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STOCK EXCHANGE ANNOUNCEMENT

24 December 2015

BANNERMAN FILES 43-101 TECHNICAL REPORT FOR ETANGO URANIUM PROJECT OPTIMISATION STUDY

Perth, Australia – Bannerman Resources Limited (ASX: BMN, TSX: BAN, NSX: BMN) ("**Bannerman**" or the "**Company**") advises that it has today filed the attached Technical Report on the Company's Etango Uranium Project with the Canadian securities regulators in accordance with the Company's ongoing disclosure obligations for the Toronto Stock Exchange listing.

The Technical Report is a comprehensive summary of the recently-completed Optimisation Study ("**OS**") on the geological modelling and mine planning aspects of the Etango Definitive Feasibility Study ("**DFS**"), which was compiled in April 2012. The OS also reflects updated capital and operating cost estimates. The report has been prepared with the assistance of independent consulting firms AMEX Foster Wheeler and Optiro in accordance with Canadian National Instrument 43-101. The OS has significantly enhanced the project economics and re-enforces Etango to be a globally significant uranium project with average annual forecast production of 7.2 million pounds U_3O_8 per year over a 15.7 year mine life. The project is located in Namibia, a premier uranium mining jurisdiction with nearby infrastructure and demonstrated political and social support for uranium mining operations.

A copy of the Technical Report can be obtained from the Canadian Securities Administrators. SEDAR filing system at www.sedar.com or from Bannerman's website at www.bannermanresources.com.

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About Bannerman *- Bannerman Resources Limited is an ASX, TSX and NSX listed exploration and development company with uranium interests in Namibia, a southern African country which is a premier uranium mining jurisdiction. Bannerman's principal asset is its 80%-owned Etango Project situated near Rio Tinto's Rössing uranium mine, Paladin's Langer Heinrich uranium mine and CGNPC's Husab uranium mine currently under construction. A definitive feasibility study and an optimisation study have confirmed the technical, environmental and financial (at consensus long term uranium prices) viability of a large open pit and heap leach operation at one of the world's largest undeveloped uranium deposits. In 2015-16, Bannerman is conducting a large scale heap leach demonstration program to provide further assurance to financing parties, generate process information for the detailed engineering design phase and build and enhance internal capability. More information is available on Bannerman's website at www.bannermanresources.com.*

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Etango Uranium Project, Namibia
Optimisation Study

National Instrument 43-101 Technical Report

Bannerman Resources Limited
Date: 24 December 2015

IMPORTANT NOTICE

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Forward-looking Statements

This technical report incorporates forward-looking statements and assumptions that are subject to numerous risks, uncertainties and other factors relating to the Etango Optimisation Study that may cause future results to differ materially from those expressed or implied in such forward-looking statements. The following are important factors that could cause actual results to differ materially from those expressed or implied by such forward-looking statements: fluctuations in uranium prices and currency exchange rates; uncertainties relating to interpretation of drill results and the geology, continuity and grade of mineral deposits; uncertainty of estimates of capital and operating costs, recovery rates, production forecasts and estimated economic return; general market conditions; the uncertainty of future profitability; approval of licences by Government authorities; and the uncertainty of access to the required capital. Readers are cautioned not to place undue reliance on forward-looking statements. Amec Foster Wheeler and Optiro expressly disclaim any intention or obligation to update or revise any forward-looking statements whether as a result of new information, future events or otherwise

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1 SUMMARY

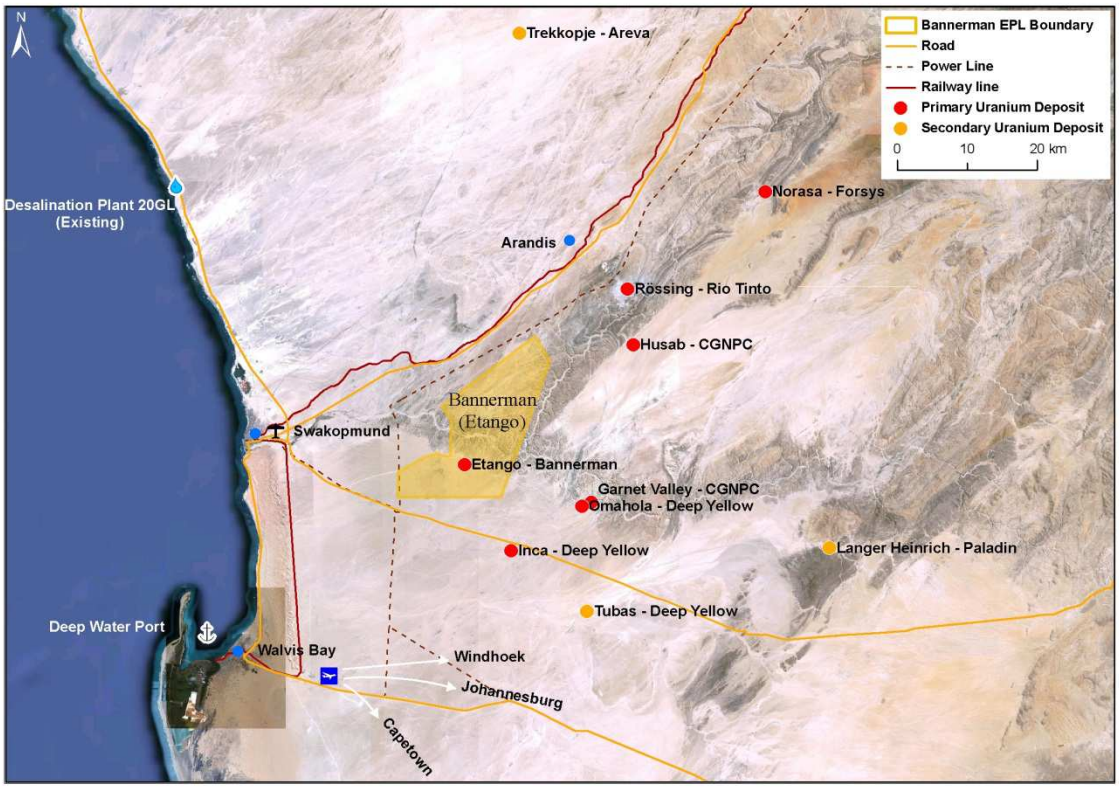
1.1 INTRODUCTION

This report documents the results of the Optimisation Study (OS) which builds upon the DFS completed in May 2012. The OS is based on an updated geological model and includes updated capital and operating costs reflecting the 2015 cost environment. In addition, mine planning consisting of open pit designs and mine production schedules were updated to incorporate the above mentioned changes in input parameters.

1.2 BACKGROUND

Bannerman Resources Limited (Bannerman, or the Company) is a Namibian-focused uranium exploration and development company. Bannerman's primary asset is its 80% owned Etango uranium project (Etango Project) in the coastal Erongo region of Namibia (Figure 1.1). The company announced on the 11 November 2015 that, subject to shareholder approval it will acquire the remaining 20% of the asset. The Etango Project lies within exclusive prospecting licence 3345 (EPL 3345), otherwise known as the Etango tenement.

Figure 1.1
Etango Project – Project Location Plan



Following the positive results of a Scoping Study completed in September 2007, a Preliminary Feasibility Study (PFS) was undertaken with results released in late 2009 and, following additional work, the Company released an update to the PFS (PFSU) in December

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2010. The Company completed a Definitive Feasibility Study (DFS) in 2012 which was documented in a technical report dated 24 May 2012.

Following the Fukushima event and subsequent weakness in the uranium industry, Bannerman has investigated potential ways of improving the project, with results summarised in an Optimisation Study (OS), as detailed in this report. Subject to licensing, and project financing (both of which are contingent on positive developments in the uranium market), Bannerman is planning to commission the Etango Project in 2020.

All monetary amounts expressed in this report are in United States of America dollars ($) unless otherwise stated.

1.3 GEOLOGY AND MINERAL RESOURCES

The Etango mineralisation (comprising the combined Anomaly A, Oshiveli and Onkelo deposits, which were at one time also referred to as the Goanikontes area) is related to uraniferous leucogranites, locally referred to as alaskites, intruded into metasediments of the Nosib and Swakop Groups of the Neoproterozoic (pre-550Ma) to early Palaeozoic (c500Ma) Damara Supergroup.

The Etango deposit forms a 6 kilometre (km) long continuous zone of uranium mineralisation, trending generally north-northwest to north-northeast and dipping to the west. The mineralised zone lies on the flank of the Palmenhorst Dome, occurring in dilatational sites in high-strain zones. Limited faulting is recognised on a deposit scale.

The dominant primary uranium mineral is uraninite (UO_2), with minor primary uranothorite ((Th, U) SiO_4) and some uranium in solid solution in thorite (ThO_2). This mineralisation occurs as microscopic disseminations throughout the alaskite, at crystal interfaces, and as inclusion within other minerals. Larger (up to 350µm) individual crystals occur intermittently, contributing to local higher grades.

Secondary uranium minerals such as coffinite ($U(SiO_4)(OH)_4$) and betauranophane ($Ca(UO_2)_2(SiO_3OH)_2\ 5H_2O$) occur as replacements of the primary minerals or as coatings along fractures. These are present within low to high grade samples, and throughout the entire depth range (0-487m).

QEMSCAN analysis indicates that about 81% of the uranium present is as primary uraninite, while 13% is in coffinite and 5% is in betauranophane (Freemantle, 2009).

Optiro Pty Ltd (Optiro) estimated the most recent resource for Etango as summarised in Table 1.1, reported (in November 2015) at a cut-off grade of 55ppm U_3O_8.

Table 1.1
Etango Deposit, Etango Project, Namibia – November 2015 Resource Estimate

Classification	Tonnes above cut-off (Mt)	U_3O_8 (ppm)	Contained U_3O_8 (t)	Contained U_3O_8 (M lb)
Measured	33.7	194	6,500	14.4
Indicated	362.0	188	68,100	150.2
Measured plus Indicated	**395.7**	**189**	**74,600**	**164.6**
Inferred	144.5	196	28,300	62.5

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In addition, adjacent uranium deposits at Ondjamba and Hyena were estimated to contain Inferred Mineral Resources of 85.1Mt at 166ppm U_3O_8 and 33.6Mt at 166ppm U_3O_8 respectively, both reported above a 100ppm U_3O_8 lower cut-off grade.

Coffey Pty Ltd (Coffey Mining) reviewed drill sampling and data quality control procedures, and validated the database used for resource modelling during the DFS.

The uranium mineralisation was modelled using both lithological and grade constraints above a lower cut-off grade of 50 ppm U_3O_8, resulting in grade shells which separately capture alaskite-dominant (AD) and alaskite sub-dominant mineralisation (ASD). Grade estimation, following variogram analysis and limited grade capping, was via ordinary kriging into 25x25x8m (XYZ) panels. Post-processing was via uniform conditioning followed by localisation into a size of 6.25x12.5x4m (XYZ) to reflect the likely grade control approach of truck scanning. An average density of 2.64t/m^3 was used as in the previous estimate.

The Mineral Resources were classified by Optiro and Bannerman using estimation quality measures, drill spacing and geological confidence, according to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) resource and reserve definition standards (2010).

1.4 MINING METHODS AND RESERVES

The mining method preferred for the Etango open pit will be a high tonnage (100Mtpa), low cost, traditional open pit truck and backhoe operation employing 550t diesel hydraulic excavators, off road 220t haul trucks and 203mm down the hole (DTH) hammer diesel drills.

The pit will be mined in a series of cutbacks to deliver 20Mtpa of ore to the heap leach operation and lower the amount of waste movement required during the early years of the project.

Selective mining for the Etango Project consists of drilling and blasting on a 12m bench, with loading out in three flitches of equal height, which will nominally be 4.5m high, after allowing for swell from blasting. The mining selectivity recommended should minimise ore loss and dilution but, at the same time, allow the 100Mtpa mining rate to be achieved cost efficiently. There is also a clear advantage from a safety point of view for loading in 4.5m flitch heights.

Bannerman estimated the JORC and NI43-101 Reserves for the Etango Staged Design at 303.3Mt at 195ppm U_3O_8 reported above a 55ppm U_3O_8 lower cut-off. The reserve consists of 32.3Mt at 196ppm U_3O_8 of proven mineral reserve and 271.0Mt at 194ppm U_3O_8 of probable mineral reserve.

1.5 METALLURGICAL TEST WORK

A series of bench-scale metallurgical test work programs have been completed since 2008, with emphasis on optimisation of comminution, leaching, and solvent extraction (SX) and other flowsheet parameters.

Significant conclusions that have shaped the proposed development of the project are:

- Pre-concentration of the ore through such processes as scrubbing and screening, flotation, heavy media separation or gravity beneficiation of fines is not practical or cost effective, and is therefore not included in the preferred process design
- Both agitated tank leaching and heap leaching have been tested in the laboratory in acidic environments. Heap leaching is the preferred method for extracting uranium

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from the ore on a cost-benefit basis, the 1-2% reduction in recovery (compared to agitated tank leaching) being offset by reduced capital and operating costs

- Optimal economics for the heap leach were achieved from ore crushed to -8mm (P_{80}=5.3mm), using high pressure grinding rolls (HPGR) as the final stage of crushing. Column tests indicate that, for a heap height of 5m, a recovery of 86.9% can be achieved over a period of 30 days with an acid consumption of 17.6kg/t H_2SO_4.

SX test work was conducted using 5% volume for volume (v/v) Alamine and 2.5% v/v isodecanol, operating at 20ºC and 35ºC. It was concluded that:

- Extractions approaching 100% can be achieved
- Temperature does not appear to increase extraction efficiency
- Extraction is unaffected by the presence of additional salts other than chloride
- The pregnant leach solution (PLS) spiked with chloride showed a decrease in extraction, indicating that control of chloride levels is required in operations
- Ammonium sulphate stripping and ammonia precipitation of uranium is recommended and has been used for engineering design.

1.6 HEAP LEACH DEMONSTRATION PLANT TEST WORK

On 24 March 2015 Bannerman opened the Etango Heap Leach Demonstration Plant.

Demonstration plant test work is scheduled to proceed in phases. The first four phases will focus on demonstrating the design and assumptions of the DFS. Subsequent phases will focus on value engineering. Phases one and two have been completed whilst the third phase is currently underway. The first two phases are briefly summarised below.

- Phase 1 of the program commenced mid-April 2015 and involved an open circuit leach operation of four Cribs (Cribs 1 to 4), each with two Columns running in parallel. The results were released on 15 July 2015.
- Phase 2 of the program commenced mid July 2015 and involved an open circuit leach operation of two Cribs (Crib 5 & 6), each with two Columns running in parallel. All Cribs and Columns were operated under the same conditions as for Phase 1. The results were released on 23 November 2015.

1.6.1

Conclusions from Demonstration Plant test work Phase 1 & 2 are as follows:

- Generally the demonstration plant results are similar or better that those obtained in previous test work performed at similar conditions
- The agglomeration process performed with a agglomerating drum unit using DFS parameters have been validated
- Despite slight segregation of particles there was no evidence of channelling and the agglomerated material retained their integrity
- All cribs achieved uranium extractions over 90% and with an average of 93.3%. This recovery result may be slightly optimistic given the size distribution of the test material of 4.0 mm P80 are finer than the DFS parameter of 5.3mm
- The final acid consumption (after drain, rinse and drain) for all cribs were less than 17kg/t and on average achieved 15.5kg/t

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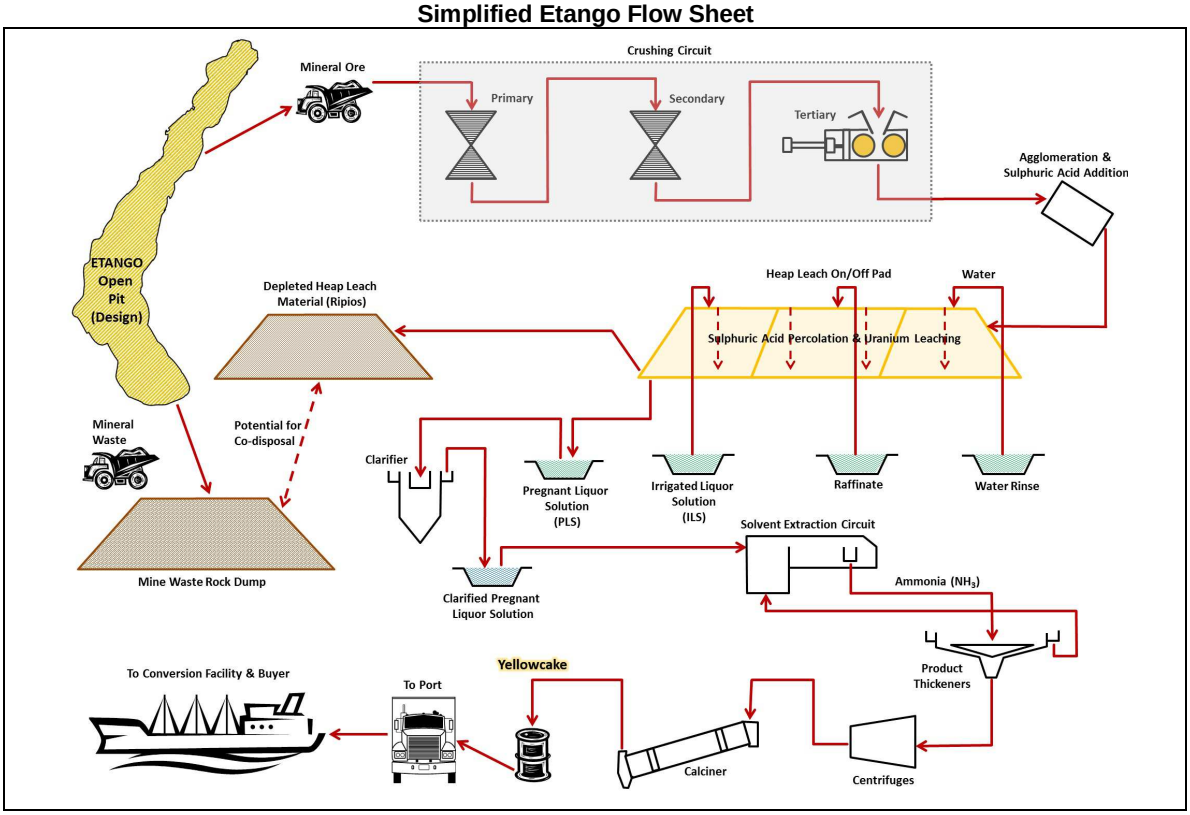
- The cribs and columns results show good correlation with respect to previous columns with a low differential in final extraction rate and acid consumption
- Cribs results are considered to be more representative, since the conditions are more representative to the agglomeration, stacking, irrigation and drainage methodology expected during a commercial heap leach operation and as such would provide a more accurate picture of the expected results for the full scale plant

1.7 PLANT AND INFRASTRUCTURE DESIGN

1.7.1 Processing

The process flowsheet (Figure 1.2) comprises a crushing circuit, reusable (on-off) heap leach pad for sulphuric acid leaching of the ore, and a uranium SX and recovery circuit to produce U_3O_8 yellowcake.

Figure 1.2
Simplified Etango Flow Sheet



Comminution

Ore is delivered to a gyratory primary crusher, followed by secondary cone crushing, and tertiary crushing by HPGR to produce the target P_{80} product size of 5.3mm.

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Agglomeration and Stacking

Crushed ore is transferred via fine ore bins to two agglomerating drums. Water, sulphuric acid and binder agent are added and the agglomerated ore is transferred to the heap leach stacking system.

The stacking system comprises an overland conveyor and a fixed stacking conveyor with tripper to transfer ore to a stacking bridge supported on a crawler undercarriage. The maximum stacking height is 5m.

The reclaim system is of similar design, fed by a bucket wheel excavator. The leached residue (Ripios) is transferred by overland conveyors to the Ripios stacking system.

Ripios Stacking

A tripper conveyor allows Ripios to be transferred to the Ripios pad boom stacker that places the Ripios onto the unlined drain Ripios pad.

Drainage from the Ripios pad is collected in the Ripios emergency pond and recycled to the heap leaching system. The pond has a double high density polyethylene (HDPE) liner with drainage net in between for leak detection.

Heap Leach Management

The heap leach pad is composed of a compacted sub-base layer, a low permeability clay impregnated geotextile lining and a HDPE liner. Draincoil piping rests on the HDPE layer and is overlain with a drainage layer.

The ore is stacked in modules (52 in total), where each module represents one day of stacking. The first three modules are designed for stacking, ore rest and dripper installation. The next 15 modules are irrigated with intermediate leach solution (ILS). The liquor from these modules produces the PLS, which is pumped to the SX circuit for uranium recovery. The subsequent 15 modules are irrigated with raffinate solution, which drains to the ILS pond and is recirculated to the heap to build up uranium tenor. Thereafter there are 12 modules for draining and rinsing. Solution from these modules is recirculated to the rinse modules and also to the ILS and raffinate as water make-up. The remaining modules are spares and used for dripper removal and reclaiming.

The ponds are designed for a residence time of 6 hours for the raffinate, ILS, and PLS ponds, and 4 hours for the rinse water pond. An emergency pond is provided to contain 24 hours drainage from the heap and a 24 hour maximum rainfall event run-off. The construction of the ponds is a clay-impregnated geotextile low permeability base liner overlain by a double HDPE liner with a drainage net for leak detection. For the rinse pond, a single layer HDPE liner overlies the clay-impregnated geotextile layer.

Solvent Extraction

PLS is pumped to a single train SX circuit which consists of two extraction, two scrubbing, four stripping, one organic regeneration and one crud removal stage. Bateman pulsed columns are used for extraction and conventional mixer/settlers are used for all other contacting duties.

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Precipitation, Calcination and Packaging

SX loaded strip liquor is pumped to the precipitation circuit where anhydrous ammonia raises pH to ~7, causing precipitation of ammonium diuranate (ADU) which is thickened, whilst barren liquor is clarified to remove suspended ADU solids.

ADU thickener underflow solids are dewatered further to remove soluble impurities, washed in centrifuges and then calcined. Calcined solids (U_3O_8) are discharged from the furnace and transferred to the product bin.

From the product bin, U_3O_8 is measured into 200L steel drums and periodically loaded into 20ft sea containers for transport to customers.

Reagents and Services

Reagents comprise sulphuric acid, hydrogen peroxide, diluent, extractant and modifier, ferrous sulphate, coagulant, sodium hydroxide, sodium carbonate, anhydrous ammonia, binding agent and flocculant. Engineering design has allowed for delivery, receipt, preparation, storage and distribution around the plant. Storage for 30 days has been catered for in all cases.

The dominant reagent by volume, bulk concentrated sulphuric acid (98% by weight), is shipped in and transferred to storage tanks at the port of Walvis Bay. From there, the acid is transported to site and transferred to four storage tanks, this being sufficient for 28 days of operations.

Services include water and air provided to the individual process plant areas or reticulated throughout the plant in the case of plant and instrument air, drinking and safety showers water and fire water.

The general layout is shown in Figure 1.3.

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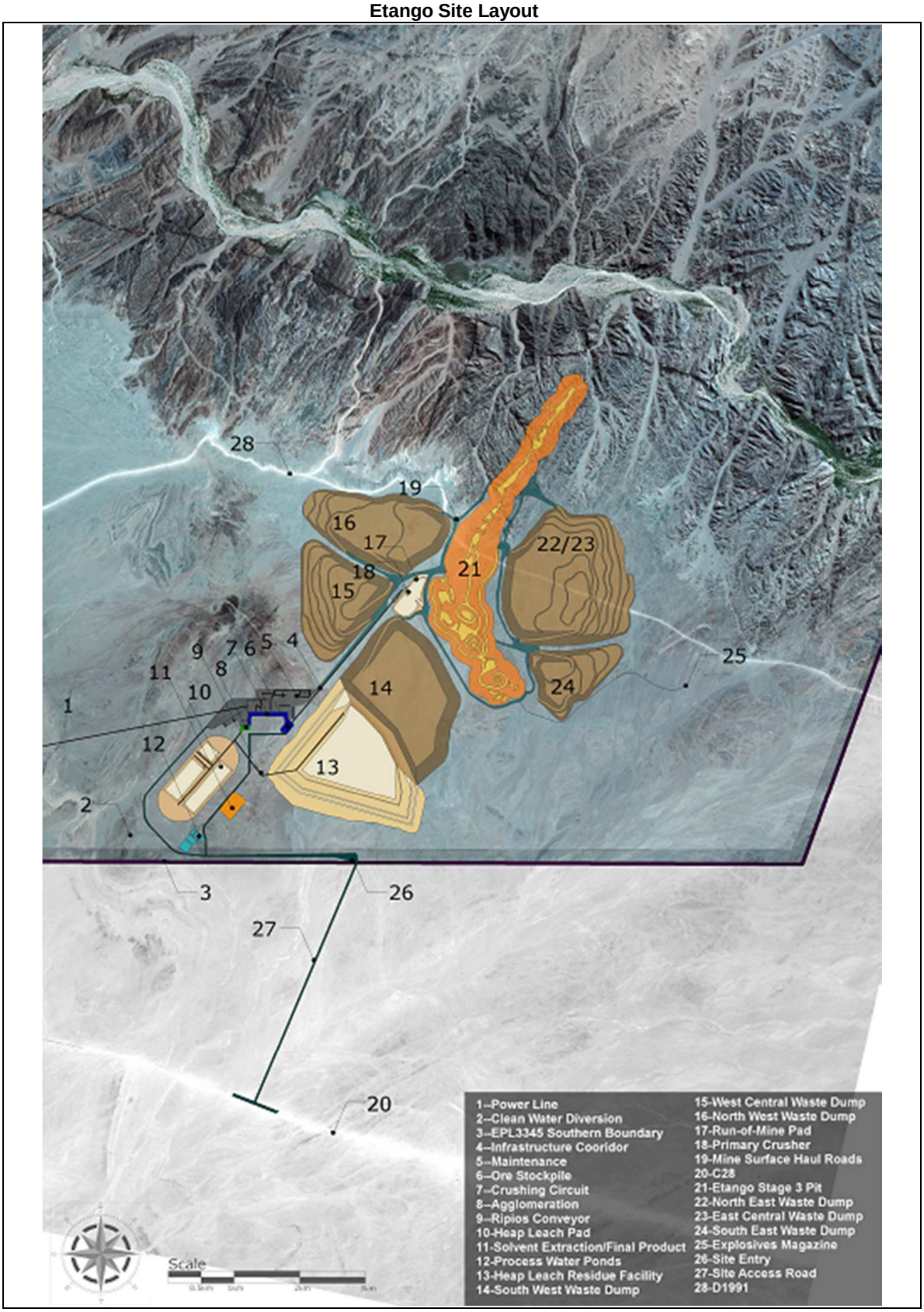
Figure 1.3
Etango Site Layout



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1.7.2 Infrastructure

Power

Total project installed power is nearly 50MW. The Namibian power utility, NamPower, previously confirmed its ability to provide power to the Etango Project and has offered a 30MVA supply for the Project. NamPower has been approached regarding the increased requirement.

The power system, supplied and installed by NamPower, is to be fully operational 24 to 30 months from the signing of the Power Supply Agreement between Bannerman and NamPower.

Construction power supply will be via temporary generator sets on site.

Water

Total operating water consumption is estimated to be 4.72Mm^3/a, of which 70% is to meet process requirements. Supply is to be provided by NamWater using water pumped from a desalination plant north of Swakopmund to a reservoir on site.

During the construction phase, the water requirement of 860m^3/day will be trucked in until the permanent supply is available.

Roads

Access to the mine site will be via a 7km unsealed spur linking to the existing C28 gravel road that leads to the town of Swakopmund and thence to the port at Walvis Bay.

Accommodation

Facilities in the towns of Swakopmund, Walvis Bay and Arandis will support the Etango operations. Bannerman is considering ways of assisting in provision of housing in these towns to overcome the shortage of suitable existing accommodation.

A temporary construction camp to house a peak workforce of 1500 workers will be constructed on site, and sold at the completion of the development phase.

1.8 WATER MANAGEMENT AND WASTE DISPOSAL

SLR Environmental, trading as Metago Environmental Engineers (Australia) Pty Ltd (Metago), undertook DFS level design for management of site water and disposal of Ripios. The initial work involved option studies to determine the most efficient method of disposal, seepage and stormwater management, taking account of environmental impacts, operational issues and capital and operating costs (including closure costs).

The Etango Project is located in a part of Namibia characterised by low rainfall, high humidity and sparse vegetation. The average annual rainfall in the district is 0-50mm, but rainfall is dominated by rare, intense events of as much as 100mm in 24 hours. For design purposes a 1000-year event of 110mm over 24 hours is estimated, with a design storm intensity of 37.8mm/hr and duration of 12.5 minutes.

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Storm water flow rates and volumes were modelled, with assumptions regarding infiltration and evaporation rates based on general soil and climate knowledge for the area. Principal conclusions were:

- Relatively small amounts of surface water is generated due to low rainfall and high infiltration rates
- No substantial runoff is generated from waste dumps
- Large trenches and containment ponds are not required.

Design of management systems maintains separation of clean and dirty water, and incorporates a combination of 'V' drains, trenches, seepage cut-off trenches and storage ponds of suitable size.

Clean water is diverted east and west of the operation. Dirty water drainage and seepage are directed to containment ponds during operations, but, where possible, are redirected to the open pit during decommissioning. Elsewhere, evaporation ponds will be constructed as part of the final landform.

1.8.1 Waste Geochemical Characterisation

Samples of waste rock and two Ripios samples were submitted for geochemical investigation. Results indicated that:

- Waste rock is devoid of sulphides and the potential to produce acid is negligible. Weathering of this material will enhance the medium to long term neutralising potential of the waste rock, but also yield aluminium into the seepage stream
- Ripios samples showed sulphur/sulphide levels capable of producing acid, and relatively low neutralising potential ratios, indicating potential for acid drainage and metal leaching
- However, the groundwater is highly saline and the effect of seepage on groundwater quality should be insignificant.

1.8.2 Groundwater Chemistry

Analysis of samples during the 2012 DFS from 27 boreholes in the area has shown groundwater to be highly saline with various metal/metalloid levels exceeding the WHO DWQG (2008) for As, B, Fe, Mo, Pb, U. None of the natural groundwater sources is currently fit for domestic, agricultural, or livestock use.

According to modelling by ERM (ERM, 2012), the waste rock seepage is expected to blend in with the natural groundwater in a 1:100 (seepage:groundwater) volumetric ratio and will, therefore, have little effect on the saline and mineralised pre-mining quality of the natural groundwater. The groundwater model indicates that most of the seepage will migrate to the open pit, increasing as the pit deepens and the hydraulic gradient steepens. Smaller volumes are expected to move northwards to the Swakop River alluvium, and southwards along palaeochannels. The present Swakop River alluvial groundwater is naturally enriched with uranium and the proposed mining project is unlikely to increase this enrichment significantly.

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1.8.3 Ripios Seepage Modelling

A net percolation study and basal seepage analysis was undertaken, based on 18 years of climate data, including a 99mm 24 hour storm event. The basal seepage model was run over an 80-year period. The results of this work indicated:

- Percolation rates within the Ripios dump are low (<7mm/a)
- Seepage from the Ripios dump will be high for the initial layer, due to the water content within the Ripios. However, seepage will decrease significantly after placement of the basal layer
- Rainfall has minimal percolation into the Ripios dump, due to high evaporation rates and a salt crust forming on the surface.

1.8.4 Ripios Dump Design

The final footprint of the Ripios dump is approximately 3.6mm^2 with capacity of 151Mm^3. The Ripios dump design consists of two lifts of front stacks and back stacks at 20m high and 10m high, respectively. The final Ripios dump will be 60m high, in keeping with environmental requirements. The additional ore tonnage in the OS means that the Ripios dump will need to be extended by approximately 200m, which was considered during the updated capital estimate.

Internal stormwater 'V' drains and delineation bunds will be constructed to direct stormwater runoff to a localised collection pond.

Prior to detailed design, a full geotechnical investigation is recommended to be carried out to determine soil strata and foundation conditions below the Ripios dump.

1.9 CAPITAL COSTS

1.9.1 Mining Capital Costs

The majority of mining capital expenditures were derived by Bannerman from quotations obtained from major equipment suppliers such as Komatsu and Caterpillar, with the balance being derived from the 2012 DFS estimates.

Capital cost estimates include $131M in preproduction capital and $267M in sustaining capital. The capital cost estimate was based on Q3 2015 quotations and has been completed to an accuracy of ±15%.

1.9.2 Plant and Infrastructure Capital Costs

In 2012 the DFS comminution, heap leach plant and site infrastructure capital costs were estimated by Amec Foster Wheeler, with Bateman Engineering Pty Ltd (Bateman) now part of the Tenova SPA Group estimating the cost of the SX/metal recovery section of the plant in the 2012 DFS. Owner's costs to cover corporate, management and administrative costs, as well as capitalised pre-production operating costs, were supplied by Bannerman. Subsequently an updated estimate was prepared by Amec Foster Wheeler for Bannerman to align the 2012 DFS with 2015 pricing. This exercise provided Bannerman with an accurate (±20%) estimate of what the current Project would cost in 2015, but it does not constitute an updated DFS. The accuracy is well within the accuracy tolerance required in a pre-feasibility study (PFS).

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Total plant and infrastructure capital costs are estimated to be $636.2M as at Q2 2015.

The estimate includes Direct and Indirect costs, engineering accuracy provisions and costs for engineering, procurement and construction management (EPCM) by an independent contractor. No provision has been included for inflation. The estimate includes $61.3M for a Project or Owner's contingency which was subsequently removed by Bannerman.

1.9.3 Owner's Capital Costs

Owner's costs have been determined by Bannerman to be $38.9M, and include:

- Pre-production staff recruitment and training
- Owner's Project Team
- Corporate costs for the Perth office and costs for Swakopmund support
- Consultants
- Housing allowance (nominal $6M)
- Environmental site assessment and monitoring
- Insurance
- Sterilisation drilling and on-site metallurgical testing
- Closure costs

1.9.4 Total Project Capital Costs

These are summarised in Table 1.2. Pre-production capital costs total $792.7M, whilst there is a further requirement for $172.3M in working capital before positive cash flow occurs.

Table 1.2
Capital Cost Expenditure Summary
($M)

Area	Pre-production	Sustaining	Total
Mining	131.33	267.22	398.55
Process Plant	321.36	-	321.36
Site Infrastructure	74.79	4.51	79.31
External Infrastructure	46.00	0.75	46.75
Miscellaneous	37.77	-15.19	22.57
Indirects	142.51	-7.30	135.21
Owner's Costs	38.90	32.50	71.40
Owner's Contingency	-	-	-
Total Project	**792.65**	**282.49**	**1 075.14**

Sustaining capital of $282.49M allows for expanding the mining fleet as production levels increase, and for mining equipment replacement. Negative numbers relate to income from sale of the construction camp and recovery of first fill materials and reagents as they are recovered via operating costs at the end of the project.

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1.10 OPERATING COSTS

1.10.1 Mine Operating Costs

The total material movement as derived from the life of mine (LOM) mine production schedule was used to determine the mine equipment requirements over time. The total mining operating costs were estimated to be $1 934.7M as at Q3 2015.

A breakdown of the mine operating costs is provided in Table 1.3. Diesel costs are the largest single component of mine operating cost.

Table 1.3
Summary of LOM Mining Operating Costs

Item	Cost ($M)	Cost ($/t mined)	% of Cost
Fixed	383.2		20
Drill and Blast	412.7		21
Load and Haul (including ancillary equipment)	1 138.8		59
Total	**1 934.7**	**1.69**	**100**

1.10.2 Plant Operating Costs

The process operating costs reflect operation at a head grade as per the life of mine plan and the throughput rate as per the life of mine plan. The accuracy of this operating cost estimate is ±20% and reflects the plant operating at design capacity.

Operating costs are estimated at $6.79/t crushed on a LOM basis.

The various plant operating costs are summarised in Table 1.4. Acid is the largest single cost (25%), followed by other power and reagents (21% and 15%).

Table 1.4
Summary of Plant Operating Costs

Item	Cost ($M/a)	Cost ($t/ore)	Cost ($/lb U_3O_8)	% of Cost (%)
Acid	33.29	1.72	4.63	25.4
Reagents	19.10	0.99	2.66	14.6
Power	26.89	1.39	3.74	20.5
Labour	10.51	.054	1.46	8.0
Maintenance Materials	16.01	0.83	2.23	12.2
Water	15.94	0.83	2.22	12.2
Consumables	7.16	0.37	1.00	5.5
Miscellaneous	2.23	0.12	0.31	1.7
Total	**131.13**	**6.79**	**18.23**	**100.0**

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1.10.3 Owner's Operating Costs

Owner's operating costs total $18.40M annually, equivalent to $0.96 per tonne crushed (for the average LOM production rate), as summarised in Table 1.5.

Table 1.5
Summary of Owner's Operating Costs

Item	Annual Cost ($ M/a)	Unit Cost ($/t Crushed)
Corporate and Owner's Labour	9.72	0.51
Total Site Office Administration	0.23	0.01
Total Personnel Expenses	3.05	0.16
Total Insurances and Government Fees	4.25	0.22
Site-Catering Facilities	0.53	0.03
Environmental Monitoring	0.30	0.02
Total Transportation Costs	0.20	0.01
Community Relations / Corporate Responsibility	0.12	0.01
Total	**18.40**	**0.96**

Principal costs are for Corporate and Owner's Labour ($9.72M), Training ($3.1M) and insurances ($4.25M).

1.10.4 Total Project Operating Costs

Total operating costs for the Project are $14.15/t ore or $37.99/lb U_3O_8 over LOM (Table 1.6).

Table 1.6
Summary of Total Project Operating Costs

Item	Cost ($/t of ore LOM)	Cost ($/lb U3O8 LOM)	% of Cost
Mining	6.38	17.13	45.10
Process	6.79	18.23	48.00
Owners & Infrastructure	0.98	2.62	6.90
Total	14.15	**37.99**	**100.00**

1.11 PROJECT FINANCIAL MODELLING

1.11.1 Base Case

The Base Case model uses the mining and processing schedules and capital and operating costs. The uranium oxide price used is $75/lb. This price is within the range quoted by various banking institution as the forecast price for when the project will come into production in 2020. A state royalty of 3% and off-site costs of $1.10/lb are included.

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The key outputs from the financial model based on the above assumptions are reported for the first 5 years of full production of the modelled operation (shown as 2021 to 2026 excluding the ramp-up year of 2020) and for the life of mine (2020 to 2035) in Table 1.7.

Based on the above, at a throughput rate of 20Mtpa, the Project is modelled to produce between 6 to 10Mlb U_3O_8 per year. The average cash operating cost in the first 5 years of full production is estimated at \$32.99/lb U_3O_8 and over the life of mine is estimated at \$37.99/lb U_3O_8.

Pre-production capital is estimated to comprise \$792.7M of capital equipment, engineering, construction and development costs. In addition, approximately \$172.3M of working capital is required in order for the project to be funded to first positive operating cash flow. Over the life of the mine (LOM) there is additional mine fleet of \$267.2M, sustaining capital of \$0.75M, and mobile equipment is replaced in Year 8 for \$4.515M. Various capital items are modelled as being recouped over the LOM as follows: the temporary construction camp is assumed to be sold for \$7.3M in Year 1 of operations and the capital outlaid on first fills of \$15.2M is recouped in the final year.

The payback period is estimated to be approximately 4.4 years with the NPV of the project, at an 8% real discount rate, estimated to be \$419.1M after tax. The internal rate of return of the project is estimated at 15.3% after tax.

Table 1.7
Key Financial Model Outputs

	First 5 Full Production Years (Yrs 2 -6)	**Life of Mine**
Project Economics		
NPV (after tax) at a Discount Rate of 8% (\$M)	-	419.1
Internal Rate of Return (after tax) (%)	-	15.3%
Payback Period from Start of Production (Q1 2020)	-	4.4
Production		
Quantity Ore Treated (Mt)	99.6	303.3
Uranium Oxide Produced (t U_3O_8)	20 810	51 234
Uranium Oxide Produced (Mlb U_3O_8)	45.9	112.95
Revenue		
Average U_3O_8 Base Price (\$/lb U_3O_8)	75	75
Net Revenue (\$M, after Government royalties)	3 185	8 093
Operating Unit Costs		
On-Site Costs/tonne Ore Treated (\$/t ore)		
Mining	7.46	6.38
Processing	6.77	6.79
Owners costs (including infrastructure maintenance)	0.97	0.98
Total Operating Costs (\$/t ore)	**15.19**	**14.15**
Total Operating Costs (\$/lb produced)	**32.99**	**37.99**
Marketing, freight and conversion	1.10	1.10

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1.11.2 Sensitivity Analysis

The financial sensitivity analysis demonstrates that the economic performance of the Etango Project is most sensitive to changes in the uranium price, followed by operating costs. This is unsurprising given the large scale and relatively modest grade of the deposit.

The project is therefore affected by factors which have the greatest bearing upon cash operating margins. Accordingly, the highest sensitivity is to uranium prices, followed by sensitivity to operating costs and lastly to capital costs. Each component is discussed briefly below.

Sensitivity to Changes in U_3O_8 Prices

As noted, the Etango Project is most sensitive to changes in uranium prices. Negative movements of 10% and 20% from the base case assumption of \$75/lb U_3O_8 result in the post-tax NPV reducing from \$419.1M to \$170.0M and minus \$84.4M respectively.

Likewise, positive movements of 10% and 20% from the base case assumption of \$75/lb U_3O_8 produce significant changes in the post-tax NPV from \$419.1M to \$666.3M and \$913.5M respectively, the latter with a post-tax IRR of 22.4%.

A 20% increase in the U_3O_8 price reduces the payback period by 1 year (to 3.5 years) and a 20% decrease in the U_3O_8 price results in payback increasing to 8.5 years.

Sensitivity to Changes in Total Operating Costs

As noted above, given the large annual throughput of the project, the financial performance is also very sensitive to changes in total operating costs.

Increases of 10% and 20% in the base case cost assumptions produce significant adverse changes in the post-tax NPV from \$419.1M to \$285.4M and \$149.7M respectively, the latter with a post-tax IRR of 10.8%.

Likewise, cost reductions of 10% and 20% from the base case assumptions result in the post-tax NPV increasing from \$419.1M to \$552.9M and \$687M respectively, the latter with a pre-tax IRR of 19.4%.

A 10% decrease in operating costs reduces the payback period by 0.4 years (to 4 years) and a 10% increase in capital costs results in payback occurring in 4.8 years.

Sensitivity to Changes in Capital Costs

The sensitivity of the Etango Project to changes in capital costs is driven by the scale and timing of the up-front construction and development expenditure. For the purposes of the sensitivity analysis, capital costs excluding working capital were varied in accordance with the nominated percentage changes. Working capital is a function of operating expenditure and lagged revenues, and has therefore not been varied in the capital cost sensitivity analysis.

Increases of 10% and 20% in the base case capital cost assumptions produce adverse changes in the post-tax NPV from \$419.1M to \$350.5M and \$280.9M respectively, the latter with a pre-tax IRR of 12.3%.

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Likewise, capital cost reductions of 10% and 20% from the base case assumptions result in the post-tax NPV increasing from $419.1M to $487.8M and $556.4M respectively, the latter with a post-tax IRR of 19.3%.

A 10% decrease in capital costs reduces the payback period by 0.4 years (to 4 years) and a 10% increase in capital costs results in payback period increasing by 0.3 years (to 4.7 years).

1.12 ENVIRONMENTAL AND PERMITTING

1.12.1 Environmental Approvals

Bannerman received Environmental Clearance in July 2012 for its plans to establish the Etango Project as a 20Mtpa heap leach operation as described in the 2012 DFS. The Environmental Clearance was valid for 3 years and expired in July 2015 upon which renewal of the Environmental Clearance was requested. The updated Environmental Clearance was granted on 11 November 2015, valid for 3 years.

The Environmental Clearance for the location and design of infrastructure ancillary to the Etango Project (including access roads, a water pipeline and power lines) was granted by the Ministry of Environment and Tourism on 11 February 2013 valid for 3 years. An application for renewal has been lodged.

1.12.2 Mining Licence Application

Bannerman submitted a mining licence (ML) application for the Etango Project in December 2009, based on the December 2009 PFS. The granting of the mining licence is subject to securing financing for the project.

1.13 PROJECT DEVELOPMENT.

A project development schedule has been outlined as part of the 2012 DFS, indicating completion of engineering design, procurement, transport and construction over a 32 month period with ramp-up to design tonnages after 50 months from commencement of early works (Table 1.8).

The schedule includes a contingency of 3 months, and is conditional upon the upgrade of access roads, establishment of the construction village and other basic infrastructure being in place to support the construction effort within 14 months.

Table 1.8
Project Development Milestones

Task	Month
Board Approval	0
Commence early works	1
Project approval, i.e. receipt of regulatory approvals/project financing	7
Commence site works	17
Commence commissioning	33
First shipment	40
Ramp-up to design tonnages	51

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The key drivers of the development schedule are Project approval followed by the timely delivery of long lead equipment with a number of long lead items such as mining haul trucks and the stacker, reclaimers and conveyors associated with the heap leach system having current delivery times greater than 18 months.

1.14 PROJECT RISK ASSESSMENT

A range of economic, engineering and other technical risks to the Project have been considered. Those risks assessed as Moderate to High, High or Major are summarised in Table 1.9 arranged in general order of likelihood and importance.

Table 1.9
Non-Resource/Mining Economic and Technical Risk Assessment

Item	Assessed Risk to Project
U_3O_8 price	High – Major risk to Project
Power supply not available	High – Major risk to Project
Mining equipment under-performance	Moderate to High
Capital cost over-run	Moderate to High
Operating cost overrun – diesel	Moderate to High
Operating cost over-run – power	Moderate to High
Operating cost over-run – acid	Moderate to High

The two highest risks to the project are considered to be:

- A long-term contract price of \$75/lb U_3O_8 has been assumed in the OS, which is at the top end of the range of current projections. Most market analysts expect the fundamentals of the uranium market to improve and the uranium price to increase from current long-term levels of approximately \$44/lb to levels around \$75/lb U_3O_8 over the next 3 to 5 years. Bannerman intends to seek a strategic partnership with an established industry end-user such that specified quantities of future production can be sold at minimum prices consistent with levels of \$75-80/lb.
- Non-availability of power supply: Bannerman believes that the additional supply will be developed based on statements by NamPower, but this remains a serious risk until NamPower has commenced construction of new capacity

In addition a number of opportunities have been identified during the OS.

- Generally the larger scale demonstration plant results are similar or better than those obtained in previous smaller scale test work performed under similar conditions. This will be further investigated in 2016.
- Preliminary geotechnical review suggests that there may be an opportunity to steepen the open pit slope angles in selected areas of the open pit.
- Benchmarking the Etango flow sheet with similar projects suggests that there is opportunity to further reduce capital. This will be further investigated in 2016.

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1.15 CONCLUSIONS

The results of the OS indicate that the Etango Project is technically feasible to develop as a simple, large open pit mining, heap leach and SX recovery operation. No technical or environmental fatal flaws have been identified.

The OS has significantly enhanced the project economics when compared to the DFS. These improvements are largely attributed to improvements made in the mine planning and the more favourable capital and operating costs. The latter is a function of the cost environment normalising following a decade of exceptionally high input costs caused by the China driven commodities boom.

The grade of the deposit is relatively low, throughput is high and the Project is capital intensive. Consequently, a higher uranium price is required than prevails currently. However, the uranium price required to incentivise the project, of \$75/lb U_3O_8, is in line with those forecasted by a number of independent analysts.

The results of test work conducted in the demonstration plant continue to re-inforce the assumptions made in the original DFS and OS. A number of opportunities have been identified during the OS that will be further investigated in 2016.

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2 INTRODUCTION

2.1 BANNERMAN RESOURCES LIMITED

This Technical Report has been prepared for Bannerman Resources Limited (Bannerman), a public company listed on the Toronto and Australian Stock Exchanges. Bannerman's corporate office is Unit 1, 2 Centro Avenue, Subiaco, Western Australia, 6008.

2.2 SCOPE OF WORK

Following completion of the DFS geology model, limited additional drilling was undertaken at the Etango deposit. This information was incorporated into a new geological model, which in addition, was updated to reflect the selectivity associated with the proposed mining method which incorporates radiometric truck scanning. This geological model in turn formed the basis for an update of the mine plan. In addition, large scale metallurgical test work was undertaken to verify the metallurgical parameters at an industrial scale. Amec Foster Wheeler was then commissioned to update capital and operating cost estimates to ensure the project reflects changes that have occurred in cost parameters since completion of the DFS. Bannerman updated the financial cash flow model to reflect all of the above changes. The Optimisation Study scope required Capital and Operating costs to be estimated to an accuracy of ±20%.

Optiro contributed the 2015 Mineral Resource model.

Mine Technics conducted a geotechnical review of the updated pit design to ensure that it honours the slope design criteria as defined in the DFS.

Bannerman publicly announced the results of Optimisation Study in October 2015, and has assembled this Technical Report, incorporating inputs from the above-noted parties, in support of the public announcement.

This report complies with disclosure and reporting requirements set forth in the Toronto Stock Exchange Listing Rules, Canadian National Instrument 43-101, Companion Policy 43-101CP, and Form 43-101F1. Mineral Resource and Mineral Reserve classifications conform to those adopted by CIM Council in November 2004 and revised in 2010. The classification and reporting of resources and reserves is also consistent with the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' of December 2012 as prepared by the Joint Ore Reserves Committee of The Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia (JORC Code 2012).

2.3 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 Bannerman regarding the geology, drilling, sampling and other exploration procedures and processes adopted by the Company
- Metallurgical test work undertaken by recognised test work laboratories, notably ALS Ammtec in Perth, Western Australia, and Bureau Veritas in Swakopmund, Namibia
- Information from Leon Fouché, Bannerman Study Manager, in relation to past history and previous studies on the Etango Project

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- Field observations, reports and data obtained during field trips 2015 by Mr Ian Glacken, Optiro and Mr Leon Fouché, Bannerman Resources
- Etango Project Environmental and Social Impact Assessment and Environmental Management Plan prepared by ERM in April 2012, including appendices by Metago Environmental Engineers
- RPS Aquaterra completed groundwater modelling to define groundwater conditions in the open pit area, and the effects from mining
- Various published historic, technical and scientific papers and reports
- Digital exploration data
- The Etango Definitive Feasibility Study and published information relevant to the Etango Project area and the region in general
- The Etango Optimisation Study.

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

2.4 PARTICIPANTS

The following QPs have been involved in compilation of the NI43-101 report:

- Bannerman:
  - Leon Fouché – Study Manager. Responsible for Sections 2, 3, 4, 5, 6, 15, 16, 19, 20, 22 and 23.
- Amec Foster Wheeler:
  - Peter Nofal – Manager, Studies. Responsible for Sections 13, 17, 18 (excluding mining costs) and 21.
  - Dean David – Process Consultant. Responsible for those aspects of Sections 13 and 17 of the report relating to ore comminution.
- Optiro:
  - Ian Glacken – Director Geology. Responsible for Sections 7 – 12 and 14.

Each of the abovementioned QPs is individually responsible for relevant parts of Section 1 (Summary), Section 24 (Other Relevant Data and Information), Section 25 (Interpretations and Conclusions), Section 26 (Recommendations) and Section 27 (References).

2.5 SITE VISIT

Leon Fouché undertook a site visit to the Etango Project in August 2015.

Ian Glacken of Optiro undertook a site visit to the Etango Project in September 2015.

Amec Foster Wheeler Personnel, Dean David and Peter Nofal, participated in site visits in April 2011 and June 2011, respectively, to assess process, plant engineering and infrastructure issues.

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2.6 QUALIFICATIONS AND EXPERIENCE

Leon Fouché, overall compiler of the report, lead QP and the professional responsible for the geotechnical, mining and Mineral Reserve parts of this report, is a full time employee of Bannerman Resources and a mining engineer with over 20 years' experience. Mr Fouché holds a Bachelor degree, majoring in Mine Engineering and a Masters' in Business Administration. Mr Fouché is a Fellow of the Australasian Institute of Mining and Metallurgy, and has the appropriate relevant qualifications, experience to be generally considered a Qualified Person as defined in the Instrument.

Amec Foster Wheeler is an international engineering company, with a strong world-wide background in mineral resource engineering partly through its purchase of the Australian company GRD Minproc Limited (GRD Minproc) in 2009. GRD Minproc specialised in resource and mining studies, process design, engineering, cost estimation and feasibility studies for the minerals industry, focusing on gold, base metals, iron ore, mineral sands and uranium, including extensive involvement with the Langer Heinrich uranium project in Namibia.

- Dean David, Technical Director Process for Amec Foster Wheeler, is responsible for process test work and design relating to the proposed comminution circuit. Mr David has over 30 years of experience in mineral processing research, operations, management and consulting. He has visited the site twice including once in 2015. Mr David has a BSc. In Metallurgy, is a Fellow of the Australasian Institute of Mining and Metallurgy and has the appropriate relevant qualifications, experience and independence to act as a Qualified Person as defined in the Instrument
- Peter Nofal, Manager Studies for Amec Foster Wheeler, has visited the site, and is responsible for those sections of the report relating to process test work, engineering design and costing, with the exception of comminution and also of mine design and costs. Mr Nofal has a BSc in Engineering and a B.Comm majoring in Business Economics, and is a Fellow of the Australasian Institute of Mining and Metallurgy. Mr Nofal has the appropriate relevant qualifications, experience and independence to act as a Qualified Person as defined in the Instrument.

Optiro is an integrated Australian-based consulting and advisory firm which has been providing services and advice to the international mineral industry and to financial institutions since 2008.

- Ian Glacken is a geologist and Geostatistician who is Director of Geology at Optiro. Mr Glacken visited the Etango site in September 2015. Mr Glacken has degrees in geology (honours), masters' degrees in mining geology and geostatistics and qualifications in computing. He has over 30 year's international experience in the mining industry after graduation. Mr Glacken has the appropriate qualifications and experience to be considered as a Qualified person as defined in the Instrument.

2.7 ABBREVIATIONS

Quantities are generally stated in SI (International System of Units) metric units, including metric tons (tonnes, t), kilograms (kg) or grams (g) for weight; kilometres (km), metres (m), centimetres (cm) and millimetres (mm) for distance; square kilometres (km^2) or hectares (ha) for area; and parts per million (ppm) for uranium oxide grade (ppm U_3O_8).

A listing of abbreviations used in this report is provided in Table 2.1.

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Table 2.1
List of Abbreviations

Abbreviation	Description
%	Percent
$	United States of America dollars
$/a	Dollars per annum
$/lb	Dollars per pound
$/t	Dollars per tonne
‴	Inches
µ	Microns
3D	three dimensional
AAS	atomic absorption spectrometry
ADU	Ammonium diuranate
ASEC	A. Speiser Environmental Consultants
Bcm	bank cubic metres
Ca	Calcium
CC	correlation coefficient
Cm	Centimetre
Cps	Counts per second
CV	coefficient of variation
DDH	diamond drill hole
DFS	Definitive Feasibility Study
Epangelo	Epangelo Mining Company
EPCM	Engineering, procurement and construction management
EPL	Exclusive Prospecting Licence
ERM	Environmental Resources Management
ESIA	Environmental and Social Impact Assessment
ESMP	Environmental and Social Management Plan
G	Gram
g/m^3	grams per cubic metre
g/t	grams per tonne
GRD Minproc	GRD Minproc Limited, now Amec Foster Wheeler
h	hour
ha	Hectares
HARD	half the absolute relative difference
HDPE	high density polyethylene
HSEC	Health, Safety, Environment and Community Plan
K	Potassium
NQ	size of diamond drill rod/bit/core
HPGR	High pressure grinding rolls
hr	Hours
HRD	half relative difference
ILS	Intermediate leach solution
ISO	International Standards Organisation

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Table 2.1
List of Abbreviations

Abbreviation	Description
kg	Kilogram
kg/t	kilogram per tonne
km	Kilometres
km²	square kilometres
kW	Kilowatts
L	Litre
LOM	Life of mine
LUC	Localised Uniform Conditioning
M	Million
m	Metres
Ma	million years
MARC	Maintenance and repair contracts
MDRL	Mineral Deposit Retention Licence
Mg	Magnesium
mL	Millilitre
ML	Mining Licence
Mlb	million pounds
mm	Millimetres
Mt	million tonnes
Mtpa	million tonnes per annum
N$	Namibian dollars
N (Y)	Northing
Na	Sodium
Nb	Niobium
NEPL	Non-Exclusive Prospecting Licence
Ni	Nickel
NPV	net present value
NQ_2	size of diamond drill rod/bit/core
ºC	degrees centigrade
OK	Ordinary Kriging
OS	2015 Optimisation Study
Pd	Palladium
PFS	Preliminary Feasibility Study
PFSU	PFS Update
PLS	Pregnant leach solution
ppb	parts per billion
ppm	parts per million
psi	pounds per square inch
PVC	poly vinyl chloride
QAQC	Quality assurance, quality control
QC	quality control

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Table 2.1
List of Abbreviations

Abbreviation	Description
QQ	quantile-quantile
RAB	Rotary Air Blast
RC	reverse circulation
RL	Reconnaissance Licence
RL (Z)	reduced level
RQD	rock quality designation
SD	standard deviation
SEA	Strategic Environmental Assessment
SG	Specific gravity
Si	Silica
SI	International System of Units
SMU	selective mining unit
t	Tonnes
t/m^3	tonnes per cubic metre
Th	Thorium
tpa	tonnes per annum
U	Uranium
$	United States of America dollars
U_3O_8	uranium oxide
w:o	waste to ore ratio
XRF	x-ray fluorescence analysis

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3 RELIANCE ON OTHER EXPERTS

Several experts with qualifications that fall outside the definition of Qualified Person under the NI43-101 regulations have contributed information relied upon by Bannerman, Amec Foster Wheeler and Optiro in preparation of the Technical Report. References to specific input are carried in Section 27.

VBKom Namibia employed experienced and qualified mining engineers to conduct pit optimisation, pit design and mine scheduling activities under the supervision of Mr Leon Fouché of Bannerman. VBKom Namibia is experienced in mining projects in Namibia in particular uranium projects of the style of mineralisation of the Etango Deposit.

Mine Technics employs experienced geotechnical engineers with particular experience in mining projects in Namibia, in particular uranium projects of the style of mineralisation of the Etango Deposit.

Mineral process test work has been undertaken by several well-established, competent and well-recognised laboratories as identified in the body of this document, primarily Ammtec Ltd of Perth, Western Australia, and Bureau Veritas of Swakopmund, Namibia. Mineralogical examinations using QEMSCAN and SEM techniques were undertaken by qualified personnel at University of Witwatersrand. Current heap leach testing and reporting is being conducted by Bannerman personnel with assay results reported by various laboratories as mentioned above. Amec Foster Wheeler has relied on the results of test work reported by these operators.

Environmental and Social Impact Assessment has been undertaken by qualified professionals employed by A. Speiser Environmental Consultants cc (ASEC) and Environmental Resources Management (ERM) and its sub-consultants. Both companies are highly experienced in environmental and social impact evaluation/analysis for mining projects in Southern Africa, and Amec Foster Wheeler has relied on the results and conclusions from their respective studies in this Technical report.

ERM and RPS Aquaterra of Perth, Western Australia, employed experienced and qualified hydrologists to undertake assess hydrogeological conditions and undertake modelling of the groundwater regime surrounding the proposed open pit. Bannerman has relied on the results of this work as part of the open pit design.

SLR Consulting Australia Pty Ltd, trading as Metago Environmental Engineers (Australia) Pty. Ltd. undertook or managed work related to waste and Ripios characterisation, seepage, surface and groundwater management and Ripios dump design. The company employed well-qualified professionals to undertake these studies, and has relied on their findings and estimates of quantities of materials in preparing this Technical Report.

The financial model has been prepared for Bannerman by Mr Leon Fouché, a MBA qualified mining engineer employed by Bannerman.

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4 PROPERTY DESCRIPTION AND LOCATIONS

4.1 INTRODUCTION

Bannerman holds an exclusive prospecting licence (EPL) over the Etango Project within the central Swakopmund district of Namibia, through an 80%-owned Namibian-registered subsidiary company (Bannerman announced on the 11 November that, subject to shareholder approval, it will acquire the remaining 20%). This district hosts one of the world's largest open cut uranium mines at Rössing (majority owned by Rio Tinto), as well as Paladin Resources Limited's Langer Heinrich uranium operation. In addition the Husab uranium mine is currently under construction.

The Etango EPL contains a number of identified uranium prospects and uranium anomalies. The Etango Project itself is based around resources in the three main identified prospects (Anomaly A, Oshiveli and Onkelo), while additional resources have been identified at the Ondjamba and Hyena prospects.

4.2 BACKGROUND INFORMATION ON NAMIBIA

Namibia is a stable, independent republic with a total surface area of approximately 825,000km², situated north of South Africa, west of Botswana and south of Angola. It is bordered to the west by the Atlantic Ocean (Figure 4.1). Namibia forms part of the Southern African Region.

Figure 4.1
Geography of Namibia

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Namibia gained independence from South African mandate on 21 March 1990, following multi-party elections and the establishment of a constitution. This independence was the outcome of a war fought by the South West Africa People's Organisation (SWAPO), against South African rule, that commenced in 1966 and a United Nations peace plan for the region that was agreed in 1988. The inaugural President, Sam Nujoma, served for the first three terms (14 years). The current (3rd) president Hage Geingob was elected president in November 2014 in a landslide victory after Hifikepunye Pohamba stepped down after serving two terms. Namibia was the first country in the world to incorporate the protection of the environment into its constitution.

The capital city of Windhoek has a population of 368,000 and is located in the Khomas Region in the centre of the country. The largest harbour is located at Walvis Bay, on the central west coast, south of Swakopmund.

The country is mostly arid or semi-arid, comprising a high inland plateau bordered by the Namib Desert along the coast and by the Kalahari Desert to the east.

The population comprises approximately 87.5% indigenous people, 6% people of European descent and 6.5% of mixed origin. About 50% of the population belong to the Ovambo tribe and 9% to the Kavangos tribe. Other ethnic groups include the Herero (7%), Damara (7%), Nama (5%), Caprivian (4%), Bushmen (3%), Baster (2%) and Tswana (0.5%).

The official language is English; however, Afrikaans is the common language for most of the population and German is spoken by one-third of the population. Various indigenous languages are also spoken, including Oshivambo, Herero and Nama. According to World Bank standards, 82% of the population is literate.

The economy is heavily dependent on the extraction and processing of minerals for export. Mining accounts for approximately 11.5% of GDP. Significant operating mines are present at Rössing (uranium), Langer Heinrich (uranium), Skorpion (zinc), and Navachab (gold), while a significant quantity of diamonds are produced from on- and off-shore diamond fields. Namibia also has important fishing and cattle industries, and a traditional subsistence agricultural sector.

Namibia is serviced by a network of sealed highways connecting Windhoek with the coast at Walvis Bay, and with Botswana, Angola and South Africa. Generally unsealed but well-maintained roads provide regional access throughout Namibia. Power is available via an extensive regional electricity grid originating in South Africa. A railway line extends from the port of Walvis Bay to Tsumeb, where a copper smelter is in operation. Mobile phone communication is well established near most population centres.

Water is sourced by industry and communities from underground aquifers and, recently, from a desalination plant constructed on the coast to the north of Swakopmund. The Government water authority, NamWater, provides assistance in the development of water resources for existing and potential new users.

4.3 MINERAL TENURE

In Namibia, all mineral rights are vested in the State. The Minerals (Prospecting and Mining) Act of 1992 regulates the mining industry in the country. The Mining Rights and Mineral Resources Division in the Directorate of Mining is usually the first contact for investors, as it handles all applications for and allocation of mineral rights in Namibia.

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An individual Exclusive Prospecting Licence (EPL) can cover an area of up to 1000km² and the specific mineral group being explored for must be stated. According to Section 140 of the Minerals (Prospecting and Mining) Act, 1992A, Part 5, uranium mineralisation is classified under the nuclear fuel minerals group.

An EPL is valid for an initial term of 3 years, with two renewals of 2 years each, plus additional periods with relevant ministerial approval. The size of the EPL should be reduced after 3 years and the size of the reduction is at the discretion of the Mining Commissioner. There may be scope, if the Commissioner sees reason, to waive the reduction of the size of the EPL's after the initial 3 year period of the licences. There is currently no set reduction size, and an approved Mining Licence may count as a reduction in size of the EPL.

Section 67 of the Minerals (Prospecting and Mining) Act, 1992A details the rights of the holder of an EPL. These include entitlement to carry out prospecting (in respect of the mineral group specified in the licence) and to remove mineral samples (except for sale or disposal and other than controlled minerals).

Other licence types include:

- Non-Exclusive Prospecting Licence (NEPL) – valid for 12 months and permitting non-exclusive prospecting on any open ground which is not restricted by other mineral groups
- Reconnaissance Licence (RL) – which allows remote sensing techniques and is valid for 6 months
- Mineral Deposit Retention Licence (MDRL) – allowing the prospector to retain rights to mineral deposits that are uneconomic to exploit immediately, for future mining operations. These are valid for up to 5 years and can be renewed subject to work and expenditure obligations for up to 2 years at a time
- Mining Licence (ML) – which allows the holder to carry on mining operations. This can be awarded to accredited agents, companies registered in Namibia or any Namibian citizen. It is valid for life of the mine, or an initial period of up to 25 years, and is renewable for successive periods of up to 15 years.

Granting of licences is determined by a committee of the Ministry of Mines and Energy, such granting being based on the committee's perception as to the ability and intention of the applicant to complete exploration as outlined in the licence application, and the validity of the proposed program to determine resources. Each licence must outline commodities of interest (in this case 'Nuclear Fuels' covers uranium) and the licence granted only pertains to those commodities. Grant determination generally takes at least 6 months from the time of application.

An environmental contract must be completed with the Department of Environment and Tourism by applicants for EPLs, MDRLs and MLs. Environmental impact assessments (where relevant) must be made with respect to land disturbance, protection of flora and fauna, water supply, drainage and waste water disposal, air pollution and dust generation.

4.4 PROJECT LOCATION

The Etango Project is located approximately 41km (by road) east of the regional town of Swakopmund and approximately 73km (by road) northeast of the deep-water port of Walvis Bay (Figure 4.2).

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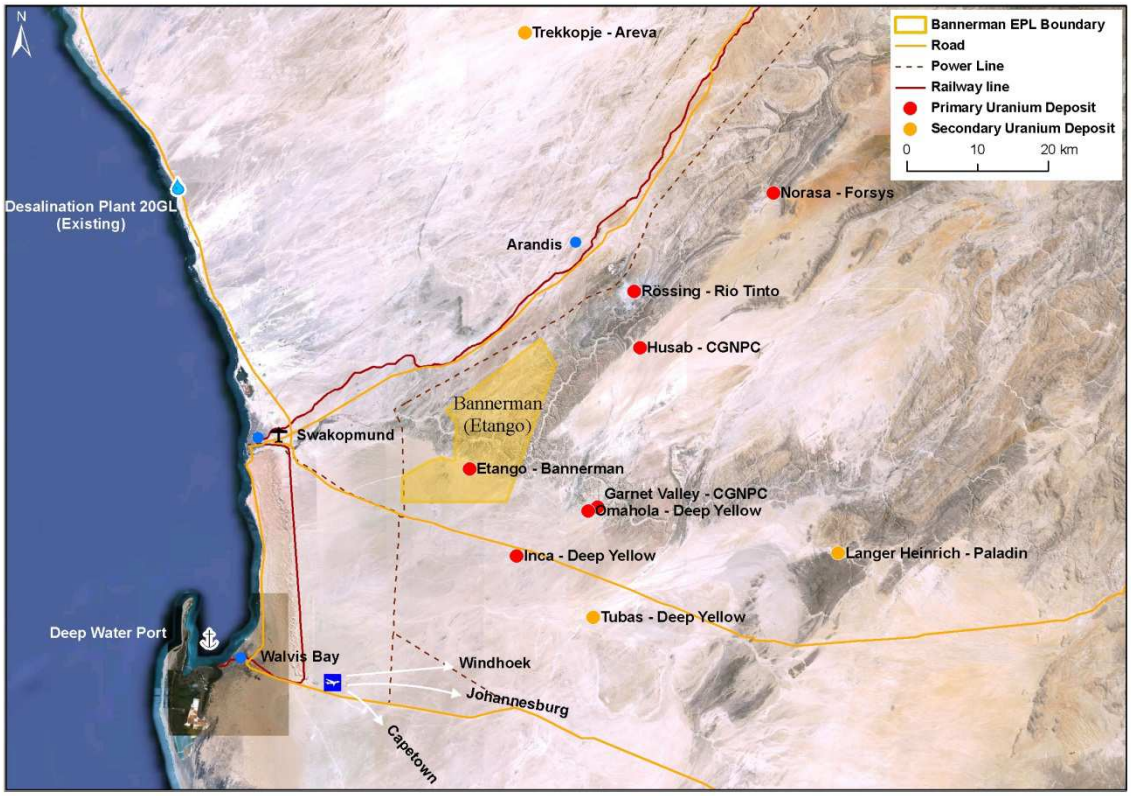
Figure 4.2
Etango Project – Project Location Plan



A sealed highway (C14) connects Swakopmund to Walvis Bay, while sealed highway B2 connects Swakopmund to the capital city of Windhoek. Access to the Etango Project from Swakopmund is gained via the B2 highway and then the partially sealed/unsealed road C28, thence by the well-maintained unsealed road D1991 into the Namib-Naukluft National Park area.

The Etango Project is situated on the Namib peneplain approximately 5km south of the Swakop River. To the north of the peneplain, erosion associated with the Swakop River has resulted in deeply incised gullies.

4.5 TENEMENT STATUS

4.5.1 Licences

The Etango Project EPL 3345 is owned by the Namibian company Bannerman Mining Resources (Namibia) (Pty) Ltd (Bannerman Namibia). Bannerman owns 80% of Bannerman Namibia, while the remaining 20% is held by the original vendor. Bannerman has reached an agreement to acquire the remaining 20% minority interest from the current owners (represented by Mr Clive Jones). The transaction, subject to shareholder approval, is expected be concluded by 31 December 2015.

EPL 3345 was granted to Turgi Investments (Pty) Ltd (Turgi) with effect from 27 April 2006 to explore for Nuclear Fuels. Renewals have been granted extending tenure to 26 April 2015. An application for a further renewal was lodged on 26 January 2015 and is expected to be renewed in due course.

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The delayed renewal is not deemed to be an issue as Regulation 71 (3) (a) from the Minerals (prospecting and Mining) Act (Act 33 of 1992) states "an exclusive prospecting licence shall not expire during a period during which an application for the renewal of such licence is being considered, until such application is refused or the application is withdrawn or has lapsed, whichever occurs first..." still applies.

EPL 3345 is now 24 326 ha in size and the tenement schedule is included as Table 4.1 and tenement coordinates as Table 4.2. Figure 4.3 shows the outline of EPL 3345.

Table 4.1
Etango Project – Tenement Schedule

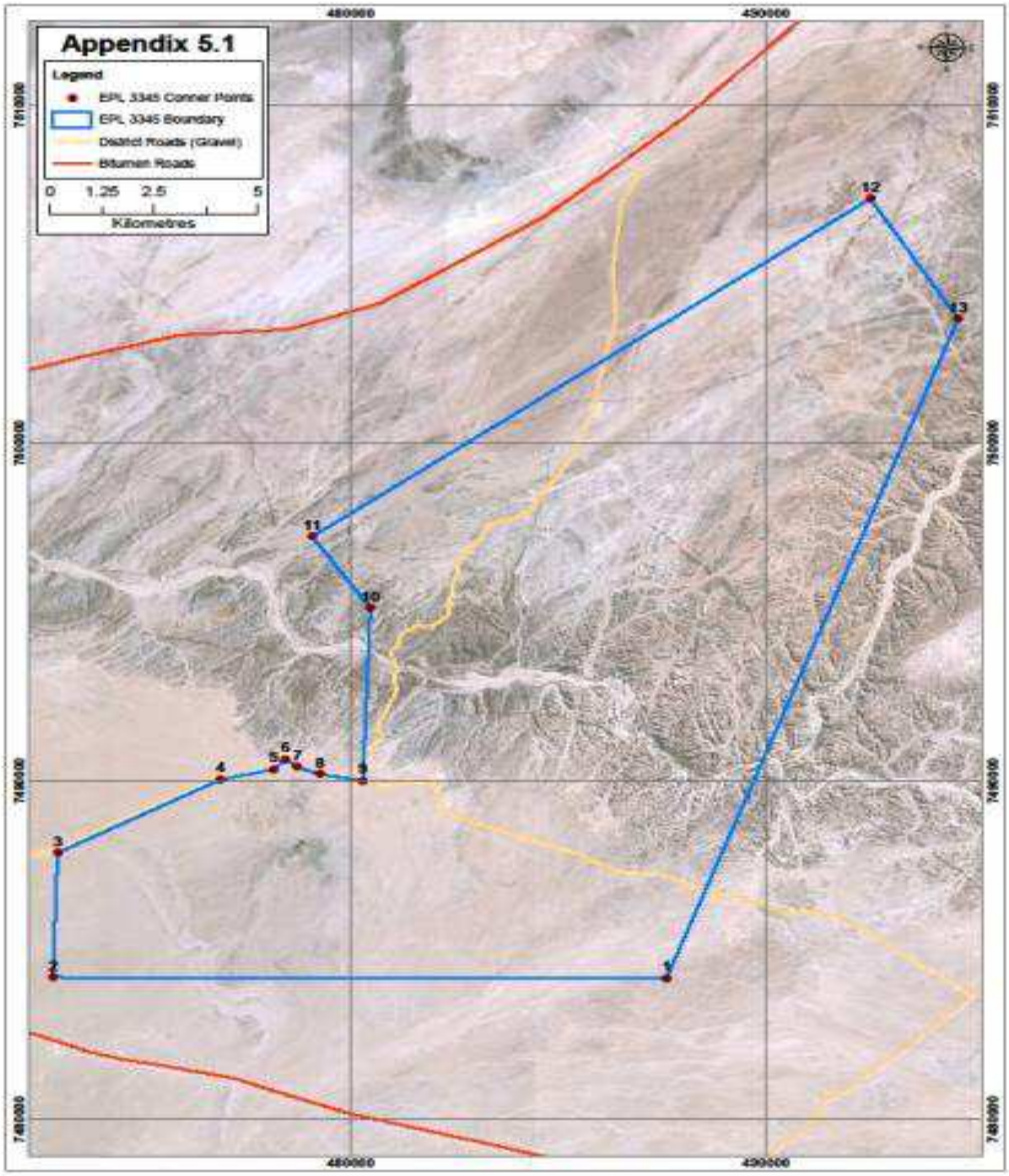
Tenement Type	Tenement No.	Grant Date	Holder	Area (ha)	Minimum Expenditure 2015 (N$)	Minimum Expenditure 2016 (N$)
EPL	3345	27.04.2006	Bannerman Mining Resources (Namibia) (Pty) Ltd	24 326	7,550,000	7,640,000

Table 4.2
Etango Project – Tenement Coordinate Summary

	Point	Latitude^	Longitude^
EPL 3345 (Etango) Licence Area – 24,326ha	1	-22.48345173	14.74459553
	2	-22.48454238	14.82167082
	3	-22.53845976	14.86468342
	4	-22.53505101	14.86932801
	5	-22.57336466	14.84251864
	6	-22.56012272	14.86757698
	7	-22.51976334	14.91324166
	8	-22.57366601	14.94763130
	9	-22.74979035	14.87921802
	10	-22.74935995	14.73544175

^ Latitude and Longitude are in Bessel 1841 Spheroid.

Figure 4.3
Etango Project – EPL 3345 Tenement Outline



On 17 December 2008, Bannerman announced that Bannerman Namibia had entered into an agreement to settle litigation previously brought by a competing claimant, Savanna Marble CC (Savanna) and certain associated parties. Under the terms of the settlement agreement, Savanna agreed to discontinue its review application in the High Court of Namibia by which Savanna had sought a declaration that the grant by the Minister of Mines and Energy of Namibia of EPL 3345, on which the Etango Project is situated, was void. This settlement involves payments and the issue of shares to Savanna (as Bannerman has previously disclosed in public documents) and removed the threat to Bannerman's title to the Etango Project.

On 21 December 2009, Bannerman lodged an application for a Mining Licence (ML 161) over the Etango Project area with the Namibian Ministry of Mines and Energy. Bannerman continues to liaise with the Ministry regarding the grant of the Mining Licence.

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4.6 AGREEMENTS AND ROYALTIES

4.6.1 Third Party Royalty

On 11 November 2015 Bannerman announced that the company had, subject to certain conditions being satisfied including shareholder approval and the renewal of EPL 3345, entered into various agreements to gain 100% ownership of the Etango Project, eliminate the existing A$12 million corporate debt and raise net A$4 million to fund the operation of the heap leach demonstration plant program and corporate working capital requirements. The proposed elimination of the debt through part conversion into Bannerman shares and the issuance of a royalty will result in a 1.5% gross revenue royalty on the future gross revenue derived from the Etango Project.

4.6.2 Government Royalties

According to Section 114, Part 1(c) of the Minerals (Prospecting and Mining) Act, 1992A, a royalty rate of 'not exceeding five per cent, as may be determined by the Minister from time to time by notice in the Gazette, of the market value, determined as provided in subsection (3), of such mineral or group of minerals' will be payable. Section 114, Part 3, defines the market value as:

1) *Determined in accordance with any term and condition, if any, of the licence of the holder concerned; or*
2) *If no such term and condition exists, determined in writing by the Minister, having regard to the value agreed between the holder in question and the person to whom such mineral or group of minerals was sold or disposed of in an at arm's length sale and prices which were in the opinion of the Minister at the time paid on international markets for such mineral or group of minerals, less any amounts deducted in respect of fees, charges or levies which are in the opinion of the Minister charged on international markets.*

The mining royalty is currently stipulated by the Namibian Government to be 3% of revenue.

4.6.3 Namibian Government Acquisition of Interest

In 2008, the Government of Namibia established Epangelo Mining Company (Epangelo) as a private company wholly owned by the Namibian Government. The mission of Epangelo is 'To ensure national participation in the discovery, exploitation and benefit of Namibia's mineral resources whilst developing and consolidating a portfolio of high quality assets and services for the benefit of its stakeholders' (*Epangelo Mining Company, 2010*).

In April 2011, the Mines and Energy Minister announced in Parliament that future mining and exploration rights for strategic minerals, including uranium, would be exclusive to Epangelo. Established exploration and mining companies expressed concern about this announcement but were assured that their existing exploration and mining licences should be unaffected (*Business Report, 2011*). In recent months, Epangelo has announced partnerships with Namibia Rare Earths Limited and talks with PE Minerals (part owners of Rosh Pinah zinc and lead mine) and Extract Resources (developers of Swakop Uranium) for shares in the respective mines.

In April 2012, Bannerman announced signing of a binding Term Sheet with Epangelo to acquire an initial 5% interest and, upon a mine development decision, a further 5% interest in Bannerman's Namibian subsidiary. Epangelo had 4 months in which to complete due

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diligence into the Project and obtain the necessary acquisition finance (approximately A$3.9M). Epangelo elected not to pursue the opportunity.

4.7 ENVIRONMENTAL LIABILITIES AND PERMITTING STATUS

4.7.1 Existing Liabilities

There are no identified existing environmental liabilities on the property.

4.7.2 Permit Requirements

The southern portion of the Etango Project Area (EPL 3345) falls within the Namib-Naukluft National Park and the northern portion of the tenement falls within the Dorob National Park.

Activities in the licence area are covered by a number of acts, policies and bills, including: the Minerals (Prospecting and Mining) Act, No 33 of 1992; the Environmental Assessment Policy, 1994; the Environmental Management Bill, 2004; the Environmental Management Act (Act No. 7 of 2007); the list of activities that may not be undertaken without an environmental clearance certificate and the Environmental Impact Assessment regulations promulgated on 6 February 2012 (Government Gazette No. 4878). South African Legislation still in force since Namibian independence in 1990 – specifically the Nature and Conservation Ordinance, No. 4 of 1975; and the Policy for Prospecting and Mining in Protected Areas and National Monuments.

Environmental and Social Impact Assessment

An ESIA, reflecting the project detailed in the 2012 DFS, was prepared by A. Speiser Environmental Consultants cc (ASEC) and Environmental Resource Management (ERM) ASEC and ERM and submitted in April 2012. Environmental Clearance was granted in July 2012 valid for 3 years. An application for renewal of the environmental permit was made in July 2015 and the Environmental Clearance was issued on the 11 November 2015 valid for 3 years (from the issue date). Environmental clearance for linear infrastructure was granted in February 2013 valid for 3 years. An application for a renewal has been submitted.

No substantiative legislative, environmental or social impacts have been identified for development of the Etango Project. The Erongo region already hosts other uranium producing operations, and uranium mining and processing is well understood in the local communities and by Government regulatory authorities. The Etango Project enjoys local community support and is expected to have a significant positive impact on the Erongo Region and Namibian national economies, including local employment and skills training.

Current Permits and Applications

The current status of the EPL is discussed in Section 4.5.1.

Visitors to the Namib-Naukluft National Park are required to obtain a park entry permit. Bannerman has ongoing Park Entry Permits (one for each employee) which are updated on an annual basis.

The proposed new Project access road will cross an existing tenement held by Reptile Uranium. A letter of 'in principle agreement' to allow construct of the road across this land has been received from Reptile Uranium, while an allowance has been included in the capital cost estimate for sterilisation drilling.

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5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 PROJECT ACCESS

The Etango Project is located approximately 31km east of the town of Swakopmund and 47km northeast of the port town of Walvis Bay (Figure 4.1). Year-round access to the Project area is gained by the sealed and unsealed C28 road from Swakopmund, then by well-maintained unsealed road (D1991) into the Namib-Naukluft National Park area.

5.2 PHYSIOGRAPHY AND CLIMATE

The Project area is located in the western region of the Namib Desert at an altitude of 150m above sea level (asl). The bulk of the project area lies on the Namib Peneplain where there is poor soil development over eluvial, colluvial and alluvial material, and bedrock. Due to the very low rainfall, these soils have gypsum crusts over large areas and vegetation is very sparse, often consisting of lichen, low bushes or shrubs.

The area of the Etango deposit is generally flat (Figure 5.1) with occasional low undulating hills with sparse outcropping bedrock. Remnant shallow drainage channels are present in the Project area. The region to the north of the deposit, around the Swakop River, is characterised by deep gully erosion and exposure of outcrops of the underlying rock sequences. There is good access to the areas of the desert plains and the Etango deposit, whilst access to river valleys can be difficult.

