Etango Uranium Project, Namibia

Feasibility Study

National Instrument 43-101 Technical Report

Bannerman Resources Limited

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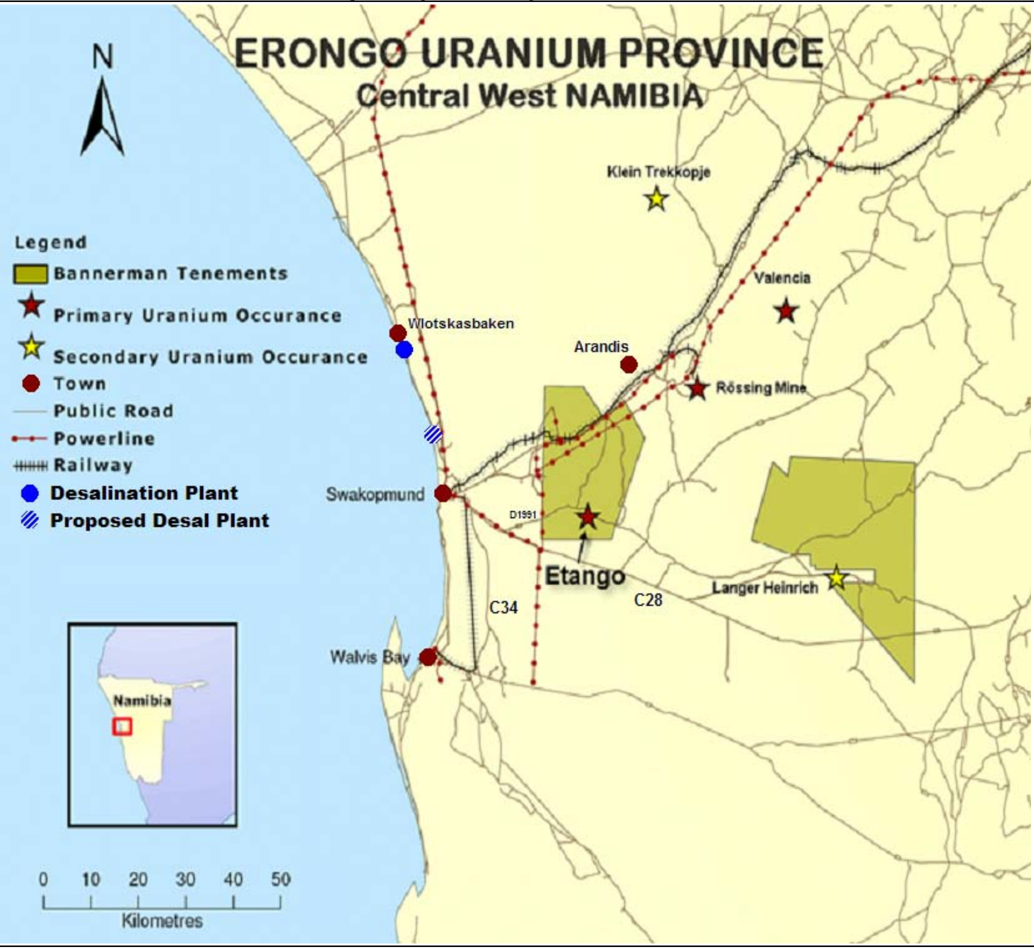
1 SUMMARY

1.1 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 Etango Project lies within exclusive prospecting licence 3345 (EPL 3345), otherwise known as the Etango licence.

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 2010.

Work commenced on a Definitive Feasibility Study (DFS) in April 2011, with results as detailed in this report. Subject to licensing and project financing, Bannerman is planning to commission the Etango Project in 2016.

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

1.2 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 six kilometre (km) long contiguous 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).

In October 2010, Coffey Mining Pty Ltd (Coffey Mining) estimated the most recent mineral resource for Etango as summarised in Table 1-1, reported at a cut-off of 100ppm.

Table 1-1
Etango Deposit, Etango Project, Namibia – October 2010 Resource Estimate

Classification	Lower Cut (ppm)	Tonnes Above Cut-off (Mt)	U_3O_8 (ppm)	Contained U_3O_8 (t)	Contained U_3O_8 (M lb)
Measured	100	62.7	205	12,900	28.3
Indicated	100	273.5	200	54,600	120.4
Meas+Ind	**100**	**336.2**	**201**	**67,500**	**148.7**
Inferred	100	45.7	202	9,200	20.3

In addition, Inferred Mineral Resources were estimated for adjacent uranium deposits at Ondjamba and Hyena in Table 1-2.

Coffey Mining reviewed drill sampling and data quality control procedures, and validated the database used for resource modelling.

Table 1-2
Ondjamba and Hyena Deposits – October 2010 Resource Estimate (100ppm cut-off)

Classification	Deposit	Tonnes Above Cut-off (Mt)	U_3O_8 (ppm)	Contained U_3O_8 (t)	Contained U_3O_8 (M lb)
Inferred	Ondjamba	85.1	166	14,100	31.1
	Hyena	33.6	166	5,600	12.3
Total Inferred		**118.7**	**166**	**19,700**	**43.4**

Resources were modelled using ordinary kriging of 3m composites within mineralisation envelopes guided by 75ppm U_3O_8 content and the alaskite boundaries. Variographic analysis was undertaken to provide kriging parameters and search radii. Block dimensions were 25x25x10m (XYZ) with sub-blocking to 6.25x6.25x1.25m. An average density of 2.64t/m^3 was utilised throughout, since statistical analysis demonstrated no meaningful variation with depth, grade or geological variable.

The resources were classified by Coffey Mining according to Canadian National Instrument 43-101 (NI43-101) criteria.

On a regional scale, the Etango deposit lies within the Southern Central Zone of the northeast-trending branch of the Damaran orogenic belt, an area that includes the Rössing mine and similar uranium-enriched alaskites at Husab (Rössing South).

1.3 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 narrow 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.

Coffey Mining estimated the JORC and NI43-101 Mineral Reserves for the Etango Staged Design at 279.6Mt at 194ppm U_3O_8 reported above a 70ppm U_3O_8 lower cut-off. The reserve consists of 64.2Mt at 194ppm U_3O_8 of Proven Mineral Reserve and 215.3Mt at 193ppm U_3O_8 of Probable Mineral Reserve.

1.4 Metallurgical Testwork

A series of bench-scale metallurgical testwork programs have been completed since 2008, with emphasis on optimisation of comminution, leaching, 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 leaching and heap leaching have been tested in the laboratory in acidic environments. Heap leaching is the preferred method for extracting uranium from the ore on a cost-benefit basis, the 1-2% reduction in recovery (compared to agitated 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 testwork 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.

1.5 Plant and Infrastructure Design

1.5.1 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.

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.

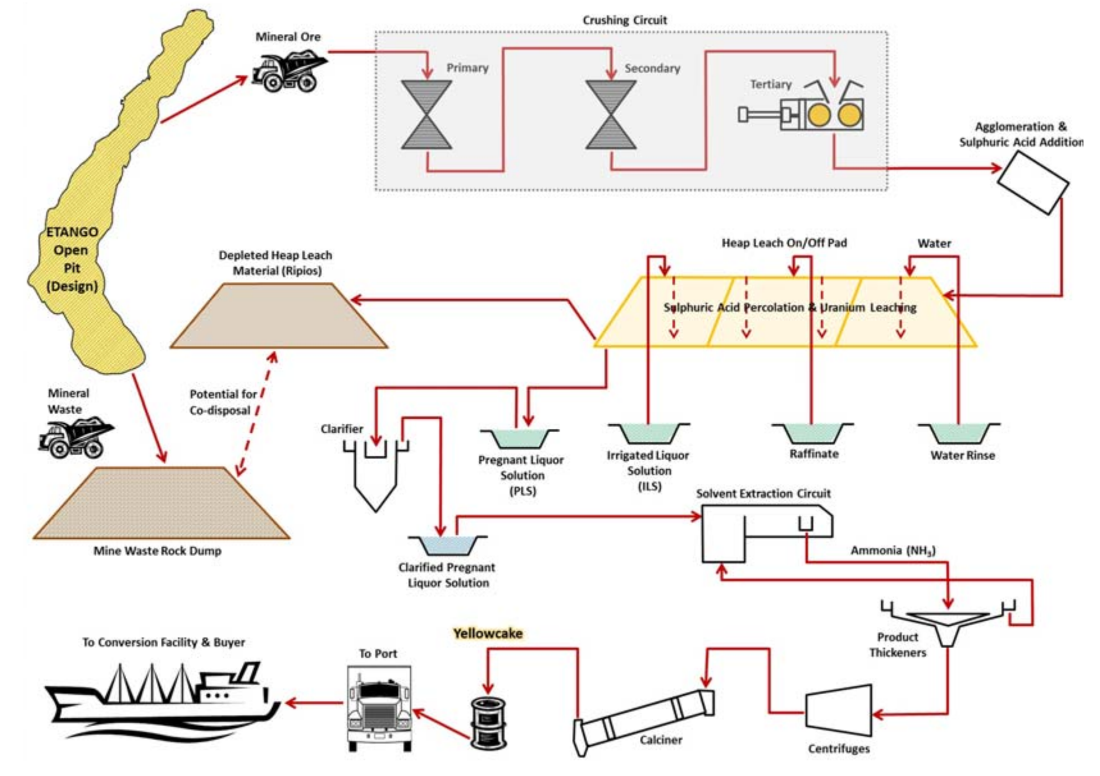
Agglomeration and Stacking

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.

Figure 1-2

Simplified Etango Flowsheet



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 a shiftable conveyor and the Ripios pad boom stacker that places the depleted material onto the unlined 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, 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.

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.

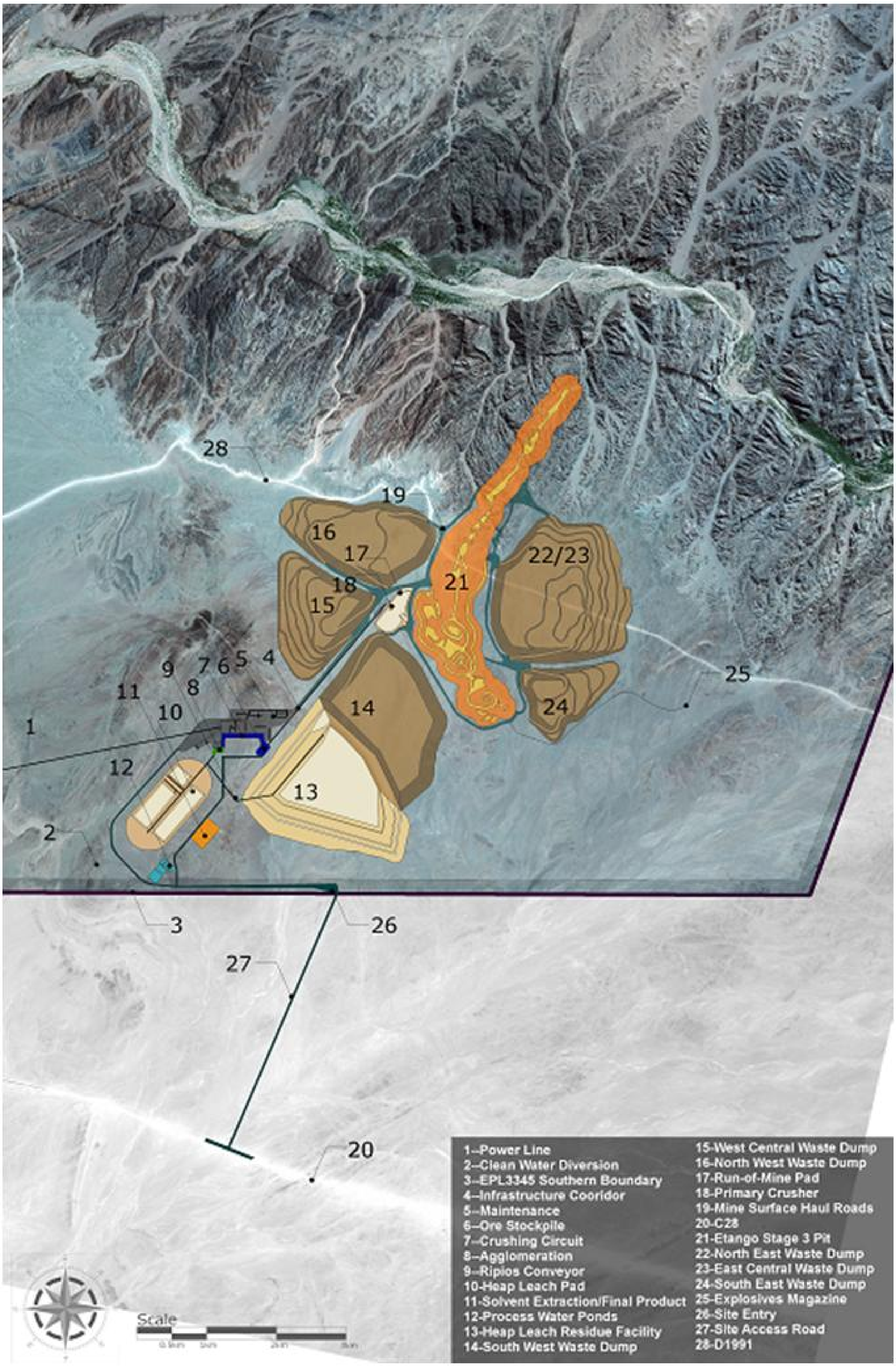
The dominant reagent by volume, bulk concentrated sulphuric acid (98% by weight), is shipped in and transferred to storage tanks at the port of Walvis Bay. From there, the acid is transported to site and transferred to four storage tanks, this being sufficient for 28 days of operations.

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.

Figure 1-3

Etango Site Layout



1.5.2 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 proposed new 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 is 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.

1.6 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.

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 are constructed as part of the final landform.

1.6.1 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.
- 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.

1.6.2 Groundwater Chemistry

Analysis of samples 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.

1.6.3 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.

1.6.4 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.

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

1.7 Capital Costs

1.7.1 Mining Capital Costs

The majority of mining capital expenditures were derived from quotations obtained from major equipment suppliers such as Komatsu and Caterpillar, with the balance being derived from Coffey Mining's in-house cost database and estimates supplied by Bannerman or AMEC Australia Pty Ltd (AMEC).

Mining capital cost estimates include $126.6M in preproduction capital and $361.3M in sustaining capital (including a $25.2M salvage credit at the end of life). The capital cost estimate was based on Q3 2011 quotations and has been completed to an accuracy of ±15%.

1.7.2 Plant and Infrastructure Capital Costs

Comminution, heap leach plant and site infrastructure capital costs have been estimated by AMEC, with Bateman Engineering Pty Ltd (Bateman) estimating the cost of the SX/metal recovery section of the plant. Owner's costs to cover corporate, management and administrative costs, as well as capitalised pre-production operating costs, have been supplied by Bannerman.

Total plant and infrastructure capital costs are estimated to be $660.5M (excluding contingency) as at 1 December 2011.

The estimate has been completed to an accuracy of ±15% and includes Direct and Indirect costs, engineering accuracy provisions (averaging 10.1% of Direct costs or $53.53M) and costs for engineering, procurement and construction management (EPCM) by an independent contractor. No provision has been included for inflation, nor for a Project or Owner's contingency.

1.7.3 Owner's Capital Costs

Owner's costs have been determined by Bannerman to be $40.0M, 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 development seed capital (nominal $6M)
- Environmental site assessment and monitoring
- Insurance
- Sterilisation drilling and on-site metallurgical testing
- Closure costs.
- Closure costs show as sustaining capital, and are incurred primarily at the end of the Project.

1.7.4 Total Project Capital Costs

These are summarised in Table 1-3. Pre-production capital costs total $870.3M, whilst there is a further requirement for $161.5M in working capital to cover early period sustaining capital items (primarily fleet build-up) and operating costs before positive cash flow occurs.

Sustaining capital of $380.94M 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, sale of selected items of the mining fleet and recovery of first fill materials and reagents as they are recovered via operating costs at the end of the project.

Table 1-3 Project Capital Cost Expenditure Summary			
Area	**Pre-production**	**Sustaining**	**Total**
Mining	126.63	361.35	487.98
Process Plant	354.44	-	354.44
Site Infrastructure	91.10	5.77	96.87
External Infrastructure – Port	3.36	0.91	4.27
External Infrastructure – other	43.22	-	43.22
Miscellaneous	44.31	(12.29)	32.02
Indirects	113.73	(7.30)	106.43
Accuracy Provision	53.53	-	53.53
Owner's Costs[1]	40.01	32.50	72.51
Owner's Contingency	-	-	-
Total Project	**870.33**	**380.94**	**1,251.27**

1.8 Operating Costs

1.8.1 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.

A breakdown of the mine operating costs is provided in Table 1-4. Diesel costs are the largest single component of mine operating cost.

Table 1-4 Summary of Life of Mine Operating Costs			
Item	**Cost ($M)**	**Cost ($/t mined)**	**% of Cost**
Fixed	326.7	0.27	13.7
Drill and Blast	737.0	0.61	30.8
Load and Haul (including ancillary equipment)	1,328.0	1.09	55.5
Total	**2,391.7**	**1.97**	**100.0**

1.8.2 Plant and Infrastructure Operating Costs

The process operating costs reflect operation at a throughput of 20Mtpa.

The various process plant operating costs are summarised in Table 1-5.

[1] Including rehabilitation costs.

Table 1-5 Summary of Plant and Infrastructure Operating Costs			
Item	**Cost (M$/a)**	**Cost ($/t of ore[2])**	**% of Cost (%)**
Acid	35.88	1.79	25.4
Reagents	25.72	1.29	18.2
Power	26.11	1.31	18.5
Labour	12.23	0.61	8.7
Maintenance Materials	18.93	0.95	13.4
Water	12.92	0.65	9.2
Consumables	6.90	0.34	4.9
Miscellaneous	2.43	0.12	1.7
Total	**141.12**	**7.06**	**100.0**

Table 1-5 does not take account of annual variations in mining and processing tonnages and grades. The impact of these changes is reflected in Table 1-7.

1.8.3 Owner's Operating Costs

Owner's operating costs are equivalent to $1.21/t crushed, as summarised in Table 1-6.

Table 1-6 Summary of Owner's Costs		
Item	**Annual (Average) ($ M/a)**	**Unit Cost LOM ($/t Ore)**
Corporate and Owner's Labour	12.09	0.673
Total Site Office Administration	0.23	0.013
Total Personnel Expenses	4.22	0.230
Total Insurances and Government Fees	4.25	0.232
Site-Catering Facilities	0.44	0.024
Environmental Monitoring	0.30	0.016
Total Transportation Costs	0.20	0.011
Community Relations / Corporate Responsibility	0.12	0.006
Other	0.04	0.002
Total	**21.85**	**1.207**

Principal costs are for Corporate and Owner's Labour, Training and Insurances.

1.8.4 Total Project Operating Costs

Total operating costs for the Project are $16.93/t ore or $45.71/lb U_3O_8 over the LOM (Table 1-7).

[2] Based on 20Mt/a throughput.

Table 1-7 Summary of Total Operating Costs – LOM					
Item	**Cost ($/t of ore Yr 1-5)**	**Cost ($/t of ore LOM)**	**Cost ($/lb U_3O_8 Yr 1-5)**	**Cost ($/lb U_3O_8 LOM)**	**% of Cost**
Mining	7.87	8.55	19.83	23.09	50.5
Processing and Infrastructure	7.10	7.17	17.90	19.36	42.4
Owner's Costs	1.24	1.21	3.12	3.26	7.1
Total	**16.21**	**16.93**	**40.85**	**45.71**	**100**

1.9 Project Financial Modelling

1.9.1 Base Case

Financial modelling has been undertaken on a Project (100% equity) basis using a cash flow model, with and without taxation.

The Base Case model uses the DFS mining and processing schedules and capital and operating costs. The Base Case uranium oxide price used is $75/lb as provided by Bannerman, based on a review of forecasts supplied by banking institutions and broking firms. A state royalty of 3% and off-site costs of $1.10/lb are included.

The key outputs from the financial model based on the above assumptions are reported for the first 5 years of the modelled operation and for the life of mine in Table 1-8.

A discount rate of 8% is used.

Base Case after-tax net present value (NPV) is $68.7M, with an internal rate of return (IRR) of 9.2% and payback after 6 years. For the pre-tax case, NPV is $238.1M and IRR 11.6%, while payback remains at 6 years.

Table 1-8
Key Financial Model Outputs

	First 5 Years	Life of Mine (Excluding Tax)	Life of Mine (Including Tax)
Project Economics			
NPV at a Discount Rate of 8% ($M)	-	238.1	68.7
Internal Rate of Return (%)	-	11.6%	9.2%
Payback Period from Start of Production	-	6	6
Production			
Quantity Ore Treated (Mt)	89.3	279.6	
Uranium Oxide Produced (t U_3O_8)	16,209	46,980	
Uranium Oxide Produced (M lb U_3O_8)	35.7	103.6	
Revenue			
Average U_3O_8 Base Price ($/lb U_3O_8)	75	75	
Net Revenue ($M, after royalties)	2,378	7,421	
Operating Unit Costs			
On-Site Costs/tonne Ore Treated			
Mining	7.87	8.55	
Processing (including infrastructure maintenance)[3]	7.10	7.17	
Owners costs (including administration)	1.24	1.21	
Total Operating Costs ($/t ore)	**16.21**	**16.93**	
Total Operating Costs ($/lb produced)	**40.85**	**45.71**	
Marketing, freight and conversion	1.10	1.10	

The Project is modelled to produce between 6 and 9Mlb U_3O_8 per year. The average cash operating cost in the first 5 years is estimated at $40.85/lb U_3O_8 and over the life of mine is estimated at $45.71/lb U_3O_8.

1.9.2 Sensitivity Analysis

Revenue

The financial sensitivity analysis demonstrates that the economic outcome of the Etango Project is highly sensitive to changes in the uranium price or other revenue factors (grade and recovery). A negative movement of 10% from the base case assumption of $75/lb U_3O_8 results in the pre-tax NPV reducing from $238M to negative $121M. Conversely, the Project would benefit greatly from increases in U_3O_8 prices with an increase of 10% in price yielding an NPV of $597M.

Operating Costs

Financial performance is also very sensitive to changes in operating costs. Increases of 10% and 20% in the base case operating cost assumptions produce significant adverse changes in the pre-tax NPV from $238M to $9M and negative $220M respectively, the latter with a pre-tax IRR of 4.4%.

[3] Difference from Table 1-5 due to assumption of head grade

Likewise, cost reductions of 10% and 20% from the base case assumptions result in the pre-tax NPV increasing from $238M to $467M and $696M respectively, the latter with a pre-tax IRR of 17.7%.

The largest individual component of operating cost is diesel cost, which represents 15.3% of total operating cost. Acid and power costs are also significant components.

Capital Costs

Increases of 10% and 20% in the base case capital cost assumptions (excluding working capital) produce adverse changes in the pre-tax NPV from $238M to $137M and $37M respectively, the latter with a pre-tax IRR of 8.5%.

Likewise, capital cost reductions of 10% and 20% from the base case assumptions result in the pre-tax NPV increasing from $238M to $339M and $440M respectively, the latter with a pre-tax IRR of 15.8%.

Cumulative Impact

Only one parameter at a time was varied in the financial analysis. However, it is possible that several aspects could vary from the base case at the same time, the result of which could be magnification or mitigation of the economic impact.

1.10 Environmental and Permitting

1.10.1 Environmental Approvals

Bannerman has received Environmental Clearances for an earlier concept to establish the Etango Project, based on an Environmental and Social Impact Assessment (ESIA) and Environmental and Social Management Plan (ESMP) completed in December 2009.

A revised ESIA has been submitted for the expanded operation. This submission also takes into account recommendations from the Uranium Rush Strategic Environmental Assessment (SEA) regarding cumulative impacts to the Erongo Region; revised visual, air quality and noise studies incorporating additional project data and extended groundwater monitoring and hydrogeological groundwater modelling investigations.

1.10.2 Mining Licence Application

Bannerman submitted a mining licence (ML) application for the Etango Project in December 2009, based on the December 2009 PFS. Since that time, the mineral resource estimate for the Etango Project has expanded and the site layout and processing flowsheet have undergone changes as noted above.

Upon receipt of an updated Environmental Clearance for development of the Etango Project, Bannerman will lodge supplementary information with the Namibian Ministry of Mines and Energy in further support of the existing Etango ML application.

1.11 Project Development

A project development schedule has been outlined as part of the DFS, indicating completion of engineering design, procurement, transport and construction over a 33 month period following Project approval, with ramp-up to design tonnages after 45 months (Table 1-9). 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 1-9
Project Development Milestones

Task	Month
Commence early works	-6
Project approval, i.e. receipt of regulatory approvals/project financing	**0**
Commence site works	9
Commence commissioning (includes 3 month contingency)	30
Commence ramp-up (with contingency)	36
First shipment (with contingency)	42
Ramp-up to design tonnages	45

The key drivers of the development schedule are Project approval followed by the timely delivery of long lead equipment – 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.

1.12 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 1-10 arranged in general order of likelihood and importance.

Table 1-10
Non-Resource / Mining Economic and Technical Risk Assessment

Item	Assessed Risk to Project
U_3O_8 price	Major
Water supply not available	Major
Mining equipment under-performance	High
Mine operating costs over-run (sustained increase in labour / materials costs)	High
Operating cost overrun - diesel	High
Capital cost overrun	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 utilised in the DFS. A number of market analysts expect the fundamentals of the uranium market to improve and the uranium price to increase from current long-term levels of approximately $60/lb to

$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 water supply: Bannerman believes that the additional supply will be developed based on discussions with NamWater, but this remains a serious risk until NamWater has commenced construction of the proposed new desalination plant. Other options may exist, but there are a number of competing projects in the region.

Operating cost overruns would have serious implications for the project. Principal components are diesel (15.5% of total operating cost), sulphuric acid usage (11%), and power costs (7.7%)

1.13 Conclusions

The results of the DFS indicate that the Etango Project is 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 grade of the deposit is relatively low, throughput is high and the Project is capital intensive. Consequently, a uranium price of $75/lb or more is required in order to generate a reasonable return.

The Project is highly sensitive to variations in revenue (U_3O_8 price, grade, recovery), and also to operating costs, in particular to diesel, power and acid costs.

2 INTRODUCTION

2.1 Bannerman Resources Limited

This Technical Report has been prepared for Bannerman Resources Limited (Bannerman), a public company listed on the Australian, Toronto and Namibian Stock Exchanges. Bannerman's corporate office is Level 1, Suite 18, 513 Hay Street, Subiaco, Perth, Western Australia, 6008.

2.2 Scope of Work

Following completion of the PFS Update (PFSU) in 2010, undertaken by AMEC, additional metallurgical testwork was undertaken, after which AMEC was commissioned to complete process and engineering design, capital and operating cost estimation and compilation of a DFS incorporating a financial cash flow model developed by Bannerman. The DFS scope required capital and operating costs to be estimated to an accuracy of ±15%.

Coffey Mining contributed the 2010 Mineral Resource model, which remains current, and undertook geotechnical input, mine design, mining cost estimation and Mineral Reserve estimation for the DFS.

A. Speiser Environmental Consultants (ASEC) and Environmental Resources Management (ERM) provided environmental and social impact studies as part of the ESIA.

Bannerman publicly announced results of the DFS in April 2012, and AMEC 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 Manual, Canadian National Instrument 43-101, Companion Policy 43-101CP, and Form 43-101F1. Mineral Resource and Mineral Reserve classifications conform to those adopted by the CIM Council in November 2004. The report is also consistent with the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' of December 2004 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).

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 testwork undertaken by recognised testwork laboratories, notably ALS Ammtec and SGS in Perth, Western Australia, and Bureau Veritas in Swakopmund, Namibia.
- Information from Bannerman personnel in relation to past history and previous studies on the Etango Project not undertaken by AMEC or Coffey Mining.

- Field observations, reports and data obtained during field trips in 2007, 2008, 2009, 2010, 2011 and 2012 by Mr Neil Inwood, Mr Brian Wolfe and other Coffey Mining personnel.
- Etango Project Environmental and Social Impact Assessment and Environmental and Social Management Plan prepared by ERM in April 2012.
- SLR, operating in Australia as Metago Environmental Engineers (Australia) Pty Ltd undertook stormwater engineering and Ripios management studies.
- RPS Aquaterra completed groundwater modelling to define groundwater conditions in the open pit area, and the effects on mining.
- Various published historic, technical and scientific papers and reports.
- Digital exploration data.
- Published information relevant to the Etango Project area and the region in general.
- A letter report by H.D.Bossau & Co of Windhoek, Namibia, dated 15 December 2011, regarding legal opinion on the status of Bannerman Mining Resources (Namibia) (Proprietary) Limited and the Etango tenement.

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

2.4 Participants

The following qualified persons (QPs) have been involved in compilation of the NI43-101 report:

- AMEC
  - Peter Nofal – Manager, Studies. Responsible for Sections 5, 13, 17 to 19, 21 (excluding mining costs), 22 and 24.
  - Dean David – Technical Director, Process. Responsible for those aspects of Sections 13 and 17 of the report relating to ore comminution.
  - Dan Greig – Principal Geologist. Responsible for overall compilation of the report, with specific responsibility for Sections 2, 3, 4, 6, 20 and 27.
- Coffey Mining
  - Mr Brian Wolfe – Principal Resource Geologist of Coffey Mining. Responsible for Sections 7-12, 14 and 23.
  - Mr Harry Warries – Principal Mining Consultant. Responsible for Section 15, 16 and those parts of Section 21 relating to mining costs.

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

Coffey Mining personnel undertook site visits to the Etango Project in August 2007, April 2008, October 2009, August 2010, September 2011 and March 2012, to review the data collection procedures and geological and mining aspects of the Project. The most recent visit was undertaken by Brian Wolfe in March 2012.

AMEC 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.

2.6 Qualifications and Experience

AMEC 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, full-time Process Consultant for AMEC, is responsible for process testwork and design relating to the proposed comminution circuit. Mr David has over 30 years experience in mineral processing research, operations, management and consulting. He has visited the site. Mr David is a Fellow of the Australasian Institute of Mining and Metallurgy, and has the appropriate relevant qualifications, experience and independence to be generally considered a Qualified Person as defined in the Instrument.

- Peter Nofal, Manager Studies for AMEC, has visited the site, and is responsible for those sections of the report relating to process testwork, 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 BComm majoring in Business Economics, and has 30 years of experience. He is a Fellow of the Australasian Institute of Mining and Metallurgy. Mr Nofal has the appropriate relevant qualifications, experience and independence to be generally considered a Qualified Person as defined in the Instrument.

- Dan Greig, overall compiler of the report, is Principal Geologist and an employee with AMEC, having worked with the company for over 16 years. Mr Greig is a professional geologist with 42 years' experience in mining and resource geology, and feasibility study management. He is a Member of the Australian Institute of Geoscientists, and has the appropriate relevant qualifications, experience and independence to be generally considered a Qualified Person as defined in the Instrument.

Coffey Mining is an integrated Australian-based consulting firm, which has been providing services and advice to the international mineral industry and financial institutions since 1987. In September 2006, Coffey International Limited acquired RSG Global. Coffey International Limited is a highly respected Australian-based international consulting firm specialising in the areas of geotechnical engineering, hydrogeology, hydrology, tailings disposal, environmental science and social and physical infrastructure.

- Brian Wolfe, the author of the geological and resource sections of this report, is a full time employee of Coffey Mining and a professional geologist with 20 years' experience in mining and resource geology. He has a BSc (Hons) in Geology and a Postgraduate Certificate in Geostatistics (2007). Mr Wolfe is a Member of the Australian Institute of Geoscientists, and has the appropriate relevant qualifications, experience and independence to be generally considered a Qualified Person as defined in the Instrument.
- Harry Warries, the professional responsible for geotechnical, mining and Mineral Reserve parts of this report, is a full time employee of Coffey Mining and a professional mining engineer with over 20 years' experience. Mr Warries holds a Masters degree, majoring in Mine Engineering. Mr Warries is a Fellow of the Australasian Institute of Mining and Metallurgy, and has the appropriate relevant qualifications, experience and independence to be generally considered a Qualified Person as defined in the Instrument.

2.7 Independence

AMEC, Coffey Mining and their employees are considered independent from Bannerman as outlined under Section 1.4 of the Instrument.

None of the parties have any material interest in Bannerman or related entities or interests. Their relationship with Bannerman is solely one of professional association between client and independent consultant. The report was prepared in return for fees based upon agreed commercial rates and the payment of these fees is in no way contingent on the results of the relevant sections.

2.8 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.

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
AMEC	AMEC Australia Pty Ltd

Table 2-1
List of Abbreviations

Abbreviation	Description
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
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
kg	kilogram
kg/t	kilogram per tonne
km	kilometres
km^2	square kilometres
kW	kilowatts
L	Litre
LOM	Life of mine
M	million
m	metres
Ma	million years
MARC	Maintenance and repair contracts
MDRL	Mineral Deposit Retention Licence
Mg	magnesium
mL	millilitre
ML	Mining Licence

Table 2-1
List of Abbreviations

Abbreviation	Description
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
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
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
US$	United States of America dollars
U_3O_8	uranium oxide
w:o	waste to ore ratio
XRF	x-ray fluorescence analysis

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 AMEC and Coffey Mining in preparation of the Technical Report. References to specific input are carried in Section 27.

AMEC and Coffey do not accept responsibility for the accuracy of such input by third parties.

That section of the report concerning mineral tenement status and legal issues associated with the Etango Project is based on a written opinion dated 15 December 2011, supplied by H.D. Bossau & Co, Legal practitioners in Windhoek, Namibia. AMEC has reviewed this document, and has relied on the opinions expressed therein.

Mineral process testwork 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. AMEC has relied on the results of testwork 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 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. Coffey Mining 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 AMEC has relied on their calculations and findings in preparing this Technical Report.

The U_3O_8 price used in the DFS was based on forecasts provided to Bannerman by several banking institutions and broking houses.

The financial model has been prepared by Bannerman using a qualified accountant. AMEC has confirmed the inputs from the DFS used in the financial model e.g. mine and process production schedules and capital and operating costs, but has relied on the cashflow modelling and sensitivity analysis work reported by Bannerman.

4 PROPERTY DESCRIPTION AND LOCATION

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. This district hosts the world's largest open cut uranium mine at Rössing (majority owned by Rio Tinto), as well as Paladin Resources Limited's Langer Heinrich uranium operation.

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 recently 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. The following description is based largely upon information from the World Fact Book (*The World Fact Book, 2007*).

Figure 4-1
Map of Namibia

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) and was then succeeded by the current President, Hifikepunye Pohamba, in March 2005 following a peaceful election. 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 230,000 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, 84% of the population is literate.

The economy is heavily dependent on the extraction and processing of minerals for export. Mining accounts for approximately 25% 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.

An individual Exclusive Prospecting Licence (EPL) can cover an area of up to 1,000km² 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 up to 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 the initial licence period and again after the first renewal period, each time by 25%. There may be scope, if the Minister sees reason, to waive the reduction of the size of the EPL after the initial 3 year period of the licence. 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 the Minister of Mines and Energy, on recommendation by an advisory committee, such granting being based on the 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.

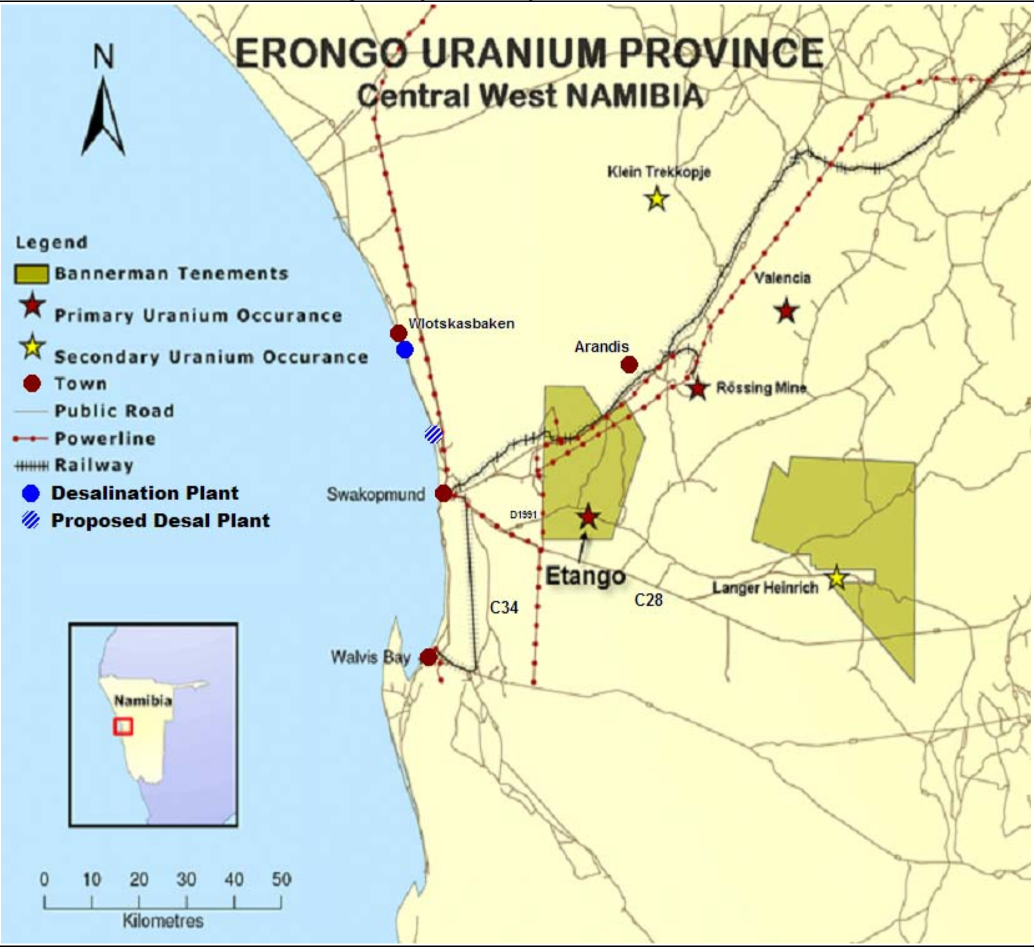
An environmental contract must be completed with the Ministry of Environment and Tourism by applicants for EPLs, MDRLs and MLs. Environmental impact assessments (where relevant) must be made with respect to land disturbance, protection of flora and fauna, water supply, drainage and waste water disposal,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

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 and Swakop River EPL 3346 are owned by the Namibian company Bannerman Mining Resources (Namibia) (Pty) Ltd (Bannerman Namibia), previously called Turgi Investments (Pty) Ltd (Turgi), which manages the Project. Bannerman owns 80% of Bannerman Namibia, while the remaining 20% is held in the name of Mr C. Jones of Perth, Australia.

