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Laramide Resources Ltd.

Westmoreland Uranium Project

Preliminary Economic Assessment Update

National Instrument 43-101 Technical Report

3325-GREP-001

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LARAMIDE RESOURCES LTD.

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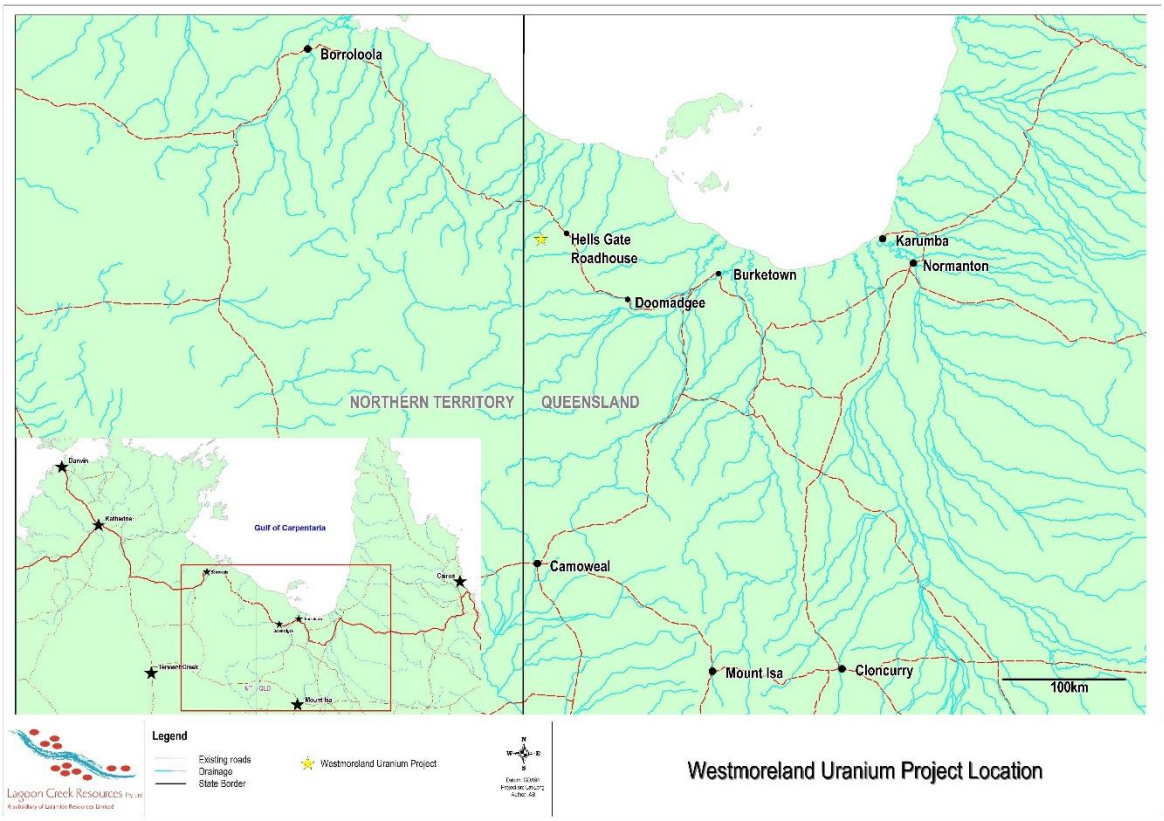
1.0 SUMMARY

1.1 Introduction

The Westmoreland Uranium Project (the 'Project') is located within a group of mineral tenements controlled by Laramide Resources Ltd (LAM) that extend for approximately 30 kilometres (km) east-west and 22 km north-south, adjacent to the Queensland (QLD)-Northern Territory (NT) border and within the North West Queensland Minerals Province. The Project is located in a region known as the Gulf Country, which includes the southern shores of the Gulf of Carpentaria and the country around with the many rivers that flow into the Gulf. It is the largest tropical savannah region in Australia, with an area of 425,000 square kilometre (km^2).

Figure 1.1.1 Project Location



The Project is readily accessed from the Savannah Highway, a road running from Burketown in QLD to Borroloola in the NT (Figure 1.1.1). The highway is a two-lane sealed road in QLD from Burketown to the Hells Gate Roadhouse, which is approximately 50 km from the NT border, thereafter the road remains unsealed. A network of gravel roads and pastoral tracks provides access to the property from Hells Gate Roadhouse, which is approximately 15 km from the closest significant mineral deposit (Long Pocket (LP)) and approximately 30 km from the Redtree (RT) Deposit. During periods of intense rainfall in the summer wet season, some road creek crossings may be impassable and this may include sections of the Savannah Highway

The Westmoreland region was probably first prospected in the 1890s, after the discovery in 1887 of silver-lead deposits at Lawn Hill, 100 km south. Copper was discovered in 1911 at Settlement Creek and at the nearby Redbank lode in the NT in 1916. In 1912, the Packsaddle and Bauhinia copper lodes were discovered near Wollogorang homestead. Pitchblende has been mined in the Peters Creek Volcanics, which overlie the Westmoreland Conglomerate, 20 to 30 km west of RT.

The mineral resource estimate (MRE) has been classified under the Canadian Institute of Mining, Metallurgy and Petroleum's (CIM) code of ore classification, and has now been restated to comply with the Joint Ore Reserves Committee (JORC) Code 2012 (refer to notes and other details in Section 14).

Table 1.1.1 Westmoreland Mineral Resource Estimate by Deposit

Deposit	Tonnes t	Density g/m³	U_3O_8 ppm	U_3O_8 Mlb
Indicated				
Redtree	14,000,000	2.5	880	27
Huarabagoo	2,500,000	2.6	890	4.9
Junnagunna	10,000,000	2.5	640	15
Long Pocket	1,300,000	2.5	420	1.2
Total Indicated	**27,800,000**	**2.5**	**770**	**48.1**
Inferred				
Redtree	3,000,000	2.5	800	5.2
Huarabagoo	3,100,000	2.6	870	6.0
Junnagunna	3,000,000	2.5	620	4.2
Long Pocket	2,700,000	2.5	380	2.3
Total Inferred	**11,800,000**	**2.5**	**680**	**17.7**

1.2 Ownership

LAM owns 100% of the Westmoreland Uranium Project through its wholly owned Australian subsidiary, Tackle Resources Pty Ltd (Tackle).

A Schedule of Tenements has been provided by LAM. The ownership and status of the tenements has not been independently verified, apart from a search of the Queensland Interactive Resource and Tenement Map (IRTM) online database. A schedule of relevant Exploration Permits for Minerals (EPMs) and Mineral Development Licences (MDLs) is presented in Table 1.2.1 below.

Table 1.2.1 Laramide Tenements in Queensland as of August 2026

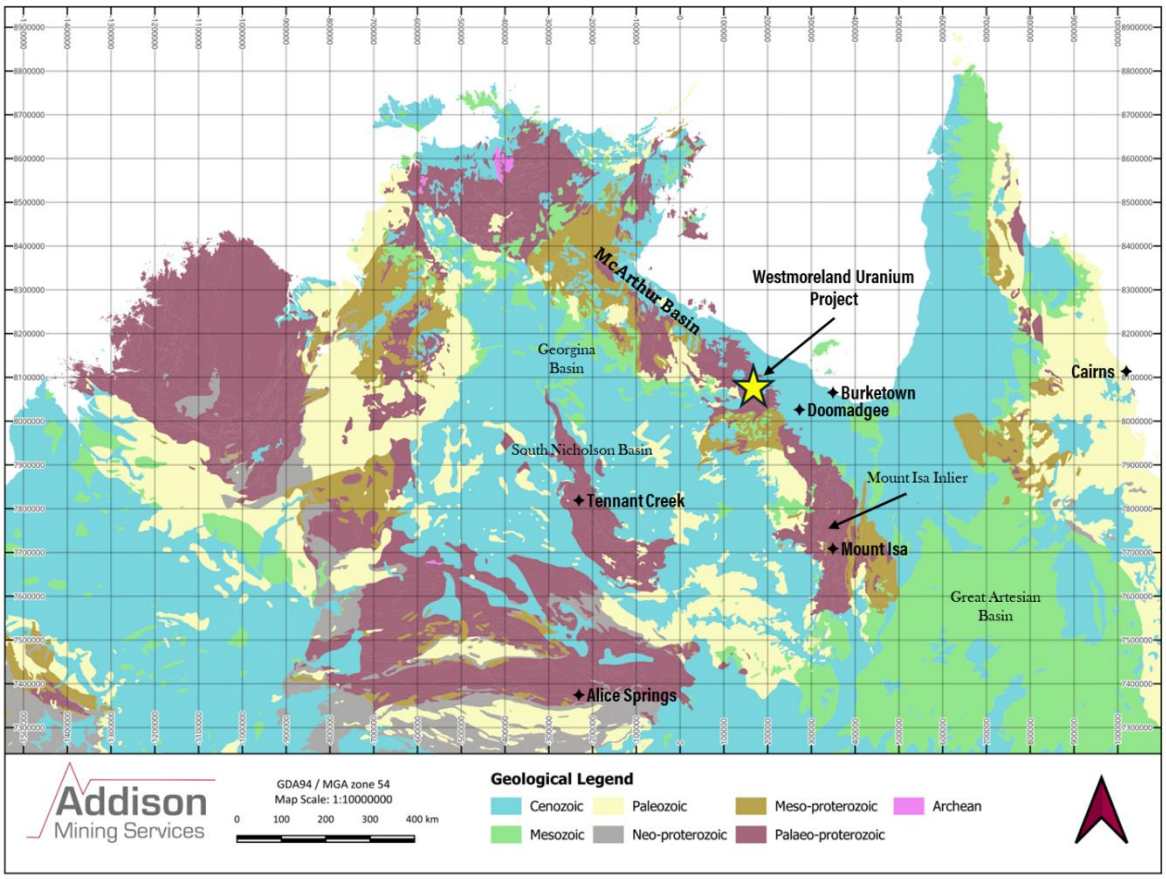
Lease	Grant Date	Expiry	Area km²	No. Sub-Blocks	Licence Holder	Approx. Centre GDA94 UTM Zone 54
EPM14558	26.07.2005	24.07.2030	186.00	58	Tackle Resources Pty Ltd	199398 mE, 8063064 mN
EPM14672	26.07.2005	24.07.2030	84.90	26	Tackle Resources Pty Ltd	186039 mE, 8060721 mN
EPM28807	04.12.2024	03.12.2029	326.96	100	Tackle Resources Pty Ltd	212576 mE, 8067059 mN
EPM28834	Under Application		326.75	100	Tackle Resources Pty Ltd	193960 mE, 8048406 mN
MDL2026	01.07.2025	30.07.2030	111.14	N/A	Tackle Resources Pty Ltd	194849 mE, 8063392 mN

The Project is centred about 380 km north-northwest (NNW) of Mt Isa City in northwest Queensland. All Mineral Resources for the Project which are described in this report are located within EPM14558, which is considered the most material EPM to the Project.

1.3 Geology and Mineralization

The Project is located on the southeastern (SE) margin of the southern McArthur River Basin, a thick sequence of Palaeo-Mesoproterozoic sedimentary and volcanic rocks deposited on the North Australian Craton.

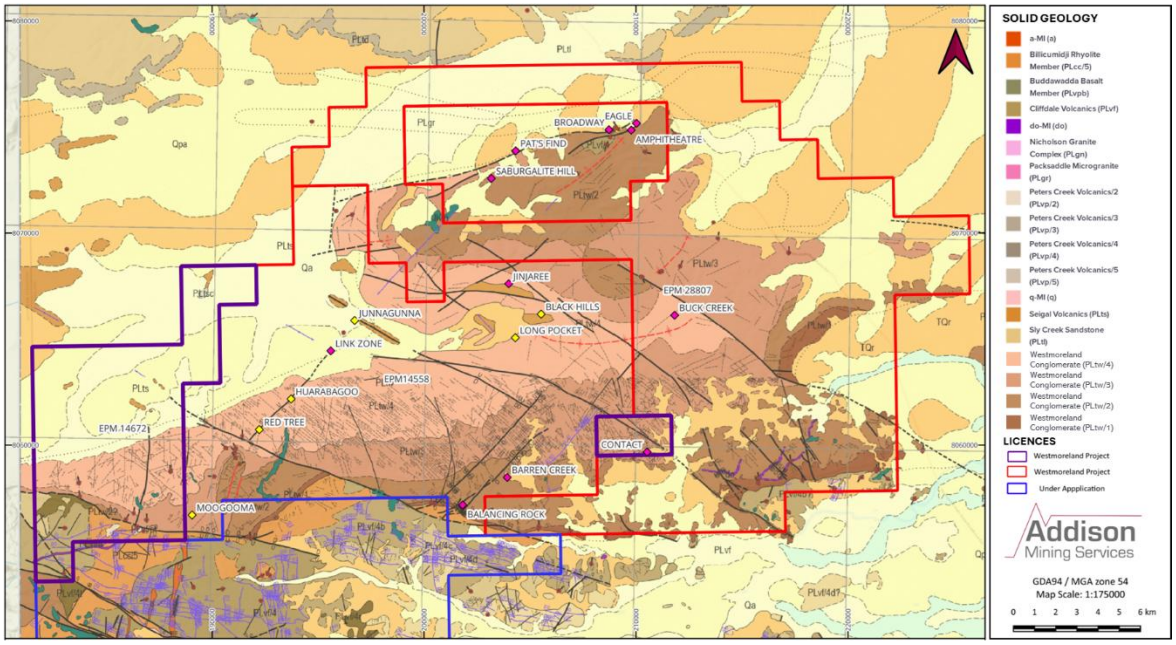
Figure 1.3.1 Geological Setting of Northern Australia



The Westmoreland–Pandanus Creek uranium field has four main types of uranium (U) occurrences, including those at contacts between volcanic units and the Westmoreland Conglomerate (Type A), near dyke contacts with the conglomerate (Type B), within the Cliffdale Volcanics beneath the conglomerate unconformity (Type C), and in fractures within the Seigal Volcanics above the conglomerate (Type D). These sandstone-hosted 'Westmoreland-style' deposits are thought to have formed from basinal brines circulating through the Westmoreland Conglomerate, leaching uranium and precipitating it in favorable locations, possibly through reduction by diagenetic chlorite, with movement along unconformities and into shear zones. Another style, 'Eva-style', is associated with shear zone mineralization in the altered Cliffdale Volcanics.

Locally, the Westmoreland tenements are situated over the outcropping Westmoreland Conglomerate (Tawallah Group) where the southern McArthur Basin overlies the Cliffdale Volcanics (Murphy Inlier). The licence EPM14558 contains the major deposits Redtree (RT), Junnagunna (JG), Huarabagoo (HB) and Long Pocket (LP), collectively termed the Redtree Group. All of these deposits are hosted within the shallow-dipping Westmoreland Conglomerate, a thick sequence of fining-upward fluvial units whose basal conglomerate is suggested as a potential source for uranium mineralization, with NE-trending (NE) dolerite-filled fractures crosscutting it. Conformably overlying the conglomerate are the Seigal Volcanics, and NE-trending dolerite dykes, potentially feeders to the volcanics, intrude both units.

Figure 1.3.2 Geology of the Westmoreland Project



Regionally, there are 13 other known mineralized zones with no resources at the property of lower materiality but are worthy targets for future exploration. Situated in the EPM14558, the most significant of these are Moogooma, Black Hills, Uranium Valley, Amphitheatre and Eagle 3, which have undergone various stages of exploration and suggest the mineralization styles are similar to that of the primary Redtree Group. Further exploration is warranted on these regional prospects.

1.4 Mining

The production schedule is planned at 2.9 million tonnes per annum (Mtpa) of mill feed, with constant annual material movement and an aim to balance ore and waste and mining fleet within total material movement peaking at 16.7 Mtpa in Year 3. Mining is undertaken over 11 years (including first year pre-strip), supplying 11 years of mill feed with a total ore production of 28.3 million tonnes (Mt) at an average grade of 810 parts per million (ppm). Deswik Pseudoflow (Deswik) was used to define optimal pits for the four Westmoreland uranium deposits based on the mineral resource model.

There are four pit shells in four distinct areas, as shown in Figure 1.4.1:

- North – Junnagunna (JG).
- Central – Huarabagoo (HB).
- South – Redtree.(RT).
- Northeast – Long Pocket (LP).

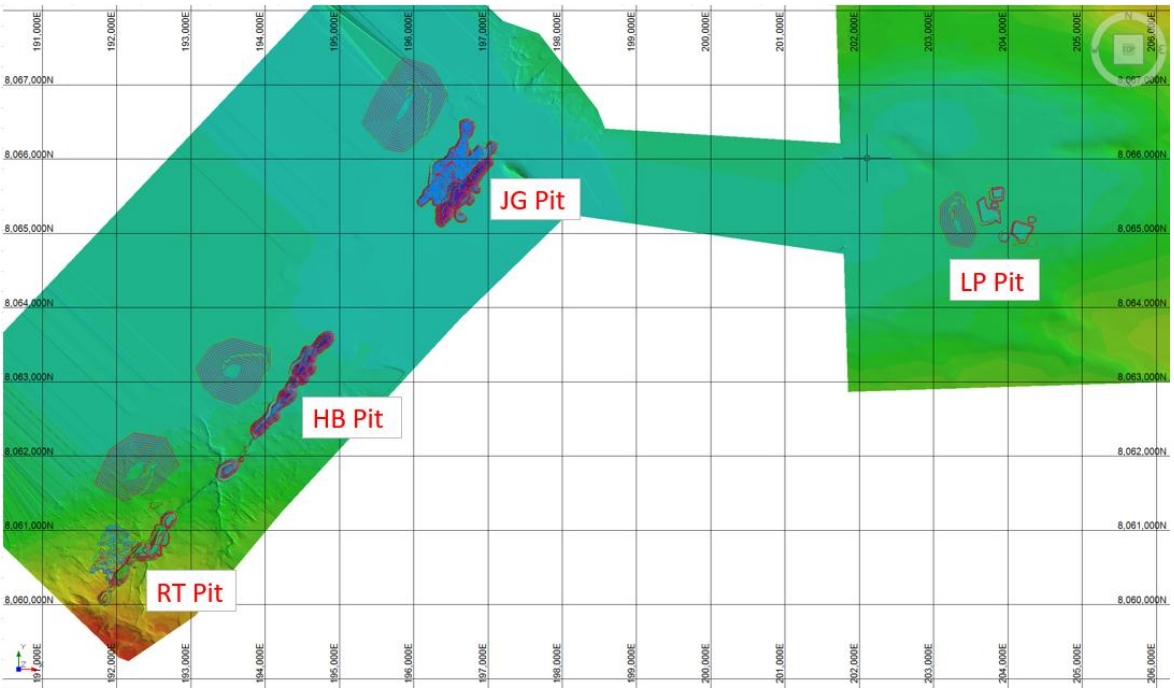
The mining methodology is based on conventional methods and is summarized below:

- Open-pit mining using three excavators loading off-highway haul trucks - two EX2000 excavators and one EX1200 excavator, all loading CAT777 trucks. Four trucks were allocated to each excavator over the life of mine plan (LOMP). The excavators were supported by an appropriate ancillary fleet.
- Conventional drill and blast (D&B) with truck and shovel (T&S) operation mining 5 metre (m) benches with 2.5 m flitches.
- Sufficient working areas to allow for simultaneous D&B and T&S operation. Flexibility in the scheduling is required.

The mining methodology is based on the following material movement schedule:

- A total of 147 Mt total material moved (TMM) will be moved over 11 years of mining, with 118 Mt of waste and 28.3 Mt of production target being produced grading 810 ppm. The life of mine (LOM) strip ratio of the four pits is 4.2 t:t. The strip ratio varies substantially between the pits with RT pit at 1.7 t:t, LP pit at 4.5 t:t, JG pit at 5.3 t:t, and HB pit at 10 t:t.
- Mill feed: 2.9 Mtpa achieved in the second year onwards for the full mine life. The mill throughput reduces to 1 Mtpa in the final year of production.
- Mining commences in RT pit to establish an initial tailings emplacement area.

Figure 1.4.1 Westmoreland Deposits and Pit Shells



The annual production statistics by pit are presented graphically in Figures 1.4.2 to 1.4.4 below.

Figure 1.4.2 Total Tonnes Mined by Pit

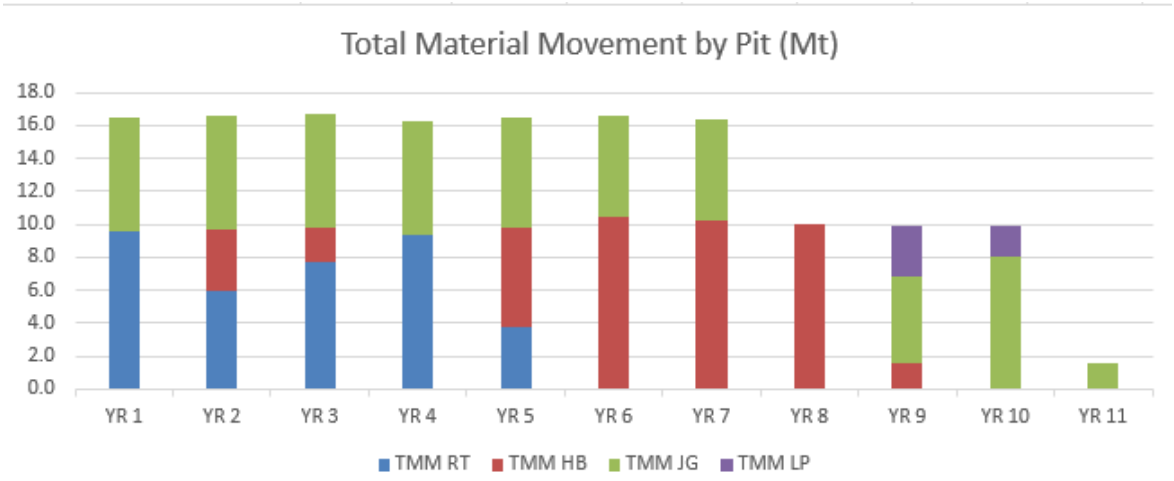


Figure 1.4.3 Ore Tonnes and Grade Milled by Year

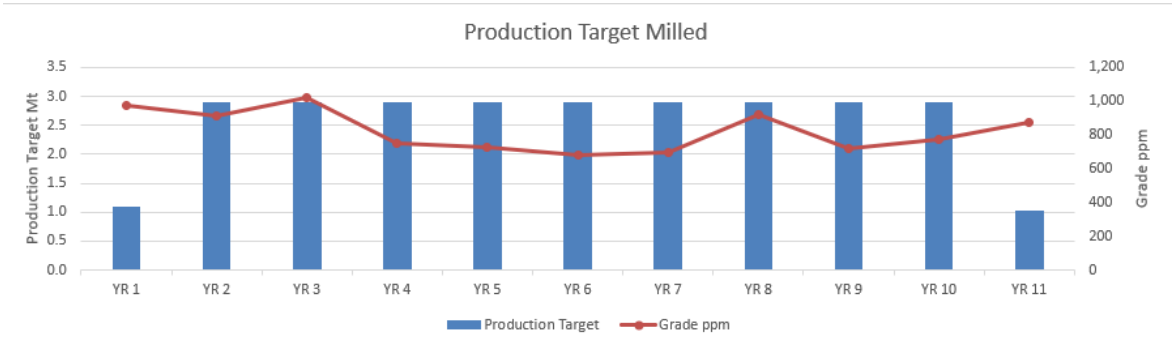
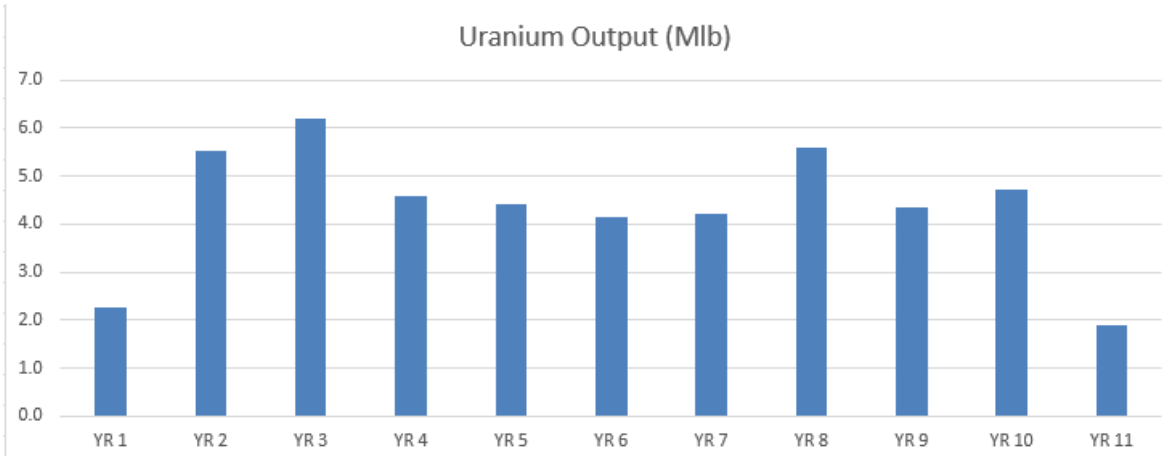


Figure 1.4.4 Uranium Production by Year



1.5 Metallurgical Testing

Metallurgical testwork programs and evaluations on the Westmoreland deposit have been conducted by several laboratories over the period since 1989. These programs included work completed by AMDEL (1989), ANSTO (1992, 1993, 1994, 1995 and 2011), JKTech (1993), and SGS (2008).

Key findings of the metallurgical testwork programs include:

- The Westmoreland material generally acid leaches very well with modest acid consumption and high uranium extractions.
- Co-leaching of gangue elements is not considered to present any problems for downstream processing.
- The Jack Lens material was the only exception and extraction was improved by adding ferric iron to assist oxidation of tetravalent uranium in the material. Further optimization of leach conditions is expected to improve performance on Jack Lens material.
- The grind size distribution required is relatively coarse, which favors milling power consumption and filtration performance.
- The leach kinetics are reasonably fast.
- Recovery of the uranium from the leached slurry can be undertaken by several methods including continuous ion exchange.
- Precipitation of the uranium as a concentrate to be sold to the market can be of a good quality and can be treated by any of the three main converters that will be treating the material.
- Pulp settling rate is reasonable with a high solids underflow density and a relatively clear overflow pregnant leach solution (PLS).
- The use of ion exchange (IX) technology has been tested and selected. The use of solvent exchange (SX) technology has also been tested and would be a technically viable treatment option.

1.6 Recovery Methods

The process flowsheet selected as the basis for the treatment of Westmoreland ore has been chosen after evaluation of the available geology, mineralogy and metallurgical testwork results. The objective has been to select a robust treatment method that provides a sound technical basis and that takes advantage of recent trends in uranium milling technology. A simplified overall flowsheet is shown in Figure 1.6.1

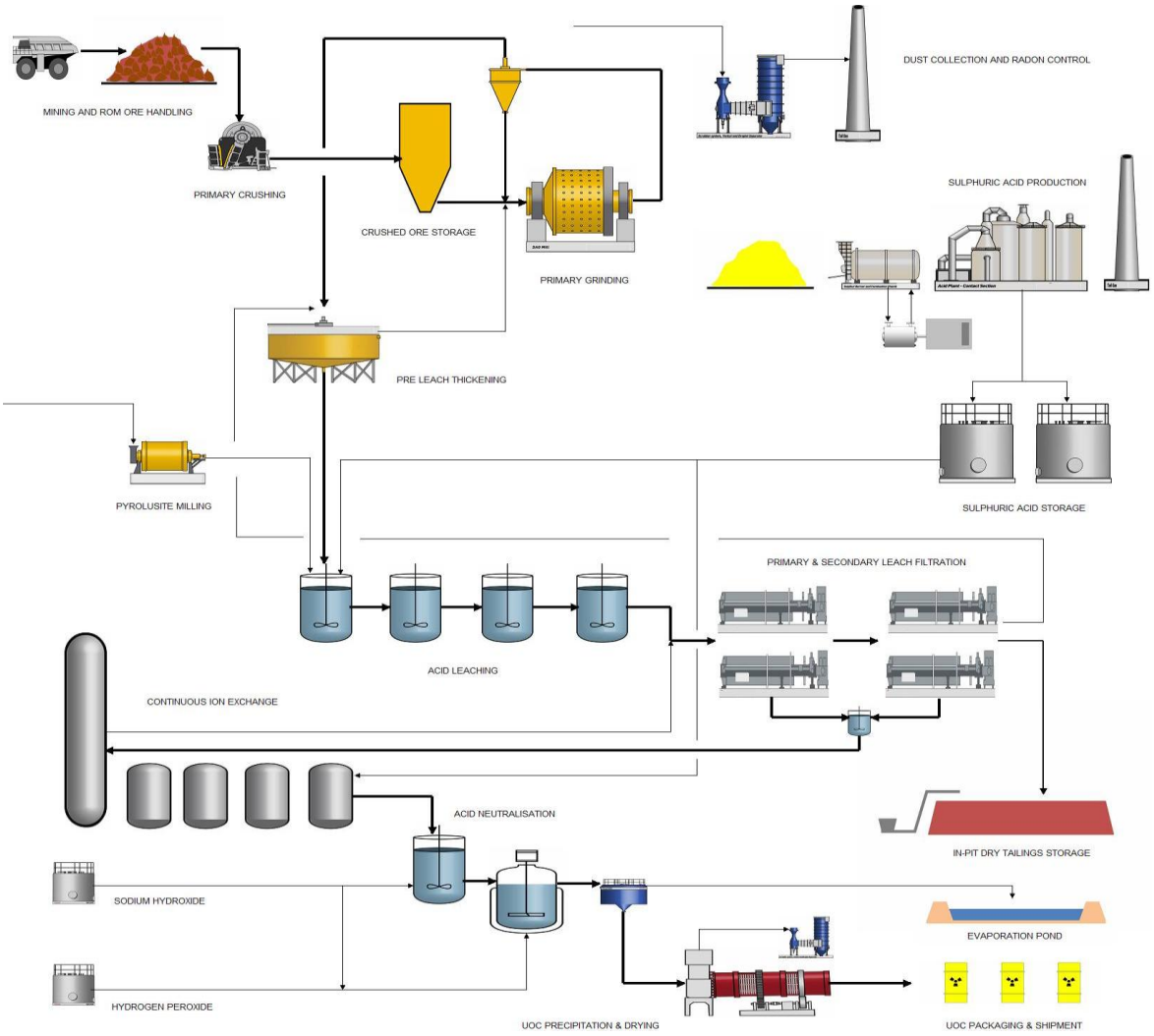
The overall process flowsheet includes feeding a blended ore mix to a single stage jaw crusher and SAG mill with pebble crusher (SS SAG/C) comminution circuit, which is in closed circuit with cyclones to achieve the final product size. The cyclone overflow stream will flow by gravity to a linear trash screen and then a pre-leach thickener. Barren solution leaving the continuous ion exchange circuit is recycled as wash to the leach residue filters and then, via the filter washgate, returns to the pre-leach thickener where most of the contained acidity is neutralized by the gangue in the ore. Pre-leach thickener overflow, which is very mildly acidic, is used as dilution water in the milling circuit. The thickened slurry is pumped to the leach circuit where it is mixed with concentrated sulfuric acid for uranium leaching. Manganese dioxide (as high quality milled pyrolusite) is added to the leach circuit to control the redox potential.

The uranium leach step is carried out at a temperature of 40 degrees Celsius (°C), and this temperature is provided partly from the heat of dilution of sulfuric acid and partly by live steam addition to the leach tanks. The leach tailings stream is filtered and washed to recover the pregnant liquor before being conveyed as a wet cake to the tailings storage facility. Pregnant leach solution flows to a continuous IX circuit where the uranium and some iron, silicon and molybdenum are adsorbed onto the resin. Impurity elements of iron, aluminium, calcium, manganese, arsenic, etc., are entrained with the resin in the interstitial fluid. The majority of the interstitial fluid is displaced prior to elution.

The uranium, together with the minor elements as impurities is eluted from the resin with concentrated sulfuric acid to produce a concentrated eluate solution containing approximately 7.5 grams per litre (g/L) uranium. The impurities present in the eluate solution are then removed, firstly by partial acid neutralization with sodium hydroxide to produce a precipitate containing uranium, iron, aluminium, and arsenic. This precipitate is recycled to the leach step to recover the uranium. The partially neutralized solution is then treated with sodium hydroxide and hydrogen peroxide to precipitate crude uranium oxide concentrate (UOC) with associated impurity levels acceptable for sale to a convertor. The barren solution produced, containing the remaining impurities including sodium and sulfate is discharged to an evaporation dam for disposal.

The UOC is then washed to remove entrained mother liquor, dried, and then packaged into clean, thick wall 200 litre (L) drums, and prepared for shipment.

Figure 1.6.1 Simplified Overall Treatment Flowsheet



1.7 Project Infrastructure

Infrastructure for the Project includes water supply, electric power supply, tailings storage, access roads, sewage treatment, an accommodation village, an airport and administration, process, and mine infrastructure buildings. It is proposed to construct a tailings storage facility (TSF) by emplacement of filtered tailings in RT pit. This strategy allows the pit to be backfilled sequentially during mining operations, effectively eliminating the final mining void and minimizing the project's overall disturbed environmental footprint.

A conceptual strategy has been adopted to eliminate the operational complexity and cost of re-handling the material. This optimized approach avoids the need for a temporary paddock facility by placing the filtered tailings immediately outside the RT pit footprint during the initial stages, then extending placement into the RT pit as soon as it becomes available. At closure, this produces a single, continuous landform without requiring any material to be rewon or relocated. The resulting plan is a conceptual design of integrated ex-pit / in-pit tailings filtered stack facility.

1.7.1 Water Supply

A project water balance indicates an average water demand for the mine and treatment facility of 211 cubic metres per hour (m^3/h). A further 2 m^3/h of potable quality water will be required for the accommodation village. An assessment of water supply options was undertaken by Groundwater Science Pty Ltd (GWS), and they noted the following:

- Estimated in-pit rainfall runoff is significant and may exceed water demand in some months.
- Estimated groundwater seepage to the mine pits is negligible.
- Sufficient water supply from local borefields is likely to be available.

Calculated average annual runoff volume based on the total estimated pit area ranges from 450,000 cubic metres (m^3) to 830,000 m^3. In-pit runoff will be pumped from the pit for use in the process plant. The water will exhibit low salinity and high suspended solids. Runoff may exceed demand for two to four months (December through March) per year, which can be managed by:

- Storage dams at surface to contain excess water.
- Mine pit scheduling to provide a lower sump / bench that can be inundated for one to two months per year.

Two borefield options for water supply exist:

- From the Great Artesian Aquifer 45 km east of the project site.
- Near mine aquifers from Westmoreland conglomerate sandstone located 20 km east of the project site.

1.7.2 Power Supply

The Project maximum continuous power draw has been estimated at 16.1 megawatts (MW) with an average continuous power draw of 11.4 MW. A power supply options assessment was undertaken looking at three options for site power supply:

- Standalone diesel generation.
- A hybrid diesel plant with solar photovoltaic (PV) array and battery energy storage system (BESS).
- A 155 km extension of the 220 kilovolt (kV) transmission line that runs from Mount Isa to the Century zinc mine as part of the Mount Isa isolated network.

The hybrid solar / BESS option at 40 percent (%) renewable fraction (RF) achieves the lowest levelized cost of electricity and has been selected for the Project.

1.7.3 Access Roads, Accommodation Camp, Airstrip, Buildings, and Sewage

Allowance has been made for an access road from the Savannah Way to the mine site and accommodation camp. The first half of the road follows an existing track, and the remainder of the roads are new. The existing part of the road travels along the Lagoon Creek and crosses the creek twice. Including the access road extensions to the mine site and accommodation camp, the total length of is approximately 30 km.

Allowance has been made for a 250 man camp to provide accommodation for personnel during construction and operations. An all-weather gravel airstrip will be provided for transport of personnel and emergency medivac purposes.

1.7.4 Tailings Storage Facility

Recent reviews of tailings management practices have identified the disposal of tailings as a filtered (dry) material and elimination of the supernatant pond as being the Best Available Technology (BAT). This technology presents a significantly lower risk of failure compared to conventional tailings disposal, reduction in seepage potential and additionally leads to significant water savings for the Project. Knight Piésold (KP) examined options for disposal of filtered tailings at the Project. Following discussions with LAM and their mining Consultant (Mining Associates), it was determined that it will be feasible to dispose of the tailings as a dry stack, with the stack being constructed within the RT pit. This allows for backfilling the pit sequentially as the pit is being mined, eliminating the final void and reducing the disturbed footprint of operations at the site.

The LOM landform has been developed by Mining Associates based on the mine design. This landform can be constructed in parallel with mining operation, and provides sufficient capacity for storage of tailings and a portion of waste rock to be generated as part of mining operations of the RT pit and other pits at the Project. The landform will be constructed from the base up over the LOM, which will allow for progressive rehabilitation of the facility.

1.8 Operating and Capital Cost Estimates

Operating cost for the process treatment plant and infrastructure has been estimated by Lycopodium (APAC) Pty Ltd (Lycopodium; formerly Lycopodium Minerals Pty Ltd) and is summarized in Table 1.8.1.

Table 1.8.1 Westmoreland Process Plant Operating Cost Summary

Cost Centre	A$ /y	A$ /t ore	US$ /lb U_3O_8
Processing Labour	12,676,636	4.37	1.85
Power	21,039,969	7.26	3.08
Consumables	81,015,976	27.94	11.85
Maintenance Materials	7,168,330	2.47	1.05
Laboratory	1,453,030	0.50	0.21
General and Administration	20,419,091	7.04	2.99
Total	**143,773,032**	**49.58**	**21.03**

Operating cost for mining, based on a Contract Mining strategy, has been estimated by Mining Associates and is summarized in Table 1.8.2.

Table 1.8.2 Mining Operating Cost Summary

Mining Unit Cost	Units	Value
Total LOMP Mining Cost	A$...M	782,6
Total Material Movement	Mt TMM	146.6
Total Mining Unit Cost	**A$ /t TMM**	**$5.34**

The total capital cost for the Project has been estimated as of third quarter 2026 (3Q26) at A$766 million (M) including contingency. The distribution of capital by major project area is summarized in Table 1.8.3.

Table 1.8.3 Capital Estimate Summary (3Q26, ±35%)

Main Area		A$
0	Construction Indirects	43,565,778
1	Treatment Plant Costs	245,078,075
2	Reagents and Plant Services	97,684,726
3	Infrastructure	58,810,692
5	Management	60,538,940
6	Owners Project Costs (excl. Mining)	140,597,759
Subtotal		**646,275,971**
Contingency		119,913,227
Fees, Taxes and Duties		-
Escalation		-
Grand Total		**766,189,198**

Sustaining capital costs are included in the overall project economic evaluation. Sustaining capital costs for the life of mine (LOM) are estimated at A$119.1M (US$84), and includes:

- Tailings dam construction and operations costs.
- Process plant sustaining capital costs.

1.9 Economic Analysis

A financial model for evaluating the Project was developed by Mining Associates. Lycopodium reviewed the model logic, consistency of input assumptions and integrity of the calculations. All costs are estimated at 3Q26 in Australian dollars (A$) with no provision for inflation escalation.

The annual cash flow projections were estimated over the Project's production life based on production schedule, sales revenue, production costs, capital expenditures, and corporate costs (taxation, royalties, etc.). The financial indicators examined included after-tax cash flow (ATCF), net present value (NPV) at 7.5% discount rate, internal rate of return (IRR), and payback period.

Table 1.9.1 lists the principal assumptions made in performing the economic analysis.

Table 1.9.1 Principal Economic Analysis Assumptions

Item	Detail
Mill Ore Throughput, tpa	2,900,000
Mine Life, years	11
Mill Head Grade, % U_3O_8	As per mine schedule
Uranium Selling Price, US$ /lb	90
Exchange Rate, A$:US$	0.7
Taxes	
• Corporate Tax	30%
• Depreciation	Over asset useful life
• GST	Not Applicable
Royalties	
• QLD State Government	5%
• Royal Gold	1% (up to A$10M indexed)
Salvage Value	Nil
Inflation	Not Included
Discount Rate	7.5%

The results of economic analysis are shown in Table 1.9.2.

Table 1.9.2 Results of Economic Analysis

	A$	US$
Capital Cost	$891M	$623M
Operating Cost	$2,182M	$1,527M
Royalty	$318M	$223M
Tax	$831M	$582M
Revenue	$6,162M	$4,314M
Pre-Tax NPV	$1,605M	$1,126M
Pre-Tax IRR	43%	43%
Post Tax NPV	$1,059M	$741M
Post Tax IRR	33%	33%
U Production Mlb	47.9	47.9

The project has a 11 year mine life, with mining completed in 11 years (including pre-strip). Project payback period is approximately 2.5 years.

Sensitivity analysis shows that the Project's economics are most sensitive to commodity pricing assumptions and foreign exchange rate assumptions.

1.10 Conclusions and Recommendations

The opportunity for establishing a uranium mine and process facility at LAM's Westmoreland prospect has been investigated at the scoping study (SS) (±35% accuracy) level. Sufficient exploration drilling and modelling has been conducted to establish an Indicated Category resource estimate of 27,8000500 t at 0.077% uranium oxide (U_3O_8), and an Inferred Category resource estimate of 11,800,000 t at 0.068% U_3O_8.

Limited metallurgical testwork and associated sampling has been performed to date, but sufficient for SS level treatment flowsheet selection, development of material and energy balances, equipment sizing, and capital (Capex) and operating (Opex) cost estimates at the ±35% accuracy level.

Financial modelling based on the capital and operating cost estimates and market assessment of future uranium prices indicates that, at the SS level of accuracy, a project based on the Westmoreland prospect would have sound economics.

It is recommended that LAM consider preceding with the Project by progressing to the next project phase, i.e. a pre-feasibility study (PFS) (±25 % accuracy level).

2.0 INTRODUCTION

2.1 Laramide Resources Ltd.

This report has been prepared for LAM for their Westmoreland uranium prospect. LAM is a uranium development company listed on the Toronto Stock Exchange (TSX) and the Australian Stock Exchange (ASX).

2.2 Background to the Report

In May 2007, GRD Minproc completed a Preliminary Assessment Report for LAM on the development of the Westmoreland uranium prospect located in NW QLD, Australia. The Preliminary Assessment Report was prepared in general conformance with the Canadian *National Instrument 43-101* (NI 43-101) *Standards of Disclosure for Mineral Projects*.

In May 2015, following further exploration and metallurgical testwork, LAM retained Lycopodium to undertake a formal SS as a further step in the development of the Westmoreland prospect, and to prepare a Technical Report conforming to the requirements of the NI 43-101 *Standards of Disclosure for Mineral Projects*.

In April 2026, LAM engaged Lycopodium to undertake an update to the Preliminary Economic Assessment (PEA) for the Westmoreland prospect, and to prepare a Technical Report conforming to the requirements of the NI 43-101 *Standards of Disclosure for Mineral Projects*.

2.3 Contributors to This Report

Table 2.3.1 lists the scope of work split between the various contributors in the preparation of this report.

Table 2.3.1 Work Split Between Contributors for Report Preparation

Section No.	Section Name	Author
0	Cover	Lycopodium
1	Summary	Lycopodium / LAM / Addison Mining Services / Mining Associates
2	Introduction	Lycopodium
3	Other Experts	Lycopodium
4	Property Description	Addison Mining Services
5	Accessibility Climate	Addison Mining Services
6	History	Addison Mining Services
7	Geological Setting and Mineralization	Addison Mining Services
8	Deposit Types	Addison Mining Services

Section No.	Section Name	Author
9	Exploration	Addison Mining Services
10	Drilling	Addison Mining Services
11	Sample Preparation	Addison Mining Services
12	Data Verification	Addison Mining Services
13	Metallurgical Testing	Lycopodium
14	Mineral Resource Estimate	Addison Mining Services
15	Mineral Reserve Estimate	N/A
16	Mining	Mining Associates
17	Recovery Methods	Lycopodium
18	Project Infrastructure	Lycopodium / LAM / KP
19	Marketing Studies and Contracts	LAM
20	Environment	LAM
21	Capital and Operating Costs	Lycopodium / Mining Associates
22	Economic Analysis	LAM / Mining Associates
23	Adjacent Properties	Addison Mining Services
24	Other Relevant Data	Lycopodium / LAM / Addison Mining Services
25	Conclusions	Lycopodium / LAM / Addison Mining Services / Mining Associates
26	Recommendations	Lycopodium / LAM / Addison Mining Services / Mining Associates / KP
27	References	Lycopodium / LAM / Addison Mining Services / Mining Associates
	Signature Page	Sign-off by QP's
	Consent Certificates	Each Qualified Person

2.4 Principal Sources of Information

Information used in this report has been gathered from a variety of sources including:

- Information provided by qualified geologists employed by LAM regarding the geology, drilling, sampling and other exploration procedures and processes adopted by the Company.
- Metallurgical testwork undertaken by recognised testwork laboratories, notably the Australian Nuclear Science and Technology Organisation (ANSTO).
- Information from LAM personnel in relation to past history and previous studies on the Project.

A listing of the principal sources of information is included in Section 27 of this document.

2.5 Site Visit

No site visit to the Westmoreland property was undertaken for this 2026 update to the project PEA.

A site visit was conducted for the previous 2015 project report in July 2015, and attended by Dr. Geoff Duckworth—Lycopodium, and Andrew Vigar—Mining Associates. Mr. Siddle, the QP, completed a site visit to the Project area between 21 and 23 January 2025.

2.6 Qualifications and Experience

The individuals presented in Table 2.6.1, by virtue of their education, experience and professional association, are considered Qualified Persons (QP) as defined in NI 43-101, for this report. The QPs meet the requirements of the independence as defined in NI 43-101. Section responsibilities are also listed below.

Table 2.6.1 Persons Who Prepared this Technical Report

Qualified Persons Responsible for the Preparation of this Technical Report						
Qualified Person	**Position**	**Employer**	**Independent of LAM**	**Date of Last Site Visit**	**Professional Designation**	**Report Sections**
Ms. S. Hunter	Process Manager	Lycopodium (APAC) Pty Ltd	Yes	-	FAusIMM(CP), RPEQ	0, 1, 2, 3, 13, 17, 18.2-18.6, 19, 20, 21.1, 21.3, 21.4, 24, 25.3-25.8, 26.3-26.6, 27
Mr. R. J. Siddle	Principal Geologist	Addison Mining Services Ltd	Yes	21 - 23 January 2025	MAIG	1, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23, 24, 25.1, 26.1, 27
Mr. A. Stepcich	Associate Principal Mining Engineer	Mining Associates	Yes	-	FAusIMM(CP), RPEQ	1, 16, 21.1.3, 21.2, 21.4.1, 22, 25.2, 25.7, 26.2, 27
Mr. T. Rowles	Regional Manager	Knight Piésold	Yes	-	FAusIMM(CP), MAIG (RPGeo), RPEQ	1, 18.1, 18.7,26.5.3

Other experts upon whose contributions the QPs have relied on are presented in Table 2.6.2.

Table 2.6.2 Persons Who Contributed to this Technical Report

Expert	Position	Employer	Independent of LAM	Date of Last Site Visit	Report Sections
Mr. G. Dunn	Process Consultant	Orway Mineral Consultants	Yes	-	13

3.0 OTHER EXPERTS

3.1 Reliance on Other Experts

The legality and currency of mining tenure is outside the expertise of the Project team. For Section 4 on property tenure, Lycopodium has relied entirely on the advice of LAM.

No additional metallurgical testing was undertaken for this phase of the PEA update. Lycopodium has relied upon the previous 2015 testwork study evaluation and interpretation completed by Mr. G. Dunn for the metallurgical testing—Section 13 of the NI 43-101 report.

Lycopodium has relied upon the water supply study performed by GWS for LAM for the relevant components Sections 18 and 21 of the NI 43-101 report.

Lycopodium has relied upon the market studies from LAM for Section 19 of the NI 43-101 report.

Lycopodium has relied upon the environmental advice from LAM for Section 20 of the NI 43-101 report.

Lycopodium has relied upon the product transportation study performed by C7 International for the relevant components of Section 21 of the NI 43-101 report.

Lycopodium has relied upon the financial analysis from LAM and Mining Associates for Section 22 of the NI 43-101 report. Lycopodium has reviewed the inputs and basis for the financial analysis.

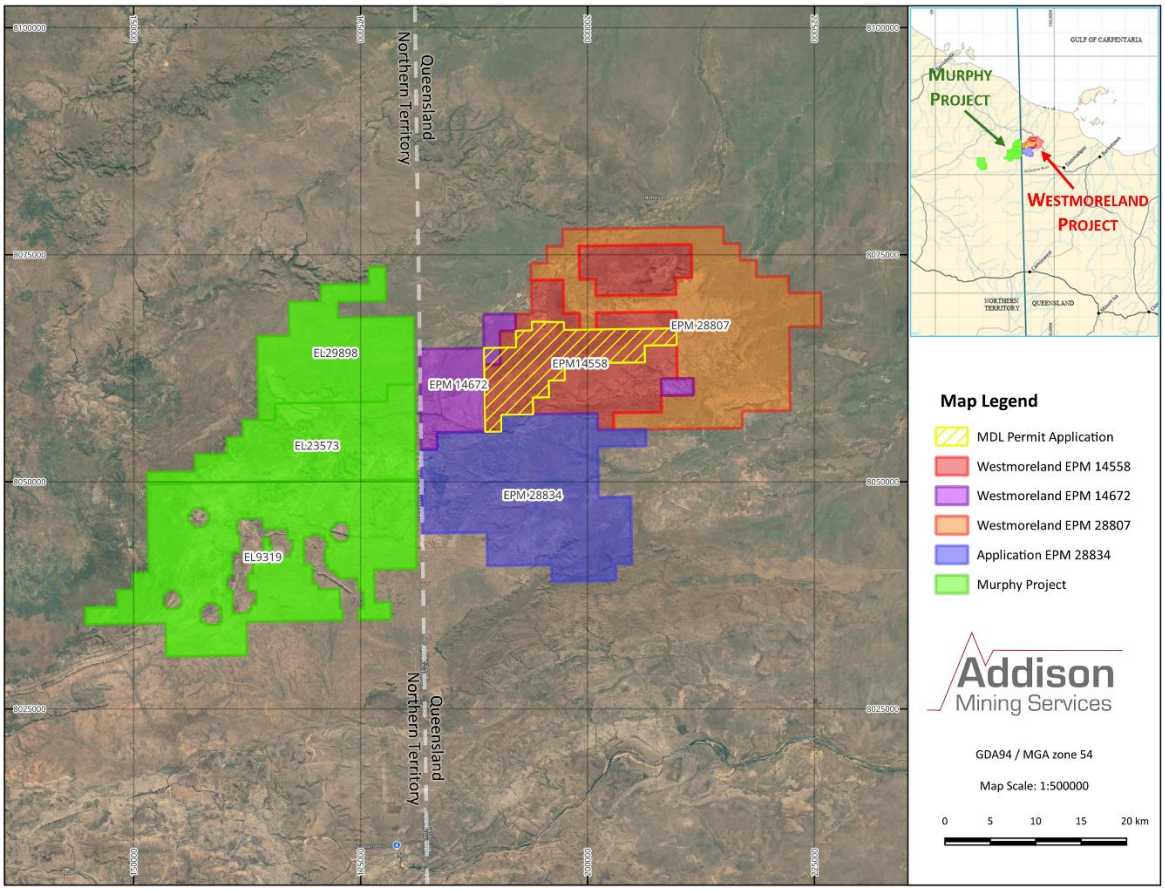
Lycopodium has relied upon the other relevant data advice from LAM and Addison Mining Services for Section 24 of the NI 43-101 report.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Project is located in NW QLD, Australia and comprises three granted EPMs, namely EPM14558, EPM14672, and EPM28807. A further EPM (28834) is in application and is yet to be granted at the time of writing. A MDL, MDL2026, was granted in July 2025 over ground dominantly held within EPM14558 with a minor portion (approximately 2%) of land in EPM14672. The issuer also holds additional adjacent mineral tenure in the NT of Australia; these properties form the Murphy Project and are considered separate to the Westmoreland Project for the purpose of this report. Figure 4.1 shows the positioning of these licences.

Further information relating to the Westmoreland project is provided in the following sub sections.

Figure 4.1 Location Map, Company Licences



4.1 Property Ownership and Location

LAM (the Issuer,) through its wholly owned Australian subsidiary Tackle, owns 100% of the Westmoreland Uranium Project. LAM also owns 100% of Lagoon Creek Resources Pty Ltd (LCR) and Westmoreland Resources Pty Ltd (WRPL) which, together, owns 100% of the Murphy Project Tenements in the NT.

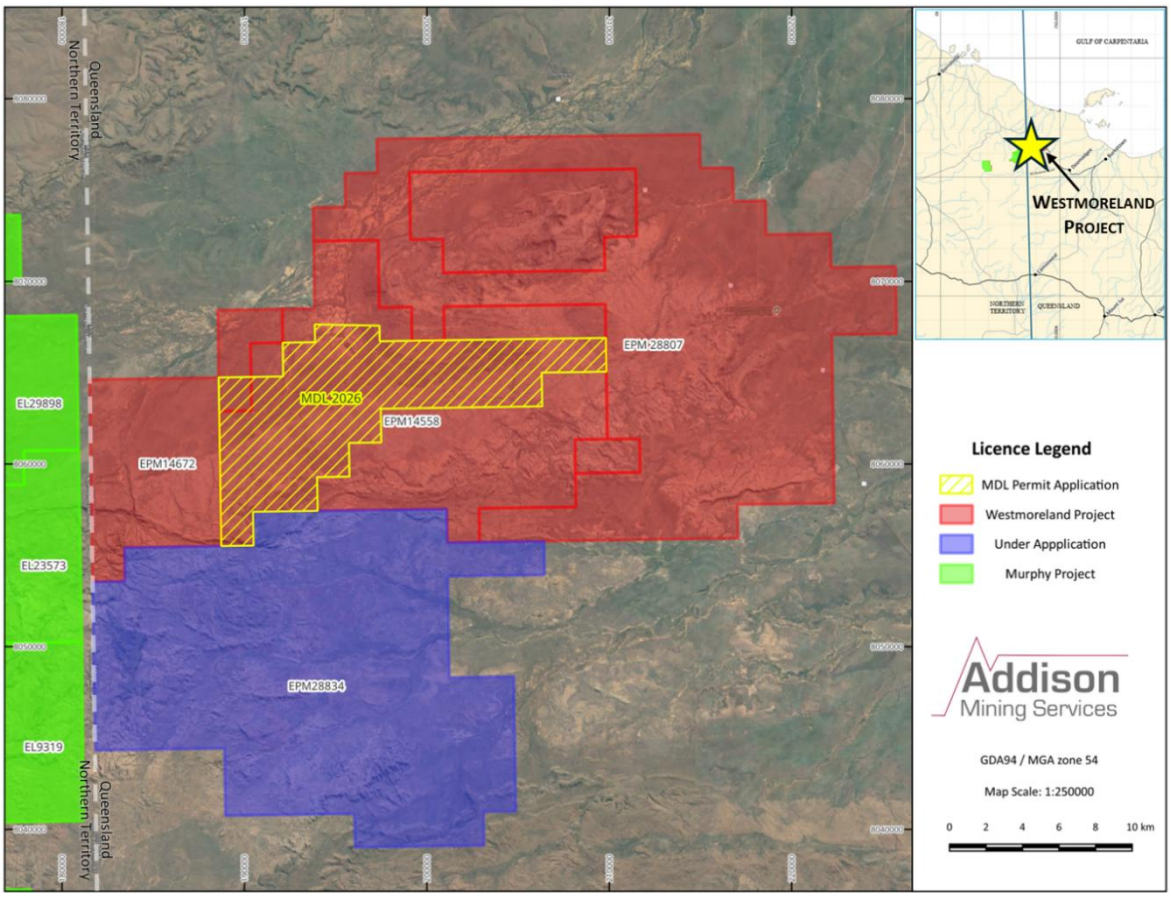
A schedule of relevant EPMs and MDLs is presented in Table 4.1.1 below and shown in Figure 4.1.1.

Table 4.1.1 Laramide Tenements in Queensland as of August 2025

Lease	Grant Date	Expiry	Area km^2	No. Sub-Blocks	Licence Holder	Approx. Centre GDA94 UTM Zone 54
EPM 14558	26.07.2005	24.07.2030	186.00	58	Tackle Resources Pty Ltd	199398 mE, 8063064 mN
EPM 14672	26.07.2005	24.07.2030	84.90	26	Tackle Resources Pty Ltd	186039 mE, 8060721 mN
EPM 28807	04.12.2024	03.12.2029	326.96	100	Tackle Resources Pty Ltd	212576 mE, 8067059 mN
EPM 28834	Under Application		326.75	100	Tackle Resources Pty Ltd	193960 mE, 8048406 mN
MDL 2026	01.07.2025	30.07.2030	111.14	N/A	Tackle Resources Pty Ltd	194849 mE, 8063392 mN

The Project is centred about 380 km NNW of Mt Isa City in northwest QLD. All Mineral Resources for the project which are described in this report are located within EPM14558 and this is considered the most material EPM to the project. EPM28807 has been recently granted (end of 2024) to LAM and has not undertaken any material exploration to date. MDL2026 encompasses the major deposits that form the Westmoreland Uranium Project in Queensland.

Figure 4.1.1 Location Map of Westmoreland Uranium Project



4.2 Surface and Access Rights

In Queensland, ownership of an EPM does not grant ownership of the land nor permission to access the property. Access to the Project requires a Conduct and Compensation Agreement (CCA) with relevant pastoral land holders. Furthermore, an Indigenous Land Use Agreement (ILUA) is in place with the relevant native title groups who are the traditional custodians of the land upon which the Project is located.

4.3 Mineral Rights in Queensland

Under the federal system, the Commonwealth Government regulates overall economic policy, tax, foreign investment, corporate law, and uranium mining and sale (environmental and safety aspects). State and Territory Governments own and allocate mineral rights, regulate operations, and collect royalties. The ownership of minerals is vested in the State, however ownership of minerals passes to the mining lease holder at the time of extraction.

The regulatory authorities and other parties involved in the mining leases and exploration permits include:

- Queensland Department of Natural Resources and Mines, Manufacturing, and Regional and Rural Development (NRMMRRD).
- Queensland Department of Environment, Tourism, Science and Innovation (DETSI).
- Burke Shire Council.
- Various pastoral lease holders and native title groups.

In Queensland, exploration requires a Queensland EPM, a tenure that can lead to an MDL or Mining Lease (ML) if exploration is successful. EPMs are granted for up to five years and may be renewed. Registered native title parties have consultation, objection, and negotiation rights regarding the proposed EPM.

Under the Environmental Protection Act 1994, 'Mining Activity' is classified as an 'Environmentally Relevant Activity'. An EPM is granted only after the EPA issues an Environmental Authority. An EPM allows activities to determine the existence, quality, and quantity of minerals, including prospecting, geophysical surveys, drilling, and sampling. Following resource identification, an MDL may be obtained for further economic viability evaluation. Full-scale mining requires an ML, with the term based on identified reserves and projected mine life.

4.4 Property Obligations and Fees

Under the Queensland Mining Act (Mineral Resources Act 1989), EPM holders must comply with conditions to maintain tenure, the most important of which regarding the Issuer's EPMs are as follows:

- Conduct of activities in accordance with EPA requirements.
- Compliance with all compensation agreements and making compensation payments as required.
- Supply technical and financial capability.

There is a minimum exploration expenditure requirement associated with the property; permit holders are required to show progress in line with an annual work program which sets out plans to advance the Project. Rent was previously required for EPMs, however under recent Queensland Government initiatives, rent relief has been applied to all EPMs forming the project and no annual rent is due until the 31 August 2028.

4.5 Royalties

4.5.1 State Royalties

A royalty is payable to the State government when a mineral is sold, disposed of, or used. In Queensland, the Mineral Resources Act 1989 requires that the holder of an ML to lodge a royalty return and any royalty payable at least annually for all MLs held, even if no production took place. Larger producers are required to pay royalties on a quarterly basis, while smaller producers generally pay royalties on an annual basis.

In the Mineral Resources Regulations 2013 - Schedule 3, S.13 includes a royalty rate of 5% if the average price per kilogram of uranium sold is A$220 or less. Above A$220 /kg, the rate increases up to a maximum of 10%.

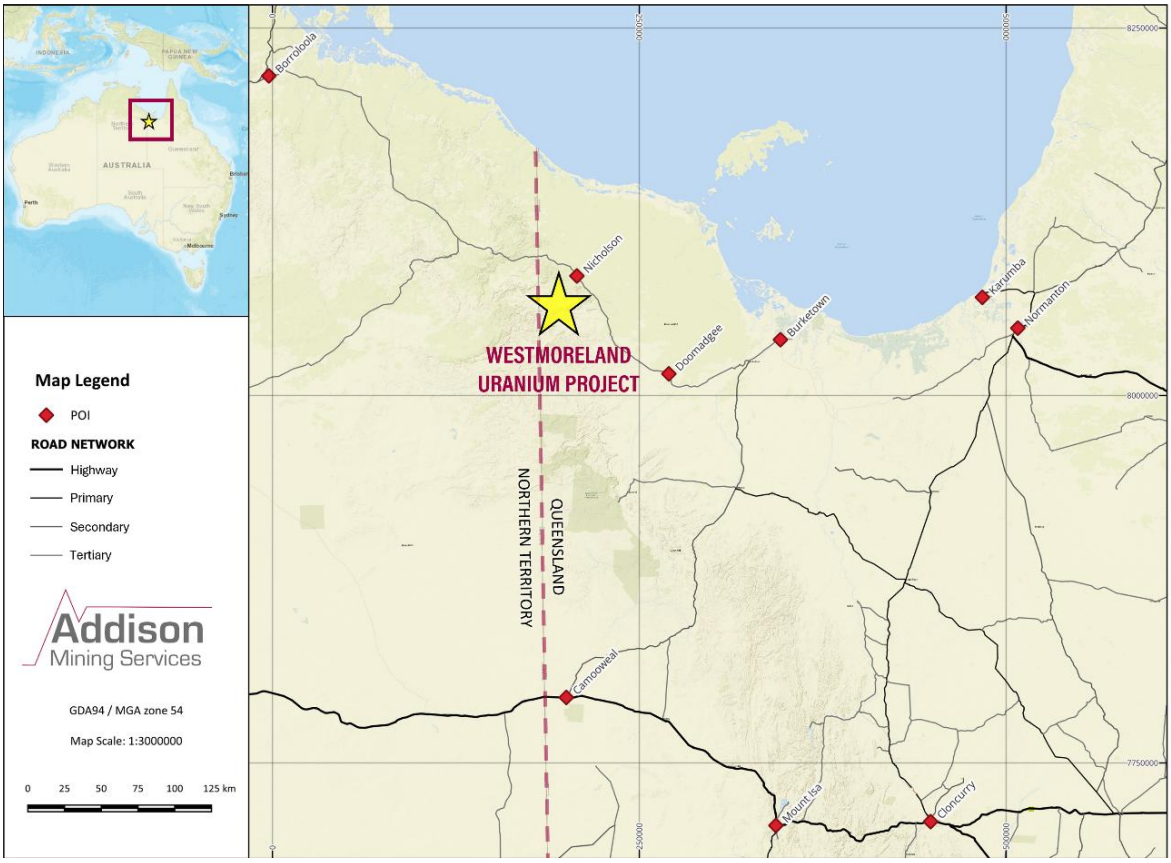
4.5.2 Other Royalties and Back in Rights

As part of the Data Licencing Agreement (DLA), dated 20 Dec 2005, a 1% Net Smelter Royalty (NSR) was agreed payable to Royalty Gold with cumulative payments capped at A$10M indexed to inflation. In 2008, Rio Tinto sold the royalty to IRC. In February 2010, IRC was fully acquired by Royalty Gold.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

Figure 5.1.1 Map of Regional Accessibility to Westmoreland Project

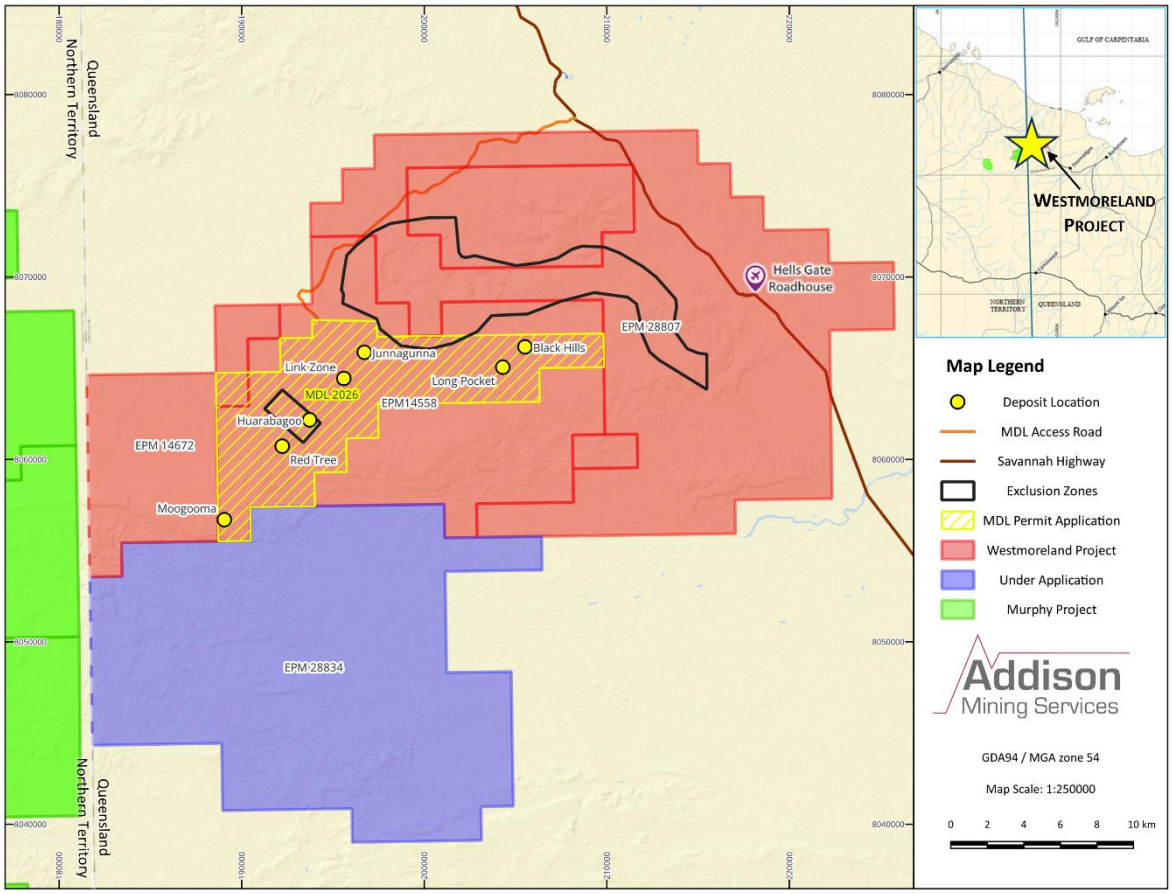


The Project is readily accessed from the Savannah Highway, a road running from Burketown in QLD to Borroloola in the NT. The highway is a two-lane sealed road in QLD from Burketown to the Hells Gate Roadhouse, which is approximately 50 km from the NT border; thereafter the road remains unsealed. A network of gravel roads and pastoral tracks provides access to the property from Hells Gate Roadhouse which is approximately 15 km from the closest significant mineral deposit (LP) and approximately 30 km from the RT deposit. Hells Gate is currently used as a base of operations for exploration activities and has an airstrip suitable for medium twin-engine aircraft. During periods of intense rainfall in the summer wet season, some road creek crossings may be impassable and this may include sections of the Savannah Highway. On the property approximately 2 km north of the HB deposit, there is a disused two runway crossed airstrip; however, this is not currently serviceable and would require upgrade to be operational. Access routes to the property are shown in Figure 5.1.2.

An activity exclusion zone exists at the southern end of the HB deposit, which will require further negotiation for future access and exploration activities.

The region has two ocean ports—Burketown (non-trading) and Karumba. Karumba Port, at the Norman River mouth, is strategically important for mining exports (e.g. Century mine zinc), live animal exports, and it supports fishing fleets. It also handles general freight for coastal communities and is a major live cattle export hub to Asia.

Figure 5.1.2 Local Access with Main Deposit Locations and Access Routes



5.2 Regional Population Centres

Mt Isa, the largest city in the region with a population of 18,317 (2021 Census, Table 5.2.1), is a significant mining and industrial hub. It benefits from daily flights via Qantas and Regional Express to Brisbane, Townsville and Cairns, and is connected to Townsville, QLD's largest city outside the capital, by major road and rail links.

While the primary land use in the broader region is pastoral, mining is the dominant economic driver, supported by large operations like the Mount Isa copper mine and the McArthur River and Century lead-zinc mines. The fishing industry also contributes significantly to employment. Any skilled workforce required for mining developments in the region would likely be sourced from Mt Isa.

The towns situated near the Westmoreland tenement block are smaller, offering essential amenities such as small health clinics, supermarkets, and a domestic airport. Notably, until 2016, these settlements were entirely powered by diesel generators. However, Ergon Energy, the state's energy provider, secured funding in 2023 to install an additional 4.5 MW centralized solar farm, aiming to supply over half of the communities' energy needs from renewable sources by the end of 2025.

Table 5.2.1 Population Centres

Town	Population	Distance Radial km	Principal Activity
Mt Isa	18,317	400	Mining
Cloncurry	3,167	440	Mining
Normanton	1,391	300	Fishing
Karumba	487	275	Port / Fishing
Doomadgee	1,387	95	Indigenous
Borroloola	871	250	Pastoral
Burketown	204	145	Pastoral

2021 census, except Borroloola 2016 census.

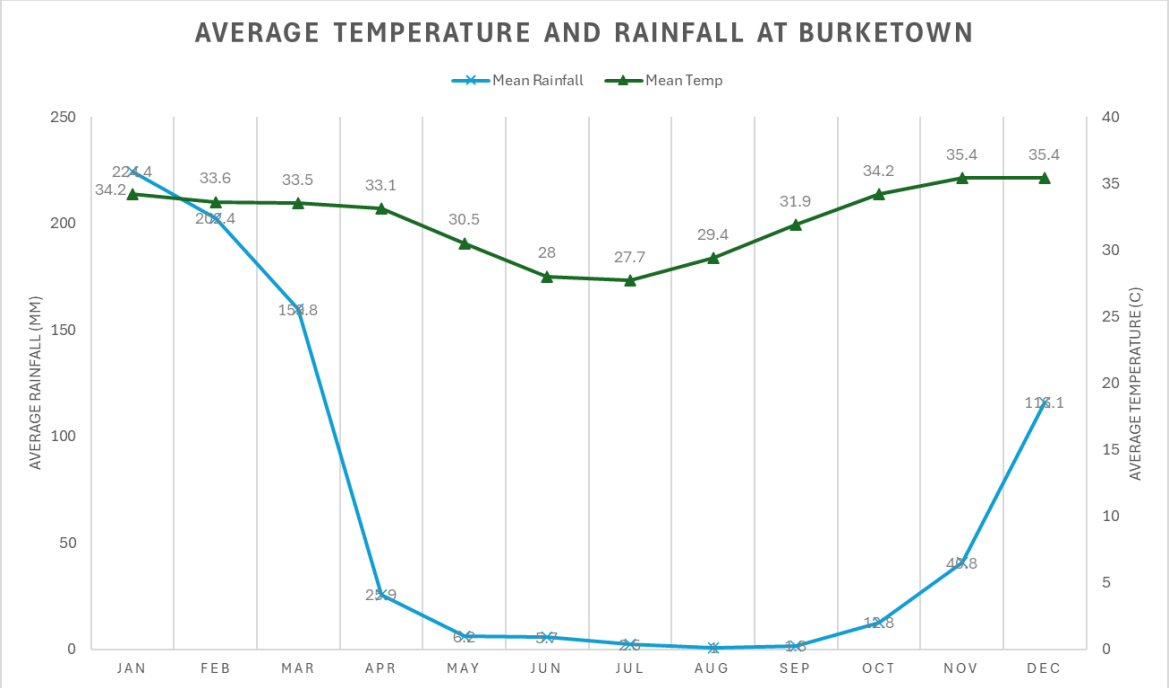
Population data acquired from the Queensland Government Statisticians Office (2021).

5.3 Climate

The Gulf region exhibits climatic gradients along the coast and north-south. Summer rainfall varies from 400 millimetres (mm) in the south to 800 mm in the north, with significant annual variability. Summer temperatures are hot (around 36°C), dropping to a mean of around 28°C in July.

Burketown, 150 km east of the tenement block, shows climate data influenced by the Gulf, moderating inland extremes (Figure 5.3.1). Most rain falls during the December to March monsoon, with January averaging 224 mm but historically reaching up to 1,000 mm.

Figure 5.3.1 Average Temperature and Rainfall at Burketown



Drawn by O. Ford from data provided by the Australian Bureau of Meteorology (2025).

5.4 Local Resources and Infrastructure

The property is not serviced by local mains power water or sewerage. Existing and groundwater wells on the property are able to provide water all year round for pastoral activities, as do the local dirt roads.

5.5 Physiography

Westmoreland is in a region known as the Gulf Country, which includes the southern shores of the Gulf of Carpentaria and the country around the many rivers that flow into the Gulf. The tenements are situated in remote, sparsely populated, rugged hill country. Topography ranges from broad gentle valleys covered by open woodland dominated by grey box eucalypt trees, to steep rugged east-west trending ridges on the flanks of valleys. The terrain ranges in elevation from 80 m to 360 m.

6.0 HISTORY

6.1 Ownership

The property has undergone an extensive history that evolved through a mix of independent exploration, joint ventures (JVs), corporate acquisitions, and reorganization, reflecting the shifting dynamics of mining exploration and resource focus in the area. A detailed summary is described in Vigar and Jones (2006) and is summarized as follows:

1. 1890s to 1950s: Nearby discoveries saw early prospecting efforts by independent explorers. Mount Isa Mines Limited (MIM) obtained Authority to Prospect in 1956 and discovered the RT deposit shortly after.

2. 1959: MIM formed a JV with Consolidated Zinc Pty Ltd. Consolidated Zinc later became CRA Ltd (CRA), eventually gaining full ownership of the leases.

3. 1967 to 1996: Queensland Mines Ltd (QML) conducted substantial exploration before shifting focus elsewhere. QML partnered with various entities, including Urangesellschaft Australia Pty Ltd (UGA), Anglo Australian Resources NL, Omega Mines Ltd (Omega), and CRA, with CRA assuming full control of the JV by 1996.

4. Post-1990: CRA merged into the Rio Tinto group, which held significant tenements before relinquishing them in 2000. These areas were subsequently claimed by Tackle Resources Pty Ltd.

6.1.1 1890s to 1910s – Early Regional Work

The Westmoreland region was probably first prospected in the 1890s, after the discovery in 1887 of silver-lead deposits at Lawn Hill, 100 km south. Copper was discovered in 1911 at Settlement Creek and at the nearby Redbank lode in the NT in 1916. In 1912, the Packsaddle and Bauhinia copper lodes were discovered near Wollogorang homestead. Pitchblende has been mined in the Peters Creek Volcanics, which overlie the Westmoreland Conglomerate, 20 to 30 km west of RT (Syvret, 1957).

6.1.2 1956 - Mount Isa Mines (MIM)

In August 1956, MIM secured Authority to Prospect (AP) 46M, covering 1,800 square miles (mi^2) near the QLD-NT border, targeting copper and uranium. Following up on Bureau of Mineral Resources (BMR) airborne scintillometer survey anomalies, MIM prospector A. Blackwell discovered a 'promising occurrence of torbernite' in the Westmoreland Conglomerate in November 1956, naming the deposit RT.

Due to the low grade and remote location, MIM relinquished the AP but secured mining leases over RT and other surface uranium mineralization in 1959, in a 50:50 JV with Consolidated Zinc Pty Ltd (later CRA, which acquired 100% interest).

6.1.3 1967 - Queensland Mines Ltd (QML)

QML took control of the licence area in 1967 and carried out exploration drilling, pitting and shaft sinking work programs in the region (see Section 9.1.1 for detail). The HB deposit was also discovered during this period.

Following the discovery of the Nabarlek deposit in 1971, QML ceased exploration at Westmoreland to concentrate their efforts in the Alligator Rivers area of the NT.

6.1.4 1975 – Joint Venture Work

In 1975, QML formed a JV with UGA, Anglo Australian Resources NL and CRA. UGA discovered the deposit in the period 1976 to 1983 when they were managing the JV. Omega entered the JV in 1982. In 1990, CRA took over management and purchased 100% of the JV in 1996. Prior to this time, CRA had purchased a 100% interest in the old MIM mining leases at RT.

6.1.5 1960 to 1980

From 1960 to 1980, 14 EPMs were held and explored within the boundary of the present EPM14558, generating 60 open file reports. Several operators were involved in the area at the time:

- 1967 to 1973 BHP
- 1968 to 1970 US Steel International
- 1970 Westmoreland Minerals Limited
- 1971 to 1972 Esso Mineral Enterprises Australia Ltd
- 1973 to 1979 Mt Arthur Molybdenum NL
- 1975 to 1981 Savage Exploration Pty Ltd
- 1977 to 1979 Mines Administration Pty Ltd

Exploration programs are summarized in Table 9.1.1 in Section 9.1.2.

6.1.6 1980 to 1990

The surge in gold exploration from 1980 to 1990 was reflected in the increased tempo of exploration in the Westmoreland area. Ten EPMs were granted in the area now covered by EPM14558; 35 open file reports record the work done through this decade. The following companies were operating during this decade:

- 1980 to 1982 Minatome Australia Pty Ltd

- 1982 to 1989 Uranerz Australia Pty Ltd
- 1983 to 1984 Total Mining Australia Pty Ltd
- 1983 to 1989 Central Electricity Generating Board Exploration (Australia) Pty Ltd
- 1984 to 1985 International Mining Corporation NL
- 1987 CSR Ltd
- 1988 to 1989 Golden Plateau NL

Significant exploration work is summarized in Table 9.1.2 in Section 9.1.3.

6.1.7 1990 to 2000

Exploration slowed post-1990, with only seven EPMs granted in the area of EPM14558 in the last 15 years. The GSQ received only 15 open file reports during this time, all from CRA detailing their previous work. CRA held a major tenement position until its absorption into Rio Tinto, which relinquished the tenements in 2000. Tackle then applied for these licence areas.

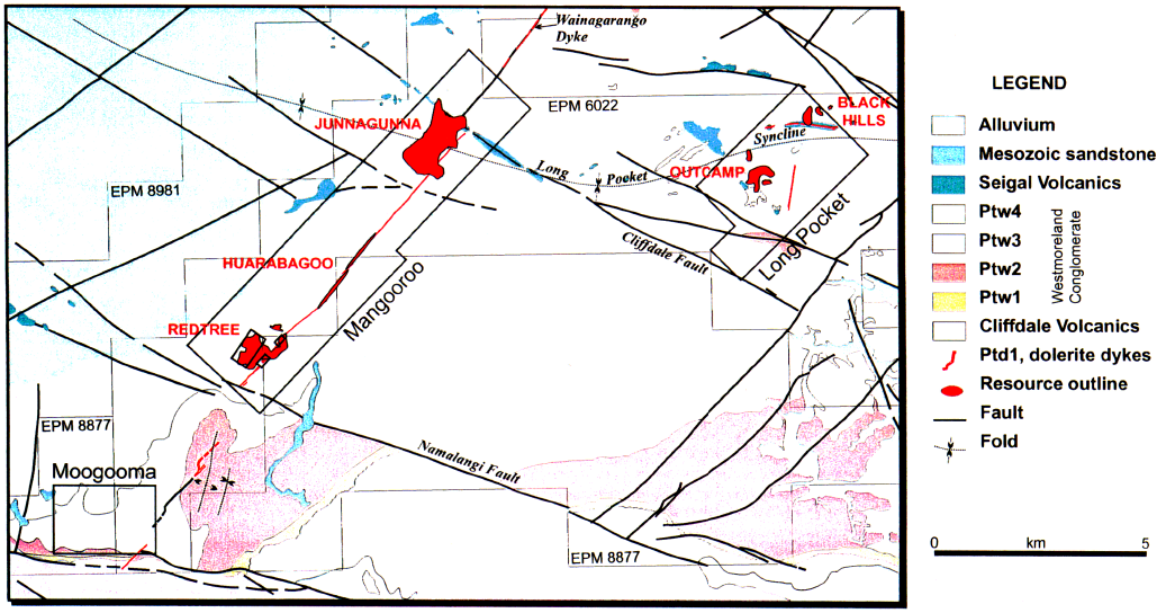
6.2 Previous Resource and Reserve Estimates

Previous operators conducted numerous mineral resource studies at RT over a 25-year period (1969 to 1994). These estimates are detailed in Vigar and Jones (2006), with the main estimates summarized below.

6.2.1 Early Estimates

- QML conducted the earliest estimate in 1969 for the Jack Lens at RT, using a basic polygonal method on 123 drillholes.
- In 1990, 1994 and 1995, CRA produced several internal estimations, improving their methodology with each attempt. They initiated a comprehensive review of the Westmoreland tenements (Figure 6.2.1) and digital terrain model was created from aerial photos (1990 to 1993). By 1993, 1,249 drillholes were in the database.

Figure 6.2.1 CRA's Westmoreland Uranium Deposits



Minenco, 1995.

- In 1995 Minenco were commissioned by CRA to update their earlier PFS using a lower cut-off grade of 0.05%.

The estimations produced were not prepared in accordance with CIM definition standards and not reported under NI 43-101. The 1996, the estimate by Minenco was never signed off by a Competent Person. Additional estimates were produced during this period but were based on less data than what is available today. Therefore, to avoid misleading the reader, these historical estimates are not included here.

6.2.2 2006 – Mining Associates Estimation

In 2006, Mining Associates (Vigar and Jones, 2006) created three dimensional (3D) geological interpretations using a 0.02% (200 ppm) cut-off and 1 m minimum width. The project area was split into three sub-areas based on mineralization and drilling trends. Geological wireframes defined domains of similar style to constrain the resource model. Mining Associates classified resources above a 0.02% U_3O_8 economic cut-off, deemed appropriate for the shallow, flat deposit.

Table 6.2.1 Resource Estimates (Above 0.02% U_3O_8) from 2006

Category	Deposit	Tonnes	U_3O_8 Uncut	U_3O_8 Cut	U_3O_8 kt	Mlb
Inferred	Redtree	10,928,500	0.094%	0.093%	10.2	22.4
	Huarabagoo	2,925,250	0.122%	0.108%	3.2	7.0
	Junnagunna	2,149,500	0.077%	0.075%	1.6	3.6
Total Inferred w/ Grade Average		**16,003,250**	**0.097%**	**0.094%**	**14.9**	**32.9**
Indicated	Redtree	3,672,250	0.096%	0.096%	3.5	7.8
	Huarabagoo	0	-	-	0.0	0.0
	Junnagunna	4,364,750	0.082%	0.081%	3.5	7.8
Total Indicated w/ Grade Average		**8,037,000**	**0.088%**	**0.088%**	**7.1**	**15.6**

6.2.3 2009 – Mining Associate Update

In 2009, a resource update was undertaken by Mining Associates (Vigar and Jones, 2009) which included the 2007 and 2008 drilling results.

Mining Associates (Vigar and Jones, 2016) published a SS (PEA) in 2016. This update, reviewed by a 2009 QP, stated that the drilling undertaken in 2009, 2010 and 2012 confirmed earlier work but did not significantly change the resource. The 2009 estimate was restated in the April 2016 study with no change to Indicated or Inferred tonnages. The May 2009 estimate (Table 6.2.2) was reported in accordance with NI 43-101 *Standards of Disclosure for Mineral Projects* and prepared under CIM Definition Standards. The estimate was also reported in accordance with the *Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves* ('the JORC Code' 2012 edition.)

Table 6.2.2 Westmoreland Mineral Resource Estimates, May 2009

Deposit	Tonnes t	Density g/m^3	U_3O_8 ppm	U_3O_8 Mlb
Indicated				
Redtree	12,900,000	2.5	900	25.5
Huarabagoo	1,460,000	2.5	830	2.7
Junnagunna	4,360,000	2.5	810	7.8
Long Pocket	-	-	-	-
Total Indicated	**18,700,000**	**2.5**	**880**	**36**
Inferred				
Redtree	4,460,000	2.5	670	6.6
Huarabagoo	2,400,000	2.5	1,090	5.8
Junnagunna	2,150,000	2.5	750	3.6
Long Pocket	-	-	-	-
Total Inferred	**9,000,000**	**2.5**	**800**	**15.9**

6.2.4 2016 – Scoping Study and Preliminary Economic Assessment (PEA)

In April 2016, Lycopodium and Mining Associates completed a SS and PEA for the Project (Vigar and Jones 2016). The study outlined scope for open-cut mining for a 13-year LOM and considered other aspects of the project, including pit optimization, mine scheduling, processing methodologies and throughputs, infrastructure requirements, and economic analysis. The economic inputs used in this study are no longer considered current.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Project licences are located on the SE margin of the southern McArthur River Basin. Mapping of the area done in the late 1970s is covered by the Geological Survey of Queensland's (GSQ) 1:250,000 scale 'Westmoreland Geological Sheet' and the BMR's 1:100,000 scale 'Seigal NT and Hedleys Creek Qld' sheet.

The Palaeo-Mesoproterozoic McArthur Basin is a substantial (5 to 10 km thick) sequence of largely unmetamorphosed sedimentary and volcanic rocks deposited on the North Australian Craton between approximately 1800 and 1575 megaannum (Ma). The southern extent of this basin is defined by the Murphy tectonic ridge (Murphy Inlier), which separates it from the Mt Isa Inlier. The east-west trending Urapunga fault zone marks the boundary between the southern and northern parts of the McArthur Basin. The oldest sediments in the southern basin unconformably overlie the Cliffdale Volcanics (part of the Murphy Inlier), the Scrutton Volcanics, and the Urapunga Granite.

Sediment deposition occurred in a variety of intracratonic settings, including fluvial, coastal, and shallow marine environments. The southern McArthur Basin is lithostratigraphically divided into the Tawallah, McArthur, Nathan, and Roper Groups. However, on the Wearyan Shelf, where the Westmoreland project is situated, only the Tawallah Group is significantly represented, with the younger groups largely absent (except for the Karns Dolomite of the Nathan Group).

The stratigraphy of the southern McArthur Basin is further categorized into three regionally correlatable 'superbasins': the Leichhardt (ca. 1800 to ca. 1740 Ma), the Calvert (ca. 1710 to 1690 Ma), and the Isa (ca. 1670 to 1575 Ma), separated by approximately 20 to 25 million year gaps. Jackson et al. (2000) proposed an eight-fold subdivision of older Statherian rocks within these superbasins across an 800 km outcrop belt. The Leichhardt Superbasin comprises the five older associations (A to E), and the Calvert Superbasin includes the younger three. These superbasins are separated by a basin inversion event.

The lithology of the Calvert and Leichhardt superbasins in the southern McArthur Basin is characterized by shallow marine and fluvial siliciclastic successions, outer-ramp carbonate successions, turbiditic siliciclastic units, and bimodal igneous rocks. These sediments can be broadly classified into four facies associations, with the proximal fluvial facies being prominent near the base of the Leichhardt superbasin, particularly in formations like the Westmoreland Conglomerate, lower Yiyinti Sandstone, and Sly Creek Sandstone, where they form thick sequences.

Figure 7.1.1 Geological Setting of Northern Australia with Westmoreland Licences

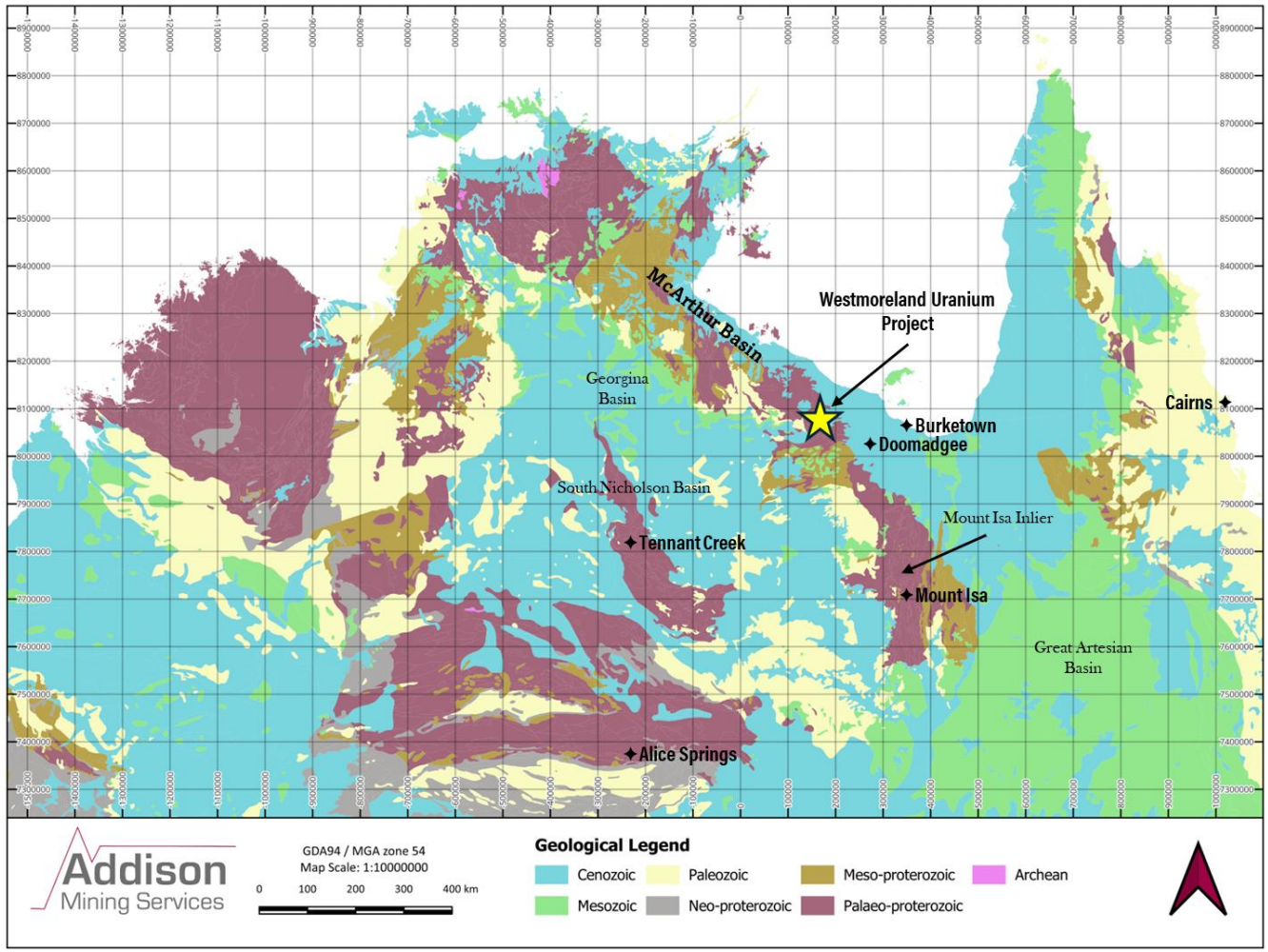
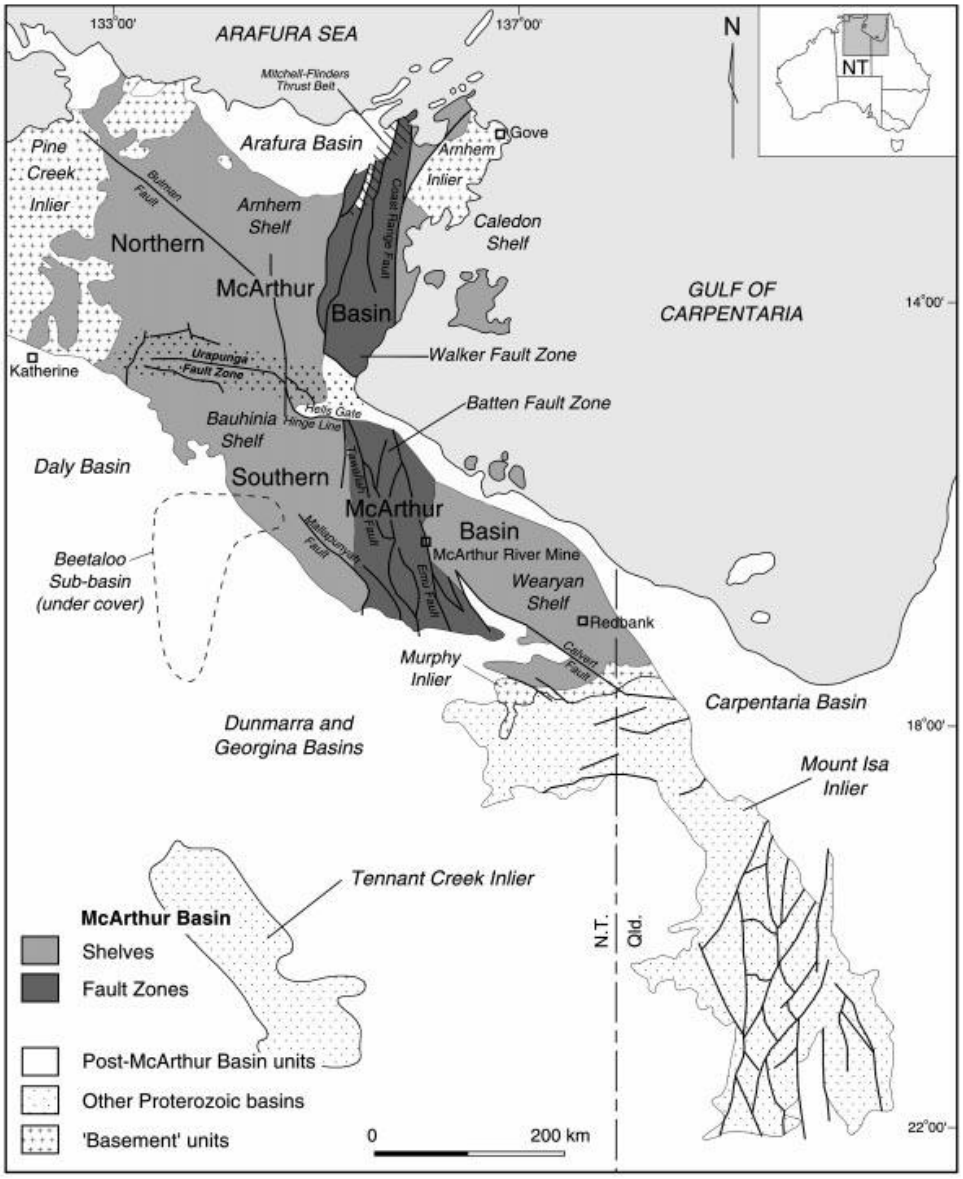


Figure 7.1.2 McArthur Basin – Westmoreland Located North Flank of Murphy Inlier



Rawlings, 1999.

