BANNERMAN FILES UPDATED 43-101 TECHNICAL REPORT

Perth, Australia – Bannerman Resources Limited (ASX: BMN, TSX: BAN, NSX: BMN) ("**Bannerman**" or the "**Company**") advises that it has today filed the attached Technical Report on the Company's Etango Uranium Project in Namibia with the Canadian securities regulators. The report has been prepared in accordance with Canadian National Instrument 43-101 and forms part of the Company's ongoing disclosure obligations for its listing on the Toronto Stock Exchange.

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

About Bannerman - *Bannerman Resources Limited is an emerging uranium development company with interests in two properties in Namibia, a southern African country considered to be a premier uranium mining jurisdiction. Bannerman's principal asset is its 80%-owned Etango Project situated southwest of Rio Tinto's Rössing uranium mine and to the west of Paladin Energy's Langer-Heinrich mine. Etango is one of the world's largest undeveloped uranium deposits. Bannerman is focused on the feasibility assessment and development of a large open pit uranium operation at Etango. More information is available on Bannerman's website at www.bannermanresources.com.*

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Etango Uranium Project, Namibia

National Instrument 43.101 Technical Document

Bannerman Resources Limited

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Effective Date:	30 June 2011
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1 SUMMARY

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 in the coastal Erongo region of Namibia (**Etango Project**). The Etango Project lies within the Etango tenement, exclusive prospecting licence 3345 (**EPL 3345**).

Following the positive results of a Scoping Study completed in September 2007, work commenced on a Preliminary Feasibility Study (**PFS**) to consider the development of a uranium mine at the Etango Project. The results of the PFS were released in late 2009 and, in 2010, the Company undertook further PFS activities involving various technical consultants. Bannerman released the results of an Update to the PFS (**PFS Update**) in December 2010, and work in 2011 commenced on a Definitive Feasibility Study (**DFS**). Subject to ultimate completion of a successful DFS, licencing and project financing, Bannerman is targeting to commission the Etango Project in 2014/15.

As part of Bannerman's continuous disclosure and annual reporting requirements, this report summarises the work undertaken on the Etango Project updated mineral resource estimate in October 2010 and the results of the PFS Update reported in December 2010.

In October 2010, Coffey Mining Pty Ltd (**Coffey Mining**) estimated an updated resource for the Etango deposit (comprising the combined Anomaly A, Oshiveli and Onkelo deposits, which were previously referred to as the Goanikontes area) totalling 62.7 million tonnes (**Mt**) at 205 parts per million (**ppm**) U_3O_8 of Measured Mineral Resources, 273.5Mt at 200ppm U_3O_8 of Indicated Mineral Resources and 45.7Mt at 202ppm U_3O_8 of Inferred Mineral Resources, reported above a 100ppm U_3O_8 lower cut-off. The Etango deposit forms a six kilometre long contiguous zone of uranium mineralisation. In addition, adjacent uranium deposits at Ondjamba and Hyena were estimated to hold Inferred Mineral Resources of 85.1Mt at 166ppm U_3O_8 and 33.6Mt at 166ppm U_3O_8 respectively, both reported above a 100ppm U_3O_8 lower cut-off.

Other areas within the Etango licence (EPL 3345), in the vicinity of the Etango Project, also have the potential to host additional uranium resources, including the southern portions of the Etango licence where there is soil and colluvium cover. The western flank of the Palmenhorst Dome, which incorporates the Cheetah and Ombepo prospects in addition to the Etango deposit, constitutes a prospective strike length of over 15km.

The PFS Update identified a conventional owner operated open pit mining operation that could be economically viable treating mined ore either in heap leach or agitated tank leach configurations. For the preferred heap leach configuration, the PFS Update concluded that Etango could produce an average of 5-7 million pounds (**Mlbs**) U_3O_8 per year over a circa 20 year mine life for a pre-production capital cost of US$702 million (excluding working capital and financing charges) and an average life-of-mine operating cost of US$42/lb U_3O_8.

Following completion of the PFS Update, Bannerman in early 2011 undertook a project improvement review and, based on the results of that review, commenced the DFS in April 2011. Key changes in the project configuration stemming from the improvement work include a 33% increase in the plant throughput, simplification of the mining approach and revisions to the project layout. These changes are expected to result in the DFS production estimates increasing by approximately 20% from 5-7Mlbs U_3O_8 per annum in the PFS Update to 6-8Mlbs U_3O_8 per annum. A reduction in estimated average life-of-mine operating costs from US$42/lb to US$38/lb U_3O_8 is being targeted by the Company.

Bannerman also has an 80% interest in the Swakop River exclusive prospecting licence in Namibia, however, based upon the demonstrated potential of the Etango Project, the Swakop River project is not a material asset of Bannerman and only brief comments are provided within this report.

2 INTRODUCTION

2.1 Scope of Work

In August 2010, Coffey Mining was requested by Bannerman to update the resource estimate for the Etango Project which incorporates the Anomaly A, Oshiveli and Onkelo uranium deposits and prepare an Independent Resource Update. Coffey Mining was also requested by Bannerman to prepare initial resource estimates for the Ondjamba and Hyena uranium deposits.

Coffey Mining has previously prepared an Independent Technical Report (**ITR**) on Bannerman's Namibian operations in 2007 and prepared updated resource and ITRs in January and September 2008, February, September and December 2009 and March and October 2010.

Based upon the demonstrated potential of the Etango Project, located on the Etango (previously called Welwitschia) licence in Namibia, the other project at Swakop River is currently not considered to be a significant material asset of the Company and only brief comments are provided on this project.

This report is intended to comply with disclosure and reporting requirements set forth in the Toronto Stock Exchange Manual, National Instrument 43-101, Companion Policy 43-101CP, and Form 43-101F1.

This report complies with Canadian National Instrument 43-101, 'Standards of Disclosure for Mineral Projects' dated 30 June 2011 (the **Instrument**), and the resource and reserve classifications adopted by 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 (the **Code**) 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**).

Furthermore, this report has been prepared in accordance with the 'Code for the Technical Assessment and Valuation of Mineral and Petroleum Assets and Securities for Independent Expert Reports' of 2005 (the **Valmin Code**) as adopted by the Australasian Institute of Mining and Metallurgy (**AusIMM**). The satisfaction of requirements under both the JORC and Valmin Codes is binding on the authors as Members of the Australasian Institute of Mining and Metallurgy and the Australian Institute of Geoscientists.

2.2 Principal Sources of Information

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

- Information from Mr Kieron Munro who was formerly Head of Geology and is now a part time consultant to the Company, and his knowledge of internal procedures and processes obtained by working for the Company.
- Information from John Turney, Bannerman Project Director and Paul Henharen, Bannerman Study Manager in relation to feasibility studies on the Etango Project.
- Field observations, reports and data obtained during field trips in 2007, 2008, 2009, 2010 and 2011 by Mr Inwood and other Coffey Mining personnel.
- Information provided by Bannerman and extensive discussions with Bannerman's exploration crews.
- Various published historic, technical and scientific papers and reports.
- Digital exploration and resource modelling data.
- Published information relevant to the Etango Project area and the region in general.

The various sections of the report have been internally reviewed to identify any material errors or omissions prior to lodgement.

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

2.3 Participants

Bannerman Resources Limited was responsible for preparation of all portions of this report apart from Sections 12, 14 and 26.1 and the associated text in the summary, conclusions and discussion, which were prepared by Coffey Mining.

The following personnel took part in the preparation of this report:

- Mr John Turney – Project Director, Bannerman Resources. Overall responsibility for the report and specifically for Sections 1-3, 13 and 16-26 and the associated text in the summary, conclusions and recommendations.
- Mr Kieron Munro – Geological consultant to Bannerman (Formerly Head of Geology). Responsible for Sections 1-12 and 23 and the associated text in the summary, conclusions and recommendations.
- Mr Neil Inwood – Principal Resource Geologist of Coffey Mining. Responsible for Sections 12, 14 and 26.1 and the associated text in the summary, conclusions and recommendations.

2.4 Site Visit

Site visits to the Etango and Swakop River Projects were undertaken by Mr Neil Inwood and other representatives of Coffey Mining, between August 21st and 23rd 2007, during which period they reviewed the data collection procedures and geology, mining, processing, environmental and waste disposal aspects of the Projects, and again by Mr Inwood between April 21st and 25th 2008, October 13th and 15th 2009, August 9th and 15th 2010, and in

September 2011. Mr Turney also made numerous site visits and infrastructure surveys in the period September 2009 to September 2011.

Mr Kieron Munro is a consultant to Bannerman and has worked at the Etango Project property and surrounding areas since May 2009. During this period, he has managed the work on various geological activities as required by his position as Head of Geology for Bannerman, including attending on site in Namibia at various times between June 2009 and March 2011.

2.5 Qualifications and Experience

Mr John Turney is an engineer with 35 years' experience in project development, construction and operation in Australia, Europe, North America and South America. Mr Turney is a Fellow of the AusIMM and has the appropriate relevant qualification and experience and independence to be generally considered a Qualified Person as defined in the Instrument. He has, however, less than five years direct experience in uranium geology, mineralogy and metallurgy.

Mr Kieron Munro is a professional geologist with over 30 years' experience in exploration, mining and resource geology in Australia, New Zealand, South East Asia and Africa. 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. He has, however, less than five years direct experience in uranium geology and uranium exploration.

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.

The author of the resources section of this report (Section 14) is Mr Neil Inwood, a full time employee of Coffey Mining and a professional geologist with 16 years' experience in mining and resource geology in Australia, Canada, USA, Europe and Asia. Mr Inwood is a Fellow of the AusIMM, and has the appropriate relevant qualifications, experience and independence to be generally considered a Qualified Person as defined in the Instrument, however he has less than five years direct experience in uranium geology and uranium exploration.

2.6 Independence

Mr John Turney is an employee of Bannerman and is not considered independent as outlined under Section 1.4 of the Instrument.

Mr Kieron Munro is a consultant to Bannerman and not considered independent as outlined under Section 1.4 of the Instrument.

Neither Coffey Mining nor Mr Inwood 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 sections of this report for which Mr Neil Inwood is responsible were 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.

2.7 Abbreviations

All monetary amounts expressed in this report are in United States of America dollars (US$) unless otherwise stated. The current exchange rate from US$ to Namibian dollars (N$) is approximately 8.1. 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 (km²) 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 below.

Table 2-1
List of Abbreviations

	Description		Description
$	United States of America dollars	Mlbs	million pounds
"	inches	mm	millimetres
µ	microns	Mtpa	million tonnes per annum
3D	three dimensional	$N	Namibian dollars
AAS	atomic absorption spectrometry	N (Y)	northing
bcm	bank cubic metres	Na	sodium
Ca	calcium	Nb	niobium
CC	correlation coefficient	Ni	nickel
cm	centimetre	NPV	net present value
cps	Counts per second	NQ_2	size of diamond drill rod/bit/core
CV	coefficient of variation	ºC	degrees centigrade
DDH	diamond drillhole	OK	Ordinary Kriging
DTM	digital terrain model	Pd	palladium
EPL	Exclusive Prospecting Licence	ppb	parts per billion
g	gram	ppm	parts per million
g/m³	grams per cubic metre	psi	pounds per square inch
g/t	grams per tonne	PVC	poly vinyl chloride
HARD	half the absolute relative difference	QC	quality control
HDPE	high density polyethylene	QQ	quantile-quantile
K	potassium	RAB	Rotary Air Blast
NQ	size of diamond drill rod/bit/core	RC	reverse circulation
hr	hours	RL (Z)	reduced level
HRD	half relative difference	RQD	rock quality designation
ISO	International Standards Organisation	SD	standard deviation
kg	kilogram	SG	Specific gravity
kg/t	kilogram per tonne	Si	silica
km	kilometres	SMU	selective mining unit
km²	square kilometres	t	tonnes
kW	kilowatts	t/m³	tonnes per cubic metre
kWhr/t	kilowatt hours per tonne	Th	thorium
l/hr/m²	litres per hour per square metre	tpa	tonnes per annum
M	million	U	Uranium
m	metres	U_3O_8	Uranium Oxide
Ma	million years	w:o	waste to ore ratio
Mg	magnesium	XRF	x-ray fluorescence analysis
ml	millilitre		

3 RELIANCE ON OTHER EXPERTS

The authors of this report are not qualified to provide extensive comment on legal issues associated with the Etango Project and other projects discussed in this report.

Similarly, the authors of this report are not qualified to provide extensive comment on hydrological, environmental or financial issues associated with the Etango Project and other projects referred to in this report. The assessment of these aspects has relied heavily on information provided and prepared by other independent consultants such as Coffey Mining and A. Speiser Environmental Consultants, and copies of government approval documents (Lindeque, 2006 and Permanent Secretary, 2006).

The responsible Qualified Person for the estimation of Mineral Resources is Mr Neil Inwood of Coffey Mining. Mr Inwood's Certificate for the Estimation of Mineral Resources is included in this report (Appendix 3).

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

4.1.1 Namibian Projects

Bannerman, through an 80%-owned Namibian-registered subsidiary company, holds two exclusive prospecting licences within the central Swakopmund district of Namibia, which hosts the world's largest open cut uranium mine at Rössing (majority owned by Rio Tinto), and Paladin Resources Limited's Langer Heinrich uranium operation.