In March 2012, Bannerman announced signing of a binding Term Sheet with Epangelo Mining Company (Epangelo), or its nominee, to acquire an initial 5% interest and, upon a mine development decision, a further of 5% interest in Bannerman's Namibian subsidiary.

EPL 3345 was granted to Turgi with effect from 27 April 2006 for an initial 3 year period to explore for Nuclear Fuels. The first application for renewal for EPL 3345 was granted on 26 April 2009 for an additional 2 years without any reduction in area. The second application for renewal for EPL 3345 was granted with effect from 27 April 2011 for an additional 2 years with no reduction in area. Following settlement of litigation proceedings with a competing claimant (refer below), a small area was excised from the northeast portion of EPL 3345. EPL 3345 is now 48,690ha in size and has an expenditure commitment of N$11,566,000 in the first year and N$6,550,000 for the second year.

EPL 3346 was also granted to Turgi with effect from 27 April 2006 for an initial 3 year period to explore for Nuclear Fuels. The first application for renewal for EPL 3346 was granted with effect from 27 April 2009 for an additional 2 years without any reduction in area. The second application for renewal for EPL 3346 was granted on 27 April 2011 for an additional 2 years without any reduction in area. The Licence is 80,826ha in size and has an expenditure commitment of N$1,100,000 for the first year and N$750,000 thereafter.

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.

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, which has been delayed by changes to the proposed project and lodging of the new ESIA.

Table 4-1
Etango Project – Tenement Schedule

Tenement Type	Tenement No.	Grant Date	Holder	Area (ha)	Minimum Expenditure First Year (N$)	Minimum Expenditure Subsequent Years (N$)
EPL	3345	27.04.2006	Bannerman Mining Resources (Namibia) (Pty) Ltd	48,690	11,566,000	6,550,000
EPL	3346	27.04.2006	Bannerman Mining Resources (Namibia) (Pty) Ltd	80,826	1,100,000	750,000

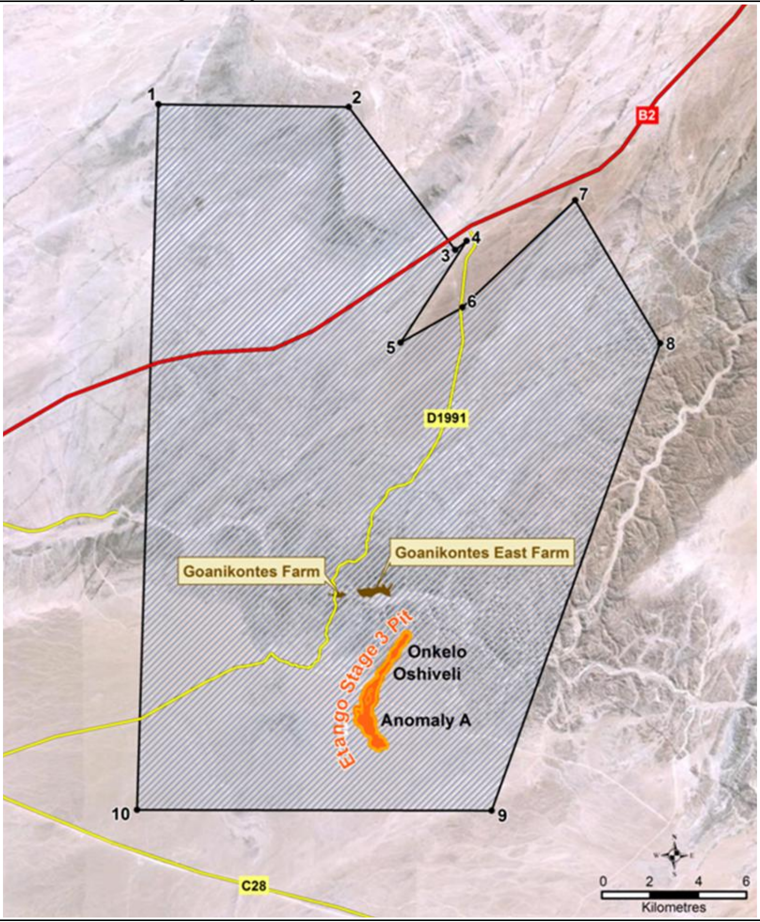
Table 4-2
Etango Project – Tenement Coordinate Summary

	Point	Latitude^	Longitude^
EPL 3345 (Etango) Licence Area – 48,690ha	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
EPL 3346 (Swakop River) Licence Area – 80,826ha	1	-22.61710054	15.21121351
	2	-22.64138218	15.24063254
	3	-22.6077662	15.24682426
	4	-22.61745087	15.50036088
	5	-22.99988448	15.50006678
	6	-22.93333082	15.4499958
	7	-22.8252111	15.32554331
	8	-22.82496517	15.41903374
	9	-22.80253449	15.41892416
	10	-22.80248000	15.29736824
	11	-22.79460073	15.29709610
	12	-22.79453151	15.28736164
	13	-22.77647406	15.28736508
	14	-22.77660623	15.25061415
	15	-22.75034518	15.16668166

^ Latitude and Longitude are in Bessel 1841 Spheroid.

Figure 4-3

Etango Project – EPL 3345 Tenement Outline



4.6 Agreements and Royalties

4.6.1 Third Parties

Bannerman owns 80% of Bannerman Namibia, which in turn holds 100% of both EPL 3345 and EPL 3346. The remaining 20% is owned by another party (see Section 4.5.1).

There are no other land holders over the proposed mine site, and as such no land access agreements are required. However, there are privately owned farms elsewhere within the area of EPL 3345.

4.6.2 Sole Funding and Vendor Royalty

In accordance with the terms of the Share Sale Agreement dated May 2005 governing the relationship between Bannerman, Bannerman Namibia and the 20% shareholder of Bannerman Namibia (refer Section 4.5.1), Bannerman is required to sole fund Bannerman Namibia until completion of a bankable feasibility study. Upon cessation of the sole funding period, the 20% shareholder may elect to contribute to Bannerman Namibia's costs or otherwise dilute in accordance with a pre-set formula. Upon the 20% shareholder's holding in Bannerman Namibia falling below 5%, the shareholding immediately reduces to nil and effectively converts into a 2% royalty on the net revenue of total production from the relevant project.

4.6.3 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:

a) *determined in accordance with any term and condition, if any, of the licence of the holder concerned; or*

b) *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.

4.6.4 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 March 2012, Bannerman announced signing of a binding Term Sheet with Epangelo (or its nominee) to acquire an initial 5% interest and, upon a mine development decision, a further of 5% interest in Bannerman’s Namibian subsidiary. The Epangelo equity would come pro rata from Bannerman Namibia and the private equity owner, and Epangelo would be required to fund its purchase and ongoing expenditure.

Epangelo has 4 months in which to complete its due diligence into the Project and obtain the necessary acquisition finance (approximately A$3.9M).

The agreement with Epangelo reflects the constructive relationship between Bannerman, Epangelo and the Government of Namibia in relation to the future development of the Etango Project. Bannerman will work actively with Epangelo over the coming months to finalise Epangelo’s initial investment and to pursue the next steps for advancing the Etango Project.

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; 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

In 2009, Bannerman lodged an ESIA for development of a smaller version of the Etango Project with the Namibian Ministry of Environment and Tourism (MET). The ESIA was conducted and reviewed by independent environmental consultants, in accordance with the Environmental Protection Act of Namibia. Formal environmental clearance for development of the Etango Project as described in the ESIA was received in April 2010.

An updated ESIA, based on the enlarged mining and heap leaching operation, was submitted in April 2012.

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.

The Minerals Act requires the submission of a Mining Licence application to be supported by an ESIA, including completion of an ESMP to manage the adverse impacts identified, as well as a feasibility study. An environmental clearance has already been received for an earlier version of the Etango project and a rejection of the application submitted in April 2012, whilst possible, is considered unlikely.

The Act does not stipulate a timeframe within which the Ministry for Mines and Energy needs to process the application. However, the recent permitting of two projects in the Erongo region has occurred within a 3 to 12 month period.

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 (Namibia) Pty Ltd (Reptile). A letter of 'in principle agreement' to allow construct of the road across this land has been received from Reptile, while an allowance has been included in the capital cost estimate for sterilisation drilling.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Project Access

The Etango Project is located approximately 41km (by road) 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.

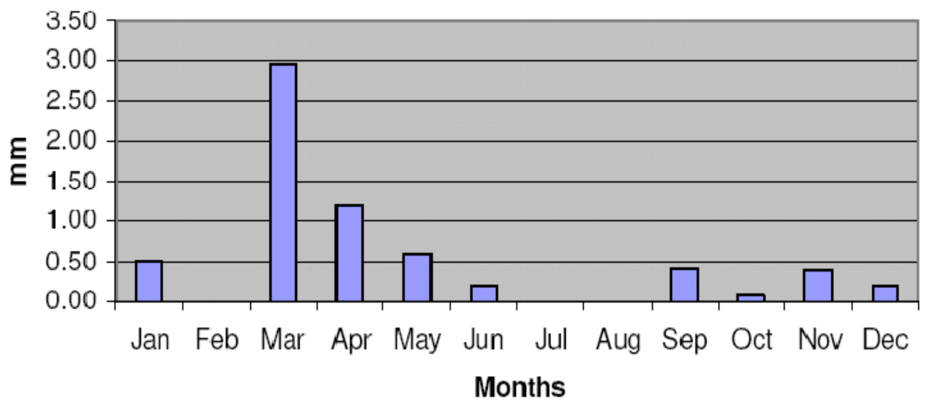
The area of the Etango deposit is generally flat (Figure 5-1) with occasional low undulating hills with sparse outcropping bedrock. Remnant shallow drainage channels are present in the Project area. The region to the north of the deposit, around the Swakop River, is characterised by deep gully erosion 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 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 Monthly Rainfall (1996-2005)

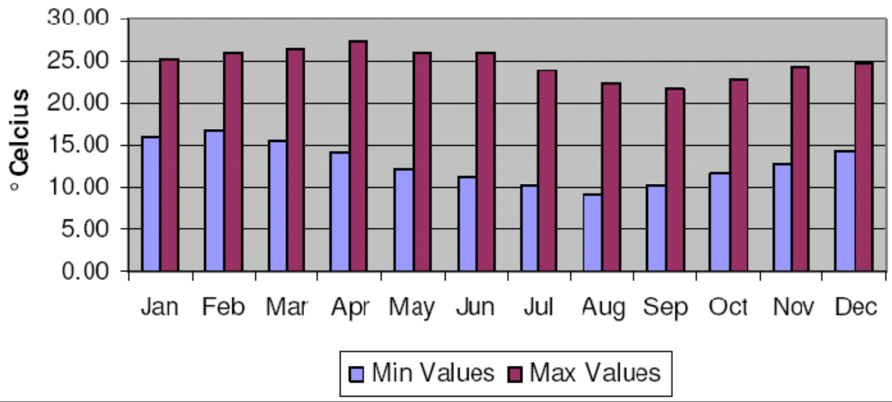


(Speiser, 2006)

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.

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.

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.

The national water utility, NamWater, has discussed plans with several mining companies to install a 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.

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 and 3346. 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 (now comprising the Etango deposit) 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 10.

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 and then March 2010 (*Inwood, 2010*). These estimates have now been superseded by the current (October 2010) resource estimation study, which included the Ondjamba and Hyena satellite deposits.

Since June 2007, metallurgical testwork 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 Rössing flowsheet (2007).
- 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 haul trucks, 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%.

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.

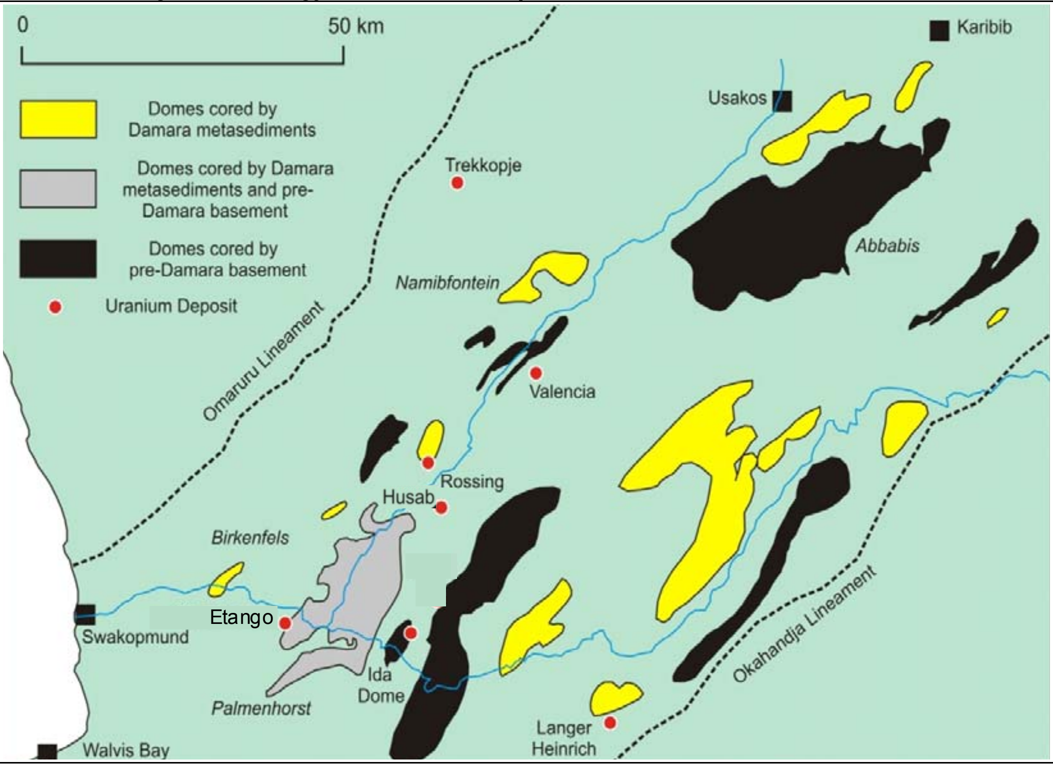
During the Damara Orogenic event, the metasedimentary cover was subjected to multiple phases of deformation, resulting in overturning of the succession and development of a prominent gneissosity and lineation which is generally sub-concordant with original bedding. This gneissosity was further deformed, with 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 is post-alaskite intrusion, but pre-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.

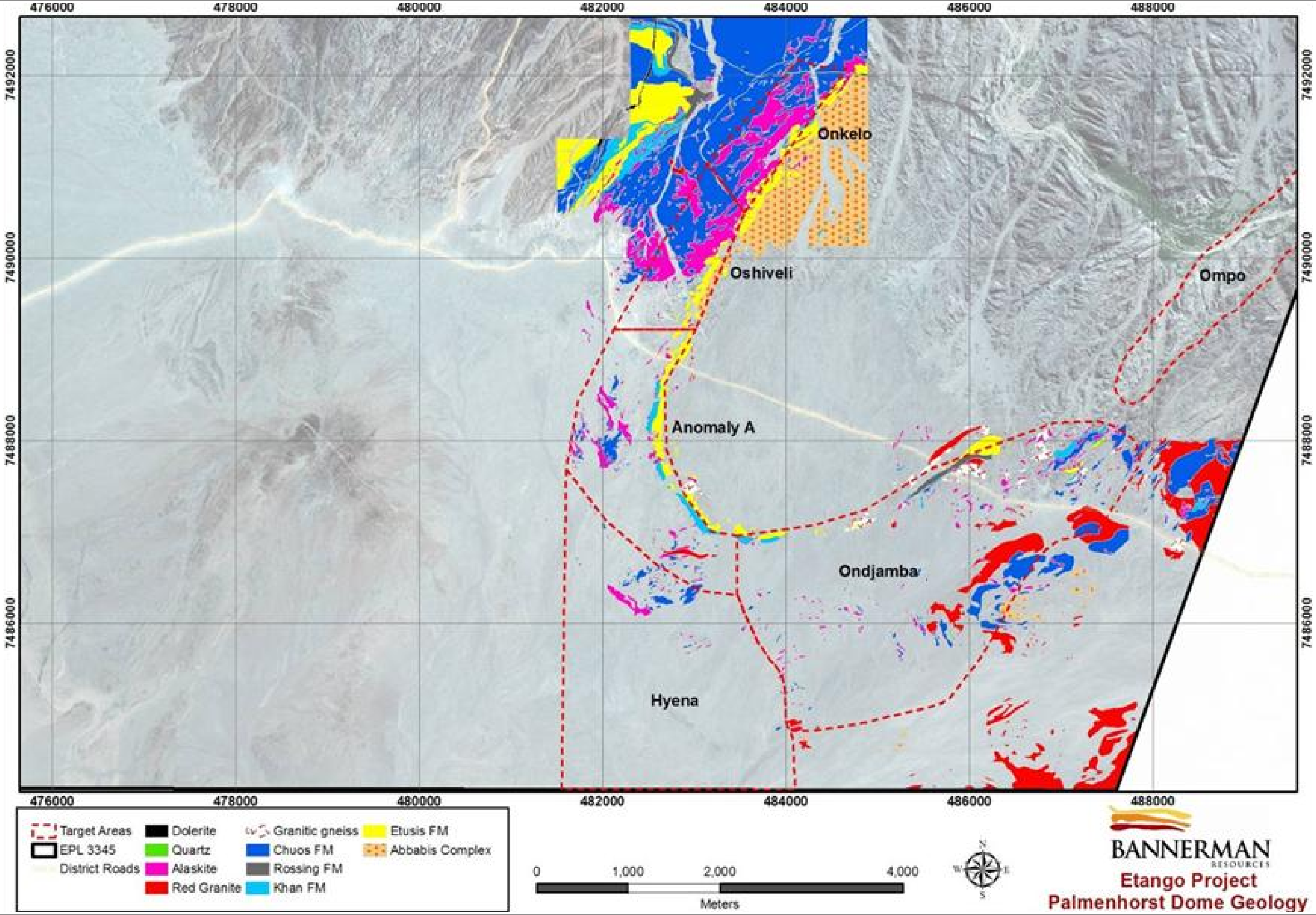
The Type D alaskites have a generally irregular and anastomosing geometry, are white to grey in colour, equigranular and contain smoky quartz, with accessory topaz.

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.

Figure 7-2

Project Geology Around the Palmenhorst Dome



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.

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.

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 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 2,100m.

Omitara also completed a total of 6,800m 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 undertook a process of capturing and digitising 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 Palmenhorst Dome. Subsequently, exploration has extended to the north from Anomaly A to the Oshiveli and Onkelo Prospects.

9.2.2 Drilling

As of 30 June 2011, Bannerman had completed a total of 1,240 RC, 141 diamond and 21 RAB drill holes for a total of over 303,500m, 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 23-480m 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 30-45º 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 testwork.

Since the previous Technical Report, a further eight RC drill holes have been completed for exploration purposes at Onkelo. These have not been included in the resource estimate as they are not currently considered material. Total additional metres drilled are 1,614m and the majority have been drilled to the southeast with one hole drilled vertically.

Twenty-eight drill holes were completed in HQ core diameter (63.5mm) for metallurgical testwork; 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 testwork.

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.

Figure 9-1

Drilling Completed at the Etango Project for the October 2010 Resource Estimate

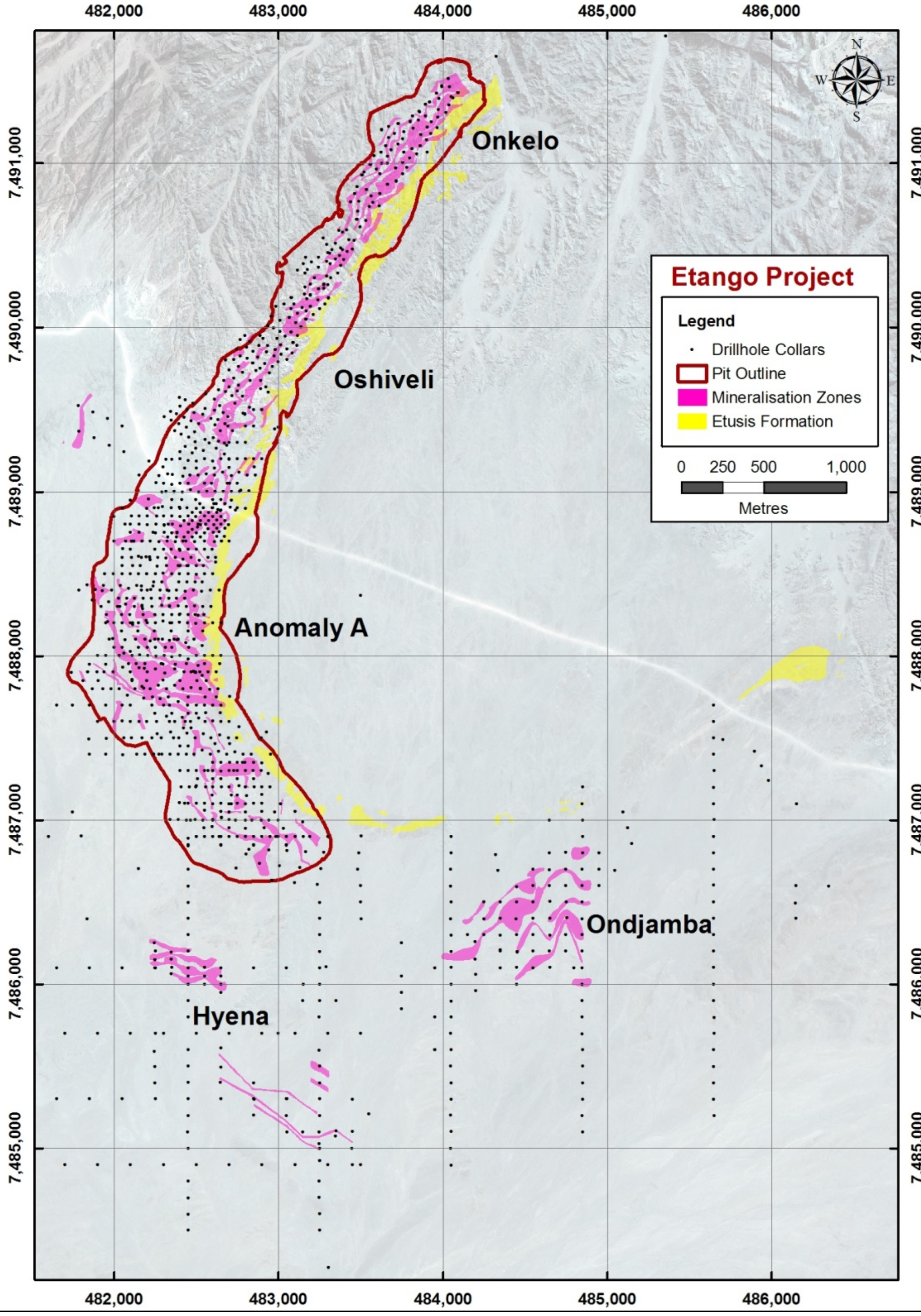
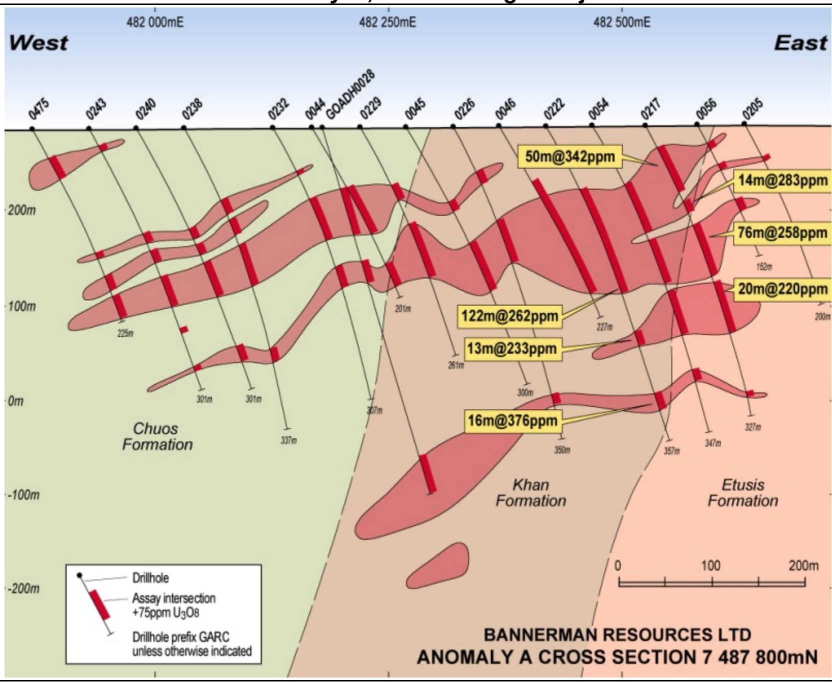


Table 9-1 Drilling by Bannerman in the Etango Project Area, up to 30 June 2011		
Drill Type	**Number**	**Metres**
RC Anomaly A	582	145,287
RC Oshiveli	152	40,069
RC Onkelo	92	18,983
RC Ondjamba	182	30,536
RC Hyena	112	18,292
RC Other	120	10,723
RC Total	**1,240**	**263,890**
DD Resource	84	26,079
DD Geotechnical	29	7,079
DD Metallurgy	28	4,857
DD Total	**141**	**38,015**
RAB Total	**21**	**1,875**
Grand Total	**1,402**	**303,780**

Figure 9-2
Typical Cross-section Through the Mineralisation at Anomaly A, at the Etango Project



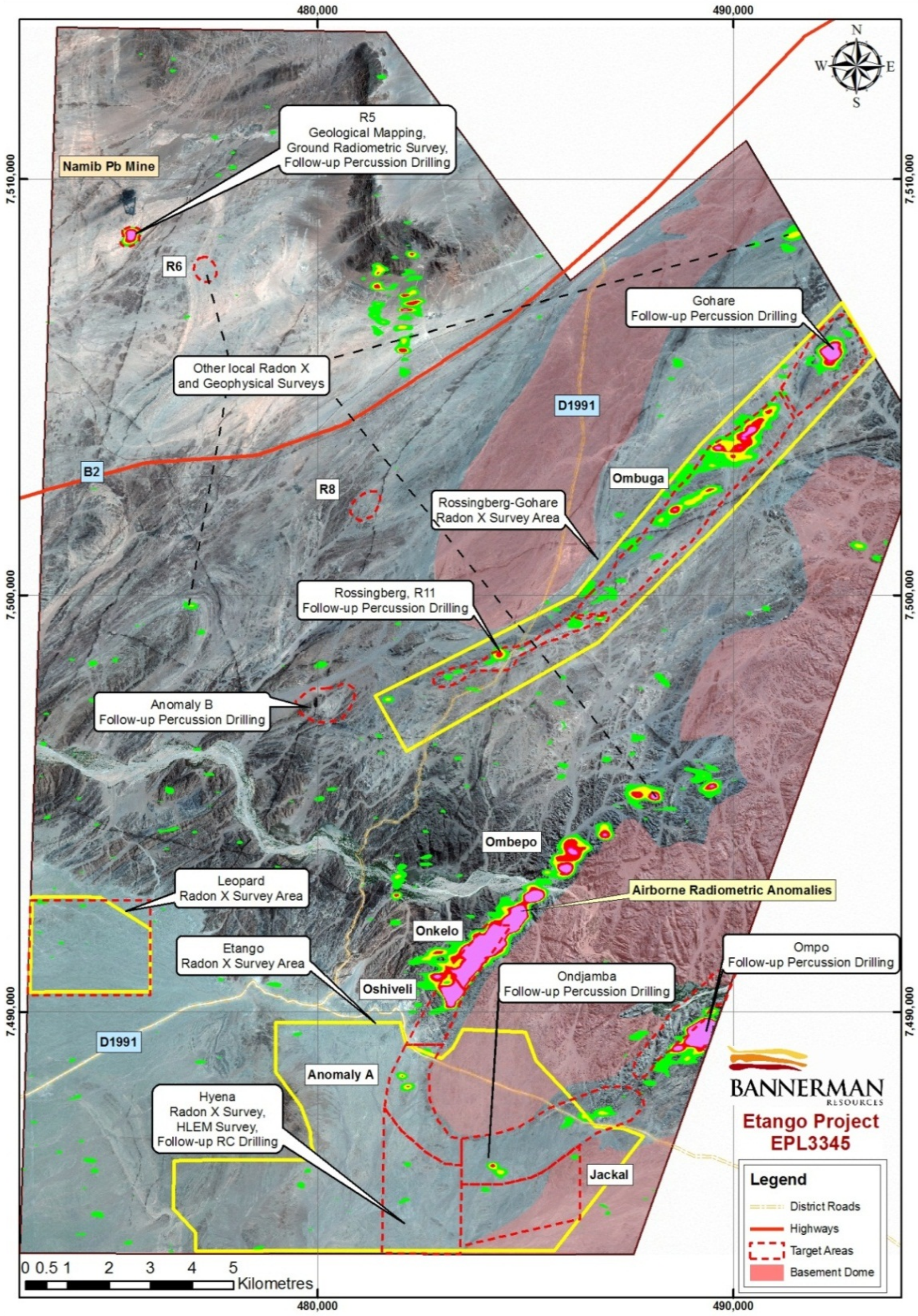
9.2.3 Ongoing Exploration

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 and where Bannerman is continuing its exploration activities. The western flank of the Palmenhorst Dome, which incorporates the Anomaly A, Oshiveli and Onkelo deposits, constitutes a prospective strike length of over 20km.

In 2010 and 2011, exploration has continued at the Etango Project and elsewhere within the Etango (EPL 3345) and Swakop River (EPL 3346) licences in Namibia. Further exploration is planned in 2012 and into the future. Figure 9-3 shows the details of some of the recent work in the Etango licence in 2010 and 2011.

Figure 9-3

Exploration at the Etango Licence (2010)

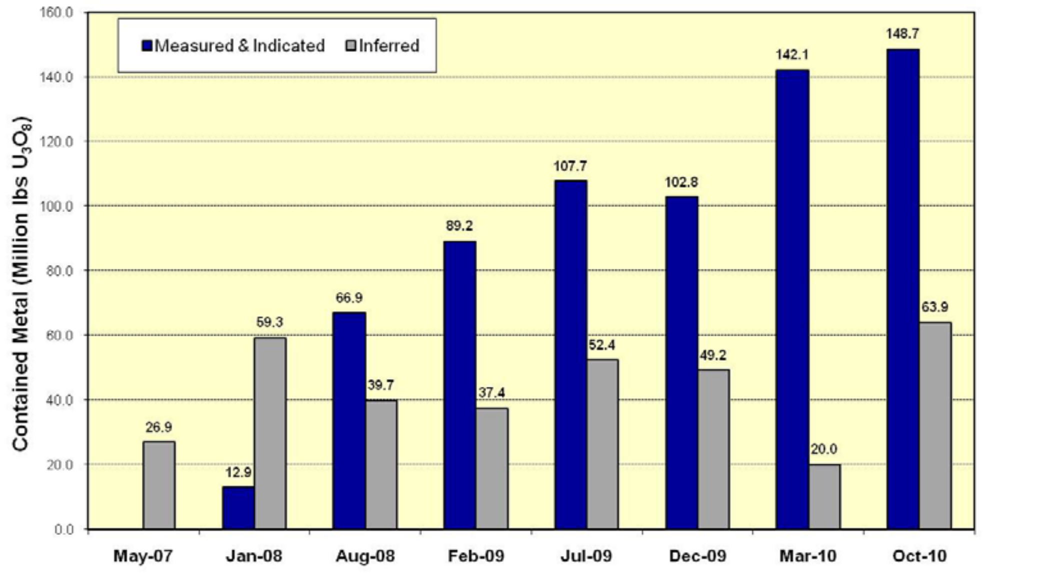


9.2.4 Previous Mineral Resource Estimates

In May 2007, an Inferred Mineral Resource was estimated by Bannerman for the Anomaly A deposit, based on the historical and recent drilling. Bannerman has continued an aggressive drilling program over the resource area up to the present time, and exploration work continues in the area. All of these drilling and exploration works are supervised by Bannerman staff geologists.

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, March 2010 and, most recently, in October 2010, Figure 9-4 (see also Section 14).

Figure 9-4
Growth of Etango Mineral Resources with Time
(Reported at a Cut-off Grade of 100 ppm U_3O_8)



10 DRILLING

10.1 Drilling by Previous Owners

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 mineral resource work. 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 sample all intersected alaskite intervals and a few metres of metasediment on either side. The location of the sampling for the resource studies is shown in Figure 9-1.

10.2 Drilling by Bannerman

As of 30 June 2011, Bannerman had drilled a total of 1240 RC, 141 diamond and 21 RAB drill holes for a total of over 303,500m in the vicinity of the Etango Project. The RC drill holes range from 23m to 480m 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 whilst modelling for these holes which were not assayed. The RC drill holes were drilled by Metzger Drilling using a bit diameter of 4.72" to 5.5". The bulk of the RC drilling has been designed 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-mineable 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 was drilled using NQ diameter core barrels (47.6mm core). Twenty-nine holes were drilled using a NQ3 core barrel (45.1mm core) for geotechnical purposes. All geotechnical samples were sent to Rocklab in Johannesburg for testwork. The majority of the core is orientated by spearing after each run. Ten drill holes were completed in HQ core diameter (63.5mm core) for metallurgical testwork; the entire HQ core was sent to Ammtec Laboratories in Perth.

Due to the shallow dip (approximately 30°-44° 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 Rössingberg, 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.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sampling Method and Approach

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.

- Samples are 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.
- 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 and, since 12 January 2009, to SGS Johannesburg for assaying.
- Since December 2008, samples have been 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.
- 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 is 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 for 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.

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 done on whole diamond core samples of 10 cm 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, 5,889 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 (Table 11-1).

Analysis of the results indicates that there is no significant change in density with depth, apart from a small reduction in 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 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.

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, 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 1 hole in 5 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 be 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 mineral resource area are competent with very few cavities. Based on the results of the investigation Bannerman determined that a 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.

11.2 Sample Preparation and Analysis

11.2.1 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 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 1 sample in 20.
- 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.

11.2.2 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 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.

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 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.

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 has 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 Survey

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* is a simple 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 used.

- *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.

- 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 will discuss the analysis of both the Bannerman and laboratory inserted standards.

<u>Bannerman Submitted Standards</u>

Bannerman has routinely inserted blanks and certified standards into their 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.

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, 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 Submitted 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**
Standard 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=1,632) exhibits some minor contamination throughout the sample runs and possible incorrect sample identification / submission, with eleven samples reporting above 10ppm U. The laboratory blank (n=6,877) 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.

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 Submitted Standards (U ppm)

Standard	XRF – Genalysis Perth							
	BL-1	**SARM1**	**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**
Standard 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.

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

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 reports 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 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%).

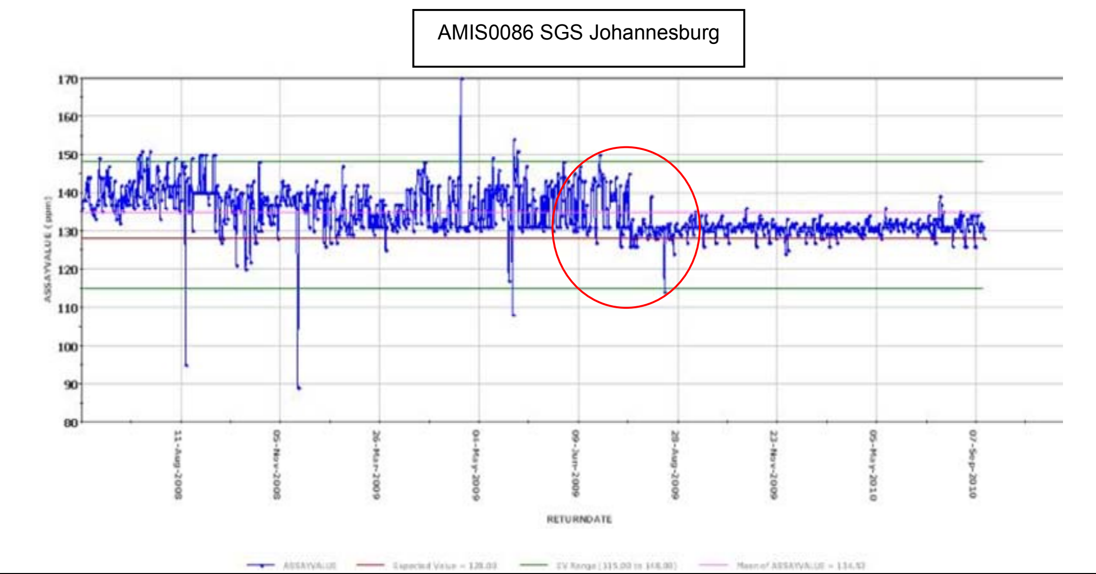
Discussion

Analyses of the Bannerman standards indicate 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 duplicates 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 (Table 14-5).

The trend of the bias seen at SGS Johannesburg is of minor concern. However, this is tempered with 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 Figure 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 (Figure 12-2). Once tagged, the bags were sent to Bannerman's sample storage yard for processing.

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 U_3O_8 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
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 estimations.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Significant testwork 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 testwork 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}=5.3µm), 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 SX characteristics.

13.2 Sample Description

Samples were provided as whole HQ[4], ½ NQ2 and ¼ NQ core. NQ core was retained for planned variability testing to follow the current program of testwork.

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

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.

[4] HQ = 96mm drill hole; NQ = 75.7mm drill hole

13.2.1 Ore Types

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

Metallurgical testing proceeded on the basis of selecting intervals of core above a cut-off grade of 100ppm U_3O_8. Typically, sediments 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 metallurgical testwork considering the two main alaskite lithologies found little or no significance difference between them 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 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 close to 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, likely due to wind transport.

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, assays represent generally low levels. The iron assay represents the amount of natural total iron in the ore. Ferric iron is required to promote oxidative.

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	%	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 from the following intervals in drill hole GOADH0048: 29-34m, 48-52m, 54-58m, 74-77m and 90-94m.

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.

Uraninite occurred typically as: 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 basal cleavage planes of phyllosilicate minerals, biotite and chlorite; with numerous sub-20µm strips within the core of biotite; 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.

Liberation Analysis

QEMSCAN liberation class data is provided in Table 13-6, split into a separate analysis for each of the two dominant groups, uraninite and uranium silicates and also for all uranium mineral phases identified.