Figure 5.1
Etango Uranium Project – Drilling in the Namib Desert at Anomaly A

Rainfall in the area is sporadic. The highest monthly rainfall in the 10 years from 1996 to 2005 occurred in March 2000 (21.8mm). Figure 5.2 summarises the average monthly rainfall for the years 1996 to 2005. The Project area also receives moisture from fogs caused when

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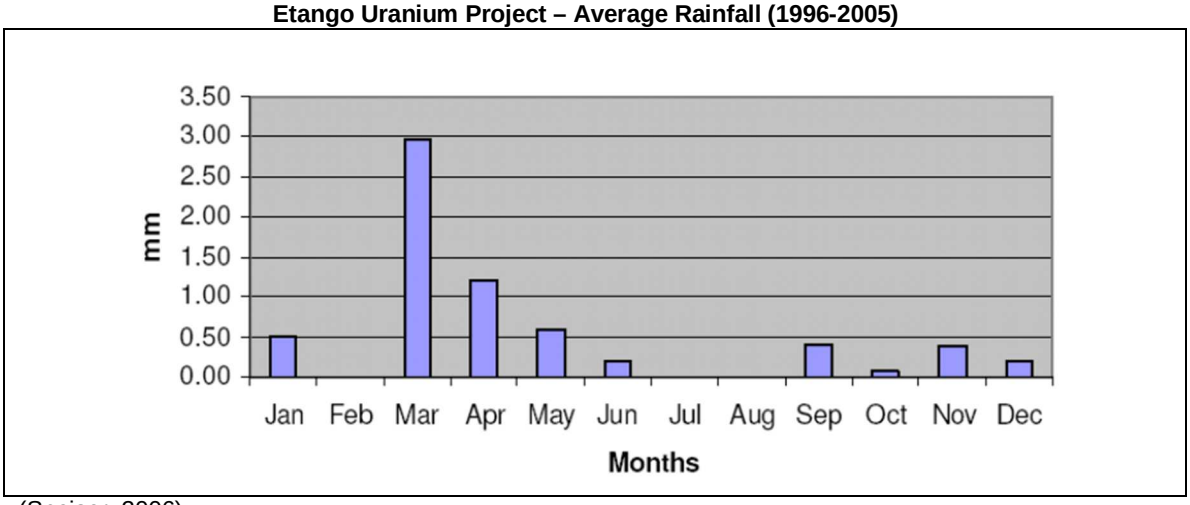
moist air which has been cooled by the Benguela Ocean current is blown onshore. As a result of the moist air feeding off the Atlantic, the air along the coast line remains humid throughout the year (between 60% and >80% relative humidity). The nearby town of Walvis Bay experiences more than 125 fog days per year (*Speiser, 2006*).

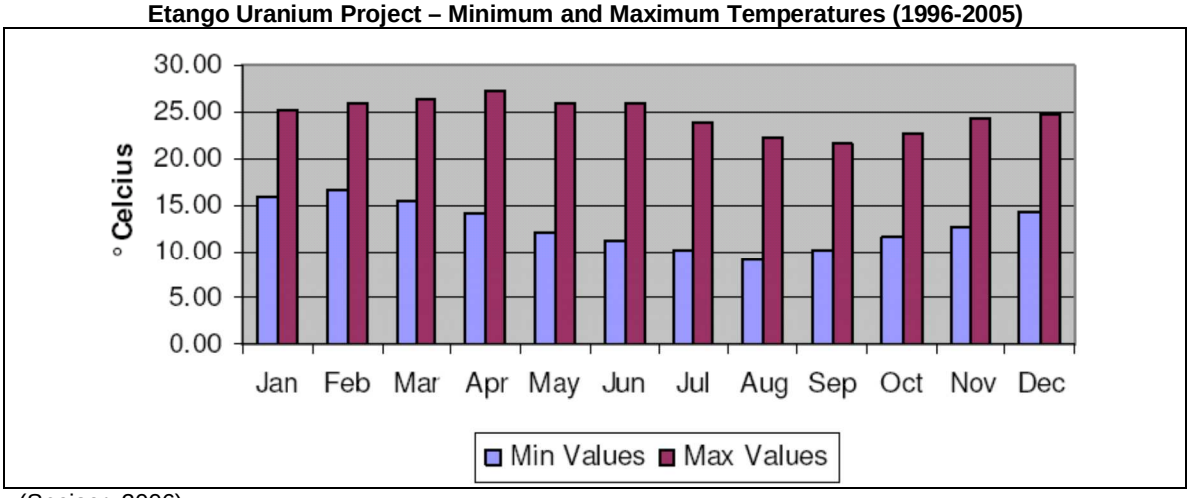
Figure 5.2
Etango Uranium Project – Average Rainfall (1996-2005)



(Speiser, 2006)

The Namib Desert region does not experience the extremes of temperatures that are typical to many other deserts, due to the presence of the cold offshore current. However, the temperature can peak at over 40ºC in the summer months, while in the coldest month (August) the minimum can fall to 9ºC (Figure 5.3). The hottest month is April with an average maximum temperature of 27ºC (*Speiser, 2006*).

Figure 5.3
Etango Uranium Project – Minimum and Maximum Temperatures (1996-2005)



(Speiser, 2006)

There are no seasonal climatic restrictions to year-round operations.

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5.3 LOCAL INFRASTRUCTURE AND SERVICES

The town of Swakopmund, with a population of approximately 50,000 people, has excellent services and infrastructure. Services include financial, shopping, construction, trades and medical support. Swakopmund serve as the regional hub for the Namibian Uranium Industry.

The port city of Walvis Bay is located 30km south of Swakopmund along sealed highway C14. Locally trained technical and non-technical personnel are employed from Windhoek and Swakopmund, while expatriate workers in the area typically reside in Swakopmund. Bannerman has an office in Swakopmund, and a field office and storage complex on site at Etango which it uses as a base for the Etango Project.

Drilling services and water for drilling are supplied by a local drilling contractor (Metzger Drilling) which owns the nearby Weitzenberg and Goanikontes Farms on the Swakop River.

AREVA, owner of the Trekkopje mine has installed a 20 Giga litre per annum desalination plant located north of Swakopmund. AREVA is currently supplying water to a number of uranium projects whilst its project is on care and maintenance. The national water utility, NamWater, has discussed plans to purchase the desalination plant to supply water for industrial purposes.

Power lines are located near the Project area and the national power utility, NamPower, has plans to increase power supplies to the region to cope with expected future demand. NamPower has recently commissioned the Caprivi Link Interconnector, allowing Namibia access to the electricity networks of Zambia, Zimbabwe, the Democratic Republic of the Congo and Mozambique.

Additional information on regional infrastructure is provided in Section 18.

5.4 LAND AVAILABILITY FOR PROJECT DEVELOPMENT

There is sufficient land available to develop the deposit and site infrastructure. Waste dumps will be arranged immediately adjacent to the open pit, with the plant site and Ripios dump lying on near-level ground to the south of the deposit (Section 17.6). All required ground lies within EPL 3345, and a Mining Licence application has been submitted.

There are no conflicting land uses on the Project area.

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6 PROJECT HISTORY

EPL 3345 has been the target of significant previous exploration which included both ground geological/geochemical work (traverses and drilling) and aerial- and ground-based geophysical investigations. However, no mining production has taken place on the property.

While uranium mineralisation was first discovered in the Central Zone of the Damara Orogen in the early 1900s, there was no further exploration in the area until the 1950s. In the 1960s, Rio Tinto South Africa commenced an extensive exploration program in the area; a regional airborne radiometric survey and subsequent detailed spectrometer-magnetometer survey were conducted by the South West African Geological Survey in the 1970s.

A broad uranium anomaly along the western flank of the Palmenhorst Dome was identified and this was followed up by an initial exploration program in 1975. From 1976 to 1978, Omitara Mines (Omitara – a joint venture between Elf Aquitaine SWA and B & O Minerals) completed extensive reconnaissance drilling along the western Palmenhorst Dome position, with much of the work in the Anomaly A area.

A dramatic fall in the price of uranium in the 1980s resulted in exploration for uranium all but ceasing in the area (*Mouillac et al, 1986*) until 2005.

In 2005, Turgi applied for, and was granted, the titles for nuclear fuels (including uranium) over EPLs 3345. The area around the Anomaly A, Oshiveli and Onkelo deposits was identified as being prospective, due to the earlier work completed, including a non-JORC resource reported for the area by *Mouillac et al (1986*).

After acquiring its interest in EPL 3345 in 2006, Bannerman undertook a process of capturing and digitising the historic drill hole, geological mapping and ground geophysical data that was obtained from the Namibian Geological Survey and the Geological Survey of South Africa. Airborne radiometric and geophysical data was purchased from the government and reprocessed for uranium, identifying anomalous trends along the western flank of the Palmenhorst Dome. This dataset was part of the Erongo survey conducted by World Geoscience in 1994/1995.

Bannerman also sourced a high resolution Quickbird satellite image that covers the region of EPL 3345. A detailed mapping program was then completed along the flanks of the Palmenhorst Dome. An extensive program of reverse circulation (RC) and diamond core drilling has since been completed at the Etango Project. The main focus for this exploration has been to drill out and develop the Anomaly A, Oshiveli and Onkelo uranium prospects (in the previously explored Goanikontes area) and to determine continuity of mineralisation along strike, at depth and to the west of the Palmenhorst Dome. The drilling completed is discussed in more detail in Section 9.

In April 2007, Bannerman estimated a maiden Inferred Resource of 56Mt at 219ppm U_3O_8 above a 100ppm U_3O_8 lower cut-off (*Inwood, 2007*). Subsequent resource estimation studies were completed in January and September 2008, February, July and December 2009, March 2010 and then October 2010 (*Inwood, 2010*). These estimates have now been superseded by the current (November 2015) resource estimation study.

Since June 2007, metallurgical test work and a series of studies have been undertaken by or on behalf of Bannerman, including, principally:

- Scoping study into an agitated leach process similar to the Rossing flowsheet (2007)

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- Trade-off studies to evaluate heap leach potential, and potential to upgrade ore prior to agitated leaching (2008/2009)
- PFS evaluating heap leach and agitated leach options at 15Mtpa throughput, at ±25% level of accuracy, for process selection (2008/9). The PFS involved geotechnical, hydrological and mining assessments, and was completed in November 2009. Nine oriented drill holes formed the basis of the geotechnical work and both Owner mining and contract mining was assessed
- Trade-off study comparing conventional tertiary crushing with an HPGR option for heap leach product (2009)
- PFSU in 2011 taking account of the October 2010 resource model, finalising the heap leach or agitated leach comparison and leading to selection of the heap leach option (2010/2011). Further mining studies were undertaken to address aspects such as open pit bench heights, diesel versus electric, shovel versus backhoe, drill and blast, grade control, mine infrastructure, mine dewatering, mining production rate, waste dump design and mobile equipment requests for tender (RFQs)
- DFS for a 20Mtpa heap leach project with recovery by SX and calcining to produce U_3O_8 for shipment (2011/2012). Mining studies included further geotechnical study based on 26 orientated holes, and additional optimisation, design, schedule and cost process to complete a mining DFS estimate to an accuracy level of ±15%
- OS for a 20Mtpa heap leach project which complements the DFS and was completed in October 2015. The OS employed an updated geological model and updated the mine planning (including pit design and mine schedules). In addition the capital cost and operating cost estimates were updated to 2015 terms.

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7 GEOLOGICAL SETTING AND MINERALISATION

The geological setting and mineralisation of the Etango deposit is described in detail in the previous NI43-101 report (*Bannerman, September 2011*). The following is a summary of the salient features.

7.1 GEOLOGICAL SETTING

Primary uranium mineralisation is related to uraniferous leucogranites, locally referred to as alaskites. These are often sheet-like, and occur both as cross-cutting dykes and as bedding and/or foliation-parallel sills, which can amalgamate to form larger, composite granite plutons or granite stockworks, made up of closely-spaced dykes and sills. These alaskite intrusions can be in the form of thin (cm-wide) stringers or thick bodies up to 200m in width.

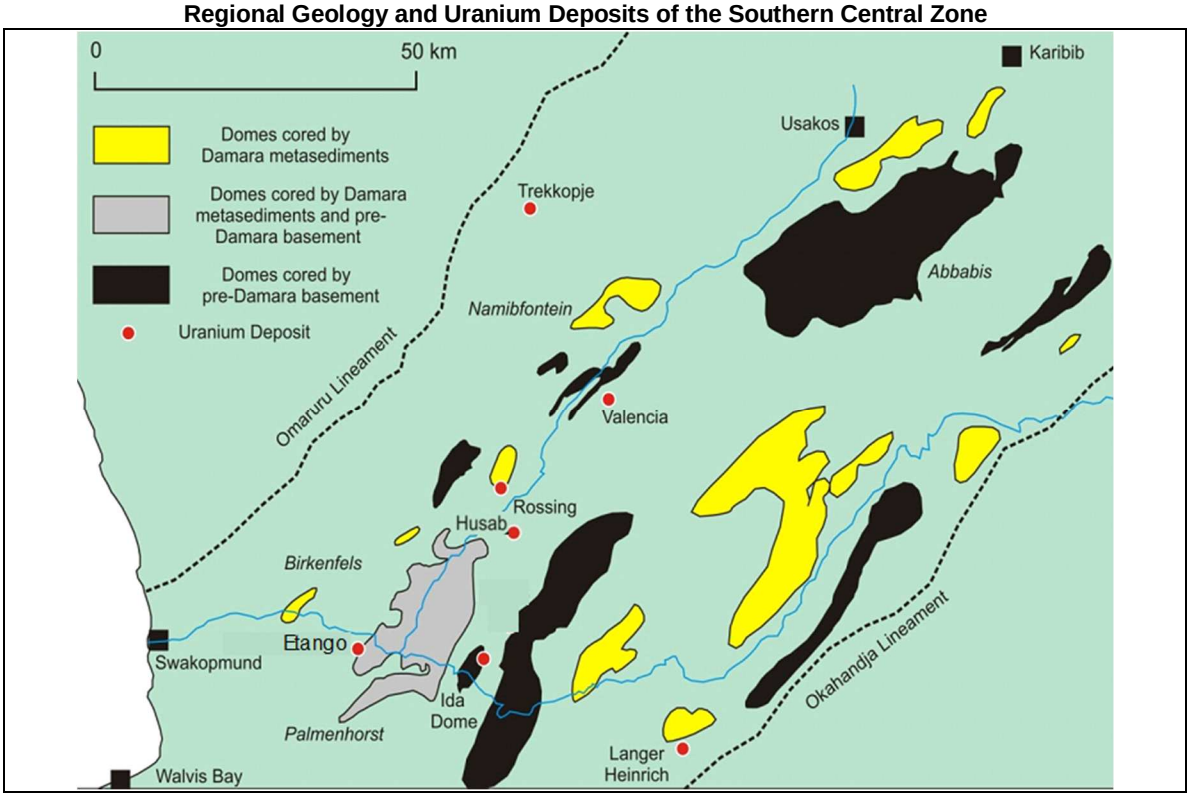
The alaskite bodies have intruded into the metasediments of the Nosib and Swakop Groups of the Neoproterozoic (pre-550Ma) to early Palaeozoic (c500Ma) Damara Supergroup. These metasediments and alaskite intrusions flank the Palmenhorst Dome, which is cored by Mesoproterozoic (1.7-2.0Ga) gneisses, intrusive rocks and meta-sediments of the Abbabis Metamorphic Complex.

During the Damara Orogenic event, the metasedimentary cover was subjected to multiple phases of deformation, resulting in overturning of the succession and development of a prominent gneissosity and lineation which is generally sub-concordant with original bedding. This gneissosity was further deformed leading to the formation of elongate basement-cored domes. Uraniferous alaskite sills and bodies that wrap around the Palmenhorst Dome are confined to dilatational sites in high-strain zones, with the alaskite sills generally striking from north-northwest to north-northeast and dipping to the west.

Limited faulting is recognised on a deposit scale. The high-strain zone is bounded in the west by a 35-45° northwest-dipping fault zone. The fault zone post-dates alaskite intrusion, but pre-dates the Karoo age, and is cut by Karoo-age dolerite dykes. Narrow, sub-vertical faults are also common. These faults display both north-down and south-down displacement; maximum displacements observed in the field are only about 2m. Fault strike extents do not exceed 100m.

On a regional scale, the Etango deposit lies within the Southern Central Zone of the northeast-trending branch of the Damaran orogenic belt. Domal structures are relatively widespread within the Southern Central Zone, where the Rössing, Palmenhorst and Ida Domes host notable uranium-enriched alaskites (Figure 7.1).

Figure 7.1
Regional Geology and Uranium Deposits of the Southern Central Zone



7.2 DEPOSIT GEOLOGY

The localised geological setting is depicted in Figure 7.2, and the uranium occurrences at the contiguous Anomaly A, Oshiveli and Onkelo Prospects can be seen to wrap around the western edge of the Palmenhorst Dome. Uranium mineralisation occurs almost exclusively in the alaskite, although minor uranium mineralisation can be found in metasediments close to the alaskite contacts, probably from metasomatic alteration and in minor thin alaskite stringers within the metasediments.

The sheeted alaskite bodies have been classified into six types (A to F) by *Nex, et al. (2001*). Under this classification, Types D and E are host to the bulk of the uranium mineralisation.

The Type D alaskites have a generally irregular and anastomosing geometry, are white to grey in colour, equigranular and contain smoky quartz, with accessory topaz.

Type E alaskites are distinguished by a reddish colouration and the presence of ubiquitous oxidation haloes (or alteration rings) which are irregular sub-circular features with a red rim and a grey core. Smoky quartz is common and the reddened parts of the oxidation haloes may contain more biotite and iron-titanium oxides than the rest of the alaskite.

However, extensive petrological, mineralogical and metallurgical study has failed to find any significant difference between these two types, apart from colour. Also mapping shows that they cross-cut, grade into each other and are of insufficient size to be separated into mining or processing units.

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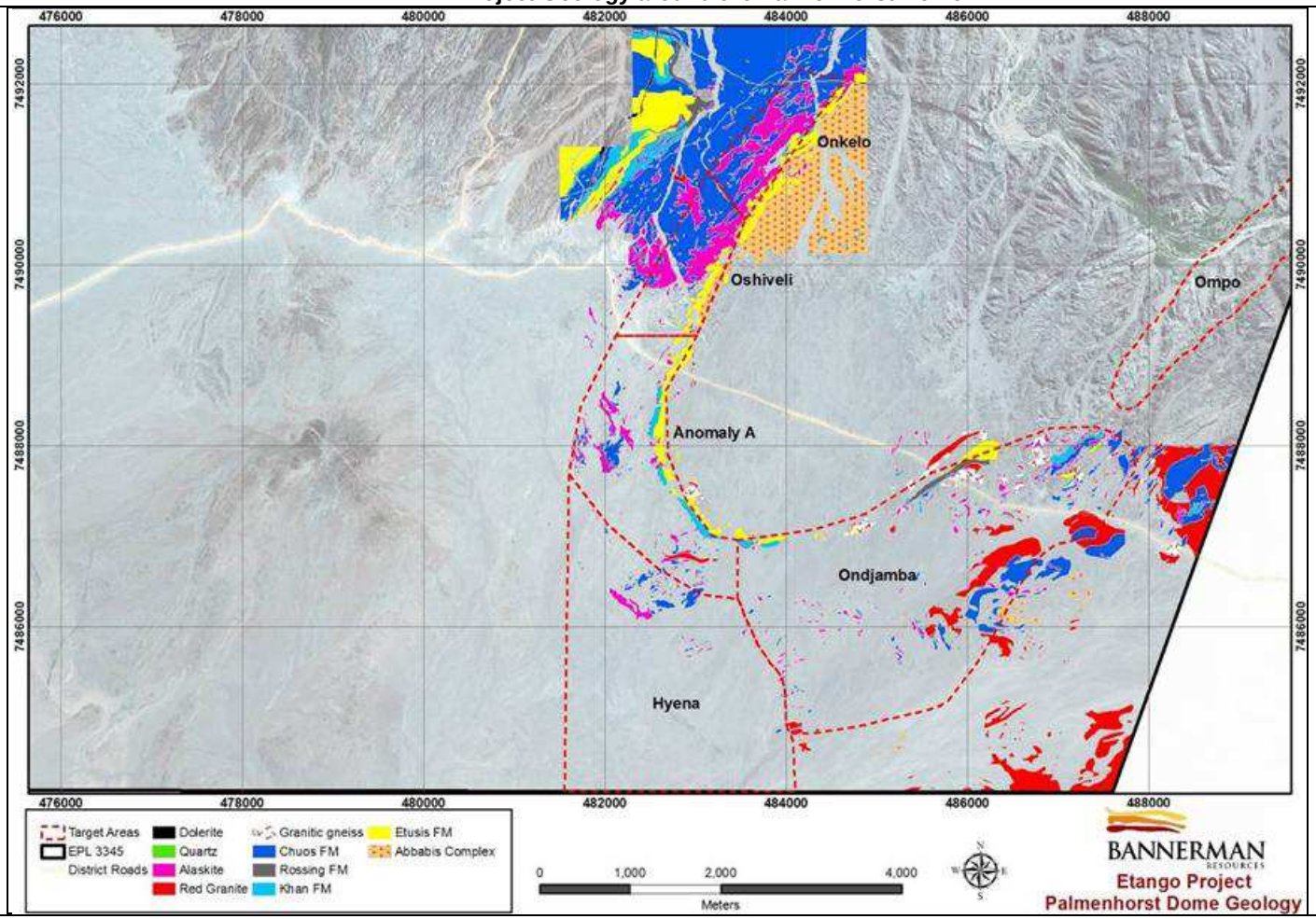
Figure 7.2

Project Geology around the Palmenhorst Dome



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The dominant primary uranium mineral is uraninite (UO_2), with minor primary uranothorite ((Th, U) SiO_4) and some uranium in solid solution in thorite (ThO_2). Uraninite is commonly associated with chloritised biotite in the alaskites, and with ilmenite and magnetite within foliated alaskites.

The primary uranium mineralisation occurs as microscopic disseminations throughout the alaskite, at crystal interfaces, and as inclusion within other minerals. Secondary uranium minerals such as coffinite ($U(SiO_4)(OH)_4$) and betauranophane ($Ca(UO_2)_2(SiO_3OH)_2\ 5H_2O$) occur as replacements of the primary minerals or as coatings along fractures. QEMSCAN analysis indicates that about 81% of the uranium present is in primary uraninite, while 13% is in secondary coffinite and 5% is in secondary betauranophane (*Freemantle, 2009*). The remaining 1% of the uranium occurs in various minor phases including brannerite, betafite and thorite. Very minor amounts of uranium are also present in solid solution in monazite, xenotime and zircon. A very minor amount of the primary refractory mineral betafite $(Ca,U)_2(Ti,Nb,Ta)_2O_6(OH)$ is also present.

In the Etango deposit the Th/U ratio averages about 0.25 and this decreases at higher uranium levels (e.g. >400ppm U_3O_8) to be between 0.05 and 0.25. Nuclides of the uranium decay series have been found to be in equilibrium or near-equilibrium (*Mouillac, et. al., 1986*).

Uraninite is not always observed in mineralised samples under the microscope, as it is thought to be present as a low-grade background scatter of largish (up to 350µm) individual crystals. Uranothorite is seen more often, probably because it is generally finer-grained and more dispersed, and hence more easily observed.

The secondary uranium-bearing minerals coffinite and betauranophane often occur in the same sample. Coffinite is more common, and, on occasions, is seen to rim uraninite as an alteration product. The highest grade samples almost always contain coffinite, while betauranophane appears to be more evenly distributed within low to high grade samples. Both secondary minerals occur together throughout the depth range (0-487m), although there is some suggestion that coffinite is more common at shallow depths and betauranophane at greater depths.

There is no evidence for any identifiable discrete enrichment or depletion zones in any uraniferous (or other) minerals in any areas of the Etango deposit. Equally, there is no perceived zonation of uranium mineralogy with depth, grade, location, bulk rock chemistry, mineralogy or any other feature. However, uranium grades decline systematically to the west down plunge along the leucogranite bodies.

The Etango deposit comprises a very large number of analyses in the 100-175ppm U_3O_8 range, with a small number of much higher grade analyses which bring the average up to the mean ore grade of around 200ppm. This is reflected in the deposit mineralogy, with a large volume of leucogranite containing a very small amount of uraninite and uranothorite being enriched by a small quantity of leucogranite bearing encrustations of secondary coffinite and betauranophane minerals, i.e. a large low-grade background of primary uranium minerals has been overprinted, partially replaced and upgraded by a more patchy and erratic, secondary mineralising event, as represented by locally abundant uranium silicate minerals, coffinite and betauranophane.

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8 DEPOSIT TYPES

Uranium mineralisation at the Etango Project (Anomaly A, Oshiveli and Onkelo deposits) occurs within a stacked sequence of leucogranite (alaskite) dykes, of varying thickness, that have intruded into the host Damara Sequence of metasedimentary rocks. This style of primary uranium mineralisation is commonly referred to as 'Rössing type' mineralisation. Other nearby examples of this style of mineralisation include the Rössing uranium mine, the Valencia deposit, and the Husab (Rössing South) deposit, which is also under development.

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9 EXPLORATION

9.1 PREVIOUS EXPLORATION

While uranium minerals were first discovered in the Central Zone of the Damara Orogen in the early 1900s, there was no intensive exploration in the area until the 1950s. In the 1960s, Rio Tinto South Africa commenced an extensive exploration program in the area; and a regional airborne radiometric survey and subsequent detailed spectrometer-magnetometer survey were conducted by the South West African Geological Survey in the 1970s.

A broad uranium anomaly along the western flank of the Palmenhorst Dome was identified in an airborne radiometric survey in 1974, and this was followed up by a program of 134 percussion drill holes in 1975. From 1976 to 1978 Omitara Mines (Omitara - a joint venture between Elf Aquitaine SWA and B & O Minerals) drilled 224 percussion drill holes, mostly short and vertical, on a reconnaissance grid of fences at 200-400m spacing (north) by 75-100m east along the western Palmenhorst Dome position, with the closer-spaced fences being near the Anomaly A area. These percussion drill holes totalled 13,383m with depths ranging from 50-100m. An additional nine diamond drill holes were completed for a total of 2100m.

Omitara also completed a total of 6800m of trenching to obtain exposure of the lithologies under cover at Anomaly A.

From 1982 to 1986, Western Mining Group (Pty) Ltd conducted regional mapping and drilled 22 percussion drill holes for 1,017m and conducted surface scintillometer surveys.

9.2 EXPLORATION BY BANNERMAN RESOURCES

9.2.1 Preliminary Work

After securing its interest in the Etango lease (EPL 3345) in 2006, Bannerman captured and digitised the historical drill hole, geological mapping and ground geophysical data that was obtained from the Namibian Geological Survey and the Geological Survey of South Africa. Airborne radiometric and geophysical data was purchased from the government and reprocessed for uranium, identifying anomalous trends along the western flank of the Palmenhorst Dome. This dataset was part of the Erongo survey derived from an airborne survey conducted by World Geoscience in 1994 and 1995.

Bannerman also sourced a high resolution Quickbird satellite image that covers the area of EPL 3345. Reprocessing of this image in the areas near the Swakop River has enabled exposure of the alaskite granites to be readily identified; this, together with the airborne radiometric data, has been an essential aid for further mapping and target generation.

An Airborne Lidar Survey was also conducted over the lease to the south of the Swakop River and a 10cm accurate surface digital terrain model (DTM) has been created over the entire Etango Project area.

The core from the nine diamond drill holes drilled earlier by Omitara was re-logged, but was deemed unsuitable for re-assay. A detailed mapping program was completed along the western and eastern flanks of the Palmenhorst Dome. The main focus for this initial exploration was to develop and drill out the previously identified Anomaly A uranium anomaly (previously explored as Goanikontes in the late 1970s and early 1980s), and to determine the continuity of uranium mineralisation along strike, at depth and to the west of the

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Palmenhorst Dome. Subsequently, exploration has extended to the north from Anomaly A to the Oshiveli and Onkelo Prospects.

9.2.2 Drilling

As of 31 October 2015, Bannerman had completed a total of 1248 RC, 141 diamond and 21 RAB drill holes for a total of over 306,999m, in the vicinity of the Etango Project (Figure 9.1 and Table 9.1). This drilling provided the geotechnical, hydrological, structural, lithological and uranium grade data over the Anomaly A, Oshiveli, Onkelo, Ondjamba and Hyena Prospects, and over the plant site area that is the subject of this feasibility study. Further RC drilling has also been completed at exploration prospects to the southwest of Etango, along the Rössingberg-Gohare line of prospects and at Ombepo and Cheetah in the licence area.

The RC drill holes range from 23-497m in depth and the diamond drill holes range from 101-528m in depth. The RC drill holes were drilled by Metzger Drilling, using bit diameters of 4.72" to 5.5". This RC drilling has been conducted on a nominal 50m x 50m, to 50m x 100m drill spacing, with the bulk of the 50m x 50m drilling being completed in the area of the potentially open-minable resource. A small area of 25m x 50m spaced drilling has also been completed in the centre of the Project area. Drilling along strike and down-dip of the main mineralisation has targeted extensions to the mineralised zones and has been drilled on a nominal spacing of 100m x 50m.

Due to the shallow dip of the mineralised alaskite bodies (approximately 15-45º to the west) and the inclination of the RC and diamond drill holes (generally 60° to the east), the length of the drill hole intercepts are close to the true thickness of the mineralised intervals (Figure 9.2).

Most of the diamond drill holes for resource delineation and grade estimation purposes were drilled using NQ diameter core barrels (47.6mm core), with the bulk of the core being orientated by spearing after each run. A total of 29 diamond drill holes were drilled for geotechnical purposes using a NQ3 core barrel (45.1mm core). All geotechnical samples were sent to Rocklab in Johannesburg for test work.

Since the previous Technical Report a further eight RC drill holes have been completed for exploration purposes at Onkelo. These have now been included in the 2015 Mineral Resource estimate. The total additional metres drilled are 1614m and the majority have been drilled to the southeast, with one hole drilled vertically. The additional drilling is represented by the blue dots in Figure 9.1.

Figure 9.1
Drilling Completed at the Etango Project
for the November 2015 Resource Estimate

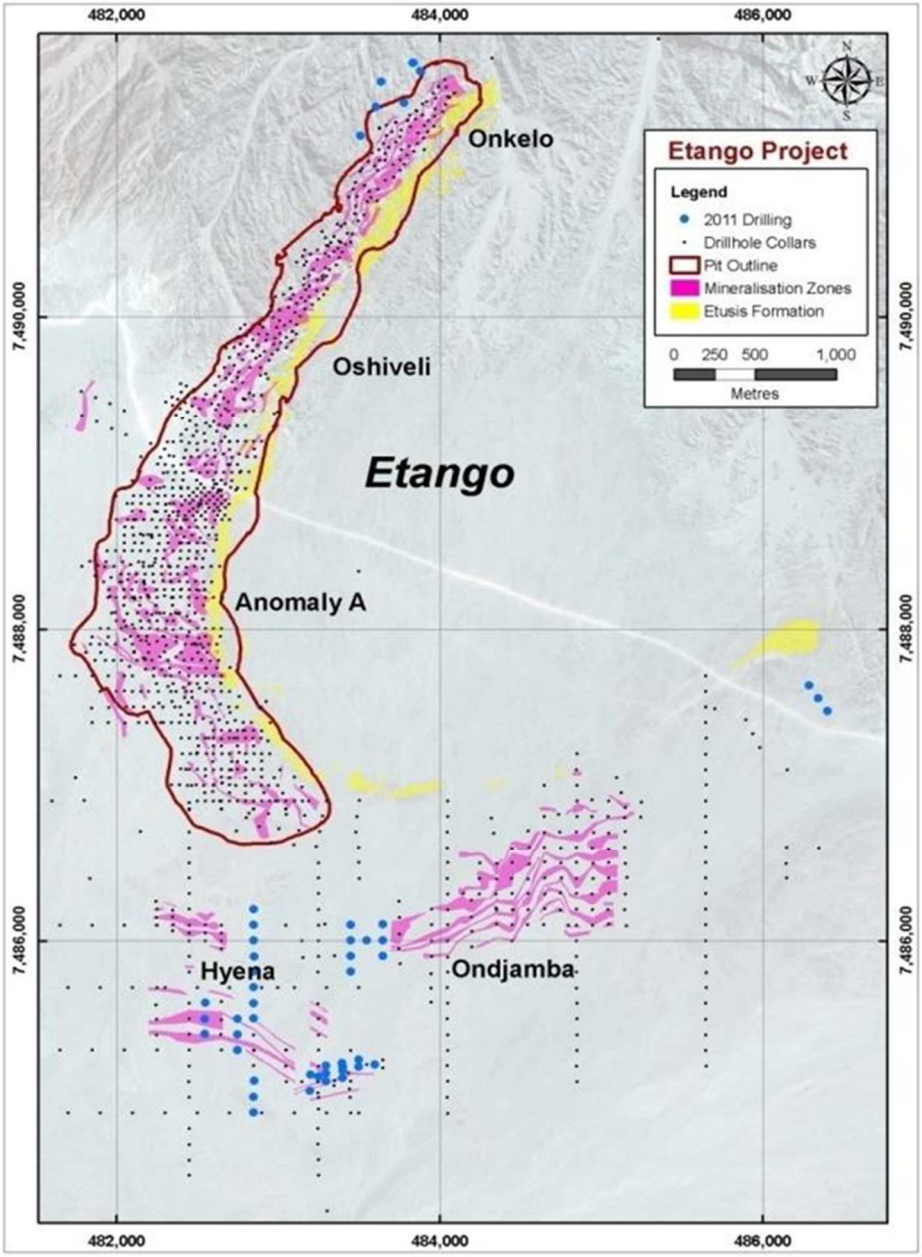

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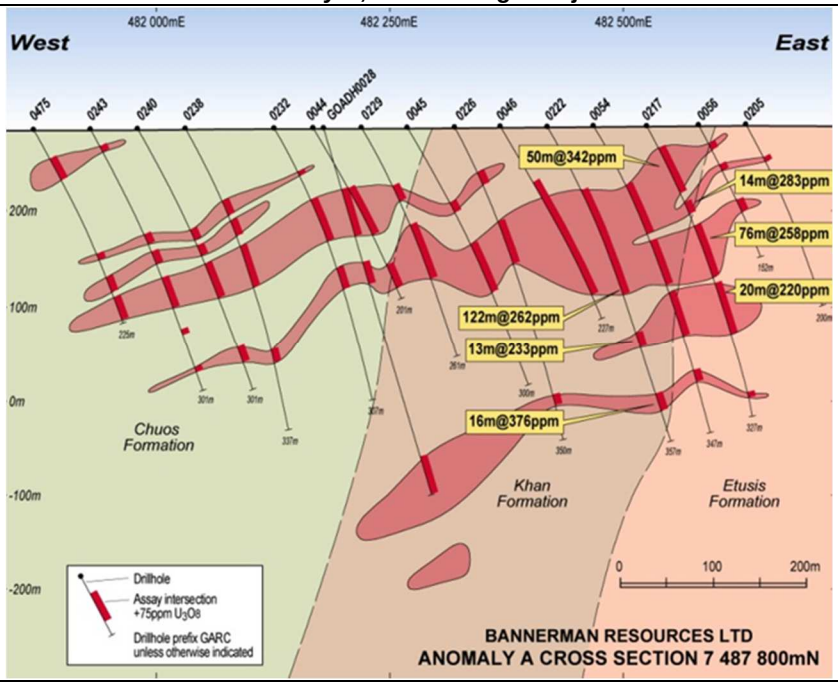
Figure 9.2
Typical Cross-section through the Mineralisation at Anomaly A, at the Etango Project



Twenty eight drill holes were completed in HQ core diameter (63.5mm) for metallurgical test work; the entire HQ core was sent to Ammtec Laboratories in Perth. Selected core from a total of 22 of the resource definition drill holes was also used for metallurgical test work.

Table 9.1
Drilling by Bannerman in the Etango Project Area, up to 31 October 2015

Drill Type	Number	Metres
RC Anomaly A	582	146,897
RC Oshiveli	152	40,069
RC Onkelo	100	20,592
RC Ondjamba	182	30,536
RC Hyena	112	18,292
RC Other	120	10,723
RC Total	**1,248**	**267,109**
DD Resource	84	26,079
DD Geotechnical	29	7,079
DD Metallurgy	28	4,857
DD Total	**141**	**38,015**
RAB Total	**21**	**1,875**
Grand Total	**1,410**	**306,999**

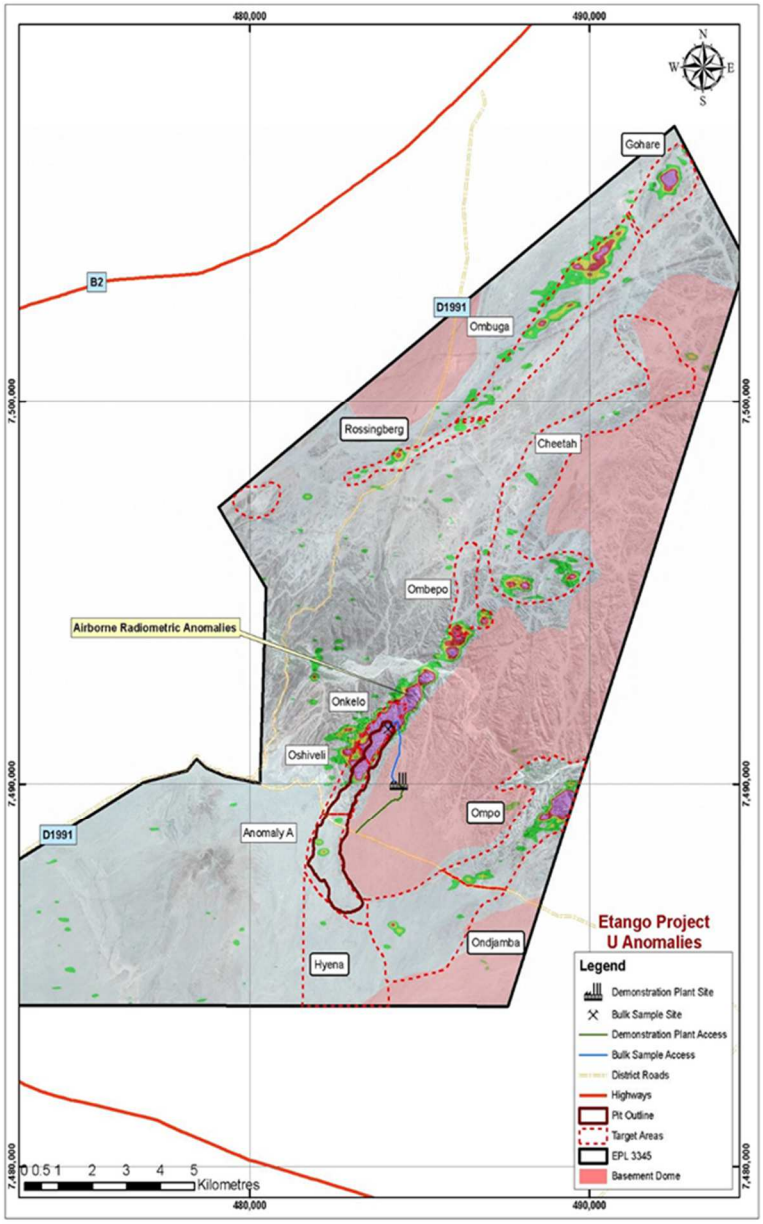
All drill hole collars have been surveyed by licensed surveyors after drilling. Downhole directional surveys were initially taken using an Eastman single shot camera at nominal 30m intervals (the first few holes only); however, for the vast majority of holes, the practice has

been to survey drill holes using a three-component Fluxgate Magnetometer survey tool following completion of the drilling.

9.2.3 Ongoing Exploration

Other areas within tenement (EPL 3345) in the vicinity of the Etango Project, have the potential to host additional uranium resources, especially in the southern portions of the lease where there is soil and colluvium cover. Work continues to focus on the Mineral Resources in the Hyena and Ondjamba areas shown in Figure 9.3.

Figure 9.3
Exploration Targets within EPL 3345



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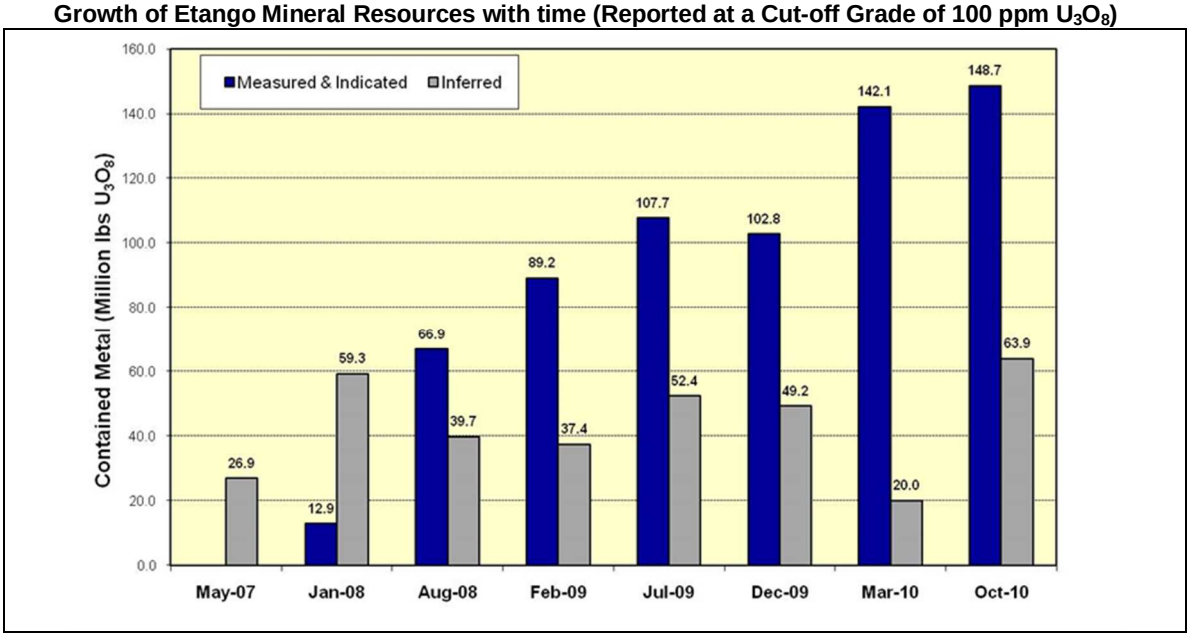
9.2.4 Previous Mineral Resource Estimates

In May 2007, an Inferred Mineral Resource was estimated by Bannerman for the Anomaly A deposit, based on the historical and recent drilling.

In January and August 2008, Coffey Mining independently estimated mineral resources for the Anomaly A/Oshiveli area based only on the recent Bannerman drilling. Further Coffey Mining Mineral Resource estimates were completed for the Anomaly A, Oshiveli and Onkelo areas in February 2009, July 2009, and March 2010 and, most recently, in October 2010 (Figure 9.4). This estimate has been superseded by the November 2015 estimate, described in Section 14.

Figure 9.4

Growth of Etango Mineral Resources with time (Reported at a Cut-off Grade of 100 ppm U_3O_8)



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10 DRILLING

10.1 DRILLING BY PREVIOUS OWNERS

Details of the exact sampling methods used for the historic drilling are not available and are not considered relevant to this report, as this drilling has not been included in any modelling or resource estimates. For the Omitara drilling, the percussion holes were typically sampled on 1m intervals, as discussed further in the following section.

The following discussion details the sampling methods used by Bannerman. Bannerman routinely samples all intersected alaskite intervals and a few metres of metasediment on either side. The location of the sampling for the resource estimates is shown in Figure 9.1.

10.2 DRILLING BY BANNERMAN

As of 31 October 2015, Bannerman had drilled a total of 1248 RC, 141 diamond and 21 RAB drill holes, for a total of over 306,999m in and around the Etango Project. The RC drill holes range from 23m to 497m in depth and the diamond drill holes range from 84m to 528m in depth. A total of 28 diamond holes were drilled for metallurgical testing purposes, 29 diamond holes for geotechnical testing purposes and 21 RAB holes for hydrogeological purposes. Lithological contacts were considered during modelling for those holes which were not assayed. The RC drill holes were drilled by Metzger Drilling using bit diameters of 4.72" to 5.5". The bulk of the RC drilling has been executed on a nominal 50m by 50m, to 50m by 100m drill spacing. The bulk of the 50m by 50m drilling has targeted the area of the likely open-pittable resource. Drilling along strike and down dip of the main mineralisation has targeted extensions to the mineralised zones and has been drilled on a nominal 100m by 50m spacing.

The majority of the diamond drilling for resource delineation and grade estimation purposes used NQ diameter core barrels (47.6mm). Twenty-nine holes were drilled using a NQ3 core barrel (45.1mm) for geotechnical purposes. All geotechnical samples were sent to Rocklab in Johannesburg for test work. The majority of the core is orientated by spearing after each run. Twenty-eight drill holes were completed using HQ core diameter (63.5mm) for metallurgical test work; the entire core was sent to Ammtec Laboratories in Perth.

Due to the shallow dip (approximately 15°-45º to the west) of the mineralised alaskites and the angle of intercept of the RC and diamond drill holes, the true thickness of the significant intercepts is close to the stated mineralised interval.

Drilling of other target areas within EPL 3345 is in progress and to date 84 RC drill holes have been completed at the Rosenberg, Ombuga, Gohare, Ombepo, Cheetah and R5 prospect areas.

10.3 SURVEYING

All drill hole collars are surveyed by licensed surveyors after drilling.

For diamond drill holes, downhole surveys were taken using an Eastman single shot camera at nominal 30m intervals up to drill hole GOADH0022. The practice is now for all drill holes to be surveyed by a Verticality magnetic survey tool performed by G Symons of Geophysics/Terratec contract geophysicists.

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11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 SAMPLING METHOD AND APPROACH

Details of the exact sampling methods used for the historic drilling are not available and are not considered relevant to this report, as this drilling has not been included for resource estimation purposes. For the Omitara drilling, the percussion holes were typically sampled on 1m intervals. When taken, chip samples were assayed by X-ray fluorescence. Downhole gamma ray spectrometry was also taken for selected intervals from most of the drill holes.

The following discussion details the sampling methods used by Bannerman. Bannerman routinely samples all intersected alaskite intervals.

11.1.1 RC Drilling

The following methodology has been applied to the RC drill hole sampling:

- Drill samples are collected off the rig cyclone in large plastic bags at 1m intervals. The sample bags are pre-marked and tags are also prepared for the laboratory sample which identifies the sample number
- The 1m sample is split in the field by Bannerman staff using a 75/25 riffle and the 75% sample is placed into a bulk sample bag from which rock chip samples are taken and placed into a chip tray for logging by the geologist
- The primary sample sent to the laboratory is obtained by splitting the 25% sample until a sample of approximately 500g to 1kg is obtained. A count per minute (CPM) reading is taken from this sample using a handheld scintillometer and recorded along with the sample condition (wet, dry, and moist). If the bulk sample is wet, a spear sample is taken
- The sample that is to be sent to the laboratory for analysis is placed into a clear plastic bag that is labelled with the drill hole identification and sample. A number of samples are placed into larger plastic bags for transport to the secure sample storage facility on site at Etango
- A library reference sample is obtained by again splitting the reject of the 25% split until another 500g to 1kg sample is obtained. The reference sample is stored in Bannerman's warehouse on site at the Etango Project
- Sample sheets are drawn up by the responsible geologist and given to the Senior Field Technician. He assigns the sample string numbers to the relevant samples. The primary sample is transferred into a new clear plastic bag which has the reference sample number written on the bag and a sample stream ticket is placed within the bag
- Sampling details are sent to the assaying laboratories electronically, while a paper copy accompanies the samples. A sample submission sheet is sent with each sample dispatch
- Up until November 2011, samples were sent from the secure sample storage facility on site at Etango to SGS Lakefield in Johannesburg (SGS Johannesburg) and Genalysis Laboratory Services in Johannesburg (Genalysis Johannesburg) three times a week via Coastal Couriers. Since 01 December 2011, due to various reasons, including the closure of SGS's sample preparation facility in Swakopmund, the practice has been to submit samples to the Bureau Veritas Analytical Laboratory in Swakopmund for sample preparation and analysis

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- Field duplicate samples sourced from the 75% reject are taken at a rate of 1 in every 20 primary samples. The sampling method is the same as used for the primary sample. Field duplicate samples are sent to Genalysis Johannesburg for assaying
- From December 2008 up until November 2011, samples were sent from the Bannerman sample storage facility directly to the SGS Sample Preparation Facility in Swakopmund (SGS Swakopmund). The sample is prepared by SGS Swakopmund and a smaller pulp sample is then sent to the relevant facility in Johannesburg for assaying. As of 01 December 2011, due to several reasons, including the closure of SGS's sample preparation facility in Swakopmund, Bannerman started submitting samples from the Etango Project to the Bureau Veritas Laboratory in Swakopmund for preparation and analysis. This represents less than 2% of the overall assay dataset for the project
- Up until September 2009, the RC chip trays and reference samples were stored in a secure facility in Swakopmund; however, since October 2009, all chip trays and reference samples have been stored at a secure sample storage facility on site at Etango
- Since December 2007, standards and blanks have been routinely inserted into the sampling stream at a nominal rate of 1:20.

11.1.2 Diamond Drilling

The following methodology has been applied to the diamond drill hole samples:

- After drilling, the diamond core is placed into core trays by the drilling contractor
- The core is then taken to the Bannerman core logging and storage facility on site at Etango, where it is orientated, measured, marked for sampling and logged by the staff geologists
- Sample intervals are determined by the geologist after logging. The sample lengths are nominally 1m; however, shorter intervals are sampled where a lithological boundary is intersected. No sampling is undertaken across lithological boundaries
- Up to drill hole GOADH0022, the core was cut in half using a diamond saw, with the primary sample sent to SGS Johannesburg for crushing and analysis. Subsequent to GOADH0022, only quarter core has been used for primary analysis. The core depths (in metres), sample intervals and sample numbers are marked on the core for later identification, as shown in Figure 11.1
- Field duplicates are taken every 20th sample. Where a field duplicate is taken, ¼ core is submitted to the laboratory. One ¼ core sample is sent to SGS Johannesburg for primary analysis, whilst the other ¼ core sample is sent to Genalysis Johannesburg for preparation. Since January 2009, all field duplicates have been sent to SGS Johannesburg for assaying. As with the RC samples, the diamond samples are placed in numbered bags for dispatch.

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Figure 11.1
Sampled Core from Anomaly A

11.1.3 Density Determinations

Bannerman has built up a large database of drill core density data over the course of its exploration program at the Etango Project. This data has been collected by Bannerman staff using three bulk density determination methods, namely the calliper method, the water immersion method and whole tray density method. Density estimates have also been made on selected pulp samples from the RC drilling programs by Genalysis Laboratory Services in Perth using the gas pycnometer method.

The calliper and water immersion methods are carried out on whole diamond core samples of 10cm length, while the whole tray method is applied to entire trays of core sample. The core diameters vary from NQ to NQ3 to HQ in diameter.

A total of 11,113 calliper, 5889 water immersion and 782 whole tray density measurements have now been collected. The majority of the density data (75% of calliper, 78% of water immersion and 42% of whole tray) was collected from the alaskites that host the bulk of the uranium mineralisation at Etango (Figure 11.1).

Analysis of the results indicates that there is no significant change in density with depth, apart from a small reduction relating to highly weathered alaskite near the surface. The latter is statistically insignificant due to the generally limited degree of weathering at Etango, especially in the Oshiveli and Onkelo areas. Density is not related to uranium grade (due to the very low levels of uraninite content). Any differences in density with depth, uranium grade, weathering, alteration, rock hardness and structural deformation are small and the number of samples involved is very small, so these lower density samples do not cause large differences from the global means of the various rock types at Etango and are therefore regarded as negligible. Consequently, global mean values have been used for the density values in the mineral resource modelling and estimation.

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Table 11.1
Breakdown of the Collected Bulk Density Data and Data Analysis

		Bulk Density Determination Method					
		Caliper		Water Immersion		Whole Tray	
		Number	% of data	Number	% of data	Number	% of data
Rock types at the Etango deposit	GALD	2437	21.9	524	8.9	65	8.3
	GALE	5577	50.2	4036	68.5	266	34.0
	GALF	50	0.4	3	0.1		
	GALU	68	0.6	28	0.5		
	Mixed alaskites	243	2.2				
	Combined Alaskites	**8375**	**75.4**	**4591**	**78.0**	**331**	**42.3**
	GRUN	82	0.7	58	1.0	9	1.2
	GRED	4	0.0	0	0.0	1	0.1
	QZ	10	0.1	3	0.1		
	CGN	2346	21.1	1003	17.0	384	49.1
	KGN	149	1.3	112	1.9	28	3.6
	EGN	147	1.3	122	2.1	29	3.7
Total		**11113**		**5889**		**782**	

11.1.4 Downhole Radiometric Surveys

Bannerman undertakes downhole radiometric observations on all drill holes, with this data being collected under contract by Terratec Geophysical Services.

Two types of downhole radiometric data are collected, from the Auslog Probe and the GRS Probe (Gamma Ray Spectrometer). Following the completion of drilling, drill holes are surveyed with the Auslog Probe, while, up until June 2008, approximately one hole in five was also resurveyed with the GRS probe. At the time of collection, the gamma log is collected on both the downhole and uphole transit of the probe.

Auslog collects a Gamma log in total Counts per Second, while the GRS Probe is a multi-channel instrument which collects the Total Count Gamma Log, a Gamma Ray count on uranium and Gamma Ray count on thorium. The GRS probe has been used as a quality assurance, quality control (QAQC) check on the Auslog Data.

11.1.5 Adequacy of Procedures

The drilling, sampling and storage procedures used by Bannerman meet industry acceptable standards and the samples are considered by Coffey Mining to be of good quality and accuracy for the purposes of Mineral Resource estimation (*Inwood, 2010b*).

RC samples observed in the field were of suitable size and generally of consistent high recovery. Coffey Mining previously recommended that the RC sample recovery is routinely recorded and entered into the drill hole database. Based on this recommendation, Bannerman field staff undertook an analysis of the RC sample recovery in 2008. The samples were weighed before they were split and all samples returned a weight ±20kg. The rocks in the resource area are competent with very few cavities. Based on the results of the investigation Bannerman determined that routine recording of this data was superfluous as the RC sample recoveries are very high.

It is worth noting that recovery is recorded and entered into the drill hole database from all the diamond holes. From this data, it is clear that the rock is very competent with very low levels of core sample loss.

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11.2 SAMPLE PREPARATION AND ANALYSIS

11.2.1 Bureau Veritas

Bannerman has been submitting drill samples from the Etango Project to the Bureau Veritas Analytical laboratory in Swakopmund since 01 December 2011 for sample preparation and analysis. Bureau Veritas is a SADCAS/SANAS ISO/IEC 17025:2005 accredited laboratory. The samples are analysed by pressed pellet X-ray fluorescence (XRF) for uranium (and then converted to uranium oxide (U_3O_8) by calculation), niobium (Nb) and thorium (Th). Due to a lull in field exploration activities since 2012, to date the samples analysed by this laboratory account for less than 2% of the total dataset for the Etango Project.

The procedure for analysis is as follows:

- Each sample is tagged upon arrival for tracking purposes
- Each sample is dried in an electric oven at ~105°C, after which it is crushed to -2mm. The sample is then split and 500g retained for further analysis
- The 500g sample is then pulverised to 85% <75µm using a LM2 pulveriser. Four out of every ten samples are screen-checked to determine the percentage passing 75µm
- After pulverising, a 250g sub-sample is retained for analysis. From this sub-sample, approximately 7.2g is mixed with 1.2g of binder and pressed into a pellet for XRF analysis
- As part of the analytical process, the lab routinely inserts in-house blanks, standards and repeat samples for quality assurance and quality control (QAQC) purposes
- A check pulp duplicate sample is sent to Genalysis Johannesburg at the rate of one sample in twenty.

11.2.2 SGS

Initially, all primary RC and diamond core samples were sent to SGS in Johannesburg for crushing, pulverisation and chemical analysis. SGS Johannesburg is a SANAA accredited laboratory (T0169). The samples were analysed by pressed pellet X-ray fluorescence (XRF) for uranium (and then converted to uranium oxide (U_3O_8) by calculation), niobium (Nb) and thorium (Th); and by borate fusion with XRF for calcium (Ca) and potassium (K). Since December 2008, the sample preparation stages have been completed at SGS Swakopmund and then pulp samples have been forwarded to SGS Lakefield (Johannesburg) for the analysis. Analysis for Ca and K was discontinued in March 2009.

The procedure for analysis is as follows:

- Upon arrival at the laboratory, a barcode is attached to each sample to enable tracking during the preparation and analysis process
- The primary sample is dried in an electric oven at ~105°C, then crushed to -2mm and pulverised to 95% <75µm using a Labtech LM2 pulveriser
- Barren rock is run through the crushing and pulverisation circuit after every sample. The last barren rock sample from each batch is analysed using XRF and the value reported for QAQC purposes
- After pulverisation, a 200g sub-sample is retained. From this sub-sample, approximately 20g is taken for XRF analysis and 0.5g to 2g for inductively coupled

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plasma (ICP) mass spectrometry analysis. Typically, the laboratory conducts an ICP analysis in conjunction with the XRF analysis on every fifth submitted sample

- SGS Johannesburg also includes a standard and blank sample at the rate of 1:22 into the sample stream
- Replicate samples from the 200g pulverised sub-samples are taken at the rate of 2:20
- A pulp duplicate sample is sent to Genalysis Johannesburg at the rate of one sample in twenty
- For U_3O_8, Nb and Th, by XRF analysis, each sample (of approximately 17g) is combined with approximately 3g of wax binder then pressed for 2 minutes to produce a compact pellet. The pellet press is cleaned after each pellet is processed. The Bannerman samples are analysed using a Panalytical Axios XRF machine
- For Ca and K analyses by borate fusion with XRF, approximately 0.2g to 0.7g of sample is mixed with a borate flux and cast, followed by the analysis by XRF. The Ca and K analyses were discontinued in March 2009, as the values simply reflect the relative levels of calcic and potassic feldspar in the alaskite leucogranite, rather than any contribution from marble or carbonate rock in the deposit
- During periods of high demand, some of the 200g sub-samples have been sent from SGS Johannesburg to SGS Perth for the XRF analysis. The procedures used in the SGS Perth laboratory were similar to those used in the SGS Johannesburg laboratory.

11.2.3 Genalysis

The procedure for analysis at Genalysis is as follows:

- Sample preparation at Genalysis Johannesburg consists of drying the samples at ~105°C and then milling the entire sample in a LM2 pulveriser (as at SGS Johannesburg)
- A barren silica flush is put through the mill after each sample
- Every 20th pulverised sample is screen-checked to determine the percentage passing 75µm
- Analyses for U_3O_8, Th and Nb are determined by pressed pellet XRF using any of a group of Philips PW1480, PW1400 and PW2400 Axios XRF machines
- Samples are prepared using 20g of sample with 3g of binder which are mixed in a grinding vessel for 4 minutes and then pressed into a pellet in a 20t hydraulic press
- One sample of pulp is re-analysed for every 20 samples (as a duplicate) and one reference standard inserted for every 20 samples
- One blank sample is inserted per shift by the laboratory.

11.3 SAMPLE SECURITY

11.3.1 Security

The prepared and packaged diamond core and RC samples for assaying were stored in Bannerman's secure storage facility on site at Etango prior to pick up via courier. All crushing, pulverising and splitting of the samples, subsequent to the original field splitting, was performed by a reputable assaying laboratory. RC samples were taken daily from the field to the secure storage facility after the initial field splitting.

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11.3.2 Adequacy of Procedures

Drilling and sampling operations are supervised by Bannerman geologists and samples are promptly bagged. Previously, samples were taken to the storage facility in Swakopmund but they are now sent to the secure onsite storage facility at Etango prior to shipment to the assay laboratory. It is considered that Bannerman currently has appropriate provisions in place to safeguard the sample security.

Coffey Mining has visited the SGS Johannesburg facility and considers it to be well run and that the preparation and analytical methods used by SGS Johannesburg are appropriate.

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12 DATA VERIFICATION

The quality control analysis of the Bannerman assaying information has relied upon field duplicates, pulp duplicates, blanks and standards submitted by Bannerman to an umpire laboratory. Internal laboratory replicates, blanks and duplicate samples have also been analysed. The QAQC procedures undertaken have been described in detail in the previous NI 43-101 report entitled 'Etango Uranium Project, Namibia, National Instrument 43-101 Technical Document' (28 September 2011). Appendix 1 of that report presented all relevant QAQC plots and has not been duplicated in this report.

12.1 COLLAR AND DTM SURVEYS

A topographic survey has been conducted over the project area. The survey was performed by licensed surveyors using the following main instruments:

- Six Ashtech dual frequency GPS receivers
- Leica RTK 1200 GPS System (two receivers)
- Leica TC1000 single second Total Station with 3' accuracy
- Leica TC600 single second Total Station with 5' accuracy.

All survey controls were surveyed and calibrated using the Post Processing method employing the Ashtech GPS receivers and Ashtech Solutions' proprietary software.

Most of the drill hole collars were surveyed prior to the resource estimate using the Leica RTK GPS or the Leica Total Stations.

12.2 ASSESSMENT OF QUALITY CONTROL DATA

The quality control data related to RC and diamond core drilling has been assessed statistically using a number of comparative analyses for each dataset. The objectives of these analyses were to determine relative precision and accuracy levels between various sets of assay pairs and the quantum of relative error. The results of the statistical analyses are presented as summary statistics and plots, which include the following:

- *Thompson and Howarth Plot*, showing the mean relative percentage error of grouped assay pairs across the entire grade range, used to visualise precision levels by comparing against given control lines
- *Rank % HARD Plot*, which ranks all assay pairs in terms of precision levels measured as half of the absolute relative difference from the mean of the assay pairs (% HARD), used to visualise relative precision levels and to determine the percentage of the assay pairs population occurring at a certain precision level. For pulp-based duplicate samples, a limit of 10% HARD is a useful limit to compare and analyse precision from different datasets. For field duplicates, a limit of 20% HARD is a useful limit to compare and analyse precision from different datasets
- *Correlation Plot,* which is a simple scatter plot of the value of assay 1 against assay 2. This plot allows an overall visualisation of precision and bias over selected grade ranges. Correlation coefficients are also determined
- *Quantile-Quantile (Q-Q) Plot* is a means where the marginal distributions of two datasets can be compared. Similar distributions should be noted if the data is unbiased

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- For standards and blanks, the *Standard Control Plot* shows the assay results of a particular reference standard over time. The results can be compared to the expected value, and the tolerance limits (usually ± two standard deviations) precision lines are also plotted, providing a good indication of both precision and accuracy over time.

12.2.1 Standards Analysis

This section covers the analysis of both the Bannerman and laboratory inserted standards.

Bannerman Submitted Standards

Bannerman has routinely inserted blanks and certified standards into its sampling stream since December 2007. The standards include two certified commercial standards by African Mineral Standards (AMIS) (AMIS0029 and AMIS0045) sourced from the Dominion Reef and Witwatersrand area; and two AMIS certified standards sourced from Anomaly A mineralised material (ANMIS0085 and AMIS0086). The Anomaly A Standards were prepared by AMIS for commercial use and have been subject to an international round robin test regime.

Most of the datasets analysed exhibited outlying results, the majority of which approximated other known standards and can be attributed to sample mixing during the sample submission/ recording process. Those results were trimmed from the sample population prior to analysis. The summary statistics for these standards are presented in Table 12.1.

Table 12.1
Statistics for Bannerman Submitted Standards (U ppm)

Standard	XRF – U ppm						
	AMIS0029		AMIS0045		AMIS0085	AMIS0086	Blank
	SGS_J	GEN_P	SGS_J	GEN_P	SGS_J	SGS_J	SGS_J
Expected Value (EV)	890	890	87	87	266	128	5
EV Range	862-918	862-918	75-99	75-99	250.6-284	115-148	0 - 10
Count	238	83	241	47	912	908	3463
Minimum	795	840	81	85	93	89	5
Maximum	962	924	104	94	386	170	215
Mean	**927**	**892**	**93**	**88**	**270**	**135**	**5.5**
Std Deviation	16	28	3.5	1.7	12.9	6	7.6
% in Tolerance	19	58	94	100	93	97.6	99
% Bias	4	0.2	7	1	2	5	9.9

Standard AMIS0029, sourced from the Dominion Reef, has a known complex mineralogy and metallurgy which may be affecting the expected value (EV) of the batches analysed. Results for both Genalysis Perth and SGS Johannesburg exhibit similar positive biases. AMIS0029 standards were submitted to SGS Johannesburg up to August 2008, when potential issues with this standard were first identified, and then submitted briefly during May 2009. Results for these later submissions indicate the same problems with bias, and no more of these standards were submitted to SGS Johannesburg after this period. Results from Genalysis Perth for December 2008 onwards exhibit a pronounced switch from a positive bias to a negative bias, possibly as a result of re-calibration or change of standard batch material used by the laboratory.

Both AMIS0085 and AMIS0086 assay data reported by SGS Johannesburg exhibit a distinct change toward a much lower positive bias from approximately July 2009 onwards.

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AMIS standards submitted to SGS Johannesburg (the primary laboratory) exhibit a positive bias ranging from 1% to 8%. The same standards submitted to the Umpire laboratory (Genalysis Perth) exhibit 0 to 2% bias. The SGS standards, with the exception of AMIS0029 (which has known issues), report >93% within tolerance limits.

The majority of the blanks submitted to SGS Johannesburg report assays less than 5ppm U. Removal of outliers close to values of known standards produced 25 assays reporting greater than 10ppm U and up to 70ppm U. Some of the higher grade results are considered to reflect the mixing of blanks with actual samples during the sampling process, and potentially due to sample contamination.

SGS Internal Standards

Three certified standards (UREM2, UREM4, and UREM9) and two blank standards (Waste Rock and Lab Blank) were identified in the database for internal use by SGS Johannesburg. One blank standard (Waste Rock) and one certified standard (SY3) were identified for SGS Perth. The summary statistics for these standards are shown below in Table 12.2.

Table 12.2
Statistics for SGS internal Standards (U ppm)

Standard	SGS Johannesburg – XRF					SGS Perth - XRF	
	UREM2	UREM4	UREM9	Waste Rock	Lab Blank	SY3	Waste Rock
Expected Value (EV)	428	84	219	1	1	645	1
Expected Value Range	364-492	72-98	186-252	0-15	0.9-1	580-709	0-15
Count	1084	1534	672	1626	6877	148	188
Minimum	416	69	191	1	1	634	1
Maximum	460	99	238	20	1	656	13
Mean	**435**	**88**	**223**	**1**	**1**	**641**	**2.1**
Std Deviation	7.9	3.3	6.1	1	0	4.2	1.8
% in Tolerance	100	100	100	100	100	100	100
% Bias	1.6	3.9	2.1	4.3	0	-0.6	116

The certified UREM standards used by SGS Johannesburg all report within tolerance limits with overall positive bias ranging between 1% and 4%. Both UREM2 and UREM4 exhibit a marked reduction in bias from approximately July 2009 onwards. This correlates with trends observed for the Bannerman submitted standards.

The SGS Johannesburg blank standard Waste-Rock (n=1632) exhibits some minor contamination throughout the sample runs and possible incorrect sample identification / submission, with 11 samples reporting above 10ppm U. The laboratory blank (n=6877) reports consistently at 0ppm U. The blank samples indicate no significant contamination during the assaying process.

The internal certified standard (SY3) results by SGS Perth display acceptable accuracy. All results report within acceptable tolerance with less than 1% overall bias.

The blank standard Waste Rock from SGS Perth (n=188) has nine samples over 5ppm, indicating minor contamination. The majority of these results are restricted to the reporting period for June 2007. The results are considered acceptable.

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Genalysis Perth Internal Standards

Seven internal standards (BL-1, SARM1, UREM1, UREM2, UREM4, UREM9 and UREM11) and one laboratory blank were identified in the database, Table 12.3.

Table 12.3
Etango Project – Statistics for Genalysis Perth internal Standards (U ppm)

Standard	XRF – Genalysis Perth							
	BL-1	SARM 1	UREM 1	UREM 2	UREM 4	UREM 9	UREM 11	Control Blank
Expected Value (EV)	220	15	28.8	428	84.8	218.8	58.5	1
Expected Value Range	187-242	13-17	24-33	364-492	72-98	186-252	50–67	0.9/1.1
Count	56	90	7	50	18	15	8	210
Minimum	214	12	26	410	81	204	55	1
Maximum	229	24	34	463	93	223	58	5
Mean	**223**	**16**	**28**	**421**	**84**	**215**	**56.5**	**1**
Std Deviation	4.02	2.79	2.51	10.21	3.39	5.56	1.12	0.3%
% in Tolerance	100%	79%	86%	100%	100%	100%	100%	99.5%
% Bias	1.3%	6.3%	-2.8%	-1.5%	-0.4%	-1.8%	-3.4%	1.9%

All of the standards, except SARM1, report good accuracy with the bulk of the samples returning assays within the set precision limits. Bias in the laboratory standards varies from -3.5% to 6.3%. Control blank standards (n=210) were identified for analysis. Only one of the control blank results exhibited signs of contamination.

12.2.2 Duplicates and Umpire Assaying Analysis – Precision

The database for the Etango deposit contains duplicate sample information for field re-splits (RC, ½ and ¼ diamond core); umpire pulp re-assays and laboratory pulp replicate assays. No intra-laboratory pulp re-splits were identified.

Original samples collected prior to 2009 were crushed and pulverised at SGS Johannesburg and analysed at either SGS Johannesburg or SGS Perth. From March/April 2009, original samples have been crushed at the sample preparation facility in Namibia, and from July 2009 samples were no longer analysed at SGS Perth. The field duplicate samples were crushed and pulverised at Genalysis Johannesburg. All primary field duplicate and umpire pulp samples were analysed at Genalysis Perth prior to 2008. From January 2008, field duplicate samples are crushed, pulverised and analysed by SGS.

The summary statistics for the duplicate analyses are shown in Table 12.4. A lower limit of 0ppm U was applied to the data prior to precision analysis.

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Table 12.4
Etango Project – Summary of Data Precision for SGS and Genalysis Laboratories for XRF Analysis of Uranium U (ppm)

Sample Type	Number of Data pairs		Comparative Means (ppm) (Original Lab/duplicate Lab)		% within Rank HARD Limits (10% / 20%)	
	SGS - JB	SGS - Perth	SGS - JB	SGS - Perth	SGS - JB	SGS – Perth
Umpire RC Field Duplicates [1]	3,175	401	91/89	99/110	60 / 74	57/ 72
Umpire Diamond Field Duplicates [1]	430	-	108/109	-	57 / 73	- / -
Umpire RC Pulp Duplicates [2]	4,606	257	81/77	75/80	66 / 78	54 / 70
Umpire Diamond Pulp Duplicates [2]	512	7	86/83	24/19	71 / 78	43 / 57
Internal RC Laboratory Pulp Repeats [3]	6,243	682	74/73	80/79	93 / 96	66 / 81
Internal Diamond Laboratory Pulp Repeats [3]	842	37	102/102	57/56	96 / 97	57 / 65

1 Duplicate samples crushed at SGS Johannesburg and analysed at Genalysis Perth
2 Pulp duplicates analysed at Genalysis Perth
3 Pulp repeats analysed at SGS

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Table 12.5 summarises the results of a series of separate campaigns (undertaken in September 2008) of check duplicate analysis to gauge the relative precision and accuracy of Setpoint laboratories in Johannesburg and ALS Chemex in Johannesburg as well of comparing the difference between XRF and ICPMS analysis at SGS Perth.

Table 12.5
Etango Project – Inter Laboratory Pulp Comparisons U (ppm)

Sample Type	No. of Data Pairs	Mean % HARD	Median % HARD	% Within Rank HARD Limits (10%/20%)	Comparative Means (ppm) (Original Lab/ Duplicate Lab)
ALS JB versus Setpoint JB – XRF	920	12.4	10.1	49/87	197/230
SGS JB versus Setpoint JB – XRF	488	15.3	8.3	58/80	202/203
SGS JB vs. ALS JB – XRF	459	14.8	9.2	50/75	214/188
SGS Perth – XRF versus ICPMS	406	10.8	6.1	67/86	174/184

Umpire Field Duplicates

The umpire laboratory field duplicates overall exhibit moderate precision. Samples assayed at SGS Johannesburg show moderate to good precision with the Genalysis duplicates; 74% of RC field duplicates and 73% of the diamond duplicates lie within a 20% Rank HARD limit. Both laboratories also reported similar means for each dataset (91ppm versus 89ppm U for the RC and 108ppm versus 109ppm U for diamond duplicates).

SGS Perth exhibits moderate precision when compared to Genalysis with 72% of the RC duplicates within a 20% Rank HARD limit. The SGS Perth RC samples report a significantly lower mean of 99ppm U versus 110ppm U, indicating a 9% bias. The bias is most pronounced for original samples having greater than 500ppm U.

Umpire Pulp Duplicates

Correlation coefficients contained in this section of the report are listed as Pearson then Spearman values unless otherwise stated.

The RC pulp duplicates for SGS Johannesburg exhibit moderate precision, with 66% of RC pulp duplicates within a generally acceptable limit of 10% Rank HARD, and correlation coefficients of 0.99 and 0.97 respectively. Comparative means between the two laboratories of 81ppm versus 77ppm U indicate a 5% overall relative positive bias in the results from SGS Johannesburg.

The diamond core pulp duplicates for SGS Johannesburg exhibit moderate precision, with 71% of the data within a generally acceptable limit of 10% Rank HARD and correlation coefficients of 0.98 and 0.96. Comparative results between the two laboratories are close, with means of 86ppm versus 83ppm, indicating a 3% overall positive bias in the results from SGS Johannesburg.

The RC pulp duplicates for SGS Perth exhibit poor to moderate precision, with 54% of the data within a generally acceptable limit of 10% Rank HARD, and correlation coefficients of 0.98 and 0.96. Comparative means between the two laboratories of 75ppm versus 80ppm U

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for SGS Perth and Genalysis Perth respectively indicates a 6% relative bias between the two laboratories. The relative bias is most pronounced for samples above 300ppm U.

The diamond pulp duplicates for SGS Perth, although analysed, are considered to be too few in number (n=7) to provide a meaningful comparison.

Laboratory Pulp Repeats (Replicates)

The internal laboratory RC and diamond core pulp replicates for SGS Johannesburg exhibit a high precision, with 93% and 96% of the data within a 10% Rank HARD limit. Correlation coefficients are 0.98 for the RC repeat pulps and 1.00 for diamond pulp repeats. The means for the original and repeat samples are comparable, with 73.87ppm U and 73.33ppm U for RC samples, and 101.99ppm U and 101.95ppm U for diamond samples.

RC pulp repeats for SGS Perth exhibit poor to moderate precision, with 66% of data within a 10% Rank HARD limit, and correlation coefficients of 0.99 and 0.95. The means are comparative, 80.49 ppm U and 78.78ppm U respectively, with an indicated 2% bias. Diamond pulp repeats exhibit generally poor to moderate precision, with 57% of data within a 10% Rank HARD limit, and correlation coefficients of 1.00 and 0.93. Consideration should be given to the relatively small population of diamond pulp repeats (n=37) used for analysis.

Inter-laboratory and XRF versus ICPMS Comparisons

The results from the inter-laboratory comparison conducted in September 2008 indicate that for all laboratories, relatively low to moderate precision (47% to 55% of the data within a 10% Rank HARD precision limit) is achieved when comparing the pulp samples.

The results indicate that Setpoint and SGS report similar means (203ppm versus 202ppm U, n=488) and that both Setpoint and SGS report higher than ALS-Chemex (ALS) with the comparison of Setpoint versus ALS (n=920) reporting means of 230ppm U versus 197ppm U (a 16% relative global bias); and the comparison of SGS versus ALS (n=459) reporting means of 214ppm U versus 188ppm U (a 14% relative global bias).

The comparison of XRF to ICPMS analysis conducted at SGS Perth indicates that for the 406 samples analysed, the ICPMS method results in a slightly higher global mean for 184ppm versus 174ppm U (or 5.7%).

Discussion

Analyses of the Bannerman standards indicates that the SGS Johannesburg laboratories are reporting a relative bias of between 1% and 8% higher than the expected values for these standards. It is also noted that the SGS internal standards exhibit a bias of 1% to 4%. Genalysis reports a negative bias of ~-2% for the same standards (UREM 2, 4 and 9).

The duplicate data for SGS Johannesburg indicates that whilst the internal repeatability is excellent for replicates, there is an overall bias of 5% compared with pulp duplicates sent to Genalysis Perth. This bias is not seen, however, with the field duplicates sent to Genalysis (particularly when outliers are removed) as the means are comparable. It is interesting to note that the Inter-laboratory comparison conducted in September 2008 shows that ALS and Setpoint in Johannesburg report similar means overall, and both laboratories report 14% to 16% higher than ALS (Figure 12.1).

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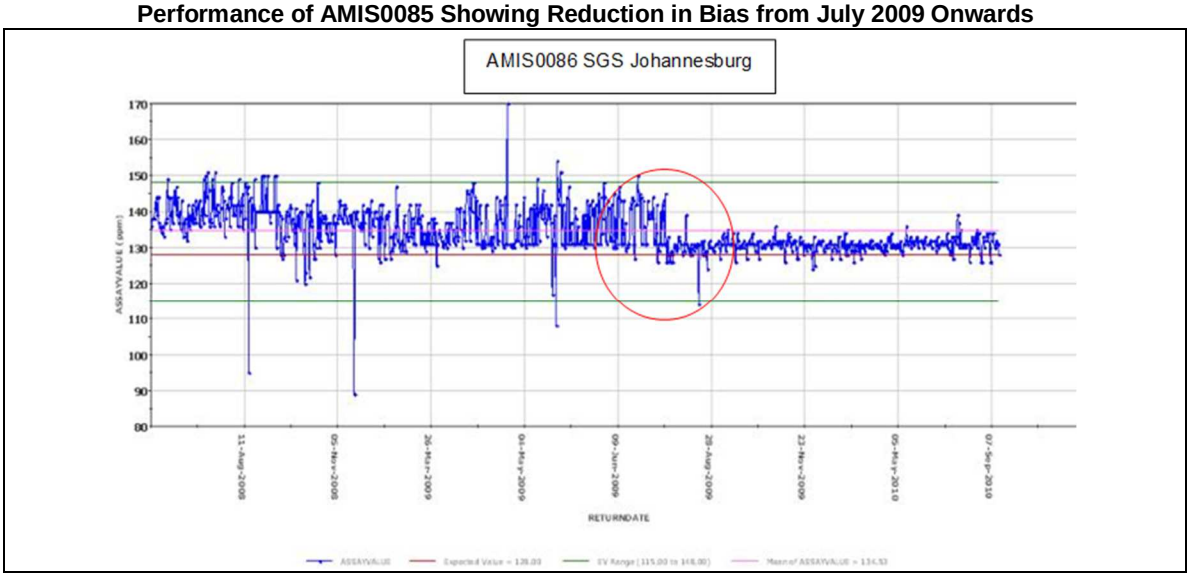
The trend of the bias seen at SGS Johannesburg is of minor concern. However, this is tempered by the relatively good correlation seen with the field duplicates; the overall similar correlation seen between the SGS and Setpoint assays; and the generally good standards performance from SGS Johannesburg. Of particular note is the marked improvement and reduction in bias for standards since mid-2009. This change is exhibited for both AMIS 0085 and AMIS0086 standards submitted by Bannerman, and in the SGS lab standards UREM2 and UREM4 (see Table 12.1).

Figure 12.1

Performance of AMIS0085 Showing Reduction in Bias from July 2009 Onwards



The results of the pulp duplicates for SGS Perth indicate a general negative bias with respect to Genalysis in the order of 6%. This potential bias should be tested with the insertion of industry standards to the SGS Perth laboratory for any future samples sent and further action taken as necessary.

The following recommendations are made in relation to the QAQC protocols for the Etango Project:

- Follow-up investigations should be undertaken with SGS Johannesburg regarding the cause of the potential bias seen in the internal laboratory standards and umpire assaying
- Standards AMIS0085 and AMIS0086 (and any other Bannerman standards) should be sent regularly to Genalysis along with the regular umpire duplicate samples
- Intra-laboratory blind pulp replicates should be undertaken at a nominal rate of 1:20
- A further high grade standard should be sourced to supplement AMIS0029.

12.3 INDEPENDENT SAMPLING

Coffey Mining visited the Anomaly A/Oshiveli site during April 2008 and collected samples for the purposes of independent sampling. A total of 40 RC samples from GARC0362 were placed into plastic bags with numbered security tags attached directly after drilling and splitting in the field. Once tagged, the bags were sent to Bannerman's sample storage yard for processing.

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Ten diamond samples were also collected from GOADH042. These were collected from the core tray located at Bannerman's core shed, then placed in plastic bags with numbered security tags attached. The tagged samples were then sent to the SGS Johannesburg laboratories where the security tags were inspected by Coffey Mining personnel, prior to sample preparation.

Figure 12.2
Etango Project – Samples Tagged for Independent Sampling

The assay results from the samples are shown in Table 12.6. The results illustrate typical examples of mineralisation from the property, with a maximum value of 1,392ppm U3O8 from sample A26295. The average of the 40 RC samples collected from hole GARC0361 was 235ppm U_3O_8. The average of the 10 diamond samples collected was 13ppm U_3O_8.

Table 12.6
Etango Project – Independent Sampling Results

Hole ID	From	To	Sample ID	U_3O_8 (ppm)	Hole ID	From	To	Sample ID	U_3O_8 (ppm)
RC Samples									
GARC0362	0	1	A26281	4.99	GARC0362	20	21	A26302	24
GARC0362	1	2	A26282	4.99	GARC0362	21	22	A26303	76
GARC0362	2	3	A26283	16	GARC0362	22	23	A26304	232
GARC0362	3	4	A26284	30	GARC0362	23	24	A26305	137
GARC0362	4	5	A26285	15	GARC0362	24	25	A26306	127
GARC0362	5	6	A26286	14	GARC0362	25	26	A26307	194
GARC0362	6	7	A26287	14	GARC0362	26	27	A26308	610
GARC0362	7	8	A26288	173	GARC0362	27	28	A26309	584
GARC0362	8	9	A26289	176	GARC0362	28	29	A26310	62
GARC0362	9	10	A26290	156	GARC0362	29	30	A26311	135

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Table 12.6
Etango Project – Independent Sampling Results

Hole ID	From	To	Sample ID	U_3O_8 (ppm)	Hole ID	From	To	Sample ID	U_3O_8 (ppm)
GARC0362	10	11	A26291	162	GARC0362	30	31	A26312	178
GARC0362	11	12	A26292	217	GARC0362	31	32	A26313	35
GARC0362	12	13	A26293	557	GARC0362	32	33	A26314	141
GARC0362	13	14	A26294	1008	GARC0362	33	34	A26315	292
GARC0362	14	15	A26295	1392	GARC0362	34	35	A26316	377
GARC0362	15	16	A26296	453	GARC0362	35	36	A26317	211
GARC0362	16	17	A26297	446	GARC0362	36	37	A26318	200
GARC0362	17	18	A26298	151	GARC0362	37	38	A26319	410
GARC0362	18	19	A26299	299	GARC0362	38	39	A26321	4.99
GARC0362	19	20	A26301	87	GARC0362	39	40	A26322	12
Diamond Samples									
GOADH0042	6.79	7.79	J2436	4.99	GOADH0042	11.79	12.79	J2441	4.99
GOADH0042	7.79	8.79	J2437	4.99	GOADH0042	12.79	13.79	J2442	20
GOADH0042	8.79	9.79	J2438	4.99	GOADH0042	13.79	14.79	J2443	62
GOADH0042	9.79	10.79	J2439	4.99	GOADH0042	14.79	15.79	J2444	13
GOADH0042	10.79	11.79	J2440	4.99	GOADH0042	15.79	16.79	J2445	4.99

12.4 ASSESSMENT OF PROJECT DATABASE

Based upon Coffey Mining's analysis of the duplicates data and the laboratory-based standards data, the Bannerman assaying is considered to meet industry acceptable standards for sample accuracy and precision and is acceptable for use in resource estimation studies.

