NI 43-101 Technical Report on the Paroo Station Lead Carbonate Mine, Wiluna, Western Australia

Effective Date: February 28, 2018

Report Date: April 12, 2018

Report Prepared for

Lead*FX*

Report Prepared by

srk consulting

SRK Consulting (Australasia) Pty Ltd

LFX001

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1 Summary

The purpose of this report is to provide a summary of the technical and economic feasibility of constructing and operating a Hydrometallurgical Facility at the Paroo Station Lead Mine (“Paroo Station” or the “Mine”) in Wiluna, Western Australia that integrates the existing mine and processing facility with a new hydrometallurgical facility to produce lead metal in ingot form pursuant to NI 43-101 and other rules of the Canadian Securities Administrators.

The Mine is 100% owned by Rosslyn Hill Mining Pty Ltd (“RHM”), a 100% owned subsidiary of Lead*FX* Inc. (“Lead*FX*”) listed on the Toronto Stock Exchange (“TSX”).

This report on the Mine is an update to an earlier report dated March 10, 2015 that was compiled by SRK Consulting (Australasia) Pty Ltd (“SRK”) with the assistance and contribution of various appropriately qualified mining industry consultants that are specifically displayed in the relevant sections of the document. SRK has been engaged in the review, editing and finalisation of this update.

On June 20, 2017 The Sentient Group (“Sentient”), Lead*FX*’s major shareholder, and InCoR Technologies Limited and InCoR Energy Materials Limited (together “InCoR”), an unrelated group, entered into an Umbrella Agreement that provided for the transfer of lead hydrometallurgical processing technologies to Lead*FX*. Under the terms of the Umbrella Agreement InCoR agreed to undertake and finance a Definitive Feasibility Study ("DFS") for the development of a Hydrometallurgical Facility at the Mine. Upon the successful completion of the DFS, Lead*FX* would have exclusive rights to use and sub-license InCoR's lead refining technologies worldwide.

Lead*FX* issued two separate common share purchase warrants (the “Warrants”) to InCoR to acquire up to 28,750,000 common shares in the capital of Lead*FX*. The Warrants would be exercisable, for no additional consideration, on and subject to the occurrence of the following triggering events:

- 80% of the Warrants are to be exercisable only on completion of a successful DFS. The DFS will be deemed to be completed and successful if it meets strict criteria including (i) a demonstrable Mine life of no less than ten years and (ii) Mine life of gross operating cash flows minus refinery capital expenditures of no less than US $450 million (”M”).
- The remaining 20% of the Warrants are to be exercisable only upon receipt of definitive environmental approvals by LeadFX to construct a lead refinery at the Mine.

InCoR contracted SNC-Lavalin Australia Pty Ltd (“SNC-Lavalin”) to prepare the DFS and on February 28, 2018 the Company released the results that demonstrated the technical and economic feasibility of constructing and operating a Hydrometallurgical Facility at the Mine and the success criteria for the issue of the first tranche of the Warrants were met.

1.1 Property Description and Ownership

The Mine is located 30 km west of Wiluna, and 2 km directly north of the Wiluna – Meekatharra road in the northeastern goldfields region of Western Australia (Figure 1). The Mine is located in the East Murchison Mineral Field on mining leases M53/501, M53/502, M53/503, M53/504 and M53/1002 and various miscellaneous and exploration licences (the “Mine”). The leases and licences cover in excess of 30,000 hectares (ha), including 2,447 ha of Mining Leases.

The Mine deposits are situated at approximately 26° 31‟ S latitude and 119° 57‟ E longitude.

Five lead carbonate deposits have been discovered to date, namely the Magellan (including Gama), Cano and Pinzon deposits (collectively referred to as “Magellan Hill”). Initial discovery of the Magellan deposit was in 1993, followed by Cano in 2001 and Pinzon in 2004. Two outlying deposits, Drake and Pizarro were discovered approximately 10 – 20 km to the south of Magellan Hill and are not part of the current Mine plan.

The Mine and concentrator plant produces lead carbonate concentrate from deposits of the mineral cerussite (lead carbonate), with subordinate anglesite (lead sulphate) and minor amounts of other more “exotic” lead and lead-manganese oxide and hydroxide minerals and phosphates, which are concentrated in weathered sedimentary rocks in the near-surface environment. The DFS conducted by SNC-Lavalin, was undertaken and completed in February 2018, to explore the technical and financial viability of converting the lead carbonate concentrate into lead metal through a new Hydrometallurgical Facility involving acid leach, electro win and melting processes, at the Mine site. The DFS summary and economic outputs are included in this document.

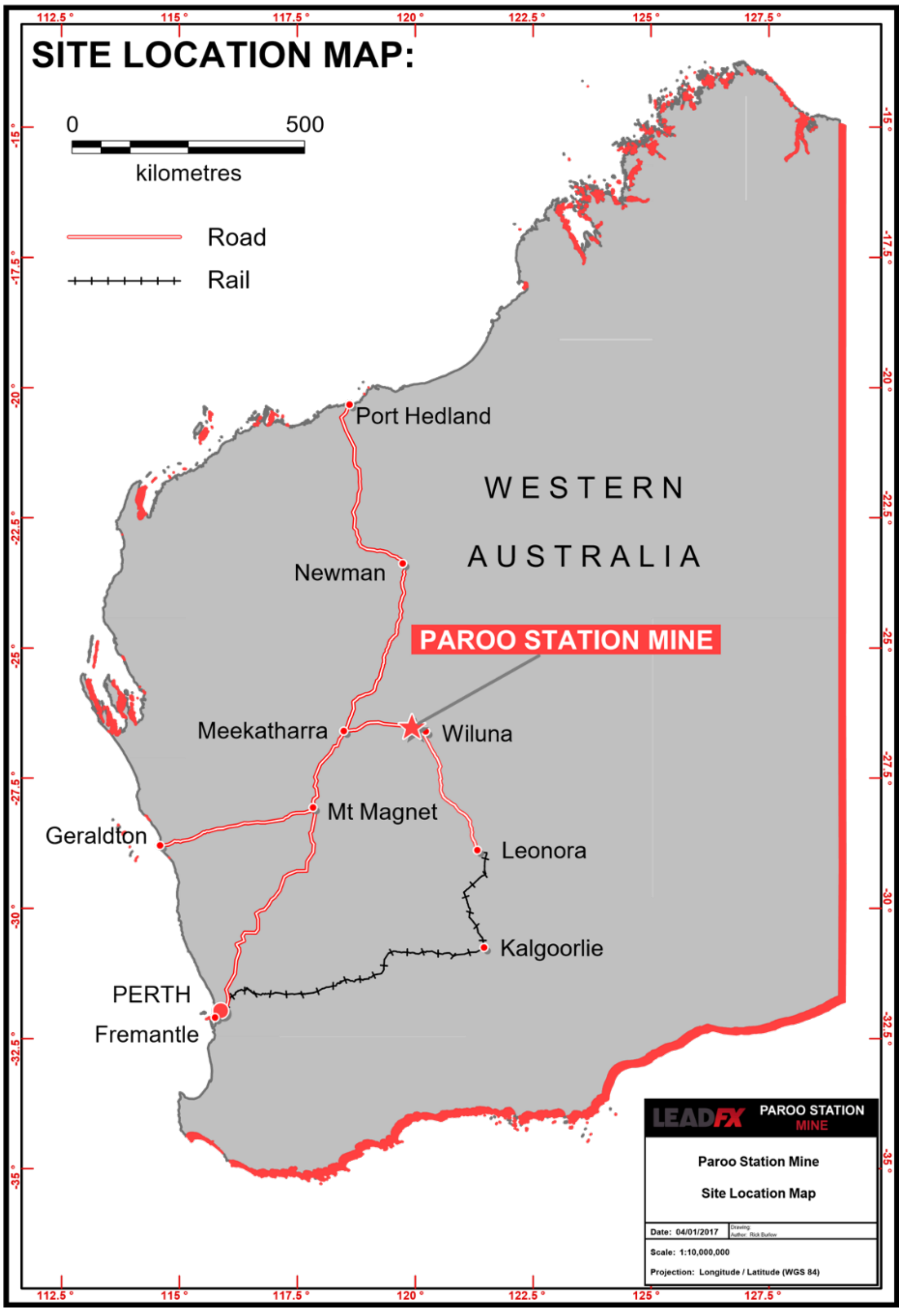


Figure 1: Location map

Source: RHM (2018)

1.2 Geology and Mineralisation

The Magellan Hill lead carbonate deposits are situated in outlier rocks of the Earaheedy Group (Earaheedy Basin) overlying the south-eastern corner of the Palaeoproterozoic Yerrida Basin, at the northern margin of the Archean Yilgarn Craton. The Yerrida Basin is one of several Proterozoic basins between the Pilbara and Yilgarn cratons (McQuitty and Pascoe, 1998).

Pirajno and Burlow (2009) refer to the larger individual Magellan deposit as a large stratabound lead deposit, and describe it as unusual. The mineralisation at Magellan is a sulphide-free supergene lead deposit accompanied by silicification, argillic (illite, kaolinite) and sericitic alteration of the host sandstone and stromatolitic dolomite of the Yelma Formation. The mineralisation is located close to, or at the disconformable contact with the underlying Maraloou Formation (Pirajno et al., 2010).

Sibbel (2009) notes that the Magellan Hill deposits are contained in a mesa outcrop 5 km by 2.5 km, comprising the Yelma formation which hosts the lead mineralisation. The mineralised unit is a quartz clay breccia with fragments of completely silicified carbonate with relict stromatolitic structures, siltstone, and euhedral and colliform banded quartz in a white clay rich matrix (up to 35 m thick).

1.3 Status of Exploration, Development and Operations

1.3.1 Exploration

Renison Goldfields Consolidated (Renison) initiated exploration for base metals in the Mine area in 1990 and carried out geochemical sampling, mapping, and geophysical survey programs in addition to drilling. Anomalous values of between 0.1% and 3.15% Lead ("Pb") from holes drilled at the southwestern edge of Magellan Hill lead to the discovery of the deposit in June 1991.

The majority of exploration work has been drilling and since discovery, non-drilling exploration has comprised extensive soil geochemical surveys; conventional and portable XRF, detailed ground gravity surveys, aerial photography and photogrammetry, and an aerial time-domain electromagnetic ("TDEM") survey.

The Magellan (including Gama), Cano and Pinzon deposits have been explored via a series of drilling campaigns dating back to the early 1990s with the distant Pizarro and Drake deposits also initially drilled in the early 1990s and small numbers of holes were drilled through the following two decades until a significant infill programs were completed in 2010.

All drilling prior to the 2015 drilling campaign have been fully disclosed in the previous Technical Report (SRK, 2015)

In 2015, two drilling programs were completed at the Paroo Station Mine and surrounding exploration prospects. RC drilling was undertaken using face-sampling hammers and auxiliary air compressors to optimise sample recovery.

During June and July 2017, a large-diameter (PQ3) diamond drilling program was conducted at the Magellan and Pinzon lead deposits using PQ3 rod and bit technology (triple tube), with core retrieved using split sets inside 3 m core barrels to maximise recovery of the core. Control drilling techniques were used to limit penetration rates and maximise core recovery.

The diamond drill sites were planned to twin existing RC holes containing known mineralisation across the projected life of mining plan with the aim of collecting annual feed composite samples for variability and metallurgical testing as part of the DFS.

Samples were delivered by road freight trucks from the Mine directly to the laboratory for processing. RC samples were delivered to Intertek Genalysis Laboratories Pty Ltd ("Genalysis") in Perth for sample preparation and subsequent assaying. Diamond core and bulk ore samples were delivered to

Australian Laboratory Services ("ALS") in Perth for processing and test work for the DFS.

No aspect of sample preparation was conducted by an employee, officer, director or associate of the issuer.

RHM has used a combination of duplicates, standards and blanks to ensure suitable quality control of assay testing. RHM's procedures of quality assurance and quality control ("QA/QC") management are consistent with industry practice and are deemed fit for purpose.

1.3.2 History and Ownership

Renison initially discovered the Magellan deposit in June 1991 by stream sediment sampling whilst exploring the region for base metal mineralisation.

The Magellan deposit was acquired from Renison by Westralian Sands Ltd in 1998, subsequently renamed Iluka Resources Limited ("Iluka"). RHM (then known as Magellan Metals Pty Ltd "Magellan Metals"), 100% owned by Polymetals Pty Ltd ("Polymetals") committed to develop a mine and plant pursuant to a farm-in agreement dated January 23, 1997 between Renison and the antecedent company Magellan Metals. This action secured the rights to a 100% interest in the Mine and the Renison Properties (Mining leases M53/501 to M53/504 and various exploration licences) were subsequently transferred to RHM during 2002.

On April 20, 1999, Lead*FX* (formerly Ivernia Inc.) agreed to invest in the Mine by acquiring a direct 15.7% equity interest in RHM from Polymetals.

In September 2000, Lead*FX* acquired a 90% equity interest in Polymetals and acquired the remaining equity ownership in Polymetals in 2003. In May 2003, Lead*FX* entered into a Termination Agreement with Iluka pursuant to which all of Iluka's remaining rights under the 1997 farm-in agreement, including the Renison Royalties, were terminated in consideration of a one-time payment to Iluka of AUD 2.1 M.

During 2003, Lead*FX* and Sentient formed an undertaking whereby Sentient agreed to provide financing to RHM in exchange for a 40% interest in RHM. In 2005, Lead*FX* acquired Sentient then 49% interest in RHM thereby becoming the sole owner of the Mine through its 100% interest in RHM.

A Feasibility Study on the development of the Mine was completed in 2001 by RHM (Magellan Metals, 2001), and updated in 2004 (Watters, 2004).

The Mine was constructed during 2004, commissioned during 2005, and achieved commercial production on October 1, 2005 producing lead concentrate.

The Mine remained operational until April 2007 when it was placed on care-and-maintenance following the initiation of government investigations into bird fatalities in the vicinity of the Port of Esperance where lead concentrates were being exported.

RHM recommenced exporting lead concentrate from existing stockpiles through Fremantle Port in September 2009 and productive mining of lead carbonate commenced in March 2010. In January 2011 the Minister for the Environment ordered RHM to cease transportation to investigate possible loss of lead concentrate from the inside of shipping containers. A source and extent investigation determined a laboratory error. The Department of Water and Environmental Regulation ("DWER") gave permission for RHM to recommence transport in February 2011. RHM went into voluntary temporary closure in April 2011 to conduct a complete end to end review of operations.

Magellan Metals changed its name to Rosslyn Hill Mining Pty Ltd, and changed the name of the Mine to the Paroo Station in November 2012.

Operations resumed in April 2013, with mining, processing and concentrate exporting continuing successfully through to the beginning of 2015.

1.3.3 Operational History

The Mine was constructed during 2004, commissioned during 2005, and achieved commercial production on October 1, 2005. From the start of production until it was placed on care-and-maintenance in April 2007 following the initiation of government investigations into bird fatalities in the vicinity of the Port of Esperance. Approximately 181,100 dry metric t (“dmt”) of lead carbonate concentrate was produced at the Mine, with the majority of concentrate being sold to third party smelters in China.

Production recommenced in late February 2010 and the mine experienced a steady increase of quarterly production through 2010, with 874,000 t of ore processed and 44,100 t of contained lead in concentrate produced for the 12 months ending December 31, 2010.

The operation ceased production again on January 5, 2011, following an order from the Minister for Environment to halt transportation to enable investigation of reports of potential lead egress to the inside of sealed transport containers. No lead egress was found and a thorough investigation resulted in discovery of a laboratory error. The Minister for Environment announced lifting of the order on February 23, 2011, allowing the operation to recommence as soon as practical after that date.

RHM voluntarily placed the Mine onto care-and-maintenance during April 2011 to conduct an ‘end to end’ review of all operational activities. A parallel review under section 46 of the EP Act was undertaken by the OEPA and the review report was published on October 3, 2011. This report resulted in changes to conditions of approval by issue of Environmental Protection (“EP”) Act Ministerial Statement 905 in July 2012. Ministerial Statement 905 superseded all previous conditions and procedures and became the operational regime for the Mine.

On March 28, 2013, RHM announced that it was recommencing processing operations operating under Ministerial Statement 905. Milling and processing operations recommenced on April 5, 2013 and the mining contractor remobilised to site and mining recommenced at the end of April 2013.

The operation experienced a steady increase of quarterly production through 2013 with no significant disruptions to production or transportation. In 2013, 835,900 t of ore was processed, 44,000 t of contained lead in concentrate was produced and 47,700 t of contained lead in concentrate was sold.

The average plant recovery was 74.6% through 2013 with quarterly production records set in the fourth quarter following the introduction of concentrate bagging in 2009 (Lead*FX* 2014).

In 2014, 1,440,000 t of ore was processed at an average head grade of 7.0% Pb producing 80,900 t of contained lead in concentrate, with an overall plant recovery of 79.3%.

In 2015, prior to the Mine entering care and maintenance, 171,200 t (“t”) of ore were processed at an average head grade of approximately 7.4% lead yielding 14,000 tonnes of concentrate containing 9,900 t of contained lead.

Table 1 sets out the production achieved during the three operational periods between 2005 and 2015.

Table 1: Production 2005 - 2015

Production Physicals	Unit	Period			
		2005 - 2007	2010 - 2011	2013 - 2015	Total
Ore milled	t	2,197,400	1,035,000	2,447,100	5,679,500
Head grade	%	7.3	6.8	7.1	7.1
Recovery	%	71.7	73.8	77.6	74.6
Concentrate produced	t	181,100	80,700	202,000	463,800
Con grade	%	64.0	64.8	66.8	65.4
Con Pb content	t	115,900	52,200	134,800	302,900

Source: Lead*FX* – Various historic documents

1.3.4 Current Status

The operation remains in care and maintenance and no ore was processed at site since February 2015. Activities have been conducted to support the DFS into the technical and commercial merits of converting lead carbonate concentrate on site, to lead metal in ingots.

1.4 Mineral Processing and Metallurgical Testing

The existing Mine process plant is a conventional mineral concentrator consisting of crushing, grinding, sulphidisation, flotation and concentrate dewatering.

Throughput of between 1.4 Mtpa and 1.7 Mtpa has been demonstrated through the concentrator. Annualised 2014 throughput exceeded 1.4 Mtpa.

Testwork undertaken in relation to the DFS study for the Hydrometallurgical Facility, which also included some additional flotation testwork, was conducted in 2017, predominantly by ALS in Balcatta, Western Australia. The test program included:

- Proof of concept testwork
- Variability testwork
- Pilot Plant testwork
- Concentrator testwork including pilot plant.

The first three programs are associated specifically with the Hydrometallurgical Facility. A number of other testwork programs were commissioned by RHM, including:

- An Electrowinning testwork program at the University of British Columbia
- A Liquid-Solids separation testwork program with Waterex
- An Acid Recovery testwork program with Eco-Tec.

A significant flotation concentrator testwork program, not part of the Hydrometallurgical Facility DFS, was carried out under the direction of the owners to evaluate flotation performance in order to produce appropriate concentrate samples for testing relating to the Hydrometallurgical Facility.

The key production data from the DFS testwork includes:

- Ore throughput = 2 Mtpa
- Updated Flotation Pb recovery algorithm (following modifications).

Recovery = 73.5% + (1.55 x % Ore Grade)

- 70% Pb concentrate
- Hydrometallurgical Facility Pb recovery = 97.9%
- Lead ingot = 70,000 tpa @ 99.97 % purity

1.5 Mineral Resource Estimate

The Mineral Resource inventory was updated by Optiro Pty Ltd in 2017 and has not changed since it was last reported as at December 31, 2016, (refer to Table 2). The Mineral Resource estimate includes the main Magellan Hill deposits of Magellan (now including Gama), Cano and Pinzon and the outlying Pizarro and Drake satellite deposits (collectively “the Mine deposits”) located approximately 10 km south and 11 km south-west of the existing Mine infrastructure respectively.

The Mineral Resource estimate for the Mine deposits are reported under the Joint Ore Reserves Committee (“JORC”) Code 2012. CIM recognises “use of foreign code” including the JORC Code 2012.

Stockpiles have been tabulated from actual mine production data.

The 2017 Mineral Resource estimate includes all depletion due to mining and processing activities before the Mine was put onto care-and-maintenance during January 2015 due to low commodity prices.

No new drilling or other exploration work has been added to the Mineral Resource estimate, which, depletion aside, remains unchanged from the 2014 estimate presented in the last Technical Report (SRK, 2015).

Data verification has been specifically outlined in Section 12 of this report.

A further satellite deposit, Drake, has been included in the compilation using an existing legacy estimate produced under JORC 2004 by CSA Global Pty Ltd (“CSA”) as no new work has been completed on that deposit since 2007 when the original deposit COG of 2.5% Pb (FinOre, 2005), was lowered to 2.1% Pb (CSA, 2008). This approach is consistent with the application of the 2012 edition of the JORC Code.

Table 2: Mineral Resource estimate as at December 31, 2016

Deposit	Resource Category	Tonnes (Mt)	Grade (Pb%)	Contained Pb Metal (kt)
Magellan (including Gama)	Measured	3.5	4.8	170
	Indicated	13.1	4.6	600
	Total Measured + Indicated	**16.6**	**4.6**	**770**
	Inferred	2.5	4.5	115
Cano	Measured	1.2	4.0	50
	Indicated	1.2	2.9	35
	Total Measured + Indicated	**2.4**	**3.5**	**85**
	Inferred	0.4	3.0	10
Pinzon	Measured	0.1	6.4	5
	Indicated	8.4	4.4	370
	Total Measured + Indicated	**8.5**	**4.4**	**375**
	Inferred	1.7	3.8	65

Deposit	Resource Category	Tonnes (Mt)	Grade (Pb%)	Contained Pb Metal (kt)
Pizarro	Measured	0	0.0	0
	Indicated	3.1	3.6	115
	Total Measured + Indicated	3.1	3.6	115
	Inferred	1.1	3.6	40
Drake	Inferred	2.7	4.1	110
Stockpiles	Measured	1.5	3.0	45
Total	**Measured**	**6.3**	**4.3**	**270**
	Indicated	**25.8**	**4.3**	**1,120**
	Total Measured + Indicated	**32.0**	**4.3**	**1,390**
	Inferred	**8.4**	**4.0**	**340**

Source: Optiro (2017)

1 All Mineral Resources have been reported in accordance with the 2012 JORC Code reporting guidelines and are inclusive of Ore/Mineral Reserves.

2 All Mineral Resources have been reported using a cut-off grade of 2.1% lead and depleted for mining to December 31, 2015. There has been no mining or processing of material during the 2016 calendar year.

3 The stockpiled Mineral Resource is based on mine production data.

4 The Mineral Resource figures are based on the Mineral Resource Report which has been prepared by Mr. Kahan Cervoj (MAusIMM, MAIG), who is an employee of Optiro, and is a “Competent Person” as defined by the 2012 JORC Code. He is a “Qualified Person” (“QP”) for purposes of NI 43-101 and he supervised the preparation of and verified the above Mineral Resource figures prepared by the Company’s consultants, including the underlying sampling, analytical, test and production data. Data was verified by site visits and reviews of the Company’s and consultants’ data.

5 Mr. Cervoj was the Competent Person for the Magellan Hill 2014 Mineral Resource that is the basis for the December 2015 Mineral Resource estimate and participated in a site visit in the last week of July 2014.

6 Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.

7 Table entries are rounded to reflect the precision of the estimate and differences may occur due to this rounding.

1.6 Mineral Reserve Estimate

The Mine has been in commercial operation over several operation phases before being placed in care and maintenance in January 2015 due to low commodity prices. As a result, the “QP” has relied on both historic and more recent production information and financial inputs to support the mine planning and confirm that economic extraction of the Mineral Resource is feasible when integrating the Hydrometallurgical Facility with the existing mining and flotation concentration activities.

The Mine plan was revised to support the Mineral Reserve estimate with updated open pit optimisation incorporating accepted product pricing and current costs and operational parameters. The open pit optimisation underpinned revised mine staging, mine designs and mine production scheduling.

The Mineral Reserve estimate was developed under the 2012 Edition of the JORC Code. CIM recognises “use of foreign codes” including the JORC Code.

Multiple pit optimisation runs were undertaken to establish sensitivity to commodity pricing, mining, processing (including the existing flotation concentration and the Hydrometallurgical Facility) and offsite costs. The results of these ancillary runs establish the key drivers to the development of the mining process suited to the extraction of the deposits’ potentially economic mineralisation.

The base case was selected for parameter variance to explore the sensitivity of output shell size and corresponding financial metrics. The parameters investigated within the sensitivity are:

- Metal pricing (AUD 2490/t Pb to AUD 3735/t Pb range).
- Mining cost (-20% to +20% range with 10% increments).
- Processing cost (-20% to +20% range with 10% increments).

The sensitivity analysis demonstrated that out of the variables selected, the Mineral reserve estimate tonnage is most affected by metal pricing, followed by processing costs with mining costs having the least significant impact.

The Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Mineral Reserves. Inferred Mineral Resources are, by definition, always additional to Mineral Reserves.

The Mineral Reserve is presented in Table 3.

Table 3: Mineral Reserve Statement as at February 28, 2018

Deposit	Reserve Category	Tonnes (Mt)	Grade (Pb%)	Contained Pb Metal (kt)
Cano	Proved	1.4	3.5	47
	Probable	1	2.6	27
	Total	**2.4**	**3.1**	**74**
Magellan	Proved	3.9	4.3	169
	Probable	13.1	4.1	538
	Total	**17**	**4.2**	**707**
Pinzon	Proved	0.1	5.8	5
	Probable	8.8	3.9	343
	Total	**8.9**	**3.9**	**348**
Stockpiles	Proved	2.9	2.4	70
	Probable	-	-	-
	Total	2.9	2.4	70
Total	**Proved**	**8.3**	**3.5**	**291**
	Probable	**22.9**	**4**	**908**
	Total	**31.2**	**3.8**	**1,199**

Source: AMC (2018)

1 *Mineral Reserves are a subset of Measured and Indicated Mineral Resources. The Mineral Reserve Estimate was developed to JORC (2012) standards which are accepted CIM under the use of a Foreign Code. The 2012 JORC Code uses the terms "Ore Reserve" and "Proved" which are equivalents to the terms "Mineral Reserve" and "Proven" respectively, as defined in NI 43-101.*

2 *The Mineral Reserve Estimate was developed by Mr Adrian Jones, a full time employee of AMC*

Consultants Pty Ltd (AMC). Mr Jones is the Competent Person for the 2015 Paroo Station Ore Reserve estimate under the 2012 JORC Code. Mr Jones supervised preparation of the estimate with assistance from specialists in each area of the estimate. Mr Jones is a Member of The Australasian Institute of Mining and Metallurgy. He has sufficient experience relevant to the style of mineralisation, type of deposit under consideration, and in open pit mining activities, to qualify as a Competent Person as defined in the JORC Code. Mr Jones consents to the inclusion of this information in the form and context in which it appears.

3 *Mr Lawrie Gillett FAusIMM of AMC is a QP for the purposes of NI 43-101 and he also supervised and verified the above Mineral Reserve figures prepared by Mr Jones, including the underlying sampling, analytical test and production data.*

4 *Mr Jones participated in a site visit in the second week of March, 2015.*

5 *The pit limits for the open pit were selected through optimisation using the Gemcom Whittle Four-X implementation of the Lerchs-Grossman algorithm. The optimisation considered Measured and Indicated Mineral Resources only. Pit designs followed the optimisation shell outline that developed the largest undiscounted cashflow for the evaluation parameters.*

6 *The process recovery of lead is linked to lead head grade. The following recovery formula was used in the analysis: Flotation Pb Recovery = 73.5% + (1.55 x % Ore Grade), Hydrometallurgical Facility Recovery 98.17%.* [1] *The average overall recovery is 80%.*

7 *Dilution of the resource model and an allowance for ore loss are included in the Ore Reserve estimate, and were introduced through applying a selective mining unit of 6.25 x 6.25 x 2.5m. Within the Ore Reserve pit design, the application of dilution resulted in inclusion of 5.59% dilution and results in an ore loss of 6.43%. Metal pricing of USD 2,250/t Pb plus USD 85/t Pb premium was used in the mine planning.*

8 *The Proved Ore Reserve Estimate is based on Mineral Resources classified as Measured, after consideration of all mining, metallurgical, social, environmental, statutory and financial aspects of the Mine. The Probable Ore Reserve Estimate is based on Mineral Resources classified as Indicated, after consideration of all mining, metallurgical, social, environmental, statutory and financial aspects of the Mine.*

9 *Table entries are rounded to reflect the precision of the estimate and differences may occur due to this rounding.*

1.7 Mining Methods

Ore at the Mine is extracted via drilling and blasting from a series of open pits on Magellan Hill. Excavators are then used to dig and load ore and waste into 85 t haul trucks. Ore is mined concurrently from a number of faces to provide a homogenous blend to the concentrator, and ore is stockpiled and further blended on the run of mine (“ROM”) pad. Grade control is enhanced by testing every blast hole in the orebody and in the near vicinity of the ore body.

Short term planning is based on additional grade control drilling and sampling of blast holes ahead of mining. The waste dumps are located adjacent to the Cano and Magellan pits.

The mining has been carried out to date by a mining contractor. Figure 2 shows the Mine layout.

[1] The Hydrometallurgical Facility recovery used in the determination of the Mineral Reserves Estimate is 98.17%. The final Hydrometallurgical Facility recovery has been estimated at 97.91%. The difference has a life of Mine net impact of 2,478 t of lead metal, being less than 0.3% of the recovered lead metal over the life of Mine.

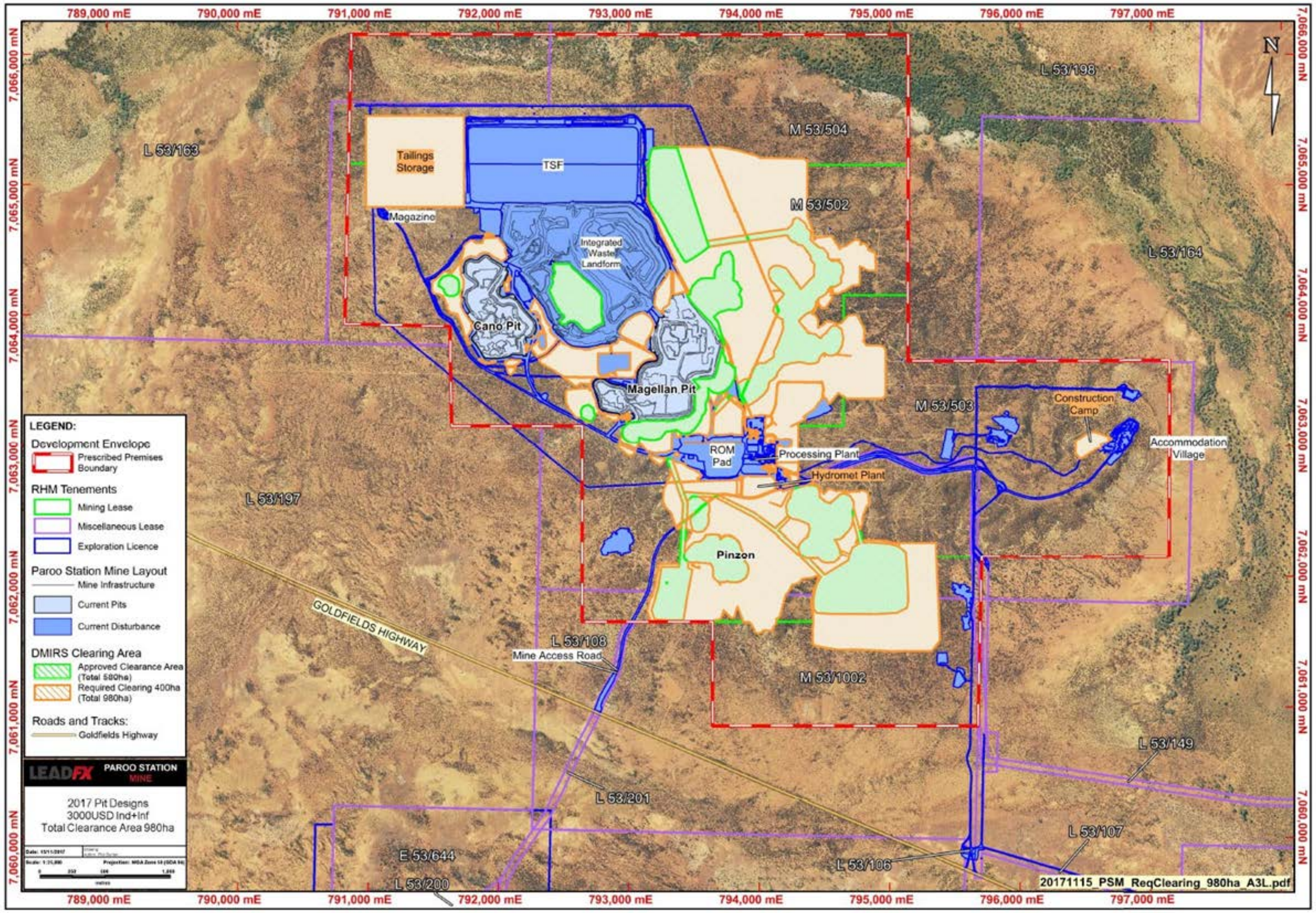


Figure 2: Mine layout

Source: RHM (2018)

1.8 Recovery Methods

All open pit ore production from the Mine was previously processed through the Mine concentrator.

Ore was processed through a conventional flow sheet consisting of the following main steps:

- Crushing and Grinding
- Flotation, Dewatering and Drying

The flow sheet for the concentrator (without planned modifications identified in the DFS) is shown in Figure 3.

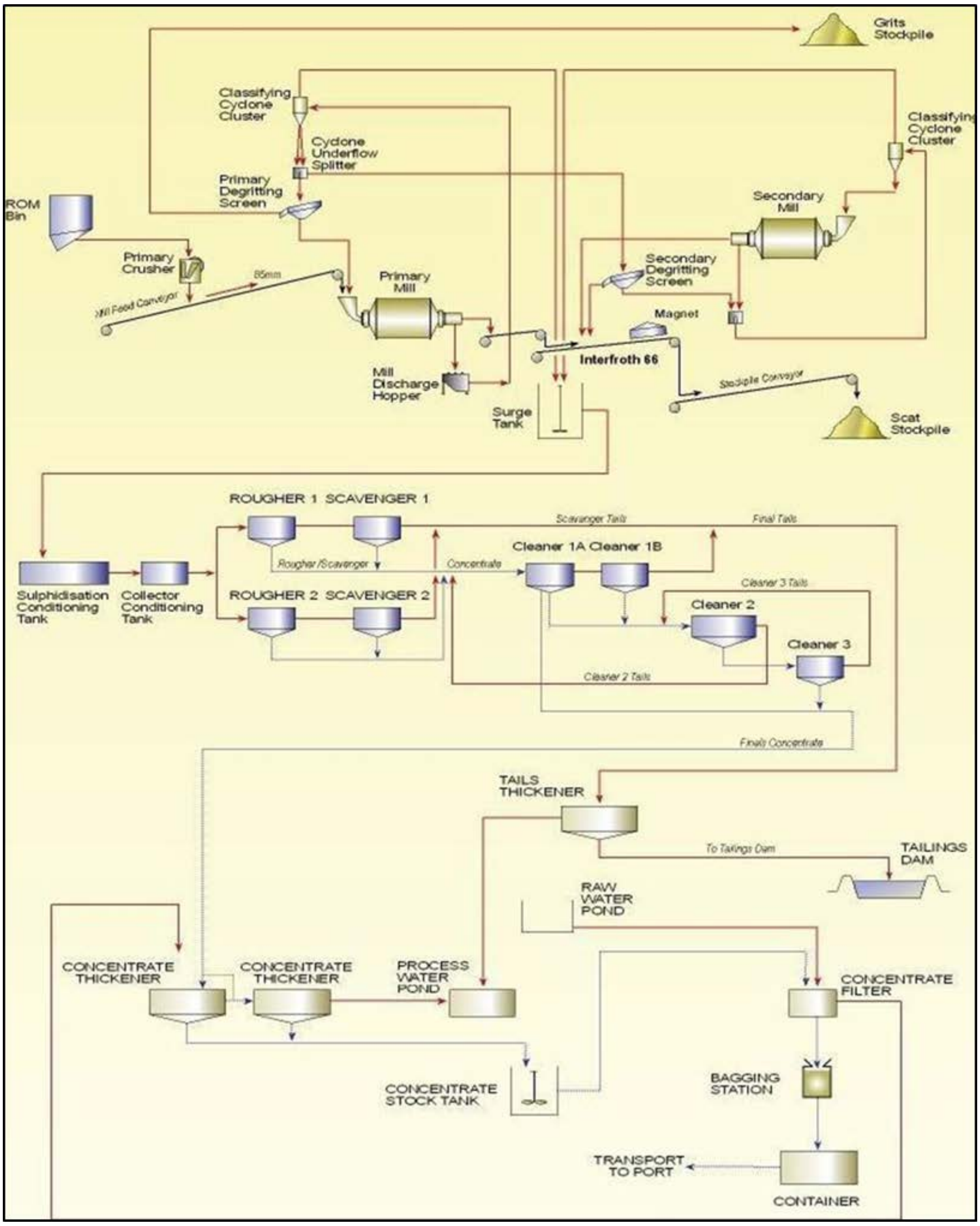


Figure 3: The Mine concentrator processing block flow diagram

Note: The primary mill trommel can discharge directly to a scats bunker and use of the degritting screens depends on ore type.

1.8.1 Recovery Models

In 2017, a metallurgical testwork program comprising batch and pilot plant works was carried out under the direction of RHM to provide the design data required to develop the Hydrometallurgical Facility flowsheet. The owners selected the samples for this work based on the current Mine plan and included samples for years 1 – 10.

An initial flotation development program was executed by RHM to prepare concentrate samples for the Hydrometallurgical Facility testwork. In the process of generating these samples significant potential for improvement in the flotation performance was identified and incorporated into the concentrator design by RHM

Flotation Recovery Model

A Concentrator Metsim Model has been developed by RHM to reflect the proposed modified flotation flowsheet and has been used to evaluate process parameters based on the flotation testwork executed at ALS. The flotation flowsheet modifications included converting the grinding circuit from a Semi Autogenous Mill / Ball Mill (“SAB”) to SAB and Pebble Crusher (“SABC”), a modified flotation circuit using existing equipment and addition of a flotation column to produce a final concentrate in the range 68 – 72% lead grade to feed the Hydrometallurgical Facility. A grade recovery algorithm for the revised flotation flowsheet was developed for use in assessing flotation concentrator performance across the range of anticipated ore feed compositions.

Figure 4 depicts the modified flotation concentrator flowsheet.

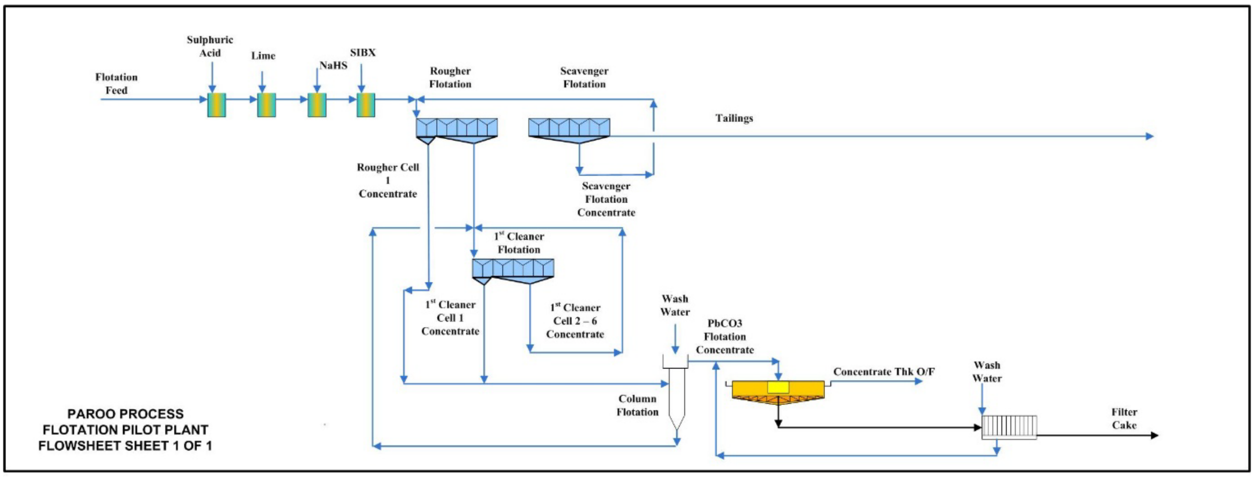


Figure 4: Modified flotation concentrator flowsheet

Source: RHM (2018)

Hydrometallurgical Facility Model

A Metsim Model was developed by SNC-Lavalin for a “base case” mineralogy comprising 78.6% lead carbonate (cerussite) and 9.7% lead sulphate (anglesite) at overall concentrate grade of 70% lead. The other major minerals (based on XRD analysis) are Pyromorphite (1%), Galena (2.6%, Leadhillite (0.6%), Kaolinite (1.0%), Magnetite (3.2%) and Quartz (2.7%). On completion of the variability testwork, additional Metsim Models were run for the assumed minimum (3%) and maximum (15%) anglesite levels, which have been run to assess the impact of the changing concentrate mineralogy on mass balance flows and operating costs. The galena in the concentrate is present as a result of the sulphidisation reaction preceding flotation not as a primary mineral.

The following figures depict the proposed flowsheet for the Hydrometallurgical Facility.

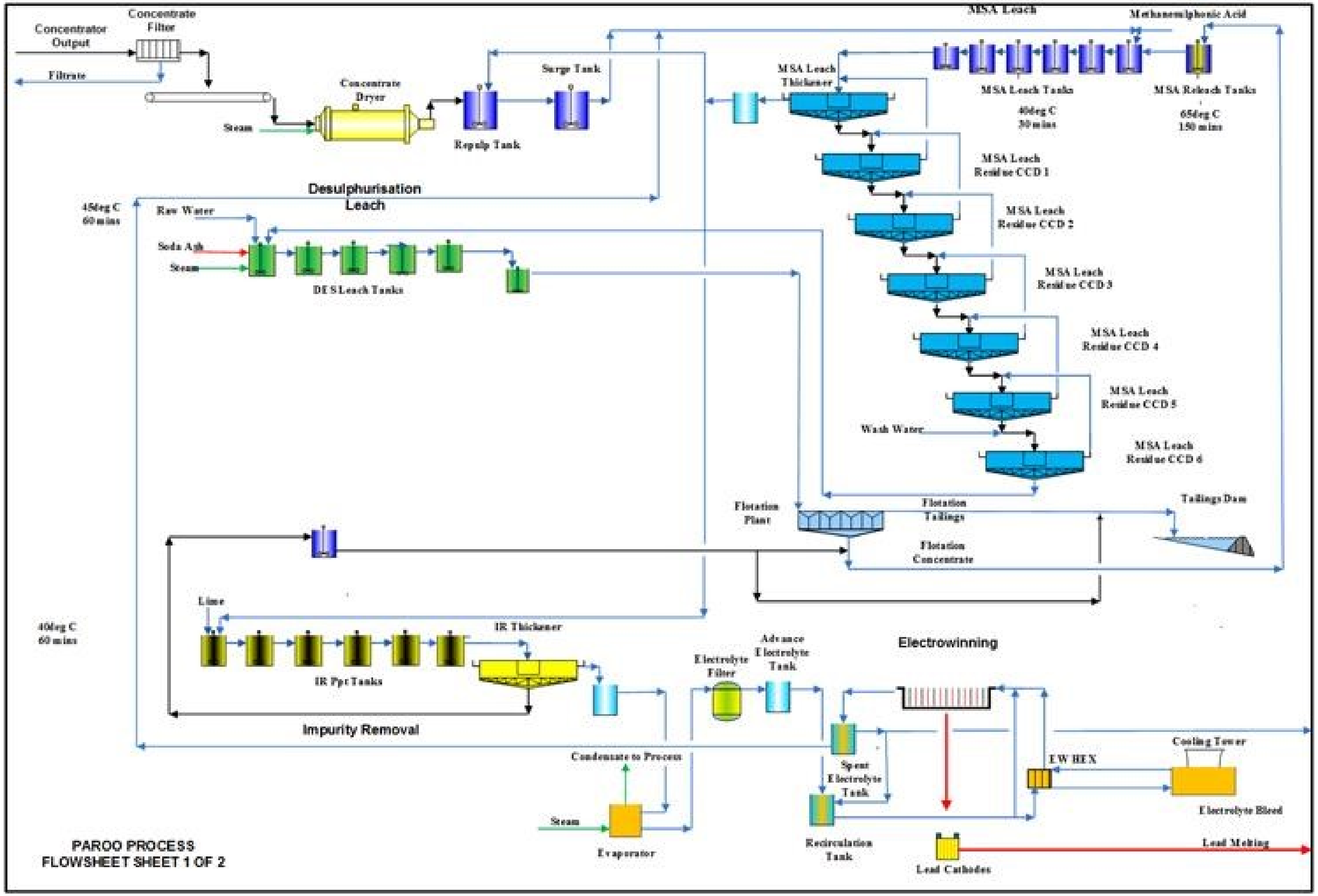


Figure 5: Hydrometallurgical facility flowsheet (1 of 2)

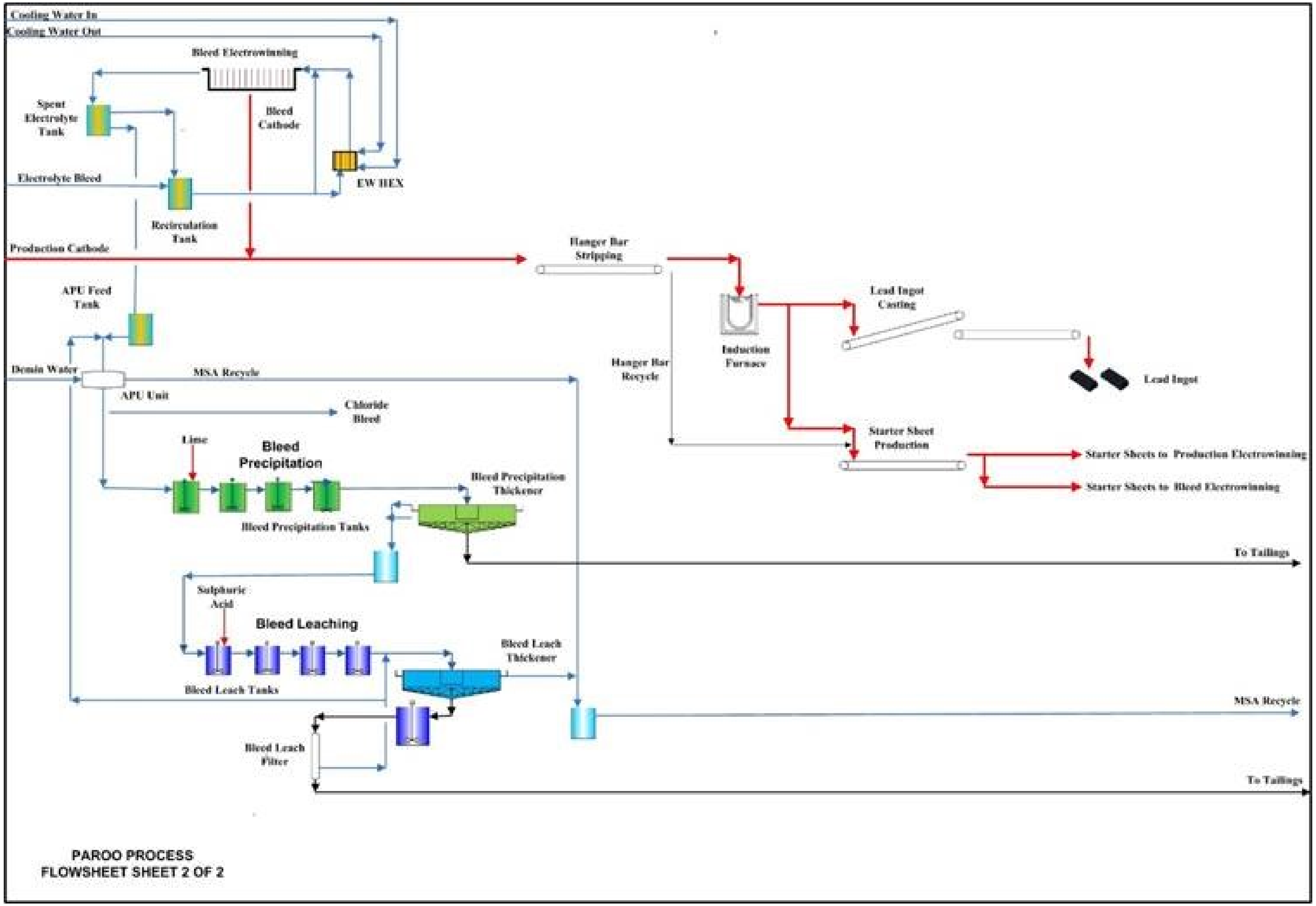


Figure 6: Hydrometallurgical facility flowsheet (2 of 2)

1.9 Mine Infrastructure

As the Mine was operating up till February 2015, all required infrastructure to support the concentrate operation is currently in place and operational. This includes:

- Processing facilities
- Power station and infrastructure
- Tailing Storage Facility (“TSF”) pipeline
- Gas pipeline and infrastructure
- Stores, maintenance and laboratory
- Fuel and chemical storage
- Magazine
- Contractor workshop
- Landfills
- Waste water treatment facilities
- Reverse osmosis (“RO”) plant
- Offices and accommodation village.

The planned infrastructure developments for the site include:

- Hydrometallurgical Facility
- Upgraded power station

1.10 Environmental Studies and Permitting

The Mine operates in accordance with the requirements of State legislation, standards and codes of practice. Specifically, operations are undertaken in accordance with the Mines Safety and Inspection Act 1994, Mines Safety and Inspection Regulations 1995, Mining Act 1978, Mining Regulations 1981, EP Act 1986 and the Environmental Protection Regulations 1987.

The Company regularly collects and reports occupational health, safety and environmental information to the following State Departments:

- EPA – Environmental Protection Authority
- DWER – Department of Water and Environmental Regulation
- DMIRS – Department of Mines Industry Regulation and Safety
- DoH – Department of Health
- DoT – Department of Transport

Currently an application (“Referral Document”) is before the State Environmental Protection Authority (“EPA”), under Part IV of the EP Act, to increase the disturbance footprint by 400 ha, taking the total disturbance footprint to 980 ha, within the development envelope. The Referral Document also includes for an increase of 19 Mt tailings storage capacity, taking the total storage capacity to 35 Mt, to meet the needs of the new forecast mine life. The document also describes the Hydrometallurgical Facility and the proposed new electricity generation plant at site. On April 4, 2018 the EPA determined their Level of Assessment of the Referral Document. The level set was “Referral Information” (the information contained within the Referral Document being sufficient for their purposes), with a request for some additional information to be supplied by RHM.

The construction and operation of the Hydrometallurgical Facility is currently the subject of a Works

Approval with DWER. A prescribed premises licence amendment is expected to follow the commissioning of the Hydrometallurgical Facility.

Updates to the DMIRS approved (2016), Mining Proposal will include the Hydrometallurgical Facility and expanded footprint areas. This will be progressed during 2018. An updated Mine Closure Plan will also be developed to meet DMIRS requirements.

No risks to completion of these approval processes have been identified.

1.11 Capital and Operating Costs

A summary of the independent assessment of capital cost for the Hydrometallurgical Facility together with an estimate for the proposed modifications to the existing concentrator plant (other capex), and other start-up costs for the Mine is presented in Table 4.

Table 4: Capital Cost Summary

Description	Estimate (USD)	Comments
Hydrometallurgical Facility Capex	151,129,061	SNC-Lavalin estimate
Other Capex	7,334,805	RHM estimate
Owner's Costs	11,941,331	RHM estimate
Company Costs to First Production	6,763,676	RHM estimate
Total Estimate (USD)	**177,168,872**	

Source: DFS (2018)

The summarised annual operating costs for the Mine are presented in Table 5. This is based on DFS work and updated historic operating costs from the Mine.

Table 5: Annual Operating Cost Summary (averaged over life of Mine)

Item	Total (USD M)
Royalties	3.30
Utilities (ex Hydrometallurgical Facility)	3.80
Mining	27.71
Concentrator	21.54
Hydrometallurgical Facility	17.19
Supply & Logistics	11.99
Other Fixed Opex	11.91

Source: DFS (2018)

1.12 Economic Analysis

The financial results from the detailed economic model prepared by RHM are estimated on the following basis:

- Real US dollars (i.e. no escalation of revenues and costs for inflation);
- No assumption regarding debt financing, and as such the cashflows presented are ungeared;
- Australian corporate tax rate of 30%;
- AUD: USD exchange rate of 0.75 for the entire Mine life;
- LME cash price of USD 2,250/t Pb for the Mine life based on the long-term price in the Wood Mackenzie global lead long-term outlook, and refined lead premia for lead ingot sales estimated by RHM;

- Hydrometallurgical Facility capital costs and operating costs estimated by SNC-Lavalin;
- Mining costs and concentrate processing operating costs and all other capital and operating costs outside of the Hydrometallurgical Facility estimated by RHM;
- Lead ingot transportation costs estimated by SNC-Lavalin; and
- Mineral reserves estimate and production schedule by AMC Consultants Pty Ltd.
- Only Mineral reserves which are a subset of the Measured and Indicated Mineral Resources have been used in the economic model.

At the February 28, 2018 LME spot lead price of USD 2,575/t Pb for life of Mine, the financial returns are set out in Table 6.

Table 6: Financial Returns (LME lead price USD 2,575/t)

Description	Estimate (USD)	Comments
Total cost to first production	USD 177 M	To start of operations
Payback Period (from start of operations)	3.0 years	From start of operations
Internal Rate of Return	31.3 % pa	From start of construction
After-tax cashflow		
- Undiscounted cashflow	USD 838 M	From start of operations
- GPV (8%pa real discount rate) *	USD 470 M	From start of construction
- NPV (8%pa real discount rate) ^	USD 303 M	From start of construction
- GPV Value (8%pa real discount rate) *	USD 5,271 M	From start of operations
- NPV (8%pa real discount rate) ^	USD 350 M	From start of operations

Further sensitivities have been applied at + 20% of CAPEX, OPEX, LME Pb Price, discount rate and exchange rates with Figure 7 showing the results. The LME lead price shows the largest single impact to the financial results followed by exchange rates[2] and OPEX respectively. For completeness, a correlated LME lead price and exchanges rates sensitivity was also produced.

[2] Equal proportionate depreciation / appreciation of USD to both AUD and EUR for the Exchange rates sensitivity has been used.

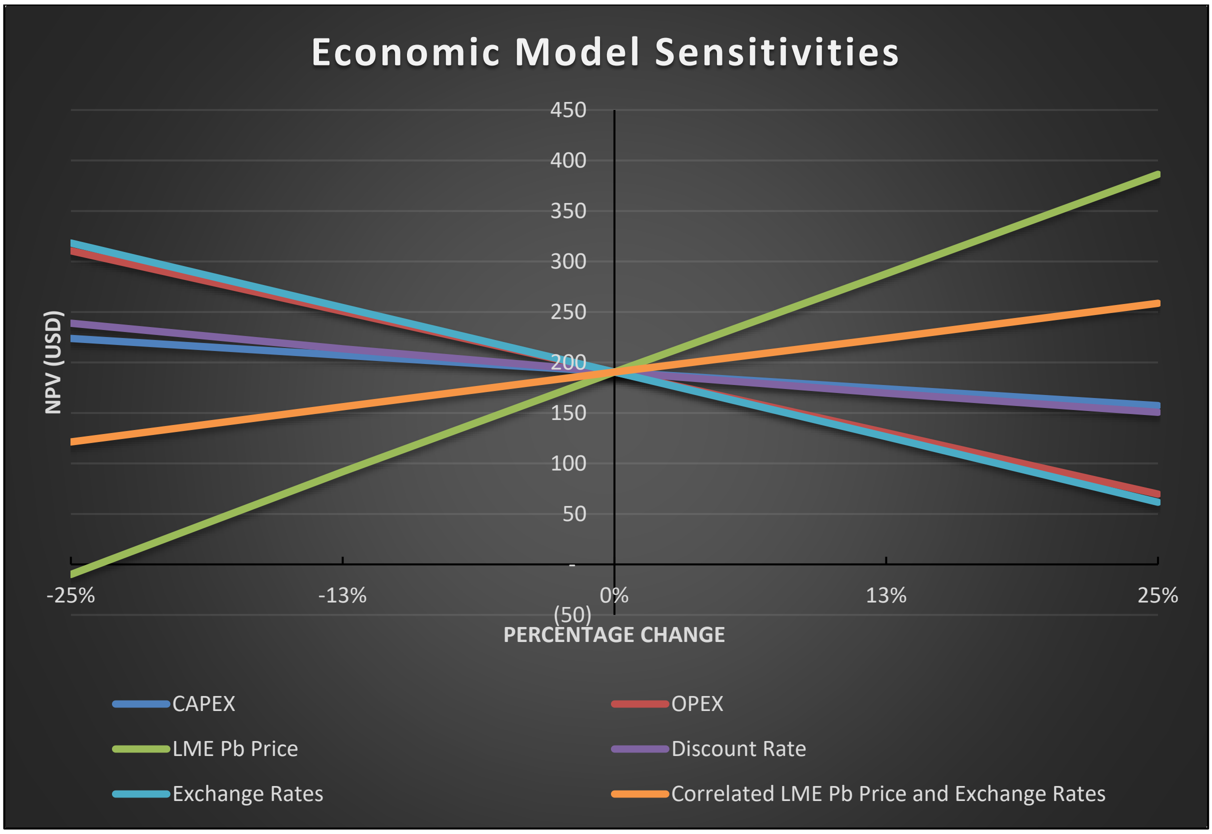


Figure 7: Economic SENSITIVITIES

1.12.1 Lead Sales Price

LME lead prices are based on the Wood Mackenzie Q4 2017 Long-Term Outlook.

The forecast sales price of lead ingot produced from the Hydrometallurgical Facility is based on the LME Price Forecast plus refined lead premia of USD 85 per t of lead ingot based on a minimum lead ingot grade of 99.97% Pb. RHM expects refined lead premia of at least USD 110 per t of lead ingot based should a lead ingot grade of 99.99% Pb be exceeded.

The forecast sales prices and sales value for lead ingot by year of production is set out in Table 7.

Table 7: Product and Sales Revenue

Year	Sales Price			Sales Value	
	LME Price (USD/t Pb)	Metals Premia (USD/t Pb)	Sales Price (USD/t Pb)	Sales Amount (t Pb)	Sales Value (USD millions)
Year 1	2,250	85	2,335	45,275	105.72
Year 2	2,250	85	2,335	69,813	163.01
Year 3	2,250	85	2,335	69,813	163.01
Year 4	2,250	85	2,335	69,813	163.01
Year 5	2,250	85	2,335	69,813	163.01
Year 6	2,250	85	2,335	69,813	163.01
Year 7	2,250	85	2,335	69,813	163.01
Year 8	2,250	85	2,335	69,813	163.01
Year 9	2,250	85	2,335	69,813	163.01
Year 10	2,250	85	2,335	69,813	163.01

Year	Sales Price			Sales Value	
	LME Price (USD/t Pb)	**Metals Premia (USD/t Pb)**	**Sales Price (USD/t Pb)**	**Sales Amount (t Pb)**	**Sales Value (USD millions)**
Year 11	2,250	85	2,335	57,488	134.23
Year 12	2,250	85	2,335	49,681	116.01
Year 13	2,250	85	2,335	47,672	111.31
Year 14	2,250	85	2,335	48,169	112.48
Year 15	2,250	85	2,335	47,494	110.90
Avg / Total	**2,250**	**85**	**2,335**	**924,093**	**2,158**

Cashflows by year of operation are set out in Table 8 and

Table 9.

Table 8: Annual Revenue and Costs (USD M)

Year	Sales Revenue	Royalties	Utilities (ex Hydromet)	Mining	Concentrator	Hydromet Facility	Supply & Logistics	Other Fixed Opex
Year 1	105.72	-3.00	-3.30	-22.32	-19.44	-13.37	-8.23	-11.95
Year 2	163.01	-4.58	-3.80	-27.71	-21.54	-17.19	-11.99	-11.91
Year 3	163.01	-4.58	-3.80	-28.00	-21.54	-17.19	-11.99	-11.91
Year 4	163.01	-4.58	-3.80	-28.00	-21.54	-17.19	-11.99	-11.91
Year 5	163.01	-4.58	-3.80	-22.98	-21.58	-17.21	-11.99	-11.95
Year 6	163.01	-4.58	-3.80	-22.95	-21.54	-17.19	-11.99	-11.91
Year 7	163.01	-4.58	-3.80	-23.02	-21.54	-17.19	-11.99	-11.91
Year 8	163.01	-4.58	-3.80	-20.59	-21.54	-17.19	-11.99	-11.91
Year 9	163.01	-4.58	-3.80	-20.64	-21.58	-17.21	-11.99	-11.95
Year 10	163.01	-4.58	-3.80	-20.62	-21.54	-17.19	-11.99	-11.91
Year 11	134.23	-3.79	-3.80	-15.92	-21.54	-15.26	-10.10	-11.91
Year 12	116.01	-3.29	-3.80	-15.82	-21.54	-14.04	-8.91	-11.91
Year 13	111.31	-3.16	-3.80	-15.82	-21.58	-13.75	-8.60	-11.95
Year 14	112.48	-3.19	-3.80	-16.17	-21.54	-13.81	-8.67	-11.91
Year 15	110.90	-3.15	-3.80	-10.67	-21.54	-13.70	-8.57	-26.02
Total	**2,158**	**-61**	**-56**	**-311**	**-321**	**-239**	**-161**	**-193**

Table 9: Annual Cashflows (USD M)

Year	Sales Revenue	Variable Opex	Fixed Opex	Ongoing Capex	Gross Cashflow	Income Tax	Net Cashflow
Year 1	105.72	-44.53	-36.33	-0.75	24.10	0.00	24.10
Year 2	163.01	-61.73	-36.24	-0.75	64.30	0.00	64.30
Year 3	163.01	-62.02	-36.24	-0.75	64.00	-8.02	55.99
Year 4	163.01	-62.03	-36.24	-0.75	64.00	-15.57	48.43
Year 5	163.01	-57.00	-36.33	-0.75	68.93	-15.57	53.36
Year 6	163.01	-56.97	-36.24	-0.75	69.05	-17.05	52.01
Year 7	163.01	-57.04	-36.24	-0.75	68.99	-17.08	51.91
Year 8	163.01	-54.61	-36.24	-0.75	71.42	-17.06	54.35
Year 9	163.01	-54.66	-36.33	-0.75	71.27	-17.79	53.47
Year 10	163.01	-54.64	-36.24	-0.75	71.38	-17.75	53.63
Year 11	134.23	-45.33	-36.24	-0.75	51.92	-17.78	34.13
Year 12	116.01	-42.31	-36.24	-0.75	36.71	-11.94	24.76
Year 13	111.31	-41.56	-36.33	-0.75	32.66	-7.38	25.28
Year 14	112.48	-42.10	-36.24	-0.75	33.39	-6.17	27.22
Year 15	110.90	-36.35	-36.24	-14.85	23.46	-9.79	13.67
Total	**2,158**	**-773**	**-544**	**-25**	**816**	**-179**	**637**

1.13 Conclusions and Recommendations

The DFS and associated economic analysis has demonstrated that the construction and operation of a Hydrometallurgical Facility on site to produce lead metal in ingot form is economically feasible and attractive. It is recommended that the following activities are initiated by RHM.

- Secure the environmental approvals for the construction and operation of the Hydrometallurgical Facility.
- Undertake a closed cycle pilot plant testwork to optimise to the concentrator and Hydrometallurgical Facility flow sheets.
- Undertake FEED work and appoint the Hydrometallurgical Facility construction engineer.
- Arrange financing for the commencement of the construction of the Hydrometallurgical Facility.
- Undertake additional work to convert Inferred Mineral Resources to Measured and Indicated to potentially increase the Mine life.

SRK reviewed the actual and projected product sales and operating cost data for the Mine. Based on this review and the above-defined variables, SRK concluded that the Mine has a positive NPV; therefore, the Mineral Reserve statement in Section 15 is valid.

2 Introduction

2.1 Terms of Reference and Purpose of the Report

The Mine, located in the Wiluna District of Western Australia is 100% owned by LeadFX, through its wholly owned subsidiary RHM. Lead*FX* is an international base metal mining company listed on the TSX.

The purpose of this report is to provide a summary of the technical and economic feasibility of constructing and operating a Hydrometallurgical Facility at the Mine in Wiluna, Western Australia that integrates the existing mine and processing facility with a new Hydrometallurgical Facility to produce lead metal in ingot form pursuant to NI 43-101 and other rules of the Canadian Securities Administrators. The quality of information, conclusions, and estimates contained herein is based on:

1 information available at the time of preparation,

2 data supplied by outside sources, and

3 the assumptions, conditions, and qualifications set forth in this report.

This report is intended for use by LeadFX subject to the terms and conditions of its contract with the QPs, Competent Persons (“CPs”) and relevant securities legislation.

The contract permits Lead*FX* to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party’s sole risk.

The responsibility for this disclosure remains with Lead*FX*. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

This report provides Mineral Resource and Mineral Reserve estimates, and a classification of resources and reserves prepared in accordance with the JORC 2012 Code of Practice. The Canadian Institute of Mining, Metallurgy and Petroleum (“CIM”) accepts the JORC Code under the use of a Foreign Code.

2.2 Qualifications of Consultants

The Consultants responsible for this Technical Report are specialists in many recognised mining industry fields of study but are not necessarily limited to those of geology, exploration, Mineral Resource and Mineral Reserve estimation and classification, mining, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the Consultants or any associates employed in the preparation of this report has any beneficial interest in Lead*FX*. The Consultants are not insiders, associates, or affiliates of Lead*FX*.

The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Lead*FX* and the Consultants. The Consultants are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101 standard, for this Technical Report, and are members in good standing of appropriate professional institutions.

The QPs are responsible for specific sections as follows:

- Scott McEwing is the QP responsible for the preparation of the report and Sections 1 to 6, 13[3], 16, 17[4], and 18 to 28 summarised there from, of this Technical Report.
- Alan Taylor is the QP responsible for Sections 13[5] and 17[6] summarised there from, of this Technical Report.
- Lawrie Gillett is the QP responsible for the Mineral Reserve – specifically, Section 15 of this Technical Report.
- Kahan Cervoj is the QP responsible for the Mineral Resource estimate specifically Sections 7 to 12 and 14 of this Technical Report.

The Mineral Resource and Mineral Reserve estimates prepared for this report were completed by the following consultants to the JORC Code standard:

- Kahan Cervoj, MAusIMM, MAIG is the consultant responsible for the preparation of a JORC (2012) standard Mineral Resource estimate.
- Adrian Jones, MAusIMM is the consultant responsible for preparation of a JORC (2012) standard Ore Reserve estimate.

2.3 Details of Inspection

Scott McEwing is responsible for the content, and editing of this Technical Report. The Certificates of the Authors are provided in Appendix A.

A site visit to the Mine was conducted by Scott McEwing on November 11 and 12, 2014 when it was in production. The site visit consisted of visiting the mining operations, reviewing mine data and information and observing plant operations.

A site visit to the Mine was conducted by Kahan Cervoj from July 23 - 25, 2014 when it was in production. The site visit consisted of reviewing deposit geology, logging and sampling protocols.

A site visit to the Mine was conducted by Adrian Jones on March 10, 2015 shortly after the Mine was placed in care and maintenance. The site visit consisted of visiting the recently shut mining operations and inspecting the Mine infrastructure.

Details of the site visits undertaken are provided in Table 10.

Table 10: Site Visits

Personnel	Company	Expertise	Date(s) of Visit	Details of Inspection
Scott McEwing	SRK	Mining Engineer	November 11 and 12, 2014	Site visit, review and observation of operations
Adrian Jones	AMC Consultants Pty Ltd	Mining Engineer	March 10, 2015	Site visit and inspection of Mine infrastructure
Kahan Cervoj	Optiro Pty Ltd	Geologist	July 23 to 25, 2014	Review of deposit geology, logging and sampling protocols

[3] With respect to the flotation components contained within.
[4] With respect to the flotation components contained within.
[5] With respect to the hydrometallurgical components contained within.
[6] With respect to the hydrometallurgical components contained within.

2.4 Sources of Information

Each QP's opinion contained herein is based on information provided to such QP by Lead*FX* and RHM personnel throughout the course of the investigations.

The QPs reviewed the available Mine data and incorporated the results thereof, with appropriate comments and adjustments as needed, in the preparation of this Technical Report. Standard industry professional review procedures were used throughout in the preparation of this Technical Report.

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

A list of documents used to support the Technical Report includes:

- Financials and operating budgets including lead market studies provided by Lead*FX* and/or RHM personnel, referring specifically to Items 21 and 0. The information was provided in the form of internal financial models and net smelter return (NSR) calculations.
- All geological data including deposit description, past exploration, drilling results, sample preparation and analysis, data verification, and legacy Mineral Resource reports provided by Lead*FX* and/or RHM personnel, referring specifically to sections 7, 8, 9, 10, 11, 12, and 14
- Mining methods, provided by Lead*FX* and/or RHM, referring specifically to Section 16.
- Mineral Processing, metallurgical testing and recovery methods provided by Lead*FX* and /or RHM personnel, referring specifically to Items 13 and 17. The information was provided in the form of operational data, the previous NI43-101 Technical Report dated March 10, 2015, and the DFS prepared by SNC-Lavalin.