The Etango Project contains a number of identified uranium prospects and uranium anomalies. The Anomaly A, Oshiveli, Onkelo and Rössingberg Anomalies are identified in historic reports and papers, dating from the 1970’s. The Etango Project is based around the main three of the identified prospects (Anomaly A, Oshiveli and Onkelo), with other newly discovered mineralisation at the Ondjamba and Hyena prospects. The Etango Project contains alaskite hosted mineralisation similar to the significant Rössing open cut uranium mine, located 20km to the northeast, and is the subject of this report.

The Swakop River project licence surrounds Paladin Resources Ltd.'s Langer Heinrich uranium mine, which contains an extensive paleochannel with carnotite mineralisation in calcrete and channel sediments. Limited exploration drilling has been completed, targeting similar uranium mineralisation, within the Swakop River licence. Swakop River is not currently considered to be a material asset of Bannerman and will be commented on only briefly.

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).

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 the Kalahari Desert to the east.

Figure 4-1 Etango Uranium Project

Geography of Namibia

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. Major operating metalliferous 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 an important fishing and cattle industries and a traditional subsistence agricultural sector.

Namibia is serviced by a network of sealed highways connecting Windhoek in the central plateau region of Namibia with the coast at Walvis Bay, and with Botswana, Angola and South Africa. Generally unsealed but well-maintained access roads provide regional access throughout Namibia and power is available via an extensive regional electricity grid originating in South Africa. A railway line also 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 the town of Swakopmund. The

Government water authority, NamWater, provides assistance in the development of water resources for existing and potential new users.

Areas within the Namib-Naukluft National Park, which includes the Etango and Swakop River Projects, are granted for exploration, subject to appropriate environmental commitments.

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 Act has been designed to facilitate and encourage the private sector to evaluate and develop mineral resources. 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. This is defined as any 'source material containing - (a) uranium, expressed as uranium oxide (U_3O_8), of more than 0.006 per cent; (b) thorium, expressed as thorium oxide (ThO_2), of more than 0.5 per cent, and of which the mass is more than a half kilogram'.

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

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

Other licence types include:

- Non-Exclusive Prospecting Licences ('NEPL') – Which are valid for 12 months and permit non-exclusive prospecting on any open ground which is not restricted by other mineral groups.
- Reconnaissance Licences ('RL') – Which allow remote sensing techniques and are valid for 6 months.
- Mineral Deposit Retention Licences ('MDRL') – Which allow 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 two years at a time.

- Mining Licences ('ML') – Which allow the applicant to carry on mining operations. These can be awarded to accredited agents, companies registered in Namibia or any Namibian citizen. These are valid for life of the mine, or an initial period of up to 25 years, and are renewable for successive periods of up to 15 years.

Granting of licences is determined by a committee of the Ministry of Mines and Energy and granting is based on the committee's perception as to the ability and intention of the applicant to complete exploration as outlined in the application and the validity of the proposed programme to determine resources. Each licence must outline commodities of interest (in this case "Nuclear Fuels" covers uranium) and the licence granted only pertains to these commodities. Therefore, overlapping licences for differing commodities may coexist. Licences may list multiple commodity categories. Grant determination generally takes at least six months from the time of application.

An environmental contract must be completed with the Department of Environment and Tourism by applicants for EPL's, MDRL's and ML's. 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

4.4.1 The Etango Project Area (EPL 3345)

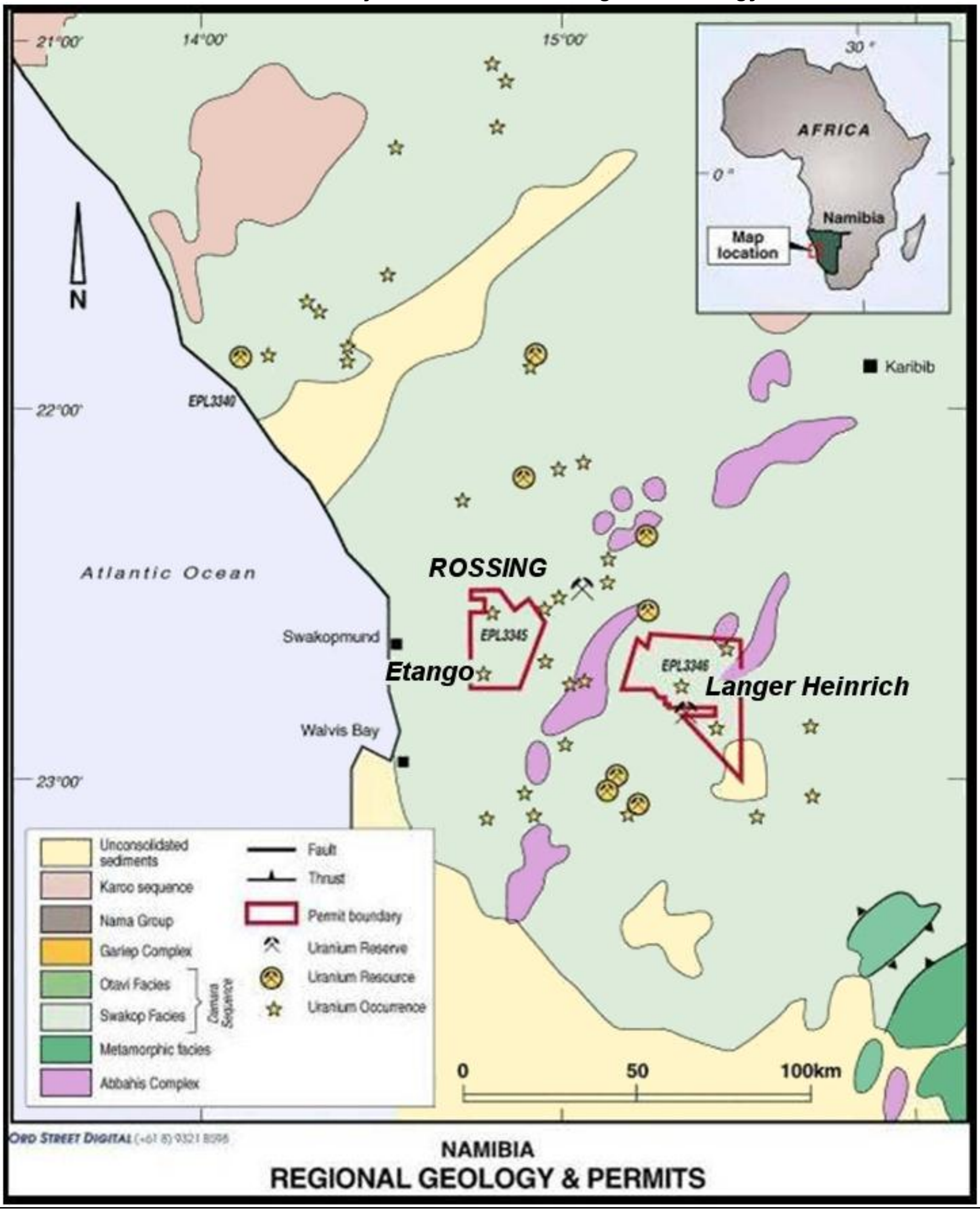
The main focus of the Etango Project comprises the Anomaly A, Oshiveli and Onkelo Prospects, located approximately 41km (by road) east of the regional town of Swakopmund and approximately 73km (by road) northeast of the deep-water port town of Walvis Bay (Figure 4.2).

The sealed C14 highway connects Swakopmund to the port at Walvis Bay and the sealed B2 highway 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 C28 road, then by the well-maintained unsealed D1991 road into the Namib-Naukluft National Park area.

The Etango Project is situated on the flat Namib Desert sands of 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.

Figure 4-2 Etango Project

Namibian Project Locations and Regional Geology



4.4.2 Swakop River Project Area (EPL 3346)

The Swakop River project area (Figure 4.2) is located approximately 67km east of Swakopmund. Access is gained by the sealed and unsealed C28 road, then by unsealed road into the Namib-Naukluft National Park area.

The Swakop River project area is not currently considered to be a material asset of Bannerman and is not discussed in any detail in the remainder of this document.

4.5 Tenement Status

4.5.1 Licences

The Etango Project EPL 3345 and Swakop River EPL 3346 (Figure 4.2) are owned by the Namibian company Bannerman Mining Resources (Namibia) (Pty) Ltd (**Bannerman Namibia**), previously called Turgi Investments (Pty) Ltd (Turgi), which manages these Projects. Bannerman owns 80% of Bannerman Namibia, while the remaining 20% is held in the name of Mr C. Jones of Perth, Australia.

EPL 3345 was granted to Turgi, now Bannerman Namibia, on 27 April 2006 for an initial three year period to explore for Nuclear Fuels. The first application for renewal for EPL 3345 was granted on 26 April 2009 for an additional two years without any reduction in area. The second application for renewal for EPL 3345 was granted on 29 March 2011 for an additional two 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, and 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 on 27 April 2006 for an initial three year period to explore for Nuclear Fuels. The first application for renewal for EPL 3346 was granted on 20 April 2010 for an additional two years (from the 27 April 2009) without any reduction in area. The second application for renewal for EPL3346 was granted on 29 March 2011 for an additional two 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 co-ordinates as Table 4-2.

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

On 17 December 2008, Bannerman announced that Bannerman Namibia had entered into an agreement to settle the 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 the 21st December 2009, Bannerman lodged an application for a Mining Licence 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.

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 area of the Anomaly A, Oshiveli, Onkelo, Ondjamba and Hyena Prospects (which contain Measured, Indicated and Inferred Mineral Resources), 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 on one of Bannerman Namibia's projects. 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.

4.7 Environmental Liabilities

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.

According to Speiser (2006), activities in the licence area are covered by a number of acts, policies and bills. These include (amongst others):

- The Namibian Constitution – Article 95.
- 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.
- The Policy for Prospecting and Mining in Protected Areas and National Monuments.

In 2009, Bannerman lodged an Environmental and Social Impact Assessment (**ESIA**) for development 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. The ESIA was based on an open pit mining operation focused on the Anomaly A deposit (forming the majority of the Etango deposit), with processing proposed to be undertaken by heap leach methods.

In April 2010, Bannerman received notification of formal environmental clearance from MET for development of the Etango Project as described in the ESIA.

An updated ESIA is scheduled to be submitted in late 2011 or early 2012. It is envisaged that a final ESIA will be lodged and an amendment to the environmental clearance requested.

No substantiative legislative, environmental or social impacts have been identified for development of the Etango Project. The Erongo region already hosts a number of other large 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.

4.8 Permitting Status

The status of the EPLs is discussed in Section 4.5.1 and the ESIA is discussed in Section 4.7. Other permits which are current include:

- Park Entry Permits – Ministry of Environment and Tourism (Etango and Swakop River Areas). 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.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Project Access

The Etango Project is located approximately 31 kilometres east of the major town of Swakopmund and 47 kilometres 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 on the D1991 into the Namib-Naukluft National Park area.

5.2 Physiography and Climate

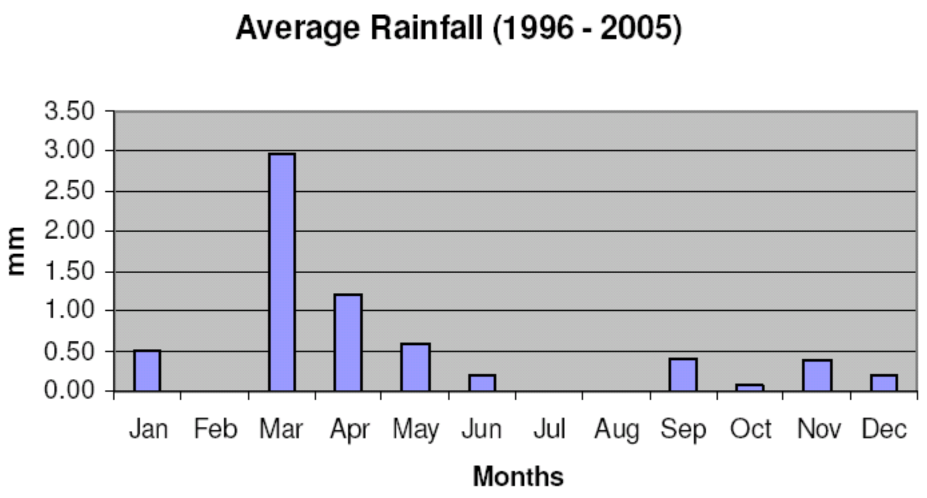
The Project area is located in the western region of the Namib Desert. The bulk of the project area lies on the Namib Peneplain where there is poor soil development over eluvial, colluvial and alluvial material, and bedrock. Due to the very low rainfall, these soils have gypsum crusts over large areas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

The area of the Etango deposit is generally flat (refer Figure 5-1) with occasional low undulating hills with sparse sub-crop of bedrock. Remnant shallow drainage channels in the desert can also be seen around the Project area. The region 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 areas of the river valleys can be difficult.