From November 2007, Bannerman has used the Acquire commercial database software system to manage its drill hole data. The use of such database management software is considered to be of high industry standard as it enables the incorporation of large datasets into an organised, auditable structure. Checks by Coffey Mining have identified no material issues with the database and it is considered acceptable for use in resource estimation.

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13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

Significant test work had been completed and reported in the previous Bannerman 43-101 report (Bannerman, September 2011). Conclusions documented in the aforementioned report are the foundation for the additional test work that has been completed.

The key conclusions that have shaped the development of the project are:

- Pre-concentration of the ore is not practical or cost effective and is therefore not included in the preferred process design. The following beneficiation options have been tested at bench-top scale:
  - Scrubbing and screening
  - Flotation
  - Heavy media separation of a coarse (+0.5mm) fraction
  - Gravity beneficiation of fines through either a Knelson or Falcon concentrator.
- Both agitated leaching and heap leaching have been tested in acidic environments in the laboratory, and heap leaching is the preferred method for extracting uranium from the ore on economic grounds
- Optimal economics for the heap leach were achieved from ore crushed to -8mm (P_{80}=5.3mm), using HPGR as the final stage of crushing
- A suite of standard comminution and crushing tests have been completed on a range of samples. This database of standard indices is suitable for designing an appropriate crushing and/or grinding circuit
- Extraction isotherm tests were conducted on the leach solution from column tests, indicating good solvent extraction (SX) characteristics.

13.2 SAMPLE DESCRIPTION

Samples were provided as whole HQ[1], ½ NQ2 and ¼ NQ core. NQ Core was retained for planned variability testing to follow the current program of test work.

Whole HQ core was selected, drilled and supplied specifically for metallurgical testing and formed the basis for the test work.

A number of composite samples have been tested throughout the various programs. The sample descriptions and source locations are summarised in Section 13.2.2.

13.2.1 Ore Types

The bulk of resource tonnage is present in four material types, of which alaskites Type D and E (see section 7.2 for more details) represent approximately 65% and 22.5% of the current Mineral Resource respectively. The remainder is made up of peripheral metasediments.

[1] HQ = 96mm drill hole, core diameter of 63mm; NQ = 75.7mm drill hole, core diameter of 53mm

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Metallurgical testing proceeded on the basis of selecting intervals of core above a cut-off grade of 100ppm U_3O_8. Typically, Metasediments were represented at the boundaries of Type D and Type E alaskite intervals as ore grade or as waste grade, based on a 10% allowance for dilution.

Earlier test work considering the two main alaskite types found little or no significance difference for metallurgical performance. Subsequent work has focussed on whole of ore performance.

13.2.2 Ore Characterisation

A composite Type D and Type E alaskite sample was prepared in a ratio of 2:1, and submitted for ICP multi element scan and chemical assay. The results for major analytes are provided in Table 13.1.

Table 13.1
Etango Alaskite Composite Sample – Assay Uranium and Potential Organic Co-extracted Species

Analytes	Species	Method	Detection Limit	Unit	Assay
Uranium	U_3O_8	ICP-MS	0.05	ppm	251
Uranium	U_3O_8	XRF	0.001	ppm	240
Vanadium	V_2O_5	ICP-OES	2	ppm	25
Niobium	Nb	ICP-MS	0.2	ppm	5
Molybdenum	Mo	ICP-MS	0.1	ppm	1
Silicon	Si	-	-	%	34.7
Arsenic	As	ICP-MS	1	ppm	2
Zircon	Zr	ICP-OES	5	ppm	92
Tungsten	W	ICP-MS	1	ppm	4
Bismuth	Bi	ICP-MS	0.1	ppm	<0.1
Thorium	Th	XRF	0.001	ppm	62

The composite was prepared to approximate the intended head grade for the study.

Low levels of potential impurity elements are present and the level of silica is considered typical, given the mineralogy of the host rock.

The analyses of potential organic loading retardants are shown in Table 13.2, and represents generally low levels.

Chloride analysis was extended to include total and water soluble forms. Chloride levels were shown to increase up to 2500ppm within the weathered part of the resource (0m to 40m), and were also shown to be present as 80% to 90% water soluble species, likely present due to wind transport from the Atlantic ocean.

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Table 13.2 Assay Potential Organic Loading Retardants for Alamine 336 Extractant					
Analytes	**Species**	**Method**	**Detection Limit**	**Unit**	**Assay**
Phosphorous	P_2O_5	ICP-OES	30	ppm	252
Sulphur	S	ICP-OES	20	ppm	100
Chloride	Cl	-	-	ppm	70

Other potential loading retardants such as N (as NO_3^-), F and SCN^- will be measured during further large scale and leach variability testing.

In the context of SX with Alamine 336, the species shown in Table 13.3 are typically rejected and, again, represent generally low levels. The iron assay represents the amount of natural total iron in the ore. Ferric iron is required to promote oxidative leaching.

Table 13.3 Ore Assays for Elements Rejected by Alamine 336 SX Extractant					
Analytes	**Species**	**Method**	**Detection Limit**	**Unit**	**Assay**
Iron	Fe	ICP-OES	0.1	%	1.02
Magnesium	Mg	ICP-OES	0.002	%	0.11
Calcium	Ca	ICP-OES	0.01	%	0.88
Sodium	Na	ICP-OES	0.005	%	1.55
Potassium	K	ICP-OES	0.01	%	5.11
Aluminium	Al	ICP-OES	0.01	ppm	7
Titanium	Ti	ICP-OES	10	ppm	370
Chromium	Cr	ICP-OES	50	ppm	110
Manganese	Mn	ICP-OES	10	ppm	150
Cobalt	Co	ICP-MS	2	ppm	2
Nickel	Ni	ICP-OES	5	ppm	7
Copper	Cu	ICP-OES	1	ppm	2
Zinc	Zn	ICP-OES	5	ppm	13

13.3 MINERALOGY

Mineralogical identification and deportment were first assessed using SEM/EDS after which a quantitative evaluation was performed using QEMSCAN.

13.3.1 SEM Analysis

Mineralogical identification and deportment was evaluated based on core samples selected over 100m of HQ drill hole (GOADH0048).

Samples were classified as coarse grained biotite granites (uraniferous alaskite) dominated by feldspars mostly in the range 2-4mm. Biotite/chlorite flakes were noted as typically sub-500µm in size. Dominant acid soluble mineralisation was identified as uraninite (UO_2) and uranothorite $(U,Th)SiO_4$; minor proportions of complex refractory double oxides brannerite $(U,Ca,Ce)(Ti,Fe)_2O_6$ and polycrase $(Y,Ca,Ce,U,Th)(Ti,Nb,Ta)_2O_6$ were also identified.

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Uraninite occurred typically as grains ranging from sub-20µm up to 100-200µm grains within fractures partially in-filled with carbonates, typically calcite, and secondary silicates; variable length (10-1000µm) narrow veins through quartz, plagioclase and chlorite: and as narrow (10-40µm) bands at quartz plagioclase contacts.

Brannerite occurred as: 50-100µm lenticular grains within the basal cleavage planes of phyllosilicate minerals, biotite and chloriteas numerous sub-20µm strips within the core of biotite and in minor proportions as 100µm strips in plagioclase feldspar cleavages.

Uranothorite occurred as 90-100µm discrete grains either in plagioclase or at quartz potash feldspar contacts. Notably, individual mineral grains were either surrounded or intersected by fractures through plagioclase and feldspar, indicating the potential for uranium mineral exposure at coarse size.

Polycrase and uraniferous monazite were present in minor to trace amounts.

13.3.2 QEMSCAN Analysis

QEMSCAN analysis was performed by the University of Witwatersrand. Samples of core were prepared as size fractions: -355µm/+208µm and -208µm/+90µm.

Uranium Deportment by Mineral Phase

The deportment of uranium associated with each uraniferous mineral phase is shown in Table 13.4, with the dominant mineralisation identified as uraninite and the uraniferous silicates coffinite, boltwoodite and uranothorite. Uraniferous phosphate mineralisation was identified as autunite.

Table 13.4
Uranium Deportment by Mineral Phase

Sample Number	**DH-010-2**	**DH-010-5**	**DH-010-7**	**DH-010-7**
Size Fraction (µm)	**-355µm/+208µm**	**-208µm/+90µm**	**-355µm/+208µm**	**-208µm/+90µm**
Mineral	**% Uranium Hosted by Phase**			
Uraninite	41.68	52.66	84.14	95.64
Uranium Silicates	53.25	43.86	12.43	3.78
Uranium Phosphates	4.73	3.21	3.16	0.54
Betafite/Pyrochlore	0.33	0.27	0.26	0.04
Total	**100.00**	**100.00**	**100.00**	**100.00**

Mineral Abundance

QEMSCAN modal analysis is presented in Table 13.5 and is consistent with the SEM analysis of metallurgical core.

Table 13.5
QEMSCAN Modal Abundance

Sample Number	**DH-010-2**	**DH-010-5**	**DH-010-7**	**DH-010-7**
Size Fraction (µm)	**-355µm/+208µm**	**-208µm/+90µm**	**-355µm/+208µm**	**-208µm/+90µm**
Mineral	**Mass (%)**	**Mass (%)**	**Mass (%)**	**Mass (%)**
Uraninite	0.04	0.11	0.01	0.01

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Table 13.5 QEMSCAN Modal Abundance				
Sample Number	**DH-010-2**	**DH-010-5**	**DH-010-7**	**DH-010-7**
Size Fraction (µm)	**-355µm/+208µm**	**-208µm/+90µm**	**-355µm/+208µm**	**-208µm/+90µm**
Mineral	**Mass (%)**	**Mass (%)**	**Mass (%)**	**Mass (%)**
U – Silicates	0.08	0.10	0.01	0.00
U – Phosphates	0.01	0.01	0.00	0.00
Betafite / Pyrochlore	0.00	0.00	0.00	0.00
Quartz	34.9	32.5	25.7	28.8
K_Feldspar	14.5	36.4	52.1	54.9
Ab_Feldspar	40.2	24.9	13.7	11.4
Chlorite	1.9	1.8	1.1	0.9
Biotite	6.1	0.7	0.2	0.1
Muscovite	0.5	1.1	1.3	1.0
Calcite	0.1	0.1	4.4	1.6
Fe Oxides / Hydroxides	0.3	0.1	0.1	0.0
Ilmenite / Rutile	0.0	0.0	0.0	0.0
Apatite	0.2	0.6	0.1	0.0
Zircon	1.2	1.5	1.3	1.1
Gypsum	0.0	0.0	0.0	0.0
Other	0.0	0.0	0.0	0.0
Total	**100.0**	**100.0**	**100.0**	**100.0**

Liberation Analysis

QEMSCAN liberation class data is provided in Table 13.6. Results for the two groups were similar, showing liberation of the upper size fraction at relatively coarse fragmentation. This will also result in exposure of the fine grained uranium mineralisation concentrated at the grain boundaries and fracture planes.

Table 13.6 provides the results for all uraniferous minerals identified.

Table 13.6 Liberation Class Data: All Uraniferous Phases					
Classification		**Locked**	**Middling's**	**Liberated**	**Total**
Sample Number	**Size Fraction**	**Area <= 30%**	**Area >30% <=80%**	**Area >80%**	
DH-010-2	-355µm/+208µm	60.1	8.4	31.5	100.0
DH-010-5	-208µm/+90µm	24.7	21.7	53.7	100.0
DH-010-7	-355µm/+208µm	99.6	0.0	0.4	100.0
DH-010-7	-208µm/+90µm	24.7	42.2	33.1	100.0

13.4 COMMINUTION CHARACTERISTICS

The comminution properties of Etango ore were characterised based on selected intervals of whole HQ core. Diamond hole locations were selected to intersect the main ore body and

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represent the ore along and across the resource and at depth. The following core provided samples for the comminution test work:

- Preliminary characterisation based on selected intervals from GOADH0048
- Variability testing utilised intervals derived from GOADH0048, GOADH0058, GOADH0059 and GOADH0060
- HPGR Pilot test work was performed on selected intervals from GOADH0062, GOADH0063, GOADH0064, GOADH0065 and GOADH0066.

General sample locations are identified in Table 13.7.

Table 13.7
Comminution Test work Samples – Composition and Source Location

Hole ID	Prospect	Approximate Location in Etango Orebody	Depth	Drilling End Date
GOADH0048	ANOMALY_A	Central area	101.25	25-Apr-08
GOADH0058	ANOMALY_A	Northern End	190.19	30-Jun-08
GOADH0059	ANOMALY_A	Central area	219.31	7-Jul-08
GOADH0060	ANOMALY_A	Southern End	102	10-Jul-08
GOADH0062	ANOMALY_A	Central area	111	17-Jul-08
GOADH0063	ANOMALY_A	Northern End	165.26	22-Jul-08
GOADH0064	ANOMALY_A	Central-west area	84	24-Jul-08
GOADH0065	ANOMALY_A	Northern End	213.7	4-Aug-08
GOADH0066	ANOMALY_A	Northern End	198.29	6-Aug-08

Interval selection was based on 10% dilution. Dilution typically occurred either as Khan and Chuos metasediments at the edges of ore grade boundaries, or waste grade Type D and Type E alaskite.

13.4.1 Preliminary Characterisation GOADH00488

The following abbreviations have been used to describe routine comminution tests performed.

- UCS – Unconfined Compressive Strength
- DWT – JK proprietary impact breakage test
- SMC – SMCC impact breakage test similar to the JK DWT
- CWi – Bond Crushing Work Index
- RWi – Bond Rod Mill Work Index
- BWi – Bond Ball Mill Work Index
- Ai – Bond Abrasion Index.

Approximately 100m of whole HQ core from diamond hole GOADH0048 was used in the preliminary characterisation. Composites of 5m to 6m were prepared.

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Discussion of results is largely limited to those relevant for heap leach processing, i.e. crushing and HPGR testing.

JK Drop Weight Test Comp-48 DWT

A single 6m composite was prepared across all intervals and subjected to a full JK Drop Weight test. In this test particles in five size ranges are tested for impact breakage resistance.

Results of the Composite Comp-48DWi test are shown in Table 13.8, and indicate relatively low resistance to impact breakage. The abrasion resistance t_a value of 0.48 indicates a medium resistance to abrasion-style comminution.

Table 13.8
JK Drop Weight Test GOADH0048

Parameter	A	b	A*b	t_a
Value	65.9	1.21	79.7	0.48

13.4.2 Comminution Variability

Comminution variability testing was performed on whole HQ test core from GOADH0048, GOADH0058, GOADH0059 and GOADH0060 and results considered for the two main alaskite types, Type D and Type E, and the overall set of 20 test samples that includes two samples from other ore types. The results for each test type are summarised in Table 13.9 in terms of the average measured value, the coefficient of variation of that value and the typical design selection point. The coefficient of variation (COV) is simply the ratio of the standard deviation to the mean value and is a relative measure of variability amongst the test samples.

Table 13.9
Comminution Properties Summary

Comminution Measure	Alaskite Type D	Alaskite Type E	Overall
BWI Average (kWh/t)	14.5	14.9	14.7
BWI COV (%)	5	4	5
BWI 80th Percentile			15.2
RWI Average (kWh/t)	12.2	12.3	12.0
RWI COV (%)	2	6	8
RWI 80th Percentile			12.7
DWI Average (kWh/m^3)	3.5	3.2	3.3
DWI COV (%)	15	18	19
DWI 80th Percentile			3.8
CWI Average (kWh/t)	8.3	8.1	8.1
CWI COV (%)	13	19	15
CWI 80th Percentile			9.3
Ai Average	0.34	0.27	0.30
Ai COV (%)	5	35	27
Ai 80th Percentile			0.35
UCS Average (MPa)	77	69	71
UCS COV (%)	24	29	27
UCS Max	100	119	119

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The results suggest that there are only minor differences between the two alaskite types and that for the purposes of comminution design they can be considered as part of a single ore type population. The most significant difference is for the Ai results where the variability is much higher (35% COV) than for the Type D Alaskite (only 5% COV).

For a heap leach circuit the main processing action is crushing and there is no grinding. Consequently the BWI and RWI have little relevance to the design calculations. The UCS and crushing work index (CWi) values are comparatively low and this means the ore will crush easily. The DWI is also low confirming low crushing power requirement. The Ai value is modest and indicates that crusher liners will be subject to intermediate rates of wear.

None of these design values is extreme and no special considerations are required when designing comminution equipment.

13.4.3 High Pressure Grinding Rolls Pilot Test work

Leaching and mineralogical test work demonstrated that the ore exhibits a high degree of uranium exposure (liberation for leaching purposes) at coarse size. SEM investigations showed that the both of the main material types presented uranium on the natural fracture boundaries within the mineral structure. HPGR crushing preferentially breaks minerals on natural fracture boundaries, and comminution indices indicate that the ore is suited to HPGR comminution. On this basis, a HPGR pilot test work program was undertaken, using Polysius equipment fitted with studded rolls to maximise throughput and minimise wear.

Samples

A total of 186m of whole HQ core was used to prepare a 1500kg master composite comprising 124m of Type D and 62m of Type E. Details of the master composite are summarised in Table 13.10.

Table 13.10
HPGR Pilot Test work Master Composite

Material Type	Hole ID	Metres	% Mass	U_3O_{8e} (ppm)
Type D	GOADH0062	19	10.2	378
	GOADH0063	7	3.8	496
	GOADH0064	-	-	-
	GOADH0065	37	19.9	414
	GOADH0066	61	32.8	287
Total Type D		**124**	**66.7**	**351**
Type E	GOADH0062	5	2.7	214
	GOADH0063	27	14.5	259
	GOADH0064	4	2.2	507
	GOADH0065	26	14.0	304
	GOADH0066	-	-	-
Total Type E		**62**	**33.4**	**291**
Total Composite		**186**	**100.0**	**331**

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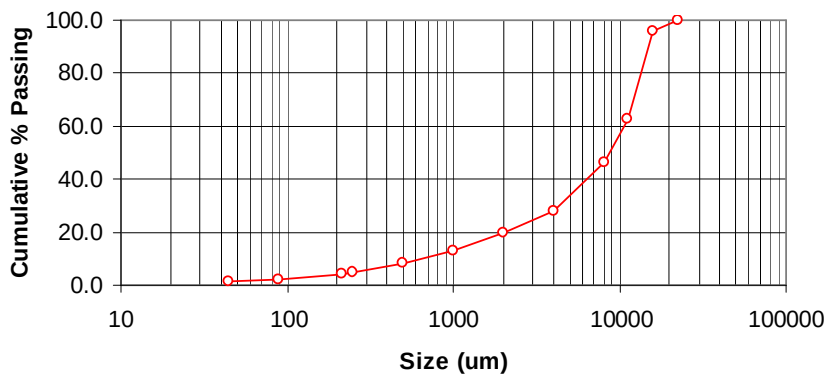
Sample Preparation

The master composite was prepared by control crushing to -22.4mm. The prepared composite particle size distribution, with an F_{80} of 13.7mm is shown in Figure 13.1 and represents the feed to the HPGR unit.

Figure 13.1
HPGR Pilot Composite Particle Size Distribution



HPGR Open Circuit Trial

Following initial pressure determination tests performed at 55bar and 75bar, a series of four open circuit trials were conducted based on the parameters tabled in Table 13.11. These tests were conducted to investigate the effect of two specific pressure settings, roll speeds and moisture levels.

Table 13.11
HPGR Open Circuit Test Parameters

Test No.	Pressure Setting (bar)	Specific Grinding Force (N/mm^2)	Roll Speed (m/s)	Moisture (%)
1	55	2.90	0.2	6.0
2	40	2.10	0.2	6.0
3	55	2.98	0.4	6.0
4	55	2.99	0.2	3.0

Visual inspection during the trials showed that while the product did tend to form a cake, the lack of clays and the particulate nature of the feed resulted in the cake readily breaking up. This indicated that issues related to cake formation and cake disagglomeration would likely not be of concern.

The specific throughput rates and specific energy achieved in open circuit testing indicated that Etango ore is amenable to comminution by HPGR. Stable specific throughput rates were maintained at an elevated moisture level of 6%, and an increase in roll speed from 0.2m/s to 0.4m/s resulted in little change in the specific throughput. Size distributions of the products are shown in Figure 13.2 from Test 4.

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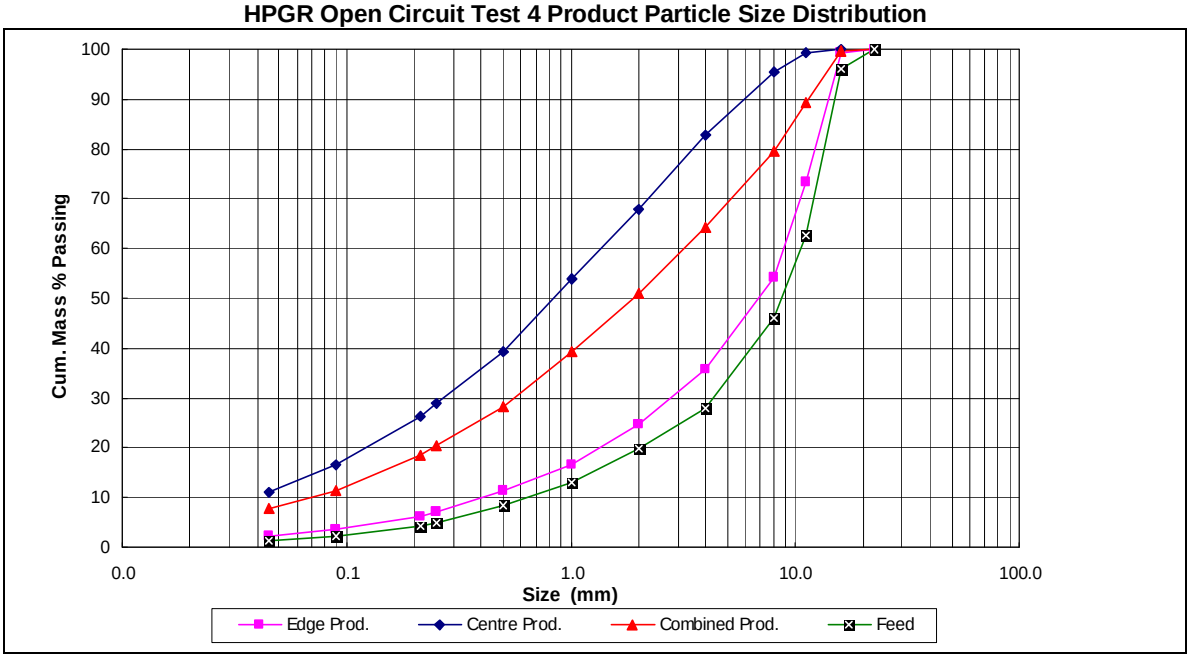
Figure 13.2
HPGR Open Circuit Test 4 Product Particle Size Distribution



The edge material is coarser than expected because it is almost identical to the feed size distribution and is contrasted strongly to the much finer centre product. This means that the combined product size distribution generated in this pilot test unit will be coarser than the product from wider rolls units where the edge material will represent a smaller proportion of the combined product.

HPGR Closed Circuit Trial

Heap leach investigations were performed on closed circuit HPGR product. The target of the HPGR preparation was a P_{80} size of 4mm, which was chosen as the optimal crush product from the initial heap leach investigations. The detailed results of the closed circuit HPGR preparation are summarised in Table 13.12.

Table 13.12
HPGR Closed Circuit Pilot Test Data

Parameter	Test 1	Test 2	Test 3	Test 4
Roll Diameter (m)	0.5	0.5	0.5	0.5
Roll Length (m)	0.3	0.3	0.3	0.3
Roll Speed (m/s)	0.2	0.2	0.2	0.2
Moisture (%)	3.0	3.0	3.0	3.0
Specific Grinding Force (N/mm^2)	2.55	2.48	2.48	2.54
Operating Gap (including zero gap) (mm)	13.5	13.2	13.2	13.0
Specific Dry Throughput (ts/hm^3)	235.1	229.2	224.2	228.8
Net Specific Energy (kWh/t)	1.13	1.14	1.12	1.14
Specific Power (kWs/m^3)	265	261	250	262
Centre Product (% Mass)	60.0	60.4	60.0	60.0
Edge Product (% Mass)	40.0	39.6	40.0	40.0
-8mm in HPGR Discharge (% Mass)	77.1	74.4	74.1	74.6

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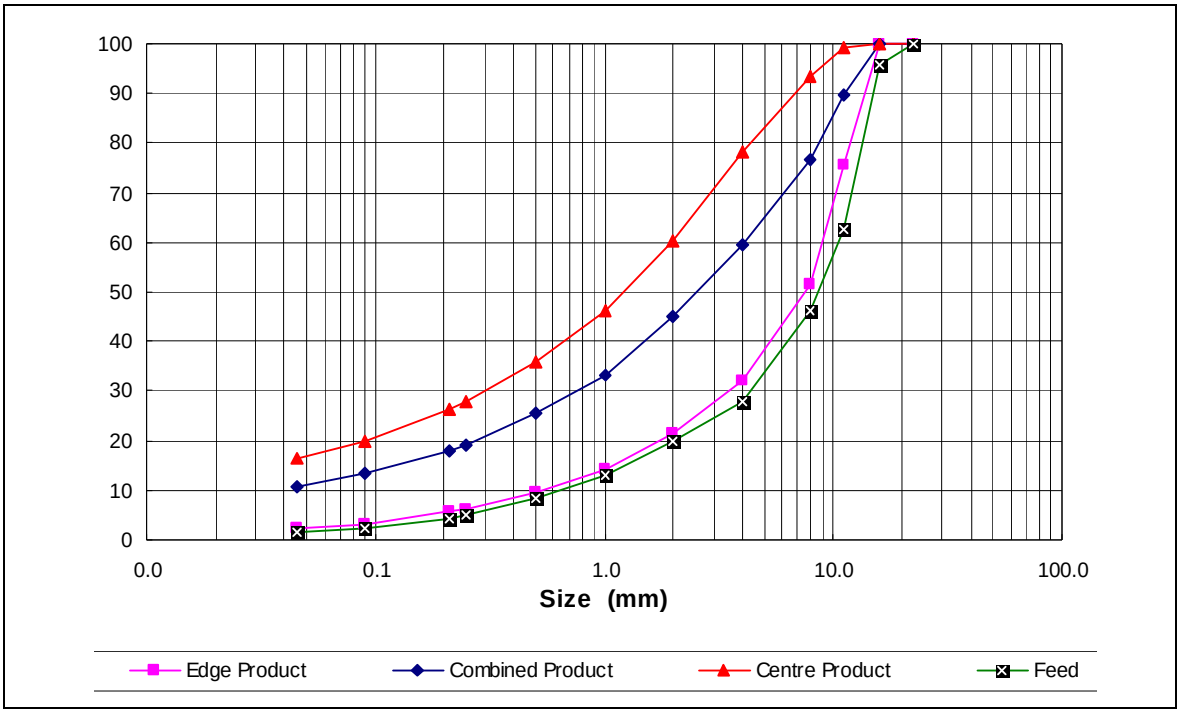
Table 13.12
HPGR Closed Circuit Pilot Test Data

Parameter	Test 1	Test 2	Test 3	Test 4
-8mm in HPGR Edge Product (% Mass)	53.7	48.8	47.6	50.0
-8mm in HPGR Centre Product (% Mass)	92.7	91.2	91.9	91.1

Figure 13.3
HPGR Closed Circuit Product Particle Size Distributions



Closed circuit testing was conducted by screening at 8mm and recycling oversize product at the end of each cycle. Stable conditions were achieved after four test cycles with a roll speed of 0.2m/s and specific pressing force of 2.54N/mm^2, resulting in a final closed circuit centre product P_{80} of 4mm at a specific throughput rate of 229ts/hm^3 and specific energy of 1.14kWh/t.

HPGR product size distributions for roll discharge streams generated at the fourth test cycle are shown in Figure 13.4

Figure 13.4. For the test roll the centre product represents 60% of the roll discharge and the other 40% (20% per end) is edge material. The combined 8mm screen undersize generated during the test (not shown) had a P80 of 4.2mm.

For the test roll the centre product represents 60% of the roll discharge and the other 40% (20% per end) is edge material. The combined 8mm screen undersize generated during the test (not shown) had a P_{80} of 4.2mm.

The size distribution differential between edge and centre product in the locked cycle work is similar to that seen in the open circuit trials (Figure 13.2). Another feature of note is that topsize in the centre zone has effectively been eliminated while the edge product contains particles as coarse as 16mm (24% +12mm). The centre product topsize is effectively the

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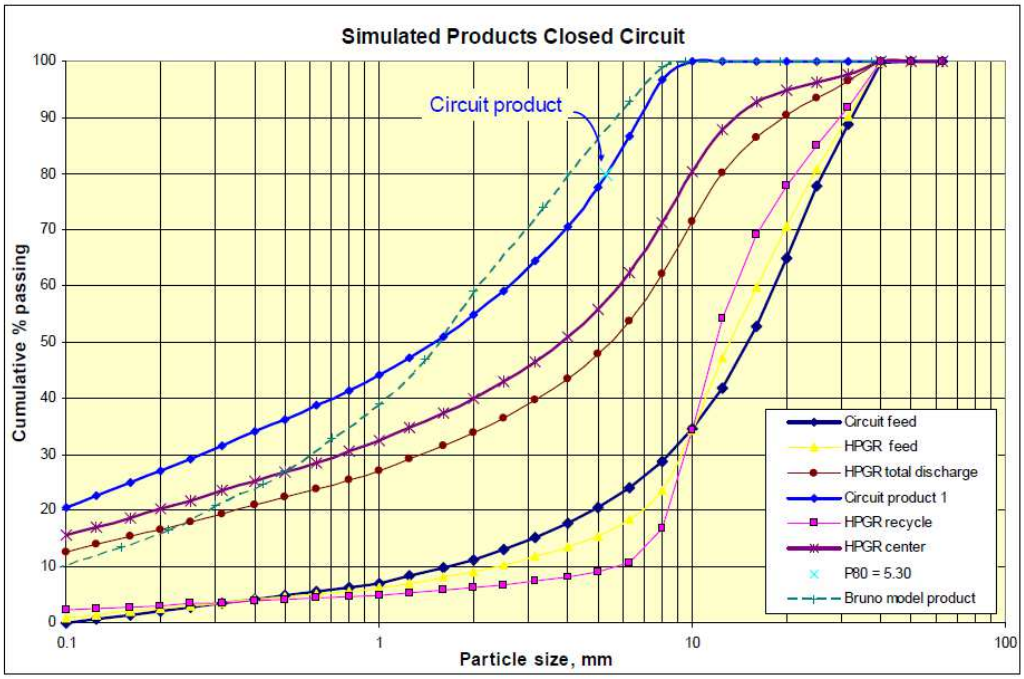
same as the measured operating gap at 13mm and this is expected. These larger particles in the edge material suggest that particles significantly larger than the operating gap were able to report to product and this is not expected.

The mass split between centre and edge product more typical of a full size HPGR is 80:20. The calculated 8mm screen undersize product from such a roll would have a P_{80} of 3.6mm.

13.4.4 Product size distribution simulations

Using these locked cycle results Polysius were able to conduct simulations representing the performance of full sized HPGR units. Polysius conducted a number of simulations at different closing screen sizes. The simulation conducted with 10mm closing screens is shown in Figure 13.4

Figure 13.4
Polysius Simulated Size Distributions with full sized HPGR and 10 mm closing Screen



The predicted circuit product with 10mm screens has a P_{80} of 5.3mm. The size distributions simulated with an 8mm closing screen are shown in Figure 13.5.

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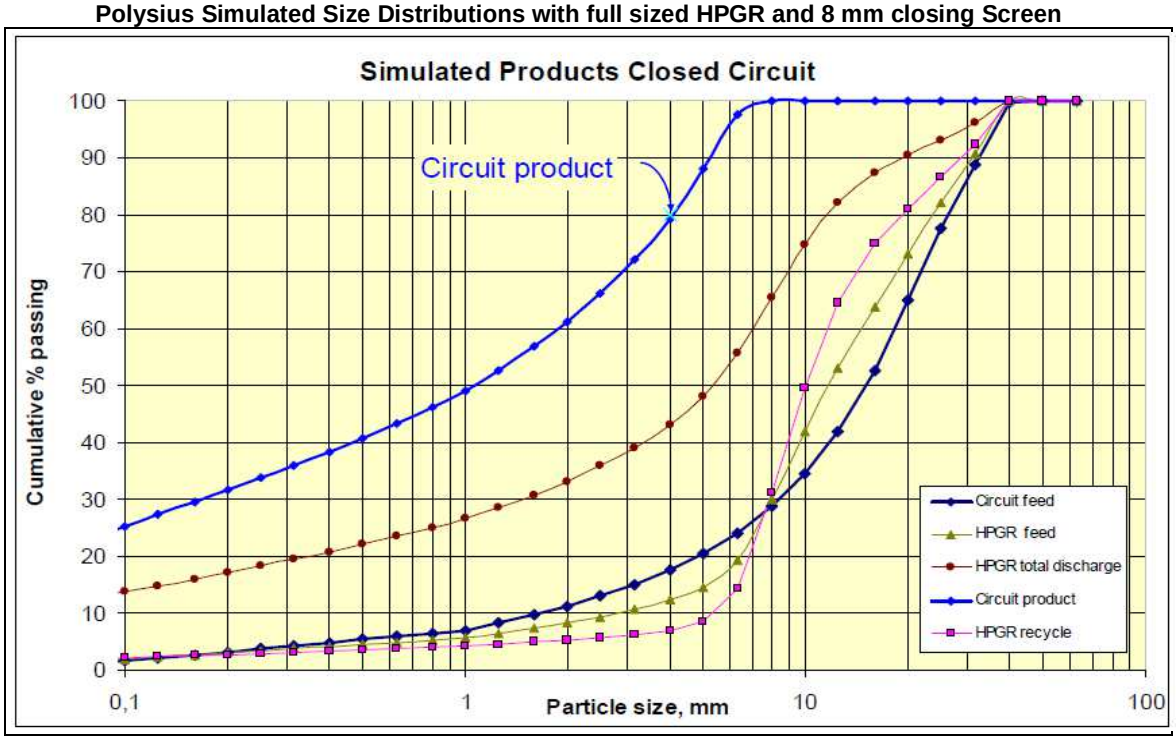
Figure 13.5
Polysius Simulated Size Distributions with full sized HPGR and 8 mm closing Screen



The predicted circuit product with 8 mm screens has a P80 of 4.0 mm.

13.4.5 Comminution – Conclusions

Comminution characterisation demonstrated that Etango ore is amenable to conventional crushing and HPGR comminution. The ore displays a generally low to moderate competency, low crushing power requirements and moderate abrasiveness. A low level of variability in comminution behaviour was evident in the core tested and the comminution properties of the two predominant ore types were shown to be similar.

Pilot scale HPGR testing proved the amenability of the ore to this processing pathway and has allowed simulation of full scale HPGR circuit performance.

13.5 PRE-CONCENTRATION TESTING

A number of pre-concentration options have been tested and reported previously (*Bannerman, September 2011*). These included scrubbing and screening, flotation, heavy media separation and gravity beneficiation of fines with either a Knelson or Falcon concentrator.

The overall conclusion from this work is that none of the beneficiation options are suitable where heap leaching is the preferred method of downstream uranium leaching.

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13.6 HEAP LEACH TESTING

Column leach testing of Etango ores has been on-going since early 2009, demonstrating uranium extractions in excess of 90% on HPGR-prepared ore, under the following conditions:

- HPGR preparation of ore to 100% less than 8mm (P_{80} ~ 4.3mm)
- Agglomeration with the following chemicals:
  - 6kg/t of H_2SO_4
  - 250g/t of Magnafloc 351
  - Sufficient water to achieve a maximum of 12% moisture in the agglomerates.
- Irrigation rates of 15L/m^2/hr
- Acid addition sufficient to maintain free acid in column discharge of greater than 8g/L H_2SO_4
- Sufficient ferric iron available to ensure that U^{4+} can be oxidised to U^{6+}.

The early heap leach test work was reported in detail in *Bannerman (September 2011)*, and these results are presented with the latest data as the discussion requires.

13.6.1 Extended Heap Leach Test work Program

A number of additional heap leach programs have been completed at ALS Ammtec. The scope of the column testing programs was broad in order to calculate the likely extractions achievable on an operating heap, and also to test a number of operational variables.

The following two programs were initiated at ALS Ammtec using sub-samples of available composite ore that was crushed to 100% - 8mm via HPGR.

Table 13.13
ALS Ammtec Program Number 12889 – Initiated in August 2010

Column Identifier	Objective
Column A (7m)	Leach profile of 7m column – Open Circuit with H_2O_2 as an oxidant
Column B (4m)	First stage of a two stage leach – Open Circuit with H_2O_2 as an oxidant
Column C (4m)	Second stage of a two stage leach – Open Circuit irrigated with ILS from Column B after re-oxidising with H_2O_2 and re-acidifying to 20g/L with H_2SO_4
Column D (2m)	Closed Circuit with pyrolusite as an oxidant
Column E (2m)	Closed Circuit with pyrolusite as an oxidant – reproducibility of Column D
Column F (2m)	Closed Circuit with H_2O_2 as an oxidant. Irrigated with PLS treated with Alamine 336

Table 13.14
ALS Ammtec Program Number 13313 – Initiated in May 2011

Column Identifier	Objective
13313 Column A (2m)	6kg/t acid in Agglomeration, 20g/L acid in irrigation liquor
13313 Column B (2m)	3kg/t acid in Agglomeration, 20g/L acid in irrigation liquor
13313 Column C (2m)	0kg/t acid in Agglomeration, 20g/L acid in irrigation liquor
13313 Column D (2m)	6kg/t acid in Agglomeration, 20g/L acid in irrigation liquor – >212µm mass

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Table 13.14 ALS Ammtec Program Number 13313 – Initiated in May 2011	
Column Identifier	**Objective**
	removed
13313 Column E (2m)	6kg/t acid in Agglomeration, 20g/L acid in irrigation liquor – 100% >3.35mm
13313 Column F (2m)	0kg/t acid in Agglomeration, 20g/L acid in irrigation liquor – Repeat of Column C

The 12889 test work program was designed to assess the following:

- A 7m column to assess the leaching performance of a column built to the maximum height considered by the project
- Generate sample for geotechnical test work to assess:
  - The permeability/percolation of freshly agglomerated ore under load equivalent to a 7m height
  - The permeability/percolation of spent ore under load equivalent to a 7m height
  - The competency and stability of a 40m heap constructed with spent ore.
- Two x 4m columns in series to assess the effect of a two-stage leaching configuration on initial leaching kinetics
- Duplicate 2m columns (closed circuit) testing the effect of pyrolusite as the oxidant and the reliability of column test methods – two columns testing the same conditions
- Control 2m column (closed circuit) to re-establish the baseline performance of a 2m column using standard conditions derived from the earlier programs
- Bench scale agitated leach test on the same sample tested in the column. This was to provide a direct comparison to the leaching performance and analytical methods used for assessing columns tests and agitated leach tests.

The 13313 program was then designed to assess:

- The effect of agglomeration acid
- The effect of changes to the particle size distribution.

The program of column test work at Bureau Veritas in Swakopmund included two separate runs which are described generally as follows:

Run No. 1

Practise tests where the laboratory can learn the operation and measurements required for a column test.

No usable results were obtained from Run No. 1.

Run No. 2

Four x 2m columns designed to test variations on the following standard conditions:

- Open circuit

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- 250kg/t of Magnafloc 351 binder in agglomeration stage
- 6kg/t of acid in agglomeration stage
- 11g/L of free acid in irrigation liquor – designed to maintain discharge free acid of greater than 8g/L
- 15L/m^2/hr irrigation rate
- delayed the addition of oxidant and ferrous sulphate to investigate whether maximum extraction could be achieved without additional ferric
- Ore sample was a sub-sample of the ore that was previously tested at Ammtec (Programs 12889 and 13313).

The following variables were investigated:

- Increased acid in agglomeration
- Decreased concentration of acid in irrigation liquor (11g/L)
- Liquor recycle.

13.6.2 Operational Performance Evaluation Tests.

Estimation of Operational Heap Performance from Column Tests

The variability test work at Bureau Veritas provided important column test work data from a range of samples made up to represent the first 3 years of operation. However, at this time, there are a limited number of column tests that represent realistic conditions for the operating heap and can therefore be used to estimate the operational performance. These tests are presented in Table 13.15, and were undertaken on sub-samples of the same composite under standard conditions, without recycling of liquor.

Table 13.15
Column Tests Relevant to Operational Heap Performance

Test Program	Column ID	Column Height (meters)	FA in Irrig. Liq. (g/L)	Excess Fe^{3+}	Comments
A12151	MH8366	2	10	Yes	Shorter than design, low FA
A12151	MH8360	2	20	Yes	Shorter than design, high FA
A12889	Column A	7	20	Yes	Taller than design, high FA
A12889	Column B	4	20	Yes	Shorter than design, high FA
A12889	Column D	2	20	Yes	Shorter than design, high FA
BV Run #2	Column 1	2	11	No	Shorter than design, low FA

The uranium extraction curves for each of these tests are presented in Figure 13.6.

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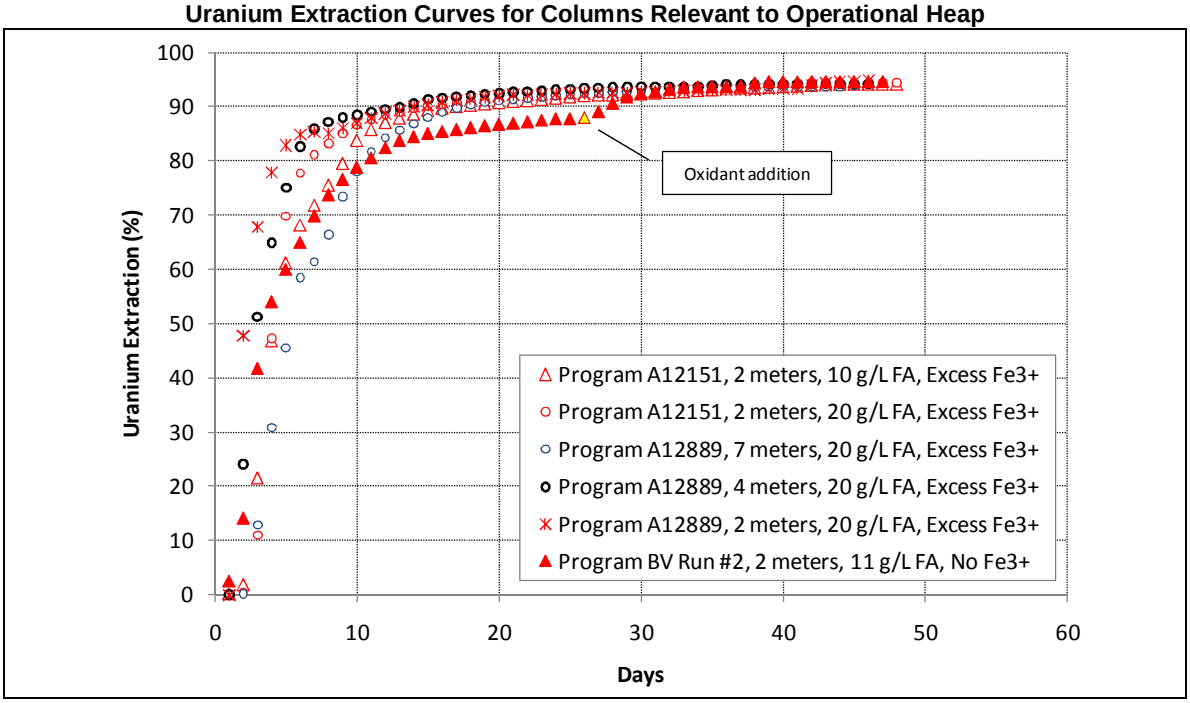
Figure 13.6
Uranium Extraction Curves for Columns Relevant to Operational Heap



From examination of the recovery curves, the following conclusions can be drawn:

- Ferric ions are required to realise maximum uranium extraction and rate of extraction
- Column tests can achieve greater than 93% uranium extraction under these conditions
- The tallest column (7m), which is expected to demonstrate the slowest rate of extraction, realised 90% extraction by Day 18 and consumed 14kg/t of acid.

The acid efficiency curves for the same tests are presented in Figure 13.7. This shows that the quantity of acid required to extract a comparable amount of uranium is relatively insensitive to the height of the column and irrigation liquor free acidity. To make a simpler comparison between the test results, Table 13.16 presents the associated leach time and calculated acid consumption for specific uranium recovery points.

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Figure 13.7
Acid Efficiency Curves for Columns Relevant to Operational Heap

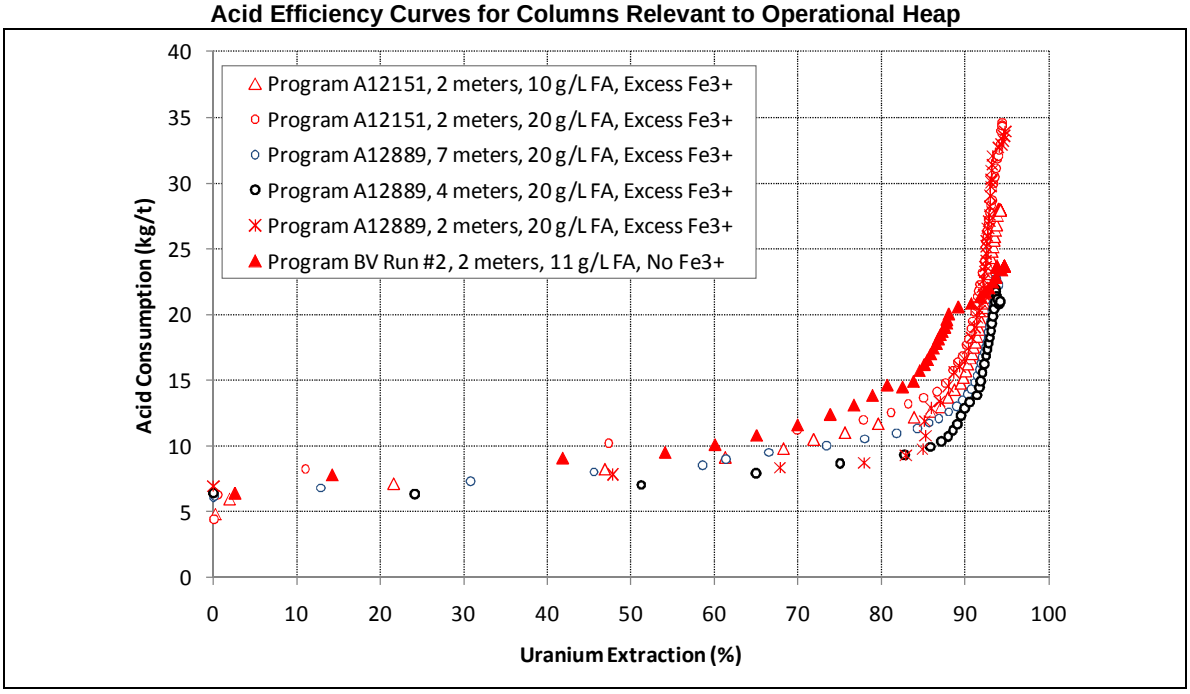

Table 13.16
Key Uranium Extraction Points from Column Tests Relevant to Operational Heap Performance

Program	Column ID	Height (m)	90% U Extraction Point		92% U Extraction Point	
			Leach Time (days)	Acid Cons. (kg/t)	Leach Time (days)	Acid cons. (kg/t)
A12151	MH8366	2	17	15.7	26	20.3
A12151	MH8360	2	15	17.7	24	23.2
A12889	Column A	7	18	13.9	24	16.7
A12889	Column B	4	14	13.3	18	15.6
A12889	Column D	2	15	17.4	21	21.5
BV Run No. 2	Column 1	2	28	20.8	30	21.6

Table 13.16 clearly demonstrates that the absence of ferric in the initial stages (BV Run No. 2 Column 1) results in a longer leach time to achieve an acceptable extraction (90 or 92%), and, consequently, the acid consumption is also higher. Disregarding the test where ferric addition was delayed, the average acid consumption required to achieve 90% uranium extraction on this sample is 15.6kg/t.

It is also notable that the 7m column achieved 90% extraction in 18 days, and 92% extraction in 24 days.

Comparable Agitated Leach Test

To date, significant variability work has been conducted on Etango ore using the agitated leaching system, so an effort has been made to link the agitated leach tests with the expected heap leaching performance by completing comparable tests on the same sample

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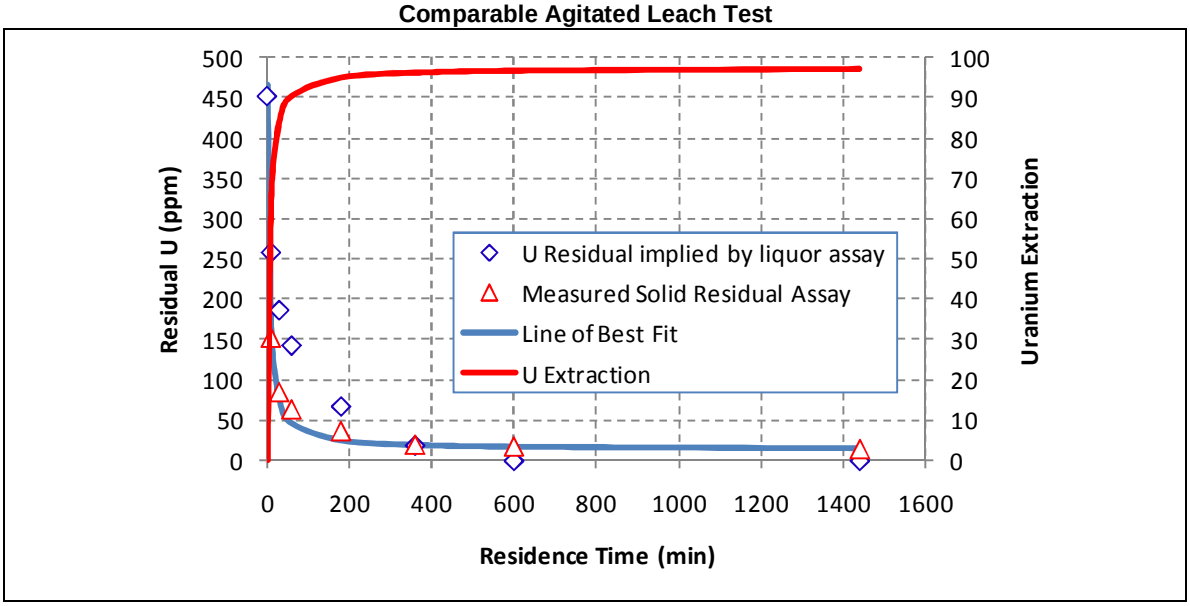
as the column tests that were reported in Table 13.15. Standard conditions for the agitated leach test are:

- Primary Grind P_{80}: 700µm
- Temperature: Maintained at: 45ºC – ambient temperature
- Water: Distilled water
- Solids density – 50% (w/w)
- Free Acid: Controlled to 5g/L throughout the leach test
- Oxidant addition as milled Pyrolusite maintaining +500mV (std calomel)
- Ferrous sulphate addition maintaining a minimum 500ppm ferric.

Figure 13.8
Comparable Agitated Leach Test



The agitated leach results presented in Figure 13.8 demonstrate marginally higher uranium extractions, while kinetics are significantly faster, as expected. Table 13.17 compares the performance of the agitated leach with the 7m column test on the same ore sample, chosen as it is theoretically the most inefficient system because of its height, and represents the extreme for leaching time.

Table 13.17
Comparison of Agitated Leach and Column Leach Tests

Test	Column ID	90% U Extraction Point		Final Ultimate Extraction		
		Leach Time (Days)	Acid cons. (kg/t)	Leach Time	Uranium (%)	Acid cons. (kg/t)
Agitated Leach	MH8597			24 hours	94.4	13.1
7m Column	Column A	18	13.9	36 days#	94.0	22.2

#Leach time refers to days under leaching conditions. Ultimate extraction estimates include an additional 12 days for washing and draining the column.

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Table 13.17 shows that the column test can achieve comparable extractions to the agitated leach test, given enough leaching time and with the consumption of more reagents.

After 18 days in the 7m column, the acid consumption for the 7m column was comparable to the overall consumption from the agitated leach, and the extraction was only decreased by 4.4%.

Figure 13.9 presents the comparative leach test into context against the other agitated leach variability tests.

Figure 13.9
Comparison of Comparative Leach with Variability Program Uranium Extractions

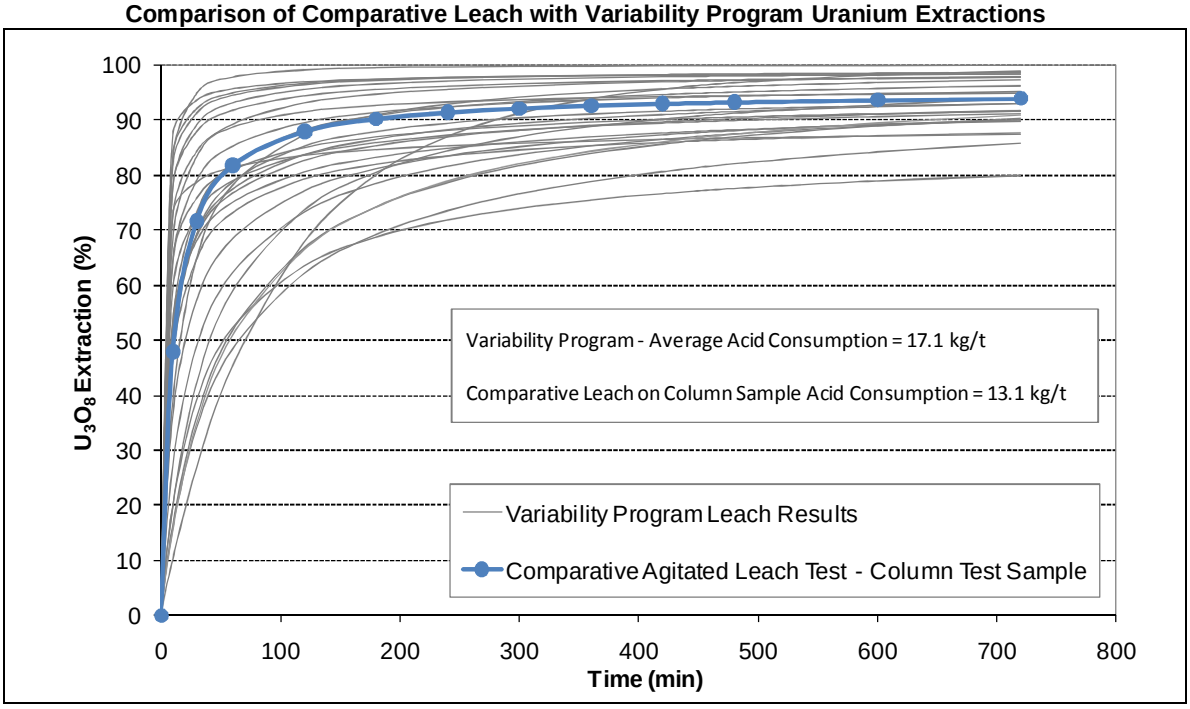

Figure 13.9 suggests that the comparative leach exhibits average uranium leach kinetics and overall extractions in comparison to the full suite of variability tests, however its acid consumption is lower than average.

In the absence of variability data from column tests to estimate operational performance, it is feasible to use the acid consumption estimates from the agitated leach program and discount the uranium extraction estimates by 4-5%.

<u>Conclusions from Operational Heap Performance Tests</u>

The conclusions from the tests designed to estimate operational heap performance are:

- Comparison of the agitated leach result suggests that the sample used for column testing exhibits average uranium extraction rate and recoveries
- With appropriate preparation (crushing and agglomeration) the sample tested consistently achieved greater than 90% extraction. This was achieved for a range of

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free acid concentrations in the irrigation liquor, however it is significant that the free acidity of the discharge liquor was consistently greater than 8g/L

- 90% extraction of uranium was realised in a 7m column within 18 days of leaching
- Ferric ions are required to oxidise U^{4+} to U^{6+} and thereby maximise the rate and extraction of uranium. In an open circuit system, this requires the manual addition of ferric (via ferrous sulphate and an oxidant), however a closed circuit may generate enough ferric such that additional reagents are not required.

13.6.3 Diagnostic Column Tests

In addition to the column tests used to estimate operational performance, a number of sub-programs were executed to investigate the 'trigger points' of the system. Specifically, the programs enabled the following assessments:

- The reproducibility of the test procedure
- The effect of column height on metallurgical performance
- The effect of different oxidants for ferric oxidation
- The effect of liquor recirculation
- The effect of particle size
- The effect of agglomeration acid.

Unless modified to investigate a specific effect, the standard conditions for the column tests were:

- Sample agglomerated using water, 6kg/t of H_2SO_4 and 250g/t of Magnafloc 351. The target agglomeration moisture was 12% (w/w).
- After agglomeration, the samples were allowed to cure for 3 days prior to loading into the column.
- Irrigation rate of 15L/m^2/hr.
- The fresh lixiviant / raffinate was re-acidified to 20g/L.
- Oxidant and ferrous sulphate were added intermittently throughout the program.

Reproducibility

A pair of 2m columns were prepared and operated comparably in order to assess the reproducibility of column performance. The resultant acid consumption over time and acid consumption / uranium extraction relationships for both tests were closely comparable.

Effect of Different Oxidant

The effect of using pyrolusite and peroxide to maintain solution Eh was compared, demonstrating comparable rates of acid consumption. Figure 13.10 suggests that the initial rate of uranium extraction is marginally increased using pyrolusite, but the difference is negligible from Day 7 onwards and may simply be a result of experimental error.

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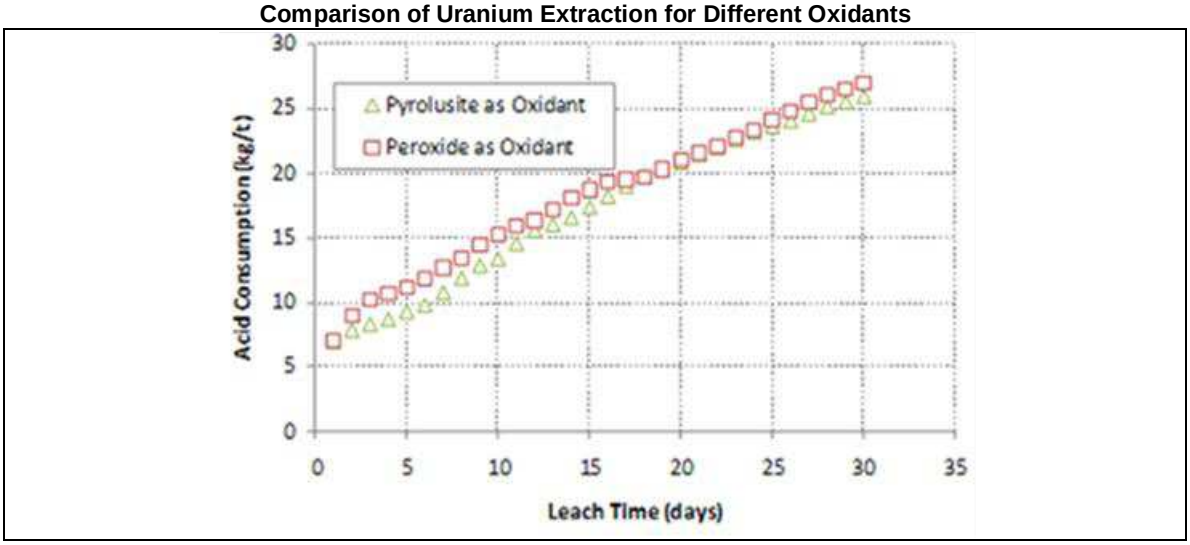
Figure 13.10
Comparison of Uranium Extraction for Different Oxidants



From this data, it is not possible to conclude that one oxidant is clearly superior to the other, and therefore the selection of oxidant should be based upon the most favourable economics of supply and delivery to the heap.

Comparison of results between a tests where oxidant was added from Day0 compared to one where it was added once the uranium extraction had reached a plateau (where other conditions were comparable) suggests that excess ferric appears to increase the rate of uranium recovery; however, the overall extraction is comparable.

Measured free acid in discharge is marginally lower for the column where no oxidant was used, however this has resulted in a significant increase in calculated acid consumption.

The increase in acid consumption has not been adequately explained to date as theory suggests that, if anything, acid consumption should be marginally increased with the addition of an oxidant, not the other way around as this data suggests.

Effect of Liquor Recirculation

The effect of re-acidified recirculating liquor on metallurgical performance was assessed compared with using fresh liquor. The resultant acid consumption over time is unaffected, and while uranium extraction is initially slowed down by using recirculated liquor, overall extraction is not affected.

Effect of Column Height

Three columns with heights of 7m (Column A), 4m (B) and 2m (C) were operated under otherwise identical conditions

Figure 13.11 and Figure 13.12 present the resultant acid consumption rate and uranium extraction rate curves for each of these tests.

As expected, the rate of uranium extraction and the rate of acid consumption (kg/t) both decreased as the column height increased. However, it is notable that the final uranium extraction achieved is very consistent across the tests. This result demonstrates that the

hydrodynamics (permeability, liquor retention) of a 7m column/heap will not limit uranium extraction.

Figure 13.11
Acid Consumption Rate for Different Column Heights

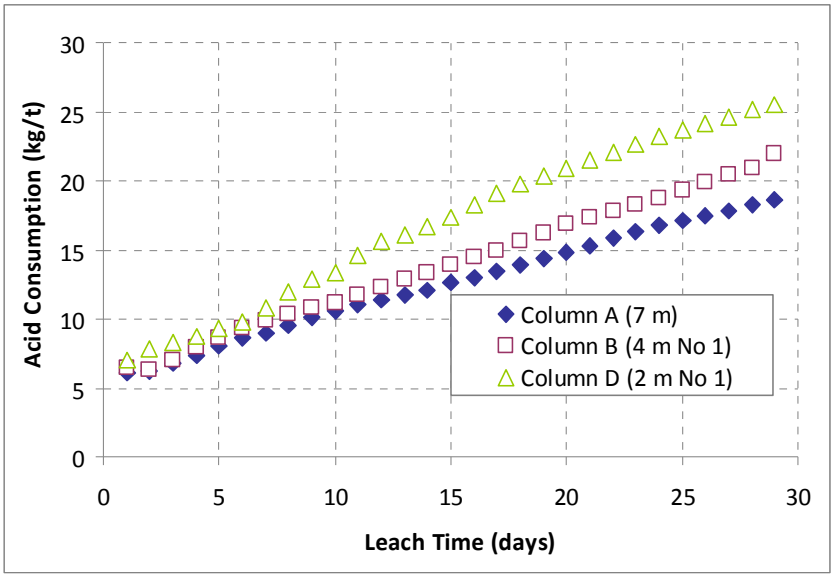
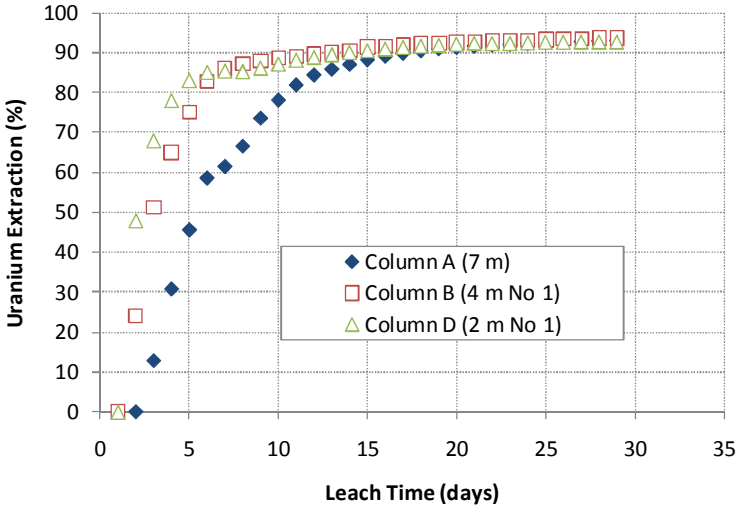


Figure 13.12
Uranium Extraction Rate for Different Column Heights



The efficiency of acid consumption is defined as the quantity of acid consumed (kg/t) in the extraction of a percentage of uranium. These curves typically show a relatively low consumption of acid for the initial fast extraction of uranium (up to ~80-85% extraction), climbing as uranium extraction slows. The acid efficiency curves for the different column heights are presented in

Figure 13.13.

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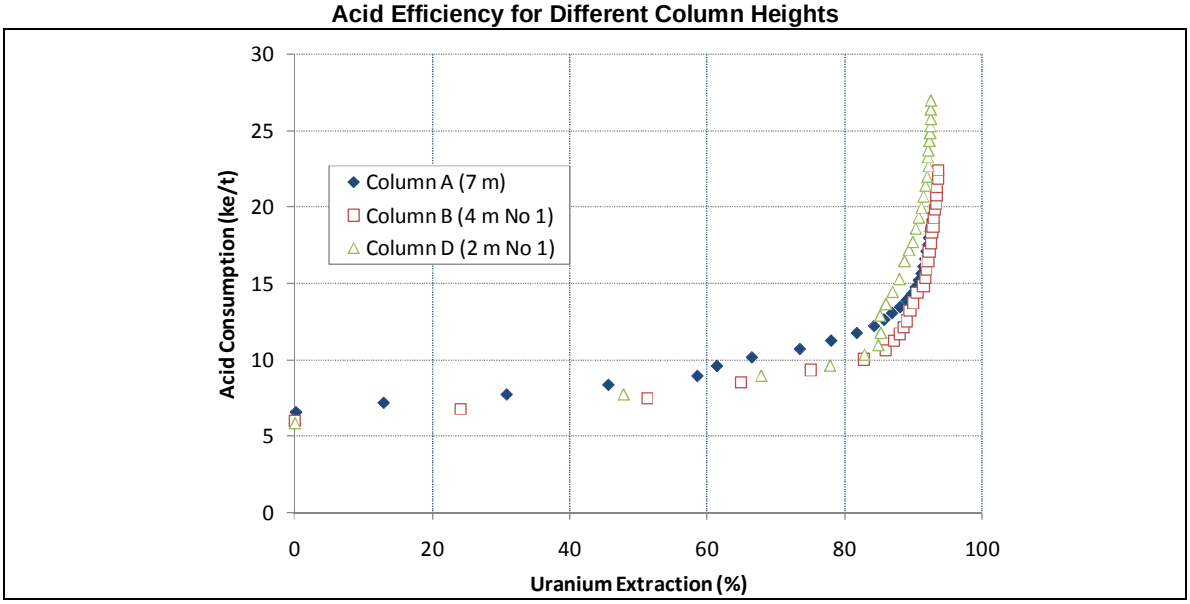
Figure 13.13
Acid Efficiency for Different Column Heights



It is notable that, within the accuracy of the measurements, the efficiency of acid consumed in the extraction of uranium is comparable for all three columns. It can, therefore, be concluded that the amount of acid required to achieve a comparable uranium extraction is independent of column height.

It should be noted that there were some notable operational differences with the resultant Free Acid measurement in the discharge.

Figure 13.14 shows that once the 7m column achieved a relatively steady state (from Day 12 onwards) the free acid in the discharge was ~8g/L. For the 2m and 4m columns, the free acid in the discharge was ~12g/L and ~16g/L, respectively, once they had reached a relatively steady state. While it cannot be measured, it can be reasonably assumed that the average free acid concentration within the column is higher for the shorter columns.

Assuming that the 7m column was exposed to a lower average free acid concentration, the overall uranium extraction was not detrimentally affected (Figure 13.12), however there may be subtle implication on the rate of uranium extraction.

It is concluded that increased free acid concentration will result in increased acid consumption, therefore there may be an opportunity to reduce acid consumption in shorter columns/heaps by using a lower concentration of free acid in the irrigating liquor yet maintaining a discharge liquor free acidity of greater than 8g/L. The results presented in

Figure 13.13 suggests that if this benefit exists, then it is likely to be only a subtle difference as the acid efficiency is comparable for the columns of different heights and different average free acid concentrations.

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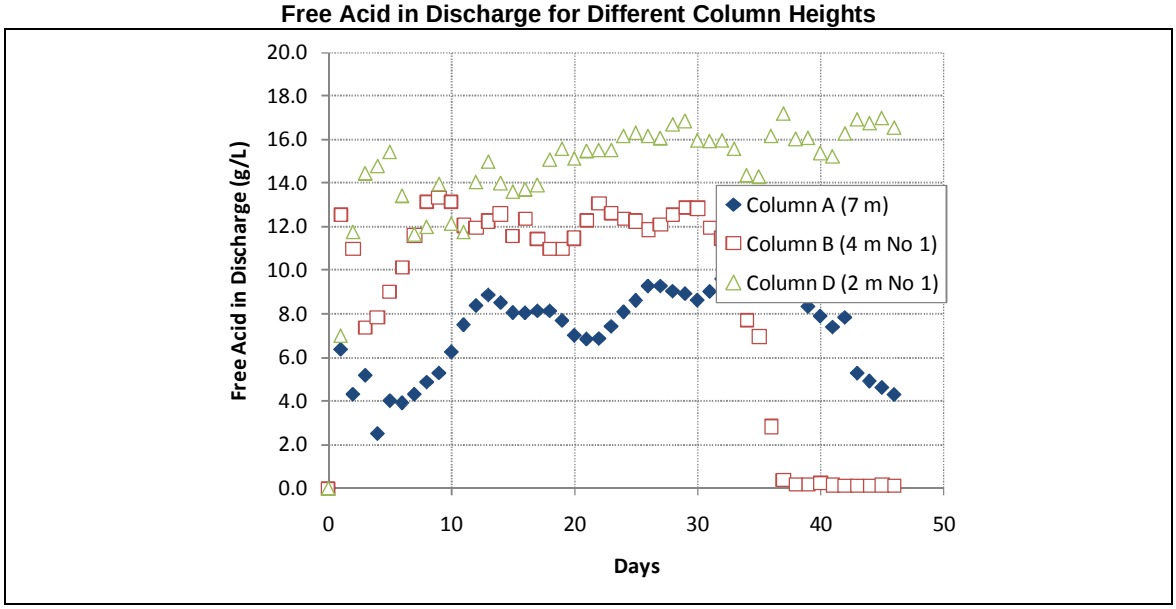
Figure 13.14

Free Acid in Discharge for Different Column Heights



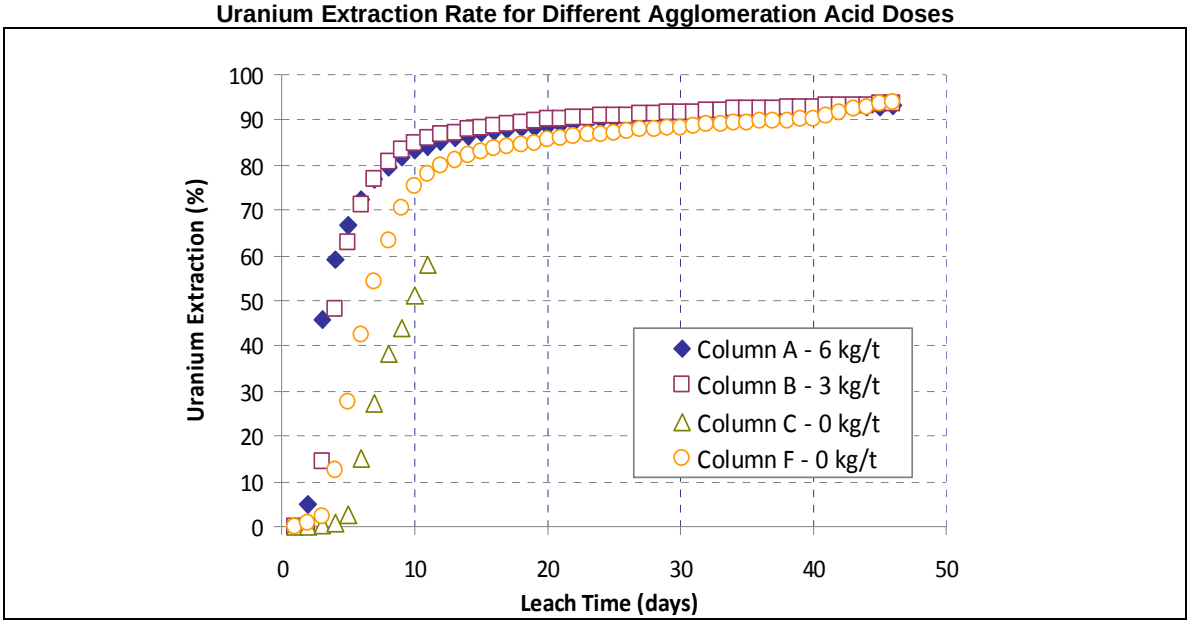
Effect of Agglomeration Acid

The effect of agglomeration acid using 6kg/t, 3kg/t and 0kg/t respectively was investigated. Unfortunately, due to operational issues, acid was not added on days 11, 16, 19, 21, 22 and 24 for Column C, and therefore this test is only comparable up until Day 10. Column F was commissioned to replace Column C as the test with 0kg/t of agglomeration acid.

Figure 13.15 and Figure 13.16 present the acid consumption rate and uranium extraction rates for the relevant tests with varying doses of agglomeration acid.

Figure 13.15

Uranium Extraction Rate for Different Agglomeration Acid Doses



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Whilst not conclusive, both the rate of acid consumption and the rate of uranium extraction trends slower as agglomeration acid is decreased. Although the rate of uranium extraction is different, all uranium extraction curves are converging at comparable overall uranium extractions greater than 90%.

Because both the rate of acid consumption and the rate of uranium extraction are changing, the acid efficiency chart is a more demonstrative comparison of performance (

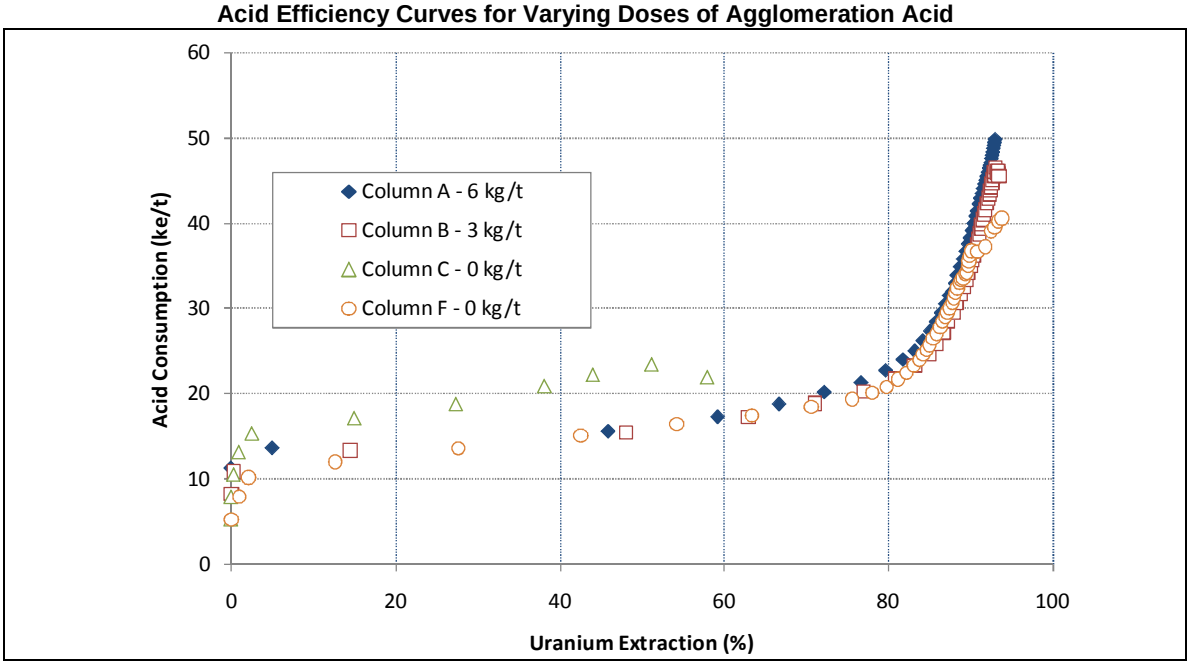
Figure 13.16).

This shows that, apart from the aborted test, the acid efficiency curves are comparable of all doses of agglomeration acid. The implications of these curves are:

- Even though decreasing the agglomeration acid may decrease the average daily consumption of acid, it will not improve the relationship between acid consumption and uranium extraction
- Varying the quantity of agglomeration acid does not affect overall uranium extraction, provided that sufficient acid is added during the irrigation phase
- Increased acid in agglomeration may decrease the amount of irrigation time required to achieve the target uranium extraction
- Using 6kg/t of agglomeration acid will not increase the cost of acid, but may decrease the cycle time on the leaching pad compared to lower doses of agglomeration acid.

Figure 13.16

Acid Efficiency Curves for Varying Doses of Agglomeration Acid



Effect of Particle Size Distribution

Two tests were undertaken to assess the effect of particle size in the heap leach environment. In one of these (Column D) the sample was screened at 212µm to remove fine particles, while in the second (Column E) +3.35mm particles were re-crushed so that the

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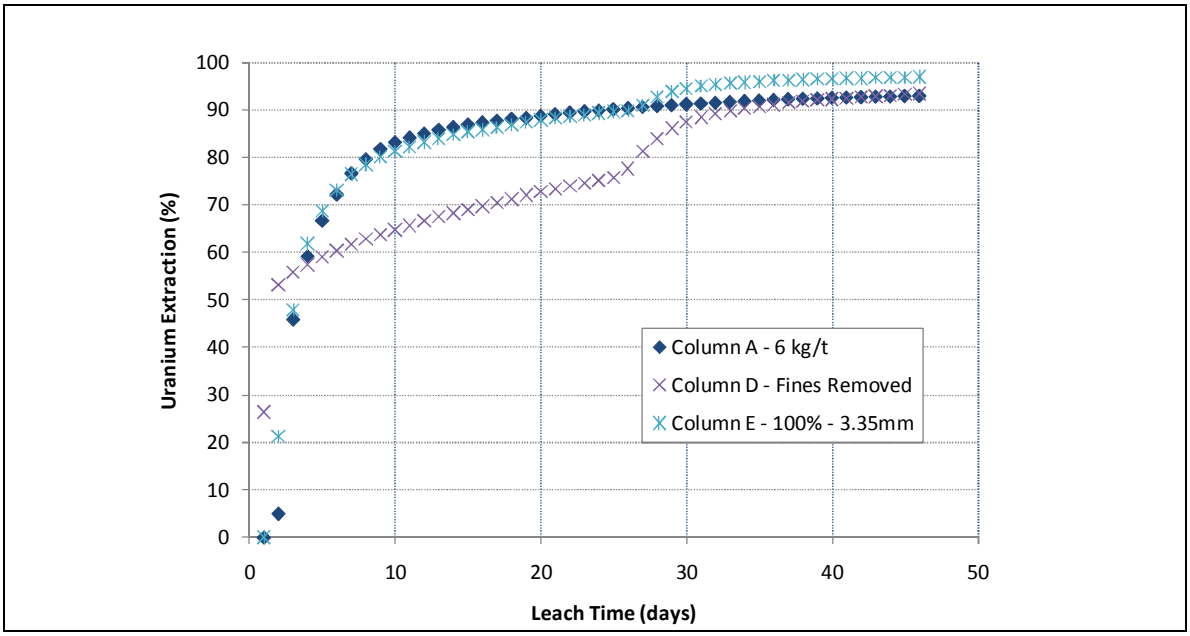
column feed was 100% -3.35mm. Results were compared with a third column (Column A) in which the feed size had not been modified.

The test data did not confirm the expectation that higher acid consumption accompanies increasing fines content. The sample with fines removed (Column D) consumed the least amount of acid. However, Column E was expected to have the highest rate of acid consumption, but, unexpectedly, Column A displayed the highest acid consumption despite having a comparable quantity of -212µm particles to Column E and significantly more coarse particles.

This result has raised the question over sample representivity between these tests. Before drawing any conclusions on acid consumption, the effect of experimental and analytical error must be duly considered.

With regards to uranium extraction (Figure 13.17), the rate of uranium extraction is the slowest for the ore with the fine particles removed (Column D), but the ultimate extraction is only marginally lower, achieving greater than 90%.

Figure 13.17
Uranium Extraction Rate for Different Particle Size Distributions



Both Column D and E achieve a significant increase in the rate of uranium extraction after Day 26 when oxidant was added to the system. This is an interesting result that suggests a condition exists where the addition of a chemical oxidant is required. At this time, this condition has not been sufficiently explained and, therefore, the operation should plan conservatively for the addition of an oxidant.

Despite the presence of less coarse particles in Column E, the overall uranium recovery and rate of uranium recovery is comparable to baseline performance (Column A) up until the point where oxidant and ferrous sulphate were added, resulting in an immediate increase in the rate of uranium extraction.

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Conclusions from Diagnostic Test work Programs

Conclusions from the diagnostic component of the column test work program are:

- The column test procedure applied at ALS Ammtec provides reproducible results
- Using conditions comparable to the heap design, the calculated uranium extractions from the column tests were consistently greater than 90%, irrespective of column height tested
- No physical limitation of liquor flow was noted in any of the column tests up to 7m in height
- Hydrogen peroxide and pyrolusite were tested as the oxidant and no significant difference in leach performance was observed
- Liquor recirculation (closed circuit) slowed the initial extraction of uranium, however it did not affect the overall extraction of uranium that is achievable
- Increasing the quantity of agglomeration acid up to 6kg/t did not result in increased uranium extraction or a measurable decrease in acid efficiency. However, it did increase the rate of extraction, and, therefore, should be included in the design basis
- Although a relationship between fines and increased acid consumption is likely on a theoretical basis, the test data cannot confirm the hypothesis.