2.5 Effective Date

The effective date of this report is February 28, 2018.

2.6 Units of Measure

The International System for weights and units has been used throughout this report. Tonnage is reported in metric tonnes. All currency is in United States Dollars ("USD") unless otherwise stated.

3 Reliance on Other Experts

The QPs' opinion contained herein is based on information provided to the QPs by Lead*FX* and/or RHM throughout the course of the investigations. The QPs have relied upon the work of other consultants in support of this Technical Report.

The QPs relied upon the work of others to describe the following sections:

- Mine and corporate history, provided by Lead*FX* and/or RHM personnel, referring specifically to Section 6
- Environmental, regulatory permitting, social or community impact (including Native Title), Mine infrastructure and general area resources, provided by Lead*FX* and/or RHM personnel, referring specifically to sections 4, 5, 18 and 20
- Land tenure and land title, referring specifically to Section 4
- Royalties, Agreements and Encumbrances in Section 4.3

These submissions have not been independently verified by the QPs and the QPs did not seek an independent legal opinion of these items.

4 Property Description and Location

4.1 Property Location

The Mine is located 30 km west of Wiluna, and 2 km directly north of the Wiluna–Meekatharra road in the northeastern goldfields region of Western Australia (Figure 8). The Mine is located in the East Murchison Mineral Field on mining leases M53/501, M53/502, M53/503, M53/504 and M53/1002 and various miscellaneous and exploration licences (the Site). The leases and licences cover in excess of 30,000 hectares (ha), including 2,447 ha of Mining Leases.

The Mine deposits are situated at approximately 26° 31" S latitude and 119° 57" E longitude.

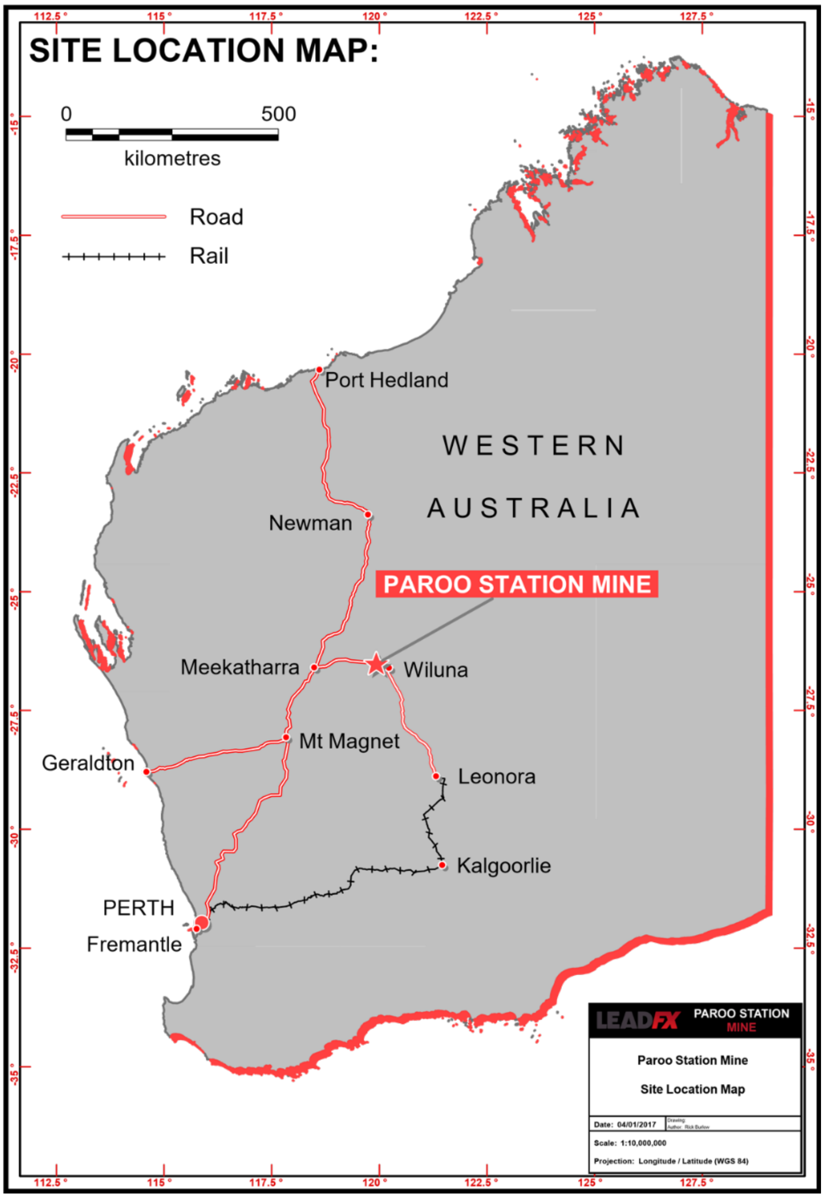


Figure 8: Location Map

Source: RHM (2018)

4.2 Mineral Titles

mining Leases are current over the mining operations and the Mineral Resources within the life of mine ("LOM") plan are on mining leases, with the exception of the Pizarro deposit that is on a Mining Lease Application.

The tenements comprising the Magellan Hill properties provide sufficient surface and mining rights to operate over the Mine life.

A mining lease application; M53/1100 has been submitted to cover a very small portion of the Mineral Resource that extends onto Exploration Licence E53/644.

Table 11: Tenement Position

Type	Title No.	Issue date	Expiry date	Approximate Hectares/ Blocks	State
Rosslyn Hill Mining Pty Ltd. – Granted Tenement Holdings					
Exploration Licence	E53/1528	04-Apr-11	03-Apr-21	280 Hectares	Western Australia
Exploration Licence	E53/644	01-Jul-96	30-Jun-18	840 Hectares	Western Australia
Exploration Licence	E53/1560	03-Feb-12	02-Feb-22	280 Hectares	Western Australia
Miscellaneous Licence	L53/106	09-Dec-99	08-Dec-20	1 Hectares	Western Australia
Miscellaneous Licence	L53/107	09-Dec-99	08-Dec-20	43 Hectares	Western Australia
Miscellaneous Licence	L53/108	09-Dec-99	08-Dec-20	5 Hectares	Western Australia
Miscellaneous Licence	L53/149	30-May-06	29-May-27	195 Hectares	Western Australia
Miscellaneous Licence	L53/163	20-June-13	19-June-34	3,994 Hectares	Western Australia
Miscellaneous Licence	L53/164	20-June-13	19-June-34	8,254 Hectares	Western Australia
Miscellaneous Licence	L53/197	12-Jan-15	11-Jan-36	4,680 Hectares	Western Australia
Miscellaneous Licence	L53/198	12-Jan-15	11-Jan-36	9,211 Hectares	Western Australia
Miscellaneous Licence	L53/200	17-Dec-15	16-Dec-36	32 Hectares	Western Australia
Miscellaneous Licence	L53/201	17-Dec-15	16-Dec-36	23 Hectares	Western Australia
Mining Lease	M53/1002	22-Jun-04	21-Jun-25	191 Hectares	Western Australia
Mining Lease	M53/501	05-May-99	04-May-20	356 Hectares	Western Australia
Mining Lease	M53/502	05-May-99	04-May-20	975 Hectares	Western Australia
Mining Lease	M53/503	05-May-99	04-May-20	499 Hectares	Western Australia
Mining Lease	M53/504	05-May-99	04-May-20	426 Hectares	Western Australia
Prospecting Licence	P53/1528	15-Apr-11	14-Apr-19	22 Hectares	Western Australia
Ivernia Australia Pty Ltd. – Granted Tenement Holdings					
Exploration Licence	E53/695	15-May-97	14-May-17	840 Hectares	Western Australia
Rosslyn Hill Mining Pty. Ltd. – Tenement Applications					
Miscellaneous Licence	L53/191	Applied On: 15-Aug-14		8.4128 Hectares	Western Australia
Miscellaneous Licence	L53/192	Applied On: 15-Aug-14		0.9612 Hectares	Western Australia
Miscellaneous Licence	L53/193	Applied On: 15-Aug-14		5.2566 Hectares	Western Australia
Miscellaneous Licence	L53/194	Applied On: 15-Aug-14		1.0612 Hectares	Western Australia
Miscellaneous Licence	L53/195	Applied On: 15-Aug-14		0.2424 Hectares	Western Australia
Miscellaneous Licence	L53/196	Applied On: 15-Aug-14		1.4153 Hectares	Western Australia
Ivernia Australia Pty Ltd. – Tenement Applications					
Mining Lease	M53/1088	Applied On: 01-Jul-08		614 Hectares	Western Australia
Mining Lease	M53/1100	Applied On: 15-Sep-17		50 Hectares	Western Australia

Source: RHM (2018)

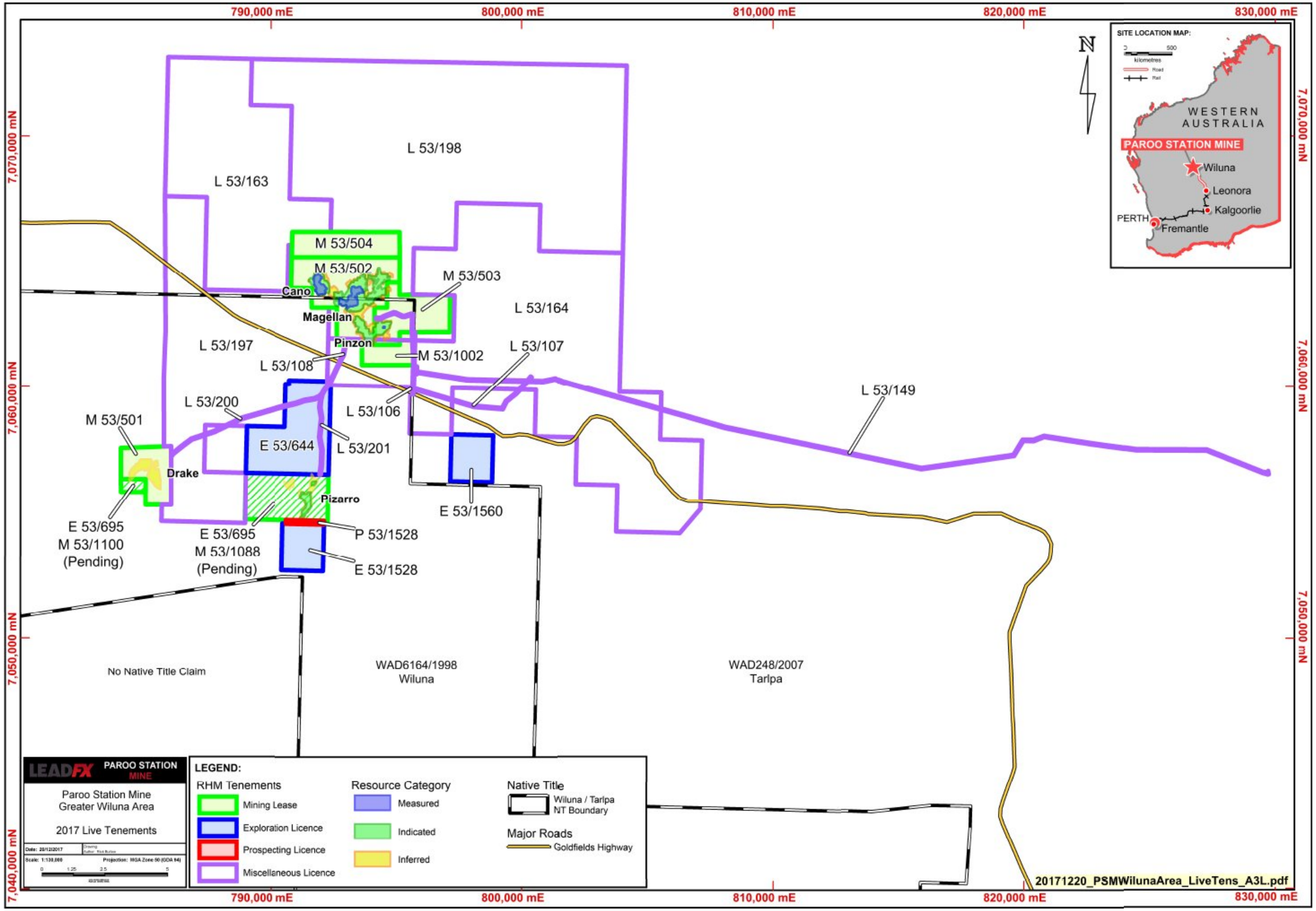


Figure 9: Land Tenure Map as at April 2018

Source: RHM (2018)

4.2.1 Nature and Extent of Issuer's Interest

In Western Australia, mineral rights belong to the State. The government issues and administers mining tenements under the relevant mining legislation, and mining companies must pay royalties to the government based on saleable production.

Exploration and mining titles in Western Australia are granted in accordance with the Mining Act 1978 (WA), which is administered by the Department of Mines Industry Regulation and Safety ("DMIRS").

Australian law generally requires that all necessary Native Title approval be obtained before a Mining Lease can be granted and mining operations can commence. Lead*FX* has been granted Mining Leases supporting its current Mineral Reserve.

There are two applications pending grant being M53/1088 (Pizarro) and M53/1100 (southern tip of Drake) which will require the successful negotiation of grant conditions with the Traditional Owners of this lease area. The negotiations are continuing successfully. The risk of an unsuccessful outcome of the negotiation is considered low.

Production from Pizarro and Drake is not scheduled in the current Mine plan contained herein. The risk of an unsuccessful outcome of these negotiations is considered low.

4.3 Royalties, Agreements and Encumbrances

RHM reports that there are two royalty payments which are applicable to the Mine.

Under the Mining Regulations 1981 (WA), RHM is required to pay a royalty to the State Government at the prescribed rate of 2.5%.

In accordance with the terms of the Wiluna Land Access Agreement of 2006 (which superseded the Heritage Agreement dated September 25, 1998 between RHM and the Milangka Native Claimant Group), RHM is required to make a royalty payment of AUD 0.04/t of all ore milled from the Mine into the Wiluna Claimant Trust Fund. Another Land Use Agreement, dated December 16, 1998 between RHM and the now unregistered Wanmulla Group, provides for a further AUD 0.04/t of all ore milled from the Mine, which may be payable if a descendent claim from the Wanmulla claim is registered.

A second agreement with the Wiluna claimants, over the gas pipeline route, requires an annual compensation payment into the Wiluna Claimant Trust Fund for use of the gas pipeline tenement area. The annual payment of AUD 20,000 was made initially in July 2006 and subsequent annual payments have been made, indexed at the consumer price index ("CPI") rate for Perth, Australia.

4.4 Environmental Liabilities and Permitting

4.4.1 Environmental Liabilities

In March 2018, the Company filed its Compliance Assessment Report ("CAR"), along with its three Annual Environment Reports ("AER") for 2017 to the four regulatory authorities.

The CAR and the AERs are the key annual environmental disclosure documents produced by RHM and submitted to the Western Australian regulatory authorities.

RHM disclosed that there are no outstanding environmental issues.

RHM has identified the anticipated closure costs required for the Mine, based on best available information. The cost estimate takes into account all aspects of rehabilitation and closure activities utilising third-party contractor rates including the Hydrometallurgical Facility.

RHM has a fully-costed closure cost estimate that is "commercial in confidence" between RHM and the respective Western Australian government departments overseeing this aspect of the operation.

4.4.2 Required Permits and Status

The Company regularly collects and reports occupational health, safety and environmental information to the following State Departments:

- EPA – Environmental Protection Authority
- DWER – Department of Water and Environmental Regulation
- DMIRS – Department of Mines Industry Regulation and Safety
- DOH – Department of Health
- DoT – Department of Transport

Operating conditions and licences for the Mine have been granted and the following are currently in force:

- Ministerial Statement 905 and 1042
- DWER – Prescribed Premises Licence – L8493/2010/2
- DWER – Licence to Extract Water – GWL96342(4).
- Australian Communications & Media Authority – Licences 1970164 and 1970178/1
- DMIRS – Dangerous Goods Site Licence – DGS020079
- DMIRS – Mining Tenement conditions
- DMIRS – Pipeline Licence – PL73
- Radiological Council – Licences LX58/2006 15145 and RS28/2005 14619

New and/or updated operating permits will be required with the construction and operation of the Hydrometallurgical Facility.

4.4.3 Other Significant Factors and Risks

Currently further regulatory approvals are being sought for the construction and operation of a Hydrometallurgical Facility at the Mine site and the necessary changes to support the increased Mine life.

The approvals being sought relate to:

- construction and operation of a new Hydrometallurgical Facility, broadly consisting of acid leach, electro winning and melting, to convert the lead carbonate concentrate currently approved to be produced at the Mine site, to an estimated 70,000 t per annum of lead metal
- transporting of lead metal ingots (approximately) 25 kg each, from the Mine site for export
- increasing of the on-site power generation capacity to 18 MW installed, to be fuelled from the existing natural gas spur line
- increasing the existing approved disturbance footprint by 400 ha, taking the total to 980 ha within a 2094 ha Development Envelope
- increasing the tailings storage capacity by 19 Mt taking the total to 35 Mt within the 2094 ha Development Envelope.

The approvals will not require changes to mining methods, concentrating methodology, tailings storage methodology, waste rock storage methodology, mining below the water table or the current water abstraction licence.

Currently the Referral Document is before the State EPA, under Part IV of the EP Act, to increase the disturbance footprint by 400 ha, taking the total disturbance footprint to 980 ha, within the development envelope. The Referral Document also includes for an increase of 19 Mt tailings storage capacity, taking the total storage capacity to 35 Mt, to meet the needs of the new forecast mine life. The document also describes the Hydrometallurgical Facility and the proposed new electricity generation plant at site. On April 4, 2018 the EPA determined their Level of Assessment of the Referral Document. The level set was “Referral Information” (the information contained within the Referral Document being sufficient for their purposes), with a request for some additional information to be supplied by RHM.

In the future, to allow access to Magellan Hill orebodies below the water table, and both the Pizarro and Drake orebodies to the south, a further application under Section 38 of the EP Act is planned. The application will be developed and submitted prior to access requirements. Groundwater and dewatering impact studies are required to be undertaken, together flora and fauna studies for the southern orebodies, with consultation with relevant Government Departments.

5 Accessibility, Climate, Local Resources, Infrastructure & Physiography

5.1 Topography, Elevation and Vegetation

The Mine is located within the Glengarry land system. Land systems define an area with a recurring pattern of landforms soils and vegetation (Mabbutt, *et al.,* 1963). Mabbutt et al. (1963) characterised the Glengarry land system as stony undulating plateaus, concave hill slopes with small breakaways, and wide drainage floors with minor channels.

The soils that are associated with this land system tend to be shallow stony red earths in the plateau and hill slopes, and deep red earths in the drainage floors and channels (Table 12). The main landform of the Mine area is the north west-facing arm of the Finlayson Range and isolated hill formations of Proterozoic sediment outliers from the Finlayson Range. Such formations include Mount Russell (599 metres Australian Height Datum (mAHD)), Mount Bartle (584 mAHD) and the hills containing the Magellan lead deposit (565 mAHD) (KH Morgan & Associates, 1999).

Table 12: Description of landforms, soil and vegetation associations within Glengarry land system

Landform	Description	Soils	Vegetation
Summits/ Stony Plateau	Strongly undulating surfaces up to 1.6 km wide and extending up to 5 km along strike; regional slope gradients are up to 2%; surfaces are locally dissected up to 10 m, with valley slopes up to 5%, stony surfaces with rocky outcrops.	Very shallow stony red clayey sands	Dense Acacia aneura (Mulga), Acacia pruinocarpa (Gidgee), and other Acacia spp., with scattered tall mallee in some areas, many shrubs, some Triodia schinzii (feathertop Spinifex), and other perennial grasses.
Hill slopes	Concave, mainly to 15%, small breakaways and benches up to 6 m high on massive quartzite or silicified rock, and minor steep slopes in kaolinised rock; stony surfaces, in part gullied to 10 m depth.	Outcrop with little adjacent soil	Open mulga with dense shrubs unpalatable perennial grasses, forbs, and short annual grasses.
Lower slopes	Concave, 1 – 5% and up to 160 m long, lightly dissected surfaces with rock outcrops in upperparts.	Shallow, stony soils on hard pan or rock	Open mulga and dense shrubs, patches of Triodia pungens (soft Spinifex) and short annual grasses.
Drainage floors	Up to 100 m wide, gradients 1 in 50 to 1 in 150; mainly with channeled tracts up to 30 m wide; concave marginal slopes up to 0.55, with lightly sealed alluvial surfaces and stony patches.	Red earths, locally deep and without hard pan	Mulga of variable density, with edible and inedible shrubs, various perennial grasses with clumps of Triodia spp., abundant herbage, and short annual grasses.
Channels	Up to 10 m wide and 2 m deep, braiding locally, gradients 1 in 15 to 1 in 150.	Bed loads range from sand to boulders on hard pan or bedrock	Similar to drainage floors, but with fewer perennial grasses.

5.1.1 Soils

Soils found within the Wiluna–Meekatharra region are derived from sediments that fill the Glengarry Basin lying between the Pilbara and Yilgarn cratons. Intensive weathering in the Wiluna–Meekatharra area has led to the development of laterite and silcrete during the Tertiary period and the outstanding feature of soils within this region is their heavily leached nature, and the presence of a cemented or siliceous hardpan layer (Keith Lindbeck and Associates, 1999).

Hart *et al.,* (1999) describes the Mine area as a low stony plateau within an area of loamy plains where the soils on the plateau are best described as sandy or skeletal, with numerous stones.

5.2 Accessibility and Transportation to the Property

5.2.1 Regional

The township of Wiluna is approximately 30 km east of the Mine site. Wiluna is the principal centre in the Shire of Wiluna known predominantly as a mining and pastoral area. The population of the town of Wiluna is approximately 300, with the last official census reporting the population of the Shire as 1644, which includes several mining villages that mainly operate on a fly-in fly-out basis (Shire of Wiluna website).

Access to the site is via the Goldfields Highway from Wiluna or via Geraldton (Figure 8), with 50% of the 30 km section between the Mine and Wiluna consisting of sealed road, and the remaining 50% being well maintained, gravel pavement. The tenement area straddles the highway and a well maintained, 3 km-long gravel road links the Mine operations to the highway.

The Mine's proximity to the highway means easy access to services operating out of Wiluna, as well as Kalgoorlie, Geraldton and Perth.

5.2.2 Mine Site

Graded mine site roads provide access to the Magellan and Cano pits, waste rock landform, tailings storage, processing plant, offices and accommodation village. Access to the undeveloped deposits (Pinzon, Pizarro and Drake) is via pastoral station and exploration tracks.

The gravel roads are subject to closure during times of heavy rainfall. The closures can last between 24 and 72 hours however, closures are normally less than 36 hours and can happen during the summer months (December to March) when cyclonic activity is at its peak.

5.2.3 Workforce

The workforce is accommodated on site in a purpose-built accommodation village and has been managed on a fly-in fly-out ("FIFO") basis (typically 8 days on, 6 days off; 4 days on, 3 days off, 12-hour shifts), with the majority of the workforce living in Perth. All flights are in and out of the Wiluna airport, located approximately 30 km from the site.

5.3 Climate and Length of Operating Season

5.3.1 Climate

The Mine is located in the semi-arid climatic region in the northern goldfields region of Western Australia, approximately 30 km west of Wiluna. The mean maximum daily temperatures range from 37.9°C in January to 19.4°C in July, with the mean minimum daily temperature ranging from 5.4°C in July to 22.9°C in January (Wiluna Bureau of Meteorology ("BOM") Number 013012,2014). The mean annual evaporation rate at Wiluna is estimated at 4,072 mm (Department of Agriculture 1987), thus exceeding mean annual rainfall by around 3,800 mm.

The average annual rainfall for Wiluna is 259.2 mm (Wiluna BOM Number 013012, 2014).

(Table 13). The region receives the majority of the rainfall in the summer (Figure 10), which is often associated with subtropical thunderstorms and cyclonic events. The annual rainfall varies markedly from year to year, with high rainfall years associated with high intensity long-duration rainfall events, which often exceed the average annual rainfall.

Table 13: Climatic data for Wiluna weather station (Bureau of Meteorology)

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sept	Oct	Nov	Dec
Average Rainfall (mm)	37.4	38.5	37.4	28.9	25.5	23.7	14.9	9.9	5.0	7.3	11.9	22.3
Average Daily Evaporation (mm)	11.0	9.5	7.8	5.6	3.7	2.5	2.6	3.7	5.7	7.9	9.3	10.1
Average Maximum Temperature (°C)	37.9	36.5	34.0	29.3	23.8	19.9	19.4	21.9	26.3	30.3	34.0	36.8
Average Minimum Temperature (°C)	22.9	22.1	19.6	15.1	10.0	6.7	5.4	6.8	9.9	13.9	17.9	21.1

Source: Bureau of Meteorology (2018)

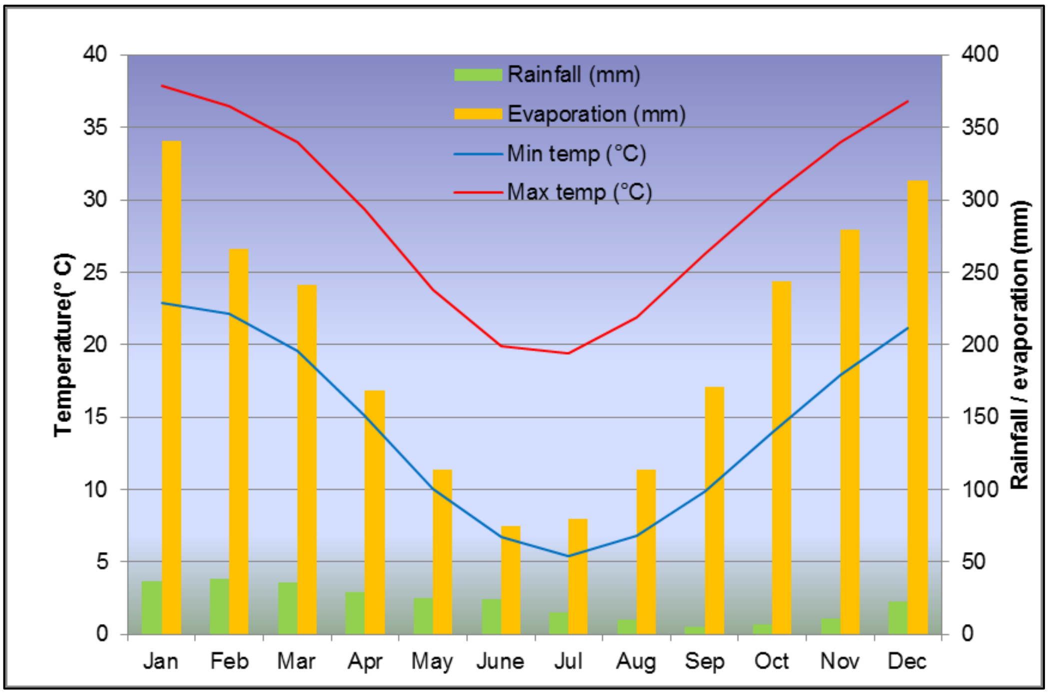


Figure 10: Climate data for Wiluna

Source: Bureau of Meteorology (2018)

5.4 Sufficiency of Surface Rights

RHM has been granted Mining Leases that support the current Mineral Reserves. Mining Lease Applications M53/1088 and M53/1100 cover the Pizarro deposit and the southern tip of the Drake deposit, respectively, both which are not part of the current Mine plan. The granting of M53/1088 and M53/1100 will require the successful negotiation of grant conditions with the Traditional Owners of these lease areas. The negotiations are continuing successfully. The risk of an unsuccessful outcome of the negotiation is considered low.

5.5 Infrastructure Availability and Sources

The current infrastructure in the Mine area is sufficient to fully support mining and lead carbonate concentrate production activities (refer Section 18). The Mine operated at a nameplate capacity of approximately 1.6 M t per annum (“Mtpa”) for three operational phases between January 2005 until April 2007, March 2010 to January 5, 2011 and April 4, 2013 to January 16, 2015.

It operated on a continual basis from April 2013 until being put onto care-and-maintenance at a capacity greater than 1.4 Mtpa and for periods of up to 1.7 Mtpa.

Additional power is required from a new 18 megawatt (“MW”) power plant to support the planned Hydrometallurgical Facility. Additional construction camp accommodation would be needed to support the construction of the Hydrometallurgical Facility.

5.6 Existing Infrastructure

5.6.1 Water

Water is supplied from an established borefield, with onsite treatment for the supply of potable water and is sufficient for historical throughput levels, together with the planned water usage for the Hydrometallurgical Facility. No increase to the water licence quantity is required.

5.6.2 Electricity

Current diesel power generation plant will be refurbished and used as back-up and emergency supply.

5.6.3 Tailings Storage

A paddock style TSF currently exists on site with regulatory approval to store tailings within the waste rock landform, known as an Integrated Waste Landform (“IWL”).

5.6.4 Accommodation Village

The accommodation village for site personnel is sufficient for approximately 170 personnel.

5.7 Planned Infrastructure

5.7.1 Water

No increase to the water licence quantity or borefield supply system is required to facilitate the operation of the Mine with the Hydrometallurgical Facility.

5.7.2 Electricity

Electricity will be generated on site with nine, 2 MW natural gas fuelled engines producing 18 MW of installed capacity.

5.7.3 Tailings Storage

As part of Mine restart, some civil works will be undertaken to create the start of the approved IWL to create a tailings storage cell within the existing waste rock landform.

5.7.4 Accommodation Village

Some upgrading and expansion of the accommodation village is planned to cater for increased numbers of site personnel as a result of the operation of the Hydrometallurgical Facility.

6 History

6.1 Prior Ownership and Ownership Changes

The Magellan deposit was, discovered in 1991 by Renison and acquired in 1998 by Westralian Sands Ltd, subsequently renamed Iluka.

RHM had the right to acquire a 100% interest in the Renison Properties subject to payment to Renison of the Renison Royalties pursuant to a Farm-in Agreement between Renison and RHM dated January 23, 1997. It was agreed that the acquisition by RHM of a 100% interest was conditional upon RHM completing a bankable feasibility study for the Mine by January 2002 and committing to develop a mine and plant with a design capacity of not less than 300,000 t of ore per annum.

In September 2001, following the completion of such a feasibility study, RHM committed to develop a mine and plant with the required capacity, and thereby secured its rights to a 100% interest in the Mine. The Renison Properties were transferred to RHM during 2002.

On April 20, 1999, LeadFX agreed to invest in the Mine by acquiring a direct 15.7% equity interest in RHM from Polymetals, the sole shareholder of RHM.

In September 2000, LeadFX acquired a 90% equity interest in Polymetals and acquired the remaining equity ownership in Polymetals in 2003.

In May 2003, LeadFX entered into a termination agreement with Iluka, pursuant to which all of Iluka’s remaining rights under the 1997 farm-in agreement, including the Renison Royalties, were terminated in consideration of a one-time payment to Iluka of AUD 2.1M.

In 2003, LeadFX and Sentient formed a Joint Venture under which, Sentient agreed to provide financing to RHM in exchange for a 40% interest. The Sentient share of the Joint Venture was increased to 49% in 2004.

In April 2005, LeadFX acquired Sentient 49% interest in RHM, thereby becoming the sole owner of the mine through its 100% interest in RHM.

6.2 Exploration and Development Results of Previous Owners

Renison initially discovered the deposit by stream sediment sampling whilst exploring the region for base metal mineralisation. A series of regional rotary air blast (“RAB”) holes to the north and south of the Magellan Hill returned anomalous values of between 0.1% and 3.1% Pb, and follow-up work on these holes led to the discovery of the Magellan deposit in June 1991 (Sibbel, 2009).

Renison completed several programs of reverse circulation (“RC”) drilling and later, diamond drilling to follow up the anomalous RAB results. A total of 42 conventional RC holes for 2,576 m and 22 diamond holes for 1,763 m were drilled between November 1991 and February 1995. The drilling supported an initial Mineral Resource estimate for the Magellan deposit.

RHM has completed several drill programs within the Magellan Hill area since 1997 for exploration, resource evaluation and sterilisation purposes, and has discovered an additional five deposits within the Mine area.

6.3 Historic Mineral Resource and Mineral Reserve Estimates

6.3.1 Historical Summary

Previous Mineral Resource estimates ("MREs") for the deposits showed the continued improvement in the understanding of the deposits and the increase in recoverable product with continued exploration activity from the initial estimate completed by Renison in 1994. This progression led to a Feasibility Study in 2003 (Watters, 2004). A summary history of the Mine's MREs is provided in Table 14.

Table 14: Summary of MREs

Deposit	Year	Author	Method
Magellan	1994	RGC Manual	Planimeter
Magellan	1996	PL Kitto Block Model	ID^2
Magellan	1997	PL Kitto Block Model	ID^2
Magellan	1999	PL Kitto Block Model	Ordinary kriging
Magellan	2000	MRT Block Model	Multiple indicator kriging
Magellan	2000	Snowden Block model	Ordinary kriging
Magellan	2000	Snowden	Conditional simulation
Cano	2001	Micromine Block Model	ID^2
Cano	2001	Micromine Block Model	Ordinary kriging
Cano	2003	Snowden Block Model	Ordinary kriging
Cano	2004	Snowden (Blair, 2004)	Ordinary kriging
Magellan	2004	Snowden (Blair, 2004)	Ordinary kriging
Unknown	2004 - 2010	Unknown internal revisions	
Drake	2005	FinOre (Williams, 2005)	$ID^{2.5}$
Drake	2007	CSA (Titley and Schaap, 2008)	COG change only
All Deposits	2011	CSA (Shi & Elliott, 2011)	Ordinary kriging
All Deposits excluding Drake	2014	Optiro (Cervoj,2015)	Ordinary kriging
Drake	2016	Optiro (Cervoj, 2016)	2012 JORC update

Source: LeadFX – Various historic documents and compilations

JORC Code is a professional code of practice that sets minimum standards for Public Reporting of minerals Exploration Results, Mineral Resources and Ore Reserves. The JORC Code is consistent with Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") definition standards. MRE's estimated prior to 1999 were reported using the industry conventions of the time.

All Mineral Resources from the Feasibility Study (2001) onwards have been estimated in accordance with the JORC Code. Those reported from 1999 – 2004 used the 1999 version of the code, those completed from 2004 – 2011 were reported under the 2004 version, and the estimate completed in 2014 and 2016 is reported under JORC (2012).

The current MRE, dated December 31, 2016 is discussed in detail Section 14.

6.3.2 2005 – 2010

The June 2005 update was based on new drilling and included revised in situ density parameters, revised top-cuts and cut-off grade of 2.5% Pb. A Resource at Drake was reported for the first time.

The December 31, 2006 update was essentially the June 2005 model depleted by mining as at December 31, 2006.

A similar update was completed for December 31, 2007 where a new mineralised envelope was developed to account for new drilling. Drake cut-off grade lowered to 2.1% Pb.

6.3.3 2010 – 2014

In 2010, CSA completed a revised Ordinary Kriged resource model using the most current exploration and grade control data available at the time. The model used a new set of grade-constrained ‘mineralised lodes’ to establish detailed 1% grade boundaries and to limit inclusion of internal waste lenses.

This differed from the previous 1% grade envelopes which encompassed internal waste and led to a suspected overestimation of the waste blocks. This was a recognisable improvement from the older 2007 model which underestimated total metal by 17% (SRK, 2011).

CSA compiled a report for the 2012 Mineral Resource estimate where previously-generated models used for the 2010 MRE were further depleted by mining based on surfaces constructed from surveys of the mining outlines to the end of April 2011 (CSA, unpublished 2013).

6.3.4 2014 – Present

In 2014 Optiro was commissioned to build revised Ordinary Kriged Magellan Hill and Pizarro resource models and report accompanying MREs. The models were built using updated parameters suitably designed and matched to reconciled mining and milling data from the 2010 – 2011 and 2013 – 2014 operations periods.

6.4 Historic Production

The Mine has been operated over three operational periods; 2005 – 2007, 2010 – 2011, and 2013 to 2015. Table 15 sets out the production achieved during these periods.

Table 15: Production 2005 to 2015

Production Physicals	Unit	Period			
		2005 - 2007	2010 - 2011	2013 - 2015	Total
Ore milled	t	2,197,400	1,035,000	2,447,100	5,679,500
Head grade	%	7.3	6.8	7.1	7.1
Recovery	%	71.7	73.8	77.6	74.6
Concentrate produced	t	181,100	80,700	202,000	463,800
Con grade	%	64.0	64.8	66.8	65.4
Con Pb content	t	115,900	52,200	134,800	302,900

Source: Lead*FX* – Various historic documents

6.4.1 2004 – 2012

The Mine was constructed during 2004, commissioned during 2005, and achieved commercial production on October 1, 2005. From the start of production until it was placed on care and maintenance in April 2007, approximately 181,100 dmt of lead carbonate concentrate was produced at the Mine, with the majority of concentrate being sold to third party smelters in China.

When the Mine was placed on care-and-maintenance following the initiation of government investigations into bird fatalities in the vicinity of the Port of Esperance. The Department of Water and Environmental Regulation (DWER, formerly, the Department of Environment and Conservation), issued a prevention notice on the Esperance Port Authority on March 15, 2007 pursuant to s 73A of the EP Act, 1986 (WA) which precluded the RHM from making any further bulk exports of lead concentrate through the Port of Esperance. As a result, RHM was obliged to pursue alternative shipping arrangements to ship its concentrate through another port in Western Australia and to keep the Mine on care and maintenance until such arrangements had been approved by the DWER.

RHM submitted a formal proposal to the DWER (formally the Office of the Environmental Protection OEPA), in August 2007 to allow for shipment of sealed bags within shipping containers through Fremantle Port. These changes were formally accepted in 2009 when Ministerial Statement 783 was issued.

Production recommenced in late February 2010 and the mine experienced a steady increase of quarterly production through 2010, with 874,000 t of ore processed and 44,100 t of contained lead in concentrate produced for the 12 months ending December 31, 2010.

The operation ceased production again on January 5, 2011, following an order from the Minister for Environment to halt transportation to enable investigation of reports of potential lead egress to the inside of sealed transport containers. No lead egress was found and a thorough investigation resulted in discovery of a laboratory error. The Minister for Environment announced lifting of the order on February 23, 2011, allowing the operation to recommence as soon as practical after that date.

RHM voluntarily placed the Mine onto care-and-maintenance during April 2011 to conduct an ‘end to end’ review of all operational activities. A parallel review under section 46 of the EP Act was undertaken by the OEPA and the review report was published on October 3, 2011. This report resulted in changes to conditions of approval by issue of EP Act Ministerial Statement 905 in July 2012. Ministerial Statement 905 superseded all previous conditions and procedures and became the operational regime for the Mine. 12,700 t of concentrate was produced for the calendar year 2011.

Table 16: Production 2005 to 2007

Production Physicals	Unit	Period			
		2005 Jan – Dec	2006 Jan – Dec	2007 Jan – Apr	2005 – 2007 Total
Ore milled	t	743,900	1,060,100	393,400	2,197,400
Head grade	%	6.5	7.9	7.3	7.3
Recovery	%	64.8	75.5	74.6	71.7
Concentrate produced	t	49,200	99,100	32,800	181,100
Con grade	%	63.6	63.7	65.3	64.0
Con Pb content	t	31,300	63,200	21,400	115,900

Source: Lead*FX* – Various historic documents

Table 17: Production 2010 to 2011

Production Physicals	Unit	Period		
		2010 Feb - Dec	2011 Jan - Mar	2010 - 2011 Total
Ore milled	t	874,000	161,000	1,035,000
Head grade	%	6.8	6.9	6.8
Recovery	%	74.0	73.0	73.8
Concentrate produced	t	68,000	12,700	80,700
Con grade	%	65.0	64.0	64.8
Con Pb content	t	44,100	8,100	52,200

Source: Lead*FX* – Various historic documents

6.4.2 2013 – 2015

On March 28, 2013, RHM announced that it was recommencing processing operations operating under Statement 905. Milling and processing operations recommenced on April 5, 2013 and the mining contractor remobilised to site and mining recommenced at the end of April 2013.

The operation experienced a steady increase of quarterly production through 2013 with no significant disruptions to production or transportation. In 2013, 835,900 t of ore was processed, 44,000 t of contained lead in concentrate was produced and 47,700 t of contained lead in concentrate was sold.

The average plant recovery was 74.6% through 2013 with quarterly production records set in the fourth quarter following the introduction of concentrate bagging in 2009 (Lead*FX* 2014).

In 2014, 1,440,000 t of ore was processed at an average head grade of 7.0% Pb producing 80,900 t of contained lead in concentrate, with an overall plant recovery of 79.3%.

In 2015, prior to the Mine entering care and maintenance, 171,200 t of ore were processed at an average head grade of approximately 7.4% lead yielding 14,000 t of concentrate containing 9,900 t of contained lead.

No ore was processed in 2016 or 2017 due to the Mine being in care and maintenance.

Table 18: Production 2013 to 2015

Production Physicals	Unit	Period			
		2013 Apr - Dec	2014 Jan - Dec	2015 Jan - Feb	2013 - 2015 Total
Ore milled	tonnes	835,900	1,440,000	171,200	2,447,100
Head grade	%	7.1	7.0	7.4	7.1
Recovery	%	74.6	79.3	77.3	77.6
Concentrate produced	tonnes	68,000	120,000	14,000	202,000
Con grade	%	65	67.4	70.4	66.8
Con Pb content	tonnes	44,000	80,900	9,900	134,800

Source: Lead*FX* – Various historic documents

On December 23, 2014, the Company announced that the decline in the London Metals Exchange (“LME”) lead price to levels not seen since mid-August 2012 was a significant factor affecting profitability and cash flow from operations and that, in line with a general downturn in commodity prices, Lead*FX* was experiencing a drop in realised sales prices for lead concentrate.

On January 16, 2015 Lead*FX* further announced that it would wind down the operations to care-and-maintenance. Milling continued until January 31, 2015 and the processing plant and the mine moved to full care-and-maintenance status during early February 2015.

7 Geological Setting & Mineralisation

McQuitty and Pascoe (1998) first described the geology of the Magellan lead carbonate deposit. Updated detailed geology and stratigraphy were produced by Elliott, *et al.* in an unpublished Ivernia Feasibility Report Update (2003). A description of the geology and geological setting was published in the Geological Survey of Western Australia ("GSWA") Record 2009/4 (Pirajno and Burlow, 2009) and with a proposed genetic model in the journal, Ore Geology Reviews (Pirajno *et al.,* 2010).

7.1 Regional Geology

The Mine deposits are situated in outlier rocks of the Earaheedy Group (Earaheedy Basin) overlying the southeastern corner of the Paleoproterozoic Yerrida Basin, at the northern margin of the Archean Yilgarn Craton (Figure 11). The Yerrida Basin is one of several Proterozoic basins between the Pilbara and Yilgarn cratons (McQuitty and Pascoe, 1998).

Hooper (2010) notes that the Yerrida Basin is a part of the Capricorn Orogen, a zone of low- to high-grade metamorphic rocks, magmatic belts, and low-grade volcanosedimentary basins that were formed as a result of an oblique collision between the Pilbara and Yilgarn cratons about 1.8 Giga annum (billion years) ("Ga"). It was probably formed at approximately 2.2 Ga and was affected by the Capricorn Orogeny. The Yerrida Basin has a faulted contact with the Bryah Basin in the west (Goodin Fault) and the Marymia Inlier in the north, and is unconformably overlain by rocks or the Earaheedy Basin in the east (Hooper, 2010).

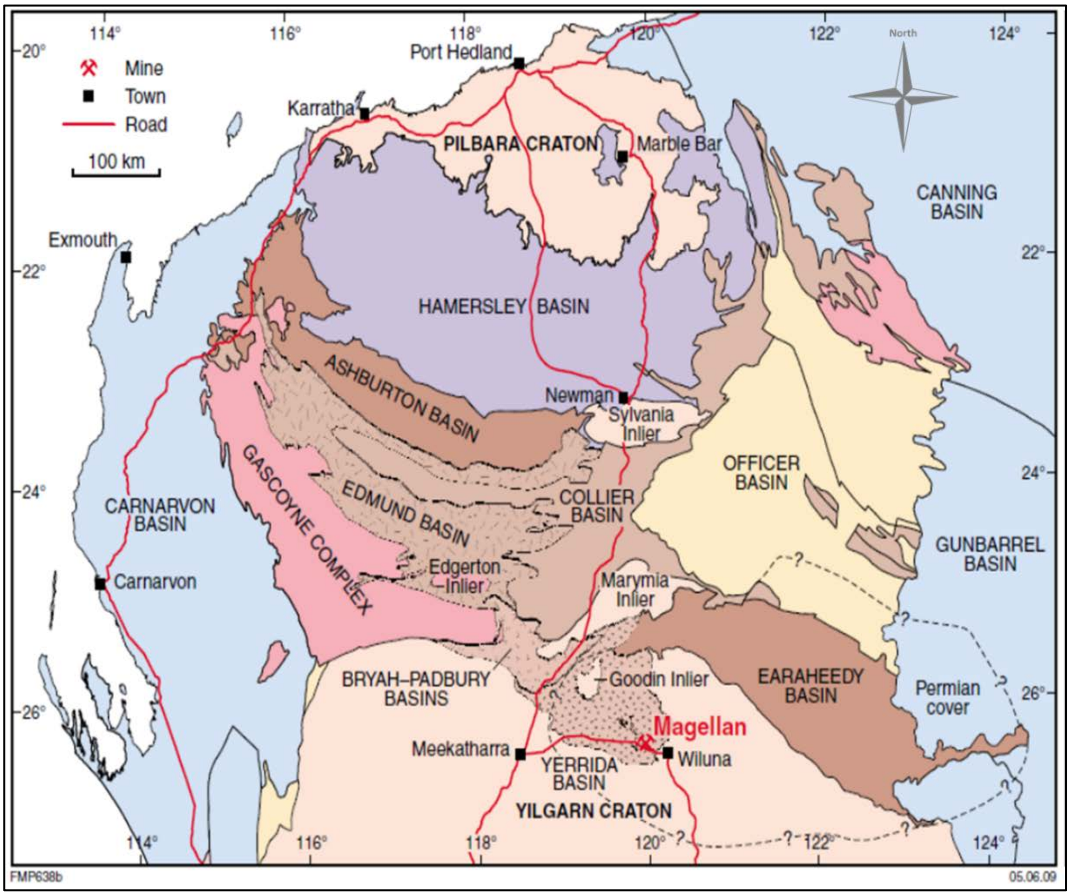


Figure 11: Regional geological setting of Magellan Pb deposit

Source: Pirajno and Burlow (2009)

Pirajno *et al.,* (2010) note that the <1.84 Ga Earaheedy Basin (Figure 11) lies at the eastern end of the Capricorn Orogen and unconformably overlies rocks of the Yilgarn Craton, the Yerrida Basin and possibly the Bryah Basin. Scattered outliers indicate that the basin originally extended much further to the south east and south west, and to the north and north east beneath the later Proterozoic Collier and Officer basins (dashed outline in Figure 11).

The stratigraphy of the Yerrida and Earaheedy Basins is presented in (Figure 12). Within the Yerrida Basin, the Mooloogool Group overlies the Windplain Group and contains the Thaduna, Doolgunna, Killara, and Maraloou formations, which were deposited in a high-energy environment, probably in a widening rift structure, surrounded by uplifted Archean rocks of the Marymia and Goodin Inliers (Hooper, 2010).

The underlying Windplain Group contains the Juderina and Johnson Cairn formations, which include siliciclastic rocks, evaporates, argillites and locally turbidites, with the depositional environment thought to be a shallow epicontinental sea, locally with sabkha environments (Hooper, 2008).

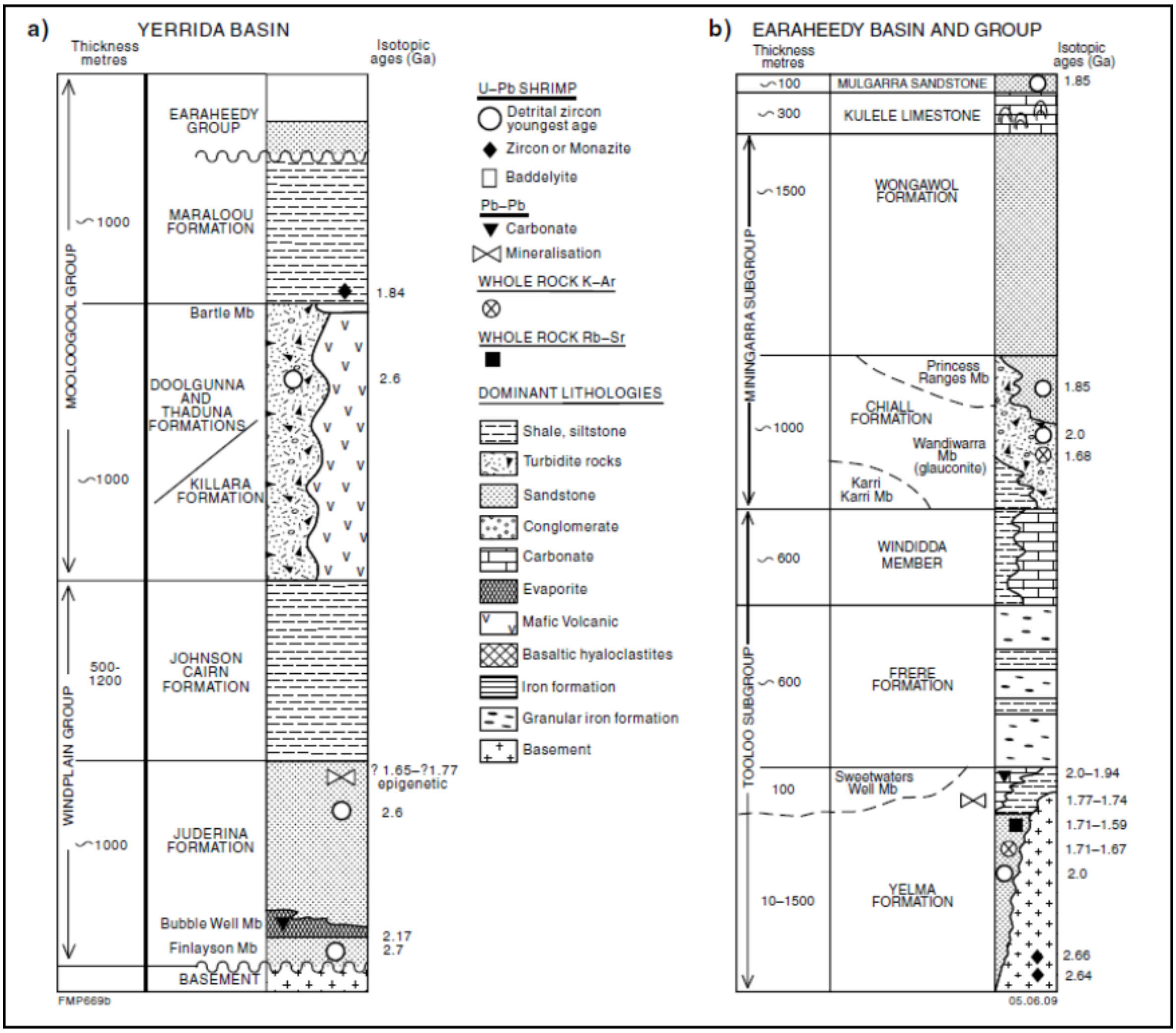


Figure 12: Stratigraphy and geochronology of Yerrida Basin (a) and Earaheedy (b) Basin

Source: Pirajno and Burlow (2009)

Pirajno and Burlow (2009) note that the Earaheedy Basin contains the Earaheedy Group (Figure 12), which they describe as "*a 5 km-thick succession of shallow marine clastic and chemical sedimentary rocks that are unconformable on the Yilgarn Craton and the ca. 1.84 Ga Mooloogool Group (Yerrida Basin).*"

The Earaheedy Group is made up of the (lower) Tooloo Subgroup and the (upper) Miningarra Subgroup (Figure 12). Pirajno *et al.,* (2010) note that the Tooloo Subgroup consists of the basal Yelma

Formation (sandstone, siltstone and stromatolitic carbonates) overlain successively by the Frere Formation (Lake Superior-type granular iron formation and shale), and the Windidda Member (iron-rich shale and carbonates).

The overlying Miningarra Subgroup (in ascending order) consists of the Chiall Formation (silty and sandy mature clastic units, commonly glauconitic), Wongawol Formation (fine-grained clastic and carbonate rocks), Kulele Limestone, and Mulgarra Sandstone (Pirajno and Burlow, 2009).

7.2 Local Geology

The Mine deposit occurs at the base of the Earaheedy Group, overlying the Mooloogool Group, with similar style mineralisation located at smaller prospects that lie south and south west of the Mine, mainly along the unconformity surface between the Juderina Formation (Windplain Group) and small outliers of the Earaheedy Group (Figure 13).

The Yerrida Group is represented by two formations within the Mine area, namely the lowermost Juderina Formation (Finlayson and Bubble Well Members) and the unconformably overlying Maraloou Formation (carbonaceous shale). Yelma Formation sandstone and carbonate of the Earaheedy Group unconformably overlie the Yerrida Group in the Mine area (Hooper, 2010).

The Finlayson Member consists of a thin (<100 m) and widespread basal quartz arenite unit, which commonly displays herringbone and trough cross-bedding and multi-directional ripple marks. The Finlayson Member is overlain by and/or intercalated with chertified stromatolitic carbonate and evaporitic sedimentary units of the Bubble Well Member (Hooper, 2010). Sediments of the Windplain Group are exposed approximately 10 km south of the Wiluna–Meekatharra road as a prominent E–W–trending ridge (Finlayson Range).

Unconformably overlying the Juderina Formation in the Mine area is the Maraloou Formation of the Mooloogool Group, which consists of carbonaceous shale, finely laminated siltstone, argillaceous dolomitic limestone and interbedded siltstone with thin beds of limestone and dolomite (Hooper, 2010).

Exposure of the shale and siltstone is poor due to preferential weathering (Hooper, 2010), and much of the unit surrounding the Mine area is covered by alluvial plain. Dolerite sills of the Killara Formation (~0 – 700 m thick) intrude the Maraloou Formation to the north west of the area (Hooper, 2010), but are not recorded within the Magellan Hill deposit sequence (Burlow, 2015).

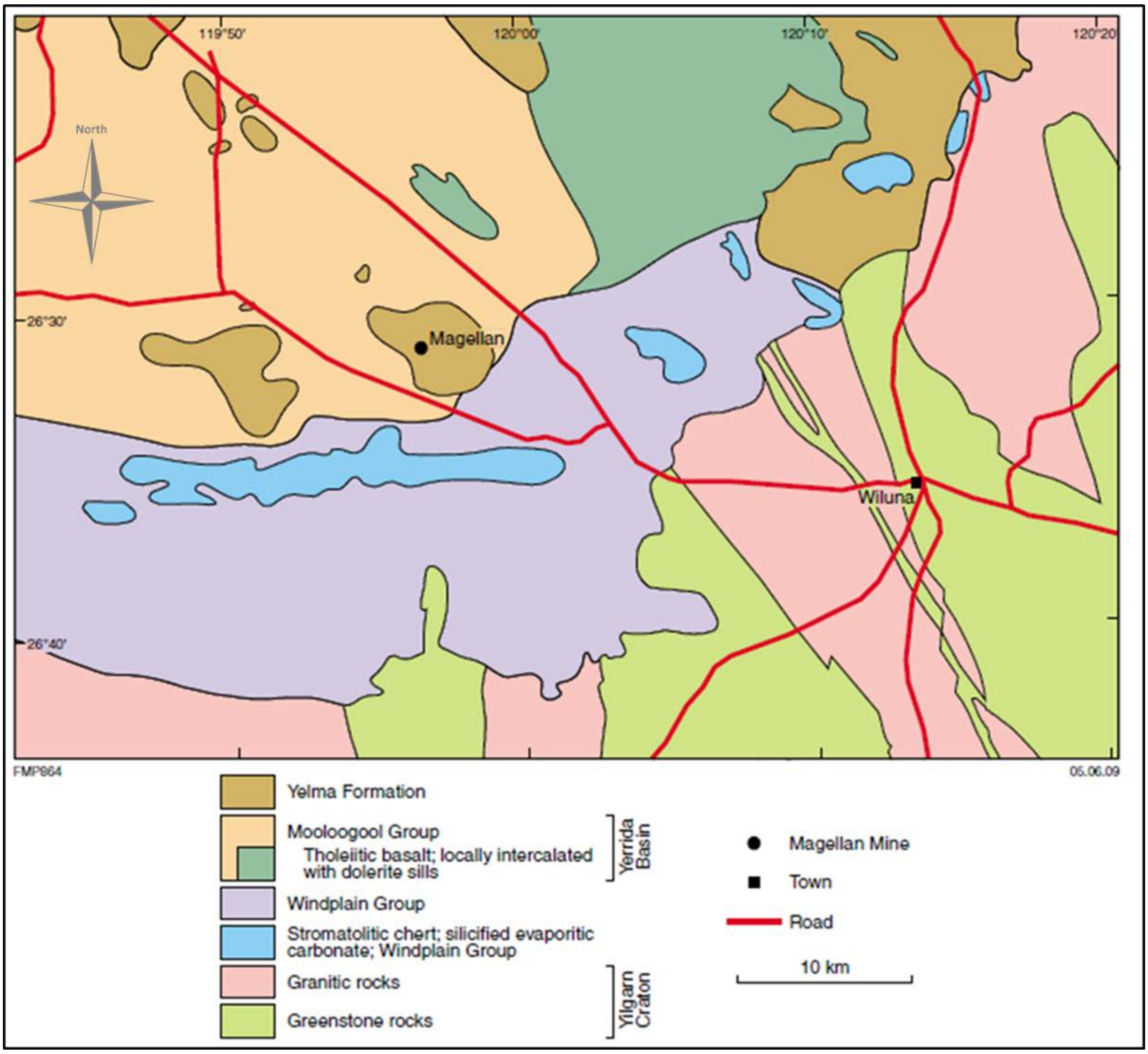


Figure 13: Schematic geological map of the southeastern part of the Yerrida Basin

Source: Pirajno and Burlow (2009)

The Yerrida sediments are commonly flat-lying to moderately dipping to the north and west, and the dominant structural feature is NE- and SE- trending faults. Folding is very gentle, and where described comprises N–NW and NE open folds. The Earaheedy sediments appear to have undergone relatively minor structural deformation (Hooper, 2010). The underlying basement contains major structures orientated N–S, NNW–SSE and E–W and these are likely to have played a major role in controlling basin structure and the location of primary mineralisation (Looi, 2010).

7.3 Property Geology

The Mine includes six non-sulphide lead deposits being Magellan, Cano, Pinzon and Gama (now a subset of the Magellan pit), within the Magellan Hill area, and Pizarro and Drake within the Finlayson Range area. The mineralisation is mainly hosted by Paleoproterozoic (1.8 Ga) quartz-clay breccia, clay, siltstone and sandstone units of the Yelma Formation, although mineralisation occurs in sediments of both the Yelma Formation and the underlying Juderina Formation in the Pizarro and Drake deposits of the Finlayson Range, (Looi, 2010).

The Yelma Formation outcrops as a mesa of approximately 5 x 2.5 km in the Magellan Hill area, and is raised above the surrounding alluvial plain by 25 – 50 m. The surface of the mesa is covered by weathered material from the underlying quartz clay breccia, mainly silcrete, quartz and chert colluvium and scree. Erosion along the flanks of the mesa and within gullies has removed the upper units and exposed the shallowly-dipping sandstone sequence (Looi, 2010).

The Maraloou Formation sediments underlying the Magellan Hill mesa are exposed in road cuttings (Mine access road) and in incised gullies at the foot of the Pinzon south eastern breakaway slope.

The southern breakaway margins of the Cano, Magellan and Pinzon deposits show a well-developed (natural) secondary dispersion Pb geochemical anomaly. The magnitude of the Pb anomaly is greatest where mineralisation approaches or intersects the surface.

Along the western flank of Magellan and the southern margins of the Pinzon deposits, this gives rise to common distinct vegetative anomalies where the ubiquitous mulga shrub land degrades suddenly to patches of ephemeral spinifex grass. These areas often display Pb in soil values exceeding 2% Pb, restricting the long-term growth of the long-lived mulga in favour of the shorter-lived spinifex (Burlow 2015, Elliott 2015).

The soil Pb anomalism tapers gradually down slope towards the shallow alluvial plain at the foot of the mesa and can be seen to swing around to the south east, influenced by the seasonal sheetwash and West Creek drainage towards Lake Way.

Minor, surficial Pb in soil anomalism can be found fixed in patches of calcrete formation along the southern West Creek drainage south of the Magellan mesa. A map showing a compilation of surface Pb in soils anomalism is shown at Figure 14.

The satellite lead deposits at Pizarro and Drake show similar, though less well-developed, dispersion anomalies.

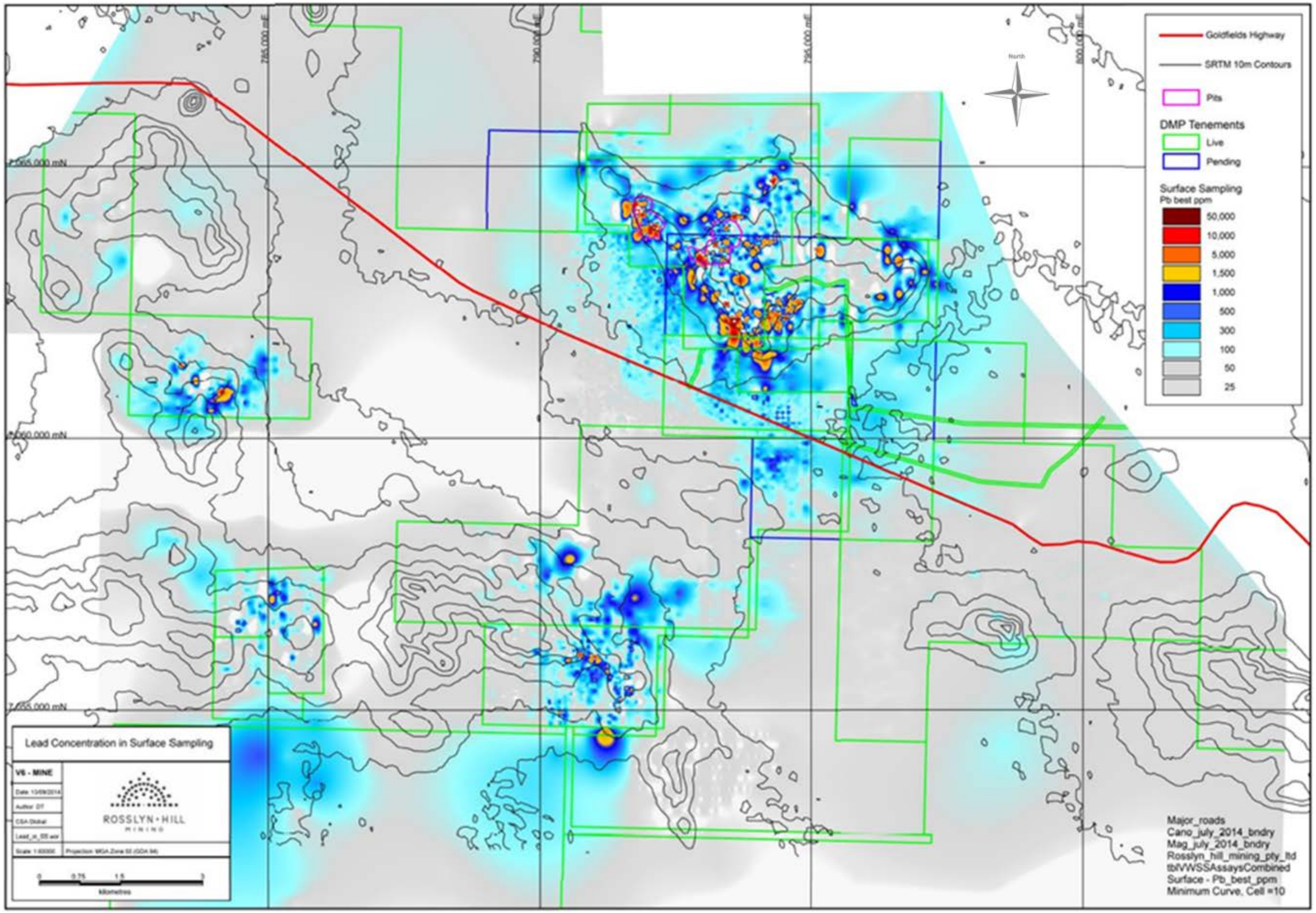


Figure 14: Map of naturally occurring Pb in soil anomalism compiled from portable XRF data and from surface (0 - 1 m) RC/RAB drill assays

Source: Burlow and Corry (2014)

A generalised description of the stratigraphy of the Magellan Hill area is provided in Figure 15 and indicates the units that the Yelma Formation has been divided into for mapping purposes on the mine.

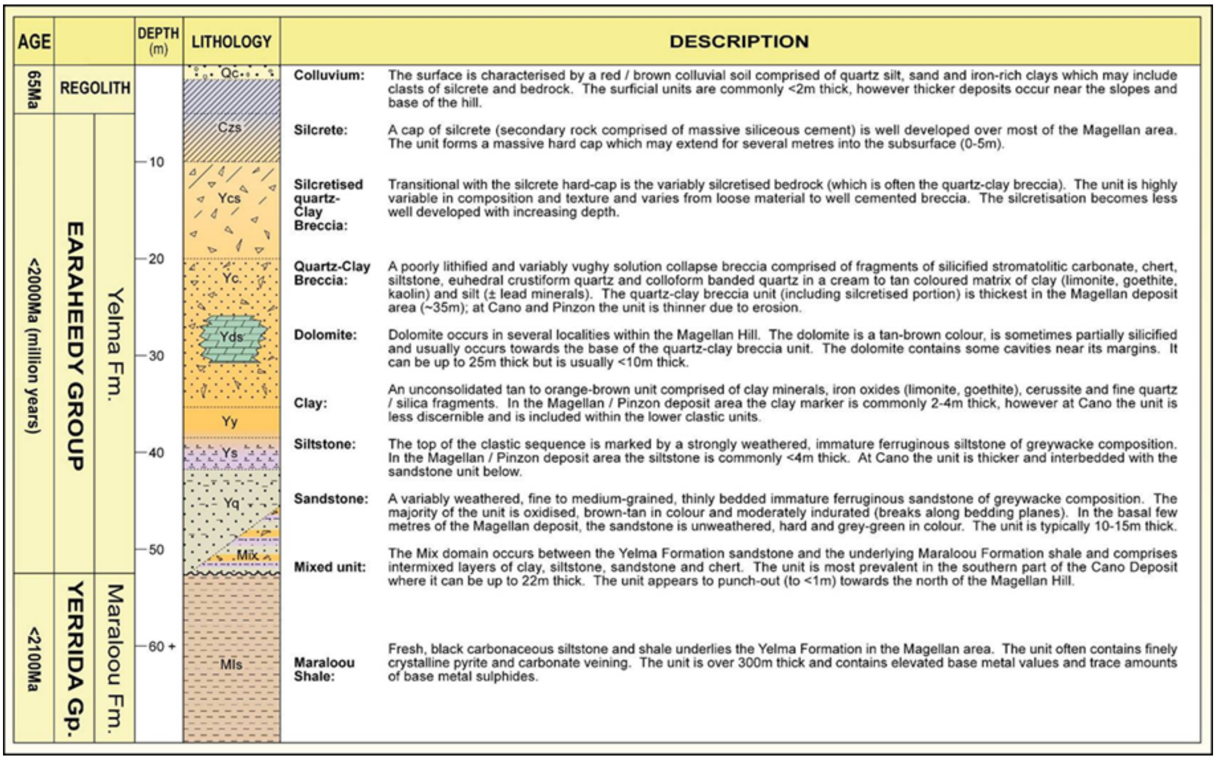


Figure 15: Generalised stratigraphy of the Magellan Hill area

Source: Looi (2010) after Elliott *et al.,* (2003)

Figure 16 depicts the mine stratigraphy in some detail, and divides the mine into the following components with increasing depth from surface:

- laterite cap
- quartz-clay breccia
- saprolitic clay zone
- saprock siltstone and sandstone; and
- Maraloou Formation.

The breccia unit is interpreted as the highly altered and weathered Sweetwaters Well Member of the Yelma Formation (Pirajno and Burlow, 2010).

The complex overprinted effects of sedimentary facies variation, gentle interference folding, mineralisation, deep oxidation and erosion mean not all units are present or identifiable in all locations across the Magellan Hill with clear and recognisable boundaries. At the Cano and Magellan deposits' southwestern margins, the upper silcretised and quartz-clay breccia units are locally absent due to erosion, while significant volumes of stromatolitic dolomite 'intrude' the clay-quartz breccia unit throughout the east of Magellan.

The simplified mine sequence legend is highly subjective, relying on geologists' interpretation rather than description, but greatly simplifies geological mapping and logging tasks during drilling campaigns.

The Pizarro deposit and nearby Columbus prospect occur within the Finlayson Range, a prominent E–W-trending series of hills comprised of siliceous rocks of the Juderina Formation. Small areas of subcropping Yelma Formation quartz clay breccia occur in the Pizarro area, although much of the area is covered by loamy colluvium deposits (Looi, 2010).