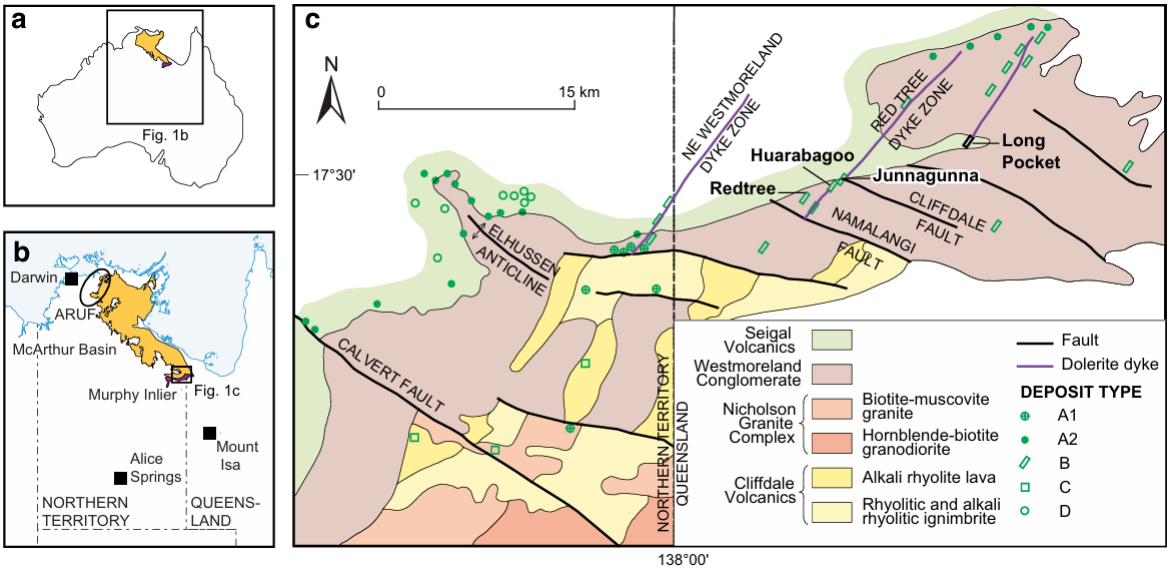
7.1.1 Westmoreland-Pandanus Creen Uranium Mineralization

The Westmoreland–Pandanus Creek uranium field has four types of uranium occurrences (Figure 7.1.3):

- Type A: At the reverse-fault contact between the Cliffdale Volcanics (hanging wall) and Westmoreland Conglomerate (Type A1), or at the contact between the Seigal Volcanics and the conformably overlying Westmoreland Conglomerate (Type A2).
- Type B: Near a contact between impermeable vertical mafic dykes and the Westmoreland Conglomerate.

- Type C: Hosted by the Cliffdale Volcanics, beneath an exhumed unconformable contact with the overlying Westmoreland Conglomerate.
- Type D: Hosted by fractures in the Seigal Volcanics, at some distance above the contact with the Westmoreland Conglomerate.

Figure 7.1.3 Geological Setting for the Westmoreland Uranium Field



Modified from Lally and Bajwah (2006).

a) Location of the McArthur Basin and Murphy Inlier.

b) Westmoreland (black rectangle).

c) Simplified geology map.

Sandstone Hosted Westmoreland-Style

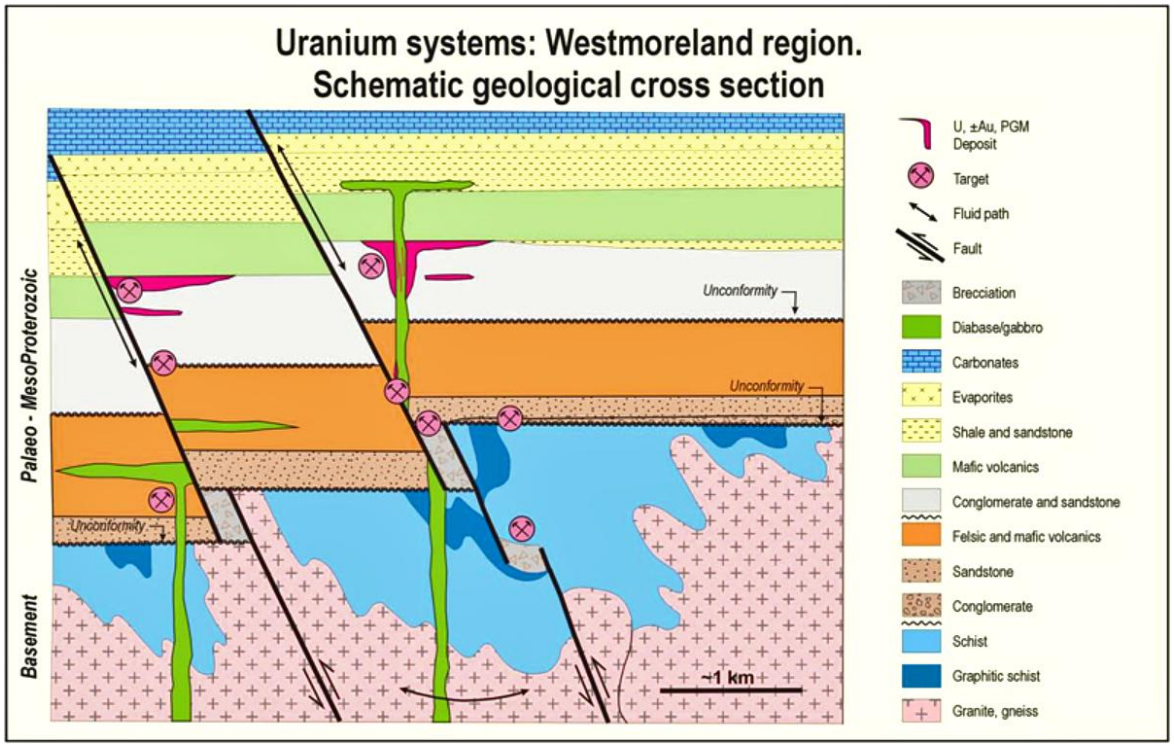
Polito et al. (2005) and Polito et al. (2006) proposed that poorly sorted fluvial sandstones and conglomerates in the southern McArthur Basin evolved into diagenetic aquifers hosting basinal brines similar to those in lead-zinc (Pb-Zn) and U deposits. These brines, formed primarily in the Westmoreland Conglomerate and parts of the Calvert Superbasin, could leach Pb-Zn ± copper (Cu) and U from sediments and volcanics. Well-sorted sediments typically become aquitards at depth due to quartz cementation, limiting their brine source potential. Conversely, initially poor aquifers with heterogeneous composition can develop secondary porosity through framework dissolution at depths exceeding 4 km, becoming conduits for metal-bearing brines. Diagenetic alteration replaces unstable detrital minerals with stable phases like illite and chlorite, indicative of aquifer formation. Increased silicate solubility at depth created secondary porosity in the Westmoreland Conglomerate, facilitating fluid flow. Basinal brines in the Leichhardt Superbasin reached ~200°C (corresponding to 5 to 9 km burial depth) and were open to fluid migration from 1680 ± 21 Ma to 1541 ± 8 Ma, encompassing the formation of the RT-JG U deposits (~1650 Ma) (see Section 7.3).

Cliffdale Volcanic-Hosted Eva-Style

Lally and Bajwah (2006) cited a 1987 classification dividing Westmoreland-Pandanus Creek uranium occurrences into five types based on hydrology and geology. McKay et al. (2001) used a local system classifying Eva as shear zone mineralization in altered acid volcanics (Cliffdale Volcanics). All uranium mineralization in the area is within or near the Westmoreland Conglomerate.

Prospect geology and isotope studies suggest mineralizing basinal brines moved through the Westmoreland Conglomerate, along unconformities (reverse faults), and into shear zones and permeable sandstones within the Cliffdale Volcanics, where uranium precipitated via reduction, possibly by diagenetic chlorite.

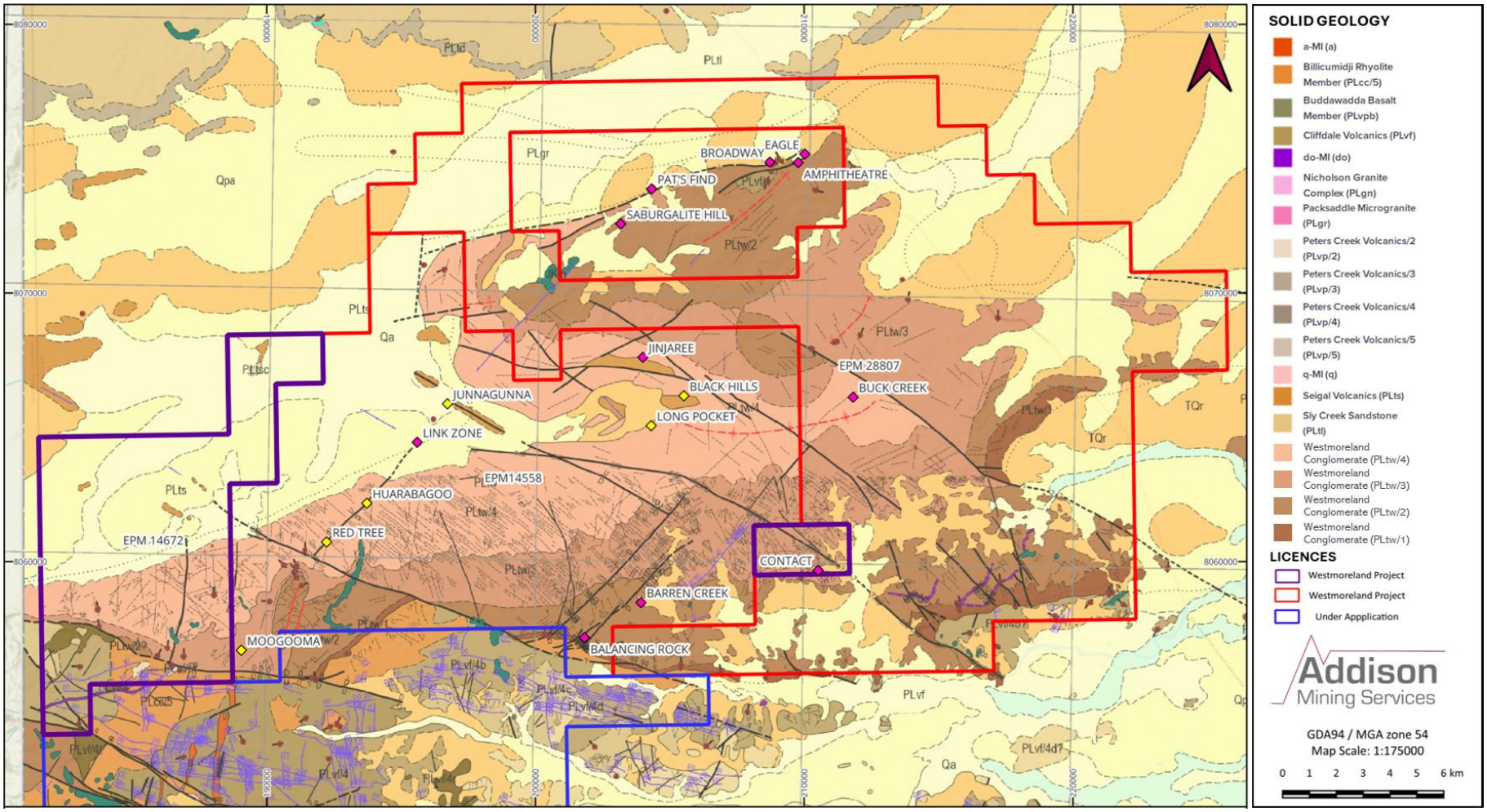
Figure 7.1.4 Uranium Systems of the Westmoreland Region: Schematic Geological Cross-Section



7.2 Local Geology

The Westmoreland tenements are centred about the outcropping Westmoreland Conglomerate of the Tawallah Group, where the southern McArthur basin on laps the Cliffdale Volcanics of the Murphy Inlier (e.g. Blaikie and Kunzmann (2020)). The RT deposit is located in the southwest (SW) of EPM14558.

Figure 7.2.1 Local Geology of Westmoreland Project



The Westmoreland uranium deposits (RT, JG, HB) are primarily hosted within the shallow-dipping Westmoreland Conglomerate. This conglomerate, up to 1,800 m thick, is divided into five fining-upward units (PTW1-PTW4), each representing proximal fluvial deposits (debris flows, alluvial fans, braided rivers) overlain by well-sorted sandstone. Angular unconformities or disconformities mark breaks, with new units often starting with pebble / boulder conglomerate. The basal conglomerate contains cobbles and coarse sand (reworked quartz veins, chert, felsic / mafic volcanics from the Murphy Inlier or similar basement) and it is suggested that this detrital material and the lithic clasts themselves likely acted as a source for the uranium mineralization found within the deposit (Polito et al., 2005). NE-trending fractures, some filled with dolerite, crosscut the conglomerate.

Conformably overlying the Westmoreland Conglomerate is the Seigal Volcanics unit. This unit is characterized by predominantly massive flows of tholeiitic basaltic lava, although localized occurrences of amygdaloidal textures are noted, along with minor interbeds of siltstone and sandstone.

Aphyric, medium-grained dolerite dykes, often NE-trending and potentially linked to basement weaknesses intrude both the Westmoreland Conglomerate and the underlying basement units of the Murphy Inlier. These dykes, like the RT dyke, weather easily and are often obscured at the surface. In its unaltered state, the dyke rock is typically a dark green dolerite and in core shows brecciation, shearing, and alteration at contacts. The RT dyke's geochemistry resembles the Seigal Volcanics, suggesting it might have been a feeder (Polito et al., 2005; Rheinberger et al., 1998). The RT dyke zone is a series of en echelon dykes (typically <20 m thick, 1 km long) extending for 15 km.

Figure 7.2.2 Showing WNW and NW Faults and Mineralization as a Function of Thresholded U/Th

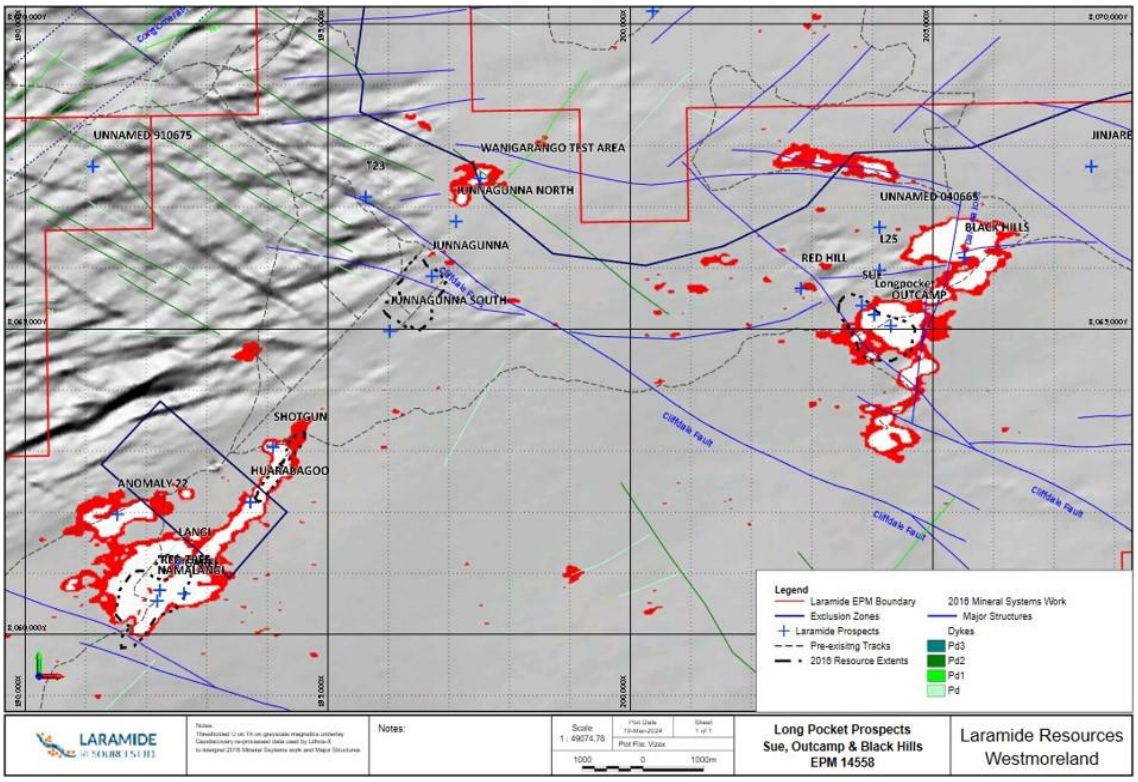


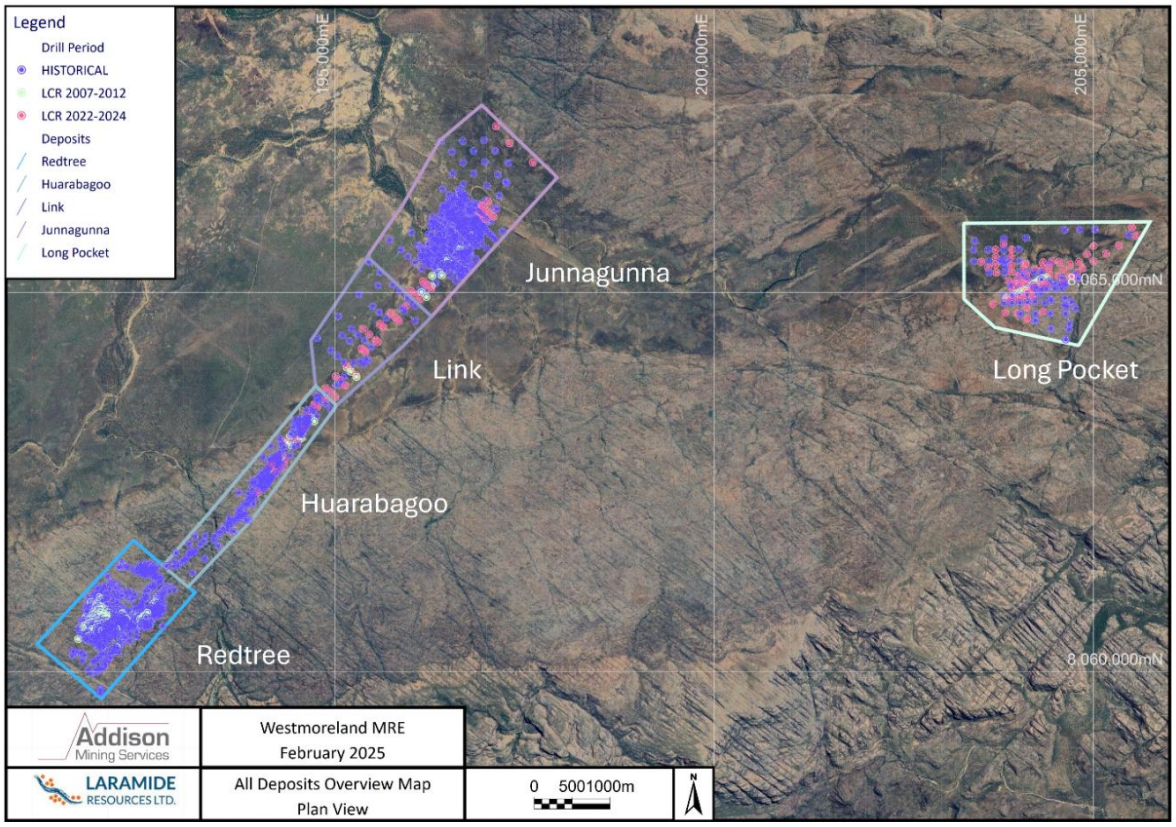
Figure 7.2.3 Westmoreland Conglomerate Outcrop



7.3 Significant Mineralized Areas

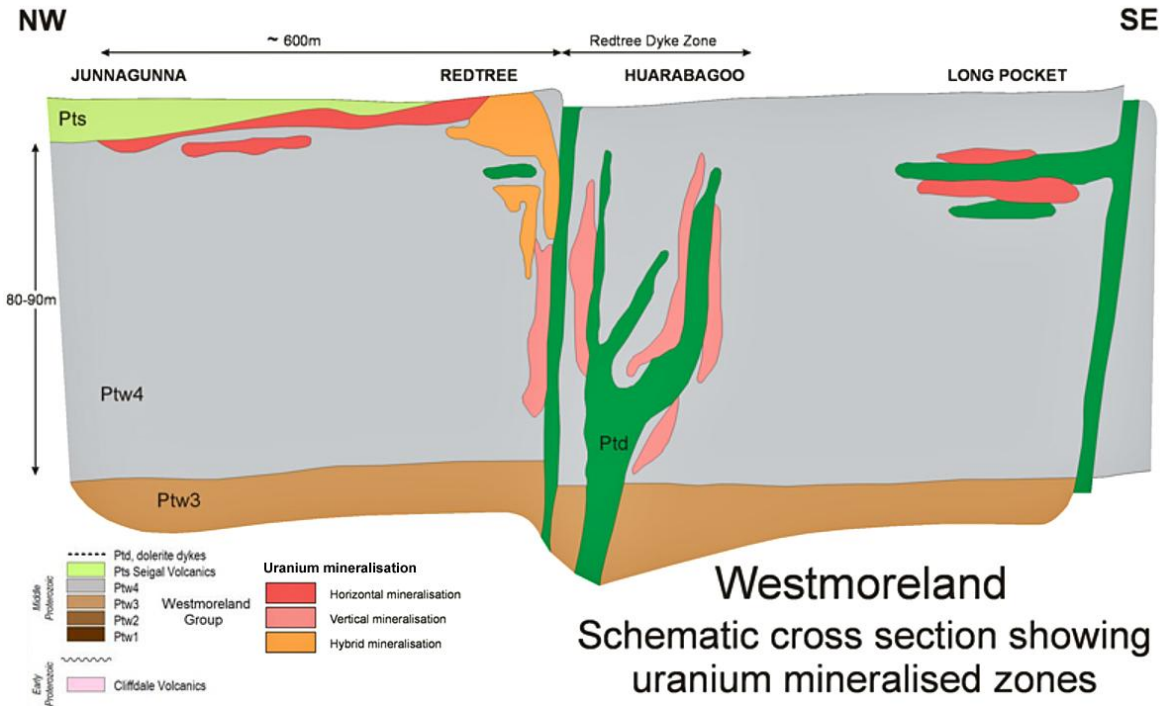
The four main deposits located in the prospect area are all hosted within the shallow dipping Westmoreland Conglomerate with alluvial overburden cover and overlying Seigal Volcanics in part (Figure 7.3.1). RT, JG, HB and LP, collectively termed the RT Group, which have undergone the most exploration, have Mineral Resource Estimates and are at resource drilling stage. They are described in the following sub-sections. A schematic cross section showing the geological relationship between the deposits is shown in Figure 7.3.2.

Figure 7.3.1 Deposit Geology of the Westmoreland Uranium Field



Redrawn after RTZ (1997) and Cuney (2009).

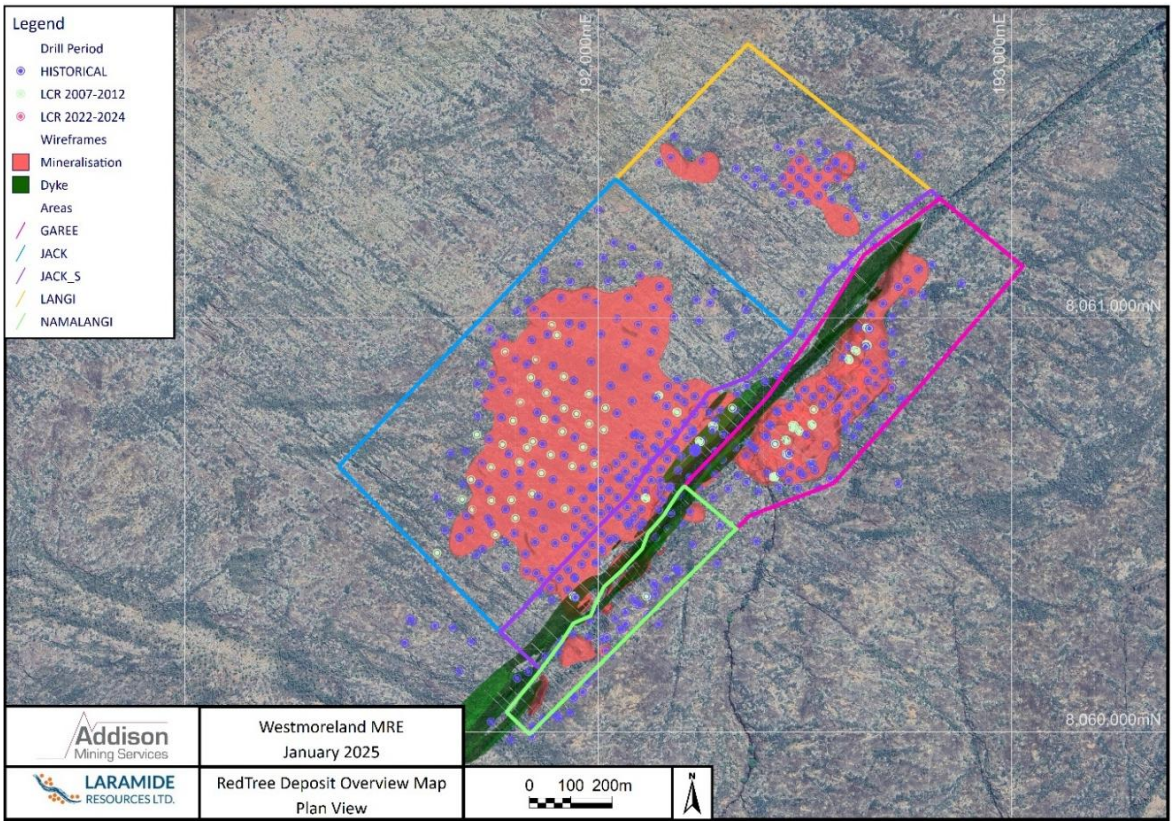
Figure 7.3.2 Uranium Mineralization Styles at Westmoreland after Rheinberger et al. (1998)



7.3.1 Redtree

The RT uranium deposit flanks the RT dyke zone immediately north of the NW-trending Namalangi fault. The deposit is made up of four mineralized lenses (Figure 7.3.1 and 7.3.2) - horizontal mineralization in the Jack, Garee, and Langi lenses, and vertical mineralization in the Namalangi lens. The Jack Lens becomes locally steep close to the dyke edge and there is minor mineralization in the dyke. Grades range from 0.15% to >2% U_3O_8 (Rheinberger et al., 1998).

Figure 7.3.3 Redtree Showing Four Major Lenses of Mineralization



Mineralization and alteration contacts are mostly bedding parallel and gently dipping. The central deposit area lacks significant steep or dyke-related mineralization. Shallow, flat-lying mineralization is often linked to a continuous pebble conglomerate layer, including the Jack and upper Garee Lenses (Figure 7.3.3), associated with moderate to strong hematite alteration and lesser chlorite / sericite. Deeper mineralization, linked to more variable stratigraphy, also shows higher grades in coarser sandstone / pebble conglomerate units and is commonly associated with chlorite-altered sediments with less hematite / sericite.

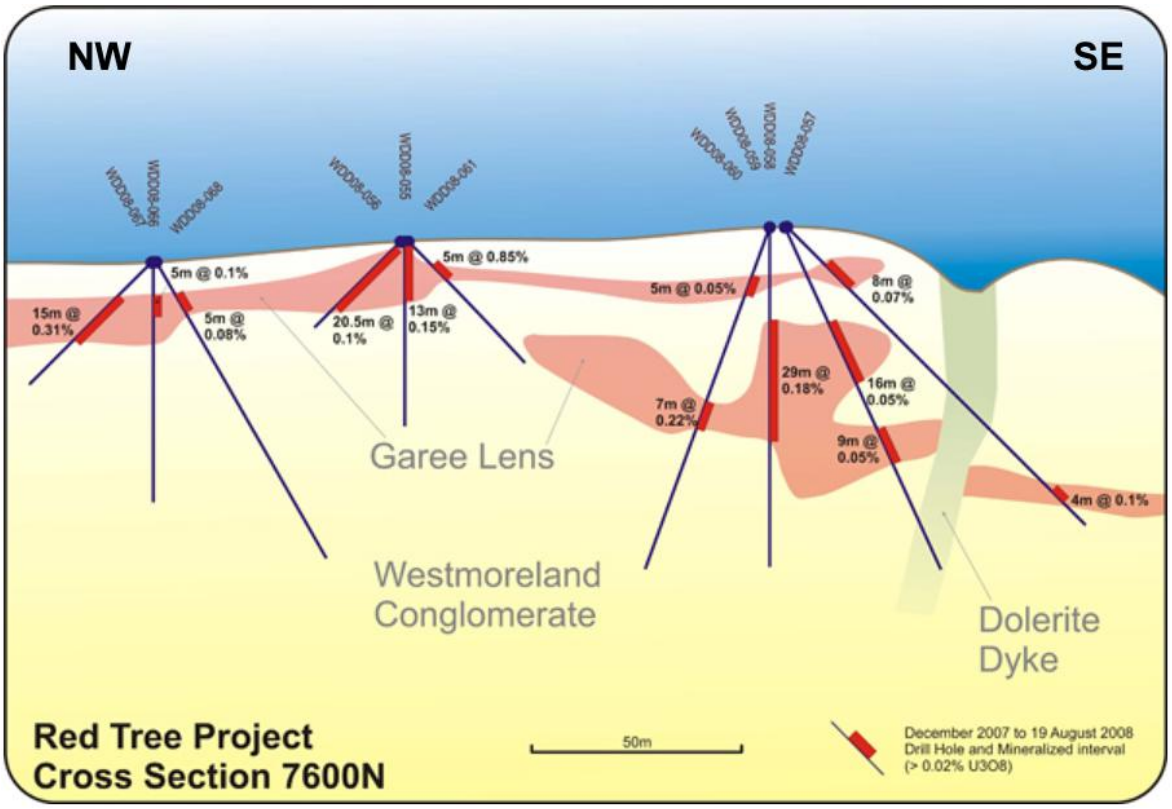
The horizontal mineralization in Jack and Langi lenses on the NW side of the dyke zone is entirely hosted within PTW4 of the Westmoreland Conglomerate. It forms a sheet of mineralization 0 to 10 m below ground surface (mBGS) (less than 20 m below the projected basal contact of the now removed Seigal Volcanics) up to 15 m thick (increasing with proximity to the dyke zone) and up to 500 m wide.

The Garee Lens consists of a mix of horizontal and vertical mineralization in the PTW4 of the Westmoreland Conglomerate on the eastern side of the dyke zone. Mineralization is 5 to 30 mBGS, up to 50 m thick adjacent to the dyke and thins to the east (away from the dyke).

Vertical mineralization at the Namalangi Lens occurs over a strike length of more than 700 m within the dyke zone, particularly within the sandstone wedge between the two dykes.

Scanning electron microscope (SEM) analysis shows consistent uranium mineralogy across drill-intersected lenses. Uraninite dominates (60 to 80%), with coffinite (2 to 15%) and meta-autunite (1 to 6%) as the main remainder. Coffinite was found to be proportionally more abundant in two samples in lower Garee Lens (40% and 32%). Brannerite was absent or present in trace amounts (<1% of uranium).

Figure 7.3.4 Redtree Cross-Section 7600 N



Source: Laramide.

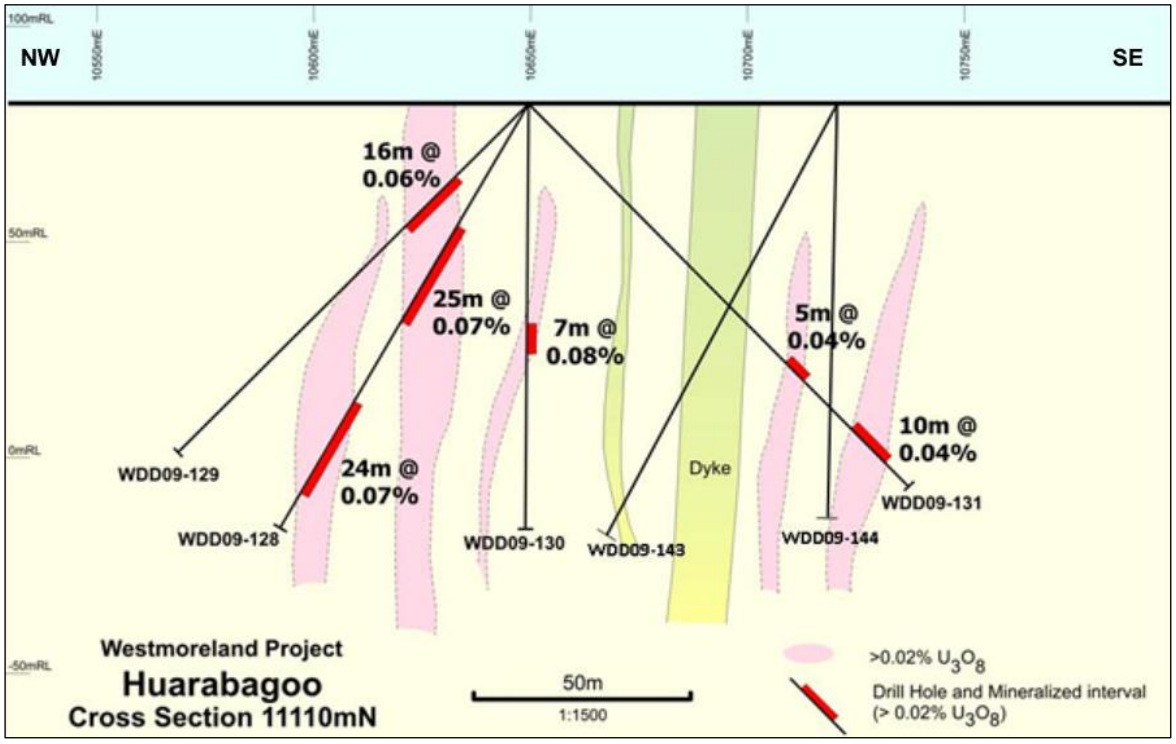
Drilling at RT intersected primarily the upper unit of the Westmoreland Conglomerate (PTW4). Lithologies intersected within this unit were predominantly coarse quartz arenites with intervals grading into pebble conglomerate. These lithologies are underlain by coarser cobble conglomerates at depth.

7.3.2 Huarabagoo

Situated 3 km NE of RT along the RT dyke zone, HB straddles the contact of the Seigal Volcanics with the Westmoreland Conglomerate (Figure 7.3.1).

The mineralization outcrops at the southern end and is concealed to the north under 2 to 3 m of sandy alluvium, and 5 to 8 m of weathered basalt of the Seigal Volcanics. The deposit comprises a 3 km zone of vertical mineralization associated with a complex dyke geometry with vertical and horizontal branches between the two principal dykes (Figure 7.3.5). Some 75% of the mineralization is within the flanking PTW4 sandstone (the remainder in the dykes) with individual lenses up to 20 m thick, 100 to 500 m long, and extending to a depth of about 80 m. Mineralization rarely extends beyond the PTW4 into PTW3 conglomerate.

Figure 7.3.5 Huarabagoo Cross-Section 11110 N



Source: Laramide.

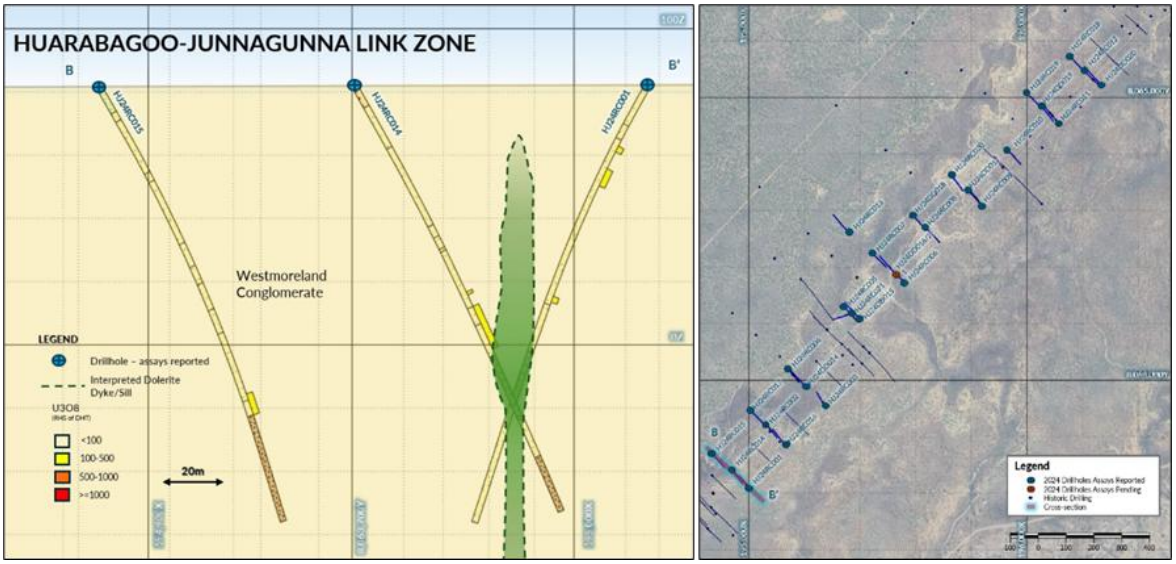
7.3.3 Huarabagoo-Junnagunna Link Mineralization

Mineralization in this zone is mainly hosted within the PTW4 unit of the Westmoreland Conglomerate, peripheral to the RT dyke system (Figure 7.3.6), and typically associated with strong hematite alteration.

The HB-JG Link is a mineralized structural corridor along strike of the RT dyke system, connecting the HB and JG deposits (Figure 7.3.6). Mineralization identified to date in this zone is predominantly hosted within the PTW4 unit of the Westmoreland Conglomerate peripheral to the RT dyke system, similar to the other Westmoreland uranium deposits, and typically associated with strong hematite alteration. Overlain by alluvial cover, it lacks surface radiometric response.

The Westmoreland Conglomerate is buried under 2 to 10 m of sandy alluvium and 5 to 15 m of weathered Seigal Volcanics. The PTW3 unit is found at 100 to 110 m depth. Drilling targeting dyke continuity intersected the RT dyke but identified offsets, suggesting cross-faulting. Further drilling is expected to improve definition of this zone and associated vertical mineralization shadowing the intrusive body, and test continuity between the deposits to potentially increase the overall resource size.

Figure 7.3.6 2024 Drilling at the Huarabagoo-Junnagunna Link

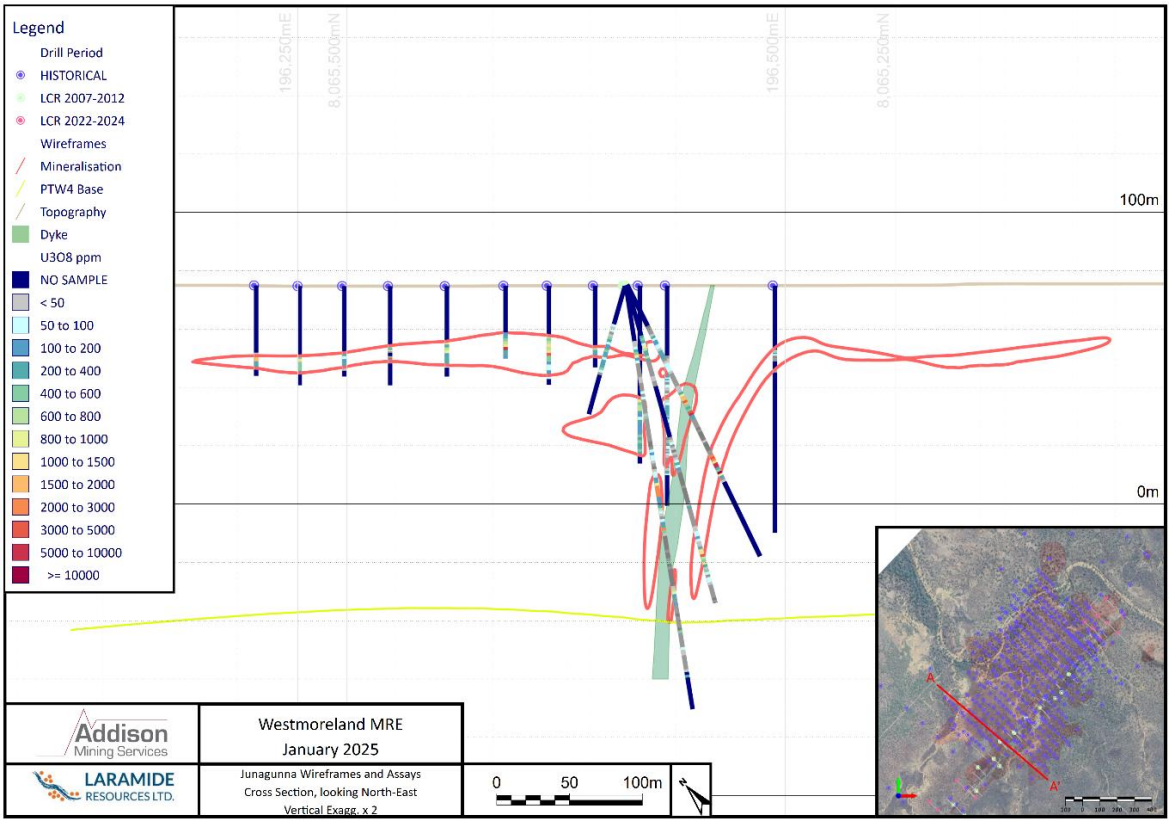


7.3.4 Junnagunna

At JG, the Westmoreland Conglomerate is beneath Seigal Volcanics, which are overlain by ~8 m of quaternary alluvium / colluvium. Weathered basalt extends to 10 to 25 m depth. The JG Conglomerate lacks the coarse pebble units seen at RT, with the upper sequence dominated by medium to coarse sandstone, underlain by a coarse sandstone with scattered pebbly clasts. The JG uranium deposit is located at a fault intersection west of the RT dyke zone and south of the Cliffdale fault, obscured by 3 to 10 m of alluvial sand and 5 to 20 m of weathered / fresh Seigal Volcanics. Extensive flat-lying mineralization in PTW4 sandstone occurs on both sides of the RT dyke, typically 0.5 to 10 m thick, just below the Seigal-Westmoreland contact. Mineralization becomes steep and rolls over approximately 20 m form the dyke contact (Figure 7.3.7).

Strongest mineralization is linked to chlorite and hematite-altered coarse pebbly sandstones, similar to RT. Highly silicified and fractured rock near the dyke is generally poorly mineralized. While alteration and mineralization contacts show a bedding parallel aspect, the overall mineralization geometry indicates vertical control. Higher grades likely occur within favorable horizons but are laterally confined by a broad structural corridor.

Figure 7.3.7 Example Cross-Section Showing Geometry of Junnagunna Mineralization

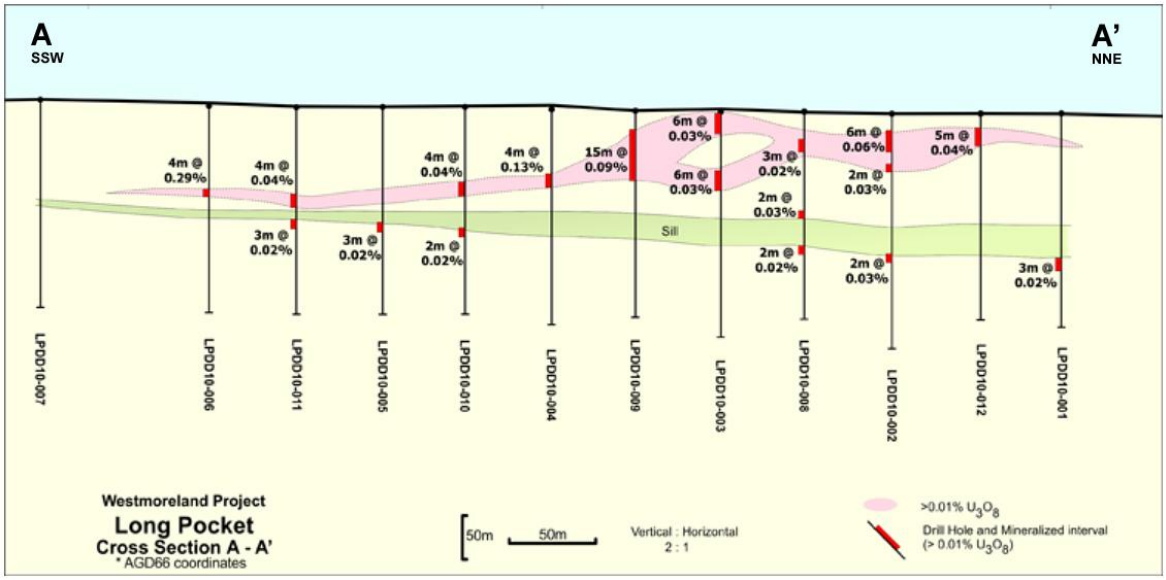


7.3.5 Long Pocket

Uranium mineralization occurs on the northern side of the Cliffdale fault and the eastern side of the RT dolerite dyke zone. The LP deposits (Outcamp, Sue, and Black Hills) are situated 8 km east of the JG deposit.

LP mineralization is sub-horizontal over a 500 m strike length to the E-NE above a dolerite sill and immediately below the underlying sill contact (Figure 7.3.8). Zones of higher-grade uranium are generally intercepted within the broader coherent mineralized envelope above the dolerite sill. Outcamp and Sue were historically viewed as separate zones but by 2020, Laramide viewed Outcamp and Sue as a single contiguous zone.

Figure 7.3.8 Long Pocket Schematic Cross-Section



7.3.6 Oxidization and Weathering

Determining oxidation and weathering at RT is challenging due to the sandstone's resistance. Competent, partly silicified sandstone is at the surface. Oxidation occurred early diagenetically. Clays are diagenetic, not weathering products. Weathering is assessed by chlorite and hematite alteration, making a consistent weathering front definition difficult. Sulfur analyses provide a more quantitative oxidation measure, with low sulfur correlating to the upper deposit. The base of oxidation is defined by logging and sulfur data.

Slight weathering (chlorite to smectite, minor oxidation) is common but doesn't affect rock competency or density. The upper RT deposit (east of the dyke) contains primary uranium minerals (uraninite, coffinite), not secondary ones.

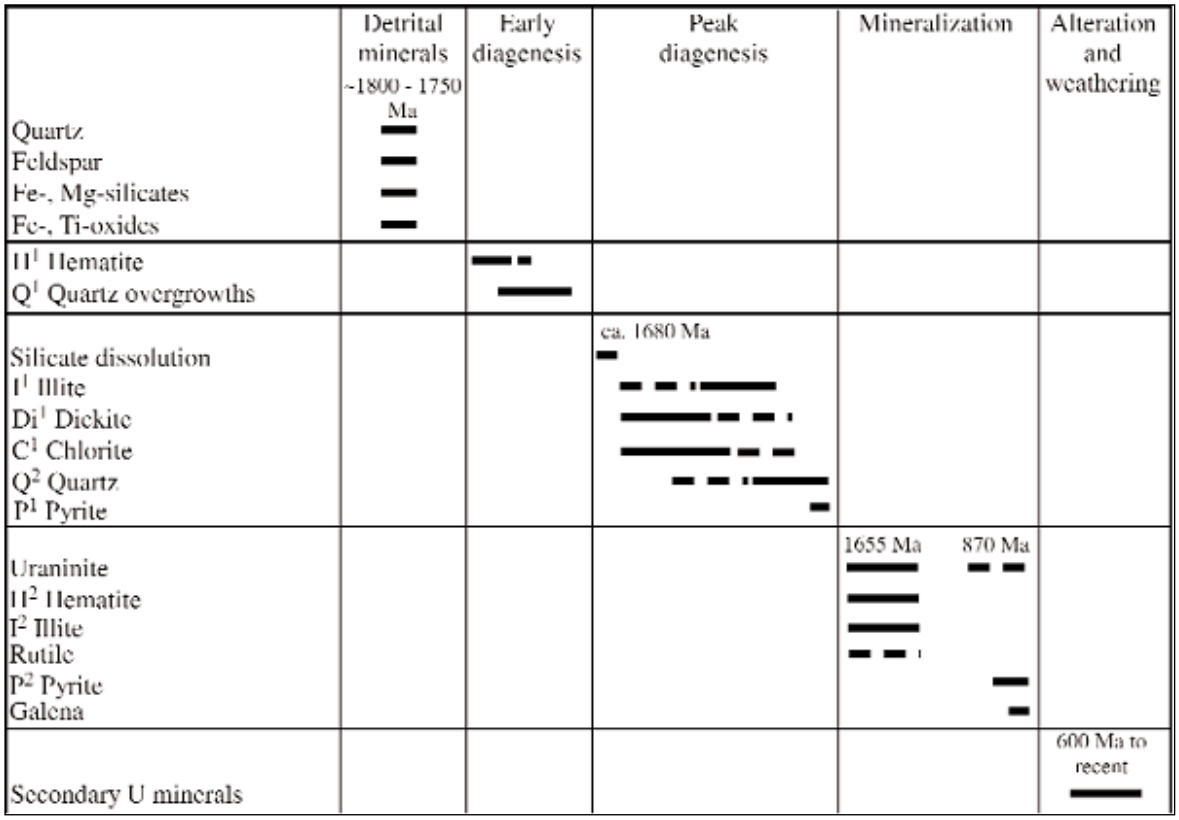
The base of oxidation, based on sulfur assays, ranges from 0 to 26 m, but is typically 5 to 15 m below ground. At JG, the upper deposit is extremely weathered alluvium / colluvium over moderately to strongly weathered basalt. Alluvial thickness is 6 to 23 m (generally 8 to 15 m), and the basalt base is usually 15 to 28 m. Underlying sandstones are slightly weathered to fresh, but oxidation state determination is complex.

7.3.7 Uranium Mineral and Alteration Mineralogy

The uranium mineralization is characterized by late-phase uraninite, hematite, illite, and minor rutile. Uraninite and hematite act as matrix cement between quartz grains. Uraninite also occurs as micron-sized grains within hematite (Polito et al., 2005) Hematite is dominant, imparting a red-brown color to samples. Some uraninite fills fractures in pyrite. Pyrite appears contemporaneous with some uraninite but also occurs as brecciated grains cemented by uraninite.

Secondary uranium minerals found at RT, HB, JG and LP include torbernite, met-torbernite, carnotite, coffinite, autinite, bassetite, and ningyoite.

Figure 7.3.9 Paragenesis in the Westmoreland Uranium Field, After Polito et al (2005)



The PTW4 subunit of the Westmoreland Conglomerate contains most of the identified uranium at Westmoreland. It is porous, coarse grained quartz sandstone, with cross-bedding and conglomerate portions. It is brown colored in outcrop and white to pale grey when fresh. Within the deposit area, it is about 80 m thick with a basal discontinuous tuffaceous fine grained laminated siltstone.

At RT, while primary uraninite exists, the ore is mainly supergene altered to silicate (sklodowskite, boltwoodite, beta-uranophane) and minor phosphate (saleeite, autunite, tobernite) minerals. Shallow mineralization is linked to moderate to strong hematite alteration with less chlorite and sericite.

HB shows zones of strong silicification by secondary quartz. Late-stage limonitic oxides are common with clays on joint surfaces. Dusty hematite is widespread in the matrix, with stronger, near-opaque hematite alteration associated with mineralization. Sericite selectively fills fine-grained clays, likely replacing feldspar.

JG's strongest mineralization is in chlorite and hematite-altered coarse pebbly sandstones, similar to RT. Highly silicified and fractured rock near the dyke is generally poorly mineralized.

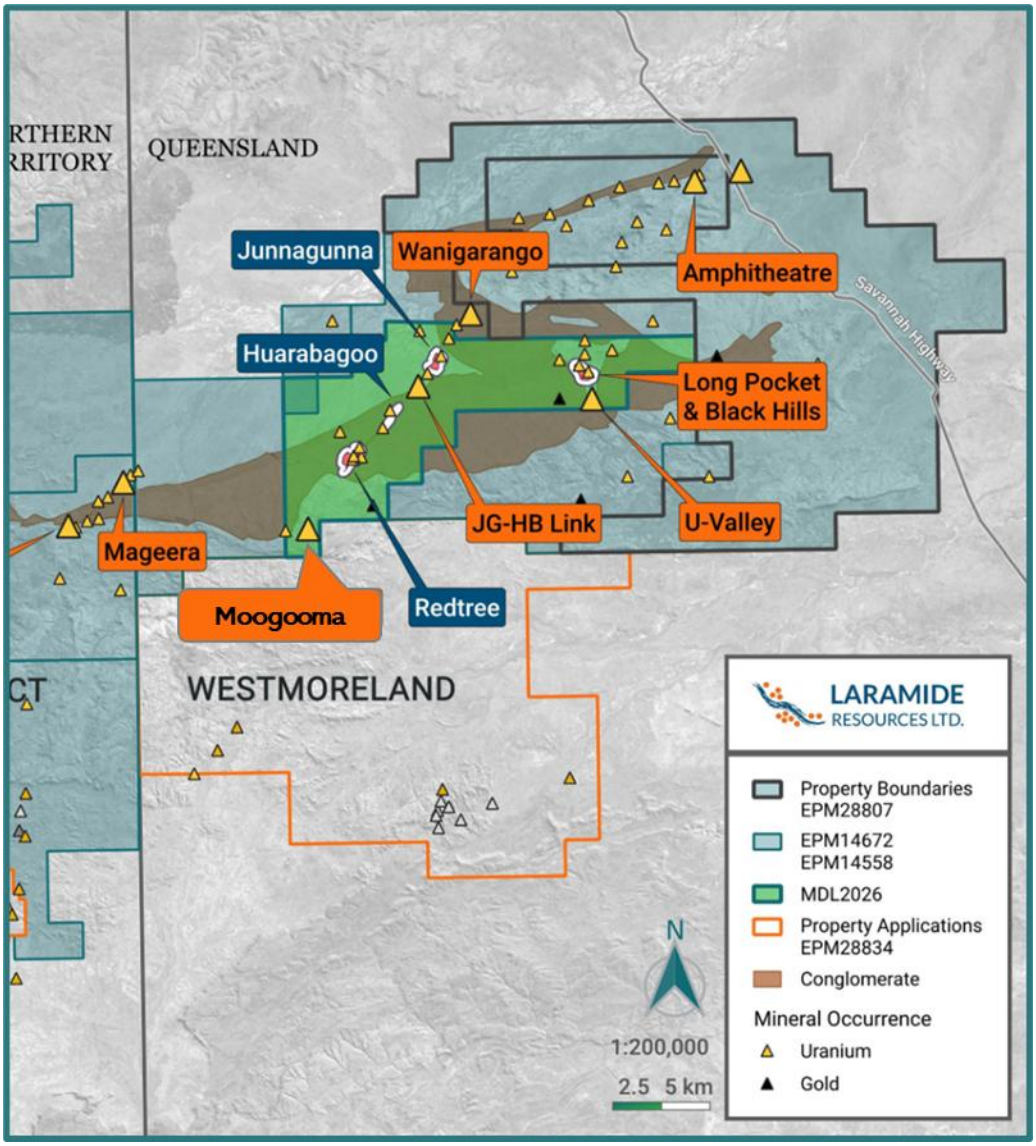
LP's PTW4 is a moderately sorted, quartz-cemented sub-litharenite (as classified by Barren (2008)), often pale pinkish-brown due to hematite staining. Alteration is generally mild sericite-hematite-rutile, with strongest mineralization associated with strong hematite alteration.

7.4 Other Mineralized Areas on the Property

In addition to the primary identified deposits of RT, HB, JG and LP, 13 other zones of known mineralization have been identified on the property and are illustrated in Figure 7.4.1. No MREs have been completed for these prospects.

The Moogooma, Black Hills and Uranium Valley, Amphitheatre, and Eagle prospects are deemed most significant and warrant immediate further exploration, as described below.

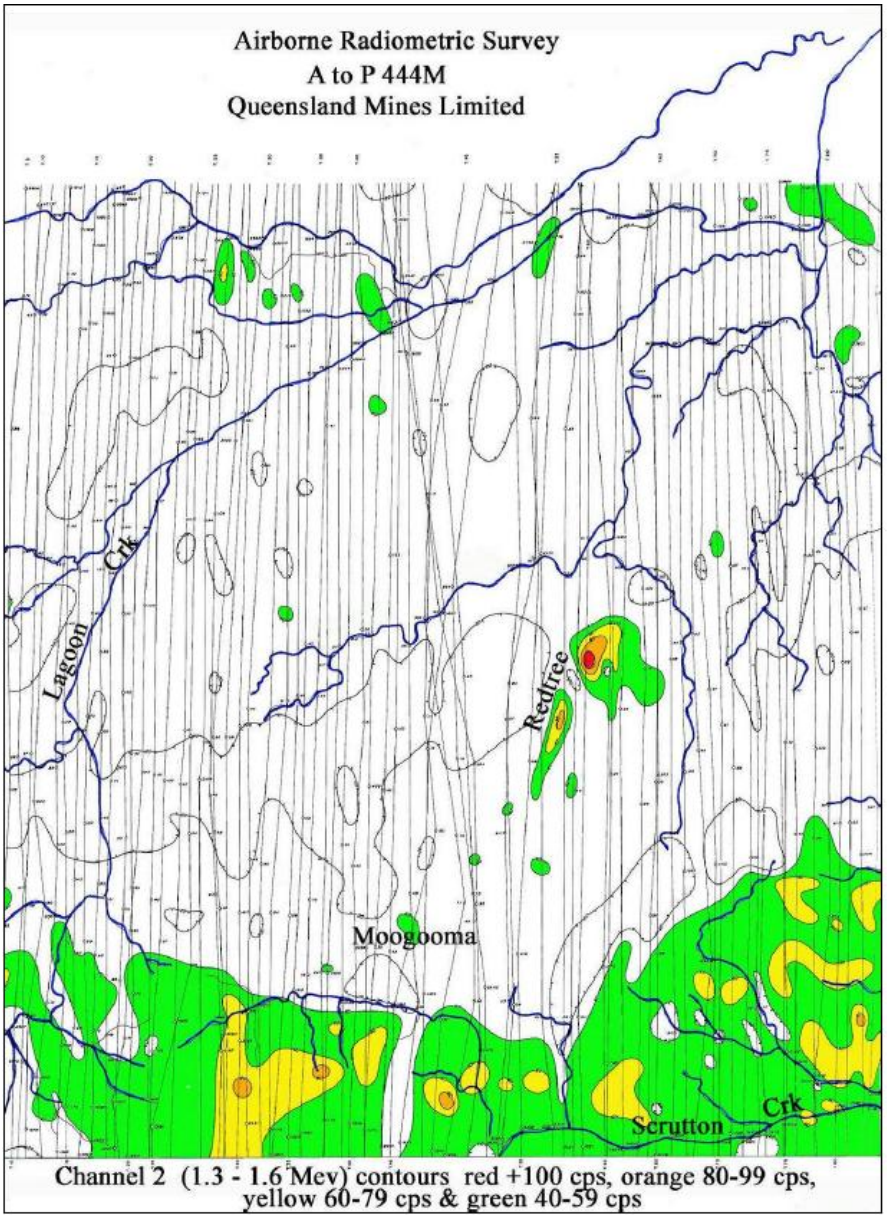
Figure 7.4.1 Map of Westmoreland Deposits and Prospects



7.4.1 Moogooma

Moogooma is a radiometric anomaly located approximately 5 km SW of RT along strike of the RT dyke (Figure 7.4.1). Exploration in 1973 conducted by QML revealed a radiometric anomaly to the SW of Moogooma in the Cliffdale Volcanics, reporting over 1,200 cps in a total count aerial survey, and a uranium anomaly of 40 cps (Figure 7.4.2). The QML radiometrics show the main U anomalies (except JG), and the high background of the Cliffdale Volcanics. It has been postulated that the Channel 2 U anomaly along structure from RT-Moogooma might have primary uranium implications.

Figure 7.4.2 EPM444 Redtree Area



Airborne Radiometric Survey, QML, Channel 2 Contours - Sheet 1 (after QML, 1974, CR5053).

Work by CRA Exploration Pty Ltd (CRAE) in 1996 noted the uranium mineralization at Moogooma was confined to a series of isolated hills, and represented the eroded remnants of a former extensive shallow dipping sheet of mineralization, similar to the one at RT. Mineralization was pertained relative to the Moogooma dyke hosted in the Westmoreland Conglomerate, reported to consist of interstitial, pebble coatings and fracture filling secondary U mineralization. Although the assay data recorded is limited with only modest notable intercepts (Table 10.3.1, Section 10) the data suggests that the mineralization is open to the west. Furthermore, the target has only been explored to shallow depths and has not been defined in a north-south direction.

Rossiter (1976) reported as a result of regional work conducted by BMR in 1979, that U-As-Au stream sediment geochemistry is an informative exploration strategy, showing strong U anomalies from known mineralization zones RT, HB, and LP, as well as Moogooma.

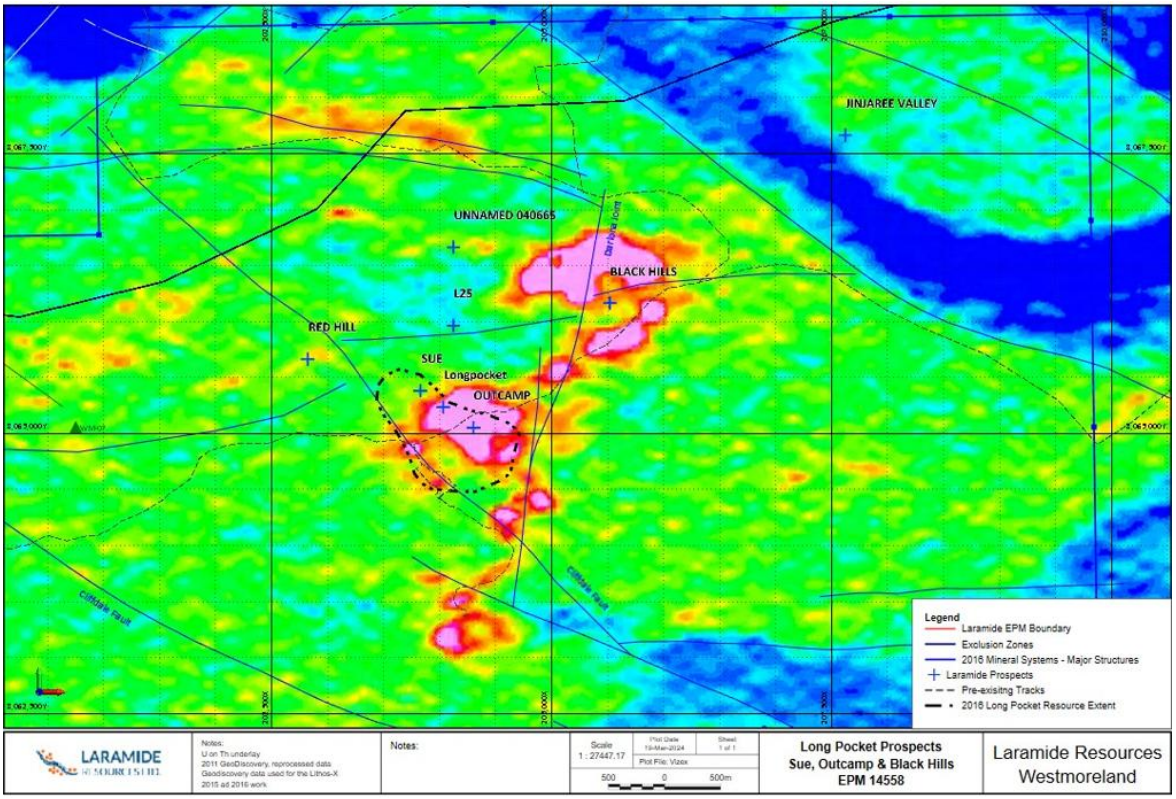
Arsenic is associated with uranium in all these areas. A weak gold association is noted at RT and HB, but not elsewhere. Mineralization is likely sourced from the Cliffdale Volcanics. The radiometric anomaly at Moogooma might represent a smaller, near-surface deposit with a strong radiometric signature due to erosion exposing remnants of a shallow, once extensive mineralized sheet.

Additional radiometrics conducted by LAM over QLD tenements identified several U-Th targets, with data analysis highlighting U over K. These U-emphasized targets correlate with the NE-trending RT dyke system and are also seen SW of RT at Moogooma. Anomalies exist along the Main Ridge Fault south of Moogooma, between the Westmoreland Conglomerate and Cliffdale Volcanics. Moogooma is a priority target for further Laramide exploration.

7.4.2 Black Hills and U-Valley

Black Hills and Uranium Valley (U-Valley) located 9 km east of HB and 12 km NE of RT, along with LP, form a 4 km x 2 km area of NE-trending U/Th radiometric anomalies (Figure 7.4.3). U-Valley, a significant radiometric anomaly SW of these prospects, shows strong Uranium and U/Th anomalism within PTW4 of the Westmoreland Conglomerate, linked to the NE-trending Darlona Joint structure off the Cliffdale Fault.

Figure 7.4.3 U on Th, GeoDiscovery Reprocessed Data (2011)



Classified as strong for both Uranium and Uranium of Thorium anomalism hosted in PTW4 associated with the Darlona Joint structure running to the NE. 2016 in house resource overlain.

Black Hills, 1.5 km NE of LP, is a broad 1.5 km x 1 km E-W radiometric anomaly (Figure 7.3.4), forming an E-W quartz ridge similar to one north of JG. The ridge is associated with Seigal Volcanics (Pts) and is located on the axis of the LP Valley syncline. The contact between PTW4 and Pts is inferred through alluvial cover in places of lower topography. The deposit is hosted by the PTW4 sandstone and is typically adjacent to the contact with the overlying Seigal Volcanics. Rock chip sampling shows anomalous U values and visible mineralization similar styles to other prospects in the Project area.

Historical drilling shows mineralization spatially related to the E-W Black Hills dyke, with varying associations to the dyke (footwall, hanging wall, distal). The results of the 2023 LAM scout drilling campaign (see Section 10.3.1), combined with a review of historical data from the 1970s, promote Black Hills to one of LAM's priority exploration targets which will include validation and qualification of historical work.

7.4.3 Long Pocket Gorge

South of LP is the 'LP Gorge', a surface expression of the Darlona joint extending SW, displaying magnetite and iron-rich lithology, possibly eroded intrusive sills. These exposures are limited to the gorge's SW edge, approximately 2 to 5m thick and visible for 20 m, often covered by boulder debris from the overlying PTW4. The Cliffdale Fault is evident south and west of LP, marked by quartz-filled veins. Uranium Valley, a valley in PTW4 south of where these structures diverge (1.3 km south of LP), likely formed due to Cliffdale Fault movement. Its southern edge is a small sandstone escarpment (10 m high) and a notable 0.7 km x 0.7 km radiometric anomaly (Figure 7.4.3).

Initial field work and pit sampling has suggested sandstone hosted mineralization with elevated U, seen on a spectrometer, and secondary mineralization in fractures. Due to its proximity to a known mineralized zone and limited data, Uranium Valley is considered an untested satellite prospect warranting further exploration.

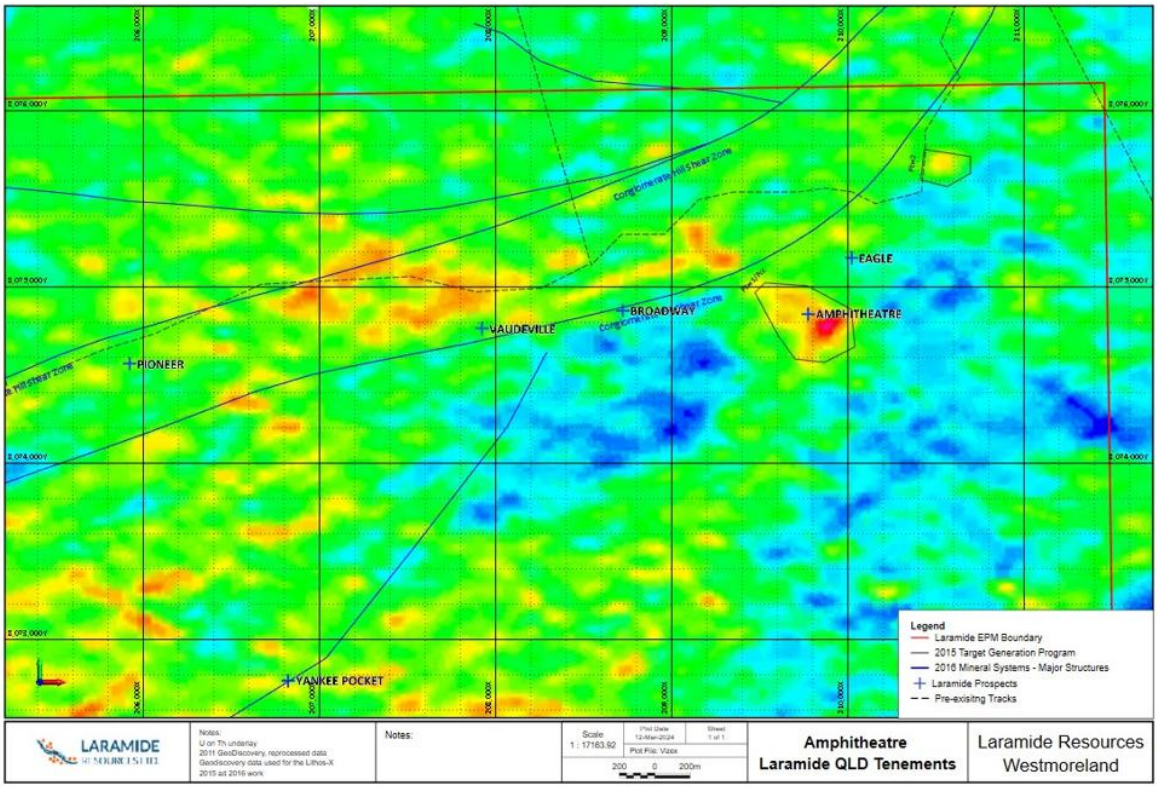
7.4.4 Amphitheatre

The Amphitheatre prospect is 11 km NE of LP, in the NE part of EPM14558, located within Cliffdale Volcanics and Westmoreland Conglomerate (primarily PTW1). It is bound to the east by an interpreted NS trending fault. This fault, covered to the north, may extend or be truncated by the Conglomerate Hill Shear Zone.

Geophysically, it's a 400 m x 300 m radiometric anomaly with weak to moderate U-U/Th signatures (Figure 7.4.4). Historical exploration (late 1960s / early 1970s) established a stratigraphic control, showing an eastern downthrow with no lateral movement (PTW2 expressing to the east and PTW1 to the west). Historical exploration in the late 1960s and early 1970s (percussion / diamond drilling, costeaning, pitting) established stratigraphic control showing an eastern downthrow with no lateral movement. Mineralization is interpreted as structurally and stratigraphically controlled, with extensive remobilization and secondary mineralization (carnotite, autunite, torbenite) west of the NS structure, potentially from down-dip solution flow. East of the fault, both primary and secondary mineralization were found, but no feeder dyke was identified as a driver.

Recent drilling (post-2022) intersected disseminated uraninite and torbernite associated with hematite alteration and silicification. Hole AMD009 intersected a potential feeder mafic dyke at ~190 m, considered a key driver for mineralization, similar to the Westmoreland deposit. Quartz veining with epithermal textures correlates with Au assay results.

Figure 7.4.4 U on Thorium, GeoDiscovery Reprocessed Data (2011)



Lithos-x target generation areas identifying Amphitheatre as a moderate priority drill target in 2015 due to moderate U on Th signature of the Amphitheatre prospect in proximity to the Conglomerate Hill.

7.4.5 Eagle

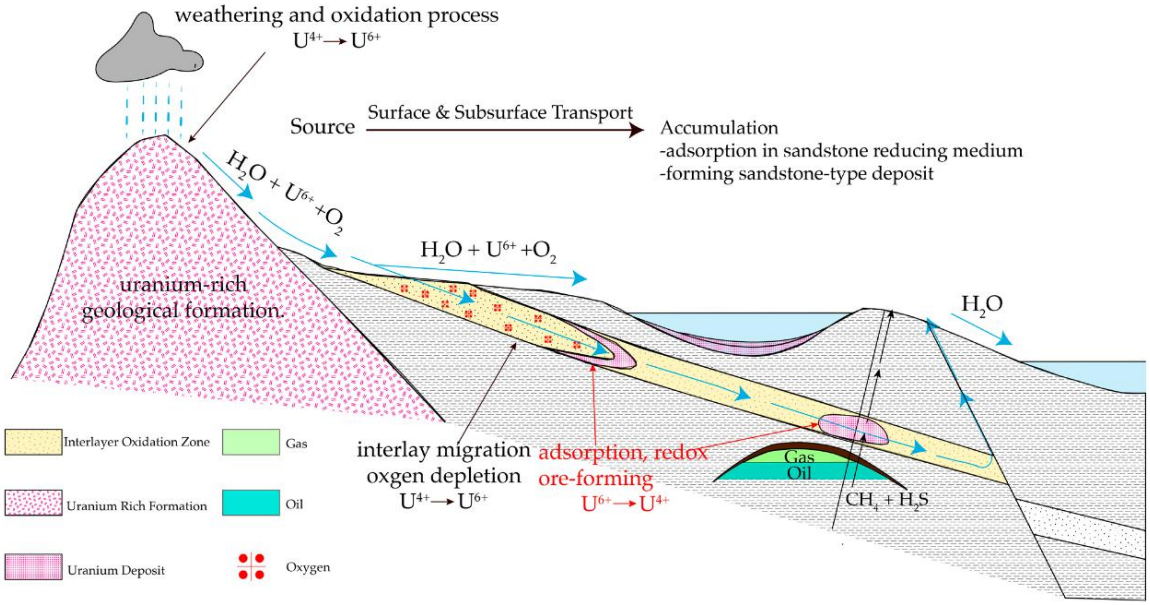
Eagle is the most northeasterly target within EPM14558. The geology is similar to Amphitheatre with gently north-dipping bedding and veins along NS and EW joint sets. Hematite alteration is present, sometimes strata-bound rather than constrained to veins and joints. Anomalous mineralization has been identified in historic exploration work, including drilling targeting northward fault / shear continuations which intersected uraninite, confirming northward continuation of mineralization. As a result, the Eagle prospect is a priority target and warrants further exploration.

Mineralization throughout the Westmoreland region is generally recognized stratigraphically within the PTW4 unit of the Westmoreland Conglomerate; however, with mineralization confirmed at the contact of PTW1 and PTW2 at both Amphitheatre and Eagle prospects, it demonstrates potential for mineralization to be stratigraphically unconstrained (and likely structurally controlled in close association with apparent feeder mafic dyke systems), presenting opportunities for improved exploration strategies and targeting across a vast Westmoreland Region.

8.0 DEPOSIT TYPES

Uranium in sandstone-hosted deposits, like the Westmoreland deposits in Australia, is typically precipitated through a process involving the interaction of uranium-rich oxidizing fluids with reducing environments within the sandstone (Figure 8.1). Several deposit controls are discussed in the below section after Guihe et al. (2023), with detailed particularities of the Westmoreland deposits themselves detailed in previous sections.

Figure 8.1 Graphical Abstract of Metallogenic Mechanisms of Sandstone-Type Uranium Deposits



Guihe et al, 2023.

8.1 Host Lithologies

These deposits occur in medium to coarse-grained, poorly sorted sandstones, often of fluvial (continental) or marginal-marine origin. A key characteristic is the presence of pyrite (FeS_2) and organic matter (plant-derived), which can be disseminated within the sandstone matrix or concentrated in lignite seams.

8.2 Redox Control on Precipitation

Uranium (U^{6+}) is mobile under oxidizing conditions as uranyl complexes. Precipitation occurs when these fluids encounter a reducing environment, converting U^{6+} to less soluble U^{4+} phases (e.g. uraninite).

8.3 Sources of Reductants

Hydrogen sulphide (H_2S) is a highly effective reductant. It can be generated in situ via anaerobic microbial decomposition of organic matter, a process often associated with sulfate-reducing bacteria. Alternatively, H_2S can migrate into the host sandstone from underlying or overlying hydrocarbon reservoirs, creating a reducing front.

8.4 Temporal Favorability

Post-Silurian continental sandstones are considered more prospective due to the widespread terrestrialization of plants from the Silurian onwards, providing a significant source of organic matter for in-situ H_2S generation. Proterozoic deposits, such as those at Westmoreland, formed in the absence of significant terrestrial organic matter, relying instead on the presence of abundant ferrous iron (Fe^{2+}) within the host rocks as the primary reductant.

8.5 Hydrogeological Setting

Gently dipping sandstone beds, typical of continental basin margins and coastal plains, are favored. This geometry promotes slower groundwater flow rates, preventing the destruction of reducing conditions by oxygenated water. Low dips also maximize the surface area for the interaction of uranium-bearing groundwater with the reducing environment.

8.6 Deposit Classifications

Sandstone uranium deposits are broadly categorized based on orebody geometry, depositional setting, and structural controls. These include:

- Tabular: Relatively flat, tabular orebodies often controlled by stratigraphic boundaries.
- Roll-front: Crescent-shaped orebodies formed at the interface between oxidizing and reducing groundwater regimes.
- Tectonic-Lithologic (Shear-Hosted): Occur along permeable fault zones that intersect sandstone-mudstone sequences. Mineralization forms tongue-shaped zones within permeable sandstone layers adjacent to the fault, often with multiple vertically stacked zones. The Westmoreland deposits are examples of this type.

8.7 Resource Characteristics

Sandstone-hosted deposits represent a significant portion of global uranium resources, although individual deposits are often of low to medium grade (typically 0.05 to 0.4% U_3O_8). Uranium provinces or basins can host numerous small to medium-sized deposits, with some reaching up to 50,000 t of U_3O_8.

The cumulative uranium tonnage within a province (e.g. the Colorado Plateau) can be substantial, reaching several hundred thousand tonnes.

9.0 EXPLORATION

9.1 Significant Historic Exploration

9.1.1 1967 – Queensland Mines Ltd (QML)

QML carried out pitting and shaft sinking at the RT prospect during 1967 to 1969, and indicated continuous primary uranium mineralization between minimum depths of 15 m and maximum depths of 135 m extending for at least 4,800 m along a major joint system.

At the same time, BHP carried out an airborne radiometric survey of 1,224 km line cutting across the strike of the Westmoreland Conglomerate. Minor anomalies were recorded.

9.1.2 1960 to 1980

The ownership of the licence area changed hands multiple times during this period, and the respective exploration programs are summarized in Table 9.1.1.

Table 9.1.1 Summary of Exploration Activities - 1960 to 1980

Year(s)	Company	Exploration Work Program
1967 to 1973	BHP	Airbourne radiometrics
1968 to 1970	US Steel International	Stream sampling for base metals around the Gulf of Carpentaria, as part of a manganese-uranium search
1970	Westmoreland Minerals Limited	Field inspection of base metal anomalies in Hedley's Creek
1973 to 1979	Mt Arthur Molybdenum NL	Reconnaissance radiometrics, including 170 km of Track Etch lines
1975 to 1981	Savage Exploration Pty Ltd	Soil geochemistry, airborne radiometrics
1977 to 1979	Mines Administration Pty Ltd	Stream sediment geochemistry and ground radiometrics for uranium, tin and tungsten

9.1.3 1980 to 1990

An upsurge in gold exploration during this period is reflected in the increase in exploration work, with multiple companies actively exploring the area, now covered by EPM14558. Table 9.1.2 summarizes the operator and exploration program details.

Table 9.1.2 Summary of Exploration Activities - 1980 to 1990

Year(s)	Company	Exploration Work Program
1980 to 1982	Minatome Australia Pty Ltd	Ground geophysics, costeans
1982 to 1989	Uranerz Australia Pty Ltd	BLEG sampling for gold and ground geophysics
1983 to 1984	Total Mining Australia Pty Ltd	Ground geophysics (including Track Etch) for uranium in the Westmoreland area
1983 to 1989	Central Electricity Generating Board Exploration (Australia) Pty Ltd	BLEG sampling for gold and soil gas sampling for radon
1984 to 1985	International Mining Corporation NL	Stream sediment sampling for gold, diamonds, uranium, and base metals
1987	CSR Ltd	BLEG and rock chip sampling for epithermal gold in the Cliffdale Volcanics
1988 to 1989	Golden Plateau NL	BLEG and rock chip sampling for gold

9.2 Laramide Exploration Programs

LAM has held prospective tenure in the Westmoreland District since 2005. The exploration targeting strategy is informed heavily by geophysical (particularly radiometric) anomalies and anomalous geochemistry in historic drilling and surface geochemistry.

During this period, the Company has conducted significant exploration activities ranging from regional geophysical surveys and soil geochemistry sampling to prospect scale geophysics, and drilling (discussed separately in Section 10.1.6). Extensive geophysical surveys including regional FALCON gravity, magnetic, and radiometric surveys have been undertaken.

Table 9.2.1 Summary of Exploration Work Carried Out by Laramide Since 2005

Period	Description of Work Completed
2005 to 2006	Detailed airborne magnetic and radiometric survey was flown over the Westmoreland area by UTS Geophysics Pty Ltd. Compilation of historical drillhole and geological data from open file reports. Acquisition and processing of relevant Landsat and SPOT satellite imagery. Reprocessing of existing open file geophysical data. A geological review was undertaken by Mining Associates Pty Ltd. Application was made to Carpentaria Land Council Aboriginal Corporation (CLCAC) for clearance to conduct initial reconnaissance on the ground. A Westmoreland Project Desktop Baseline study was undertaken.
2006 to 2007	Initial sacred site clearance took place alongside the CLCAC enabling a soil sampling and geological mapping program to commence in August 2007. Independent resource estimate was undertaken by Mining Associates Pty Ltd. A desktop conceptual study for mining and processing completed by GRD Minproc.

Period	Description of Work Completed
2007 to 2008	Soil sampling, geological mapping of the JG and HB Prospects. Petrological examination of a selection of rock samples.
2008 to 2009	Resource estimate update by Mining Associates, incorporating the results of recent drilling. A total resource (Indicated and Inferred) of 27.7 Mt @ 0.09 % U_3O_8 for 51.9 Mlb contained U_3O_8. A regional stream sediment sampling program over the project area commenced. SEM analysis on assay pulps by SGS Minerals Services. Additional petrography and hyperspectral portable infrared mineral analyzer (PIMA) analysis of selected drill core. Radiological surveys were undertaken comprising ground gamma surveys.
2009 to 2010	Continued regional stream sediment sampling program. A total of 142 samples were collected from 71 sample sites. Ground scintillometer surveys and geological mapping on radiometric anomalies in the LP area. Quantitative Evaluation of Materials by Scanning Electron Microscopy (QEMSCAN) analysis of samples undertaken by SGS. Review of project metallurgy and feasibility study (FS).
2010 to 2011	Soil sampling program with ground scintillometer points over the HB–JG Prospects. A total of 1,050 soil samples were collected from 525 sample sites. Reprocessing of the 2005 Westmoreland Airborne Survey Data by GeoDiscovery Pty Ltd. ANSTO Minerals commissioned to undertake a metallurgical test and recovery program.
2011 to 2012	Ground radiometric surveys over two areas, Southern Valley and Southern Black Hills (LP area). Rock chip sampling and geological mapping. An updated conceptual mining study of the Westmoreland Project area.
2012 to 2013	Geological database review and targeting exercise of the main Westmoreland resource and surrounding area. Predictive modelling for the discovery of further sandstone hosted uranium mineralization as found at the Westmoreland uranium deposit.
2013 to 2014	Work on a SS and PEA was undertaken.
2014 to 2015	The Company commenced a project examining the Mineral Systems which relate to the deposition of uranium in the region. Development of this comprehensive model incorporates a detailed review of influencing factors on mineralized deposition and brings together the various aspects of structural architecture, sedimentology, fluid sources and pathways from regional to prospect scale. This geological modelling not only focussed on uranium indicators, but also investigated vectors in relation to other commodities. New information gathered as part of this modelling included an extensive short wave infrared (SWIR) spectral data collection and assay of significant data points from previously drilled core in and around the Westmoreland Project. Gold Occurrence Review. Work on a SS and PEA was undertaken incorporating an updated mine plan and costings for open-cut operation.

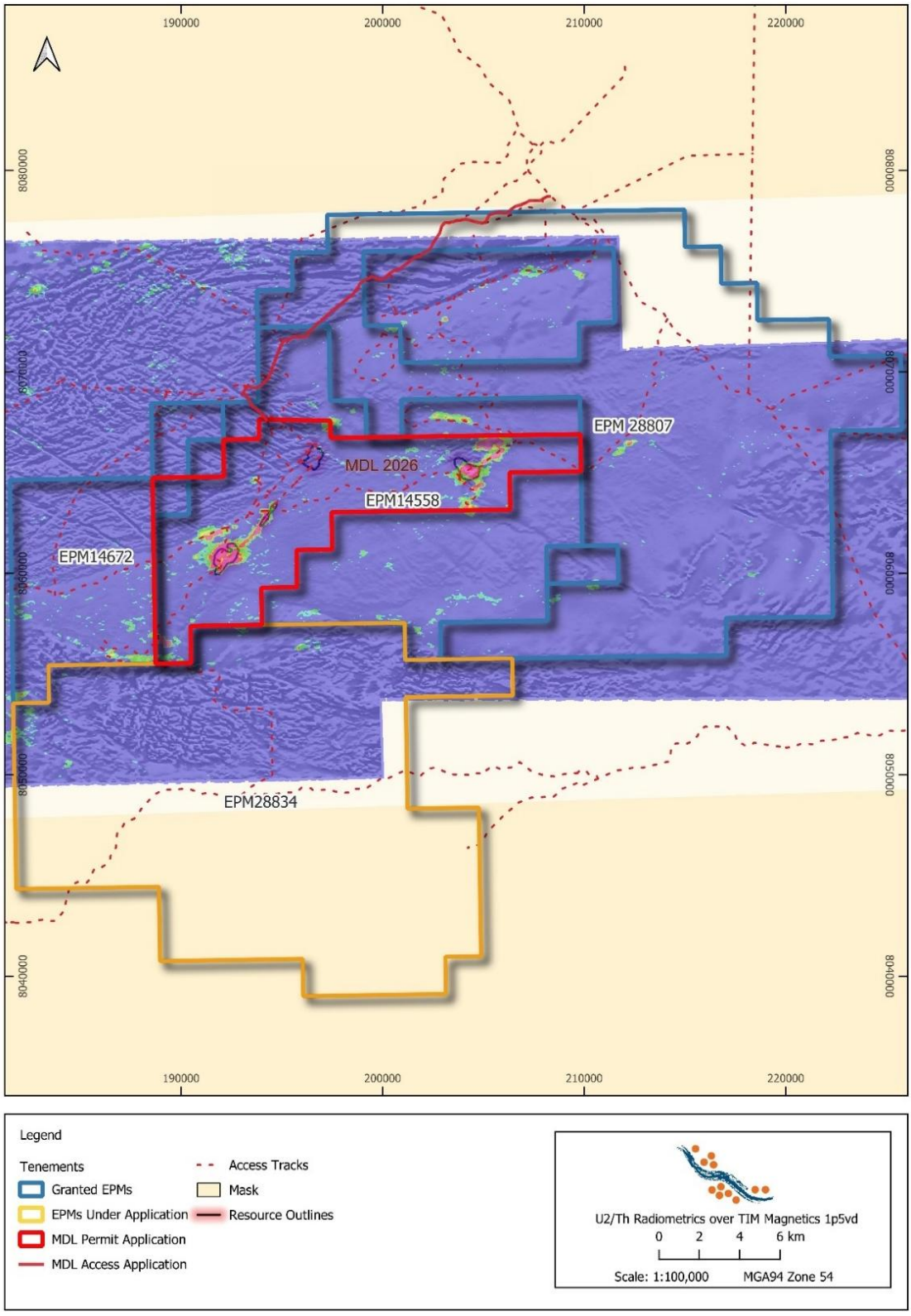
Period	Description of Work Completed
2015 to 2016	Continued work on the Mineral Systems project that commenced in the previous year. This project had the objective of expanding and refining the regional targeting model to expand the overall resource potential of the region. The Minerals System Review incorporated detailed solid geology and stratigraphical interpretations over the Westmoreland region, and a number of targets were identified for exploration away from the main resource area. Completion of the PEA on the Westmoreland Project.
2016 to 2017	SWIR spectral data collection and assay including a total of 4,207 spectra samples taken from 63 drillholes at Westmoreland.
2017 to 2018	3D magnetic inversion modelling study utilizing established detailed airborne magnetic survey data. The aim of the modelling was to closely examine the geological contact between the Seigal Volcanics and the underlying Westmoreland Conglomerate. The study could provide a way of estimating the depth to mineralization most likely located at the base of the Seigal Volcanics and indicate subtle structural offsets in the base of the Seigal Volcanics that may be related to mineralization at Westmoreland. Planning a FALCON Airborne Gravity Gradiometry, Magnetics and DTM Survey.
2018 to 2019	Completion of FALCON Airborne Gravity Gradiometry, Magnetics and DTM Survey. Follow up investigations included extensive geophysical modelling and 3D inversions of the aeromagnetic and airborne gravity data.
2019 to 2020	Several technical and targeting reviews were completed to assist with future exploration and development planning. These included a preliminary resource estimate for the 'LP' prospect, a vanadium potential review, and a Pb- isotope study.
2020 to 2021	172 soil and 20 rock chip samples with supplementary data acquisition, including a portable x-ray fluorescence analyzer (pXRF), an Exploranium G-110 scintillometer, and a Gamma Surveyor spectrometer. Furthermore, the previously drilled core stored at Camp Caroline was resampled for petrography, primary chemical assays, or Pb-isotopic characterization.
2021 to 2022	Interpretation of soil geochemistry results from the fieldwork conducted in the prior reporting year. Follow-up soil programs were proposed.
2022 to 2023	Field reconnaissance, earthworks at two prospects—Amphitheatre and LP.
2023 to 2024	Field reconnaissance at four prospects—HB, Amphitheatre, LP and Black Hills.

9.2.1 Airborne Geophysics

Radiometrics

Three airborne radiometric surveys have been completed concurrently with magnetics across the tenement areas. CRA completed a survey in 1988 using Austirex Internationals Pt Ltd, followed by LCR in 2005 (UTS contractor); followed by GSQ in 2007 (Fugro, Figure 9.2.1).

Figure 9.2.1 U^2/Th Radiometrics Overlaying Magnetics RTP-1VD with Mineral Occurrences and Resource Outlines



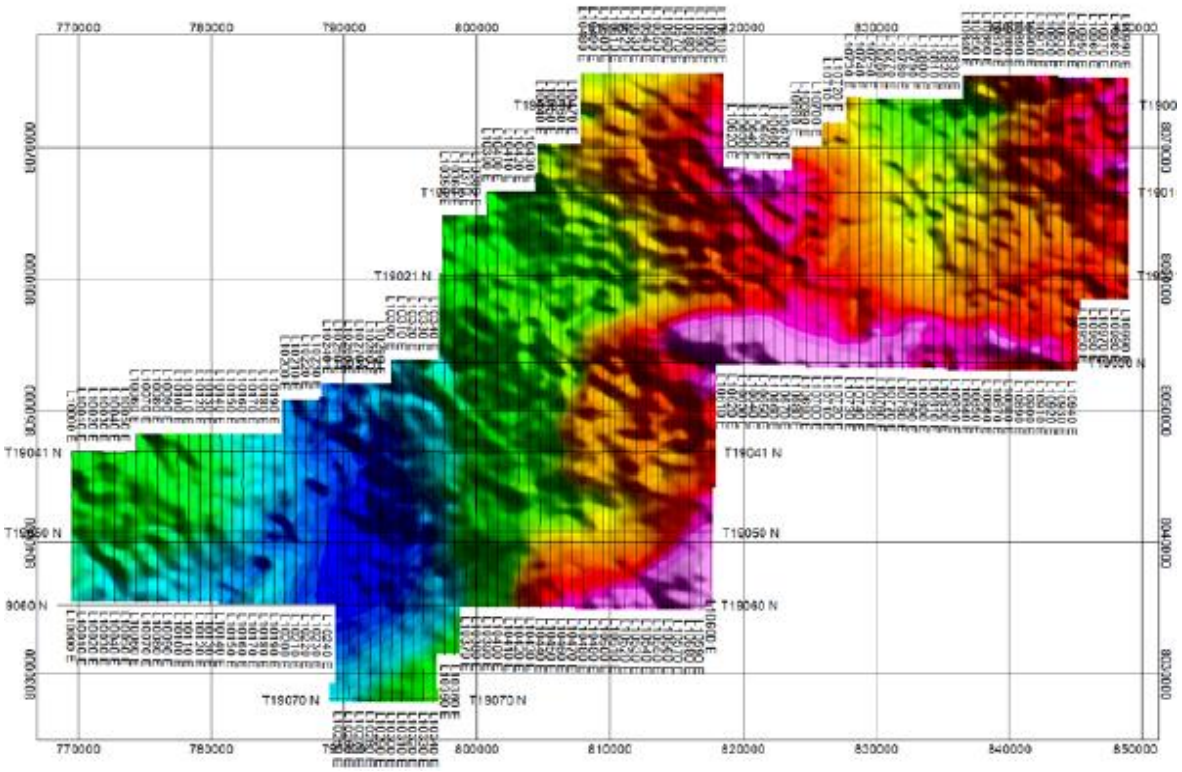
Gravity

Geoscience Australia released a South Nicholson Gravity Survey in 2016. The grid was derived from gravity observations stored in the Australian National Gravity Database. Grid spacing was approximately 800 m.

As part of the Company's Mineral Systems investigations, it was identified that structural positions have provided a conduit for mineralizing fluids into fault segments and a gravity survey would greatly assist with understanding the structure regime of this area. CGG Aviation Australia Pty Ltd was contracted to conduct the FALCON Airborne Gravity Gradiometry, Magnetics and DTM Survey in 2018 (Figure 9.2.2).

The survey area covered a total of 3,007 line kilometres of data acquired. Traverse line spacing was 800 m with a tie line spacing of 6,500 m.

Figure 9.2.2 Gravity Image (Unprocessed)



Magnetics

Four magnetic / radiometric surveys have been completed over the history of the tenement areas. An initial airborne magnetic survey was completed by CRA in 1988 by Austirex Internationals Pty Ltd, followed by LCR in 2005 (UTS contractor); followed by GSQ in 2007 (Fugro); and lastly a FALCON AGG system in 2018. A 3D inversion of TMI Magnetics was also completed covering the RT, HB and JG deposits (Figure 9.2.1).

9.2.2 Geochemical and Radiometric Surveys

Geochemical Surveys

LAM has acquired a large database of soil geochemistry samples during the company's tenure with a mix of conventional x-ray fluorescence (XRF) analyzed soils and laboratory analyses. Over 2,000 samples have been taken within the area proposed to be covered under the hyperspectral survey. LAM utilized the geochemistry analysis results to identify mineralized systems, validate geophysical anomalies, and for lithogeochemical analysis of host geology.

Selection of the appropriate analytical technique is dependent on the presence and thickness of alluvial cover overlying Proterozoic geology. Conventional soils, in areas of outcropping to sub cropping proterozoic lithology, samples were analyzed by LAM staff using pXRF. Where assay for Au following pXRF analysis is required, samples have been sent for fire assay.

Scintillometer Surveys

Ground scintillometer surveys have been captured across the tenure areas over the decades, with approximately 5,500 sites captured in recent years. The acquisition of this data both confirms and defines radiometric anomalies which assist with more detailed prospect and drillhole targeting.