13.6.4 Geotechnical Considerations

Heap Stability

Golder Associates (Golder) undertook laboratory-scale geomechanical testing on agglomerated composite ore using the feedstock for heap leach column test work at Ammtec over the period August to November 2010.

Golder tested and reported the load-permeability and load-percolation rate relationships for the agglomerated feed ore and the final residue (bottom 1m) of the 7m tall Column A, being the two extremes in material structure (Chapman, 2010b).

The conclusions arising from the Golder study were as follows:

- The load-permeability of the feed ore indicated a marked trend of decreasing permeability up to a height of ~4m. Thereafter the permeability did not significantly reduce with additional load
- The void ratio of the agglomerated ore does not reduce significantly when subjected to loads greater that ~80kPa. This is typical of sandy materials which have low compressibility. Consistent with this behaviour, the permeability of the heap material does not significantly decrease with further load
- The results indicate that a percolation rate of 15L/m^2/hr is achievable for a 7m high heap
- The load percolation tests on 'undisturbed' heap leach residue indicated that an application rate of 160L/m^2/hr was achievable before ponding occurred
- The 'disturbed' heap leach residue test displayed ponding at an application rate of 5L/m^2/hr.

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Stability Analysis for Ripios Storage

Golder conducted geotechnical stability analysis for the proposed Ripios storage facility, using residue product from the Ammtec 7m column trial, of August-October 2010.

The test work objective was to identify an appropriate slope angle for construction. A concurrent study also considered long term heap stability following closure.

The preliminary results (Chapman 2010a) indicate:

- The stability of the heap is highly dependent on the height of the phreatic surface (water table) that may form in the heap
- Provided the phreatic level can be managed to 10% of heap height, the outer slope of the stacked Ripios can be formed at a maximum batter of 2.5H:1V (~22°). The underdrainage system should be designed to maintain phreatic level to 10% of heap height
- Slope configuration should also facilitate closure and long-term stability of the final landform
- The design should also consider the geotechnical stability of the foundation below the Ripios storage facility
- Retaining regular benches on the outer slope is not recommended as it will concentrate flow of water and lead to erosion.

13.7 SOLVENT EXTRACTION TESTING

SX testing was undertaken by ALS Ammtec in Perth, Western Australia. The purpose of the test work was to produce a quantity of typical Etango heap leach PLS, with appropriate levels of contaminants (Al, Ca, Fe, K, Mg, Mn and chloride) that might be produced from raffinate recycling to leach, and to conduct SX tests on that PLS.

The sample was part of a larger composite sample prepared in early 2010 from HPGR samples. The material received was crushed to <3.35mm and agglomerated with sulphuric acid, binder solution (dilute flocculant solution) and additional water.

Head assay of the ore sample is shown in Table 13.18.

Table 13.18
Head Assay of SX Test work Ore

Sample	U (ppm)	V (ppm)	P (ppm)	Th (ppm)	Fe (%)	Mg (%)	Al (%)	Ca (%)
PLS composite	207	13	267	81	0.8	0.4	6.60	1.10

Three medium scale (4m high by 190mm diameter) column acid leach tests were conducted, operating in closed circuit with fresh leach solution added to the first column only. The PLS was contacted with Alamine 336 (5% v/v in narrow cut kerosene) and the raffinate recycled to the column. The first column was operated for a period of 12 days, after which the raffinate was introduced into the second column. The solution was recirculated through the second column, with uranium recovered by contacting with Alamine as before, for 13 days,

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after which the raffinate was introduced into the third column. The third column was operated for only 8 days then allowed to drain for a further 2 days. The final PLS and drain solutions recovered were analysed and combined as appropriate to form a feed solution for SX testing.

Based on previous test work, the column irrigation rate chosen was 15L/m^2/hr. The initial leach feed solution (40L) was prepared from Perth tap water with 20g/L H_2SO_4 plus 500mg/L Fe^{2+} added (as ferrous sulphate). The redox of the solution was adjusted to +500mV (Ag/AgCl) using hydrogen peroxide solution before introducing into the column. Raffinate solutions were adjusted back to 20g/L H_2SO_4 before recycling back to the columns.

The as-produced PLS was analysed as shown in Table 13.19.

Table 13.19
Analysis of As-produced PLS

Sample	U (Mg/L)	Al (Mg/L)	Fe (Mg/L)	Fe^{2+} (Mg/L)	Mg (Mg/L)	P (Mg/L)	Si (Mg/L)	SO_4 (g/L)	Cl (Mg/L)	H_2SO_4 (g/L)
PLS composite	336	2,911	4,923	1,550	2,675	915	616	47.12	567	8.67

Extraction isotherm tests were conducted on the PLS using 5% v/v Alamine 336 and 2.5% v/v isodecanol in low aromatic kerosene.

The isotherms tests conducted at 20ºC and 35ºC for the as-produced PLS are shown graphically in Figure 13.18 and Figure 13.19.

Figure 13.18
Uranium Loading Isotherm
(as-produced PLS at 20ºC)

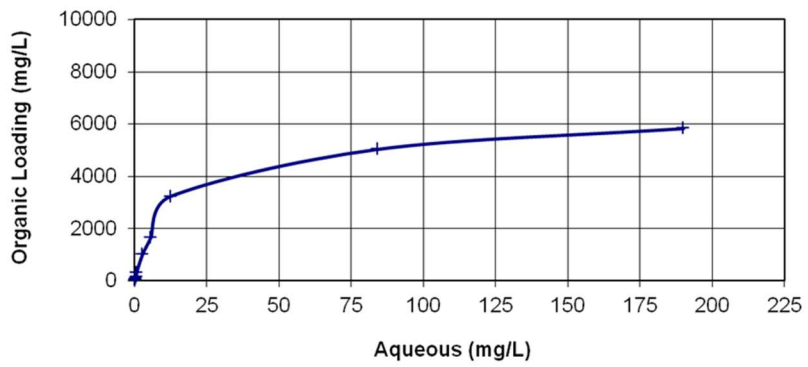

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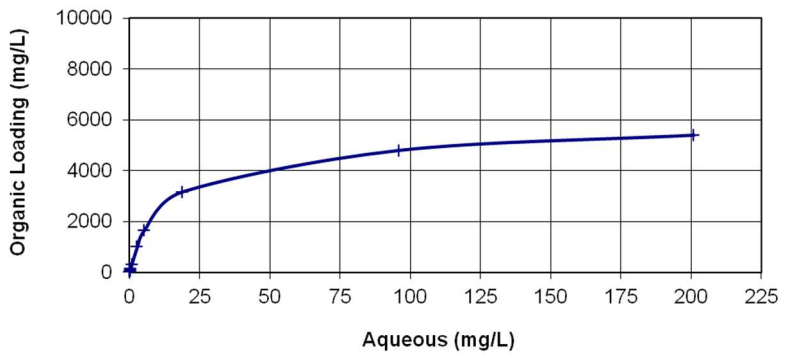
Figure 13.19
Uranium Loading Isotherm
(as-produced PLS at 35ºC)



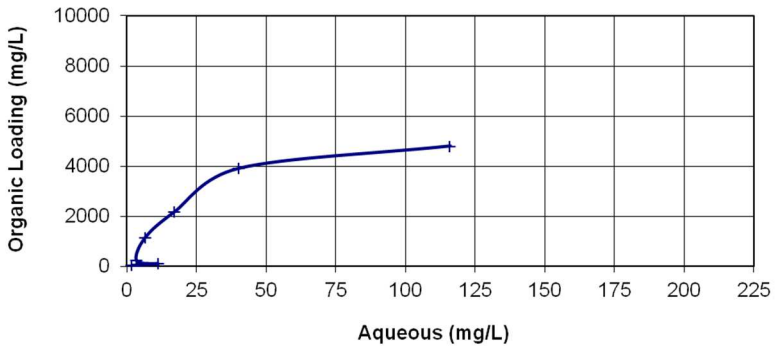
A 2L portion of the as-produced PLS was 'spiked' with salts to increase contaminants such as Al, Ca, Fe^{2+}, Fe^{3+}, K, Mg and Mn (Table 13.20).

Table 13.20
As-produced PLS 'Spiked' with Salts

Sample	U (Mg/L)	Al (Mg/L)	Fe (Mg/L)	Fe^{2+} (Mg/L)	Mg (Mg/L)	P (Mg/L)	Si (Mg/L)	SO_4 (g/L)	Cl (Mg/L)	H_2SO_4 (g/L)
Spiked PLS Composite	236	25,720	13,280	6,430	32,320	578	467	312.0	890	7.04

Results of extraction isotherm testing on the spiked PLS are shown in Figure 13.20.

Figure 13.20
Uranium Loading Isotherm
(Spiked PLS at 20ºC)



Sodium chloride (2.48g/L Cl) was also added to the spiked PLS and isotherm testing repeated (Figure 13.21).

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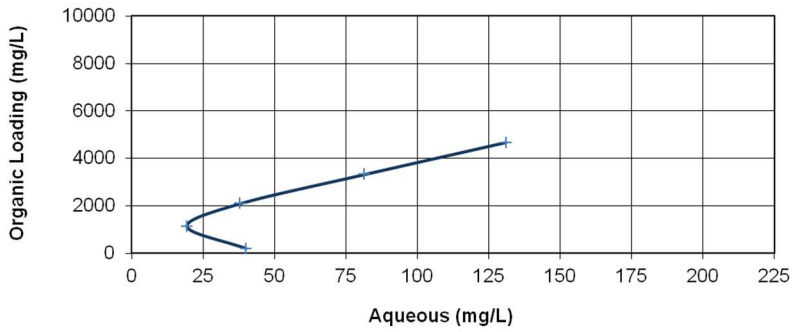
Both isotherm tests on spiked PLS produced an erroneous point. The raffinate solutions were re-analysed and found to be correct. The calculated organic loadings obtained from the difference between the PLS and raffinate analyses agreed with the organic loadings obtained from back-stripping the organic samples. The organic loadings for these two erroneous points appear to be of the correct magnitude, and the raffinate levels appear to be incorrect.

Figure 13.21
Uranium Loading Isotherm
(Spiked PLS plus Chloride at 20ºC)



13.8 MISCELLANEOUS TESTING

13.8.1 Chloride Analysis

Total and water soluble chloride analysis was performed on surface samples and at depth. Results are reported in Table 13.21 and Table 13.22. These indicate moderate levels at surface, with the concentration reducing significantly at depth.

Table 13.21
Etango Surface Ore Total and Water Soluble Chloride

Analysis	Unit	GOADH0048 (0-9m) Test 1	GOADH0048 (0-9m) Test 2
Water Soluble Chloride	mg/kg	2,023	2,026
Total Chloride	mg/kg	2,200	2,500

Table 13.22
Etango Ore Total and Water Soluble Chloride at Depth

Analysis	Unit	GOADH0048 (35-40m)	GOADH0048 (61-66m)	GOADH0048 (95-100m)
Water Soluble Chloride	mg/kg	20	25	36
Total Chloride	mg/kg	50	70	70

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13.9 HEAP LEACH DEMONSTRATION PLANT TEST WORK

This section of the report was written by Amec Foster Wheeler, based on the following test work reports prepared by Bannerman:

- Demonstration Plant, Phase 1 Report, June 2015
- Demonstration Plant, Phase 2 Performance Report, September 2015.

All test work considered in this section has been performed and managed by Bannerman personnel. Assays were completed by different laboratories while others have been performed on site by Bannerman personnel. While the content of those test work reports has been generally reviewed by Amec Foster Wheeler for inclusion into this report the results have not been fully audited or sought to be verified by Amec Foster Wheeler.

13.9.1 Introduction

On 22 September 2014 Bannerman announced the award of the major contracts to construct and operate the Etango Heap Leach Demonstration Plant. Activities at the site commenced in early October, with the completion of the construction and official opening on 24 March 2015.

Bannerman decided to undertake the demonstration plant test work in five phases with the objectives and activities for each stage as summarised in Table 13.23.

Table 13.23
Demonstration Plant Program Activities and Objectives

Phase	Objectives	Activities	Status
1 Commissioning	• Commissioning of Plant • Validate leaching parameters	• Open cycle operation of all cribs and columns. • Identify issues and correct plant and operating procedures as required	Completed
2 Reproducibility	• Demonstrate consistent operation of plant • Validate leaching assumptions in DFS	• Operate 2 cribs and 4 columns • Utilize same blended sample in both cribs	Completed
3 Solution recycle	• Simulate the heap leach pad cycle to generate Pregnant Leach Solution (PLS) • Assess the build-up of deleterious elements emanating from the recycling of solution	• Operate three cribs in closed cycle • Analyse the possible build-up of deleterious elements. • Generate and store sufficient PLS to enable the validation of SX assumptions in Phase 4	Underway
4 Solvent Extraction	• Demonstrate the solvent extraction process and assumptions in the	• Operate SX plant in laboratory in Swakopmund.	Pending

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Table 13.23
Demonstration Plant Program Activities and Objectives

Phase	Objectives	Activities	Status
	DFS		
5 Value Engineering	• Conduct optimisation studies	• Primarily utilize columns to evaluate various opportunities to improve the project economics.	Pending

The test work completed to date and reviewed in this section, comprises the first two phases of the demonstration plant test work, namely:

- Phase 1 of the program commenced mid-April 2015 and involved an open circuit leach operation of four Cribs (Cribs 1 to 4), each with two Columns running in parallel
- Phase 2 of the program commenced mid July 2015 and involved an open circuit leach operation of two Cribs (Crib 5 & 6), each with two Columns running in parallel. All Cribs and Columns were operated under the same conditions as for Phase 1. Phase 2 Cribs test work was performed in Crib 1 and Crib 2, but for convenience and to avoid confusion with Phase 1, have been labelled in this report as Crib 5 and 6 respectively.

13.9.2 Demonstration Plant Description

The flowsheet of the demonstration plant resembles the front end of the processing plant up to the heap leaching stage. Provisions have been made to install either a pulse column or mixer-settler unit to demonstrate the solvent extraction component of the flowsheet, in due course.

The plant is comprised:

- Agglomeration unit consisting of the agglomeration drum (0.725m diameter by 2.5m length), feed hopper, vibrating feeder and feed conveyor
- Luffing conveyor
- Four cribs of 2x2x6m with a capacity of approximately 30 dry tonnes of ore each, at the nominated definitive feasibility study (DFS) height of 5m
- Each crib is equipped with one dedicated irrigation (feed) positive displacement pump
- Eight, 5m high columns with an internal diameter of 0.185m and with a capacity of approximately 200kg were installed, enabling direct comparison of the leaching performance between the columns and the cribs.

The plant site viewed from the north east side is shown in Figure 13.22. The plant is self-sufficient with respect to electricity and operates on a continuous cycle. The cribs are able to be operated in open (i.e. individually) or closed loop (i.e. series) circuit.

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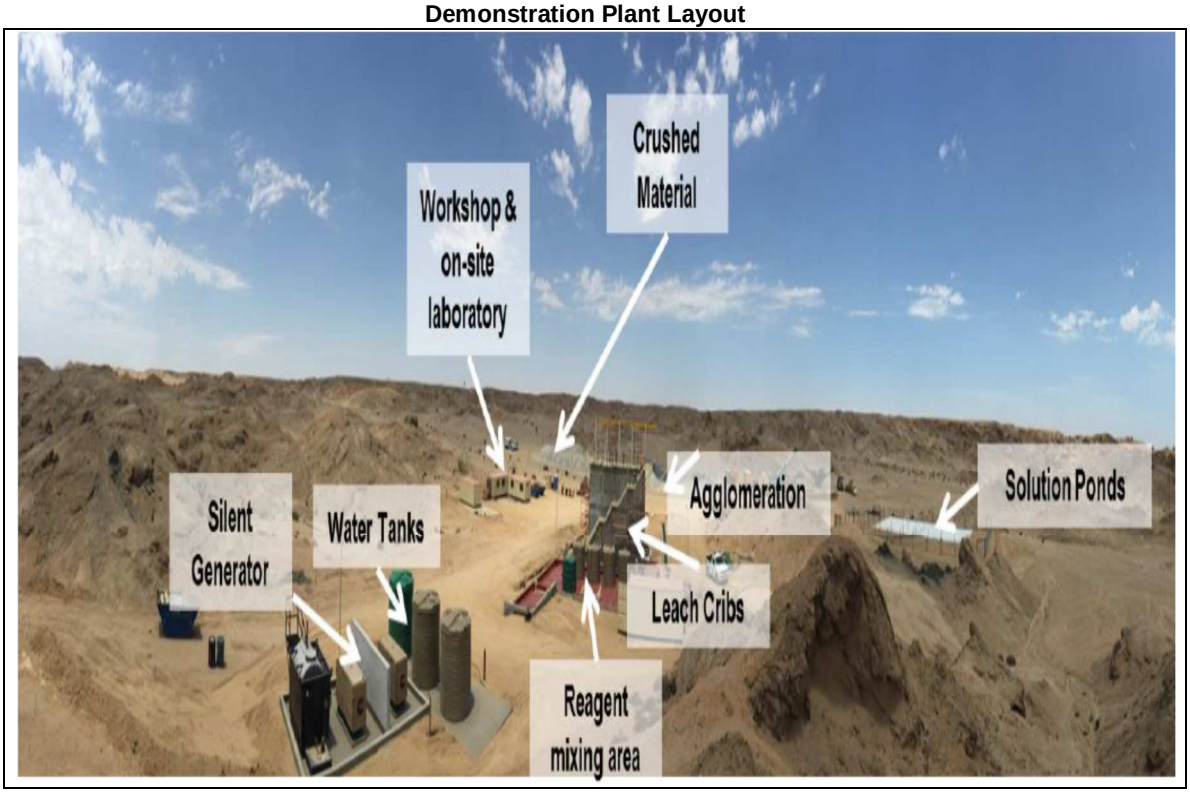
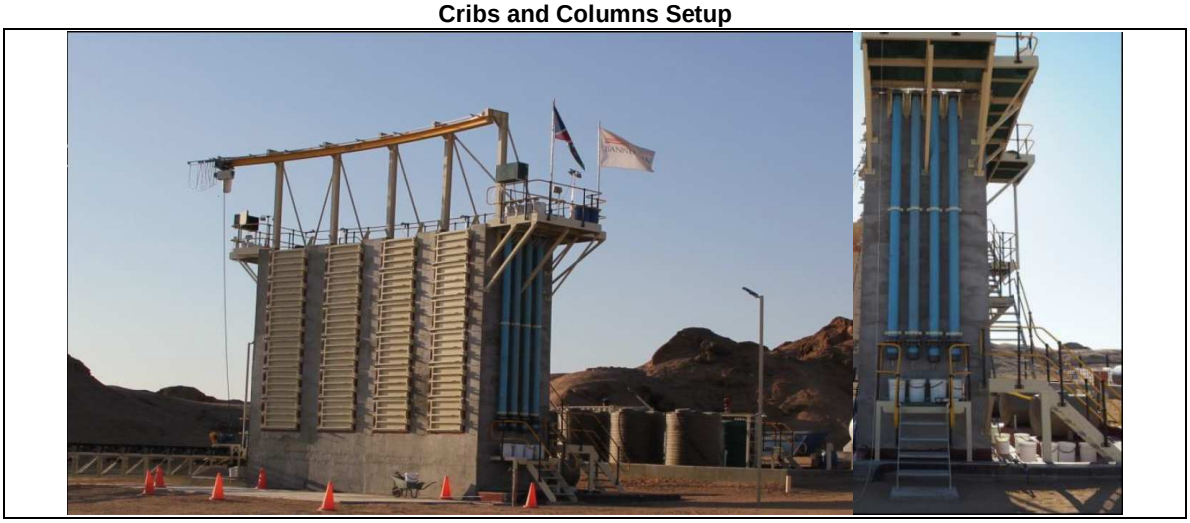
Figure 13.22
Demonstration Plant Layout



The setup of the cribs and the columns is shown in Figure 13.23. The series of gates on the front of the cribs allow for the progressive stacking from the bottom up, instead of dropping the material in from the top. The gates also permit the unloading of the ripios (leach residue) by sections allowing the determination of extraction and or moisture at different heights (e.g. by meter).

Figure 13.23
Cribs and Columns Setup



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13.9.3 Ore Sample

The bulk sample (approximately 3,000 tonnes) for the demonstration program was sourced from the northern end of the Etango orebody. This area was selected because the ore grade was representative of the orebody and the cost of the excavation was low due to the orebody outcropping in this area. The clean ore allows for the controlled blending with waste material to achieve the specified dilution.

The area was mined using a blast pattern with a burden and spacing between the holes of 2.0m and 1.8m respectively with a total of 98 blast holes drilled. Each hole was assayed providing an average grade of U_3O_8 at 202.4ppm.

Figure 13.24
Bulk sample area and Mined ore rocks

The ore sample was diluted with gneiss rock (typical waste) sourced from the planned pit area to generate a blended sample representative of the projected run of mine ore feed to the plant.

Phase 1 and Phase 2 used 10% dilution (as indicated in previous test work). A total of five blast holes from the waste block were sampled and assayed providing the average grade of U_3O_8 in the waste of 4.94 ppm.

The same waste material was used as material to form a drainage layer but with a particle size distribution of -22mm and +8mm fraction.

13.9.4 Mineralogy

Previous mineralogical studies done on extensive drill samples from across the Etango deposit ranging from 3m to 487m below surface have concluded:

- There is no apparent variation in any uranium minerals from north to south in the Etango deposit or within the D or E-type alaskites which host more than 90% of the mineralisation
- There is also no apparent difference in chemistry, mineral content, and grain size, texture and grade variation between the D and E-type alaskites.
- The variability of properties within the alaskite as a whole is low

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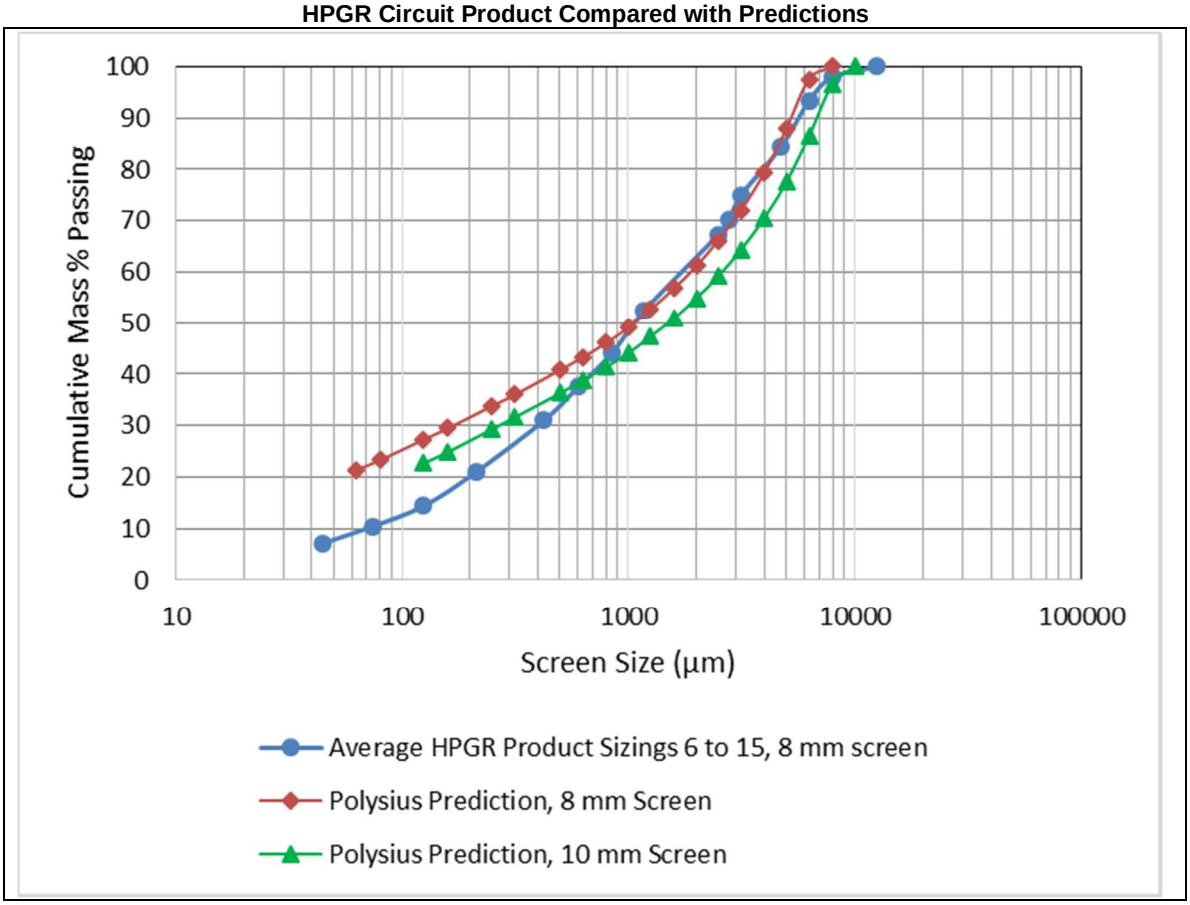
As such no further mineralogical studies were considered necessary for either the bulk ore sample or waste rock (from the Chuos formation, which makes up the bulk of the Etango waste rock) used for the Demonstration Plant test work. The bulk sample is thus considered representative of the typical mineralised alaskites at the Etango deposit.

13.9.5 Crushing and particle size

The flowsheet used to generate the crushed ore sample was similar to the one considered in the DFS design. The bulk sample was crushed using conventional primary and secondary crushing equipment (mobile crushing plant) to generate a product of P_{100}<22mm. This product was then tertiary crushed by means of a High Pressure Grinding Roll (HPGR) unit to generate the final particle size. The targeted particle size was intended to be the same as that considered in the DFS design of P_{80}=5.3mm but the use of an 8 mm screen resulted in a finer product. Approximately 2000 tonnes of ore and waste crushed in this way.

The screen chosen to achieve a 5.3 mm P80 was 8 mm and this was finer than the 10 mm screen aperture size envisaged in the DFS. Consequently the resultant PSD for the trials was consistent (except for the fines sizes) with the simulated prediction by Polysius when using an 8 mm screen as can be seen in Figure 13.25

Figure 13.25
HPGR Circuit Product Compared with Predictions



The waste material used for dilution, was crushed in a similar way and using the same crushing configuration and equipment.

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The ore particle size, as measured at the feed to agglomeration, was slightly finer than the HPGR circuit product measurements and has an average P_{80} of 3.65mm. The Individual cribs particle size P_{80} values are shown in Table 13.28.

From Figure 13.25 it can also be noted that the final crushed product was finer than DFS design assumptions for all the fractions larger than 0.9mm and coarser for the fractions below 0.9 mm. The crushed particle size determines the leaching response and is discussed further below.

In addition two samples of approximately 500 tonnes were crushed by different methods and to a different particle size. One sample was crushed by means of an HPGR but targeting a coarser particle size, achieving on average a P_{80}=4.8mm. The second sample was crushed by conventional crushing targeting a P_{80}= 5.3mm and achieving a P_{80}=4.6mm. Both of these samples will be used in planned value engineering work.

13.9.6 Bottle Roll Tests

A composite sample obtained during the crushing campaign was ground to a P_{80}=850 microns and used for bottle roll testing. The purpose was to establish the leach performance of the material to be used during the demonstration plant leach test work (cribs and columns).

Three subsamples were tested at the same conditions with the conditions set as follows.

- Sample mass: 500 grams
- Solids Density: 50%w/w
- Temperature: 45°C
- Time: 9 hours of total leaching time, with samples of solution taken every 2 hours
- Leach condition: Constant pH of 1.65 and redox potential >475mV (Ag/AgCl).

The results of these test are summarised in Table 13.24. These results provide a base in terms of maximum extraction that could be expected for the Cribs and columns.

Table 13.24
Bottle roll results

Sample	Head Grade, U_3O_8	Residue Grade, U_3O_8	U_3O_8 Extraction % (from solid assay)	Acid consumption kg/t
1	215	10	95.3	7.75
2	215	12	94.4	7.75
3	215	9	95.8	7.75
Average			95.2	7.75

13.9.7 Crib test procedure

Agglomeration

The crushed ore was fed into the agglomerator feed hopper by front end loader. The ore was the fed into the agglomeration drum by a vibrating feeder and a conveyor. To maintain

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a constant feed rate into the agglomeration drum, the level in the hopper was kept constant at all times.

The ore was agglomerated at a controlled feedrate of sulphuric acid (98% purity), binder (Magnafloc 351) prepared to 0.3% dilution and water added as required. Samples of the agglomerated ore were taken from the luffing conveyor during stacking on an hourly basis for visual inspection then analysed for moisture content. The agglomeration process is illustrated in Figure 13.26

.

The parameters used in the agglomeration process were as established during the previous phases of test work and are summarised in Table 13.25. It is important to note that all previous column test work has been performed by agglomerating small amounts of sample at a time in a batch mode. This test work represent the first test work performed in an agglomeration drum unit and in a continuous manner.

Figure 13.26
Agglomeration Drum

Table 13.25 **Agglomeration Parameters**	
Acid addition, kg/t	6.0
Binder addition, g/t	250
Final Moisture, %w/w	12%

Crib Loading

A 0.35m layer of waste material was laid down at the bottom end of each crib to act as a drainage layer. A slotted corrugated drainage pipe (110mm inside diameter) was laid in the middle of the crib to drain the leach solution out of the crib. The drainage layer has to date not been replaced since the initial loading but was inspected verifying its integrity.

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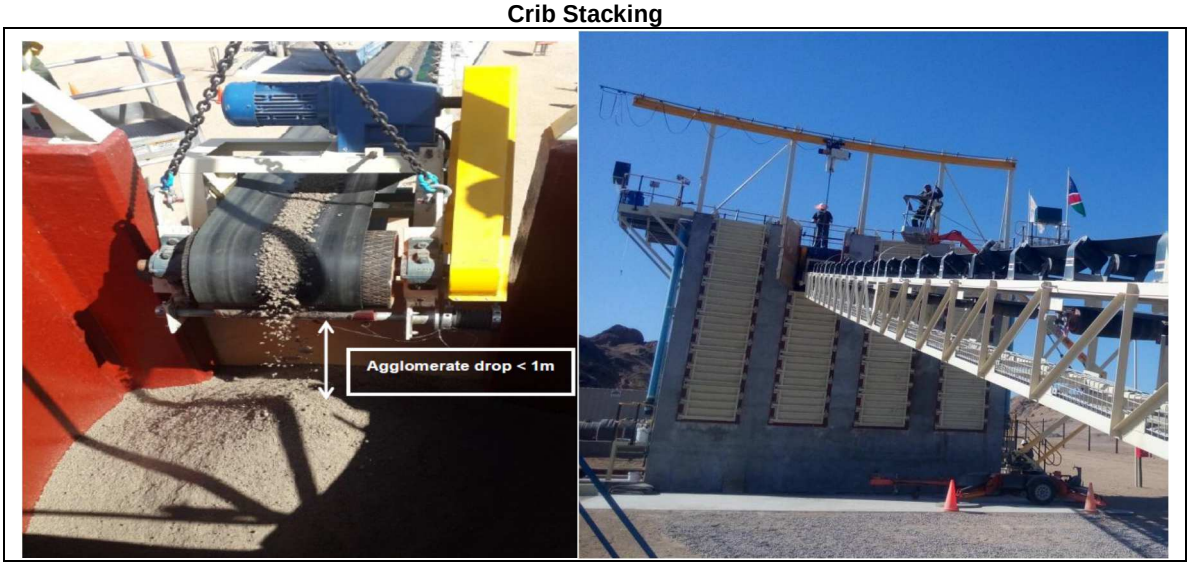
Crib loading occurred concurrently with the agglomeration process using a luffing conveyor. To minimise compaction and segregation, a maximum drop height of 1m from the discharge of the luffing conveyor was maintained during the stacking process as depicted in Figure 13.27. The luffing conveyor was continuously raised as each layer of ore was placed inside cribs. The front end gates were positioned as the staking progressed.

Figure 13.27
Crib Stacking



Irrigation

The leach solution used as feed for the cribs and columns was prepared in a mix tank and then transferred to one of the four irrigation feed tanks (one per each group of a crib and columns) located in the reagent mixing area. Phases 1 and 2 of the test work were completed in open circuit with freshly made up feed solution. The feed solution was assayed for its acid, uranium, ferrous and ferric concentration, pH and redox potential at regular intervals.

Leach solution was discharged at the top of each crib via dripper lines and targeted a constant irrigation flow rate of 15L/h.m^2. The solution was pumped to each crib by a dedicated positive displacement pump. The irrigation flows to the crib and columns were regularly checked by tanks level measurements and by measuring the time required to fill a certain volume in a graduated cylinder. Targeted irrigation parameters are summarised in Table 13.26.

Table 13.26
Irrigation Parameters

Acid in feed solution, g/L	20
Ferric in feed solution, g/L	3
Flow of solution to each crib, L/h	60
Irrigation area, m^2	4.35
Flow rate, L/h.m^2	15
Redox of discharge solution, mV Ag/AgCl	> 450

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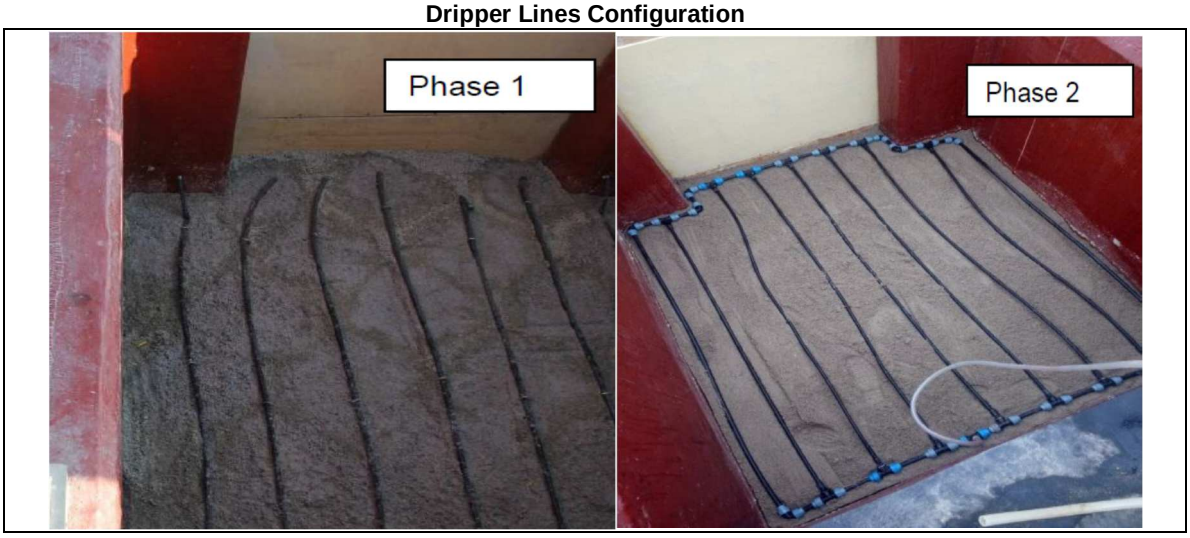
Dripper lines with dripper spacing of 0.3m were used in both Phase 1 and Phase 2. For Phase 2, however, the irrigation system was modified to a closed loop setup thereby ensuring a more uniform irrigation across the surface of the crib (Figure 13.28).

After the stacking was finished, the ore was left curing for three days before the irrigation was restarted for Phase 1 and for one day in Phase 2. The leach irrigation period was run on average for 20 days. The leach period was then followed by a period of drainage, rinse and drainage, each of 2, 3 and 5 days respectively. The purpose was to provide enough time for all solution to drain out of the Crib and to recover any dissolved uranium and acid present in the solids residue before unloading it out of the Cribs/columns.

The rinse was performed using solution at an acid strength of 2g/L and at the same target irrigation rate of 15L/h.m^2.

Figure 13.28
Dripper Lines Configuration



The drained solution from the Cribs was collected in individual tanks and the solution again sampled and assayed for acid, uranium, ferrous and ferric concentrations, pH and redox potential at regular intervals during the drainage cycle. The solution flow rates from the crib and columns for Phase 1 were regularly checked using the same method as the irrigation flow rate. For Phase 2 the discharge tanks were mounted on a 2t platform scale to weigh the discharge solution providing a more accurate measurement. The density of the solution was also regularly measured to convert the weight measurements into volume of solution.

The analytical services were provided by Bureau Veritas in Swakopmund with some analyses (pH and redox potential) performed at the on-site laboratory.

Unloading

At the end of the leaching cycle i.e. after the final drain, the cribs and columns were carefully unloaded in a manner which enabled sampling to assay for uranium content, moisture and size distribution. The unloading was done manually with shovels removing the ripios in one meter intervals and then weighed. The location of these samples was accurately recorded to establish a profile of the leach performance at different sections and at depth. This information was also used to determine the final extraction achieved in each crib. A

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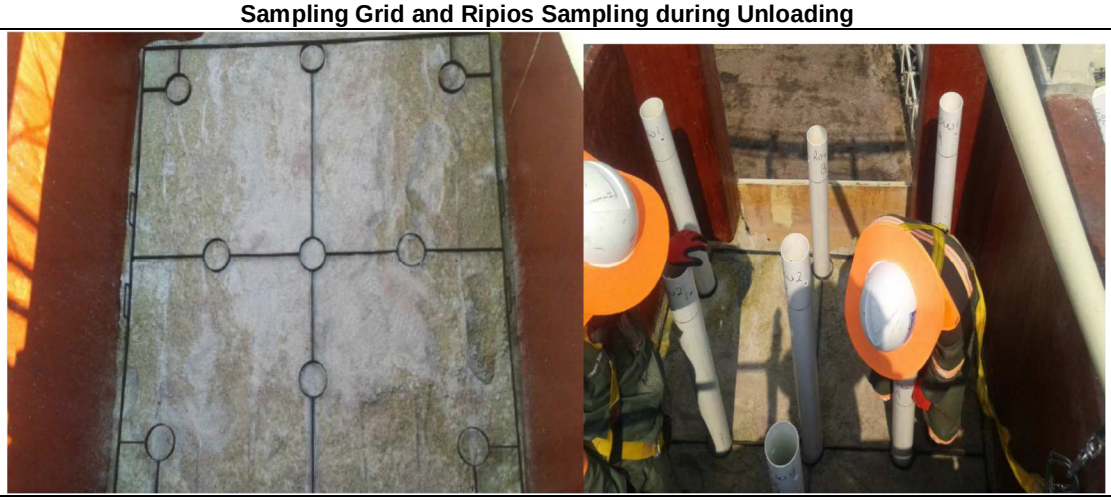
sampling grid (Figure 13.29) was used to keep consistency of the sampling within the various crib and with respect to other cribs.

Figure 13.29
Sampling Grid and Ripios Sampling during Unloading



Columns

The column test procedure in general follows the same basis as the Cribs with the following exceptions:

- Loading: Agglomerated ore from the agglomerating drum was collected into buckets. The ore was then gradually added into the column using a bag and rope-pull methodology. A drop height of approximately 1m was maintained through the loading process in-line with the crib stacking drop height
- Unloading: The unloading of the column was performed by opening the bottom end of the column
- Irrigation: Columns were irrigated from a separate feed pump and feed tank. The solution was discharged at the top centre of the column and in single point.

Operational Improvements implemented in Phase 2

As previously mentioned Phase 1 was a commissioning period with the key objectives to familiarise and educate the operators on the plant operation as well as obtain preliminary results to identify improvements to the procedures and methodology of the demonstration plant. The Phase 1 results led to several improvements to be implemented for the Phase 2 campaign with the key changes being:

- Installation of larger capacity discharge tanks. This improved the solution volume accounting required for progressive extraction calculations in the metallurgical accounting system. The larger tanks were mounted on a 2t platform scale with an accuracy of 0.2kg for volume measurements resulting in accurate volume accounting
- Curing time was reduced from 3 days to 1 day. The change was essential to enhance the initial leach kinetics. The significant volume of solution drained from the cribs and

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columns during the initial 3 day period resulted in an acidity reduction in both the crib and column inventory

- The closed loop irrigation system was adopted to ensure uniform irrigation across the surface of the crib.

13.9.8 Phase 1 and 2 Results

Ore Feed and Agglomeration

Samples were taken during the feed preparation for each crib with a total of four samples for Cribs 1 to 4 and 12 samples for Cribs 5 and 6. The samples were assayed providing an average head grade for all cribs of between 175 and 207ppm (refer to Table 13.27).

The achieved agglomerated material properties for each Crib is summarised in Table 13.27.

Table 13.27
Head grade and Agglomerated Ore Parameters

Parameter	Target	Crib 1	Crib 2	Crib 3	Crib 4	Crib 5	Crib 6
Agglomeration Feed. P_{80} mm	5.3 (*)	3.5	3.5	3.6	3.7	4.0	3.7
Head grade, U_3O_8ppm	205(*)	207.1	195.0	193.1	194.7	174.9	184.8
Acid addition, kg/t	6.0	6.2	4.8	5.3	3.3	6.6	6.0
Binder addition, g/t	250	244	238	282	220	228	245
Moisture, %w/w	12.00	9.15	10.50	10.05	10.00	9.96	10.56
Note (*): Represent the values considered in the DFS design.							

The achieved agglomerated material product (agglomerates) for all cribs visually appeared to be competent and of the right moisture (i.e. not too wet or too dry) as indicated in Figure 13.30. The competency of the agglomerates was later confirmed when during the unloading of the cribs at the end of the leach cycle the agglomerates were found to be generally still present as loaded.

Figure 13.30
Sampling Grid and Ripios Sampling during Unloading

Stacking

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The height of the crib loads were measured at the end of stacking and the bulk density calculated for each Crib as summarised in Table 13.28.

Table 13.28
Cribs Stacking Results

Parameter	Crib 1	Crib 2	Crib 3	Crib 4	Crib 5	Crib 6
Initial mass (wet), t (*)	32.5	33.8	34.4	33.8	33.6	33.9
Initial height (end of tacking), m	5.10	5.10	5.10	5.10	5.19	5.17
B. density ore as stacked (wet), t/m3	1.46	1.52	1.54	1.52	1.49	1.50
Note (*): Calculated based on final residue (ripios) mass.						

Additionally the height of the ore inside the Crib, for Cribs 5 and 6, was measured at different stages of the leach cycle and the slumpage fraction calculated which is summarised in Table 13.29.

The degree of slumpage is slight which suggests that the agglomerates once placed in the Crib are stable, contributing to a good hydraulic conductivity.

Table 13.29
Cribs Height (m) and Slumpage fraction (%)

	Crib 5		Crib 6	
Day	Recorded Height	Slumpage	Recorded Height	Slumpage
0	5.19	0.0%	5.17	0.0%
5	5.11	1.5%	5.14	0.6%
10	5.07	2.4%	5.05	2.3%
15	5.05	2.6%	5.04	2.5%
20	5.05	2.8%	5.04	2.6%
25	5.04	2.9%	5.04	2.6%
30	5.04	2.9%	5.03	2.7%
Final	5.04	3.0%	5.03	2.8%
Final B. density, t/m^3	1.5		1.49	

Ripios

Samples obtained during ripios unloading were again analysed for uranium and moisture content as well as size distribution. The location of these samples were accurately recorded allowing the final uranium extraction in each crib to be determined. For Cribs 1, 5 and 6, nine samples per meter of segment were submitted to the laboratory for the requested analysis. For Cribs 2, 3 and 4, four samples from each meter segment were submitted for analysis primarily to reduce analysis costs. The average final moistures and grades per meter segment for each crib are summarised in Table 13.30 and Table 13.31.

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Table 13.30
Cribs Ripios Moisture at different depths

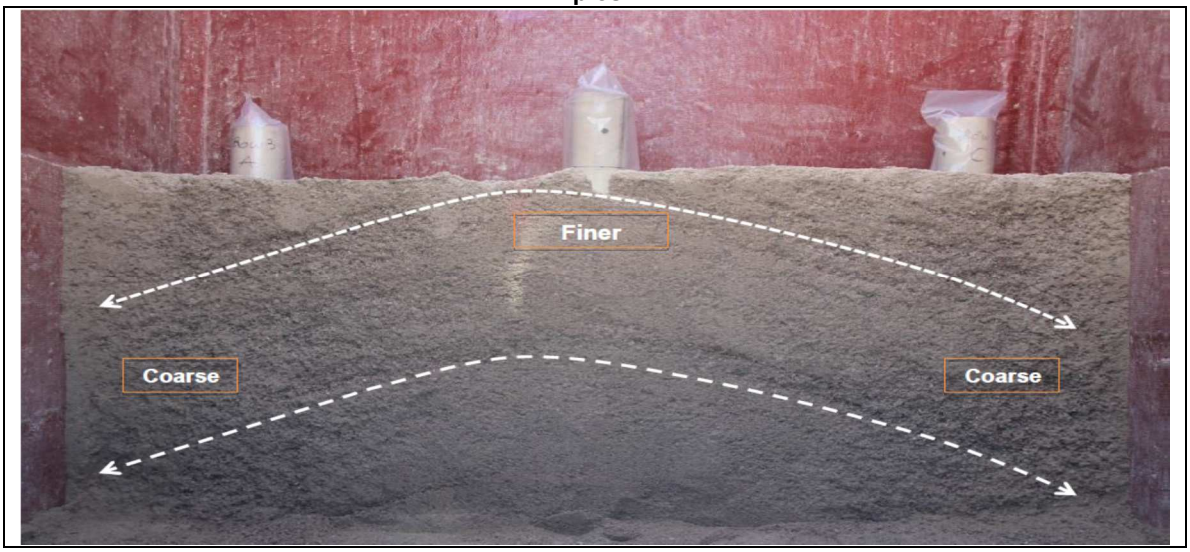
Segment from Top	Crib 1	Crib 2	Crib 3	Crib 4	Crib 5	Crib 6
5m	6.1%	5.7%	5.6%	6.2%	6.6%	6.5%
4m	6.4%	7.0%	6.8%	7.3%	7.3%	7.1%
3m	6.5%	6.7%	7.1%	8.4%	8.0%	7.3%
2m	6.8%	7.7%	7.2%	7.4%	8.4%	7.5%
1m	8.2%	7.4%	8.4%	8.0%	9.4%	8.6%
Avg. moisture	6.8%	6.9%	7.0%	7.5%	8.0%	7.4%

Table 13.31
Ripios grade (U_3O_8 ppm) at different depths and Calc. Head grade

Segment from Top	Crib 1	Crib 2	Crib 3	Crib 4	Crib 5	Crib 6
5m	8.5	7.9	10.6	9.3	10.9	9.6
4m	9.2	7.4	11.2	10.1	11.1	13.7
3m	12.8	13.1	13.1	11.5	12.6	13.3
2m	11.7	12.3	12.2	12.9	11.1	13.7
1m	12.0	13.1	16.2	14.2	11.1	13.7
Avg. ripios grade	11.6	10.8	12.7	11.6	12.4	13.9
Calc. H. grade	183.4	189.3	177.0	184.8	180.3	177.1

In general the moisture content and grade of the ripios increased with depth. Ore stability was evident during unloading with agglomerates still observed. Slight segregation of ore was noticed with the finer material located in the centre section of the cribs (Figure 13.31). The slight segregation didn't appear to have an impact in the flow of solution through the crib with no percolation issues observed during the test work.

Figure 13.31
Ripios



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Leaching

The average results achieved per key parameter, during leaching are shown in Table 13.32. It is noted that the irrigation flows obtained are lower than the targeted 15L/h.m^2 due to some operational difficulties. Effective heap leaching depends primarily on the solution flow and a lower flow can slowdown the leach kinetics. The impact is difficult to evaluate however since all of the cribs were operated at the slower flow rates. Changes in key parameters compared to the DFS target parameters make the evaluation of results in predicting an industrial scale heap more difficult.

Table 13.32
Cribs Average irrigation and Drainage Parameters During Leaching

Parameter	Crib 1	Crib 2	Crib 3	Crib 4	Crib 5	Crib 6
Irrigation flow during leaching, l/h/m^2	13.1	12.8	13.6	13.2	12.9	12.9
Acid in solution to Crib, g/L	17.6	17.0	16.2	16.6	18.9	19.1
Acid in solution out of Crib, g/L	5.1	4.7	4.3	3.5	7.5	7.5
Redox of discharge solution, mV Ag/AgCl	449	471	454	435	450	451
Curing period, days	3	3	3	3	1	1
Leaching period, days	19	18	20	18	20	20
Drain, days	2	2	2	2	2	2
Rinse, days	3	4	3	3	3	3
Final drain, days	6	7	3	3	10	13

The average uranium content left in residue together with the uranium present in the discharge solution were used to establish the calculated head grade (Table 13.33). All extraction rates shown in this section of the report are based on the calculated head grade i.e. the total uranium contained in solution plus the uranium in final residue, unless otherwise stated.

Table 13.33
Calculated Head Grade for the Cribs (U_3O_8ppm)

Parameter	Crib 1	Crib 2	Crib 3	Crib 4	Crib 5	Crib 6
Grade, U_3O_8	183.4	189.3	177.0	184.8	180.3	177.1

Extractions at different depth inside the cribs, were also estimated, based on the residue mass and grade for each segment with the results presented in Table 13.34. It can be observed that the final extraction decreases with depth, which is expected, since the acid concentration in the solution decreases with depth as the acid is consumed. The extraction varied from an average of 95% at the top of the crib (5m segment) to an average of 92.5% at the bottom (1m segment).

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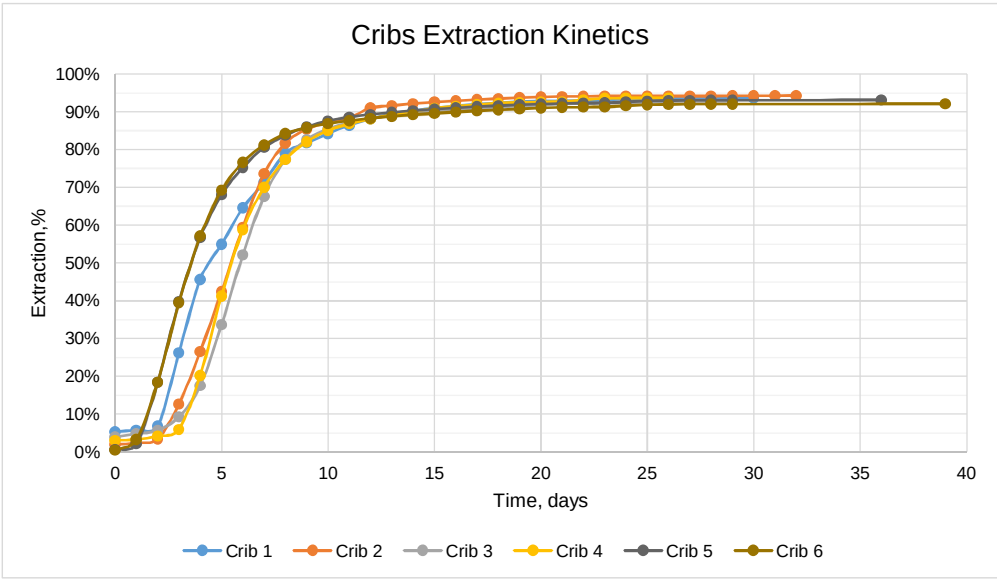
Table 13.34
Cribs Extraction at different depths

Segment from Top	Crib 1	Crib 2	Crib 3	Crib 4	Crib 5	Crib 6
5m	95.4%	95.8%	94.0%	95.0%	94.0%	94.6%
4m	95.0%	96.1%	93.7%	94.5%	93.8%	92.3%
3m	93.0%	93.1%	92.6%	93.8%	93.0%	92.5%
2m	93.6%	93.5%	93.1%	93.0%	93.8%	92.3%
1m	93.4%	93.1%	90.9%	92.3%	93.9%	92.3%
Average	93.6%	94.3%	92.8%	93.7%	93.1%	92.1%

All cribs achieved uranium extractions of over 90% with a combined average of 93.3%. Crib 2 achieved the highest extraction of 94.3%. Figure 13.32 below indicates that by day 11 all cribs had achieved over 85% extraction with a combined average of 87.5%. By day 15 of leaching, all cribs had achieved 90% extraction with a combined average of 90.9%. Please note that all kinetics and acid consumption rates are reported from the end of the curing period unless otherwise stated. Acid consumption also includes the acid loss associated with the ripios unless otherwise stated.

Figure 13.32
Cribs Extraction kinetics



The final extraction rates for all the Cribs are slightly lower than compared with the extraction rate achieved at the bottle roll test of 95.2%. The bottle roll results could be interpreted as the maximum leaching extraction possible for the cribs and columns since the test represents much more aggressive leaching conditions (namely finer particle sizes used, agitation applied, and higher acid solution content).

On the basis that the bottle rolls achieves the maximum extraction possible the cribs results indicate that on average 98% of the leachable uranium was recovered.

From Figure 13.32, it can also be noticed that Cribs 5 and 6 achieved a quicker initial extraction during the first four days when compared with the Phase 1 Cribs (Cribs 1 to 4).

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This is attributable to an early start to irrigation implemented for the Phase 2 test work i.e. curing of 1 day instead of 3 days.

Acid consumption shows a linear gradient in Figure 13.33 up to the end of the leaching period (day ~20) and is similar to the gradients developed during previous column test work.

Due to the drain and rinse periods after the leaching period, acid consumption decreases slightly and linearly to 3.2% on average by the end of the leach cycle. The final acid consumption including the drain, rinse and drain cycles for all cribs was less than the target 17kg/t and averaged 15.5kg/t (Table 13.35.

Figure 13.33
Cribs Acid Consumption

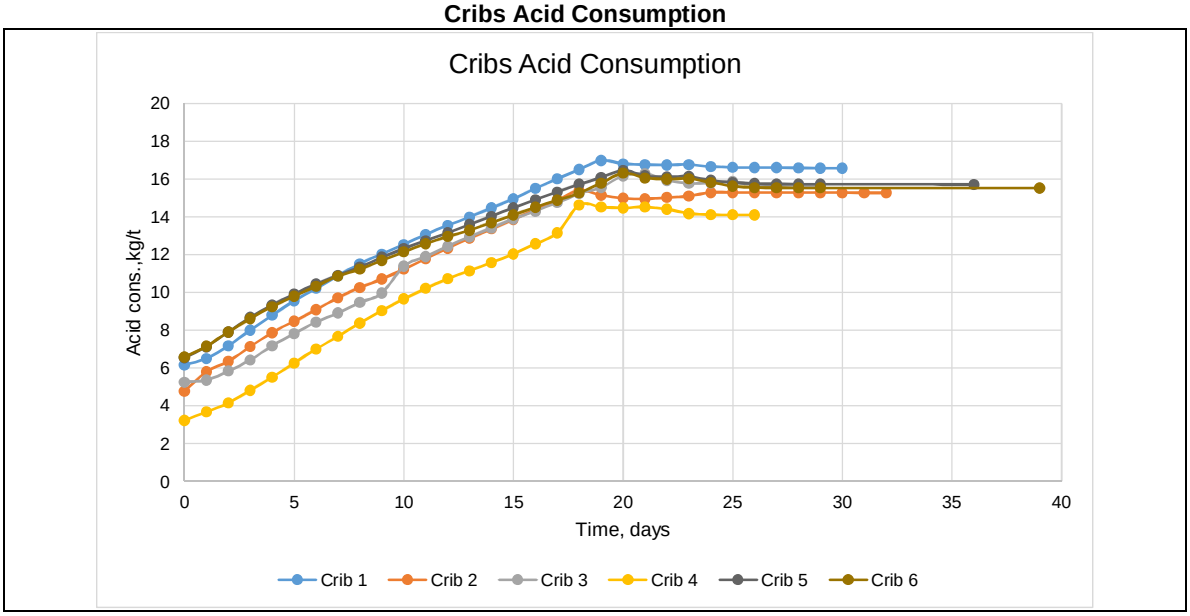

Table 13.35
Cribs acid Consumption, Kg/t

Description	Crib 1	Crib 2	Crib 3	Crib 4	Crib 5	Crib 6
Leaching period	17.0	15.4	16.3	14.6	16.5	16.3
Final, after rinse and drain	16.6	15.3	15.7	14.1	15.7	15.5

Acid consumption is expressed as kilograms of acid at 100% purity, per tonne of dry ore. The mass values used in the calculation of acid consumption rates for the Cribs are the discharge ripios mass since this reflects the more accurate measurement. The columns utilised the initial measured mass due to the lower amount of material being easier to weight accurately.

In general the column results correlate well with the results obtained for the cribs as depicted in Figure 13.34. The columns do show a faster extraction kinetic at the initial stage of leaching up to day 10 after which extraction rates becomes almost identical. The final extraction differs by less than 2.8% for crib and columns test 5 and less than 1.4% for Crib & Columns test 6. The acid consumption trends are similar during the entire leach cycle with a final acid consumption difference of less than 6.2% between crib and columns test 5 and less than 4.4% between crib and columns test 6. These low differences between the results of

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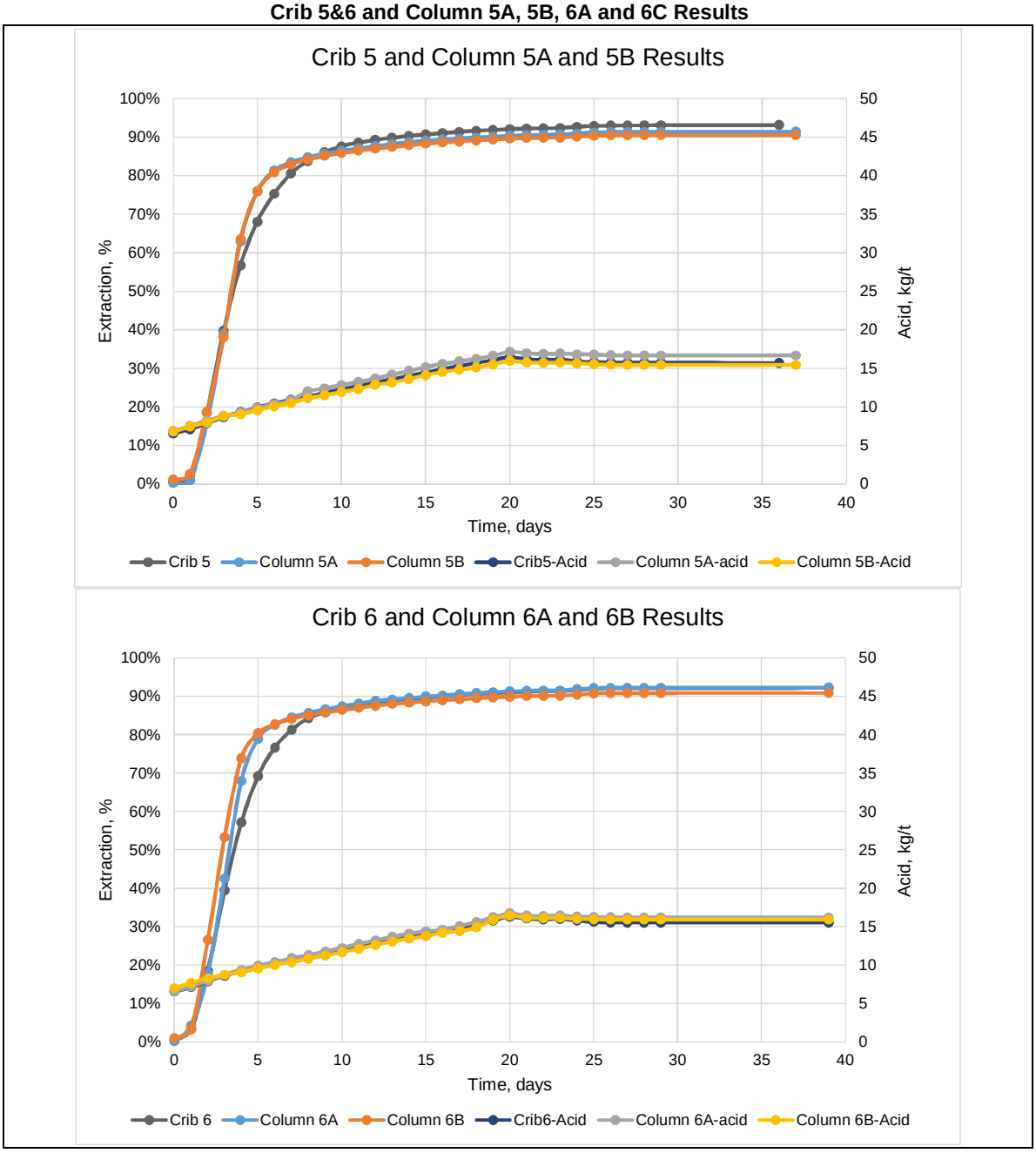
the columns (~200 kg sample) and the Cribs (~30 tonne sample) indicate that a low scale-up factor can be applied for extraction rates and acid consumption.

Demonstration Plant Columns results

Figure 13.34
Crib 5&6 and Column 5A, 5B, 6A and 6C Results



Comparison with previous test work-5m columns in open circuit

Two tests in 5m columns were conducted at the end of 2011 and early 2012 as part of the DFS test work (Refer to Orway Mineral Consultants Report "Bureau Veritas Column Test Results Update Runs 1 to 4" February 2012). The columns were part of Run 3 and Run 4 of

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that test work campaign and identified as column 8. These two column runs were performed under slightly different conditions those used in the demonstration plant and are summarised in Table 13.36. The leach kinetic data for the columns are presented in Figure 13.35 and are plotted against the leaching ratio (m^3/t) to account for the differences in irrigation flow.

The Run 3 and Run 4 columns exhibit much slower kinetics than the demonstration plant columns and this is likely due to the absence of redox control applied until well into the leaching process and to the lower acid concentration used. Column 4 achieved a similar extraction to columns 5A and 6A with an irrigation rate above 0.7 m3/t. The final extraction for Columns 5A and 6A is similar to Run 4 at the equivalent irrigation flow with Run 3 column requiring an additional irrigation volume of approximately 28% to achieve the same extraction. The acid consumption follows the same linear trend with an initial difference due to the shorter curing period used for columns 5A and 6A. The final acid consumption for Columns 5A and 6A is similar to Run 4 at the equivalent irrigation flow.

Table 13.36
Selected Current and Previous Test work columns at 5m

Description	Column 5A	Column 6A	Run 3 Column 8	Run 4 Column 8
Column diameter, mm	185	185	200	200
Sample particle size P_{80}, mm	4.0	3.7	3.35	3.35
Curing period, days	1	1	2	2
Avg. Irrigation flow, L/h.m^2	12.7	13.0	15.0	14.9
Avg. acid in solution to Column, g/L	18.9	19.1	15	15
Redox control	From day 1	From day 1	From day 23 onwards	From day 7 onwards
Redox control method	Fe^{3+} solution	Fe^{3+} solution	Fe^{2+} solution $+H_2O_2$	Fe^{2+} solution $+H_2O_2$
Leaching period, days	20	20	32	32
Final extraction	91.4%	92.3%	93.6%	94.3%

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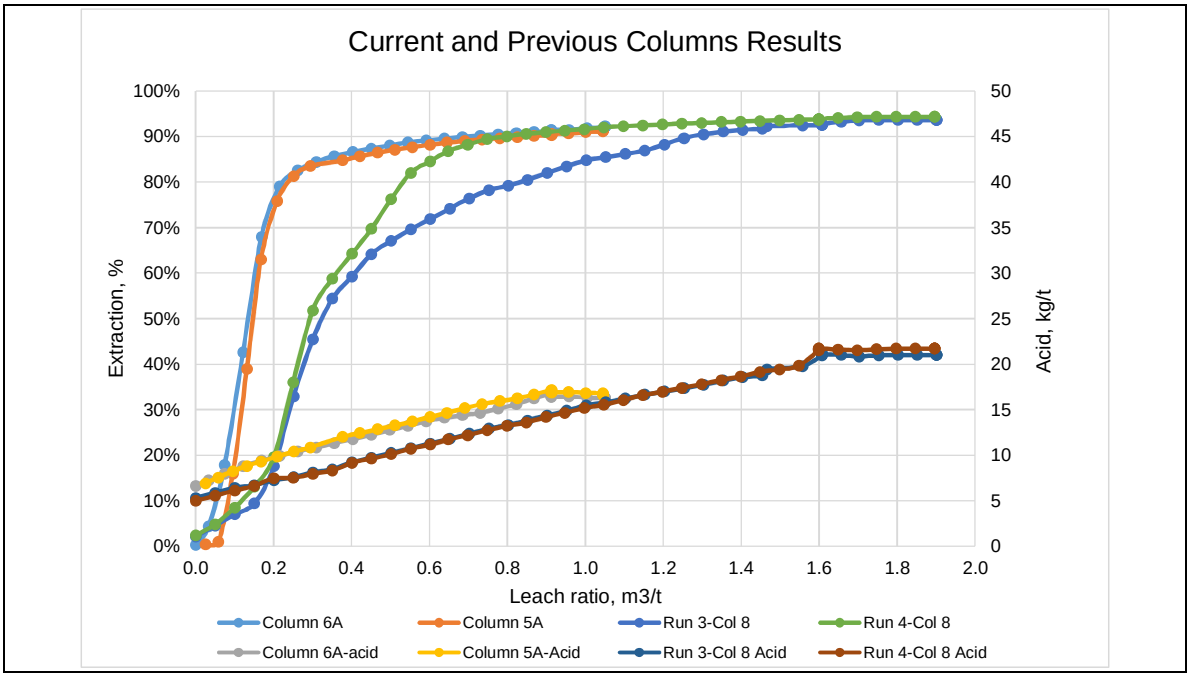
Figure 13.35
Current and Previous Test work columns at 5m



Comparison with previous test work 4m columns in open and closed circuit

As mentioned previously Phase 1 and Phase 2 of the demonstration plant test work were performed in open circuit i.e. utilising only freshly made feed solution. The extraction kinetics in an open circuit are generally faster than in a closed circuit. Phase 3 of the demonstration plant test work underway will provide the first closed circuit results.

As noted in In Section 13.6.3 notes previous test work phases were run with columns in closed circuit, recirculating solution from one column to another (i.e. one recirculation). The results of those tests indicated that the acid consumption over time is unaffected and that while uranium extraction is initially slowed down by using recirculated liquor the overall extraction is not affected. This is not necessarily the case with higher recycling of solution equivalent to the steady state condition of the proposed industrial scale heap.

Demonstration plant results and DFS Design Parameters

The 2012 DFS design parameters are summarised in Table 13.37. The particle size achieved was finer than that considered in the DFS design. Generally a finer particle size would achieve a higher extraction and a higher acid consumption but it is difficult to quantify the difference without further test work.

The DFS design also considered the final extraction rate from the heap to be 87%. The number was obtained applying scale up factors which are derived from the industry experience of other heap leach operations across other commodities primarily copper. In this respect the slight difference between the columns and cribs results are encouraging, indicating that the scale up factor applied in the DFS should be reviewed but after the closed circuit test work results are analysed and preferably with a sample with a similar particle size to the one considered in the DFS design. The continuous closed circuit test work should also be extended to achieve a leaching time that maximise recovery i.e. when no additional

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increase in recovery is obtained as the scale up factors I apply from the terminal extraction point.

Table 13.37
Main DFS design Parameters

Description	
Particle size P_{80}, mm	5.3
Leach time (leach only), days	32
U_3O_8 Extraction, %	87
Acid consumption, kg/t	17.6

13.9.9 Conclusions from Demonstration Plant test work Phase 1 & 2

- Generally the demonstration plant results are similar or better that those obtained in previous test work performed at similar conditions
- The agglomeration process performed with agglomerating drum unit using DFS parameters have been validated
- Despite slight segregation of particles there was no evidence of channelling and the agglomerated material retained their integrity
- During the curing period, a significant amount of solution drained form the cribs, indicating that the agglomerated ore reached it saturation moisture content
- The reduction in the curing period from 3 days to 1 day improved the initial kinetics while reducing the overall cycle
- All cribs achieved uranium extractions over 90% and with an average of 93.3%. This recovery result will be slightly optimistic if 5.3 mm F80 leach feed is targeted rather than the 4.0 mm P80 leach feed tested to date
- The testing of 4.0 mm leach feed rather than 5.3 mm leach feed means that all factors apart from the leach recovery are conservative for heap leaching
- The final acid consumption (after drain, rinse and drain) for all cribs were less than 17kg/t and on average achieved 15.5kg/t
- The acid consumption during leaching is linear with respect to the time which indicates that the longer the time required to achieve a specific extraction target, the higher the acid consumption is going to be
- The crib extraction rates indicates that 98% of the leachable uranium (based on bottle rolls results) was recovered
- The cribs and columns results show good correlation with respect to previous columns with a low differential in final extraction rate and acid consumption
- Cribs results are considered to be more representative, since the conditions are more representative to the agglomeration, stacking, irrigation and drainage methodology expected during a commercial heap leach operation and as such would provide a more accurate picture of the expected results for the full scale plant

13.9.10 Recommendations

- The dissolved composition of the leach solution may have a significant impact on the performance of both the heap leach and SX circuits. To date limited closed circuit test

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work has been conducted and then at low recycling rates. The impact of continuous recycling of solution between the heap and the solvent extraction should be studied in more detail in the close circuit campaign and run for at least three cycles i.e. running three sets of two columns or cribs tests. The test work should incorporate continuous SX contact of column PLS to generate raffinate for return to the columns

- The 2012 DFS assumed that no bleed of solutions to control impurities was required and the only bleed stream from the plant is the solution contained within the leach residue. This assumption should be studied in more detail during a close circuit campaign, as described above. This will allow for the establishment of an impurity concentration profile and investigation of the effect on the heap. Potentially compounds such as jarosite could precipitate in the driplines or within the heap impacting the solvent extraction efficiency and potentially the final product specification
- It is recommended to start future leach test work with the two coarser samples available as soon as possible. This test work should be done at a column level first and later in the cribs. The coarser HPGR result will provide recoveries consistent with the use of 10 mm screen as envisaged in the 2012 DFS design.

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14 MINERAL RESOURCE ESTIMATES

14.1 ETANGO MINERAL RESOURCE

The November 2015 Resource update (Table 14.1) represents a significant increase in the Etango Mineral Resource endowment; the previous estimate was completed in October 2010.

This estimate includes the results of an additional 8 RC holes to the October 2010 update, along with 11 existing holes which were deepened.

An in situ dry bulk density of 2.64t/m^3 was used to report the estimate.

Table 14.1
Etango Deposit, Etango Project, Namibia – November 2015 Resource Estimate

Resource Category	Tonnes (Mt)	Grade (U_3O_8 ppm)	Contained U_3O_8 (Mlbs)
Measured	33.7	194	14.4
Indicated	362.0	188	150.2
Inferred	144.5	196	62.5
Total	540.2	191	227.1

Note: Figures have been rounded. Conversion of lb to kg = x 2.20462
Constrained within the resource pit shell, reported using a 55ppm U_3O_8 cut-off

14.1.1 Introduction

In March 2015, International Resource Solutions Pty Ltd (IRS) completed an update of the Mineral Resource for the Etango uranium project at the request of Bannerman. This estimate was later reviewed and endorsed by the Qualified Person for the resource, Mr Ian Glacken of Optiro in June 2015. Optiro's review highlighted that IRS had not modelled the mineralisation outside of the alaskite bodies and in November 2015 a final Mineral Resource update, encompassing all of the mineralisation at Etango, was completed.

This section details the steps taken in preparing the final November 2015 OK and local uniform conditioning (LUC) estimate.

This section concentrates on the estimate methodology undertaken. The QA/QC, geology, sampling and drilling procedures are discussed in detail in previous sections of this Technical Report.

14.1.2 Mineral Resource Estimate

In November 2015, Optiro finalised the Mineral Resource estimate for the Etango Project (comprising the Anomaly A, Oshiveli and Onkelo prospects). Resource estimates have previously been completed in 2008, 2009, March 2010 and October 2010. The previous Mineral Resource was estimated using ordinary block kriging inside manually generated solids defined at a 75ppm U_3O_8 cut-off. The November 2015 model has been estimated using Ordinary Kriging, followed by post-processing using uniform conditioning (UC) and finally localised into SMUs using a local uniform conditioning (LUC) algorithm.

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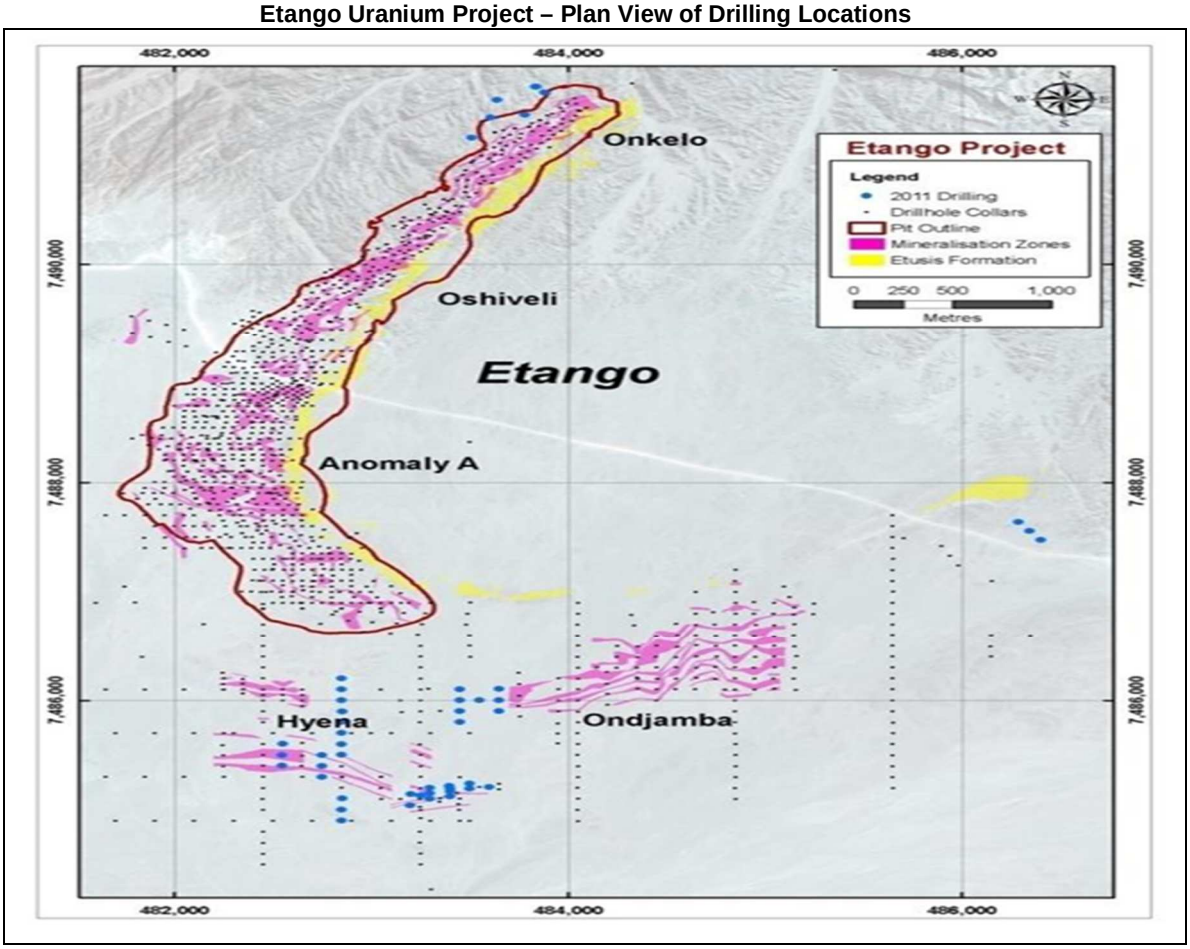
The Qualified Person with respect to the Etango Project resource estimate is Mr Ian Glacken (Director Geology) who is employed by mining consultancy Optiro.

14.1.3 Resource Database and Validation

Database

The drill hole database used for the November 2015 Etango resource estimate consists of 939 RC and 105 diamond drill holes for 239,032m which includes a total of eight new holes and extensions to 11 pre-existing holes for a total of 3219m drilled after October 2010. Only holes drilled by Bannerman have been used in the estimate. Figure 14.1 displays the location of the drill holes used in the estimate and highlights the additional holes used for the November 2015 update.

Figure 14.1
Etango Uranium Project – Plan View of Drilling Locations



The drill holes were drilled typically at -60° to either the east or southeast (UTM grid), with a drill spacing ranging from 25m x 50m, to 50m x 50m and 50m x 100m.

A density value of 2.64t/m^3 was used for the mineralised zones. This value was chosen after analysis of 8883 density determinations from the mineralised zones by water immersion and calliper methods.

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All primary RC and diamond core samples are sent to SGS Johannesburg, a SANAA accredited laboratory (T0169), for crushing, pulverisation and chemical analysis. Samples are analysed by pressed pellet XRF for U_3O_8, Nb, Th, and by borate fusion with XRF for Ca and K. Some pulverised samples are also analysed for uranium in Perth, Australia by SGS.

Where the chemical assays were returned as 'below detection limit', half of the detection limit was assigned to the intervals (2ppm or 5ppm U_3O_8). Intervals which were not sampled were assigned a value of 0.001ppm U_3O_8, affecting 4,040 by 1m intervals.

Validation

The November 2015 drill hole database was validated using a variety of methods including:

- In-field verification of the collar locations of a selection of drillholes by Optiro in September 2015
- Database and visual comparison of assay, collar and survey data against the 2010 validated database
- 3D analysis of collar positions and downhole survey traces.

No significant data related issues were identified and the resulting database is considered to be robust and appropriate for use in resource estimation.

Bulk Density Data

The bulk density readings were taken from 76 diamond drill holes located along the trend of the deposit (Figure 14.2) with a total of 5889 water immersion measurements and 11,113 calliper measurements available. Summary statistics for the mineralised zone and sediment bulk density measurements are shown in Table 14.2.

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Figure 14.2
Location of Density Readings

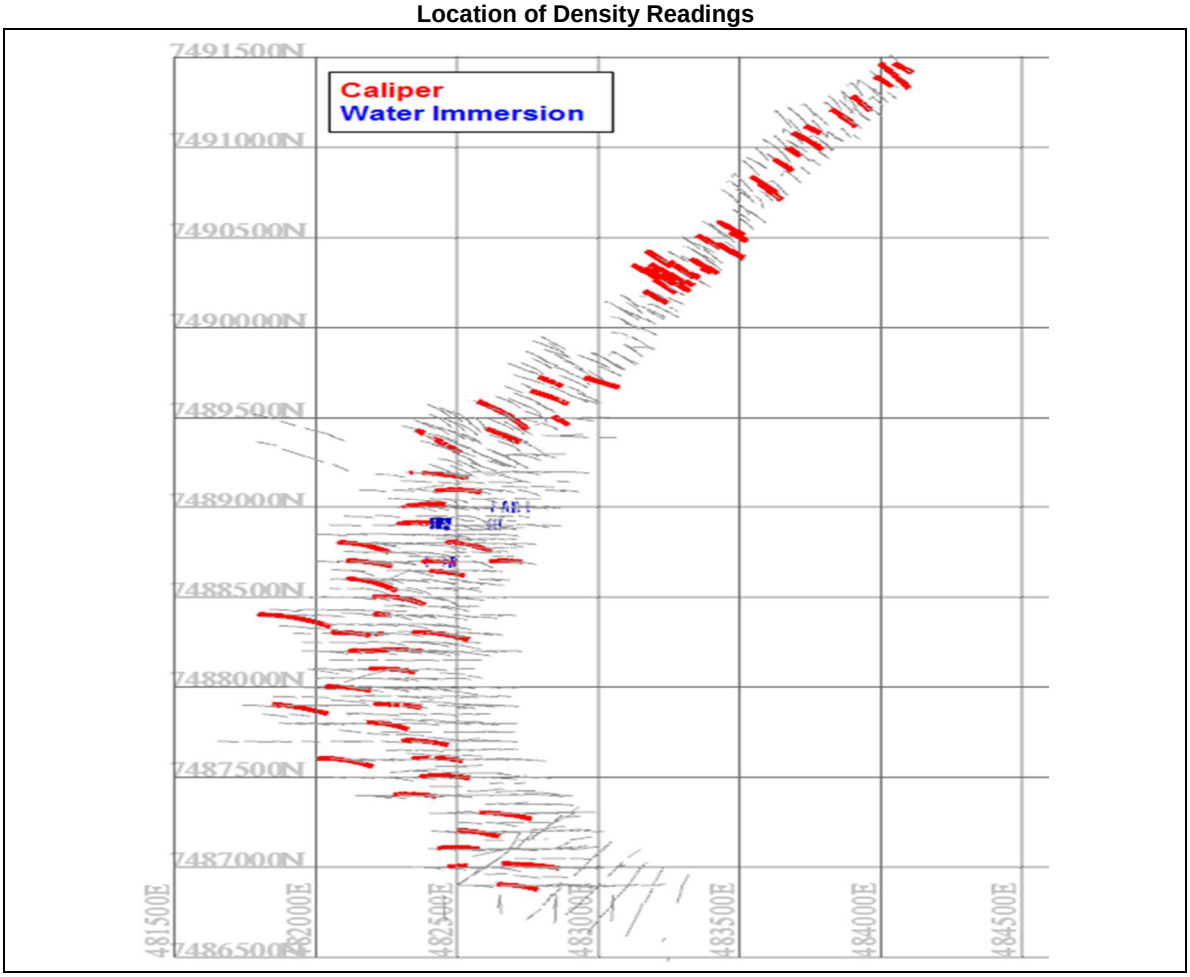

Table 14.2
Summary Statistics for Bulk Density Data (Calliper and Water Immersion) (t/m³)

Item	All Mineralised Zones	All Mineralised Zones <15m from Surface	Alaskites	Chuos (CGN)	Khan (KGN)	Etusis (EGN)
Count	4,369	141	6,559	1,987	126	118
Minimum	1.95	2.50	1.01	1.42	2.59	1.77
Maximum	5.37	2.89	5.37	3.83	3.32	3.40
Mean	**2.64**	**2.65**	**2.63**	**2.71**	**2.86**	**2.81**
Median	2.63	2.64	2.63	2.71	2.83	2.78
Standard Deviation	0.08	0.05	0.09	0.10	0.14	0.18
Variance	0.01	0.00	0.01	0.01	0.02	0.03
Coefficient of Variation	0.03	0.02	0.03	0.04	0.05	0.06

The mineralised zones are predominantly alaskite with minor metasedimentary units. For the mineralised zones, the bulk density measurements averaged 2.64t/m³. Based upon the

water immersion and calliper readings, the Chuos, Khan and Etusis units had average bulk density values of 2.71t/m³, 2.86t/m³ and 2.81t/m³ respectively.

Figure 14.3 shows histogram plots of the mineralised zone bulk density data. Figure 14.4 shows histogram plots of the meta-sedimentary unit bulk density data.

