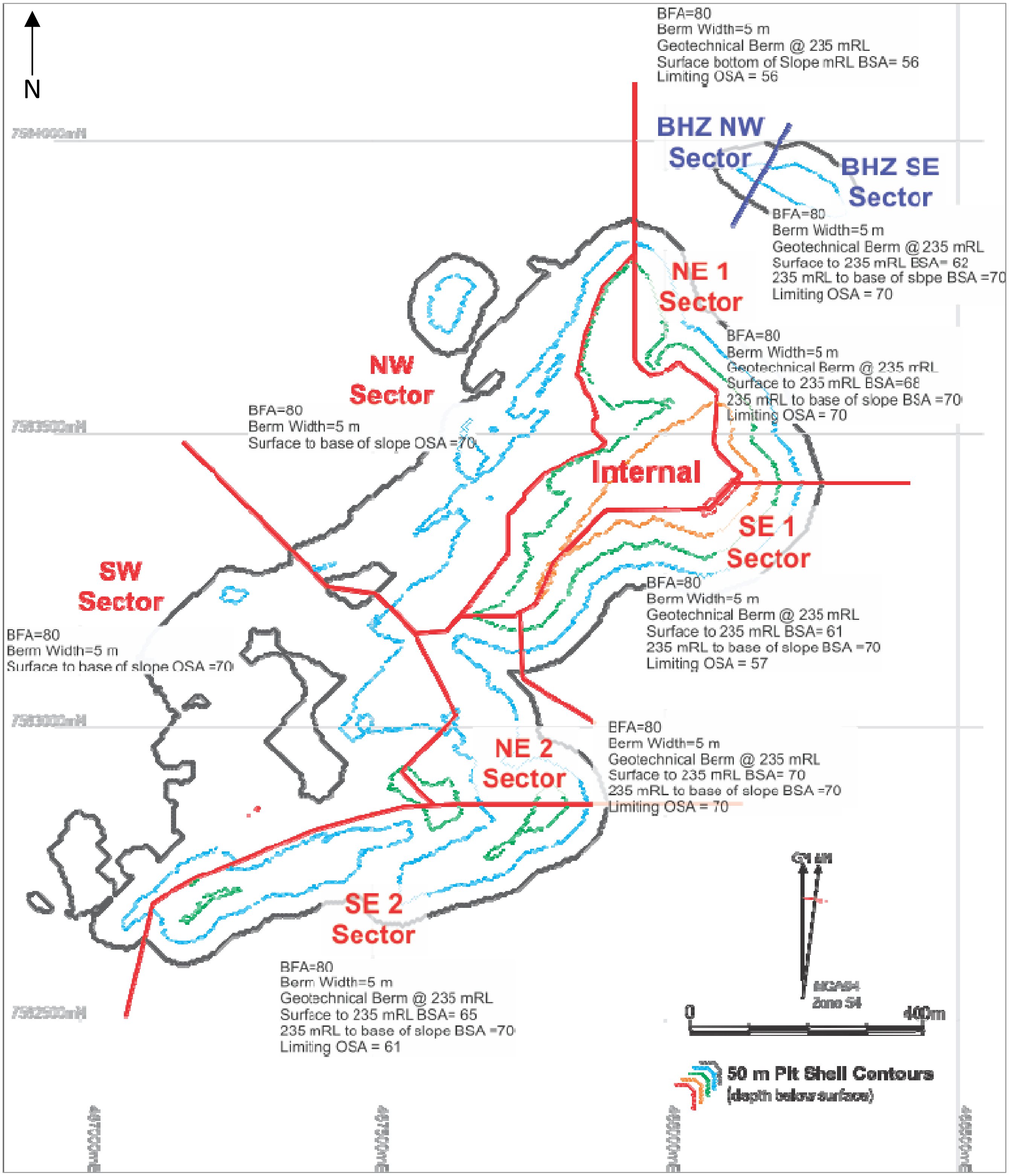
The slope design recommendations for the Main Zone and BHZ are summarized in Table 16.1.

Table 16.1 Slope design recommendations

Sector	Main						BHZ	
	SW	**NW**	**NE 1**	**SE 1 & Internal**	**NE 2**	**SE 2**	**NW**	**SE**
Double bench height (m)	20	20	20	20	20	20	20	20
BFA (deg)	80	80	80	80	80	80	80	80
Min. berm width (m)	5	5	5	5	5	5	5	5
Geotechnical berm elevation or ramp	None	None	235 mRL	235 mRL	235 mRL	235 mRL	None	235 mRL
BSA weathered (deg)	70	70	68	61	70^	65	70^	62
BSA fresh (deg)	N/A	N/A	70^	70^	70^	70^	70^	70^
Limiting OSA (deg)	70^	70^	70^	57	70^	61	56	70

Note: 70^ indicated above 70° and outside of the SSR design chart.

Figure 16.1 Geotechnical sectors



Source: Elysium Mining Ltd.

16.3.3 Open pit mining method

AMC proposes to mine the open pits using a conventional truck and excavator mining method. AMC has assumed that a 10 m bench height would be adopted for waste rock, and that mineralized material will be mined in 2.5 m flitches to increase mining selectivity. Mining of mineralized material will occur using a hydraulic excavator in backhoe configuration (example: Komatsu PC1250). Hydraulic excavators in front-shovel configuration (example: Komatsu PC2000) will be used in areas of bulk waste rock to increase the rate of mining in waste rock and reduce operating costs.

The average productivity of the PC1250 excavators in blasted mineralized material is expected to be approximately 1,080 t/op hr. The productivity of the PC2000 in fresh waste is expected to reach 1,513 t/op hr.

Hauling of mineralized material and waste will be undertaken by 90 t trucks (example: CAT 777). Mineralized material will be hauled to the run-of-mine (ROM) pad or the process plant. Low grade material will be hauled to the low-grade long-term stockpiles.

16.3.4 Open pit optimization

16.3.4.1 Cut-off calculation

AMC estimated mineralized material and waste mining costs based on high level assumptions. The open pit mine operations are planned to be executed by a mining contractor.

Table 16 displays the calculation of the open pit cut-off on an NSR basis. The parameters used to estimate NSR are provided in Table 16.3.

Table 16.2 Open pit cut-off calculation

		Unit	Value
Mineralized material mining costs	Contractor mineralized material mining costs	A$/t Mineralized material mined	3.36
	Grade control	A$/t Mineralized material mined	0.70
	Run of mine re-handle	A$/t Mineralized material mined	0.25
	Total mineralized material mining costs	A$/t Mineralized material mined	4.31
Waste mining costs	Contractor waste mining costs	A$/t waste mined	2.90
Processing and site G&A costs	Processing cost	A$/t Mineralized material mined	28.49
	Site G&A	A$/t Mineralized material mined	2.00
NSR	NSR cut-off	A$/t Mineralized material mined	31.90

Table 16.3 NSR parameters for pit optimization

		Unit	Main				BHZ			
			Transition		Sulphide		Transition		Sulphide	
			Lead conc.	Zinc conc.	Lead conc.	Zinc conc.	Lead conc.	Zinc conc.	Lead conc.	Zinc conc.
Processing parameters	Lead metallurgical recovery	%	91.3	2.2	89.7 – 91.8	1.9 – 2.5	80.6	1.2	88.0	3.0
	Zinc metallurgical recovery	%	10.2	75.2	9.3 – 12.1	71.3 – 75.5	22.3	19.3	7.4	78.5
	Silver metallurgical recovery	%	46.6	3.9	65.5 – 94.9	1.4 – 2.6	46.6	3.9	92.3	5.3
Metal price	Lead price	US$/lb	0.95							
	Zinc price	US$/lb	1.05							
	Silver price	US$/oz	16.50							
General	Process plant throughput	Mtpa	1							
	Exchange rate		US$0.75:A$1.0							
Concentrate parameters	Lead concentrate grade	%	72.5	5.1	66.3 – 68.2	3.2 – 4.5	57.0	3.4	67.7	3.5
	Zinc concentrate grade	%	2.5	53.3	2.9 – 3.6	54.6 – 54.9	14.0	48.9	3.1	51.2
	Concentrate moisture	%	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0
Concentrate costs	Transport, storage, and loading	A$/wmt	50	106.5	50	106.5	50	106.5	50	106.5
	Marketing	US$/wmt	5	5	5	5	5	5	5	5
	TCRC	US$/wmt	170	180	170	180	170	180	170	180
	Total concentrate costs	US$/dmt	231.0	287.9	231.0	287.9	231.0	287.9	231.0	287.9
Penalties	Penalties	US$/dmt	2	5.5	8 – 10	1.5 – 5.5	2	4.5	6	5
Refining	Silver refining charge	A$/oz Ag	1.67	1.67	1.67	1.67	1.67	1.67	1.67	1.67
Metal payable	Lead payable	%	95	50	95	50	95	50	95	50
	Zinc payable	%	50	85	50	85	50	85	50	85
	Silver payable	g/t	95	70	95	70	95	70	95	70
Metal deductions	Lead minimum deduction	%	3	0	3	0	3	0	3	0
	Zinc minimum deduction	%	0	8	0	8	0	8	0	8
	Silver minimum deduction	g/t	50	93	50	93	50	93	50	93
Royalties	Lead QLD Government royalty	%	4.28							
	Lead vendor royalty	%	1.5							
	Zinc QLD Government royalty	%	2.92							
	Zinc vendor royalty	%	1.5							
	Silver QLD Government royalty	%	5% if Au grade in concentrate > 100 g/t							
	Silver vendor royalty	%	1.5							

Note: Parameters for sulphide material in the Main area vary by zone.

The NSR was determined on a block by block basis. The formula to determine NSR is as follows:

NSR = (value of recovered payable Pb metal) * (1 - Pb royalty %) + (value of recovered payable Zn metal) * (1 - Zn royalty %) + (value of recovered payable Ag metal) * (1 - Ag royalty %).

Value of recovered payable Pb metal = (Pb contained metal x Pb metallurgical recovery % x Pb payability % x Pb Price) - Concentrate costs.

Value of recovered payable Zn metal = (Zn contained metal x Zn metallurgical recovery % x Zn payability % x Zn Price) - Concentrate costs.

Value of recovered payable Ag metal = (Ag contained metal x Ag metallurgical recovery % x Ag payability %) x (Ag Price – Ag refining charge).

16.3.4.2 Dilution and mining recovery factors

A mining dilution of 5% and a mining recovery of 95% were assumed. The mining dilution and recovery were applied as factors during the pit optimization process and to estimate open pit tonnages for the schedule. The dilution material is assumed to have zero value.

Dilution method of the block models should be further investigated in the next study stage.

16.3.5 Pit shell selection

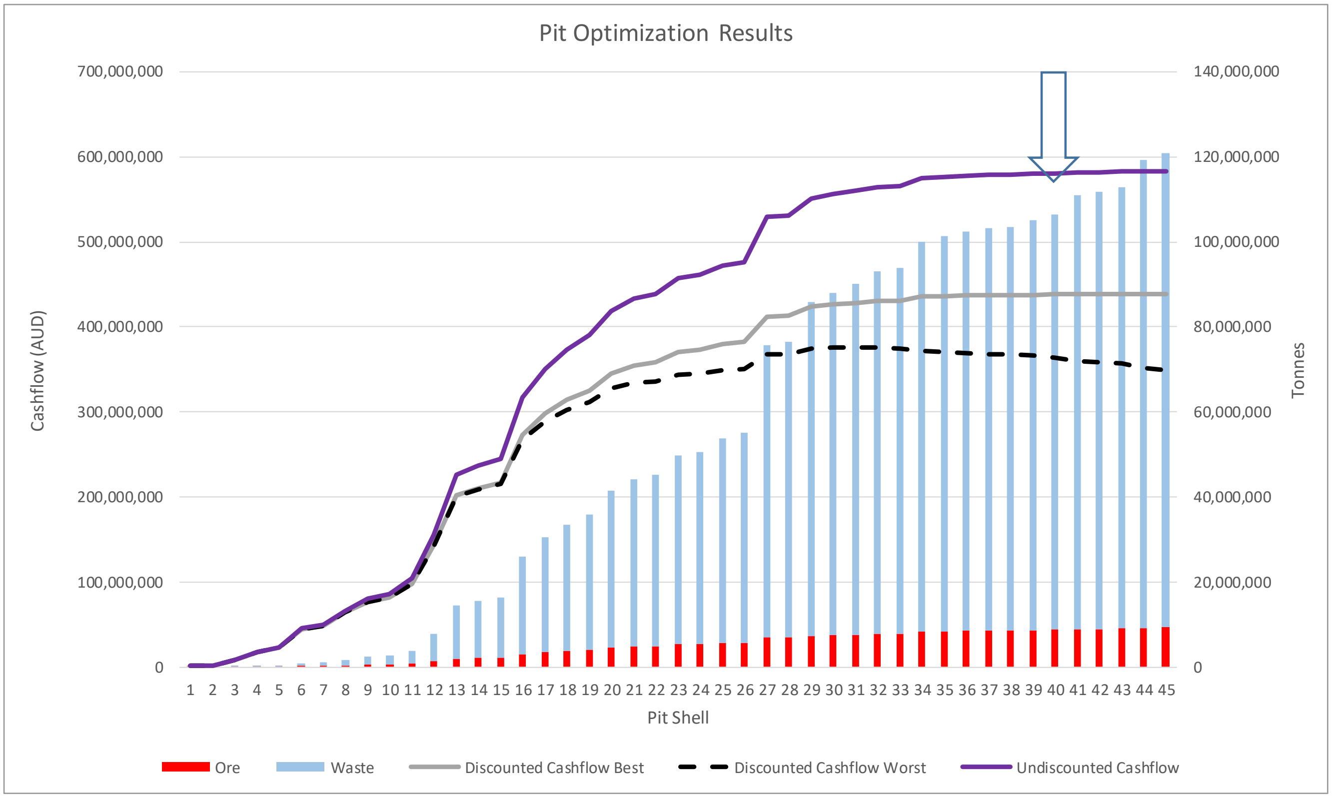
The Lerchs-Grossmann pit optimization algorithm as implemented in the Whittle software was used to define the ultimate pit shell for the Main Zone and BHZ. The selected pit shells were then used to produce pit designs and the mining schedule.

16.3.5.1 Main Zone ultimate pit selection

The Main Zone optimization results are provided in Figure 16.2. The graph shows discounted pit values for "best case" and "worst case", and undiscounted values. The best case gives the maximum discounted value and requires that each shell be mined sequentially. The worst case assumes that the deposit is mined on a bench by bench basis and gives the lowest discounted value.

The Main Zone can be developed as a combination of open pit and underground. An open pit to underground interface analysis was conducted (see Section 16.4) and it was determined that additional value is generated by mining the upper portion of the zone 3B mineralization using an open pit mining method. AMC analyzed specific zones of the deposit separately and selected shell 40 as the ultimate shell, based on overall project value generated by the open pit, underground, and taking into consideration a 15 m crown pillar between the open pit and underground operations.

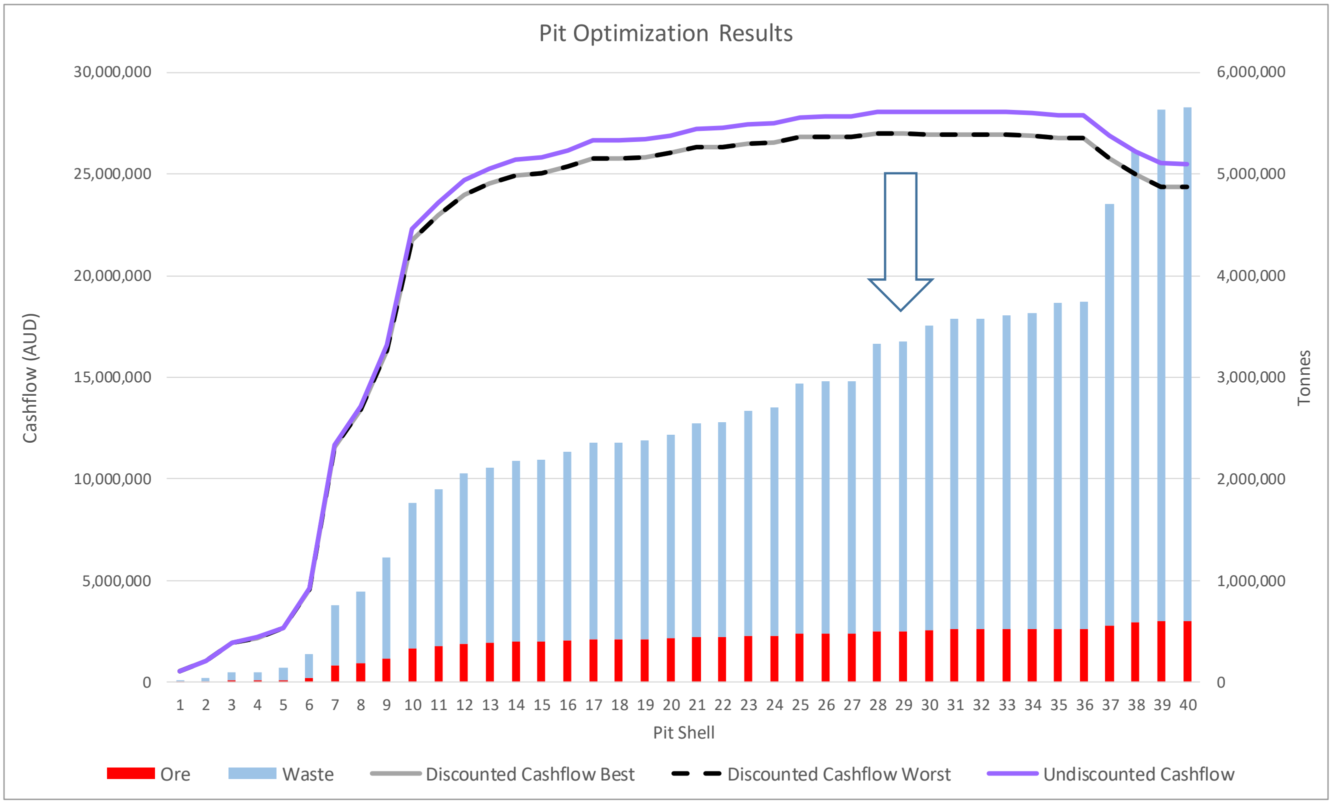
Figure 16.2 Main Zone optimization results



16.3.5.2 BHZ ultimate pit selection

The BHZ optimization results are provided in Figure 16.3. The pit shell corresponding to the 100% revenue factor is pit shell 30. As the BHZ pit is to be mined in one single pushback the maximum discounted worst cashflow shell 29 was selected as the basis for the mining schedule.

Figure 16.3 BHZ optimization results



16.3.6 Pit design

Pits were designed based on the selected optimization shells. Four pits have been designed, one for BHZ and three in the Main area.

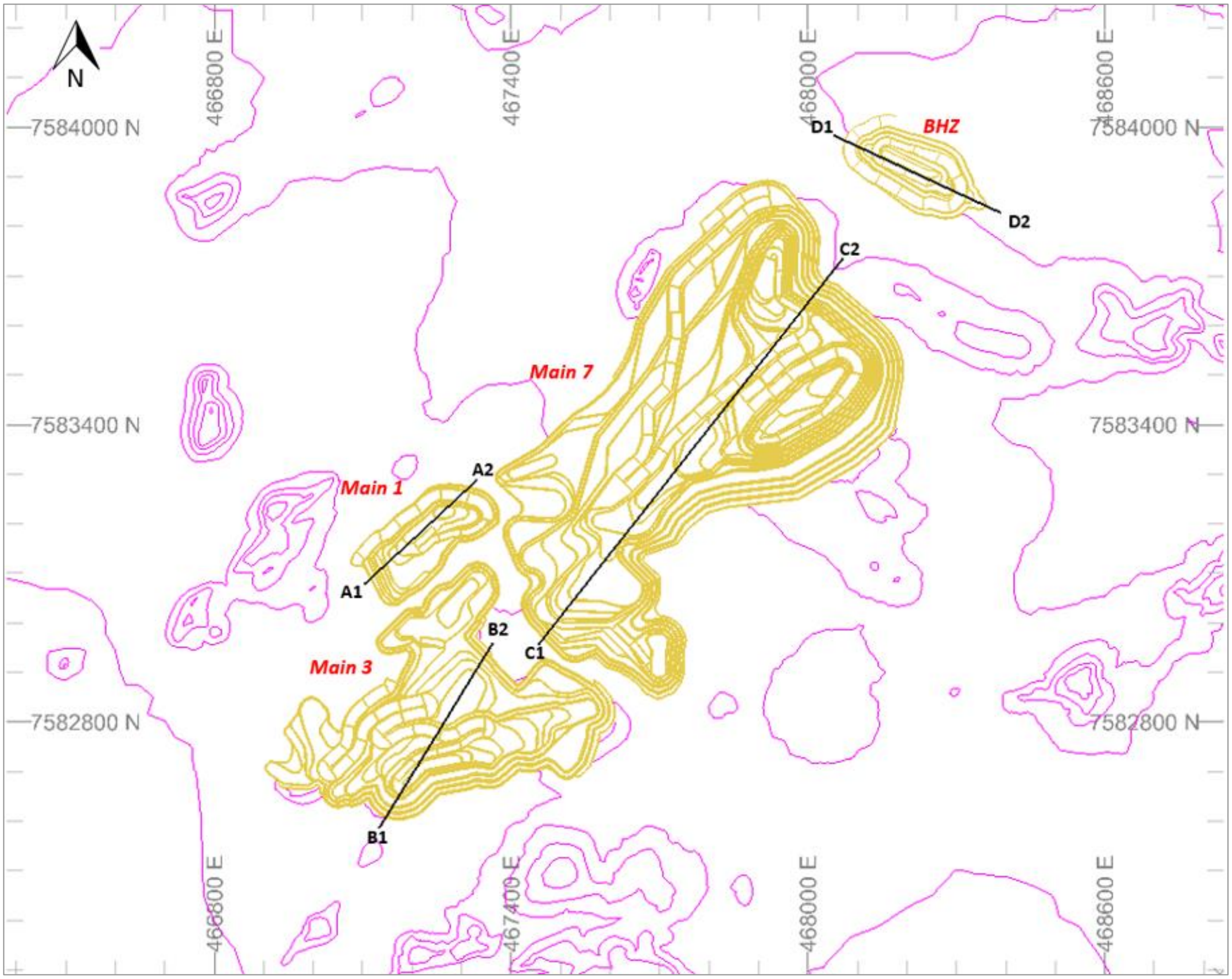
Tailings from the processing plant are to be stored in three of the mined-out pits (BHZ, Main 1, and Main 3). AMC assumed that a minimum 15 m pillar at pit crest level is to be left between the tailings pits and other pits. A geotechnical study should be conducted in the future to confirm appropriate pillar dimensions.

Haulage ramps will be 23.1 m wide at a 10% gradient. Single way ramps of 13.3 m width were designed for the bottom twenty vertical meters. The final pit designs for the Main Zone and BHZ are presented in Figure 16.4. Sections displaying NSR values for the mineralized material are presented in Figure 16.5 to Figure 16.8.

Figure 16.9 shows the two stages (Main 2 and Main 3) of the Main 3 pit.

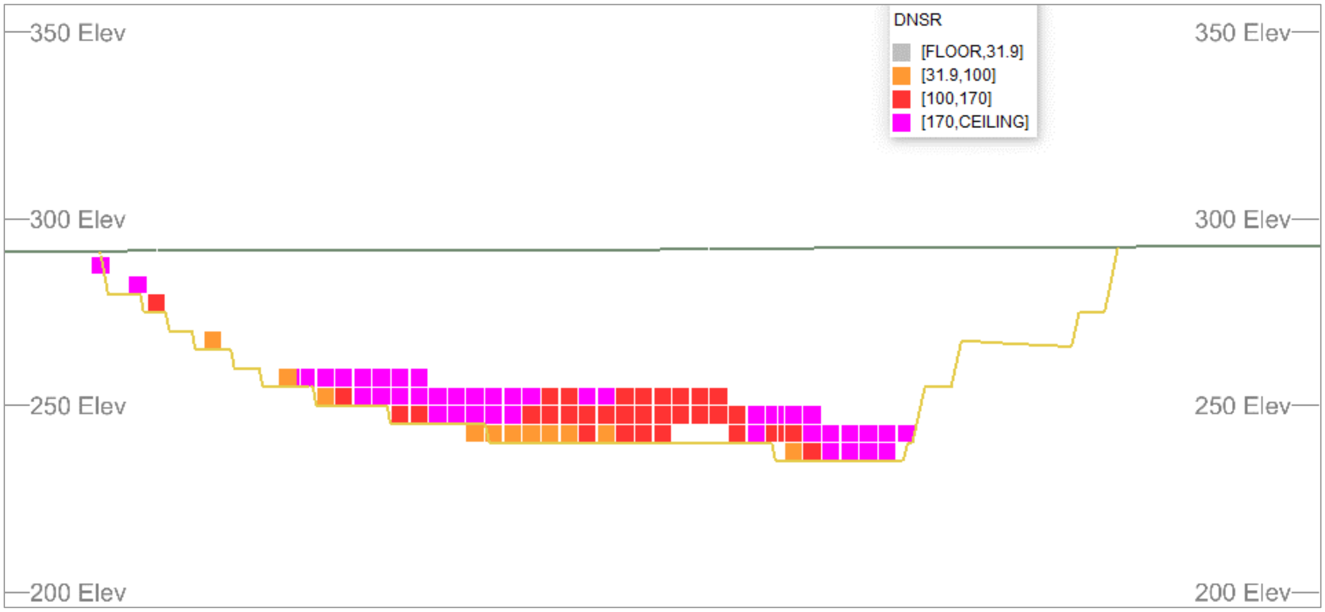
The Main 7 pit is the largest pit measuring approximately 930 m in length, 330 to 550 m in width, and 200 m at its deepest point. The Main 7 pit consists of four stages (Main 4, Main 5, Main 6, and Main 7) and is presented in Figure 16.10.

Figure 16.4 Final pit design



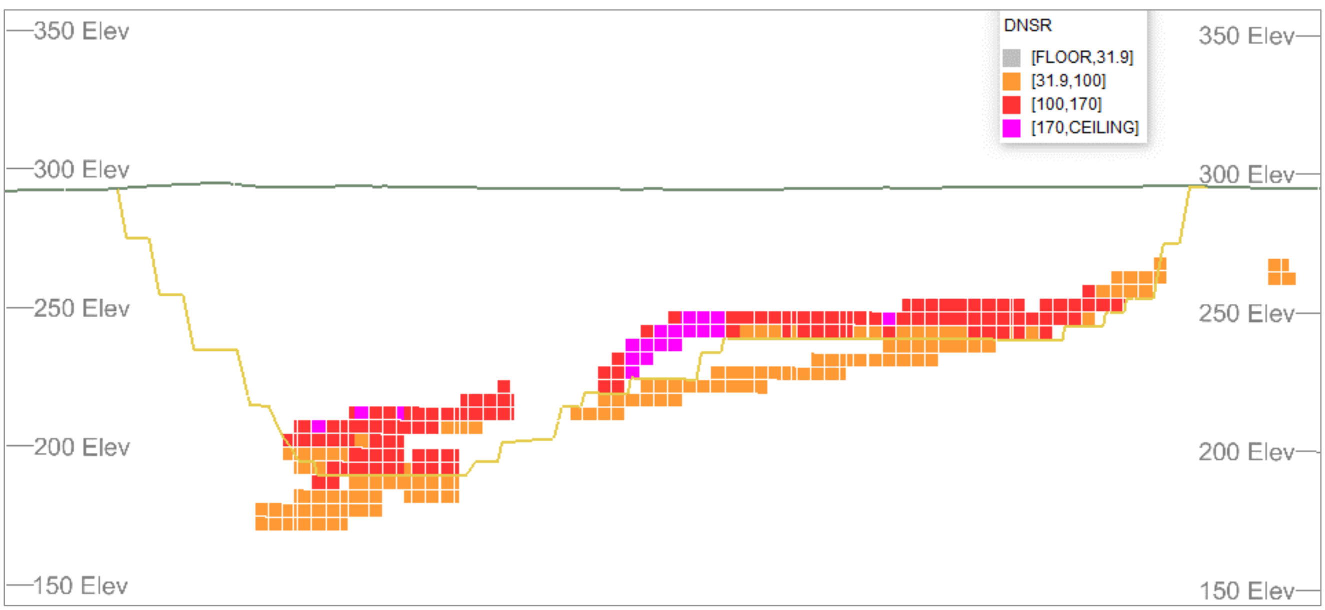
Source: AMC

Figure 16.5 Section view A1-A2 with NSR values (A$)



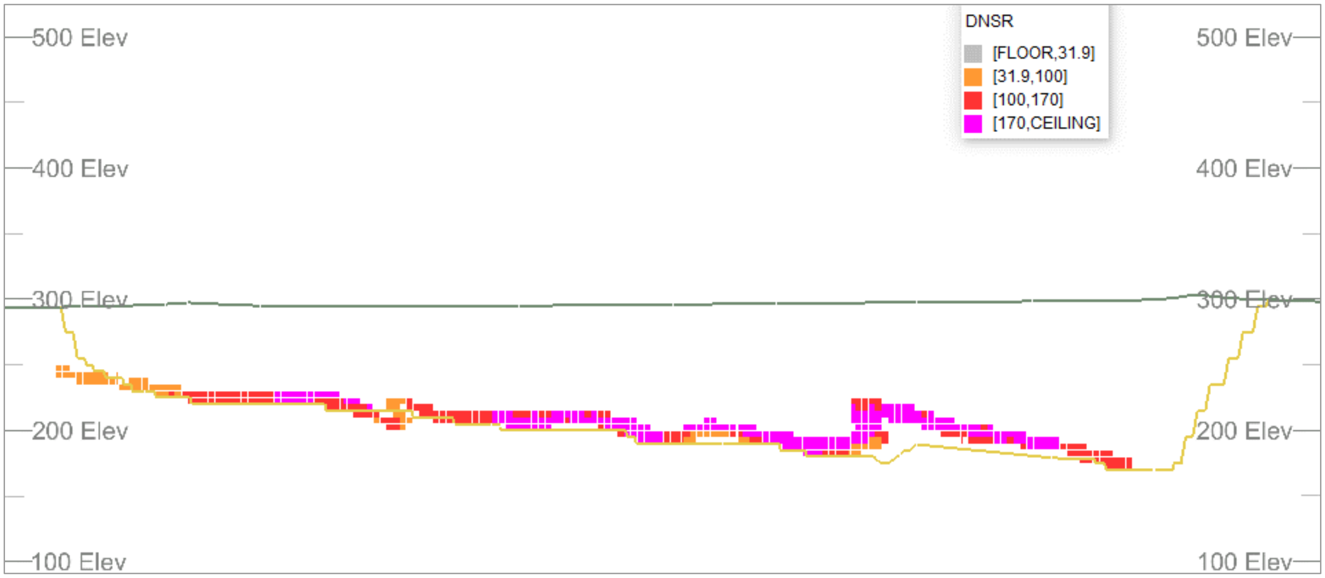
Source: AMC

Figure 16.6 Section view B1-B2 with NSR values (A$)



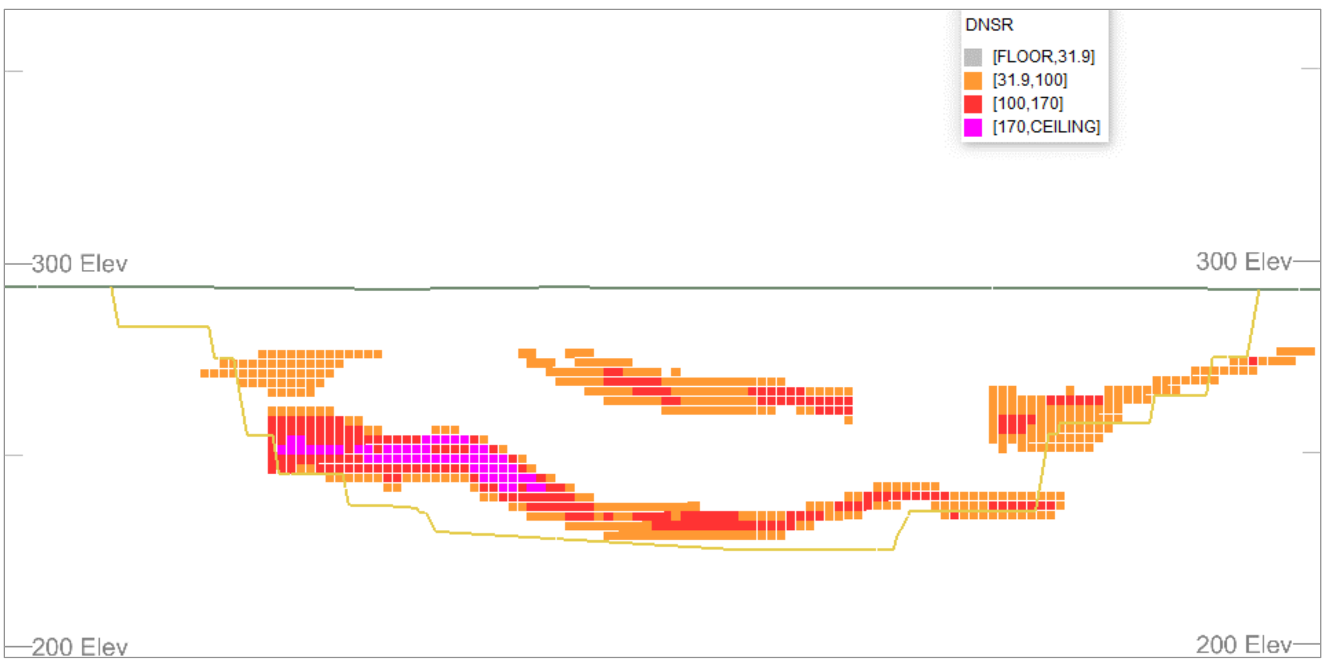
Source: AMC

Figure 16.7 Section view C1-C2 with NSR values (A$)



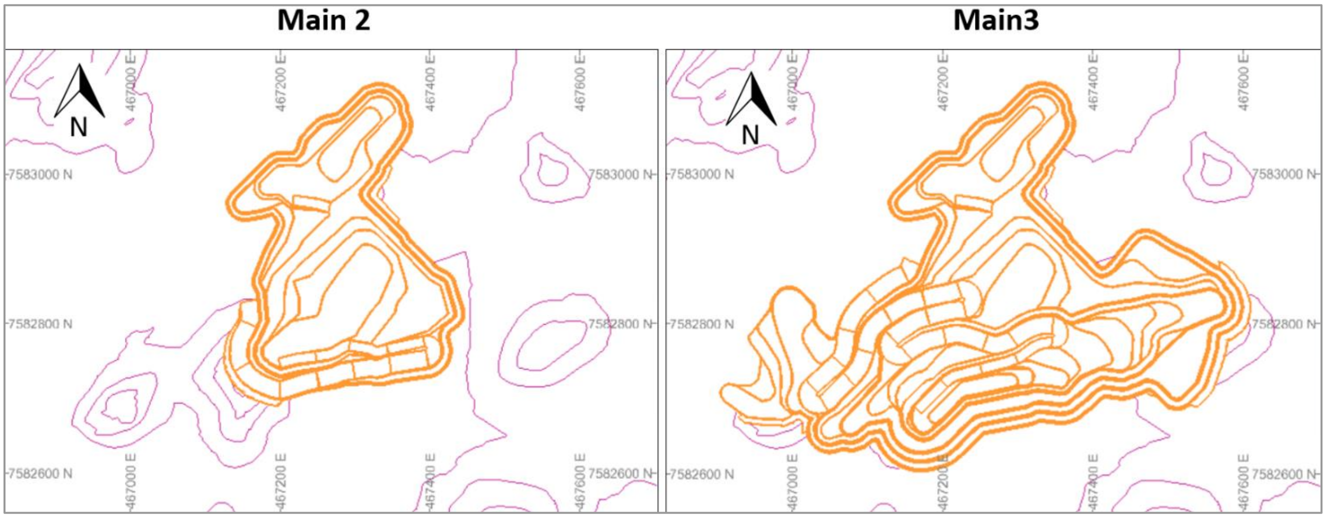
Source: AMC

Figure 16.8 Section view D1-D2 with NSR values (A$)



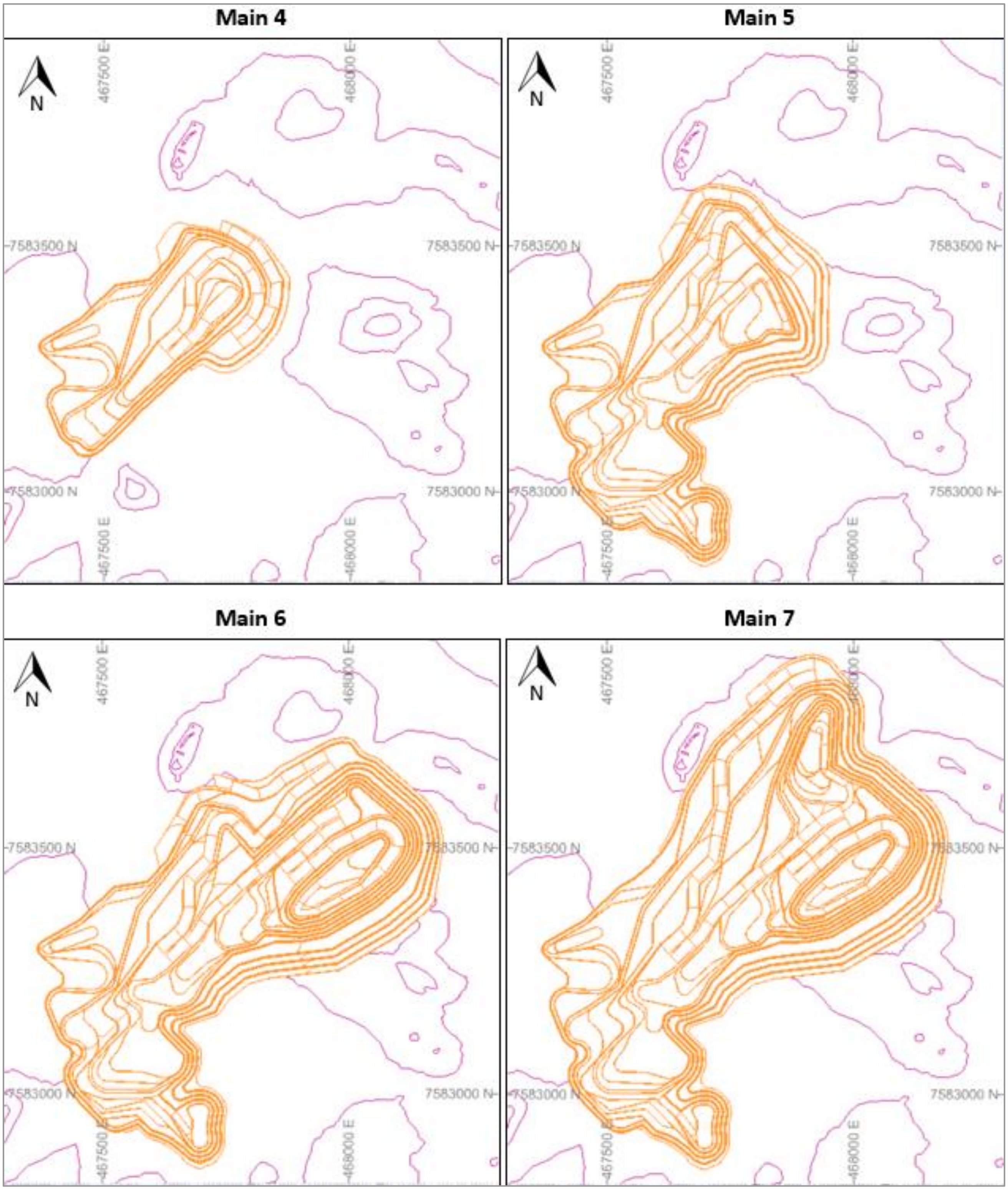
Source: AMC

Figure 16.9 Main 3 pit stages



Source: AMC

Figure 16.10 Main 7 pit stages



Source: AMC.

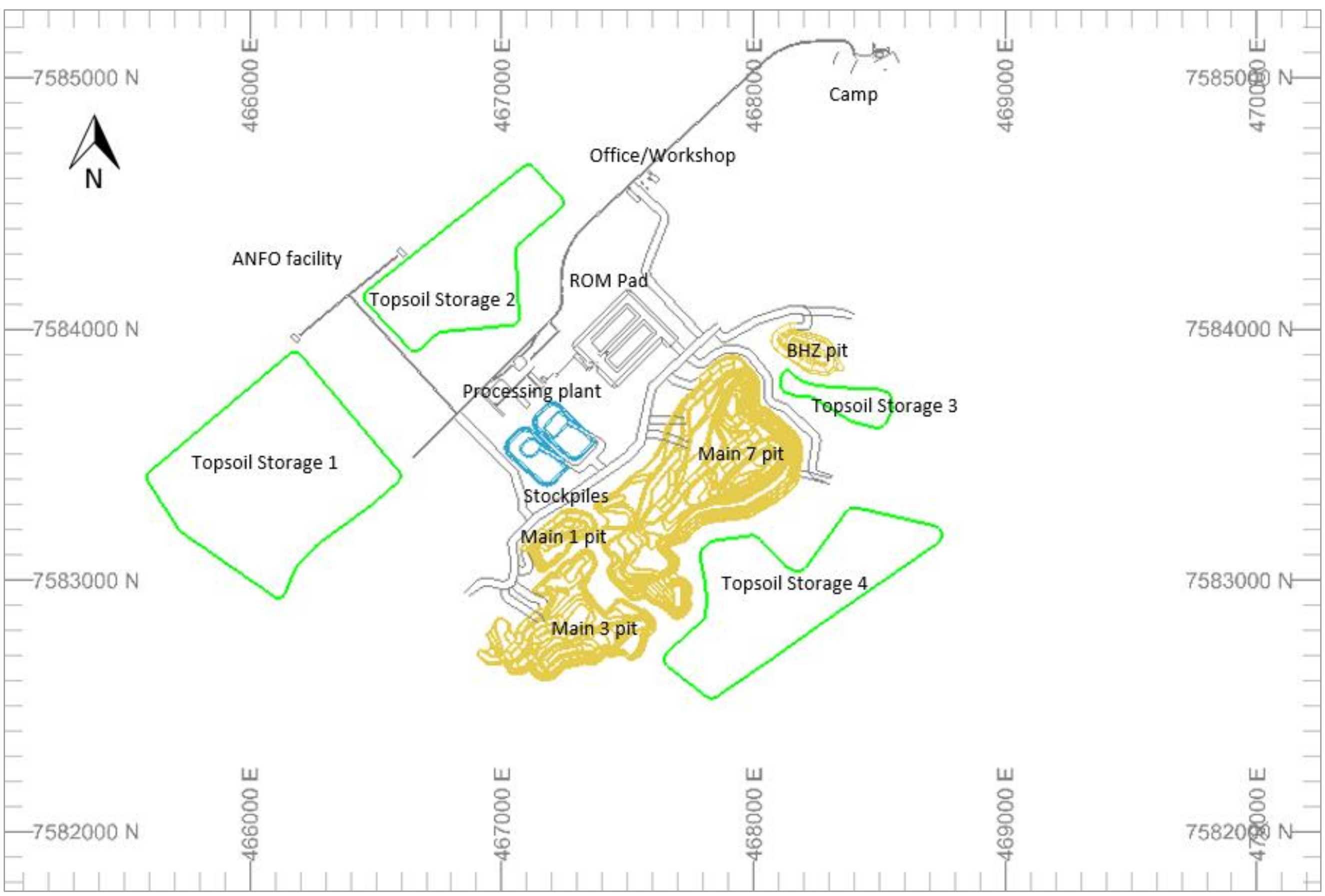
16.3.7 Layout of other mining related facilities

Three out-of-pit dumps, waste dumps 1 to 3, will store waste mined from the pits. Waste dump 2 will extend over the BHZ pit as it is mined-out and tailings backfilling operations are concluded. AMC recommends that a geotechnical stability analysis is conducted on Waste dump 2 in the next study stage to ascertain if any specific geotechnical guidelines would have to be adopted for this concept.