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
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**

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**	**Middlings**	**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 represent the ore along and across the resource and at depth. The following core provided samples for the comminution testwork:

- Preliminary characterisation based on selected intervals from GOADH0048

- Variability testing utilised intervals derived from GOADH0048, GOADH0058, GOADH0059 and GOADH0060
- HPGR Pilot testwork 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 Testwork Samples – Composition and Source Location

Hole ID	Prospect	Approximate Location in Etango Orebody	Final 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	17/7/2008
GOADH0063	ANOMALY_A	Northern End	165.26	22/7/2008
GOADH0064	ANOMALY_A	Central-west area	84	24/7/2008
GOADH0065	ANOMALY_A	Northern End	213.7	4/8/2008
GOADH0066	ANOMALY_A	Northern End	198.29	6/8/2008

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
- DWi – JK proprietary impact breakage test
- SMC – JK proprietary impact breakage test
- 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 drill hole GOADH0048 was used in the preliminary characterisation; composites of 5m to 6m were prepared.

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 DWi

A single 6m composite was prepared across all intervals and subjected to a full JK Drop Weight test.

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

Specific energy values and crusher appearance function data are used to develop comminution design criteria. These data are presented in Table 13-9 and Table 13-10. The data indicates that for particles less than 63mm, there is a trend to increasing impact breakage resistance with decreasing size.

Table 13-9
Crusher Model Appearance Function Data

Percentage Passing t_{10}	PSD of ore ground to 10, 20 and 30% passing 1/10th Original Size – as Cumulative % Passing				
	Passing Size (Relative to Initial Size ie t_{75}=1/75th orig. size; t_2= ½ original size)				
	t_{75}	t_{50}	t_{25}	t_4	t_2
10%	3.01	3.85	5.77	19.52	48.97
20%	5.52	7.19	11.16	40.10	79.43
30%	7.96	10.46	16.50	59.87	95.54

Table 13-10
Specific Comminution Energy

Percentage Passing t_{10} (1/10th orig. size)	Feed Size Fraction (mm)				
	-16/+13.2mm	-22.4/+19.0mm	-31.5/+26.5mm	-45.0/+37.5mm	-63.0/+53.0mm
	Specific Energy E_{cs} (kWh/t)				
10%	0.17	0.16	0.16	0.13	0.10
20%	0.37	0.36	0.34	0.29	0.22
30%	0.58	0.57	0.56	0.48	0.40

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.

UCS

A total of 20 UCS tests were performed. UCS values for separate Type D and Type E alaskite samples exhibit a relatively wide variation, while low variability exists between the two main lithology types.

Maximum UCS values of 99.7Mpa for Type D and 119.0Mpa for Type E alaskite are typically classified as low to moderate and indicate no issues with standard crushing preparation.

Bond Crushing Index

A total of 18 Bond Crushing Work Index determinations were performed, each determination being based on 20 specimens from each selected 5m interval. Note that individual particle results are not considered when analysing CWI results. A single test result is the average of the outcomes for the 20 particles. Due to the stepwise nature of the test and the irregular particle shapes, there is a high and unrepresentative scatter in individual particle breakage outcomes.

As for UCS test data, crushing index values for separate Type D and Type E samples indicate low variability between the two main lithology types and are consistent at depth.

Average values of 8.0kWh/t and 8.3kWh/t for Type D and Type E respectively confirm both ores are relatively soft. Maximum values 110.3kWh/t for Type D and 10.7kWh/t for Type E are also typical of soft ores and confirm that crusher design will not be limited by crushing power requirements.

The Ai value is used to calculate the wear rate of crusher liners in the Etango heap leach circuit.

Bond Abrasion Index

A total of 18 Bond Abrasion Index (Ai) determinations were performed utilising residues from UCS testwork.

The values showed that Type D alaskite is the more abrasive, while both indicate an ore feed with moderate abrasion potential. Average Ai values are 0.336 and 0.274 for Type D and Type E, respectively.

JK SAG Mill Comminution SMC Testing

The JK SMC test is a reduced version of the full Drop Weight Index test and applied to a single size fraction, in this case nominally -31.0mm/+26.5mm. The SMC tests data was calibrated using the results of the full Drop Weight Test (reported in Section 13.4.1) and the results provide competence values for assessment of variability in SAG mill comminution behaviour.

A total of 18 SMC tests were performed. The distribution of values indicated relatively consistent SAG mill comminution performance at depth and also indicates DWi values slightly higher for Type D than Type E alaskite. Impact resistance as measured by the average A*b value for Type D alaskite of 78.1 is consistent with the full JK DWi test value on GOADH0048 Type D at 79.7.

Minimum A*b values of 63.3 for Type D and 69.8 in the case of Type E indicate a moderate to soft ore from an impact breakage perspective, and, as for the original calibration test on GOADH0048, are indicative of a relatively low resistance to impact breakage. These results are consistent with the CWI values.

The SMC results are most commonly used for SAG mill design, but for the heap leach circuit they are most relevant to cone crusher and HPGR design.

Bond Rod Mill Work Index

RWi variability data was developed based on a total of 18 tests. The distribution of RWi values appears very consistent with depth and indicates a low variability in the global RWi index for both types of alaskite. Data indicates that there is virtually no difference between the type D and Type E ores given the repeatability of the test is typically ±0.5kWh/t.

Average values of 12.1kWh/t for Type D and 12.3kWh/t in the case of Type E indicate that breakage in the 12mm to 1 mm range (relevant to HPGR operation) requires low to moderate energy.

Bond Ball Mill Work Index

A total of 18 BWi tests were performed. The distribution of values indicated relatively low variability in global ball mill comminution behaviour as well as a low variability between Type D and Type E Alaskite.

Average values of 14.5kWh/t and 14.8kWh/t for Types D and E respectively classify the ore as moderately hard from a fine grinding perspective, with little variation at depth.

No fine grinding is occurring in the heap leach circuit but the BWI value is included as it is a common comminution reference.

13.4.3 High Pressure Grinding Rolls Pilot Testwork

Initial testwork demonstrated that the ore exhibits a high degree of liberation 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 from the variability testwork indicated that the ore would likely respond well to HPGR comminution. On this basis, pilot HPGR pilot testwork was undertaken. Polysius equipment was used, based on the expected higher specific throughput available via Polysius studded rolls design compared to smoother roll surface alternatives.

Samples

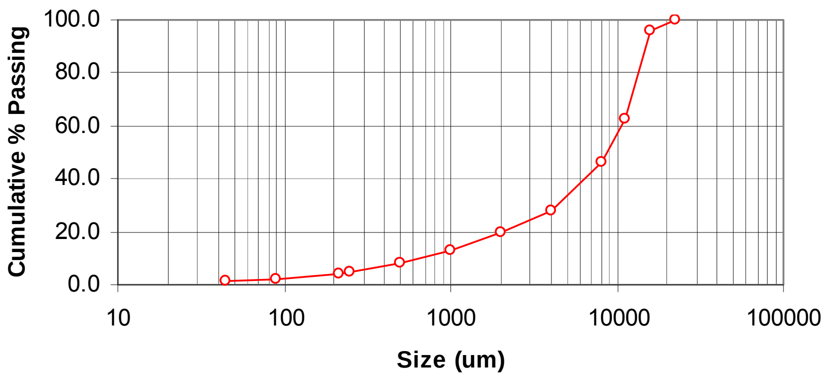
A total of 186m of whole HQ core was used to prepare a 1500kg master composite comprising 124m of Type D and 62m of Type E. Details of the master composite are summarised in Table 13-11.

Sample Preparation

The master composite was prepared by control crushing, using jaw and cone crushers, 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 test unit.

Table 13-11 HPGR Pilot Testwork 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**

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 13-12. These tests were conducted to investigate the effect of two specific pressure settings, roll speeds and moisture levels.

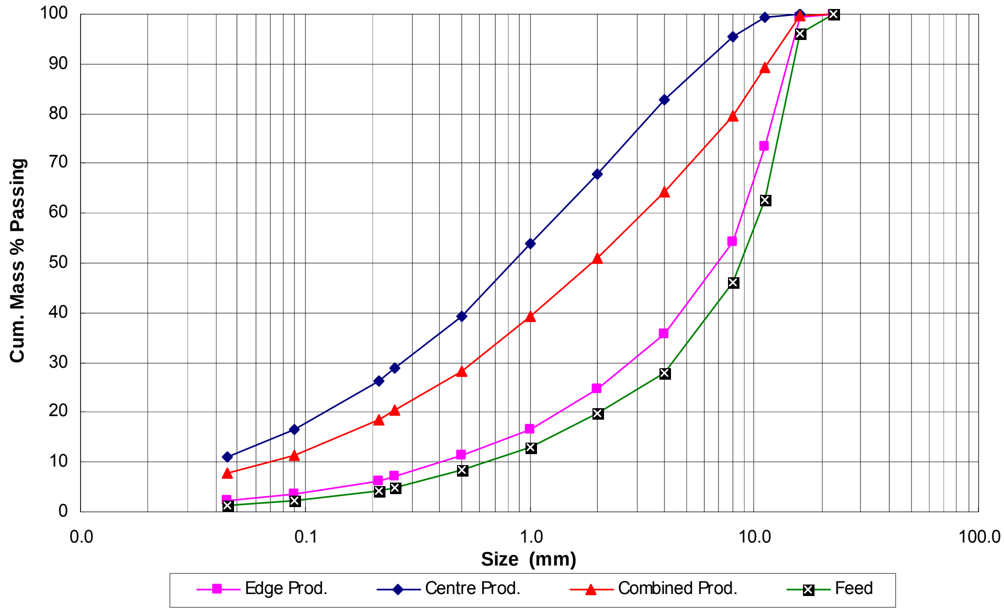
Table 13-12 HPGR Open Circuit Test Parameters				
Test No.	**Pressure Setting (bar)**	**Specific Pressure (N/mm^2)**	**Roll Speed (m/s)**	**Moisture (%)**
1	55	2.90	0.2	6.0
2	40	2.10	0.2	6.0
3	55	2.98	0.4	6.0
4	55	2.99	0.2	3.0

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 (especially the need for disagglomeration) will not influence the design.

The specific throughput rates (250ts/hm3) and specific energy (1.2kWh/t) 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 from Test 4 are shown in Figure 13-2.

Figure 13-2

HPGR Open Circuit Test 4 Product Particle Size Distribution



HPGR Closed Circuit Trial

Closed circuit HPGR operation was used to prepare a sample for heap leach investigations. 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 13-13.

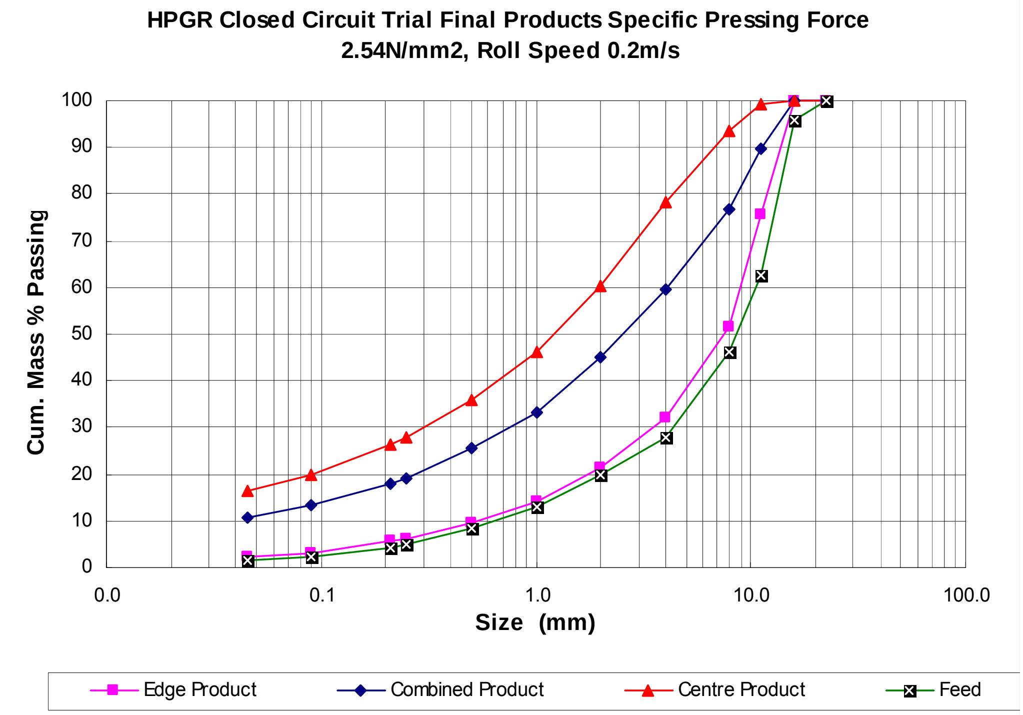
Closed circuit testing was conducted by screening at 8mm and recycling oversize product at the end of each cycle. Stable conditions were achieved after four test cycles with a roll speed of 0.2m/s and specific pressing force of 2.54N/mm^2, resulting in a final closed circuit centre product P_{80} of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

HPGR product size distributions are shown in Figure 13-3 for final products generated at the fourth test cycle.

Table 13-13
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
-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



13.4.4 Comminution – Conclusions

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

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.

13.6 Heap Leach Testing

Column leach testing of Etango ores has been ongoing 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 Magnaflocc 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 (FA) 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 testwork 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 Testwork Program

A number of additional heap leach programs have been completed at ALS Ammtec, and further testing is underway at Bureau Veritas in Swakupmond. 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 (Table 13-14 and Table 13-15) were initiated at ALS Ammtec using sub-samples of available composite ore that was crushed to 100% - 8mm via HPGR.

Table 13-14 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 13-15 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 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 testwork 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 testwork 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 testwork at Bureau Veritas in Swakupmond is in progress with results available only from Run 2, which was as follows:

Run No. 2

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

- Open circuit
- 250kg/t of Magnaflocc 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

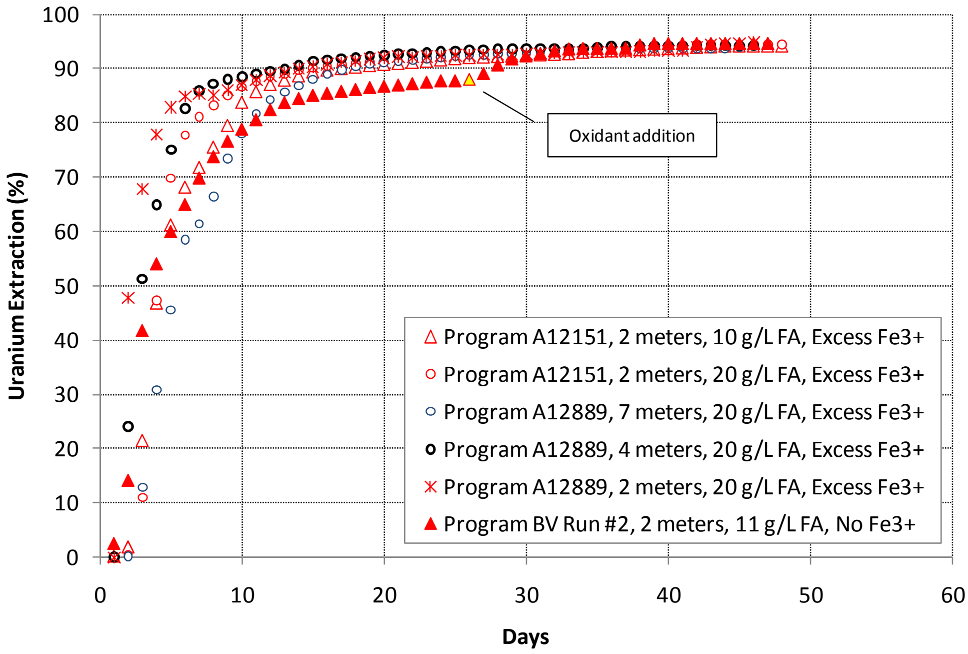
The variability testwork currently ongoing at Bureau Veritas will provide important column testwork data from a range of samples made up to represent the first three years of operation. However, at this time, there are a limited number of column tests that represent realistic conditions for the operating heap and can therefore be used to estimate the operational performance. These tests are presented in Table 13-16, and were undertaken on sub-samples of the same composite under standard conditions, without recycling of liquor.

The uranium extraction curves for each of these tests are presented in Figure 13-4.

Table 13-16
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 No. 2	Column 1	2	11	No	Shorter than design, low FA

Figure 13-4
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 13-5. 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 13-17 presents the associated leach time and calculated acid consumption for specific uranium recovery points.

Figure 13-5

Acid Efficiency Curves for Columns Relevant to Operational Heap

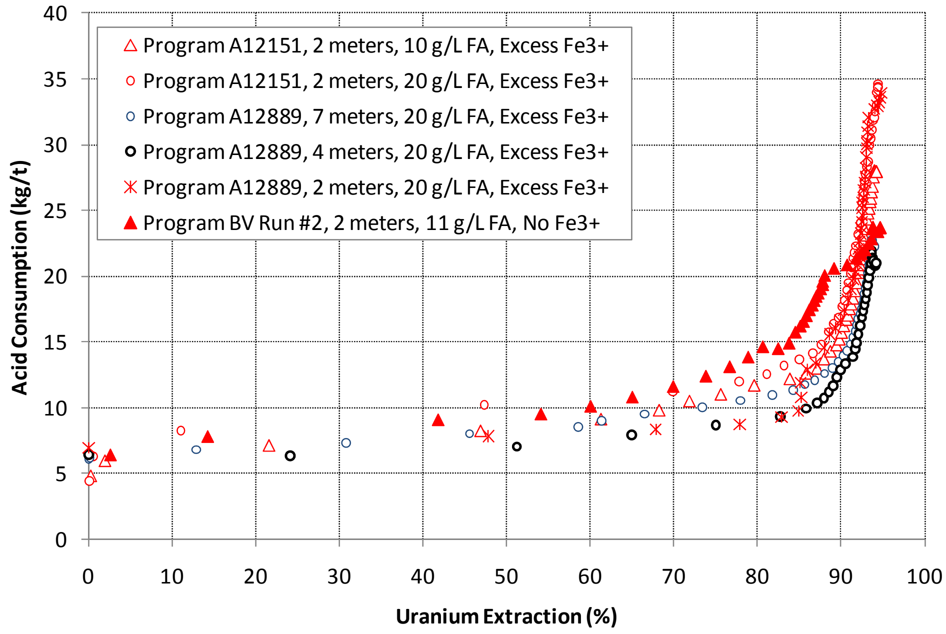


Table 13-17

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

Program	Column ID	Height (m)	90% U Extraction Point Leach Time (days)	90% U Extraction Point Acid Cons. (kg/t)	92% U Extraction Point Leach Time (days)	92% U Extraction Point 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 13-17 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 as the column tests that were reported in Table 13-16. 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.

The agitated leach results presented in Figure 13-6 demonstrate marginally higher uranium extractions, while kinetics are significantly faster, as expected. Table 13-18 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.

Figure 13-6

Comparable Agitated Leach Test

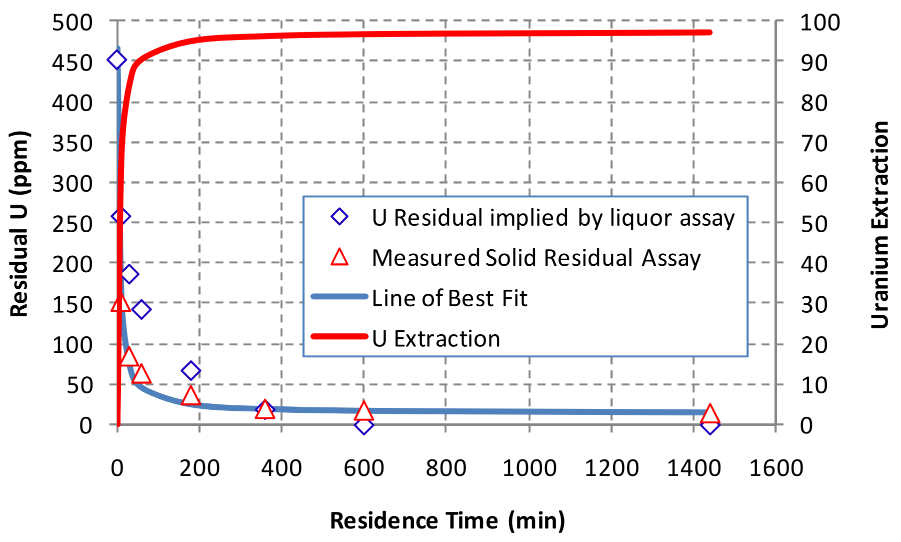


Table 13-18 shows that the column test can achieve comparable extractions to the agitated leach test, given enough leaching time and with the consumption of more reagents.

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 13-7 presents the comparative leach test into context against the other agitated leach variability tests.

Table 13-18 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.

Figure 13-7
Comparison of Comparative Leach with Variability Program Uranium Extractions

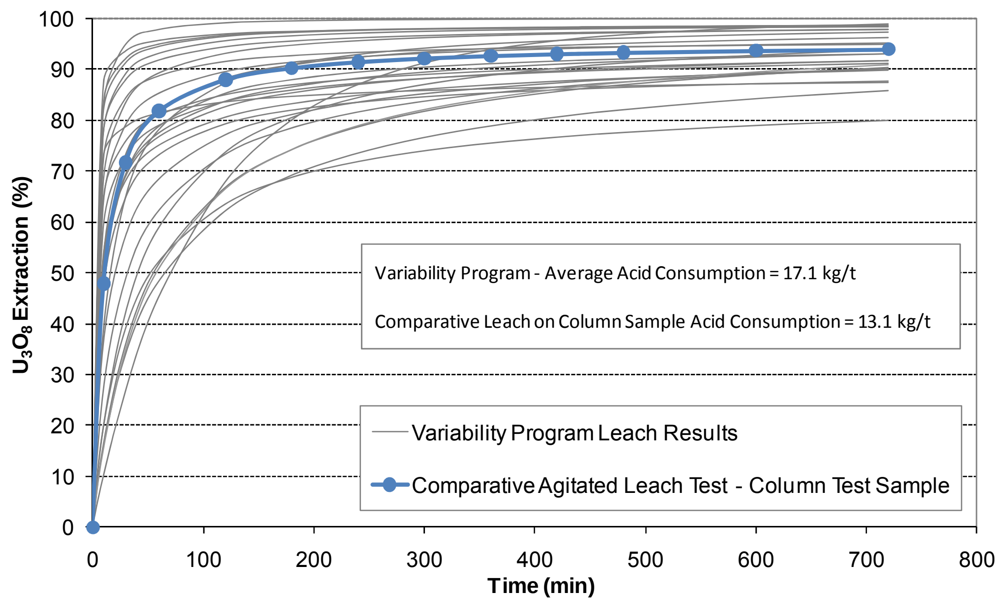


Figure 13-7 suggests that the comparative leach exhibits average uranium leach kinetics and overall extractions in comparison to the full suite of variability tests, however its acid consumption is lower than average.

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

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.

<u>Reproducibility</u>

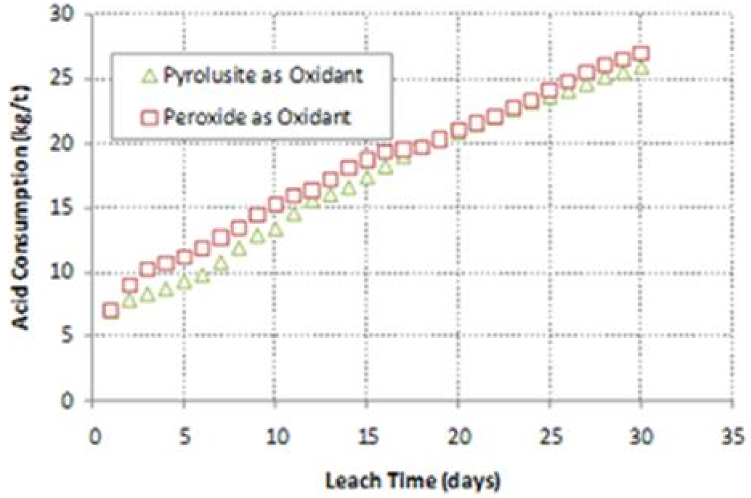
A pair of 2m columns were prepared and operated comparably in order to assess the reproducibility of column performance. The resultant acid consumption over time and acid consumption / uranium extraction relationships for both tests were closely comparable.

<u>Effect of Different Oxidant</u>

The effect of using pyrolusite and hydrogen peroxide to maintain solution Eh was compared, demonstrating comparable rates of acid consumption. Figure 13-8 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.

Figure 13-8

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 test 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.

<u>Effect of Liquor Recirculation</u>

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.

<u>Effect of Column Height</u>

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

Figure 13-9 and Figure 13-10 present the resultant acid consumption rate and uranium extraction rate curves for each of these tests.

Figure 13-9

Acid Consumption Rate for Different Column Heights

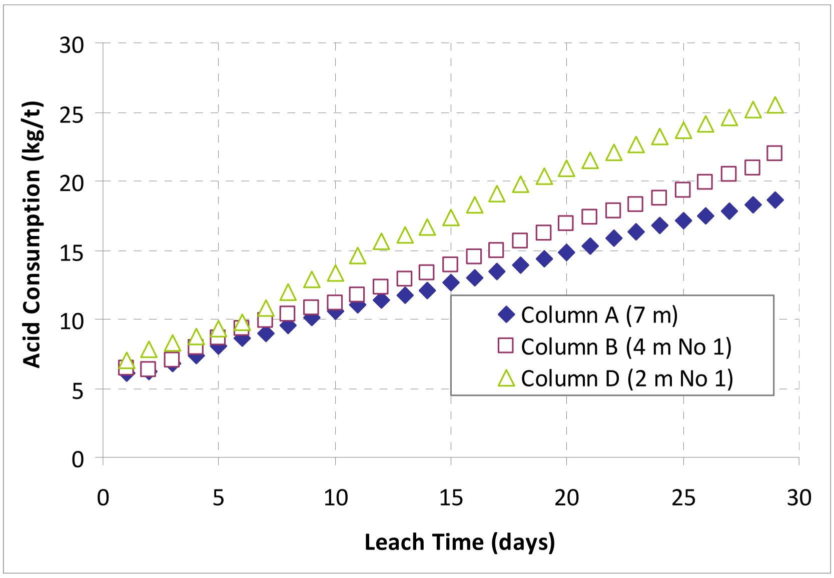
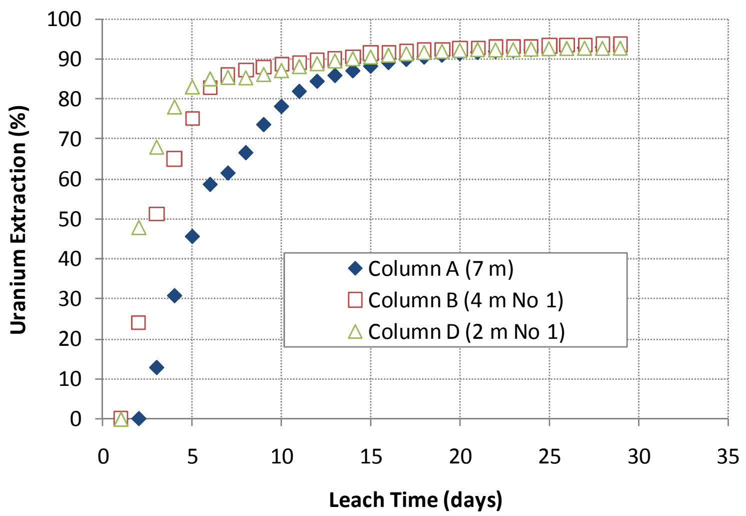


Figure 13-10

Uranium Extraction Rate for Different Column Heights

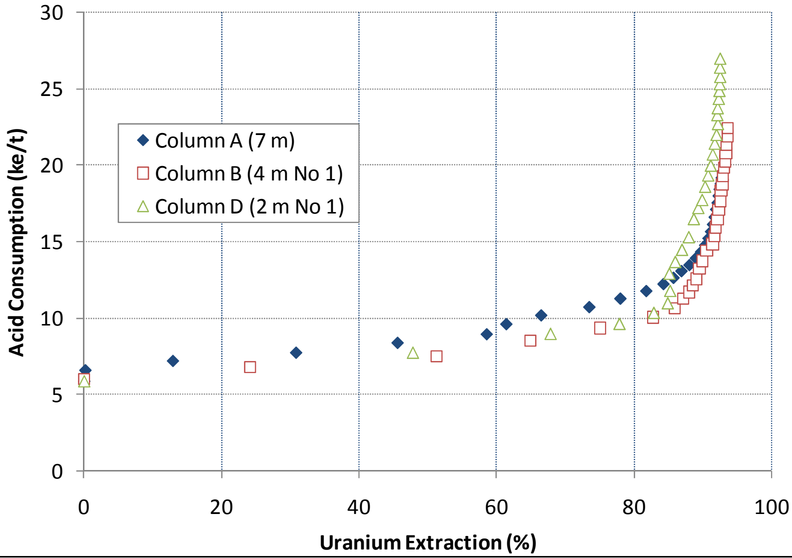


As expected, the rate of uranium extraction and the rate of acid consumption (kg/t) both decreased as the column height increased. However, it is notable that the final uranium extraction achieved is very consistent across the tests. This result demonstrates that the hydrodynamics (permeability, liquor retention) of a 7m column / heap will not limit uranium extraction.

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 13-11.

Figure 13-11

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.

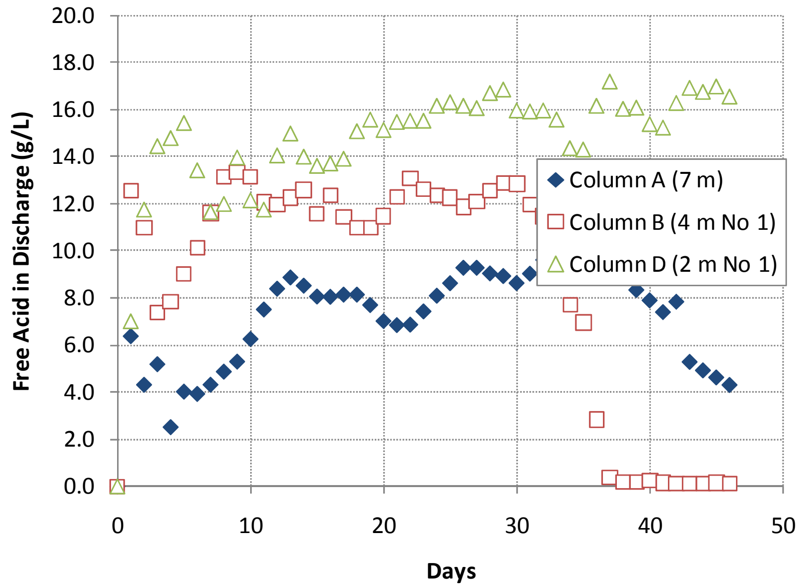
It should be noted that there were some notable operational differences with the resultant Free Acid measurement in the discharge. Figure 13-12 shows that once the 7m column achieved a relatively steady state (from Day 12 onwards) the free acid in the discharge was ~8g/L. For the 2m and 4m columns, the 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 13-10), 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 13-11 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.

Figure 13-12

Free Acid in Discharge for Different Column Heights



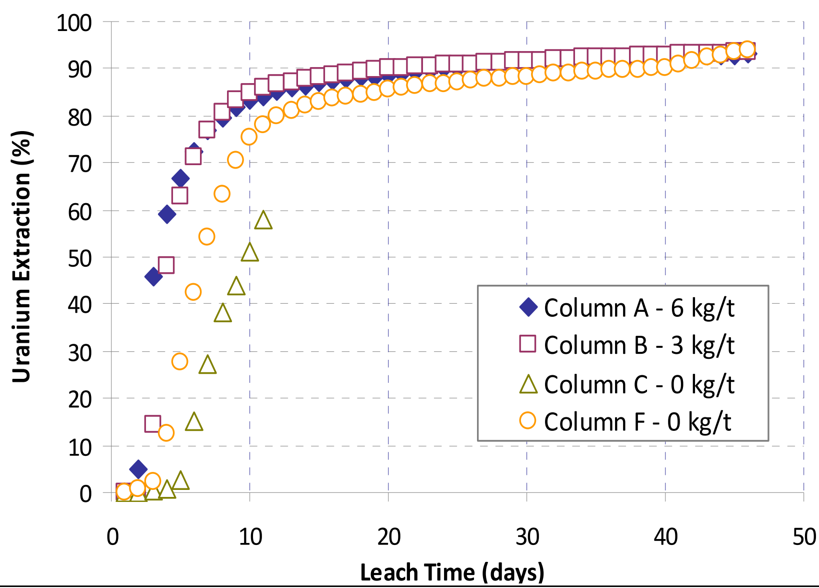
Effect of Agglomeration Acid

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 13-13 presents the uranium extraction rates for the relevant tests with varying doses of agglomeration acid.

Figure 13-13

Uranium Extraction Rate for Different Agglomeration Acid Doses



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%.

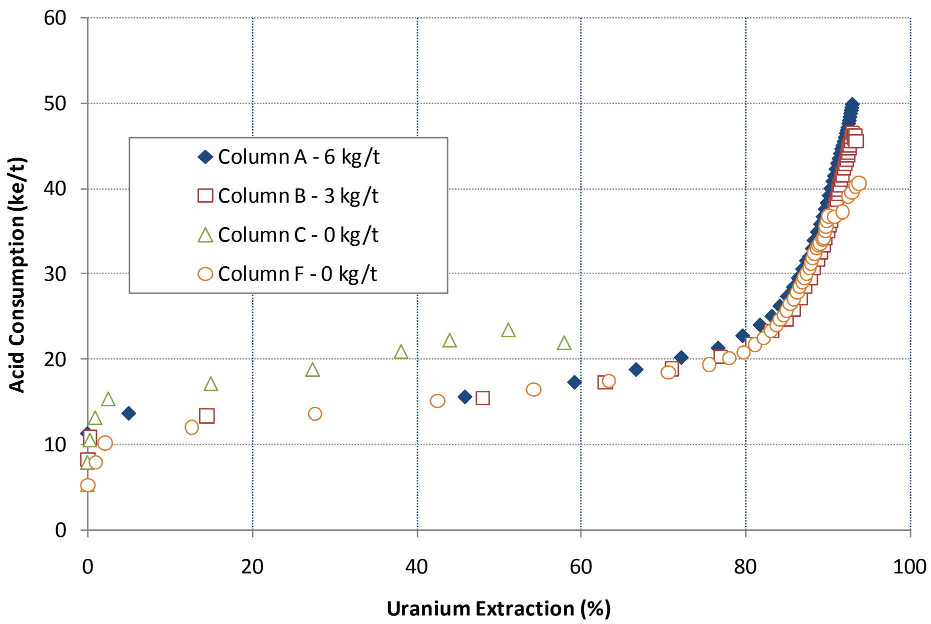
Because both the rate of acid consumption and the rate of uranium extraction are changing, the acid efficiency chart is a more demonstrative comparison of performance (Figure 13-14).

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 13-14

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

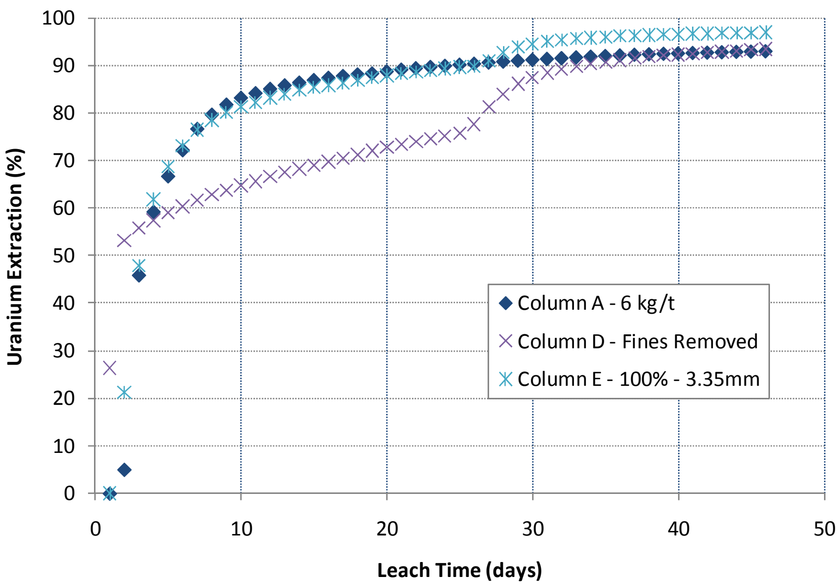
The test data did not confirm the expectation that higher acid consumption accompanies increasing fines content. The sample with fines removed (Column D) consumed the least amount of acid. However, Column E was expected to have the highest rate of acid consumption, but, unexpectedly, Column A displayed 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 13-15), 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 13-15

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.

Conclusions from Diagnostic Testwork Programs

Conclusions from the diagnostic component of the column testwork 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 testwork 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.

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 testwork 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^o). 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 testwork 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 13-19.

Table 13-19
Head Assay of SX Testwork 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, 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 testwork, 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 13-20.

Table 13-20
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 13-16 and Figure 13-17.

Figure 13-16
Uranium Loading Isotherm
(as-produced PLS at 20ºC)

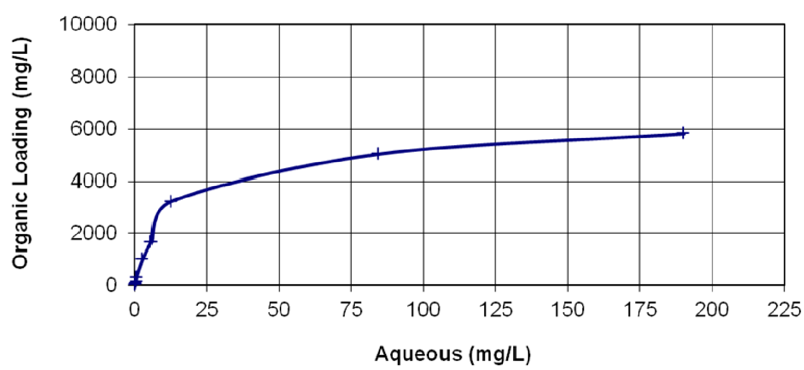
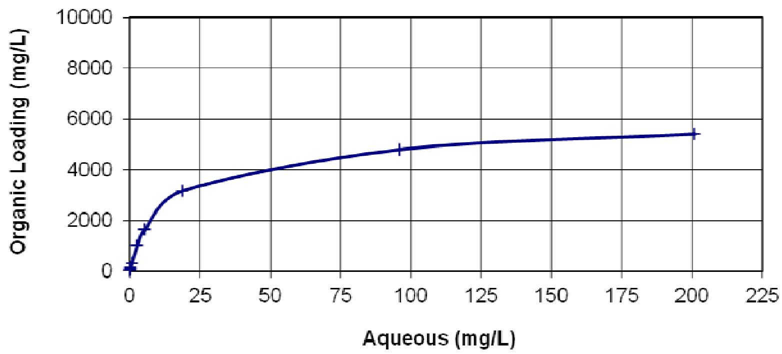


Figure 13-17
Uranium Loading Isotherm
(as-produced PLS at 35ºC)



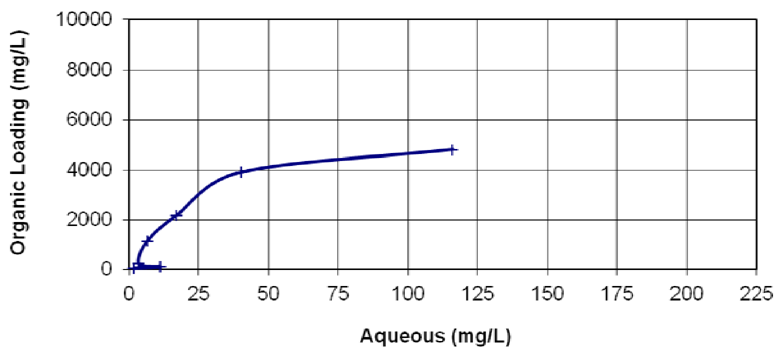
A 2L portion of the as-produced PLS was 'spiked' with salts to increase contaminants such as Al, Ca, Fe^{2+}, Fe^{3+}, K, Mg and Mn (Table 13-21).