Figure 5-1
Etango Uranium Project
Drilling in The Namib Desert at Anomaly A

Rainfall in the area is very sporadic. The highest annual rainfall in the ten years from 1996 to 2005 occurred in March 2000 with 21.8mm of rainfall. Figure 5-2 summarises the average monthly rainfall for the years 1996 to 2005. The Project area also receives moisture from fogs which are caused when moist air which has been cooled by the Benguela oceanic current is blown on-shore. As a result of the moist air feeding off the Atlantic, the air along the coast line remains humid throughout the year (between 60% and >80% relative humidity). The nearby town of Walvis Bay experiences more than 125 fog days per year (Speiser, 2006).

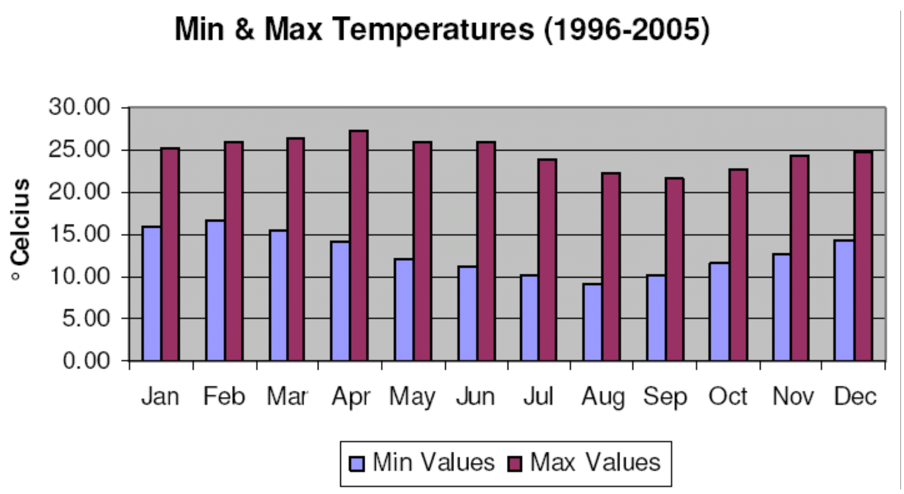
Figure 5-2 Etango Uranium Project Average Rainfall



(Speiser, 2006)

The Namib Desert region does not experience the extremes of temperatures that are typical to most other deserts, due to the presence of the cold current offshore. However, the temperature can peak at over 40ºC in the summer months, while in the coldest month of August, the minimum can fall to 9ºC (Figure 5-3). The hottest month on average is April with an average maximum temperature of 27ºC (Speiser, 2006).

Figure 5-3 Etango Uranium Project Min & Max Temperatures (1996-2005)



(Speiser, 2006)

5.3 Local Infrastructure and Services

The town of Swakopmund (31km west of the Project area) has excellent services and infrastructure, with a population of approximately 50,000 people (Figure 5-4). Services include financial, shopping, construction, trades and medical support. The port city of Walvis Bay is located 30km south of Swakopmund along the sealed C14 highway. 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. Most other mining and exploration companies in the area also utilise Swakopmund as the base for their operations.

Drilling services and water for the 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.

Figure 5-4 Etango Project
Municipality Building In Swakopmund

6 PROJECT HISTORY

The area of EPL 3345 has been the target of significant previous exploration which included both ground work (traverses and drilling) and aerial and ground based geophysical investigations.

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

A dramatic decrease 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 Investments (Pty) Ltd (**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). Turgi was later renamed Bannerman Mining Resources (Namibia) (Pty) Ltd which is 80% owned by Bannerman Resources Ltd.

After acquiring its interest in EPL 3345 in 2006, Bannerman undertook a process of capturing and digitising the historic drillhole, 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 1994/1995.

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

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Introduction

Exploration for uranium within the Central Zone of the Damara Orogen in Namibia has been conducted, periodically, since the discovery of uranium bearing beryl near Rössing Mountain in the early 1900s. In 2006, Bannerman acquired an interest in two Exclusive Prospecting Licences in this Central Zone and in proximity to the Rössing and Langer Heinrich uranium mines. One of these tenements also covered the previously discovered and explored Goanikontes Project, later renamed the Etango Project. Since 2006, Bannerman has been actively exploring in these two tenements and is now examining the possibility of mining uranium mineralisation at the Etango Project. There are three contiguous, anomalous areas in the Etango Project, which are, from north to south, the Onkelo, Oshiveli and Anomaly A Prospects.

Primary uranium mineralisation in the Etango Project area is related to uraniferous leucogranites, locally referred to as alaskites. The alaskites 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.

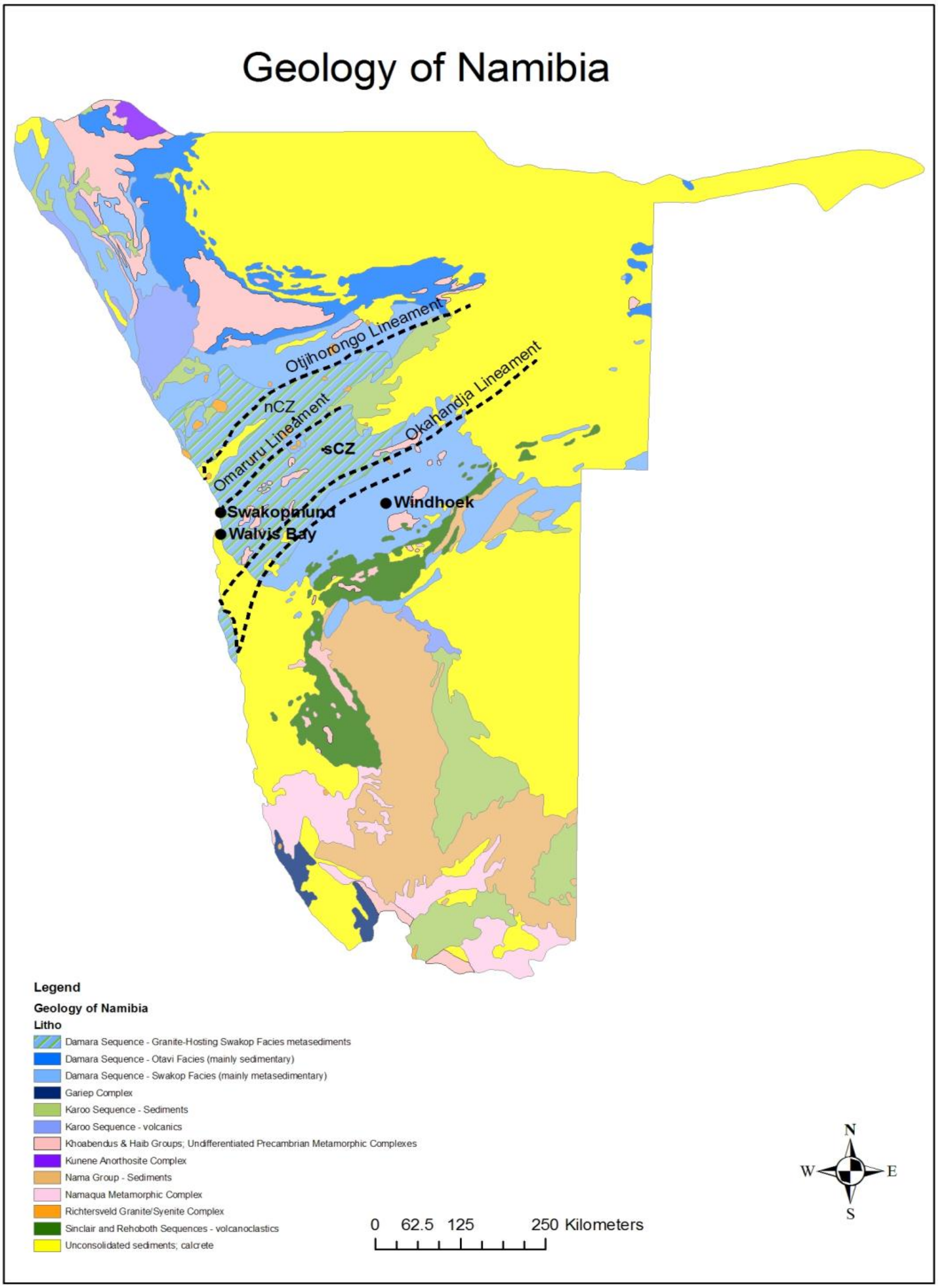
The alaskite bodies have intruded into the metasediments of the Nosib and Swakop Groups of the Damara Supergroup. These metasediments and alaskite intrusions flank the Palmenhorst Dome which is cored by Mesoproterozoic (1.7-2.0 Ga) gneisses, intrusive rocks and meta-sediments of the Abbabis Metamorphic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

An exploration and resource definition drilling programme of 1,240 RC, 141 diamond and 21 RAB drillholes, totalling 303,780 m, has been undertaken in the Anomaly A, Oshiveli and Onkelo Prospect areas at the Etango Project. This work includes some 1,253 holes for resource definition, 28 for metallurgical testwork, 29 for geotechnical studies and 21 for hydro-geological monitoring. The lithological contacts in these non-resource holes were also considered for geological modelling, even though the holes had not been assayed. In total, 1,356 drillholes have been completed for 295,981m of drilling in the Etango area. This drilling work has provided the geotechnical, hydrological, structural, lithological and grade data for this study. This effort is focussed on a 6km long segment of a larger 20km long structural and stratigraphic setting which follows and wraps around the Palmenhorst Dome.

7.2 Regional Geology

The Neoproterozoic (pre-550 Ma) to early Palaeozoic (c500 Ma) Damara Orogen consists of an N-trending coastal branch, and a NE-trending intracontinental branch which runs from Walvis Bay, through Namibia towards Botswana and Zimbabwe, Figure 7-1. In Namibia the Damara Orogen has been interpreted as a result of the collision between the Congo, Kalahari and Rio de la Plata Cratons (South America) around 550-500 Ma.

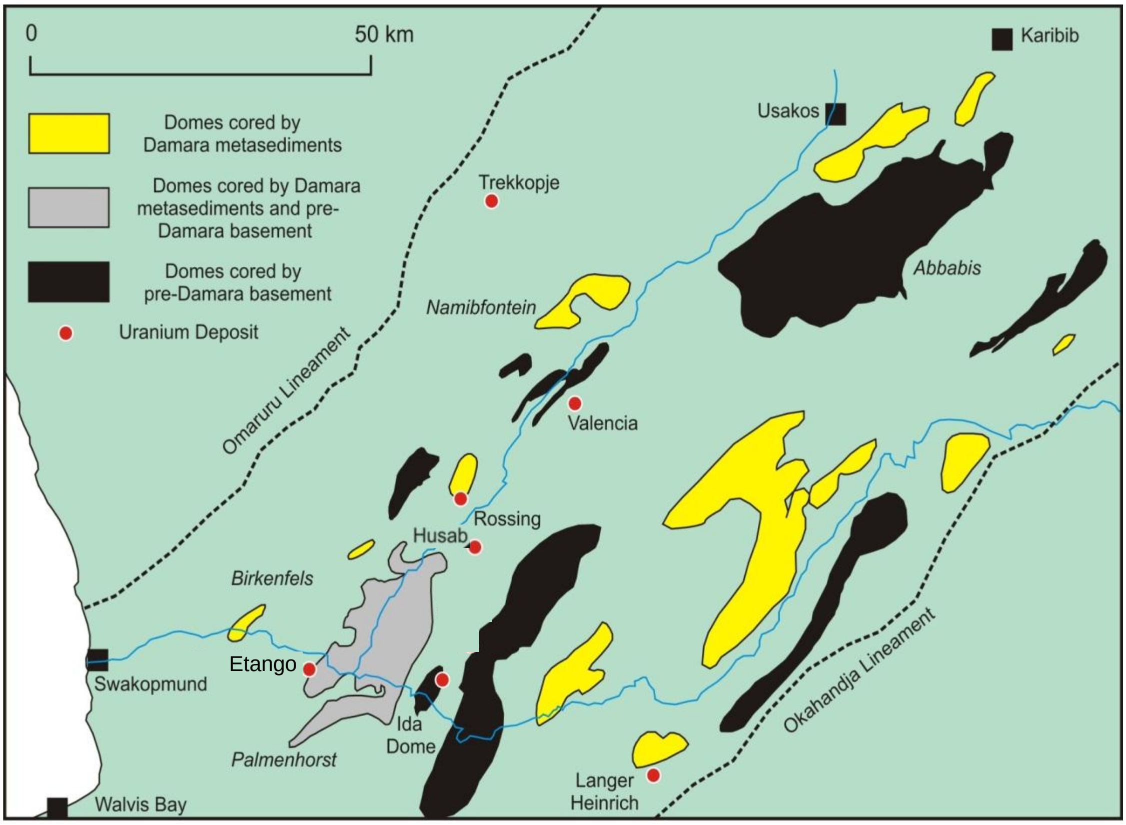
Figure 7-1 Geology of Namibia



Nex, 1997, suggests that events that led to the formation of the Damara Orogenic Belt can be summarised as follows. The pre-Gondwanaland continent rifted and the segments parted, accompanied by minor volcanic activity. Fluvial material was deposited within the rift valley and, as the basin deepened, sedimentation evolved to include marine and carbonaceous sediments, marine or terrestrial glacial deposits, and argillaceous marine sedimentation. The tectonic regime then changed from divergence to convergence, including subduction, with the onset of a major orogenic event, including polyphase deformation and associated metamorphism and igneous activity.