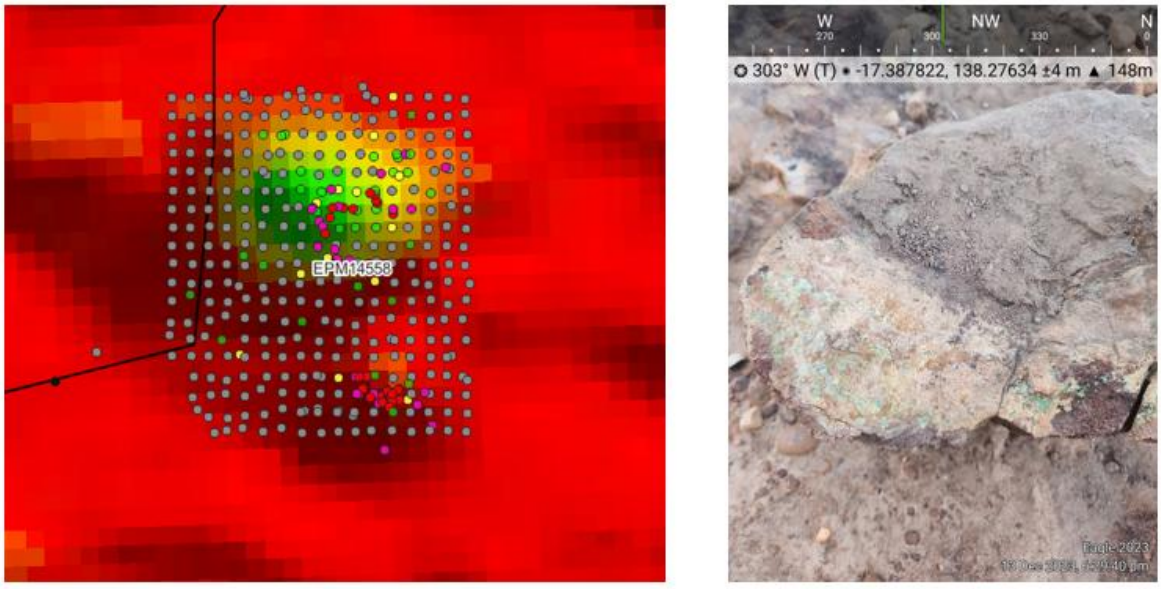
The most recent scintillometer grids were completed at Amphitheatre and Eagle prospects.

At Amphitheatre, two scintillometer grids were conducted in May and December in 2023. The grids were designed to test NS structures preferentially over the EW structures (which are regionally linked with dolerite dyke emplacement associated with the most economically significant uranium mineralization). Interpretations from the survey results delineate a radiometric high, skirting a unit expression in the SE of the Amphitheatre basin outcrop. Elevated readings also appear to correlate with northerly structures, in comparison to the EW trends. EW structures with elevated readings appear to be discrete and isolated.

A scintillometer grid was also completed at the Eagle Prospect. Bedding is observed as gently north dipping with veins associated with broadly perpendicular joint sets, orientated NS and EW. Hematite alteration was also observed strata bound. Elevated scintillometer readings (1,000 to 3,800 cps) in the NE of the grid, are found predominantly associated with a pervasively hematite altered conglomerate unit, roughly correlating with a U^2/Th high. Elevated readings (2,000 to 42,000 cps) to the SE of the grid correlate with a weak but distinct U peak observed in the U count radiometric geophysical data.

A sample from the highest reading was taken (42,000 cps), with visible torbernite (Figure 9.2.3). Elevated readings appeared to continue further to the south, and an extension of the grid is warranted.

Figure 9.2.3 Scintillometer Grids at Eagle Prospect



Left: Categorized field data over radiometric U; right: rock chips sample, recording 42,000 cps from SE cluster of elevated readings within Eagle Scintillometer grid.

9.2.3 Petrological and SEM Analysis

A petrological, mineragraphic examination and semi quantitative electron microscopic identification of selected encrusting uranium bearing secondary minerals was undertaken in 2008. The work was based upon a selection of rock samples and a selection of quarter core samples chosen from the 2007 Phase 1 drilling. Thin sections were prepared, and a report was compiled by an independent expert (Barren, 2008). Additional samples were selected in 2022 from LP drill core, were submitted to Dr. Carol Simpson in Raworth, NSW, for petrological description (Simpson, 2023). Descriptions of the intrusive sill unit observed at LP identified a preserved textural similarity between the dolerite samples, which would suggest that the various sills observed in the prospect area are likely to be part of a single magmatic event.

A SEM analysis was conducted on assay pulps by SGS Minerals Services. The study included a general mineralogical analysis and specific uranium study of uranium-bearing species and associated gangue minerals. The purpose was to provide accurate characterization of the uranium species present in the RT and JG mineralization.

QEMSCAN analysis of samples was also undertaken. SGS was contracted to analyse a number of uranium bearing ore samples to quantify their mineralogy, particularly with respect to uranium bearing minerals (SGS, 2009).

In 2015, analysis of drill core was undertaken using a hyperspectral PIMA. The purpose was to physically examine and interpret the mineralized intervals of the Westmoreland Conglomerate, and to gain information on clay alteration halos present (Stevenson, 2009).

10.0 DRILLING

10.1 Historic Drilling Campaigns

Due to the Project changing hands multiple times since the late 1950s, there have been many phases of exploration drilling under the respective licence holders. For the purpose of this report, hole numbers and metres quoted are from inside the three EPM licences (Figure 10.1.1 and Table 10.1.1). It is worth noting that there are additional drilling campaigns external of the Westmoreland licences in QLD that are not referred to in this report.

The major drilling campaigns are outlined in the following section.

10.1.1 Early Drilling – Mount Isa Mines

In 1958, MIM initially drilled 13 'wagon' percussion holes, totalling 350 m, at RT, encountering visible torbernite. MIM reported up to 12 mineralized horizons in the secondary mineralization, averaging 7.3 m thick over a 430 m by 90 m area, with grades ranging from 0.05% to 0.5% U_3O_8 (Brooks, 1960).

10.1.2 1969 to 1971 – Queensland Mines Ltd (QML)

Following on from MIM, QML carried out ~17,000 m of percussion and ~25,000 m of diamond drilling (DD) across the RT Group prospects, in addition to the other exploration campaigns carried out (detailed in Section 9.1.1). The average width of mineralization was quoted to be 9.5 m. Assays varied between 0.05% and 1%, averaging 0.2% U_3O_8 (Culpeper et al., 1999).

10.1.3 1969 to 1971 – BHP

At the same time as QML, BHP drill tested some regional prospects outside the Rt Group, paying particular emphasis on Amphitheatre. BHP drilled a total of 5,680 m (2,275 m diamond, 3,404 m percussion), with 2,534 m of those total metres at Amphitheatre. Amphitheatre is not included in the MRE update.

10.1.4 1975 – Mount Isa Mines (MIM) and Minad

In 1975, MIM returned back to the licence area and drilled 57 diamond holes at RT for 2,251 m. Minad also drilled two diamond holes at RT for 108 m and eight diamond holes at HB for 594 m.

10.1.5 1975 to 1989 – Joint Venture Work

In 1975, five companies formed a JV (QML, UGA, Anglo Australian Resources NL and CRA, and Omega (later, in 1982)) to work the licence area together. As a result, there were multiple drilling campaigns, often at the same time, the outcomes of which are noted below.

1976 to 1980 Urangesellshaft (UGA)

UGA drill tested all RT Group deposits (RT, HB, JG, and LP), the HB-JG link, and other regional prospects:

- RT – two diamond holes for 154 m.
- HB – two percussion holes for 174 m, 133 diamond holes for 11,034 m.
- JG – 212 diamond holes for 11,520 m.
- HB-JG link- 27 diamond holes for 1,823 m.
- LP – 35 percussion holes for 1,082 m, 45 diamond holes for 3,045 m.
- Other prospects – six percussion holes for 114 m, 89 diamond holes for 7,744 m.

1977, 1985 -1986 – Omega

In 1977, Omega drilled two diamond holes at HB for 121 m and one diamond hole at RT for 61 m. In 1982, Omega joined the JV and drilled again in 1985 to 1986 with an additional 45 diamond holes targeting HB for 1821 m and one isolated diamond hole at a regional prospect (51 m).

1995 – CRA Exploration Ltd.

CRA Exploration Ltd (CRAE) took control of the ground in 1990, and in 1995 undertook a large drilling campaign, comprising 5,129 m of percussion, 11,064 m reverse circulation (RC) and 1,350 m of diamond drilling over the RT Group deposits to define the resource. The RC holes were sampled at 1 m intervals over a riffle splitter for dry samples. Wet samples were bulked sampled and then spear sampled at 1 m intervals. Intervals to be sampled were selected based on radiometric response using a scintillometer (URTEC UR135).

Much of the 1995 MRE was based on this RC program, and it appears that most of the core holes drilled by CRAE were used for metallurgical sampling and were not used for estimation.

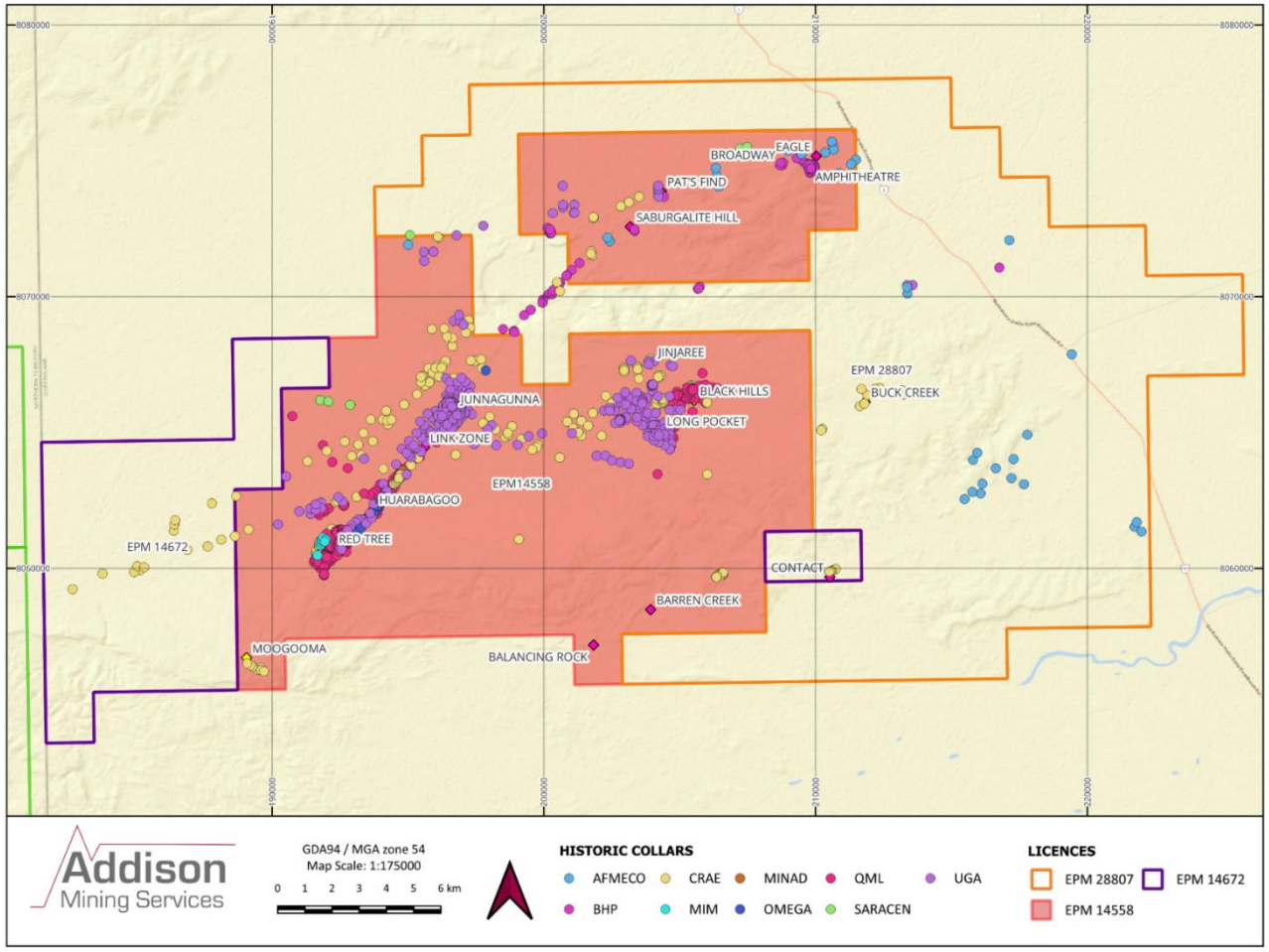
CRAE also drill tested Black Hills, a regional prospect 1.6 km NE of LP, with an extensive 8,754 m percussion, RC and diamond drilling campaign. Black Hills is not part of the MRE update but is discussed in Section 7.4.2.

Various other companies during this period drilled other regional prospects which are not discussed but are summarized in Table 10.1.1.

Table 10.1.1 Drillhole Summary of Collars Inside the Laramide Uranium Project EPMs, QLD

Deposit	Company	Period	Open Hold Percussion		Reverse Circulation		Diamond Core		Total	
			No. Holes	Metres m	No. Holes	Metres m	No. Holes	Metres m	No. Holes	Metres m
Redtree	MIM	1958	13	349.58	-	-	-	-	**13**	**349.58**
	QML	1969 - 1971	286	14,675	-	-	30	5,021.9	**316**	**19,697**
	MIM	1969 – 1970	-	-	-	-	57	2,251.1	**57**	**2,251.1**
	Minad	1975	-	-	-	-	3	107.9	**3**	**107.9**
	UGA	1976 – 1978	-	-	-	-	2	154.05	**2**	**154.05**
	Omega	1977	-	-	-	-	1	61.3	**1**	**61.3**
	CRAE	1990 - 1995	83	2,870.4	-	-	17	797.08	**100**	**3,667.5**
Huarabagoo	QML	1969 – 1971	30	999.44	-	-	105	14	**135**	**15,703**
	Minad	1975	-	-	-	-	8	14,704	**8**	**594.14**
	UGA	UGA	2	174.28	-	-	133	594.14	**135**	**11,208**
	Omega	1977, 1985 – 1986	-	-	-	-	45	11,034	**45**	**1,821.3**
	CRAE	1990 – 1995	8	397.53	23	1,895	4	326.53	**35**	**2,619.1**
Junnagunna	UGA	1976 – 1978	-	-	-	-	212	11,520	**212**	**11,520**
	CRAE	1990 – 1993	13	1,154.9	205	8,500	3	149.7	**221**	**9,804.6**
JG-HB Link	QML	1969	-	-	-	-	9	1,586.8	**9**	**1,586.8**
	UGA	1976 – 1979	-	-	-	-	27	1,822.5	**27**	**1,822.5**
	CRAE	1993	-	-	9	669	-	-	**9**	**669**
Long Pocket	QML	1970	54	1,315.2	-	-	21	4,165	**75**	**5,480.1**
	UGA	1976 – 1980	35	1,082	-	-	45	3,045.1	**80**	**4,127.1**
	CRAE	1988, 1993	9	216	-	-	1	38.4	**10**	**254.4**
Other	BHP	1969 – 1971	51	2,178.2	-	-	11	967.4	**62**	**3,145.6**
	QML	1971 – 1976	19	603.18	-	-	4	1,007.8	**23**	**1,611**
	AFMECO	1973	34	3,375.3	-	-	-	-	**34**	**3,375.3**
	SARACEN	1976	2	130.03	-	-	5	1,021	**7**	**1,151**
	UGA	1984	6	114	-	-	89	7,744.4	**95**	**7,858.4**
	OMEGA	1986	-	-	-	-	1	51.5	**1**	**51.5**
	CRAE	1989 - 1994	72	2,368.6	93	6,367.7	1	18.06	**166**	**8,754.4**

Figure 10.1.1 Map of Current Targets



Shows the historic collars and the company responsible.

10.1.6 Example Historic Drilling Cross Sections

The figures below (Figure 10.1.2 to Figure 10.1.5) show example cross sections of historic drilling results for the RT Group of deposits and LP.

Figure 10.1.2 Redtree Example Cross-Section of Historic Drilling Results

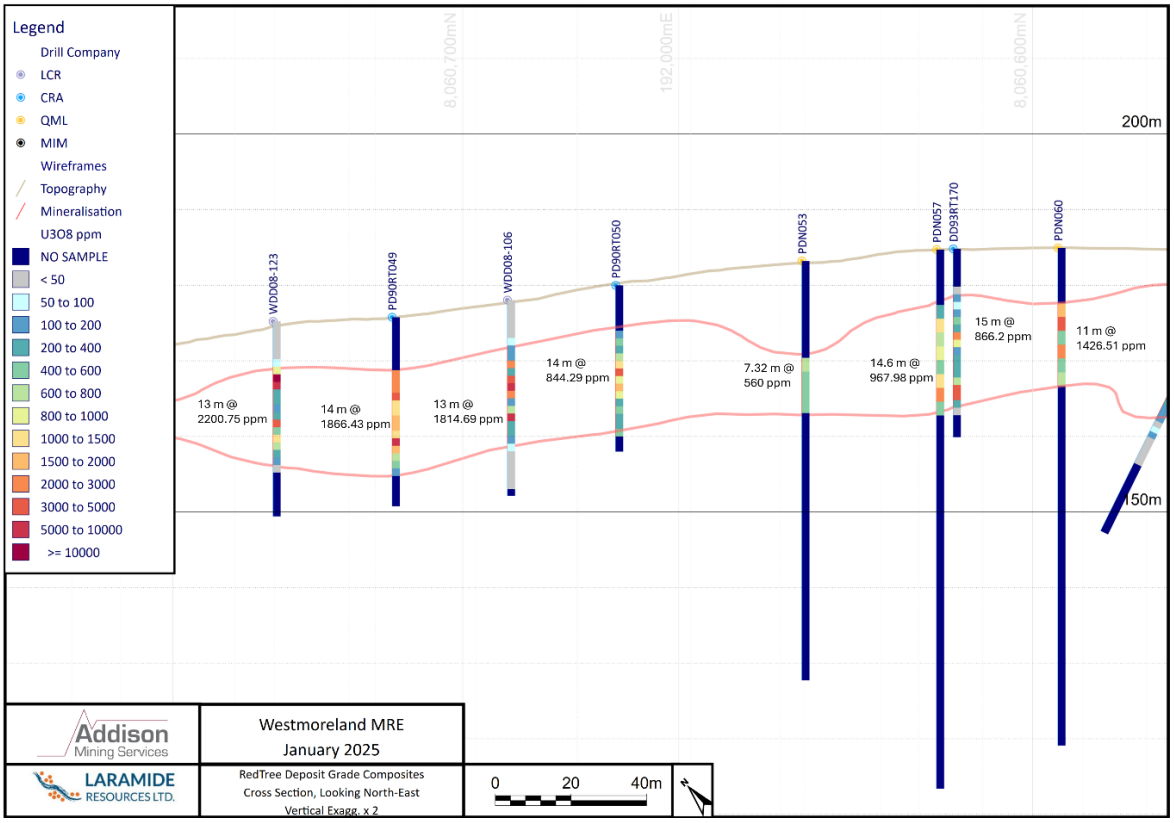


Figure 10.1.3 Huarabagoo Example Cross-Sectio of Historic Drilling Results

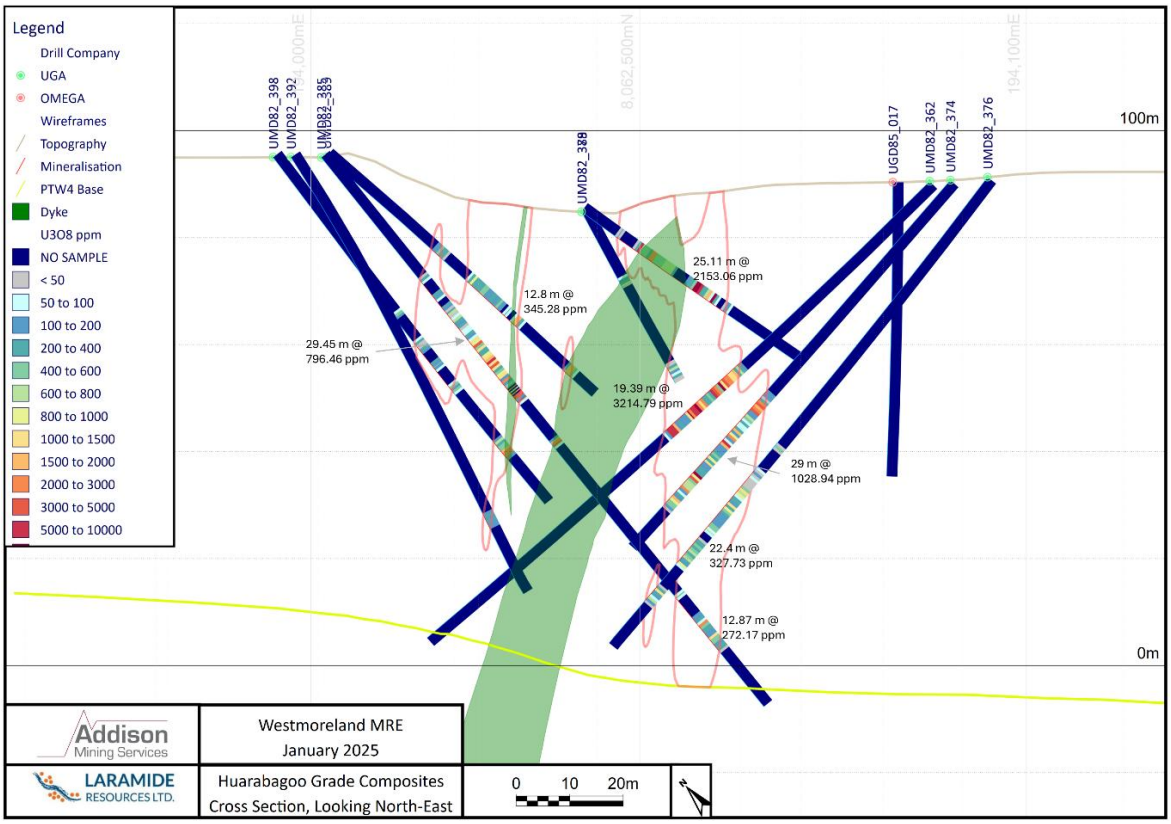


Figure 10.1.4 Junnagunna Example Cross-Section of Historic Drilling Results

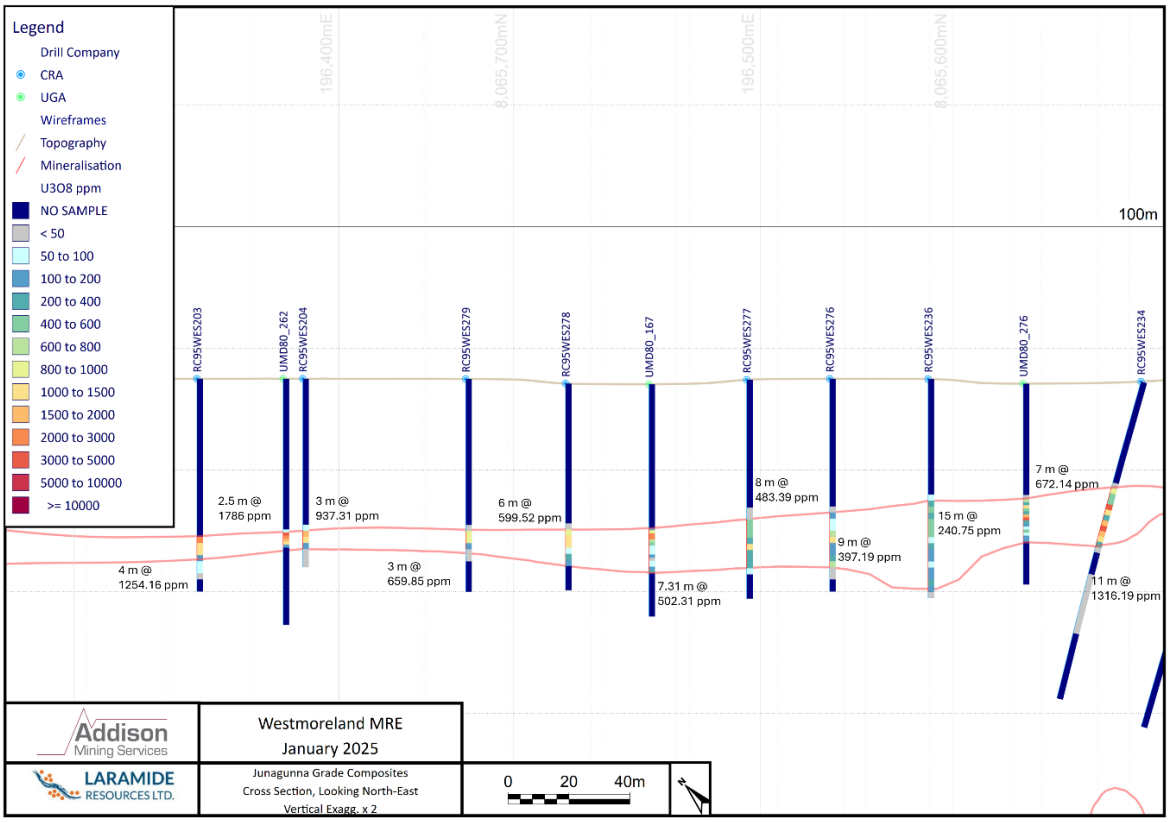
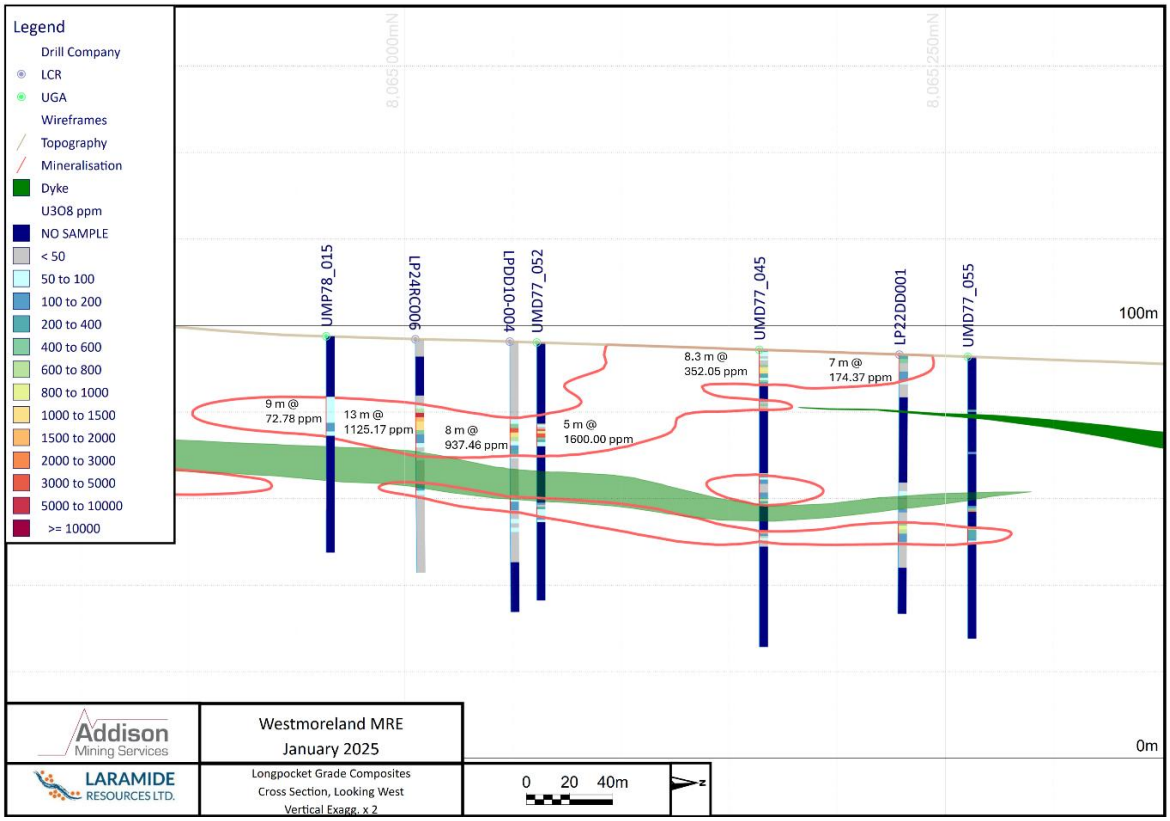


Figure 10.1.5 Long Pocket Example Cross-Section of Historic Drilling Results



10.2 Laramide Drilling Campaigns

LAM has undertaken multiple drilling campaigns as part of its exploration strategy since 2005. The drilling took place in several stages and is summarized in Table 10.2.1 and detailed in the following sections.

Table 10.2.1 Summary of Laramide Drilling Campaigns

Period	Description of Drilling Activity Completed
2007 to 2008	2007 to 2008 campaign at RT and JG Prospects examined totalling 161 holes and 12,272 m.
2008 to 2009	Phase 2 helicopter supported drilling at RT, focussing on the Jack Lens. A total of 925.9 m of diamond core drilling was completed in 39 drillholes.
2009 to 2010	Diamond core drilling at the northernmost part of the HB and the southern extension of the JG deposit between November and December 2009. A total of 1,871.2 m was completed in 31 drillholes.
2010 to 2011	Diamond core drilling program consisting of 19 drillholes for 1,377.9 m was completed. The program comprised 630.4 m in seven holes at HB, and 747.5 m in 12 holes at LP in August 2010.
2012 to 2013	Diamond drilling program comprising 31 drillholes for 4,117.9 m over the HB prospect and the HB-JG 'Structural Corridor'.
2016 to 2017	SWIR spectral data collection and assay including a total of 4,207 spectra samples taken from 63 drillholes at Westmoreland.
2020 to 2021	Drilled core stored at Camp Caroline was resampled for petrography, primary chemical assays, or Pb-isotopic characterization.
2022 to 2023	>1,250 m of diamond drilling at Amphitheatre and LP.
2023 to 2024	4,468 m of diamond drilling at four prospects—HB, Amphitheatre, LP and Black Hills.
2024+	104 diamond and RC drillholes completed for a total of 10,959.46 m across Amphitheatre, LP, HB, the link zone between HB and JG, and JG prospects.

10.2.1 2007 to 2008

The first LAM drilling program commenced in December 2007 and continued into 2008. Two prospects—RT and JG—were drilled by an LF70 rig and two-man portable rigs. The primary objectives of the drilling program were:

- To provide quality controlled drill data within the RT and JG deposits from which to assess the accuracy and validity of historical drilling.
- To improve geological understanding of lithology, alteration, and structural controls on mineralization.
- To provide closer spaced drilling to improve confidence levels on resource estimates.

At RT, the drilling was helicopter assisted using a combination of Bell 206 Jet Rangers and UH1 Huey helicopters. Drilling was completed in early July 2008 totalling 161 holes for 12,272 m. As part of the drilling program, downhole gamma data was collected (discussed in Section 10.4.1).

Samples were collected from both RT and JG, and were submitted for chemical analysis at ALS Laboratories. In addition, samples were sent for petrological analysis, SEM analysis, and a number of holes were drilled for metallurgical testing.

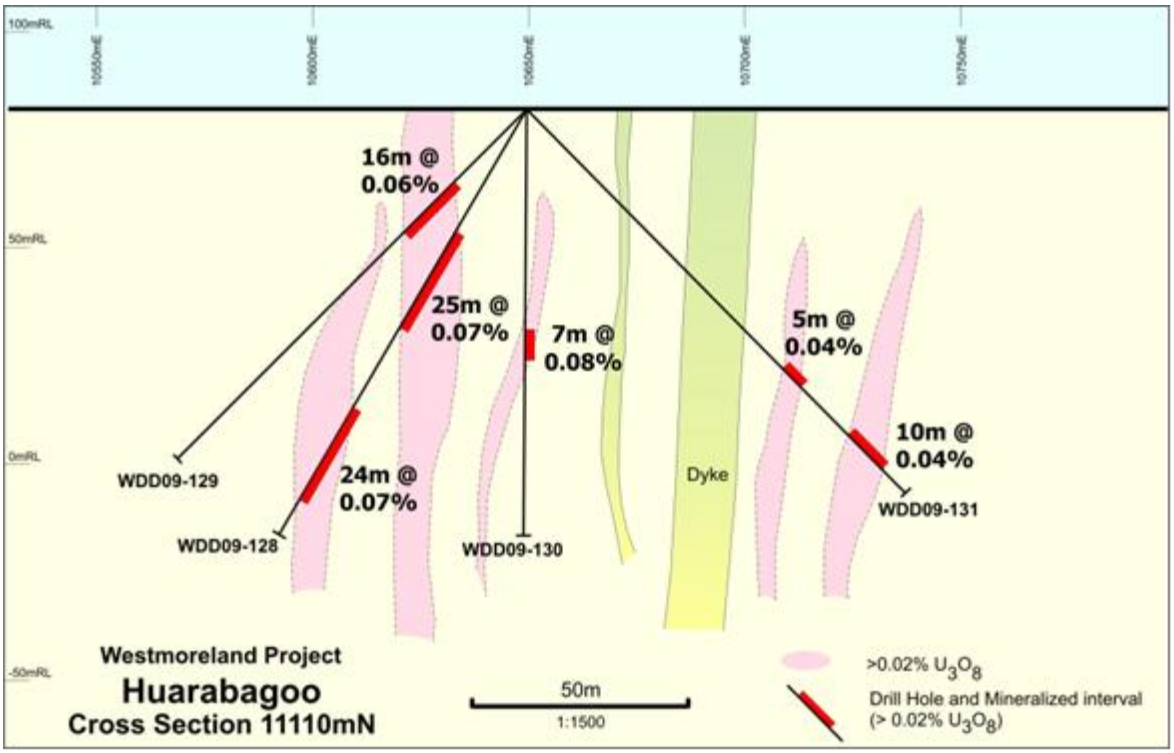
Drilling in 2008 focussed on the Jack Lens at RT, comprising of 39 holes for 925.9 m completed between September and October. The program was helicopter supported due to the difficult terrain. The drilling aimed to validate historical drillhole data and provide information about structure and mineralogy of the prospect. The results confirmed continuity of mineralization identified in historical drilling, and also indicated the potential for high grade lenses within the broader mineralized envelope.

Following the program, an updated independent resource estimate was undertaken by Mining Associates (Vigar and Jones, 2009). This resource estimation incorporated the results of the drilling. A total resource (Indicated and Inferred) of 27.7 Mt at 0.09% U_3O_8 for 51.9 Mlb contained U_3O_8 was estimated. This document was compliant with NI 43-101 requirements and was released to the TSX.

10.2.2 2009

A diamond core drilling program was undertaken between November to December 2009. Drilling targeted the northern most part of the HB prospect, structurally controlled mineralization within the JG deposit and the southern extension of the JG deposit. A total of 1,871.2 m was completed for 31 holes. The results confirmed continuity of mineralization identified in historical drilling and also indicated the potential for high grade lenses within the broader mineralized envelope.

Figure 10.2.1 Huarabagoo Cross-Section 111110 mN



10.2.3 2010

A diamond core drilling program was undertaken in August 2010 to further understand the HB deposit and investigate the Sue and Outcamp prospects at LP. The program consisted of 19 drillholes for 1,377.9 m which comprised seven holes for 630.4 m at HB, and 12 holes for 747.5 m at the Sue and Outcamp prospects at LP, approximately 7 km east of the resource area.

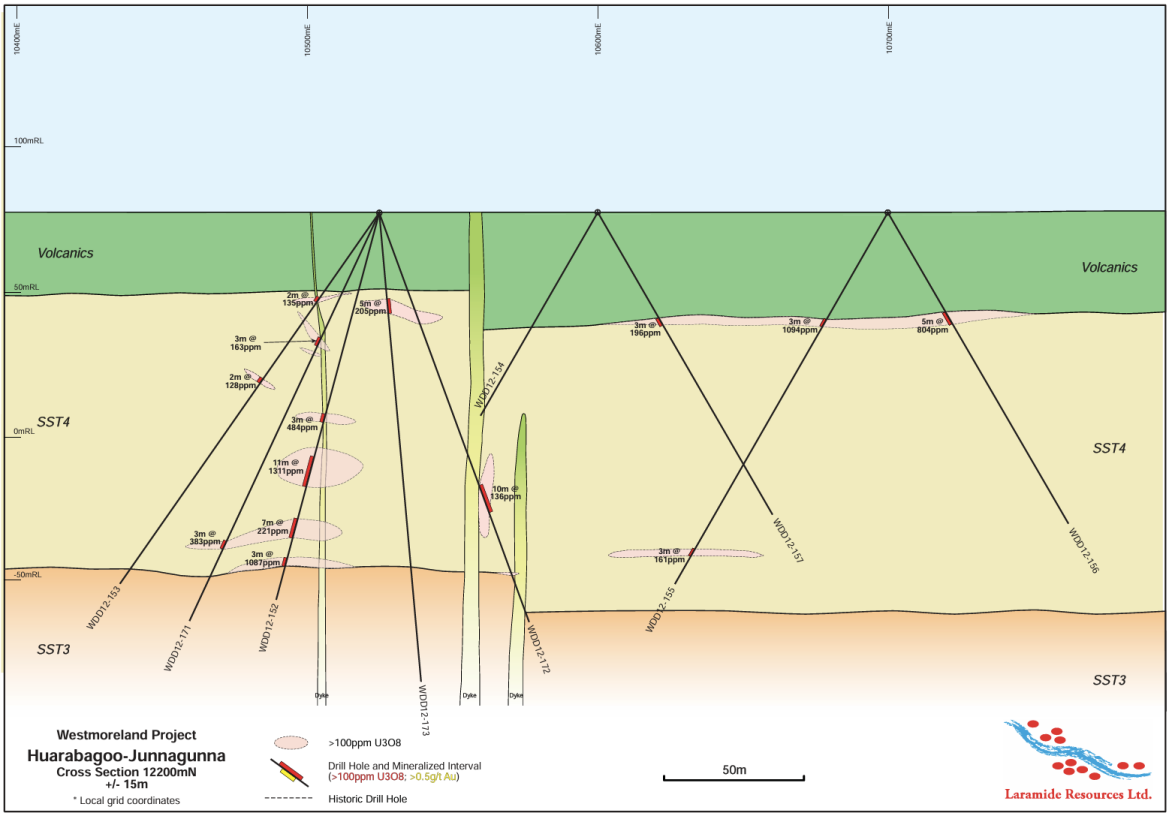
The HB drilling confirmed the mineralization is bound by steep structures broadly parallel to the RT dyke, with indications of horizontal mineralization in coarser more permeable sandstone facies. The drilling at LP confirmed the presence of a broad flat-lying and a relatively shallow zone of uranium mineralization.

10.2.4 2012

A diamond drilling program was undertaken between August and November 2012 at the HB prospect and the HB-JG link 'Structural Corridor'. This program of diamond drilling comprised 31 drillholes for 4,117.9 m.

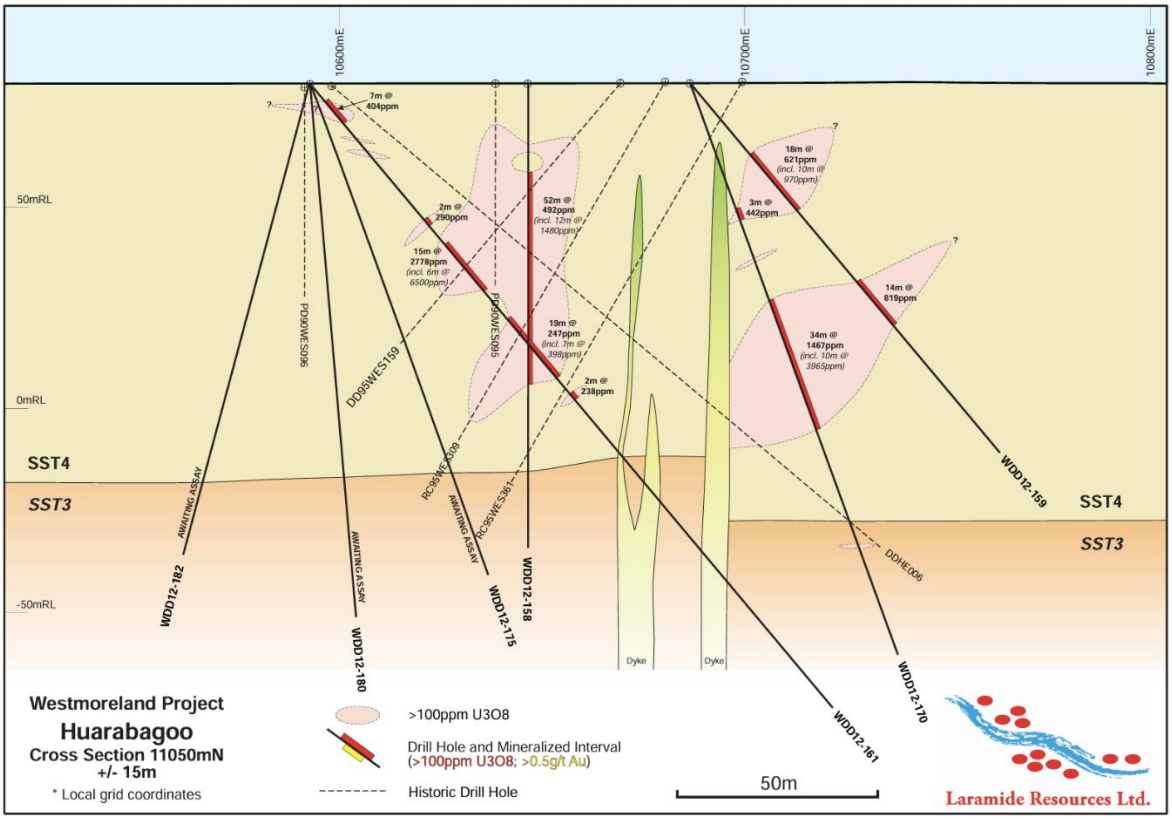
One phase of the drilling program was focused on the highly prospective structural corridor that connects the HB and JG deposits – an area not extensively targeted in the past by LAM. This drilling was one component of a broader program to assess the potential for additional uranium resources at Westmoreland. Initial drilling in the corridor resulted in the discovery of a new zone of mineralization that was not previously known to the Company. In addition, a shallowly dipping zone of mineralization, similar in style to the shallow mineralization at JG, was intersected in the structural corridor and shows the potential to further increase the overall size of the resource (Figure 10.2.2).

Figure 10.2.2 Cross-Section within HB-JG Link Structural Corridor



The second phase of the drilling program focused on the HB deposit both in the existing resource and in the northern section outside of the resource area. Drilling was designed to better define the structurally controlled mineralization in this area and, potentially, increase the resource within the existing deposit and along strike. Drilling delivered significant widths and grades at HB. The drilling confirmed the HB mineralization is controlled by steep structures broadly parallel to the RT dyke (Figure 10.2.3). Drilling in the northern portion of the prospect successfully identified a new intersection east of the dyke.

Figure 10.2.3 Cross-Section at Huarabagoo



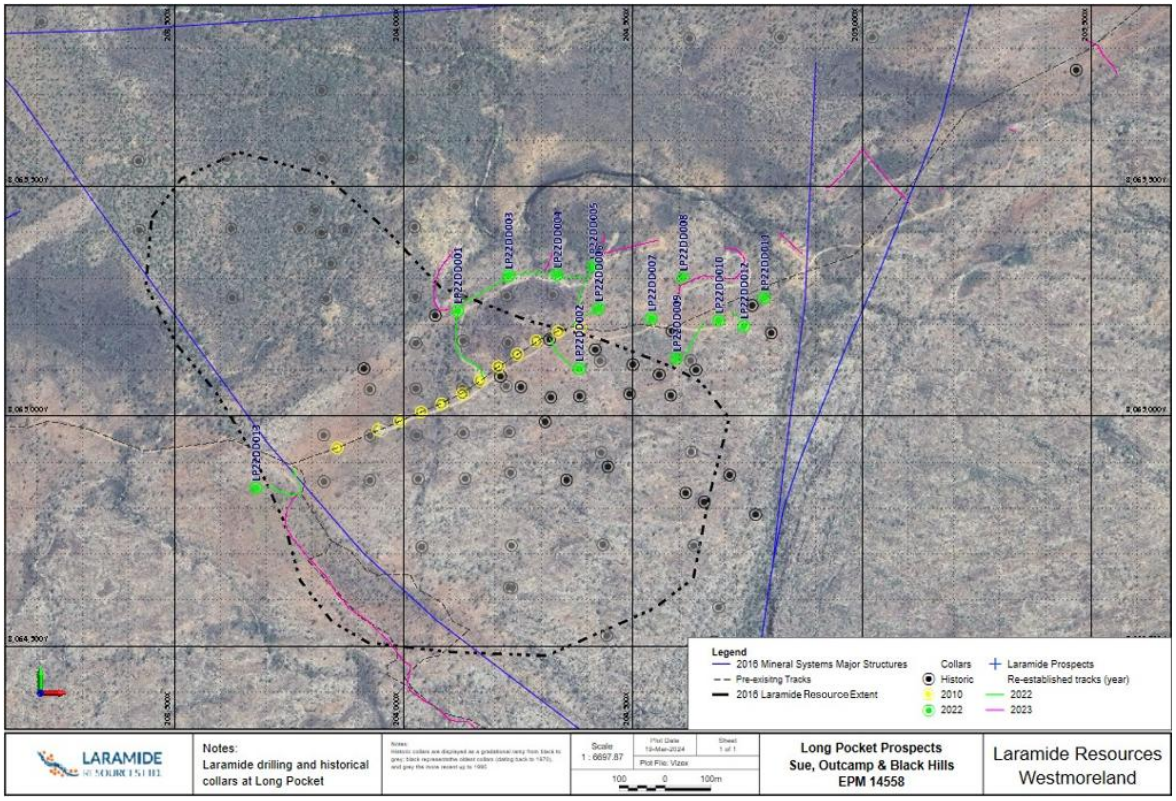
Shows mineralization subparallel with RT dyke.

10.2.5 2022

Drilling in 2022 comprised of 18 diamond drillholes totalling 1,413.5 m across LP and Amphitheatre prospects. The drilling contractor used was owner operated Schonknecht Drilling, utilizing a truck mounted DR800 and Tulla Drilling Pty Ltd using a Sandvik DE710 track-mounted rig.

LP totalled 13 holes for 727.5 m completed by Schonknecht Drilling. Drilling aimed to extend the mineralization envelope to the northeast. Results from drillholes provided immediate step outs of over 100 m from known shallow, flat-lying mineralization, and also suggested a potential mineralized corridor to the Black Hills uranium prospect.

Figure 10.2.4 2022 Long Pocket Drilling with Historical Collars

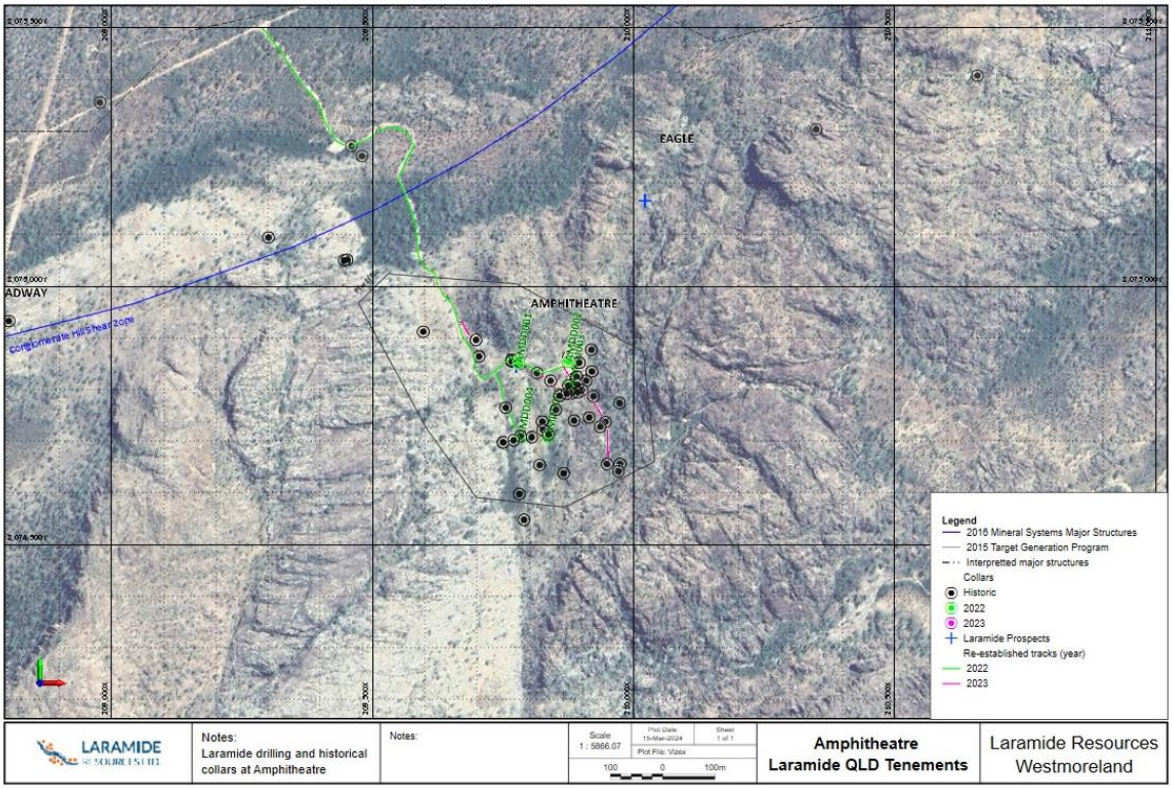


The Amphitheatre uranium prospect expresses as a 400 x 300 m radiometric anomaly and was subject to historical exploration in the late 1960s and early 1970s. Exploration included drilling percussion and diamond holes, costeaning, and 'pitting' up to a depth of 34 feet (ft).

Reconnaissance fieldwork by the company resulted in the 're-discovery' of the Amphitheatre prospect in 2021.

Subsequently, five drillholes were completed at Amphitheatre prospect in May 2022 for a total of 686 m. Drilling was completed by Tulla Drilling Pty Ltd using a Sandvik DE710 track-mounted rig. Drilling intersected mostly sedimentary sequence with minor mafic packages. Significant results (>200 ppm U_3O_8) included: AMDD001 – 3 m @ 507 ppm U_3O_8 from 59 m, including 1 m @ 1,072 ppm (0.107%) U_3O_8; AMDD004 – 4 m @ 277 ppm U_3O_8 from 34 m; and AMDD005 – 2 m @ 413 ppm U_3O_8, including 1 m @ 601 ppm U_3O_8 from 89 m. True widths are not yet understood.

Figure 10.2.5 2022 Amphitheatre Drilling with Historical Collars



10.2.6 2023

A larger diamond drilling campaign was designed for 2023, as detailed in Table 10.2.2.

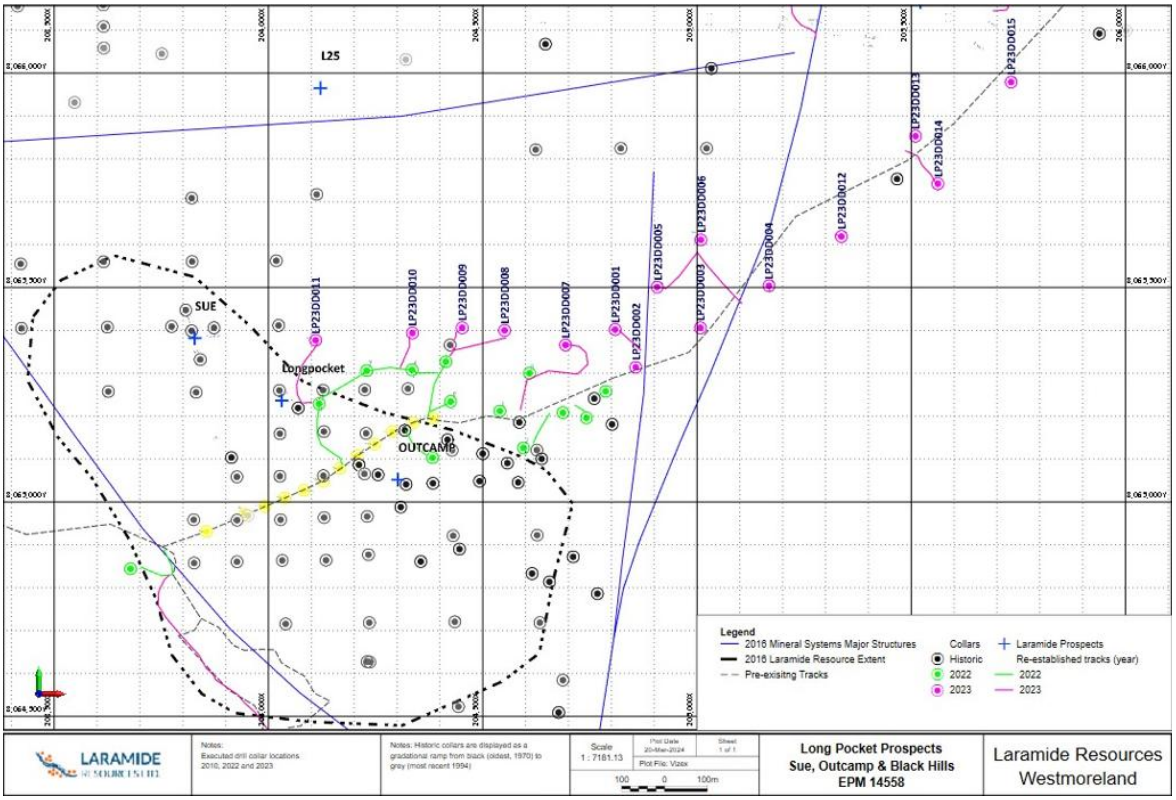
Table 10.2.2 Key Drilling Metrics for the 2023 Programs Broken Down by Prospects

Prospect	Total Metres Drilled m	Number of Holes Completed
Amphitheatre	856	9
Long Pocket	1,016	15
Black Hills	646	3
Huarabagoo	1,590.50	13
Total	**4,109**	**40**

At LP, the program aimed to further test extents of the mafic sill and associated mineralization to the NNE, stepping out ~100 to 200 m spacings following a radiometric anomaly extending towards the Black Hills prospect. The drilling contractor used was owner operated Schonknecht Drilling, utilizing a truck mounted DR800.

Only six significant uranium intercepts (>100 ppm U_3O_8) were identified in this step out program. Near resource significant intercepts were typical of LP mineralization styles, albeit low grade (148 ppm and 108 ppm respectively). High grade intervals were isolated to the east, with no clear trend in mineralization style.

Figure 10.2.6 Plan View of Drillhole Locations



Completed at LP in 2023.

At HB, drilling aimed to infill zones that had been historically drilled, and to test for potential extensions of mineralization to the NE, beyond the footprint of the existing resource. Typical drillhole geology showed a weakly weathered sandy-gravelly conglomerate from surface as the PTW4 of the Westmoreland Conglomerate series. All holes intercepted multiple zones of mineralization (>100 ppm U_3O_8), with some zones displaying grades exceeding 1.0% U_3O_8. The high-grade mineralization intercepted improved the confidence in the integrity of the deposit with tighter spacing of less than 50 m in some places, and supported extensions to over 250 m strike of a gold zone identified in 2012.

Figure 10.2.7 Plan View of 2023 Huarabagoo (HB23) Drillholes and Respective Assay Results

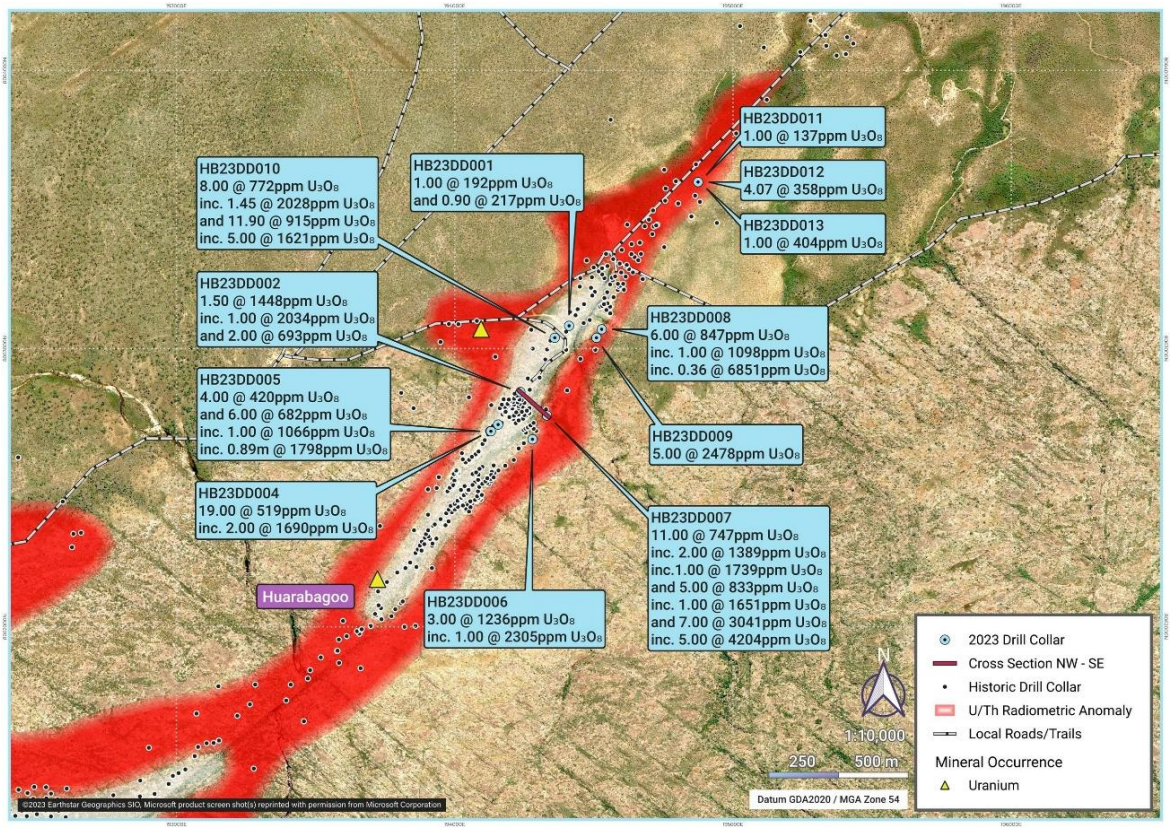
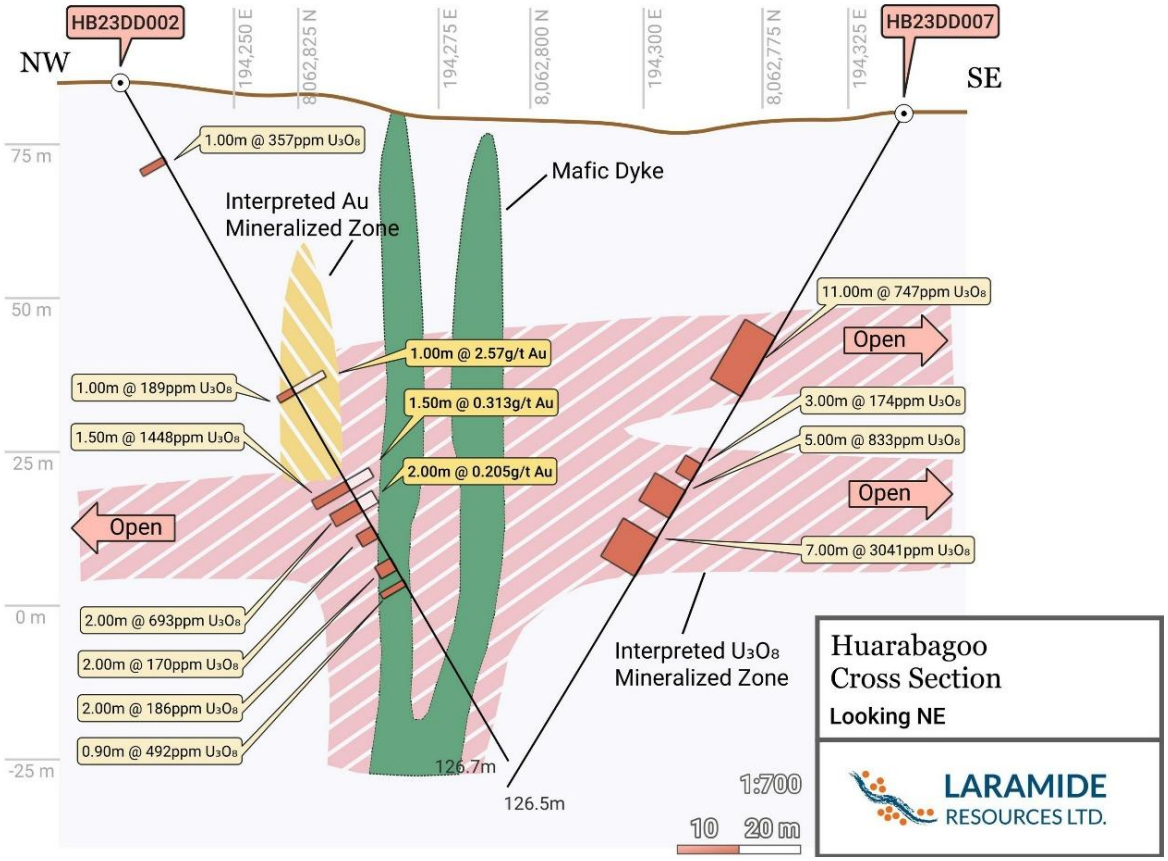
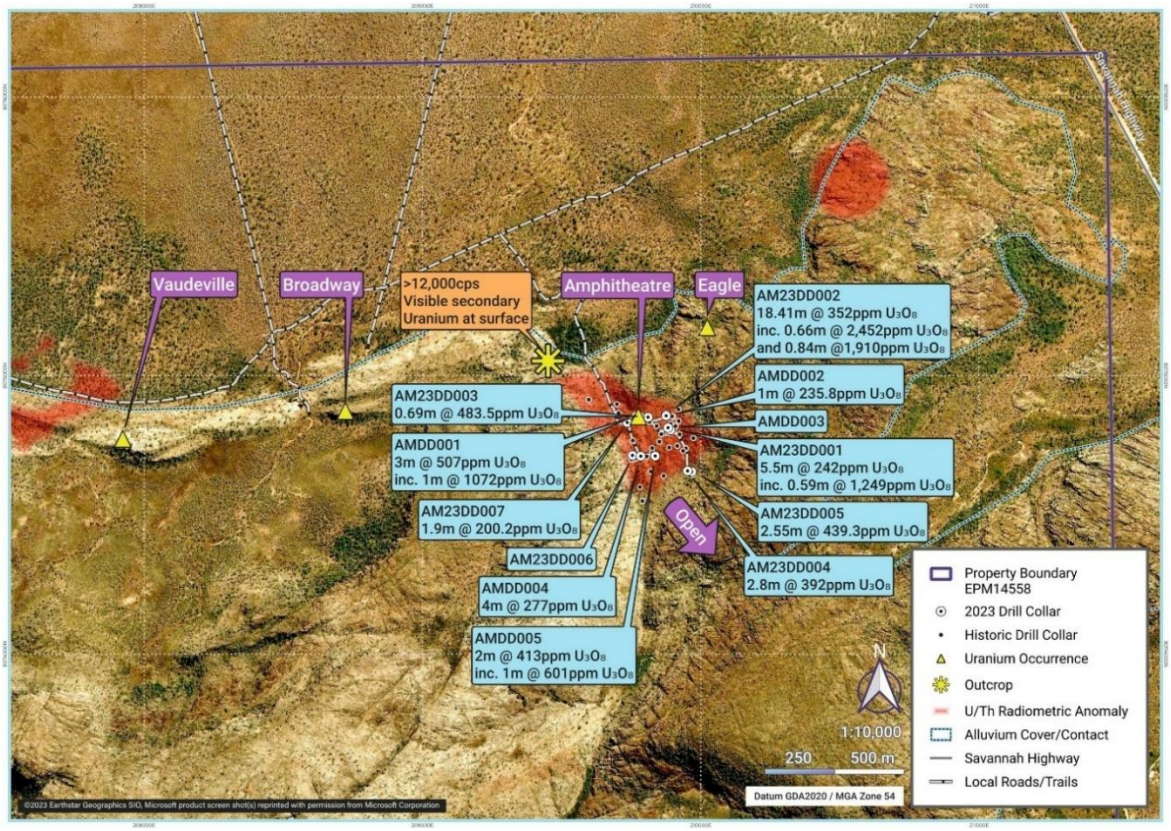


Figure 10.2.8 Huarabagoo Cross-Section Showing Broad Mineralized Intercepts



The drilling program at Amphitheatre aimed to test its potential as a satellite deposit. Drilling intersected predominately sandstones, minor siltstones, mudstones and clays, with the latter typically associated with apparent fault zones and secondary alteration. Alteration is variable but dominated by hematite alteration and found in close association with uranium mineralization. Results showed multiple zones of shallow mineralization, and highlighted mineralization over 200 m south of previous drilling which remained unconstrained to the east and south.

Figure 10.2.9 Summary of Drilling Results at Amphitheatre



10.2.7 2024

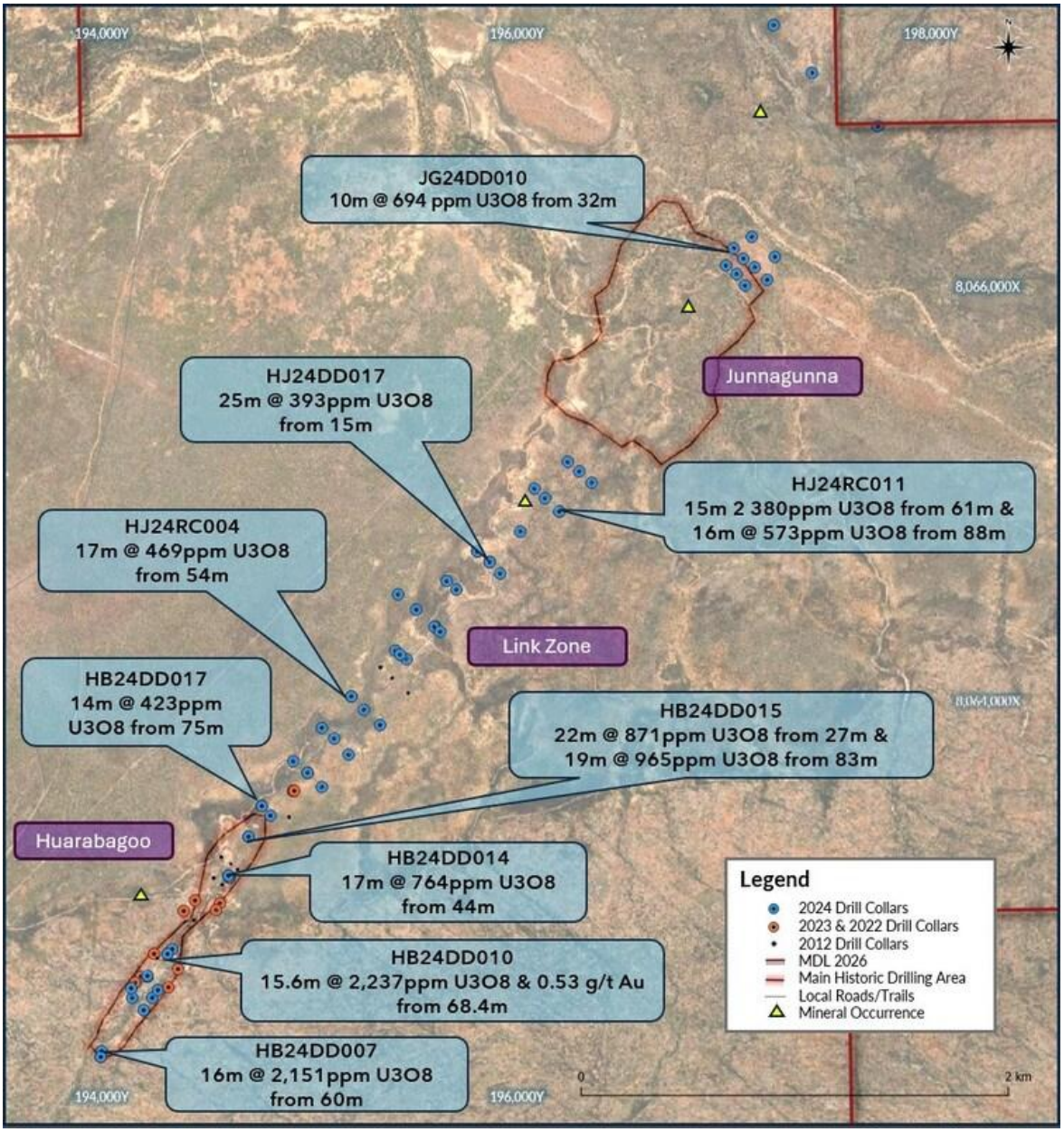
A substantial drilling campaign across various prospects was also completed in 2024 (Table 10.2.3). Planetary Drilling Pty Ltd completed diamond drilling at Amphitheatre and HB utilizing a Sandvik DT600 SPMP and AED Pty Ltd completed both RC and diamond at LP, the HB-JG (HJ) link zone and JG, utilizing a McCullochs multipurpose URD650.

Table 10.2.3 Drilling Details for 2024 Campaign

Prospect	Amphitheatre	Long Pocket	Huarabagoo	HJ-Link Zone		Junnagunna		Totals		
Drill Type	DD	RC	DD	RC	DD	RC	DD	RC	DD	Total
No. Holes	8	38	17	21	8	1	11	60	44	104
No. Metres	1,336	2,136	1,827.16	3,096	1,124.1	114	1,326.65	5,346	5,613.46	10,959.5

Infill and extension resource drilling was completed at LP, HB and JG with positive results contributing to this resource upgrade (examples shown in Figure 10.2.10).

Figure 10.2.10 Map Showing Examples of Broad Mineralized Intercepts from 2024 Drilling



Further exploration drilling was also completed at Amphitheatre, testing to the east of the prospect area for primary and secondary uranium mineralization. The program also aimed to define structural controls on mineralization, and test for potential extensions to the north, obscured by tertiary sediments. Encouraging results were returned and warrant follow up in 2025 and 2026.

10.3 Other Prospect Campaigns

10.3.1 Laramide Scout Drilling - Black Hills and U-Valley

While LAM's recent focus has been on LP, three 2023 scout holes (BH23DD001 to BH23DD003) were drilled to orient the dolerite body and verify historical intercepts at the Black Hills exploration prospect.

Multiple zones of mineralization were identified, with results including:

- BH23DD001 with 3.0 m at 259 ppm U_3O_8 from 29 m.
- BH23DD001 with 0.8 m at 505 ppm U_3O_8 from 120.12 m.
- BH23DD002 with 2.0 m at 591 ppm U_3O_8 from 209 m, including 0.9 m at 1,154 ppm U_3O_8 from 210 m.
- BH23DD003 with 3.0 m at 1,844 ppm U_3O_8 from 88 m, including 2.0 m at 2,671 ppm U_3O_8 from 89 m.

Mineralization identified in the 2023 scout holes was predominantly hosted in the coarse-grained granular Westmoreland Conglomerate, with the higher grades associated with the fractured footwall contact of the intrusive dolerite dyke. Significant intercepts reported in BH23DD003 associated with the dolerite dyke, recorded anomalous Ti, V and Y in both the hanging and footwall.

10.3.2 Historical Campaigns

Moogooma

CRAE drilled seven percussion holes in the general Moogooma area in 1990. Example intercepts (U_3O_8) are reported in Table 10.3.1. The holes presented in Table 10.3.1 are a series of shallow holes over a 620 m WNW-ESE section and are listed east to west. The following interpretations can be made from the assay data available:

- The best values are observed in the western hole at 6 to 7 m in a 26 m deep hole.
- Next best values are reported at 2 to 3 m in 19 m deep hole located further to the east.
- No economic assays are identified in the middle holes that ranged from 19 to 40 m deep.
- A marginal assay is recorded in the most eastern hole.

Table 10.3.1 Moogooma Drillholes and Limited U_3O_8 Assays by XRF (Rio Tinto, 2000)

Hole ID	From m	To m	U_3O_8 %	Au ppm	Gamma Log
PD90WES082 EOH Depth 26 m	3.0	4.0	0.0034	0.02	
	4.0	5.0	**0.011**	0.02	
	5.0	6.0	0.007	0.02	
	6.0	7.0	**0.10**	**0.14**	
PD90WES081 EOH Depth 19 m	0	1.0	0.003	0.02	
	1.0	2.0	0.0075	0.04	
	2.0	3.0	**0.025**	**0.08**	
	3.0	4.0	0.0085	0.02	
PD90WES080 EOH Depth 19 m	No recorded assay data				
PD90WES077 EOH Depth 40 m	0	1.0	0.0018	0.02	Low level ~50 - 150 cps from 3 - 21 m
	1.0	3.0	0.0044	0.02	
PD90WES076 EOH Depth 40 m	No recorded assay data				Low level ~50 – 150 cps from 3 – 23 m
PD90WES078 EOH Depth 23 m	No recorded assay data				Low level ~50 – 150 cps from 3 – 23 m
PD90WES079 EOH Depth 61 m	35.0	36.0	0.0044	0.02	Elevated frp, 36 – 39 m not elsewhere Peak ~100 cps
	36.0	37.0	**0.021**	0.02	
	55.0	56.0	0.003	0.02	
	56.0	57.0	-	0.02	

10.4 Laramide Logging and Data Capture

Holes drilled during the LAM drilling are generally gamma logged at the completion of each hole. All holes are geotechnically and geologically logged and photographed. Based on geological logging and scintillometer readings, 1 m lengths of core were selected for chemical analysis. Selection is based mainly on the gamma log, supported by hand-held scintillometer checking. Any section >300 cps (based on the modern REFLEX gamma log) is sent for assay, plus a 1 m buffer either side of that interval (sampled in 1 m ½ core intervals). Historically, a 50 ppm eU_3O_8 sampling threshold was used.

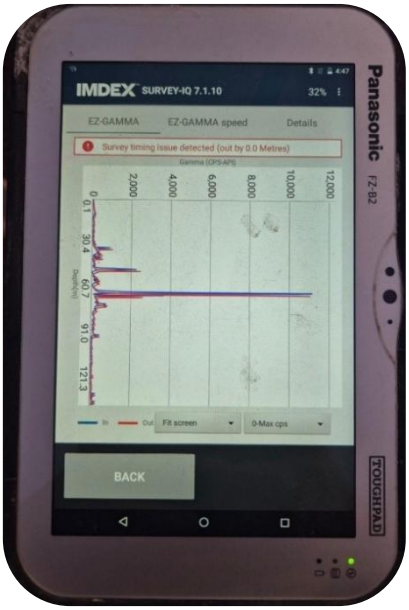
Petrology samples were selected to provide a representative suite of mineralization and alteration across all mineralized zones. Metallurgical samples were collected by taking half HQ core 5 m composite samples.

10.4.1 Downhole Gamma Logging

Logging of drillholes was conducted within steel casing after filling the hole with clean water. An Auslog W450-1 (T070) probe (2008 to 2024) and a REFLEX EZ-Gamma (007) probe (2024 onwards) were used, both recording raw cps readings at 10 centimetre (cm) intervals.

Calibration holes with steel casing are re-logged before each drilling season, showing consistent readings.

Figure 10.4.1 Example of the REFLEX DH Gamma Data Survey Results



From 2008 to 2023, an approximate K-factor was calculated by comparing Auslog W450-1 (T070) logging with assay results from early drillholes. The resulting approximate eU_3O_8 values guided initial interpretation and sample interval selection (broad intervals >50 ppm eU_3O_8 with a 2 to 5m buffer into barren material; entire intervals sampled where multiple zones occurred). These values are not for resource estimation.

From 2024, the REFLEX tool provided effective real-time uranium mineralization indication on a tablet. Sections >300 cps were sent for assay at ALS Laboratories with a 2 m barren material buffer.

Gamma log and assay relationships were monitored throughout the program to check for disequilibrium and ensure proper sampling.

Good correlation was observed with the Auslog W450-1 (T070) from 2008 to 2023, with no significant disequilibrium identified. The REFLEX tool in 2024 occasionally produced false positives, leading to over-assaying. While useful for identifying broad mineralization trends, the REFLEX tool is not for resource estimation. No under-sampling occurred due to the REFLEX tool or geologist oversight.

Gamma logging results are stored electronically in the Company's cloud-based drillhole database (MX Deposit, MXD), linked to individual holes.

10.4.2 Geological Logging

Logging data for all RC and DD drillholes is captured in the Company's cloud database (MXD) following standard procedures. Data is entered into standardized tables in MXD using field tablets assigned per hole to prevent overwriting. After logging, tablets are synced to the cloud database, and the assignment is released. Automated validation checks ensure data accuracy. A supervising geologist verifies log and data integrity before marking each hole as complete and locking it. The cloud database is backed up daily by the provider (Seequent), and application programming interface (API) integrations enable separate daily backups to LAM's SharePoint servers.

Reverse Circulation

All RC drillholes have data tabs for collar, survey, lithology and alteration, recovery, scintillometer and downhole gamma, and sampling (primary, standard, duplicate).

All 1 m RC intervals were sampled (bulk and split from cyclone) and organized at the drill site. Field weights of 1 m splits were recorded in the MXD recovery table. Lab weights were also recorded, and a comparison between these datasets was used to identify potential sampling errors, showing good correlation in the 2024 RC program.

Each 1 m RC interval was geologically logged, with chips sieved and stored in trays. High-resolution photos of chip trays are stored on SharePoint. A downhole gamma survey was conducted at the end of each RC hole to check for mineralization, and data is stored in MXD.

Diamond Drill Core

Diamond drillholes have data tabs for collar, survey, lithology and alteration, geotechnical (including core recovery and rock quality data (RQD)), structure, bulk density, scintillometer and downhole gamma, petrology and SEM, and sampling (primary, standard, duplicate).

As per the standard operating procedures (SOP) mentioned above, core trays are laid out sequentially for logging. Steel V-bars create channels on the logging table to reconstruct and orient the core. Manmade breaks are marked with an X and included in RQD calculations.

Geotechnical logging is followed by geological logging after core assembly and orientation.

All drill core is photographed shortly after drilling, with the reference line facing down. Photos are taken under uniform conditions (dry and wet) using a standard camera connected to the logging tablet via Bluetooth. High-resolution images are stored on LAM's SharePoint server.

10.4.3 Downhole SWIR

In May 2015, 3,086 SWIR spectra (one spectra taken at each metre interval) were measured from 33 drillholes, forming several cross sections through the various mineralized zones at Westmoreland. The instrument used was an ASD Terraspec.

From this initial SWIR data collection (Round 1), the drillholes generally showed a strong sericite signature with some chlorite developed locally. From these spectra it was considered additional SWIR analysis was required to determine if there is a clear footprint / zonation in the data.

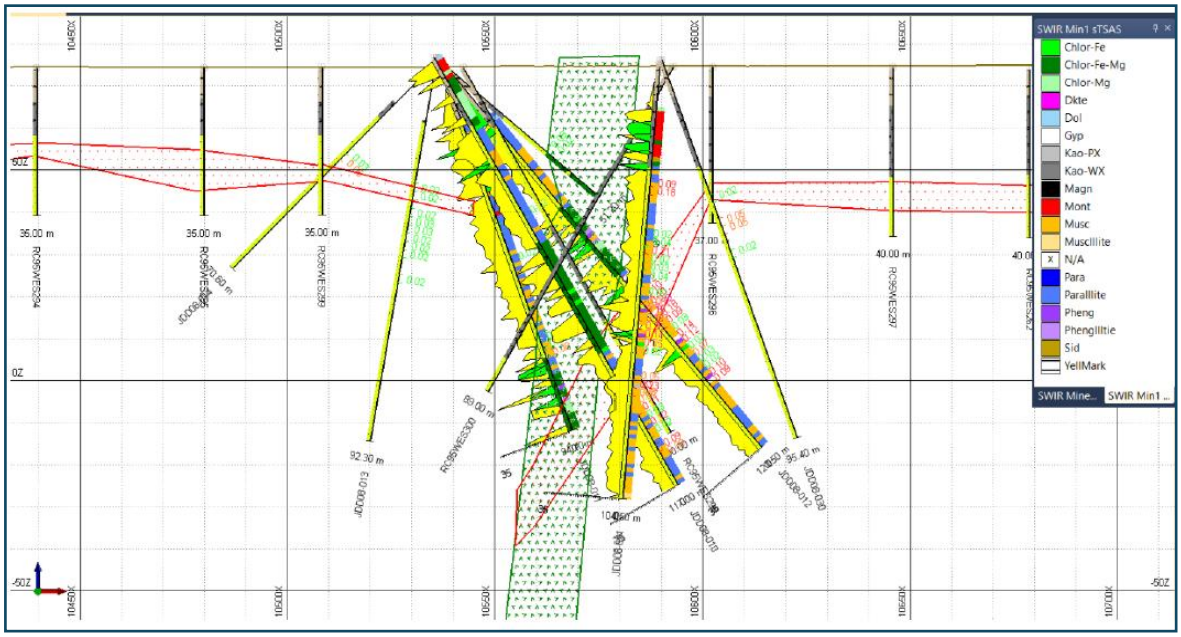
Subsequently, in July 2015, an additional 12,387 spectra samples were collected from 110 drillholes, as part of Round 2. These drillholes analyzed included both QLD and NT drillholes, with a total of 4,207 spectra samples taken from 63 drillholes at Westmoreland. The initial review of the Westmoreland resource area drillholes is as detailed below.

Some of the results were plotted in sections (Figure 10.4). The data reviewed was SWIR mineralogy, and sericitic and chloritic wavelength compositions. The mineralogy reviewed comprised of the following:

- Chlorite.
- Dickite.
- Dolomite.
- Kaolinite.
- Montmorillonite.
- Sericite.
- Sericitic-chlorite.
- Sericitic-kaolinite.

In summary, results indicated that is difficult to draw any firm conclusions for mineralized areas; however, the holes with mineralization generally appear to have some SWIR mineralogy (i.e. sericitic-kaolinite, montmorillonite, etc.) within PTW, whereas unmineralized holes (WDD12-182) do have a simpler mineralogy of generally only sericite in PTW. This is a slightly tenuous conclusion and additional scanning of unmineralized / 'background' holes should help further.

Figure 10.4.2 Junnagunna Drillhole Cross-Section with SWIR and Assay Values Plotted Downhole



10.4.4 Sampling for Chemical Analysis

Assay sampling intervals are selected after logging and photography, primarily based on gamma logs and supported by scintillometer readings (Figure 10.4.1).

Any section exceeding 300 cps on the REFLEX probe is assayed, with a buffer on either side. The sample table is populated with primary samples, non-sampled intervals, and controls (standards, blanks, duplicates).

Reverse Circulation

For sampled RC intervals, split samples in pre-labelled bags were sent to ALS Laboratories for pulverization, with pulps retained. Non-sampled 2024 splits were stored sequentially in labelled green RC bags at the core processing yard for potential future use.

During chip sieving for geological logging, fines from each 1-m interval were collected in numbered geochem packets. These are boxed per drillhole and stored at LAM's core processing facility for potential future pXRF analysis.

Diamond Drill Core

Core is typically sampled at 1-m intervals, respecting unit changes. All core is sawn using an Almonte diamond saw and placed in pre-labelled bags (LCDXXXXXX). The core is cut along the geologist's orientation lines. If no orientation line exists, a cutting line is marked. Core is then crushed and pulverized by ALS Laboratories, with pulps retained for potential additional assays.

Figure 10.4.3 Example Drill Core Marked Up for Cutting and Sampling



10.4.5 Petrology

Sample off-cuts were routinely selected for petrologic examination during core logging. Samples were given a sample number and their location recorded in drill logs and entered into the database. Samples are stored in sample bags and are currently housed in the Brisbane storage facility.

A selection of samples were submitted to Dr. Jane Barron in St Ives, NSW, for petrological description in 2008 (Barron, 2008). Petrographic descriptions of samples selected from the RT area describe the sediments as variously silicified, poorly to moderately sorted quartz-rich arenites. The sandstones and conglomerates are considered to be formerly permeable quartz arenites which have become aquifer sediments in which variable proportions of cement (quartz, clay, chlorite) were deposited in pore spaces. Cementing clays are dominantly smectite and kaolinite. Alteration minerals occurring in the matrix are dominantly chlorite, sericite and hematite.

Additional samples were selected in 2022 from LP drill core, were submitted to Dr. Carol Simpson in Raworth, NSW, for petrological description (Simpson, 2023). Sediments selected from the LP area describe the sediments as predominantly coarse-grained to pebbly quartzose sedimentary successions hosting fine grained intervals. Descriptions of the intrusive sill unit observed at LP identified a preserved textural similarity between the dolerite samples, which would suggest that the various sills observed in the prospect area are likely to be part of a single magmatic event. Minor differences between the various dolerite samples occur in the estimated proportions of the main original minerals, especially in the amount of interstitial ex-glass relative to plagioclase and ferromagnesian minerals, and in grainsize but most of the differences between the dolerite samples occurs in the overprinting secondary alteration assemblage.

10.4.6 SEM Samples

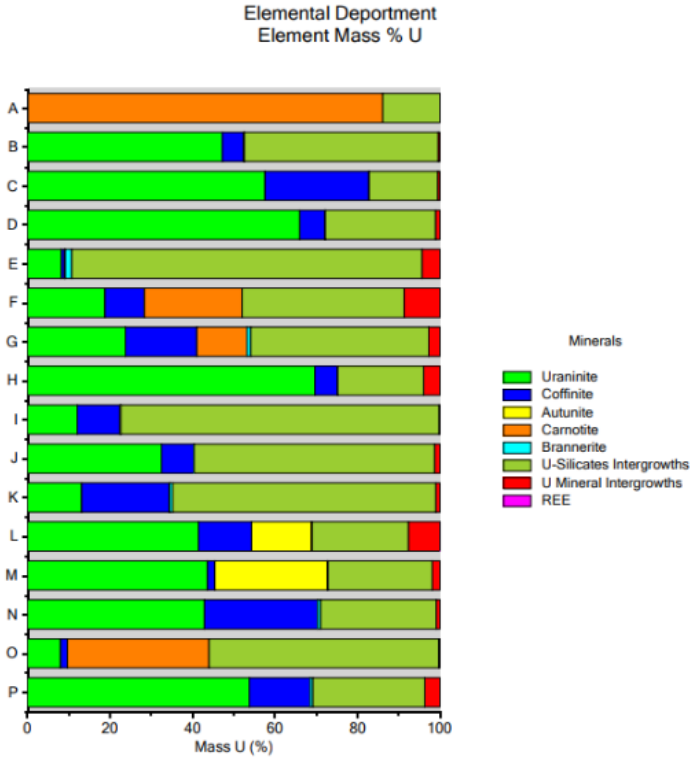
The objectives of the SEM analysis (QEMSCAN) were to provide accurate characterization of the uranium species present in the RT and JG deposits, and a scanning electron microscope analysis was conducted on assay pulps from 20 mineralized intervals in 2009. The study included a general mineralogical analysis and specific uranium study of uranium bearing species and associated gangue minerals (SGS, 2009).

The predominant uranium bearing mineral was found to be uraninite (UO_2) which with coffinite (U(SiO4)1-x(OH)4x) generally comprise >80 weight percentage (% wt) of uranium minerals (Figure 10.4.4). The species autunite (Ca(UO_2)2(PO4)2.10-12H2O) and the dehydrated product meta-autunite is the other distinguishable uranium mineral of note, reporting up to 10% wt of the uranium.

The proportion of brannerite ((U,Ca,Ce)(Ti,Fe)2O6) is less than 1% of uranium mineralization in 15 of 19 mineralized samples with values from 1 to 3.2% in four samples.

The results are consistent with previous SEM analyses undertaken by Rio Tinto in 1994.

Figure 10.4.4 Elemental Deportment of the Uranium Between the Uranium-Bearing Minerals in all the 2009 SEM Samples



SGS, 2009.

10.4.7 Metallurgy

A total of 16 drillholes were drilled with HQ core size during the 2007 to 2008 program to obtain samples that may be submitted for metallurgical testwork. The holes were selected to provide representative samples throughout the RT deposit and to a lesser extent, JG (Baker et al., 2011). Samples consisted of five metre intervals of half HQ core. The samples were delivered to ANSTO at Lucas Heights in sealed 200 L drums.

10.4.8 Discussion

The QP considers the procedures employed by LAM to be well managed and thought out. Rock quality data indicated the core was competent and drill core recovery is excellent (average 97%). Areas of poorer recovery can be identified as near surface unconsolidated sediments (up to 20 m thick in places) or zones of poorer recovery below surface are generally associated with the dolerite dyke contact.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Historical Sampling

The exact procedures employed by previous explorers is not well understood, procedures that are known are described in the following sub sections. The assay methods used by previous explorers was captured in the access databases and migrated to LAM's MX Deposit system and are presented in Table 11.1.1.

Table 11.1.1 Summary of Analysis Techniques Used by Previous Explorers

Value	Description	Company
GS3	SCINTREX GS3 SCINTILLOMETER	QML
RAD-MIM	SCINTILLOMETER – MIM	MIM
XRF-1	XRF ALS	UGA, QML
XRF-5	XRF ANALABS/PILBARA	Omega
XRF-A	XRF AMDEL	CRA, MINAD, QML, UGA
XRF-G	XRF GEOMIN LABORATORY	UGA
XRF-MIM	XRF MT ISA MINES LABORATORY	MIM

11.1.1 Queensland Mines Limited

Percussion and core samples from the Jack and Garee lenses were bulked over 3 ft. (0.91 m) or 6 ft. (1.83 m) lengths. The samples were analyzed on site using a Scintrex GS-3 spectrometer and any samples >0.05% U_3O_8 were sent to Amdel in Adelaide for analysis by XRF. The holes were also probed using a Berthold LB1200 gamma logger, and anomalous intervals were used as a check against readings from the GS-3 spectrometer.

11.1.2 Urangesellschaft

The UG percussion samples were bulked into 2-m lengths and a representative sample of approximately 1 kg obtained by quartering the bulk sample at the drill site. Core was sawn in half at Camp Ridgeway. Lengths ranged from 0.1 m to 0.5 m; most were bulked into 0.5 m core lengths and sent for assay. During 1976, all UG core samples were sent to ACS in Adelaide and analyzed for U_3O_8 by XRF. The 1977 core samples were analyzed by Geomin in Sydney using fluorometric and colorimetric techniques. Sixteen duplicate samples were checked by ALS in Brisbane; these averaged 10% higher assay than Geomin. In 1978, UG sent all core and percussion samples to ALS in Brisbane for XRF analysis of U_3O_8. All the samples sent to Geomin in 1977 were re-assayed by ALS using both fluorometric / colorimetric and XRF techniques. Check analyses were carried out by Amdel in Adelaide using XRF. From 1978 onwards, all UG samples were assayed by ALS in Brisbane using XRF. Twenty duplicate samples were sent in 1980 to Pilbara Laboratories in Perth. The Pilbara results averaged 6% less than the ALS assays.

11.1.3 CRA Exploration Pty Ltd

The CRAE percussion (RC) holes were sampled at 1 m intervals over a riffle splitter for dry samples. Wet samples were bulked sampled and then spear sampled at 1 m intervals. Intervals to be sampled were selected based on radiometric response using a scintillometer (URTEC UR135). No mention was made of techniques used to sample drill core. It appears that most of the core holes drilled by CRAE were used for metallurgical sampling and were not used for resource estimation.

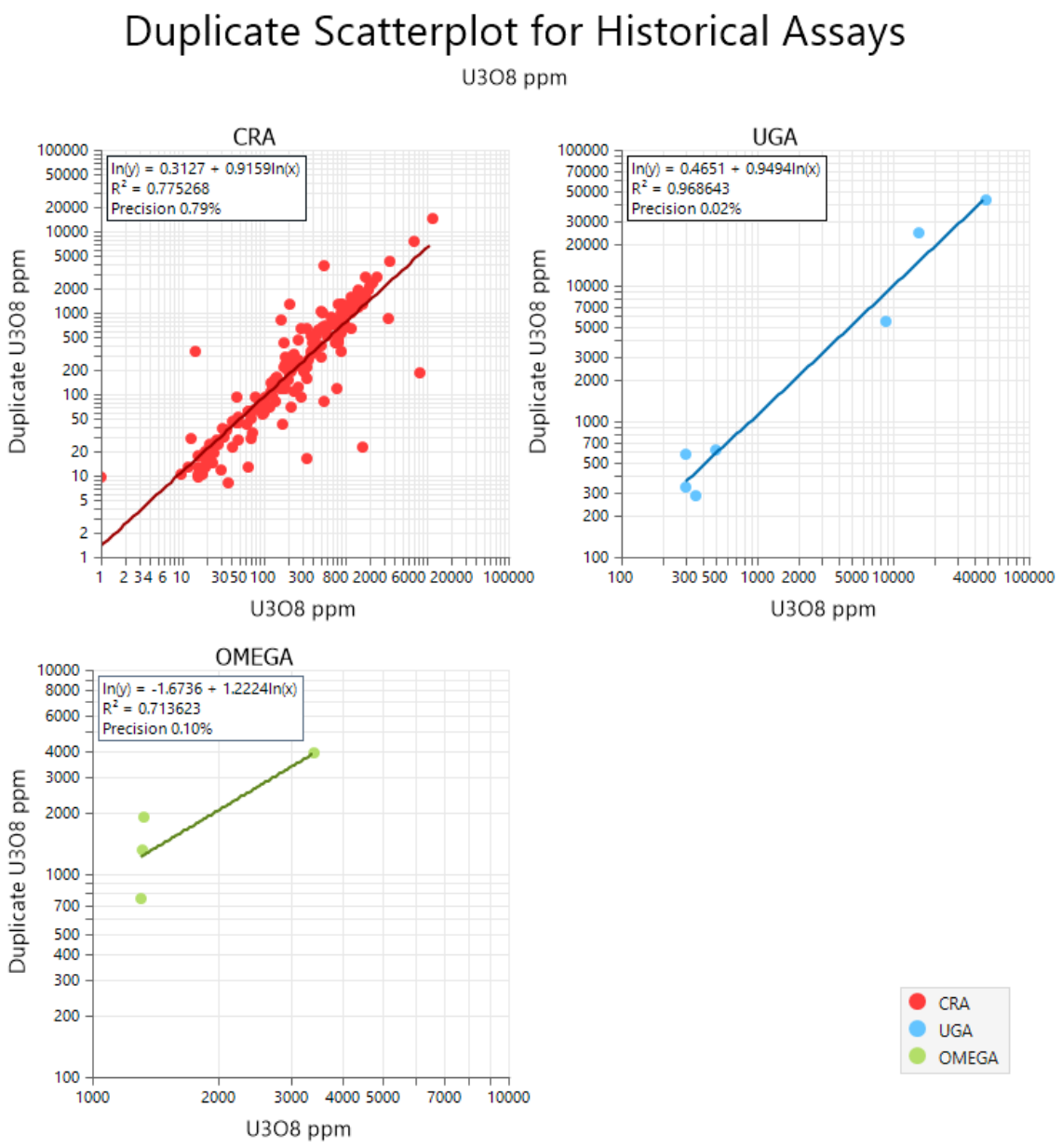
11.1.4 Historical Quality Control

There is limited quality control (QC) data for the historical data. A 'duplicate assays' sheet was identified in historical access database pack which contained hole identification (ID), depth from and depth to uranium assay results for each sample. These were paired with the parent samples in the database based the hole ID and depths resulting 182 sample pairs with the parent sample assay method shown as follows:

- XRF AMDEL (XRF-A) 170 CRA samples.
- XRF ALS 7 (XRF-1) UGA samples and one CRA sample.
- XRF ANALABS/PILBARA (XRF-5) four OMEGA samples.