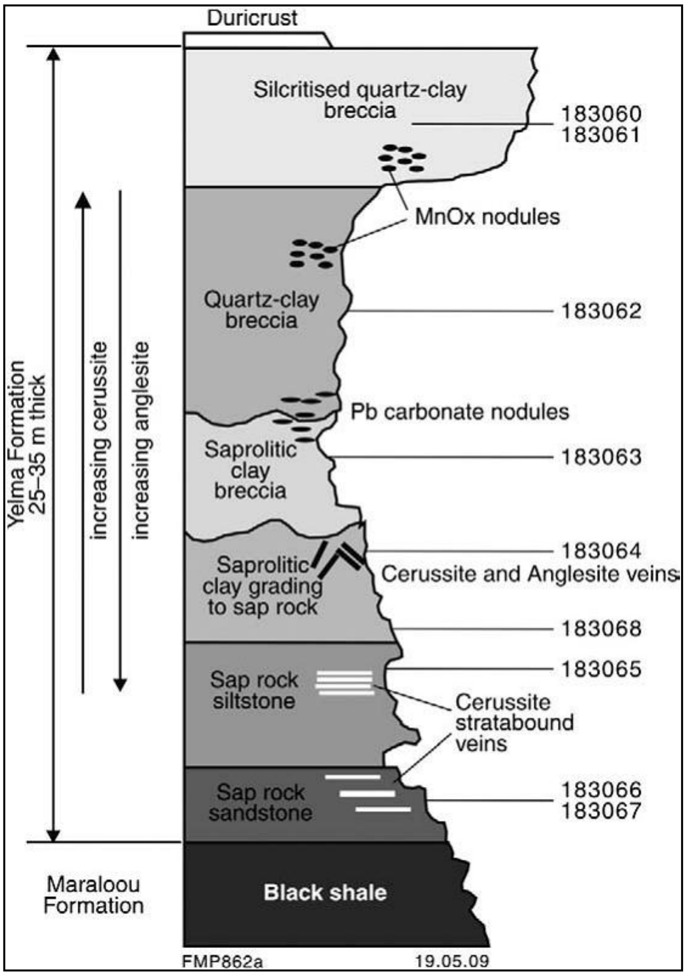


Figure 16: Simplified mine stratigraphic sequence

Source: Pirajno and Burlow (2009)

7.4 Significant Mineralised Zones

Mineralisation is located in five defined deposits and a number of outlying prospects. The Gama deposit is now included with Magellan as recent drilling in 2014 has confirmed that they are actually components of the same deposit.

The deposits are listed below in general order of size and shown in Figure 17:

Magellan Hill (main deposits under development):

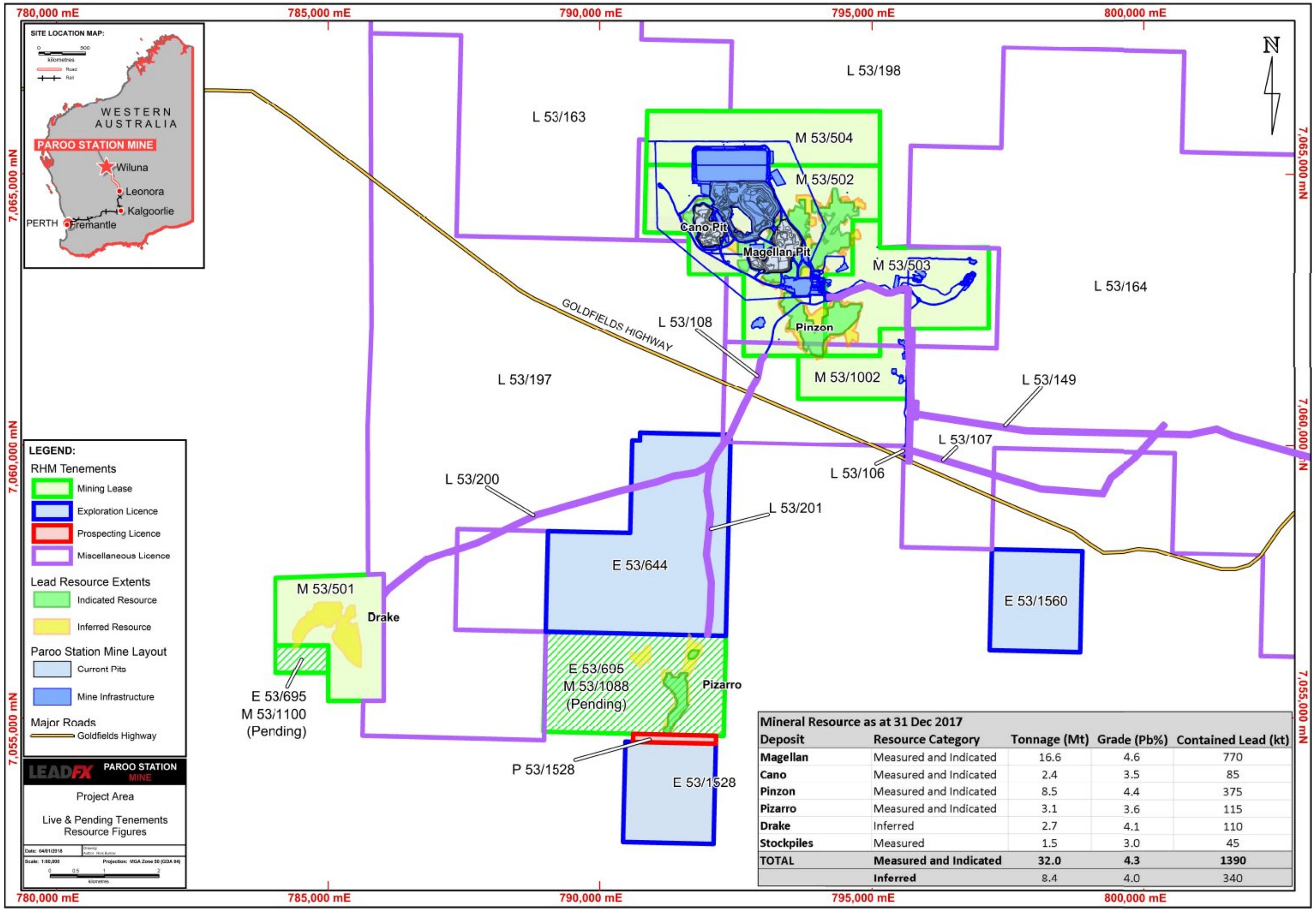
- Magellan (now includes Gama)
- Cano
- Pinzon

Finlayson Range deposits (located approximately 10 km south of the Magellan Hill group):

- Pizarro
- Drake

Finlayson Range prospects (in the exploration phase):

- Columbus (not shown in Figure 17)



Mineral Resource as at 31 Dec 2017				
Deposit	**Resource Category**	**Tonnage (Mt)**	**Grade (Pb%)**	**Contained Lead (kt)**
Magellan	Measured and Indicated	16.6	4.6	770
Cano	Measured and Indicated	2.4	3.5	85
Pinzon	Measured and Indicated	8.5	4.4	375
Pizarro	Measured and Indicated	3.1	3.6	115
Drake	Inferred	2.7	4.1	110
Stockpiles	Measured	1.5	3.0	45
TOTAL	**Measured and Indicated**	**32.0**	**4.3**	**1390**
	Inferred	8.4	4.0	340

Figure 17: The Mine Deposits

Source: Lead*FX* (2018)

Pirajno and Burlow (2009) refer to the Magellan deposit as a large stratabound lead deposit, and describe it as unusual. The Magellan Hill mineralisation is accompanied by silicification, argillic (illite, kaolinite) and sericitic alteration of the host sandstone and stromatolitic dolomite of the Yelma Formation and is located close to, or at the disconformable contact with, the underlying Maraloou Formation (Pirajno *et al.,* 2010).

The orebody at the Mine is contained in a mesa outcrop 5 x 2.5 km, comprising the Yelma Formation which hosts the lead mineralisation, the majority of which is contained within a quartz clay breccia up to 35 m thick. The mineralised unit is described as an upper quartz clay breccia with fragments of completely silicified carbonate with relict stromatolitic structures, siltstone, and euhedral and coliform banded quartz in a white clay-rich matrix (up to 35 m thick) (Sibbel 2009).

7.4.1 Magellan Hill

The Magellan deposit extends for approximately 1,200 m in a northerly direction with an average width of approximately 650 m and an average vertical thickness of economic mineralisation of approximately 12 m (Figure 18).

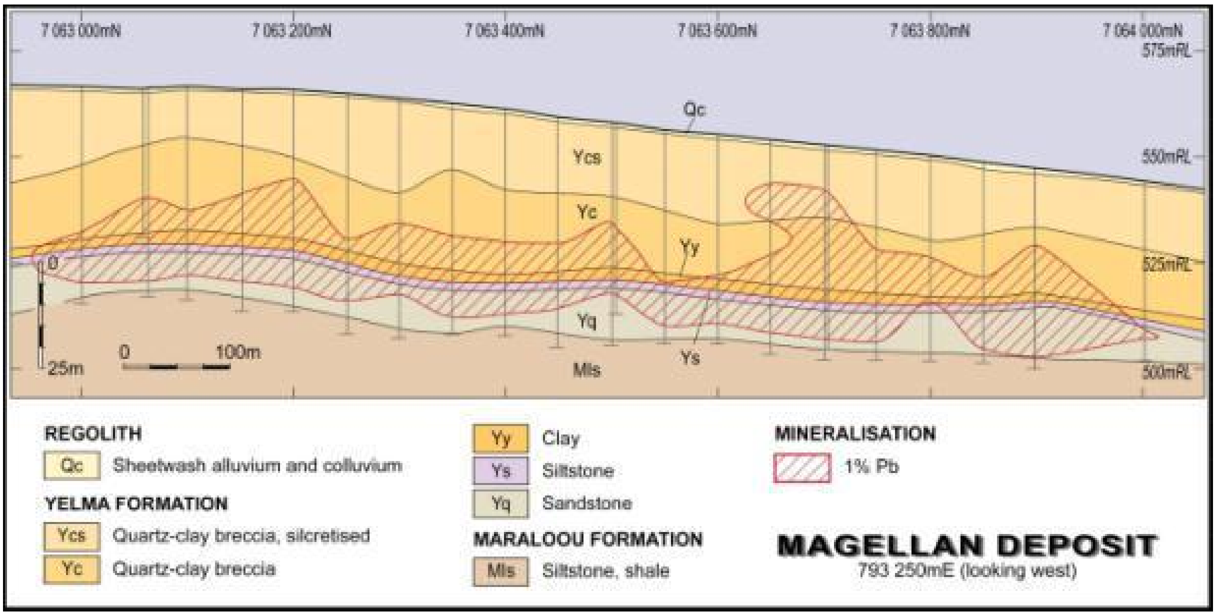


Figure 18: Schematic drill hole section through thicker part of Magellan deposit

Source: Lead*FX* (2011)

The Cano deposit lies along a north-west axis, extending for approximately 800 m with an average width of 400 m and an average vertical thickness of approximately 7 m.

The Pinzon deposit (Figure 19) comprises two zones of mineralisation, one trending in an N–NW direction and the second on a north-east trend. Both zones are approximately 1,000 m long by 200 m wide with an average vertical thickness of 5 m.

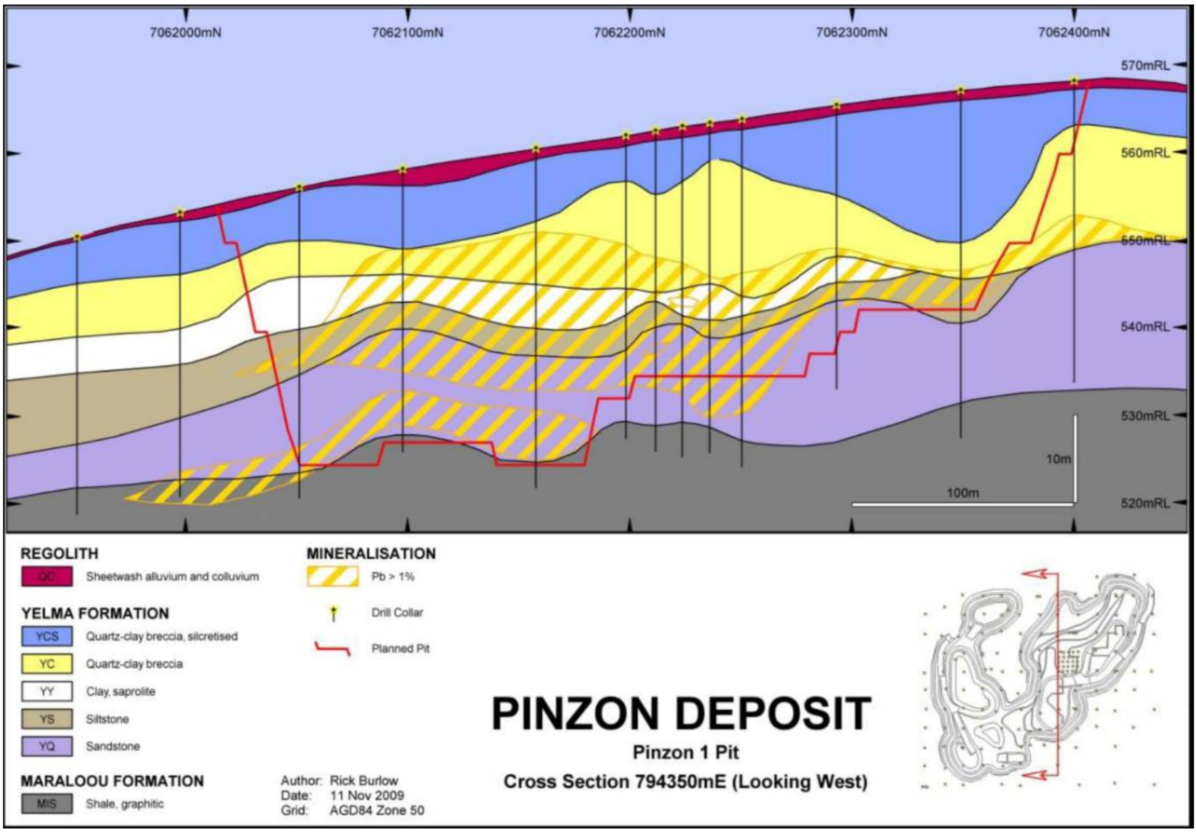


Figure 19: Cross section through the Pinzon deposit

Source: Lead*FX* (2011)

The Gama deposit has now been shown to coalesce with the eastern flank of Magellan and further extends for 1,200 m in a north easterly direction with an average width of 300 m and an average thickness of approximately 5 m.

Mineralisation at Magellan and Cano is consistent and continuous, compared to the mineralisation at Pinzon which displays more variability, with mineralisation presenting as semi-continuous higher-grade elongate bodies within a lower grade halo (Figure 19).

7.4.2 Finlayson Range Deposits / Prospects

Mineralisation at Pizarro and Drake is similar to that described above; however, lateral extents to these deposits are currently restricted to less than 500 m.

7.4.3 Ore Mineralogy

The lead mineralisation occurs predominantly as cerussite ($PbCO_3$) with lesser anglesite ($PbSO_4$). Minor amounts of other lead minerals such as pyromorphite ($Pb_5(PO_4)_3Cl$), coronadite ($(Pb,H_20)_2Mn_5O_{10}$), plattnerite (PbO_2) and plumbogummite ($PbAl_3(PO_4)_2(OH)_5\bullet(H_2O)$) are known from petrographic analysis of ore, concentrate and tailings samples. Minor amounts of sphalerite (ZnS) and galena (PbS) occur in the underlying Maraloou Formation.

The mineralisation is typically very fine grained and indistinctive with very little or no visible recognition in hand specimen or at mine-scale in the open pits.

It occurs as replacement of the host rocks forming relatively flat-lying, continuous sheets

(Figure 18). Other forms of mineralisation identified include breccia, vein, concretionary, nodular and coarse crystalline (Looi, 2010).

7.4.4 Grade Distribution

The major host to the mineralisation is the lower part of the Yelma Formation quartz-clay breccia unit (“Yc”) and underlying Yelma Formation clay (“Yy”) unit. The quartz-clay breccia is the residue of a mixed carbonate sequence and is silica rich (quartz crystals, chert and silcrete) in the upper portions, tending towards clay-rich in the lower portions.

The lower clay unit Yy has historically been distinguished during RC drilling on its physical attributes; however, it is likely to be the lower part of the Yc unit and/or the top of the strongly weathered Yelma Formation siltstone (“Ys”) unit. The underlying siltstone and Yelma Formation sandstone (“Yq”) sequence can also be a significant host to mineralisation, especially when deeply weathered as is the case with the Cano and Pinzon deposits.

Ore zones can have both gradational and sharp grade boundaries (Figure 18 and Figure 19). The highest grade areas are often concentrated in the clay-rich or strongly weathered units.

Mineralisation is typically weaker within the upper silcretised portion and in areas where the effects of weathering are minor and is confirmation of significant supergene processes.

Mineralisation in the Maraloou Formation is generally not present in economic grades as the appearance of this unit appears to indicate the effective limits of economic mineralisation to the deposits.

High-grade zones throughout the deposits are generally thought to reflect the position of relict primary mineralised structures; however, hydromorphic (porosity and permeability) and geochemical factors are likely to be important controls on grade distribution (Looi, 2010).

Some zonation of ore minerals has been identified within the Magellan deposit and observations from drill samples, mineral identification studies and geochemical data suggest that anglesite is more prevalent in the upper parts of the deposit.

Coronadite is also more prevalent in the upper parts of the deposits and it is commonly associated with the upper contact or exposed mineralisation. Pyromorphite (as veins and needle-like clusters) has only been identified within the lower clastic sequence.

8 Deposit Type

The Mine's lead deposit most likely represents the final weathered remnant of a wallrock replacement-type non-sulphide zinc-lead deposit. McQuitty and Pascoe (1998) first described the Magellan deposit, with further characterisation being made during later exploration and mining campaigns.

8.1 Mineral Deposit

The Mine's lead deposits are unusual for base metal mineralisation, owing to its almost complete lack of economic metals, other than lead. The mineralisation displays very low zinc grades that are generally less than 500 ppm Zn.

The Mine's lead deposits are almost entirely sulphide free, consisting only of carbonate and oxide lead mineral species, and as such falls into the category of non-sulphide ore systems as defined by Hitzman et al., (2003). Some extremely minor relic sulphide (hand specimen size) was discovered in 2013, protected from oxidation by a silica-rich rind and has not been encountered since.

The Mine's deposits likely represent a new category within the class of supergene non-sulphide mineral systems. There is no known analogue of the Mine's deposits, but they show a strong similarity to non-sulphide zinc deposits, of which there are several examples worldwide (Figure 20).

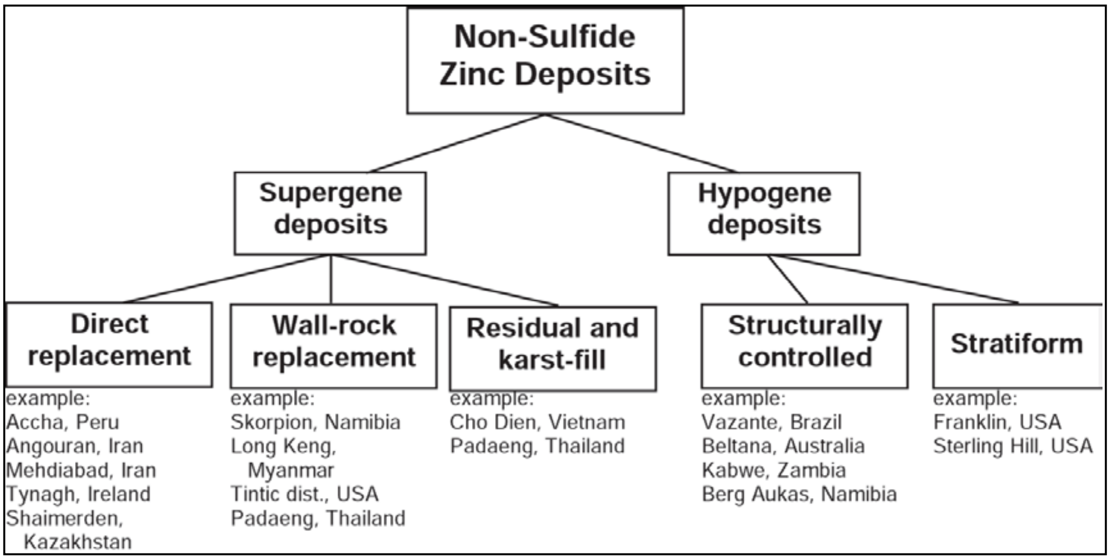


Figure 20: Classification of non-sulfide zinc deposits

Source: Hitzman *et al.,* (2003)

Supergene non-sulfide zinc deposits, which are generated via oxidation of sulphide and non-sulphide zinc deposits, are the most common type of non-sulphide zinc deposits and have a worldwide distribution (Figure 21).

Most supergene non-sulphide zinc deposits occur in carbonate host rocks owing to the high reactivity of carbonate minerals with acidic, oxidised, zinc-rich fluids derived from the breakdown of sphalerite-rich bodies. The majority of supergene deposits have either a Mississippi Valley type – ("MVT") or a high-temperature, carbonate replacement-type sulphide progenitor, although supergene deposits may form from a variety of sphalerite-rich deposits. These sulphide progenitors often contain significant quantities of lead in the form of galena lead sulphide ("PbS").

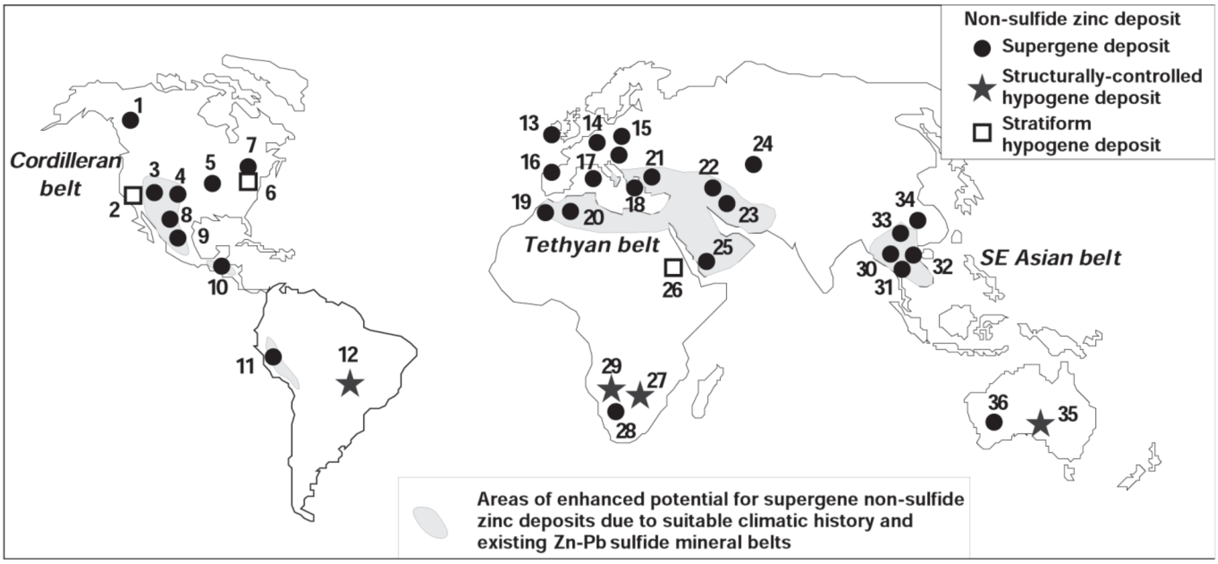


Figure 21: Global distribution of non-sulphide zinc-lead deposits

Source: Hitzman *et al.,* (2003) Note: Magellan is number 36

The Mine deposits appear to fall within the wallrock-replacement grouping of supergene deposits. Supergene wallrock replacement zinc deposits form adjacent to, and down groundwater flow gradient from, the original sulphide body and related direct-replacement deposits (Figure 22b) and as sulphide bodies are progressively oxidised, acidic groundwater containing zinc migrates out into the calcareous wallrock where it reacts and precipitates zinc carbonates (Figure 22c).

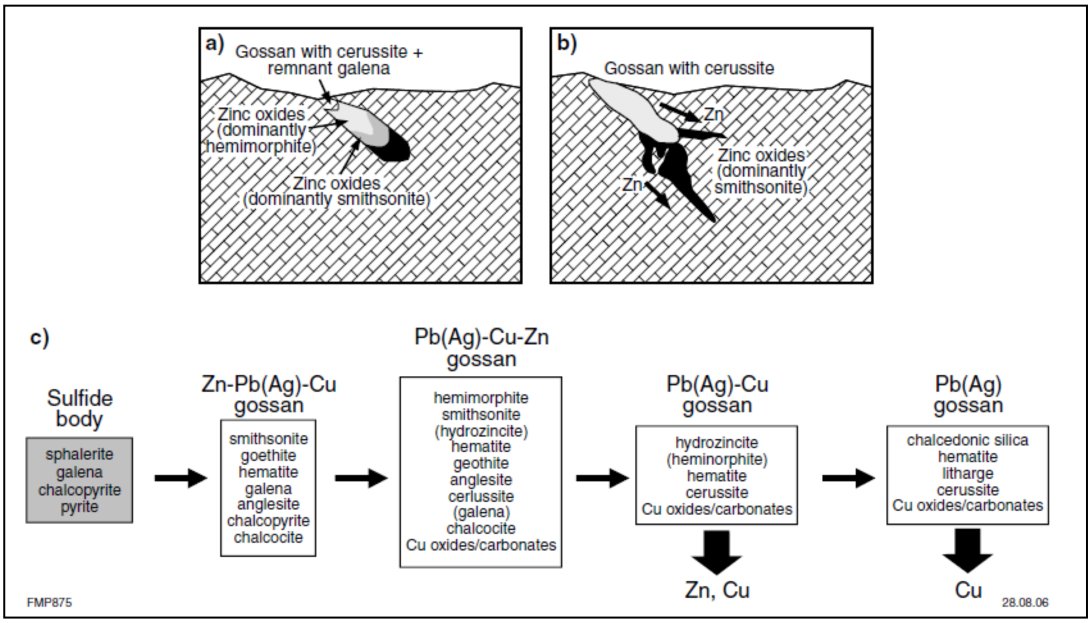


Figure 22: Genetic models for the formation of non-sulphide minerals systems: a) direct replacement type; b) wallrock replacement type (applicable to the Mine); c) mineralogical changes related to progressive replacement of sulphides

Source: Pirajno and Burlow (2009), Hitzman *et al.,* (2003)

In areas of deep, mature weathering, residual lead deposits with a silica-clay gangue may form by reduction of the land surface and essentially complete removal of zinc from the system, and the cerussite-anglesite mineralisation in the Mine's deposits could be an example of this process (Hitzman *et al.,* 2003).

Zinc and other metals such as silver may have been mobilised by groundwater interactions to such an extent that they are no longer present within the deposits, leading to the stable, oxidised lead minerals remaining as the major species.

Constant top-down flushing of the deposit by meteoric waters containing dissolved carbon dioxide (“CO_2”) may have evolved anglesite-rich mineralisation to a more cerussite-dominant assemblage, assisting the remobilisation of upper mineralisation downwards towards the favourable clay-rich portions of the quartz-clay breccia and clay units while depleting the upper, silcretised breccia unit.

No weathered, altered sulfide or relic sulphide textures were observed during early exploration or mining of the Magellan and Cano deposits (Lead*FX*, 2011). In late 2013, a small (~10 cm) specimen of relic galena was discovered by CSA and RHM geologists during mining of the lower Magellan mineralised horizon. The sulphide, preserved with a rind of carbonate inside a crystalline and chalcedonic quartz vugh immediately proved the presence of at least small quantities of primary sulphide mineralisation. The flat-lying, low-deformation position of the sulfidic precursor deposit at the Mine, combined with prolonged weathering at or just above the groundwater table may have contributed to the near-perfect conversion of sulphide galena to carbonate and other oxide species.

8.2 Geological Models and Exploration

The discovery of the Magellan lead deposit in 1991 established the Yelma Formation as a significant host for potential MVT-style mineralisation (McQuitty and Pascoe, 1998).

The Magellan Hill and outlying lead deposits display a characteristic pattern of Pb-in-soil anomalism around marginal breakaway slopes where the hardcap has eroded and portions of the mineralised zone are exposed. Apart from these local situations, the ore-grade mineralisation is “blind” with limited surface physical or geochemical expression. Gravity survey data shows a weak correlation between mineralisation and local gravity lows, from a likely mass removal event during brecciation of the mineralised sequence but is considered a poor predictor of lead accumulations.

Exploration across the local tenements since discovery has focused on identification of similar remnant Yelma (and Juderina) Formation outliers as exploration targets. Coverage by conventional and portable X-ray fluorescence (“XRF”) soil geochemical surveys has accompanied wide-spaced, shallow RAB and RC drilling and led to the discovery of the Cano and Pinzon deposits on the Magellan Hill and the satellite deposits Pizarro and Drake 10 km to the south and south-west respectively.

9 Exploration

Renison initiated exploration for base metals in the Mine in 1990 and carried out geochemical sampling, mapping, and geophysical survey programs in addition to drilling. Anomalous values of between 0.1% and 3.15% Pb from holes drilled at the southwestern edge of Magellan Hill lead to the discovery of the deposit in June 1991.

The majority of exploration work has been drilling and since discovery, non-drilling exploration has comprised extensive soil geochemical surveys; conventional and portable XRF, detailed ground gravity surveys, aerial photography and photogrammetry, and an aerial TDEM survey.

9.1 Relevant Exploration Work

9.1.1 Soil Geochemical Surveys

Following early geochemical surveys by Renison and CSA, a campaign using portable XRF mineral analyser units was carried out during 2008 and 2009.

Measurements in these later surveys were collected at a spacing of 50 m, along N–S lines spaced 200 m apart. Each sample station had the surface topsoil removed to a depth of 2 – 5 cm so that the instrument could scan the soil surface at each station. A physical soil sample was collected at a frequency of 1: 20 samples to provide a baseline for the survey (Sergeev, 2008). Basic soil type and subcrop / outcrop geology was also noted and a number of rock chip portable XRF measurements taken.

The combined portable XRF survey areas cover almost the entire Magellan Hill, with the exception of existing waste landform and disturbed mine areas (as at 2009). In addition, most of the known outlying lead deposits have been surveyed. The following summarises the sample density across all prospective areas.

- Magellan Hill (Magellan, Cano, Gama and Pinzon area) 1,877 stations
- Drake (Drake deposit) 425 stations
- Pizarro (Pizarro and Columbus prospect areas) 782 stations
- Cortez West[7] (Cortez prospect and North Pizarro area) 610 stations E53/1560 (11.5 km SE of Magellan Hill)

In 2014, all conventional and portable XRF data was merged with surficial RC drilling to produce a high-quality combined dataset (previously presented as Figure 14).

The combined surface geochemical dataset for lead shows a detailed, far-ranging picture of the mine, near-mine and locality scale lead-in-soil anomalism. Importantly, all samples used reflect natural anomalism free of possible surficial mine contamination.

The southern breakaway margins of the Cano, Magellan and Pinzon deposits show a well-developed (natural) secondary dispersion Pb geochemical anomaly and correspond closely with observed anomalous vegetation.

[7] Following RHM Management review in 2015, the portion of the exploration licence that covered the Cortez prospect was relinquished as the mineralisation data collected was unlikely to support a Mineral Resource.

The Pb anomalism displays a strong NW linear trend along the western margin of the Cano deposit that corresponds with large-scale structures observed in the open pits.

Several plumes arising from mechanical transportation down slope from the mesa’s south and western breakaway into the broad West Creek drainage channel can be observed. Several subordinate East-North-East (“ENE”) to North-East (“NE”) alignments also exist and preferential erosion of susceptible strata may be related to structural trends.

The magnitude of the Pb anomaly is greatest where mineralisation approaches or intersects the surface and the resultant dispersion anomaly is weaker and more confined towards the north where the breakaways are poorly developed.

The satellite lead deposits at Pizarro and Drake show similar, though less well developed, dispersion anomalies. An outlier hill east of Pizarro also shows anomalous Pb-in-soil anomalism and represents an exploration drilling target.

9.1.2 Ground Gravity Surveys

A ground gravity survey was carried out in late 2007 with additional infill surveying over areas of interest carried out in early 2008 (Sergeev, 2008). Station spacings range from 50 metres north (“mN”) x 50 metres east (“mE”) over the Magellan deposit, to 50 mN x 200 mE at the other deposits.

Gravity measurements were collected using Scintrex CG3 Autograv instruments, with carrier phase global positioning system (“GPS”) data collected using Trimble 4000 series geodetic receivers (Hooper, 2009). The Bouger anomaly processing was carried out by Fugro Surveys using a country rock density of 2.67 g/cm^3.

The processed results of the survey are presented in Figure 23. Apparent gravity lows associated with the Magellan and Cano deposits are less well defined than previously suggested and the lack of associated gravity lows with the other known deposits (e.g. Drake, Pizarro, Pinzon) implies that the deposits cannot be directly detected from gravity data alone. However, the high-resolution gravity data does enable the identification of many structural features that appear to be related to mineralisation.

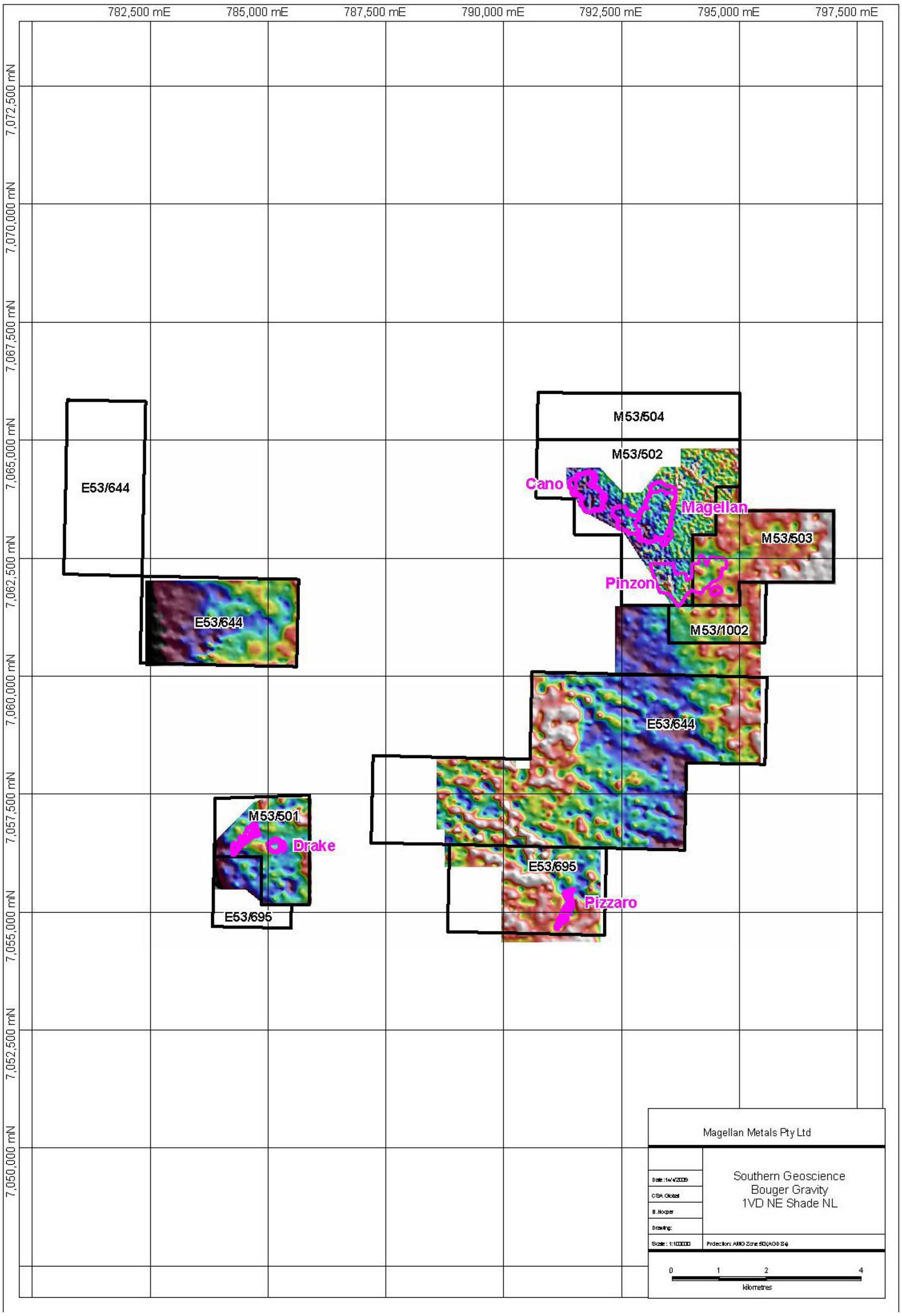


Figure 23: Bouguer anomaly 1st vertical derivative from merged gravity data; levelled and processed with outlined major lead deposits

Source: Sergeev (2008)

9.1.3 Aerial Photography and Photogrammetry

The most recent satellite imagery and airborne photography which is documented in the previous Technical Report (SRK, 2015) consisted of the following:

- February 2012 – Geo-Eye-1 collection of satellite imagery data by AAM Pty Ltd
- May 2014 – detailed aerial photographic dataset by Fugro Spatial Solutions.

All aerial photography including the 2012 and 2014 datasets is available to RHM geologists as digital colour photographic plates, a combined ortho-rectified image for use in Global Information System ("GIS") applications in GeoTIFF and ECW, and ancillary data such as detailed aeromagnetic, radiometric and altitudinal data.

The 2014 Fugro altitudinal data was processed into a detailed digital terrain model ("DTM") and contour set. The DTM model is shown in Figure 24.

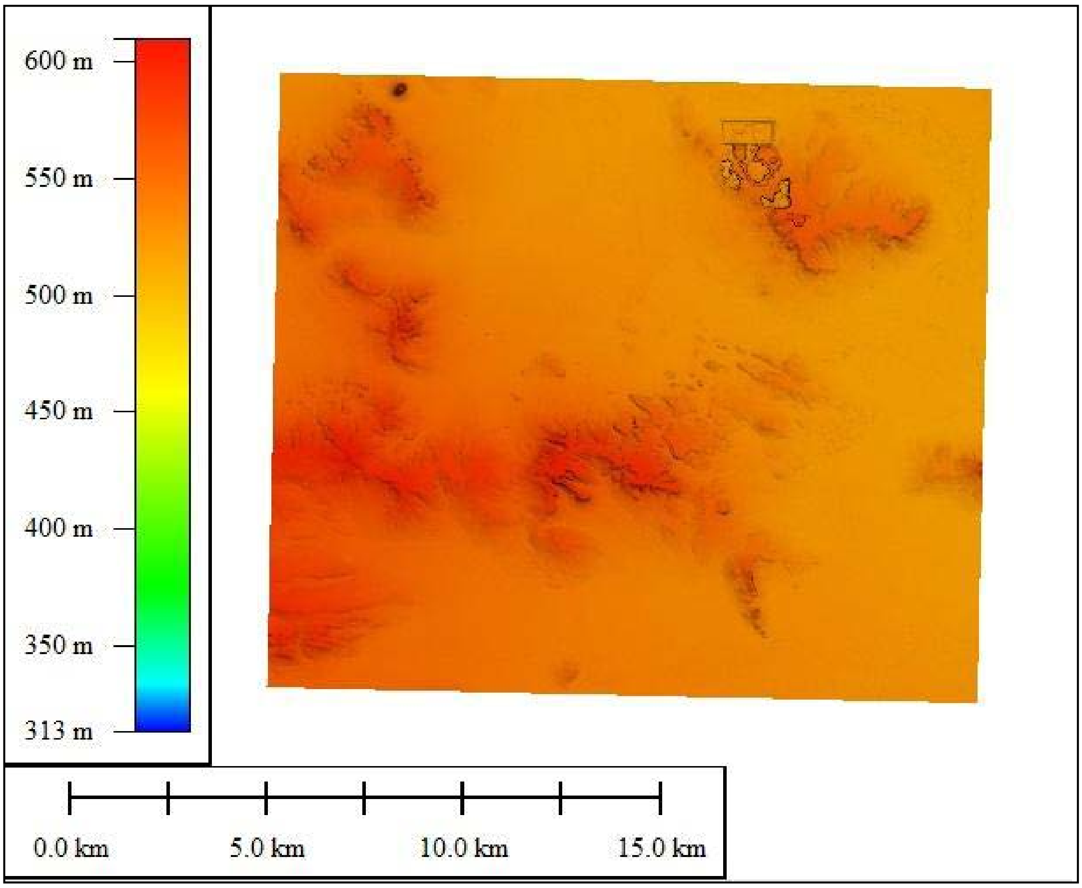


Figure 24: Digital terrain model produced from 2014 aerial photography / altitude data

Source: Fugro (2014)

9.1.4 Aerial Time Domain Electromagnetic Survey

In September 2014, GPX Surveys Pty Ltd ("GPX") performed a XTEM helicopter electromagnetic survey over the Mine and surrounds as part of the work associated with securing future palaeochannel water supplies for the processing of additional discoveries and/or processing plant expansions. The survey was flown using a Eurocopter AS350 BA Squirrel helicopter (Figure 25).

The data acquisition equipment comprised a XTEM time domain airborne electromagnetic survey system. XTEM consists of a carbon fibre and plywood frame that is suspended 30 m below the helicopter. A transmitter loop is attached to the outside arms of the rig and a receiver coil is located at the centre of the rig. A magnetometer sensor is mounted on the XTEM frame and the rig flown at a nominal height of 35 m above the terrain. Helicopter survey speed is between 45 and 50 knots and the along-line sample interval is between 2 and 5 m.

The XTEM receiver outputs 30-channel windowed data for subsequent processing.

Figure 25: Photo of aerial XTEM survey equipment

Source: GPX (2014)

A preliminary processed image is shown as Figure 26.

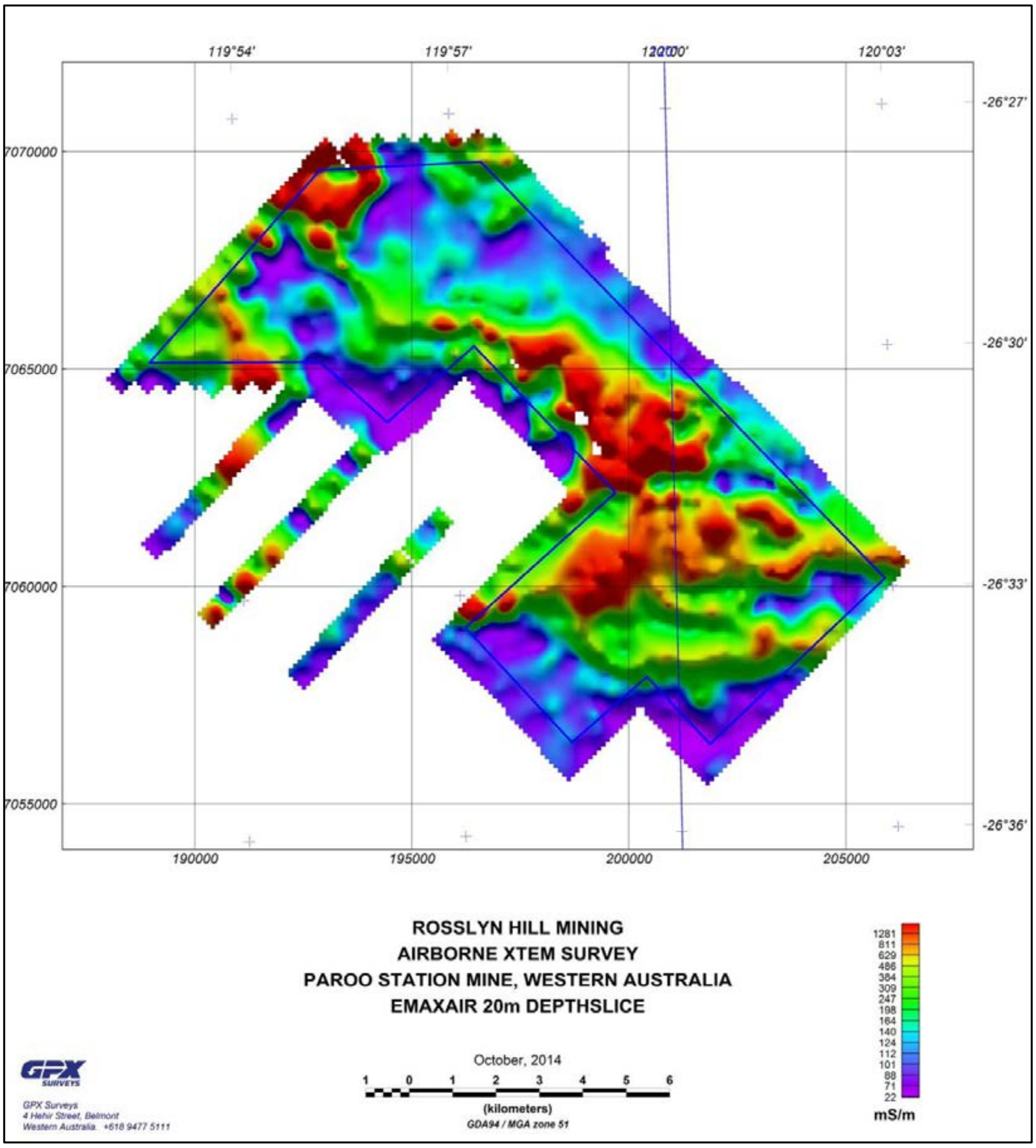


Figure 26: Preliminary airborne XTEM survey results (October 2014)

Source: GPX (2015)

9.2 Significant Results and Interpretation

All non-drilling forms of exploration have contributed directly to the targeting of additional mineralisation, either as extensions to known deposits, or to discovery of new deposits.

Geochemical surveys, including the conventional, portable XRF and combined datasets presented in Section 9.1.1 have greatly assisted in generating new drill targets.

In addition, the surveys have assisted in assessing the distribution of naturally-occurring lead in the environment, contributing to mine closure planning and environmental documentation.

Gravity surveys have generated new drilling targets around Drake and Pizarro (Sergeev, 2008). Several gravity targets were drilled at the Drake prospect in late 2013, with encouraging results.

Aerial photography and DTM generation have aided exploration through mapping of local geological contacts and has been used in land use studies as part of the mine closure planning documentation and environmental compliance.

10 Drilling

10.1 Summary Statistics

The Magellan Hill lead deposits have been explored and delineated by a series of drilling campaigns dating back to the early 1990s. Typical drill patterns have varied from 50 x 50 m to a staggered 50 x 100 m.

Grade control drilling at Magellan and Cano has infilled the exploration drilling data to a 12.5 x 12.5 m and 16.7 x 16.7 m patterns since the commencement of mining in 2005.

Table 19 summarises the RHM drill hole database by drill method as at 14 February 2018.

Table 19: Drill Hole Database Summary

Drilling Type	Number of Holes	Total Metres
Air Core (AC)	43	1,305
Rotary Air Blast (RAB)	1,318	30,868
Reverse Circulation (RC)	4,598	141,729
Diamond Drill (Core)	92	5,351

Source: RHM (2018)

10.2 Drilling 2015 – 2018

All drilling prior to the 2015 drilling campaign have been fully disclosed in the previous Technical Report (SRK, 2015)

In 2015, two drilling programs were completed at the Mine and surrounding exploration prospects; Table 20 outlines the drilling programs completed.

Table 20: Recent Drilling Programs

Program	Year	Number of holes	Total Metres
Exploration Drilling RC (Drake)	2015	7	315
Exploration Drilling RC (South Pizarro)	2015	9	405
Metallurgical Diamond (Magellan, Pinzon)	2017	22	730

Source: RHM (2018)

RC drilling was undertaken using face-sampling hammers and auxiliary air compressors to optimise sample recovery.

All diamond coring was conducted using PQ3 rod and bit technology (triple tube), with core retrieved using split sets inside 3 m core barrels to maximise recovery of the core. Control drilling techniques were used to limit penetration rates and maximise core recovery.

Figure 27, Figure 28 and Figure 29 indicate the location of the holes drilled during the period 2015 to early 2018; details of each program are outlined below.

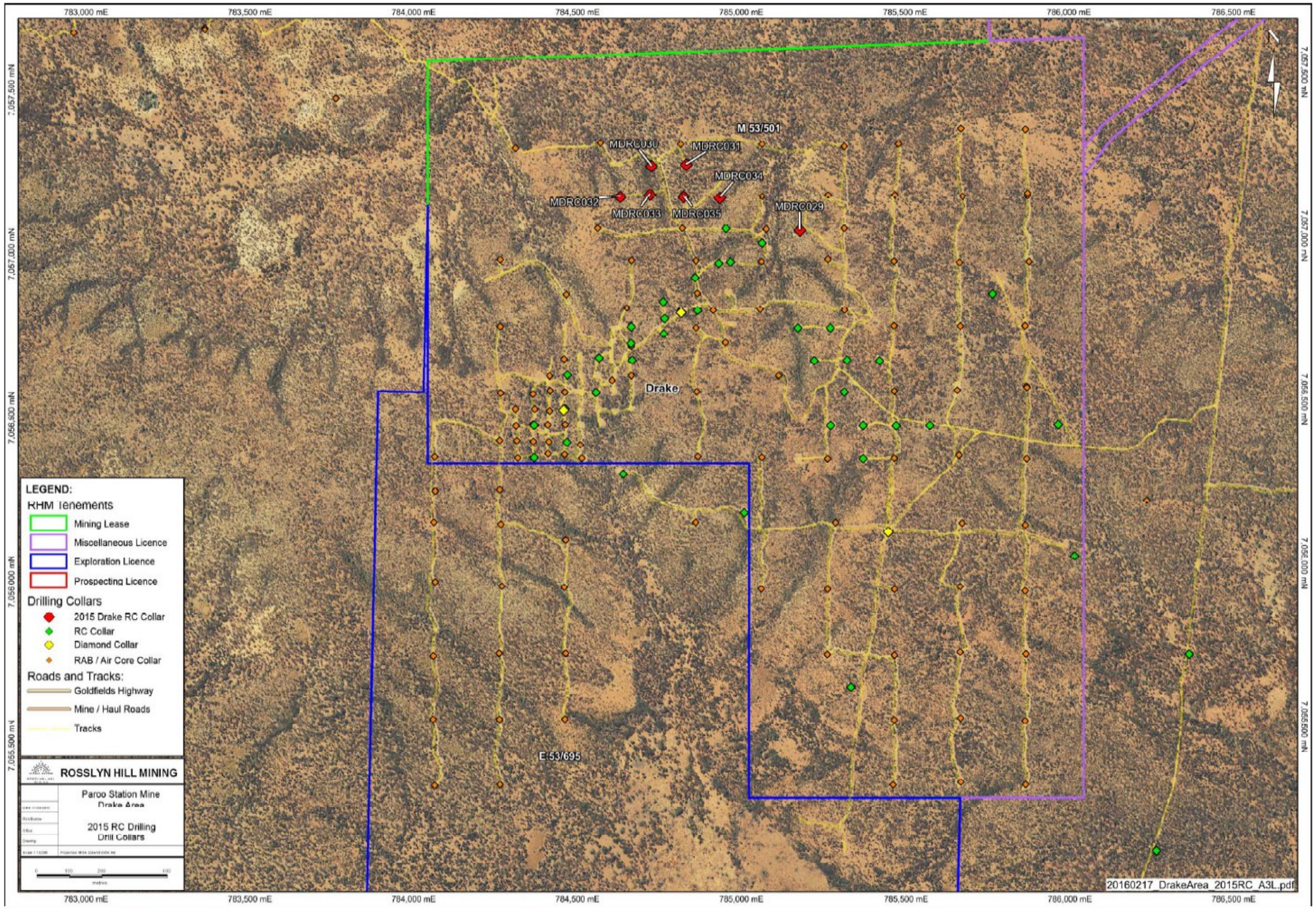


Figure 27: Location map of drill hole collars - Drake 2015 drilling program

Source: RHM (2016)

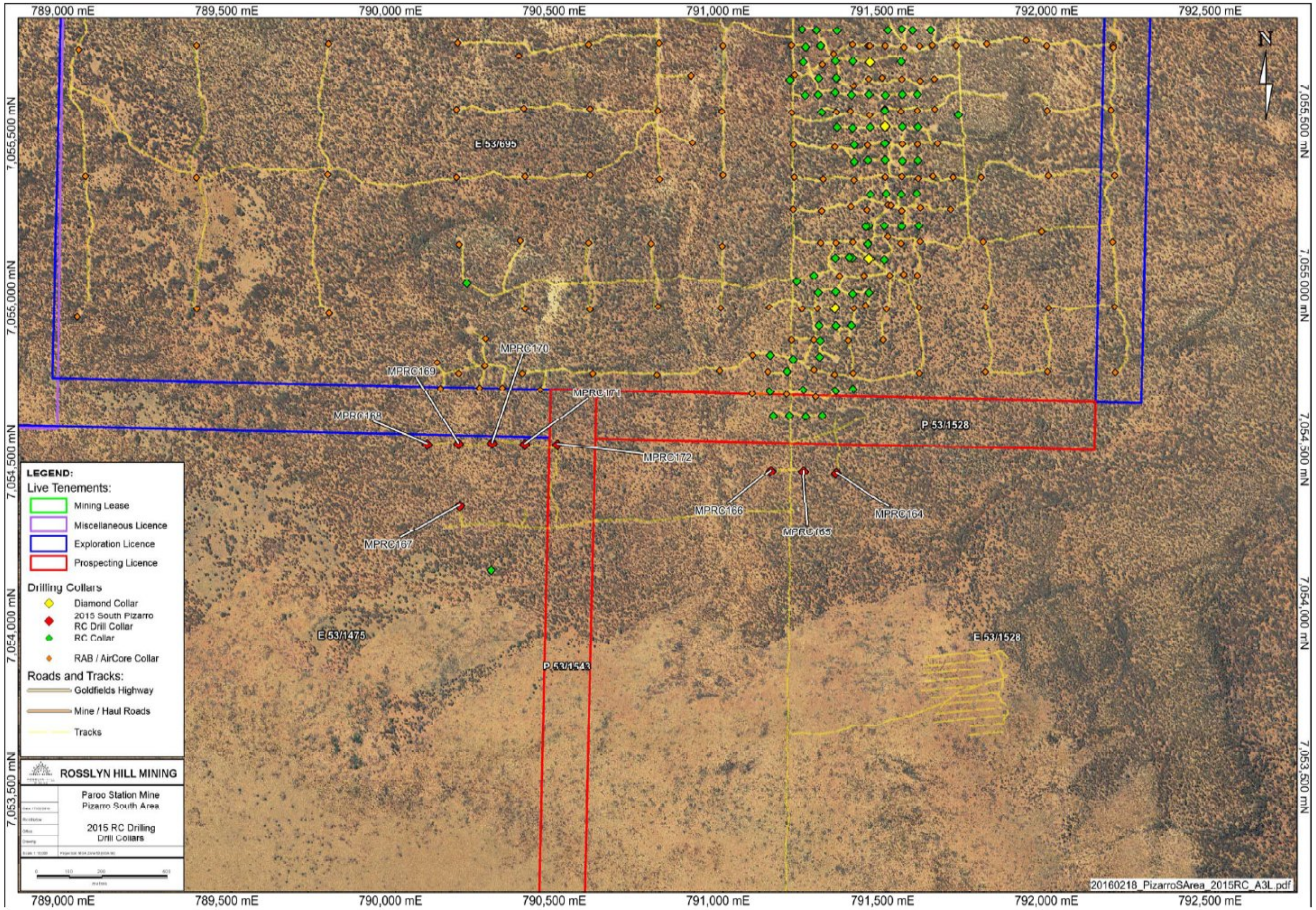


Figure 28: Location map of drill hole collars - South Pizarro 2015 drilling program

Source: RHM (2016)

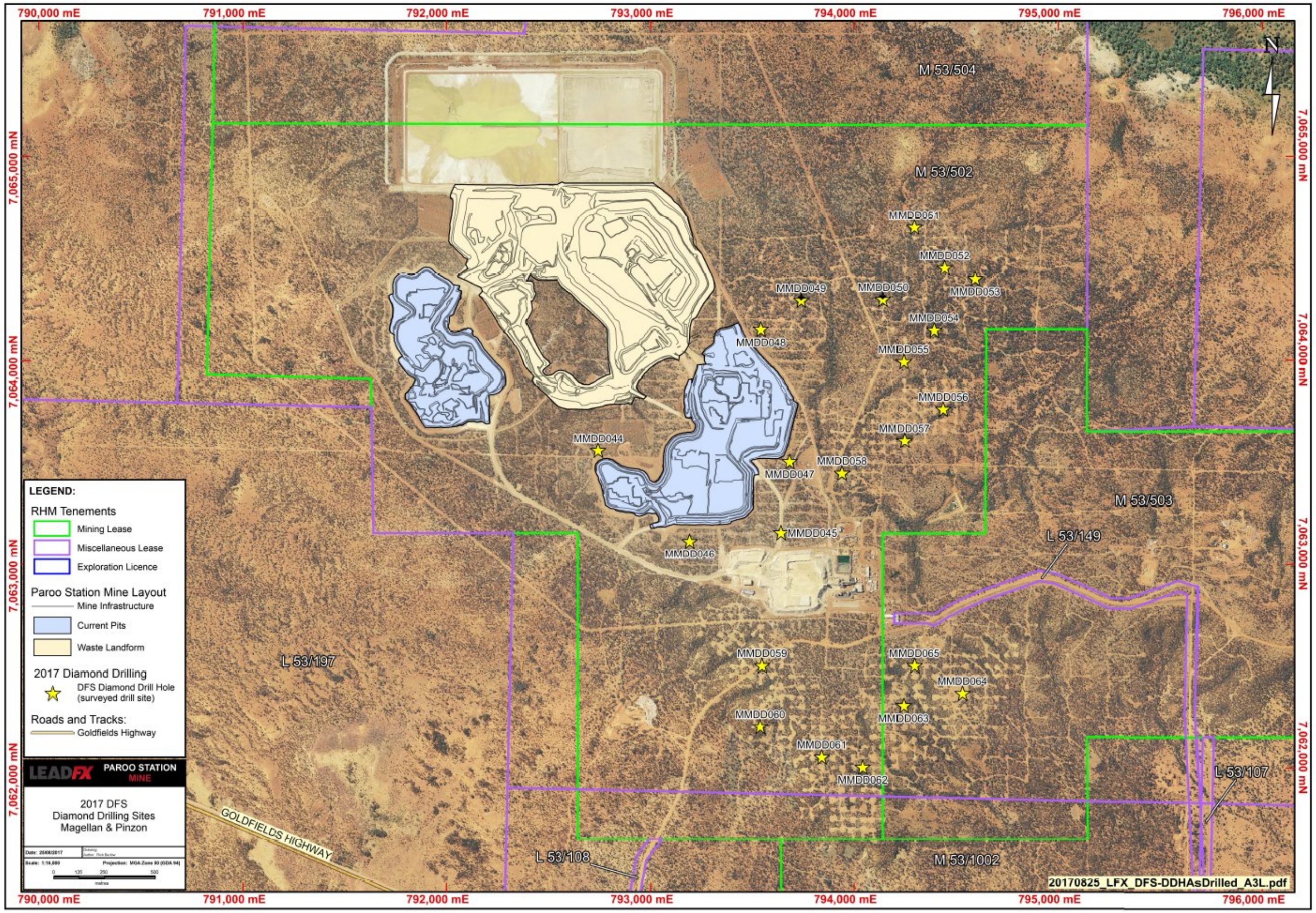


Figure 29: Location map of drill hole collars - Magellan-Pinzon 2017 metallurgical diamond drilling program

Source: RHM (2017)

10.2.1 RC Exploration Drilling (Drake)

RHM completed a RC drilling programme at the Drake lead deposit (M53/501) during March 2015 (refer Figure 27 above). Seven vertical RC holes were drilled for a total of 315 m (MDRC029-035). Each hole was drilled to a set depth of 45 m, for a total of 315 m of drilling. The drilling programme tested the northern extension of the secondary NW-SE Drake trend.

All holes were logged in their entirety using the standard RHM legend. Geological analysis was limited to review of geological logs and assays. All sampling and data collection used the RHM standard procedures detailed in Section 10.3.

10.2.2 RC Exploration Drilling (South Pizarro)

RHM completed an exploration RC drilling program at the Pizarro south area during March 2015. Nine RC drill holes were completed for a total of 405 m on tenements E53/1475, P53/1543 and E53/1528 (refer Figure 28 above).

On E53/1475, five vertical 45 m holes were completed for a total of 225 m in the extreme north-east of the tenement (MPRC167-171). On P53/1543, a single vertical 45 m hole was completed in the extreme northern portion of the tenement (MPRC172). The six holes tested for the continuation of mineralisation detected west of the main Pizarro trend.

On E53/1528, 3 vertical holes were completed for a total of 135 m (MPRC164-166). The drilling tested the continuation of the main Pizarro trend to the south of the Pizarro deposit in E53/695.

All holes were logged in their entirety using the standard RHM legend. All sampling and data collection referred to in Section 10.3. Geological analysis was limited to review of geological logs and assays.

10.2.3 Magellan-Pinzon Metallurgical Diamond Drilling Program

During June and July 2017, a large-diameter (PQ3) diamond drilling program was conducted at the Magellan and Pinzon lead deposits (refer Figure 29 above)

The diamond drill sites were planned to twin existing RC holes containing known mineralisation across the projected life of mining plan with the aim of collecting annual feed composite samples for variability and metallurgical testing as part of the DFS.

In all, 22 holes were drilled for a total of 730.1 metres by drill contractor West Core Drilling. All holes were drilled using PQ3 (triple-tube), averaging 16.2 m penetration per day of drilling. Mineralised sample intervals were identified using portable XRF analysis. A total of 383.45 metres of whole PQ3 core samples were selected and dispatched to the ALS Metallurgical Laboratory in Perth, Western Australia. Table 21 provides a summary of the drilling and the general location of the work according to deposit and tenement.

Table 21: 2017 Metallurgical Diamond Drilling Program

Tenement	Deposit	Number of holes	Drilled Metres
M53/502	Magellan	15	491.6
M53/502	Pinzon	4	138.1
M53/503	Pinzon	3	100.4
Total		22	730.1

Source: RHM (2017)

The results of the laboratory test work are discussed in Chapter 13.

10.3 Procedures

All 2015 – 2018 exploration and resource upgrade RC drilling was conducted using the procedures described below.

The sample preparation, analysis and security described in this section of the report refer to the current procedures employed by RHM.

Where historical procedures, results or analyses differed from current practice described in the 2015 Technical Report (SRK, 2015), these have been outlined.

10.3.1 Survey Control

All collar locations were set out using hand-held GPS units to an approximate accuracy of +/- 3 m. Tracks were set out according to plans approved by RHM's Native Title and Government departmental approvals process.

Once drilling and sampling was completed, drill hole locations were surveyed using RTK Differential GPS (RTK DGPS) equipment used by the mine surveyors or an appointed contractor surveyor. Nominal accuracy on drill collar locations is +/- 10 cm. For close-spaced grade control RC drilling, hole locations were both set out and picked up by RTK DGPS.

All holes are set up and drilled vertical to test the sub-horizontal mineralisation. As vertical drillholes, there is no requirement to downhole surveys the completed hole.

Hole divergence is minimal over the short, vertical drill holes <50 m in length and the use of vertical holes is appropriate for the sub-horizontal attitude of the mineralisation. To this end, downhole mineralisation thicknesses will provide a reasonable approximation of the true mineralisation thickness. The absence of downhole surveying for the vertical, relatively short drilling has been endorsed by several external consultants involved with the Mine (SRK 2011, Optiro 2015).

10.3.2 Sample Collection – RC Drilling

The 2015 RC drilling was completed by Intercept Drilling Pty Ltd, using a track-mounted Furukawa HRD 2000 RC drilling rig with on-board Sullair air compressor (1350 CFM / 500 psi) with a truck mounted 800 psi auxiliary booster, 3.5 inch rods, 5 inch down-hole hammer and up to 5 5/8 inch face-sampling RC drill bits with cyclone and pneumatic sample drop door to adjustable cone splitter.

Primary RC samples were collected at 1 m intervals based upon 1 m marks on the rig's feed chains. A shutter installed at the base of the cyclone was closed at the marked 1 m interval to minimise cross-contamination between samples. The shutter was reopened once the previous metre's sample bag was removed and the next metre bag was in place. The shutter opened to a cone splitter, which split a $^{1}/_{8}$ subsample in to a calico bag, and the remaining sample into a large plastic bag.

The indistinct nature of the cerussite and anglesite mineralisation makes the visual differentiation between mineralised and unmineralised material at the Paroo Station deposits difficult. The portable XRF is used to identify subsamples to be submitted for laboratory analysis (SRK, 2015).

For the geological logging, a cut length of PVC pipe was used to obtain a 'spear' subsample from each bulk sample bag. Where possible, samples are taken from the mid-point of the sample bag to the corner of the sample bag. If the sample has a very high ratio of chip to fines, preventing the spear from reaching the bottom of the bag, the bag was angled to the side in order to get more of a representative sample.

The logging subsample was sieved with a 200 mm medium size sieve with 2 mm mesh. An estimate of the percentage of the remaining chips and other information is recorded and a representative sample of the content was placed in plastic 20-compartment chip trays. All chip trays are stored at the

Mine in the chip tray building (SRK, 2015).

Table 22 shows sample details and submitted QAQC sampling for the RC drilling programs at Drake and South Pizarro.

Table 22: Sample details for 2015 South Pizarro RC Drilling Program

Prospect	Tenement	Date Drilled	Hole ID	Total Hole Depth (m)	Laboratory Assays	QAQC
Drake	M53/501	22/03/2015	MDRC029	45	7	2
		23/03/2015	MDRC030	45	17	2
			MDRC031	45	15	2
			MDRC032	45	18	2
			MDRC033	45	11	2
		24/03/2015	MDRC034	45	16	0
			MDRC035	45	13	2
Pizarro South	P53/1543	20/03/2015	MPRC172	45	21	2
	E53/1528		MPRC164	45	27	2
			MPRC165	45	22	2
			MPRC166	45	27	2
	E53/1475	20/03/2015	MPRC171	45	21	2
		21/03/2015	MPRC170	45	28	2
			MPRC169	45	13	2
			MPRC168	45	15	2
			MPRC167	45	10	2
		Total	**15**	**720m**	**281**	**34**

Source: RHM (2015)

10.3.3 Sample Collection – Diamond Drilling

The holes drilled for the 2017 metallurgical test work were drilled by West Core Drilling, using a Boart Longyear LF90D track mounted diamond drilling rig using a wireline drilling method. To provide the largest possible sample volume for metallurgical work and to maximise core recovery, PQ (83 mm) diameter triple tube was selected. The target depths were taken from the identified mineralisation in each twin RC hole. Primary drill core samples were collected during the 2017 metallurgical diamond drilling program according to the following protocol;

- Core was collected from the drill rig 2 – 3 times a day, during which the driller was consulted about progress, ground conditions, core recovery etc.
- The core was removed from the barrel and the triple tube barrel liner and then placed in Impala 2 and 3 plastic core trays which were used for safety and ease of handling.
- Core trays were covered and strapped to a 4WD for transportation back to a covered shed and placed on core racks for subsequent processing.
- Prior to logging and sampling the core was deliberately not cleaned to prevent washing out of loose/small particles.
- An initial geotechnical log was undertaken that recorded;

  - o Interval lengths (drill runs) were taken from driller's core blocks.
  - o The amount of core physically recovered for each interval was measured and recorded.
  - o Total core recovery was then calculated as a percentage (recovery/run length x 100).
  - o The sum total amount of core >10 cm per drill run was measured and recorded.
  - o RQD was calculated as a percentage (core >10 cm/run length x 100).
  - o Fracture frequency was counted as the number of joints per metre. Where fracture frequency was >20, the count was estimated as a percentage of broken core relative to competent core over that metre (i.e. 100% where the interval is entirely rubble; 50% where half the core is rubble, 25% where a quarter is rubble/broken).
- Drill core was marked with metre marks for sample cutting according to the driller's core blocks. Drill core was not oriented due to the vertical nature of the holes.
- A geological log using the RHM geological legend was recorded, including colour, grain size, major and minor lithological unit, alteration type and intensity, weathering and comments.

Mineralised intervals within the core were identified using a portable XRF instrument (Olympus Innov-X Delta, Serial No. 500138). The instrument was calibrated daily and checked against local matrix-matched standard samples. Two or more portable XRF readings were taken for each metre (or geological interval where <1 m) from surface to approximately 5 m above known mineralisation identified in the twin RC drill hole assay results. Three to four portable XRF readings were taken for each metre (or geological interval where <1 m) from (approx.) 5 m up-hole to (approx.) 5 m down-hole of known mineralisation.

Portable XRF results were recorded manually into a database during data collection along with the date and reading identification number. Assays were separated by depth into corresponding geological intervals. The portable XRF results were downloaded from the portable XRF instrument and tabulated into an MS Excel spreadsheet.

The manually recorded results were cross-referenced with uploaded results (date, reading number and Pb %). The portable XRF assay results were used in conjunction with geological interval, alteration logs and RC twin hole assay data to assign a mineralised interval for each diamond core. The interval was marked and packed as whole core for transport to the laboratory for analysis.

All drill core was photographed with a Pentax K20-D digital SLR camera. Photos were taken of the core in wet and dry states under well-lit conditions.

10.3.4 Sample Collection – Bulk Metallurgical Samples

A bulk ore sample was prepared at the Mine for testwork related to support the smelter metallurgical testing. The bulk ore sample was intended to create a typical run-of-mine lead concentrate for use in leaching and electrowinning test studies.