Figure 14.3
Histogram Plot of the Mineralised Zones Bulk Density Measurements

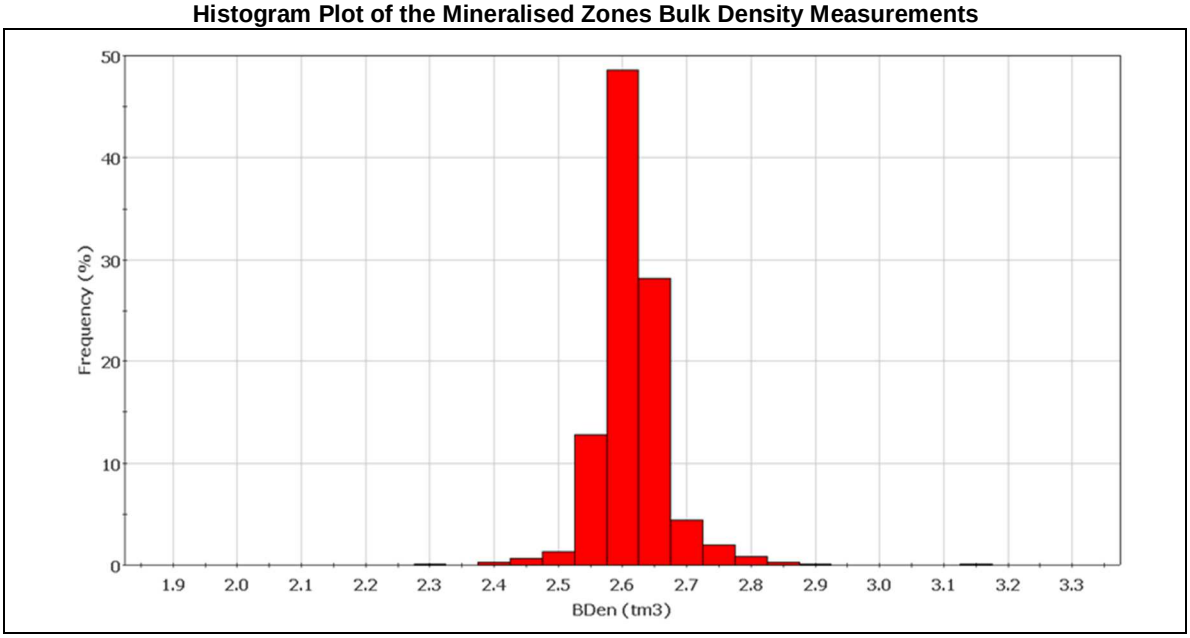

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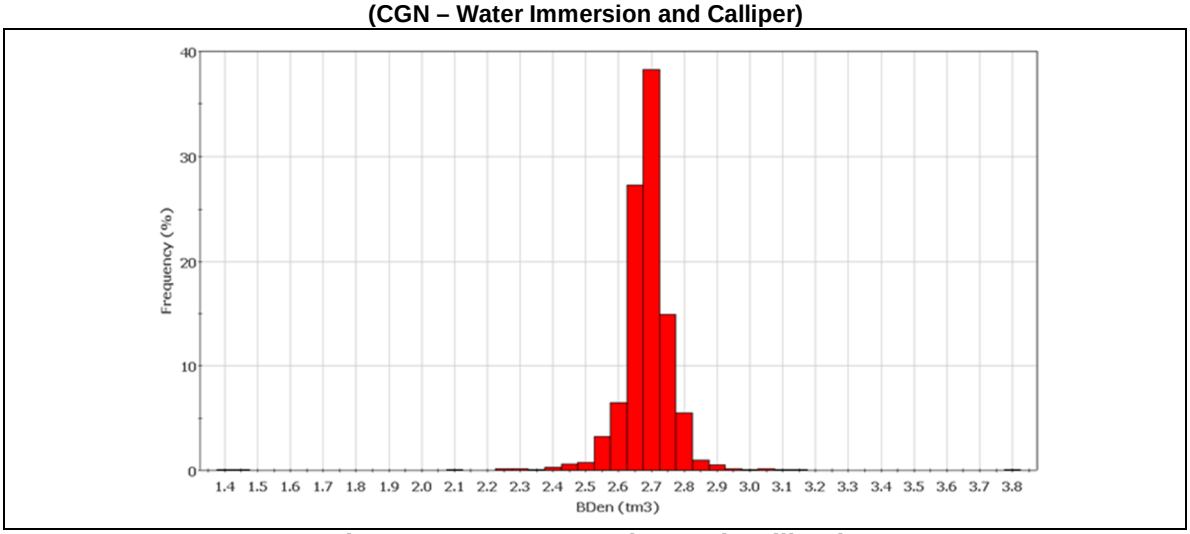
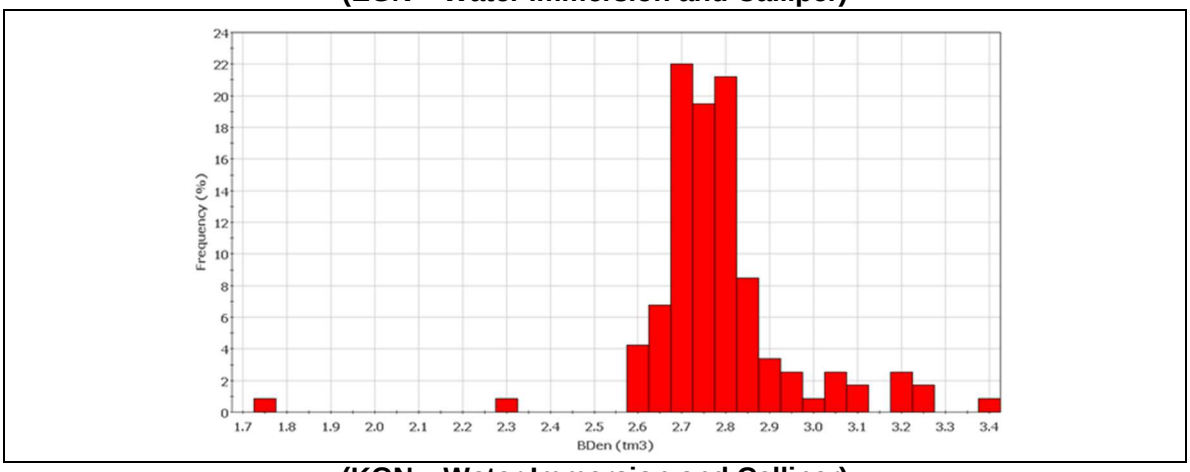
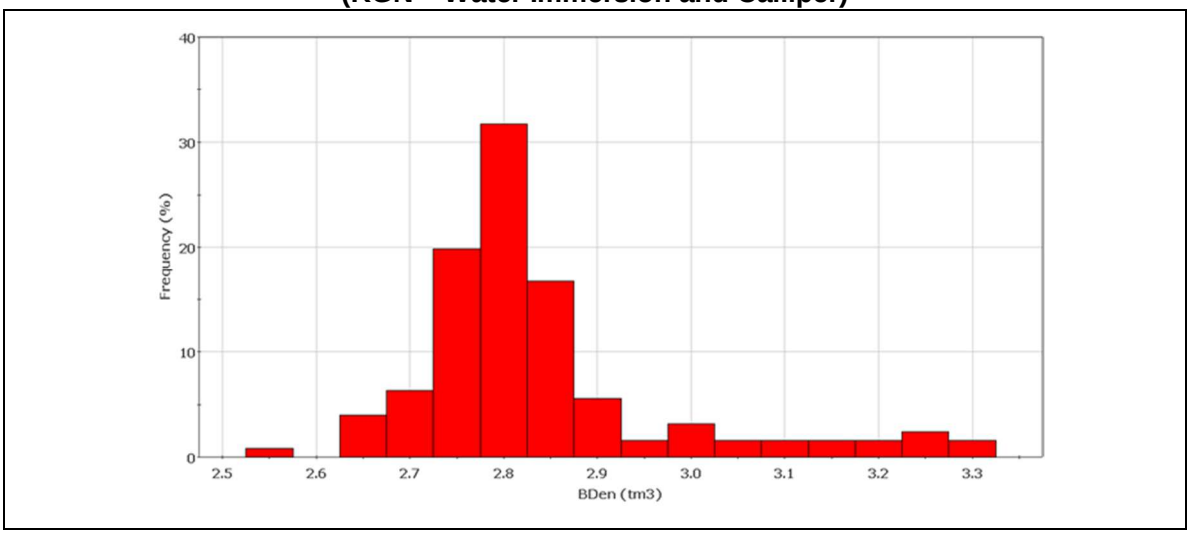
Figure 14.4
Histogram Plot of Bulk Density Readings from the Metasediments
(CGN – Water Immersion and Calliper)



(EGN – Water Immersion and Calliper)



(KGN – Water Immersion and Calliper)



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14.1.4 Geological Interpretation and Domaining

Weathering Profile

The pedolith mainly consists of less than 1m of transported sands. In places minor calcrete or gypcrete is encountered within the transported sand, and, where present, often binds the sand grains together to form a surface cap.

At Anomaly A/Oshiveli, the base of the weathering profile in the alaskites and surrounding meta-sediments was logged to extend typically less than 50m from the surface. At Onkelo, the base of weathering where recorded was typically at 3m or less.

Some leaching of uranium from the alaskites near surface was evident. This is thought to be associated with oxidation observed in the upper parts of the deposit. Based upon the available core density measurements, the effect of weathering on density within the profile is considered to be negligible (e.g. the average density of the 55 density readings taken within 5m from surface was 2.64t/m^3).

Geological Interpretation and Mineralisation Domaining

The majority of the uranium mineralisation (over 90% by both metal content and by sample count) is associated with the alaskite bodies and typically follows the trends of the alaskite contacts. This type of mineralisation is termed alaskite-dominant (AD) mineralisation. Minor uranium mineralisation is also found in the metasedimentary sequences adjacent to the alaskite contacts, associated with metasomatic alteration and in thin alaskite stringers within the metasediments. This style of mineralisation is termed alaskite sub-dominant (ASD) mineralisation.

For the purposes of the November 2015 estimate, Etango has been broadly separated into three mineralised areas, referred to as north, mid and south domains (Figure 14.5). These areas are based on local changes in strike and dip directions of the mineralised trend throughout the deposit. The north domain is defined as areas >7,488,950mN, the mid domain is defined as ≤7,488,950mN and ≥7,487,450mN and the south domain as <7,487,450mN.

Previous resource estimates at Etango have focussed on the manual interpretation of numerous three dimensional (3D) wireframe models using geological and grade criteria (within alaskite, parallel to the alaskite trend and above a 75ppm or 100ppm U_3O_8 threshold). Manual generation of these solids is considered time-consuming not without risk.

Geological logging supports the general interpretation of the mineralised trends striking to the southeast (south domain), north (mid domain) and northeast (north domain) and dipping between 20° to 40° to the west, following the western flank of the Palmenhorst Dome. Locally however, continuity is commonly difficult to determine with numerous, plausible interpretations available at the current nominal drillhole spacing of 50m. In order to best navigate this risk, the constraining mineralisation model used in the November 2015 resource estimate was based upon two grade shells (Figure 14.6), representing the two distinct styles of mineralisation, and generated using a two-stage Categorical Indicator Kriging technique.

Figure 14.5

Etango Uranium Project – November 2015 Mineralisation and Estimation domains

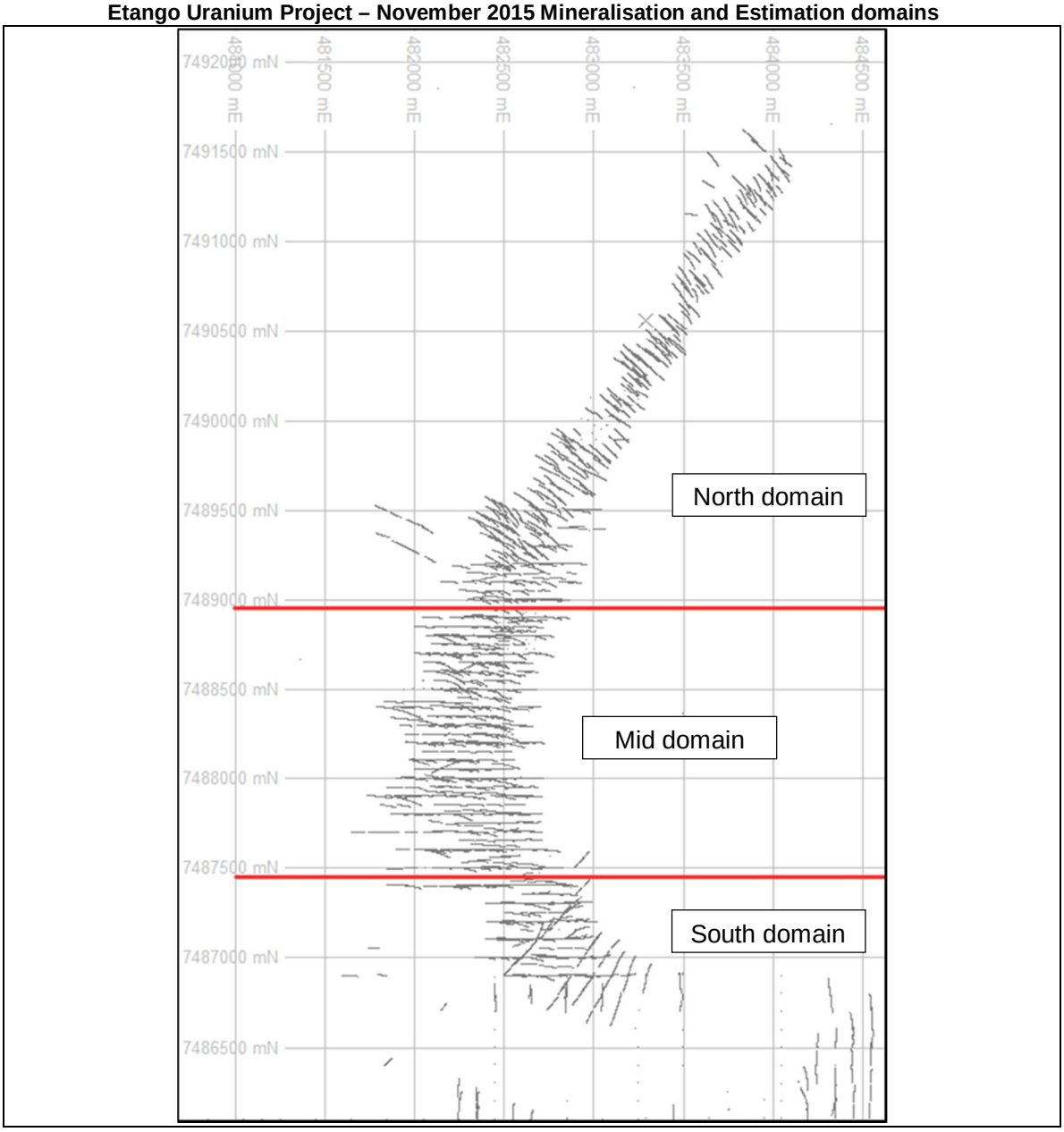

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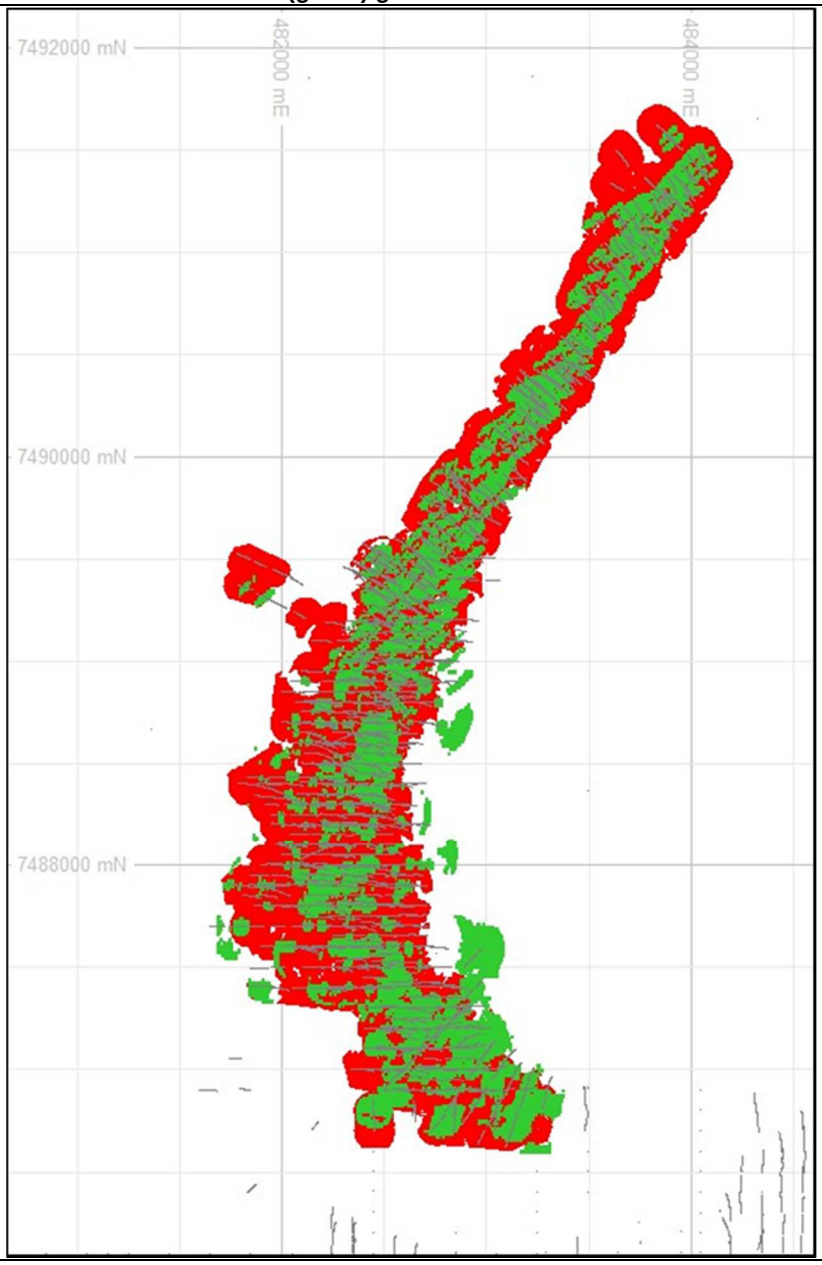
Figure 14.6

Etango Uranium Project – November 2015 Alaskite Dominant (red) and Alaskite Sub-Dominant (green) grade shells



Alaskite Dominant Mineralisation

The AD grade shell was created in Vulcan using 3m downhole composites, taking into account the distribution of both the mineralisation (above a 50ppm U_3O_8 threshold) and the dominant host lithology. Intersections where logged alaskite lithology contributed more than 33% of a 3m drilling composite were flagged (LITH_N=1) and then a second flag was applied (IND_50=1) for composites above a 50ppm U_3O_8 threshold, effectively removing unmineralised alaskite samples. A probability variable (0/1) was then estimated for each orientation domain (north, mid and south) using OK. Parameters for the CIK AD estimate are presented in Table 14.3. Variography used for the AD grade shell estimate was adopted

from that calculated in Isatis from 5m composites of the data above 50ppm U_3O_8 (regardless of lithology) as it appeared to smooth the small scale variability. The indicator variography for each estimation domain is presented in Figure 14.7. The final AD grade shell was subsequently created around areas that had greater than a 40% probability of achieving the target grade and host lithology criteria.

Table 14.3
Alaskite Dominant Grade Shell – CIK estimation parameters

Domain	Pass	Axis orientation (°)			Search distance			Min No of Comp	Max No of Comp	Max No of Comp per Hole
		Major	Semi Major	Minor	Major	Semi Major	Minor			
North	1	215	0	-20	100	100	40	1	8	4
Mid		190	0	-20	100	100	40			
South		250	-20	0	100	100	40			

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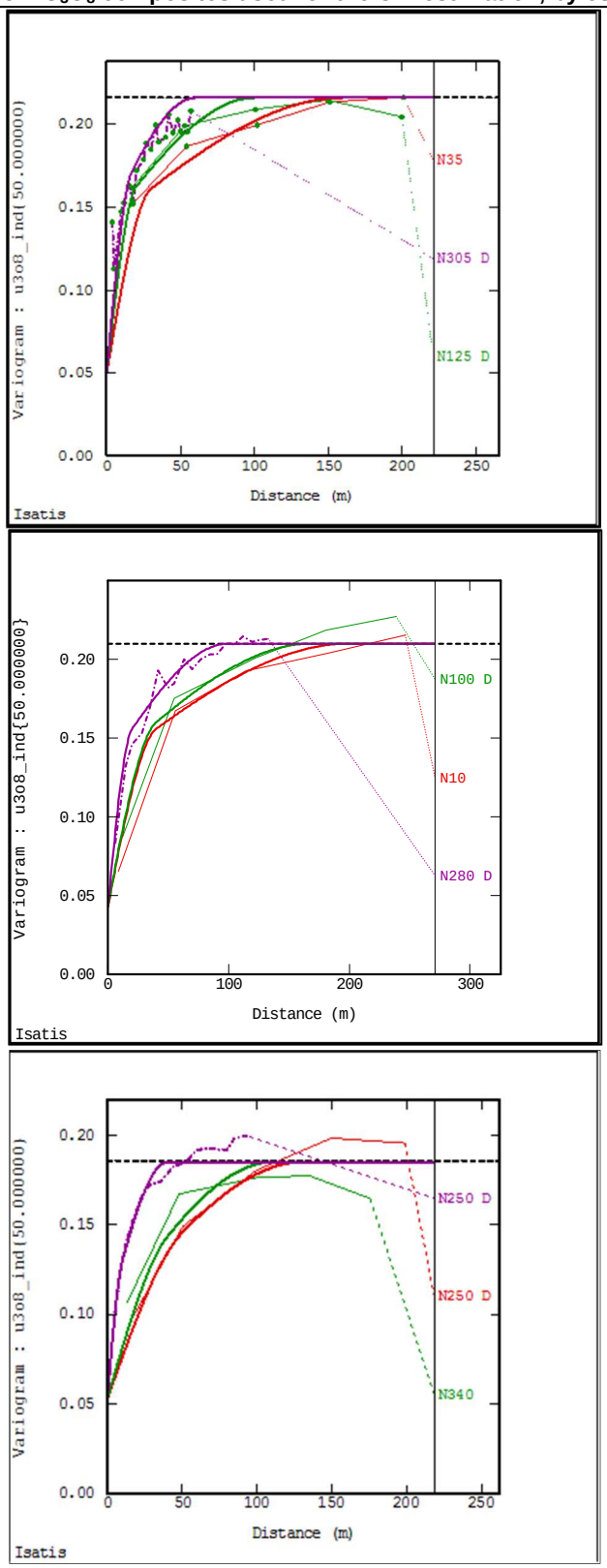
Figure 14.7

Variograms of the 5m U_3O_8 composites used for the CIK estimation, by estimation domain



North

Mid

South

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Alaskite Sub-Dominant Mineralisation

For the ASD mineralisation, all intersections outside of the AD grade shell and above a 50ppm U_3O_8 threshold were used, regardless of lithology. As with the AD mineralisation, a probability variable (0/1) was then estimated by orientation domain using OK. The ASD CIK model was estimated using Datamine Studio 3 into a panel block model of 6.25mE by 12.5mN by 4mRL. Parameters for the CIK ASD estimate are presented in Table 14.4. Indicator variography was completed in Supervisor using the 3m ASD coded composite data by orientation domain, and is summarised in Table 14.5. Blocks above a 40% probability of being above the 50ppm U_3O_8 threshold were integrated into the final ASD mineralisation grade shell.

Table 14.4
Alaskite Dominant Grade Shell – CIK estimation parameters

Domain	Pass	Axis orientation			Search distance			Min No of Comp	Max No of Comp	Max No of Comp per Hole
		Major	Semi Major	Minor	Major	Semi Major	Minor			
North	1	140	160	0	100	25	6	4	24	4
Mid		100	170	-30	185	135	18			
South		50	170	-130	100	85	9.5			

Table 14.5
Alaskite Sub-Dominant Grade Shell – Variography of 3m U_3O_8 composites

Domain	Axis	Direction	Nugget	Structure 1		Structure 2	
				Sill	Range	Sill	Range
			C_0	C_1	A_1	C_2	A_2
North	Direction 1	00°→230°	0.45	0.26	15	0.29	100
	Direction 2	-20°→320°			8		25
	Direction 3	-70°→140°			5		6
Mid	Direction 1	-05°→220°	0.42	0.33	85	0.25	185
	Direction 2	-09°→>310°			115		135
	Direction 3	-80°→100°			3.1		18
South	Direction 1	-08°→270°	0.51	0.12	70	0.37	100
	Direction 2	06°→360°			63.5		85
	Direction 3	-80°→050°			6.9		9.5

14.1.5 Statistical Analysis

Radiometric Data Factoring

The vast bulk of the assays (>90%) used in the resource estimate were analysed by XRF, with the remainder being factored gamma log eU_3O_8 analysis sourced from the Auslog tool.

As the radiometric data constituted a relatively small portion of the resource dataset, the factors obtained from the 2008 resource study were applied to the radiometric data (after checking).

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The linear regressions used for the factoring of the Auslog eU_3O_8 data to minimise any relative bias are shown below:

- Bin 1 – 0ppm to 1100ppm eU_3O_8:
  - Factored Auslog = Auslog eU_3O_8ppm * 0.86 - 26.
- Bin 2 – 1100ppm to 1700ppm eU_3O_8:
  - Factored Auslog = Auslog eU_3O_8ppm * 1.03 - 67.
- Bin 3 – >1700ppm:
  - Factored Auslog = Auslog eU_3O_8ppm * 0.96 – 79.
- Any factored data that was less than 5ppm was given a grade of 5ppm U_3O_8.

Statistical Analysis of Composites and Top Cuts

The bulk of the sampled intervals were 1m in length. After consideration of the geological setting and mining, including the likely mining selectivity and bench/flitch height, a regular 3m U_3O_8 (downhole) composite length was selected. Compositing for the grade estimation was stopped at the grade shell boundaries and composites with residual intervals of less than 1.2m were retained by combining with the previous composite. Summary statistics of the U_3O_8 composites are presented in Table 14.6.

Histogram plots for each estimation domain are presented in Figure 14.8. All plots demonstrate the strong positive tail typical of the deposit; however, all datasets also have moderate coefficients of variation (standard deviation/mean) of between 0.92 and 1.41, indicating that positive outliers do not necessarily heavily impact upon the mean of the data population.

Table 14.6
Summary statistics of the 3m U_3O_8 composites, by estimation domain

Domain	Zone North		Zone Mid		Zone South	
	Alaskite-dominant AD	**Alaskite-Subdominant ASD**	**Alaskite-dominant AD**	**Alaskite-Subdominant ASD**	**Alaskite-dominant AD**	**Alaskite-Subdominant ASD**
Count	8,835	803	10,376	384	2,673	333
Minimum	0.001	0.001	0.001	0.001	0.001	0.001
Maximum	2,282.33	2,624.60	2,841.67	1,046.98	1,943.67	1,958.00
Mean	**167.61**	**125.48**	**188.29**	**151.14**	**174.79**	**147.08**
Standard Deviation	170.04	177.84	177.33	175.31	161.2	200.28
Variance	28,914.71	31,625.3	31,445.99	30,734.4	25,985.37	40,111.8
Coefficient of Variation	1.01	1.41	0.94	1.16	0.92	1.36

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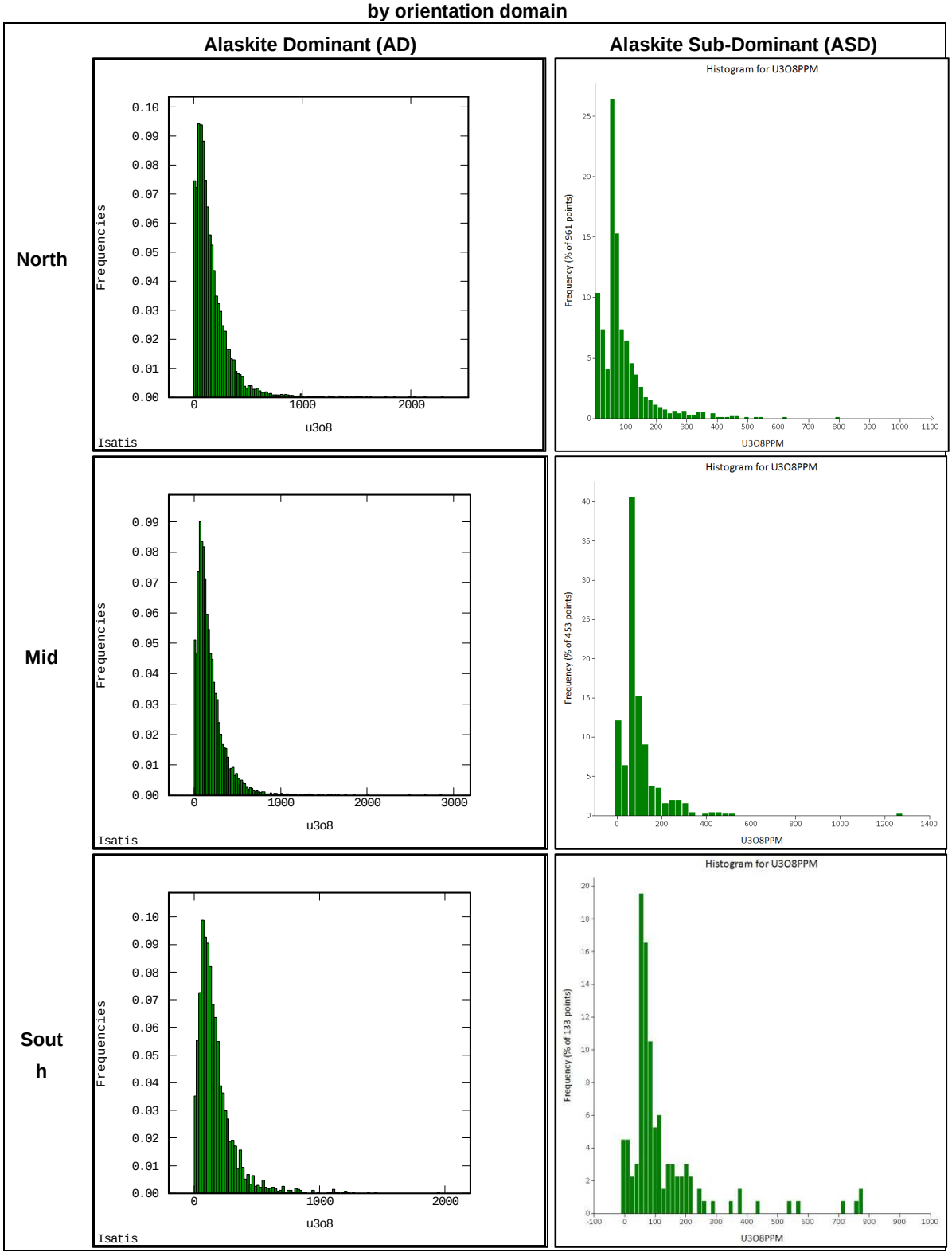
Figure 14.8

Histograms of 3m U_3O_8 Composites for Alaskite Dominant and Alaskite Sub-Dominant Mineralisation, by orientation domain



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The effects of the highest grade composites on the mean grade and standard deviation of the uranium dataset for each of the estimation domains were investigated using a combination of compilation and reviewing statistical plots (histograms and probability plots), assessing the effect of potential top cuts on the population statistics and review of the population disintegration at higher grades. A top cut of 1700ppm was applied to the north and mid domain for the AD mineralisation, which was reduced to 1300 for the south domain. For the ASD mineralisation domains, a top cut of 900ppm was applied to all domains prior to estimation.

Composite data was viewed in 3D to determine the existence of clustering or otherwise of the highest grades observed in each domain to assess the appropriateness of the high grade cut. A list of the top cuts applied and their impact on the mean grades of the domains is provided in Table 14.7. A cell declustering algorithm was applied to derive the declustered statistics.

Table 14.7
Summary statistics of the 3m U_3O_8 composites, by estimation domain

Domain	Zone North		Zone Mid		Zone South	
	Alaskite-dominant AD	**Alaskite-Subdominant ASD**	**Alaskite-dominant AD**	**Alaskite-Subdominant ASD**	**Alaskite-dominant AD**	**Alaskite-Subdominant ASD**
Count	8,835	803	10,376	384	2,673	333
Minimum	0.001	0.001	0.001	0.001	0.001	0.001
Maximum	1,700	900	1,700	900	1,300	900
Mean	**161.91**	**118.72**	**176.54**	**149.89**	**173.21**	**137.07**
Standard Deviation	167.48	128.65	167.27	147.42	160.39	158.38
Variance	28,050.4	16,551.3	27,979.6	21,732.9	25,725.7	25,083.8
Coefficient of Variation	1.03	1.08	0.95	0.98	0.93	1.16
No of Samples cut	0	2	0	1	0	0

14.1.6 Geostatistical Analysis

<u>Alaskite Dominant Mineralisation</u>

Traditional semi-variograms were used to analyse the spatial variability of U_3O_8 for the AD domains and were calculated on top cut and declustered (3m) composite data using the geostatistical software, Isatis. Modelled variograms were generally shown to have good structure, were used throughout the OK estimation and also were used for the change of support process as input to the Uniform Conditioning (UC) post-processing. Interpreted anisotropy directions correspond well with the modelled geology and overall geometry of the interpreted domains. The results of the variography analysis are summarised in Table 14.8.

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Table 14.8
Alaskite Dominant Mineralisation – U_3O_8 Variogram Parameters

Domain	Rotation			Nugget	Structure 1				Structure 2			
					Sill	Range (m)			Sill	Range (m)		
	Major	Semi-Major	Minor	C_0	C_1	Major	Semi-Major	Minor	C_2	Major	Semi-Major	Minor
North	215	0	-20	0.36	0.32	30	30	5	0.32	130	110	25
Mid	190	0	-20	0.36	0.25	40	30	7	0.39	110	80	18
South	250	-20	0	0.35	0.19	70	40	5	0.46	120	70	25

All zones exhibited a well-structured downhole variogram with a relative nugget of approximately 35%. The variography in the major and semi-major axes generally had moderately defined structure and was modelled with a first structure at ranges of between 30m to 70m in the major axis. This has typically resulted in the zones having between 30% and 50% of the total variance modelled within the range of the first structure. Incorporating the second structure, the total range of the major axis ranges from 110m to 130m. Figure 14.9 shows the obtained variography from the AD mineralisation domains.

Alaskite Sub-Dominant Mineralisation

Grade variography was completed using Supervisor on the ASD mineralised composites for each domain. Variograms were transformed using Normal Scores in order to improve the variogram resolution, with the resulting parameters back-transformed before use using a Gaussian anamorphosis. A summary of the results of the variography analysis is presented in Table 14.9.

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Figure 14.9
Alaskite Dominant Variogram Plots by domain

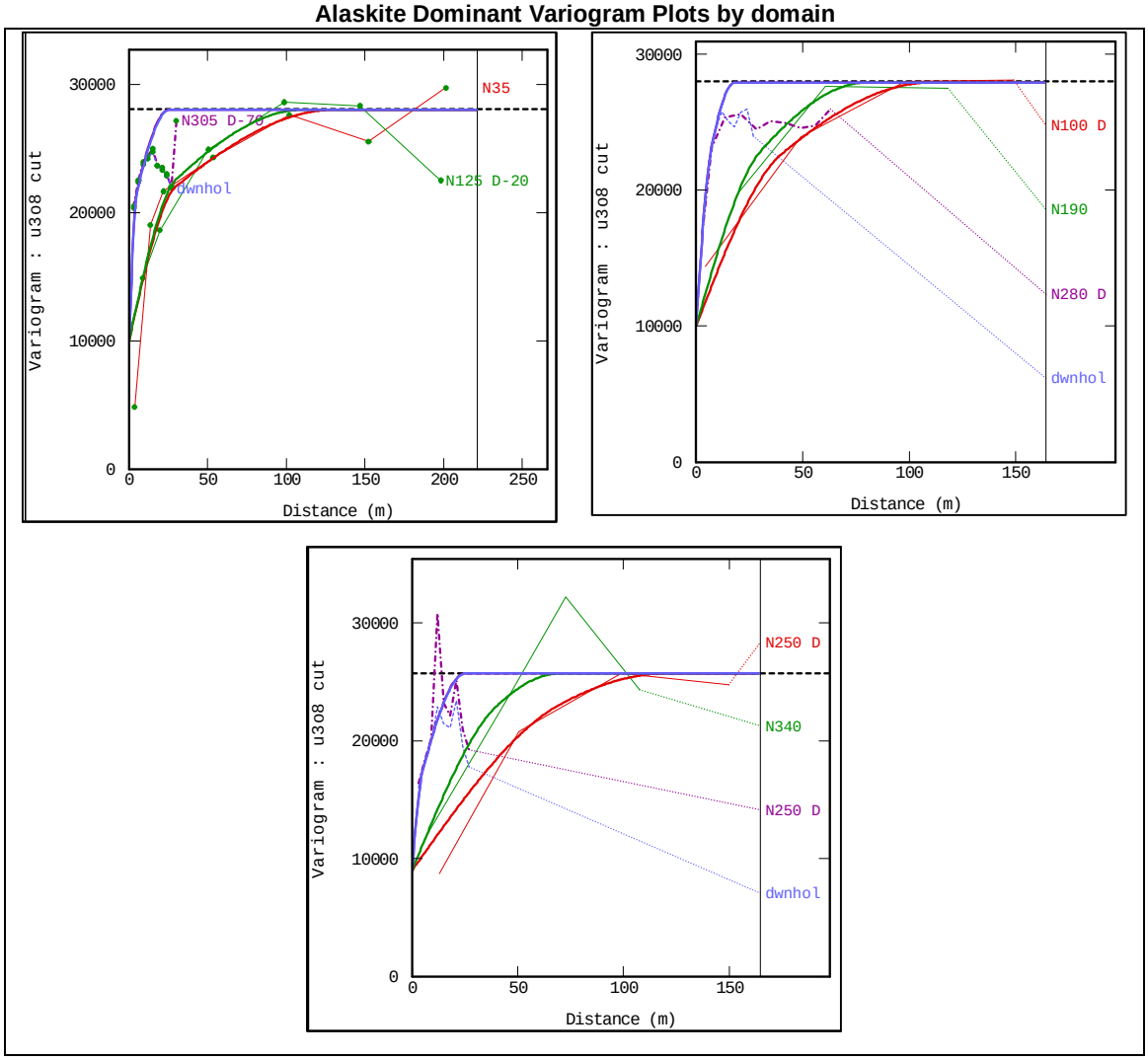

Table 14.9
Alaskite Sub Dominant Mineralisation – U_3O_8 Variogram Parameters

Domain	Axis	Direction	Nugget	Structure 1		Structure 2	
				Sill	Range	Sill	Range
			C_0	C_1	A_1	C_2	A_2
North	Major	00°→220°	0.35	0.43	55	0.22	170
	Semi-Major	10°→130°			95		95
	Minor	-80°→130°			2.5		8.5
Mid	Major	-03°→320°	0.30	0.31	400	0.39	515
	Semi-Major	-09°→>230°			185		185
	Minor	80°→250°			2.8		4.3
South	Major	-20°→240°	0.34	0.31	170	0.35	200
	Semi-Major	00°→330°			115		155
	Minor	-70°→060°			3		6.5

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14.1.7 Block Model Construction

The Etango block model was created with parent block dimensions of 25 m E by 25 m N by 8 m RL selected on the basis of the average drill spacing across the deposit. The block model was subcelled down to 6.25 m E by 12.5 m N by 4 m RL (the SMU size) to ensure adequate volume resolution of the mineralised domains. The model covered all the interpreted mineralisation zones and included suitable additional waste material to allow later pit optimisation studies. The block model parameters are presented in Table 14.10.

Block coding was completed on the basis of the block centroid, wherein a centroid falling within any wireframe was coded with the wireframe solid attribute. No rotation was applied to the block model.

Table 14.10
Etango Block Model Parameters

	Block model extents		Number of Blocks	Block size (m)	SMU/subcell (m)
	Minimum	Maximum			
Easting	481687.5	484212.5	101	25	6.25
Northing	7486587.5	7491812.5	209	25	12.5
Elevation	-218	326	69	8	4

14.1.8 Grade Estimation - Ordinary Kriging

Grade estimation on the panel scale was completed using Ordinary Kriging (OK) within the defined indicator mineralisation shells for both the Alaskite Dominant and Alaskite Sub-Dominant domains. Grade estimation for the AD mineralisation was carried out using Isatis and Datamine Studio 3 for the ASD mineralisation.

Alaskite Dominant Mineralisation

Ordinary Kriged (OK) estimates of U_3O_8 were completed for each of the AD orientation domains (north, mid and south) using the respective grade variogram model, and a set of ancillary parameters controlling the source and selection of composite data. The sample search parameters were defined based on the variography and the data spacing, and a series of sample search tests were performed in Isatis. The final set of estimation parameters is detailed in Table 14.11.

Soft domain boundaries between the orientation domains were used throughout. A two-pass estimation strategy was applied, utilising progressively larger and less restrictive searches and only considering un-estimated blocks for each successive pass. All estimations were completed at the parent cell scale using a discretisation grid of 7 (X) by 7 (Y) by 5 (Z). There was no restriction on the number of drillholes used per block estimate.

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Table 14.11 Alaskite Dominant Mineralisation – OK Sample Search Parameters					
Domain Name	**Axis Orientation**			**Search Pass 1**	**Search Pass 2**
	Bearing	**Plunge**	**Dip**		
North	215°	0°	-20°	100m by 100m by 40m 24 to 36 samples	500m by 500m by 120m 12 to 24 samples
Mid	190°	0°	-20°	100m by 100m by 40m 24 to 36 samples	300m by 300m by 100m 12 to 24 samples
South	250°	-20°	0°	100m by 100m by 40m 24 to 36 samples	500m by 500m by 200m 12 to 24 samples

Alaskite Sub-Dominant Mineralisation

A total of three estimation search passes were used to estimate U_3O_8 ppm for the ASD mineralisation domains. The first search pass was set to the range of the variogram in each direction, with the exception of the mid domain, where the search radii were reduced to approximately half that of the variogram (and similar to the other domains) to avoid over-smoothing of the grades. The first search utilised a minimum of 3 and a maximum of 24 samples. The second search pass was set to double that of the first pass and the minimum number of samples was reduced to two. For the third search pass the search radii was increased by a factor of 10, utilising a minimum of two samples and a maximum of 24. For each pass, no more than 10 samples from one drillhole could be used. Block discretisation was set to 10 (E) by 10 (N) by 4 (Z). All estimations were completed at the parent block scale. A full list of the estimation parameters used to interpolate grades into the ASD domains is presented in Table 14.12.

Table 14.12 Alaskite Sub-Dominant Mineralisation – OK Sample Search Parameters						
Domain	**Axis Orientation**			**Search Pass 1**	**Search Pass 2**	**Search Pass 2**
	Bearing	**Plunge**	**Dip**			
North	220°	0°	-10°	170m by 95m by 8.5m 3 to 24 samples	170m by 95m by 8.5m 12 to 24 samples	1700m by 950m by 85m 2 to 24 samples
Mid	140°	0°	-10°	200m by 140m by 4.3m 3 to 24 samples	200m by 140m by 4.3m 2 to 24 samples	2000m by 1400m by 43 m 2 to 24 samples
South	240°	-20°	-20°	200m by 150m by 6.5m 3 to 24 samples	200m by 150m by 6.5m 2 to 24 samples	2000m by 1500m by 65m 2 to 24 samples

Validation

A detailed visual and statistical review of the final estimate was conducted, including:

- Visual and graphical comparison of the input composites data with the block grade estimates in various cross section views and in plan. Figure 14.10 shows an example of the validation plots for the North domain

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- A comparison of the block model whole block estimate versus the mean of the composited dataset). Overall, the comparisons obtained were all within acceptable thresholds.

Overall, the grade estimates showed a good reproduction of the composite datasets with internal grade zonation domains being appropriately delineated.

14.1.9 Recoverable Resources

In order to calculate the recoverable resources for Etango, Uniform Conditioning (UC) was applied using Isatis (AD mineralisation) and an in-house software package in Datamine Studio 3 for the ASD mineralisation. The in-house UC algorithm has been benchmarked against the Isatis UC with favourable results.

Uniform conditioning of the OK estimate for the AD mineralisation domains has been undertaken using the following lower cut-off grades (ppm U_3O_8): 50, 75, 100, 125, 150, 175, 200, 250, 300 and 500ppm U_3O_8. Support correction has been undertaken on the basis of an SMU dimension of 6.25mE x 12.5mN x 4mRL. The information effect, on the basis of 6.25m E x 12.5m N x 4m RL grade control spacing for the AD mineralisation has also been modelled into the support correction and was found to have a minor effect. No tonnage corrections (exclusion of very small tonnages per block) have been made to the estimates.

Local uniform conditioning was performed on both the ASD and AD domains. A support correction using an SMU of 6.25mE x 12.5mN x 4mRL was conducted. The information effect for the ASD mineralisation was set at 5m E by 5m N by 4m RL, reflecting the likely blasthole spacing during mining.

Validation

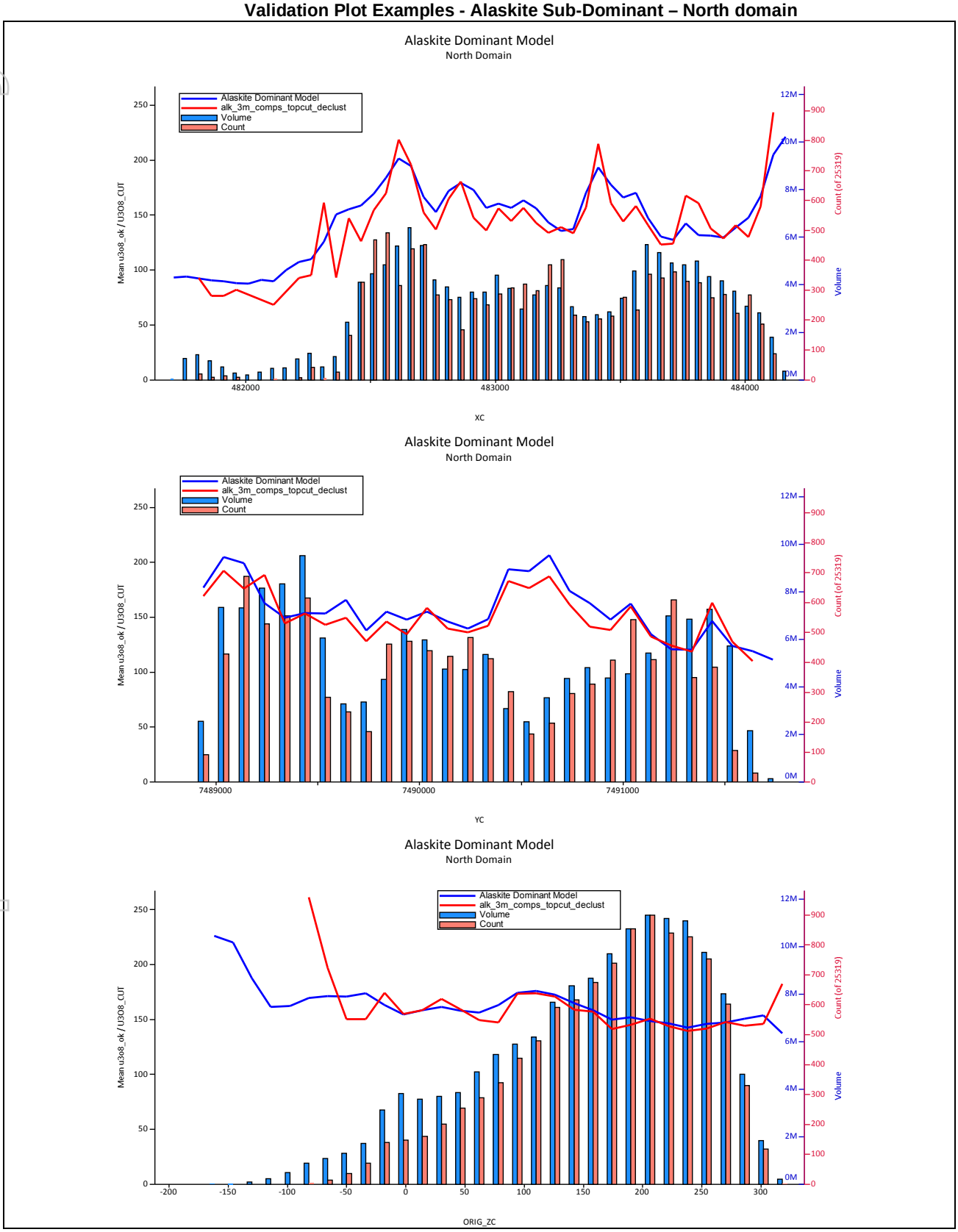
The following validation was completed on the UC and LUC models:

- Checks that the UC model at zero cut-off grade reflects the OK model
- Checks that the proportions of blocks are no smaller than a single SMU as a fraction of the panel size; and
- Checks that the LUC and UC models show the same tonnage-grade curve for selected test blocks.

Optiro is satisfied that the November 2015 model fairly reflects the input grades that that no artefacts have been introduced via either the UC or the LUC post-processing steps.

Figure 14.10

Validation Plot Examples - Alaskite Sub-Dominant – North domain



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14.1.10 Bulk Density

The bulk density values used for the resource model were based upon the data analysed in Section 14.1.3. A value of 2.64t/m³ was used for all mineralised material, both within the modelled alaskite bodies and metasediments.

Based upon the available core density measurements, the effect of weathering on the bulk density of the profile is considered to be minor and no change was applied to the bulk density of the different lithologies based upon the weathering profile.

14.1.11 Etango Resource Reporting and Classification

Introduction

The resource estimate for the Etango Project has been categorised in accordance with the criteria laid out in the Canadian National Instrument 43-101 (incorporating the CIM Guidelines, 2007) and the 2012 JORC Code. Previous Resource Estimates were classified in accordance with the 2004 JORC Code. Effectively, there are two main differences between the two Codes from a classification viewpoint:

- The 2004 Code requires geological or grade continuity for Indicated Resources, whereas the 2012 Code requires geological **and** grade continuity
- The 2012 Code states that there should be a reasonable expectation that Inferred Resources will eventually be upgraded to Indicated or Measured Resources.

The 2015 classification has attempted to follow the criteria applied by Coffey in its 2010 resource estimate (which was based on the 2004 JORC Code) as closely as possible, since there has been almost no change to the database used for the estimation between 2010 and 2015. It is important to note that the Coffey classification from 2010 could not be applied directly to the 2015 estimate as categories were associated with individual wire framed lenses, and the 2015 domain shapes were in different places, using a different approach (categorical modelling) and being generated at a different cut-off (50ppm in 2015 versus 75ppm in 2010).

A combination of Measured, Indicated and Inferred Resources have been defined in the November 2015 Mineral Resource. The classification criteria are as outlined below.

Measured Resource

A Measured category was assigned based on blocks estimated in pass one or two of the estimate, for mineralised zones with a strong geological understanding, consistent mineralisation shape and grade tenor, good OK estimation quality (as defined by a slope of regression >0.9), and a nominal 25m x 25 m or 25m x 50m drill hole coverage.

Indicated Resource

An Indicated category was assigned based on blocks estimated in pass one or two of the estimate, for mineralised zones with a strong geological understanding, consistent mineralisation shape and grade tenor, moderate OK estimation quality (as defined by a slope of regression broadly between 0.3 and 0.9) and a nominal 50m x 50m to 50m x 100m drill hole coverage.

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Inferred Resource

An Inferred category was applied to all mineralisation zones which were not classified as Indicated or Measured. All of the ASD mineralisation, regardless of whether it is encapsulated in the 3D classification shapes, has been classified as Inferred. This is due to the key assumption that these lenses are small and discontinuous.

Optiro did not apply a 'potential' or 'unclassified' category as the model was trimmed, removing extrapolated blocks using an 'unclassify' solid (Figure 14.11).

Overall, the 2015 Resource Classification is broadly similar to the 2010 classification, with the exception of the Measured Resources. Although a number of separate Measured Resource zones were defined in 2015, the volume is smaller than that defined by Coffey in 2010, despite the 2015 criteria being more relaxed. A plan view of the classified block model is presented in Figure 14.12. Example cross sections demonstrating the classification philosophy are presented in Figure 14.13 and Figure 14.14.

Figure 14.11

Example section view of the 'unclassify' solid

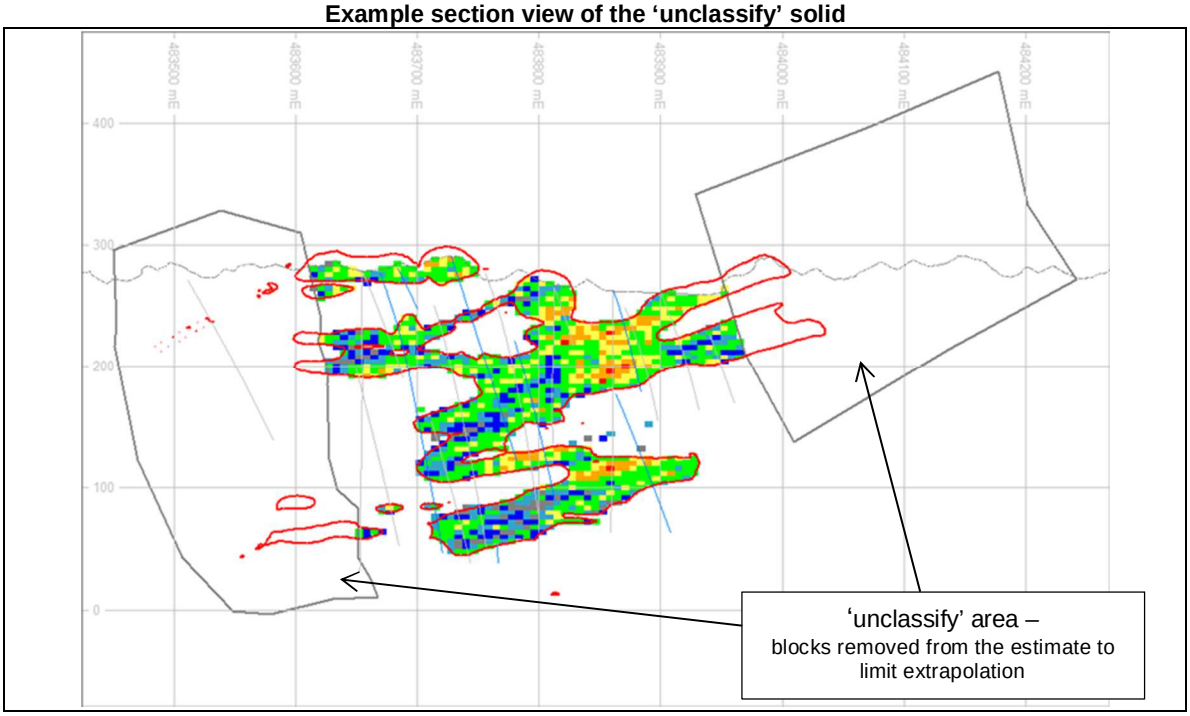

Figure 14.12
Plan View of the Classified Block Model

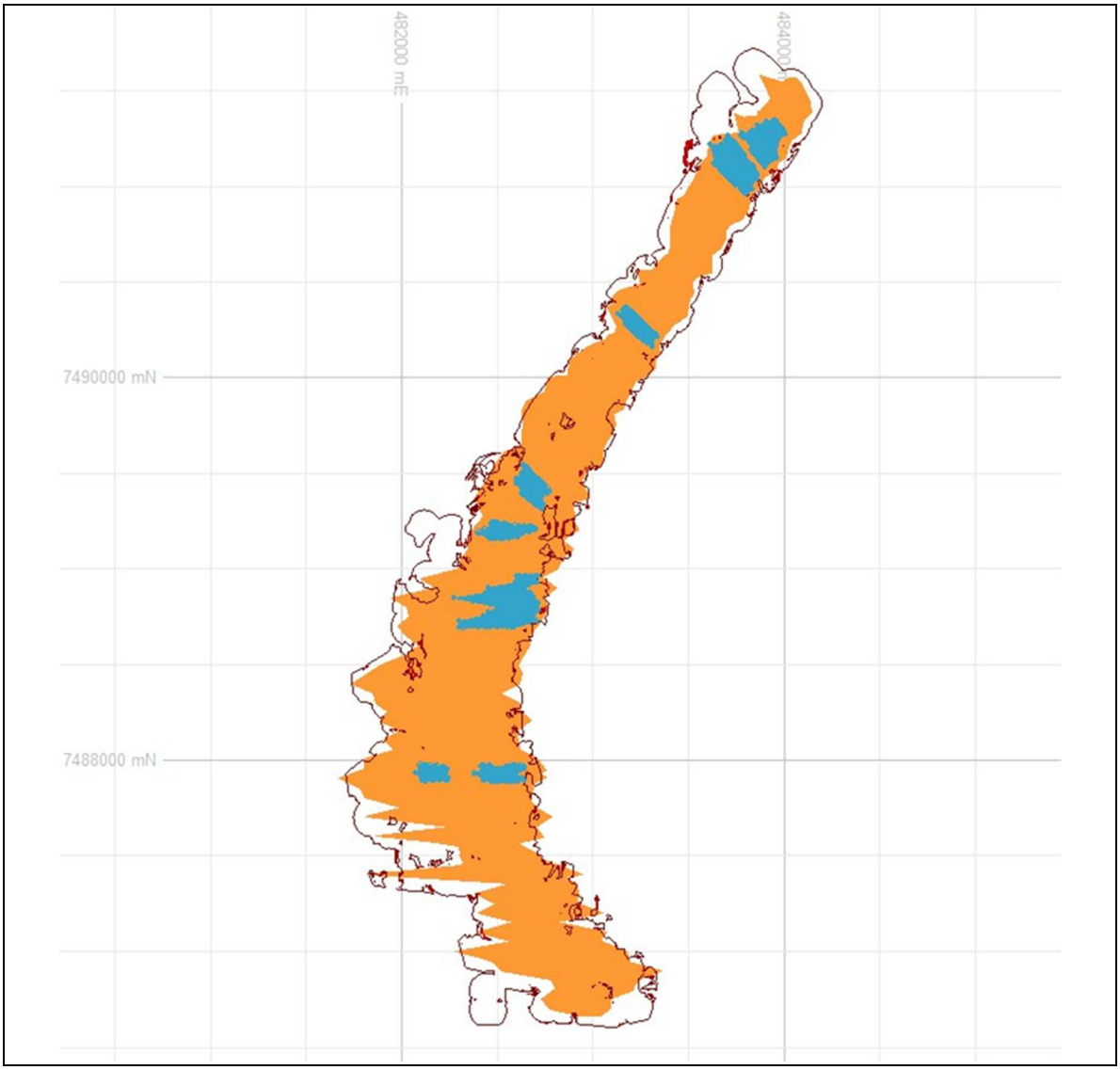

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Figure 14.13

Section View along 7,489,180 mN demonstrating the classification strategy

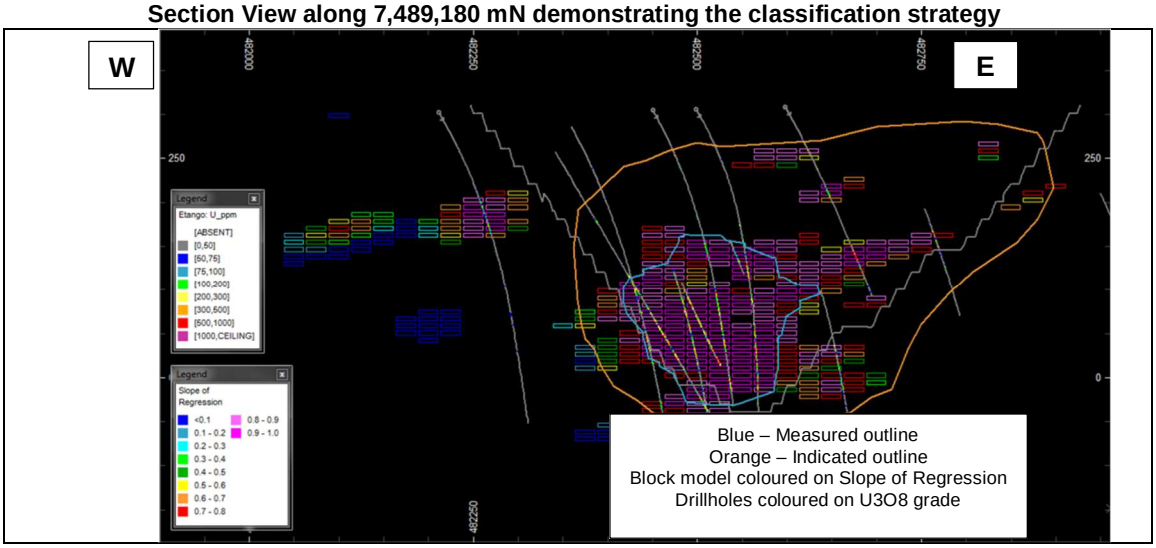
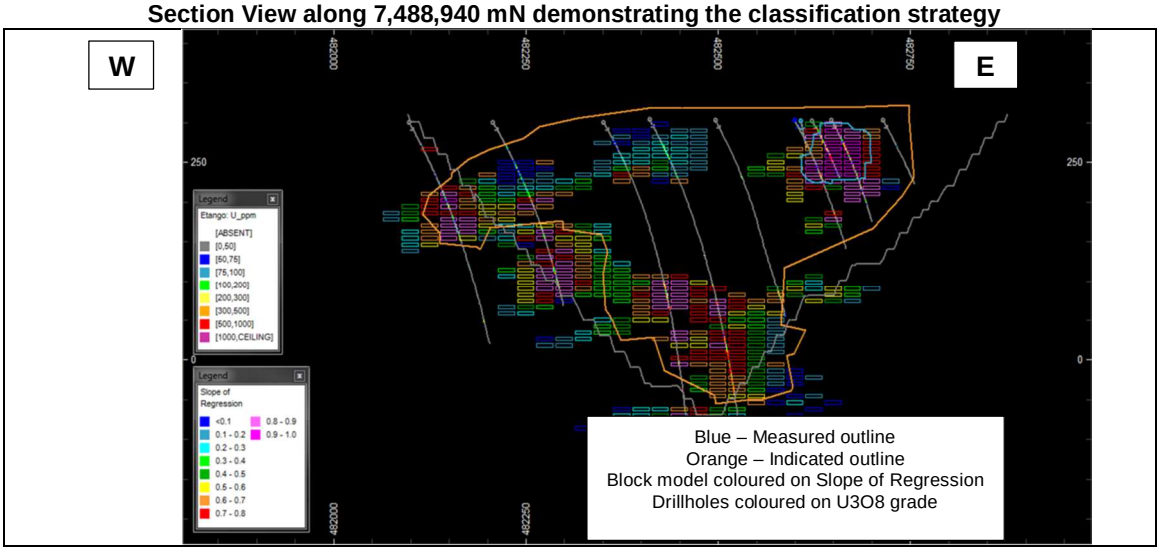


Figure 14.14

Section View along 7,488,940 mN demonstrating the classification strategy



14.1.12 Etango Grade Tonnage Reporting

The consolidated LUC model was subject to pit optimisation using a Lerchs-Grossmann algorithm. This optimisation used a uranium price of $75/lb and mining costs which were revised in mid-2015. A mining study suggested that, given the $75 price and revised scheduling parameters, a marginal cut-off of 55ppm U_3O_8 was achievable. The mineralisation was therefore reported above this cut-off grade inside the optimal pit, which was predicated on all resource categories. The results are presented in Table 14.13.

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Table 14.13 Etango Deposit, Etango Project, Namibia – November 2015 Resource Estimate			
Resource Category	**Tonnes (Mt)**	**Grade (U_3O_8 ppm)**	**Contained U_3O_8 (Mlbs)**
Measured	33.7	194	14.4
Indicated	362.0	188	150.2
Inferred	144.5	196	62.5
Total	540.2	191	227.1
Note: Figures have been rounded. Conversion of lb to kg = x 2.20462 LUC Model reported using a Bulk Density of 2.64t/m^3 Constrained within the resource pit shell, reported using a 55ppm U_3O_8 cut-off			

14.1.13 Etango Summary, Conclusions and Recommendations

The November 2015 Resource update represents a significant change in the Etango resource endowment. The change in methodology to a more geologically-constrained approach, together with the lower cut-off grade and the use of a recoverable methodology to reflect the planned grade control method, have all contributed to a more robust and appropriate estimate to that used for the DFS.

The following limitations of the modelling approach are noted:

- While the OK model has been reported for a range of cut off grades, it should be noted that the OK model is valid for a limited range of cut offs for which the model was designed (considered to be in the practical range of 50ppm to 100ppm U_3O_8).
- The estimation approach, while common and considered to be best practice for uranium deposits, is nonetheless reliant upon a number of theoretical assumptions and the underlying mathematical theory is complex.

14.2 ONDJAMBA AND HYENA MINERAL RESOURCES

Coffey Mining was requested by Bannerman to undertake a maiden resource estimation study on the Ondjamba and Hyena deposits, which are also located within EPL 3345. The Ondjamba deposit is located approximately 1km along strike to the southeast of the Etango deposit, while the Hyena deposit is located approximately 1km to the south of the Etango deposit, Figure 9.1.

The resource estimation study included a review of the available drill hole database information, geological models, statistical and geostatistical constraints, grade estimation, and classification of the estimate in accordance to the criteria laid out in the Instrument.

14.2.1 Resource Database

<u>Ondjamba</u>

The drill hole database consists of 125 RC drill holes totalling 22,231m.

The drill holes were drilled typically at 60° to the north (UTM grid) with a drill spacing ranging from 100m x 100m to 200m x 100m.

A combination of chemical assaying (11,609 samples - 58% of the total) and factored radiometric data (8252 1m composites – 42% of the total) were used for the estimation. The

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radiometric data was factored such that the mean of the eU_3O_8 data matched that of the chemical data. Within the mineralisation domains, 3220 chemical (88%) and 422 radiometric (12%) assays were used.

Hyena

The drill hole database consists of 148 RC and 4 diamond drill holes totalling 15,262m. Of those drill holes, 47 RC and 3 diamond drill holes totalling 9061m were directly used for the deposit model.

The drill holes were drilled typically at 60° to the north (UTM grid) or vertically with a drill spacing ranging from 50m x 25m to 200m x 100m.

A combination of chemical assaying (6803 samples - 67% of the total) and factored radiometric data (3311 1m composites – 33% of the total) were used for the estimation. Within the mineralisation domains 1616 chemical (99%) and 20 radiometric (1%) assays were used.

14.2.2 Geological Modelling

To establish appropriate grade continuity, the mineralisation models for the Ondjamba and Hyena deposits were based on nominal 75ppm U_3O_8 mineralisation haloes.

The mineralisation constraints were generated based on sectional interpretation and 3D analyses of the available drilling data. The vast majority of the uranium mineralisation is associated with the alaskite bodies and follows the trends of the alaskite contacts. The alaskite contacts were considered at the time of modelling and used to guide modelling of the mineralisation shapes.

The mineralisation boundaries within the alaskites bodies were often extended to the alaskite contacts for up to 3m, even if these intervals were not mineralised above the nominal 75ppm U_3O_8 cut-off. Mineralised zones which did not have more than two drill hole intersections on two consecutive sections and for which a strong geological continuity could not be established, were typically not estimated.

Ondjamba

The mineralised zones at Ondjamba (Figure 14.15) were modelled as 12 distinct zones (ranging from 1m to 70m thick, averaging 11m thick) with a SW-NE trend. The zones dip from -30° to -40° to the south-east (Figure 14.16). Individual zones were modelled from 150m to 1,750m long. Figure 14.16 is a typical sectional interpretation.

Hyena

The mineralised zones at Hyena (Figure 14.17) were modelled as 19 distinct zones in four separate domains, (ranging from 2m to 63m thick, averaging 12.6m thick) with a west-east trend. Three domains exhibit a southerly dip from -30° to -40° to the south, with domain three exhibiting a near vertical west-east trend (Figure 14.18). Individual zones were modelled from 150m to 1750m long. Figure 14.18 is a typical sectional interpretation.

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Figure 14.15

Ondjamba Mineralised Zones and Drilling

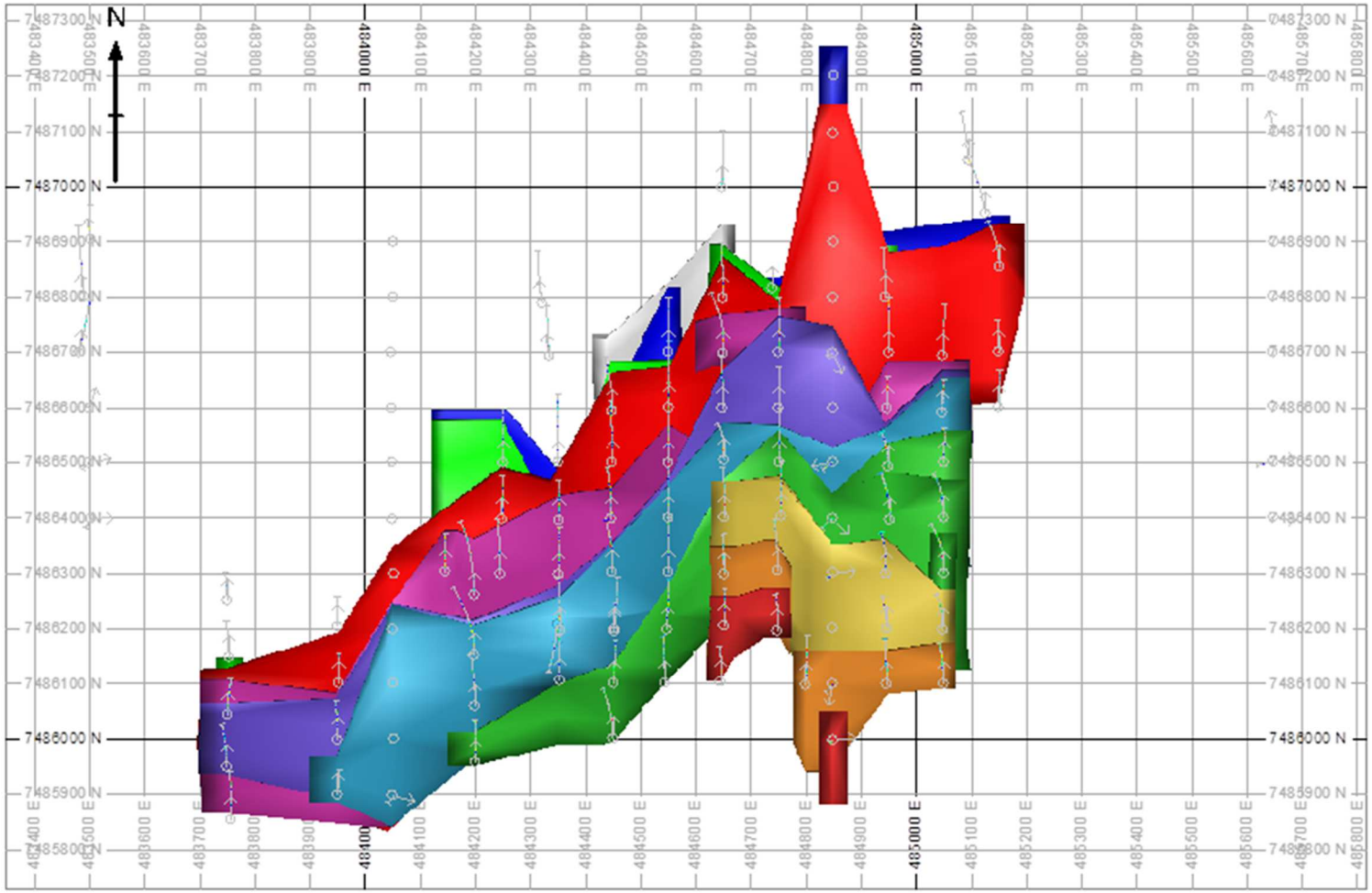

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Figure 14.16

Ondjamba South-North Sectional Interpretation (484,850mE)

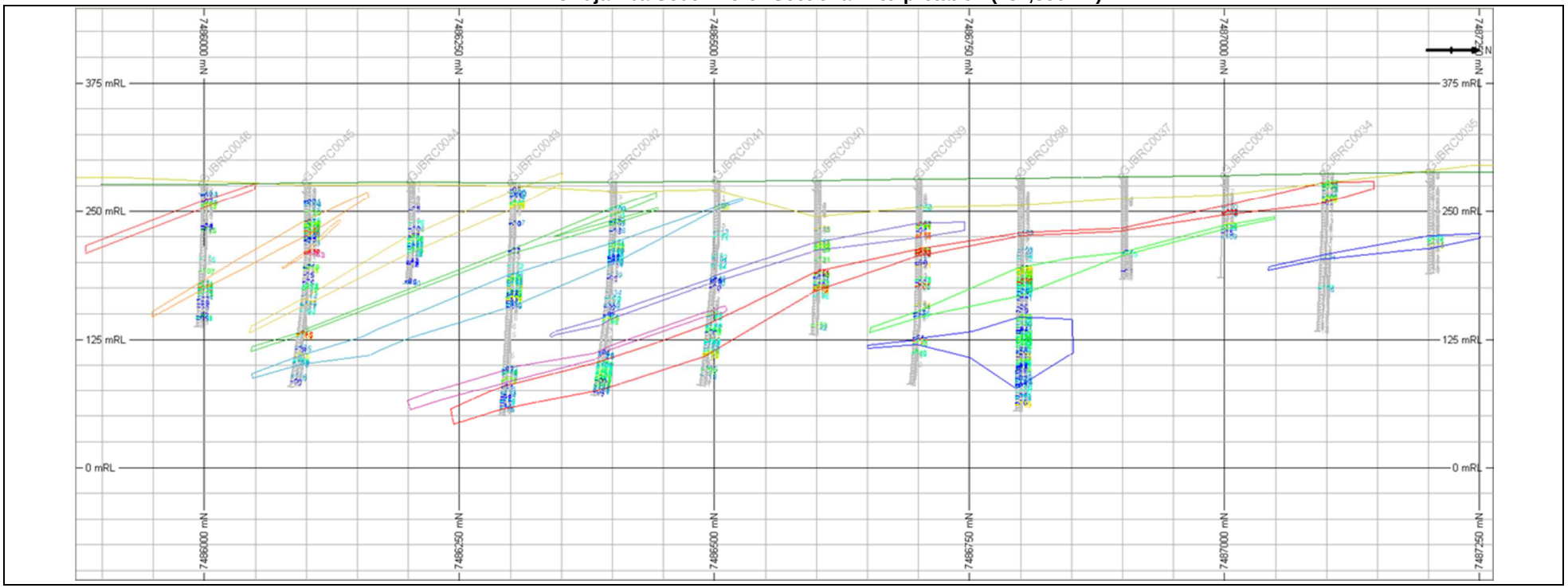

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Figure 14.17
Hyena Mineralised Zones and Drilling

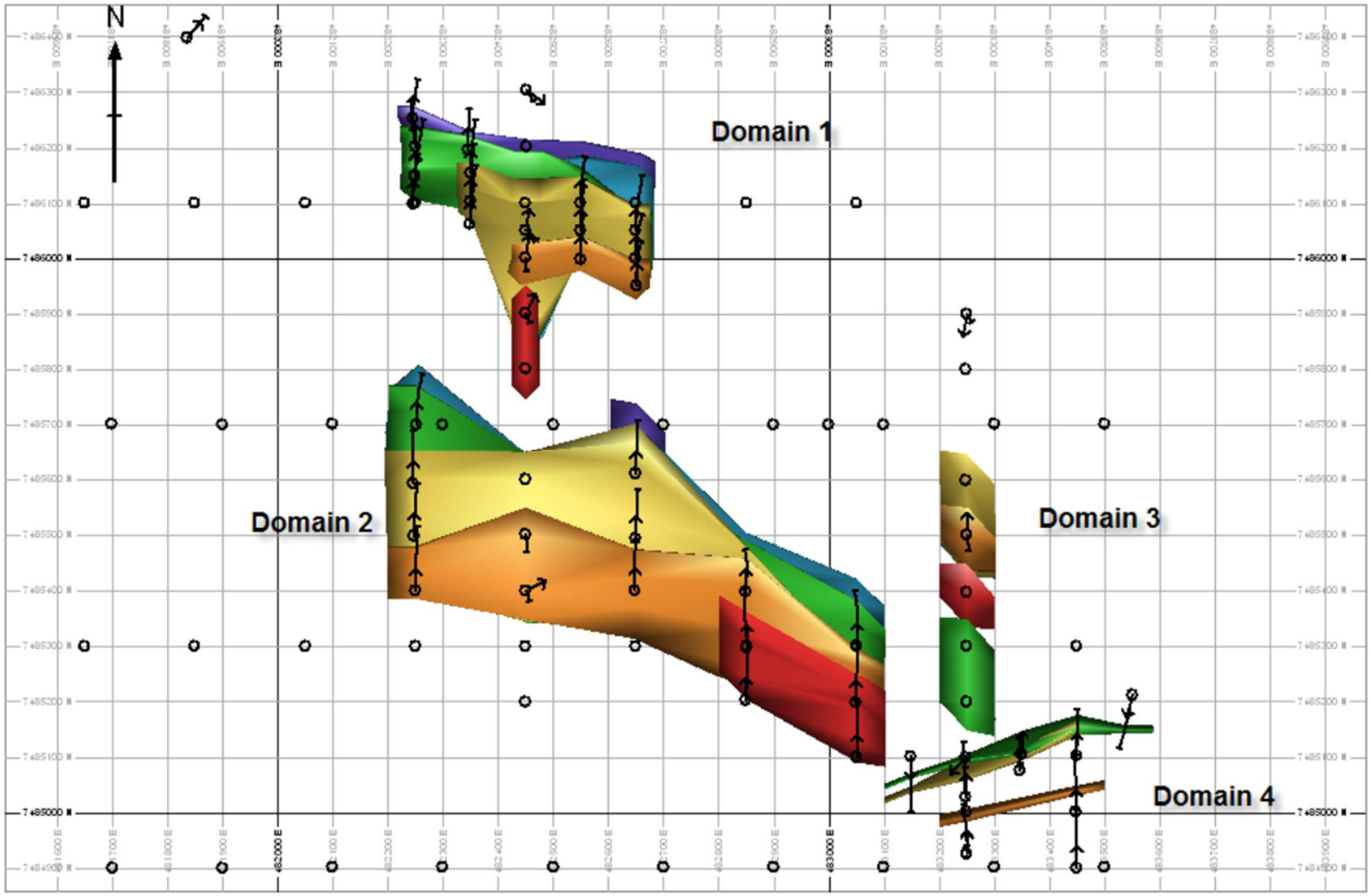

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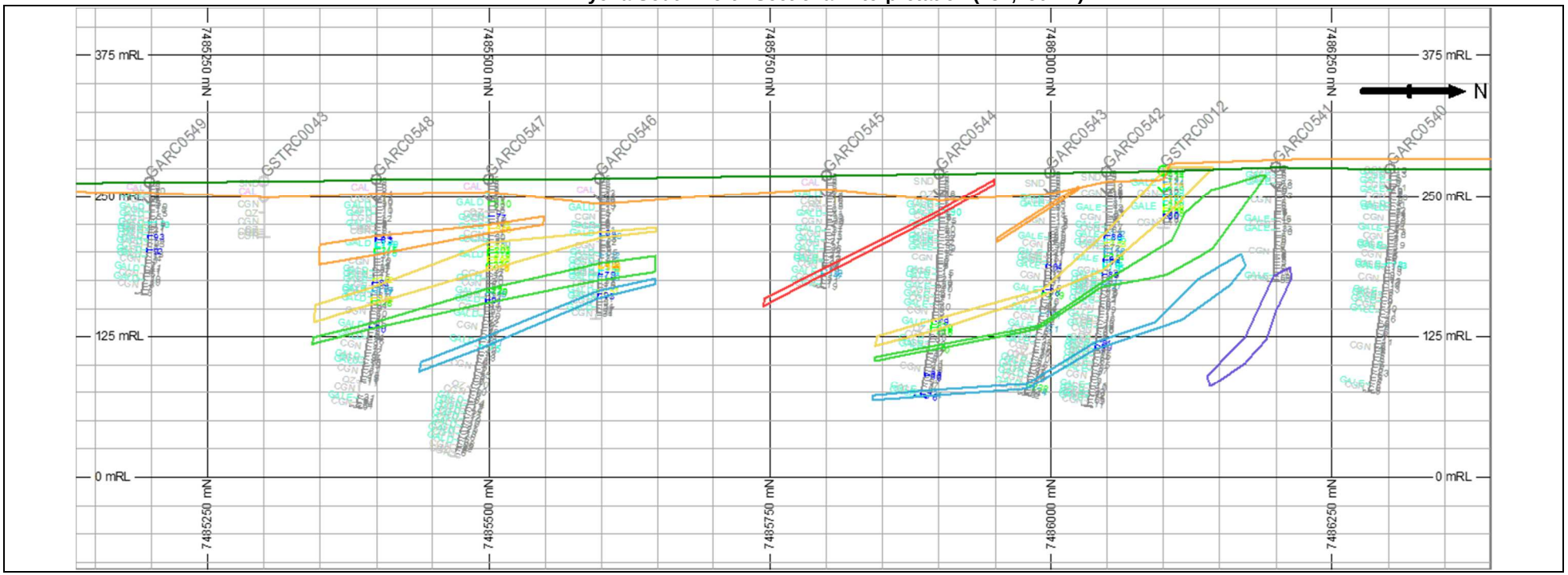
Figure 14.18

Hyena South-North Sectional Interpretation (482,450mE)



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14.2.3 Grade Estimation

The samples captured within the mineralisation shapes were composited to a regular 3m downhole composite length. Based on the 3m composite data, statistical and geostatistical investigations were completed to derive appropriate estimation parameters such as top cuts, variogram model parameters, and search ranges etc.

A single top cut of 700ppm U_3O_8 was applied to the 3m composites for all Ondjamba zones prior to estimation. The effect of the top cuts was to decrease the mean grade of the 3m composites by <1%.

At Hyena, only Domain 3 exhibited any significant high grade tail in the population distributions, therefore a top cut of 850ppm U_3O_8 was applied to the 3m composites for Hyena Domains 1, 2 and 4, and a top cut of 1250ppm was applied to Domain 3 prior to estimation. The effect of the top cuts was to decrease the mean grade of the 3m composites by <1% for Domains 1, 2 and 4 and 22% for Domain 3.

3D block models were constructed for the purposes of grade estimation for each deposit. A parent block size of 25mN by 25mE by 10mRL was selected as the appropriate block size based on the current average data spacing and the geostatistical investigations completed. Sub-celling has been limited to 3.125mN by 3.125mE by 1.25mRL in order to achieve appropriate volume definition of the mineralisation.