Three in-pit dumps will be created in Main 7 pit in order to reduce hauling costs as the out-of-pit dumps grow in size and the mined-out voids can be safely backfilled. The general site layouts excluding and including waste dumps are presented in Figure 16.11 and Figure 16.12 respectively.

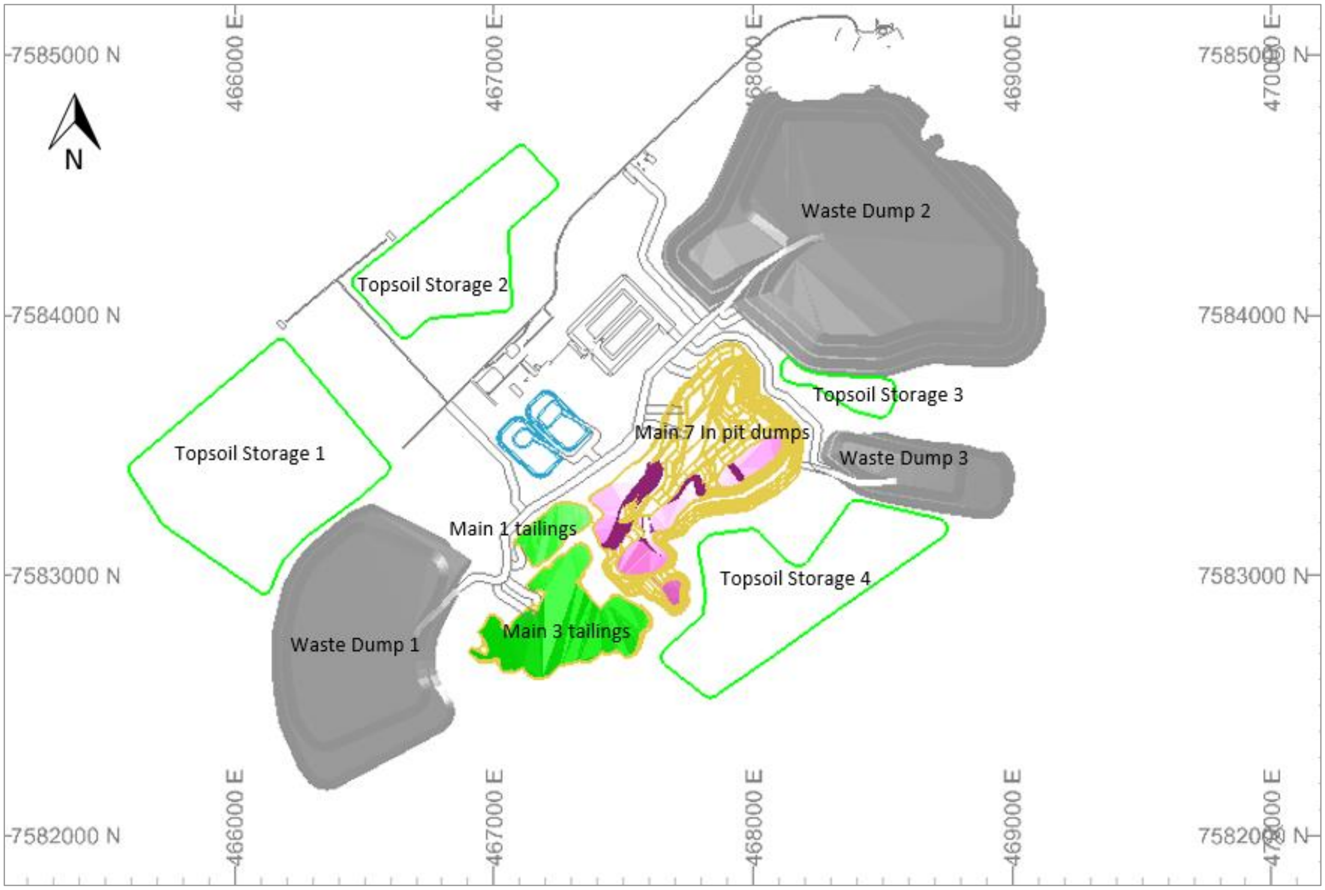
Approximately 50 cm of top soil will be removed from the surface of the footprint of the pits, mine roads and out-of-pit dumps and stored in approximately 2 m high stockpiles for future rehabilitation. Four top soil storages will be built. The ROM stockpile has been designed to accommodate up to 500 kt of mineralized material.

Figure 16.11 General site layout excluding waste dumps



Source: AMC.

Figure 16.12 General site layout including waste dumps



Source: AMC

16.3.8 Open pit mining equipment

The projected mining schedule (see Section 16.6) was used to derive the equipment requirements. The following assumptions were used to estimate loading, hauling, and drilling equipment:

- Equipment availability of 85%
- Use of availability of 88%
- Operating efficiency of 92%
- Derived utilization of 69%
- 6,028 operating hours per annum

Open pit primary equipment requirement at peak production is summarized in Table 16.4.

Table 16.4 Open pit primary equipment

Equipment type [1]	Peak No. required
Excavator for mineralized material (PC1250)	1
Wheel loader (WA600)	1
Shovel for waste (PC2000)	2
90 t haul truck (CAT777)	11
Production drill (D65)	4
Dozer (300HP)	4
Grader (CAT 140M)	2
Water truck	2
Total	**27**

Note: [1]Equipment models indicated are for sizing and costing purposes only and are not meant to be recommendations regarding equipment manufacturer for purchasing decisions.

16.3.9 Open pit mining personnel

The total number of personnel required was estimated based on the production throughput of the operation and the equipment numbers. It is assumed that the management and technical staff will be part of the owner's team. Contractor personnel numbers were estimated for mine supervision, mine operations and maintenance.

Total operator numbers were calculated based on the number of machines on site at any given time. Equipment such as trucks, excavators, drills and dozers are considered to be manned at all times.

Management, technical staff and some maintenance positions were assumed to work on an 8 on 6 off roster while mining labour is assumed to work on a two-weeks on one-week off basis. AMC has assumed two 12 hour shifts per day.

The open pit mining personnel is shown in Table 16.5.

Table 16.5 Open pit manning

Open pit manning	Peak No. required
Owner – Mining Supervision and Technical Services	
OP Mining Manager	1
Technical Services Manager	1
Mine Safety	4
Maintenance Superintendent	3
Technical Services Superintendent	1
Mine Engineer	3
Geologist	2
Surveyor	4
Sub total	**19**
Contractor – OP Supervision and Production	
Project Manager	2
Operation Supervisor	3
Mine Trainer	2
Excavator / FEL operator	15
Truck operator	33
Drill operator	12
Grader / Dozer operator	18
Ancillary equipment operator	24
Sub total	**109**
Contractor – maintenance	
Boilermaker	3
Electrician / Maintenance Planner	5
Clerks	4
Mechanic	36
Trade Assistance	3
Laborer / Forklift / Crane Operator	3
Sub total	**54**
Total OP personnel	**182**

16.4 Open pit to underground interface

In order to optimize the open pit to underground interface, AMC generated incremental pit shells and determined open pit value, which was then compared to the value generated by underground stopes on a level by level basis. If the pit value is higher than the value of the underground, after accounting for operating and development costs, the pit is selected. This analysis is repeated as the pit deepens until the optimal depth is reached.

After selection of the optimal depth of the pit relative to underground stopes, a combined value is determined to confirm that the selected depth generates the maximum value. Stopes are then clipped to the pit design to determine tonnes and grade for the underground mine.

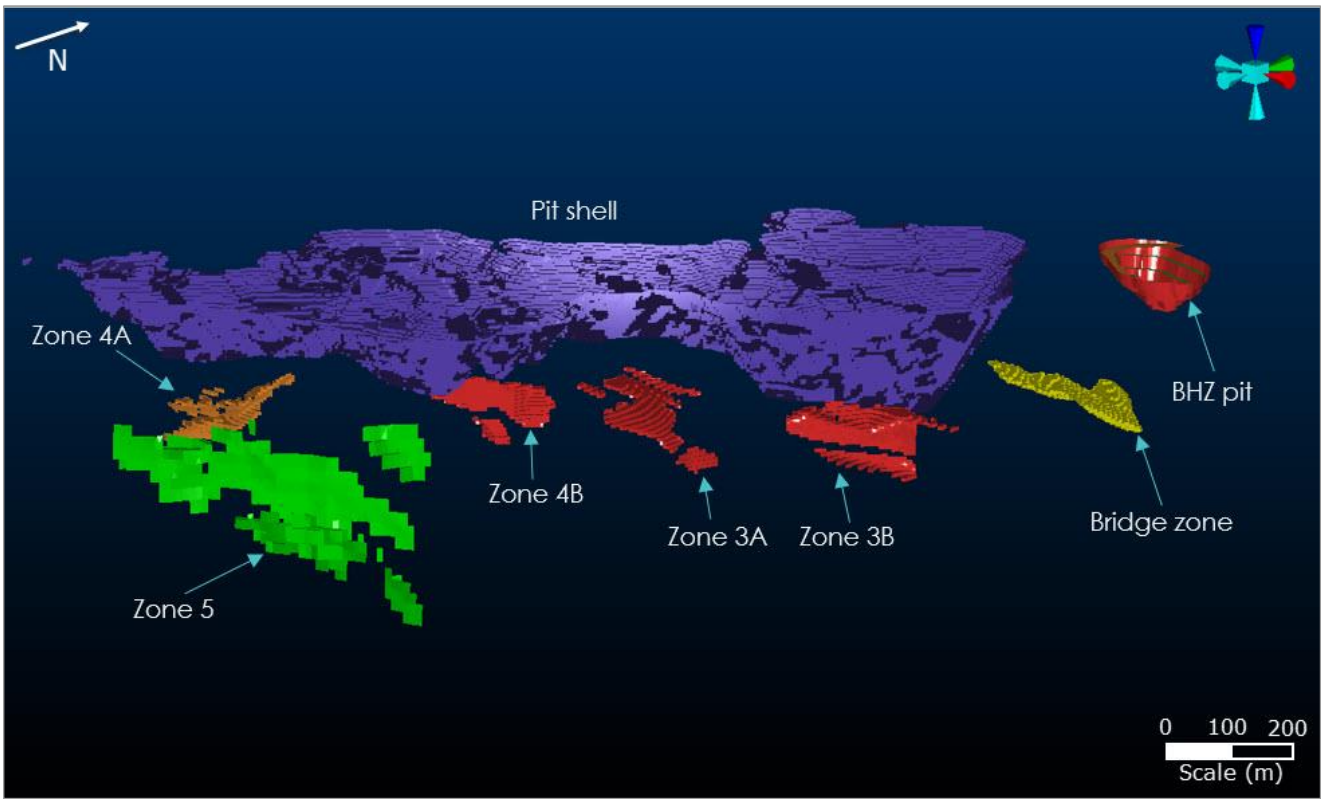
VTT indicated that a 20 m crown pillar shall be left beneath the Main Zone pit and the underground stopes. The size of the crown pillar will require confirmation with the pertinent geotechnical investigations in the future.

After this analysis AMC determined that the Bridge Zone and part of Zone 3 could be optimally mined from underground. Zone 3 has two distinct areas, one directly beneath the Main pit (termed Zone 3A) and one to the east side of the Main pit (Zone 3B). The upper portion of the Zone 3B mineralization adds more value when mined using an open pit mining method with the lower portion of Zone 3B mined from underground.

AMC notes that Zone 4A and Zone 5 are at a very preliminary stage of exploration and are not considered further in this PEA.

Figure 16.13 shows the underground zones and preliminary stope design shapes.

Figure 16.13 Underground zones



Source: AMC

16.5 Underground mining

To assess the underground mining potential, a pit shell was generated that incorporates the potential open pit reserves for the Main Zone, all mineralized material outside of the pit shell was considered for underground mining with the focus on the Bridge Zone.

The geometry of the underground resources is primarily flat dipping (23° to 30°) and varying in thickness across each zone (3 m to 12 m). However, the Zone 5 mineralization and part of Zone 3B tends to be narrow in width and fairly vertical in dip.

The geometry lends itself to room and pillar (R&P) mining in the flat dipping zones and longhole stoping in the steeply dipping zones.

16.5.1 Resource model for underground mining

AMC developed the Mineral Resource sub-celled block models for the Main and Bridge zones (reference: Peg_bm_2018_new_pit.dm and brz_bm_jun2018.dm) for evaluation of the underground mining potential.

16.5.2 Underground geotechnical considerations

The Property mineralization is hosted in a BIF. The BIF's consist of banded quartz-magnetite-fayalite-garnet-grunerite-hedenbergite-sulphide. Apatite, gahnite, and graphite are common minor minerals. Bedding is typically on a scale of 1 mm to 5 mm. In fresh rocks the main sulphide minerals are galena, sphalerite, with subordinate pyrrhotite, pyrite, and chalcopyrite. Garnet quartzite can be present at the lateral extents of the BIF, where the BIF has apparently been attenuated due to folding.

Minimal work has been done on the geotechnical aspects of underground mining. Assumptions for extraction of R&P stopes and design parameters for longhole stoping are based on the knowledge that in general, the host rock is considered to be "Good" with highly competent hangingwall and footwall. It is assumed that minimal ground support is required for the R&P operations and the longhole stopes will be filled with waste rock.

16.5.3 Preliminary underground cost assumptions

Based on the likely mining method selection, AMC used benchmark cost models for Australian metal mines over a range of production rates per day. The benchmark models represent R&P mining and range from 0.4 Mtpa to 5.0 Mtpa. AMC has based the cost on a throughput of approximately 1.0 Mtpa which represents the maximum for the combined production rate for the three areas under consideration, namely the Bridge Zone and Main Zones 3A and 3B.

AMC assumed that all mining will be undertaken by a contractor. The contractor will be responsible for providing and maintaining appropriate mining equipment. Benchmark costs for R&P mining at a production rate of 1 Mtpa are between A$50/t and A$60/t. Given the competent host rock and the minimal need for ground support the R&P operating cost selected for this study is A$50/t of mineralized material.

Minimal tonnage is mined using longhole stoping from Zone 3B and a preliminary benchmark cost of A$50/t is also assumed for mining in this area.

16.5.4 Cut-off value

For the underground operations an NSR field is generated in the Mineral Resource model. This value is used to select the cut-off value for the underground mine. The key parameters used to determine the NSR are provided in Table 16.6 and Table 16.7. An exchange rate of US$0.75:A$1.0 was used. Operating costs assumed are A$50/t of mineralized material for mining, A$28.49 for processing and A$2.0 for G&A for a total of A$80.49. Processing costs and G&A were estimated by GR Engineering Services Limited (GRES).

Table 16.6 Key parameters used to determine NSR for Bridge Zone

Metal	Price	Recovery (Pb Con)	Recovery (Zn Con)
Lead	US$0.95/lb	88.0%	3.0%
Zinc	US$1.05/lb	7.4%	78.5%
Silver	US$16.5/oz	92.3%	5.3%

Table 16.7 Key parameters used to determine NSR for Zone 3A and Zone 3B

Metal	Price	Recovery (Pb Con)	Recovery (Zn Con)
Lead	US$0.95/lb	89.7%	2.5%
Zinc	US$1.05/lb	9.3%	73.7%
Silver	US$16.5/oz	86.0%	2.3%

16.5.5 Mineable Shape Optimizer (MSO)

AMC used a function of the Datamine™ software, MSO to evaluate preliminary stope wireframes for the R&P and longhole mining methods. The following parameters were adopted to generate stope wireframes above a cut-off value of A$80.5/t of mineralized material (Table 16.8).

Table 16.8 MSO parameters

MSO parameter	Unit	R&P (Bridge Zone)	R&P (Zone 3A & 3B)	Longhole mining (Zone 3B)
Stope height	m	3	5	20
Stope width	m	Varying*	Varying*	Varying**
Stope length	m	5	5	5
Cut-off value (NSR)	A$/tonne	80.5	80.5	80.5
Hangingwall / footwall dilution thickness	m	0	0	0
Hangingwall / footwall dip angle	°	60 – 120	60 - 120	60 - 120
Drive height in mineralization	m	3	5	5
Drive width in mineralization	m	3	5	5

Notes:
* Minimum and maximum allowable stope width is 5 m and 275 m respectively.
** Minimum and maximum allowable stope width is 2 m and 30 m respectively.

16.5.6 Dilution and mining recovery factors

There are two main sources of dilution in underground stopes:

- Planned dilution. This is the dilution required to achieve the designed stope shape. Designed dilution can result from waste included:
  - To achieve minimum mining width.
  - To achieve a viable mining shape.
- Unplanned dilution. This is dilution that is outside of the designed stope shape. Depending on the mining method, it may include both overbreak and floor dilution.
  - Overbreak is typically a result of blasting practices and geotechnical conditions and can include backfill from the adjacent stope walls.
  - Floor dilution is the result of mucking backfill from the stope floor.

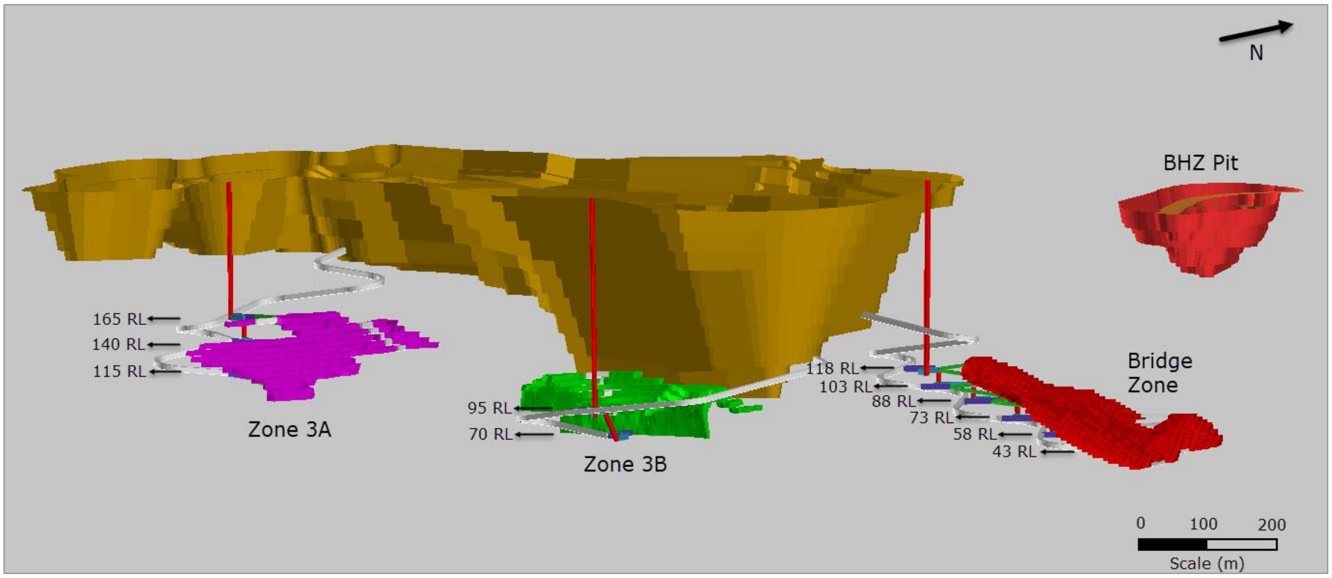
For R&P AMC has applied a dilution factor of 10% at zero grade to the Mineral Resource and a mining recovery factor of 86% has been applied to the stopes, 100% recovery is assumed for the mineralized material from development.

For longhole mining AMC has applied a dilution factor of 12% at zero grade to the Mineral Resource and a mining recovery factor of 95% has been applied to the stopes, 100% recovery is assumed for the mineralized material from development.

16.5.7 Underground mine access and design

It is assumed that the underground zones will each be accessed from the Main pit with individual declines. AMC has developed conceptual mine designs for each zone to determine mine physicals and development costs. The mine design for each zone is shown in Figure 16.14.

Figure 16.14 Underground mine design for all zones



Source: AMC

16.5.8 Underground development

The three main access declines have a total length of 2,166 m, level access development has a total length of 1,786 m and the remaining development includes return air drive access and mineralized development. Vertical development consists of return air raises with a total of 698 m. Total development for the three underground zones including vertical development is 5,547 m.

16.5.9 Stope design

Stope wireframes were generated using MSO, a check was made to remove any outlying stopes that would not be economic when the cost of access development was included. The cost of access development was determined for each level and each level was evaluated to determine whether it was economic to develop. The potential mining inventory associated with the potentially economic stopes is summarized in Table 16.9.

Table 16.9 Potential mineralized material for extraction from underground

Area	Tonnes (t)	Ag (oz/t)	Pb (%)	Zn (%)	NSR (US$)
Bridge Zone	919,075	11.0	7.1	1.9	163
Zone 3A	459,049	7.9	5.2	2.4	135
Zone 3B	360,831	12.0	5.4	1.3	124
Total	**1,738,955**	**16.0**	**3.5**	**6.7**	**148**

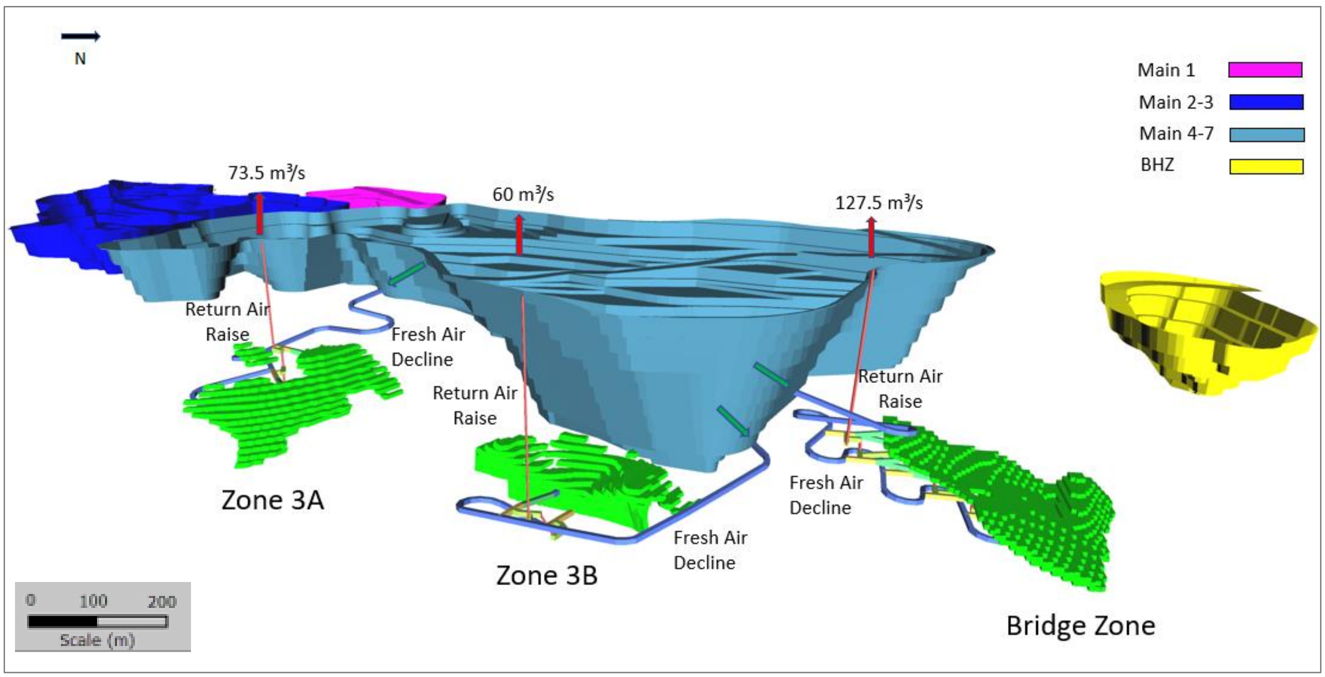
16.5.10 Ventilation

The function of the ventilation system is to dilute / remove airborne dust, diesel emissions, gases from explosives, and to maintain temperatures at levels necessary to ensure safe production throughout the life of the mine. AMC has undertaken a preliminary estimate of the ventilation requirements in consideration of the production rate, material handling and mining method. AMC estimated that the total mine airflow should be 261 m^3/s.

The mine will be ventilated by a "Pull" or exhausting type ventilation system. That is, the primary mine ventilation fans will be located on surface at the primary exhaust airways of the mine. Fresh air will enter the mine via the internal declines from the portals inside the pit and exhaust to the surface via dedicated return airways. Most production activities will require auxiliary fans and ducting with level airflows managed through regulators located at raise accesses.

Fresh air for production activities will be distributed via fresh air decline and exhausted via dedicated return air drives that connect to 2.4 m diameter raise for the Zone 3A, 2.1 m diameter raise for Zone 3B and a 3 m diameter raise to surface in the Bridge Zone. Fans for each raise will be located on surface. Figure 16.15 shows the ventilation strategy for the three zones.

Figure 16.15 Overview of the ventilation system arrangement



Source: AMC

Primary fan sizing estimate was based on:

- Raise diameter and length.
- Maximum raise airflow.
- Estimated frictional resistance assuming raisebore development.
- Estimated fan efficiencies.

Table 16.10 shows the primary fan requirements.

Table 16.10 Primary fan selection

Zone	3A	3B	Bridge
Description	**Return air raise**	**Return air raise**	**Return air raise**
Airflow (m^3/s)	73.5	60	127.5
Pressure (Pascal)	1,047.0	1,099.0	2,305.2
No of fans per installation	1	1	2
Vent raise diameter (m)	2.4	2.1	3.0
Arrangement	Single	Single	Parallel
Each fan motor size (kW)	115.3	99.7	208.0

The auxiliary fan sizing estimate was based on:

- Largest equipment concurrently operating in a heading.
- Duct diameter.
- Maximum duct length.

Table 16.11 shows the auxiliary fan requirements for each zone.

Table 16.11 Auxiliary fan selection

Description	Decline development (all zones)	Production heading (Zone 3a)	Production heading (Zone 3b)	Production heading (Bridge Zone)
Equipment	LH514 Loader TH545i Truck	LH514 Loader	LH514 Loader	LH514 Loader
Duct diameter	1.07 m	1.07 m	1.07 m	1.07 m
Maximum length	400 m	150 m	125 m	125 m
Each fan motor size	55 kW	37 kW	37 kW	37 kW
Number Required per zone	2	8	4	12

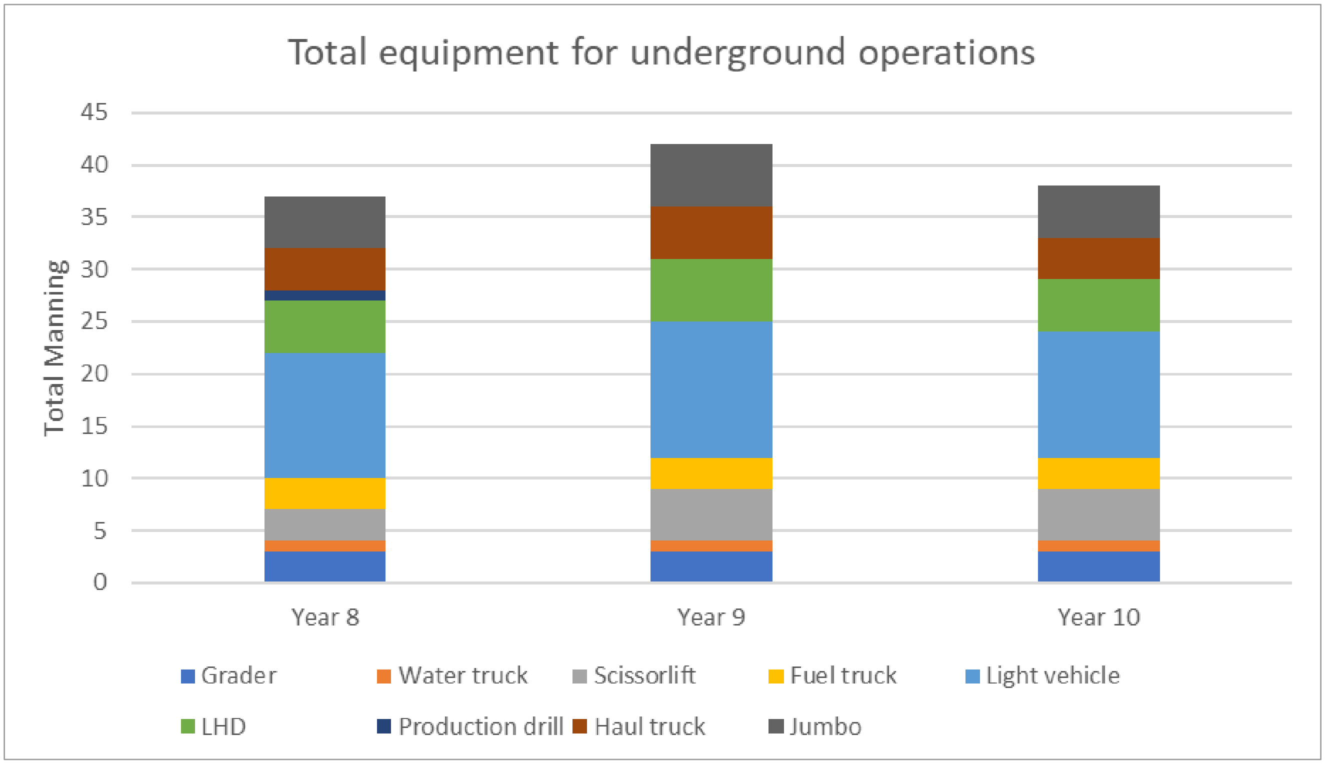
16.5.11 Backfill

There will be no backfill in the R&P areas, it is assumed that waste rock from development will be placed as backfill in the longhole stopes.

16.5.12 Underground mining equipment

AMC has completed an estimate of the main equipment required to meet the production rate (see Section 16.6) for the combined underground mining zones. A summary of the equipment by year is provided in Figure 16.16. It is assumed that the equipment will be supplied by the contractor as the open pit production ramps down the underground production ramps up.

Figure 16.16 Summary of major underground mining equipment

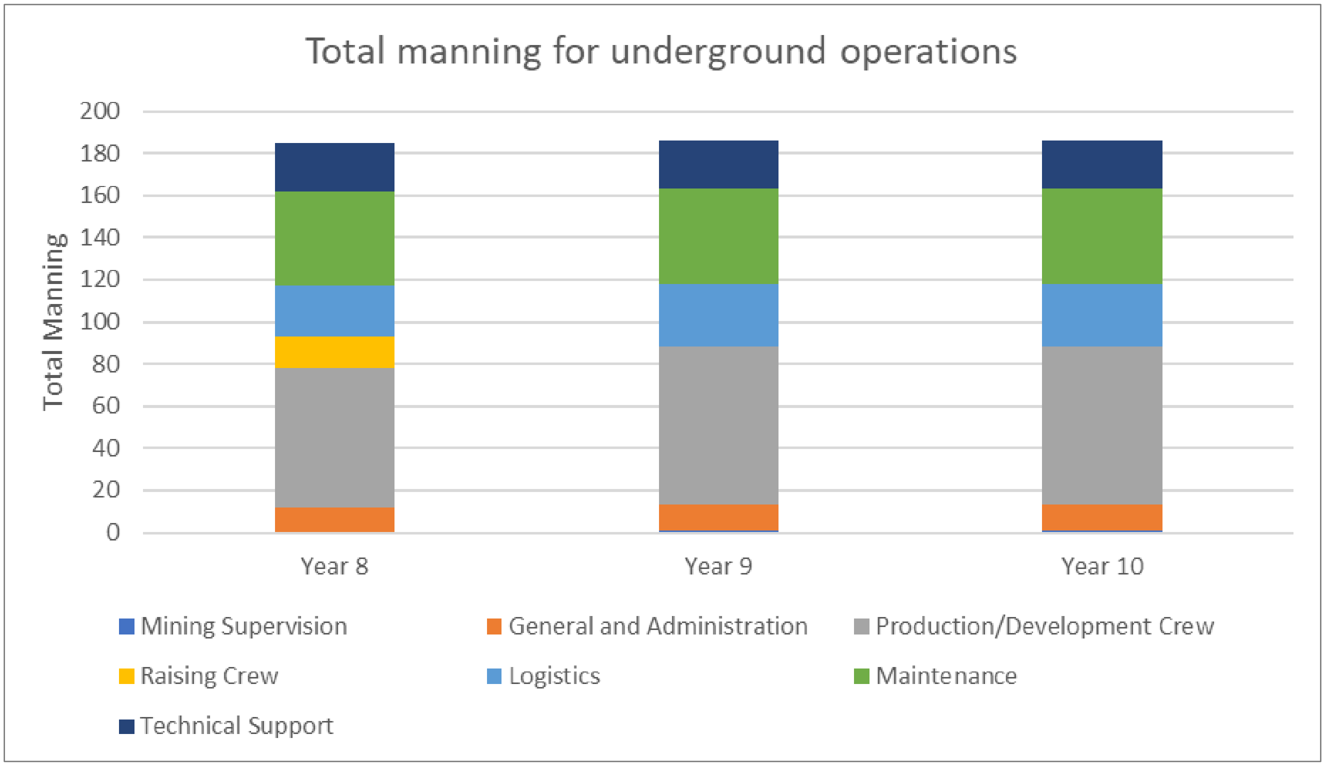


Source: AMC

16.5.13 Underground mining personnel

It is assumed that the underground mines will be operated by a contractor. To determine quantities for infrastructure and planning, AMC has provided an estimate of the manning required to meet the planned production rate from the three zones. The estimate assumes that the labour will operate on a 2 week on 1 week off roster. The maximum number of personnel hired for the underground mine operations will be 202, this includes an 8% allowance for absenteeism. Figure 16.17 provides a summary of the total underground mine manning requirements excluding absentee allowances.

Figure 16.17 Summary of underground manning



Source: AMC

16.5.14 Underground infrastructure

16.5.14.1 Power

There is a high-pressure natural gas line located approximately 16 km to the south of the project. The line runs between Osborne and Cannington Mine sites. Electrical power will be generated by gas fired generator sets, each rated at 2,500 kW at full load and expected to run at 80% load or 2,000 kW each.

Power for the project will be reticulated throughout the site at 11 kV. At each of the mine entrance portals there will be a junction box with a disconnect switch. From the portals, power will be carried underground via a 3C armoured cable. Portable 500 kVA MLC with stepdown transformers and 1,000 V distribution, will be located at each active level access. A junction box with a switch will allow individual transformers to be isolated, so they can be removed from service with minimal interruption of power in the remainder of the mine. Once a level is completed, the MLC can be moved to the next active heading.

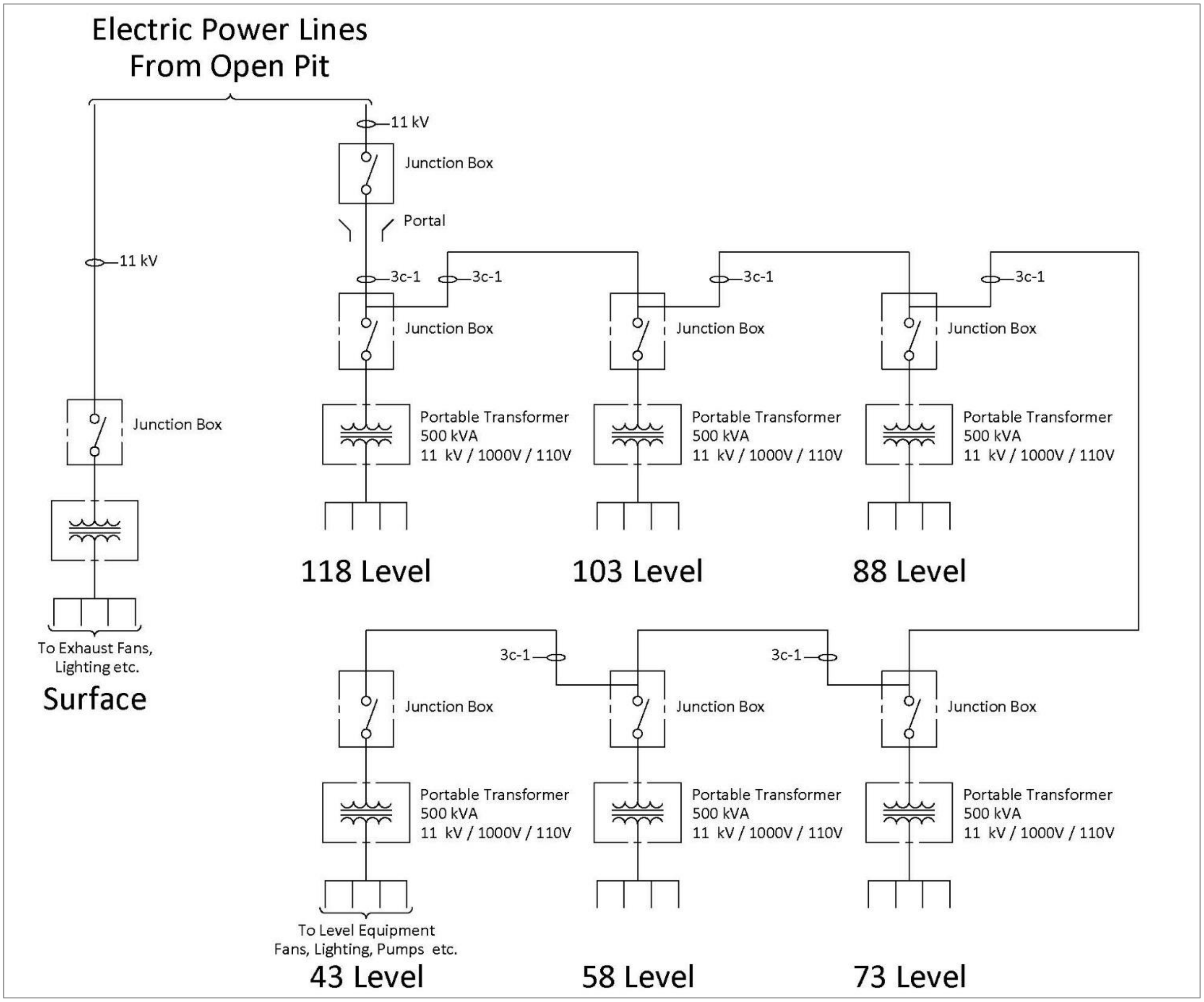
The primary power demand for the underground mine is associated with the dewatering pumps, the fresh water supply pump, and the main fans located at the top of the primary exhaust raises. All the underground zones will be in operation at the same time, the maximum demand of approximately 1.9 MW will be required for the underground operations.

Electrical block power distribution diagrams for each zone are provided in Figure 16.18 to Figure 16.20. A summary of the peak and average power demand by activity is provided in Table 16.12.