Approximately 30 t of ore was selected from two partly processed ROM ore stockpiles:

- Stockpile B39, a high grade stockpile averaging approximately 6.39%Pb
- Stockpile C20, a low-grade stockpile averaging approximately 3.47%Pb

Both stockpile B39 and C20 were sourced from Magellan pit and both contain a high proportion of clay-rich material. The B39 stockpile contains some YC breccia ores and sediment and the C20 stockpile is composed of varying sediment, including sand and siltstone.

A loader was used to obtain random buckets of material from the stockpile along the stockpile working face. Each bucket was passed through a static 'grizzly' screen to reject material coarser than ~80 mm. Oversize rejected by the screen was returned to the stockpile. The passed material was loaded into a series of 220 litre (44 gallon) lined steel drums using a small hopper. On filling, the liner was sealed and the lid attached. All drums were individually weighed with the drum number, weight and stockpile source marked on each drum and on a master record sheet.

Drums were loaded onto pallets, strapped together and the drums and pallets were washed clean and inspected before being freighted to ALS laboratory in Perth.

A total of 42 drums were sourced from stockpile B39 (high grade), for a total of 13,500 kg of material, and a total of 22 drums were soured from stockpile C20 (Low Grade), for a total of 6,837 kg of material. The combined bulk metallurgical sample shipped to ALS in Perth totalled 20,337 kg.

Further discussion regarding the metallurgical testing of the bulk metallurgical sample is provided in Section 13.

10.4 Interpretation and Relevant Results

Test results and outcomes of the Magellan-Pinzon metallurgical diamond drilling program are discussed in conjunction with the MRE in Section 14.

Significant intersections recorded by the Drake and South Pizarro RC drilling programs are shown below in Table 23 and Table 24.

Table 23: Intersections recorded for 2015 Drake RC drilling program

Prospect	Hole ID	Depth From (m)	Depth To (m)	Intersection (width / grade)	Comment
Drake (M53/501)	MDRC029	15	16	1 m @ 1.26% Pb	
	MDRC031	21	22	1 m @ 1.59 % Pb	
	MDRC032	13	15	2 m @ 1.29 % Pb	
		17	18	1 m @ 1.23 % Pb	
	MDRC033	14	18	4 m @ 2.34 % Pb	Includes 1m @ 4.50 % Pb from 14 – 15 m
	MDRC035	15	16	1 m @ 1.03 % Pb	
		18	19	1 m @ 1.32 % Pb	

Source: RHM (2016)

Table 24: Intersections recorded for 2015 South Pizarro RC Drill Program

Prospect	Hole ID	Depth From (m)	Depth To (m)	Intersection (width / grade)	Comment
Pizarro (E53/1528)	MPRC164	16	19	3m @ 2.24 % Pb	Includes 1m @ 3.24 %Pb from 17-18m
		20	25	5m @ 1.96 % Pb	Includes 1m @ 3.02 % Pb from 22-23m
	MPRC165	31	32	1m @ 1.22 % Pb	
	MPRC166	35	37	2m @ 1.20 % Pb	

Source: RHM (2016)

Geology logs and assay results were reviewed. A low grade intersection by eastern most MPRC164 recorded 3 m @ 2.24% Pb and 5 m @ 1.96% Pb from 16 m. Although not high grade, the thicker intersection is encouraging and may indicate the Pizarro trend locally turns to a south-easterly direction, similar to the changes in the northern Pizarro trend. A parallel structure (NW-SE trending) may be mineralised adjacent to MPRC164. Additional follow-up drilling is planned to test the interpreted trend. The 2015 results at Pizarro and Drake have confirmed the extensions to the known mineralisation, but the extensions to date are narrower and/or at a lower grade than the previously identified mineralisation.

Drilling in P53/1543 and E53/1475 returned no significant assays.

11 Sample Preparation, Analysis and Security

11.1 Security Measures

For all 2015-2018 drilling – RC and diamond core – all paperwork involved with sample dispatch for drill samples is prepared by the supervising geologist for each program. The sample list is compiled by the supervising geologist and a visual check is competed of all subsamples on the list prior to dispatch.

For RC drill samples, the subsamples (normally in calico bags) were placed into labelled plastic bags with an average of five subsamples per bag. The plastic bags were labelled, cable-tied and placed in one t polyweave 'bulka' bags which were also closed and cable-tied and were then readied for dispatch. For diamond core, the core is photographed wet and dry in-tray; this is done for geological record but also records the core in a 'before-shipping' state. Each core tray is neatly stacked in sequence on pallets before secure wrapping with shipping plastic to prevent any loss or tray movement. Drill core is labelled and handled as "Fragile" goods. For bulk ore samples, steel drums containing the sample were closed with lids, labelled and strapped onto pallets (4 drums per pallet) for dispatch.

Samples were delivered by road freight trucks from the Mine directly to the laboratory for processing. RC samples were delivered to Genalysis in Perth for sample preparation and subsequent assaying. Diamond core and bulk ore samples were delivered to ALS in Perth for processing and test work for the DFS.

As part of the chain of custody for each sample dispatch, the assay lab was sent a hard and digital copy of the sample submission paperwork containing details of the submission number, number of packages, number of samples, sample list, where it was sent from, consignment note, dispatch date, and the required preparation and analytical method.

The assay labs sent a confirmatory email documenting any discrepancies from the submission form such as additional or missing samples. Occasional sample discrepancies can occur, but are promptly solved due to the nature of the records kept and the processes and procedures adopted/ implemented.

11.2 Sample Preparation for Analysis

All sample preparation and analyses for the recent RC drilling programs conducted in 2015 – 2018 (discussed in Section 10) have been carried out at Genalysis (RC samples only) in Maddington, Western Australia, and at ALS in Balcatta, Western Australia (ALS, diamond core and bulk samples). These laboratories have been certified in accordance with ISO/IEC 17025:

- Genalysis date of accreditation: September 20, 1991 – Accreditation No: 3244
- ALS date of accreditation: December 22, 2015 – Accreditation No: 825

No aspect of sample preparation at Genalysis was conducted by an employee, officer, director or associate of RHM or Lead*FX*.

RC samples were received by the lab, sorted, checked and confirmed, then dried ready for pulverisation. Large samples were split down to a nominal 1.2 or 2 kg size and pulverised using a robotic pulveriser via the laboratory's sample preparation code: SP11, SP22, SP23 or SP24 depending on sample mass. RC samples were prepared in Genalysis hazardous sample preparation area, owing to the toxicity of the oxide lead content.

11.3 Sample Analysis

Before 2013, RC drill samples were analysed for lead only using an ore grade four acid digest with an Atomic Absorption finish to a detection limit of 0.01% Pb (Genalysis code: 4AH/AA).

The change to ICP-OES was made to accommodate multi-elemental data. For the 720 m of RC drilling conducted during 2015 – 2018, the primary, field duplicate and blank RC samples were analysed for aluminium, iron, lead, phosphorus and sulphur using an ore grade four acid digest with an ICP-OES finish to a detection limit of 0.05% Al, 0.01% Fe, 50 ppm/0.005% Pb, 0.01% P and 0.01% S (Genalysis code: 4AH/OE).

Details of the metallurgical diamond core sample and bulk sample analysis for DFS metallurgical test work are discussed within the DFS documentation.

11.4 QA / QC Procedures

A QA/QC program has been implemented by RHM to provide adequate confidence that sample and assay data can be used in resource estimation.

The QP has reviewed and is satisfied that the QA/QC system is sufficient to assess the data reliability, accuracy and precision. This QA/QC review relates specifically to the 2015 RC drilling program analyses undertaken by Genalysis for the assay batch 645.0/1504324 which contains all assay data for that program.

For the 2015 RC drilling at Drake and Pizarro, a total of 288 submitted laboratory samples were assayed with QA/QC samples consisting of duplicates (submitted), blanks and standards (4 submitted).

No QA/QC samples were collected for the diamond drilling program, as the purpose of drilling these cores was to obtain sample material for metallurgical test work and not Mineral Resource delineation. QA/QC protocols associated with the DFS test work are discussed within the DFS documentation.

As discussed earlier, the challenging drilling conditions at Magellan Hill mean recovery of RC samples has been variable with some sample loss observed in many drill holes. Various techniques such as close monitoring of air input and sample/outside return during drilling, collection of samples from the return hose, downhole geophysics and correlation between grade and recovery have been used to look at sample recovery. In isolation, these tests would not remove the concern of bias. However, in combination, RC sample recovery is not regarded as a significant issue for estimation of a Mineral Resource at the Magellan Hill and outlying deposits.

Results and interpretation of duplicates, standards and blanks derived from the 2015 RC drilling program are discussed below.

11.4.1 Standards

RHM submitted deposit specific reference materials as well as Geostats certified reference material (“CRM”) base metal standards for analysis with the primary RC drill samples. The in-house standards library (Mag-01 to Mag-19) are produced from pulps from the Magellan Hill deposits and are therefore matrix matched to the mineralisation. Geostats certified reference materials are sourced from various oxide and sulphide mineralisation and are not matrix matched. All submitted standards were analysed by Genalysis.

A total of 4 standards were used by RHM during the 2015 RC drilling program and were inserted at a ratio of approximately one standard to 72 primary samples. Mag-06 and Mag-18 were used from the deposit specific library, while GBM398-1 and GBM302-10 were selected from the GeoStats library.

Table 25 tabulates the standards assayed by Genalysis and the results.

Table 25: Genalysis assays of In-house and Geostats standards

Standard Type	Sample ID	Standard	Expected Pb (ppm)	Lab Assay Pb (ppm)	Comment
In-House	MQS0580	Mag-06	17,920	20,105	Continues to assay higher than the expected
In-House	MQS0581	Mag-18	250,300	251,763	Within 1 Std Deviation – OK
Geostats	QS000595	GBM398-1	26,669	27,201	Within 1 Std Deviation – OK
Geostats	QS000596	GMS302-10	55,869	54,922	Within 1 Std Deviation – OK

Source: RHM (2018)

All four standards submitted performed reasonably although as noted by Optiro (2015), Mag-06 is assaying above its expected average by some margin. This may reflect a bias in the original 20-assay result used to establish the material Pb ppm average for Mag-06 rather than any fault in the laboratory. Thus the high assay for MQS0580 is not unexpected (RHM, 2018).

The standards performance indicates that the analytical process is in control.

A range of internal and certified reference material standards were analysed by Genalysis as normal internal checks.

RHM notes Genalysis internal QA/QC assay Std05:GBM398-1 used the same GBM398-1 Geostats reference material as was used for submitted standard sample QS000596; the lab standard recorded 26,238 ppm Pb, well within 1 standard deviation (1,360 ppm Pb) of the expected value (26,669 ppm Pb).

The remaining 12 samples have not been reviewed but all data is maintained in the DataShed geological database for future reference.

11.4.2 Blanks

One blank sample is inserted into the sample stream for each drill hole; each passes through the laboratory sample preparation process and are used to monitor potential contamination during preparation. The blank material is a local uncertified waste basalt sourced from the Wiluna Gold operations which contains only trace amounts of lead (typically less than 150 ppm Pb).

A total of 15 blank samples were submitted during 2015 – 2018. Overall, blank performance is acceptable, although some anomalous values are returned. Figure 30 below shows the blank assays reported by Genalysis during 2015 – 2018.

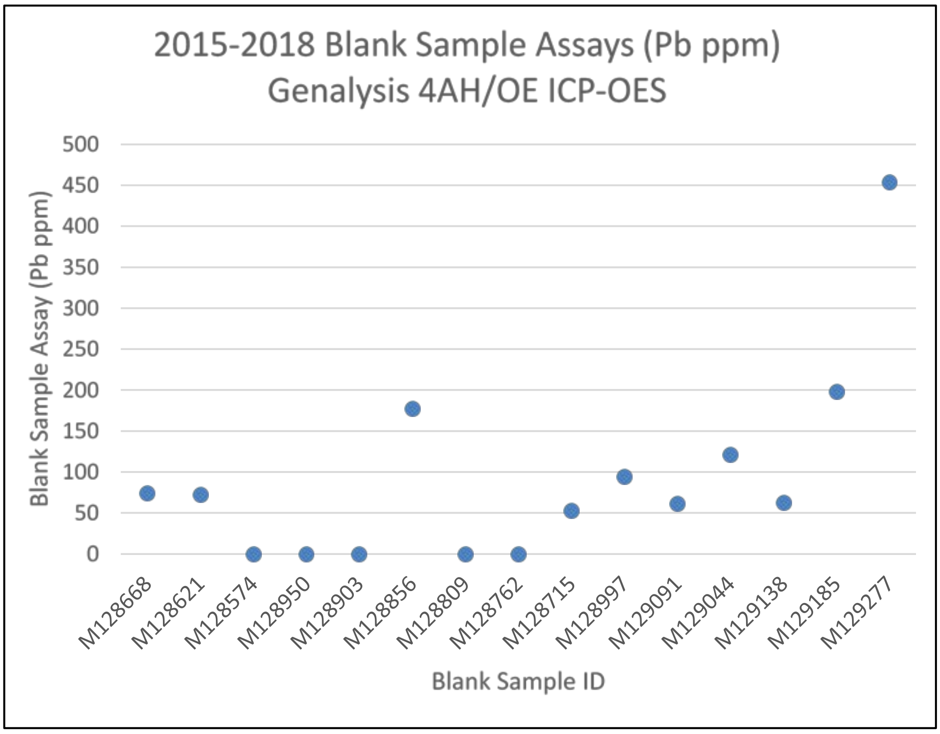


Figure 30: Blank sample assay performance

Source: RHM (2018)

Three blanks assayed over the expected maximum of 150 ppm Pb. Only M129277 exceeded the 150 ppm Pb by a significant margin with a value of 454 ppm Pb. This may be an outlier basalt sample with a slightly higher Pb content, or it may indicate a slight degree of contamination from a previous sample before the M129277 blank sample was prepared (RHM, 2018).

In comparison to the 5,385 ppm Pb (0.531% Pb) assay average for all primary samples, the one 300 ppm exceedance (0.03% Pb) is considered marginal, and would not indicate routine mishandling of the samples by the laboratory. No systematic bias was observed.

11.4.3 Duplicates

Field duplicates are inserted at a rate of approximately 1 per drill hole. 15 field duplicates were submitted to Genalysis during 2015 – 2018, making up 5.2% of the primary samples (ratio of approximately 1:19).

Figure 31 shows the performance of field duplicates vs. primary samples for RC drilling submitted during 2015 – 2018.

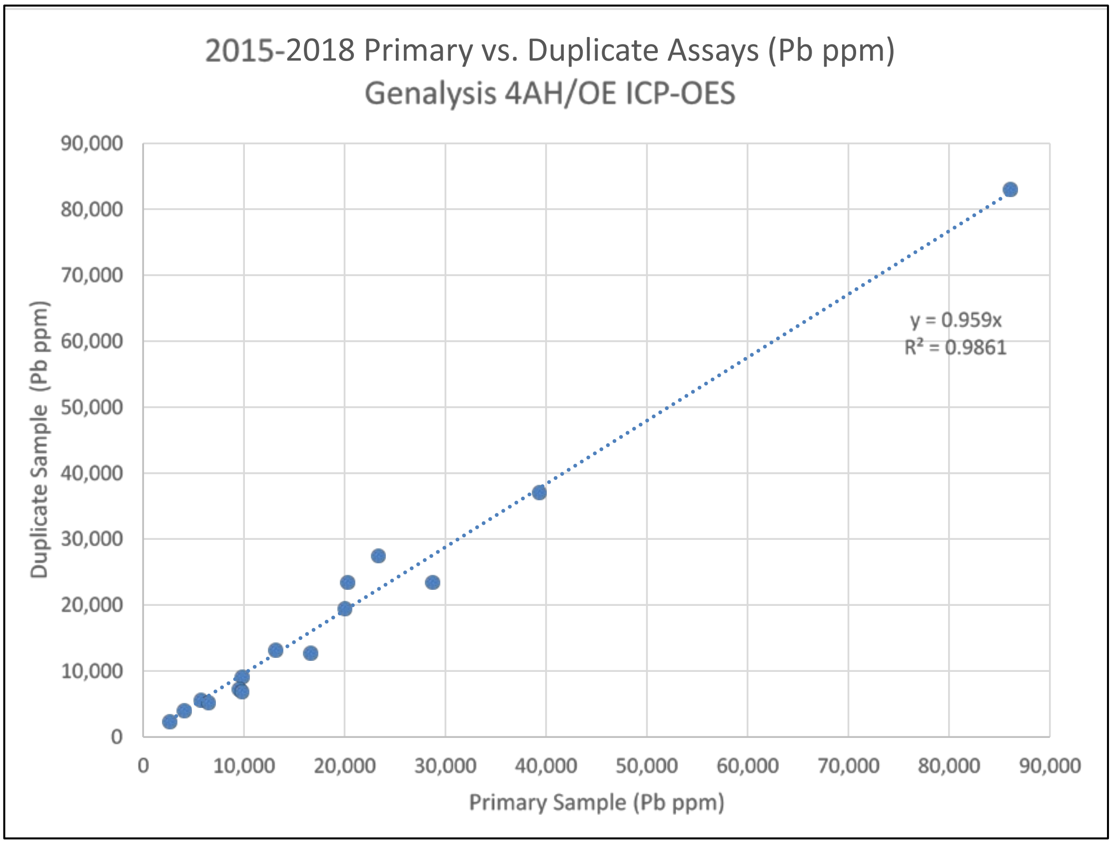


Figure 31: Field Duplicate sample assay performance

Source: RHM (2018)

The highest primary sample grade assayed was 86,054 ppm Pb (8.61% Pb) thus very high ranges above 10% Pb were not tested.

No significant assay bias is observed for the field duplicate samples. Correlation is reasonable for each primary-duplicate assay pair across most grade ranges.

11.4.4 Laboratory Pulp Checks

A total of 11 Laboratory pulp checks were assayed. This is an internal laboratory check as part of Genalysis internal QA/QC processes.

Figure 32 below shows the pulp check performance.

All exhibit very good correlation with no bias, showing adequate accuracy is being maintained during sub-sampling of the pulverised pulp, digest and assay finish.

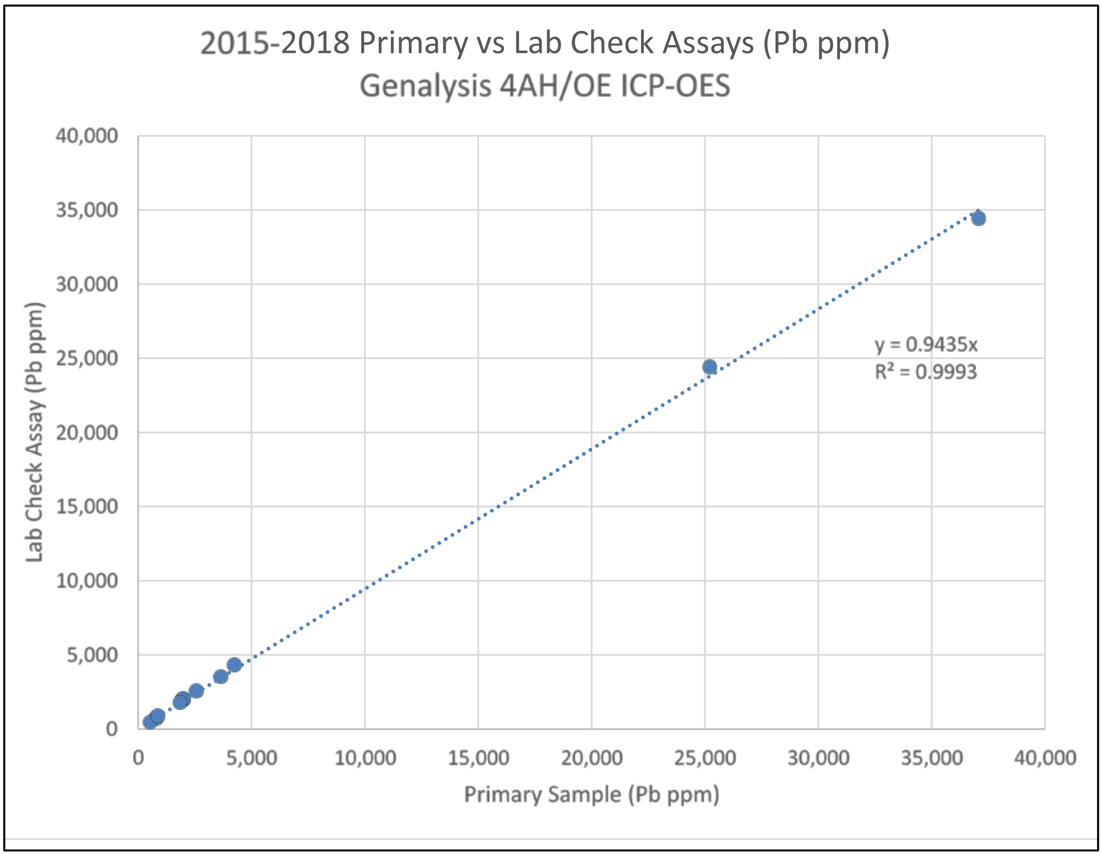


Figure 32: Pulp check performance

Source: RHM (2018)

11.4.5 Umpire Duplicates

No umpire samples were taken in the review period.

11.5 Discussion

Blank sample assay data since 2013 shows the uncertified basalt material used may not be acceptable; a metallurgical-grade blank material (certified, washed quartz gravel for example) with repeatable below-detection limit results for lead should be sourced for future blank assays.

The field duplicate data shows that an acceptable level of precision is being achieved by sample handling procedures and personnel conducting the sampling at the drill rig.

The laboratory check assays (duplicate samples taken from pulverised material) show that a high standard of precision is being achieved at the laboratory, with the proviso that the samples are selected and assayed by the laboratory and thus are not blind.

Overall, both the field and laboratory duplicates show that an acceptable level of precision has been achieved at the sampling stage and by the laboratory to date.

Confidence in the assay data is sufficient to support all geological interpretation of levels of mineralisation (RHM, 2018).

11.5.1 Actions

There are no actions envisaged at this point in time.

11.5.2 Results

The available analytical QA/QC was reviewed and no significant systematic errors were identified.

11.6 Opinion on Adequacy

Overall, the available QA/QC data demonstrated that the sample and analytical data captured is of sufficient accuracy and precision.

The results for blanks, standards and lab splits for lead have been reviewed. Overall, the QA/QC reviewed show acceptable results with no fatal flaws. Sufficient checks are in place to ensure that the assay results accurately reflect the samples.

Factors that could impact the accuracy and reliability of the results, such as drill sample spacing, recovery, moisture content and density, are adequately managed.

12 Data Verification

RHM and its consultants employ a number of QA / QC processes during drilling and sampling, these include:

- Duplicate RC sampling
- Testing of known standards and blank samples inserted into the sample stream
- Review of laboratory internal duplicate, blank and repeat samples
- Data transfer from paper logs to digital form by geologists
- Sample dispatch procedures
- Industry-leading secure geological database
- GIS and spatial data import, export procedures and visual checks

These methods have been reviewed and are considered industry standard and are reasonable for the drilling and sample methods employed and the status of the deposit as an operating mine. The same methods are used during care and maintenance periods.

The QP observed the geological and sampling work flow associated with the 2014 RC drilling program (as outlined in SRK 2015, Section 10.2.3) during various site visits in 2014. During those and other site visits, the QP observed an actual database update process and reviewed all previous QA/QC reports available from past drilling and sampling programs as described in the previous chapter.

An independent review was commissioned of the QA/QC results of 22 sample batches (4,681 drill hole samples) submitted to Genalysis in Perth, that included 538 QC samples and 977 Standard samples (CSA 2015).

The sample and geological data workflow accommodates RC and other drilling types (such as diamond, or RAB, Aircore etc.) as well as non-drilling samples such as portable XRF, soil and geochemical sampling.

All sampling associated with the 2015 RC drilling program was subject to the same processes as for the 2014 RC drilling program. QA/QC samples and results are discussed in Chapter 11.

12.1 Procedures

Electronic transfer methods employed by RHM and their consultants remove the risk of minor typographic errors. Subsequent interpretation of the data during geological modelling will identify significant batch, spatial and duplication/omission errors.

Collection, processing and importing of incoming geological and assay data is handled by RHM geologists according to formal documented procedures.

RHM employs a robust Maxwell Geosciences DataShed transactional database model for storage of all geological data, including all drill hole, surface geochemical and laboratory assay data.

The DataShed model, database and supporting software are maintained on RHM-internal server and computer systems. Database programming and direct maintenance of DataShed has been overseen by CSA and Maxwell Geosciences since the DataShed database was installed during 2009. All transactions and edits are tracked by the database. A backup copy is held internally by RHM, with a secondary backup being held offsite by Maxwell Geosciences. During operation, backups are run on a nightly, weekly and monthly basis and offer superior redundancy should one copy fail. During the current care and maintenance period, backups are performed on an ‘as-needs’ basis; usually when new data is loaded into the database.

The DataShed database contains linked libraries and import layouts designed to qualify all incoming data before allowing append operations to the database tables. The DataShed model has inbuilt constraints and triggers, ensuring that the data is validated and constrained e.g. no overlapping intervals or duplicated sample IDs (CSA, 2015).

Incoming laboratory assay data is received in laboratory-specific file formats and quarantined in DataShed’s custom assay buffer. The assay data must qualify and merge correctly with drill hole and sample interval data to enter the assay tables and be considered for export.

Exports are made from the live DataShed database to RHM-required formats such as MS Access and MS Excel. Other RHM software such as MapInfo/Discover and GEOVIA Surpac access these ‘moment-in-time’ export files rather than linking to the live database directly thus eliminating accidental editing or damage to the live database. The exports are updated as often as required to include new incoming data.

Visual checks of new geological and sample assay data are made, often spatially in 3D systems such as GEOVIA Surpac before geological interpretation.

12.2 Limitations

There appears to be no current limitations to the data verification practices used by RHM at the Mine.

12.3 Opinion on Data Adequacy

It is the QP’s opinion that the data at the Mine is adequate for use and the methods employed are considered industry standard and are reasonable for the drilling and sample methods employed and the status of the deposit as an operating mine.

13 Mineral Processing & Metallurgical Testing

13.1 Testwork

Two testwork phases are described in the following sections. The first relates to the initial and following years testwork for to the existing flotation plant at the Mine.

The second describes the testwork undertaken in relation to the DFS for the Hydrometallurgical Facility, which also included some additional flotation testwork.

13.1.1 Initial Test Work Sulphidisation Flotation

The original metallurgical test work was performed in Amdel's laboratory in Adelaide, South Australia, from late 1999 through to 2001.

Flotation was selected over gravity concentration as the processing route because of the broad size distribution of the non-sulphide lead mineralisation, especially with a significant portion of the lead being in fine particle size fractions.

Recovery of the non-sulphide lead minerals cerussite ($PbCO_3$) and anglesite ($PbSO_4$) by sulphidisation flotation has been a standard mineral separation process for around 90 years. The technical literature reports that cerussite responds better to sulphidisation flotation than anglesite because of its higher solubility. However, both test work and production operations treating either natural anglesite or that produced from leaching operations such as zinc hydrometallurgical plants show that good metallurgical results can be achieved from sulphidisation flotation of anglesite.

Sulphidisation flotation performance for the other lead minerals identified in the Mine deposits, such as pyromorphite ($Pb_5(PO_4)_3Cl$), coronadite ($(Pb,H_20)_2Mn_5O_{10}$), plattnerite (PbO_2) and plumbogummite ($PbAl_3(PO_4)_2(OH)_5 \bullet (H_2O)$), is sparsely discussed in the technical literature. Analysis of samples from pilot plant testing in 1999 (see below) on material from the Magellan deposit showed poorer metallurgical performance for pyromorphite and plumbogummite. Since these minerals constitute a minor proportion of the lead mineralisation and some of them have been identified in the lead concentrate (possible as composite particles with cerussite and/or anglesite), the net metallurgical effect from their possible low sulphidisation flotation performance is considered to be negligible.

Amdel's metallurgical testwork program for Magellan deposit material consisted of bench-scale laboratory flotation tests on samples from both RC and diamond core drilling. RC samples were treated in discontinuous runs in a mini-pilot plant with a feed capacity of 160 kg/h.

Bond Ball Mill Work Index of the Magellan deposit material was measured in the range 2 – 18 kilowatt hours per t (kWh/t) with a mean value of 8 kWh/t. The wide range of grindability values reflects the composition of the ore with fine "clay" material and coarse "siliceous" material.

Once pilot plant operation had been stabilised, surveys showed it produced a lead concentrate assaying 67 – 75% Pb with lead recoveries in the range of 77 – 88% from a head grade of 8% Pb with a flotation feed sizing 80% – 75 µm.

Bench-scale tests on a 100 kg composite sample of Cano material produced similar results to that for Magellan material.

The testwork demonstrated that sulphidisation flotation was a viable process route for treating the non-sulphide lead mineralisation in the Magellan and Cano deposits.

The concentrator was designed on the basis of the pilot plant and bench-scale testing data and commenced operation in 2005. In its first campaign production from October 2005 to April 2007, the concentrator processed 2.197 Mt of ore at a head grade of 7.3% Pb, with a 71.7% lead recovery into the lead concentrate.

There have been two main differences between the existing Mine flotation concentrator and the initial metallurgical testwork:

- Treatment of lower lead head grade ore, and
- Coarsening the flotation feed sizing to 80% passing 150 µm.

13.1.2 2017 Flotation and Hydrometallurgical Facility Testwork Program

The 2017 testwork program for the DFS was carried out predominantly by ALS in Balcatta, Western Australia. The test program included:

- Proof of concept testwork;
- Variability testwork;
- Pilot Plant testwork; and
- Concentrator testwork including pilot plant.

The first three programs are associated specifically with the Hydrometallurgical Facility. A number of other testwork programs were commissioned by RHM, including:

- An Electrowinning testwork program at the University of British Columbia;
- A Liquid-Solids separation testwork program with Waterex; and
- An Acid Recovery testwork program with Eco-Tec.

A significant flotation concentrator testwork program, not part of the Hydrometallurgical Facility DFS was carried out under the direction of RHM to evaluate flotation performance in order to produce appropriate concentrate samples for testing relating to the Hydrometallurgical Facility.

The hydrometallurgical component of the testwork program was carried out at the direction of RHM with support from SNC-Lavalin. Data analysis and incorporation of the testwork results into the design of the Hydrometallurgical Facility was the responsibility of SNC-Lavalin.

Samples of concentrate and ore for the various testwork programs were provided by RHM. Selection and composition of samples was undertaken by RHM in consultation with SNC-Lavalin.

Concept and Flowsheet Development

The initial concept flowsheet for leaching of Paroo Station lead concentrates was developed by Professor David Dreisinger and co-workers as published (US Patent 93220104 and Hydrometallurgy 142 (2014) 23-35), and included some preliminary testing of the Mine concentrate. Methane Sulphonic Acid (“MSA”) is used in lead electroplating and MSA has properties that make it particularly suitable for use in the Hydrometallurgical Facility flowsheet, the most important being the high solubility of lead. The initial concept flowsheet involved acid leaching of lead concentrate, purification of the leachate, and electrolysis for the production of lead cathode. For this DFS a testwork program was required to complete the proof of concept for the proposed flowsheet, to generate engineering design data and test the process with variable feed compositions expected to emerge from the Mine.

In addition, as the flowsheet contains novel technology, execution of a pilot plant campaign, to increase confidence in design and performance of the eventual plant, has been undertaken. This chapter summarises the result of this testwork with reference to more comprehensive testwork reports that are available for review. The different phases of the testwork are summarised below.

Proof of Concept Testwork

Initial proof of concept testwork was carried out on two samples of high grade lead concentrate held in storage by RHM after the shut-down of operations in 2015. These were provided to ALS for the initial testwork program. The objective in testing these samples was to provide proof of concept data of the overall flowsheet ahead of the more detailed variability and pilot plant testwork program.

The proof of concept testwork included three stage leaching tests, primary MSA leach of concentrate, desulphurisation leach (“DeS”) and secondary MSA leaching of DeS solids, and preliminary testing of a range of unit operations including solid/liquid separation and bleed treatment circuits, within the Hydrometallurgical Facility to provide preliminary engineering design data. Additional testwork was developed as the need arose, including Acid Recovery testwork (Eco-Tec), Settling and Filtration work (Waterex), Electrowinning (UBC) and Evaporation and Oxidation testwork at ALS.

Variability Testwork – Flotation

A drilling program was carried out on site by RHM to provide samples for a variability testwork program which was designed to generate annual feed composites across the projected Mine life. These ore samples were shipped to ALS for preparation and testing. The initial work comprised flotation testwork to prepare annual concentrate samples for hydrometallurgical testing. However, following this initial testwork, it became apparent to InCoR and RHM that the flotation recoveries were variable and additional flotation testwork was undertaken to resolve this issue. The revised flotation regime involved changes to pre-conditioning and flotation flowsheets. Some of these changes had previously been contemplated by RHM prior to the plant being put on care and maintenance.

Implementation of these changes provided an increased flotation recovery at target concentrate grades in the range 55-60% lead. However, testing of the solid/liquid separation of these concentrate samples resulted in a further change to the target concentrate grade.

The significant increase achieved in concentrate filtration rate when using cleaner column flotation prompted the targeting of a 70% lead grade as the feed to the Hydrometallurgical Facility. The other driver for this change was to improve on the poor filtration characteristics of leach residues of all three leaching stages. Changes to the preconditioning and operating pH led to significant improvements in lead recovery and flotation kinetics even at the higher concentrate grade of 70%. Column flotation was specifically used for the final concentrate cleaning stage to allow the concentrates to be washed using a positive bias ratio on the column to maximise removal of gangue slimes, which were negatively impacting on the concentrate filtration characteristics.

Inclusion of the column flotation step resulted in a significant drop in leach residue mass generation and improved concentrate filtration characteristics due to the absence of clays.

Variability Testwork – Hydrometallurgical Testing

The initial Hydrometallurgical Facility flowsheet followed the sequential plant operations of three consecutive leaching operations, followed by an impurity removal step and lead electrowinning. As noted the above column flotation improved the initial concentrate filtration however, leaching of the concentrate increased the slimes concentration in the leach residues and led to increasing problems with residue thickening and filtration. This situation led to a change in flowsheet whereby the MSA residue was treated in a CCD circuit and the DeS leach residue would be reintroduced into the flotation concentrator circuit to recover the lead carbonate produced in the DeS leach thus avoiding the poor filtration characteristics of filtration steps following both leaches. The flotation concentrate can then be introduced back into the primary MSA leach. The flotation recoveries in a single stage of flotation were 94 – 95% and this flowsheet change was adopted which allowed for the deletion of two of the three solid liquid separation circuits.

Cleaning the flotation concentrates in a second stage using column flotation resulted in a significant reduction in the magnitude of gangue leaching side reactions by a factor of approximately 20, essentially eliminating gangue components in the concentrate as a source of variability in the leaching operations.

The key variable remaining is the relative proportions of cerussite and anglesite in the concentrate which impacts both reagent consumption in the DeS leach and the mass of leach residue that needs to be thickened, filtered and washed between leaching stages. In addition, the presence of galena and pyromorphite in the concentrates received more attention in evaluating the variability samples.

The impurity removal unit operation is approaching the point of being redundant, given the extremely low levels of iron and aluminium removed in this circuit, to the extent that, subject to electrowinning performance with these metals present, the need for this circuit should be considered in a future optimisation exercise.

The key unit operations tested in the variability program were the thickening and filtration unit operations to ensure that the range of residue masses produced and variations in filtration flux rates were covered by the available filtration capacities in the design. Thickening was initially a consideration to the extent that the underflow densities impact filtration rates, whereas the sizing of the thickeners themselves is rise rate controlled given the very low feed densities of the leach residues.

As a result of variable and poor filtration rates and poor washing efficiencies achieved during the variability and pilot plant test program, a late change to the flowsheet was the introduction of counter current decantation (CCD) in the MSA Leach Residue Solid/Liquid Separation flowsheet. Thickening testwork was essential in defining settling rates and predicted underflow densities.

Pilot Plant Testwork – Flotation

Bulk samples from stockpiled ore on the ROM pad comprising 12.0 dry t @8.4 % Pb and 5.8 dry t at 5.3% Pb were shipped to ALS. The two samples were individually blended crushed and stored in drums prior to commencing the first stage of the pilot plant which was preparation of a bulk lead flotation concentrate. A range of concentrate lead grades were developed from the piloting.

ALS assembled a pilot milling and flotation plant and both samples were treated separately through the pilot plant using the reagent regime and flowsheet modifications developed during the variability testwork program. During initial operation of the pilot plant, as a result of improved flotation kinetics, a decision was taken to separately recover a high-grade concentrate from the front cell of both the rougher and first cleaner cells and the residence time of the first cleaner was reduced by 50%. The cleaner circuit was also reconfigured so that the concentrates produced following the first cell were recycled to the head of the rougher flotation circuit. This revised pilot plant flowsheet produced approximately 6% better lead recoveries than the equivalent batch flotation test that preceded the pilot plant at similar concentrate grades. The improved lead recovery is attributed to the removal of approximately 60% of the lead in the 1st rougher without any requirement for further cleaning and the decision to close the 1st cleaner circuit. Washing in column flotation also is a significant driver in producing a high concentrate grade with minimal cleaning losses. The first rougher and first cleaner concentrates were combined, and the lead grade approximated 60% for both samples.

After conducting initial hydrometallurgical testwork on the 60% lead concentrates, it became apparent that both the concentrate filtration and the filtration of leach residues were extremely problematic. Further cleaning of the concentrate was required. Consequently, both high grade and low-grade concentrates were subsequently cleaned through a continuous laboratory column flotation circuit, which was highly effective in rejecting gangue slimes and increasing lead grades in the concentrates. Based on the need for slimes rejection to alleviate low filtration rates in the Hydrometallurgical Facility a concentrate target grade of 70% ± 2% has been set for the reconfigured flotation plant based on the results achieved in column cleaning. The final concentrates produced were also washed with demineralised water to remove residual flotation reagents and trace chloride content.

Pilot Plant Testwork – Hydrometallurgical Testing

The pilot plant operation was run in two stages. During the first stage excessive frothing was noted in the MSA leach reactors. Adding an anti-foaming agent reduced the frothing but caused further issues with solid/liquid separation and electrowinning unit operations. On further investigation, residual flotation reagents in the concentrate were indicated as the root cause of the problem. Drying of the concentrate could remove the majority of the undesirable reagents before leaching.

The second stage of pilot plant operations used a dried concentrate which proved effective in controlling frothing, without the added expense of the anti-foaming agent. Concentrate drying was adopted for the commercial flowsheet.

The busbar and hanger bar systems in the electrowinning cell were also upgraded due to overheating of the initial systems at the high current density. Smoothing agent types and addition rates were also adjusted to improve cathode quality and graphite and DSA anodes were trialed separately.

Overall the second pilot plant run was successful in operation of the MSA leaching circuit in closed circuit with the electrowinning cell, the latter achieving uninterrupted lead cathode production. The cathodes deposits were nodular, however chemically of acceptable quality. Lead ingots were also produced for assay. Significant dross formation during this step at lab scale was ascribed to the small scale of the operation. Flotation of the DeS Residue gave lead recoveries in the [94-95%] range which confirmed that the DeS residue solid/liquid separation circuit and the MSA Re-leach circuit and solid/liquid separation circuits could be deleted from the flowsheet.

13.2 Metsim Modelling

The Hydrometallurgical Facility flowsheet was initially modelled in Metsim by SNC-Lavalin based on the proof of concept testwork carried out in part during the scoping study. The base case model was further developed based on additional batch testwork carried out during the early stages of the DFS. The flowsheet has evolved as further testwork data became available to define the process parameters required for the flowsheet. This data has been incorporated into an optimised base case model which reflects the basis of the flowsheet design.

The original concept testwork on which the flowsheet is based identified a requirement for three separate leaching circuits, one to leach lead carbonates, a conversion leach to react lead sulphate with sodium carbonate to produce lead carbonate, and a final lead carbonate leach. Little additional work was carried out on the remaining flowsheet elements. Further work on the detail of the flowsheet identified a need to incorporate an impurity bleed into the flowsheet and further recover MSA from various metal MSA salts produced by gangue leaching reactions in order to contain operating costs.

Testwork identified an opportunity to simplify the original concept flowsheet by eliminating the MSA releach circuit and floating the DeS conversion residue to produce a cerussite flotation concentrate for recycle to the MSA leach. This approach eliminated two problematic solid/liquid separation circuits.

The overall water balance was also an issue with the need to incorporate an evaporator into the overall flowsheet to maintain a closed water balance, which is driven by steam generated from waste heat from the power station.

During the DFS, a series of three Metsim models were developed for the Mine as follows:

- A Base Case model incorporating all of the flowsheet elements required to operate the Hydrometallurgical Facility. The base case grade of 70% lead was selected following investigation of concentrate treatment at grades of 55% and 60% Pb. The concentrate feed input data to the model is based on average life of Mine data derived from the variability testwork program.
- Individual models were run using the base case model with different input data based on testwork

results generated by the variability testwork program for a high and low anglesite proportion of the base case feed mineralogy consistent with the range in anglesite proportion of the concentrates in normal concentrator operations.

Outputs from these models were used to validate the process design to ensure that the range of operating conditions under which the Hydrometallurgical Facility would be required to function were incorporated into the process design.

A drilling program was undertaken to provide representative samples of each year of production for the Mine with the proposed Hydrometallurgical Facility based on a revised mine cut-off grade calculated by RHM.

These samples were then treated according to the operating practice of the existing flotation concentrator to produce a range of concentrate samples to be evaluated according to the revised Hydrometallurgical Facility flowsheet. The concentrate grades produced for this testwork program were compared to the typical concentrates previously produced for shipment to a smelter to increase overall lead recovery given that the concentrates could now be treated on site to produce lead ingot. An analysis of the flotation results indicated that the concentrate grade that minimised slimes recovery to the flotation concentrate was of the order of 70% lead, up from the 67% – 68% lead grade targeted for sale to a smelter. With the revised flotation regime flotation recovery was increased relative to historical concentrator performance.

13.3 Process Design

A metallurgical testwork program comprising batch and pilot plant works was carried out to provide the design data required to develop the Hydrometallurgical Facility flowsheet. Samples for this work were selected to represent ore quality from years 1 – 10 of forecast mining.

An initial flotation development program was executed to prepare concentrate samples for the Hydrometallurgical Facility testwork. In the process of generating these samples significant potential for improvement in the flotation performance was identified and incorporated into the concentrator design.

The Process Design Criteria for the Hydrometallurgical Facility have been developed by SNC-Lavalin to establish a Metsim Model and for mass balance and equipment sizing calculations. Select process design data for the major process areas are also presented in this chapter.

13.3.1 Flotation Recovery Model

A Concentrator Metsim Model has been developed to reflect the proposed modified flowsheet and has been used to evaluate process parameters for the proposed flowsheet based on the flotation testwork executed at ALS. The flowsheet modifications included converting the mill from SAB to SABC, a modified flotation circuit using existing equipment and addition of a flotation column to produce a final concentrate in the range 68 – 72% lead grade to feed the Hydrometallurgical Facility. A grade recovery algorithm for the revised flowsheet was developed for use in assessing flotation concentrator performance across the range of anticipated feed compositions.

13.3.2 Hydrometallurgical Facility Model

A Metsim Model was developed by SNC-Lavalin for a “base case” mineralogy comprising 71.3% lead carbonate (cerussite) and 11.9% lead sulphate (anglesite) at overall concentrate grade of 70% lead. The other major minerals assumed to be in the concentrate (based on XRD analysis) are Pyromorphite (4.6%), Galena (2.9%), Leadhillite (0.9%), Kaolinite (1.0%), Hematite (5.3%) and Quartz (2.2%). On completion of the variability testwork, additional Metsim Models were run for the assumed minimum (3%) and maximum (15%) anglesite levels, which have been run to assess the impact of the changing concentrate mineralogy on mass balance flows and operating costs.

Testwork interpretation for design criteria for both modifications to the existing concentrator and for the Hydrometallurgical Facility are detailed in the section on Recovery.

13.4 Processing Plant Description

13.4.1 Existing Concentrator Plant

Ore is processed through a conventional flowsheet consisting of the following main steps:

- Crushing and Grinding
- Flotation, Dewatering and Drying

The processing block flow diagram is shown in Figure 33 (not including proposed modifications).

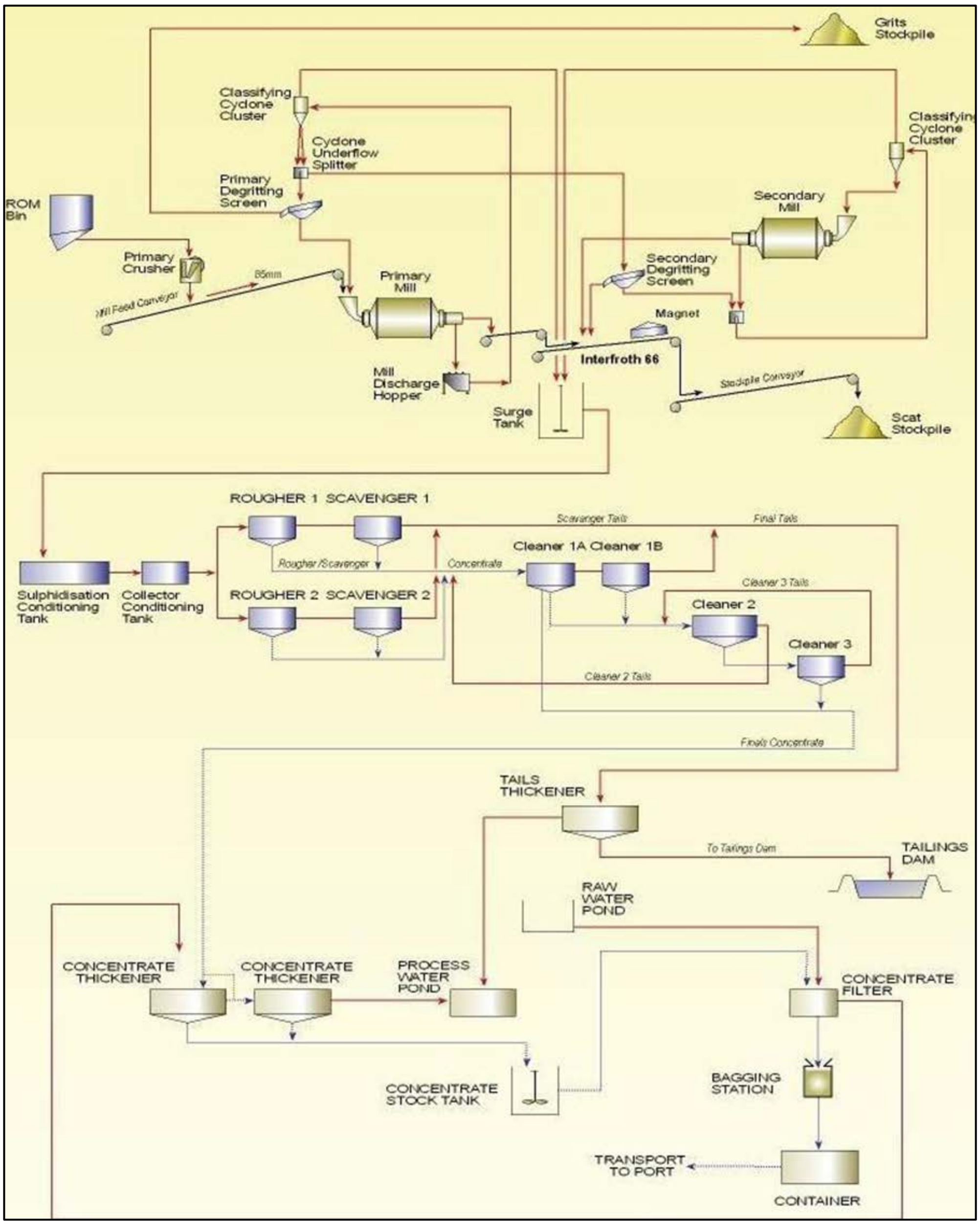


Figure 33: Existing Concentrator Plant flow diagram

Source: RHM (2018)

Crushing and Grinding

Ore is delivered to the ROM pad by haul trucks from the Mine. It is separated into ore stockpile "fingers" and withdrawn by front-end loader and blended as required. Stockpiled ore is fed into the ROM bin over a static grizzly screen.

The ROM bin provides approximately one hour of residence time. Ore is withdrawn from the ROM bin via an inclined apron feeder and fed over another grizzly which is currently covered. A single toggle Goodwin Barsby 1219 x 914 mm primary jaw crusher with a 132 kW motor capacity and a closed side setting of 110 mm processes in excess of 300 t/h of ore producing a product with an 80% passing size (P_{80}) of approximately 45 mm.

The primary crushing facility has significant excess capacity due to the relatively low ore competence. The ore characteristically has a bimodal particle size distribution with a coarse siliceous fraction probably derived from the upper portion of the quartz-clay breccia and clay units, and a fine fraction predominantly coming from the clay-rich lower horizons of the deposit.

Dust control is managed by ducting and covers over the conveyor.

Crushed ore is conveyed to a 2-stage closed circuit grinding section. Primary crushed ore is fed directly to the semi-autogenous grinding ("SAG") mill feed chute and ground in a Morgårdshammar 4.2 m diameter x 5.2 m grate discharge SAG mill with a 1350 kW motor using 105 mm diameter grinding media.

The mill is lined with rubber-metal composite liners and operates in closed circuit with a cluster of five x Krebs gMax 15 hydrocyclones for classification. While hydrocyclone overflow goes to the flotation feed surge tank, the underflow is divided with two thirds going to the degritting screen before returning to the SAG mill feed chute and the remaining one third going to the ball mill. The ball mill can be isolated at low throughput rates.

The ball mill was installed in series with the SAG mill in August 2006 to increase nominal milling capacity to ~1.9 Mtpa. This secondary mill is also a Morgårdshammar mill (3.5 m diameter x 4.9 m) with a 1350 kW motor using 65 mm diameter grinding media and also lined with rubber-metal composite liners. Motor power draw is currently restricted to 650 kW because of mechanical issues. This ball mill is operated in closed circuit with a cluster of four Krebs gMax 15 hydrocyclones.

The overflow product from both hydrocyclone clusters targeting a product sizing of 80% (P_{80}) minus 150 microns discharges directly into a flotation feed surge tank with a live volume of 500 m^3.


Figure 34: Paroo Station crushing and grinding sections

Source: RHM (2015)

Low grade ore scats from the SAG mill trommel and “siliceous grit” from the oversize of the degritting screens can be stockpiled and treated if the lead content warrants it.

Flotation, Dewatering and Drying

Concentration of non-sulphide lead-bearing minerals is carried out by sulphidising froth flotation. In this process a sulphidising reagent is added to the slurry of ground ore effectively coating the surfaces of the lead-bearing mineral particles such as cerussite and anglesite and changing the surfaces to a lead sulfide. This enables the particles so “sulphidised” to be recovered by conventional sulphide flotation.

Slurry from the flotation feed surge tank is pumped to the sulphidisation conditioning tank where the sulphidising agent, sodium hydrosulfide (“NaHS”) is added to maintain a set oxidation reduction potential (“ORP”). While most of the NaHS is added at this this stage there are smaller dosing points at the scavenger flotation cells. The overflow from the sulphidisation conditioning tank goes to the collector conditioning tank where the collector sodium isobutyl xanthate (“SIBX”) and frother (Interfroth 50) are added.

Flotation pH is controlled to around 7 – 7.5 using caustic soda or lime, with sodium silicate dispersant occasionally used. The flotation circuit consists of two parallel rougher-scavenger stages and three cleaner stages. Slurry from the collector conditioning tank with the ORP raised to

150 – 170 mV gravitates to rougher-scavenger flotation which is done in two parallel rows of 6-cell banks of Outokumpu OK-38 machines (38 m^3 per cell) arranged as 3 x cell rougher and 3 x cell scavenger.

These six-cell banks are on rougher and scavenger flotation duties with the rougher/ scavenger concentrate gravitating to the head of the first cleaner cells and the tailings reporting to final tailings. The first cleaner stage also consists of six OK-38 cells, with the second and third cleaner stages in closed circuit comprising six and four Denver DR-300 flotation cells (8 m^3 per cell) respectively. Final lead concentrate assays around 64 to 66% lead.

Flotation concentrate is thickened to 67% w/w solids by two thickeners in series: 6 m diameter thickener followed by 9 m diameter clarifier with the clarifier overflow gravitating to the process water dam. Underflow from both thickeners is pumped to a thickened concentrate storage tank and then filtered to 8.5% w/w moisture.

In 2007, a Metso Vertical Plate Filter model VPA 1530-36 was installed to replace the original two belt filters on concentrate dewatering duty.

Final tailings is thickened to ~48% w/w solids in a 16 m diameter high rate thickener with the underflow pumped in two stages to the TSF and the overflow going to the process water pond for recycling.

Process monitoring and control is done by a Citect SCADA system with the concentrator having typical measurement equipment for a modern flotation plant such as a multi stream on-stream analyser unit. The most important variable to control in the flotation section is the ORP where the ground ore is sulphidised.

Figure 35: Paroo Station flotation section

Source: RHM (2015)

Figure 36: Paroo Station lead concentrate filter

Source: RHM (2015)

13.4.2 Modifications to Concentrator Plant

The existing flotation concentrator will be modified prior to recommencing operations to achieve two objectives:

- Increased throughput;
- Improved metallurgical performance.

The process areas comprising of the modified flowsheet are:

- Primary Crushing;
- Pebble Crushing;
- Milling;
- Rougher Flotation;
- 1st Cleaner Flotation;
- 2nd Cleaner Flotation;
- Concentrate Thickening;
- Concentrate Filtration;
- Tailings; and
- Reagents.

Services are not included in the description below, as no modifications have been made to these areas of the flotation concentrator.

Primary Crushing

The existing primary crushing facility is largely unmodified. Two minor updates have been incorporated into the revised flotation concentrator flowsheet. A static grizzly has been introduced ahead of the primary jaw crusher to allow fines at -100 mm to bypass the crusher. Additionally, the drive on the SAG Mill Feed Conveyer has been upgraded to accommodate the additional capacity required for the pebble crusher discharge.

Open pit ore is loaded onto trucks and hauled to a ROM pad situated close to the crushing facility where ore is dumped onto the ROM stockpile to allow for blending of the plant feed for both grade and ore hardness. Finger stockpiles are used for blending the ore between the mine and plant. Ore is fed into the ROM bin by the Front-End Loader (“FEL”). Loading of ore to the ROM bin is controlled by the crusher operator who activated tipping light indicators from the control room. Live capacity of the ROM Bin is 480 t giving approximately 100 mins surge capacity at design crusher throughput.

The objective of the Coarse Crushing area is to reduce ROM ore in one stage of crushing to a size suitable for further size reduction in the SAG Milling circuit.

The crusher is capable of crushing ore at a maximum rate of 285 t/hr with an average feed rate of 157 t/hr.

The Primary Crusher discharges to the Mill Feed Conveyer.

Dust emission control in the Crushing area is provided covered conveyors. The feed to the SAG Mill is monitored and controlled by the SAG Mill Feed Weightometer.

Pebble Crushing

The existing SAG Mill discharge grade will be pebble ported to allow coarse rock in the SAG Mill media charge Oversize discharged from the SAG Mill trommel onto the Pebble Crusher Feed Conveyer which feeds to the Pebble Crusher Feed Bin, Belt Magnets and Pebble Crusher Metal Detector are provided on the Pebble Crusher Feed Conveyer to remove tramp steel, predominantly ball scats ahead of the Pebble Crusher.

If the pebble crushing circuit is off line a diverter gate and discharge chute is provided to divert the SAG mill discharge screen oversize to the existing concrete bunker.

The recycle crusher produces a 10 – 12 mm product, which is discharged directly to the SAG Mill Feed Conveyer.

A monorail maintenance hoist will be provided for crusher maintenance.

Rougher Scavenger Flotation

The rougher scavenger circuit comprises two parallel trains of six OK30 flotation cells. The following description details the equipment in Train 1. An identical equipment set is provided for Train 2.

Process slurry is received from the Milling Train cyclone overflow and enters into the first of four conditioning tanks. In the first tank the slurry pH is adjusted to 5.5 with sulphuric acid pumped from the hydrometallurgical circuit. In the second tank the pH is raised to 6.0 with lime pumped from the hydrometallurgical circuit. In the third tank NaHS is added to condition the surfaces of the lead minerals ahead of collector addition. The final tank is a collector conditioning tank.

The objective of this circuit is to produce two product streams for subsequent cleaning and recycle to flotation feed. The fast-floating component of the lead concentrate is recovered off the 1st rougher cell and gravitates to the column feed pump box.

Concentrate from the remaining rougher cells and the scavenger circuit gravitates to the Rougher Concentrate Pump Box and the duty Rougher Concentrate Pump pumps the combiner rougher scavenger concentrate to the 1st Cleaner circuit.

Float level indicators are used to control flotation cell level by adjusting valves located on the tails discharge of every second flotation cell. Frother is added to rougher flotation cell 1 and scavenger flotation cell 1.

The flotation cells are forced aspirated with air supplied by the duty blower.

1st Cleaner Flotation

1st Cleaner

The 1st Cleaner Scavenger Flotation Cell comprise a single Outokumpu OK30 flotation cells of 30 m^3 capacity in a bank of 6 identical cells. Frother is added as required to the feed box of the first flotation cell. The flotation cells remain sized on the concentrate carrying capacity requirements and provide 12.5 mins residence time to allow for recovery of a fast-floating high-grade concentrate.

The concentrate gravitates to the feedbox of the column flotation cell.

1st Scavenger Cleaner Flotation

The Cleaner Scavenger Flotation Cells comprise a bank of 5 Outokumpu OK30 flotation cells of 30 m^3 capacity. The flotation cells remain sized on the residence time requirements, and provide 37.5 mins residence time to recover slow floating lead minerals.

The duty Cleaner Scavenger Concentrate Pump, pumps the cleaner scavenger concentrate to the feed box of the 1st cleaner flotation circuit.

Cleaner scavenger tailings gravitate to the tailings thickener.

Float level indicators are used to control flotation cell level by adjusting valves located on the tails discharge of every bank of cells.

Column Flotation

Concentrate from the 1st rougher cells and the 1st cleaner cells gravitates to the Column Feed Pump Box. The duty Column Feed Pump, pumps the rougher concentrate to the column flotation cell. Process water is added to the feed hopper as required for density adjustment.

The Column Flotation Cell comprises a single operating column. Cleaner Concentrate gravitates to the Concentrate thickener. Column cleaner tailings in each case are pumped to the feed to the 1st cleaner scavenger circuit.

A float level indicator is used to control flotation cell level by adjusting a single valve located on the discharge of the flotation column.

The column is run with a high positive bias ratio on the wash water to ensure removal of gangue slimes from the final concentrate.

DeS Residue Flotation

Flotation of the DeS residue replaces the MSA Releach circuit in the original flowsheet and is designed to accomplish separate recovery of:

- Sulphur
- Galena
- Remaining Lead Minerals

While sulphur and galena recovery may not be required this provision is made to ensure any circulating load of sulphur and galena can be stripped from the recirculated DeS residue.

The flotation circuit will utilise the existing 2nd and 3rd cleaner cells to accomplish these duties. The slurry pH is adjusted with acid to pH 5.5 which will neutralise any residual soda ash and then lime is added to raise the pH to 6.5.

Sulphur is generated in the MSA leach by oxidation of galena. To an extent this sulphur will be oxidised in the MSA leach but nonetheless residual sulphur needs to be removed from the circuit to avoid build-up of a circulating load. This is accomplished in 2 x 8 m^3 flotation cells using just frother to achieve sulphur recovery.

Residual galena is removed next in the next 2 x 8 m^3 flotation cells using SIBX and frother to recover the lead.

The flotation tailings from these two steps are conditioned with NaHS and SIBX prior to flotation in 4 x 8 m^3 cells to recover the remaining lead minerals, predominantly anglesite and cerussite. This concentrate may be further cleaned by column flotation to remove gangue slimes.

Concentrate Thickening

A new larger Concentrate Thickener will be installed as part of the concentrator upgrade replacing two existing units that are badly corroded to produces a thickened concentrate underflow containing approximately 65% (w/w) solids. The concentrate is then pumped to the filtration circuit where a filter cake is produced as a feed to the Hydrometallurgical Facility.

Flocculant addition to final concentrate thickener is controlled using a flow element linked to the duty flocculant pump speed controller that receives a cascaded signal from the concentrate thickener bed level controller. The level controller will function to maintain a bed level in the concentrate thickener based on the flow ratio pre-set by the operator to supply the required flocculant addition rate.

The duty flocculant pump will have its delivery rate adjusted to maintain an operator input setpoint for the bed level in the thickener using the vendor supplied interface level transmitter to provide the control signal.

The speed of the duty Concentrate Thickener U/F Pump is controlled by the setpoint of the mass flow calculation on the pump discharge line.

Concentrate pumped to the Filter Feed Surge Tank which provides 12 hrs surge capacity between the thickening and filtration circuits.

Thickener overflow gravitates to the process water dam.

Concentrate Filtration

Thickened flotation concentrate slurry is pumped to the Concentrate Filter Feed Tank at a variable flow rate dependant on the plant head grade and the performance of the final concentrate thickener. The storage tank is used to provide surge capacity between the flotation circuit and the concentrate filter, allowing for maintenance of the filter. The slurry storage tank level is measured by an ultrasonic type level transmitter but is not controlled.

The Concentrate Filter Feed Pump deliver slurry into a manifold connected to the concentrate filter. The pump delivers concentrate slurry to the filter at an operating pressure of 6Bar.

The Concentrate Filter is a 37 chamber plate and frame filter with plates of 1.5 m x 1.5 m. The filter is fully automatic and is expected to operate on a 15 – 17 min cycle time producing up to a maximum 15 t/h of filter cake at a moisture content of 8%.

The filter is provided with two stainless steel bomb doors that are closed during cloth washing to prevent spray water entering the cons filter discharge chute. The doors are open during cake discharge.

The filter cake is washed three times with condensate to remove chloride from the contained moisture prior to feeding the concentrate to the Hydrometallurgical Facility.

The filter cake discharges to a hopper in batches up to 11 t every fifteen minutes when the filter is in operation. The concentrate falls under gravity from the Cons Filter Discharge Chute to the Concentrate Feeder.

Tailings

The Tailing Thickener is fed under gravity from the discharge of the final scavenger flotation cell in each train, and with minor flows from other sources for water reclaim.

Flocculant addition to tailings thickener is controlled using a flow element linked to the flocculant pump speed controller that receives a cascaded signal from the thickener bed level controller. The level controller will function to maintain a bed level in the tailings thickener based on the flow ratio pre-set by the operator to supply the required flocculant addition rate.

The duty flocculant pump will have its delivery rate adjusted to maintain an operator input setpoint for the bed level in the thickener using the vendor supplied interface level transmitter to provide the control signal.

The speed of the duty Tailings Thickener U/F Pump is controlled by the setpoint of the mass flow calculation on the pump discharge line. Tailings are pumped to the integrated waste facility.

The tailings thickener overflow gravitates to the process water dam.

Hydrometallurgical Facility

The Hydrometallurgical Facility will be constructed adjacent to the current concentrator plant and will receive concentrate via the currently installed concentrate filter press.

The Hydrometallurgical Facility includes:

- Concentrate preparation
- Leaching and solid/liquid separation
- Impurity removal and electrolyte preparation
- Lead electrowinning
- Bleed treatment
- Lead melting, casting and load-out
- Utilities and reagents
- Power station, heat recovery and steam generation.

The overall process plant is designed to achieve capacity and the availabilities shown in Table 26.

Table 26: Hydrometallurgical Facility Nominal Production and Availability Criteria

Production Criteria	Units	Parameter
Plant Concentrate Feed Rate Design	dry tpa	103,000
Concentrate Lead Grade	%	66 - 72
Concentrate Lead Grade -Design	%	70
Nominal Lead Metal Production Capacity	tpa	70,000
Recovery of lead from concentrate	%	97.0
Operating Schedule		
Days per Annum		365
Days per Week		7
Operating Hours	per day	24
Process Plant		
Design Availability	%	92
Operating Hours	per annum	8,059

The flowsheet for the facility is described in Figure 37 and Figure 35, with the general layout portrayed in Figure 39.

The following sections describe the elements of the Hydrometallurgical Facility.

