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

Effective Date: February 15, 2019

Report Date: April 5, 2019

Report Prepared for

Lead*FX*

Report Prepared by

SRK Consulting (Australasia) Pty Ltd

LFX002

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1 Summary

The purpose of this report is to provide a technical summary of the mineral assets and summary of the proposed changes to the processing of ore to produce lead metal through the construction and operation of a Hydrometallurgical Facility at the Paroo Station Lead Mine ("Paroo Station Mine", "Paroo Station", the "Mine" or the "Project"), near Wiluna, Western Australia, pursuant to NI 43-101 and other rules of the Canadian Securities Administrators.

This report is an update of the earlier report dated April 12, 2018 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 finalization of this update.

1.1 Property Description and Ownership

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

The Mine is located 30 km west of Wiluna, and 2 km directly north of the Wiluna–Meekatharra road in the north-eastern goldfields region of Western Australia and 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 licenses (the Site). The leases and licenses cover in excess of 30,000 ha, including 2,447 ha of Mining Leases.

The Mine is 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, Pinzon (collectively referred to as Magellan Hill), Pizarro and Drake deposits. Initial discovery of the Magellan deposit was in 1993, followed by Cano in 2001 and Pinzon in 2004. Two outlying deposits, Pizarro and Drake were discovered approximately 8–11 km to the south and southwest of Magellan Hill. The Pizarro deposit is included within the current Mine plan, while Drake is not.

The Mine and existing processing plant produces lead carbonate concentrate from deposits of the mineral cerussite (lead carbonate), with subordinate anglesite (lead sulfate) 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. This technical report is based on an updated Definitive Feasibility Study ("DFS Update") of February 2019 which was a progression on the Definitive Feasibility Study ("DFS") of February 2018.

The DFS Update, integrates proposed changes and additions to the existing Flotation Concentrator and infrastructure on site with the proposed new Hydrometallurgical Facility and provides capital and operating cost estimates for the Project, summarizes proposed mining operations, metallurgical testwork and engineering development and a construction execution plan.

The DFS Update is based upon the new Hydrometallurgical Facility having a nominal design capability of 70,000tpa of lead ingots with the capacity of production up to 80,000 tpa lead ingots.

The mining operation concept remains as per previous operating periods of the Mine. The Mineral Reserve estimate of 36.3Mt at 3.7% Pb, containing 1,344.6 kt of contained Pb metal supports a 17-year life of mine (LOM) and was prepared to the 2012 Edition of JORC Code.

The existing concentrator will be modified in several key areas as noted in this report to improve concentrate grade and recovery based on metallurgical testwork undertaken in 2017 and 2018. The objective of these changes is to target a high-grade concentrate with maximum lead recovery from a lower lead head grade than was previously treated.

Lead concentrate from the modified Flotation Concentrator will be processed through a new Hydrometallurgical Facility built adjacent to the existing processing facilities.

The new Hydrometallurgical Facility will utilize a proprietary process which has been licensed to Lead FX. The process utilizes Methanesulphonic Acid (MSA) to leach the lead from the concentrates produced through the existing Flotation Concentrator. In broad terms, the lead rich solution from the MSA leaching is treated through an electrowinning circuit to produce lead cathode. The cathode is subsequently melted in a furnace with the molten lead cast into ingots as well as starter sheets for the electrowinning circuit.

A testwork program involving proof of concept, variability and piloting was designed, commissioned and managed by InCoR to provide design data for the Original DFS covering just the Hydrometallurgical Facility. Subsequently further batch testwork and a larger scale demonstration plant has been carried out by RHM in support of the DFS Update covering the entire Project flowsheet.

The primary product from the Project will be premium quality lead ingot shipped into existing markets, however, lead cathode as a product could be sold should a market be identified.

Figure 1: Location map

1.2 Geology and Mineralization

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 mineralization at Magellan is a sulfide-free supergene lead accompanied by silicification, argillic (illite, kaolinite) and sericitic alteration of the host sandstone and stromatolitic dolomite of the Yelma Formation. The mineralization 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 mineralization. The mineralized unit is a quartz clay breccia containing 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 Gold Consolidated (RGC) 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 between 0.1% and 3.15% Pb from holes drilled at the south-western 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 Paroo Station lead carbonate deposits (Magellan – including Gama, Cano and Pinzon) have been explored via a series of drilling campaigns dating back to the early 1990s. The distant Pizarro and Drake deposits were also initially drilled in the early 1990s and small numbers of holes were drilled during the following two decades until significant infill programs were completed in 2010.

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

In 2015, two drilling programs were completed at the Paroo Station Mine and surrounding exploration prospects. Reverse circulation (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 mineralization 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 (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 testwork for the DFS. These laboratories have been certified in accordance with ISO/IEC 17025.

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

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 Goldfields Corporation (Renison) initially discovered the Magellan deposit in June 1991 from stream sediment sampling while exploring the region for base metal mineralization.

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) 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 Pty Ltd. This action secured the rights to a 100% interest in the Paroo Station Lead Mine and the Renison properties were subsequently transferred to RHM during 2002.

On April 20, 1999, Ivernia agreed to invest in the Paroo Station project by acquiring a direct 15.7% equity interest in RHM from Polymetals Pty Ltd (Polymetals), the sole shareholder of RHM.

In September 2000, Ivernia acquired a 90% equity interest in Polymetals and acquired the remaining equity ownership in Polymetals in 2003. In May 2003, Ivernia 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 A$2.1 million (M).

During 2003, Ivernia and the Sentient Group Limited (Sentient) formed an undertaking whereby Sentient agreed to provide financing to RHM in exchange for a 40% interest. In 2005, Ivernia acquired Sentient's then 49% interest in RHM thereby becoming the sole owner of the Paroo Station Mine through its 100% interest in RHM.

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

The Paroo Station Mine was constructed during 2004, commissioned during 2005, and achieved commercial production on October 1, 2005.

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.

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. No lead egress was found and a thorough investigation resulted in discovery of a laboratory error. The 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 Pty Ltd changed its name to Rosslyn Hill Mining Pty Ltd, and changed the name of the mine to the Paroo Station Mine in November 2012.

Operations resumed in April 2013, with mining and processing continuing successfully through to the end of 2014 as world metal prices fell.

On January 16, 2015, Lead*FX*, announced its decision to move the Mine and processing plant into a care-and-maintenance phase to the TSX.

Milling continued until January 31, 2015. The Mine was transitioned to care-and-maintenance status during early February 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 dmt of lead carbonate concentrate was produced at the Mine by open pit methods, 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 discovered 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 project 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 *Environmental Protection Act* (*EP Act*) was undertaken by the Office of the Environmental Protection Authority (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 project.

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, 835,800 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, 1,437,958 t of ore was processed at an average head grade of 7.0%Pb producing 80,915 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% Pb yielding 14,000 t 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 between 2005 and 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**
Conc. grade	%	64.0	64.8	66.8	**65.4**
Conc. Pb content	t	115,900	52,200	134,800	**302,900**

1.3.4 Current Project Status

The operation remains in care-and-maintenance and no ore has been processed at site since February 2015. Activities have been conducted to support the DFS Update in terms of updating 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 Paroo Station Mine process plant is a conventional mineral concentrator consisting of crushing, grinding, sulfidization, 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 Update was conducted in 2018 predominantly by Australian Laboratory Services (ALS) in Balcatta, Western Australia. The test program included:

- Variability testwork – Hydrometallurgical testing
- Batch testwork on Flotation Concentrator
- Batch testwork on Hydrometallurgical Facility
- Demonstration Plant testing – Flotation
- Demonstration Plant testing – Hydrometallurgical Facility.

Previous testwork programs (2017 and 1999–2001) are also described in Section 13.

The key production data from the DFS Update testwork includes:

- Ore throughput = 2.185 Mtpa
- Flotation Pb recovery (following enhancements) = 83% producing 72% Pb concentrate
- Hydrometallurgical Facility Pb recovery = 98%–99%
- Lead ingot = 70,000 tpa (plus design margin up to 80,000 tpa) at 99.99% purity.

1.5 Mineral Resource Estimate

The Mineral Resource estimate for the Mine includes the main Magellan Hill deposits and the outlying Pizarro and Drake satellite deposits, located approximately 8 km south and 11 km south-west respectively from the Paroo Station Mine infrastructure.

The Magellan Hill, Pizarro and Drake Mineral Resources have been reported in accordance with the guidelines of the JORC Code (2012). Further detail can be found in previous Technical Reports (SRK 2015 and SRK 2018).

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 as part of a 2016 Mineral Resource update.

For the Magellan Hill deposits and the Pizarro deposit, no additional exploration data have been incorporated into any of the Mineral Resource estimates.

The Drake Mineral Resource was originally estimated in 2005 and reported in accordance with the guidelines of the JORC Code (2004). As part of the 2016 Mineral Resource update, the QP reviewed the Drake Mineral Resource estimate and associated documentation. The Drake Mineral Resource was subsequently updated and reported in accordance with the guidelines of the JORC Code (2012) reporting code.

In 2019, all Mineral Resources associated with the Paroo Station Mine have used an updated reporting cut-off, as a result of the new processing opportunities and changed economics. The Mineral Resources are now reported using a cut-off of 1.3% Pb (the previous cut-off used was 2.1% Pb).

Table 2: Mineral Resource estimate as at February 15, 2019

Deposit	Resource Category	Tonnes (Mt)	Grade (% Pb)	Contained Pb Metal (kt)
Magellan (including Gama)	Measured	4.5	4.2	185
	Indicated	14.5	4.3	625
	Total Measured + Indicated	19.0	4.3	810
	Inferred	3.3	3.9	130
Cano	Measured	1.6	3.4	55
	Indicated	2.1	2.4	50
	Total Measured + Indicated	3.7	2.9	105
	Inferred	0.8	2.3	15
Pinzon	Measured	0.1	6.1	5
	Indicated	9.5	4.1	390
	Total Measured + Indicated	9.5	4.1	395
	Inferred	2.0	3.5	70
Pizarro	Measured	0	0.0	0
	Indicated	4.6	3.1	140
	Total Measured + Indicated	4.6	3.1	140
	Inferred	2.0	2.8	55
Drake	Inferred	3.7	3.4	125
Stockpiles	Measured	2.9	2.4	70
Total	Measured	9.1	3.5	315
	Indicated	30.6	3.9	1,205
	Total Measured + Indicated	39.7	3.8	1,520
	Inferred	11.7	3.4	396

Source: Optiro (2019).

Notes:

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 1.3% Pb and depleted for mining to December 31, 2015. There has been no mining or processing of material during the 2016–2018 calendar years.
3. The stockpiled Mineral Resource is based on mine production data.
4. The Mineral Resource figures are based on the Mineral Resource Report prepared by Mr Kahan Cervoj (MAusIMM, MAIG), who is an employee of Optiro Pty Ltd, 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 January 2019 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.
8. All resources are reported inclusive of Ore Reserves/Mineral Reserves.

1.6 Mineral Reserve Estimate

The Paroo Station 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 historical as well as more recent production information, including current cost, revenue and metallurgical recoveries generated as part of the DFS Update 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 updated open pit optimization incorporating accepted product pricing and current project costs and operational parameters. The open pit optimization underpinned revised mine staging, mine designs and mine production scheduling.

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

Open pit optimization was used to identify the optimum economic pit shape based on the highest project cashflow. The pit optimization process seeks a solution to a complex 3D mathematical relationship involving the resource model, geotechnical slope guidelines, product revenue, project constraints, modifying factors and costs.

The key inputs into the optimization process include:

- Product prices
- Mining costs
- Processing, realization and administration costs
- Process recoveries
- Pit slope angles
- Prepared model.

The resource model was converted to a mining model by a process of regularization to account for dilution and ore losses. The diluted model has been used as the basis for optimization, pit evaluation and scheduling. Further preparation included adding cost, recovery, royalties and revenue drivers to 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 optimization process. These include mining costs per bench, processing costs, metallurgical recovery formulas and expected metal price.

The Whittle Four-X software package was used to develop the pit optimization shells.

Multiple pit optimization 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 processes suited to the extraction of the potentially economic mineralization.

Changes in the undiscounted cashflow and ore tonnage variance for each parameter have been plotted, where a steeper slope on any curve represents greater sensitivity to the parameter represented by that curve. The curve is defined over a ±20% variability from the base case for each parameter.

Table 3: Mineral Reserve statement as at February 15, 2019

Deposit	Reserve Category	Tonnes (Mt)	Grade (% Pb)	Contained Pb Metal (kt)
Cano	Proved	1.5	3.3	51.6
	Probable	1.2	2.4	29.4
	Total	2.8	3.1	81
Magellan	Proved	4.3	4.2	177.6
	Probable	13.1	4.1	540.2
	Total	17.4	4.1	717.9
Pinzon	Proved	0.1	5.9	5
	Probable	9.2	3.8	350.4
	Total	9.2	3.8	355.4
Pizarro	Proved	0.0	0.0	0.0
	Probable	3.9	3.1	120.7
	Total	3.9	3.1	120.7
Stockpiles	Proved	2.9	2.4	69.6
	Probable	0.0	0.0	0.0
	Total	2.9	2.4	69.6
Total	**Proved**	**8.8**	**3.4**	**303.8**
	Probable	**27.5**	**3.8**	**1,040.8**
	Total	**36.3**	**3.7**	**1,344.6**

Source: AMC (2019).

Notes:

1. Mineral Reserves are a subset of Measured and Indicated Mineral Resources. The Mineral Reserve estimate was developed to JORC Code (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 mineralization, 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 Laurie 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 10, 2015.
5. The pit limits for the open pit were selected through optimization using the Gemcom Whittle Four-X implementation of the Lerchs-Grossman algorithm. The optimization considered Measured and Indicated Mineral Resources only. Pit designs followed the optimization shell outline that developed the highest 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 = (-0.1017 × % Ore Grade^2 + 2.7556 × % Ore Grade + 73.5%)/100 limited to a maximum of 92.5%, Hydrometallurgical Plant recovery 97.87%.
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 50 cm skin around the 1.60% Pb cut-off grade envelope. Within the Ore Reserve pit design, the application of dilution resulted in inclusion of 9.92% dilution and results in an ore loss of 1.83%. Metal pricing of US$2,269/t Pb plus US$94/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 project. 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 project.
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 Paroo Station 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 orebody.

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.

1.8 Recovery Methods

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

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

- Crushing
- Grinding
- Concentration of non-sulfide lead-bearing minerals is carried out by sulfidising froth flotation
- Lead concentrate product dewatering and handling.

1.8.1 Recovery Models

An integrated METSIM model developed as part of the DFS Update combines the Flotation Concentrator model and the Hydrometallurgical Facility model. This has been used to develop the overall plant design.

Initial data for the model was derived from the testwork database available at that time. The data in the model has subsequently been updated with data from the Demonstration Plant operations.

The flotation concentrate elemental composition was derived from an average of the concentrate produced from the Demonstration Plant flotation plant operations. Reagent consumptions for the Flotation Concentrator were set to reflect quantities derived from the Demonstration Plant flotation operations.

Leaching extents for the lead minerals were derived from batch testwork and Demonstration Plant data. Minor element leaching extents were determined from the Demonstration Plant leaching chemistry.

After development of the base case model, which is set to the expected average lead mineralogy, two other models were developed for the minimum and maximum anglesite content. Minor lead minerals were assumed to be constant across all three models. While the throughput parameters described below reflect a specific nominal case, the process plant has been designed to produce a nominal 70,000 tpa lead ingot.

Flotation Recovery Model

A Concentrator METSIM model developed to reflect the proposed modified flowsheet has been used and validated by SNC-Lavalin 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 (semi-autogenous grinding) to SABC (SAB/Crush), a modified flotation circuit using existing equipment with relocated concentrate streams off first rougher and first cleaner cells and addition of a flotation column to produce a final concentrate in the 71%–74% Pb grade range to feed the Hydrometallurgical Facility.

A grade recovery algorithm for the revised flowsheet was developed for use in assessing the Flotation Concentrator performance across the range of anticipated feed compositions as part of the Demonstration Plant testwork. The Demonstration Plant flotation circuit was configured to mimic the proposed concentrator flowsheet.

Hydrometallurgical Facility Model

A METSIM model was developed for a 'base case' mineralogy, based on a weighted average of the annual variability samples, comprising predominantly 80.0% Pb carbonate (cerussite) and 9.17% Pb sulfate (anglesite) at an overall concentrate grade of 71.8% Pb. Other minor minerals assumed to be in the concentrate, based on X-ray diffraction (XRD) analysis, are pyromorphite (1.38%), galena (2.22%), leadhillite (0.54%), kaolinite (1.06%), hematite (0.48%) and quartz (4.03%). The flowsheet is described in detail below.

On completion of the variability testwork, additional METSIM models were run for the assumed minimum (3.05%) and maximum (21.4%) anglesite levels, which have been run to assess the impact of the changing concentrate mineralogy on mass balance flows and operating costs.

The following figures depict the flowsheet for the Flotation Concentrator and the Hydrometallurgical Facility.

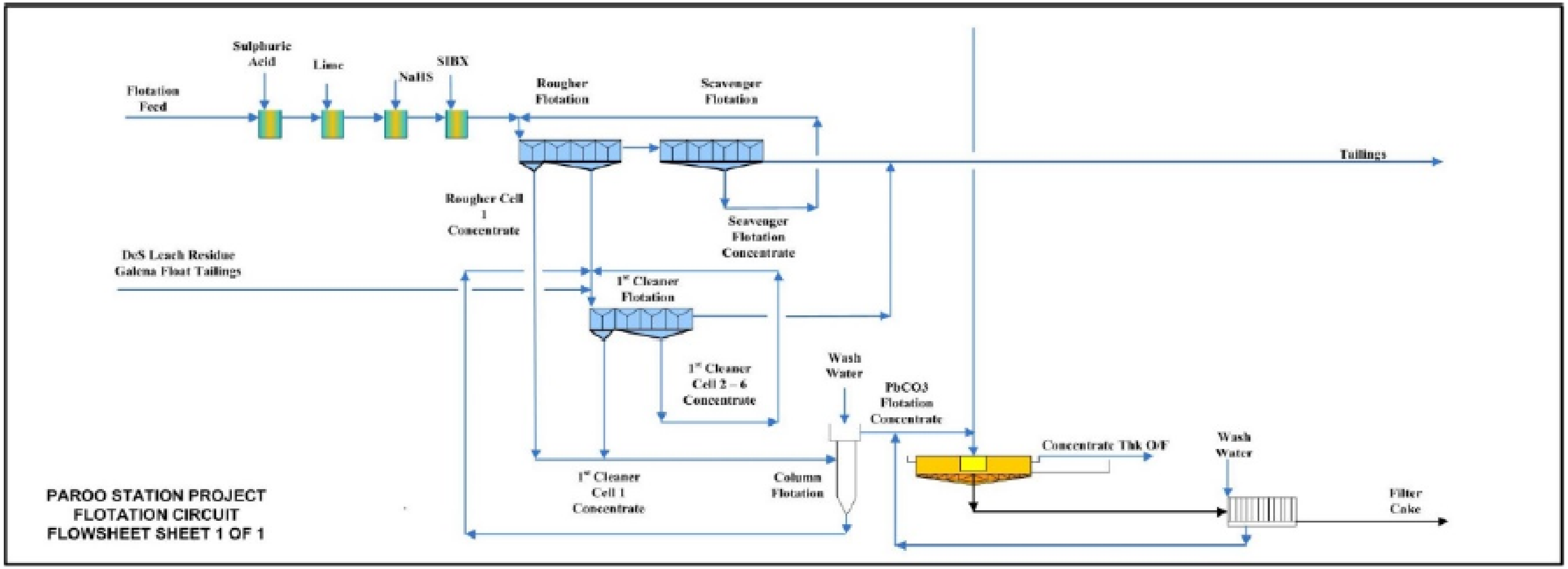

Figure 2: Paroo Station Project – flotation flowsheet

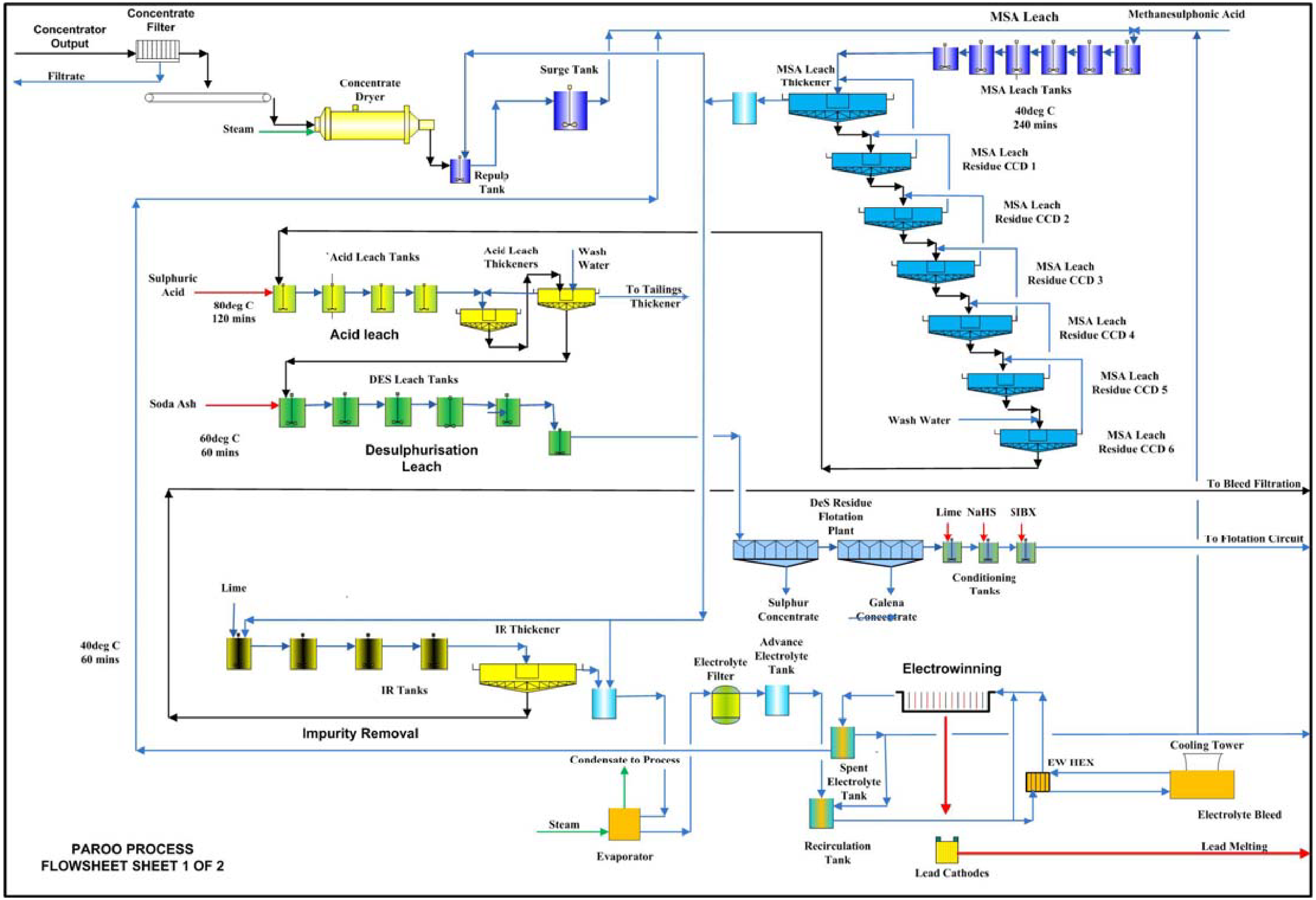

Figure 3: Paroo Station Project – Hydrometallurgical process flowsheet – Sheet 1

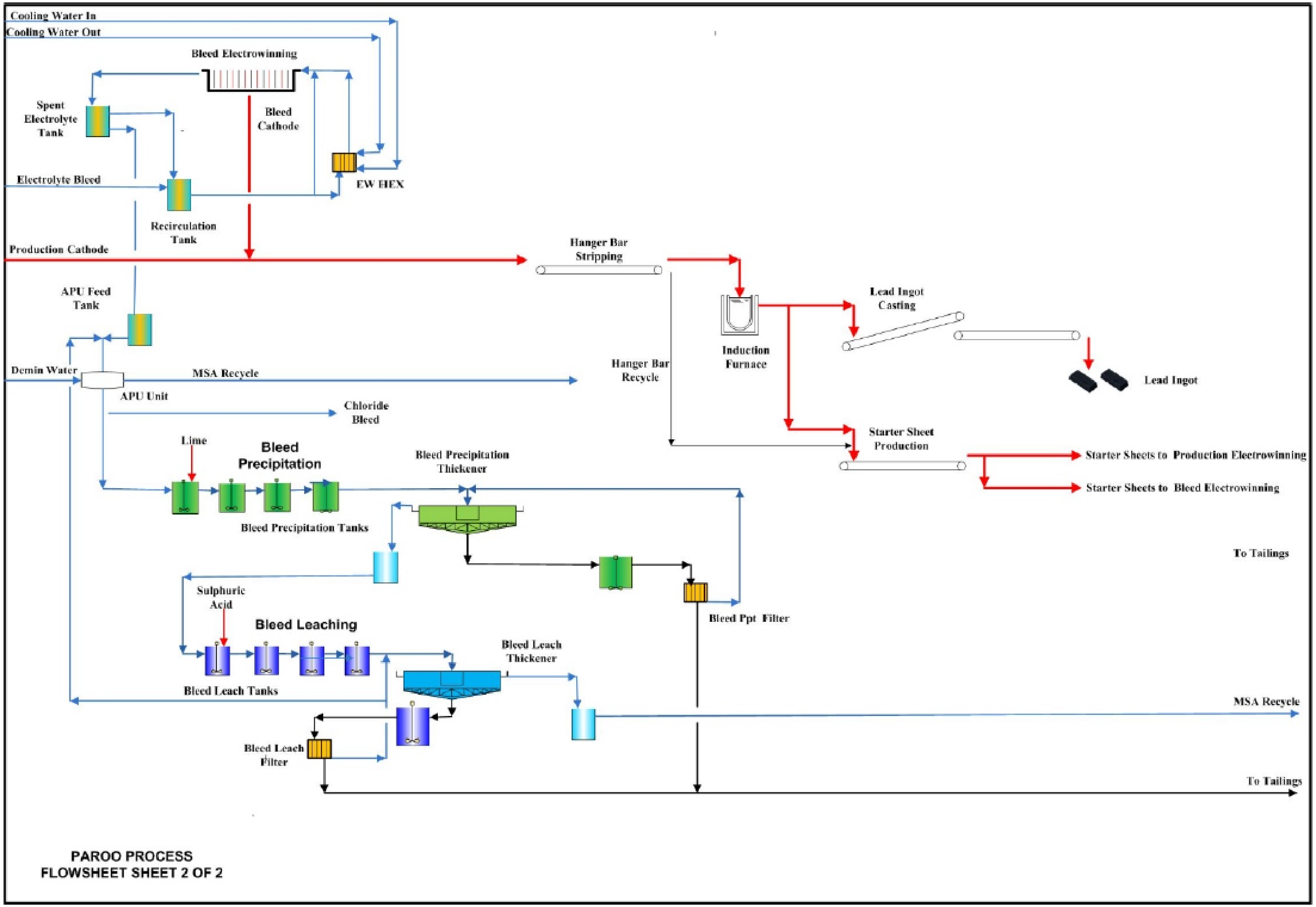

Figure 4: Paroo Station Project – Hydrometallurgical process flowsheet – Sheet 2

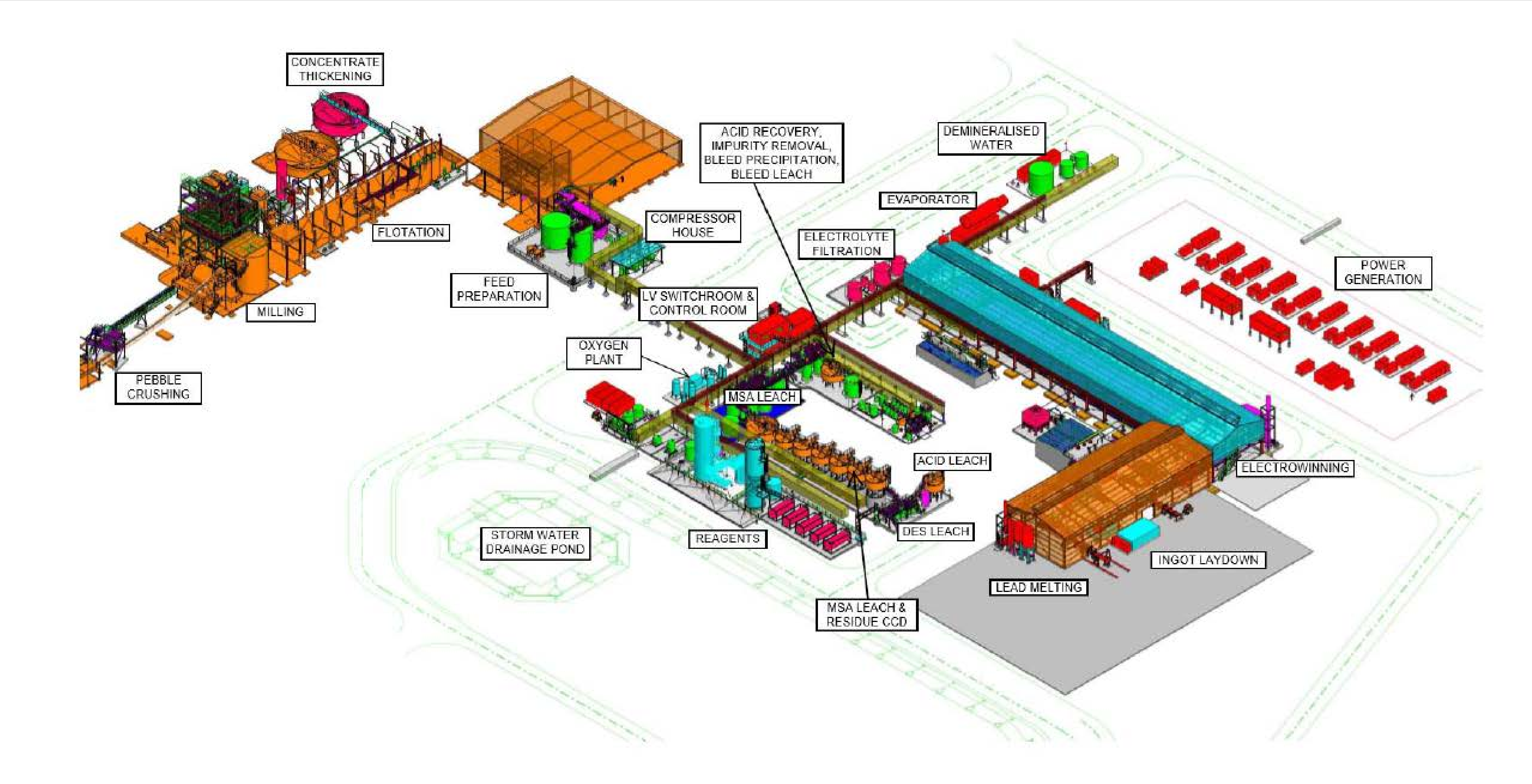

Figure 5: General layout of Flotation Concentrator and proposed Hydrometallurgical Facility

1.9 Project Infrastructure

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

- Processing facilities
- Power station and infrastructure
- Tailings storage facility (TSF) and 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:

- Department of Water and Environmental Regulation (DWER)
- The Department of Mines Industry Regulation and Safety (DMIRS)
- Department of Health (DoH)
- Department of Transport (DoT).

Ministerial Statement 1083 was signed by the Minister for the Environment on September 25, 2018, following release of EPA report and recommendations 1620. The EPA report considered the Rosslyn Hill Mining 'Hydromet Facility & Mine Extension Proposal' referral document of April 20, 2018. The Ministerial Statement provides environmental Conditions for the proposal (Project), which includes an increase the disturbance footprint by 400 ha, taking the total disturbance footprint to 980 ha, within the development envelope. The approval 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 revised forecast LOM. The approval also includes the Hydrometallurgical Facility and the proposed new electricity generation plant at site.

The Mining Proposal for the Hydrometallurgical Facility was approved by the DMIRS on October 31, 2018, under the West Australian *Mining Act 1978*. The approval was granted to commence the development and operation of the project in accordance with revised mining tenement conditions. The revised tenement conditions reflect the material provided within the RHM, Mining Proposal document describing the Hydrometallurgical Facility and associated mining and operational changes to the project. The approval allows for onsite activities under the *Mining Act*, but does not allow for construction activities to commence.

A Works Approval for the Hydrometallurgical Facility was approved by the DWER on November 30, 2018, under Part V of the *EP Act 1986*. The approval was granted to allow the construction of the Hydrometallurgical Facility, subject to conditions. RHM currently holds a Prescribed Premises License L8493/2010/2, permitting the control of emissions and discharges to the environment, and the monitoring and reporting of them. The Works Approval specifies emission levels such that during testing and commissioning of the Hydrometallurgical Facility, the proposed emissions described in the Works Approval document are confirmed to allow the issuing of a Prescribed Premises License Amendment, with nominated emission limits.

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

1.11 Economic Analysis

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

- The capital cost estimate to build the proposed Hydrometallurgical Facility, make modifications to the existing Flotation Concentrator and associated infrastructure for the project is US$183.7M (including Owner's costs of US$20.8M and a contingency of US$14.9M and growth allowances of US$6.6M).
- The average operating cost to produce a 99.99% Pb ingot is US$1,276.28/t (including overheads and sustaining capital over the 17-year life of mine).
- The developed flowsheet and recoveries for the operation are for production of up to 80,000 tpa of 99.99% Pb ingot.
- A Mineral Reserve estimate is 36.3Mt at a grade of 3.7% Pb over a 17-year LOM.
- Concentrate grades in the order of 72% Pb were achieved over a range of head grades from 3% Pb to 11% Pb. An average recovery of 83% Pb was achieved at a head grade of 4% Pb.
- Impurity elements in the concentrate were significantly reduced, resulting in lower Hydrometallurgical Facility operating costs.
- Lead extraction in MSA (methane sulfonic acid) leach averages approximately 90% across all lead mineralogy based on a single pass through the leach circuit. The lead extraction from the MSA leach residue averages 98%, resulting in an overall extraction of 81.3%.
- Lead cathode was produced at current densities of 300–350 A/m^2, which equates to a cathode plating rate of 70,000–80,000 tpa. Cathode quality exceeds 99.99% Pb.

Table 4: Financial returns

Description	Estimate	Comments
Total cost to first production	US$184M	To start of operations
Payback period	4.0 years	From start of operations
Internal rate of return	24.6% per annum	From start of construction
After-tax project cashflow		
Project revenue	US$2,584M	From start of operations
- Less all-in sustaining costs	-US$1,487M	From start of operations
Cashflow before tax	US$1,096M	From start of operations
- Less income tax	-US$253M	From start of operations
Cashflow after tax	US$843M	From start of operations
Present value		
- GPV (8.25% real discount rate)[1]	US$430M	From start of construction
- NPV (8.25% real discount rate)[2]	US$257M	From start of construction

Notes:
1 – GPV = gross present value = present value of cashflow after tax.
2 – NPV = net present value = present value of total cost to production

The following revenue assumptions were used:

- Wood Mackenzie price curve with long-term average price of US$2,350/t
- Lead premia based on Fastmarkets Metal Bulletin pricing for 99.97% Pb purity adjusted for 99.99% Pb to Southeast Asia
- Ocean freight netback based on RHM quotes.

1.12 Capital and Operating Costs

A summary of the independent assessment of capital costs for the Hydrometallurgical Facility together with the modifications to the Concentrator Plant is provided in Table 5.

Table 5: Capital costs summary

Item	Total (US$ M)
Direct costs	114,560,102
Indirect costs	33,390,911
Subtotal base costs	147,951,013
Contingency	14,931,924
Subtotal including contingency	162,882,937
Owner's costs	20,833,000
Total	**183,715,937**

Source: RHM (2019).

The operating costs for the Paroo Station Mine are summarized in Table 6. The estimate base date is October 1, 2018.

Table 6: Operating costs summary

Cost Centre	Life of Mine		
	US$	US$/t ore (feed)	US$/t Pb (ingot)
Mining	362,791,836	10.00	332.02
Flotation Concentrator	394,655,986	10.88	261.18
Hydrometallurgical Facility	332,459,946	8.89	295.11
Supply and Logistics	149,702,419	4.13	137.00
Sustainability	45,657,670	1.26	41.78
Corporate and General and Administration	86,411,841	2.38	79.08
Sustaining Capital	32,882,477	0.91	30.09
Total	**1,394,562,175**	**38.43**	**1,276.28**

Source: RHM (2019).

1.12.1 Lead Sales

Prior to production, RHM plans to enter into an offtake contract with a major global trading company for the sale and purchase of 100% of the annual lead ingot production. Under such offtake arrangements, RHM will sell lead ingot on a free on board (FOB) basis Fremantle, i.e. RHM will be responsible for delivering the lead ingots to container vessels in Fremantle, but not responsible for subsequent ocean freight and delivery to end-users.

The sales price received by RHM will comprise (i) the prevailing LME (London Metals Exchange) cash price for lead, plus (ii) a premium. Premiums are typically benchmarked from cost, insurance and freight (CIF) Index Premiums for specific destinations quoted by Fastmarkets MB, and will be agreed by RHM and offtakers based on the lead ingot specification and subject to adjustment for offtaker costs for managing risk, marketing, freight, insurance and other costs.

The notional CIF sales price is adjusted to an actual delivered FOB basis by a freight netback for the offtaker's cost of containerized ocean freight from Fremantle to index premium destinations. The financial analysis is based on forecasts from the Wood Mackenzie Lead Market Assessment for:

- Long-term LME lead prices
- Long-term lead premiums for index premium destinations
- Freight costs to index premium destinations.

Lead index premiums in the financial analysis are based on 99.99% Pb ingot, CIF index premiums and freight netbacks are based on Southeast Asian destinations, and index lead premiums are adjusted for the aforementioned offtaker costs.

Table 7 shows the forecast sales values.

Table 7: Forecast sales values

Production Year	Sales Price (US$/t Pb)	Sales Amount (t Pb)	Sales Value (US$ M)
Year 1	2,133	33,462	71.39
Year 2	2,116	70,915	150.08
Year 3	2,054	78,063	160.35
Year 4	2,228	78,491	169.05
Year 5	2,154	76,356	186.61

Production Year	Sales Price (US$/t Pb)	Sales Amount (t Pb)	Sales Value (US$ M)
Year 6	2,444	75,226	183.85
Year 7	2,444	72,352	176.83
Year 8	2,444	79,519	194.35
Year 9	2,444	74,403	181.84
Year 10	2,444	70,370	171.98
Year 11	2,444	65,845	160.92
Year 12	2,444	62,166	151.93
Year 13	2,444	70,097	171.32
Year 14	2,444	57,998	141.75
Year 15	2,444	49,293	120.47
Year 16	2,444	43,662	106.71
Year 17	2,444	34,462	84.23
Average/ Total	**2,365**	**1,092,681**	**2,584**

1.12.2 Financial Analysis

Project cashflows by year of operation are set out in Table 8 and Table 9.

Table 8: Annual revenue and costs

Production Year	Total Revenue (US$ M)	Royalties (US$ M)	Mining (US$ M)	Flotation Concentrator (US$ M)	Hydromet Facility (US$ M)	Supply & Logistics (US$ M)	Other Opex (US$ M)
Year 1	71.39	-2.09	-16.63	-22.34	-15.34	-5.26	-7.79
Year 2	150.80	-4.30	-26.66	-23.45	-19.66	-9.58	-7.79
Year 3	160.35	-4.50	-26.92	-23.46	-20.60	-10.40	-7.79
Year 4	169.05	-4.83	-26.01	-23.52	-20.68	-10.45	-7.81
Year 5	186.61	-5.60	-26.08	-23.54	-20.53	-10.20	-7.79
Year 6	183.85	-5.52	-25.93	-23.55	-20.40	-10.07	-7.79
Year 7	176.83	-5.31	-26.15	-23.57	-20.07	-9.74	-7.79
Year 8	194.35	-5.83	-25.38	-23.58	-20.93	-10.57	-7.81
Year 9	181.84	-5.46	-23.64	-23.55	-20.31	-9.98	-7.79
Year 10	171.98	-5.17	-21.36	-23.58	-19.84	-9.51	-7.79
Year 11	160.92	-4.84	-21.79	-23.62	-19.31	-8.99	-7.79
Year 12	151.93	-4.58	-20.93	-23.72	-18.91	-8.57	-7.81
Year 13	171.32	-5.15	-17.93	-23.59	-19.80	-9.48	-7.79
Year 14	141.75	-4.27	-15.23	-23.69	-18.39	-8.09	-7.79
Year 15	120.47	-3.65	-15.78	-23.77	-17.36	-7.08	-7.79
Year 16	106.71	-3.24	-15.20	-23.91	-16.72	-6.44	-7.81
Year 17	84.23	-2.57	-11.17	-18.21	-13.62	-5.29	-7.36
Total	**2,584**	**-77**	**-363**	**-395**	**-322**	**-150**	**-132**

Table 9: Annual cashflows

Production Year	Sales Revenue (US$ M)	Variable Opex (US$ M)	Fixed Opex (US$ M)	Ongoing Capex (US$ M)	Gross Cashflow (US$ M)	Income Tax (US$ M)	Net Cashflow (US$ M)
Year 1	71.39	-23.81	-45.63	-2.97	-1.03	2.90	1.88
Year 2	150.08	-43.13	-48.31	0.00	58.65	0.00	58.65
Year 3	160.35	-45.24	-48.43	-2.70	63.98	0.00	63.98
Year 4	169.05	-45.21	-48.09	0.00	75.75	-17.12	58.63
Year 5	186.61	-45.59	-48.16	-8.55	84.33	-22.97	61.36
Year 6	183.85	-45.10	-48.15	-3.75	86.85	-23.55	63.30
Year 7	176.83	-44.45	-48.17	0.00	84.20	-22.87	61.33
Year 8	194.35	-45.99	-48.11	0.00	100.25	-26.41	73.84
Year 9	181.84	-43.60	-47.12	-8.40	82.71	-22.81	59.90
Year 10	171.98	-41.02	-46.23	-0.15	84.59	-22.30	62.29
Year 11	160.92	-40.06	-46.28	0.00	74.59	-19.66	54.93
Year 12	151.93	-38.60	-45.91	0.00	67.42	-17.76	49.66
Year 13	171.32	-38.37	-45.36	0.00	87.58	-23.02	64.57
Year 14	141.75	-32.05	-45.41	-8.40	55.89	-14.47	41.41
Year 15	120.47	-29.94	-45.49	-0.15	44.90	-11.36	33.54
Year 16	106.71	-27.63	-45.68	0.00	33.40	-8.51	24.89
Year 17	84.23	-22.44	-35.78	-13.65	12.35	-3.60	8.74
Total	**2,584**	**-652**	**-786**	**-49**	**1,096**	**-253**	**843**

1.13 Conclusions and Recommendations

The Paroo Station Lead Mine was shut down in early 2015 due to the low lead 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 cost exposure to third parties processing the concentrate offshore.

The DFS Update confirms the following:

- The capital cost estimate to build the proposed Hydrometallurgical Facility, make modifications to the existing Flotation Concentrator and associated infrastructure is US$183.7M (including Owner's costs of US$20.8M, contingency of US$14.9M and growth allowances of US$6.6M).
- The average operating cost to produce 99.99% Pb ingot is US$1,276.28/t (including overhead and sustaining capital over the 17-year life of mine).
- The developed flowsheet and recoveries for the operation are for the production of up to 80,000 tpa of 99.99% Pb lead ingot
- A Mineral Reserve estimate is 36.3Mt at a grade of 3.7% Pb for a 17-year LOM.
- Concentrate grades in the order of 72% Pb were achieved over a range of head grades from 3% Pb to 11% Pb. An average recovery of 83% Pb was achieved at a head grade of 4% Pb.

- Impurity elements within the concentrate were significantly reduced, resulting in lower Hydrometallurgical Facility operating costs.
- Lead extraction in MSA leach averages approximately 90% across all lead mineralogy based on a single pass through the leach circuit. The lead extraction from the MSA leach residue averages 98% resulting in an overall extraction of 81.3%.
- Lead cathode was produced at current densities of 300–350 A/m^2, which equates to a cathode plating rate of 70,000–80,000 tpa. Cathode quality exceeds 99.99% Pb.
- RHM recommends proceeding with financing of the Hydrometallurgical Facility and concentrator modifications.

SRK reviewed the actual and projected product sales and operating cost data for the Project. Based on this review and the above-defined variables, SRK concluded that the Project 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 Paroo Station Lead Mine (the Mine), located in the Wiluna district of Western Australia, is 100% owned by Lead*FX* Inc. (Lead*FX*) through its wholly owned subsidiary, Rosslyn Hill Mining Pty Ltd (RHM). Lead*FX* is an international base metal mining company listed on the TSX.

The purpose of this report pursuant to NI 43-101 and other rules of the Canadian Securities Administrators is to provide a technical summary of the mining and exploration assets in relation to the Paroo Station Lead Mine in Western Australia.

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

3 Assumptions, conditions, and qualifications set forth in this report.

This report is intended for use by Lead*FX* subject to the terms and conditions of its contract with the Qualified Persons (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 Joint Ore Reserves Committee (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 preparing this Technical Report are specialists in many recognised mining industry fields of study that 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.

Other than Dr David Dreisinger, none of the consultants or any associates employed in the preparation of this report has any beneficial interest in Lead*FX*. Dr David Dreisinger is a director of Lead*FX*. The remaining consultants are not insiders, associates, or affiliates of Lead*FX*.

The results of this Technical Report are not dependent on 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 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, 16, 19 to 28 of this Technical Report.
- Dr David Dreisinger is the QP responsible for sections 13, 17 and 18 of this Technical Report.
- Laurie 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 Section 14 of this Technical Report.

The Mineral Resource and Ore Reserve estimates prepared for this report were completed by the following consultants to the JORC Code (2012) standard:

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

2.3 Details of Inspection

Scott McEwing is responsible for the content, preparation, compilation, and editing of this Technical Report. The Certificates of Qualified Persons and Consents of Qualified Persons are provided in Appendix A.

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

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

A site visit to the Mine was conducted by Adrian Jones on 10 March, 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 project 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 (as QP)	SRK Consulting Ltd	Mining Engineer	November 11–12, 2014	Site visit, review and observation of operations
Kahan Cervoj	Optiro Pty Ltd	Geologist	July 23–25, 2014	Review of logging and sampling protocols
Adrian Jones	AMC Consultants Pty Ltd	Mining Engineer	March 10, 2015	Site visit, inspection of recently shut mining operations and project infrastructure
Dr David Dreisinger	Lead FX Inc	Metallurgical Engineer	January 16, 2017	Site visit and inspection of project infrastructure

2.4 Sources of Information

The opinion of each QP in this report is based on information provided to the QP by Lead*FX* and RHM personnel throughout the course of the investigations.

The QPs reviewed the available project 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 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:

- Project financials including lead market studies provided by Lead*FX* and/or RHM personnel, referring specifically to Section 21. The information was provided in the form of the Financial Model 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.

2.5 Effective Date

The effective date of this report is February 15, 2019.

2.6 Units of Measure

The International System for weights and units has been used throughout this report. Tonnage is reported in metric tonnes. Unless otherwise stated, all currency is in United States dollars (US$).

3 Reliance on Other Experts

The QPs' opinions in this report are 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 the project areas in support of this Technical Report.

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

- Project 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), project 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 Project is located 30 km west of Wiluna, and 2 km directly north of the Wiluna–Meekatharra road in the north-eastern Goldfields region of Western Australia. The Project 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 licenses (the Site). The leases and licenses cover in excess of 30,000 ha, including 2,447 ha of Mining Leases.

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

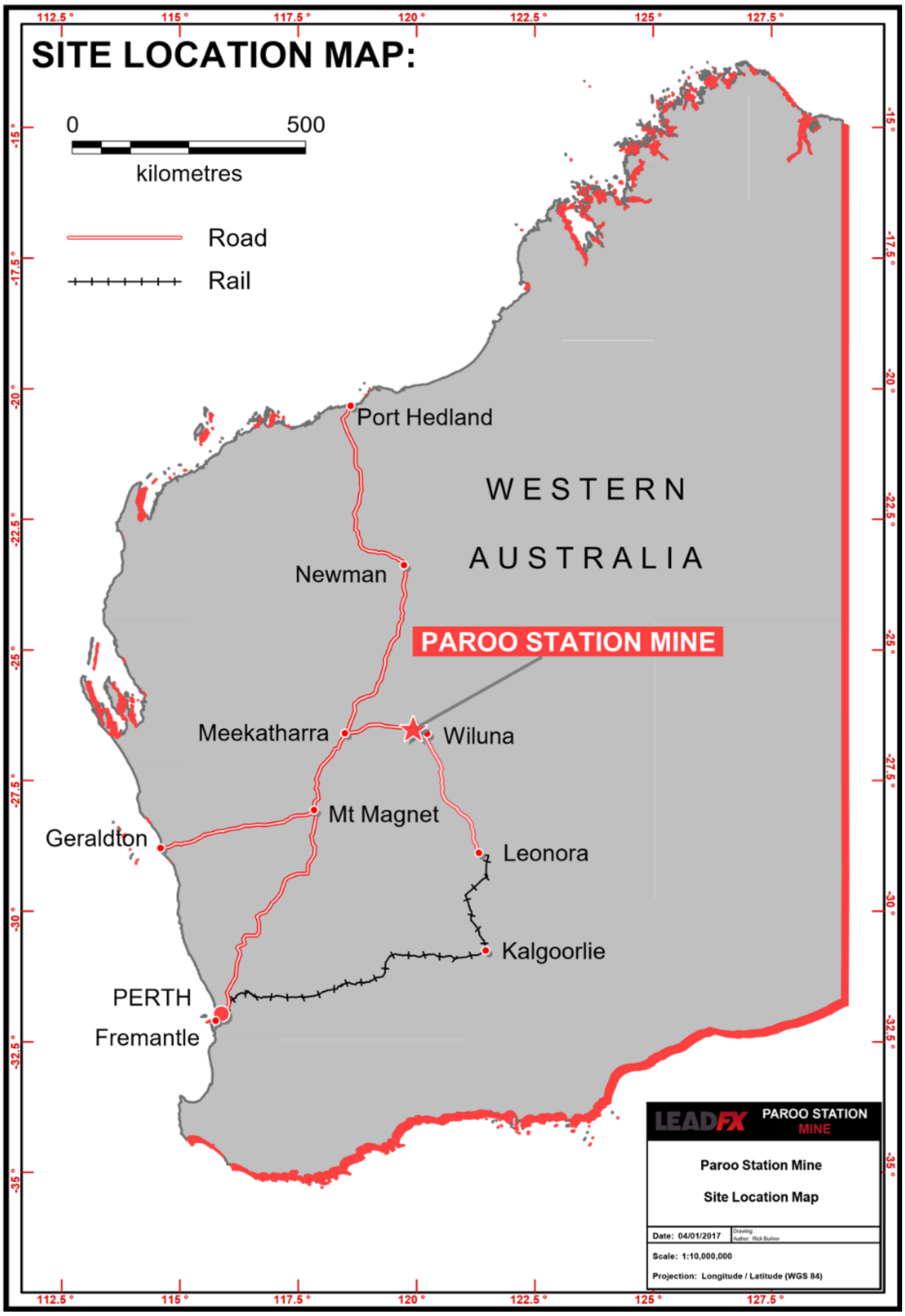

Figure 6: Location map

Source: RHM (2018).

4.2 Mineral Titles

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

The tenements listed in Table 11 are regarded as Core Project Tenements for the purposes of the LOM plan.

Table 11: Tenement position

Type	Title No	Status	Date issued	Date of expiry	Hectares
Rosslyn Hill Mining Pty Ltd – Granted Tenement Holdings					
Mining Lease	M53/502	Granted	05-May-99	04-May-20	975
Mining Lease	M53/503	Granted	05-May-99	04-May-20	499
Mining Lease	M53/504	Granted	05-May-99	04-May-20	426
Mining Lease	M53/1002	Granted	22-Jun-04	21-Jun-25	191
Miscellaneous License	L53/106	Granted	09-Dec-99	08-Dec-20	1
Miscellaneous License	L53/107	Granted	09-Dec-99	08-Dec-20	43
Miscellaneous License	L53/108	Granted	09-Dec-99	08-Dec-20	5
Miscellaneous License	L53/149	Granted	30-May-06	29-May-27	195
Miscellaneous License	L53/163	Granted	20-Jun-13	19-Jun-34	3,994
Miscellaneous License	L53/164	Granted	20-Jun-13	19-Jun-34	8,254
Miscellaneous License	L53/197	Granted	12-Jan-15	11-Jan-36	4,680
Miscellaneous License	L53/198	Granted	12-Jan-15	11-Jan-36	9,211
Miscellaneous License	L53/200	Granted	17-Dec-15	16-Dec-36	32
Miscellaneous License	L53/201	Granted	17-Dec-15	16-Dec-36	23
Prospecting License	P53/1528	Granted	15-Apr-11	14-Apr-19	22
Retention License	R53/4	Granted	18-Mar-19	17-Mar-24	614
Rosslyn Hill Mining Pty Ltd – Tenement Applications					
Miscellaneous License	L53/191	Application	15-Aug-14	21 years from date of grant	8.4
Miscellaneous License	L53/192	Application	15-Aug-14	21 years from date of grant	0.9
Miscellaneous License	L53/193	Application	15-Aug-14	21 years from date of grant	5.2
Miscellaneous License	L53/194	Application	15-Aug-14	21 years from date of grant	1
Miscellaneous License	L53/195	Application	15-Aug-14	21 years from date of grant	0.2
Miscellaneous License	L53/196	Application	15-Aug-14	21 years from date of grant	1.4

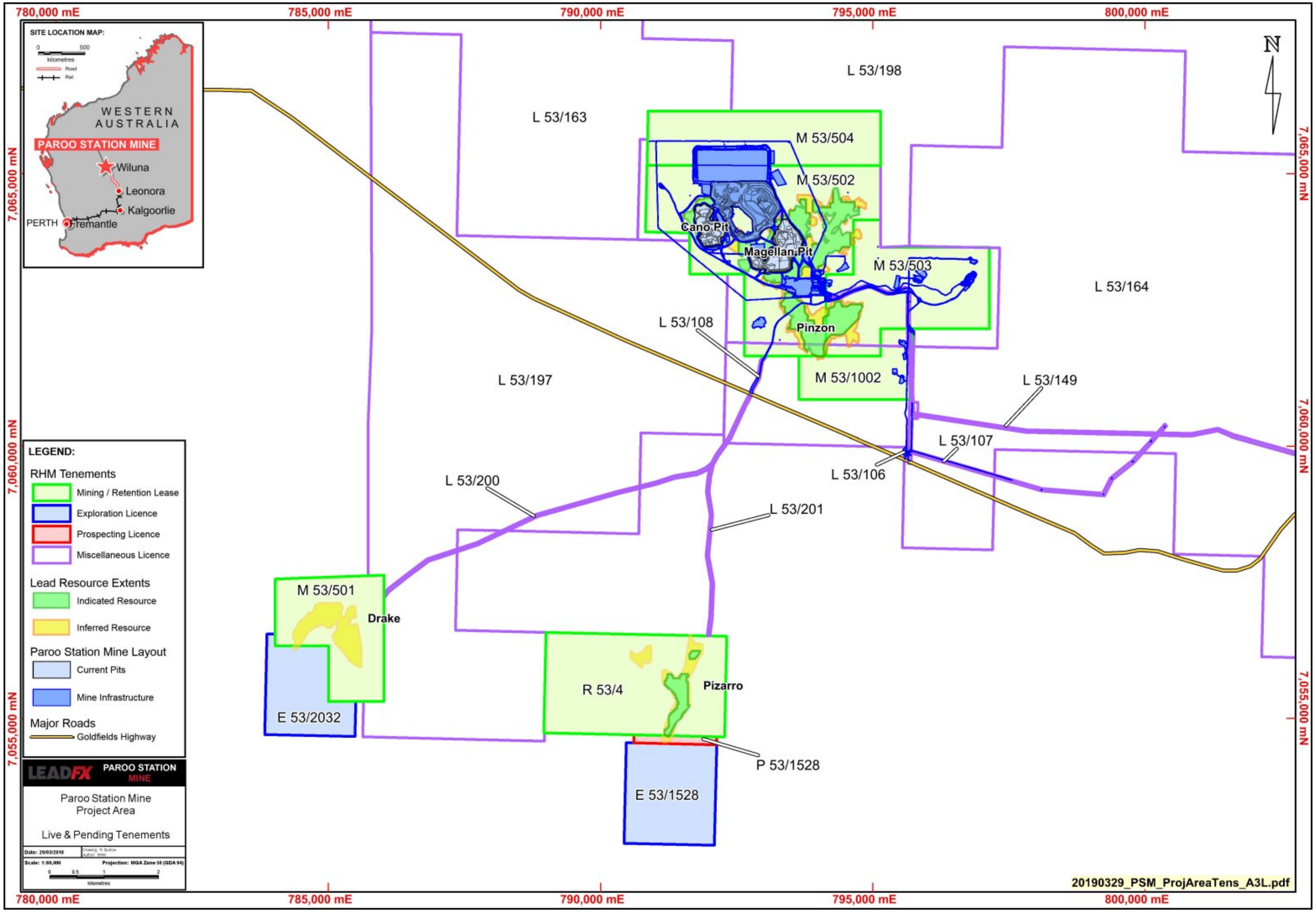

Figure 7: Land tenure map as at February 2019

Source: RHM (2019).

4.2.1 Nature and Extent of Issuer's Interest

In Western Australia, mineral rights belong to the State. The State Government issues and administers mining tenements under the relevant mining legislation, and mining companies must pay royalties to the State 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 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. With the exception of the Pizarro deposit on Retention License (R53/4), Rosslyn Hill Mining has been granted Mining Leases supporting its current Mineral Reserve.

Although production from Pizarro is scheduled in the current mine plan (although late in the program), the grant of the Mining Lease and settlement of Native Title matters are not finalised and will be required.

4.3 Royalties, Agreements and Encumbrances

RHM reports that there are two royalty payments 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 5.0%.

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 A$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 A$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 Rosslyn Hill Mining's gas pipeline route, requires an annual compensation payment into the Wiluna Claimant Trust Fund for use of the gas pipeline tenement area. The initial annual payment of A$20,000 was made in July 2006, and subsequent annual payments, indexed at the Consumer Price Index (CPI) rate for Perth, Western Australia, have been made.

4.4 Environmental Liabilities and Permitting

4.4.1 Environmental Liabilities

In March 2019, the Company filed its Compliance Assessment Report (CAR), along with three Annual Environment Reports (AERs) for 2018 to the 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 project, based on best available information. The cost estimate accounts for all aspects of rehabilitation and closure activities using third-party contractor rates.

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

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

- Department of Water and Environmental Regulation (DWER)
- The Department of Mines Industry Regulation and Safety (DMIRS)
- Department of Health (DoH).

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

- Ministerial Statement 1083
- DWER – Prescribed Premises License – L8493/2010/2
- DWER – License to Extract Water – GWL96342(4)
- Australian Communications & Media Authority – Licenses 1970164 and 1970178/1
- DMIRS – Dangerous Goods Site License –DGS020079
- DMIRS – Mining Tenement conditions
- DMIRS – Pipeline License – PL73
- Radiological Council – Licenses LX58/2006 15145 and RS28/2005 14619.