Table 16.12 Summary of underground power demand

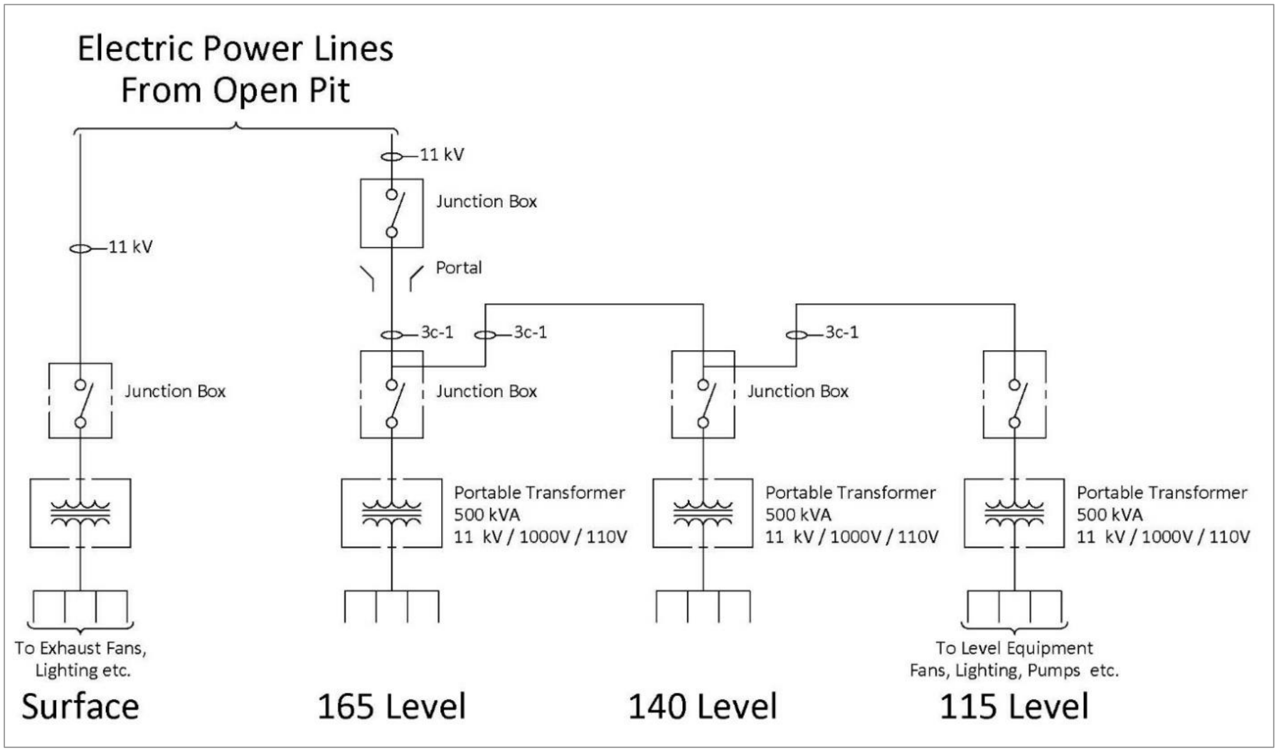
Zone	Bridge		3A		3B	
Description	**Peak power (kW)**	**Average power (kW)**	**Peak power (kW)**	**Average power (kW)**	**Peak power (kW)**	**Average power (kW)**
Mine dewatering and water supply	283	242	189	161	94	81
Mine Ventilation	625	576	308	275	230	207
Other underground infrastructure	631	188	484	129	382	70
Power	**1,538**	**1,006**	**981**	**565**	**706**	**357**

Figure 16.18 Bridge Zone underground electrical block power distribution diagram



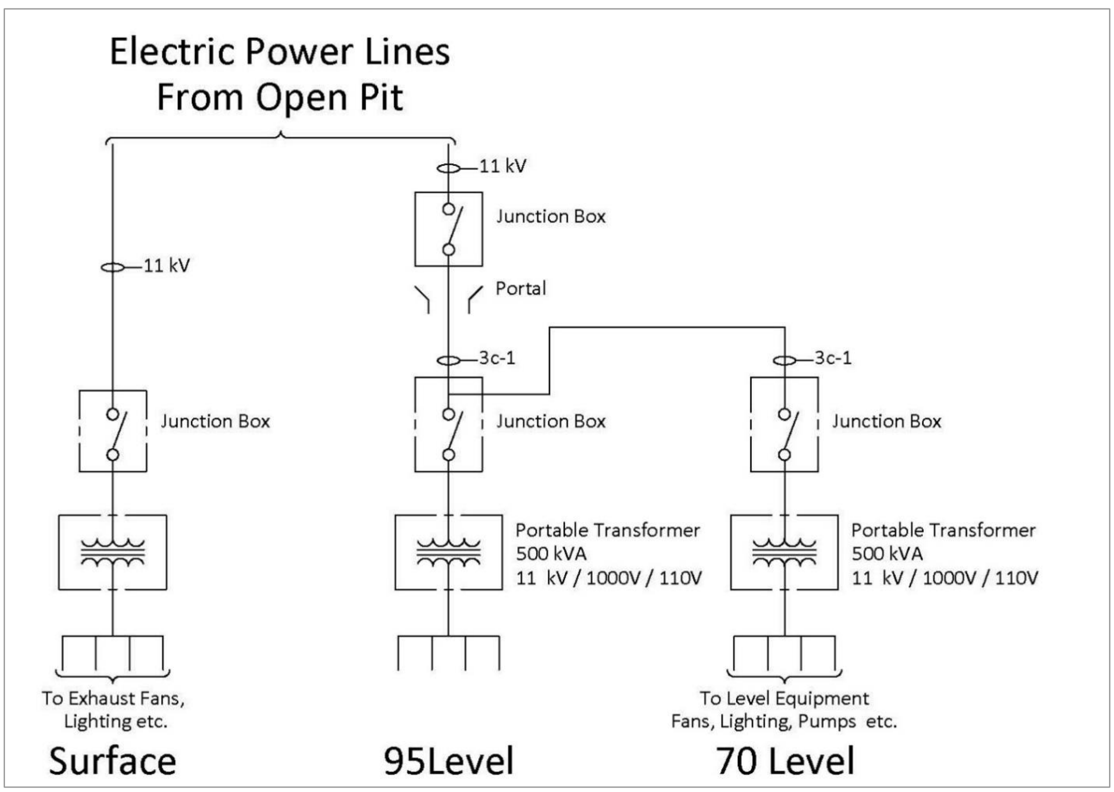
Source: AMC

Figure 16.19 Zone 3A underground electrical block power distribution diagram



Source: AMC

Figure 16.20 Zone 3B underground electrical block power distribution diagram



Source: AMC

16.5.14.2 Potable water

Potable water will be generated onsite from the raw water supply via a reverse osmosis plant before being pumped to the plant and mining amenities as well as the accommodation village.

16.5.14.3 Service water

Raw water for the Project shall be sourced from a dedicated bore field. A 4-inch HDPE line will be installed in stages down the declines to provide service water for use in the mine. Every 40 m vertically, a pressure reducing valve will be installed to control the pressure in the line. A combination of hoses and 2-inch HDPE piping will extend out onto the levels to provide utility water. Service water will be required for drilling, and watering muck piles, and access routes for dust control. A minimum requirement of 2 l/s of service water is required to operate the underground equipment.

16.5.14.4 Dewatering

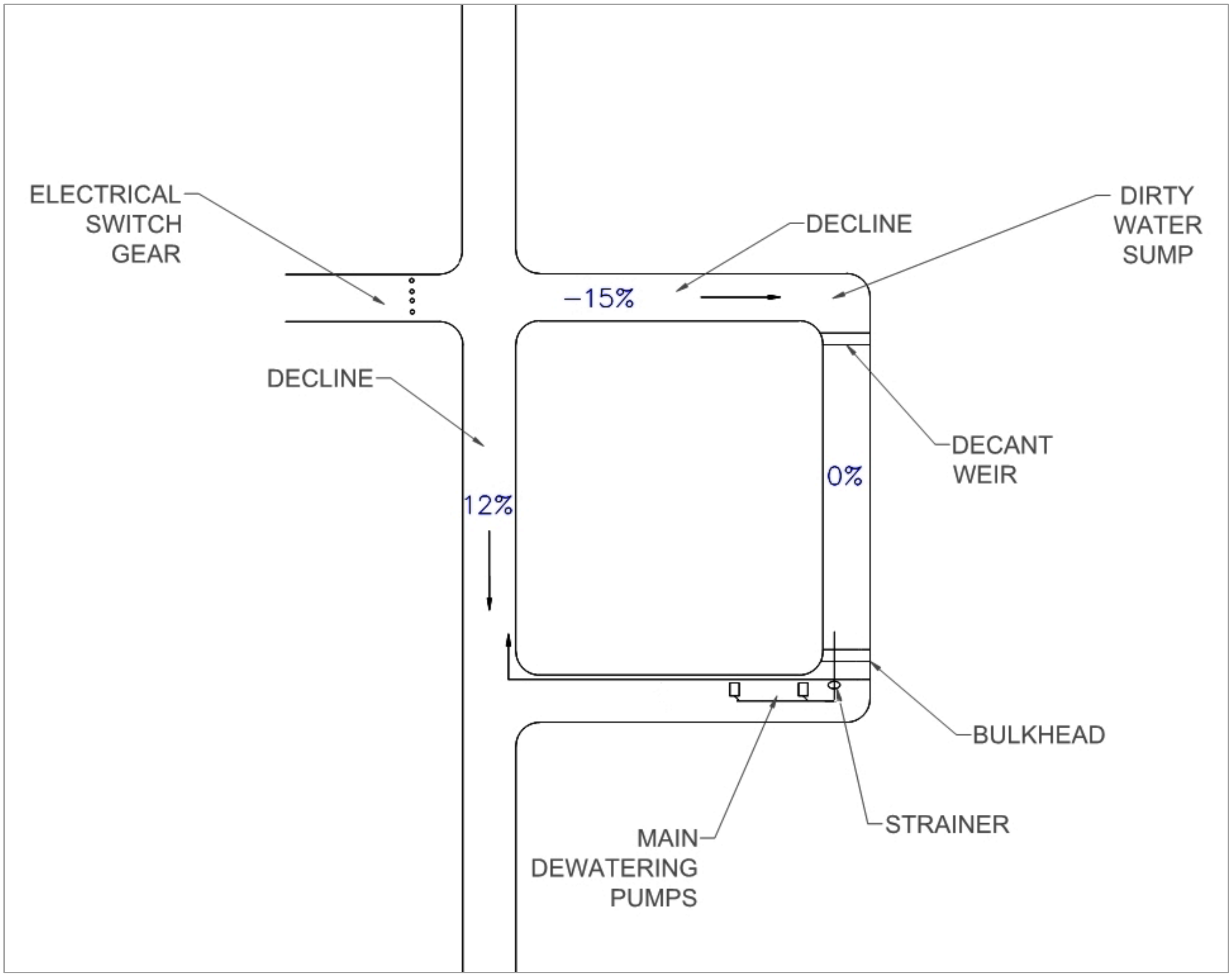
Based on a nearby aquifer and water required for carrying out mining activities a total capacity of 10 l/s is required for the dewatering system for each mine.

During development of the declines and prior to establishment of the main sumps, any water will be collected and discharged using submersible pumps. The pumps will be arranged in a cascading fashion to bring the water to the portals. Submersible pumps and pipes have been costed for this purpose. The three mines will have a main dewatering sump installed near the intersection of the main access decline and the lowest access to the main working levels. At the main sump there will be two – 75 kW driven centrifugal multi-stage discharge pumps, one running and the other on standby. This provides an online spare pump, and in an emergency upset condition can provide additional capacity to the system.

Water will flow from the stoping levels to level access sumps. Drain holes will be drilled between the sumps to transfer water down to the main dewatering sump at the lowest elevation of the mine. Mining lifts below the decline and stope access intersection will use a submersible trash pump to bring the water to the level sump.

Mine water will be placed into a dirty water sump where a decant weir allows solid particles to settle out. Cleaner water will be allowed to decant into a holding sump. Water will be then pumped up to the portal in a 150 mm Schedule 40 pipe where it will discharge into existing open pit dewatering system. Figure 16.21 shows a plan view of a typical main dewatering sump.

Figure 16.21 Plan view of the main dewatering sump



Source: AMC

16.5.14.5 Compressed air

Portable air compressors (one for each zone in operation) will be moved together with the primary mining equipment. The compressors will be sized so that they will be able to supply four operating drills.

16.5.14.6 Communications

A leaky feeder system will provide means for communication underground. All vehicles will be fitted with radios. The underground mine personnel will also have radios and cap lamps equipped with emergency warning systems.

16.5.14.7 Workshops and magazines

The workshop located on surface and surface magazine will be used to support the underground workings.

16.5.14.8 Mine escape and rescue

Portable 20 personnel refuge stations will be located appropriately relative to operating levels. Lunchrooms near the maintenance area will also serve as refuge stations. Self-rescue storage will be provided in the lunchrooms as well as first aid kits at the refuge stations.

Return air raises will be equipped with the Safescape System and will act as the second egress for each zone. The return air raises between levels will also be equipped with Safescape System and will provide escape routes between levels.

16.5.14.9 Fire detection and suppression systems

The mine ventilation systems will be provided with an ethyl mercaptan (stench gas) system (activated manually or remotely) to warn underground personnel in the event of an emergency. Radio contact via the leaky feeder system provides an alternative method of communication. The main ventilation fans can be shut down or adjusted to assist with fire control systems in the mine.

16.6 Projected open pit and underground life-of-mine (LOM) production schedule

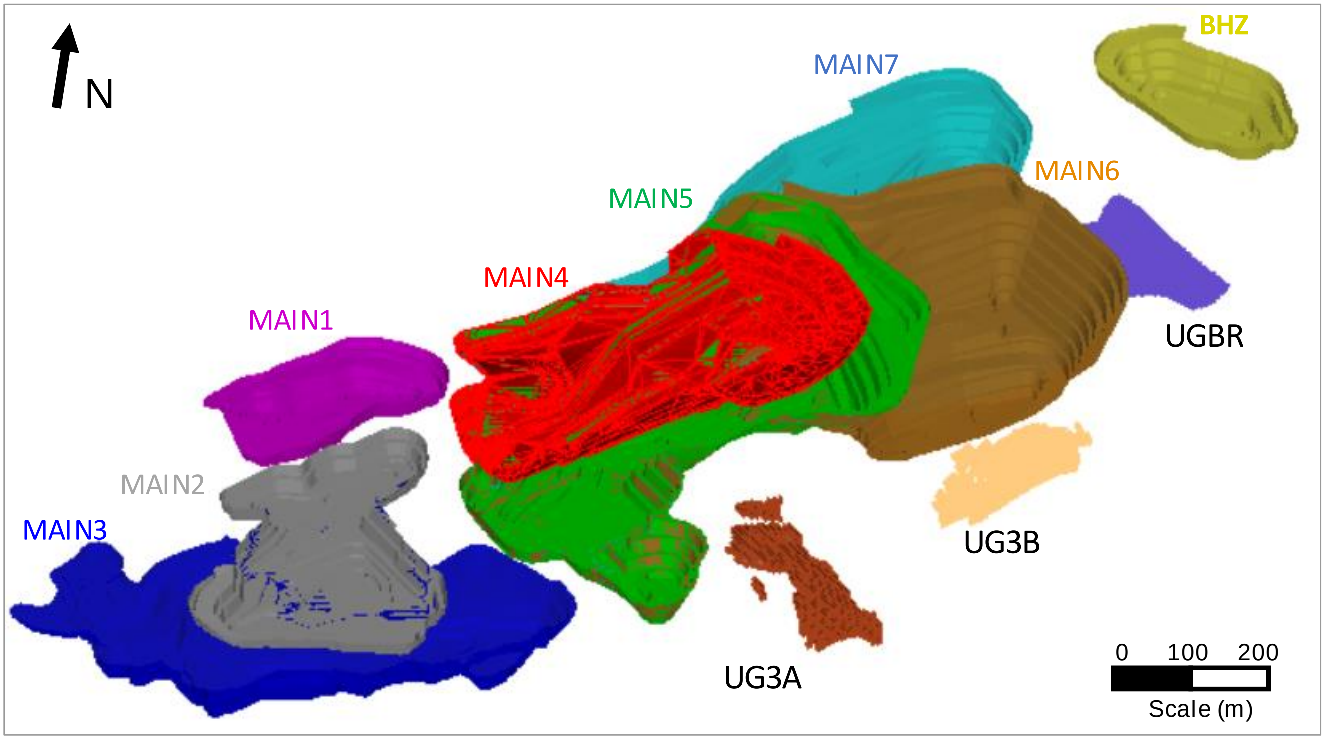
AMC completed a combined open pit and underground mine plan using the Minemax Scheduler 6 (Minemax) software. Minemax seeks to maximize the discounted operating cash flows while honoring constraints related to processing and mining inputs.

The target of the LOM schedule is to maximize the net present value of the plan while maintaining a 1.1 Mtpa steady state processing plant throughput. The schedule was developed on a yearly basis using a discount rate of 8% per annum. To determine the optimum strategy in terms of value, each underground area was evaluated together with production from the open pit operations.

16.6.1 Inventory by mining area

The total inventory includes 121.7 Mt of rock, of which 10.6 Mt are mineralized material at NSR grade of A$135.3/t. The mine plan includes eight open pit stages and three underground zones as shown in Figure 16.22.

Figure 16.22 Open pit and underground mining areas



Source: AMC.

The inventory by mining area and their associated mineralized material and waste quantities is presented in Table 16.13.

Table 16.13 Total inventory

Mining area	**Mineralized material tonnes**	**NSR grade**	**Waste tonnes**	**Total rock tonnes**
	Mt	**A$/t**	**Mt**	**Mt**
BHZ	0.5	108.3	3.2	3.7
Main 1	0.3	146.7	2.9	3.2
Main 2	1.0	108.3	8.3	9.3
Main 3	1.2	123.2	12.8	14.0
Main 4	1.4	135.0	13.5	14.9
Main 5	1.4	145.9	20.8	22.3
Main 6	1.5	161.4	34.4	35.8
Main 7	1.6	120.5	15.0	16.6
UG3A	0.5	134.7	0.1	0.6
UG3B	0.4	123.8	0.1	0.4
UGBR	0.9	163.3	0.1	1.0
Total	**10.6**	**135.3**	**111.1**	**121.7**
Total OP	**8.9**	**132.9**	**110.8**	**119.7**
Total UG	**1.7**	**147.5**	**0.3**	**2.0**

The open pits contain approximately 0.54 Mt of mineralized material above the transition material NSR cut-off grade with grades of 4.57% Pb, 1.44% Zn, and 7.31g/t Ag. This material is not considered in this PEA and has not been assigned any processing recovery.

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

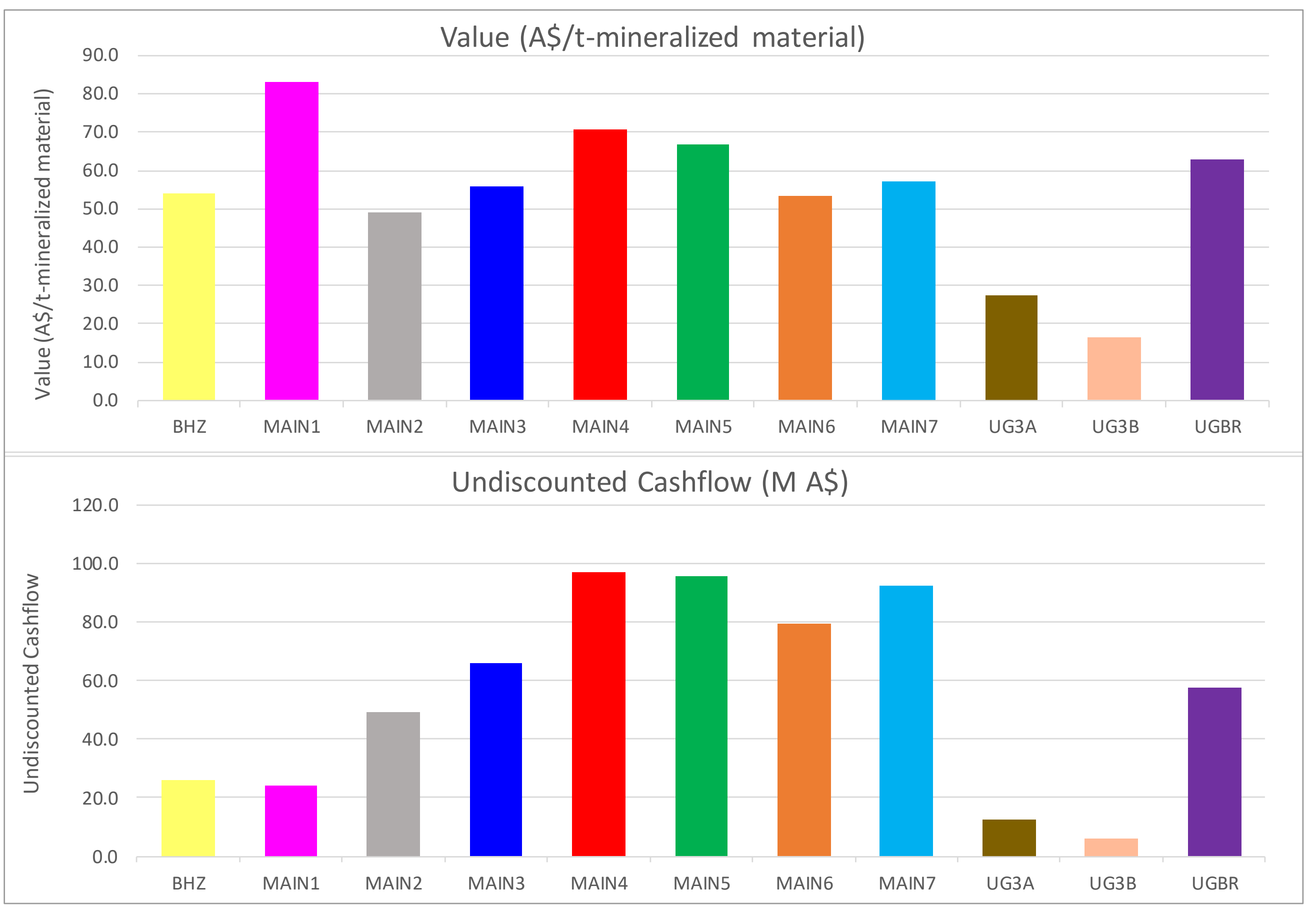
16.6.2 Mine sequence considerations

16.6.2.1 Value of mining areas

In order to optimize the overall value of the project and the sequence of mining, AMC has estimated the value for each pit stage and each underground zone. The value, defined as the indicative undiscounted cashflow per tonne of mineralized material, accounts for preliminary mining costs, G&A, and processing costs. The value of the three underground zones include capital development costs and infrastructure capital expenses.

The projected value from each source and consideration of practical scheduling constraints provided a basis for the order in which the pits and underground zones are scheduled. The indicative value by mining area is shown in Figure 16.23.

Figure 16.23 Indicative value by mining area



Source: AMC

16.6.2.2 Tailings pits

In addition to considering the value by mining area when developing the preferred mining sequence, a focus to mine the three tailings pits (BHZ, Main 1, and Main 3) early in the mine life was maintained in order to enable placement of process tailings. The storage capacity of the three tailings pits are presented in Table 16.14. The capacity equivalent to material processed was estimated based on the pit volume and an approximate generation of tailings equivalent to 85% of ore volume processed having a tailings density of 2 t/bcm. It is assumed that tailings can be placed up to 5 m below the pit crest's lowest point. There is enough capacity to store tailings from all open pits and underground within the three selected tailings pits. AMC assumed that no liner would be needed prior to tailings placement and that the tailings would not affect the water quality of any aquifers. These assumptions would have to be confirmed in further studies. In addition, AMC recommends that condemnation or infill drilling is conducted in advance of mining the tailings pits to ensure that the mineralization is fully understood and mined appropriately, to avoid sterilization of any potential mineralized material.

The BHZ pit has an overall higher value than the Main 1 pit and is scheduled to be mined first. In addition, as BHZ is a secluded open pit it will allow to identify any risks or changes to the tailings deposition method prior to moving to the interconnected Main tailings pits and pushbacks. Waste dump 2 will be extended over the BHZ pit once filled with tailings and properly dewatered.

The combined capacity of both the BHZ pit and the Main 1 pit is equivalent to 3.2 Mt of mineralized material processed, sufficient to store tailings production until Year 3 of production. The Main 3 pit must be completely mined and prepared for tailings disposal by the end of Year 3 to manage additional process tailings.

Table 16.14 Capacity for tailings

Mining area	Volume available for tailings	Capacity equivalent to processed mineralized material
	M m³	Mt
BHZ	1.0	1.6
Main 1	0.9	1.5
Main 3	6.2	10.5
Total	**8.0**	**13.6**

16.6.2.3 Open pit and underground precedences

It is assumed that the underground zones will each be accessed from the Main pit with individual declines as shown in Figure 16.14.

The following open pit and underground precedences are required in order to provide access to underground operations from the Main pit:

- Pit Main 5 to be totally mined before UG3A.
- Pit Main 6 to be totally mined before UG3B and UGBR.

16.6.3 Strategic schedules

Three high-level production schedules were developed to compare various operational scenarios to determine the ultimate strategy for the project. The following scenarios were assessed:

- Scenario 1: Maximum 18 Mt of total OP material mined per year. OP and UG precedences to allow access to UG zones from the Main pit.

- Scenario 2: Maximum 17 Mt of total OP material mined per year. OP and UG precedences to allow access to UG zones from the Main pit.
- Scenario 3: Maximum 17 Mt of total OP material mined per year. No OP and UG precedences.

The parameters and assumptions common to all scenarios are as follows:

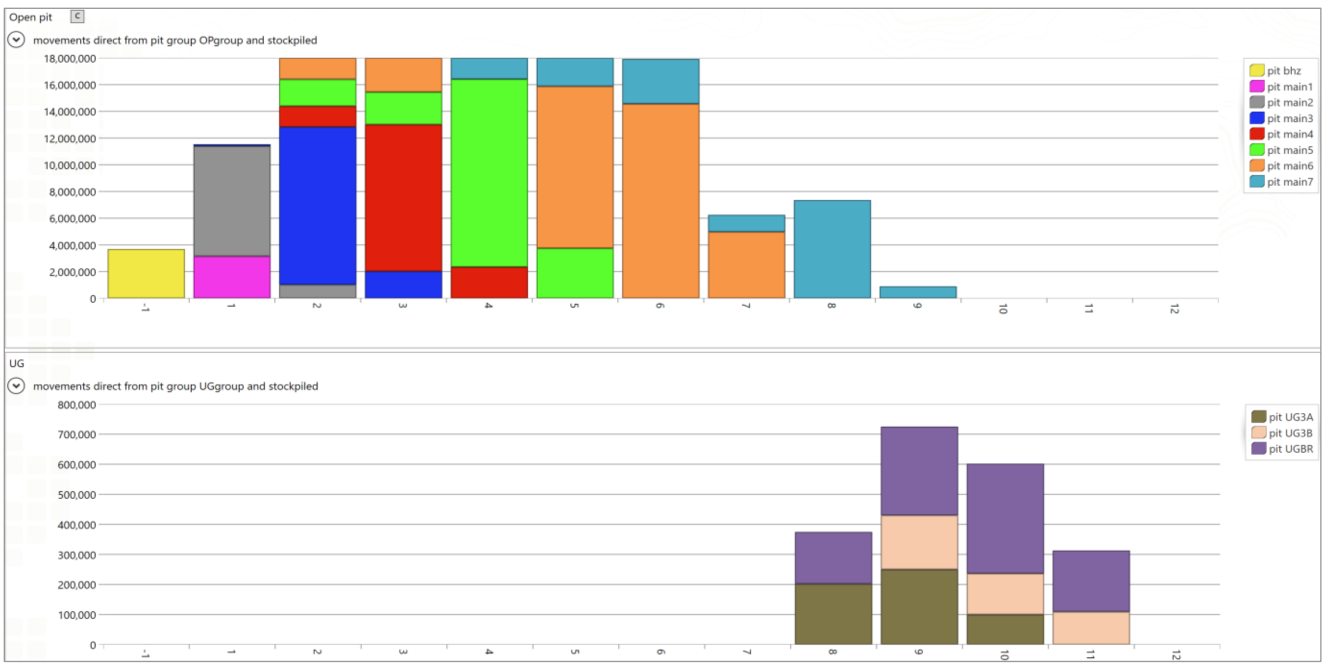
- Yearly scheduling period.
- Process plant capacity of 0.98 Mt in Year 1, and 1 Mtpa from Year 2 onwards.
- Mining starts with the BHZ pit during the pre-production year (Year -1).
- The tailings pit Main 1 is completely mined by end of Year 1 and the tailings pit Main 3 is completely mined by end of Year 3.
- Vertical advance rate limit of 70 m per year.
- No constraints on stockpiling of mineralized material.

For the purposes of comparing the initial scenarios an indicative discounted cashflow was generated including G&A, operating mining, and processing costs. The operating cost estimates are preliminary but differ only marginally from the final costs presented in Section 21; the analysis undertaken is valid on a comparative basis.

The indicative cashflows include underground capital development cost, with decline access from the pit, and infrastructure capex; however, they do not include any other capital costs.

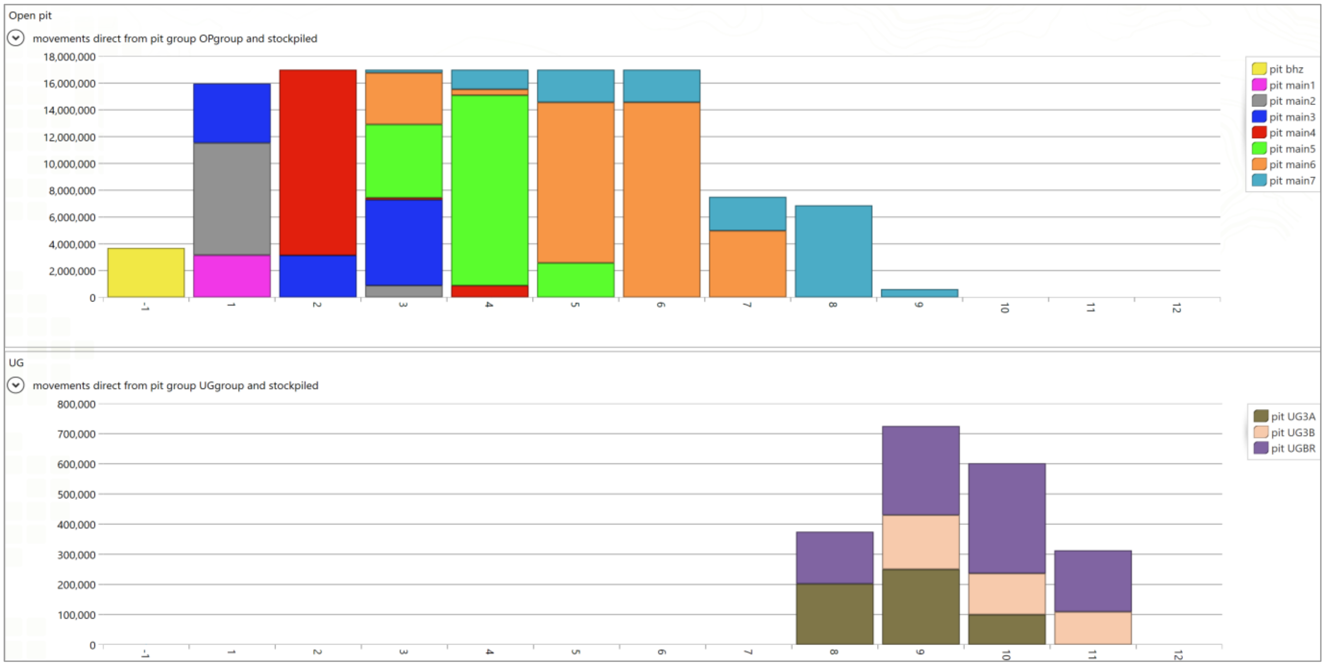
The results of the strategic schedules are presented in Figure 16.24 to Figure 16.27, and Table 16.15.

Figure 16.24 Scenario 1 – sequence and tonnes mined



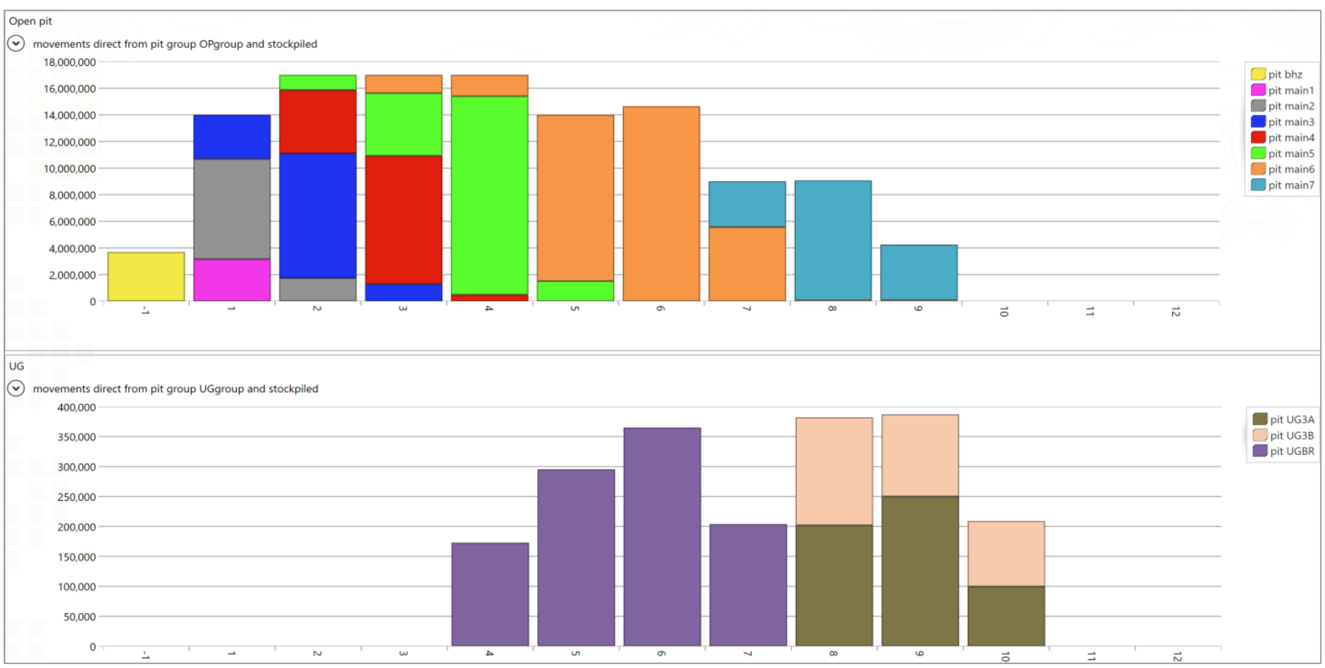
Source: AMC

Figure 16.25 Scenario 2 – sequence and tonnes mined



Source: AMC

Figure 16.26 Scenario 3 – sequence and tonnes mined



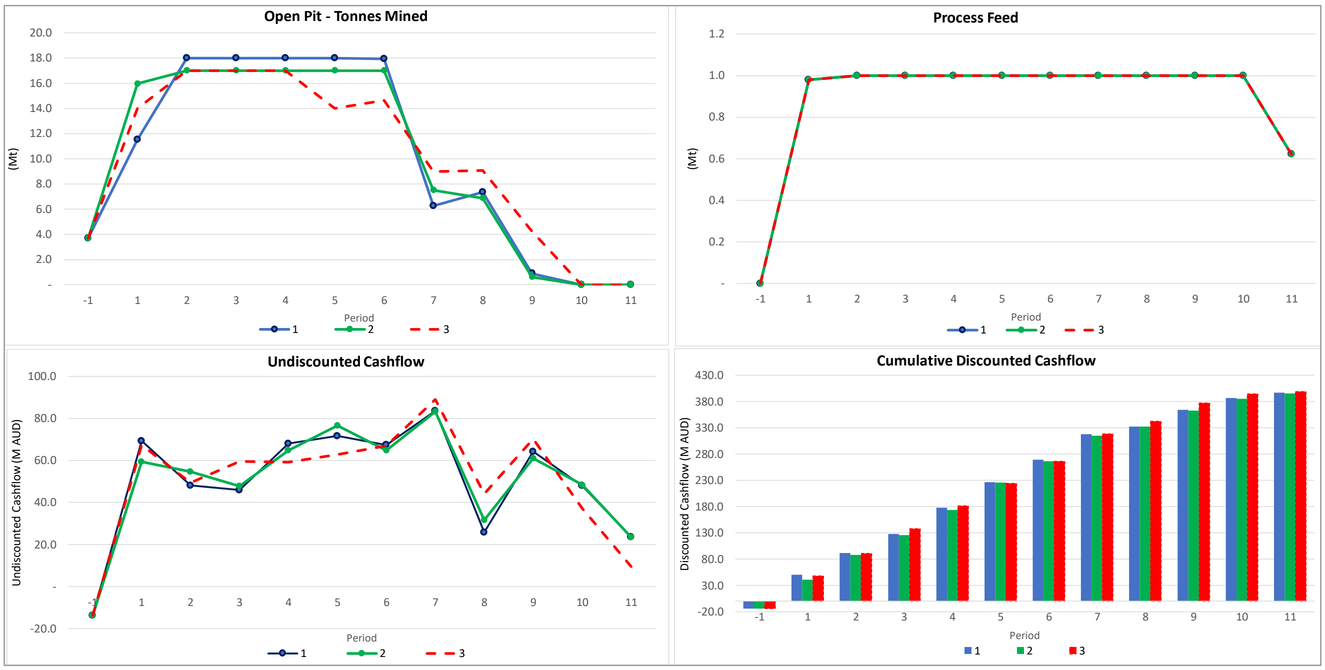
Source: AMC

Table 16.15 Summary of strategic schedules

Scenario	Description	Indicative undiscounted cashflow (A$M)	Indicative discounted cashflow (A$M)	Maximum stockpile size (Mt)
1	Max. OP TMM 18 Mtpa. OP / UG precedences	602.0	396.7	1.1
2	Max. OP TMM 17 Mtpa. OP / UG precedences	602.0	395.3	1.2
3	Max. OP TMM 17 Mtpa. No OP / UG precedences	601.7*	399.0*	1.4

Note: *Scenario 3 cashflows do not include additional cost of decline from surface to Bridge Zone.

Figure 16.27 Scenarios results comparison



Source: AMC

Scenarios 1 and 2 involve UG operations at almost the end of open pit mining, with 1.5 years of overlap, and will have the following operational benefits:

- Full use of existing infrastructure including workshops, magazines, and offices.
- More operational headings and flexibility while mining two to three underground zones concurrently.
- The total underground production rate is higher than Scenario 3, with the benefit of cost savings due to economy of scale.

Scenario 3 involves mining of the underground Bridge Zone starting in Year 4. Scenario 3 generates higher discounted cashflow (A$2.3M and A$3.7M higher than Scenario 1 and 2 respectively). However, Scenario 3 will require an independent decline from surface of 1.3 km in length, resulting in an additional development cost of A$6.5M, which is currently not included in the value presented in the strategic schedules. Scenario 3 would therefore probably be less attractive than Scenarios 1 and 2 on a discounted basis once this cost is included. Scenario 3 operational benefits are as follows:

- UG equipment can be moved at the end of mining of the Bridge Zone to Zones 3A and 3B reducing mobilization costs.

- Some underground infrastructure (i.e. fans, electrical substations, pumps) can be moved from the Bridge Zone to Zones 3A and 3B.
- No OP / UG interaction when mining the Bridge Zone as there will be independent ramp access from surface; although OP / UG interaction should also be minimal for Scenario 1 and 2.

Although all scenarios are equivalent in value given the precision of the estimate, Scenario 2 was selected as the most attractive development strategy since it involves lower open pit material movement and provides greater flexibility for underground operations.

16.6.4 Combined open pit and underground production schedule

After the three strategic schedules were produced to feed the plant at 1 Mtpa, VTT decided to target an increased plant throughput rate of 1.1 Mtpa.

AMC undertook a value analysis to optimize the combined open pit and underground production schedules. Based on this analysis, it was concluded that the optimal combination is to mine the underground zones near the end of the open pit operations and modify the underground production profile to mine the three zones in only three years.

16.6.5 Final underground development and production schedule

To derive the final underground scenario, three production schedules were developed, one for each zone. The development and production schedules for each underground zone were developed on yearly periods and are summarized in Table 16.16 to Table 16.18.

Table 16.16 Production and development schedule for Zone 3A

	Total	**Year 8**	**Year 9**	**Year 10**
Mineralized material (t)	459,049	116,633	244,486	97,930
NSR (A$/t)	134.66	139.26	136.02	125.77
Pb (%)	5.16	5.35	5.26	4.69
Zn (%)	2.41	2.48	2.38	2.43
Ag (g/t)	7.85	8.37	7.99	6.91
Development				
Ramp (m)	715	715		
Level access (m)	625	383	158	84
RAR access (m)	127	127		
RAR (m)	182	182		
Total (m)	1,649	1,407	158	84
Waste (t)	**96,684**	**88,808**	**5,174**	**2,702**

Table 16.17 Production and development schedule for Zone 3B

	Total	Year 8	Year 9	Year 10
Mineralized material (t)	359,104	126,984	130,404	101,717
NSR (A$/t)	123.99	134.86	126.22	107.54
Pb (%)	5.38	5.86	5.42	4.74
Zn (%)	1.28	1.44	1.37	0.97
Ag (g/t)	12.02	11.31	12.62	12.13
Development				
Ramp (m)	546	546		
Level access (m)	376	130	114	132
RAR access (m)	80	80		
RAR (m)	245	245		
Mineralized material (m)	375	375		
Total (m)	1,622	1,376	114	132
Waste (t)	**65,343**	**57,769**	**4,525**	**3,049**

Table 16.18 Production and development schedule for Bridge Zone

	Total	Year 8	Year 9	Year 10
Mineralized material (t)	919,075	85,164	425,276	408,636
NSR (A$)	163.25	155.24	160.72	167.56
Pb (%)	7.07	6.65	6.94	7.30
Zn (%)	1.93	1.91	1.92	1.94
Ag (g/t)	11.02	10.78	10.91	11.19
Development				
Ramp (m)	905	905		
Level access (m)	785	541	164	80
RAR access (m)	315	315		
RAR (m)	271	271		
Total (m)	2,276	2,032	164	80
Waste (t)	**142,212**	**129,270**	**8,570**	**4,372**

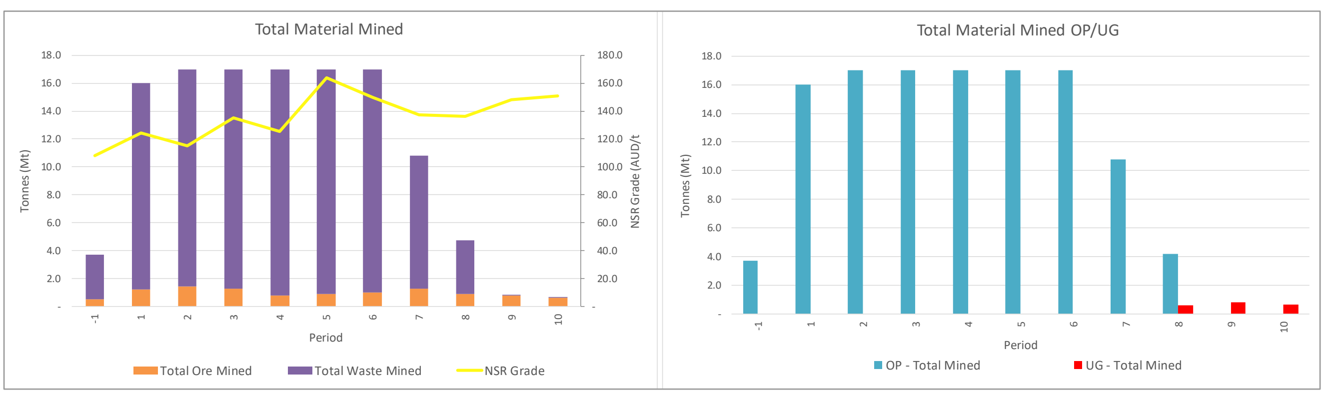
Mining operations extend over 11 years, including the pre-production period. The total annual ex-pit material mined peaks at 17 Mtpa from Year 2 to Year 6, before dropping to approximately 11 Mtpa at the end of the open pit mine life. The underground operations start in Year 8 and peak at 0.8 Mtpa in Year 9.

The projected combined schedule is summarized in Table 16.19 and Figure 16.28.