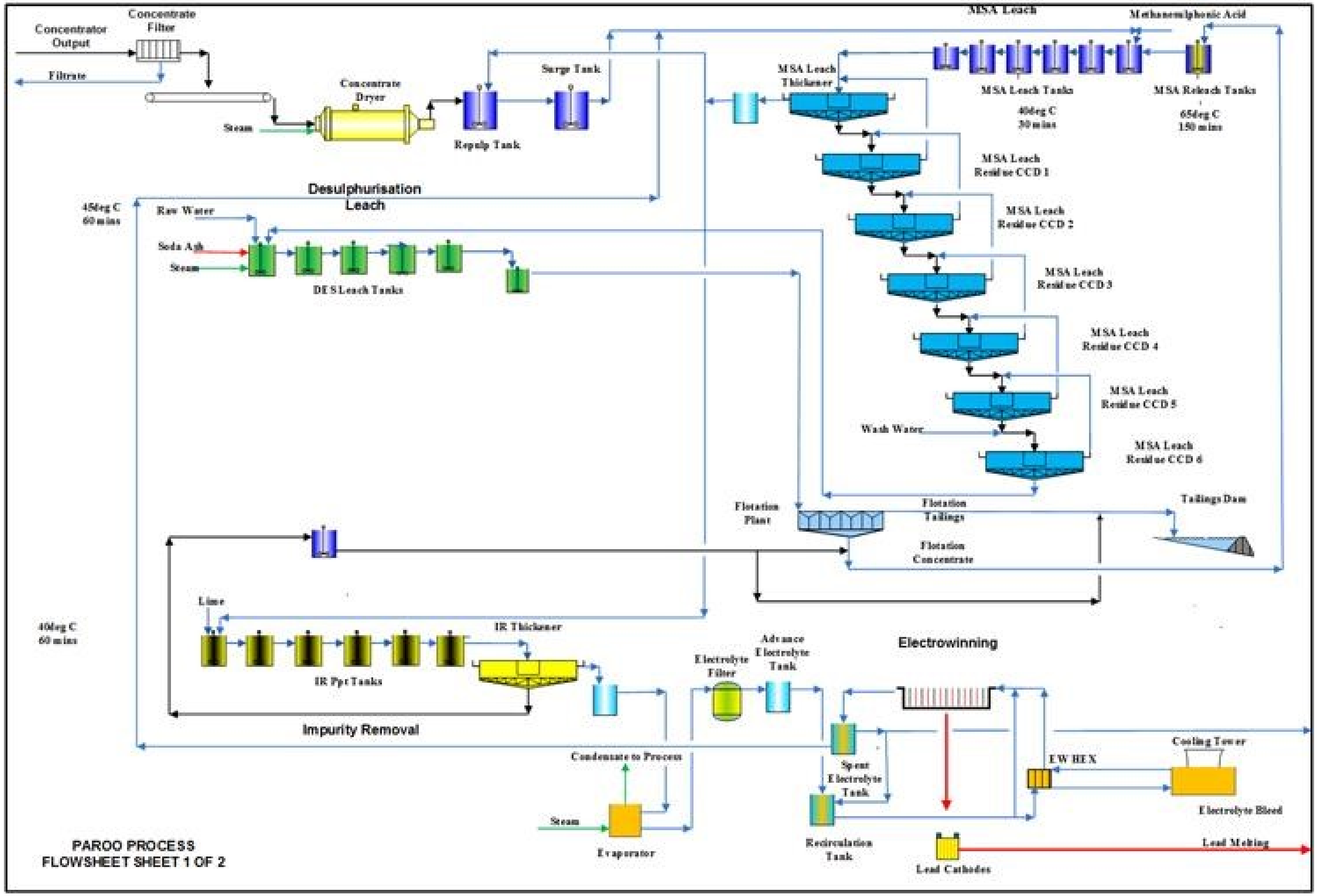


Figure 37: Hydrometallurgical Facility Flowsheet (1 of 2)

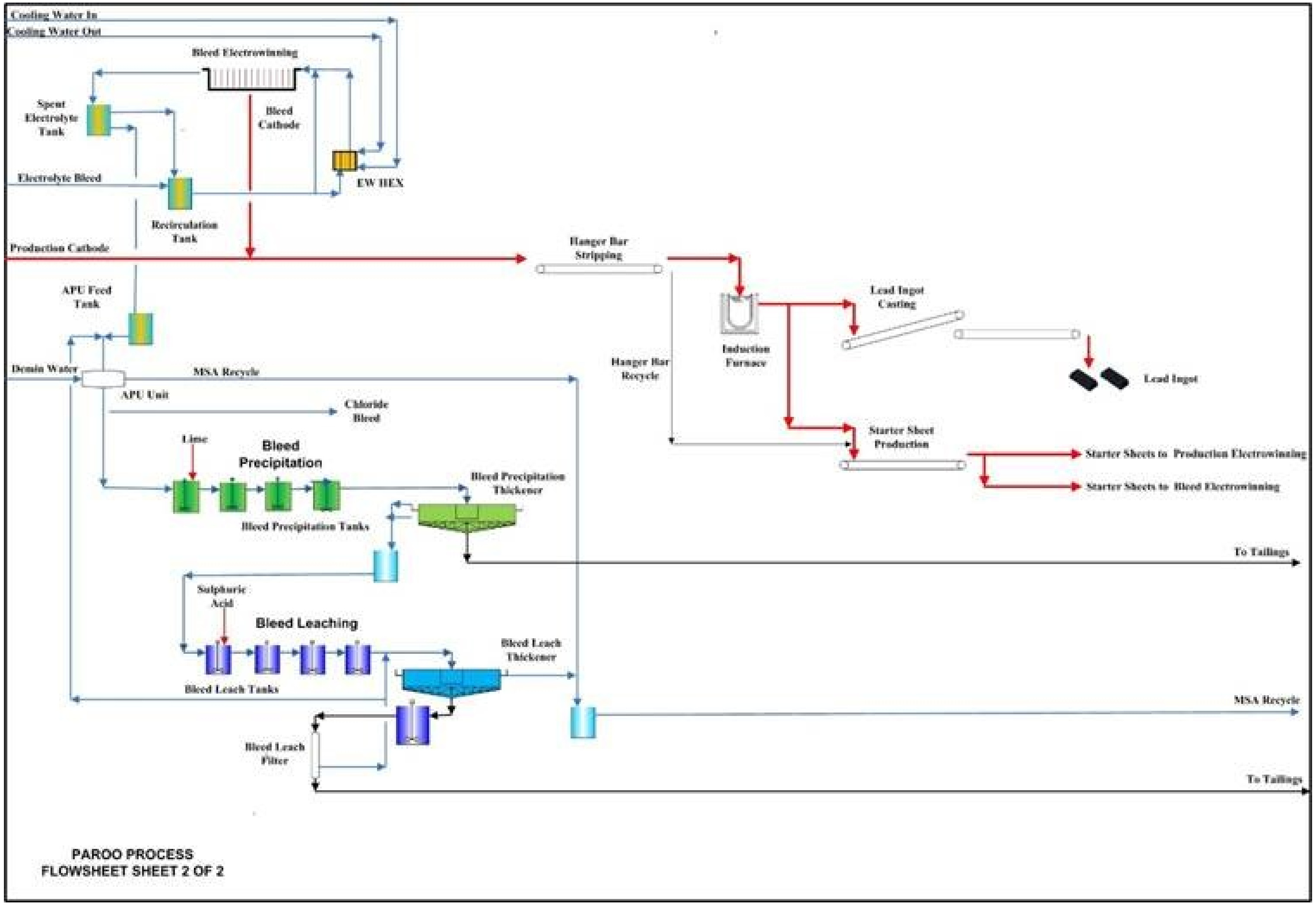


Figure 38: Hydrometallurgical Facility Flowsheet (2 of 2)

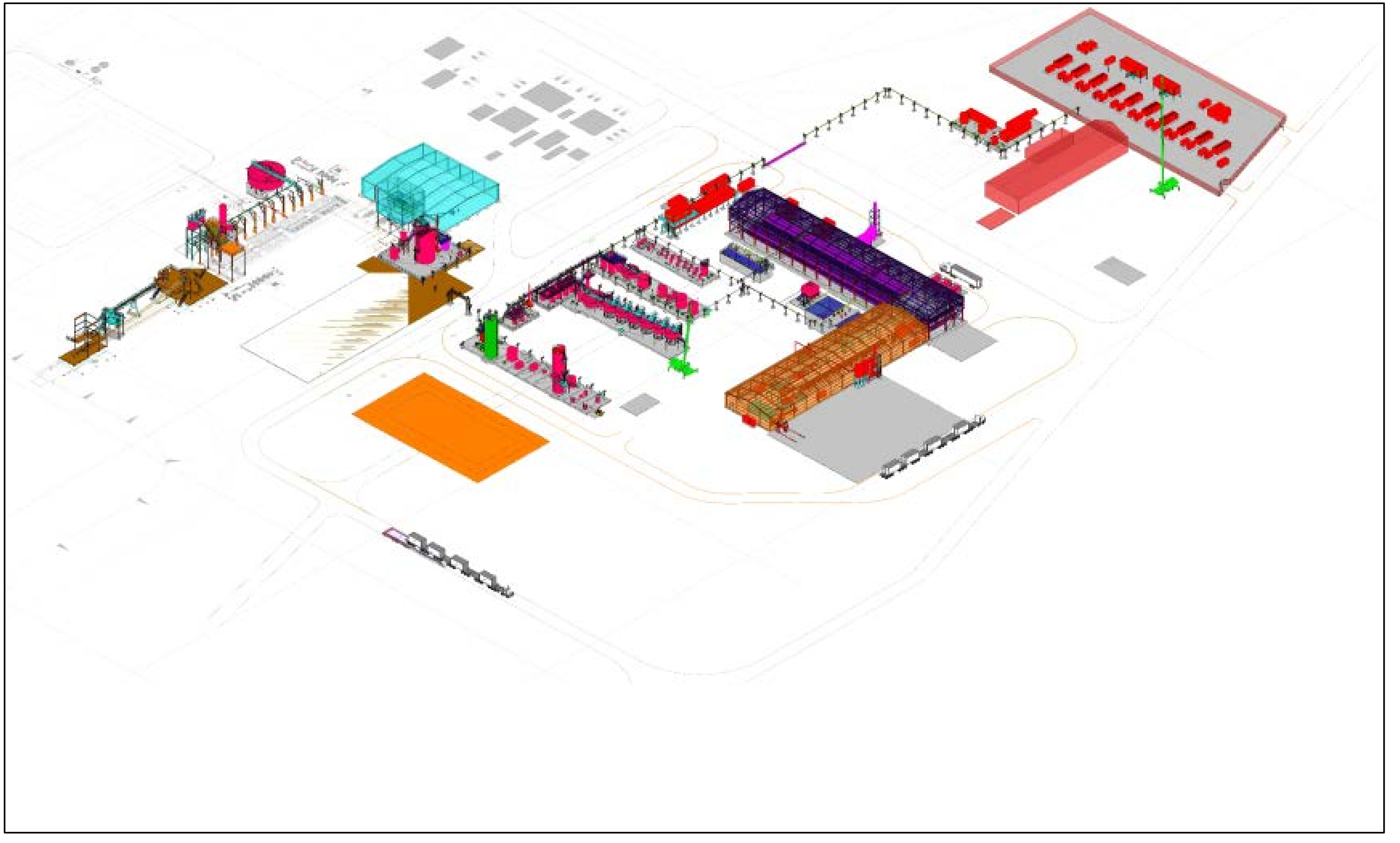


Figure 39: Hydrometallurgical Facility General Layout

The Hydrometallurgical Facility will consist of a series of acid and alkali leach circuits, impurity filtration, electro winning tank house and a cathode melt shop to produce 99.97% pure lead metal ingots at an estimated quantity of 70,000 t per annum.

The first leach circuit uses MSA as the key reagent. The majority of the lead carbonate is dissolved through this process achieving 99-100% leach extraction. The second leach circuit uses sodium carbonate as the key reagent which converts the lead residue from the first leach circuit to a lead carbonate. This product is fed to a repurposed part of the existing flotation concentrator, floating the lead carbonate which is fed into a train of MSA Releach tanks, discharging to the initial MSA leach circuit. The leaching reactions generate quantities of CO_2 gas, which is vented to a dedicated scrubber. An impurity removal circuit treats and adds lime to neutralise any residual acid to precipitate iron and aluminium ahead of the electrowinning circuit.

The electrolyte from the leach circuit is then pumped to the electro winning cells within the tank house. Gas generation (principally O_2), from the electrowinning cells is contained and directed to the dedicated scrubber to capture any airborne particulates. Lead from the electro winning cells is washed to remove any residual acid and then melted in an induction furnace in a fully automated lead ingot casting area. Exhaust gases from the induction furnace are directed to a baghouse to capture any airborne particulates.

The Hydrometallurgical Facilities will be bunded to contain any stormwater runoff and spills. The water will report to a new lined storage pond via sumps and sump pumps and will be recovered and reused in the process stream. A reverse osmosis plant will treat raw borefield water to supply various sections of the Hydrometallurgical Facility.

The Hydrometallurgical Facility will require additional electricity generation capacity. The existing diesel fuelled power station will remain as a back-up and emergency standby. A new natural gas fuelled (from an existing natural gas spur pipeline (“Magellan Lateral”) to the Goldfields Gas Transmission line), electricity generation system will be constructed with an installed capacity of 18 MW. The electrical distribution system will be upgraded to supply electricity loads at the existing accommodation village and the existing borefield, replacing local diesel fuelled generators.

Lead metal ingots will be transported from the Mine site for export.

14 Mineral Resource Estimate

The Mineral Resource estimate for the Mine consists of the main Magellan Hill deposits and the outlying Pizarro and Drake satellite deposits, located approximately 10 km south and 11 km south-west respectively from the existing mine infrastructure.

The Magellan Hill and the Pizarro Mineral Resources were estimated in 2014. The Mineral Resource was depleted for mining and processing activities up until the Mine was placed in care and maintenance in 2015. The Mineral Resource estimate was estimated by Optiro Pty Ltd and is current as at 31 December 2017.

The Magellan Hill, Pizarro and Drake Mineral Resources have been reported in accordance with the JORC Code 2012. CIM recognises “use of foreign code” including the JORC Code 2012.

For the Magellan Hill deposits and Pizarro, no additional exploration data have been incorporated into any of Mineral Resource estimates, which, depletion aside, remains unchanged from the 2014 estimate presented in SRK, 2015.

The Drake Mineral Resource was originally estimated in 2005 and reported in accordance with the 2004 JORC code. As part of the December 31, 2016 Mineral Resource update, Optiro Pty Ltd conducted a review of the Drake Mineral Resource estimate and associated documentation, concluding that there was sufficient confidence in the data, interpretation, estimation and available documentation, to support the reporting of the 2005 Drake Mineral Resource in accordance with the JORC 2012 reporting code. Data verification has been specifically outlined in Section 12 of this report.

14.1 Drillhole Database

The Magellan Hill (Magellan including Gama, Cano and Pinzon) deposits and the outlying Pizarro deposit Mineral Resource estimates were updated in late 2014, using the available RC and diamond drillholes, including available RC grade control data. Some RAB lithology information was used to inform geological interpretations at Pizarro, but no RAB assay data was used for estimation. No new resource drilling has been added to the Magellan Hill deposits.

The available data supporting the 2014 Magellan Hill and Pizarro estimates is presented in Table 27.

Table 27: Drillhole Statistics for Magellan Hill and Pizarro December 2014 Mineral Resource estimate

Hole type	Magellan Hill				Pizarro			
	Number of holes	Metres drilled	% holes	% metres	Number of holes	Metres drilled	% holes	% metres
Air core	24	804	0.03	0.16				
Blastholes	70,556	351,959	92.2	71.9				
Diamond drilling	41	2,287	0.05	0.47	4	402	1	3
Ditch Witch	325	325	0.42	0.07				
Piezometer	7	58	0.01	0.01				
Rotary air blast	514	10,456	0.67	2.14	227	7,484	57	49
Reverse Circulation	4,177	123,311	5.5	25.18	167	7,289	42	48
Rip lines	854	535	1.1	0.11				
Total	76,498	489,734			398	15,175		

Source: Optiro (2015)

At the Drake deposit, RAB assay data along with available RC and diamond drillhole data have been used to inform the 2005 Mineral Resource estimate. Additional RAB and diamond drilling has been completed at Drake post the 2005 estimate. On review, this additional data does not materially change the Mineral Resource estimate. The 2016 statistical test work confirmed that at Drake, the correlation between RAB and RC data was sufficient to support an Inferred Mineral Resource and reported in accordance with the JORC 2012 reporting code. Table 28 below shows the available data informing the 2005 Drake estimate and available for the 2016 review.

Table 28: Drake Mineral Resource public reporting at December 31 2015 (>2,1% Pb)

Drill Type	2005 Data			Post 2005 Data			All		
	Nos	Length (m)	Pb%	Nos	Length (m)	Pb%	Nos	Length (m)	Pb%
RAB	38	3.2	3.5	16	2.6	2.70	54	3.0	3.3
RC	21	4.6	3.87	21	2.7	3.44	42	3.7	3.7
DDH	1	6.0	1.88	5	5.3	6.92	6	5.4	6.0
All	60	3.7	3.62	42	3.0	3.94	102	3.4	3.73

Source: Optiro (2016)

Additional RC exploration drilling on the peripherals to the Pizarro and Drake deposits has been conducted in early 2015 as documented in sections 10 and 11. All of the updated drilling was outside of the currently defined Mineral Resource limits and hence the Mineral Resource estimates have not been updated for the 2015 drilling.

14.2 Geologic Model

The regional, local and prospect scale geology is described earlier in Chapter 7.

The 2016 Mineral Resource update for the Paroo Station deposits is based upon the 2014 geological model, whose characteristics are included in the following sections for completeness.

14.2.1 2014 Geological Modelling

The 3D geological modelling was carried out using ARANZ *Leapfrog* Geo Version 2.0.2.

The models are substantial in size and locally complex. They contain the main sedimentary units along with clay zones, cavities and dolomite horizons.

Magellan Hill

The overall package is tabular and sub horizontal; however, some units exhibit localised steeper dipping sections and distinct layering that can be split into hanging wall and footwall subdomains.

An oblique cross section through the NE–SW Cano-Magellan trend (looking east) is shown in Figure 40. Vertical exaggeration of 5 times has been applied to show detail.

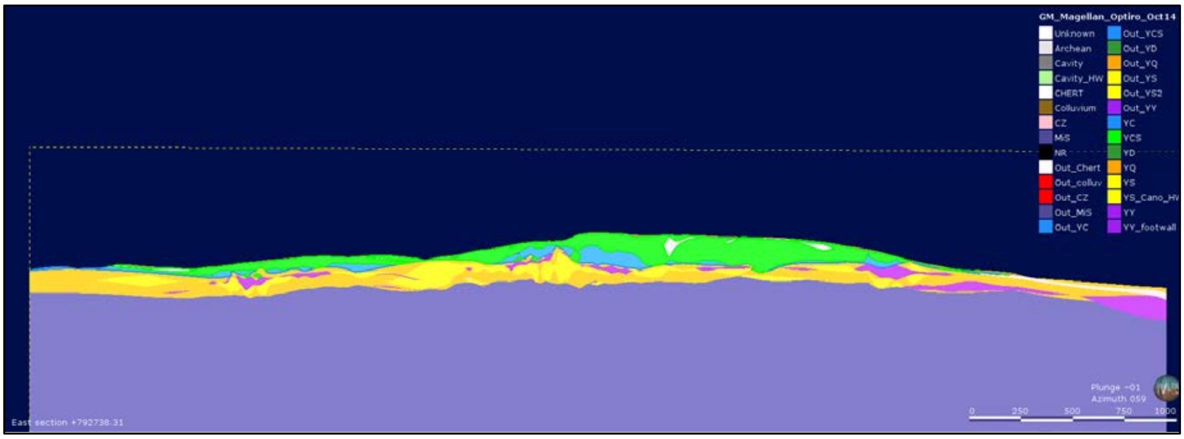


Figure 40: Magellan Hill geology model oblique cross section looking east

Source: Optiro (2015)

Pizarro

The geological package at Pizarro consists of a footwall YQ unit that is overlain/ intercalated with a shallow NE-dipping YD unit of approximately 100 m vertical thickness. A narrower YQ unit sits above the YD, and this has a complex zone of intercalated flat dipping lithologies.

The resultant interpretation is a mixture of discrete horizons that follow narrow zones of logged geology and these are often separated into main and footwall units. Larger and more extensive units, such as the footwall YQ and YD units, were modelled as more continuous strata that enclose the smaller zones of clay or chert.

The siltstone and silicified siltstone unit does not appear to be as laterally consistent as it is at Magellan Hill and it was modelled as such.

An oblique cross section view of the Pizarro model is shown in Figure 41. A vertical exaggeration of 5 times has been used to show detail.

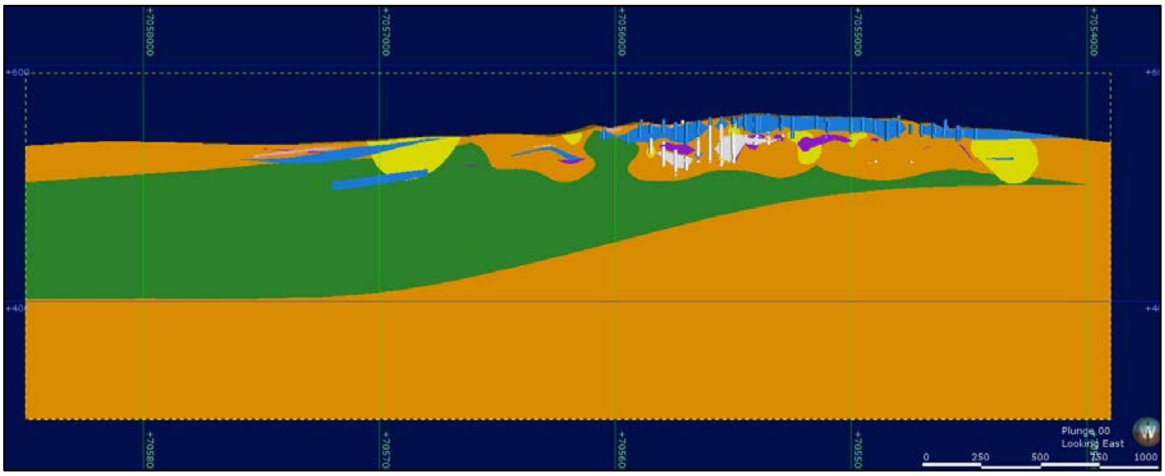


Figure 41: Pizarro geology model oblique cross section looking east

Source: Optiro (2015)

Drake

At Drake, the oxidised lead mineralisation that comprises the deposit is of a secondary nature and forms a supergene blanket which transects all of the rock types and appears to be similar to the Cano deposit (FinOre, 2005).

The mineralisation is hosted within highly weathered remnants of the Yelma Formation and the underlying Juderina Formation sediments. The mineralisation is pervasive and the current data does not provide any evidence for lithological control, other than the degree of weathering.

At the Drake deposit, the highest grade zone has a strong NE orientation which shows similar characteristics to the Magellan Hill structural controls.

14.3 Assay Capping and Compositing

For the Magellan Hill and Pizarro deposits, assay data was composited to 1.0 m downhole length, using the interpreted mineralised domain and lode as hard boundaries.

For Drake, the assay data was treated as a 'semi-soft' boundary by creating 1.0 m downhole composite samples which incorporated 0.25 m of waste either side of the mineralised interval.

14.3.1 Magellan Hill Statistics

The prior to and post-composite statistics demonstrate that there is no significant change in the metal during composite creation. Table 30 lists the Magellan Hill lodes and the respective number of samples within each area. Lode 3 is the major lode, representing a continuous zone across all three areas and encompasses the main part of the Magellan Hill mineralisation. Lode 0 is the waste or non-mineralised domain.

Table 29: Magellan Hill number of composite samples by area and lode code

Lode	Number of samples			Total
	Cano	Magellan	Pinzon	
0	11,857	53,698	9,301	74,856
3	10,411	24,594	2,831	37,836
4		25		25
5			63	63
6		137		137
7		20		20
8		17		17
9		12		12
10		13		13
11		0		8
13		14		14
14		6		6
16		23		23
17	66	0		66
18		8		8
19		33		33
Total	**6,799**	**78,600**	**12,195**	**113,137**

Source: Optiro (2015)

The statistics for the Magellan Hill deposits show that all domains have relatively low variance and skew, with low coefficients of variation ("CV"). Cano and Pinzon have similar statistical parameters, which are broadly similar to Magellan's, but the mean and median grades are slightly lower than at Magellan.

Table 30: Magellan Hill composite statistics by Zone

Lead (PB_PREF)	Global	Waste	(Mineralised (+1% Lead)		
			Cano	Magellan	Pinzon
Samples	113,137	74,856	10,477	24,902	2,902
Minimum	0.00	0.00	0.00	0.01	0.12
Maximum	66.60	20.36	57.20	66.60	48.23
Mean	1.79	0.21	4.44	5.13	4.28
Standard deviatio	3.86	0.37	5.27	5.53	4.77
CV	2.16	1.79	1.19	1.08	1.11
Variance	14.89	0.14	27.74	30.53	22.75
Skewness	4.24	15.35	2.87	2.60	3.00
Log mean	-1.38	-2.66	1.00	1.17	1.02
Log variance	5.33	2.77	0.94	0.95	0.83
Geometric mean	0.25	0.07	2.72	3.24	2.77
10%	0.01	0.01	0.95	1.04	0.99
20%	0.02	0.01	1.20	1.36	1.16
30%	0.06	0.02	1.46	1.75	1.43
40%	0.14	0.05	1.80	2.35	1.92
50%	0.29	0.08	2.33	3.11	2.59
60%	0.57	0.14	3.16	4.13	3.49
70%	1.09	0.22	4.50	5.60	4.53
80%	2.22	0.36	6.68	7.90	6.19
90%	5.30	0.59	10.99	12.10	9.87
95%	9.27	0.77	15.09	16.40	13.61
97.5%	13.54	0.89	19.60	20.63	17.16
99%	19.04	0.99	25.90	26.10	23.97

Source: Optiro (2015)

The box-and-whisker plot in Figure 42 provides a visual comparison of the respective lode grade distribution.

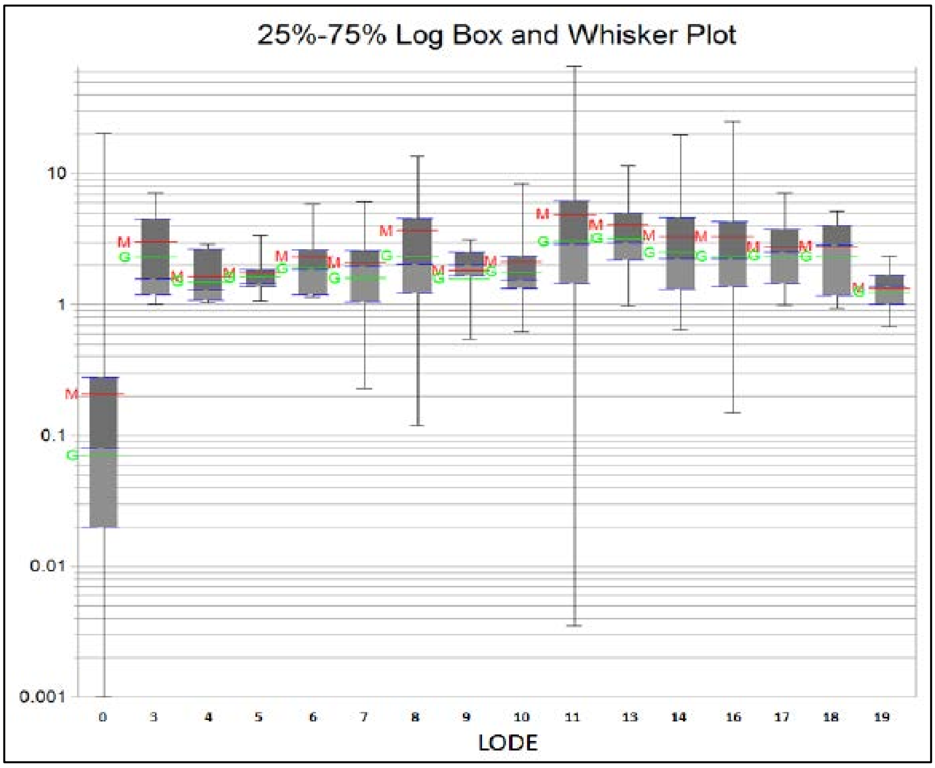


Figure 42: Magellan Hill box-and-whisker plot

Source: Optiro (2015)

14.3.2 Pizarro Statistics

An analysis of the prior to- and post-composite statistics demonstrates that there has been no significant change to the metal as the result of the compositing process. Table 31 lists the Pizarro lodes and the respective number of samples within each area. Lode 1 is the major mineralised domain and lode 0 is the waste or non-mineralised domain.

Table 31: Pizarro number of composite samples by lode code

Lode	Number of composite samples			
	0	1	10	Total
0	6,799			**6,799**
1		1,036		**1,036**
2		102		**102**
3		4		**4**
53		0	19	**19**
54			37	**37**
55			10	**10**
56			8	**8**
Total	**6,799**	**1,142**	**74**	**8,015**

Source: Optiro (2015)

The statistics for Pizarro are is shown in Table 32 and the overall statistical parameters are similar to the Magellan Hill (low variance, relatively low skew and very low coefficient of variance). Figure 43 illustrates the different grade distributions for the respective lodes at Pizarro.

Table 32: Pizarro composite statistics by Zone and Lode

Lead Grade % Statistics	All	Waste Zone/Lode 0	Low Grade Zone = 1 (Lode 1,2,3)				Subdomain		High Grade Zone				
			Zone 1	Lode 1	Lode 2	Lode 3	Non-min	Min	Zone 10	Lode 53	Lode 54	Lode 55	Lode 56
Samples	6,944	5,728	1,142	1,036	102	4	451	691	74	19	37	10	8
Minimum	0	0	0.03	0	0	1	0.03	0.32	1.29	7	1	5	9
Maximum	28.89	2.43	16.85	16.85	14.85	1.57	2.66	16.85	28.89	17.22	28.89	13.01	28.22
Mean	0.52	0.13	1.75	1.77	1.51	1.28	0.57	2.52	11.24	10.67	11.33	7.68	16.61
Standard Deviation	1.6	0.19	1.81	1.72	2.61	0.19	0.31	1.97	6.17	3.08	7.18	2.44	6.92
CV	3.11	1.45	1.04	0.97	1.73	0.15	0.55	0.78	0.55	0.29	0.63	0.32	0.42
Variance	2.57	0.04	3.28	2.95	6.79	0.04	0.098	3.865	38.05	9.5	51.49	5.97	47.85
Skewness	8.43	2.81	2.93	2.74	3.26	1.94	1.042	2.718	1.48	0.77	1.37	1.3	0.6
Log Samples	6,582	5,366	1,142	1,036	102	4	451	691	74	19	37	10	8
Log Mean	-2.37	-2.96	0.13	0.2	-0.55	0.24	-0.763	0.713	2.29	2.33	2.25	2	2.73
Log Variance	3.75	2.41	0.97	0.82	2.05	0.02	0.557	0.381	0.27	0.08	0.38	0.08	0.17
Geometric Mean	0.09	0.05	1.14	1.22	0.58	1.27	0.466	2.040	9.87	10.27	9.5	7.37	15.4
10%	0.01	0.01	0.35	0.42	0.1	1.16	0.16	1.05	6.08	7.03	5.36	5.85	9.26
20%	0.01	0.01	0.57	0.6	0.16	1.16	0.3	1.2	6.84	7.85	6.59	5.87	9.59
30%	0.02	0.01	0.76	0.8	0.23	1.2	0.38	1.36	7.51	9.12	7.33	6.08	12.27
40%	0.04	0.03	0.97	0.99	0.37	1.2	0.49	1.56	8.51	9.38	7.84	6.77	12.44
50%	0.08	0.05	1.18	1.23	0.63	1.2	0.58	1.85	9.38	9.63	8.58	7.03	17.49
60%	0.15	0.08	1.43	1.49	1.04	1.2	0.636	2.21	9.76	10.39	9.56	7.51	17.49
70%	0.27	0.14	1.87	1.95	1.28	1.2	0.74	2.8	12.27	12.03	10.51	9.72	20.15
80%	0.48	0.23	2.6	2.65	1.66	1.57	0.82	3.4311	14.62	13.97	17.37	9.76	23.49
90%	1.1	0.38	3.85	3.89	3.35	1.57	0.93	4.79	20.15	16.08	24.29	13.01	28.22
95%	2.23	0.52	5.2	5.11	6.61	1.57	0.98	6.28	28.22	17.22	28.71	13.01	28.22
97.50%	4.04	0.67	6.54	6.41	11.3	1.57	1.16	8.39	28.71	17.22	28.89	13.01	28.22
99%	7.84	0.78	8.93	8.66	12.6	1.57	1.43	10.87	28.89	17.22	28.89	13.01	28.22

Source: Optiro (2015)

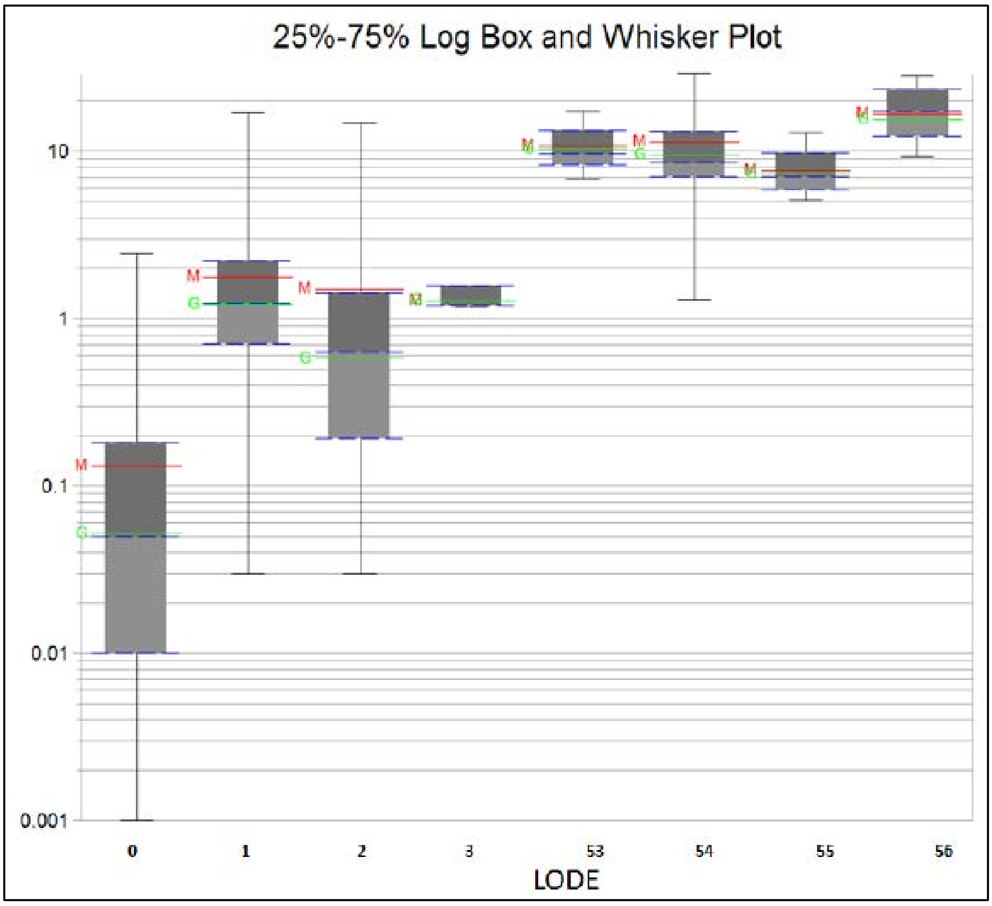


Figure 43: Pizarro box-and-whisker plot

Source: Optiro (2015)

14.3.3 Drake Statistics

The statistical summary for the Drake composite sample set including the 0.25 m waste buffer is presented in Table 33.

Table 33: Drake summary statistics all samples data available for 2005 Mineral Resource estimate

Statistic	Pb %
Number of samples	277
Minimum	0.04
Maximum	25.50
Mean	2.84
Median	1.56
Variance	10.77
Standard deviation	3.28
Coefficient of variation	1.16
Geometric mean	1.51
Sichel mean	3.20

Source: Optiro (2016)

14.3.4 Capping

Magellan Hill

The mineralisation statistics and grade distribution for the Magellan Hill deposits were reviewed and a cap of 35% Pb was independently derived using a combination of log-histogram and log-probability plots, as well as the disintegration of the grade distribution with increasing grades. This is the same cap value used in the 2011 estimate (SRK 2011).

For the waste domain (zone/lode=0), the grade distribution for the waste lode (zone/lode=0) is shown in Figure 44. A cap of 1.0% lead was applied to minimise the impact of the limited number of high grade samples (approximately 99.09% of samples have composite grades less than 1% lead).

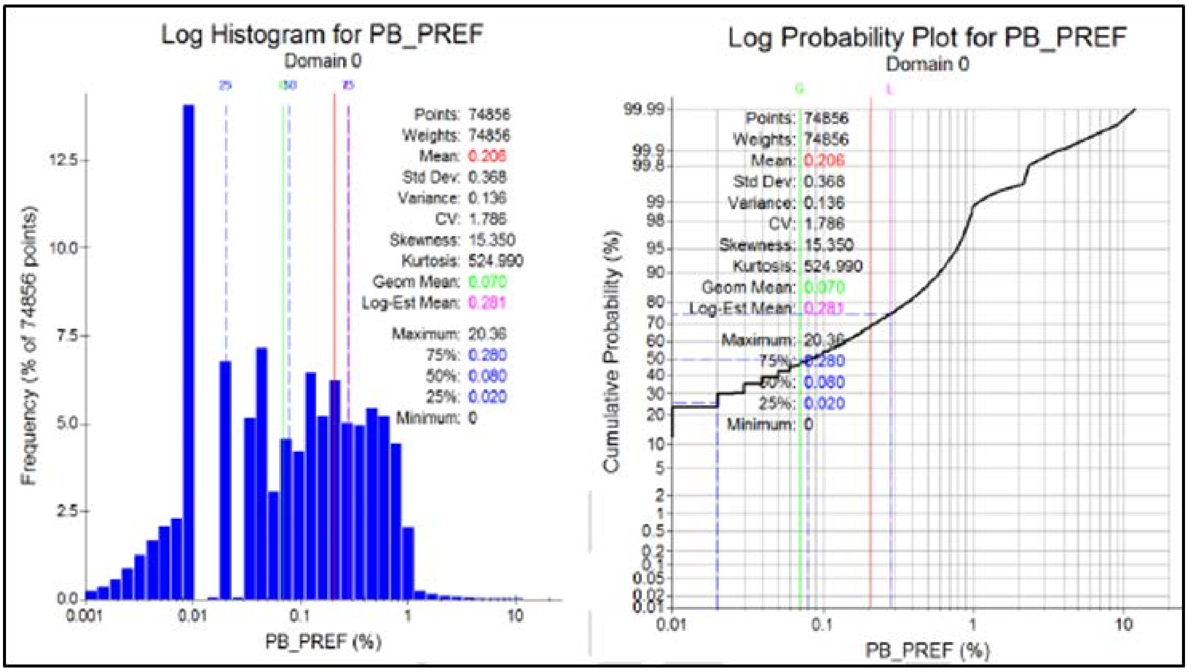


Figure 44: Magellan Hill waste grade distribution

Source: Optiro (2015)

The grade distribution and low coefficient of variation for the Pizarro mineralisation was such that no cap was applied.

Drake

For Drake, a cap of 20% Pb was derived from the grade distribution for the mineralised domain and applied to the composited sample data prior to grade interpolation. A single composite sample of 25.5% Pb was capped to 20% Pb.

14.4 Density

14.4.1 Magellan Hill

Dry bulk density for the Magellan Hill deposits is applied using a lithology based grade-density algorithm that has been fully documented in SRK (2011) and is unchanged. The density-Pb correlation is based on the testing of whole diamond drill cores in an off-site accredited laboratory.

Table 34: Magellan Hill bulk density - lithology based algorithm

Unit	Density regression	Density range (t/m^3)
Colluvium	Pb x 0.3 + 1.60	1.60 - 1.99
Cavity (0 density and 0.0% lead grade)	0.00	0.00
Silcretised quartz-clay breccia	Pb x 0.3 + 2.00	2.00 - 2.45
Quartz-clay breccia	Pb x 0.3 + 1.90	1.90 - 2.59
Dolomite	Pb x 0.3 + 2.00	2.00 - 2.45
Chert	Pb x 0.3 + 1.90	1.90 - 2.32
Clay	Pb x 0.3 + 1.70	1.70 - 2.32
Siltstone	Pb x 0.3 + 1.70	1.70 - 2.39
Sandstone	Pb x 0.3 + 2.00	2.00 - 2.69
Maraloou Shale	Pb x 0.3 + 2.10	2.10 - 2.61

Source: Optiro (2015)

14.4.2 Pizarro

Dry bulk density for the Pizarro deposit is assigned based on the modelled lithology that has been fully documented in SRK (2011) and is based on the testing of whole diamond drill cores in an off-site accredited laboratory.

Table 35: Pizarro bulk density - lithology based values

Unit	Density range (t/m^3)
Colluvium	1.6
Indurated material	2.0
Cavity (0 density and 0.0% lead grade)	0.0
Silicretised quartz-clay breccia	1.9
Quartz-clay breccia	1.9
Siltstone/Shale	2.0
Clay	1.7
Chert Laterite	1.9
Siltstone	2.0
Dolomite	2.0
Sandstone	2.0

Source: Optiro (2015)

14.4.3 Drake

No bulk density data is available for the Drake deposit due to the nature of the RC and RAB drilling techniques. The following bulk density algorithm obtained from the Cano deposit prior to 2005 was applied:

Dry Bulk density = 1.8 + (0.04 x Pb%)

The Cano 2005 lead-density correlation results in density values ranging from 1.8 to 2.6 t/m^3 and this density range is considered representative of the Paroo Station deposits and appropriate for an Inferred Mineral Resource.

14.5 Variogram Analysis and Modelling

For the Magellan Hill deposits, the traditional variography highlighted an unexpectedly high nugget structure (approximately 35% to 45% of the total sill) and although the horizontal directions were prominent, they were not conclusive. As a result, the grade continuity was modelled using normal-score variography which provided more conclusive variogram directions. The resultant variogram models were back-transformed from Gaussian to traditional variogram models for use in estimation.

For Pizarro, indicator variography at 1.0% Pb was prepared to differentiate sub 1% and +1% material, and subsequent traditional variography was prepared for the respective subdomains.

Variography at Drake was modelled using the median indicator variogram, which although poorly structured, coincided with the observed geology.

14.5.1 Magellan Hill

As the most dominant mineralised domain, variography was prepared for the major lode (lode=3) only as the other lodes did not have sufficient samples for reliable directional variography. Separate variograms were prepared for the three resource areas (Cano, Magellan and Pinzon).

The final variogram models were then back transformed from Gaussian to traditional variogram models. All of the variography had a horizontal dip plane, with 2 dominant directions in the horizontal plane. At Cano, the major direction of continuity was towards the north-west and the intermediate direction towards the north east. At Magellan and Pinzon, the direction of major continuity was oriented towards the north east, with the intermediate direction orientated towards the north-west. The back-transformed models for the three Magellan Hill deposits have similar sills, but the overall variogram ranges and anisotropies are significantly different. The major direction at Cano is 2.7:1 times that of the intermediate direction, whereas at Magellan and Pinzon, the ratio of major to intermediate axis is almost 1:1.

14.5.2 Pizarro

At Pizarro, only the main lode (lode 1 and 2) had sufficient samples to create robust variography, and this was applied to all other domains, including the non-mineralised/waste domain.

Three dimensional consistent interpretations at 1% Pb included variable amounts of sub-1% Pb material. To differentiate the two populations, indicator variography was prepared at a +1% Pb categorical indicator. The dip plane for the indicator variogram at Pizarro was horizontal, with the maximum direction of continuity orientated north-west and the intermediate direction towards the south west.

A threshold of 0.5 was subsequently used to discriminate between the sub 1% Pb subdomain (flagged as “NONM”) and +1% Pb subdomain (flagged as “MIN”) were modelled. Variography for the two domains was then prepared. The directions of continuity for both subdomains were identical and were aligned parallel to the indicator variogram, with the only difference between the two being that the nugget structure for the +1% subdomain was slightly higher than the nugget for the non-mineralised subdomain.

14.5.3 Drake

Median indicator variograms were generated for the composited Drake assay data. The resultant variography was poorly structured, primarily due to the limited data (277 composites). However, the variography did broadly coincide with the observed geological continuity and provided guidance towards selecting an appropriate search ellipse for grade interpolation (FinOre, 2005).

14.6 Block Modelling

The 2014 Magellan Hill block model was constructed from a 'first-principles' basis by Optiro and reported in accordance with JORC 2012 and NI43-101 as reported in SRK (2015). The Mineral Resource was updated in 2016 to include all depletion due to mining and processing activities before the mine was placed on care and maintenance in February 2015.

The Pizarro block model was reported in accordance with JORC 2012 and NI43-101 as reported in SRK (2015).

The Drake block model was estimated by FinOre in 2005 and reported in accordance with JORC 2004 and NI43-101 at the time. In 2016, Optiro conducted a review of the Drake 2005 Mineral Resource, finding that there was sufficient documentation and confidence in the estimate, to update the report of the 2005 estimate in accordance with JORC 2012.

14.6.1 Depletion for Mining

Optiro prepared the Mineral Resource estimate for the Paroo Station deposits when the operation was being mined in 2014, and depleted the model for mining to the November 30, 2014.

Due to the long term decline in the lead metal price, the Paroo Station was placed on care and maintenance in early 2015. Mining ceased on the January 16, 2015, and processing of ROM stocks ceased on the February 2, 2015 (Optiro, 2016).

In 2016, Optiro depleted the block model and reported the insitu Mineral Resource and available surface stocks, to the December 31, 2015, as detailed in the following sections.

Magellan Hill

The 2014 Mineral Resource for the Paroo Station deposits was initially depleted for mining to the November 30, 2014. Between November 2014 and the cessation of mining on the January 16, 2015, mining was undertaken in the Magellan pit exclusively and an excavated pit survey was completed when mining ceased.

In January 2016, Optiro depleted the Mineral Resource for mining between November 30, 2014 and January 16, 2015, and updated the remaining surface stocks for processing up until the February 2, 2015, when processing ceased (refer Figure 45 below).

The Cano pit survey is unchanged from that used in the December 2014 Mineral Resource update sand no additional depletion was required. Mining has not commenced at the Pinzon deposit.

The updated model is the same block model fully documented in the 2014 Mineral Resource documentation, but with the "MINED" field updated for mining to the end of December 2015 (Optiro, 2016).

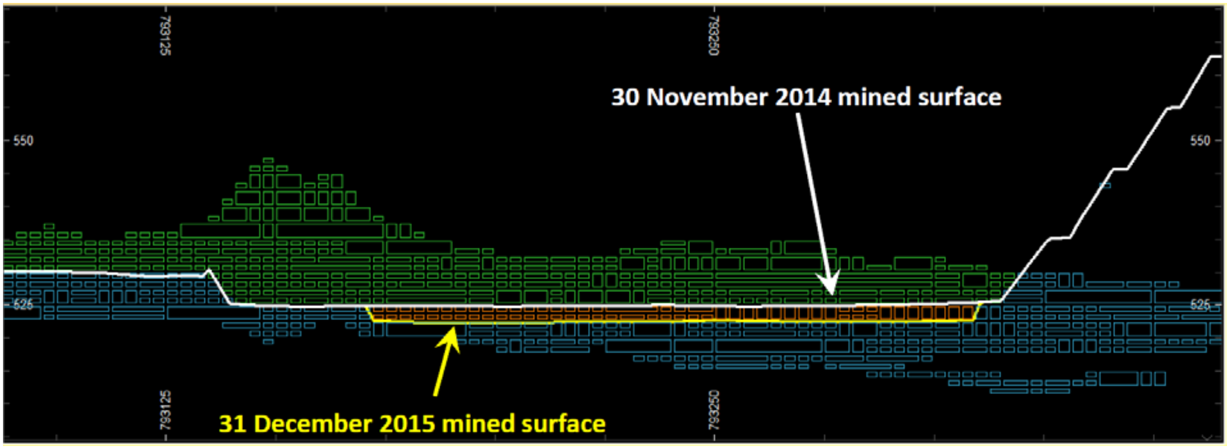


Figure 45: Section 7063260mN, looking west through Magellan pit showing depleted block model and mined surfaces

Source: Optiro (2016)

Pizarro

There has been no mining at Pizarro and there have been no changes to the Pizarro block model since the Mineral Resource update in 2014.

Drake

There has been no mining at Drake and there have been no changes to the Drake block model since the Mineral Resource update in 2014. However, additional documentation has been prepared to support the reporting of the Mineral Resource in accordance with the JORC 2012 reporting code.

14.6.2 Magellan Hill Block Model

The Magellan, Cano and Pinzon deposits were modelled in a single block model and the block model prototype parameters is shown in Table 36. These parameters were derived by kriging neighbourhood analysis (KNA) testing in 2014. Mineralisation was defined by a 1% Pb boundary which was treated as a 'hard' estimation boundary.

Table 36: Magellan Hill model prototype

	X	Y	Z
Origin	791050	7061200	490
Parent cell	25	25	2.5
Minimum subcell	3.125	3.125	1.25
Number of parent cells	156	140	34

Source: Optiro (2015)

14.6.3 Pizarro Block Model

A 2-stage bock model estimation approach was employed at Pizarro, whereby regions designated as 'exploration area' and supported by wider spaced exploration drilling (drillhole spacing greater than 100 mE x 100 mN) was estimated using 100 mE x 100 mN x 5 mRL parent cell size. The mineralisation and adjacent areas supported by closer spaced drilling (nominally less than or approaching 50 mE x 50 mN) was estimated using the KNA defined parent cell size of 25 mE x 25 mN x 2.5 mRL (Table 37).

Mineralisation was defined by a 1% Pb boundary, which encapsulated a high grade +10% Pb boundary. All mineralisation boundaries were treated as 'hard' estimation boundaries.

Table 37: Pizarro model prototype

	Infill model			Exploration model		
	X	Y	Z	X	Y	Z
Origin	789537.5	705937.5	480	789537.5	705937.5	480
Parent cell	25	25	2.5	100	100	5.0
Minimum subcell	3.125	3.124	1.25	12.5	12.5	0.5
Number of parent cells	176	176	60	44	44	30

Source: Optiro (2015)

14.6.4 Drake Block Model

Parent cells were created with dimensions of 25 mE x 25 mN x 2.5 mRL. Sub-blocking of the model to 5 mE x 5 mN x 2.5 mRL was completed to provide extra control to the volume reporting. Further sub-celling to 0.5 m in the vertical was used to achieve more accurate volume control.

Only blocks flagged as being within the 1% Pb wireframe were interpolated with grade (FinOre, 2005), with the boundary treated as a 'semi-soft' boundary incorporating 0.25 m of waste dilution either side of the 1% Pb contact.

14.7 Estimation Methodology

14.7.1 Magellan Hill

For the 2014 model update, grade estimation was controlled by the area and lode fields. The composite data and model was coded by an 'area' field which differentiated Magellan, Cano and Pinzon, but this was treated as a soft 'boundary' to ensure that no edge artefacts were introduced at the boundaries. This was achieved by translating the area boundaries by approximately half the respective modelled variogram distance in the X-Y plane, and making samples within the overlap area available to both adjoining areas (Figure 46).

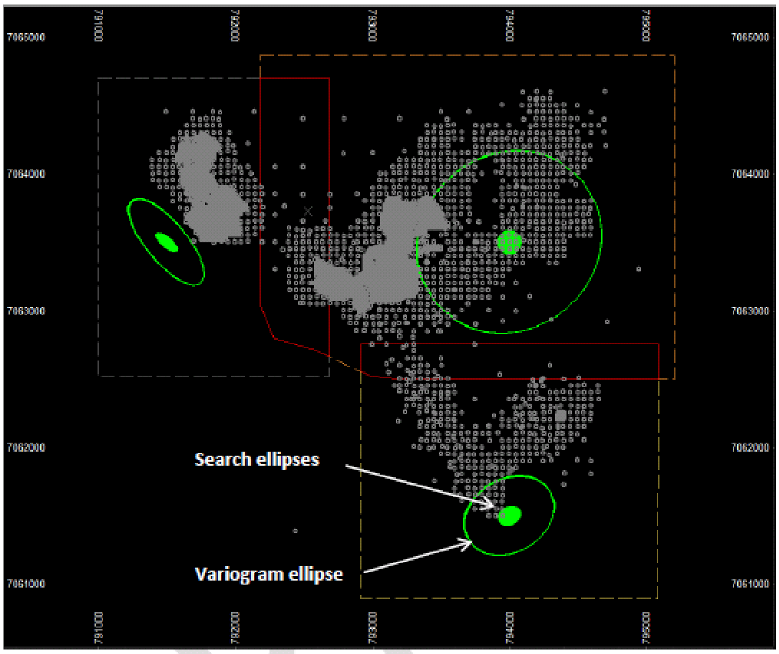


Figure 46: Magellan Hill expanded estimation overlap areas (red), search (solid discs) and variogram ellipses (lines)

Source: Optiro (2015)

A multi-pass search method was used for grade estimation and is summarised in Table 38.

Table 38: Magellan Hill search parameters

Search	Zone/ Lode	Area	Datamine Rotations 3-1-3	First Pass		Second Pass		Third Pass		Samples per hole
				Search 1x2x3	No. of samples	Search 1x2x3	No. of samples	Search 1x2x3	No. of samples	
Waste	0	All	000°,000°, -90°	500x 20x 40	4-40	750x 300x 60	4-40	1,500x 600x 120	4-40	NA
Well drilled/ Sampled	1/3	Cano LG	000°,000°, -130°	100x 37.5x 5	8-48	150x 56.25x 7.5	8-48	300x 112.5x 15	8-48	4
		Mag LG	000°,000°, -40°	87.5x 87.5x 5	8-44	131.25x 131.25x 7.5	8-44	262.5x 262.5x 15	8-44	4
		Pnzn LG	000°,000°, -30°	87.5x 67.5x 5	8-40	131.75x 101.25x 7.5	8-40	262.5x 202.5x 15	8-40	4
Poorly sampled lodes <5 drillholes and/or <12 samples	1/>3	Cano LG	000°,000°, -130°	100x 37.5x 5	2-48	150x 56.25x 7.5	2-48	300x 112.5x 15	2-48	NA
		Mag LG	000°,000°, -40°	87.5x 87.5x 5	2-44	131.25x 131.25x 7.5	2-44	262.5x 262.5x 15	2-44	NA
		Pnzn LG	000°,000°, -30°	87.5x 67.5x 5	2-40	131.75x 101.25x 7.5	2-40	262.5x 202.5x 15	2-40	NA

Source: Optiro (2015)

Ordinary kriging was used for grade estimation. The search directions were based on the variography and the search distance for the first pass based on the results of the KNA.

For the poorly sampled lodes, the minimum number of samples was reduced to 2, to optimise the proportion of cells that received and estimate.

For any cell (whether mineralised or not) that was not estimated after the third pass, the nearest estimated grade was assigned and the PB_SV field set to '4'.

Within the waste domain there were a small number of cells that either did not receive a grade estimate or received a negative grade estimate. If the cell did not receive an estimate the PB_SV field was set to '5' or if negative, the PB_SV field set to '6' and a default lead grade of 0.2% lead was assigned.

Parent cell estimation was used for all estimates at Magellan Hill.

14.7.2 Pizarro

Grade estimation for the +1% Pb mineralised domain used a categorical indicator estimation method to differentiate very low grade from elevated mineralised material within the 1% Pb boundary. An indicator grade of 1% was selected and the proportion above/below this indicator was estimated using a multiple pass search strateg, with the ellipse orientation controlled by a dynamic anisotropy process.

A threshold of 0.5 was used to discriminate below 1% from above 1% Pb subdomains within the overall mineralised domain. Pb grades for each subdomain were then separately estimated using the search ellipse. The +10% Pb grade domain was estimated using a conventional ordinary kriging with a single search ellipse. No restriction on the number of samples used per drillhole was used.

Parent cell estimation was used for all estimates at Pizarro. The search parameters are summarised in Table 2.

Table 39: Pizarro search parameters

	Datamine Rotations	First Pass		Second Pass		Third Pass		Dynamic Anisotropy
Zone/Lode	3-1-3	Search (1-2-3)	Nos. Samples	Search (1-2-3)	Nos. Samples	Search (1-2-3)	Nos. Samples	
Categorical Indicator	0, 0, 55	175 x 175 x 5	4 to 12	262.5 x 262.5 x 7.5	4 to 12	525 x 525 x 15	4 to 12	Yes
SUB_DOM = MIN	0, 0, 60	200 x 50 x 15	8 to 32	300 x 75 x 22.5	8 to 32	600 x 150 x 45	4 to 32	No
SUB_DOM = NONM	0, 0, 60	200 x 50 x 15	8 to 32	300 x 75 x 22.5	8 to 32	600 x 150 x 45	4 to 32	No

14.7.3 Drake

Grade estimation used 1.0 m composites and a 0.25 m softening skin, to inform an inverse distance (power of 2.5) interpolation technique.

A flat search ellipse of 150 x 80 x 3 m was used and composited samples were length weighted during the grade interpolation process.

No more than three samples from any one drillhole were used per block estimate and all subcells received their parent cell grade estimate.

Two search passes were used during the grade interpolation. The first pass was carried out using the previously outlined ellipse and was followed by a second pass search for those blocks not estimated in the first pass, using an ellipse double the size of the first.

14.7.4 Stockpiles

Estimates of the stockpiles have been produced from actual mine production and survey figures obtained from the Magellan and Cano open pits and production estimates of ROM stocks at the end of processing on February 2, 2015 (Table 40).

The ROM ‘finger’ stockpiles constructed during January 2015 were partially depleted for processing and the remaining volume/t were not surveyed. The ROM finger tonnage remaining is solely based on the claimed t and grade. Any variance between the predicted and actual ROM fingers is not considered to be material (Optiro, 2016).

Prior to 2016, the mineralised waste (green) stockpiles were constructed from material below the processing cut-off of 2.5% Pb, but above the incremental cut-off 2.1% Pb. With the cessation of mining and processing in 2015, a review of the stockpiles included the mineralised waste stockpile material into the Mineral Resource, as this material is above the 2.1% lead reporting cut-off and is available for future processing.

Table 40: Stockpile inventory as at December 31, 2015

Location	Stockpile	Ore Tonnes	Lead Grade	Lead Tonnes
ROM Fingers	Fingers B39 and C20	19,944	5.9 %	1,176
Other Ore Stockpiles	LG Spile	657,318	2.90 %	19,062
	FINGERZ001	50,740	4.06%	2,060
	MGSPILE	343,810	3.80 %	13,065
	Total Other Stockpiles	**1,051,868**	**3.25 %**	**34,187**
GREEN MW Stockpiles	MWGREENCAN	126,411	2.32 %	2,933
	MWGREENMAG	145,368	2.36 %	3,431
	MWGREENMAG2	177,832	2.47 %	4,392
	Total MW Green	**449,611**	**2.39 %**	**10,756**
Total stockpiles (including ROM Fingers)		**1,521,423**	**3.03 %**	**46,119**

Source: Optiro (2016)

Table entries are rounded to reflect the precision of the estimate and differences may occur due to this rounding.

14.8 Model Validation

Model validation consisted of initial on-screen visual validation of the estimate. This was followed by a comparison between global naïve and declustered composite averages with the block model averages (comparative statistics). The final validation step was the preparation of swath plots by easting and northing showing the naïve and declustered composite against the block model averages.

14.8.1 Visual Comparison

Magellan Hill

Initial validation was undertaken by visually inspecting easting and northing sections of the composite sample data and the estimated block model, which raised no significant concerns (Figure 47).

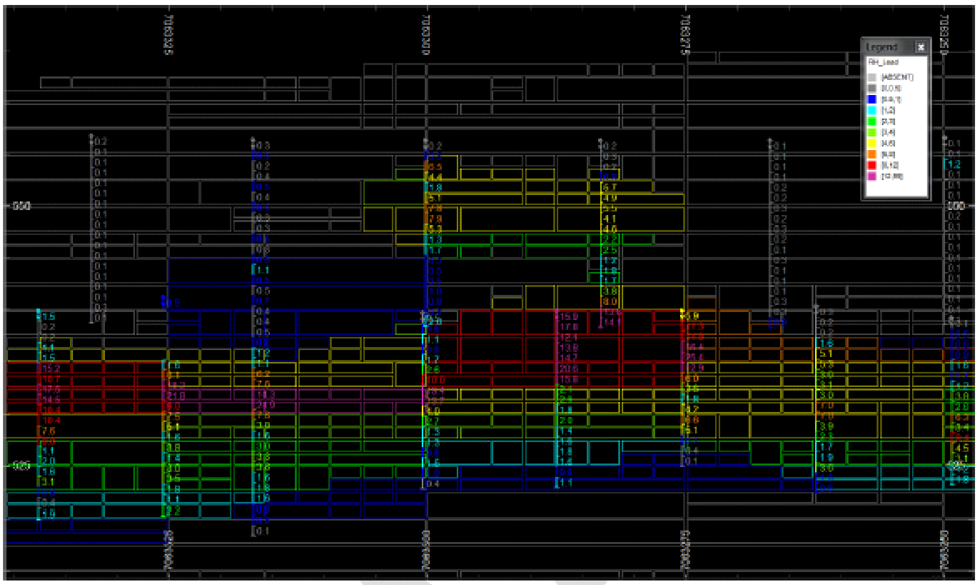


Figure 47: Magellan section 793066mE showing composite samples and block model

Source: Optiro (2015)

Pizarro

Initial validation was undertaken by visually inspecting easting and northing sections of the composite sample data and the estimated block model (Figure 48) which identified no significant concerns.

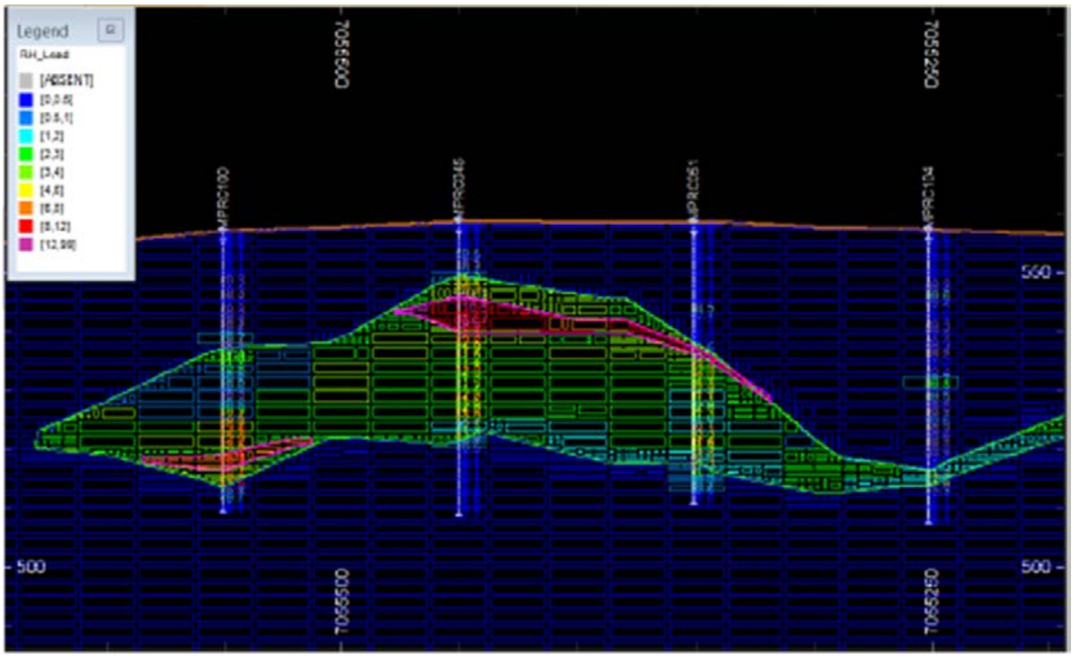


Figure 48: Pizarro section showing composite sample and block model

Source: Optiro (2015)

Drake

Visual validation of the Drake estimate confirms that there is good correlation between the estimate and available drilling. (Figure 49).

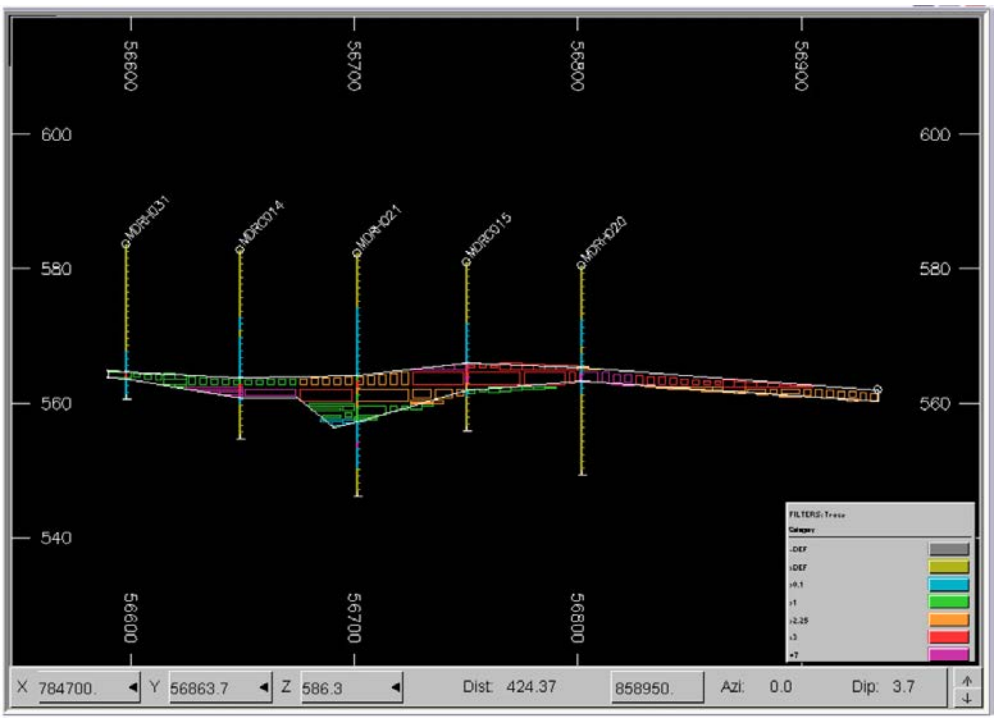


Figure 49: Drake section showing composite sample and block model

Source: FinOre (2005)

14.8.2 Comparative Statistics

Magellan Hill

The sample averages were compared against global estimated averages on a lode by lode basis, as well as reporting the estimate by search pass (Table 41).

There is good correlation between the global comparison and the comparison by search pass. The correlations are poorer for passes 2, 3 and 4 as expected, as these represent much more discrete areas that are not as well supported by sampling (i.e. extrapolation).

Table 41: Global validation - Magellan Hill

			Model Average								Relative Difference								
Lode	**Sample Average**		**Global**	**Search Pass**							**Global**		**Search Pass**						
	Naïve	**Declustered**		**1**	**2**	**3**	**4**	**5**	**6**	**9**	**vs Naïve**	**vs Declustered**	**1**	**2**	**3**	**4**	**5**	**6**	**9**
0	0.20		0.12	0.13	0.11	0.08	0.03	0.20	0.20		38%		31%	43%	59%	85%	-2%	-2%	
3	4.88	4.36	4.14	4.29	3.39	2.88	2.76			2.17	15%	5%	12%	30%	41%	44%			56%
4	4.07		3.80	3.80	1.95						7%		6%	52%					
5	3.36		3.31	3.31	2.79						2%		1%	17%					
6	3.32		3.15	3.18	2.97	3.34					5%		4%	10%	-1%				
7	2.77		2.86	2.87	1.99	2.48					-3%		-4%	28%	10%				
8	2.78		2.84	2.84	2.90						-2%		-2%	-4%					
9	1.34		1.42	1.42	1.11						-6%		-6%	17%					
10	3.03		3.24	3.24							-7%		-7%						
11	1.63		1.53	1.52	2.20						6%		7%	-35%					
13	1.73		1.94	1.96	1.27	1.44					-12%		-13%	27%	17%				
14	2.32		2.17	2.24	1.84	2.11					6%		3%	21%	9%				
16	2.10		2.21	2.21							-5%		-5%						
17	3.69		3.32	3.69	2.44	2.33	1.61				10%		0%	34%	37%	56%			
18	1.81		1.93	1.93							-6%		-6%						
19	2.14		1.90	1.90							11%		11%						

Source: Optiro (2015)

Pizarro

The sample domain averages were compared against the block model average on a lode by lode basis, both globally and by search pass.

As at Magellan Hill, the comparison for the composite sample average and estimate average within the first search pass correlate well with the sample grades for each area and lode combination, and the correlations are poorer for the other passes as a function of the degree of extrapolation (Table 42).

Table 42: Global Validation – Pizarro

Estimate type	Lode	Comp. Mean Lead %	Model Grade							Relative difference %						
Traditional			**Global**	**Search 1**	**Search 2**	**Search 3**	**Search 4**	**Search 5**	**Search 6**	**Global**	**Search 1**	**Search 2**	**Search 3**	**Search 4**	**Search 5**	**Search 6**
Traditional	0	0.13	0.07	0.08	0.06	0.06		0.01	0.05	50%	36%	54%	57%		96%	62%
Cat. Indicator	1	1.77	1.82	1.79	1.98	2.20	0.29			-2%	-1%	-12%	-24%	84%		
	2	1.51	1.33	1.19	1.39	1.83	2.40			12%	21%	8%	-21%	-59%		
Traditional	3	1.28	1.20						1.20	6%						6%
	53	10.67	10.65	10.7	9.2					0%	0%	14%				
	54	11.33	10.40	10.4	12.6					8%	8%	-11%				
	55	7.68	7.80	7.8	7.3					-2%	-2%	4%				
	56	16.61	17.15	17.2	11.9					-3%	-4%	29%				
LODE = 1 Transitional Lead Model (PB-V1)																
Traditional	1	1.77	1.81	1.78	2.00	2.13	1.47			-2%	-1%	-13%	-20%	17%		
	2	1.51	1.38	1.23	1.39	1.93	2.36			9%	18%	8%	-28%	-56%		

Source: Optiro (2015)

Drake

For Drake, the comparative statistics between the composite and model averages are presented in Table 43 and there is good correlation between the composite samples and the estimated grade. No further validation was undertaken for Drake.

Table 43: Drake comparative statistics

	% Pb			Relative difference
	Composite Average	Model Average	Difference	
+1% mineralised domain	2.84	2.98	0.14	4.9%

14.8.3 Swath Plots

Magellan Hill

Swath plots were prepared by easting and northing to ensure spatial grade trends were maintained during estimation; these are shown in Figure 50 for Lode 3.

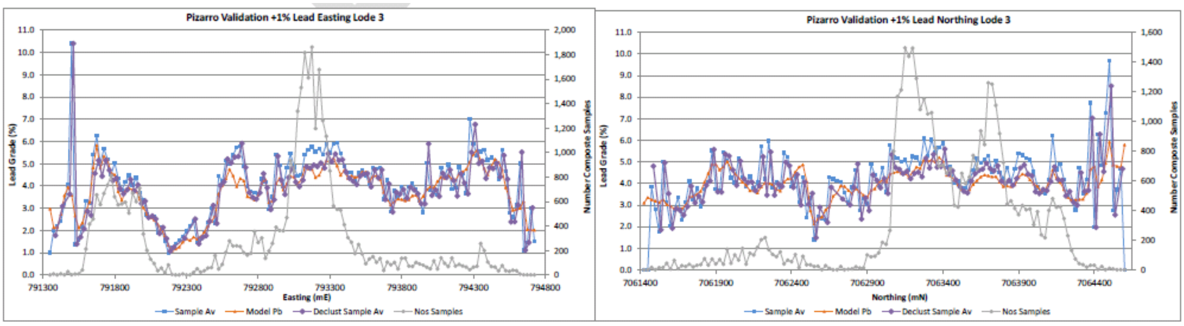


Figure 50: Swath plots for Lode 3 - Magellan Hill

Source: Optiro (2015)

The swath plots for the main mineralised lode (Lode 3) were also compared against the declustered averages, using a cell declustering approach with a cell size of 25 (X) x 25 (Y) x 2.5 m (Z). There is good correlation in easting and northing between the naïve and declustered sample grade and the modelled grade, and the sample grade trends have been maintained in the model estimate.

Pizarro

Swath plots by easting and northing to test that sample trends had been maintained during estimation were prepared and the plots for Lode 1 are shown in Figure 51.

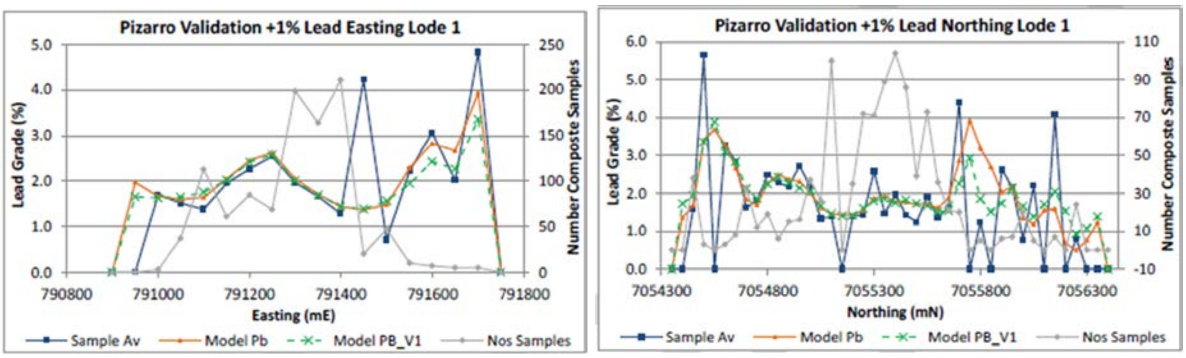


Figure 51: Swath plots for Lode 1 showing linear and categorical indicator estimates – Pizarro

Source: Optiro (2015)

For Pizarro, the available composite data and the estimated model grades correlates well and the composite grade trends have been maintained in the block model.

14.9 Resource Classification

14.9.1 2016 Mineral Resource

The 2016 Mineral Resource update has been classified in accordance with the CIM 2005 definitions and standards. The Mineral Resource classification is unchanged from that described in SRK 2015 and is summarised in Table 44.