Table 13-21
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 13-18.

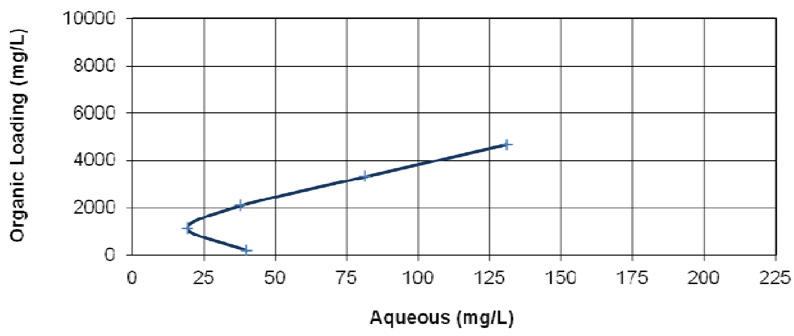
Figure 13-18
Uranium Loading Isotherm
(Spiked PLS at 20ºC)



Sodium chloride (2.48g/L Cl) was also added to the spiked PLS and isotherm testing repeated (Figure 13-19).

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 13-19
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 13-22 and Table 13-23. These indicate moderate levels at surface, with the concentration reducing significantly at depth.

Table 13-22
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 13-23
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

14 MINERAL RESOURCE ESTIMATES

14.1 Etango Project Mineral Resource

The October 2010 Resource update (Table 14-1) represented an incremental increase in the Etango Mineral Resource endowment; a previous estimate was completed in March 2010.

This estimate included the results of an additional 27 (10 diamond and 17 RC) holes to the March 2010 update, plus additional chemical assays not available for the previous update.

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 – October 2010 Resource Estimate

Classification	Lower Cut	Tonnes Above Cut-off (Mt)	U_3O_8 (ppm)	Contained U_3O_8 (t)	Contained U_3O_8 (Mlb)
Inferred	100	45.7	202	9,200	20.3
	125	40.3	214	8,600	19.0
	150	34.7	226	7,800	17.3
Indicated	100	273.5	200	54,600	120.4
	125	238.6	212	50,700	111.7
	150	193.7	230	44,500	98.1
Measured	100	62.7	205	12,900	28.3
	125	56.6	215	12,200	26.8
	150	47.5	230	10,900	24.0

Note: Figures have been rounded. Conversion of lbs to kg = x 2.20462
OK Model Reported at Various Cut offs Using a Bulk Density of $2.64t/m^3$
Panel dimensions of 25m N by 25m E by 10m RL

14.1.1 Introduction

In August 2010, Coffey Mining was asked to undertake a Resource update of the Etango Project. This section details the steps taken in preparing the October 2010 OK estimate.

This update follows on from the March 2010 resource update which was also undertaken by Coffey Mining.

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 October 2010, Coffey Mining completed a resource estimate for the Etango Project (comprising the Anomaly A, Oshiveli and Onkelo prospects). Resource estimates have previously been completed in 2008, 2009, and March 2010; and this work has now again been updated. OK was used as the method for estimating the resource.

The Qualified Person with respect to the Etango Project resource estimate is Mr Brian Wolfe (Principal Resource Consultant) who is employed by Coffey Mining.

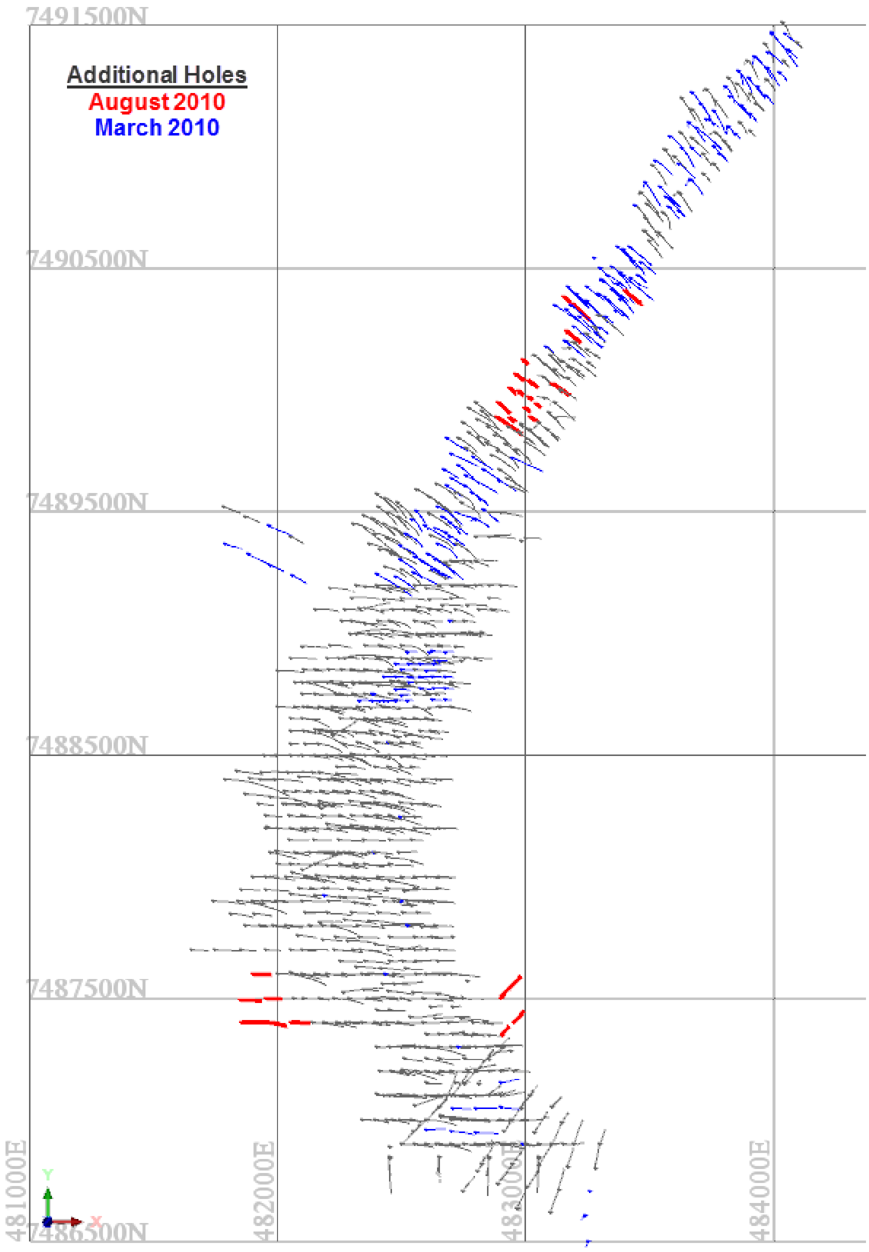
14.1.3 Resource Database and Validation

Database

The drill hole database used for the October 2010 resource estimate consists of 913 RC and 145 diamond drill holes for 246,950m. 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 October 2010 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 total of 58,065 chemical (93%) and radiometric (7%) assays were used in the estimate.

A density value of 2.64t/m^3 was used for the mineralised zones. This value was chosen after analysis of 8,883 density determinations from the mineralised zones by water immersion and calliper methods.

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 internal to mineralised zones were treated as null values (i.e. no samples), affecting 156 by 1m intervals.

Validation

The October 2010 drill hole database was checked by a variety of methods including:

- Checks of the top 200 assays against original laboratory certificates
- Database and visual comparison of assay, collar and survey data against the 2008 validated database
- 3D analysis of collar positions and downhole survey traces.

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

14.1.4 Geological Interpretation and Modelling

Geological and Mineralisation Model

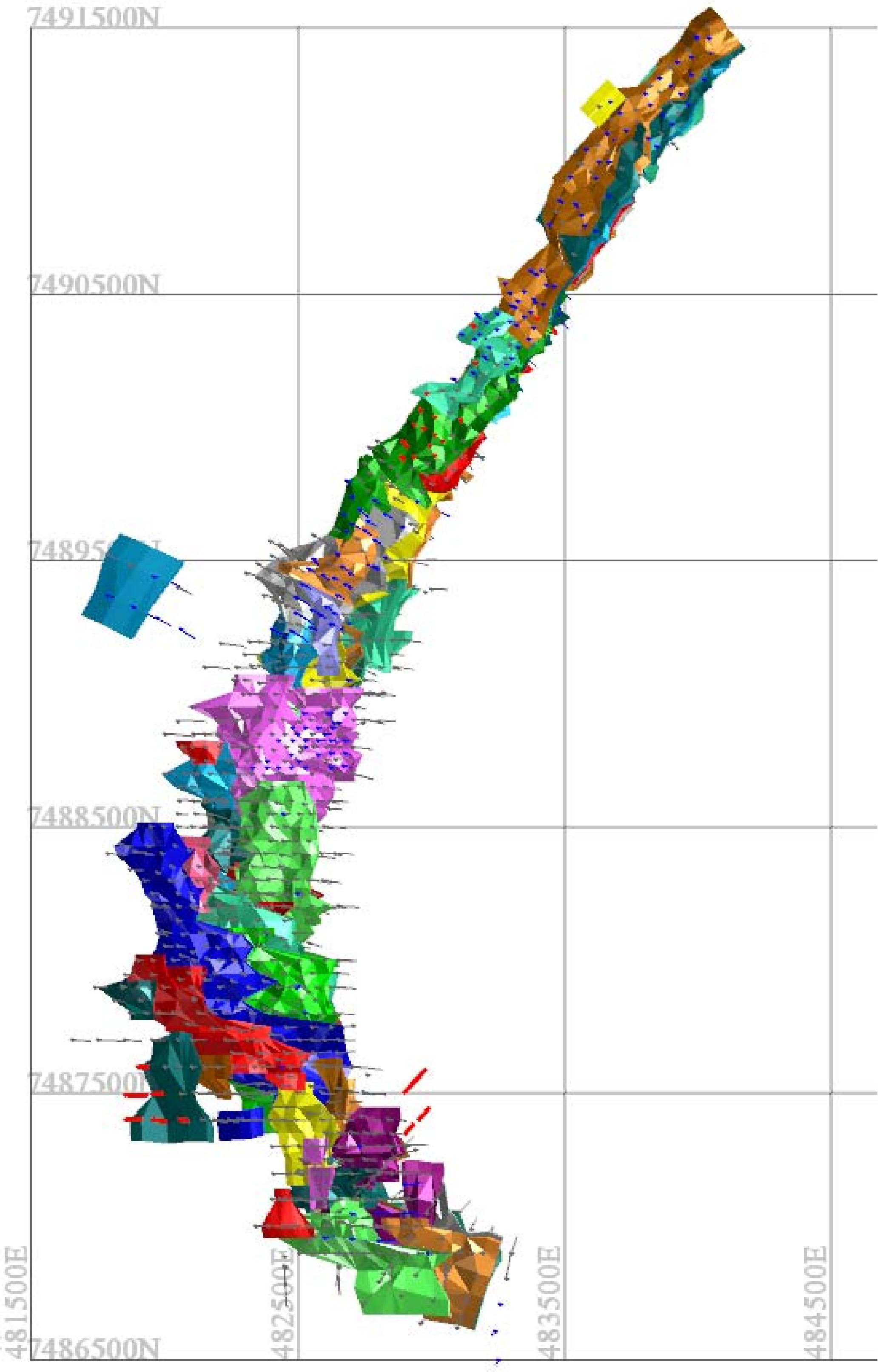
Separate three dimensional (3D) wireframe models were created for both the alaskite bodies and the mineralised zones (Figure 14-2). The majority of the uranium mineralisation (93% by metal content, 85% by sample count) is associated with the alaskite bodies and follows the trends of the alaskite contacts, typically with little coherent mineralisation occurring in the surrounding sediments. The alaskite contacts were therefore considered at the time of wireframe modelling and used to guide 3D modelling of the mineralisation shapes.

To establish appropriate grade continuity, the mineralisation model was based upon a nominal 75ppm U_3O_8 mineralisation halo. This nominal mineralisation outline typically also represented the natural cut-off of U_3O_8 mineralisation exhibited in the drill holes, with grades generally falling below 20-30ppm U_3O_8 away from the logged alaskite contacts.

The mineralisation boundaries within the alaskites bodies were often extended for up to 3m to the alaskite contacts, even if these intervals were not mineralised above the nominal 75ppm U_3O_8 cut-off.

Figure 14-2

Etango Uranium Project – Modelled Mineralised Zones



The mineralisation constraints were generated based upon sectional interpretation and three dimensional analyses of the available drilling data. The mineralised zones were modelled as 68 distinct zones (comprising 110 validated 3D shapes ranging from 3-135m thick – averaging 20m thick) with strike trends to the southeast, north and northeast following the western flank of the Palmenhorst Dome. The zones dip from 20° to 40° to the west. Individual zones were modelled with strike lengths ranging from 150-1,400m.

Weathering Profile

The pedolith mainly consists of <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).

14.1.5 Statistical Analysis

Radiometric Data Factoring

The vast bulk of the assays (93%) used in the resource estimate were analysed by XRF, with the remainder being factored gamma log e U_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).

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 1,100ppm e U_3O_8
  - Factored Auslog = Auslog eU_3O_8ppm * 0.86 - 26
- Bin 2 – 1,100ppm to 1,700ppm eU_3O_8
  - Factored Auslog = Auslog eU_3O_8ppm * 1.03 - 67
- Bin 3 – >1,700ppm
  - 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. To emulate a potential mining sub-bench size (i.e. 2.5m) it was decided to use 3m U_3O_8 composites for the estimation with a minimum allowable length of 1.5m. Statistical analysis was undertaken on the dataset with the residuals (<1.5m length) excluded. It was determined that inclusion of the residuals had a negligible effect on mean grades and therefore any residuals were not used in the estimates. Further statistical investigations were performed upon the 3m U_3O_8 composites from within each of the mineralised zones.

Summary statistics of the U_3O_8 composites are presented in Table 14-2.

Table 14-2
OK Resource – Summary Statistics for 3m U_3O_8 Composites (ppm)

Uncut 3m Composites									Cut 3m Composites		
Zone	**Number**	**Minimum**	**Maximum**	**Mean**	**Median**	**Std Dev**	**Variance**	**Coeff. Var.**	**Cut**	**Mean**	**% Change**
1	256	5	1,364	**209**	153	187	35,071	0.9	900	**207**	1
2	1433	4	1,104	**172**	139	130	16,823	0.8	850	**171**	0
3	1528	5	1,632	**213**	176	162	26,316	0.8	900	**212**	1
4	252	5	740	**143**	117	105	10,974	0.7	600	**142**	1
5	671	5	1,944	**210**	158	204	41,735	1.0	1000	**206**	2
6	82	5	607	**188**	163	131	17,078	0.7		**188**	-
7	53	23	1,142	**263**	163	250	62,437	1.0	850	**255**	3
8	18	77	255	**142**	134	49	2,375	0.3		**142**	-
9	361	5	1,695	**217**	150	216	46,806	1.0	1000	**213**	2
10	212	3	485	**158**	151	102	10,307	0.6		**158**	-
11	99	5	496	**138**	119	87	7,535	0.6		**138**	-
12	210	5	468	**113**	104	78	6,111	0.7		**113**	-
13	553	5	2,495	**182**	136	171	29,121	0.9	650	**175**	3
14	836	4	2,842	**257**	181	258	66,434	1.0	1350	**252**	2
15	127	33	749	**216**	184	120	14,434	0.6		**216**	-
16	149	5	1,340	**272**	226	192	36,970	0.7	850	**269**	1
17	85	5	1,055	**280**	211	222	49,369	0.8		**280**	-
18	2596	2	1,908	**215**	171	186	34,606	0.9	1400	**215**	0
19	63	9	339	**113**	82	86	7,334	0.8		**113**	-
20	456	5	2,132	**251**	208	227	51,413	0.9	1200	**249**	1
21	118	5	1,105	**168**	129	159	25,239	0.9	600	**161**	4
22	10	62	357	**135**	101	93	8,651	0.7		**135**	-
23	922	5	1,838	**210**	157	195	37,949	0.9	1150	**208**	1
24	155	5	855	**209**	183	158	24,922	0.8	700	**206**	1
25	576	5	2,137	**214**	177	202	40,930	0.9	1100	**209**	2

Table 14-2

OK Resource – Summary Statistics for 3m U_3O_8 Composites (ppm)

Uncut 3m Composites									Cut 3m Composites		
Zone	**Number**	**Minimum**	**Maximum**	**Mean**	**Median**	**Std Dev**	**Variance**	**Coeff. Var.**	**Cut**	**Mean**	**% Change**
26	584	5	2,282	**238**	198	217	47,229	0.9	1200	**235**	2
27	254	5	1,492	**222**	176	191	36,587	0.9	800	**217**	2
28	22	5	450	**166**	151	110	12,105	0.7		**166**	-
29	280	5	2,602	**178**	135	203	41,102	1.1	900	**172**	4
30	280	5	1,127	**173**	160	107	11,476	0.6	600	**171**	1
31	148	5	1,478	**218**	160	219	48,042	1.0	800	**210**	4
32	141	5	279	**103**	99	54	2,907	0.5		**103**	-
33	157	5	1,188	**186**	142	159	25,294	0.9	800	**184**	1
34	477	5	2,165	**161**	120	175	30,776	1.1	900	**156**	3
35	180	5	3,132	**251**	157	327	106,713	1.3	1000	**234**	7
36	121	5	789	**150**	111	148	21,877	1.0		**150**	-
37	28	56	404	**134**	106	81	6,562	0.6		**134**	-
38	55	5	1,417	**256**	197	243	58,869	0.9	800	**244**	5
39	210	5	1,169	**173**	131	169	28,507	1.0	800	**169**	2
40	33	5	396	**149**	129	100	10,064	0.7		**149**	-
41	92	5	719	**149**	118	124	15,416	0.8	600	**148**	1
42	43	2	1,574	**200**	137	254	64,393	1.3	800	**182**	9
43	40	9	415	**109**	98	74	5,416	0.7		**109**	-
44	70	70	489	**222**	203	92	8,552	0.4		**222**	-
45	41	5	370	**153**	130	110	12,157	0.7		**153**	-
46	119	5	520	**124**	99	101	10,292	0.8		**124**	-
47	16	66	317	**145**	127	65	4,198	0.4		**145**	-
48	17	36	323	**127**	114	81	6,575	0.6		**127**	-
49	17	5	922	**178**	124	213	45,287	1.2		**178**	-
50	973	5	1,675	**176**	131	173	29,757	1.0	1200	**175**	0

Table 14-2

OK Resource – Summary Statistics for 3m U_3O_8 Composites (ppm)

Uncut 3m Composites									Cut 3m Composites		
Zone	**Number**	**Minimum**	**Maximum**	**Mean**	**Median**	**Std Dev**	**Variance**	**Coeff. Var.**	**Cut**	**Mean**	**% Change**
51	278	5	2,033	**194**	128	232	53,688	1.2	1100	**188**	3
52	37	5	176	**96**	82	46	2,102	0.5		**96**	-
53	136	5	1,075	**170**	130	155	24,156	0.9	700	**166**	2
54	33	16	812	**218**	185	184	33,696	0.8		**218**	-
55	191	5	1,457	**177**	97	202	40,704	1.1	850	**172**	3
56	74	10	986	**205**	138	192	37,017	0.9	800	**201**	2
57	547	5	1,532	**165**	121	158	25,072	1.0	1200	**164**	1
60	649	5	1,004	**157**	129	126	15,828	0.8	700	**155**	1
61	657	5	1,339	**208**	166	167	27,879	0.8	1000	**207**	0
62	434	5	999	**191**	152	159	25,193	0.8	900	**191**	0
63	190	5	986	**200**	154	173	29,904	0.9	800	**198**	1
64	120	5	853	**194**	137	167	27,801	0.9		**194**	-
65	8	53	149	**96**	82	32	1,050	0.3		**96**	-
66	68	8	933	**192**	174	153	23,488	0.8	700	**188**	2
67	289	5	1,469	**182**	134	170	28,980	0.9	800	**177**	3
68	78	13	448	**120**	92	92	8,498	0.8		**120**	-
69	75	7	1,291	**224**	143	233	54,283	1.0	900	**215**	4
70	165	5	1,400	**274**	214	255	65,221	0.9	1000	**268**	2

Figure 14-3 shows typical histogram plots of the 3m U_3O_8 composite data from within Zones 2 and 5 respectively. Both plots demonstrate the strong positive tail typical of the deposit; however both datasets also have relatively low coefficients of variation (standard deviation/mean) of 0.75 for Zone 2 and 0.97 for Zone 5, indicating that positive outliers do not necessarily heavily impact upon the mean of the data population.

Assessment of the high grade U_3O_8 composites was completed on the zone grade populations to determine the requirement for high-grade cutting to be used for resource estimation. The approach taken included:

- Detailed review of histogram and probability plots, with significant breaks in populations used to interpret possible outliers
- Detailed review of spatial distribution plots
- Ranking of the composite data and the investigation of the influence of individual composites on the mean and standard deviation.

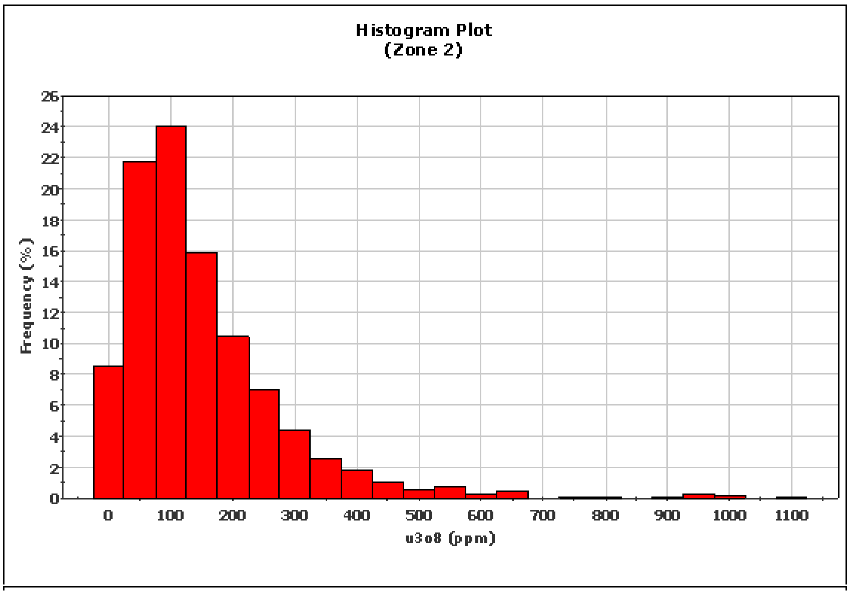
The top cuts used and their effect on the mean of the mineralised zones average grade are shown in Table 14-2. The effect of applying top cuts to the bulk of the zones was to reduce the naïve mean typically by between 1 to 4%. However, some zones were highly sensitive to the cutting of a relatively few high grade samples (e.g. Zone 42, where high grade cutting resulted in a 9% decrease in the mean) due to high-grade outliers.

Figure 14-3

Resource Estimate Histogram of 3m U_3O_8 Composites for Zones 2 and 5

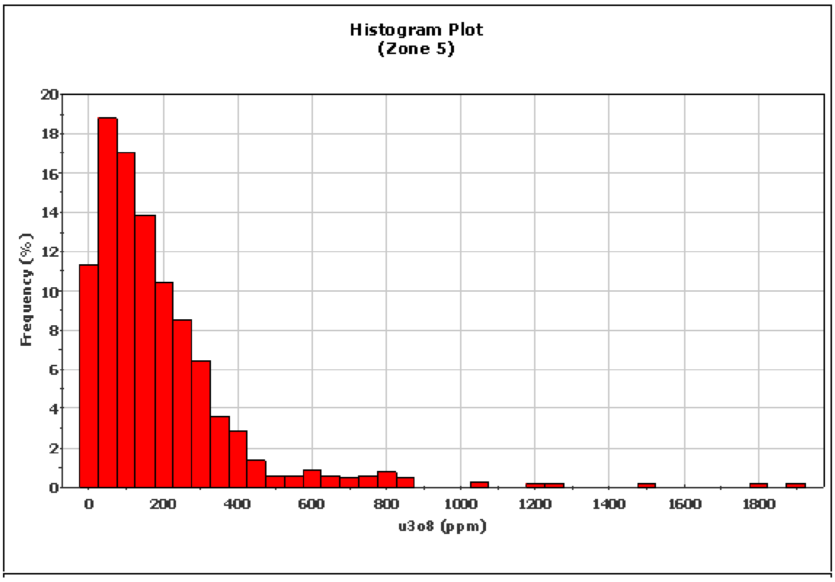
(Zone 2)

	Unweighted	Weighted	Units
Samples:	1,433	N/A	
Minimum:	3.67	N/A	ppm
Maximum:	1,104.33	N/A	ppm
Mean:	171.87	N/A	ppm
Median	139.53	N/A	ppm
Std. Deviation:	129.66	N/A	ppm
Coefficient of Variation:	0.75	N/A	



(Zone 5)

	Unweighted	Weighted	Units
Samples:	671	N/A	
Minimum:	5.00	N/A	ppm
Maximum:	1,943.67	N/A	ppm
Mean:	209.94	N/A	ppm
Median	158.33	N/A	ppm
Std. Deviation:	204.14	N/A	ppm
Coefficient of Variation:	0.97	N/A	



Bulk Density Data

The bulk density readings were taken from 76 diamond drill holes located along the trend of the deposit (Figure 14-4) with a total of 5,889 water immersion measurements and 11,113 calliper measurements available. Summary statistics for the mineralised zone and sediment bulk density measurements are shown in Table 14-3. The location of the bulk density readings are shown in Figure 14-4.

Figure 14-4
Location of Density Readings

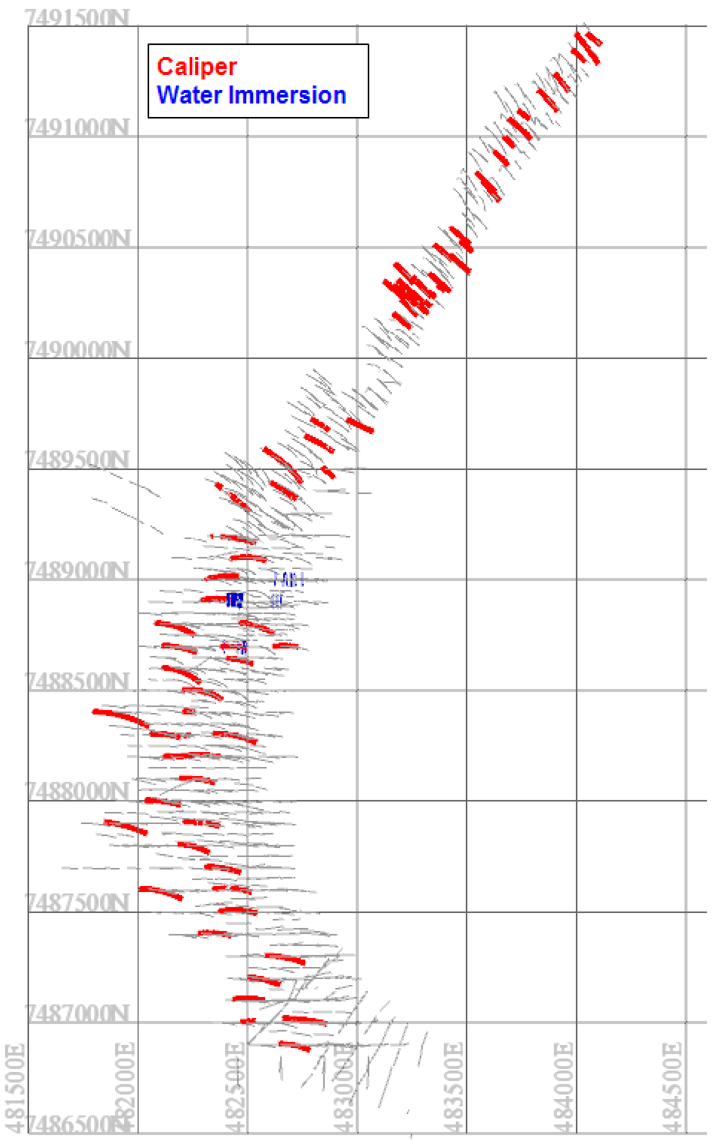


Table 14-3
Summary Statistics for Bulk Density Data
(Calliper and Water Immersion) (t/m³)

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 consist predominantly of alaskite lithologies 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 14-5 shows histogram plots of the mineralised zone bulk density data. Figure 14-6 shows histogram plots of the meta-sedimentary unit bulk density data.

Figure 14-5
Histogram Plot of the Mineralised Zones Bulk Density Measurements

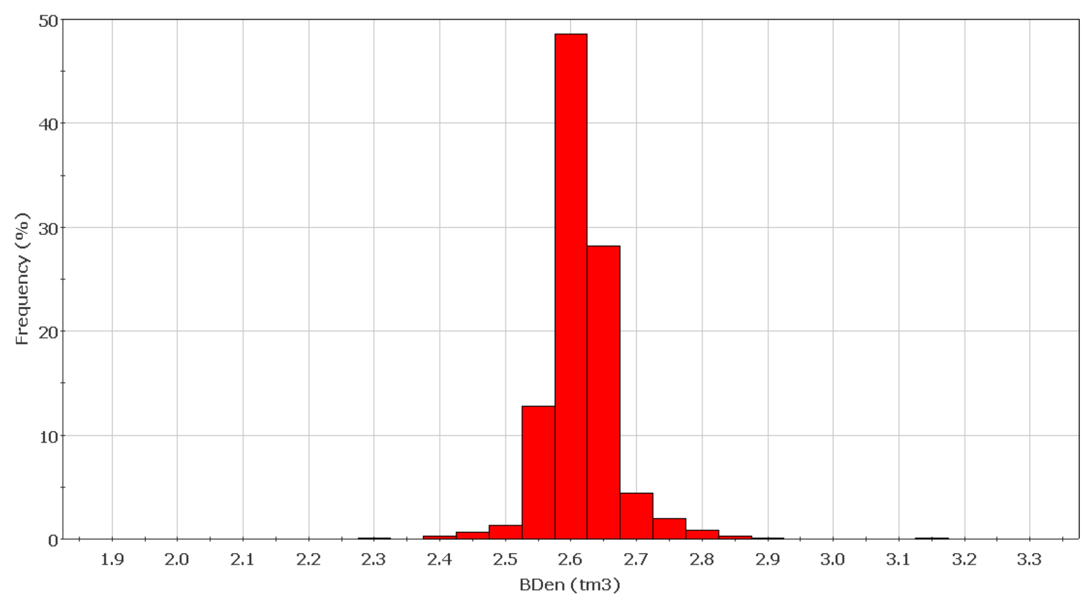
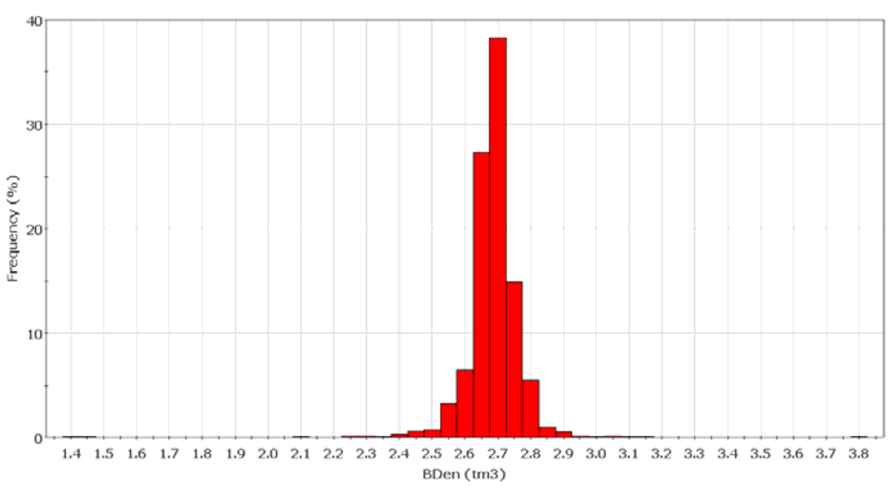


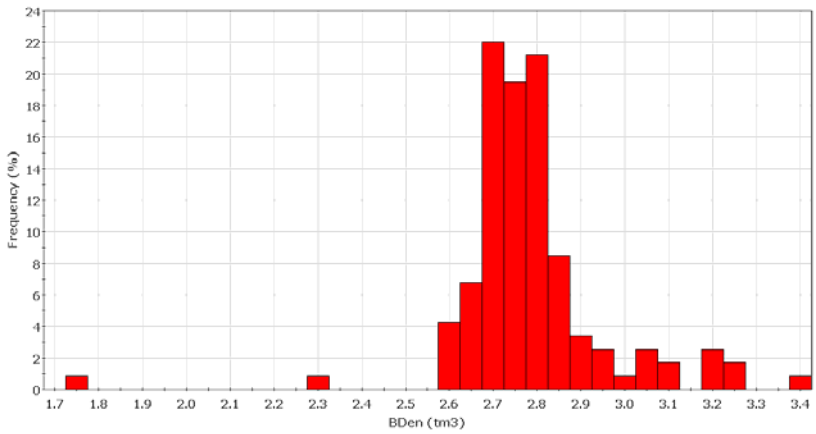
Figure 14-6

Histogram Plot of Bulk Density Readings from the Metasediments

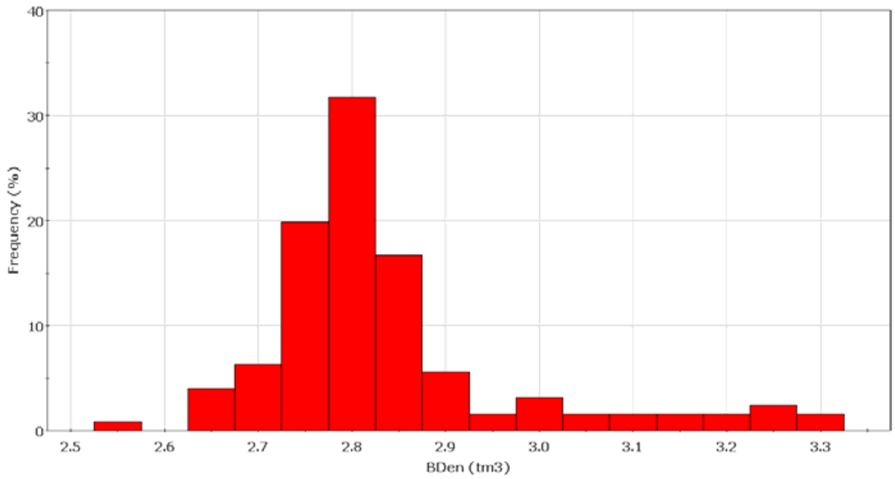
(CGN – Water Immersion and Calliper)



(EGN – Water Immersion and Calliper)



(KGN – Water Immersion and Calliper)



14.1.6 Variography

In this document, the term 'variogram' is used as a generic word to designate the function characterising the variability of variables versus the distance between two samples. Isatis geostatistical software was used throughout. Both traditional semi-variograms and correlograms were used to analyse the spatial variability of U_3O_8 for 3m composites from the mineralised zones. Downhole variography was calculated and used to determine the nugget for each of the zones, Table 14-4.

Table 14-4
OK Resource – Variogram Parameters

Zone	Zones Applied To	Co (%)	C1 (%)	First Spherical Structure Range (m)			C2 (%)	Second Spherical Structure Range (m)		
				Major	Semi Major	Minor		Major	Semi Major	Minor
2	2; 4; 5; 6; 7; 15; 21; 36; 40	31	40	30	30	8	29	100	100	28
3	3; 7; 8; 9; 10; 11; 16; 38; 41; 42; 43; 45	35	40	40	40	13	26	144	134	31
13	13	32	45	50	50	11	23	150	150	25
14	14; 17; 27	27	41	40	40	12	32	120	90	30
18	1; 18; 32; 37	40	35	40	40	12	25	140	85	36
23	20; 23; 28; 35; 44	35	39	36	36	14	26	135	100	33
25	12; 19; 22; 24; 25; 26; 29; 46; 50; 51; 52; 53; 54; 55; 56; 57	35	35	40	40	8	30	120	120	22
30_34	21; 30; 31; 33; 34; 39; 47; 48; 49	34	43	30	30	15	23	130	130	30
60	60, 70	20	50	60	60	10	30	140	130	20

Variography was calculated based for key domains, namely Zones 2, 3, 13, 14, 18, 23, 25, 30/34, and 60. Table 14-5 summarises the resulting variogram models used in the resource estimate.