Table 16.19 Combined open pit and underground material mined

	Unit	Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Total mineralized material mined	Mt	10.6	0.5	1.2	1.4	1.3	0.8	0.9	1.0	1.3	0.9	0.8	0.6
NSR grade	A$/t	135.3	108.3	124.4	115.3	135.1	125.2	163.9	149.9	137.2	136.3	148.1	150.6
Total waste mined	Mt	111.1	3.2	14.8	15.6	15.7	16.2	16.1	16.0	9.5	3.8	0.01	0.01
Total mined	Mt	121.7	3.7	16.0	17.0	17.0	17.0	17.0	17.0	10.8	4.8	0.8	0.6
Open pit													
OP - mineralized material mined	Mt	8.9	0.5	1.2	1.4	1.3	0.8	0.9	1.0	1.3	0.6	-	-
OP - NSR grade	A$/t	132.9	108.3	124.4	115.3	135.1	125.2	163.9	149.9	137.2	134.1	-	-
OP - waste mined	Mt	110.8	3.2	14.8	15.6	15.7	16.2	16.1	16.0	9.5	3.6	-	-
OP - total mined	Mt	119.7	3.7	16.0	17.0	17.0	17.0	17.0	17.0	10.8	4.2	-	-
Underground													
UG - mineralized material mined	Mt	1.7	-	-	-	-	-	-	-	-	0.3	0.8	0.6
UG - NSR grade	A$/t	147.5	-	-	-	-	-	-	-	-	140.3	148.1	150.6
UG - waste mined	Mt	0.3	-	-	-	-	-	-	-	-	0.25	0.01	0.01
UG - total mined	Mt	2.0	-	-	-	-	-	-	-	-	0.6	0.8	0.6
UG – lateral development	m	4,849	-	-	-	-	-	-	-	-	4,117	435	297
UG – vertical development	m	698	-	-	-	-	-	-	-	-	698	-	-

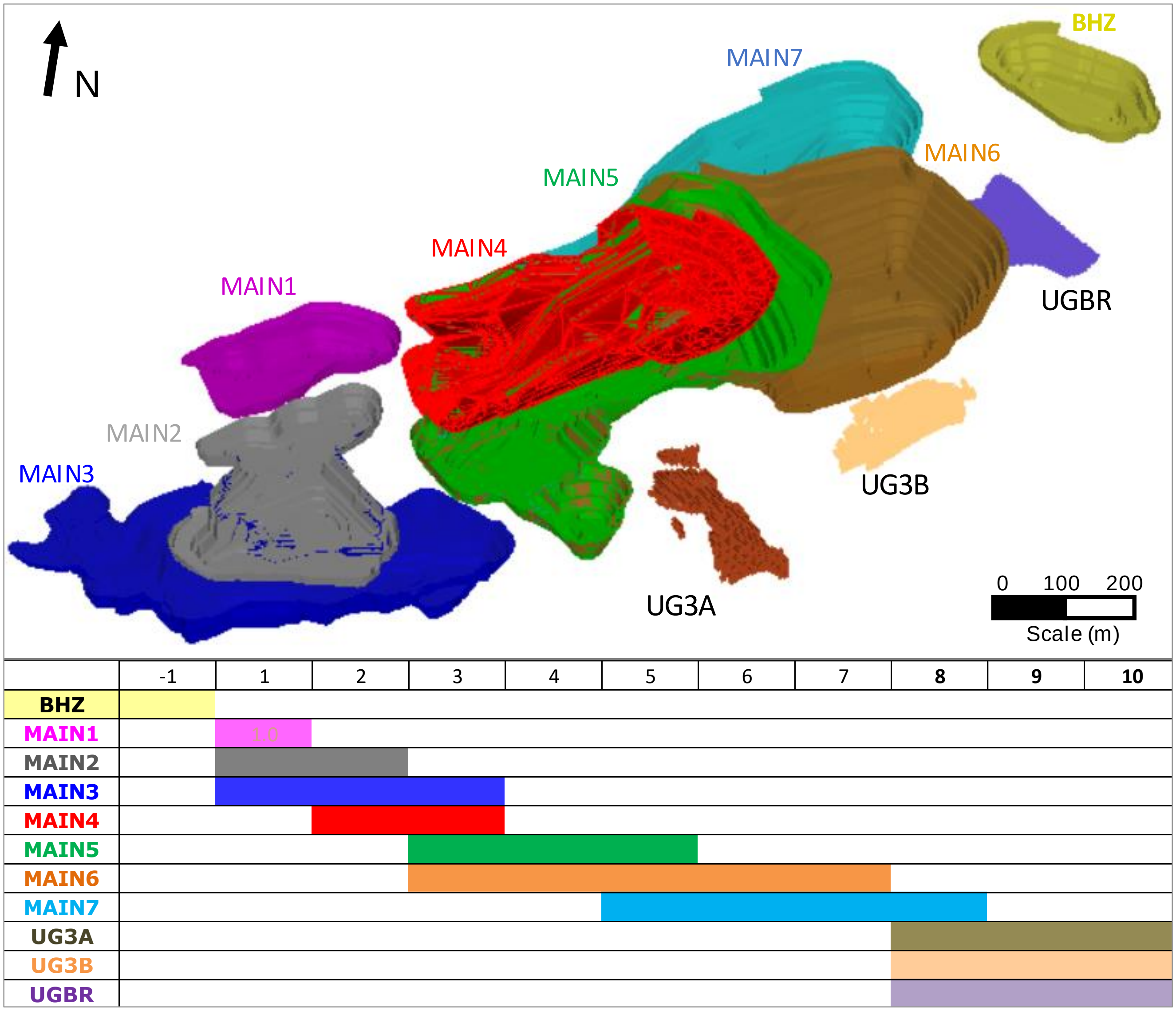
Figure 16.28 Life-of-mine production schedule



Source: AMC

The mining sequence by mining area is presented in Figure 16.29.

Figure 16.29 Mining sequence



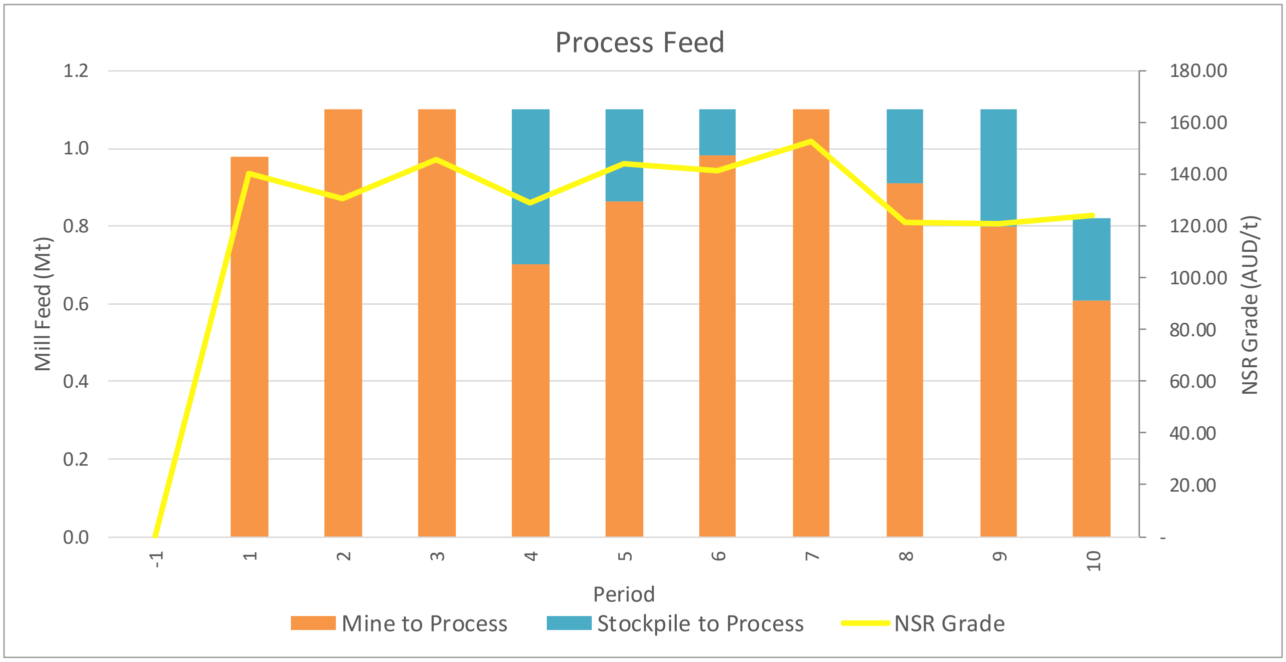
Source: AMC

16.6.6 Projected process plant feed schedule

Projected production from the BHZ open pit is stockpiled in Year -1, during the construction of the process plant. It has been assumed that the process plant would be capable of producing 0.98 Mt during Year 1 and 1.1 Mtpa from Year 2.

As presented in Figure 16.30, the targeted process feed is achieved on a yearly basis. The final two years of production, Years 10 and 11, are comprised of processing underground material and low-grade material from the long-term stockpiles.

Figure 16.30 Process feed schedule



Source: AMC

The combined process feed is summarized in Table 16.20. Note that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 16.20 Combined open pit and underground mill feed

	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Mill feed (Mt)	10.6	0.98	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	0.82
NSR (A$/t)	135.3	140.2	130.9	145.9	128.8	144.1	141.5	152.6	121.6	120.7	124.1
Pb (%)	5.3	5.7	4.8	5.2	4.7	5.7	5.8	6.3	4.9	5	5.2
Zn (%)	2.2	2	2.5	3.1	2.6	2.4	1.9	2	1.7	1.7	1.6
Ag (g/t)	8.8	6.8	7.5	7.7	7.6	8.7	10.4	11	10.5	8.5	8.9
OP mill feed (Mt)	8.9	1	1.1	1.1	1.1	1.1	1.1	1.1	0.8	0.3	0.2
NSR (A$/t)	132.9	140.2	130.9	145.9	128.8	144.1	141.5	152.6	113.6	47.5	47.5
Pb (%)	5.1	5.7	4.8	5.2	4.7	5.7	5.8	6.3	4.5	1.73	1.7
Zn (%)	2.2	1.9	2.5	3.1	2.6	2.4	1.9	2	1.6	0.9	0.9
Ag (g/t)	8.4	6.8	7.5	7.7	7.6	8.7	10.4	11	10.6	3.7	3.7
UG mill feed (Mt)	1.7	-	-	-	-	-	-	-	0.3	0.8	0.6
NSR (A$/t)	147.5	-	-	-	-	-	-	-	140.3	148.1	150.6
Pb (%)	6.2	-	-	-	-	-	-	-	5.8	6.2	6.4
Zn (%)	1.9	-	-	-	-	-	-	-	1.9	2	1.9
Ag (g/t)	10.4	-	-	-	-	-	-	-	10.2	10.3	10.6

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

17 Recovery methods

17.1 Introduction

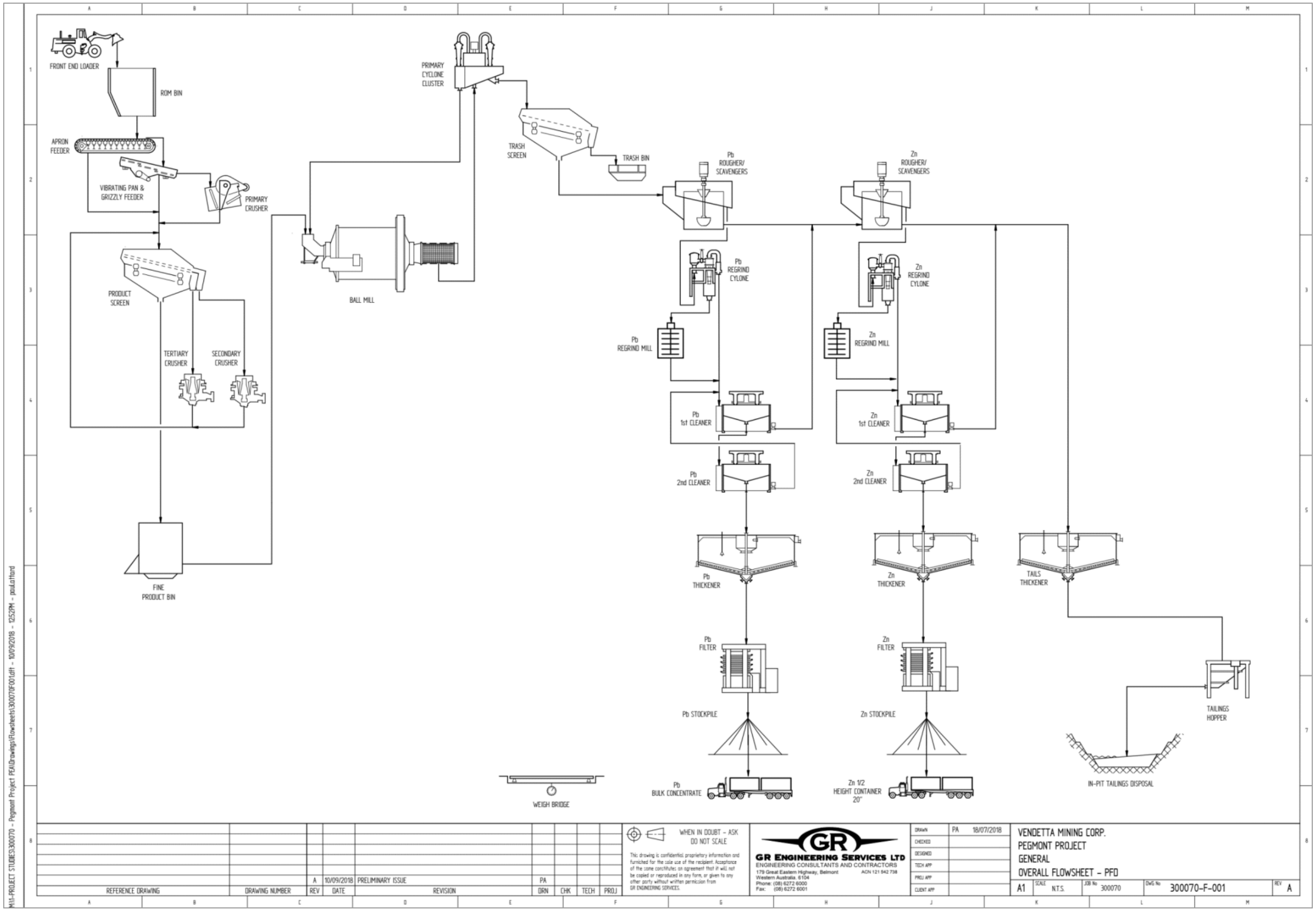
A conventional sequential flotation circuit has been selected for the recovery of the lead and zinc minerals from the Pegmont deposit.

The process plant shall consist of a conventional three stage crushing and a single stage ball mill grinding circuit, followed by differential flotation of the lead and zinc minerals to produce separate saleable lead and zinc concentrates. The concentrates from the lead and zinc flotation circuits will be thickened and subsequently filtered on site for road transport.

The lead concentrate will be transported by road to the Mt Isa, while the zinc concentrate will be transported by road to a rail siding located at the nearby town of Malbon, and then transported by rail to Townsville, Queensland.

An overall simplified process flow diagram is included in Figure 17.1.

Figure 17.1 Simplified process flow diagram



Source: GRES

17.2 Process design basis

The plant has been designed on the basis of treating open pit material from the identified mineralized zones.

The process plant has used the following key criteria as the basis of design development.

Table 17.1 Process design basis

Key criteria	Units	Process values
Annual throughput	t/a dry	1,100,000
Ore SG	t/m^3	3.9
Feed grades – design		
Lead	%	8.3
Zinc	%	3.0
Silver	g/t	10
Lead concentrate		
Lead grade	%	69
Lead recovery	%	91
Recovery of silver to lead concentrate	%	70
Zinc concentrate		
Zinc grade	%	54
Zinc recovery	%	73
Recovery of silver to zinc concentrate	%	3
Annual operating hours		
Crushing circuit	h	6570
Process plant	h	8059
Treatment rate		
Crushing circuit	t/h	176
Process plant	t/h	138
Bond rod mill work index (benchmark)	kWh/t	22
Bond ball mill work index (85th percentile)	kWh/t	20.3
Primary grind size (P80)	µm	106
Concentrate regrind grind size (P80)	µm	20
Lead rougher scavenger laboratory flotation time	min	11
Lead cleaner laboratory flotation time	min	13
Lead recleaner laboratory flotation time	min	8.6
Zinc rougher scavenger laboratory flotation time	min	11
Zinc cleaner laboratory flotation time	min	7.7
Zinc recleaner laboratory flotation time	min	4

17.3 Process plant description

17.3.1 Overview

Fresh open pit mineralization will be mined and delivered to a Run of Mine (ROM) pad for processing through a three-stage crushing plant designed to provide a crushed product size suitable as feed to the single stage ball mill. A three-stage crushing plant and ball mill comminution circuit has been selected based on the available comminution data. JKMRC drop weight tests or SMC comminution tests have yet to be performed and it is recommended that these tests are undertaken in the next phase of project development to confirm the comminution approach applied.

Crushed material will be stored in the fine product bin for reclaim via a conveyor system to the primary ball mill. The single stage ball mill will operate in a closed circuit with hydrocylones to produce a target P80 size of 106 µm, suitable for the downstream flotation circuit. A differential flotation circuit for lead and zinc will be utilized with rougher concentrate regrind stages for both the lead and zinc circuits to produce saleable grade lead and zinc concentrate.

Tailings from the flotation circuit will be thickened for the recovery of process water. The thickened tailings will be initially pumped to an in-pit tailings storage facility for deposition, using one of the smaller starter open pits. Latter operation will use mined out sections of the main open pit mine for tailings storage.

Concentrates from the lead and zinc flotation circuit will be thickened and subsequently filtered for road and road / rail transport. Lead concentrate will be transported by road from site to Glencores's Mt Isa Smelter. Zinc concentrate will be loaded into half height containers and sealed at site for road transport to the Malbon rail loading facility. The zinc concentrate half height containers will be stored at Malbon, prior to being railed once per month the Sun Metals Zinc Refinery in Townsville, Queensland.

Raw water will be supplied to the process plant from the artesian basin via a network of bore-field pumps. The process water circuit will comprise of circuit water recovered from the tailings and concentrate thickeners, in combination with return water from the in-pit tailings storage facilities. Raw water will also be used as make-up water for the process water system. Separate lead / zinc flotation water circuits have not been considered in the initial design concept.

17.3.2 Crushing

A three-stage crushing circuit has been selected based on applications treating similar material and the hardness values measured in the test work conducted for this study which indicate hard and potentially highly competent material will be treated.

Open pit material will be reclaimed by a FEL from the ROM stockpile into the ROM bin. The open pit material will be reclaimed from the ROM bin by an apron feeder and vibratory grizzly pan feeder to remove fines from the feed to the primary jaw crusher. The open pit material will be reduced in size from an F80 of 317 mm to a P80 of 120 mm by the primary jaw crushing, based on a close side setting (CSS) of 100 mm.

Primary crushed material will be conveyed to a double deck product screen. The top deck oversize material will be directed to the secondary cone crusher, with the lower deck material directed to a tertiary cone crusher. The secondary crusher will operate with a CSS of 32 mm, whilst the tertiary crusher CSS will be set to 14 mm.

The secondary and tertiary crushing stages will operate in closed circuit with the product screen. The final product size will be controlled by the aperture of the bottom deck of the product screen to generate a ball mill feed size having a P80 of approximately 10 mm.

The crushed material will be conveyed to a fine product bin.

17.3.3 Grinding

Crushed material will be milled in a single stage ball mill operating in closed circuit with hydrocyclones. Crushed material will be reclaimed from the fine product bin by a variable speed conveyor for mill feed rate control. The single stage ball mill, with dimensions of 5.23 m Ø x 6.67 m EGL, will be equipped with a fixed speed 3300 kW mill motor. The cluster of 400 mm diameter hydrocyclones will have an overflow size P80 of 106 µm and will be designed to operate at a recirculating load of up to 300%.

Overflow from the hydrocylones will be pumped to the lead flotation conditioning tank. Cyclone underflow will gravitate back to the single stage ball mill for further size reduction.

Lime will be added when required in the milling circuit stage maintain a slurry pH prior to flotation of 8.5 to 9.0. Sodium Monophosphate (SMP) will also be added to the milling circuit to limit the pre-activation of sphalerite prior to lead flotation.

17.3.4 Lead flotation circuit

Cyclone overflow from the grinding circuit will gravitate via a trash screen to the lead rougher conditioning tank. The lead flotation circuit will consist of a conditioning stage, roughing and scavenger flotation, regrinding of the lead rougher and scavenger concentrate, followed by two stages of cleaning. The final lead concentrate will be pumped to the lead concentrate dewatering circuit and the lead rougher flotation tailings and lead cleaner tailings will be pumped to the zinc flotation circuit.

The trash screen undersize slurry will gravitate into the 25 m^3 capacity agitated lead rougher conditioning tank and will provide a maximum of 5 minutes conditioning residence time at the design pulp flow rate.

Reagent additions in the lead flotation circuit will include:

- Aerophine 3418A collect to the rougher conditioning tank, the lead rougher / scavenger circuit and the lead cleaner flotation cells.
- Frother to the feed of the lead rougher cells and at the head of the cleaner cells.
- Lime to the rougher conditioning tank and lead cleaner cells.

Slurry will gravitate from the lead rougher conditioning tank to the feed box of the lead rougher flotation cells. Flotation will take place in a conventional lead rougher, cleaner and recleaner flotation circuit using mechanically agitated, forced air cells. The lead flotation circuit will consist of the following equipment:

- One 25 m^3 rougher conditioning tank.
- Two 30 m^3 lead rougher mechanically agitated, forced air tank cells in series.
- Three 30 m^3 lead scavenger mechanically agitated, forced air tank cells in series.
- One 355 kW stirred media detritor (SMD) lead regrind mill.
- One 2 m^3 lead cleaner conditioning tank.
- Five 16 m^3 mechanically agitated, forced air conventional lead cleaner cells.
- Four 4.3 m^3 mechanically agitated, forced air conventional lead recleaner cells.

Five cells in series were selected for lead rougher and scavenger stage to minimize losses due to short circuiting. The 30 m^3 cells will give a combined residence time of approximately 31 minutes. The cell arrangement will be Feedbox - One cell - Pinch valve - One cell - Pinch valve - One cell - Pinch valve – Two cells – Pinch Valve. The lead rougher and scavenger concentrate will be pumped to the lead concentrate regrind mill. The lead scavenger tailings will discharge into the lead scavenger tailings hopper along with the lead cleaner tailings and pumped to the zinc flotation circuit. The lead rougher and scavenger concentrate will be reground by a SMD mill to a product target P80 size of 20 µm to allow an upgrade to occur in the lead cleaner circuit. Stirred mills have been chosen specifically to use inert media which will not promote galvanic interactions with other sulphide minerals.

The lead rougher concentrate will be classified by cyclones prior to the regrind stage with the cyclone underflow reporting to the lead regrind mill. The cyclone underflow will be ground to a P_{80} size of 20 µm in the lead regrind mill which will operate in open circuit. The lead regrind cyclone overflow will gravitate to the lead regrind mill discharge tank, combining with the mill discharge. The lead regrind mill discharge product will be pumped to the lead cleaner conditioning tank.

Reground lead rougher / scavenger slurry in the lead cleaner conditioning tank will overflow to the lead cleaner feed box. A two bank (two + three) cleaner cell arrangement has been selected to minimize short circuiting. Lead cleaner concentrate from the lead cleaner circuit will be pumped to the conventional lead recleaner cells, arranged in a single bank, and the lead cleaner tails will report to the lead scavenger tailings hopper. Lead recleaner concentrate will be pumped to the lead concentrate thickener and the lead recleaner tailings will gravitate to the lead cleaner feed box.

The lead flotation cells will be supplied with air mass flow control to enable each bank of cells to be controlled to an operator set point. Level control will be via dart valves and level elements controlled to an operator set point.

17.3.5 Zinc flotation circuit

The lead scavenger tailings and lead cleaner tailings will be pumped to the zinc flotation circuit. The zinc flotation circuit will consist of the zinc roughing and scavenger circuit followed by a zinc regrind of the rougher and scavenger concentrate and two stages of cleaning. The zinc concentrate will be pumped to the zinc concentrate thickener and the final tailings will be pumped to the tailings thickener.

Tailings from the lead flotation circuit will discharge into the first of two 25 m³ agitated zinc rougher conditioning tanks. Each tank will provide 4 minutes of conditioning residence time with a total of 8 minutes for the feed to the zinc circuit.

Reagent additions in the zinc flotation circuit will include:

- Lime pH modifier to the first zinc rougher conditioning tank, rougher / scavenger circuit, cleaner conditioning tank, and as staged additions to the zinc cleaner circuit.
- Copper sulphate activator to the second zinc rougher feed conditioning tank.
- SIBX collector to the second zinc rougher feed conditioning tank.
- Frother to the feed box of the first zinc rougher cell, and stage addition to the cleaner circuit.

Slurry will gravitate from the second zinc rougher conditioning tank to the feed box of the zinc roughers. Flotation will take place in a conventional zinc rougher, cleaner and recleaner flotation circuit using mechanically agitated, forced air cells. The zinc flotation circuit will consist of the following equipment.

17.3.5.1 Zinc rougher – scavenger circuit

- Two 30 m³ zinc rougher mechanically agitated, forced air tank cells in series.
- Four 30 m³ zinc scavenger mechanically agitated, forced air tank cells in series.
- One 185 kW SMD zinc regrind mill.

17.3.5.2 Zinc cleaner and zinc recleaner circuit

- One 5.0 m³ agitated zinc cleaner conditioner tank.
- Five 8 m³ mechanically agitated, forced air conventional zinc cleaner cells.
- Three 4.25 m³ mechanically agitated, forced air conventional zinc recleaner cells.

Six cells in series were selected for zinc rougher and scavenger stage to minimize losses due to short circuiting. The 30 m^3 cells will give a combined residence time of approximately 32 minutes. The cell arrangement will be Feedbox - One cell - Pinch valve - One cell - Pinch valve - Two cells - Pinch valve – Two cells – Pinch Valve. Slurry from the zinc rougher conditioning tanks will report to the zinc rougher and scavenger circuit. The zinc rougher and scavenger concentrate will be pumped to the zinc concentrate regrind mill. The zinc scavenger tail will then form the bulk of the final plant tailings stream.

The zinc rougher and scavenger concentrate will be reground by a SMD mill to a product target P_{80} size of 20 µm to allow an upgrade to occur in the zinc cleaner circuit. The zinc rougher concentrate will be classified by cyclones prior the regrind stage with the cyclone underflow reporting to the zinc regrind mill. The cyclone underflow will be ground to a P_{80} size of 20 µm in the zinc regrind mill which will operate in open circuit. The zinc regrind cyclone overflow will gravitate to the zinc regrind mill discharge tank, combining with the mill discharge. The zinc regrind mill discharge product will be pumped to the zinc cleaner conditioning tank.

Zinc cleaner conditioning tank overflow will gravitate to the zinc cleaner feed box. A two bank (two + three) cleaner cell arrangement has been selected to minimize short circuiting. Zinc cleaner concentrate from the zinc cleaner will be pumped to the conventional zinc recleaner cells, arranged in a single bank, and the zinc cleaner tails will report to the zinc scavenger tailings hopper as final tails. The zinc recleaner concentrate will be pumped to the zinc concentrate thickener and the zinc recleaner tailings will gravitate to the zinc cleaner feed box. The zinc flotation cells will be supplied with air mass flow control to enable each bank of cells to be controlled to an operator set point. Level control will be via dart valves and level elements controlled to an operator set point.

17.3.6 Concentrate production

17.3.6.1 Concentrate thickening

Final lead and zinc concentrates from the flotation recleaner circuits will be pumped to 10 m and 8 m diameter high rate concentrate thickeners respectively. No concentrate thickening test work has been completed and a conventional specific settling rate of 0.25 $t/m^2.h$ has been used to size the concentrate thickeners based on benchmark data from similar applications – a 1.25 safety factor on required area has been applied in addition given the fineness of the reground concentrates

Flocculant will be added to increase the settling rate and underflow density to a target of 60% and 57% solids (w/w) for lead and zinc respectively. Thickened underflow will then be pumped to the associated lead and zinc agitated concentrate storage tanks by peristaltic pumps. To protect the lead and zinc filters, in-line strainers will be installed on the discharge lines from the thickener underflow pumps to remove trash. Overflow from lead and zinc concentrate thickeners will gravitate to the process water pond.

Control systems for all thickeners will include bed level indication to control the flocculant addition rates and bed mass pressure indication to control the thickener underflow pump speed. Rake torque will be measured and the system will be able to automatically raise or lower the rakes as required.

17.3.6.2 Concentrate filtering

The filtration circuits for the lead and zinc concentrates will be similar. Each concentrate filtration circuit will consist of an agitated concentrate storage tank, filter feed pumps and plate and frame type pressure filters with ancillaries. No filtration test work has been completed. Pressure filters have been selected as having a superior ability to achieve low residual moisture levels in the filter product. In the absence of specific test work the filtration rates used for determination of the filter sizing have been benchmarked against similar applications (315 kg/m^2h for Pb and 254 kg/m^2h for Zn).

The lead concentrate storage tank will have a live volume of 200 m^3, and the live volume of the zinc concentrate storage tank will be 100 m^3.

Thickened lead and zinc concentrate slurries will be pumped from the respective concentrate storage tanks for dewatering in the pressure filters. The filtration area requirements are:

- 80 m^2 expandable to 100 m^2 for the lead concentrate filter.
- 44 m^2 expandable to 60 m^2 for the zinc concentrate filter.

The pressure filter will dewater the slurry to produce a filter cake with low residual moisture suitable for transport. Based on vendor advice on similar application, moisture levels in the range of 9% to 12% (w/w) can be expected for the concentrates. The filtrate will be pumped to its respective concentrate thickener to recover contained solids and water reclamation. The dewatered lead and zinc filter cakes will be discharged automatically from each filter press into the separate sections of the concentrate storage shed to minimize cross contamination.

17.3.6.3 Concentrate loadout facility

The lead and zinc filter cake will be stored on separate stockpiles inside the concentrate storage shed. The structure will be sized to accommodate inventory for both the lead and zinc filter products with provision to move the stockpiles within the enclosed building to extend the storage inventory or to provide a drying mitigation area, should the concentrates not meet the transportable moisture limit (TML) specifications and require further drying.

The concentrate loadout facility will comprise of a direct load out truck weighbridge for the lead and zinc concentrates. Lead concentrate will be loaded onto bulk concentrate trucks for dispatch to Glencore’s Mt Isa Lead Smelter. Zinc concentrate will be loaded into half height containers equipped with sealable lids and transported by road to the Malbon rail loading facility. Zinc concentrate will be stored at Malbon for monthly rail haulage to Sun Metals Zinc refinery in Townsville, Queensland or to the Port of Townsville for export shipment to other customers.

17.3.7 Tailings thickening

The final tailings, consisting the zinc scavenger and zinc cleaner tails will be transferred to a 16 m diameter high rate final tailings thickener. Flocculant will be added to increase the settling rate and produce an underflow density of 50 – 55 % solids (w/w). A flux rate of 1.0 $t/m^2.h$ has been used to size the tailings thickener – a 1.2 safety factor on required area has also been applied in the absence of specific test work. Underflow will then be pumped to tailings storage facility which will consist of the first of the smaller open pits used in the initial years of operation. Thickener overflow will gravitate to the process water dam.

17.4 Reagents

Treatment of the sulphide mineralization by the differential flotation process will require a specific suite of flotation reagents to ensure activation and depression of mineral species, frother for froth stability and collectors for specific mineral recovery. A summary of the reagent usages is listed in Table 17.2.

Table 17.2 Annual reagent consumption

Reagent	Consumption g/t
Flotation reagents	
Frother (MIBC)	120
Copper sulphate	110
Lead collector (3418A)	30
Sodium Monophophate	150
Sodium Isobutyl Xanthate (SIBX)	29
Flocculant	15
Anti-scalant	6
Hydrated lime	466
Grinding media	
Grinding media (2.5 mm)	200
Grinding media (80 mm Steel)	1410

17.4.1 Aero 3418A

The lead collector will be a dialkyl dithiophosphinate with the trade name Aerophine 3418A. The lead collector will be supplied as a 100% concentrated solution in 1,000 litre bulk boxes.

Lead collector will be pumped undiluted to a head tank for distribution to the lead flotation circuit. Flow control will be provided using variable speed dosing pumps. Flow rates will be manually measured.

17.4.2 MIBC

The frother will be a Methyl Isobutyl Carbinol (MIBC). The frother will be supplied as a 100% concentrated solution in 1,000 litre bulk boxes.

Frother will be pumped undiluted to a head tank for distribution through the flotation circuits. Flow control will be provided using variable speed dosing pumps.

17.4.3 Sodium Isobutyl Xanthate (SIBX)

SIBX is a collector for sulphide minerals. SIBX will be supplied as solid pellets in 160 kg drums or 1,000 kg bulk bags.

SIBX mixing will be completed manually by the operator. The drums will be lifted by forklift, emptied by a drum tipping device and then mixed with raw water to generate a 20% (w/v) solution. The hood over the mixing tank will be fitted with an extraction fan.

The SIBX mixing tank is divided into separate mixing and storage sections. The mix section is larger and agitated while the smaller storage section is not. The SIBX solution will be automatically transferred from the mixing tank to a head tank for distribution to the zinc flotation circuit. The SIBX transfer pump normally draws from the agitated section of the tank but when a low level is reached, the draw point is changed to the storage section of the tank. Excess solution overflows to the storage section of the dual-purpose tank.

SIBX will be pumped to a head tank for distribution. Flow control will be provided using variable speed dosing pumps.

17.4.4 Lime hydrated

Lime will be used to increase the pH of the process streams. The lime will be supplied as hydrated lime in bulk as a powder. Upon receipt at site, it will be pneumatically transferred into a 50 tonne capacity silo. The hydrated lime will then be mixed with water in the lime mixing tank to produce a 20% w/w hydrated lime slurry. The hydrated lime slurry will be automatically transferred from the mixing tank to the storage tank. Both the mixing and storage tanks will be agitated. Hydrated lime slurry will be pumped via a ring main for distribution throughout the plant. Individual addition points will be controlled by an automatic valve and timer.

17.4.5 Copper sulphate

Copper sulphate pentahydrate will be used to activate and promote the flotation of the zinc sulphide minerals. Copper sulphate pentahydrate will be supplied in granular form in 1,200 kg bulk bags.

Copper sulphate mixing will be completed manually by the operator. The bulk bags will be lifted by the reagents hoist, split, and then mixed with raw water to generate a 15% (w/v) solution.

The copper sulphate mixing tank is divided into separate mixing and storage sections. The mix section is larger and agitated while the smaller storage section is not. The copper sulphate solution will be automatically transferred from the mixing tank to a stainless-steel head tank for distribution to the zinc flotation circuits. The copper sulphate transfer pump will normally draw from the agitated section of the tank but when a low level is reached, the draw point is changed to the storage section of the tank. Excess solution overflows to the storage section of the dual-purpose tank.

The mixed copper sulphate solution will be distributed to the plant via variable speed dosing pumps.

17.4.6 Antiscalant

Antiscalant will be added to the discharge of both the process water pumps. The antiscalant will be supplied as a solution in 1,000 litre bulk boxes.

Antiscalant will be distributed undiluted to the discharge of process water pumps and mining slurry water pump via variable speed dosing pumps.

17.4.7 Flocculant

The flocculant will be an anionic flocculant. Flocculants are long chain molecules that aid solids settling by causing individual particles to stick together thereby forming larger, heavier particles. Flocculant will be supplied as a powder in 20 kg bags or as bulk bags of 800 kg.

The flocculant mixing system will be a proprietary packaged plant consisting of a dry flocculant hopper, powder feed and wetting system, mixing tank, transfer pump and storage tank. Flocculant will be mixed automatically with raw water on a batch basis to generate a 0.25% (w/v) solution. The flocculant solution will be automatically transferred from the mixing tank to the storage tank.

Flocculant solution will be pumped by individual progressive cavity dosing pumps for distribution to the thickeners. Additional dilution water is injected into a mixing device located in the dosing pump discharge lines near the flocculant dose point. Flow rates will be manually measured.

17.4.8 Sodium Monophoshate (SMP)

Sodium Monophosphate (SMP) is a modifying / depressant agent to limit the pre-activation of the zinc sulphide minerals by free lead and copper ions. Depressants change the surface chemistry of selected minerals to depress the action of collectors in flotation. SMP will be supplied in powder form in 1,000 kg bulk bags.

SMP mixing will be completed manually by the operator. The bulk bags will be lifted by the reagents hoist, split and then mixed with raw water to generate a 20% (w/v) solution. The hood over the mixing tank will be fitted with an extraction fan.

SMP will be pumped to a head tank for distribution. Flow control will be provided using variable speed dosing pumps.

17.5 Services

17.5.1 Control systems

The Pegmont process plant will be operated on a continuous basis. Operators will monitor and run the plant from PC based human-machine interface (HMI) systems located in the process plant control room. Operations will be monitored and controlled from the process plant control room via PC based Supervisory Control and Data Acquisitioning (SCADA) screens using Citect software. The ability to remotely start and stop plant equipment from the operator screens via start and stop sequences or by "manually" starting or stopping individual equipment will be provided. Recording of plant data by operations personnel will be required on a shift by shift basis, with the Courier system providing online assay results.

Motor starters, main isolators and distribution board feeders will be located in the Motor Control Centres (MCCs). Each MCC will contain a programmable logic controller (PLC) which with the SCADA system, will make up the Process Control System (PCS). The PCS will provide the interface between drives and instrumentation and the operators. Hardwired outputs and inputs for plant equipment and field devices will be interfaced through these PLCs in which plant startup / shutdown sequences and interlocks will be programed. The plant functions will be controlled automatically by the PLC and the operators will be required to monitor the system performance and perform manual checks.

All equipment will be equipped with local start and stop buttons. Start and stop will also be possible from HMI. It is intended that there will be two modes of operation for each piece of equipment: local and remote. The status of each piece of equipment (i.e. whether in local or remote mode) will be displayed at the HMI. Local or remote mode will be selected using a switch mounted on the local control panel. The stopped / started status of each piece of equipment will be displayed at the HMI.

17.5.2 Online analysis

The performance of the flotation circuits will be monitored by an On Stream Analysis (OSA) system – Outotec Courier 5i SL. Samplers, mostly pressure pipe samplers, will direct representative slurry samples to the OSA for analysis and where not possible the streams will be pumped to the OSA by dedicated sample pumps. The OSA will be located with sufficient elevation to allow streams to gravitate to their destination and where not possible the streams will be pumped to their destination by dedicated pumps. The OSA will measure the copper, lead, zinc, iron, and silver concentrations and slurry density of each of the sample streams as follows:

- Lead rougher feed
- Lead rougher / scavenger concentrate
- Lead scavenger tail
- Lead final concentrate
- Lead cleaner tail
- Zinc rougher / scavenger concentrate
- Zinc scavenger tail
- Zinc final concentrate
- Zinc cleaner tail
- Final tails

A dedicated PC will be provided for the OSA system to provide continuous monitoring of assays and allow system development and calibration tasks to be undertaken. Assays will be updated every 5 to 10 minutes. A mimic page on the SCADA PC's will also be provided to show the status of the OSA systems and the assay results obtained which will also appear on the pages for the relevant flotation circuit.

A Multiplexer will be fitted to the OSA to provide for metallurgical samples on each stream analysis for shift composite samples for metallurgical accounting. The sample cut timing will be adjusted from the dedicated OSA PC and the sampling will occur automatically. There will be provision to manually operate the cutter to obtain samples for calibration.

17.5.3 Compressed air

Plant and instrument air systems will have separate air receivers and distribution network. Plant and instrument air to the processing facility will be supplied by two screw compressors arranged as one duty and, one standby machine. A bleed stream from the plant air receiver will provide the feed for a dedicated air dryer and instrument air receiver.

An additional dedicated air compressor will provide air services to the filter presses as high-pressure air for drying purposes and a separate receiver will be housed at the filter building for this duty.

Flotation air to the flotation circuits will be supplied by two dedicated blowers arranged in a duty-standby configuration.

17.5.4 Raw water

Raw water will be sourced from the artesian basin via a borefield located 28 km to the south of the project. Raw water supplied from the borefield will be stored in a new 1,000 m^3 raw water tank at the plant site. Raw water will be used for the following purposes:

- Process water make up
- Reagent mixing
- Flocculant dilution
- Cloth washing in filtration
- Gland water
- Regrind milling cooling water
- Potable water treatment plant feed and safety showers
- Fire water

Raw water is pumped to the raw water tank and process water dam via the raw water submersible pumps and the raw water booster pump.

Two raw water distribution pumps arranged in a duty-standby configuration will be located at the raw water tank. These pumps will distribute the raw water throughout the entire processing facility. In addition, there will be two dedicated pumps, arranged in a duty-standby configuration, for gland water.

The lower portion of the raw water tank will provide a dedicated fire water reservoir for the fire water system. The fire water system will include two pumps, an electric fire water pump and a diesel driven fire water pump to ensure a continuous supply of water to the fire system in the event of a power failure.

Raw water will also provide feed supply for a potable water treatment plant.

17.5.5 Potable water

Potable water will be provided by a reverse osmosis plant designed for a production rate of 3 m^3/h and will provide for the potable water and safety shower. The plant will also provide potable water to the accommodation village.

18 Project infrastructure

18.1 Infrastructure and services

A concept has been developed for the site layout, based on the location of the mining pits and the local topography. The mill and flotation plant are separated from the open pit by a blast exclusion zone of 500 m. The ROM and crushing plant will fall within a 250 m to 500 m exclusion zone that will require the area to be vacated during blasting only.

Administration and Mine offices will be located near the processing plant area outside of the 500 m exclusion zone.

The accommodation camp will be located to the north of the project area and will be naturally shielded from noise and light from the processing and mining operations by an outcrop of hills. This is shown in Figure 18.1.