The construction and operation of the Hydrometallurgical Facility will require some new and/or updated minor operating permits to be obtained.

4.5 Other Significant Factors and Risks

Currently all regulatory approvals for the construction and operation of a Hydrometallurgical Facility at the Mine site and the necessary changes to support the extended LOM, have been gained.

The following approvals were gained during 2018:

***EP Act* (EPA governed – signed 25 September 2018):**

- Construction and operation of a new Hydrometallurgical Facility, broadly consisting of acid leach, electrowinning and melting, to convert the lead carbonate concentrate currently approved to be produced at the Mine site, to an estimated 70,000 tpa of lead metal
- Transporting of lead metal ingots (approximately 25 kg each) from the Mine site for export
- Increasing the onsite power generation capacity to 18 MW installed, to be fueled 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.

Works Approval (DWER – signed 30 November 2018):

- Works approval to construct and commission the new Hydrometallurgical Facility; following commissioning, an amended Prescribed Premises License that includes the Hydrometallurgical Facility will be issued.

Mining Proposal (DMIRS – signed 31 October 2018):

- Mining Proposal updated to reflect the new Hydrometallurgical Facility and the required mining and related infrastructure.

To allow future access to Magellan Hill orebodies below the water table and the Pizarro orebodies to the south, RHM plans to submit a further application under Section 38 of the *EP Act*. This will require groundwater and dewatering impact studies and flora and fauna studies for the southern orebodies to be undertaken, as well as consultation with relevant Government Departments to address any risk of failure to achieve the necessary environmental approvals.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Topography, Elevation and Vegetation

The Project 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 plateaus and hill slopes, and deep red earths in the drainage floors and channels (Table 12). The main landform of the project area is the north-west facing arm of the Finlayson Range and isolated hill formations of Proterozoic sediment outliers of the Finlayson Range. Such formations include Mount Russell (599 m Australian height datum (m AHD)), Mount Bartle (584 m AHD) and the hills containing the Magellan lead deposit (565 m AHD) (KH Morgan & Associates, 1999).

Table 12: Description of landforms, soil and vegetation associations in the 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 the upper parts	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: 50 to 1: 150; mainly with channelled tracts up to 30 m wide; concave marginal slopes, 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: 15 to 1:150.	Bed loads range from sand to boulders on hard pan or bedrock	Similar to drainage floors, but with fewer perennial grasses

Source: Adapted from Mabbutt, et al., 1963.

5.1.1 Soils

Soils in 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 features of soils in this region are their heavily leached nature and presence of a cemented or siliceous hardpan layer (Keith Lindbeck & Associates, 1999).

Hart et al. (1999) describes the project 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 center 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 (FIFO) basis (Shire of Wiluna website).

Access to the site is via the Goldfields Highway from Wiluna or via Meekatharra (Figure 6), 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 mining tenement area is immediately north of the highway and a well-maintained, 3 km gravel road links the Magellan Hill operations to the highway.

The operation'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, TSF, 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 typically 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 is managed on a FIFO basis (typically 8 days on, 6 days off, on 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 Project 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), station number 013012, 2014) as shown in Table 13. The mean annual evaporation rate at Wiluna is estimated at 4072 mm (Department of Agriculture, 1987), thus exceeding mean annual rainfall by approximately 3,800 mm.

The average annual rainfall for Wiluna is 259.2 mm (Wiluna BOM station number 013012, 2014) (Table 13). The region receives most of its rainfall in the summer (Figure 8), 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 that often exceed the average annual rainfall.

Table 13: Climatic data for Wiluna weather station

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).

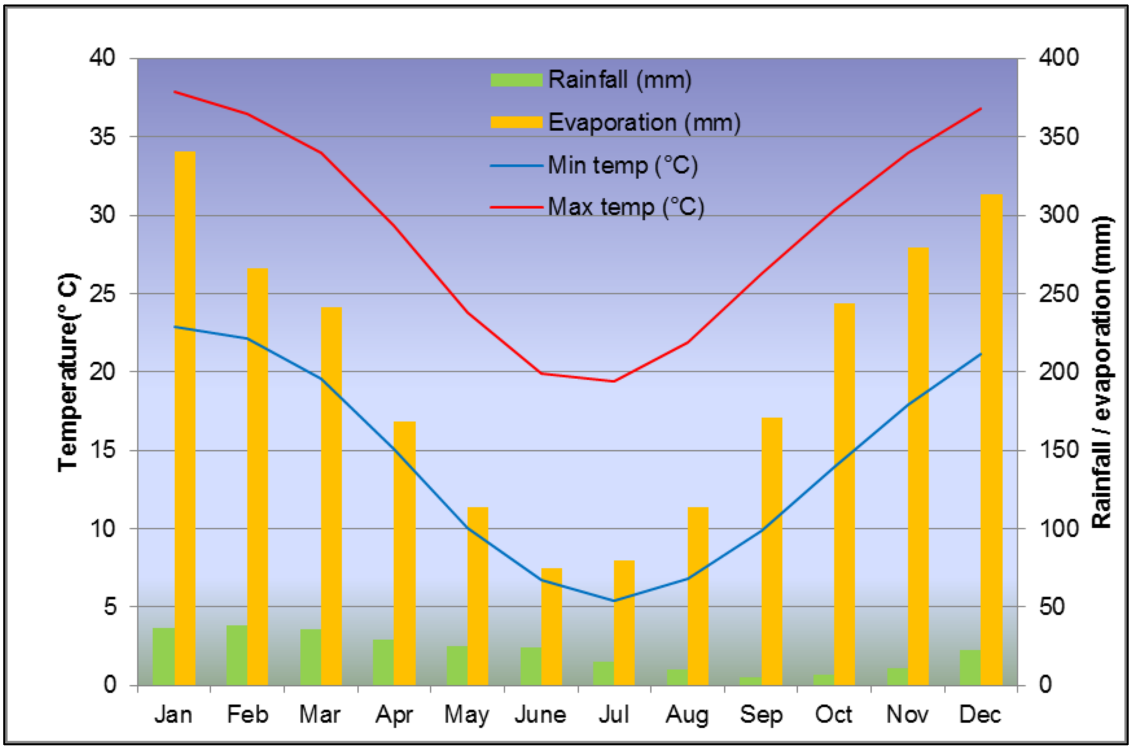

Figure 8: Climate data for Wiluna

Source: Bureau of Meteorology (2018).

5.4 Sufficiency of Surface Rights

With the exception of the Pizarro deposit on Retention License (R53/4), RHM has been granted Mining Leases that support the current Mineral Reserves. The granting of R53/4 will require inclusion of the area into an existing access agreement with the Traditional Owners of this area.

5.5 Infrastructure Availability and Sources

The current infrastructure in the Mine area is sufficient to support a fully operational mine and concentrator (Section 18). The Mine operated at a nameplate capacity of approximately 1.6 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.

5.6 Existing Infrastructure

5.6.1 Water

Water supply from an established borefield, with onsite treatment for the supply of potable water, is sufficient for historical throughput levels, as well as the planned water consumption at the Hydrometallurgical Facility. No increase to the water license quantity is required.

5.6.2 Electricity

The current diesel power generation plant will be partly refurbished and used as a back-up and emergency supply.

5.6.3 Tailings Storage

A paddock-style TSF currently exists on site.

5.6.4 Accommodation Village

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

5.7 Planned Infrastructure

The Hydrometallurgical Facility and related infrastructure is planned to be constructed to produce lead ingots.

5.7.1 Water

The operation of the mine with the Hydrometallurgical Facility does not require an increase to the water license quantity or borefield supply system.

5.7.2 Electricity

Electricity will be generated on site with new natural gas fueled engines.

5.7.3 Tailings Storage

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

5.7.4 Accommodation Village

The increased numbers of site personnel as a result of the operation of the Hydrometallurgical Facility will require some upgrading and refurbishment of the accommodation village.

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 on RHM completing a Bankable Feasibility Study (BFS) for the Paroo Station Mine by January 2002 and committing to develop a mine and plant with a design capacity of not less than 300,000 tpa ore.

In September 2001, following the completion of the BFS, 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, Lead*FX* (formerly Ivernia Inc.) agreed to invest in the project by acquiring a direct 15.7% equity interest in RHM from Polymetals, the sole shareholder of RHM.

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 A$2.1M.

In 2003, Lead*FX* 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, Lead*FX* acquired Sentient's 49% interest in RHM, thereby becoming the sole owner of the Mine through its 100% interest in RHM.

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. The 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 Company from making any further bulk exports of lead concentrate through the Port of Esperance. As a result, the Company was obliged to pursue alternative shipping arrangements to ship its concentrate through another port in Western Australia and to place the Mine on care-and-maintenance in April 2007 until such arrangements had been approved by the DWER.

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

RHM recommenced exporting lead concentrate from existing stockpiles through Fremantle Port in September 2009 and 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. No lead egress was found and a thorough investigation resulted in discovery of a laboratory error. The DWER gave permission for RHM to recommence transport in February 2011. RHM went into voluntary temporary closure in April 2011 to conduct an end-to-end review of operations. In parallel, the DWER conducted a Section 46 Ministerial Review of the implementation conditions to see whether the conditions should be changed. Interim conditions were issued on February 23, 2011 and superseded Ministerial Statement 559 by Ministerial Statement 783 until July 27, 2012 when the Minister for the Environment issued Ministerial Statement 905.

Operations resumed in April 2013, with mining and processing continuing successfully through to the end of 2014 as world metal prices fell.

On January 16, 2015 Lead*FX* announced the decision to move the mine and processing plant into a care-and-maintenance phase to the TSX.

Milling continued until January 31, 2015. The Mine was transitioned to care-and-maintenance status during early February 2015.

On May 12, 2017 Lead*FX* announced an agreement with InCoR Technologies Limited and InCoR Energy Materials Limited (InCoR) related to the transfer of lead refining technologies to Lead*FX* for the development of a lead refinery at Paroo Station Mine via a Definitive Feasibility Study (DFS). InCoR, subject to performance criteria, would earn a 43% share of Lead*FX*.

The DFS, funded by InCoR, examined the technical and financial viability of producing up to 70,000 tpa of lead ingots from the Mine. Lead ingots would be produced from a purpose-built Hydrometallurgical Facility to be constructed on site, adjacent to the existing Concentrator Facility, which would treat the flotation concentrate to produce lead ingot.

The DFS was prepared by global engineering and construction firm, SNC-Lavalin (Perth office) to AACE class 3 criteria for engineering, design and estimation. The engineering, design and estimating results coupled with the results of testwork programs feed into the capital and operating cost models that support the overall financial model.

The DFS supported the performance criteria for InCoR earning the agreed portion of shares in Lead*FX*.

6.2 Exploration and Development Results of Previous Owners

Renison initially discovered the deposit by stream sediment sampling while exploring the region for base metal mineralization. A series of regional rotary air blast (RAB) holes to the north and south of the Magellan Hill returned anomalous values between 0.1% Pb 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 the Maiden Mineral Resource estimate for the Magellan deposit.

RHM has completed several drill programs in the Magellan Hill area since 1997 for exploration, resource evaluation and sterilization purposes, which led to discovery of an additional five deposits.

6.3 Historic Mineral Resource and Mineral Reserve Estimates

6.3.1 Historical Summary

Previous Mineral Resource estimates for the deposits in the area 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 project's Mineral Resource estimates is provided in Table 14.

Table 14: Summary of Mineral Resource estimates

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)	JORC Code (2012) update

Source: Ivernia – Various historical documents and compilations.

The JORC Code is a professional code of practice that sets minimum standards for Public Reporting of Exploration Results, Mineral Resources and Ore Reserves. The JORC Code is consistent with the CIM definition standards. Mineral Resource estimates prior to 1999 were reported using the industry conventions of the time.

All Mineral Resource estimates from the Feasibility Study onwards (2001) have been carried out under the guidelines of the JORC Code. Those reported in the 1999–2004 period used the 1999 version of the code, those completed in the 2004–2011 period were reported under the 2004 version, and the estimates completed in 2014 and 2016 are reported under JORC Code (2012).

The current Mineral Resource estimate dated 2018 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 mineralized envelope was developed to account for new drilling. The cut-off grade for the Drake deposit was reduced to 2.1% Pb.

6.3.3 2010–2014

In 2010, CSA Global (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 'mineralized lodes' to establish detailed 1% Pb grade boundaries and to limit inclusion of internal waste lenses.

This differed from the previous 1% Pb grade envelopes which encompassed internal waste and led to a suspected overestimation of the waste blocks. This was a recognizable improvement from the 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 Mineral Resource estimate were further depleted by mining based on surfaces constructed from surveys of the mining outlines to the end of April 2011 (CSA, unpublished report, 2013).

6.3.4 2014–2015

In 2014, Optiro was commissioned to build revised ordinary kriged Magellan Hill and Pizarro resource models and report accompanying the Mineral Resource estimates. The models were built using updated parameters suitably designed and matched to reconciled mining and milling data from the 2010–2011 and 2013–2014 periods of operation.

6.3.5 2016 - Present

The January 2019 Mineral Resource estimate includes all depletion due to mining and processing activities when the Mine was put onto care-and-maintenance during January 2015 due to low commodity prices. The reporting cut-off grade was lowered to 1.3% Pb in January 2019. Stockpiles have been tabulated from actual mine production data.

No new data from drilling or other exploration work has been added to the Mineral Resource estimate, which, other than depletion and revision of the cut-off grade, remains unchanged from the 2014 estimate.

6.4 Historic Production

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 following the initiation of government investigations into bird fatalities in the vicinity of the Port of Esperance, approximately 181,100 dmt of lead carbonate concentrate was produced by open pit methods, 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 thereafter.

RHM voluntarily placed the project onto 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* Ministerial Statement 905 in July 2012. Ministerial

Statement 905 superseded all previous conditions and procedures and became the operational regime for the project.

6.4.2 2013–2015

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, the mining contractor mobilised 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, 835,800 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 (Ivernia, 2014).

In 2014, 1,437,958 t of ore was processed at an average head grade of 7.0%Pb to produce 80,915 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 was processed at an average head grade of approximately 7.4% Pb yielding 14,000 t of concentrate containing 9,900 t of contained lead.

No ore was processed in 2016, 2017 or 2018 due to the Mine being placed in care-and-maintenance.

6.4.3 Care-and-Maintenance 2015

On December 23, 2014, the Company announced that the decline in the LME lead price to levels not seen since mid-August 2012 was a significant factor affecting profitability and cashflow from operations and that, in line with a general downturn in commodity prices, Lead*FX* was experiencing a drop in sales prices realized 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 and Mineralization

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).

The regional, local and property-scale geological setting are discussed in the following sections.

7.1 Regional Geology

The Paroo Station Mine lead deposits are situated in rocks of the Earaheedy Basin overlying the south-eastern corner of the Paleoproterozoic Yerrida Basin, at the northern margin of the Archean Yilgarn Craton in central Western Australia (Figure 8). The Yerrida Basin is one of several Proterozoic basins that formed between the Pilbara and Yilgarn cratons (McQuitty and Pascoe, 1998).

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 formed after an oblique collision between the Pilbara and Yilgarn cratons about 1.8 Ga. The Yerrida Basin was probably formed in a widening rift at approximately 2.2 Ga and was later affected by the Capricorn Orogeny. It 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 of the Earaheedy Basin in the east (Hooper, 2010).

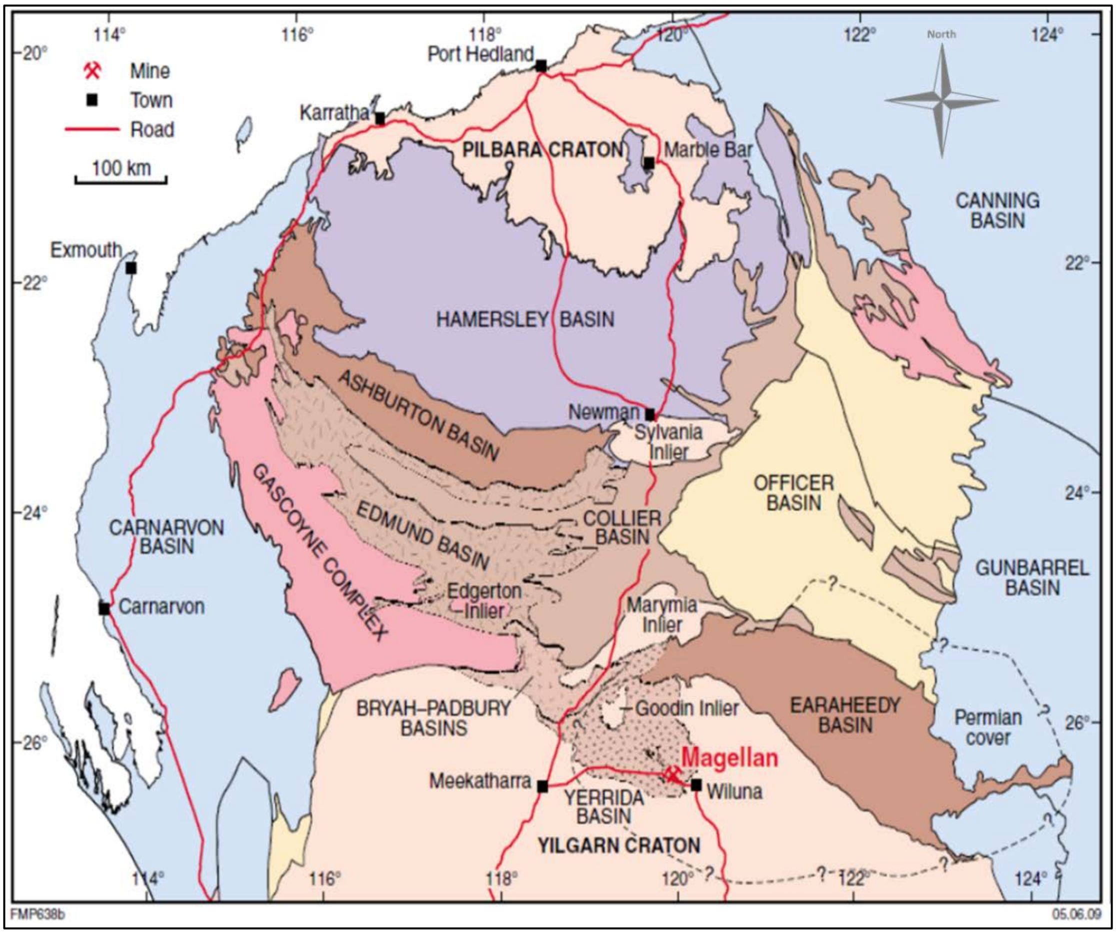

Figure 9: Regional geological setting of Magellan lead deposit

Source: Pirajno and Burlow (2009).

Pirajno et al. (2010) note that the <1.84 Ga Earaheedy Basin (Figure 9) 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 across the Yerrida Basin, and to the north and north-east beneath the later Proterozoic Collier and Officer basins (dashed outline in Figure 9).

The stratigraphy of the Yerrida and Earaheedy basins is presented in Figure 10. Within the Yerrida Basin, the Mooloogool Group overlies the basal 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 stable, low-relief environment (Pirajno et al., 2010).

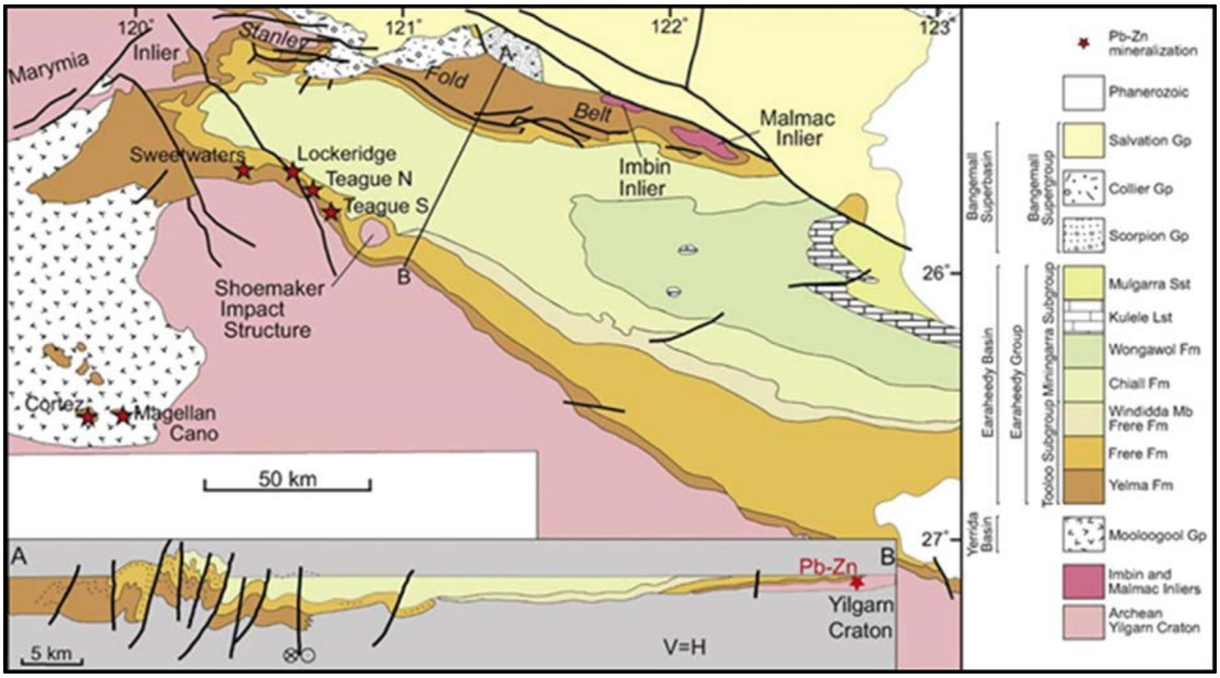

Figure 10: Simplified geological map and stratigraphy of the Earaheedy Basin showing the locations of lead-zinc deposits

Source: Sergeev et al. (2016), after Pirajno and Burlow (2009).
Note: The Paroo Station Mine (Magellan and Cano) lead deposits are shown in lower left corner.

Pirajno and Burlow (2009) note that the Earaheedy Basin contains the Earaheedy Group (Figure 10), 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 Mooloogool Group (Yerrida Basin).'

The Earaheedy Group is made up of the (lower) Tooloo Subgroup and the (upper) Miningarra Subgroup. 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 Paroo Station Mine lead deposits occur at the base of the Earaheedy Group, overlying the Mooloogool Group of the Yerrida Basin. Mineralization of a similar style is located in the smaller deposits, Pizarro and Drake, that lie south and south-west of Magellan, mainly along the unconformity surface between the Juderina Formation (Windplain Group) and small outliers of the Earaheedy Group (Figure 11).

The Yerrida Group is represented by two formations in the Mine area; the lowermost Juderina Formation (Finlayson and Bubble Well Members) is unconformably overlain by the Maraloou Formation. Yelma Formation sandstone and carbonate of the Earaheedy Group unconformably overlie the Yerrida Group in the 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 east–west trending ridge (Finlayson Range).

Unconformably overlying the Juderina Formation in the Magellan 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 Maraloou shale and siltstone is poor due to preferential weathering, and much of the unit surrounding the Mine area is covered by alluvial plain and sheet wash deposits. 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 in the Paroo Station Mine project area (Burlow, 2015).

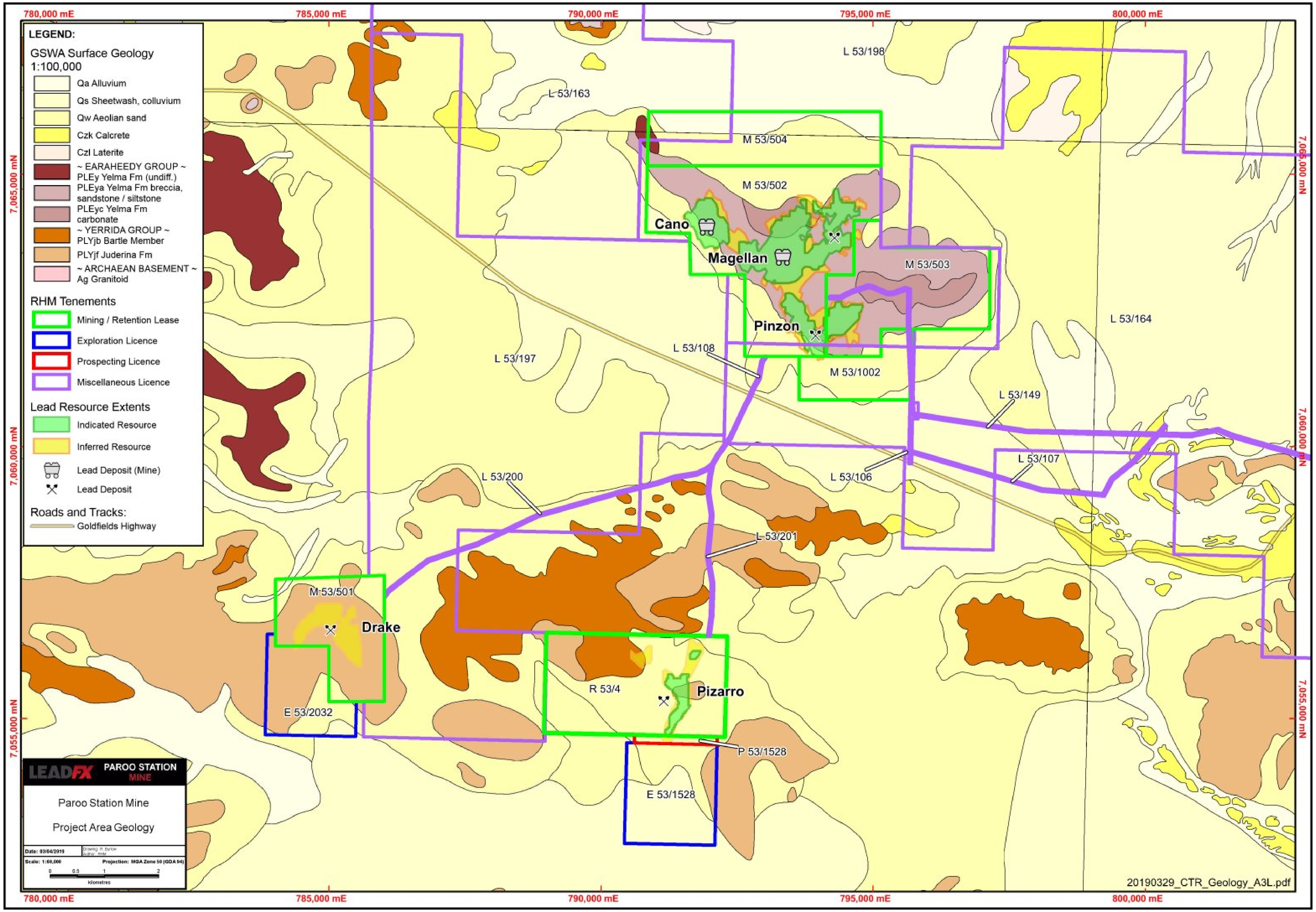

Figure 11: Schematic geological map of the Paroo Station Mine project area

Source: RHM (2019).

Note: Rock units adapted from Geological Survey of WA (GSWA) 1:100,000 surface geology map sheets 2844, 2845, 2944 and 2945.

The Yerrida sediments are commonly flat-lying to moderately dipping to the north and west, and within the project area, the dominant structural features are faults trending north-east and south-east. 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. 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 mineralization (Looi, 2010).

7.3 Property Geology

The project includes five lead deposits – Magellan, Cano, Pinzon in the Magellan Hill area, and Pizarro and Drake in the Finlayson Range south of the Paroo Station Mine (Figure 11). Other small lead mineral occurrences (i.e. Cortez) are present across the local area.

The Paroo Station Mine area contains remnant discontinuous outliers of the Yelma Formation (Earaheedy Basin), forming low hills surrounded by shales of the Maraloou Formation (Yerrida Basin). The informally-named Magellan Hill – a mesa of approximately 5 × 2.5 km rises 25–50 m above the surrounding alluvial plain. A relatively thin (up to 60 m) sequence of the Yelma Formation sediments unconformably overlies the Maraloou shale. The Yelma sequence includes a basal fining-upwards clastic sandstone-siltstone sequence that is overlain by a silcretized quartz-clay collapse breccia with relics of dolostones at the base of the unit. The Yelma Formation is overprinted by surficial massive silcrete several meters thick and a thin colluvial soil (Sergeev et al., 2017).

At the Pizarro and Drake lead deposits south of the Magellan Hill group, lead mineralization occurs in sediments of the Yelma Formation and the underlying Juderina Formation (Yerrida Basin) (Looi, 2010).

The surficial lead anomalism, mine lithologic sequence and mineralization styles are discussed in the following sections.

7.3.1 Surficial Lead Anomalism

In the Paroo Station Mine project area, natural surface lead-in-soil anomalism is widespread. Erosion along the flanks of the Magellan Hill mesa and in seasonal stream gullies has removed the upper units and exposed the shallow-dipping sandstone sequence (Looi, 2010). The erosion commonly forms a ragged breakaway slope capped by the upper silcrete horizon.

The southern margins of the Cano, Magellan and Pinzon deposits show a well-developed secondary dispersion geochemical anomaly. The magnitude of the lead anomaly is greatest where mineralization approaches or intersects the surface. Distinct vegetation anomalies occur where the ubiquitous ‘mulga’ acacia shrub land degrades suddenly to open patches of spinifex grass. These areas often display values of lead-in-soil exceeding 20,000 ppm Pb, restricting the growth of the long-lived mulga in favor of the shorter-lived spinifex (Burlow 2015, Elliott 2015). The lead-in-soil anomalism decreases over distances of 2 km or more, tapering gradually down slope towards background levels of 50–75 ppm Pb in the shallow alluvial plain at the foot of the mesa. The anomalies swing around to the south-east, influenced by the seasonal sheet wash and regional West Creek drainage eastwards towards Lake Way.

Minor, surficial lead-in-soil anomalism identified in patches of the calcrete formation along the southern West Creek drainage south of the Magellan mesa indicates local scavenging of lead by the calcrete carbonates.

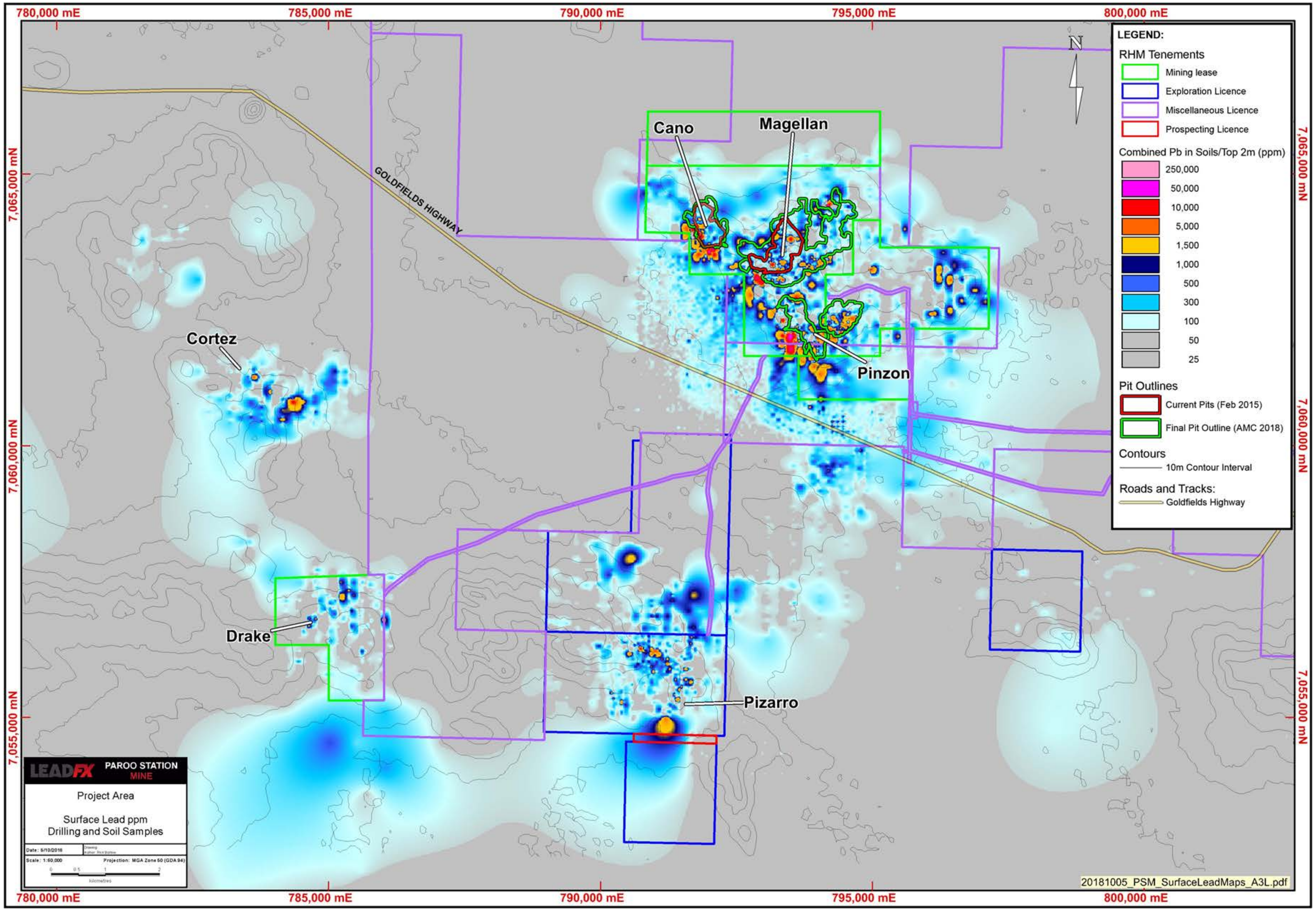

Figure 12: Map of naturally occurring lead-in-soil anomalism compiled from portable XRF data and from surface (0–2 m) RC/RAB drill assays

Source: RHM (2018).

Note: The Magellan Hill group and Finlayson Range lead deposits are as labelled.

A map showing the distribution of surface lead-in-soil is shown at Figure 12. The map was compiled from portable X-ray fluorescence (pXRF) surveys, conventional soil sample surveys and assays from the top 2 m of all drill holes. Effective detection limits are approximately 50 ppm Pb.

The satellite lead deposits at Pizarro, Drake and Cortez show similar, though smaller, anomalies.

Paroo Station Mine Stratigraphy

A general description of the stratigraphy of the Paroo Station Mine sequence is shown in Figure 13. The informal mine sequence and lithostratigraphic names were established by Pascoe and Edgar (1995).

The mine host sequence is divided into the following components with increasing depth from surface:

- Laterite and silcrete caprock
- Quartz-clay breccia
- Saprolitic clay zone
- Saprock siltstone and sandstone
- Maraloou Formation.

Each lithology is described in the sections below (parentheses indicate Rock Codes).

Laterite/ Silcrete Cap (CZL, CZS)

This surface unit is from 0.5 m to 3 m thick, consists of variably cemented, pisolitic, clayey, red-brown lateritic material and contains nodular fragments from 2 mm to 15 mm in diameter as well as lithic fragments. An uneven colluvial cover is also present across much of the Mine area (Pirajno et al., 2010). A cap of silcrete (secondary rock comprised of massive siliceous cement) is well developed over most of the Magellan area. The unit is often exposed at surface where it forms a massive cement that may extend for several meters into the subsurface (0–5 m). The silcrete becomes less well developed with increasing depth and its lower contacts are transitional with the underlying bedrock units (Elliott in RHM, 2003). Across the top of Magellan Hill, a thin skeletal sandy soil is interspersed with cobbles of eroded silcrete. At the edges of the hill, the silcrete forms a breakaway with boulder scree, particularly along the south-western flank of the Pinzon and Magellan deposits.

Quartz-clay Breccia (YCS, YC)

A poorly lithified and variably vughy solution collapses breccia comprised of fragments of silicified stromatolitic carbonate, chert, siltstone, euhedral crustiform quartz and colloform banded quartz in a white-grey to tan colored matrix of clay (limonite, goethite, kaolin) and silt (± cerussite, anglesite). The top part of the unit (up to 20 m) is variably syncretized and is referred to as ‘syncretized quartz-clay breccia’ (YCS) and is transitional with the units above and below.

At the Cano and Magellan deposits’ south-western margins, the upper syncretized and quartz-clay breccia units are locally absent due to erosion. The quartz-clay breccia is thickest in the central Magellan deposit area (~35 m); the thickness at Cano varies relative to topography (erosion level) from 0 m to 16 m thick (Elliott in RHM, 2003).

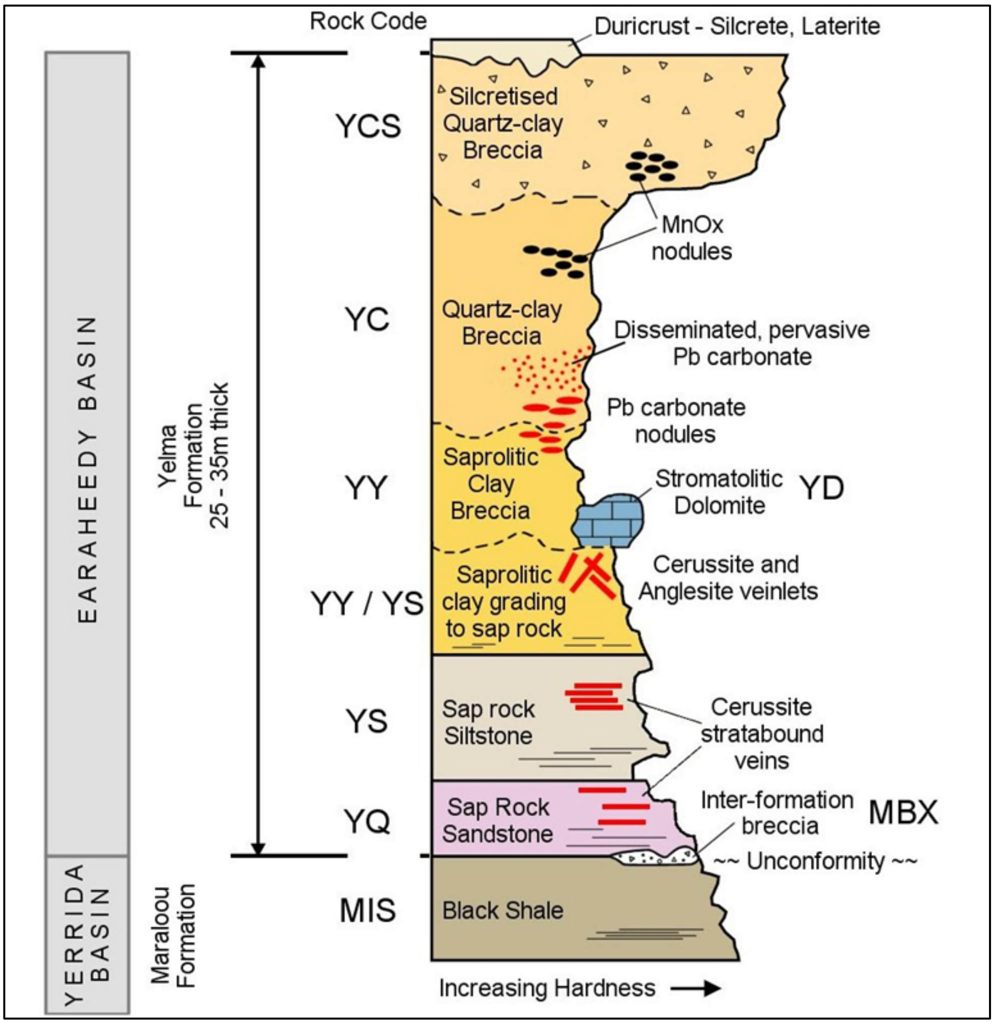

Figure 13: Generalized stratigraphy of the Magellan Hill area

Source: RHM, 2018.

Dolomite (YD)

Sergeev et al. (2017) add that several lenses of stromatolitic dolomite also outcrop within the Magellan open pit at the north-eastern and southern flanks of the Magellan deposit. The dolomite forms a series of dome-shaped lenses up to 15 m thick that are composed of dark-grey massive to laminated micro- to medium-crystalline stromatolitic dolomite. A 5–10 cm thick moderately weathered rind forms at the upper contact of the dolomite with the quartz-clay breccia.

Preserved stromatolite taxa including *Asperia digitata*, *Pilbaria deverella*, *Ephyaltes edingunnensis*, *Yelma digitata* and *Yandilla meekatharrensis* have been identified in the breccia and in relic dolomite lenses flanking the Magellan Hill mineralized zones. The stromatolite presence indicates a marine lagoonal or supra-tidal ephemeral lake/ sabkha depositional environment similar in nature to the modern environments seen at Shark Bay, Western Australia.

The breccia unit and relic dolomite lenses are interpreted as the highly altered and weathered Sweetwaters Well Member of the Yelma Formation (Pirajno et al., 2010).

The dolomite lenses are largely barren of lead mineralization, although the thin outer weathered rind and fractures can have grades in excess of 10% Pb.

Clay (YY)

An unconsolidated clay unit marks the boundary between the basal clastic units and the upper breccia unit. The clay is tan to orange-brown in color and is comprised of kaolin with accessory iron oxides (limonite, goethite), cerussite and fine quartz/ silica fragments. The clay unit is transitional to the quartz-clay breccia unit above and is thought to represent the residue after dissolution of a carbonate precursor. In the Magellan deposit, the clay unit is commonly 2–4 m thick; however, the unit at Cano is less discernible and may be included in the lower clastic units (Elliott in RHM, 2003). The unit locally grades downwards to saprock, partially retaining its original sedimentary fabric (Pirajno et al., 2010); in these zones it is identified as weathered sediment from the underlying siltstones.

Siltstone (YS)

Underlying the clay unit, marking the top of the clastic sequence is a strongly weathered, immature ferruginous siltstone of greywacke composition. At Magellan, the siltstone is commonly <4 m thick. At Cano, the unit is recorded as an interbedded siltstone/ sandstone unit (Elliott in RHM, 2003).

The siltstone is grey to white, finely laminated very fine to fine-grained rock composed of up to 0.1 mm clasts of quartz, detrital sericite and lenses of iron oxyhydroxides in a quartz-feldspar-clay matrix. The lamination reflects layer variations in quartz grain sizes. Siltstone laminae are ferruginous in places, mostly of hematite composition. Iron oxyhydroxide pseudomorphs and voids after carbonate crystals locally occur indicating the original carbonate-bearing composition of the fresh siltstones (Sergeev et al., 2017).

Sandstone (YQ)

The lowermost sandstone unit marks the base of the Yelma sequence at the Magellan Hill. This comprises partly oxidized fine- to medium-grained sandstone, interbedded with siltstone; a rhythmic succession is observed, steadily fining upwards. Microscopic examination reveals a well-sorted feldspathic hematitic sandstone, consisting of rounded to subrounded quartz and microcline grains, cemented by a fine quartz–feldspar–hematite granular matrix, with selective sericitic alteration, mostly in the matrix. Sericite crosscutting veinlets are present. Carbonate spots and coatings of lead minerals, probably cerussite-anglesite intergrowths are observed.

The sandstone and siltstone units display a shallow 0°–15° northerly dip with gentle interference folding about axes trending north-west and north-east. At Cano, the sandstone shows common mud-cracks, ripple marks and trough/ trough cross laminations, suggesting a very shallow to emergent depositional environment (Pirajno and Burlow, 2009, Pirajno et al., 2010).

Ripple mark orientations from the Cano Pit suggest a general NW–SE paleocurrent trend. Stratabound zones and veins of anglesite and cerussite are up to 10 mm thick, appear to replace beds and laminae, and exhibit grades of 10% Pb or more. More intense folding is seen locally in the south-west portion of the Cano Pit, and is possibly due to the presence of a nearby NNW-trending fault (Pirajno et al., 2010).

Maraloou Shale (MIS)

The Maraloou Formation surrounds the Magellan Hill mesa, forming a low-lying plain with outcrop largely obscured by colluvium and scree material. The unit is known primarily from drill intersections beneath the Yelma Formation outlier that contains the Magellan deposit and is present within the mine area as finely laminated and fissile, black graphitic shale and shaly siltstone. It contains abundant disseminated framboidal and euhedral pyrite (Pirajno et al., 2010). At the uppermost contact, the unconformity between the Maraloou shales and the overlying sandstones of the Yelma Formation, a chaotic zone of rip-up breccia is commonly observed in drill holes.

The Maraloou shales are largely barren of lead mineralization, with some rare drill holes showing remobilized lead (as cerussite) within the shale unit beneath the Pinzon deposit.

7.4 Significant Mineralized Zones

Lead mineralization occurs in five main deposits with outlying prospects and mineral occurrences (Figure 14). The deposits are listed below in general order of size:

Magellan Hill group (comprises the Paroo Station Mine):

- Magellan (including Gama)
- Cano
- Pinzon.

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

- Pizarro
- Drake.

Finlayson Range prospects and mineral occurrences:

- Cortez.

The Gama deposit referred to in earlier literature is now considered an extension of the Magellan mineralization.

The Columbus mineralization referred to in earlier literature is now considered part of the Pizarro deposit.

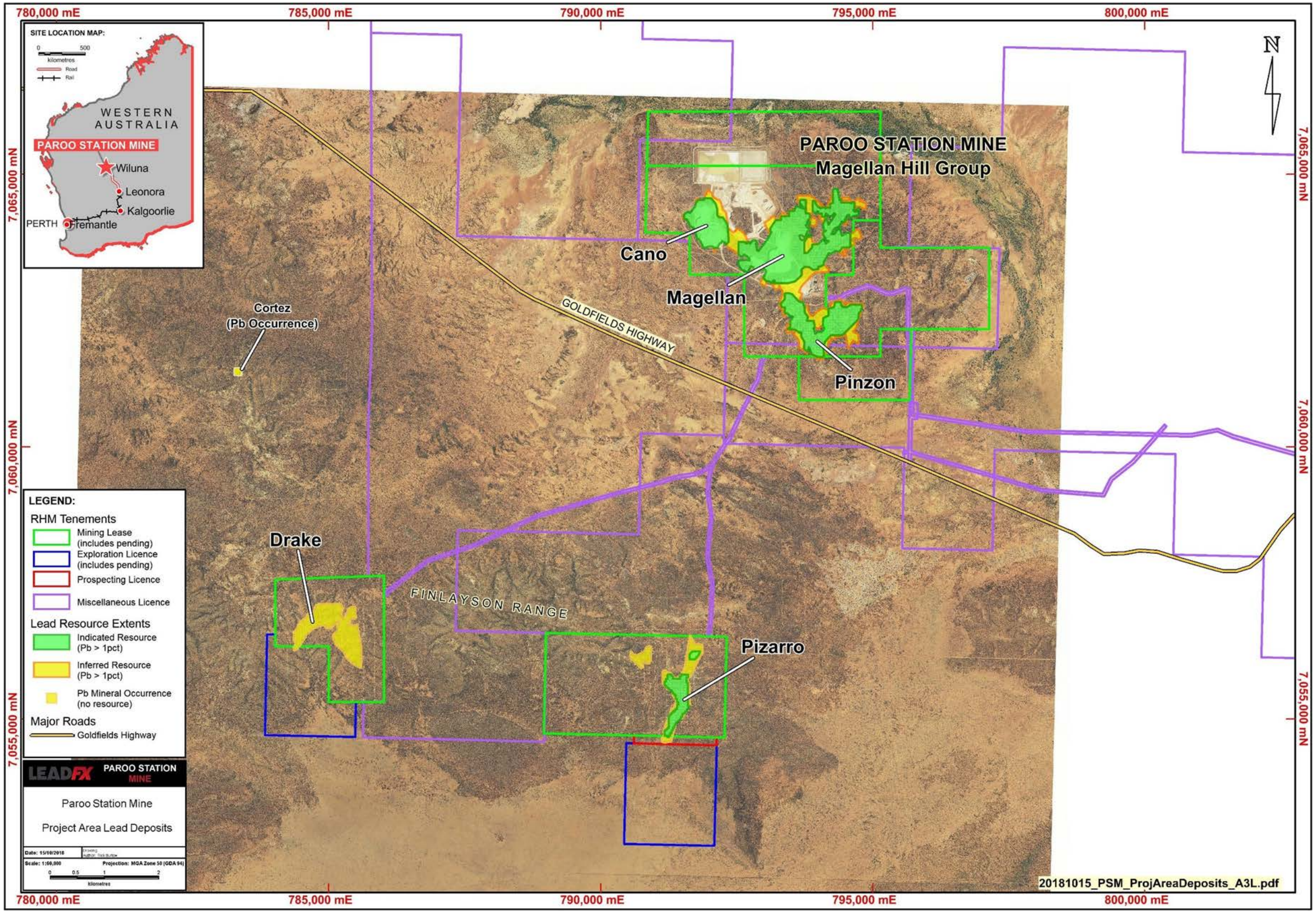

Figure 14: Lead deposits of the Paroo Station Mine area, showing the Magellan Hill group (Cano, Magellan and Pinzon) and the Finlayson Range deposits (Pizarro and Drake) to the south and southwest of the Mine

Source: RHM (2018).

7.4.1 Magellan Hill Group

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

The Magellan Hill group of lead deposits – Magellan, Cano and Pinzon – are contained in a mesa outcrop with dimensions of 5 × 2.5 km, comprising the Yelma Formation which hosts the lead mineralization, the majority of which is contained in a quartz-clay breccia and sediment sequence up to 35 m thick. The mineralized unit is described as an upper quartz-clay breccia with fragments of completely silicified carbonate with relict stromatolitic structures, siltstone, and euhedral and colloform banded quartz in a white clay-rich matrix up to 35 m thick (Sibbel, 2009).

The Magellan deposit extends for approximately 1,600 m in a NNE direction with an average width of approximately 900 m and an average vertical thickness of economic mineralization of approximately 12 m (Figure 15).

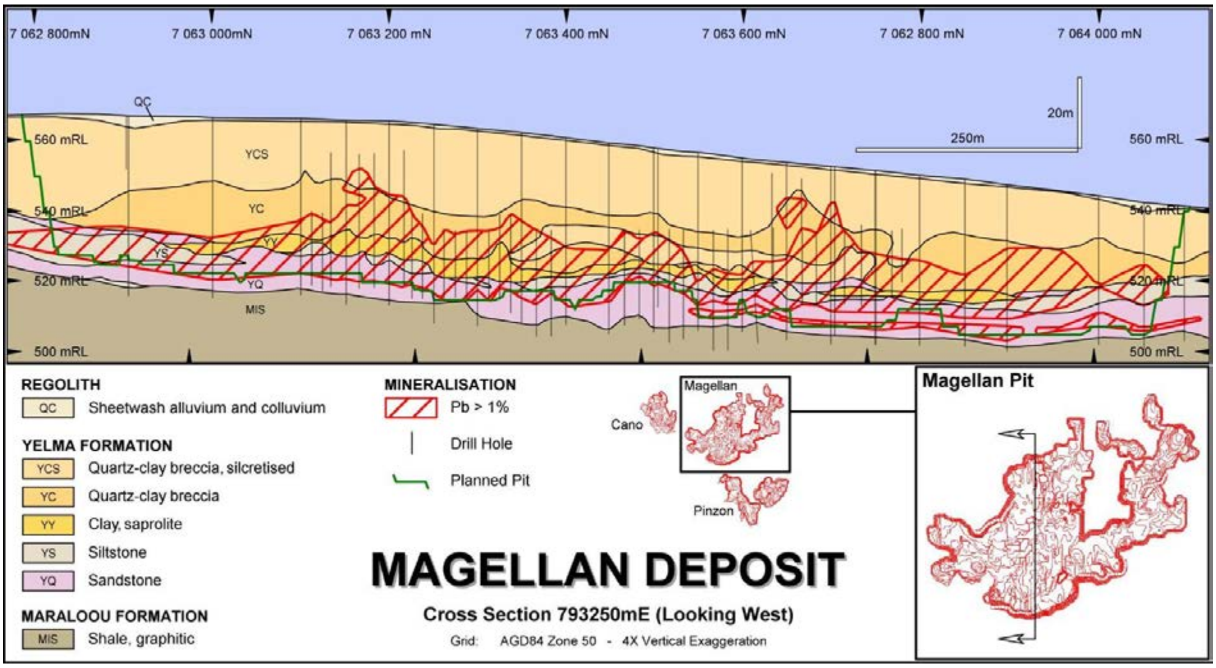

Figure 15: Schematic cross section looking west through Magellan deposit 793250 mE (AGD84). Note: 4 × vertical exaggeration.

Source: RHM (2018).

The Cano deposit lies along a north-west axis, extending for approximately 850 m with an average width of 430 m and an average vertical thickness of approximately 7 m.

The Pinzon deposit comprises two zones of mineralization, one trending in an N–NW direction and the second on a north-east trend (Figure 16). Both zones intersect in a V-shaped body and are approximately 1,000 m long by 200 m wide with an average vertical thickness of 5 m.

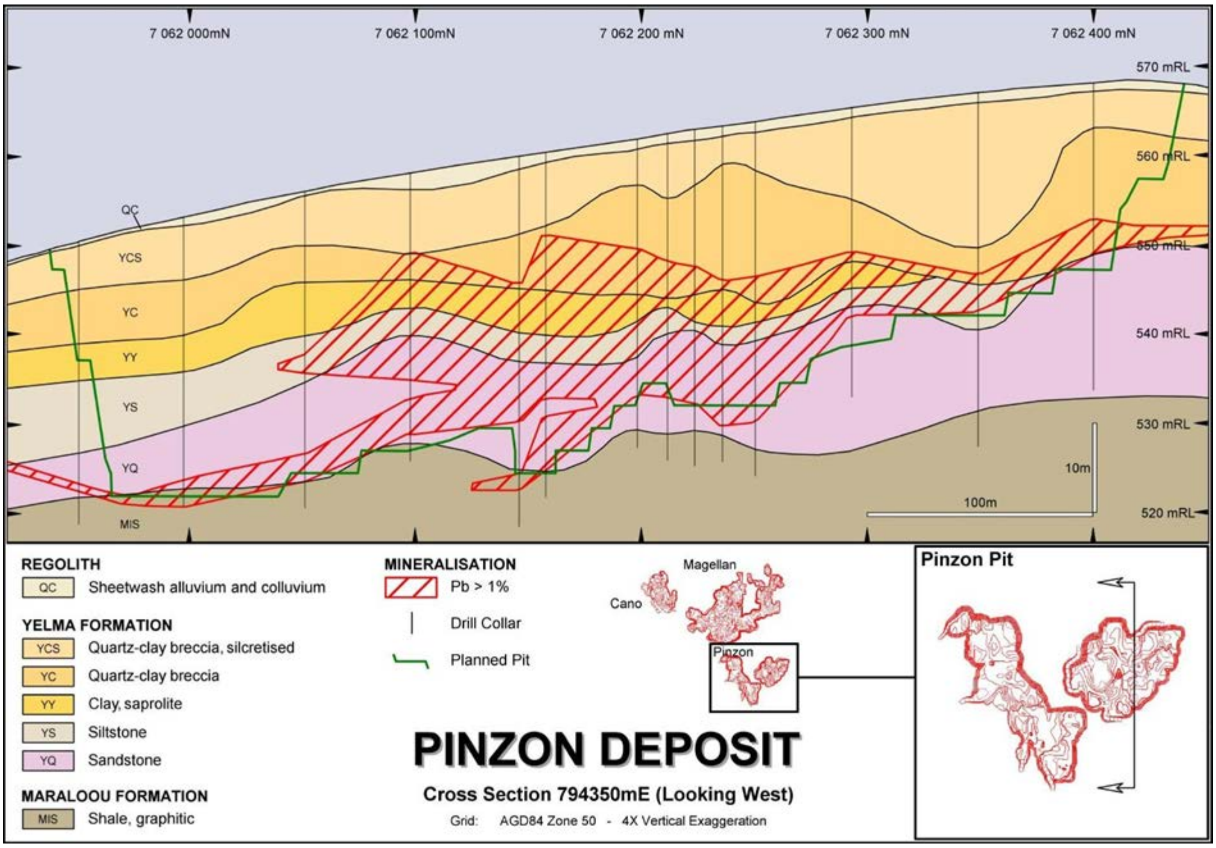

Figure 16: Schematic cross section through the Pinzon deposit 794350 mE (AGD84). Note: 4 × vertical exaggeration.

Source: RHM (2018).

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.

In comparison with the mineralization at Pinzon which displays more variability, mineralization at Magellan and Cano is consistent and continuous, presenting as semi-continuous higher-grade elongate bodies within a lower grade halo.

7.4.2 Finlayson Range Deposits/ Prospects

Mineralization styles at Pizarro and Drake are similar to the Magellan Hill group of deposits described above; however, their stratigraphic position and internal geological relationships show different settings.

Pizarro

The Pizarro lead deposit is located 7.8 km SSW of the Paroo Station Mine, occurring within the Finlayson Range, a prominent east–west trending series of hills comprised of rocks of the Juderina Formation. While small areas of sub-cropping Yelma Formation quartz-clay breccia occur in the Pizarro area, much of the unit is covered by loamy colluvium deposits (Looi, 2010).

The deposit main mineralized trend strikes NNE over a length of approximately 1,950 m, with an average width of 230 m. A secondary trend extending 620 m north-west bisects and offsets the main trend by 270 m in a sinistral fashion. The trends are interpreted to be primary mineralizing structures or fluid pathways, possibly faults related to Yerrida and/ or Earaheedy basinal rifting. A smaller mineralized body, 300 m in diameter, lies 850 m to the north-west of Pizarro. Formerly known as the Columbus prospect, it may be a continuation of the secondary north-west trend, disconnected by groundwater remobilization of the mineralization and partly by erosion. It is now considered part of the Pizarro deposit.

The geological package at Pizarro consists of a footwall Yelma Formation sandstone unit that is overlain/ intercalated with a shallow north-east dipping dolomite unit of approximately 100 m vertical thickness. A narrower sandstone unit sits above the dolomite, 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. The footwall sandstone and dolomite units are 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 the Magellan Hill group.

Like the Magellan Hill group, the Pizarro mineralization is completely oxidized, consisting of cerussite with minor anglesite and pyromorphite.

Pyromorphite (as veins and needle-like clusters) has been identified as an accessory mineral within the lower clastic sequence.

Drake

The Drake lead deposit is located 11 km south-west of the Paroo Station Mine, in the Finlayson Range. The main mineralized trend strikes north-east for approximately 680 m and is 200 m wide. A secondary, diffuse and lower grade lobate trend arcs south-east from the northern limits of the main trends.

At Drake, the oxidized lead mineralization that comprises the deposit is of a secondary nature and forms a supergene blanket which transects all host rock types and is similar to the Cano deposit (FinOre, 2005). The mineralization is hosted within highly weathered remnants of the Yelma Formation and the underlying Juderina Formation sediments.

The mineralization is pervasive and current data does not provide any evidence for lithological control, other than the degree of weathering. The Drake main trend has a strong north-east orientation which shows similar characteristics to the Magellan Hill and Pizarro structural controls (Optiro, 2015). The deposit may also be controlled by the unconformable contact between the Yelma and Juderina formations and this may represent a fluid pathway that is important to the genesis of the Drake deposit.

7.4.3 Ore Mineralogy

The present Paroo Station Mine mineralization is essentially supergene in origin and relatively simple in composition: cerussite ($PbCO_3$) and anglesite ($PbSO_4$) are the dominant minerals, with minor pyromorphite ($Pb_5(PO_4)_3Cl$) and plumbogummite ($PbAl_3(PO_4)(PO_3OH)(OH)_6$) and occasional coronadite ($PbMn_8O_{16}$). Plattnerite (PbO_2) was also recorded in trace amounts (McQuitty and Pascoe, 1998; Pirajno et al., 2010).