The 400 km wide inland branch of the Damara Orogenic Belt, between the Congo and Kalahari Cratons, is divided into a number of zones (Miller, 1983) based on lithostratigraphic, structural and metamorphic criteria. The Central Zone occupies a broad, central region in the Damara Belt and is bounded by the Otjihorongo Lineament in the north and the Okahandja Lineament in the south, Figure 7-1. The Omaruru Lineament divides the Central Zone into a Northern Central Zone, in which the higher stratigraphic formations of the Damara Sequence are exposed, and a Southern Central Zone that exposes the lower stratigraphic formations of the Damara Sequence, with large windows of basement gneisses and numerous intrusive Pan-African granitoids, Figure 7-2.

Figure 7-2 Regional Geology and Uranium Deposits of the Southern Central Zone



The Central Zone, indicated in Figure 7-1 as the Northern Central Zone (nCZ) and the Southern Central Zone (sCZ), contains voluminous granites and gneisses, including basement (Abbabis) augen gneiss, synmetamorphic red granite, the Salem granite suites and late to post-kinematic intrusions such as the Donkerhuk Granite (Basson & Greenway, 2004). Domal structures are relatively widespread within the Southern Central Zone (sCZ) and the Rössing, Palmenhorst and Ida Domes host notable uranium-enriched, sheeted leucogranites known as alaskites. This zone is characterized by elongate basement-cored domes, abundant granitoid intrusions and a metasedimentary cover sequence which has been metamorphosed at high temperature and low pressure to upper amphibolite–granulite facies (Miller, 1983 & Nex, et al., 2002).

7.2.1 Regional Stratigraphy

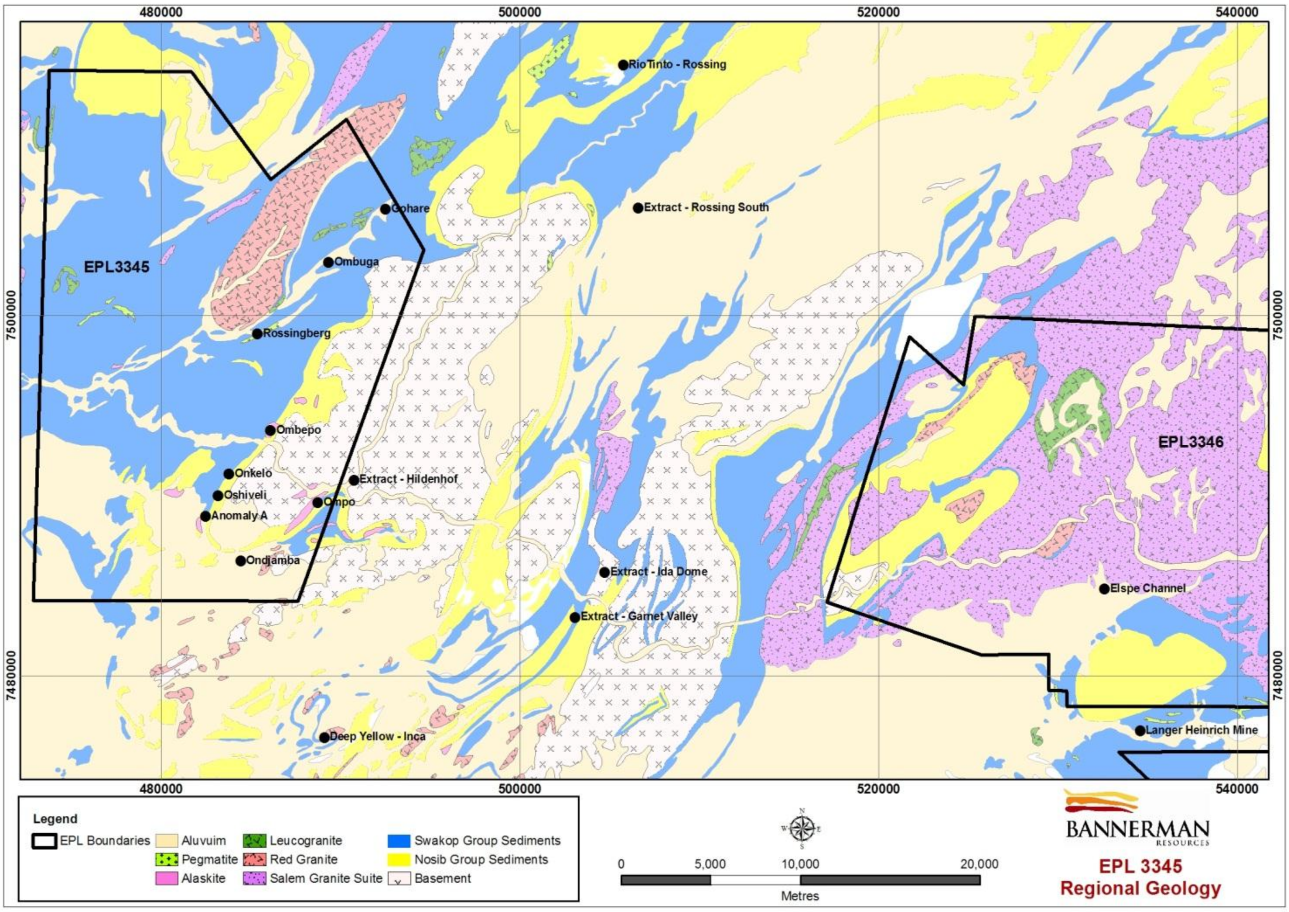
The stratigraphy of the Damara Sequence is divided into two major groups: the basal Nosib Group (comprising the Etusis and Khan Formations) and the upper Swakop Group (comprising the Rössing, Chuos, Karibib and Kuiseb Formations). The Project area stratigraphy is summarised in Table 7-1 and shown in Figure 7-3; and, for the Khan Formation, is after Nex (1997); while the Rössing Formation is after Nash (1971). The Damara Sequence unconformably overlies the gneissic and migmatitic lithologies of the Mesoproterozoic Abbabis Metamorphic Complex. A map of the geology within and around EPL3345 and EPL 3346 is shown in Figure 7-3.

Table 7-1
Stratigraphic Column of the Damara Supergroup (Roesener and Schreuder, 1997)

Group	Subgroup	Formation	Maximum Thickness	Lithology
Swakop	Khomas	Kuiseb	>3000	Pelitic and semi-pelitic schist and gneiss, migmatite, calc-silicate rock, quartzite. Tinkas member: Pelitic and semi-pelitic schist, calc-silicate rock, marble, para-amphibolite.
		Karibib	1000	Marble, calc-silicate rock, pelitic and semi-pelitic schist and gneiss, biotite amphibolite schist, quartz schist, migmatite.
		Chuos	700	Diamictite, calc-silicate rock, pebbly schist, quartzite, ferruginous quartzite, migmatite.
	Discordance			
	Ugab	Rössing	200	Marble, pelitic schist and gneiss, biotite-hornblende schist, migmatite, calc-silicate rock, quartzite, metaconglomerate.
	Discordance			
Nosib		Khan	1100	Migmatite, banded and mottled quartzo-feldspathic clinopyroxene-amphibolite gneiss, hornblende-biotite schist, biotite schist and gneiss, migmatite, pyroxene-garnet gneiss, amphibolite, quartzite, metaconglomerate.
		Etusis	3000	Quartzite, metaconglomerate, pelitic and semi-pelitic schist and gneiss, migmatite, quartzo-feldspathic clinopyroxene-amphibolite gneiss, calc-silicate rock, metaphyllite.
	Major unconformity			
	Abbabis Complex			Gneissic granite, augen gneiss, quartzo-feldspathic gneiss, pelitic schist and gneiss, migmatite, quartzite, marble, calc-silicate rock, amphibolite.

The Damara Sequence in the Central Zone was deposited unconformably on the Abbabis Metamorphic Basement Complex, which consists of augen gneiss, migmatic gneiss, granite gneiss, biotite schist and amphibolite (Basson & Greenway, 2004). The basement rocks have variable radiogenic ages, between 1955 Ma and 960 Ma (Kukla, 1992) and are mainly exposed in the cores of the domal structures that occur within the southern Central Zone.

Figure 7-3 Regional Geological Plan in the vicinity of the Etango Project



The Nosib Group forms the base of the Damara Sequence and comprises the earliest rift-fill sediments and volcanic rocks that were deposited discordantly on the pre-Damara basement. The initial Etusis Formation sediments are discontinuous fluvial deposits and acidic to alkali volcanic rocks, which are dated at 750 Ma (Kukla, 1992). These earliest rocks are then overlain by the marine quartzites, meta-arkoses and minor meta-conglomerates of the Etusis Formation that grade upwards into semi-pelitic lithologies of the Khan Formation. The Khan Formation comprises mainly amphibole-clinopyroxene gneisses that are interpreted to represent a change to a more calcareous and less clastic sedimentary protolith (Martin, 1983). The Nosib Group is unconformably overlain by the Swakop Group metasediments.

The Rössing Formation, comprising marbles, quartzites and various meta-pelitic rocks forms the base of the Swakop Group and overlies the Khan Formation para-conformably and disconformably (Basson & Greenway, 2004). The Chuos Formation then unconformably overlies the Rössing Formation and is thought to be of a glacio-marine origin, comprising mainly of pebble and boulder-bearing diamictite. The Chuos Formation is overlain by marbles of the Karibib Formation, which are in turn overlain by thick pelitic schists of the Kuiseb Formation (Kinnaird and Nex, 2008). The upper calcareous Swakop Group consists of various meta-pelitic and carbonate rocks that are interpreted to have formed on a spreading sea floor during rifting.

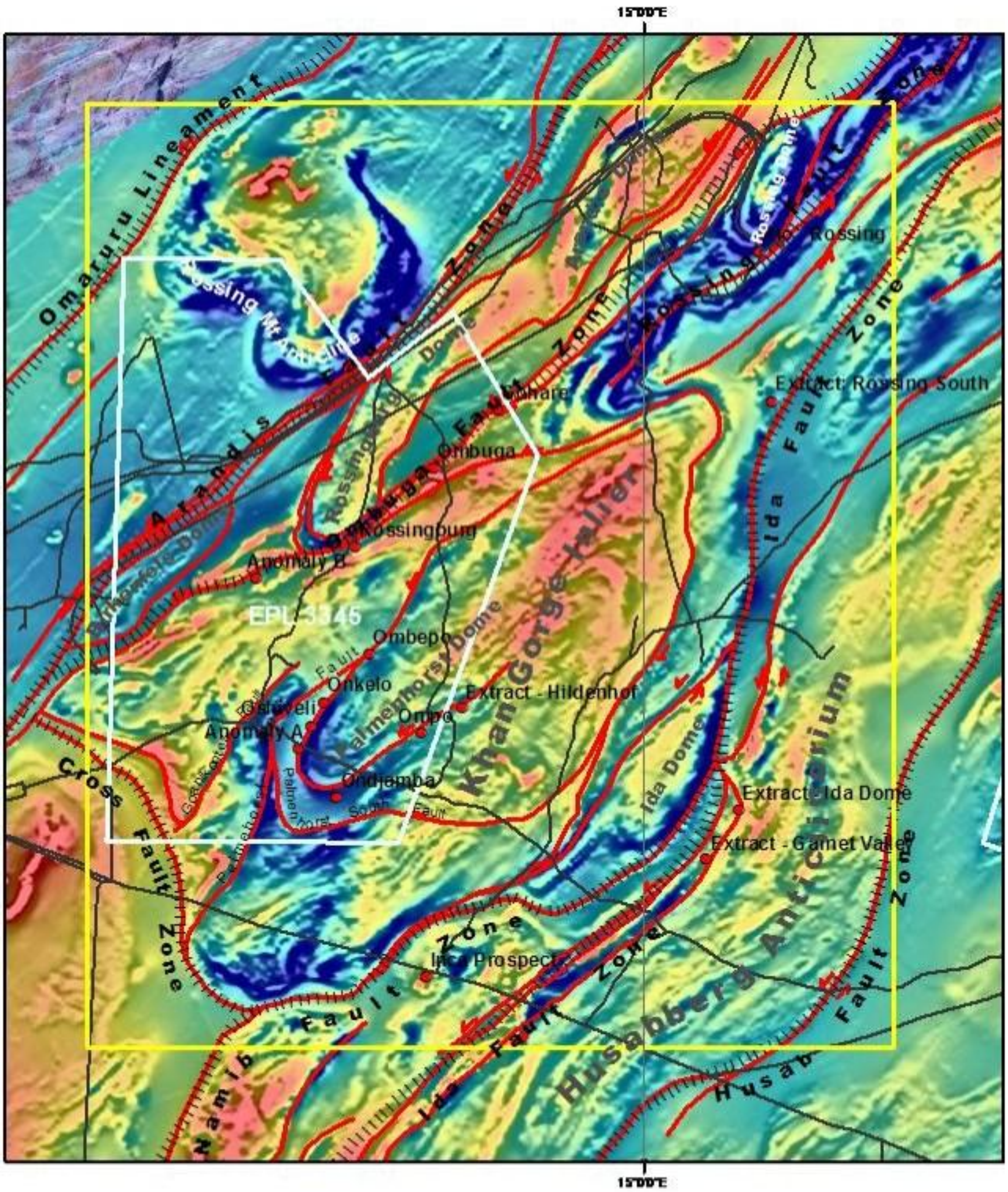
7.2.2 Regional Structure

During the Damara Orogenic event, the metasedimentary cover was subjected to numerous phases of deformation, commencing with an early folding (F_1) which produced overturned and recumbent structures that were accompanied by thrusting and shearing. The second major deformation event (D_2) resulted in a prominent gneissosity (S_2) and lineation (L_2) which is generally close to parallel to the earlier S_1 and S_0 (bedding) layering. This gneissosity was then further deformed by a later D_3 deformation event which resulted in the elongate basement-cored domes which are characterized by constrictional fabrics. Uraniferous alaskite sills and bodies that wrap around the Palmenhorst Dome are confined to dilatational sites in the D_2 high-strain zones, with the alaskite sills generally trending from north-northwest to north-northeast in strike and dipping to the west. The airborne magnetic image, Figure 7-4, clearly shows the underlying structures.