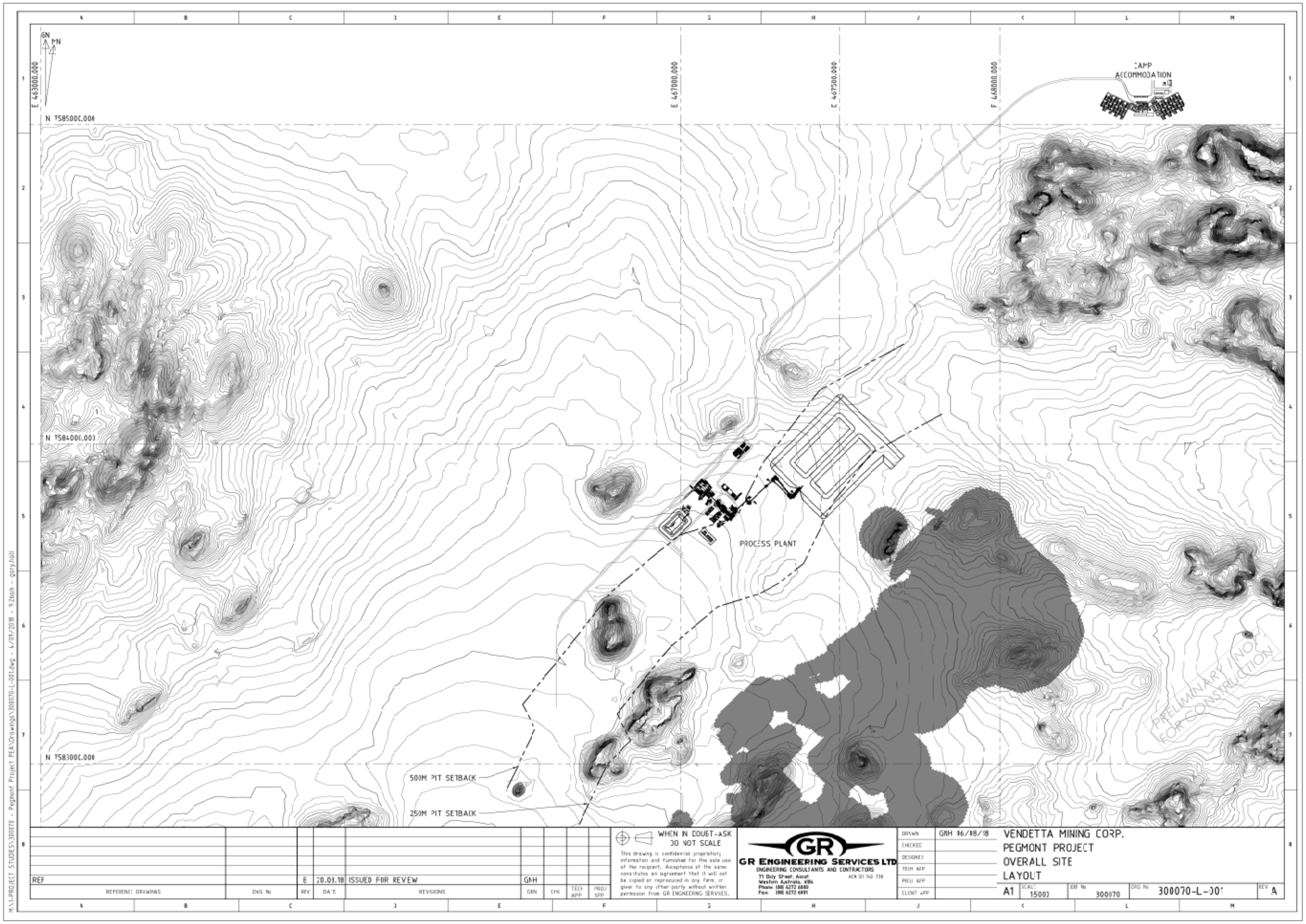
Existing infrastructure at site includes:

- Unsealed public and private roads to the site location.
- Pegmont exploration base camp facility.
- Local bore water pumps and tanks, servicing site, and drilling activities.

Area wide infrastructure:

- High Pressure Natural Gas line located approximately 16 km to the South of the project. The line runs between Osborne and Cannington Mine sites.
- Fibre Optic Cable also approximately 16 km to the South running in parallel to the high-pressure gas line, offset by 150 m.
- Osborne Mine – Accommodation camp facility with +300 capability for shared service with Osborne, approximately 45 minutes by road to the construction site.
- Osborne Airport – Fully seal all weather airport, currently servicing Chinova – Osborne operations. It is capable of servicing jet powered aircraft.
- Artesian Basin for water supply is expected to be located approximately 27 to 32 km south of the plant location.

Figure 18.1 Site schematic layout



Source: GRES

18.2 Buildings

New site buildings will be required to support the operation and will include:

- Project administration office located at the plant site.
- Plant workshop and store.
- Reagents storage shed.
- Plant maintenance and stores offices.
- Laboratory.
- Training facility, first aid, and paramedic building.
- Plant control room.
- Plant switchrooms.
- Mining administration and geology offices.
- Mine workshop including segregated heavy and light vehicle maintenance bays.
- A mine workshop maintenance office.
- ANFO storage shed.
- Booster storage shed.
- Crib room facilities for both the plant and mining areas.
- Ablution facilities for both the plant and mining areas.
- New carpark facilities at both the plant and mining office buildings.

Most of the buildings will be transportable style sandwich panel buildings constructed on a steel sub frame. The buildings will be pre-wired for telephone, data, light, and 240-volt power. The location of these buildings is shown on the site plan.

The plant workshop, stores, reagents, and mine workshop sheds will be of steel framed and clad construction.

The ANFO and booster sheds will be of steel framed and clad construction with 2 m high precast concrete blast walls inside the shed.

18.3 Power supply

18.3.1 Load

Electrical power for the operation, estimated to be an average load of 6.1 MW for the processing plant and associated services, and excludes the future underground mining requirement. Electrical power will be generated by gas fired generator sets, each rated at 2,500 kW at full load and expected to run at 80% load and 2000 kW each. Including mining and camp demand power, nominally four sets will be required to be running with five sets installed for demand and standby application.

Electrical power for the operation will by gas fired generator sets running at 80% load with 2000 ekW output. Each set will consume 8.43 MJ/ekW.h with a consumption rate of 443 Nm3/h of natural gas. Figures have been based on Caterpilla G3520H gas genset technical data.

18.3.2 HV Power distribution

Power for the Project will be reticulated throughout the site at 11 kV. Stepdown transformers will be installed at each Motor Control Centre (MCC) to further reduce the voltage to 415 V (3 phase supply) for the plant equipment and to 110 volts for single phase supply as required.

Power factor correction equipment will be installed as part of the new works to ensure a minimum power factor of 0.95 lagging.

18.4 Water supply

18.4.1 Raw water

Raw water for the Project shall be sourced from a dedicated bore field. For the purpose of this study an area in between the existing Cannington Silver Lead Zinc Mine and the Osborne Copper-Gold Mine bore fields has been selected as the location. The bore field is assumed to comprise of five bores sunk in the Great Australian Artesian Basin spread over 5 km. Each bore is expected to deliver 7 to 10 L/s with a total capacity in excess of the Plants water requirements including when return water from the tailings storage facilities is not available.

The bores shall report to a transfer tank to be pumped via a 27 km long pipe line following the Selwyn Toolebuc Road to the plant site and stored in a new 1,000 m^3 raw water tank located adjacent to the process water pond.

The system will work on a constant overflow arrangement to the process water pond ensuring that the raw water tank is fully maintained for raw water and fire service demands.

18.4.2 Potable water

Potable water will be generated onsite from the raw water supply via a reverse osmosis plant before being pumped to the plant and mining amenities as well as the accommodation village.

18.4.3 Sewage

A sewerage system is required to convey and treat wastewater produced from toilets, showers, and sinks at the plant site and in the accommodation village. Sewerage systems are regulated by the local district councils in accordance with the QLD Building Codes and the Department of Health.

Separate packaged proprietary sewerage treatment system will be installed to treat both the accommodation village and the processing plant / mining demands. These systems will consist of a number of in-ground collection pits located adjacent to the plant and mining buildings where waste water is generated. Waste water will be plumbed into the pits from where it will be pumped to each main treatment unit collection tank using submersible macerator pumps. Each pump will be automatically started and stopped by float switches to prevent overflow and maximize pump life.

The packaged treatment systems will be located within a 500 m vicinity of camp and process plant locations. Clear effluent produced by the plant will be discharged by a spray irrigation system. Ongoing operation of the system will require only general maintenance of the pumps, agitators, and air blower.

18.5 Communications

A buried fibre optic communication line is located approximately 16 km south of the proposed site. It is proposed to connect a branch from the fibre to the proposed site to provide for telecommunications and data requirements.

There is a series of mobile telephone towers within the area that supply some communication, but these are limited in coverage, and the preferred option will be the supply of fibre to the site. During the site visit it was observed that mobile coverage was sporadic and required a local booster at the base camp to get a working signal. To ensure reliable mobile wireless coverage of the site for broadband internet connectivity it is proposed to install a radio repeater at the site.

For internal communications on site it is proposed that mobile radios.

18.6 Accommodation

A new 204-man accommodation village will be built to the north of the project and shielded from both the noise and light generated at the plant by a series of local hills. The village will be located approximately 2 km from the processing plant, providing ease of access for personnel. A new unsealed access road will be constructed between the camp and the mine administration.

18.7 Security

New security fencing will be erected around the stores yard and remote borefield installations. Stock fencing shall be erected around process water dam. Due to the remoteness of the operation no allowance has been made for security fencing around entire plant or for access security boom gates.

18.8 Roads

18.8.1 Site access road

The Project includes developing a 10.5 km all-weather unsealed road including a crossing of Sandy creek from the Selwyn Toolebuc Road to the plant which then continues onto the accommodation village. This new road will enable two trucks to pass and will also cater for the operations' light vehicle traffic. A tee junction intersection will be provided on Selwyn Toolebuc Road in accordance with the requirements of the local authority. This is a public road and it is currently used by trucks and light vehicles within the area.

18.8.2 Haulage and general site roads

Internal roads will not be sealed but will be made from locally sourced crushed aggregate.

18.9 Tailings storage

For this study it has been assumed that tailings generated by the process will be safely pumped from the processing plant to mined-out pits.

The settling characteristics of combined locked cycle rougher and cleaner tails for composites from Zones 1, 2, 3, BHZ, and Bridge Zone were assessed using static cylinder method for durations of 1 hour by ALS Metallurgy. A standard anionic flocculants (910VHM) was used with three doses (0.5, 2, and 3 ml) per composite, a second flocculant (945SH) was tested but settled densities were lower than for 910VHM at the same dosage.

All five composites exhibited rapid settling rates. Using a 0.5 ml dosage of 910VHM solids content ranged from 61.6 – 66.1% with settled densities between 1.82 to 1.95 t/m^3, averaging 64.3% solids and 1.89 t/m^3. A settled density of 1.7 t/m^3 was assumed for the determination of the in-pit tails capacity.

The approximate volume and processed material equivalent capacity for the tainlings pits is presented in Table 18.1.

Table 18.1 Tailings pit approximative capacity

Tailings pit name	Volume available for tailings (excl. top 5 m)	Mill feed equivalent capacity
	M m^3	Mt
BHZ	1.0	1.4
MAIN1	0.9	1.3
MAIN2-MAIN3	6.2	8.9
Total	**8.0**	**11.6**

A 5 m freeboard was used to limit the maximum height of tails to allow for rain events. In addition, 10 m pillars were left between tailings pits and other pushbacks.

AMC recommends that additional studies (environmental, hydrogeological) be conducted to understand the detailed mechanisms associated with floor and pit wall preparation to ensure this tailings disposal method is sound from an engineering perspective and environmentally viable.

19 Market studies and contracts

Marketing study for the concentrate products to be produced from Pegmont was contributed to by Ocean Partners.

19.1 Global Zn and Pb concentrate markets

Recent Zinc concentrate market tightness was unstainable, and the market is in the process of transitioning to increased concentrate supply, driven in part by new mine production. Spot treatment charges (TC) reflect this recent shift. These less extreme market conditions are expected to prevail through the medium term. While recent lead concentrate market tightness is similarly expected to be unsustainable, the transition to improved concentrate supply has been lagging that of Zinc.

While the medium and longer-term concentrate markets for both Zinc and Lead are expected to be less extreme, they will continue to be supportive of new mine production emerging, such as that from Pegmont. Critically, the Chinese are expected to remain structurally short of Zinc and Lead concentrate and will therefore need to continue to draw on supply from the western world in order to satisfy their growing demand. Further, key industry participants have demonstrated a focus on reducing mine supply when not warranted by market conditions. Accordingly, it is expected that both from a price and TC perspective, the markets will be supportive in the longer term, affording higher prices and lower TC's versus historical levels.

19.2 Pegmont concentrate quality

The Pegmont Zn and Pb concentrate qualities are both reasonably clean, and as such are expected to be marketable to a wide number of smelters and other potential buyers. The Project will produce two concentrates, being a 65-70% lead concentrate with payable silver credits, a 49 – 55% zinc concentrate.

Potential buyers are expected to attribute value to Pegmont's potential as a blend with more complex feeds. However, this needs to be weighed against the fact that the clean nature of the concentrates, both lead and zinc concentrates, provide little by-product value for smelters.

The lead and zinc concentrate specifications shown in Table 19.1 and Table 19.2 respectively are based upon the metallurgical test work carried out to date, as described in Section 13. Though it is noted that concentrate grades will be impacted by the blend of feed entering the plant as well as by the flotation conditions at that time, and hence will vary over time.

China imposes an importation limit on cadmium levels in the zinc concentrate of 3,000 ppm. Several of the zones at Pegmont exceed 3,000 ppm. Achieving levels below 3,000 ppm, through blending and scheduling, will allow for a China deliverable quality which provides access to a much larger, and very aggressive market.

The following payables, and treatment and refining charges have been assumed for purposes of the PEA.

Lead concentrate:

- Lead payment: 95%, min deduction three units.
- Silver payment: 95%, min deduction 50 grams/dmt.
- Silver refining charge: US$0.6 – US$1/payable oz.
- Treatment charges: US$170/wmt.
- Potential penalties:
  - Cloride (C) and Fuoride (F): US$2/100 ppm > 500 ppm combined.

Zinc concentrate:

- Zinc payment: 85%, minimum deduction eight units.
- Treatment charges: US$180/wmt.
- Potential penalties:
  - Fe: US$1.50/1% > 9%.
  - Pb: US$2/1% > 3.5%.
  - Cd: US$1.50/0.1 > 0.4%.

Table 19.1 Lead concentrate specifications

Flotation testing method		Zone 1 sulphide	Zone 1 transition	Zone 2 sulphide	Zone 3 sulphide	Bridge Zone sulphide	BHZ S sulphide	BHZ transition	Zone 5 lens B sulphide	Zone 5 lens C sulphide
		Locked cycle	Locked cycle	Locked cycle	Locked cycle	Locked cycle	Batch	Batch	Batch	Batch
Zn	%	3.7	2.62	3.61	3.0	3.5	3.75	10.7	3.11	3.02
Pb	%	66.9	67.9	65.5	66.5	67.2	70.5	65.9	65.6	66.1
Ag	ppm	117	92.8	94.7	112	119	126	82	79	89
F	ppm	<1000	<1000	<1000	<1000	<1000	147	<50	240	245
Cl	ppm	800	500	800	900	700	600	500	100	600
SiO2	%	3.55	3.12	4.36	4.2	4.07	3.37	1.17	4.63	NSS
Hg	ppm	<1	1	<1	<1	1				
Al	%	0.18	0.15	0.27	0.21	0.22	0.11	0.06	0.21	0.34
As	ppm	42.7	8.8	29.7	24.7	17.2	<50	<50	<50	<50
Ba	ppm	10	10	20	10	10	<50	<50	<50	<50
Be	ppm	6.29	5.16	6.89	8.74	6.18	<10	<10	10	10
Bi	ppm	99.8	16.65	6.75	11.3	23.4	210	90	100	250
Ca	%	0.96	0.64	0.78	0.72	1.02	0.34	0.06	0.48	0.58
Cd	ppm	248	187.5	261	204	209	190	740	210	260
Ce	ppm	19.9	17.85	24.1	21	27.8				
Co	ppm	10.5	2.1	18.5	4	6.8	<10	<10	<10	<10
Cr	ppm	37	19	70	22	26	30	190	20	30
Cs	ppm	0.23	0.37	0.51	0	0.36				
Cu	ppm	2530	20.5	213	114	322	1820	2130	1120	970
Fe	%	4.73	3.58	4.47	5	4.22	3.03	3.45	4.27	5.26
Ga	ppm	0.64	0.44	0.8	1	0.69	<50	<50	<50	<50
Ge	ppm	0.1	0.1	0.1	0	0.1				
Hf	ppm	0.4	0.6	0.6	1	0.5				
In	ppm	0.502	0.195	0.329	0	0.424				
K	%	0.02	0.01	0.07	0	0.03	<0.1	<0.1	<0.1	0.1
La	ppm	17.3	14.9	19.2	18	24.3	<50	<50	<50	<50
Li	ppm	0.7	0.4	0.9	1	0.7				
Mg	%	0.11	0.06	0.11	0	0.11	0.05	<0.05	0.14	0.16
Mn	ppm	5480	3320	5400	5780	5330	2460	1430	5400	6070
Mo	ppm	19	19	25.6	23	20	30	40	10	10
Na	%	0.01	0.01	0.02	0	0.01	<0.05	<0.05	<0.05	<0.05
Nb	ppm	0.6	0.4	0.8	1	0.5				
Ni	ppm	15.5	7.9	15.8	15	15	10	10	20	10

Flotation testing method		Zone 1 sulphide	Zone 1 transition	Zone 2 sulphide	Zone 3 sulphide	Bridge Zone sulphide	BHZ S sulphide	BHZ transition	Zone 5 lens B sulphide	Zone 5 lens C sulphide
		Locked cycle	Locked cycle	Locked cycle	Locked cycle	Locked cycle	Batch	Batch	Batch	Batch
P	ppm	4030	2670	3250	2830	3580	1010	210	1640	2060
Rb	ppm	1.5	1.2	4.6	2	2				
Re	ppm	0.003	0.002	0.004	0	0.004				
S	%	12.9	12.95	12.8	13	12.85	13.2	17.1	12.1	12.2
Sb	ppm	31.8	10.85	35.2	40	79.3	<50	<50	<50	<50
Sc	ppm	0.3	0.3	0.3	0	0.4	<10	<10	<10	<10
Se	ppm	8	6	6	6	6	10	10	10	10
Sn	ppm	4.4	9.8	1.8	2	5.4				
Sr	ppm	10.6	7.3	8.5	7	9	10	10	10	10
Ta	ppm	0.06	<0.05	0.06	0	<0.05				
Te	ppm	1.26	0.51	0.58	1	0.5				
Th	ppm	1.8	1.77	2.72	2	2.49	<50	<50	<50	<50
Ti	%	0.012	0.01	0.016	0	0.011	<0.05	<0.05	<0.05	<0.05
Tl	ppm	0.17	0.18	0.17	0	0.38	<50	<50	<50	<50
U	ppm	5.8	5.5	5.6	6	9.3	<50	<50	<50	<50
V	ppm	12	9	12	12	12	10	<10	10	10
W	ppm	1.4	1.4	0.9	1	3.3				
Y	ppm	1.3	1.4	1.5	2	1.8				
Zr	ppm	16	21.2	20	18	19.7				

Table 19.2 Zinc concentrate specifications

Flotation testing method		Zone 1 sulphide	Zone 1 transition	Zone 2 sulphide	Zone 3 sulphide	Bridge Zone sulphide	BHZ sulphide	BHZ Transition	Zone 5 lens B sulphide	Zone 5 lens C sulphide
		Locked cycle	Locked cycle	Locked cycle	Locked cycle	Locked cycle	Batch	Batch	Batch	Batch
Zn	%	54.5	53.3	54.9	54.8	52.3	51.9	48.9	49.2	50
Pb	%	3.21	5.15	3.6	4.55	5.43	1.32	3.37	1.56	1.8
Ag	ppm	7.86	27.70	5.01	7.29	10.4	13	18	7	9
F	ppm	<1000	<1000	<1000	<1000	<1000	50	27	81	79
Cl	ppm	100	100	<100	<100	200	<100	200	200	100
SiO2	%	1.71	1.75	2.33	1.64	1.74	1.74	1.86	3.42	2.15
Hg	ppm	5	4	6	6	5				
Al	%	0.09	0.10	0.14	0.1	0.12	0.07	0.08	0.21	0.12
As	ppm	22.5	3.90	21	12.5	7.8	<50	<50	50	<50
Ba	ppm	10	10	10	10	10	<50	<50	<50	<50
Be	ppm	4.0	3.1	4.8	4.3	3.0	<10	<10	10	<10
Bi	ppm	6.29	2.28	0.65	1.92	2.82	30	20	<20	20
Ca	%	0.23	0.27	0.27	0.26	0.32	0.18	0.11	0.31	0.15
Cd	ppm	3510	3250	3640	3490	2980	2830	2740	3580	3830
Ce	ppm	5.24	6.76	9.39	4.87	5.97				
Co	ppm	26.8	13.50	21.8	15.2	54	20	10	10	10
Cr	ppm	29	21	44	21	26	50	70	60	30

Flotation testing method		Zone 1 sulphide	Zone 1 transition	Zone 2 sulphide	Zone 3 sulphide	Bridge Zone sulphide	BHZ sulphide	BHZ Transition	Zone 5 lens B sulphide	Zone 5 lens C sulphide
		Locked cycle	Locked cycle	Locked cycle	Locked cycle	Locked cycle	Batch	Batch	Batch	Batch
Cs	ppm	0.11	0.23	0.18	0.3	0.24				
Cu	ppm	674	350	448	617	1085	1210	950	520	600
Fe	%	6.71	5.85	5.71	5.53	6.75	9.09	11	10.65	9.96
Ga	ppm	0.44	0.41	0.58	0.46	0.5	<50	<50	<50	<50
Ge	ppm	0.08	0.09	0.07	0.08	0.08				
Hf	ppm	0.2	0.3	0.3	0.2	0.3				
In	ppm	5.76	3.88	4.98	4.88	6.52				
K	%	0.01	0.01	0.02	0.01	0.01	<0.1	<0.1	<0.1	<0.1
La	ppm	4.3	5.50	7.4	3.9	4.9	<50	<50	<50	<50
Li	ppm	0.4	0.40	0.5	0.4	0.5				
Mg	%	0.05	0.03	0.06	0.05	0.04	<0.05	<0.05	0.12	0.06
Mn	ppm	2990	2220	3770	2600	3240	1880	2550	6000	3340
Mo	ppm	2.05	2.04	2.61	1.71	2.3	<10	10	10	<10
Na	%	0.03	0.03	0.03	0.03	0.03	<0.05	<0.05	<0.05	<0.05
Nb	ppm	0.5	0.30	0.5	0.4	0.4				
Ni	ppm	13.8	6.8	15.3	11.1	10.6	10	40	20	20
P	ppm	760	910	940	870	790	320	310	810	410
Rb	ppm	0.6	0.80	1.3	1.7	1				
Re	ppm	<0.002	<0.002	<0.002	<0.002	<0.002				
S	%	27.1	30.40	27.2	27.2	26.7	29.8	31.4	28.8	29.8
Sb	ppm	9.25	3.56	10.9	13	16.95	<50	<50	<50	<50
Sc	ppm	0.1	0.20	0.2	0.2	0.2	<10	<10	<10	<10
Se	ppm	11	11	11	11	11				
Sn	ppm	1.1	1.4	0.9	1.3	2				
Sr	ppm	2.3	2.9	2.7	2.2	2.3	10	10	10	10
Ta	ppm	0.08	<0.05	<0.05	<0.05	0.05				
Te	ppm	0.06	0.06	0.06	0.05	0.05				
Th	ppm	0.83	0.91	1.72	0.95	1.01				
Ti	%	0.007	0.01	0.009	0.007	0.008	<0.05	<0.05	<0.05	<0.05
Tl	ppm	0.43	0.16	0.28	0.19	0.23	<50	<50	<50	<50
U	ppm	2.5	2.70	3.3	2.7	3.1	<50	<50	<50	<50
V	ppm	5	5	7	4	6	<10	<10	10	<10
W	ppm	1.2	0.9	0.6	0.8	3.2	<50	50	50	<50
Y	ppm	0.5	0.6	0.8	0.6	0.8				
Zr	ppm	6.7	10	9.9	7.3	9.1				

19.3 Contracts

There are no contractual arrangements for smelting at this time, nor are there any contractual arrangements for the sale of lead, zinc, or silver products, such as streaming or off-takes at this time.

19.4 Metal price and exchange rate forecasts

For the Pegmont PEA a number of sources were reviewed to determine an appropriate long-term price of lead, zinc, and silver. The assessment included market forecast information (institutional consensus prices), current metal prices and rolling three-year London Metal Exchange (LME) averages.

Historical lead, zinc and silver prices are shown in Figure 19.1, Figure 19.2, and Figure 19.3. Historical exchange rate trends are plotted in Figure 19.4.

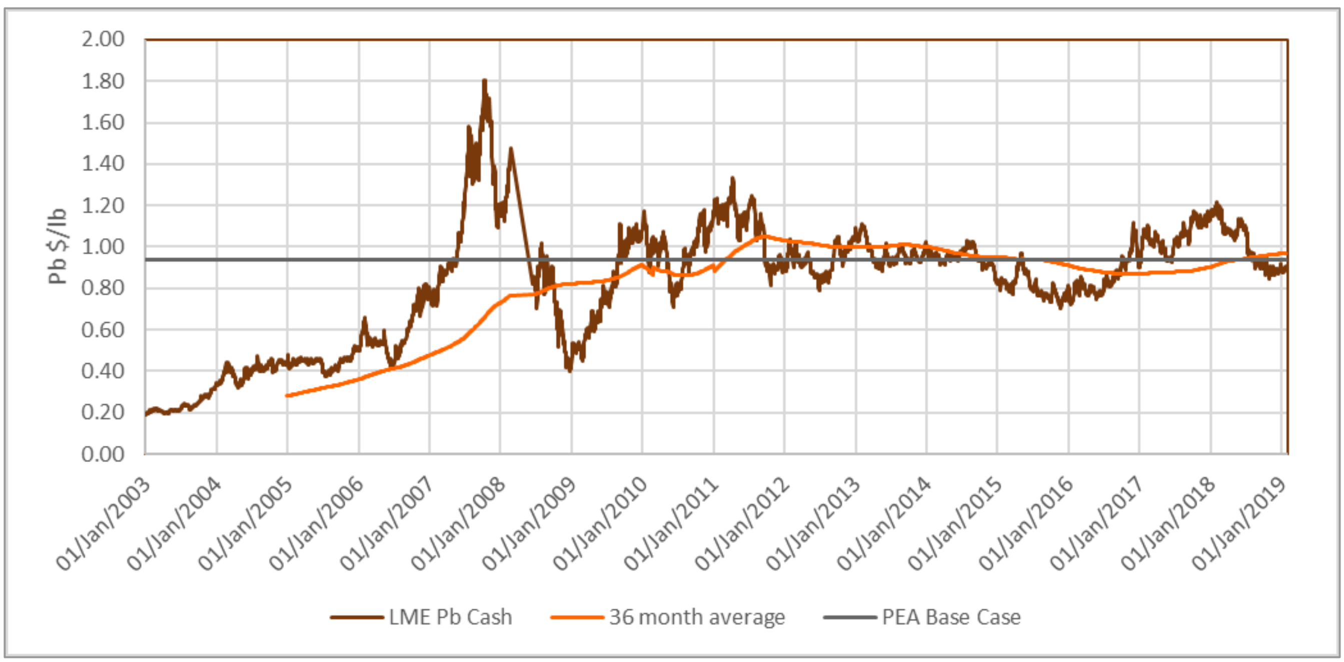
The metal prices used in the PEA economic analysis base and spot price cases are given in Table 19.3. It must be noted that metal prices are highly variable and are driven by complex market forces and are extremely difficult to predict. The spot prices for lead, zinc and silver prices are based on the LME cash buyer at the close of trading on 22 January 2019. The spot exchange rate is the official Reserve Bank of Australia rate on the same day.

A sensitivity analysis on metal prices and exchange rates was completed as part of the overall economic analysis. The results of this are discussed in Section 22.

Table 19.3 Metal price and exchange rate

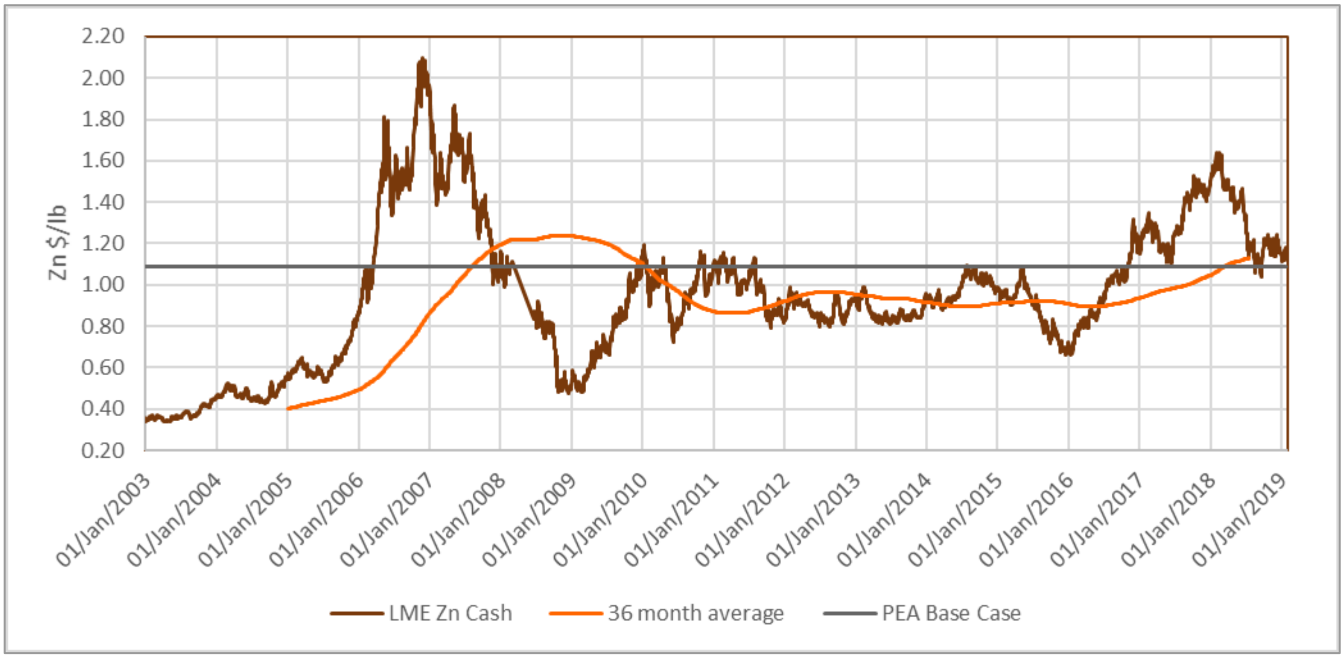
	Unit	Base case value	Spot price case (22 Jan 2019)
Lead price	US$/lb	1.09	1.18
Zinc price	US$/lb	0.94	0.91
Silver price	US$/oz	16.50	15.31
Exchange rate	US$:A$	0.75	0.71

Figure 19.1 LME Lead cash price, trailing 3-year average, data source: LME



Source: Ocean Partners

Figure 19.2 LME zinc cash price, trailing 3-year average, data source: LME

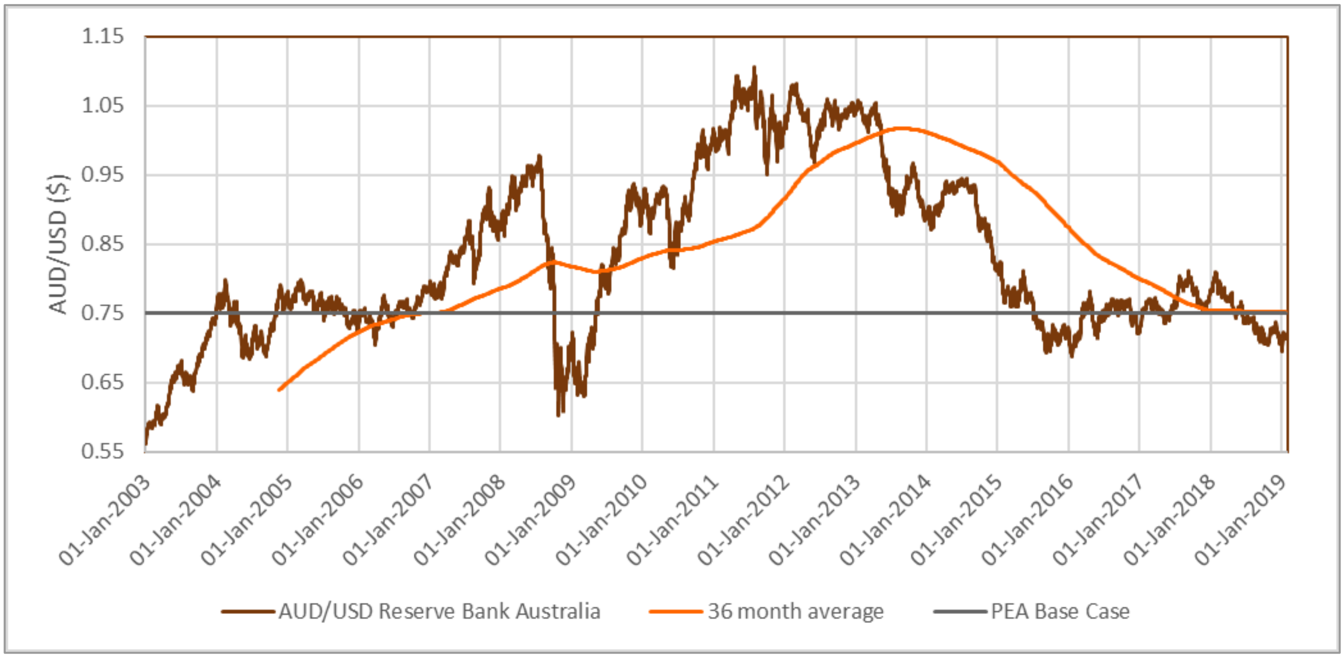


Source: Ocean Partners

Figure 19.3 Silver price - London fixing, trailing 3-year average, data source: London Bullion Market Association (LBMA)

Source: Ocean Partners

Figure 19.4 US$:A$ exchange rate and trailing 3-year average, data source: Reserve Bank of Australia



Source: Ocean Partners

20 Environmental studies, permitting, and social or community impact

20.1 Relevant legislation

The environmental and native title management of mining and exploration activities in Queensland is covered by a number of state and Commonwealth Acts, including:

- National Greenhouse and Energy Reporting Act 2007 (NGER Act) (Commonwealth).
- Native Title Act 1993 (Commonwealth).
- Environment Protection and Biodiversity Conservation Act 1999 (EPBC Act), Environment Protection and Biodiversity Conservation Regulation 2000, Environment Protection and Biodiversity Conservation Amendment (Independent Expert Scientific Committee on Coal Seam Gas and Large Coal Mining Development) Act 2012 and Environmental Offsets Policy 2012 (Commonwealth).
- Aboriginal and Torres Strait Islander Heritage Protection Act 1984 and Native Title Act 1993 (Commonwealth).
- Aboriginal Land Act 1991 (Queensland).
- Aboriginal Cultural Heritage Act 2003 (Queensland).
- Native Title (Queensland) Act 1993 (Queensland).
- Land Act 1994 (Queensland).
- Regional Planning Interests Act 2014 and Regional Planning Interests Regulation 2014 (Queensland).
- Soil Conservation Act 2014 (Queensland).
- Biosecurity Act 2014 and Biosecurity Regulation 2016 (Queensland).
- Environmental Protection Act 1994 (EP Act) and the related Environmental Protection Regulation 2008 (EP Reg) (Queensland).
- Nature Conservation Act 1992, Nature Conservation (Wildlife Management) Regulation 2006, Nature Conservation (Protected Plants) Conservation Plan 2000 and the Nature Conservation (Wildlife) Regulation 2006 (Queensland).
- Vegetation Management Act 1999 and Vegetation Management Regulation 2012 (Queensland).
- Environmental Offsets Act 2014, Environmental Offsets Regulation 2014 and the Environmental Offsets Policy 2014 (Queensland).
- Queensland Biosecurity Act 2014 (Queensland).
- Water Act 2000, Water Regulation 2002, Water Plan (Gulf Basin) 2007 (Queensland).
- Mineral Resources Act 1989 (MR Act) (Queensland).

Environmental approvals and enforcement in Queensland primarily fall within the scope of the EP Act. The Queensland Department of Environment and Science (DES) has the responsibility of regulating this Act. Other state departments may have permitting jurisdiction under other legislation, subject to nature of proposed activities. The Project may also require approval from the Federal Environment Minister if the action has, will have, or is likely to have a significant impact on a Matter of National Environmental Significance (MNES).

20.2 Approvals and permitting

20.2.1 Existing approvals and permits

20.2.1.1 Environmental authorities

PML currently holds two Environmental Authorities (EA) for the Project which are described as EPSX00957013 and EPSL00057813. The Project currently encompasses three ML that are subject to EA EPSX00957013 (EPM 14491, EPM 15106 and EPM 26210) and three EPMs that are subject to EA EPSL00057813 (ML 2620, ML 2621 and ML 2623). It should be noted that EPM 14491 and EPM 15106 have been amalgamated with EPM 26210.

EA EPSX00957013 is subject to standard conditions contained in the document Eligibility criteria and standard conditions for exploration and mineral development projects – Version 2. EA EPSL00057813 is subject to Standard Environmental Conditions contained in the Code of Environmental Compliance for Mining Lease Projects (EM588) (EHP 2001).

20.2.1.2 Groundwater extraction licenses

The project has a water extraction permit (405106) issued by the DERM and is in good standing until 2020, allowing the extraction of 2 megalitres per annum from mining lease ML 2620. This is sufficient to conduct the planned exploration activities. If the demand for water increases, then further supply options will be subject to approval under the Water Act 2000.

20.2.2 Approvals strategy

To develop the Project further, the environmental approvals process must be completed. The approvals process for the Project would involve combining the two EAs, into a single site-specific EA that is Level one, non-code compliant. This process would involve the creation of a new ML over part of the existing EPM 26210. Subject to the final mine design and potential impacts, the Project EA is expected to be amended by the 'Major EA Amendment with no Environmental Impact Statement (EIS) required process.

20.2.2.1 Major amendment approvals

Prior to commencing the Major EA Amendment and Supporting Information approvals process, PML will apply for a Pre-lodgment meeting with DES to seek direction and advice on whether a proposed application will meet the legislative application requirements.

Under the Environmental Protection Act 1994 (EP Act), DES must decide whether the proposed amendment is a major amendment or a minor amendment. This decision is called the assessment level decision. It is not envisioned the Project would allow for a minor amendment process, therefore timelines provided in Section 20.2.2.2 are for the major amendment process only.

20.2.2.2 Additional approvals

The EA amendment to allow mining on additional ML(s) is only valid once the ML is granted. The EA and ML processes occur simultaneously, including lodgment of applications and the public notification process. The ML can only be granted once other approvals are in place, including landowner agreements (Conduct and Compensation Agreements) and native title agreements (an Indigenous Land Use Agreement and a Cultural Heritage Management Plan).

Other approvals which may be required as part of the project are road use agreements for transport of product, or off lease approvals for transport / utility corridors.

20.2.2.3 Approvals timeline

An EA amendment will be sought to authorize the additional ML, change in mining activities, inclusion of additional environmentally relevant activities (such as mining of ore) and allowance for the associated disturbance footprint.

Once a major amendment has been approved and an amended EA has been issued, the EA holder must submit a replacement Plan of Operations and updated financial assurance to be approved by DES prior to the commencement of operations.

The Major Amendment process and timeline is summarized in Table 20.1. A flow chart detailing the complete major amendment process and timeline, including the objections and appeals processes is provided in Attachment 1. The timeline assumes no impacts on MNES. As the Major Amendment process does not allow for a bilateral agreement between the Commonwealth and State, if there are impacts of MNES, a separate 12 months approvals process will be required.

Table 20.1 Major amendment summary timeline

Major amendment (including public notification)		
Task	**Business days**	**EP act section**
Completion of technical studies and drafting of EA Amendment Application	Up to 6 months	-
Submit EA Amendment Application to DES and lodge ML Application to DNMRE	-	-
Notice of Assessment Level Decision (ALD)	20	228, 229
Information Request Issued	20	144
Applicant Responds to Information Request	Up to 6 months	-
Public Notice / Declaration of Compliance (for EA and ML)	40	157 - 159
Decision to Approve	25	168, 181
Issue of amended EA (objections and land court hearing if applicable)	25	195
Granting of additional ML (subject to subject to compensation agreements, native title agreements, objections and land court hearing if applicable)	Up to 18 months	-
Total	**155**	**-**
Total	**8 – 24 months**	**-**

20.3 Summary of environmental values / constraints

As part of the EA amendment approval process the proponent will be required to complete baseline studies describing conditions for all environmental values of the Project site, the likely impacts of the project and proposed mitigation and management strategies. The following section provides a summary of Environmental Studies that have been or will likely need to be undertaken to support the Major EA Amendment process.

20.3.1 Existing environmental studies and data

Environmental studies completed on the Project site to date have consisted of one baseline terrestrial ecology survey. An early wet season (Nov – Jan) survey was undertaken over the Project site in November 2017. A progress update letter was issued to PML in December 2017 summarizing the findings of the baseline ecology survey. An early dry season (May – July) survey is to be completed to align with the current best practice fauna trapping methodology. Following the completion of the second survey, a baseline Terrestrial Ecology report will be prepared for the Project.

The findings of the early wet season ecology study are summarized below.

20.3.1.1 Biodiversity values and species of conservation significance

The findings of the flora and fauna survey are summarized as follows:

- Seven Regional Ecosystems (REs) occur on the Project site, all of which are listed as 'Least Concern' under the Vegetation Management Act and 'No Concern at present' under the Queensland Biodiversity Status.
- No Threatened Ecological Communities (TECs) as listed under the EPBC Act occur within the Study area.
- The suite of fauna species recorded on site included 41 birds, 19 reptiles, 8 mammals, and 2 amphibians.
- Of the 41 birds recorded on site, 7 species are listed as wetland indicator species as identified in the DES WetlandInfo database. The presence of these species reflects the wetland values on the Project site, which assists in determining environmental offsets under offsets legislation.
- No flora or fauna species listed as threatened under the NCWR or EPBC Act have been identified to date.
- One marine bird species, the Rainbow Bee-eater (Merops ornatus) was recorded on the Project site. The Rainbow Bee-eater is listed under the EPBC Act as a marine species. This species occupies much of mainland Australia, several near-shore islands, and is known to migrate to Papua New Guinea and eastern Indonesia for the non-breeding period.
- Three introduced fauna species, including the House Mouse (Mus musculus), European Rabbit (Oryctolagus cuniculus) and the One-humped camel (Camelus dromedaries) have been recorded on site.

While further technical assessment of the ecological values is still required, the values described above are considered unlikely to present a significant constraint to Project approval.

20.3.2 Environmental studies to be completed

Several baseline studies and management plans will be required to support the Project development application. A summary of baseline studies and reports yet to be completed is provided in Table 20.2.