The Magellan Hill group of deposits is interpreted to have been formed by the prolonged weathering of a precursor sulfide body. Only one relic hand specimen of galena was found in 2013 at Magellan in the lower quartz-clay breccia horizon, contained in a remnant clast of quartz broken open by blasting. Trace amounts of galena are also recorded in close intergrowths with anglesite and cerussite in flotation plant ore samples.

The supergene lead mineralization is typically very fine grained and indistinctive at mine and hand specimen scale, mainly occurring as disseminations, veinlets and matrix replacements with minor other types including concretionary, nodular and coarse crystalline forms.

The mineralization shows vertical zoning at the Cano and Magellan deposits. Anglesite prevails over cerussite in the quartz-clay breccia in the upper parts of the deposit (Figure 17). The deposits also demonstrate a high degree of supergene lead mobility, resulting in almost complete destruction of the primary lead distribution pattern and development of the supergene-type pattern, consisting of a coupled upper depletion zone and subhorizontal enrichment blanket below (Sergeev et al., 2017).

Looi (2010) notes that manganese-rich coronadite is also more prevalent in the upper parts of the deposits and it is commonly associated with exposed mineralization along the eroded western flanks of the Pinzon deposit. Manganiferous concretion nodules form within fine-grained siltstones at Cano and Magellan and are also present as ‘stringer’ zones throughout the silcretized and lower quartz-clay breccia units.

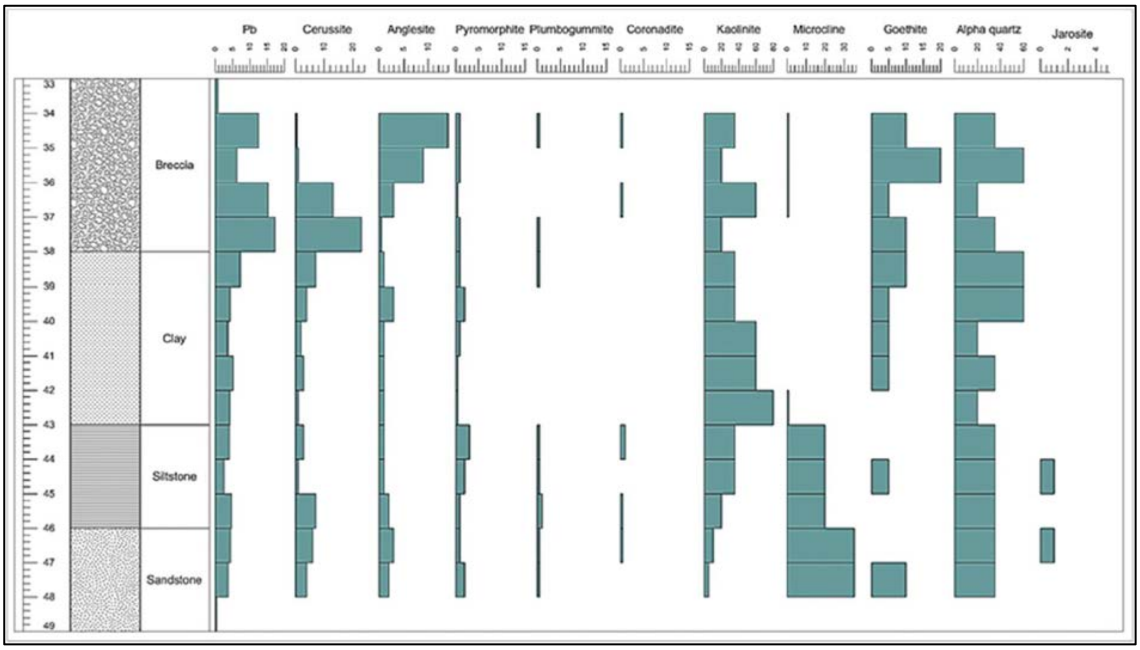

Figure 17: Typical mineral zonation, RC drill hole MMRC582, Magellan deposit

Source: Sergeev et al. (2017).

Minor amounts of sphalerite (ZnS) and galena (PbS) occur in the underlying Maraloou Formation. At Pinzon, supergene mineralization is present in the Maraloou Formation black carbon shales as recorded in several RC drill holes. It occurs as granular cerussite replacing carbonate grains and infilling vughs between laminae.

7.4.4 Grade Distribution

The major host to the mineralization is the lower part of the quartz-clay breccia unit (YC) and underlying 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 based on its physical attributes; overall clay content and lack of texture. However, it is likely to represent the lower part of the YC unit and/ or the top of the strongly weathered siltstone (YS) unit, depending on weathering. The underlying siltstone and sandstone (YQ) sequence can also be a significant host to mineralization, especially when deeply weathered as is the case with the Cano and Pinzon deposits.

Ore zones can have both gradational and sharp grade boundaries. The highest grades are often concentrated in the clay-rich or strongly weathered units, with grades reaching 45% Pb or more in YC and YY of the Magellan orebody.

Mineralization is typically weaker within the upper silcretized breccia and in areas where the effects of weathering are minor and is confirmation of significant supergene processes.

Mineralization in the Maraloou Formation is generally not present in economic grades and the appearance of this unit appears to indicate the effective limits of economic mineralization to the deposits.

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

7.4.5 Genetic Model of Mineralization

The lead deposits of the Magellan Project are thought to have formed due to extensive weathering, volume reduction, and supergene enrichment of primary (sulfide), carbonate-hosted base metal mineralization. The quartz-clay breccia unit that hosts mineralization at the Magellan deposit is interpreted to represent the precursor carbonate horizon (which probably also contained evaporates) that has undergone dissolution, collapse, and silica enrichment. Through supergene enrichment processes, oxide lead minerals have been deposited as matrix replacements in the quartz-clay breccia and into the underlying basal clastic unit (siltstone and sandstone) of the Yelma Formation (Elliott in RHM, 2003).

The stratabound primary sulfide mineralization may have been of low to moderate grade initially, being enriched by a process of remobilization, migration and volume reduction by weathering of the host sequence.

The mineralization occurs within the Sweetwaters Well dolomite and is thought to be similar in style and formed synchronously (1.88–1.80 Ga) with the Mississippi Valley-type (MVT) prospects in the Teague area of the Earaheedy Basin. Deep basinal fluids related to the Capricorn Orogeny brought lead and lesser quantities of other base metals into contact with the reactive dolostone host rocks. Both base metal occurrences are related to the Capricorn Orogeny (Sergeev et al., 2017).

During the long weathering history of the Paroo Station Mine deposits, lead and other base metals gradually moved downwards through a multi-stage remobilization mechanism, re-precipitating at the lowering dolomite weathering front. Lead mobility increased after complete dissolution of the dolomite. Lead migrated further down into the underlying clastic unit and where close to groundwater tables, downslope through permeable layers of weathered sandstones (Sergeev et al., 2017).

Finally, periods of lateritization formed a silica-rich hardcap over the mineralized areas; subsequent erosion brought the deposits close to the surface, with a breakaway forming around the edges of the hill. The current mineralization is sited almost entirely above the current water table, which generally sits close to the unconformity between the Maraloou Formation and the overlying Yelma sediments.

8 Deposit Type

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

8.1 Mineral Deposit

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

The Mine’s lead deposits are almost entirely sulfide free, consisting only of carbonate and oxide lead mineral species, and as such falls into the category of non-sulfide ore systems as defined by Hitzman et al. (2003). Some extremely minor relic sulfide (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-sulfide mineral systems. There is no known analogue of the Mine’s deposits, but they show a strong similarity to non-sulfide zinc deposits, of which there are several examples worldwide (Figure 18).

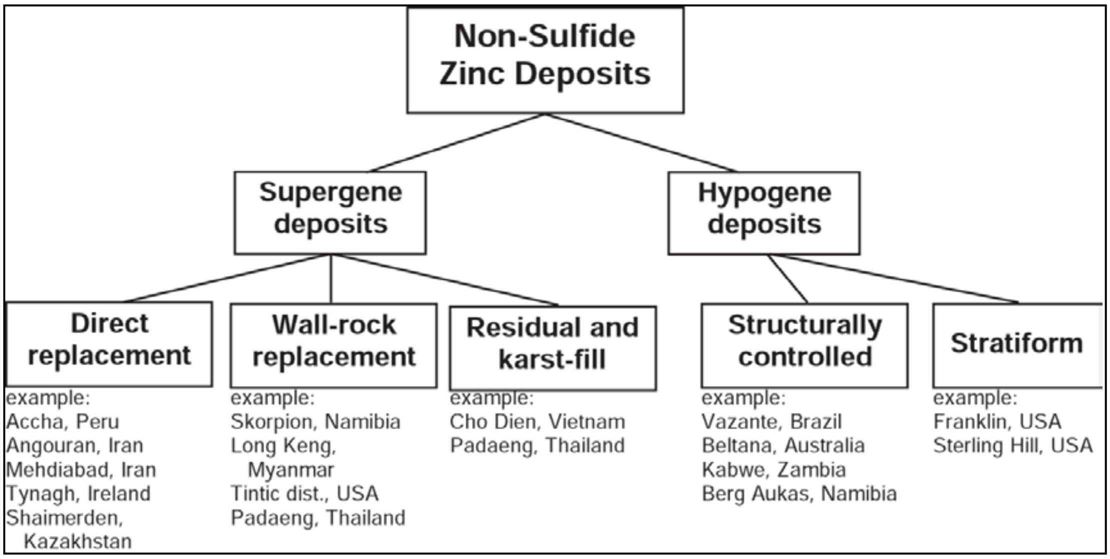

Figure 18: Classification of non-sulfide zinc deposits

Source: Hitzman et al. (2003).

Supergene non-sulfide zinc deposits, which are generated via oxidation of sulfide and non-sulfide zinc deposits, are the most common type of non-sulfide zinc deposits and have a worldwide distribution (Figure 19).

Most supergene non-sulfide 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 are either MVT or have a high-temperature, carbonate replacement-type sulfide progenitor, although supergene deposits may form from a variety of sphalerite-rich deposits. These sulfide progenitors often contain significant quantities of lead in the form of galena lead sulfide (PbS).

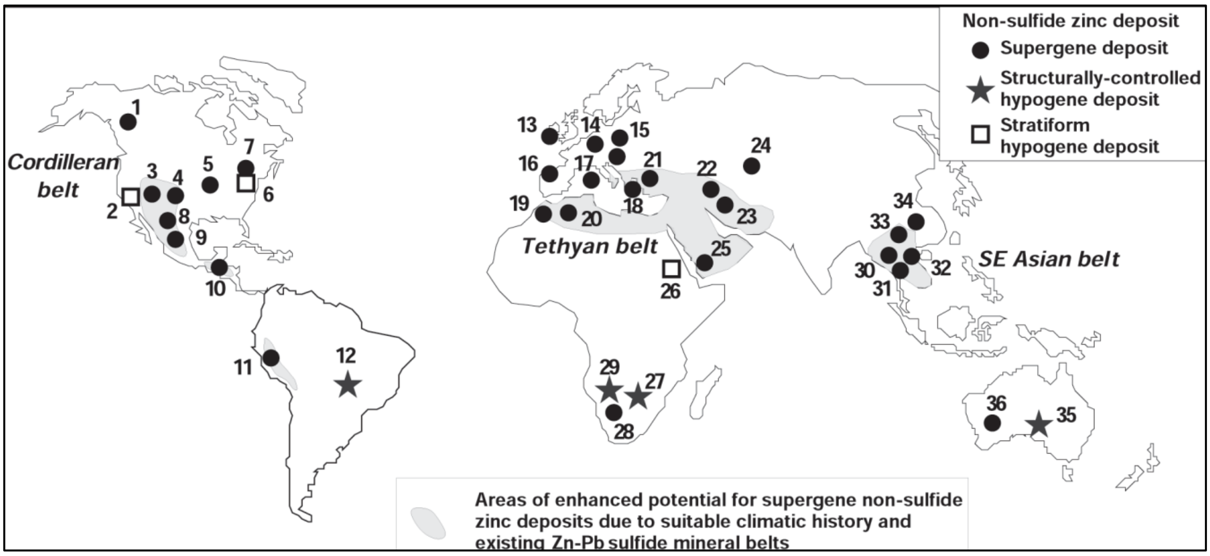

Figure 19: Global distribution of non-sulfide zinc-lead deposits

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 sulfide body and related direct-replacement deposits (Figure 20b) and as sulfide bodies are progressively oxidized, acidic groundwater containing zinc migrates out into the calcareous wallrock where it reacts and precipitates zinc carbonates (Figure 20c).

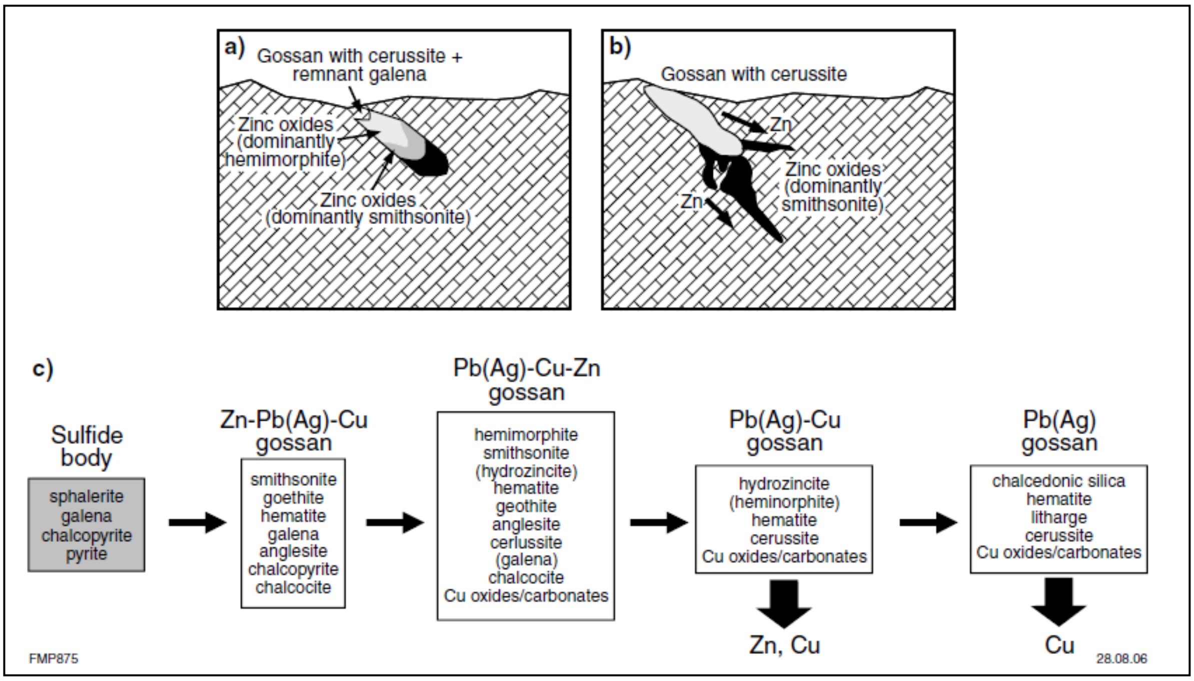

Figure 20: Genetic models for the formation of non-sulfide minerals systems

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

Notes:
a) Direct replacement type
b) Wallrock replacement type (applicable to the Mine)
c) Mineralogical changes related to progressive replacement of sulfides.

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 mineralization 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 mineralization to a more cerussite-dominant assemblage, assisting the remobilization of upper mineralization downwards towards the favourable clay-rich portions of the quartz-clay breccia and clay units while depleting the upper, silcretized breccia unit.

No weathered, altered sulfide or relic sulfide 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 mineralized horizon. The sulfide, preserved with a rind of carbonate inside a crystalline and chalcedonic quartz vugh immediately proved the presence of at least small quantities of primary sulfide mineralization. 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 sulfide 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 mineralization (McQuitty and Pascoe, 1998).

The Magellan Hill and outlying lead deposits display a characteristic pattern of lead-in-soil anomalism around marginal breakaway slopes where the hardcap has eroded and portions of the mineralized zone are exposed. Apart from these local situations, the ore grade mineralization is 'blind', with limited surface physical or geochemical expression. Gravity survey data shows a weak correlation between mineralization and local gravity lows from a likely mass removal event during brecciation of the mineralized 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% Pb and 3.15% Pb from holes drilled at the south-western 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 Gold Consolidated (RGC) and CSA Global on behalf of Magellan Metals Pty Ltd, a campaign using field portable XRF (FP-XRF) mineral analyzer 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 (current as at 2009). In addition, most of the known outlying lead deposits have been surveyed. The following summarizes 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[1] (Cortez prospect and North Pizarro area): 610 stations
- E53/1560 (11.5 km south-east of Magellan Hill): 1,007 stations

In 2014, all conventional and portable XRF data was merged with surficial RC drilling to produce a high-quality combined dataset (Figure 12).

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 lead geochemical anomaly and correspond closely with observed anomalous vegetation.

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

The lead anomalism displays a strong north-west 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 downslope from the mesa's south and western breakaway into the broad West Creek drainage channel can be observed. Several subordinate ENE–NE alignments also exist and preferential erosion of susceptible strata may be related to structural trends.

The magnitude of the lead anomaly is greatest where mineralization 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 lead-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 m north (mN) × 50 m east (mE) over the Magellan deposit, to 50 mN × 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 21. 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 mineralization.

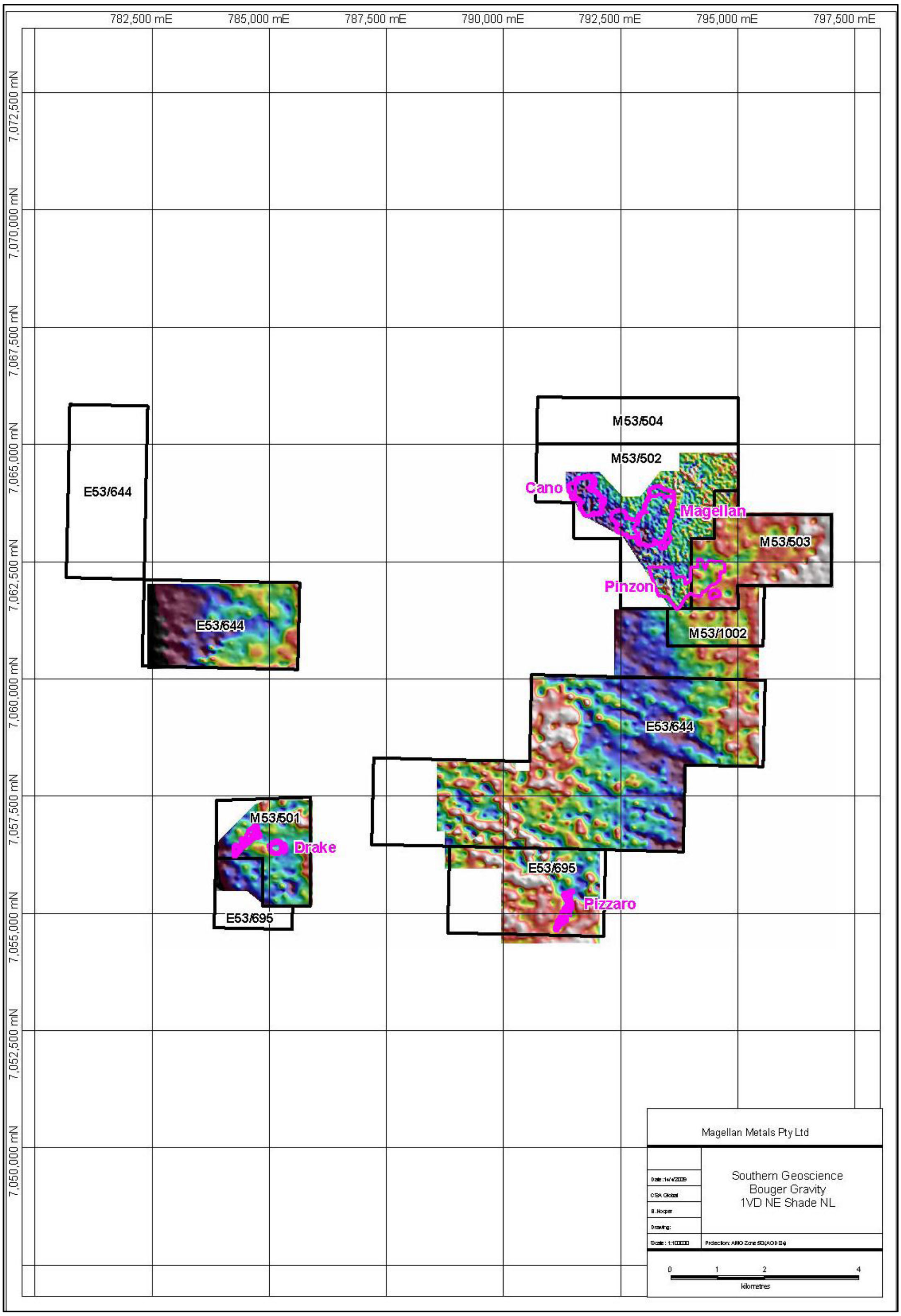

Figure 21: Bouguer anomaly first vertical derivative from merged gravity data; levelled and processed with outlined major lead deposits

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 GIS (Global Information System) 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 22.

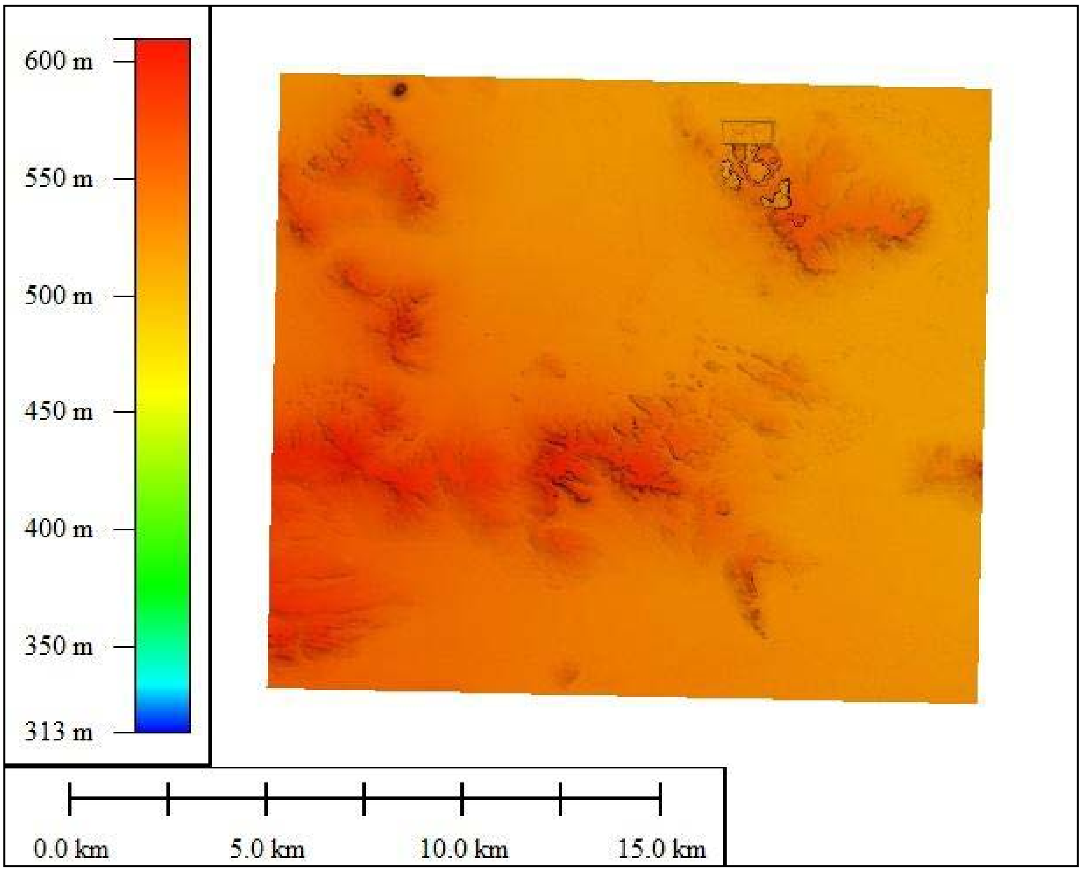

Figure 22: Digital terrain model produced from 2014 aerial photography/ altitudinal data

Source: Fugro (2014).

9.1.4 Aerial Time Domain Electromagnetic Survey

In September 2014, GPX Surveys Pty Ltd (GPX) performed an 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 23).

The data acquisition equipment comprised an XTEM time domain airborne electromagnetic survey system. The 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 center 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 45–50 knots and the along-line sample interval is between 2 m and 5 m.

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

Figure 23: Photo of aerial XTEM survey equipment

Source: GPX (2014).

A preliminary processed image is shown as Figure 24.

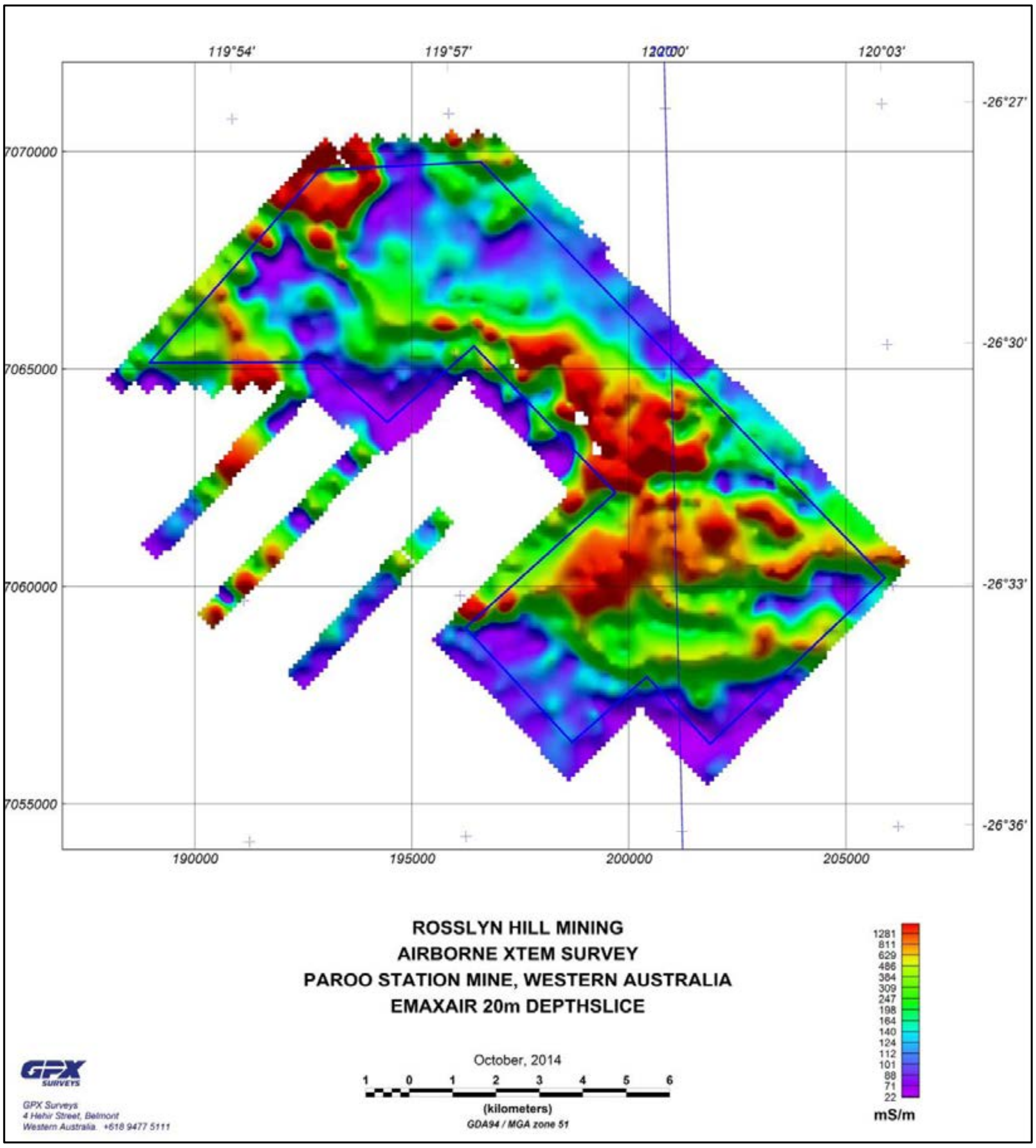

Figure 24: Preliminary airborne XTEM survey results (October 2014)

Source: GPX (2015).

9.2 Significant Results and Interpretation

All non-drilling forms of exploration have contributed directly to the targeting of additional mineralization, 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 × 50 m to a staggered 50 × 100 m.

Grade control drilling as part of mining operations of the Magellan and Cano has infilled the exploration drilling data to 12.5 × 12.5 m and 16.7 × 16.7 m patterns since the commencement of mining in 2005.

Table 15 summarises the RHM drill hole database by drill method as at February 15, 2019.

Table 15: Drill hole database summary

Drilling type	Number of holes	Total meters
Air core (AC)	43	1,305
Rotary air blast (RAB)	1,318	30,868
Reverse circulation (RC)	4,623	142,653
Diamond drill (Core)	92	5,351

Source: RHM (2018).

10.2 Drilling – 2018

Between April, 28 and May, 2, 2018, an exploration RC drilling program was completed across exploration areas at the Drake lead deposit (M53/501). Drilling was also completed at the East Cortez (E53/644), South Pizarro (E53/1528, P53/1528) and Bubble Well (E53/1560) prospects. In all, 25 RC drill holes were completed for a total of 924 m. Table 16 outlines the drilling programs completed.

All drilling prior to the 2018 RC program have been fully disclosed in the previous Technical Report (SRK, 2018).

Table 16: Recent drilling programs

Program	Year	Number of holes	Total meters
Exploration drilling RC (Drake)	2018	10	324
Exploration drilling RC (East Cortez)	2018	5	200
Exploration drilling RC (South Pizarro)	2018	6	240
Exploration drilling (Bubble Well)	2018	4	160
Total		**25**	**924**

Source: RHM (2018).

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

Figure 25, Figure 26 and Figure 27 depict the location of the holes drilled during 2018; details of each program are outlined below.

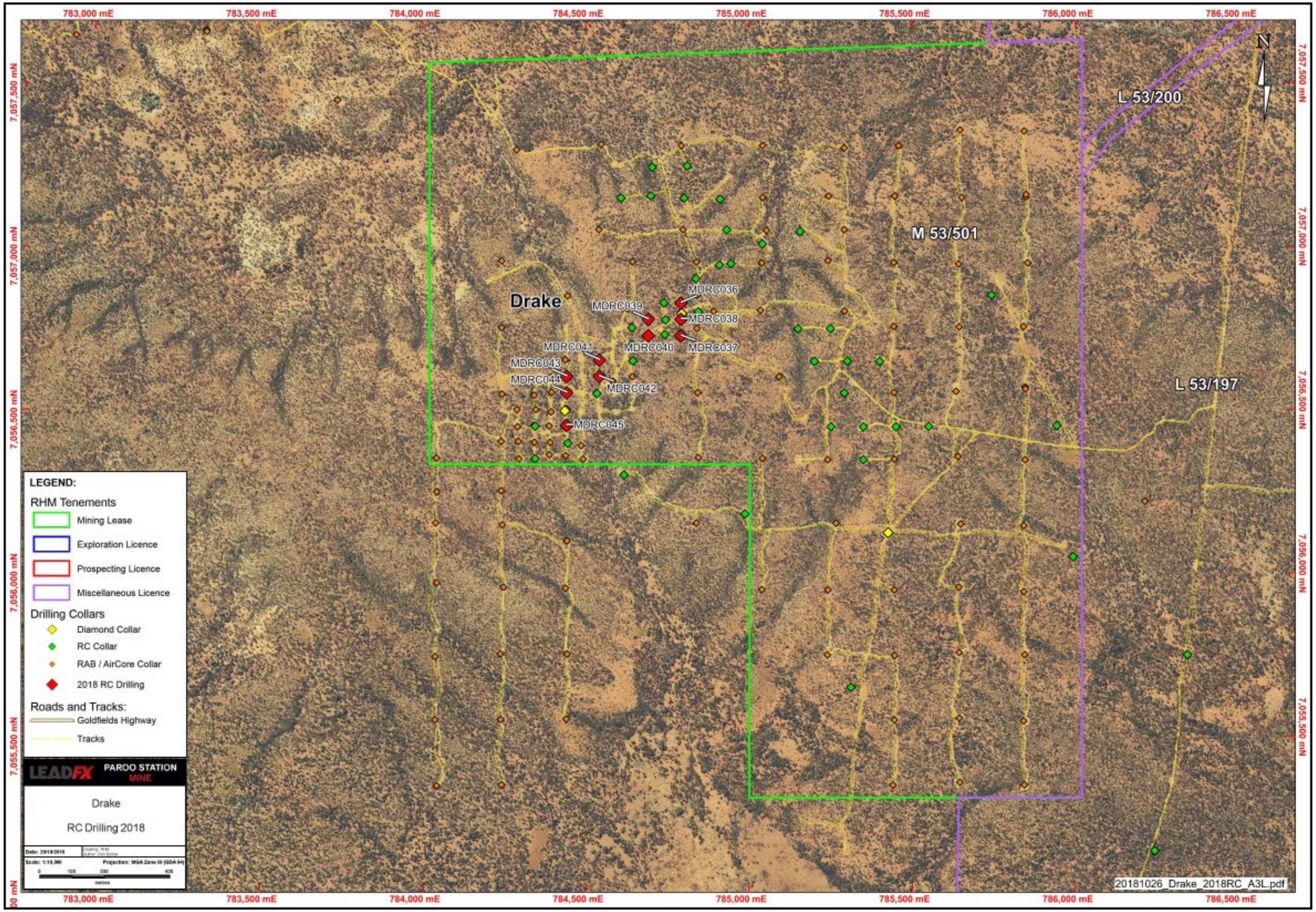

Figure 25: Location map of drill hole collars – Drake 2018 RC drilling program

Source: RHM (2019).

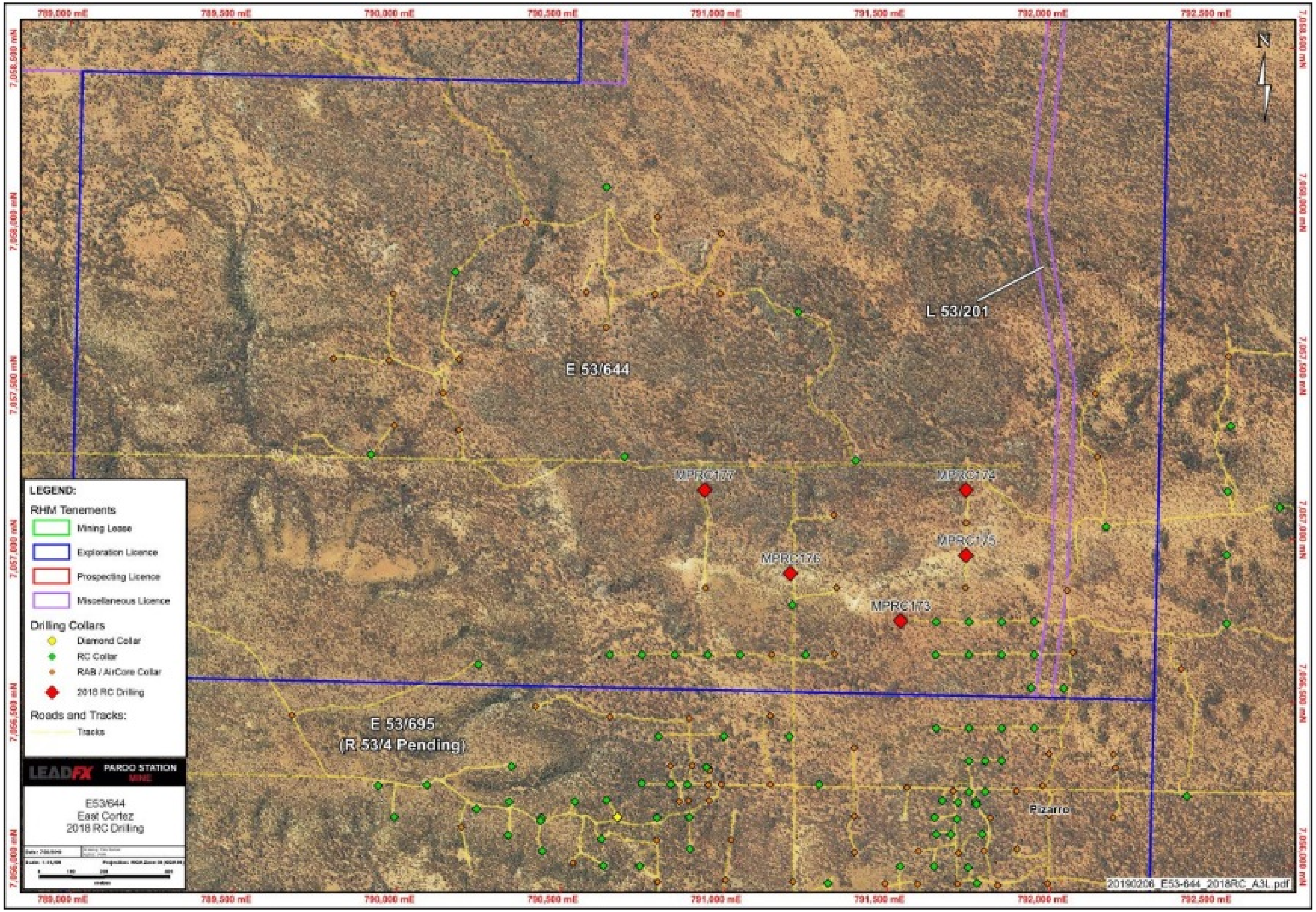

Figure 26: Location map of drill hole collars – East Cortez 2018 RC drilling program

Source: RHM (2019).

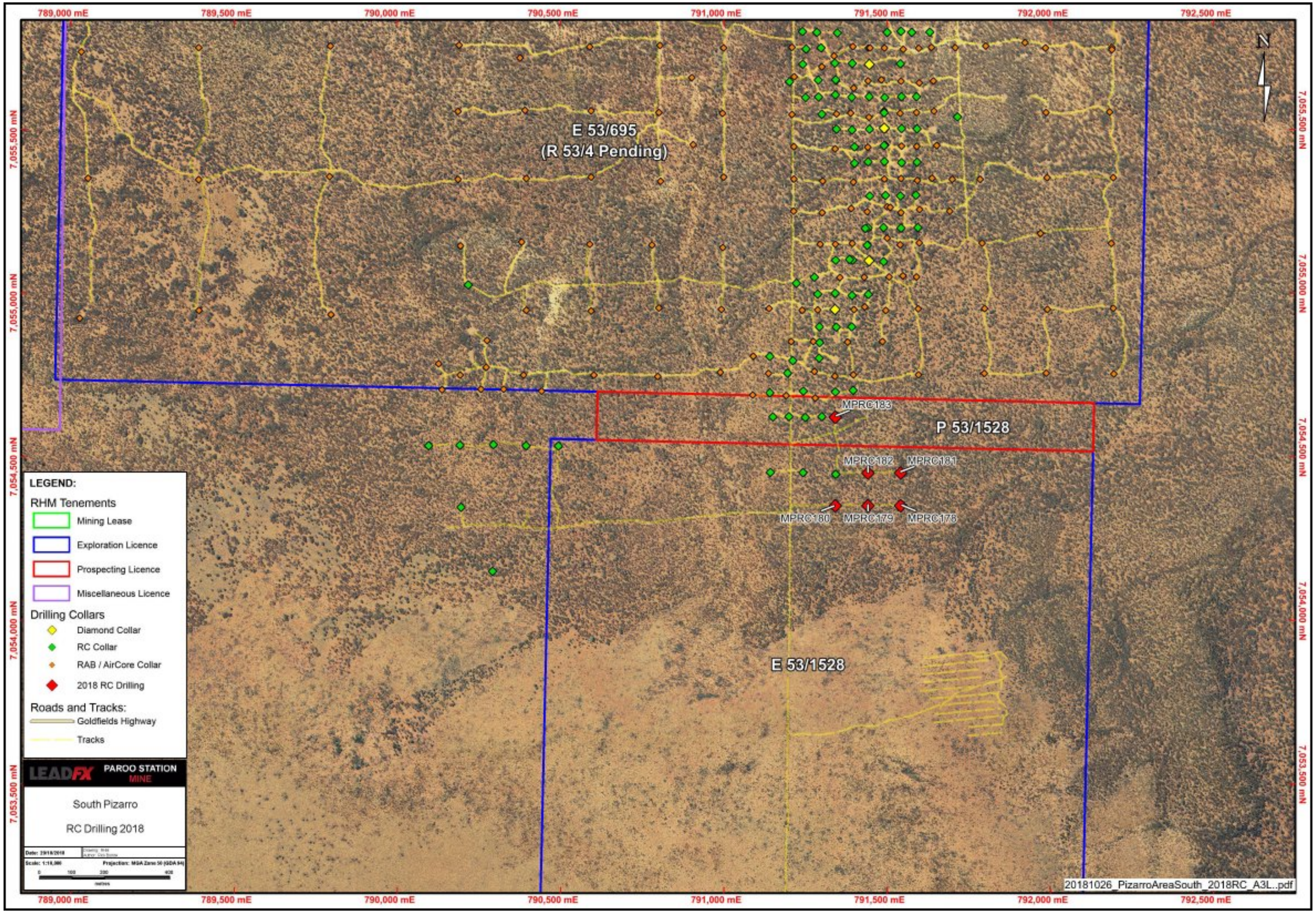

Figure 27: Location map of drill hole collars – South Pizarro RC drilling program

Source: RHM (2019).

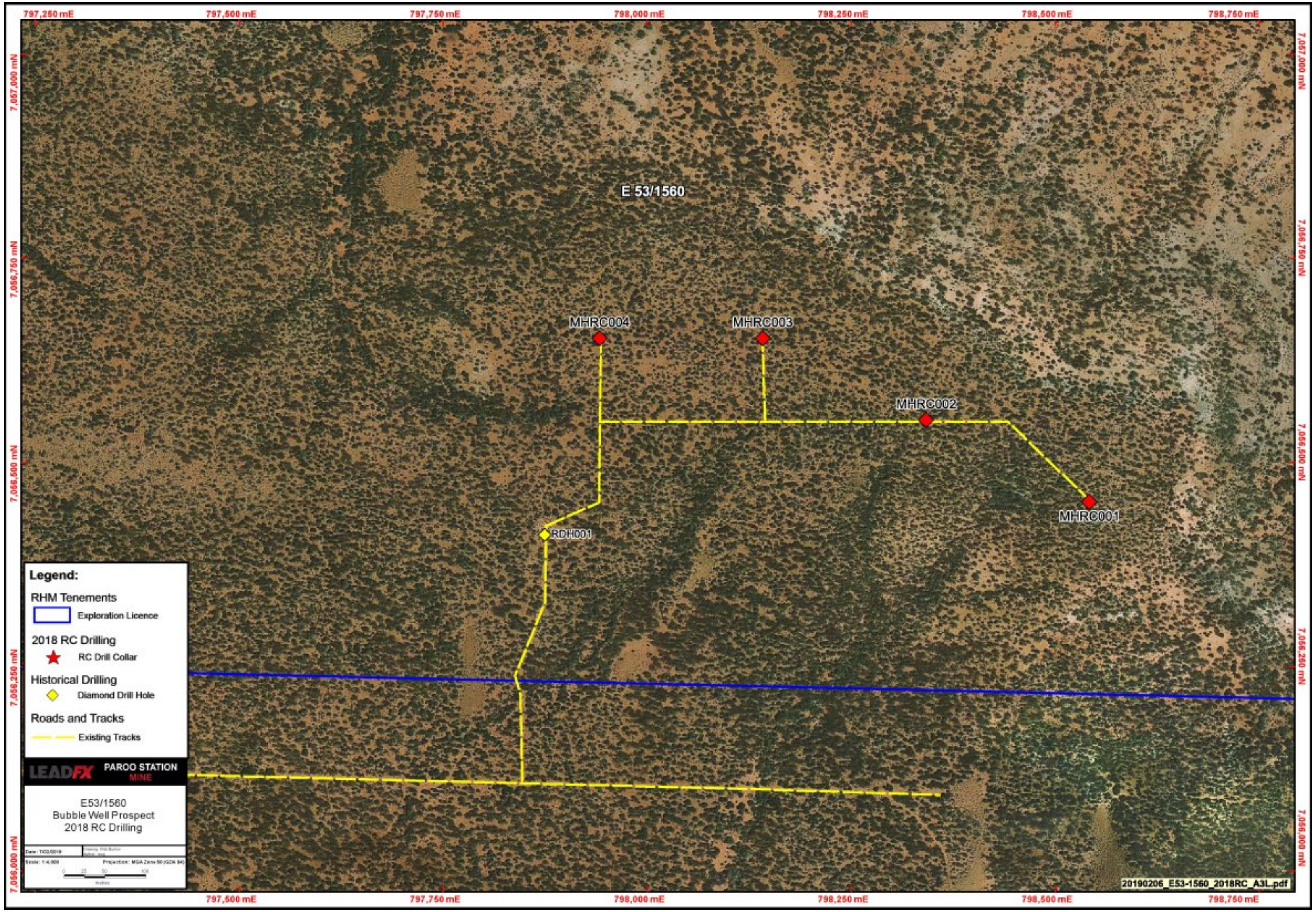

Figure 28: Location map of drill hole collars – Bubble Well RC drilling program

Source: RHM (2019).

10.2.1 RC Exploration Drilling (Drake)

A program of 10 RC drill holes were completed at the Drake deposit (M53/501) for a total of 324m. Drilling twinned two pre-existing RAB holes and re-drilled two incomplete RC drill holes. The remainder infilled the Drake main trend on a 50 m resource drill spacing.

Drilling was notionally separated into a north eastern group of 5 holes (MDRC036-040) and a south western group of 5 holes (MDRC041-045).

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 (East Cortez)

At E53/644, five RC holes tested the broad east–west trending PZ4 geochemical anomaly located north of the Pizarro lead deposit. A total of 200 m was drilled.

All holes were logged in their entirety using the standard Rosslyn Hill 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 RC Exploration Drilling (South Pizarro)

A total of six RC holes were completed at South Pizarro (P53/1528 and E53/1528) for a total of 240 m. The drill holes investigated a possible change in the Pizarro trend direction from south-west towards the south-east indicated by earlier drilling.

10.2.4 RC Exploration Drilling (Bubble Well)

At Bubble Well (E53/1560), four RC holes were completed for a total of 240 m. The drilling targeted a small outlier hill south-east of the Paroo Station Mine which had an untouched volume of prospective Yelma Formation rocks. A weak secondary dispersion lead-in-soil anomaly was present along the northern breakaway slope, indicating mineralization may be present within the mesa.

10.3 Procedures

All 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.

Historical procedures, results or analyses that differed from current practice described in the 2015 Technical Report (SRK, 2015) 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 real-time kinetic 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 mineralization. As there are vertical drill holes, there is no requirement to carry out downhole surveys on the completed hole.

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

10.3.2 Sample Collection – RC Drilling

The 2018 RC drilling was completed by Blue Spec Drilling using a Schramm-style multi-purpose RC/ diamond drilling rig with remote rod-handler. The on-board air compressor was boosted with a truck-mounted auxiliary compressor, with the hole being drilled using 3½ inch rods with a 4¾ inch (122 mm) downhole hammer. The RC sample was then returned via a cyclone and pneumatic sample drop door to an adjustable cone splitter.

Primary RC samples were collected at 1 m intervals based on 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 minimize cross-contamination between samples. The shutter was reopened once the previous meter's sample bag was removed and the next sample bag was in place. The shutter opened to a cone splitter, which split a ⅛ subsample in to a calico bag, and the remaining sample into a large plastic bag.

The indistinct nature of the cerussite and anglesite mineralization makes the visual differentiation between mineralized and unmineralized 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 chips to fines, preventing the spear from reaching the bottom of the bag, the bag was angled to the side in order to get a more 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 17 shows sample details and submitted QA/QC sampling for the 2018 RC drilling programs.

Table 17: Sample details for 2018 RC drilling program

Prospect	Tenement	Drill hole ID	Total hole depth (m)	Laboratory assays	QA/QC assays
Drake	M53/501	MDRC036	30	16	2
		MDRC037	35	11	2
		MDRC038	35	28	2
		MDRC039	30	10	2
		MDRC040	35	14	2
		MDRC041	35	14	2
		MDRC042	35	15	2
		MDRC043	32	21	2
		MDRC044	32	23	2
		MDRC045	25	23	2

Prospect	Tenement	Drill hole ID	Total hole depth (m)	Laboratory assays	QA/QC assays
East Cortez	E53/644	MPRC173	40	21	2
		MPRC174	40	16	2
		MPRC175	40	5	0
		MPRC176	40	16	2
		MPRC177	40	15	2
South Pizarro	E53/1528	MPRC178	40	14	2
		MPRC179	40	12	2
		MPRC180	40	23	2
		MPRC181	40	6	0
		MPRC182	40	15	2
	P53/1528	MPRC183	40	25	2
Bubble Well	E53/1560	MHRC001	40	12	0
		MHRC002	40	12	2
		MHRC003	40	10	0
		MHRC004	40	12	2
Total		**25**	**924**	**389**	**50**

Source: RHM (2018).

10.3.3 Sample Collection – Diamond Drilling

No additional diamond drilling for exploration has been undertaken subsequent to the 2017 Technical Report.

As discussed in the 2017 Technical Report, diamond drilling for metallurgical testwork was conducted in 2017. These 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 maximize core recovery, PQ (83 mm) diameter triple tube was selected. The target depths were taken from the identified mineralization 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 or 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 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 as follows:
  - Interval lengths (drill runs) were taken from driller’s core blocks.
  - The amount of core physically recovered for each interval was measured and recorded.
  - Total core recovery was then calculated as a percentage (recovery/run length × 100).
  - The sum total amount of core >10 cm per drill run was measured and recorded.
  - Rock quality designation (RQD) was calculated as a percentage (core >10 cm/run length × 100).

- Fracture frequency was counted as the number of joints per meter. Where fracture frequency was >20, the count was estimated as a percentage of broken core relative to competent core over that meter (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 1 m 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 color, grain size, major and minor lithological unit, alteration type and intensity, weathering and comments.

Mineralized 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 meter (or geological interval where <1 m) from surface to approximately 5 m above known mineralization identified in the twin RC drill hole assay results. Portable XRF readings were taken (three to four for each 1 m, or geological interval where <1 m) from approximately 5 m up-hole to approximately 5 m down-hole of known mineralization.

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 grade). The portable XRF assay results were used in conjunction with geological interval, alteration logs and RC twin hole assay data to assign a mineralized 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.

10.4 Interpretation and Relevant Results

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

Significant intersections recorded by the Drake and South Pizarro RC drilling programs are shown in Table 18.

Table 18: Intersections recorded for 2018 RC drilling programs

Prospect	Tenement	Drill hole ID	Intersections
Drake	M53/501	MDRC036	5 m at a grade of 5.47% Pb from 19 m
		MDRC037	3 m at a grade of 5.92% Pb from 20 m
		MDRC038	23 m at a grade of 11.63% Pb from 9 m
		MDRC039	7 m at a grade of 2.23% Pb from 15 m
		MDRC040	3 m at a grade of 4.29% Pb from 20 m
		MDRC041	3 m at a grade of 7.3% Pb from 20 m
		MDRC042	1 m at a grade of 1.59% Pb from 27 m
		MDRC043	1 m at a grade of 1.36% Pb from 13 m
		MDRC044	10 m at a grade of 4.46% Pb from 21 m
		MDRC045	11 m at a grade of 6.79% Pb from 8 m

Prospect	Tenement	Drill hole ID	Intersections
East Cortez	E53/644	MPRC173	3 m at a grade of 1.65% Pb from 12 m
		MPRC174	No significant intersection
		MPRC175	No significant intersection
		MPRC176	1 m at a grade of 0.85%Pb from 12 m
		MPRC177	No significant intersection
South Pizarro	E53/1528	MPRC178	No significant intersection
		MPRC179	No significant intersection
		MPRC180	3 m at a grade of 3.03% Pb from 16 m
		MPRC181	No significant Intersection
		MPRC182	No significant intersection
	P53/1528	MPRC183	1 m at a grade of 1.8% Pb from 26 m
Humboldt	E53/1560	MHRC001	No significant intersection
		MHRC002	No significant intersection
		MHRC003	No significant intersection
		MHRC004	No significant intersection

Source: RHM (2019).

Geology logs and assay results were reviewed. A low-grade intersection in easternmost hole, MPRC164, recorded 3 m at a grade of 2.24% Pb and 5 m at a grade of 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 mineralized 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 mineralization, but the extensions to date are narrower and/or at a lower grade than the previously identified mineralization.

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

10.4.1 Drake (M53/501)

Mineralization along the Drake main NE–SW trend was extended to the north and south of diamond hole MMDD023. MDRC038 encountered heavy mineralization, returning 23 m at a grade of 11.63% Pb from 9 m, showing strong short-range continuity of the high-grade intersection seen in the diamond hole 18 m to the north.

The trend remains open to further extension north of MDRC036 (5 m at a grade of 5.47% Pb from 19 m) and south of MDRC037 (3 m at a grade of 5.92% Pb from 20 m).

MDRC039 and MDRC040 showed weaker mineralization on the 784550 mE line. MDRC039 reported 7 m at a grade of 2.23% Pb from 15 m and MDRC040 reported 3 m at a grade of 4.29% Pb from 20 m; both holes indicate the Drake main trend has good mineralization continuity over the 50 m line spacing. The trend remains open to the north and south of the two holes.

The second group of Drake drill holes continued work at the south-western end of the main Drake trend.

Two RC holes re-drilled earlier incomplete or abandoned RC holes. Re-drill MDRC041 returned assays of 3 m at a grade of 7.3% Pb from 20 m, where MDRC007 had failed to penetrate past 13 m depth. The result improves the central part of the Drake trend; however, infill drill hole MDRC042 located 50 m to the south returned 1 m at a grade of 1.59% Pb from 27 m. Further RC infill drilling is required on the 384750 mE line.

Re-drill MDRC043 reported 1 m at a grade of 1.36% Pb from 13 m and did not improve the MDRC005 intersection which was incomplete at the end of hole at a grade of 2.92% Pb.

Re-drills of previous RAB holes MDRH024 and MDRH030 provided a partial twin test of the older RAB drilling intersections. MDRC044 returned a near-perfect repeat of 10 m at a grade of 4.46% Pb from 21 m, confirming the earlier MDRH024 RAB intersection of 9 m at a grade of 3.89% Pb from 21 m. MDRC045 reported 11 m at a grade of 6.79% Pb from 8 m, reinforcing MDRH030's weaker RAB intersection of 7 m at a grade of 3.41% Pb from 14 m. This result highlights the common high variation at very short 5–10 m range within the Drake mineralization, in parallel to similar short-range variability seen in the Magellan Hill group of deposits.

A comparison between the drilling results and current block model prediction is presented in Table 19. The Drake Mineral Resource is classified as an Inferred Mineral Resource at best, which is reflected in the variance between the actual drilling and the current Mineral Resource.

Table 19: Drake actual drilling and model predicted intersections

Hole ID	2018 drilling			Current Mineral Resource		
	From (m)	Length (m)	Lead grade (%)	From (m)	Length (m)	Lead grade (%)
MDRC036	19	5.0	5.5	15	7.0	5.4
MDRC037	20	3.0	5.9	No intersection predicted		
MDRC038	9	23.0	11.6	15	17.0	4.1
MDRC039	15	7.0	2.2	15	4.0	3.6
MDRC040	20	3.0	4.3	15	11.0	4.9
MDRC041	20	3.0	7.3	22	1.0	5.4
MDRC042	27	1.0	1.6	22	1.0	5.4
MDRC043	13	1.0	1.4	17	4.0	3.2
MDRC044	21	10.0	4.5	20	5.0	6.9
MDRC045	8	11.0	6.8	14	6.0	3.2

The 2018 RC drill program at Drake improves local knowledge for the Drake lead resource. The Drake resource model has not been updated due to the limited amount of drilling. An update will be considered in the future.

10.4.2 East Cortez (E53/644)

Assays show thin, shallow mineralization that explains the observed soil geochem anomaly. Best results are from MPRC173 (3 m at a grade of 1.65% Pb from 12 m) and MPRC176 (1 m at a grade of 0.85% Pb from 12 m). The new drill data agrees with other scattered drilling in the area and indicates broad sub-cropping lead-mineralized sediments that have been interpreted as a lead-scavenging horizon, possibly related to groundwater levels.

10.4.3 South Pizarro (E53/1528, P53/1528)

At South Pizarro, limited ore grade mineralization was intercepted, but no appreciable shift in the trend direction was detected.

On P53/1528, MPRC183 reported 1 m at a grade of 1.8% Pb from 26 m. Mineralization is still open to the south-west.

On E53/1528, the encouraging result from MPRC180 (3 m at a grade of 3.03% Pb from 16 m) shows potential for ore grade mineralization far south of the main Pizarro trend. The 2015 block model did not predict mineralization in this drill hole and the intercept shows the potential that the trend is still open to the south and west.

10.4.4 Bubble Well (E53/1560)

The drilling targeted a small outlier hill south-east of the Paroo Station Mine which had a largely untested volume of prospective Yelma Formation rocks. A weak secondary dispersion lead-in-soil anomaly was present along the northern breakaway slope, implying mineralization may be present within the mesa.

All four RC holes were drilled to their target depth of 40 m without issue. However, the drilling did not intersect mineralization within the targeted outlier mesa with no ore grade or other significant intersections noted.

11 Sample Preparation, Analysis and Security

Details of the sample preparation, analysis and security for drilling prior to 2015 have been disclosed in SRK's 2015 report. The sample preparation, analysis and security details for drilling between 2015 and 2017 have been disclosed in SRK's 2018 report.

11.1 Security Measures

All paperwork associated with the sample dispatch for the 2018 RC and diamond drilling campaign was prepared by the supervising geologist for each program. This involved compiling a sample list for each submission and conducting a visual check 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 1 t capacity polyweave 'bulka' bags which were closed with cable-ties and readied for dispatch. For diamond core, the core was photographed wet and dry in the tray; this is done for geological record but also records the core in a 'before shipping' state. Each core tray is stacked in sequence on pallets before secure wrapping with shipping plastic to prevent any loss or tray movement. The 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 (four drums per pallet) for dispatch.

Samples were delivered by road freight trucks from the Paroo Station Mine directly to the laboratory for processing. The RC samples were delivered to Intertek Genalysis Laboratories (Genalysis) in Perth for sample preparation and subsequent assay. Diamond core and bulk ore samples were delivered to Australian Laboratory Services (ALS) in Perth for processing and testwork for the DFS.

As part of the chain of custody for each sample dispatch, the assay laboratory was sent hardcopies as well as digital copies of the sample submission paperwork containing 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 laboratory is 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 resolved due to the nature of the records kept and the processes and procedures adopted and implemented. Additional samples are added to the submission lists and missing samples are sent in later submissions if required.

11.2 Sample Preparation for Analysis

All sample preparation and analyses for RC drilling programs conducted in 2018 (discussed in Section 10) were carried out at Genalysis in Maddington, Western Australia. All diamond core and bulk metallurgical samples were received, prepared and assayed/ tested at ALS in Balcatta, Western Australia. These laboratories have been certified in accordance with ISO/IEC 17025, as follows:

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

No aspect of sample preparation at Genalysis was conducted by an employee, officer, director or associate of RHM, Ivernia or LeadFx Incorporated.