The chronological sequence of structural events in the area can be summarised as:

- Bedding (S_0) is well developed and most prominent between lithologically distinct formations as well as in formations with distinct compositional layering.
- Low angle (bedding-sub parallel) shearing (D_1) and tight to isoclinal folding resulted in the discontinuous strike extent of some formations, and the out of sequence stratigraphy in places close to the basement-cover interface.
- S_1 is the earliest tectonic fabric and is defined by the preferred orientation of metamorphic minerals or the grain-shape-preferred orientation of flattened quartz-feldspar aggregates.
- D_2 related fabrics, and associated structures, are most pervasively developed along the NW limb of the Palmenhorst Dome. High-strain D_2 fabrics are represented by linear fabrics (L_2) such as rodded and stretched minerals, clasts and earlier intrusive rocks. D_2 deformation is also associated with F_2 folding which occurs on a variety of scales. The folds have shallow to moderate, NW dipping axial planes with shallow NNE plunges, parallel to the mineral stretching lineation (L_2). This suggests that the folds have been rotated into the direction of maximum stretch.
- The Palmenhorst Dome has formed in response to D_3 refolding of formations, earlier fabrics and folds. The dome has a slight SE vergence in the project area and the SW hinge is overturned with the axis of the dome plunging towards the NE.
- The D_2 high strain zone is bounded in the west by a fault zone. This fault zone dips at 35-45° towards the NW, parallel to the main S_1/S_2 gneissosity developed in the Khan Formation and is discordant to rocks in the hanging wall. The fault zone is post-alaskite intrusion, but pre-Karoo age, with dolerite dykes cross-cutting the fault zone. E-W trending, sub-vertical faults are also common. These faults are narrow, both N-down and S-down displacement occurs, and maximum displacements observed in the field are only about 2 m. Fault strike extents do not exceed 100 m.

Figure 7-4 Regional Structural Geology Plan in the vicinity of the Etango Project



Two main styles of alaskite geometry are seen; these are stringer or sheet like, and the more massive bodies or "blows". Alaskite intrusions are semi-concordant with the enveloping gneissosity (S_1/S_2) and are seen to intrude along the axial planes of F_2 folds.

The massive bodies are located close to and above the interface between the Etusis and Khan Formations. These massive bodies have a lensoidal shape, which generally lies within the enveloping S_1/S_2 gneissosity, and may either taper towards their lateral terminations, or terminate abruptly. The bulk of the alaskite bodies are seen to have intruded during deformation (D_2), while there are also some late-stage, cross-cutting dykes which are post-deformation.

There is a clear tendency for the alaskite bodies to become more stringer-like further to the west and structurally higher up in the stratigraphic section.

7.3 Project Geology

The uranium occurrences at the contiguous Anomaly A, Oshiveli and Onkelo Prospects, which comprise the Etango Project area, wrap around the western edge of the Palmenhorst Dome, as shown below on Figure 7-5 and below in detail at the Anomaly A and Oshiveli and Onkelo areas in Figure 7-6. The dome consists of pre-Damaran basement with a core that is commonly referred to as red granite gneiss (quartz, microcline and accessory plagioclase and biotite). The core is surrounded by migmatites and a variety of basement rock types (Mouillac, et. al., 1986). A series of conformable to disconformable metasedimentary rocks of the Nosib and Swakop Groups surround the dome, and these are intruded by the alaskite-hosted uranium mineralisation.

These deposits are located in a high strain zone along the south-western flank of the Palmenhorst Dome. The current geology map of the area (Figure 7-6) is mainly based on Nex's (1997) lithostratigraphical subdivision which is supported by the latest 1:50 000 scale regional map produced by the Geological Survey of Namibia in 2001. The geology in the area comprises uraniferous sheeted leucogranite bodies (locally termed alaskites) that have intruded into metasediments of the older Nosib (Etusis and Khan Formations) and younger Swakop Group (Chuos Formation) of the Damara Sequence. The Damara metasediments are wrapped around the highly eroded Palmenhorst Dome comprising basement lithologies of the Abbabis Metamorphic Complex. These basement lithologies comprise quartzo-feldspathic augen gneiss with minor pelitic biotite schist and discontinuous amphibolite pods, intruded by fine to medium grained equigranular granites (Kinnaird and Nex, 2008).

The stratigraphy of the Damara Supergroup in the Etango Project area is discussed in more detail below. Figure 7-7 shows the detailed geology in the outcropping Oshiveli and Onkelo areas.

7.3.1 Etusis Formation

Immediately adjacent to the Palmenhorst Dome are the metasedimentary rocks of the Etusis Formation. These consist of quartzites and meta-arkoses with a pale pink coloration and cross-bedding on a small scale. The sediments are fine to medium-grained and contain mm scale laminations of Fe-Ti oxide minerals which pick out the cross-bedding.

A high-potassium, reddish granite referred to as the 'Red Granite' occurs between the basement migmatites and the Etusis Formation and as dykes and plugs in the Lower Khan Formation. This granite is a separate unit to the red granite gneiss found in the core of the dome (Mouillac, et al., 1986).

Figure 7-5 Satellite Image of EPL 3345

showing the main prospects and airborne radiometric anomalies

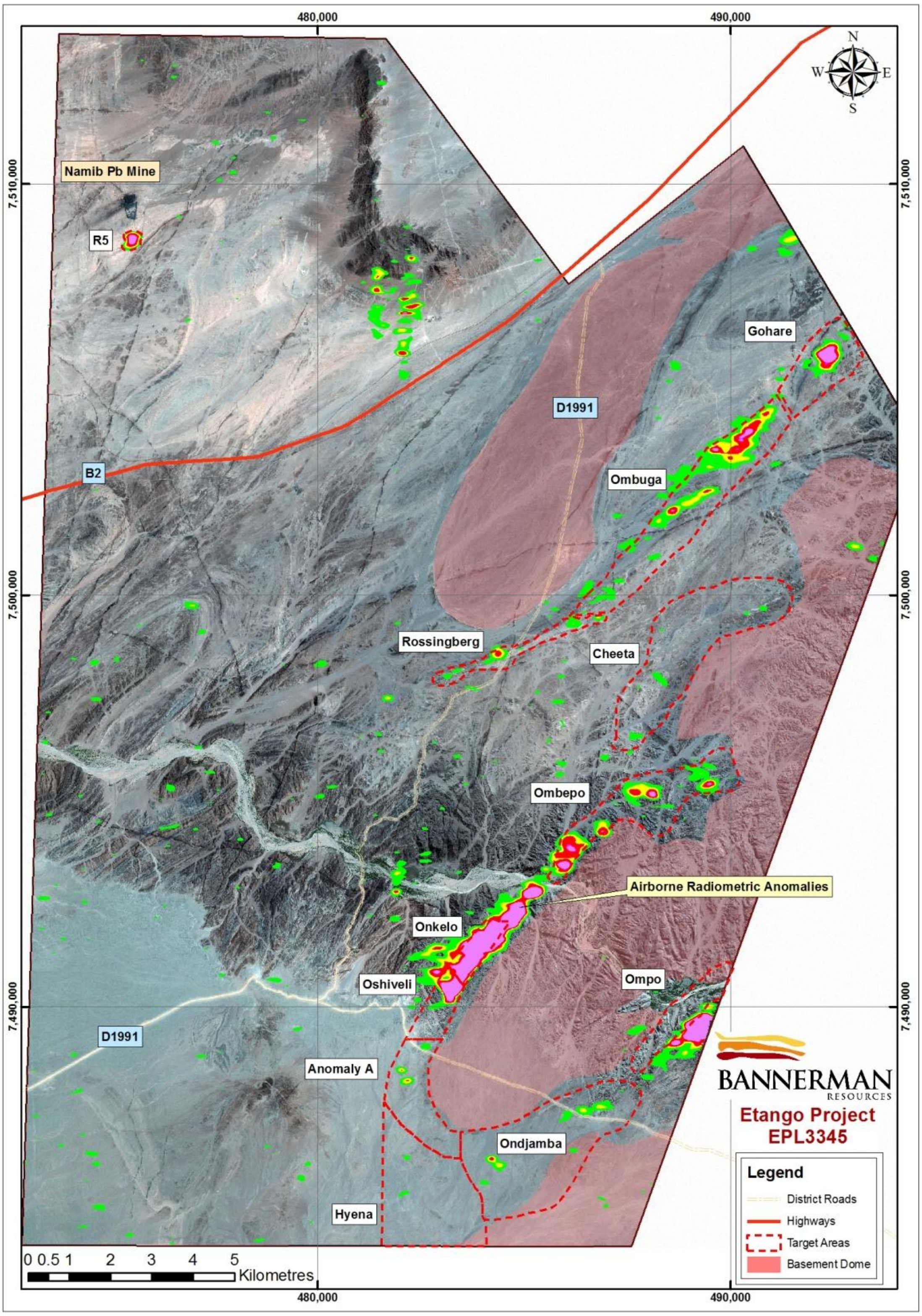
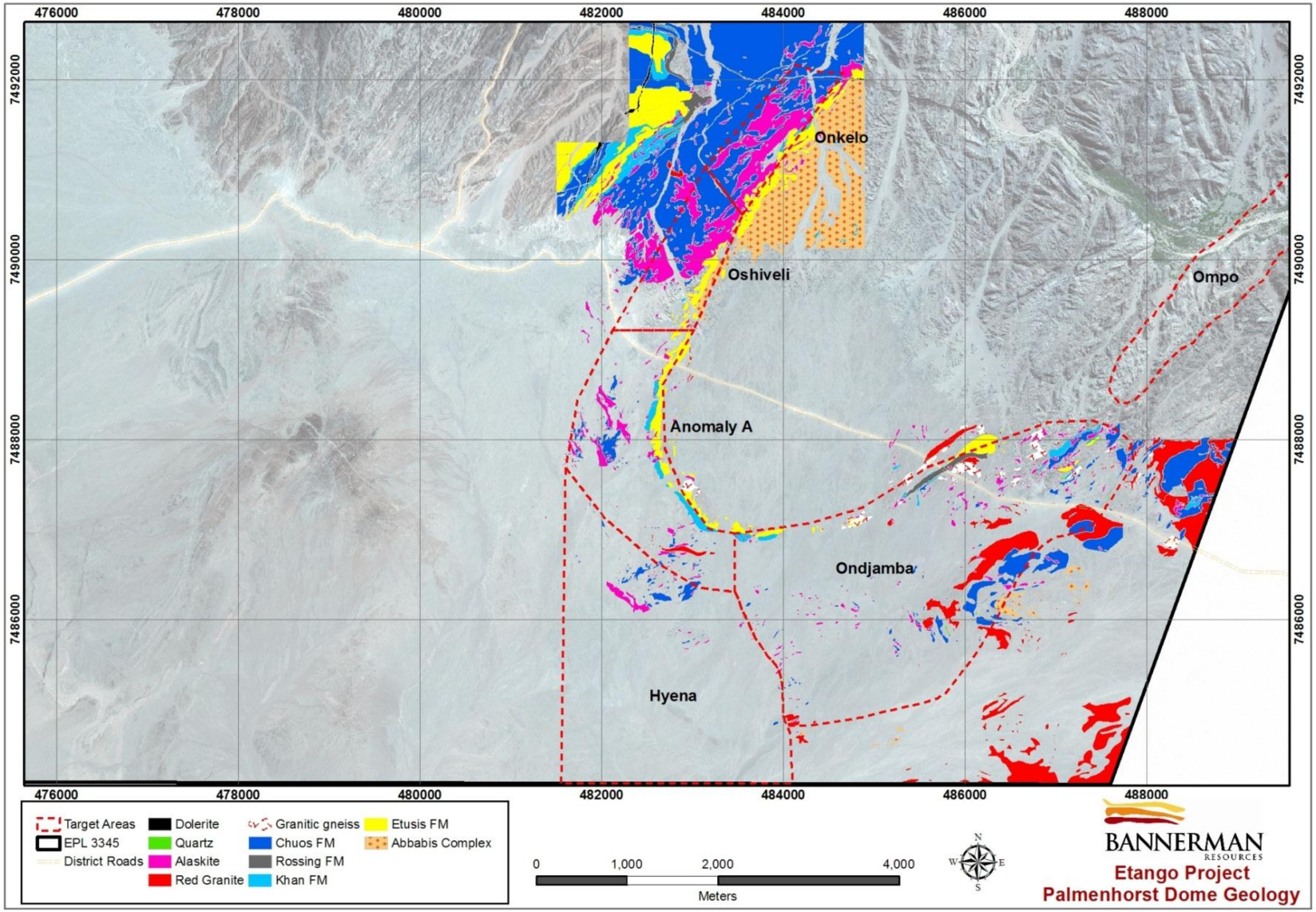


Figure 7-6 Project Geology around the Palmenhorst Dome



The contact with the underlying basement units is transitional and migmatitic in nature, while the upper boundary of the Etusis Formation is arbitrarily defined by the presence of dark biotite gneiss indicating the presence of the more pelitic, and iron-rich, Khan Formation (Mouillac, et al., 1986). Towards the boundary with the Khan Formation schistose bands occur which contain a significant increase in disseminated Fe-Ti oxides. This formation passes gradationally upwards into the Khan Formation over some 20 m with the appearance of, and increase in, the biotite and cordierite content (Nex, 1997).