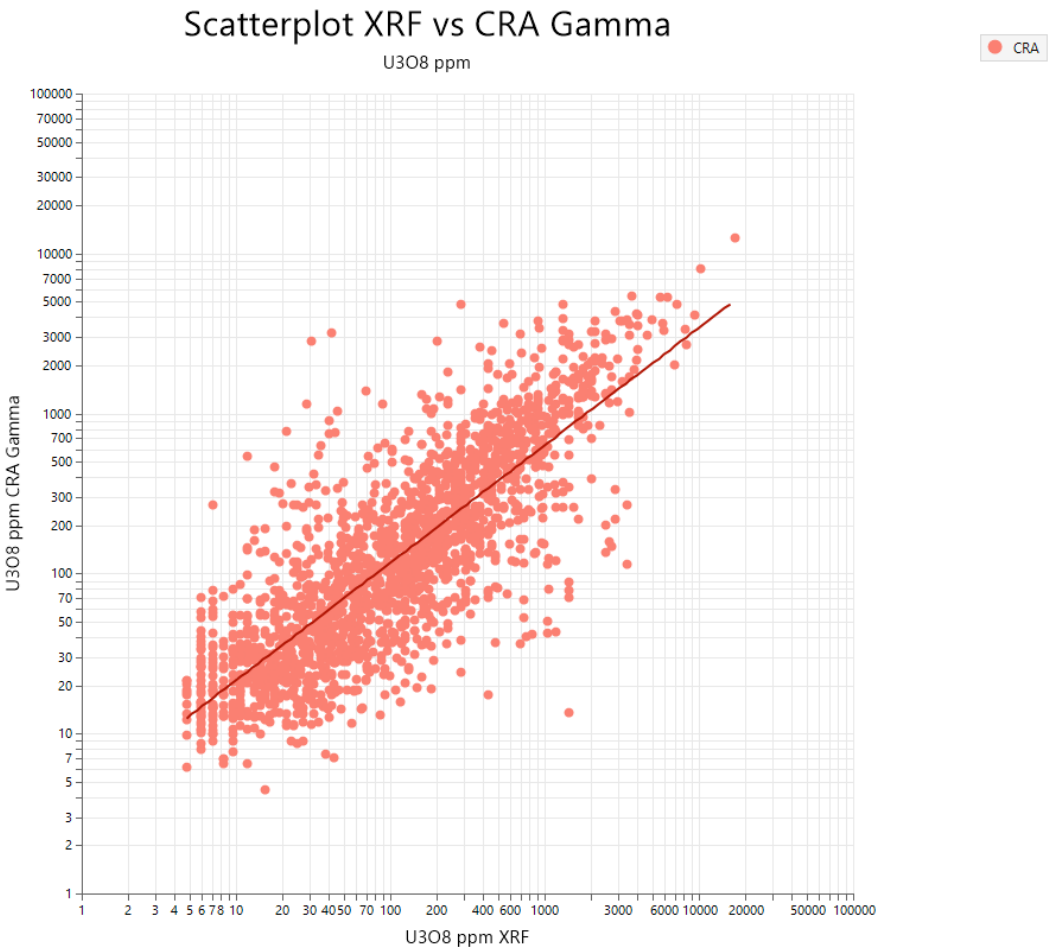
Figure 11.1.1 shows a scatter plot of the available data, with the CRA data being most meaningful. The correlation is generally good with some outliers; however, it is not known what method was used for the duplicate analysis. Furthermore, the duplicate samples only have indication of hole ID and depth from and to - no parent sample numbers were assigned, so there is a risk that keystroke errors may exist in the duplicate data.

Figure 11.1.1 Scatterplot of Original Values Against Duplicates for Historical Assays from Respective Companies



A further 1,876 sample pairs were identified for the CRA downhole gamma and XRF analysis (Figure 11.1.2), the duplicates compare well overall with some dispersion and outliers. It is possible some drillholes may have been examined after drilling, with a potential accumulation of radon present. The probed intervals also did not exactly match the original sample intervals and were joined by means of compositing two interval files; this may result in some discrepancy for the outliers. Analytical error or calibration issues with the probe may also account for the outliers.

Figure 11.1.2 Scatterplot XRF Values Against Rio 1995 Gamma Results



11.2 Laramide Sampling

Sample numbers and intervals are recorded by the geologist into the sampling table in MXD, before being produced as a cut sheet for the sampler.

Blanks are inserted at the start of each dispatch and at an insert rate of 1:50, or at least one blank per hole preferably at the end of a mineralized zone. A coarse blank reference material, 'rainbow quartz', sourced from Dynamics G-EX, is currently in used. Historically (2008 to 2012), a stockpile of un-mineralized core was used as a coarse blank material.

A duplicate sample is added at an insert rate of 1:50, or at least one in every drillhole, and preferably across a range of mineralized grades.

A stockpile of certified reference materials (CRMs) has been purchased by LAM from Ore Research & Exploration Pty Ltd (Oreas), Bays Water North, Australia. LAM currently uses five different CRMs stocked by Oreas, which are detailed in Table 11.2.1. These are inserted at a rate of 1:25 original samples, where the CRM selection should match the local grade in the primary sample sequence.

Table 11.2.1 U and Au CRMs Used in the LAM's Sampling Programs

Standard	Description	Years Incorporated
OREAS 101a	Medium grade U (410 ppm) Mt Gee granitic and haematitic breccia bodies, IOCG variant.	2007 to 2012
OREAS 101b	Medium grade U (387 ppm) Mt Gee granitic and haematitic breccia bodies, IOCG variant.	2007 to 2012
OREAS 120	Low grade U (39.6 ppm) Nyota Prospect Karoo sandstone-hosted tabular deposit.	2022 to present
OREAS 121	Resource grade U (206 ppm) Nyota Prospect Karoo sandstone-hosted tabular deposit.	2023 to present
OREAS 123	Moderate to high grade U (825 ppm) Nyota Prospect Karoo sandstone-hosted tabular deposit.	2023 to present
OREAS 124	High grade U (1,779 ppm) Nyota Prospect Karoo sandstone-hosted tabular deposit.	2022 to present
OREAS 233b	Orogenic lode gold mineralization style, Au bearing ore blended with barren greenstone (1.07 ppm), ore sourced from Frogs Leg Gold Mine, WA.	2023 to present

Individual samples are packed into prelabelled calico bags. Sample bags are packed, five at a time, into white polyweave sacks. About 100 to 150 samples are then packed into a larger bulk bag. These are mounted onto pallets and loaded for transport as complete holes. Depending on the mode of transport (Toyota Landcruiser with or without a trailer; truck or semitrailer), one or more complete holes were transported at a time to Mt Isa for sample preparation by ALS Laboratories.

11.2.1 Security

A sample submission and chain of custody form is held by the driver for each batch of samples transported to Mt Isa and handed over to the laboratory on arrival at the laboratory. Prior to despatch, the metal sample boxes are checked with a Ludlum 2241-3 rate meter, to ensure that the external radiation level is less than 5 microsieverts (µSv) otherwise special labelling is required. Judicious packing (high-grade samples are packed at the centre surrounded by low-grade material) ensures that this standard is never exceeded. The laboratory manager is notified of the despatch before the driver departs. The journey takes 8 to 10 hours (h), and the driver maintains contact with the camp by satellite phone or mobile telephone to ensure security of the cargo during transport.

11.2.2 Laboratory Sample Preparation and Analysis

The ALS laboratory in Mount Isa employs experienced staff for conventional and uranium sampling and assaying. ALS Brisbane and Perth are NATA accredited (No. 825) and ISO/IEC 17025:2005 certified for ore and mineral analysis using AAS, AES (ICP), ICP / MS, XRF, and classical techniques. ALS Mt Isa and Townsville hold ISO9001:2015 accreditation (No. ISO9001-0058569).

Upon arrival at the Mt Isa lab, sample submission forms are checked, and samples are scanned with a ROTEM-GENE dosimeter to determine radioactivity and appropriate handling according to ALS's NORM protocols.

Non-radioactive samples (<1 microsieverts per hour (µSv/h)) are processed in Mt Isa, barcoded, and weighed. A work order (WO) is created (ALS batch numbers with MI prefix). Sample preparation occurs in Mt Isa before dispatch for analysis. Au-AA26 analysis is done in ALS Townsville, and ME-MS61 analysis in ALS Brisbane.

Radioactive samples (>1 µSv/h) are prepared at the Perth Hazardous Materials Prep Facility (ALS batch numbers with PH prefix). ME-MS61 and Au-AA26 analysis for these samples is conducted by ALS Perth, which has specialized equipment and safety protocols for handling radioactive materials, including isolated dust capture and mandatory personal protective equipment (PPE).

All samples are weighed upon receipt for riffle splitting calculations. Samples are crushed, and a portion (~0.5 kg) is split for pulverization, with the remainder kept as a coarse reject. The 0.5 kg aliquot is pulverized to 85% passing 75 microns (µm) in a ring grinder. Silicon wash is applied if requested.

The 0.5 kg pulp is subsampled, and 30 g is placed in a labelled, barcoded wire top packet. Barcodes are generated using ALS software. Labelled pulp samples are weighed, packed, and a corresponding barcode label is applied to the box for chain of custody tracking via ALS's LIMS.

Boxes arriving at ALS Brisbane, Townsville, or Perth are scanned, opened, and individual sample barcodes are scanned into LIMS. Boxes are repacked and stored in a balance room. Before analysis, samples are scanned and weighed, cross-referencing preparation weights. A sub-sample is extracted, weighed, and digested using four-acid for base metals and fire assay for gold.

ALS Brisbane inserts two standards and two duplicates per 36 Client samples, analyzed in batches of 40.

ICP-MS and ICP-AES are used for broad base metal concentration ranges (e.g. uranium 0.1-10,000 ppm). Gold analysis uses fire-assay fusion with AAS finish.

ALS uses internal standardization to monitor instrument drift. Historically, europium was used for drift correction and matrix effect adjustments. Older ME-XRF05 samples were analyzed by pressed pellet XRF, where samples were weighed post-compression for barcode cross-reference. Two standards were included per 40-sample batch for drift monitoring. Modern XRF typically uses fused-based methods.

Table 11.2.2 Analytical Methods Used by Laramide by Year

Method Code	Element(s)	Campaign(s)
ME-XRF05 ME-XRF07 ME-XRF10s	U	2008
ME-XRF15b		2012
ME-XRF30		2024
ME-ICP61s	Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sr, Ti, U, V, W, Zn	2008
ME-ICP61		2008 to 2012, 2022 to 2024
ME-MS61	Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cs, Cu, Fe, Ga, Ge, Hf, In, K, La, Li, Mg, Mn, Mo, Na, Nb, Ni, P, Pb, Rb, Re, S, Sb, Sc, Se, Sn, Sr, Ta, Te, Th, Ti, Tl, U, V, W, Y, Zn, Zr	2022 to 2024
ME-MS61r	Ag, Al, As, Ba, Be, Bi, Ca, Cd, Ce, Co, Cr, Cs, Cu, Dy, Er, Eu, Fe, Ga, Gd, Ge, Hf, Ho, In, K, La, Li, Lu, Mg, Mn, Mo, Na, Nb, Nd, Ni, P, Pb, Pr, Rb, Re, S, Sb, Sc, Se, Sm, Sn, Sr, Ta, Tb, Te, Th, Ti, Tl, Tm, U, V, W, Y, Yb, Zn, Zr	2024
PGM-ICP23 PGM-ICP27	Au, Pd, Pt	2008
PGM-MS23		2007
P-OG62	P	2023
Zr-MS85	Zr	2022, 2023
Au-AA25	Au	2008 to 2012, 2022, 2023
Au-AA26		2023, 2024

11.2.3 Quality Assurance

LAM has a detailed standard operating procedure which has been reviewed and discussed with the QP. There are no concerns with LAM's procedures.

Laboratory procedures include:

- Silicon wash after milling high grade samples and scanning of mill for residue, conducted on request.
- Ability to use sample weight as additional check on sample number.
- Analysis by two methods (ICP / MS or AES depending on grade versus XRF).
- Insertion of two standards and two duplicates every 36 samples.

The QP has not completed any laboratory inspections.

11.2.4 Quality Assurance

Onsite procedures include the following control insert rates as a minimum requirement. Additional controls may be added at the discretion of the logging geologist.

- Blank sample of un-mineralized material at the beginning of each batch and immediately following a mineralized sample, or at a minimum at an inset rate of 1 per 50 samples.
- Mineralized duplicate to be taken every 50 samples, or at least one in every drillhole, and preferably across a range of mineralized grades.
- Certified reference material inserted every 25 samples, grade matched to local primary samples.
- Sampling of barren material either side of mineralized zones.

Standards

2007 to 2012

One hundred and eleven samples of Standard 101a and 107 samples of 101b were submitted for analysis during the 2007 to 2012 drilling programs. Table 11.2.3 shows the summary statistics of the analysis of these samples. During LAM's drilling programs between 2008 and 2012, ALS Fusion Method ME-XRF05 was nominated as the preferred method for all uranium assay results.

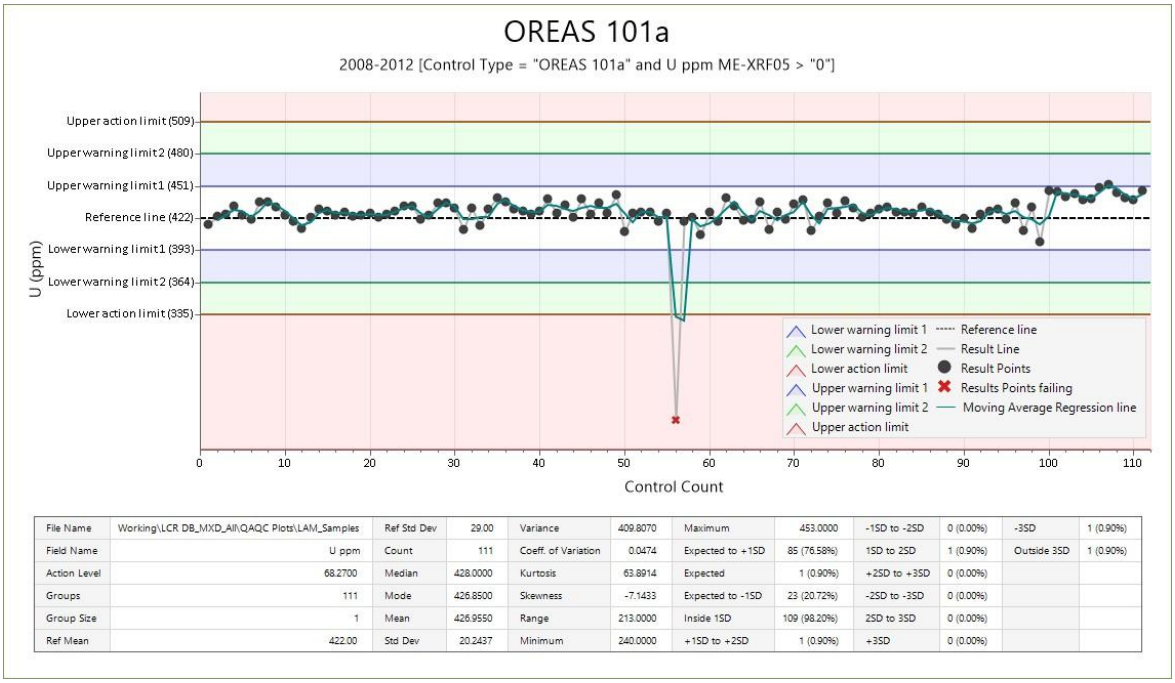
Table 11.2.3 Summary of LAM's Certified Standards from 2008 to 2012

Standard	CRM		Control Assays Results	
	Certified Value* U ppm	2 x Standard Dev. Certified Value* U ppm	Mean U ppm	2 x Standard Dev. U ppm
Oreas 101a	422	58	427.0	40.5
Oreas 101b	396	29	403.8	20.2

***Certified value and Standard Deviation from each CRM reported under Fusion.**

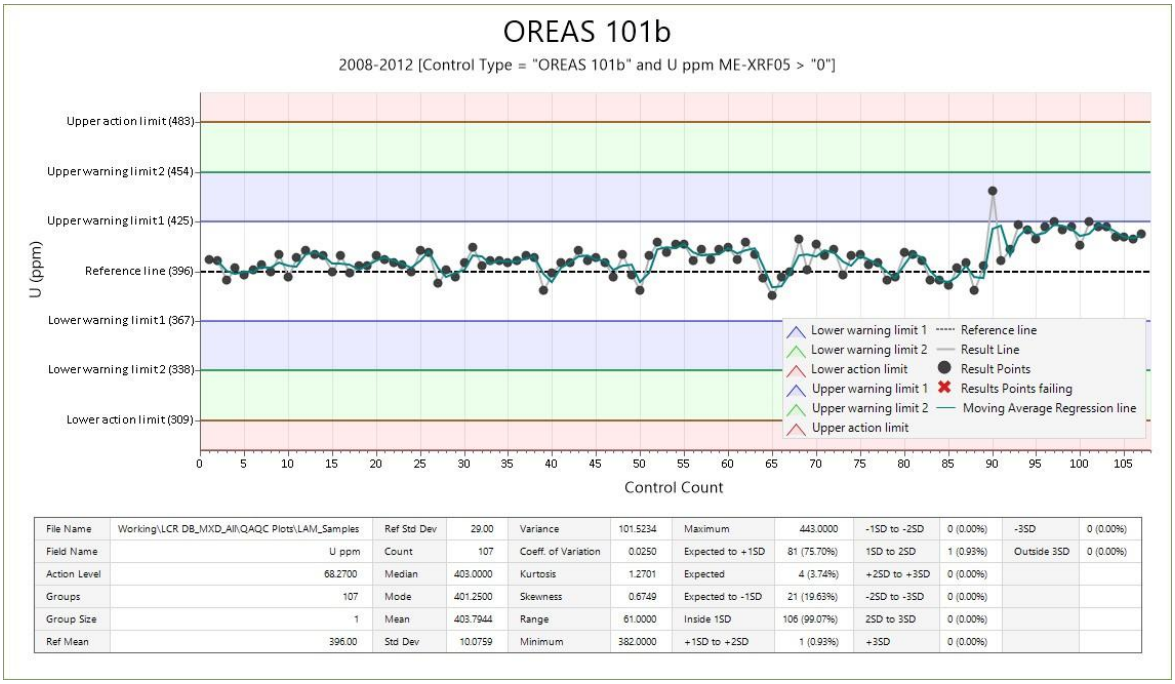
Assay results for submitted standards from 2008 to 2012 are provided in Figure 11.2.1 and Figure 11.2.2. The certified value is shown as a dashed black line and the blue, green and red lines indicate one, two and three standard deviations from the mean value, respectively.

Figure 11.2.1 Results for Standard 101a



File Name	Working\LCR DB_MXD_All\QAQC Plots\LAM_Samples	Ref Std Dev	29.00	Variance	409.8070	Maximum	453.0000	-1SD to -2SD	0 (0.00%)	-3SD	1 (0.90%)
Field Name	U ppm	Count	111	Coeff. of Variation	0.0474	Expected to +1SD	85 (76.58%)	1SD to 2SD	1 (0.90%)	Outside 3SD	1 (0.90%)
Action Level	68.2700	Median	428.0000	Kurtosis	63.8914	Expected	1 (0.90%)	+2SD to +3SD	0 (0.00%)		
Groups	111	Mode	426.8500	Skewness	-7.1433	Expected to -1SD	23 (20.72%)	-2SD to -3SD	0 (0.00%)		
Group Size	1	Mean	426.9550	Range	213.0000	Inside 1SD	109 (98.20%)	2SD to 3SD	0 (0.00%)		
Ref Mean	422.00	Std Dev	20.2437	Minimum	240.0000	+1SD to +2SD	1 (0.90%)	+3SD	0 (0.00%)		

Figure 11.2.2 Results for Standard 101b



File Name	Working\LCR DB_MXD_All\QAQC Plots\LAM_Samples	Ref Std Dev	29.00	Variance	101.5234	Maximum	443.0000	-1SD to -2SD	0 (0.00%)	-3SD	0 (0.00%)
Field Name	U ppm	Count	107	Coeff. of Variation	0.0250	Expected to +1SD	81 (75.70%)	1SD to 2SD	1 (0.93%)	Outside 3SD	0 (0.00%)
Action Level	68.2700	Median	403.0000	Kurtosis	1.2701	Expected	4 (3.74%)	+2SD to +3SD	0 (0.00%)		
Groups	107	Mode	401.2500	Skewness	0.6749	Expected to -1SD	21 (19.63%)	-2SD to -3SD	0 (0.00%)		
Group Size	1	Mean	403.7944	Range	61.0000	Inside 1SD	106 (99.07%)	2SD to 3SD	0 (0.00%)		
Ref Mean	396.00	Std Dev	10.0759	Minimum	382.0000	+1SD to +2SD	1 (0.93%)	+3SD	0 (0.00%)		

Generally, the LAM-submitted standards returned excellent accuracy during 2008 to 2012, with the average grade well within the specified range of the certified grade, indicating reliability in the assay method.

2022 to 2024

Oreas 101a and Oreas 101b were unavailable for purchase at the beginning of the 2022 field season, and suitable alternatives were sourced from Oreas, Bays Water North, Australia (Table 11.2.4). Laramide now has a selection of five uranium CRMs in use across a range of grades. Table 11.2.4 summarizes the statistics of the analysis of these CRMs. All CRMs from 2022 to 2024 were reported by ALS under their 4-acid digest ME-MS61 method of analysis.

Table 11.2.4 Summary of LAM's Certified Standards from 2022 to 2024

Standard	Count*	CRM		Control Assays Results	
		Certified Value# ppm	2 x Standard Dev. Certified Value* ppm	Mean U ppm	2 x Standard Dev. ppm
Oreas 120	50	39.6 U	3.02 U	40.2	4.34
Oreas 121	47	206 U	14.0 U	208.1	14.64
Oreas 123	50	825 U	70.0 U	823.5	62.52
Oreas 124	33	1,779 U	180 U	1,754.7	148.70
Oreas 233b	44	1.07 Au	0.086 Au	1.07	0.06

***CRMs used in drill programs contributing to the resource update.**

#All uranium standards report CVs under 4-Acid Digestion. Oreas 233b (Au) reports to a Pb-Fire Assay CV.

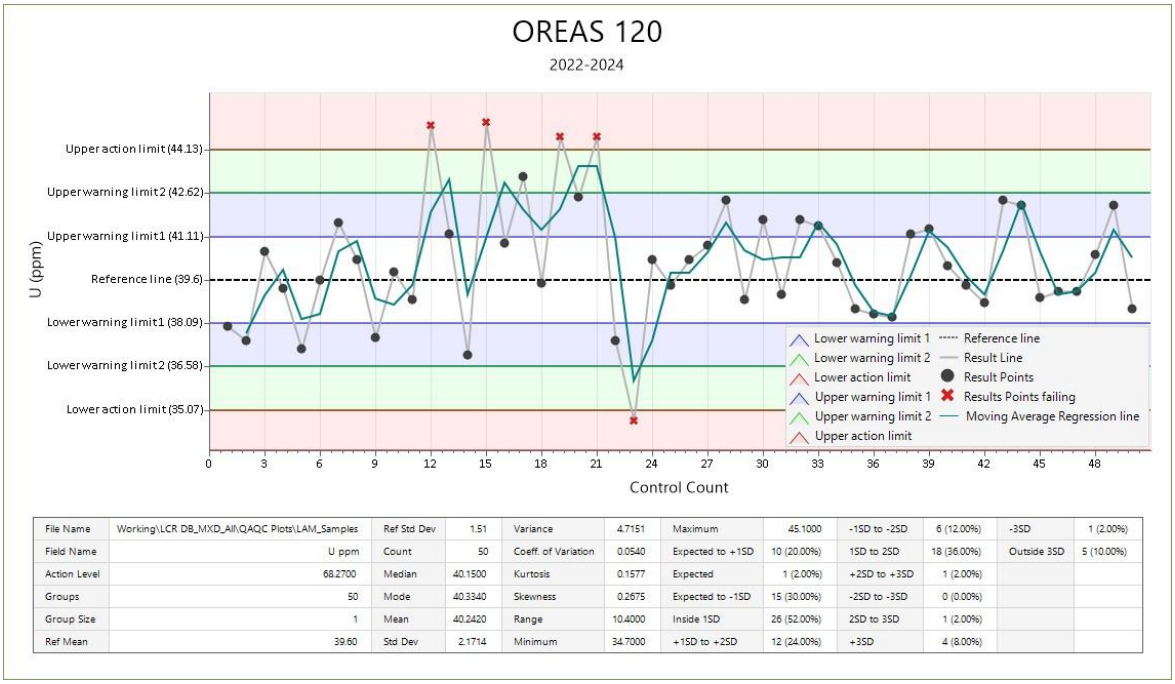
Assay results for submitted standards from 2022 to 2024 are provided in Figure 11.2.3 to Figure 11.2.7. The certified value in each case, is shown as a dashed black line and the blue, green and red lines indicate one, two and three standard deviations from the mean value, respectively. Generally, each CRM has reported an average grade within the specified range of the certified grade.

Oreas 120 (50) has seen five instances where the CRM failed (reported more than 3SD from the CV). However, due to the nature of this low-grade CRM (39.6 ppm U) and the tight SD increments attributed (1.51 ppm U), it has never warranted further investigations. Broadly, the CRM has performed well, with a mean assay result of 40.24 ppm U, which is within the specified range of the certified grade.

All other CRMs have not observed any reported failures. Oreas 121 (47), which represents a near cut-off grade of 206 ppm U, observed a very modest positive bias which equates to less than 5 ppm U on average (mean assay results 208 ppm U), and so is not considered a cause for concern for the reliability of the assay method. There have been four instances of the CRM reporting beyond 2SD but not beyond 3SD from the CV, and no further investigations were required.

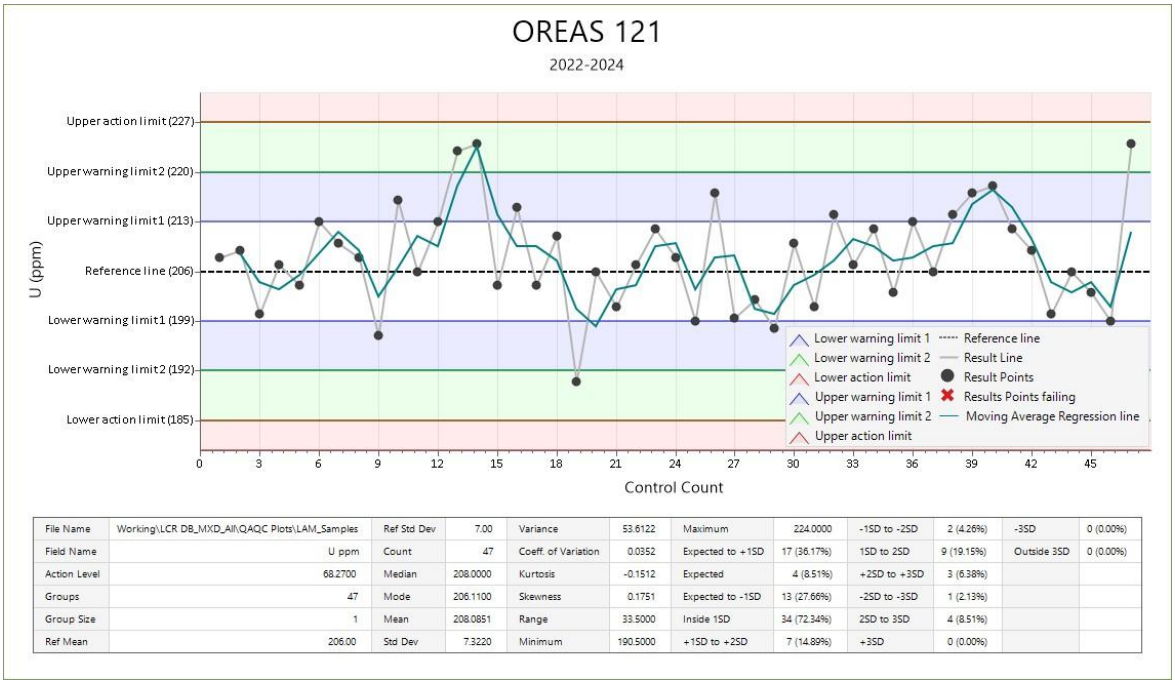
Generally, Oreas 123 reported well, with only three instances of the CRM reporting beyond 2SD but not beyond 3SD from the CV, and no further investigations were required. Oreas 124 appears to slightly under report with a control assay average of 1,754.6 ppm U. This under reporting equates to less than 25 ppm U variance, and has not been considered a cause for concern for the reliability of the assay method.

Figure 11.2.3 Results for Standard 120



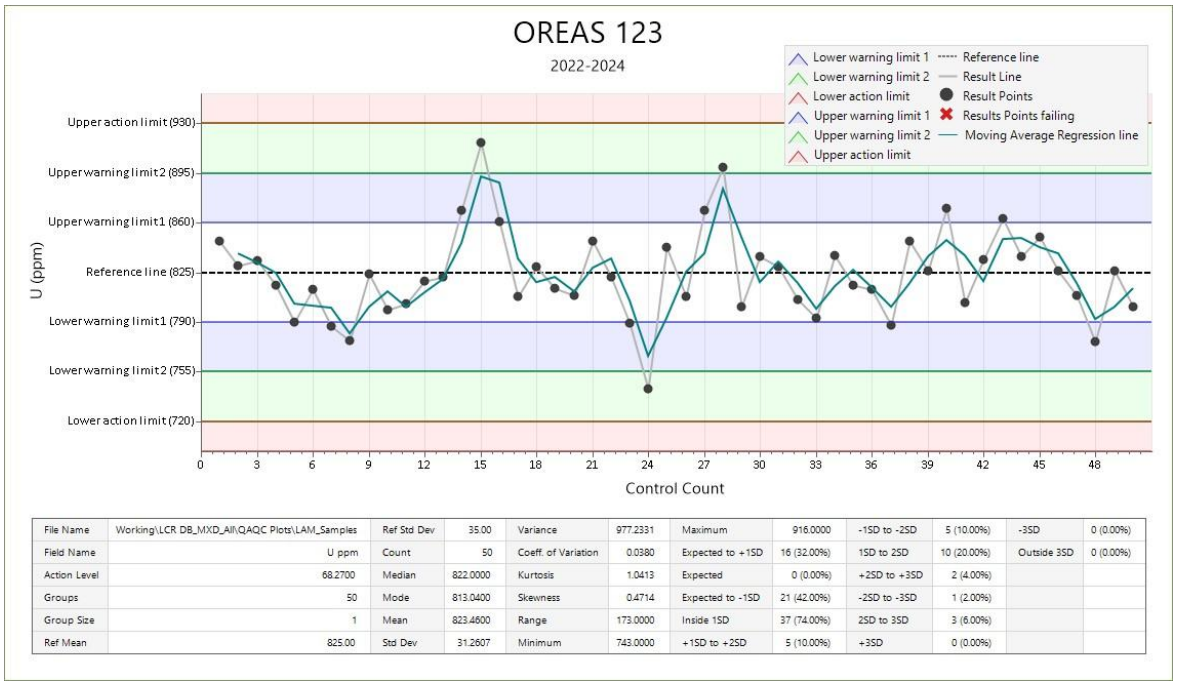
File Name	Working\LCR DB_MXD_All\QAQC Plots\LAM_Samples	Ref Std Dev	1.51	Variance	4.7151	Maximum	45.1000	-1SD to -2SD	6 (12.00%)	-3SD	1 (2.00%)
Field Name	U ppm	Count	50	Coeff. of Variation	0.0540	Expected to +1SD	10 (20.00%)	1SD to 2SD	18 (36.00%)	Outside 3SD	5 (10.00%)
Action Level	68.2700	Median	40.1500	Kurtosis	0.1577	Expected	1 (2.00%)	+2SD to +3SD	1 (2.00%)		
Groups	50	Mode	40.3340	Skewness	0.2675	Expected to -1SD	15 (30.00%)	-2SD to -3SD	0 (0.00%)		
Group Size	1	Mean	40.2420	Range	10.4000	Inside 1SD	26 (52.00%)	2SD to 3SD	1 (2.00%)		
Ref Mean	39.60	Std Dev	2.1714	Minimum	34.7000	+1SD to +2SD	12 (24.00%)	+3SD	4 (8.00%)		

Figure 11.2.4 Results for Standard 121



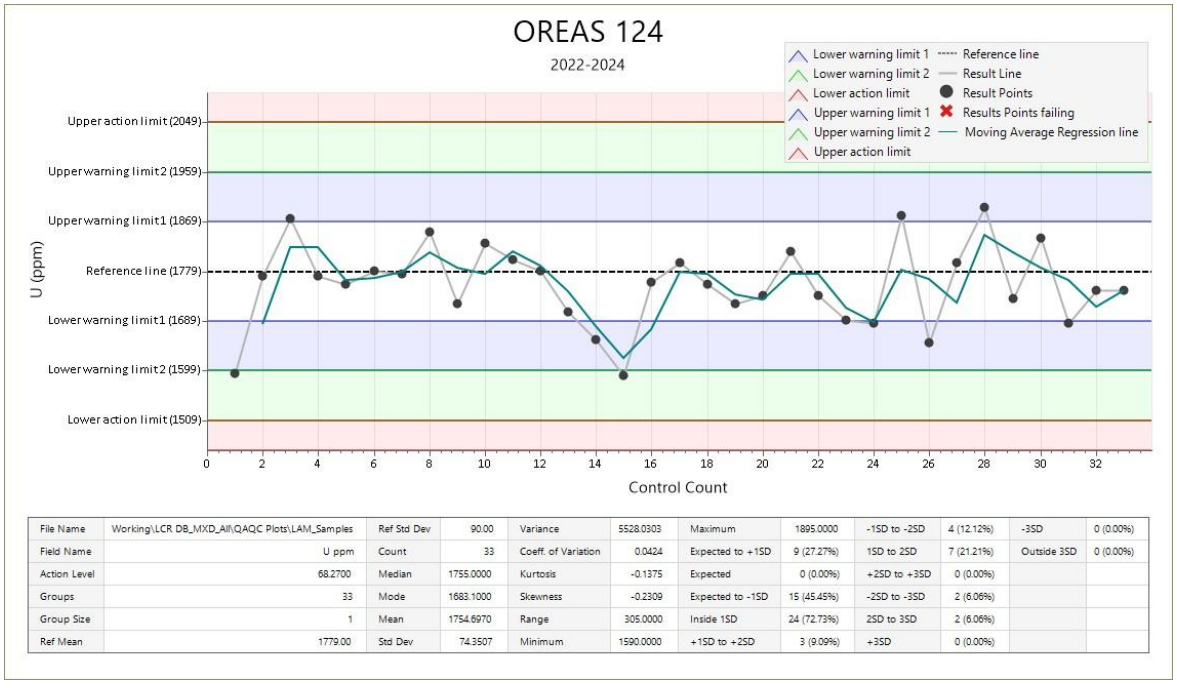
File Name	Working\LCR DB_MXD_All\QAQC Plots\LAM_Samples	Ref Std Dev	7.00	Variance	53.6122	Maximum	224.0000	-1SD to -2SD	2 (4.26%)	-3SD	0 (0.00%)
Field Name	U ppm	Count	47	Coeff. of Variation	0.0352	Expected to +1SD	17 (36.17%)	1SD to 2SD	9 (19.15%)	Outside 3SD	0 (0.00%)
Action Level	68.2700	Median	208.0000	Kurtosis	-0.1512	Expected	4 (8.51%)	+2SD to +3SD	3 (6.38%)		
Groups	47	Mode	206.1100	Skewness	0.1751	Expected to -1SD	13 (27.66%)	-2SD to -3SD	1 (2.13%)		
Group Size	1	Mean	208.0851	Range	33.5000	Inside 1SD	34 (72.34%)	2SD to 3SD	4 (8.51%)		
Ref Mean	206.00	Std Dev	7.3220	Minimum	190.5000	+1SD to +2SD	7 (14.89%)	+3SD	0 (0.00%)		

Figure 11.2.5 Results for Standard 123



File Name	Working\LCR DB_MXD_All\QAQC Plots\LAM_Samples	Ref Std Dev	35.00	Variance	977.2331	Maximum	916.0000	-1SD to -2SD	5 (10.00%)	-3SD	0 (0.00%)
Field Name	U ppm	Count	50	Coeff. of Variation	0.0380	Expected to +1SD	16 (32.00%)	1SD to 2SD	10 (20.00%)	Outside 3SD	0 (0.00%)
Action Level	68.2700	Median	822.0000	Kurtosis	1.0413	Expected	0 (0.00%)	+2SD to +3SD	2 (4.00%)		
Groups	50	Mode	813.0400	Skewness	0.4714	Expected to -1SD	21 (42.00%)	-2SD to -3SD	1 (2.00%)		
Group Size	1	Mean	823.4600	Range	173.0000	Inside 1SD	37 (74.00%)	2SD to 3SD	3 (6.00%)		
Ref Mean	825.00	Std Dev	31.2607	Minimum	743.0000	+1SD to +2SD	5 (10.00%)	+3SD	0 (0.00%)		

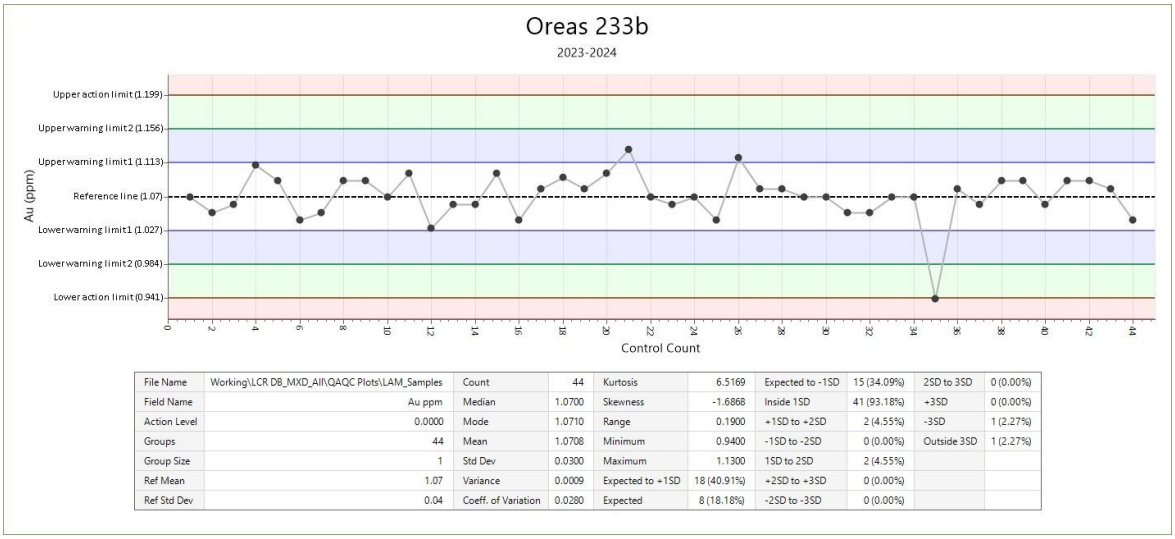
Figure 11.2.6 Results for Standard 124



File Name	Working\LCR DB_MXD_All\QAQC Plots\LAM_Samples	Ref Std Dev	90.00	Variance	5528.0303	Maximum	1895.0000	-1SD to -2SD	4 (12.12%)	-3SD	0 (0.00%)
Field Name	U ppm	Count	33	Coeff. of Variation	0.0424	Expected to +1SD	9 (27.27%)	1SD to 2SD	7 (21.21%)	Outside 3SD	0 (0.00%)
Action Level	68.2700	Median	1755.0000	Kurtosis	-0.1375	Expected	0 (0.00%)	+2SD to +3SD	0 (0.00%)		
Groups	33	Mode	1683.1000	Skewness	-0.2309	Expected to -1SD	15 (45.45%)	-2SD to -3SD	2 (6.06%)		
Group Size	1	Mean	1754.6970	Range	305.0000	Inside 1SD	24 (72.73%)	2SD to 3SD	2 (6.06%)		
Ref Mean	1779.00	Std Dev	74.3507	Minimum	1590.0000	+1SD to +2SD	3 (9.09%)	+3SD	0 (0.00%)		

Oreas 233b has been used as a gold CRM, which has only one instance of reporting more than 2SD but less than 3SD from the CV. The CRM has performed well, with a mean assay result of 1.071 ppm Au, which is within the specified range of the certified grade, indicating reliability in the fire assay method used for Au.

Figure 11.2.7 Results for Standard 233b



File Name	Working\LCR DB_MXD_All\QAQC Plots\LAM_Samples	Count	44	Kurtosis	6.5169	Expected to -1SD	15 (34.09%)	2SD to 3SD	0 (0.00%)
Field Name	Au ppm	Median	1.0700	Skewness	-1.6868	Inside 1SD	41 (93.18%)	+3SD	0 (0.00%)
Action Level	0.0000	Mode	1.0710	Range	0.1900	+1SD to +2SD	2 (4.55%)	-3SD	1 (2.27%)
Groups	44	Mean	1.0708	Minimum	0.9400	-1SD to -2SD	0 (0.00%)	Outside 3SD	1 (2.27%)
Group Size	1	Std Dev	0.0300	Maximum	1.1300	1SD to 2SD	2 (4.55%)		
Ref Mean	1.07	Variance	0.0009	Expected to +1SD	18 (40.91%)	+2SD to +3SD	0 (0.00%)		
Ref Std Dev	0.04	Coeff. of Variation	0.0280	Expected	8 (18.18%)	-2SD to -3SD	0 (0.00%)		

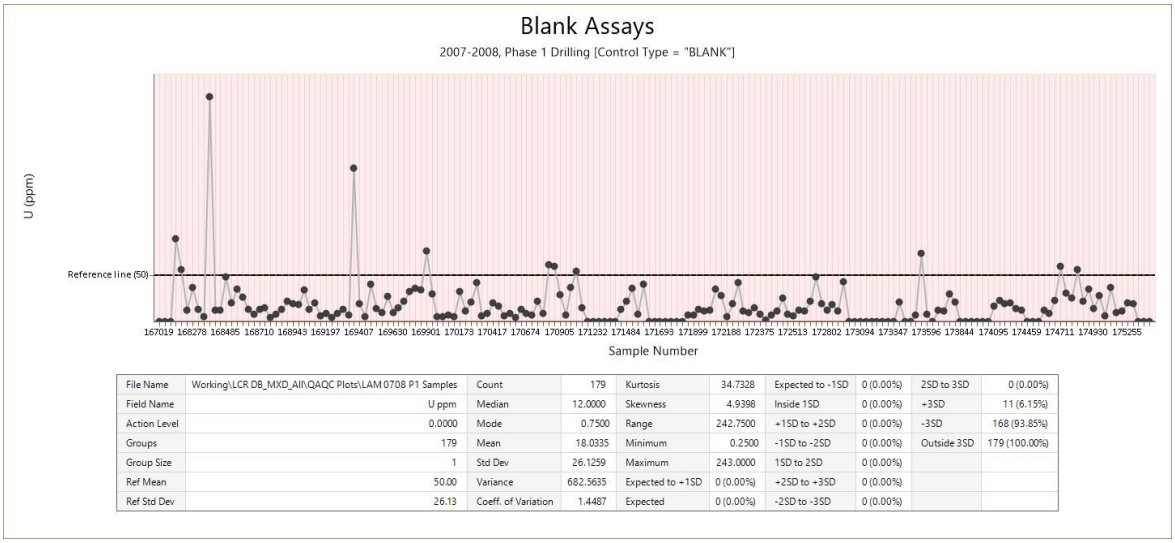
Blanks

2007 to 2012

From 2007 to 2012, blanks were prepared from half core samples obtained from exploration drillholes of equivalent material (Westmoreland Conglomerate). Blanks of this nature were selected using a scintillometer to obtain samples having less than 50 ppm U (56 ppm U_3O_8). Samples were then stored in a drum before being inserted into the sampling sequence. Blanks were generally inserted within the mineralized intervals of each drillhole.

A total of 179 blanks were analyzed during the first phase of drilling 2007 to 2008 (Figure 11.2.8). Some 92% of the blanks returned values less than 50 ppm U. Eleven samples returned values greater than 50 ppm U. Six of these results were returned from the first two sample batches. Investigation of these results by re-assay of coarse splits and examination of samples indicated that some weakly mineralized material had been included in the blanks drum. Subsequently, each blank sample was screened individually before being sent to the laboratory.

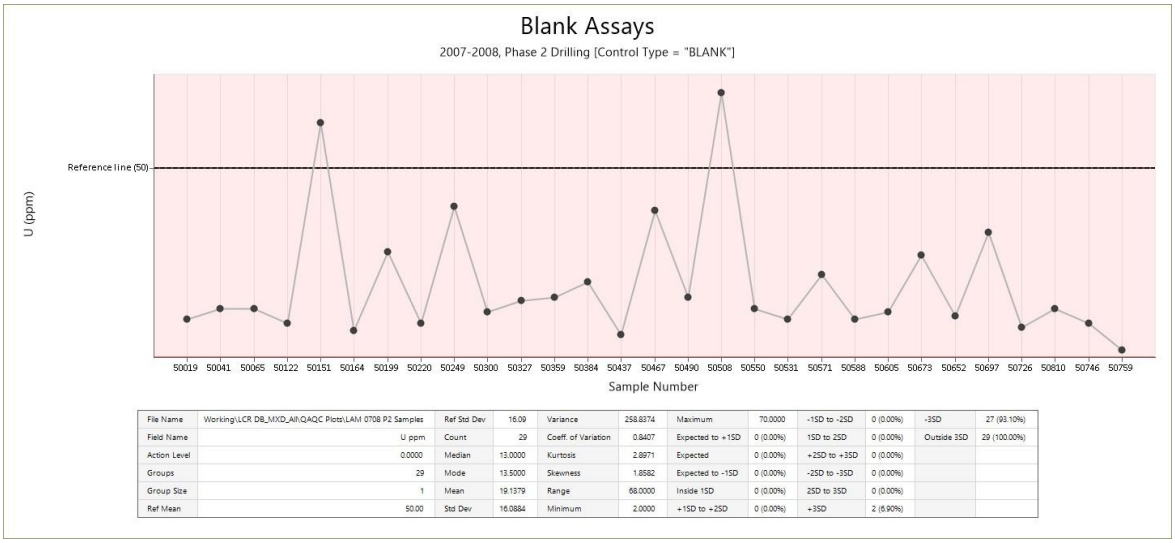
Figure 11.2.8 Phase 1, Drilling 2007 to 2008, Blank Assay Results



File Name	Working\LCR DB_MXD_All\QAQC Plots\LAM 0708 P1 Samples	Count	179	Kurtosis	34.7328	Expected to -1SD	0 (0.00%)	2SD to 3SD	0 (0.00%)
Field Name	U ppm	Median	12.0000	Skewness	4.9398	Inside 1SD	0 (0.00%)	+3SD	11 (6.15%)
Action Level	0.0000	Mode	0.7500	Range	242.7500	+1SD to +2SD	0 (0.00%)	-3SD	168 (93.85%)
Groups	179	Mean	18.0335	Minimum	0.2500	-1SD to -2SD	0 (0.00%)	Outside 3SD	179 (100.00%)
Group Size	1	Std Dev	26.1259	Maximum	243.0000	1SD to 2SD	0 (0.00%)		
Ref Mean	50.00	Variance	682.5635	Expected to +1SD	0 (0.00%)	+2SD to +3SD	0 (0.00%)		
Ref Std Dev	26.13	Coeff. of Variation	1.4487	Expected	0 (0.00%)	-2SD to -3SD	0 (0.00%)		

The second phase of drilling in 2007 to 2008, aimed to use blank material with concentrations closer to the detection limit (Figure 11.2.9). Core consisting of sandstone and conglomerate of the Westmoreland Conglomerate was selected from exploration drillholes and 29 representative samples sent for analysis. A comparison of blanks assayed during the second phase of drilling with the previous sample in the sequence was undertaken (Figure 11.2.10).

Figure 11.2.9 Phase 2, Drilling 2007 to 2008, Blank Assay Results



File Name	Working\LCR DB_MXD_All\QAQC Plots\LAM 0708 P2 Samples	Ref Std Dev	16.09	Variance	258.8374	Maximum	70.0000	-1SD to -2SD	0 (0.00%)	-3SD	27 (93.10%)
Field Name	U ppm	Count	29	Coeff. of Variation	0.8407	Expected to +1SD	0 (0.00%)	1SD to 2SD	0 (0.00%)	Outside 3SD	29 (100.00%)
Action Level	0.0000	Median	13.0000	Kurtosis	2.8971	Expected	0 (0.00%)	+2SD to +3SD	0 (0.00%)		
Groups	29	Mode	13.5000	Skewness	1.8582	Expected to -1SD	0 (0.00%)	-2SD to -3SD	0 (0.00%)		
Group Size	1	Mean	19.1379	Range	68.0000	Inside 1SD	0 (0.00%)	2SD to 3SD	0 (0.00%)		
Ref Mean	50.00	Std Dev	16.0884	Minimum	2.0000	+1SD to +2SD	0 (0.00%)	+3SD	2 (6.90%)		

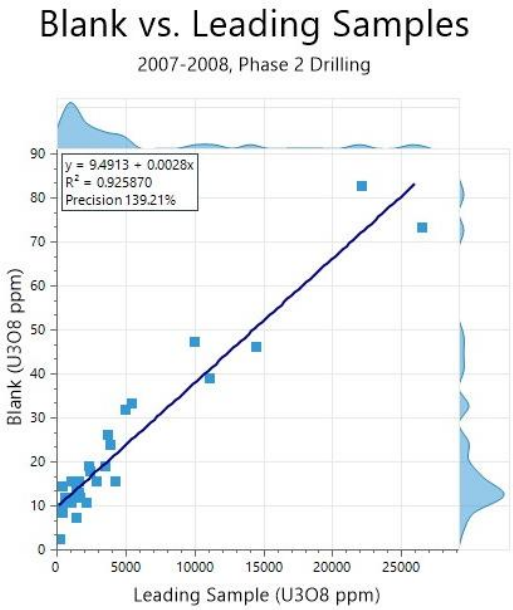
The comparison shows very low-level contamination for blanks inserted after samples having a grade less than 4,000 ppm U_3O_8. For these samples the level of contamination is considered to be less than 10 ppm U_3O_8.

For blanks inserted following samples having a grade of about:

- 5,000 ppm U_3O_8 there is about 35 ppm (0.6%) contamination.
- 10,000 ppm U_3O_8 there is about 45 ppm (0.5%) contamination.
- 20,000 ppm U_3O_8 (i.e. 2% grade) is about 50 to 75 ppm (0.4%) contamination.

It should be noted that the precision of the analytical technique for grades above 10,000 ppm is 100 ppm, which is greater than the indicated level of contamination.

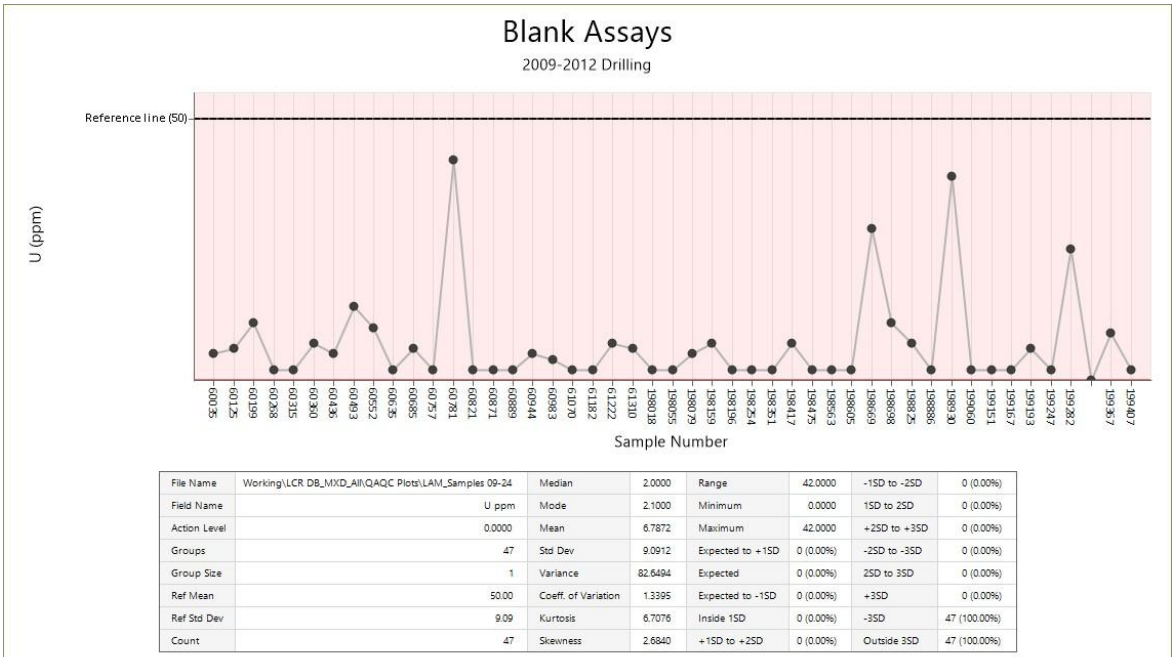
Figure 11.2.10 Low Level Contamination in Blanks, Phase 2 Drilling, 2007 to 2008



Number of Pairs	29	Corr Coeff (R)	0.962	**X: U3O8 ppm Leading**		Variance	41670517	Min	2.358
Model Type	Linear	R^2	0.926	Count	29	Std Dev	6455	Max	82.544
Model	$y = b_0 + b_1x$	Rank Corr Coef (%)	0.898	Mean	4624.009	**Y: U3O8 ppm Blank**		Variance	360
b0	9.491298	Precision %	139%	Min	227.586	Count	29	Std Dev	19
b1	0.0028279	X Var / Y Var	115778	Max	26532.000	Mean	22.567		

The method of utilizing half core samples obtained from exploration drillholes of equivalent material was used up until 2012, with 47 blanks inserted during these programs (Figure 11.2.11).

Figure 11.2.11 Drilling 2009 to 2012, Blank Assay Results



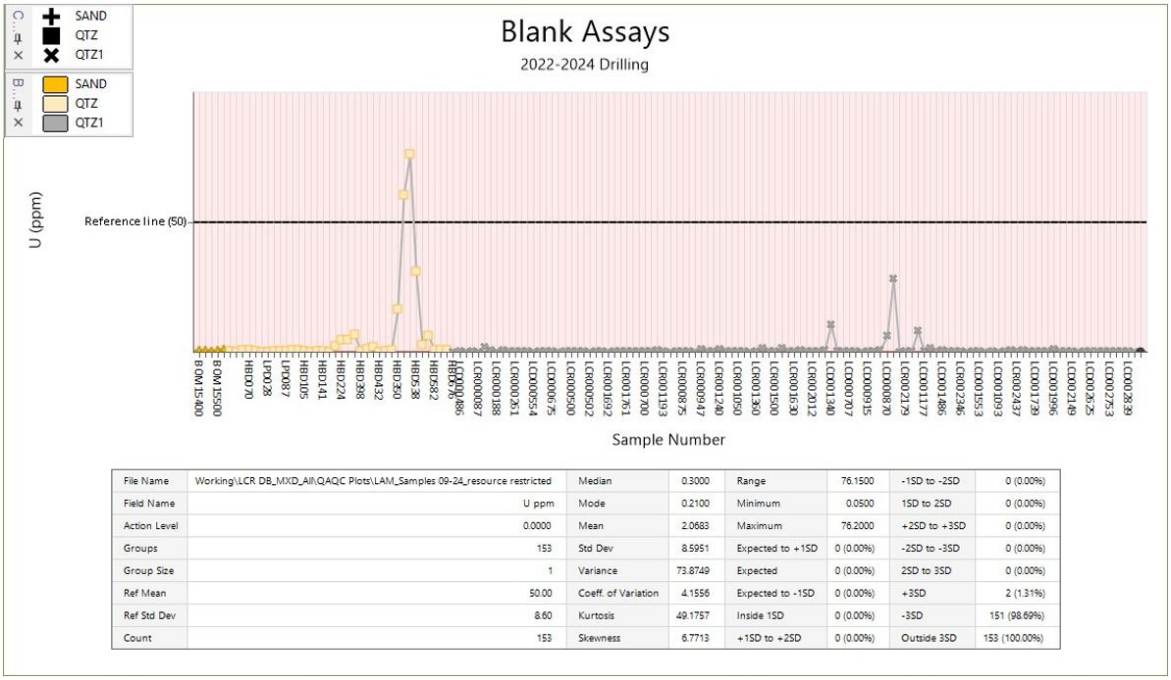
File Name	Working\LCR DB_MXD_All\QAQC Plots\LAM_Samples 09-24	Median	2.0000	Range	42.0000	-1SD to -2SD	0 (0.00%)
Field Name	U ppm	Mode	2.1000	Minimum	0.0000	1SD to 2SD	0 (0.00%)
Action Level	0.0000	Mean	6.7872	Maximum	42.0000	+2SD to +3SD	0 (0.00%)
Groups	47	Std Dev	9.0912	Expected to +1SD	0 (0.00%)	-2SD to -3SD	0 (0.00%)
Group Size	1	Variance	82.6494	Expected	0 (0.00%)	2SD to 3SD	0 (0.00%)
Ref Mean	50.00	Coeff. of Variation	1.3395	Expected to -1SD	0 (0.00%)	+3SD	0 (0.00%)
Ref Std Dev	9.09	Kurtosis	6.7076	Inside 1SD	0 (0.00%)	-3SD	47 (100.00%)
Count	47	Skewness	2.6840	+1SD to +2SD	0 (0.00%)	Outside 3SD	47 (100.00%)

In summary, the analysis of blank / low grade samples submitted throughout the 2007 to 2012 drilling programs indicates that laboratory cross sample contamination is within acceptable limits and considered inconsequential.

2022 to 2024

In 2022, unconsolidated play sand acquired from a local homeware store was utilized as a blank material (SAND). It was recognized that unconsolidated sands would be unlikely to pass through the first mass reduction phase at the laboratory's preparation facilities, and so would not be testing the whole circuit. In 2023, a new coarse pebble quartz material was acquired (QTZ). Due to supply limitations, in 2024 a new pebble coarse blank material was sourced from a new provider (QTZ1). A total of 153 blanks were inserted across the various drill programs from 2022 to 2024, 96.7% of which reported <10 ppm U (Figure 11.2.12).

Figure 11.2.12 Drilling 2022 to 2024, Blank Assay Results



File Name	Working\LCR DB_MXD_All\QAQC Plots\LAM_Samples 09-24_resource restricted	Median	0.3000	Range	76.1500	-1SD to -2SD	0 (0.00%)
Field Name	U ppm	Mode	0.2100	Minimum	0.0500	1SD to 2SD	0 (0.00%)
Action Level	0.0000	Mean	2.0683	Maximum	76.2000	+2SD to +3SD	0 (0.00%)
Groups	153	Std Dev	8.5951	Expected to +1SD	0 (0.00%)	-2SD to -3SD	0 (0.00%)
Group Size	1	Variance	73.8749	Expected	0 (0.00%)	2SD to 3SD	0 (0.00%)
Ref Mean	50.00	Coeff. of Variation	4.1556	Expected to -1SD	0 (0.00%)	+3SD	2 (1.31%)
Ref Std Dev	8.60	Kurtosis	49.1757	Inside 1SD	0 (0.00%)	-3SD	151 (98.69%)
Count	153	Skewness	6.7713	+1SD to +2SD	0 (0.00%)	Outside 3SD	153 (100.00%)

Coarse blank control samples reporting greater than 5 ppm U were reviewed on a case-by-case basis, taking into consideration the blank sample size and leading sample grades. In each case, the preceding three leading sample grades and weights were reviewed under a 1% sample preparation carry-over limit as per the equation below. The same formula is used by ALS to discern acceptable carryover and/or trigger further in-laboratory investigations where required.

$$Blank\ Result = \frac{((Result\ prior\ to\ sample\ \times Carry\ over\ weight) + (Expected\ blank\ results\ \times Blank\ sample\ weight))}{(Carry\ over\ weight + Blank\ sample\ weight)}$$

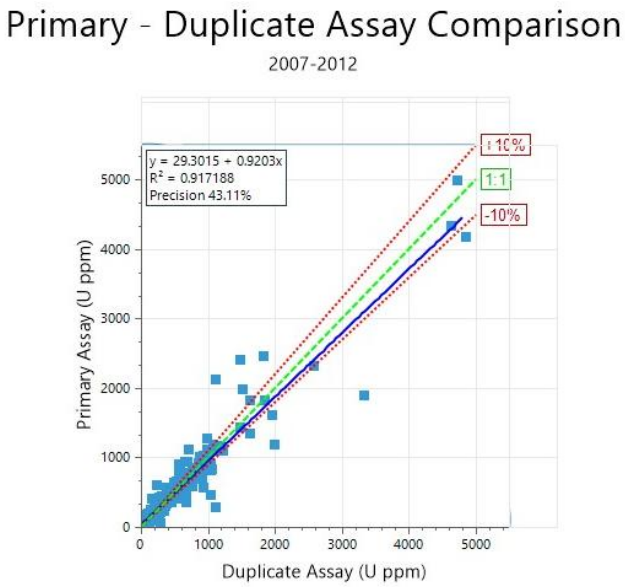
In summary, the analysis of blank / low grade samples submitted throughout the 2022 to 2024 drilling programs indicates that laboratory cross sample contamination is within acceptable limits and considered inconsequential.

Duplicates

2007 to 2012

A total of 294 primary duplicates were sampled during the 2007 to 2012 drilling programs (Figure 11.2.13), which displays the greatest variation in paired results, this is not unexpected and is within accepted tolerances. Primary duplicates were made up of the remaining ½ core, after the primary sample was collected.

Figure 11.2.13 Primary Duplicate Assay Comparison, 2007 to 2012

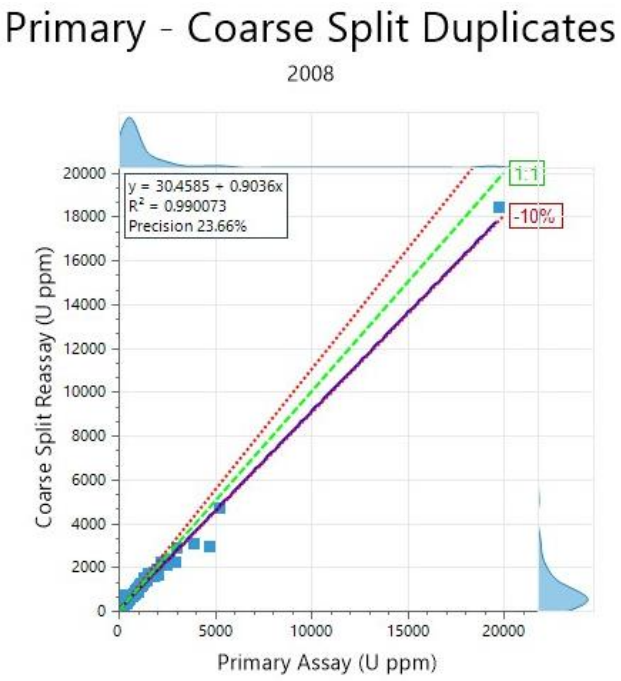


Number of Pairs	294	Corr Coeff (R)	0.958	**X: Duplicate sample result**		Variance	384867	Min	8.0
Model Type	Linear	R^2	0.917	Count	294	Std Dev	620	Max	4980.0
Model	$y = b_0 + b_1x$	Rank Corr Coef (%)	0.964	Mean	414.2	**Y: Original sample result**		Variance	355406
b0	29.30149	Precision %	43%	Min	8.0	Count	294	Std Dev	596
b1	0.9203141	X Var / Y Var	1.08	Max	4860.0	Mean	410.5		

Coarse Split Residue Duplicates

A total of 100 samples of retained coarse material from initial jaw crushing were resubmitted as duplicates from 2008 drillholes. The comparison of uranium assays between the primary and sample and the coarse split duplicate is presented in the following graph (Figure 11.2.14). The comparison shows a generally good correlation indicating good repeatability. The regression line indicates a positive bias towards the primary sample over the coarse split residue re-assayed. However, it is thought this has been skewed by a handful of high-grade samples and is not considered significant.

Figure 11.2.14 Comparison of Coarse Split Duplicates, 2008



Number of Pairs	100	Corr Coeff (R)	0.995	**X: Parent U ppm**		Variance	4416984	Min	24
Model Type	Linear	R^2	0.99	Count	100	Std Dev	2102	Max	18400
Model	$y = b_0 + b_1x$	Rank Corr Coef (%)	0.976	Mean	1174.18	**Y: U ppm**		Variance	3642615
b0	30.45854	Precision %	24%	Min	23	Count	100	Std Dev	1909
b1	0.9036021	X Var / Y Var	1.21	Max	19800	Mean	1091.45		

2022 to 2024

During the 2022 to 2024 field programs, 132 duplicates pairs were nominated across each of the mass reduction phases including the ALS processing circuit; field, coarse or pulp duplicates.

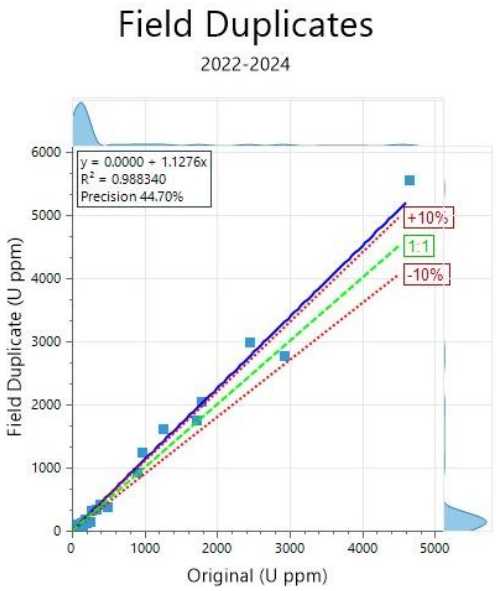
Field Duplicates

A total of 61 primary field duplicates were sampled during the 2022 to 2024 drilling programs (Figure 11.17). The field duplicate pairs display the greatest variation in paired results, which is not unexpected and is only just beyond a +10% tolerance. Primary field duplicates were made up of the ¼ cut core, after the primary sample was collected.

Coarse Split Residue Duplicates

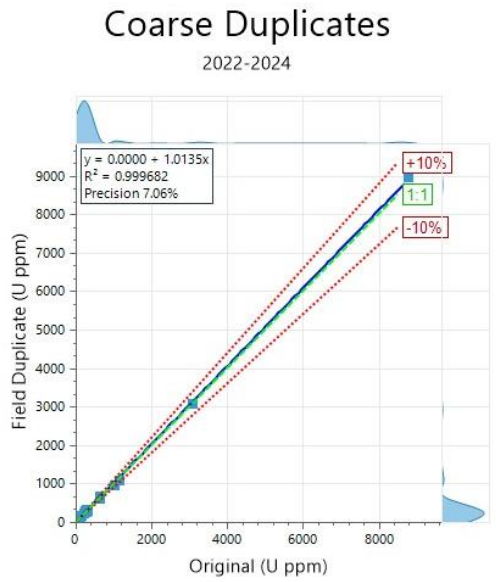
A total of 36 coarse split residue duplicates were sampled during the 2022 to 2024 drilling programs (Figure 11.18). Samples of retained coarse material from initial jaw crushing were resubmitted as duplicates. The coarse duplicate pairs from 2022 to 2024 indicate strong repeatability and in hand strong laboratory performance at their first mass reduction phase in the ALS processing circuit.

Figure 11.2.15 Primary Field Duplicate Assay Comparison, 2022 to 2024



Number of Pairs	61	Corr Coeff (R)	0.994	**X: Original sample result**		Variance	664765	Min	1.2
Model Type	Linear	R^2	0.988	Count	61	Std Dev	815	Max	5540.0
Model	$y = b_0 + b_1x$	Rank Corr Coef (%)	0.991	Mean	334.2	**Y: Duplicate sample result**		Variance	867205
b0	0	Precision %	45%	Min	1.1	Count	61	Std Dev	931
b1	1.127557	X Var / Y Var	0.767	Max	4650.0	Mean	361.3		

Figure 11.2.16 Comparison of Coarse Split Duplicates, 2022 to 2024

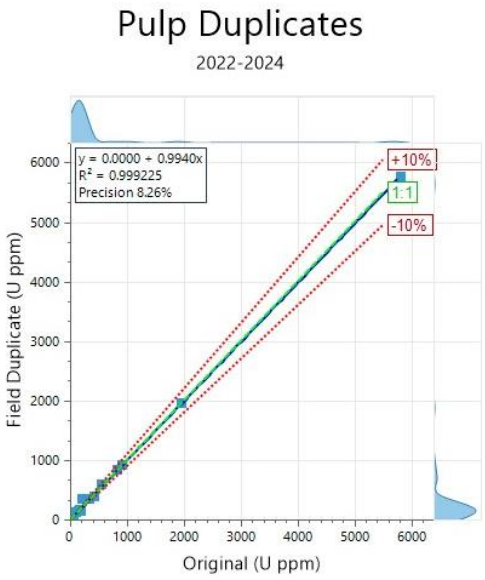


Number of Pairs	36	Corr Coeff (R)	1	**X: Original sample result**		Variance	2317467	Min	2.6
Model Type	Linear	R^2	1	Count	36	Std Dev	1522	Max	8950.0
Model	$y = b_0 + b_1x$	Rank Corr Coef (%)	0.996	Mean	511.0	**Y: Duplicate sample result**		Variance	2389735
b0	0	Precision %	7.06%	Min	2.6	Count	36	Std Dev	1546
b1	1.013467	X Var / Y Var	0.97	Max	8780.0	Mean	509.8		

Pulp Duplicates

A total of 35 pulp duplicates were tested during the 2022 to 2024 drill programs. A second aliquot of the pulp material was spilt at the final mass reduction phase, to make a comparison between two pulp duplicates and test the laboratory's analytical reproducibility. The comparison between these pulp duplicate pairs is presented in Figure 11.19. The comparison indicates a strong performance in the laboratory's analytical precision under the ME-MS61 method of analysis for uranium.

Figure 11.2.17 Comparison of Pulp Duplicates, 2022 to 2024

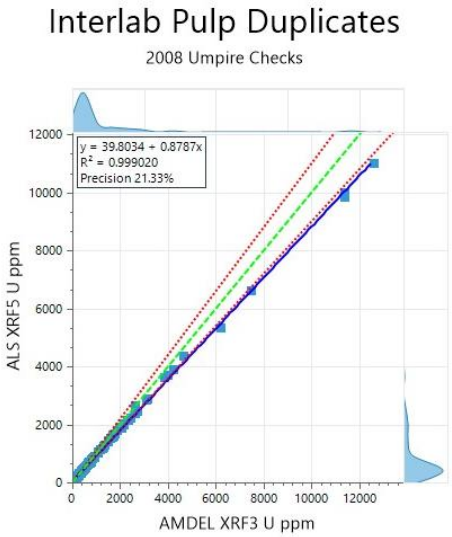


Number of Pairs	35	Corr Coeff (R)	1	**X: Original sample result**		Variance	1044153	Min	2.6
Model Type	Linear	R^2	0.999	Count	35	Std Dev	1022	Max	5760.0
Model	$y = b_0 + b_1x$	Rank Corr Coef (%)	0.998	Mean	354.5	**Y: Duplicate sample result**		Variance	1029169
b0	0	Precision %	8.26%	Min	2.7	Count	35	Std Dev	1014
b1	0.9939879	X Var / Y Var	1.01	Max	5810.0	Mean	356.9		

Inter-Laboratory Checks

One hundred and forty-four pulp duplicates originally sent to ALS (Mt Isa) during the 2007 to 2008 drilling program were re-submitted to AMDEL Laboratories in Mount Isa for uranium analysis (XRF). A comparison of the two sets of analyses is presented in Figure 11.20. A review of these 144 inter-laboratory checks indicates that ALS potentially under reports in comparison to AMDEL. However, given the performance of all other controls, it was not considered that this variance warranted further investigations but has been noted, particularly as it is only just beyond a -10% tolerance. No inter-laboratory checks have been conducted on sampled material collected from 2022 to 2024.

Figure 11.2.18 Comparison of Inter-Laboratory Repeats, 2008



Number of Pairs	144	Corr Coeff (R)	1	**X: Parent U ppm**		Variance	3762553	Min	121
Model Type	Linear	R^2	0.999	Count	144	Std Dev	1940	Max	11000
Model	$y = b_0 + b_1x$	Rank Corr Coef (%)	0.999	Mean	1130.8402777777778	**Y: AMDEL XRF3 ppm U**		Variance	2908004
b0	39.80345	Precision %	21%	Min	131	Count	144	Std Dev	1705
b1	0.8787056	X Var / Y Var	1.29	Max	12600	Mean	1033.4791666666667		

11.2.5 Discussion

The QP considers that LAM's exploration programs have been well executed and managed with particular attention to detail. While there are minor signs of contamination in blank material, the concentrations are not of material concern. Precision and accuracy appear suitable for use in the MRE.

12.0 DATA VERIFICATION

Mr. Siddle, the QP, completed a site visit to the project area between 21 and 23 January 2025, and inspected representative sections of drill core, visited rehabilitated drill sites and inspected selected outcrop geology. Mineralization is clearly visible in drill core and inspection of assay results against the drill core with the aid of LAM's Exploranium G-110 scintillometer served to confirm the depth and relative grade of the reported results. The QP has had sight of LAM's laboratory assay certificates, although he has not completed a laboratory inspection. The QP has also been involved in periodic discussion with LAM's exploration team since 2022.

No drill core from previous explorers is available for inspection as it was disposed of by CRA. The QP inspected the Microsoft Access databases which LAM inherited with the property, along with various Company reports. The greatest support for the legacy data was the drilling undertaken by LAM, which serves to confirm the tenor of grade and thickness seen in legacy data. During this time, historic collars at RT were located and surveyed.

The QP worked for LAM as an exploration geologist on the Project in 2007 and 2008, and has firsthand knowledge of the drilling in those periods. He is now independent of the issuer and has been since January 2009.

While there cannot be absolute certainty in the historical data, the QP considers it suitable for use in the MRE.

13.0 METALLURGICAL TESTING

13.1 Introduction

Metallurgical testwork programs and evaluations on the Westmoreland deposit have been conducted by several laboratories over the period since 1989. These programs included work completed by AMDEL (1989), ANSTO (1992, 1993, 1994, 1995 and 2011), JKTech (1993), and SGS (2008). No additional testwork has been undertaken since 2008.

The evaluation and interpretation of results reported here is based largely on the most recent ANSTO 2011 report, with reference to other sources as noted. This work was reviewed by Lycopodium's Process Consultant (Mr. Grenvil Dunn) in 2016. Key relevant aspects of the report, together with Lycopodium's comments, are extracted and included in the sections that follow.

13.2 Previous Metallurgical Testwork

Mr. R.W. Nice (2012) undertook a review of the past testwork performed on the Westmoreland prospect. A brief summary of his findings are presented here.

13.2.1 Comminution

Very little comminution testwork has been done. Some comminution testwork was reported by JKTech 1993 '*Crushing Simulations for a Uranium Heap Leach Project*', JKTech (JKMRC Commercial Division), May 1993. The results presented in Table 13.2.1.

Table 13.2.1 Comminution Test Results

Material	Bond Ball Work Index kWh/t	JK Parameters		
		A	b	Axb
Oxide	17.2	69.8	2.10	146.6
Fresh	19.4	79.0	1.80	142.2

The Bond ball work index (BBWi) values and the JKMRC determined ore impact parameter (Axb) results are contradictory, suggesting that the tests were performed on different samples. In the ANSTO 2011 report, reference is made to the samples as competent ore, producing little fines during crushing, and consequently, for the purpose of the SS the Bond ball mill results have been accepted as representative.

13.2.2 Heap Leaching

Early testwork conducted by ANSTO and AMDEL looked at heap leaching techniques. The use of heap leaching was discounted by LAM for good reason; the extractions of the Fresh material were very low.

Table 13.2.2 Heap Leach Test Results

Material	Leach Time days	Extraction %	Size % -300 µm	Acid kg/t	Peroxide kg/t
Oxide	34	92 – 93	9.3	4.2	0.6
Fresh	44	77 – 78	8.7	12.9	1.1

13.2.3 Agitated Leach Testwork

ANSTO previously carried out extensive agitated leaching testwork and mineralogy on ore samples from several deposits in the Westmoreland area in 1992 to 1995. The testwork conditions used concentrated sulfuric acid (H_2SO_4) as the leachate, and hydrogen peroxide (H_2O_2) as the oxidant. The standard conditions were:

- Temperature: 40°C.
- Hydrogen ion exponent (pH): 1.5.
- Redox potential (ORP): 475 mV.
- Slurry density: 55 percent weight by weight (% w/w).
- Grind size: ~35% -75 µm.

Some optimization tests were conducted looking at changes to each of these variables. The four separate samples tested were a low grade oxide, a low grade fresh, a high grade oxide, and a high grade fresh. The optimization tests were conducted on a 1:1 blend of low and high grade oxide materials. Table 13.2.3 summarizes the various agitated leach test results.

Table 13.2.3 Agitated Leach Test Results – Blend Low and High Grade Oxides

#	Grind % -75 ppm	Leach h	Slurry % Solids	Temp °C	Extraction %	pH	Acid kg/t	ORP mV	H_2O_2 kg/t	Oxidant
1	27	24	55	40	90.7	1.5	7.7	475	0.8	H_2O_2
2	36	24	55	40	92.5	1.5	9.5	475	1.2	H_2O_2
3	37	24	55	40	92.3	1.5	10.1	475	1.8	H_2O_2
4	41	24	55	40	93.0	1.5	10.3	475	2.1	H_2O_2
5	49	24	55	40	93.4	1.5	9.5	475	0.9	H_2O_2
6	36	24	55	40	98.2	0.5	26.4	475	1.2	H_2O_2
7	36	24	55	40	96.6	1.0	12.0	475	1.1	H_2O_2
8	36	24	55	40	92.5	1.5	9.5	475	1.2	H_2O_2
9	36	24	55	40	83.8	1.9	6.1	475	1.1	H_2O_2
10	36	24	55	40	83.0	1.5	10.5	270/337	1.2	None
11	36	24	55	40	92.5	1.5	9.5	475	1.2	H_2O_2
12	36	24	55	40	92.9	1.5	9.0	550	1.9	H_2O_2
13	36	24	55	40	95.6	1.5	8.3	479/530	1.2	H_2O_2/Fe^{3+}
14	36	24	55	40	93.6	1.5	14.2	478/485	3.1	Pyrolusite
15	36	24	55	40	94.0	1.5	9.8	498/643	1.2	NaClO3
16	36	24	55	40	93.2	1.5	12.5	624/669	2.0	NaClO3
17	36	24	55	40	95.4	1.5	11.0	433/602	1.1	NaClO3
18	36	24	55	40	85.9	1.5	6.3	430/478	0.7	H_2O_2
19	36	24	55	30	91.3	1.5	9.3	475	1.0	H_2O_2
20	36	24	55	40	92.5	1.5	9.5	475	1.2	H_2O_2
21	36	24	55	60	96.1	1.5	9.8	475	1.1	H_2O_2
22	36	24	55	40	92.5	1.5	9.5	475	1.2	H_2O_2
23	36	24	65	40	94.3	1.5	11.0	475	1.7	H_2O_2
Average		**24**	**55**	**40**	**92.4**	**1.5**	**10.5**	**475**	**1.3**	**H_2O_2**

13.2.4 Solid Liquid Separation

Limited settling and filtration tests were carried out by ANSTO (ANSTO Report C1206) on slurries from JG and RT (Garee) generated in the laboratory program at the base case grind of P_{80} = 250 µm and 30°C (pH 1.5, 500 mV). The batch tests were performed in a 1 L measuring cylinder. Magnafloc E10 at a concentration of 0.025% wt was the flocculant used.

These preliminary flocculant and thickener requirements indicate that solid / liquid separation will not be an issue.

13.2.5 U_3O_8 Recovery

The early IX testwork looked at two ion exchange resins, both of which performed well. The resin loaded to about 60 grams per litre of wet settled resin (g/L wsr) with uranium recovery of about 76%. About 20 bed volumes (BV) of liquor were treated before the uranium in the barren solutions started to climb (breakthrough). Elution of the resin was also tested and about 12 BV of 1 molar (M) sodium chloride (NaCl) and 0.1 M sulfuric acid eluant was used. The resulting eluate solution was relatively clean with only phosphorus and arsenic as significant impurities. SX tests were conducted and the extraction rates were very good after three stages of extraction.

A test of UOC production by direct precipitation was conducted on the leach solutions using hydrogen peroxide. The precipitate produced contained 53% U_3O_8 equivalent with very high amounts of iron (4.6% Fe) and aluminium (6.4% Al), suggesting that direct precipitation would not be technically viable.

13.2.6 Product Preparation

One test investigated precipitation with ammonia based compounds to produce an ammonium diuranate (ADU). Using ammonium hydroxide (NH_4OH) over 99.5% of the U_3O_8 equivalent was precipitated. The mean grade of the ADU was 78% U_3O_8 with the only significant impurity being arsenic at 1.2%. The product grade of 78% U_3O_8 equivalent is above the minimum specification of 65% U (76.6% U_3O_8). The arsenic level at 1.2% is well above the Cameco product specification of 0.05% with a reject level of 0.15% As. However, the Comurhex specification allows 1% As and rejects at 2.5% As. Further testwork is necessary to reduce this arsenic level.

13.3 ANSTO 2011 Metallurgical Testwork Program

For the most recent testwork (2011), ANSTO was requested to undertake a metallurgical test program on the extraction of uranium from four composite lens samples (JG, RT (Garee) Upper, RT (Garee) Lower, and Jack) of the Westmoreland deposit. The overall aim of this work was to obtain data on process options for the recovery of uranium. A conceptual design flowsheet, which comprises conventional acid leaching followed by IX or SX and uranium product recovery, was examined in this test program. Petrological SEM work and analysis was also undertaken by ANSTO as part of the metallurgical study.

The results were reported by ANSTO in the June 2011 document titled *The Extraction of Uranium from the Westmoreland Deposits*. The following program formed the basis of the testwork:

- Undertake quantitative x-ray diffraction analysis (XRD) on the four lens samples to identify the proportions of major / minor gangue minerals. Four selected leach residues were similarly assessed.
- Undertake dilute leach tests on samples from each lens to determine the limit for uranium extraction under typical and more severe leach conditions.
- Develop laboratory grind calibration curves for the RT (Garee) and JG composites.
- Undertake a series of tests to determine optimum leaching conditions for the RT (Garee) and JG composites.
- Carry out two to three slurry leach tests on a sample of Jack Lens composite.
- SEM examination of four selected leach residues to assist in identifying any factors limiting uranium extraction during leaching.
- Prepare a 'bulk' composite for leaching and for the generation of pregnant liquor for use in uranium recovery work.
- Undertake batch laboratory IX equilibrium, loading and elution tests.
- Undertake batch laboratory solvent extraction equilibrium and stripping tests.
- Produce uranium oxide concentrates from the IX and SX routes.

The individual samples were crushed to <25 mm and then combined to prepare composites for each of the four lenses:

- JG Lens.
- RT (Garee) Upper Lens.
- RT (Garee) Lower Lens.
- Jack Lens.

The four crushed composite lens samples were split to provide sub-samples. One sub-sample for each lens was used to determine size versus uranium distribution, and to conduct scrubbing tests. A second sub-sample was crushed to <2 mm to provide samples for assay and leach testwork. The remaining sub-samples were retained.

13.3.1 Samples Tested

Four sets of samples were compiled by LAM from their 2008 drilling program for testwork by ANSTO. Table 13.3.1 summarizes the make-up of these samples.

Table 13.3.1 Testwork Sample Details

Deposit	Drillhole	From m	To m	Total m	U_3O_8 ppm	Weight kg
Junnagunna	JDD08-023	45	65	20	2,250	70
	JDD08-023	80	90	10	2,910	34
	JDD08-026	20	70	50	850	179
				80	**1,443**	**283**
Redtree (Garee) Upper Lens	WDD08-009	30	50	20	540	69
	WDD08-012	35	55	20	540	68
	WDD08-037	12	36	24	610	86
	WDD08-040	16	36	20	5,270	74
				84	**1,739**	**297**
Redtree (Garee) Lower Lens	WDD08-011	62	82	20	2,580	73
	WDD08-012	60	80	20	510	68
	WDD08-040	88	103	15	3,210	74
				55	**3,018**	**215**
Jack Lens	WDD08-054	11.5	20	18.5	90	35
	WDD08-055	0	25	25	1,050	69
				43.5	**727**	**104**

The samples were chosen to be representative intervals of specific recognizable lenses, which account for the majority of the resource base. Only limited leach testwork was done on the Jack Lens composite as it was considered to be surface oxidized ore.

The size by size analysis of each composite sample of crushed rock over the range 1 to 19 mm indicated that uranium was uniformly distributed in each size fraction, in proportion to the sample mass distribution, with a slight enrichment in the <1 mm fraction. Therefore, upgrading of ore could not be achieved by a size based separation.

It was noted that the weighted average grades of the samples were well above the nominated 'resource average' of 1,000 ppm U_3O_8 and consequently the samples were not strictly representative of the ore to be treated over the LOM. None the less, the samples are considered to be sufficiently representative of the deposit for the purpose of a SS.

13.3.2 Mineralogy

LAM sent 20 samples from the 2008 drilling program to SGS Mineral Services (SGS) in Perth for mineralogical study. The results of this work indicated that uranium occurred predominantly as uraninite and coffinite with lesser torbenite and autunite. No ningyoite was identified.

Quantitative XRD indicated that quartz was the dominant gangue mineral in all ore samples. Its relative concentrations varied from 88 to 92% wt. The minor constituents (less than 5% each) were illite, hematite, jarosite, chamosite, and hydroxylapatite. Chamosite (Fe rich chlorite), an acid consuming mineral, was found in four ores, whereas hydroxylapatite was detected only in JG ore. The uranium-bearing minerals were not abundant enough to be detectable by XRD.

SEM analysis on leach residues showed other gangue minerals such as rutile / anatase (TiO_2), zircon ($ZrSiO_4$), monazite ($(Ce,La,Nd,Th)PO_4$), florencite ($(Ce,La)Al_3(PO_4)_2(OH)_6$), pyrite (FeS2), galena (PbS), iron copper sulphide, copper sulphide and barite (BaSO4) were also present in the samples. The uranium-bearing minerals are closely associated with the quartz phases and, to a lesser degree, to the iron rich phases. In the near surface 'weathered' profiles carnotite which is a potassium and vanadium oxide mineral is also present.

13.3.3 Scrubbing Tests

Following the crushing tests, ANSTO assessed that the material was quite competent due to the surprisingly few 'fines' produced during crushing. On this basis, it was deemed that scrubbing would have no significant effect in any attempt to upgrade leach feed, and no further testwork on scrubbing was performed.

13.3.4 Size-by-Size Deportment

Each composite was crushed to <25 mm and then screened with each fraction analyzed for uranium. The results are shown in Table 13.3.2. There is very little upgrading on a size-by-size basis with uranium distributed on a similar basis to the particle size fraction. As a result, upgrading by screening would not be beneficial.

Table 13.3.2 Size-by-Size Uranium Distribution

Size mm	Junnagunna			Redtree (Garee) Lower			Redtree (Garee) Upper			Jack Lens		
	Weight	Uranium		Weight	Uranium		Weight	Uranium		Weight	Uranium	
	%	ppm	%	%	ppm	%	%	ppm	%	%	ppm	%
19	25.5	1,223	25.1	24.3	1,139	21.8	21.9	1,967	28.0	23.7	616	18.0
16	13.2	1,246	13.3	14.1	1,013	11.3	12.5	864	7.0	14.9	674	12.4
12.5	12.2	1,277	12.6	13.8	1,390	15.2	13.2	1,467	12.6	12.3	902	13.7
9.4	12.3	1,159	11.5	12.7	1,505	15.1	12.0	1,103	8.6	12.3	872	13.2
7.4	9.5	1,330	10.2	10.2	1,347	10.8	9.6	1,353	8.4	10.0	737	9.1
2.0	7.6	1,170	7.2	7.8	1,294	7.9	8.5	1,195	6.6	7.9	868	8.4
1.0	4.0	1,099	3.5	4.0	1,121	3.5	5.6	1,299	4.8	5.2	739	4.7
<1.0	15.7	1,321	16.7	13.0	1,399	14.4	16.5	2,246	24.1	13.8	1,209	20.6
Calc		1,241			1,269			1,543			812	
Assay		1,138			1,170			1,579			737	

13.3.5 Grind Calibration

Comminution tests were not conducted during the recent programme. However, grind calibration tests were conducted to determine the time required to reduce the <2 mm material to four different size distributions equivalent to eighty percent passing (P_{80}=) 350, 250, 150, and 75 µm for the purpose of sample preparation.

13.3.6 Dilute Acid Leach Tests

ANSTO conducted dilute acid leach tests – two on each composite pulverized to determine the ultimate uranium extraction and provide an estimate of the propensity for gangue dissolution. The two tests per composite comprised:

1. A base case leach with hydrogen ion exponent (pH) at 1.5, temperature at 40°C, and an ORP of 500 mV by adding 1.5 g/L of ferric iron.

2. An extreme case leach with pH at 1.0, temperature at 60°C and an ORP of 500 mV by adding 1.5 g/L of ferric iron.

The results are summarized in Table 13.3.3. The uranium was readily leached with a similar high recovery of ~99% under either base case or extreme conditions. The relative dissolution of gangue can be assessed by comparing the concentrations of ions in the final dilute leach liquors. For the dilute base case conditions, the concentrations are relatively low, decreasing in the order Ca>Si>Al>K>Mg. Gangue dissolution was greatest for RT (Garee) Lower Lens, and lowest for Jack Lens, noting that Fe dissolution could not be estimated because iron was added to the leach solution.

Table 13.3.3 ANSTO Dilute Leach Test Results

Lens	Base Case			Extreme Case		
	Head ppm[1]	Residue ppm[1]	Extraction %	Head ppm[1]	Residue ppm[1]	Extraction %
Junnagunna	1,370	14	99.0	1,370	9	99.3
Redtree (Garee) Lower	1,380	19	98.6	1,380	12	99.1
Redtree (Garee) Upper	1,862	21	98.9	1,862	14	99.2
Jack	929	22	97.6	929	14	98.5

[1]All values as U_3O_8 determined by DNA.

13.3.7 Conventional Leach Tests

Conventional laboratory scale leach tests were then conducted on the composites. In addition, these tests were also optimized for acid addition, oxidant type and addition, temperature, and leach times. Sydney tap water was used in all tests as no site water was available. The following 'base case' conditions were used based on previous historical testwork and general conventional acid leach conditions:

- Solids Density 55% solids by weight (w/w).
- Temperature 40°C.
- Leach Time 24 h.
- Grind Size Distribution P_{80} = 250 µm.
- Acidity (24 h) pH = 1.5.
- ORP 500 mV.

The oxidant used by ANSTO in the laboratory was sodium permanganate. This oxidant is not used in operating plants but is used in the laboratory for convenience and ease of control. The results of these base case tests are summarized in Table 13.3.4.

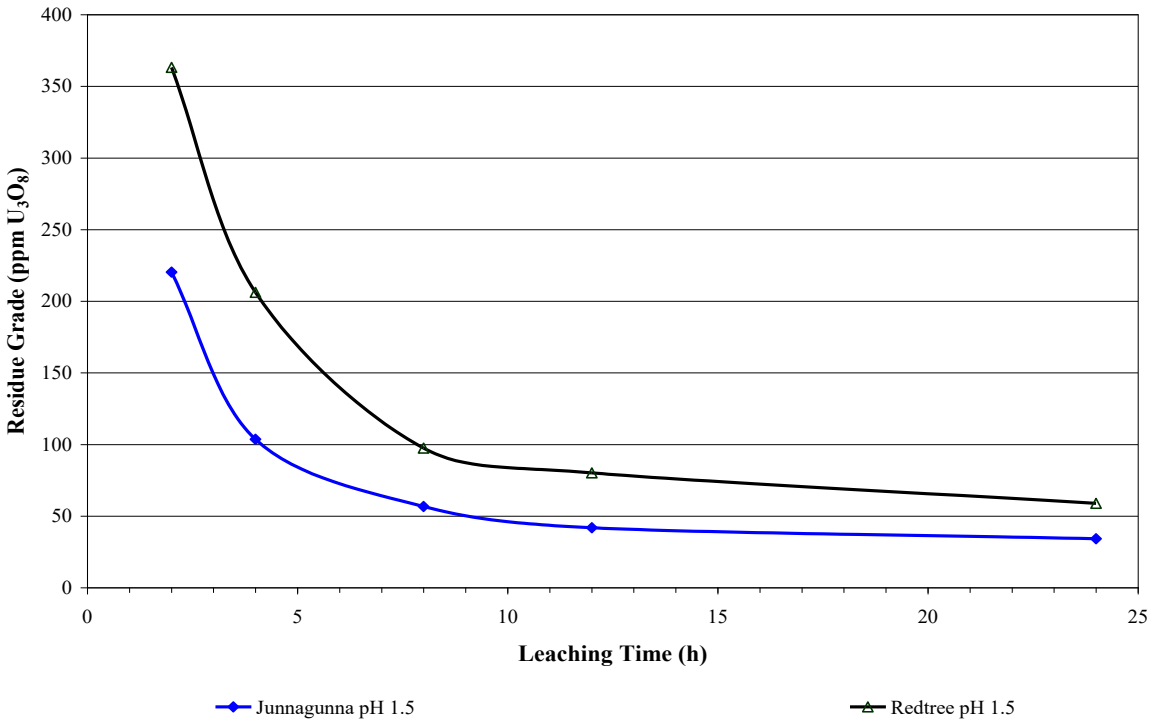
Table 13.3.4 ANSTO Conventional Leach Test Results

Composite	Head Grade ppm U_3O_8	Residue Grade ppm U_3O_8	Extraction %	Acid Add'n kg/t	Oxidant Add'n kg/t[1]
Junnagunna	1,370	34	97.5	20.6	1.6
Redtree (Garee)	1,704	59	96.5	17.1	1.6
Jack Lens	929	119	87.2	5.5	0.4

[1]Sodium permanganate is an acid consumer in the leach being responsible for 1 kg per kg of oxidant applied.

The preliminary results are good for the two main composites and less so for the Jack Lens results. The Jack Lens leaching tests are discussed in more detail in Section 13.3.13 below. The results confirmed that the ore is amenable to leaching with sulfuric acid. The leach kinetics indicate that leaching will be complete after 12 h as shown in Figure 13.3.1.

Figure 13.3.1 ANSTO Base Case Conventional Leach Test Kinetics



Optimization tests were then undertaken to ensure the correct leach conditions were established. A total of 43 tests were conducted looking at a range of parameters, as indicated below.

Table 13.3.5 Range of Parameters

Grind Size P_{80} = µm	Acidity pH	Oxidant ORP mV	Oxidant Type	Temperature °C
350	2.0	450	Sodium permanganate	50
150	1.7	550	pyrolusite	30
75	1.3	450 + 1.0 g/L Fe^{3+}		

Fe^{3+} indicates ferric iron added as ferric sulfate. Pyrolusite is a natural mineral containing about 75% manganese dioxide (MnO_2) and is commonly used by operating uranium extraction plants.

13.3.8 Effect of Grind Size

The effect of grind size on uranium extraction was examined at varying grind sizes of P_{80} 350, 250, 150, and 75 µm under base case conditions (pH 1.5, 40°C and ORP of 500 mV) for both the JG and RT (Garee) composites by maintaining the pH, temperature, ORP, and leach time constant. Table 13.3.6 summarizes the results for both composites. As shown, grind size has very little effect on the leach extraction.

Table 13.3.6 ANSTO Grind Size Optimization Leach Test Results

Grind Size P_{80} = µm	Head Grade ppm U_3O_8	Residue Grade ppm U_3O_8	Extraction %	Acid Add'n kg/t	Oxidant Add'n kg/t[1]
Junnagunna	1,370				
350		40	97.1	18.8	1.4
250		34	97.5	20.6	1.6
150		41	97.0	19.4	1.5
75		27	98.1	19.8	1.7
Redtree (Garee)	1,704				
350		56	96.7	16.4	1.4
250		59	96.5	17.1	1.6
150		56	96.7	16.4	1.5
75		54	96.8	17.3	1.7

[1]Potassium permanganate oxide.

13.3.9 Effect of pH

The effect of pH on leaching performance for JG and RT (Garee) was examined in four tests. For JG, except for leaching at pH 2, 24 h uranium extractions were very similar at pH 1.3 to 1.7. Optimum conditions were leaching at pH 1.5 to 1.7 for 12 h. However, for the RT (Garee) ore, the extraction increased with decreasing pH. The 24 h extraction increased from 92% to 98% when the leaching pH was decreased from pH 2.0 to pH 1.3. The pH also had an impact on the initial leaching rate. For this ore, optimum conditions were leaching for 12 h at pH 1.3 to 1.5. For both ores, acid requirements were relatively low, with an acid addition of 20 kilograms per tonne (kg/t) sufficient at the 'optimum' conditions. Thus, the JG composite is less sensitive to the acidity than the RT (Garee) composite, however both show that if the pH is held at 1.7 or lower, the extractions are quite good. Table 13.3.7 summarizes the results for both composites.