The RC samples were received by the laboratory, sorted, checked and the delivered samples confirmed. The samples were then dried ready for pulverization.

Large samples were split down to a nominal 1.2 kg or 2 kg size and pulverized using a robotic pulverizer via the laboratory's sample preparation code:

- SP11 (dry crush ~10 mm pulverize up to 300 g)
- SP22 (dry crush ~2 mm, robotic preparation, pulverize 300 g up to 1,2 kg) depending on sample mass, SP23 (dry crush up to 3 kg, split, robotic preparation 1,2 kg) or SP24 (if >3,000 g, dry crush, ~2 mm, split, robotic preparation 1,2 kg).

Owing to the toxicity of the carbonate lead content, the RC samples were prepared in Genalysis' hazardous sample preparation area.

Details of the diamond core sample and bulk sample preparation for DFS metallurgical testwork are included in Section 3.4.2 of the InCor DFS documentation and ALS Report A18236, supplied as part of the DFS documentation.

11.3 Sample Analysis

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

After 2013, assaying finishes were performed by inductively-coupled plasma–optical emission spectrometry (ICP-OES) to accommodate multi-elemental data. For drilling conducted during 2018, the primary, field duplicate and blank RC samples were analyzed for aluminum, iron, lead, phosphorus and sulfur using an ore grade 4-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 diamond core sample and bulk sample analysis for DFS metallurgical testwork are discussed in the DFS documentation.

11.4 Quality Assurance/Quality Control 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 demonstrates sufficient sample and analytical accuracy and precision to support estimation of a Mineral Resource. A QA/QC review was conducted specifically for the 2018 RC drilling program and subsequent analyses undertaken by Genalysis.

For the 2018 RC drilling at Drake and Pizarro, a total of 432 samples were assayed, including 382 primary 1 m samples and 50 QA/QC samples consisting of 21 field duplicate samples, 21 blank samples and 8 standards. The laboratory carried out a further 15 internal pulp checks.

No QA/QC samples were collected for the 2017 diamond drilling program, as the purpose of drilling these cores was to obtain sample material for metallurgical testwork and not Mineral Resource delineation. The QA/QC protocols associated with the DFS testwork are discussed within the DFS documentation.

The challenging drilling conditions at Magellan Hill results in varied RC sample recovery, 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 improve sample recovery. In isolation, these tests would not remove the concern of bias. However, in combination, the 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 2018 RC drilling program are discussed below.

11.4.1 Standards

RHM submitted project-specific reference materials as well as Geostats Pty Ltd (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 mineralization. Geostats certified reference materials are sourced from various oxide and sulfide mineralization and are not matrix matched.

A total of four in-house standards and four Geostats standards were submitted for the 2018 RC drilling program and were inserted at a combined ratio of approximately one standard to 48 primary samples (Table 20).

Of the four Geostats standards, three were less than 2 standard deviations from the expected mean values and one was between 2 and 3 standard deviations.

Of the four in-house standards, two of the standards were less than 2 standard deviations, one was between 2 and 3 standard deviations and one was greater than 3 standard deviations.

Table 20: Genalysis assays of in-house and Geostats standards

Standard Type	Sample ID	Standard	Assay Pb (ppm)	Expected Pb (ppm)	Comment
Geostats	QS000597	GBM302-10	6.01	5.59	Between 2 and 3 std.dev.
	QS000598	GBM903-13	2.29	2.15	Less than 2 std.dev.
	QS000599	GBM398-4	1.19	1.17	Less than 2 std.dev.
	QS000600	GBM398-1	2.76	2.67	Less than 2 std.dev.
In-house	MQS0582	Mag-06	2.17	1.79	Greater than 3 std.dev.
	MQS0583	Mag-10	6.31	5.56	Between 2 and 3 std.dev.
	MQS0584	Mag-16	0.63	0.54	Less than 2 std.dev.
	MQS0585	Mag-01	0.14	0.12	Less than 2 std.dev.

Source: RHM (2019).

The standards submitted performed reasonably with the exception of Mag-06 which exceeded 3 standard deviations from the expected mean. The poor performance of Mag-06 has been previously noted by the QP in the 2015 QA/QC review. The poor performance is believed to reflect a bias in the original 20-assay certification process that was used to establish the expected average and variance for Mag-06, rather than an analytical accuracy problem.

In addition to the RHM submitted standards, Genalysis analyzed a range of internal and certified reference material standards as part of its internal checks.

The performance of the standards demonstrates that that the analytical process is largely in control and an appropriate degree of analytical accuracy is being achieved.

11.4.2 Blanks

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

A total of 21 blank samples were submitted during 2018, seven of which failed to meet the acceptance criteria (Table 2). Of the samples that failed, six exhibited a geochemical signature that demonstrated the sampled material was not the waste basalt blank material. Subsequent investigation confirmed that the wrong material had been sampled and submitted as blank material. A single true blank sample (M132344) returned a lead assay approximately 10 times more than the expected value and was proceeded by a sample with a grade of 22.65% Pb. While this was potentially an instance of cross-sample contamination, the magnitude is not considered material. It is also possible that the result is a natural variation of the waste basalt material used as a blank.

Table 21: Genalysis assays of in-house and Geostats standards for RC drilling 2018

Description	Al %	Fe %	P %	Pb %	S %	Average
M132032	3.41	3.6	0.04	5.76	0.15	Al=3.7%, Fe=3.6%, P=0.06%, Pb=3.75%, S=0.27%
M132069	3.54	4.19	0.06	4.67	0.23	
M132106	2.91	4.08	0.06	2.66	0.19	
M132138	4.22	4.47	0.07	3.84	0.5	
M132175	3.72	2.78	0.06	3.10	0.31	
M132212	4.32	3.87	0.05	2.46	0.23	
M132249	7.09	1.88	0.03	0.029	0.2	Al=7.4%, Fe=2.2%, P=0.03%, Pb=0.04, S=0.20%
M132283	7.02	1.82	0.03	0.022	0.19	
M132317	7.18	2.59	0.03	0.030	0.21	
M132344	7.42	2.27	0.03	0.149	0.19	
M132386	7.57	2.44	0.03	0.046	0.18	
M132428	7.47	2.52	0.03	0.057	0.22	
M132512	6.71	2.04	0.03	0.014	0.21	
M132554	7.34	2.48	0.03	0.075	0.2	
M132596	7.88	2.15	0.04	0.045	0.2	
M132638	7.59	2.35	0.03	0.043	0.22	
M132680	7.75	2.21	0.03	0.036	0.19	
M132764	7.73	2.24	0.04	0.063	0.2	
M132806	7.29	2.04	0.01	0.014	0.21	
M132890	7.6	1.85	0.03	0.005	0.19	
M132974	7.72	1.98	0.03	0.022	0.2	

Of the remaining confirmed blank samples, 12 blanks returned assays over the expected maximum of 150 ppm Pb.

Overall, the analytical performance of the blank material is considered acceptable with no systematic bias observed in the true blank material.

11.4.3 Duplicates

Field duplicates are inserted at a rate of approximately one per drill hole, with a total of 21 field duplicates submitted to Genalysis during 2018, with an approximate submission rate of one in 20.

Figure 29 shows the performance of field duplicates vs primary samples for RC drilling submitted during the 2015–2018 period.

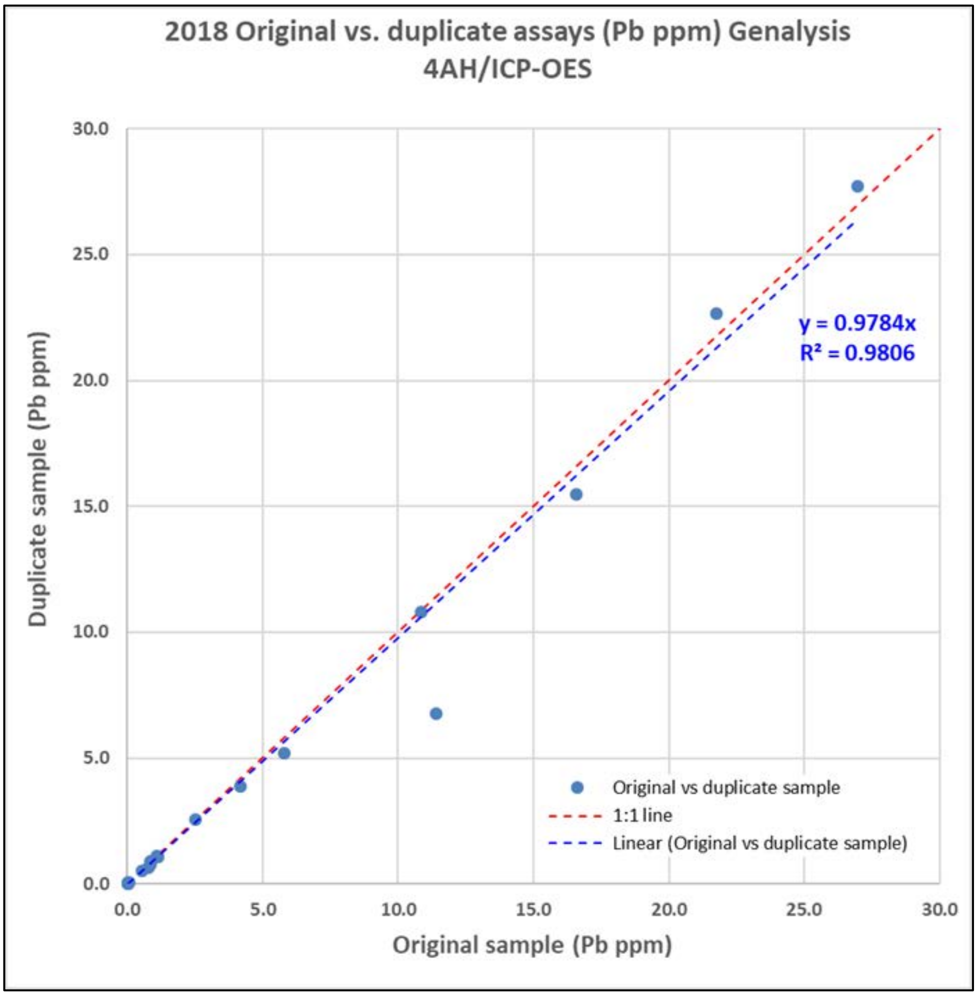

Figure 29: Field duplicate sample assay performance

The original sample grade assayed ranged from 50 ppm Pb (detection limit) to 277,512 ppm Pb (0.005% Pb to 27.75% Pb), testing the full range of lead grades.

No significant assay bias is observed for the field duplicate samples, with good correlation across most grade ranges.

11.4.4 Laboratory Pulp Checks

A total of 15 laboratory pulp checks were assayed. This is an internal laboratory check as part of Genalysis' internal QA/QC processes. A comparison is presented in Figure 30.

All pulp checks exhibit very good correlation with no material bias, showing good sampling precision is being maintained during subsampling of the pulverized pulp, digest and assay finish.

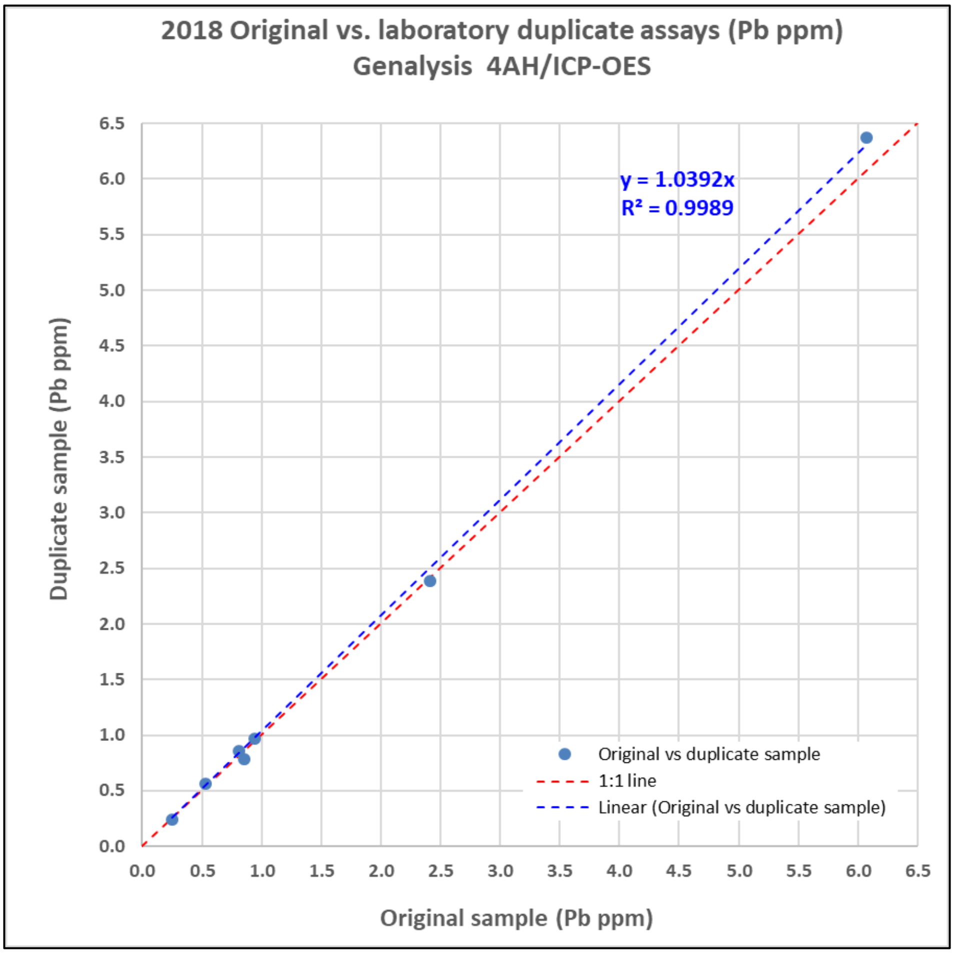

Figure 30: Pulp check performance

11.4.5 Umpire Duplicates

No umpire samples were taken in the 2015–2018 period.

11.5 Discussion

The performance of the RHM and in-house Genalysis standards demonstrates that an appropriate degree of analytical accuracy has been achieved for the samples submitted in 2018.

Although there were problems with the submitted blank samples, existing protocols were able to identify and resolve the problem. The remaining blank sample assays did not identify any systematic cross-contamination. The use of the waste basalt as a lead blank should be revised and a less variable, lower grade, lead blank material sourced. This maybe some other ad hoc material or a metallurgical grade blank material.

The field duplicate data show that the sample protocols achieve an acceptable level of sample and analytical precision has been achieved.

The laboratory duplicate data (duplicate samples taken from pulverized material) show that the laboratory achieves an acceptable standard of analytical precision, 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 the laboratory has achieved an acceptable level of precision at the sampling and analytical stages to date. Confidence in the assay data is sufficient to support all geological interpretation of mineralization.

11.5.1 Actions

No changes to the current sample protocol are currently required.

The material used for the lead blanks should be replaced with material that is less variable and of a lower average grade than the current waste basalt material used.

11.5.2 Results

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

11.6 Opinion on Adequacy

The results for blanks, standards and laboratory 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.

Overall, the available QA/QC data demonstrated that the sample and analytical data captured is of sufficient accuracy and precision to support the estimation and classification of the Mineral Resource as Measured, Indicated and Inferred Mineral Resources.

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
- 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 have been employed during care-and-maintenance periods.

The QP observed the geological and sample work flow associated with the 2014 RC drilling program (as outlined in SRK 2015, Section 10.2.3) during 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 Section 11.

A 2015 independent review of the QA/QC results of 22 sample batches (4,681 drill hole samples) submitted to Genalysis in Perth was commissioned. These included 538 QC samples and 977 standard samples (Optiro, 2015) and found that the QA/QC data demonstrated that the sample and analytical precision and accuracy was appropriate for the estimation of Mineral Resources.

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

All sampling associated with subsequent RC drilling programs in 2015 and 2018 was subject to the same processes as for the 2014 RC drilling program. Sampling of diamond drill core was completed in 2017 for metallurgical testing was handled as part of a DFS conducted during 2017 and 2018.

QA/QC sampling and review is discussed in Section 11.4.

12.1 Procedures

Electronic transfer methods employed by RHM and its consultants remove the risk of minor typographic errors. Subsequent interpretation of the data during geological modelling identifies 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's 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 needed 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 identification (Optiro, 2015).

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

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 opinion of the QP that the data at the Paroo Station Mine is appropriate for use and the methods employed are considered industry standard. The drilling, sampling and analytical methods are considered appropriate for the geological modelling, estimation and subsequent mining of the deposits.

13 Mineral Processing and Metallurgical Testing

13.1 Testwork

Three areas of testwork are described in the following sections. The first section relates to the initial and following year's testwork for the existing flotation plant at the Paroo Station Mine. The second describes the testwork undertaken during 2017 in relation to the 2018 DFS study for the Hydrometallurgical Facility, which included some additional flotation testwork. The third describes the testwork undertaken for the Demonstration Plant which was run in 2018 for the 2019 DFS Update, to provide confirmation of the final flowsheet design.

13.1.1 Original Metallurgical Testwork

The original metallurgical testwork was done 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-sulfide lead mineralization, especially with a significant portion of the lead being in fine particle size fractions.

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

The technical literature contains limited discussion on sulfidization 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 ($PbAl_3(PO_4)_2(OH)_5$•(H_2O)). 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 mineralization 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 sulfidization flotation performance is considered negligible.

Amdel's metallurgical testwork program for the Magellan deposit material consisted of bench-scale laboratory flotation tests on samples from 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.

The Bond Ball Mill Work Index (BBMWI) of the Magellan material measured in the 2–18 kWh/t range, 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 the pilot plant operation had been stabilized, surveys showed it produced a lead concentrate assaying 67% Pb –75% Pb with recoveries in the range of 77% Pb –88% Pb from a head grade of 8% Pb with a flotation feed sizing of 80% passing -75 µm.

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

The testwork demonstrated that sulfidization flotation was a viable process route for treating the non-sulfide lead mineralization 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. Up to April 2007, it processed 2.172 Mt of ore at a head grade of 7.3% Pb, with a 71.7% Pb recovery into the lead concentrate.

There are two main differences between the production plant and the initial metallurgical testwork:

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

13.1.2 Flotation and Hydrometallurgical Facility Testwork Program (2017)

The 2017 testwork program was carried out 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 InCoR, 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 InCoR to evaluate flotation performance 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 InCoR 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.

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

1 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 9,322,104 and Hydrometallurgy 142 (2014) 23-35), and included some preliminary testing of the Paroo concentrate. Methane sulfonic acid (MSA) is used in lead electroplating and MSA has properties that make it particularly suitable for use in the Paroo flowsheet, the most important being the high solubility of metals. The simplified technology 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 the variable feed compositions expected.

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 project are summarised below.

2 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 original University of British Columbia (UBC) paper provided results on leaching and electrowinning but no solid/ liquid separation testwork had been undertaken.

The proof of concept testwork included three stage leaching tests (primary MSA leach of concentrate, DeS (desulfurization) leach and secondary MSA leaching of DeS solids). Preliminary testing of a range of unit operations including solid/ liquid separation and bleed treatment circuits in the Hydrometallurgical Facility to provide preliminary engineering design data, was conducted. 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.

3 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 LOM. These ore samples were shipped to ALS for preparation and testing. This work comprised two phases. The initial work comprised flotation testwork to prepare annual concentrate samples for hydrometallurgical testing. However, following 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 resulted in an increased flotation recovery at target concentrate grades in the range 55%–60% Pb. 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 InCoR/ RHM to target a 70% Pb 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 pre-conditioning and operating pH lead to significant improvements in lead recovery and flotation kinetics even at the higher concentrate grade of 70% Pb. Column flotation was specifically used for the final concentrate cleaning stage to allow the concentrates to be washed to maximize removal of gangue slimes, which were affecting 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.

4 Variability Testwork Hydrometallurgical Testing

The initial hydrometallurgical testing flowsheet followed the sequential plant operations of three consecutive leaching operations, followed by an impurity removal step and lead electrowinning. However, ongoing issues with residue thickening and filtration led to a change in flowsheet whereby the DeS leach residue would be reintroduced into the Flotation Concentrator circuit to recover the lead carbonate produced in the DeS leach. The flotation recoveries in a single stage of flotation were excellent, and this flowsheet change was adopted which allowed two of the three solid/ liquid separation circuits to be eliminated. Operationally, this revised circuit will be much simpler, and would incur lower operating and capital costs.

Some of the most significant improvements to the Hydrometallurgical Facility flowsheet are in fact based on changes to the operating philosophy of the Flotation Concentrator as the leaching operations are very simple and robust, whereas the solid/ liquid separation processes are more difficult and are largely dictated by the physical characteristics of the flotation concentrate.

The InCoR/ RHM decision to clean 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, this eliminates 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 aluminum 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 optimization 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 controlled by the rise rate, 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.

5 Pilot Plant Testwork – Flotation

Bulk samples from stockpiled ore on the ROM pad comprising a high grade (12.0 dmt at a grade of 8.4% Pb) and lower grade sample (5.8 dmt 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. From the piloting exercise, a range of concentrate lead grades were developed.

ALS assembled a pilot milling and flotation plant and both samples were treated separately through the pilot plant using the reagent regime developed during the variability testwork program and also adopting the revised flowsheet. During initial operation of the pilot plant, as a result of the improved flotation kinetics, a decision was taken by InCoR 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 subsequent to the first cell were recycled to the head of the rougher. 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 first rougher without any requirement for further cleaning and the decision to close the first cleaner circuit. Washing in column flotation is also a significant driver of concentrate grade, with minimal cleaning losses. The first rougher and first cleaner concentrates were combined, and the lead grade approximated 60% for both the high grade and low grade samples.

After conducting initial hydrometallurgical testwork on the 60% Pb concentrates, it became apparent that both the concentrate filtration and the filtration of leach residues were 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% Pb ±2% Pb has been set for the reconfigured flotation plant based on the results achieved in column cleaning of the high grade and low-grade concentrates. The final concentrates produced were also washed with demineralized water to remove residual flotation reagents and trace chloride content.

6 Pilot Plant Testwork – Hydrometallurgical Testing

The pilot plant operation was run in two stages. The initial reliability run was of short duration due to an early failure of the electrowinning rectifier which proved ultimately to be fortuitous. This initial run experienced a number of extreme levels of frothing in the MSA leach reactors and difficulties with solid/ liquid separation and electrowinning unit operations. The delay in repairing the rectifier allowed these issues to be addressed in a systematic manner, which resulted in the second pilot plant run being much smoother.

An anti-foaming agent was initially tried as a means of froth control in the leach reactors, and while successful, residual agent caused significant issues with downstream solid/ liquid separation. The reagent was also costly. However, an alternative approach of drying the concentrate proved to be at least equally effective in controlling frothing and proved that residual flotation reagents were the root cause of the problem. Concentrate drying was adopted for the second pilot plant run and included in 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, but chemically of acceptable quality. Lead ingots were also produced for assay. Significant dross formation during this step at the laboratory 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.

13.1.3 Flotation and Hydrometallurgical Facility Testwork Program (2018)

Variability Testwork Hydrometallurgical Testing

The initial hydrometallurgical testing flowsheet followed the three consecutive leaching steps with intermediate solid/ liquid separations, followed by an impurity removal step and lead electrowinning. However, ongoing issues with residue thickening and filtration at each stage led to a change in the flowsheet concept whereby the DeS leach residue would be reintroduced into the Flotation Concentrator circuit to recover the lead carbonate produced in the DeS leach. This change eliminated two of the solid/ liquid separation steps and the second MSA leach step, significantly simplifying the overall flowsheet. Flotation recoveries of the DeS leach residue in a single stage of flotation were excellent and this flowsheet change was adopted.

The most significant improvement in the Hydrometallurgical Facility performance are in fact based on changes to the operation of the Flotation Concentrator as the leaching operations are very simple and robust whereas the solid/ liquid separation processes proved to be significantly difficult. These issues are dictated by the physical characteristics of the fine gangue slimes in the flotation concentrates.

The decision to clean the flotation concentrates in a second cleaner stage of column flotation resulted in an almost complete rejection of the gangue components in the concentrate, eliminating these as a source of variability in the leaching operations.

The key variable remaining which influences the performance of the Hydrometallurgical Facility is the relative proportions of cerussite and anglesite in the concentrate. This ratio impacts 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, recovery of lead contained in minor 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 in solution to the extent that, subject to electrowinning performance with these metals present, the need for this circuit could be considered in a future optimization exercise.

The key unit operations tested in the variability program were 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 are controlled by the rise rate, given the very low feed densities of the leach residues in all of the leaching steps. Subsequent changes to the flowsheet has made this work redundant.

As a result of variable and poor filtration rates and poor washing efficiencies achieved during the variability and pilot plant test program, the flowsheet was changed through the introduction of counter current decantation (CCD) of the MSA Leach Residue. Thickening testwork was essential in defining settling rates and predicted underflow densities. In conjunction with this change, flotation of the DeS Leach residue was tested on all samples.

Batch Testwork on Flotation Concentrator – DFS Update

Primary Grind Size Testwork

Analysis of lead losses in the Flotation Concentrator tailings identified potentially recoverable lead losses in the coarse end of the size range. Closing the milling circuit with screens in the pilot plant operation showed a significant improvement in lead recovery in comparison to the equivalent batch testwork. The value of the screens in improving flotation performance was not initially recognized until size by size assay data on the flotation tailings became available.

A series of batch flotation tests were carried out, which involved pre-screening each flotation test feed prior to milling across a range of screen sizes of interest with the screen oversize ground to pass the screen. The results of these tests indicated significant improvements in lead recovery could be achieved by closing the milling circuit with screens.

Sulfur and Galena Flotation

Elemental sulfur is generated in the MSA leach as a result of the leaching of galena and would otherwise build up in the circulating load in the flowsheet if a means of removal was not provided.

The quantity of sulfur generated is small, and operation of the flotation circuit need not be continuous as the sulfur can be allowed to build up to a level which improves flotation performance. Sulfur flotation is slow and in order to remove entrained gangue, a small flotation column is provided.

Galena flotation is not considered necessary in other than a major process upset as prior to the flotation step, galena is leached in the MSA leach so the likelihood of a requirement to float galena is considered low.

Nonetheless, testwork was carried out on a residue sample containing galena to evaluate the flotation performance and reagent requirements.

Batch Testwork on Hydrometallurgical Facility – DFS Update

A range of batch testwork programs arising out of recommendations in the DFS were carried out in advance of the demonstration plant operation in order to define the operation parameters of the Demonstration Plant.

The focus of the programs was to maximize lead recovery from the minor lead minerals across the Hydrometallurgical Facility flowsheet.

Electrowinning

Further testwork was carried out at UBC evaluating a range of lead smoothing agents to be used in the tankhouse. The testwork was carried out on simulated electrolytes which reflected the solution compositions in the overall plant model. A combination of two smoothing agents was identified for use in the demonstration plant – Aloes and EW 50.

Aloes is the primary smoothing agent, a natural product used by Tech Resources at their Trail Operations lead electrorefining circuit.

EW50 is a secondary smoothing agent produced by Solvay for use in copper electrowinning tankhouses.

Coupon Testing

Coupon testing was carried out at ALS on a range of stainless steel and plastics to confirm Materials Of Construction (MOC). This work was carried out over a period of two months at two temperatures covering the range of expected plant operation. No issued were identified with the MOCs selected.

DeS Leaching

Further testwork was carried out to maximize the conversion of anglesite to cerussite in the DeS leach unit operation. A number of tests were carried out over a range of temperatures to assess whether improved anglesite conversions could be achieved. This testwork resulted in a decision to increase the DeS leach temperature from 40°C to 60°C to maximize conversion rates.

Boiling Point Elevation

Boiling point elevation testwork was carried out to provide equipment vendors with information to design the evaporator based on the final solution compositions proposed for the circuit.

Ca Solubility

The leaching reagent used in this circuit is also used industrially as a descaling reagent and is known to hold calcium in solution. A range of tests were carried out to assess the relationship between MSA and calcium concentration in solution to confirm earlier work which is incorporated in the bleed circuit design for MSA recovery.

Acid Leaching

Acid leaching of MSA leach residues is required to convert minor lead minerals in the leach residue into lead sulfate which is amenable to conversion to lead carbonate followed by re-leaching to recover the remaining lead. Testwork has proven that pyromorphite and galena in the leach residues can be converted to lead sulfate. Pyromorphite is also generated by unwanted side reactions in the impurity removal circuit and this circuit provides a means of recovering the lead contained in these minerals.

A series of acid leach tests were carried out to evaluate the operating conditions necessary to oxidize pyromorphite with sulfuric acid. Galena was also found to be substantially oxidized under the conditions identified. A temperature of 80°C and a residence time of 2 hours was found to be necessary to maximize pyromorphite and galena oxidation to lead sulfate.

Demonstration Plant Testing – Flotation – DFS Update

Three bulk samples were shipped to ALS to prepare the required composites for the operation of the demonstration plant flotation circuit. The flowsheet tested was essentially identical to that used in the pilot plant testwork with the exception that the column flotation column was now fully integrated into the flowsheet such that the tailings stream from the column was recycled within the overall flowsheet.

Four composites were produced across a range of lead grades:

- Composite 1 11.0% Pb
- Composite 2 6.47% Pb
- Composite 3 4.50% Pb
- Composite 4 2.80% Pb.

The objectives for the flotation circuit were:

- To produce approximately 2.5 t of flotation concentrate to provide a feedstock to the hydrometallurgical demonstration plant
- To develop a grade recovery curve for the final Flotation Concentrator flowsheet to be used in financial modelling
- To confirm the flotation recovery improvements identified in batch testwork attendant on closing the milling circuit with a final screen size and all modifications proposed for the flotation circuit.

These objectives were satisfactorily met during the operation of the Demonstration Plant and consistent recoveries were achieved from each composite over the duration of the run. Higher concentrate grades in a tight range around 72% Pb were generated regardless of the incoming ROM head grade which has largely eliminated any concentrate variability issues except the proportion of anglesite in the concentrate.

A grade recovery curve was generated which provided a consistent dataset across the range of feed grades from 3% Pb to 11% Pb. This data has been used to derive reserve cut-off grades and metallurgical recoveries across the predicted mine plan.

Demonstration Plant testing – Hydrometallurgical Facility – DFS Update

The Demonstration Plant was run to provide confirmation of the final flowsheet design. Changes from the pilot plant flowsheet were not substantial, being mainly residence times and operating conditions. The major change to the flowsheet was replacing filtration of the MSA leach residues in the pilot plant with a 7-stage CCD circuit.

The reduction in minor element leaching as a result of the improved flotation performance noted above has generated a significant improvement in the extent of minor element leaching. This has reduced process risk of not meeting the required product specification.

In addition, the primary focus of the Demonstration Plant was to confirm the operating plant parameters for the electrowinning circuit and to test suitable vendor equipment for the tankhouse anodes. To this end, two electrowinning cells operating in parallel has allowed side-by-side testing of anodes. The quality of cathode produced by the Demonstration Plant confirms expectations that the overall process is capable of producing 99.99% Pb cathode.

13.2 METSIM Modelling

The final version of the METSIM model has been developed progressively over the course of the Scoping Study, DFS, Early Works Engineering and the Pilot Plant and Demonstration Plant testwork to include all of the operations of both the Flotation Concentrator and the Hydrometallurgical Facility. The model has evolved with the flowsheet as a range of unit operations have been considered and either included or removed from the flowsheet.

The original UBC testwork on which the hydrometallurgical flowsheet is based identified a requirement for three separate leaching circuits: one to leach lead carbonates (cerussite), a conversion leach to react lead sulfate (anglesite) 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 to contain operating costs.

Testwork identified an opportunity to simplify the UBC flowsheet by eliminating the MSA re-leach 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, the following METSIM models for the Hydrometallurgical Facility have been developed:

- A Base Case model incorporating all the flowsheet elements required to operate the Hydrometallurgical Facility. The base case concentrate grade of 71.8% Pb was selected following investigation of concentrate treatment at grades of 55% Pb and 60% Pb. The concentrate feed input data to this model is based on average LOM data derived from the variability testwork program.
- Individual models were run using the Base Case model with concentrate changes for a high and low anglesite 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.

Following the DFS, the model was expanded to include the Flotation Concentrator and all proposed upgrades and modifications to the concentrator flowsheet such that a model of the entire process plant was constructed. This allowed the interfaces between the Flotation Concentrator and the Hydrometallurgical Facility to be examined in detail.

The current Mass Balance is derived from the integrated model which also provides input into the Design Criteria in the DFS Update.

The Mineral Reserve base used to define the initial METSIM model did not extend to the projected LOM under the revised operating cost scenario and lower cut-off grades. RHM therefore undertook a drilling program to provide representative samples of each year of production for the proposed new LOM, 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 minimized slimes recovery to the flotation concentrate was of the order of 70% Pb, up from the 67%–68% Pb grade targeted for sale to a smelter. With the revised flotation regime flotation, recovery was increased relative to historical concentrator performance. The composition of these concentrates is described below.

The impact of mineralogical variability in the ROM ore has largely been eliminated as a result of the improved flotation performance, which has largely eliminated gangue components from the concentrate. The key remaining variable in the concentrate is the relative proportions of cerussite and anglesite in the concentrate.

13.3 Process Design

The process facilities described in this section are for the treatment of lead carbonate ores, mined from the Paroo Station Mine open pits, at a nominal treatment rate of approximately 2.1 Mtpa. The flowsheet is comprised of an existing Flotation Concentrator, producing a high-grade lead carbonate concentrate for treatment in a Hydrometallurgical Facility to produce lead ingot at an annualized rate of 70,000 t.

A metallurgical testwork program comprising batch, Pilot Plant and Demonstration Plant works was carried out under the direction of RHM and InCoR to provide the design data required to develop the Flotation Concentrator and Hydrometallurgical Facility flowsheet.

The Process Design Criteria for the overall process plant has been developed by SNC-Lavalin to provide a basis to develop a METSIM model and for Mass Balance and equipment sizing calculations. Data from batch testwork, Pilot Plant testwork and Demonstration Plant testwork has been incorporated in the METSIM model.

The nominal design capacity of the new Hydrometallurgical Facility is 70,000 tpa lead ingot with sufficient design margin built into all aspects of the overall design, to allow a maximum of 80,000 tpa of lead ingot to be produced.

Annual production will be dependent on the quantum and lead feed grade producing a 72% Pb ±1% Pb concentrate by the modified Flotation Concentrator. The modified Flotation Concentrator will be capable of receiving up to 2,185,000 tpa of ore, but its throughput will be constrained by feed grade to the mill and a target 72% Pb grade for concentrates so as not to exceed the maximum 80,000 tpa production capability of the new Hydrometallurgical Facility.

The LOM plan is based on meeting the ramp-up curves of the Flotation Concentrator and the new Hydrometallurgical Facility with annualized production of 70,000 tpa of lead ingot produced in Month 13 of operations and annualized production of 80,000 tpa lead ingot achieved in Month 24 of operations.

13.4 Flotation Concentrator description

13.4.1 Modifications to Flotation Concentrator

The existing flotation concentrator will be modified prior to recommencing operations to achieve two objectives – increased throughput and improved metallurgical performance.

Modifications to the following process areas were made:

- Closing the milling circuit with screens
- Revising the flotation flowsheet to improve metallurgical performance
- Including column flotation in the cleaning circuit
- Replacing equipment as required ensuring plant availability.

13.4.2 Primary Crushing

The objective of the Primary Crushing area is to reduce ROM ore in one stage of crushing to a size suitable for further size reduction in the Milling circuit. Operating at 92% availability (on average 22.1 hours a day), the Primary Crushing circuit delivers crushed ore at a P80 of 45 mm to the semi-autogenous grinding (SAG) Mill.

Open pit ore is loaded onto trucks and hauled to a ROM Pad situated close to the crushing facility where it is dumped onto finger stockpiles. The stockpiles are used for blending the ore between the mine and plant for grade control and ore hardness control. A front-end loader (FEL) feeds ore into the ROM Bin. Loading of ore to the ROM bin is controlled by the crusher operator who activates tipping light indicators from the control room. Live capacity of the ROM Bin is 150 t, giving approximately 30 minutes' surge capacity at the design crusher throughput.

The ROM Bin discharges to the Apron Feeder which delivers the ore at a controlled rate to a Static Grizzly with an aperture of 130 mm designed to bypass final product sized ore past the jaw crusher. Grizzly undersize passes directly to the SAG Mill Feed Conveyer. Grizzly oversize gravitates to the Jaw Crusher.

The Jaw Crusher discharges to the SAG Mill Feed Conveyer which transfers the crusher product at a nominal rate of 248 tph to the mill. The apron feeder discharge rate is controlled by the SAG Mill Feed Weightometer which regulates feed to the SAG Mill Feed Conveyer.

A monorail maintenance hoist is provided for jaw crusher maintenance.

Dust emission control in the Crushing area is provided by a separate collection system within the Crushing building. The dust emission control system consists of a single insertable dust collection unit for the Primary Crusher.

Provision has been made for the future installation of a fixed rock breaker. The Crushing plant floor will be sloped and graded to a floor sump. A single sump pump delivers spillage to the SAG Mill discharge hopper. This pump is sized for a flow rate of 35 m^3/h.

13.4.3 Pebble Crushing

Oversize is discharged from the SAG Mill trommel onto the Pebble Crusher Feed Conveyer. Two Belt Magnets and a Metal Detector are provided to remove tramp steel, predominantly ball scats ahead of the Pebble Crusher.

If the pebble crushing circuit is offline, a diverter gate and emergency chute is provided to divert the SAG Mill discharge screen oversize to a concrete bunker.

The pebble crusher produces a 10–12 mm product that discharges to the SAG Mill Feed. A monorail maintenance hoist is provided for pebble crusher maintenance.

A pebble crusher feed weightometer monitors the pebble recycle rate to allow the new feed rate to the SAG Mill to be calculated by difference.

13.4.4 Milling

The objective of the Milling circuit is to liberate the lead minerals contained in the ore through a process of particle size reduction. Once liberated, lead minerals are amenable to separation and upgrade by froth flotation. The Milling circuit comprises a SABC flowsheet operating in closed circuit with screens.

Mill Feed System

The feed rate of crushed ore provided by the primary crushing circuit is measured by a weightometer on the SAG Mill Feed Conveyer. The feed rate setpoint to the SAG Mill will be controlled by a combination of mill power, mill load, possibly ore grade and/ or other variables by an algorithm to be defined at a later date.

Grinding Media

Balls are added to each mill on a batch basis as required to maintain the ball charge in each mill. Individual monorail hoists are provided to load balls into each mill from a 1 t ball kibble.

SAG Mill

The ore is fed to a 7.32 m diameter by 3.96 m Effective Grinding Length (EGL), 1.35 MW SAG Mill by the SAG Mill Feed Conveyer. The mill feed rate is controlled at the set point of a nominal 248 tph or as required via a control loop with input signal from the weightometer and a feedback signal to the relevant feeder speed.

SAG Mill speed is fixed at 65% of critical speed. The plant operator maintains the mill charge at a set weight read by load cells under the mill mountings via feedback control to the SAG Mill feed rate. These control loops are used by the operator to adjust the milling parameters to meet the grind size targets at various feed conditions and throughputs required by the operation.

SAG Mill discharge of nominally -21 mm (via a grate); the grate also contains pebble ports at approximately 80 mm to provide a feed to the pebble crusher. Pebbles are passed either to the pebble crusher feed conveyer or to the scats bay if the pebble crushing system is offline for any reason.

The extent of SAG Mill oversize is expected to be variable based on the ore hardness, and is collected by a recycle conveyor for pebble extraction and crushing.

SAG Mill discharge slurry is collected in the rubber-lined 2-compartment SAG Mill Discharge Hopper. Variable speed metal lined Screen Feed Pumps deliver the slurry to a distributor box which splits the flow between the two safety screens.

Ball Mill

The Ball Mill receives screen underflow all or in part from the stack sizer underflow box. The Ball Mill has 1.35 MW installed power at a mill size of 5.49 m diameter and 8.53 m EGL, running at a speed of 75% critical. The ball charge is approximately 35%. The Ball Mill discharge is laundered to the 2-compartment Ball Mill Discharge Hopper. A trommel on the Ball Mill discharge recovers ball scats which are discharged to a concrete bunker at floor level for removal. A maintenance access corridor is provided along the front of the milling train to facilitate routine operations around the mills.

Ball Mill discharge slurry is collected in the rubber-lined 2-compartment Ball Mill Discharge Hopper. Variable speed rubber lined Screen Feed Pumps deliver the slurry to a distributor box which splits the flow between the two safety screens.

Safety Screens

Two horizontal vibrating Safety Screens are provided to remove oversize from the stack sizer feed. The screen has a 5 mm slotted aperture. Screen oversize discharges to the stack sizer oversize launder to recycle back to the mills. Screen undersize gravitates to the Stack Sizer Primary Distributor.

Classification

The primary distributor is a 10-port splitter, eight live ports and two blocked off against a future expansion of the stack sizer cluster. Each line gravitates to one of eight stack sizer distributors, each 5-port distributor feeding a single stack sizer.

Eight stack sizers are provided to screen the combined SAG Mill and Ball Mill discharge. Screen undersize gravitates to the Rougher Flotation Feed Tank. Each stack sizer has five decks that cut at 150 microns. Stack sizer underflow is collected in two pipe launders that gravitate to the Screen Underflow Splitter Box. Screen oversize is distributed between the SAG Mill feed and the Ball Mill feed chutes to balance the power draw of the two mills.

Screen underflow is sampled by automatic sample cutters to provide metallurgical accounting samples and feed to the on-stream analyzer (OSA).

Water Addition to Mills

Process water addition to the SAG Mill feed is aligned to the new mill feed rate to maintain mill pulp density in the range of 65%–70%. The Ball Mill receives screen underflow at approximately 66% solids, which will nominally not require water addition. Process water is added to the SAG Mill Discharge trommel screen sprays to ensure that a clean separation is achieved on the trommel.

A regulated water addition to the SAG Mill discharge is set to maintain the stack sizer feed density at a target density to achieve the required screen underflow density, which is targeted at between 34% and 35% solids to control the product size and keep the pulp density in a range suitable for flotation.

Water addition to the Ball Mill feed is made if required to maintain the Ball Mill pulp density in the range of 65%.

13.4.5 Rougher and Scavenger Flotation

The rougher scavenger circuit comprises a conditioning train of four tanks after which the flotation feed is split into two parallel trains of tank flotation cells.

Changes to the reagent regime compared to the existing flowsheet have resulted in four sequential conditioning tanks being provided ahead of rougher flotation. The conditioning stages are:

- Acid conditioning at a pH of 5.5 achieved by the addition of the acid leach thickener discharge to the flotation feed
- Adjustment with lime to pH 6.5
- Conditioning with NaHS targeting a final pH range of 8.0–8.5. Flotation recovery is extremely sensitive to pH levels above 8.5 with high falls in recovery noted if flotation is carried out above a pH level of 8.5
- Conditioning with Sodium Isobutyl Xanthate (SIBX)

The objective of this circuit is to produce two separate concentrate streams per train for subsequent cleaning and recycling to flotation feed. The first concentrate stream is derived from rougher cells 1–2 in each train, which recover approximately 60% of the lead in the feed at a grade in the range of 55%–60% Pb. This concentrate stream is taken straight to the second cleaner column to minimize cleaning losses. The remaining rougher and scavenger concentrates comprising the second concentrate stream are collected in a pump box and are pumped to the first cleaner circuit feed box.

The flotation circuit comprises four agitated Rougher Conditioning Tanks, three Rougher Flotation Cells on each train, and three Scavenger Flotation Cells. The flotation cells are three cell banks of 38 m^3 capacity.

Process slurry received from the Rougher Flotation Feed Pumps enters into the rougher conditioning train detailed above where it is mixed with reagents, collectors and dilution water prior to gravitating into the flotation feed Splitter Box. Frother is added to the feed box of the rougher flotation cells.

Concentrate from rougher cells 1–2 gravitates to the Cleaner Concentrate Pump Box and the duty Column Flotation Feed Pump pumps the rougher concentrate to the second Cleaner Circuit.

Concentrate from the remaining rougher/ scavenger cells gravitates to the Rougher/ Scavenger Concentrate Pump Box and the duty Rougher/ Scavenger Concentrate Pump pumps the rougher/ scavenger concentrate to the first Cleaner Circuit.

Float level indicators are used to control flotation cell level by adjusting valves located on the tails discharge of every third flotation cell. The flotation cells are forced aspirated by blowers.

The rougher/ scavenger flotation area floor will be adequately sloped and graded to four floor sumps. One sump pump will be provided in a location adjacent to the rougher conditioning tank and two adjacent to the scavenger circuit discharge. These pumps will be sized to accommodate a flow rate of 30 m^3/h.

On-stream Analysis System

A Courier 6 slurry analyzer is provided for Wavelength Dispersive X-ray Fluorescence (WDXRF) measurement of iron, lead and sulfur. Sixteen samples flow under gravity or are pumped to a multiplexer above the measurement unit where each stream is sequentially sampled and analyzed. The samples are combined into two streams, the larger of which is pumped to the rougher conditioning tank. Sample points will be single-stage or 2-stage sampling points as required by the size of the process flows.

Streams sampled are:

- Sample Point – Flotation Feed Train 1
- Sample Point – Flotation Tailings Train 1
- Sample Point – Rougher Concentrate Train 1
- Sample Point – Scavenger Concentrate Train 1
- Sample Point – Flotation Feed Train 2
- Sample Point – Flotation Tailings Train 2
- Sample Point – Rougher Concentrate Train 2
- Sample Point – Scavenger Concentrate Train 2
- Sample Point – Second Cleaner Concentrate
- Sample Point – Third Cleaner Concentrate
- Sample Point – Cleaner Tailings.

Sulfur Flotation

The Sulfur Flotation cells comprise a bank of three Denver flotation cells of 8 m^3 capacity, each comprising of a bank of three cells, a feed box and a drop box. The launders will be reconfigured to allow the concentrate stream to gravitate to the Sulfur Rougher Concentrate Pump Box. Frother is added as required to the feed box of the first flotation cell. Float level indicators are used to control the sulfur rougher flotation cell level by adjusting valves located on the tails discharge of the bank of cells.

The Sulfur Rougher Concentrate Pump pumps the sulfur rougher concentrate to the Sulfur Cleaner Flotation Column. The sulfur cleaner tailings gravitates to the sulfur rougher feed box. Sulfur column concentrate gravitates to the flotation tailings area sump pump.

A positive bias ratio is maintained on the sulfur cleaner column to remove entrained gangue slimes from the column concentrate.

Air supply to the base of the sulfur column is mixed with a circulating slurry flow generated by the Sulfur Column Circulating Pump. Medium pressure blower air is supplied to the column.

Cavitation tube sparging generates 'pico-bubbles' substantially increasing the surface area available to the desired target particles. The cavitation tube sparger is at the heart of aeration systems used for bubble generation in the column flotation cells. The 'Cav-Tube' is a hydrodynamic sparger shaped to maximize fine bubble generation, effectively increasing the superficial surface area rate (Sb) of air moving through a column flotation cell, thus maximizing metallurgical performance.

Galena Flotation

The Galena Flotation Cells comprise a bank of three Denver flotation cells of 8 m^3 capacity, each comprising of a bank of three cells, a feed box and a drop box. The launders will be configured to allow the concentrate stream to gravitate to Galena Rougher Concentrate Pump Box. Frother and SIBX are added as required to the feed box of the first flotation cell. Float level indicators are used to control the galena rougher flotation cell level by adjusting valves located on the tails discharge of the bank of cells.

The Galena Rougher Concentrate Pump pumps the galena rougher concentrate to the Galena Cleaner Flotation Column. The Galena cleaner tailings gravitate to the galena rougher feed box. Galena column concentrate gravitates to the rougher flotation area sump pump.

A positive bias ratio is maintained on the galena cleaner column to remove entrained gangue slimes from the column concentrate.

Air supply to the base of the sulfur column is mixed with a circulating slurry flow generated by the Column Circulating Pump. Medium pressure blower air is supplied to the column.

13.4.6 First Cleaner Flotation

The first Cleaner Flotation Cells comprise a bank of 6 × OK38 flotation cells of 38 m^3 capacity, each comprising of a bank of three cells, a drop box and a further bank of three cells. The launders on the first cell will be configured to allow the concentrate stream to gravitate to second Cleaner Column feed. The remaining first cleaner concentrate gravitates to the first Cleaner Pump Box which is an in-ground sump. Frother is added, as required, to the feed box of the first flotation cell. The first Cleaner Concentrate Pump pumps the first cleaner concentrate to the head of the first cleaner circuit. First Cleaner tailings gravitate to the tailings thickener.

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

13.4.7 Second Cleaner Flotation

Rougher concentrate from cells 1–2 of each rougher/ scavenger train plus concentrate from the first cleaner are combined in the Column Flotation Feed Pump Box and pumped by the Column Flotation Feed Pump to the Cleaner Flotation Column. Column concentrate gravitates to the Concentrate Thickener.

Discharge from the column controls the operating level and is driven by the differential head between two pressure sensors on the upper body of the column. Column tailings discharge to the first cleaner cell 1 Column Flotation Tailings Pump Box. Column flotation tailings is pumped by the Column Flotation Tailings Pump to the second cell in the first cleaner train.

A positive bias ratio is maintained on the column to remove entrained gangue slimes from the column concentrate.

Air supply to the base of the column is mixed with a circulating slurry flow generated by the Column Circulating Pump. Medium pressure blower air is supplied to the column. Cavitation tube sparging generates ‘pico-bubbles’, substantially increasing the surface area available to the desired target particles, thus maximizing metallurgical performance.

13.4.8 Concentrate Thickening

The Concentrate Thickener 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 for treatment in the Hydrometallurgical Facility.

Flocculant addition to the 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 maintains a bed level in the concentrate thickener based on the flow ratio pre-set by the operator to supply the required rate of flocculant addition.

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 Underflow Pump is controlled by the set point of the mass flow calculation on the pump discharge line.

Concentrate pumped to the Concentrate Stock Tank which provides 15 hours’ surge capacity between the thickening and filtration circuits. Thickener overflow gravitates to the Process Water Pond.

The floor sump level is monitored ultrasonically. The Concentrate Thickener Area Sump Pump is automatically started, stopped and alarmed using the alarm contacts in the level sensor. The pump will be sized to accommodate a flow rate of 25 m^3/h.

13.4.9 Concentrate Filtration

Thickened flotation concentrate slurry is pumped to the agitated Concentrate Stock Tank at a variable flow rate, dependent on the Flotation Concentrator head grade and the performance of the concentrate thickener. The stock tank is used to provide surge capacity between the flotation circuit and the concentrate filter, allowing filter maintenance to be carried out. The stock tank level is measured by an ultrasonic-type level transmitter but is not controlled.

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

The Concentrate Filter is a 36-chamber plate-and-frame filter with plates of 1.5 m × 1.5 m and a screen area of 81 m^2. The filter is fully automatic and is expected to operate on a 15–17 minutes’ cycle time producing a maximum of 17.8 tph at a moisture content of 8% solids.

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

The filter cake discharges to a hopper in batches up to 6 t every 15 minutes when the filter is in operation. The concentrate falls under gravity from the Concentrate Filter Discharge Chute to the Concentrate Feeder.

13.4.10 Lead Concentrate Composition

The plant will produce approximately 100,000 dry tpa of lead concentrate at an approximate grade of 72% Pb. The concentrate is comprised of two major lead minerals; cerussite at 85% of the total lead and anglesite at 12% of the total lead. Minor lead minerals make up the remainder of the concentrate. Anglesite in the ore varies between about 3% and 30% of the lead in the ROM feed to the Flotation Concentrator on an unblended basis, so the expected range of anglesite composition will be narrower that the in-pit range. The Hydrometallurgical Facility has been designed to accommodate a suitable range of anglesite composition.

13.4.11 Concentrate Storage Shed

The existing concentrate storage shed has sufficient capacity to hold 10,000 t of lead concentrate. It is not envisaged that storage of that mass will be required; however, using the shed to provide buffer storage between the Flotation Concentrator and the Hydrometallurgical Facility is prudent, particularly during ramp-up.

A concentrate reclaim facility will recover concentrate to the Hydrometallurgical Facility feed preparation area.

13.4.12 Flotation Tailings

The Tailings Thickener is fed under gravity from the discharge of the final scavenger flotation cell in each train and the first cleaner tailings.

Flocculant addition to the tailings thickener is controlled using a flow element linked to the flocculant pump speed controller which 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 delivery rate is adjusted to maintain an operator input set point 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 Underflow Pump is controlled by the set point of the mass flow calculation on the pump discharge line. Tailings are pumped to the tailings disposal dam.

The tailings thickener overflow gravitates to the Process Water Pond from which the duty Process Water Pump pumps the overflow back to addition points within the plant, predominantly the Milling circuit. The process water pond also receives overflow from the concentrate thickener, decant return from the tailings dam, and raw water makeup.

An ultrasonic type instrument measures the floor sump level. The Tails Thickener Area Sump Pump is automatically started, stopped and alarmed using the alarm contacts in the level sensor.

13.5 Hydrometallurgical Facility

The proposed process for the leaching of lead concentrates is a patented technology that uses MSA to leach lead carbonate. Minor lead sulfate is converted to lead carbonate using sodium carbonate. Lead methanesulfonate is electrowon to produce lead cathode noting that this process has also been used as an electrolyte to electrowin the lead commercially since about 1980 in lead electroplating. Lead cathode is melted to produce lead ingot.

The unit operations in the flowsheet that involve process chemistry are listed below:

- Feed preparation
- MSA leach
- MSA solid/ liquid separation
- Acid leach
- DeS leach
- Impurity removal
- Lead electrowinning
- Bleed treatment.

13.5.1 Feed Preparation

The feed battery limit for the study is the concentrate discharge from the existing lead concentrate filter.

A variable speed Concentrate Feeder, will withdraw concentrate at a controlled rate from the Concentrate Filter Discharge Chute, and deliver the concentrate to the Concentrate Dryer. The concentrate is dried to zero moisture and heated to 150°C to remove residual flotation reagents. The dried filter cake discharges to the MSA Leach Feed Re-pulp Tank.

Two agitators provide a high intensity environment in which to re-pulp the filter cake in MSA leach liquor at a density of 65% w/w solids. The concentrate filter discharge is intermittent and the variable speed drive on the concentrate feeder serves to smooth the intermittent discharge from the filter to achieve a relatively constant feed to the re-pulp tank. To an extent, the dryer will also smooth out the flow to the re-pulp tank.

The concentrate feeder discharge rate is controlled by the Concentrate Weightometer. The MSA contained in the electrolyte will be consumed in leaching the concentrate to the extent of the available acid in the re-pulped concentrate.

The re-pulped concentrate is pumped to the Leach Feed Surge Tank by the duty MSA Leach Feed Re-pulp Transfer Pump where the concentrate is mixed the MSA Leach Feed Surge Tank Agitator. A live storage residence time of 18 hours is provided to accommodate the intermittent feed from the concentrate filter, which is currently operational for 12 hours per day.

Re-pulped concentrate is pumped by the duty MSA Leach Feed Pump to the head of the MSA Leach circuit.

The Concentrate Feed Re-pulp Area Sump Pump pumps spillage from the bunded area to the Leach Filter Surge Tank or to the flotation plant tailings thickener, as required, for water recovery.

13.5.2 MSA Leach

Lead concentrate slurry is pumped from the surge tank to the MSA Leach Feed Box (2010-BXF-0001) that mixes all feed streams to the MSA leach and allows distribution of leach feed to either the first or second tank in the leach train through operation of dart valves.

Leaching is carried out in a single train of six agitated atmospheric leach tanks. MSA Leach Tank Agitators provide the necessary mixing and solids suspension for optimum reaction kinetics and oxygen dispersion. The last tank in the train discharges to the MSA Leach Discharge Thickener. The design includes provision to bypass any of the MSA leach tanks for maintenance or de-scaling purposes.

MSA in the spent electrolyte is used to leach most of the available lead carbonate, targeting little or no free MSA in the leach discharge, to minimize reagent consumption in the impurity removal stage. Provision is made to stage add MSA to multiple reactors if required.

The leaching reactions generate significant levels of CO_2 gas that is vented to a dedicated scrubber for all atmospheric leach circuits in the refinery. The tanks are covered and have a high freeboard to accommodate the expected frothing levels.

A sampler is provided on the MSA leach feed and leach discharge to determine the leach feed metal concentrations.

The MSA Leach Area Sump Pump pumps spillage from the bunded area to the leach feed pump box or the flotation plant tailings thickener, as required, for water recovery.

13.5.3 MSA Leach Solid/ Liquid Separation

MSA leach residue gravitates to the MSA Leach Residue Thickener Feed Tank where it is dosed with flocculant and thereafter gravitates to the MSA Leach Residue Thickener where the leach residue is thickened. Thickener overflow gravitates to the MSA Leach Residue Thickener Overflow Tank. Thickener overflow is pumped to the impurity removal circuit by the duty MSA Leach Residue Thickener Overflow Pump. Thickener underflow is pumped by the duty MSA Leach Residue Thickener Underflow Pump to the feed tank to CCD Thickener 1.

A train of six counter-current washing thickeners is provided to wash the MSA leach residue. Wash water is added to CCD Thickener 6 and gravitates to the MSA Leach Residue Thickener.

Each CCD Thickener has an agitated Feed Tank.

Duty and standby pumps are provided on each thickener underflow.

The CCD Area floor will be sloped and graded to two floor sumps. Two sump pumps deliver spillage to the CCD Thickeners.

13.5.4 Acid Leach

The acid leach circuit decomposes pyromorphite to lead sulfate for conversion to lead carbonate in the following DeS leach circuit. Residual galena from the MSA leach can also be leached in the acid leach and converted to lead sulfate.

Leaching reactions are carried out in a single train of four agitated atmospheric leach tanks.

Two electric immersion heaters are provided to heat the slurry to 80°C.

Sulfuric acid is added to the circuit at a controlled rate to maintain 24 g/L acid in the leach discharge.

Acid Leach Agitators provide the necessary mixing and solids suspension. The discharge of the last tank in the train gravitates to the Acid Leach Discharge Hopper and is subsequently pumped to the Acid Leach Thickeners. Provision is made in the design to bypass any of the acid leach tanks for maintenance or de-scaling purposes. The last tank in the train discharges to Acid Leach Discharge Tank. The slurry is then pumped by the duty Acid Leach Discharge Pump to Acid Leach Thickener 1.

The duty Acid Leach Thickener 1 Underflow Pump pumps thickener underflow to the feed of Acid Leach Thickener 2. The duty Acid Leach Thickener 2 Underflow Pump pumps thickener underflow to the DeS Leach Feed Tank. Acid Leach Thickener 1 O/F is collected in the Acid Leach Thickener O/F Tank and is pumped to the DeS Residue Flotation Acid Conditioning Tank. Wash water is added to each thickener at a ratio of 10:1 to solution in the thickener underflow.

13.5.5 Desulfurization Leach

Acid leach residue is combined with sodium carbonate solution in the agitated DeS Leach Feed Tank. Two electric immersion heaters are provided to heat the process slurry to 60°C.

The reaction of the anglesite with sodium carbonate is carried out in a single train of four agitated DeS Leach Tanks. The DeS Leach Agitators provide the necessary mixing and solids suspension. The design makes provision to bypass any of the DeS leach tanks for maintenance or de-scaling purposes. The last tank in the train discharges to DeS Leach Discharge Tank. The slurry is then pumped by the duty DeS Leach Discharge Pump to the DeS Flotation Area.