7.3.2 Khan Formation

Overlying the Etusis Formation is a biotite-quartz gneiss which changes gradationally into a blue-grey quartzo-feldspathic cordierite gneiss with the cordierite porphyroblasts giving a mottled appearance to the rock. This Khan Formation gradationally changes into banded gneiss with the blue-grey banding formed by the colour variations of amphibole and diopside (Nex, 1997).

The Khan Formation can be subdivided into two units: the lower unit is characterised by dark grey biotite-amphibole-pyroxene schist and gneiss (with amphibolite and calc-silicate beds); while the upper unit is characterised by scattered quartz pebbles and is lighter in colour due to a higher quartz and feldspar content and a lower proportion of biotite, amphibole and pyroxene (Mouillac, et al., 1986).

The banded gneiss passes gradationally upward into coarse amphibole-pyroxene gneiss which is also banded with prominent pale green diopside-rich bands. An amphibole-biotite schist forms the topmost unit of the Khan Formation and this can contain minor matrix-supported pebble bands.

7.3.3 Rössing Formation

The Rössing Formation is not prominent in the immediate Anomaly A, Oshiveli and Onkelo Prospect areas and is absent from the proposed mining area. Where present, it has a restricted lateral extent and consists of alternating sequences of diopside marble, quartzite and biotite-garnet schist (Mouillac, et al., 1986).

The contact between the Khan and Rössing Formations is gradational with no indication of an unconformity. In the region, the Rössing Formation has an extremely variable stratigraphy and contains many diverse lithologies; with the base of the Rössing Formation correlated with the first marble band. This marble is frequently impure and dominated by serpentinised porphyroblasts of forsterite. This basal unit is overlain by a succession of interbedded marble, chert, and biotite-amphibole schist with minor impure quartzite. The marble units become increasingly pure higher up in the formation, with the uppermost marble band being succeeded by porphyroblastic schists which are themselves overlain by matrix-supported conglomerates.

Above the conglomerates are impure quartzites which grade upward into distinctive, interbedded, relatively pure, fine-grained, white quartzites and amphibole-diopside bearing calc-silicate rocks.

This unit is succeeded by impure quartzite with minor cm scale beds of conglomerate. Above the impure quartzite is a prominent sulphide-rich quartzite with visible pyrite and chalcopyrite. This is in turn overlain by impure quartzites, biotite schists and porphyroblastic schists, followed by a 7 m thick serpentine-rich marble band. The topmost units of this formation are composed of quartzite which exhibits a prominently eroded surface at its contact with the overlying Chuos Formation (Nex, 1997).

7.3.4 Chuos Formation

The Chuos Formation disconformably overlies the Rössing Formation and is the most prominent marker horizon within the Damara Sequence throughout the inland branch. The basal lithologies of the Chuos Formation consist of pelitic garnet-biotite-cordierite schist. These pass gradationally upward into schists containing thin, laterally impersistant marble and amphibole-rich horizons above which are found the more typical Chuos diamictite.

The Chuos Formation is traditionally described as a tillite and consists of pebbles and boulders of variable size and composition in a brown pelitic matrix. The rocks have an aluminosilicate character and contain abundant biotite, sparse diopside and brownish green amphibole. It is suggested that the Chuos Formation formed as glacio-marine sediments rather than as purely glacial sediments. It consists of a granular unbedded mixtite containing pebbles and boulder-sized clasts, termed diamictite.

The Karibib and Kuiseb Formations, which overlie the Chuos Formation, are not found near the Etango Project area (Mouillac, et al., 1986).

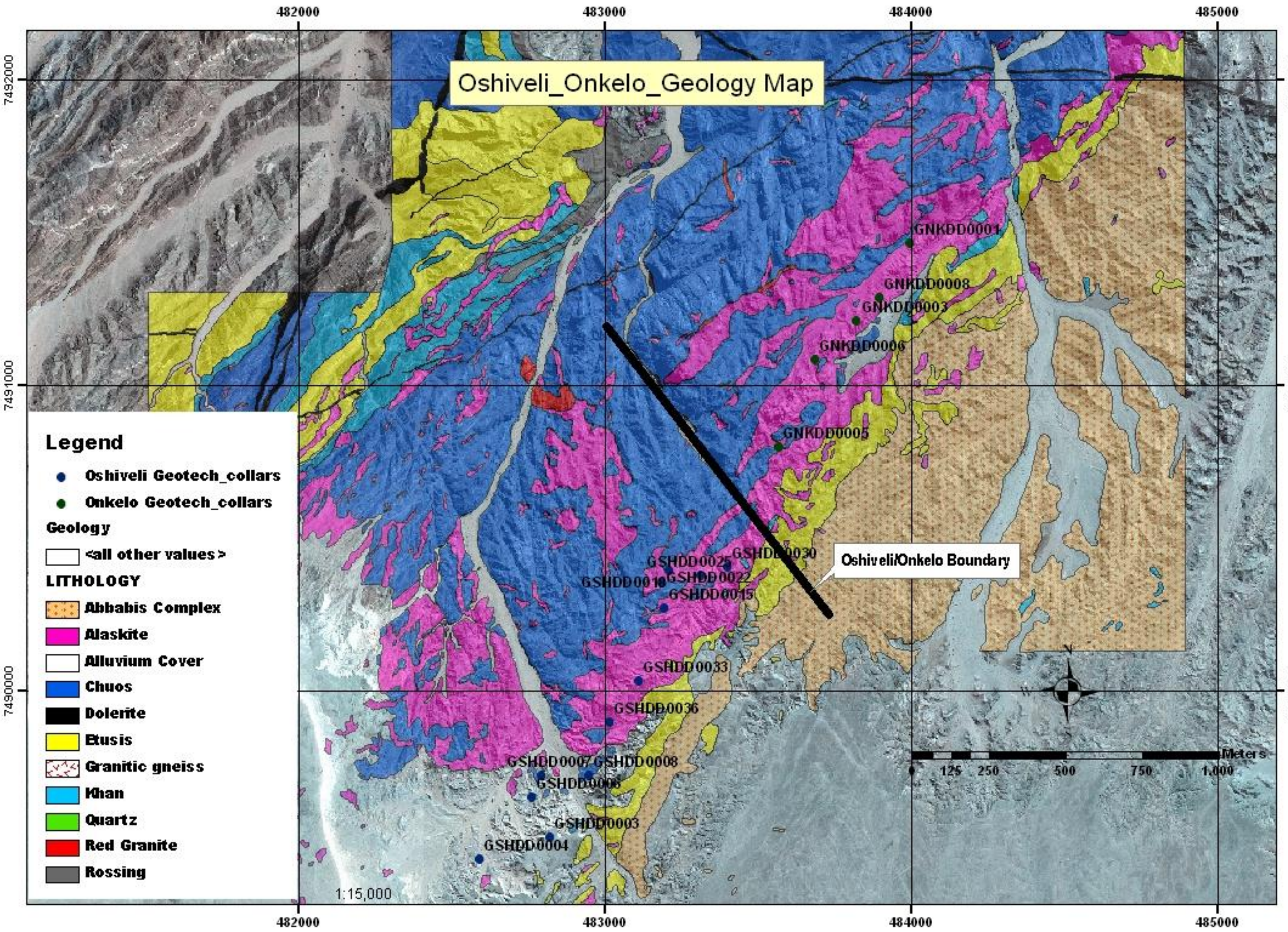
7.3.5 Alaskite

The uraniferous sheeted leucogranite bodies intrude a high strain zone between the pre-Damaran basement and the Damara Supergroup metasedimentary sequence. These bodies are generally referred to as alaskite, which is defined petrologically as a granitic rock that contains less than 5% mafic minerals (Mouillac, et. al., 1986).

The uraniferous intrusive alaskites are late-stage leucocratic granites that often have a pegmatitic texture. However, in the field, local variations in texture and mineralogical composition are common and the composition can vary from alkali-feldspar granite to tonalite (Nex et al., 2001). Mineralogically, the alaskites consist mainly of quartz and feldspar with minor, but variable accessory minerals. Accessory minerals include ilmenite, biotite, apatite, topaz, garnet, tourmaline, uraninite, betafite, zircon, and monazite. Quartz varies in colour from colourless through smoky to almost black (indicating the presence of higher grade uranium mineralisation).

The alaskites are associated with the regional D3 tectonic event and have intruded the Nosib and Swakop Group metasedimentary sequences. They generally occur as bodies parallel to the main S2 foliation (but can sometimes be transgressive to the foliation) and can vary in thickness from a few centimetres to 200m. The alaskite bodies can have a strike continuity of up to several hundred metres, although along the down-dip direction, they can exhibit bifurcation and can truncate after several tens of metres. Crystallisation of the alaskites is interpreted to have occurred pre-, syn- and post- the regional D3 deformation (Mouillac, et al., 1986).

Figure 7-7 Detailed Geological Plan of the Oshiveli and Onkelo areas of the Etango Project showing the abundant alaskite intrusions



Uraniferous alaskite bodies on the NW limb of the Palmenhorst Dome are thought to be confined to re-activated D2 high-strain fabrics with the alaskites generally trending to the north-northeast.

Figure 7-6 illustrates the outcropping surface geology in the EPL3345 tenement area and around the Palmenhorst Dome. Figure 7-7 shows the mapped distribution of alaskites along the western flank of the Abbabis Complex.

7.4 Mineralisation

Uranium mineralisation in the Etango Project area occurs almost exclusively in the alaskite granites. Minor uranium mineralisation is also found in the metasedimentary sequences close to the alaskite contacts, probably from metasomatic alteration and in minor thin alaskite stringers within the metasediments. The main mineralised alaskite bodies are associated with the Khan Formation and the lower part of the Chuos Formation and occur within 400 m of the contact between the Etusis and Khan Formations (Mouillac, et. al., 1986) as shown in Figure 7-8. Major alaskite blows are also found along the Khan/Etusis and Khan/Chuos contacts.

Figure 7-8 View of outcropping alaskite intrusions (light colours) in Etusis Formation (brown on left), Khan Formation and Chuos Formation (grey to right). View looking south from the Onkelo deposit. Width of view about 1km.

The sheeted alaskite bodies have been classified into six types (A to F) by Nex, et al. (2001). In this classification, Types D and E are host to the bulk of the uranium mineralisation at Etango. 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 recognised 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. The "alteration rings" have been interpreted to have formed as oxidation fronts which have affected the distribution of uranium therein (Mouillac, et. al., 1986). Smoky quartz is common and the reddened parts of the oxidation haloes may contain more biotite and Fe-Ti 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. Consequently, these 'types' are irrelevant to the mining and processing of the alaskite.

The dominant primary uranium mineral is uraninite (UO_2), with minor primary uranothorite ((Th, U) SiO4) and some uranium in solid solution in thorite (ThO_2). The 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. 400 ppm 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).

Recent SGS mineralogy (Youlton, et. al., 2010) and earlier scanning electron microscopy studies by Townend (2008) on the mineralised alaskites have also identified other uraniferous minerals. However, there is broad agreement in all of the mineralogy work completed to date on the Etango Project. The most compelling results are those of the SGS (Youlton, et. al., 2010) work which has seen 71 samples studied in detail and hence represents the bulk of the work completed to date.

The minerals uraninite and uranothorite, often with associated weakly-uraniferous thorite, are the main primary uranium-bearing minerals present. Uraninite is not always seen in samples under the microscope, as it is thought to be present as a low-grade background scatter of largish (up to 350 microns) individual crystals. Uranothorite is seen more often, probably because it is generally finer-grained and more dispersed, and hence more easily observed. These two minerals are also associated with thorite, which can be weakly uranium-bearing, and rarely with brannerite. Minor uranium is also present in the minerals monazite, xenotime and zircon, either as minute inclusions or in crystal lattice substitution. Fine crystals of uraninite and uranothorite are also intergrown with monazite, xenotime and biotite, and as inclusions in feldspar, zircon and other minerals. Betafite, which is also a primary mineral, is also present rarely.

The uranium silicate minerals, coffinite and betauranophane, comprised all of the observed secondary uranium-bearing minerals, the two often occurring in the same sample. Coffinite is seen slightly more often, and on occasions, coffinite is observed riming uraninite as an alteration product. The highest grade samples almost always contain coffinite, while betauranophane appears to be more evenly distributed from low to high grade samples.