Table 14-5
Variogram and Search Ellipse Orientation Parameters

Zones	Axis Orientation (dip→dip direction)		
	Major	Semi-Major	Minor
12	15→000	43→255	43→104
1, 6, 7, 8, 9, 11, 16, 19, 21, 21, 31, 32, 35, 44, 45	00→000	24→270	66→090
10	00→000	30→270	60→090
24	15→180	23→276	61→059
5, 15, 18, 36, 37, 38, 39, 40, 47, 48, 49	20→180	22→278	59→052
25	00→025	24→295	66→115
23	00→025	45→295	45→115
50, 52, 54, 61-70	00→030	30→300	60→120
55, 57	15→048	29→309	56→162
22, 53	05→220	30→313	170→121
26, 28, 29, 46, 56	10→220	57→315	58→113
51	15→048	29→309	56→162
13	00→130	30→220	60→040
2, 3, 4, 17, 27, 30, 34, 41, 42, 43	00→140	24→230	66→050
33	00→140	45→230	45→050

All zones exhibited a well-structured downhole variogram with a relative nugget between 20% and 40%. The variography in the major and semi-major axes generally had moderately defined structure and were modelled with a first structure at ranges of between 30m to 60m in the major axis. This has typically resulted in most of the zones having between 68% and 77% 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 100m to 150m.

Figure 14-7 and Figure 14-8 show an example of the obtained variography from Zones 2 and 23.

Figure 14-7

Zone 2 Variogram Plot

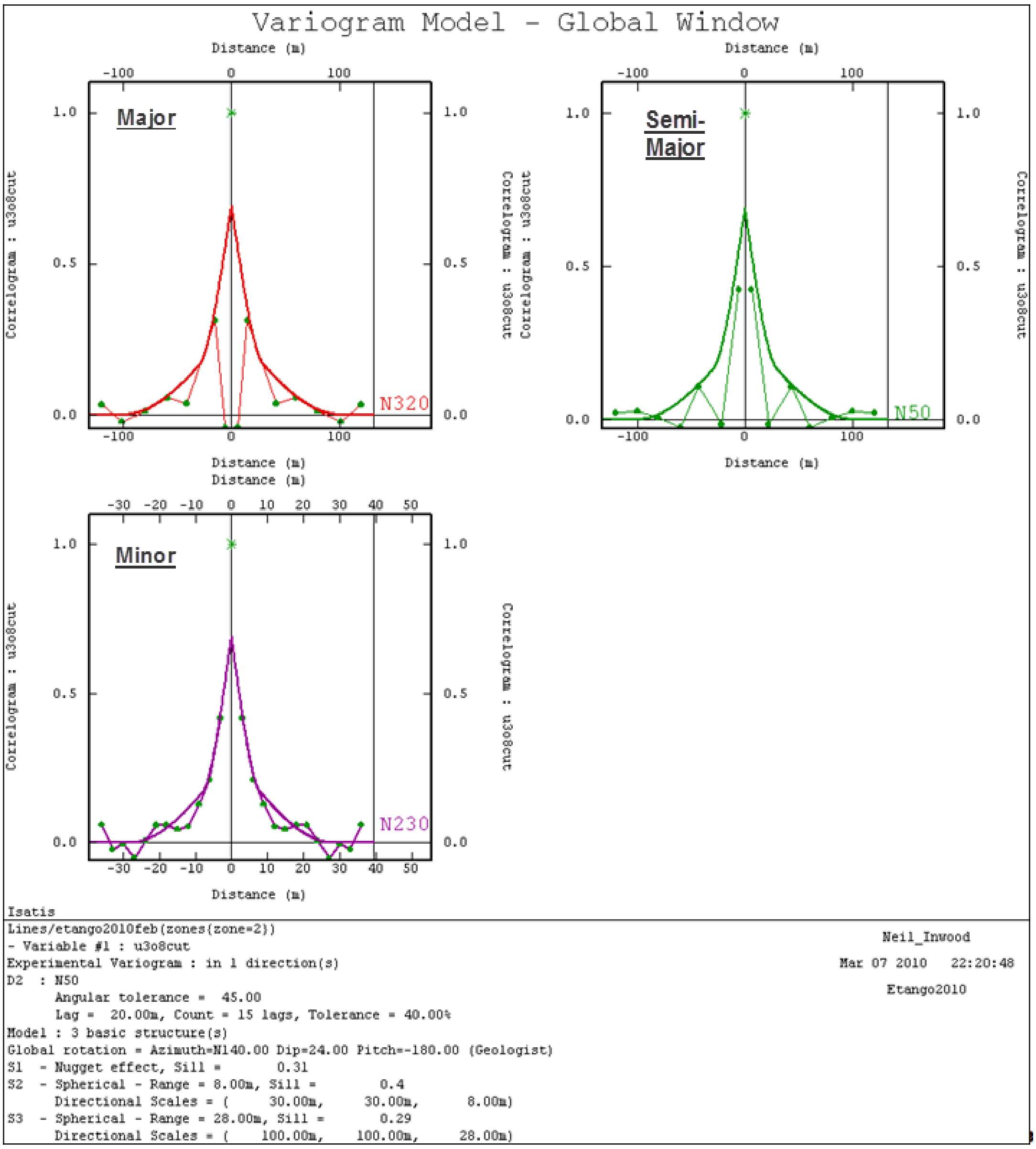
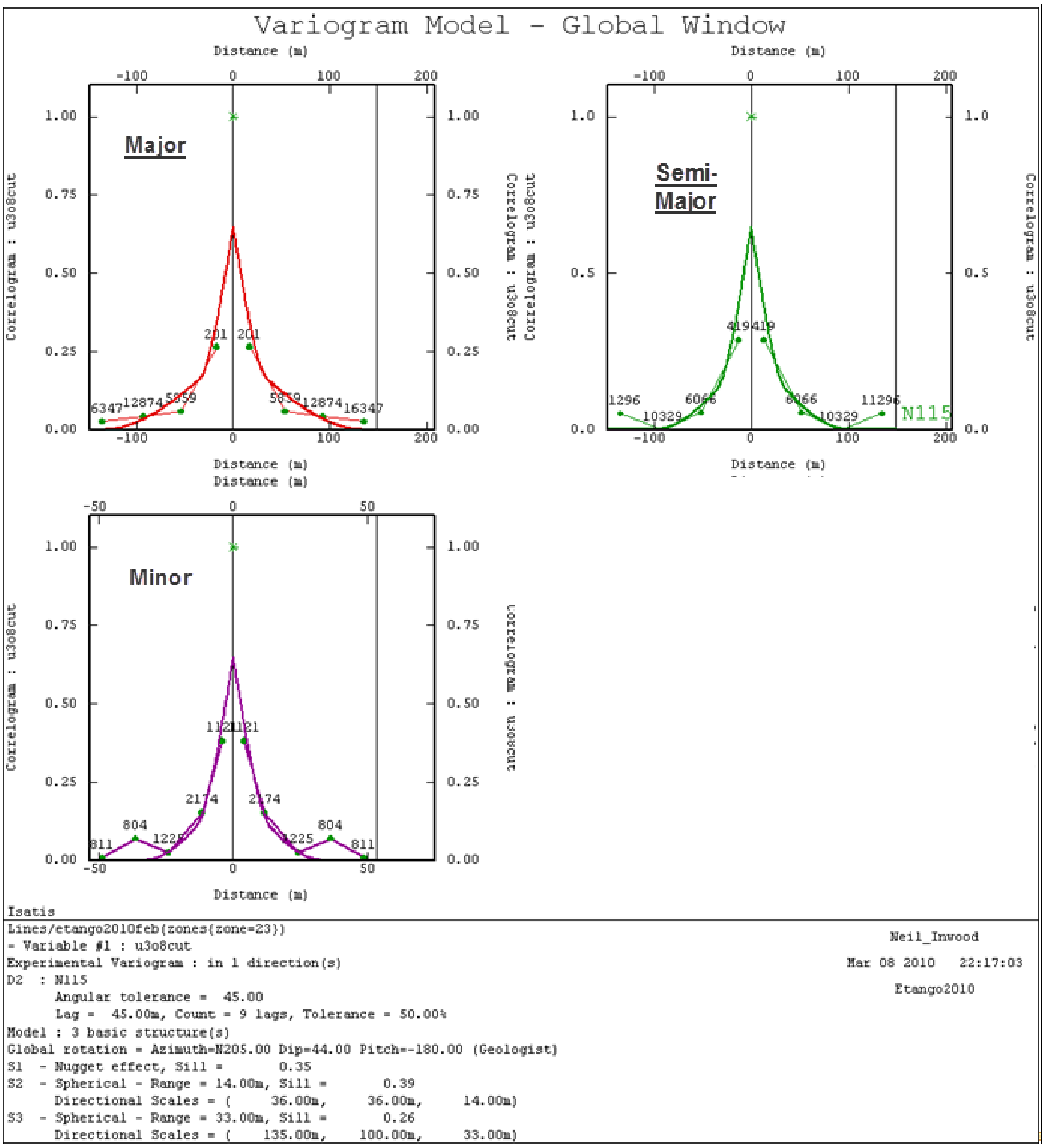


Figure 14-8

Zone 23 Variogram Plot



14.1.7 Block Model Construction

A block model was created using Surpac mining software with a parent cell size of 25mE by 25mN by 10mRL, which was sub-blocked to 6.25 x 6.25 x 1.25m. No rotation was applied to the block model. The block model parameters are summarised in Table 14-6. Variables were coded into the model to allow for grade estimation with service variables added to allow for statistical analysis and validation of the grade estimate and assessment of the quality of the estimate.

Table 14-6 Block Model Parameters			
	Easting (X)	**Northing (Y)**	**RL (Z)**
Minimum Coordinates	481,500	7,486,500	-300
Maximum Coordinates	484,800	7,492,000	350
Block size (m)	25	25	10
Sub Block size (m)	6.25	6.25	1.25

14.1.8 Grade Estimation

OK Estimate

Grade was estimated into the block models by OK for U_3O_8 using Surpac mining software. Sample neighbourhood testing was conducted, to determine an appropriate search strategy for the OK estimation. Neighbourhood testing included investigations into the minimum and maximum number of samples used for estimation, negative kriging weights, the slope of regression and the resulting kriging variance.

As the Bannerman drilling had been completed on a regular grid pattern, drill hole data clustering was not a significant problem, and similar sample selection criteria were used for all mineralised zones. The sample search was orientated according to the variogram and search ellipse orientations shown in Table 14-5. A staged sample search strategy was applied, as summarised in Table 14-7.

Table 14-7
Sample Search Parameters – Ordinary Kriging

Zones	Pass	Search Radii			Number of Samples		
		Major Axis (m)	**Semi-Major Axis (m)**	**Minor Axis (m)**	**Min**	**Max**	**Maximum / Hole**
All	1	65	65	32.5	12	24	5
	2	130	130	65	12	24	5
	3	260	260	130	6	24	5

The sample selection criteria are presented in Table 14-7. The variogram parameters used for the estimation were based upon the variography discussed in Section 14.1.6 and are summarised in Table 14-5.

Hard domain boundaries were used during estimation for the individually numbered zones (i.e. 68 separate grouped zones), although soft boundaries were used for separately modelled subsets of the same zone number. Discretisation of 5Nx5Ex5RL was used for block estimates.

Validation

A detailed visual and statistical review of the OK 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 14-9 shows an example of the validation plots.

- A comparison of the block model whole block estimate versus the mean of the composited dataset (Table 14-8).

Figure 14-9

Validation Plot Examples

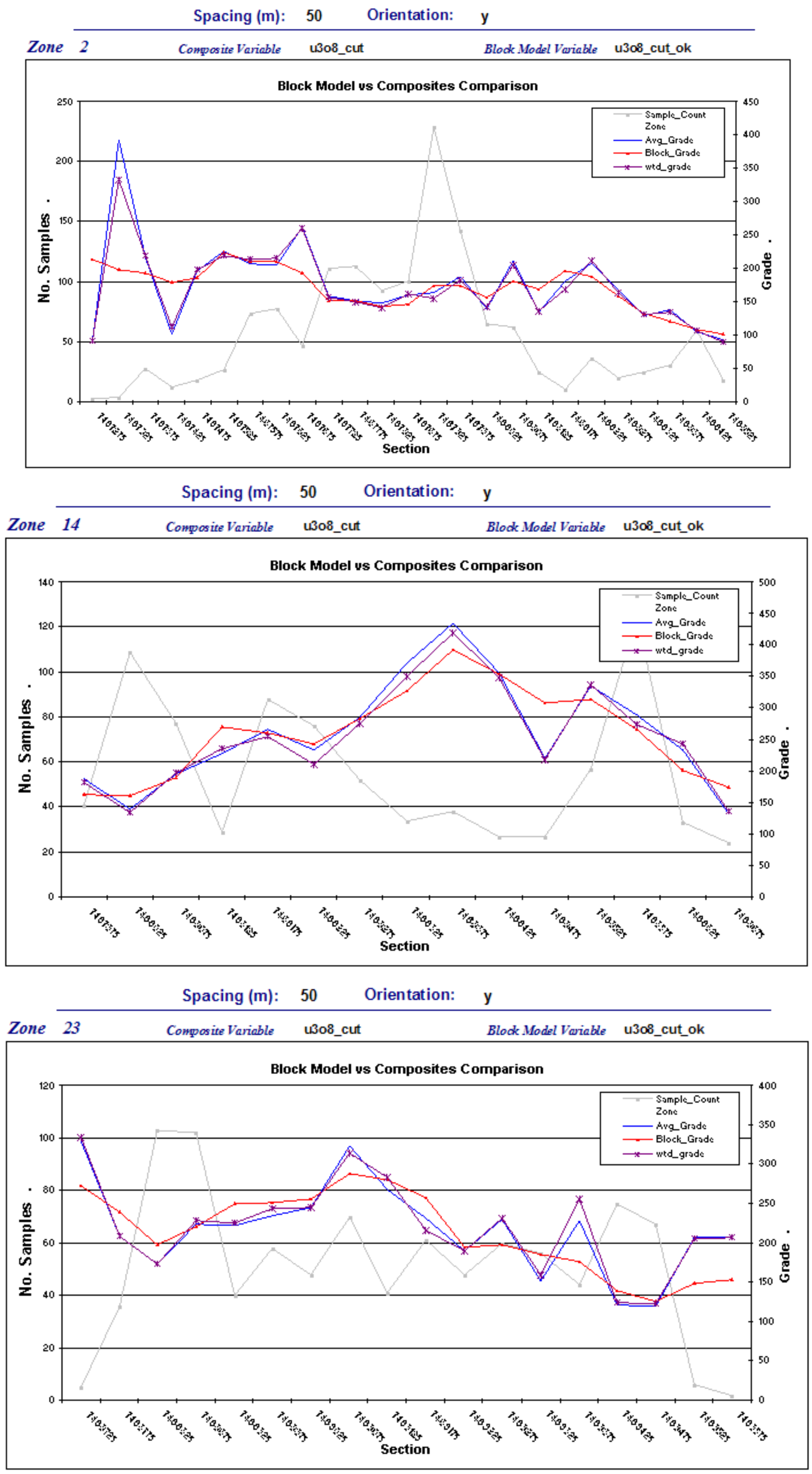


Table 14-8
OK Block Estimates Versus 3m Composite Data Comparison

Zone	Block Grade	Naïve Composite Mean	Declustered Composite Mean	BM % Difference to Naïve Mean (%)	BM % Difference to Declustered Mean (%)
1	198	207	209	-4	-5
2	168	171	170	-2	-1
3	206	212	208	-3	-1
4	144	142	143	1	0
5	203	205	207	-1	-2
6	196	188	191	4	3
7	274	255	259	7	6
8	142	142	143	-1	-1
9	212	213	208	0	2
10	156	158	156	-1	0
11	116	138	134	-16	-13
12	112	113	111	-1	1
13	172	175	170	-2	1
14	252	252	246	0	2
15	227	216	215	5	6
16	274	269	264	2	4
17	292	280	280	4	4
18	211	215	211	-2	0
19	114	113	116	1	-2
20	245	248	247	-1	-1
21	138	161	162	-14	-15
22	136	135	136	1	0
23	212	208	212	2	0
24	202	206	207	-2	-3
25	209	210	211	0	-1
26	231	235	229	-2	1
27	215	217	211	-1	2
28	160	166	166	-3	-3
29	167	172	170	-3	-1
30	168	171	170	-2	-2
31	208	210	212	-1	-2
32	106	103	105	3	1
33	184	183	186	0	-1
34	160	156	158	3	1
35	257	231	233	11	10
36	157	150	156	5	1
37	137	134	133	2	3
38	258	244	260	6	-1
39	176	169	177	4	0
40	151	149	154	2	-2
41	133	148	147	-10	-10

Table 14-8
OK Block Estimates Versus 3m Composite Data Comparison

Zone	Block Grade	Naïve Composite Mean	Declustered Composite Mean	BM % Difference to Naïve Mean (%)	BM % Difference to Declustered Mean (%)
42	166	182	165	-9	0
43	120	109	109	9	10
44	220	222	221	-1	-1
45	157	153	156	3	1
46	127	124	127	2	0
47	144	145	147	-1	-2
48	133	127	129	4	3
49	200	153	164	31	22
50	170	170	168	0	1
51	195	187	186	4	5
52	97	96	97	0	-1
53	163	166	170	-1	-4
54	207	214	226	-3	-8
55	164	171	173	-4	-5
56	227	198	196	15	16
60	162	155	157	4	4
61	202	208	202	-3	0
62	183	191	190	-4	-4
63	203	198	191	3	6
64	190	194	191	-2	-1
65	88	96	90	-8	-2
66	179	188	173	-5	3
67	175	177	179	-1	-2
68	122	120	124	2	-1
69	227	215	208	5	9
70	261	268	268	-3	-3

Zones which exhibited unexpected grade differences to the input composites were checked in 3D for potential errors, these differences typically being found to result from the proportional effect of a low number of composites in smaller areas of irregular geometries (e.g. Zone 49).

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

Bulk Density

The bulk density values used for the resource model were based upon the data analysed in Section 14.1.5. A value of 2.64t/m³ was used for all material within the modelled alaskite bodies. The same value was coded into all modelled mineralised zones. Bulk densities of 2.70t/m³, 2.86t/m³ and 2.80t/m³ were coded for the Chuos, Khan and Etusis lithologies respectively.

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.

14.1.9 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 and the JORC Code. A combination of Measured, Indicated and Inferred Resources have been defined using definitive criteria determined during the validation of the grade estimates, with detailed consideration of the NI43-101 categorisation guidelines.

Criteria for Resource Categorisation

The resource has been classified as a combination of Measured, Indicated and Inferred Mineral Resources based on the confidence level of the key criteria that were considered during resource classification as presented in Table 14-9. Figure 14-10 illustrates the classification applied to the mineral resource block model.

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 high slope of regression), and a nominal 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, and a nominal 50m x 50m to 50m x 100m drill hole coverage.

Inferred Resource

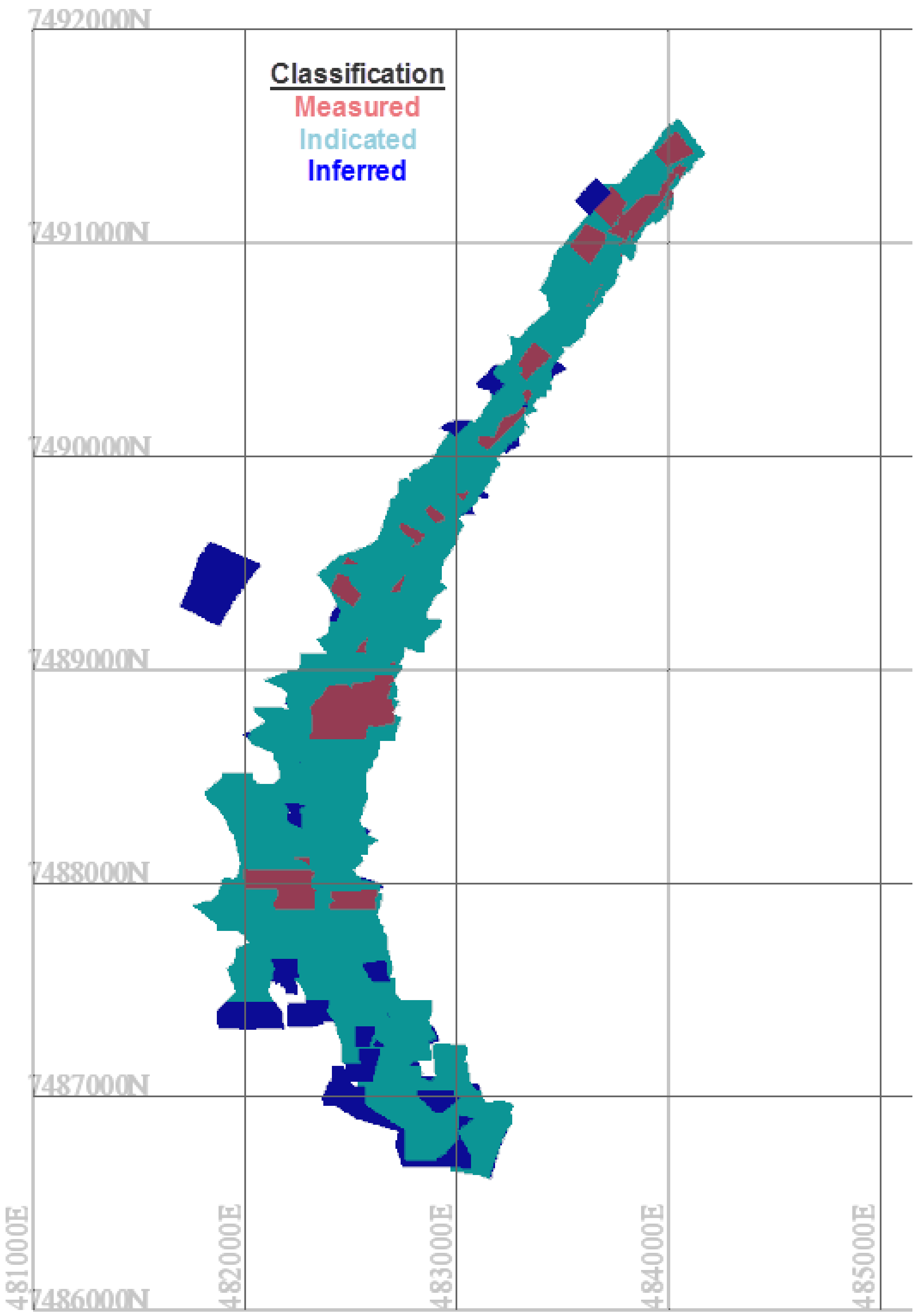
An Inferred category was applied to all mineralisation zones which were not classified as Indicated or Measured.

Table 14-9
Confidence Levels of Key Categorisation Criteria

Items	Discussion	Confidence
Drilling Techniques	RC/Diamond – industry standard approach.	High
Logging	Standard nomenclature applied with recording and apparent high quality.	High
Drill Sample Recovery	Acceptable recoveries determined for the majority of the drilling.	High
Sub-sampling Techniques and Sample Preparation	Industry standard for both RC and diamond drilling	High
Quality of Assay Data	Good internal laboratory and external quality control data available for the majority of the chemical assaying. Factored radiometric data is considered to be globally equivalent to chemical assaying, but can show local differences.	Moderate
Verification of Sampling and Assaying	Twinning of selected RC and diamond holes indicates diamond drilling results are similar to RC results.	High
Location of Sampling Points	Most drill hole collars surveyed by GPS surveyed and most drill holes have been downhole surveyed.	High
Data Density and Distribution	The deposit defined on a notional 50mE x 50mN to 50mE x 100mN with some 25m E x 25mN to 25mE to 50mN infill drill hole spacing with most holes drilled through the mineralised zones.	Moderate - High
Audits or Reviews	Coffey Mining has reviewed the site drilling and sampling procedures. The model has not been externally audited.	High
Database Integrity	No material errors identified.	High
Geological Interpretation	The interpreted lithological and mineralisation boundaries are considered reasonably robust. Infill drilling continues to vary interpretations slightly with respect to both structural and grade continuity. Some low grade mineralisation of presumed limited extent is not able to be directly interpreted and modelled.	Moderate
Estimation and Modelling Techniques	Estimates based on detailed statistical and geostatistical analysis. Estimation by Ordinary Kriging is satisfactory	Moderate
Cutoff Grades	Range of cutoff grades reported. The OK model is valid for a limited range of cutoffs for which the model was designed. The tenor of mineralisation will result in sensitivity of the reported tonnages and grades to the cutoff grade chosen.	Moderate
Mining Factors or Assumptions	Whole block estimates for all mineralised regions completed for 25mE by 25mN by 10mRL size blocks. The OK model does not incorporate edge dilution, ore loss, nor does it represent an SMU model.	Moderate

Figure 14-10

Plan View of the Classified Block Model



14.1.10 Etango Grade Tonnage Reporting

The OK resources for the Etango Project reported above various cut-offs are summarised in Table 14-10. 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.

Coffey Mining is unaware of any mining, metallurgical, infrastructure or other relevant factors which may materially affect the resources.

Table 14-10

Etango Deposit, Etango Project, Namibia – October 2010 Resource Estimate

Classification	Lower Cut	Tonnes Above Cut-off (Mt)	U_3O_8 (ppm)	Contained U_3O_8 (t)	Contained U_3O_8 (M lb)
Inferred	100	45.7	202	9,200	20.3
	125	40.3	214	8,600	19.0
	150	34.7	226	7,800	17.3
Indicated	100	273.5	200	54,600	120.4
	125	238.6	212	50,700	111.7
	150	193.7	230	44,500	98.1
Measured	100	62.7	205	12,900	28.3
	125	56.6	215	12,200	26.8
	150	47.5	230	10,900	24.0

Note: Figures have been rounded. Conversion of lb to kg = x 2.20462
OK Model Reported at Various Cut-offs Using a Bulk Density of 2.64t/m^3
Panel dimensions of 25m N by 25m E by 10m RL

14.1.11 Etango Summary, Conclusions and Recommendations

The October 2010 Resource update represents an incremental increase in the Etango resource endowment. Additional infill drilling and improved understanding of the mineralisation (particularly in the Onkelo region) have resulted in increased Measured and Indicated material in the updated estimate.

The following limitations of the OK model 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 100ppm to 150ppm U_3O_8).
- The tenor of the Etango mineralisation will result in sensitivity of the reported tonnages and grades to the cut-off grade chosen.
- The OK model represents whole block estimates for all mineralised regions completed using 25mE by 25mN by 10mRL parent blocks.
- The OK model does not incorporate edge dilution or ore loss, nor does it represent an SMU model (adjusted for mining scale selectivity).

- The OK model possibly omits a small amount of what is considered to be low grade mineralisation having limited extent.

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

Ondjamba

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 (8,252 1m composites – 42% of the total) were used for the estimation. The radiometric data was factored such that the mean of the eU_3O_8 data matched that of the chemical data. Within the mineralisation domains, 3,220 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 9,061m 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 (6,803 samples - 67% of the total) and factored radiometric data (3,311 1m composites – 33% of the total) were used for the estimation. Within the mineralisation domains 1,616 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 14-11) 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 14-12). Individual zones were modelled from 150m to 1,750m long. Figure 14-12 shows a typical sectional interpretation.

Hyena

The mineralised zones at Hyena (Figure 14-13) were modelled as 19 distinct zones in 4 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 3 exhibiting a near vertical west-east trend (Figure 14-14). Individual zones were modelled from 150m to 1,750m long. Figure 14-14 shows a typical sectional interpretation.

Figure 14-11

Ondjamba Mineralised Zones and Drilling

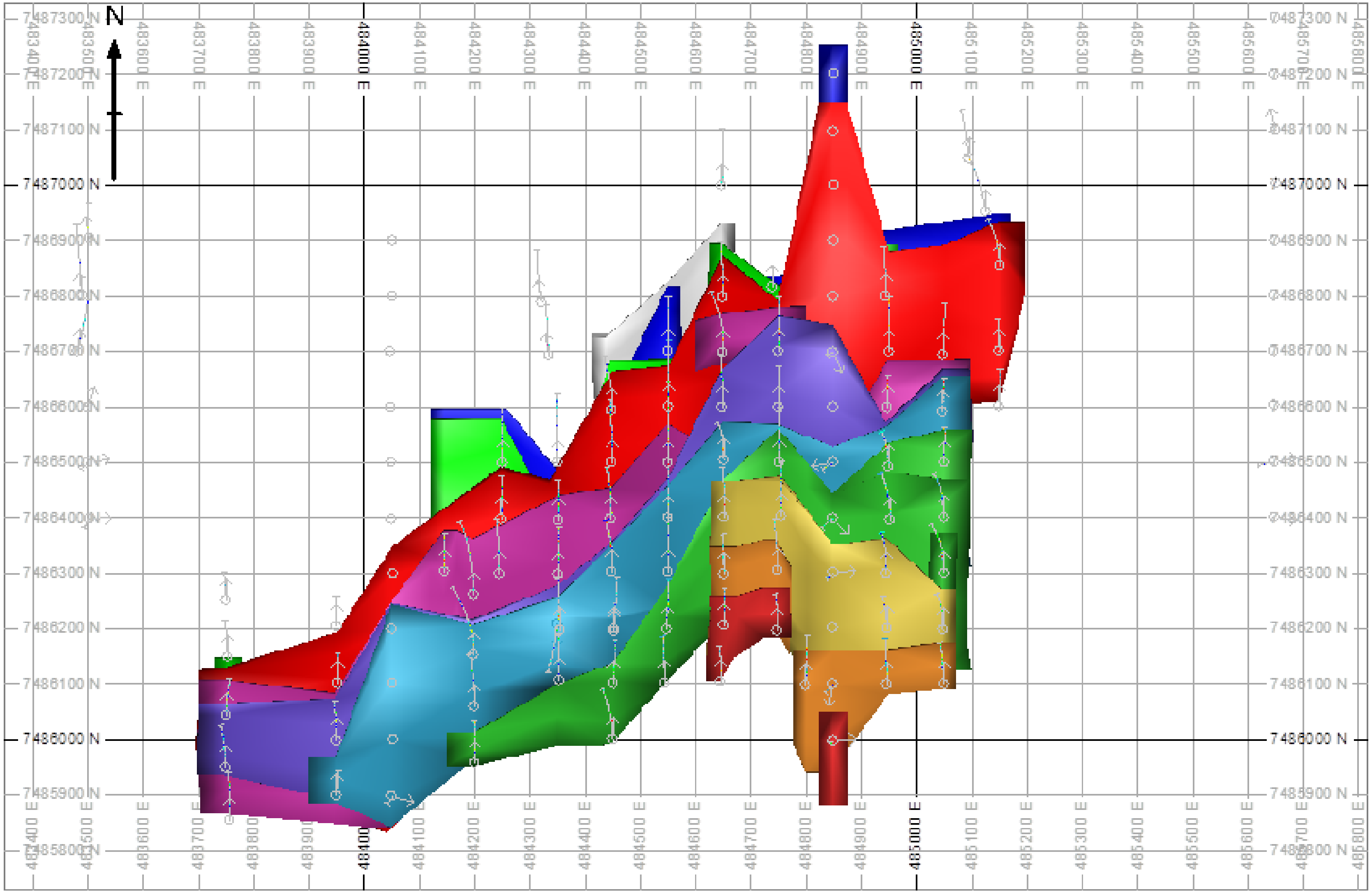


Figure 14-12

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

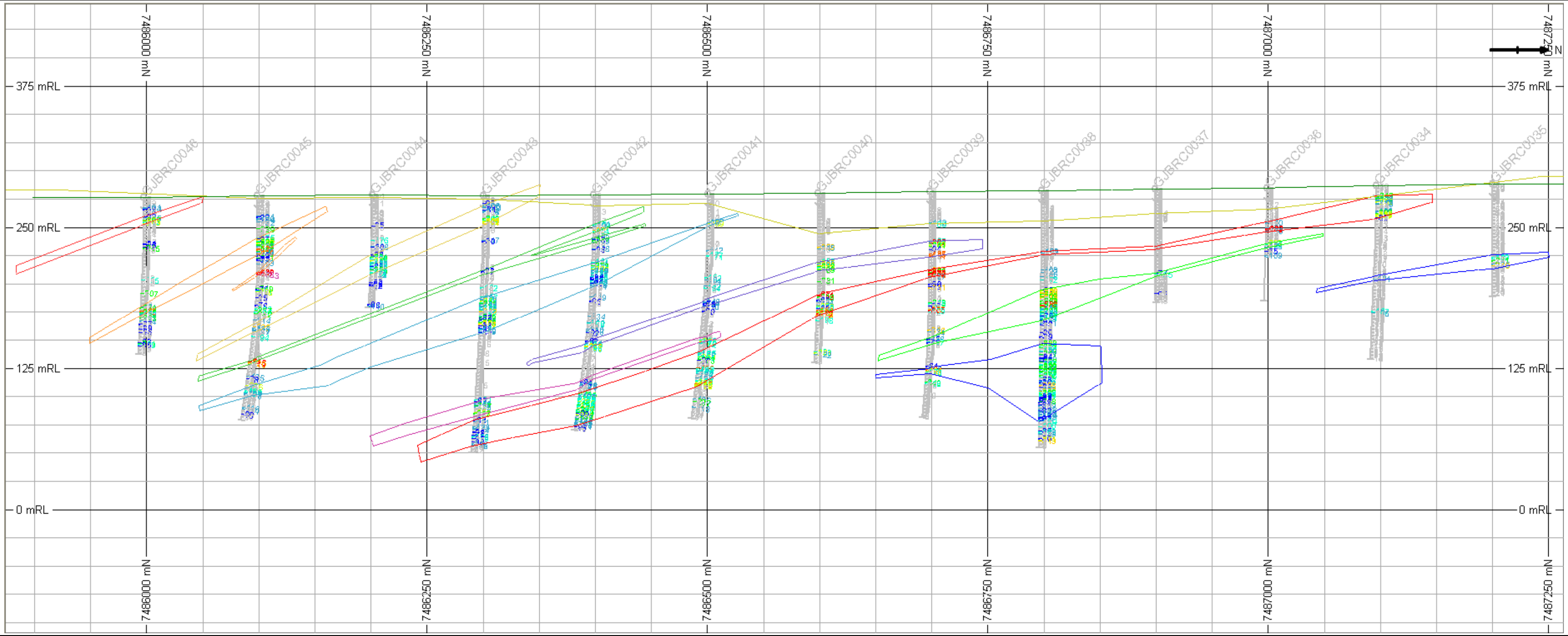


Figure 14-13

Hyena Mineralised Zones and Drilling

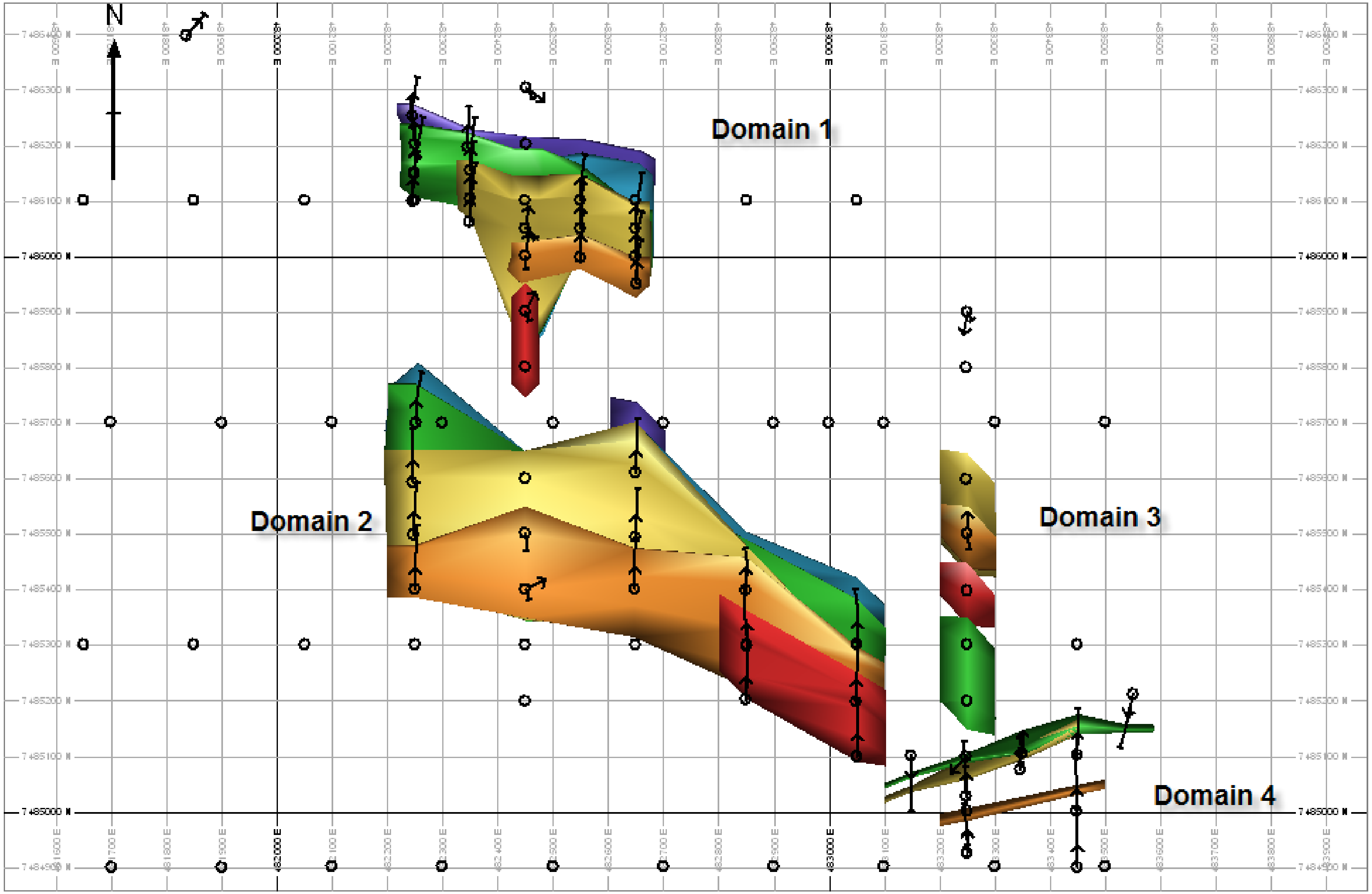
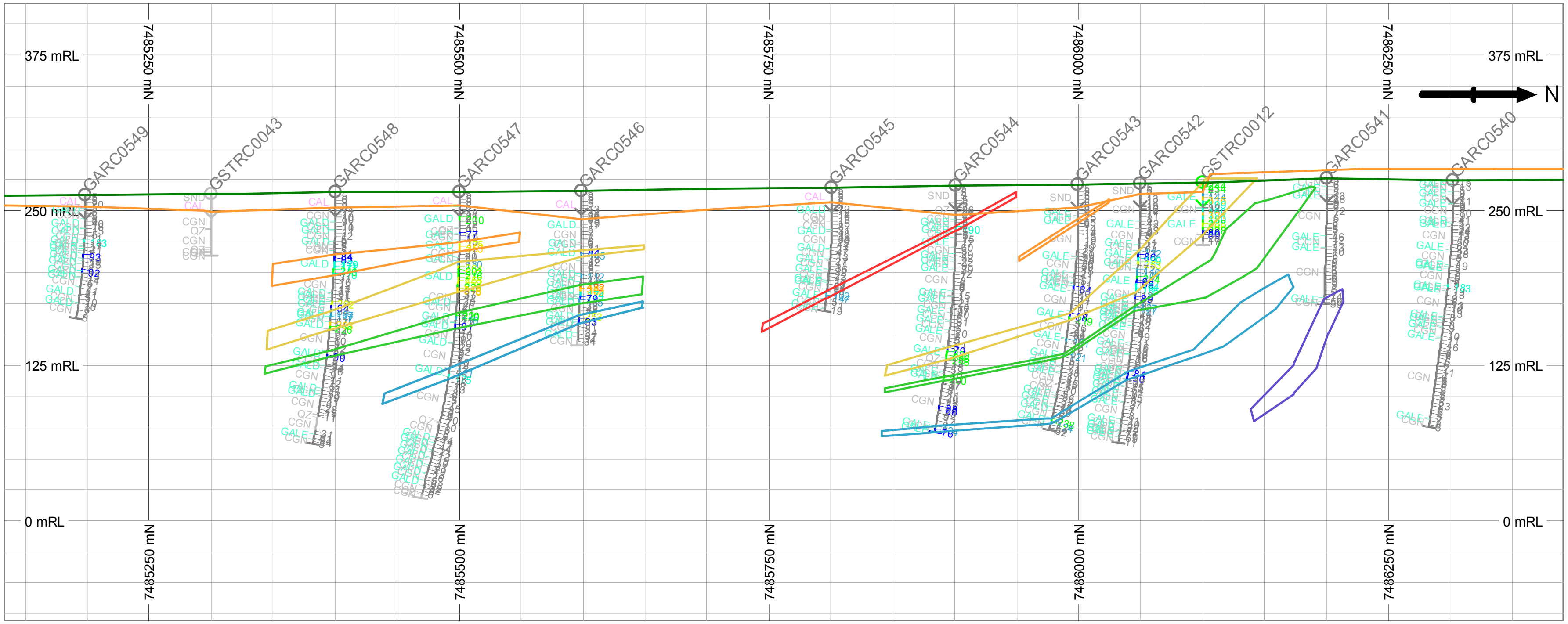


Figure 14-14

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



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 high-grade cuts, variogram model parameters, and search ranges etc.