OK was chosen as the appropriate method for estimating grade, based upon the top cut 3m U_3O_8 composites. Due to an insufficient number of assays available to generate interpretable correlograms, variogram (correlogram) parameters for Hyena were derived from the Etango deposit models and applied to all zones individually with hard assay boundaries. Correlograms for the combined zones assays were derived for the Ondjamba mineralisation and applied to the individual zones with hard boundaries (each zone was only estimated using assays within the same zone). In all cases search axes of 120mx80mx40m for Hyena and 240mx160mx80m for Ondjamba, were orientated into the dip plane of the mineralisation. Second and third search passes at 2x and 3x multipliers were applied. The bulk of the blocks filled within the first and second search passes.

14.2.4 Ondjamba and Hyena Resources

Categorisation of the grade estimate was undertaken according to the criteria laid out in NI43-101. The Resources were classified as Inferred using the criteria determined during the validation of the grade estimates, with detailed consideration of the NI43-101 (CIM) guidelines.

Blocks were classified as Inferred considering issues such as geological and grade continuity and within a nominal 100m x 100m drill hole spacing. Blocks not classified as Inferred were left as Unclassified. Two zones at Ondjamba and five zones at Hyena were not classified where drill hole spacing became too broad. A default in situ bulk density value of 2.64t/m^3 was used when reporting the resource. No mining has occurred at either of the deposits.

The Mineral Resource for the Ondjamba and Hyena deposits reported above various cut-offs are summarised below (Table 14.14 and Table 14.15). Based upon the style of modelling undertaken and the understood economics of the deposit, it is recommended that the resource be reported above 100ppm U_3O_8.

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Table 14.14 Ondjamba Deposit, Etango Project, Namibia – October 2010 Resource Estimate			
Cut-off Grade	**Tonnes Above Cut-off (Mt)**	**U_3O_8 (ppm)**	**Contained U_3O_8 (Mlb)**
Inferred			
75	86.6	165	31.5
100	85.1	166	31.3
125	73.5	174	28.3
150	50.8	190	21.3
Note: Figures have been rounded Reported at Various Cut-offs Using a Bulk Density of 2.64 t/m^3 Ordinary Kriged Estimate Based Upon 3m Cut U_3O_8 Composites Block Dimensions of 25m NS by 25m EW by 10m RL			

Table 14.15 Hyena Deposit, Etango Project, Namibia – October 2010 Resource Estimate			
Cut-off Grade	**Tonnes Above Cut-off (Mt)**	**U_3O_8 (ppm)**	**Contained U_3O_8 (Mlb)**
Inferred			
75	33.8	165	12.3
100	33.6	166	12.3
125	30.1	172	11.4
150	20.6	186	8.4
Note: Figures have been rounded Reported at Various Cut-offs Using a Bulk Density of 2.64 t/m^3 Ordinary Kriged Estimate Based Upon 3m Cut U_3O_8 Composites Block Dimensions of 25m NS by 25m EW by 10m RL			

14.3 ETANGO PROJECT COMBINED MINERAL RESOURCES

The combined Etango Project November 2015 Mineral Resource estimate includes the Etango Mineral Resource, reported within an optimal resource pit at a cut-off grade of 55ppm U_3O_8, and the Ondjamba and Hyena Mineral Resource estimates (unchanged since October 2010), reported at a 100ppm U_3O_8. The final November 2015 estimate comprises Measured plus Indicated resources of 395.7Mt at 189ppm for 164.6Mlb of contained U_3O_8, and Inferred resources of 263.2Mt at 182ppm for 106.1Mlb of contained U_3O_8.

The Etango Mineral Resource estimate has been prepared in accordance with the Australian JORC Code 2012 guidelines and Canadian National Instrument 43-101 by Optiro. The Ondjamba and Hyena Mineral Resource estimates have been classified and reported in accordance with the Australian JORC Code 2004 guidelines and Canadian National Instrument 43-101 and remain unchanged since the previous October 2010 estimate.

The combined Mineral Resource estimate is tabulated below (Table 14.16) by individual deposit area.

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Table 14.16
Etango Project Mineral Resource Estimate November 2015

		Measured			Indicated			Inferred		
Deposit	**Cut-off Grade (U3O8 ppm)**	**Tonnes (Mt)**	**Grade (U_3O_8 ppm)**	**In situ U_3O_8 (Mlbs)**	**Tonnes (Mt)**	**Grade (U_3O_8 ppm)**	**In situ U_3O_8 (Mlbs)**	**Tonnes (Mt)**	**Grade (U_3O_8 ppm)**	**In situ U_3O_8 (Mlbs)**
Etango[1]	55	33.7	194	14.4	362	188	150.2	144.5	196	62.5
Ondjamba[2]	100							85.1	166	31.3
Hyena[2]	100							33.6	166	12.3
Total		**33.7**	**194**	**14.4**	**362**	**188**	**150.2**	**263.2**	**182**	**106.1**

Note 1: The Etango estimate has been reported in accordance with JORC 2012 and is constrained within the November 2015 optimal resource pit shell.

Note 2: Ondjamba and Hyena remain unchanged from the October 2010 estimate and have therefore been reported in accordance with JORC 2004. The resource has not been constrained within a pit shell.

Note 3: The figures may not add due to rounding. A bulk density of 2.64 t/m^3 has been used.

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15 MINERAL RESERVE ESTIMATES

15.1 INTRODUCTION

This section provides a summary of the methodology used and the economic criteria applied to derive at the Mineral Reserves as tabulated in this section.

Further detail on the economic criteria is provided in Section 16 through to Section 22.

The Mineral Reserves were determined as part of the OS that was completed in 2015.

The OS was based on an update of the Etango Deposit Mineral Resources as of November 2015.

The OS was based on mine planning work that was undertaken utilising the Measured and Indicated Resources only.

15.2 CIM DEFINITION OF MINERAL RESERVES

The CIM Standing Committee on Reserve Definitions, which forms part of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), developed the 'CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines', which was updated on 27 November 2010.

These guidelines state the following:

15.2.1 Mineral Reserve

Mineral Reserves are subdivided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve.

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, and economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

Mineral Reserves are those parts of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Qualified Person(s) making the estimates, is the basis of an economically viable project after taking account of all relevant processing, metallurgical, economic, marketing, legal, environment, socio-economic and government factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the treatment plant or equivalent facility. The term ‘Mineral Reserve’ need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

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15.2.2 Probable Mineral Reserve

A 'Probable Mineral Reserve' is the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

15.2.3 Proven Mineral Reserve

A 'Proven Mineral Reserve' is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

Application of the Proven Mineral Reserve category implies that the Qualified Person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect potential economic viability.

15.3 ECONOMIC CRITERIA

The term 'Economic Criteria' is defined to include mining, processing, and metallurgical, economic, marketing, legal, environmental, social and governmental considerations.

The sources for the Economic Criteria are summarised in Table 15.1.

Table 15.1
Etango Uranium Project – Source Economic Criteria
Used for Mineral Reserve Determination

Item	Source
Mining Cost	Bannerman
Metallurgical Aspects	AMEC, Bateman Engineering
Processing Cost	AMEC, Bateman Engineering
Residue Storage Facility	SLR/Metago
Commodity Price	Bannerman
General and Administration Cost	Amec Foster Wheeler, Bannerman
Social and Environmental	Bannerman, A. Speiser Environmental Consultants
Mine Closure Cost	Bannerman
Government	Bannerman
Hydrology and Hydrogeology	Aquaterra
Geotechnical	Mine Technics
Site Water Balance	SLR/Metago
Mining Dilution and Recovery	Bannerman
Discount Rate	Bannerman

Unless otherwise stated all costs are quoted in US$.

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The Mineral Reserves as determined for the Project were based on the Economic Criteria as summarised in Table 15.2.

Table 15.2
Etango Uranium Project – Summary Economic Criteria Used for Mineral Reserve Determination

Item	Unit	Value
Crusher Feed	Mtpa	20
Uranium Price	$/lb	75
Royalty (Government & Third Party)	%	4.5
Processing Cost (inclusive of General & Administration)	$/t ore	6.79
Processing Recovery	%	87
Average Mining Cost	$/t mined	1.69
Mining Dulution[2]	%	0%
Mining Recovery[3]	%	100%
Overall Pit Wall Slope Angle (inclusive of a ramp system)	Degrees	43 – 51
Initial Project Capital	M$	793
Sustaining Capital	M$	282
Closure Costs	M$	32.5
Discount Rate	%	8

[2] Included in the Mineral Resource model.
[3] Included in the Mineral Resource model.

The mining costs were based on an owner mining scenario, assuming a leased mining fleet. Furthermore, it was assumed that, based on the geotechnical information available, 100% of the material will require blasting.

15.4 MINERAL RESERVE SUMMARY

This Reserve estimate has been determined and reported in accordance with Canadian National Instrument 43-101, 'Standards of Disclosure for Mineral Projects' of June 2011 (the Instrument) and the classifications adopted by CIM Council in November 2010.

The Mineral Reserve was based on a cut-off of 55ppm U_3O_8 and was determined as of 1 December 2015. As these have been no production, this reserve remains current as of 31 December 2015.

All stated reserves are completely included within the Resources as shown in Table 15.3 provides a summary of the Mineral Reserve determined for the Project.

Table 15.3
Etango Uranium Project – Mineral Reserves Summary

Classification	Ore Reserves	
	Tonnes (Mt)	Grade U_3O_8 (ppm)
Proved	32.3	196
Probable	271.0	194
Total	**303.3**	**195**

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The reported Mineral Reserves have been compiled by Mr Leon Fouché. Mr Fouché is a Fellow of the Australian Institute of Mining and Metallurgy and an employee of Bannerman. He has sufficient experience, relevant to the style of mineralisation and type of deposit under consideration and to the activity he is undertaking, to qualify as a Qualified Person as defined in the CIM Definition Standards, as well as a Competent Person as defined by the 'Australasian Code for Reporting of Mineral Resources and Ore Reserves' of December 2012 ('JORC Code') as prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, the Australian Institute of Geoscientists and the Minerals Council of Australia.

15.5 DISCUSSION

A number of factors may materially affect the reserve estimates. These factors include, but are not limited to, environmental, permitting, legal, title tax, socio-economical, marketing and political, economic or other factors. In the case of the Project, most of these factors are well understood and have been described in other sections of this report.

Nevertheless, it is noted that the economic parameters that have an impact on the revenue stream of the Project, have the largest impact on the Project economic viability. The three parameters identified that adversely affect the revenue stream of the Project the most are listed below:

15.5.1 Uranium Price

The current long term contract price for U_3O_8 is around $44/lb. Numerous market analysts, ranging from industry organisations, banking institutions, specialist uranium pricing reporting firms and producers currently expect the fundamentals of the uranium market to improve significantly, with uranium spot price projections ranging from $65/lb to $80/lb.

15.5.2 Uranium Grade

The resource delineation at the Project has been undertaken over a number of years and it is based on extensive RC drilling, which resulted in a good understanding of the mineralisation style and grade tenor. As such, it is believed that the uranium grade at the Project is well understood.

15.5.3 Metallurgical Recoveries

Metallurgical recoveries have been determined by laboratory test work. Subsequent to the DFS large section column leach testing has been undertaken which have supported the DFS assumptions. It is believed that, with the current available metallurgical data, the metallurgical recoveries are sufficiently well understood for reporting of mineral reserves.

It is the opinion of Bannerman that, excepting the parameters discussed above, there are no other material factors that may affect the mineral reserve estimates.

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16 MINING METHODS

16.1 INTRODUCTION

The mining study that was undertaken as part of the OS covered built upon an existing 2012 DFS. Following a review of the DFS work a number of the DFS aspects were left unchanged including:

- Geotechnical and hydrological assessment
- Open pit bench height
- Drill & blast parameters
- Equipment selection.

The OS revisited and updated the following aspects:

- Mine planning including pit optimisation, final pit design, pit staging, dump design, mine production scheduling and geotechnical review of the updated pit design
- Mine operating and capital cost estimation to an order of accuracy of ±15%.

The Study was based on:

- A uranium price of $75/lb
- The 2015 Localised Uniform Conditioning block model (LUC Model) of the combined Onkelo, Oshiveli, and Anomaly A (herein Etango) prospects
- Geotechnical assessment of 26 geotechnical holes and surface mapping
- On/Off Heap Leach process with a combined recovery of 87% and a capacity of 20Mtpa
- Q3 2015 market price for:
  - Explosives, fuel, lubricants, mobile equipment and earthmoving tyres
  - Vendor-provided services for mobile maintenance and 'down hole' explosives.
- Traditional open pit truck and backhoe operation.

16.2 MINE OPERATIONS

The mineralisation at Etango stretches over a strike length of around 6km, is up to 1km wide and extends to a known depth of approximately 400m below surface, with the orebody outcropping on surface in some areas. The deposit is, therefore, conducive to an open pit mining method rather than an underground method.

The overall operating strategy for the Etango open pit will focus on delivering a high tonnage, low cost operation. The grade of the ore is low, thus the mine operation needs to be cost conscious with a high degree of certainty in production capacity to meet the required ore processing rate and support the underlying cost structure. The mine operation strategy is to maximise NPV by maximising the available grade of the ore processed and minimising the waste movement.

To ensure business outcomes, the 2012 DFS has focused on a predictable, high performance mine with a conventional approach to mining. The manning strategy of Etango

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reflects this focus. In the long term the mine will need to sustain operations with local talent. Allowances have been made for the initial years of the mine to employ expatriate personnel, after this period it was assumed that national employees will replace the majority of the expatriates.

A vigilant focus on quality is required due to the geometry of the alaskite intrusions (source of the recoverable mineralisation) and the high total material movement of 100Mtpa. Considerations for backhoe excavation, downhole gamma logging, blast movement monitoring, RC grade control calibration, and truck scanners feature as important tools to ensure tonnes delivered to the primary crusher are of the planned grade and dilution from the mine is minimised.

Operational and mechanical performance needs to be monitored, continuously improved, and accurately reported in real time. To this end, the OS includes dispatch systems to ensure implementation costs are accurately reflected. Manning for the system has been included to supervisory and senior engineering level in both condition monitoring and production dispatching.

To minimise the on-going cost of waste haulage, detailed dump-build simulations were undertaken to minimise the truck requirements to deliver tonnes. This comes at a cost of additional capital for haul road construction and dump maintenance.

16.2.1 Geotechnical and Hydrogeological Review

A geotechnical assessment to provide pit slope design parameters for the Etango project was completed by Coffey Mining to a DFS level during the 2012 DFS. The geotechnical parameters were used during the OS for updating the pit designs and subsequently reviewed by Mine Technics against the original Coffey recommendations. The 2012 DFS geotechnical analysis that informed the OS is discussed below.

The geotechnical data from which the geotechnical domains have been derived is based primarily on geotechnical logging of 26 oriented drill hole cores and surface structural mapping. The geotechnical data collection was undertaken by Bannerman staff geologists under the guidance of Coffey Mining. Geotechnical data collected from drill core has the following limitations:

- The data is heavily biased; the dominant sample direction (drill hole azimuth) is toward the east. There are only four drill holes which have westerly azimuths and these drill holes intersect the east wall of Anomaly A
- The majority of drill holes intersect the toe of the footwall (east wall) pit slope; the drill holes have been designed with a resource focus
- There is a paucity of data for the hanging wall of the deposit in the proposed location of the west wall of the pit (only two drill holes sampling the rock mass near the toe of the hanging wall pit slope.)

Uranium mineralisation on the Etango Project is associated with late-staged leucocratic granites referred to as Alaskites which are the principal host of the uranium mineralisation. The Alaskites intrude the host metasedimentary formation, dipping at a shallow angle (30°) toward the west. The fault model provided by Bannerman comprises 17 fault planes. Broken zones representing possible faults were identified from the cored geotechnical drilling. The fault planes generally dip at shallow to moderate angles toward the west and are interpreted to daylight on both the southeast and northeast walls.

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Stability analysis of the overall/inter-ramp slope geometry assumed partly de-watered slopes and depressurised batter slopes in the pit walls. The analysis suggests that the stability of the overall / inter-ramp slope is very sensitive to changes in the groundwater assumptions.

An examination of the GSHAP seismic hazard maps available on public domain established that Etango is in an area where only a very low level of seismic activity is expected. The seismic hazard maps suggest a peak particle acceleration (PPA) value in the range of 0.02g to 0.04g (0.2m/s² to 0.4m/s²) for a 10% probability of exceedance in a 50 year time period, representing a return period of 1-in-475 years. The seismic coefficient resulting from a magnitude 4.6 event to be applied in the open pit stability analysis is determined to be 0.01 (in software requiring horizontal accelerations with respect to g) or 0.1 (in software requiring horizontal accelerations with respect to m/s²).

A reliable material properties database has been developed, based primarily on laboratory test work which has been completed to appropriate international material testing standards.

There is overall uniformity in the rock mass properties with little difference between alaskite and host metasediments. The weathered rock mass is a 'poor' quality rock mass with a 'weak' intact rock strength, while the fresh rock mass is a 'good' quality rock mass with a 'strong' intact rock strength.

The Etango deposit has been divided into geotechnical domains based on discontinuity patterns (North Domains and South Domains), subdivided into weathering (weathered and fresh rock), and into design sectors based on pit wall orientation (North, East, South and West).

Assessment of batter slope geometry has been undertaken by examining the kinematics of potential structurally controlled failures and selection of a design batter slope angle to minimise under-cutting of daylighting structural planes. Stability of overall and inter-ramp slope geometries have been undertaken for the Etango domains using Rocscience software Slide (*Rocscience, 2002*).

The berm width design for the weathered and fresh rock is based on Modified Ritchie's Criterion and the Martin-Piteau method to provide rock fall catch protection and to provide sufficient catch width to retain a bulked failure volume, based on the interpreted controlling failure mechanism. The assessment of berm width using a FOS risk-based approach with Modified Ritchie's Criterion suggests that a berm width of 9.5m (determined using the Martin-Piteau assessment method) would be satisfactory for containing bulked material volumes arising from batter scale failures. The 9.5m berm width is appropriate for the proposed 24m batter height and with batter face angles of up to 70° for almost all design sectors. The berm width assessment for the southeast design sector of North Domain suggests a minimum berm width of 10.2m.

The recommended inter-ramp slope angle (IRSA) is calculated from the recommended batter height/batter angle/berm width configuration for each geotechnical domain.

For the weathered rock mass there is one slope design for all domains, comprising 55° batter face angles, 12m batter heights and 6m berm widths for an IRSA of 39.8° over an inter-ramp slope height (IRSH) of 20m.

For the fresh rock mass, there is one slope design for all domains. The slope design comprises 70° batter face angles, 24m batter heights and 9.5m berm widths for an IRSA of 52.8°. The slope should be de-coupled at every 5th berm with either a geotechnical berm (minimum width of 15m) or placement of the haul ramp, limiting the IRSH to 120m

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(vertically). The overall slope angle for the pit depth of 380m is calculated to be approximately 50.5°.

The recommended slope design for the west and east waste dumps comprises an overall slope angle of 30° for a maximum waste dump height of 100m, a batter slope angle of 35°, a lift height of 20m and berm width of 10m. An examination of the sensitivity of FOS on the water level was undertaken and it shows that FOS reduces with an increase in the water level and full friction angle. The FOS of the dumps increases as additional lifts are added as the overall slope angle reduces. The waste dump design is based on assumed material properties sourced from general mining engineering literature.

An assessment of the excavation characteristics, for the completely and highly weathered rock mass indicates that excavation can be achieved by mechanical means of digging and blasting with reduced powder factors. For the moderately weathered rock mass, the evaluation shows that most will require blasting ('blast to loosen'). For the fresh and slightly weathered rock mass, blasting ('blast to fracture') will be required for excavation.

The Etango project will undergo four stages of mining. Three pits would be exposed during Stage 1; two pits would be expanded during Stage 2; two pits would be mined during stage 3 and final wall cuts during Stage 4. This would provide the opportunity to confirm design assumptions and check stability experience from mined faces during Stages 1, 2 and 3.

The recommended pit slope design developed for the Etango project is presented in Table 16.1.

Table 16.1
Etango Slope Design

Domain	Design Sector	Weathering	BFA (°)	BW (m)	BH (m)	IRSA (°)	IRSH/ De-Couple (m)	OSH (m)	OSA (°)
North/ South	All Slopes	Weathered	55	6	12	39.8	20	380	50.5
		Fresh	70	9.5	24	52.8	120		

Legend
BFA Batter Face Angle
BW Berm Width
BH Batter Height
IRSA Inter-Ramp Slope Angle
IRSH Inter-Ramp Slope Height
OSH Overall Slope Height
OSA Overall Slope Angle

Modelling completed by RPS Aquaterra demonstrated the potential for depressurisation through natural drainage, to the pit excavation only, for a range of expected hydrogeological conditions at the Etango mine. The bedrock into which the open pit is to be excavated is massive, with limited structures. Bedrock aquifer permeability's are low (0.01 to 0.0001m/day).

The results show that the amount of depressurisation or reduction in pore pressures is sensitive to the assigned aquifer parameters and the rate of mining (i.e. the advance of maximum pit depth with time). For both the Base Case and Low Case, natural drainage to the pit faces is not expected to result in any significant depressurisation or lowering of piezometric heads. The modelling has assisted in identifying those areas where pressures will be high and where potential additional depressurisation might be required.

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Blasting was estimated to switch from dry conditions to wet conditions at the 176RL.

16.2.2 Dilution and Mining Loss

Previous resource estimates for the Etango deposit had employed an ordinary kriging ('OK') methodology into a large number of individually interpreted wireframes. Fundamentally, this type of approach results in an in situ mineral resource which does not represent the selectivity anticipated at the time of mining and to which modifying factors such as dilution and ore loss must be applied to enable reserve calculation. The optimisation study adopted a recoverable resource approach utilising Uniform Conditioning (UC) to model a recoverable resource. This model then formed the basis for employing Local Uniform Conditioning in order to create a uniform block size model suitable for use in mine planning as discussed in Section 14 of this report.

Following a review of the geological model it was decided to add no further dilution or mining loss to the model for the following reasons;

- Firstly, the resource modelling incorporated a recoverable resource modelling approach which models the tonnage and grade within an SMU
- Secondly, the process of creating the grade shells used for estimating the panel grades of the resource model incorporated dilution into the grade shells by applying a below economic cut-off grade of 50ppm for the mineralised zones.
- Thirdly, the block size or SMU of the 2015 LUC resource model is 6.25mE x 12.5mN x 4mRL which equates to 3.7 times the truck size. As a haul truck load will effectively be the SMU of the grade control process by employing radiometric truck scanning, dilution and mining loss has been incorporated by the larger block size. The ratio of resource block size to truck size (of 3.7) correlates well with the range of 3.5 to 8 reported in the literature of open pit uranium mines employing radiometric truck scanning.

16.2.3 Equipment Selection

Alternative truck sizes were considered in previous studies. A 220t truck provides the best mix of flexibility and equipment count for the material movement required. The mine is long, narrow, and centres on an orebody of inconsistent grade and strip ratio. Trolley assist was ruled out early due to the geometric variations of the mine.

Excavation of bench heights of 4-5m at production rates required leads to the selection of a large hydraulic excavator. Earlier studies showed that a five pass bucket selection on a 550t diesel hydraulic excavator is the best candidate for this excavation rate.

A poll of excavator manufacturers and end users was conducted during the DFS to better understand the preferences between the two choices of bench height (4m or 5m). The deciding factor was based on safety consideration from one of the largest users of backhoe excavators in Western Australia. The recommendation from this user was to limit cuts to 4m to 4.5m depth. Operators had concerns for heights greater than the recommended height. The DFS is based on a 4m excavation height (flitch) pre blast or 4.5m post blast.

The next decision was to determine an appropriate bench height given a 4m flitch. Where the flitch height governed the selection of hydraulic backhoe excavators, the bench height decision is governed by blasting considerations.

As part of the original DFS, a study of the geotechnical parameters of the materials in conjunction with an 'ideal' particle size distribution for a 60-89 gyratory crusher was

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undertaken. In the absence of blasting field trials, a modified Kuz-Ram cumulative distribution curve was calculated to form an understanding of the particle size distribution. Although the outcomes show that a 165mm hole provides the best potential outcomes, a 203mm hole for production was adopted to minimise the number of drills required and reduce operating costs. The 165mm hole remains as the planned diameter in trim shots.

With the hole diameter nominated, the trade-offs in blasting could be measured. Blasting outcomes are a trade-off between energy distribution, explosive confinement, and energy level. Two options for bench height were considered, namely 8m (two 4m flitches) or 12m (three 4m flitches). The best balance of the trade-offs is achieved with a 12m flitch, which improves both confinement and distribution for the same energy level.

16.3 MINE PLANNING

Mine planning covers the optimisation, pit design and mine production schedule of the open pit optimisation and shell selection, Table 16.2 summarises the inputs into the pit optimisation.

Table 16.2
Inputs into the Optimisation

Item	Unit	Value
Mill throughput	Mtpa	20
Uranium price	$/lb	75
Royalty	%	3.0
Transport, shipping, penalties, marketing and sales	$/lb	1.10
Processing and General & Administration costs	$/t ore	8.62
Average mining cost	$/t ore	1.80
Processing recovery (agitated tank leach)	%	87
Overall pit wall slope angle (inclusive of a ramp system)	Degrees	43 to 51

The LUC model, which reflects the result expected from the use of a truck scanner, formed the basis of the pit optimisations. A number of pit optimisation was carried out utilising the functionality of the software to determine pushbacks and apply mining width constraints to the interim pit shells. In addition sensitivity analysis was conducted on the Total Resource, including Inferred Resources and on Measured and Indicated Resources only. The latter formed the basis for the subsequent pit designs.

The results from the Measured and Indicated Resource optimisation are summarised in Table 16.3 and Figure 16.1. The optimisation results were smoothed without ramps, then taken through a series of Milawa Balanced (Whittle's scheduling routine) schedules to nominate a series of pit shells for design. Milawa Balanced runs suggest that Pits 28 provided the highest value and Pits 9, 11 and 19 were good candidates to provide adequate size and meet the mining targets and constraints (discussed further in the Mine Schedule section). Summaries of these shells are provided in Table 16.3.

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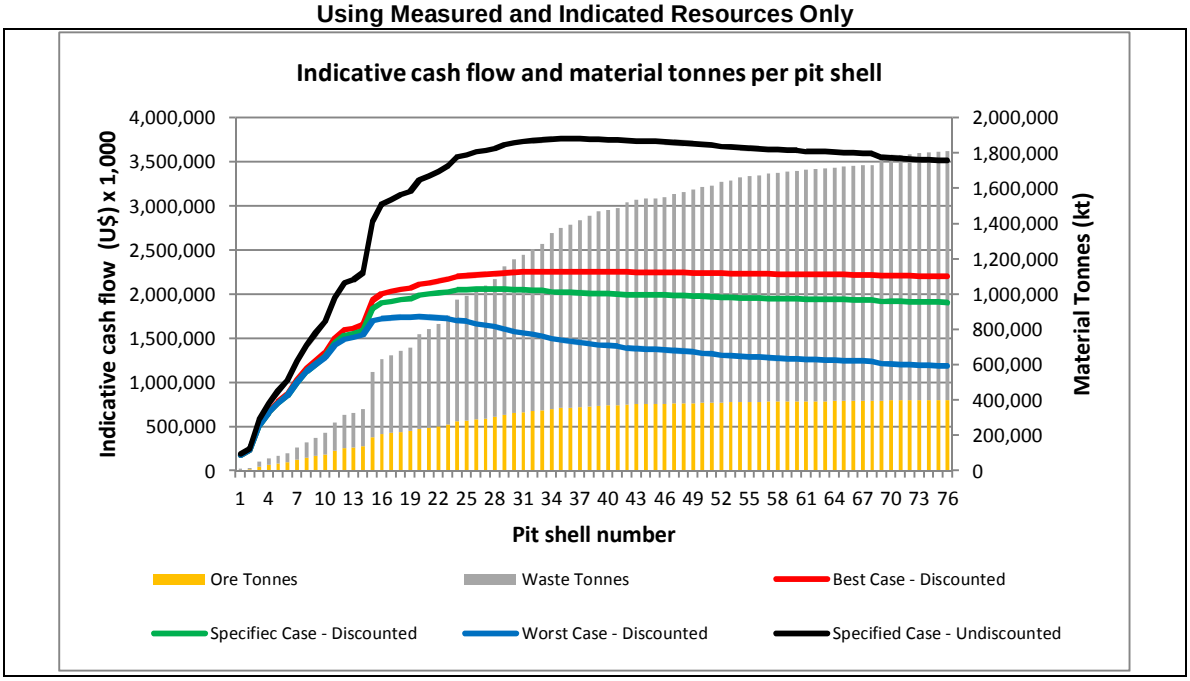
Table 16.3
Etango Uranium Project – Summary of Optimisation Shells Used in Whittle Generated Schedule after Mining Width Adjustment Measured and Indicated Resource Only

Shell	Total Material	Waste	Strip Ratio	Mill Feed			Cash Flow ($M)			
				Tonnes (in situ)	U_3O_8 Grade	U_3O_8 Output	Undisc CF	Best	Worst	Spec
	(Mt)	(Mt)	(w:o)	(Mt)	(ppm)	(lb x 1000)	($M)			
9	223	130	1.4	93	220	39,093	1,622	1,232	1,232	1,232
11	326	202	1.6	124	213	50,842	2,014	1,465	1,432	1,452
19	752	519	2.2	233	203	90,561	3,172	1,959	1.702	1,915
28	1,084	780	2.6	304	196	114,302	3,643	2,092	1,636	2,026

Figure 16.1
Etango Uranium Project – Summary Pit Optimisation Results Using Measured and Indicated Resources Only



16.3.1 Pit Design

As discussed above, Whittle pit shell 28 was selected as a basis for the final pit design depicted below in Figure 16.2. The final pit was designed with three pit exit ramps serving the waste dump and crusher positions as defined in the 2012 DFS.

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Figure 16.2
Etango Uranium Project – Final Pit Design

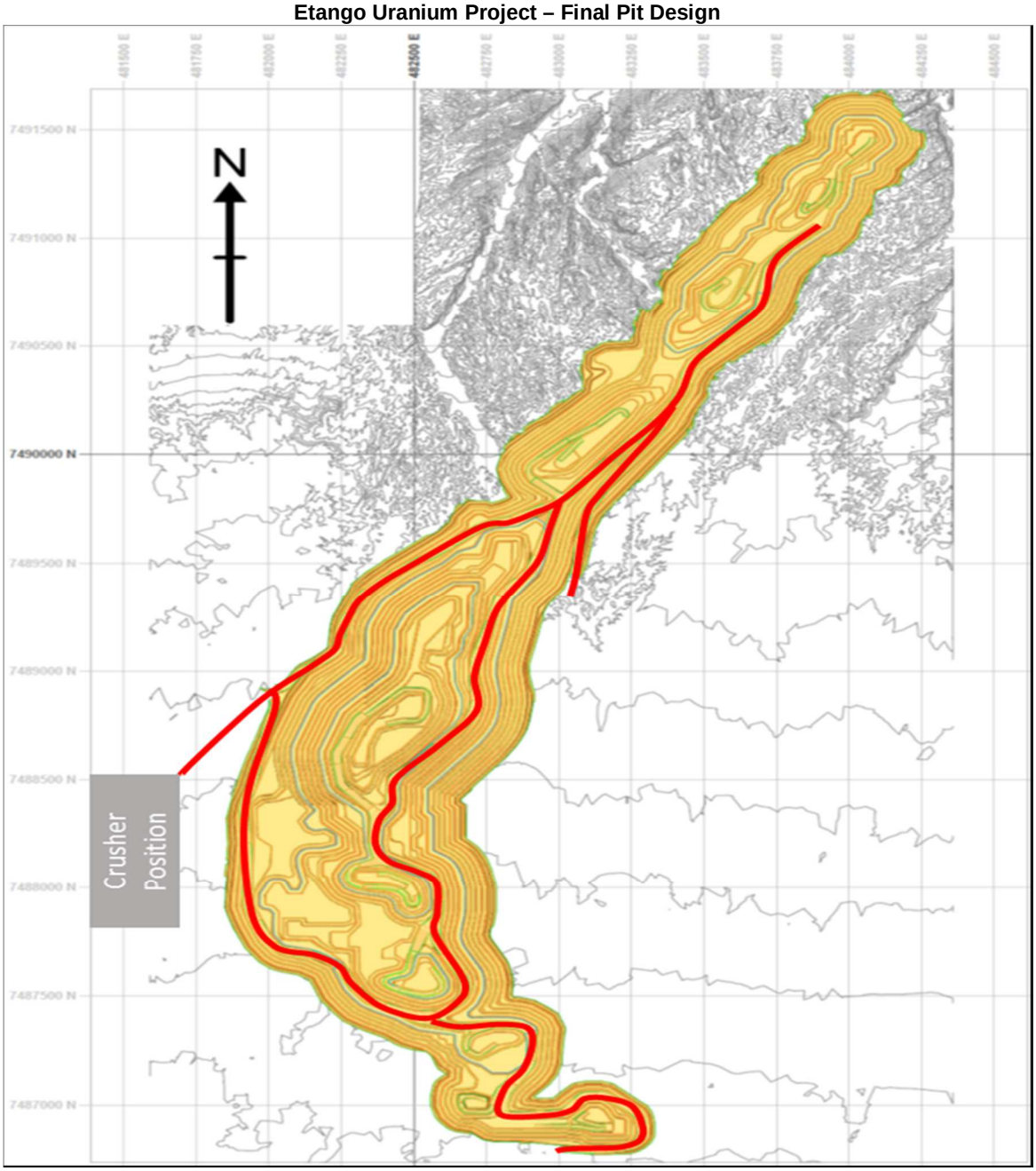

Table 16.4 provides a summary of the design criteria used for the detailed pit design work.

The geotechnical design parameters require a 15m 'decoupling berm' to be left when the inter-ramp slope height exceeds 150m. Where possible, pit ramps have been used to fulfil requirement to reduce waste stripping.

Stages 1, 2, 3 and 4 (LOM pit) were designed to aid a staged development of the open pit.

The material inventory is shown in Table 16.5.

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Table 16.4
Pit Design Specifications

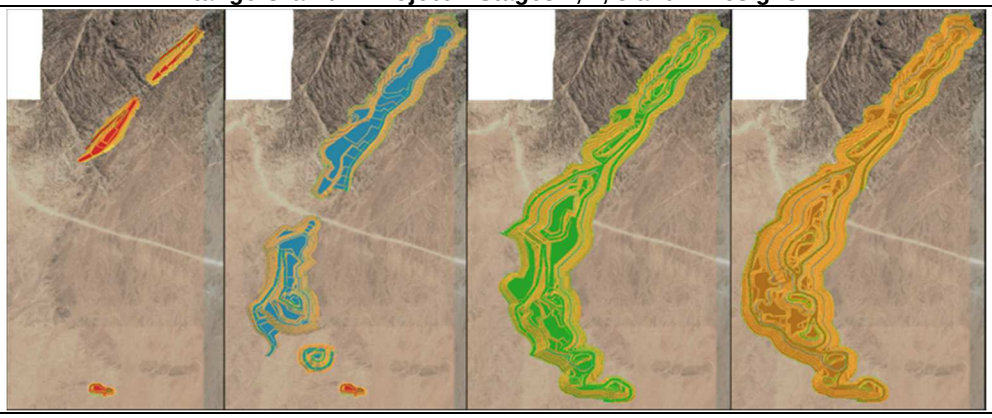
Item		Unit (m)	Value
Minimum working Width		**m**	**50**
Bench Height	Weathered Rock Fresh Rock	m	12 24
Batter angle	Weathered Rock Fresh Rock	deg.	55 70
Berm Width	Weathered Rock Fresh Rock	m	6 9.5
Decoupling Bench		m/vertical m	15m every 150m
Total Width	Dual Carriage Way Single Carriage Way Trough Ramp (Drop Cut)	m	30 17 30
Running Width	Dual Carriage Way Single Carriage Way Trough Ramp (Drop Cut)	m	24 11 28

Table 16.5
Staged Design Material Inventory

Stage	Ore		Waste	Total Material
	Tonnes (Mt)	Grade (ppm)	(Mt)	(Mt)
Stage 1	18.8	185	16.3	35.0
Stage 2	120.5	200	278.2	398.7
Stage 3	100.8	191	369.6	470.3
Stage 4	63.3	193	178.0	241.3
Total	**303.3**	**195**	**842.1**	**1145.4**

Figure 16.3 provides an overview of the three pit stages that were developed.

Figure 16.3
Etango Uranium Project – Stages 1, 2, 3 and 4 Designs



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16.3.2 Dump Design

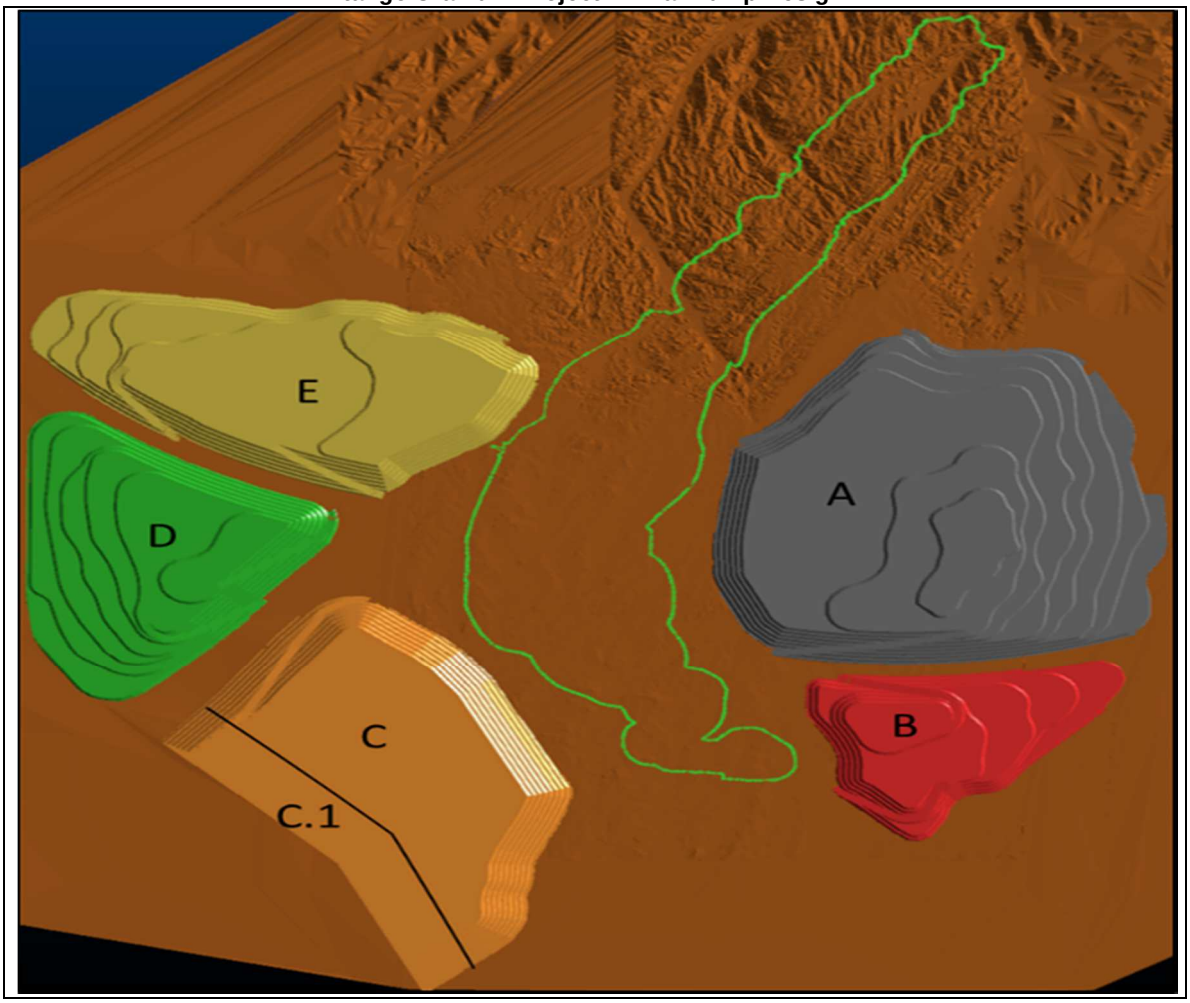
The 2012 DFS waste dump designs were used as a basis for this Optimisation Study. The updated pit design has a lower strip ratio and therefore less waste dump space is required for the OS.

As a result the southern waste dump (designated Dump C in Figure 16.4) was reduced in size to allow for a larger Ripios dump associated with increased ore tonnage in the pit design. The Ripios dump will have to be extended by approximately 150m and will therefore occupy the area shown as C1 in Figure 16.4.

The waste dump landform takes account of geotechnical and geochemical characteristics as well as consideration of the land character and vegetation, surface water and catchments, ground water, archaeology, topsoil requirements and other infrastructure needs.

Waste rock is non-acid forming. The visual (height limit) constraint is within 15m (vertical) of recommendation; the remainder of the constraints to the waste dump were met by applying a series of exclusion zones to derive the final dump design (Figure 16.4).

Figure 16.4
Etango Uranium Project – Final Dump Design



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16.3.3 Mine Production Schedule

Material was scheduled in Geovia's MineSched software utilising a re-blocked model (12.5m x 12.5m x 12.0m) to speed up the scheduling process.

The constraints set for the schedule were:

- Crusher feed rate of 20Mtpa
- Defer waste movement
- Maximum vertical advance rate of six benches per stage per annum
- Maximum total material movement (ex-pit) of ~100Mtpa.

A variable cut-off policy was adopted during the scheduling process. This allows for the cut-off grade to be flexed on an annual basis to maximise metal production in the early years of the mine life.

Figure 16.5 displays the total material mined for the Etango Project within the Stage 4 design. It is evident from the graph, that ore production in the early years exceed the crusher feed rate. This approach allows for applying a higher cut-off grade to the plant feed thereby allowing for a higher grade to report to the plant. The lower grade material is stockpiled and reclaimed in the later years of the mine life.

Figure 16.5
Etango Uranium Project – Total Tonnes Mined

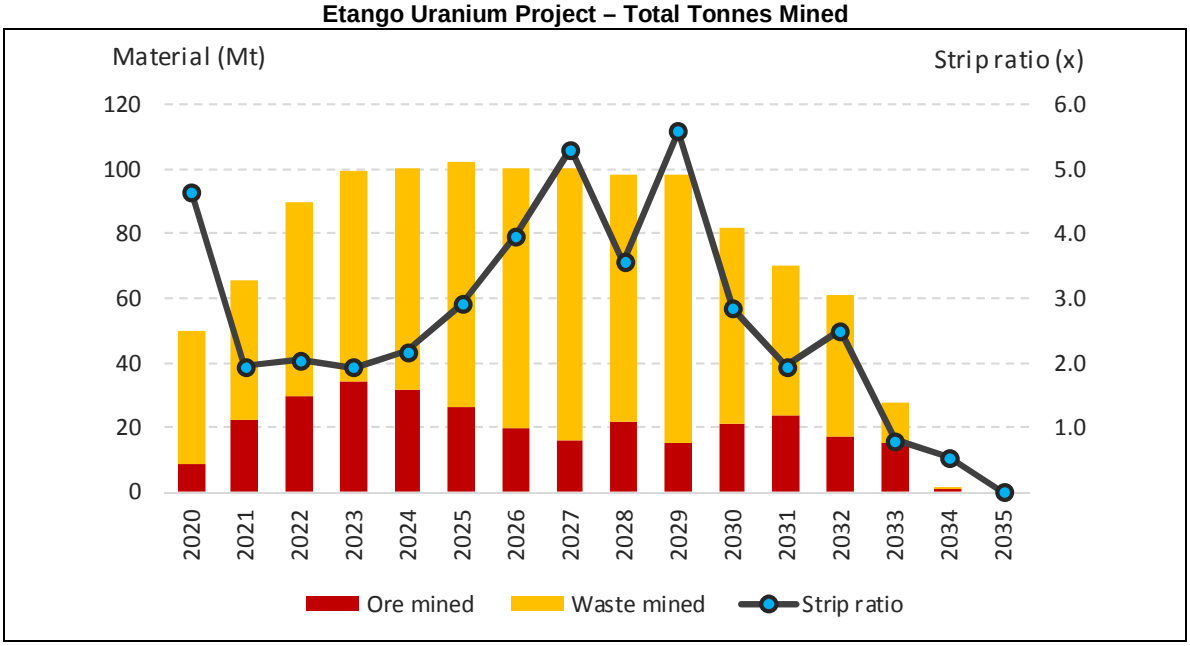
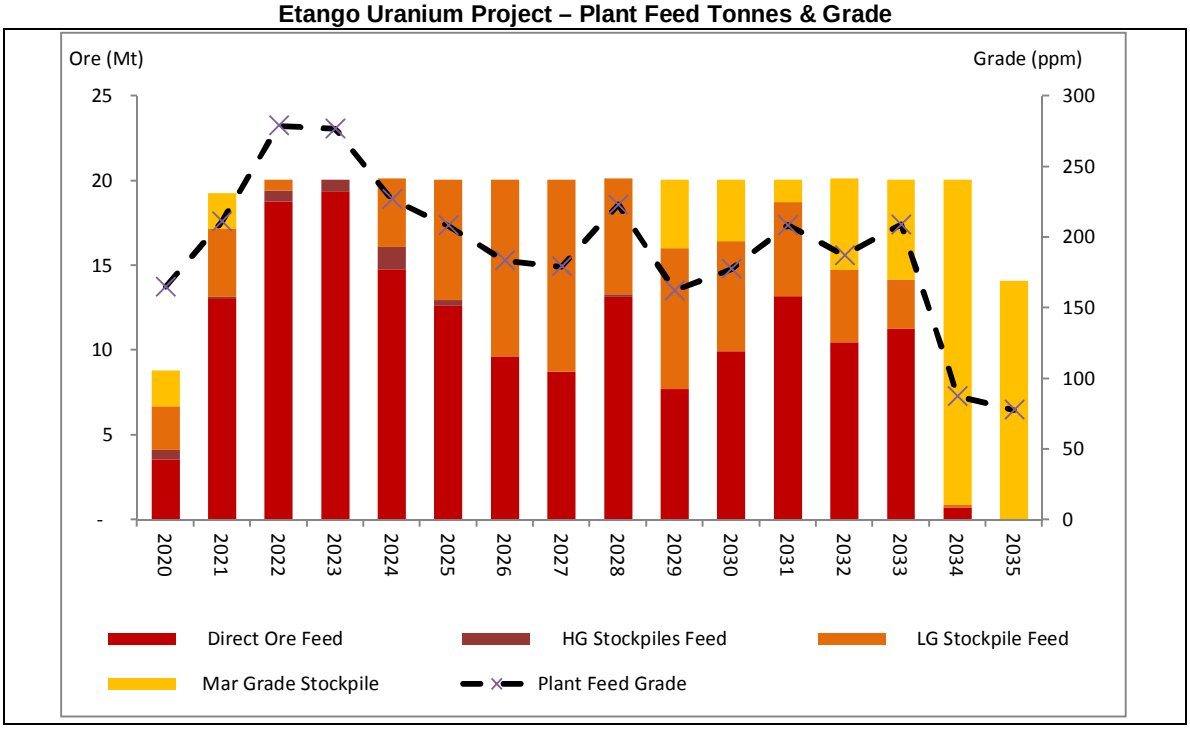


Figure 16.6 depicts the plant feed tonnage and grade. The effect of the variable cut-off policy is clearly evident high grades processed in the first five years of the mine life. The grade subsequently drop as lower grade material is fed from stockpiles.

Figure 16.6
Etango Uranium Project – Plant Feed Tonnes & Grade



A summary of the mine production schedule, along with the mill feed scenario, is provided in Table 16.6. Rehandle and stockpiling is based on an average volume of 30% of the ex-pit feed. Equipment required to achieve the production schedule is provided as Table 16.7.

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Table 16.6
Etango Uranium Project – Summary Mine Production Schedule

Model	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Total
Total Tonnes (kt)	50,136	65,370	89,561	99,566	100,273	102,079	99,917	99,872	98,285	98,294	82,019	70,081	61,112	27,406	1,431		1,145,402
Ore Tonnes (kt)	8,893	22,225	29,411	33,919	31,574	26,159	20,073	15,817	21,574	14,910	21,342	23,837	17,473	15,171	933		303,312
Grade (ppm)	164	197	226	210	174	176	172	186	209	178	170	189	203	251	289		195
In situ Metal – (klb)	3,221	9,639	14,651	15,690	12,096	10,166	7,615	6,472	9,959	5,845	7,999	9,914	7,821	8,386	594		130,070
Direct Tip Mill Feed(kt)	3,566	13,093	18,795	19,346	14,770	12,640	9,628	8,728	13,142	7,708	9,943	13194	10,457	11,280	737		167,026
Direct Tip Grade(ppm)	238	259	286	277	254	256	248	252	275	248	250	257	269	299	338		266
Direct Tip In situ Metal klb	1,871	7,476	11,831	11,813	8,265	7,130	5,264	4,841	7,953	4,215	5,472	7,487	6,211	7,447	549		97,825
Rehandle Live (kt)	5,250	6,159	1,280	729	5,360	7,435	10,447	11,347	6,988	12,367	10,132	6,881	9,673	8,795	19,338	14,104	136,286
Rehandle Live Grade(ppm)	115.55	109.52	181.07	265.31	152.51	126.95	123.70	123.53	126.00	108.68	106.66	114.55	98.03	92.88	78.05	77.77	107
Rehandle Live (klb)	1,337	1,487	511	427	1,802	2,081	2,849	3,090	1,941	2,963	2,383	1,738	2,090	1,801	3,327	2,418	32,246
Mill Feed (kt)	8,816	19,252	20,075	20,075	20,130	20,075	20,075	20,075	20,130	20,075	20,075	20,075	20,130	20,075	20,075	14,104	303,312
Mill Feed Grade (ppm)	165	211	279	277	227	208	183	179	223	162	177	208	187	209	88	78	195
Mill Feed In situ metal (klb)	3,208	8,963	12,342	12,239	10,067	9,211	8,113	7,931	9,894	7,178	7,855	9,225	8,301	9,248	3,876	2,418	130,070
Closing Stockpile (kt)	77	3,049	12,385	26,229	37,674	43,758	43,756	39,498	40,942	35,777	37,044	40,806	38,149	33,245	14,103		
Closing Stockpile Grade	78	103	110	112	102	98	93	86	84	79	78	78	78	78	78		
Closing Stockpile klb	13	689	2,999	6,450	8,478	9,433	8,935	7,477	7,542	6,208	6,352	7,042	6,562	5,700	2,418		

Table 16.7
Etango Uranium Project – Major Equipment Requirements for the Revised 3 Stage Pit Schedule

Equipment Type	Peak	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16
Sandvik D55SP	18.0	7	10	13	14	15	16	16	16	15	16	13	10	8	5	2	0
Komatsu PC5500	6.0	3	4	5	6	6	6	6	6	6	6	5	5	5	2	1	0
Komatsu 830E-AC	39.0	14	22	34	39	37	35	38	35	39	32	33	27	33	16	4	3
Komatsu WA1200	2.0	2	1	1	2	2	2	2	2	2	2	2	1	1	1	2	2
Komatsu WD600	4.0	2	2	3	3	3	4	4	3	3	4	3	3	2	2	1	1
Komatsu D375A-6	9.0	4	6	7	7	7	8	8	8	8	7	7	6	6	3	2	2
Komatsu GD825A-2	6.0	3	4	5	6	6	5	6	6	6	5	5	4	5	3	3	2
Komatsu HD785-7	4.0	2	2	3	4	3	3	3	3	3	3	3	3	3	2	2	2
Total	**84.0**	**37**	**51**	**71**	**81**	**79**	**79**	**83**	**79**	**82**	**75**	**71**	**59**	**63**	**34**	**17**	**12**

16.1 MINE CAPITAL AND OPERATING COSTS

The Mining Capital and operating costs are set out in Section 21.

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17 RECOVERY METHODS

17.1 OVERVIEW

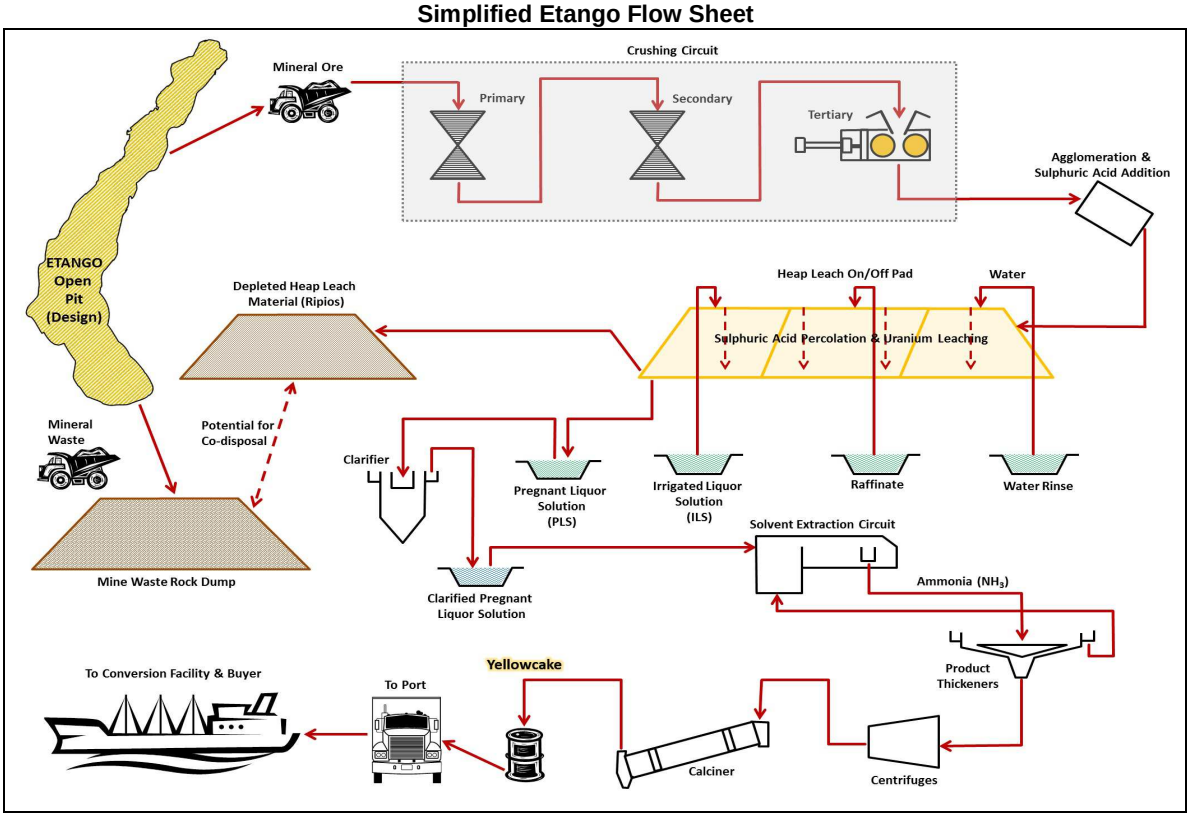
Fundamental process design criteria have been determined based on test work defined in the 2012 DFS and as described in Section 13 include:

- Heap leach crush size - P_{80} 5.3mm
- Leach duration - 30 days
- U_3O_8 recovery - 86.7%
- Acid consumption - 18 kg/t.

The basic flowsheet is shown in Figure 17.1 and comprises:

- Crushing and heap leaching of ore using sulphuric acid
- Recovery of uranium from leach liquor by SX, stripping, precipitation and calcination
- Removal and storage of leached ore.

Figure 17.1
Simplified Etango Flow Sheet



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17.2 HEAP LEACH

17.2.1 Primary Crushing

Ore is delivered directly to the run of mine (ROM) bin which has a live capacity of two truckloads and can be fed from two sides simultaneously. A rock breaker is installed to deal with oversize material. The ROM bin feeds directly into the gyratory primary crusher. The gyratory crusher is equipped with a 600kW motor and has a maximum capacity of 4800tph at an open side setting of 190mm.

The crushed ore discharges into the primary crusher vault and onto an apron feeder which discharges on to the primary crusher discharge conveyor. Primary crushed ore passes under the primary crusher tramp magnet where tramp metal is removed and diverted via a chute and discarded. The ore is transferred to the stockpile feed conveyor fitted with a weightometer, transferred to a second conveyor and discharged onto the coarse ore stockpile.

17.2.2 Stockpile and Secondary Crushing and Screening

The coarse ore stockpile has a live capacity of approximately 16 hours. A reclaim system consists of three hoppers and feeders which transfer ore onto the stockpile reclaim conveyor. The stockpile reclaim conveyor, which is fitted with a weightometer, transfers ore to the secondary screening feed bins via the shuttle head conveyor. The secondary screening feed bins, which have a total capacity of 1644m^3 live (30 minutes), feeds ore to the secondary screens via three vibrating feeders. The screens are installed with polyurethane mesh panels with 90kW motors. The oversize is conveyed to the secondary crushers, while undersize is transferred to the tertiary crushing circuit.

The secondary screening oversize conveyor is fitted with a weightometer, a magnet and a metal detector. Ore is directed to the secondary screening oversize shuttle head conveyor which discharges to the secondary crusher feed distribution bins that have a total capacity of 354m^3 live (15 minutes). Ore is transferred from the bin to the secondary crushers via belt feeders. The two secondary crushers are Metso MP 1000 standard head cone crushers. The crushers are equipped with 750kW motors and are set to a closed side setting of 35mm. Both crushers discharge on to the secondary screening feed conveyor.

17.2.3 Tertiary Screening and Crushing

Undersize from the secondary screens discharges onto the tertiary crushing feed conveyor which transfers the ore to the tertiary crushers. A weightometer is located between the secondary screens and the tertiary screens, to monitor the undersize throughput from the tertiary screens, while another weightometer is used to monitor the total feed. A magnet and metal detector are fitted to the tertiary crushing feed conveyor. The tertiary crusher feed ore is discharged, via a shuttle head conveyor, to the tertiary crushed feed distribution bins. The bins have a total capacity of 826m^3 live (15 minutes).

Belt feeders feed the two tertiary crushers. The HPGR units are Polysius 24/17-8, each fitted with two motors, each 2500kW. Crushed ore is discharged to the tertiary screen bin feed conveyor, which conveys the ore to the tertiary screening feed distribution bins via a shuttle conveyor. The bins, which have a total capacity of 1650m^3 live (30 minutes), distribute the crushed ore to five vibrating double deck banana screens via vibrating feeders. They are equipped with polyurethane mesh panels and 55kW motors. The top deck provides a protective screen with 15mm apertures, while the bottom deck screens have

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10mm apertures to provide the target P_{80} product size of nominally 5.3mm. There is provision in the layout for additional screens should a finer product size be targeted.

Undersize from the tertiary screens is discharged to the tertiary screens fine ore conveyor, which transfers the ore to the agglomerators. A weightometer fitted to the conveyor allows monitoring of fine ore produced by the crushing circuit. Oversize from the tertiary screens is added to undersize from the secondary screens on the tertiary crushing feed conveyor.

17.2.4 Crushing Dust Suppression and Extraction

Dust suppression sprays are used for all transfer points in all crushing and screening areas to minimise fugitive dust emissions. The dust suppression sprays are supplied from the raw water header.

Wet dust scrubbers with water tanks are located at the primary crushing, reclaim, secondary crushing and secondary and tertiary screening areas to extract and remove dust from the various relevant transfer points and equipment areas. In each case, the scrubbers recycle a large proportion of the water required, and a slurry (containing the removed dust) is bled from each of the scrubber tanks and transferred to two evaporation ponds. The evaporation ponds are not lined and are expected to be periodically cleaned out by mechanical means.

17.2.5 Agglomeration

Fine ore from the tertiary screens is transferred to two fine ore bins via the shuttle head conveyor. The fine ore bins, with a total capacity of 940m^3 live (30 minutes), feed ore to belt feeders, which are used to transfer ore to two agglomerating drums via their respective feed conveyors. Weightometers are fitted to each conveyor. Water, sulphuric acid and binder agent are added in the agglomerating drums which are 3.6m in diameter and 10m in length, fitted with 400kW motors. The agglomerated ore is transferred to the heap leach stacking system via conveyor.

17.2.6 Stacking and Reclaiming

The stacking and reclaiming system is a race-track type system, which comprises an overland conveyor and a fixed stacking conveyor with tripper to transfer ore to a stacking bridge arrangement equipped with a conveyor and tripper and stacker conveyor. The stacking bridge is supported on a five crawler undercarriages with a maximum speed of 2m/min. The maximum stacking height is 5m. A tripper travels along the top chord of the frames to place material anywhere along the length of the mobile stacking conveyor.

The reclaiming system is a similar race-track type system. A bucket wheel excavator is used to reclaim the ore from the heap and transfer the ore to the bucket wheel excavator conveyor. Ore is transferred via a mobile hopper to the reclaiming bridge equipped with a conveyor, which is supported on a five crawler undercarriage as for the stacking bridge. The Ripios is then transferred via a mobile hopper to the reclaiming overland conveyors via the heap leach reclaiming conveyor to the Ripios stacking system.

17.2.7 Ripios Stacking

Ripios is transferred from the heap leach reclaiming overland conveyors to the residue pad feed conveyor. A tripper conveyor allows residue to be transferred to the residue pad shiftable conveyor. A tripper conveyor on the shiftable conveyor transfers Ripios to the residue pad boom stacker that places the material onto the Ripios pad.

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The Ripios dump design consists of two lifts of front stacks and back stacks at 20m high and 10m high, respectively. The final Ripios dump will be 60m high, in keeping with environmental requirements. The first lift consists of eleven front stacks and nine back stacks. The second lift consists of six front stacks and five back stacks. Stacking gradually moves the Ripios towards the western outer boundary of the footprint.

The Ripios dump is unlined, based on results of geochemical characterisation and water seepage studies. The dump design includes the following infrastructure:

- Construction of a conveyor platform starter embankment. The height of the platform will be dependent on the quantity of suitable material available from the open pit, but it will be in excess of 5m
- Construction of a ramp using under- or oversized crushed gneiss from the heap leach drainage pad construction
- Construction of internal stormwater 'V' drains and delineation bunds to direct storm-water runoff from the Ripios dump to a localised collection pond
- Construction of external seepage and stormwater management systems.

Drainage from the Ripios pad is collected in the Ripios emergency pond and recycled to the heap leaching system. The pond has a double HDPE liner with drainage net in between for leak detection.

17.2.8 Heap Leach Solution System

The heap leach pad is constructed using several layers comprising: a compacted sub-base layer of around 300mm thickness; a 7mm thickness low permeability clay-impregnated geotextile lining; and a 1.5mm HDPE liner. Draincoil piping is laid at 4m spacings onto the HDPE layer and overlain with a fine drainage layer (around 400mm) followed by a 600mm thick coarse drainage layer. These drainage layers serve to both protect the liner and drainages pipes from the stacking and reclaiming system tracks and to provide a suitable medium for heap leach solution drainage to the draincoil system and subsequent channels and ponds.

The ore is stacked onto the prepared pads in modules, where each module represents one day of stacking. There are a total of 52 modules (26 modules per heap) with each module being equivalent to one stacking day. The first three modules are designed for stacking, ore rest and dripper installation. The next 15 modules are irrigated with ILS. The liquor from these modules produces the PLS, which is pumped to the SX circuit for uranium recovery. The subsequent 15 modules are irrigated with raffinate solution. The liquor from these modules is drained to the ILS pond and recirculated to the heap to build up uranium tenor. Following raffinate irrigation are 12 modules for draining, rinsing and draining of the rinse water. Solution from these modules is recirculated to the rinse modules. The remaining modules are spares and used for dripper removal and reclaiming.

The raffinate, ILS and PLS pumps are all designed for around 2200m^3/hr flowrate. Each area is irrigated at 15L/hr/m^2; twin drop drippers are used for irrigation.

The raffinate, ILS and PLS ponds are designed for a residence time of 6 hours, with 4 hours for the rinse water pond. The emergency pond is designed to contain 24 hours drainage from the heap and a 24 hour maximum rainfall event run-off.

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The construction of the PLS, ILS, raffinate and emergency ponds includes a clay-impregnated geotextile low permeability base liner (7mm), followed by double HDPE liner (1.5mm upper and 1mm lower) with a drainage net (3-4mm) between for leak detection. For the rinse pond, a single layer HDPE liner (1mm) over the clay impregnated geotextile layer (7mm) is used.

17.3 SOLVENT EXTRACTION, PRECIPITATION, CALCINATION AND PACKAGING

17.3.1 Solution Clarification

The clarification circuit consists of two feed tanks and pinned bed clarifiers (PBC), run in parallel. PLS from the heap leach is passed through a pinned bed clarifier to remove ultra-fine solid particles that might influence the performance in the SX circuit. Sulphuric acid is added to the feed stream to maintain a free acid of approximately 12g/L, and flocculant and/or coagulant are added to control the solids content in the clarifier overflow.

17.3.2 Solvent Extraction

The SX circuit consists of four process steps; extraction, scrubbing, stripping, and organic regeneration. These steps allow for continuous recovery of uranium from a low tenor aqueous solution into an organic phase which is then stripped to produce a higher tenor aqueous solution with reduced impurity levels.

The SX plant consists of:

- Three extraction Bateman Pulsed Columns (BPC), in parallel, with counter-current flow
- Scrubbing in three Bateman Reverse Flow Mixer Settlers (RFMSs) with counter-current flow
- Stripping in four RFMSs with counter-current flow
- Regeneration in one RFMS with counter-current flow.

All four stages involve crud removal that warrants further treatment for organic recovery and waste disposal.

All equipment and pipe lines that handle organic solutions are electrically grounded to earth for the purpose of removing static electricity as part of the fire protection strategy. Organic lines are constructed from SS316 or conductive FRP.

Clarified PLS overflows from the PBC and is fed by gravity (1329m^3/h) to a SX feed tank where the PLS is mixed with the spent scrub solution from the scrubbing stage. The clarified PLS is then pumped to the top of the three parallel BPCs. The PLS contains 241-397mg/L U_3O_8, approximately 12g/L H_2SO_4, between 0.63 and 2g/L Cl^-, Mg, Fe as well as other impurities.

Fresh organic (consisting of 5% Alamine 336 and 2.5% Iso-decanol in a (mainly) aliphatic kerosene diluent – Shellsol 2325) is prepared in the organic make-up tank; barren organic also flows into the tank. The barren organic is pumped to the bottom of the BPC at a total design flow rate of 169m^3/h.

The PLS is contacted counter-currently in the active section of the BPC with the organic. Uranium transfers to the organic and the depleted aqueous (raffinate), containing about 10mg/L U_3O_8, flows by gravity to the after-settler from where it is pumped to the barren pond.

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The after-settler has facilities for decanting any entrained organic. The raffinate flow is controlled to match the PLS flow rate and maintain a stable interface level in the lower decanter.

The loaded organic containing about 3150mg/L U_3O_8 overflows from the top of the BPC to the loaded organic tank from where it is pumped to scrubbing.

Pulsation air is generated by positive displacement blowers. The pressure of the air is raised to about 50kPag, raising the temperature to the vicinity of 95ºC. The air is cooled to 40ºC in a water-cooled, finned type cooler. The pressure in the line after the cooler is maintained at 45kPag, by releasing some air through a breaker valve.

Air vessels act as surge buffers for the pulsation air. The pulsation in the columns is achieved by a set of three 4-way valves switching between air (at 15–40kPag in the air vessel) fed to the pulsation legs of the columns, and venting via the 4-way valves from the columns to the atmosphere. The pulsed columns are operated in an organic continuous dispersion. The target hold-up of the aqueous phase is 20-35%.

The loaded organic is scrubbed in RFMS with dilute sulphuric acid (iron removal), demineralised water (chloride removal) and 90g/L ammonium hydroxide to maintain pH below 2.2 (Si removal) in order to remove any impurities which may transfer with the uranium during extraction.

The scrubbed organic is transferred directly to stripping, while the combined spent scrub solutions are transferred to the extraction circuit. The uranium is stripped from the scrubbed solvent using barren solution from the ADU plant containing a minimum of 120g/L ammonium sulphate and a maximum of 30mg/L U_3O_8 in four RFMS. The pH in the mixers is controlled using 90g/L ammonium hydroxide solution. The pH increases from fully loaded solvent (pH = 3) to fully stripped solvent (pH = 5.5). The strip discharge phase ratio O:A = 6:1. The resulting OK liquor has >18g/L U_3O_8. The OK liquor flows to the OK liquor after-settler where the majority of any entrained organic is removed. The OK liquor then gravitates to the OK liquor tank from where it is pumped to the precipitation circuit

The full organic stream is regenerated in a single RFMS using either 25g/L sodium hydroxide or 25g/L sodium carbonate or a mixture. The spent regeneration solution flows to the regeneration solution tank and a 10% bleed is sent to the effluent tank. The regenerated organic flows to an after-settler where any entrained aqueous is removed. The barren organic then returns by gravity to the barren organic tank.

Crud is removed periodically from the pulsed column by gravity and from the settlers using a mobile air diaphragm pump and flexible snorkel. It is transferred to the crud surge tank, and may also be accumulated in the crud holding tank. After settling and separation of the organic and aqueous phases, the crud is batch treated in the agitated crud treatment tank. The treatment includes the addition of reagents (sulphuric acid, diluent, demineralised water and filter aid) intended to enhance phase separation. The final procedure adopted will be based on experience and will be determined during commissioning.

Discharge from the crud tank can be separated into drained aqueous, decanted organic or mixed phase crud. The mixed phase crud is pumped to the plate and frame type crud filter for further treatment. The recovered liquid phases are sent to the drain separation system for separation and recovery, and the solids are drummed for disposal.

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17.3.3 Ammonium Diuranate Precipitation

This plant area incorporates three stages; precipitation, product wash and water removal by centrifuge. This section also includes a reagent mixing section.

The area contains higher concentrations of radioactive material, and so is isolated from general access via a high fence with security clearance required. It is monitored by a camera system. Special change rooms with washing facilities for clothes and personnel permits minimum contamination outside the enclosed area.

The loaded strip liquor reports to the ADU precipitation tank, fitted with an agitator to provide mixing and suspend the precipitate in solution.

Ammonia is added to the ADU precipitation reactor and reacts with the uranyl sulphate to form a precipitate. The continuous feed to the reactor causes the operating level to rise to the tank overflow. Additional tanks allow for additional residence time to complete the precipitation reaction.

Overflow from the ADU precipitation reactors reports to the ADU thickener. The thickener produces underflow slurry of 50% w/w which is pumped to the next processing circuit. The thickener overflow (barren strip) flows into the overflow tank from where it is pumped to the ADU polishing filter to remove any solids that may remain in solution prior to returning to the SX circuit.

There is a facility to recycle the thickener overflow stream if large quantities of ADU particles in suspension report to this stream.

The ADU storage tanks have the primary function of providing buffer storage capacity large enough to ensure that upstream processes can operate continuously during operations and during minor plant maintenance outages.

The centrifuges have a liquid discharge (filtrate) and a solids discharge. The filtrate is gravity-fed back to the ADU thickeners. The solids discharge consists of ammonium diruranate slurry with a paste-like consistency and a solids content of approximately 70% w/w. The centrifuge solids discharge into the corresponding screw feeder that feeds the corresponding ADU product kiln.

The ADU, which is not a saleable product, needs to be calcined at 800ºC to produce U_3O_8.

Spillage sump pumps are located at the ADU precipitation area and the ADU slurry thickener. These ensure that products are isolated in the particular area and cross-contamination is eliminated.

A fire water ring hydrant is provided for use during a fire event.

17.3.4 Reagent Mixing

Small mixing tanks are used to provide dilute solutions of:

- Sodium hydroxide (10%) in a mixing tank with an agitator
- Sodium hydroxide (25g/L NaOH) for regeneration
- Sodium carbonate (25g/L Na_2CO_3) for regeneration

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- Ammonium hydroxide (10%) in a mixing tank
- Sulphuric acid dilution will be completed using in-line mixers.

All solutions are diluted with demineralised water to minimise contamination of the SX product.

17.3.5 Product Preparation and Packaging

Calcination

Calcination is undertaken to convert wet ADU to dry U_3O_8 as a saleable product.

The Calciner screw feeders transfer the ADU to an electrically heated kiln which oxidises the ADU to U_3O_8 and reduces the moisture content of the feed to ≤1%w/w. The product kiln operates on a continuous 24 hour cycle.

Product is discharged from the kilns into a common product storage bin via rotary discharge feeders.

The kiln discharge chute has sufficient capacity to contain the contents of the retort tube to cater for any problem downstream with the drum packing plant, when transfer to the product storage tank is not possible.

Uranium Packing Plant

The empty drums are manually loaded onto the feed conveyor, from where filling, lidding, washing and weighing are automated. The drums first pass through an air lock into the packing module under negative pressure to ensure no product dust is able to leave the area. The drums are then conveyed to the filling position where the product is loaded at a controlled rate until the weightometer detects the target drum weight.

The drum packing plant module operates automatically. Drums are also automatically washed and dried once they have been filled and lidded. A label is printed with the appropriate details and this is manually stuck to the drum by the operator.

Approximately 44 drums can be loaded into each 20 foot sea-container.

Off-gas and Dust Scrubbing

The off-gas system has two main duties, namely dry dust extraction scrubbing and off-gas scrubbing. Dust scrubbing occurs after maintaining negative pressures in the various plant areas and in the process technician work areas. The off-gas scrubbing duty primarily handles the captured kiln off-gases. Individual scrubbing modules are proposed for each calciner (two units) and a third system for reagent and ADU vent gas scrubbing and building dust management.

17.4 REAGENTS

There are 11 major reagents used in the process plant, listed as follows:

- Peroxide
- Diluent – Shellsol 2325

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- Extractant and Modifier – Alamine 336 extractant and Isodecanol modifier
- Ferrous Sulphate
- Coagulant
- Sodium Hydroxide
- Sodium Carbonate
- Anhydrous Ammonia
- Binding Agent – Magnafloc 351
- Flocculant – two different flocculants for PLS and for ADU.

These are delivered in bulk tankers or containers and there is sufficient storage space for each on site for 30 days of operation.

Mixing to required concentration levels takes place on-site. Spillage containment systems are in place, with sumps and pumps to return spillage to mixing tanks or to appropriate parts of the operating plant.

Fire protection systems are provided for flammable compounds as appropriate.

17.5 SITE SERVICES

The services areas include water and air provided to the individual process plant areas or reticulated throughout the plant in the case of plant and instrument air, drinking and safety showers water, gland seal water and fire water.

Water

Desalinated water is pumped to the site discharging into six raw water tanks providing a total residence time of 24 hours. A small fraction of that water is directed to the potable water plant.

Fire water is supplied with three fire water pumps, which include a diesel powered pump, which withdraw water from raw water tanks five and six to supply fire water for the fire water ring main and the SX fire systems. The water contained in the bottom part of these tanks is allocated solely for fire water use and equates to a total of 576m^3. This volume of water provides the SX foam system with 10 minutes of operation, and four water hydrants for 4 hours.

The potable water plant provides 150L of water per person per day. The potable water tank provides 24 hours storage capacity.

The demineralised water plant is fed from the main raw water header and discharges into the demineralised water tank, which provides a storage capacity of 24 hours. Duty/standby pumps are used to transfer the demineralised water to the SX and precipitation areas.

Separate raw water storage tanks are provided for the primary crushing and fine crushing circuits. Make-up water is supplied from the main plant raw water header. Fire water for the fine crushing circuits is supplied by three fire water pumps, which include a diesel powered pump.

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Raw water is transferred from the fine crushing water tank to the crushing water tank by transfer pumps and distributed to the primary crushing area and mine water trucks from this tank by various pumps.

Air

Air services are split into three plant areas: plant, primary crushing and SX/reagents. Duty/ standby compressors are provided for each of these systems. Individual air receivers then distribute this air to either a header for general use in their respective plant areas or to an instrument air dryer and subsequent instrument air receiver.

A separate compressor and receiver are used to provide high pressure air to the precipitation area.

Diesel

A diesel storage tank and a fuel bowser are provided to receive diesel and distribute it to site.

Sulphuric Acid

Bulk concentrated sulphuric acid (98% w/w) is transported to site from Walvis Bay by a trucking contractor.

On site the acid is transferred to four mild steel, sulphuric acid storage tanks, providing storage for 28 days usage. Acid is withdrawn by the duty/standby sulphuric acid distribution pumps for delivery to the agglomeration, heap leach, and SX and precipitation areas.

The sulphuric acid unloading and storage area is suitably bunded and serviced by a sump and sump pump, transferring any spillage and wash-down to the raffinate pond. Two safety showers are also provided in this area.

17.6 SITE LAYOUT

The Etango process plant takes up an area of approximately 8km^2. The site layout is shown in Figure 17.2, and the layout design philosophy is discussed in the remainder of this section.

The site layout took account of a number of factors, including a requirement to remain within current licence area.

The layout took account of an environmental exclusion zone located north of plan, plus a preference to remain south of the watershed into the Swakop River system. Further environmental restrictions were minimisation of visual impact within National Park and reducing effects on the Welwitschia plant locations.

With these restrictions, the waste rock dumps were sited adjacent to the open pit to minimise haulage costs which are the largest single component of the operation. This led to the coarse ore stockpile and process plant being located 3km from the coarse crusher, linked by an overland conveyor.

Other features include:

- Burying the primary crusher to lower the height of the ROM pad
- Aligning the fine crushing plant east-west, taking account of the prevailing wind

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- Location of significant structural loads (HPGRs, cone crushers and vibrating screens) outside of the palaeo-channel
- The heap leach pads located southwest of the main plant to suit the topography of the site and minimise earthworks
- The collection ponds for the heap are located such that the heaps drain to the ponds
- Solvent Extraction/Reagents plant is located adjacent to the heap leach operation on competent ground to the northeast of the ponds, providing close proximity of PLS ponds for pumping into the plant
- Any bleed streams from the solvent extraction plant can drain by gravity to the heap leach ponds
- Water storage located adjacent to the SX ponds close to the mine lease boundary

The Ripios Storage facility is located at the extremity of the final waste rock dump profile, adjacent to the heap leach pad at the southern end of the mining waste dumps. This area suits the radial stacking arrangement.

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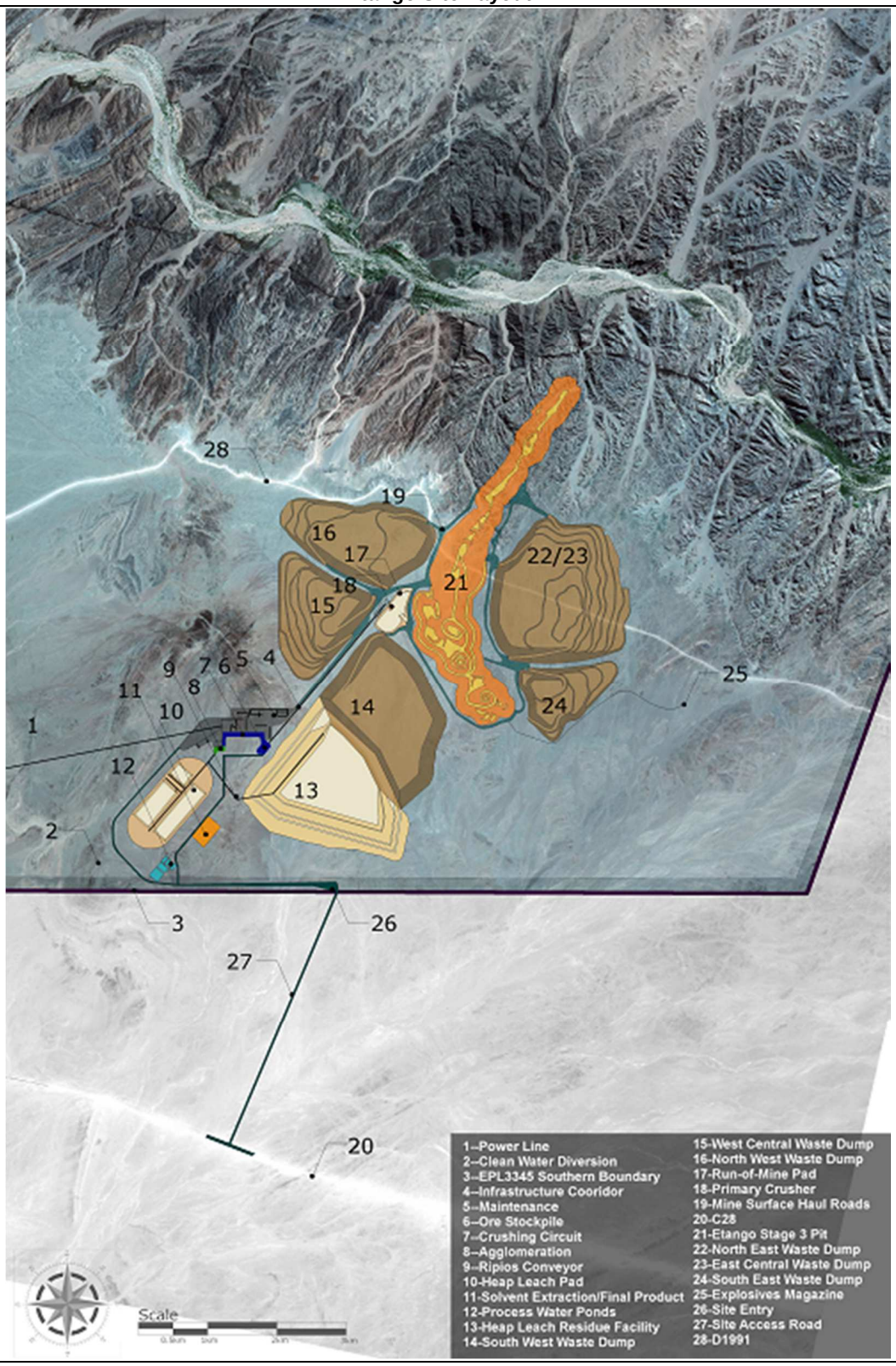
Figure 17.2
Etango Site Layout



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18 PROJECT INFRASTRUCTURE

The existing external infrastructure arrangement is shown in Figure 18.1.

Figure 18.1
Etango Project – Existing External Infrastructure

18.1 SITE INFRASTRUCTURE

Site infrastructure includes provision of:

- Heap leach pad, as discussed in Section 17.2.8, as part of the process plant description
- Waste rock dumps, described under Mining in Section 16.3.2
- Ripios disposal dump, described as part of heap leaching (Section 17.2.7)
- Site services, i.e. water, air, diesel and sulphuric acid, as described in Section 17.5.