Table 13.3.7 ANSTO Acidity Optimization Leach Test Results

Acid pH	Head Grade ppm U_3O_8	Residue Grade ppm U_3O_8	Extraction %	Acid Add'n kg/t	Oxidant Add'n kg/t[1]
Junnagunna	1,370				
2.0		52	96.2	9.8	1.2
1.7		36	97.3	14.7	1.5
1.5		34	97.5	20.6	1.6
1.3		28	97.9	25.0	1.7
Redtree (Garee)	1,704				
2.0		130	92.4	11.8	1.0
2.0		116	93.2	9.5	1.0
1.7		73	95.7	11.4	1.3
1.5		59	96.5	17.1	1.6
1.5		55	96.8	16.7	1.4
1.3		31	98.2	20.4	1.6

[1]Potassium permanganate oxidant.

13.3.10 Effect of Pulp Temperature

The effect of leaching temperature was investigated for JG and RT (Garee). These tests were carried out under similar base case conditions with temperature as the only variable. As expected, the uranium leaching rate increased with increasing temperatures from 30°C to 50°C. For both ores, leaching at 30°C significantly decreased the extraction rate and, to a lesser extent, the final extraction of uranium. The initial rate of leaching was reduced at 40°C, but extractions were quite similar to those at 50°C after 12 h. Slurry pulp temperatures were varied while keeping other parameters constant. Table 13.3.8 summarizes these results.

Table 13.3.8 ANSTO Slurry Treatment Optimization Leach Test Results

Temperature °C	Head Grade ppm U_3O_8	Residue Grade ppm U_3O_8	Extraction %	Acid Add'n kg/t	Oxidant Add'n kg/t[1]
Junnagunna	1,370				
30		55	96.0	14.3	1.1
40		34	97.5	20.6	1.6
50		27	98.0	24.1	1.8
Redtree (Garee)	1,704				
30		98	94.2	12.4	1.1
40		59	96.5	17.1	1.6
50		41	97.6	19.0	1.8

[1]Potassium permanganate oxidant.

Although temperature has a significant effect on the initial extraction rate, there is also a significant relative increase in the acid addition. At the highest temperature, after 8 h leaching, the rate of gangue dissolution, as reflected in the acid addition, is much greater than the decrease in the uranium residue grade. Whereas at 30°C, the relative rates of uranium and gangue dissolution are still reasonably favorable after 24 h.

13.3.11 Effect of Oxidation Potential

The effect of ORP on the leaching of the JG and RT (Garee) composites is summarized in Table 13.3.9.

Table 13.3.9 ANSTO ORP Optimization Leach Test Results

ORP mV	Head Grade ppm U_3O_8	Residue Grade ppm U_3O_8	Extraction %	Acid Add'n kg/t	Oxidant Add'n kg/t
Junnagunna	1,370				
550		28	97.9	18.3	1.8
500		34	97.5	20.6	1.6
500		38	97.2	20.0	2.9*
450		61	95.5	18.0	0.6
Redtree (Garee)	1,704				
550		44	97.4	17.5	1.8
500		59	96.5	17.1	1.6
500		53	96.9	17.0	2.8*
450		106	93.8	15.0	0.8

***Indicates substitution of pyrolusite for the potassium permanganate.**

For both samples, there is a significant increase in oxidant demand for increasing the ORP from 450 to 500 mV, but only a further small addition is required to achieve 550 mV. The oxidant demand for both samples was very similar for both samples. Extraction is lower at the lowest ORP level of 450 mV. However, the differences between 500 mV and 550 mV are not so significant. The purpose of controlling ORP is to control the concentration of the ferric (Fe^{+3}) iron. As shown in Figures 13.3.2 and 13.3.3, at an ORP of 450 mV, the ferric ion concentration is <1 g/L for much of the leach, while at the higher ORP levels the Fe^{+3} reaches as much as 2.5 g/L enabling higher leach extraction kinetics.

Figure 13.3.2 ANSTO Ferric Ion Concentrations Junnagunna Composite

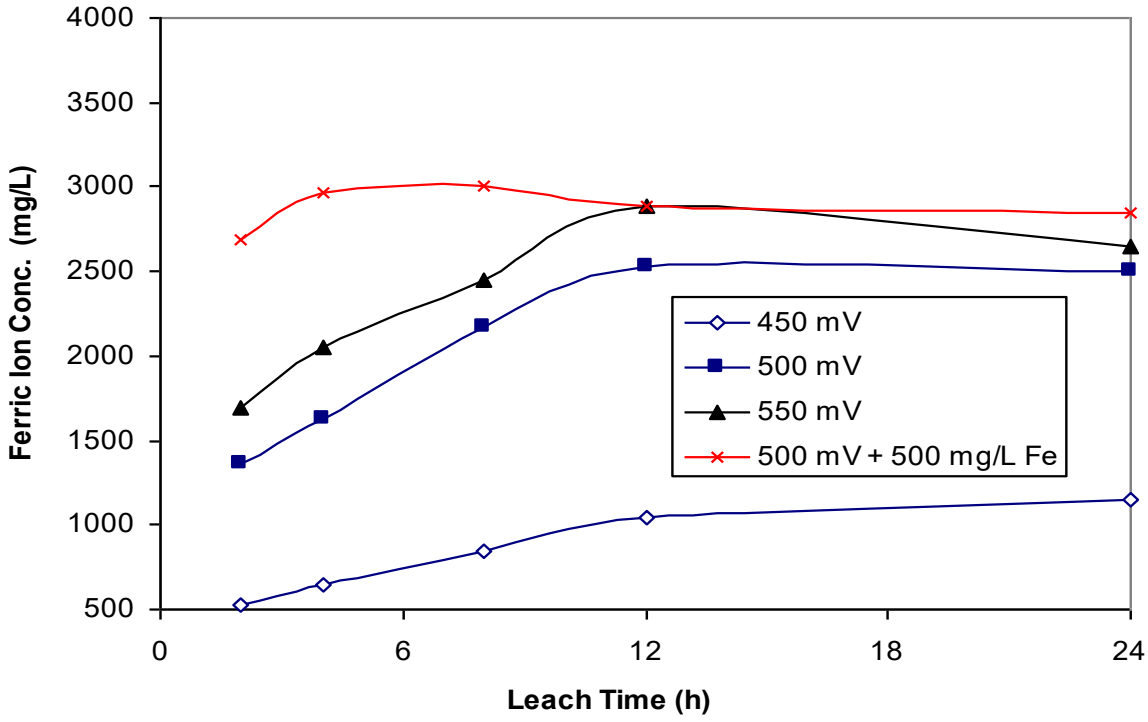
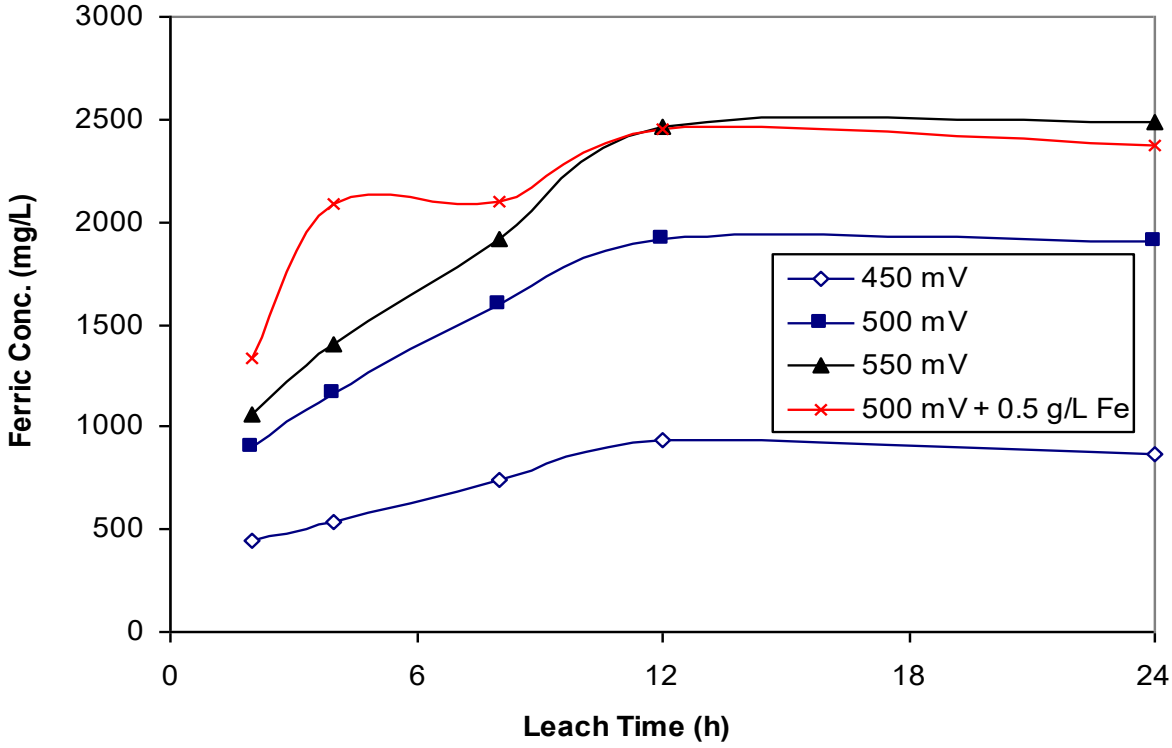


Figure 13.3.3 ANSTO Ferric Ion Concentrations Redtree (Garee) Composite



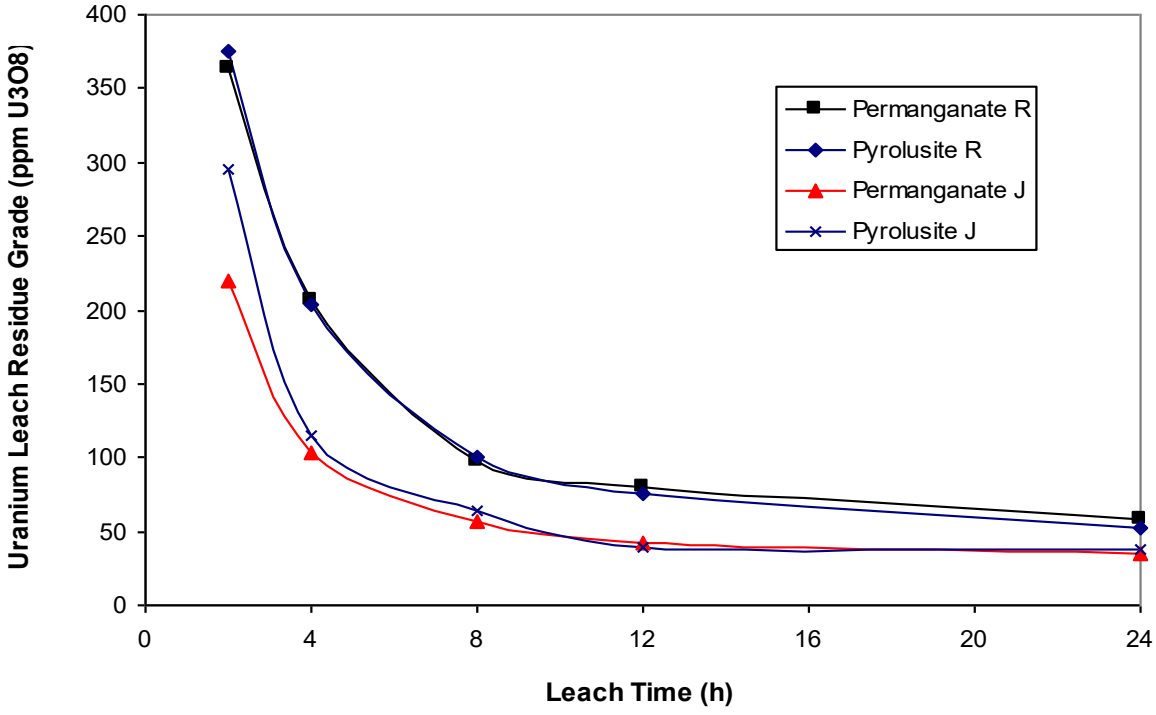
13.3.12 Effect of Oxidant Type

For ease of control in the laboratory, ANSTO used sodium and potassium permanganate in most leach tests. Base case leaches for JG and RT (Garee) were also carried out using pyrolusite to demonstrate that both oxidants gave equivalent results. The leach kinetics for each composite with the two oxidants are almost identical, as shown in Figure 13.3.4. ANSTO calculated that a pyrolusite containing 75% MnO_2 would theoretically report a consumption of about 3.3 kg/t, which is close to the consumption experienced in the testwork.

Table 13.3.10 ANSTO Oxidant Comparison Leach Test Results

Composite	Oxidant Type	ORP 24 h mV	Head Grade ppm	Residue Grade ppm	Extraction %	Acid Add'n kg/t	Oxidant Add'n kg/t
Junnagunna	Permang	485	1,370	34	97.5	20.6	1.6
	Pyroluste	481		38	97.2	20.0	2.9
Redtree (Garee)	Permang	477	1,704	59	96.5	17.1	1.6
	Pyrolusite	472		53	96.9	17.0	2.8

Figure 13.3.4 ANSTO Comparison of Oxidants Leach Kinetics



13.3.13 Leaching of Jack Lens Material

The Jack Lens composite material was tested separately from the JG and RT (Garee) composites because of the relatively small amount of this material expected to occur within the resources / reserves.

Initially, only one leach test was conducted on the Jack Lens material. These results were using 'base case' conditions and are reported in Table 13.3.4. The results were poor when compared to the JG and RT (Garee) results. Subsequently, some optimization tests were conducted as summarized in Table 13.3.11. These tests were conducted at a constant 40°C, and a grind size distribution of P_{80} = 250 µm, except for one test at P_{80} = 150 µm, and all at 24 h.

It is evident that to increase extraction from the Jack Lens material there will be a need to add ferric iron. Lowering the operating pH to 1.2 from 1.5 produced a better extraction than adding the ferric iron, but almost doubled the acid addition. Lowering pH as well as adding ferric iron has no benefit and increases acid consumption. Reducing the grind size distribution also provides little benefit. The optimum conditions for Jack Lens would be either lower pH or the addition of ferric iron. ANSTO suggest that if Jack Lens was blended with the JG and/or RT (Garee) materials, the natural ferric from these materials would be enough to allow the Jack Lens extractions to improve.

ANSTO recommended further work to identify methods of improving extraction on the Jack Lens ore.

Table 13.3.11 ANSTO Jack Lens Optimization Leach Test Results

ORP mV	pH	Ferric Add'n g/L	Residue Grade ppm U_3O_8	Extraction %	Acid Add'n kg/t	Oxidant Add'n kg/t
500	1.5	0	119	87.2	5.5	0.37
500	1.2	0	79	91.5	9.8	0.28
500	1.5	1.0	83	91.0	4.0	0.14
500	1.2	1.0	81	91.3	8.9	0.11
500	1.5	1.0	82	91.2*	4.3	0.28

***Indicates the grind size distribution of P_{80} = 150 µm.**

13.3.14 Leach Liquor Composition

ANSTO analyzed the leach liquor for major and minor element composition. The concentrations of the minors, and elements that could report to final product as penalty elements, e.g. Mo, V, Zr, are low. Arsenic was at the greatest concentration (especially from RT ore) and may warrant additional attention in regard to waste water treatment. Ferric concentrations are reasonably high in JG and RT (Garee), which is a positive for leaching, but will result in some degree of iron loading if IX is used for uranium recovery. However, ANSTO note that none of the gangue element concentrations in solutions would be expected to result in downstream processing problems.

13.3.15 Leach Residues

ANSTO analyzed the leach residues on a size-by-size basis and it was determined that the uranium was well leached from all fractions when compared with the size-by-size analyzes of the feed material. There is a slight decrease of extraction in the coarser particle sizes but the differences are quite minor and do not warrant finer grinding to possibly allow more extraction. The uranium-bearing minerals in the residues of JG and RT (Garee) were predominantly enclosed within quartz. They did not appear altered by leaching. It is likely that the acid solution could not penetrate the enclosing quartz, since no liberated or partially exposed uranium minerals were found.

13.3.16 Bulk Leach Tests

LAM decided to have a bulk sample leached using a blended sample from the four lenses being investigated. A 70.4 kg sample was composited using the following mix:

- JG 22.3 kg.
- RT (Garee) Upper 13.5 kg.
- RT (Garee) Lower 13.5 kg.
- Jack 22.1 kg.

The leach parameters were as a result of the optimization leach testwork:

- Grind Size P_{80} = 250 µm.
- Slurry Solids 55%.
- Duration 12 h.
- pH 1.5.
- ORP 550 mV.
- Oxidant pyrolusite.
- Temperature 40°C.

The assayed head grade of this composite was 1,360 ppm U_3O_8, compared with a calculated grade of 1,560 U_3O_8. Table 13.3.12 summarizes the leach results.

Table 13.3.12 ANSTO Bulk Sample Leach Test Results

Head Grade ppm U_3O_8	Residue Grade ppm U_3O_8	Extraction %	Acid Add'n kg/t	Oxidant Add'n kg/t
1,360	52	96.2	23.7[1]	6.44

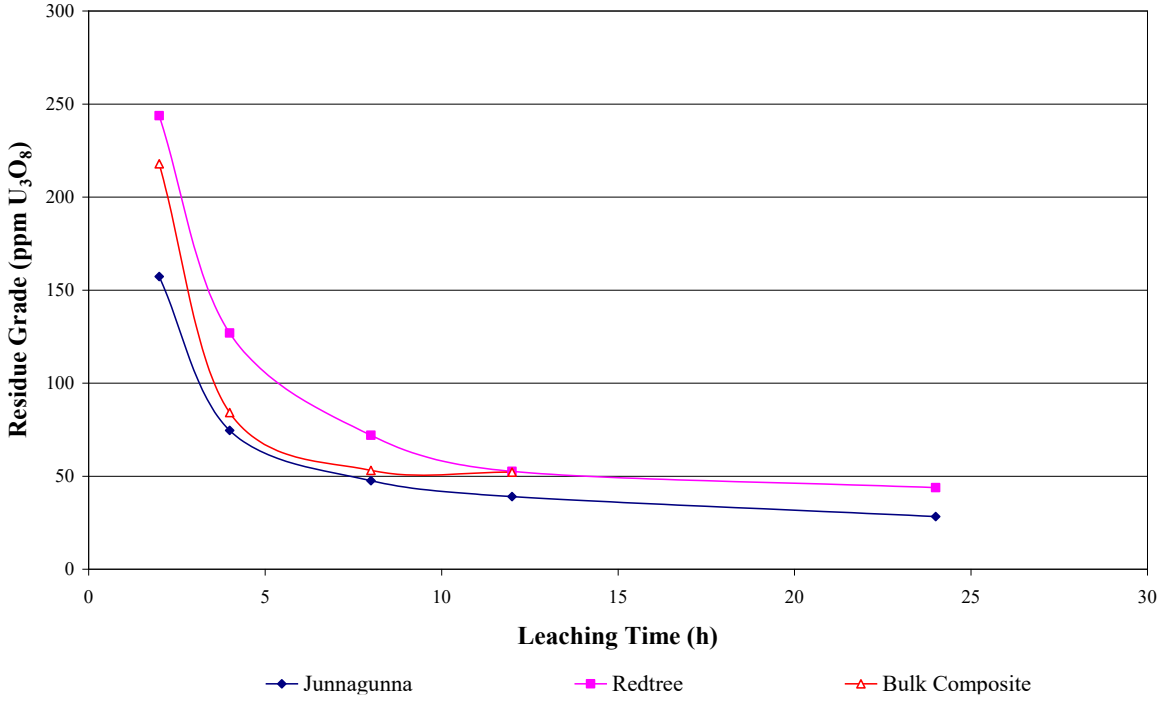
[1]23% of the acid was employed to leach the oxidant.

The uranium extraction was 96.2% for the bulk leach. This was below the extraction achieved on the JG and RT (Garee) ores, but higher than the extraction achieved on Jack ore. From the three tests conducted under similar conditions on the individual ores, the calculated extraction expected in the bulk leach is 95.6%. ANSTO considered that this increase may be due to the elevated ORP and iron levels enhancing the leaching of uranium from the Jack ore component of the composite. The acid consumption is higher than expected as was the oxidant consumption. Leach kinetics were as expected as shown in Figure 13.3.5.

ANSTO explain the higher acid and oxidant consumptions as due to the extra iron introduced by the mild steel grinding media used by Metcon when dry grinding the sample. The ORP was also held at a higher level for the bulk leach (550 mV) compared to the individual sample leaches (500 mV). The kinetics curve further confirms that 12 h leach time should be adequate. The bulk leach liquors report much higher manganese due to the use of the pyrolusite as an oxidant.

In summary, it can be said that the bulk leach tests was a strong confirmation of the individual sample testwork results and allows confident process design based on the results.

Figure 13.3.5 ANSTO Comparison of Bulk Leach Kinetics and Individual Sample Leach Kinetics



13.3.17 Settling and Filtration Tests

Vendor thickening and filtration tests were undertaken by FLSmidth (FLS) on slurries from the bulk leach tests. ANSTO also undertook some batch cylinder settling tests. Table 13.3.13 summarizes the FLS and ANSTO test results. FLS quoted an overflow solids concentration of less than 100 ppm.

Table 13.3.13 ANSTO / FLS Settling Test Results

Parameter	Units	FLS g/L	ANSTO	
			Junnagunna	Redtree (Garee)
Feed Rate	tph	30	30	30
Feed Solids	% (w/w)	45	-	-
Feedwell Solids	% (w/w)	7.5	7.5	6.6
Flocculant Addition	g/t	50 – 100	62.5	71.6
Flocculant Type (Magnafloc)	-	800 HP	E10	E10
Rise Rate	m/h	4.1	-	-
Free Settling Rate	m/h	30	5.9	5.9
Expected Underflow Solids	% (w/w)	60 – 61	39.9	37.6
Underflow Stress Yield	Pa	14 – 19	-	-
Flux Rate	$t/m^2/h$	0.38	0.103	0.142
Thickener Diameter (at 30 tph)	M	10	19.3	16.4
Thickener Diameter (at 125 tph)	M	20	-	-

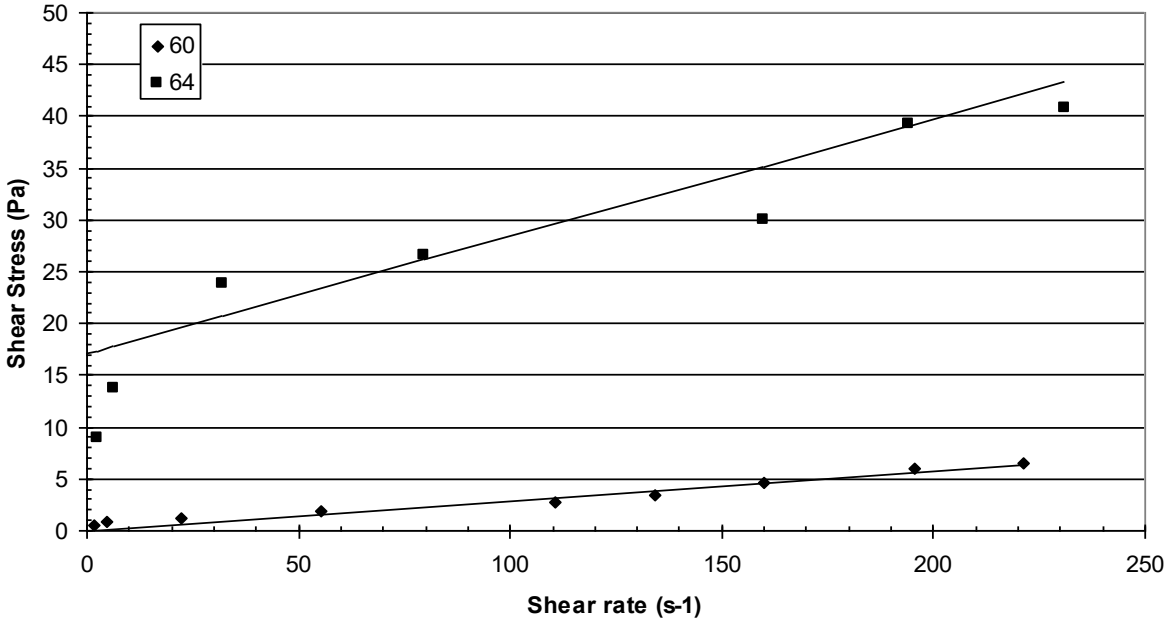
Filtration Tests

FLS conducted some preliminary filtration tests using the thickened samples from the settling tests. At a feed percent solids of 60%, the filtering rate was 0.472 tonnes per square metre per hour ($t/m^2/h$).

13.3.18 Pulp Rheology

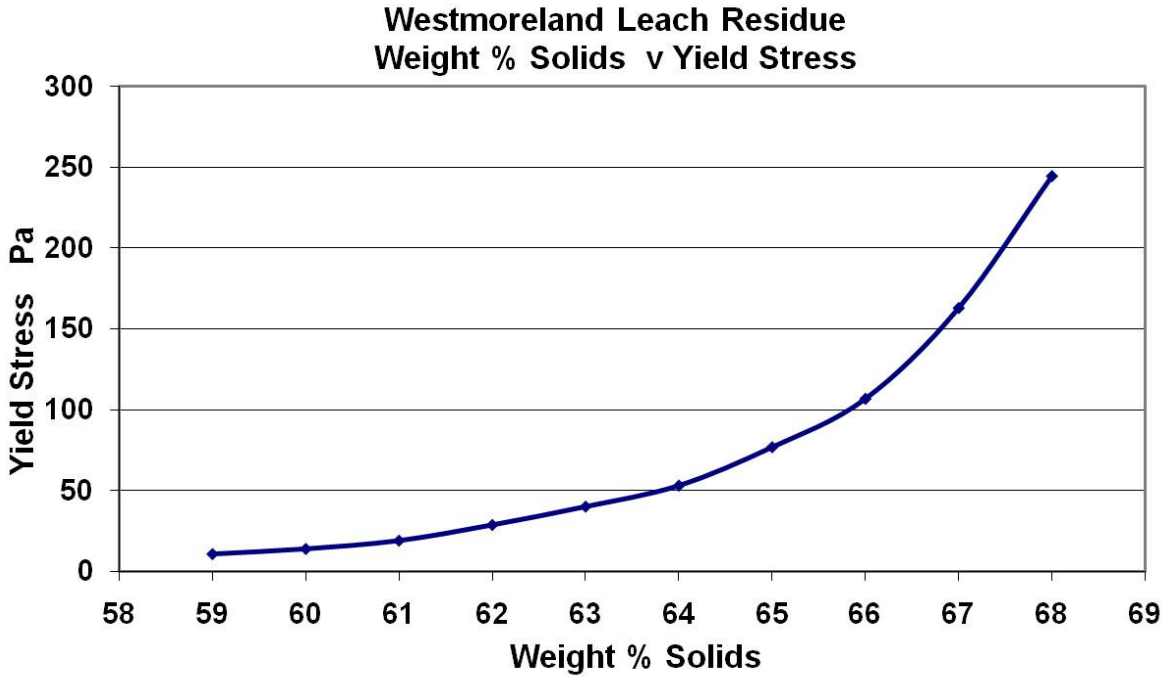
FLS conducted viscosity tests at various slurry pulp densities as well as determining the stress versus shear rate relationship. Figure 13.3.6 illustrates the shear rate versus shear stress relationship.

Figure 13.3.6 Thickened Pulp Shear Rate vs. Shear Stress – Bulk Sample



The relationship for shear stress and slurry pulp density is illustrated in Figure 13.3.7. ANSTO concluded, based on these results, that the product slurry settled reasonably well and the filtration testwork conducted by FLS indicated that the leach product slurry was amenable to filtration. The slurry filtration rate was reasonable and the filter cake could be washed to recover more than 99% of the soluble uranium without excessive wash water.

Figure 13.3.7 Thickened Pulp Yield Stress vs. Slurry Solids Density – Bulk Sample



13.3.19 Uranium Recovery Tests

Liquor generated from the bulk leach testwork was used to investigate alternative methods to recover the U_3O_8. The use of IX and SX techniques were tested, as well as precipitation of the U_3O_8 as an ammonium diurinate or as a uranyl peroxide.

The liquors for testing were as indicated in Table 13.3.14.

Table 13.3.14 IX Feed Liquor Compositions (mg/L/ppm)

Sample	pH	U_3O_8	Fe	Al	As	Si	V	Mn	S	Ca	Mg	Mo	K
Leach Liquor	1.8	1,605	5,350	609	100	589	21	3,750	9,910	312	170	13	236
RIP Feed	1.5	1,540	4,510	933	96	618	27	3,690	10,080	329	269	14	2
IX Feed	1.5	939	3,240	680	56	432	19	2,650	7,490	244	194	10	2

The IX feed has been diluted to simulate the effect of the wash water that would normally be used while the resin-in-pulp (RIP) feed solution is the PLS with no dilution.

13.3.20 Ion Exchange Testwork

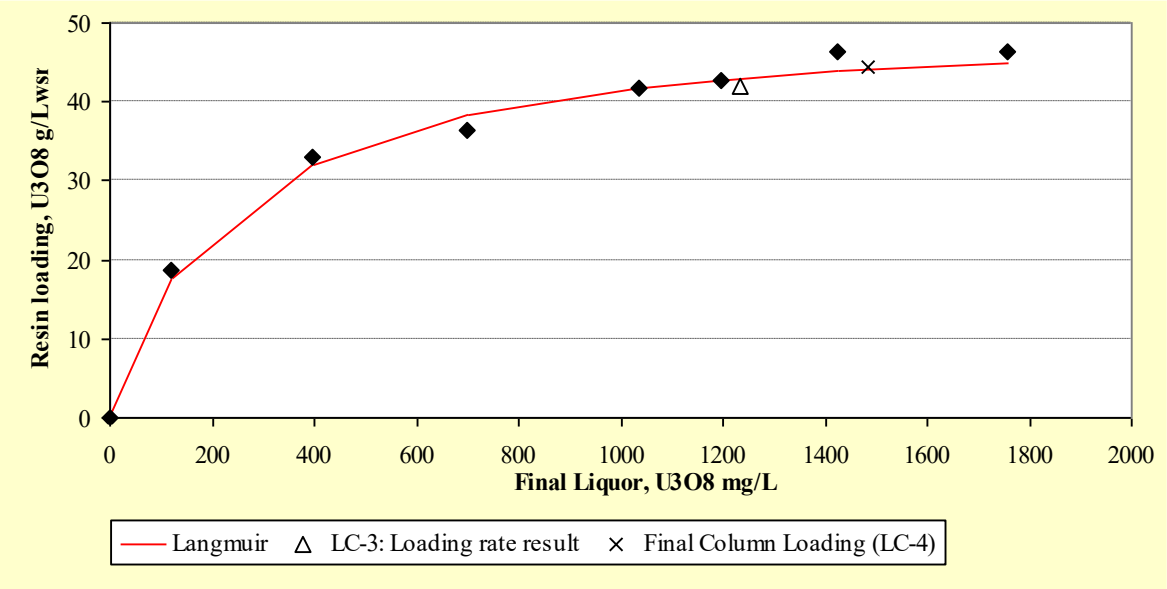
The IX testwork looked at two options: RIP, which adsorbs the uranium ions to the surface of specific resins while still within the leach pulps; and straight IX using clarified liquors, which are then put in contact with a resin and the uranium ions adsorb.

The resin chosen by ANSTO for the RIP simulated testwork was Ambersep 920, which had a mean particle size diameter of 0.75 mm to 0.95 mm. The resin chosen for the IX simulated testwork was Amberjet 4400, which had a mean particle size diameter of 0.58 mm. Prior to use, both resins were conditioned by contacting them with sulfuric acid and water to convert exchange sites to the sulfate form from the chloride form. The Ambersep 920 was screened at 600 µm. The Amberjet 4400 was screened at 600 µm and at 300 µm. The test results reported for the Ambersep 920 were derived by treating the PLS, rather than placing the resin in the leached pulp. In addition to this limitation in the test representation, Lycopodium considers that further testwork around the highly siliceous nature of the ore to determine if high rates of resin abrasion would be experienced in simulated RIP conditions. RIP has not been selected as the preferred process for the SS.

Resin Loading

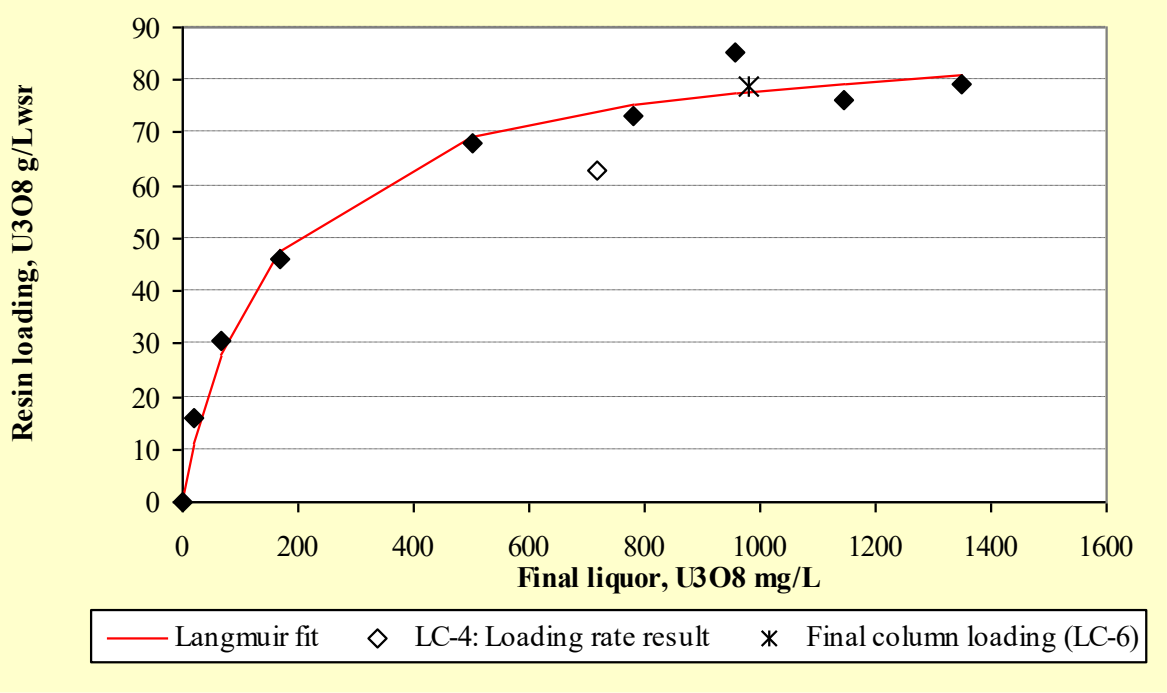
Figure 13.3.8 illustrates the loading curve for the RIP resin, Ambersep 920. The maximum loading for the RIP resin was 51 g/L wsr. The anticipated loading from a solution concentration of 1,450 g/L U_3O_8 would be about 42 g/L wsr Ambersep 920.

Figure 13.3.8 Uranium Resin Loading – RIP Resin Ambersep 920



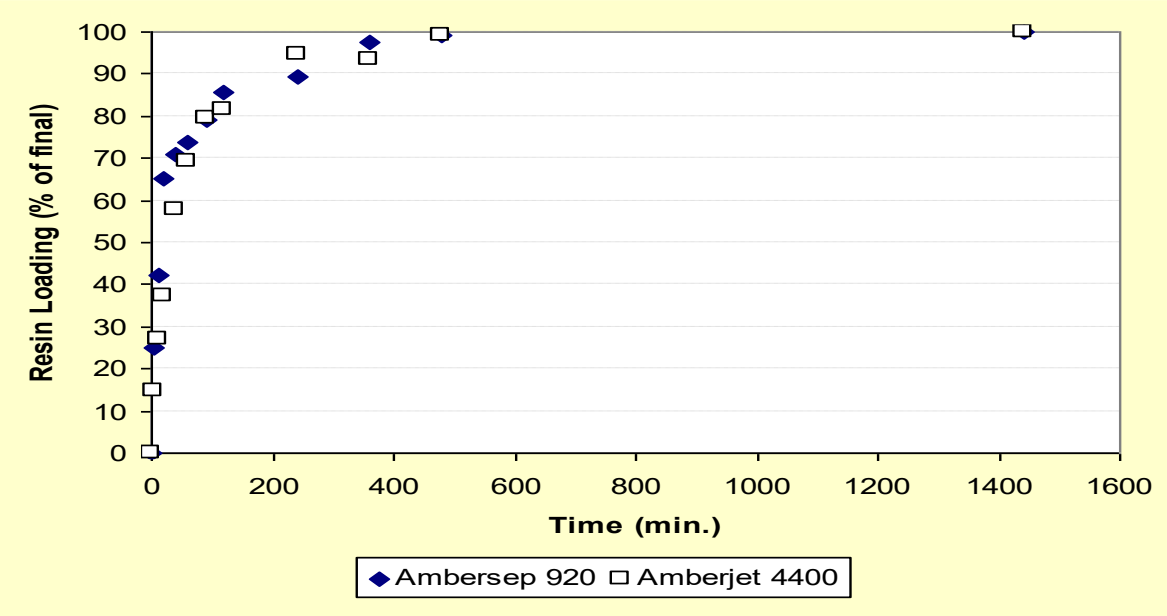
The equivalent curve for the Amberjet 4400 is shown in Figure 13.3.9.

Figure 13.3.9 Uranium Resin Loading – IX Resin Amberjet 4400



The loading for the Amberjet 4400 is higher than that of the Ambersep 920. ANSTO also measured the loading rates for both resins as indicated in Figure 13.3.10.

Figure 13.3.10 Uranium Resin Loading – RIP Resin Ambersep 920 and IX Resin Amberjet 4400



As can be seen the loading rate for the Ambersep 920 is slightly faster than that of the Amberjet 4400.

The kinetic parameters as calculated by ANSTO for the two resins are summarized in Table 13.3.15.

Table 13.3.15 Loading Kinetic Parameters Ambersep 920 and Amberjet 4400

Resin	T_{50} min	T_{75} min	Final Resin Loading g/L wsr U_3O_8	k
Ambersep 920	14.5	64.5	42	21.0
Amberjet 4400	32.0	77.0	63	26.6

This data indicates that the Ambersep 920 loads slightly quicker albeit to a lower maximum resin loading.

Column breakthrough curves were produced for each resin using leach solution delivered downflow to the column at a flow rate of 4 bed volumes per hour (BV/h) (1.05 m/h). The loading was conducted at 35°C for delivery of 100 BV of feed. A fraction of column effluent was taken every 2 BV and analyzed for uranium and impurities. Figure 13.3.11 summarize the curves and shows that the Ambersep 920 requires about 50 BV to reach saturation loading, and that loading occurs at 45.3 g/L which confirms the previous determination of 42 g/L wsr. The Amberjet 4400 reaches saturation at approximately 100 BV and loads to 79 g/L wsr, which is higher than previously determined. All the tests indicate the Amberjet 4400 has a higher capacity than Ambersep 920, with a ratio of about 1.4:1.0 equivalent per litre wsr.

Figure 13.3.11 Uranium Resin Breakthrough Curves

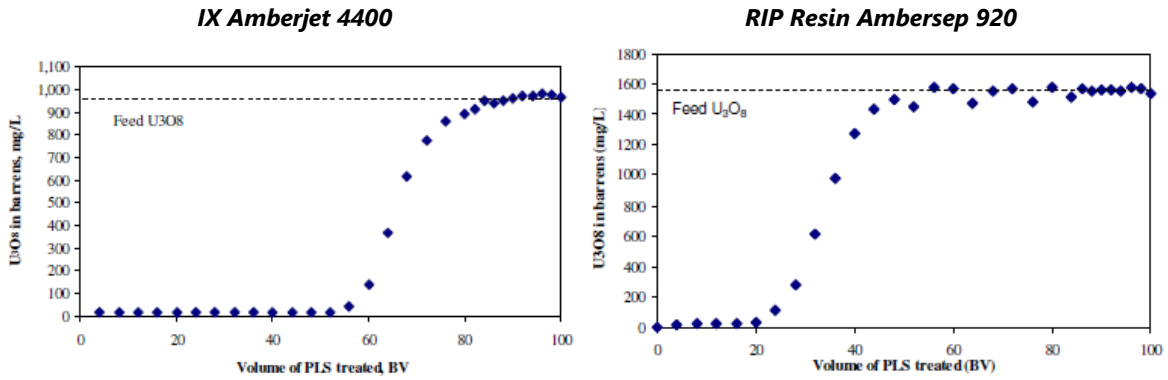


Table 13.3.16 indicates resin loadings for some of the other elements in the PLS. Scrubbing stages may be required in the process flowsheet if these elements deport to the eluate in excessive amounts. It is noted that the high apparent silica loading on the Ambersep 920 might suggest that a RIP process would not be appropriate for the adsorption and recovery of uranium.

Table 13.3.16 Metal Ions Loading for Ambersep 920 and Amberjet 4400

Resin	U_3O_8 g/L wsr	Fe g/L wsr	SO_4 g/L wsr	Si g/L wsr	P g/L wsr
Ambersep 920	45.3	1.6	59.4	17.6	0.4
Amberjet 4400	78.7	0.5	94.3	1.7	0.6

Resin Elution

The elution behavior of each resin was characterized by performing equilibrium measurements, elution rate measurements, and column elution behavior with 1 M sulfuric acid and at 35°C. The isotherms for the resins are given in Figures 13.3.12 and 13.3.13.

Figure 13.3.12 Uranium Elution Isotherm – RIP Resin Ambersep 920

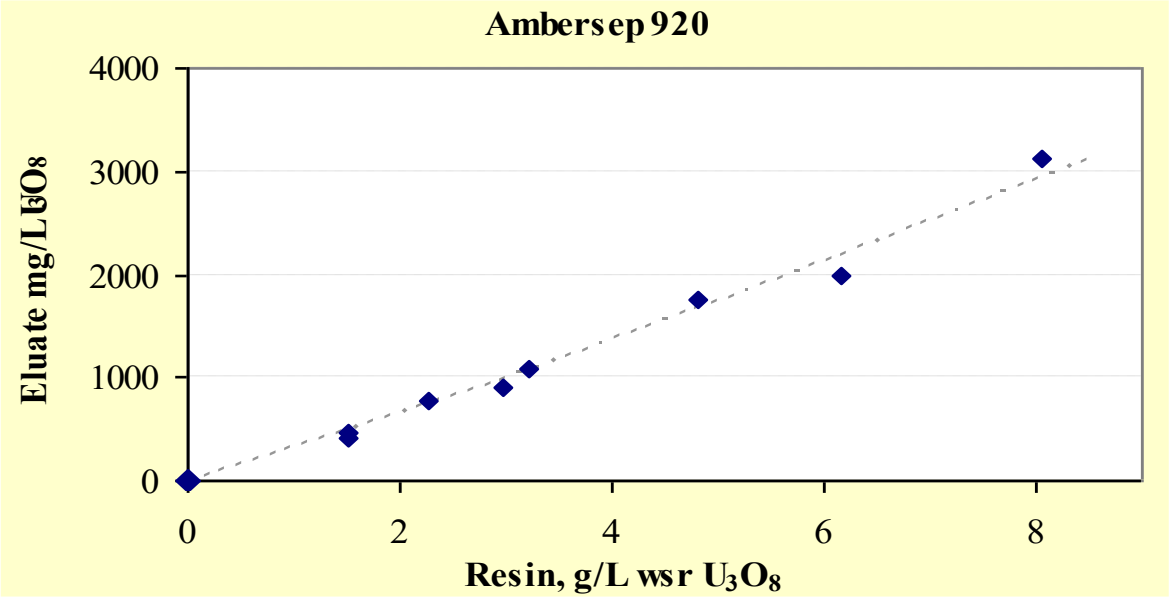
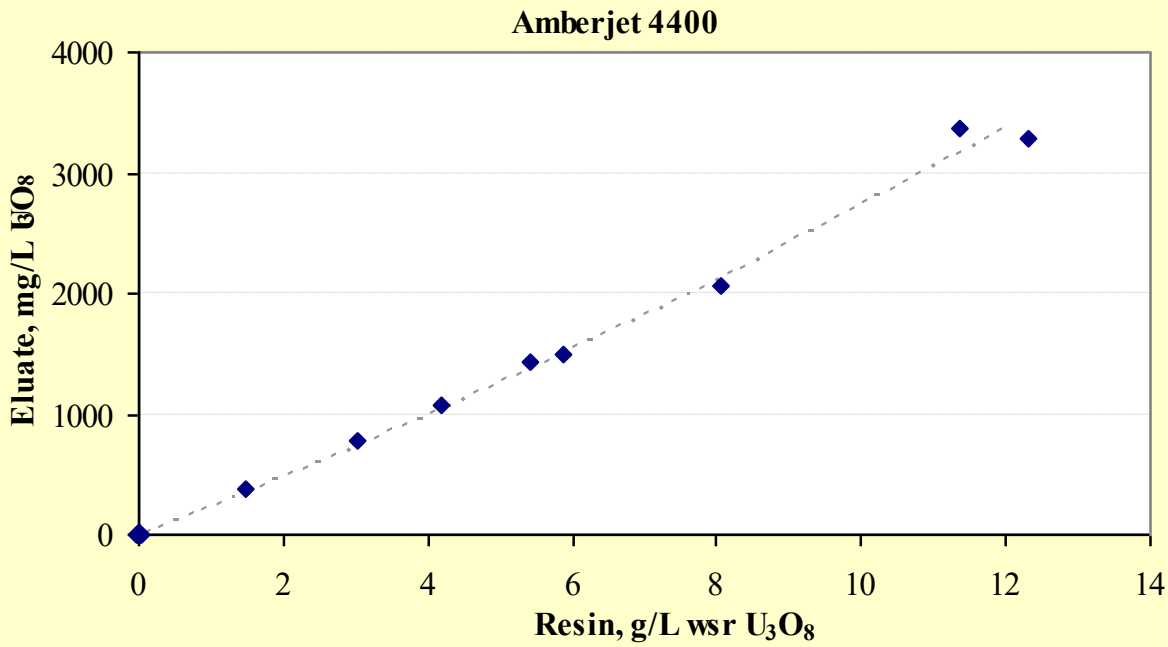


Figure 13.3.13 Uranium Elution Isotherm – IX Resin Amberjet 4400



Uranium loaded Ambersep 920 and Amberjet 4400 resins were contacted with 1 M sulfuric acid in bottle roll tests at 35°C. The rate of uranium elution was determined by monitoring the variation in the uranium concentration of the eluant over 24 h. The elution kinetic parameters are shown in Table 13.3.17.

Table 13.3.17 Elution Kinetic Parameters Ambersep 920 and Amberjet 4400

Resin	T_{50} min	T_{75} min	Final Resin Loading (g/L wsr U_3O_8) Initial	k
Ambersep 920	9	22	45.3	0.8
Amberjet 4400	26	53	78.7	1.4

The elution rates were also determined as shown in Figures 13.3.14 and 13.3.15 indicate that both resins were eluted efficiently by sulfuric acid. A final resin concentration of 1 g/L wsr U_3O_8 indicated practically complete elution.

ANSTO also conducted column elution testwork. These elution curves (at 35°C) are shown in Figures 13.3.16 and 13.3.17. The eluant was delivered to the column at a flow rate of 1 BV/h (0.09 m/h). ANSTO concluded that both elution curves indicate that uranium elution is achieved well within 20 BV of eluant delivered to the column with a stripped resin composition of 1 g/L wsr U_3O_8 reached after 7 BV of eluate for the Ambersep 920 and 14 BV for Amberjet 4400. The elution process is kinetically impaired at lower temperatures and a minimum temperature of 40°C is recommended. The elution behavior of the two impurities—iron and phosphorous—are also included. Iron appears to elute prior to the uranium, particularly for Amberjet 4400. Ferric iron in its ferric sulfate complex form is less strongly bound compared to the uranium oxide sulfate complex and can be eluted with a weak acid or a reductant to effect a 'scrubbing step' thereby improving the U:Fe ratio. Phosphorous elution is coincident with uranium for both resins and this may impact to some extent on precipitate purity during product recovery when direct precipitation of the uranium from the eluate is undertaken.

The variation of the uranium concentrations in the bulk eluate for both resins are compared in Figure 13.3.18, and show that bulk eluates can contain up to 8.8 g/L and 11.3 g/L U_3O_8 for the Ambersep and Amberjet after collection of 4 and 5 BV, respectively.

Figure 13.3.14 Uranium Elution Rate – RIP Resin Ambersep 920

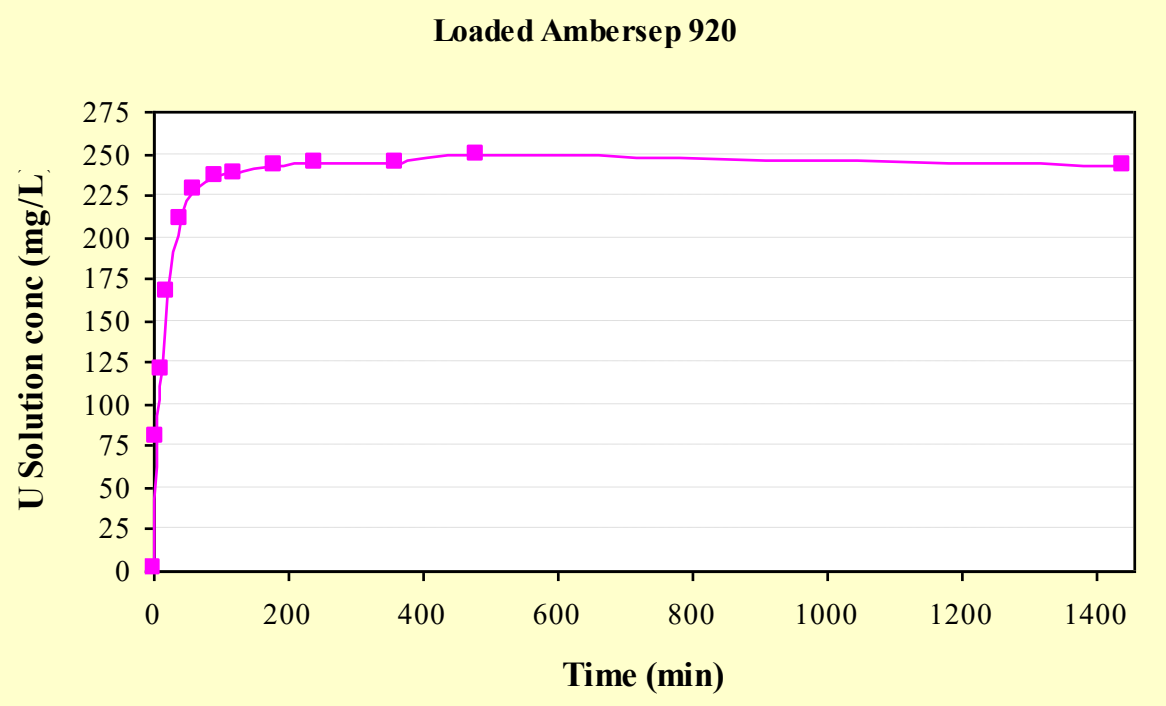


Figure 13.3.15 Uranium Elution Rate – IX Resin Amberjet 4400

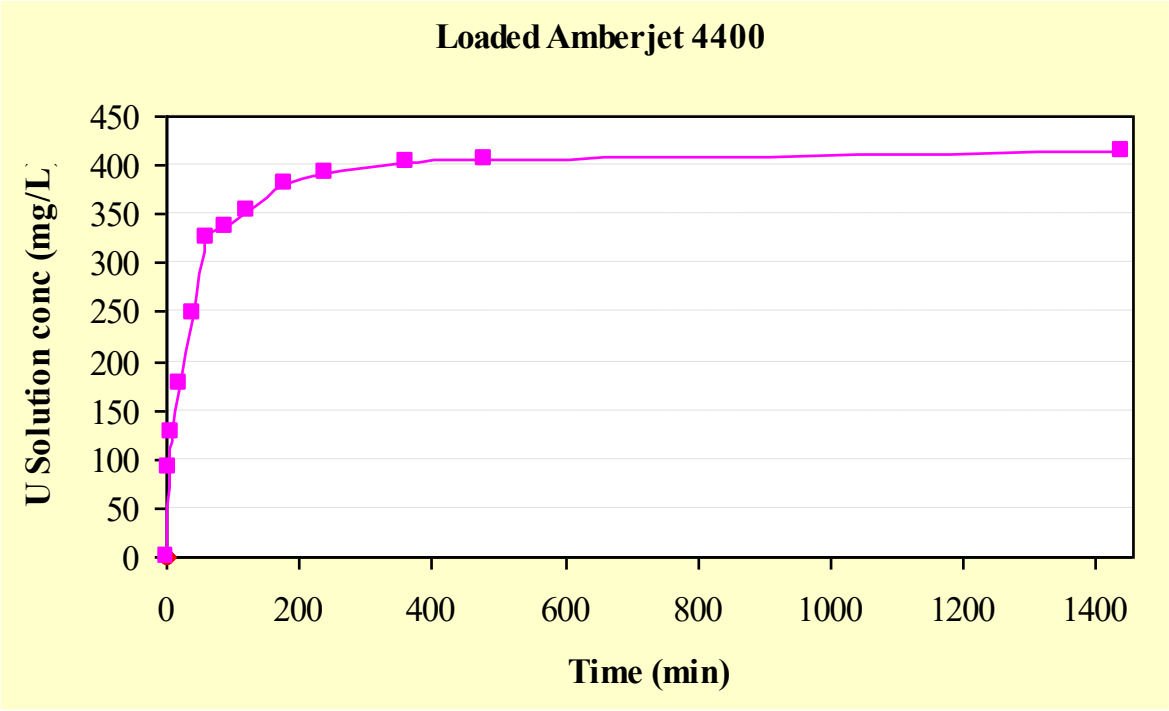


Figure 13.3.16 Uranium and Impurity Elution – RIP Resin Ambersep 920

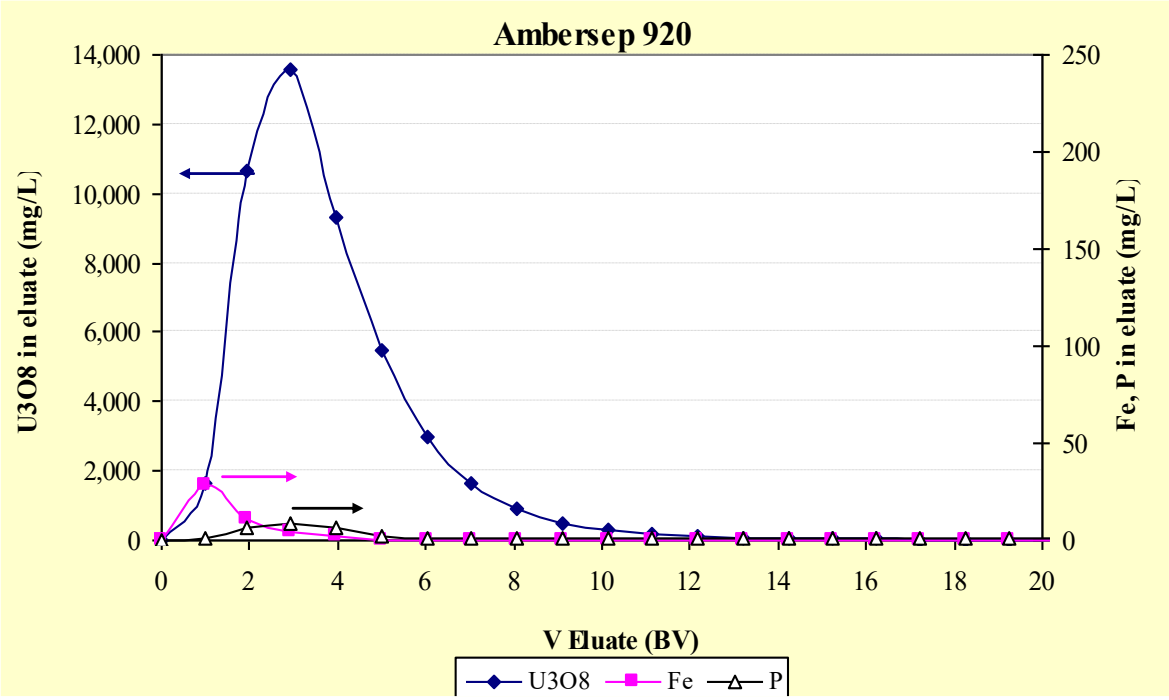


Figure 13.3.17 Uranium and Impurity Elution – IX Resin Amberjet 4400

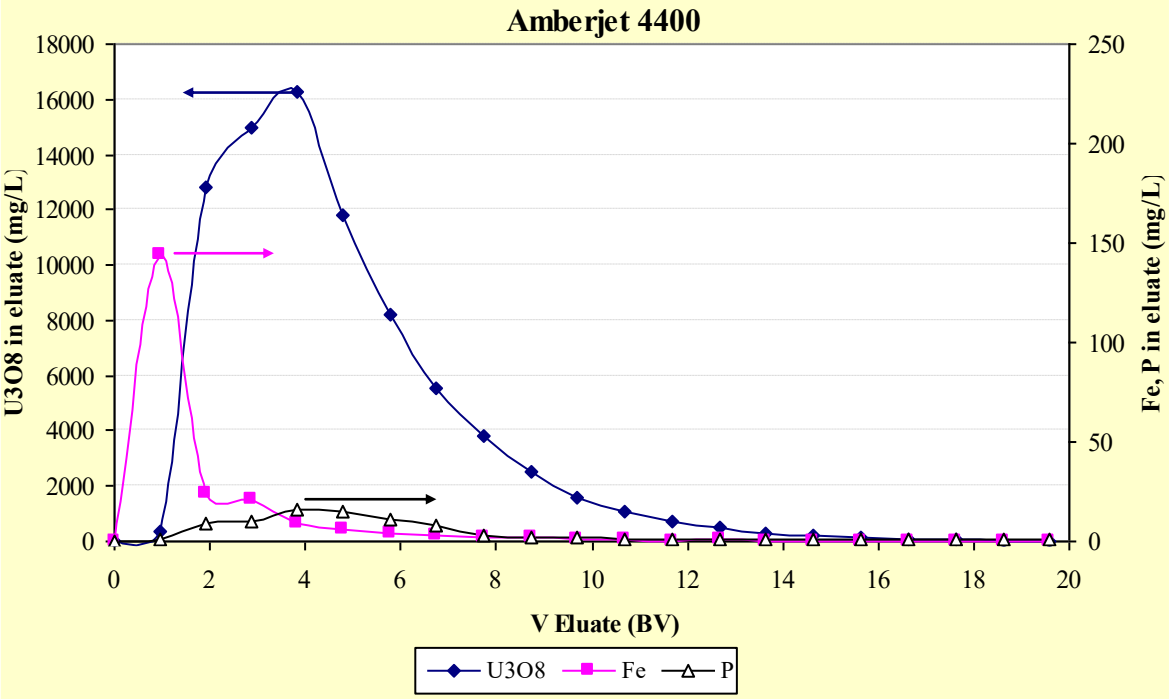
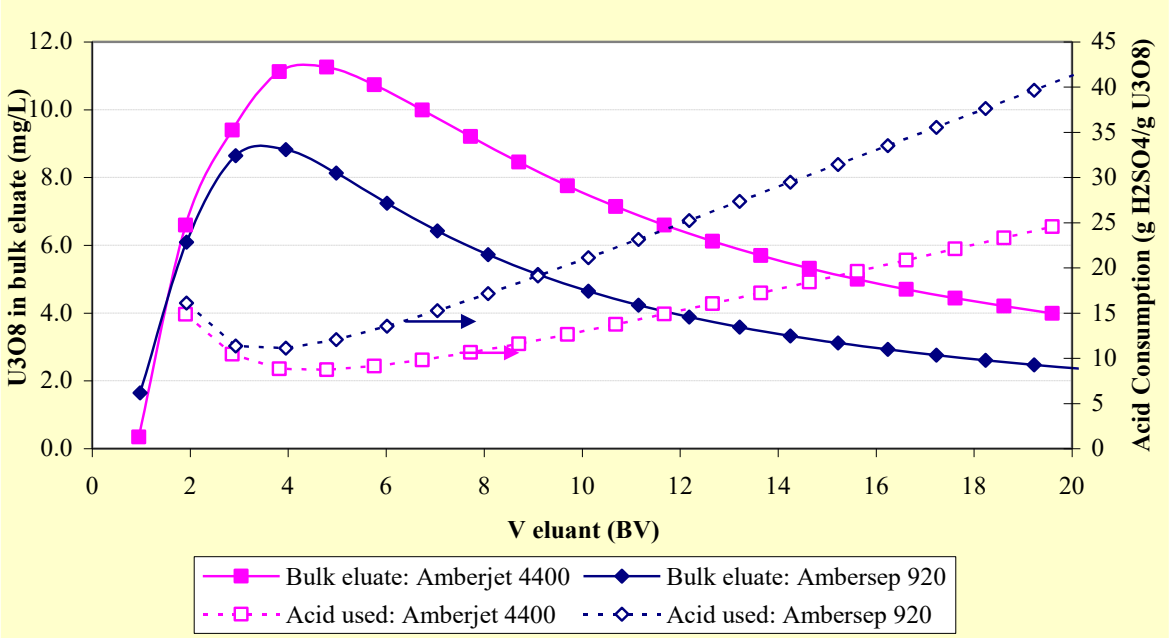


Figure 13.3.18 Bulk Elute Concentrations – RIP Resin Ambersep 920 and IX Resin Amberjet 4400



13.3.21 Uranyl Peroxide Precipitation

ANSTO conducted some non-optimized precipitation tests using the eluate generated in the elution testwork. These results, with their previous experience, was used to indicate impurity deportment and uranium levels.

The method was based on a two stage process where gypsum (pH 1.5) and iron hydroxide (pH 3.5) were precipitated first and removed, prior to uranyl peroxide precipitation ($UO_4.2H_2O$) using hydrogen peroxide at pH 3.5 and 35°C. This final production reaction follows the equation below.

$$UO_2(SO_4)_3 + H_2O_2 + 2NaOH + xH_2O \longrightarrow UO_4.xH_2O + 2H_2SO_4 + Na_2SO_4$$

Table 13.3.18 presents the analyzes of the products from the precipitation tests for both resins, and compares them to the specifications provided from the three converter companies—Comurhex, Converdyne, and Cameco. Not all elements are shown, only the more evident ones. Iron nearly met specification, but phosphorous exceeded reject specifications for Cameco and attention needs to be paid to managing it. The gypsum / iron cake composition indicated that some uranium losses occurred during the preliminary precipitation. XRF assays show that the gypsum from the Amberjet eluate contained 0.67% U_3O_8 and from the Ambersep 920 eluate, the gypsum contained 0.44% U_3O_8. The uranium in gypsum represented 12% and 6% of the uranium in feed for Amberjet and Ambersep, respectively. In practice this slurry could be recycled to the leach circuit and the wash liquor combined with the filtrate to recover this uranium. ANSTO believed that the high sulfate ion content resulted from poor cake washing during filtration. An alternative would be to employ magnesia or sodium hydroxide, with an iron precipitation step to control phosphate, but uranium recycle in the precipitate would be inevitable. The use of limestone should be avoided as a consequence of this reagent introducing calcium, and also manganese depending upon the source of limestone.

Table 13.3.18 Uranyl Peroxide Compositions (as % of U)

Element	Ambersep 920	Amberjet 4400	Cameco		Comurhex		Converdyne	
			Accept	Reject	Accept	Reject	Accept	Reject
U*	71.0	71.2	-	-	-	-	-	-
V_2O_5	n/m	n/m	-	-	0.30	0.30	-	-
V	<0.03	<0.03	0.05	0.10	-	-	0.01	0.05
As	<0.04	<0.03	0.05	0.15	1.00	2.50	0.01	0.04
B	<0.04	<0.02	0.01	0.15	0.20	0.20	0.01	0.10
C	n/m	n/m	-	-	0.20	1.00	0.01	0.20
Ca	0.32	0.47	-	-	-	-	-	-
Cl	<0.35	0.44	-	-	0.15	0.25	0.05	0.10
CO_3	n/m	n/m	-	-	2.00	3.00	0.20	0.50
F	<0.04	<0.03	-	-	0.15	0.30	0.01	0.10
Fe	0.23	0.19	1.00	2.00	-	-	0.15	0.50
K	0.41	0.26	1.00	2.00	-	-	0.20	1.00
Mg	<0.04	<0.03	3.00	4.00	-	-	0.02	0.50
Mo	<0.03	<0.02	0.10	0.30	0.10	0.30	0.10	0.30
Na	0.20	0.26	1.00	2.00	1.00	7.50	0.50	3.00
PO_4	0.86	0.83	-	0.50	1.00	1.00	0.10	1.00
S	4.08	2.56	1.00	3.50	0	0	0	0
Se	<0.03	<0.02	-	-	-	-	0.01	0.04
SiO_2	0.43	0.28	1.07	2.00	0.50	2.50	0.50	2.00
SO_4	12.24	7.68	-	-	3.00	10.00	1.00	4.00
Th	<0.03	<0.02	0.50	2.00	-	-	0.01	0.05
Ti	<0.04	<0.03	0.05	0.10	-	-	0.01	0.05
Zr	<0.03	<0.02	0.10	0.50	0.20	2.00	0.01	0.50
Cd	<0.04	<0.02	-	-	-	-	0.01	0.04

***The U measured is the percent of dried product.**

Based on these results, ANSTO concluded that precipitation of uranyl peroxide from the eluates generated a product for which the composition compared favorably to a Cameco, Comurhex, and Converdyn (upper limit) purity specification. Iron phosphate precipitation during the iron removal stage or resin scrubbing prior to elution, may provide a solution to the high levels of phosphorus in the uranyl peroxide product.

13.3.22 Solvent Extraction Testwork

Bench scale solvent extraction testwork was also conducted by ANSTO and demonstrated that the ore was suited to this approach to uranium recovery. However, for environmental reasons, LAM's preference was for a flowsheet that excludes the use of ammonia and consequently the focus of the SS was on the use of ion exchange. Details of the solvent extraction testwork performed can be found in ANSTO Westmoreland Final Report 25 July 2011.

13.4 Conclusions and Recommendations

The principal conclusions reached are:

1. Due to the limited comminution testwork completed to date, the following ore comminution parameters were used for estimating the comminution circuit selection:
   - BBWi of 19.4 kWh/t
   - Bond Rod mill work index (BRWi) of 21.0 kilowatt hours per tonne (kWh/t).
2. The Westmoreland material generally acid leaches very well with modest acid consumption and high uranium extractions.
3. Co-leaching of gangue elements is not considered to present any problems for down-stream processing.
4. The Jack Lens material was the only exception and extraction was improved by adding ferric iron to assist oxidation of tetravalent uranium in the material. Further optimization of leach conditions is expected to improve performance on Jack Lens material, for example treat as a blend with the other ores that release iron in the leach.
5. The grind size distribution required is relatively coarse, which favors milling power consumption and filtration performance.
6. The leach kinetics are reasonably fast.
7. Recovery of the uranium from the leached slurry can be undertaken by several methods including continuous ion exchange.
8. Precipitation of the U as a concentrate to be sold to the market can be of a good quality and can be treated by any of the three main converters that will be treating the material.
9. Pulp settling rate is reasonable with a high solids underflow density and a relatively clear overflow PLS.
10. The use of IX technology has been tested and selected. The use of SX technology has also been tested and would be a technically viable treatment option.

A significant metallurgical test program, including closed circuit piloting will be required for the next phase, including collection and testing of representative samples and composites, variability tests on specific zones of the deposits, and comminution testwork over the range of lithologies expected to be encountered. ANSTO specifically recommended the following:

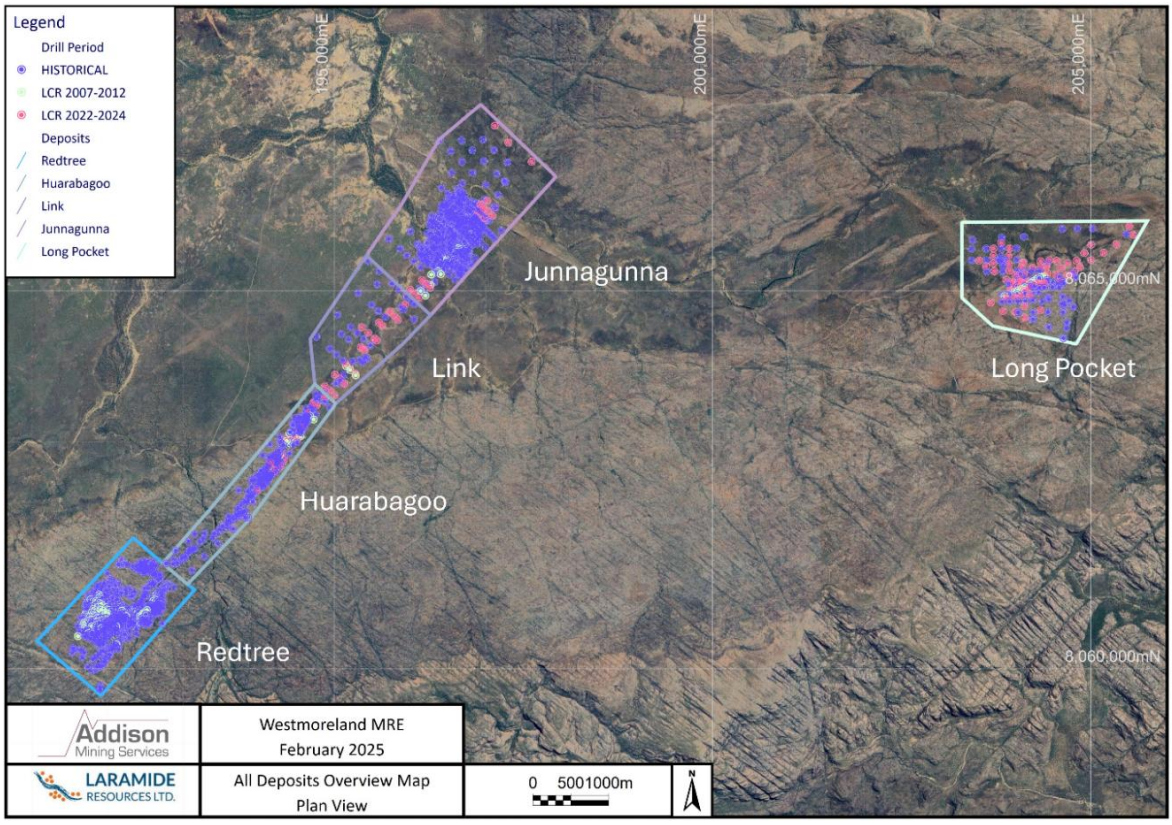
- Conduct leach tests using solution either from site or a synthetic solution to simulate expected leach make-up solution.
- Conduct optimization tests on the expected composite feed, and use these blends for the pilot plant test program.
- Conduct downstream neutralization testwork, on liquors generated from RT (Garee) ore and a composite of all three ores, to ensure that the arsenic can be effectively immobilized into an iron precipitate.
- Conduct a continuous pilot operation on the expected feed composite to confirm data generated in batch tests, and to generate slurry / solution for continuous downstream piloting.
- Conduct filtration, settling and rheology testwork on the product slurry from the continuous testwork.
- Conduct downstream continuous testwork, i.e. IX and/or SX.
- Consider tailings neutralization treatment and recycle of liquor.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

An update to the MRE for the Project, QLD, Australia, has been prepared by Addsion Mining Services of the United Kingdom (UK) on behalf of LAM. The updated MRE has an effective date of 31 January 2025, and was prepared using Micromine Origin and Beyond 2025 software. MREs were prepared for the RT, HB, JG and LP deposits (Figure 14.1.1). No MREs are reported for other areas. A review of the gold assay data was also completed for HB; due to sampling of gold being non-systematic by previous explorers, continuity was difficult to identify and it was considered that any estimate of a gold credit was not meaningful.

Figure 14.1.1 Westmoreland – Mineral Resource Estimate Areas



No Mineral Resources are reported for the Link area.

14.2 Input Data Summary

A summary of drillholes by deposit and drilling period used to inform the estimate is summarized in Table 14.2.1. It should be noted that the Project drillhole database contains 2327 collars for 153,146 m of drilling; only those drillholes in the areas of interest are summarized below. A more detailed breakdown of historical drilling is provided in Section 10 of this report. The estimate incorporated all relevant drilling completed by LAM up to and including the 2024 drilling. No new drilling has been completed by LAM since 2008 at RT. Additionally, Table 14.2.2 summarizes the drilling by type in each area.

A summary of the assay type falling within the mineralization wireframe models is shown in Table 14.2.3. The XRF method was used for the analysis of 87% of the samples, with 7% by scintillometer methods and 6% by ICP-MS.

Table 14.2.1 Summary of Drillhole Information

Deposit	Drill Period	No. of Drillholes	Minimum Depth m	Maximum Depth m	Total Metres m	Mean Depth m
Redtree	Historical	504	3.66	245.24	26,640.34	53
	LAM 2007-2008	126	13.5	302.70	8,855.00	70
	ALL	630	3.66	302.70	35,495.34	56
Huarabagoo	Historical	357	3.66	216.10	31,785.55	89
	LAM 2007-2012	39	80.00	201.00	4,616.39	118
	LAM 2022-2024	30	37.80	141.10	3,417.80	114
	ALL	426	3.66	216.10	39,819.74	93
Junnagunna	Historical	443	25.00	154.05	22,115.56	50
	LAM 2007-2012	41	50.00	152.40	4,18.40	102
	LAM 2022-2024	18	98.60	158.05	2,286.75	127
	ALL	502	25.00	158.05	28,570.71	57
Long Pocket	Historical	83	9.14	217.93	4,942.60	60
	LAM 2007-2012	12	60.00	71.20	747.50	62
	LAM 2022-2024	65	42.70	132.70	3,730.10	57
	ALL	160	9.14	217.93	9,420.20	59

Table 14.2.2 Summary of Drillhole Type Used to Inform MRE

Deposit	Type	No. of Drillholes	Minimum Depth m	Maximum Depth m	Total Metres m	Mean Depth m	% Holes	% Metres
Redtree	ALL	630	3.66	302.70	35,495.34	56.34	100.0	100.0
	DD	237	9.14	302.70	17,278.41	72.90	37.6	48.7
	PD	367	3.66	128.02	17,517.77	47.73	58.3	49.4
	WD	26	15.03	36.73	699.16	26.89	4.1	2.0
Huarabagoo	ALL	426	3.66	216.10	39,819.74	93.47	100.0	100.0
	DD	363	13.50	216.10	36,352.49	100.14	85.2	91.3
	PD	40	3.66	110.28	1,571.25	39.28	9.4	3.9
	RC	23	40.00	114.00	1,896.00	82.43	5.4	4.8
Junnagunna	ALL	502	25.00	158.05	28,570.71	56.91	100.0	100.0
	DD	276	28.77	158.05	18,004.84	65.23	55.0	63.0
	PD	13	31.00	108.00	1,154.87	88.84	2.6	4.0
	RC	213	25.00	150.00	9,411.00	44.18	42.4	32.9
Long Pocket	ALL	160	9.14	217.93	9,420.20	58.88	100.0	100.0
	DD	84	35.03	217.93	6,054.25	72.07	52.5	64.3
	PD	38	9.14	51.00	1,228.95	32.34	23.8	13.0
	RC	38	48.00	96.00	2,137.00	56.24	23.8	22.7

DD = diamond drilling; PD = percussion drilling; WD = 'wagon' percussion drilling; RC = reverse circulation.

Table 14.2.3 Summary of Assay Methods Used in MRE by Deposit and Company

Deposit	Company	Method Code	Assay Type	No. of Assays	% of Assays	Min. U_3O_8 ppm	Max. U_3O_8 ppm	Mean. U_3O_8 ppm	CoV
Redtree	CRA	XRF-A	XRf	731	9.1	1.00	13,000.00	766.82	1.71
	LAM	ME-XRF05	XRF	3,847	48.1	5.90	11,792.00	791.93	1.78
	LAM	U-XRF07	XRF	25	0.3	12,145.76	32,899.68	17,447.44	0.30
	MIM	RAD-MIM	Scint.	156	1.9	10.00	6,390.00	641.41	1.52
	MIM	XRF-MIM	XRF	686	8.6	1.00	14,770.00	1,102.72	1.64
	MINAD	XRF-A	XRF	94	1.2	100.00	8,200.00	734.04	1.61
	QML	GS3	Scint.	866	10.8	10.00	4,900.00	358.88	0.73
	QML	XRF-A	XRF	1,411	17.6	50.00	135,800.00	1,729.74	2.52
	UGA	XRF-1	XRF	112	1.4	25.00	16,500.00	1,093.36	2.39
	UGA	XRF-A	XRF	32	0.4	200.00	22,800.00	3,499.69	1.44
	UGA	XRF-G	XRF	44	0.5	10.00	910.00	110.23	1.32
Huarabagoo	CRA	XRF-A	XRF	769	11.8	10.61	22,518.90	643.49	2.04
	LAM	ME-MS61	ICP-MS	521	8.0	4.01	10,353.38	702.13	1.78
	LAM	ME-XRF05	XRF	847	13.0	2.36	9,280.30	546.82	1.59
	LAM	ME-XRF15b	XRF	3	0.0	15,801.28	23,171.28	18,316.91	0.23
	LAM	ME-XRF30	XRF	1	0.0	14,209.36	14,209.36	14,209.36	0.00
	MINAD	XRF-A	XRF	268	4.1	100.00	8,800.00	604.48	1.63
	OMEGA	XRF-5	XRF	119	1.8	11.00	17,000.00	2,490.48	1.26
	QML	GS3	Scint.	345	5.3	40.00	4,380.00	382.12	1.04
	QML	XRF-1	XRF	192	3.0	1.00	10,900.00	704.62	1.81
	QML	XRF-A	XRF	697	10.7	50.00	96,000.00	2,247.63	2.39
	UGA	XRF-1	XRF	2,733	42.1	10.00	48,700.00	1,263.56	2.38
Junnagunna	CRA	XRF-A	XRF	1,528	30.6	1.00	17,095.50	592.19	1.68
	LAM	ME-MS61	ICP-MS	137	2.7	8.84	4,610.67	396.05	1.71
	LAM	ME-XRF05	XRF	922	18.4	2.36	11,756.62	591.58	1.95
	LAM	U-XRF07	XRF	5	0.1	12,735.36	16,744.64	14,857.92	0.12
	UGA	XRF-1	XRF	2,034	40.7	1.00	19,100.00	850.81	1.72
	UGA	XRF-A	XRF	365	7.3	1.00	1,0000.00	426.28	1.91
	UGA	XRF-G	XRF	9	0.2	20.00	220.00	145.56	0.47
Long Pocket	CRA	XRF-A	XRF	8	0.8	66.00	1,250.00	386.38	1.07
	LAM	ME-MS61	ICP-MS	443	41.9	2.24	8,525.62	249.39	2.11
	LAM	ME-XRF05	XRF	177	16.7	2.36	8,183.65	317.39	2.57
	QML	GS3	Scint.	11	1.0	170.00	480.00	340.91	0.36
	QML	XRF-A	XRF	17	1.6	150.00	2,600.00	1,102.94	0.59
	UGA	XRF-1	XRF	250	23.7	8.00	4,800.00	456.52	1.62
	UGA	XRF-G	XRF	151	14.3	5.00	5,5070.00	917.41	5.22

Only assays falling within the mineralized wireframes are included. Scint = scintillometer. See Section 11 for further details of analytical methods.

14.3 Geological Interpretation and Mineralization Modelling

Mineralization is hosted almost exclusively in the Paleoproterozoic 'PTW4' Westmoreland sandstone conglomerate, with minor mineralization in the basaltic RT dyke and the basaltic Seigal Volcanics. Typically, mineralization is steep to sub-vertical within proximity to the Westmoreland dyke system which spans the length of RT, JG and HB, rolling over to horizontal to sub-horizontal mineralization approximately 20 m from the dyke edges and can extend laterally for 100 to 500 m or more perpendicular to the dyke. At HB, mineralization is dominantly sub-vertical and proximal to the dyke walls. At LP, mineralization is hosted in the PTW4 sandstone conglomerate, and is sub-horizontal and sits under intrusive basaltic sills.

Mineralization was modelled using implicit modelling techniques and using 100 to 200 ppm U_3O_8 cut-offs over a minimum 2 m interval to generate a mineralized shell. Non-assayed intervals were inserted into the assay file and given a value of 0.05 ppm U_3O_8 to prevent the model from extending through areas of null value. Variable anisotropy was used during implicit modelling of wireframes and was guided by the proximity to the dyke, volcanics or sills. Due to the scale of the Project, each area was modelled individually. Further details are provided in the following sub-sections.

14.3.1 Redtree

At RT, the dyke was modelled along with the base of PTW4 / roof of PTW3 which outcrops at surface; no other geological units are present. Mineralization is present on the SE and NW of the PTD1 dyke in PTW4, and is divided into the following lenses—Jack, Garee, Langi and Namalangi (Figure 14.3.1). The Jack Lens was modelled continuously but was subdivided into a steep (Jack S) and flat component to aid with variography and modelling. Minor additional mineralization was present in the dyke itself, and this was modelled separately and contained to the dyke. Thirty-seven drillholes were ignored due to the proximity of more recent drilling or due to lack of assay data (CRA metallurgical drillholes) (Table 14.3.1). Example cross sections are shown in Figure 14.3.2 and Figure 14.3.3.

Figure 14.3.1 Redtree Mineralization and Dyke Wireframes

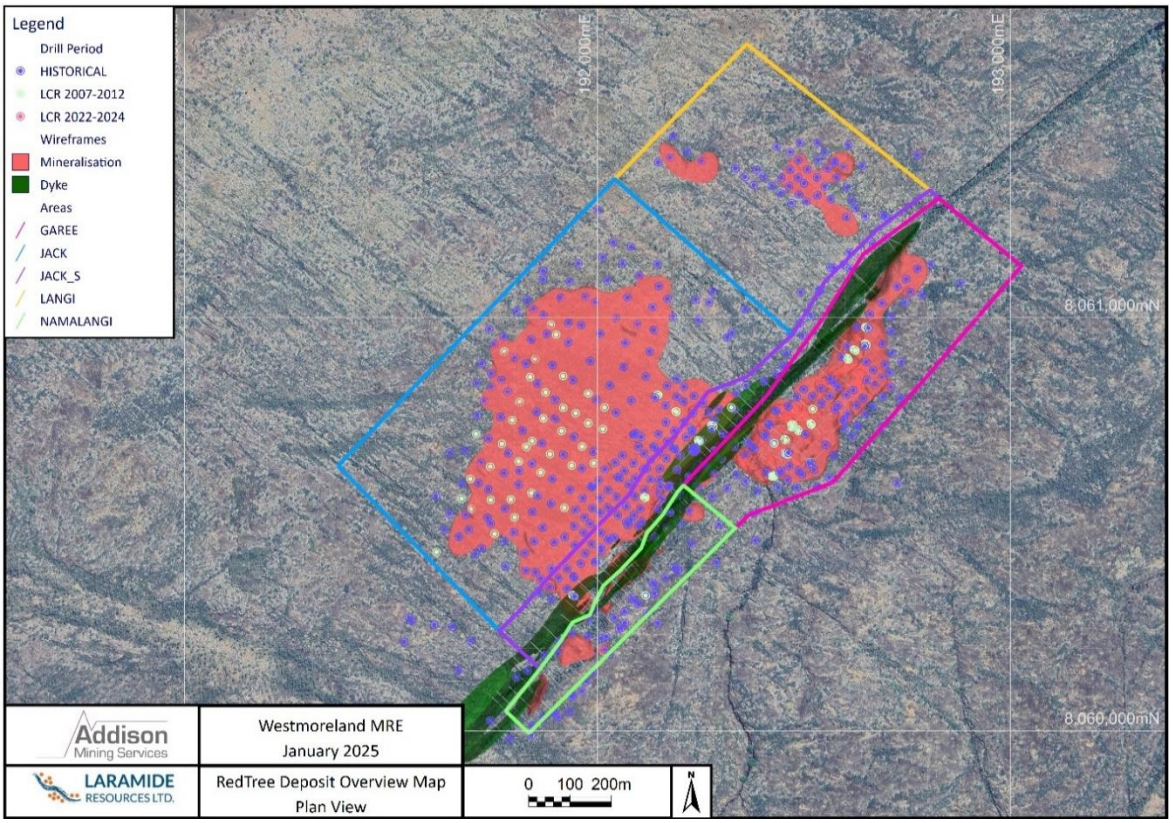


Table 14.3.1 Ignored Drillholes Redtree

Hole ID	Hole Type	Year	Company
DD011	DD	1969	MIM
DD92RT104	DD	1992	CRA
DD92RT105	DD	1992	CRA
DD92RT106	DD	1992	CRA
DD92RT107	DD	1992	CRA
DD92RT108	DD	1992	CRA
DD92RT109	DD	1992	CRA
DD92RT110	DD	1992	CRA
DD92RT111	DD	1992	CRA
DD92RT112	DD	1992	CRA
DD92RT113	DD	1992	CRA
DD92RT114	DD	1992	CRA
DD92RT115	DD	1992	CRA

Hole ID	Hole Type	Year	Company
DD92RT116	DD	1992	CRA
DD92RT129	DD	1992	CRA
DD011	DD	1969	MIM
DDI016	DD	1970	MIM
PD90RT017	PD	1990	CRA
PD90RT023	PD	1990	CRA
PD90RT024	PD	1990	CRA
PD90RT026	PD	1990	CRA
PD90RT036	PD	1990	CRA
PD90RT037	PD	1990	CRA
PD90RT055	PD	1990	CRA
PD90RT064	PD	1990	CRA
PDNL005	PD	1970	QML

Figure 14.3.2 Redtree Example Cross-Section of Wireframe Interpretation - Garee Lens

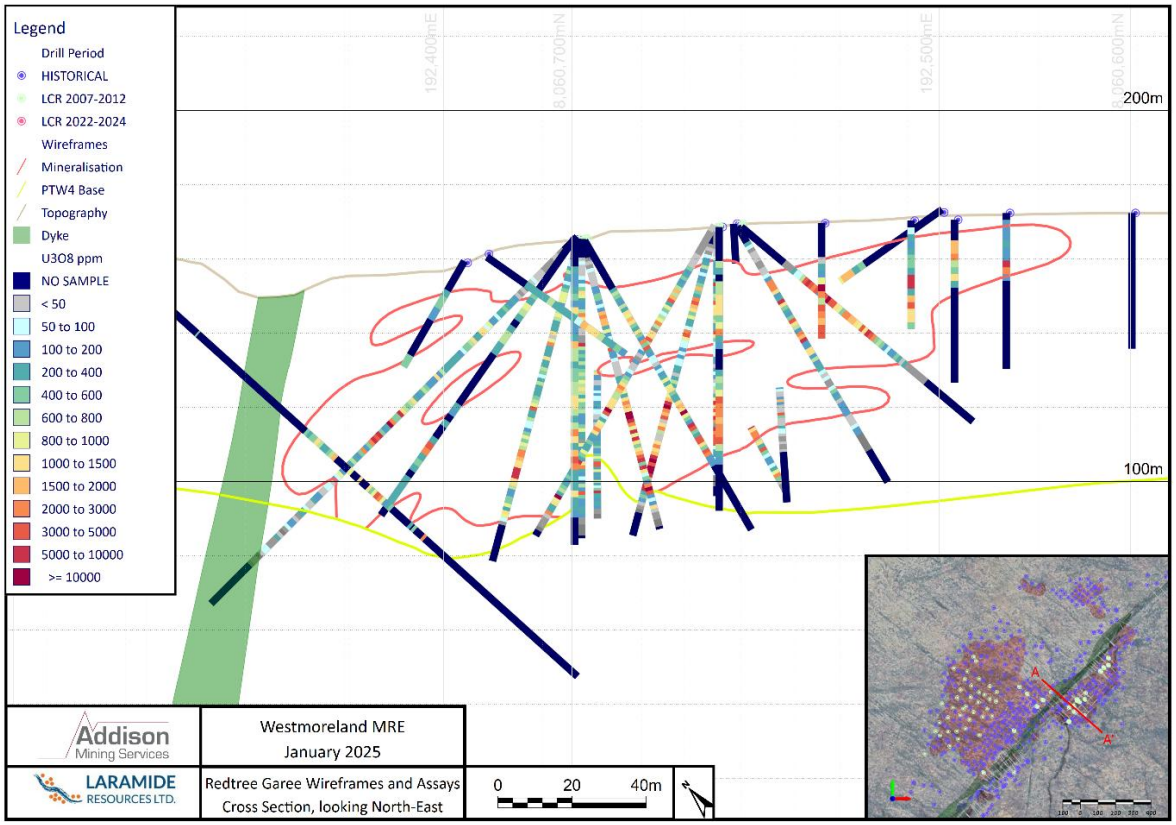
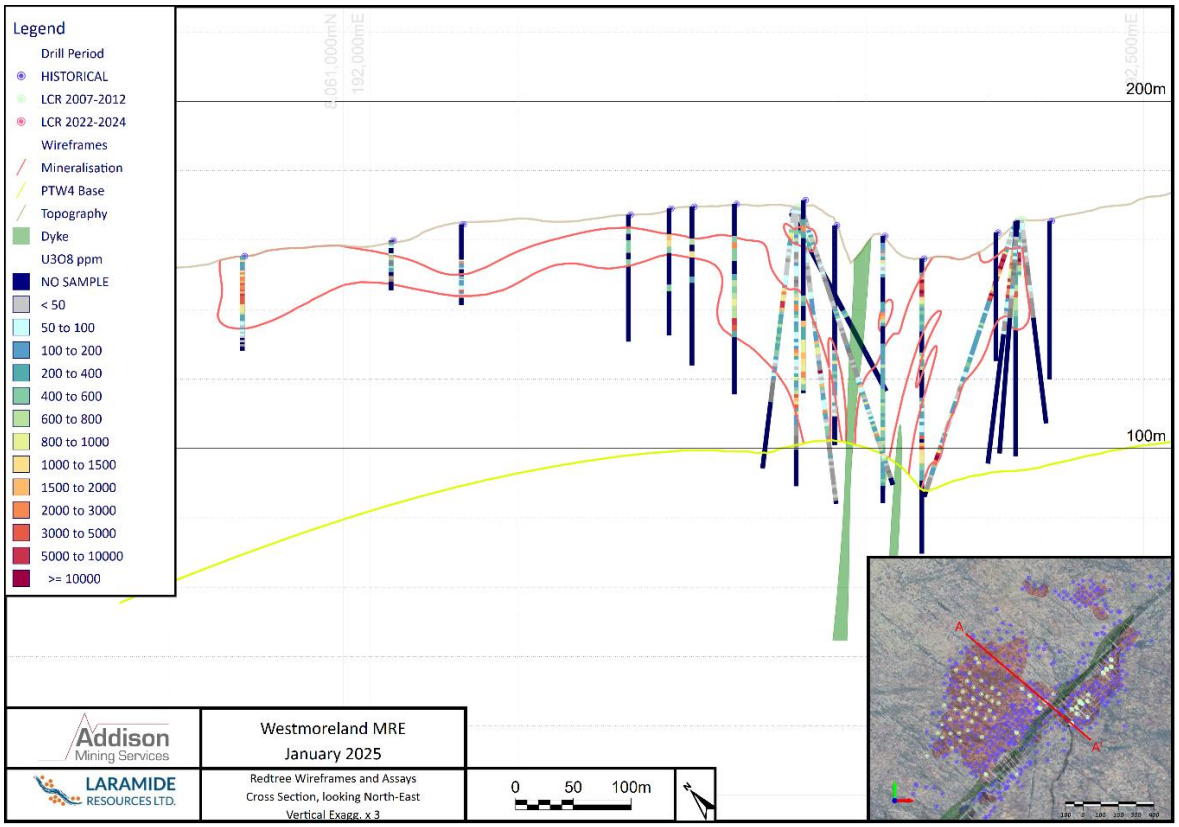


Figure 14.3.3 Redtree Example Cross-Section of Wireframe Interpretation – Garee, Jack and Jack Steep Lenses



Vertical exaggeration times three.

14.3.2 Huarabagoo

Mineralization at HB is interpreted to be steep and proximal to the walls of the dyke. The dyke was modelled by correlation of major dyke material drilling intercepts as six dominant units which appear to have been emplaced with a dextral sense of shear. The correlated dyke units were modelled using the Micromine Implicit Vein Network tool, individual centre of mass surfaces were generated for each body, and these surfaces were in turn used to create dynamic anisotropy models to guide the mineralization around the geometry of the dyke. Due to the geographic size of the deposit with a strike length of approximately 2.4 km, the deposit was split into three sections for modelling to allow the implicit models to be generated at a finer mesh size and maximize computing power. A mineralized shell was generated for the uranium mineralization using the dynamic model, and was truncated by the base of the PTW4 unit. In the north of the model area, alluvial cover is present and this was used to trim the model also. Example images are shown in Figure 14.3.4 and Figure 14.3.5.