There is no evidence in any of the petrology and mineralogy work for any clearly definable leucogranite types (i.e. D or E-types) that could be mined as discrete visually, mineralogically or chemically identifiable units. Similarly, 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. There is no perceived zonation of uranium mineralogy with depth, grade, location, bulk rock chemistry, mineralogy or any other feature.

3D computer modelling reveals that the Etango uranium grades are frequently highest within the leucogranite dykes in the Khan Formation and immediate adjacent to the basement and Etusis Formation contact, on the eastern margin of the deposit. The grades then systematically fall to the west down plunge along the leucogranite bodies. This well documented grade distribution is surmised to be an artefact of the deposit genesis process, and hence formed during the deposit origin. While the grade is highest in the east, there is no evidence of this being related to the preferential development of any specific mineral, but rather to a greater abundance of the same minerals.

Another aspect of the Etango grade distribution that should be borne in mind is the fact that within the mineralisation boundaries the average grade, of about 200ppm U_3O_8, is not present as a large number of samples of uniform tenor. Rather the deposit comprises a very large number of analyses in the 100-175 ppm U_3O_8 range, with a small number of much higher grade analyses which bring the average up to the mean ore grade. For instance one assay of 400ppm U_3O_8 and five of 150ppm U_3O_8, (or one assay of 800ppm U_3O_8 and ten of 150ppm U_3O_8) represents the average grade for the deposit. This will be reflected in the deposit mineralogy with a large volume of visually ordinary 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. In other words, it appears that there is a large low-grade background of primary uranium minerals (uraninite and uranothorite). That is then overprinted, partially replaced and upgraded by a, probably more patchy and erratic, secondary mineralisation (event?) which is represented by the locally abundant uranium silicate minerals, coffinite and betauranophane.

No obvious depth variation can be seen in the primary uranium minerals, with uraninite and uranothorite being present in samples from 3m to 487m depth. However, with the secondary uranium minerals, there is some suggestion that coffinite is more common at shallow depths and betauranophane at greater depths, although both actually occur together in the shallowest sample collected at 3m and the deepest at 487m depth.

There is no apparent variation in any uranium minerals from north to south in the Etango deposit, or with the D or E-type logging descriptions.

From a metallurgical perspective it is concluded that the uranium minerals present are amenable to leaching, except for rare brannerite and betafite. Uraninite and uranothorite are observed as fine intergrowths with and as inclusions within monazite, xenotime, biotite, zircon and other minerals. This may lead to their not being accessible, or totally accessible, to leaching, and thereby lower recoveries.

The iron minerals (biotite, chlorite and iron oxides/hydroxides) and carbonate minerals will consume acid during the leaching process, but these minerals are present in low quantities. Therefore the leaching should consume relatively little acid. The presence of minor phosphate minerals could potentially cause uranium to re-precipitate during leaching.

8 DEPOSIT TYPES

Uranium mineralisation at the Etango Project (Anomaly A, Oshiveli and Onkelo deposits) and Ondjamba and Hyena 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 programme 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 drillholes in 1975. From 1976 to 1978 Omitara Mines (a joint venture between Elf Aquitaine SWA and B & O Minerals) (Omitara) drilled 224, mostly short and vertical, percussion drillholes on a reconnaissance grid of fences at 200-400 m spacing (north) by 75-100 m east along the western Palmenhorst Dome position, with the closer spaced fences near the Anomaly A area. These percussion drillholes totalled 13,383 m with depths ranging from 50-100 m. An additional nine diamond drillholes were also drilled for a total of 2,100 m.

Omitara Mines also completed a total of 6,800 m 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 drillholes for 1,017 m and conducted surface scintillometer surveys.

9.2 Exploration by Bannerman Resources

9.2.1 Preliminary Work

After securing its interest in the Etango lease (EPL3345) in 2006, Bannerman undertook a process of capturing and digitising the historical drillhole, 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 the Etango tenement (EPL3345). Reprocessing of this image in the areas near the Swakop River has enabled exposure of the alaskite granites to be readily identified; and together with the airborne radiometric data this 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 10 cm accurate surface digital terrain model (DTM) has been created over the entire Etango Project area.

The core from the nine diamond drillholes drilled earlier by Omitara was re-logged, but was deemed unsuitable for re-assay. A detailed mapping program was then completed along the western and eastern flanks of the Palmenhorst Dome; and the results can be seen in Figure 7-6, above. 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 1970's and early 1980's) and to determine the continuity of uranium mineralisation along strike, at depth and to the west of the Palmenhorst Dome. Subsequently, the 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 drillholes for a total of over 303,500 m, 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 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 drillholes range from 23-480 m in depth and the diamond drillholes range from 101-528 m in depth. The RC drillholes were drilled by Metzger Drilling, using a bit diameter of 4.72" to 5.5". The RC drilling has been conducted on a nominal 50 m x 50 m, to 50 m x 100 m drill spacing, with the bulk of the 50 m x 50 m drilling being completed in the area of the likely open-minable resource. A small area of 25 x 50 m 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 100 m x 50 m spacing.

Due to the shallow dip of the mineralised alaskite bodies (approximately -30-45º to the west) and the inclination of the RC and diamond drillholes (generally -60° to the east), the length of the drillhole intercepts are close to the true thickness of the mineralised intervals, Figure 9-2.

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

Figure 9-1
Drilling Completed at the Etango Project
for the October 2010 Resource Estimate

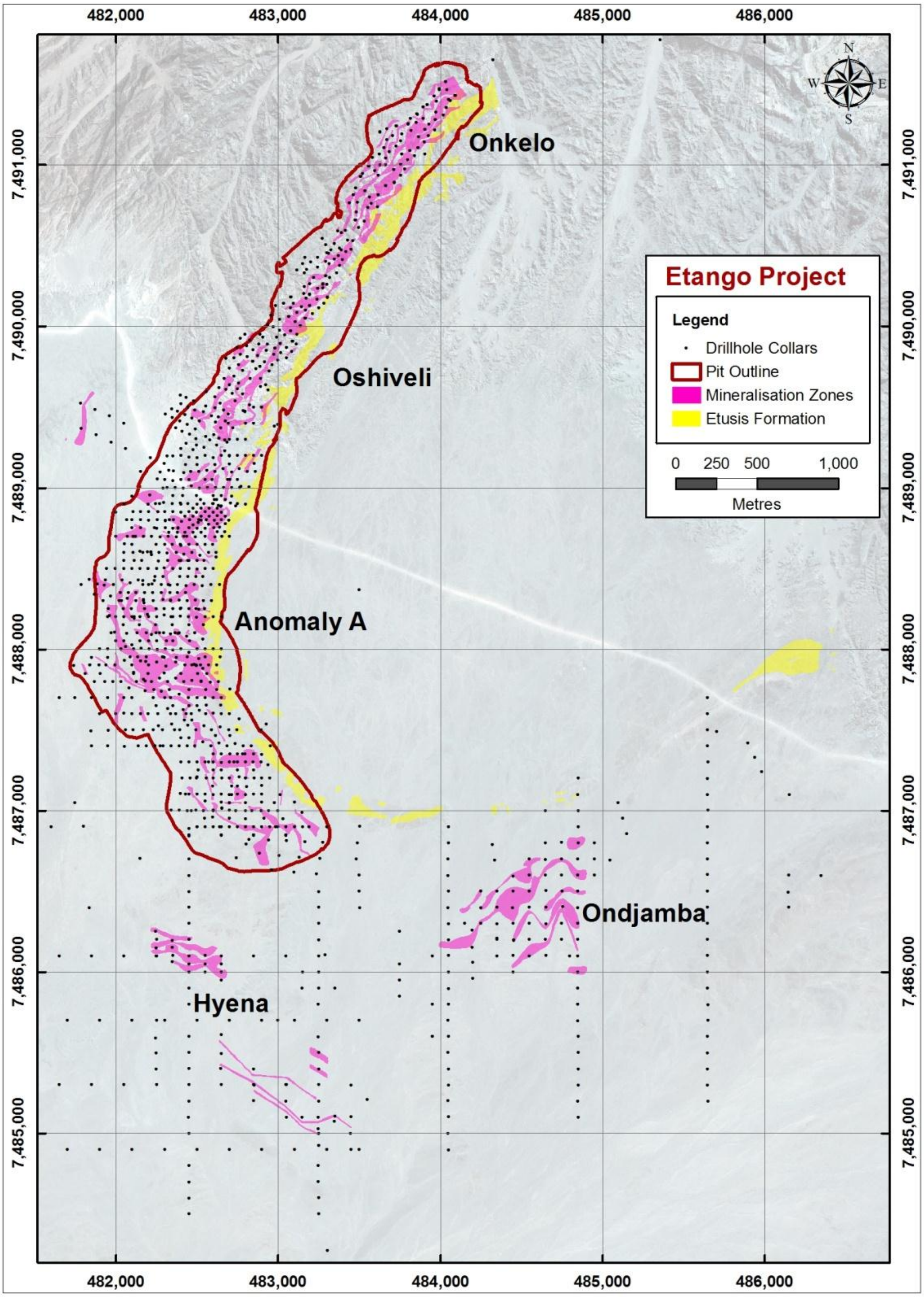
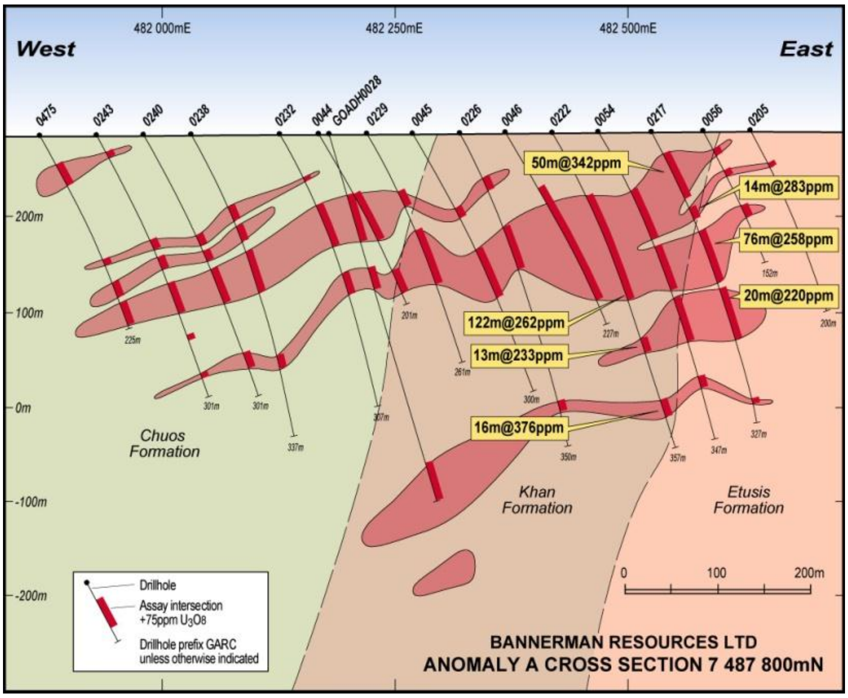


Figure 9-2 Typical Cross-section through the mineralisation at Anomaly A, at the Etango Project