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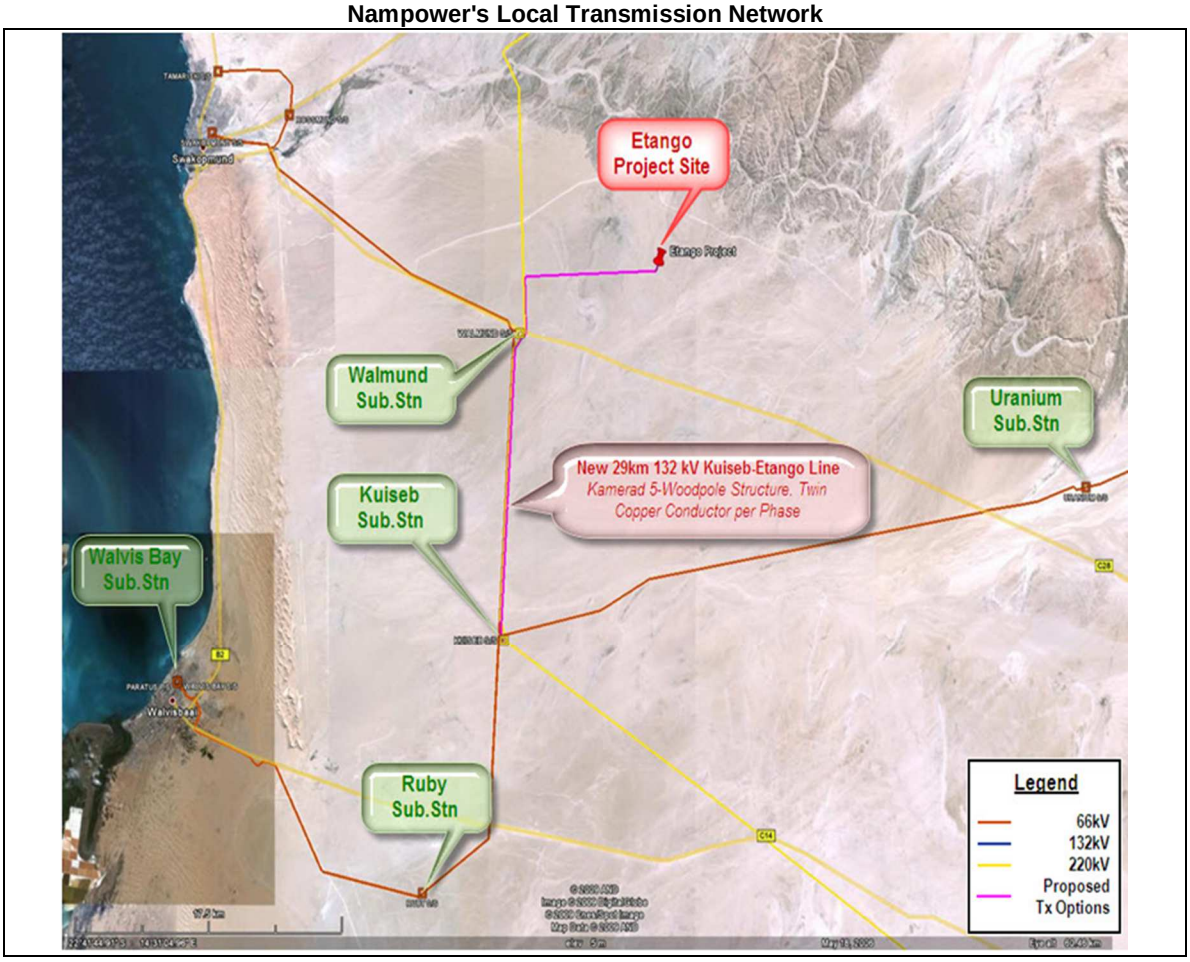
18.2 POWER

18.2.1 Project Power Demand and Supply

The process plant has installed power of approximately 49.5MW with an average operating demand of approximately 39.1MW. The largest single drives are the HPGRs of which there are two, each with two by 2.5MW motors.

Power for the Etango site will be fed by NamPower from the 220kV national grid through its substation located at Kuiseb (Figure 18.2), which is to be upgraded to 160MVA capacity. Distribution from Kuiseb is currently at 66kV, which can be upgraded to a distribution voltage of 132kV. NamPower has proposed a 29km 132kV transmission line from the Kuiseb substation to the Etango project site where a 132/33kV switchyard, transformer(s) and 40MVA indoor Etango substation will be installed.

Figure 18.2
Nampower's Local Transmission Network



The power system, supplied and installed by NamPower, is expected to be fully operational 24 to 30 months from the signing of a Power Supply Agreement between Bannerman and NamPower. The commercial arrangements between NamPower and Bannerman will involve the capital cost of the power line being paid by Bannerman, and a schedule of rates and payments.

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Construction power is by transportable temporary generator sets provided by the construction contractor.

18.2.2 Namibian Grid Capability and Expansion Plans

Namibian Demand and Grid Capacity

Current maximum generation capacity in Namibia is 508MW (May 2015), although this is markedly reduced in the dry season when the Ruacana hydro-electric power capability is reduced significantly.

Peak power demand in Namibia (2015) was 550MW. In the past Namibia has imported up to 60% of its power requirements from South Africa and other neighbouring countries, although increased demand within South Africa has limited available power in the region.

The proposed development of the Husab uranium mine, is likely to increase demand by an estimated 50MW in the Erongo region alone.

NamPower is considering a number of alternatives to increase power generation capacity, including:

- Combined cycle gas-fired power station (Kudu Gas) – 400MW to 800MW (earliest 2019)
- Coal-fired power station in the Erongo region – 250MW
- Baynes Hydropower Station – 360MW to 550MW (50-50 split between Namibia and Angola)
- Renewables – 30MW Solar capacity and 44 MW in Wind capacity.

None of these projects is well advanced, although some have commenced the process of Environmental and Social Impact Assessments.

Power prices are expected to rise significantly to fund this additional generation capacity and to offset increases in supply tariffs.

Power Pricing

An electricity price of $0.104/kWh has been used in the OS, this being the price current in 2015 escalated at double the Namibian consumer price index (CPI) to the end of the decade and discounted to 2015 terms.

18.3 WATER

18.3.1 Project Demand

Total usage during operations is estimated to be 4.72Mm^3/a (Table 18.1), equating to a daily requirement of 12,930m^3/day.

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Table 18.1 **Operational Water Requirements**	
Area	**Annual Usage (Mm^3)**
Mining	0.40
Process	4.12
Infrastructure/Administration	0.20
Total	**4.72**

Water requirements for the mining operation are primarily for dust suppression. Process plant requirements are for agglomeration, reagents, and heap leach make-up, as well as dust management. Infrastructure/administration requirements cover ablution and sewage treatment facilities. No provision has been allocated for any future expanded water usage.

Construction water requirements are estimated to be 627,000m^3 at an average of 860m^3/day, including demand from the 1500 man construction camp on site. Ablution and crib facilities are to be constructed and operated for the construction phase. Domestic water effluent will be treated via six 250 person sewage treatment plants distributed throughout the Etango works site.

18.3.2 Namibian Water Supply Capacity

Namwater can currently supply up to 14Mm^3/a in the Erongo coastal region drawing from two aquifer systems located north and south of Walvis Bay. It is understood that there is no additional supply capability from these sources.

Areva, a French nuclear energy focused company, has constructed a desalination plant at Wlotzkasbaken, 30 km north of Swakopmund, to support its developing Trekkopje uranium mine. The desalination plant has an installed freshwater capacity of 20 Mm^3/a with potential to expand to 25 Mm^3/a. Trekkopje is currently mothballed and the water is being sold to bulk users in the Erongo region. Namwater is currently in negotiations with Areva to acquire the desalination plant.

18.3.3 Etango Site Water Supply

The Etango water scheme is expected to comprise two pump stations, one at Swakopmund and one along the pipe route, each installed with three variable speed pumps.

The above-ground delivery line is expected to be 32km long and 400mm diameter. Covered 'Pioneer style' water tanks will be erected at site.

18.3.4 Water Tariff

The water tariff of \$3.5/m^3 used in the OS is based on the cost of desalinated water delivered to site.

18.4 ROADS

The C28 gravel road from Swakopmund to Windhoek passes approximately 5km south of Etango, and is adequate for the Project's transport requirements. A 7km unsealed spur road will be constructed to link the existing road to the Etango site (Figure 18.3).

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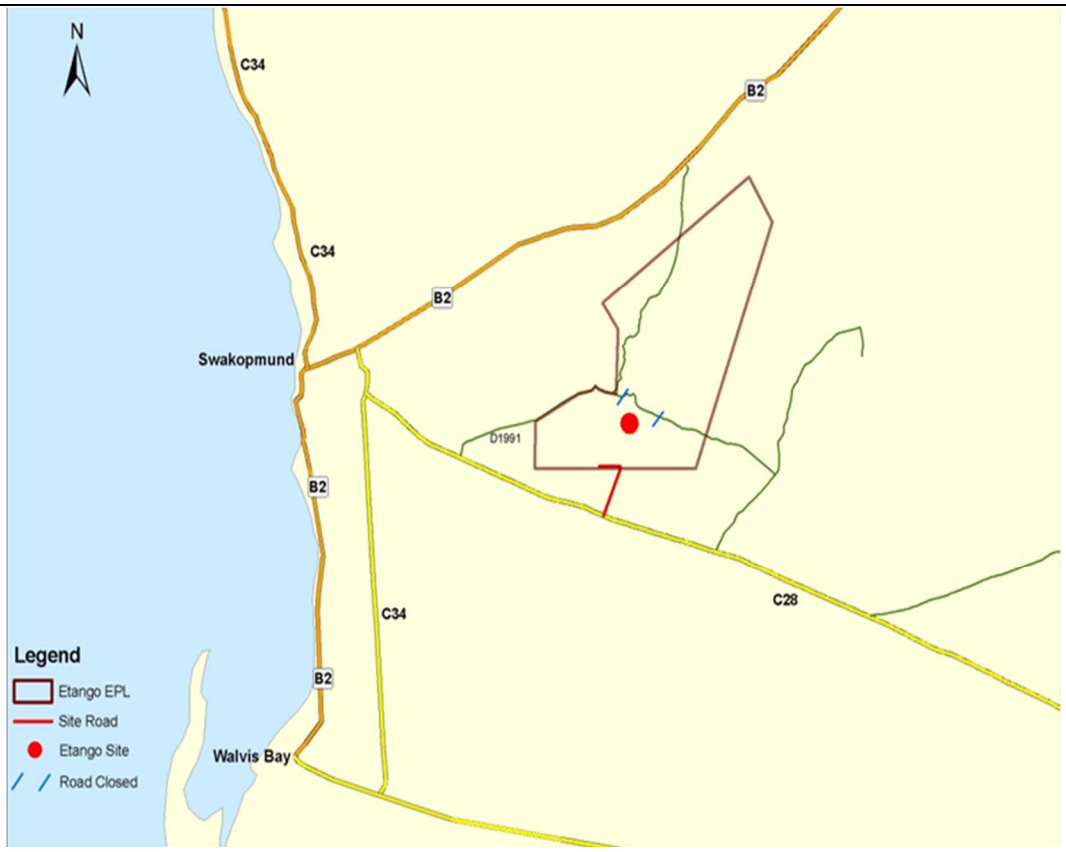
The road will cross an existing tenement held by Reptile Uranium. A letter of 'in principle agreement' has been received from Reptile Uranium, while an allowance has been included in the capital cost estimate for sterilisation drilling.

The capital cost for the access road has been allowed for in the cost estimate.

Figure 18.3
Local Area Roads



18.5 PORT OF WALVIS BAY

Walvis Bay is Namibia's largest commercial port, receiving approximately 3000 vessel calls each year and handling about 5Mt of cargo. It is a sheltered deepwater harbour largely unaffected by bad weather. The area of Berths 1, 2 and 3, the turning basin and the approach channel are at a depth of 12.8m below chart datum. From Berths 4 to 8, the depth is 10.6m below chart datum.

The port comprises:

- A container terminal that can handle approximately 250,000 containers per annum. Capacity is being expanded to about 400,000 container movements per annum. These facilities will accommodate the requirements of the Etango Project
- Tank storage for sulphuric acid; four tanks, currently utilised in part by Rossing Mine, but with capacity available to Bannerman

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- Bulk Shipping Terminal. Bulk receipts and transhipping will be handled through the existing Walvis Bay facilities.

NamPort has previously provided support to Etango, and ongoing negotiations will facilitate construction of new facilities, and upgrading of existing facilities as required to receive and to tranship Etango bulk shipments to site.

An allowance for minor upgrades, including addition of one extra acid storage tank (15,000t) has been included in the capital cost estimate.

18.6 COMMUNITY FACILITIES

Facilities in the towns of Swakopmund, Walvis Bay and Arandis will support the Etango operations, and Bannerman will participate in community based activities and initiatives.

The Swakopmund office will cover housing management, recruitment and administration activities. Retention of this office provides Bannerman with a face to the community and reduces the number of people reporting to the security gate on-site on an adhoc basis.

Provision of the independently managed radiation testing facility proposed by Bannerman will be an important asset, not only to limit costs of radiation management, but as a symbol of safety in the community. The possibility exists to share this facility and the cost of operation with other uranium producers in the area.

18.7 PERMANENT HOUSING

Discussions are being held considering the role of Bannerman in the provision of accommodation in existing townships for permanent employees. The blanket provision of housing is fraught with political issues, but the shortage of suitable existing accommodation in these townships will affect the ability of Bannerman to attract and retain the services of quality personnel to match the planned staffing and ramp-up activities.

A sum of $6M has been allowed in the Owner's capital cost estimate to assist in provision of housing, but details have yet to be developed in conjunction with local authorities.

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19 MARKET STUDIES AND CONTRACTS

19.1 PRODUCT SPECIFICATIONS

The processed product from the Etango Project will be uranium oxide (U_3O_8), known as 'yellow cake', contained in standard drums each holding up to 450kg of U_3O_8 depending on the density of the final product. Yellow cake is inert and mildly radioactive, emitting alpha radiation which is absorbed by the drum. It is non-toxic and would be dangerous to humans only if ingested in quantity. A range of regulations governs the transport of the drums, including Namibian and international transportation regulations.

Arrangements for the sampling and assaying of the yellow cake within the shipped drums will be made with the relevant conversion facilities. Penalties and surcharges exist at these facilities for impurities.

19.2 PRODUCT SHIPPING AND CONVERSION

19.2.1 Shipping

The drums of processed yellowcake will be packed into sea containers at the mine site and transported by road to the port of Walvis Bay. Drums of yellow cake have been exported from Namibia through Walvis Bay for approximately 40 years, the material being sourced from Rössing and, in recent years, also from the Langer-Heinrich operation.

Specialist shipping agents exist for yellow cake and other nuclear materials, located in Europe and the USA. Consistent with standard practice, Bannerman expects to pay for all shipping and transport to the conversion facility, and then for the weighing, sampling and assaying at the converter.

The cost estimates as determined in the DFS (provided by the nuclear fuel transport division of NY-based nuclear fuels trading company ICAP Energy) have been used for the purposes of the OS. The rationale being that the costs are likely to reduce, given the reductions in the oil price and consequently transport costs. Utilising the DFS estimate can therefore be considered conservative.

19.2.2 Conversion

The drums of yellow cake will be shipped to one of three or four established conversion facilities throughout the world, with the primary ones located in France (Areva/Comurhex), US (Honeywell/Converdyn) and Canada (Cameco/Port Hope/Blind River). At the conversion facility, the U_3O_8 is converted into a gas (uranium hexafluoride, UF_6), placed in canisters and either stored, sold or shipped to an enrichment facility.

Title to the yellow cake typically passes from the producer to the buyer upon delivery to the conversion facility. The producer receives a credit to its metal account at the conversion facility for the majority of the delivered quantity soon after delivery, with the balance determined after weighing, sampling and assaying. Sale of the final determined quantity of uranium occurs in accordance with the producer's relevant sales contracts.

All conversion facilities have pre-set specifications for yellow cake. Before signing up with a particular conversion facility, sample quantities will be sent to each conversion facility for analysis and acceptance. Ultimately a contract will be negotiated between the producer and each of the conversion facilities utilised. The contract covers the procedures for weighing,

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sampling and assaying of the yellow cake, and the terms for storage, as well as the details of surcharges for deleterious mineral content. There is typically a free storage period with additional charges for longer term storage.

Test work carried out on the Etango ore to date does not indicate that the final yellow cake product will contain above-standard levels of impurities which would typically attract penalty surcharges at the relevant conversion facilities.

19.3 SALES AND MARKETING

19.3.1 Sales and Marketing Strategy

Bannerman Namibia expects to form an in-house sales and marketing function to administer the Etango Project's uranium sales arrangements and revenues. This function will be supported by specialist uranium marketing groups as required and, potentially, other uranium producers seeking to market the Etango Project's uranium production.

Cost allowances for in-house and external marketing services have been allowed in the operating cost estimates for the Project.

The yellow cake sold from the Project will be sold under a mix of spot (short term sales and delivery), medium term (1-2 years to delivery) and long term (3+ years to delivery) sales contracts. Based on current estimates and advice from uranium marketing consultants, it is expected that approximately 20 key sales contracts will be required at any one time to cover the majority of the expected annual production of 6-9Mlb of U_3O_8. Initial marketing efforts are expected to involve the negotiation of sales contracts with 'ramp up' features allowing for some flexibility in the development timetable as production and sales volumes increase with the establishment of stable operations.

The buyers of the U_3O_8 product from the Etango Project will largely comprise nuclear power utilities in various nations which generate power using nuclear facilities including China, South Korea, USA, Japan, France, UAE, Saudi Arabia, UK, Finland, Sweden, Spain and Russia. In addition to nuclear power utilities, sales are expected to occur to nuclear fuel brokers and potentially other producers seeking to build inventories for their own contractual obligations or investment purposes. The Project owner has already established relationships with the majority of the above parties and is positioned to enter into contractual negotiations at the appropriate time.

19.3.2 Sales and Marketing Costs

Estimated sales-related costs incorporated in the DFS total \$1.10/lb U_3O_8, as shown in Table 19.1. This was assumed unchanged for the purpose of the OS.

Table 19.1
Sales and Marketing Costs

Item	Basis	\$/lb U_3O_8
Freight and Shipping		
From mine gate to Walvis Bay port	Quote sent 6 March 2012 from Wesbank for N\$4,650 per sea container (40 x 450kg drums per sea container). Round up to (say) \$0.02/lb to include insurance and any other incidentals.	\$0.02
From Walvis Bay port to UF_6	Rates provided by NY-based nuclear fuels	\$0.36

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conversion facilities (Europe or North America).	shipping agent ICAP Energy, ranging between $13,390/container (to Converdyn, US) and $14,440/container (to Comurhex, France). Equates to $0.35-0.37/lb, allow $0.36/lb.	
Marine and transport insurance	Estimated marine and transport insurance premium cost on a sea container of 38,800lbs U_3O_8 valued at approx. $2.7M = 0.03% x $2.7M, equates to $0.02/lb.	$0.02
Conversion Facility Charges and Penalties		
In accordance with Cameco and Converdyn specifications	Weighing and sampling fee is generally expressed as a rate per kilogram, including the weight of the loaded drums. Cameco charged a rate of $0.43/kg in 2009, equating to approximately $0.20/lb U_3O_8. Allowing for price escalation since 2009, an estimate of $0.25/lb U_3O_8 is assumed.	$0.25
	Impurity penalties: Given the results of the work undertaken to date, an allowance of $0.10/lb U_3O_8 has been made.	$0.10
Sales and Marketing	Estimate of $2.5M/year to cover labour and other fixed costs for average annual sales of 7Mlb U_3O_8.	$0.35
Total		$1.10/lb

19.4 URANIUM DEMAND AND SUPPLY FORECASTS

Extensive studies and analyses of the global nuclear power and uranium markets are frequently published by industry analysts and capital markets institutions. The following subsections provide an overview of recent views regarding the global uranium market and associated price forecasts.

19.4.1 Uranium Market

Uranium oxide is used, primarily, in the generation of electricity within nuclear power facilities. Based on data from the World Nuclear Association, total uranium consumption in 2014 was approximately 171Mlb U_3O_8 and total uranium production was approximately 146Mlb U_3O_8 representing 85% of demand. Total uranium consumption is expected to grow in 2015 to approximately 174Mlb U_3O_8.

The supply deficit is presently filled from secondary supplies including commercial stockpiles, nuclear weapons stockpiles, recycled uranium and plutonium from reprocessing used fuel and some from re-enrichment of depleted uranium tails.

Following the natural disasters in Japan in March 2011 and the resultant operating issues with the Fukushima Daiichi nuclear power facility, uranium spot and long term contract prices weakened. However, the clean nature of nuclear power for base load electricity generation

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remains a key alternative and growth area for the world's industrialised and fast-developing nations. This fact is expected by numerous analysts to drive higher future uranium prices.

Recent key events supporting market analysts' views of higher uranium prices include:

Japan

Japan has restarted two of the existing 43 operable nuclear reactors with a further 24 in the process of restart approvals. The low utilisation of the Japanese nuclear fleet continue to place considerable stress on Japan's trade balance due to significantly increased imports for its fossil fuel power generation facilities. In June 2015 the government approved the draft electricity generation plan to 2030; this has nuclear at 20-22%.

Commitment to Nuclear Power

Various nations have in recent months confirmed their commitment to nuclear power. In particular, the Chinese government has a stated target of nuclear energy capacity of 58GW by 2020, compared with approximately 19GW in 2014. India expects to have 14GW nuclear capacity on line by 2020, from the current installed capacity of approximately 5GW. Other nations to reaffirm their commitment to nuclear power include the United Arab Emirates, Saudi Arabia and the USA where six new reactors are expected to come on line by 2020;

Secondary Supplies

The current supply deficit is being satisfied through the sale of uranium from inventories and secondary sources. However, the 1993 'Megatons to Megawatts' program between Russia and the USA for the down-blending of highly enriched uranium from dismantled Russian nuclear warheads ceased in December 2013 and is unlikely to be renewed.

Incentive Prices for New Mine Development

Mining of uranium is subject to many of the same cost pressures as other mining operations but, unlike other commodities, uranium mining carries increased environmental and safety management obligations and associated development timeframes. The development of new mines and the expansion of existing operations will, in the view of various uranium producers and developers, require higher uranium prices to incentivise development and expansion commitments.

19.4.2 Uranium Price Forecasts

Spot prices and long-term contract prices were approximately $36/lb and $44/lb respectively, at the end of 2015.

Various banking institutions and broking firms prepare periodic forecasts of future uranium spot and long term contract prices. Forecast spot prices for the next 5 years presently range from approximately $65 up to $80/lb U_3O_8. Historic short and long term prices are shown in Figure 19.1.

The economic assessment within this OS utilises a base case uranium price, stated in (real) December 2015 dollars, of $75/lb U_3O_8. Sensitivity analyses have been run at various prices either side of the base case price.

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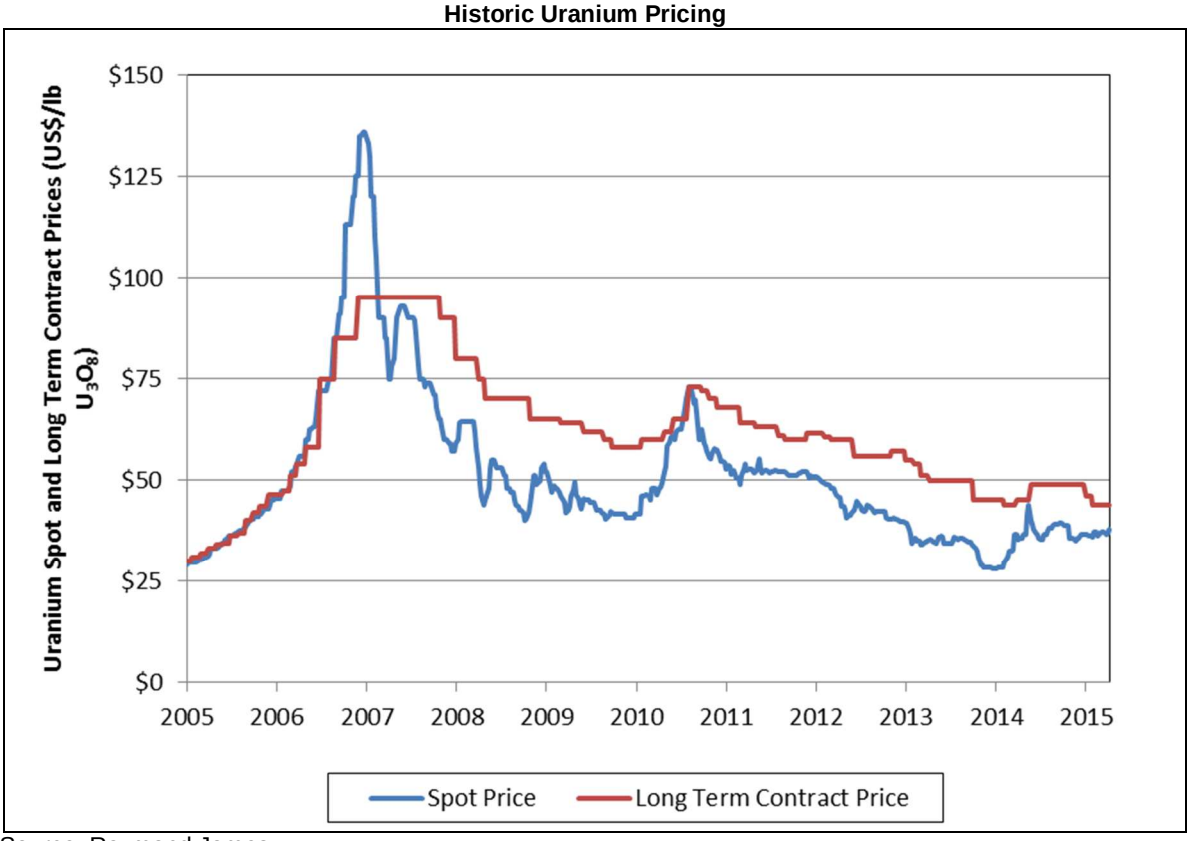
Figure 19.1
Historic Uranium Pricing



Source: Raymond James

19.5 CONTRACTS

At this time, Bannerman advises that no contracts exist between it and third parties regarding the Project.

Obtaining a Mining Licence over the Project area is a necessary first step in Project Development, and the ESIA of April 2012 has been submitted and approved as part of this process.

The next stage of the Project requires Bannerman board approval for Project Development, at which point it will become necessary to obtain negotiate a number of fundamental agreements and contracts, including:

- EPCM contract for Project construction, including early engineering activities
- Uranium sales contracts (short, medium and long-term)
- MARC type contracts for the supply and servicing of the major pieces of mobile mining equipment
- Supply contracts with NamPower and NamWater for provision of power and water to site
- Supply and service contracts are expected for major reagent supplies, in particular sulphuric acid and the various reagents for the SX process.

The particulars of the relevant contracts will be prepared as and when the Project is developed.

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20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

Bannerman received Environmental Clearance in July 2012 for its plans to establish the Etango Project as a 20Mtpa heap leach operation as described in the DFS. The Environmental Clearance was valid for 3 years and expired in July 2015 upon which a renewal of the Environmental Clearance was submitted. Updated Environmental Clearance was issued on the 11 November 2015 valid for 3 years from date of issue.

The Environmental Clearance for the location and design of infrastructure ancillary to the Etango Project (including an access road, a water pipeline and power lines) was granted by the Ministry of Environment and Tourism on 11 February 2013 valid for 3 years. A request for renewal has been submitted.

20.2 BRIEF SUMMARY OF THE PROPOSED ETANGO PROJECT

20.2.1 Mining Operations and Life of Mine

The mining follows a conventional open pit drill, blast, and load and haul truck and excavator/ shovel operation. Blasting will occur at nominated times at an expected frequency of three to four times a week.

The deposit will be mined in a series of four stages. The total rock mined will rise from a nominal 50Mtpa in Year 1 to just over 100Mtpa between Year 4 and Year 10, before declining over the last 6 years of operations. The final pit will be ~6km in length, 1km wide and, at its deepest, approximately 400m deep.

All waste rock is to be dumped externally to the open pit at the waste dumps planned, on either side of the pit. Rehabilitation work will be carried out progressively and rock-lined drains will be constructed, where required, to ensure excess run-off is controlled and directed down to sediment traps.

On mine closure, no future alternative land uses are likely, as the mine is within the Namib-Naukluft National Park. The mine will be made safe and potentially hazardous areas will be permanently closed off to the public. The Etango Project will set cumulative finances aside, from construction onwards, to pay for all mine closure and post-mine closure costs, such as the ongoing monitoring of groundwater.

20.2.2 Processing

Ore will be delivered to the ROM stockpile. The ROM ore will be crushed, mixed with water, sulphuric acid and binding chemicals, and transported via conveyor belts onto a heap leach pad. The heap leach pad is composed of a compacted sub-base layer, a low permeability clay-impregnated geotextile lining and a HDPE liner. Draincoil piping is laid on the HDPE layer and overlain with fine and coarse drainage layers. These drainage layers serve to both protect the liner and drainages pipes from the stacking and reclaiming system tracks and to provide a suitable medium for heap leach solution drainage to the draincoil system and subsequent channels and ponds

The stacked ore is 'drip irrigated' from the top with a mild solution of sulphuric acid. The liquid percolates through the heap, leaching the uranium into solution. It is collected in the drainage layer and delivered to the collection ponds. After the leach cycle is complete, the

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barren ore is successively drained, washed and drained again with water to recover the uranium-bearing solution.

The leached residue is reclaimed from the heap and conveyed to the Ripios Storage Facility. The final size of the Ripios Storage Facility will be approximately 2500m by 2000m, with an average stacking height of 45m. Seepage from the Ripios Storage Facility is collected in two lined ponds and recycled to the active heap leaching system.

The uranium-rich leaching solution is pumped from the collection ponds to the SX plant, where the uranium is absorbed (loaded) onto an organic reagent. The loaded solution is stripped of uranium which is then precipitated, thickened and calcined to produce yellow cake which is packed into drums for transport off site is recovered.

Eleven reagents are used in the process, including sulphuric acid, peroxide, sodium hydroxide, sodium carbonate and anhydrous ammonia. The ESMP details how all fuels and chemicals will be received, prepared, handled, stored and distributed.

20.2.3 Site Water Management

The Etango Project is located in a part of Namibia characterised by low rainfall, high humidity and sparse vegetation. There is no weather station on site, but the average annual rainfall in the district is 0-50mm. Annual totals are variable as rainfall is dominated by rare, intense events of as much as 100mm in 24 hours. Engineering designs were set to manage a 1000-year 24-hour event.

The Project is located over a watershed, hence there are no significant upstream catchments to deal with. Storm-water run-off from up-stream catchment areas will be diverted around the site (Figure 20.1).

Storm water flow rates and volumes were modelled, and it was concluded that:

- Relatively small amounts of surface water are generated due to low rainfall and high infiltration rates
- No substantial run-off is generated from waste dumps
- Large trenches and containment ponds are not required.

Stormwater management systems are designed to maintain separation of clean and dirty water, and incorporate a combination of 'V' drains, trenches, seepage cut-off trenches and storage ponds of suitable size.

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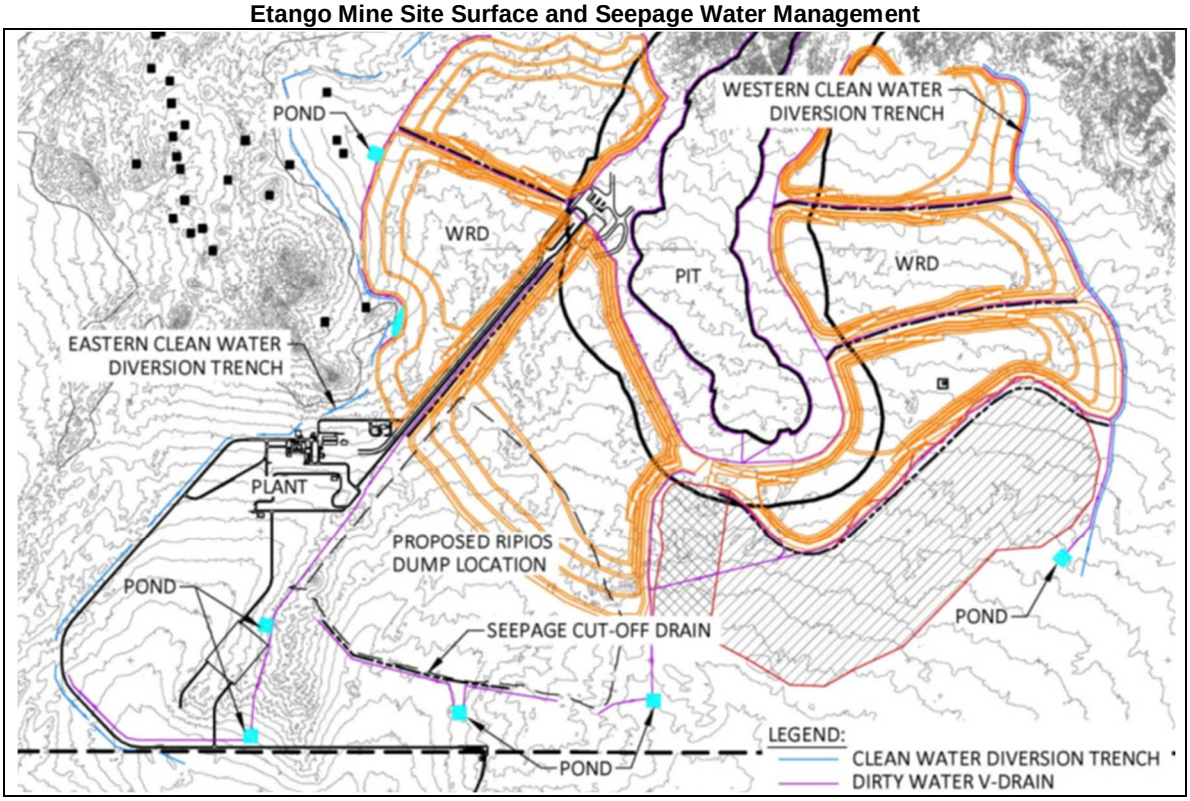
Figure 20.1
Etango Mine Site Surface and Seepage Water Management



Dirty water drainage is directed to containment ponds during operations, but, where possible, it is redirected towards the open pit during decommissioning. Elsewhere, evaporation ponds will be constructed as part of the final landform.

Dirty water 'V' drain design includes HDPE lining to minimise infiltration. Dirty water storage ponds include HDPE linings, as well as a 500mm high compacted earth embankment to keep out clean water.

Seepage water cut-off trenches lie around the southern portion of the waste rock dumps where topography slopes to the south. These, too, are designed to allow discharge into the pit after cessation of mining. HDPE liner and a bund are included in design.

20.2.4 Off-site Infrastructure

NamPower is proposing a permanent power supply to be sourced from the Kuiseb Substation, which is to be upgraded to 160MVA capacity (Section 18.1).

The Project operational water demand is approximately 5Mm^3 per annum. This will be sourced from Namwater's supply. A dedicated pipeline will bring desalinated water to the mine site, running adjacent to the proposed 7km mine access road.

Water requirements for the construction phase will be trucked to site until the pipeline to site is in operation. Peak demand during construction will be 27ML/month during bulk earthworks for compactions and dust suppression.

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20.2.5 Activities during the Construction Phase

An average of approximately 800 construction workers will be required during construction, with numbers peaking at approximately 1500. Bannerman proposes to set up a temporary construction camp on a site which will later be impacted upon during operation, e.g. the location of the future western waste rock dump. The camp includes ablution and kitchen facilities. The sewage treatment plant installed during construction will be re-used during operations.

Prior to construction, a final site layout plan indicating the different areas, e.g. lay-down areas, access route, camp and batch plant will be required to be approved by the MET.

20.2.6 Operations Workforce

Just over 1000 people will be employed directly by the mine and its contractors during operations.

No employees or contractors will be accommodated on site. Instead, the mine workforce will be transported from Swakopmund, Walvis Bay, Arandis and environs, in company-provided transport.

20.2.7 Decommissioning

The proposed decommissioning activities will be listed in the ESMP and developed in more detail prior to the commencement of construction. However, in essence, all plant, equipment and foundations will be removed, and the plant site rehabilitated. Waste rock dumps will be shaped to minimise erosion and runoff. The surface of the Ripios dump will be compacted and covered with a metre of waste rock. Appropriate drainage systems will be in place to intercept and direct dirty water runoff and seepage to the abandoned open pit.

Groundwater monitoring systems will be retained for a period to detect any contamination leaving site, although studies indicate there will be negligible impact on the existing groundwater quality which is very poor.

20.3 ENVIRONMENTAL IMPACT ASSESSMENT PROCESS

The Environmental Impact Assessment (EIA) methodology included an Environmental Scoping process and specialist studies which informed the ESIA and ESMP. Since the beginning of the ESIA process in 2008, Bannerman and ASEC have engaged in an ongoing public participation process (PPP) as summarised in Table 20.1. Publicising a cell phone number for receiving comments and suggestions by SMS has broadened the feedback.

Table 20.1
Public Consultations to Date

Period	Purpose	Public Participation Process
October 2008	Review draft Environmental Scoping Report	Public meetings in Arandis, Swakopmund, Walvis Bay and Windhoek attended by 230 people. Two focus-group meetings, including site visits, with neighbours and Coastal Tourism Association of Namibia (CTAN).
June 2009	Updating Stakeholders	Short progress report circulated to

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Table 20.1 Public Consultations to Date		
Period	**Purpose**	**Public Participation Process**
		approximately 400 Interested and Affected Parties (IAPs). Meetings with Regional and local Town Councils Focus-group meetings with neighbours and CTAN.
October 2009	Review draft ESIA, specialist studies and draft ESMP	Public meetings in Arandis, Swakopmund, Walvis Bay and Windhoek attended by 90 people. Focus-group meetings with neighbours and CTAN.
July 2010	Review draft Environmental Scoping Report for all linear infrastructure to the mine	Interim Background Information document circulated to approximately 400 IAPs. Public meetings in Swakopmund and Windhoek attended by 82 people. Focus-group meetings with neighbours and CTAN.
February 2011	Interim update on ESIA	Public meetings in Swakopmund and Windhoek attended by 48 and 9 people, respectively. Focus-group meeting with neighbours.
February 2012	Review draft Amendment ESIA and ESMP	Public meetings in Arandis, Walvis Bay, Swakopmund and Windhoek. Meetings with local and regional government, neighbours and CTAN.
January 2014	Review draft ESIA and ESMP for the Demonstration Plant	Public meeting in Swakopmund; Focus-group meeting with neighbours at the Goanikontes Oasis and meetings with local government and CTAN.

20.4 MAIN ISSUES RAISED

Interested and affected parties have shown great interest in the Project. Issues raised most frequently were:

- The mine's power and water requirements and where these will be sourced
- The current overuse of the aquifers and the potential pollution of groundwater
- The negative impacts on the sense of place at the Moon Landscape, the Swakop River and the Welwitschia Plains. This was most often verbalised in terms of noise and visual impacts, but also in loss of bio-diversity
- The impact of dust and potential radiation on the towns and workforce
- The closing of the road beyond the D1991 turnoff, used to access the Welwitschia Plains and neighbouring farms
- The impact of increased traffic on other roads

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- The implications for tourism in the region
- The cumulative impacts of the Etango Project and of several other proposed uranium mines on the Namib Naukluft National Park and the Erongo Region (e.g. loss of sense of place, the impacts caused by the influx of job-seekers on the provision of towns' services, rising house prices and salaries on existing businesses and residents)
- The size of the workforce, the need for employment, developing the necessary skilled labour force to maximise the use of local labour
- The need for confidence that the mine will put aside adequate resources to implement a full mine closure plan.

20.5 ALTERNATIVES TO THE PROJECT

During the course of project planning, a number of alternatives were considered, notably:

- Agitated acid leach process and heap leach processing options
- Positioning of waste rock dumps close to the proposed pit and alternative placement of these dumps to minimise both the visual impact and the potential impact to the Swakop River catchment
- Initially the D1991 was proposed as the access road to the mine, but a new spur off the C28 is planned to minimise the impacts from the Moon Landscape
- Rail and road options for bulk transport.

20.6 LEGISLATION, POLICIES AND THE URANIUM RUSH

The ESIA summarises relevant Namibian legislation and policies. Of particular relevance to the Etango Project is the Uranium Rush: Strategic Environmental Assessment (SEA) which sets out the likely cumulative impacts of mine-related developments in the Namib. It describes the Environmental Quality Objectives (EQOs) or 'desired state' and makes recommendations as to how this desired state can be achieved and maintained. The sections on water, energy, tourism and recreation, and biodiversity are most relevant to this project.

The location of the mine and processing plant, close to the Moon Landscape and the Swakop River, puts it in 'Red Flag' and 'Yellow Flag' sensitive areas for biodiversity and tourism, as set out in the SEA. The SEA states that red and yellow flag areas should be unavailable for mining and prospecting unless an extraordinary mineral deposit of national importance occurs in the area. Given the size of the Etango Project, it is of national importance and the ESIA addresses these sensitive sites in detail to ensure that all the necessary mitigation and control measures are put in place to minimise negative impacts.

20.7 SUMMARY OF BIOPHYSICAL AND HUMAN ENVIRONMENT IMPACTS

A detailed description of the biophysical environment has been developed, along with the background of the human environment. These are summarised in Table 20.2.

Impacts and mitigation/enhancement measures are included in the ESIA report.

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Table 20.2 **Summary of Biophysical and Human Environment Impacts**	
Biophysical/Human Element	**Impact**
Geology and soil	***No new findings since May 2012*** Mining will permanently change the geological formation at the site. This resource will not be available for future generations. Desert soils are generally nutrient-poor, but they are still able to support a surprising array of plant life. Disturbed soil may not be conducive for burrowing activity, which could result in localised extinctions of burrowing animals such as invertebrates, lizards, scorpions and rodents.
Surface water / geohydrology	***No new findings since May 2012*** Water is a scarce resource and the over-abstraction of ground water is a national concern. An east-west striking watershed separates the Swakop River and the Tumas Catchment areas. Most of the Etango mine site lies on the latter where shallow drainage lines drain the terrain in a south-westerly direction towards the Swakopmund-Walvis Bay Dune Belt. These drainage lines are poorly defined and are conspicuous by the perennial plants they support. Sporadic surface water flows plays a major role in structuring and driving desert ecosystems. Any disruption of surface-water flow patterns at the Etango site therefore has the potential to negatively impact on downstream communities of plants and animals. Three aquifer systems were identified within the project area: • The primary palaeochannel aquifer to the south of the project area • The Swakop alluvial aquifer to the north of the study area • The secondary fractured aquifer hosted in the Damara Supergroup and Abbabis Complex (basement aquifer), consist mainly of meta-sediments and igneous rock. Radionuclide analyses confirm the relatively high background of uranium and other daughter elements in the regional groundwater. The ratios of U234 and U238 indicate that the Swakop River alluvium is unaffected by pollution from upstream sources. A conceptual site model, describing potential linkages between contamination sources, pathways and receptors was formulated for the project area by BIWAC Namibia. ERM translated this conceptual groundwater model into a numerical flow and transport model to simulate the potential groundwater impacts as a result of the Project. The results suggest that: • Water levels have not have stabilised after 92 years of recovery,

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Table 20.2
Summary of Biophysical and Human Environment Impacts

Biophysical/Human Element	Impact
	indicating a very slow water level recovery as a result of pit dewatering • The extent of the uranium and sulphate plumes are not significantly influenced by the different Ripios options (base case vs. waste rock base and cover). However, they are marginally influenced by the seepage rate, which is essentially determined by the placement water contents of the Ripios, and whether the Ripios is lined or not • For the worst-case scenario modelled (no Ripios liner, 15% moisture content for the Ripios and no adsorption) uranium concentrations remain within a maximum of 1km from the Ripios and HLP, after 60 years. Simulated sulphate concentrations stay significantly below background concentrations • Modelling results suggest that it is unlikely for any contamination plume to reach the alluvium and the palaeochannels in the modelled time period and beyond. The main recommendations which Bannerman has incorporated into the revised design are: • Position infrastructure away from the palaeochannel and to avoid obstructing surface drainage where possible • Locate Waste Rock Dumps to the south of the Swakop River catchment so run-off flows into the Tumas Catchment • Ensure that mine infrastructure (particularly the Heap Leach Residue Facility) can withstand and contain runoff from a 1:50 year rainfall event • Line the heap leach pad • Routinely monitor groundwater quality and levels to manage potential impacts and to continuously refine model input • Continued testing of samples to test the Acid Rock Drainage potential of the Ripios material, and neutralising potential of waste rock, to verify the model results.
Land use	***No new findings since May 2012*** The Etango Project is located in the Namib Naukluft National Park (NNNP). The current land use is conservation and eco-tourism. Etango is located in close proximity to some of the park's most important tourist attractions, namely the Moon Landscape (dramatic landscapes), Swakop River (dramatic landscape and linear oasis for plants and animals) and Welwitschia flats (home to one of the largest populations of Welwitschia in the world). Mining has the potential to conflict with land uses such as conservation and eco-tourism, during the Life of Mine.

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Table 20.2 **Summary of Biophysical and Human Environment Impacts**	
Biophysical/Human Element	**Impact**
Air quality	***No new findings since May 2012*** Hourly average wind speed, wind direction, temperature and rainfall data was measured over a four year period and dust fallout data and ambient PM10 concentrations have been collected for more than 2 years. The SEA found that 82% of background PM10 concentrations and dust fallout in the region could be attributed to natural windblown dust while vehicles contribute approximately 13%. The impact assessment was done through dispersion modelling of Total Suspended Particulates, PM10 (particulate matter with an aerodynamic diameter of less than 10µm which has potential for human health risk, carbon monoxide (CO), diesel particulate matter (DPM), oxides of nitrogen (NOx) and sulphur dioxide (SO_2). Mitigation measures considered for each phase include: • Crushing and Screening – Hooding with scrubbers, 75% control efficiency • Drilling – Water sprays, 70% control efficiency • Materials handling – Water sprays, 50% control efficiency • Unpaved roads – Water sprays and(or) dust suppressants, 75% and 90% control efficiency respectively • Wind-blown dust – Natural crusting, 30% control efficiency.
Radiation exposure	***No new findings since May 2012*** The radiological safety and impact of the Etango Project to members of the public was evaluated in terms of two exposure conditions; namely a Tourist Exposure Condition and a Farmer Exposure Condition. These two exposure conditions evaluated the contribution of the atmospheric pathway; an additional variation was also considered for the Farmer Exposure Condition where in addition to the atmospheric pathway, a groundwater pathway was also considered. Model results suggest that public exposure to radiation is well below international limits. No specific mitigation measures are required to reduce the levels of radiation exposure to members of the public. However, as the atmospheric pathway contributes to radiation dose, the mitigation measures to reduce dust levels, proposed in the air quality impact assessment must be applied to also maintain radiation doses to a minimum. In addition, Bannerman will work according to a Radiation Management Plan, as per the required legislation (which is already in force for the exploration phase of this project).
Biodiversity	Biodiversity (i.e. the plants and animals living in an area and the ecosystems that support them) will be affected by mining activities. The process plant site and the majority of Anomaly A pit are located within the flat sandy gravel plains with shallow ephemeral washes. The

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Table 20.2
Summary of Biophysical and Human Environment Impacts

Biophysical/Human Element	Impact
	northern pit extension of Oshivelo and Onkelo are situated in deep valleys with steep, largely unvegetated slopes but with washes that run through them that support diverse vegetation. Important ecological drivers supporting the ecosystem and habitats in this EPL are rain, fog, rivers and springs and nutrient/food resources. The main impacts include a loss and fragmentation of habitats, reduced plant and soil crust productivity, interference with animal movements, possible blocking of surface flows in ephemeral water courses and increased poaching. These impacts can all cumulatively reduce the populations of animals in the area. Dust fallout and long-term radiation dust, remote noise, availability of permanent water and vehicular activity are likely to have direct negative impacts on biodiversity which are rated as moderate or low. The mine could potentially destroy up to an estimated 10% of the habitat (and therefore population) of a newly discovered gecko species which is ecologically and taxonomically unknown. This is potentially an impact of moderate significance. Similarly, mining will have negative direct and cumulative impacts on the total population of numerous identified, unidentified and unknown invertebrate species, some of which are classified as endangered. This impact is therefore major and remains so, even with mitigation as the Precautionary Principle requires that significant reduction of a range restricted habitat for certain species is assumed until further information proves otherwise. Mining will reduce the populations of Swakopmund Commiphora (Commiphora oblanceolata) that occurs in the Onkelo and Oshiveli deposit areas and their rescue and relocation should be considered. Proposed mitigation measures can help reduce the severity of these impacts.
Socio-economic	***No new findings since May 2012*** The mine's construction will generate 800 to 1500 short-term jobs over a 2 year period. Mining operations will generate 1000+ direct jobs and an estimated further 1600 jobs created by suppliers and contractors of goods and services to the mine over its 14 year lifespan. The mine will therefore contribute to Namibia's economic and social upliftment, in a country where unemployment is estimated at 51.2%. Skills acquisition/upgrading will provide greater opportunities for the local labour force to participate in the project and will make a crucial contribution towards long-term sustainability of employment in the area, beyond the life of mine. The inward migration of employees and job-seekers will increase pressure on the availability and adequacy of education and health services. Affordable housing is already a major issue for residents.

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Table 20.2 **Summary of Biophysical and Human Environment Impacts**	
Biophysical/Human Element	**Impact**
	It is possible that the loss of sense of place in the national park may negatively impact on tourism in Swakopmund and on guest farms neighbouring the mine. Bannerman is working closely with the local tourism sector to minimise the negative impacts of mining on the tourism industry. Mitigation measures could ultimately result in increased tourism to the region but at this stage, as the Uranium Rush may have cumulative impacts on tourism, the overall residual impact on tourism is rated of minor negative significance.
Archaeology and cultural	***No new findings since May 2012*** Four archaeological sites located within the proposed site provide links to the overall picture of the re-population of the Namib Desert in the late Pleistocene and Middle Stone Age. They are regarded as not important sites and have a low or indirect threat regarding the acceptable vulnerability ranking used. Once the final layout has been proposed, a further site study is required to ensure that no archaeological site is destroyed without recording.
Visual	***No new findings since May 2012*** The visual specialist study incorporates the environmental objectives and mitigation measures proposed in the central Namib 'Uranium Rush' Strategic Environmental Assessment (SEA). A viewshed analysis was undertaken for the major project activities and it was found that the zone of visual influence for the combined project would be strong and that the following receptor points would be impacted: ▪ C28: Main access route to the area ▪ Welwitschia Flats: SEA red flag area ▪ Swakop River: SEA red flag area ▪ Moon Landscapes viewing point: SEA red flag area ▪ D1991 southbound: due to proximity to Moon Landscape red flag area. For most activities, the visual objective can be met with mitigation. The exceptions are the lights at night, blasting and placement of certain Waste Rock Dumps. The current night time context is associated with a 'dark sky' sense of place and the combined lighting from the mine will probably create a pool of light which may alter the current dark sky sense of place. The height and movement of blast plumes will be clearly visible to the surrounds and would change the visual sense of place within the foreground/middle ground zone when blasting takes place. Bannerman is proposing lunchtime blasting, to minimise disturbance to its workforce, neighbours and tourist activities in the area. The visual specialist study has also recommended heights above the horizon as seen from the Moon Landscape and geometries of the outside facades of certain Waste Rock Dumps to limit the visual impacts from, in particular the Moon Landscape.

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Table 20.2 Summary of Biophysical and Human Environment Impacts	
Biophysical/Human Element	**Impact**
	The Welwitschia Flats and the Moon Landscape area to the north of the Swakop River were highlighted by the Central Namib Uranium Rush SEA as tourism and biodiversity red flag areas, requiring that the existing character of the landscape be preserved and that level of change to the characteristic landscape should be very low and must not attract attention. However, both these receptors do fall outside the 6km foreground zone where landscape modifications are less visible and as such have less potential to influence the landscape character in these areas. The Oshiveli and Onkelo pits, which are located in the Swakop River area, would generate major impacts during construction and operation. With mitigation, these impacts can be reduced to moderate levels should mitigation be effectively implemented but it must be realised that the existing wilderness sense of place would be lost. The Etango Project will adhere, as far as possible, to all corridor plans of the SEA to minimise visual and bio-diversity impacts.
Noise	***No new findings since May 2012*** The present ambient noise levels in the environment of the Etango project tend to be extremely low to very low, i.e. typically in the range of 20dBA to 40dBA, depending on the wind speed and presence of human activity in the area. The extremely low ambient noise levels contribute to the sense of place in this desert environment. Noise will come from in-pit mining, mineral processing operations, the haulage of ore and waste rock, and road and rail traffic. Noise impact will be greater during the night, when sound travels further and background noise levels tend to be at their lowest. During construction the extent of the noise impact will be quite considerable due to the fact that all the activities will be above ground level and there will be no acoustic screening by pit walls or waste rock dumps. However, since the resulting ambient noise levels are well below the guideline limits the noise impact is still considered to be low. Operational activities would result in a moderately significant, negative direct impact on pre-development ambient noise levels. It is most unlikely that any of the continuous noise emissions during all the investigated periods will be 'audible' at the Welwitschia Plains. During decommissioning there will be no increase in ambient noise levels at the nearest noise sensitive points. Research on the impact of noise on wildlife suggests that the noise

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Table 20.2 Summary of Biophysical and Human Environment Impacts	
Biophysical/Human Element	**Impact**
	levels have to be much higher than those generated in the environment by the Etango mining operation. Mitigation measures include: • Effective maintenance program for all diesel powered equipment • Enclosure of noisy equipment and processes • Adherence to a published blasting schedule • The replacement of reverse hooters by flashing strobe lights or broadband noise sources, and the training of operators to prevent 'bucket slams' and similar events.
Sense of place	***No new findings since May 2012*** Sense of place is formed from a combination of attributes such as space, visual, noise, biodiversity and archaeology. The combined impacts (both direct and indirect) of all the activities taking place in and around the proposed Etango site have the potential to profoundly alter the way people perceive and use this section of the National Park, affecting both peoples' psyche and some people's livelihoods. The visual specialist study showed that the relocation by a few metres lower down from the current 'main viewpoint of the Moon Landscape' will shield the view of the waste rock dump. It is proposed to upgrade the site, i.e. providing benches, improved signage, and shade.

Cumulative impacts in the Erongo Region will be controlled and reduced only through the combined efforts of all the mines in reducing their individual zones of influence. To this end, it is of vital importance that Bannerman contribute significantly to the implementation of the Strategic Environmental Management Plan as defined in the Central Namib Uranium Rush SEA.

It is of critical importance that further mining is managed in such a way that it does not detract from the elements which define significant landscape character specifically relating to the eco-tourist industry within the region and the country.

20.8 SUMMARY OF IMPACTS ASSESSMENT FINDINGS

A summary of all assessed environmental and human aspects with major and moderate effects after mitigation measures is given in Table 20.3.

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Table 20.3
Summary of Impacts Assessment Findings

Impact / Environmental Quality Objectives (EQOs) from SEMP	Phase	Significance Pre Mitigation	Significance After Mitigation
Key: C = construction, O = operation, D = Decommissioning/post-closure, cum = cumulative			
Surface Water			
Restriction of Surface Water Flow/EQO 7/2	C, O, D	**Major negative**	**Moderate negative**
Air Quality			
Potential PM10 Health Impacts/EQO 4/2	O	**Major negative**	**Moderate negative**
	cum	**Major negative**	**Major negative**
Potential CO, DPM and SO2 Health Impacts	cum	**Moderate negative**	**Moderate negative**
Biodiversity			
Potentially significant reduction of the population of an undescribed Pachdactylus gecko species/EQO 8/1 EQO 8/2 EQO 8/3 EQO 8/4	C, O, D	**Moderate negative**	**Unknown, remains moderate negative**
Disturbance to and Reduction of Populations of Invertebrates at a Local and Regional Level		**Major negative**	**Major negative**
Loss of plants and habitat due to direct physical destruction/EQO 8/2	C, O	**Moderate negative**	**Moderate negative**
Morbidity/Mortality of plants and habitat degradation due to loss of surface and subsurface water flow/EQO 8/6 regarding water and EQO 8/1 regarding monitoring	C, O, D	**Major negative**	**Moderate negative**
Economic	**C, O**		
The direct economic impacts of the project are the sales of the products by the mine itself, the wages and salaries of the people directly employed, profits of the mine itself, as well as the taxes and royalties the mine pays/EQO 1/1 EQO 5/2		**Major positive**	**Major positive**
The indirect economic impacts of the project are the mine purchases for construction and operations (inputs) and their inputs, backwards down the supply chain, and other services bought etc. Induced impacts arise from the spending of wages – (greater on locally produced goods)/ EQO 1/1	C, O	**Major positive**	**Major positive**
Government revenue from VAT, BLNS, PAYE, SSC, WCF, Royalties; local council taxes and profits from providing utilities and services to residents (mine and supplier employees)/EQO 1/1	C, O	**Major positive**	**Major positive**

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Table 20.3
Summary of Impacts Assessment Findings

Impact / Environmental Quality Objectives (EQOs) from SEMP	Phase	Significance Pre Mitigation	Significance After Mitigation
Key: C = construction, O = operation, D = Decommissioning/post-closure, cum = cumulative			
Impact on tourism			
Closure of road between D1991 and Welwitschia Flats, closing circular route used mainly by tour operators and tourists; but also providing access route for direct neighbours/ EQO 9/1	C, O, D, P	**Major negative**	**Moderate negative**
Mine closure: Job losses, reduced business turnover of suppliers and service industries and retail businesses, reduced government revenue	D, P	**Major negative**	**Major negative**
Social			
Increased employment opportunities with the mining company and with suppliers of goods and services to the mine and wider communities; opportunities to expand skills in the labour force/EQO 5/2 EQO 6/1	C,O	**Major positive**	**Major positive**
On mine closure, loss of employment at the mining company and with suppliers of goods and services to the mine and wider communities/EQO 2/1	D, C	**Moderate negative**	**Moderate negative**
Increased demand for school services required for children of employees and other migrants leading to overstretched services – notably too few classrooms and competent teachers to deliver quality education/EQO 5/1	D	**Moderate positive**	**Moderate positive**
Promotion of best management practices that promote common interests and improved service delivery through collaboration with key stakeholders	C, O	**Moderate to Major positive**	**Major positive**
	D	**Minor positive**	**Moderate positive**
Visual			
Visual impact of Pit, Dust and Blasting	C	**Major negative**	**Moderate negative**
	O	**Major negative**	**Major negative**
	D	**Major negative**	**Major negative**
Visual impact of heap leach residue facility	O	**Major negative**	**Moderate negative**
Visual impact of waste rock dumps	O	**Moderate negative**	**Moderate negative**
Visual impact of access roads	C, O	**Moderate negative**	**Moderate negative**
Impact on pre-development ambient noise	O	**Moderate negative**	**Moderate negative**

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Table 20.3 Summary of Impacts Assessment Findings			
Impact / Environmental Quality Objectives (EQOs) from SEMP	**Phase**	**Significance Pre Mitigation**	**Significance After Mitigation**
Key: **C = construction, O = operation, D = Decommissioning/post-closure, cum = cumulative**			
levels			
Sense of Place			
Loss of sense of place due to visual impact characteristics for the mine site areas/ EQO10/1 EQO10/2	C, O	**Major negative**	**Moderate negative**
Loss of sense of place due to noise impacts/ EQO10/1 EQO10/2	O	**Major negative**	**Moderate negative**

20.9 ENVIRONMENTAL MONITORING

20.9.1 Groundwater

Analysis of pre-mining groundwater from 27 boreholes in the area has shown it to be highly saline (many sources are comparable in quality with seawater), with levels exceeding the WHO DWQG (2008) for As, B, Fe, Mo, Pb, U. None of the natural groundwater sources is currently fit for domestic, agricultural, or livestock use.

The uranium concentrations in the pre-mining groundwater are very much higher than the WHO drinking water quality guideline of 0.015mg/L: the median value (0.18mg/L) is about 10 times higher than the WHO limit and the 90 percentile value (1.6 mg/L) about 100 times higher.

A series of groundwater bore-holes is being monitored to establish a baseline of groundwater quality. This data will be used as a datum reference during production to monitor any effects of the operation on local groundwater quality.

Post closure, a number of these bore-holes will continue to be operated, and water quality analysed and monitored. The extent, frequency and bore-holes to be used will be selected during the mine closure phase, based on the data trends and any history of pollution.

20.9.2 Run-off

Surface run-off from all areas of the plant and works will be collected and reused in the process. Data from periodic samples will define the nature and extent of any pollutants in this water.

Site demolition and rehabilitation will end the need for run-off monitoring on the site, post closure, as the dirty water will be redirected to the abandoned open pit or evaporation ponds.

20.9.3 Waste and Heap Leach Waste Seepage

Seepage from the waste dumps and heap leach waste storage facility are collected in sump ponds on the downgrade side. During operations this seepage will be sampled and monitored for pollutant levels, and pumped back to the plant for recycling.

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Where feasible post closure, these ponds will be provided with drainage to the pit. This will provide a safe and effective evaporation facility for this water.

20.9.4 Ambient Dust

Ambient dust monitoring data will be collected from selected sample points around the site prior to construction, during construction and for the life of the mine. Ambient dust samples will be taken every 5 years for 15 years to test the effectiveness of the post closure rehabilitation measures.

20.10 ENVIRONMENTAL STUDIES – CONCLUSIONS

No substantive legislative, environmental or social impediments have been identified for development of Etango. The region already hosts a number of large uranium operations and uranium mining and processing is well understood in the local communities and Government authorities.

20.11 MINING LICENCE

A Mining Licence is required before mining may commence.

Bannerman submitted its initial mining licence application for the Etango Project to the Namibian Ministry of Mines and Energy in December 2009, based on the December 2009 PFS for open pit mining and heap leaching of the Anomaly A area within the Etango deposit.

Since that time, the mineral resource estimate for the Etango Project has expanded and the site layout and processing flowsheet have undergone changes. The ESIA was submitted in April 2012 and the latest Environmental Clearance Certificate was issue on the 5th of July 2015.

Bannerman lodged the supplementary information, including the 2012 DFS, with the Ministry of Mines and Energy in further support of the existing Etango mining licence application.

20.12 CLOSURE BOND

Currently there is no requirement for a closure bond to be posted.

Although no detailed closure plan yet exists, Bannerman has made provision to set aside a total of $32.5M for this purpose, including allowances for capping of dumps where possible during operations.

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21 CAPITAL AND OPERATING COSTS

21.1 OVERVIEW

Capital and operating costs have been determined by:

- Bannerman: Mining capital and operating costs, excluding mining infrastructure estimated by Amec Foster Wheeler. Owner's Costs, power, water and labour rates
- Amec Foster Wheeler: Plant and infrastructure capital and operating costs, excluding SX, precipitation, calcining and packaging plant capital costs for equipment determined by Bateman. Mining infrastructure requirements, Ripios pad and water management system quantities were determined by others but estimated by Amec Foster Wheeler.

All costs are quoted in US$ as of the 3rd quarter 2015. Where budget prices were obtained in currencies other than US$, the exchange rates shown in Table 21.1 have been used.

Table 21.1
Exchange Rates

Currency	Rate	Source
US Dollar to Namibian Dollar	US$1.00 = N$ 12.25	Bannerman
US Dollar to South African Rand	US$1.00 = ZAR 12.25	Bannerman
US Dollar to Australian Dollar	US$1.00 = A$ 1.28	Bannerman
US Dollar to Euro	US$1.00 = EUR 0.88	Bannerman
US Dollar to Japanese Yen	US$1.00 = YEN 124.00	Bannerman

21.2 CAPITAL COSTS

21.2.1 Mining Capital Costs

Introduction

The capital cost attributable to mining can be divided into mobile mine equipment, site infrastructure, and capitalised operating costs. Year by Year capital requirements are presented in Table 22.2.

Table 21.2
Etango Uranium Project – Miscellaneous Mine Capital
($M)

Year	Mobile Mining Equipment	Site Infrastructure	Capitalised Operating Costs	Total
-3			1.000	1.000
-2			3.803	3.803
-1	90.905	30.222	5.400	126.527
1	72.648	23.309		95.957
2	69.221	10.597		79.818
3	31.630	7.194		38.824
4	1.889	1.434		3.324
5	6.028			6.028

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Table 21.2 Etango Uranium Project – Miscellaneous Mine Capital ($M)				
Year	**Mobile Mining Equipment**	**Site Infrastructure**	**Capitalised Operating Costs**	**Total**
6	0.575			0.575
7	0.531			0.531
8	10.508			10.508
9	6.242			6.242
10	1.369			1.369
11	11.492			11.492
12	9.337			9.337
13				
14	3.212			3.212
15				
16				
Total	**315.587**	**72.756**	**10.203**	**398.547**

Mobile Mining Equipment

Major mobile equipment used directly in the mining operation such as trucks, excavators, drills, tracked and tyred dozers, graders, and water carts were estimated in accordance with the mine schedule.

The mobile equipment requirements were determined to match the mining requirements. A schedule of fleet build-up and replacement was developed to determine the capital cost requirements over the project life.

Note that for the financial model mobile equipment capital was expended in the prior year to reflect uncertainties in delivery.

Request for quotations (RFQ) documentation was sent to South African-based vendors (of the equipment) for quotations. Cost estimates for the major mobile equipment include all manufacture, transport, insurance, assembly, and commissioning costs. Excluded from the quotation was any local tax due. Costs of the equipment were quoted in $, with the vendor providing exchange rate assumptions.

RFQ returns were assessed for completeness and competitiveness. Based on the outcomes of the assessment, two vendors were selected: A vendor to supply equipment for drilling activities; and one vendor to supply equipment for load, haul, and ancillary activities. Work continued with the nominated vendors to ensure the cost structure is accurately reflected in the Study.

The capital costs for minor equipment such as lighting plants, support equipment such as heavy maintenance trucks required for field support services, and other support mobile equipment such as small equipment for other support activities (such as stemming or tyre maintenance) are based on the DFS estimates.

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Like the minor equipment, light vehicle and transport requirements from various population centres have been included in the mobile equipment costs.

Site Infrastructure

As part of the RFQ process, market prices for key consumables were requested from Southern African suppliers. RFQ documentation was assembled for mine tyres, diesel, lubricants, and explosives. In addition to supply of goods, onsite storage infrastructure was included as part of the RFQ process.

Based on mine plan requirements, surface roads within the mine for the first 3 years of operation were estimated by Amec Foster Wheeler. These estimates were based on the design width of surface roads and provided the total amount of cut, cut that would require blasting, and fill that would be required to establish the mine road network. A cost for road establishment is included in the site infrastructure costs. An allowance of $3M has been included for the road network establishment required for operations. Additional road construction and maintenance activities required (after the mine operation is established) are covered in the operating costs.

In the northern portion of the mine, bench establishment costs will be incurred. Since all bench establishment will occur within the open pit boundary, these costs are a premium on top of the mine costs already established in the operating costs. An allowance has been added for the costs of pioneering works for bench establishment.

With the exception of the fuel farm, lube storage, explosives plant and magazines; site infrastructure was estimated as part of the Amec Foster Wheeler scope of works and is discussed in Section 21.2.2.

Other miscellaneous capital requirements included within the site infrastructure are based on 2012 DFS estimates. These costs include the costs for drilling activities, technical services warehouse fit-out and first fills, and sustaining capital.

Drilling activities include geotechnical and grade control. An allowance for geotechnical drilling into the Etusis formation has been included. Geotechnical drilling costs and timing ensure sufficient lag prior to excavation of the final hanging wall. Ongoing grade control costs include the capital required for the purchase of gamma-logging equipment, truck scanners, and ongoing RC drilling and laboratory analysis.

Technical services costs (above those described above in drilling activities) includes the cost of off-the-shelf purchase of a dispatch and high precision GPS systems for excavators and drills; specialised mining software; and survey equipment required for ongoing mining operations.

Warehouse and first fills included allowances for the initial purchase of shelving, mine haul tyres, ground engaging tools, lubricants, fuel, and other initial stock required.

Capitalised Mine Operating Costs

Detailed engineering costs include an allowance for the completion of works associated with the design of the open pit and completion of supply contracts make up part of the capitalised operating costs.

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The remainder of the capitalised operating costs for the mine include continued pre-production labour required to oversee the mining start-up in the first 2 years prior to operations.

21.2.2 Process Plant and Infrastructure

Introduction

The capital cost estimate for the process plant and site infrastructure was developed with an Engineering, Procurement and Construction Management (EPCM) contracting strategy in mind. The plant design is based on a heap leach facility with final treatment via an SX plant, through to final product packaging.

Work undertaken by Amec Foster Wheeler includes all costs associated with the process plant from ROM bin to discharge pipeline into the SX plant, reagents storage facility, all associated infrastructure at the Mine and Walvis Bay port, temporary services and facilities for construction, first fills and spares.

Excluded from this part of the estimate are:

- All costs associated with mining other than mining infrastructure (see sub-section 21.2.1)
- Capital costs for external infrastructure comprising power supply and water supply
- Owner's costs relating to corporate, management and administration costs associated with the operation, as well as costs associated with capitalised operating costs for operating and support staff employed pre-production.

All costs are presented in US$ at 10 August 2015.

The estimate for the plant and infrastructure scope of work is $635.3M. In Addition Bannerman has increased the plant mobile equipment requirement adding $0.92M bringing the total estimate to $636.24

Estimate Categories

The capital cost estimate is structured to encompass the following major categories:

- Direct Costs: Expenditures incurred during the construction of the process plant and infrastructure. The costs include materials and equipment, freight to site, construction labour and equipment (including contractors' supervision, overheads and profit), temporary construction facilities, construction mobile equipment, accommodation of construction labour, and contractor mobilisation and demobilisation
- Indirect Costs: Expenditures for engineering design, procurement, project management, site construction management and commissioning supervision by the EPCM contractor and its consultants. The indirect costs include appropriate allowances for the EPCM contractor's overhead contribution
- Accuracy Provisions/Growth Allowances: Accuracy provision or growth allowances are included within an estimate to cover unknown but expected increases in quantity and costs following detailed design
- Escalation: Excluded from the estimate

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- Contingency: Additional to the base cost estimate to cover unforeseeable elements of cost, risk and uncertainty within the defined scope of work.

Estimate Summary

The estimated total costs are summarised in Table 21.3.

Table 21.3
Capital Cost Estimate – Plant and Site Infrastructure, Summarised by Area

Area No.	Area Description	Direct Man hours (Hours)	Total Cost ($M)
00000	General	7,980	10.23
03010	General Site Works	262,398	15.01
03011	Primary Crushing	292,387	32.76
03012	Stockpile and Secondary Crushing and Screening	323,072	47.29
03013	Tertiary Screening and Crushing	247,286	48.34
03021	Agglomeration	114,165	17.53
03022	Stacking and Reclaiming	182,608	54.37
03030	Leach Residue Stacking	103,704	32.16
03040	Heap Leach Solution Handling	526,637	36.62
03051	Heap Leach Solution Clarification	32,054	7.72
03052	Solvent Extraction	294,629	36.48
03061	Precipitation	42,111	6.20
03062	Calcination and Packaging	26,725	14.39
03080	Peroxide	11,363	0.87
03081	Diluent, Extractant and Modifier	3,811	0.31
03082	Ferrous Sulphate	4,560	0.43
03083	Coagulant	3,603	0.29
03084	Sodium Hydroxide and Sodium Carbonate	6,800	0.60
03085	Ammonia	7,089	2.41
03086	Binding Agent	16,414	2.77
03087	Flocculants	4,868	0.58
03090	Water distribution	75,681	9.26
03091	Air Distribution	28,185	2.86
03092	Diesel and Power Generation	4,397	0.38
03100	Sulphuric Acid Handling at Plant Site	110,062	7.20
03130	Electrical Distribution Process plant	50,496	9.97
03140	Communications	22,923	5.11
03150	Process Controls	4,224	1.61
03450	Buildings at Plant Site	151,806	22.43
03800	Spares	0	17.30
03810	First Fills and Opening Stocks	0	16.71
03900	Temporary Services and Facilities	23,400	20.15
03910	Temporary Construction Camp	156,750	12.53

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Table 21.3 Capital Cost Estimate – Plant and Site Infrastructure, Summarised by Area			
Area No.	**Area Description**	**Direct Man hours (Hours)**	**Total Cost ($M)**
03920	Mobilisation and Demobilisation	0	6.77
03930	Vendor Representatives	21,894	5.36
03990	Owner's Costs – Pre-production	0	2.48
10300	Facilities at Port Site	34,132	3.11
	Direct Costs – Subtotals	**3,198,132**	**509.82**
	EPCM		63.43
	Contingency Allowance		61.30
	Totals		**635.32**

On review, Bannerman has adjusted the mobile equipment list and added $0.92M for additional equipment bringing the total Capital cost to $636,2M

Estimate Accuracy

The estimate has been prepared in accordance a targeted accuracy of ±20%. In order to achieve the targeted accuracy the following was completed:

- Level of engineering 15 to 25% complete
- Multiple quotes sourced for equipment and bulk materials supply
- Detailed material take-offs (MTOs) prepared for all bulk materials
- Labour rates based on information received from contractors and industry agreements
- Labour productivity calculations based on information from contractors currently active in the region
- Indirect construction costs, including temporary facilities and construction support, calculated in detail.

EPCM costs calculated at high level.

Estimate Methodology

The capital cost estimate has been assembled using the general methods of development described below:

- Earthworks:
  - Unit Rates: Based on actual pricing from contractors recently active in the region of Namibia
  - Original quantities: Determined from material take-offs based on general arrangement drawings. Bannerman appointed consultant, SLR, provided the quantities for the Ripios storage pad. Quantities for the SX plant area were provided by Bateman Engineering.
- Concrete:

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- Unit Rates: Based on actual pricing from contractors recently active in the region of Namibia
- Original quantities: Determined from material take-offs based on general arrangement drawings. Quantities for the SX plant area were provided by Bateman.

- Structural Steelwork Fabrication and Installation:
  - Unit Rates: Based on actual pricing from contractors recently active in the region of Namibia
  - Original quantities: Determined from material take-offs based on general arrangement drawings. Quantities for the SX plant area provided by Bateman.
- Platework Fabrication and Installation:
  - Unit Rates: Based on actual pricing from contractors recently active in the region of Namibia
  - Original quantities: Quantities for platework determined from material take-offs or based on similar structures proposed on past or current engineering studies. Quantities for the SX plant area provided by Bateman.
- Equipment Installation:
  - Unit Rates: Installation man hours were derived from three sources: Current budget pricing from installation contractors active on similar projects in the region; Amec Foster Wheeler database information for unit man hours for installation of equipment and installation information provided by equipment suppliers
  - Quantities and specifications: Determined from the latest available mechanical equipment list plus additional allowances for minor equipment items.
- Equipment Supply Costs:

  Equipment costs are current as at Q3 2015. Multiple source quotes were sought for some major equipment items. Updated pricing for balance of equipment costs was sourced from updated vendor pricing from original preferred vendors.

- Pipework:

  Quantities for piping were determined from original material take-offs of general arrangement drawings. Unit rates were based on current database information from various suppliers. Original quantities for SX plant area provided by Bateman.

- Electrical and Instrumentation

  The electrical estimate was developed from the mechanical equipment list, electrical load list, electrical equipment list, single line diagrams and electrical general arrangement drawings. An instrumentation and control valves list is also integrated into the estimate. The use of fibre optic back bone, process control system, CCTV, access control system, fire alarm system and radio communication system block diagrams were also used to develop the estimate.

  Updated budgetary pricing was obtained for major electrical equipment from original preferred vendor, with in-house historical database pricing used for minor equipment.

  Electrical bulk material quantities are compiled using of electrical layout drawings and priced by estimating using recent project database information.

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Installation man hours were updated based on Amec Foster Wheeler's Australian database values and recent information from contractors active and familiar with the region. A productivity factor for local site conditions has been applied.

- Buildings:
  - Unit Rates: Based on budget pricing information from contractors with experience on similar projects in the region.

    Building costs include allowances for building fit-out, loose furniture, etc. Fit-out of the laboratory is covered under mechanical equipment supply.

    Buildings allowed in the estimate include port offices and ablutions, lime storage, administration offices, change house and laundry, plant laboratory, emergency services centre, covered parking area, main gatehouse and security gatehouse, central control room, crushing and screening amenities, mobile crib rooms, ablutions for offices, mine and warehouse offices and ablutions, maintenance workshop, warehouse, reagents store, dangerous goods storage, core shed, heavy and light vehicle workshop, rubber lining, hoses and belting workshop, and tyre and battery store and workshop.
- Freight:

  Freight allowances are based on information provided by suppliers or equipment fabricators, or as historical information for projects in Namibia. Allowances range between 3% and 10% of supply value, depending on source of equipment or material.

  It is assumed that, where possible, all bulk materials and most of the equipment will be sourced locally in Namibia or from the southern African region.
- Labour Rates:

  Composite labour costs are determined per discipline and per work package. The installation costs are based on estimated crew man hours priced at crew rates developed for each of the disciplines.

  Labour rates are inclusive of labour base rate plus direct labour on-costs (including wages, pension, payroll tax, insurances, fringe benefits, accommodation, travel time and R&R allowances); construction tools and equipment (including cranage); indirect labour (including supervision, management, etc.); site facilities (including offices, storage, ablutions, communications and scaffolding); miscellaneous costs (including consumables, maintenance and services); and home office costs including administration.

  The base labour rates are based on information provided by contractors in their budget pricing proposals for a similar project, and the Construction Industries Federation of Namibia for local labour.

  The labour rate is based on a 55 hour week. Allowances for site accommodation, travelling and R&R costs have been built into the composite labour rate.

  It is assumed that labour will be sourced, in the first instance from local sources in Namibia. Thereafter labour will be sourced from Southern Africa. For this estimate, it is assumed all unskilled and semi-skilled labour will be sourced locally and that no accommodation is required for this labour. A lunch meal allowance is included for all direct labour on site.
- Labour Productivity Factor:

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The unit man hours applied to the estimate quantities for all disciplines are based on Australian norms for engineering construction, factored to suit local conditions.

Appropriate productivity factors were applied per discipline, taking account of geographic location, level of skills, level of unionisation of site, project complexity, climatic conditions, cultural considerations, etc.

Productivity factors ranged from 1.3 (earthworks) up to 3.0 (piping installation), averaging 2.4 overall.

- First Fills and Spares:

  First fill reagents and consumables have been assessed to suit requirements for first 30 days of production, with the exception of sulphuric acid and binder where a different ramp up period is assumed.

  Spares allowance are based on information provided by suppliers in their budget proposals, determined by mechanical engineering, or (for minor equipment) an allowance of 4% for capital spares and 1% for commissioning spares.

- Assistance with Commissioning of Plant (Direct Labour):

  An allowance has been made for a crew for 12 weeks, to cover minor modifications, improvements, changes, etc., related to safety, operations enhancements and related Client requirements.

- Temporary Construction Services:

  Temporary Construction Services are based on a detailed assessment of requirements for temporary services, facilities and consumables for the 30 month construction period. Major services and facilities include security and medical services, maintenance of roads, services and temporary structures, temporary power supply (diesel generators), diesel and diesel storage facilities, offices, ablutions, stores, crib rooms, etc., including fit-out, site communications system, EPCM contractor vehicles and general EPCM stores vehicles, waste handling and disposal, transport on site, and messing and accommodation of EPCM site-based team.

- Heavy Lift Cranage:

  No allowance for heavy lift cranage is included in this estimate. The cost of contractor's use of cranes is built into the all-in labour rate, and, in addition, the project's 400t crane will be available to assist with heavy lifts or long reach lifts, if required.

Engineering, Procurement and Construction Management

The EPCM estimate is based on a high level assessment of personnel man hours and expenses required to support project construction. The estimate includes for mobilisation and demobilisation of the EPCM contractor's workforce and consultants required to supplement design engineering and construction.

Engineering and drafting man hours are based on engineering deliverables while project management, procurement, construction management and commissioning management man hours are time-based according to the implementation schedule.

Current market rates which include overheads recovery and margin for the EPCM contractor have been used to price these services.

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The expenses provision within the EPCM cost estimate include costs such as, project office rental and outgoings, utilities, couriers and postage, reproduction of documents, stationery, entertainment, computer hardware and software, travel and accommodation of personnel in transit. On-site accommodation of personnel is covered elsewhere under temporary services and facilities.

Estimate Accuracy Provision / Growth Allowance

The accuracy provisions reflect the level of definition available relating to the scope of work, process design, conceptual engineering design and cost data at the time of the capital estimate development, and make appropriate allowances for uncertain elements of cost, for estimating errors and omission in quantification, thereby reducing the risk of cost variation within the required accuracy level.

The accuracy provisions are an integral component of the capital cost estimate and must be considered as part of the overall costs necessary for implementation of the project. This allowance is not intended to cover contingency issues such as, abnormal or inclement weather, acts of God, industrial disturbances, etc. Provision for these major undefined issues are included separately as a 'below-the-line' item in this estimate.

The accuracy provisions have been assessed at discipline level on a line-by-line basis to reflect the level of accuracy of material take-offs and design detail available at the time of the estimate. The overall accuracy provision is $45.21M, representing 9.7% of direct costs and is included in the direct cost in Table 22.4.

Owner's Costs

The Owner's cost estimate takes account of costs for the Owner's project team, pre-production recruitment/manning, training, housing allowances, environmental site assessments and monitoring during construction, Swakopmund support, insurance and consultants, as summarised in Table 21.4. Costs also include allowances for geotechnical and sterilisation drilling in support of construction, and further metallurgical test work on site.

Table 21.4
Summary of Owner's Costs

Item	Cost ($M)
Owner's Project Team	9.96
Corporate, Perth	1.80
Owner's Pre-production Staff	6.73
Capitalised Recruitment costs	1.52
Training and Training Manuals	4.00
Consultants	1.46
Housing Allowance	6.00
Environmental Site Assessment	0.92
Swakopmund Support	0.81
Insurance	3.62
Drilling – geotechnical and sterilisation	2.00
Metallurgical Test work (on site)	0.10
Total	**38.90**

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Contingency Allowances

- Project Contingency:

  Project contingency is an allowance additional to the base cost estimate to cover unforeseeable elements of cost, risk and uncertainty within the defined EPCM contractor's scope of work. This is an allowance to cover possible costs that cannot be explicitly foreseen or described at the time the estimate is prepared due to lack of complete, accurate and detailed information.

  An allowance for project contingency was included in the Amec Foster Wheeler estimate, representing 12% of Direct costs or $61.30M (Table 21.3). However, it was omitted from the capital cost in the base case financial model.