Sodium carbonate solution is added to the circuit at a controlled rate to maintain 3.5 g/L sodium carbonate in the leach tanks.

The DeS Leach Area Sump Pump pumps spillage from the bunded area to the DeS Leach Feed Tank or the spillage pond, as required, for water recovery.

13.5.6 Leach Area Scrubber

The MSA leach, DeS leach and Impurity Removal tanks vent gases are collected and passed through the Leach Area Scrubber. A caustic scrub liquor is used to maximize acid and impurity recovery from the gas stream. The scrubber bleed passes to the Refinery Bleed Tank and is pumped along with other minor bleed streams to the feed to the tailings thickener by the duty Refinery Bleed Pump.

13.5.7 Impurity Removal

An impurity removal circuit provides impurity control and acid balance. This circuit treats a split of the liquor discharge of the MSA circuit, typically in the range 0%–5% of the total flow and uses slaked lime to precipitate iron and aluminum ahead of the lead electrowinning circuit. Raising the pH also generates a precipitation reaction between lead and orthophosphoric acid which removes all the orthophosphoric acid from solution and a small proportion of the lead as the hydroxyl form of pyromorphite.

Precipitation reactions are carried out in a single train of one agitated atmospheric leach tank. The Impurity Removal Agitator provides the necessary mixing and solids suspension.

The discharge of the last tank in the train gravitates to the Impurity Removal Thickener.

The design makes provision to bypass the impurity removal tank for maintenance or de-scaling purposes.

Thickener underflow production is very low given the predicted impurity levels so in normal operation it is proposed to run the impurity removal circuit in recycle for most of the operating time to build up a bed of settled solids. A batch of thickened precipitate will be periodically fed to filtration via the Bleed Precipitation Filter Surge Tank.

The duty Impurity Removal Thickener Underflow Pump pumps thickener underflow to the bleed precipitation area for filtration and washing on an intermittent basis. Impurity thickener Overflow is collected in the Impurity Thickener Overflow Tank and is pumped to the evaporator circuit by the duty Impurity Removal Thickener Overflow Pump.

The Impurity Removal Area Sump Pump pumps spillage from the bunded area to the impurity removal feed box or to the spillage pond as required for water recovery.

13.5.8 Electrolyte Purification

The strong electrolyte is pumped from the Evaporator Discharge Tank by the duty Strong Electrolyte Pump to a filtration system comprising two co-matrix filters operating in parallel. The filtered advance electrolyte gravitates to the Advance Electrolyte Tank. Filtered electrolyte is pumped to the electrowinning circuit by the Advance Electrolyte Pumps.

Filtered electrolyte is periodically directed to the Filter Backwash Tank and is pumped by the Filter Backwash Pumps to the electrolyte filters during the periodic backwash cycle.

Filter backwash is collected in the Filter Backwash Product Tank, and is pumped to the impurity removal circuit by the Backwash Product Pumps.

The Electrolyte Area Sump Pump pumps spillage from the bunded area to the Backwash Product Tank or to the spillage pond as required for water recovery.

13.5.9 Bleed Treatment

A bleed stream from the production lead tankhouse is treated though a number of stages for reagent and lead recovery. These stages are described below.

Bleed Treatment – Electrowinning

Most of the lead in the bleed stream is recovered in this step in a batch electrowinning process which strips the lead concentration of the bleed treatment feed from 66 g/L to 3 g/L.

Three circulating tanks are provided in this circuit on the following basis:

- One tank is always filling with spent electrolyte bleed over a projected 16-hr cycle time. The tank's live storage is sufficient for the proposed bleed rate
- One tank operates in closed circuit with 10 electrowinning cells to strip the lead level in the bleed stream of lead over approximately 16 hours of electrowinning
- One tank of lead depleted spent electrolyte is being pumped forward for acid recovery. On completion of a batch electrowinning cycle, the 10 electrowinning cells are drained down into this tank which should now be empty.

The three duties are cycled between tanks as required.

Three circulating pumps are linked by a common header to all three tanks and operate in closed circuit with the bleed electrowinning cells. Three bleed spent electrolyte pumps are linked by a common header to all three tanks and pump bleed spent electrolyte to the acid recovery circuit.

Bleed Treatment – Acid Recovery

The Acid Purification Unit (APU) is used to de-acidify the bleed spent electrolyte solution in order to recycle the recovered MSA. The electrolyte bleed solution from the bleed cells is filtered through a multimedia filter to remove suspended solids from the solution.

Filtered solution is then transferred to the APU Feed Tank. Feed solution is pumped through the APU by the APU Feed Pump where acid is adsorbed by the ion exchange resin material while the metallic salt impurities pass through to the by-product stream which is collected in the APU By-product Tank.

This by-product stream consists of the metallic salts and a small amount of free MSA which is then pumped to the precipitation circuit by the duty APU By-product Pump. Water used for 'regeneration' of the ion exchange resin is then pumped down through the APU resin bed from the APU Elution Tank by the Elution Water Pumps so that the MSA is desorbed as a product solution. This purified solution exits the APU and is collected for re-use in the Recovered MSA Tank.

Bleed Treatment – Precipitation

This leaching circuit is the third component of the overall bleed treatment flowsheet used to provide impurity control and acid balance. This circuit treats the liquor discharge of the preceding acid recovery circuit, and reacts lime with metal methanesulfonates to precipitate a range of metal hydroxides and generate soluble calcium methanesulfonate as a reaction product.

Precipitation reactions are carried out in a single train of four agitated atmospheric leach tanks (2058-TKR-0001 to 0004). Bleed Precipitation Agitators provide the necessary mixing and solids suspension. The discharge of the last tank in the train discharges to the Bleed Precipitation Thickener (2058-THM-0001). The design makes provision to bypass any of the precipitation tanks for maintenance or de-scaling purposes.

The duty Bleed Precipitation Thickener Underflow Pump pumps thickener underflow to the agitated Bleed Precipitation Residue Filter Feed Surge Tank.

Precipitation thickener Overflow is collected in the Bleed Precipitation Thickener Overflow Tank and is pumped to the bleed leaching circuit by the duty Bleed Precipitation Thickener Overflow Pump.

The duty Bleed Precipitation Residue Filter Feed Pump discharges to the Bleed Precipitation Residue Filter. The filter cake is washed and discharged to a bin that is removed to tailings on a routine basis. The filtrate gravitates to the thickener feed.

The Bleed Precipitation Area Sump Pump pumps spillage from the bunded area to the Bleed Precipitation Tank 1 or to the spillage pond as required for water recovery.

Bleed Treatment – Leaching

This leaching circuit is the fourth component of the overall bleed treatment flowsheet used to provide impurity control and acid balance for the main part of the flowsheet. This circuit treats the liquor discharge from the preceding precipitation circuit and uses sulfuric acid to react with calcium methanesulfonate to precipitate gypsum and regenerate MSA. Strontium is also precipitated as strontium sulfate, liberating additional MSA.

Leaching reactions are carried out in a single train of four agitated atmospheric leach tanks. The discharge of the last tank in the train discharges to the Bleed Leach Thickener. The design makes provision to bypass any of the bleed leach tanks for maintenance or de-scaling purposes.

The duty Bleed Leach Thickener Underflow Pump pumps thickener underflow to the agitated Bleed Leach Residue Filter Surge Tank. Thickener O/F is collected in the Bleed Leach Thickener O/F Tank and is pumped to the MSA leach circuit by the duty Bleed Leach Thickener O/F Pump.

The duty Bleed Leach Residue Filter Feed Pump discharges to the Bleed Leach Residue Filter. The filter cake is washed and discharged to a bin that is removed to tailings on a routine basis. Filtrate gravitates to the feed to the Bleed Leach Thickener.

13.5.10 Lead Electrowinning

The lead electrowinning tankhouse is designed for a nominal lead cathode production of 70,000 tpa to match the proposed production schedule, with a maximum production rate of 80,000 tpa potentially achievable by increasing cathode current density from the design operating level of 300 A/m^2 to a maximum of 350 A/m^2. The tankhouse comprises two parallel trains of cells with the cathode handling operation located in at the end of the cell banks. One fully automatic crane is dedicated to the harvesting operation.

Filtered electrolyte is pumped to the Circulating Electrolyte Tank and then pumped by the two duty Circulating Electrolyte Pumps at 1,100 m^3/h to the electrowinning cells in the tankhouse. Plating agents (EW50 and Aloes) are fed by dosing pumps into the circulation tank.

The Circulating Electrolyte Pumps distribute the electrolyte to the electrowinning cells. The electrolyte solution within each cell is distributed via an individual manifold located at the base of each cell at a flow rate of between 1.85 L/min/m^2 and 2.24 L/min/m^2 depending on the current density. The electrolyte distribution manifolds are designed at 2.5 L/min/m^2 should higher current densities be employed in the future.

Each cell is fitted with 45 lead starter sheets and 46 DSA anodes. The tankhouse is broken into two sections with one electrical circuit overall powered by a single 230 V, 27,300 A transformer-rectifier, providing the power requirements for lead electroplating. Plating will take place over a cycle time dictated by the requirement to reach the specified cathode thickness of approximately 16 mm. Cathode thicknesses of up to 18 mm can be considered, which would result in a maximum cathode weight of 180 kg.

Cathode handling involves lifting one third (15) of the plated cathodes from each cell at a time with an overhead Lead Cathode Crane and lifting bale and transporting them to the cathode storage conveyers. At the cathode storage conveyers, the cathodes are cleaned of electrolyte with hot water sprays and the washed cathodes are transferred to conveyors feeding the cathode handling machine(s). Starter sheets that have previously been fabricated replace the cathodes removed.

Solution from each electrowinning cell overflows into a common pipe header by which the spent electrolyte gravitates to the spent electrolyte tank. The recirculating electrolyte solution is passed through a plate heat exchanger, with the cooled recirculating electrolyte being returned to the electrowinning circuit.

The unavoidable evolution of oxygen at the anodes gives rise to an effervescent bubbling effect at the electrolyte surface. This causes a ‘mist’ of lead and acid aerosols in the atmosphere above the cells. To minimize the impact on personnel and the tankhouse building, the tankhouse will be constructed with a mist capture system that serves to remove these aerosols from the tankhouse.

A critical operation impacting on the current efficiency of an electrowinning cell house is improved by detecting and correcting short circuits (‘shorts’) between anode/ cathode pairs and poor contacts. An infra-red scanner is placed on-board the lead cathode crane, which traverses the cells and allows a full infra-red scan to be transmitted to a computer system. Shorts and poor contacts can be individually identified, allowing the operator to take remedial action.

The Electrowinning Area Sump Pumps pump spillage from the bunded area to the Circulating Electrolyte Tank.

13.5.11 Lead Melting

Cathode Handling

Loaded cathodes are removed from the tankhouse and are washed to remove acidic liquor from the surface of the cathodes. The copper hanger bars are automatically stripped from the cathodes and passed to the lead starter sheet package bin for re-use. Cathodes are then transferred to the furnace at a rate of approximately 1 cathode per minute.

Furnace

A packaged lead furnace comprising a single 15 tph lead furnace supported by a 600 kW induction power unit and all ancillaries, is provided to convert lead cathode to molten lead.

The furnace is stationary, non-tilting and metal movement is carried out by immersed metal pumps. Individual metal pumps are provided for the lead ingot casting and starter sheet production duties.

The 600 kW power supply provides a melting system designed for maximum efficiency in power conversion matched to the required melting rate. The power unit can maintain full power to a well-charged furnace throughout the melt cycle with no operator adjustments to the panel.

The furnace has a strong furnace cover with a slot to allow cathode sheets to be fed one at a time into the furnace. The furnace cover can be designed to accept cathode plates directly from a horizontal conveyer or from a cathode charging device from above.

The furnace will typically melt one 180 kg cathode per minute. The 600 kW melting furnace has additional melt capacity to melt approximately one cathode every 42 seconds. This higher production capacity ensures the furnace can raise temperature quickly to maintain constant bath temperatures and permit catch-up on production if required.

A set of flexible water-cooled power leads carry power from the cabinet to the furnace coil.

The 600 kW power supply is entirely contained within a pre-wired steel cubicle with gasketed doors. The unit requires a 415 V, 3-phase, 50 Hz, 690 kVA power source. The cubicle includes the following:

- Circuit isolation system
- DC to AC inverter
- Safety isolation transformer
- Capacitor bank
- Ground/ molten leak detector
- Control system components.

The cooling system consists of one enclosed dry air cooling system and includes the following items:

- An industrial dry air cooler and fans
- Motor control center for the cooling system control
- Duty and standby cooling water recirculation pumps
- Side stream deionizer
- Emergency cooling supply.

Ingot Casting

Lead ingot casting is a fully automated process from receipt of molten lead to stacking of cast ingots. Molten lead passes via a heated launder to the star feeder at the head of the lead ingot casting line. The speed of the star feeder and the lead ingot molds passing beneath the feeder is controlled such that the flow of lead from the furnace just fills the mold. Multiple molds are clamped to a chain conveyer, which passes the molds beneath the feeder. Cooling water and air are applied to the molds after filling to facilitate solidification of the lead ingot. At the head of the conveyer, the lead molds are rotated to the upside-down position and tapped to release the ingots.

The ingots fall onto another chain conveyer that delivers the ingots to the stacking robot. The stacking robot stacks the ingots for shipment.

Starter Sheet Casting

Molten lead from the smaller induction furnace is laundered to a holding tank in which a water-cooled drum rotates to produce a thin sheet of lead. The sheet is guillotined into cathode-sized pieces and a copper hanging bar is automatically fitted to the lead sheet before the prepared cathodes are stacked on a rack for use in the lead tankhouse.

The general layout is shown in Figure 31. The flowsheets for the facility are described in Figure 32, Figure 33 and Figure 34.

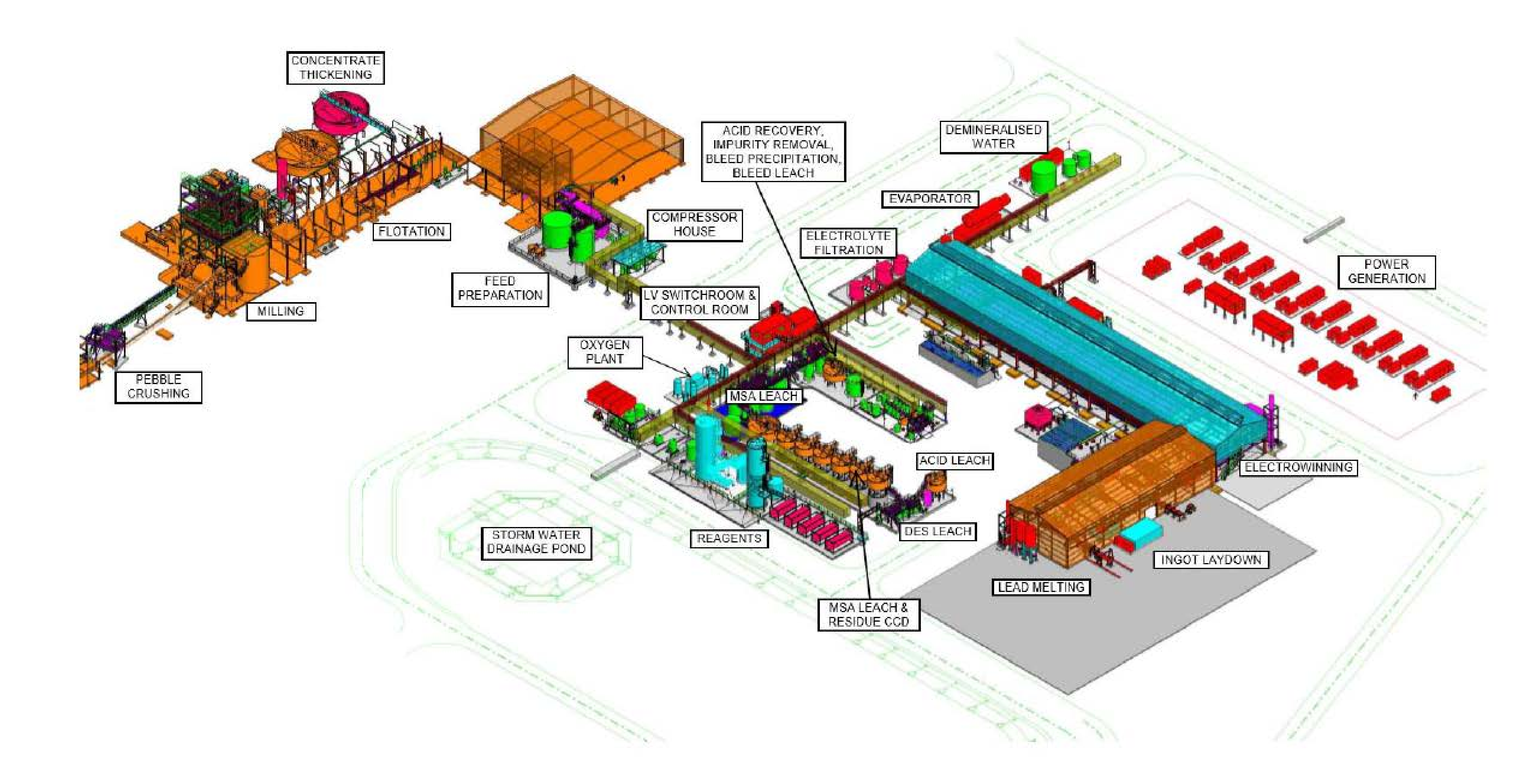

Figure 31: General layout of Flotation Concentrator and proposed Hydrometallurgical Facility

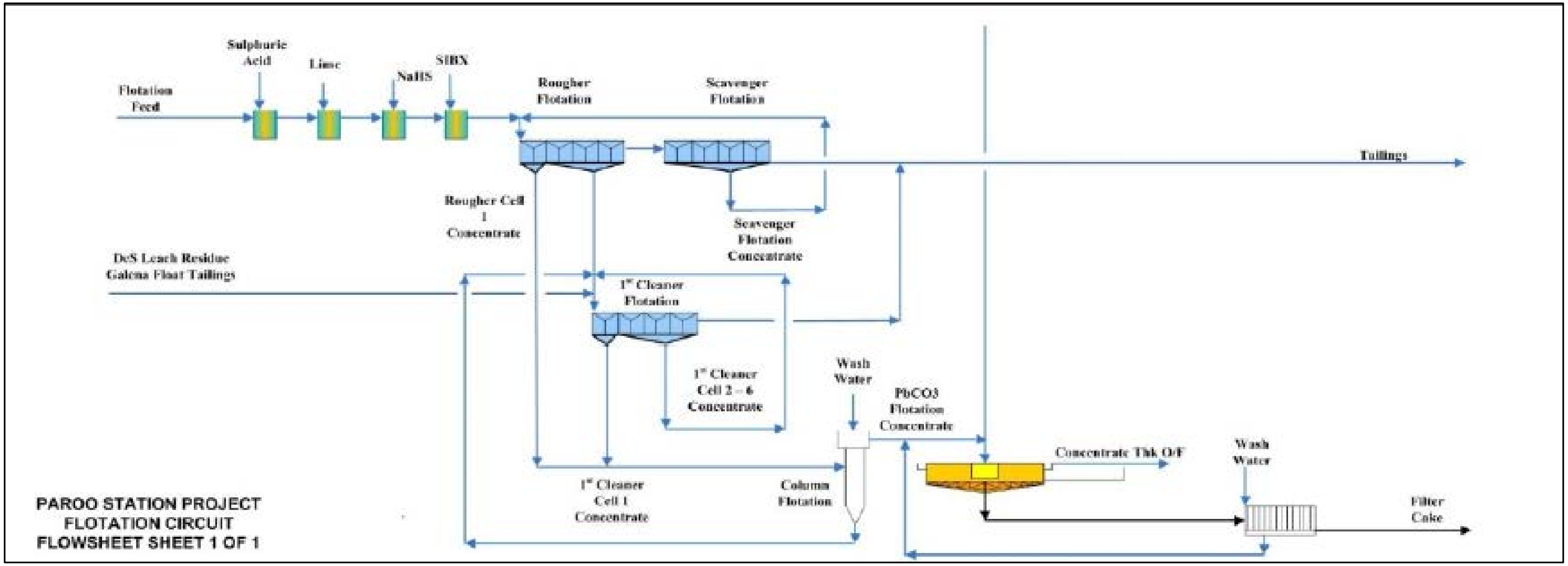

Figure 32: Paroo Station Project – Flotation flowsheet

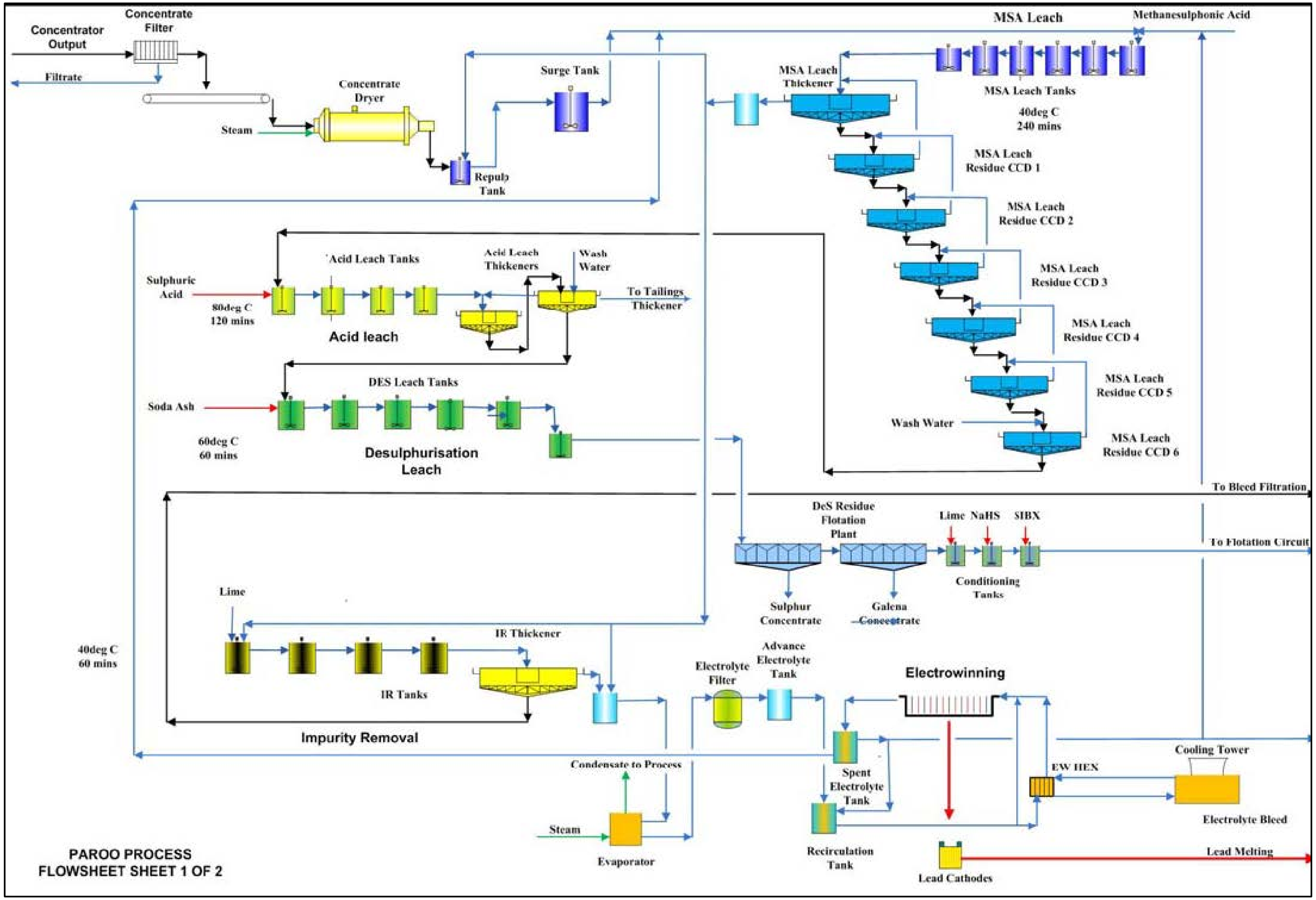

Figure 33: Paroo Station Project – Hydrometallurgical process flowsheet – Sheet 1

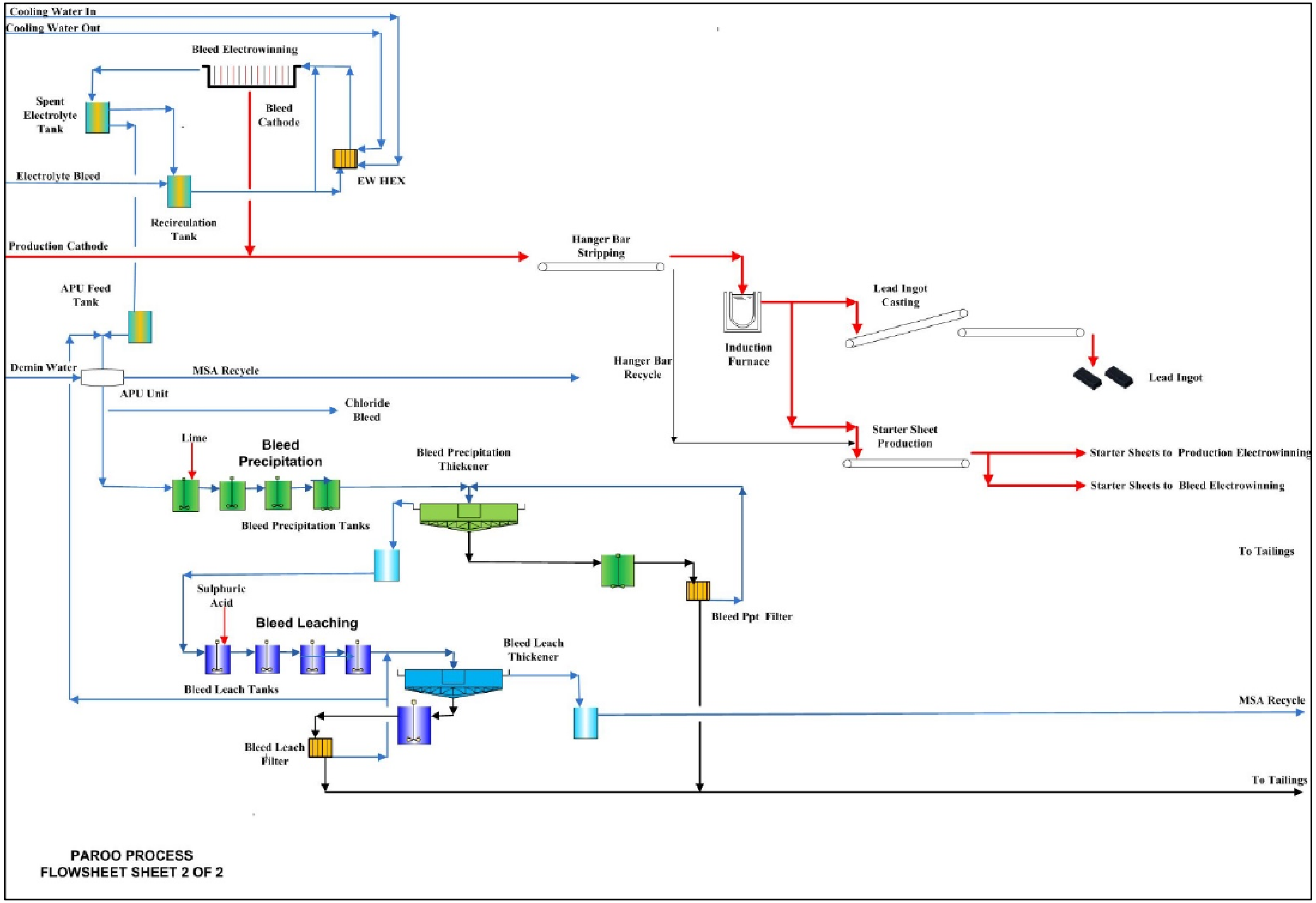

Figure 34: Paroo Station Project – Hydrometallurgical process flowsheet – Sheet 2

14 Mineral Resource Estimate

The Mineral Resource estimate for the Mine includes the main Magellan Hill deposits and the outlying Pizarro and Drake satellite deposits, located approximately 8 km south and 11 km south-west respectively from the Paroo Station Mine infrastructure.

The Magellan Hill, Pizarro and Drake Mineral Resources have been reported in accordance with the guidelines of the JORC Code (2012). Further detail can be found in previous Technical Reports (SRK 2015 and SRK 2018).

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 as part of a 2016 Mineral Resource update.

For the Magellan Hill deposits and the Pizarro deposit, no additional exploration data have been incorporated into any of the Mineral Resource estimates.

The Drake Mineral Resource was originally estimated in 2005 and reported in accordance with the guidelines of the JORC Code (2004). As part of the 2016 Mineral Resource update, the QP reviewed the Drake Mineral Resource estimate and associated documentation. The reporting of the Drake Mineral Resource was subsequently updated and reported in accordance with the guidelines of the JORC Code (2012) reporting code.

In 2019, all Mineral Resources associated with the Paroo Station Mine have used an updated reporting cut-off, as a result of the new processing opportunities and changed economics. The Mineral Resources are now reported using a cut-off of 1.3% Pb (the previous cut-off used was 2.1% Pb).

14.1 Drill hole Database

The Mineral Resource estimates for the Magellan Hill (Magellan including Gama, Cano and Pinzon) deposits and the outlying Pizarro deposit were updated in late 2014, using the available RC and diamond drill holes, including available RC grade control data. Some rotary air blast (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. Available data supporting the 2014 Magellan Hill and Pizarro estimates is presented in Table 22.

Table 22: Drill hole statistics for Magellan Hill and Pizarro - December 2014 Mineral Resource estimate

Hole type	Magellan Hill				Pizarro			
	Number of holes	Meters drilled	% holes	% meters	Number of holes	Meters drilled	% holes	% meters
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, RC and diamond drill hole data have been used to inform the 2005 Mineral Resource estimate. Additional RAB, RC and diamond drilling has been completed at Drake since the 2005 estimate which has not materially changed the Mineral Resource estimate. The 2016 review confirmed that at Drake, the correlation between RAB and RC data was sufficient to support classifying the resource as an Inferred Mineral Resource and it has been reported in accordance with the guidelines of the JORC Code (2012) reporting code. Table 23 shows the available data informing the 2005 Drake estimate and available at the time of the 2016 review.

Table 23: Drake Mineral Resource public reporting at December 31, 2015 (>2.1% Pb)

Drill Type	2005 Data			Post 2005 Data			All		
	Number	Length (m)	Grade (% Pb)	Number	Length (m)	Grade (% Pb)	Number	Length (m)	Grade (% 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).

Subsequent to this, two drilling programs have been conducted at Pizarro and Drake – in 2015 and 2018.

2015 RC Drilling

A program of RC exploration drilling on the periphery of the Pizarro and Drake deposits was conducted in early 2015. At Pizarro, distal extensions to the southern limits of the main trend were tested, with the trend shown to continue, albeit at a lower grade.

At Drake, extensions to the weaker NW–SE mineralized trend were tested with no significant economic intersections recorded.

2018 RC Drilling

A further 16 RC drill holes were drilled at Pizarro and Drake during mid-2018.

At Pizarro, six RC drill holes for 240 m were completed south of the Pizarro resource, testing a possible change in the direction of the main Pizarro trend. The drilling falls outside of the Pizarro resource limits and is not material to the Pizarro resource. The resource estimate was not updated to include this drilling.

At Drake, 10 RC holes were drilled for a total of 324 m along the Drake main mineralized trend. The program twinned historical RAB holes with RC drill holes and the resource drill spacing was partly infilled. The 2018 drilling at Drake extended the known mineralization. As an Inferred Mineral Resource, the predicted estimated lead grade correlates well with the sample assays from the 2018 drilling, as presented in Chapter 10, Table 19.

The Drake Mineral Resource estimate has not been updated for drilling completed since 2005. As further drilling is planned and mining is currently not planned for some time, the Drake resource estimate will be updated on an as required basis, going forward.

14.2 Geologic Model

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

The 2016 Mineral Resource estimate is based on 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 sills.

Magellan Hill

The overall package is tabular and subhorizontal; however, some units exhibit 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 35. A vertical exaggeration of 5 times has been applied to show detail.

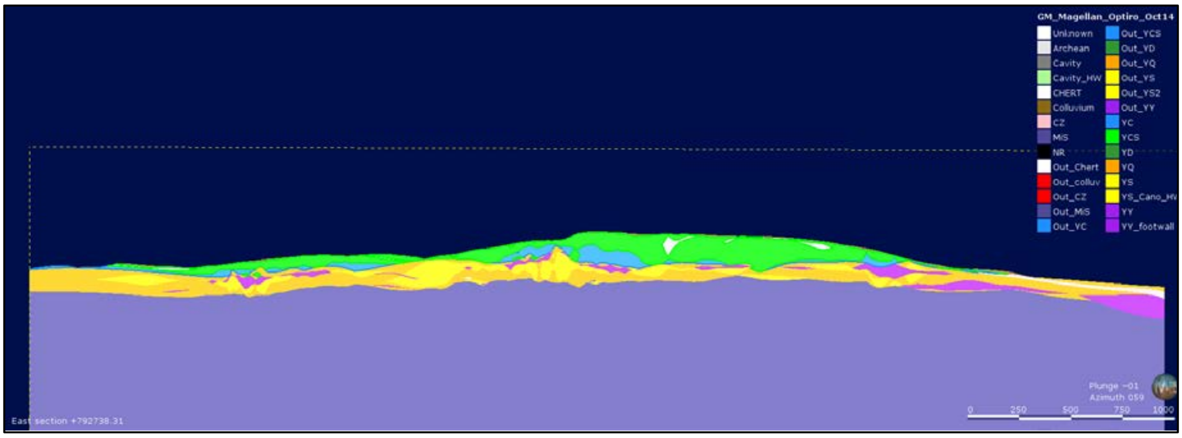

Figure 35: Magellan Hill geology model oblique cross section looking east

Pizarro

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

The resultant model is a mixture of discrete vein models that follows the 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 “deposit” contacts that enclosed 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 an “intrusive” unit.

An oblique cross section view of the Pizarro model is shown in Figure 36. A vertical exaggeration of 5 times has been used to show detail.

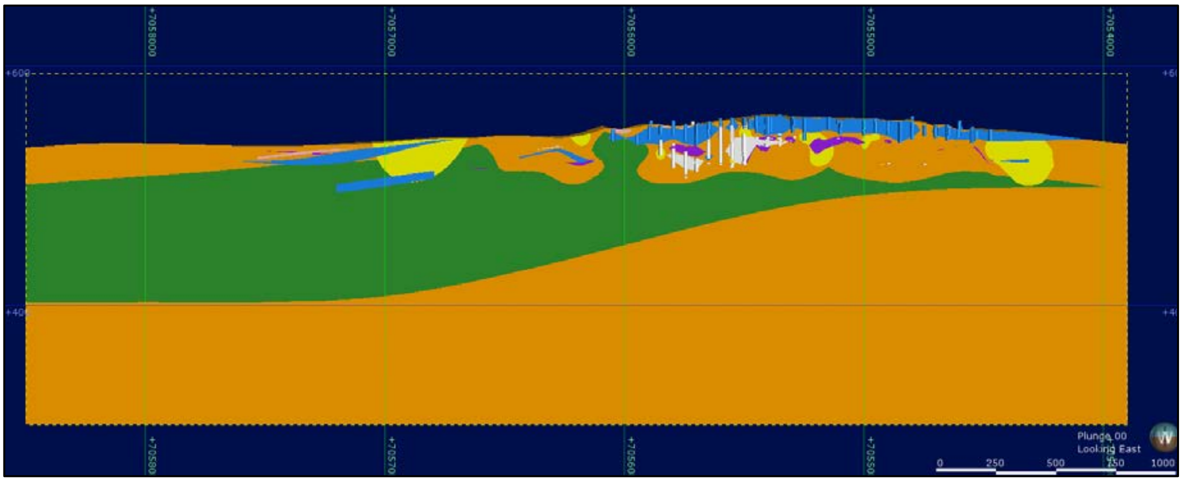

Figure 36: Pizarro geology model oblique cross section looking east

Source: Optiro (2015).

Drake

The Drake geology is informed by the drilling available up to 2005 only. The oxidized lead mineralization 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 mineralization is hosted within highly weathered remnants of the Yelma Formation and the underlying Juderina Formation sediments. The mineralization is pervasive and the current data does not provide any evidence for lithological control, other than the degree of weathering.

In the Drake deposit, the highest grade zone has a strong north-east orientation which shows similar characteristics to the Magellan Hill structural controls.

The classification of this deposit as an Inferred Mineral Resource is viewed as appropriate given the geological continuity observed by the mineralization, but there is considerable grade variation encountered by drilling. Only 40 mineralized intervals above 3% Pb have been used for this estimate.

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 mineralized 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 mineralized interval.

14.3.1 Magellan Hill Statistics

The statistics before and after compositing demonstrate that there is no significant change in the metal as a result of the compositing process. Table 24 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 mineralization. Lode 0 is the waste or non-mineralized domain.

Table 24: 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**

The statistics for the Magellan Hill data show that all domains have relatively low variance/ standard deviation and skew, with low coefficients of variation (CV). Cano and Pinzon have similar statistical parameters, which are broadly similar to parameters for Magellan, but the average and median grades are higher.

Table 25: Magellan Hill composite statistics by Zone

Lead (PB_PREF)	Global	Waste	Mineralized (+1% Pb)		
			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 deviation	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

Lead (PB_PREF)	Global	Waste	Mineralized (+1% Pb)		
			Cano	Magellan	Pinzon
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 37 provides a visual comparison of the respective domain statistics.

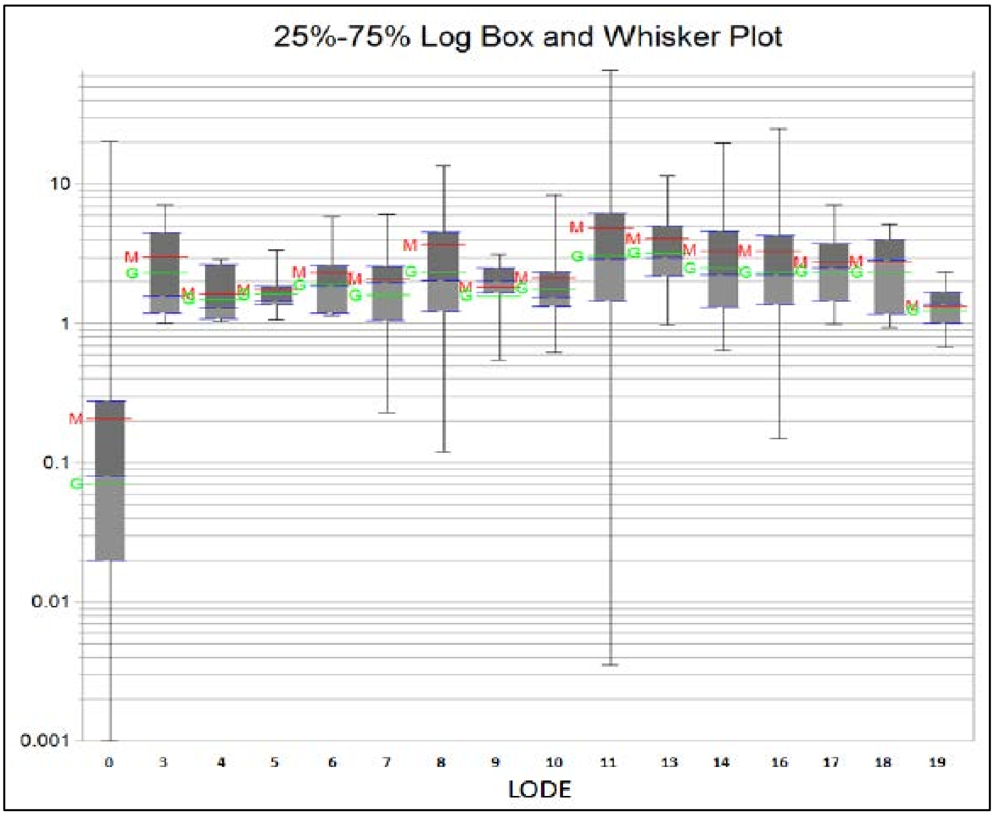

Figure 37: Magellan Hill box-and-whisker plot

Source: Optiro (2015).

14.3.2 Pizarro Statistics

An analysis of the pre- and post-composite statistics demonstrates that there has been no significant change to the metal as the result of the compositing process. Table 26 lists the Pizarro lodes and the respective number of samples within each area. Lode 1 is the major mineralized domain and Lode 0 is the waste or non-mineralized domain.

Table 26: Pizarro number of composite samples by lode code

Lode	Number of composite samples			
	Lode 0	Lode 1	Lode 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**

The statistics for Pizarro are shown in Table 27. The overall statistical parameters are similar to Magellan Hill (low variance, relatively low skew and very low CV). A combination of the high-grade lode and the relatively low variability meant that no top-cut was required.

Table 27: 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).

The box-and-whisker plot in Figure 38 provides a visual comparison of the respective domain statistics.

Figure 38: 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 28.

Table 28: Drake summary statistics – all samples – data available for 2005 Mineral Resource estimate

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

Source: Optiro (2016).

14.3.4 Assay Grade Capping

Magellan Hill

The mineralization 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 cap value is unchanged from that used in the 2011 estimate.

For the waste domain (Zone/ Lode=0), the grade distribution for the waste lode (Zone/ Lode=0) is shown in Figure 39. A cap of 1.0% Pb was applied to minimize the impact of the limited number of high grade samples (approximately 99.09% of samples have composite grades below 1.0% Pb).

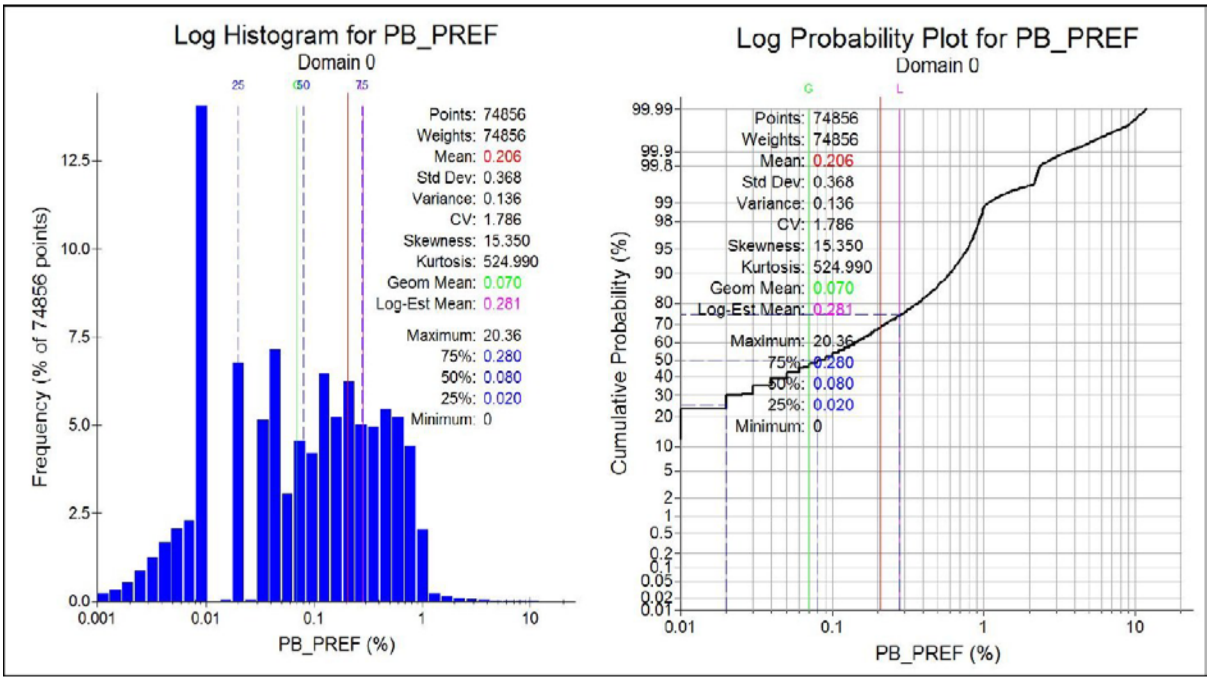

Figure 39: Magellan Hill waste lode grade distribution

Source: Optiro (2015).

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

Drake

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

14.4 Density

14.4.1 Magellan Hill

Dry bulk density for the Magellan Hill deposits was applied using a lithology-based grade-density algorithm 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 29: Magellan Hill bulk density – lithology-based algorithm

Unit	Density regression	Density range (t/m^3)
Colluvium	Pb × 0.03 + 1.60	1.60-1.99
Cavity (0 density and 0.0% Pb grade)	0.00	0.00
Silcretized quartz-clay breccia	Pb × 0.03 + 2.00	2.00-2.45
Quartz-clay breccia	Pb × 0.03 + 1.90	1.90-2.59
Dolomite	Pb × 0.03 + 2.00	2.00-2.45
Chert	Pb × 0.03 + 1.90	1.90-2.32
Clay	Pb × 0.03 + 1.70	1.70-2.32
Siltstone	Pb × 0.03 + 1.70	1.70-2.39
Sandstone	Pb × 0.03 + 2.00	2.00-2.69
Maraloou Shale	Pb × 0.03 + 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 30: 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% Pb grade)	0.0
Silicretized 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 used. The following bulk density algorithm obtained from the Cano deposit prior to 2005 was applied – dry bulk density = 1.8 + (0.04 × Pb%).

The Cano 2005 lead density correlation results in density values ranging from 1.8 t/m^3 to 2.6 t/m^3 and this density range is considered representative of the Paroo Station deposits and appropriate for the reporting of 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%–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 <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 mineralized 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 Cano, Magellan and Pinzon resource areas.

The final variogram models were then back-transformed from Gaussian to traditional variogram models. All of the variography had a horizontal dip plane, with two 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 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 Lode 2) had sufficient samples to create robust variography, and this was applied to all other domains, including the non-mineralized/ waste domain.

Three-dimensional consistent interpretations at 1% Pb included variable amounts of <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 <1% Pb subdomain (flagged as ‘NONM’) and >1% Pb subdomain (flagged as ‘MIN’) were modelled. Variography for the two domains was then undertaken. 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-mineralized 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 for selecting an appropriate search ellipse for grade interpolation (FinOre, 2005).

14.6 Block Model

The 2014 Magellan Hill block model was constructed from first principles by Optiro. The block model is reported in accordance with the JORC Code (2012) and NI 43-101 reporting guidelines as reported in SRK (2015). Optiro updated the 2014 Magellan Hill block model in 2016 to include all depletion due to mining and processing activities before the mine was placed on care-and-maintenance in January 2015.

The Pizarro block model was reported in accordance with JORC Code (2012) and NI 43-101 as reported in SRK (2015).

The Drake block model was estimated by FinOre in 2005 and reported at the time in accordance with the JORC Code (2004) and NI 43-101. 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 2005 estimate in accordance with the JORC Code (2012) reporting guidelines.

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 30 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 16 January 16, 2015, and processing of ROM stocks ceased on the February 2, 2015 (Optiro, 2016).

In 2016, Optiro depleted the block model and stockpiles, updating the remaining Mineral Resources and available surface stocks, to December 31, 2015, as detailed below.

14.6.2 Magellan Hill

The 2014 Mineral Resource for the Paroo Station deposits was initially depleted for mining to November 30, 2014. Between November 2014 and the cessation of mining on 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 February 2, 2015, when processing ceased.

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

The updated model is the same Datamine 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).

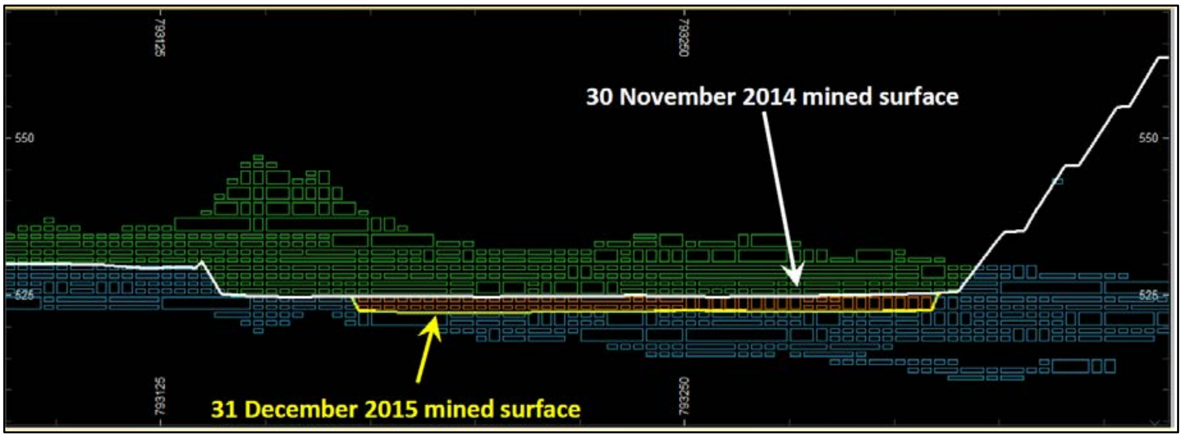

Figure 40: Section 7063260 mN, looking west through Magellan pit showing depleted block model and mined surfaces

Source: Optiro (2016).

14.6.3 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.

14.6.4 Drake

There has been no mining at Drake and there have been no changes to the Drake block model since the original Mineral Resource was estimated in 2005. However, additional documentation has been prepared to support the reporting of the Mineral Resource in accordance with the JORC Code (2012) reporting code. Additional RC drilling has been completed at Drake in 2018 but the Mineral Resource estimate has not been updated.

14.6.5 Magellan Hill Block Model

The Magellan, Cano and Pinzon deposits were modelled in a single block model, ‘Magellan Hill’. The block model prototype parameters are shown in Table 31. These parameters were derived by kriging neighborhood analysis (KNA) testing in 2014. Mineralization was defined by a 1% Pb boundary which was treated as a ‘hard’ estimation boundary.

Table 31: 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).

14.6.6 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 (drill hole spacing greater than 100 mE × 100 mN) was estimated using 100 mE ×100 mN × 5 mRL parent cell size. The mineralization and adjacent areas supported by closer-spaced drilling (nominally less than or approaching 50 mE × 50 mN) was estimated using the KNA-defined parent cell size of 25 mE × 25 mN × 2.5 mRL (Table 32).

Mineralization was defined by a 1% Pb boundary, which encapsulated a high grade +10% Pb boundary. All mineralization boundaries were treated as ‘hard’ estimation boundaries.

Table 32: 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

14.6.7 Drake Block Model

Parent cells were created with dimensions of 25 mE × 25 mN × 2.5 mRL. Sub-blocking of the model to 5 mE × 5 mN × 2.5 mRL was completed to provide extra control to the volume reporting. Further subcelling 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 constrained by the area and lode fields. The area field 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 marking samples within the overlap area to both adjoining areas (Figure 41).

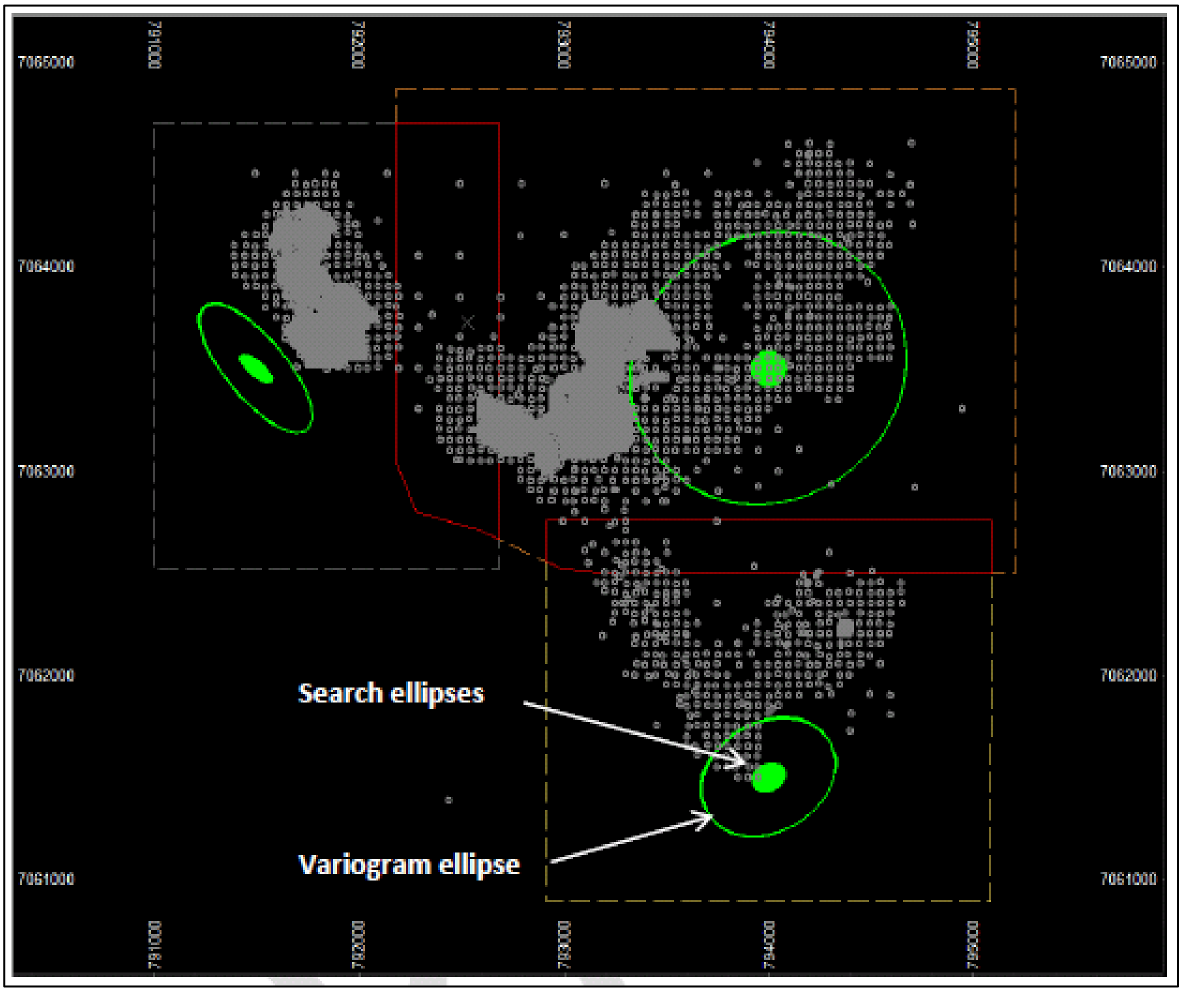

Figure 41: Magellan Hill expanded estimation overlap areas (red), search (solid discs) and variogram ellipses (lines)

Source: Optiro (2015).

A multi-pass search method was used for grade estimation as summarized in Table 33.

Table 33: Magellan Hill search parameters

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

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

For any cell (whether mineralized 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 grade of 0.2% Pb was assigned.

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

14.7.2 Pizarro

Grade estimation for the >1% Pb mineralized domain used a categorical indicator estimation method to differentiate very low grade from elevated mineralized 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 strategy, with the ellipse orientation controlled by a dynamic anisotropy process.

A threshold of 0.5 was used to discriminate <1% from >1% Pb subdomains within the overall mineralized domain. 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 drill hole was used.

Parent cell estimation was used for all estimates at Pizarro. The search parameters are summarized in Table 34.

Table 34: Pizarro search parameters

Zone/ Lode	Datamine Rotations 3-1-3	First Pass	No. of Samples	Second Pass	No. of Samples	Third Pass	No. of Samples	Dynamic Anisotropy
		Search (1-2-3)		Search (1-2-3)		Search (1-2-3)		
Categorical Indicator	0, 0, 55	175 × 175 × 5	4 to 12	262.5 × 262.5 × 7.5	4 to 12	525 × 525 × 15	4 to 12	Yes
SUB_DOM = MIN	0, 0, 60	200 × 50 × 15	8 to 32	300 × 75 × 22.5	8 to 32	600 × 150 × 45	4 to 32	No
SUB_DOM + NONM	0, 0, 60	200 × 50 × 15	8 to 32	300 × 75 × 22.5	8 to 32	600 × 150 × 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 × 80 × 3 m was used, and composited samples were length-weighted during the grade interpolation process.

No more than three samples from any one drill hole were used per block estimate and all subcells received the 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.

Parent cell estimation was used for all estimates at Drake.

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.

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

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

In January 2019, Optiro revised the cut-off grade for resource reporting down to 1.3% Pb from 2.1% Pb. Stockpiles of mineralized waste ('blue') constructed during mining from material between 1.2% Pb and the then-operating cut-off of 2.1% Pb have an average grade of 1.68% Pb. These are available for future processing and were incorporated into the current Mineral Resource.

Table 35 shows a summary of stockpile inventory as at January 2019.

Table 35: Stockpile inventory as at January 2019

Stockpile	Ore (t)	Grade (% Pb)	Contained Lead (t)
ROM Ore Stockpiles	1,071,812	3.29	35,226
Green Stockpiles	449,611	2.39	10,756
Blue Stockpiles	1,405,138	1.68	23,641
Total stockpiles (including ROM fingers)	**2,926,561**	**2.38**	**69,622**

Source: Optiro (2016, 2019).

Note: 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 an 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 42).

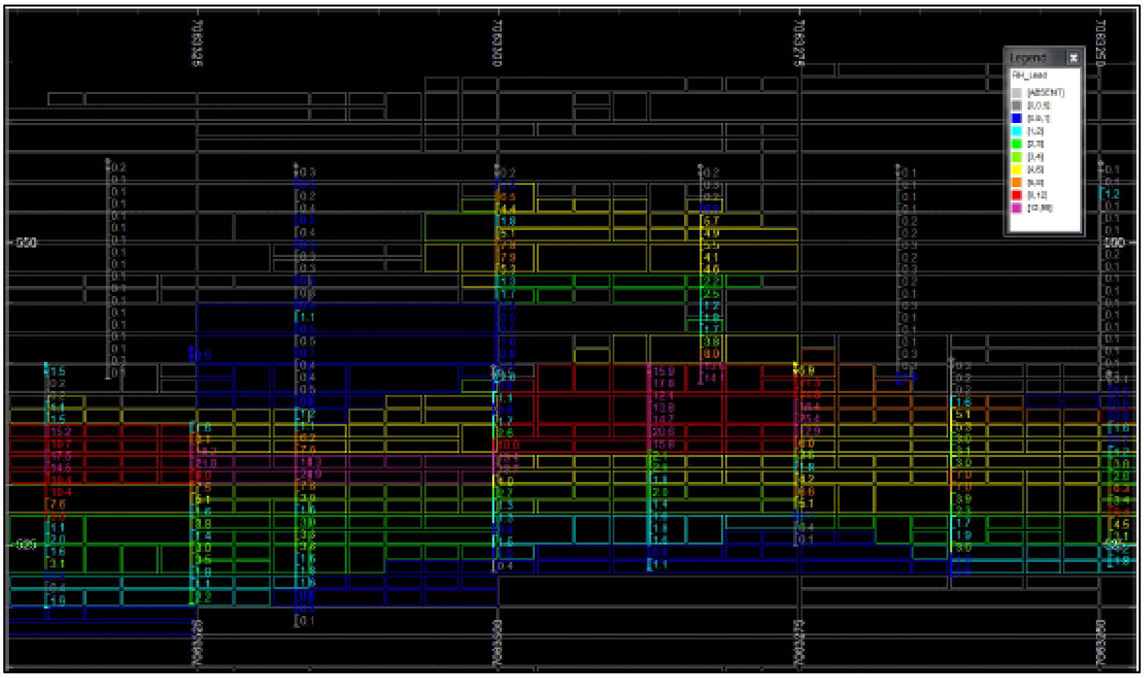

Figure 42: Magellan section 793066 mE showing composite samples and block model

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 43) which identified no significant concerns.