Table 20.2 Baseline studies progress to date

Study aspect	Further requirements
Environmental Offsets	No assessment of offset obligations completed to date. Final offset obligations will be determined based on findings of field surveys consistent with industry guidelines; Queensland Environmental Offsets Policy v1.1 2014, which is based on the Environmental Offsets Act 2014. The offsets assessment should consider offsets required under the EPBC Act Environmental Offsets Policy 2012 if required. This will be included in the Supporting Information Document.
Aquatic Ecology	No aquatic surveys have been undertaken at the Project site. Aquatic sampling and trapping surveys will be undertaken within the Project site and it's receiving environment to identify relevant values potentially impacted by the proposal. A baseline Aquatic Ecology report will be prepared.
EPBC Referral	Requirements for EPBC Act referral to will confirmed following completion of field surveys and preparation of infrastructure plan. Desktop assessment of footprint to identify potential impacts to Matters of National Environmental Significance as verified during field surveys will be undertaken.
Land	Soil surveys and land suitability assessment to be completed.
Air and Greenhouse Gas	Baseline dust monitoring, air modelling and reporting to be completed.
Noise and Vibration	Baseline noise monitoring, noise modelling and reporting to be completed.
Surface Water	Baseline surface water monitoring will be undertaken to establish background water quality. Designated surface water monitoring sites will be established, and sufficient relevant baseline monitoring data will be collected to support the assessment. Flood modelling, water balance modelling and a water management plan will be completed.
Groundwater	Baseline groundwater monitoring will be undertaken to collect information about existing aquifer conditions. Groundwater monitoring bores will be installed at designated locations to enable the collection of sufficient baseline groundwater quality data required to support the assessment. Groundwater monitoring and an underground water impact report will be completed. Make good agreements between the landowner and groundwater user (PML) may be required if it is assessed landowner bores may be impacted.
Indigenous and Non-indigenous Cultural Heritage	Surveys to be undertaken to identify areas of historical cultural significance. Develop a Cultural Heritage Management Plan in consultation and cooperation with Native Title holders. An Indigenous Land Use Agreement may be required. Non-indigenous cultural heritage assessments will only be required if there are relevant heritage areas mapped over the proposed site.
Waste Rock, Ore and Residue Characterization	Waste rock testing will be undertaken to characterize waste rock, ore and residue. Waste Rock Characterization report to be prepared.
Water Supply	Desktop assessment and reporting will be undertaken. Further water licenses may need to be obtained if feasibility studies indicate more than 2 megalitres per annum are required for processing.
Rehabilitation	A suitable rehabilitation strategy will be formulated following preparation of the mine plan.
Mine Engineering Design	Mine Engineering Design documents will be drafted, in accordance with proposed mine plane.

20.3.3 Environmental constraints identified (to date)

With the exception of one Commonwealth listed marine species (Rainbow Bee-eater), no species or communities of conservation significance have been identified on the Project site, to date. The Rainbow Bee-eater is a highly mobile and seasonally abundant species, with a widespread distribution throughout mainland Australia. It is unlikely that Project development will significantly impact this species.

As highlighted in Section 20.3.2, several environmental studies will need to be completed in order to identify all environmental constraints relevant to the Project.

20.4 Native title and cultural heritage

Native title is defined as the rights and interests that are possessed under the traditional laws and customs of Aboriginal and Torres Strait Islander peoples, and that are recognized by common law.

Generally, applications for a resource permit must nominate a native title process at the time of making the application. Some types of tenures, including the valid grant of tenements prior to 1 January 1994, have had the existence of native title extinguished and are not subject to the native title process. Given that the three existing mining leases were granted prior to this date, applications over these leases are not subject to a native title process. This remains in effect for all subsequent renewals on the same terms as the original grant.

The two exploration leases were granted after 1 January 1994 and native title may continue to exist over the EPMs. On granting of the EPMs, a set of conditions have been attached known as the “Native Title Protection Conditions”, this describes the process of engagement with native title parties on undertaking exploration activities including procedures for cultural heritage finds and native title parties field inspection of the intended exploration area.

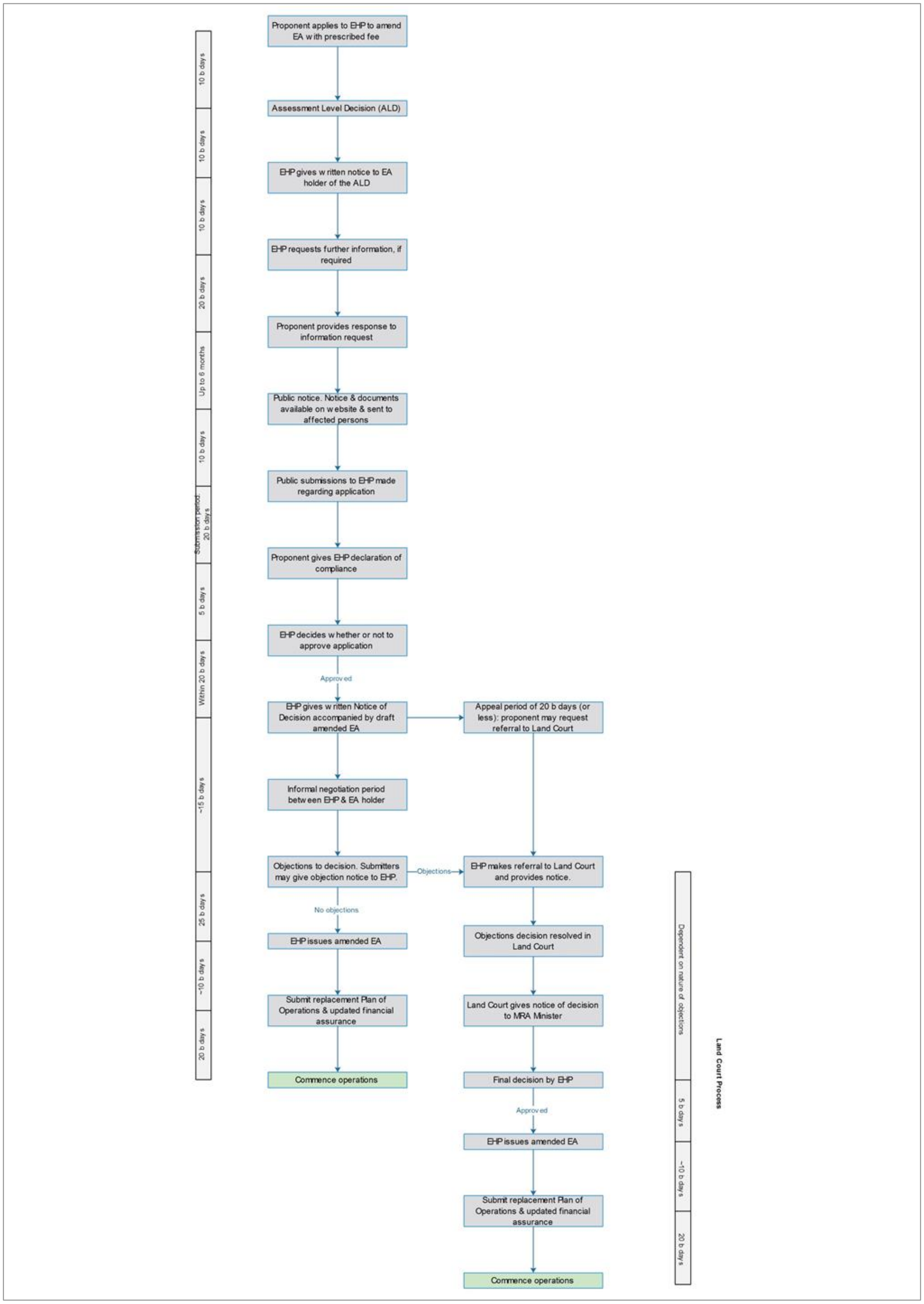
Future application for mining licenses over the existing EPMs or variation of the terms of the existing MLs would require a statutory negotiation process to be undertaken at the time.

The Pegmont project carries a duty of care under the Aboriginal Cultural Heritage Act 2003 (QLD) to protect Aboriginal cultural heritage when carrying out exploration activities on both the EPMs and MLs.

20.5 Closure and remediation

The conditions of an EA granted for the Project would require the Proponent to provide Financial Assurance in the amount and form agreed upon by the administering authority. This amount will act as a source of financial security to cover any costs or expenses incurred in taking action to prevent or minimize environmental harm or rehabilitate or restore the environment should the EA holder fail to meet their EA obligations. To minimize financial assurance commitments, rehabilitation would be undertaken progressively where permitted by the mine design and activities. Rehabilitation methods are expected to be best practice in line with the arid conditions in the North-west Highlands bioregion. The FA will need to be reviewed throughout the life of the mine when a plan of operations is amended or replaced, or the EA is amended. The FA may also be discharged at the conclusion of activities and at the time of surrendering the EA, subject to successful rehabilitation.

Figure 20.1 Major EA amendment flowchart



Source: AARC

20.5.1 Capture of topsoil

On commencement of operations, topsoil will be recovered from all mining and infrastructure areas. It is anticipated that topsoil will be recovered to a depth of 0.5 m and then stored in an area away from creeks or rivers. The soil should not be stacked or driven on by heavy equipment in order to preserve the seed bank. A windrow should be built around the stockpile as a measure to prevent the soil from being washed away during heavy rain events.

20.5.2 Waste dumps

The three major waste dumps will require re-sloping. Final rehabilitation designs for all the dumps will be done to 15 m high batters at 21 degrees. Berms will be set at 8 m in width. Following this, topsoil will be spread over the dumps to a thickness of 0.3 m prior to revegetation.

20.5.3 In-pit waste dumps

Three in-pit waste dumps will be constructed in the Main 7 pit. The rehabilitation plan for Main 7 is to allow for water to fill the pit to a level 50 m below the pit crest. This will effectively submerge two of the dumps, leaving one to be rehabilitated. This in-pit dump will need to have its top 10 m pushed down to create the rehabilitation slopes at 21 degrees. The top of the rehabilitated structure will be sloped at 3 degrees to avoid pooling of water. The areas of the in-pit dump which will be submerged will be left at 37 degrees and require no rehabilitation work. All re-sloped areas on this in-pit dump will then be covered with 0.3 m of topsoil and revegetated.

20.5.4 Tailings storage

Tailings from the processing plant is to be stored in three of the mined-out pits. The tailings will be filled to a pre-determined level and then allowed to evaporate and be dewatered. Once sufficiently stable they will level be capped with waste. Additional waste will then be added to allow for a slope to be created on the footprint of the former pit. The slope will be built at around three degrees and allow for water to run off the pit’s footprint, preventing pooling on top of the tailings and waste. These areas will then be covered with 0.3 m of topsoil and revegetated.

20.5.5 Stockpiles and roads

It is expected that roads and stockpiling areas will be cleared upon cessation of the mining operations. These areas will not require any major re-leveling. All former roads and stockpiling areas will be covered by 0.3 m of topsoil and revegetated.

20.5.6 Infrastructure

Infrastructure, including the workshop, camp, processing Plant, ANFO facility, etc., will need to be demobilized. The area will be re-sloped to the same dimensions as the Waste Dump, covered in 0.3 m of topsoil and then revegetated.

20.5.7 Pits

Main 7 will be the only pit that is not used for tailings storage. Two of the three waste dumps in this pit will be submerged when groundwater is allowed to infiltrate the pit. The pit will require a bund be constructed around it prior to being handed back to the state government as a lake.

20.5.8 Closure and remediation costs

Closure and remediation costs have been estimated at a high-level based on the requirements described in the previous paragraphs. The total closure costs have been estimated at approximately A$15M.

21 Capital and operating costs

21.1 Operating costs

21.1.1 Processing costs

Operating costs for the processing plant are deemed to be of a level of accuracy consistent with industry standards for a PEA. The costs are presented in A$ and are based on prices obtained during the third quarter of 2018 (3Q18) and exclude the GST cost component.

The costs cover the processing of mineralized material from the ROM pad battery limit. This includes the sections covering crushing, milling, flotation, dewatering, concentrating handling and trucking, site services (power, air, and water), and administration costs.

21.1.1.1 Summary

Operating costs have been developed using the parameters specified in the process design criteria in conjunction with the resource model. Annual operating costs and costs per tonne have been developed for the processing plant and are summarized in Table 21.1.

The operating cost estimate has been developed on the basis of a process plant feed tonnage of 1,100,000 tonnes per annum.

The operating cost for the processing plant is A$28.93M per annum or A$26.30 per tonne material processed excluding concentrate transport.

Concentrate transport of the lead has been based on A$50 per wet concentrate tonne rate generated from pricing developed from discussions with haulage companies from the project site to Mt Isa. This equates to a A$4.45 per tonne of ROM processed material. Zinc concentrate transport consist of both trucking and rail components and is based on a new siding facility at Malbon, south of Cloncurry. At the rail facility the sealed half height concentrate containers will be transferred to a 48 carriage train for transport to Sun Metals at Townville on a monthly basis. The transport cost has been estimated on information provided by the trucking company, rail infrastructure charges, rail operator charges in Gross Tonne Kilometres (GTK), and loading and offloading charges. This equates to a transport cost of A$100.58 per wet concentrate tonne or A$3.71 per tonne of ROM treated material.

The adjusted operating cost inclusive of concentrate transport is A$37.90M per annum or A$34.46 per tonne of material processed.

Table 21.1 Annual processing costs (OPEX)

Cost centre	Annual cost A$M	Unit cost A$/t
Processing costs (processing plant and admin)		
Crushing	1.64	1.49
Fine ore storage and handling	0.10	0.09
Grinding and classification	8.87	8.06
Flotation (includes regrinding stages)	5.33	4.85
Concentrate dewatering	0.13	0.12
Concentrate filtering	0.33	0.30
Concentrate load out	0.19	0.17
Reagent mixing and dosing	0.09	0.08
Process water	0.44	0.40
Raw water	0.11	0.10
Tailings thickening and disposal	0.21	0.19
Tailings return	0.09	0.08
Air services supply & reticulation	0.32	0.29
Processing administration	11.02	10.01
Lighting and workshop	0.08	0.07
Sub total	**28.93**	**26.30**
Concentrate transport		
Lead concentrate	4.90	4.45
Zinc concentrate	4.08	3.71
Total operating cost	**37.90**	**34.46**

21.1.2 Qualifications and exclusions

The operating costs presented have been calculated from first principles and budget quotations for supply of chemicals, materials, and services. The following items have been excluded from the operating cost estimate:

- All head office costs and corporate overheads.
- Exchange rate variations.
- Escalations.
- Project financing costs.
- Interest charges.
- Political Risk Insurance.

The Australian Goods and Services Tax (GST) will not be a cost to the Project as an income generated by the Project in the form of product sales is GST free (as the products are considered export commodities) and will not attract output credits. However, the GST has cash flow implications for the Project, as GST will be applied to most inputs, including consumable costs. Any GST payable on these inputs listed below can be claimed back from the Australian Taxation Office (ATO):

- Land compensation / land owner's costs.
- Subsidies to the local community.
- Rehabilitation costs.
- Amortization and depreciation charges.

21.1.2.1 Tailings storage

- In-pit tailings deposition has been adopted using one of the early pits with the major amount of future tailings accommodated for within the main pit. Costs associated with tailings disposal have been allocated to mining.

21.1.2.2 Product

- Product transport from site by truck has been included in the operating costs as follows
  - Lead concentrate will be trucked in specifically designed bulk haulage side tippers to Mt Isa smelter.
  - Zinc concentrate will be trucked in sealed half height sea containers to Malbon siding, for transfer to trains for shipment to Sun Metals or the Port of Townsville.

21.1.2.3 Environmental

- Plant site rehabilitation costs have been excluded from the processing operating cost estimate.
- Tailings rehabilitation costs have been excluded from the processing operating cost estimate. Environmental sampling and monitoring costs have been estimated and include assay costs, and an allowance for consultant's costs and general consumables.

21.1.2.4 Labour

- Overtime allowances have been excluded.
- Union fees have been excluded.
- The labour schedule and roster are based on:
  - A three panel, 14 days on and 7 days off fly-in and out, shift roster.
  - A back to back 8 days on and 6 days off fly-in and out, dayshift roster for maintenance and management.
- An on-cost rate of 28.92% has been applied and includes long service leave, payroll tax, and superannuation.
- Salary basis is drawn from HAYS 2018 Salary Review and references mining and processing for Queensland for the relevant positions.

21.1.2.5 Consumables

- The consumption of reagents and other consumables are based on rates from the metallurgical test work and GRES experience.
- All reagent and consumable costs have been sourced from suppliers and are calculated as free-in-store at the Project site.

21.1.2.6 Utilities

- The power schedule is based on the equipment list prepared by GRES. GRES has calculated the power draw based on standard utilization factors.
- The power cost is based on site generated power at a unit rate of A$0.204/kWh. A figure of A$0.204/kWh has been generated from first principles based on on-site generation by gas fired generating sets using standard fuel burn rates and a current gas price of A$10.30/GJ including a delivery component price from the gas line operator. The price includes the likely maintenance cost for the generators.

21.1.2.7 Laboratory

- An allowance has been made for chemical and metallurgical laboratory facility onsite to process metallurgical and daily assays.

21.1.2.8 General and administration

- General and administration costs for the processing plant operations have been estimated from other similar scale operations.

21.1.3 Processing and maintenance costs

21.1.3.1 Labour

There will be a total of 56 personnel comprising twenty operations personnel, thirteen maintenance personnel, seven logistics / stores / safety personnel and sixteen supervisory or management personnel. The workforce will reside in onsite accommodation and will travel a short distance to the operating plant on a daily basis. The workforce has been established on a fly-in and fly-out arrangement out of a Townsville based hub. Working cycles have been established to provide to the minimum number of total employees by using current Australian shift cycles.

Pegmont personnel will work the following proposed rosters:

- Management and administration 8 days on 6 days off
- Processing shift operators 14 on 7 off, 3 panel roster
- Maintenance crew 8 days on 6 days off
- Engineers 8 days on 6 days off

The total operating salaries inclusive of on cost factor is $7.22M per annum, with an extended cost inclusive of site accommodation and flights of $9.37M per annum. Salaries account for 24.9% of the operating cost with flights and camp accommodation accounting for 7.5% of the overall cost.

21.1.3.2 Power

The power demand has been calculated to be 6.23 MW, with and annual usage in 44,755 MWh for the processing plant.

The calculated total power cost for the Pegmont project is A$9.13M per annum and equates to 31.6% of the plant operating cost.

21.1.3.3 Consumables

Reagents and media consumption make up 20.7% of the operating cost estimate at A$5.99M per annum.

All reagent costs have been based on dry bulk delivery to site with onsite mixing required. Some of the reagents such as MIBC and Cytec 3418A will be supplied in liquid form in 1,000 liter liquid bulk containers.

The grinding media consumption and cost have been based on similar applications and advised by the vendors. The grinding media steel consumption is based on similar operations and equates to 1.26 kg/t.

The unit costs and consumptions used for the reagent cost estimate are summarized in Table 21.2.

Table 21.2 Reagent usage

Reagent	Unit cost A$/kg	Consumption kg/t	Consumption t/y	Cost A$M/y	Cost A$/t
Frother (MIBC)	4.84	0.12	132	0.64	0.581
Copper sulphate	2.53	0.11	121	0.31	0.278
Lead collector 3418A	16.64	0.03	33	0.55	0.499
Sodium Isobutyl Xanthate	3.59	0.03	32	0.11	0.104
Flocculant	4.63	0.02	17	0.08	0.069
Antiscalant	3.00	0.01	7	0.02	0.018
Hydrated lime	0.683	0.47	513	0.35	0.318
Sodium monophosphate	7.333	0.15	165	1.21	1.100
Reagents sub total				**3.27**	**2.23**
Grinding media (2.5 mm Ceramic)	4.67	0.40	79	0.368	0.335
Grinding media (80 mm balls)	1.69	1.26	1,391	2.353	2.14
Grinding media sub total				**2.721**	**2.47**
Total				**5.97**	**5.44**

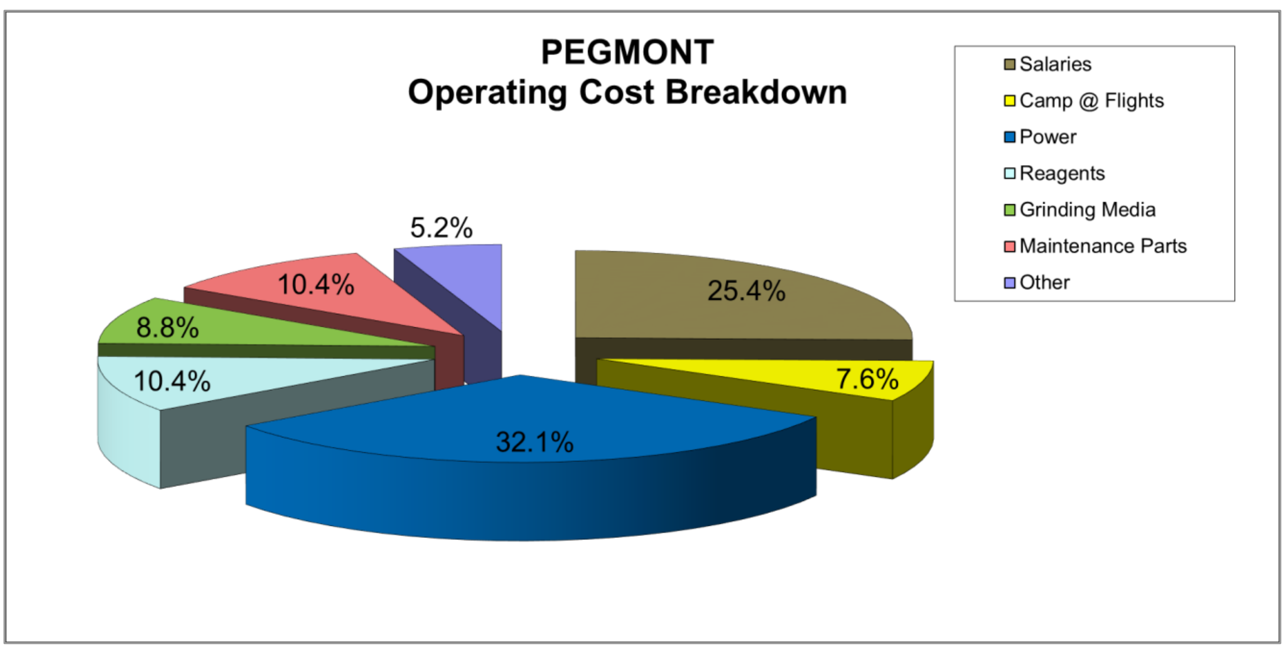
21.1.3.4 Plant maintenance

Plant maintenance (excluding labour) has been based estimated by factoring spares and consumables from the capital cost estimate for each section of the plant and equates to 10.3% of the operation cost at A$2.97M.

21.1.3.5 Laboratory

An allowance has been made for a chemical and metallurgical laboratory facility to conduct daily assay and metallurgical assaying requirements. Wet assaying costs have been estimated at A$0.67M per annum and are included within the Administration costs.

Figure 21.1 Pegmont process plant operating cost



Source: GRES

21.1.4 General and administration costs

The general and administration costs include:

- Salaries and vehicle costs for the follow site positions: The General Manager, a Site Accountant, the payroll and accounts personnel, an Environmental Superintendent and two environmental officers.
- Accommodation costs have been calculated on expected site workforce and roster using a cost of A$70 per man day in camp.
- Flight costs have been calculated on expected site workforce and roster using a rate of A$995 per return trip. This has been based on private charter operating between the Osborne Mine Airport and the Townsville airport. The service would utilize a SAAB 340b aircraft and includes departure and airport taxes.

A cost of approximately A$6.24/t for G&A were estimated by GRES.

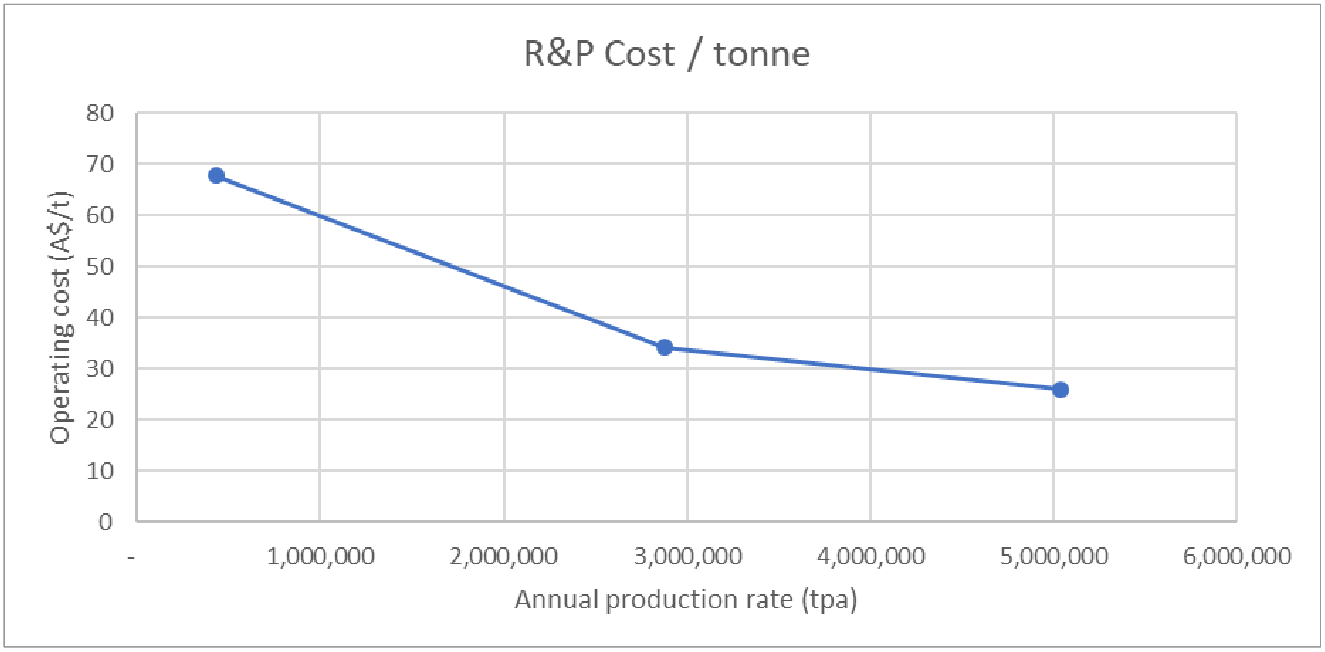
21.1.5 Underground operating costs

Underground operating costs are based on benchmark costs for R&P and Longhole stoping over a range of annual throughputs. AMC assumed that all underground mining will be undertaken by a contractor. The operating cost estimates include all labour, material, equipment supply and operation, supervision, administration, and on-site management. Labour costs include all benefits and employment taxes.

The benchmark models represent R&P mining and range from 0.4 Mtpa to 5.0 Mtpa. AMC has based the cost on a throughput of approximately 1.0 Mtpa which is representative of the combined production rate for the three areas under consideration, namely the Bridge Zone and Main Zones 3A and 3B.

Benchmark costs for varying production rates for R&P mining are provided in Figure 21.2. Given the competent host rock and the minimal need for ground support the R&P operating cost selected for this study is A$50/t of mineralized material. This benchmark cost compares well with similar-sized operating mines in AMC's database.

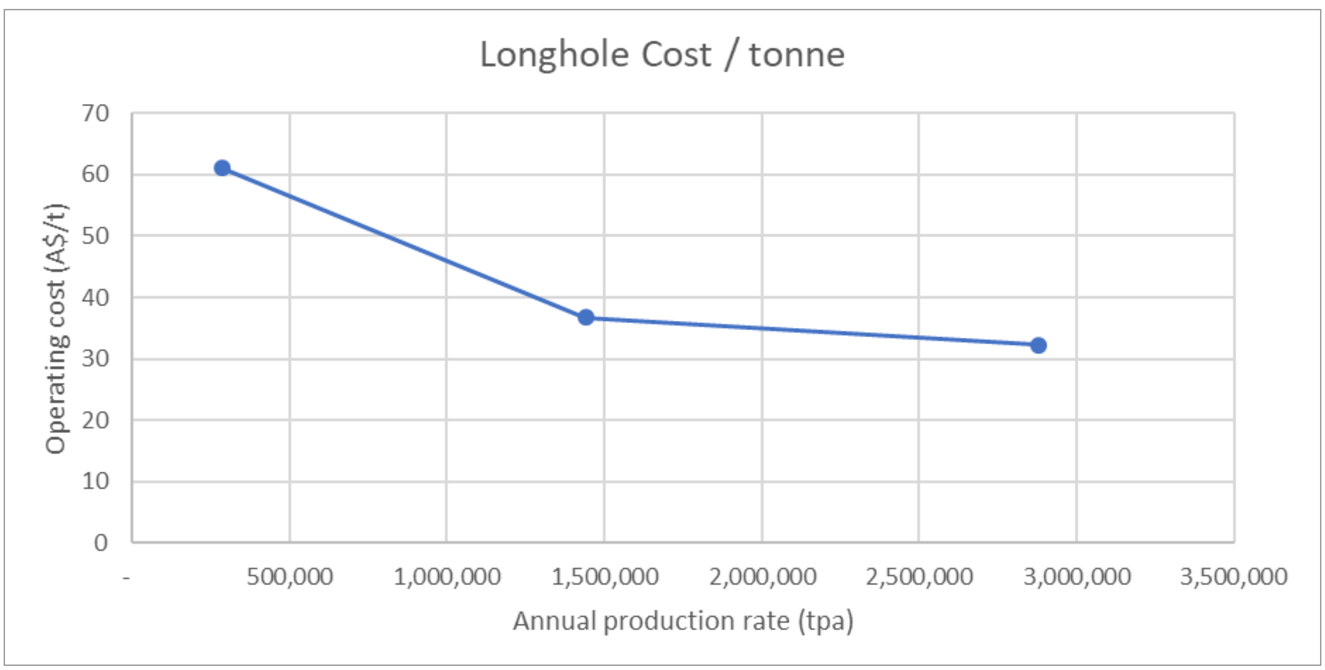
Figure 21.2 Bench mark cost data for Australian R&P operations



Source: AMC

Part of Zone 3B is the only longhole mining area and production is very limited. A representative cost of A$50/t of mineralized material was assumed for longhole mining based on benchmark data provided in Figure 21.3.

Figure 21.3 Benchmark cost data for Australian longhole mining operations



Source: AMC

An underground mining cost of A$50/t of mineralized material has been selected for both R&P and longhole stoping.

21.1.6 Open pit operating costs

As a contractor operation was estimated, equipment purchase costs were translated into leasing costs and included in the contractor mining costs using an interest rate of 5%. A 15% contractor markup cost was added to the estimate to account for the contractor's profit margin. Supplementary information was estimated by AMC based on its internal database for similar types of equipment or activities.

The open pit contractor operating cost estimate covers the following activities:

- Drill and blast.
- Load and haul to waste dumps, process plant and stockpiles.
- Presplit.
- Grade control.
- ROM re-handle.
- Auxiliary operations such as clearing of the open pits and waste dump footprint area, mine haul road construction, maintenance of benches, road and waste dumps, and dewatering.
- Maintenance of the mine fleet.
- Open pit contractor management and supervision.
- Mining and maintenance personnel.
- Consumables including fuel, parts, explosives, etc.
- Equipment ownership costs expressed as a lease cost per operating hour.

Costs associated with operating the office for the technical services team and contractors are included in the open pit mining G&A costs and are estimated at A$8.2M per year.

As presented in Table 21.3 and Table 21.4, the mining cost per tonne of material mined within the pits over the LOM averages A$3.54/t for mineralized material and A$3.08/t for waste.

Table 21.3 Summary of open pit operating costs by year

Open pit cost (A$M)	**(A$/t)**	**Total (A$M)**	**Y-1**	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**	**Y7**	**Y8**	**Y9**	**Y10**
Waste-drill	0.24	26.44	0.76	3.53	3.72	3.76	3.88	3.85	3.82	2.27	0.86	-	-
Waste pre-split	0.14	15.11	0.64	1.51	2.09	2.11	1.90	1.89	1.88	1.33	1.77	-	-
Waste-blast	0.44	49.07	1.42	6.54	6.91	6.97	7.19	7.15	7.09	4.21	1.59	-	-
Waste-load	0.28	31.30	0.91	4.17	4.40	4.45	4.59	4.56	4.52	2.68	1.01	-	-
Waste-haul	0.81	89.87	1.97	7.30	10.22	11.33	13.13	14.04	17.47	10.56	3.86	-	-
Waste-ancillary	0.60	66.85	1.93	8.92	9.41	9.50	9.80	9.74	9.66	5.73	2.17	-	-
Mineralized material-drill	0.23	2.04	0.11	0.28	0.32	0.29	0.18	0.20	0.23	0.30	0.13	-	-
Mineralized material-blast	0.49	4.37	0.24	0.60	0.69	0.62	0.38	0.43	0.49	0.64	0.29	-	-
Mineralized material-load	0.29	2.55	0.14	0.35	0.41	0.36	0.22	0.25	0.28	0.37	0.17	-	-
Mineralized material re-handle	0.30	2.68	0.15	0.37	0.43	0.38	0.23	0.26	0.30	0.39	0.18	-	-
Mineralized material-grade control	0.15	1.34	0.07	0.18	0.21	0.19	0.11	0.13	0.15	0.20	0.09	-	-
Mineralized material haul	0.91	8.06	0.38	0.85	0.99	1.09	0.63	0.79	1.08	1.56	0.68	-	-
Mineralized material-ancillary	0.60	5.35	0.29	0.74	0.85	0.76	0.46	0.52	0.59	0.78	0.35	-	-
OP - mine G&A*	0.53	63.25	4.08	8.16	8.16	8.16	8.16	8.16	8.16	8.16	2.04		-
Sub total OP mining costs (A$M)		**368.29**	**13.09**	**43.52**	**48.81**	**49.97**	**50.85**	**51.96**	**55.72**	**39.18**	**15.18**	**-**	**-**
Stockpile reclaiming cost	0.50	0.73	-	-	-	-	0.20	0.12	0.06	-	0.09	0.15	0.11
Total open pit (A$M)		**369.01**	**13.09**	**43.52**	**48.81**	**49.97**	**51.05**	**52.08**	**55.78**	**39.18**	**15.28**	**0.15**	**0.11**

Note: *OP - mine G&A includes Mine Supervision (Mine Manager, Superintendents), Technical Team (Engineers, Geologist, Surveyors, Safety), Workshop (Boilermakers, Trade assistants, Maintenance Planners, Electricians, clerk), and Administration and Office Expenses (office clerks, expenses, and consulting).

Table 21.4 Summary of open pit operating costs by activity

Activity		Material type	
		Mineralized material	Waste
Drill	A$/t	0.23	0.24
Pre-split	A$/t	0.00	0.14
Blast	A$/t	0.49	0.44
Load	A$/t	0.29	0.28
Re-handle	A$/t	0.30	0.00
Grade control	A$/t	0.15	0.00
Haul	A$/t	0.91	0.81
Ancillary	A$/t	0.60	0.60
Mining G&A (fixed at 18.6M/annum)	A$/t	0.56	0.56
Total mining cost per tonne	**A$/t**	**3.54**	**3.08**
Stockpile reclaiming cost	A$/t	0.50	0.00

The mine will be operating in two 12-hour shifts. Two rosters are planned for the operation; a 14/7 roster for operators and an 8/6 roster for management, supervisors and the technical services team.

The estimate return flight cost per person is A$994.70 with a daily charge A$75 per man-day for accommodation, village management, maintenance, operating the wet mess, meals and cleaning. A summary of the number rooms required is presented in Table 21.5, and camp and flight costs in Table 21.6.

Table 21.5 Summary of number of rooms required

	Roster*	Total	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
UG manning	**14-7**	**156**	-	-	-	-	-	-	-	-	156	156	156
	8-6	**30**	-	-	-	-	-	-	-	-	29	30	30
Open pit manning	**14-7**	**117**	57	93	102	102	108	111	117	93	66	18	18
	8-6	**65**	33	56	59	59	62	65	65	56	10	2	2
Total OP & UG rooms required		**194**	**76**	**125**	**136**	**136**	**144**	**149**	**155**	**125**	**193**	**194**	**194**

Table 21.6 Summary of camp and flight costs by year

Camp & flight costs	Total (A$M)	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Camp	30.64	2.08	3.42	3.72	3.72	3.94	4.08	4.24	3.42	0.91	0.55	0.55
Flight	26.62	1.84	3.05	3.29	3.29	3.47	3.60	3.70	3.05	0.62	0.36	0.36
Total camp and flight*	**57.26**	**3.92**	**6.47**	**7.01**	**7.01**	**7.41**	**7.67**	**7.94**	**6.47**	**1.53**	**0.91**	**0.91**

Note: *UG camp and flight costs already included in the mining cost rate.

Mine closure items discussed in Section 16 comprise the following activities:

- Re-sloping the waste dumps, in-pit dumps, and ROM pad and placement of 30 cm of top soil.
- Pit, stockpile, road access and tailings pits rehabilitation.
- Revegetation.
- Post closure monitoring.

Mining-related closure costs have been estimated at A$14.49M and are incurred at the end of operations in Year 10. Waste dump re-shaping could start earlier in the project life as dozing equipment is available and waste is directed to the in-pit waste dumps. A cost of A$2,500/ha for re-slope, A$3.41/m^3 for top soil cover with 0.30m top soil thickness, A$4,000/ha for revegetation and A$2.20/$m^3$ for waste backfill were estimated by AMC. The summary of mining related closure costs is presented in the Table 21.7.

Table 21.7 Summary of mining related closure costs by activity

Type	Total (A$M)	Re-sloping (A$M)	Top soil (A$M)	Revegetation (A$M)	Waste backfill (A$M)
Waste dump	3.17	0.23	2.10	0.84	0.00
Stockpile	0.09	0.00	0.07	0.03	0.00
Pit tailings	7.96	0.06	0.22	0.09	7.60
Infrastructure	2.99	0.00	1.67	0.65	0.67
Roads	0.27	0.00	0.20	0.08	0.00
Total	**14.49**	**0.30**	**4.25**	**1.68**	**8.27**

21.2 Capital costs estimate

21.2.1 Capital cost contribution

The capital estimate for Pegmont has been developed by the following contributors;

- GR Engineering Services: process plant, plant infrastructure, non-process infrastructure and compilation of overall capital estimate.
- Queensland Rail: Malbon rail siding.
- Wasco (Australia) Pty Ltd: natural gas pipeline.
- Energy Power Systems: power station.
- Commins Contracting: Selwyn Toolebuc Road to site access road development.
- AMC: mine underground development and infrastructure costs, pre-stripping costs.
- VTT: owner's costs.

21.2.2 Foreign exchange rates

The following rates were used in the estimate for conversion to A$.

Table 21.8 Foreign exchange rates

Currency	Exchange rate
US$	0.75
EURO (€)	0.67

21.2.3 Capital cost estimate summary

Capital costs have been summarized by main work break down structure in Table 21.9 and presented as initial and sustaining capital costs in Table 21.10.

The capital cost estimate includes all costs before the commencement of production (initial capital costs) and sustaining capital costs incurred over the LOM.

Table 21.9 Capital cost breakdown

Cost centre		A$M
Open pit mining	Mine preproduction development	13.1
	Earthworks	0.1
	Roads	0.4
	Fuel storage & distribution	1.0
	Mine services workshop / stores / explosives magazine	3.3
	Project management	0.0
	Engineering and drafting	0.3
	Site supervision and management	0.1
Underground development	UG Zone 3A	8.2
	UG Zone 3B	6.2
	UG Bridge	11.4
	Contingency	1.2
Underground infrastructure	UG Zone 3A	2.9
	UG Zone 3B	2.3
	UG Bridge	5.9
Processing	Earthworks	1.0
	Crushing & screening	9.8
	Coarse & fine ore storage & handling	3.3
	Grinding & classification	7.5
	Flotation	11.4
	Concentrate thickening	1.2
	Concentrate filtering	2.7
	Concentrate storage	3.3
	Reagent mixing & distribution	2.0
	Power reticulation - plant	10.0
	Water storage & reticulation	0.9
	Raw water supply	2.4
	Tails (thickening &) disposal	1.2
	Tailings return water	0.2
	Air services supply & reticulation	0.7
	Plant administration buildings & offices	1.1
	Plant workshop / stores	0.8
	Laboratory	1.7
	Plant piping	7.7
	Commissioning	1.0
	Sustaining capital	2.1
Infrastructure	Roads	3.0
	Concentrate transport infrastructure	2.0
	Gas pipeline	5.0
	Power station	12.7
	Permanent camp facilities	11.7
	Communications	1.4
	Mobile equipment	2.4
	Spare parts	1.5
	Sustaining capital	1.2

Cost centre		A$M
Indirects	Rehabilitation	14.5
	Site supervision and management	4.0
	Project management & engineering design	11.4
	Mobilization / demobilization / indirect costs	5.7
	Site construction cranes & equipment	4.6
	Site construction facilities	0.8
	Initial fills	0.6
	Owner's costs - G&A	1.3
	Owner's costs - mining	3.9
Contingency & escalation	Open pit contingency	0.4
	Underground infrastructure contingency	3.9
	Process contingency	6.8
	Infrastructure contingency	3.0
Capital expenditure total		**229.0**

Table 21.10 Total project capital cost summary

	LOM (A$M)	Initial (A$M)	Sustaining (A$M)
Open pit mining	18.3	18.3	-
Underground mining	37.0	-	37.0
Processing	72.0	69.9	2.1
Infrastructure	40.1	39.6	1.2
Indirects	32.3	32.3	-
Closure costs	14.5	-	14.5
Contingency	14.1	10.2	3.9
Total	**229.0**	**170.3**	**58.7**

21.2.4 Site infrastructure and processing capital costs

The Project capital cost estimate has been developed for the Preliminary Economic Assessment is based upon an EPC / M approach for the construction and commissioning of the project facilities. This includes mine plant and infrastructure, the process plant and infrastructure, general mine infrastructure and roads and upgrades to the Malbon rail siding for zinc concentrate loading.

The estimate includes all the necessary costs associated with process engineering, design engineering and drafting, procurement, construction and construction management, commissioning of the operational facilities and their associated infrastructure, mining services establishment, first fills of reagents, consumables and spare parts to design, procure, construct and commission all of the facilities required to establish the Project.

The estimate is based upon preliminary engineering, quantity take-offs, budget price quotations for major equipment and bulk commodity costs achieved recently on similar projects undertaken in the Australian minerals processing industry. Unit rates for installation were benchmarked to those achieved recently on similar projects undertaken in the Australian minerals processing industry.

The estimate pricing was obtained predominantly during the third quarter 2018 (3Q18) in A$. The capital cost estimates presented in this document are deemed to be of a level of accuracy consistent with industry standards for a PEA based on the following:

- Developed engineering quantities from preliminary calculations and Layout drawings.
- Budget quotations obtained for major items and site-based contract works.
- The capital cost estimate was broken down using a conventional Work Breakdown Structure ("WBS") with plant areas (i.e. crushing, grinding and classification, flotation, filtration and concentrate handling, etc.) as sub-categories.
- In addition, the capital cost estimate was broken down into commodity components (i.e. concrete, structural, mechanical, etc.).