A single upper cut of 700ppm U_3O_8 was applied to the 3m composites for all Ondjamba zones prior to estimation. The effect of the upper 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 an upper cut of 850ppm U_3O_8 was applied to the 3m composites for Hyena Domains 1, 2 and 4, and an upper cut of 1,250ppm was applied to Domain 3 prior to estimation. The effect of the upper 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, the geostatistical investigations completed, and the parameters are in common with the nearby Etango model. 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 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 reported resource for the Ondjamba and Hyena deposits reported above various cut-offs are summarised below (Table 14-11 and Table 14-12). 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. If cut-off grades substantially higher than the Coffey Mining preferred cut-off grade are to be used for public reporting (e.g. >150ppm U_3O_8), the resource classification will need to be reviewed to accommodate the different risk profile.

Table 14-11
Ondjamba Deposit, Etango Project, Namibia – October 2010 Resource Estimate

Lower Cut	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 14-12
Hyena Deposit, Etango Project, Namibia – October 2010 Resource Estimate

Lower Cut	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 Combined Mineral Resources

The combined October 2010 mineral resource estimate, reported at a cut-off grade of 100ppm U_3O_8, comprises Measured and Indicated resources of 336.2Mt at 201ppm for 148.7Mlb of contained U_3O_8, and Inferred resources of 164.6Mt at 176ppm for 63.9Mlb of contained U_3O_8.

The mineral resource estimate has been prepared in accordance with the Australian JORC Code guidelines and Canadian National Instrument 43-101 by Coffey Mining.

The combined mineral resource estimate is tabulated below, firstly (in Table 14-13) by individual deposit area (at a cut-off grade of 100ppm U_3O_8) and, secondly (in Table 14-14), for the total Project estimate at a range of cut-off grades.

Table 14-13 Etango Project Mineral Resource Estimate October 2010 – By Deposit Reported At A Cut-Off Grade Of 100ppm U_3O_8												
	Measured Resources				Indicated Resources				Inferred Resources			
Deposit	Tonnes	Grade	Contained U_3O_8		Tonnes	Grade	Contained U_3O_8		Tonnes	Grade	Contained U_3O_8	
	(Mt)	(ppm U_3O_8)	(Tonnes)	(Mlbs)	(Mt)	(ppm U_3O_8)	(Tonnes)	(Mlbs)	(Mt)	(ppm U_3O_8)	(Tonnes)	(Mlbs)
Etango	62.7	205	12,900	28.3	273.5	200	54,600	120.4	45.7	202	9,200	20.3
Ondjamba	-	-	-	-	-	-	-	-	85.1	166	14,200	31.3
Hyena	-	-	-	-	-	-	-	-	33.6	166	5,600	12.3
Total	**62.7**	**205**	**12,900**	**28.3**	**273.5**	**200**	**54,600**	**120.4**	**164.6**	**176**	**29,000**	**63.9**

Table 14-14 Etango Project Mineral Resource Estimate October 2010 – Total Estimate Reported at a Range of Cut-off Grades												
	Measured Resources				Indicated Resources				Inferred Resources			
Cut-off Grade	Tonnes	Grade	Contained U_3O_8		Tonnes	Grade	Contained U_3O_8		Tonnes	Grade	Contained U_3O_8	
(ppm U_3O_8)	(Mt)	(ppm U_3O_8)	(Tonnes)	(Mlbs)	(Mt)	(ppm U_3O_8)	(Tonnes)	(Mlbs)	(Mt)	(Mt)	(ppm U_3O_8)	(Tonnes)
100	**62.7**	**205**	**12,900**	**28.3**	**273.5**	**200**	**54,600**	**120.4**	**164.6**	**176**	**29,000**	**63.9**
125	56.6	215	12,200	26.8	238.6	212	50,700	111.7	143.9	185	26,600	58.6
150	47.5	230	10,900	24.0	193.7	230	44,500	98.1	106.1	201	21,400	47.1

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 DFS that was completed in May 2012.

The DFS was based on an update of the Etango Deposit Mineral Resources as of October 2010.

The DFS 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, 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.

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, 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	Coffey Mining
Metallurgical Aspects	AMEC, Bateman Engineering
Processing Cost	AMEC, Bateman Engineering
Tailings Storage Facility	Coffey Mining
Commodity Price	Bannerman
General and Administration Cost	AMEC, Bannerman
Social and Environmental	Bannerman, A. Speiser Environmental Consultants
Mine Closure Cost	Bannerman
Government	Bannerman
Hydrology and Hydrogeology	Aquaterra
Geotechnical	Coffey Mining
Site Water Balance	SLR/Metago
Mining Dilution and Recovery	Coffey Mining
Discount Rate	Bannerman

Unless otherwise stated all costs are quoted in US$.

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	%	5
Processing Cost (inclusive of General & Administration)	$/t ore	7.86
Processing Recovery	%	87
Average Mining Cost	$/t mined	1.97
Mining Dilution	%	3%
Mining Recovery	%	97%
Overall Pit Wall Slope Angle (inclusive of a ramp system)	degrees	43 – 48
Initial Project Capital	M$	870.3
Sustaining Capital	M$	348.4
Closure Costs	M$	32.5
Discount Rate	%	8

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 70ppm U_3O_8 and was determined as of 10 April 2012. This reserve remains current as of 24 May 2012.

All stated Mineral Reserves are completely included within the Mineral Resources as shown in Table 14-10.

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)	Contained U_3O_8 (tonnes)	Contained U_3O_8 (Mlbs)
Proven	64.2	194	12,455	27.5
Probable	215.3	193	41,553	91.6
Total	**279.5**	**194**	**54,223**	**119.54**

The reported Mineral Reserves have been compiled by Mr Harry Warries. Mr Warries is a Fellow of the Australasian Institute of Mining and Metallurgy and an employee of Coffey Mining Pty Ltd. 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 2004 ('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 mineral reserve estimates. These factors include, but are not limited to, environmental, permitting, legal, title tax, socio-economical, marketing and political, economical 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 most significantly are listed below:

15.5.1 Uranium Price

The current long term contract price for U_3O_8 is around \$60/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 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

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 Coffey Mining that, excepting the parameters discussed above, there are no other factors that may materially affect the mineral reserve estimates.

16 MINING METHODS

16.1 Introduction

The mining study that was undertaken as part of the DFS covered the following aspects:

- Geotechnical and hydrological assessment
- Open pit bench height and dilution (incorporating truck scanning) study
- Equipment selection
- Mine planning: optimisation, final pit design, pit staging, dump design and mine production scheduling
- 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 October 2010 Ordinary Kriged block model (OK 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 84.5%[5] and a capacity of 20Mtpa
- Q3 2011 market price for:
  - Explosives, fuel, mobile equipment and earthmoving tyres
  - Vendor-provided services for mobile maintenance, 'down hole' explosives and fuel management
- 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 as shown in Table 16-1, 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.

[5] Subsequently increased to 87% for process design and financial modelling

Table 16-1 Etango Uranium Project – Breakdown of Measured and Indicated Resource by Grade Range					
U_3O_8 Grade (ppm)		Plant Feed (Mt)	Average Grade (ppm)	Percent of M&I Within Range	
From	To			Plant Feed Tonnes (%)	Contained Metal (%)
70	100	27.6	86	10	4
100	200	149.8	150	52	41
200	300	81.2	241	28	36
300	+	27.5	366	10	19

To ensure business outcomes, the DFS has focused on a predictable, high performance mine with a conventional approach to mining. The manning strategy of Etango 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 on-board 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 study took dispatch systems to RFQ level 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 ongoing 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 has been completed by Coffey Mining to the DFS level.

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.

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 exceedence 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 factor of safety (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 (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 three stages of mining. Six pits would be exposed during Stage 1; four pits would be expanded during Stage 2 and final wall cuts during Stage 3. This would provide the opportunity to confirm design assumptions and check stability experience from mined faces during Stages 1 and 2.

The recommended pit slope design developed for the Etango project is presented in Table 16-2.

Table 16-2
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 permeabilities 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.

Blasting was estimated to switch from dry conditions to wet conditions at the 176RL.

16.2.2 Bench Height and Dilution Study

A bench height and dilution study was undertaken by as part of the inputs into the work for the DFS. The aim of the study was twofold. The first object was to understand (at the resolution presented within the OK Model) the impact of bench height selection on dilution and ore loss. The second object was to develop a probabilistic model that quantitatively accounted for radiometric scanners.

The study employed traditional methods of re-blocking the model to a common selective mining unit (SMU) to account for the equipment size selected. The OK estimation was undertaken with a parent support size of 25m north-south by 25m east-west by 10m vertical. To increase the resolution of boundaries (between mineralised and unmineralised materials), the ordinary kriged estimation was sub-blocked to 6.25m north-south by 6.25m east-west by 1.25m vertical. The study tested various configurations of bench height to understand the impact on dilution and loss, with results in Table 16-3.

Table 16-3
Etango Uranium Project – Bench Level Dilution with a Comparison to Resource
(at 70ppm U_3O_8 Reported as Inside the PFSU Pit Design)

Mining Model	Tonnes (Mt)	Grade (ppm)	Contained Metal (Mlb)	Resource		
				Tonnes (Mt)	Grade (ppm)	Contained Metal (Mlb)
4m bench	301.6	187	124.5	291.7	199	128.3
5m bench	303.6	185	123.8			
6m bench	306.2	182	122.6			
4m and 8m bench combination	305.9	180	121.4			
5m and 10m bench combination	306.7	178	120.4			
6m and 12m bench combination	307.9	176	119.2			
8m bench	309.8	177	120.6			
10m bench	313.6	172	118.9			
12m bench	316.5	168	116.9			

The second objective was to determine the final tonnes and grade to be presented to process given radiometric scanning. The process preserves the parcel tonnes and grade from the OK resource model. For blocks impacted by edge dilution (18% of the ore blocks), tonnes and grade of the load were calculated based on the probability of the bucket load being ore or waste. Loads of ore and waste were accumulated for each block from this calculation. These accumulated tonnes of ore and waste form the bases of the mine planning model.

The conclusion of the bench height and dilution study shows that an excavation in 4m or 5m cuts provide the greatest economic value to the project.

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 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. Further outcomes from the dilution study show 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 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 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 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.

Section 16.3.3 provides detail of the selected mining fleet and number required for the production schedule.

16.3 Mine Planning

Mine planning covers the optimisation, pit design, dump design, and mine production schedule of the open pit optimisation and shell selection.

Table 16-4 summarises the inputs into the pit optimisation.

Table 16-4
Inputs into the Optimisation

Item	Unit	Value
Plant throughput	Mtpa	20
Uranium price	$/lb	70
Royalty	%	5.0
Transport, shipping, penalties, marketing and sales	$/lb	1.20
Processing and General & Administration costs	$/t ore	6.83
Average mining cost	$/t ore	1.91
Processing recovery	%	84.5
Overall pit wall slope angle (inclusive of a ramp system)	Degrees	43 to 48

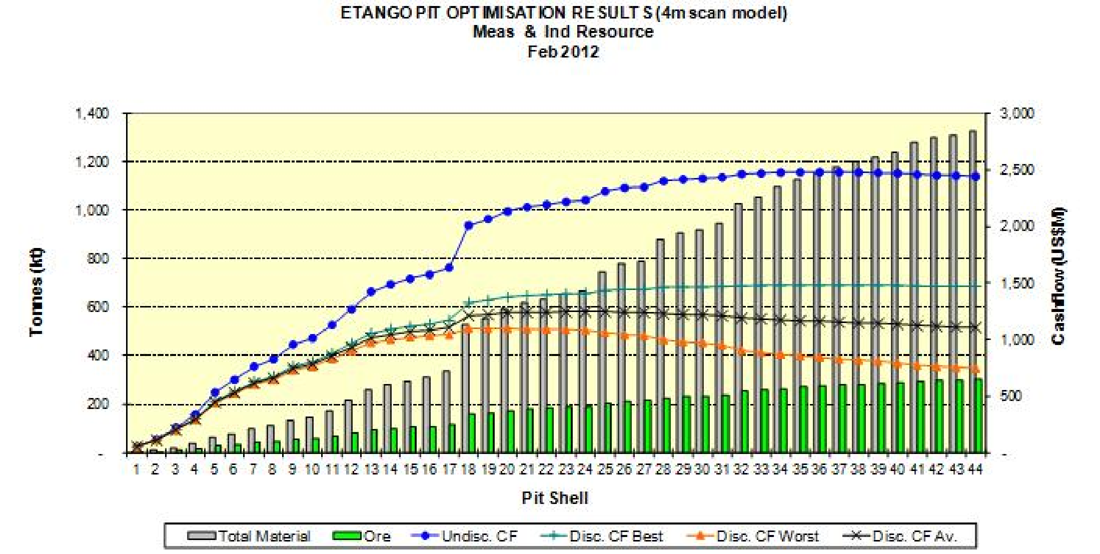
The 4m bench height, diluted resource model adjusted to reflect the result expected from the use of a truck scanner, as supplied by AMC and described in Section 16.2, formed the basis of the pit optimisations. Two pit optimisations were carried out with the first based on the Total Resource, including Inferred Resources and the second based on Measured and Indicated Resources only.

The results from the Measured and Indicated Mineral Resource optimisation are summarised in Table 16-5 and Figure 16-1.

Table 16-5
Etango Uranium Project – Summary of Optimisation Shells Used in Miliwa Generated Schedule – Measured and Indicated Resource Only

Shell	Total Material	Waste	Strip Ratio	Plant Feed			Cash Flow ($M)			
				Tonnes (in-situ)	U_3O_8 Grade	U_3O_8 Output	Undisc. CF	Best	Worst	Avg
	(Mt)	(Mt)	(w:o)	(Mt)	(ppm)	(lb x 1000)	($M)			
7	140	100	1.5	60	240.2	18,062	763	628	603	616
17	337	221	1.9	116.4	210.5	45,355	1,636	1,169	1,041	1,105
36	1,151	876	3.2	274.7	193.8	98,585	2,479	1,478	842	1,160

Figure 16-1
Etango Uranium Project – Summary Pit Optimisation Results Using Measured and Indicated Resources Only



The optimisation results were smoothed without ramps, then taken through a series of Miliwa Balanced (Whittle's scheduling routine) schedules to nominate a series of pit shells for design. Miliwa Balanced runs suggest that Pits 7, 17, and 36 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 16-5.

16.3.1 Pit Design

Figure 16-2 shows the final pit design.

Figure 16-2
Etango Uranium Project – Final Pit Design

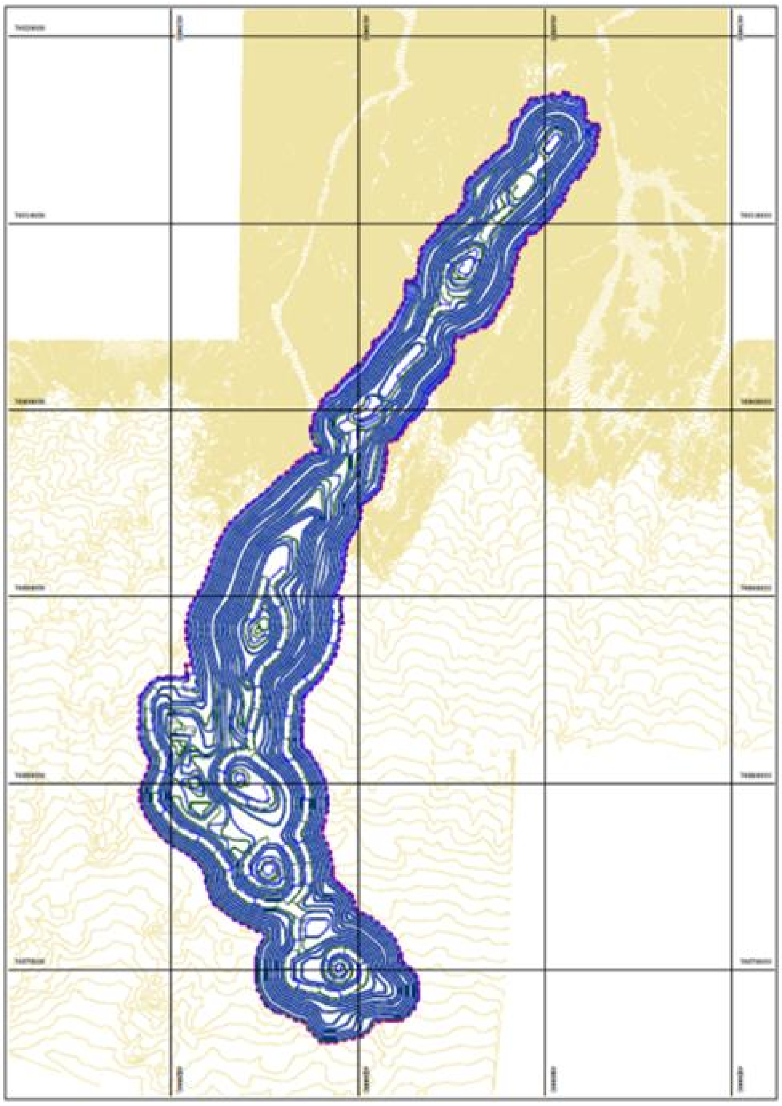


Table 16-6 provides a summary of the design criteria used for the detailed pit design work.

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.

Shells 7, 17, and 36 were used as a guide for the design of Stages 1, 2 and 3 (LOM pit). The material inventory is shown in Table 16-7.

Figure 16-3 provides an overview of the three pit stages that were developed.

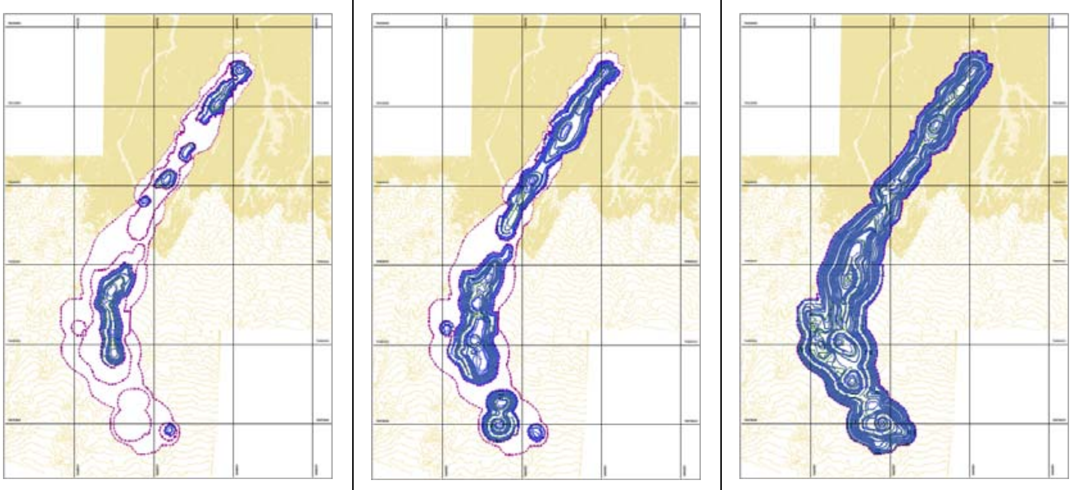
Table 16-6 Pit Design Specifications			
Item		**Unit (m)**	**Value**
Minimum Working Width		**m**	**40**
Bench Height	Weathered Rock Fresh Rock	m	12 24
Batter Angle	Weathered Rock Fresh Rock	deg.	55 70
Berm Width	Weathered Rock Fresh Rock	m	6 9.5
Decoupling Bench		m/vertical m	15m every 150m
Total Width	Dual Carriage Way Single Carriage Way Trough Ramp (Drop Cut)	m	32 17 30
Running Width	Dual Carriage Way Single Carriage Way Trough Ramp (Drop Cut)	m	25 10 28

Table 16-7
Staged Design Material Inventory

Stage	Ore		Waste	Total Material
	Tonnes (Mt)	Grade (ppm)	(Mt)	(Mt)
Stage 1	31.5	237	52.6	84.1
Stage 2	85.4	199	215.0	300.4
Stage 3	162.7	182	667.3	830.0
Total	**279.6**	**194**	**935.0**	**1,214.6**

Figure 16-3
Etango Uranium Project – Stages 1, 2 and 3 Designs

Stage 1	Stage 2	Stage 3



16.3.2 Dump Design

The waste dump landform is based on meeting a set of ten criteria, including:

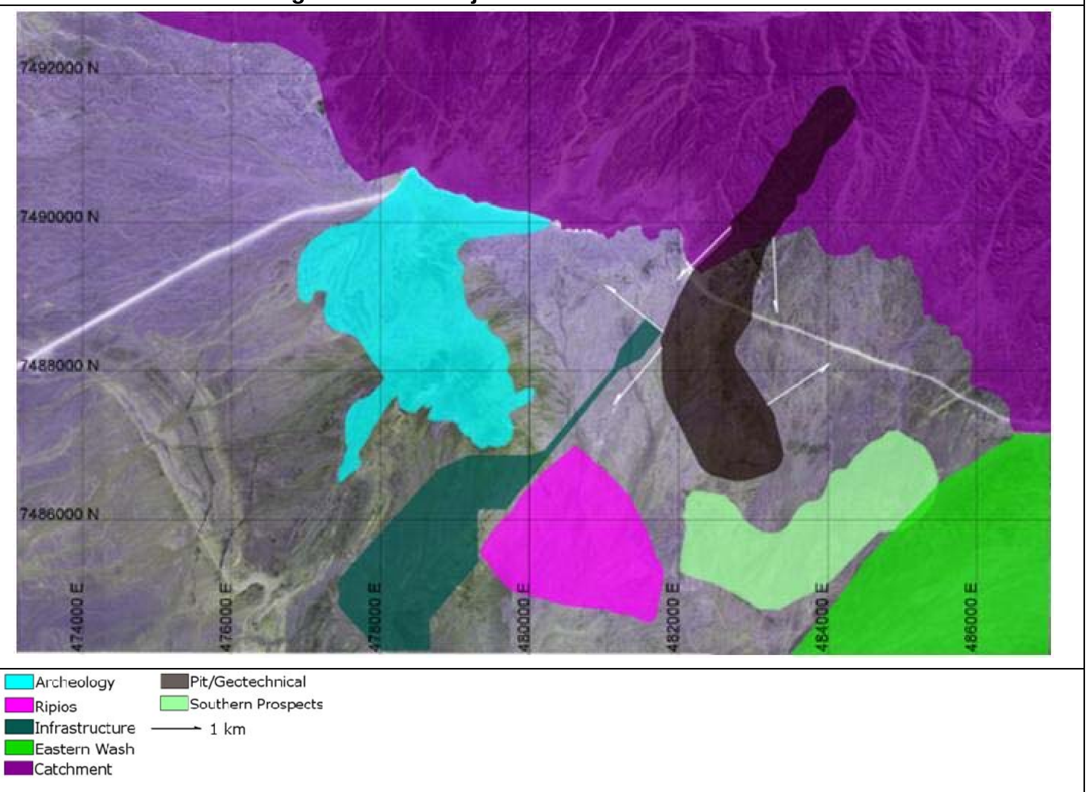
- Geotechnical
- Geochemical
- Consideration for the land character
- Surface water and catchments
- Ground water
- Archaeology
- Visual
- Topsoil requirements
- Vegetation
- Other infrastructure needs.

After taking each into account, there is sufficient land available for waste tipping.

Land available for dumping and the final landform (based on minimising waste costs) is displayed in Figure 16-4.

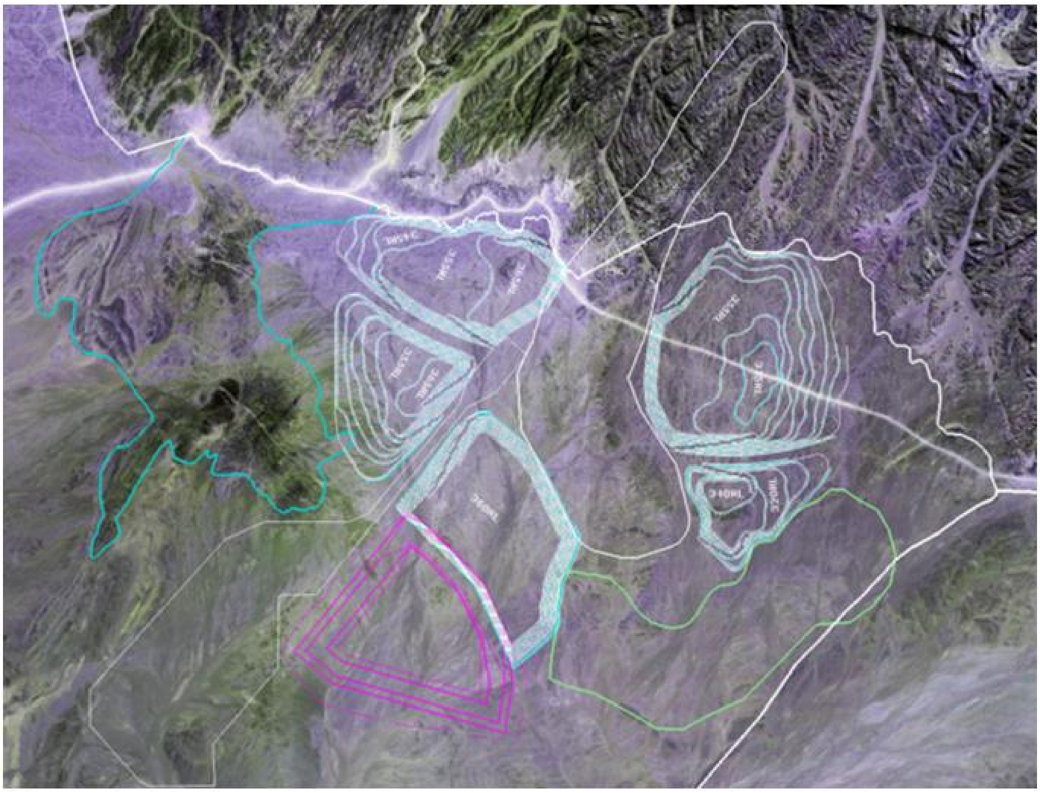
Figure 16-4

Etango Uranium Project – Available Land for Waste



The waste rock at Etango 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 16-5).

Figure 16-5
Etango Uranium Project – Final Dump Design



16.3.3 Mine Production Schedule

Material was scheduled by stage and by bench and a physicals schedule was developed in Microsoft Excel.

The constraints set for the schedule were:

- Crusher feed rate of 20Mtpa
- Defer waste movement
- Maximum vertical advance rate of eight benches per stage per annum
- Maximum total material movement (ex-pit) of 100Mtpa.

A summary of the mine production schedule, along with the crusher feed scenario, is provided in Table 16-8. Rehandle and stockpiling is based on an average volume of 40% of the ex-pit feed. Equipment required to achieve the production schedule is provided as Table 16-9.

Table 16-8

Etango Uranium Project – Summary Mine Production Schedule

Model	Q1 Y1	Q2 Y1	Q3 Y1	Q4 Y1	Q1 Y2	Q2 Y2	Q3 Y2	Q4 Y2	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)	8,408	10,912	10,436	10,228	15,079	14,946	20,015	20,037	100,335	100,016	100,457	100,307	99,703	100,290	100,290	100,488	90,239	74,637	60,439	60,209	15,955	1,125	**1,214,549**
Plant Feed Tonnes (kt)	853	3,165	2,951	3,094	5,020	4,972	5,000	5,000	19,985	20,013	19,983	20,038	19,967	20,037	20,000	18,842	20,342	20,497	19,539	22,321	7,322	647	**279,587**
Grade (ppm)	183	197	197	217	235	234	245	261	207	189	200	196	181	151	171	170	172	191	196	210	268	311	**194**
In-situ Metal – (klb)	344	1,375	1,282	1,482	2,597	2,562	2,697	2,872	9,129	8,328	8,801	8,638	7,978	6,688	7,558	7,060	7,729	8,620	8,433	10,328	4,332	444	**119,275**
Direct Tip Crusher Feed(kt)	512	1,899	1,771	1,856	3,012	2,983	3,000	3,000	11,991	12,008	11,990	12,023	11,980	12,022	12,000	11,305	12,205	12,298	11,723	13,393	4,393	388	**167,753**
Direct Tip Grade(ppm)	183	197	197	217	235	234	245	261	207	189	200	196	181	151	171	170	172	191	196	210	268	311	**194**
Direct Tip In-situ Metal klb	206	825	769	889	1,558	1,537	1,618	1,723	5,478	4,997	5,281	5,183	4,787	4,013	4,535	4,236	4,637	5,172	5,060	6,197	2,599	266	**71,565**
Rehandle Live (kt)	288	101	1,229	1,144	1,988	1,989	2,000	2,000	7,994	5,492	7,993	7,977	7,987	5,478	8,000	7,537	7,795	7,702	7,816	6,607	2,929	260	**102,305**
Rehandle Live Grade(ppm)	183	197	197	217	235	234	245	261	207	189	200	196	181	151	171	170	172	191	196	210	268	311	**194**
Rehandle Live (klb)	116	44	534	548	1,028	1,025	1,079	1,149	3,652	2,285	3,521	3,439	3,191	1,829	3,023	2,824	2,962	3,239	3,373	3,057	1,733	178	**43,827**
Plant Feed (kt)	**800**	**2,000**	**3,000**	**3,000**	**5,000**	**5,000**	**5,000**	**5,000**	**20,000**	**20,000**	**20,000**	**20,000**	**20,000**	**20,000**	**20,000**	**20,000**	**20,000**	**20,000**	**20,000**	**20,000**	**10,140**	**648**	**279,587**
Plant Feed Grade (ppm)	**183**	**197**	**197**	**217**	**235**	**234**	**245**	**261**	**207**	**190**	**200**	**196**	**181**	**155**	**171**	**167**	**172**	**191**	**195**	**210**	**250**	**311**	**194**
Plant Feed In-situ metal **(klb)**	**322**	**869**	**1,303**	**1,437**	**2,586**	**2,574**	**2,697**	**2,872**	**9,136**	**8,376**	**8,808**	**8,622**	**7,991**	**6,830**	**7,558**	**7,376**	**7,599**	**8,412**	**8,611**	**9,254**	**5,598**	**445**	**119,275**
Closing Stockpile (kt)	53	1,218	1,170	1,264	1,284	1,255	1,255	1,256	1,240	1,253	1,236	1,274	1,240	1,277	1,277	119	461	957	496	2,818			
Closing Stockpile Grade	183	196	196	198	198	198	198	198	198	179	179	179	179	124	124	124	160	176	176	204			
Closing Stockpile klb	21	527	506	551	562	549	549	549	543	494	488	504	491	348	348	33	162	371	192	1,267			

Table 16-9

Etango Uranium Project – Major Equipment Requirements for the Revised 3 Stage Pit Schedule

Equipment Type	**Peak**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Year 12**	**Year 13**	**Year 14**	**Year 15**	**Year 16**
Drill	18	6	11	15	15	15	16	17	18	16	17	14	12	9	9	3	2
Excavator	6	2	5	6	6	6	6	6	6	6	6	6	5	5	4	1	1
Haul truck	39	10	16	25	26	30	32	34	36	38	39	38	35	32	33	11	9
Wheel loader	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Rubber tyre dozer	3	1	2	3	3	3	3	3	3	3	3	3	2	2	2	1	1
Tracked dozer	6	3	5	6	6	6	6	6	6	6	6	6	5	5	4	2	2
Grader	8	3	4	5	5	6	6	7	7	7	8	7	7	6	7	3	2
Water truck	4	2	2	3	3	3	3	3	4	4	4	4	4	3	3	2	2
Total	**87**	**29**	**47**	**65**	**66**	**71**	**74**	**78**	**82**	**82**	**85**	**80**	**72**	**64**	**64**	**25**	**21**

16.4 Mine Capital and Operating Costs

Mining Capital and operating costs are set out in Section 21.

17 RECOVERY METHODS

17.1 Overview

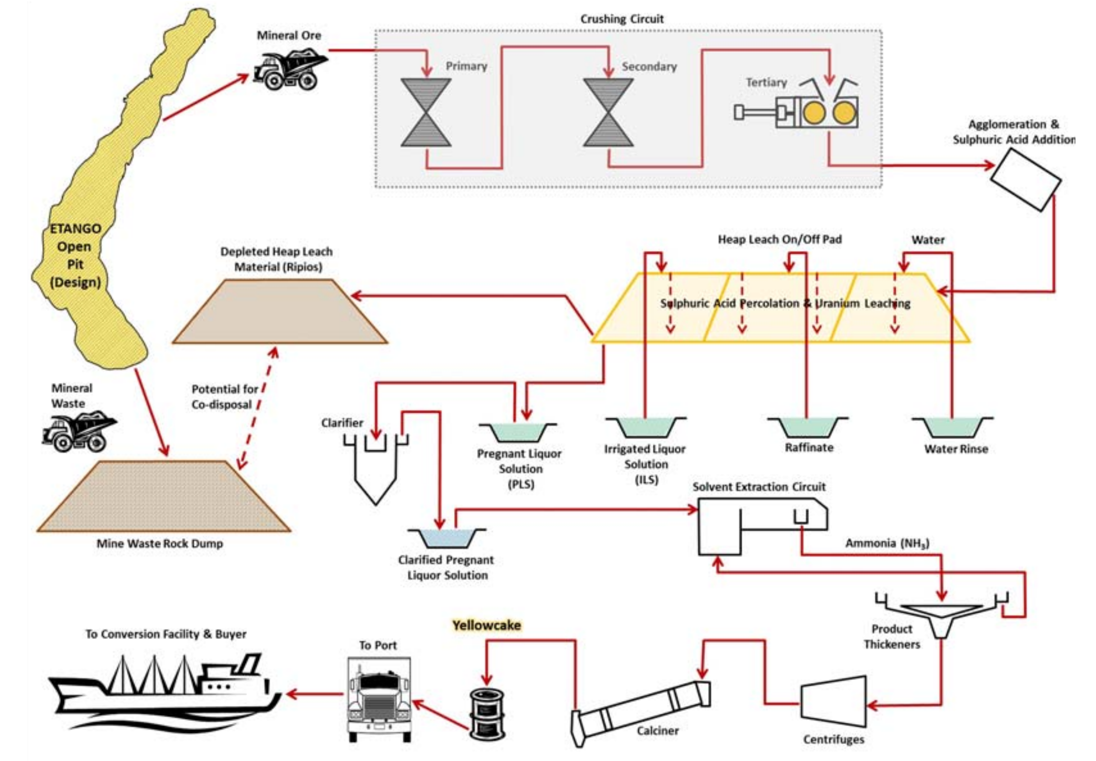
Fundamental process design criteria have been determined based on testwork as described in Section 13. Key parameters include:

- Heap leach crush size: P_{80} 5.3mm
- Leach duration: 30 days
- U_3O_8 recovery: 86.9%
- Acid consumption: 17.6 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 Flowsheet



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 20/17-8, each fitted with two motors of 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 10mm apertures to provide the target P_{80} product size of 5.3mm.

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 drive-in 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. 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. 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 Heap Leach Residue (Ripios) Stacking

Ripios is transferred from the heap leach reclaiming overland conveyors to the Ripios 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.

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 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 1m thick coarse (25-40mm) drainage layer. The drainage layer protects 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 is 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.

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. Sulphuric acid is added to the PLS 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.

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^-, and 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.

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 and a water-cooled, finned type cooler. The pressure in the line after the cooler is maintained at 45kPag, by means of a breaker valve.

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).

The scrubbed organic is transferred 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 and the settlers. 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.

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.

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.

Ammonia is added, 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 underflow slurry (50% w/w) is pumped to the next processing circuit. Overflow (barren strip) flows into an overflow tank from where it is pumped to the ADU polishing filter, 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 a 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 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.

<u>Uranium Packing Plant</u>

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.

<u>Off-gas and Dust Scrubbing</u>

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:

- Sulphuric acid
- Hdrogen peroxide
- Diluent – Shellsol 2325
- 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[6] 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

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 four 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.

[6] Except Magnafloc 351 – 7 days storage

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, SX and precipitation areas.

The sulphuric acid unloading and storage area is suitably bunded and serviced by a sump and sump pump, transfering any spillage and washdown 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.