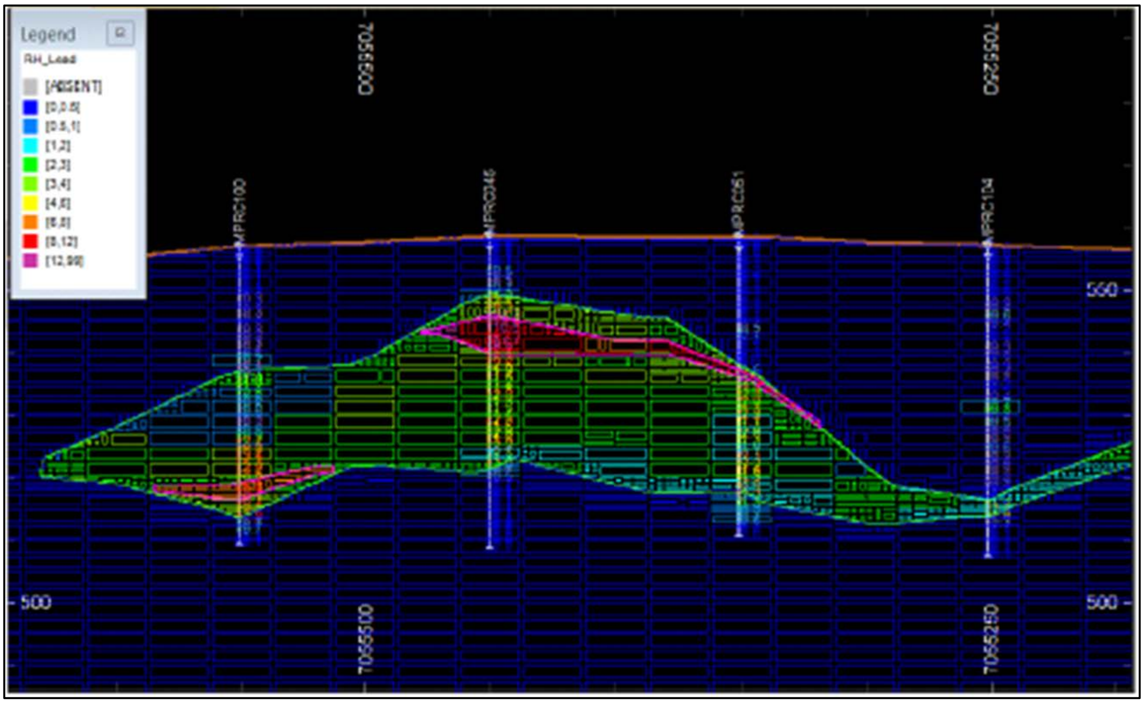

Figure 43: Pizarro section showing composite sample and block model

Source: Optiro (2015).

Drake

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

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 37).

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. areas that were extrapolated.

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 in 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 38).

Drake

For the Drake deposit, the comparative statistics between the composite and model averages are presented in Table 36. There is good correlation between the composite samples and the estimated grade. No further validation was undertaken for Drake.

Table 36: Drake comparative statistics

	Grade (% Pb)			Relative difference
	Composite Average	Model Average	Difference	
+1% mineralized domain	2.84	2.98	0.14	4.9%

Table 37: Global validation – Magellan Hill

Lode	Sample Average		Model Average								Relative Difference									
			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).

Table 38: 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).

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 44 for Lode 3.

The swath plots for the main mineralized lode (Lode 3) were also compared against the declustered averages, using a cell declustering approach with a cell size of 25 (X) × 25 (Y) × 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.

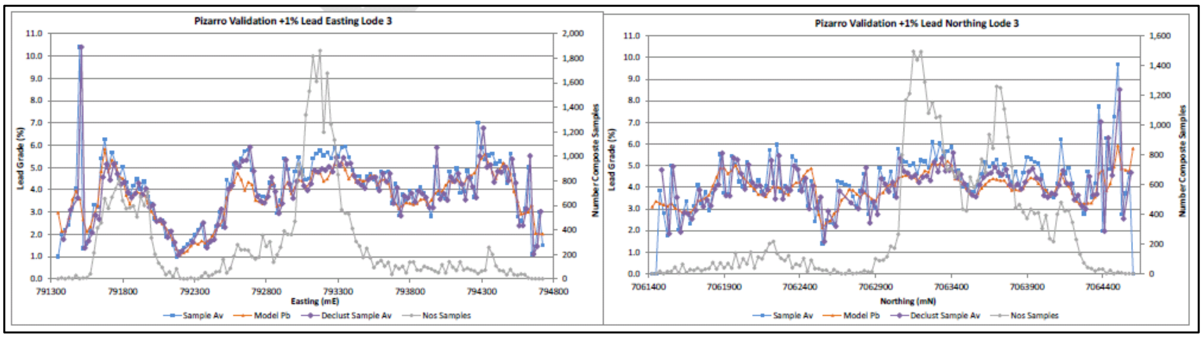

Figure 44: Swath plots for Lode 3 – Magellan Hill

Source: Optiro (2015).

Pizarro

Swath plots by easting and northing to test that sample trends had been maintained during estimation were prepared. The plots for Lode 1 are shown in Figure 45.

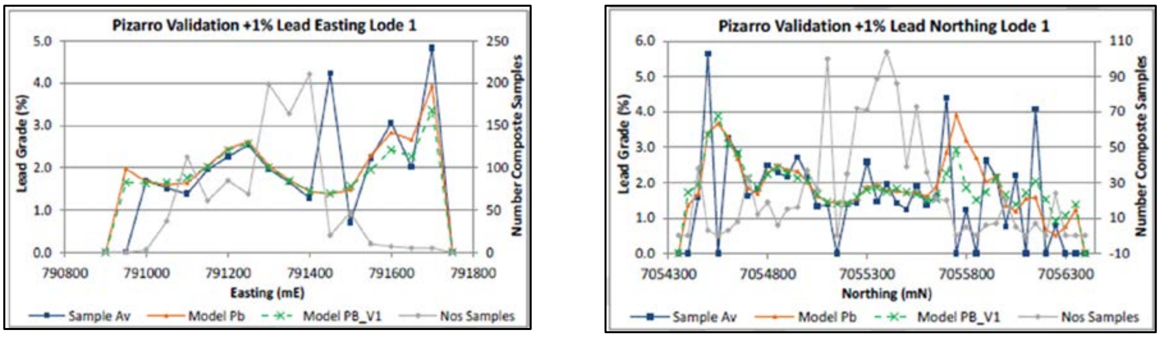

Figure 45: Swath plots for Lode 1 showing linear and categorical indicator estimates – Pizarro

Source: Optiro (2015).

Overall, the validation identified no errors with the block model and the block model correlates well with the available information.

14.9 Resource Classification

14.9.1 2014 Estimation

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 summarized in Table 39.

Table 39: Mineral Resource classification criteria 2014

Data quality and intrinsic value	Search pass	Hole spacing (m)	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 × 25	High	Measured
		<50 × 50	Moderate	Indicated
		<100 × 100	Low	Inferred
		>100 × 100	NA	Unclassified
	2	<100 × 100	NA	Inferred
		>100 × 100		Unclassified
	3	<100 × 100	NA	Unclassified
		>100 × 100		Unclassified
	4	NA	NA	Unclassified
Maraloou Shale		Unclassified		

Source: Optiro (2015).

Mineralization within the Maraloou Shale or similar basal unit is unclassified (not a Mineral Resource) due to the recognition of poor metallurgical recovery.

Magellan Hill

Figure 46 depicts a plan view of the Magellan Hill model colored 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 Measured Mineral Resources. Indicated Mineral Resources are those areas informed in search pass 1 and informed by sampling spaced less than 50 × 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 mineralization is too narrow to support the eventual economic extraction.

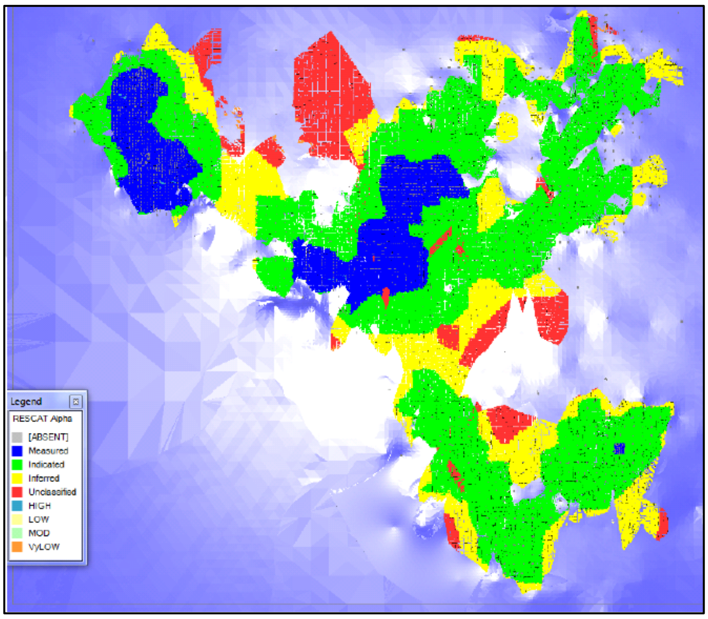

Figure 46: Plain view model colored by confidence/ classification showing top of Maraloou shale and drilling – Magellan Hill

Source: Optiro (2015).

At Pizarro, 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 Resource at Pizarro (Figure 47). Where the estimate is informed in the first pass and the drilling density is less than 50m × 50m, there is sufficient confidence to classify the mineralization as an Indicated Mineral Resource.

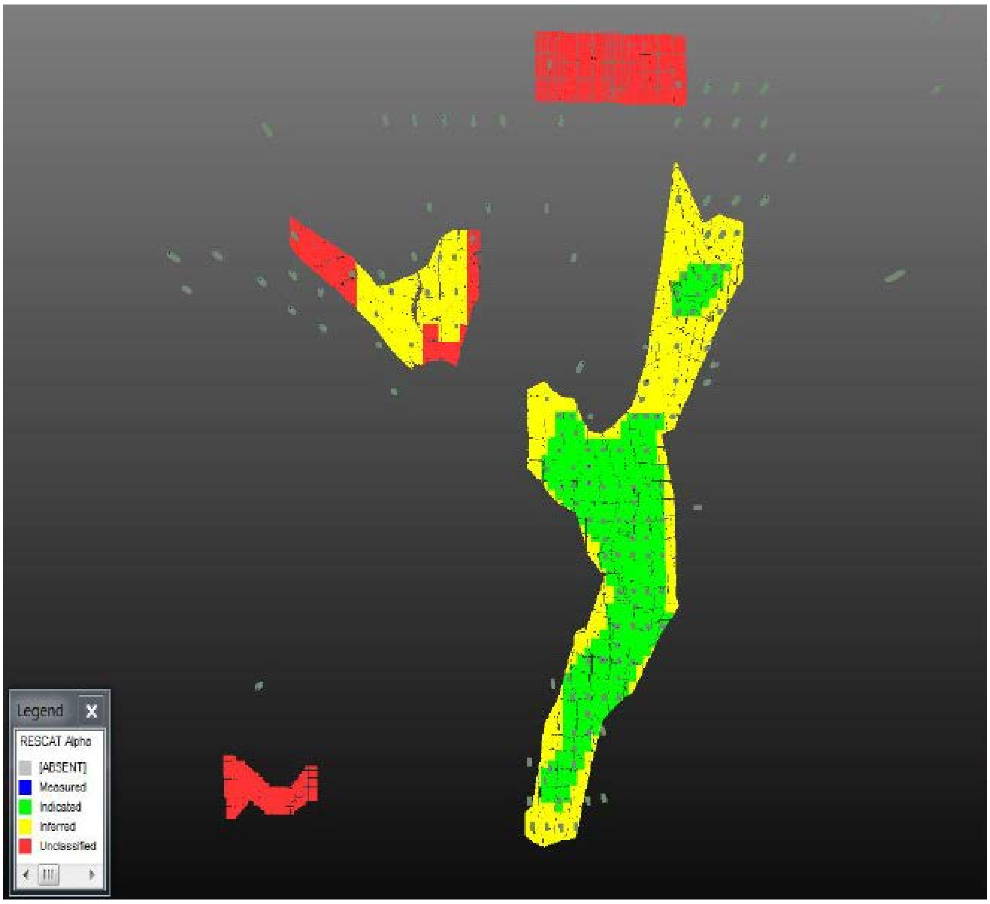

Figure 47: Model plan view colored by confidence/ classification and drilling – Pizarro

Source: Optiro (2015).

Drake

The Drake deposit has been classified in accordance with the JORC Code (2012) as an Inferred Mineral Resource, as the 2005 block model lacked detailed topography and because geological and grade continuity have not been fully demonstrated. As infill drilling confirms the assumed geological and grade continuity, there are reasonable expectations for the resource classification to be upgraded with additional drilling (Figure 48).

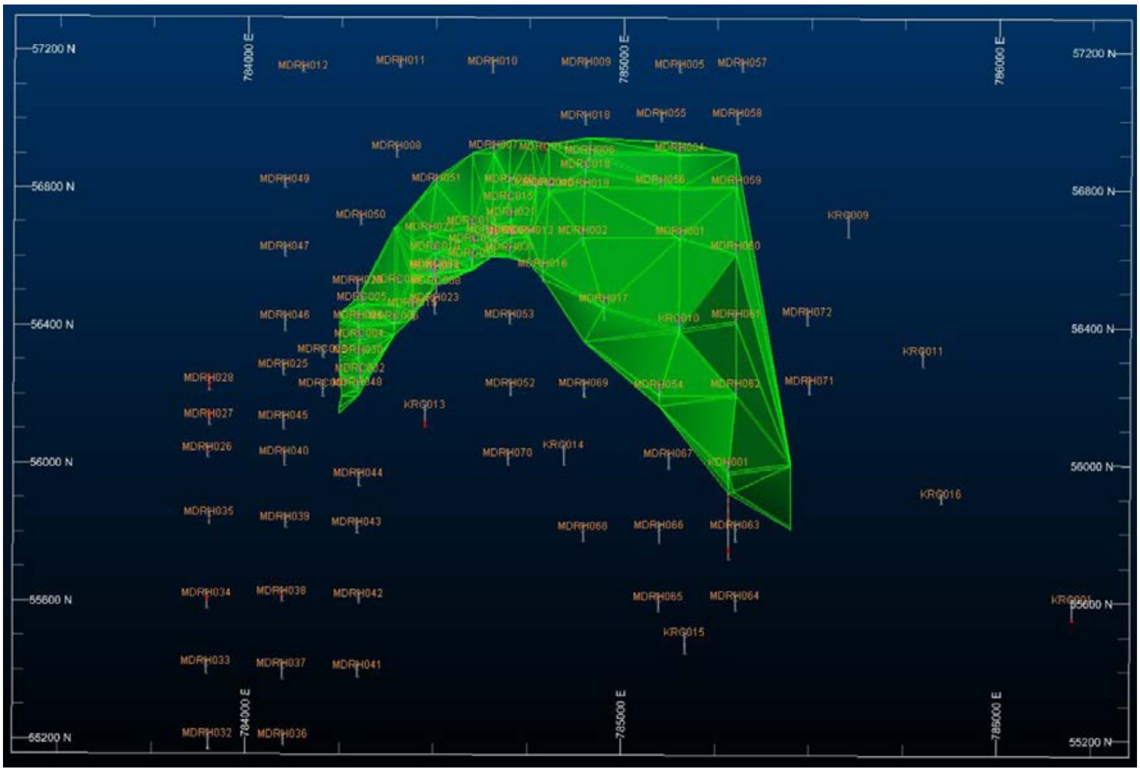

Figure 48: Plan view of Drake deposit showing available drilling as at 2005 and Inferred Mineral Resource (green polygon)

Source: Optiro 2016 after FinOre 2005.

14.9.2 Revision of Mineral Resource Reporting Cut-off Grade

During previous operating periods, a 2.1% Pb cut-off was appropriate for the reporting Mineral Resources as it suitably reflected the prevalent economics of the operation, production and sale of a mixed-oxide lead carbonate concentrate.

Following completion of the DFS Update conducted in 2018, the Mineral Resource reporting cut-off grade was revised from 2.1% Pb to 1.3% Pb to reflect the updated predicted economics of the operation.

Optiro reviewed the available documentation and costs from the DFS Update. The review assessed the proposed hydrometallurgical processing route, which lowers of overall costs and increases revenue by the sale of lead ingot and the subsequent impact on the reporting of the Mineral Resources. Changes to the geological models for the Magellan Hill, Pizarro or Drake deposits to facilitate the change of reporting cut-off grade, were not required.

The new 1.3% Pb cut-off grade reflects an appropriate balance between the current understanding of the geology/ mineralization and the likely life of mine economics (Optiro, 2019). The new Mineral Resource tabulation prepared is presented in Section 14.10.

14.10 Mineral Resource Statement

The Mineral Resource inventory was updated in January 2019 to reflect a lowering of the reporting cut-off grade (Table 40).

The Mineral Resource estimate includes the main Magellan Hill deposits of Magellan (now including Gama), Cano and Pinzon and the outlying Finlayson Range satellite deposits, Pizarro and Drake, located approximately 8 km south and 11 km south-west of the existing Mine infrastructure respectively. Collectively, the five areas are known as the ‘Paroo Station Mine deposits’.

The Mineral Resource estimate for all Paroo Station Mine deposits are reported under the JORC Code (2012).

The January 2019 Mineral Resource estimate includes all depletion due to mining and processing activities when the Mine was put onto care-and-maintenance during January 2015 due to low commodity prices. The reporting cut-off grade was lowered to 1.3% Pb in January 2019. Stockpiles have been tabulated from actual mine production data.

No new data from drilling or other exploration work has been added to the Mineral Resource estimate, which, other than depletion and revision of the cut-off grade, remains unchanged from the 2014 estimate.

Table 40: Mineral Resource estimate as at February 15, 2019

Deposit	Resource Category	Tonnes (Mt)	Grade (% Pb)	Contained Pb Metal (kt)
Magellan (including Gama)	Measured	4.5	4.2	185
	Indicated	14.5	4.3	625
	Total Measured + Indicated	19.0	4.3	810
	Inferred	3.3	3.9	130
Cano	Measured	1.6	3.4	55
	Indicated	2.1	2.4	50
	Total Measured + Indicated	3.7	2.9	105
	Inferred	0.8	2.3	15
Pinzon	Measured	0.1	6.1	5
	Indicated	9.5	4.1	390
	Total Measured + Indicated	9.5	4.1	395
	Inferred	2.0	3.5	70
Pizarro	Measured	0	0.0	0
	Indicated	4.6	3.1	140
	Total Measured + Indicated	4.6	3.1	140
	Inferred	2.0	2.8	55
Drake	Inferred	3.7	3.4	125
Stockpiles	Measured	2.9	2.4	70
Total	Measured	9.1	3.5	315
	Indicated	30.6	3.9	1,205
	Total Measured + Indicated	39.7	3.8	1,520
	Inferred	11.7	3.4	396

Source: Optiro (2019).

Notes:

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 1.3% Pb and depleted for mining to December 31, 2015. There has been no mining or processing of material during the 2016-2018 calendar years.
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 Pty Ltd, and a 'Competent Person' as defined by the 2012 JORC Code. He is a 'Qualified Person' ('QP') for purposes of NI 43-101 and 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 January 2019 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.
8. All resources are reported inclusive of Ore Reserves/Mineral Reserves.

14.11 Mineral Resource Sensitivity

14.11.1 Mineral Resource Classification

For all deposits, the resource categories are unchanged from previously reported Mineral Resources. The resource category applied is logical, consistent and is closely linked to the spatial coverage of the collected data (Figure 49).

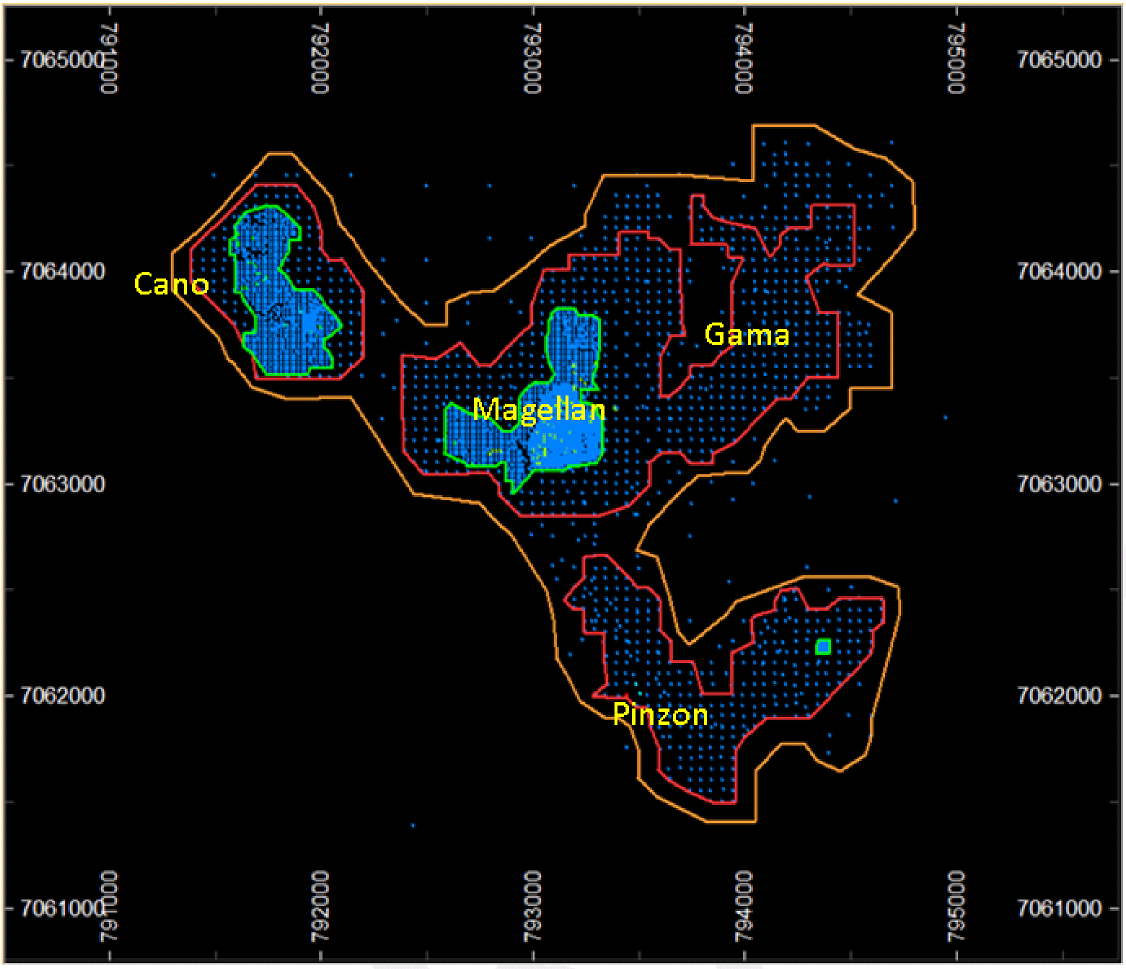

Figure 49: Magellan Hill Mineral Resource classification – December 31, 2014

Source: Optiro (2014).

Notes: Green = Measured; Red = Indicated; Orange = Inferred.

14.11.2 Inventory Changes from 2015 to 2019

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

Since 2014, the Magellan Hill deposits haven been updated for mining and processing depletion to the Paroo Station Mine's placement on care-and-maintenance in January and February 2015.

The 2005 Drake resource estimate has been reviewed and reported in accordance with the JORC Code (2012).

In 2018, the DFS Update demonstrated an improved cost and revenue structure for Mine, driven by the change to on-site hydrometallurgical processing. In 2019, the Mineral Resource reporting cut-off grade has been lowered to 1.3% Pb (from 2.1% Pb in 2017), to reflect the changed economics. This has added 10.8 Mt at 1.7% Pb for a total of 190 kt of lead metal.

Table 41: Change in Mineral Resources from December 2017 at 2.1% Pb cut-off to February 2019 at a 1.3% Pb cut-off

Deposit	Resource Category	Tonnes (Mt)	Grade (% Pb)	Contained Pb metal (kt)
Magellan Hill stockpiles	Measured	1.4	1.7	24
Magellan Hill (Magellan, Cano and Pinzon)	Measured	1.4	1.7	24
	Indicated	3.4	1.8	60
	Total Measured + Indicated	4.8	1.8	84
	Inferred	1.5	1.8	26
Pizarro	Measured	0		
	Indicated	1.4	1.8	26
	Measured + Indicated	1.4	1.8	26
	Inferred	0.8	1.7	14
Drake	Measured	0		
	Indicated	0		
	Measured + Indicated	0		
	Inferred	1.0	1.6	16
All	Measured	2.8	1.7	48
	Indicated	4.8	1.8	85
	Measured + Indicated	7.6	1.8	133
	Inferred	3.3	1.7	56

The QP considers the updated lead reporting grade of 1.3% Pb to be appropriate for the given mineralization style, planned mining and processing opportunities and future metal price assumptions.

14.12 Relevant Factors

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

15 Mineral Reserve Estimate

The Paroo Station 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 historical as well as more recent production information, including current cost, revenue and metallurgical recoveries generated as part of the DFS Update, 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 an updated open pit optimization incorporating accepted product pricing, current project costs and operational parameters. The open pit optimization underpinned a revised mine staging, mine designs and mine production scheduling.

The Mineral Reserve estimate was developed under the 2012 Edition of the JORC Code. The CIM recognizes the 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° 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 Mine*, Peter O'Bryan & 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.

15.1.2 Hydrological

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 end of expected LOM. A hydrological review is required to confirm there will be no likely adverse impact on the stability of the pit walls.

15.1.3 Open Pit Optimization

Open pit optimization was used to identify the optimum economic pit shape based on the highest project cashflow achievable. The pit optimization process seeks a solution to a complex 3D mathematical relationship involving the mineral resource model, geotechnical slope guidelines, product revenue, project constraints, modifying factors and costs. The key inputs into the optimization process include:

- Product prices
- Mining costs
- Processing, realization and administration costs
- Process recoveries
- Pit slope angles
- Diluted resource model.

The mineral resource model was converted to a mining model by a process of regularization to account for dilution and ore losses. The diluted model has then been used as the basis for optimization, pit evaluation and scheduling. Further preparation included adding cost, recovery, royalties and revenue drivers to individual blocks within the model. Net smelter return (NSR) inputs and formulas required to calculate the economic value for each block were used in the optimization process.

A net present value (NPV) discount rate of 10%, which is comparable with Australian projects of similar scale and size, has been applied.

The Whittle Four-X software package was used to develop the pit optimization 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 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.

The final pits were designed with the Magellan Hill pits divided into nine progressive pit stages, and the Pizarro pit divided into three progressive pit stages, to assist with achieving the production schedule targets. Each stage has its own ramp access while following the minimum mining width, so the stages can be mined independently.

Plan views of the Cano, Magellan and Pinzon deposits are shown in Figure 50 to Figure 52, and a plan view of all pit stages is shown in Figure 53. The pit design for Pizarro is shown in Figure 54.

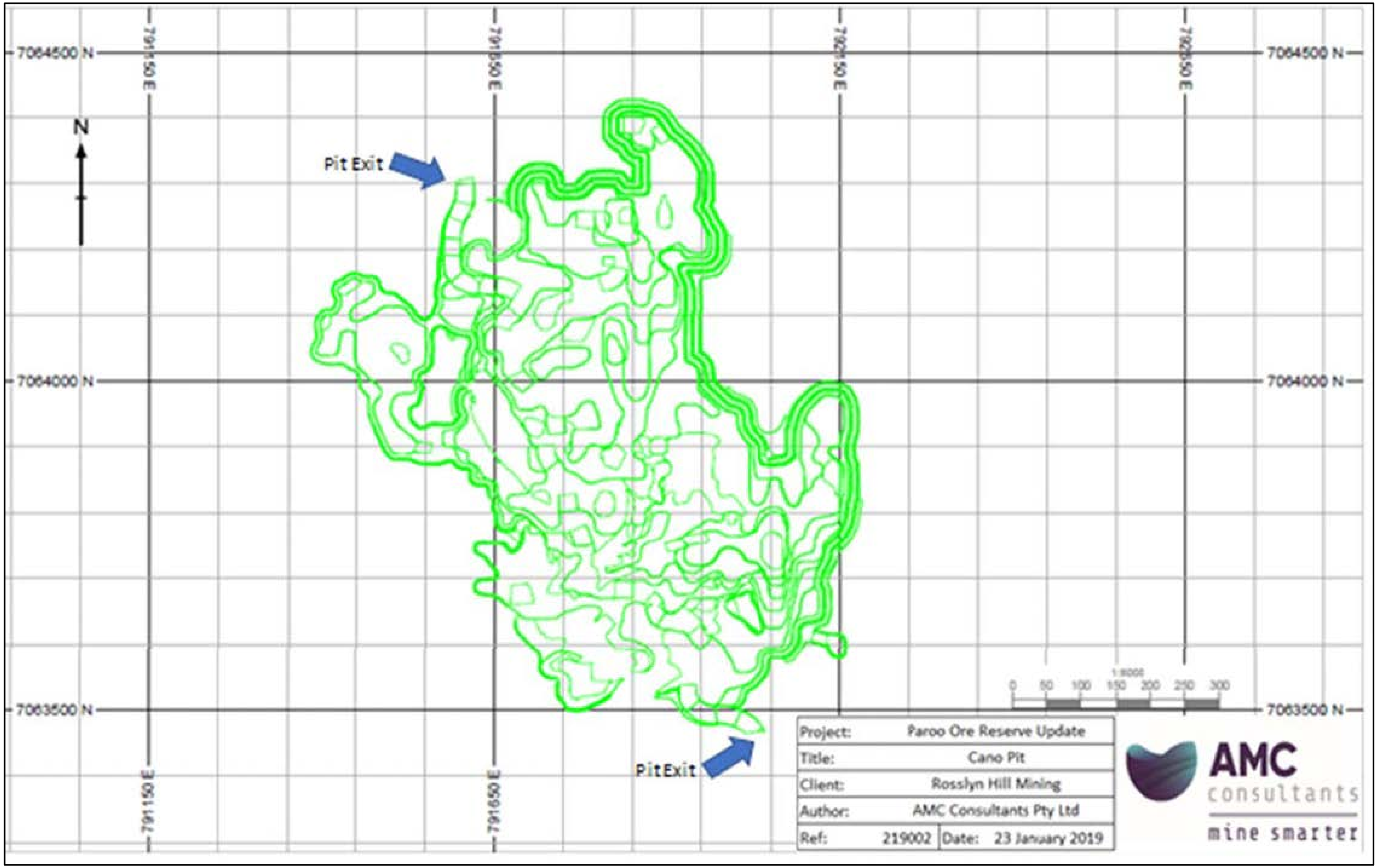

Figure 50: Pit design for Cano

Source: AMC (2019).

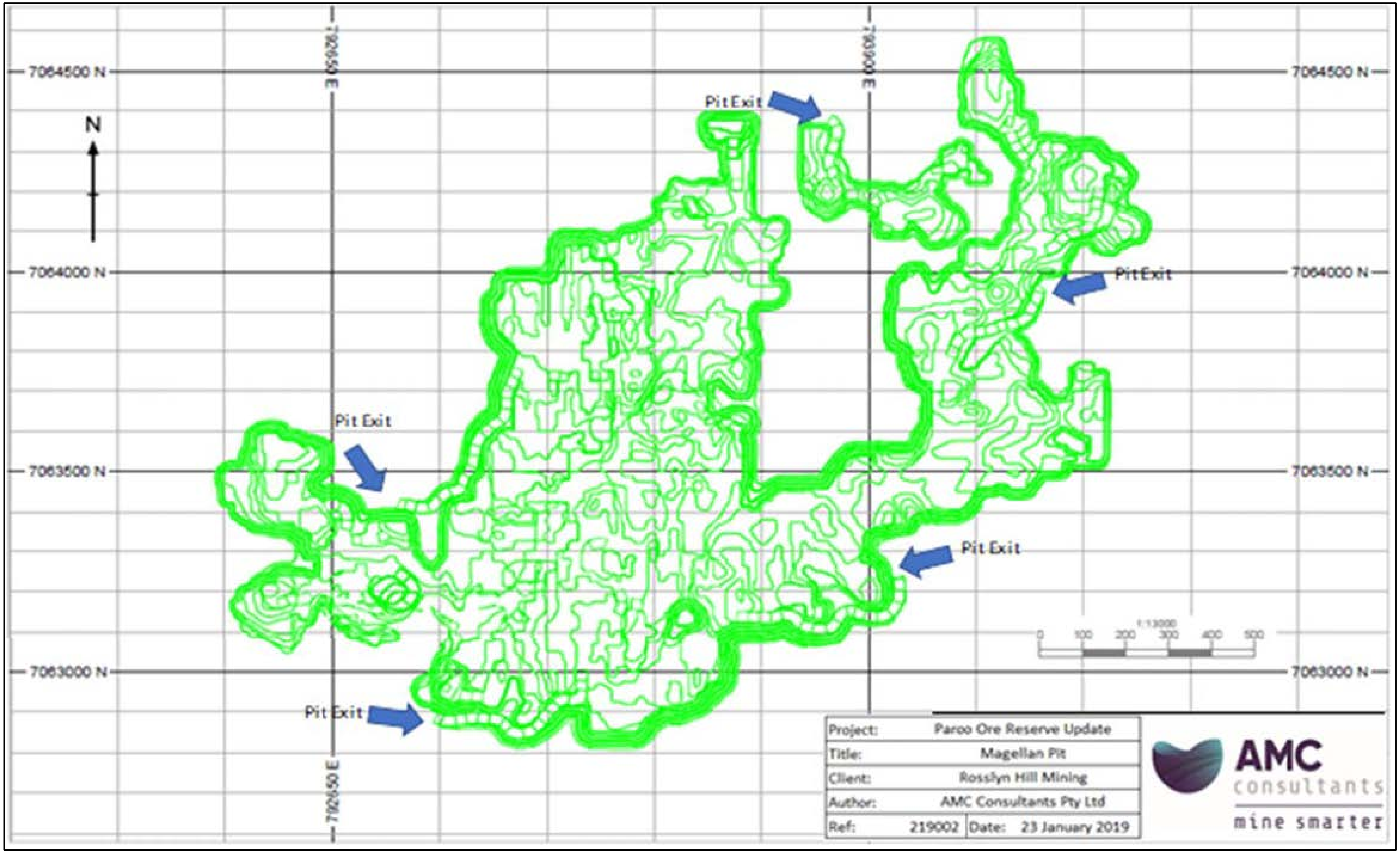

Figure 51: Pit design for Magellan

Source: AMC (2019).

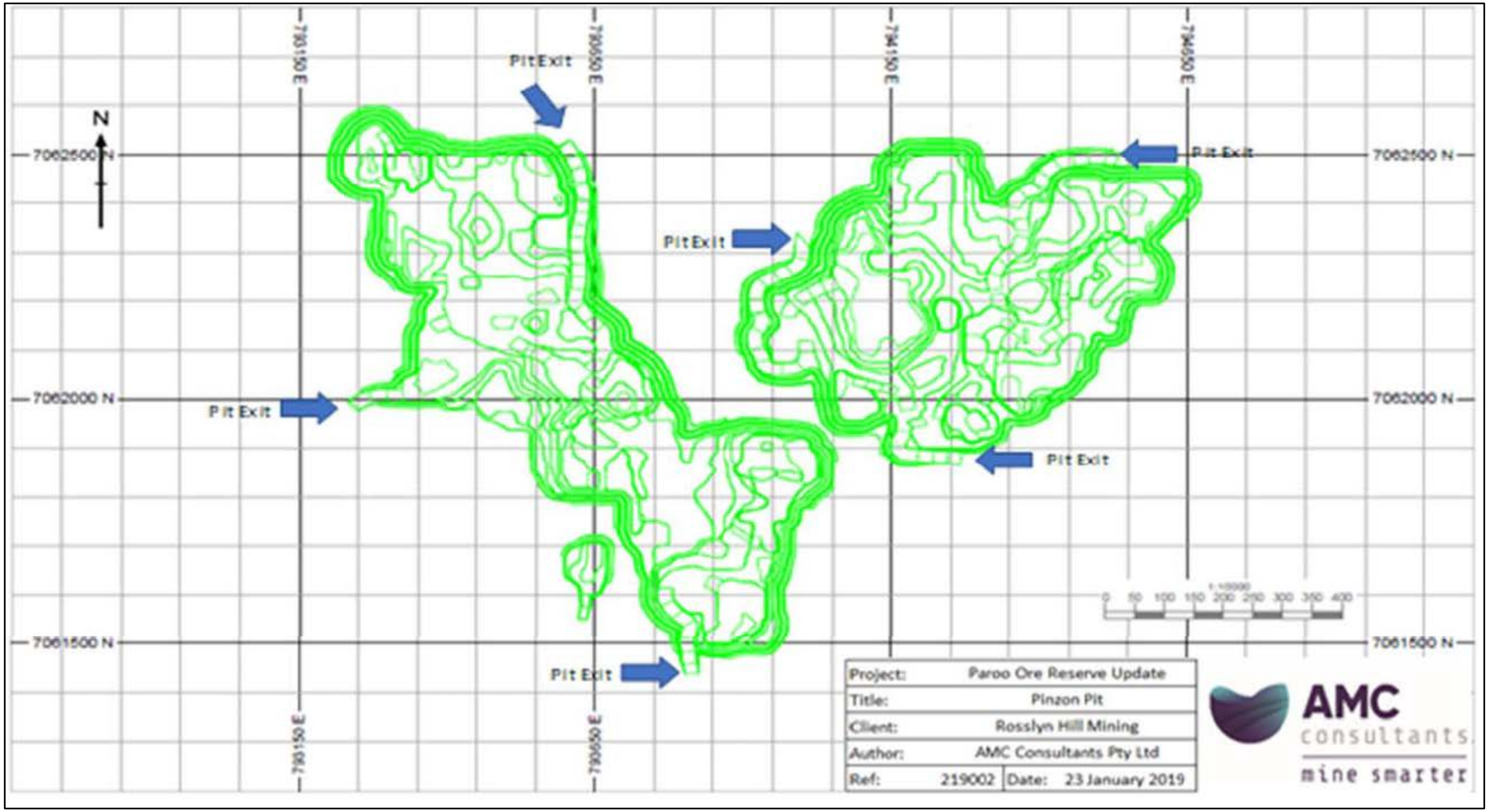

Figure 52: Pit design for Pinzon

Source: AMC (2019).

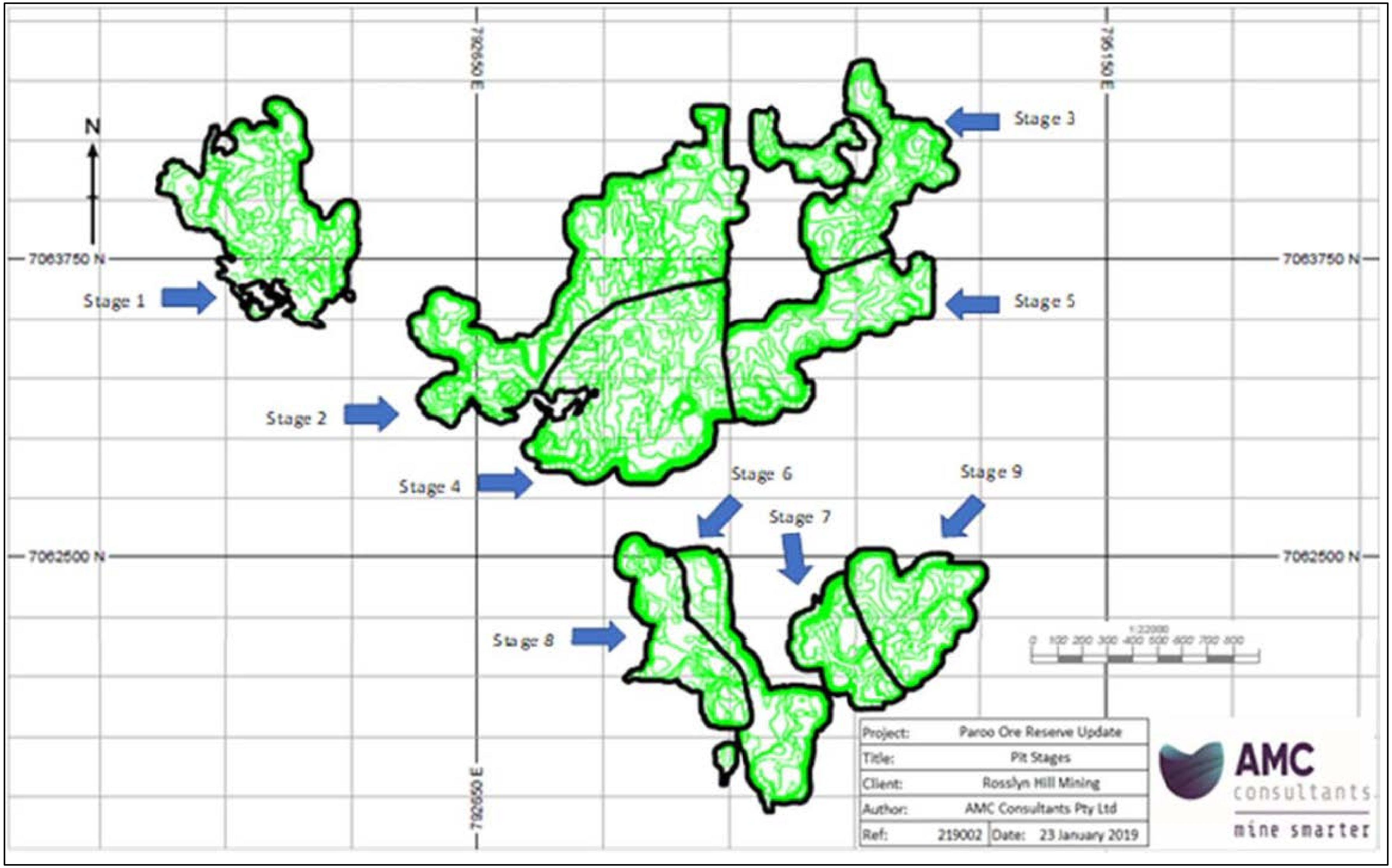

Figure 53: All pit stages

Source: AMC (2019).

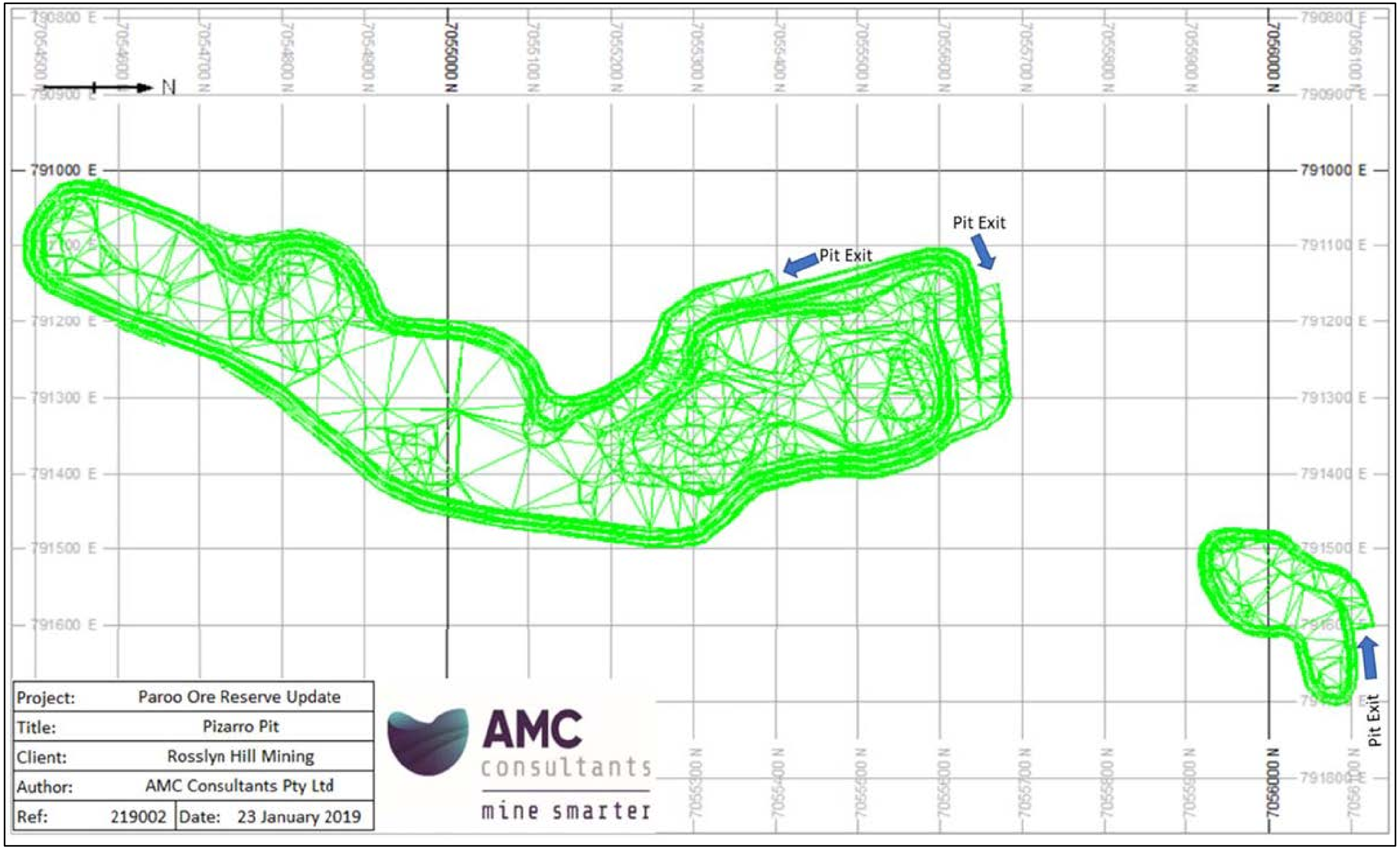

Figure 54: Pit design for Pizarro

Source: AMC (2019).

Each pit stage design was evaluated using the mining model to produce each pit inventory as displayed in Table 42.

Table 42: Inventory summary by pits

Pit design	Waste (t)	Ore (t)	Strip Ratio	Pb (%)	Contained Pb (t)
Cano	3,852,455	2,776,132	1.39	2.92	81,040
Magellan	53,781,487	17,422,555	3.09	4.12	717,863
Pinzon	26,003,680	9,247,261	2.81	3.84	355,366
Pizairn	9,575,828	3,914,999	2.45	3.08	120,727
Total Design	93,213,450	33,360,947	2.79	3.82	1,274,996
Optimisation	85,116,965	33,612,635	2.53	3.85	1,293,510
Design vs Optimisation (%)	9.51	-0.75	10.34	-0.69	-1.43

Source: AMC (2019).

15.3 Mine Production Scheduling

Strategic mine production schedules were developed using MineMax software, to produce quarterly increment schedules for the LOM. MineSched software was then used, with reference to the strategic quarterly schedule, to generate a monthly increment schedule for the first five years of operation, followed by a quarterly schedule thereafter.

This schedule was developed based on:

- Diluted Magellan Hill and Pizarro models with Measured and Indicated Mineral Resource categories only
- Annual schedule where start date is October 1, 2020
- Mill capacity of 2.185 Mtpa after an initial ramp
- Achieving production creep to support a maximum 80 ktpa of lead ingot production
- 5 m benches
- Use of existing stockpiles as ore feed for the commissioning and ramp-up of the flotation and hydrometallurgical facilities.

Figure 55 and Figure 56 show total the material moved during the LOM and the source of plant feed. These movements integrate with Section 21 (Capital and Operating Costs) and Section 22 (Economic Analysis). Operations will commence at the completion of wet commissioning, with stockpile rehandle from existing stockpiles being the basis of plant feed. Mining commences in the Cano pit in Month 6 of operations. The Cano pit will be mined out in the first 18 months of mining before Pinzon and Magellan are depleted over the remaining LOM. Mining at Pizarro commences in Year 10 of operations. Throughout the LOM, ore from existing operations and ore from new stockpiles is used to supplement the direct ore feed. Excluding stockpiles on hand at the start of operations, a total of 33.4 Mt of ore will be mined at an average grade of 3.82% Pb, at a stripping ratio of 2.79:1, for contained lead of 1.344 Mt.

Salient points from the schedule include:

- Mine life of 17 years of plant throughput; 16 years of mining
- Mill kept at capacity until near the end of the LOM
- Total material movement limited to the first year due to stockpile feed. Thereafter the production adjusts to suit the ore requirements over the life of mine.

Figure 55: Total material movement

Source: AMC (2019).

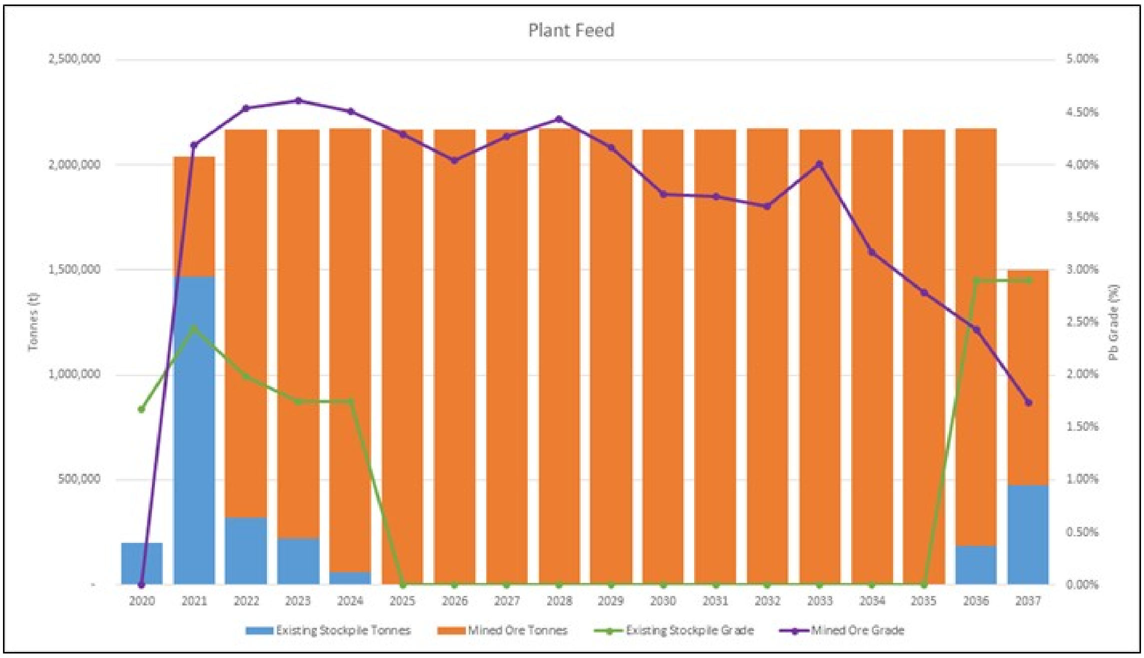

Figure 56: Annual plant feed

Source: AMC (2019).

15.4 Waste and Stockpile Design

Preliminary waste dumps were designed to ensure sufficient ex-pit dumping capacity. The following design parameters and assumptions have been applied:

- 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.

While the integrated waste landform (IWL) embankment will provide 11.4 Mm^3 of waste rock storage capacity, a conservative approach has been adopted and the design of the waste dumps is such that there is sufficient volume available to contain all waste produced. In addition to this, opportunities for in-pit dumping that will realize both cost savings from shorter hauls and reduced dump footprints and/or heights have not been factored into the waste dump designs. The current development of the pits is such that in-pit dumping opportunities will be realized shortly after recommencement of mining operations.

The proposed location of the waste dump are shown in Figure 57.

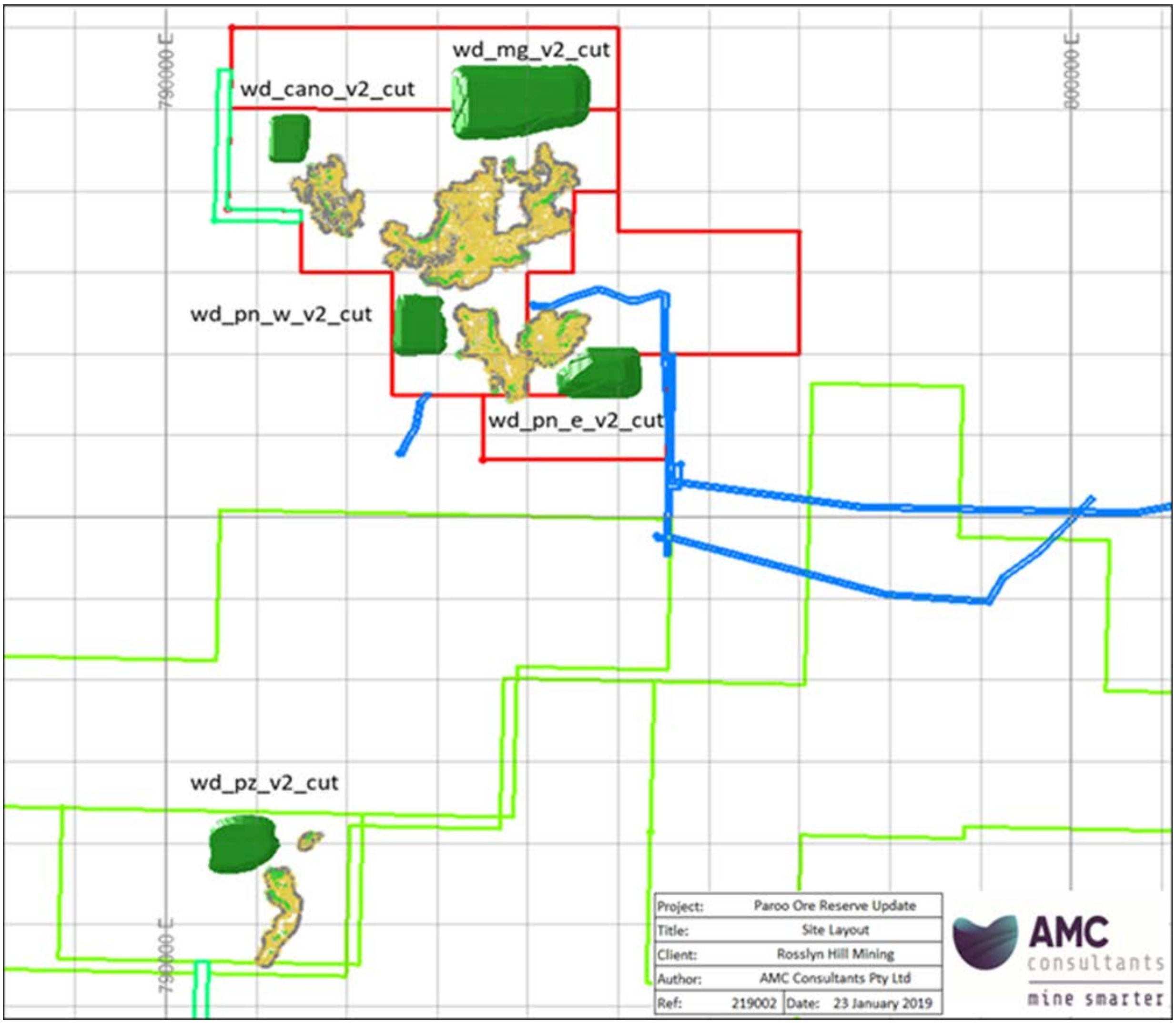

Figure 57: Waste dump layout

The volumes for each of the waste dump were evaluated as shown in Table 43.

Table 43: Capacity of waste dumps

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)
Cano	Cano Dump	1,999,972	2,599,964	2,859,960	2,892,303	252,785
Magellan	Magellan Dump	27,710,327	36,023,425	39,625,768	41,258,809	1,260,645
Pinzon	Pinzon Dump 1 & 2	13,477,373	17,520,585	19,272,644	21,210,219	922,950
Pizarro	Pizarro Dump	5,016,227	6,521,096	7,173,205	8,078,966	462,620
Total		**48,203,900**	**62,665,070**	**68,931,577**	**73,440,297**	**2,899,000**

Source: AMC (2019).

15.5 Mineral Reserve Estimate

15.5.1 2019 Mineral Reserve Estimate

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

The Ore Reserve estimate was developed under the JORC Code (2012) which is recognized by the CIM as a Foreign Code.

The Mineral Reserve statement is presented in Table 44.

Table 44: Mineral Reserve statement as at February 15, 2019

Deposit	Reserve Category	Tonnes (Mt)	Grade (% Pb)	Contained Pb Metal (kt)
Cano	Proved	1.5	3.3	51.6
	Probable	1.2	2.4	29.4
	Total	**2.8**	**3.1**	**81**
Magellan	Proved	4.3	4.2	177.6
	Probable	13.1	4.1	540.2
	Total	**17.4**	**4.1**	**717.9**
Pinzon	Proved	0.1	5.9	5
	Probable	9.2	3.8	350.4
	Total	**9.2**	**3.8**	**355.4**
Pizarro	Proved	0.0	0.0	0.0
	Probable	3.9	3.1	120.7
	Total	**3.9**	**3.1**	**120.7**
Stockpiles	Proved	2.9	2.4	69.6
	Probable	0.0	0.0	0.0
	Total	**2.9**	**2.4**	**69.6**
Total	**Proved**	**8.8**	**3.4**	**303.8**
	Probable	**27.5**	**3.8**	**1,040.8**
	Total	**36.3**	**3.7**	**1,344.6**

Source: AMC (2019).

Notes:

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 mineralization, 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 Laurie Gillett FAusIMM of AMC is a Qualified Person 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 10, 2015.
5. The pit limits for the open pit were selected through optimization using the Gemcom Whittle Four-X implementation of the Lerchs-Grossman algorithm. The optimization considered Measured and Indicated Mineral Resources only. Pit designs followed the optimization shell outline that developed the highest 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 = (-0.1017 × % Ore Grade^2 + 2.7556 × % Ore Grade + 73.5%)/100 limited to a maximum of 92.5%, Hydrometallurgical Plant Recovery of 97.87%.
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 50 cm skin around the cut-off grade 1.60% Pb envelope. Within the Ore Reserve pit design, the application of dilution resulted in inclusion of 9.92% dilution and results in an ore loss of 1.83%. Metal pricing of US$2,269/t Pb plus a US$94/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 project. 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 project.
9. Table entries are rounded to reflect the precision of the estimate and differences may occur due to this rounding.

15.5.2 Inventory Changes from 2018 to 2019

The 2019 Mineral Reserve is materially different to the 2018 Mineral Reserve estimates.

The Mineral Reserve estimate has previously been estimated as at February 28, 2018 from a Technical Report undertaken by SRK dated April 12, 2018. RHM has updated the estimate following enhancements to the flotation recovery, refining of the cost and revenue inputs and the inclusion of Mineral Resources from the economic Pizarro satellite deposit.

An increase of approximately 5.1 Mt in Mineral Reserves is noted between the February 28, 2018 estimate and the current estimate.