The quantity of man hours and therefore costs for engineering design, procurement, construction management and commissioning were all estimated from first principles using in-house data and experience gained from similar projects.

21.2.5 Capital cost estimate summary – site infrastructure (on and off site)

Table 21.11 details a summary by area of the capital cost estimate for the design, construction and commissioning of the site infrastructure facilities.

Table 21.11 Site infrastructure capital cost estimate summary

Area	**Capital cost A$M**
201 Roads	3.0
348 Concentrate Transport Infrastructure	1.9
410 Gas Pipeline	5.0
450 PowerStation	12.7
480 Permanent Camp Facilities	11.6
490 Communications	1.4
570 Mobile Equipment	2.4
603 Spare Parts	1.5
Grand total (excluding GST)	**39.6**

21.2.5.1 Road upgrades

Estimate provided by a local Cloncurry contractor "Commins Contracting" after inspecting the route.

21.2.5.2 Concentrate transport infrastructure – rail loading

Queensland Rail verbally provided typical unit costings, within the required level of accuracy for the Malbon siding upgrade.

21.2.5.3 Gas pipeline

The design, supply, installation, and commissioning of the 16.7 km long 80 nb steel high pressure gas pipeline was quoted by Wasco.

21.2.5.4 Power station

The estimate includes for 5 x 2,500 kVa natural gas fired generators quoted by Energy Power Systems (Caterpillar) in an n+1 configuration.

21.2.5.5 Permanent and temporary camp

A 204-room permanent camp will be constructed with 200 single ensuite rooms and 4 double rooms along with messing and recreation facilities to suit the expected number of personnel on site. Supply and installation costs were factored from a similar recent tender from ATCO for a camp at Mt Isa.

Temporary accommodation until the permanent camp is available will be at the existing Osborne Camp located 35 km away.

21.2.5.6 Communications

The communications area makes allowances for External Coms, Mobile Radio, Borefield Telemetry, WAN & LAN, a Telephone System, Entertainment system for the Village, and a communications shelter based on similar recent Australian projects.

21.2.5.7 Spares

Commissioning spares are included as 1.5% of the mechanical supply costs.

Mechanical capital spares are included in the estimate as 4% of the mechanical supplied costs.

Electrical capital spares are included in the estimate as 4% of the supplied costs excluding the power station.

21.2.6 Capital cost estimate summary – mineral processing

Table 21.12 details a summary by processing area of the capital cost estimate for the design, construction and commissioning of the new processing facilities before the commencement of production excluding contingency.

Table 21.12 Processing capital cost estimate summary

Area	Capital cost A$M
200 Earthworks	1.0
310 Crushing & screening	9.8
320 Coarse & fine ore storage & handling	3.3
330 Grinding & classification	7.4
336 Flotation	11.4
338 Concentrate thickening	1.2
342 Concentrate filtering	2.7
346 Concentrate storage	3.3
360 Reagent mixing & distribution	1.9
370 Power reticulation – plant	10.0
390 Water storage & reticulation	0.9
391 Raw water supply	2.5
400 Tails (thickening &) disposal	1.1
402 Tailings return water	0.2
420 Air services supply & reticulation	0.7
430 Plant administration buildings & offices	1.1
440 Plant workshop / stores	0.8
460 Laboratory	1.7
499 Plant piping	7.7
505 Commissioning	1.0
Grand total (excluding GST)	**69.9**

21.2.6.1 Estimate basis

The capital cost estimate presented in this study relate to capital works required to construct a new processing plant and support infrastructure facilities. Design criteria and flow sheets for the process plant and infrastructure were developed using historical data, metallurgical test work data, and in-house experience.

From the developed processing route, plant equipment selections were made, and plant layouts were developed. Sufficient preliminary engineering design was undertaken to ensure the functionality of the proposed layouts, suitability of equipment specifications, and to enable construction material quantities to be estimated within the desired level of accuracy.

The estimate is prepared on a commodity basis and reported by area. Details of the structure of the estimate are provided by commodity in this section.

Current market pricing for equipment, labour and bulk rates are incorporated into the estimate. The installation rates include all charges necessary to deliver the requirements of the Project.

21.2.6.2 Earthworks

Plant and infrastructure site earthworks quantities were estimated off the plant layout. An overall bill of quantities for the Project was compiled and a costed schedule of rates from similar projects used for the estimate. The basis assumes a single sub-contractor on multiple work fronts performing all the nominated earthworks for the Project (roads, plant, pipeline, and camp works).

21.2.6.3 Concrete

The concrete quantities were calculated for each area from the general arrangement drawings, layout drawings and preliminary designs developed for the Project.

All in market rates from similar projects were used against the calculated bill of quantities for concrete supply and installation. The rates used are inclusive of a batching plant and are reflective of the overall quantity of the scope.

Concrete works durations verified GRES developed man-hours applied within the estimate by applying an industry man hour rate to reverse calculate the material and installation distribution.

21.2.6.4 Structural steelwork and plate work

Quantities were calculated, on an area by area basis, from the general arrangement drawings and layout drawings developed for the Project.

Market rates from recent similar projects against the calculated bill of quantities to give a schedule of rates pricing for all-inclusive supply. The rates utilized for structural steelwork includes heavy, medium, and light members, conveyor gantries, and trestles. The rates utilized for plate work includes small bis-alloy lined bins and chutes, rubber lined chutes, hoppers and various other. Separate rates are used for grid mesh flooring, hand railing, and stair treads.

East coast Australian rates were used due to the total tonnage calculated for the Project that would make offshore supply unviable.

The supply rates include materials supply, shop detailing, fabrication, surface preparation, final painting in the shop, and identification tagging.

21.2.6.5 Equipment

The process design criteria were used to develop the mechanical equipment list that defines the requirements and sizes of all the mechanical equipment, plate work and tank items. Specifications and data sheets were developed for all major equipment.

Written budget quotations from enquiries accompanied by engineering specifications and data sheets were requested from recognized suppliers for all of the following major equipment in the plant:

- Primary crusher
- Secondary crushers
- Ball mill
- Regrind mills
- Flotation cells
- Vibrating screens
- Hydrocyclones
- Tailings and concentrate thickeners
- Concentrate filters
- Sump, slurry and process pumps
- Power station

Costs derived from recent projects completed to beyond PEA study level and preliminary vendor enquiries were used to estimate costs for the following major items of process equipment:

- Accommodation village
- Conveyor components
- Flocculant plant as a package
- Lime storage and mixing system
- Samplers and analysers
- Weightometers
- Compressed air equipment
- Flotation blowers
- Generators
- Weighbridge
- RO water treatment plant
- Waste water treatment plant
- Galvanized tanks
- Fire water package
- Minor generators

21.2.6.6 Piping

The piping estimate was factored against the total plant capital cost estimate and benchmarked against similar recent Australian projects.

Overland piping quantities have been calculated from process requirements and geographical locations of the water sources, gas pipeline locations and pit location for the tailings. Database rates provided by piping suppliers for various specification pipe and valves and fittings were applied to the developed quantities.

21.2.6.7 Electrical and instrumentation

The electrical, instrumentation and control quantities have been compiled from the Project scope, single line diagrams, layouts, equipment list, and load list. The instrument list was developed from typical P&ID's for this style of plant.

Cable and material quantities were estimated based on layout, switch room locations, equipment specific requirements and drive requirements, and schedules produced.

Database pricing has been used for all electrical components (i.e. switch rooms and MCC's).

21.2.6.8 Transport and freight

Major equipment suppliers were requested to provide pricing to transport equipment to site. Where suppliers have not included transport, an allowance based on the estimated size or weight of each item has been allowed.

Sea freight and packing charges have been included to transport all imported materials and equipment from place of manufacture to site. These costs are included in the supply cost of the item.

Transport for fabricated steel items has been allowed based on quantities expected to be loaded per truck and anticipated truck price between selected suppliers and site. Where possible, fabricated steel components will be suitably packed onto skids and containerized to minimize handling costs.

21.2.6.9 Installation labor

Estimates for installation labor was based on estimated man-hours associated with the equipment and fabricated items to be installed in each area of the plant. The estimated hours for installation reflect the labor force productivity for Australian construction sites for the minerals industry and the application of industry standard labor rates for the type of work involved. The rates were developed with due consideration to the rates actually achieved on recent similar minerals processing projects.

Labor crew rates were built-up including an appropriate mix of supervision, skilled and unskilled personnel. Each crew rate included the costs of mandatory meetings and breaks, small tools, statutory labor costs, PPE, and clothing.

The construction labor rates for SMP and E&IC are based on the current GRES site installation rates. The rates developed are relevant to the current industry and the location of the Project.

21.2.6.10 Plant services

The capital cost estimates compiled for the plant services / infrastructure components of the project are based on requirements dictated by the current process plant design / capacity and plant layout. Dimensions and details are provided in the mechanical equipment list.

The main scope of work items covered by the capital cost estimate are summarized as follows:

- Process plant office, crib room, ablution, and a central plant control room.
- Laboratory building, office, and equipment.
- Training and first aid facility.
- Workshop, stores sheds, and offices.
- RO potable water treatment plant.

21.2.6.11 Commissioning

Applicable engineering labour rates have been adopted for the commissioning team. The team will consist of engineers from various disciplines, the relevant vendor commissioning representatives and a team of tradesman selected form the construction workforce. A travel allowance has been made to transport the dedicated commissioning team to and from the site.

Certain items of major equipment such as the crushers, ball mill, regrind mills, flotation cells, filters, and the thickeners will require vendor representation on site during final commissioning. A vendor commissioning hire cost and travel allowance have been included in the estimate for this purpose. This expenditure will ensure that equipment warranties are preserved, and any early operating issues are resolved ahead of hand over. The vendor commissioning rates used in the capital estimate were supplied to GRES with the equipment tenders.

21.3 Mining capital costs

Escalation has not been considered in the estimate (no inflation is applied); costs are in 2018 real dollars.

21.3.1 Underground capital cost estimate

Key capital cost areas for the underground mine include access development and related underground infrastructure. The total capital required for the underground mine is A$40.9M. The underground mine will be operated by a contractor and all equipment costs are included in the operating cost.

Capital cost estimates are based on mine physical take-offs from the mine design and either benchmark costs or recent vendor quotes. Development is costed at a unit rate of A$5,000/m. A summary of the development capital cost estimate is provided in Table 21.13.

Table 21.13 Summary of underground capital development costs

Area	Total (A$M)	Year 8 (A$M)	Year 9 (A$M)	Year 10 (A$M)
Zone 3A	8.2	7.0	0.8	0.4
Zone 3B	6.2	5.0	0.6	0.7
Bridge Zone	11.4	10.1	0.8	0.4
Total	**25.9**	**22.2**	**2.2**	**1.4**

The underground infrastructure includes electrical distribution and equipment, dewatering and service water distribution, communications, portals for each Zone, refuge chambers and second egress. Total capital for the underground infrastructure is approximately A$15.0M. A summary of the capital cost for underground infrastructure is provided in Table 21.14. Expenditure for infrastructure will start in Year 7 for all Zones. AMC notes that all the refuge chambers are costed under the Bridge Zone and these will be utilized as needed between the three Zones.

Table 21.14 Summary of underground infrastructure capital costs

Description	Total (A$M)	Bridge Zone (A$M)	Zone 3A (A$M)	Zone 3B (A$M)
Electrical distribution and equipment	5.7	3.2	1.5	0.9
Dewatering and service water	1.7	0.6	0.6	0.5
Communications	0.4	0.3	0.03	0.03
Portal	1.9	0.6	0.7	0.7
Refuge chambers	0.9	0.9	-	-
Second egress	0.5	0.2	0.1	0.2
Subtotal	**11.1**	**5.9**	**2.9**	**2.3**
Contingency (35%)	3.9	2.0	1.0	0.8
Total	**15.0**	**8.0**	**3.9**	**3.1**

21.3.2 Open pit capital costs estimate

The open pit operations are planned to be executed by a contractor. All mining equipment are assumed to be provided by the open pit contractor; they have been included in the operating costs as a leasing cost. Fixed mining infrastructure such as workshops and explosive magazines were estimated by GRES.

Open pit capitalized stripping is currently reflected in the open pit operating cost estimate.

21.3.3 Capital cost estimate summary – project indirects (EPCM and owner costs)

Indirect costs were estimated by GRES with VTT providing the Owner’s costs. The costs were based on quotes from vendors and historical costs from previous projects. The summary indirect costs are shown in Table 21.15.

Table 21.15 Indirect cost summary

Area	Total A$M
502 Site supervision and management	3.9
500 Project management & engineering design	11.4
840 Mobilization / demobilization / indirect costs	5.7
503 Site construction cranes & equipment	4.6
504 Site construction facilities	0.8
602 Initial fills	0.6
Owner’s costs - G&A	1.3
Owner’s costs – mining	3.9
Grand total	**32.3**

21.3.3.1 Cranes and equipment costs

Estimates for cranes and equipment costs are based on estimated hours of utilization for major cranes and equipment items associated with installation in each area of the plant, and the application of industry standard charge-out rates for the various cranes and equipment types involved. The applied rates do not include for fuel; however, this cost has been estimated separately. The charge-out rates were taken from in-house construction equipment recently hired in Queensland.

21.3.3.2 Construction facilities

The estimate includes for EPCM offices for the duration of construction on site. An allowance has been made for the hire, establishment, operation and removal of temporary construction offices, crib rooms and ablutions for the construction workforce. There is included an SMP and E&I subcontractor offices and facilities for the durations of these construction works. Consumables and services are included for the duration of the construction period. The charge-out rates were taken from in-house recent hire on Queensland projects.

21.3.3.3 Construction fuel

Construction fuel has been allowed for the duration of construction at A$1.10 per liter. Construction fuel is included in the earthworks and concrete all in rates.

21.3.4 Mobilization and demobilization

Estimates of the mobilization and demobilization costs for each of the construction contractors have been included in the capital cost estimates. These costs have been based on data sourced from recent projects and make allowance for materials, equipment, and personnel.

21.3.4.1 R&R flights, meals, and accommodation

The existing camp at Osborne shall be used as temporary construction accommodation and messing facilities until the permanent Pegmont accommodation facility is available.

Flights, meals and accommodation have been included for all direct labour and indirect construction personnel for the duration of the construction period. The flights for the EPCM personnel are allowed for at A$1,900 return inclusive the A$995 per return trip on a charter flight Townsville to Osborne. Direct labour sourced out of Queensland includes for A$1,550 return including the A$995 per return trip on a charter flight Townsville to Osborne.

21.3.4.2 Engineering, procurement, and construction management

The EPCM estimate was developed from first principles for the personnel required to perform all activities associated with the Project, for the planned duration of the Project. The involvement of each team member was individually estimated based on the complexity of the task and benchmarked against the EPCM costs incurred on recent similar projects. The design, procurement and management aspects of the EPCM work will be undertaken off site in the offices of the EPCM engineer. The EPCM team will work in collaboration with the owner's team to complete the project.

The capital cost estimate contains an allowance for a site-based construction management team and site supervisors to be engaged for the duration of the site works. Travel, site vehicles, and site accommodation costs for these people have been included in the study estimate.

21.3.4.3 Initial fills

First fills and operating stocks are calculated and costed for the project.

21.3.5 Contingency and escalation

Contingency costs were estimated by AMC for the open pit and GRES for the Infrastructure and Process Plant. The summary contingency costs are shown in Table 21.16.

Table 21.16 Contingency and escalation cost summary

Area	Total A$M
Open pit contingency	0.4
Underground infrastructure contingency	
Process contingency	6.8
Infrastructure contingency	3.0
Grand total	**10.3**

21.3.5.1 Engineers contingency as growth

Engineer's contingency costs and allowances commensurate with the level of design and estimating confidence have been excluded, GRES would normally expect 5 – 10% contingency on the standard plant items, with higher contingency attributable to the less defined infrastructure items (e.g. roads and the raw water borefield). This contingency allowance would not include for changes to the process flow sheet, process plant design, or major equipment selections.

21.3.5.2 Escalation

Escalation is excluded from the estimate.

21.3.6 Capital cost estimate clarifications

The following qualifications apply to the capital cost estimate presented in this Study:

- No allowance has been made for sunk costs incurred by the Principal prior to project implementation.
- No allowance has been made for interest charges or capital financing costs.
- No allowance has been made in the estimate for exchange rate fluctuations with respect to equipment sourced from outside Australia.
- The capital cost estimate has been compiled to represent an EPCMM project execution by a competent and proven engineering and construction organization capable of delivering within the schedule and required budget accuracy in conjunction with the client's owner's team.

- The EPCM capital cost is representative of the effort required to design, manage and execute the scope not deemed to be turn-key scopes of work that will be managed directly by the owner's team therefore requiring minimal input from the EPCM engineering team. A project and construction manager provided by the EPCM contractor for the duration of the works will be a single combined role and will work in conjunction with the client's counterpart.
- All bulk earthworks materials general borrow / fill material, wear course, and select fill material will be sourced at site.
- The main access road to the site is designed to meet the requirements of the project. Internal roads that service the mine services area and the camp are designed for their application. The access to the area identified as the Borefield is via the existing basic access tracks and public roads.
- There is no allowance for construction of a tailings storage facility, as in pit storage will be utilized.
- Concrete supply at the site will be by mobilized batching plant with cement and aggregates sourced from Cloncurry suppliers. Construction water to be provided by the owner for use by the sub-contractors is assumed to be of good quality. Any costs associated with improving the existing construction water source or sourcing good quality water for construction from offsite has not been included.
- The power station for the project will comprise of five (four duty and one spare) synchronized 2,500 kVa gas fired generators.
- Power at the camp is provided from the power station with a self-contained 500 kVa diesel generator as emergency backup.
- The area nominated in the study as the Borefield is located approximately 27 km from the plant site and contains 5 x bores within a 5,000 m radius. The capital included for all surface equipment, diesel generators, telemetry, transfer pumps and tank.
- Drilling and casing of the bores at the area nominated as the Borefield is not included in the estimate.

22 Economic analysis

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

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

22.1 Assumptions

All currency is in A$ unless otherwise stated. The cost estimate was prepared with a base date of year 1, assumed to be 2019 and does not include any escalation beyond this date. For net present value (NPV) estimation, all costs and revenues are discounted at 8% from the base date. The economic model shows the Project under construction for two years, which is considered development and then in production for the balance of the projected cash flows, which is considered operating.

A regular Australian Federal corporate tax rate of 30% is applied. No tax planning has been applied, all historical tax attributes such as any loss carry forwards, recapture, mineral property, exploration costs or net tax basis of capital assets are ignored. Taxes are paid in the year they are incurred. Withholding taxes on repatriation to VTT in Canada are not considered as all after tax profits are assumed to remain in an Australian holding company.

The estimates of capital and operating costs have been developed specifically for this project and are summarized in Section 21 and Section 22 of this report (presented in 2018 dollars). The economic analysis has been run with no inflation (constant dollar basis). No project financing costs are included.

Project revenue is derived from the sale of lead and zinc concentrate into the international market. Details regarding the refining of these concentrates can be found in the market studies presented in Section 19. Allowances for penalty elements in the lead and zinc concentrates have been made.

Metal prices and exchange rate assumptions for the base and spot price cases are discussed in Section 19.

Royalties rates are discussed in Section 4.2. A 1.5% NSR royalty is paid to the vendor and the graduated Queensland state royalties for lead and zinc are paid at a rate of 5% based on the metal prices assumed, less allowable deductions for in state refining.

A discount rate 8.0% was deemed appropriate for the Project. Discount rates applied to projected cash flows also recognize the time value of money as well the risks and variables associated with the project, such as metal price fluctuation, marketability of the commodity, location of the project, stage of development, and experience of the owner. The prevailing “risk free” interest rates are currently quite low and given the Project’s location in the Mount Isa region of Queensland, country risk component is negligible. In selecting an appropriate discount rate the largest component of risk for the project are the confidence levels of both the PEA and the use, in part, of Inferred Mineral Resources.

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

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

Reclamation costs are assumed to be incurred at the end of the mine life.

22.2 Economic analysis

AMC conducted a high-level economic assessment of the proposed open pit and underground operation of the Pegmont Deposit. The project is projected to generate approximately A$201M pre-tax NPV and A$124M post-tax NPV at 8% discount rate, pre-tax IRR of 32% and post-tax IRR of 24%.

Project capital is estimated at A$229M with a payback period of 2.7 years (discounted pre-tax cash flow from start of production in Year 1). Key assumptions and results of the economics are provided in the Table 22.1. The LOM production schedule, average metal grades, recovered metal, and cash flow forecast is shown in Table 22.2.

Table 22.1 Pegmont Deposit – key economic input assumptions and cost summary

	Unit	Value
Total open pit plant feed	kton	8,862
Total underground plant feed	kton	1,739
Total open pit waste moved	kton	110,791
Lead grade	%	5.3
Zinc grade	%	2.2
Silver grade	g/t	8.8
Lead recovery to lead concentrate	%	90
Zinc recovery to zinc concentrate	%	74
Silver recovery to lead concentrate	%	77
Zinc price – base case	US$/lb	$1.09
Lead price – base case	US$/lb	$0.94
Silver price – base case	US$/oz	$16.50
Exchange rate – base case	A$/US$	0.75
Zinc price – spot case	US$/lb	$1.18
Lead price – spot case	US$/lb	$0.91
Silver price – spot case	US$/oz	$15.31
Exchange rate – spot case	A$/US$	0.71
Discount rate	%	8
Lead payable metal	%	95
Zinc payable metal	%	85
Lead minimum deduction	%	3
Zinc minimum deduction	%	8
Payable lead metal	M lbs	1,069
Payable zinc metal	M lbs	317
Payable silver metal	M oz	1.1
Moisture content	%	8
Australian federal corporate tax	%	30
Prestart capital costs	$M	170.3
Sustaining capital costs	A$M	58.7
Total operating costs	A$M	787.7
Open pit mine operating costs	A$M	355.8
Underground mine operating costs	A$M	86.9
Process and tails operating costs	A$M	278.8
General and administrative costs	**A$M**	**66.2**

1. As of 22 January 2019, spot lead, zinc and silver prices are London Metal Exchange cash buyer, and exchange rate is Reserve Bank of Australia official rate.
2. Cash costs include all operating costs, smelter, refining and transportation charges, net of by-product (zinc and silver) revenues.
3. All-in Sustaining Costs (AISC) include total cash costs and all sustaining capital expenditures.

Table 22.2 Summary of economic results

	Unit	Base case	Spot case
Cash cost	$/lb payable lead	0.65	0.60
AISC cost	$/t lb payable lead	0.71	0.66
Gross sales revenue	$M	1,826	1,912
Realization costs (royalties, transport smelting)	$M	398	405
Site operating costs (mining, processing, G&A)	$M	788	788
EBITDA	$M	640	720
Total taxes	$M	123	147
EBDA	$M	517	572
	% lead	73.4	71.3
Revenue split by commodity	% zinc	25.3	27.5
	% silver	1.3	1.2
Pre-tax NPV at 8%	$M	201	249
Pre-tax IRR	%	31	37
Pre-tax payback period	Years	2.7	2.4
LOM cash flows (undiscounted)	$M	288	343
After-tax NPV at 8%	$M	124	158
After-tax IRR	%	24	27
After-tax payback period	**Years**	**3.5**	**3.0**

A total payable metal production of 1,069 Mlbs of lead, 317 Mlbs of zinc, and 1,102 koz of silver are projected to be produced during the mine life. Figure 22.2 shows a breakdown of the payable lead, zinc, and silver produced by year during the mine life.

Figure 22.1 Tonnes milled and head grades by year

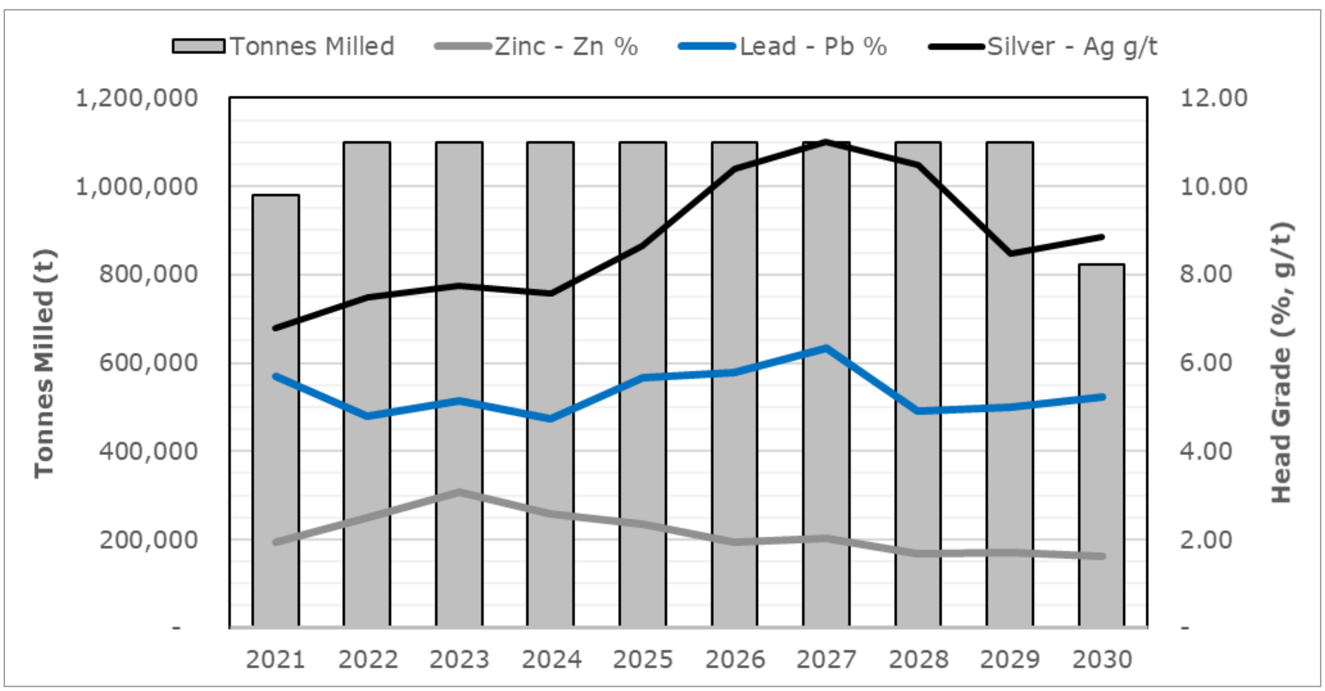


Figure 22.2 Metal production by year

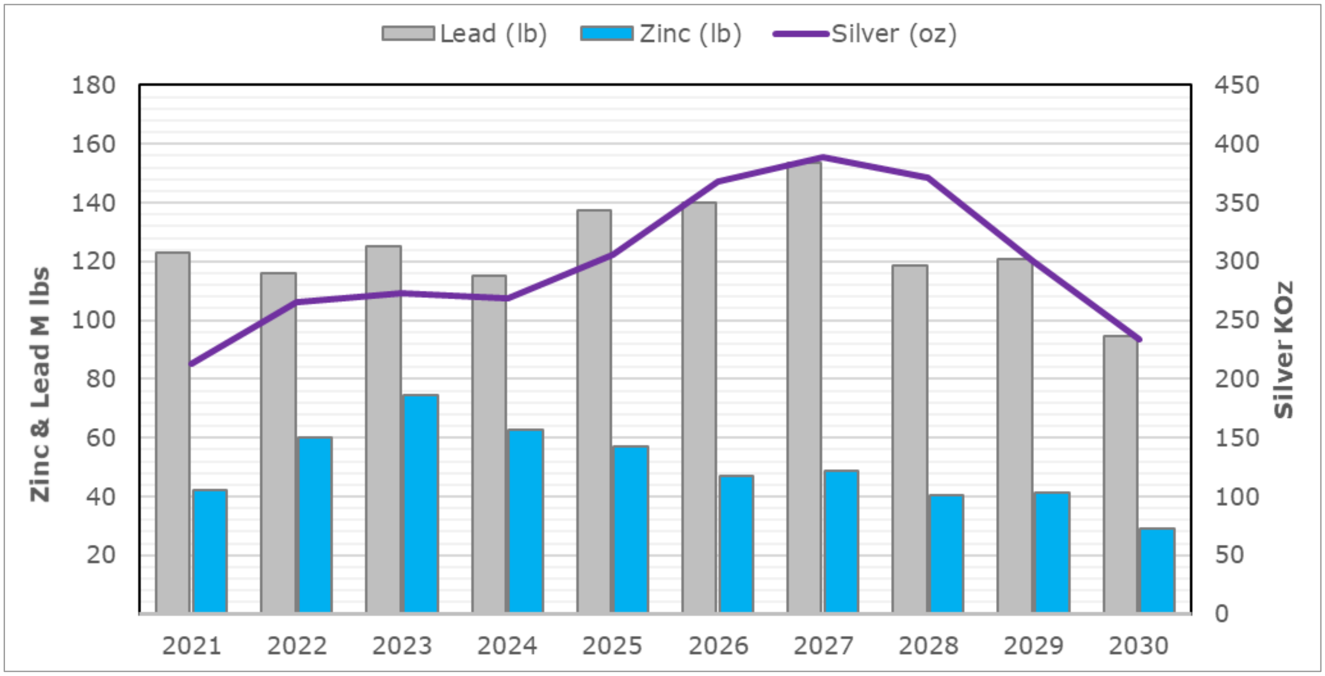
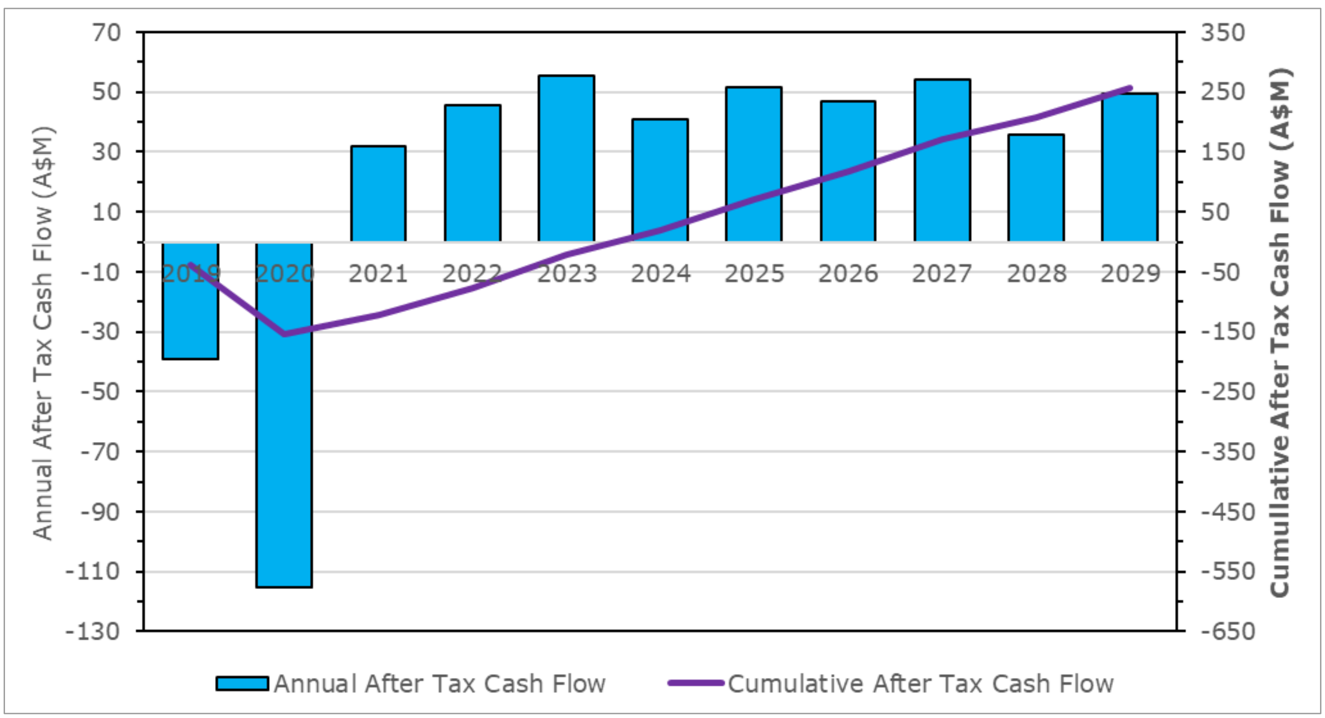


Figure 22.3 Base case annual post-tax cash flow



22.3 Sensitivity analysis

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

The results show that the post-tax NPV is robust and remains positive for the range of sensitivities evaluated.

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

Lead and zinc metal prices often move in tandem, sensitivities to lead and zinc prices are shown in Table 22.4 and Table 22.5 for the base and spot price cases respectively.

Table 22.3 Pegmont project base case economic sensitivity analysis – post tax

Input	**Input factor**						
	85%	**90%**	**95%**	**100%**	**105%**	**110%**	**115%**
Lead Price (US$/lb)	42.7	70.0	97.3	124.4	151.1	177.8	204.5
Zinc Price (US$/lb)	95.3	105.1	114.8	124.4	134.0	143.6	153.2
Capex (LOM)	145.9	138.7	131.6	124.4	117.1	109.7	102.2
Opex (per tonne milled)	174.9	158.1	141.2	124.4	107.4	90.2	73.1
Exchange rate (US$:A$)	**234.7**	**197.9**	**161.2**	**124.4**	**87.1**	**49.5**	**12.0**

Figure 22.4 Pegmont project base case economic sensitivity analysis – post tax

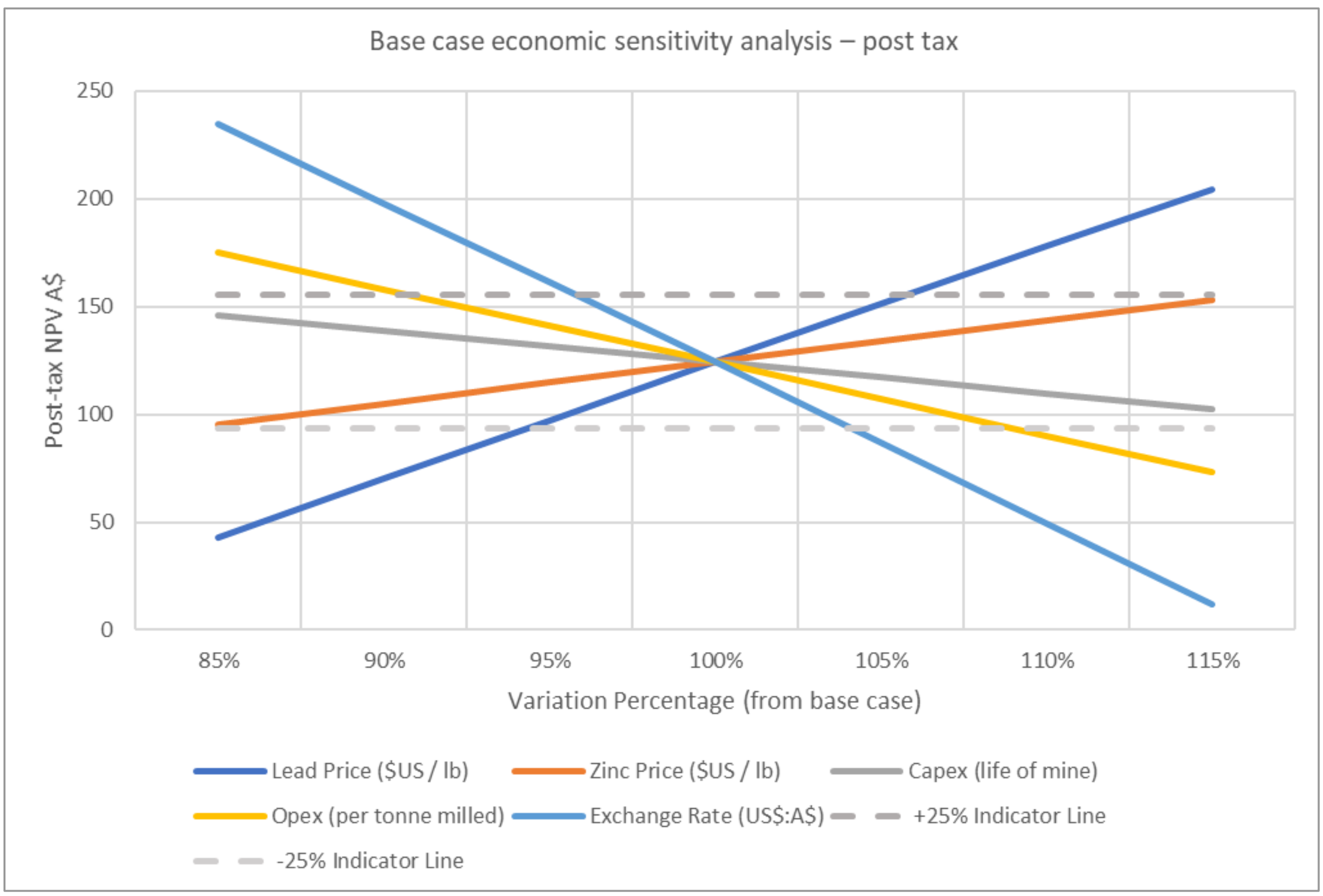


Table 22.4 Lead and zinc price base case economic sensitivity analysis – post tax

Lead price ($/lb)	Zinc Price ($/lb)				
	0.85	**0.95**	**1.09**	**1.15**	**1.25**
0.75	(24,285,557)	(7,357,432)	16,081,656	26,084,078	42,727,372
0.85	32,118,974	48,946,983	72,386,072	82,388,494	99,031,788
0.94	84,294,396	101,122,405	124,422,975	134,257,335	150,620,986
1.05	147,323,749	163,869,008	186,914,267	196,748,627	213,112,278
1.15	204,134,014	220,679,274	243,721,980	253,553,522	269,912,485

Table 22.5 Lead and zinc price spot price case economic sensitivity analysis – post tax

Lead price ($/lb)	Zinc Price ($/lb)				
	0.85	**0.95**	**1.09**	**1.18**	**1.25**
0.75	4,722,861	22,429,891	47,066,199	62,826,575	75,037,530
0.91	101,129,258	118,753,368	142,975,712	158,471,268	170,477,044
0.94	119,253,790	136,663,287	160,885,632	176,381,188	188,386,964
1.05	184,923,496	202,332,993	226,555,337	242,048,538	254,050,874
1.15	244,620,622	262,025,131	286,240,535	301,731,651	313,733,988

23 Adjacent properties

23.1 Introduction

There are a number of mines, projects and resources, owned by third parties within the Cloncurry district within a 3040 km radius of the Pegmont project. While these deposits are not directly related to the structural setting, or mineralization at Pegmont, the proximity of the deposits presents an opportunity to share infrastructure. In all cases there are no immediately adjacent properties or tenement boundaries that influence or abut the Pegmont project.

Note that any Mineral Resource and Ore Reserve figures in the tables in this section are based on the JORC Code December 2012 and are not CIM and NI 43-101 compliant.

23.2 Cannington mine

The Cannington lead zinc silver mine is 100% owned by South32. It is accepted as an example of a Broken Hill Type deposit, that although of a different age to Pegmont displays many similarities, particularly with respect to the base metal zonation (Walters and Bailey, 1998).

Cannington is one of the world's largest producers of silver and lead. Concentrate production commenced in 1997. The underground mine feeds a beneficiation processing facility with a nominal milling capacity of 3.4 Mtpa that produces silver / lead and zinc concentrates. Power is generated by an on-site gas-fired power station operated under contract. The site is accessed via public roads and by the company owned all weather landing strip (Trepell Airport).

In FY2018, (ending 30 June 2018), Cannington processed 2.35 Mt for ore, producing concentrates containing 104,4 kt of lead, 41.3 kt of zinc, and approximately 11.98 million ounces of silver. Production decreased from the previous year by 20%, 21%, and 41% for lead, zinc, and silver respectively due to technical constraints in the underground mine. The information above and Table 23.1 is taken from the South32 Annual Report, 2017, as of 30 June 2018.

Table 23.1 Cannington Mineral Resources and Ore Reserves

Cut-off on NSR basis	Material type	Category	Tonnes (Mt)	Pb (%)	Zn (%)	Ag (g/t)
Mineral Resource						
A$40/t	Open cut sulphide	Measured	27	3.30	2.42	105
		Indicated	3.1	2.69	1.76	68
		Inferred	1.5	2.35	1.31	65
A$100/t	Underground sulphide	Measured	51	5.43	3.39	194
		Indicated	3.4	3.80	2.56	116
		Inferred	0.7	4.12	2.42	85
Ore Reserve						
A$130/t	Underground sulphide	Proved	22	5.47	3.41	184
		Probable	0.3	4.93	2.28	174

23.3 Osborne copper gold operations

Osborne was an important copper-gold deposit hosted in silica altered banded iron stones within albertized quartzites. From early exploration to production significant refinements of the geological model were made (Tullemans and Voulgaris, 1998) and these were the basis VTT re-interpretation of Pegmont.

The Osborne mine commenced operation in 1995 and operated continuously until it was placed on care and maintenance by then owner Barrick Australia Ltd. The mine was purchased by Ivanhoe

Australia in September 2010, and production recommenced in February 2012. It was then 100% acquired through a takeover of duel ASX & TSX listed Inova Resources (previously Ivanhoe Mines) by Shanxi Donghui Coal Coking & Chemicals Group Co. in November 2013, to form Chinova Resources Pty Ltd (Chinova). Chinova sourced ore from Osborne and Kulthor underground mines and from the Starra 276 underground mine, which was hauled 53 km to the Osborne mill. Chinova completed underground operations and milling in February 2016. In September 2015 an expansion of the existing open pit commenced with pre-stripping waste, ore was first mined in February 2016, open pit mining was completed in March 2017. Chinova are currently processing low grade stockpiles, 1.66 Mt were processed during 2017 (Chinova Resources, 2018).

The Osborne copper-gold flotation plant can process 2 Mtpa. Power is supplied via a 19 megawatt natural gas and diesel fuelled power station. The site is accessed via a sealed concentrate haul road from Phosphate Hill and by company owned all-weather landing strip.

Current Mineral Resources at Osborne is not known, Chinova updated the Kulthor Mineral Resource estimate during 2017, and these are shown in Table 23.2.