Figure 14.3.4 Huarabagoo Dyke and Mineralization Wireframes Plan Map

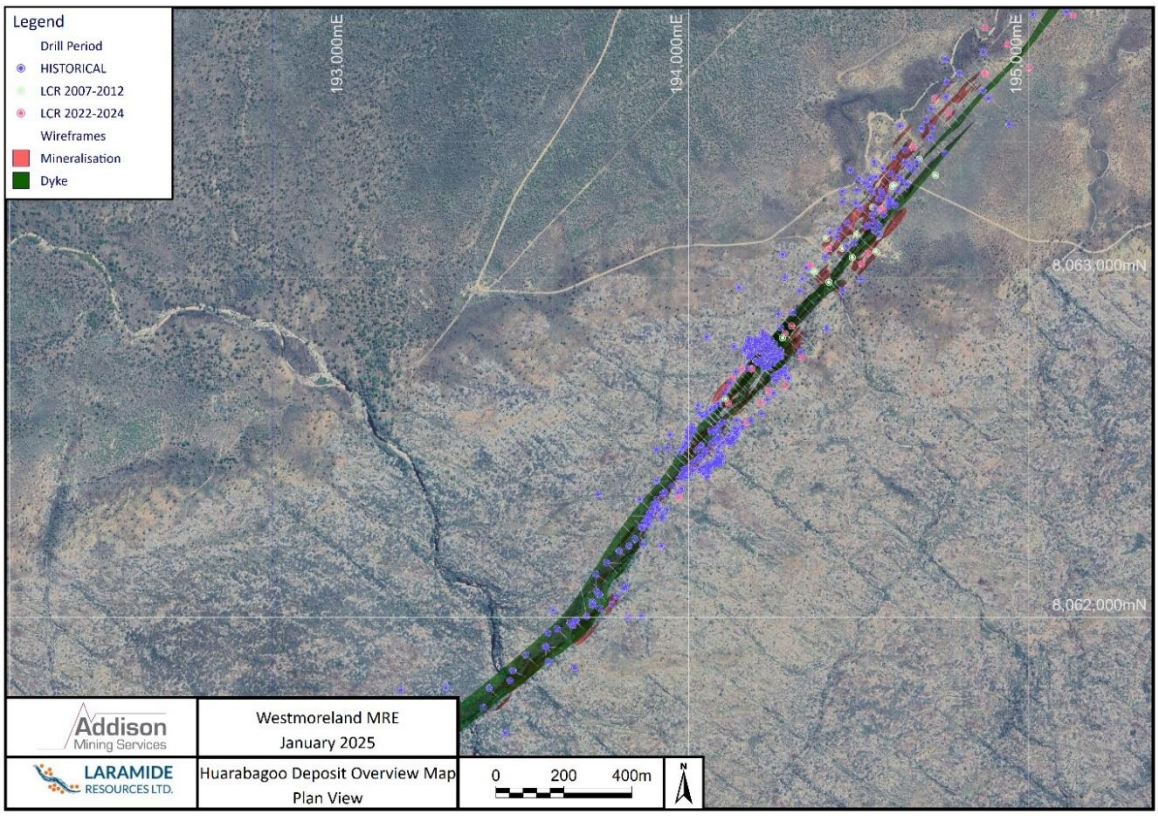
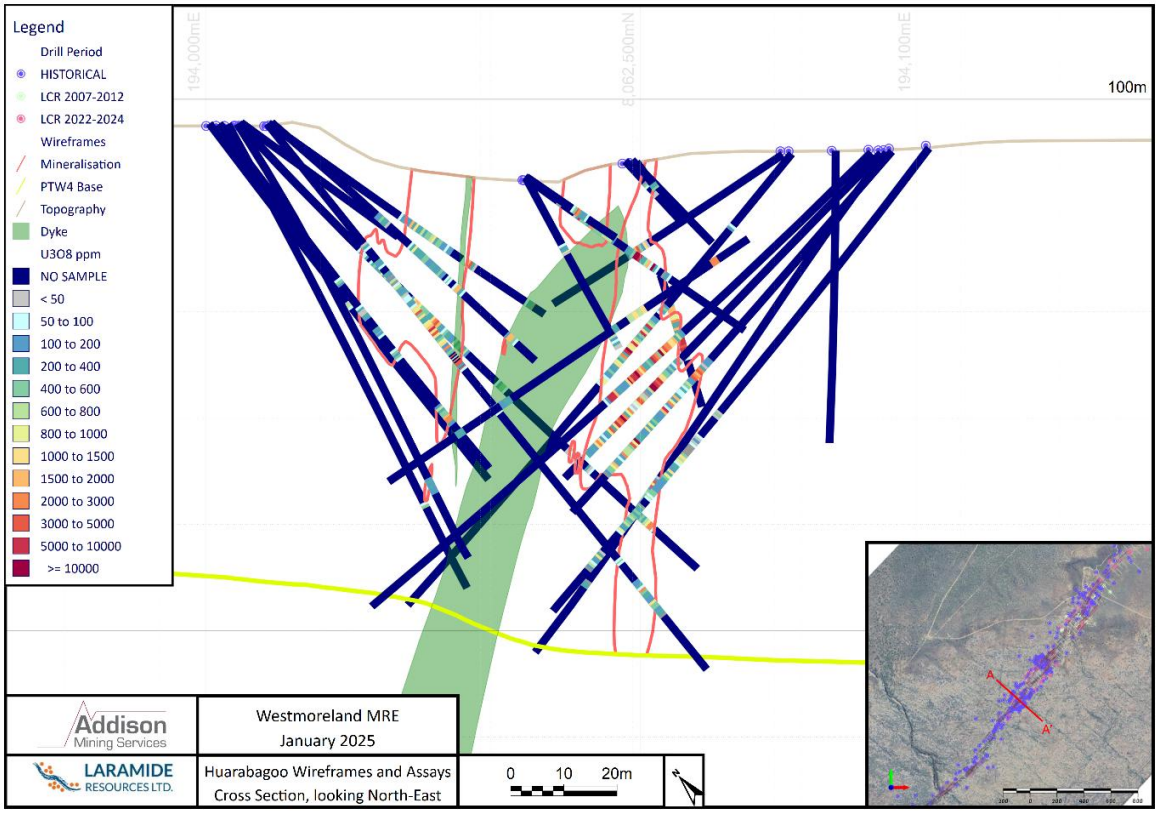


Figure 14.3.5 Huarabagoo Example Cross-Section



14.3.3 Junnagunna

At JG, mineralization is flat on either size of the dyke and extends 400 to 500 m from the dyke to the NW and up to 200 m in the SE. Mineralization sits under the 'PTS' volcanics unconformity and rolls over to become steep 20 to 30 m from the edge of the dyke. The dyke was modelled using the vein modeller and the centre of mass surface used to generate a dynamic trend model along with surfaces representing the trend of the flatter mineralization. The dynamic trend model was used to generate a mineralized wireframe shell. Example images are shown in Figure 14.3.6 and Figure 14.3.7.

Figure 14.3.6 Junnagunna Dyke and Mineralization Wireframes Plan Map

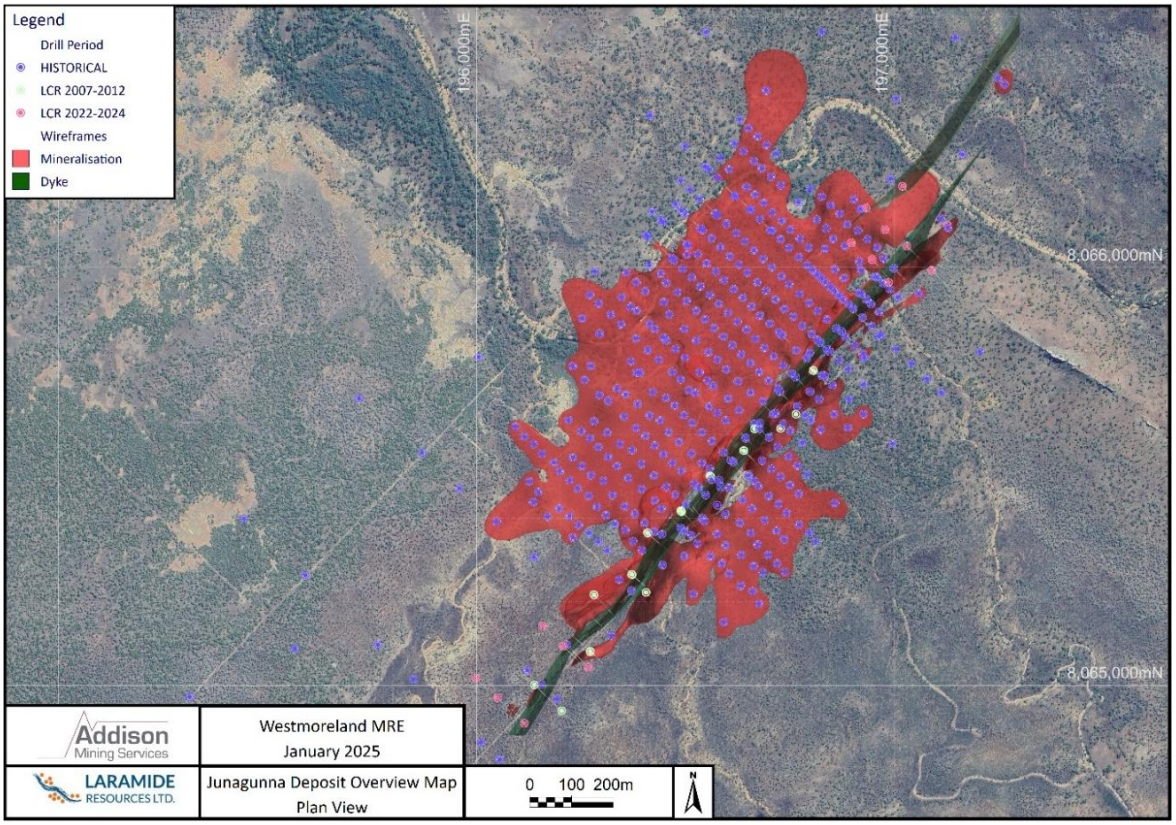
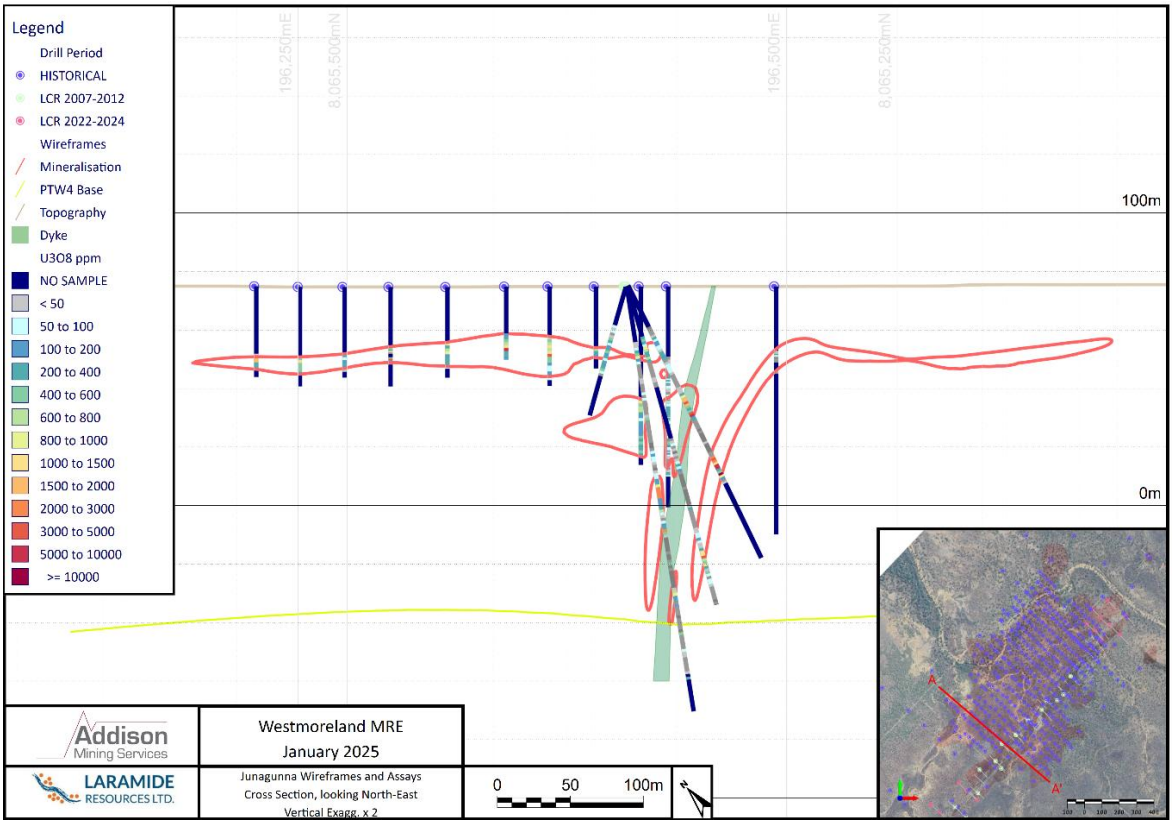


Figure 14.3.7 Junnagunna Example Cross-Section



Vertical exaggeration times two.

14.3.4 Long Pocket

At LP, mineralization is hosted in the PTW4 sandstone conglomerate and is sub horizontal and sits under intrusive PTD1 basaltic sills. The sill units were correlated and modelled using the vein implicit vein modelling tool and the centre surfaces used to create a dynamic model; the dynamic model was then in turn used to model the mineralization. Example images are shown in Figure 14.3.8 and Figure 14.3.9.

Figure 14.3.8 Long Pocket Dyke / Sill and Mineralization Wireframes Plan Map

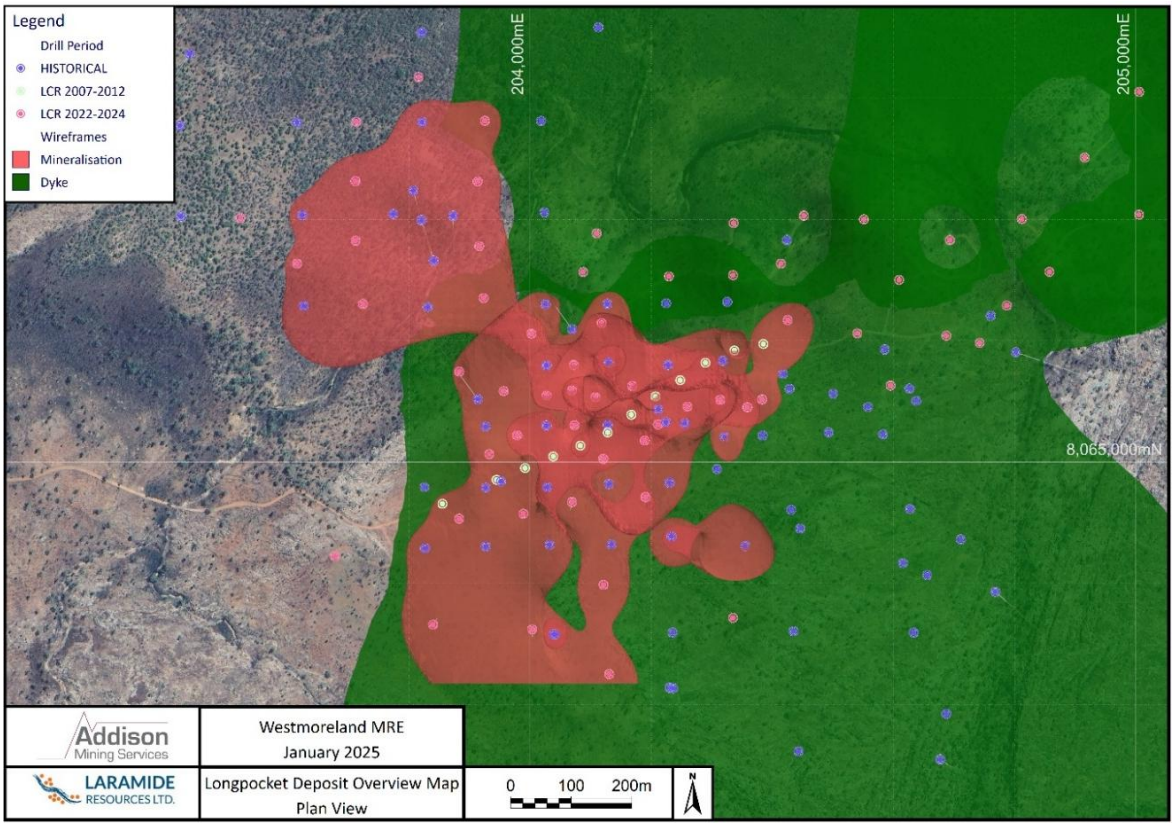
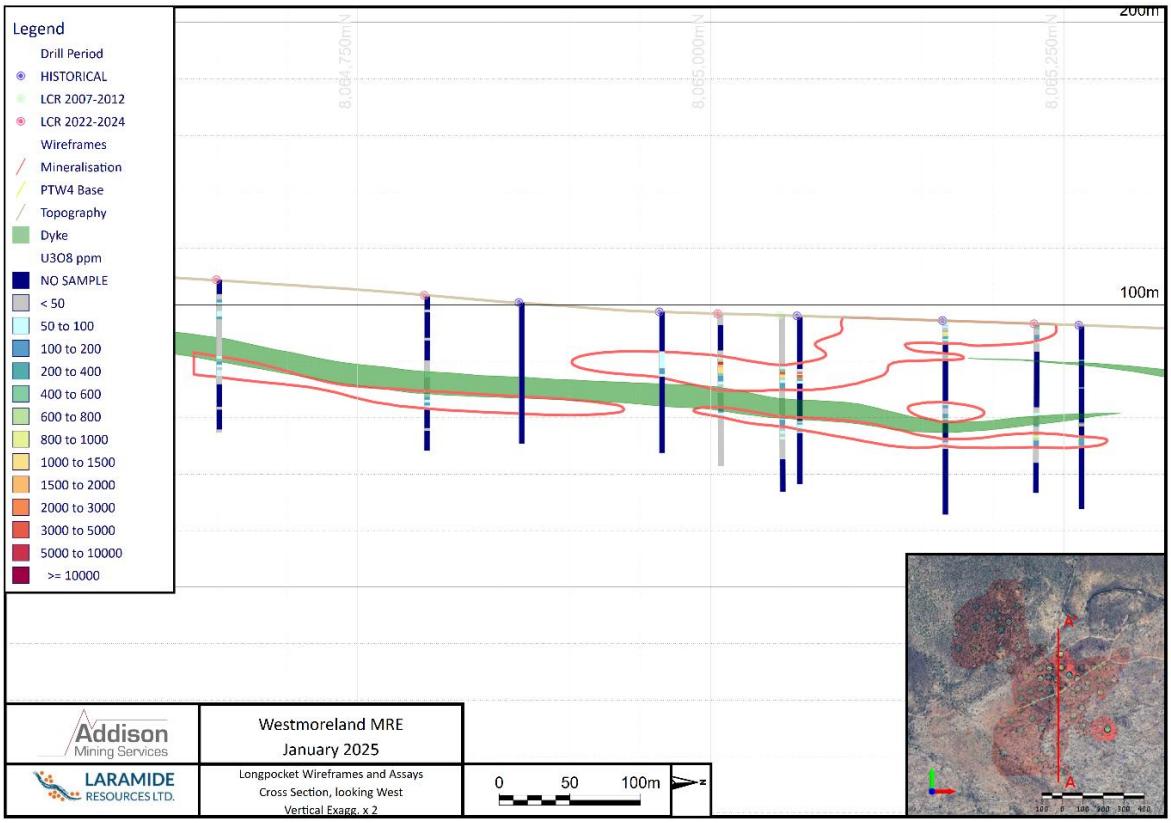


Figure 14.3.9 Long Pocket Example Cross-Section



Vertical exaggeration times two.

14.4 Grade Compositing and Variography

All assay data falling within the wireframe was composited to 2 m intervals. The minimum accepted length was 1 m, and residual values were added to the last composite; length weighted averaging was used.

Directional semi-variograms were generated for the RT Jack, Jack Steep and Garee lenses as well as for HB, JG and LP. Normal scores transform or median indicator transform models were used according to which type gave the most meaningful structures. The normal scores models were back transformed by Micromine Origin and Beyond software prior to estimation. At RT, meaningful models could not be generated for the dyke and Namalangi lenses, which used the Jack Steep semi-variogram models during estimation, and the Langi Lens which used the Jack semi-variogram model. Example experimental models and fitted models are shown in Figure 14.4.1 to Figure 14.4.4.

Table 14.4.1 Semi-Variogram Parameters and Axis Directions

Variogram	Nugget	Sill 1	Sill 2	Axis 1		Axis 2		Axis 3	
				Range 1	Range 2	Range 1	Range 2	Range 1	Range 2
RT Jack	0.3727	0.5078	0.7028	51.7	76.2	52.8	87.3	3.9	8.24
RT Jack Steep	0.28	0.5353	0.4064	32	94	10	27	7.53	8.58
RT Garee	0.1075	0.0475	0.095	5.9	18.67	8.82	29.41	4.3	10
HB	0.1	0.083	0.067	15	45	10	20	5	15
JG	0.09	0.098	0.062	50	104	13	62	3.72	6.4
Long Pocket	0.5	0.25	0.25	48	88	35	60	3	8
	Model Type			Azimuth	Plunge	Azimuth	Plunge	Azimuth	Plunge
RT Jack	Normal Scores			230	25	320	25	320	20
RT Jack Steep	Normal Scores			43.2	3.8	137.8	49.7	130.0	-40.0
RT Garee	Median Indicator			240.2	3.8	330.9	10.2	310.0	-79.0
HB	Median Indicator			216	0	306	86.06	306	-3.94
JG	Median Indicator			44	0	134	0	0	-90
Long Pocket	Normal Scores			60	0	150	0	0	-90

Figure 14.4.1 Garee Semi-Variograms

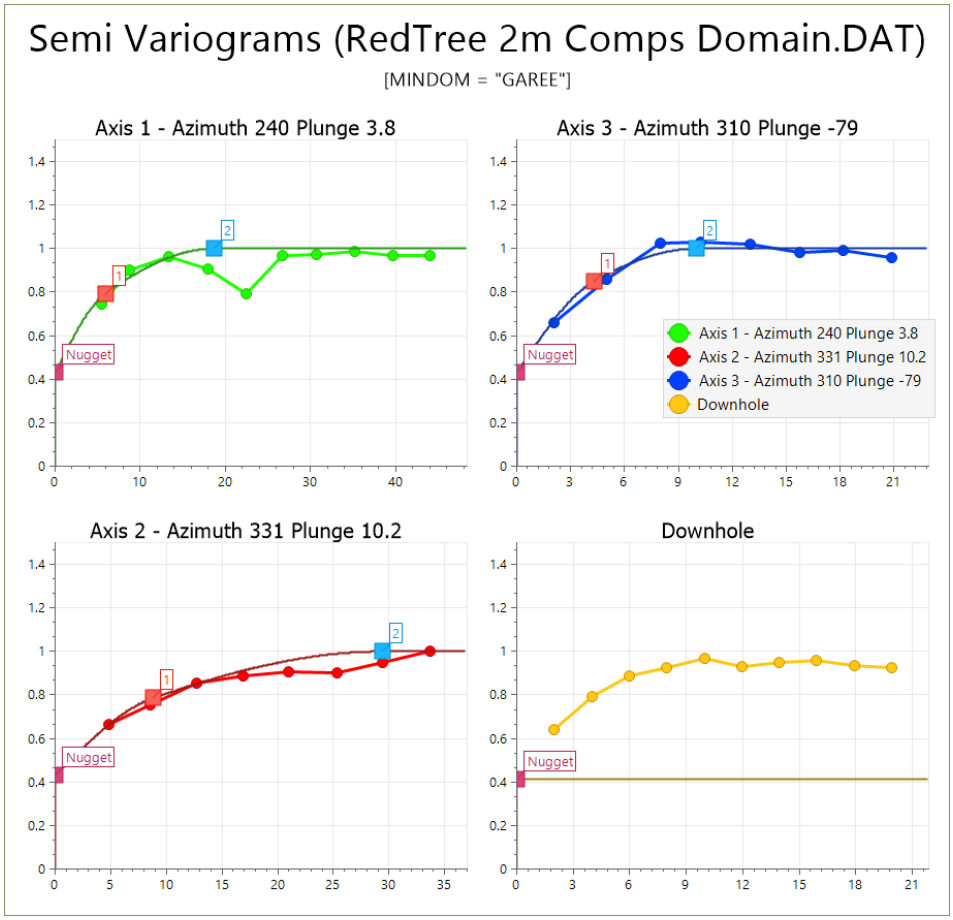


Figure 14.4.2 Huarabagoo Semi-Variograms

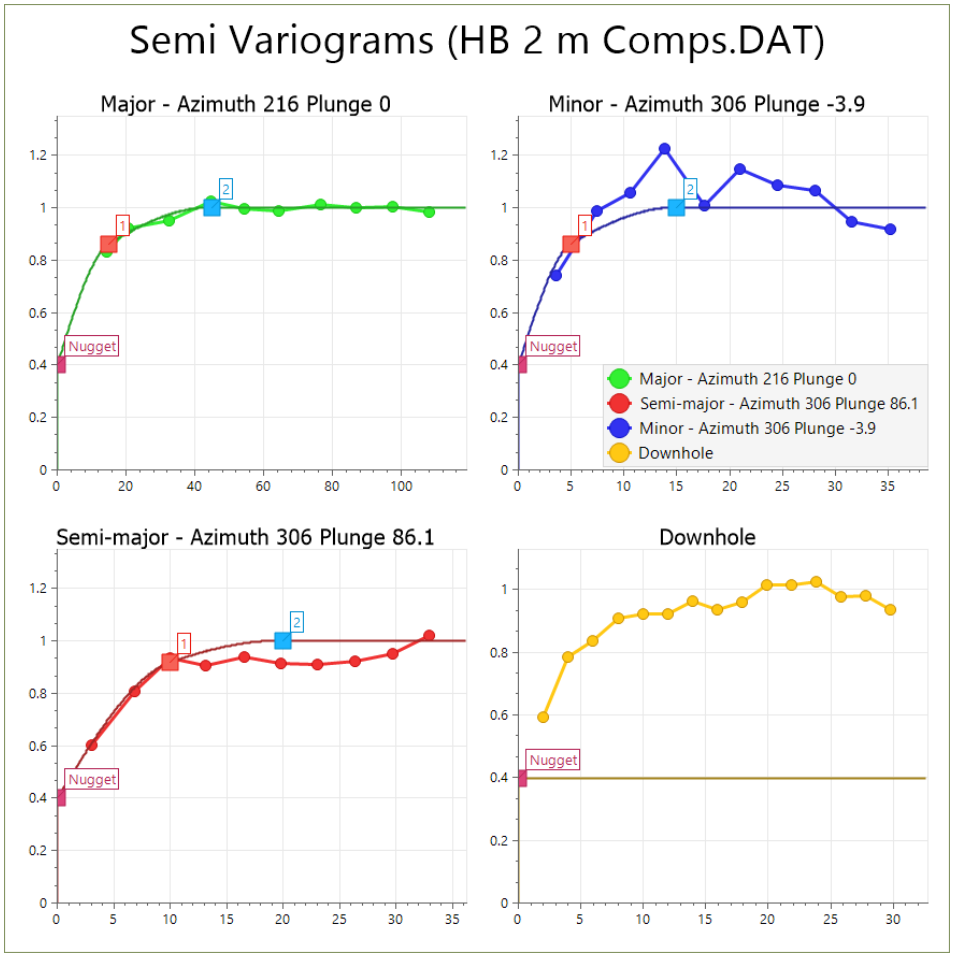


Figure 14.4.3 Junnagunna Semi-Variograms

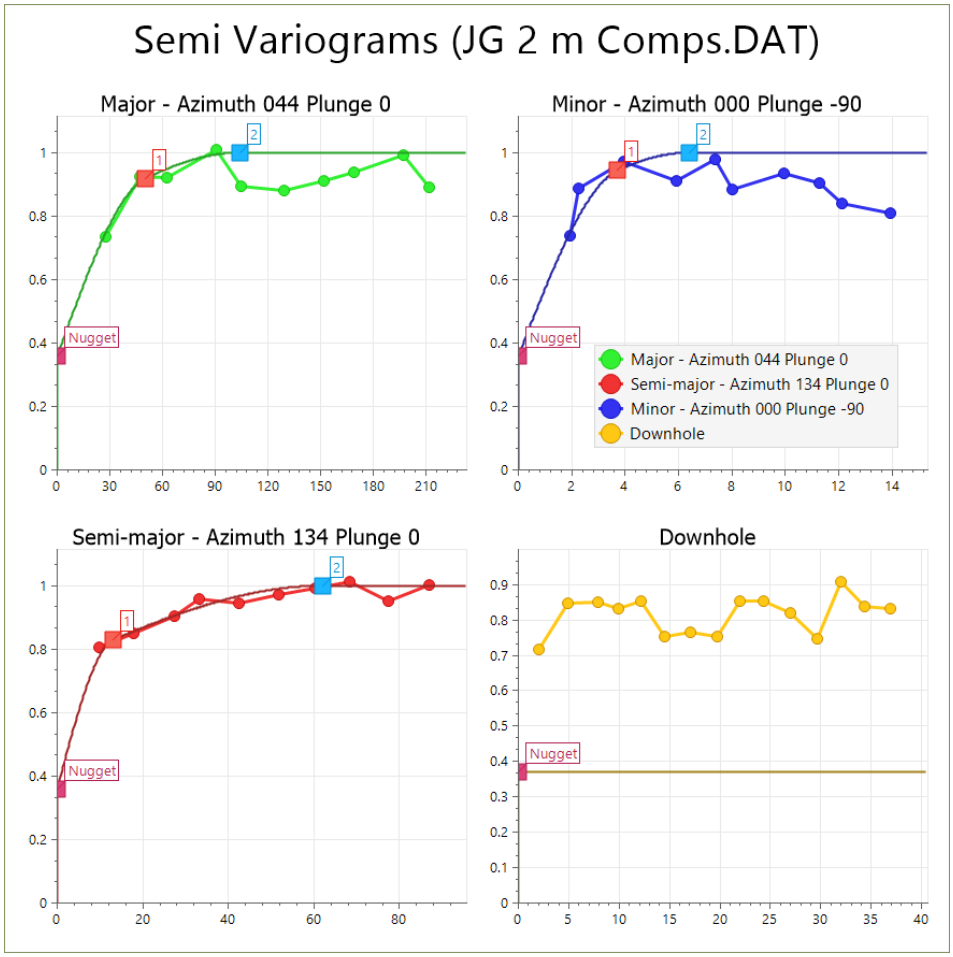
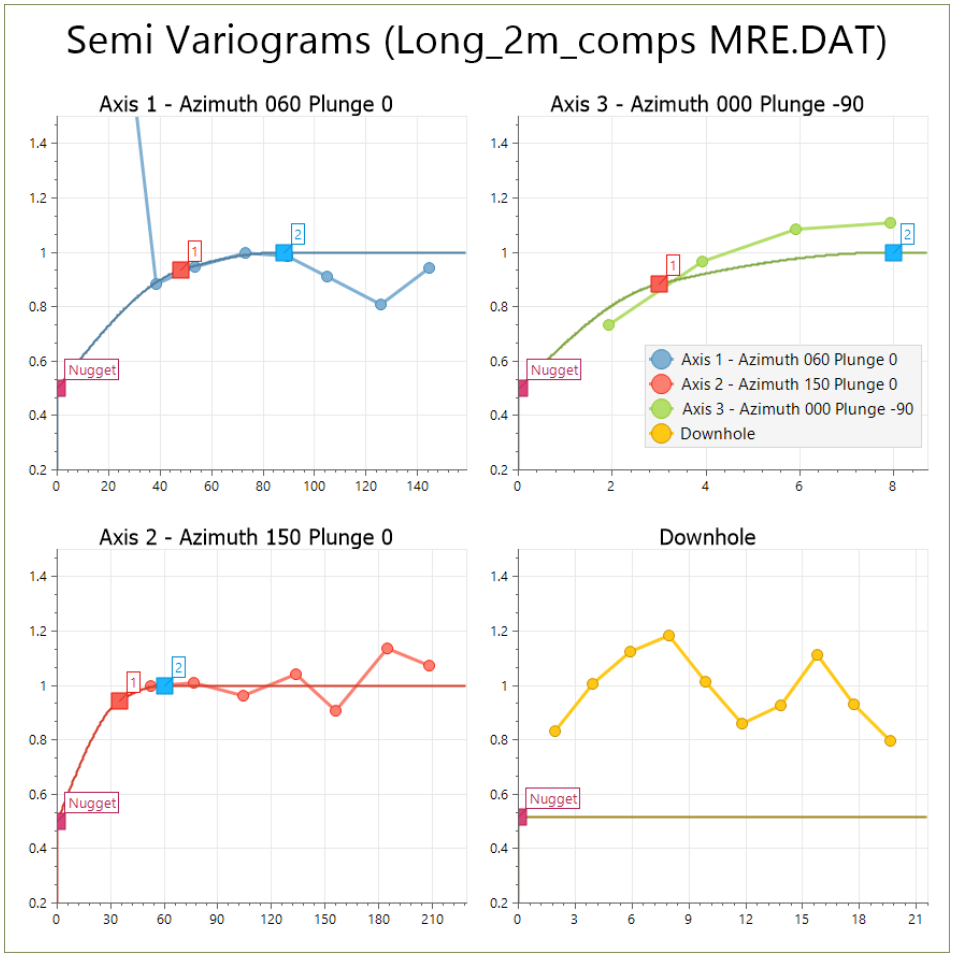


Figure 14.4.4 Long Pocket Semi-Variograms



14.5 Block Model Parameters

A blank block model was created for each deposit. The dynamic anisotropy models were written to the block model for each deposit for use in block estimation, and the block models were then restricted to the mineralization wireframes and sub-blocked accordingly. The RT Group of deposits (RT, HB and JG) were rotated 40° clockwise around the Z axis to better honor the strike of the mineralization. Block model parameters are shown in Table 14.5.1.

Table 14.5.1 Block Model Parameters

Direction	Min. Centre	Block Size m	Max. Centre	Azimuth Rotations	No. of Sub Block Divisions
Redtree					
East	191630	10	192810		5
North	8060040	10	8061430	40	5
Z	50	5	210		2
Huarabagoo					
East	193375	5	194890		2
North	8061700	10	8063630	40	2
Z	-40	5	120		5
Junnagunna					
East	196010	10	197280		2
North	8064900	15	8066760	40	3
Z	-40	5	65		5
Long Pocket					
East	203589	20	204469		10
North	8064555	20	8065595	40	10
Z	45	4	105		2

14.6 Grade Estimation

All block models were estimated using ordinary kriging on a domain-by-domain basis and at parent block scale. Within the RT Jack and Jack Steep lenses, a soft boundary was used to allow samples to inform block on either side of the boundary. Dynamic anisotropy was used which rotates the search and variogram axis during estimation of each block. Discretization of points within each block was applied accordingly to honor change of support effects when estimating from points to blocks. A single sector search was used in each estimate and the maximum number of points in each drillhole restricted. Incrementally larger searches were used and a minimum number of drillholes applied. At LP, the parent cell size was doubled in areas of more sparse drilling. Table 14.6.1 sets out the estimation parameters for each domain. The handling of high-grade values is discussed in the following section.

Table 14.6.1 Block Estimation Parameters

Domain	Pass	Radius 1 m	Radius 2 m	Radius 3 m	Max. Samples	Min. Hole Count	Max. Sample per Hole	Block Size Multiplier	Discretization X, Y, Z
RT Dyke	1	100	100	10	24	3	3	1	5,5,2
RT Dyke	2	100	100	10	24	1	3	1	5,5,2
RT Garee	1	40	40	10	24	3	3	1	5,5,2
RT Garee	2	40	40	10	24	1	3	1	5,5,2
RT Jack / Steep	1	60	40	10	24	3	3	1	5,5,2
RT Jack / Steep	2	60	40	10	24	1	3	1	5,5,2
RT Langi	1	60	40	10	24	3	3	1	5,5,2
RT Langi	2	60	40	10	24	1	3	1	5,5,2
RT Namalangi	1	60	40	10	24	3	3	1	5,5,2
RT Namalangi	2	60	40	20	24	1	3	1	5,5,2
Huarabagoo	1	30	30	10	24	3	3	1	2,5,2
Huarabagoo	2	50	50	10	24	3	3	1	2,5,2
Huarabagoo	3	50	50	10	24	2	3	1	2,5,2
Huarabagoo	4	100	100	20	24	2	3	1	2,5,2
Junnagunna	1	75	45	10	36	3	4	1	5,5,2
Junnagunna	2	100	60	15	36	3	4	1	5,5,2
Junnagunna	3	150	150	20	36	2	4	1	5,5,2
Junnagunna	4	150	150	20	36	1	4	1	5,5,2
Long Pocket	1	50	50	10	24	3	3	1	10,10,2
Long Pocket	2	75	75	15	24	1	3	1	10,10,2
Long Pocket	3	75	75	15	24	2	3	2	10,10,2
Long Pocket	4	150	150	30	24	1	3	2	10,10,2

14.7 Top Cutting and Grade Distance Thresholds

In order to control the influence of high-grade samples, a process of distance thresholds was applied whereby a composite sample is allowed to keep its original value within a given percentage of the search radius; outside of this radius, the grade is capped to the threshold value. The grade thresholds were selected after inspection of domain histograms and review of the block model and input data following test kriging runs. The thresholds and distances are presented in Table 14.7.1.

Table 14.7.1 Grade Capping Distance Threshold

Domain	Range /m								
	Axis 1	**Axis 2**	**Axis 3**	**Axis 1**	**Axis 2**	**Axis 3**	**Axis 1**	**Axis 2**	**Axis 3**
Threshold U_3O_8 ppm	**10,000**			**8,100**			**5,000**		
RT Dyke	10	10	1	20	20	2	60	60	6
RT Garee	4	4	1	8	8	2	24	24	6
RT Jack / Steep	6	4	1	12	8	2	36	24	6
RT Langi	6	4	1	12	8	2	36	24	6
RT Namalangi	6	4	1	12	8	2	36	24	6
Threshold U_3O_8 ppm	**10,000**			**8,000**			**-**		
Huarabagoo	10	10	3	20	20	6	-	-	-
Junnagunna	10	10	3	20	20	6	-	-	-
Threshold U_3O_8 ppm	**3,000**			**1,500**			**-**		
Long Pocket	25	25	5	56	56	10	-	-	

14.8 Block Model Validation

The block models were validated by comparison of input and output data histograms (Figure 14.8.1 to Figure 14.8.4), global statistics, and inspection of the models in cross section (Figure 14.8.5 to Figure 14.8.9) and 3D. Table 14.8.1 shows statistics for the composites and block models by domain. Particular attention was paid to the mean and weighted mean of the composite data versus volume weighted mean of the block models. The weighting of the composites was assigned by cell delustering to remove bias brought about by data clustering that results from favorably drilling high grade areas. The weighted mean values compare favorably with some discrepancy in RT Langi, Namalangi and the dyke where the number of data points are comparatively few.

Table 14.8.1 Domain Composite and Block Model Statistics

	RT Jack / Steep		RT Garee		RT Langi		RT Namalangi	
	Comps.	**BM**	**Comps.**	**BM**	**Comps.**	**BM**	**Comps.**	**BM**
Min. Value	7.3	67.8	5.9	117.7	115.0	115.0	65.0	189.7
Max. Value	15,000.0	6,572.8	15,000.0	6,436.8	3,069.5	1,921.9	15,000.0	4,367.9
2nd Highest	15,000.0	6,390.2	15,000.0	6,436.8	3,043.0	1,908.3	13,278.5	4,053.0
3rd Highest	12,446.5	5,281.4	15,000.0	6,436.8	2,932.6	1,792.7	10,700.5	3,865.4
4th Highest	9,457.2	5,162.3	13,608.0	4,656.8	2,864.0	1,792.7	10,056.8	3,865.4
No. of Points	1,640.0	21,865.0	2,452.0	18,430.0	57.0	1,453.0	152.0	3,532.0
Mean	938.0	819.3	889.5	746.3	829.0	664.2	1,477.5	1,069.4
Wtd. Mean	860.0	878.4	791.7	797.0	836.2	708.3	1,310.9	1,092.6
Wtd. Std. Dev.	1,174.9	523.9	1,385.7	580.8	695.5	374.7	2,312.8	627.8
Std. Dev.	1,271.0	452.2	1,523.0	558.4	737.3	365.2	2,480.4	636.8
CoV	1.4	0.6	1.7	0.7	0.9	0.5	1.7	0.6
Median	509.2	724.1	387.6	563.7	642.1	597.7	513.8	1,027.6
	RT Dyke		**Huarabagoo**		**Junnagunna**		**Long Pocket**	
	Comps.	**BM**	**Comps.**	**BM**	**Comps.**	**BM**	**Comps.**	**BM**
Min. Value	46.6	78.1	0.1	45.6	0.1	62.7	2.5	57.1
Max. Value	15,000.0	4,548.7	33,914.5	9,796.3	10,565.6	3,836.6	9,276.8	1,979.4
2nd Highest	7,000.0	4,548.7	20,952.3	9,528.7	9,900.0	3,836.6	5,501.0	1,979.4
3rd Highest	6,498.0	4,519.7	18,490.0	9,528.7	7,812.2	3,729.5	5,052.9	1,979.4
4th Highest	4,937.0	4,486.2	17,676.6	9,528.7	7,125.0	3,729.5	2,892.0	1,979.4
No. of Points	62.0	1,744.0	2,724.0	23,203.0	1,984.0	28,576.0	449.0	74,927.0
Mean	1,627.1	1,446.2	917.3	867.8	610.9	600.2	303.5	270.8
Wtd. Mean	1,468.2	1,615.9	885.6	848.2	600.9	614.8	287.4	273.1
Wtd. Std. Dev.	2,008.6	1,305.1	1,780.0	643.0	817.0	361.0	566.0	169.0
Std. Dev.	2,436.6	1,115.4	1,754.0	652.0	902.0	346.0	650.0	167.0
CoV	1.5	0.8	1.9	0.8	1.5	0.6	2.1	0.6
Median	496.5	1,101.6	365.5	685.2	300.4	530.2	133.3	231.9

Figure 14.8.1 Redtree Composite and Block Model Histograms

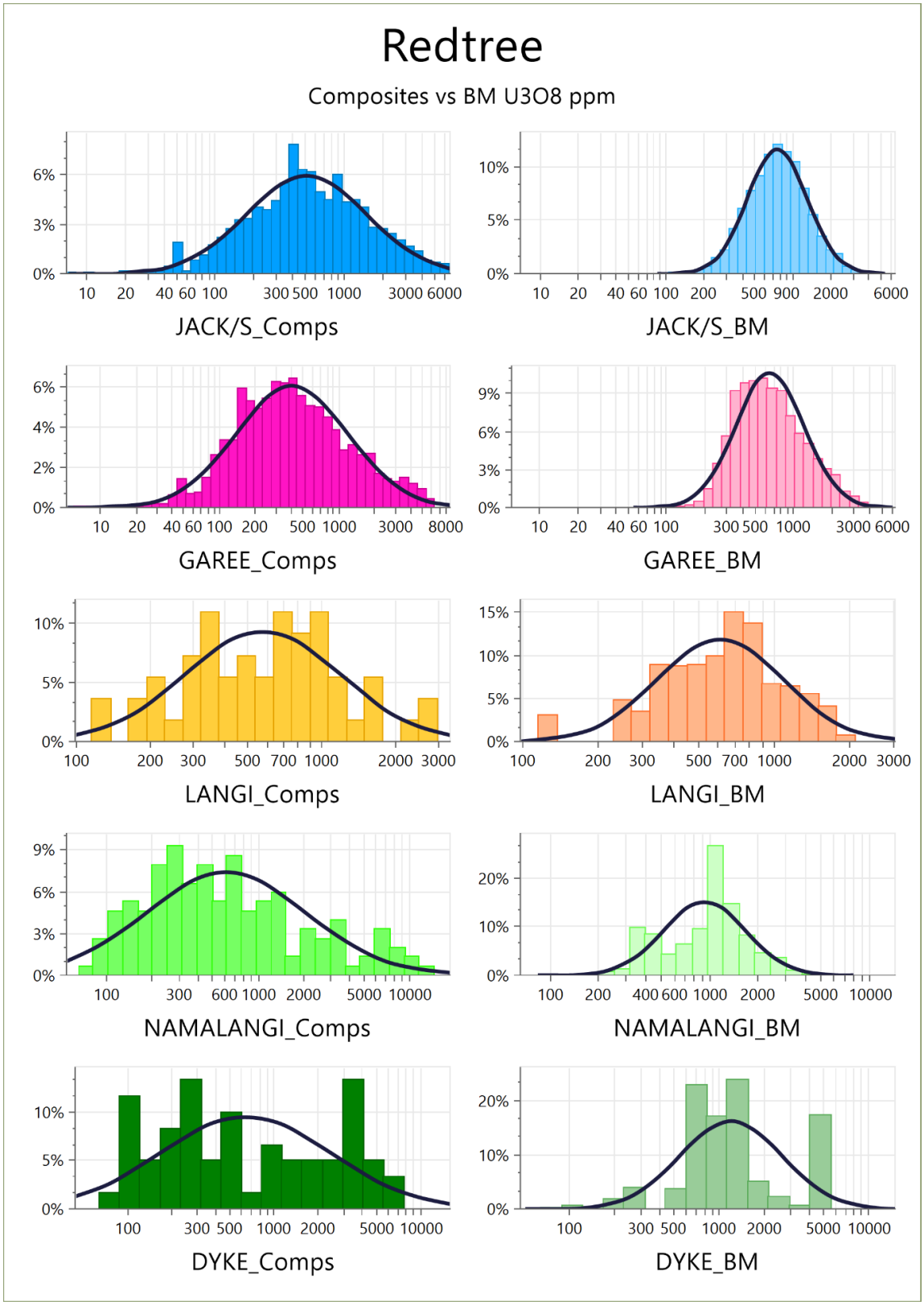


Figure 14.8.2 Huarabagoo Composite and Block Model Histograms

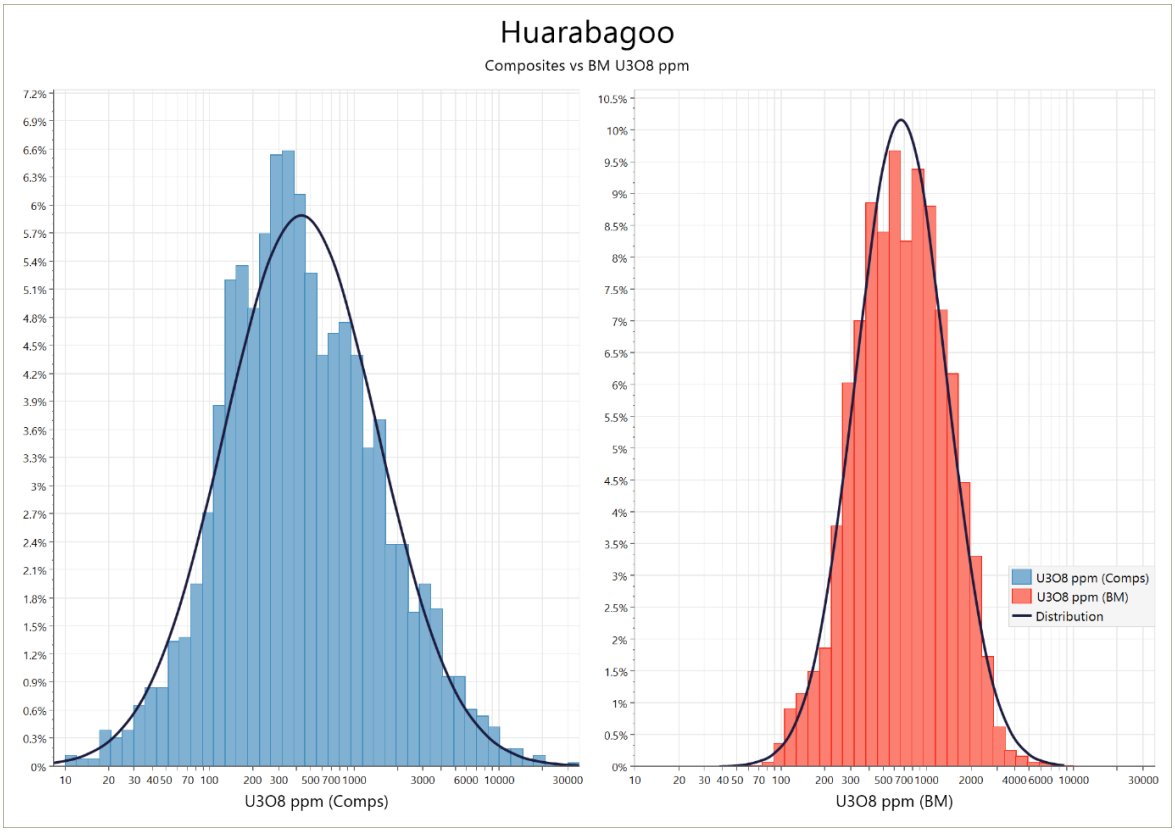


Figure 14.8.3 Junnagunna Composite and Block Model Histograms

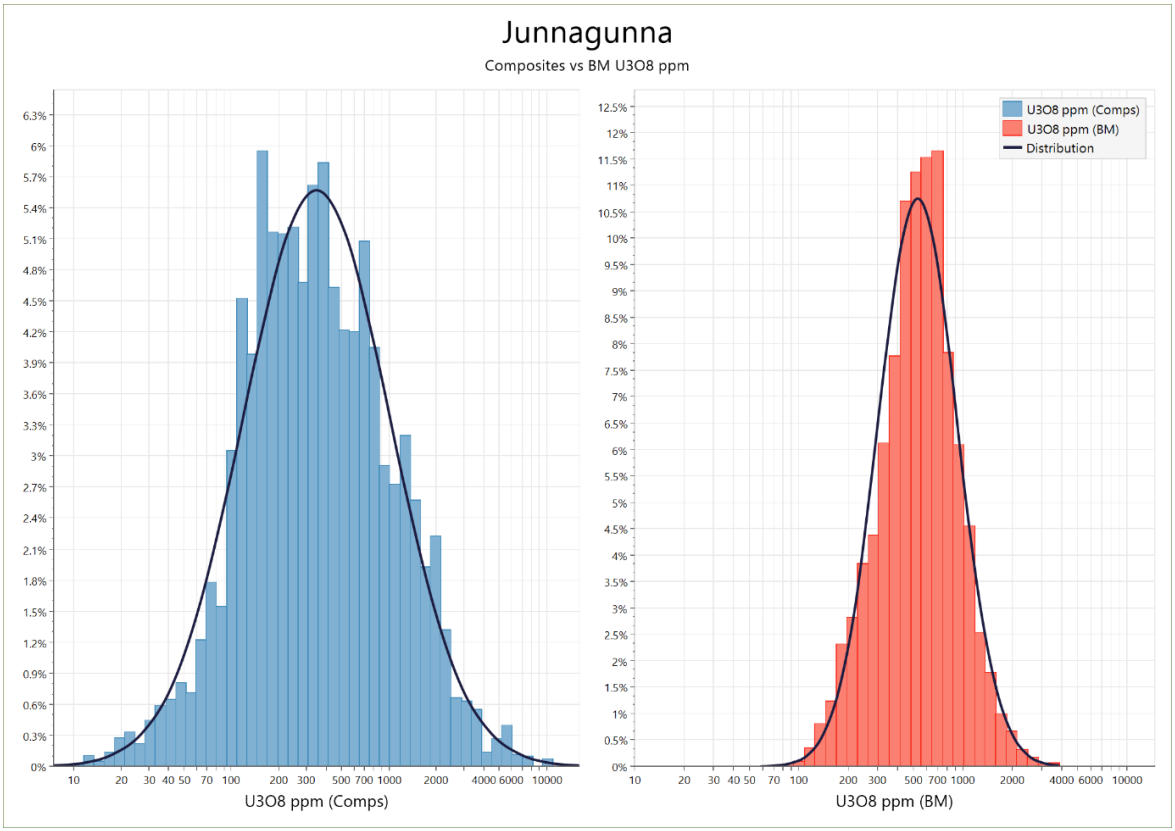


Figure 14.8.4 Long Pocket Composite and Block Model Histograms

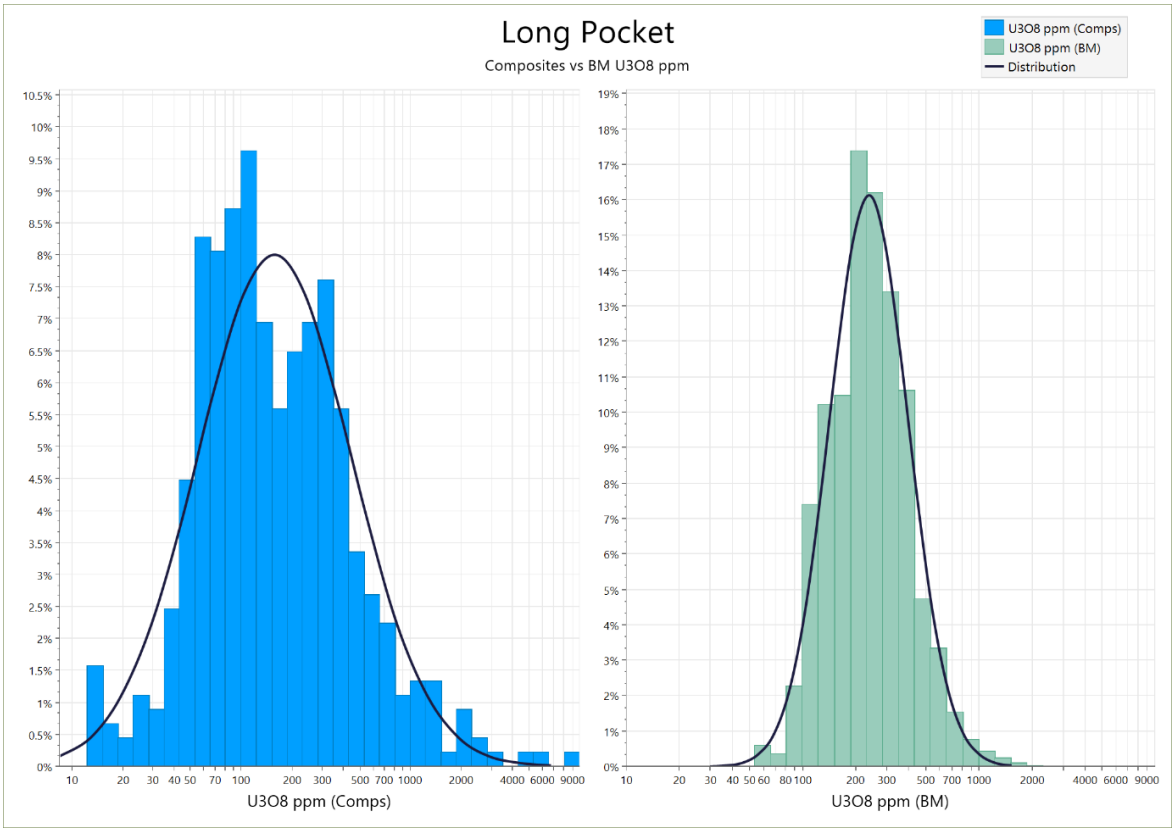


Figure 14.8.5 Redtree Garee Block Model Cross Section

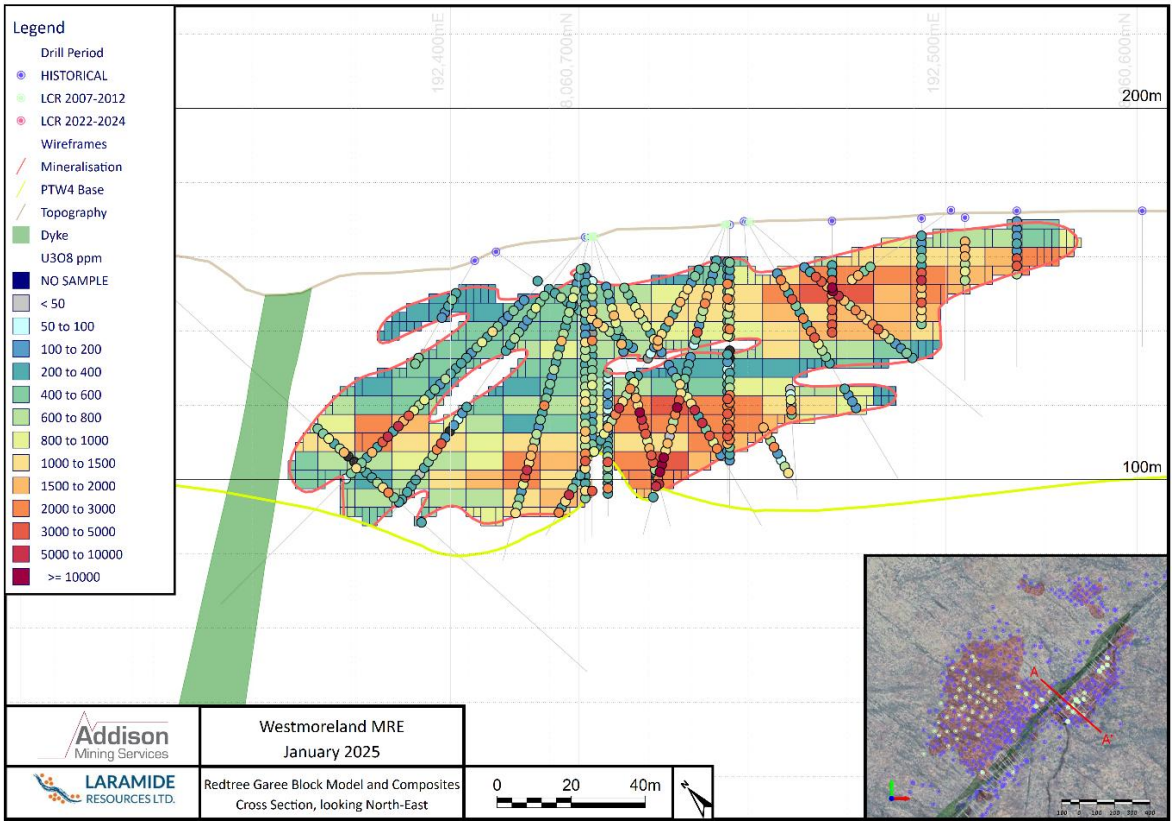
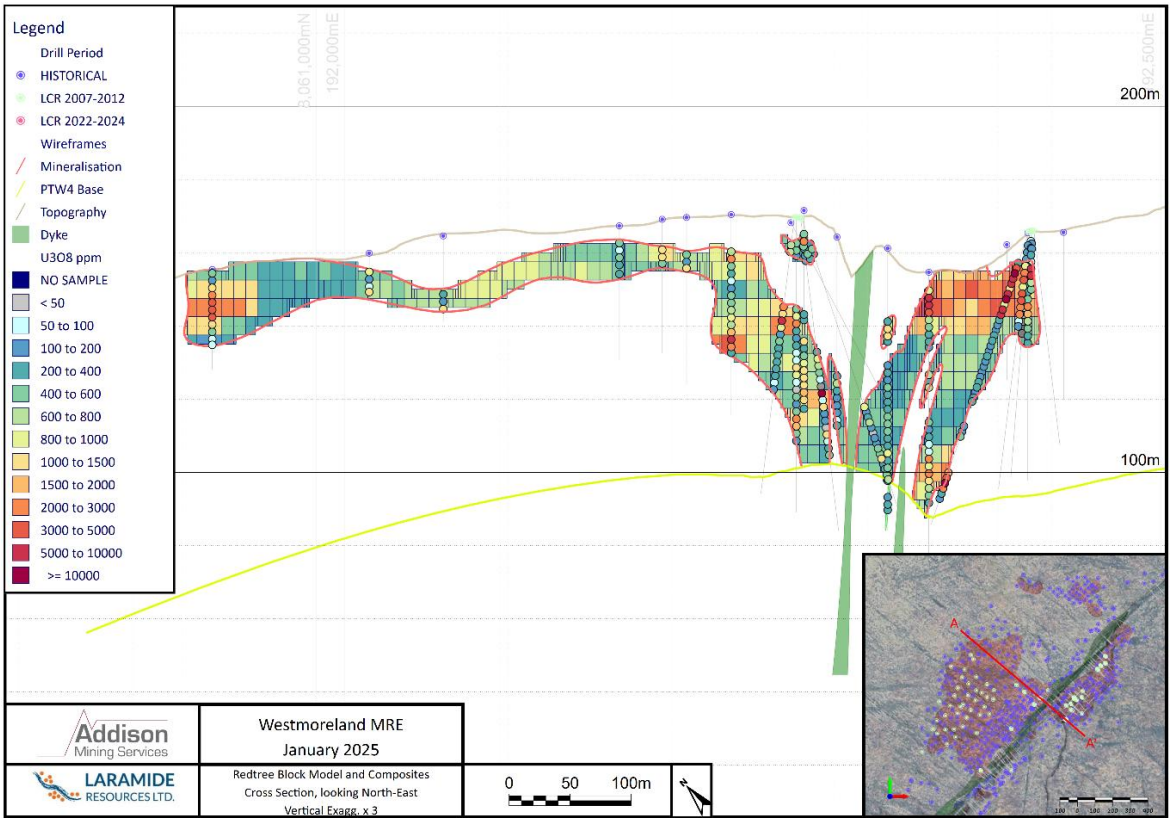


Figure 14.8.6 Redtree Jack-Garee Block Model Cross Section



Vertical exaggeration x3.

Figure 14.8.7 Huarabagoo Block Model Cross Section

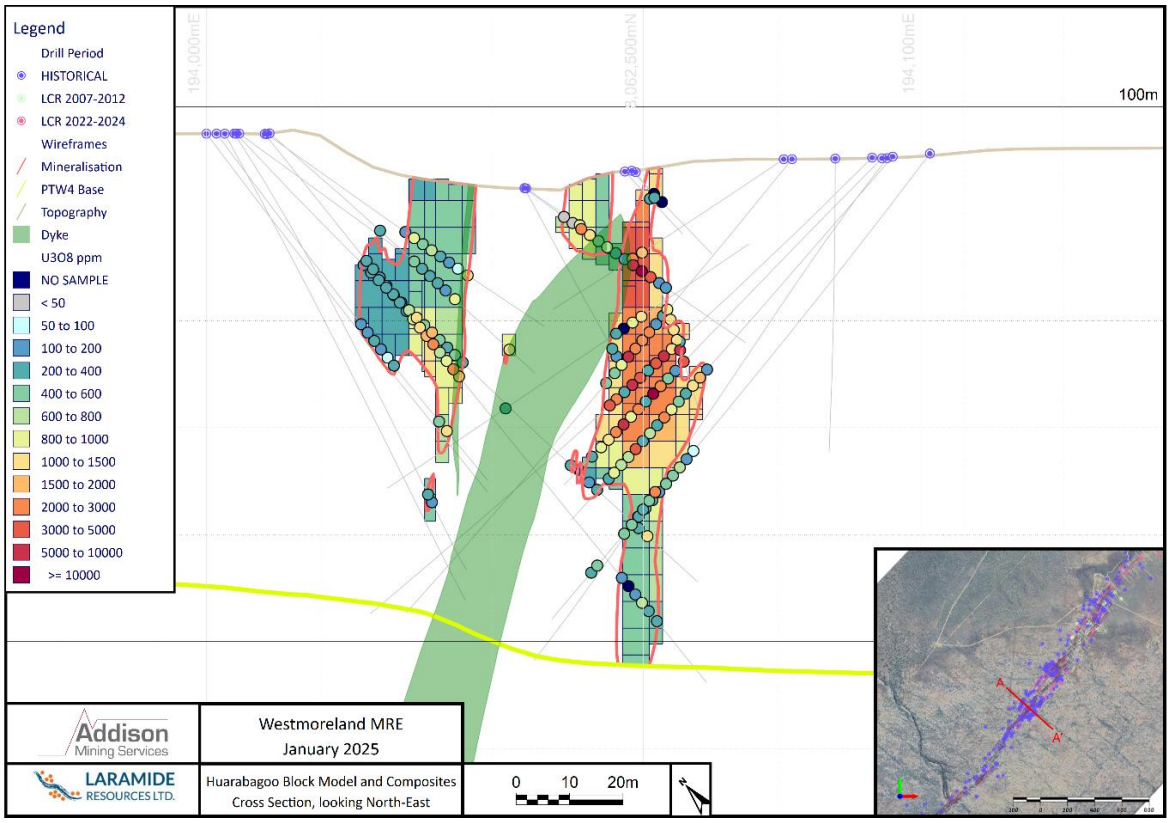
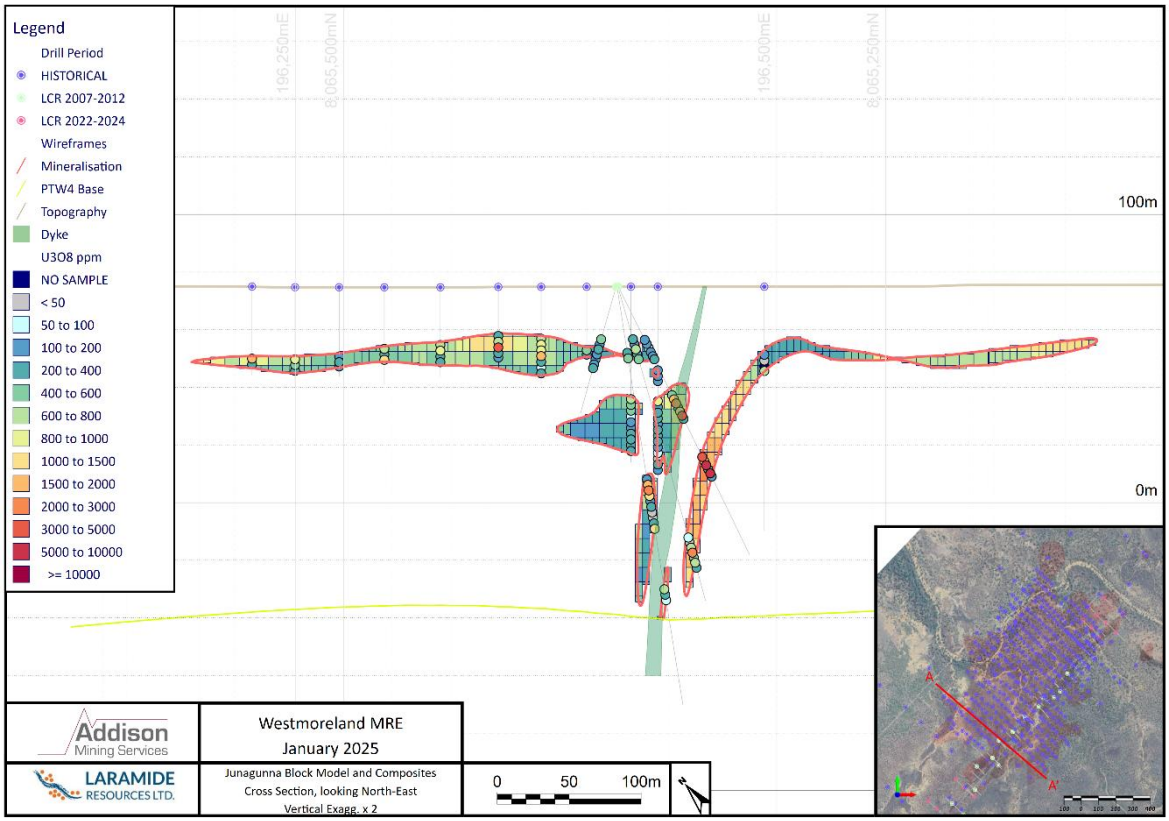
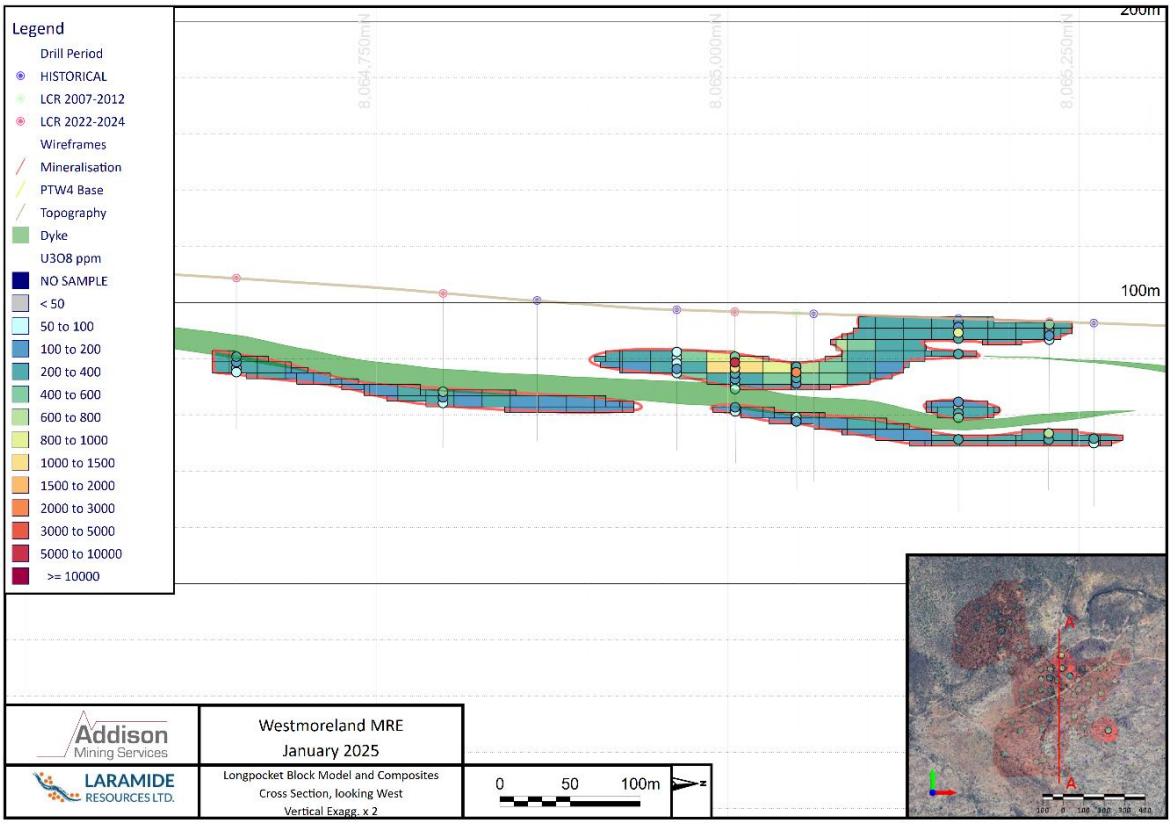


Figure 14.8.8 Junnagunna Block Model Cross Section



Vertical exaggeration x2.

Figure 14.8.9 Long Pocket Block Model Cross Section



Vertical exaggeration x2.

14.9 Bulk Density

There are 1,428 bulk density determinations in the Westmoreland database across different lithologies and zones of mineralization. The mean value is 2.57 g/cm^3. No relationship was found between density and grade. Analysis showed a mean value of 2.7 g/cm^3 was appropriate for material in the dyke while PTW4 sediment were given values of 2.5 g/cm^3 at RT and LP, 2.6 g/cm^3 at HB, and 2.54 g/cm^3 at JG.

14.10 Cut-off Grade and Reasonable Prospects of Eventual Economic Extraction

Reasonable prospects of eventual economic extraction contemplates mining by open pit mining methods with mineral processing by conventional leaching. Mining costs are estimated at approximately US$3 /t, mineral processing at US$30 /t, and general and administrative cost at US$5 /t processed. Considering a U_3O_8 price of US$80 /lb, a breakeven cut-off grade of 200 ppm is used for reporting and aligns with the cut-off grade used in previous estimates. Process recovery is estimated at 97.5%.

Pit optimization tests showed that all mineralized material above cut-off grade within the RT, JG and HB deposit block models has reasonable prospect of being extracted by open pit methods. At LP, an ultimate pit shell was used to constrain the estimate of reported Mineral Resources.

14.11 Resource Classification

The estimate was classified according to the QPs view of the estimation confidence. Indicated Resources are reported in areas where the spacing and quality of data are sufficient to allow estimation to a level of confidence which can be used for mine planning and economic evaluation. Those areas classified as Indicated Resources are typically informed by data with spacing 30 to 50 m, or in the case of the JG flat mineralization 30 m across strike and 60 to 70 m along strike. Indicated Resources are estimated into blocks approximately one third of the data spacing, and are typically informed in the first estimation pass using a minimum of three drillholes. Inferred Resources are classified in areas typically with spacing 50 to 100 m.

Measured Resources were thought not warranted due to the reliance on legacy data and the accuracy of the digital terrain model, which is material at RT where the topography is more variable. At RT, in the most densely drilled parts of the Garee Lens which LAM has also drilled extensively, Measured Resources may be warranted in the future with addition of improved topographic controls.

Figure 14.11.1 Redtree Resource Classification

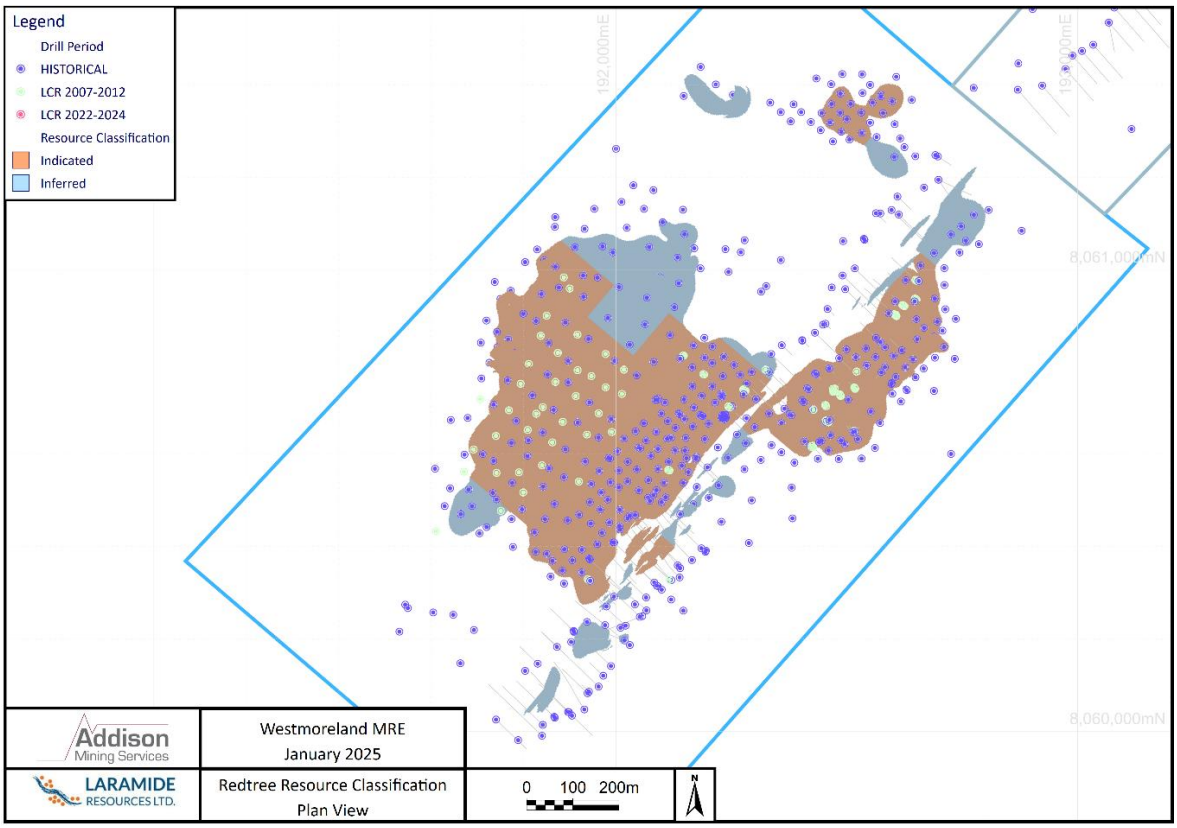


Figure 14.11.2 Huarabagoo Resource Classification

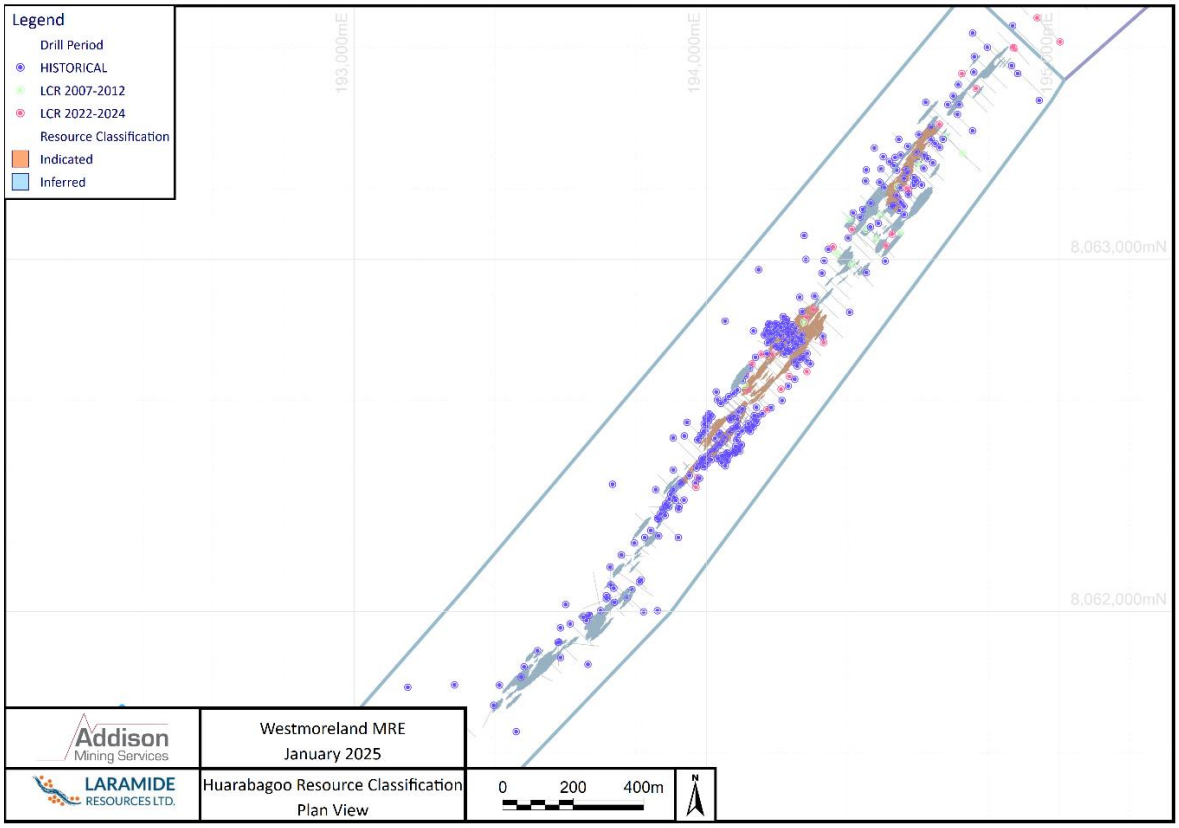


Figure 14.11.3 Junnagunna Resource Classification

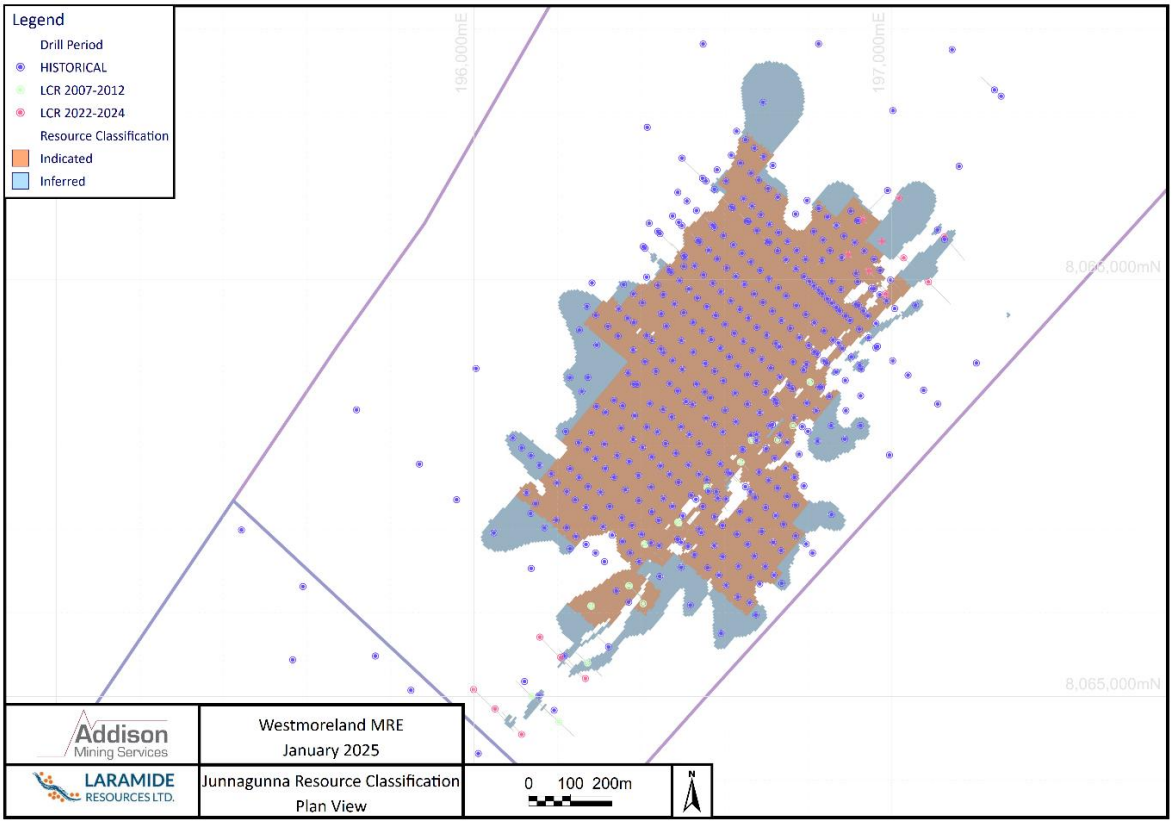
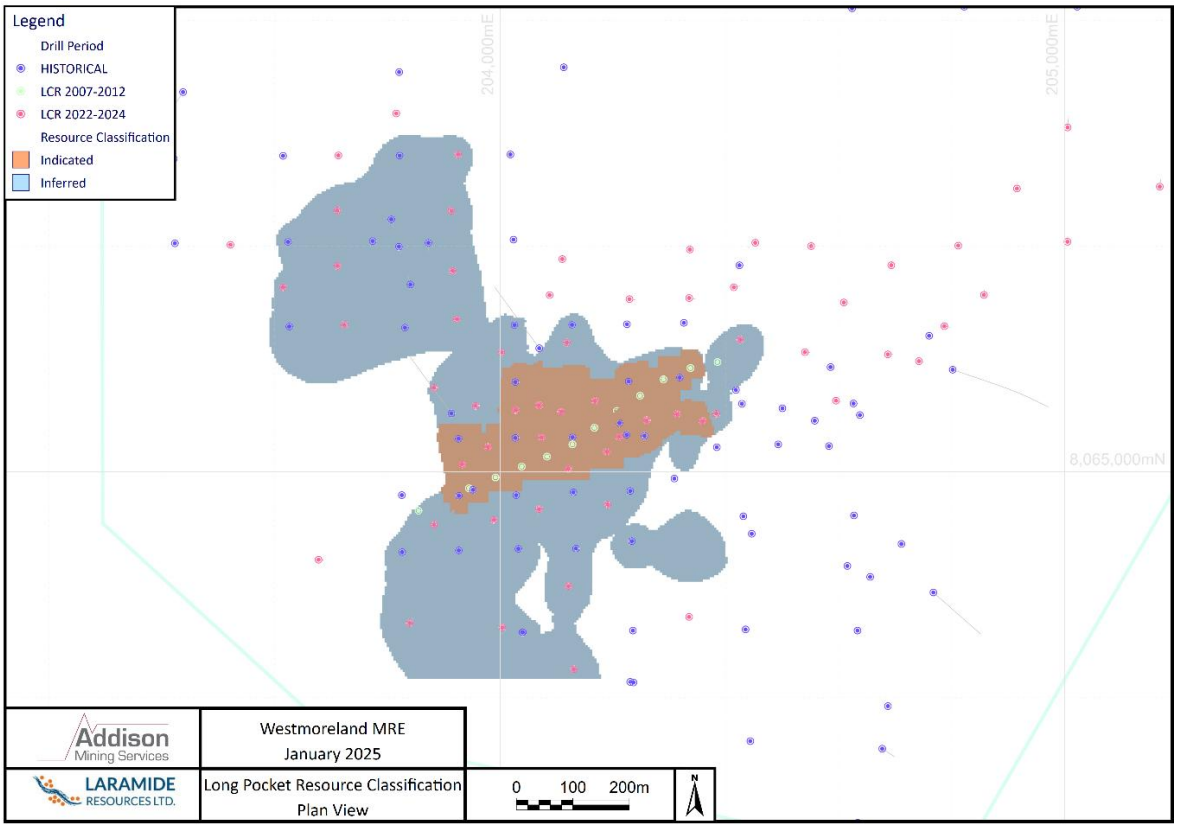


Figure 14.11.4 Long Pocket Resource Classification



14.12 Resource Estimate Results

The updated MRE has an effective date of 31 January 2025, and is reported above a cut-off grade of 200 ppm U_3O_8 and comprises of:

- Indicated Resources of 27.8 Mt at an average grade of 770 ppm U_3O_8 for 48.1 million contained pounds of U_3O_8.
- Inferred Resources of approximately 11.8 Mt at an average grade of 680 ppm U_3O_8 for 17.7 million contained pounds of U_3O_8.

The updated estimate supersedes all previous estimates. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. It is important to note that currently, only exploration and not mining for uranium is permitted in QLD, Australia. However, it is reasonable to expect that the policy may change in the future as there is a historical precedent for uranium mining within the State. An activity exclusion zone exists at the southern end of the HB deposit which will require further negotiation for future access and exploration activities, and effects 30% of the contained tonnage and metal of the HB Inferred Estimate. Table 14.12.1 sets out the Indicated and Inferred Mineral Resources by deposit.

Table 14.12.1 Mineral Resources by Deposit for the Westmoreland Uranium Project

Deposit	Tonnes	Density g/m³	U_3O_8 ppm	U_3O_8 Mlb
Indicated				
Redtree	14,000,000	2.5	880	27
Huarabagoo	2,500,000	2.6	890	4.9
Junnagunna	10,000,000	2.5	640	15
Long Pocket	1,300,000	2.5	420	1.2
Total Indicated	**27,800,000**	**2.5**	**770**	**48.1**
Inferred				
Redtree	3,000,000	2.5	800	5.2
Huarabagoo	3,100,000	2.6	870	6.0
Junnagunna	3,000,000	2.5	620	4.2
Long Pocket	2,700,000	2.5	380	2.3
Total Inferred	**11,800,000**	**2.5**	**680**	**17.7**

Reported above a cut-off grade of 200 ppm U_3O_8. Effective 31 January 2025.

14.13 Comparison to Previous Estimate

The previous MRE had an effective date of 11 May 2009 and is superseded by this estimate dated effective 31 January 2025. Differences in the estimate are shown in Table 14.13.1. Differences in the estimate are attributed to the following items.

- Additional drilling and receipt of assay results post May 2009 to January 2025 at JG and HB.
- Additional exploration at LP, which was not included in the previous estimate.
- Application of more sophisticated 3D modelling techniques, including implicit modelling which was not commonly employed in 2009, and is more adept at modelling complex geometry than traditional cross section interpretation.
- The RT deposit, which has not seen further exploration has no material change and this helps to support the veracity of the estimates.

Table 14.13.1 Comparison to Previous Estimate

2009 MRE				
Deposit	**Tonnes**	**Density g/m³**	**U_3O_8 ppm**	**U_3O_8 Mlb**
Indicated				
Redtree	12,900,000	2.5	900	25.5
Huarabagoo	1,460,000	2.5	830	2.7
Junnagunna	4,360,000	2.5	810	7.8
Long Pocket	-	-	-	-
Total Indicated	**18,700,000**	**2.5**	**880**	**36**
Inferred				
Redtree	4,460,000	2.5	670	6.6
Huarabagoo	2,400,000	2.5	1,090	5.8
Junnagunna	2,150,000	2.5	750	3.6
Long Pocket	-	-	-	-
Total Inferred	**9,000,000**	**2.5**	**800**	**15.9**
Relative Difference %				
Deposit	**Tonnes**		**Grade**	**Contained Metal**
Indicated				
Redtree	9%		-2%	6%
Huarabagoo	71%		7%	81%
Junnagunna	129%		-21%	92%
Long Pocket	-		-	-
Total Indicated	**49%**		**-13%**	**34%**
Inferred				
Redtree	-33%		19%	-21%
Huarabagoo	29%		-20%	3%
Junnagunna	40%		-17%	17%
Long Pocket	-		-	-
Total Inferred	**31%**		**-15%**	**11%**
Absolute Difference %				
Deposit	**Tonnes**		**Grade ppm**	**Contained Metal Mlb**
Indicated				
Redtree	1,100,000	-	-20	1.5
Huarabagoo	1,040,000	0.1	60	2.2
Junnagunna	5,640,000	-	-170	7.2
Long Pocket	1,300,000	2.5	420	1.2
Total Indicated	**9,100,000**	**-**	**-110**	**12.1**
Inferred				
Redtree	-1,460,000	-	130	-1.4
Huarabagoo	700,000	0.1	-220	0.2
Junnagunna	850,000	-	-130	0.6
Long Pocket	2,700,000	2.5	380	2.3
Total Inferred	**2,800,000**	**-**	**-120**	**1.8**

15.0 MINERAL RESERVE ESTIMATES

This section is not applicable.

16.0 MINING

16.1 Mining Model and Modifying Factors

The mineral resources used in the study are summarized in Table 16.1.1.

Table 16.1.1 January 2025 Mineral Resources

Deposit	Tonnes t	Density g/m³	U_3O_8 ppm	U_3O_8 Mlb
Indicated				
Redtree	14,000,000	2.5	880	27
Huarabagoo	2,500,000	2.6	890	4.9
Junnagunna	10,000,000	2.5	640	15
Long Pocket	1,300,000	2.5	420	1.2
Total Indicated	**27,800,000**	**2.5**	**770**	**48.1**
Inferred				
Redtree	3,000,000	2.5	800	5.2
Huarabagoo	3,100,000	2.6	870	6.0
Junnagunna	3,000,000	2.5	620	4.2
Long Pocket	2,700,000	2.5	380	2.3
Total Inferred	**11,800,000**	**2.5**	**680**	**17.7**

Mineral Resources as of 31 January 2025, above a CoG of 200 ppm.

The mineral resources for the JG, HB, RT and LP deposits that comprise the Project were estimated using separate block models. To convert the resource models to mining models, each resource model was regularized to a selective mining unit (SMU) and then a global production target loss was applied to the block model.

- The JG deposit was regularized to a 2.5 x 5 x 2.5 m model and no global production target loss was applied.
- The HB deposit was regularized to a 2.5 x 5 x 2.5 m model and 3% global production target loss was applied.
- The RT deposit was regularized to a 2.5 x 5 x 2.5 m model and 3% global production target loss was applied.
- The LP deposit was regularized to a 4 x 8 x 2 m model and 3% global production target loss was applied.

The regularization process over the four deposits resulted in a 4.1% reduction in Resource tonnes, a 2.3% reduction in grade, and an overall uranium metal loss of 6.3% from the Resource Model to the Mining Model. A global 6.3% overall metal loss was considered to be adequate for this PEA and no further loss and dilution factors were applied to the Mining Model.

The Deswik Pseudoflow Optimizer was used to define optimal pit shells for the four Westmoreland uranium deposits—JG, HB, RT and LP—based on the Mining Block Model generated. The costs coded into the block model for the optimization are shown below in Table 16.1.3.

The cut-off grade used in the optimization was a marginal cut-off grade. If the revenue received from a processed SMU block was greater than the processing cost and general & administration (G&A), then that SMU was processed through the processing plant. The cut-off grade used for the optimization was 228 ppm.

Table 16.1.2 Cut-Off Grade Estimate

CoG Calculation		
Uranium Price US$ /lb	$75.00	
A$:US$	0.7	
U_3O_8 Price	$0.24	A$ /ppm
Royalty State 5%	$0.01	A$ /ppm
Net U_3O_8 Price	$0.22	A$ /ppm
Processing Cost and G&A	$48.00	A$ /t Production Target
Recovery	95.0%	
Marginal Cut-Off Grade	**228**	ppm

Table 16.1.3 Optimization Cost Centres

Optimization Cost Centres	Units	Rates
Uranium Price Assumption	US$ /lb	$75.00
Exchange Rate	A$:US$	$0.70
Private Royalty Assumption	1% NSR	Up to A$10M
QLD Gov. Royalty Assumption	% Ad Valorem	5%
Mining Load and Haul	A$ /t TMM	$4.46
Mining Drill and Blast Oxide	A$ /t Oxide	$0.35
Mining Drill and Blast Fresh	A$ /t Fresh	$0.86
Mining Grade Control	A$ /t Production Target	$0.66
Diesel Cost	A% /L	$1.35
Processing Cost and Site G&A	A$ /t Production Target	$48.00
Uranium Processing Recovery	%	95%

From each of the four optimizations, an optimal shell was selected to proceed to the pit design process. As an example, the optimization result from the RT pit is shown in Figure 16.1.1 and Table 16.1.4.

Figure 16.1.1 RT Pit Optimization Result

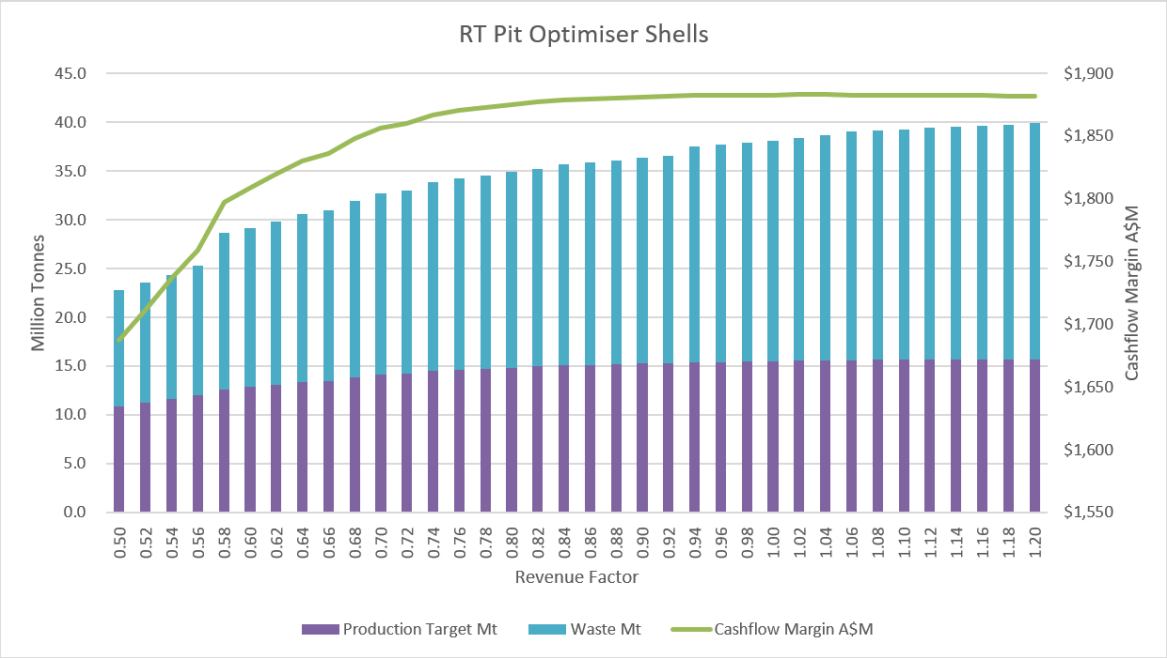


Table 16.1.4 RT Pit Optimization Result

Revenue Factor %	Revenue A$...M	Opex A$...M	Cashflow Margin A$...M	Production Target Mt	Waste Mt	Strip Ratio t:t
0.50	$2,480	$793	$1,687	10.8	11.9	1.10
0.52	$2,529	$818	$1,711	11.2	12.3	1.10
0.54	$2,583	$846	$1,737	11.6	12.7	1.09
0.56	$2,632	$873	$1,759	12.0	13.3	1.10
0.58	$2,720	$923	$1,797	12.6	16.0	1.27
0.60	$2,748	$939	$1,809	12.9	16.3	1.27
0.62	$2,776	$956	$1,820	13.1	16.8	1.28
0.64	$2,802	$973	$1,830	13.3	17.3	1.29
0.66	$2,819	$983	$1,836	13.5	17.4	1.29
0.68	$2,856	$1,008	$1,848	13.8	18.0	1.30
0.70	$2,881	$1,025	$1,856	14.1	18.6	1.32
0.72	$2,894	$1,034	$1,860	14.2	18.8	1.32
0.74	$2,920	$1,053	$1,867	14.5	19.3	1.33
0.76	$2,932	$1,062	$1,870	14.6	19.6	1.34
0.78	$2,943	$1,070	$1,873	14.7	19.8	1.34
0.80	$2,953	$1,078	$1,875	14.9	20.0	1.35
0.82	$2,962	$1,085	$1,877	15.0	20.3	1.35
0.84	$2,971	$1,093	$1,879	15.1	20.6	1.37
0.86	$2,978	$1,099	$1,880	15.1	20.8	1.37
0.88	$2,984	$1,103	$1,881	15.2	20.9	1.38
0.90	$2,988	$1,107	$1,881	15.3	21.1	1.38
0.92	$2,992	$1,110	$1,882	15.3	21.2	1.39
0.94	$3,003	$1,121	$1,882	15.4	22.2	1.44
0.96	$3,006	$1,123	$1,883	15.4	22.3	1.44
0.98	$3,009	$1,126	$1,883	15.5	22.4	1.45
1.00	$3,013	$1,130	$1,883	15.5	22.6	1.46
1.02	$3,016	$1,133	$1,883	15.5	22.8	1.47
1.04	$3,019	$1,136	$1,883	15.6	23.1	1.48
1.06	$3,023	$1,140	$1,883	15.6	23.4	1.50
1.08	$3,025	$1,142	$1,883	15.6	23.5	1.50
1.10	$3,026	$1,144	$1,883	15.7	23.6	1.51
1.12	$3,028	$1,145	$1,883	15.7	23.7	1.52
1.14	$3,029	$1,147	$1,882	15.7	23.9	1.52
1.16	$3,030	$1,147	$1,882	15.7	23.9	1.52
1.18	$3,031	$1,149	$1,882	15.7	24.0	1.53
1.20	$3,032	$1,150	$1,882	15.7	24.2	1.54

16.2 Pit Designs

The optimal shells selected to proceed to design were: RT pit the 86% revenue factor (RF) shell; JG pit the 86% RF shell; HB pit the 78% RF shell; and LP pit the 96% RF shell.