Table 44: Mineral Resource classification criteria 2014

Data quality and intrinsic value	Search pass	Hole spacing	Slope of regression / Kriging efficiency	Resource classification
Data has acceptable levels of precision and accuracy and is understood to be representative, excluding the Maraloou Shale	1	<25 x 25	High	Measured
		<50 x 50	Moderate	Indicated
		<100 x 100	Low	Inferred
		>100 x 100	NA	Unclassified
	2	<100 x 100	NA	Inferred
		>100 x 100		Unclassified
	3	<100 x 100	NA	Unclassified
		>100 x 100		Unclassified
	4	NA	NA	Unclassified
Maraloou Shale		Unclassified		

Source: Optiro (2015)

Mineralisation within the Maraloou Shale or similar basal unit is unclassified (not a Mineral Resource) as a function of the poor metallurgical recovery from this horizon.

Magellan Hill

Figure 52 depicts a plan view of the Magellan Hill model coloured by the applied resource classification, showing the available drilling and the top of the Maraloou Shale. Only areas informed by search pass 1 and supported by grade control sampling are considered to be Measured Mineral Resources. Indicated Mineral Resources are those areas informed in search pass 1 and informed by sampling spaced less than 50 m x 50 m. All other material with search pass 1 has been classified as Inferred Mineral Resource.

Material estimated outside of the first search pass has not been classified as a Mineral Resource either because of a lack in confidence of the interpreted geological and/or grade continuity or because of concerns that the width of mineralisation is too narrow to support the eventual economic extraction.

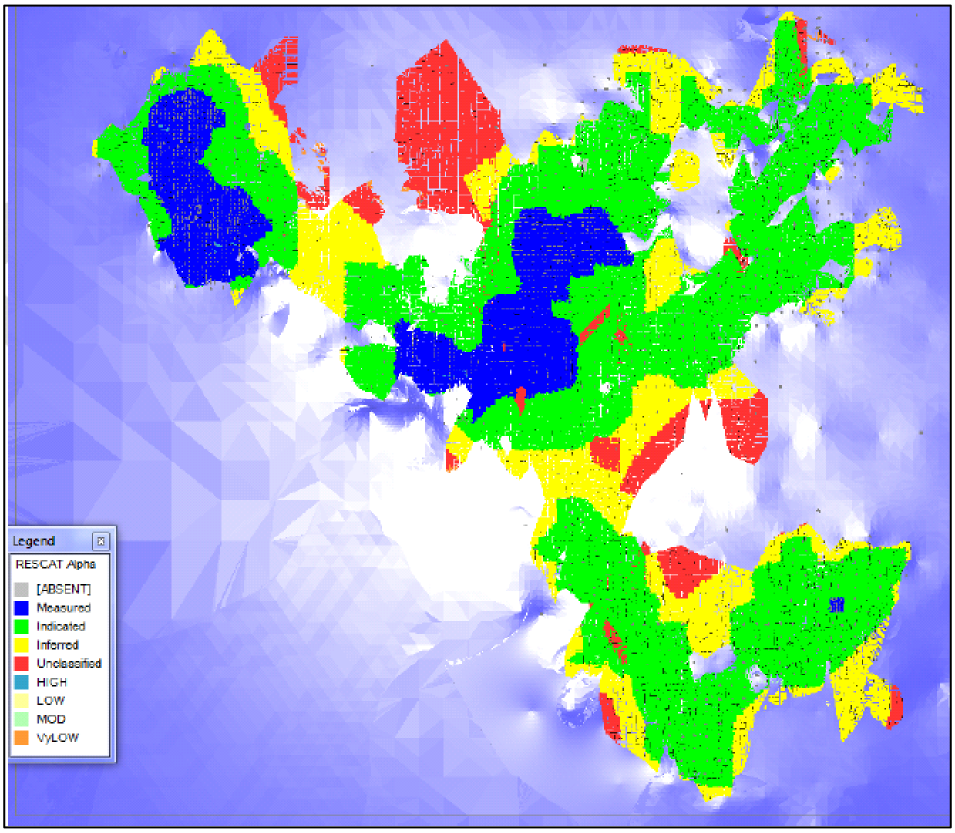


Figure 52: Plan view model coloured by confidence / classification showing top of Maraloou Shale and drilling - Magellan Hill

Source: Optiro (2015)

Pizarro

Figure 53 is a plan view for Pizarro where there is no basal unit analogous to the Maraloou Shale to truncate the Mineral Resource. Due to the lack of grade control drilling, there is no Measured Mineral Resources at Pizarro. Where the estimate is informed in the first pass and the drilling density is less than 50 mE x 50 mN, there is sufficient confidence to classify the mineralisation as Indicated Mineral Resources.

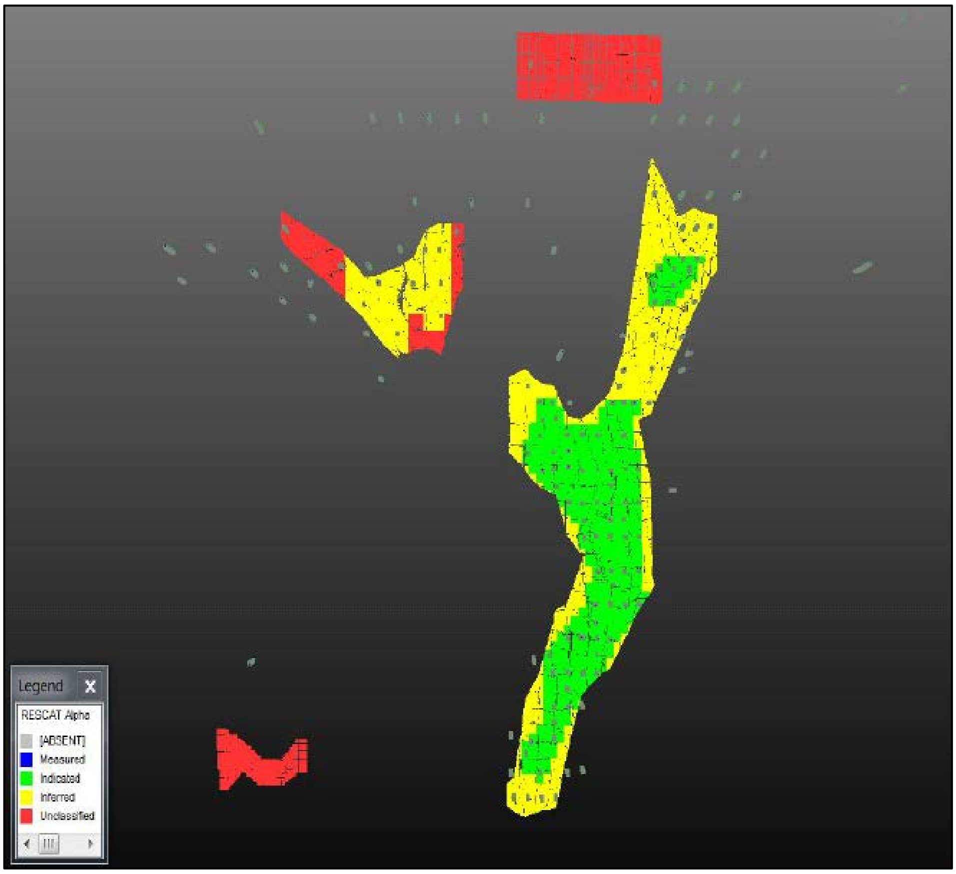


Figure 53: Model plan view coloured by confidence / classification and drilling – Pizarro

Source: Optiro (2015)

Drake

Due to the absence of a suitably detailed topography and because the geological and grade continuity has not been demonstrated at Drake, the Mineral Resource has been classified in accordance with the JORC 23012 code as an Inferred Mineral Resource. If infill drilling confirms the assumed geological and grade continuity, there are reasonable expectations for classification to be upgraded (Figure 54).

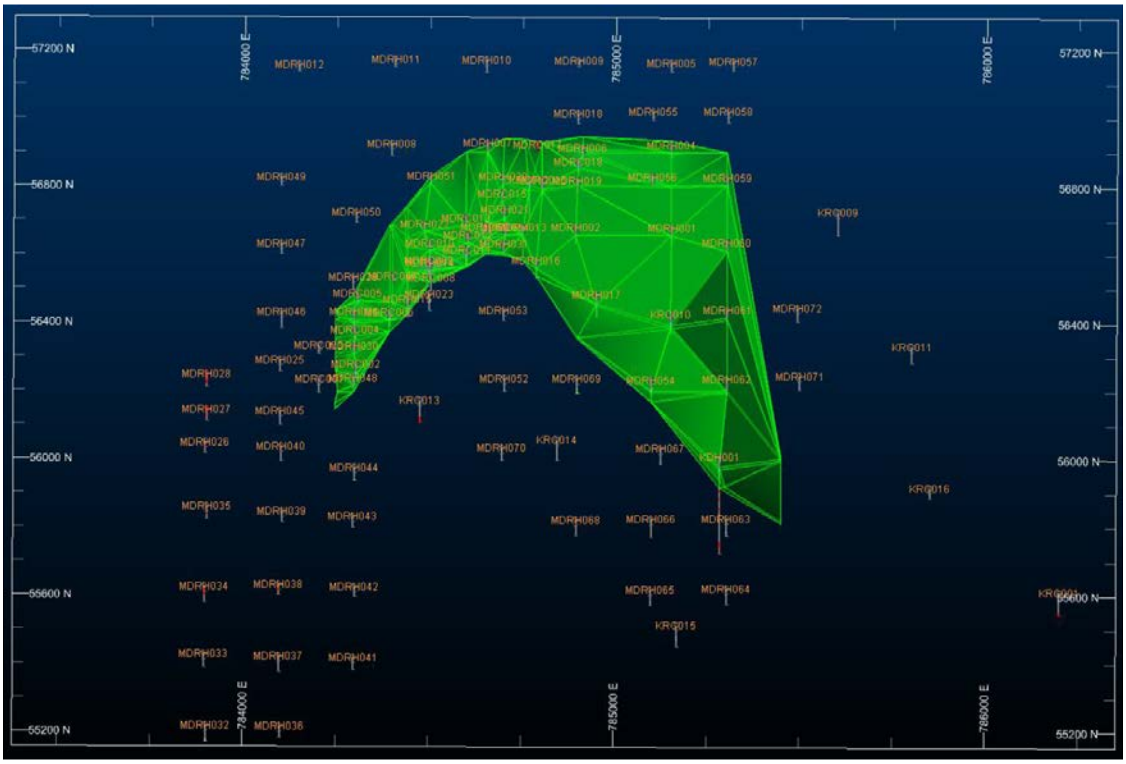


Figure 54: Drake lead deposit plan view showing available drilling as at 2005 and Inferred Mineral Resource (green polygon)

Source: Optiro (2016) and FinOre (2005)

14.10 Mineral Resource Statement

Table 45 depicts the December 31, 2016 Mineral Resource tabulation, at a 2.1% lead cut-off and rounded to reflect the precision of the estimate. The estimate has been prepared by Optiro Pty Ltd and is as per the Technical Report dated March 10, 2015 prepared by SRK, net of mining depletion (totalling 5 kt of contained lead) in January and February 2015 and reported in the Company's Annual Information Form dated March 28, 2018.

Table 45: Mineral Resource estimate as at December 31, 2016

Deposit	Resource Category	Tonnes (Mt)	Grade (Pb%)	Contained Pb Metal (kt)
Magellan (including Gama)	Measured	3.5	4.8	170
	Indicated	13.1	4.6	600
	Total Measured + Indicated	**16.6**	**4.6**	**770**
	Inferred	2.5	4.5	115
Cano	Measured	1.2	4.0	50
	Indicated	1.2	2.9	35
	Total Measured + Indicated	**2.4**	**3.5**	**85**
	Inferred	0.4	3.0	10
Pinzon	Measured	0.1	6.4	5
	Indicated	8.4	4.4	370
	Total Measured + Indicated	**8.5**	**4.4**	**375**
	Inferred	1.7	3.8	65
Pizarro	Measured	0	0.0	0

Deposit	Resource Category	Tonnes (Mt)	Grade (Pb%)	Contained Pb Metal (kt)
	Indicated	3.1	3.6	115
	Total Measured + Indicated	3.1	3.6	115
	Inferred	1.1	3.6	40
Drake	Inferred	2.7	4.1	110
Stockpiles	Measured	1.5	3.0	45
Total	**Measured**	**6.3**	**4.3**	**270**
	Indicated	**25.8**	**4.3**	**1,120**
	Total Measured + Indicated	**32.0**	**4.3**	**1,390**
	Inferred	**8.4**	**4.0**	**340**

1. *All Mineral Resources have been reported in accordance with the 2012 JORC Code reporting guidelines and are inclusive of Ore/Mineral Reserves.*
2. *All Mineral Resources have been reported using a cut-off grade of 2.1% lead and depleted for mining to December 31, 2015. There has been no mining or processing of material during the 2016 calendar year.*
3. *The stockpiled Mineral Resource is based on mine production data.*
4. *The Mineral Resource figures are based on the Mineral Resource Report which has been prepared by Mr. Kahan Cervoj (MAusIMM, MAIG), who is an employee of Optiro, and is a "Competent Person" as defined by the 2012 JORC Code. He is a QP for purposes of NI 43-101 and he supervised the preparation of and verified the above Mineral Resource figures prepared by the Company's consultants, including the underlying sampling, analytical, test and production data. Data was verified by site visits and reviews of the Company's and consultants' data.*
5. *Mr. Cervoj was the Competent Person for the Magellan Hill 2014 Mineral Resource that is the basis for the December 2015 Mineral Resource estimate and participated in a site visit in the last week of July 2014.*
6. *Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.*
7. *Table entries are rounded to reflect the precision of the estimate and differences may occur due to this rounding.*

14.11 Mineral Resource Sensitivity

14.11.1 Mineral Resource Classification

Revision of the resource confidence in 2014 for the Paroo Station deposits is logical and consistent, and is closely linked to the spatial coverage of the collected data (Figure 55).

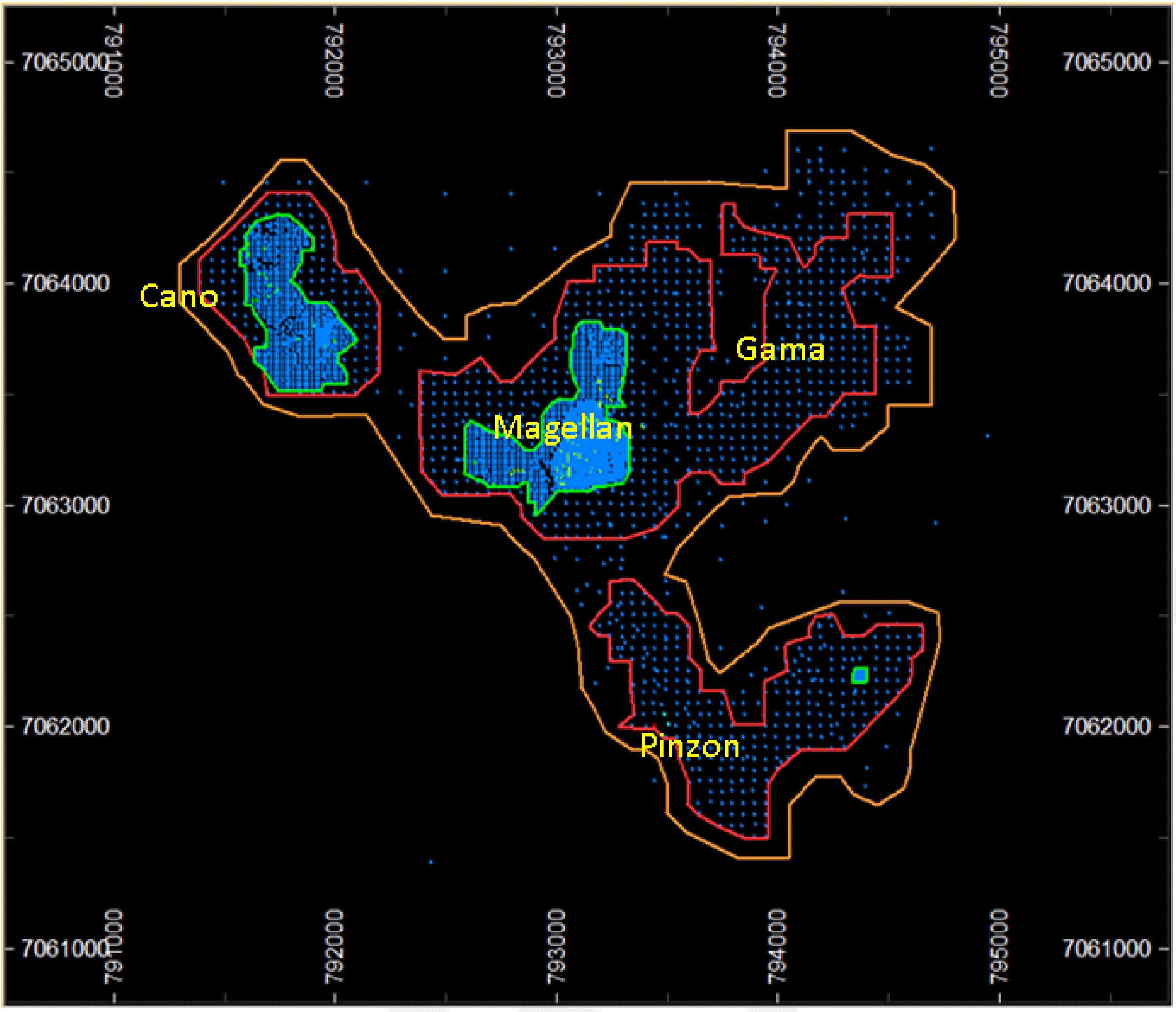


Figure 55: Mineral Resource classification - December 31, 2014

Source: Optiro (2014)

Note: Green = Measured, Red = Indicated and Orange = Inferred

14.11.2 Inventory Changes from 2015 to 2018

The December 31, 2016 Mineral Resource estimate as described earlier has been prepared in accordance with the 2012 edition of the JORC Code.

The 2014 Mineral Resource estimate update employed a “first principles” approach by utilising all geological work available including significant drilling, production observations and interpretive compilations to update estimates of the lead mineralisation on RHM’s tenements.

Since 2014, the Magellan Hill deposits haven been updated for mining and processing depletion prior to the Paroo Station placement on care and maintenance in February 2015, which reflects the only change to inventory for these deposits since the previous technical report (SRK 2015).

In 2016 the Drake Mineral Resource was reviewed and reported in accordance with the JORC 2012 reporting code, and the resource has been reported at a 2.1% Pb cut-off.

The classification of the Mineral Resource suitably reflects the relative confidence in the Mineral Resource estimate.

14.12 Relevant Factors

The construction of the Hydrometallurgical Facility at the Mine is expected to provide a positive impact on the Mine economics. In the future, this may provide justification to lower the Mineral Resource reporting cut-off, from the current 2.1% Pb. This will be assessed when the capital and operational costs are better understood post commissioning of the Hydrometallurgical Facility.

Topography at the Drake deposit is currently based on the available surveyed drillhole collar locations, which is reflected in the current Mineral Resource classification. Detailed topographical data is now available and will be incorporated with next update of the Drake Mineral Resource.

Other factors that could influence the reported Mineral Resource have been discussed in the sections above. No further relevant factors have been noted.

15 Mineral Reserve Estimate

The Mine has been in commercial operation over several operation phases before being shut down in January 2015 due to low commodity prices. As a result, the QP has relied on both historic and more recent production information and financial inputs to support the mine planning and confirm that economic extraction of the Mineral Resource is feasible when integrating the Hydrometallurgical Facility with the existing mining and flotation concentration activities.

The Mine plan was revised to support the Mineral Reserve estimate with updated open pit optimisation incorporating accepted product pricing and current costs and operational parameters. The open pit optimisation underpinned revised mine staging, mine designs and mine production scheduling.

The Mineral Reserve estimate was developed under the 2012 Edition of the JORC Code. CIM recognises "use of foreign codes" including the JORC Code.

15.1 Parameters Relevant to Mine or Pit Designs and Plans

15.1.1 Geotechnical

An overall slope angle of 40^0 has been applied to the optimization process. All final pit designs produced have incorporated the recommended geotechnical pit slope design parameters from geotechnical interpretations undertaken and presented in Review of Wall Design Parameters Paroo Station, Peter O'Bryan and Associates, (Jan 2015):

- Bench face height 10 m – from surface to 30 m depth.
- Bench face height 15 m – below 30 m depth from surface.
- Face angle 600 throughout.
- Minimum berm width of 5 m at 10 m and 20 m depth intervals.
- Minimum berm width of 6 m at 30 m and 45 m depth intervals.

The existing pit wall designs are based on 10 m-high, 50^0 face angle batters separated by 5 m-wide berms.

15.1.2 Hydrogeological

The as-mined pits do not currently intersect the water table; however, the water table will be partially intersected when pits are mined to the ultimate design, at the conclusion of the expected mine life. Regulatory approval is required to mine beneath the water table. A hydrogeological review is required to confirm there will be no likely adverse impact on the stability of the pit walls.

15.1.3 Open Pit Optimisation

Open pit optimisation was used to identify the optimum economic pit shape based on the cashflow analyses. The pit optimisation process seeks a solution to a complex 3D mathematical relationship involving the Mineral Resource model, geotechnical slope guidelines, product revenue, Mine constraints, modifying factors and costs.

The key inputs into the optimisation process include:

- Product prices.
- Mining costs.
- Processing, realisation and administration costs.

- Process recoveries.
- Pit slope angles.
- Prepared model.

The Mineral Resource model was converted to a mining model by a process of regularisation to account for dilution and ore losses. The diluted model has then been used as the basis for optimisation, pit evaluation and scheduling. Further preparation included; adding cost, recovery, royalties and revenue drivers to the individual blocks within the model.

A net present value (NPV) discount rate of 8%, which is comparable with Australian projects of similar scale and size, has been applied.

Net smelter return (“NSR”) inputs and formulas required to calculate the economic value for each block were used in the optimisation process. These include mining costs per bench, processing costs, metallurgical recovery formulas, expected metal price etc.

The Whittle Four-X software package was used to develop the pit optimisation shells.

15.2 Mine Design

The following design parameters were used in all final pits:

- Dual lane ramps of 25 m wide at 10% gradient.
- Batter angle 600.
- 10 m bench height from surface to 30 m depth.
- 15 m bench height below 30 m depth.
- 5 m bench width at 10 m and 20 m depths
- 6 m bench width at 30 m and 45 m depths.
- Minimum mining width approximately 40 m.

A final pit was designed and divided into nine progressive pit stages, in order to assist with achieving the schedule targets. Each stage has its own ramp access, whilst complying with the minimum mining width, so they can each be mined independently.

A plan view of the Cano, Magellan and Pinzon pits are shown in Figure 56 to Figure 58 and all Pit Stages are shown in Figure 59.

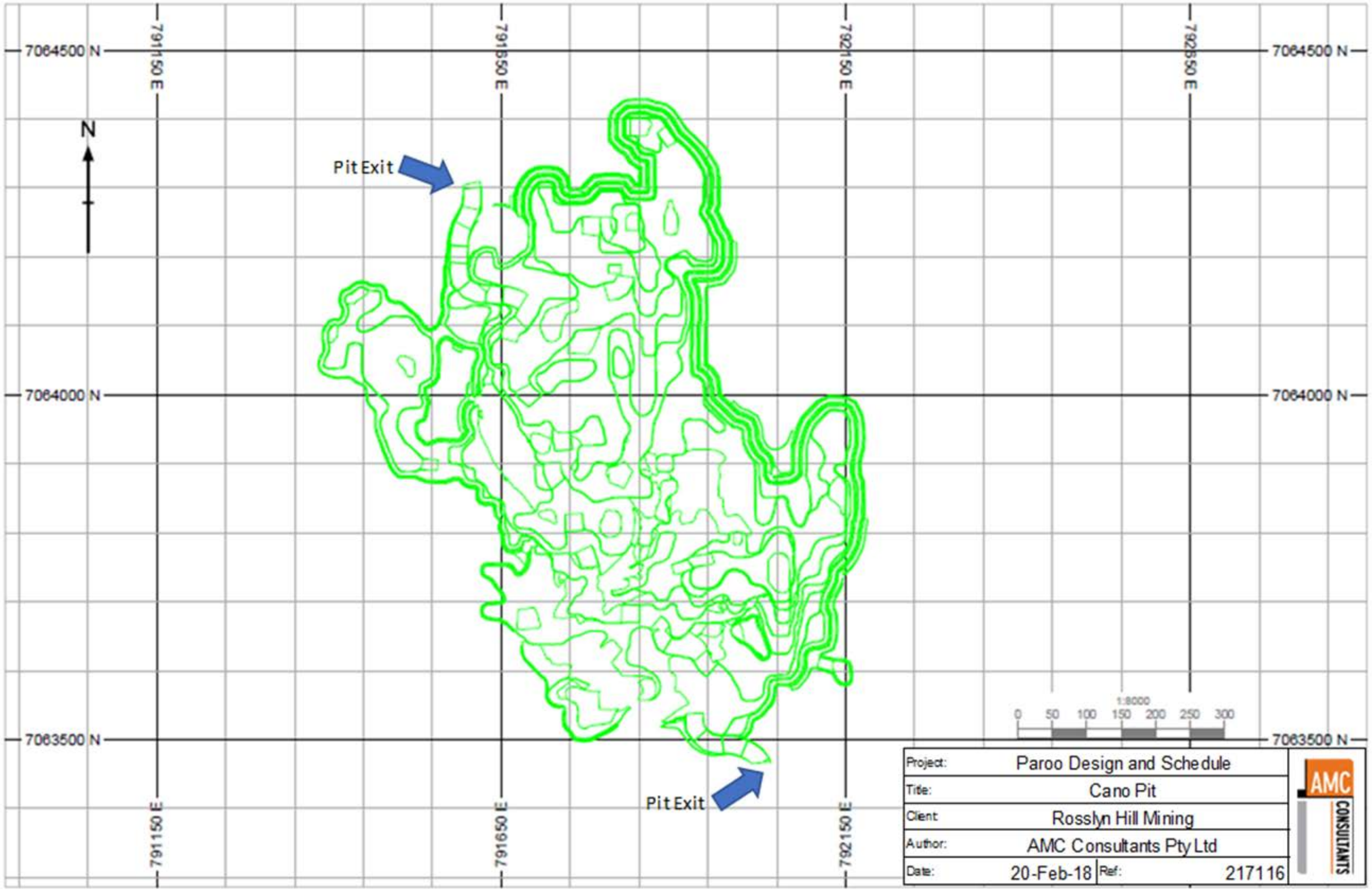


Figure 56: Cano Pit Design

Source: AMC (2018)

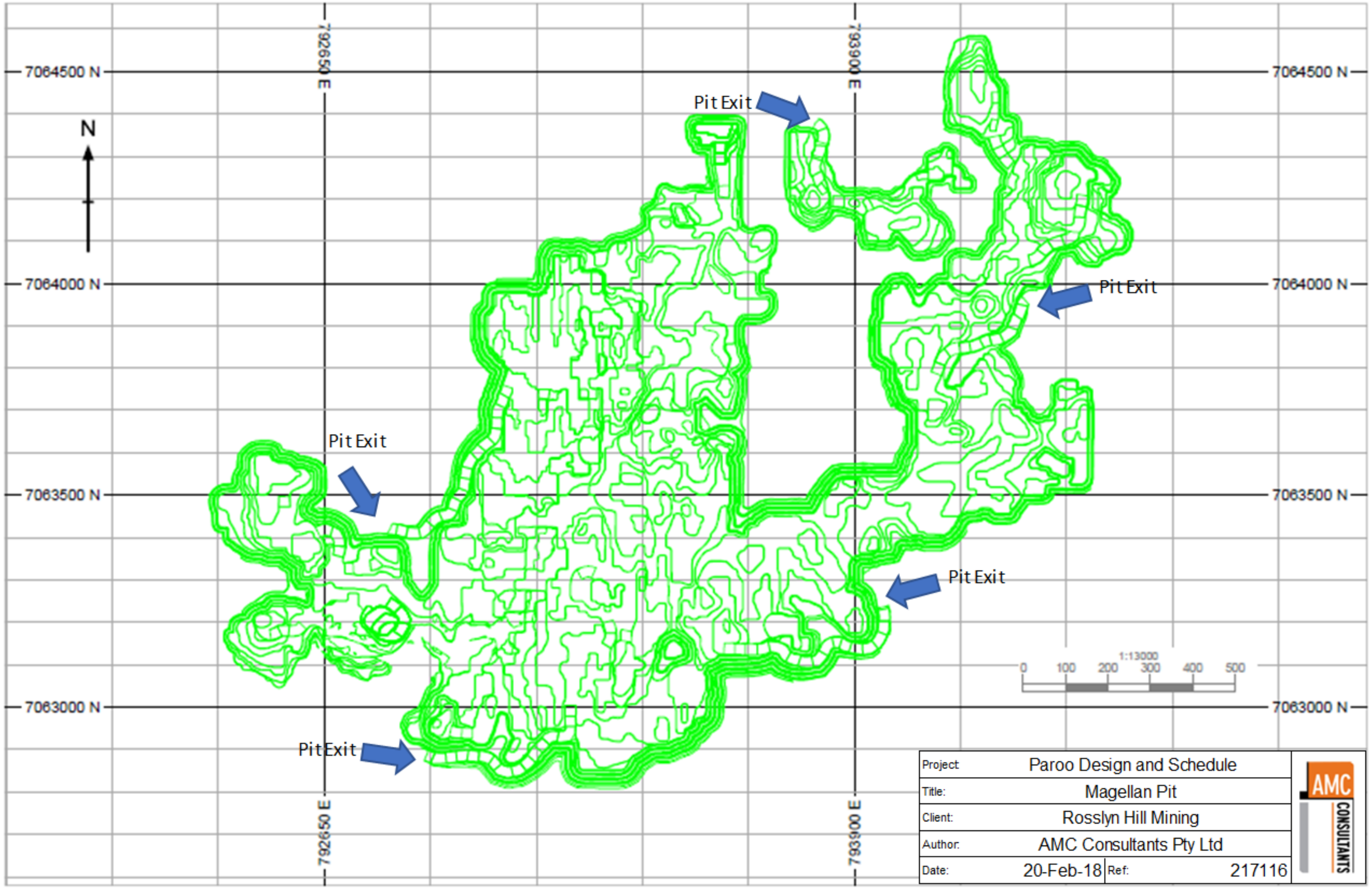


Figure 57: Magellan Pit Design

Source: AMC (2018)

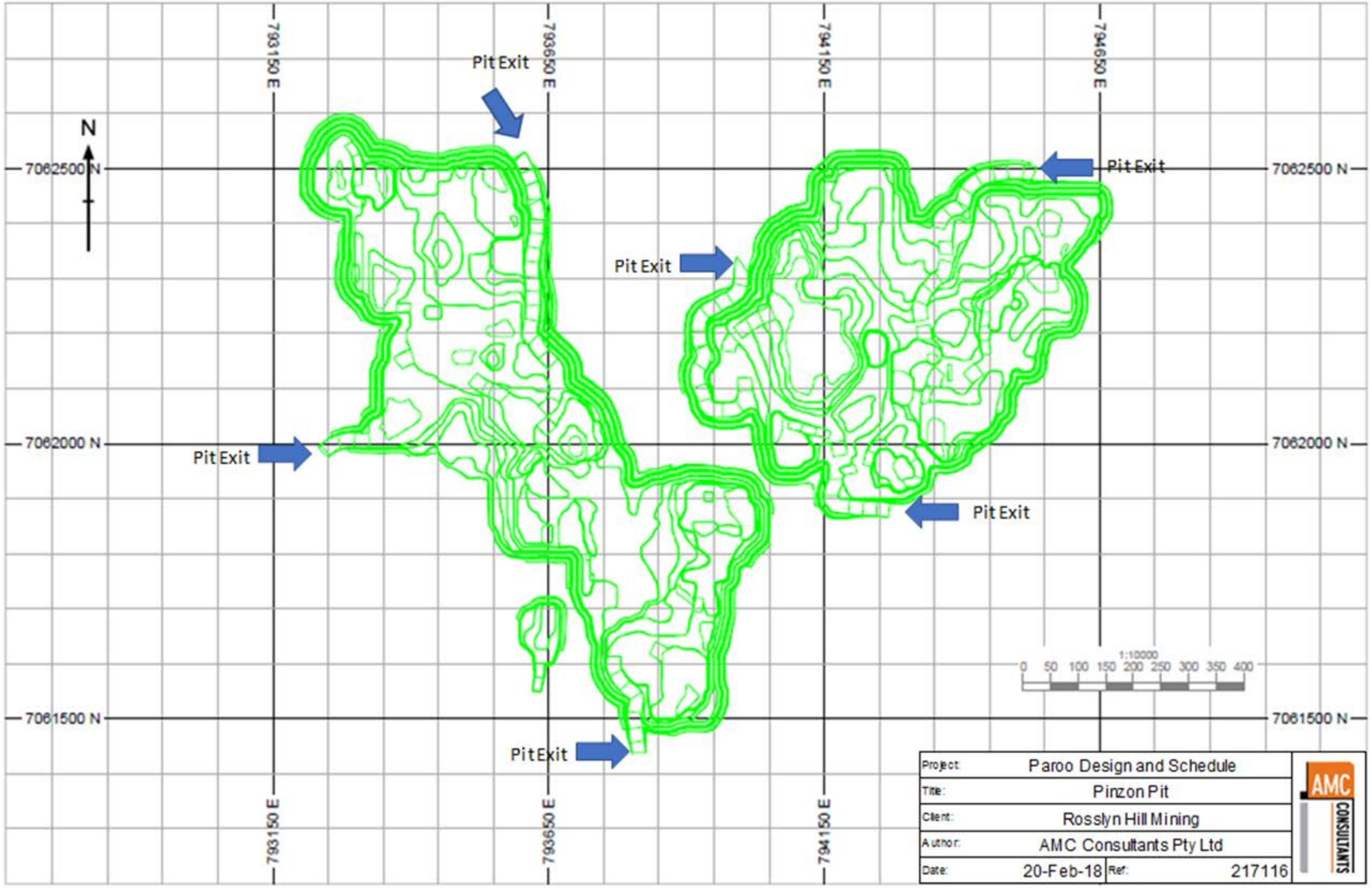


Figure 58: Pinzon Pit Design

Source: AMC (2018)

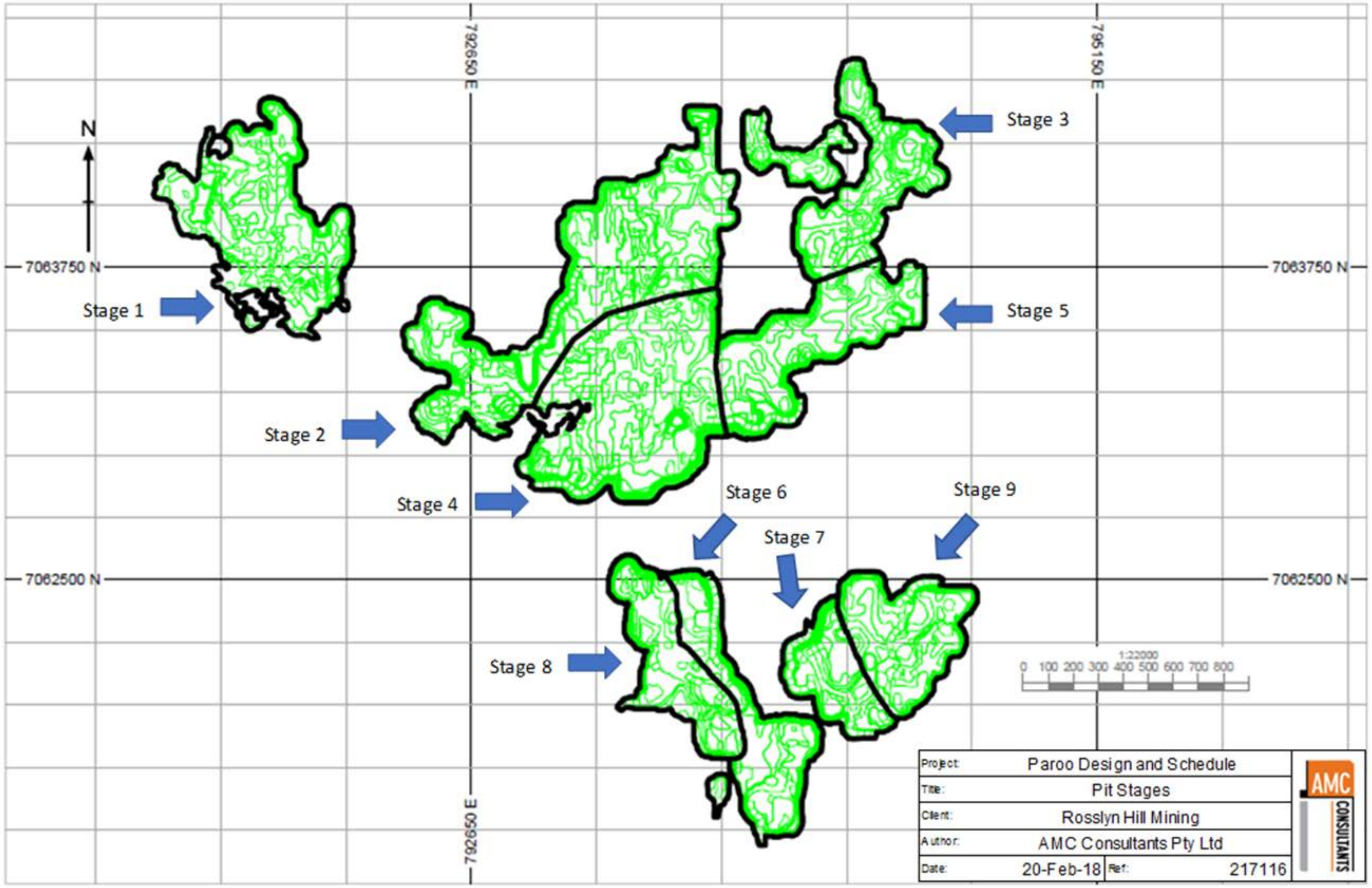


Figure 59: All Pits showing stages

Source: AMC (2018)

Each pit stage design was evaluated using the mining model to produce the results in Table 46.

Table 46: Inventory summary by pit

Pit design	Waste (t)	Ore (t)	Strip Ratio	Pb (%)	Contained Pb (t)
Cano	4,115,548	2,393,051	1.72	3.10	74,293
Magellan	54,024,336	16,990,259	3.18	4.16	706,606
Pinzon	26,335,366	8,841,224	2.98	3.94	347,910
Total Design	84,475,061	28,227,534	2.99	4.00	1,128,809
Optimisation	81,043,591	28,805,607		3.97	1,144,242
Design versus Optimisation	4.2%	-2.0%		-1.3%	0.7%

15.3 Mine Production Scheduling

Mine production schedules were developed using MineMax software. Several iterations of the schedule were evaluated in order to balance the mill targets and minimise stockpiles while keeping a steady production profile.

This schedule was developed based on:

- Diluted model with Measured and Indicated Mineral Resource categories only.
- Annual schedule where start date is irrelevant.
- Mill capacity of 2.0 Mtpa is constant from the second year of production after an initial ramp up from 1.4 Mt for the first year.
- Achieving 70 ktpa of lead ingot production as consistently as possible.
- Maximum total material movement limited to 8 Mtpa in the first year.
- 5 m benches.
- Maximum vertical advance rate of ten benches per year.
- 40 m minimum cutback distance.
- Minimise stockpiles from in-pit under 1.5 Mt.

The results of the schedule are shown in Figure 60 and Figure 61.

Salient points from the schedule include:

- Constant monthly positive cashflow
- Mine life of +15 years
- Mill capacity met by mining production until near the end of the mine life
- Stockpile maximum not exceeding 2.3 Mt of ore which includes existing stockpiles as well as mineralised waste stockpiles
- Total material movement ("TMM") limited to 8 Mtpa in the first year. The production then increases to 10.5 Mtpa for the following three years and then gradually stepping down until the end of the mine life
- The schedule assumes ideal conditions where the potential of the specified mining fleet can be achieved and that stockpiles will be manageable. It does not take into account and delays due to calendar events, unfavourable weather conditions, issues with maintenance and permits etc.

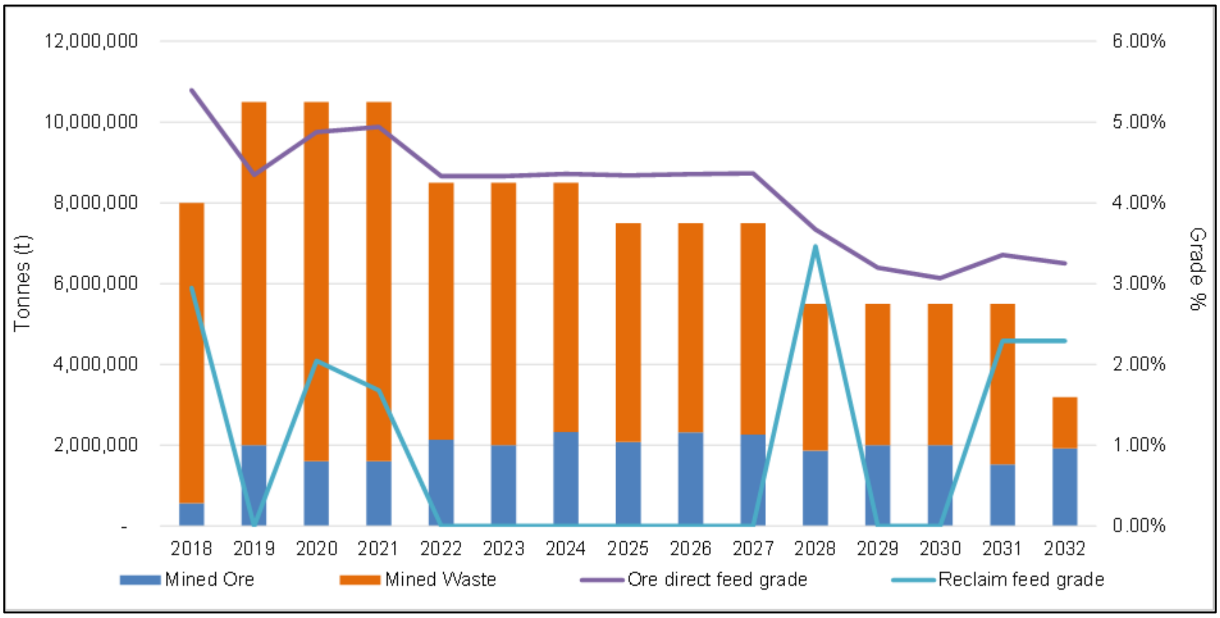


Figure 60: Total Material Movement

Source: AMC (2018)

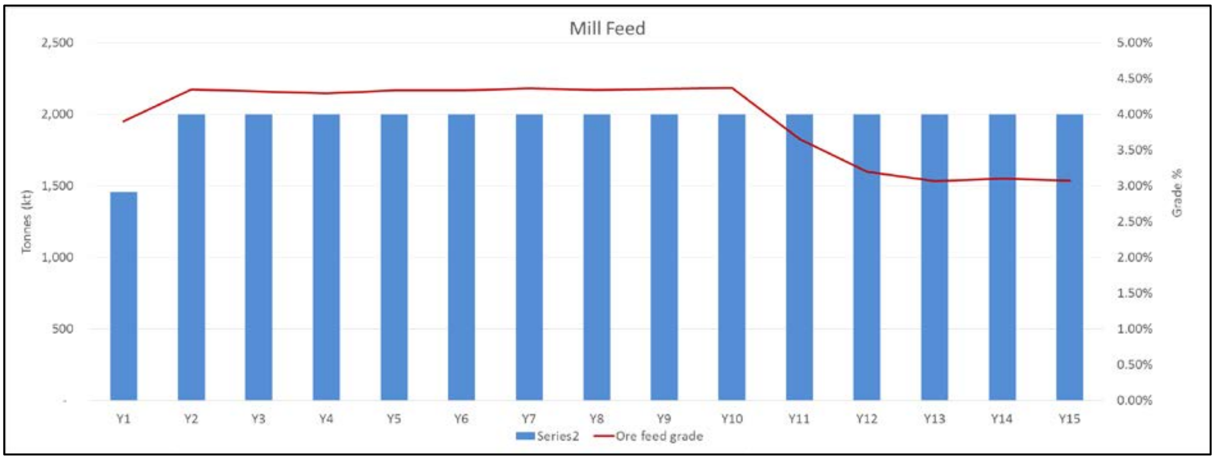


Figure 61: Annual Mill Feed

Source: AMC (2018)

15.4 Waste and Dump Design

Preliminary designs for the waste dumps were prepared to ensure sufficient ex-pit dumping capacity. The design parameters and assumptions are:

- Batter or face angle of 180
- 5 m berm every 10 m lifts
- Maximum total height of 50 m
- Minimum of 50 m away from the pit boundary.

A plan of each dump design is shown together with the pits and site layout in Figure 62. The IWL embankment detailed in section 18, will provide 11.4 Mm^3 of waste rock storage capacity. Not factored into the dump designs are the opportunities that are present for in-pit dumping that will realise both cost savings from short hauls and reduced dump footprints and / or heights.

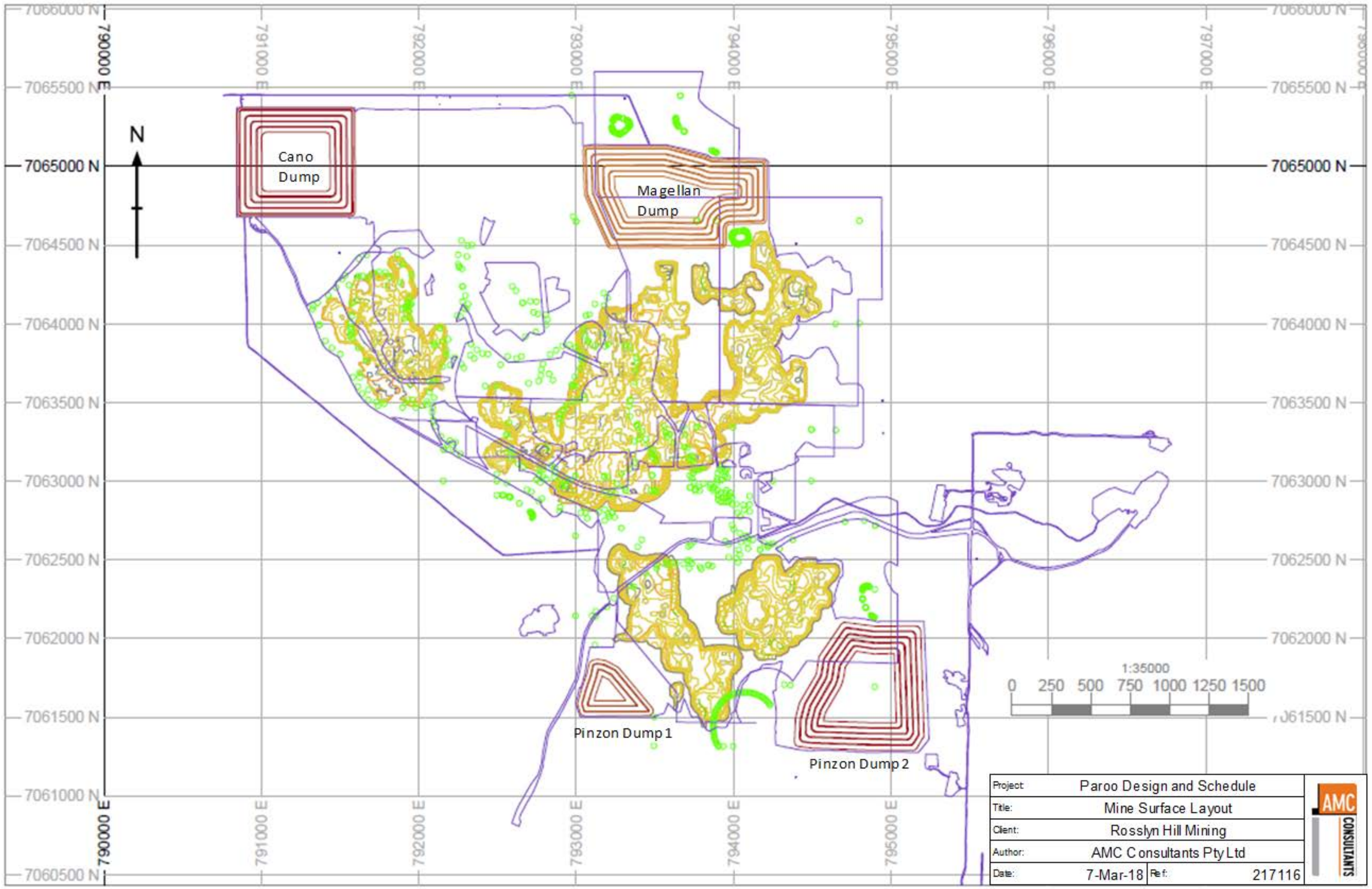


Figure 62: Site Layout

Source: AMC (2018)

The volumes for each of the waste dump designs were evaluated and are shown in Table 47.

Table 47: Waste dump volume capacity for each location

Pit	Dump Design Name	In situ volume (m^3)	Volume after 30% swell factor (m^3)	Volume after 10% contingency (m^3)	Design volume (m^3)	Design area (m^2)
Magellan	Magellan Dump	27,846,361	36,200,269	39,820,296	19,837,153	663,126
Cano	Cano Dump	2,138,089	2,779,516	3,057,467	14,232,033	514,910
Pinzon	Pinzon Dump 1 & 2	13,671,648	17,773,142	19,550,457	18,708,157	674,320
IWL Embankment					11,400,000	
Total		43,656,098	56,752,927	62,428,220	64,177,343	1,852,356

Source: AMC (2018)

15.5 Mineral Reserve Estimate

15.5.1 2018 Reserve Estimate

The Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Mineral Reserves. Inferred Mineral Resources are, by definition, always additional to Mineral Reserves.

Based on the analysis, the QP therefore concludes that there is a reasonable expectation that the Mine will reopen.

The Mineral Reserve estimate was classified and reported under the 2012 Edition of the JORC Code Code. CIM recognises “use of foreign code” including the JORC Code.

The Mineral Reserve is presented in the Table 48.

Table 48: Mineral Reserve Statement as at February 28, 2018

Deposit	Reserve Category	Tonnes (Mt)	Grade (Pb%)	Contained Pb Metal (kt)
Cano	Proved	1.4	3.5	47
	Probable	1.0	2.6	27
	Total	**2.4**	**3.1**	**74**
Magellan	Proved	3.9	4.3	169
	Probable	13.1	4.1	538
	Total	**17.0**	**4.2**	**707**
Pinzon	Proved	0.1	5.8	5
	Probable	8.8	3.9	343
	Total	**8.9**	**3.9**	**348**
Stockpiles	Proved	2.9	2.4	70
	Probable			
	Total	2.9	2.4	70
Total	**Proved**	**8.3**	**3.5**	**291**
	Probable	**22.9**	**4.0**	**908**
	Total	**31.2**	**3.8**	**1,199**

Source: AMC (2018)

1. *Mineral Reserves are a subset of Measured and Indicated Mineral Resources. The Mineral Reserve Estimate was developed to 2012 JORC Code standards which are accepted CIM under the use of a Foreign Code. The 2012 JORC Code uses the terms "Ore Reserve" and "Proved" which are equivalents to the terms "Mineral Reserve" and "Proven" respectively, as defined in NI 43-101.*
2. *The Mineral Reserve Estimate was developed by Mr Adrian Jones, a full-time employee of AMC Consultants Pty Ltd (AMC). Mr Jones is the Competent Person for the 2015 Paroo Station Ore Reserve estimate under the 2012 JORC Code. Mr Jones supervised preparation of the estimate with assistance from specialists in each area of the estimate. Mr Jones is a Member of The Australasian Institute of Mining and Metallurgy. He has sufficient experience relevant to the style of mineralisation, type of deposit under consideration, and in open pit mining activities, to qualify as a Competent Person as defined in the JORC Code. Mr Jones consents to the inclusion of this information in the form and context in which it appears.*
3. *Mr Lawrie Gillett FAusIMM of AMC is a QP for the purposes of NI 43-101. He is a full-time employee of AMC and he also supervised and verified the above Mineral Reserve figures prepared by Mr Jones, including the underlying sampling, analytical test and production data.*
4. *Mr Jones participated in a site visit in the second week of March 2015.*
5. *The pit limits for the open pit were selected through optimisation using the Gemcom Whittle Four-X implementation of the Lerchs-Grossman algorithm. The optimisation considered Measured and Indicated Mineral Resources only. Pit designs followed the optimisation shell outline that developed the largest undiscounted cashflow for the evaluation parameters.*
6. *The process recovery of lead is linked to lead head grade. The following recovery formula was used in the analysis: Flotation Pb Recovery = 73.5% + (1.55 x % Ore Grade), Hydrometallurgical Facility Recovery 98.17%[8]. The average overall recovery is 80%.*
7. *Dilution of the resource model and an allowance for ore loss are included in the Ore Reserve estimate, and were introduced through applying a selective mining unit of 6.25 m x 6.25 m x 2.5m. Within the Ore Reserve pit design, the application of dilution resulted in inclusion of 5.59% dilution and results in an ore loss of 6.43%. Metal pricing of USD 2,250/t Pb plus USD85/t Pb premium was used in the mine planning.*
8. *The Proved Ore Reserve estimate is based on Mineral Resources classified as Measured, after consideration of all mining, metallurgical, social, environmental, statutory and financial aspects of the Mine. The Probable Ore Reserve estimate is based on Mineral Resources classified as Indicated, after consideration of all mining, metallurgical, social, environmental, statutory and financial aspects of the Mine.*
9. *Table entries are rounded to reflect the precision of the estimate and differences may occur due to this rounding.*

[8] The Hydrometallurgical Facility recovery used in the determination of the Mineral Reserves Estimate is 98.17%. The final Hydrometallurgical Facility recovery has been estimated at 97.91%. The difference has a life of mine net impact of 2,478 t of lead metal, being less than 0.3% of the recovered lead metal over the life of Mine.

15.6 Inventory Changes from 2016 to 2018

The Mineral Reserve is materially different to the legacy Mineral Reserves estimates.

The Mineral Reserve estimate has previously been estimated as at December 31, 2014 from a Technical Report undertaken by SRK dated March 10, 2015. RHM has updated the estimate following mining depletion in January 2015 as reported in the 2015 and 2016 Lead*FX* Annual Information Reports.

An increase of approximately 24.7 Mt Mineral Reserves is noted between the December 31, 2016 estimate and the current estimate.

An increase of approximately 745 kt Pb is noted between the December 31, 2016 estimate and the current estimate.

The material differences are considered to be due to the overall operating cost reduction associated with the application of the Hydrometallurgical Facility producing lead ingot on site. The reduced operating costs allows significantly enhanced exploitation of the Mineral Resources.

15.6.1 Mineral Reserve Sensitivity

Multiple pit optimisation runs were undertaken to establish the Mine's sensitivity to pricing, mining and processing costs. The results of these ancillary runs establish the key drivers to the development of the mining process suited to the extraction of the deposits potentially economic mineralization.

The base case was selected for parameter variance to explore the sensitivity of output shell size and corresponding financial metrics. The parameters investigated within the sensitivity are:

Metal pricing (AUD 2,490/t Pb to AUD 3,735/t Pb range).

- Mining cost (-20% to +20% range with 10% increments).
- Processing cost (-20% to +20% range with 10% increments).

Figure 63 plots the effect of the sensitivity analysis on the Mineral Reserve estimate ore tonnage.

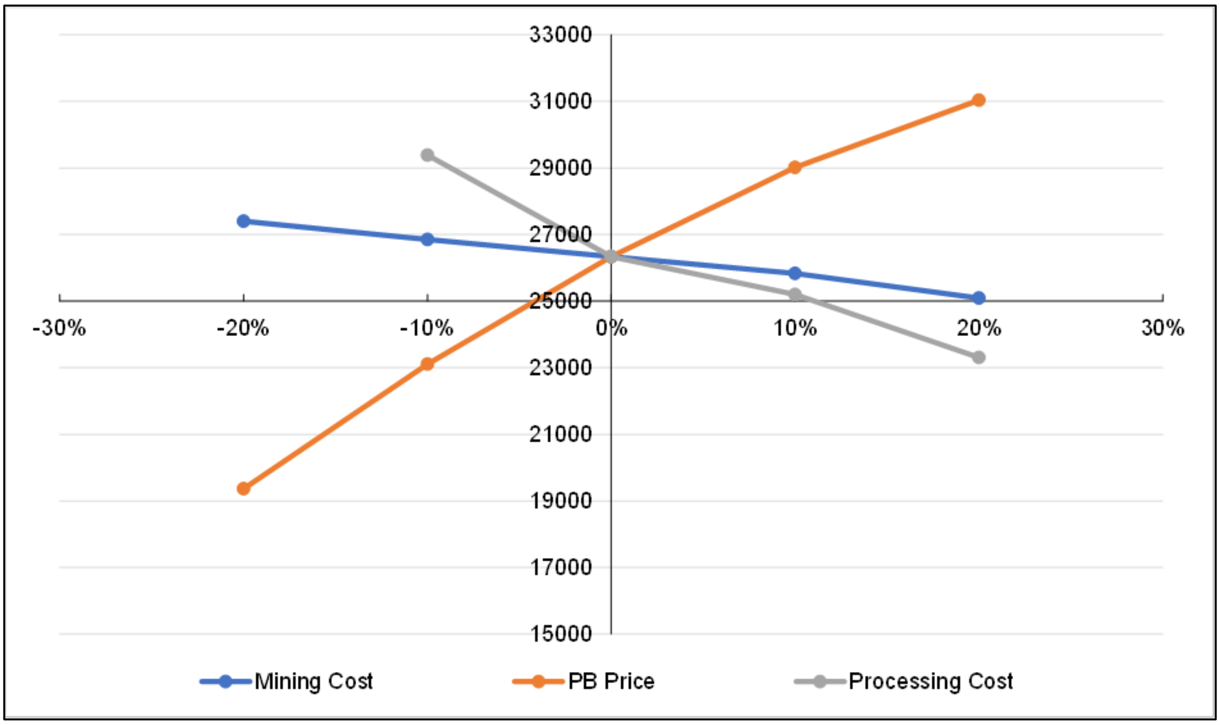


Figure 63: Sensitivity analysis graph - ore tonnes(kt)

The sensitivity analysis demonstrated that out of the variables selected, the Mineral Reserve estimate tonnage is most affected by metal pricing, followed by processing costs with mining costs having the least significant impact.

16 Mining Methods

16.1 Previous Operation

The following descriptions are provided on the mining methods which were undertaken during the last operational period from 2013 to 2015. No material changes are proposed to the mining methods when operations recommence.

Ore at the Paroo Station was extracted from a series of open pits on Magellan Hill. Drilling and blasting is required so that excavators can be used to dig and load ore and waste into 85 t haul trucks. Ore was mined concurrently from a number of faces to provide a homogenous blend to the concentrator, and ore is stockpiled and further blended on the ROM pad.

Grade control is enhanced by sampling every blasthole in the orebody and in the near vicinity of the orebody. Mining was based on 2.5 m flitches within 5 m benches.

This method is eminently suitable for the flat-lying shallow geometry of the ore.

Short-term planning is based on grade control and blasthole sampling and appears to provide a reasonable level of control to the mining operations.

Figure 64: Mining operations in the Magellan open pit

Source: RHM (2013)

16.2 Mining Fleet and Requirements

16.2.1 General Requirements and Fleet Selection

MACA Limited held the mining contract to provide ROM ore feed, drill and blast and load and haul of ore and waste. The fleet selection was based on minable ore block size and volume / tonnage requirements. Circa 100 t fleet configuration is deemed most appropriate for these variables.

16.2.2 Drilling

Drilling has historically been performed by 1 x GD5000 drill operated on double shift, nominally 102 mm holes, single pass 5.0 m benches with 0.5 m subdrill. Pattern size is from 3 x 3.5 m burden and spacing in the hard cap rock to 4 x 4.5 m burden and spacing in the softer rock sequences. Wall control is achieved with batter holes, nominally 5 m depth and 2 m spacing and buffer/ stab holes nominally 2.5 m depth and 1.5 m spacing.

16.2.3 Blasting

Blasting was primarily performed using ANFO due to the dry conditions with powder factors typically ranging from 0.2 – 0.5 kg/bcm. Single hole firing was used to minimise movement and dilution of the ore. It is anticipated that a reduction in holes size to 89 mm will be required in order to keep the powder factor down as generally the effort required to blast reduces with depth.

16.2.4 Loading

Loading has been previously performed by 1 x 120 t class backhoe configuration excavator operating double shift. Productivity in excess of 8 Mtpa of ore and waste can be achieved with this size machine with the digging conditions presented. The 5 m benches are mined in 2 x 2.5 m flitches with the differing material types being defined by mark-out tape and paint as designated by the site geologists.

16.2.5 Hauling

85 t class dump trucks have been used to haul the ore, waste and mineralised waste materials to their respective destinations; ROM pad, waste dump and stockpiles. The leads for ore, waste and mineralised waste differ depending on the pit and stage location and can vary from 2 to 4 trucks hauls.

16.2.6 Auxiliary Equipment

Haul road, pit floor, waste dump and drill and blast pattern preparation have been previously performed by a combination of an articulated water cart, grader and bulldozer. Other minor equipment such as IT loaders, support trucks and explosives trailers support the drill and blast and mobile equipment maintenance activities.

16.3 Mine Dewatering

The as-mined pits do not currently intersect the water table; however, they will do so when mined to the final design. Prior to commencing any mining below the water table, a ground water investigation will need to be performed identifying to the effects on the hydrogeological regime of the ground water resource, effects on the potential groundwater dependent ecosystems within the drawdown zone and the effects on any other existing or approved groundwater users. Once these impacts have been assessed and appropriate action plans identified, RHM will apply to the EPA and DMIRS for approval to mine below the water table. As part of this study the water data sources, surface water, groundwater and the dewatering system will be considered.

17 Recovery Methods

17.1 Metallurgical Performance

During the last operational phase from April 2013 to January 2015, all open pit ore production from the Mine was processed through the Paroo Station concentrator.

Metallurgical performance for the last operational campaign is shown in Table 49, Table 50 and Table 51.

Table 49: Metallurgical Performance 2013

Actual	Unit	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Ore milled	dmt				17,160	54,116	89,029	99,919	119,253	111,990	100,690	122,676	121,034	835,867
Head grade	%				8.60	10.50	6.60	7.00	6.60	7.30	8.40	6.80	5.40	7.10
Annualised rate	Mtpa				0.21	0.64	1.08	1.18	1.4	1.36	1.19	1.49	1.43	0.84
Recovery	%				62.50	68.10	74.80	72.50	72.40	75.70	78	77	75	74
Con produced	dmt				1,469	6,079	6,575	8,173	8,766	9,507	10,165	9,864	7,455	68,053
Con grade	%				62.80	63.90	63.90	63.60	65.10	65.00	65	65	65	65
Con Pb content	dmt				923	3,881	4,201	5,194	5,711	6,183	6,636	6,448	4,481	44,018
Con Moisture	%				11.85	11.60	9.70	9.60	9.77	9.73	9.40	9.47	9.80	9.85
Plant availability	%				30	70	79	82	86	92	83	90	85	58
Plant usage	%				36	47	71	74	85	77	74	86	87	53

Source: RHM (2015)

Table 50: Metallurgical Performance 2014

Actual	Unit	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Ore milled	dmt	129,458	116,977	117,202	103,549	103,328	87,346	118,661	103,271	128,807	137,108	143,029	149,222	1,437,958
Head grade	%	5.70	6.00	6.90	7.70	8.40	9.40	7.70	7.30	6.60	6.90	6.70	6.60	7.00
Annualised rate	Mtpa	1.52	1.52	1.38	1.26	1.22	1.06	1.4	1.26	1.5	1.61	1.74	1.76	1.44
Recovery	%	78.20	73.00	78.80	84.20	86.20	82.10	77.80	83.70	82.00	76.30	74.20	76.00	79.00
Con produced	dmt	8,860	7,793	9,659	11,026	11,180	9,740	10,419	9,436	10,060	10,567	10,525	10,720	119,985
Con grade	%	65.40	65.00	66.40	66.30	67.20	68.40	67.80	67.00	69.30	68.50	67.60	69.50	67.40
Con Pb content	dmt	5,792	5,066	6,411	7,312	7,516	6,661	7,064	6,324	6,975	7,234	7,113	7,447	80,915
Con Moisture	%	9.40	9.60	9.10	9.60	9.10	9.00	8.80	9.90	8.60	9.20	8.90	8.90	9.20
Plant availability	%	90.90	92.34	82	76	87	84	89	82	92	87	95	96	88
Plant usage	%	86.81	85.49	87.28	89	73	65	81	79	89	96	95	98	85

Source: RHM (2015)

Table 51: Metallurgical Performance 2015

Actual	Unit	Jan	Feb	Total
Ore milled	dmt	166,305	4,852	171,157
Head grade	%	7.43	6.1	7.39
Annualised rate	Mtpa	1.96	0.06	0.17
Recovery	%	77.80	61.00	77.32
Con produced	dmt	13,621	392	14,013
Con grade	%	70.50	64.70	70.37
Con Pb content	dmt	9,607	253	9,860
Con Moisture	%	8.52	8.63	8.52
Plant availability	%	95.77	12.10	9.06
Plant usage	%	105.85	24.50	10.87

Source: RHM (2015)

17.2 DFS Testwork

17.2.1 Introduction

A metallurgical testwork program comprising batch and pilot plant works was carried out to provide the design data required to develop the Hydrometallurgical Facility flowsheet. Samples for this work were selected to represent ore quality from years 1 – 10 of forecast mining.

An initial flotation development program was executed to prepare concentrate samples for the Hydrometallurgical Facility testwork. In the process of generating these samples significant potential for improvement in the flotation performance was identified and incorporated into the concentrator design.

The Process Design Criteria for the Hydrometallurgical Facility have been developed by SNC-Lavalin to establish a Metsim Model, a mass balance and for equipment sizing calculations. Select process design data for the major process areas are also presented in this chapter.

17.2.2 Recovery Models

Flotation Recovery Model

A Concentrator Metsim Model has been developed by RHM to reflect the proposed modified flotation flowsheet and has been used to evaluate process parameters based on the flotation testwork executed at ALS. The flotation flowsheet modifications included converting the grinding circuit from a Semi Autogenous Mill/ Ball Mill (“SAB”) to SAB and Pebble Crusher (SABC), a modified flotation circuit using existing equipment and addition of a flotation column to produce a final concentrate in the range 68 – 72% lead grade to feed the Hydrometallurgical Facility. A grade recovery algorithm for the revised flotation flowsheet was developed for use in assessing flotation concentrator performance across the range of anticipated ore feed compositions.

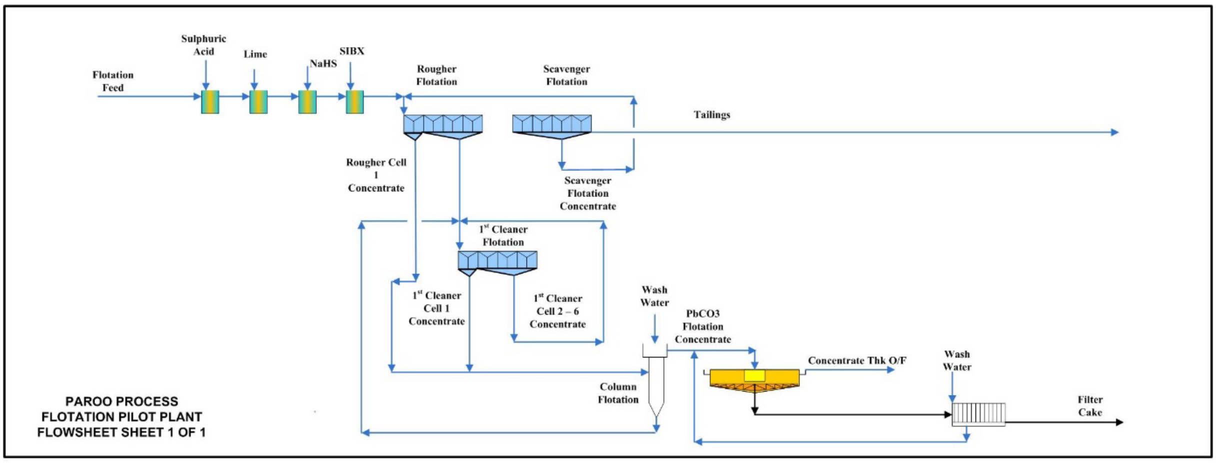


Figure 65: Modified Flotation Concentrator Flowsheet

Source: RHM (2018)

Hydrometallurgical Facility Model

A Metsim Model was developed by SNC-Lavalin for a "base case" mineralogy comprising 78.6% lead carbonate (cerussite) and 9.7% lead sulphate (anglesite) at overall concentrate grade of 70% lead. The other major minerals assumed to be in the concentrate (based on XRD analysis) are Pyromorphite (1%), Galena (2.6%, Leadhillite (0.6%), Kaolinite (1.0%), Magnetite (3.2%) and Quartz (2.7%). On completion of the variability testwork, additional Metsim Models were run for the assumed minimum (3%) and maximum (15%) anglesite levels, which have been run to assess the impact of the changing concentrate mineralogy on mass balance flows and operating costs.

17.2.3 Testwork Interpretation – Existing Concentrator Modifications

Comminution

Estimates of grinding power requirements are based on the comminution characteristics of the ore within the first 10 years of operation, although it should be noted that there is little difference in the work indices of the early years of operation. The ore becomes significantly harder in the latter years of operation. Because the ore is also significantly bimodal in terms of ore hardness the milling operation can be throughput- limited if significant silica pebble build-up occurs in the mills.