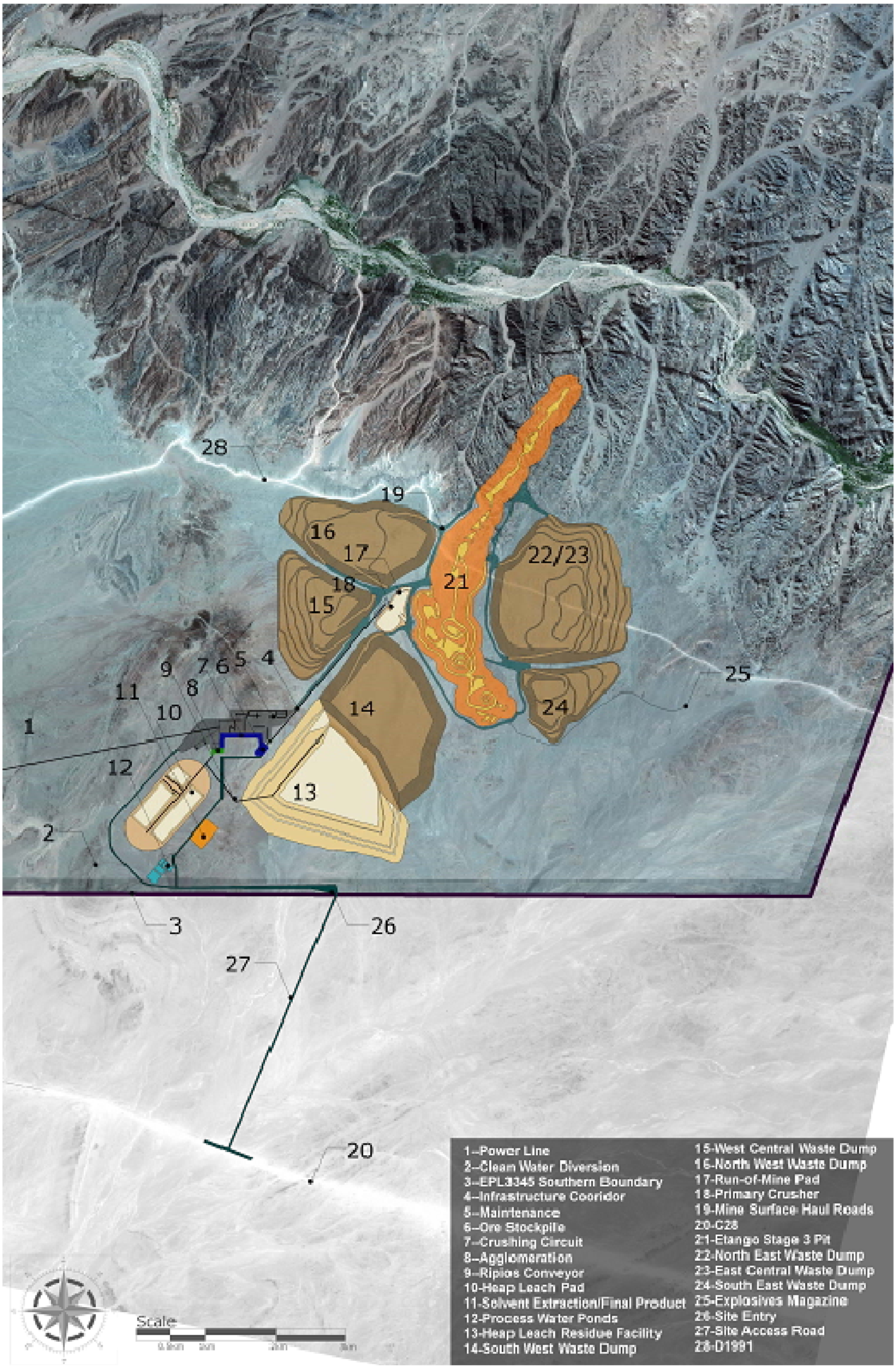
The site layout takes account of a number of factors, including a requirement to remain within current licence area.

The layout takes 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 are minimisation of visual impact within National Park and reduction of effects on the Welwitschia plant locations.

With these restrictions, the waste rock dumps are sited adjacent to the open pit to minimise haulage costs which are the largest single component of the operation. This leads to the coarse ore stockpile and process plant being located 3km from the coarse crusher, linked by an overland conveyor.

Figure 17-2

Etango Site Layout



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
- Location of significant structural loads (HPGRs, cone crushers and vibrating screens) outside of the palaeo-channel
- The heap leach pads are 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 drain by gravity to the heap leach ponds
- Water storage is located adjacent to the SX ponds close to the mine lease boundary.

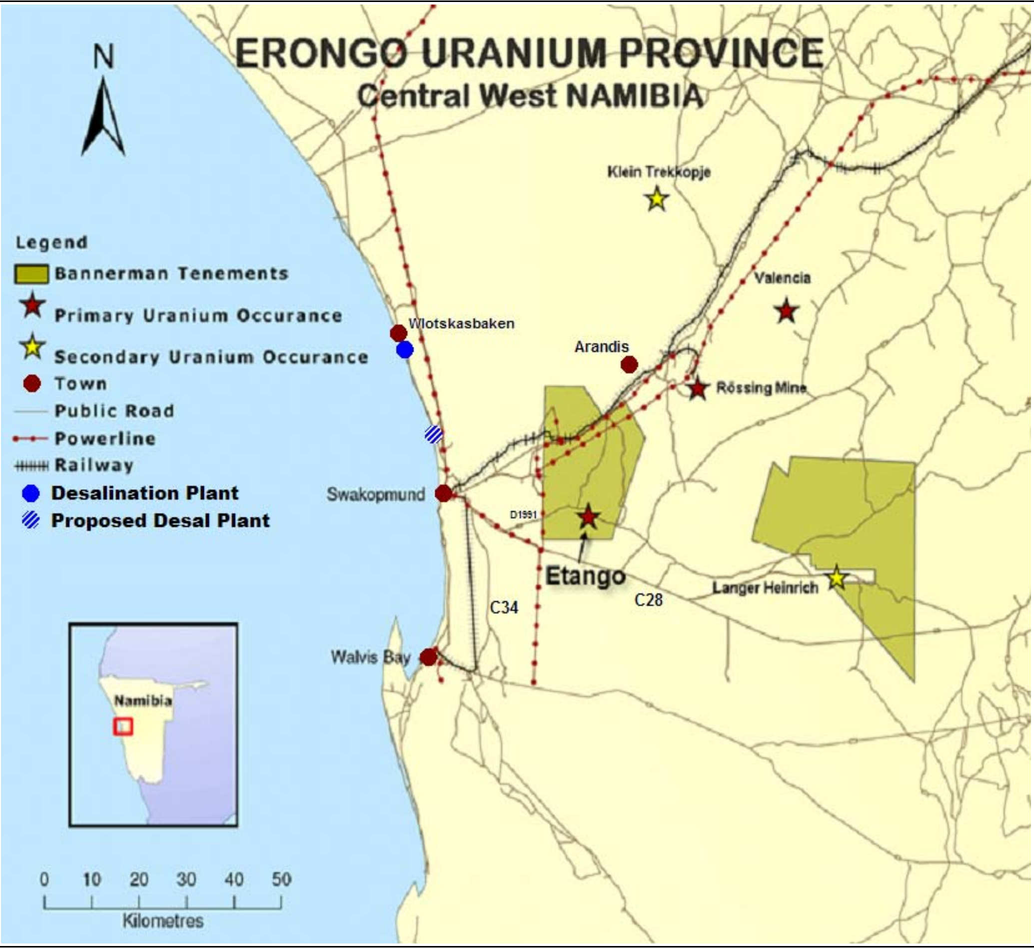
The Ripios Storage facility is located at the extremity of the final waste rock dump profile, adjacent to the heap leach pad at the southern end of the mining waste dumps. This area suits the radial stacking arrangement.

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, namely water, air, diesel and sulphuric acid, as described in Section 17.5.

18.2 Power

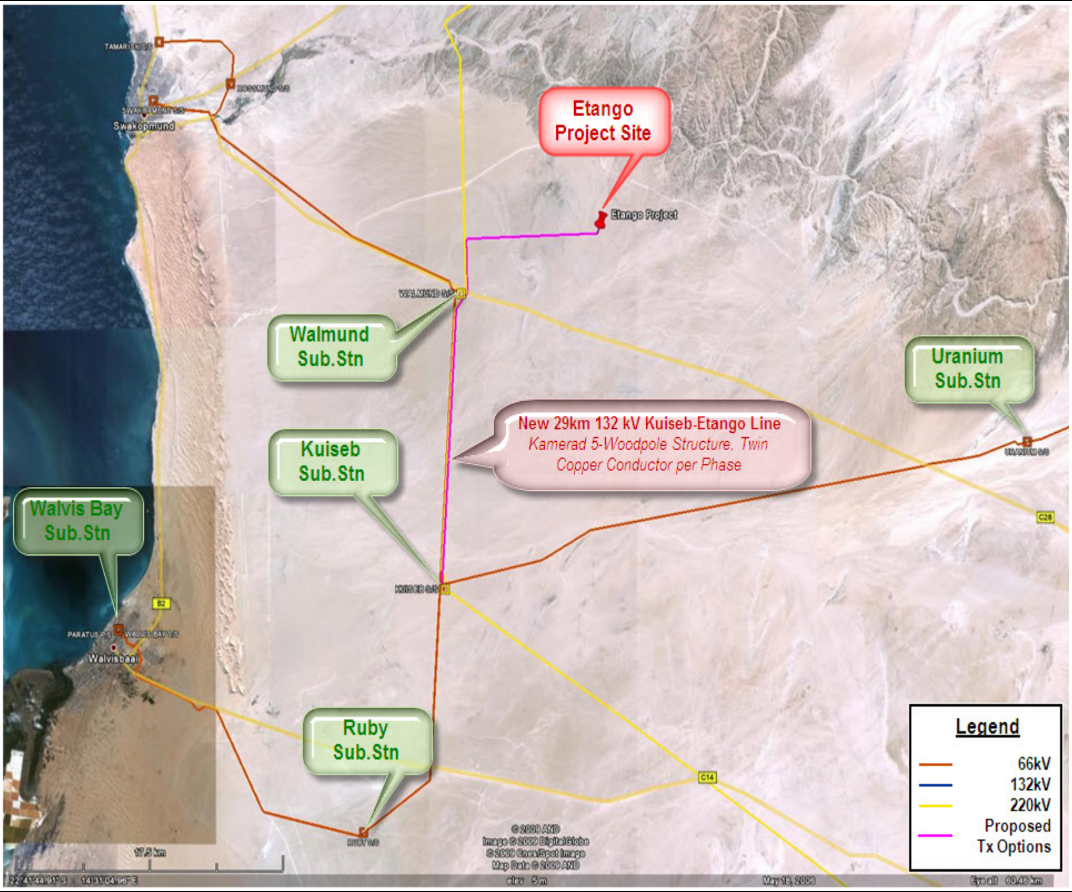
18.2.1 Project Power Demand and Supply

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 220 kV national grid through its substation located at Kuiseb (Figure 18-2), which is to planned 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 Project site where a 132/33kV switchyard, transformer(s) and 40MVA indoor 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 is expected to involve the capital cost of the power line being paid by Bannerman, and a schedule of rates and payments.

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 580MW (June 2011), although this is reduced in the dry season when the Ruacana hydro-electric power capability is reduced significantly.

Peak power demand in Namibia (2009) was 550MW, although this figure is out-dated. In the past Namibia has imported up to 60% of its power requirements from South Africa and other neighbouring countries, but increased demand within South Africa has limited available power in the region.

The proposed development of new uranium mines, a desalination plant and the expansion plans of existing facilities are likely to increase demand by an estimated 300MW in the Erongo region alone.

NamPower predicts a shortfall of as much as 300MW by 2015, and is considering a number of alternatives to increase power generation capacity, including:

- Combined cycle gas-fired power station (Kudu Gas) – 400MW to 800MW (earliest 2016)
- Coal-fired power station at Walvis Bay – 400MW
- Diesel peaking station at Walvis Bay – 50MW
- Lower Orange River small hydropower stations – 108MW
- Baynes Hydropower Station – 360MW to 550MW (50-50 split between Namibia and Angola).

The need to increase base-load power supplies is clearly recognised by NamPower. None of the above 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 imported supply tariffs.

Power Pricing

An electricity price of $0.0975/kWh has been used in the DFS, this being the price current in 2011.

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.

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 constructed and operated for the construction phase. Domestic water effluent is 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. When fully developed, Trekkopje is expected to use 10-12 Mm^3/a, but the remainder of the output is likely to go to Rössing, Langer Heinrich and the proposed Husab project.

NamWater is proposing to establish a new Sea Water Reverse Osmosis (SWRO) desalination plant north of the Swakopmund municipal area, on the Atlantic coastline (Figure 18-1). The plant is planned to produce approximately 25Mm^3/a of potable water with a lifespan of 20 years. The proposed project incudes a tie-in to the existing Omdel-Swakopmund pipeline and a new 44km long distribution line.

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 $2.74/m^3 used in the DFS is based on discussions between Bannerman and NamWater, and reflects the estimated cost of desalination and delivery 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 is to be constructed to link the existing road to the Etango site (Figure 18-3).

The road crosses an existing tenement held by Reptile. A letter of 'in principle agreement' has been received from Reptile, 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 1000 vessel calls each year and handling about 2.5Mt 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 150,000 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 Rössing Mine, but with capacity available to Bannerman.
- 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 ad hoc 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. Details have yet to be developed in conjunction with local authorities.

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.

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 35 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 for the DFS have been provided by nuclear fuel transport operators.

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, sampling and assaying of the yellow cake, and the terms for storage, as well as the details of surcharges for impurities. There is typically a free storage period with additional charges for longer term storage.

Testwork 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 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.

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. 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.

19.3.2 Sales and Marketing Costs

Estimated sales-related costs incorporated in the DFS total \$1.10/lb U_3O_8, comprising freight, shipping, insurance, sales and marketing and an allowance for conversion impurities.

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 2011 was approximately 162Mlb U_3O_8 and total uranium production was approximately 140Mlb U_3O_8. Total uranium consumption is expected to grow in 2012 to approximately 177Mlb U_3O_8.

The supply deficit is presently filled from secondary supplies including the sale of US Government inventories and the down-blending of highly enriched uranium (HEU) from nuclear weapons as part of the 1993 US / Russian 'Megatons to Megawatts' program.

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, in 2012, it is now emerging that although a minority of nuclear power generating countries may seek reductions or deferrals of their own nuclear programs, the clean nature of nuclear power for base load electricity generation 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

None of Japan's existing 51 nuclear reactors are currently operating, placing considerable stress on Japan's trade balance due to significantly increased imports for its fossil fuel power generation facilities. Japan's Trade Minister recently approved the re-start of two reactors in western Japan and approvals from the local authorities are being sought. The re-starting of Japan's reactors, in particular for the northern hemisphere summer months, represents a key short term catalyst for the uranium market;

Commitment to Nuclear Power

Various nations have in recent months confirmed their commitment to nuclear power. In particular, the National Energy Administration of China forecast that China's nuclear energy capacity will increase to 80MkW by 2020, compared with approximately 12MkW currently. India has also announced that it is seeking to increase its nuclear power capacity from the current installed capacity of approximately 4MkW to 63MkW by 2032. Other nations to reaffirm their commitment to nuclear power include the United Arab Emirates, Saudi Arabia and the USA where the Nuclear Regulatory Commission recently issued construction and operating licences for two more nuclear reactors;

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 is due to end in 2013 and is unlikely to be renewed at its present volumes.

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 \$52/lb and \$60/lb respectively, at the end of 2011.

Various banking institutions and broking firms prepare periodic forecasts of future uranium spot and long term contract prices, which have been used by Bannerman in establishing its price expectations.

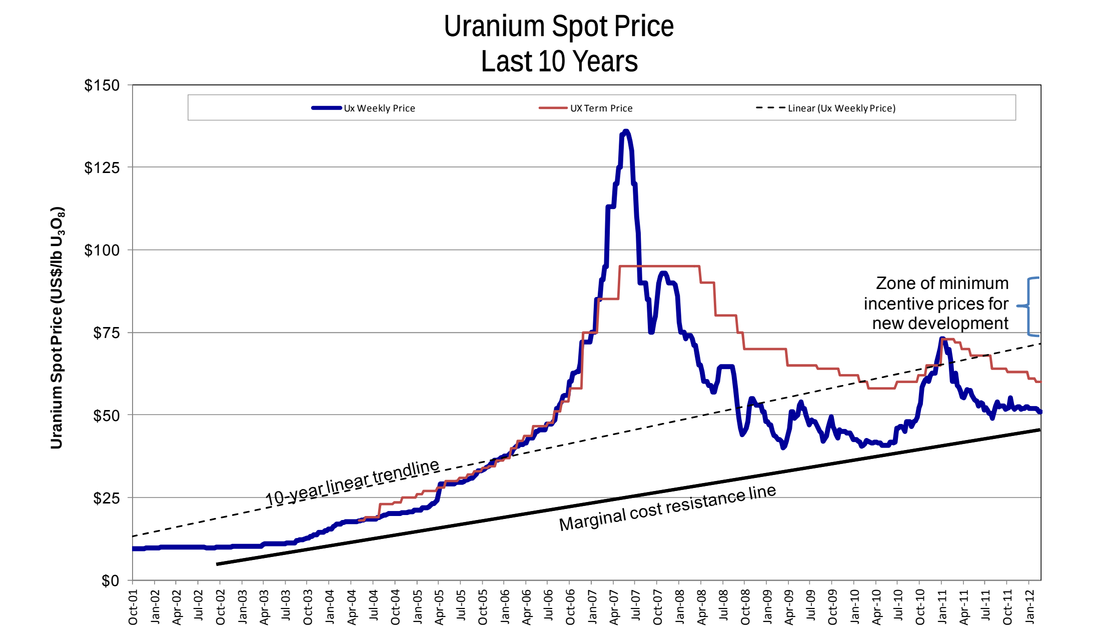
Forecast spot prices from the above sources over the next 5 years presently range from approximately \$65 up to \$80/lb U_3O_8 and the range of forecast long term contract prices is slightly higher (\$70 to \$90/lb U_3O_8). Historic short and long term prices are shown in Figure 19-1.

The economic assessment within the DFS utilises a base case uranium price, stated in (real) December 2011 dollars, of \$75/lb U_3O_8. Sensitivity analyses have been run at various prices either side of the base case price.

AMEC has reviewed the uranium market data and considers that, while the long-term price assumed in the base case financial analysis is at the high end of current expectations, it falls within the range of predicted prices. The financial analysis confirms that the Etango Project is highly sensitive to uranium price, and a long-term floor price of \$75/lb appears to be a minimum for the Project to be development as an economic proposition.

Figure 19-1

Uranium Pricing Outlook



19.5 Contracts

At this time, Bannerman advises that no contracts exist between it and third parties regarding development of the Project.

Obtaining a Mining Licence over the Project area is the key initial step in Project development, and the latest ESIA of April 2012 has been submitted as part of this process. The DFS will provide additional support in this regard.

The next stage of the Project requires Bannerman board approval and obtaining of finance for Project development, at which point it will become necessary to 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.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

Bannerman received Environmental Clearance in March 2010 for its plans to establish the Etango Project as a 15Mtpa heap leach operation as described in the PFS. The Environmental Clearance was issued, based on the ESIA and ESMP which were developed between October 2007 and December 2009 by ASEC and ERM, with a team of 14 specialists.

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 26 July 2011.

Following further exploration and testwork, Bannerman has:

- Expanded the mine design to include the deposits of Oshiveli and Onkelohas
- Refined and enlarged the heap leach processing operation
- Altered the mine site layout as per Figure 17-2.

A revised ESIA was submitted in April 2012, describing the potential positive and negative impacts that the updated project may have on the physical, biological and social environment.

The main changes in the revised ESIA are:

- Extended specialist studies to include the latest project updates, amendments made to Waste Rock Dump placements and the Ripios Storage Facility
- Incorporation of recommendations from the Uranium Rush Strategic Environmental Assessment (SEA) regarding cumulative impacts to the Erongo Region
- Revised visual, air quality and noise studies incorporating additional project data
- Extended groundwater monitoring and hydrogeological groundwater modelling investigation.

This has improved the understanding of the project's impact on the receiving environment and the findings have been incorporated into the mine design, the ESIA, ESMP, and Radiation Management Plan.

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, 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 three stages. The total rock mined will rise from a nominal 40Mtpa in Year 1 to just over 100Mtpa between Year 3 and Year 10, before declining over the last six years of operations. The final pit will be ~6km in length, 1km wide and, at its deepest, approximately 380m 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 runoff 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.

The Mine Closure Framework will be detailed in the amended ESMP.

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 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 2,500m by 2,000m, with an average stacking height of 60m. 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.

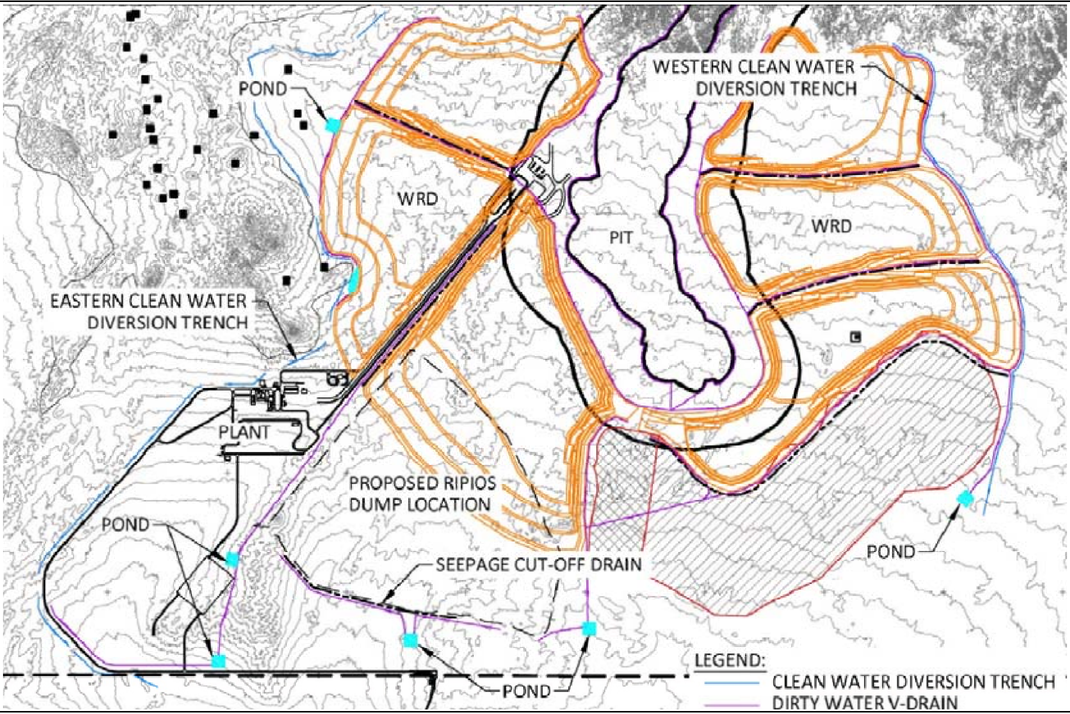
20.2.3 Site Water Management

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 runoff from up-stream catchment areas will be diverted around the site (Figure 20-1).

Figure 20-1

Etango Mine Site Surface and Seepage Water Management



Stormwater 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 runoff 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.

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 proposed desalination plant north of Swakopmund. 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 Desalination plant and pipeline to site are in operation. Peak demand during construction will be 27ML/month during bulk earthworks for compactions and dust suppression.

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 1,500. 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 1,000 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 draft 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.

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 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.

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.
- 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 Biophysical and Human Environment Impacts

A detailed description of the biophysical environment has been developed, along with the background of the human environment. Impacts and mitigation / enhancement measures are included in the ESIA report.

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 20-2.

Table 20-2
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 PM_{10} Health Impacts / EQO 4/2	O	**Major negative**	**Moderate negative**
	cum	**Major negative**	**Major negative**
Potential CO, DPM and SO_2 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**

Table 20-2

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.			
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**
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**

Table 20-2
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.			
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 levels	O	Moderate negative	Moderate negative
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 (1.6mg/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 runoff 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.

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 has been revised to encompass these changes and was submitted in April 2012.

Upon receipt of an updated Environmental Clearance for development of the Etango Project, Bannerman will lodge supplementary information, including the 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 Ripios.

21 CAPITAL AND OPERATING COSTS

21.1 Overview

Capital and operating costs have been determined by:

- Coffey Mining: Mining capital and operating costs, excluding mining infrastructure identified by Coffey Mining but estimated by AMEC.
- AMEC: plant and infrastructure capital and operating costs, excluding SX, precipitation, calcining and packaging plant capital costs for equipment which have been estimated by Bateman. Mining infrastructure requirements, Ripios storage facility and water management system quantities were determined by others but estimated by AMEC.
- Bannerman: Owner's Costs, and power, water and acid prices.

All costs are quoted in US$ as of the 4th quarter 2011, other than Mining which was to 3rd quarter 2011. 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$1.00 = N$ 8.60	Bannerman
US Dollar to South African Rand	US$1.00 = ZAR 8.60	Bannerman
US Dollar to Australian Dollar	US$1.00 = A$ 1.20	Bannerman
US Dollar to Euro	US$1.00 = EUR 0.80	Bannerman
US Dollar to Japanese Yen	US$1.00 = YEN 90.00	Bannerman
US Dollar to Brazilian Real	US$1.00 = REA 1.73	Bannerman

The estimates are considered to have an accuracy of ±15%.

21.2 Capital Costs

21.2.1 Mining Capital Costs

Mining capital cost estimates were estimated primarily by Coffey Mining. Infrastructure cost estimates for buildings including all workshops, wash bays, warehouse, field crib and abolition facilities with the exception of the turn key infrastructure (below) were completed by AMEC, as were costs for services including road network (to the mine but not within the operating limits of the open pit), power, and water supply.

Capital costs of 'turnkey' facilities that are maintained and managed through contract arrangements include the bulk explosives plant and storage facilities, mobile (explosives) manufacturing, explosive magazines, fuel depot, field fuel depot, and lube storage facilities, which were derived from requests for tender (RFQs). These costs appear as operating costs through the fixed monthly fee component of the maintenance and repair contracts (MARC) quotes that were obtained.

The cost of all mining equipment, minor equipment, specialised tooling, and ore definition and geotechnical drilling requirements within the pit were developed by or collated into the mining capital cost estimate by Coffey Mining. Some specific capital items (for example the training costs including simulators) were estimated by Bannerman and added to the capital estimate. Capital costs estimated completed by Coffey Mining for the open pit fall into four categories, namely:

- Cost estimates that were derived from RFQ (91% of the costs)
- Informal budgetary level queries from vendors (3% of the costs)
- In-house cost data (4% of the costs)
- Nominal allowances (3% of the costs).

Mining capital cost estimates include \$126.6M in preproduction capital and \$361.3M in sustaining capital (including a \$25.2M salvage credit at the end of life).

With the exception of the \$50.4M of mining equipment purchased near the end of the mine life, all mobile equipment was considered scrap with zero salvage at the end of its useful life. This equipment (purchased during Years 12 and 13) attracted a \$25.2M salvage value.

Year by year capital requirements are presented in Table 21-2.

Table 21-2
Etango Uranium Project – Mine Capital Cost
(\$M)

Year	Mobile Mining Equipment	Site Infrastructure	Capitalised Operating Costs	Total
-3			1.000	1.000
-2	14.008	3.837	4.248	22.093
-1	76.409	20.927	6.198	103.535
1	92.168	17.301		109.469
2	77.209	4.601		81.810
3	9.720	2.264		11.984
4	20.227	1.394		21.621
5	11.250	2.956		14.206
6	7.177	1.348		8.524
7	23.091	0.448		23.539
8	9.720	3.722		13.442
9	20.378	1.667		22.045
10	3.942	0.736		4.677
11	19.578	2.990		22.568
12	10.236	0.917		11.152
13	40.183	1.094		41.277
14		0.245		0.245
15				
16	-25.209			-25.209
Total	**410.089**	**66.445**	**10.446**	**487.980**

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 where estimated early in the DFS.

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 is modelled one year earlier than required in order to reflect uncertainties in delivery.

RFQ documentation was sent to South African-based equipment vendors 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 US$, with the vendor providing exchange rate assumptions.

RFQ returns were assessed for completeness and competitiveness by Coffey Mining. 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 small mobile equipment such as stemming or tyre maintenance vehicles, are based on Namibian-based dealership quotations or from Coffey Mining's cost database.

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 and lubricants, and explosives. In addition to supply of goods, onsite storage infrastructure was included as part of the RFQ process.

Vendor submissions were assessed for completeness and competitiveness by Coffey Mining. The cost of the infrastructure is included as part of the site infrastructure.

Based on mine plan requirements, surface roads within the mine for the first 3 years of operation were estimated by AMEC. 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 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 section 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.

Site infrastructure required by mining operations includes: offices, ablutions, messing facilities, workshops, wash pad, warehouses, fuel farm, lube storage, explosives plants and magazines. With the exception of the fuel farm, lube storage, explosives plant and magazines, site infrastructure was estimated as part of the AMEC scope of works and is discussed in Section 21.2.2.

Other miscellaneous capital requirements included within the site infrastructure are based on quotations and 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. 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.

<u>Capitalised Mine Operating Costs</u>

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.

The remainder of the capitalised operating costs for the mine include pre-production labour required to oversee the mining start-up in the first two years prior to operations.

21.2.2 Process Plant and Infrastructure

<u>Introduction</u>

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.

Work undertaken by AMEC 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.

Bateman developed the estimate for the SX plant, precipitation, calcination and final packaging components.

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, including power and water supplies (Section 18 and Sub-section 21.2.6)
- 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 (Sub-section 21.2.3).

Estimate Categories

The capital cost estimate for plant and site infrastructure 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.
- Contingency: additional to the base cost estimate to cover unforeseeable elements of cost, risk and uncertainty within the defined scope of work. Excluded from capital cost in financial model.

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 Description	**Total Cost ($M)**
General and General Site Works	36.00
Crushing	146.48
Agglomeration	20.63
Stacking and Reclaiming	60.29
Leach Residue Stacking	30.60
Heap Leach Solution Handling	41.32
Leach Solution Clarification and SX	46.43
Precipitation, Calcination and Packaging	17.40
Reagents	9.38
Water distribution	11.58
Air Distribution	3.37
Diesel and Power Generation	0.45
Sulphuric Acid Handling at Plant Site	9.77
Electrical Distribution Process Plant	12.37
Communications	6.00
Process Controls	2.25
Buildings at Plant Site	26.19
Spares	18.77
First Fills and Opening Stocks	13.51
Temporary Services and Facilities	26.57
Temporary Construction Camp	15.87
Mobilisation and Demobilisation	15.42
Vendor Representatives	5.35
Owner's Costs – Pre-production	2.48
Facilities at Port Site	3.77
Direct Costs – Subtotals	**582.25**
EPCM	72.08
Contingency Allowance	69.88
Total (AMEC Estimate)	**724.21**
Plus	
Additional Mobile Equipment (Bannerman)	1.17
Pre-production Processing Labour (Bannerman)	4.98
Less	
Contingency Allowance	(69.88)
Total Plant and Infrastructure Capital Cost	**660.48**

Some components of the AMEC estimate (Table 21-3) 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's First Fill, Spares, Mobilisation / Demobilisation and Commissioning costs

- Indirect costs in the model comprise AMEC's Temporary Services and Facilities, Temporary Construction Camp, Vendors Representatives, Owner's Pre-production costs and EPCM costs
- AMEC's Accuracy Provision costs have been moved from Direct Costs into Indirects.

Table 21-4 provides a breakdown of the Plant and Infrastructure cost by function, reflecting these changes.

Table 21-4
Capital Cost Estimate – Plant and Site Infrastructure, Summarised by Function

Function Description	Total Cost ($M)
Process Plant	354.44
Site Infrastructure	91.10
External Infrastructure – Port only	3.36
Miscellaneous	44.31
Indirects	113.73
Accuracy Provision	53.53
Total	**660.48**

Estimate Accuracy

The estimate has been prepared in accordance a targeted accuracy of ±15%. 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. Approximately 80% of equipment pricing was from multiple budget price quotes, while all bulk material unit rates were based on prices supplied by reputable fabricators, suppliers and contractors
- 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.

All quantities and equipment costs for the SX and U_3O_8 recovery plant area were produced by Bateman and are not considered in AMEC's assessment of estimate accuracy.

First Fills and Spares

First fill reagents and consumables have been assessed to suit requirements for the 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 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 manhours are based on engineering deliverables while project management, procurement, construction management and commissioning management manhours 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.

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.

EPCM costs for the SX plant area have been provided by Bateman as a lump sum value.

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 $53.53M, representing 10.1% of Direct costs.

Contingency Allowance

An allowance for project contingency was included in the plant and infrastructure estimate, representing 12% of Direct costs or $69.87M (Table 21-3). This allowance is additional to the base cost estimate to cover unforeseeable elements of cost, risk and uncertainty within the defined scope of work, to cover possible increased costs that cannot be explicitly foreseen or described at the time of the estimate. However, Bannerman elected not to include this contingency in the capital cost in the base case financial model.

21.2.3 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 21-5. Costs also include allowances for geotechnical and sterilisation drilling in support of construction, and further metallurgical testwork on site.

No allowance for Owner’s contingency was made in the capital cost estimate in base case financial model.

Estimate Assumptions and Qualifications

Amongst other things, the capital cost estimate assumes:

- 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.

- 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.

Table 21-5
Summary of Owner's Costs

Item	Cost ($M)
Owner's Project Team	9.96
Corporate, Perth	1.80
Owner's Pre-production Staff	7.84
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 Testwork (on site)	0.10
Total	**40.01**

21.2.4 Sustaining Capital Costs

There is limited sustaining capital required for the plant and infrastructure, the majority being required for build-up and replacement of the mining fleet and other mobile equipment (Table 16-9).

A sum of $7.3M is recovered from the sale of the construction camp on construction completion, and $12.3M expended for first fill in the capital estimate is recovered at the end of the project life.

21.2.5 Closure and Rehabilitation Capital Costs

A detailed closure plan will be developed at a later date as part of the ESMP. High-level consideration has been given to the closure requirements. Bannerman intends to set aside a total of $32.5M for this purpose.

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.

21.2.6 Total Project Capital Cost

The total Project capital cost estimate as used in the base case financial model is set out in Table 21-6.

Table 21-6
Project Capital Cost Expenditure Summary

Area	Pre-production	Sustaining	Total
Mining	126.63	361.35	487.98
Process Plant	354.44	-	354.44
Site Infrastructure	91.10	5.77	96.87
External Infrastructure - Port	3.36	0.91	4.27
External Infrastructure - other	43.22	-	43.22
Miscellaneous	44.31	(12.29)	32.02
Indirects	113.73	(7.30)	106.43
Accuracy Provision	53.53	-	53.53
Owner's Costs[7]	40.01	32.50	72.51
Owner's Contingency	-	-	-
Total Project	**870.33**	**380.94**	**1,251.27**

This cost incorporates the AMEC plant and infrastructure estimate, excluding the project contingency recommended by AMEC. It also includes the Coffey estimate for mining capital and Bannerman's estimate of Owner's Costs.

It should be noted that the total project capital cost excludes working capital requirements, which are however included in the financial model.

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 (mining) or December 2011.

Exchange rates were provided by Bannerman and are reported in Section 21.1.

Operating costs are estimated to an accuracy of ±15%.

21.3.2 Contributors

Contributors to the operating cost estimate were as follows:

- Mine operating costs: Coffey Mining

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[7] Including rehabilitation costs

- Plant and site infrastructure operating costs: AMEC, including equipment specified by Bateman
- External infrastructure (power and water) costs: Bannerman
- Owner's (G&A) costs: Bannerman
- Acid costs: Bannerman.

21.3.3 Operating Cost Estimate Summary

The estimated annual operating costs are presented in Table 21-7. These costs average \$16.93/t processed or \$45.71/lb U_3O_8 produced. Operating costs in the first 5 years are estimated to be \$16.21/t of ore or \$40.85/lb of U_3O_8 produced, with the peak cost of \$61.52/lb occurring in Year 8, when the ore grade is at its lowest.

Table 21-7
Operating Cost Summary

Area	LOM Operating Cost ($M)	Unit Operating Cost ($/t Processed)
Mining	2,391.74	8.55
Process plant	1,999.71	7.15
Site Infrastructure	2.48	0.01
External Infrastructure	2.88	0.01
Owner's Cost (G&A)	337.44	1.21
Total	**4,734.24**	**16.93**

21.3.4 Mine Operating Costs

<u>Summary</u>

The mining 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.

The LOM operating costs are summarised in Table 21-8.

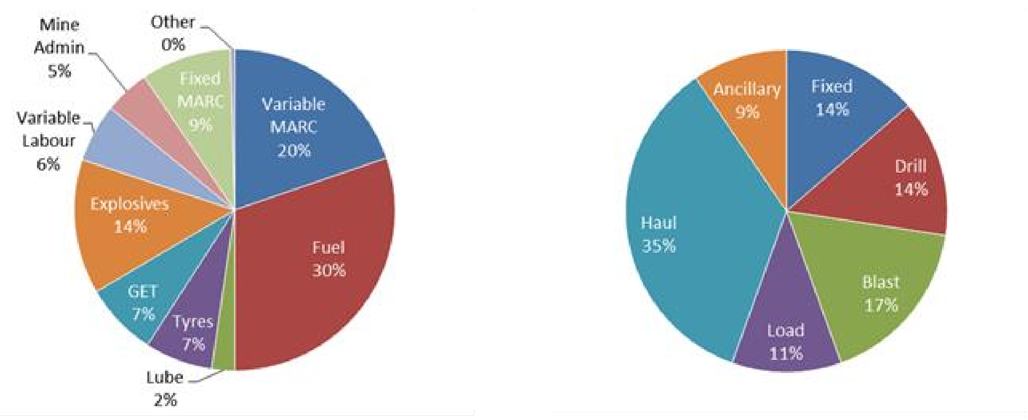
Direct mining fixed operating costs include MARC, supervision and service management fees, as well as some allowances for dewatering, and labour. Grade control has been capitalised.

Fuel, variable MARC, and explosives make up ~65% of the direct expense of the mining costs. A breakdown of the unit cost of mining is provided in Figure 21-1.