An increase of approximately 145 kt Pb is noted between the February 28, 2018 estimate and the current estimate.

15.6 Mineral Reserve Sensitivity

Multiple pit optimization runs were undertaken to establish the project's sensitivity to pricing, mining and processing costs. The results of these ancillary runs establish the key drivers to the development of the mining processes suited to the extraction of the deposits' potentially economic mineralization.

Changes in the undiscounted cashflow and ore tonnage variance for each parameter have been plotted, where a steeper slope on any curve represents greater sensitivity to the parameter represented by that curve. The curve is defined over a ±20% variability from the base case for each parameter. The sensitivity results are plotted in the graphs illustrated in Figure 58 and Figure 59.

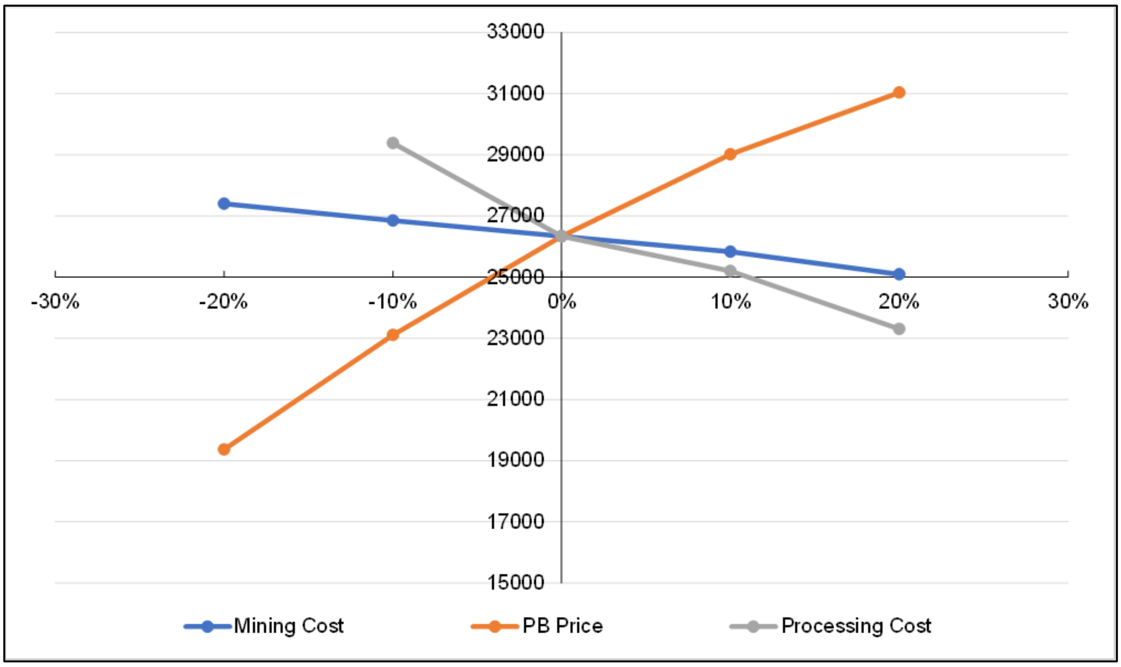

Figure 58: Sensitivity analysis graph – ore tonnes (kt)

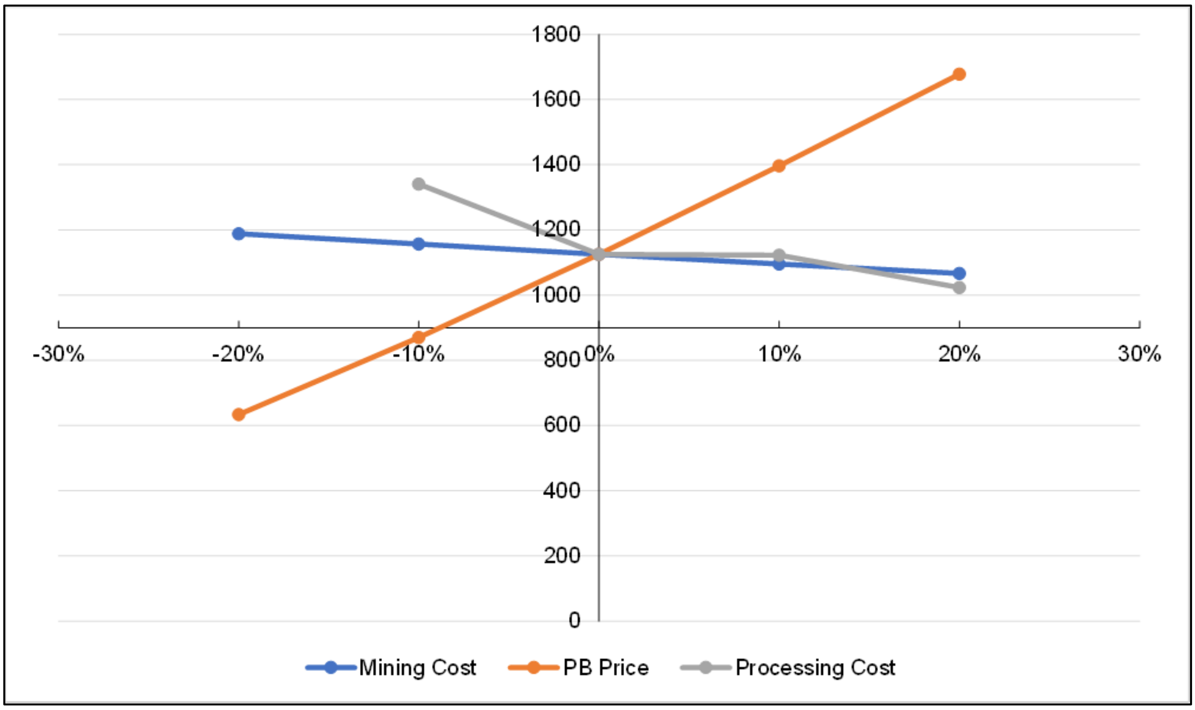

Figure 59: Sensitivity analysis graph – undiscounted cashflow (US$M)

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. Mining costs have 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 Mine 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 orebody.

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 60: Mining operations in the Magellan open pit

Source: RHM.

16.2 Mining Fleet and Requirements

16.2.1 General Requirements

MACA Mining Pty Ltd (MACA) hold the mining contract (currently in suspension) to provide ROM ore feed, drill and blast and load and haul of ore and waste.

An assessment has been made comparing the benefits of applying an owner operator model verses maintaining the mining contract.

The owner operator model assessed would utilize hired earthmoving equipment on a fully maintained basis for load and haul, and crusher feed activities, RHM (or labour hire) operational personnel and contracted drill & blast services.

Budget rates have been obtained from reputable WA firms for equipment supply and maintenance, and drill and blast activities.

For the purposes of the DFS Update, the owner operator model costing has been applied.

The RHM Mining department mainly comprised technical personnel (engineers, geologists and surveyors) to control and administer the mining operations and mining contract.

The following aspects of the mining operation were controlled by RHM:

- All geological functions
- All survey functions
- All mine design functions
- All mine planning functions (long-, medium- and short-term)
- Contract management
- Quarry Manager obligations as per the *Mines Safety and Inspection Act and Regulations*.

The following aspects of the mining operations were controlled by MACA:

- Equipment supply and maintenance for drill & blast, load & haul, and crusher feed operations
- Operators and supervision
- Support functions for the MACA operations, i.e. administration, OSH&E (Occupational Health, Safety & Environment) and training
- Project management.

16.2.2 Drilling

Drilling has historically been performed by a single GD5000 drill operated on double shift, nominally 102 mm holes, single pass 5.0 m benches with 0.5 m sub drill. Pattern size is from 3m × 3.5m burden and spacing in the hardcap rock to 4m × 4.5m 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.

Future drilling requirements are planned to be met by a similar class of drill.

16.2.3 Blasting

There are no planned changes to the blasting practices from the prior operations. Blasting was primarily performed using ANFO (ammonium nitrate fuel oil) due to the dry conditions with powder factors typically ranging from 0.2 kg/bcm to 0.5 kg/bcm. Single-hole firing was used to minimize movement and dilution of the ore. It is anticipated that a reduction in hole size to 89 mm will be required to keep the powder factor down as generally the effort required to blast reduces with depth. There will be no change to the blasting practices for the future operations.

16.2.4 Loading

Loading has been previously performed by a single 120 t class backhoe configuration excavator operating on double shift. Productivity in excess of 8 Mtpa of ore and waste can be achieved with this size of machine with the digging conditions presented. The 5 m benches are mined in 2 × 2.5 m flitches with the differing material types being defined by mark-out tape and paint as designated by the site geologists.

The updated 2019 mine planning requires the use of a larger 200 t class excavator to meet the volume movement requirements for the first 10 years of operations. The mine layout including the ore body parameters are well suited to this size machine given the reduction of cutoff grade.

16.2.5 Hauling

The operations are proposed to continue to use 85 t class dump trucks to haul the ore, waste and mineralized waste materials to their respective destinations – ROM Pad, waste dump and stockpiles. The hauls for ore, waste and mineralized waste differ depending on the pit and stage location and can vary from 2 trucks to 4 trucks hauls. 85 t class dump trucks will continue to be used with the larger 200 t class excavator.

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 integrated tool carrier 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 groundwater investigation will need to be performed to identify the effects on the hydrological regime of the groundwater resource, effects on the potential groundwater dependent ecosystems (GDEs) 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 Environmental Protection Authority (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 Mine concentrator.

Metallurgical performance for the last operational campaign is shown in Table 45 to Table 47 (2013–2015).

Table 45: Paroo Station Mine 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
Annualized 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
Conc. produced	dmt				1,469	6,079	6,575	8,173	8,766	9,507	10,165	9,864	7,455	68,053
Conc. grade	%				62.80	63.90	63.90	63.60	65.10	65.00	65	65	65	65
Conc. Pb content	dmt				923	3,881	4,201	5,194	5,711	6,183	6,636	6,448	4,481	44,018
Conc. 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 46: Paroo Station Mine 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	1,1437,958
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
Annualized 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
Conc. 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
Conc. 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
Conc. 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
Conc. 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 47: Paroo Station Mine metallurgical performance – 2015

Actual	Unit	Jan
Ore milled	dmt	166,305
Head grade	%	7.40
Annualized rate	Mtpa	1.96
Recovery	%	77.80
Conc. produced	dmt	13,621
Conc. grade	%	70.50
Conc. Pb content	dmt	9,607
Conc. Moisture	%	8.50
Plant availability	%	95.77
Plant usage	%	105.85

Source: RHM (2015).

17.2 Definitive Feasibility Study Update Testwork

17.2.1 Basis of Recovery Calculations

The estimation of metallurgical recoveries from variability concentrates is based on the testwork on column flotation concentrates produced for the bulk metallurgical testwork. For this calculation, the following results have been used:

- Lead recovery from the MSA bulk leach test
- Flotation recovery from DeS leach tests residues to concentrate
- Leach recovery from DeS flotation concentrates.

The semi-quantitative XRD (X-ray diffraction) mineralogy of the variability samples at various stages of processing were used to complement the metallurgical test results.

17.2.2 Mineralogy

The calculated distribution of lead across different minerals in the feed concentrate, the MSA leach residues and the DeS leach residues is reflected in Table 48 and Table 49, as calculated from XRD assays. Note that galena has been formed during the sulphidization process prior to flotation. The Paroo Station ore samples contain negligible primary galena.

These results have been used to track dissolution of minerals during the process.

Note that the pyromorphite mineral distribution has been re-calculated based on phosphorus assays for each sample as the original XRD assays were found to overestimate the pyromorphite content of the sample.

Table 48: Distribution of lead across lead Minerals in MSA leach feed

Mineral	Year 1	Year 2	Year 3/4	Year 5	Year 6	Year 7/8/9	Year 10	Average
Galena	2.33	2.56	0.01	3.71	3.68	5.22	4.41	3.13
Cerussite	92.3	74.4	95.1	90.9	64.8	88.7	73.8	82.9
Hydrocerussite	0.0	0.0	0.59	0.0	0.0	0.0	0.0	0.084
$NaPb_2(CO_3)_2OH$	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Anglesite	3.68	21.2	3.01	3.91	30.0	4.11	18.6	12.1
Leadhillite	1.05	1.16	0.00	0.56	1.11	0.59	1.33	0.83
Pyromorphite	0.60	0.66	1.31	0.96	0.47	1.35	1.89	1.03
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

In the feed, most of the lead is contained in cerussite; however, minor lead minerals, predominantly anglesite, still contribute a variable proportion of the lead in each concentrate.

Table 49 shows the lead distribution amongst minerals in the MSA Leach residue, with all cerussite leached.

While anglesite is dominant in the samples with a low lead extraction, galena and pyromorphite occur in significant quantities in the other residues.

Table 49: Distribution of lead across lead minerals in MSA leach residue

Mineral	Year 1	Year 2	Year 3/4	Year 5	Year 6	Year 7/8/9	Year 10	Average
Galena	54.1	27.3	43.4	55.9	55.9	21.4	15.9	39.1
Cerussite	0.00	0.0	0.0	0.0	0.0	0.0	0.0	0.00
Hydrocerussite	0.00	0.0	0.0	0.0	0.0	0.0	0.0	0.00
$NaPb_2(CO_3)_2OH$	0.00	0.0	0.0	0.0	0.0	0.0	0.0	0.00
Anglesite	19.7	57.4	30.8	16.0	16.0	59.2	67.0	38.0
Pyromorphite	26.2	15.3	25.7	28.1	28.1	19.3	17.1	22.8
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

Based on feed and residue mineralogy detailed in Table 49, the estimated leach extractions achieved for each mineral are listed in Table 50.

Table 50: Calculated mineral extractions in MSA leach

Mineral	Year 1	Year 2	Year 3/4	Year 5	Year 6	Year 7/8/9	Year 10	Average
Galena	25.6	59.2	35.6	7.07	11.0	54.0	68.9	40.5
Cerussite	100	100	100	100	99	100	100	100
Hydrocerussite	100	100	100	100	100	100	100	100
$NaPb_2(CO_3)_2OH$	100	100	100	100	100	100	100	100
Anglesite	82.8	89.6	56.8	74.7	21.2	-61.0	68.9	26.8
Pyromorphite	-45.1	11.2	17.23	-88.6	-40.2	-62.8	22.0	-29.5
Lead Extraction	96	92	96	95	70	89	93	90

On average, approximately 90% of the contained lead is recovered in the primary MSA leach stage.

It is questionable whether anglesite would leach in the MSA leach, as circulating Spent and MSA solution would be expected to be saturated with sulfate. Some galena leaching was expected due to the addition of hydrogen peroxide to the leach. Pyromorphite precipitation can be expected in the presence of phosphate and chloride under low free acid conditions.

Conversion of the anglesite to cerussite in the DeS results in distribution of lead amongst minerals in the DeS leach residues is shown in Table 51.

Table 51: Distribution of lead across lead minerals in DeS leach residue

Mineral	Year	Year	Year	Year	Year	Year	Year	Average
	1	2	3/4	5	6	7/8/9	10	
Galena	42.8	8.19	28.8	59.8	7.21	18.2	27.1	27.4
Cerussite	21.3	31.8	30.1	0.0	54.9	26.1	29.1	27.6
Hydrocerussite	0.00	7.58	0.0	12.78	0.00	0.00	10.0	4.34
$NaPb_2(CO_3)_2OH$	3.96	2.28	0.0	1.92	31.5	6.06	13.5	8.47
Anglesite	3.75	45.2	15.1	0.00	4.27	25.8	4.3	14.1
Pyromorphite	28.2	4.9	26.0	25.5	2.08	23.9	16.0	18.1
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

Recovery of lead from these residues is initially achieved by floating the residue to reject gangue minerals, after which the flotation concentrate is recombined with the MSA leach.

17.2.3 Estimation of Metallurgical Recovery from Variability Testwork

The recovery of lead has been calculated based on the results of the various tests on variability samples (Table 52).

Flotation recoveries have been reduced by 1.5% to allow for losses during the cleaning stage. The initial estimate of recoveries is based on the actual leach test results achieved. However, three of the seven tests returned poor conversions of anglesite to cerussite.

If it is assumed that the test conditions can be modified and optimized for better conversion, the unconverted anglesite can be added to the recovery. It is strongly recommended that this assumption be followed up with further testwork.

Two overall recoveries have been calculated. The first one using test results only, resulted in a weighted average lead recovery of 96.8% and when it is assumed that anglesite can be fully converted to cerussite, the recovery rises to 97.9%.

These recoveries represent a first-pass recovery across all the flowsheet unit operations. It is probable that unleached lead minerals may be recycled. It should be considered that the MSA re-leach residue returning to the process will be subjected to further DeS leaching and flotation, so that a second pass through the process is likely.

This could increase recoveries to the extent that the only losses are effectively the flotation tailings, and the 97.9% recovery would therefore appear to be a reasonable target. However, conducting sensitivity analyses on recovery in the financial model is recommended.

The DeS float MSA Leach Recovery has been adjusted for Years 2, 3/4, 7/8/9 based on recycling and reflotation of the residue.

Table 52: Estimated recoveries from variability samples

Prospect Aspect	Year 1	Year 2	Year 3/4	Year 5	Year 6	Year 7/8/9	Year 10	Weighted Average
MSA Leach Recovery	96.0	92.0	96.0	95.0	70.0	89.0	92.6	90.5
Recovery from Solution	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9
Float Recovery	89.2	91.5	95.0	89.3	96.6	81.1	92.1	89.2
DeS Float MSA Leach Recovery (actual)	88.1	39.1	73.2	88.6	92.4	59.5	85.1	71.8
DeS Float MSA Leach Recovery (adjusted)	88.1	83.3	87.3	88.6	92.4	84.2	85.1	86.5
Overall Recovery (testwork)	**99.0**	**94.8**	**98.7**	**98.9**	**96.7**	**94.2**	**98.3**	**96.8**
Overall Recovery (estimate)	**99.0**	**98.0**	**99.2**	**98.9**	**96.7**	**96.4**	**98.3**	**97.9**

A Concentrator METSIM model was been developed by InCoR to reflect the proposed modified flowsheet and has been used and validated by SNC-Lavalin to evaluate process parameters for the proposed flowsheet based on the flotation testwork done by ALS. The flowsheet modifications included converting the mill from SAB to SABC, a modified flotation circuit using existing equipment with relocated concentrate streams off first rougher and first cleaner cells and addition of a flotation column to produce a final concentrate in the range 71%–74% Pb 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 as part of the Demonstration Plant testwork. The Demonstration Plant flotation circuit was configured to mimic the proposed concentrator flowsheet.

1. **Hydrometallurgical Facility Model**

   A METSIM model was developed for a ‘base case’ mineralogy, based on a weighted average of the annual variability samples, comprising predominantly 80.0% Pb carbonate (cerussite) and 9.17% Pb sulfate (anglesite) at overall concentrate grade of 71.8% Pb. The other minor minerals assumed to be in the concentrate (based on XRD analysis) are pyromorphite (1.38%), galena (2.22%), leadhillite (0.54%), kaolinite (1.06%), hematite (0.48%) and quartz (4.03%). The flowsheet is described in detail below.

   On completion of the variability testwork, additional METSIM models were run for the assumed minimum (3.05%) and maximum (21.4%) anglesite levels, which have been run to assess the impact of the changing concentrate mineralogy on Mass Balance flows and operating costs.

2. **Integrated Model**

   An integrated METSIM model developed after the DFS combines the Flotation Concentrator model and the Hydrometallurgical Facility model. This has been used to develop the overall plant design.

   Initial data for the model was derived from the testwork database available at that time. The data in the model has subsequently been updated with data from the Demonstration Plant operations.

   The flotation concentrate elemental composition was derived from an average of the concentrate produced from the Demonstration Plant flotation plant operations. Reagent consumptions for the Flotation Concentrator were set to reflect quantities derived from the Demonstration Plant flotation operations.

   Leaching extents for the lead minerals were derived from batch testwork and Demonstration Plant data. Minor element leaching extents were determined from the Demonstration Plant leaching chemistry.

After development of the base case model, which is set to the expected average lead mineralogy, two other models were developed for the minimum and maximum anglesite content. Minor lead minerals were assumed to be constant across all three models. The throughput parameters described below reflect a specific nominal case. However, the Process Plant has been designed to produce a nominal 70,000 tpa lead ingot.

Table 53 summarizes the key parameters in the three METSIM models.

Table 53: METSIM models – key parameters

Parameter	Units	Low Anglesite	Base Case Average Anglesite	High Anglesite
Flotation Concentrator				
ROM Throughput	tph	255	255	255
	tpa	2,058,833	2,058,833	2,058,833
Lead Head Grade	% Pb	3.997	3.990	4.000
Cerussite Recycle	tph	1.70	2.40	4.02
	tpa	13,698	19,353	32,382
Lead Head Grade	% Pb	32.8	46.7	60.7
Flotation Concentrate	tph	12.4	13.1	14.6
	tpa	100,242	105,354	118,046
Lead Head Grade	% Pb	71.8	71.8	71.8
Cerussite	%	85.5	80.1	69.8
Anglesite	%	3.05	9.2	21.4
Galena	%	2.28	2.28	2.00
Pyromorphite	%	1.425	1.380	1.226
Leadhillite	%	0.532	0.541	0.476
Flotation Recovery	%	83.0	83.0	83.1
Concentrate Leaching				
Concentrate Feed	tpa	100,242	105,354	118,046
Lead Grade	% Pb	71.8	71.8	71.8
MSA Leach				
MSA Residue	tph t/h	1.75	2.54	4.36
Lead Grade	% Pb	31.1	43.7	55.8
Lead Leached	%	94.6	88.4	76.2
Iron In	g/L	1.65	1.65	1.65
Iron Out	g/L	1.82	1.82	1.82
MSA Leach Residue CCD				
Number Stages		7	7	7
Target Wash Efficiency		99.95	99.93	99.82
Wash Ratio		1.89	1.89	1.65
Underflow Density	%w/w	45	45	45
Acid Leach				
Pyromorphite Conversion	%	89–90	89–90	89–90
DeS Leach				

Parameter	Units	Low Anglesite	Base Case Average Anglesite	High Anglesite
Anglesite Conversion	%	98	98	98
Evaporator				
Evaporator Requirement	m^3/h	7.2	8.19	10.9
Lead Tankhouse				
Lead Electroplated	tpa	72,633	71,580	69,823
Lead Melting				
Ingot Production	tpa	67,403	66,426	64,796
Starter Sheet Production	tpa	5,230	5,154	5,027
Metallurgical Performance				
Leach Extraction	%	99.8	99.7	99.5
Overall Extraction	%	82.8	82.7	82.7

17.2.4 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 later 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 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 sulfuric 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 second cleaning and typically contains 55%–65% of the recoverable lead at a grade of approximately 60% Pb. 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% Pb can be produced in this circuit. A key feature of the Rosslyn Hill 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 addition of small amount of sulfuric acid to reduce the pH to 5.5. Empirically and inexplicably, addition of a small amount 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 SIPX conditioning steps, as per the current plant arrangement.

Flotation Feed Density

In general, a flotation feed density of 35% has historically been employed at the flotation plant. The current testwork highlighted improvements in flotation recovery if the flotation feed density was reduced to 30% solids which 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 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 Oxidation/Reduction Potential (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 led to an equally significant increase in SIPX consumption.

Flotation Concentrate Thickening

A new concentrate thickener will be installed based on the design parameters of 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 accommodate variability in 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; however, introduction of column flotation into the circuit has improved the filtration such that the revised duty including concentrate washing is well within the capacity of the existing Metso VPA Filter press. Filtration rates now average approximately 5 t/m^2/h, compared with the original 200 kg/m^2/h achieved when the Flotation Concentrator was operating.

Concentrate Particle Size Distribution, PSD

The size distribution of the lead concentrate produced in the pilot plant has a P80 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 plant. It is proposed that existing cleaner flotation equipment in the current plant be used for this duty.

Sulfur Flotation

Elemental sulfur is formed by the reaction of galena with ferric methanesulfonate and while the formation rate can be measured as a few tens of kg per hour, it will be necessary to separately recover Sulfur on an intermittent basis to avoid a build-up in the circulating load. Sulfur 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 Sulfur which can be passed to the flotation tailings.

Flotation of Lead Minerals from DeS Leach Residue

The tailings from the Elemental Sulfur float will be conditioned using sulfuric 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.

17.2.5 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 Heat Recovery Boiler (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 re-pulp washed flotation concentrate in MSA thickener overflow to 65% solids which contains minimal MSA to avoid gas evolution in either the re-pulp 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 re-leach discharge slurry at 65°C is added to the head of the leach train so that the residual acid and ferric ion can be used 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 ore being processed. The residue mass flow variability is largely attributable to a variable proportion of anglesite in the MSA leach residue, which does not 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 Proof of Concept (POC) and variability testwork. Based on Pilot Plant performance, underflow densities are expected to range between 35% and 50%. Thickener underflow 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 6-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 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 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 aluminum 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 aluminum from the advance electrolyte, with the circuit operating in the pH range of 4.0–4.5. In operation, it is expected that the impurity removal circuit could be bypassed, eliminating the lime requirement, depending on the level of impurities in solution. However, the impurity removal circuit is currently used to build up iron in the MSA leach circuit to effect oxidation of galena.

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/m^2/h at 30% solids (w/w) was selected for design purposes. The design of this thickener is controlled by the rise rate.

Electrolyte Filtration

The advance electrolyte is filtered to remove 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 maximize 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 methanesulfonate salts and generate calcium methanesulfonate which will be regenerated to MSA 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 methanesulfonate and generate MSA. 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

To maintain a positive water balance, it is necessary to evaporate water from the process liquor.

18 Project Infrastructure

18.1 Onsite infrastructure

Key infrastructure for the project includes:

- Processing facilities
- Hydrometallurgical facility (to be constructed)
- Power station and infrastructure
- TSF and 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.

Figure 61 shows the key site infrastructure overlain on a regional aerial photograph of the operation.

18.1.1 Processing Facilities

The lead ore processing facilities have been described in Section 17 and consist of infrastructure to allow lead ore to be processed through a series of crushing, milling, flotation concentration, filtration moisture reduction and drying operations. The concentrate bagging operation from the previous operating period will be maintained as a standby as needed.

18.1.2 Hydrometallurgical Facility

The Hydrometallurgical Facility has been described in Section 17 and will consist of infrastructure to allow lead concentrate to be processed into lead ingot by acid leaching, solid/ liquid separation, electrowinning and melting operations.

18.1.3 Mine Offices

The 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
- Ablution facilities.

The transportable buildings are connected via concrete footpaths and wooden walkway.

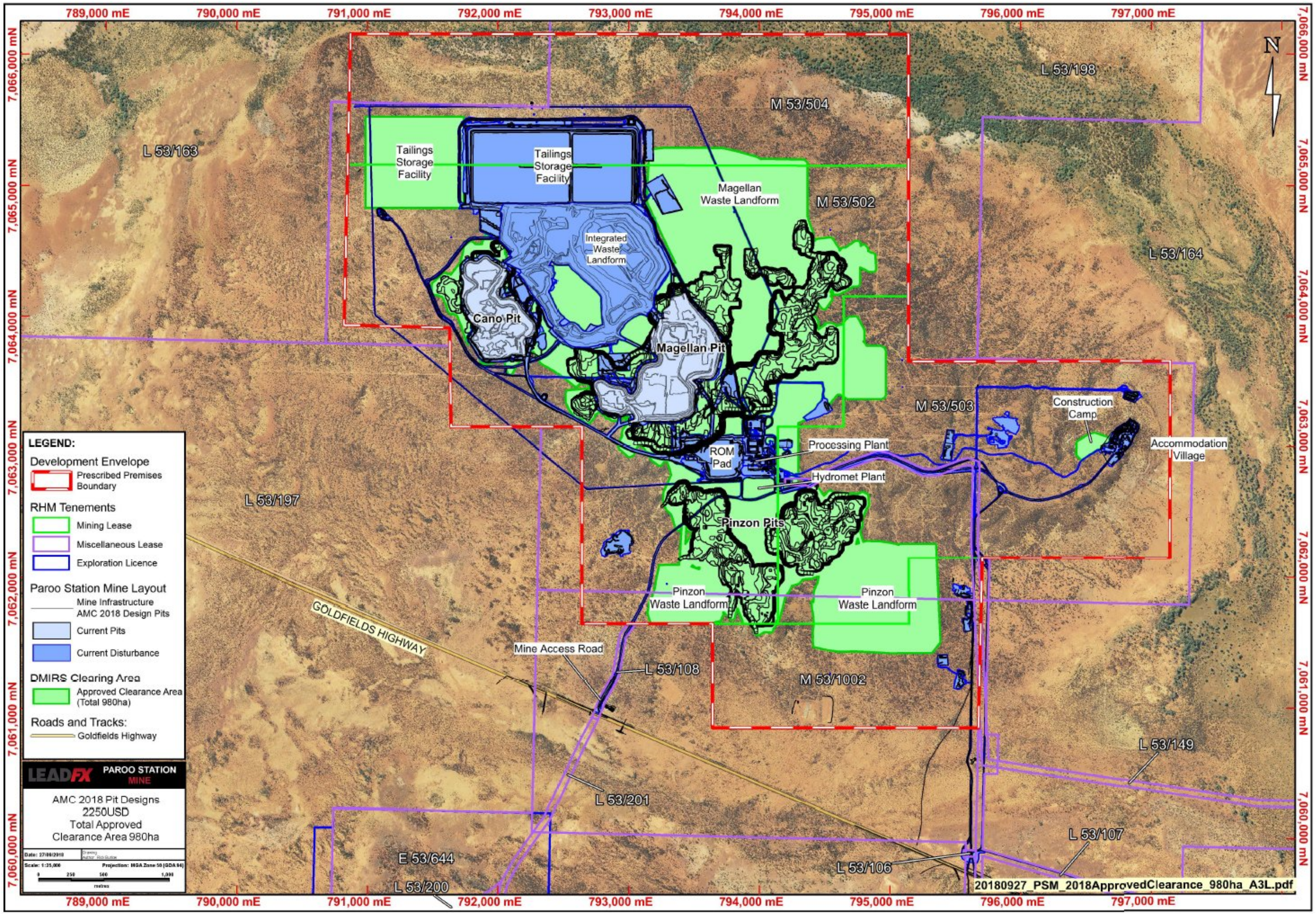

Figure 61: Site layout showing key infrastructure

Source: RHM (2018).

18.2 Water Supply and Management

18.2.1 Borefield

Processing water requirements for the operation are currently met from a production borefield located approximately 4 km south-east of the Mine (Figure 62).

The borefield comprises four production bores (PB01 to PB04). Production bores PB01, PB02 and PB03 are installed to depths between 12 m 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 Update to install dedicated overhead powerlines to each production bore for future power supply.

The water is of variable quality (total dissolved solids ranging from 1,000 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 paleochannel 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 license is for 2.5 GL per annum.

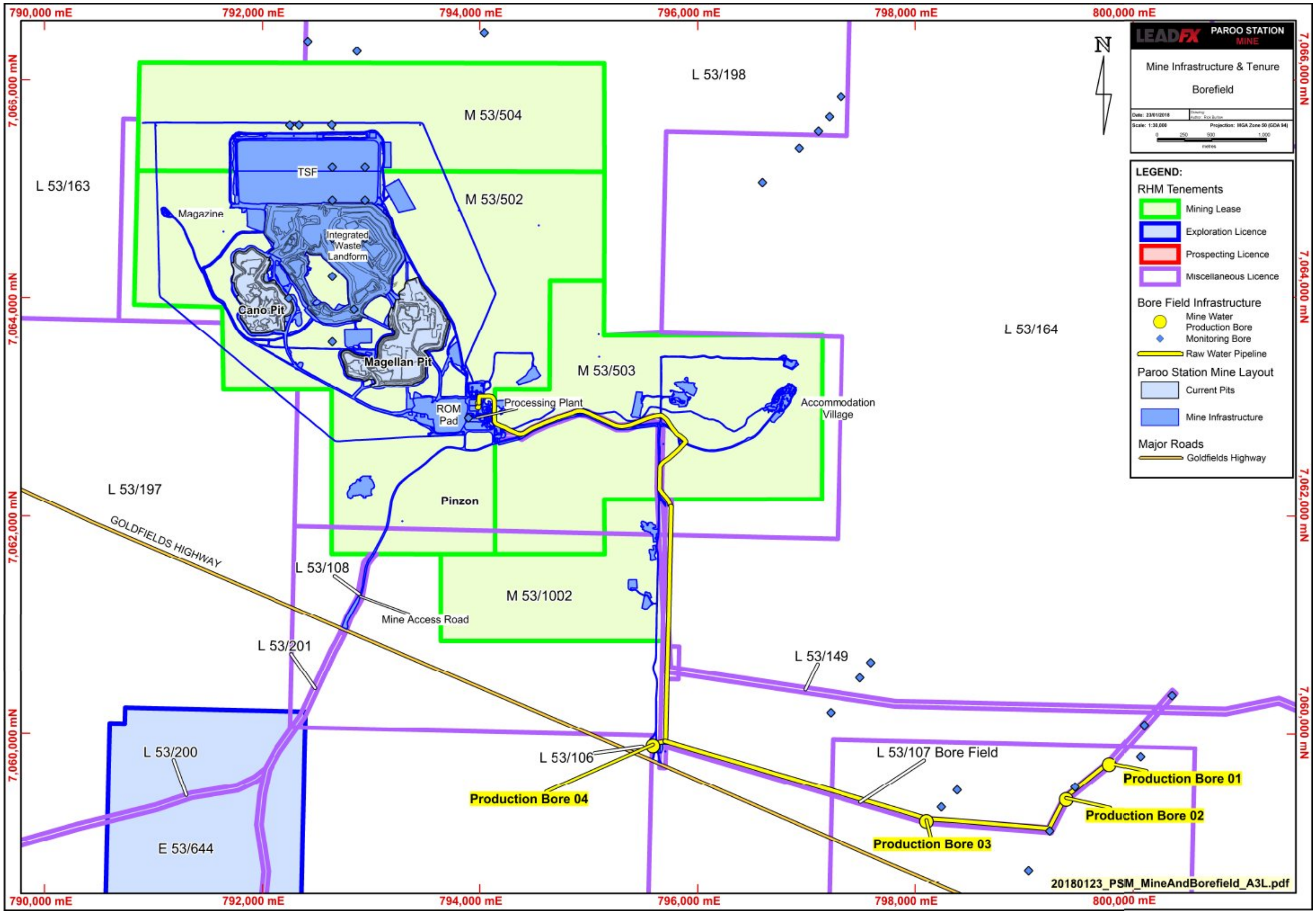

Figure 62: Borefield location

Source: RHM (2018).

18.2.2 Reverse osmosis plant

A reverse osmosis (RO) desalination plant is located within a sea container located next to the raw water dam.

RO treated water is stored in three 16 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 new Hydrometallurgical Facility and all potable water requirements for the site. To meet project demand, allowance for an additional RO plant has been included in the capital cost estimate.

18.2.3 Waste water treatment facilities

Waste water treatment facilities include a sewage farm, comprising a 2-celled cascading water treatment installation approximately 150 m south-east 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 silcretized/ quartz-clay breccia with clean mine waste used 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 north-east corner of Mining Lease M53/502, within a fenced and secure compound. When in operation, the facilities include two ventilated transportable buildings for storage of ANFO. 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 washdown 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 2019 by Golder Associates. The establishment of an integrated waste landform (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 there are 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 current 17-year LOM will require additional tailings storage volume. The expected additional tailings storage volume is 19 Mt for the mine extension, taking the total stored volume to 35 Mt. The storage volume of 35 Mt has been approved under Part IV of the *EP Act* and now forms part of the new Ministerial Statement for the project. The approval was based on the Golder Associates (2018) tailings storage options study for the planned total storage volume. The most favored option is an IWL amalgamation consisting of the existing TSF, the currently approved IWL and the Cano pit once mining in that pit is complete. The amalgamated IWL will include progressive waste rock walls lifts to the outer margins of the three structures such that the amalgamated IWL will end up as a single tailings storage structure.

Detailed design and regulatory approvals will be required for the amalgamated IWL which can occur once the Mine is operational. Current tailings deposition approvals are in place and will cover no less than the first 4.5 years of operations.

RHM developed an estimate to undertake the approved IWL work based on a technical study undertaken by Golder Associates with an estimated implementation cost of US$1,5M. The capital expenditure forms part of sustaining capital.

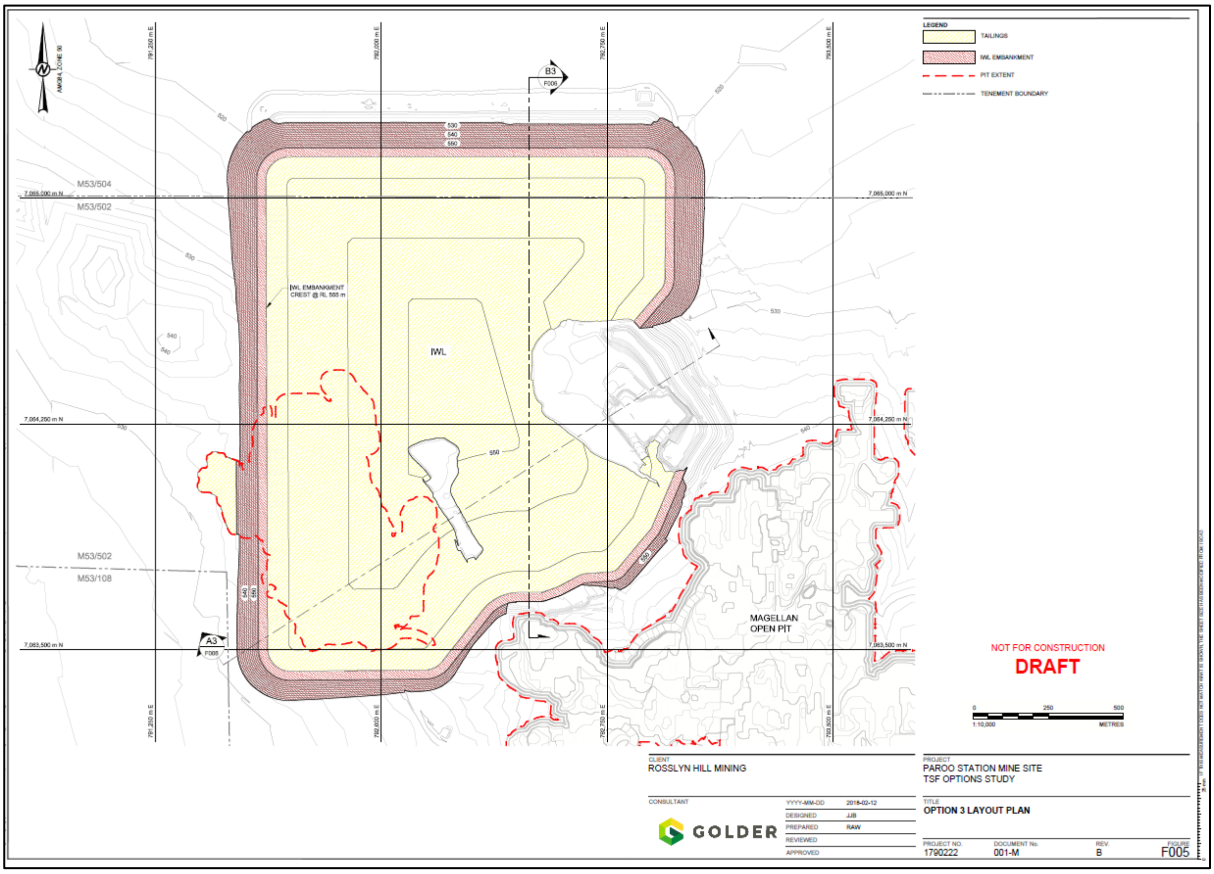

Figure 63: Golder Associates TSF options study – Option 3

18.5.2 Stores, maintenance workshop and laboratory

The stores area is located close 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.

While the laboratory facilities are appropriate for sample preparation and analysis for the mining and Flotation Concentrator requirements, installation of additional laboratory and associated equipment has been planned to support the new Hydrometallurgical Facility requirements. Costs have been included in the capital cost estimate accordingly.

18.5.3 Reagent and fuel storage

Processing reagents are transported to the project area and stored in four storage tanks contained in 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 wrapped 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 accommodation generating set.

18.6 Additional Support Facilities

18.6.1 Accommodation village

The majority of the project 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. It is located approximately 3.5 km east of the processing plant and covers an area of 3.62 ha (Figure 64).

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 prevent cattle entering the area.

Figure 64: 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 were leased.

As part of the Hydrometallurgical Facility DFS Update, a complete operation power study was completed. A new power station is proposed comprising of 9 gas-fueled generator sets. The generator sets are likely to be nominal 2 MW units and will power 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 site.

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's subsidiary, Redback Pipelines Pty Ltd, with the Paroo Station Lead Mine 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 machines being retained as reserve supply.

The sizing of the pipeline with a licensed 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 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 Offsite Infrastructure and Logistics 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 and Contracts

19.1 Overview

RHM 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 summarize 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 stable elements.

Approximately 86% of global lead metal consumption is due 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 vital as a back-up 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 12 V accessories unrelated to the drive line power generation of the vehicles.

19.3 Historic Commodity Prices

19.3.1 1975–2000

Lead metal demand stagnated over a 20-year period (1975–1995) as a consequence of the prevalent relatively large market for lead in gasoline and paints being almost completely removed.

Demand during this period 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 2000, the price of lead metal remained relatively stable. After 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–Present

Since 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 US$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 US$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.

The declines of late 2014 were maintained in 2015, with the price of lead at approximately US$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 US$2,600/t in early 2018.

19.4 Life of Mine Planning Assumptions

Prior to production, RHM plans to enter into an offtake contract with a major global trading company for the sale and purchase of 100% of its annual lead ingot production. Under such offtake arrangements, RHM will sell lead ingot on an FOB basis Fremantle. RHM will therefore be responsible for delivering the lead ingots to container vessels in Fremantle, but not responsible for subsequent ocean freight and delivery to end-users.

The sales price received by RHM will comprise (i) the prevailing LME cash price for lead, plus (ii) a premium.

Premiums are typically benchmarked off CIF Index Premiums for specific destinations quoted by Fastmarkets MB, and will be agreed by RHM and offtakers based on the lead ingot specification and subject to adjustment for offtaker costs for managing risk, marketing, freight, insurance and other costs.

The notional CIF sales price is adjusted to an actual delivered FOB basis by a freight netback for the offtaker's cost of containerized ocean freight from Fremantle to index premium destinations. The financial analysis is based on forecasts from the Wood Mackenzie Lead Market Assessment for:

- Long-term LME lead prices
- Long-term lead premiums for index premium destinations
- Freight costs to index premium destinations.

Lead index premiums in the financial analysis are based on 99.99% Pb ingot, CIF index premiums and freight netbacks are based on South East Asian destinations, and index lead premiums are adjusted for the aforementioned offtaker costs.

Table 54: Ingot sales price (US$/t Pb)

Calendar year	LME price Forecast	SE Asia net Premium CIF basis	Sale Price CIF basis	Freight Netback for FOB sale	Sale Price FOB basis
2021	2,039	145	2,184	-51	2,133
2022	2,022	145	2,167	-51	2,116
2023	1,960	145	2,105	-51	2,054
2024	2,060	145	2,205	-51	2,154
2025	2,350	145	2,495	-51	2,444
2026	2,350	145	2,495	-51	2,444
2027	2,350	145	2,495	-51	2,444
2028	2,350	145	2,495	-51	2,444
2029	2,350	145	2,495	-51	2,444
2030	2,350	145	2,495	-51	2,444
2031	2,350	145	2,495	-51	2,444
2032	2,350	145	2,495	-51	2,444
2033	2,350	145	2,495	-51	2,444
2034	2,350	145	2,495	-51	2,444
2035	2,350	145	2,495	-51	2,444
2036	2,350	145	2,495	-51	2,444
2037	2,350	145	2,495	-51	2,444

19.5 Contracts and Status

At the time of writing RHM has a liability of A$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 Mining Pty Ltd (in suspension)
- Camp management – Australian Camp Services Pty Ltd (in suspension)
- Gas Pipeline Maintenance – APA Group Pty Ltd.

20 Environmental Studies, Permitting and 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:

- Environmental Protection Authority (EPA)
- Department of Water and Environmental Regulation (DWER)
- The Department of Mines Industry Regulation and Safety (DMIRS)
- Department of Health (DoH)
- Department of Transport (DoT).

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

- Ministerial Statement 1083
- DWER – Prescribed Premises License – L8493/2010/2
- DWER – License to Extract Water – GWL96342(4)
- Australian Communications & Media Authority – Licenses 1970164 and 1970178/1
- DMIRS – Dangerous Goods Site License –DGS020079
- DMIRS – Mining Tenement conditions
- DMIRS – Pipeline License – PL73
- Radiological Council – Licenses 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 15 July 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 DMP 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 5 January 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 23 February 2011, allowing the operation to recommence as soon as practical after that date.

RHM voluntarily placed the project 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 Ministerial Statement 905 in July 2012. Ministerial Statement 905 supersedes all previous conditions and procedures. The project remained in care-and-maintenance until April 2013, and has operated under Ministerial 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 project 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 26 July 2024 and changed Condition 18 reducing the financial assurance to 2 million 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 Integrated Waste Landform (IWL)), under s 45C of the EP Act, to allow for the anticipated 4.5 year remaining LOM at the time. A license amendment to construct and operate the IWL was achieved on 14 February 2017 from the Department of Environmental Regulation (now the DWER).

Ministerial Statement 1083 was signed by the Minister for the Environment on 25th September 2018, following release of EPA report and recommendations 1620. The EPA report considered the Rosslyn Hill Mining 'Hydromet Facility & Mine Extension Proposal' referral document of 20th April 2018. The Ministerial Statement provides environmental Conditions for the proposal (Project), which includes an increase the disturbance footprint by 400 ha, taking the total disturbance footprint to 980 ha, within the development envelope. The approval 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 LOM. The approval also includes the Hydrometallurgical Facility and the proposed new electricity generation plant at site.

Mining Proposal, Hydrometallurgical Facility was approved by Department of Mines Industry Regulation and Safety (DMIRS), on October 31, 2018, under the West Australian *Mining Act 1978*. The approval was granted to commence the development and operation of the Project in accordance with revised Mining Tenement Conditions. The revised tenement conditions reflect the material provided within the RHM, Mining Proposal document describing the Hydrometallurgical Facility and associated mining and operational changes to the Project. The approval allows for the onsite activities under the Mining Act, however do not allow for construction activities to commence.

A Works Approval for the Hydrometallurgical Facility was approved by the Department of Water and Environmental Regulation (DWER), on 30th November 2018, under Part V of the *Environmental Protection Act 1986*. The approval was granted to allow the construction of the Hydrometallurgical Facility, subject to conditions. RHM currently holds a Prescribed Premises License L8493/2010/2, permitting the control of emissions and discharges to the environment, and the monitoring and reporting of them. The Works Approval specifies emission levels such that during testing and commissioning of the Facility, the proposed emissions described in the Works Approval document, are confirmed to allow the issuing of a Prescribed Premises License Amendment, with nominated emission limits.

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 project area had changed (Outback Ecology, 2010b).

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 Project 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 Project area (Maia 2015). Collectively these recent surveys have now described the vegetation within the entire development envelope. 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 affected by the proposed new disturbance footprint of 980ha 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 led to positive results. 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 (the Paroo Station mine site), 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.

Paroo Station Mine is using the Landscape 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 Project is unique and complex due to naturally occurring (i.e. pre-mining disturbance) high baseline levels of lead in topsoil associated with elevated mineralized 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 (DEC, 2002).

The Bamford desktop survey identified an assemblage of 295 vertebrate fauna species potentially occurring in the Rosslyn Hill Mining 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 Malleefowl.

Species of conservation significance

Significant species expected to be present at least occasionally within the project 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 Rosslyn Hill 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 Rosslyn Hill 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 project 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
- 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 project 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 project 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 c*ease." 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. The CAR and AER's for 2018 will be submitted on or before March 31, 2019.

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

Site environmental monitoring and reporting requirements are being undertaken to ensure compliance with the relevant approvals and license conditions.

By letter dated February 2, 2015, the DWER advised the Company that it could cease sampling programs along the transport route and Fremantle Port while operations are in care-and-maintenance.

Site inspections and audits have been undertaken at various times in the past by DMIRS and DWER.

20.4.2 Mine Closure Plan

The Company submitted an updated Mine Closure Plan (MCP) on March 3, 2015 to DMIRS. The Plan was approved on 30th June 2015. The approved version has superseded all previous versions.

20.5 Post-Performance or Reclamations Bonds

20.5.1 Mining Rehabilitation Fund

On September 26, 2014, the Company was refunded A$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.

20.6 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 project 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 Environmental 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 1,300 km concentrate transport route.

Stakeholder consultation has been ongoing and has had a recent focus with proposed hydrometallurgical facility and mine extension.

20.7 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.

20.8 Reclamation and Closure Cost Estimate

20.8.1 Costing Methodology

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

The above assumptions and methodologies have been applied to the operation 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.

20.8.2 Estimated Cost

RHM has a fully-costed closure cost estimate that is consistent with the current Mine Closure Plan.

21 Capital and Operating Costs

This section outlines the capital and operating costs for the design, supply and construction of the new Hydrometallurgical Facility, modifications at the existing Flotation Concentrator, and other onsite and offsite infrastructure and support services. Other finance and related costs and inputs are discussed.

The capital cost (Capex) estimate is in US dollars, has been prepared to AACE Class 3 and has a base date of October 1, 2018.

The operating cost (Opex) estimate is current as at October 1, 2018 and there have been no material adverse changes since this date to any material items. The estimates are presented in US dollars (US$).

21.1 Capital Costs

The estimated costs for the new Hydrometallurgical Facility and modifications to the existing Flotation Concentrator are summarized in Table 55.

Table 55: Capex breakdown

Description	Costs (US$)	
DIRECT COSTS		
Site Development		2,253,240
Concentrator Modifications	9,974,255	9,974,255
Metallurgical Plant		90,540,171
Area 2005 – Feed Preparation	4,073,968	
Area 2010 – MSA Leach	1,453,212	
Area 2015 – MSA Leach Residue	4,786,462	
Area 2017 – Acid Leach	1,583,775	
Area 2020 – DeS Leach	869,920	
Area 2030 – Leach Area Scrubber	233,345	
Area 2040 – Tailings	31,992	
Area 2045 – Impurity Removal	982,154	
Area 2050 – Electrolyte Filtration	1,194,439	
Area 2056 – Bleed Treatment Electrowinning	3,557,328	
Area 2057 – Bleed Treatment Acid Recovery	2,577,370	
Area 2058 – Bleed Treatment Precipitation	761,681	
Area 2059 – Bleed Treatment Leaching	921,574	
Area 2060 – Lead Electrowinning	25,964,121	
Area 2065 – Lead Melting	14,103,097	
Area 2070 – Reagents	6,678,909	
Area 2080 – Oxygen	453,929	
Area 2090 – Pipe Racks	561,072	
Area 6000 – Services	6,011,440	
Area 6300 – Power Supply	7,357,855	
Area 6500 – Evaporator	5,951,747	
Area 7110 – Buildings	430,782	
Area 7200 – Owner's Direct Costs		5,357,835

Description	Costs (US$)	
Growth Allowance		6,434,602
Subtotal Direct Costs		114,560,104
INDIRECT COSTS		
Field Indirects		1,414,625
Freight		5,086,495
Spare Parts		3,120,680
First Fills		4,153,329
Construction Infrastructure		1,775,655
Construction Support		733,644
EPCM		16,921,667
Growth Allowance		184,815
Subtotal Direct Costs		33,390,910
Contingency		14,931,924
Total Estimate excl. Owner's Costs		162,882,937
Owner's Costs		20,833,000
Total Estimate		183,715,937

Outside of the new Hydrometallurgical Facility, certain modification works have been identified as being required in the existing concentrator facilities. The estimated costs for the modifications to the existing concentrator are summarized in Table 56.

Table 56: Other Capex estimate

Description	Estimate (US$)
Area 1000 – Plant Wide	1,574,320
Area 1005 – Primary Crushing	29,487
Area 1010 – Pebble Crushing	1,009,436
Area 1015 – Milling	3,569,835
Area 1020 – Rougher Flotation	542,192
Area 1025 – DES Flotation	1,229,363
Area 1035 – 2nd Cleaner Flotation	354,539
Area 1045 – Concentrate Thickening	1,253,841
Area 1045 – Concentrate Filtration	210,463
Area 1050 – Tailings Thickening and Pumping	9,579
Area 1060 – Concentrator Reagents	78,346
Area 6023 – Blower	112,854
Total Direct Costs Estimate	**9,974,255**

Owner's costs associated with the development of the project, including pre-production care-and-maintenance, working capital, and provisions for project and Lead*FX* overhead costs to production are set out in Table 57.

Table 57: Owner's costs

Description	Estimate (US$)
Corporate	2,396,474
Care-and-maintenance	
Tenements	72,202
Processing	79,861
OHS&E	125,984
Site Administration	3,381,809
Project Development	
Project Construction	780,456
Owners Project Team	3,390,067
Site Establishment	444,896
Temporary Facilities	152,461
Pre-Production	
Mining	870,111
Production	3,488,458
Sustainability	1,546,185
Supply & Logistics	638,618
Temporary facilities during construction	2,470,414
Contingency (5%)	992,044
Total	**20,833,000**

The total costs to develop the project to production are set out in Table 58. These costs exclude existing debts of RHM and financing costs during project construction and project reserve accounts estimated to be in the order of US$63M.

Table 58: Company costs to first production

Description	Estimate (US$)
Hydrometallurgical Facility	152,908,682
Concentrator Modifications	9,974,255
Owner's Costs	20,833,000
Total	**183,715,937**

21.2 Operating Costs

21.2.1 Basis of Estimate

The primary basis for the operating cost estimate is the mining schedule described in Chapter 6, the process design criteria described in Chapter 9, the labor schedule for the entire Project provided by RHM, consumables and reagents provided by RHM and/or determined by SNC-Lavalin, power costs based on third party quotes of unit costs and power loads determined by SNC-Lavalin together with general estimation undertaken by SNC-Lavalin and RHM in their areas of respective expertise.

The key operational physicals used to calculate unit operating costs are presented in Table 59.

Table 59: Operational physicals

Physicals	Unit	Life of Mine
Ore mined	BCM's	16,800,351
Waste mined	BCM's	48,203,654
Ore mined	dmt	33,360,947
Ore from existing stockpiles	dmt	2,926,561
Ore feed	dmt	36,287,508
Ore feed grade	% Pb	3.71
Flotation recovery	%	82.92
Concentrate produced	dmt	1,552,898
Hydrometallurgical Recovery	%	98.00
Lead ingot produced	Tonnes	1,092,681

The accuracy of the operating cost estimate is calculated to be within -10% +15% as deemed appropriate for a DFS.

21.2.2 OPEX Estimate

Table 60 summarizes the operating costs for the project presented by cost center. The build-up of each of these cost centers is explained in detail below. The same costs reported by cost center are reported by the cost category in Table 61.

Table 60: Operating cost summary

Cost Centre	Life of Mine		
	US$	US$/ t ore (feed)	US$/ t Pb (ingot)
Mining	362,791,836	10.00	332.02
Flotation Concentrator	394,655,986	10.88	361.18
Hydrometallurgical Facility	332,459,946	8.89	295.11
Supply and Logistics	149,702,419	4.13	137.00
Sustainability	45,657,670	1.26	41.78
Corporate and General and Administration	86,411,841	2.38	79.08
Sustaining Capital	32,882,477	0.91	30.09
Total	**1,394,562,175**	**38.43**	**1,276.28**

Table 61: Common cost categories

Common Cost Categories	Life of Mine		
	US$	US$/ t ore (feed)	US$/ t Pb (ingot)
Labour	341,093,548	9.40	312.16
Flights & Accommodation	89,417,623	2.46	81.83
Non-physicals	103,833,019	2.86	95.03
Power	187,091,207	5.16	171.22
Maintenance	97,010,255	2.67	88.78
Reagents & Consumables	177,239,215	4.88	162.21
Mining (fixed & variable)	239,962,492	6.61	219.61
Freight to Port and Port charges	126,032,340	3.47	115.34
Sustaining Capital	32,882,477	0.91	30.09
Total	**1,394,562,175**	**38.43**	**1,276.28**

21.2.3 All-in Sustaining Life of Mine Costs

The All-in Sustaining Costs (AISC), for the life of mine of US$1,394,562,175 are presented graphically against ore feed and ingot production in Figure 65 and Figure 66 respectively. On a unit basis, this can be represented as US$38.43/t ore feed or US$1,276.28/t of ingot production.

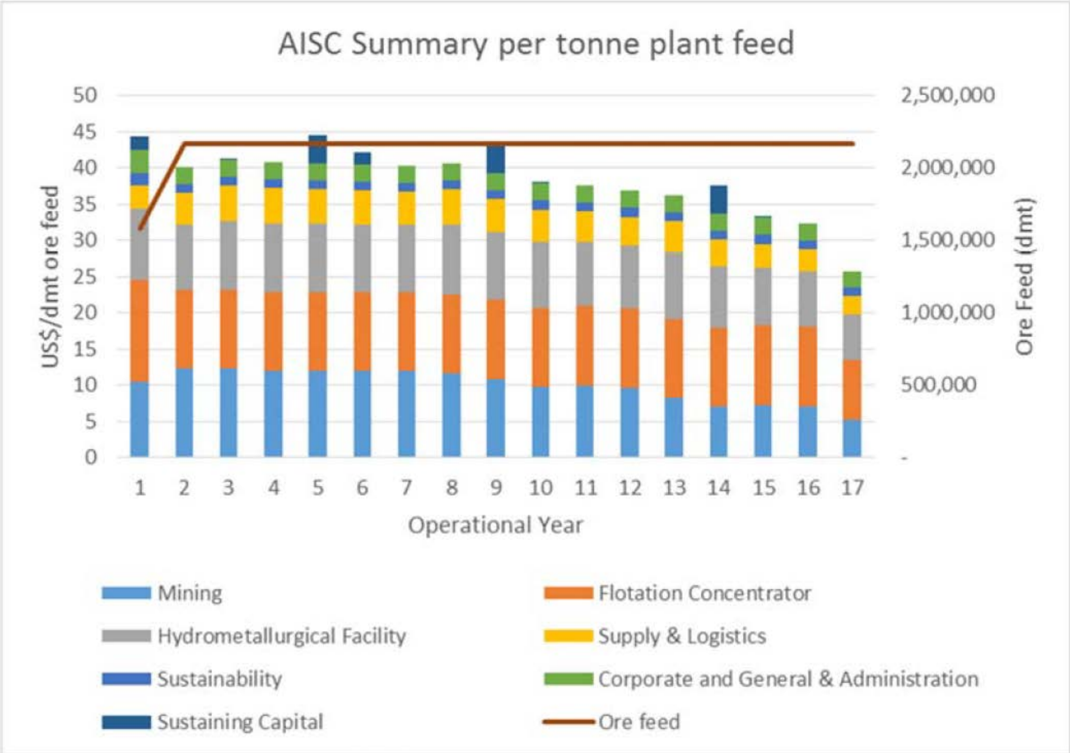

Figure 65: AISC summary by Ore feed

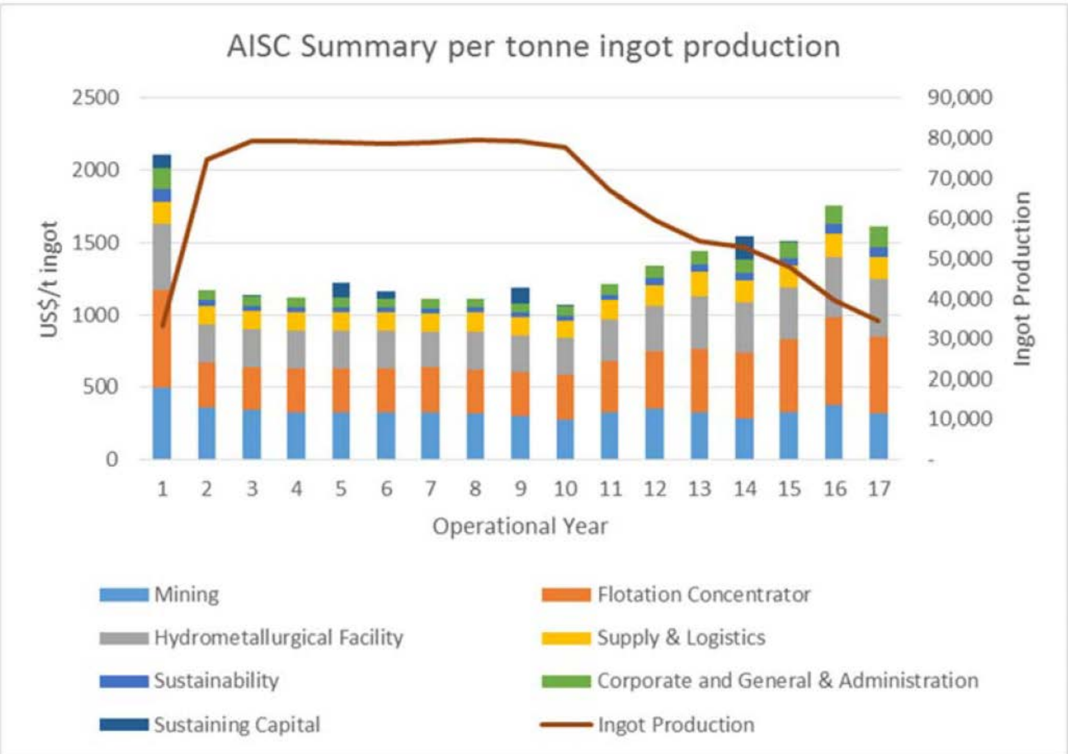

Figure 66: AISC summary by Ingot production

21.3 Sustaining Capital and Decommissioning

RHM will develop an Integrated Waste Landform (IWL) within the existing waste rock landform. There is sufficient existing area for the first 6–12 months of operations (operations ramp-up period), and during this period a new cell (IWL) will be constructed. An allowance of A$2,000,000 is provided for construction and A$200,000 for construction management in the financial analysis during the third quarter of the first year of operations.