The preferred option to accommodate the hard component of the ore is to pebble port the SAG Mill and institute pebble crushing of the SAG Mill pebble product, to minimise the SAG Mill circulating load.

Flotation

The flotation circuit design has been based on an analysis of the batch variability testwork and the pilot plant operation. The initial phase of flotation testwork on the variability composites identified a strong negative relationship between pH and flotation kinetics and recovery, when a target pH for the flotation feed was not set. A testwork report provided by RHM indicated that conditioning with sulphuric acid and lime sequentially appeared to improve flotation recovery. When tested, this proved to be the case.

The existing circuit is a rougher/scavenger circuit followed by three stages of cleaning to produce a final concentrate. The rougher scavenger circuit and the first cleaner circuit operate in open circuit.

A revised circuit design has been developed as follows: Rougher concentrate from the first rougher cell passes directly to a second cleaning circuit and typically contains 55 – 65% of the recoverable lead at a grade of approximately 60% lead.

The remaining rougher scavenger concentrate passes to a cleaner/cleaner scavenger circuit with a second high grade concentrate being produced from the first cleaner and the concentrate from the cleaner scavenger recycled to the head of the first cleaner circuit. The cleaner scavenger circuit operates in open circuit.

The combined first rougher and first cleaner concentrates are combined and treated in a single stage of column flotation which essentially operates as a slimes rejection circuit. Column concentrate passes to the concentrate thickener. Column tailings are recycled to the head of the cleaner scavenger circuit. Lead concentrate grades in the range 68 – 72% lead can be produced in this circuit. A key feature of the RHM ores is the presence of high levels of ultrafine clay slimes that typically report to the final concentrate. These slimes have a significant negative impact on flotation concentrate filtration rates in the concentrate filter and also in the one filter circuit in the Hydrometallurgical Facility, where the slimes are concentrated after the MSA leach.

Flotation Feed Conditioning

Additional conditioning steps were introduced into the flotation circuit to control the final flotation pH. The flotation pH significantly impacts both flotation kinetics and overall lead recovery. When the pH increases to above 8.5, flotation performance was impacted negatively, and, depending on the ore type being processed, the final flotation feed pH needed to be controlled to an optimum level. An initial acidification step was employed using a small addition of sulphuric acid to reduce the pH to 5.5. Empirically and inexplicably, a small addition of lime to raise the pH to 6.5 – 7.5 ahead of the NaHS conditioning step was also found to be beneficial to overall lead recovery of some ores. The conditioned feed then passed through NaHS and SIBX conditioning steps, as per the current plant arrangement.

Flotation Feed Density

Generally, a flotation feed density of 35% has been employed historically on the flotation plant. The current testwork highlighted improvements in flotation recovery if the flotation feed density was reduced to 30% solids. This would occur in practice on an as needs basis dictated by operational experience, predominantly when ores containing a high proportion of fines are treated. The improved flotation kinetics achieved using the conditioning steps detailed in Section 5.3.3 above negate any impact of reduced residence time on the flotation recovery, and also reduce slimes entrainment in the rougher and cleaner concentrates.

Flotation Reagent Selection

The existing flotation concentrator reagent regime was applied to all testwork. NaHS addition was placed under ORP control in the testwork whereas on site the NaHS is ratioed to the lead feed grade. This approach reduced the NaHS addition significantly but at the expense of an equally significant increase in SIPX consumption.

Flotation Concentrate Thickening

A new concentrate thickener will be installed based on the design parameters of the revised duty plus a reasonable safety margin. A specific settling rate of 0.15 t/m^2/h has been used for the thickener sizing calculation plus a 50% design margin to allow for variable concentrate production rates.

Concentrate Filtration and Concentrate Properties

Concentrate Filtration

Concentrate filtration of the original concentrate was always an issue due to the high slimes content of the concentrates derived from conventional cleaner flotation, however, introduction of column cleaner flotation into the circuit has improved the filtration such that the revised filtration duty including concentrate washing is well within the capacity of the existing Metso VPA Filter press. Filtration rates now average about 5 t/m^2/h, compared with the original 200 kg/m^2/h achieved when the flotation concentrator was operating.

Concentrate Analyses

The major element analyses for concentrates produced from the two pilot plant runs and the variability samples has been undertaken.

Concentrate Particle Size Distribution, PSD

The size distribution of the lead concentrate produced in the pilot plant has a P_{80} of 103 microns and exhibits a distinctly bimodal size distribution due to the presence of ultrafine clays.

DeS Leach Residue Flotation

The DeS Residue from the Hydrometallurgical Facility will be returned to the concentrator to recover lead minerals for reintroduction into the Hydrometallurgical Facility. It is proposed that existing cleaner flotation equipment in the current plant be used for this duty.

Sulphur Flotation

Elemental sulphur is formed by the reaction of galena with ferric methane sulphonate and while the formation rate can be measured as a few tens of kg per hour, it will be necessary to separately recover Sulphur on an intermittent basis to avoid a build-up in the circulating load. Sulphur will float readily with frother only as a reagent scheme and the existing 3rd stage cleaner which is currently redundant can be used to recover Sulphur which can be passed to the flotation tailings.

Flotation of Lead Minerals from DeS Leach Residue

The tailings from the Elemental Sulphur float will be conditioned using sulphuric acid, lime and NaHS to activate the lead minerals and float a concentrate. Flotation will be incorporated into existing equipment. The recycled lead concentrate will be added to the concentrate thickener and re-join the concentrate stream into the concentrate filter.

17.2.4 Testwork Interpretation for Hydrometallurgical Facility

Feed Preparation

Filter cake from the existing concentrate filter is dried at 110°C to drive off residual flotation reagents which would otherwise cause frothing issues in the MSA leach circuits. Steam from the HRB system will provide the required heat input and condensate will be returned to the boiler feed tank. Dryer offgas is scrubbed in a venturi scrubber to recover any entrained concentrate dust.

The objective of the feed preparation circuit is to repulp washed flotation concentrate in MSA thickener overflow to 65% solids which contains minimal MSA to avoid gas evolution in either the repulp tank or the surge tank.

MSA Leaching

The objective of the MSA leach is to dissolve all the lead minerals present in the concentrate that are soluble in MSA and to liberate any lead minerals encapsulated in cerussite, predominantly anglesite. The lead in a typical concentrate is predominantly present as cerussite (85 - 90%) with the remainder of the lead predominantly present as anglesite. Galena and Pyromorphite can also be present.

The initial MSA leaching step is required to liberate the anglesite ahead of the following DeS leach so that the anglesite can be converted to cerussite. Much of the anglesite is enclosed in cerussite and is not amenable to conversion in the first instance. The MSA Releach discharge slurry at 65^{O}C is added to the head of the leach train so that the residual acid and ferric ion can be utilised in the MSA leach. After the leach the slurry is degassed to enable effective liquids-solids separation in the subsequent thickener and CCD circuit.

MSA Leaching Solid / Liquid Separation

Based on the batch variability and pilot plant testwork the solids mass feed rate to the MSA leach residue thickener is expected to be variable depending on the lead mineralogy of the concentrate being processed. The residue mass flow variability is largely attributable to the variable proportion of anglesite in the MSA leach residue, which does not however impact on the thickener sizing. Once degassed, the leach residues settle quickly.

The pilot plant data from a larger thickener unit is considered more reliable than the smaller cylinder tests carried out during the POC and variability testwork. Based on pilot plant performance, underflow densities are expected to range between 35% to 50%. Thickener U/F densities of 40 w/w % have been assumed as a design basis.

The MSA leach residue thickener size is dictated by the liquor rise rate due to the variable but low thickener feed density. Thickener overflow passes directly to impurity removal.

Thickener underflow is passes down a six-stage CCD circuit, which is used to wash the MSA leach residue ahead of the DeS leach.

DeS Leaching

The objective of the DeS leach is to use sodium carbonate to react with anglesite in the MSA leach residue to produce lead carbonate that can be floated in the existing flotation concentrator and returned to the MSA leach circuit. There is very little solids mass loss across the DeS leach circuit. The discharge slurry is returned to the existing concentrator to a dedicated flotation process to recover the remaining lead minerals to concentrate.

Leach Area Scrubber

The objective of the leach area scrubber is to remove any entrained lead and MSA from the carbon dioxide gas stream evolved in the MSA leaching circuits using a wet packed bed scrubber. The lead concentration in the offgas will be monitored and held below 0.5 mg/m^3.

The scrubber has a design entrained lead discharge level of 0.3 mg/m^3.

Impurity Removal

The impurity removal circuit is designed to precipitate iron, arsenic and aluminium from the MSA Leach residue thickener O/F using lime in a series of six reactors so that the resultant precipitate can then be thickened and the Impurity Removal Thickener O/F passes to electrolyte filtration.

Lime is used as the neutralising agent to completely remove residual acid, iron and aluminium from the advance electrolyte, with the circuit operating in the pH range 4.0 – 4.5. In operation it is expected that the impurity removal circuit could possibly be bypassed eliminating the lime requirement, depending on the level of impurities in solution. However, currently the impurity removal circuit is used to build up iron in the MSA leach circuit to effect oxidation of galena by recycling thickener underflow back to the head of the MSA Leach circuit.

The design basis for this area relies on the precipitation and thickening data derived from the pilot plant operations. A flux rate of 0.02 t/m2/h at 30% solids (w/w) was selected for design purposes. The design of this thickener is rise rate controlled.

Electrolyte Filtration

The advance electrolyte is filtered to remove any residual suspended solids before passing the clarified solution to lead electrowinning.

Bleed Treatment

A small bleed stream of 3 m^3/h spent electrolyte is treated through successive treatment stages to recover the contained lead and MSA and to precipitate a range of impurities such that the precipitates can be thickener filtered and washed to recover the contained acid.

Bleed Electrowinning

A bleed stream of spent electrolyte is required to remove minor impurities from the overall process flowsheet. In order to treat these impurities, it is first required to plate out the lead contained in the bleed stream to the minimum level sustainable.

Acid Recovery

A packaged acid recovery plant is used to maximise initial acid recovery using resin bed technology.

Once the lead and acid are depleted from the bleed stream minor element removal can be undertaken.

Bleed Precipitation

The bleed precipitation circuit design parameters are based on ALS testwork. The objective of the circuit is to precipitate metal methane sulphonate salts and generate calcium methane sulphonate which will be regenerated to methane sulphonic acid in the next stage of bleed treatment.

Bleed Leaching

The bleed leach circuit design parameters are based on ALS testwork. The objective of the circuit is to precipitate calcium and strontium methane sulphonate and generate methane sulphonic acid. The gypsum precipitate is recovered and washed before the precipitate is pumped to disposal.

Lead Electrowinning

Lead electrowinning design parameters are based on current state of the art numbers which, to the extent possible have been replicated in the pilot plant testwork, however it is not expected that pilot plant operations will provide any design parameters other than to confirm the ability to generate high purity lead cathode.

Lead **Melting**

There are three key areas within this area:

- Lead Melting
- Lead Casting
- Lead Starter Sheet Preparation

Evaporator

In order to maintain a positive water balance, it is necessary to evaporate water from the process liquor.

18 Project Infrastructure

18.1 On-site Infrastructure

Key infrastructure for the Mine includes:

- Processing facilities
- Hydrometallurgical Facility
- Power station and infrastructure
- TSF pipeline
- Gas pipeline and infrastructure
- Stores, maintenance and laboratory
- Fuel and chemical storage
- Magazine
- Contractor workshop
- Landfills
- Waste water treatment facilities
- Reverse osmosis plant
- Offices and accommodation village.

Figure 66 shows the key site infrastructure overlain on a regional aerial photograph of the operation.

18.1.1 Processing Facilities

The existing lead processing facilities have been described earlier (refer Section 13) and consist of infrastructure to allow lead ore to be processed through a series of crushing, milling, flotation concentration, and filtration.

18.1.2 Hydrometallurgical Facility

The Hydrometallurgical Facility has been described in detail earlier (refer to Section 13) and will consist of infrastructure to allow lead concentrate to be processed into lead ingot by acid leaching, solid liquid separation, electro winning and melting operations.

18.1.3 Mine Offices

Mine offices comprise 14 transportable buildings used for the following purposes; administration, first aid room, crib room, meeting rooms, clean/ dirty change area(s), laundry, and ablution facilities.

The transportable buildings are connected via concrete footpaths and wooden walk trellising.

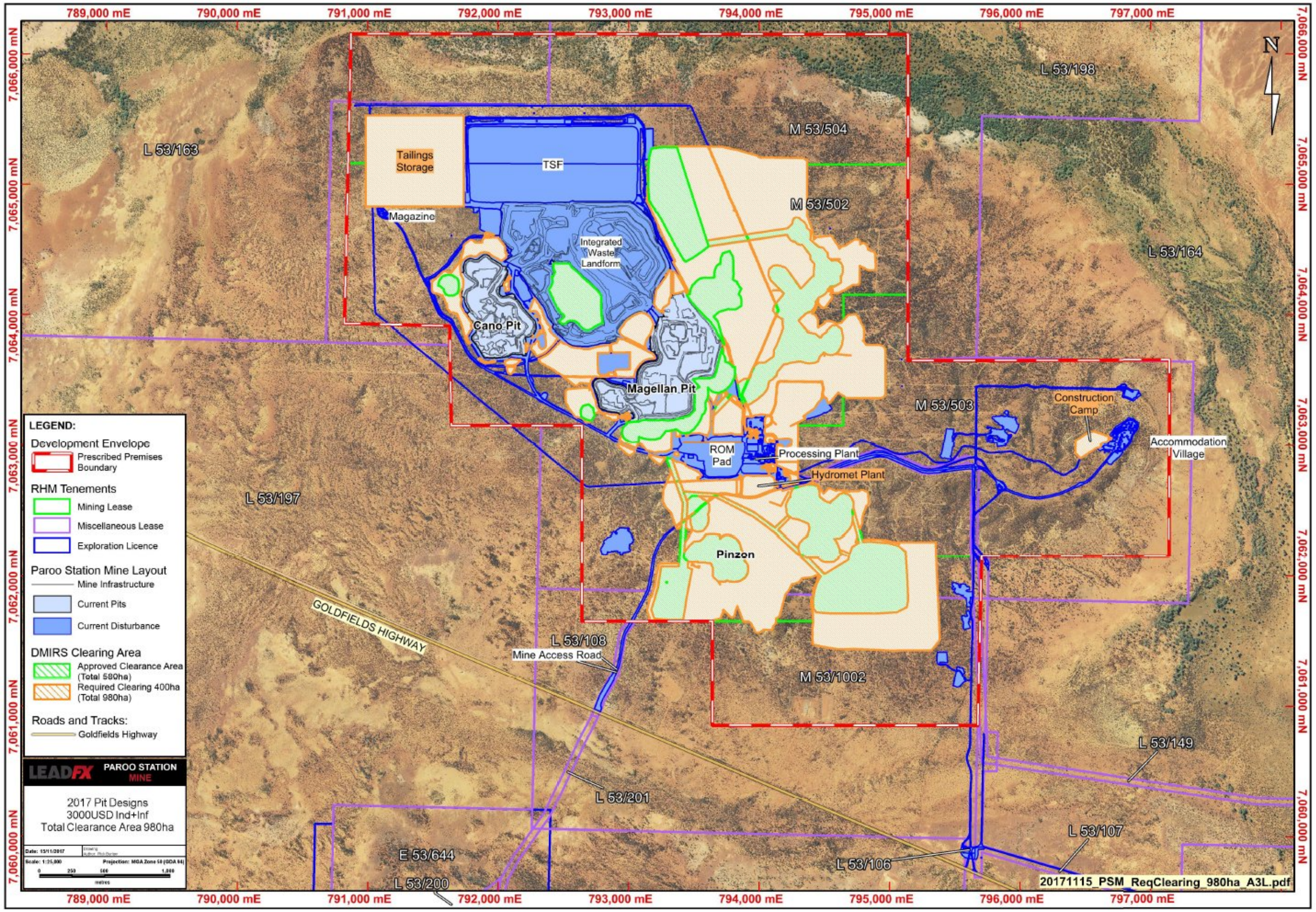


Figure 66: Site Layout

Source: RHM (2018)

18.2 Water Supply & Management

18.2.1 Borefield

Processing water requirements for the operation are currently met from a production borefield located approximately 4 km southeast of the Mine (Figure 67).

The borefield comprises four production bores (PB01 to PB04) with production bores PB01, PB02 and PB03 installed to depths between 12 and 18 m below the surface and draw water from a calcrete aquifer. Production bore PB04 is installed to 84 m below the surface and draws water from a fractured rock formation.

Each production bore has an individual generating set that can be operated remotely to supply power for each pump. It is in the scope of the Hydrometallurgical Facility DFS to install dedicated overhead power lines to each of the production bore for future power supply.

The water is of variable quality (total dissolved solids ranging from 1,000 milligrams per litre (mg/L) in PB01 to 12,000 mg/L in PB04); however, there are no known constraints on water quality for processing supply.

Future mine production increases would result in an increased demand for processing water supply and preliminary exploration undertaken during 2014 using airborne survey equipment has identified an area prospective for a potential palaeochannel aquifer to the north of the TSF. Further work is required to locate and define water in suitable quantities and with acceptable quality.

The current groundwater abstraction licence is for 2.5 Gl per annum.

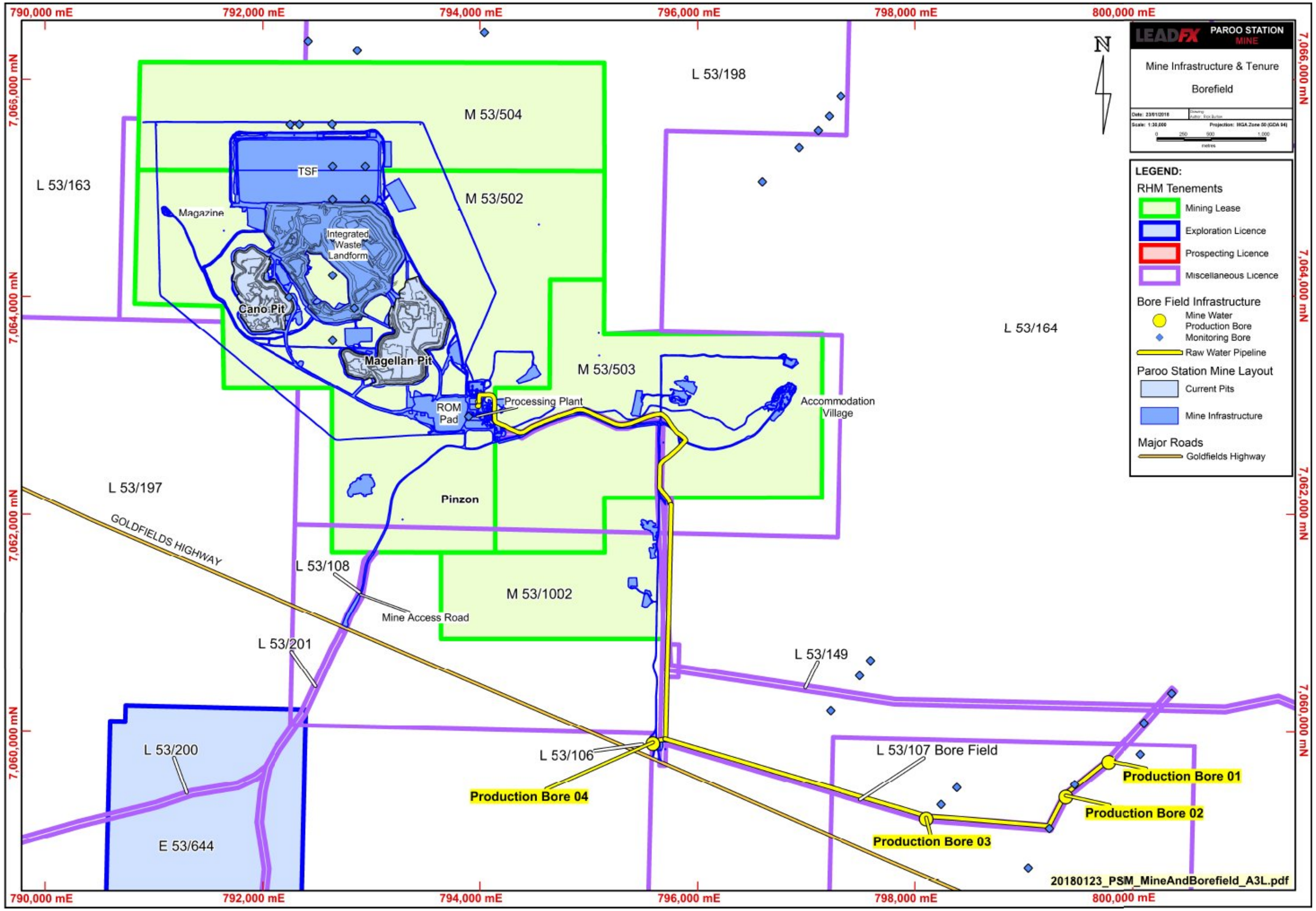


Figure 67: Borefield Location

Source: RHM (2018)

18.2.2 RO Plant

A RO desalination plant is located within a sea container located next to the raw water dam.

RO treated water is stored within three 16 kilolitre (“kL”) holding tanks located adjacent to the RO plant before distribution through a series of surface and subsurface PVC pipelines to four additional holding tanks located across site. A dedicated RO plant will provide the necessary water supply for the Hydrometallurgical Facility.

18.2.3 Waste Water Treatment Facilities

Waste water treatment facilities include a sewage farm, comprising a two celled cascading water treatment installation approximately 150 m southeast of the mine site administration block.

A second treatment facility includes an aerobic disc water treatment plant which sends treated water via a pipeline to an evaporation pond located 300 m north of the accommodation village.

18.3 Service Roads and Bridges

The operation is situated on elevated area that is significantly above the level of the surrounding plains. No bridgework is required on the operations tenements.

18.3.1 Roads

A well-maintained gravel access road of approximately 5 km extends from the Goldfields Highway to the processing plant, mine administration area and the accommodation village.

18.4 Mine Operations and Support Facilities

18.4.1 Haul Roads

The haul road consists of a compacted silcretised / clay quartz breccia with clean mine waste utilised for bunding positioned along the edge of the road. Further haul roads are planned to coincide with mine expansion through the development of the unmined deposits.

18.4.2 Magazine

A magazine area is located in the northeast corner of mining lease M53/502, within a fenced and secure compound. When in operation, the facilities include two ventilated transportable buildings for ammonium nitrate/fuel oil storage. Explosives are transported to site by a contractor when required.

18.4.3 Mining Contractor Workshop

The mining contractor workshop is located approximately 200 m east of the processing plant area. The facilities include:

- Hydrocarbon storage sea container
- Large shed/workshop area with concrete apron and footpaths
- Truck and light vehicle washdown bay and triple interceptor oil water separator
- Two 53 000 L double sheath wrapped fuel tanks
- Change rooms
- Lunch, administration and ablution buildings.

18.4.4 Truck Washdown

A truck washdown area is located adjacent to the main administrative block and includes concrete apron, drainage sump, water storage and sump pump. Water from the truck wash down bay, laundry and change area showers is collected via sump pumps and pumped to the TSF via the tailings discharge line.

18.5 Process Support Facilities

18.5.1 Tailings Storage

The TSF, is a conventional paddock impoundment design located approximately 2.5 km N-NW of the main administration area, consisting of two cells with multi point spigotting and occupying an area of 85 ha. Each cell has a central decant and decant water is returned to a process water dam via a submersible pump return pipeline. Annual geotechnical and operational audits are conducted, with the most recent completed in March 2018. The establishment of an IWL to store tailings was approved under Part V of the EP Act and DMIRS in 2017. The IWL will be embedded within the existing waste rock landform south of the existing TSF. The IWL will be concurrently constructed as the waste rock is placed. The waste rock will thus provide a substantial portion of the tailings confining embankments.

The tailings storage methodology will remain unchanged. Tailings material characteristics are not expected to change as no changes in geology of the waste materials or the concentrator plant operation. The tailings discharge from the Hydrometallurgical Facility will be pH amended and equate to approximately 1% of the total tailings stream.

The expected additional tailings storage volume is 19 million t (“Mt”) for the Mine extension, taking the total stored volume to 35 Mt.

Golder (2018), have undertaken a tailings storage options study for the planned total storage volume. The most favoured option is an IWL consolidation consisting of the existing TSF, the currently approved IWL and the Cano pit once mining in that pit is complete as depicted in Figure 68. The consolidated IWL will include progressive waste rock walls lifts to the outer margins of the three structures such that the consolidated IWL will end up as one tailings storage structure. Detailed design and regulatory approvals will be required for the consolidated IWL which can occur once the Mine is operational as current tailings deposition approvals are in place and will cover no less than the first 5 years of operations.

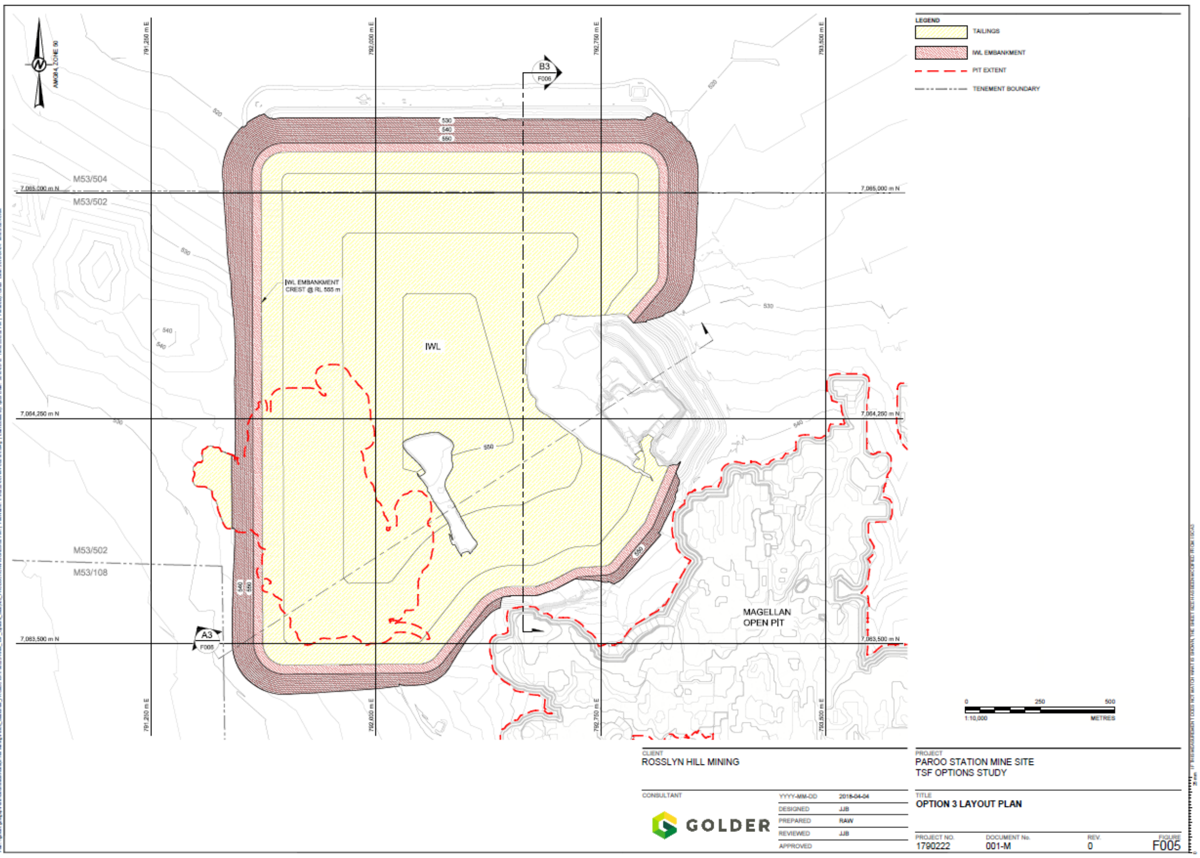


Figure 68: Tailings Storage Layout - Option 3

Source: Golder (2018)

18.5.2 Stores, Maintenance Workshop and Laboratory

The stores area is located in close proximity to the processing facility. The stores area contains warehouse facilities, hydrocarbon storage areas, laydown areas, and dangerous goods storage.

Maintenance facilities include a workshop and supporting infrastructure to service fixed and mobile plant maintenance requirements.

The laboratory facilities are appropriate to allow for sample preparation and analysis for the mining and processing requirements.

18.5.3 Reagent and Fuel Storage

Processing reagents are transported to the Mine area and stored in four storage tanks contained within a concrete lined apron and bunded reagent area.

There are three fuel storage facilities in the vicinity of the processing facilities and one at the accommodation village.

Diesel storage facilities comprise:

- Two 110,000 L double-sheathed wrap tanks
- One 16,000 L day tank for the power station reserve generating sets
- One 16,000 L day tank located at the accommodation village to power the accommodation generating set.

18.6 Additional Support Facilities

18.6.1 Accommodation Village

The majority of the workforce is sourced from Perth and works on a fly-in / fly-out rotational basis. The accommodation village provides accommodation for up to 230 people on rotation and is located approximately 3.5 km east of the processing plant and covers an area of 3.62 ha (Figure 69).

Facilities include:

- Wet and dry mess
- Camp kitchen
- Small swimming pool
- Gymnasium
- Common television, phone, internet room
- Car park
- Camp management transportable buildings
- Contractor storage shed
- Two laundries.

All facilities are connected by concrete footpaths.

The village is fully fenced and a cattle grid is in place to minimise cattle entering the area.

Figure 69: Accommodation Village

Source: RHM (2018)

18.7 Power Supply and Distribution

During the last operational phase, power was generated on site via a natural gas fired power station supplemented by diesel power generation facility. The power station consisted of five natural gas powered QSK60G Cummins generators (1,375 kVA each) with six Cummins K50 diesel generators (1,000 kVA each). The facility comprising the six diesel powered generators is owned by RHM while the five natural gas powered generators are leased.

As part of the Hydrometallurgical Facility DFS, a complete operation power study was completed. A new power station is proposed comprising nine gas fuelled engines generator sets. The engines are nominal 2 MW, are self-contained units and will power both the Hydrometallurgical Facility, the existing concentrator plant and all other ancillary loads on site.

A pipeline provides for delivery of natural gas to the operation (for the new electricity generation units), from the Goldfields Gas Pipeline, which passes to the east of Mine.

18.7.1 Gas Pipeline and Infrastructure

The natural gas pipeline extends 37 km from the Goldfields Gas Pipeline east of Wiluna to the operation. The pipeline is wholly owned by RHM subsidiary Redback Pipelines Pty Ltd with the Paroo Station as the sole user.

Construction of the 37 km gas pipeline commenced in September 2006 and was completed in December 2006 with hydrostatic testing completed in March 2007.

In 2014, the gas generators outlined in Section 18.7 were added as primary power generation, with the existing diesel powered generators being retained as reserve supply.

The sizing of the pipeline with a licenced capacity of 4.9 TJ/d is more than adequate to meet the future needs including the Hydrometallurgical Facility as the forecast daily consumption is 3.6 TJ/d.

18.8 Transport

Road and rail transport services will be provided by a contractor(s) to both supply reagents to site and for the transport and shipment of lead ingots. It is expected the lead ingots will be transported by a combination of road and shipping to the point of sale with the 25 kg ingots likely to be packaged into in 1 t bundles for transport from the mine site.

18.9 Off-site Infrastructure and Logistic Requirements

Logistic support to the operation will be provided by a combination of the onsite and offsite RHM and Lead*FX* resources and industry consultants as required.

19 Market Studies & Contracts

19.1 Overview

The Mine is currently moving from a lead concentrate market to a London Metal Exchange (LME) grade lead metal market, through the construction and operation of an onsite Hydrometallurgical Facility. The following sections summarise aspects of the lead metal market.

19.2 Lead Markets

Lead is used in lead-acid batteries, building construction, bullets and shot, weights, as part of solders, pewters, fusible alloys and as a radiation shield. Lead has the highest atomic number of all the stable elements.

Approximately 86% of global lead metal consumption is attributed to the production of lead-acid batteries which are used in most vehicles, and as back-up and storage media for renewable energy sources, such as wind and solar.

Lead acid batteries are also vital as a backup emergency power supply for critical infrastructures in hospitals, telephone networks and for emergency services when main electricity supplies fail.

Electric vehicles, hybrids and other renewable energy vehicles require lead acid batteries to power the 12V accessories unrelated to the drive line power generation of the vehicles.

Market analysts are predicting that there will be a global deficit in mine product of lead metal as represented in Figure 70 over the coming years.

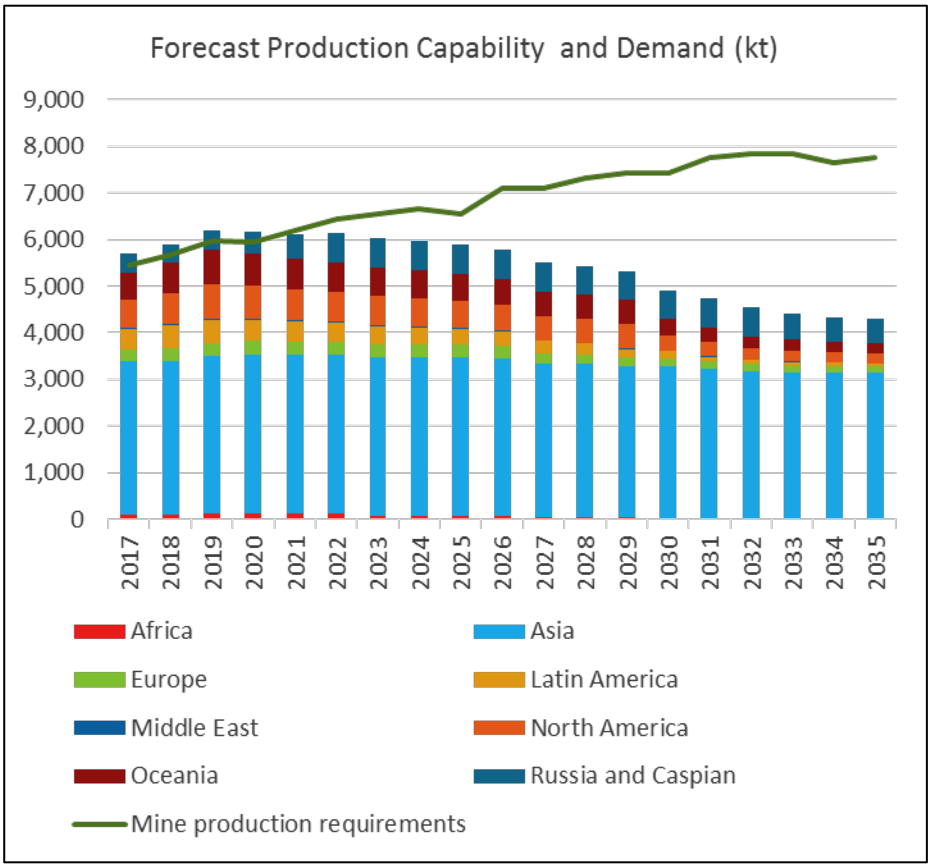


Figure 70: Forecast Production Capability and Demand from Existing Sources

Source: Wood Mackenzie (2017)

19.3 Historic Commodity Prices

19.3.1 1975 to 2000

Lead metal demand stagnated over a 20-year period (1975 – 1995) as a consequence of the then relatively large market for lead in gasoline and paints being almost completely removed.

During this period demand was almost flat with the growth in battery use balancing the lost market segments. Large excess stocks were accumulated which have only been recently exhausted.

Prior to the year 2000, the price of lead metal remained relatively stable. After the year 2000, China began to emerge as a dominant producer and user in the market which caused a significant change in the supply and demand fundamentals of lead.

19.3.2 2000 to Present

Since the year 2003, the price of lead metal has been volatile and is generally affected by international economic and political conditions, levels of supply and demand, producer, LME and other inventory levels such as unofficial Chinese inventories, inventory carrying costs and currency exchange rates.

During 2007, the market established new all-time highs for the price of lead metal, reaching approximately USD 3,980/t on the LME. In 2008, lead prices declined dramatically, along with other base metals due primarily to the global economic crisis, reaching a low of USD 880/t.

In 2010, the price was extremely volatile as growth concerns in China coupled with questions on the viability of the economic recovery of the Western world pervaded the marketplace.

In 2011 and 2012, the price of lead was again extremely volatile due to projected slower growth in Chinese lead consumption and global economic uncertainty related to the European sovereign debt crisis.

2015 has seen the declines of late 2014 maintained with the price of lead at approximately USD 1,725/t for cash buyers. The voluntary and involuntary mine production cuts in 2015 and 2016, compounded by tougher Chinese environmental clampdowns curtailing output, continued to cause a severe drawdown of global lead concentrate inventories throughout 2017 leading to prices above USD 2,600/t in early 2018.

19.4 Life of Mine Planning Assumptions

An analysis of short and longer-term supply and demand dynamics by reputable industry leaders in commercial intelligence in the base metals sector was undertaken and appropriate forecasts of lead metal price and AUD:USD exchange rates were selected. The LOM planning work developed by RHM in late 2017 is based on a medium USD 2,250/t lead price. Market analysts are forecasting demand for refined lead to grow at a steady pace, averaging 2.0% p.a. in the long term to 2035. Many base metal specialists still predict lead metal prices increasing to >USD 2,400/t. The World Bank Group's Commodity Markets Outlook (October 2017) suggests that lead prices should remain above USD 2400/t till 2023 with a softening to USD 2300/t by 2030.

19.5 Economic Analysis

The economic analysis undertaken in sections 21 and 22 has assumed a short- to medium-term pricing of USD 2,250/t which is supported by both the 36-month trailing average and consensus pricing. Figure 71 shows a long-term lead price forecast.

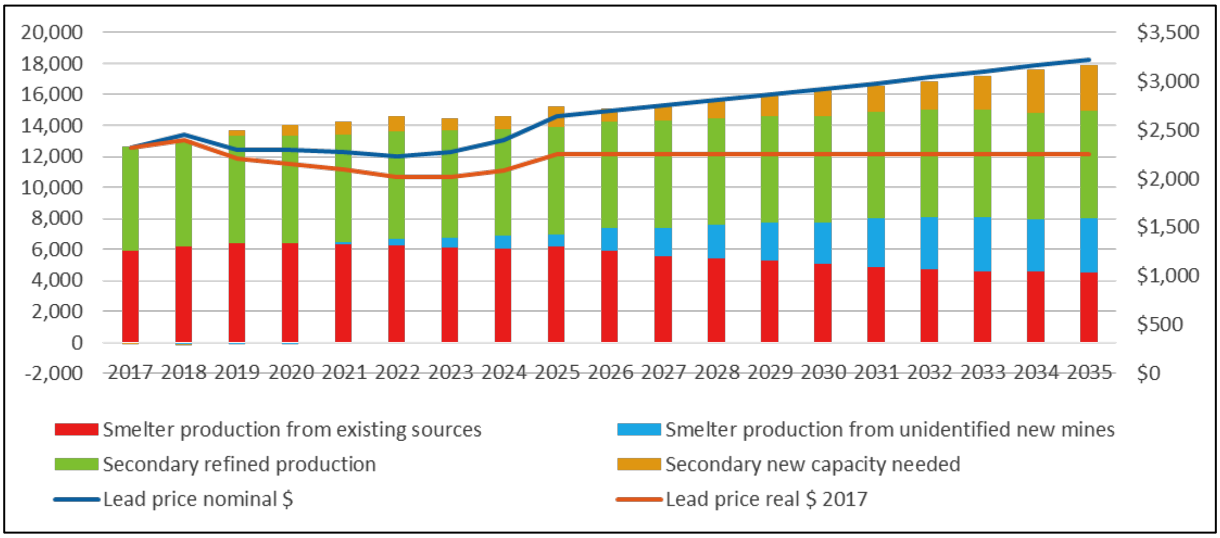


Figure 71: Consensus Long-Term Lead Price Forecasts

Source: Wood Mackenzie (2017)

19.6 Contracts and Status

At the time of writing RHM has a liability of AUD 767,689 owed as early termination of the Aurizon land transport contract. There are no other contracts which incur a liability to either RHM or Lead*FX*.

Other contracts of significance in effect and/or suspension are;

- Mining contract – MACA Limited (in suspension)
- Camp management – Australian Camp Services Pty Ltd (in suspension)
- Gas Pipeline Maintenance – APA Group Pty Ltd

20 Environmental Studies, Permitting & Social or Community Impact

When in operation, the Mine operates in accordance with the requirements of State legislation, standards and codes of practice. Specifically, operations are undertaken in accordance with the Mines Safety and Inspection Act 1994, Mines Safety and Inspection Regulations 1995, Mining Act 1978, Mining Regulations 1981, EP Act 1986 and the Environmental Protection Regulations 1987.

The Company regularly collects and reports occupational health, safety and environmental information to the following State Departments:

- EPA – Environmental Protection Authority
- DWER – Department of Water and Environmental Regulation
- DMIRS – Department of Mines Industry Regulation and Safety
- DOH – Department of Health
- DoT – Department of Transport

Operating conditions and licences for the Mine have been granted and the following are currently in force:

- EPA – Ministerial Statement 905 and 1042
- DWER – Prescribed Premises Licence – L8493/2010/2
- DWER - Licence to Extract Water – GWL96342(4)
- ACMA – Australian Communications & Media Authority – Licences 1970164 and 1970178/1
- DMIRS – Dangerous Goods Site Licence –DGS020079
- DMIRS – Mining Tenement conditions
- DMIRS – Pipeline Licence – PL73
- Radiological Council – Licences LX58/2006 15145 and RS28/2005 14619

20.1 Required Permits and Status

The operation was originally approved under the EP Act, 1986 with Statement 559. The original Mining Proposal required for the mine was presented to Department of Industry and Resources (DoIR; now DMIRS) in September 1999 and the mine was subsequently approved under the Mining Act in July 2004.

On July 15, 2005, RHM received approval from the State Mining Engineer for the amended TSF design from 50 ha to a total area of 64 ha (one 25 ha cell and one 39 ha cell (Cell 2)), which increased the capacity of the TSF to 10.4 Mt. Redesign of the second cell to reflect the DMIRS approval has occurred via changing the footprint area from a square to an oblong, giving the increased capacity. No change to the initial design wall lift method and final crest height was proposed in the redesign.

Production recommenced in late February 2010 and on January 5, 2011 the operation ceased again following an order from the Minister for Environment to cease transportation to enable investigation of reports of potential lead egress to the inside of sealed transport containers. No lead egress was found and a thorough investigation resulted in discovery of a laboratory error. The Minister for Environment announced lifting of the order on February 23, 2011, allowing the operation to recommence as soon as practical after that date.

RHM voluntarily placed the Mine into care and maintenance during April 2011 to conduct an ‘end to end’ review of all operational activities. A parallel review under s 46 of the EP Act was undertaken by the OEPA and the review report was published on October 3, 2011.

This report resulted in changes to conditions of approval by issue of EP Act Statement 905 in July 2012. Statement 905 supersedes all previous conditions and procedures. The Mine remained in care and maintenance until April 2013, and has operated under Statement 905 since then.

On December 9, 2014, the EPA approved an increase in the approved area of disturbance to 456 ha under s 45C of the EP Act to allow for an increase in the size of pits and related infrastructure. A development envelope of 2094 ha (comprising the mining tenements M53/504, M53/502, M53/503 and M53/1002) was also nominated.

In January 2015 RHM voluntarily placed the Mine into care and maintenance due to depressed world metal prices. The operation remains in care and maintenance as at the date of publishing of this report.

On November 15, 2016 Ministerial Statement 905 was amended by Ministerial Statement 1042 which changed Condition 3A to allow the export of lead carbonate concentrate through the Port of Fremantle till July 26, 2024 and changed Condition 18 reducing the financial assurance to 2 M dollars. Additionally, the approved area of disturbance was increased to 580 ha, the tailings storage volume increased to 16 Mt, (through the construction of an IWL), under s 45C of the EP Act, to allow for the anticipated 4.5 year remaining mine life at the time. A licence amendment to construct and operate the IWL was achieved on February 14, 2017 from the Department of Environmental Regulation, (now the DWER).

Currently the Referral Document is before the State EPA, under Part IV of the EP Act, to increase the disturbance footprint by 400 ha, taking the total disturbance footprint to 980 ha, within the development envelope. The Referral Document also includes for an increase of 19 Mt tailings storage capacity, taking the total storage capacity to 35 Mt, to meet the needs of the new forecast mine life.

The document also describes the Hydrometallurgical Facility and the proposed new electricity generation plant at site. On April 4, 2018 the EPA determined their Level of Assessment of the Referral Document. The level set was “Referral Information” (the information contained within the Referral Document being sufficient for their purposes), with a request for some additional information to be supplied by RHM.

The construction and operation of the Hydrometallurgical Facility is currently the subject of a Works Approval with DWER. A prescribed premises licence amendment is expected to follow the commissioning of the Hydrometallurgical Facility.

Updates to the DMIRS approved (2015), Mining Proposal will include the Hydrometallurgical Facility and expanded footprint areas. This will be progressed during 2018. An updated Mine Closure Plan will also be developed to meet DMIRS requirements.

20.2 Environmental Study Results

20.2.1 Flora

A comprehensive survey of the flora of the project area was undertaken in 1999 to provide baseline data, identify any issues of conservation significance, and inform environmental management (Hart *et al.,* 1999). A total of 178 native species were recorded spread over 93 genera and 39 families. The survey also recorded seven weed species in seven genera and families. No Declared Rare Flora or Priority species were recorded.

In October 2009, a desktop assessment was undertaken by Outback Ecology of the future mining areas to determine if the conservation status of species having the potential to occur within the Mine area had changed (Outback Ecology, 2010).

The development envelope area has also been subject to two recent detailed vegetation surveys. The first survey was undertaken in June, September, October and December 2011 primarily to the west of the existing Mine area (Western Botanical 2012), and the second survey was undertaken in May and October 2014 and focused on the area to the east of the existing Mine area (Maia 2015). Collectively these recent surveys have now described the vegetation within the entire development envelope of the Mine. The methods used for this assessment consisted of a review of existing reports followed by an interrogation of available ecological databases as well as detailed site surveys using transects and quadrat methods.

No Threatened Ecological Communities or Priority Ecological Communities or Declared Rare Flora are expected to be impacted by the proposed new disturbance footprint of 980 ha within the development envelope.

Assessment of closure related issues

Rehabilitation of surfaces has been undertaken progressively during the life of the Mine to the extent possible without affecting operations. This has allowed rehabilitation methods to be tested and refined to determine the most suitable and successful method for final rehabilitation.

Recent on-site rehabilitation trials using a suitable growth medium has been highly effective. The growth medium is generated during vegetation and topsoil clearing, where the addition of the upper layer of naturally occurring silcrete is incorporated into the residual topsoil. The topsoils of the Mine area, are skeletal in nature and overlay a generally impervious layer of silcrete. The incorporation of the silcrete to create the growth medium significantly adds to the volume of available growth medium material.

The Mine uses the Land Function Analysis (“LFA”) method (Tongway, Hindley 2004) to actively monitor and record rehabilitation of the waste rock landform (“WRL”) domain. The landform rehabilitation monitoring plan is guiding monitoring activities that make use of the LFA and sets out the required steps when conducting monitoring activities in the field and nominates the right LFA ‘tools’ for each step.

The LFA assessment model has been used to monitor the effectiveness of the progressive natural revegetation on the rehabilitated surfaces (currently a total of 14.74 ha), on the various lifts at the WRL (IWL) using fixed transects.

Planned further studies

The Mine is unique and complex due to naturally occurring (i.e. pre-mining disturbance) high baseline levels of lead in topsoil associated with elevated mineralised outcrops in the development envelope area. Existing industry guidelines and standards for rehabilitation do not prescribe criteria for sites with elevated naturally occurring lead levels. Substantial work is being undertaken by RHM in consultation with key stakeholders to define appropriate mine closure standards and criteria for lead in topsoil that reflect the naturally occurring lead levels in the topsoil unique to the area.

RHM proposes that future iterations of the Mine Closure Plan will refine post-mining land use to reflect outcomes of the work.

20.2.2 Fauna

A comprehensive fauna study was undertaken in 2014 (Bamford 2017), of the development envelope and other selected local habitats. Original baseline studies were conducted in the area in 1999 (Hart *et al.,* 1999).

The Murchison bioregion is rich and diverse in fauna however, most species are wide ranging and usually occur in adjoining regions. More than 40% of the Murchison's original mammal fauna is now regionally extinct. Feral predators (cats and foxes), changed fire regimes and vegetation loss are the threatening processes that affect vertebrate animals (December 2002).

The Bamford desktop survey identified an assemblage of 295 vertebrate fauna species potentially occurring in the Mine area. This comprised 11 frogs, 90 reptiles, 156 birds, 30 native mammal and eight introduced mammal species. A total of 145 species have been confirmed from the site, including seven frogs, 40 reptiles, 76 birds, 17 native mammals and five introduced mammals. A total of 30 vertebrate species of conservation significance fauna species are expected to occur in the study area, with 25 of these considered as currently extant within the region. The assemblage is considered to be relatively intact, within a relatively intact, largely uncleared landscape. Some mammal species are considered locally extinct and a number of species are likely to have been impacted by long-term pastoralism, including Mallee fowl.

Species of Conservation Significance

Significant species expected to be present at least occasionally within the Mine area include two reptiles, up to 19 birds and four mammal species. Species of note include:

- Australian Bustard; recorded in May 2014 – widespread and not reliant on VSAs expected to be impacted by the proposed expansion;
- Bush Stone-curlew; recorded in May and October 2014 – from riparian woodlands in October, away from the proposed expansion area;
- Rainbow Bee-eater; recorded in October 2014 – common and widespread, recorded from areas outside of the proposed expansion areas;
- Brush-tailed Mulgara; a burrow recorded in May 2014, but no individuals trapped in October 2014 – potential to be impacted as its preferred areas of spinifex on sandplain may be impacted by the proposed development (within VSA 1).
- Long-tailed Dunnart; an individual trapped in October 2014 – potential to be impacted as its preferred areas of rocky outcrops may be preferentially impacted by the proposed development (within VSA 1).
- Greater Bilby; recorded by RHM personnel prior to the May 2014 site visit – although the record appears genuine, it is considered most likely to be a dispersing male from a fauna release program north-east of Wiluna. The species is considered unlikely to rely on the dominant VSAs within the Mine area.

These are the species most likely to be impacted by the proposed expansion, although impacts are expected to be minimal.

Vegetation and Substrate Associations

Four VSAs were identified across the Mine area and surrounding landscape:

- Plateau Mulga on cobbles and loam; high in the landscape with some incised drainage lines;
- Mulga woodland on slopes and plains; mid to low in the landscape with occasional emergent Bowgada and very occasional eucalypts;
- Open shrubland on clay flats; lies low in the landscape adjacent to the main paleo-drainage system; and
- Riverine woodland; Eucalypt woodland along major drainage line to north and east; effectively a broad gallery forest.

None appears to be restricted to the proposed expansion areas and most of the areas of interest support VSAs 1 and 2. The riverine woodland, as a narrow corridor, has the most potential to be impacted, however it lies outside the expected areas of mine impact.

Patterns of biodiversity

Biodiversity is likely to be spread across the VSAs, with the most significant areas for fauna considered to be the riverine woodland and fringing shrubland/woodlands, and Mulga over spinifex on red sandy loam. Although the Plateau Mulga on cobbles and loam VSA is considered to have relatively low biodiversity, some areas within it are expected to be important for different fauna taxa e.g. rocky hills (Long-tailed Dunnart) and dense vegetation along seasonal watercourses (birds).

Key Ecological Processes

One of the dominant ecological processes currently affecting the fauna assemblage in the Mine area is hydrology, with other less significant processes including fire, feral species and interactions with native species, habitat degradation due to weed invasion and connectivity.

Stygofauna

Annual stygofauna sampling was commenced under the Stygofauna Sampling Plan (SSP) in November 2004 with the sampling of a number of existing bores and wells within and outside of the Mine impact area (Biota, 2005).

The requirement for the continued implementation of the Stygofauna Sampling Plan was not carried forward into Ministerial Statement 905 when it was issued in July 2012. EPA Report 1415 dated October 2011 stated *"The EPA considers that scientific knowledge has increased and there is no adverse impact on stygal communities. Sampling can therefore cease."* Stygofauna sampling and reporting have not been conducted since that date.

20.3 Environmental Issues

In March 2018, the Company filed its Compliance Assessment Report ("CAR"), along with its three Annual Environment Reports ("AER") for 2017 to the four regulatory authorities.

The CAR and the AERs are the key annual environmental disclosure documents produced by RHM and submitted to the Western Australian regulatory authorities.

RHM disclosed that there are no outstanding environmental issues.

20.4 Operating and Post-Closure Requirements and Plans

20.4.1 Environmental Monitoring and Reporting

The Mine environmental monitoring and reporting requirements are being undertaken to ensure compliance with the relevant approvals and licence conditions.

By letter dated February 2, 2015, the DWER advised the RHM that it could cease sampling programs along the transport route and Fremantle Port while operations are in care and maintenance.

Mine site inspections and audits have been undertaken at various times in the past by DMIRS and DWER.

20.4.2 Mine Closure Plan

RHM submitted an updated Mine Closure Plan (MCP) on March 3, 2015 to DMIRS. The MCP was approved on June 30, 2015. The approved version has superseded all previous versions.

20.5 Post-performance or Reclamation Bonds

20.5.1 Mining Rehabilitation Fund

On September 26, 2014, the RHM was refunded AUD 2.6M in bonding as a result of the coming into force of the Mining Rehabilitation Fund Act of 2012, which requires the payment of an annual levy each year.

Annual payments to the DMIRS (due in September each year), are now based on the disturbed footprint minus the rehabilitated footprint.

Social and Community

There are a number of stakeholders that may be affected by the operation and eventual closure of the Mine. The stakeholders identified for the Mine are:

- Toro Energy Limited (leaseholder of Lake Way Pastoral Station)
- Paroo Station Pastoral Company (leaseholder of Paroo Station)
- Traditional Owners/Native Title Parties (TMPAC)
- Shire of Wiluna
- Department of Water and Environment Regulation (DWER)
- Environmental Protection Authority (EPA)
- Department of Health (DoH)
- Department of Primary Industries and Regional Development (DPIRD)
- Department Mines Industry Regulation and Safety (DMIRS)
- Native Title Tribunal
- Department of Planning Lands and Heritage (DPLH)
- Meat and Livestock Australia
- Local community groups.
- 20 local government authorities along the 1300 km concentrate transport route

Stakeholder consultation has been ongoing and has had a recent focus with proposed Hydrometallurgical Facility and Mine extension.

20.6 Closure Monitoring

Closure performance monitoring is undertaken throughout the rehabilitation of completed land surfaces with an annual monitoring report. Closure monitoring is expected to continue for up to 10 years following final Mine closure, when relinquishment of tenements is successfully approved.

Figure 72: Rehabilitated Landform

20.7 Reclamation and Closure Cost Estimate

20.7.1 Cost Methodology

RHM has identified the anticipated closure costs required for the Mine including the Hydrometallurgical Facility, based on best available information. The cost estimate takes into account all aspects of rehabilitation and closure activities utilising third-party contractor rates.

The above assumptions and methodologies have been applied to the Mine and RHM acknowledges that further investigation and stakeholder consultation is required to refine the post mining land use. As the post mining land use is refined the closure costing will be reviewed to ensure it continues to adequately address infrastructure and maintenance costs for the post mining land use.

Key areas used in the costing assessment are presented in the following sections below:

- Land forms
- Industrial Infrastructure
- Mining Infrastructure
- Water containment facilities
- Groundwater Infrastructure
- Roads
- Exploration
- Water treatment – post-closure
- Post-closure monitoring
- Owner's management (closure and post-closure)
- Contingency

RHM will calculate and continually update the mine closure cost model as information becomes available.

20.7.2 Estimated Cost

RHM has a fully-costed closure cost estimate that is consistent with the current MCP.

21 Capital and Operating Costs

This section outlines the financial analysis for the construction and operation of the new Hydrometallurgical Facility integrated with existing facilities including mining activity and concentrator plant.

Capital and operating costs for the Hydrometallurgical Facility are based on the SNC-Lavalin's estimates. Capital and operating costs outside of the Hydrometallurgical Facility have been estimated by RHM based on historical operating costs and contract structures and incorporating the changes to the flotation regime from the Flotation Recovery Model.

21.1 Capital Costs

The estimated costs for the new Hydrometallurgical Facility are summarised in Table 52.

Table 52: Hydrometallurgical Facility Capex Estimate

Description	Estimate (USD)	
DIRECT COSTS	**USD**	**USD**
Site Development		2,942,512
Hydrometallurgical Facility		78,988,026
Area 2005 - Feed Preparation	6,393,207	
Area 2010 - MSA Leach	1,216,870	
Area 2015 - MSA Leach Residue Thickening and Filtration	5,013,288	
Area 2020 - DeS Leach	822,615	
Area 2030 - Leach Area Scrubber	157,588	
Area 2035 – MSA Re-leach	429,811	
Area 2041 – Tailings	33,299	
Area 2045 - Impurity Removal	1,684,537	
Area 2050 - Electrolyte Filtration	1,181,951	
Area 2056 - Bleed Treatment Electrowinning	4,688,073	
Area 2057 - Bleed Treatment Acid Recovery	2,588,132	
Area 2058 - Bleed Treatment Precipitation	860,004	
Area 2059 - Bleed Treatment Leaching	718,110	
Area 2060 - Lead Electrowinning	22,067,121	
Area 2065 - Lead Melting	11,496,313	
Area 2070 – Reagents	2,883,273	
Area 2071 – Lime	1,527,749	
Area 2090 - Pipe Racks	605,736	
Area 6010 – Services	1,536,516	
Area 6300 - Power Supply	4,592,601	
Area 6500 – Evaporator	3,691,232	
Construction Market Pricing Adjustment	4,800,000	
Power Station		21,840,517
Mobile Equipment		837,325
Subtotal Direct Costs		**104,608,380**

Description	Estimate (USD)	
INDIRECT COSTS		
Temporary Facilities		2,584,000
Freight		1,003,143
Vendor Representation		1,055,344
Spare Parts		2,931,211
First Fills		2,306,186
Accommodation Camp		1,000,000
Heavy Cranage		105,000
EPCM		15,594,546
Subtotal Indirect Costs		**26,579,430**
CONTINGENCY		
Contingency		19,941,251
Total (USD)		**151,129,061**

Outside of the new Hydrometallurgical Facility, certain modification works have been identified as being required in the existing concentrator facilities to increase throughput and improve recoveries (refer to Chapters 13 and 17). The estimated costs for the modifications to the existing concentrator, re-energisation of RHM's lateral gas pipeline and construction of an IWL for future tailings and waste rock disposal are summarised in Table 53.

Table 53: Other Capex Estimate

Description	Estimate (USD)
Pebble Crushing	1,775,100
Replacement Thickener	1,943,026
Flotation Modifications	1,366,679
Lateral Gas Pipeline Recommissioning	750,000
Integrated Waste Rock Landform	1,500,000
Total Estimate (USD)	**7,334,805**

Owner's Costs associated with the Hydrometallurgical Facility are set out in Table 54.

Table 54: Owner's Costs

Description	Estimate (USD)
Post-DFS Pilot Plant	750,000
Project Execution	2,000,000
Insurance during Construction	500,000
Pre-Production Working Capital (prior to ramp-up period)	8,691,331
Total Estimate (USD)	**11,941,331**

Provision has been made for care and maintenance costs and Lead*FX* corporate costs ("Company Costs") to Hydrometallurgical Facility first production, set out in Table 55.

Table 55: Company Costs to First Production

Description	Estimate (USD)	Comments
Care and Maintenance Costs	5,940,176	1 June 2018 to 31 December 2019
Lead*FX* Corporate Costs	823,500	1 June 2018 to 31 December 2019
Total Estimate (USD)	**6,763,676**	

The summary of total costs to Hydrometallurgical Facility first production are set out in Table 56.

Table 56: Total Cost to First Production

Description	Estimate (USD)	Comments
Hydrometallurgical Facility Capex	151,129,061	SNC-Lavalin estimate
Other Capex	7,334,805	RHM estimate
Owner's Costs	11,941,331	RHM estimate
Company Costs to First Production	6,763,676	RHM estimate
Total Estimate (USD)	**177,168,872**	

21.2 Fixed Operating Costs

21.2.1 Labour

The staffing structure was determined by RHM, and for the Hydrometallurgical Facility in conjunction with SNC-Lavalin.

The workforce for the Hydrometallurgical Facility is based on the same operating principles as the historic mining and processing operations, two 12-hour shifts per day, and seven days a week.

The workforce is accommodated on site in a purpose-built accommodation village and is managed on a fly-in fly-out basis (typically 8 days on, 6 days off; 4 days on, 3 days off, 12-hour shifts), with the majority of the workforce living in Perth. All flights are in and out of the Wiluna airport located approximately 30 km from the site.

Many of the historical management and administration positions at the Mine will be shared between the concentrator and the Hydrometallurgical Facility. This includes Operational Superintendents and Shift Supervisors. The cost of employment for local staff has been supplied by RHM and includes applicable oncosts. Table 57 sets out a summary of the annual labour budget by functional area.

Table 57: Annual Labour Budget Summary

Description	Staff (# people)	Salaries (AUD)	Oncosts (AUD)	Total Costs (AUD)
Operations	4	494,450	85,999	580,449
Mining	13	1,739,125	302,466	2,041,591
Production incl. maintenance (Concentrator)	61	7,148,564	1,243,139	8,391,703
Production incl. maintenance (Hydromet Facility)	37	3,862,650	671,663	4,534,313
Sustainability	13	1,530,100	266,095	1,796,195
Supply & Logistics	7	743,900	129,368	873,268
Finance & Commercial	12	1,313,650	228,445	1,542,095
Total	147	16,832,439	2,927,174	19,759,613

21.2.2 Maintenance and Operating Supplies/Spares

Maintenance costs for the Hydrometallurgical Facility have been calculated by SNC-Lavalin using a fixed percentage of the maintainable CAPEX. Maintainable CAPEX cost includes the installed mechanical equipment, electrical, instrumentation and piping costs. Each area of the Hydrometallurgical Facility has an assigned maintenance cost for the equipment in the respective area. The percentages of CAPEX applied for each section of the Hydrometallurgical Facility are shown in Table 58. The power station operating costs is included in the power generation cost.

Table 58: Hydrometallurgical Facility Maintenance Costs

Description	Process Plant CAPEX Mechanical ENG	Allow (%)	Maintenance USD
Total	**51,801,329**	**2.31%**	**1,196,919**

Maintenance costs outside the Hydrometallurgical Facility have been estimated by RHM using the last operationally prepared budget (2015) as a basis. The costs take into account historical knowledge of the maintenance requirements across each functional area and are set out in Table 59.

Table 59: Other Mine Maintenance Costs

Area	AUD pa
Engineering	878,059
Dewatering	487,586
Crushing	227,347
Grinding Primary	1,402,993
Grinding Secondary	614,510
Flotation	338,835
Tailing Storage	294,663
Power	99,095
Water Bore Fields	456,118
Laboratory	27,110
Village	531,188
Water Potable	129,847
Administration	61,865
Mobile Equipment	927,285
Total	**6,476,499**

21.2.3 Other Fixed Costs

The Other Fixed Costs associated with the Mine have all been estimated using the last operationally prepared budget (2015) as a basis except for Hydrometallurgical Facility costs which have been calculated by SNC-Lavalin. Annual costs by department are set out in Table 60.

Table 60: Other Fixed Costs

Area	AUD pa
Mining	
Mining	582,299
Grade Control	503,460
Processing	
Processing	2,472,069
Laboratory	165,232
Metallurgy	142,779
Hydrometallurgical Facility	840,933
Sustainability	
Sustainability	1,018,786
Environmental	386,862
OSH	515,750
Stakeholder Engagement	107,150
Risk Management and Assurance	121,933
Supply & Logistics	
Supply & Warehouse	860,000
Operational Support	
Flights	3,177,027
Administration	887,344
Village – labour	3,336,423
Village – other	346,220
Corporate	
Corporate Labour	675,050
Burswood Office	205,200
Corporate	360,000
Finance & Commercial	420,000
Total	**17,112,517**

21.3 Variable Operating Costs

21.3.1 Royalties

Royalties are payable to the State of Western Australia on sale of lead ingots at the rate of 2.5% of the gross invoice value minus allowable deductions (allowable deductions being ocean transportation costs from the export port in Western Australia to the purchaser).

Royalties are payable to The University of British Columbia at a rate of 1% of the purchase cost of MSA.

In accordance with the terms of the Wiluna Land Access Agreement of 2006 (which superseded the Heritage Agreement dated September 25, 1998 between RHM and the Milangka Native Claimant Group), RHM is required to make a royalty payment of AUD 0.04/t of all ore milled from the Paroo Station to the trustee for the Wiluna native title holders, Tarlka Matuwa Piarku (Aboriginal Corporation) RNTBC (TMPAC), Another Land Use Agreement, dated December 16, 1998 between RHM and the now unregistered Wanmulla Group, provides for a further AUD 0.04/t of all ore milled from the Mine, which may be payable and is a subject of the consolidation of land access agreements discussed below.

A second agreement with the Wiluna native title holders over Magellan Lateral gas pipeline route, requires an annual compensation payment for use of the gas pipeline tenement area. An annual payment of AUD 20,000 was made initially in July 2006 and subsequent annual payments have been made, indexed at the CPI rate for Perth, Australia. RHM is currently endeavouring to consolidate all of the land access agreements to cover the tenure that hosts the Mineral Resources and has assumed royalties under such native title agreement will be as set out in Table 61.

Table 61: Native Title Royalty Rates

Production Royalty	Units	Unit Cost
LME Price below USD 2,000/t	% of gross invoice value minus allowable deductions	0.25%
LME Price between USD 2,000/t and USD 2,300/t		0.30%
LME Price above USD 2,300/t		0.45%
Exploration Royalty	% of exploration costs	15%
Annual Fixed Payment	AUD / year	126,720

21.3.2 Power

The power demand was calculated from a load analysis generated from the mechanical equipment list, and various process data as required to calculate pump heads. The load list contains the vendor-calculated absorbed power for each item of equipment for the Hydrometallurgical Facility, where available. Where this is not available an allowance has been made for absorbed power by the use of load factors applied to the estimated installed motor power consumption. The absorbed power in all cases is multiplied by the utilisation to obtain the actual power consumed. Standby equipment is considered to have no utilisation (i.e. zero hours). The distribution of power consumption for different areas of the Mine is detailed in Table 62.

Table 62: Power Consumption

Concentrator and Other	Fixed (kWh/a)	Variable (kWh/dmt ore)
Primary Crushing		0.75
Mill Feed Systems		0.05
Pebble Crushing		0.48
Milling		8.11
CPS Rougher/Scavenger Flotation	8,065,947	
1st Cleaner	2,151,804	
2nd Cleaner	659,626	
Cerussite Recleaner	966,255	
Concentrate Thickening and Storage	201,632	
Final Con Filtration	587,313	
Tailings		0.11
Reagents	39,217	
Water Services	200,310	
Fire Water Services	216	
Air Services	401,335	
Fuel Services	14,628	
Buildings	3,238,272	
Borefield	1,621,555	
Camp	1,728,000	

Hydrometallurgical Facility	Fixed *(kWh/a)	Variable (kWh/t Pb)
Area 2005 – Feed Preparation	421,930	
Area 2010 – MSA Leaching	124,640	
Area 2015 – MSA Leach Residue S/L Separation	239,036	
Area 2020 – DeS Leaching	122,222	
Area 2030 – Leach Ventilation Scrubber	23,049	
Area 2041 – Spillage Pond	16,320	
Area 2045 – Impurity Removal	1,752	
Area 2050 – Electrolyte Filtration	271,633	
Area 2056 – Bleed Lead Electrowinning	169,491	
Area 2057 – Acid Recovery	2,201,401	
Area 2058 – Bleed Precipitation	22,159	
Area 2059 – Bleed Leach	35,729	
Area 2060 – Production Lead Electrowinning		691
Area 2065 – Lead Melting and Casting		56.4
Area 2070 – Reagents ^	17,378	
Area 2071 – Lime ^	43,137	
Area 6000 – Services	797,513	
Area 6050 – Evaporator	1,399,625	
Area 2900 – Buildings	55,188	

* power costs modelled as fixed costs represent approximately 29% of total power costs

^ reagents and lime are variable depending on the proportion of anglesite, and shown here as fixed consumption for the base case anglesite content.

A unit power cost of AUD 0.130 per kWh (USD 0.098 per kWh) has been applied based on:

- gas purchase cost of AUD 5.48/GJ on a long-term contracted basis;
- gas transportation cost of AUD 5.44/GJ covering transmission via the Dampier to Bunbury Natural Gas Pipeline, the Goldfields Gas Pipeline, and RHM's lateral gas pipeline;
- gas supply management cost of AUD 0.08/GJ;
- heat rate of 9,657 kJ/kWh for the gas generating set selected by SNC-Lavalin for the DFS, and
- gas generating set operating and maintenance costs of AUD 0.024 per kWh specified by SNC-Lavalin.

21.3.3 Mining

RHM's costs associated with mining activity are included under Other Fixed Costs in 21.2.3.

Mining contractor rates are set out in Table 63.