Table 21-8 Etango Uranium Project – Mining LOM Operating Costs ($M)							
Category	**Fixed**	**Drill**	**Blast**	**Load**	**Haul**	**Ancillary**	**Total**
Variable Maintenance		55.4		102.1	226.6	88.5	472.5
Fuel	46.2	93.7		114.9	389.4	78.7	722.8
Lube	2.3	9.3		7.6	30.9	5.8	55.9
Tyres					148.3	15.1	163.5
GET		143.3		20.8		12.5	176.5
Explosives			323.9				323.9
Variable Labour		27.6	26.0	18.4	41.0	27.6	140.6
Fixed Labour	108.8						108.8
MARC and Contract Labour	159.2		57.9				217.1
Other	10.2						10.2
Total	**326.7**	**329.2**	**407.8**	**263.8**	**836.1**	**228.1**	**2,391.7**

Unit Cost $/t Mined							
Category	**Fixed**	**Drill**	**Blast**	**Load**	**Haul**	**Ancillary**	**Total**
Variable Maintenance		0.05		0.08	0.19	0.07	0.39
Fuel	0.04	0.08		0.09	0.32	0.06	0.60
Lube	0.00	0.01		0.01	0.03	0.00	0.05
Tyres					0.12	0.01	0.13
GET		0.12		0.02		0.01	0.15
Explosives			0.27				0.27
Variable Labour		0.02	0.02	0.02	0.03	0.02	0.12
Fixed Labour	0.09						0.09
MARC and Contract Labour	0.13		0.05				0.18
Other	0.01						0.01
Total	**0.27**	**0.27**	**0.34**	**0.22**	**0.69**	**0.19**	**1.97**

Figure 21-1
Etango Uranium Project - Mining LOM Operating Breakdown by Expense and Activity



Major Consumables

The fuel price was determined from the Engen and Puma RFQ submissions combined with projections for the long term diesel price. Applicable taxes, tariffs, duties, and bulk purchase discounts were applied. Lubricant costs were part of the RFQ. A diesel price of $0.94/L was adopted for the Study.

Explosives costs were based on modified RFQ costs assuming ammonium nitrate can be bulk imported through the port of Walvis Bay. Bulk explosives prices are based on constituent costs of ammonium nitrate and emulsion costs. An average bulk explosives price of $838/t was adopted for the Study.

Variable maintenance rates for major mobile equipment were obtained through the MARC RFQ process.

Local mining labour and mining administration 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 Goodyear Tyre.

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 Coffey Mining in-house cost database and expected life.

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 Coffey Mining. Powder factors are estimates, based on the results of the Kuz-Ram fragmentation analysis, geotechnical assessment, and crusher requirements. 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 21-9.

Table 21-9
Equipment Costs –$/Engine Hour (Excluding Labour Cost)

Equipment	Total	Variable Maintenance	Fuel	Lube	GET	Tyres	Rebuild
Drill	340	63	105	10	161		0
Excavator	739	305	339	22	61		11
Truck	288	81	132	12		57	7
Wheel loader	299	76	129	12	32	45	5
Tyred dozer	161	49	63	6	17	26	0
Tracked dozer	91	30	51	3	7		0
Grader	60	24	19	2	7	8	0
Water cart	113	40	47	5		19	1

21.3.5 Plant and Infrastructure Operating Costs

Summary Costs

The plant operating costs developed by AMEC are summarised in Table 21-10 and reported as functions of total annual cost and cost per tonne of ore crushed. The table shows that acid costs are the major 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, the price of which was determined from the Engen and Puma RFQ submissions.

The major reagent cost is for sulphuric acid, for which three quotes were received and the lowest quote was selected.

Table 21-10
Summary of Plant and Infrastructure Operating Costs (Average, based on 20Mt/a throughput)

Item	Cost (M$/a)	Cost ($/t of ore)	% of Cost (%)
Acid	35.88	1.79	25.4
Reagents	25.72	1.29	18.2
Power	26.11	1.31	18.5
Labour	12.23	0.61	8.7
Maintenance Materials	18.93	0.95	13.4
Water	12.92	0.65	9.2
Consumables	6.90	0.34	4.9
Miscellaneous	2.43	0.12	1.7
Total	**141.12**	**7.06**	**100.0**

Power Supply

The power consumption is estimated from the installed power values presented in the Mechanical Equipment Lists (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 (9.75 c/kWh), represents NamPower's 2011 cost, supplied by Bannerman.

Approximately 70% of the power requirement and cost lies in the crushing and agglomeration process.

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's 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 $2.74/m^3. This unit cost was provided by Bannerman from discussions with NamWater.

Consumables

Consumables costs and 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 supplied by the vendor and the rolls transport cost quote supplied by Wesbank. HPGR checkplates cost and replacement frequency were supplied by the HPGR vendor. The cost for the heap leach drippers was based on a vendor quote, and considered replacement of the drippers every two cycles. Packaging drum cost was based on the production and on unitary cost per drum based on costs from other projects. The drainage layer replacement cost assumed 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, testwork, maintenance contractors, equipment hire, environmental audits, mobile equipment maintenance and fuel and safety equipment.

External Infrastructure

An annual allowance of $192,000 has been made by Bannerman for National Parks and Road maintenance.

21.3.6 Other Owner's Operating Costs

Owner’s operating costs estimated by Bannerman total $29.52M annually, equivalent to $1.08/t crushed, as summarised in Table 21-11.

Table 21-11
Summary of Owner's Costs

Item	Average Annual Cost ($ M/a)	Unit Cost ($/t of ore LOM)
Corporate and Owner's Labour	12.09	0.673
Total Site Office Administration	0.23	0.013
Total Personnel Expenses	4.22	0.230
Total Insurances and Government Fees	4.25	0.232
Site-Catering Facilities	0.44	0.024
Environmental Monitoring	0.30	0.016
Total Transportation Costs	0.20	0.011
Community Relations / Corporate Responsibility	0.12	0.006
Other	0.04	0.002
Total	**21.85**	**1.207**

Principal costs are for Corporate and Owner’s Labour, Training and Insurances.

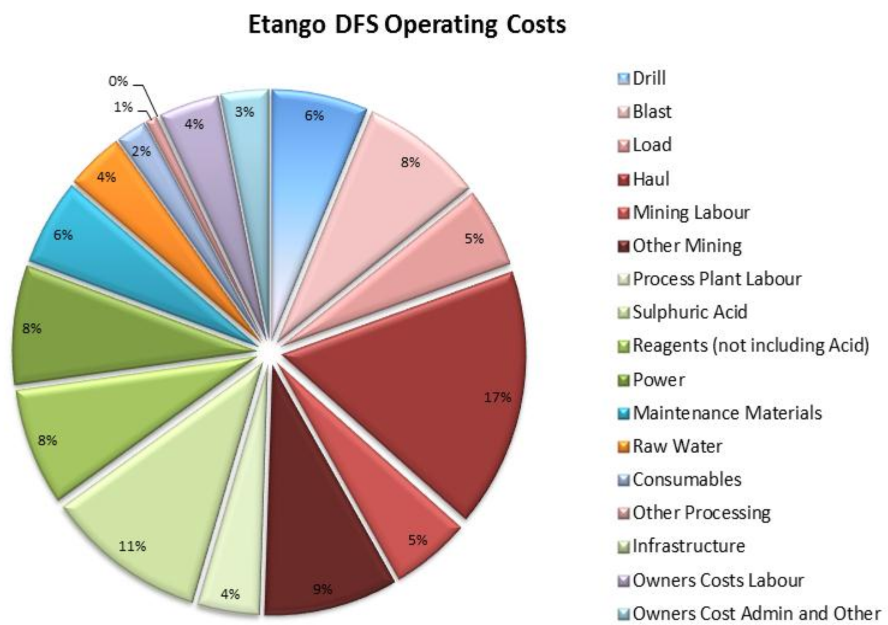
21.3.7 Total Project Operating Costs

Total operating costs for the project are summarised in Table 21-12 for the life of mine (LOM) and for the first five years. A breakdown highlighting individual components of the LOM operating cost is shown in Figure 21-2.

Table 21-12
Summary of Total Operating Costs - LOM

Item	Cost ($/t of ore Yr 1-5)	Cost ($/t of ore LOM)	Cost ($/lb U_3O_8 Yr 1-5)	Cost ($/lb U_3O_8 LOM)	% of LOM Cost
Mining	7.87	8.55	19.83	23.09	50.5
Processing and Infrastructure	7.10	7.17	17.90	19.36	42.4
Owner's Costs	1.24	1.21	3.12	3.26	7.1
Total	**16.21**	**16.93**	**40.85**	**45.71**	**100**

Figure 21-2
Total Project Operating Cost Breakdown



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 KPMG 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 (Table 21-8), metallurgical parameters, capital and operating costs identified earlier in this report. These have been reviewed by AMEC 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 can be turned on or off. 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 pre-tax basis.

All revenue and cost estimates are expressed in US$ and are based on real December 2011 quarter 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 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 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.

22.2.3 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.

22.2.4 Revenue

For the Base Case, uranium output is sold at a long term contract price of $75/lb of U_3O_8 (Section 19). 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.

22.2.5 Royalties

The financial model assumes a Namibian Government gross royalty of 3.0% of sales revenue in accordance with current Namibian legislation.

No vendor royalty is shown, i.e. it is assumed that the 20% private interest in Bannerman Namibia does not dilute to a contractual 2% net smelter return royalty.

22.2.6 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.

A royalty of 3.0% of gross sales is expected to be applicable.

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.

Table 22-1

Etango Base Case Financial Model – Cash Flow Summary

ETANGO DEFINITIVE FEASIBILITY STUDY (April 2012)

Financial Year Ending 31 December	TOTAL	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Physicals																				
Mining, Milling and Production (Mt)																				
Ore	279.6				10.1	20.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0	18.8	20.3	20.5	19.5	22.3	7.3	0.6
Waste	935.0				29.9	50.1	80.4	80.0	80.5	80.3	79.7	80.3	80.3	81.6	69.9	54.1	40.9	37.9	8.6	0.5
Total Material Mined	1,214.5				40.0	70.1	100.3	100.0	100.5	100.3	99.7	100.3	100.3	100.5	90.2	74.6	60.4	60.2	16.0	1.1
Grade (U_3O_8 ppm)	193.5				202.0	243.4	207.2	188.7	199.8	195.5	181.2	151.4	171.4	170.0	172.3	190.8	195.8	209.9	268.3	311.3
Strip Ratio	3.3				3.0	2.5	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.3	3.4	2.6	2.1	1.7	1.2	0.7
Ore Feed	279.6				9.3	20.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0	19.6	20.0	20.0	20.0	20.0	10.0	0.6
Grade (U_3O_8 ppm)	193.5				203.4	243.3	207.2	188.7	199.8	195.5	181.3	155.9	171.4	166.0	172.3	190.8	195.5	209.9	251.6	311.2
Recovery					86.0%	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%	86.9%
U3O8 (t)	46,980				1,635	4,226	3,599	3,278	3,470	3,396	3,148	2,707	2,977	2,829	2,994	3,314	3,395	3,645	2,190	175
U3O8 (000lbs)	103,573				3,605	9,317	7,934	7,228	7,650	7,488	6,941	5,968	6,564	6,237	6,600	7,305	7,485	8,037	4,827	386
Revenue (US$ M)																				
Price ($/lb)	75.0				75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0	75.0
Sales (000lbs)	103,573				1,772	7,875	8,564	7,463	7,509	7,542	7,123	6,293	6,366	6,346	6,479	7,070	7,425	7,853	5,897	1,995
Gross Revenue	7,768				133	591	642	560	563	566	534	472	477	476	486	530	557	589	442	150
Royalties and Conversion Costs	347				6	26	29	25	25	25	24	21	21	21	22	24	25	26	20	7
Net Revenue	7,421				127	564	614	535	538	540	510	451	456	455	464	507	532	563	423	143
Operating Expenditure (US$ M)																				
Mining	2,392				79	121	161	167	181	188	196	204	202	201	188	164	143	137	51	10
Processing	2,000				73	141	141	141	141	141	141	141	141	139	141	141	141	141	78	16
Infrastructure	5				0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
Ow ners Costs	337				23	23	22	22	22	22	22	22	22	22	22	22	22	22	22	8
Total Operating Expenditure	4,734				175	285	325	330	345	351	359	367	366	362	351	327	306	301	151	34
Pre-Tax Cash Flow	2,687				(48)	279	289	204	193	189	151	84	91	92	113	179	226	262	272	109
Tax	512										27	25	26	29	36	62	75	92	97	44
Post-Tax Cash Flow	2,175				(48)	279	289	204	193	189	124	59	65	63	77	117	151	171	175	66
Capital Expenditure (US$ M)																				
Mining Direct Costs	488	1	22	104	109	82	12	22	14	9	24	13	22	5	23	11	41	0		(25)
Processing Direct Costs	354		140	215																
Mobilisation and Demobilisation	14		6	8																
First Fills and Opening Stocks			5	7																(12)
Spares & Commissioning	18		7	11																
Site Infrastructure Direct Costs	97		36	55								6								
Ow ners Direct Costs	40	6	10	24																
External Infrastructure	47		23	23																
Sustaining Capital	1				0	0	0	0	0	0	0	0	0	0	0	0	0	0		
Pre-Prod Ow ners, EPCM & Accuracy Provision	128	13	51	64																
Temporary Services & Facilities, Const Camp and Vendor Reps	32		16	24	(7)															
Rehabilitation	33						1			1			1			1		1		30
Import Duties																				
Ow ners Contingency																				
Total Capital Expenditure	1,251	20	316	535	102	82	13	22	14	9	24	19	23	5	23	12	41	1		(7)
Post Capital Expenditure Cash Flow	923	(20)	(316)	(535)	(150)	198	276	183	179	180	101	40	42	59	54	105	109	170	175	73
Operating Cost (US$/t ore)	**16.93**				**18.80**	**14.24**	**16.24**	**16.52**	**17.24**	**17.57**	**17.97**	**18.36**	**18.28**	**18.47**	**17.54**	**16.36**	**15.29**	**15.03**	**15.03**	**51.82**
Operating Cost (US$/lb U_3O_8)	**45.71**				**48.50**	**30.57**	**40.94**	**45.72**	**45.07**	**46.92**	**51.78**	**61.52**	**55.68**	**58.11**	**53.17**	**44.79**	**40.86**	**37.40**	**31.20**	**86.98**

22.2.7 Foreign Exchange Rates

Capital and operating items in foreign currency were converted to US$ using assumed long term exchange rates based on available economic research from various investment and banking institutions including Rand Merchant Bank and Investec. The base case assumption is that US$1.00 equals A$1.20; N$8.60; ZAR8.60 and €0.80.

22.2.8 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.

22.2.9 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.

22.2.10 Financial Parameters

Net Present Value (NPV)

Project NPVs are calculated on both annual before- and 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 before- and after-tax net cash flows.

Payback Period

The payback period is defined as the period of time in which the cumulative undiscounted before or 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.

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 2031 based on the February 2011 mineral resource estimate
Annual throughput	20Mt
Fuel price	$0.94/L, plus freight
Sulphuric acid price	$99.50/t delivered to site
Raw water cost	$2.74/m^3, comprising $2.50/m^3 delivered to Swakopmund reservoir and $0.24/m^3 for O&H costs ex Swakopmund reservoir to site
Power cost	$0.098/kWh
Production rate	Between approximately 6 to 9 Mlb of U_3O_8 per year
Exchange rates	US$1.00 : A$1.20 : N$8.60 : R8.60 : €0.80

The key outputs from the financial model based on the above assumptions are reported for the first 5 years of the modelled operation and for the life of mine in Table 22-3.

Table 22-3
Key Financial Model Outputs

	First 5 Years	Life of Mine (Excl. Tax)	Life of Mine (Incl. Tax)
Project Economics			
NPV at a Discount Rate of 8% ($M)	-	238.1	68.7
Internal Rate of Return (%)	-	11.6%	9.2%
Payback Period from Start of Production (years)	-	6	6
Production			
Quantity Ore Treated (Mt)	89.3	279.6	
Uranium Oxide Produced (t U_3O_8)	16,209	46,980	
Uranium Oxide Produced (Mlb U_3O_8)	35.7	103.6	
Revenue			
Average U_3O_8 Base Price ($/lb U_3O_8)	75	75	
Net Revenue ($M, after royalties)	2,378	7,421	
Operating Unit Costs			
On-Site Costs/tonne Ore Treated ($/t ore)			
Mining	7.87	8.55	
Processing (including infrastructure maintenance)	7.10	7.17	
Owners costs (including administration)	1.24	1.21	
Total Operating Costs ($/t ore)	**16.21**	**16.93**	
Total Operating Costs ($/lb produced)	**40.85**	**45.71**	
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 8Mlb U_3O_8 per year. The average cash operating cost in the first 5 years is estimated at \$40.85/lb U_3O_8 and over the life of mine is estimated at \$45.71/lb U_3O_8.

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 22-4 presents the results of the sensitivity analysis.

Table 22-4 Sensitivity Analysis (Pre-tax Basis)					
Parameter/Variation	**Value**		**Project**		
U_3O_8 Price	**U_3O_8 Price ($/lb)**		**Pre-tax IRR (%)**	**Pre-tax $NPV_{8\%}$ ($M)**	**Payback Period (years)**
-30%	52.50		0.0%	(839.4)	N/A
-20%	60.00		0.0%	(480.2)	N/A
-10%	67.50		6.0%	(121.1)	11
0%	75.00		11.6%	238.1	6
10%	82.50		16.4%	597.2	4
20%	90.00		20.7%	956.4	3
30%	97.50		24.7%	1315.5	3
Total Capital Costs	**Project Capital Costs ($M)**				
-30%	1627		7.2%	(64,2)	10
-20%	1502		8.5%	36.6	8
-10%	1376		9.9%	137.3	6
0%	1251		11.6%	238.1	6
10%	1126		13.5%	338.8	5
20%	1001		15.8%	439.6	4
30%	876		18.5%	540.3	4
Operating Costs	**Average Operating Costs ($/lb U_3O_8)**				
	First 5 Years	**Life of Mine**			
-30%	$53.11	$59.42	0.1%	(449.4)	15
-20%	$49.02	$54.85	4.4%	(220.3)	13
-10%	$44.94	$50.28	8.1%	8.9	9
0%	$40.85	$45.71	11.6%	238.1	6
10%	$36.77	$41.14	14.8%	467.2	5
20%	$32.68	$36.57	17.7%	696.4	4
30%	$28.60	$32.00	20.5%	925.6	4
Mining Costs	**Average Mining Costs ($/lb U_3O_8)**				
	First 5 Years	**Life of Mine**			
-30%	$25.78	$30.02	6.2%	(109.4)	12
-20%	$23.80	$27.71	8.1%	6.5	9
-10%	$21.82	$25.40	9.9%	122.3	6
0%	$19.83	$23.09	11.6%	238.1	6
10%	$17.85	$20.78	13.2%	353.9	5
20%	$15.87	$18.47	14.8%	469.7	5
30%	$13.88	$16.16	16.3%	585.5	4
Processing Costs	**Average Processing Costs ($/lb U_3O_8)**				
	First 5 Years	**Life of Mine**			
-30%	$23.20	$25.10	7.1%	(52.1)	11
-20%	$21.42	$23.17	8.7%	44.6	8
-10%	$19.63	$21.24	10.2%	141.4	6
0%	$17.85	$19.31	11.6%	238.1	6
10%	$16.06	$17.38	13.0%	334.1	5
20%	$14.28	$15.45	14.3%	431.5	5
30%	$12.49	$13.52	15.6%	528.2	4

Figure 22-1, Figure 22-2 and Figure 22-3 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.

Figure 22-1

Project Net Present Value

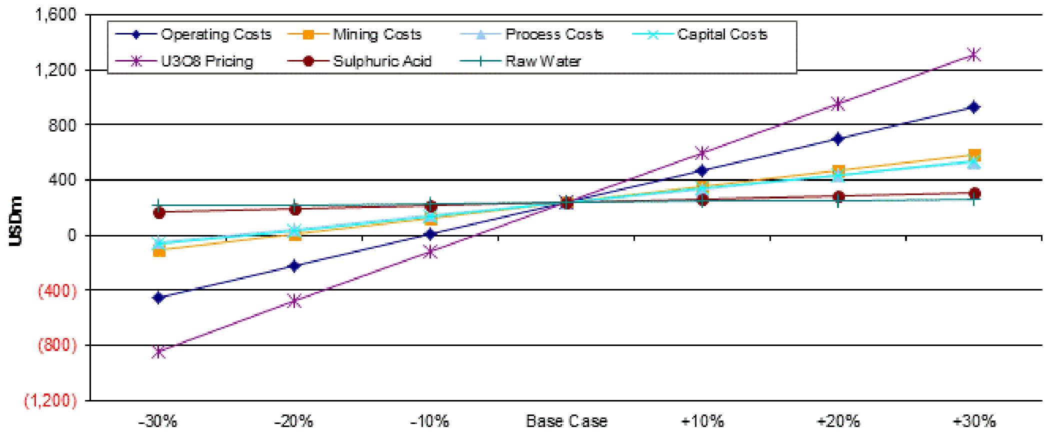


Figure 22-2

Project Internal Rate of Return

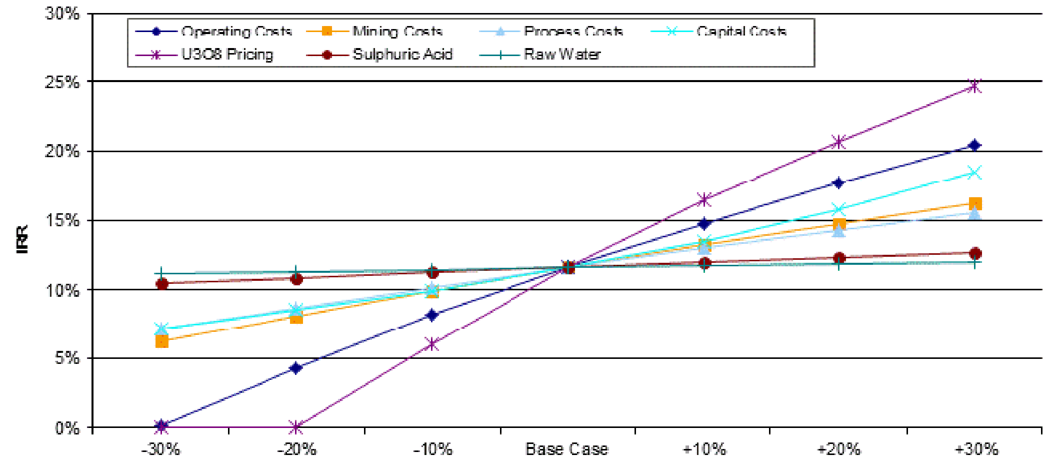
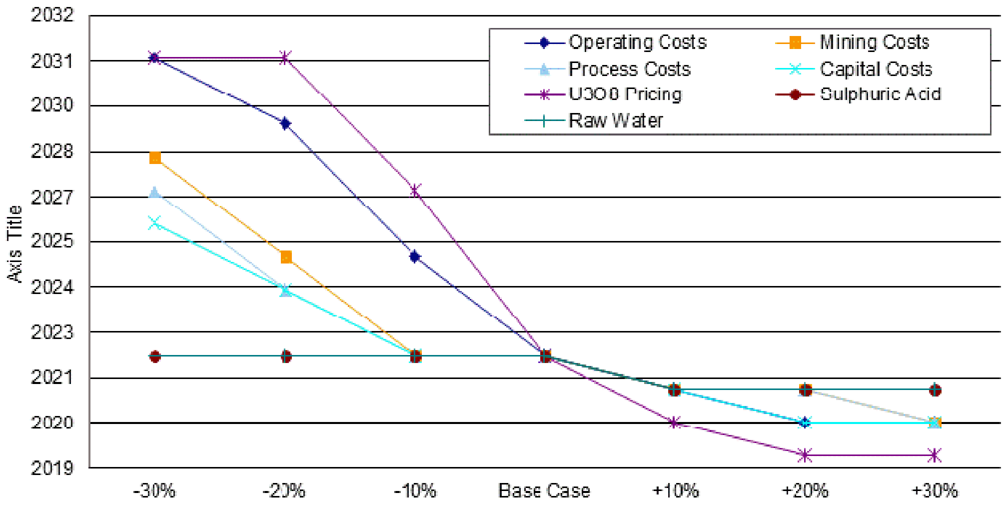


Figure 22-3

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 pre-tax NPV reducing from $238M to minus $121M and minus $480M respectively.

Likewise, positive movements of 10% and 20% from the base case assumption of $75/lb U_3O_8 produce significant changes in the pre-tax NPV from $238M to $597M and $956M respectively, the latter with a pre-tax IRR of 20.7%.

A 20% increase in the U_3O_8 price reduces the payback period by 3 years (to 3 years) and a 20% decrease in the U_3O_8 price results in payback not occurring during the life of mine.

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.

Increases of 10% and 20% in the base case cost assumptions produce significant adverse changes in the pre-tax NPV from $238M to $9M and minus $220M respectively, the latter with a pre-tax IRR of 4.4%.

Likewise, cost reductions of 10% and 20% from the base case assumptions result in the pre-tax NPV increasing from $238M to $467M and $696M respectively, the latter with a pre-tax IRR of 17.7%.

A 10% decrease in total operating costs reduces the payback period by 1 year (to 6 years) and a 10% increase in total operating costs increases the payback period by 3 years (to 10 years).

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.

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 pre-tax NPV from $238M to $137M and $37M respectively, the latter with a pre-tax IRR of 8.5%.

Likewise, capital cost reductions of 10% and 20% from the base case assumptions result in the pre-tax NPV increasing from $238M to $339M and $440M respectively, the latter with a pre-tax IRR of 15.8%.

A 10% decrease in capital costs reduces the payback period by 1 year (to 6 years) and a 10% increase in capital costs results in payback still occurring in 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.

23 ADJACENT PROPERTIES

The Bannerman Etango Project is situated within the highly mineralised southern Central Zone of the Damara Orogenic Belt, which is currently 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.

The Coffey Resource QP has not personally inspected these properties and has relied on public information for the following comments.

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 northeastern 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 mining 5.2Mt of rock to produce 3,628t of U_3O_8 in 2011.

23.2 Langer Heinrich Mine

The Langer Heinrich Mine, which is owned by a subsidiary of Paladin Energy Ltd, is located directly adjacent to Bannerman's Swakop River EPL 3346. 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 2011, Paladin reported the remaining Measured and Indicated Mineral Resources at the Langer Heinrich Mine to be 110.3Mt at 550ppm U_3O_8 for 133.5Mlbs of U_3O_8, at a 250ppm U_3O_8 cut-off grade. An additional 15.6Mt at 500ppm U_3O_8 for 16.4Mlbs was estimated to exist in stockpiles. The remaining mineral reserves were estimated at 109.2Mt at 550ppm U_3O_8 for 131.7Mlbs of U_3O_8, at a 250ppm U_3O_8 cut-off grade, of which approximately 10% was in existing stockpiles.

23.3 Husab (Rössing South) Project

The Husab project is owned by a subsidiary of Extract Resources Ltd (Extract) at the current time. 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).

In August 2011, Extract publically reported a mineral resource upgrade for the Husab (Rössing South) Project comprising a Measured Resource of 74Mt at 510ppm U_3O_8, an Indicated Resource of 281Mt at 440ppm U_3O_8 and Inferred Resources of 228Mt at 310ppm U_3O_8, above a 100ppm U_3O_8 lower cut-off (Extract, 2011). Within this, mineral reserves were estimated at a proven 62.7Mt at 569ppm U_3O_8 and a probable 217.3Mt at 504ppm U_3O_8, for a total of 319.9Mlbs of U_3O_8.

The Husab mineralisation is of an identical alaskite-hosted type to Bannerman's Etango Project.

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, construction and commissioning is expected to take approximately 36 months (including a 3 month contingency) from Project Approval.

Table 24-1
Project Development Milestones

Task	Date (Month)
Commence early works	-6
Project approval i.e. receipt of regulatory approvals/project financing	0
Commence site works	9
Commence commissioning (with 3 month contingency)	30
Commence ramp-up (with 3 month contingency)	36
First shipment (with 3 month contingency)	42
Ramp-up to design tonnages (with 3 month contingency)	45

The schedule shows some early works activities being undertaken, which is expected to include additional testwork 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

For the purposes of the DFS, 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 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 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 Bannerman 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 organizations 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.

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 three stages.

Waste dumps have sufficient footprint available to accommodate the LOM tonnage from a 3 Stage pit. The dump design and schedule allows for Closure considerations for drainage and rehabilitation. There is potential for storage of around 33Mt of waste as backfill in the Stage 3 north area.

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.

Testwork has been generally restricted to a bulk composite, with limited variability test work completed. A range of crush sizes, column heights, acid curing, addition rates and free acid levels, leaching periods, oxidants etc have been investigated, and the results are relatively consistent.

However, limited testwork has been completed under the final (selected) process parameters, and little variability testwork has been undertaken. Consequently, there remain some uncertainties regarding the process parameters adopted for DFS-level engineering.

25.3.2 SX and Uranium Recovery

Extraction testwork 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, testwork work has not yet been undertaken to confirm equipment performance and sizing.

Build up of contaminants in the return raffinate and impurity levels in the U_3O_8 product over the longer terms have not yet been determined.

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 an 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, and an ESIA submitted to support the application of a Mining Licence.

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.

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
Water supply	High – Major risk until assured
Mining equipment under-performance	High
Mine operating costs overrun (sustained increase in labour/ materials costs)	High
Capital cost over-run	Moderate to High
Operating cost over-run – power	Moderate to High
Operating cost over-run - acid	Moderate to High
Timely availability of water supply	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
Failure to achieve plate throughput	Low to Moderate
Ramp-up schedule over-run	Low to Moderate
Project construction delays	Low to Moderate
Adequacy of power 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
- Poorer than expected equipment performance and/or availability which would lead to failure to meet the production schedule and increased unit costs
- Sustained unbudgeted increase in labour / materials / consumables costs.

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 utilised in the DFS, higher than current long-term price predictions of \$65-70/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 US\$ 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, and an accuracy provision allowance made for expected, unidentified additional costs once detailed engineering has been undertaken. These provisions are based on AMEC'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. This has not been included by Bannerman in the base case financial model.

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 ***High***.

25.7.6 Operating Cost Overrun

Diesel costs are the highest single component of the operating cost constituting 15.3% of the total. Consequently, increases in oil prices will impact significantly on project economics, and, given the volatility of oil prices in recent years, this is considered a ***High*** risk area.

The base case annual consumption of sulphuric acid is approximately 340,000tpa. At the assumed delivered price of $100/t, this represents the highest process operating cost item, and some 10.6% of total operating costs. 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 2011 price of 9.75c/kWh has been applied to determine operating costs, but there is significant pressure on NamPower to increase the tariff well above CPI for the next 4 to 5 years, which has not been accounted for. Electricity costs account for 7.7% of the total operating cost and this is a ***Moderate*** risk to the Project.

25.7.7 Process

Although only a modest amount of metallurgical testwork has been carried out for the base case heap leach option by DFS standards, the results are generally consistent. Further testwork should be carried out in both leaching and the SX / precipitation / thickening / calcining area, to increase confidence in and improve equipment specification and 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, as there is little precedent for full scale HPGR operation in hard rock applications and because scale-up from testwork is not well proven.
- Heap Leaching: heap leach testwork 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 testwork to-date indicate relatively consistent results over a range of conditions, in line with the design criteria.
- SX and recovery testwork has been limited, and the design includes numerous assumptions regarding equipment and performance that remain to be quantified through additional testwork 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.

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 ***Low to Moderate***.

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

NamWater and the existing and potential mines in the Erongo area have formed the Erongo Mines Water Users Group (EMWUG). The Ministry of Agriculture, Water and Forestry, in conjunction with NamWater and EMWUG is in the process of developing a 20ML seawater desalination plant, at Mile 6, to provide the extra capacity required in the Region. Tenders for the construction and operation of the new desalination plant are reportedly due in mid-2012. Until there is resolution on the supply of water from Mile 6 or the existing desalination plant at Wlotzkasbaken, water supply remains a ***Major*** risk to the project.

The delivery water supplies within the proposed timeframe is also of concern given the number of stakeholders in the equation and the importance of an adequate and affordable water supply to the Project.

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 30 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, whereafter the royalty will revert to the standard rate of 3%. The royalty for all other mines remained unchanged as gazetted in 2006. The DFS 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.

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 TESEF initiatives 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.
- The ramp-up schedule for the process plant of 12 months is not aggressive.

26 RECOMMENDATIONS

Although the DFS has 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.

26.1.1 Metallurgical Testwork

Additional metallurgical testwork is recommended as follows:

Leaching Testwork

- Crib Pilot Plant

  Pilot operation consisting of a set of cribs (4 sets of 2, each 2x2x6m) to treat composite samples, running in a closed circuit with a pilot SX plant, each for a period of about 50 days. The DFS design values of P_{80}=5.3mm and height = 5m should be used. In this regard it should be noted that it will be difficult to obtain an operational HPGR product size distribution using smaller HPGR units.

  This work should be combined with downstream process testwork recommended by Bateman 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, acid use and percolation rates.

- Heap Leach – Variability Program

  This program is designed to assess more fully the range of characteristics of the ore across the deposit. 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.

- Heap Leach – Materials Testing

  Laboratory testwork 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 banded gneiss material from the mine can be used as a drainage layer (high resistance to acid attack) and, if so, determine the optimum thickness for the drainage layer.
  - Simulate the whole pad, liner and drainage sequence in respect to stability.

Metallurgical Testwork – Solvent Extraction, Precipitation and Product Quality

For design purposes, Bateman recommends that a pilot plant run should be completed with the chosen equipment to confirm engineering parameters for the design.

Bateman 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.

Details of the testwork programs remain to be developed, but preliminary cost estimates are as shown in Table 26-1.

Table 26-1
Recommended Process Testwork Budget

Item	**Cost ($)**
Crib pilot plant	1,250,000
Leaching variability	100,000
Heap leach materials testing	25,000
Pilot SX / precipitation / recovery	250,000
Total	**1,625,000**

26.1.2 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.

26.1.3 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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Appendix 1
Certificates

Certificate of Qualified Person

As an author of the report entitled "*Etango Uranium Project, Feasibility Study, National Instrument 43-101 Technical Report"* dated 24 May 2012, on the Etango Project, property of Bannerman Resources Limited (the 'Technical Report'), I hereby state:

1. My name is David Denis Greig and I am a Principal Geologist with the firm of AMEC Australia Pty Ltd of 140 St Georges Terrace, Perth, WA, 6000, Australia.
2. I am a practising geologist and a Member of the Australian Institute of Geoscientists (1722).
3. I am a graduate of St Andrews University in Scotland with a BSc (Hons) in Geology obtained in 1969.
4. I have practiced my profession continuously since 1969.
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 have not visited the Etango Project property.
7. I have managed overall preparation of the Technical Report, with specific responsibility for Sections 2, 3, 4, 6, 20 and 27, and the associated text in the Summary, Interpretation and 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 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 May 2012.

[signed]

David Denis Greig
Principal Geologist, AMEC Australia Pty Ltd

BSc (Hons) Geology)

MAIG

Certificate of Qualified Person

As an author of the report entitled "*Etango Uranium Project, Feasibility Study, National Instrument 43-101 Technical Report*" dated 24 May 2012, on the Etango Project property of Bannerman Resources Limited (the "Technical Report"), I hereby state:

1. My name is Peter Nofal and I am a Group Manager Studies with the firm of AMEC Australia Pty Ltd of 140 St Georges Terrace, Perth, WA, 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. In 1992 I graduated from the University of South Africa with a BCom (Hons), majoring in business economics.
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 for 1 day and surrounding areas in for 5 days in July 2011.
7. I contributed to and am responsible for Sections 5, 13, 17-19, 21 (excluding mining costs), 22 and 24 of the Technical Report, and the associated text in the Summary, Interpretation and 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 May 2012.

[signed]

Peter Nofal — BSc(Eng)
Manager Studies, AMEC Australia Pty Ltd — BCom(Hons)

Certificate of Qualified Person

As an author of the report entitled "*Etango Uranium Project, Namibia, Feasibility Study, National Instrument 43.101 Technical Report*" dated 24 May 2012, 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 Australia Pty Ltd, Level 14, 140 St Georges Terrace, Perth, 6000 Australia.
2. I am a practising metallurgist and a Fellow 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.
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 May 2012.

[signed]

Dean David
Technical Director – Process
(AMEC Australia Pty Ltd)

B App Sc (Metallurgy)
FAusIMM

Certificate of Qualified Person

As an author of the report entitled "*Etango Uranium Project, Namibia, National Instrument 43-101 Technical Report*" dated 24 May 2012, on the Etango Project property of Bannerman Resources Limited (the "Technical Report"), I hereby state:

1. My name is Brian Richard Wolfe and I am a Principal Resource Geologist with the firm of Coffey Mining Pty. Ltd. of 1162 Hay Street, West Perth, WA, 6005, Australia.

2. I am a practising geologist and a Member of the AIG (4629).

3. I am a graduate of the National University of Ireland, Dublin, with a BSc Degree (Hons) in Geology (1992) and hold a Postgraduate Certificate in Geostatistics (2007)

4. I have practiced my profession for a total of 18 years since 1993.

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 have personally visited the Etango Project property and surrounding areas for a total of 3 days in March 2012. I have performed consulting services and reviewed files and data supplied by Bannerman Resources between February 2012 and April 2012.

7. I contributed to and am responsible for Sections 7 to 12, and Sections 14 and 23 of the Technical Report, 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 May 2012.

[signed]

Brian Wolfe | BSc (Hons, Geology)
Principal Resource Geologist | Post Grad Cert (Geostatistics)

Certificate of Qualified Person

As an author of the report entitled "*Etango Uranium Project, Namibia, National Instrument 43-101 Technical Report*" dated 24 May 2012, on the Etango Project property of Bannerman Resources Limited (the "Technical Report"), I hereby state:

1. My name is Harry Warries, MSc (Mine Engineering), FAusIMM and I am Manager Mining - Perth of Coffey Mining Pty. Ltd, 1162 Hay Street, West Perth, WA, 6005, Australia.
2. I am a graduate of Delft University of Technology, Holland, and hold a Masters degree, majoring in Mine Engineering (1989).
3. I am a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM, # 111318).
4. I am a practising mining engineer and have practiced my profession continuously since 1990. My relevant experience for the purpose of the Technical Report is:
   - Operational experience on numerous mines in Western Australia
   - Mine planning experience on a large number of projects, including Africa
   - 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 Section 15, Section 16 and parts Section 21 related to mining costs in the Technical Report, and responsible for related parts of Section 1, Section 21, Section 25, Section 26 and Section 27.
7. I have personally visited the Etango Uranium Project from 21 August 2007 to 24 August 2007.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. 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.
10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
11. 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.
12. 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 May 2012.

[signed]

Harry Warries MSc (Mining)
Manager Mining – Perth (FAusIMM)
Coffey Mining Pty Ltd