To date, there have been no geotechnical studies undertaken at Westmoreland. High level geotechnical assumptions were made for both the optimization overall slope angles (OSA) and pit designs, as shown in Table 16.2.1 below. Geotechnical assessments will need to be made in any future studies.

Table 16.2.1 Geotechnical Assumptions

Geotechnical Assumptions	Units	Parameter
Optimization OSAs		
OSA Oxide	degrees	36
OSA Fresh	degrees	50
Pit Design Parameters		
Oxide Batter	degrees	45
Fresh Batter	degrees	70
Berm Width	m	4
Bench Heights	m	20
777 Dual Ramp Width	m	28
777 Single Ramp Width	m	19

The pits were designed using the pseudoflow optimizer shells as discussed above. A comparison between the optimizer physicals for the four deposits and the subsequent pit design physicals is shown in Table 16.2.2.

Table 16.2.2 Optimization Versus Pit Design Physicals

Optimizer vs. Design Physicals	Units	Optimizer Physicals	Pit Design Variance	Variance %
Waste	Mt	108.10	118.32	9.5%
Production Target	Mt	23.15	28.24	-12.2%
Uranium Grade	%	805	810	0.7%
Total Material Movement	Mt	140.25	146.56	4.5%
Strip Ratio	t:t	3.36	4.19	24.6%

There are four pit designs in four distinct areas within the mining lease as shown in Figure 16.2.1 to Figure 16.2.5:

- North – JG.
- Central – HB.
- South – RT.
- Northeast - LP.

Figure 16.2.1 Four Pits – RT, HB, JG, LP

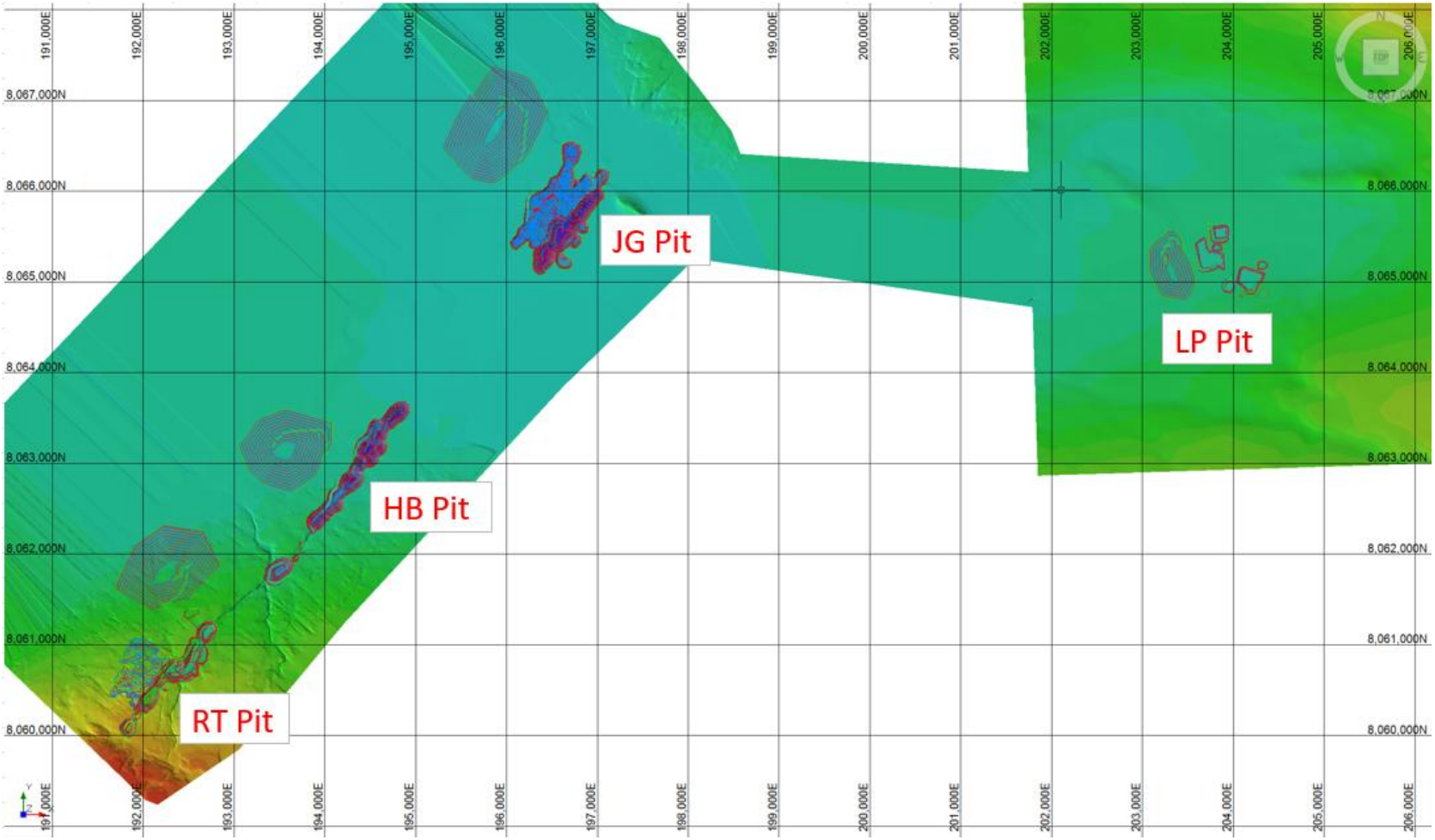


Figure 16.2.2 North Pit- Junnagunna (JG)

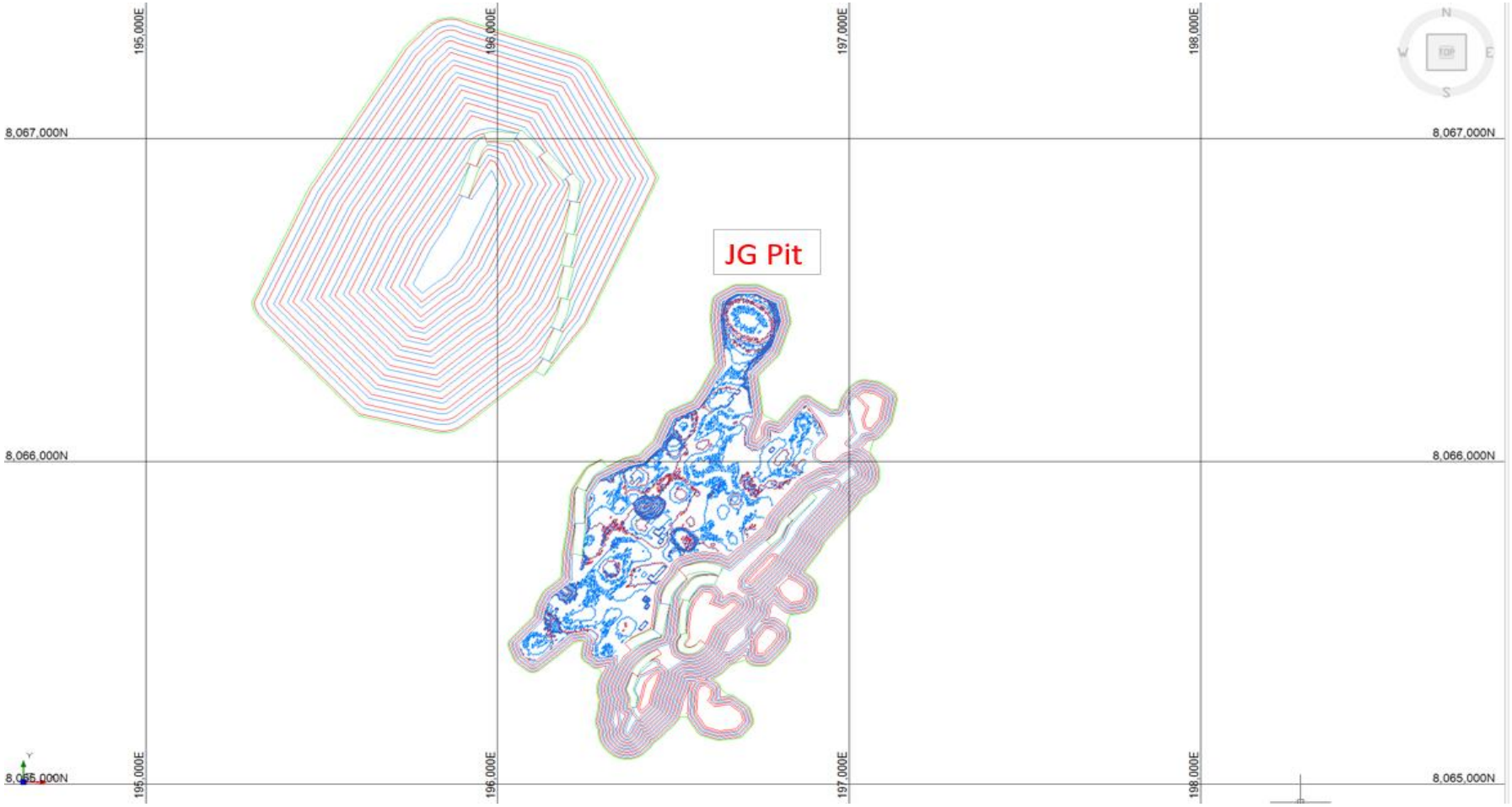


Figure 16.2.3 Central Pit – Huarabagoo (HB)

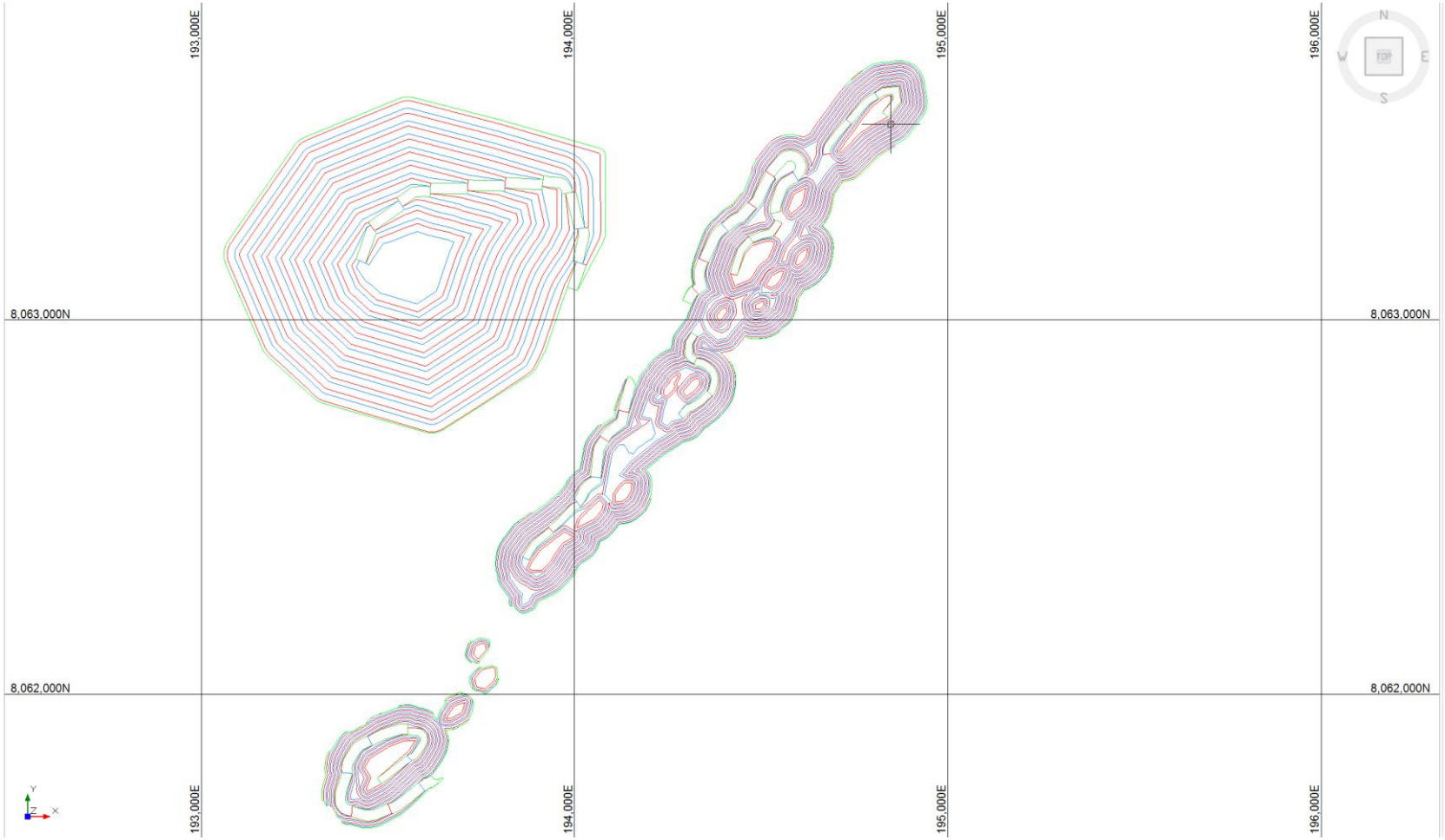


Figure 16.2.4 South Pit – Redtree / Garee (RT)

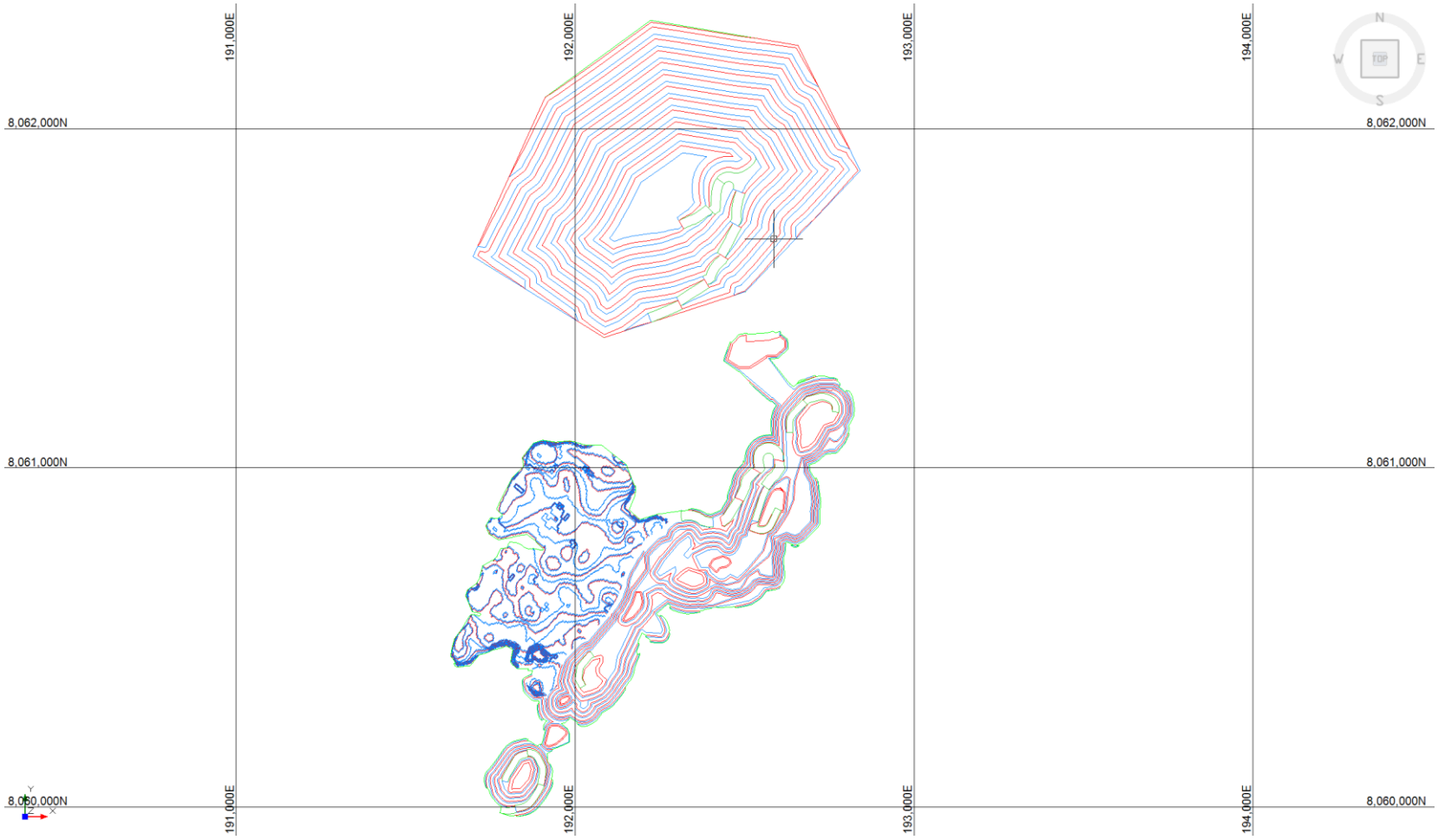


Figure 16.2.5 Northeast Pit – Long Pocket (LP)



Figure 16.2.6 Westmoreland Deposits – South Pit in Foreground



Source: Site visit 2015.

16.3 Mining Methodology

The mining methodology is based on conventional methods and is summarized below:

- Open-pit mining using three excavators loading off-highway haul trucks - two EX2000 excavators and one EX1200 excavator, all loading CAT777 trucks. The average production rate for the EX2000's was 1073 tonnes per hour (tph) and the production rate for the EX1200 was 558 tph. Four trucks were allocated to each excavator over the LOMP. The excavators were supported by an appropriate ancillary fleet.
- Conventional D&B with T&S operation mining 5 m benches with 2.5 m flitches.
- Sufficient working areas to allow for simultaneous D&B and T&S operation. Flexibility in the scheduling is required.

The mining methodology is based on the following material movement schedule:

- A total of 147 Mt TMM will be moved over 11 years of mining with 118 Mt of waste and 28.3 Mt of production target being produced grading 810 ppm. The LOM strip ratio of the four pits is 4.2 t:t. The strip ratio varies substantially between the pits with RT pit at 1.7 t:t, LP pit at 4.5 t:t, JG pit at 5.3 t:t, and HB pit at 10 t:t.
- Mill feed: 2.9 Mtpa achieved in the second year onwards for the full mine life. The mill throughput reduces to 1 Mtpa in the final year of production.
- Mining commences in RT pit to establish an initial tailings emplacement area.

16.4 Production Schedule

The production schedule is planned at 2.9 Mtpa of mill feed. Mining is undertaken for 11 years with a total production target of 28.3 Mt at an average grade of 810 ppm (Table 16.4.1).

Table 16.4.1 LOMP Physicals

LOMP Physicals	Units	LOMP
Waste	Mt	118.32
Production Target	Mt	28.24
Uranium Grade	ppm	810
Total Material Movement	Mt	146.6
Strip Ratio	t:t	4.2

The total annual production and annual production by pit are presented in Table 16.4.2. Figures 16.4.1 to Figure 16.4.10 graphically shows the schedule in detail. Over the LOMP, 79% of the production target comes from the Indicated Resource category and 21% from the Inferred Resource category. There is a low level of geological confidence associated with Inferred Resources and there is no certainty that further exploration work will result in the determination of Indicated Resources, or that the production target itself will be realized.

Figure 16.4.1 Total Material Mined

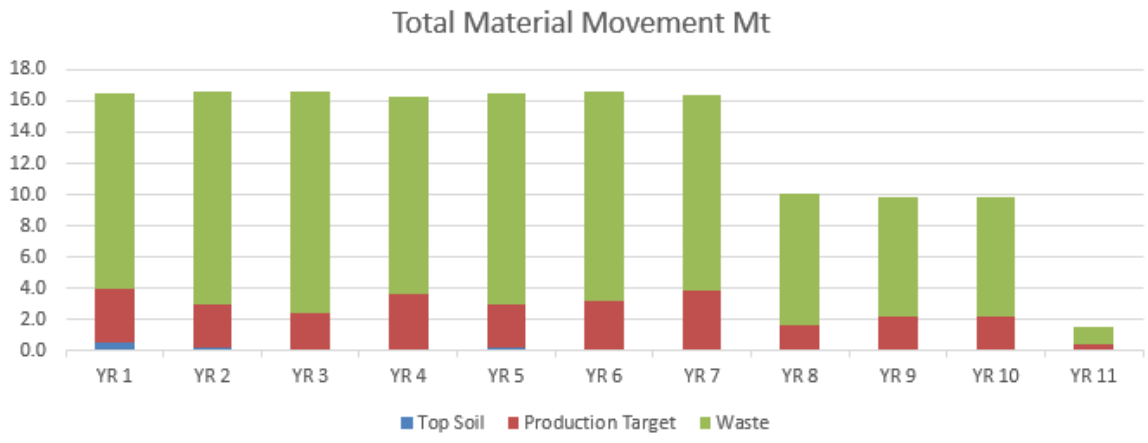


Table 16.4.2 LOMP Schedule

Physicals	Units	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
RT Pit													
Waste	Mt	22.8	6.2	3.3	5.5	5.8	2.0	0.0	0.0	00	0.0	0.0	0.0
Production Target	Mt	13.6	3.4	2.7	2.1	3.6	1.8	0.0	0.0	0.0	0.0	0.0	0.0
Uranium Grade	ppm	882	976	909	1,033	757	731	0	0	0	0	0	0
Total Material Movement	Mt	36.4	9.6	6.0	7.7	9.4	3.8	0.0	0.0	0.0	0.0	0.0	0.0
Strip Ratio	t:t	1.7	1.8	1.3	2.6	1.6	1.1	0.0	0.0	0.0	0.0	0.0	0.0
HB Pit													
Waste	Mt	40.1	0.0	3.6	2.0	0.0	5.7	9.9	9.4	8.4	1.1	0.0	0.0
Production Target	Mt	4.0	0.0	0.1	0.1	0.0	0.3	0.5	0.9	1.6	0.5	0.0	0.0
Uranium Grade	ppm	979	0	931	1,077	0	924	959	985	991	976	0	0
Total Material Movement	Mt	44.1	0.0	3.7	2.1	0.0	6.0	10.4	10.3	10.0	1.5	0.0	0.0
Strip Ratio	t:t	10.0	0.0	27.4	21.6	0.0	19.4	18.9	10.7	5.1	2.4	0.0	0.0
JG Pit													
Waste	Mt	51.4	6.9	6.9	6.8	6.8	5.9	3.5	3.2	0.0	4.0	6.4	1.1
Production Target	Mt	9.7	0.0	0.0	0.0	0.0	0.8	2.7	2.9	0.0	1.2	1.7	0.4
Uranium Grade	ppm	678	0	0	445	435	617	627	610	0	650	865	947
Total Material Movement	Mt	61.2	6.9	6.9	6.8	6.8	6.6	6.2	6.1	0.0	5.3	8.1	1.5
Strip Ratio	t:t	5.3	0.0	0.0	1,520.5	247.6	7.9	1.3	1.1	0.0	3.2	3.9	2.6
LP Pit													
Waste	Mt	4.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	2.7	1.3	0.0
Production Target	Mt	0.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.4	0.5	0.0
Uranium Grade	ppm	410	0	0	0	0	0	0	0	0	365	438	0
Total Material Movement	Mt	4.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	3.1	1.8	0.0
Strip Ratio	t:t	4.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	7.7	2.3	0.0

Physicals	Units	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
Total Material Movement													
Waste	Mt	118.3	13.0	13.8	14.4	12.6	13.6	13.4	12.5	8.4	7.8	7.7	1.1
Production Target	Mt	28.2	3.4	2.8	2.2	3.6	2.8	3.2	3.8	1.6	2.0	2.2	0.4
Uranium Grade	ppm	810	976	910	1,033	755	721	681	697	991	673	760	947
Total Material Movement	Mt	146.6	16.4	16.6	16.7	16.2	16.4	16.6	16.3	10.0	9.9	9.9	1.5
Strip Ratio	t:t	4.2	3.8	4.9	6.4	3.5	4.8	4.1	3.3	5.1	3.8	3.5	2.6
Measured	Mt	0.00	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Indicated	Mt	22.2	3.1	2.1	1.7	3.2	2.3	2.7	2.9	0.9	1.4	1.6	0.3
Inferred	Mt	6.00	0.3	0.7	0.5	0.4	0.5	0.5	.9	0.8	0.6	0.6	0.1
Processing Schedule													
Processed Production Target	Mt	28.2	1.1	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	1.0
Uranium Grade	Ppm	810	976	913	1,020	755	725	681	697	921	719	777	878
Recovered U lb	Mlb	47.9	2.2	5.5	6.2	4.6	4.4	4.1	4.2	5.6	4.4	4.7	1.9

Figure 16.4.2 Production Target by Pit

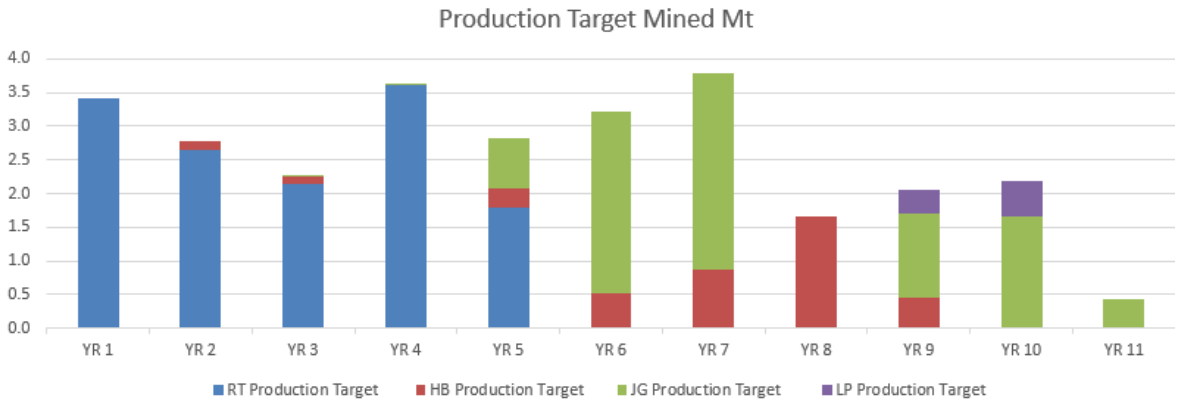


Figure 16.4.3 TMM by Pit

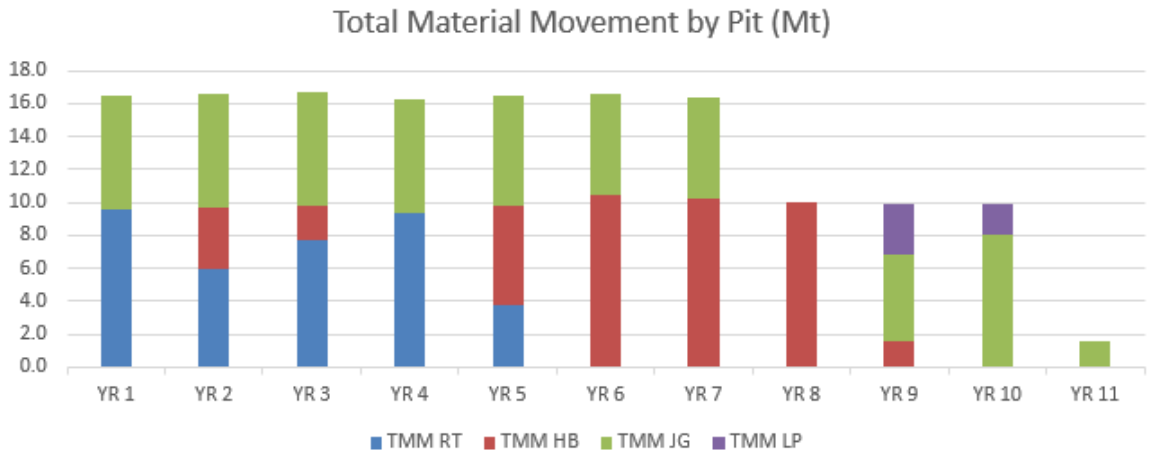


Figure 16.4.4 Production Target Mined

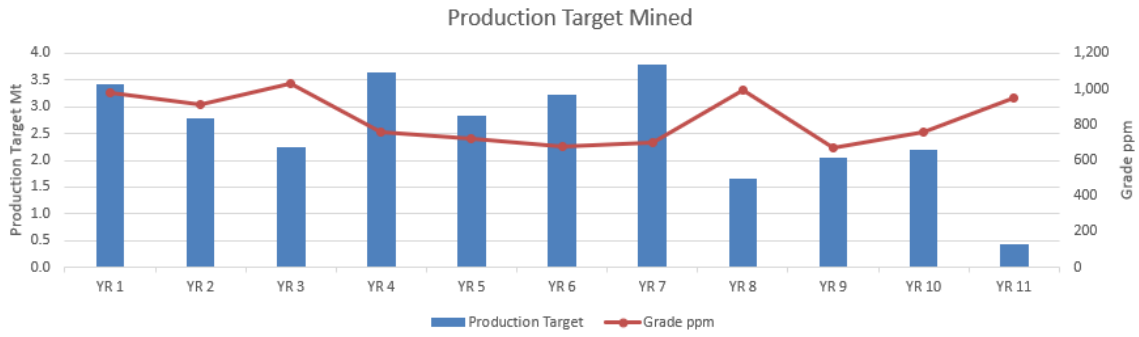


Figure 16.4.5 Production Target Milled

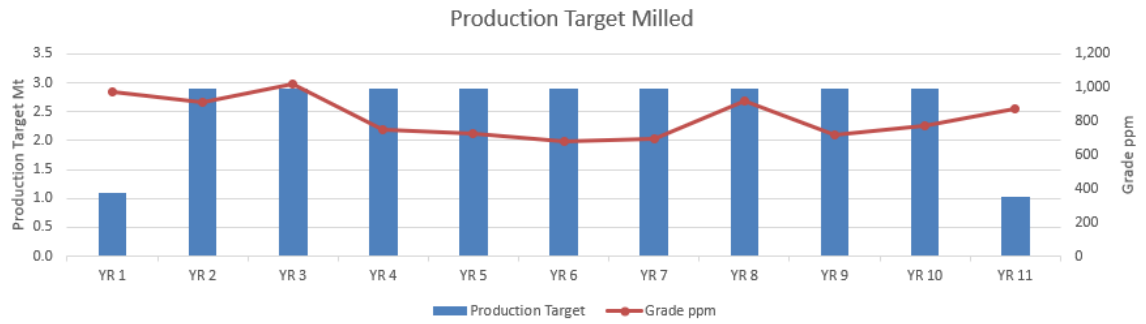


Figure 16.4.6 Closing ROM Stockpile

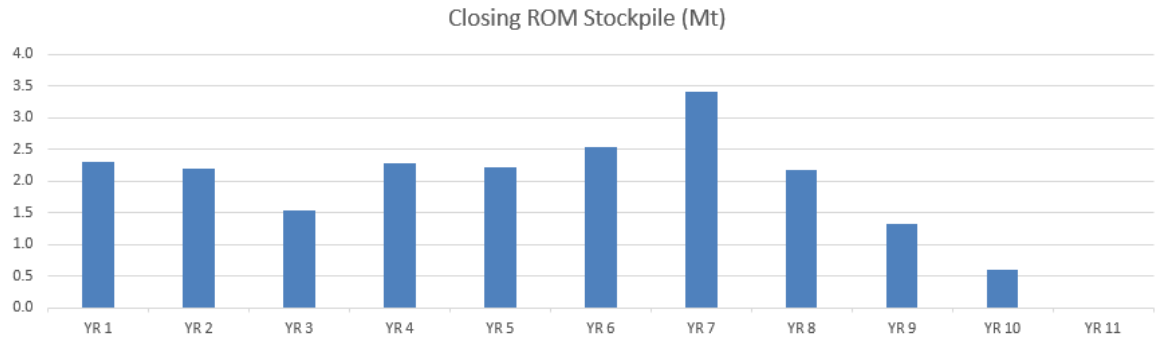


Figure 16.4.7 Metal Produced

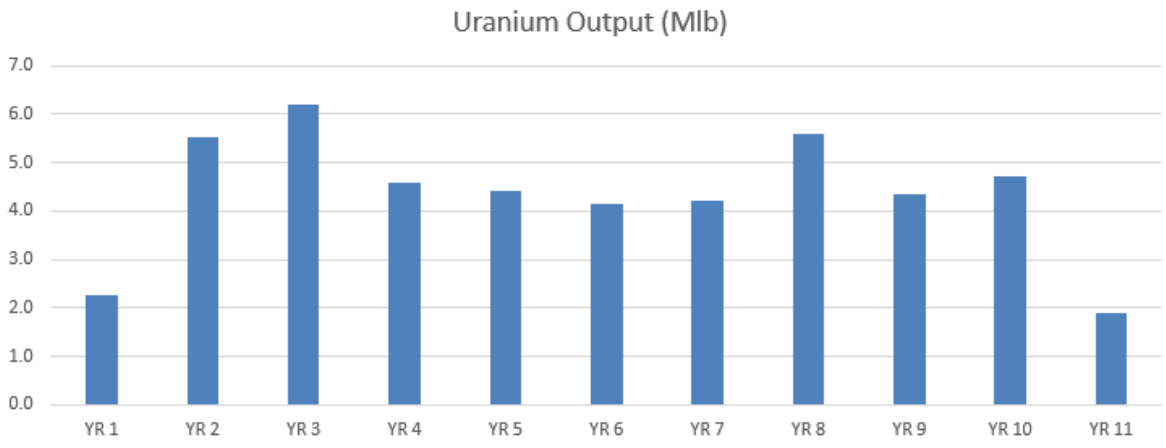


Figure 16.4.8 Resource Category (Mt)

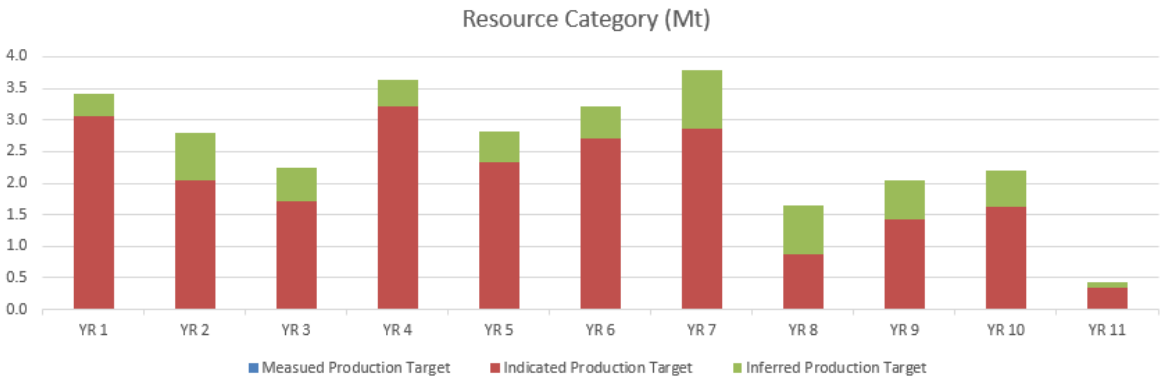


Figure 16.4.9 Resource Category (%)

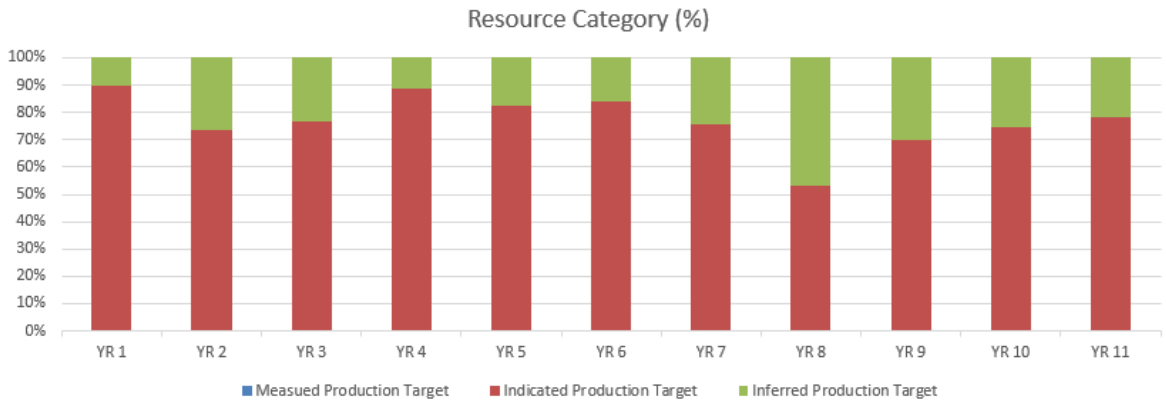


Figure 16.4.10 Excavator TMM (Mt)



17.0 RECOVERY METHODS

17.1 Process Design

The process flowsheet selected as the study basis for the treatment of Westmoreland ore has been chosen after evaluation of the available geology, mineralogy and metallurgical testwork results (see Section 13). The objective has been to select a robust treatment method that provides a sound basis for estimating capital and operating costs at the SS level of accuracy, but also that takes advantage of recent trends in uranium milling technology.

17.1.1 Selected Process Flowsheet

The treatment plant design incorporates the following unit process operations:

- ROM pad blending of different ore types for optimum leach recovery.
- Single stage primary crushing in a jaw crusher to produce a crushed product size of P_{80} 130 mm.
- Crushed ore bin (COB) with a nominal 1,900 t live capacity to provide 5 h of operation at design plant throughput.
- Crushed ore from the COB is reclaimed by an apron feeder positioned under the bin to feed the grinding circuit.
- The grinding circuit is a SS SAG/C type, which consists of a closed circuit SS SAG mill and pebble crusher. The semi-autogenous grinding (SAG) mill is in closed circuit with hydro-cyclones to produce a grind size of P_{80} 180 µm.
- A pre-leach thickener is included to increase slurry density to the leach circuit, minimize leach tank volume requirements and to partly neutralize filter wash, which is returned to the milling circuit.
- Leach residue filtration to recover leached uranium in pregnant liquor and produce a washed dry filter cake containing acid gangue for final disposal.
- Continuous ion exchange for the separation of soluble uranium from the leach pregnant solution and the production of barren liquor for recycle to the leach process.
- Impurity removal to separate predominantly iron and some manganese, silicon and aluminium by acid neutralization with caustic soda prior to UOC production.

- Crude UOC production, washing and drying. Crude UOC is precipitated from the cleaned liquor by treating with hydrogen peroxide and caustic soda. The precipitated UOC is separated from the mother liquor, washed, dried, calcined and stored in preparation for packaging.

- The dry UOC product is weighed and loaded in to 200 L drums, which are then packed into 20 t containers in preparation for shipment.

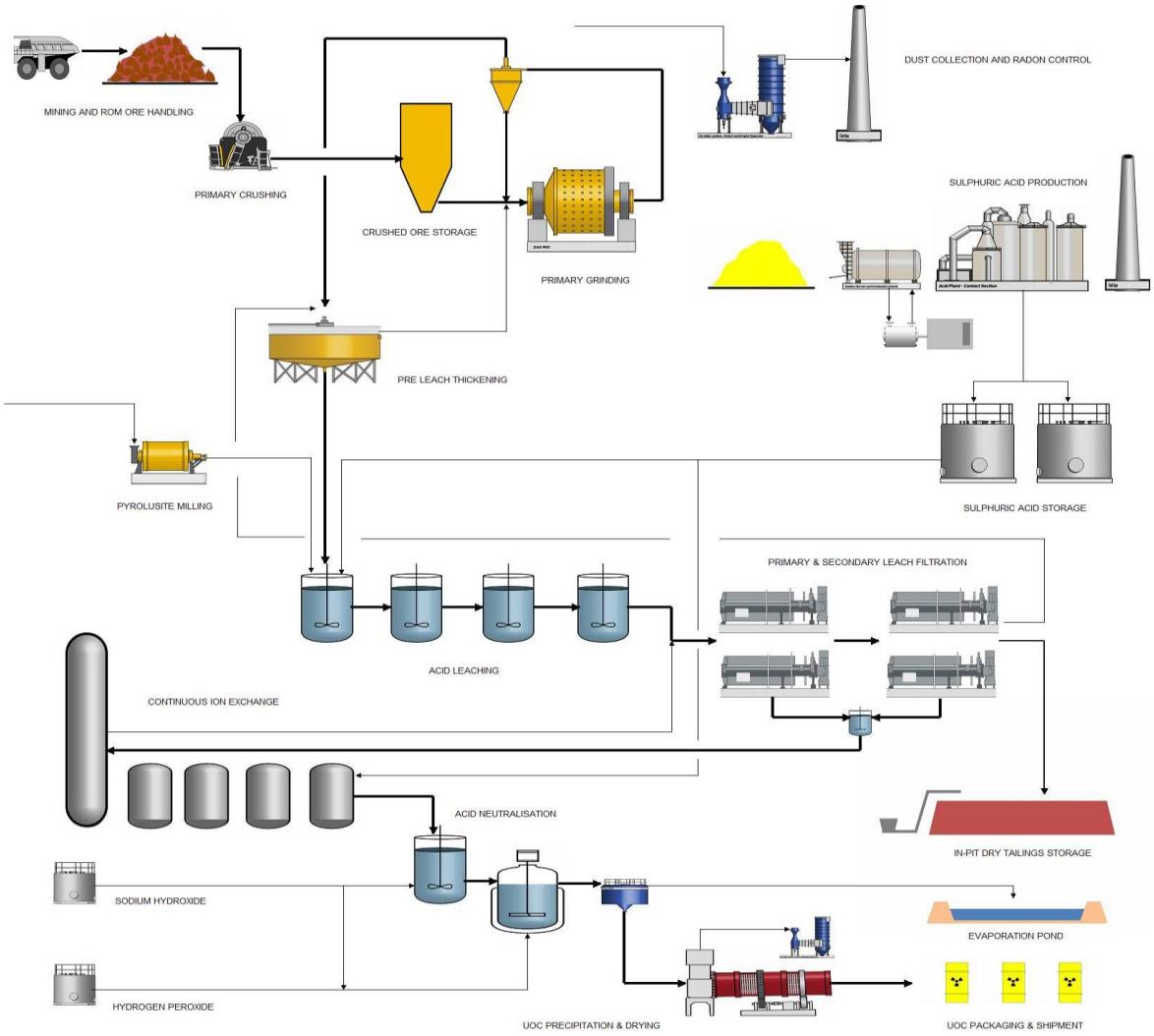
A schematic overall process flow diagram depicting the unit operations incorporated in the selected process flowsheet is presented in Figure 17.1.1. The key issues considered in process and equipment selection are outlined in the next section. The key process design criteria listed in Table 17.1.1 form the basis of the detailed process design criteria and mechanical equipment list.

Table 17.1.1 Summary of Key Process Design Criteria

	Units	Design	Source*
Plant Throughput	tpa	2,900,000	Client
Head Grade	U ppm	788	Client
U3O8 Production	lb/y	4,786,098	Calculation
	kg/h	271	Calculation
Overall Recovery	%	95	Client
Crushing Plant Availability	%	80.0	Client
Plant Availability	%	91.3	Client
Bond Ball Mill Work Index (BWi)	kWh/t	19.4	Testwork
Bond Rod Mill Work Index (RWi)	kWh/t	21.0	Assumed
Bond Abrasion Index (Ai)		0.2	Assumed
Grind Size	µm	180	Client
Mill Discharge pH		4.0	Lycopodium
Pre-Leach Thickener Solids Loading	$t/m^2.h$	0.38	Testwork
Leach Residence Time	h	12	Testwork
Leach Temperature	°C	40	Testwork
Leach pH	-	1.5	Testwork
Leach [Fe]	g/L	>2.5	Testwork
Leach ORP	mV	500	Testwork
Acid Addition	kg/t	24	Testwork
Leach extraction	%	96	Testwork
Leach Filter Flux	kg d.s./m^2.h	315	Assumed
Pyrolusite Grind Size P_{80}	um	45 - 54	Testwork
Pyrolusite Available MnO_2	%	75	Assumed
Pyrolusite Addition Rate	kg/t	6.4	Testwork
Ion Exchange Adsorption	-	Mod NIMCIX	Assumed
Ion Exchange Resin	-	Amberjet 4400	Testwork

	Units	Design	Source*
Exchange Form	-	SO4 - HCO3 form	Assumed
Total Exchange Capacity	eq/L	> 1.4	Assumed
Design U load	g/L(wsr)	40	Assumed
Elutriation Volume	m^3/m^3 resin	7.5	Assumed
Acid Neutralization pH	-	2 – 3	Assumed
Acid Neutralization Temperature	˚C	40	Assumed
Refinery Reagent Addition Rate			
• 100% NaOH to UOC	kg/lb U_3O_8	4.94	Calculated
• 50% H_2O_2 to UOC	kg/lb U_3O_8	0.21	Calculated
Acid Neut. Thickener Area Flux	$t/m^2.h$	16	Assumed
Acid Neut. Residue Filter Flux	kg d.s./m^2.h	75	Assumed

Figure 17.1.1 Schematic Overall Process Flow Diagram



17.2 Process and Plant Description

The overall process flowsheet includes feeding a blended ore mix to a single stage jaw crusher and SAG mill with pebble crusher (SS SAG/C) comminution circuit, which is in closed circuit with cyclones to achieve the final product size. The cyclone overflow stream will flow by gravity to a linear trash screen and then a pre-leach thickener. Barren solution leaving the continuous ion exchange circuit is recycled as wash to the leach residue filters and then, via the filter washate, returns to the pre-leach thickener where most of the contained acidity is neutralized by the gangue in the ore. Pre-leach thickener overflow, which is very mildly acidic, is used as dilution water in the milling circuit. The thickened slurry is pumped to the leach circuit where it is mixed with concentrated sulfuric acid for uranium leaching. Manganese dioxide (as high quality milled pyrolusite) is added to the leach circuit to control the redox potential. The uranium leach step is carried out at a temperature of 40°C, and this temperature is provided partly from the heat of dilution of sulfuric acid and partly by live steam addition to the leach tanks. The leach tailings stream is filtered and washed to recover the pregnant liquor before being conveyed as a wet cake to the tailings storage facility. Pregnant leach solution flows to a continuous ion exchange circuit where the uranium and some iron, silicon and molybdenum are adsorbed onto the resin. Impurity elements of iron, aluminium, calcium, manganese, arsenic, etc., are entrained with the resin in the interstitial fluid. The majority of the interstitial fluid is displaced prior to elution.

The uranium, together with the minor elements as impurities is eluted from the resin with concentrated sulfuric acid to produce a concentrated eluate solution containing approximately 7.5 g/L U. The impurities present in the eluate solution are then removed, firstly by partial acid neutralization with sodium hydroxide to produce a precipitate containing uranium, iron, aluminium, and arsenic. This precipitate is recycled to the leach step to recover the uranium. The partially neutralized solution is then treated with sodium hydroxide and hydrogen peroxide to precipitate crude uranium oxide concentrate UOC with associated impurity levels acceptable for sale to a convertor. The barren solution produced, containing the remaining impurities including sodium and sulfate is discharged to an evaporation dam for disposal.

The UOC is then washed to remove entrained mother liquor, dried, and then packaged into clean, thick wall 200 L drums, and prepared for shipment.

17.2.1 Ore Recovery and Crushing

Run-of-mine (ROM) ore from the open pit, at maximum lump size of 900 mm, will be transported to the plant by rear tipping dump trucks. The trucks will tip directly into the ROM bin. When trucks are unable to tip into the ROM bin, the truck load will be dumped onto the ROM pad. The ROM pad will be primarily utilized for emergency storage and ore blending if required. ROM ore will be reclaimed to the ROM bin by a front-end loader (FEL).

A rock breaker will be installed to assist in breaking down, oversize material retained on the grizzly above the ROM bin. Ore will be withdrawn from the ROM bin via an apron feeder and passes to a vibrating grizzly ahead of the jaw crusher. Primary fines separated by the vibrating grizzly pass to the primary crusher discharge conveyor belt. Vibrating grizzly oversize is crushed by the jaw crusher and passes to the crusher discharge conveyor.

The crushed ore will be conveyed, via the crushed ore bin feed conveyor, to the crushed ore bin. The crushed ore bin feed conveyor will be fitted with a weightometer, to monitor primary crusher throughput, and for control of the apron feeder variable speed drive.

The crushing circuit will be serviced by a common dust collection system, consisting of multiple extraction hoods, ducting, and a dust scrubber. Dust collected by this system will be slurried in the dust scrubber and pumped by the dust scrubber disposal pump to the pre-leach thickener feed box. Radon gas collected by the extraction hoods will pass through the scrubber, and be vented to atmosphere via a 30 m tall scrubber vent stack.

17.2.2 Crushed Ore Bin

The crushed ore bin will have a live capacity of approximately 1,900 t (equivalent to 5 h of mill feed at 2.9 Mtpa).

Crushed ore will be reclaimed from the bin, by a variable speed apron feeder. The feeder will discharge onto the SAG mill feed conveyor, which will convey the crushed ore to the SAG mill feed chute. The SAG mill feed conveyor will be fitted with a weightometer, used for controlling the speed of the reclaim feeder and for mass accounting of feed presented to the grinding circuit.

Dust extraction hoods are provided at the crushed ore bin and feed and discharge conveyor transfer points to convey dust and radon gas to a dust scrubber and vent stack.

17.2.3 Grinding and Classification

The grinding circuit is a SS SAG/C circuit, comprised of a single, variable speed, SAG mill. The single stage SAG mill will operate in closed circuit with hydrocyclones and a pebble crusher. The target product particle size exiting the grinding circuit (cyclone overflow) will contain P_{80} 180 µm material.

To meet the design throughput, and achieve the required leach product size, a 9.14 m diameter (dia.) x 7.51 m SAG mill, with 10.5 MW installed motor power is required.

Crushed ore, reclaimed from the crushed ore bin, will be conveyed to the SAG mill feed chute via the SAG mill feed conveyor. Process water will be added to the SAG mill feed chute, to control the in-mill pulp density. The process water is recycled from the pre-leach thickener overflow and consists of a mixture of fresh make-up water, acidic barren liquor recycled from the continuous ion exchange (CIX) circuit via the tailings filter wash, and scrubber bleed water from the ore prep dust scrubber and the leach circuit scrubber. The SAG mill will be fitted with discharge grates, which will allow slurry to pass through the mill and will also relieve the mill of pebble build-up. The SAG mill product will discharge to a mill trommel screen for pebble dewatering.

Grinding media (125 mm balls) will be added to the SAG mill via the SAG mill feed chute, utilizing a dedicated media hoist, kibble, and feed chute.

SAG mill discharge trommel screen oversize will be conveyed to a pebble crushing circuit. Undersize from the trommel screen will flow by gravity to the cyclone feed pump box. The slurry will then be pumped to the cyclone cluster by one of two (duty / standby) variable-speed cyclone feed pumps. Process water will be added to the cyclone feed pump box for cyclone feed density control.

The cyclone cluster overflow will flow by gravity through a metallurgical sampler then onto two vibrating trash screens in a parallel configuration. The trash screen undersize will be directed to the leach thickener feed whilst trash screen oversize will pass to a trash kibble and associated liquor will flow to a mill area sump pump. Slurry from the cyclone underflow launder, will be returned to the SAG mill feed chute.

Spillage within the grinding circuit will be managed, utilizing a dedicated drive-in sump and sump pump. Any spillage generated in the grinding area will be returned to the cyclone feed pump box.

17.2.4 Pebble Crushing

Oversize from the SAG mill discharge trommel will be conveyed to the pebble crusher feed bin, via a series of belt conveyors. A self-cleaning belt magnet will be positioned at the head chute of the first conveyor to remove any scrap metal and steel media which could potentially damage the pebble crusher.

Downstream of the self-cleaning belt magnet, the pebbles will pass under a metal detector, prior to discharging into the pebble crusher feed bin. The feed bin will provide surge capacity ahead of the pebble crusher and allow a controlled feed to be presented to the crusher. Should the pebble crusher not be operational, a diverter gate ahead of the pebble crusher feed bin will allow pebbles to bypass the pebble bin and crusher and feed directly to the pebble crusher discharge conveyor. Similarly, should the metal detector detect tramp metal (not removed by the self-cleaning belt magnet), the diverter gate ahead of the pebble crusher feed bin will automatically allow pebbles to bypass the pebble bin and crushers and feed directly to the pebble crusher discharge conveyor.

Pebbles will be withdrawn from the pebble crusher feed bin, by a vibrating feeder, which will be variable speed. The pebble crusher will discharge crushed pebbles directly onto the pebble crusher discharge conveyor, which in turn will return the crushed pebbles to the SAG mill feed conveyor.

The pebble crusher discharge conveyor will be fitted with a weightometer, used for mass accounting of feed presented to the grinding circuit.

17.2.5 Pre-Leach Thickening

Trash screen undersize will flow by gravity directly to the pre-leach thickener feed box, where flocculant will be added to aid with particle settling. Overflow solution from the pre-leach thickener will flow by gravity to the pre-leach thickener overflow tank, and then be pumped to the SAG mill feed and cyclone feed as dilution water. Underflow slurry from the pre-leach thickener, at 55% solids density, will be pumped by dedicated thickener underflow pumps, to the pre-leach thickener underflow tank. A thickener recycle pump is included to improve thickener operational flexibility when running or ensure compaction of the thickener bed does not occur if the thickener is offline for a plant shutdown.

The pre-leach thickener area will be serviced by a dedicated floor sump pump. Spillage and wash down collected by the sump pump will be returned to the cyclone feed box.

Further testwork may negate the need for a pre-leach thickener if cyclone discharge is shown to provide adequate discharge density.

17.2.6 Leach Circuit

Pre-leach thickener underflow is pumped to a thickener underflow leach tank that provides a 2 h surge capacity ahead of the leach circuit. The leach circuit consists of six mechanically agitated rubber lined and insulated mild steel tanks in series. A bypass facility is provided between tanks to enable online maintenance. A total residence time of 12 h is provided in the six tanks. Sulfuric acid (98%) is added via a flash mixing tank to maintain a leach slurry pH (hydrogen ion exponent (measure of acidity or alkalinity)) of 1.5. The first leach tank receives approximately 4°C of heat from the diluting acid and the balance of the energy required to raise the temperature to 40°C is provided from a direct steam addition. Pyrolusite oxidant (48% soluble manganese (Mn)) is added as a finely ground slurry to the leach to maintain an ORP of 500 to 550 mV (Ag / AgCl +3.8M KCl).

The leach process produces a metastable leachate containing mainly calcium, iron, potassium and arsenic, some of which precipitate after the leach. For this reason, a fluid bed ion exchange system was selected to recover the uranium.

Emissions of dust and vapors from the atmospheric leach circuit are expected to be low and consequently no allowance has been made for vent collection and scrubbing. This needs further consideration and confirmation as the project moves into the next development stage.

17.2.7 Leach Residue Filtration

Leached slurry exits the leach circuit and passes to the leach discharge transfer tank after which it is flocculated in two stages prior to pressure filtration in vertical chambers, which are fitted with inflatable membranes. The membranes consolidate the residue as a competent cake free of ‘tear drop’’ and facilitate the dewatering process and uranium recovery in a displacement wash employing the ion exchange barren liquor. The purpose of fines flocculation is to increase the slurry filtration rate on the residue filters.

The filter feed is provided by a high head centrifugal pump (for each individual filter) and four filters are employed in parallel. The primary filters have a total filtration area of 2,380 m^2. Each filter has 62 chambers of 40 mm depth, using a 2.5 m x 2.5 m plate size. Unclarified filtrate from the primary filters flows to the CIX feed tank.

Primary filter cake is discharged to a common residue repulp box where it is repulped with secondary leach filter residue washate and then pumped to the leach residue secondary filter feed tank. The repulped residue is then pumped to four leach residue discharge secondary filters. These filters are similar to the primary filters in area and size.

Secondary filter filtrate is used to wash the primary filter cake which then reports to the CIX feed tank. CIX barren liquor is settled in a pond where after it is employed to wash the secondary filter cake and this washate then reports to the filter washate tank from where it is either pumped to the pre-leach thickener or for use as flocculant dilution in the leach residue filtration step.

The washed filter cake is removed from the filter chambers at an approximate 18.7% free moisture containing all the gangue acid solute and broken over cake breakers before being conveyed to a lined tailings facility designed to store the leach residue.

17.2.8 Continuous Ion Exchange

The continuous counter-current ion exchange step consists of a two fluid bed modified National Institute of Metallurgy continuous ion exchange (NIMCIX) extract column and eight fixed bed columns each of which is capable of being employed in resin receipt (from extract), rinse, elution, regeneration, wash and resin elutriation, and discharge to the extract column. These fixed bed columns are serviced with a multiport valve. Elutriation is independent of the multiport valve using barren liquor.

Each of the extract columns has 10 sieve trays and the PLS is intermittently stopped and a reversal in flow is accommodated to index the resin flow. From a review of the calculated PLS assay, it is quite possible that a 40 g/L (wsr) uranium loading on the resin will be achieved (much higher loadings were achieved in laboratory loading trials). The barren liquor emanating the column is passed over a guard screen to remove any broken beads or misreporting resin, and then stored in a pond to settle out fines and jarositic type precipitates.

Eluted and regenerated resins are restored to the top of the extract column where it commences its down flow in the column as described above.

The loaded resin is received in a column containing approximately 26 m^3 of wet resin and rinsed in a single stage with clean water to displace solutes. Some minor solute carryover is likely thus it becomes a contaminant carried forward with the concentrated eluate.

The elution process is conducted at 40°C employing a 1.2 molar sulfuric acid liquor. The uranium peak in the elution curve is harvested as a concentrated eluate while the remaining weaker eluate is returned to the eluant tank. On completion of the elution process, any eluant in the trail column is displaced with clean water.

Approximately 10% of the eluted resin flow is regenerated to remove surficial silica foulant and a weak caustic soda regenerant is employed in this duty (if required). Silica levels are kept below 2% and typically below 1.5% on the resin inventory. The regenerated resin is restored to service in the hydroxide form and is then converted to the sulfate form in the flared section of the extract column.

A variety of vessels support the continuous ion exchange system. Resin make-up requirements are influenced by housekeeping and transfer practices. Recessed impeller pumps are employed along with hydraulically pumped elutriation systems to soften the resin transfer and minimize resin degradation.

The barren liquor from the extract column is stored in a surge pond with 24 h of retention time before being employed in the leach residue filter wash steps.

17.2.9 Impurity Removal

A preference was adopted to avoid the use of patented technologies and to exclude limestone addition as an initial acid neutralization step. Sodium hydroxide was instead incorporated to simultaneously remove soluble iron transferred within the interstitial fluid. This step is a precursor to the precipitation of a UOC.

Very few other impurities are removed in this step with manganese notably being the more difficult impurity (derived from pyrolusite – although laboratory results to date do not indicate significant Mn transfer) and small quantities of silicon and aluminium co-reporting to the UOC. The UOC may attract penalties because of this modified impurity removal step. Alternative neutralizing agents exist that could be easily substituted into this duty without significant cost or process implications, which are planned to be investigated during future project evaluations.

The iron precipitate is thickened in a very small thickener. A portion is recycled to the precipitation circuit as a seed with the remainder returned to the leach circuit after filtration where it is re-leached together with any co-precipitated uranium.

Alternative impurity removal in the form of a pre-elution mild acid or reducing scrub of the resin have been shown as effective in the minimization of impurity transfer to the precipitation circuit.

17.2.10 Crude UOC Production, Washing and Drying

A crude UOC is precipitated from the iron–free pregnant eluant employing hydrogen peroxide as an oxidant and sodium hydroxide to neutralize the hydrolysis acid. The hydrogen peroxide will oxidize most polyvalent metals present in the pregnant eluant in addition to the uranium and consequently any remaining trace iron and manganese will co-report with the uranium product. The high sulfate containing matrix also results in some sulfur entrainment in the UOC, and potentially incomplete uranium precipitation. The UOC is thickened and a majority is recycled to the first tank in the precipitation cascade as a seed to coarsen the crystals and minimize the entrainment of sodium and sulfate into the product.

The UOC is washed to remove the sodium sulfate mother liquor in a combination of filtration and centrifuging process. This combination provides redundancy in the event of the failure of one of these two units. Alternately two centrifuges could be employed in a series-parallel mode. The washed UOC product is dried in a continuous horizontal kiln and transferred under gravity to a storage hopper before packaging. The use of bucket elevators and screw conveyors is avoided and the application is made of high face velocity extract vent systems in compliance with the as low as reasonably achievable (ALARA) principles that govern the engineering of the Controlled Areas in Uranium plant.

A 'clean and dirty' change facility with radiation monitoring facilities is provided. The entire Controlled Area is closed to prevent uncontrolled access.

17.2.11 UOC Packaging

The dry product is weighed into clean, thick wall 200 L drums and prepared for shipment. Approximately 300 kg of UOC is loaded into a drum. The UOC product will assay approximately 70% uranium. After packaging the drums are externally washed and dried. Filled drum storage is under roof cover and the filled drums are loaded into containers under roof. The drums are loaded into 20 t containers and strapped to the floor of the container in a prescribed securing map in compliance with regulations.

17.2.12 Sulfuric Acid Production

The Project requires approximately 98,000 tonnes per annum (tpa) (269 tonnes per day (tpd)) of 98% sulfuric acid for ore leaching and ion exchange elution. Acid will be produced from an onsite sulfur burning sulfuric acid plant, constructed as part of the project. A range of technology licence options exist for sulfuric acid production. This scoping study is based on a modular Metso acid plant, which incorporates their double-catalysis design to meet higher emission requirements and operational flexibility (40 to 100% capacity).

A brief description of the process is provided below.

The solid sulfur is melted using low-pressure steam, after which the liquid sulfur is pumped to a filtration stage to removed solid impurities. The filtered liquid is then pumped to the sulfur burner. Combustion air required for burning elemental sulfur is dried in a drying tower using concentrated sulfuric acid as the absorption medium. The dried air is then sent to the refractory-lined combustion furnace, where sulfur is oxidized to sulfur dioxide (SO_2). The hot SO_2 rich process gas is cooled in an integrated waste heat boiler (WHB) equipped with a steam drum, producing saturated steam.

Downstream of the WHB, the cooled process gas enters a four-bed catalytic converter where SO_2 is exothermically converted to sulfur trioxide (SO_3). Between the first and second catalyst beds, the hot gas is cooled in Superheater 2, which provides final superheating of steam to the required battery-limit conditions. After the second bed, the gas is further cooled in Superheater 1, where saturated steam from the WHB is superheated in the first stage before the gas is admitted to the third catalyst bed.

After passing through the third bed, the gas is cooled in a gas-gas heat exchanger and routed to the intermediate absorption tower, where SO_3 is absorbed into circulating sulfuric acid. To achieve the required inlet temperature for the fourth bed, the gas is reheated in the same gas-gas heat exchanger before re-entering the converter. Following the final conversion stage, the gas is cooled in an economizer and directed to the final absorption tower, where the remaining SO_3 is absorbed.

The cleaned tail gas passes through candle filters and is then safely vented to the atmosphere via a stack.

Circulating and product acids are cooled in dedicated heat exchangers. The final sulfuric acid product, at 40°C and 98.5% wt concentration, is withdrawn from the overall acid system and delivered to the storage tanks.

All the heat of SO_2 oxidation, and a large part of the heat of SO_3 hydration, and of acid condensation is recovered through the interbed coolers / steam system, and then used for steam export. The Metso process recovers the surplus energy as steam (up to 60 bar gauge (barg)). The steam is sent to a pass out turbo alternator, where a portion of the steam is extracted at 6 barg and used for live steam process heating in the acid leach circuit and the remainder condensed and returned to the acid plant waste heat boiler. The turboalternator generates 1.6 MW of electric power for use on the site.

17.2.13 Reagents Mixing and Storage

The major reagents utilized within the process plant will include:

- Caustic soda (NaOH) for ion exchange resin regeneration, acid neutralization, and crude UOC precipitation.
- Concentrated sulfuric acid for ore leaching and ion exchange elution and demineralizer resin regeneration.
- Hydrogen Peroxide for UOC precipitation.
- Pyrolusite (for leach circuit redox potential control).
- Flocculants for thickening and filtration.
- Sulfur for the production of sulfuric acid.
- Water treatment chemicals for boiler feed water and cooling water treatment.

Sodium Hydroxide (NaOH)

Sodium hydroxide (caustic soda – 'caustic') will be delivered to site as solid pearl in 1 t bulk bags and dissolved in water to produce a 30% w/w caustic solution for addition to the process.

Concentrated Sulfuric Acid

Concentrated sulfuric acid (98% w/w) 269 tpd will be produced onsite in a dedicated sulfur burning contact sulfuric acid plant. The product acid will be stored in two 450 m^3 acid storage tanks and pumped from storage to the leach and elution process steps.

Hydrogen Peroxide

Hydrogen peroxide (50%) will be delivered to site in intermediate bulk containers (IBCs) and offloaded into onsite storage tanks constructed for hydrogen peroxide service. The fluid will be diluted immediately prior to use in the uranium oxyhydrolysis step to approximately 30%. This lower concentration is preferred to prevent spontaneous decomposition by other polyvalent ions in the feed matrix.

Pyrolusite

The pyrolusite will be milled on site, depending on the source, and delivered as a 50% slurry with a target product grind size of P_{80} 45 µm. The pyrolusite will contain 48 to 50% acid soluble manganese as manganese dioxide. The pyrolusite slurry will be able to be dosed into three of the leach tanks.

Flocculants

Flocculants are required to aid in both slurry pre-leach thickening and leach residue filtration. Allowance has been made for two separate flocculants each being prepared in a separate plant as described below.

Flocculant micro-breads powder will be delivered to site in 1,000 kg bags and 15 kg bags. The bulk bag will be lifted, by the flocculant hoist, to the storage hopper. The smaller bags will be manually lifted to the respective storage hopper. Flocculant will be mixed in a proprietary mixing system, comprized of a bulk dry hopper, screw feeder, flocculant blower, and mixing tank. Each flocculant plant will mix flocculant powder with fresh water to achieve the required storage concentration (0.25% w/w).

Flocculant will be withdrawn from the storage hopper by the flocculant screw feeder. The screw feeder will convey flocculant to the flocculant eductor, from which the flocculant powder will be pneumatically conveyed, to the flocculant mixer, by the flocculant blower. Fresh water will be added to the mixer, to hydrate the flocculant powder, prior to discharging into the agitated flocculant mixing tank. Upon completion of the mixing cycle, the flocculant will be transferred to the flocculant storage tank, by the flocculant transfer pump.

Sulfur

Solid sulfur is required onsite for the production of sulfuric acid. Annual sulfur requirement will be 34,499 tpa of prilled / flaked sulfur. Sulfur will be trucked from Townsville and stored in a covered storage shed on site adjacent to the sulfuric acid plant.

Water Treatment Chemicals

Miscellaneous chemicals will be required for water treatment for potable water, boiler feed water, and cooling water, including the following:

- Chlorine.
- Corrosion inhibitors.
- Algaecide and biocide.
- Dispersants.

17.2.14 Water Services

The process plant will utilize fresh water, process water, demineralized water, filtered water, gland water, and potable water.

Raw Water, Filtered Water and Fire Water

Fresh water for the process plant and mining operation will be sourced from local bores. The bores will pump to the raw water storage tank. Raw water is filtered and stored in a filtered fresh water storage tank from where it is distributed to all non-acidic process water uses, including reagent make-up, gland water, cooling water make-up, and boiler feed water make-up.

The filtered fresh water tank will provide a combined filtered fresh water and fire water reserve.

Firewater will be supplied from the filtered fresh water storage tank, via a dedicated suction manifold. The firewater system will comprise:

- An electrical jockey pump.
- An electrical firewater pump.
- A diesel standby firewater pump.

The firewater system pressure will be maintained by the jockey water pump. An electric fire water pump will automatically start on a drop in line pressure. The diesel fire water pump will automatically start if the line pressure continues to drop below the target supply pressure, which will occur when there is significant fire water demand or during a power failure.

Process Water

Pre-leach thickener overflow provides the source of acidic process water. This water is used for a limited number of services including:

- SAG mill feed dilution water.
- SAG mill discharge trommel sprays.
- Pre-leach thickener / leach filter flocculant dilution water.
- Trash screen water sprays.
- Service points around the plant.

Demineralized Water

A total of 20 m^3/h of demineralized water will be required in the processing plant. Plant areas requirements include:

- High pressure steam production from waste heat generated during normal operation at the acid plant.
- Low pressure steam requirements for the uranium refinery.
- UOC thickener underflow filter wash water.
- Crude UOC repulp water.
- Flocculant make-up for uranium refinery.
- NaOH make-up water.

Gland Water

Gland water will be filtered fresh water and stored in a dedicated gland water storage tank. Dedicated gland water pumps will supply gland seal water to the required pumps across the process plant.

Potable Water

Fresh water for potable water use will be sourced from local water bores. The bores will pump to the potable water feed tank. Potable water will be distributed for human consumption across the site, and to the safety showers and eye wash stations throughout the process plant. The safety showers are reticulated from a dedicated safety shower water tank via a ring-main and dedicated safety shower water pumps. If the water temperature in the safety shower water tank becomes too high, then the ring-main will be discharged to the raw water tank whilst the water in the safety shower water tank is refreshed with replacement potable water.

Cooling Water

Cooling water is required for the acid plant operations. A 700 m^3/h system with a heat load of 8 MW is required.

17.2.15 Steam Services

A 2 tph low pressure steam boiler and associated ancillaries are required for steam services in the uranium refinery.

17.2.16 Air Services

Plant and instrument air at 700 kilopascal gauge (kPag) will be provided by two high pressure air compressors, operating in a lead-lag configuration. The entire high pressure air supply will be dried and can be used to satisfy both plant air and instrument air demand. Dried air will be distributed via one dedicated air receiver for the entire process plant.

18.0 PROJECT INFRASTRUCTURE

18.1 Introduction

Project infrastructure for the Project includes water supply, electric power supply, access roads, sewage treatment, an accommodation village, and airport. The proposed tailings disposal concept enables progressive mining and backfilling of the RT pit over the life of the operation. The proposed strategy comprises two sequential stages: an initial ex-pit tailings placement phase, followed by in-pit tailings placement once the western shallow section of the pit is excavated, lined, and ready to be backfilled.

18.2 Project Water Supply

A project water balance indicates an average water demand for the mine and treatment facility of 211 m^3/h. A further 2 m^3/h of potable quality water will be required for the accommodation village. An assessment of water supply options was undertaken by GWS during the 2015 study. These options remain unchanged and are summarized as follows:

- Estimated in-pit rainfall run-off is significant and may exceed water demand in some months.
- Estimated groundwater seepage to the mine pits is negligible.
- Sufficient water supply from local borefields is likely to be available.

Available hydrogeological data were examined to identify potential water supply targets, and provides estimates of mine pit rainfall run-off and groundwater seepage as inputs to the project water balance. The study applied an upper and lower estimated wetting threshold of 10 mm to 20 mm per months based on:

- Average four rainfall events per month in wet season (consecutive rain days are one event).
- Based on statistical analysis of rainfall data from Burketown Post office from 1887 to 2015.
- Wetting threshold per rain event of 5 mm. The recommended initial loss values for rocky steep catchments (>3% general slope) can vary from 0.5 mm to 7 mm per rainy event, depending on the wet conditions of the rock (U.S. Soil Conservation Service Hydrology, 1986).

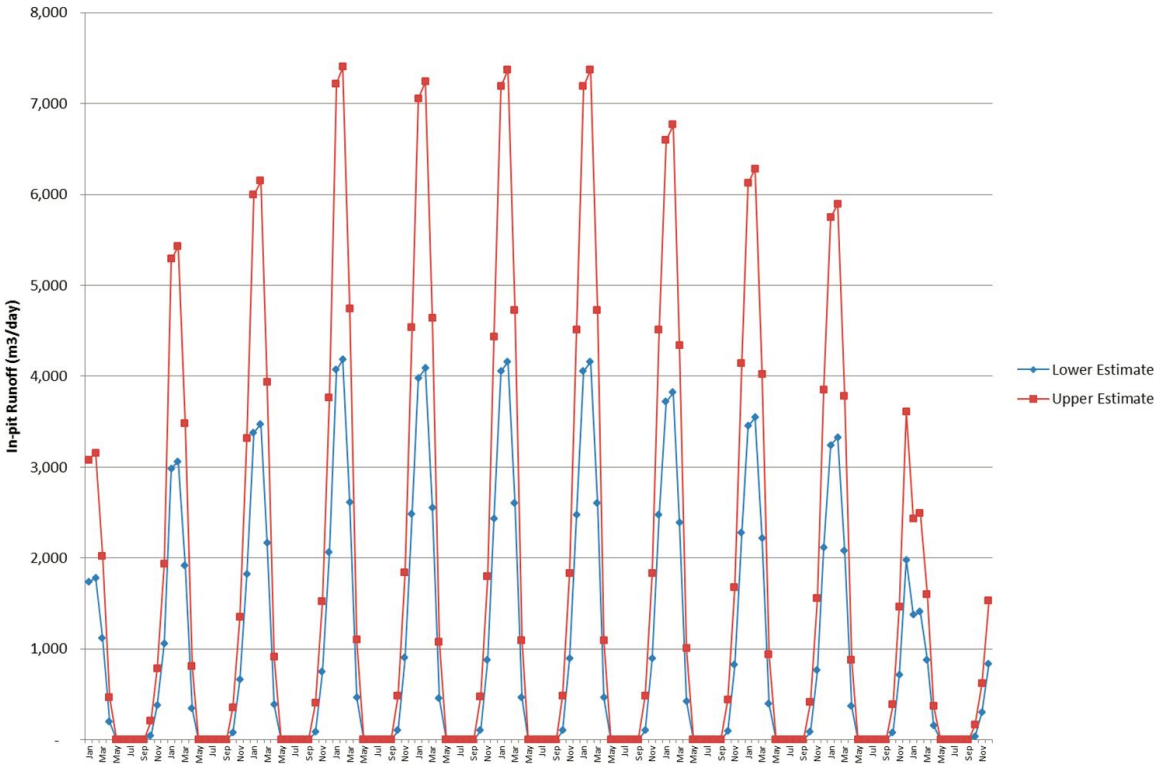
The results of the study indicated that in-pit run-off from May to September would be 0 mm (Figure 18.1.11). The maximum in-pit run-off in February ranges from 75 mm per month (lower estimate), to 133 mm per month (upper estimate).

Calculated average annual run-off volume based on the total estimated pit area ranges from 450,000 m^3 to 830,000 m^3. Maximum values range from 4,200 m^3/day (lower estimate) to 7,400 m^3/day (upper estimate) in February.

In-pit run-off will be pumped from the pit for use in the process plant. The water will exhibit low salinity and high suspended solids. Run-off may exceed demand for two to four months (December through March) per year. This can be managed by:

- Storage dams at surface to contain excess water.
- Mine pit scheduling to provide a lower sump / bench that can be inundated for one to two months per year.

Figure 18.2.1 Calculated In-Pit Rainfall Run-Off



Groundwater seepage to the mine pit was calculated using the Dupoit-Forsheimer Equation. Seepage is negligible. Calculated rates for each pit range from 10 m^3/day (best estimate) to 100 m^3/day (conservative high estimate) in the first year of mining. No management is required. Seepage will evaporate in dry months and be subsumed by rainfall run-off in wet months.

Two borefield options for water supply exist (see Figure 18.2.2.):

- From the Great Artesian Aquifer 45 km east of the Project site.

- Near mine aquifers from Westmoreland Conglomerate sandstone located 20 km east of the Project site.

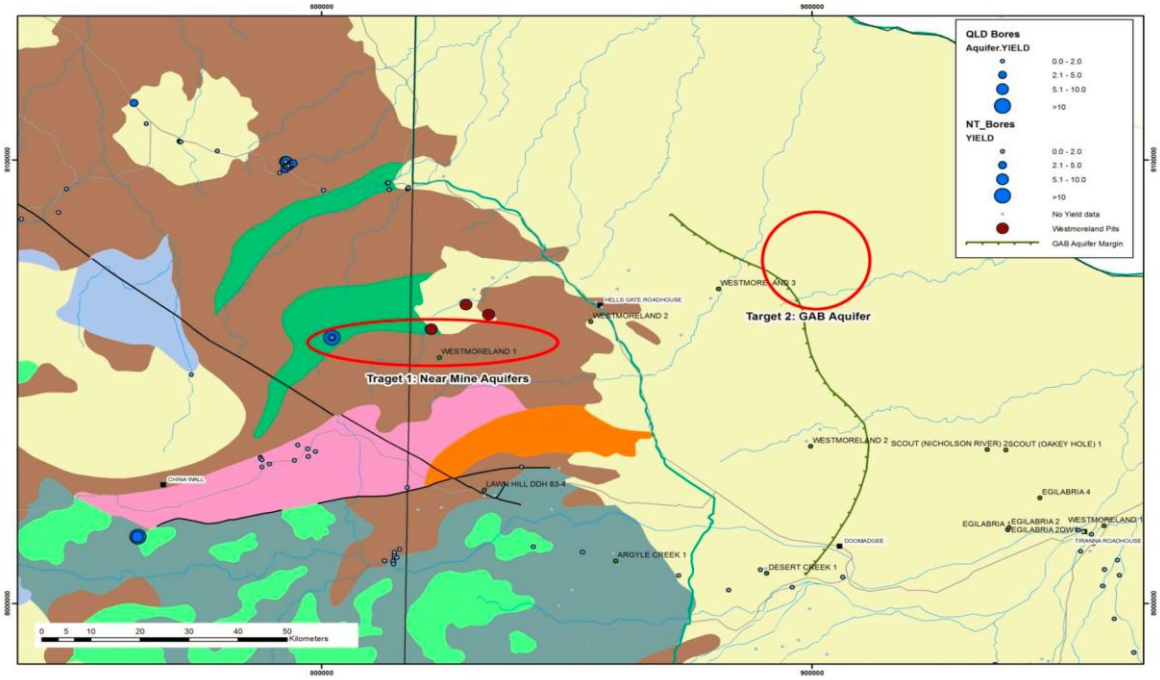
Water drawn from a borefield will vary depending on seasonal surface water harvesting. During periods of no surface water harvesting average demand on the groundwater supply will be 211 m^3/h. Demand will be reduced during periods of surface water capture.

Table 18.2.1 compares the borefield supply options. Capital and operating cost estimates were developed for each option, and details of these are presented in Table 18.2.2.

Table 18.2.1 Borefield Water Supply Operations

	Great Artesian Basin Aquifer	Near Mine Aquifers
Description	The Great Artesian Aquifer margin is located some 45 km east of the Project site (Figure 18.2.2). The presence of the aquifer in this area has not been confirmed by drillholes. The location and extent is inferred from Seismic Survey (Grimes and Slater, 1976). The absence of the aquifer further west (closer to Westmoreland) has been confirmed by BMR strat hole Westmoreland 3, drilled in 1973.	Drilling records in the NT identify bores that yield up to 12 litres per second (L/s) from sandstone described as the Westmoreland Conglomerate (Bore RN 009279), located 20 km west of the project site (Figure 18.2.2). Discussion with local water bore drillers (Kelly Drilling Pers com July 2015) suggest that bores in this formation typically yield 2 to 5 L/s. Note: QLD records of water bores are less detailed that NT, which make bore yields difficult to review.
Distance	45 km	20 km
Risk	Moderate – location of aquifer margin is not tested	Moderate – aquifer capacity has not been tested

Figure 18.2.2 Hydrogeological Setting and Groundwater Supply Targets



Capital cost estimates for each option are presented in Table 18.2.2.

Table 18.2.2 Water Supply Options Capital Estimates

		Source		Option A GAB A$	Option B Near Mine A$	Average A$
Direct Costs						
A	Supply	Consultant	-	2,939,000	2,900,000	-
B	Installation	Consultant	-	635,000	420,000	-
C	Freight	Consultant	10%	357,400	332,000	-
D = A + B +C	**Total Direct Costs**	**Calculation**		**3,931,400**	**3,652,000**	**3,791,700**
Indirect Costs						
E	Contractors / construction indirect	Consultant	-	70,000	130,000	-
F	Other indirect costs as necessary	Consultant	-	-	-	-
G = E + F	**Total Indirect Costs**	**Calculation**		**70,000**	**130,000**	**100,000**
H = D + G	**Project Cost before Contingency**	**Calculation**		**4,001,400**	**3,782,000**	**3,891,700**
I	Contingency (=x% of H)	Calculation	50%	2,000,700	1,891,000	1,945,850
K = H + I	**Total Project Installed Cost**	**Calculation**		**6,002,100**	**5,673,000**	**5,837,550**

18.3 Project Electric Power Supply

Three options were investigated as part of the power supply assessment for the Project. These options are still in the conceptual evaluation stage, with no commercial undertakings being considered at this juncture. For the purposes of the study, the options have been investigated in sufficient depth to give a reasonable degree of confidence that the preferred option can be achieved within the cost parameters specified.

The project maximum continuous power draw has been estimated at 16.1 MW, with an average continuous power draw of 11.4 MW. The power supply options considered were:

- Standalone diesel generation.
- A hybrid diesel plant with solar PV array and BESS.
- A 155 km extension of the 220 kV transmission line that runs from Mount Isa to the Century zinc mine as part of the Mount Isa isolated network.

18.3.1 Diesel Generation

A standalone diesel generation plant would consist of 6 x 3.5 MW units, providing a total installed capacity of 21.0 MW and a firm capacity of 14.0 MW with four units running. This is adequate to serve the plants average power demand while maintaining N+2 integrity.

18.3.2 Hybrid Diesel Plant with PV / BESS

The Project site lies within one of Australia's highest solar resource zones. The site receives approximately 5.8 to 6.2 peak sun hours per day averaged annually, with strong irradiance from September to April. Three renewable fraction scenarios were modelled for comparison (20%, 40% and 60% RF) with equivalent BESS capacity for each scenario. These scenarios are combined with the same diesel generating plant in Option 1 to provide 24 h power supply to the site.

18.3.3 Extension of 220 kV Transmission Line from Century Zinc

A grid connection option has been assessed based on a new 220 kV transmission line from the Mount Isa isolated network to the Westmoreland site, via the Century mine corridor, at an estimated length of 155 km. The connection would connect into the existing Mount Isa 220 kV network at the Century mine 220 kV substation, and would require a new 30 megavolt-ampere (MVA) step-down substation at the Westmoreland site.

18.3.4 Power Costs Summary

A summary of capital and operating costs and a comparative levelized cost of electricity (LCOE) for each option is provided in Table 18.3.1.

Table 18.3.1 Summary of Power Supply Options

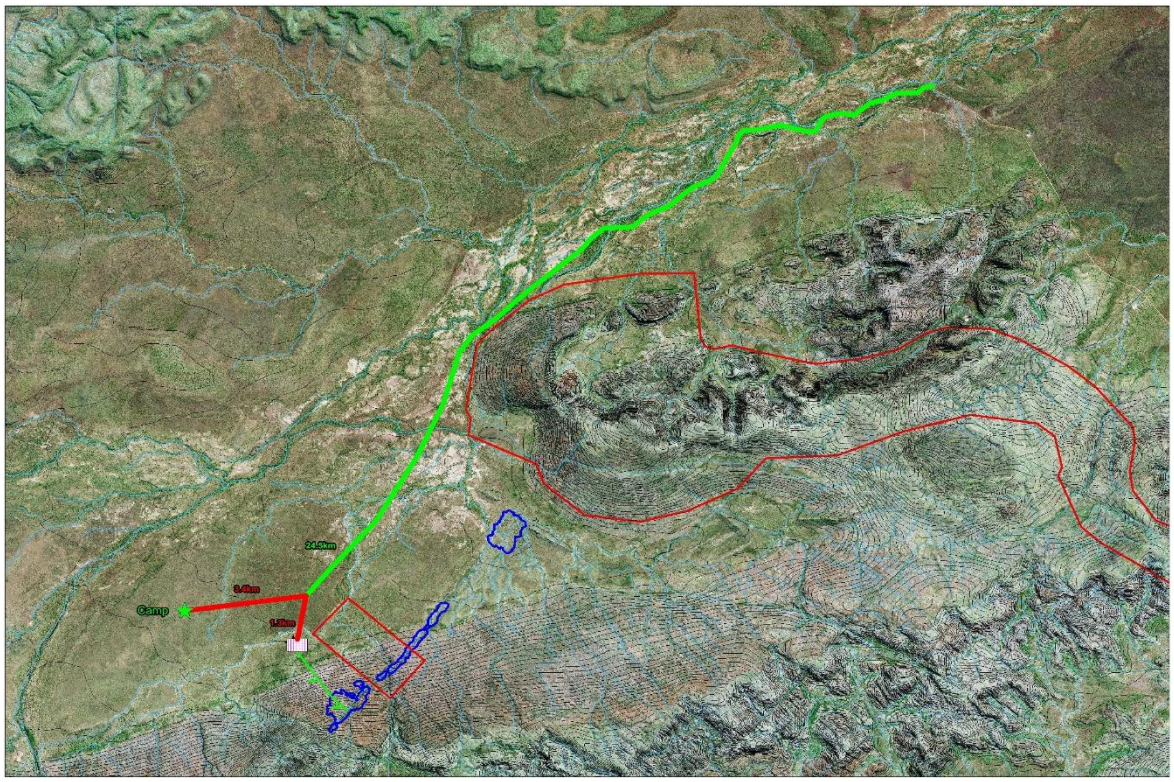
	Capex A$...M	Opex A$...M /y	Annual Cost A$...M /y	LCOE A$ /MWh
All Diesel	32.8	28.01	42.8	392
Hybrid – 20% RF	63.4	29.38	38.7	354
Hybrid – 40% RF	99.0	23.00	37.5	344
Hybrid – 60% RF	179.6	16.37	42.7	392
Grid Connection	137.6	19.38	40.2	349

The hybrid solar / BESS option at 40% RF achieves the lowest levelized cost of electricity and has been used as the basis of power supply for the Project. This option also provides a 43% reduction in Scope 1 carbon emissions compared to the diesel power generation option.

18.4 Project Access Roads

Allowance has been made for an access road from the Savannah Way to the mine site and accommodation camp; refer to Figure 18.4.1. The first half of the road follows an existing track, and the remainder of the roads are new. The existing part of the road travels along the Lagoon Creek and crosses the creek twice. Including the access road extensions to the mine site and accommodation camp, the total length of is approximately 29.2 km. Other options for the access roadway exist, but the option shown has been adopted for the purpose of the SS.

Figure 18.4.1 Site Access Road Route



18.5 Accommodation Camp and Airport

Allowance has been made for a 250 man camp to provide accommodation for personnel during construction and fly-in, fly-out (FIFO) operations and maintenance staff once plant operation commences. The accommodation camp will be fit for purpose and reflect standards of condition expected in the mining industry at the present time to employ and retain staff.

An all-weather gravel airstrip will be provided for transport of personnel and emergency medivac purposes. No allowance has been made for lighting for night landings or take-offs.

18.6 Administration, Mining and Process Buildings, Sewage Treatment

Allowance has been made for inclusion of building infrastructure covering administration, mining and exploration, process, including changerooms, ablutions, workshops, stores, explosives magazines, site roadways and carparks, and reticulation of potable water, electric power and sewerage.

18.7 Tailings Management

The tailings will leave the plant in a semi-dry state, which presents an opportunity to dispose of the material as a stacked product. While this study assumes the tailings will not be neutralized prior to disposal, detailed geochemical studies will be required in the next design phase to assess the solubility of metals and radionuclides, ensuring that any seepage water chemistry can be safely managed.

The current concept is to dispose of these filtered tailings within the RT pit. This strategy allows the pit to be backfilled sequentially during mining operations, effectively eliminating the final mining void and minimizing the Project's overall disturbed environmental footprint. The primary challenge with the in-pit backfilling strategy would be storing the tailings during the initial years of operation, before the RT pit is sufficiently excavated and prepared to receive tailings.

A conceptual strategy has been adopted to eliminate the operational complexity and cost of re-handling the material. This optimized approach avoids the need for a temporary paddock facility by placing the filtered tailings immediately outside the RT pit footprint during the initial stages, then extending placement into the RT pit as soon as it becomes available. At closure, this produces a single, continuous landform without requiring any material to be rewon or relocated. The resulting plan is conceptual design of integrated ex-pit / in-pit tailings filtered stack facility.

The proposed tailings disposal concept enables progressive mining and backfilling of the RT pit over the life of the operation. The proposed strategy comprises two sequential stages: an initial ex-pit tailings placement phase, followed by in-pit tailings placement once the western shallow section of the pit is excavated, lined, and ready to be backfilled. This approach allows tailings deposition to commence at the start of operations while aligning with the mining sequence.

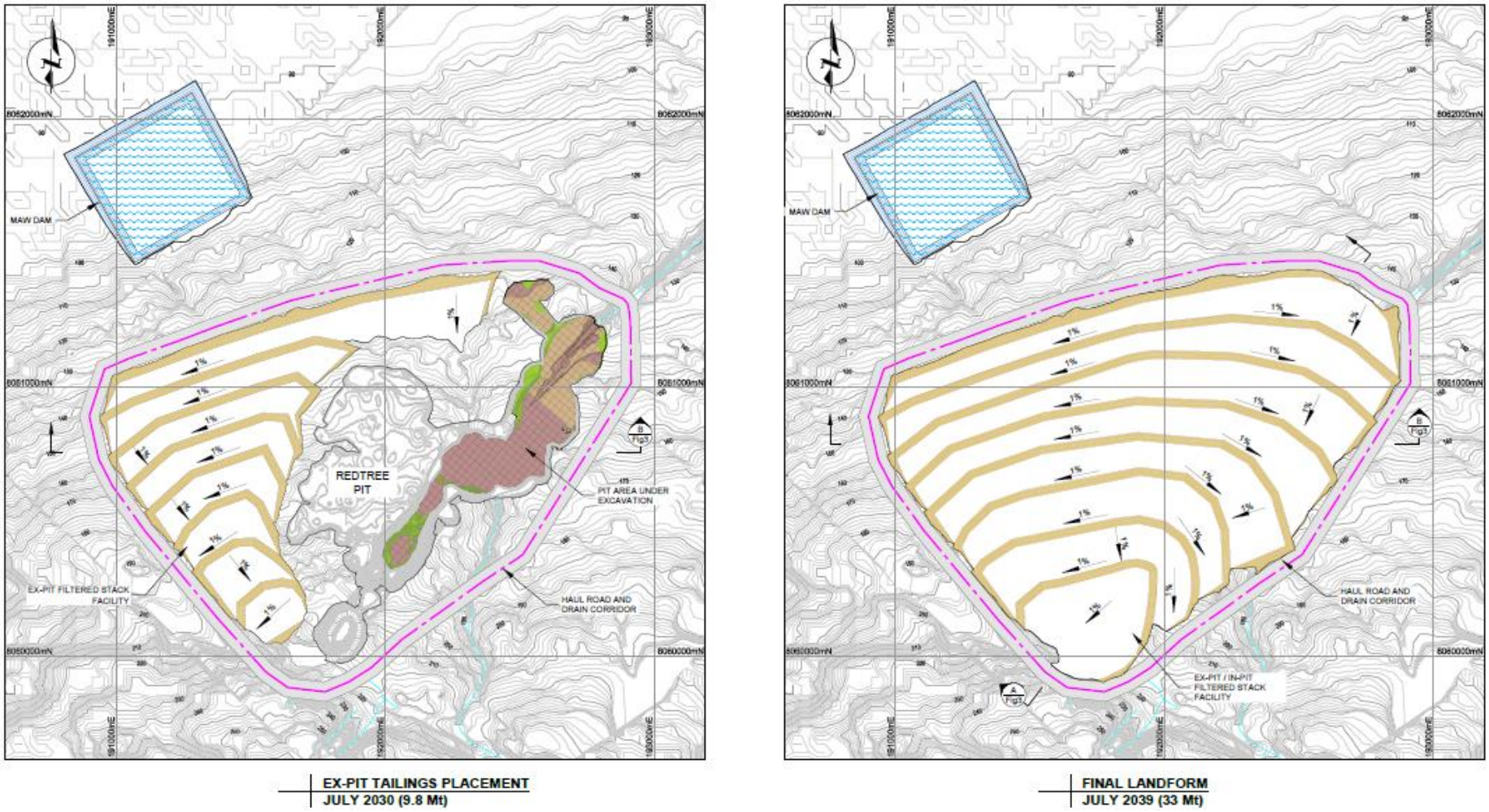
The tailings production schedule is summarized in Table 18.7.1. The facility operates over approximately 12.5 years, producing approximately 33.1 Mt of filtered tailings at a total placed volume of approximately 20.7 Mm^3.

Table 18.7.1 Conceptual Tailings Schedule

Year	Throughput t	Cumulative Throughput t	Cumulative Volume m^3	Placement Location
1	1,100,000	1,100,000	687,500	Ex-Pit
2	2,900,000	4,000,000	2,500,000	Ex-Pit
3	2,900,000	6,900,000	4,312,500	Ex-Pit
4	2,900,000	9,800,000	6,125,000	Ex-Pit
5	2,900,000	12,700,000	7,937,500	In-Pit
6	2,900,000	15,600,000	9,750,000	In-Pit
7	2,900,000	18,500,000	11,562,500	In-Pit
8	2,900,000	21,400,000	13,375,000	In-Pit
9	2,900,000	24,300,000	15,187,500	In-Pit
10	2,900,000	27,200,000	17,000,000	In-Pit
11	2,900,000	30,100,000	18,812,500	In-Pit
12	2,274,853	32,374,853	20,234,283	In-Pit
13	707,059	33,081,912	20,676,195	In-Pit

Figure 18.7.1 below outlines the tailings conceptual plan.

Figure 18.7.1 LOMP Tailings Conceptual Plan



19.0 MARKETING STUDIES AND CONTRACTS

19.1 Introduction

LAM is one of the few advanced junior mining companies well placed to benefit from market requirements for new uranium production in the years ahead. The uranium market has undergone a fundamental structural transformation driven by accelerating global demand for secure, reliable and emissions-free electricity. Nuclear energy has re-emerged as a strategic component of national energy policy as governments seek to improve energy security, reduce greenhouse gas emissions, support industrial electrification and meet rapidly increasing electricity demand associated with artificial intelligence, data centres and advanced manufacturing.

More than 30 countries have announced plans to expand or extend nuclear generation, while numerous existing reactors have received life extensions and previously retired facilities are being restarted. At the same time, significant investment is being directed toward advanced reactor technologies, including small modular reactors (SMRs), creating an additional long-term source of uranium demand.

In parallel, governments have increasingly recognized the importance of rebuilding domestic nuclear fuel supply chains. In the United States (U.S.), federal initiatives supporting uranium mining, conversion, enrichment and fuel fabrication have accelerated materially in recent years. The U.S. Department of Energy's investment in the Cameco-Westinghouse partnership to advance domestic nuclear fuel capabilities demonstrates the strategic importance being placed on securing long-term uranium supply and strengthening western nuclear fuel infrastructure.

Against this backdrop, large, long-life uranium projects located in politically stable jurisdictions are expected to play an increasingly important role in meeting future global uranium requirements.

19.2 Demand

Global nuclear generation is entering a sustained period of expansion. According to the latest figures from the World Nuclear Association (WNA), as of October 2025 there were 438 operable civil nuclear power reactors worldwide, with approximately 397 gigawatt-electric (GWe) of total generating capacity. Nuclear power generated a record 2,667 terawatt-hour (TWh) of electricity in 2024, accounting for approximately 9% of global electricity generation. Nuclear reactors provide continuous, reliable, and low-carbon baseload electricity and remain an important component of the electricity mix in more than 30 countries.