Table 23.2 Kulthor and Kulthor North 2017 Mineral Resources

Cut-off	Material type	Category	Tonnes (Mt)	Cu (%)	Au (g/t
Mineral Resource					
0.75% CuEq.	Kulthor Underground	Measured	20.98	3.30	2.42
		Indicated	3.1	2.69	1.76
		Inferred	1.5	2.35	1.31
0.75% CuEq.	Kulthor North Open Pit (1260 – 1100 mRL)	Measured	0.52	0.82	1.33
		Indicated	3.93	0.79	1.54
		Inferred	2.04	0.75	1.33
1.20% CuEq.	Kulthor North Underground (1100 – 900 mRL)	Measured	2.58	0.98	0.72
		Indicated	1.60	1.04	0.68
		Inferred	1.45	1.08	0.68

Source: Chinova Resources 2018.

23.4 Selwyn projects - Merlin project

In late 2008, Inova discovered the high-grade Merlin molybdenum and rhenium deposit below the Mount Dore copper deposit. Construction of the Merlin decline began in late 2010, and having advanced some 2,213 m decline, mining was ceased in January 2012. The Merlin maiden resource estimate was undertaken in 2010, a Pre-Feasibility study in 2011 and a Feasibility study in 2012, all documents are available on SEDAR under the Inova profile. Chinoava updated the Merlin Project in 2014 with infill drilling, an updated JORC Mineral Resource and an in-house feasibility study. The project remains on care and maintenance due to low market prices for molybdenum.

Table 23.3 Merlin project Mineral Resource

Cut-off	Material type	Category	Tonnes (Mt)	Mo (%)	Re (ppm)
0.3% Mo	Underground	Measured	0.8	2.3	34
		Indicated	4.2	1.5	26
		Inferred	1.4	1.1	24

Note: Mineral Resources, as of 29 October 2014 from "Merline Molybdenum / Rhenium Project, 2014 presentation by Neal Valk, Chinova Resources.

23.5 Selwyn projects – Mount Dore

Mount Dore is located east of the Starra mine and decommissioned Selwyn processing plant. Mount Dore mineralization is located physically above the Merlin deposit in three distinct copper mineralized zones. Ivanhoe Australia completed a NI 43-101 technical report on the resource estimates on Mount Dore in 2010 and a PEA in 2011. These were further updated in a Pre-Feasibility Study in 2012. In 2016 Chinova were in the process of updating a Feasibility Study on Mt Dore (Chinova Resources, 2017), which included updating the JORC Mineral Resource. Chinova have submitted permitting documents to the Queensland Government for a 2 Mtpa open pit and heap leach project based on a mining inventory of 8.5 Mt grading 0.85% copper (Chinova Resources, 2016).

Table 23.4 Mount Dore deposit Mineral Resources

Cut-off	Material type	Category	Tonnes (Mt)	Cu (%)	Au (ppm)
0.25% Cu	Mount Dore Nth Upper (polymetalic oxide), Sth (copper oxide) & Nth Lower (sulphide).	Measured	1.1	0.69	0.12
		Indicated	66.9	0.58	0.08
		Inferred	42.4	0.49	0.13

Notes: Mineral Resources, as of 15 December 2016 from "Mount Dore Mineral Resource Update Summary, Chinova Resources Ltd and ResEval Pty Ltd.

The information on the Mineral Resource and Mineral Reserve estimates above have not been verified by the QP, and the information is not necessarily indicative of the mineralization on the Property that is the subject of the Report.

24 Other relevant data and information

24.1 Additional drilling

The Mineral Resources reported in Section 13.1 considered drilling information up to 15 April 2018. With drilling on the Property continuing through 2018 and being reported, it is considered appropriate discuss this activity in this Section. From 15 April 2018 to the completion of the program, VTT completed an additional 32 drillholes in three main areas.

Eight drillholes (PVD172, PVRD174-176, and PVD181-184) were completed in Zone 1 transition, in and around Main 1 pit. The purpose of these drillholes was primarily to obtain metallurgical samples for variability test work, which is ongoing at ALS Metallurgy. The geological logging of these holes was complete, and the lithological data was available for the construction of the mineralized envelope wireframes used for the Mineral Resource estimate of 31 July 2018, although the assay data was not used.

At Bridge Zone five drillholes were completed. Two of these (PVRD165 and 166) were drilled to the SE of the Mineral Resource, in an attempt to extend mineralization in that direction. These holes did not intersect the Bridge Zone mineralized horizon, they did however intersect shallow sulphide BHZ mineralization. One drillhole (PVRD164) was drilled up dip of the Bridge Zone Mineral Resource, and intersected 3.3 m of barren host banded ironstone, with classic garnet selvages on both hangingwall and footwall. A drillhole to obtain geotechnical samples (PVRD191 was drilled in the centre of Bridge Zone. One drillhole (PVD171) was completed at BHZ to obtain core in the transition mineralization in an area drilled previously with drilled with RC.

Four drillholes were drilled to better scope the geometry of a fold structure discovered in late 2017, in geotechnical hole PVRD154. To obtain more structural data on the fold PVRD153 and 154 were both extended. Holes PVRD168 and PVD193 were drilled approximately 60 m along strike to the NE of PVRD154. PVRD168 deviated excessively in the RC pre-collar and the hole was abandoned. The target was later re-drilled with PVD193, cored from surface to aid targeting and intersecting the upper fold at a low angle and the steep fold limb. Holes PVRD169 and PVD170 were drilled 160 and 100 m respectively along strike the SW of PVRD154. Again, PVRD169 deviated excessively in the RC pre-collar and the hole was abandoned, the target was re-drilled with PVRD170 which successfully intersected the upper, flat laying limb of the new fold structure.

In the same area, a series of four holes were drilled to better define the geometry of the Zones 2 – 3 Z Fold structure. On one section, PVD192 intersected the lower flat limb, PVD193 first intersected 15 m of the top fold sub-parallel to layering in the low-grade halo before intersecting the steeply dipping limb also sub-parallel to layering and PVD194 successfully intersected the top flat limb of the Z Fold. PVRD190 deviated excessively and intersected an area modelled as being attenuated. PVD195 intersected the steep limb of the Z fold.

A total of four exploration RC drillholes were drilled to test for extensions to mineralization in three separate areas; none of these were successful. However, VTT would like to extend two of them as they believe they were ended prematurely without reaching target depth.

Significant assay results from the post Mineral Resource drillholes are presented in Table 24.1.

Table 24.1 Drill results post 15 April 2018

Hole name	Dip / Azimuth	From (m)	To (m)	Interval (m)	True thickness* (m)	Vertical depth below surface (m)	Grade**			
							Pb+Zn %	Pb %	Zn %	Ag g/t
BHZ										
PVD171 (transition)	-89/255	32.70	38.00	5.30	3.7	32.7	9.05	6.60	2.46	14
PVRD165	-57/207	82.00	85.00	3.00	3.0	66.9	8.25	4.75	3.51	8
PVRD166	-57/205	44.00	46.00	2.00	1.9	36.3	5.46	1.93	3.53	5
Bridge Zone										
PVRD164	-61/210	No significant result								
PVRD191	-52/206	251.10	255.43	4.33	4.2	219.2	11.97	9.73	2.24	34
Zone 1 Transition										
PVD172	-66/132	40.20	46.10	5.90	5.9	36.68	8.02	4.49	3.53	7
including		40.20	43.10	2.90	2.9	36.68	13.87	8.30	5.57	9
PVRD174	-50/140	No significant result								
PVRD175	-62/140	31.63	39.13	7.50	5.2	21.18	7.53	3.61	3.92	7
including		33.36	39.13	5.77	2.1	29.85	8.86	4.40	4.46	7
PVRD176	-87/143	41.32	47.70	6.38	6.3	41.09	9.65	6.19	3.46	11
including		41.32	46.70	5.38	5.3	41.09	10.94	6.98	3.96	13
PVD181	-51/322	41.32	47.30	7.98	5.1	32.05	9.06	5.81	3.25	9
including		41.32	47.30	5.98	3.3	32.05	11.29	7.21	4.09	11
PVD182	-51/319	29.68	43.00	13.32	12.8	23.16	9.15	7.48	1.67	17
including		30.68	40.00	9.32	8.0	23.94	12.50	10.46	2.04	24
PVD183	-84/144	16.80	24.85	8.05	6.4	16.70	7.69	5.05	2.64	10
including		19.45	23.87	4.42	3.8	19.34	12.68	8.80	3.88	16
and		29.65	37.20	7.55	5.5	29.48	10.52	7.55	5.87	10
including		30.40	36.20	5.80	4.2	30.22	16.35	10.17	6.18	17
PVD184	-85/147	25.90	32.27	6.37	5.1	25.76	9.86	7.11	2.75	15
including		25.90	30.45	4.55	3.6	25.76	13.25	9.85	3.40	21
Zones 2 & 3										
PVD153 extension	-74/139	No significant result								
PVD154 extension	-65/063	298.60	300.65	2.05	1.8	276.1	4.58	2.08	2.50	8
PVRD168		Abandoned - no significant result								

PVRD169	-88/151	No significant result								
PVRD170	-81/145	182.00	188.00	6.00		176.1	11.43	8.50	2.93	11
including		183.00	188.00	5.00		177.1	13.09	9.78	3.31	13
PVRD190	-62/309	No significant result								
PVD192	-66/321	201.16	214.91	4.75	4.1	197.3	5.46	2.33	3.13	6
including		201.16	213.16	3.00	2.6	197.3	7.06	2.44	4.62	6
and		227.16	229.16	2.00	1.7	212.7	4.97	1.76	3.21	7
PVD193	-86/150	262.14	267.14	5.00	3.0	261.2	5.72	5.26	0.46	11
including		263.14	266.14	3.00	1.8	262.2	7.48	6.95	0.54	15
and		278.93	280.93	2.00	1.1	277.9	4.51	4.23	0.28	10
and		282.93	284.93	2.00	1.0	281.8	5.66	5.11	0.56	10
PVD194	-51/320	137.43	150.43	13.00	10.7	105.4	9.48	6.74	2.74	11
including		137.43	146.45	9.02	8.3	105.4	12.85	9.05	3.80	14
PVD195	-55/348	205.36	207.40	2.04	<1	167.7	7.65	3.80	3.85	14
Exploration RC holes										
PVR177	-84/151	No significant result								
PVR178	-46/318	No significant result – needs to be deepened								
PVR185	vertical	No significant result								
PVR189	vertical	No significant result								

Notes:
*True thickness is estimated by VTT using structural measurements and three-dimensional geological modelling.
**Drill intersections are summarized intersection lengths >2.0 m, using a combined 1% lead and zinc grade with maximum 1 m internal dilution. Included intervals are at a combined 3% lead and zinc grade with no internal dilution.

24.2 Project implementation plan

It is the intention of VTT to undertake all implementation work on Pegmont project under an EPC or EPCM style of Contract.

VTT will establish a client's representative team to manage the requirements for the delivery of the Project.

The capital cost (CAPEX) estimate was developed on the basis that the Project would be implemented under an Engineer, Procure and Construct (EPC) methodology for the major process plant components, allowing for some form of performance warranty or guarantee to be included.

For the implementation phase, an implementation plan will be developed. The focus of the plan will be to investigate in detail the sequence of the project to define:

- Overall project timing.
- Project critical path(s).
- Requirements for early works and commitments.
- Areas where significant risk of schedule over-runs exist.

Optimizing the delivery methodology, particularly in respect to interfaces between construction work packages, will be investigated early in the implementation phase of the Project.

The implementation plan will include the following:

- Scheduling of the mining and infrastructure works to integrate with the construction of the process plant.
- Scheduling of the process plant works to ensure that plant is commissioned in line with ore availability and development of the mine.
- Definition of the contracting structure to be employed.
- Assessment and definition of resourcing levels required for design and construction.

25 Interpretation and conclusions

Indicative financial results indicate that the Project shows attractive potential and should be progressed to the next stage.

The PEA shows a pre-tax NPV of A$201M, and post-tax NPV of A$124M and associated post-tax IRR of 24% and payback of 3.5 years.

25.1 Observations and conclusions

25.1.1 Geology

The combination of validation of the historic drillhole data and the addition of 69 new drillholes has increased confidence in the nature and extent of the mineralization at Pegmont.

A geological model for Pegmont has developed which is robust and well understood. This is allowing VTT to maximize the use of RC pre-collars down to a depth of no closer than 20 m from the interpreted position of the hangingwall of the principal hangingwall BIF, Lens B, at which point diamond core drilling commences. This will reduce costs but gives a drillcore sample through the mineralization. In addition, structural data is collected from the core.

25.1.2 Mineral Resource estimate

The 2018 Mineral Resource of combined open pit and underground material is 5.8 Mt of Indicated material grading 6.5% Pb, 2.6 % Zn, and 11 g/t Ag and 8.3 Mt of Inferred material grading 5.1% Pb, 2.8 % Zn, and 8 g/t Ag. The cut-off grade applied to the open pit Mineral Resources is 3% Pb+Zn and that applied to the underground is 5% Pb+Zn.

AMC makes the following observations and conclusions:

Since the previous 2017 AMC Mineral Resource estimate:

- Total Indicated tonnes have increased by 156%, while the Inferred tonnes have decreased by 14%.
- The grades of lead, zinc, and silver have increased overall in the Indicated category and decreased in the Inferred category.
- There are additions due to new drilling and discovery of the Bridge Zone.
- The 2018 optimization pit used different economic parameters and different recoveries resulting in a larger open pit shell constraining the Mineral Resources.

The differences between the 2018 and 2017 Mineral Resources are due to the discovery of the Bridge zone, new drilling and different economic parameters applied to the optimization pits.

25.1.3 Processing and metallurgy

The recently completed test work program at ALS focused on metallurgical drillhole samples from Zones 1, 2, and 3 plus some from the Bridge Zone between the main mineralization zones and the BHZ zone tested in 2016. These samples were received in late 2017 and reported in 2018, with key findings summarized below:

- Mineralogical studies have shown that galena is well liberated but sphalerite less so, especially in the transition material. Regrinding of rougher concentrate is indicated.
- This is confirmed by the flotation results where more intensive regrinding of the lead concentrate is required in order to release zinc losses to the zinc circuit.
- Although batch flotation tests exhibited some variability in zinc performance, locked cycle tests which are a better predictor of plant performance, showed more consistency. Overall the

results gave 90 – 93% lead recovery to high grade concentrates ranging from 66% - 73% Pb. As expected, zinc recoveries were lower, being 70 – 75% to concentrates in the 52 – 55% Zn range.

- In benchmark terms these are typical results from complex sulphide deposits and Pegmont can be considered amenable to a conventional differential flotation route.

Pre-concentration by Heavy Media Separation or magnetic separation is not feasible. For the primary mineralization, the flowsheet will be a straightforward Pb / Zn differential flotation circuit using SMBS to depress zinc in the initial lead circuit and copper sulphate to re-activate the zinc in the zinc circuit that follows.

Good quality lead concentrates can be produced and moderate quality zinc concentrates. The main impurity in the zinc concentrate is iron, thought to be occurring as the high Fe marmatite species of sphalerite but yet to be confirmed by microprobe analysis.

The limited amount of comminution work completed on the current metallurgical composites suggest a Bond Ball Mill Work Index of 20.3 kWh/t relating to a hard ore type which would require significant grinding power. Further comminution test work has been recommended to expand on this data and enable selection of the most suitable comminution circuit design.

The progression of the metallurgical test work to include lock cycle flotation tests has demonstrated that the Mineral Resource is well suited to a differential flotation process capable of generating saleable concentrates for both lead and zinc.

Overall lock cycle tests generated positive results with lead concentrate grades of greater than 66% Pb at recoveries in the range of 87 to 92% Pb. Similarly, zinc concentrates grades in the range of 51 to 54% Zn were achievable at moderate recoveries of 70 to 75% Zn.

Zinc liberation following primary grinding to P_{80} of 106µm remains one of the key issues. Part of the recommendations include the investigation of a finer primary grind size to improve sphalerite liberation. Zinc liberation in the sulphide samples averaged 72% liberated sphalerite in the +38 µm fraction, falling to an averaged 56% liberated sphalerite in the +75 µm fraction. The assessment would also determine any effect of overgrinding of the liberated lead sulphide to both the grade and recovery of lead.

Optimization of the flotation regrind stage for a slightly coarser regrind following the finer primary grind should also be examined in the next stage of flotation test work to improve on the zinc sulphide rejection from the lead concentrate. Zinc deportment to the lead concentrate accounts for 9 to 12% of the zinc distribution.

Limited comminution test work was available during the preparation of the PEA which as directed the design towards a three-stage crushing and Ball mill configuration. There is potential for the comminution circuit to be reduced to a single or two stage crushing circuit feeding a SAG mill which would reduce the plant capital cost estimate.

Due to the high lead head grade and the increased probability of overgrinding lead sulphides in the primary mill classification circuit; the opportunity exists to examine the use of flash flotation within the grinding circuit. The next stage of test work should include the inclusion of flash flotation within the grinding circuit. The inclusion has the potential to decrease the size of the conventional lead flotation circuit design.

Equipment price enquiries were obtained from up to three suppliers. During the implementation phase, additional enquiries may be made to potential suppliers, and costs may be negotiated more favourably than during this study phase.

A large number of second-hand transportable accommodation units are currently available throughout Queensland following the completion of the various Coal Seam Gas projects, sourcing these units instead of new units, when required could reduce the capital cost of the accommodation village, with a potential saving of A$20,000 to A$30,000 per unit (A$1M to A$1.5M over the 50 accommodation blocks required).

- Other second-hand equipment may be sourced in the execution phase.
- Potential to optimize the cut and fill requirements by revisiting the village, plant, and mine infrastructure ground contours.
- An early works stage prior to contract award will help bring the critical path forward and shorten the construction schedule.
- Potential to reduce the capital cost of the 16.7 km long gas pipeline to the Project by reducing the pressure of the gas at the branch point off the Cannington lateral and running the gas pipeline in reinforced high-pressure thermoplastic pipes instead of steel. Potential capital cost saving of A$1.5M to A$2M.

If a commercially acceptable agreement could be reached with South32 to load out the Zinc concentrate into tippler type rail wagons at the existing bulk loading facility south east of Cloncurry there is potential to reduce the zinc concentrate transport costs to the refinery in Townsville.

25.1.4 Mining

AMC makes the following conclusions regarding the underground mine potential:

- There are potentially three areas economically viable to mine by underground methods.
- The majority of the underground mineralized material will be mined by mechanized R&P and the remainder will be mined using longhole stoping.
- For R&P AMC has applied a dilution factor of 10% and for longhole stoping 12%, both at zero grade. A mining recovery factor of 86% for R&P and 95% for longhole stoping has been applied to the stopes, 100% recovery is assumed for the mineralized material from development.
- The underground material will be accessed from three separate portals located within the pit. All mineralized material will be hauled up the declines for processing.
- Waste rock from development will be stockpiled on the existing waste dumps on surface or used as backfill in the longhole stopes.
- Underground mine operating costs are assumed to be A$50/t of mineralized material.
- A cut-off value of A$80.5/t of mineralized material is used for the underground mines. This includes processing and G&A costs.
- Total development for the three underground zones including vertical development is 5,547 m.
- The primary mine ventilation fans will be located on surface at the primary exhaust airways of the mine. Fresh air will enter the mine via the declines. Total airflow required is 261 m^3/s.
- All underground mining will be completed by a mining contractor who will be responsible for the supply and maintenance of equipment and supply, management and supervision of the workforce.
- The optimal extraction combination is to mine the underground zones concurrently at the end of the pit mine life with operations starting in Year 8 and mining completed by end of Year 10.
- Approximately 1.7 Mt of mineralized material at an NSR value of A$147.5 is projected to be mined from underground over a three-year period (Year 8 to Year 10).

- Underground mine services include power (peak demand of 1.9 MW), dewatering (10 l/s per Zone), service and potable water, communications, refuge chambers, and emergency egress.
- The offices, plant, surface facilities including workshop and magazine used for open pit operations, will also be used for the underground operations.
- The total capital cost estimate for all three zones is A$40.9M and includes underground access development and infrastructure.

AMC makes the following conclusions with respect to open pit mining:

- Three pits sub-divided into seven pushbacks are potentially mineable from open pit.
- The open pit mine life is approximately ten years, mining 8.9 Mt of mineralized material and 110.8 Mt of waste.
- Mining operations will be undertaken by a mining contractor at a life-of-mine average mining cost of A$3.1/t.
- Mining should focus on reducing waste movement costs by mining bulk waste areas on 10 m benches.
- Mineralized material should be mined selectively on appropriate flitches to minimize mining dilution and ore loss.
- Slope angles used for pit designs are steep and reflect the good condition of the surrounding rock but will require appropriate mining method to ensure that walls can be maintained at the proposed angle.
- Waste dump should be constructed to minimize haulage costs and opportunities to backfill mined out pits should be considered whenever appropriate.

25.1.5 Environmental and social

AMC makes the following conclusions with respect to environmental and social considerations:

- Additional seasonal surveys will have to be conducted but with the exception of one Commonwealth listed marine species (Rainbow Bee-eater), no species or communities of conservation significance have been identified on the Project site, to date
- Future application for mining licenses or variation of the terms of the existing MLs would require a statutory negotiation process to be undertaken at the time.
- Based on the information reviewed as part of this preliminary environmental assessment, there were no known significant environmental issues or sensitive receptors / features identified that could materially influence project viability, nor affect the major design components for future mine development.

25.2 Risks

Standard industry practices, equipment and processes were assumed for the PEA. The authors of the report are not aware of any unusual risks or uncertainties that could affect the reliability or confidence in the PEA results relative to the data and information available and the level of study.

Most mining projects are exposed to risks that may impact the economic outlook to varying degrees. External factors that are largely beyond the control of the project proponents can be difficult to anticipate and mitigate; although, in many instances, some reduction in risk may be achieved by regular reviews and interventions over the life of the project. Certain opportunities that can enhance project economics may also be identified during subsequent studies.

25.2.1 Mining risks

- The design and viability of the project is at risk when the production grade does not match the reserve grade. This will be mitigated during operations by planning grade control activities ahead of the mining activities.
- Other mining risks and control measures identified included a large number of typical mining risks such as heavy vehicle interaction with light vehicles and personnel, haul ramp failure, pit wall failure, excessive ore dilution, and waste dump failure.
- Controls to be implemented to mitigate these risks include development of a traffic management plan, targeted geotechnical drilling and investigation, development of a ground control management plan, surface water study, grade control drilling, and design of facilities to the required codes and standards.

25.2.2 Processing risks

The key processing risks and control measures identified include:

- Process uptime is a key driver for maintaining revenue from the Project while controlling operating costs. As such, uncontrolled downtime from major equipment failure is a significant risk which will be controlled by maintaining adequate spare parts and critical insurance spares in store. Where possible, common equipment (i.e. pumps) will be installed, and adequate surge has been assessed and incorporated in the design.
- The quantity of bore water available to the project is limited. This will be managed by constructing a process water dam at the plant and recovery of water in the process plant.
- Inexperienced or inadequately trained personnel have the potential to cause unplanned plant stoppages or cause the plant to operate sub-optimally. An operator training scheme will be implemented prior to the commencement of production and a complete set of operation manuals will be provided for plant.
- Equipment has been sized (throughput) and selected based on the Vendetta test work data.

25.2.3 Safety and health risks

The safety risks and control measures identified included a large number of typical mineral processing risks and control measures such as working at heights, falling objects, and equipment guarding which will be addressed by utilizing proven industry designs and controls. Specific safety risks associated with the design include:

- Road safety to and from the operation, including the road trains hauling to Mt Isa and Malbon. There is provision for road upgrades and maintenance in the capital and operating estimates. Road safety will be included in the traffic management plan which is to be developed.
- Personnel are not fit for work. A fitness for work system will be developed and in place prior to the commencement of plant operations.
- Lightning strike to personnel and equipment during a storm event. A procedure covering the event of a lightning storm will be in place prior to commencement of construction activities.
- The interaction between heavy vehicles and light vehicles may cause equipment damage and personnel injury. A traffic management plan will be implemented prior to the commencement of mining activities.
- Exposure to dust during operations and maintenance of the plant. Dust monitoring and operating procedures will be developed and implemented. Dust suppression systems will be incorporated into the plant design.

25.2.4 Environmental risks

- The key environmental risks identified relate to bush fires, spillage of hydrocarbons, noise emissions, contaminated water runoff, dust emissions from the operations and the disposal of plant waste streams, including tailings. These will be controlled through using proven designs to contain spillage, seepage, run off, and include dust monitoring and dust suppression systems in the plant design. Additional studies would have to be conducted to ensure that disposal of the tailings within the pits will be undertaken in an environmentally acceptable manner.

25.2.5 Organizational and project risks

Organizational and project risks and the control measures identified include:

- A delay in gaining the necessary corporate and government approvals could have a significant effect on the project start up and cash flow. This risk will be controlled by developing a detailed work plan to secure the necessary approvals or submissions.
- A delay in the construction, commissioning and performance testing of the Project would add additional costs and a delay in fulfilling sale contracts. Mitigation strategies will be developed and implemented. Long lead equipment items will be prepared for purchase early in the execution phase.
- High frequency of Department of Planning and Environment, Resources and Energy reportable incidents during construction. A safety management plan will be developed and in place. Other mitigating controls will include a high supervisor to worker ratio, use of experienced workforce, use of contractors with experience in the work environment.
- Damage to buried services caused by unauthorized excavation works. This will be controlled by developing and implementing an excavation permit process and procedure.
- Serious injury or fatality during construction and handover. An emergency response team, emergency response plan and crisis management team will be developed and implemented prior to the commencement of construction activities and operations.
- Community interface with the mining operation. This will be controlled by ongoing community communication, employment of locals where possible, implementation of health and safety procedures for personnel and vehicles arriving at and leaving the operation.
- Project financing is not achieved. Risk will be mitigated by examining methods to reduce project cost to have a lower cost curve position.

25.2.6 Financial risks

The key financial risks and control measures identified included:

- Insufficient working capital resulting from delays in receipt of product sales and the capital requirement not being budgeted appropriately. This will be controlled by detailing the working capital requirements based on typical product payment terms and durations.
- Product prices reduce to below the cash cost of production, which may be driven by market forces. This may be controlled by arranging long term off-take contracts and identifying pit stages / shells and mining activities based on price.
- Interest rates rise. This will need to be offset by further optimization of the project, investigating opportunities for higher revenue through higher production rates or grades, or lower costs through negotiated contracts and leaner operating costs.
- Obtaining project financing may not be practical if equity pricing reduces. Adopting a more aggressive high grading strategy may mitigate this risk.
- Failure to achieve closure criteria because the rehabilitated mine landscape is not deemed safe, stable, non-polluting, and self-sustaining. To prevent this outcome a Mine Closure Plan

should be developed, reviewed and revised over the life of the mine and implemented on throughout the Project life and on closure. Rehabilitation will be undertaken progressively, where practical.

- The project has allowed to use one of the existing local airstrips (either at Osborne or Mt Dore) to fly in and out personnel for both the construction and operational periods. If a commercially acceptable agreement cannot be reached with the operators of the airstrips, a new airstrip would need to be developed for the project.

26 Recommendations

The following recommendations are drawn from the QP's observations as well as input from and discussions with VTT. The majority of these activities are designed to attain best practices and should be embedded in the way the project is run. Approximate cost for undertaking some of the proposed recommendations is included in the text.

26.1 Geology

26.1.1 Data collection

- Collection of core recovery and RQD data on consistent intervals (for example, the assay interval) rather than on the logged geological intervals currently used. It is difficult to compile meaningful spatial statistics for recovery and RQD across non-uniform intervals.
- Continuation of verification of historical (pre-VTT) data.

26.1.2 QA/QC protocol

- Generation of a matrix matched CRM to better accommodate the ratio of lead to zinc and the silver values seen at Pegmont.
- Continue with the assaying of duplicate pulps at an umpire laboratory as a further check on laboratory accuracy.
- Re-assessment of assay fail criteria to enable earlier detection of problematic results.
- Generation of a crushed and homogenized bulk sample of quartz float material for use as blank material.
- A selection of RC holes should be twinned by drilling proximal (within 5 m) DD holes to verify historic drilling program assays where QA/QC data is absent or indicates some issues of bias or uncertainty.

26.1.3 Modelling

- Incorporate the use of sulphur assays of the mineralized zones to refine the boundary between mineralized zone oxidation states oxide / transition and transition / sulphide. Rebuild these modelled surfaces.
- For the purpose of accurately assigning dilution, build low grade mineralization domains after investigating the selection of the grade boundaries of mineralized domains.

26.1.4 Drilling general

- Continue use of a mix of RC drilling and DD through the target horizon.
- Further drilling should aim to establish the controls on grade distribution in the oxide/transition boundary and improve the interpretation of sub-domains to control the distribution of high-grades.
- Could be combined with collection of samples for metallurgical testing and have laboratory density measurements performed on full core sticks. Also twinning adjacent to any poorer performing historical QA/QC results should be considered.
- Continued collection of geotechnical and hydrogeological data collection in subsequent drilling and core logging activities.
- Drill selected drillholes below the current target horizon as the presence of additional BIFs below the known occurrences should not be discounted. The use of downhole geophysical techniques that potentially could identify off-hole mineralized BIFs should be investigated.

26.1.5 Exploration program

Continue drilling to further evaluate the deposit. Activities are to both infill the current Mineral Resource to increase the level of confidence and upgrade the non-classified portion of the model.

- Undertake drilling to inflll data gaps between the Zone 4 and the two Zone 3 underground panels, approximately 4,400 m has been designed and costed by VTT at approximately A$910,000.
- Further follow up exploration drilling of copper intersection and TEM anomaly on EPM14491, three RC holes drilled in the opposite direction to the previous drilling should be considered, approximately A$80,000.
- Recommendations in earlier sections above in relation to data collection should be undertaken during any further drilling to maximize the value of the data collected from each drillhole, such that it can be used in any subsequent study work.
- Exploration drilling outside of the resource area for extensions and new zones, approximate A$150,000 is required to test immediate targets.
- Expand detailed outcrop geological mapping, A$40,000.

26.2 Processing

The following recommendations are made for consideration as the project progresses to the next phase:

- Continue metallurgical test work programs, including variability test work, estimated at approximately A$100,000.
- Filtration testing on the concentrates to support filter sizing and selection. Cost approximately $5,000.
- Additional comminution tests are recommended including specific tests such as the integrated JK drop weight / SMC test to determine the AG / SAG mill parameters of DWi, Axb, Mia, Mib, Mic, and t_a. This is required to assess the suitability of the Pegmont ore to AG / SAG milling options (approximate cost A$30,000).
- Zone 1 Transitional program to cover the first open pit material. The program would include A Bond Rod Mill Index, a Bond Ball Mill Index and Bond Abrasion index (Ai) for this initial material. Batch rougher and cleaner flotation tests following T1092 program including some rougher kinetic tests. Lock cycle test on the ore zone. Mineralogy of the feed sample ground to a P80 of 106 µm and static settling tests on the lock cycle products (approximate cost A$77,500).
- Bond Crushing Work Index, Bond Rod Mill Work Index, Bond Ball Mill Work Index, and Bond Abrasion Index tests should also be undertaken in conjunction with the JKMRC tests to confirm the current process design and examine the amenability of a primary crush with AG / SAG mill option (approximate cost A$15,000).
- Future flotation test work is also recommended to better define rougher flotation kinetics to investigate use of flash flotation for recovery of fast floating galena, the effect of grind and regrind sizes on both lead and zinc flotation, optimization of zinc grade and recovery and to establish the effects of site water, mild steel media and sample aging (oxidation) (approximate cost A$50,000).
- Next phase of lock cycle flotation tests using site water is recommended (approximate cost A$39,000).
- Filtration test work will need to be undertaken on the produced concentrates to confirm the size of the selected filters. As no work has been done to date GR has used a database of similar regrind size concentrates for both lead and zinc to size the current selection (approximate cost A$10,500).

- Thickening test work on the concentrates is also recommended to assess viscosity which may impact on both pumping and filtration rates (approximate cost A$8,000).

26.3 Project implementation

As the project progresses to the next phase further consideration should be given to the sourcing of used equipment that may reduce the project implementation duration or costs. In particular, the ball mill, concentrate filters, and the accommodation village.

26.4 Open pit mining

AMC recommends that the following aspects are examined in the next study stage:

- AMC recommends that a dilution study is conducted in the next stage of study to ascertain the anticipated mining dilution and ore recovery in combination with the most appropriate mining fleet and associated costs (A$50,000).
- A geotechnical program should be continued to collect additional data for wall angle stability analysis (A$500,000).
- A geotechnical study should be undertaken to understand the offset distance to be left between the open pit and underground workings (crown pillar) and pillars for the tailings pit (A$20,000).
- A dump stability analysis should be undertaken for all waste dumps especially waste dump 2 which is to cover the BHZ tailings pit (A$20,000).
- AMC recommends that quotes from Australian mining contractors are collected to firm up the mining costs estimates for the open pit operations (A$10,000).
- Hydrological and hydrogeological studies should be conducted to better define dewatering requirements for the open pit and underground workings (A$100,000).

26.5 Underground mining

AMC makes the following recommendations for the underground mine:

- Operating costs are based on benchmarking data, AMC recommends that actual quotes be obtained from mining contractors for the next level of study (A$10,000).
- Further work is required to obtain sufficient geotechnical information to support the design criteria assumptions for mining recovery factors and mine design (A$100,000).
- Additional exploration drilling should be carried out with an aim to increasing the confidence in the underground Mineral Resources and increasing the potential throughput for the underground mine in order to fill the mill.

26.6 Environmental and social

AMC recommends that the following studies are conducted for the next study stage:

- Conduct additional baseline studies as per Table 20.2 (A$150,000).
- Review progressive restoration potential (A$50,000).
- Undertake waste rock, ore, and residue characterization (A$100,000).

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28 QP Certificates

Certificate of John Morton Shannon

I, John Morton Shannon, P.Geo., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as the General Manager / Principal Geologist with AMC Mining Consultants (Canada) Ltd. with an office located at Suite 202 – 200 Granville Street, Vancouver, BC, V6C 1S4.

2 This certificate applies to the technical report titled "Pegmont Mineral Resource Update and PEA", with an effective date of 21 January 2019, (the "Technical Report") prepared for Vendetta Mining Corp ("the Issuer").

3 I am a member in good standing of the Engineers and Geoscientists British Columbia (registration #32865) and the Association of Professional Geoscientists of Ontario (registration #0198), and a member of the Canadian Institute of Mining, Metallurgy, and Petroleum.

4 I am a graduate of Trinity College Dublin in Dublin, Ireland (BA Mod Nat. Sci. in Geology in 1971). I have practiced my profession continuously since 1971, and have been involved in mineral exploration and mine geology for over 40 years since my graduation from university. This has involved working in Ireland, Zambia, Canada, and Papua New Guinea. My experience is principally in base metals and precious metals.

5 I have read National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

6 I have not visited the Property.

7 I am responsible for the preparation of Sections 3 – 9, 23, and 24, and parts of 1, 2, 25, 26, and 27 of the Technical Report.

8 I am independent of the Issuer and related companies as described in Section 1.5 of NI 43-101.

9 I have had prior involvement with the property that is the subject of the Technical Report in that I was a qualified person for previous AMC Technical Reports on the Pegmont property.

10 I have read NI 43-101 and the section of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

11 As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 21 January 2019
Signing Date: 13 March 2019

Original Signed and Sealed by

____

John Morton Shannon, P.Geo.

Certificate of Maree Angus

I, Maree Angus, MAIG, of Brisbane, Queensland do hereby certify that:

1 I am currently employed as a Senior Geologist with AMC Consultants Pty Ltd, with an office at Level 21, 179 Turbot Street, Brisbane, Queensland, 4000, Australia.

2 This certificate applies to the technical report titled "Pegmont Mineral Resource Update and PEA", with an effective date of 21 January 2019, (the "Technical Report") prepared for Vendetta Mining Corp ("the Issuer").

3 I am a member in good standing of the Australian Institute of Geoscientists (Membership #6790) and of the Australasian Institute of Mining and Metallurgy (Membership #108282).

4 I am a graduate of James Cook University of North Queensland (BSc Hons) in Economic Geology in 1992). I have worked as a geologist for a total of 21 years since my graduation from university. I have two years' experience in operations, over nine years' experience as a resources industry geologist in project environments, and 12 years consulting experience in gold, copper, and base metals projects, including Mineral Resource estimations and audits and 43-101 report preparation.

5 I have read National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

6 I visited the Property on 16 – 17 May 2017.

7 I am responsible for the preparation of Sections 10 – 12, and parts of Sections 1, 25, 26, and 27 of the Technical Report.

8 I am independent of the Issuer and related companies as described in Section 1.5 of NI 43-101.

9 I have had prior involvement with the property that is the subject of the Technical Report in that I was a qualified person for the previous AMC Technical Report on the Pegmont property.

10 I have read NI 43-101 and the section of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

11 As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 21 January 2019
Signing Date: 13 March 2019

Original Signed and Sealed by

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Maree Angus, MAIG

Certificate of Dinara Nussipakynova

I, Dinara Nussipakynova, P.Geo., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as a Principal Geologist with AMC Mining Consultants (Canada) Ltd. with an office at Suite 202 – 200 Granville Street, Vancouver, BC, V6C 1S4.

2 This certificate applies to the technical report titled "Pegmont Mineral Resource Update and PEA", with an effective date of 21 January 2019, (the "Technical Report") prepared for Vendetta Mining Corp ("the Issuer").

3 I am a member in good standing of the Engineers and Geoscientists British Columbia (registration #37412) and the Association of Professional Geoscientists of Ontario (registration #1298).

4 I am a graduate of Kazakh National Polytechnic University in Almaty, Kazakhstan (BSc and MSc in Geology in 1987). I have practiced my profession continuously since 1987, and have been involved in mineral exploration and mine geology for a total of 30 years since my graduation from university. This has involved working in Kazakhstan, Russia, and Canada. My experience is principally in database management, geological interpretation, and resource estimation, principally in base and precious metals.

5 I have read National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

6 I have not visited the Property.

7 I am responsible for Section 14, and parts of Sections 1, 25, 26, and 27 of the Technical Report.

8 I am independent of the Issuer and related companies as described in Section 1.5 of NI 43-101.

9 I have had prior involvement with the property that is the subject of the Technical Report in that I was a qualified person for previous AMC Technical Reports on the Pegmont property.

10 I have read NI 43-101 and the section of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

11 As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 21 January 2019
Signing Date: 13 March 2019

Original Signed and Sealed by

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Dinara Nussipakynova, P.Geo.

Certificate of Gary Methven

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

1 I am currently employed as the Underground Manager and Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd. with an office located at Suite 202 – 200 Granville Street, Vancouver, BC, V6C 1S4.

2 This certificate applies to the technical report titled "Pegmont Mineral Resource Update and PEA", with an effective date of 21 January 2019, (the "Technical Report") prepared for Vendetta Mining Corp ("the Issuer").

3 I am a registered member in good standing with Engineers and Geoscientists British Columbia (registration #44471), a member of Registered Professional Engineers of Queensland (License #06839), and a member of the Australian Institute of Mining and Metallurgy (Membership #211942).

4 I graduated from University of Witwatersrand in Johannesburg, South Africa with a Bachelor of Science degree in Mining Engineering in 1993. I have relevant experience in precious and base metal deposits, mine infrastructure, design and planning, mine production and financial evaluation, reserve estimation, technical reviews, and technical studies.

5 I have read National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

6 I have not visited the Property.

7 I am responsible for the preparation of parts of Sections 1, 16, 21, 25, 26, and 27 of the Technical Report.

8 I am independent of the Issuer and related companies as described in Section 1.5 of NI 43-101.

9 I have no prior involvement with the Property that is the subject of the Technical Report.

10 I have read NI 43-101 and the section of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

11 As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 21 January 2019
Signing Date: 13 March 2019

Original Signed and Sealed by

____

Gary Methven, P.Eng.

Certificate of Philippe Lebleu

I, Philippe Lebleu, P.Eng., M.Eng., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as the Open Pit Manager and Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd. with an office located at Suite 202 – 200 Granville Street, Vancouver, BC, V6C 1S4.

2 This certificate applies to the technical report titled "Pegmont Mineral Resource Update and PEA", with an effective date of 21 January 2019, (the "Technical Report") prepared for Vendetta Mining Corp ("the Issuer").

3 I am a registered member in good standing with Engineers and Geoscientists British Columbia (registration #41544).

4 I graduated from The Royal School of Mines, Imperial College in London, England with a Masters of Mining Engineering with Rock Mechanics in 1999. I have extensive operational experience in iron ore, copper, and aggregates in Canada, Australia, Brazil, and Malaysia, with mines moving up to 90 Mtpa. My consulting experience spans various commodities and includes project and site work in the Americas, Africa, and arctic environments.

5 I have read National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

6 I visited the Property on 13 – 14 June 2018.

7 I am responsible for the preparation of Sections 15, 19, 20, and 22, and parts of Sections 1, 16, 18, 21, 25, 26, and 27 of the Technical Report.

8 I am independent of the Issuer and related companies as described in Section 1.5 of NI 43-101.

9 I have no prior involvement with the Property that is the subject of the Technical Report.

10 I have read NI 43-101 and the section of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

11 As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 21 January 2019
Signing Date: 13 March 2019

Original Signed and Sealed by

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Philippe Lebleu, P.Eng., M.Eng.

Certificate of Brendan Mulvihill

I, Brendan Mulvihill, MAusIMM (CP Met), RPEQ, residing at 26 Charles Glen Street, Daisy Hill, Queensland, Australia 4127 do hereby certify that:

1 I am a Senior Process Engineer at GR Engineering Services Limited, Building 3, Level 3, Kings Row Office Park 42 McDougall Street, Milton, Queensland, 4064, Australia.

2 This certificate applies to the technical report titled "Pegmont Mineral Resource Update and PEA", with an effective date of 21 January 2019, (the "Technical Report") prepared for Vendetta Mining Corp ("the Issuer").

3 I am a Chartered Professional Member of the Australasian Institute of Mining and Metallurgy (#309808) and Registered Professional Engineer of Queensland under the discipline of Metallurgy (#15189).

4 I graduated from the La Trobe University Bendigo, Australia (B.App.Sc. Metallurgy (Hons.), in 1995. I have practiced my profession for 23 years in the minerals industry and have experience in preliminary and feasibility studies, process optimization, process engineering design, and operation of mineral processing plants. I have been directly involved in feasibility studies and process engineering design of base metal and precious metal extraction plants in Australian and International projects.

5 I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

6 I have not visited the Property.

7 I am the co-author of this report and responsible for Sections 13 and 17, and parts of Sections 1, 2, 18, 21, 25, 26, and 27, and accept professional responsibility for those sections of this Technical Report.

8 I, as a Qualified Person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.

9 I have had no prior involvement with the Property that is the subject of the Technical Report.

10 I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith.

11 That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective Date: 21 January 2019
Signing Date: 13 March 2019

Original Signed and Sealed by

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