- Owner's Contingency or Management Contingency:

  Owner's or management contingency is sometimes included in capital cost estimates, to cover such risks as:

  – Changes of scope
  – Exchange rate variations
  – Escalation on materials supply, labour and fuel costs
  – Possible industrial relations disputes
  – Abnormal market conditions that cause unexpected rate increases or shortages of skilled manpower for either engineering design or construction
  – Unforeseen shortages, or abnormal cost increases, of construction materials, reagents, consumables, etc.
  – Abnormal weather impacts or delays
  – Unforeseen environmental or social constraints
  – Schedule impacts from late delivery of critical equipment items
  – Unforeseen geotechnical issues
  – Unforeseen changes in legislation.

No allowance for Owner’s contingency was made in the capital cost estimate in base case financial model.

Estimate Assumptions and Qualifications

The following assumptions and qualifications are to be considered for this capital cost estimate:

- Water will be available on site for use by installation contractors
- Ground conditions based on a preliminary geotechnical report are suitable for standard concrete equipment support structures, with no requirement for piling or special foundations

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- Backfill material, sand and aggregates for concrete, site earthworks, roadworks, ponds and pads are available within 5km of the site and at minimal or no cost
- Roads are to a minimum standard with no kerbing and limited stormwater drainage
- Pipe-racks are allowed within plant area and at road crossings. Overland pipelines are to be placed directly on ground, with no allowance for overland sleepers
- The plant operation is predominantly managed from a central control room with support from field operators.

Sustaining Capital Costs

There is limited sustaining capital required for the plant and infrastructure. A sum of $7.3M is recovered from the sale of the construction camp on construction completion, and $15.2M expended for first fill in the capital estimate is recovered at the end of the project life.

Closure and Rehabilitation Capital Costs

A detailed Closure plan will be developed at a later date as part of the ESMP, but high-level consideration has been given to the closure requirements. Bannerman intends to set aside a total of $32.5M for this purpose, including $2.5M in allowances for capping of dumps during operations.

Given the desert environment, scant flora and fauna and poor quality of the existing groundwater, combined with the low acid and metal generating potential of run-off and seepage, this closure cost is considered reasonable.

21.2.3 Total Project Capital Cost

The total Project capital cost estimate as used in the base case financial model is $1,075.14M, comprising $792.7M in pre-production capital and $282.487M in deferred, sustaining and closure capital (Table 21.5).

This cost incorporates the Amec Foster Wheeler plant and infrastructure estimate, except for the Project Contingency of $61.3M recommended by Amec Foster Wheeler. It also includes the estimate for Mining capital and Owner's Costs prepared by Bannerman.

Some components of the Amec Foster Wheeler estimate have been handled differently in the financial model, for example:

- Process plant Directs have been separated from Site Infrastructure Directs
- Port Facilities costs appear under External Infrastructure
- Miscellaneous category comprises Amec Foster Wheeler's First Fill, Spares, Mobilisation/ Demobilisation and Commissioning costs
- Indirect costs in the model comprise Amec Foster Wheeler's Temporary Services and Facilities, Temporary Construction Camp, Vendors Representatives, Owner's Pre-production costs and EPCM costs
- Amec Foster Wheeler's Accuracy Provision costs have been moved from Direct Costs into Indirects.

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It should be noted that the Total Project Capital Cost excludes Working Capital requirements, which are however included in the financial model. There is no provision for contingencies.

Table 21.5
Project Capital Cost Expenditure Summary ($M)

Area	Pre-production	Sustaining	Total
Mining	131.33	267.22	398.55
Process Plant	321.36	-	321.36
Site Infrastructure	74.79	4.51	79.31
External Infrastructure	46.00	0.75	46.75
Miscellaneous	37.77	-15.19	22.57
Indirects	142.51	-7.30	135.21
Owner's Costs	38.90	32.50	71.40
Owner's Contingency	-	-	-
Total Project	**792.65**	**282.49**	**1075.14**

21.3 OPERATING COSTS

21.3.1 Introduction

The operating cost estimate for Etango has been assembled by quarters for the first two years and annually thereafter. The operating costs include mining, processing, utilities, consumables, maintenance, labour, general, office, site and external infrastructure and administrative costs.

Costs are expressed in US$ as of September 2015.

Exchange rates were provided by Bannerman and are reported in Section 21.1.

Operating costs are estimated to an accuracy of ±20%.

21.3.2 Contributors

Contributors to the operating cost estimate were as follows:

- Mine operating costs: Bannerman
- Plant and site infrastructure operating costs: Amec Foster Wheeler, including equipment specified by Bateman
- External infrastructure (power and water) costs: Bannerman
- Owner's (G&A) costs: Bannerman.

21.3.3 Operating Cost Estimate Summary

The estimated annual operating costs are presented in Table 21.6. These costs average $14.15/t processed or $37.99/lb U_3O_8 produced.

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Table 21.6
Operating Cost Summary

Area	LOM Operating Cost ($M)	Unit Operating Cost ($/t Processed)
Mining	1934.72	6.38
Process plant	2059.56	6.79
Infrastructure	5.56	0.02
Owner's Cost (G&A)	291.22	0.96
Total	4291.1	14.15

21.3.4 Mine Operating Costs

Summary

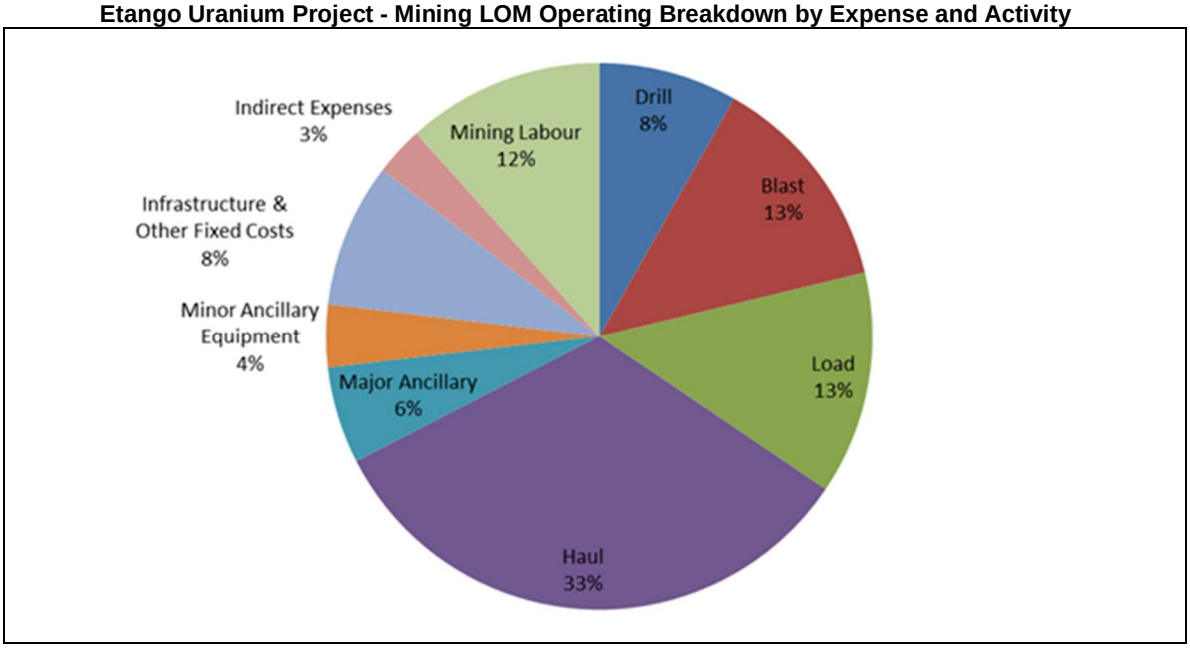
Operating costs are based on market available data for Q3 2015.

The Study considers that mining equipment will be maintained by a supplier through a MARC. MARC is split into two components – a variable component based on equipment hours, and a fixed component covering the labour, overheads and depreciation of the fixed plant as set out in the MARC contract.

In addition to the mobile equipment MARC, explosives and fuel supply contracts are included in the study. The scope of works for the explosives contract includes turnkey bulk plant, magazines, and transport and manufacture of bulk explosives 'on bench'. Fuel supply includes on-site depot for fuel and lubes, transport and management.

Fuel, variable MARC, and explosives make up the majority of the direct expense of the mining costs. A breakdown of the unit cost of mining is provided in Figure 21.1.

Figure 21.1
Etango Uranium Project - Mining LOM Operating Breakdown by Expense and Activity



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Major Consumables

The fuel price was determined from the Engen RFQ submissions. Applicable taxes, tariffs, duties, and bulk purchase discounts were applied. Lubricant costs were part of the RFQ.

Explosives costs were based on the BME RFQ costs. Bulk explosives prices are based on constituent costs of ammonium nitrate and emulsion costs.

Variable maintenance rates for major mobile equipment were obtained thru the MARC RFQ process.

Local mining labour and mining admiration labour costs were based on current salaries paid by Bannerman in Namibia along with a loading of 60%. The costs of non-Namibian labour is based on current Australian rates accounting for on costs and adjusted to US$ at the DFS exchange rates (see Table 21.1).

Tyre costs are based on submissions from Bridgestone.

Minimal power is required by the open pit. Power costs are covered in Section 21.3.5.

Other Consumables

The costs of ground engaging tools (GET) and wear items were based on RFQ submission from Komatsu and Sandvik.

Equipment Ownership Costs

Ownership costs (less insurance) are included as equipment capital costs.

Equipment Operating Assumptions

The following operating assumptions were used for equipment:

- Loading parameters, including material density, moisture content and material swell based on geological information
- Equipment availability based on RFQ provided guarantee percentages
- Bucket fill factors based on industry experience
- Excavator load cycle times based on RFQ performance data and industry standards
- Truck cycle times based on an industry standard computer mine truck modelling package (TALPAC)
- Fuel burn rates as supplied by the RFQ
- Truck cycle time parameters (rolling resistance, speeds, and spot/dump times) based on industry standards.

Drill and Blast

Drill and blast costs were developed by Bannerman. Powder factors are estimates, based on the results of the Kuz-Ram fragmentation analysis, geotechnical assessment, and crusher requirements done during the DFS. These were reviewed during the OS and deemed to be

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appropriate. Drill production rates are 28m/h irrespective of a 203mm production hole or 165mm trim hole and are consistent with industry standards.

Mine Administration

The costs of mine administration include mine supervision, technical services, and administration.

The majority costs for grade control are carried as capital items, with the operating costs covering the manning and the ongoing operating costs of gamma logging.

Equipment Operating Costs

The LOM average equipment costs for the major equipment are displayed in Table 21.7.

Table 21.7
Equipment Costs – $/SMU (Excluding Labour Cost)

Model	Equipment	Total	Variable Maintenance	Fuel	Lube	GET	Tyres	Rebuild
Sandvik D55SP	Drill	199	43	71	8	77		0
Komatsu PC5500	Excavator	795	188	285	19	294		10
Komatsu 830E-AC	Truck	245	75	123	9		31	7
Komatsu WA1200	Wheel loader	348	81	108	9	87	57	6
Komatsu WD600	Tyre dozer	103	25	53	5	5	16	0
Komatsu D375A-6	Track dozer	79	21	42	2	14		0
Komatsu GD825A-2	Grader	55	27	16	2	6	5	0
Komatsu HD785-7	Water cart	92	39	39	4		9	1

21.3.5 Plant and infrastructure Operating Costs

Summary Costs

The plant operating costs developed by Amec Foster Wheeler are summarised in Table 21.8 and are reported as functions of total annual cost and cost per tonne of ore crushed. The table shows that acid is the major cost item, followed in turn by other reagents, power and labour costs.

Reagents

The reagent consumptions are based on the mass balance and design criteria. Quotes from vendors were obtained for the unit costs for each reagent, with the exception of diesel where the price was determined from the Engen RFQ submissions.

The major reagent cost is for sulphuric acid, for which two quotes were received and the average of the two quotes was used for the OPEX.

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Table 21.8 Summary of Plant and Infrastructure Operating Costs (base on LOM average throughput rate)			
Item	**Cost ($M/a)**	**Cost ($/t of ore)**	**% of Cost (%)**
Acid	33.29	1.72	25.4
Reagents	19.10	0.99	14.6
Power	26.89	1.39	20.5
Labour	10.51	.054	8.0
Maintenance Materials	16.01	0.83	12.2
Water	15.94	0.83	12.2
Consumables	7.16	0.37	5.5
Miscellaneous	2.23	0.12	1.7
Total	**131.13**	**6.79**	**100.0**

Power Supply

The power consumption is estimated from the installed power values presented in the Mechanical Equipment List (MEL), with utility factors applied to reflect the operating power draw. Annual operating hours for relevant areas were used to determine annual power usage.

The unit operating cost (10.4 c/kWh), represents NamPower's 2015 cost escalated by twice Namibian CPI until the end of the decade and discounted to 2015 terms and supplied by Bannerman.

Approximately 70% of the power requirement and cost lies in the crushing and agglomeration section of the process plant.

Labour

The process manning schedule was supplied by Bannerman for Year 3 of operation. Manning levels reflect a four panel continuous shift roster working 12 and 8 hour shifts depending on the role.

Maintenance Materials

The maintenance materials costs are based on percentage factors applied to the total area cost of various plant areas from the capital cost estimate. The factors, based on Amec Foster Wheelers' experience range from 1.91% for buildings to 5.8% in the precipitation, calcining and packaging area.

Water Supply

The process water consumption is estimated from the mass balance, with additional allowances made for mine and general water usage. Plant usage accounts for approximately 87% of total site usage.

For this study it is assumed that water is supplied by NamWater, at a cost of $3.5/m^3. This unit cost was provided by Bannerman and is based on the current cost of desalinated water.

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Consumables

Consumables costs including the replacement frequency for crusher liners, screen panels, agglomerator liner and lifters were based on vendor information. The HPGR roll maintenance cost is the largest single item and was based on consumption data supplied by the specified vendor. In addition to the roll maintenance vendor data for the HPGR checkplates cost and replacement frequency were supplied.

The cost for the heap leach drippers was based on a vendor quote, and considered the replacement of the drippers every two cycles. Uranium packaging drum costs are based on the production and on a unitary cost per drum from similar projects.

The drainage layer replacement cost assumes a removal rate of 10mm depth per cycle. All costs are inclusive of freight.

Miscellaneous

The miscellaneous estimate includes allowances for government fees, insurances, consultants, test work, maintenance contractors, equipment hire, environmental audits, and mobile equipment maintenance and fuel and safety equipment.

External Infrastructure

An annual allowance of $192,000 has been made by Bannerman for National Park fees and Road maintenance.

21.3.6 Other Owner's Operating Costs

The Owner's operating costs estimated by Bannerman total $18.40M annually, equivalent to $0.96/t crushed, as summarised in Table 21.9.

Table 21.9
Summary of Owner's Costs

Item	Average Annual Cost ($M/a)	Unit Cost ($/t of ore LOM)
Corporate and Owner's Labour	9.72	0.51
Total Site Office Administration	0.23	0.01
Total Personnel Expenses	3.05	0.16
Total Insurances and Government Fees	4.25	0.22
Site-Catering Facilities	0.53	0.03
Environmental Monitoring	0.30	0.02
Total Transportation Costs	0.20	0.01
Community Relations / Corporate Responsibility	0.12	0.01
Total	**18.40**	**0.96**

Principal costs are for Corporate and Owner's Labour, Training and Insurances.

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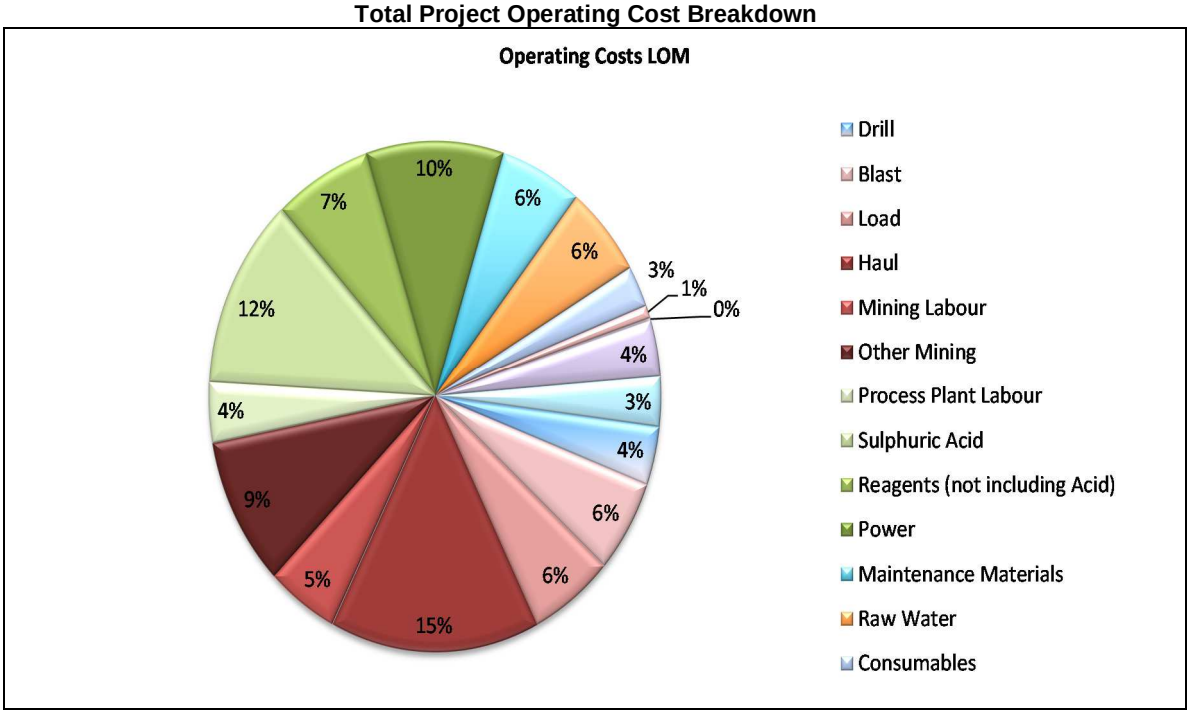
21.3.7 Total Project Operating Costs

Total operating costs for the project are summarised in Table 21.10 for the life of mine (LOM) and for the first 5 years of full production. A breakdown highlighting individual components of the LOM operating cost is shown in Figure 21.2.

Table 21.10
Summary of Total Operating Costs - LOM

Item	Cost ($/t of ore Yr 2-6)	Cost ($/t of ore LOM)	Cost ($/lb U_3O_8 Yr 2-6)	Cost ($/lb U_3O_8 LOM)	% of LOM Cost
Mining	7.46	6.38	16.19	17.13	45.1
Processing	6.77	6.79	14.69	18.28	48.1
Owner's Costs and Infrastructure	0.97	0.98	2.11	2.63	6.8
Total	**15.19**	**14.15**	**32.99**	**37.99**	**100**

Figure 21.2
Total Project Operating Cost Breakdown



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22 ECONOMIC ANALYSIS

22.1 INTRODUCTION

This section describes the financial model developed by Bannerman.

The final draft versions of the financial model were reviewed by Amec Foster Wheeler in Perth for internal accuracy and consistencies. A number of relatively minor changes were recommended and these were made.

Model inputs have been derived from the mining and plant feed schedule, metallurgical parameters, capital and operating costs identified earlier in this report. These have been reviewed by Amec Foster Wheeler and are in accordance with their relevant sections.

22.2 FINANCIAL MODEL INPUTS AND ASSUMPTIONS

The financial model has been created in Excel. Mining and processing data, and capital and operating cost estimates have been inserted into the financial model to enable the calculation of an internal rate of return (IRR) and a net present value (NPV) based on the indicative production and cash flow forecasts.

22.2.1 Basis of Financial Model

The scope of the financial model has been restricted to the project level excluding the effects of financing. Corporate taxation has been included. The financial model outputs reflect the results of the project at the Bannerman Namibia level allowing for an appropriate level of allocated administrative and corporate costs from the various ownership entities.

The financial model reflects the equity cash flows of the Etango Project without any debt financing.

The sensitivity analysis has been undertaken on a post-tax basis.

All revenue and cost estimates are expressed in US$ and are based on real 2015 quarter 3 values. Accordingly, no inflation assumption has been incorporated.

The key assumptions incorporated into the financial model for the DFS analysis are described in further detail as follows.

22.2.2 Assumed Mine Life

The period covered within the financial model commences from an assumed development start year of (late) 2017 through to construction and initial development during 2018 to 2019, with ramp up of production from (early) 2020 onwards. The mine is currently modelled to operate up to at least 2035 at which point mine closure activities are modelled to occur. This equates to an estimated mine life of approximately 16 years based on the OS resource model and other OS assumptions.

22.2.3 Production Physicals

The calculation of annual uranium oxide output is based on the mining and processing schedules which set out the appropriate parameters for these activities. Only Measured and Indicated Mineral Resources have been considered. The financial model allows for the

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variation in all key assumptions including mining rate, waste/ore stripping ratios, ore grades and metallurgical recovery (estimated at 86.85%).

Annual production is summarised in Table 22.1. Further detail was available in the relevant worksheets of the financial model.

22.2.4 Working Capital

A working capital build-up and delay between production and cash revenue receipts of 4 months has been assumed to simulate the estimated timeframe of the uranium oxide sales process.

22.2.5 Revenue

Final uranium output is assumed to be sold at a base case long term contract price of $75/lb of U_3O_8. Sensitivities have also been run at different price assumptions. Net revenue has been calculated after deducting royalties and an allowance of $1.10/lb for the estimated marketing, freight and conversion-related costs prior to sale at the relevant conversion facility.

22.2.6 Royalties

The financial model assumes a Namibian Government gross royalty of 3.0% of sales revenue in accordance with current Namibian legislation.

A contractual 1.5% royalty on gross revenue less (allowable deductions) is payable to Resource Capital Fund, the majority shareholder. Bannerman is also required to pay on the behalf of RCF the withholding tax that will be levied by the government upon the royalty. The royalty has been modelled in the financial model and has been grossed up to offset the withholding tax. As a result the effective royalty rate applied in the financial model is 1.875%. Tax advice indicates that the royalty to RCF will not be able to be expensed prior to taxation, and in response, RCF have agreed to reimburse the taxation amount to Bannerman. Subsequently, the full royalty rate of 1.875% was reduced by the corporate tax rate 37.5%.

22.2.7 Tax

An overview of the fiscal system in Namibia, outlining the principal taxes and duties expected to be payable by the project, is as follows. Taxation of the parent company, and/or individual investors is not considered in this overview.

The rate of corporate income tax payable by mining companies is 37.5%, payable on taxable profits with a capital deductions regime allowing the deduction of pre-production and other capital expenditure over a three year period.

Value Added tax (VAT) may be chargeable on sales and paid on purchases within Namibia. Where applicable, the VAT rate is 15%, although certain items are zero rated for VAT. Uranium produced by the Project will be exported, and will therefore not be subject to VAT.

22.2.8 Foreign Exchange Rates

Capital and operating items in foreign currency were converted to US$ using assumed long term exchange rates as on June 2015. The base case assumption is that US$1.00 equals A$1.28; N$12.25; ZAR12.25 and €0.88.

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22.2.9 Operating Costs

Operating costs have been estimated for each of the key functions of the project, and are detailed in the financial model.

Operating costs include all on-site costs and related overheads. As noted above, costs associated with the marketing, freighting and conversion of final product are modelled as deductions from revenue in accordance with industry and accounting practice.

22.2.10 Capital Costs

Capital costs are set out on the capital costs worksheets of the financial model. Each of the key capital cost components is set out in further detail on a separate worksheet.

The financial model for the purposes of this report does not calculate an accounting profit and, as a result, there is no non-cash depreciation or depletion calculation module for capital expenditure.

The cash operating surplus comprises net revenue less annual operating costs. Estimates of annual net cash flow are derived after deducting capital expenditure and allowances for working capital from the relevant period's cash operating surplus.

22.2.11 Financial Parameters

Net Present Value (NPV)

Project NPVs are calculated on after-tax net cash flows. The financial model is configured such that a range of discount rates can be applied and that tax can be turned on or off.

For the purposes of the base case evaluation, a real annual discount rate of 8% has been assumed.

Internal Rate of Return (IRR)

The various IRRs for the project are calculated using the annual after-tax net cash flows.

Payback Period

The payback period is defined as the period of time in which the cumulative undiscounted after-tax net cash flows ultimately becomes positive. At this point, the project will have paid back the initial development and working capital costs.

22.3 FINANCIAL MODEL OUTCOMES

The base case cash flow is shown in Table 22.1. In calculating the potential returns from the project, the fundamental assumptions shown in Table 22.2 have been made.

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Table 22.1
Etango Base Case Financial Model – Cash Flow Summary

Financial Year Ending 31 December	Units	TOTAL	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Physicals																					
Mining																					
Ore	Mt	303.3				8.9	22.2	29.4	33.9	31.6	26.2	20.1	15.8	21.6	14.9	21.3	23.8	17.5	15.2	0.9	
Waste	Mt	842.1				41.2	43.1	60.1	65.6	68.7	75.9	79.8	84.1	76.7	83.4	60.7	46.2	43.6	12.2	0.5	
Total Material Mined	Mt	1,145.4				50.1	65.4	89.6	99.6	100.3	102.1	99.9	99.9	98.3	98.3	82.0	70.1	61.1	27.4	1.4	
Grade	ppm	194.5				164	197	226	210	174	176	172	186	209	178	170	189	203	251	289	
Strip Ratio	Ratio	2.8				4.6	1.9	2.0	1.9	2.2	2.9	4.0	5.3	3.6	5.6	2.8	1.9	2.5	0.8	0.5	
Stockpile																					
Stockpile	Mt	n/a				0.1	3.0	12.4	26.2	37.7	43.8	43.8	39.5	40.9	35.8	37.0	40.8	38.1	33.2	14.1	
Grade	ppm	n/a				111	96	110	112	102	98	93	86	84	79	78	78	78	78	78	
Mill																					
Ore Feed	Mt	303.3				8.8	19.3	20.1	20.1	20.1	20.1	20.1	20.1	20.1	20.1	20.1	20.1	20.1	20.1	20.1	14.1
Grade	ppm	194.5				165.1	211	279	277	227	208	183	179	223	162	177	208	187	209	88	78
Recovery	%					85.98%	86.85%	86.9%	86.9%	86.9%	86.9%	86.9%	86.9%	86.9%	86.9%	86.9%	86.9%	86.9%	86.9%	86.9%	86.9%
Production																					
U_3O_8	t	51,234				1,256	3,531	4,862	4,822	3,966	3,629	3,196	3,124	3,898	2,828	3,094	3,634	3,270	3,643	1,527	953
U_3O_8	000lbs	112,951				2,770	7,785	10,719	10,630	8,744	8,000	7,046	6,888	8,593	6,234	6,822	8,012	7,210	8,032	3,367	2,100
Revenue																					
Price	$/lb	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0
Sales	000lbs	112,951				1,521	6,439	9,741	10,660	9,372	8,248	7,364	6,941	8,025	7,020	6,626	7,615	7,477	7,758	4,922	3,222
Gross Revenue	$000	8,471,330				114,086	482,905	730,568	799,473	702,929	618,592	552,297	520,548	601,863	526,534	496,952	571,131	560,786	581,855	369,136	241,673
Royalties and Conversion Costs	$000	378,386				5,096	21,570	32,632	35,710	31,398	27,630	24,669	23,251	26,883	23,519	22,197	25,511	25,048	25,990	16,488	10,795
Net Revenue	$000	8,092,944				108,990	461,335	697,936	763,763	671,532	590,962	527,628	497,297	574,979	503,016	474,755	545,621	535,738	555,866	352,648	230,879
Operating Expenditure																					
Mining	$000	1,934,720				78,917	111,101	138,220	156,897	170,068	166,397	170,989	149,438	165,014	146,612	142,049	115,504	128,161	70,300	16,906	8,147
Processing	$000	2,059,565				66,479	130,753	135,866	135,608	135,945	135,704	135,608	135,608	135,945	135,608	135,608	135,608	135,945	135,608	135,608	98,066
Infrastructure	$000	5,563				357	357	357	357	357	357	357	357	357	357	357	357	357	357	357	208
Owners Costs	$000	291,224				19,922	19,793	19,091	18,876	18,865	18,524	18,524	18,524	18,498	18,590	18,590	18,590	18,590	18,434	18,434	9,378
Total Operating Expenditure	$000	4,291,071				165,674	262,004	293,534	311,738	325,235	320,982	325,477	303,927	319,814	301,167	296,604	270,059	283,054	224,699	171,306	115,799
Pre-Tax Cash Flow	**$000**	**3,801,872**				**(56,685)**	**199,331**	**404,402**	**452,026**	**346,297**	**269,980**	**202,151**	**193,370**	**255,165**	**201,849**	**178,151**	**275,562**	**252,684**	**331,167**	**181,342**	**115,080**
Tax	$000	1,067,847							47,624	124,269	99,999	74,915	70,432	92,983	73,012	64,429	100,569	92,277	122,600	67,554	37,182
Post-Tax Cash Flow	**$000**	**2,734,025**				**(56,685)**	**199,331**	**404,402**	**404,402**	**222,028**	**169,980**	**127,236**	**122,938**	**162,182**	**128,836**	**113,722**	**174,992**	**160,407**	**208,566**	**113,789**	**77,898**
Capital Expenditure																					
Mining Direct Costs	$000	398,547	1,000	3,803	126,527	95,957	79,818	38,824	3,324	6,028	575	531	10,508	6,242	1,369	11,492	9,337		3,212		
Processing Direct Costs	$000	321,358		126,838	194,520																
Mobilisation and Demobilisation	$000	6,106		2,442	3,664																
First Fills and Opening Stocks	$000			6,078	9,117																(15,194)
Spares & Commissioning	$000	16,468		6,587	9,881																
Site Infrastructure Direct Costs	$000	79,309		29,918	44,877								4,515								
Owners Direct Costs	$000	38,899	6,120	9,492	23,288																
External Infrastructure	$000	45,996		22,920	23,076																
Sustaining Capital	$000	750				56	56	56	56	56	56	56	56	56	56	56	56	56	28		
Pre-Prod Owners, EPCM & Accuracy Provision	$000	111,044	10,879	44,418	55,747																
Temporary Services & Facilities, Const Camp and Vendor	$000	24,163		12,585	18,878	(7,300)															
Rehabilitation	$000	32,500						500			500			500			500		500		30,000
IMPORT DUTIES	$000																				
Owners Contingency	$000																				
Total Capital Expenditure	$000	1,075,141	17,999	265,082	509,573	88,713	79,874	39,380	3,379	6,083	1,130	587	15,078	6,798	1,424	11,547	9,892	56	3,740		14,806
RCF Royalty payment	$000	97,817				1,317	5,576	8,436	9,231	8,117	7,143	6,377	6,011	6,950	6,080	5,738	6,595	6,475	6,719	4,262	2,791
Post Capital Expenditure Cash Flow	$000	1,561,067	(17,999)	(265,082)	(509,573)	(146,715)	113,881	356,586	391,791	207,828	161,707	120,272	101,849	148,435	121,332	96,436	158,505	153,876	198,108	109,526	60,302

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Table 22.2 **Fundamental Assumptions of Financial Modelling Analysis**	
Basis	Project level, pre- or post-tax and excluding any debt financing
U_3O_8 prices	Long term contract price assumed at $75/lb U_3O_8
Development period	2 to 3 years, assuming commissioning in early 2016
Mine life	16 years, closing in 2035 based on the November 2015 mineral resource estimate
Annual throughput	20Mt
Fuel price	$0.79/L, plus freight
Sulphuric acid price	$95.6/t delivered to site
Raw water cost	$3.5/m^3
Power cost	$0.104/kWh
Production rate	Between approximately 6 to 10 Mlb of U_3O_8 per year
Exchange rates	US$1.00 : A$1.22 : N$12.25 : R12.25 : €0.88

The key outputs from the financial model based on the above assumptions are reported for the first 5 years of full production of the modelled operation (shown as 2021 to 2026 excluding the ramp-up year of 2020) and for the life of mine (2020 to 2035) in Table 22.3.

Table 22.3 **Key Financial Model Outputs**		
	First 5 Years of Full production	**Life of Mine**
Project Economics		
NPV at a Discount Rate of 8% ($M)	-	419.1
Internal Rate of Return (%)	-	15.3%
Payback Period from Start of Production (Q1 2016)	-	4.4
Production		
Quantity Ore Treated (Mt)	99.6	303.3
Uranium Oxide Produced (t U_3O_8)	20 810	51 234
Uranium Oxide Produced (Mlb U_3O_8)	45.9	112.95
Revenue		
Average U_3O_8 Base Price ($/lb U_3O_8)	75	75
Net Revenue ($M, after Gvmnt royalties)	3 185	8 093
Operating Unit Costs		
On-Site Costs/tonne Ore Treated ($/t ore)		
Mining	7.46	6.38
Processing	6.77	6.79
Owners costs (including infrastructure maintenance)	0.97	0.98
Total Operating Costs ($/t ore)	**15.19**	**14.15**
Total Operating Costs ($/lb produced)	**32.99**	**37.99**
Marketing, freight and conversion	1.10	1.10

Based on the above, at a throughput rate of 20Mtpa, the Project is modelled to produce between 6 to 10Mlb U_3O_8 per year. The average cash operating cost in the first 5 years of full production is estimated at $32.99/lb U_3O_8 and over the life of mine is estimated at $37.99/lb U_3O_8.

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Pre-production capital is estimated to comprise $792.7M of engineering, construction and development costs plus approximately $172.3M of working capital in order for the project to be funded to first positive operating cash flow. Over the life of the mine (LOM) there is additional mine fleet of $267.2M, sustaining capital of $0.75M, and mobile equipment is replaced in Year 8 for $4.515M. Various capital items are modelled as being recouped over the LOM as follows: the temporary construction camp is assumed to be sold for $7.3M in Year 1 of operations and the capital outlaid on first fills of $15.2M is recouped in the final year.

Table 22.4
Capital Costs

	Pre-Production ($M)	Life of Mine ($M)
Mining	131.3	398.5
Plant	321.4	321.4
Infrastructure (site and external)	120.8	126.1
Owner's Costs	38.9	38.9
Mobilisation/Demobilisation	6.1	6.1
Vendor Representatives, EPCM and Accuracy Provision	115.7	115.7
Spares, First Fills and Commissioning	31.6	16.4
Construction Camp and Facilities	26.8	19.5
Owner's Contingency	-	-
Rehabilitation	-	32.5
Capital Cost before Working Capital	**792.6**	**1075.1**
Working Capital – Funding Operating Cash Shortfall	172.3	-
Total Initial Capital Cost	**964.9**	**1075.1**

At an assumed sales price of $75/lb U_3O_8, operating cash flow is estimated to be $1,400M in the first 5 years of full production, and a total of $2,734M over the life of mine.

Table 22.5
Revenue and Operating Costs

	First 5 Years Full Prod	Life of Mine
Net revenue/tonne ore processed ($/t)	31.98	26.68
Net revenue/tonne U_3O_8 sold ($/t)	153,079	157,961
Mining costs/tonne U_3O_8 produced ($/t)	35.689	37,763
Mining costs/lb U_3O_8 produced ($/lb)	16.19	17.13
Processing costs/tonne U_3O_8 produced ($/t)	32,383	40,199
Processing costs/lb U_3O_8 produced ($/lb)	14.69	18.23
Operating costs/tonne U_3O_8 produced ($/t)	72,730	83,755
Operating costs/lb U_3O_8 produced ($/lb)	32.99	37.99

The payback period is estimated to be approximately 4 years (before and after tax) with the NPV of the project, at an 8% real discount rate, estimated to be $419.1M after tax. The internal rate of return of the project is estimated at 15.3% after tax.

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22.4 FINANCIAL SENSITIVITY ANALYSIS

Sensitivity analyses have been undertaken on key parameters within the financial model to assess the impact of changes upon project cash flows, NPV, IRR and payback period.

In assessing the sensitivity of the project returns, each of the parameters has been varied independently of the others. Accordingly, combined positive or negative variations in any of these parameters will have a more marked effect on the forecast economics of the project than will the individual variations considered.

The convention adopted in this analysis is that negative sensitivities are adjustments that reduce project economics or value (for example, increased capital or operating costs) and, correspondingly, positive sensitivities are adjustments that improve project economics and value.

Table 22.6 presents the results of the sensitivity analysis.

Table 22.6
Sensitivity Analysis

Parameter/Variation	Value		Project		
U_3O_8 Price	U_3O_8 Price ($/lb)		IRR (%)	$NPV_{8\%}$ ($M)	Payback Period (years)
-30%	52.50		0.2%	-355.9	15.2
-20%	60.00		6.3%	-84.4	8.5
-10%	67.50		11.1%	170.0	5.3
0%	75.00		15.3%	419.1	4.4
10%	82.50		19.0%	666.3	3.9
20%	90.00		22.4%	913.5	3.5
30%	97.50		25.5%	1158.4	3.3
Total Capital Costs	**Project Capital Costs ($M)**				
-30%	1,397.68		11.0%	210.0	5.5
-20%	1,290.17		12.3%	280.9	5.1
-10%	1,182.66		13.7%	350.5	4.7
0%	1,075.14		15.3%	419.1	4.4
10%	967.63		17.1%	487.8	4.0
20%	860.11		19.3%	556.4	3.8
30%	752.60		21.9%	625.0	3.6
Operating Costs	**Average Operating Costs ($/lb U_3O_8)**				
	First 5 Years Full Prod	**Life of Mine**			
-30%	42.89	49.39	8.2%	13.2	6.6
-20%	39.59	45.59	10.8%	149.7	5.4
-10%	36.29	41.79	13.1%	285.4	4.8
0%	32.99	37.99	15.3%	419.1	4.4
10%	29.69	34.19	17.4%	552.9	4.0
20%	26.39	30.39	19.4%	686.7	3.9

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Table 22.6 Sensitivity Analysis					
Parameter/Variation	**Value**		**Project**		
			IRR (%)	**$NPV_{8\%}$ ($M)**	**Payback Period (years)**
U_3O_8 Price	**U_3O_8 Price ($/lb)**				
30%	23.09	26.59	21.3%	820.5	3.7
Mining Costs	**Average Mining Costs ($/lb U_3O_8)**				
	First 5 Years	**Life of Mine**			
-30%	21.04	22.27	12.2%	231.6	5.0
-20%	19.43	20.55	13.2%	294.3	4.8
-10%	17.81	18.84	14.3%	356.7	4.6
0%	16.19	17.13	15.3%	419.1	4.4
10%	14.57	15.42	16.3%	481.6	4.2
20%	12.95	13.70	17.3%	544.0	4.1
30%	11.33	11.99	18.2%	606.5	4.0
Processing Costs	**Avg Processing Costs ($/lb U3O8)**				
	First 5 Years	**Life of Mine**			
-30%	19.10	23.70	12.3%	233.1	4.9
-20%	17.63	21.88	13.3%	295.2	4.7
-10%	16.16	20.06	14.3%	357.2	4.5
0%	14.69	18.23	15.3%	419.1	4.4
10%	13.22	16.41	16.2%	481.1	4.2
20%	11.75	14.59	17.2%	543.1	4.1
30%	10.28	12.76	18.1%	605.0	4.0

Figure 22.1, Figure 22.2 and show the sensitivity results on the Project's NPV, IRR and payback period to changes in U_3O_8 prices, capital costs and operating costs in graphical form.

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Figure 22.1
Project Net Present Value

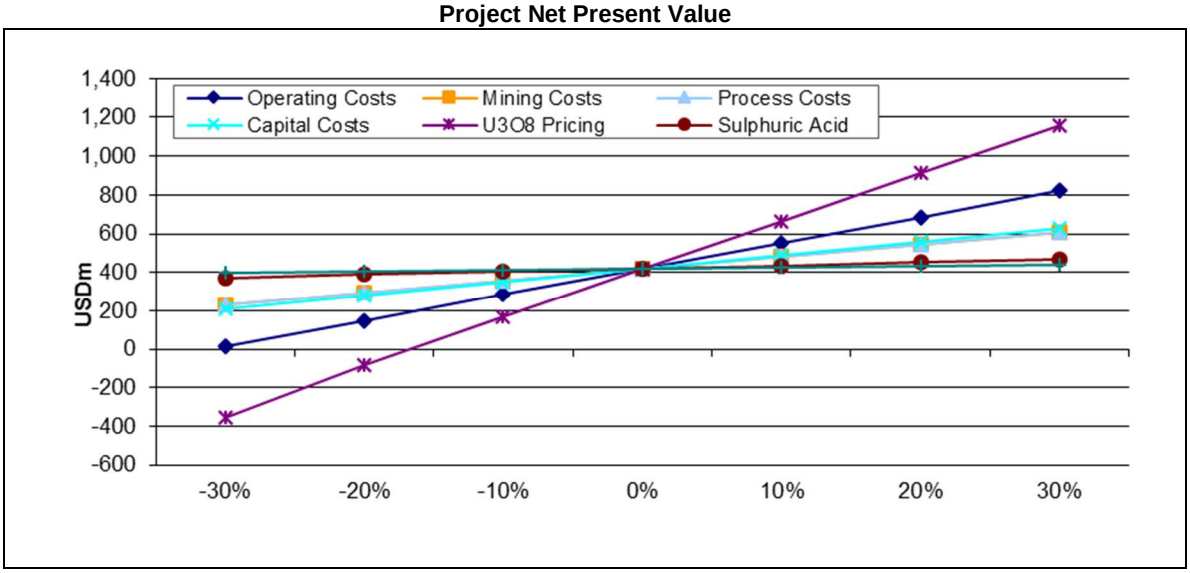

Figure 22.2
Project Internal Rate of Return

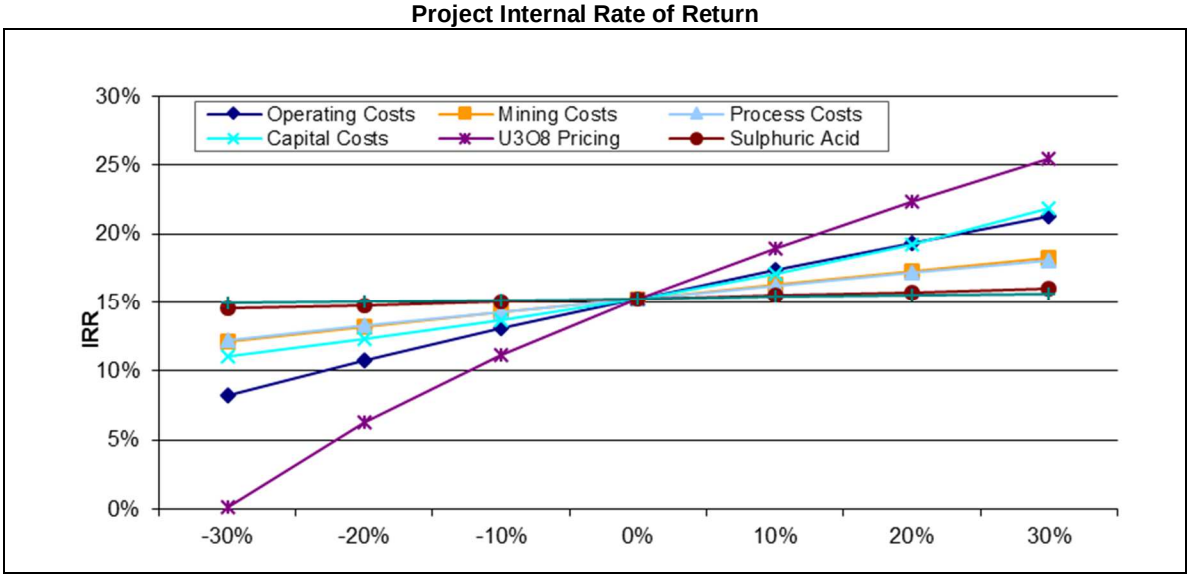

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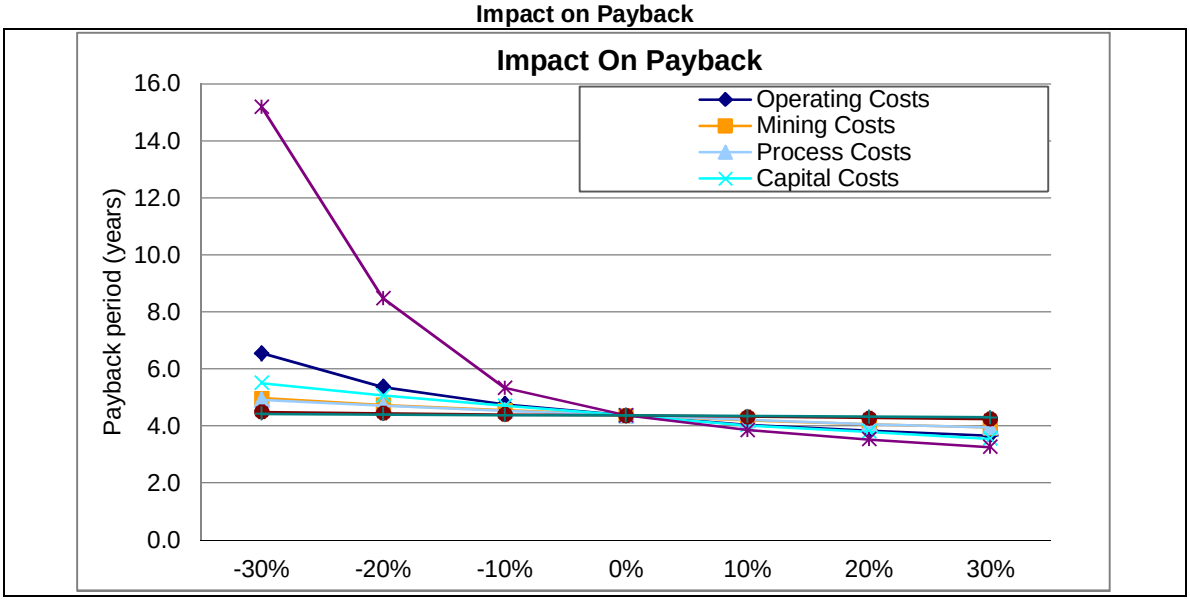
Figure 22.3
Impact on Payback



22.4.1 Relative Sensitivities

The financial sensitivity analysis demonstrates that the economic performance of the Etango Project is most sensitive to changes in the uranium price, followed by operating costs. This is unsurprising given the large scale and relatively modest grade of the deposit.

The project is therefore affected by factors which have the greatest bearing upon cash operating margins. Accordingly, the highest sensitivity is to uranium prices, followed by sensitivity to operating costs and lastly to capital costs. Each component is discussed briefly below.

Sensitivity to Changes in U_3O_8 Prices

As noted, the Etango Project is most sensitive to changes in uranium prices. Negative movements of 10% and 20% from the base case assumption of \$75/lb U_3O_8 result in the post-tax NPV reducing from \$419.1M to \$170.0M and minus \$84.4M respectively.

Likewise, positive movements of 10% and 20% from the base case assumption of \$75/lb U_3O_8 produce significant changes in the post-tax NPV from \$419.1M to \$666.3M and \$913.5M respectively, the latter with a post-tax IRR of 22.4%.

A 20% increase in the U_3O_8 price reduces the payback period by 1 year (to 3.5 years) and a 20% decrease in the U_3O_8 price results in payback increasing to 8.5 years.

Should higher prices than the base case assumption be available to the Project, then the economics become immediately and significantly more attractive.

Sensitivity to Changes in Total Operating Costs

As noted above, given the large annual throughput of the project, the financial performance is also very sensitive to changes in total operating costs.

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Increases of 10% and 20% in the base case cost assumptions produce significant adverse changes in the post-tax NPV from $419.1M to $285.4M and $149.7M respectively, the latter with a post-tax IRR of 10.8%.

Likewise, cost reductions of 10% and 20% from the base case assumptions result in the post-tax NPV increasing from $419.1M to $552.9M and $687M respectively, the latter with a pre-tax IRR of 19.4%.

For further detail, sensitivity analyses have also been undertaken on subcategories of operating costs including mining costs, processing costs and sulphuric acid costs. The results of this analysis are charted in previous figures.

A 10% decrease in operating costs reduces the payback period by 0.4 years (to 4 years) and a 10% increase in capital costs results in payback occurring in 4.8 years.

Sensitivity to Changes in Capital Costs

The sensitivity of the Etango Project to changes in capital costs is driven by the scale and timing of the up-front construction and development expenditure. For the purposes of the sensitivity analysis, capital costs excluding working capital were varied in accordance with the nominated percentage changes. Working capital is a function of operating expenditure and lagged revenues, and has therefore not been varied in the capital cost sensitivity analysis.

Increases of 10% and 20% in the base case capital cost assumptions produce adverse changes in the post-tax NPV from $419.1M to $350.5M and $280.9M respectively, the latter with a pre-tax IRR of 12.3%.

Likewise, capital cost reductions of 10% and 20% from the base case assumptions result in the post-tax NPV increasing from $419.1M to $487.8M and $556.4M respectively, the latter with a post-tax IRR of 19.3%.

A 10% decrease in capital costs reduces the payback period by 0.4 years (to 4 years) and a 10% increase in capital costs results in payback period increasing by 0.3 years (to 4.7 years)

22.5 CAUTIONARY STATEMENT

The results of the economic analysis are based on forward-looking information that are subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here.

Forward-looking information includes commodity prices and exchange rates; the proposed mine production plan; projected plant head grade and recovery rates; uncertainties and risks regarding the estimated capital and operating costs; uncertainties and risks regarding cost estimates and completion schedule for the proposed Project infrastructure, including the need to obtain permits and governmental approvals on a timely basis.

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23 ADJACENT PROPERTIES

The Bannerman Etango Project is situated within the highly mineralised southern Central Zone of the Damara Orogenic Belt, which has been subject to intensive exploration and development by a number of international mining and exploration companies. Significant nearby uranium projects include the Rössing Mine, the Langer Heinrich Mine, the Trekkopje Mine and the nearby Husab (formerly Rössing South) Project.

23.1 RÖSSING MINE

The Rössing Mine is controlled by Rössing Uranium Limited which in turn is owned by Rio Tinto (69%), the Iranian Foreign Investment Company (Government of Iran (15%)), the Industrial Development Corporation of South Africa (10%), the Namibian Government (3%) and private ownership (3%). The mine which is the third largest uranium mine in the world, and is the largest granite-hosted uranium mine, is located approximately 13km from the north-eastern boundary of EPL 3345. Production commenced in 1976. In 2009, Rössing completed a feasibility study into an expansion of the mine and a program to extend the mine life to 2023 and beyond (Aurecon, 2010).

The Rössing style of mineralisation as reported is very similar to that at the Etango Project and the structural trend which hosts the Rössing Mine is interpreted to extend into the Gohare-Ombuga-Rössingberg trend in the centre of EPL 3345, highlighting the highly prospective nature of this tenement.

Rössing reported processing 7.0Mt of rock to produce 1 543t of U_3O_8 in 2014.

23.2 LANGER HEINRICH MINE

The Langer Heinrich Mine, which is owned by a subsidiary of Paladin Energy Ltd, is located approximately 50km to the east of Bannerman's EPL 3345. The Langer Heinrich mine came into production in December 2006.

The Langer Heinrich deposit is a calcrete-hosted uranium deposit that is associated with valley fill sediments in a tertiary paleo-drainage system. The uranium mineralisation occurs as disseminations of the mineral carnotite in calcretised valley-fill sediments. The deposit occurs over a 15km strike length and has up to 8m of river sand and scree overburden.

In June 2015, Paladin reported the remaining Measured and Indicated Mineral Resources at the Langer Heinrich Mine to be 82.5Mt at 550ppm U_3O_8 for 99.1Mlbs of U_3O_8, at a 250ppm U_3O_8 cut-off grade. An additional 32.1Mt at 400ppm U_3O_8 for 28.4Mlbs was estimated to exist in stockpiles. The remaining mineral reserves were estimated at 100.7Mt at 510ppm U_3O_8 for 112.6Mlbs of U_3O_8, at a 250ppm U_3O_8 cut-off grade, of which approximately 30% was in existing stockpiles.

23.3 HUSAB PROJECT

The Husab project is owned by China General Nuclear Power Company (CGNPC). It consists of two EPLs with a total area of 637km² and is located between Bannerman's two tenements.

The Husab tenements contain primary alaskite-hosted mineralisation under extensive Aeolian sand and gravels of the Namib Plain. Mineralised alaskites occur mainly within the Rössing Formation, including clastic metasediments, calc-silicate gneisses and marbles, and

also along the contact between the Khan and Rössing Formations and the contact between the Chuos and Rössing Formations (Extract, 2008).

The Husab Mineral Resources and Mineral Reserves are no longer publically reported.

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24 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT IMPLEMENTATION

24.1.1 Development Phases and Schedule

The Project Development schedule has been prepared as indicated in Table 24.1. The execution of engineering design, procurement, transport and construction is expected to take approximately 36 months from Project Approval.

Table 24.1
Project Development Milestones

Task	Date (Month)
Commence early works	1
Project approval i.e. receipt of regulatory approvals/project financing	7
Commence site works	17
Commence commissioning	33
Commence initial ramp up	39
First shipment	40
First shipment with schedule contingency	43
Ramp-up to design tonnages	51

The schedule shows some early works activities being undertaken, which is expected to include additional test work to increase certainty in process and engineering. Site geotechnical investigations have also been recommended.

The key driver of the development schedule is the delivery of long lead equipment with a number of long lead items such as mining haul trucks and the stacker, reclaimers and conveyors associated with the heap leach system having greater than 18 months delivery. Detailed engineering is required before orders can be placed, and such work could commence pre-Project Approval.

The schedule includes a contingency of 3 months, and is conditional upon the upgrade of access roads, establishment of the construction village and other basic infrastructure being in place to support the construction effort.

A key issue is the timely receipt of Namibian Government licences and permits.

24.1.2 Execution Methodology

The OS adopted the 2012 DFS execution methodology in its entirety. It is assumed that an integrated team or an engineering, procurement and construction management (EPCM) model with a combination of horizontal construction packages and EPC packages and with 'free-issue' major or long lead equipment is adopted, in view of the limited of number of major consultants/engineering/contracting companies capable of undertaking the full scope of work.

The contracts will be a mixture of lump sums for equipment supply and EPC contracts, and cost reimbursable with performance incentives for construction. In the present market

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environment, lump sum construction contracts are not considered to be cost effective and could result in schedule slippage.

It is envisaged that the Project will be managed by two project teams, the Bannerman project team and the EPCM Provider's team.

Bannerman Owner's Team

The Owner's team will be responsible for the obtaining all necessary government approvals and permits for the construction and operation of the Project In addition it will continue liaising with local communities and government organisations including any resettlement issues.

Bannerman will develop operational and maintenance plans and procedures, undertake recruitment and training, and establish initial mining operations during the development phase.

EPCM Provider

The EPCM Provider will provide or manage engineering services for finalising process and engineering design, equipment specification, procurement and construction of the plant and infrastructure.

Pre-commissioning and commissioning of plant will be undertaken with assistance from Bannerman operations personnel, and the contractor will provide a small team of people to assist Bannerman during plant ramp-up as required.

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25 INTERPRETATION AND CONCLUSIONS

25.1 GEOLOGY AND RESOURCES

The Etango Project hosts significant uranium resources over a prospective strike length of greater than 15km along the western flank of the Palmenhorst Dome, which incorporates the Anomaly A, Oshiveli, Onkelo, Ondjamba and Hyena deposits.

Coffey Mining has reviewed the drilling, sampling and assaying procedures used by Bannerman and finds them to be acceptable by industry standards. Checks by Coffey Mining have identified no material issues with the database and it is considered acceptable for use in resource estimations.

The mineral resources have been classified in accordance with NI43-101, and the measured and indicated mineral resources are considered suitable for use in mining studies at a DFS level of accuracy.

25.2 MINING

The preferred mining method is open pit extraction utilising a conventional mining fleet comprising of 550t diesel hydraulic excavators, backed up by 220t off road dump trucks mining at a peak mining rate of around 100Mtpa to supply 20Mtpa of ore.

The optimum degree of mining selectivity for the Etango Project is drilling and blasting on a 12m bench, with loading out in three flitches of equal height, which will nominally be 4.5m high, after allowing for swell from blasting.

Detailed staged pit designs show that a practical, achievable mining sequence can be adopted which mines the deposit in four stages.

Waste dumps have sufficient footprint available to accommodate the LOM tonnage from a 4 Stage pit. The dump design and schedule allows for Closure considerations for drainage and rehabilitation.

The quantities for major consumables including major mining equipment, explosives and blasting accessories, fuel and lubricants and tyres have been estimated from the mining schedules. Through an RFQ process, preferred suppliers of the major consumables have supplied prices, and the supply of the major consumables in the required quantities and in the required timeframes has been confirmed.

25.3 METALLURGICAL

25.3.1 Heap leaching

Column leach tests have confirmed that the ore is amenable to crushing and heap leaching, demonstrating high recoveries, relatively rapid kinetics and relatively low acid consumption. Bannerman Resources has constructed a demonstration plant to conduct large section leach test work on a larger scale. Each of section (or crib) is 2m x 2m x 5m and is able to leach approximately 30t of ore. Tests in this facility have confirmed the metallurgical assumptions from previous test work.

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25.3.2 SX and Uranium Recovery

Extraction test work has demonstrated that high levels of extraction will be achieved using standard SX processes.

The circuit selected by Bateman is conventional and considered generally low risk. However, test work work has not yet been undertaken to confirm equipment performance and sizing.

Build-up of contaminates in the return raffinate and impurity levels in the U_3O_8 product over the longer terms have not yet been determined but is the subject of a test program underway utilising the demonstration plant.

25.4 GEOTECHNICAL AND HYDROLOGY

Adequate geotechnical, hydrological and hydrogeological investigations have been completed for pit design purposes. Investigations were extended to cover waste dump and Ripios dump design.

Site hydrology has been investigated to determine water control requirements.

A preliminary site investigation of geotechnical conditions for structural design indicates favourable conditions. However, more detailed investigations are required for final design, including drilling, soil and test pit sampling. This work should be extended to confirm sources of suitable construction materials.

25.5 PROJECT DEVELOPMENT

Project implementation has been completed at a high level, based on EPCM approach. A construction schedule has been determined indicating a duration of 36 months from Project approval and financial closure to first shipment of product.

25.6 ENVIRONMENTAL AND PERMITTING

Environmental studies have been completed into all aspects of the project, an ESIA submitted to support the application and an environmental clearance has been received.

No fatal flaws have been identified, and appropriate mitigation measures have been included in the project design to manage environmental issues.

25.7 REVIEW OF PROJECT RISK

A range of economic, engineering and other technical risks to the Project have been considered. Those risks are summarised in Table 25.1 arranged in general order of likelihood and importance, and are discussed by discipline in the remainder of this section.

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Table 25.1 **Non-Resource/Mining Economic and Technical Risk Assessment**	
Item	**Assessed Risk to Project**
U_3O_8 price	High – Major risk to Project
Power supply	High – Major risk until assured
Mining equipment under-performance	Moderate to High
Capital cost over-run	Moderate to High
Operating cost over-run – power	Moderate to High
Operating cost over-run - acid	Moderate to High
Operating cost over-run - diesel	Moderate to High
Mine operating costs overrun (sustained increase in labour/ materials costs)	Moderate
Operating cost over-run – HPGR wear rate	Moderate
Heap leaching under-performance	Moderate
Resource under-performs	Moderate
Mobile equipment poor availability	Moderate
Scheduling and production time not achieved	Moderate
Mine Equipment capital cost over-run	Moderate
Late completion of mine pioneering works	Moderate
Timely availability of water supply	Low to Moderate
Failure to achieve plate throughput	Low to Moderate
Ramp-up schedule over-run	Low to Moderate
Project construction delays	Low to Moderate
Adequacy of water supply	Low to Moderate
Qualified personnel availability	Low to Moderate
Operating cost over-run - water	Low
Foreign exchange variation	Low
Permits refused or seriously delayed	Low
Royalty rate increase	Low

25.7.1 Geological Interpretation and Resource

While the reported mineral resources are considered to be robust, these remain estimates and there are underlying uncertainties relating to interpretation of drill results and the geology, continuity and grade of the mineral deposits. Such risks are typical of all mining projects, and the level of risk is judged to be no more than ***Moderate***, given the general continuity in geometry and grade of the deposit.

25.7.2 Mining Risk Assessment

The risk associated with the resource estimate extends into the mining study, in terms of potential inaccuracies in deposit geometry, continuity and grade. These uncertainties will be reduced during grade control drilling prior to mining.

Other ***Moderate to High*** mining risks include:

- Late completion of pioneering works, however, the schedule allows for excess ore production in the first year - ***Moderate***

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- Poorer than expected equipment performance and/or availability which would lead to failure to meet the production schedule and increased unit costs – ***Moderate to High***

Again, such risks are typical of all mining projects, and the level of risk is not unusual.

25.7.3 Price of U_3O_8

A long-term contract price of $75/lb U_3O_8 has been assumed in the DFS, on the upper end of the current long-term price predictions of $65-80/lb.

Exposure to lower prices for U_3O_8 would be a ***Major*** risk to the project. Lower than modelled prices for U_3O_8 would reduce modelled operating cash flows and could cause the deferral of a development decision or the suspension of operations.

Conversely, higher than modelled U_3O_8 prices would have a significant positive impact on cash operating margins, as there would be minimal additional costs.

Bannerman intends to seek a strategic partnership with an established industry end-user such that specified quantities of future production can be sold at minimum prices consistent with the $75/lb minimum price.

25.7.4 Foreign Exchange Rate Exposure

The perceived risk of exchange rate exposure is considered relatively ***Low*** due to the fact that the vast majority of capital expenditure is in the SADC countries. A number of banks are predicting a significant improvement in the strength of the $ over the next few years, especially compared to the southern African currencies.

25.7.5 Capital Cost Overrun

As for any major mining project, there is a significant risk of capital cost overruns resulting from a range of factors, primarily sudden and unpredicted increases in equipment, materials or labour capital costs.

Additional risk lies in uncertainty regarding site geotechnical conditions, although no obvious issues were identified from preliminary examination.

Engineering has been taken to a level appropriate for a DFS in 2012, and an accuracy provision allowance made for expected, unidentified additional costs once detailed engineering has been undertaken. These provisions are based on Amec Foster Wheeler's experience with similar project, but there is no certainty that such provisions are sufficient. A Project Contingency of 12% in the plant and infrastructure capital cost estimate to allow for other unexpected engineering or cost issues, was recommended by Amec Foster Wheeler, but this has been omitted by Bannerman from the base case financial model. Subsequent updates focussed on re-pricing the estimate with budget quotations and constitute a PFS level of confidence.

No provision has been made for outside contingencies such as abnormal weather impacts or delays, unforeseen environmental or social constraints, schedule impacts from late delivery of critical equipment items, or unforeseen changes in legislation.

The absence of a Project Contingency in the base case financial model increases project risk from capital cost overruns to **Moderate to *High***.

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25.7.6 Operating Cost Overrun

The base case annual consumption of sulphuric acid is approximately 340,000tpa. At the assumed delivered price of $96/t, this represents the highest process operating cost item. Project economics are sensitive to changes in acid price, which constitutes a ***Moderate to High*** risk to the project.

The future cost of electricity supplied by NamPower is uncertain. The price of 10.4c/kWh has been applied to determine operating costs. This was determined by escalating the current price by double the current CPI (~6% per annum) until the end of the decade and then expressing this value in 2015 terms. Electricity costs are considered to be a ***Moderate to High*** risk to the Project.

Diesel prices were based on quotes obtained in 2015. Given the historic low of current oil prices, diesel costs are considered to be a ***Moderate to High*** risk to the project.

25.7.7 Process

During the 2012 DFS only a modest amount of metallurgical test work had been carried out for the base case heap leach option by DFS standards. The results were encouraging and consistent. Further medium scale test work is underway to finalise the heap design and it is recommended additional work be carried out in the SX/precipitation/thickening/calcining area. Work in these areas will increase confidence in equipment selections and overall engineering design, thereby further de-risking the financial model.

Moderate Process risks identified from the DFS include:

- Crushing: HPGR crushing is a relatively new technology and there remains a degree of risk whether a higher wear rate will occur or not. There is limited precedent for full scale HPGR operation in hard rock applications and wear rate scale-up from test work is not well proven
- Heap Leaching: Heap leach test work conducted during the DFS has not fully tested the selected design criteria, nor ore variability across the deposit, and there is potential that key design criteria such as recovery, extraction rates and acid consumption could be optimistic. However, results of column test work to-date indicate relatively consistent results over a range of conditions, in line with the design criteria. It is anticipated that data collected from the current test work will reduce the design risk in this area
- SX and recovery test work has been limited, and the design includes numerous assumptions regarding equipment and performance that remain to be quantified through additional test work as recommended by Bateman.

25.7.8 Utility Supply

Adequate and timely supply of water and electricity are fundamental to all activities in the construction and operation of the mine. NamPower and NamWater have a track record of supplying utilities across the country, but specific risks should be considered further by Bannerman, since the implications of late or reduced supply could be very significant.

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Electricity Supply

NamPower is planning increases to its network capacity, but there is uncertainty that sufficient power can be made available and brought to site according to the current Project timeframe. The risk is judged to be ***High***.

The largest specific risk would be catastrophic failure of a transformer during commissioning or ramp-up. Generally, arrangements can be made to share and swap spare or extra capacity, but delays would certainly occur.

Water Supply

The risks around the supply of fresh water have diminished significantly since the completion of the DFS. Areva has constructed a 20Gl desalination plant which was commissioned in 2010 to supply water to their Trekkopje mine. Since the mothballing of the Trekkopje project Areva has supplied water to other bulk users in the region. NamWater is in discussion with Areva to purchase the facility and it is expected that water from the facility will be available to the region. The water supply risk has thus been re-assessed as ***Low to Moderate*** for the OS.

25.7.9 Regulation

Namibia is very supportive of mining as can be seen from the history of diamond and uranium mining; the Rössing uranium mine has been in continuous operation for over 40 years. The issues of title to land, permitting, licences, access over public land and possible legal challenges to any of title, right to mine or right to access the licensed mining or EPL areas are all regarded as manageable and a ***Low*** risk.

Permitting

There is currently no reason to believe that the necessary permits required to enable development of the Etango Project will not be obtained in due course, and the level of risk is considered ***Low***.

Royalties and Taxes

An amendment in December 2008 to the Act has provided the Minister for Mines and Energy with the effective discretion to set the mineral royalty for all commodities for all mining projects, including nuclear fuels, at any level.

The 2006-gazetted Government royalty on nuclear fuels in Namibia is 3%. A recent decision by the Minister has resulted in a 6% royalty being imposed on Rössing Uranium Limited, however it is understood that this arrangement will be maintained until an overdue royalty obligation has been settled by Rössing, where after the royalty will revert to the standard rate of 3%. The royalty for all other mines remained unchanged as gazetted in 2006. The 2015 OS has accordingly assumed a 3% royalty to Government.

The risk of changes to royalties (and the corporate tax rate) cannot be discounted in any jurisdiction, but, given Namibia's commitment to development of the mining industry, it is considered no more than ***Low to Moderate*** for a new project.

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25.7.10 Labour and Training

Southern Africa, including Namibia, has a long history of mining developments and operations, and there is a good skill base, including in the Erongo Region. However, the proposed Husab Mine and expansions at Rössing, Langer Heinrich and Trekkopje will put considerable pressure on the pool of skilled and semi-skilled employees. Namibian legislation such as the Affirmative Action (Employment) Act 1998 and anticipated NEEEF makes this more than simply a financial issue to be solved by importing labour.

The risk of not being able to identify suitably trained personnel in any of the positions from unskilled to senior management is regarded as ***Low to Moderate***. Bannerman has every intention of contributing to the operation of technical institutions to train semi-skilled and unskilled workers, establishing training regimes and HR policies and processes that negate the potential risks.

Industrial action is a part of the labour landscape in Africa, and so is to be expected from time to time in the life of an operating mine. The democratic governance and comparative political stability of the country are counters to the possibility of long-term, debilitating industrial action.

25.7.11 Schedule Delays

Project Execution Schedule Delays

The current schedule has been built up from first principles including standard engineering design times, quoted supplier delivery times, historical installation times and industry standard float. The project area is not prone to excessive adverse weather conditions and is serviced by excellent existing infrastructure; however, the study is unable to predict international resource activity during the procurement and construction period, which can have a significant impact on the supply chain and product delivery times.

The 6 month early engineering period will allow a review of the long lead items list prevalent at the time, which will mitigate some of the risk.

The risk of excessive and costly delays to project construction are considered ***Moderate***, mitigated to some extent by a 3 month contingency allowance.

Ramp-up Delays

The risk in prolonged ramp-up to full production is considered ***Moderate***, but mitigated to some extent since:

- Commissioning the crushing circuit on drainage layer material for the heap leach circuit will assist in de-risking the ramp-up period as the materials handling circuits will be fully commissioned, and the operators fully conversant, prior to ore ramp-up
- A production buffer occurs once the solution circuits are commissioned. ILS can be recirculated through the heaps with increasing tenor if there are any delays in ramping up the SX circuit.

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26 RECOMMENDATIONS

Although the 2012 DFS and OS have been completed to an acceptable level, some additional technical investigations are recommended, to increase certainty and reduce risk in the Project's financial outcome.

Other activities that form part of Early Engineering works would not be expected to commence prior to completion of financing, and have not been included.

26.1 METALLURGICAL TEST WORK

Additional metallurgical test work is recommended as follows:

Leaching Test work

- Crib Pilot Plant:

  A test program using the pilot plant built by Bannerman has been underway since April 2015. The test work performed to date is described in Section 13 and additional results continue to be generated.

  It is recommended to treat composite samples, running in a closed circuit with a pilot SX plant and run for at least three cycles. The DFS design values particle size of nominal P80=5.3mm (ie ore prepared using a 10 mm closing screen on the HPGR) and height = 5m should be used.

- This work should be combined with downstream process test work recommended by Bateman and endorsed by Amec Foster Wheeler to minimise overall cost. In respect to heap leaching, this work generates data on a closed circuit system that achieves representative impurity concentration in the SX raffinate returned to the heaps. This testing therefore, generates more accurate information in regard to kinetics, recovery, and acid use and percolation rates Heap Leach – Variability Program:

  To assess the characteristics of the various ores. This would be conducted using 5m columns in open circuit. The first test(s) would run a column using a sub-sample of the material used in the cribs so that a set of factors can be derived to allow column results to be scaled up to plant conditions.

- Large Scale Heap test work (possible only)- Trial Heap:

  Trial heaps are designed to conduct large scale heap leaching test work, commonly using between 5 000 to 50 000 t of ore. These are generally operated in closed cycle solution circulation mode and performed at the optimum conditions established in column and cribs laboratory testing. The trial heap is constructed at the preferred operational height using the selected stacking method and using the selected pad configuration (liners and drainage layers). In case of Etango, the main points to test, other than kinetics and reagent consumptions, are: agglomeration and stacking process, solution percolation and unconstrained heap stability (not possible to test in columns or cribs, since the ore is enclosed by a wall).

  Although there are significant benefits derived from running the crib pilot plant Amec Foster Wheeler notes there are additional benefits from a trial heap. It will either confirm the current design, operability and recovery (and increase project design confidence) or it will provide timely updated parameters for incorporation into the design of the full scale plant. The decision to proceed with a trial heap will naturally depend on the level of perceived process risk that remains in the existing design following crib testing.

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In the opinion of Amec Foster Wheeler the decision to proceed or not with a trial heap should be made when the current crib test work program is nearing its completion. If the process risk level at this point is excessive and can only be mitigated by conducting a trial leach then its inclusion in the test program will be recommended.

- Heap Leach – Materials Testing:

  Laboratory test work to:

  - Test the geo-synthetic clay liner (GCL) in terms of its compatibility with the acidic leach solution, and in terms of its effects on the stability of the heap
  - To confirm whether or not native soils are suitable as a bedding layer
  - Prove that screened material from the mine or nearby burrow pits can be used as a drainage layer (high resistance to acid attack) and, if so, determine the optimum thickness for the two drainage layers
  - Simulate the whole pad, liner and drainage sequence in respect to stability.

Metallurgical Test work – Solvent Extraction, Precipitation and Product Quality

For design purposes, Amec Foster Wheeler recommends that a pilot plant run should be completed with the chosen equipment to confirm engineering parameters for the design.

Amec Foster Wheeler strongly recommends that a fully integrated heap leach and hydrometallurgical laboratory scale pilot campaign be operated for at least 14 days, and possibly up to 30 days, at steady state to fully evaluate the performance of the selected processing circuit. Various laboratory batch tests will be required prior to and in support of the pilot work.

The principal purpose of this work is to obtain:

- An understanding of the impurity build up and any deleterious effects
- Engineering design parameters (including flux rates and scrubbing/regeneration requirements)
- Confirmation of uranium recovery and purity.

The pilot plant should cover the following areas as a minimum:

- Batch bench scale tests for the precipitation and recovery of a uranium product from the loaded strip liquor produced by solvent extraction batch contact
- Confirm clarifier parameters and undertake screening of flocculants and coagulants to produce clarified feed to the SX system
- The hydraulic behaviour of the PLS should be tested for the effects of contaminants in the organic phase and for the creation of emulsifying species, like jarosites, in the aqueous phase
- Measurement of SX performance at the realistic PLS uranium and impurity concentrations produced by the integrated circuit, to prove recovery efficiencies, scale-up data and indication of likely crud issues
- Optimising and obtaining data regarding the ADU precipitation, thickening and filtration steps and therefore a measurement of the uranium recovery and uranium product quality.

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Details of the test work programs remain to be developed, but preliminary cost estimates are as shown in Table 26.1.

Table 26.1
Preliminary Process Test work Budget

Item	**Cost ($)**
Leaching variability	100,000
Heap leach materials testing	25,000
Pilot SX / precipitation / recovery	250,000
Total	**375,000**

26.1.1 Engineering

A program of site geotechnical investigations is required prior to detailed engineering design, primarily in order to provide engineering data for the design of foundation construction works for the plant, waste rock dumps, heap leach pad, leachate collection ponds and the Ripios disposal area.

These investigations would include trial pit excavation, mechanical auger drilling, standard Penetration Testing, constant and falling head permeability tests, in situ testing, sampling and laboratory testing.

Trial pits would improve understanding of surface conditions and lithology of the underlying soils, and investigate the potential borrow areas for structural earth-fill, aggregate and sand for construction.

Drilling would provide samples for laboratory testing of material strength properties and also confirm groundwater conditions across the site.

A proposal to undertake this work has been received. No costing was provided, but a preliminary cost estimate is $100,000.

26.1.2 Project Advancement

It is anticipated that Bannerman will continue to investigate sources of Project financing and continue discussions with potential purchasers of uranium. In addition Bannerman intends to:

- Maintain contact with regulatory authorities regarding licences, permitting and environmental management
- Advance discussions with NamPower, NamWater and the Port of Walvis Bay concerning supply of external infrastructure and port services.

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New Brunswick Securities Commission
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Prince Edward Island Securities Office, Consumer, Corporate and Insurance Services Division, Office of the Attorney General
Securities Commission of Newfoundland and Labrador
Yukon Registrar of Securities
Northwest Territories Registrar of Securities
Nunavut Registrar of Securities

CONSENT OF QUALIFIED PERSON – TECHNICAL REPORT, ETANGO PROJECT URANIUM PROJECT, OPTIMISATION STUDY

I, Leon Fouché, B Eng(Mining), FAusIMM consent to the public filing of the technical report titled "Etango Uranium Project, Nambia, Optimisation Study; National Instrument 43-101 Technical Report" dated 24 December 2015 (the "Technical Report") by Bannerman Resources Limited.

I also consent to any extracts from or a summary of the Technical Report in the press release of Bannerman Resources Limited dated 24 December 2015 (the "Disclosure Document").

I certify that I have read the Disclosure Document and that it fairly and accurately represents the information in the sections of the Technical Report for which I am responsible.

Dated December 24, 2015.

Leon Fouché, BEng(Mining), FAusIMM

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Certificate of Qualified Person

As an author of the report entitled "Etango Uranium Project, Namibia, Optimisation Study, National Instrument 43.101 Technical Report" dated 24 December 2015, on the Etango Project property of Bannerman Resources Limited (the "Technical Report"), I hereby state:

1. My name is Leon Fouché, BEng (Mining), FAusIMM and I am Study Manager of Bannerman Resources Ltd, Unit 1, 2 Centro Avenue, Subiaco, WA, 6008, Australia.
2. I am a graduate of University of Pretoria, South Africa and hold a Bachelor degree, majoring in Mine Engineering (1988).
3. I am a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM, # 222693).
4. I am a practising mining engineer and have practiced my profession continuously since 1989. My relevant experience for the purpose of the Technical Report is:
   - Operational experience on numerous mines in South Africa and Namibia including Rössing Uranium Ltd.
   - Mine planning experience on a large number of projects, including 12 years at Rössing Uranium Ltd.
   - Project manager for numerous feasibility studies, including projects in Africa.
5. I am a "qualified person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
6. I am responsible for Sections 2, 3, 4, 5, 6, 15 and 16 of the Technical Report, responsible for related parts of Sections 1, 21, 25, 26 and 27.
7. I have personally visited the Etango Uranium Project from 18 July to 27 July 2015.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Sections of the Technical Report set out above contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated at Perth, Western Australia, on 24 December 2015.

Leon Fouché
Study Manager
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Ontario Securities Commission
Autorité des marché financiers
New Brunswick Securities Commission
Nova Scotia Securities Commission
Prince Edward Island Securities Office, Consumer, Corporate and Insurance Services Division, Office of the Attorney General
Securities Commission of Newfoundland and Labrador
Yukon Registrar of Securities
Northwest Territories Registrar of Securities
Nunavut Registrar of Securities

Dear Sirs and Mesdames,

CONSENT OF QUALIFIED PERSON – TECHNICAL REPORT, ETANGO PROJECT, NAMIBIA: BANNERMAN RESOURCES LIMITED

I, Ian Glacken, BSc(Hons), MSc, MS, Grad.Dip.Comp, FAusIMM(CP), MIMMM, CEng, DIC, consent to the public filing of the technical report titled "Etango Uranium Project, Nambia, Optimisation Study; Updated Technical Report" dated 24 December 2015 (the "Technical Report") by Bannerman Resources Limited.

I also consent to any extracts from or a summary of the Technical Report in the press release of Bannerman Resources Limited dated 24, December, 2015 (the "Disclosure Document").

I certify that I have read the Disclosure Document and that it fairly and accurately represents the information in the sections of the Technical Report for which I am responsible.

Dated December 24, 2015.

Ian Glacken, BSc(Hons), MSc, MS, Grad.Dip.Comp, FAusIMM(CP), MIMMM, CEng, DIC

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CONSENT of AUTHOR

TO: British Columbia Securities Commission
Montreal Securities Commission
Alberta Securities Commission
Ontario Securities Commission

AND TO: Bannerman Resources Limited

I refer to the technical report dated 24 December 2015 titled "*Etango Uranium Project, Namibia, Optimisation Study, National Instrument 43-101 Technical Report*" (the "Technical Report").

I am responsible for the preparation in the Technical Report of Sections 13, 17, 18 and 21 (excluding mining costs and those parts relating to the metallurgy and plant design of the comminution section of the treatment plant) and the associated text in the Summary, Interpretation and Conclusions, and Recommendations.

I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated, this 24th day of December 2015.

(Signed)

Peter Nofal

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Certificate of Qualified Person

As an author of the report entitled "*Etango Uranium Project, Namibia, Optimisation Study, National Instrument 43.101 Technical Report*" dated 24th December 2015, on the Etango Project property of Bannerman Resources Limited (the "Technical Report"), I hereby state:

1. My name is Peter Nofal and I am Technical Director - Studies with the firm of AMEC Foster Wheeler, L7 197 St Georges Terrace, Perth, 6000 Australia.
2. I am a chemical engineer and a Fellow of the AusIMM (207660).
3. I am a graduate of The University of the Witwatersrand in South Africa with a BSc(Eng) in 1982 and a BCom(hons) from the University of South Africa in 1990.
4. I have practiced my profession continuously since 1982.
5. I am a "qualified person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
6. I visited the Etango Project property and surrounding areas in 2010.
7. I contributed to and am responsible for Sections 13, 17, 18 and 21 (excluding mining costs and those parts relating to comminution) of the Technical Report and the associated text in the summary, conclusions and recommendations.
8. As of the effective date of the Study, to the best of my knowledge, information and belief, the parts of the Study for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Study not misleading.
9. I am independent of Bannerman Resources pursuant to section 1.4 of the Instrument.
10. I have read the National Instrument and Form 43-101F1 (the "Form") and the Study has been prepared in compliance with the Instrument and the Form.
11. I do not have nor do I expect to receive a direct or indirect interest in the Etango Project property of Bannerman Resources, and I do not beneficially own, directly or indirectly, any securities of Bannerman Resources or any associate or affiliate of such company.

Dated at Perth, Western Australia, on 24 December 2015.

[signed]

Peter Nofal
Technical Director, Studies
(Amec Foster Wheeler)

BSc (Eng)
BCom (Hons)

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Certificate of Qualified Person

As an author of the report entitled "*Etango Uranium Project, Namibia, Optimisation Study, National Instrument 43.101 Technical Report*" dated 24 December 2015, on the Etango Project property of Bannerman Resources Limited (the "Technical Report"), I hereby state:

1. My name is Dean Malcolm David and I am Technical Director - Process with the firm of Amec Foster Wheeler, L7 197 St Georges Terrace, Perth, 6000 Australia.
2. I am a practising metallurgist, a Fellow and a Chartered Professional (CP) Metallurgy of the AusIMM (102351).
3. I am a graduate of South Australian Institute of Technology in South Australia with a BAppSc in Metallurgy in 1982.
4. I have practiced my profession continuously since 1982.
5. I am a "qualified person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
6. I visited the Etango Project property and surrounding areas on 17 April, 2011 and on 23 February, 2015.
7. I contributed to and am responsible for Sections 13 and Section 17 of the Technical Report (those parts relating to comminution), and the associated text in the summary, conclusions and recommendations.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of Bannerman Resources pursuant to section 1.4 of the Instrument.
10. I have read the National Instrument and Form 43-101F1 (the "Form") and the Technical Report has been prepared in compliance with the Instrument and the Form.
11. I do not have nor do I expect to receive a direct or indirect interest in the Etango Project property of Bannerman Resources, and I do not beneficially own, directly or indirectly, any securities of Bannerman Resources or any associate or affiliate of such company.

Dated at Perth, Western Australia, on 24 December 2015.

[signed] 24/12/2015

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CONSENT of AUTHOR

TO: British Columbia Securities Commission
Montreal Securities Commission
Alberta Securities Commission
Ontario Securities Commission

AND TO: Bannerman Resources Limited

I refer to the technical report dated 24 December, 2015 titled "*Etango Uranium Project, Namibia, Optimisation Study, National Instrument 43-101 Technical Report*" (the "Technical Report").

I am responsible for the preparation of those parts of Sections 1, 13, 17, 25 and 26 of the Technical Report relating to metallurgy and plant design for the comminution section of the treatment plant.

I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated, this 24th day of December 2015.

24/12/2015

(Signed)
Dean David

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