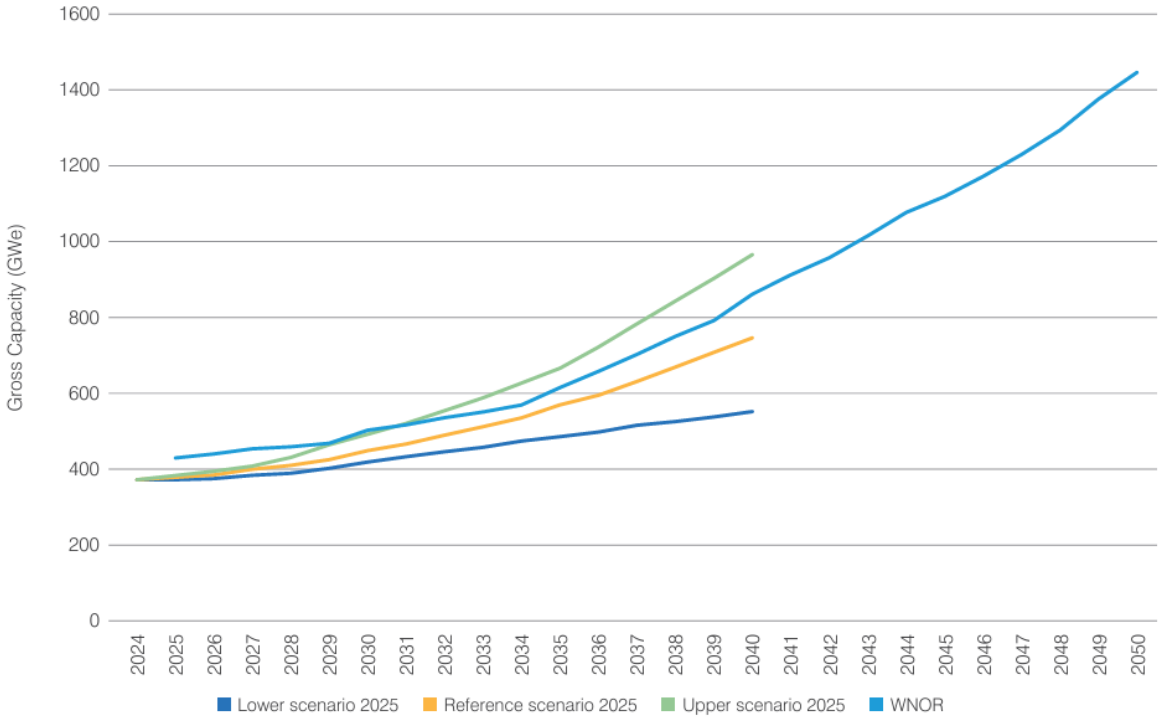
Global nuclear capacity is now entering a renewed period of expansion. WNA's latest assessment identifies approximately 76 GWe of nuclear capacity currently under construction, together with approximately 104 GWe classified as planned and a further 298 GWe as proposed. China is the principal driver of this expansion, accounting for approximately 48 GWe of the 104 GWe of planned capacity globally and 180 GWe of the 298 GWe of proposed capacity. WNA's 2026 World Nuclear Outlook indicates that global nuclear capacity could reach approximately 1,446 GWe by 2050 if announced national government targets are achieved, exceeding the roughly 1,200 GWe implied by the international goal to triple nuclear generating capacity by 2050.

While the Fukushima Daiichi accident in 2011 represented a major setback for the Japanese nuclear industry, Japan has progressively returned nuclear generation to its energy mix. Fourteen Japanese reactors had restarted by October 2025, while a further 19 operable units had yet to receive permission to restart. Japan's 2030 policy continues to target nuclear power supplying 20 to 22% of electricity, while the government's Seventh Strategic Energy Plan, approved in 2025, envisages nuclear supplying approximately 20% of electricity in 2040. This represents a significant reversal of the post-Fukushima retreat from nuclear power, and reflects renewed emphasis on energy security, decarbonization and reduced dependence on imported fossil fuels.

Growth is particularly pronounced in China, which has become the world's most active nuclear construction market. WNA reports 58 operable reactors representing approximately 60 GWe of capacity, with a further 33 reactors totalling approximately 35 GWe under construction. Between 2019 and 2024, approval was granted for 46 new Chinese reactors, with a further 10 reactors approved by the State Council in April 2025. The China Nuclear Energy Association anticipates approximately 110 GWe of nuclear capacity by 2030 and 150 GWe by 2035, while longer-term studies cited by WNA envisage capacity reaching approximately 335 GWe by 2050.

India is also pursuing a major expansion of nuclear power. WNA reports 24 operable reactors with approximately 7.9 GWe of capacity and six reactors totalling approximately 5.2 GWe under construction. India expects nuclear capacity to reach approximately 22.5 GWe by 2032 and, under the Nuclear Energy Mission for Viksit Bharat announced in 2025, is targeting 100 GWe of nuclear generating capacity by 2047. In addition to projects already under construction, pre-construction activities are underway for a further 10 units and the government has granted in-principle approval for approximately 32 GWe of additional capacity.

Figure 19.2.1 World Nuclear Fuel Report Generation Capacity Scenarios Compared to WNOR



This expansion has significant implications for uranium demand. WNA estimates that the global nuclear fleet presently requires approximately 70,000 tonnes of uranium (tU) annually. Under the Upper Scenario in the World Nuclear Fuel Report 2025, annual uranium requirements could approach 200,000 tU by 2040 to support approximately 966 GWe of nuclear capacity. Were global nuclear capacity to reach approximately 1,200 GWe, WNA estimates that annual uranium requirements could increase to approximately 250,000 tU, assuming no substantial change in the global reactor technology mix.

The longer-term outlook therefore points to sustained growth in both nuclear generating capacity and nuclear fuel demand. Expansion of the existing reactor fleet, life extensions, reactor restarts, continued large-scale construction programmes in China and India, and the emergence of new nuclear programmes and small modular reactors are collectively expected to increase requirements for uranium, conversion, enrichment and fuel fabrication services. Against this background, securing reliable and geographically diverse sources of uranium supply is becoming an increasingly important component of global energy security.

19.3 Supply

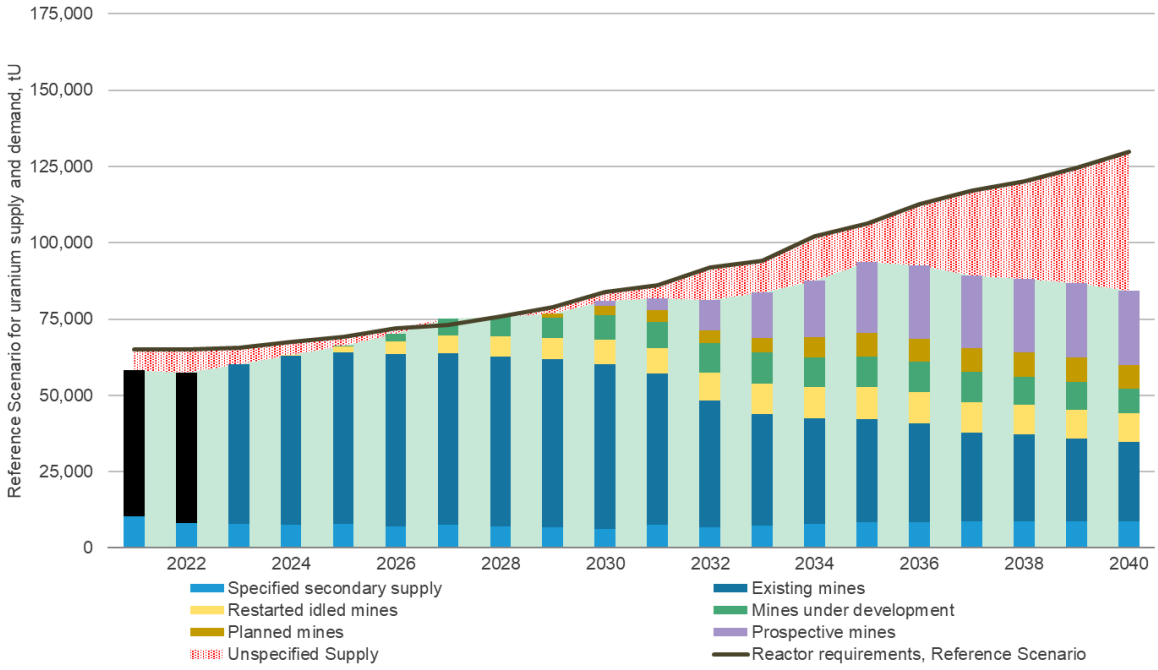
While demand expectations have strengthened considerably, uranium supply remains constrained.

Following more than a decade of depressed uranium prices after Fukushima, global investment in new uranium mine development declined significantly. As a result, relatively few greenfield uranium projects have advanced into production despite the improving demand outlook.

Current annual uranium requirements for the global operating reactor fleet are approximately 70,000 tUpa, compared with primary uranium mine production of approximately 60,000 tU/y. WNA reports that global mine production reached 60,213 tU in 2024, equivalent to approximately 90% of world reactor requirements, compared with 54,433 tU in 2023 and a recent low of 47,805 tU in 2021. The balance between reactor requirements and primary mine supply continues to be met from secondary sources, including commercial and government inventories, recycled uranium and plutonium, underfeeding of enrichment plants and, to a lesser extent, re-enrichment of depleted uranium tails.

The uranium market has historically experienced long periods in which mine production was either above or below reactor requirements. Large inventories accumulated during the early development of the nuclear industry, particularly from military programmes and periods of over-production, subsequently allowing reactor demand to exceed primary mine production for extended periods. These inventories and other secondary sources played an important role in suppressing the requirement for new mine production for several decades. More recently, however, inventories have been drawn down while nuclear generating capacity and utility contracting requirements have begun to increase.

Figure 19.3.1 Reference Scenarios for Uranium Production and Requirements



Source: World Nuclear Association, Uranium Markets / The Nuclear Fuel Report, 2025.

Uranium production remains highly concentrated geographically. In 2024, Kazakhstan accounted for approximately 39% of global mine production, Canada 24% and Namibia 12%, meaning that these three countries collectively supplied approximately three-quarters of the world's mined uranium. Kazakhstan produced approximately 23,270 tU, Canada 14,309 tU, Namibia 7,333 tU, and Australia 4,598 tU during the year. Production is also concentrated at the corporate level, with Kazatomprom, Cameco and Orano accounting for approximately 21%, 17% and 11% respectively of global uranium mine production in 2024.

The uranium supply outlook has changed significantly from that anticipated in the middle of the last decade. Higher uranium prices and renewed utility contracting have encouraged the restart and expansion of previously suspended operations, including major production increases in Canada and Kazakhstan, while a number of new projects are progressing towards development. Nevertheless, WNA emphasizes that new uranium mines will be required both to meet additional demand from reactor growth and to replace depletion from existing operations. Development timelines for new uranium mines can be lengthy because deposits must be discovered, defined, permitted, financed and supported by appropriate processing and infrastructure before commercial production can commence.

Secondary supplies remain an important component of the uranium market, although their nature has changed materially. The former U.S.-Russian 'Megatons to Megawatts' programme, which converted Russian weapons-grade highly enriched uranium into reactor fuel, ended in 2013 and at its peak had supplied a substantial proportion of global reactor requirements. WNA estimates that worldwide conversion of military highly enriched uranium supplied approximately 15% of reactor requirements prior to the programme ending. Current secondary sources are instead dominated by commercial and government stockpiles, recycled materials and enrichment-related sources.

WNA's World Nuclear Fuel Report 2025 estimates that, at the end of 2024, uranium inventories amounted to approximately 42,000 tU in the U.S., 40,000 tU in the European Union, and 65,000 tU in East Asia. Recycled uranium and plutonium are estimated to displace approximately 2,000 tU of primary uranium supply annually, while enrichment underfeeding and re-enrichment of depleted uranium can provide additional uranium-equivalent supply depending on enrichment capacity, tails assays and market economics.

The longer-term supply challenge is substantially greater. WNA estimates that approximately 70,000 tU/y is required to fuel the current global nuclear fleet. Under the Upper Scenario in *the World Nuclear Fuel Report 2025*, annual uranium requirements could increase to almost 200,000 tU/y by 2040, supporting approximately 966 GWe of nuclear generating capacity. If global nuclear capacity reaches approximately 1,200 GWe, consistent with the international objective of tripling nuclear capacity, WNA estimates uranium requirements could reach approximately 250,000 tU/y. Against current primary production of around 60,000 tU/y, this illustrates the scale of additional uranium production that may ultimately be required.

WNA considers the global uranium resource base sufficient to support significant nuclear growth, but stresses that the principal challenge is converting resources in the ground into timely commercial production. Identified recoverable uranium resources below US$130 /kg U total approximately 5.93 million tonnes of uranium (MtU) globally, with substantially greater resources available at higher cost categories. The ability of these resources to contribute to future supply will depend on uranium prices, permitting, financing, infrastructure development and the lead times required to construct new mines.

The expected expansion of nuclear generation over the coming decades therefore represents a significant structural increase in uranium demand. Existing mines, restarts and brownfield expansions can meet part of this requirement, but substantial new primary mine capacity will be required if the nuclear industry's current growth ambitions are realized. WNA's outlook indicates that uranium supply will need to grow materially from present levels, both to replace depletion of existing mines and to support new reactor capacity. This is likely to require sustained investment in uranium exploration and mine development and uranium prices capable of supporting the development of new, economically viable sources of supply.

19.4 Pricing

Unlike most commodities, uranium is predominantly marketed through long-term contracts negotiated directly between uranium producers and nuclear utilities. These contracts typically extend for periods of five to 10 years and provide both utilities and producers with long-term supply and pricing certainty.

Spot market transactions represent only a relatively small portion of total uranium purchasing activity and therefore do not provide the primary basis upon which new uranium projects are financed or developed. Instead, project economics are generally supported by long-term contract pricing negotiated in advance of production.

During 2026, published long-term uranium prices have continued to strengthen, reaching approximately US$94 /lb (UxC) and US$97 /lb (TradeTech), representing the highest reported term prices in approximately 18 years. In addition, recent disclosures by industry participants indicate that negotiated commercial contract values may exceed published term price indicators, reflecting increasing competition among utilities seeking to secure future uranium supply.

Although utility contracting activity varies from year to year, the industry continues to exhibit a substantial volume of uncovered long-term uranium requirements extending into the 2030s. As existing contracts mature and new nuclear capacity enters operation, utilities are expected to continue re-entering the term market to secure future fuel requirements.

The uranium price assumptions adopted in this PEA are intended to reflect long-term market conditions over the expected operating life of the Project rather than short-term fluctuations in the spot market. The selected pricing assumptions are considered consistent with current long-term market indicators, published industry forecasts and the incentive pricing required to support development of new primary uranium production.

LAM intends to market production from the Project primarily through a portfolio of long-term sales agreements with major nuclear utilities, utilizing a combination of fixed-price and market-related contracts. Australia maintains one of the world's most rigorous uranium export control regimes, with all uranium exports subject to International Atomic Energy Agency safeguards and bilateral nuclear cooperation agreements, ensuring that production is supplied exclusively for peaceful nuclear energy purposes.

19.5 Strategic Importance of Uranium

Over the past decade, uranium has evolved from a commodity primarily influenced by cyclical supply and demand dynamics into a strategically important material underpinning national energy security, economic competitiveness, and decarbonization objectives. This transformation has been driven by the convergence of rapidly growing electricity demand, renewed global investment in nuclear power, and increasing government recognition that secure domestic nuclear fuel supply chains are critical strategic assets.

Recent advances in artificial intelligence, cloud computing and hyperscale data centres have significantly increased long-term electricity demand forecasts, placing renewed emphasis on reliable sources of continuous, carbon-free baseload generation. Unlike intermittent renewable energy sources, nuclear power provides high-capacity, dispatchable electricity capable of supporting the growing energy requirements of digital infrastructure and industrial electrification.

Recognizing these trends, governments around the world have adopted policies intended to accelerate nuclear deployment while reducing reliance on foreign-controlled fuel supply chains. In the U.S., a series of Executive Orders and federal initiatives have identified nuclear energy as a strategic priority, supporting expansion of domestic uranium production, conversion, enrichment, and fuel fabrication capabilities. These initiatives are intended to strengthen long-term energy security while reducing dependence on geopolitical competitors for critical nuclear fuel services.

This strategic policy direction has been reinforced through direct federal investment across the nuclear fuel cycle. Notably, the U.S. Department of Energy has provided support for the Cameco–Westinghouse partnership as part of broader efforts to expand domestic nuclear fuel capabilities and strengthen the resilience of the North American nuclear supply chain. Such investments demonstrate increasing government recognition that uranium production and fuel-cycle infrastructure represent essential components of national critical infrastructure.

At the same time, geopolitical developments have highlighted the concentration of global uranium production and fuel processing capacity. Kazakhstan remains the world's largest uranium producer, while conversion and enrichment services remain concentrated in a limited number of jurisdictions. Ongoing sanctions affecting Russian nuclear fuel services, combined with broader geopolitical uncertainty, have accelerated efforts by utilities and governments to diversify sources of uranium supply and reduce dependence on politically sensitive regions.

The U.S. has also established strategic programs intended to encourage domestic uranium production, including initiatives to rebuild strategic uranium inventories and support the development of a secure domestic nuclear fuel supply chain. Similar policies are being pursued across other western jurisdictions as governments seek to ensure long-term access to uranium produced under stable regulatory regimes and high environmental standards.

Collectively, these developments have fundamentally altered the long-term outlook for the uranium market. Rather than being viewed solely as a commodity, uranium is increasingly recognized as a strategic resource essential to energy security, technological leadership and economic resilience. This evolving policy environment is expected to support continued investment in new primary uranium production, particularly from politically stable jurisdictions such as Australia, which possesses some of the world's largest undeveloped uranium resources.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Note that the current QLD government has a policy not to approve uranium mining projects. However, during the previous Liberal National Party (LNP) government of 2012 to 2015, a process preparing for the recommencement of uranium mining was undertaken. During this time, a Uranium Mining Implementation Committee was established to examine and recommend a best practice framework for the recommencement of uranium mining in QLD. Uranium mining has taken place historically in Queensland and it is reasonable to expect that policy may change in the future.

20.1 Environmental Considerations

The Project is situated in a sparsely populated region of northern Australia, straddling the NT and QLD border. The region's population is supported primarily by pastoralism, mining, commercial fishing and tourism, and consists of a high proportion of Aboriginal peoples. A few small towns (i.e. populations <3,000 people) are located within a few hundred kilometres of the Project area; the closest town is the Aboriginal community of Doomadgee, with an approximate population of 1,460 people, which is located roughly 80 km to the east. The majority of the land tenure around the Project is leasehold with minor freehold properties and Aboriginal Freehold, which is held by Aboriginal Land Trusts both in QLD and in the NT.

The Project inhabits a region with a marked wet and dry season that is subject to monsoonal conditions and occasional cyclonic activity. The average wet season (November to March) rainfall is in the order of 172 mm/month, while the average dry season rainfall (April to October) is in the order of 10 mm/month. High rainfall intensities and durations do occur during cyclonic conditions. Mean wet season maximum temperature is 33°C, with no significant variation in dry season maximums.

The Project area spans two rainfall catchments, namely the Lagoon Creek Catchment and the Nicholson Catchment. No flood mapping is available for those areas of the catchments.

Groundwater occurs at relatively shallow depth below surface in the Project area (from 7 m to standing water level (SWL)). However, depth to groundwater is expected to vary with seasonal rainfall. As 88% of the rainfall occurs in the wet season, significant recharge is expected over that period. During the dry season, little to no recharge is expected and groundwater levels are expected to recede. SWL variations of 5 m or more may be expected in some areas.

In the broader area, there are a wide range of aquifer types from high-yield, high-potential aquifers that are part of the Great Artesian Basin (GAB), to low-potential, local aquifers such as those associated with river alluvium or fractured rocks. There are nine groundwater bores in the immediate Project area and 72 in the wider area.

The soils in the Project area are mostly skeletal or shallow sands. These support native woodland vegetation with a spinifex and tussock grass understorey. Isolated patches of monsoon rainforests occur in gorges, with riparian vegetation along the rivers. This vegetation is typical of the broader region.

Preliminary environmental studies have been completed in the area, acknowledging the presence of environmental constraints such as threatened species and ecosystems on both local and regional scales. Further detailed ecological and environmental studies will be undertaken to identify and manage significant species populations and their habitats. Any potential impacts will be addressed through the relevant planning and environmental approval processes, including provisions for ongoing monitoring and management as required.

Roads of varying standards service the wider region in which the Project is situated. During the wet season (November to March), all major roads are closed for various amounts of time due to impassable river crossings. No rail lines currently service the area.

Two designated gulf ports lie within approximately 200 km of the Project area.

This Project will require a range of permits, licences and development applications covering the development in QLD, as well as approvals under Commonwealth legislation.

20.2 Social and Community Considerations

There is one native title determination over the Project area. This is a determination that native title exists in the area, and the native title holders are the Gangalidda and the Garawa peoples. Right to negotiate agreements were entered into with the Gangalidda and the Garawa peoples (who were then registered native title claimants) at the time of the grant of the exploration permits. LAM recognizes that the Gangalidda and the Garawa peoples native title holders are key stakeholders in the Project and, as part of the process towards a commencement of a mining operation, agreements will need to be negotiated under the Native Title Act. These agreements will provide for consents to mining grants and activities in return for commercial consideration, including training and employment of indigenous personnel.

There are also 51 registered cultural heritage sites registered within the Project area. To meet the cultural heritage duty of care in relation to these and other sites in the Project area, and in accordance with the Environmental Impact Assessment (EIA) process, a Cultural Heritage Management Plan (CHMP), or its equivalent as part of any native title agreement, will be sought to be developed with the Gangalidda and the Garawa peoples native title holders.

As part of the process working towards a mining operation, LAM intends to undertake a social impact assessment. A community consultation program will gather information and views from parties in the region who may be impacted by the operation. The assessment of these impacts is to identify possible beneficial and adverse impacts.

Consideration is to be given to the following:

- The impact of the project on existing pastoral land uses and land holders.
- Any potential impacts on the surrounding community.

- The potential and mechanisms for local and statewide communities and businesses to tender contracts for services and supplies for any relevant components of the construction and operation of the project.

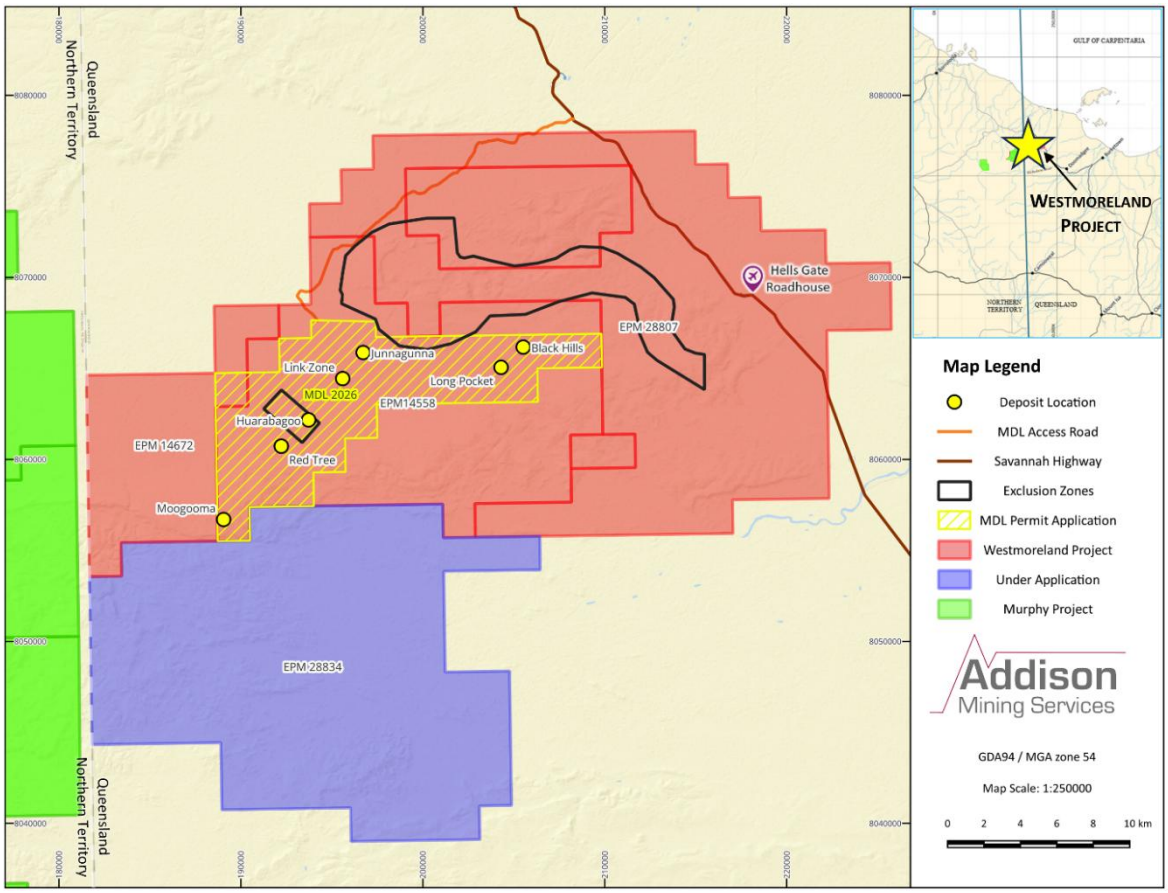
- The potential positive and negative social impacts that could result from an increased population.

- Impact on services or other development projects in the region that have relevance to this proposed operation.

The entering of an ILUA with the Gangalidda and Garrwa people in November 2022 has provided the company with a pathway for operating under our EPMs and recently applied for MDL but, importantly, recognizes the need for the negotiation of a ML ILUA for the Project to proceed.

An activity exclusion zone exists at the southern end of the HB deposit which will require further negotiation for future access and exploration activities.

Figure 20.2.1 Westmoreland Project Outline with Exclusion Zones



20.3 Occupational Health and Safety (OH&S)

20.3.1 Commitment to Health and Safety

LAM is dedicated to designing, constructing, and operating a project where health and safety (H&S) are paramount, ensuring a safe workplace for all.

Safety has been a fundamental consideration in the various project stages, with a focus on eliminating or minimizing risks to personnel. Comprehensive systems have been developed to support safe construction and operation, applying a risk management approach to ensure targeted controls where engineering solutions may be less effective.

LAM's H&S systems meet or exceed the Work Health and Safety Act (2011) and Work Health and Safety Regulations (2011), which serve as the minimum standard. In alignment with ISO 45000, LAM has implemented a Safety and Health Management System (SHMS) that identifies and controls principal mining hazards, particularly those with the potential for multiple fatalities or serious incidents.

Worker involvement is integral to LAM's H&S programs, ensuring collaborative design, implementation, and continuous improvement. Clear reporting mechanisms are in place to monitor performance, enhance accountability, and drive ongoing improvements.

20.3.2 Project Risk Review and Management

A comprehensive Project risk review will be conducted, incorporating new insights from mine planning, plant design, community consultation, and permitting progress. These assessments will follow a process map approach in compliance with AS 31000 and applied with the likelihood and consequence descriptors alongside a risk matrix. Risk ratings will be determined based on design outcomes, integrating risk reduction principles during the feasibility stage.

The scope of these assessments will be focused solely on the operational aspects of the Project and will not include financial risks such as foreign exchange fluctuations, commodity pricing, or construction costs. As the Project advances, LAM will conduct additional risk reviews in collaboration with contracting partners and specialist consultants. The risk register is a dynamic document that will be continuously reviewed and updated throughout the study, design, and construction phases. It will also play a crucial role in informing mine and plant design, as well as overall site layout, to minimize risk and embed a culture of 'safety in design'.

During core cutting, personnel near the core saw wear appropriate PPE. All staff and contractors on the Project wear personal radiation monitoring badges optically stimulated luminescence (OSL) dosimeters in accordance with Section 38 of the Queensland Radiation Safety Act 1999. LAM uses ARPANSA-provided sealed TLDs suitable for dusty conditions. The maximum annual dose for radiation workers is 20 millisievert (mSv), though actual doses are typically much lower, often below the minimum reportable dose (0.05 mSv).

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

The overall study capital cost estimate was compiled by Lycopodium and is presented here in summary format. The capital cost estimate reflects the Project scope as described in this study report. Mine capital costs (developed by Mining Associates for LAM) and are included in the estimate tables below.

All costs are expressed in A$ unless otherwise stated and based on 3Q26 pricing. The estimate is deemed to have an accuracy of ±35%.

The various elements of the Project estimate have been subject to internal peer review by Lycopodium and have been reviewed with LAM for scope and accuracy.

21.1.1 Summary

The capital estimate is summarized in Table 21.1.1 by area. The pre-production capital cost is estimated at A$646M (US$452M) excluding contingency.

The estimate work breakdown structure (WBS) is based on the standard Lycopodium WBS.

Table 21.1.1 Capital Estimate Summary by Area (3Q26, ±35%)

Main Area		A$
0	Construction Indirects	43,565,778
1	Treatment Plant Costs	245,078,075
2	Reagents and Plant Services	97,684,726
3	Infrastructure	58,810,692
5	Management	60,538,940
6	Owners Project Costs (excl. Mining)	140,597,759
Subtotal		**646,275,971**
Contingency		119,913,227
Fees, Taxes and Duties		-
Escalation		-
Grand Total		**766,189,198**

21.1.2 Estimating Currency and Base Date

The estimate is expressed in A$ based on prices and market conditions current at 3Q26. An A$ to US$ exchange rate of A$1.00 = US$0.70 is assumed for the report conversion to US$.

21.1.3 Mining Capital Costs

Mining capital costs were prepared by Mining Associates under contract to LAM on the basis of a contractor mining strategy. A factored estimate of A$5.8M was used for the mining related capital expenditure. A summary of mining costs included in the capital cost estimate is provided in Table 21.1.2 below.

Table 21.1.2 Mining Capital Costs

Mining Capex	A$
Office	250,000
Workshop	1,400,000
Fuel and Oil Farm (Go Line)	1,000,000
Washdown Area	500,000
Magazine	200,000
Power, Water, Sewer, Comms	150,000
Miscellaneous	500,000
Total	**5,800,000**

21.1.4 Infrastructure Capital Costs

Water Supply

GWS identified two borefield options for the supply of the Project's raw water requirement, via supply from the GAB aquifer, and supply from nearby local bore water sources.

Electric Power Supply

A power options assessment was completed as part of the study investigating three power supply options for the Project. The study compared a standalone diesel power station, a hybrid diesel with solar PV and BESS, and a grid connection from the Mount Isa isolated 220 kV network. The analysis assessed three RF scenarios (20%, 40% and 60%) for the hybrid options. The 40% RF hybrid option was selected as the preferred development path and include in the study scope, modelled on an Owner-operator basis.

Project Access Roads

An allowance has been made for the construction of 30 km of access road into the mine site and accommodation camp.

Accommodation Camp and Airstrip

Allowance has been made in the capital estimate for accommodation camp to house an onsite workforce of 250 personnel, and an associated airstrip. A fit-for-purpose camp accommodation has been assumed in keeping with the current state of the Australian mining industry.

Administration Mining and Process Buildings, Sewage Treatment

Allowance has been made in the capital estimate for the appropriate inclusion of building infrastructure covering administration, mining process, including change rooms, ablutions, workshops, stores, and reticulation of potable water, electric power, and sewerage.

21.1.5 Estimating Methodology

The estimate has been based on an engineering, procurement and construction management (EPCM) contracting strategy, whereby an Engineer will undertake all detailed design, procurement and onsite construction management acting for and on behalf of LAM.

21.1.6 Estimate Methods of Calculation

Capital costs were compiled using the methods of calculation described in Table 21.1.3.

Table 21.1.3 Capital Cost Methods of Calculation

Discipline	Sub Items	Method
Earthworks	Quantities	Quantities for plant site bulk earthworks have been estimated from the layout.
	Unit Rates	Rates were derived from Lycopodium's recent experience for works of this type in similar locations.
Concrete	Quantities	Quantities for concrete works were established using, plant layout prepared for the study and benchmarked against detailed quantities for similar sized reference projects and/or factored as necessary.
	Unit Rates	Rates and hours were derived from Lycopodium's recent experience for works of this type in similar locations.
Steelwork	Quantities	Quantities for structural steel were established using, plant layout prepared for the study and benchmarked against detailed quantities for similar sized reference projects and/or factored as necessary.
	Supply Rates	Based upon Lycopodium database of supply rates for recent projects. Assumes supply from South East Asian fabricators.
	Install Rates	Rates and hours were derived from Lycopodium's recent experience for works of this type in similar locations.
Platework / Tankage	Quantities	Platework and tankage quantities were determined using the sizing provided in the mechanical equipment list prepared for the study as the basis.
	Supply Rates	Based upon Lycopodium database of supply rates for recent projects. Assumes supply from South East Asian fabricators.
	Install Rates	Rates and hours were derived from Lycopodium's recent experience for works of this type in similar locations.

Discipline	Sub Items	Method
Mechanical Equipment	Supply Costs	The mechanical equipment list prepared for the study providing quantity and sizing. Budget quotations from equipment vendors for key mechanical equipment (eight packages). Costs for all other items were derived from Lycopodium's current in-house database.
	Install Rates	Rates and hours were derived from Lycopodium's recent experience for works of this type in similar locations.
Plant Pipework		The supply and installation estimate for in-plant piping was derived using factors suitable for this project. These factors are a percentage of the mechanical equipment supply and installation costs and are calculated per individual plant area. The plant piping costs allow for the supply and installation of pipe, fittings, mountings, manual valves and actuated valves.
Electrical, Instrumentation		Costs were derived from factors suitable for the Project and based on Lycopodium's in-house database and referenced from recent similar projects.
Architectural / Buildings		Pricing for plant and administration buildings sourced from recent experience for similar Lycopodium projects.
Freight	Total Cost	Factored from supply costs based on Lycopodium database of costs from similar projects.
Import Duty	Total Cost	Taxes and duties have been excluded from the estimate.
EPCM Services	Total Cost	Estimate was factored based upon Lycopodium's recent experience with similar type and size of project.
Owners Costs	Total Cost	The Owners cost estimate includes: • The Owners project management team labor. • Pre-production costs. • First fills (grinding media, lubricants, fuel, and reagents). • Opening stocks. • Plant mobile equipment. • Project spares. • Vendor representative costs for key process plant equipment.

21.1.7 Quality Development

Quantity information was derived from a combination of the following sources:

- Study Engineering: Includes quantities derived from concept or preliminary engineering for the purpose of the study. Includes equipment lists and redline mark-ups of previous project drawings, 3D model and data by engineering.

- Estimated: Includes quantities derived from sketches or redline mark-ups of previous project drawings / data by estimating or similar project go-bys.

- Factored: Quantities derived from percentages applied as a factor derived from previous projects.

21.1.8 Pricing Basis

Estimate pricing was derived from a combination of the following sources:

- Budget Quotation: Budget pricing solicited specifically for the study or project estimate.
- Database: Historical database pricing that is less than six months old.
- Estimated: Historical database pricing older than six months, escalated to the current estimate base date.
- Factored: Factored from costs with a basis.

21.1.9 Installation Costs

This component represents the cost to install the plant equipment and bulk materials onsite or to perform site activities. Installation costs are further divided between direct labor, equipment and construction indirect costs.

The labor component reflects the cost of the direct workforce required to construct the Project scope. The labor cost is the product of the estimated work hours spent on site multiplied by the cost of labor, inclusive of overtime premiums, statutory overheads, payroll burden, and contractor margin.

The equipment component reflects the cost of the construction equipment and running costs required to construct the Project. The equipment cost also includes small tools, consumables, personal PPE, and the applicable contractor margin.

Construction indirect costs encompass the remaining cost of installation and include items such as offsite management, onsite staff and supervision above trade level, crane drivers, equipment and labor mobilization and demobilization, rest and recreation (R&R), meals, and accommodation costs.

The labor gang rates, and equipment costs estimated for each major trade commodity are shown in Table 21.1.4.

Table 21.1.4 Direct Labor Gang Rates

Item	Direct Labor A$	Equipment A$	Total Hourly Rate A$
Earthworks	90.0	160.0	250.0
Concrete Installation	90.0	10.0	100.0
SMP	120.0	52.0	172.0
E&I	120.0	52.0	172.0

21.1.10 Freight

The freight estimate was derived from the following:

- By applying container rates to bulk items.
- By calculation as a percentage of supply costs for items.
- By benchmarking against similar projects located in similar regions.

21.1.11 Field Indirects

Establishment of project construction offices and workshops commensurate with the scope of the process plant construction scope are included in the capital estimate.

Construction services such as power, water, fuel and consumables are included.

Construction indirect costs for all direct labor is included in the installation rates for all works in the capital estimate. This is inclusive of PPE, travel, and clothing. Earthworks rates are inclusive of fuel, maintenance and running costs of machinery.

Construction mobile equipment and project cranes are included in the capital cost estimate.

21.1.12 EPCM Services

The EPCM services cost estimate allows for the cost of home office and site support staffing, subconsultants, office consumables, equipment, and associated project travel. The EPCM costs for the process plant component of the Project represents approximately 15.7% of the direct costs (excluding Owner's costs and management costs) of the process plant facility.

21.1.13 Owner's Costs

Included as Owners cost discipline in the estimate are the following:

- Owners project management team costs.
- First fills (grinding media, lubricants, fuel, and reagents).
- Opening stocks.
- Specialist vendor representatives expected to be required to assist with construction verification and commissioning.
- Process plant mobile equipment.
- Plant spares.

21.1.14 Spares

Spares have been calculated as a percentage of mechanical equipment supply costs.

21.1.15 Duties and Taxes

The capital estimate excludes all government taxes and duties.

21.1.16 Project Insurances

Project insurances have been excluded from the estimate.

21.1.17 Contingency

An amount of contingency has been provided in the estimate to cover anticipated variances between the specific items allowed in the estimate and the final total installed project cost. The contingency does not cover scope changes, design growth, etc., or the listed qualifications and exclusions.

It should be noted that contingency is not a function of the specified estimate accuracy, and such accuracy should be measured against the project total that includes contingency.

A deterministic contingency analysis has been applied to the estimate that considers scope definition, materials / equipment pricing and installation costs. The resultant project contingency for the Project is 18.5%.

21.1.18 Escalation

No allowance has been made for escalation to account for either:

- A project commencement date beyond the base date of 3Q26; or
- The time between project commencement and the date at which commitments will actually be incurred and payments made.

21.2 Sustaining Capital

Sustaining capital costs are included in the overall project economic evaluation. Sustaining capital costs for the LOM are estimated at A$119.1M (US$84M) and include:

- Tailings dam construction and operations costs.
- Process plant sustaining capital costs.

Tailings dam construction costs are further detailed in Table 21.2.1 below.

Table 21.2.1 LOMP Tailings Cost Estimate

Item	Description	Cost A$
1.0	Preliminary Construction Works	9,361,307
2.0	Filtered Tailings Stack	37,705,084
3.0	Surface Water Management	8,939,913
4.0	Progressive Closure and Rehabilitation	20,931,750
5.0	Contingency (25%)	21,021,618
Total Project Cost		**105,108,088**

21.3 Capital Cost Exclusions and Qualifications

The following is excluded from the Project capital costs:

- Pre-production labor costs.
- Working capital.
- Project insurances.
- Duties / taxes / fees.
- Project sunk costs.
- Project escalation.
- Closure costs.
- Permits and licences.
- Land purchase and relocation costs.
- Exchange rate variations.

21.4 Operating Costs

21.4.1 Mining Costs

Mining Associates prepared the mining operating cost for the Project based on a contractor mining strategy. Table 21.4.1 shows the cost of the contractor mining strategy based over the LOM, at a 2.9 Mtpa processing rate.

Table 21.4.1 Westmoreland Mine Operating Cost Summary

Mining Unit Cost	Units	Value
Mining Load and Haul	A$ /t TMM	$4.46
Mining D&B Oxide	A$ /t Oxide	$0.35
Mining D&B Fresh	A$ /t Fresh	$0.86
Mining Grade Control	A$ /t Production Target	$0.66
Diesel Cost	A$ /L	$1.35
Total LOMP Mining Cost	**A$...M**	**$782.6**
Total Material Movement	**Mt TMM**	**146.6**
Total Mining Unit Cost	**A$/t TMM**	**$5.34**

21.4.2 Plant and Administration Costs

Process plant and administration operating costs have been developed by Lycopodium based on a design treatment rate of 2.9 Mtpa of ore with the plant operating 24 h/d, 365 d/y with a 91.3% plant utilization (nominal 8,000 h/y) and a P_{80} grind size of 180 µm.

The operating cost estimate has been compiled from a variety of sources and is based on whole of ore treatment and a head grade of 788 ppm U_3O_8. Operating costs are presented in A$ and are based on prices obtained during 2Q26, to an accuracy of ±35%. The process plant operating costs for the facilities are summarized in Table 21.4.2.

Table 21.4.2 Westmoreland Process Plant Operating Cost Summary

Cost Centre	A$ /y	A$/t Ore	US$ /lb U_3O_8
Process Plant			
Power	21,039,969	7.26	3.08
Operating Consumables	81,015,976	27.94	11.85
Maintenance	7,168,330	2.47	1.05
Labor – Process, Maintenance and Laboratory	12,676,636	4.37	1.85
Laboratory	1,453,030	0.50	0.21
Total Processing	**123,353,941**	**42.54**	**18.04**
Labor – G&A	6,000,050	2.07	0.88
General & Administration Costs	14,419,041	4.97	2.11
Total G&A	**20,419,091**	**7.04**	**2.99**
Total Processing incl. G&A	**143,773,032**	**49.58**	**21.03**

21.4.3 Power

The power requirement for the process plant is based on the mechanical equipment list (MEL), adjusted for equipment load factor and utilization. The unit cost of supplying electricity to the Project is estimated at A$0.21 /kWh.

A summary of the power cost for the plant-by-plant area is tabulated below in Table 21.4.3.

Table 21.4.3 Westmoreland Process Plant Power Cost by Plant Area

Plant Area	A$ /y	A$ /t
Primary Crushing and Surge Bin	457,659	0.16
Mill Feed	193,301	0.07
Grinding – SAG Mill	14,427,830	4.98
Grinding Area (Excluding Mill)	1,153,830	0.40
Pebble Area Crushing	294,832	0.10
Pre-Leach Thickening and Leaching Tanks	1,007,423	0.35
Filtration – Leach Discharge	1,000,395	0.34
Tailings Conveyors and Reclaim	665,650	0.23
CIX	399,289	0.14
UOC Refinery	637,004	0.22
Reagents	1,133,161	0.39
Services - Water	540,543	0.19
Services - Air	108,993	0.04
Services - Other	171,978	0.06
Plant Buildings	908,808	0.31
Camp	883,008	0.30
Turbo-Alternator (Acid Plant Energy Recovery)	-2,943,360	-1.01
Total	**21,039,969**	**7.26**

21.4.4 Operating Consumables

Consumable requirements for the Project process plant were based on testwork consumption rates, reference plants, and industry standards. Budget quotations for reagents and consumables were received from suppliers and adjusted to a delivered at place (DAP) price and include customs and duties.

Diesel unit costs were supplied by LAM and diesel consumption for the plant mobile equipment was estimated.

The consumables cost by plant area is summarized below in Table 21.4.4.

Table 21.4.4 Westmoreland Process Plant Consumables Cost

Plant Area	A$ /y	A$ /t
Primary Jaw Crusher – Jaw liners	170,949	0.06
SAG Mill – Liners	2,605,059	0.90
SAG Mill – Grinding Media	4,833,198	1.67
Pebble Crusher – Mantle / Concave	76,311	0.03
Sulfur	15,738,028	5.43
Pyrolusite	14,848,000	5.12
Hydrogen Peroxide	1,608,033	0.55
IX Resin	376,924	0.13
Sodium Hydroxide (+99%)	34,335,850	11.84
Flocculants (Type 1) – Thickening / Filtration	860,929	0.30
Flocculant (Type 2) – UOC Refinery	3,352	0.001
Cooling Water Treatment	174,964	0.06
BFW and Steam Treatment	174,964	0.06
Mill Lubricants	104,978	0.04
General Supplies	26,245	0.01
Operator Consumables	15,540	0.01
Product Packaging Drums	2,554,286	0.88
Product Containers, Strapping and Freight	558,750	0.19
Diesel – Borefield Water Pumps	1,040,000	0.36
Diesel – Mobile Equipment	909,617	0.31
Total	**81,015,976**	**27.94**

21.4.5 Labor (Processing / Maintenance and Administration)

The labor costs for the process plant and administration are summarized in Table 21.4.5. Labor rates were based on the 2025 Hays Salary Guide and reference projects. Mining salary costs were included in the mine operating cost provided by Mining Associates. The labor rates are based on a skill level and consist of a base salary and the required overhead allowances.

Table 21.4.5 Westmoreland Process Plant and Administration Manning Levels

	People	Total Labor Cost A$ /y
Administration	36	6,000,050
Operations and Laboratory	51	8,173,538
Maintenance	24	4,503,099
Total	**111**	**18,676,686**

The roster is based on a three-panel roster for shift personnel and 12 h per shift and salaries take into account the Project FIFO requirements.

21.4.6 General and Administration (Excluding G&A Labor)

General and administration (G&A) costs were developed by Lycopodium based upon similar reference projects and are summarized in Table 21.4.6.

Table 21.4.6 Westmoreland Plant General and Administration Summary

General and Administration	A$ /y
Telecommunications	100,000
Insurances	1,363,000
Stationery	21,000
Office Cleaning	140,000
Postage, Courier and Light Freight	21,000
Computer Supplies and Support	70,000
First Aid Costs	35,375
Entertainment	35,000
Consultants and Vendors	189,000
Banking Fees	14,000
Radiation Protection and Monitoring	50,000
Safety, Clothing	51,000
Training Budget	200,000
Travel and Accommodation	30,000
DEP/ Environmental License	15,000
Community Relations Expenses	150,000
Messing and Accommodation	5,511,166
FIFO	6,602,900
Miscellaneous	20,000
Total	**14,419,041**

21.4.7 Maintenance

The maintenance cost for the Project processing plant was factored from the equipment supply capital cost and is summarized in Table 21.4.7. Allowances for plant mobile equipment, and general maintenance have been made.

Table 21.4.7 Westmoreland Plant Annual Maintenance Materials Cost

	A$ /y	A$ /t
Crushing, Surge Bin and Reclaim	346,614	0.12
Milling	1,125,497	0.39
Pebble Area Crushing	98,409	0.03
Thickening and Leaching	229,951	0.08
Filtration and TSF Conveying	710,574	0.25
CIX	40,280	0.01
UOC Refinery	516,105	0.18
Reagents – Acid Plant	1,679,454	0.58
Reagents - Other	56,675	0.02
Plant Services (Water, Air, etc.)	94,496	0.03
Infrastructure, Tailings	22,064	0.01
Buildings	119,532	0.04
Borefield	703,359	0.23
Camp, Camp Services incl. Airport	665,638	0.23
Maintenance General	256,650	0.09
Contract Labor	501,032	0.17
Total	**7,168,330**	**4.37**

21.4.8 Exclusions

The operating costs stated above makes no allowance for the following:

- All LAM head office / corporate costs.
- All import duties, withholding taxes and other taxes (corporate tax included in the financial model).
- Project financing costs.
- Any impact of foreign exchange rate fluctuations (included in the financial model under sensitivity analysis).
- Any escalation from the date of the estimate.
- Any contingency allowance.
- Any rehabilitation or closure costs (included in the financial model).
- Any licence fees or royalties (included in the financial model).

- Government monitoring / compliance costs.
- Water extraction licensing costs (pumping power and installation has been allowed).
- Transport from Townsville to buyer.

22.0 ECONOMIC ANALYSIS

22.1 Introduction

A financial model for evaluating the Project was developed by Mining Associates (MA). Lycopodium reviewed the model logic, consistency of input assumptions, and integrity of the calculations. All costs are constant in 2026 Australian dollars with no provision for inflation escalation.

The annual cash flow projections were estimated over the Project's production life based on production schedule, sales revenue, production costs, capital expenditures, and corporate costs (taxation, royalties, etc.). The financial indicators examined included pre-tax cash flow, ATCF, pre-tax NPV and after-tax net present value at a 7.5% discount rate, IRR, and payback period.

22.1.1 Principal Assumptions

The principal assumptions are detailed in this section.

The proposed mining inventory was based on the production target and mine schedule described in Sections 15 and 16 of this report. The annual throughput is at a rate of 2.9 Mtpa.

22.1.2 Uranium Sale Price

MA used a selling price of US$90 /lb in the economic model.

Prices used are non-escalated real prices. Sensitivity analysis demonstrates financial returns for the Project at a range of uranium prices.

22.1.3 Product Sales

It is intended that crude uranium oxide (U_3O_8) will be produced and transported from the Westmoreland site to Adelaide, before being shipped to overseas customers.

22.1.4 Exchange Rates

The financial model for the Project uses an A$:US$ exchange rate of $0.70.

22.1.5 Taxes

Australian corporate tax is payable on returns from the Project. The following assumptions have been applied when calculating corporate tax payable:

- 30% corporate tax rate (for the purpose of the financial model, this is applied to the Project cash flows only, i.e. the impact of any other LAM cash flows has been ignored).
- Project assets are depreciated using a 10% straight line depreciation method.

22.1.6 Goods and Services Tax

A goods and services tax (GST) at a rate of 10% is levied by the Australian Federal Government on purchases by individuals and corporations on non-exempt goods and services. Businesses can claim back GST on most business inputs. As all product sales will be to overseas customers, GST is not applicable.

22.1.7 Royalties

Queensland State royalties apply to the Project. Queensland State mineral legislation imposes a royalty on the sale of minerals. The royalty rate applicable to uranium is 5%.

The financial analysis of the Project has applied the royalty rates above, which are based on the calculations provided in Schedule 3 of the Mineral Resources Regulation 2025.

22.1.8 Other Royalties

A 1% NSR is payable to Royalty Gold, with cumulative payments capped at A$10M indexed to inflation.

22.1.9 Revenue Deductions

The cost of shipping and insurance is a revenue deduction for the purpose of determining Queensland State royalties payable.

22.1.10 Reclamation

No assumptions about the salvage value on plant and equipment have been made in the financial model.

22.1.11 Project Financing

No assumptions have been made about the Project financing in the financial model.

22.2 Financial Model

Table 22.2.1 provides the key economic assumptions used in the financial model.

Table 22.2.1 Key Economic Assumptions Used in the Financial Model

Assumption	Units	Rate
Commodity Prices	lb	US$90
Exchange Rates		
A$:US$ Exchange Rate	A$:US$	0.70
Other		
Corporate Tax Rate	%	30.0
Discount Rate	%	7.5

22.3 Model Inputs

22.3.1 Production summaries

The production schedule for the Project is described in Section 16.

22.3.2 Capital Cost Summary

The capital costs for the Project are as described in Section 21.

22.3.3 Operating Cost Summary

The operating costs for the Project are as described in Section 21.

22.3.4 Economic Results

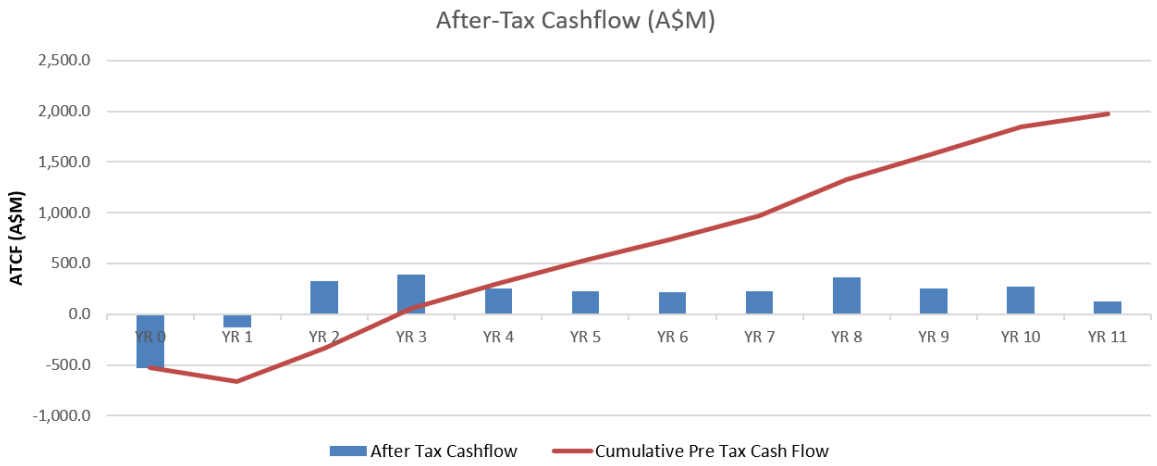
Table 22.3.1 below shows the production of uranium oxide and unit operating costs for the Project.

Table 22.3.1 Economic Analysis Results

	A$	US$
Capital Cost	$891M	$623M
Operating Cost	$2,182M	$1,527M
Royalty	$318M	$223M
Tax	$831M	$582M
Revenue	$6,162M	$4,314M
Pre-Tax NPV	$1,605M	$1,124M
Pre-Tax IRR	43%	43%
Post Tax NPV	$1,059M	$741M
Post Tax IRR	33%	33%
U Production Mlb	47.9	47.9

The Project has an 11 year mine-life, with mining undertaken over the entire 11 years (including pre-strip). The Project payback period is approximately 2.5 years into the mine-life, as shown in Figure 22.3.1.

Figure 22.3.1 Annual and Cumulative After Tax Cash Flow



The Project's economics are most sensitive to commodity pricing assumptions and foreign exchange rate assumptions.

The NPV sensitivity for the Project under various scenarios where the following factors are increased or decreased by incremental percentages are shown in Figure 22.3.2 and Table 22.3.2.

The factors adjusted for the sensitivity analysis are:

- Capital cost.
- Uranium sales price.
- Forex.
- Processing operating costs.
- Mining operating costs.

Figure 22.3.2 Pre-Tax NPV Sensitivity Analysis

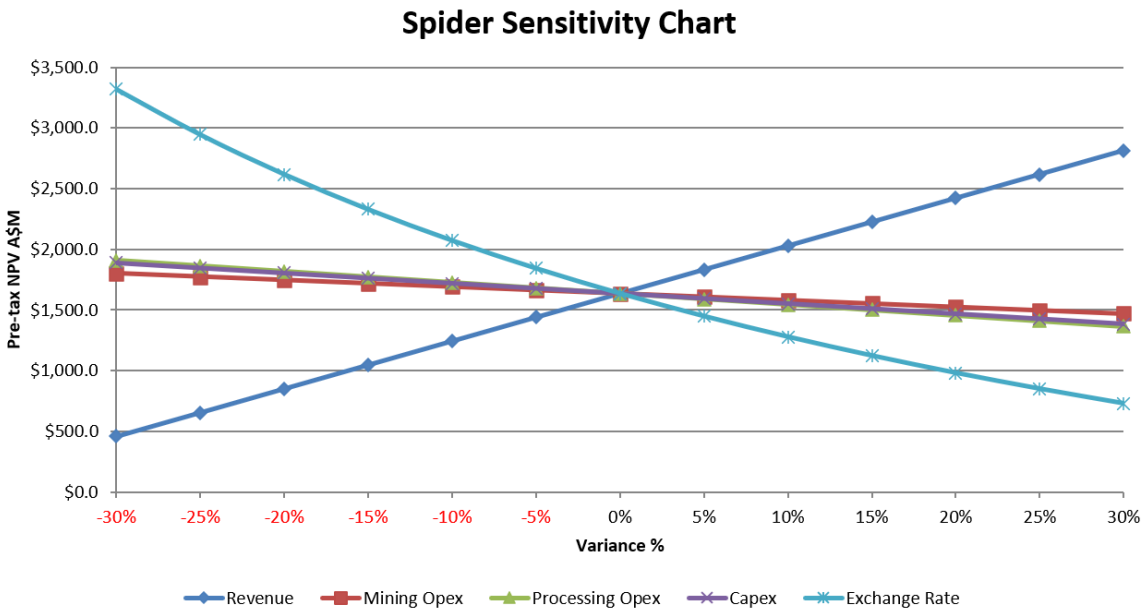


Table 22.3.2 Pre-Tax NPV Sensitivity

Variance	Revenue A$...M	Mining Opex A$...M	Processing Opex A$...M	Capex A$...M	Exchange Rate A$...M
30%	$2,784.1	$1,605.9	$1,324.6	$1,353.9	$699.5
25%	$2,587.7	$1,605.9	$1,371.5	$1,395.9	$820.4
20%	$2,391.3	$1,605.9	$1,418.4	$1,437.9	$951.3
15%	$2,195.0	$1,605.9	$1,465.2	$1,479.9	$1,093.6
10%	$1,998.6	$1,605.9	$1,512.1	$1,521.9	$1,248.8
5%	$1,802.2	$1,605.9	$1,559.0	$1,563.9	$1,418.8
0%	**$1,605.9**	**$1,605.9**	**$1,605.9**	**$1,605.9**	**$1,605.9**
-5%	$1,409.5	$1,605.9	$1,652.7	$1,647.9	$1,812.6
-10%	$1,213.1	$1,605.9	$1,699.6	$1,689.8	$2,042.2
-15%	$1,016.8	$1,605.9	$1,746.5	$1,731.8	$2,298.9
-20%	$820.4	$1,605.9	$1,793.4	$1,773.8	$2,587.7
-25%	$624.0	$1,605.9	$1,840.2	$1,815.8	$2,915.0
-30%	$427.6	$1,605.9	$1,887.1	$1,857.8	$3,289.0

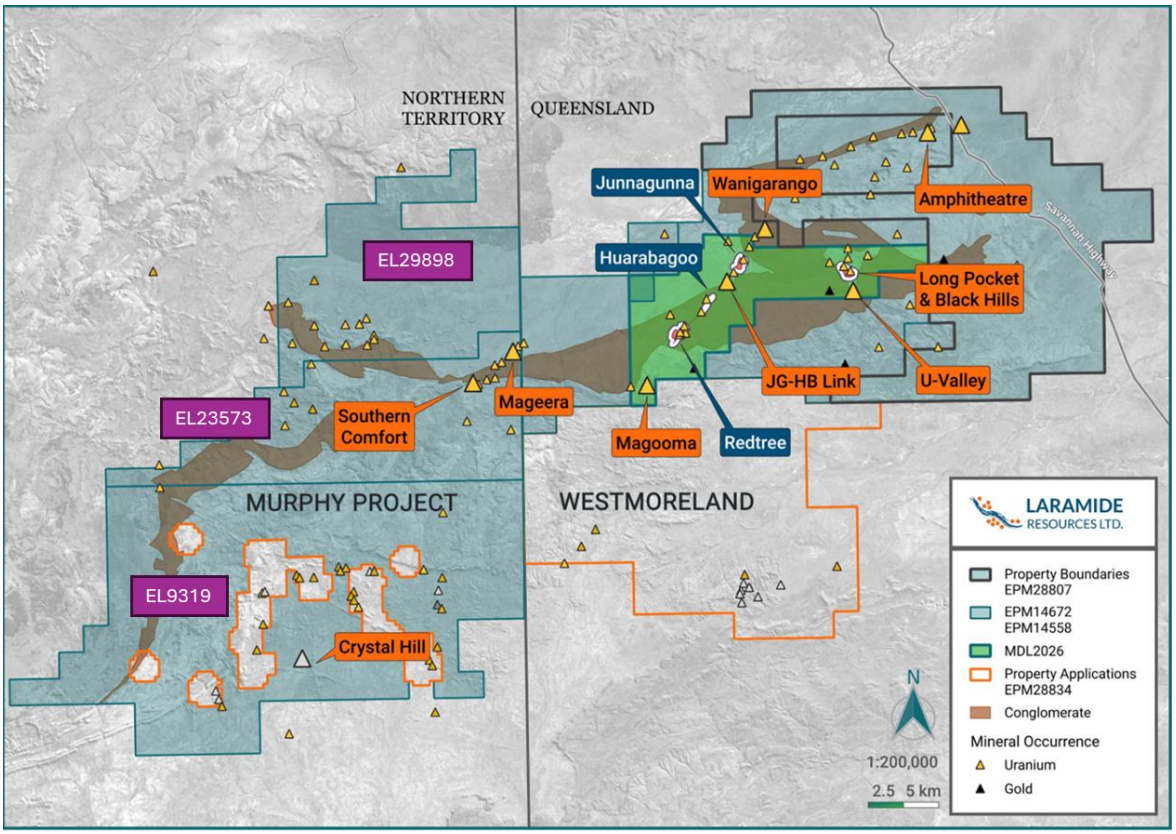
23.0 ADJACENT PROPERTIES

23.1 Introduction

LAM holds an interest in a number of NT tenements immediately adjacent to the QLD Westmoreland project tenements that have not been discussed in this report (Figure 23.1.1). The tenements are contiguous and cover a significant area of the Wearyan Shelf of the southern McArthur basin and the adjoining Murphy Inlier. The Murphy Project tenements extend up to 90 km to the west of the NT-QLD border and is considered a separate Project due to the geographic scale, topographical and access limitations. There are currently no MRE for the Murphy Project.

The Murphy Project has some geological similarities to the Project including continuation of the Westmoreland Conglomerate and Seigal volcanics units. However, it contains a variety of other mineralization styles associated with the Cliffdale volcanic units of the Murphy Inlier and is prospective for gold, copper, tin, tungsten, and other minerals.

Figure 23.1.1 LAM's QLD and NT Tenements



24.0 OTHER RELEVANT DATA

24.1 Northern Australia Development Policy and Infrastructure

The Project is located in NW QLD within the region covered by the Australian Government's Northern Australia development agenda. In 2024, the Australian Government released the *Northern Australia Action Plan 2024–2029*, which builds on the *Our North, Our Future: White Paper on Developing Northern Australia* released in 2015 and establishes a refreshed policy framework for economic development across northern Australia.

The action plan identifies six priority areas: activating the northern economy; infrastructure to unlock growth; a safe and secure north; growing the northern Australia workforce; liveable, healthy and resilient communities; and protecting the north's environment and cultural heritage. First Nations participation and outcomes are incorporated across these priorities. The framework recognizes northern Australia's natural-resource endowment and the importance of infrastructure, investment, workforce development, and resilient regional communities in supporting economic growth.

A significant development since the previous Westmoreland study is the establishment and expansion of the Northern Australia Infrastructure Facility (NAIF). NAIF was established in 2016 to support economic and infrastructure development in northern Australia. Its financing capacity was increased from A$5 billion (bn) to A$7bn in 2023 and, in July 2026, the Australian Parliament extended its investment period by 10 years to 30 June 2036. NAIF can provide flexible debt and, in certain circumstances, equity financing for eligible projects that deliver economic and public benefits to northern Australia.

Westmoreland's remote location means that access to power, transport, communications and other regional infrastructure remains an important component of project development. Government programs supporting northern Australian infrastructure and regional development could therefore provide opportunities for engagement with government and infrastructure providers as the Project advances. No assumption has been made in the SS regarding the availability of government or NAIF funding for the Project.

24.2 Queensland Uranium Mining Policy

The regulatory and policy position applicable to uranium mining in QLD remains a material consideration for the Project.

Uranium mining in QLD is not expressly prohibited by the Mineral Resources Act 1989. However, the QLD Government currently maintains a policy position under which deliberate production of uranium is not permitted. Exploration for uranium is not prohibited, as uranium commonly occurs in association with other minerals.

The QLD Government changed following the State election held in October 2024, with the Liberal National Party forming government. The current Government has adopted a generally supportive policy towards resources-sector investment, and has introduced measures intended to facilitate investment and accelerate the assessment of significant resource and critical-mineral projects. Notwithstanding this broader pro-development policy position, the QLD Government stated in February 2026 that uranium mining does not form part of its current plan for the resources sector and that deliberate uranium production remains prohibited.

Accordingly, development of the Project as a uranium mine would require a change in current QLD Government policy, in addition to obtaining the mineral tenure, environmental, Native Title and other approvals applicable to the development of a uranium mine.

The SS does not assume that such a policy change will occur. Continued exploration and technical studies can nevertheless advance the Project and improve its development readiness should the QLD Government's uranium mining policy change in the future.

24.3 Australian Uranium Policy, Exports and Safeguards

Australia is an established producer and exporter of uranium for peaceful nuclear purposes. Uranium oxide concentrate produced in Australia is subject to Commonwealth export controls, including the requirement for a Mineral Export Permission (MEP) and individual shipment approvals.

The Australian Government administers uranium exports in accordance with Australia's nuclear non-proliferation and safeguards obligations. Australian uranium may only be supplied for peaceful purposes to countries meeting Australia's safeguards and nuclear cooperation requirements. The Department of Industry, Science and Resources (DISR) administers uranium export permissions in conjunction with the Australian Safeguards and Non-Proliferation Office and other Commonwealth agencies.

Consequently, while the development and approval of an individual uranium mine is substantially affected by the policy and regulatory framework of the relevant State or Territory, an established Commonwealth framework exists for the export and international use of Australian uranium.

24.4 Uranium Market and Nuclear Energy Developments

During the period 2016 to 2026, uranium market conditions have strengthened substantially, supported by increasing nuclear generation, reactor life extensions, construction of new nuclear reactors and renewed emphasis on energy security and reliable low-emissions electricity supply.

The Australian Government's *Resources and Energy Quarterly – June 2026* describes global uranium demand as increasing strongly and forecasts reactor uranium consumption to rise from approximately 91,000 t in 2025 to 104,000 t in 2031. The same outlook forecasts uranium prices increasing from approximately US$85 /lb in the second half of 2026 to an average of approximately US$95 /lb in 2031, in real terms. Australian uranium export earnings are forecast to increase from approximately A$1.6bn in 2025 to 2026 to A$1.8bn by 2030 to 2031.

24.5 North West Queensland Energy and Infrastructure Development

A significant regional development is the proposed expansion of electricity transmission infrastructure across north and NW QLD.

The CopperString project is intended to establish a high-voltage electricity transmission connection between the Townsville region and the North West Minerals Province. The broader project has been designed to ultimately connect the region with the National Electricity Market and facilitate additional electricity generation and industrial development.

The project delivery strategy has evolved during 2025 and 2026. The current program prioritises completion of the Eastern Link between the Townsville region and Hughenden by 2032, subject to approvals. Development options for the Western Link towards Mount Isa are continuing to be assessed, with the QLD Government also establishing a A$200M North West Energy Fund to support renewable generation, battery storage, microgrids and other firming solutions in communities including Richmond, Julia Creek, Cloncurry and Mount Isa.

The QLD Government continues to identify CopperString as a major infrastructure project supporting development of the North West Minerals Province. In 2026, the Government's regional project pipeline identified the broader CopperString project as a major investment extending between Mount Isa and Townsville.

Westmoreland is not located directly on the proposed CopperString transmission corridor and a dedicated transmission connection or other power solution would still be required to supply the Project. Nevertheless, increased generation capacity and transmission development within the North West Minerals Province could materially alter the availability, cost and emissions intensity of future electricity supply to the Project. These developments should therefore continue to be monitored as Westmoreland advances through future study phases.

24.6 Queensland Critical Minerals and Resources Development

The QLD Government policy towards minerals development has increasingly focused on critical minerals, supply-chain security and development of the North West Minerals Province.

In 2026, the QLD Government released *Delivering Queensland's Critical Minerals Future 2026–30*, which seeks to accelerate project development, attract investment, strengthen regional economies and coordinate infrastructure and government support across the critical-minerals value chain. The strategy emphasises investment attraction, infrastructure development, project acceleration and development of downstream industries.

The QLD Government has also introduced legislative measures intended to provide faster and clearer decision-making pathways for major projects capable of making a significant contribution to the State's economic development, including critical-minerals projects.

Uranium is not assumed in this Technical Report to receive the same policy treatment as minerals designated as critical minerals by the QLD Government, and these initiatives do not alter the current QLD policy restricting uranium production. Nevertheless, the increased emphasis on mineral exploration, regional infrastructure and investment in NW QLD is relevant to the broader development environment in which Westmoreland is located.

24.7 First Nations Participation and Regional Development

Government policy relating to development in northern Australia increasingly emphasises partnerships with First Nations peoples and the delivery of lasting regional economic benefits.

The *Northern Australia Action Plan 2024–2029* identifies advancing First Nations outcomes as a central objective across each of its six policy priorities. Similarly, projects seeking finance through NAIF are generally required to develop an Indigenous Engagement Strategy addressing participation and economic opportunities associated with project development.

As Westmoreland advances, continued engagement with Traditional Owners, Native Title parties, local communities, landholders and government will form an important component of project planning and approvals. Potential regional benefits may include employment, training, procurement and business opportunities, together with investment in infrastructure associated with development of the Project.

25.0 CONCLUSIONS

25.1 Geology and Resource Estimate

The Westmoreland uranium deposits—RT, JG and HB—are hosted largely within the shallow dipping Westmoreland Conglomerate. The RT uranium deposit flanks the RT dyke zone immediately north of the NW-trending Namalangi fault. The deposit comprises horizontal mineralization in the Jack, Garee, and Langi lenses and vertical mineralization in the Namalangi Lens with grades ranging from 0.15% to >2% U_3O_8. The HB deposit is about 3 km NE of RT along the RT dyke zone and straddles the contact of the Seigal Volcanics with the Westmoreland Conglomerate. The deposit comprises a 3 km zone of vertical mineralization associated with a complex dyke geometry with vertical and horizontal branches between the two principal dykes. The JG uranium deposit occurs at a fault intersection west of the RT dyke zone and south of the northwest trending Cliffdale fault. Mineralization lies 0.5 to 10 m thick immediately beneath the Seigal-Westmoreland contact. Uranium mineralization occurs on the northern side of the Cliffdale fault and the eastern side of the RT dolerite dyke zone. Drilling at LP intersected horizontal uranium mineralization over a 500 m strike length above a dolerite sill and immediately below the underlying sill contact.

An update to the MRE for the Project, QLD, Australia (Figure 25.1.1 and Figure 25.1.2) has been prepared by Addison Mining Services of the UK on behalf of LAM ('the Issuer'). The Issuer is a dual listed entity on the TSX and ASX stock exchanges of Canada and Australia respectively, as such the estimate is reported in accordance with NI 43-101 *Standards of Disclosure for Mineral Projects,* and prepared under CIM Definition Standards. CIM *Definition Standards for Mineral Resources* (2014) and *Best Practices Guidelines* outline by CIM (2019) have been followed. The estimate is also reported in accordance with the *Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves* ('the JORC Code' 2012 edition.)

The updated MRE has an effective date of 31 January 2025, and is reported above a cut-off grade of 200 ppm U_3O_8 and comprises of:

- Indicated Resources of 27.8 Mt at an average grade of 770 ppm U_3O_8 for 48.1 million contained pounds of U_3O_8.
- Inferred Resources of approximately 11.8 Mt at an average grade of 680 ppm U_3O_8 for 17.7 million contained pounds. of U_3O_8.

The updated estimate supersedes all previous estimates. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. It is important to note that currently, only exploration, and not mining for uranium is permitted in QLD, Australia. However, it is reasonable to expect that the policy may change in the future as there is a historical precedent for uranium mining within the State.

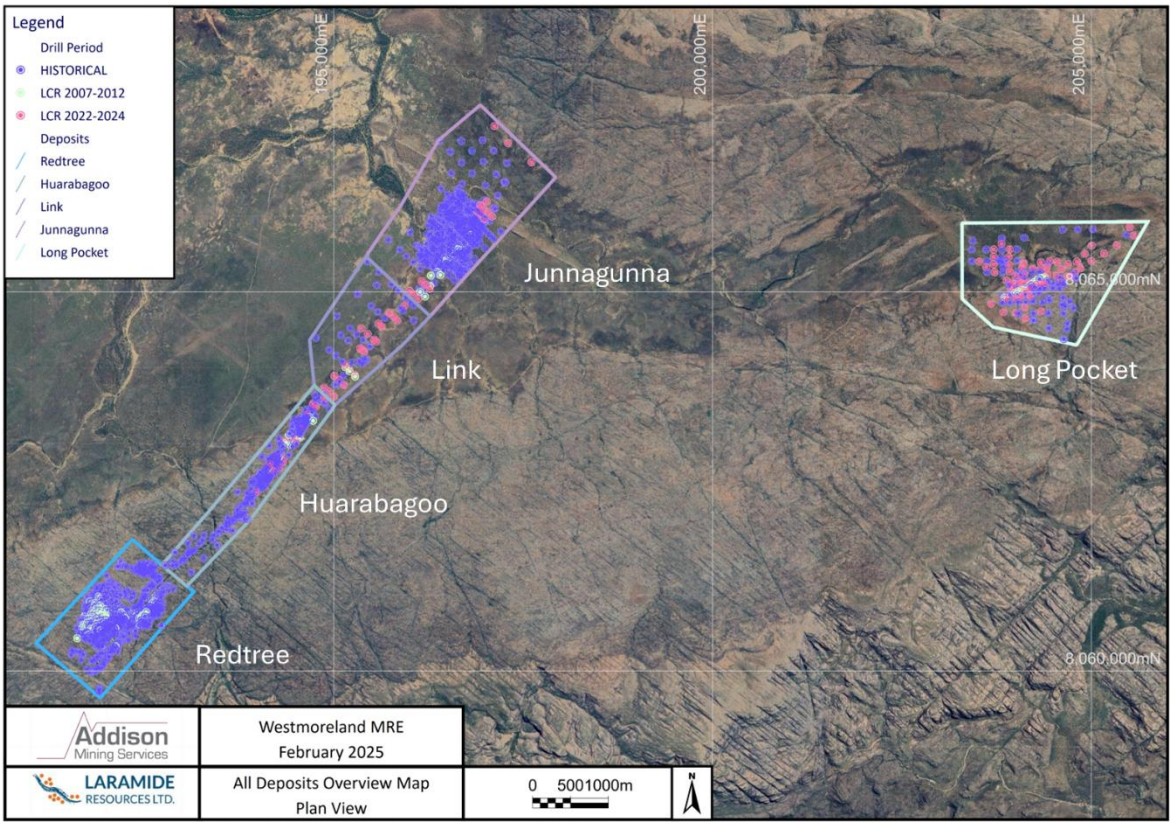
Table 25.1.1 sets out the Indicated and Inferred Mineral Resources by deposit. Readers are encouraged to review the accompanying notes and explanatory text in support of the estimate.

Table 25.1.1 Mineral Resources by Deposit for the Westmoreland Uranium Project

Deposit	Tonnes t	Density g/m³	U_3O_8 ppm	U_3O_8 Mlb.
Indicated				
Redtree	14,000,000	2.5	880	27
Huarabagoo	2,500,000	2.6	890	4.9
Junnagunna	10,000,000	2.5	640	15
Long Pocket	1,300,000	2.5	420	1.2
Total Indicated	**27,800,000**	**2.5**	**770**	**48.1**
Inferred				
Redtree	3,000,000	2.5	800	5.2
Huarabagoo	3,100,000	2.6	870	6.0
Junnagunna	3,000,000	2.5	620	4.2
Long Pocket	2,700,000	2.5	380	2.3
Total Inferred	**11,800,000**	**2.5**	**680**	**17.7**

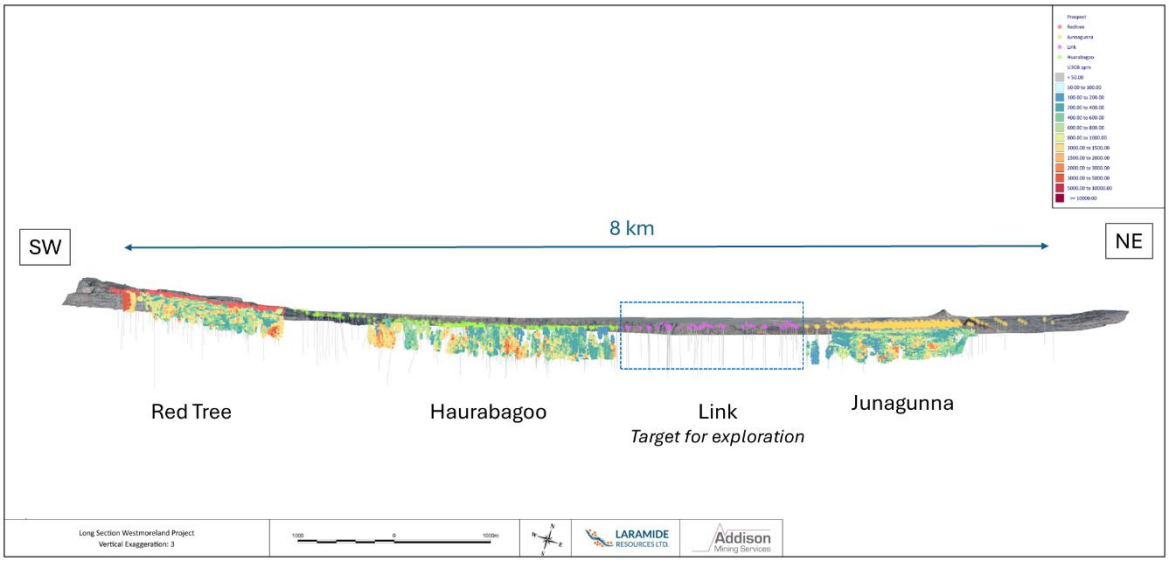
Reported above a cut-off grade of 200 ppm U_3O_8. Effective 31 January 2025.

Figure 25.1.1 Westmoreland – Mineral Resource Estimate Areas



No Mineral Resources are reported for the Link area.

Figure 25.1.2 Westmoreland Long Section



Looking NW, displaying drillholes and block models.

25.2 Mining Methods

The mining methodology is based on conventional methods and is summarized below:

- Open-pit mining using three excavators loading off-highway haul trucks - two EX2000 excavators and one EX1200 excavator, all loading CAT777 trucks. The average production rate for the EX2000's was 1,073 t/h and the production rate for the EX1200 was 558 t/h. Four trucks were allocated to each excavator over the LOMP. The excavators were supported by an appropriate ancillary fleet.
- Conventional D&B with T&S operation mining 5 m benches with 2.5 m flitches.
- Sufficient working areas to allow for simultaneous D&B and T&S operation. Flexibility in the scheduling is required.

The mining methodology is based on the following material movement schedule:

- A total of 147 Mt TMM will be moved over 11 years of mining with 118 Mt of waste and 28.3 Mt of production target being produced grading 810 ppm. The LOM strip ratio of the four pits is 4.2 t:t. The strip ratio varies substantially between the pits with RT pit at 1.7 t:t, LP pit at 4.5 t:t, JG pit at 5.3 t:t, and HB pit at 10 t:t.
- Mill feed: 2.9 Mtpa achieved in the second year onwards for the full mine life. The mill throughput reduces to 1 Mtpa in the final year of production.

Mining would commence in RT to establish an initial tailings emplacement area. The first five years focus on production at RT and pre-strip at HB and JG, production from HB and JG is scheduled during Years 5 to 9, with LP production from Year 9 to the end of mining operations in Year 11.

25.3 Metallurgical Testwork

The testwork results do not introduce any concerns regarding processing the Westmoreland materials. The following conclusions are noted:

- The Westmoreland material generally acid leaches very well with modest acid consumption and high U extractions. The leach times are relatively short.
- The Jack Lens material was the only exception and extraction was improved by adding ferric iron to assist oxidation of tetravalent uranium in the material.
- The grind size distribution is relatively coarse, but only very limited comminution testwork has been reported to date and the results are contradictory.
- Recovery of the U from the leached slurry by both Ion exchange and solvent extraction was investigated.
- Precipitation of the U as a concentrate to be sold to the market is good quality and can be treated by any of the three main converters that will be treating the material.
- Pulp settling is reasonable with a very high solids underflow density and a relatively clear overflow PLS.
- The resin chosen for IX testwork showed good loading and elution characteristics.
- Solvent extraction tests showed that the kinetics with a conventional amine extractant are good.
- Stripping the loaded SX organic is also conducted relatively easily using ammonium sulfate.

25.4 Process Plant

Based on the currently available metallurgical test data, processing Westmoreland ore at a rate of 2.9 Mtpa by a process that involves: crushing and grinding in a SS SAG circuit; leaching with sulfuric acid at 40°C; separation of the leached ore from the pregnant liquor by pressure filtration; recovery of the U by continuous IX and elution; production of a uranium oxide concentrate by neutralization of the eluate; and precipitation of UOC with hydrogen peroxide; is a feasible treatment scheme at the SS level. Attention will need to be given to the control of impurities in the final UOC concentrate if the Project moves forward to the next phase.

25.5 Infrastructure

The Westmoreland prospect is located in a remote region of NW QLD and the provision of appropriate infrastructure will be required as part of the Project. In particular the cost of provision of the projects water requirements and electric power requirements, while adequately addressed at the SS level, will require more detailed investigation if the project moves forward to the next phase.

The design of the TSF has been developed as a conceptual level only with significant testing and verification of the design required at the next stage of study. Based on the assumption adopted by KP and stated in the SS, it is considered that an integrated ex-pit / in-pit filtered tailings stack facility will be feasible at the Project.

25.6 Risks and Opportunities

The Project is at an early (SS) level of evaluation. Only limited metallurgical testwork and associated sampling has been completed. In particular, recent testwork has been based on samples with higher than average head grade for the deposit. Only very limited comminution testwork has been performed and the results were contradictory. Significantly more sampling and metallurgical testwork is required in the next phase of the Project to confirm the process design criteria and basis of design. This may result in changes to the process flowsheet and operating conditions with impacts on both the capital and operating cost estimates.

Similarly only very preliminary evaluations of project infrastructure provision have been undertaken. In the next phase of the project more detailed investigations into project water supply and electric power supply are required and these may impact both capital and operating cost estimates.

If the Project moves forward with the IX recovery flowsheet, then the opportunity exists to incorporate an acid recovery membrane plant. This will require further focused testwork to determine its applicability to Westmoreland. If successful, this flowsheet change may enable recovery / recycle of up to 65% of the sulfuric acid with significant savings in both capital and operating cost, both for sulfuric acid and caustic soda consumption (for acid neutralization). However, further testwork and engineering is required to define the capital and operating costs of the acid recovery plant, and the impact of recycled acid and the associated uranium and other impurities in the acid recycled to the remainder of the process.

25.7 Capital and Operating Costs

Total capital and operating cost estimates have been developed at the SS accuracy level of ±35%. Total capital investment for the Project as at 3Q26, subject to the limitations and provisions as stated in Section 21, is estimated to be A$646,275,971.

Total operating cost as at 3Q26, subject to the limitations and provisions as stated in Section 21, is estimated to be A$49.58 /t ore including G&A.

Total mining unit cost as at 3Q26, subject to the limitations and provisions as stated in Section 21, is estimated to be A$5.34 /t TMM.

25.8 Financial Analysis

At the SS level of analysis and based on a 11 year life the financial analysis indicates an economically sound project with an after tax NPV of A$1,058.7M at a discount rate of 7.5% and an after tax IRR of 33.0%. The Project payback period is approximately 2.5 years.

26.0 RECOMMENDATIONS

26.1 Geological

Further resource definition and expansion work to be carried out proximal to the current resource to support a more detailed mine plan as the project progresses to PFS level. Field work will need to incorporate drilling and field mapping.

Continued regional exploration should be undertaken and is likely to yield further discoveries as the geological understanding of the resource geology develops.

26.2 Mining

Further key areas of work to be completed during the next project development stage include:

- Refinement of the mine scheduling plan.
- Further refinement of the in-pit tailings design and scheduling.
- Further geotechnical evaluation work to determine pit slopes, bench heights, berms, etc. This work will also confirm pit liner materials for the in-pit tailings plan.
- Further hydrological work to assess potential water issues relating to the pit design.
- Work to improve costing and planning relating to mine closure issues.

26.3 Metallurgical Testing

It will be necessary to conduct a thorough and exhaustive metallurgical testwork programme on Westmoreland materials that are representative of what would be mined and processed. This programme would be designed to test options and to determine metallurgical characteristics and parameters for proper detailed design. Such detail would be necessary for the next stage of study, which would be at a PFS level or a definitive feasibility study (DFS) level.

26.4 Process Plant

A significant reduction in plant capital cost and operating cost may be possible if an acid recovery plant (membrane plant) can be shown to be technically and economically feasible. This could result in significant reduction in acid consumption, possibly eliminating the need for an acid plant onsite, and reducing the caustic soda requirement for eluate neutralization. It is recommended that this flowsheet option be investigated in the next project phase.

26.5 Project Infrastructure

26.5.1 Project Water Supply

Two options for water supply were highlighted for the Project. Further investigation is thus required during the next phase to confirm which option is best to advance for project development.

26.5.2 Project Electric Power Supply

The hybrid diesel and solar / BESS option at 40% RF was selected for the Project as being the most economical in the current market. It is suggested all options are reviewed again at the next phase of the Project in light of the continued government investment in the CopperString project in north QLD and the continued technology advances in power generation.

26.5.3 Tailings Storage Facility

The key aspects of the tailings management system which will need to be addressed as part of the next stage of study will be as follows:

- Initial consultation with the regulatory authorities to gain acceptance for the design concepts, as no similar tailings disposal operations are present in QLD.
- The geochemical characteristics of tailings and whether the facility can be approved without neutralization of the tailings prior to disposal.
- The geotechnical properties of the filtered tailings including strength, permeability, consolidation, and liquefaction potential.
- Detailed mining and tailings disposal plans to be developed for the pit.
- Investigation of the availability of suitable low permeability liner and capping material at site.
- Determination of the gradation of blasted waste rock at site to assess the requirement for the geocomposite liner below the basal low permeability liner, any requirement for a filter blanket or geofabrics on the upstream side of the waste rock bunds and requirement for a filter blanket or geofabrics (or geocomposite) below the low permeability cap.
- Modelling of the thickness of the inert waste capping material to ensure that no erosion and minimal desiccation of the low permeability capping occurs.

26.6 Project Development

Financial analysis at the SS level of accuracy has indicated that the Project economics are robust and would justify progressing the Project to the next phase of development.

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DATE AND SIGNATURE

Name:	Ms. Sandra (Sandy) Hunter
Degree and Professional Association:	FAusIMM(CP), RPEQ
Position:	Process Manager
Signature:	*(signed) "Sandy Hunter"*
Date:	22 July 2026

Name:	Mr. R. J. Siddle
Degree and Professional Association:	MAIG
Position:	Principal Geologist
Signature:	*(signed) "Richard Siddle"*
Date:	22 July 2026

Name:	Mr. A. Stepcich
Degree and Professional Association:	FAusIMM(CP), RPEQ
Position:	Associate Principal Mining Engineer
Signature:	*(signed) "Anthony Stepcich"*
Date:	22 July 2026

Name:	Mr. Timothy Rowles
Degree and Professional Association:	FAusIMM(CP), MAIG (RPGeo), RPEQ
Position:	Regional Manager
Signature:	*(signed) "Tim Rowles"*
Date:	22 July 2026

CERTIFICATE OF QUALIFIED PERSON

I, Sandra (Sandy) Hunter, FAusIMM(CP), RPEQ, as an author of this report entitled '*Preliminary Economic Assessment Update National Instrument 43-101 Technical Report*' for the Westmoreland Uranium Project, Queensland, Australia, prepared for Laramide Resources Ltd. and dated 22 July 2026, do hereby certify that:

1) I am a Manager Process - Brisbane with Lycopodium (APAC) Pty Ltd. My office address is Level 1, 757 Ann Street, Fortitude Valley, Queensland 4006.

2) I am a graduate of the Murdoch University in 2000 with a Bachelor of Science Honours degree in Mineral Science (Extractive Metallurgy). I am a Fellow and Chartered Professional (Metallurgy) of the Australasian Institute of Mining and Metallurgy (AusIMM) (member number 112041), and registered as a Practising Registered Professional Engineer (in the area of Metallurgy) in the state of Queensland, registration number 28941.

3) I have worked as a process engineer / metallurgist continuously since 1996. My relevant experience for the purpose of the Technical Report is:

   - review and report as a consultant on numerous process plant facilities and mining projects around the world on various commodities.
   - Principal Process Engineer for numerous projects, including preliminary economic assessments, feasibility studies and detailed designs in Australia, Africa, Asia and the Americas.
   - development, execution and interpretation of a number of metallurgical testwork programs.
   - Plant metallurgist and commissioning manager at a number of gold mines in Australia, Africa, North America and Asia.

4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have not visited the Westmoreland Uranium Project.

6) I am responsible for all of preparation of Item Nos. 0, 1, 2 3, 13, 17, 18.2-18.6, 19, 20, 21.1, 21.3, 21.4, 24, 25.3-25.8, 26.3-26.6, 27 of the Technical Report.

7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

8) I have not been involved in the previous Technical Report on the Westmoreland Uranium Project.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) 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.

Dated this 2nd day of September, 2026.



Sandra (Sandy) Hunter, FAusIMM(CP), RPEQ

Addison Mining Services Ltd
110 Brooker Road Waltham Abbey Essex EN9 1JH
United Kingdom
Company number 08883789
www.addisonmining.com

CERTIFICATE OF QUALIFIED PERSON

I, Richard John Siddle, Principal Geologist, as an author of this report entitled '*Preliminary Economic Assessment Update National Instrument 43-101 Technical Report*' for the Westmoreland Uranium Project, Queensland, Australia, prepared for Laramide Resources Ltd. and dated 22 July 2026, do hereby certify that:

1) I am a Principal Geologist with Addison Mining Services Ltd, 110 Brooker Road, Waltham Abbey, Essex, EN9 1JH, United Kingdom.

2) I graduated with a Master of Geology (Hons) from the University of Leicester, UK, in 2007. In addition, I obtained a Master of Science (merit) in Mining Geology in 2010 from the Camborne School of Mines, University of Exeter, Tremough, Cornwall, UK.

3) I am a member of the Australian Institute of Geoscientists (membership number 6802) and a fellow of the Geological Society of London My relevant experience for the purpose of the Technical Report is:

   – I have worked as a geologist for over 16 years since graduation from university. Relevant experience includes 3 years of exploration, drilling supervision and resource development in respect to uranium, gold, silver and base metal deposits in Queensland, New South Wales and Western Australia and 2.5 years as a consulting resource geologist at Micromine Consulting Services. I have since spent 11 years performing resource estimation and geological modelling for Addison Mining Services.

4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I completed a site visit to the project area between the 21st and 23rd of January 2025, and inspected representative sections of drill core, visited rehabilitated drill sites and inspected selected outcrop geology. Discussions were held with the issuer's technical teams and exploration and socio-environmental considerations discussed. No additional drilling has been completed since the date of the site visit and I consider the previous inspection to be current.

6) I am responsible for preparation of Item Nos. 4-12,14 and contribution to relevant sub-sections of items 1 and 23-27 of the Technical Report.

7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

8) I have been involved in the previous Technical Report on the Westmoreland Uranium Project dated 27 August 2025 and dated effective 31 January 2025.

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9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) 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.

Dated this 2nd day of September, 2026.



Richard John Sidlde (M.Sc., MAIG.) (AIG #6802)

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CERTIFICATE OF QUALIFIED PERSON

I, Anthony Stepcich, FAusIMM(CP) and RPEQ, as an author of this report entitled '*Preliminary Economic Assessment Update National Instrument 43-101 Technical Report*' for the Westmoreland Uranium Project, Queensland, Australia, prepared for Laramide Resources Ltd. and dated 22 July 2026, do hereby certify that:

1) I am an Associate Principal Mining Engineer with Mining Associates Pty Ltd. Mining Associates office address is PO Box 161 Spring Hill Qld 4004 Australia.

2) I earned a Bachelor of Mining Engineering from the University of Ballarat in 1992, a Master of Science in Mineral Economics from Curtin University in 1997, and a Graduate Diploma in Finance and Investment from the Securities Institute of Australia in 2002.

3) I am a Fellow and Chartered Professional of the Australasian Institute of Mining and Metallurgy (FAusIMM(CP) membership number 110954). I am a Registered Professional Engineer in the state of Queensland (RPEQ membership number 27082). I have worked as a Mining Engineer for a total of 34 years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - Reviewed and reported on numerous mining projects worldwide as a consultant, supporting due diligence and regulatory requirements.
   - Served as Principal Mining Engineer for multiple scoping, prefeasibility, and feasibility studies across Australia, South America, and Asia.
   - Worked as a site mining engineer at several metalliferous and coal mines in Australia and Asia.
   - Accumulated 18 years of consulting experience as a Principal Mining Engineer, completing studies and due diligence assignments for numerous Australian and international projects.

4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have not undertaken a Site Visit to the Westmoreland Uranium Project.

6) I am responsible for all of preparation of Item Nos. 1(part), 16, 21.1.3, 21.2, 21.4.1, 22, 25.2, 25.7, 26.2, 27 of the Technical Report.

7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

8) I have not been involved in the previous Technical Report on the Westmoreland Uranium Project.

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9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) 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.

Dated this 2nd day of September, 2026.



AusIMM
CHARTERED PROFESSIONAL
MINING
Anthony Stepcich

Anthony Stepcich,
BEng(Mining), MSc(Mineral Economics), Grad Dip(Finance & Investment), FAusIMM(CP), RPEQ.

CERTIFICATE OF QUALIFIED PERSON

I, Timothy David Rowles, FAusIMM (CP), MAIG (RPGeo), RPEQ, as an author of this report entitled '*Preliminary Economic Assessment Update National Instrument 43-101 Technical Report*' for the Westmoreland Uranium Project, Queensland, Australia, prepared for Laramide Resources Ltd. and dated 22 July 2026, do hereby certify that:

1) I am a Principal Engineer and the Regional Manager (QLD) with Knight Piésold Pty Ltd. My office address is 36 Cordelia Street, South Brisbane, Queensland 4101, Australia.

2) I graduated from the Royal School of Mines, Imperial College, London with a Bachelor of Science in Environmental Geology in 1996 and from the University of Manchester with a Masters Degree in Earth and Environmental Science in 1998. I was awarded a Professional Certificate in Tailings Management by AusIMM in 2021. I am a Fellow of the Australasian Institute of Mining and Metallurgy (No 227249) and have been awarded the status of Chartered Professional (CP) in the field of Environmental Engineering. I am a Member of the Australian Institute of Geoscientists (No 10317) and have been awarded the status of Registered Professional Geoscientist (RPGeo) in the field of Geotechnical & Engineering. I am a registered Professional Engineer Queensland (No 10166).

3) I have worked as a Geotechnical / Geoenvironmental Engineer for a total of 29 years since my graduation. My relevant experience for the purpose of the Technical Report is:

- Delivering full-lifecycle tailings management solutions — from conceptualization, investigation and design through construction, operation, and closure — for precious, base & energy metals and bulk metals & minerals.
- Principal Engineer for numerous technical studies including preliminary economic assessments, prefeasibility and feasibilities studies covering tailings systems, mine site water management systems, waste rock management and mine site infrastructure both in Australia and internationally including projects in Africa, Asia, Europe and the Americas.
- Resident Engineer providing construction supervision and quality assurance for the construction of a range of tailings storage facilities, water dams, surface water management structures, lining systems and other infrastructure.
- Engineer of Record for a range of tailings management facilities both in Australia and Internationally including upstream, centerline and downstream constructed dams, using a range of tailings technologies including unthickened, thickened and cyclone deposition.
- Member of Independent Technical Review Boards (ITRBs) for tailings storage facilities with high to extreme consequence categories both in Australia and Internationally.

4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have not personally visited the Westmoreland Uranium Project site.

6) I am responsible for preparation of portions of Section 1,18,26 of the Technical Report.

7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

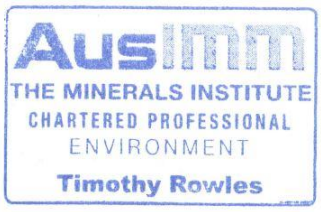
8) I have been involved in the previous Technical Report on the Westmoreland Uranium Project.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) 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.

Dated this 2nd day of September, 2026.





Timothy David Rowles, FAusIMM (CP), MAIG (RPGeo), RPEQ