The Hydrometallurgical Facility will require replacement of the anodes in the electrowinning tankhouse. Allowance has been made for replacement of anodes every five years at a cost of US$ 8,400,000 per replacement.

RHM has allowed for the purchase of five new light vehicles every five years over the LOM at an estimated cost of A$200,000 per replacement.

The sustaining capital requirements are set out in Table 62.

Table 62: Sustaining capital requirements

Description	Units	Estimate	One-off	Recurring
Balance for mobilisation	A$	327,135	Q2 operations	
Additional spares and consumables for existing plant	A$	150,000	Q2 operations	
Site access road relocation	A$	800,000	Q3 operations	
Integrated Waste Rock Landform	A$	2,200,000	Q3 operations	
Site Communications Upgrade	A$	614,500	Q4 operations	
Water Exploration	A$	725,000	Q9 operations	
IT Software & Hardware Replacement	A$	344,353	Q23 operations	
Mill upgrade for hard ore	A$	3,500,000	Y6 operations	
Anode replacement	US$	8,400,000		4.5Y operations
Light vehicle replacement	A$	200,000		5.0Y operations
Total LOM estimate	**US$**	32,882,477		

Decommission costs of A$18,805,775 are included in the financial analysis as set out in Table 63.

Table 63: Decommissioning costs

Description	Estimate (A$)
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	**18,805,775**

21.3.1 Esperance Settlement Agreement (ESA)

RHM has a contingent payment obligation of A$3.0M under the ESA between it, the State of Esperance Port Authority, and the State of Western Australia, payable when the Company reaches a cumulative EBITDA target of A$18M measured from January 1, 2009. Forecast as at March 31, 2019 RHM's cumulative EBITDA from January 1, 2009 was A$62.64M, and the Company is forecast to reach the cumulative EBITDA target in the second year of production when the payment is expected to be settled. The Company has previously made payments totaling A$6M under the agreement. The ESA contains some standard events of default that could require immediate payment of the remaining A$3M including (i) a liquidator, receiver or administrator taking possession of or is appointed over the whole or any party of the assets of the Company or (ii) the Company permanently ceases to operate the project.

21.4 Production

21.4.1 Mining Schedule

The mining schedule set out in Table 64 has been calculated by AMC Consultants Pty Ltd based on its Mineral Reserve estimate and the production ramp-up schedule.

Table 64: Mining schedule

Year	Waste mined * (dmt)	Ore mined * (dmt)	Ore Feed from mining (dmt)	Ore Feed from existing stockpile (dmt)	Ore Feed grade (%Pb)
Year 1	3,276,319	601,030	171,920	1,524,289	2.5%
Year 2	7,766,488	3,248,753	1,767,833	400,267	4.0%
Year 3	9,478,056	1,831,354	1,925,607	242,493	4.4%
Year 4	8,482,762	2,546,282	2,071,845	102,195	4.4%
Year 5	9,007,328	1,924,845	2,168,100	-	4.3%
Year 6	8,691,709	2,071,837	2,168,100	-	4.2%
Year 7	9,064,740	1,974,954	2,168,100	-	4.1%
Year 8	9,064,740	1,974,954	2,168,100	-	4.1%
Year 9	7,762,567	1,653,845	2,168,100	-	4.2%
Year 10	5,855,343	2,544,037	2,168,100	-	4.0%
Year 11	6,083,700	2,244,584	2,168,100	-	3.7%
Year 12	4,967,834	2,034,170	2,174,040	-	3.5%
Year 13	2,755,754	2,691,471	2,168,100	-	4.0%
Year 14	971,567	1,892,681	2,168,100	-	3.3%
Year 15	1,145,712	1,742,302	2,168,100	-	2.9%
Year 16	488,038	1,246,629	2,174,040	-	2.6%
Year 17	15,840	84,162	1,388,721	657,318	2.2%
Total	**93,213,450**	**33,360,947**	**33,360,947**	**2,926,561**	**3.71%**

Notes: * dry bulk density for ore of 2.00 dmt/bcm and for waste 1.94 dmt/bcm. Stripping ratio of 2.99.

21.4.2 Ingot Production

Production from the Flotation Concentrator and Hydrometallurgical Facility is set out in Table 65.

Table 65: Production

Year	Lead in Ore Feed (t) Pb	Flotation Recovery (%)	Lead in Con (t) Pb	Con Grade (%) Pb	Con Tonnage (dtm)	Hydromet Recovery (%)	Lead Ingot (t) Pb
Year 1	42,402	80.4%	34,112	71.8%	47,510	98.0%	33,430
Year 2	86,813	83.3%	72,280	71.8%	100,669	98.0%	70,834
Year 3	94,883	83.9%	79,652	71.8%	110,936	98.0%	78,059
Year 4	95,378	84.0%	80,086	71.8%	111,540	98.0%	78,484
Year 5	92,971	83.8%	77,897	71.8%	108,492	98.0%	76,340
Year 6	91,711	83.7%	76,761	71.8%	106,910	98.0%	75,226
Year 7	88,489	83.4%	73,829	71.8%	102,826	98.0%	72,352
Year s	96,527	84.1%	81,142	71.8%	113,011	98.0%	79,519
Year 9	90,814	83.6%	75,922	71.8%	105,741	98.0%	74,403
Year 10	86,283	83.2%	71,806	71.8%	100,009	98.0%	70,370
Year 11	81,138	82.8%	67,189	71.8%	93,577	98.0%	65,845
Year 12	77,015	82.4%	63,435	71.8%	88,350	98.0%	62,166
Year 13	85,923	83.2%	71,528	71.8%	99,621	98.0%	70,097
Year 14	72,257	81.9%	59,182	71.8%	82,426	98.0%	57,998
Year 15	62,173	80.9%	50,299	71.8%	70,054	98.0%	49,293
Year 16	55,514	80.3%	44,553	71.8%	62,051	98.0%	43,662
Year 17	44,325	79.3%	35,165	71.8%	48,977	98.0%	34,462
Total	**1,344,617**	**82.9%**	**1,114,837**	**71.80%**	**1,552,698**	**98.0%**	**1,092,540**

22 Economic Analysis

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).
- A$:US$ exchange rate of 0.726 for the entire LOM.
- No assumption regarding project debt financing, and as such the project cashflows presented are ungeared.
- Australian corporate tax rate of 30% and availability of the R&D tax offset incentive scheme for US$43.9M of labor expenditure during construction.
- Mineral Reserves estimate and production schedule by AMC Consultants Pty Ltd.
- Long-term LME lead cash price forecasts from the Wood Mackenzie Lead Market Assessment.
- Long-term lead premium forecasts from the Wood Mackenzie Lead Market Assessment.
- Hydrometallurgical Facility and Flotation Concentrator capital costs and operating costs estimated by SNC-Lavalin.
- Owner's costs during construction, mining costs and all other operating costs outside of the Hydrometallurgical Facility and Flotation Concentrator estimated by RHM.
- Lead ingot transportation costs estimated by RHM.

22.1.1 Lead Price Forecasts and Sales Price

Prior to production, RHM plans to enter into an offtake contract with a major global trading company for the sale and purchase of 100% of the annual lead ingot production of the Project. Under such offtake arrangements, RHM will sell lead ingot on an FOB basis Fremantle, i.e. RHM will be responsible for delivering the lead ingots to container vessels in Fremantle, but will not be responsible for subsequent ocean freight and delivery to end-users.

The sales price received by RHM will comprise (i) the prevailing LME cash price for lead, plus (ii) a premium. Premiums are typically benchmarked off CIF Index Premiums for specific destinations quoted by Fastmarkets MB, and will be agreed by RHM and offtakers based on the lead ingot specification and subject to adjustment for offtaker costs for managing risk, marketing, freight, insurance and other costs.

The financial analysis is based on forecasts from the Wood Mackenzie Lead Market Assessment for:

- Long-term LME lead prices
- Long-term lead premiums for index premium destinations
- Freight costs to index premium destinations.

Lead index premiums in the financial analysis are based on 99.99% Pb ingot, CIF index premiums and freight netbacks are based on Southeast Asian destinations, and index lead premiums are adjusted for the aforementioned offtaker costs.

Table 66: Wood Mackenzie LME price forecasts (US$/t Pb)

Calendar Year	Forecast	Lower Limit	Upper Limit
2021	2,039	1,844	2,174
2022	2,022	1,764	2,200
2023	1,960	1,647	2,176
2024	2,060	1,665	2,331
2025	2,350	1,900	2,660
2026	2,350	1,900	2,660
2027	2,350	1,900	2,660
2028	2,350	1,900	2,660
2029	2,350	1,900	2,660
2030	2,350	1,900	2,660
2031	2,350	1,900	2,660
2032	2,350	1,900	2,660
2033	2,350	1,900	2,660
2034	2,350	1,900	2,660
2035	2,350	1,900	2,660
2036	2,350	1,900	2,660
2037	2,350	1,900	2,660

Table 67: Ingot sales price (US$/t Pb)

Calendar Year	LME Price Forecast	SE Asia net CIF basis	Sale Price CIF basis	Fright Netback for FOB sale	Sale Price FOB basis
2021	2,039	145	2,184	-51	2,133
2022	2,022	145	2,167	-51	2,116
2023	1,960	145	2,105	-51	2,054
2024	2,060	145	2,205	-51	2,154
2025	2,350	145	2,495	-51	2,444
2026	2,350	145	2,495	-51	2,444
2027	2,350	145	2,495	-51	2,444
2028	2,350	145	2,495	-51	2,444
2029	2,350	145	2,495	-51	2,444
2030	2,350	145	2,495	-51	2,444
2031	2,350	145	2,495	-51	2,444
2032	2,350	145	2,495	-51	2,444
2033	2,350	145	2,495	-51	2,444
2034	2,350	145	2,495	-51	2,444
2035	2,350	145	2,495	-51	2,444
2036	2,350	145	2,495	-51	2,444
2037	2,350	145	2,495	-51	2,444

22.1.2 Sales Revenue

The forecast sales prices and sales value for lead ingot by year of production are set out in Table 68.

Table 68: Sales value

Production Year	Sales Price US$/t Pb	Sales Amount t Pb	Sales Value US$ million
Year 1	2,145	33,430	71.72
Year 2	2,120	70,834	150.20
Year 3	2,070	78,059	161.59
Year 4	2,130	78,484	167.15
Year 5	2,368	76,340	180.77
Year 6	2,444	75,226	183.85
Year 7	2,444	72,352	176.83
Year 8	2,444	79,519	194.35
Year 9	2,444	74,403	181.84
Year 10	2,444	70,370	171.98
Year 11	2,444	65,845	160.92
Year 12	2,444	62,166	151.93
Year 13	2,444	70,097	171.32
Year 14	2,444	57,998	141.75
Year 15	2,444	49,293	120.47
Year 16	2,444	43,662	106.71
Year 17	2,444	34,462	84.23
Avg/Total	**2,359**	**1,092,540**	**2,578**

22.1.3 Cashflows

Project cashflows by year of operation are set out in Table 69 and Table 70.

Table 69: Annual revenue and costs (US$ million)

Year	Total Revenue	Royalties	Mining	Flotation	Hydromet	Supply & Logistics	Other Opex
Year 1	71.39	-2.09	-16.63	-22.34	-15.34	-5.26	-7.79
Year 2	150.08	-4.30	-26.66	-23.45	-19.66	-9.58	-7.79
Year 3	160.35	-4.50	-26.92	-23.46	-20.60	-10.40	-7.79
Year 4	169.05	-4.83	-26.01	-23.52	-20.68	-10.45	-7.81
Year 5	186.61	-5.60	-26.08	-23.54	-20.53	-10.20	-7.79
Year 6	183.85	-5.52	-25.93	-23.55	-20.40	-10.07	-7.79
Year 7	176.83	-5.31	-26.15	-23.57	-20.07	-9.74	-7.79
Year 8	194.35	-5.83	-25.38	-23.58	-20.93	-10.57	-7.81
Year 9	181.84	-5.46	-23.64	-23.55	-20.31	-9.98	-7.79
Year 10	171.98	-5.17	-21.36	-23.58	-19.84	-9.51	-7.79
Year 11	160.92	-4.84	-21.79	-23.62	-19.31	-8.99	-7.79
Year 12	151.93	-4.58	-20.93	-23.72	-18.91	-8.57	-7.81
Year 13	171.32	-5.15	-17.93	-23.59	-19.80	-9.48	-7.79
Year 14	141.75	-4.27	-15.23	-23.69	-18.39	-8.09	-7.79
Year 15	120.47	-3.65	-15.78	-23.77	-17.36	-7.08	-7.79
Year 16	106.71	-3.24	-15.20	-23.91	-16.72	-6.44	-7.81
Year 17	84.23	-2.57	-11.17	-18.21	-13.62	-5.29	-7.36
Total	**2,584**	**-77**	**-363**	**-395**	**-322**	**-150**	**-132**

Table 70: Annual cashflows (US$ million)

Year	Sales Revenue	Variable Opex	Fixed Opex	Ongoing Capex	Gross Cashflow	Income Tax *	Net Cashflow
Year 1	71.39	-23.81	-45.63	-2.97	-1.03	2.90	1.88
Year 2	150.08	-43.13	-48.31	0.00	58.65	0.00	58.65
Year 3	160.35	-45.24	-48.43	-2.70	63.98	0.00	63.98
Year 4	169.05	-45.21	-48.09	0.00	75.75	-17.12	58.63
Year 5	186.61	-45.59	-48.16	-8.55	84.33	-22.97	61.36
Year 6	183.85	-45.10	-48.15	-3.75	86.85	-23.55	63.30
Year 7	176.83	-44.45	-48.17	0.00	84.20	-22.87	61.33
Year 8	194.35	-45.99	-48.11	0.00	100.25	-26.41	73.84
Year 9	181.84	-43.60	-47.12	-8.40	82.71	-22.81	59.90
Year 10	171.98	-41.02	-46.23	-0.15	84.59	-22.30	62.29
Year 11	160.92	-40.06	-46.28	0.00	74.59	-19.66	54.93
Year 12	151.93	-38.60	-45.91	0.00	67.42	-17.76	49.66
Year 13	171.32	-38.37	-45.36	0.00	87.58	-23.02	64.57
Year 14	141.75	-32.05	-45.41	-8.40	55.89	-14.47	41.41
Year 15	120.47	-29.94	-45.49	-0.15	44.90	-11.36	33.54
Year 16	106.71	-27.63	-45.68	0.00	33.40	-8.51	24.89
Year 17	84.23	-22.44	-35.78	-13.65	12.35	-3.60	8.74
Total	**2,584**	**-652**	**-786**	**-49**	**1,096**	**-253**	**843**

Note:
* Calculation of corporate income tax includes an estimated carry forward Australian tax loss of A$48.7M and written down value of existing property, plant and equipment for tax purposes of A$28.9M, forecast as at March 31, 2019, and availability of the R&D tax offset incentive scheme for US$43.9M of labor expenditure during construction.

22.1.4 Financial Returns

The financial returns based on real US dollar cashflows on an ungeared and after-tax basis, using LME cash price forecasts provided by Wood Mackenzie for LOM, are set out in Table 71.

Table 71: Financial returns

Description	Units	Estimate	Comments
Total cost to production	US$ million	184	to start of operations
Payback Period	years	4.00	from start of operations
Internal Rate of Return	%pa	24.6%	from start of construction
After-tax project cashflow:			
Project revenue	US$ million	2,584	from start of operations
Less: All-in sustaining costs	US$ million	-1,487	from start of operations
Cashflow before tax	US$ million	1,096	from start of operations
Less: Income tax	US$ million	-253	from start of operations
Cashflow after tax	US$ million	843	from start of operations
Present Value:			
- GPV (8.25%pa real discount rate) *	US$ million	430	from start of construction
- NPV (8.25%pa real discount rate) ^	US$ million	257	from start of construction

Notes:

* GPV = gross present value = present value of cashflow after tax

^ NPV = net present value (i.e. GPV - costs to production)

** All-in sustaining costs in this case includes royalties, the ESA and decommissioning.

22.1.5 Sensitivity Analysis

The sensitivity of financial returns to changes in LME price forecasts, and capital costs and operating costs are set out in Table 72.

Table 72: Sensitivity table

Description	Units	DFS Case	WoodMac Upper Limit	WoodMac Low Limit	LME Price - 10%	DFS Capex + 10%	Production - 10%	Site Opex + 10%	AUD/USD +10%	Recovery Rate - #
Total cost to production	US$ million	184	184	184	184	199	184	184	184	184
Payback Period	years	4.00	29.6%	15.8%	19.6%	22.9%	20.8%	21.7%	22.0%	23.0%
Internal Rate of Return	%pa	24.6%	3.50	5.25	4.75	4.00	4.50	4.25	4.25	4.00
After-tax project cashflow:										
Project revenue	US$ million	2,584	2,897	2,129	2,336	2,584	2,325	2,584	2,584	2,496
Less: All-in sustaining costs	US$ million	-1,487	-1,497	-1,471	-1,480	-1,487	-1,422	-1,620	-1,607	-1,477
Cashflow before tax	US$ million	1,096	1,400	659	856	1,096	903	963	977	1,019
Less: Income tax	US$ million	-253	-345	-124	-182	-249	-196	-214	-218	-230
Cashflow after tax	US$ million	843	1,055	534	674	848	707	749	759	788
Present Value:										
- GPV (8.25%pa real discount rate) *	US$ million	430	535	276	343	432	361	381	386	402
- NPV (8.25%pa real discount rate) ^	US$ million	257	362	104	171	244	188	208	214	229

Notes:

* GPV = gross present value = present value of cashflow after tax.

^ NPV = net present value = GPV – present value of total cost to production.

Recovery rate for Flotation Concentrator reduced by 2% and recovery rate for Hydrometallurgical Facility reduced by 1%.

** All-in sustaining costs in this case includes royalties, the ESA and decommissioning.

22.1.6 Annual Statistics

Annual statistics summarizing physicals, financials, revenue allocation and forecast sales price and value are provided in Tables 74– 77 respectively.

Table 73: Physicals

Description	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Production - Ore Feed							
Plant ore feed-rate	dmt ore	1,696,209	2,168,100	2,168,100	2,174,040	2,168,100	2,168,100
Ore head grade	%Pb	2.50%	4.00%	4.38%	4.39%	4.29%	4.23%
Contained lead in ore feed	t Pb	42,402	86,813	94,883	95,378	92,971	91,711
Production - Concentrate							
Flotation recovery	%	80.5%	83.4%	84.0%	84.0%	83.80%	83.7%
Lead in concentrate	t Pb	34,145	72,363	79,656	80,093	77,915	76,761
Concentrate grade	%	71.8%	71.8%	71.8%	71.8%	71.8%	71.8%
Concentrate production	dmt con	47,556	100,784	110,941	111,550	108,516	106,910
Production - Ingots							
Lead in concentrate	t Pb	34,145	72,363	79,656	80,093	77,915	76,761
Hydromet recovery	%	98.0%	98.0%	98.0%	98.0%	98.0%	98.0%
Lead ingot	t Pb	33,462	70,915	78,063	78,491	76,356	75,226
Sales Price (real)							
LME Lead Price	US$/t Pb	2,039	2,022	1,960	2,060	2,350	2,350
Net Lead Premium	US$/t Pb	94	94	94	94	94	94
Ingot Price	US$/t Pb	2,133	2,116	2,054	2,154	2,444	2,444

Description	Units	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Production - Ore Feed							
Plant ore feed-rate	dmt ore	2,168,100	2,174,040	2,168,100	2,168,100	2,168,100	2,174,040
Ore head grade	%Pb	4.08%	4.44%	4.19%	3.98%	3.74%	3.54%
Contained lead in ore feed	t Pb	88,489	96,527	90,814	86,283	81,138	77,015
Production - Concentrate							
Flotation recovery	%	83.4%	84.1%	83.6%	83.2%	82.8%	82.4%
Lead in concentrate	t Pb	73,829	81,142	75,922	71,806	67,189	63,435
Concentrate grade	%	71.8%	71.8%	71.8%	71.8%	71.8%	71.8%
Concentrate production	dmt con	102,826	113,011	105,741	100,009	93,577	88,350
Production - Ingots							
Lead in concentrate	t Pb	73,829	81,142	75,922	71,806	67,189	63,435
Hydromet recovery	%	98.0%	98.0%	98.0%	98.0%	98.0%	98.0%
Lead ingot	t Pb	72,352	79,519	74,403	70,370	65,845	62,166
Sales Price (real)							
LME Lead Price	US$/t Pb	2,350	2,350	2,350	2,350	2,350	2,350
Net Lead Premium	US$/t Pb	94	94	94	94	94	94
Ingot Price	US$/t Pb	2,444	2,444	2,444	2,444	2,444	2,444

Description	Units	Year 13	Year 14	Year 15	Year 16	Year 17	Years 1-17
Production - Ore Feed							
Plant ore feed-rate	dmt ore	2,168,100	2,168,100	2,168,100	2,174,040	2,046,039	36,287,508
Ore head grade	%Pb	3.96%	3.33%	2.87%	2.55%	2.17%	3.71%
Contained lead in ore feed	t Pb	85,923	72,257	62,173	55,514	44,325	1,344,617
Production - Concentrate							
Flotation recovery	%	83.2%	81.9%	80.9%	80.3%	79.3%	82.9%
Lead in concentrate	t Pb	71,528	59,182	50,299	44,553	35,165	1,114,981
Concentrate grade	%	71.8%	71.8%	71.8%	71.8%	71.8%	71.8%
Concentrate production	dmt con	99,621	82,426	70,054	62,051	48,977	1,552,898
Production - Ingots							
Lead in concentrate	t Pb	71,528	59,182	50,299	44,553	35,165	1,114,981
Hydromet recovery	%	98.0%	98.0%	98.0%	98.0%	98.0%	98.0%
Lead ingot	t Pb	70,097	57,998	49,293	43,662	34,462	1,092,681
Sales Price (real)							
LME Lead Price	US$/t Pb	2,350	2,350	2,350	2,350	2,350	2,271
Net Lead Premium	US$/t Pb	94	94	94	94	94	94
Ingot Price	US$/t Pb	2,444	2,444	2,444	2,444	2,444	2,365

Table 74: Financials

Description	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Ingot Sales							
LME Component	US$	68,241,071	143,415,063	153,014,718	161,674,065	179,437,236	176,781,423
Refined Lead Premium	US$	3,145,450	6,666,049	7,337,879	7,378,164	7,177,489	7,071,257
Total Revenue	US$	**71,386,522**	**150,081,112**	**160,352,597**	**169,052,228**	**186,614,726**	**183,852,680**
Operating Costs							
Royalties	US$	(2,091,516)	(4,295,742)	(4,503,316)	(4,827,343)	(5,598,718)	(5,517,214)
Mining	US$	(16,634,024)	(26,656,060)	(26,916,815)	(26,012,178)	(26,082,956)	(25,927,517)
Flotation	US$	(22,336,278)	(23,451,776)	(23,461,394)	(23,523,951)	(23,538,721)	(23,547,851)
Hydromet	US$	(15,337,255)	(19,663,446)	(20,598,018)	(20,679,159)	(20,531,458)	(20,399,795)
Supply & Logistics	US$	(5,255,567)	(9,575,501)	(10,399,867)	(10,453,122)	(10,203,061)	(10,072,709)
Sustainability	US$	(2,692,689)	(2,692,689)	(2,692,689)	(2,700,066)	(2,692,689)	(2,692,689)
Site Mgt/Support	US$	(3,269,089)	(3,269,089)	(3,269,089)	(3,278,046)	(3,269,089)	(3,269,089)
Corporate	US$	(1,827,101)	(1,827,101)	(1,827,101)	(1,832,107)	(1,827,101)	(1,827,101)
ESA	US$	-	-	(2,178,000)	-	-	-
Sustaining Capex	US$	(2,970,527)	-	(526,350)	-	(8,545,200)	(3,750,000)
Decommissioning	US$	-	-	-	-	-	-
Total Sustaining Costs	US$	**(72,414,045)**	**(91,431,404)**	**(96,372,639)**	**(93,305,972)**	**(102,288,994)**	**(97,003,966)**
Cashflow before tax	US$	**(1,027,523)**	**58,649,708**	**63,979,958**	**75,746,257**	**84,325,732**	**86,848,714**
Corporate income tax	US$	2,904,000	-	-	(17,120,345)	(22,967,075)	(23,545,989)
Cashflow after tax	US$	**1,876,477**	**58,649,708**	**63,979,958**	**58,625,912**	**61,358,657**	**63,302,725**

Description	Units	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Ingot Sales							
LME Component	US$	170,027,538	186,870,589	174,847,743	165,369,640	154,735,160	146,090,947
Refined Lead Premium	US$	6,801,102	7,474,824	6,993,910	6,614,786	6,189,406	5,843,638
Total Revenue	US$	**176,828,639**	**194,345,412**	**181,841,652**	**171,984,425**	**160,924,567**	**151,934,585**
Operating Costs							
Royalties	US$	(5,309,946)	(5,827,091)	(5,457,872)	(5,167,000)	(4,840,640)	(4,575,612)
Mining	US$	(26,149,718)	(25,379,470)	(23,644,193)	(21,364,223)	(21,788,207)	(20,929,159)
Flotation	US$	(23,569,513)	(23,582,022)	(23,552,948)	(23,584,390)	(23,622,213)	(23,717,950)
Hydromet	US$	(20,066,523)	(20,929,174)	(20,305,483)	(19,836,741)	(19,308,266)	(18,911,763)
Supply & Logistics	US$	(9,741,216)	(10,571,728)	(9,977,801)	(9,512,598)	(8,990,639)	(8,570,190)
Sustainability	US$	(2,692,689)	(2,700,066)	(2,692,689)	(2,692,689)	(2,692,689)	(2,700,066)
Site Mgt/Support	US$	(3,269,089)	(3,278,046)	(3,269,089)	(3,269,089)	(3,269,089)	(3,278,046)
Corporate	US$	(1,827,101)	(1,832,107)	(1,827,101)	(1,827,101)	(1,827,101)	(1,832,107)
ESA	US$	-	-	-	-	-	-
Sustaining Capex	US$	-	-	(8,400,000)	(145,200)	-	-
Decommissioning	US$	-	-	-	-	-	-
Total Sustaining Costs	US$	**(92,625,795)**	**(94,099,704)**	**(99,127,175)**	**(87,399,031)**	**(86,338,845)**	**(84,514,892)**
Cashflow before tax	US$	**84,202,844**	**100,245,708**	**82,714,478**	**84,585,394**	**74,585,722**	**67,419,693**
Corporate income tax	US$	(22,869,535)	(26,407,330)	(22,811,832)	(22,295,242)	(19,657,926)	(17,762,890)
Cashflow after tax	US$	**61,333,309**	**73,838,378**	**59,902,646**	**62,290,151**	**54,927,796**	**49,656,803**

Description	Units	Year 13	Year 14	Year 15	Year 16	Year 17	Years 1-17
Ingot Sales							
LME Component	US$	164,728,729	136,295,276	115,837,623	102,604,794	80,986,144	2,480,957,757
Refined Lead Premium	US$	6,589,149	5,451,811	4,633,505	4,104,192	3,239,446	102,712,056
Total revenue	US$	**171,317,878**	**141,747,087**	**120,471,127**	**106,708,986**	**84,225,590**	**2,583,669,814**
Operating Costs							
Royalties	US$	(5,147,331)	(4,274,742)	(3,646,920)	(3,241,072)	(2,572,328)	(76,894,403)
Mining	US$	(17,928,422)	(15,233,175)	(15,777,642)	(15,199,046)	(11,169,032)	(362,791,836)
Flotation	US$	(23,588,780)	(23,688,109)	(23,773,864)	(23,905,901)	(18,210,327)	(394,655,986)
Hydromet	US$	(19,802,781)	(18,391,594)	(17,361,968)	(16,717,455)	(13,619,064)	(322,459,946)
Supply & Logistics	US$	(9,481,141)	(8,085,576)	(7,081,477)	(6,435,811)	(5,294,413)	(149,702,419)
Sustainability	US$	(2,692,689)	(2,692,689)	(2,692,689)	(2,700,066)	(2,545,144)	(45,657,670)
Site Mgt/Support	US$	(3,269,089)	(3,269,089)	(3,269,089)	(3,278,046)	(3,089,961)	(55,431,216)
Corporate	US$	(1,827,101)	(1,827,101)	(1,827,101)	(1,832,107)	(1,726,986)	(30,980,624)
ESA	US$	-	-	-	-	-	(2,178,000)
Sustaining Capex	US$	-	(8,400,000)	(145,200)	-	-	(32,882,477)
Decommissioning	US$	-	-	-	-	(13,652,993)	(13,652,993)
Total Sustaining Costs	US$	**(83,737,334)**	**(85,862,075)**	**(75,575,950)**	**(73,309,504)**	**(71,880,247)**	**(1,487,287,570)**
Cashflow before tax	US$	**87,580,544**	**55,885,012**	**44,895,178**	**33,399,482**	**12,345,343**	**1,096,382,243**
Corporate income tax	US$	(23,015,271)	(14,473,804)	(11,358,936)	(8,510,399)	(3,601,697)	(253,494,272)
Cashflow after tax	US$	**64,565,273**	**41,411,208**	**33,536,242**	**24,889,083**	**8,743,646**	**842,887,972**

Note:

* Calculation of corporate income tax includes an estimated carry forward Australian tax loss of A$48.7M and written down value of existing property, plant and equipment for tax purposes of A$28.9M, forecast as at March 31, 2019, and availability of the R&D tax offset incentive scheme for US$43.9M of labor expenditure during construction.

Table 75: Revenue allocation

Description	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Revenue Allocation							
Royalties	US$/t Pb	63	61	58	62	73	73
Mining	US$/t Pb	497	376	345	331	342	345
Flotation	US$/t Pb	668	331	301	300	308	313
Hydromet	US$/t Pb	458	277	264	263	269	271
Supply & Logistics	US$/t Pb	157	135	133	133	134	134
Sustainability	US$/t Pb	80	38	34	34	35	36
Site Mgt/Support	US$/t Pb	98	46	42	42	43	43
Corporate	US$/t Pb	55	26	23	23	24	24
ESA	US$/t Pb	-	-	28	-	-	-
Sustaining Capex	US$/t Pb	89	-	7	-	112	50
Decommissioning	US$/t Pb	-	-	-	-	-	-
Taxes	US$/t Pb	(87)	-	-	218	301	313
Equity Return	US$/t Pb	56	827	820	747	804	841
Total	US$/t Pb	**2,133**	**2,116**	**2,054**	**2,154**	**2,444**	**2,444**

Description	Units	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Revenue Allocation							
Royalties	US$/t Pb	73	73	73	73	74	74
Mining	US$/t Pb	361	319	318	304	331	337
Flotation	US$/t Pb	326	297	317	335	359	382
Hydromet	US$/t Pb	277	263	273	282	293	304
Supply & Logistics	US$/t Pb	135	133	134	135	137	138
Sustainability	US$/t Pb	37	34	36	38	41	43
Site Mgt/Support	US$/t Pb	45	41	44	46	50	53
Corporate	US$/t Pb	25	23	25	26	28	29
ESA	US$/t Pb	-	-	-	-	-	-
Sustaining Capex	US$/t Pb	-	-	113	2	-	-
Decommissioning	US$/t Pb	-	-	-	-	-	-
Taxes	US$/t Pb	316	332	307	317	299	286
Equity Return	US$/t Pb	848	929	805	885	834	799
Total	US$/t Pb	**2,444**	**2,444**	**2,444**	**2,444**	**2,444**	**2,444**

Description	Units	Year 13	Year 14	Year 15	Year 16	Year 17	Years 1-17
Revenue Allocation							
Royalties	US$/t Pb	73	74	74	74	75	70
Mining	US$/t Pb	256	263	320	348	324	332
Flotation	US$/t Pb	337	408	482	548	528	361
Hydromet	US$/t Pb	283	317	352	383	395	295
Supply & Logistics	US$/t Pb	135	139	144	147	154	137
Sustainability	US$/t Pb	38	46	55	62	74	42
Site Mgt/Support	US$/t Pb	47	56	66	75	90	51
Corporate	US$/t Pb	26	32	37	42	50	28
ESA	US$/t Pb	-	-	-	-	-	2
Sustaining Capex	US$/t Pb	-	145	3	-	-	30
Decommissioning	US$/t Pb	-	-	-	-	396	12
Taxes	US$/t Pb	328	250	230	195	105	232
Equity Return	US$/t Pb	921	714	680	570	254	771
Total	US$/t Pb	**2,444**	**2,444**	**2,444**	**2,444**	**2,444**	**2,365**

Table 76: Forecast sales price and value

Year	Sales price			Sales value		
	LME Price (US$/t Pb)	Metals Premia (US$/t Pb)	Sales Price (US$/t Pb)	Sales Price (US$/t Pb)	Sales Amount (t Pb)	Sales Value (US$ M)
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 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
Average/ Total	**2,250**	**85**	**2,335**	**2,335**	**924,093**	**2,158**

23 Adjacent Properties

The Mine deposits are stratabound, occurring in the Proterozoic Earaheedy Group, with mineralization being largely restricted to the unconformable contact between the Yelma and Maraloou formations. These deposits are unusual due to the near-total absence of sulfides; 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 mineralization similar to the Mine's deposits, nearby exploration is targeting base metal and gold mineralization in the Earaheedy and Yerrida basins, as well as the local Wiluna and Joyners Find Archaean 'greenstone' belts.

Great Western Exploration Limited (GWE) is currently exploring the Chisel and Frustration Well prospects for copper mineralization at its Yerrida Basin tenement package. The Chisel prospect, located approximately 15 km north of the Mine, is defined by a gravity anomaly within favorable stratigraphy in the prospective Maraloou Formation rocks. GWE indicates volcanogenic massive sulfide (VMS), sedimentary-hosted copper-cobalt, sedimentary lead-zinc or intrusion-related base and/ or precious metals mineralization may be present (Great Western Exploration corporate website, 2018).

Blackham Resources continues to mine and explore the Wiluna greenstone belt 30 km east of the Mine for Archean lode-hosted gold deposits. Blackham's Matilda/ Wiluna Gold Operation is the centerpiece of a 1,100 km^2 tenement package with total JORC Code (2012) Mineral Resources of 65 Mt at a grade of 3.1 g/t Au for 6.5 Moz Au. Mining and exploration are focusing on the Wiluna Mine Sequence and 10 km 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) has the Wiluna West direct shipping ore (DS) iron and gold projects located in the Joyners Find greenstone belt 30 km south of the Mine. The Wiluna West iron project has a JORC Code (2004) compliant Mineral Resource totaling 130.3 Mt at an average grade of 60% Fe, including 69.2 Mt 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 Code (2004) and JORC Code (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 and 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 Paroo Station Lead Mine.

24.1 EPCM Contractor

The Engineering, Procurement and Construction Management (EPCM) model was determined to be the most appropriate project delivery method for planning the execution phase of the Project.

An EPCM Contractor will take the Project from design through to commissioning. The EPCM scope of work will include detailed engineering, procurement, construction management and commissioning of the Process Plant and its associated infrastructure.

RHM is currently investigating other contracting models such as Guaranteed Maximum Price (GMP) or EPC/Lump Sum Turnkey (LSTK), and will progress as discussions with Project lenders continue.

24.2 Construction

The proposed EPCM contract for the Hydrometallurgical Facility will include the onsite construction of the facility. The Paroo Station 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.

24.3 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.

24.4 Concentrator Startup

The concentrator plant startup will be undertaken based on the 2013 plant startup which was both successful and well documented. The startup 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.

24.5 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 months production at steady state.

24.6 Steady State Operation

Steady state operation will be achieved when the concentrator plant is achieving 102,000tpa concentrate at 70%Pb. With the planned modifications, steady state is forecast 5 months from startup. The hydrometallurgical facility steady state will be when the facility is producing the equivalent of 70,000tpa on a daily basis.

24.7 Operational Readiness

The Mine, Flotation Concentrator and Hydrometallurgical Facility operations are based on operation for 24 hours a day, 365 days a year to treat up to 2.185 million dry tonnes of ore per annum (Mtpa), producing lead concentrate for treatment in the Hydrometallurgical Facility to produce lead ingot.

Concentrator throughput will be adjusted as a function of ore head grade and lead flotation recovery to match the treatment capacity of the Hydrometallurgical Facility.

To allow for process disruptions, planned maintenance and other unforeseen factors, an overall operating utilisation factor of 92.0% has been applied to the design of the plant and to estimate steady state flow rates.

Planned outages, included in the availability factor, will be used to service equipment. During such periods, production on the remaining operating streams will be optimised to minimise the production losses.

The primary business units identified for the Operations Phase will be fully supported onsite by maintenance, engineering, technical, Health, Safety, Environment and Community (HSEC), Human Resources (HR), and finance/commercial groups of the organisation.

The Operations Phase will be staffed using a combination of recruitment and manning strategies for existing and new employees. All operations employees will be accommodated at a permanent village approximately 3 kilometres from the plant site for the duration of their onsite rotation. Employees will work a roster that suits the needs of the organisation.

For start-up, there will be in excess of 200 employees and contractors on site. An operational readiness program will be implemented to ensure the operating organisation is ready to successfully manage the facility from start-up, and that the workforce is well-trained and capable.

To achieve this, an operating organisation (operations team) will be developed during the Project implementation phase. The organisation must be well-prepared, with all support systems, processes, protocols and operational documentation in place and all risks associated with the ramp-up and sustained operations reduced or mitigated via the detailed operational readiness program.

25 Interpretation and Conclusions

25.1 Exploration

Most of the exploration work conducted by and on behalf of RHM within the project area has been drilling, for purposes of exploration, resource definition and sterilization. However, all non-drilling forms of exploration have contributed directly to the targeting of additional mineralization, 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 south-western 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 unrecognized, 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 mineralization 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, Drake and Cortez 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, 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 mineralization. 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 × 50 m to a staggered 50 × 100 m.

Grade control drilling at Magellan and Cano has infilled the exploration drilling data to a 12.5 × 12.5 m and 16.7 × 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 mineralization 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 Intertek Genalysis Laboratories (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: 20 September 1991 – Accreditation No: 3244
- ALS date of accreditation: 22 December 2015 – Accreditation No: 825.

All sample preparation and analyses for the DFS diamond core and bulk sample testwork 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 project 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 Paroo Station 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 for the Mine includes the main Magellan Hill deposits and the outlying Pizarro and Drake satellite deposits, located approximately 8 km south and 11 km south-west respectively from the existing Paroo Station Mine infrastructure.

The Magellan Hill, Pizarro and Drake Mineral Resources have been reported in accordance with the JORC Code (2012). Further detail can be found in Optiro's reports J1782_RRHMPL_Dec2014_MRE.pdf (Mineral Resource Estimate), 170223_ParooStationMineralResourceStatement2016.pdf (Mineral Resource Statement), the FinOre 2005 Drake Mineral Resource Estimate.pdf (FinOre 2005 Drake Mineral Resource Estimate) and

Optiro's 20160129_Memo_DrakeJORC2012.pdf (JORC Code 2012 Table 1 and compliance report for Drake Mineral Resource). A new Mineral Resource tabulation is presented in Optiro's Paroo Station Lead Mineral Resource Reporting Cut-off Update.

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 as part of a 2016 Mineral Resource update. The Mineral Resource estimate was estimated re-tabulated after a change in cut-off grade by Optiro was adopted (from 2.1% Pb to 1.3% Pb) and is current as at February 2019.

For the Magellan Hill deposits and Pizarro, no additional exploration data have been incorporated into any of the Mineral Resource estimates.

The Drake Mineral Resource was originally estimated in 2005 and reported in accordance with the JORC Code (2004). As part of a 2016 Mineral Resource update, Optiro 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 Code (2012) reporting code. The Drake Mineral Resource was re-tabulated in January 2019 as part of the change in cut-off grade.

25.3 Mineral Reserve Estimate

The Paroo Station Lead 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 historical as well as more recent production information, including current cost, revenue and metallurgical recoveries generated as part of the DFS Update, 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 updated open pit optimization incorporating accepted product pricing and current project costs and operational parameters. The open pit optimization underpinned revised mine staging, mine designs and mine production scheduling.

The Mineral Reserve estimate was developed under the 2012 Edition of the JORC Code. The CIM recognizes the use of Foreign Codes, including the JORC Code.

Open pit optimization was used to identify the optimum economic pit shape based on the highest project cashflow. The pit optimization process seeks a solution to a complex 3D mathematical relationship involving the mineral resource model, geotechnical slope guidelines, product revenue, project constraints, modifying factors and costs.

The key inputs into the optimization process include:

- Product prices
- Mining costs
- Processing, realization and administration costs
- Process recoveries
- Pit slope angles
- Prepared model.

The mineral resource model was converted to a mining model by a process of regularization to account for dilution and ore losses. The diluted model has been used as the basis for optimization, pit evaluation and scheduling. Further preparation included adding cost, recovery, royalties and revenue drivers to individual blocks within the model.

An NPV discount rate of 8%, which is comparable with Australian projects of similar scale and size, has been applied.

NSR inputs and formulas required to calculate the economic value for each block were used in the optimization 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 optimization shells.

25.4 Mining

25.4.1 Geotechnical and Hydrogeological

An overall slope angle of 40° 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 Mine, Peter O'Bryan & 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.

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.

The final pits were designed with the Magellan Hill pits divided into nine stages and the Pizarro pit divided into three stages to assist with achieving schedule targets. The stages have their own ramp access while following the minimum mining width, allowing the stages to be mined independently.

25.4.3 Production Schedule

Strategic mine production schedules were developed using MineMax software to produce quarterly based schedules for the LOM. MineSched software was then applied using the quarterly schedules as guidance to generate a monthly schedule for the first five years of operation, followed by quarterly schedules thereafter.

The schedules were based on the following parameters:

- Diluted Magellan Hill and Pizarro models with Measured and Indicated Mineral Resource categories only
- Annual schedule with a start date of October 1, 2020
- Mill capacity of 2.185 Mtpa after an initial ramp
- Achieving production creep to support a maximum 80 ktpa of lead ingot production
- 5 m benches
- Use of existing stockpiles as ore feed for the commissioning and ramp-up of the Flotation Concentrator and Hydrometallurgical Facility.

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 final version of the METSIM model has been developed progressively over the course of the scoping study, DFS, Early Works Engineering and the pilot plant and demonstration plant testwork to include all of the operations of both the Flotation Concentrator and the Hydrometallurgical Facility. The model has evolved with the flowsheet as a range of unit operations have been considered and either included or removed from the flowsheet.

The original UBC testwork on which the hydrometallurgical flowsheet is based identified a requirement for three separate leaching circuits: one to leach lead carbonates (cerussite), a conversion leach to react lead sulfate (anglesite) 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 to contain operating costs.

Testwork identified an opportunity to simplify the UBC flowsheet by eliminating the MSA re-leach 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 METSIM model for the Hydrometallurgical Facility have been developed for the Paroo Station Project as follows:

- A Base Case model incorporating all the flowsheet elements required to operate the Hydrometallurgical Facility. The base case concentrate grade of 71.8% Pb was selected following investigation of concentrate treatment at grades of 55% Pb and 60% Pb. The concentrate feed input data to this 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 concentrate changes for a high and low anglesite 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.

Following the DFS the model was expanded to include the Flotation Concentrator and all the proposed upgrades and modifications to the concentrator flowsheet such that a model of the entire process plant was constructed. This allowed the interfaces between the Flotation Concentrator and the Hydrometallurgical Facility to be examined in detail.

The current Mass Balance is derived from the integrated model which also provides input into the Design Criteria in the DFS Update.

The mining reserve base used to define the initial METSIM model did not extend out to the projected LOM under the revised operating cost scenario and lower cut-off grades, so RHM undertook a drilling program to provide representative samples of each year of production for the proposed new LOM, 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 minimized slimes recovery to the flotation concentrate was in the order of 70% Pb, up from the 67%–68% Pb grade targeted for sale to a smelter. With the revised flotation regime, flotation recovery was increased relative to historical concentrator performance. The composition of these concentrates is described below.

The impact of mineralogical variability in the ROM ore has largely been eliminated as a result of the improved flotation performance which has largely eliminated gangue components from the concentrate. The key remaining variable in the concentrate is the relative proportions of cerussite and anglesite in the concentrate.

25.6 Environmental

Ministerial Statement 1083 was signed by the Minister for the Environment on September 25, 2018, following release of the EPA report and recommendations 1620. The EPA report considered the Rosslyn Hill Mining ‘Hydromet Facility & Mine Extension Proposal’ referral document of April 20, 2018. The Ministerial Statement provides environmental Conditions for the Project, which includes an increase the disturbance footprint by 400 ha, taking the total disturbance footprint to 980 ha within the development envelope. The approval 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 revised forecast LOM volumes. The approval includes the Hydrometallurgical Facility and the proposed new electricity generation plant at site.

Mining Proposal, Hydrometallurgical Facility was approved by Department of Mines Industry Regulation and Safety (DMIRS), on October 31, 2018, under the West Australian *Mining Act 1978*. The approval was granted to commence the development and operation of the project in accordance with revised Mining Tenement Conditions. The revised tenement conditions reflect the material provided within the RHM, Mining Proposal document describing the Hydrometallurgical Facility and associated mining and operational changes to the project. The approval allows for the onsite activities under the Mining Act, but does not permit construction activities to commence.

A Works Approval for the Hydrometallurgical Facility was approved by the Department of Water and Environmental Regulation (DWER), on November 30, 2018, under Part V of the *EP Act 1986*. The approval was granted to allow the construction of the Hydrometallurgical Facility, subject to conditions. RHM currently holds a Prescribed Premises License L8493/2010/2, permitting the control of emissions and discharges to the environment, and the monitoring and reporting of them. The Works Approval specifies emission levels such that during testing and commissioning of the Hydrometallurgical Facility, the proposed emissions described in the Works Approval document, are confirmed to allow the issuing of a Prescribed Premises License Amendment, with nominated emission limits.

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

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

- The capital cost estimate to build the proposed Hydrometallurgical Facility, make modifications to the existing Flotation Concentrator and associated infrastructure is US$183.7M (including Owner's costs of US$20.8M, contingency of US$14.9M and growth allowances of US$6.6M).
- The average operating cost to produce 99.99% Pb ingot is US$1,276.28/t (including overhead and sustaining capital over the 17-year LOM).
- The developed flowsheet and recoveries for the operation to produce up to 80,000 tpa of 99.99% quality lead ingot.
- A mineable JORC Code (2012) Mineral Reserve estimate of 36.3 Mt at 3.7% Pb grade for a 17-year LOM.
- Concentrate grades of the order of 72% Pb were achieved over a range of head grades from 3% Pb to 11% Pb. An average recovery of 83% Pb was achieved at a 4% Pb head grade.
- Impurity elements in the concentrate were significantly reduced, resulting in lower operating costs for the Hydrometallurgical Facility.
- Lead extraction in MSA leach average approximately 90% across all lead mineralogy based on a single pass through the leach circuit. The lead extraction from the MSA leach residue average 98%, resulting in an overall extraction of 81.3%.
- Lead cathode was produced at current densities between 300 A/m^2 and 350 A/m^2, which equates to a cathode plating rate of 70,000–80,000 tpa. Cathode quality exceeds 99.99% Pb.

Table 77: Financial returns

Description	Units	Estimate	Comments
Total cost to first production	US$ million	184	To start of operations
Payback Period	years	4.00	From start of operations
Internal Rate of Return	%pa	24.6	From start of construction
After-tax project cashflow			
Project Revenue	US$ million	2,584	From start of operations
- Less all-in sustaining costs	US$ million	-1,487	From start of operations
Cashflow before Tax	US$ million	1,096	From start of operations
- Less Income Tax	US$ million	-253	From start of operations
Cashflow after Tax	US$ million	843	From start of operations
Present Value			
- GPV (8.25% real discount rate)[1]	US$ million	430	From start of construction
- NPV (8.25% real discount rate)[2]	US$ million	257	From start of construction

Notes:
1 – GPV = gross present value = present value of cashflow after tax.
2 – NPV = net present value = present value of total cost to production.

Revenue assumptions:

- Wood Mackenzie price curve with long term average price of US$2,350/t
- Lead premia based on Fastmarkets Metal Bulletin pricing for 99.97% Pb purity adjusted for 99.99% to Southeast Asia
- Ocean freight netback based on RHM quotes.

25.8 Foreseeable Impacts of Risks

The Paroo Station 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 cost exposure to third parties processing the concentrate offshore.

The mine plan has been updated and revised to reflect the operation of the Hydrometallurgical Facility which presents a more robust project that demonstrates profitability at both the current spot lead prices and medium-term conservative price forecast of US$2,350/t.

26 Recommendations

The main points concerning ongoing work and recommendations concerning future work include:

- The progression of the DFS Update design should be progressed through a selected EPCM (Engineer Procure Construct Manage) contractor. The program will focus on planning and strategic activities, including equipment and civil works, power station tenders and the progression of engineering to allow detailed design to commence and the establishment of construction systems and practices.
- Regulatory approvals and Native Title agreements will be required to allow for the development of the Pizarro deposit. Currently, the forecast access to this satellite deposit is well into the 17-year LOM.
- Exchange of information with project finance institutions seeking debt finance terms and equity partners should be progressed during 2019, with targeting a financial close for the Project in September 2019.
- The water table will be intersected when pits are mined to the final design. Prior to commencing any mining below the water table, a groundwater investigation should be performed to identify the effects of the groundwater resource on the hydrological regime, effects on the potential groundwater dependent ecosystems in 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 regulatory authorities for permission to mine below the water table.
- Prior to executing the final design, a hydrological review should be performed to ensure there is no adverse impact on the stability of the pit walls as a consequence of mining below the water table.

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28 Glossary

Abbreviation	Definition
4WD	4 wheel drive
AER	Annual Environment Report
ALS	Australian Laboratory Services
A$	Australian dollar
ANFO	ammonium nitrate fuel oil
APU	Acid Purification Unit
ARSM	Associate of the Royal School of Mines
AusIMM	Australasian Institute of Mining and Metallurgy
bcm	bank cubic meter
DOM	Bureau of Meteorology
CAR	Compliance Assessment Report
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	Desulfurization 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
EGL	Effective grinding length
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	gigaliters
GPS	Global positioning system
GPX	GPX Surveys Pty Ltd
GSWA	Geological Survey of Western Australia
GWE	Great Western Exploration Limited
ha	hectares
HRB	Heat recovery boiler
hr	hours

Abbreviation	Definition
Iluka	Iluka Resources Limited
InCoR	InCoR Energy Metal Limited
IWL	Integrated Waste Landform
JORC	Joint Ore Reserves Committee
$kg/m^2/h$	kilograms per square meter per hour
kL	kiloliters
km	kilometers
KNA	Kriging Neighborhood Analysis
kt	kilotonnes
kWh/t	kilowatt hours per tonne
L	liters
$L/min/m^2$	Liters per minute per square meter
Lead*FX*	Lead*FX* Inc.
LME	London Metals Exchange
LOM	life of mine
m	meters
M	million
Magellan Hill	Magellan (including Gama), Cano and Pinzon deposits
Magellan Metals	Magellan Metals Pty Ltd
mAHD	meters Australian height datum
mg/L	milligrams per liter
mg/m3	milligrams per cubic meter
mE	meters east
mm	millimeters
mN MOC	meters north Materials of Construction
MRE	Mineral Resource estimate
MS	Microsoft
MSA	methane sulfonic acid
Mt	million tonnes
Mtpa	million tonnes per annum
MVT	Mississippi Valley type
MW	megawatts
NaHS	sodium hydrosulfide
NE	north-east
NPV	net present value
NSR	net smelter return
NW	north-west
OEPA	Office of the Environmental Protection Authority
ORP	oxidation reduction potential
Paroo Station	Paroo Station Mine
Pb	lead

Abbreviation	Definition
PbS	galena (lead sulfide)
Polymetals POC	Polymetals Pty Ltd Proof of concept
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
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	tonnes
TDEM	time-domain electromagnetic
t/h	tonnes per hour
$t/m^2/h$	tonnes per square meter per hour
the Mine deposits	Magellan (now including Gama), Cano and Pinzon and the outlying Pizarro and Drake satellite deposits
TJ/d	terrajoules per day
TMM	total material movement
tpa	tonnes per annum
TSF	tailings storage facility
TSX	Toronto Stock Exchange
UBC	University of British Columbia
US$	United States dollar
VMS	volcanogenic massive sulfide
XRD	X-ray diffraction
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

Appendix A: Certificates of Qualified Persons and Consents of Qualified Persons

srk consulting

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Certificate of Qualified Person

To accompany the report entitled, *NI 43-101 Technical Report on the Paroo Station Lead Carbonate Mine, Wiluna, Western Australia*, with an effective date of February 15, 2019, 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 6, 16, 19 to 28, of the Technical Report.

f) I am independent of the issuer 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 previously been a QP responsible for Sections 1 to 3, 16, 18 to 19, and 21 to 28 and the preparation of the reports titled *NI 43-101 Technical Report on the Paroo Station Lead Carbonate Mine, Wiluna, Western Australia,* with Effective Dates December 31, 2014 and February 31, 2018.

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 5, 2019.

Scott McEwing, BEng(Mining), FAusIMM CP(Min)
Principal Consultant

CHARTERED PROFESSIONAL
AusIMM
AUSTRALASIAN INSTITUTE OF MINING & METALLURGY
• MINING •
SCOTT McEWING

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 15, 2019, prepared for LeadFX Inc. (the Technical Report)

I, Dr David Dreisinger, Ph.D., P.Eng., F.C.A.E., F.C.I.M., of LeadFX Inc., Parmelia House, Suite 2, Level 5, 191 St Georges Terrace, Perth WA 6000, Australia do hereby certify that:

1. I am a Director with the firm of LeadFX Inc. of Parmelia House, Suite 2, Level 5, 191 St Georges Terrace, Perth WA 6000, Australia;

2. I am a graduate of Queen's University of Kingston, Canada with a B.Sc. Metallurgical Engineering (1980) and a Ph.D. Metallurgical Engineering (1984). I have practised my profession continuously since 1980;

3. I am a Fellow of the Canadian Academy of Engineering and of the Canadian Institute of Mining, Metallurgy and Petroleum and am a member of good standing of the Association of Professional Engineers and Geoscientists of British Columbia (Registration Number 15803);

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 on January 16, 2017;

6. I have prior involvement with the property that is the subject of the Technical Report in my capacity as President and CEO of InCoR Energy Metals Ltd, a major shareholder of LeadFX Inc. and as a Director of LeadFX Inc.;

7. I am not 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 13, 17 and 18 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 Vancouver, Canada on April 5, 2019

Dave Dreisinger

Dr. David Dreisinger

Director

PROFESSIONAL
PROVINCE OF
D. B. DREISINGER
BRITISH COLUMBIA
ENGINEER

LeadFX Inc.
Rosslyn Hill Mining Pty Ltd (ABN: 68 075 523 661)
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AMC consultants
mine smarter

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 15 February 2019, 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:

- I am an employee with the firm of AMC Consultants Pty Ltd of Level 1, 1100 Hay Street, West Perth, WA 6005.
- I am a graduate of the University of Melbourne with a BE Eng (Mining) in 1975. I have practised my profession continuously since 1975.
- I am a FAusIMM (CP).
- 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.
- I have not inspected the property.
- I have no prior involvement with the property that is the subject of the Technical Report.
- I am independent of the issuer as defined in Section 1.5 of NI 43-101.
- I do not have any securities in LeadFX Inc. or its subsidiaries.
- I am responsible for the preparation of items 15 of this Technical Report.
- 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.
- 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.
- 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 5 April 2019

The signatory has given permission to use their signature in this AMC document

Lawrie Gillett

Level 1, 16 Ord Street
West Perth WA 6005
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Australia
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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 15, 2019, 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 was the QP for the previous NI43-101 Technical Report with the Effective date of February 28, 2018;

8. I am independent of the issuer as defined in Section 1.5 of NI 43-101;

9. I do not have any securities in LeadFX Inc. or its subsidiaries;

10. I am responsible for the preparation of items 7 to 12 and 14 of this Technical Report;

11. 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;

12. 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

13. 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 5, 2019

K. Cervoj

Kahan Mit-hat Cervoj

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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, *NI 43-101 Technical Report on the Paroo Station Lead Mine, Wiluna, Western Australia*, with an effective date of February 15, 2019 (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 5th day of April 2019.

Scott McEwing

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, Dr David Dreisinger 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 15, 2019 (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 5, 2019

Dave Dreisinger

Dr David Dreisinger

LeadFX Inc.
Rosslyn Hill Mining Pty Ltd (ABN: 68 075 523 661)
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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 and 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 15 February 2019 (the "Technical Report") by LeadFX Inc.

I further consent to the use of my name and to the Company making reference to and summarizing 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 5 April 2019

The signatory has given permission to use their signature in this AMC document

Lawrie Gillett

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West Perth WA 6005
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West Perth WA 6872
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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, 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 15, 2019 (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.

Signed at Perth on April 5, 2019

K. Cervoj

Kahan Mit-hat Cervoj