Table 63: Mining Contractor Rates

Description	Units	Unit Cost
Dayworks	AUD/bcm total	0.08
Drill and Blast	AUD/bcm total	2.01
Load and Haul Ore	AUD/bcm ore	4.66
Load and Haul Waste	AUD/bcm waste	3.95
Clear & Grub	AUD/bcm total	0.02
Rehabilitation	AUD/bcm total	0.09
Rehandle (ore from stockpile)	AUD/dmt ore	1.25

21.3.4 Reagents and Consumables – Concentrator

Consumption rates have been determined based on historical production and incorporating the changes to the flotation regime from the Flotation Recovery Model.

Table 64: Concentrator Reagents and Consumables

Description	Units	Consumption Rate	Units	Unit Cost
105 mm Balls	kg/dmt ore	0.485	AUD/t	1,675
65 mm Balls	kg/dmt ore	0.293	AUD/t	1,680
40 mm Balls	kg/dmt ore	0.111	AUD/t	1,607
NaHS	kg/dmt ore	0.756	USD/t	686
SIBX	kg/dmt ore	1.305	USD/t	1,582
Frother	kg/dmt ore	0.035	AUD/t	3,750
Flocculant - Concentrate	kg/dmt cons	0.040	USD/t	2,642
Flocculant - Tailings	kg/dmt (ore-con)	0.040	USD/t	2,642
Sulphuric Acid	kg/dmt ore	0.045	USD/t	188
Lime	kg/dmt ore	0.101	USD/t	254

21.3.5 Reagents and Consumables – Hydrometallurgical Facility

The reagents and consumables section of the operating cost estimate includes all reagent costs. All other equipment consumables are covered in the respective plant area maintenance cost.

Estimated rate of consumption of reagents are based on data from testwork and flowsheet METSIM modelling. In-house price data was used only for low cost items which do not significantly influence the consumable cost. Table 65 summarises the reagent consumption and supply and logistics costs.

Table 65: Hydrometallurgical Facility Reagents and Consumables

Description	Units	Consumption Rate	Units	Unit Cost
MSA Leach				
MSA	kg/dmt con	3.53	USD/t	2,042
Hydrogen Peroxide	kg/dmt con	3.02	USD/t	860
Flocculant	g/dmt con	65.7	USD/t	2,642
Des Leach				
Sodium Carbonate	kg/dmt con	68.8	USD/t	389
MSA Re-leach				
Hydrogen Peroxide	kg/dmt con	5.54	USD/t	860
Impurity Removal				

Description	Units	Consumption Rate	Units	Unit Cost
Lime	kg/dmt con	7.8	USD/t	254
Hydrogen Peroxide	g/dmt con	302	USD/t	860
Flocculant	t/year	0.55	USD/t	2,642
Bleed Treatment				
Sulphuric Acid	kg/dmt con	27.6	USD/t	188
Calcium Lignosulphonate	g/dmt con	237	USD/t	1,313
Aloes	g/dmt con	119	USD/t	7,942
Lime	kg/dmt con	6.4	USD/t	254
Balls	g/dmt con	173	USD/t	1,092
Flocculant	t/year	1.98	USD/t	2,642
Pb Electrowinning		27.6		
Calcium Lignosulphonate	kg/dmt con	3.38	USD/t	1,313
Orthophosphoric Acid	g/dmt con	59.0	USD/t	992
Aloes	g/dmt con	1,694	USD/t	7,942
Anodes	kg/dmt con	663	USD per unit	510
Utilities				
RO water	litres/year	98,952	USD/Kl	1.20
Potable Water	litres/year	2,920	USD/Kl	2.00

21.3.6 Ingot Transport Costs

Ingot produced at the Mine will be transported by a combination of road/ and shipping to a point of sale. The 25 kg ingots will be shipped in 1 t bundles by road freight from the Mine to a nominated port where the ingots will be trans-shipped for delivery. For the purposes of the DFS the shipment port is nominated as Port Hedland and the point of sale is nominated as Shanghai China. A freight cost for the DFS has been supplied by a freight forwarder and used as the basis for transport costs in the operating cost estimate as follows:

- Land freight from the Mine to Port Hedland of AUD 125/t Pb;
- Port fees of AUD 16.25/t Pb; and
- Ocean Freight from Port Hedland to China of USD 42/t Pb.

RHM has assumed inspection fees of USD 5.16/t Pb.

21.4 Sustaining Capital and Decommissioning

An allowance has been made to replace the Hydrometallurgical Facility's light vehicle fleet in full every five years following commencement of operations. Anode inventory is to be replaced every six years and is included as part of the OPEX costs. There is no other sustaining capital for the Hydrometallurgical Facility.

Sustaining capital costs of AUD 1 M per year are included as part of the financial analysis.

Decommission costs of AUD 18,805,775 are included in the financial analysis as set out in Table 66.

Table 66: Decommissioning Costs

Description	Estimate (AUD)
Land Forms	5,591,598
Industrial Infrastructure	4,746,247
Mining Infrastructure	223,083
Water Containment Facilities	105,580
Groundwater Infrastructure	59,582
Roads	282,216
Exploration	28,826
Water Treatment - Post Closure	96,794
Post Closure Monitoring	679,000
Owner's Management (closure and post closure)	5,267,175
Contingency	1,725,674
Total Estimate (AUD)	**18,805,775**

21.5 Production

21.5.1 Mining Schedule

The mining schedule has been calculated by AMC Consultants Pty Ltd based on the Mineral Reserves estimate and the production ramp up schedule, set out in Table 67.

Table 67: Mining Schedule

Year	Ore from Stockpile (dmt)	Waste Mined * (dmt)	Ore Mined * (dmt)	Ore to Plant (dmt)	Ore to Plant (%Pb)
Year 1	887,347	7,430,578	569,422	1.456.769	3.90%
Year 2	0	8,500,000	2,000,000	2,000,000	4.35%
Year 3	395,412	8,895,412	1,604,588	2,000,000	4.32%
Year 4	397,811	8,897,811	1,602,189	2,000,000	4.29%
Year 5	0	6,365,137	2,134,863	2,000,000	4.33%
Year 6	0	6,496,583	2,003,417	2,000,000	4.33%
Year 7	0	6,170,930	2,329,070	2,000,000	4.36%
Year 8	0	5,419,101	2,080,899	2,000,000	4.34%
Year 9	0	5,175,659	2,324,341	2,000,000	4.35%
Year 10	0	5,238,685	2,261,315	2,000,000	4.37%
Year 11	133,905	3,633,905	1,866,095	2,000,000	3.66%
Year 12	0	3,500,000	2,000,000	2,000,000	3.20%
Year 13	0	3,500,000	2,000,000	2,000,000	3.07%
Year 14	473,103	3,973,103	1,526,897	2,000,000	3.10%
Year 15	371,403	1,278,347	1,918,802	2,000,000	3.07%
Total	**2,658,982**	**84,475,251**	**28,221,898**	**29,456,769**	**3.94%**

* dry bulk density for ore of 2.00 dmt/bcm and for waste 1.94 dmt/bcm. Stripping ratio of 2.99

21.5.2 Ingot Production

Production from the flotation concentrator and Hydrometallurgical Facility is set out in Table 68.

Table 68: Production

Year	Lead in Ore Feed (t Pb)	Flotation recovery (%)	Lead in Con (t Pb)	Con Grade (%Pb)	Con Tonnage (dmt)	Hydromet Recovery (%)	Lead Ingot (t Pb)
Year 1	56,878	81.3%	46,243	70.0%	66,062	98.2%	45,396
Year 2	86,903	82.1%	71,307	70.0%	101,867	98.2%	70,000
Year 3	86,360	82.6%	71,307	70.0%	101,867	98.2%	70,000
Year 4	85,819	83.1%	71,307	70.0%	101,867	98.2%	70,000
Year 5	86,614	82.3%	71,307	70.0%	101,867	98.2%	70,000
Year 6	86,640	82.3%	71,307	70.0%	101,867	98.2%	70,000
Year 7	87,170	81.8%	71,307	70.0%	101,867	98.2%	70,000
Year 8	86,831	82.1%	71,307	70.0%	101,867	98.2%	70,000
Year 9	87,094	81.9%	71,307	70.0%	101,867	98.2%	70,000
Year 10	87,333	81.6%	71,307	70.0%	101,867	98.2%	70,000
Year 11	73,126	80.3%	58,718	70.0%	83,882	98.2%	57,642
Year 12	63,923	79.4%	50,744	70.0%	72,492	98.2%	49,814
Year 13	61,335	79.4%	48,692	70.0%	69,559	98.2%	47,799
Year 14	62,053	79.3%	49,200	70.0%	70,286	98.2%	48,298
Year 15	61,484	78.9%	48,510	70.0%	69,300	98.2%	47,621
Total	**1,159,563**	**81.4%**	**943,866**	**70.0%**	**1,348,380**	**98.2%**	**926,571**

22 Economic Analysis

A comprehensive economic analysis has been completed for the Mine incorporating capital costs, operating costs and revenue. This analysis includes the construction and operation of the Hydrometallurgical Facility, start up and operation of the existing mining operations and concentrator facilities, and transport to market and revenue from lead metal sales.

Key assumptions used in the analysis include; AUD:USD exchange rate of 0.75, LME cash price of USD 2,250/t Pb plus an expected premia of USD 85/t Pb based on ingot grade. The economic analysis results are summarised in Table 72.

22.1 Basis of Reporting

The financial results from the detailed economic model prepared by RHM are estimated on the following basis:

- Real US dollars (i.e. no escalation of revenues and costs for inflation);
- No assumption regarding debt financing, and as such the cashflows presented are ungeared;
- Australian corporate tax rate of 30%;
- AUD:USD exchange rate of 0.75 for the entire Mine life;
- LME cash price of USD 2,250/t Pb for the Mine life based on the long-term price in the Wood Mackenzie global lead long-term outlook, and refined lead premia for lead ingot sales estimated by RHM;
- Hydrometallurgical Facility capital costs and operating costs estimated by SNC-Lavalin;
- Mining costs and concentrate processing operating costs and all other capital and operating costs outside of the Hydrometallurgical Facility estimated by RHM;
- Lead ingot transportation costs estimated by SNC-Lavalin; and
- Mineral Reserves estimate and production schedule by AMC.

22.2 Sales Prices

LME lead prices are based on the Wood Mackenzie Q4 2017 Long-Term Outlook.

The forecast sales price of lead ingot produced from the Hydrometallurgical Facility is based on the LME Price Forecast plus refined lead premia of USD 85 per t of lead ingot based on a minimum lead ingot grade of 99.97% Pb. RHM expects refined lead premia of at least USD 110 per t of lead ingot based should a lead ingot grade of 99.99% Pb be exceeded.

The forecast sales prices and sales value for lead ingot by year of production is set out in Table 69.

Table 69: Sales

Year	Sales Price			Sales Value		
	LME Price (USD/t Pb)	Metals Premia (USD/t Pb)	Sales Price (USD/t Pb)	Sales Price (USD/t Pb)	Sales Amount (t Pb)	Sales Value (USD millions)
Year 1	2,250	85	2,335	2,335	45,275	105.72
Year 2	2,250	85	2,335	2,335	69,813	163.01
Year 3	2,250	85	2,335	2,335	69,813	163.01
Year 4	2,250	85	2,335	2,335	69,813	163.01
Year 5	2,250	85	2,335	2,335	69,813	163.01

Year	Sales Price			Sales Value		
	LME Price (USD/t Pb)	Metals Premia (USD/t Pb)	Sales Price (USD/t Pb)	Sales Price (USD/t Pb)	Sales Amount (t Pb)	Sales Value (USD millions)
Year 6	2,250	85	2,335	2,335	69,813	163.01
Year 7	2,250	85	2,335	2,335	69,813	163.01
Year 8	2,250	85	2,335	2,335	69,813	163.01
Year 9	2,250	85	2,335	2,335	69,813	163.01
Year 10	2,250	85	2,335	2,335	69,813	163.01
Year 11	2,250	85	2,335	2,335	57,488	134.23
Year 12	2,250	85	2,335	2,335	49,681	116.01
Year 13	2,250	85	2,335	2,335	47,672	111.31
Year 14	2,250	85	2,335	2,335	48,169	112.48
Year 15	2,250	85	2,335	2,335	47,494	110.90
Avg / Total	**2,250**	**85**	**2,335**	**2,335**	**924,093**	**2,158**

22.3 Cashflows

Mine cashflows by year of operation are set out in Table 70 and

Table 71.

Table 70: Annual Revenue and Costs (USD M)

Year	Sales Revenue	Royalties	Utilities (ex Hydromet)	Mining	Concentrator	Hydromet Facility	Supply & Logistics	Other Fixed Opex
1	105.72	-3.00	-3.30	-22.32	-19.44	-13.37	-8.23	-11.95
2	163.01	-4.58	-3.80	-27.71	-21.54	-17.19	-11.99	-11.91
3	163.01	-4.58	-3.80	-28.00	-21.54	-17.19	-11.99	-11.91
4	163.01	-4.58	-3.80	-28.00	-21.54	-17.19	-11.99	-11.91
5	163.01	-4.58	-3.80	-22.98	-21.58	-17.21	-11.99	-11.95
6	163.01	-4.58	-3.80	-22.95	-21.54	-17.19	-11.99	-11.91
7	163.01	-4.58	-3.80	-23.02	-21.54	-17.19	-11.99	-11.91
8	163.01	-4.58	-3.80	-20.59	-21.54	-17.19	-11.99	-11.91
9	163.01	-4.58	-3.80	-20.64	-21.58	-17.21	-11.99	-11.95
10	163.01	-4.58	-3.80	-20.62	-21.54	-17.19	-11.99	-11.91
11	134.23	-3.79	-3.80	-15.92	-21.54	-15.26	-10.10	-11.91
12	116.01	-3.29	-3.80	-15.82	-21.54	-14.04	-8.91	-11.91
13	111.31	-3.16	-3.80	-15.82	-21.58	-13.75	-8.60	-11.95
14	112.48	-3.19	-3.80	-16.17	-21.54	-13.81	-8.67	-11.91
15	110.90	-3.15	-3.80	-10.67	-21.54	-13.70	-8.57	-26.02
Total	**2,158**	**-61**	**-56**	**-311**	**-321**	**-239**	**-161**	**-193**

Table 71: Annual Cashflows (USD M)

Year	Sales Revenue	Variable Opex	Fixed Opex	Ongoing Capex	Gross Cashflow	Income Tax	Net Cashflow
1	105.72	-44.53	-36.33	-0.75	24.10	0.00	24.10
2	163.01	-61.73	-36.24	-0.75	64.30	0.00	64.30
3	163.01	-62.02	-36.24	-0.75	64.00	-8.02	55.99
4	163.01	-62.03	-36.24	-0.75	64.00	-15.57	48.43
5	163.01	-57.00	-36.33	-0.75	68.93	-15.57	53.36
6	163.01	-56.97	-36.24	-0.75	69.05	-17.05	52.01
7	163.01	-57.04	-36.24	-0.75	68.99	-17.08	51.91
8	163.01	-54.61	-36.24	-0.75	71.42	-17.06	54.35
9	163.01	-54.66	-36.33	-0.75	71.27	-17.79	53.47
10	163.01	-54.64	-36.24	-0.75	71.38	-17.75	53.63
11	134.23	-45.33	-36.24	-0.75	51.92	-17.78	34.13
12	116.01	-42.31	-36.24	-0.75	36.71	-11.94	24.76
13	111.31	-41.56	-36.33	-0.75	32.66	-7.38	25.28
14	112.48	-42.10	-36.24	-0.75	33.39	-6.17	27.22
15	110.90	-36.35	-36.24	-14.85	23.46	-9.79	13.67
Total	**2,158**	**-773**	**-544**	**-25**	**816**	**-179**	**637**

22.4 Financial Returns

The financial returns based on real US dollar cashflows on an un-geared and after-tax basis, using LME cash price of USD 2,250/t Pb for LOM based on the long-term price forecast in the Wood Mackenzie global lead long-term outlook, are set out in Table 72.

Table 72: Financial Returns (LME lead price USD 2,250/t)

Description	Estimate (USD)	Comments
Total cost to first production	USD 177 M	To start of operations
Payback Period (from start of operations)	4.0 years	From start of operations
Internal Rate of Return	23.5% pa	From start of construction
After-tax cashflow		
- Undiscounted cashflow	USD 637 M	From start of operations
- GPV (8%pa real discount rate) *	USD 357 M	From start of construction
- NPV (8%pa real discount rate) ^	USD 191 M	From start of construction
- GPV Value (8%pa real discount rate) *	USD 401 M	From start of operations
- NPV (8%pa real discount rate) ^	USD 224 M	From start of operations

* GPV = Gross Present Value (ie excluding costs to first production)

^ NPV = Net Present Value (ie including costs to first production)

22.5 Sensitivity Analysis

At the LME spot lead price at February 28, 2018 of USD 2,575/t for life of mine, the financial returns are set out in Table 73.

Table 73: Financial Returns (LME lead price USD 2,575/t)

Description	Estimate (USD)	Comments
Total cost to first production	USD 177 M	to start of operations
Payback Period (from start of operations)	3.0 years	from start of operations
Internal Rate of Return	31.3% pa	from start of construction
After-tax cashflow		
- Undiscounted cashflow	USD 838 M	from start of operations
- GPV (8%pa real discount rate)	USD 470 M	from start of construction
- NPV (8%pa real discount rate)	USD 303 M	from start of construction
- GPV Value (8%pa real discount rate)	USD 527 M	from start of operations
- NPV (8%pa real discount rate)	USD 350 M	from start of operations

Further sensitivities have been applied at + 20% of CAPEX, OPEX, LME Pb Price, discount rate and exchange rates with Figure 73 showing the results. The LME lead price shows the largest single impact to the financial results followed by exchange rates[9] and OPEX respectively. For completeness, a correlated LME lead price and exchanges rates sensitivity was also produced.

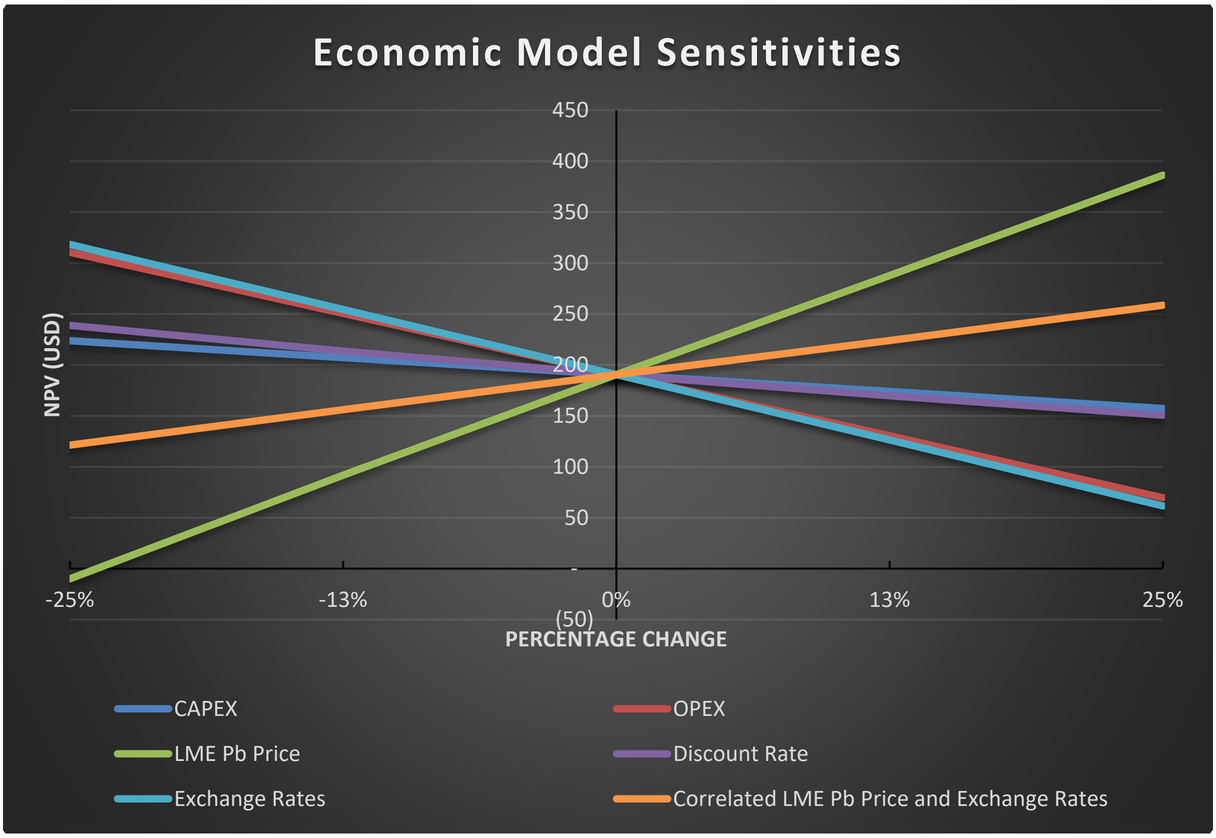


Figure 73: Economic Sensitivities

[9] Equal proportionate depreciation / appreciation of USD to both AUD and EUR for the Exchange rates sensitivity has been used.

1 Annual Statistics

Annual statistics for physicals, financial and revenue allocation are set out in Table 74, Table 75, and Table 76 respectively.

Table 74: Physicals

Description	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Production - Ore Feed									
Plant ore feed-rate	dmt ore	1,456,769	2,000,000	2,000,000	2,000,000	2,000,000	2,000,000	2,000,000	2,000,000
Ore head grade	%Pb	3.90%	4.35%	4.32%	4.29%	4.33%	4.33%	4.36%	4.34%
Contained lead in ore feed	t Pb	56,878	86,903	86,360	85,819	86,614	86,640	87,170	86,831
Production - Flotation									
Flotation recovery	%	81.3%	82.1%	82.6%	83.1%	82.33%	82.3%	81.8%	82.1%
Lead in concentrate	t Pb	46,243	71,307	71,307	71,307	71,307	71,307	71,307	71,307
Concentrate grade	%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%
Concentrate production	dmt con	66,062	101,867	101,867	101,867	101,867	101,867	101,867	101,867
Production - Refinery									
Lead in concentrate	t Pb	46,243	71,307	71,307	71,307	71,307	71,307	71,307	71,307
Refinery recovery	%	97.9%	97.9%	97.9%	97.9%	97.9%	97.9%	97.9%	97.9%
Lead ingot	t Pb	45,275	69,813	69,813	69,813	69,813	69,813	69,813	69,813
Sales Price (real)									
LME Lead Price	USD/t Pb	2,250	2,250	2,250	2,250	2,250	2,250	2,250	2,250
Refined Lead Premia	USD/t Pb	85	85	85	85	85	85	85	85
Ingot Price	USD/t Pb	2,335	2,335	2,335	2,335	2,335	2,335	2,335	2,335

Description	Units	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Years 1-15
Production - Ore Feed									
Plant ore feed-rate	dmt ore	2,000,000	2,000,000	2,000,000	2,000,000	2,000,000	2,000,000	2,000,000	29,456,769
Ore head grade	%Pb	4.35%	4.37%	3.66%	3.20%	3.07%	3.10%	3.07%	3.94%
Contained lead in ore feed	t Pb	87,094	87,333	73,126	63,923	61,335	62,053	61,484	1,159,563
Production - Flotation									
Flotation recovery	%	81.9%	81.6%	80.3%	79.4%	79.4%	79.3%	78.9%	81.4%
Lead in concentrate	t Pb	71,307	71,307	58,718	50,744	48,692	49,200	48,510	943,866
Concentrate grade	%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%
Concentrate production	dmt con	101,867	101,867	83,882	72,492	69,559	70,286	69,300	1,348,380
Production - Refinery									
Lead in concentrate	t Pb	71,307	71,307	58,718	50,744	48,692	49,200	48,510	943,866
Refinery recovery	%	97.9%	97.9%	97.9%	97.9%	97.9%	97.9%	97.9%	97.9%
Lead ingot	t Pb	69,813	69,813	57,488	49,681	47,672	48,169	47,494	924,093
Sales Price (real)									
LME Lead Price	USD/t Pb	2,250	2,250	2,250	2,250	2,250	2,250	2,250	2,250
Refined Lead Premia	USD/t Pb	85	85	85	85	85	85	85	85
Ingot Price	USD/t Pb	2,335	2,335	2,335	2,335	2,335	2,335	2,335	2,335

Table 75: Financials

Description	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Ingot sales									
LME Component	USD	101,867,871	157,078,839	157,078,839	157,078,839	157,078,839	157,078,839	157,078,839	157,078,839
Refined Lead Premia	USD	3,848,342	5,934,089	5,934,089	5,934,089	5,934,089	5,934,089	5,934,089	5,934,089
Total revenue	USD	105,716,213	163,012,929	163,012,929	163,012,929	163,012,929	163,012,929	163,012,929	163,012,929
Operating costs									
Royalties	USD	(3,004,264)	(4,580,622)	(4,580,622)	(4,580,622)	(4,580,882)	(4,580,622)	(4,580,622)	(4,580,622)
Utilities (ex- Hydromet)	USD	(3,296,923)	(3,795,133)	(3,795,133)	(3,795,133)	(3,800,451)	(3,795,133)	(3,795,133)	(3,795,133)
Mining	USD	(22,322,909)	(27,708,629)	(28,002,599)	(28,004,383)	(22,984,078)	(22,952,145)	(23,015,337)	(20,588,607)
Flotation	USD	(19,440,534)	(21,540,372)	(21,540,372)	(21,540,372)	(21,577,828)	(21,540,372)	(21,540,372)	(21,540,372)
Refinery	USD	(13,370,333)	(17,189,476)	(17,189,476)	(17,189,476)	(17,206,668)	(17,189,476)	(17,189,476)	(17,189,476)
Supply & Logistics	USD	(8,234,942)	(11,988,118)	(11,988,118)	(11,988,118)	(11,991,680)	(11,988,118)	(11,988,118)	(11,988,118)
Sustainability	USD	(2,959,093)	(2,951,008)	(2,951,008)	(2,951,008)	(2,959,093)	(2,951,008)	(2,951,008)	(2,951,008)
Operational Support	USD	(6,985,919)	(6,966,831)	(6,966,831)	(6,966,831)	(6,985,919)	(6,966,831)	(6,966,831)	(6,966,831)
Corporate	USD	(1,248,599)	(1,245,188)	(1,245,188)	(1,245,188)	(1,248,599)	(1,245,188)	(1,245,188)	(1,245,188)
Ongoing Capex	USD	(752,055)	(750,000)	(750,000)	(750,000)	(752,055)	(750,000)	(750,000)	(750,000)
Total Operating Costs	USD	(81,615,570)	(98,715,377)	(99,009,347)	(99,011,131)	(94,087,252)	(93,958,893)	(94,022,085)	(91,595,354)
EBITDA	USD	24,100,643	64,297,552	64,003,582	64,001,798	68,925,677	69,054,036	68,990,844	71,417,574
Corporate income tax	USD	-	-	(8,017,989)	(15,568,590)	(15,568,055)	(17,045,219)	(17,083,727)	(17,064,769)
Net operating cashflow	**USD**	24,100,643	64,297,552	55,985,593	48,433,208	53,357,621	52,008,817	51,907,117	54,352,805

* Calculation of corporate income tax includes an estimated carry forward Australian tax loss of AUD 25.0 M and written down value of existing property, plant and equipment for tax purposes of AUD 30.1 M, as a 31 December 2017.

Description	Units	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Years 1-15
Ingot sales									
LME Component	USD	157,078,839	157,078,839	129,347,067	111,782,586	107,260,979	108,380,703	106,861,337	2,079,210,097
Refined Lead Premia	USD	5,934,089	5,934,089	4,886,445	4,222,898	4,052,081	4,094,382	4,036,984	78,547,937
Total revenue	USD	163,012,929	163,012,929	134,233,512	116,005,483	111,313,060	112,475,086	110,898,321	2,157,758,034
Operating costs									
Royalties	USD	(4,580,882)	(4,580,622)	(3,788,707)	(3,287,131)	(3,158,271)	(3,189,986)	(3,146,599)	(60,801,075)
Utilities (ex-Hydromet)	USD	(3,800,451)	(3,795,133)	(3,795,133)	(3,795,133)	(3,800,451)	(3,795,133)	(3,795,133)	(56,444,739)
Mining	USD	(20,642,272)	(20,623,616)	(15,915,314)	(15,815,761)	(15,822,187)	(16,167,491)	(10,670,695)	(311,236,023)
Flotation	USD	(21,577,828)	(21,540,372)	(21,540,372)	(21,540,372)	(21,577,828)	(21,540,372)	(21,540,372)	(321,118,109)
Refinery	USD	(17,206,668)	(17,189,476)	(15,262,534)	(14,042,066)	(13,745,073)	(13,805,686)	(13,700,113)	(238,665,475)
Supply & Logistics	USD	(11,991,680)	(11,988,118)	(10,101,156)	(8,906,010)	(8,601,907)	(8,674,535)	(8,571,152)	(160,989,891)
Sustainability	USD	(2,959,093)	(2,951,008)	(2,951,008)	(2,951,008)	(2,959,093)	(2,951,008)	(2,951,008)	(44,297,455)
Operational Support	USD	(6,985,919)	(6,966,831)	(6,966,831)	(6,966,831)	(6,985,919)	(6,966,831)	(6,966,831)	(104,578,818)
Corporate	USD	(1,248,599)	(1,245,188)	(1,245,188)	(1,245,188)	(1,248,599)	(1,245,188)	(1,245,188)	(18,691,458)
Ongoing Capex	USD	(752,055)	(750,000)	(750,000)	(750,000)	(752,055)	(750,000)	(14,854,316)	(25,362,535)
Total Operating Costs	USD	(91,745,446)	(91,630,364)	(82,316,242)	(79,299,500)	(78,651,383)	(79,086,229)	(87,441,407)	(1,342,185,579)
EBITDA	USD	71,267,482	71,382,565	51,917,270	36,705,983	32,661,677	33,388,856	23,456,914	815,572,455
Corporate income tax	USD	(17,792,788)	(17,747,761)	(17,782,285)	(11,942,697)	(7,379,311)	(6,166,019)	(9,788,763)	(178,947,973)
Net operating cashflow	USD	53,474,694	53,634,805	34,134,985	24,763,286	25,282,366	27,222,838	13,668,151	636,624,482

Table 76: Revenue Allocation

Description	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Revenue Allocation									
Royalties	USD/t Pb	66	66	66	66	66	66	66	66
Utilities (ex-Hydromet)	USD/t Pb	73	54	54	54	54	54	54	54
Mining	USD/t Pb	493	397	401	401	329	329	330	295
Flotation	USD/t Pb	429	309	309	309	309	309	309	309
Refinery	USD/t Pb	295	246	246	246	246	246	246	246
Supply & Logistics	USD/t Pb	182	172	172	172	172	172	172	172
Sustainability	USD/t Pb	65	42	42	42	42	42	42	42
Operational Support	USD/t Pb	154	100	100	100	100	100	100	100
Corporate	USD/t Pb	28	18	18	18	18	18	18	18
Ongoing Capex	USD/t Pb	17	11	11	11	11	11	11	11
Taxes	USD/t Pb	-	-	115	223	223	244	245	244
Equity Return	USD/t Pb	532	921	802	694	764	745	744	779
Total	**USD/t Pb**	**2,335**	**2,335**	**2,335**	**2,335**	**2,335**	**2,335**	**2,335**	**2,335**

Description	Units	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Years 1-15
Revenue Allocation									
Royalties	USD/t Pb	66	66	66	66	66	66	66	66
Utilities (ex- Hydromet)	USD/t Pb	54	54	66	76	80	79	80	61
Mining	USD/t Pb	296	295	277	318	332	336	225	337
Flotation	USD/t Pb	309	309	375	434	453	447	454	347
Refinery	USD/t Pb	246	246	265	283	288	287	288	258
Supply & Logistics	USD/t Pb	172	172	176	179	180	180	180	174
Sustainability	USD/t Pb	42	42	51	59	62	61	62	48
Operational Support	USD/t Pb	100	100	121	140	147	145	147	113
Corporate	USD/t Pb	18	18	22	25	26	26	26	20
Ongoing Capex	USD/t Pb	11	11	13	15	16	16	313	27
Taxes	USD/t Pb	255	254	309	240	155	128	206	194
Equity Return	USD/t Pb	766	768	594	498	530	565	288	689
Total	**USD/t Pb**	**2,335**	**2,335**	**2,335**	**2,335**	**2,335**	**2,335**	**2,335**	**2,335**

23 Adjacent Properties

The Mine deposits are stratabound, occurring in the Proterozoic Earaheedy Group, with mineralisation being largely restricted to the unconformable contact between the Yelma and Maraloou formations. These deposits are unusual due to the near-total absence of sulphides; this has led Pirajno et al. (2010) to conclude that the Mine's deposits are likely to represent a new category within the class of supergene non-sulfide mineral systems. Although there are no known analogues of the deposits, their genesis is considered similar to non-sulfide zinc deposits occurring in areas of deep weathering formed through by reduction of the land surface (Hitzman *et al.,* 2003). Regionally, exploration targets for Magellan-style deposits are focused on dolomitic or other chemically reactive permeable horizons within the Earaheedy and Yerrida basins.

Although no projects have identified mineralisation similar to the Mine's deposits, nearby exploration is targeting base metal and gold mineralisation in the Earaheedy and Yerrida basins, as well as the local Wiluna and Joyner's Find Archaean 'greenstone' belts.

Great Western Exploration Limited (GWE) is currently exploring the Chisel and Frustration Well prospects for copper mineralisation at its Yerrida basin tenement package. The Chisel prospect, located approximately 15km north of the Mine, is defined by a gravity anomaly within favourable stratigraphy in the prospective Maraloou Formation rocks. GWE indicates volcanogenic-massive-sulphide (VMS), sedimentary hosted copper-cobalt, sedimentary lead-zinc or intrusion related base and/or precious metals mineralisation may be present (Great Western Exploration corporate website, 2018).

Blackham Resources continues to mine and explore the Wiluna greenstone belt 30km east of the Mine for Archaean lode-hosted gold deposits. Blackham's Matilda / Wiluna Gold Operation, is the centrepiece of a 1,100 km^2 tenement package with total JORC 2012 Mineral Resources of 65 Mt @ 3.1 g/t for 6.5 Moz of gold. Mining and exploration are focusing on the Wiluna Mine Sequence and 10 kilometres of strike along the Coles Find Shear, with exploration prospects at Lake Way, Carroll, Prior, Mentelle, and Monarch (Blackham Resources corporate website, 2018).

Golden West Resources (GWR) Wiluna West direct-shipping-ore iron and gold projects are located in the Joyner's Find greenstone belt 30 km south of the Mine. The Wiluna West iron project has a JORC 2004 Code compliant Resource totalling 130.3 Mt at an average iron grade of 60% Fe, including 69.2M t of Probable Reserves at 60.3% Fe. The project is currently in care and maintenance. GWR's Golden Monarch gold prospect contains a combined JORC 2004 and JORC 2012 Mineral Resource estimate of 3.5 Mt at 2.3 g/t Au for 254,000 oz Au. Portions of the resource will be mined and treated at the nearby Wiluna Gold Operation plant under a 2017 agreement with Blackham Resources. GWR is also following up comprehensive mapping and geochemical soil sampling programs at the Bowerbird, Eagle, Emu and Comedy King gold prospects with RC drilling. (Golden Western Resources corporate website, 2018).

24 Other Relevant Data & Information

The following information provides a summary of the planned activities in the lead up to the operation of both the concentrator plant and the Hydrometallurgical Facility at the Mine.

24.1 Regulatory Approvals

Currently the Referral Document is before the State EPA, under Part IV of the EP Act, to increase the disturbance footprint by 400 ha, taking the total disturbance footprint to 980 ha, within the development envelope. The Referral Document also includes for an increase of 19 Mt tailings storage capacity, taking the total storage capacity to 35 Mt, to meet the needs of the new forecast mine life. The document also describes the Hydrometallurgical Facility and the proposed new electricity generation plant at site.

On April 4, 2018 the EPA determined their Level of Assessment of the Referral Document. The level set was “Referral Information” (the information contained within the Referral Document being sufficient for their purposes), with a request for some additional information to be supplied by RHM. The EPA Report and Recommendations is likely to be provided mid-2018.

The construction and operation of the Hydrometallurgical Facility is currently the subject of a Works Approval with the DWER. The works approval will permit ground disturbing activities in relation to the construction of the Hydrometallurgical Facility. A prescribed premises licence amendment is expected to follow the commissioning of the Hydrometallurgical Facility, authorising the operation.

Updates to the DMIRS, approved (2015), Mining Proposal will include the Hydrometallurgical Facility and expanded footprint areas. This will be progressed during 2018, prior to the start of construction of the Hydrometallurgical Facility. An updated MCP will also be developed to meet DMIRS requirements.

24.2 Funding

Lead*FX* is securing funding for the planned pre financing activities. The funding will cover proposed early works, including end to end pilot plant for optimisation purposes, key owner’s team personnel and FEED.

Concurrently Lead*FX* is also securing overall funding to construct the Hydrometallurgical Facility and restart the operations.

24.3 Early Works

The planned early works prior to award of the EPCM for the Hydrometallurgical Facility include;

Operation of a pilot plant to incorporate the identified changes nominated by the DFS pilot plant and batch testwork operation. While these nominated changes were tested during the DFS pilot plant, they were not necessarily integrated into a full end to end process. The pilot plant will allow for comprehensive confirmation of the DFS design parameters in particular electrowinning performance.

FEED works will be undertaken to progress the DFS design information such that resolved engineering details can be incorporated into identified long lead items tender packages.

Appropriately qualified and industry recognised personnel are being sought to join the owners team to manage the early works and then construction.

24.4 Construction

The proposed EPCM contract for the Hydrometallurgical Facility will include the onsite construction of the facility and the required modifications to the flotation plant. The Mine has an existing suite of utilities to support the construction activities, including accommodation, power and water supply. The Hydrometallurgical Facility will be located in an area on site which is not encumbered by any existing infrastructure, yet is adjacent to the existing concentrator plant and power station.

24.5 Commissioning

The commissioning of the Hydrometallurgical Facility is planned to be undertaken over an extended period. Forecast commissioning to steady state is approximately 12 months.

24.6 Concentrator Start-up

The concentrator plant start-up will be undertaken based on the successful 2013 plant start-up which was both successful and well documented. The start-up to steady state (24hr operation), involved a core team of operators who transferred knowledge to an ever-increasing workforce until the plant was in full operation on a 24hr basis. The previous ramp up period was 4 months.

24.7 Ramp Up

The ramp up of the Hydrometallurgical Facility during the later commissioning phase will align with the concentrator plant ramp up period. The existing concentrate storage shed has sufficient buffering capacity for storage of concentrate before entering the Hydrometallurgical Facility of approximately 1 month's production at steady state.

24.8 Steady State Operation

Steady state operation will be achieved when the concentrator plant is achieving 102,000 tpa concentrate at 70% Pb. With the planned modifications, steady state is forecast 5 months from start-up. The Hydrometallurgical Facility steady state will be when the facility is producing the equivalent of 70,000 tpa.

25 Interpretation & Conclusions

25.1 Exploration

Most exploration work conducted by and on behalf of RHM within the Mine area has been drilling, for purposes of exploration, resource definition and sterilisation. However, all non-drilling forms of exploration have contributed directly to the targeting of additional mineralisation, either as extensions to known deposits, or to discovery of new deposits.

25.1.1 Geochemical Surveys

Geochemical surveys, including the conventional, portable XRF and combined datasets, have greatly assisted in generating new drill targets. In addition, the surveys have assisted in assessing the distribution of naturally-occurring lead in the environment, contributing to mine closure planning and environmental documentation.

A strong, well-defined lead-in-soil anomaly is associated with the southwestern slope of Magellan Hill, with the anomaly extending along strike both to the north-west and south east of the known deposits (Sergeev, 2008). Due to the high density, previously unrecognised, isolated, lead-in-soil anomalies to the north east and east of Magellan Hill have been identified. The Gama deposit and the north eastern portion of the Magellan deposit are essentially blind, with no clearly associated lead anomaly.

The southern breakaway margins of the Cano, Magellan and Pinzon deposits show a well-developed (natural) secondary dispersion lead geochemical anomaly. These correspond closely to observed vegetative anomalies. The magnitude of the lead anomaly is greatest where mineralisation approaches or intersects the surface. The dispersion anomaly is weaker and more confined towards the north where the breakaways are poorly developed.

Minor, surficial lead in soil anomalism can be found fixed in patches of calcrete formation south of the Mine village and along the southern West Creek drainage south of the Magellan mesa (Burlow and Corry, 2014).

The satellite lead deposits at Pizarro and Drake show similar, though less well developed, dispersion anomalies.

25.1.2 Gravity Surveys

Apparent gravity lows associated with the Magellan and Cano deposits are less well defined than previously believed, and the lack of associated gravity lows with the other known deposits (e.g. Drake, Pizarro, and Pinzon) implies that the deposits cannot be directly detected from gravity data. However, the high-resolution gravity data does enable the identification of many structural features, some of which are related to the mineralisation. Gravity surveys have generated new drilling targets around Drake and Pizarro, and several gravity targets were drilled at the Drake prospect in late 2013, with encouraging results.

25.1.3 Aerial Photography / Photogrammetry

Aerial photography and DTM generation have aided exploration through mapping of local geological contacts and providing maps for exploration program safety Maps for native title / DMIRS permit requirements have also been generated. The aerial photography has also been used in land use studies as part of the mine closure planning documentation and environmental compliance.

25.1.4 Drilling

The Magellan Hill lead deposits have been explored and delineated by a series of drilling campaigns dating back to the early 1990s. Typical drill patterns have varied from 50 x 50 m to a staggered 50 x 100 m.

Grade control drilling at Magellan and Cano has infilled the exploration drilling data to a 12.5 x 12.5 m and 16.7 x 16.7 m patterns since the commencement of mining in 2005.

All drilling prior to the 2015 drilling campaign have been fully disclosed in the previous Technical Report (SRK, 2015).

In 2015, two drilling programs were completed and another in 2017. The two programs in 2015 were on tenements not included in the current mine plan.

During June and July 2017, a large-diameter (PQ3) diamond drilling program was conducted at the Magellan and Pinzon lead deposits. The diamond drill sites were planned to twin existing RC holes containing known mineralisation across the projected life of mining plan with the aim of collecting annual feed composite samples for variability and metallurgical testing as part of the DFS.

25.1.5 Sampling

All sample preparation and analyses for the recent RC drilling programs conducted in 2015 – 2018 (discussed in Section 10) have been carried out at Genalysis (Genalysis, RC samples only) in Maddington, Western Australia, and at ALS in Balcatta, Western Australia (ALS, diamond core and bulk samples). These laboratories have been certified in accordance with ISO/IEC 17025:

- Genalysis date of accreditation: September 20, 1991 – Accreditation No: 3244
- ALS date of accreditation: December 22, 2015 – Accreditation No: 825

All sample preparation and analyses for the DFS diamond core and bulk sample test work were carried out at ALS Laboratories

No aspect of sample preparation at Genalysis or ALS was conducted by an employee, officer, director or associate of RHM or Lead*FX*.

25.1.6 Data Verification

The RHM database has inbuilt constraints and triggers, ensuring that the data is validated and constrained. Importing of incoming data is handled by RHM geologists according to documented procedures.

The data at the Mine is adequate for use and the methods employed are considered industry standard and are reasonable for the drilling and sample methods employed and the status of the deposit as an operating mine.

25.2 Mineral and Resource Estimate

The Mineral Resource estimate was last reported as at December 31, 2016 by Optiro and there have been no changes since. The Mineral Resource estimate includes the main Magellan Hill deposits of Magellan (now including Gama), Cano and Pinzon and the outlying Pizarro and Drake satellite deposits located approximately 10 km south and 11 km south-west of the existing mine infrastructure respectively.

The Mineral Resource estimate for the Paroo Station deposits are reported under the JORC Code 2012. CIM recognises “use of foreign code” including the JORC Code 2012. The 2005 Drake Mineral Resource estimate (originally produced under JORC 2004) and later updates were reviewed by Optiro in 2016 with the required ‘Table 1’ prepared for public reporting under JORC 2012.

Stockpiles have been tabulated from actual mine production data.

The 2017 Mineral Resource estimate includes all depletion due to mining and processing activities before Paroo Station was put onto care-and-maintenance during January 2015 due to low commodity prices.

No new drilling or other exploration work has been added to the Mineral Resource estimate, which, depletion aside, remains unchanged from the 2014 estimate presented in SRK, 2015.

Table 45 depicts the Mineral Resource tabulation, at a 2.1% lead cut-off and rounded to reflect the precision of the estimate. The estimate has been prepared by Optiro and is as per the Technical Report dated March 10, 2015 prepared by SRK, net of mining depletion (totalling 5kt of contained lead) in January and February 2015 and reported in the Company's Annual Information Form dated March 28, 2018. The Mineral Resource inventory is 32.0 Mt at 4.3% lead for 1,390 kt of contained lead metal classified as measured and indicated and a further 8.4 Mt at 4.0% lead for 340 kt of contained lead classified as inferred.

25.3 Mineral Reserve Estimate

The Mineral Reserve estimate is presented in Table 48 and consists of 31.2 Mt at 3.8% lead for 1,199 kt of contained lead metal.

The Mine has been in commercial operation over several phases of operation before being shut down in January 2015 due to low commodity prices. As a result, the QP has relied on both historic and more recent production information and financial inputs to support the mine planning and confirm that economic extraction of the resource is feasible.

The Mine plan was revised to support the Mineral Reserve estimate with the updated open pit optimisation incorporating accepted product pricing and current costs and operational parameters. The open pit optimisation underpinned revised mine staging, mine designs and mine production scheduling.

The Mineral Reserve estimate was classified and reported under the 2012 Edition of the JORC Code. CIM recognises "use of foreign codes" including the JORC Code.

25.4 Mining

25.4.1 Geotechnical and Hydrogeological

An overall slope angle of 40º has been applied to the optimization process. All final pit designs produced have incorporated the recommended geotechnical pit slope design parameters from geotechnical interpretations undertaken and presented in Review of Wall Design Parameters Paroo Station, Peter O'Bryan and Associates, January 2015:

- Bench face height 10 m – from surface to 30 m depth.
- Bench face height 15 m – below 30 m depth from surface.
- Face angle 60° throughout.
- Minimum berm width of 5 m at 10 m and 20 m depth intervals.
- Minimum berm width of 6 m at 30 m and 45 m depth intervals.

The existing pit wall designs are based on 10 m-high, 50° face angle batters separated by 5 m-wide berms.

The as-mined pits do not currently intersect the water table; however, the water table will be partially intersected when pits are mined to the final design at the conclusion of the expected mine life. A hydrological review is required to confirm there is no likely adverse impact on the stability of the pit walls.

25.4.2 Pit Design Criteria

The following design parameters were used in all final pits:

- Dual lane ramps of 25 m wide at 10% gradient.
- Batter angle 60°.
- 10 m bench height from surface to 30 m depth.
- 15 m bench height below 30 m depth.
- 5 m bench width at 10 m and 20 m depths
- 6 m bench width at 30 m and 45 m depths.
- Minimum mining width approximately 40 m.

A final pit was designed and divided into nine stages to assist with achieving schedule targets. The stages have their own ramp access while following the minimum mining width so they can be mined independently.

25.4.3 Production Schedule

Mine production schedules were developed using Minemax software. Several iterations were conducted to balance mill targets and minimise stockpiles while keeping a steady production profile.

This schedule was developed based on:

- Diluted model with Measured and Indicated Mineral Resource categories only.
- Annual schedule where start date is irrelevant.
- Mill capacity of 2.0 Mtpa is constant from the second year of production after an initial ramp up from 1.4 Mt for the first year.
- Achieving 70 Ktpa of lead ingot production as consistently as possible.
- Maximum total material movement limited to 8 Mtpa in the first year.
- 5 m benches.
- Maximum vertical advance rate of ten benches per year.
- 40 m minimum cutback distance.
- Minimise stockpiles from in-pit under 1.5 Mt.

25.4.4 Waste and Stockpile

Preliminary waste dumps were designed to ensure sufficient ex-pit dumping capacity. The design parameters and assumptions are:

- Batter or face angle of 18°
- 5 m berm every 10 m lifts
- Maximum total height of 50 m
- Minimum of 50 m away from the pit boundary.

25.5 Metallurgy and Processing

The Hydrometallurgical Facility flowsheet was initially modelled in Metsim by SNC-Lavalin based on the proof of concept testwork carried out in part during the scoping study. The base case model was further developed based on additional batch testwork carried out during the early stages of the DFS.

The flowsheet has evolved as further testwork data became available to define the process parameters required for the flowsheet. These data have been incorporated into an optimised base case model which reflects the basis of the flowsheet design.

During the DFS, a series of Metsim models have been developed as follows:

- A Base Case model incorporating all of the flowsheet elements required to operate the Hydrometallurgical Facility. The base case grade of 70% lead was selected following investigation of concentrate treatment at grades of 55% and 60% Pb. The concentrate feed input data to the model is based on average life of mine data derived from the variability testwork program.
- Individual models were run using the base case model with different input data based on testwork results generated by the variability testwork program for a high and low anglesite change to the base case feed mineralogy.

Outputs from these models were used to validate the process design to ensure that the range of operating conditions under which the Hydrometallurgical Facility would be required to function were incorporated into the process design.

A drilling program was undertaken by RHM to provide representative samples of each year of production for the proposed new mine life, based on a revised mine cut-off grade calculated by RHM. These samples were then treated according to the operating practice of the existing flotation concentrator to produce a range of concentrate samples to be evaluated according to the revised Hydrometallurgical Facility flowsheet. The concentrate grades produced for this testwork program were compared to the typical concentrates previously produced for shipment to a smelter to increase overall lead recovery given that the concentrates could now be treated on site to produce lead ingot.

An analysis of the flotation results indicated that the concentrate grade that minimised slimes recovery to the flotation concentrate was of the order of 70% lead, up from the 67%-68% lead grade targeted for sale to a smelter. With the revised flotation regime flotation recovery was increased relative to historical concentrator performance.

25.6 Environmental

Currently an application is before the EPA, under Part IV of the EP Act, to increase the disturbance footprint by 400ha, taking the total disturbance footprint to 980 ha, within the development envelope. Based on flora and fauna studies undertaken in development envelope, no significant impacts are anticipated.

The application also includes for an increase of 19 Mt tailings storage capacity, taking the total storage capacity to 35 Mt, to meet the needs of the new forecast mine life. The application also describes the Hydrometallurgical facility and the proposed new electricity generation plant at site.

The construction and operation of the Hydrometallurgical Facility is currently the subject of a Works Approval with DWER. A prescribed premises licence amendment is expected to follow the commissioning of the Hydrometallurgical Facility.

Updates to the currently DMIRS approved Mining Proposal, to include the Hydrometallurgical Facility and expanded footprint areas will be progressed in 2018. An updated MCP will also be developed to meet DMIRS requirements.

Closure performance monitoring is undertaken throughout the rehabilitated land surfaces. This monitoring work describes and measures the success of the progressive natural revegetation.

25.7 Projected Economic Outcomes

An economic model was developed for the Mine to support the Mineral Reserve estimation. The model incorporates the Mine production schedule and key assumptions, and outlines an operating Mine life of 15 years with positive financial outcomes.

Sensitivities were analysed using economic model at ± 25% to determine which changes have the highest impact on the economic outcomes. These included the following:

- Commodity price
- Operating costs
- Capital costs
- Exchange rate
- Discount rate.

Figure 74 presents the results from the sensitivity analysis. The LME Lead Price has the largest impact on the Mine's financial results followed by the exchange rates and then the operating costs.

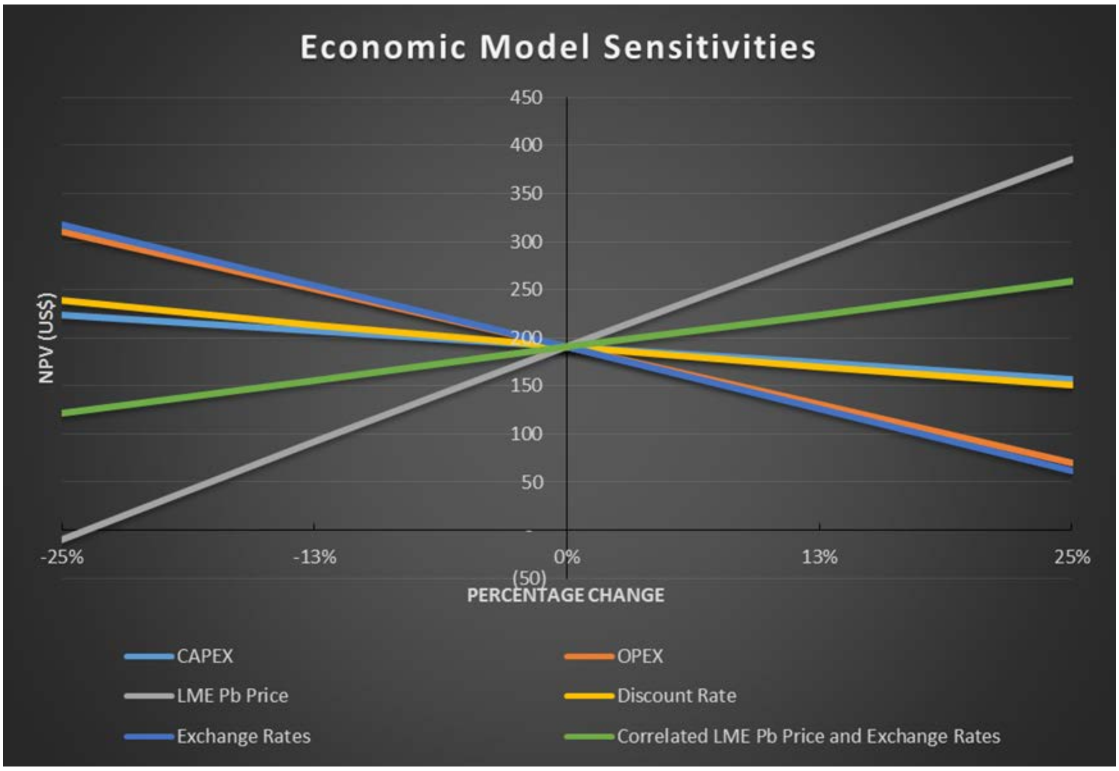


Figure 74: Economic Model Sensitivities

SRK reviewed the actual and projected product sales and operating cost data for the existing production facility. Based on this review and the above-defined variables, SRK concluded that the Mine has a positive NPV; therefore, the Mineral Reserve statement in Section 15 is valid.

25.8 Foreseeable Impacts of Risks

The Mine was shut down in early 2015 due to the low lead spot prices and was subject to very strict compliance conditions, remaining sensitive to both public and political oversight through the production and transport of lead carbonate concentrate for export.

Construction and operation of the Hydrometallurgical Facility on site to produce lead metal, eliminates lead concentrate transportation which in turn removes previous compliance and stakeholder risks to the business. Additionally, production of LME grade lead metal on site, eliminates the cost exposure of third parties off shore, to process the concentrate.

The Mine plan has been updated and revised to reflect the operation of the Hydrometallurgical Facility which presents a more robust Mine that demonstrates profitability at both the current spot lead prices and medium-term price forecast of USD 2,250/t.

26 Recommendations

The DFS and associated economic analysis has demonstrated that the construction and operation of a Hydrometallurgical Facility on site to produce lead metal in ingot form is economically feasible and attractive. It is recommended that the following activities are initiated by RHM.

- Secure the environmental approvals for the construction and operation of the Hydrometallurgical Facility.
- Undertake a closed cycle pilot plant testwork to optimise to the concentrator and Hydrometallurgical Facility flow sheets.
- Undertake FEED work and appoint the Hydrometallurgical Facility construction engineer.
- Arrange financing for the commencement of the construction of the Hydrometallurgical Facility.
- Undertake additional work to convert Inferred Mineral Resources to Measured and Indicated to potentially increase the Mine life.

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28 Glossary

Abbreviation	Definition
4WD	4 Wheel Drive
ALS	Australian Laboratory Services
AUD	Australian dollar
ARSM	Associate of the Royal School of Mines
AusIMM	The Australasian Institute of Mining and Metallurgy
DOM	Bureau of Meteorology
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CO_2	Carbon Dioxide
CPI	Consumer price index
CSA	CSA Global Pty Ltd
CV	coefficient of variation
DeS	Desulphurisation leach
DFS	Definitive Feasibility Study
DWER	Department of Water and Environmental Regulation
DMIRS	Department of Mines, Industry Regulation and Safety
dmt	dry metric tonnes
DoH	Department of Health
DTM	digital terrain model
ENE	East North East
EP (Act)	Environmental Protection (Act)
EPA	Environmental Protection Authority
Fe	Iron
FIFO	Fly In, Fly Out
Ga	Giga annum (billion years)
Genalysis	Intertek Genalysis Laboratories Pty Ltd
GIS	Global information system
Gl	Gigalitre
GPS	Global positioning system
GPX	GPX Surveys Pty Ltd
GSWA	Geological Survey of Western Australia
ha	hectare
hr	hour
Iluka	Iluka Resources Limited
InCoR	InCoR Energy Metal Limited
IWL	Integrated Waste Landform
JORC	Joint Ore Reserves Committee
kL	kilolitre
km	kilometre
KNA	Kriging Neighbourhood Analysis

Abbreviation	Definition
kt	kilotonne(s)
kWh/t	Kilowatt hours per tonne
L	Litre
Lead*FX*	Lead*FX* Inc.
LME	London Metals Exchange
LOM	life of mine
m	metre(s)
M	million
Magellan Hill	Magellan (including Gama), Cano and Pinzon deposits
Magellan Metals	Magellan Metals Pty Ltd
mAHD	Metres Australian height datum
mg/L	milligrams per Litre
mE	Metres east
mm	millimeters
mN	Metres north
MRE	Mineral Resource Estimate
MS	Microsoft
MSA	Methane Sulphonic Acid
Mt	million tonnes
Mtpa	million tonnes per annum
MVT	Mississippi Valley type
MW	Megawatt
NaHS	sodium hydrosulfide
NE	North-East
NPV	net present value
NW	North-West
OEPA	Office of the Environmental Protection Authority
ORP	oxidation reduction potential
Paroo Station	Paroo Station Mine
Pb	Lead
PbS	Galena (Lead Sulphide)
Polymetals	Polymetals Pty Ltd
QA/QC	quality assurance / quality control
QP	Qualified Person
RAB	Rotary Air Blast
RC	reverse circulation
Renison	Renison Goldfields Consolidated
RHM	Rosslyn Hill Mining Pty Ltd
RO	reverse osmosis
ROM	run of mine
RQD	Rock-Quality Designation

Abbreviation	Definition
SAB	Semi Autogenous Mill / Ball Mill
SABC	Semi Autogenous Mill / Ball Mill and Pebble Crusher
SAG	semi-autogenous grinding
Sentient	The Sentient Group
SE	South-East
SIBX	sodium isobutyl xanthate
SRK	SRK Consulting (Australasia) Pty Ltd
t	tonne(s)
TDEM	Time-domain electromagnetic
t/h	tonnes per hour
the Mine deposits	Magellan (now including Gama), Cano and Pinzon and the outlying Pizarro and Drake satellite deposits
TMM	Total Material Movement
tpa	tonnes per annum
TSF	tailings storage facility
USD	United States dollar
VMS	Volcanogenic-massive-sulphide
XRF	X-ray fluorescence
XTEM	Geophysical Survey System Transient Electromagnetic
Yc	Yelma Formation clay-quartz breccia
Yq	Yelma Formation sandstone
Ys	Yelma Foundation siltstone
Yy	Yelma Foundation clay

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Certificate of Qualified Person

To accompany the report entitled, *Technical Report on the Paroo Station Lead Carbonate Mine, Wiluna, Western Australia*, with an effective date of February 28, 2018, prepared for LeadFX Inc. (the Technical Report).

a) I, Scott McEwing, am a Principal Mining Engineer with SRK Consulting Australasia Pty Ltd, with a business address at Level 1, 10 Richardson Street, West Perth, WA 6005, Australia.

b) I am a graduate of University of Auckland, Bachelor of Engineering (Mining) in 1996. I have been practicing in my profession since 1996.

c) I am a Fellow and Chartered Professional (Mining) of the Australasian Institute of Mining and Metallurgy. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).

d) My most recent personal inspection of the Property was on 11 and 12 November 2014.

e) I am responsible for Sections 1 to 3, 16, 18 to 19, and 21 to 28, of the Technical Report.

f) I am independent of Ivernia Inc as defined by Section 1.5 of the Instrument.

g) I have previously been involved with the property that is the subject of the Technical Report; I have been a QP responsible for Sections 1 to 3, 16, 18 to 19, and 21 to 28 and the preparation of the report titled *NI 43-101 Technical Report on the Paroo Station Lead Carbonate Mine, Wiluna, Western Australia, Effective Date December 31, 2014.*

h) I have read the Instrument and the Technical Report has been prepared in compliance with the Instrument.

i) At the effective date of the technical report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

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

Signed at Perth, Western Australia, on April 12, 2018.

Scott McEwing, BEng(Mining), FAusIMM CP(Min)
Principal Consultant

CHARTERED PROFESSIONAL
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CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: Technical Report on the Paroo Station Lead Mine, Wiluna, Western Australia, with an effective date of February 28, 2018, prepared for LeadFX Inc. (the Technical Report)

I, Alan Taylor, of ALTA Metallurgical Services, 1/6 Langtree Court, Blackburn, Victoria 3130, Australia, do hereby certify that:

1. I am Managing Director of the firm of ALTA Metallurgical Services PR: JO-AL Enterprises Pty Ltd of 1/6 Langtree Court, Blackburn, Victoria 3130, Australia;

2. I am a graduate of the Durham University, UK with a Bachelor of Science in Chemical Engineering (Honours) in 1960. I have practised my profession since 1960 with a break from 1962-1964 while serving as an officer in the Royal Air Force of the UK;

3. I am a Fellow and Chartered Professional of the Australasian Institute of Mining and Metallurgy (FAusIMM (CP));

4. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101;

5. I have not inspected the property;

6. I have no prior involvement with the property that is the subject of the Technical Report;

7. I am independent of the issuer as defined in Section 1.5 of NI 43-101;

8. I do not have any securities in LeadFX Inc. or its subsidiaries;

9. I am responsible for items 13 and 17 of this Technical Report with respect to the Hydrometallurgical Facility components of those items;

10. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance with NI 43-1-1 and Form 43-101F1;

11. As of the effective date of the Technical Report, 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; and

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

Signed at Blackburn on April 12, 2018

Alan Taylor

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CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: Technical Report on the Paroo Station Lead Mine, Wiluna, Western Australia, with an effective date of February 28, 2018, prepared for LeadFX Inc. (the Technical Report)

I, Lawrie Gillett of AMC Consultants Pty Ltd, Level 1, 1100 Hay Street, West Perth, WA 6005, do hereby certify that:

1. I am an employee with the firm of AMC Consultants Pty Ltd of Level 1, 1100 Hay Street, West Perth, WA 6005;
2. I am a graduate of the University of Melbourne with a BE Eng (Mining) in 1975. I have practised my profession continuously since 1975;
3. I am a FAusIMM (CP);
4. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101;
5. I have not inspected the property;
6. I have no prior involvement with the property that is the subject of the Technical Report;
7. I am independent of the issuer as defined in Section 1.5 of NI 43-101;
8. I do not have any securities in LeadFX Inc. or its subsidiaries;
9. I am responsible for the preparation of items 15 of this Technical Report;
10. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance with NI 43-1-1 and Form 43-101F1;
11. As of the effective date of the Technical Report, 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; and
12. I consent to the filing of the Technical Report with and Stock Exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed at Perth on April 12, 2018

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CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: Technical Report on the Paroo Station Lead Mine, Wiluna, Western Australia, with an effective date of February 28, 2018, prepared for LeadFX Inc. (the Technical Report)

I, Kahan Mit-hat Cervoj of Optiro Pty Ltd, Level 1, 16 Ord Street, West Perth, WA 6872, do hereby certify that:

1. I am an employee with the firm of Optiro Pty Ltd of Level 1, 16 Ord Street, West Perth, WA 6872;

2. I am a graduate of the Curtin University of Technology with a Bachelor of Applied Science (Geology), graduating in 1991. I have practised my profession continuously since December 1990, with experience in base metal exploration, resource development, mining geology and resource estimation;

3. I am a Member of the Australian Institute of Geoscientists (AIG membership number 6302);

4. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101;

5. I have inspected the property between the July 23rd and July 25th, 2014;

6. I have previously been involved with the property that is the subject of the Technical Report; I completed the Mineral Resource estimate for the Magellan Hill and Pizarro deposits in 2015, and I was the JORC Competent Person (CP) for the Paroo Station Lead Mine at December 31, 2015 and 2016, having prepared the Mineral Resource statement for the respective periods;

7. I am independent of the issuer as defined in Section 1.5 of NI 43-101;

8. I do not have any securities in LeadFX Inc. or its subsidiaries;

9. I am responsible for the preparation of items 7 to 12 and 14 of this Technical Report;

10. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance with NI 43-1-1 and Form 43-101F1;

11. As of the effective date of the Technical Report, 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; and

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

Signed at Perth on April 12, 2018

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Consent of Qualified Person

Ontario Securities Commission
British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Financial Services Commission
Manitoba Securities Commission
Autorité des marchés financiers du Québec Nova Scotia Securities Commission
New Brunswick Securities Commission
Prince Edward Island Securities Office
Securities Commission of Newfoundland & Labrador

Dear Sirs

Re: LeadFX Inc. (the Company)
Consent Letter for Use of Technical Report

I, Scott McEwing, consent to the public filing of the technical report titled, *Technical Report on the Paroo Station Lead Mine, Wiluna, Western Australia*, with an effective date of February 28, 2018 (the Technical Report) by LeadFX Inc.

I further consent to the use of my name and to the Company making reference to and summarising or taking extracts from the part(s) of the Technical Report that I am responsible for in any document that may be required to be filed or otherwise disclosed by the Company pursuant to applicable securities laws or stock exchange policies pertaining to continuous and timely disclosure.

Dated at Perth, Western Australia, on this 12th day of April 2018.

Scott McEwing

ALTA Metallurgical Services PR: JO-AL Enterprises Pty Ltd
ABN 19 883 791 267
PO Box 1211, Blackburn North, Victoria 3130, Australia
T: +61 (0)418 126 284 alantaylor@altamet.com.au
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CONSENT OF QUALIFIED PERSON

Ontario Securities Commission
British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Financial Services Commission
Manitoba Securities Commission
Autorité des marchés financiers du Québec Nova Scotia Securities Commission
New Brunswick Securities Commission
Prince Edward Island Securities Office
Securities Commission of Newfoundland & Labrador

Dear Sirs;

Re: LeadFX Inc, (the "Company")
Consent Letter for Use of Technical Report

I, Alan Taylor consent to the public filing of the technical report titled, Technical Report on the Paroo Station Lead Mine, Wiluna, Western Australia, with an effective date of February 28, 2018 (the "Technical Report") by LeadFX Inc.

I further consent to the use of my name and to the Company making reference to and summarising or taking extracts from the part(s) of the Technical Report that I am responsible for in any document that may be required to be filed or otherwise disclosed by the Company pursuant to applicable securities laws or stock exchange policies pertaining to continuous and timely disclosure.

Dated this April 12, 2018

Alan Taylor

Alan Taylor

AMC Consultants Pty Ltd
ABN 58 008 129 164

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West Perth WA 6005
Australia

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AMC consultants mine smarter

CONSENT OF QUALIFIED PERSON

Ontario Securities Commission
British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Financial Services Commission
Manitoba Securities Commission
Autorité des marchés financiers du Québec Nova Scotia Securities Commission
New Brunswick Securities Commission
Prince Edward Island Securities Office
Securities Commission of Newfoundland & Labrador

Dear Sirs;

Re: LeadFX Inc, (the "Company")
Consent Letter for Use of Technical Report

I, Lawrie Gillett consent to the public filing of the technical report titled, Technical Report on the Paroo Station Lead Mine, Wiluna, Western Australia, with an effective date of February 28, 2018 (the "Technical Report") by LeadFX Inc.

I further consent to the use of my name and to the Company making reference to and summarising or taking extracts from the part(s) of the Technical Report that I am responsible for in any document that may be required to be filed or otherwise disclosed by the Company pursuant to applicable securities laws or stock exchange policies pertaining to continuous and timely disclosure.

Dated this April 12, 2018

LJ Gillett.

Lawrie Gillett

Level 1, 16 Ord Street
West Perth WA 6005
PO Box 1646
West Perth WA 6872
Australia
T: +61 8 9215 0000
F: + 61 8 9215 0011

CONSENT OF QUALIFIED PERSON

Ontario Securities Commission
British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Financial Services Commission
Manitoba Securities Commission
Autorité des marchés financiers du Québec Nova Scotia Securities Commission
New Brunswick Securities Commission
Prince Edward Island Securities Office
Securities Commission of Newfoundland & Labrador

Dear Sirs;

Re: LeadFX Inc, (the "Company")
Consent Letter for Use of Technical Report

I, Kahan Mit-hat Cervoj consent to the public filing of the technical report titled, Technical Report on the Paroo Station Lead Mine, Wiluna, Western Australia, with an effective date of February 28, 2018 (the "Technical Report") by LeadFX Inc.

I further consent to the use of my name and to the Company making reference to and summarising or taking extracts from the part(s) of the Technical Report that I am responsible for in any document that may be required to be filed or otherwise disclosed by the Company pursuant to applicable securities laws or stock exchange policies pertaining to continuous and timely disclosure.

Dated this April 12, 2018

K. Cervoj

Kahan Mit-hat Cervoj