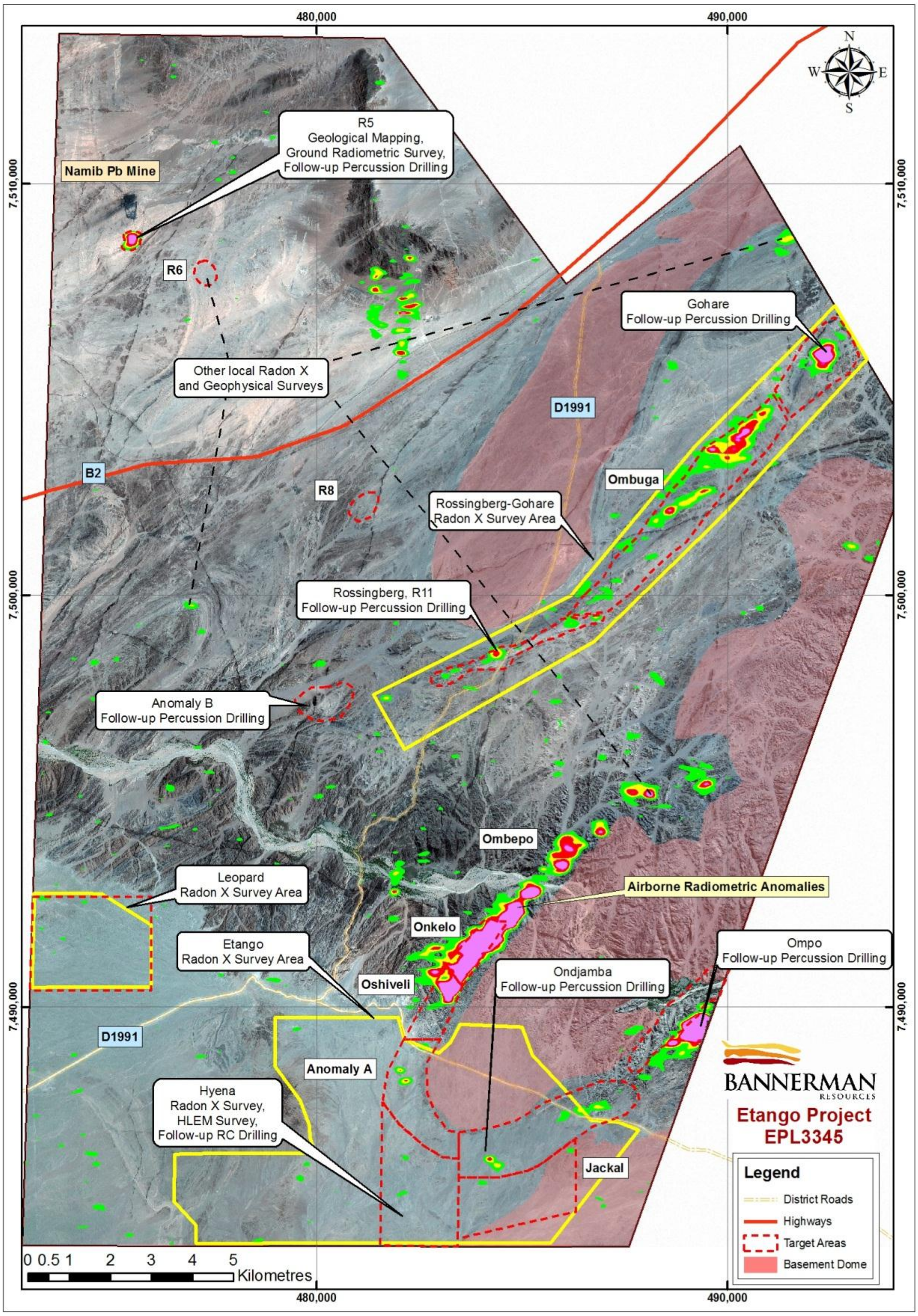
Twenty eight drillholes were also completed in HQ core diameter (63.5 mm core) for metallurgical testwork; and the whole HQ core was sent to Ammtec Laboratories in Perth. Selected core from a total of 22 of the resource definition drillholes was also used for metallurgical testwork.

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**

All drillhole collars have been surveyed by licensed surveyors after drilling. Downhole directional surveys were initially taken using an Eastman single shot camera at nominal 30 m intervals (the first few holes only); however, for the vast majority of holes the practice has been to survey drillholes using a three component Fluxgate Magnetometer survey tool following completion of the drilling.

Figure 9-3 2010 Recent Exploration at the Etango Licence



9.2.3 On-going Exploration

Other areas within the tenement (EPL 3345), in the vicinity of the Etango Project, also 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.

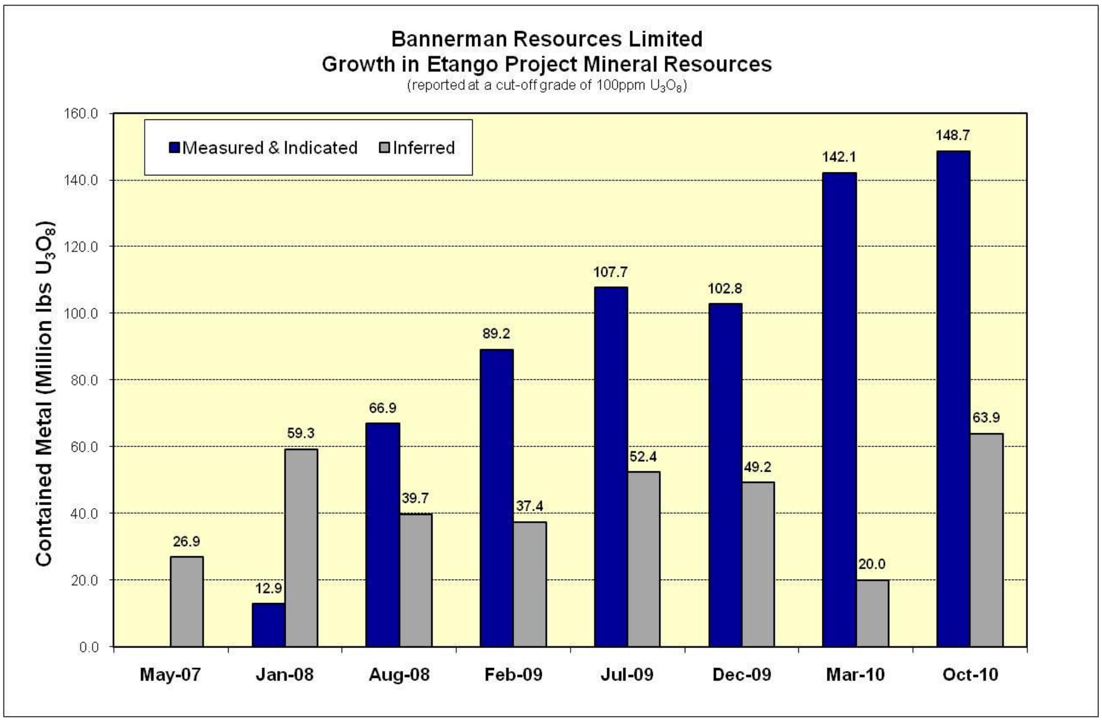
In 2010 and 2011 exploration has continued at the Etango Project and elsewhere within the Etango (EPL 3345) and Swakop River (EPL 3346) licences in Namibia. Further exploration is also planned in 2012 and into the future. Figure 9-3 (above) shows the details of some of the recent work in the Etango licence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

9.2.4 Previous Mineral Resource Estimates

In May 2007, an initial Inferred Mineral Resource was estimated by Bannerman for the Anomaly A deposit, based on the historical and recent drilling. Bannerman has since 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 then completed for the Anomaly A, Oshiveli and Onkelo areas in February 2009, July 2009, March 2010 and now, most recently, by the current Coffey Mining mineral resource estimate in October 2010, Figure 9-4 (see Section 14).

Figure 9-4 Growth of Etango Mineral Resources with Time



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. When taken, chip samples were assayed by X-ray fluorescence. Downhole gamma ray spectrometry was also taken for selected intervals from most of the drillholes.

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 Current Owners

As of 30 June 2011, Bannerman had drilled a total of 1240 RC, 141 diamond and 21 RAB drillholes for a total of over 303,500m on the vicinity of the Etango Project. The RC drillholes range from 23m to 480m in depth and the diamond drillholes 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 drillholes 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 purely 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 drillholes were also completed in HQ core diameter (63.5mm core) for metallurgical testwork; and the whole HQ core was sent to Ammtec Laboratories in Perth.

Due to the shallow dip (approximately -30° to -44º to the west) of the mineralised alaskites and the angle of intercept of the RC and diamond drillholes, 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 drillholes have been completed at the Rössingberg, Ombuga, Gohare, Ombepo, Cheetah and R5 prospect areas.

10.3 Surveying

All drillhole collars are surveyed by licensed surveyors after drilling.

For diamond drillholes, downhole surveys were taken using an Eastman single shot camera at nominal 30m intervals up to drillhole GOADH0022. The practice is now for all drillholes 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 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 drillholes.

The following discussion details the sampling methods used by Bannerman. Bannerman routinely samples all intersected alaskite intervals. The location of the sampling for the resource studies is shown in Figure 10-1.

11.1.1 RC Drilling

The following methodology has been applied to the RC drillhole 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 as shown below in Figure 11-1.
- The 1m sample is split in the field by Bannerman staff using a 75/25 riffle splitter as shown in Figure 11-1 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.

Figure 11-1 RC Sampling at Anomaly A

- 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 drillhole identification and sample depth as shown in Figure 11-1. A collection of the samples are placed into larger plastic bags for transport to the secure sample storage facility on site at Etango as shown in Figure 11-2.
- 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 as shown in Figure 11-3.
- 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 as well as a paper copy which is sent with 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, as shown in Figure 11-2, 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 the 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 January 12th 2009 to SGS Johannesburg for assaying.
- Since December 2008, samples are sent from the Bannerman sample storage facility directly to the SGS Sample Preparation Facility in 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, as shown in Figure 11-3.
- Since December 2007, standards and blanks have been routinely inserted into the sampling stream at a nominal rate of 1:20.

Figure 11-2 The Bannerman RC Sample storage area at the Etango Storage Facility

Figure 11-3 RC Drilling Chip Tray Storage at the Etango Storage Facility

11.1.2 Diamond Drilling

The following methodology has been applied to the diamond drillhole 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, as shown in Figure 11-4, 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.

Figure 11-4 The Bannerman Core Sampling and Logging area at the Etango Storage Facility

- Up to drillhole 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 11-5.
- 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 are sent to SGS Johannesburg for assaying. As with the RC samples, the diamond samples are placed in numbered bags for dispatch.

Figure 11-5 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 although this is statistically insignificant due to the generally low amount 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. 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 drillholes, 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, all 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 transit of the probe and on the 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 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 (Inwood, 2010b) to be of good quality and accuracy for the purposes of mineral resource estimation.

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 drillhole 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 little 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 drillhole 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 for the analysis. Analysis for Ca and K was discontinued in March 2009.

The procedure for analysis at SGS was as follows:

- Upon arrival at SGS Johannesburg (or Swakopmund), a barcode was attached to each sample to enable tracking during the preparation and analysis process.
- The primary sample was dried in an electric oven at ~105°C, then crushed to -2 mm and pulverised to 95% <75 µm using a Labtech LM2 pulveriser.
- Barren rock was run through the crushing and pulverisation circuit after every sample. The last barren rock sample from each batch was analysed using XRF and the value reported for QAQC purposes.
- After pulverisation, a 200 g sub-sample was retained and from this sub-sample, approximately 20 g was taken for XRF analysis and 0.5 g to 2 g for inductively coupled plasma (ICP) mass spectrometry analysis. Typically, SGS Johannesburg conducted an ICP analysis in conjunction with the XRF analysis on every fifth submitted sample.
- SGS Johannesburg also included a standard and blank sample at the rate of 1:22 into the sample stream.
- Replicate samples from the 200 g pulverised sub-samples were taken at the rate of 2:20.
- A pulp duplicate sample was also 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 17 g) was combined with approximately 3 g of wax binder then pressed for 2 minutes to produce a compact pellet. The pellet press was cleaned after each pellet was processed. The Bannerman samples were analysed using a Panalytical Axios XRF machine.
- For Ca and K analyses by borate fusion with XRF, approximately 0.2 g to 0.7 g of sample was 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 200 g sub-samples were 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 was as follows:

- Sample preparation at Genalysis Johannesburg consisted 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 was put through the mill after each sample.
- Every 20^{th} pulverised sample was screen checked to determine the percentage passing 75 µm.
- Analyses for U_3O_8, Th and Nb were determined by pressed pellet XRF using any of a Philips PW1480, PW1400 and PW2400 Axios XRF machines.
- Samples were prepared using 20 g of sample with 3 g of binder which were mixed in a grinding vessel for 4 minutes and then pressed into a pellet in a 20 t hydraulic press.
- One sample of pulp was re-analysed for every 20 samples (as a duplicate) and one reference standard was inserted for every 20 samples.
- One blank sample was also inserted per shift.

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 and were previously taken to the storage facility in Swakopmund and now 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 have 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 and internal laboratory replicates, blanks and duplicate samples.

12.1 Collar and DTM Survey

A topographic survey has been conducted over the project area. The survey was performed by licenced 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 the “Ashtech Solutions” proprietary software.

Most of the drillhole 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 +/- 2 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.

Bannerman Submitted Standards

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 standards ANMIS0085 and AMIS0086 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 these outliers returned results approximating other known standards and can be attributed to sample mixing during the sample submission / recording process. Results that closely compared to known standards were trimmed from the sample population prior to analysis. The summary statistics for these standards are presented in Table 12-1. Summary control plots are in Appendix 1.

Standard AMIS0029, sourced from the Dominion Reef, has a known complex mineralogy and metallurgy which may be affecting the 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 (see Appendix 1).

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 by Bannerman 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.

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

The majority of the blanks submitted to SGS Johannesburg report assays less than 5 ppm 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. Summary control plots are in Appendix 1.

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**
Std Deviation	7.9	3.3	6.1	1	0	4.2	1.8
% in Tolerance	100	100	100	100	100	100	100
% Bias	1.6	3.9	2.1	4.3	0	-0.6	116

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

The SGS Johannesburg blank standard Waste-Rock (n=1,632) exhibits some minor contamination throughout the sample runs and possible incorrect sample identification / submission with eleven samples reporting above 10 ppm U. The laboratory blank (n=6,877) reports consistently at 0 ppm 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 9 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**
Std Deviation	4.02	2.79	2.51	10.21	3.39	5.56	1.12	0.3%
% in Tolerance	100%	79%	86%	100%	100%	100%	100%	99.5%
% Bias	1.3%	6.3%	-2.8%	-1.5%	-0.4%	-1.8%	-3.4%	1.9%

All of the standards, except SARM1, report good accuracy with the bulk of the samples returning assays within the set precision limits. Bias in the laboratory standards varies from -3.5% to 6.3%. Control blank standards (n=210) were identified for analysis (see Appendix 1). 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 and summary charts are presented in Appendix 1. 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

3Pulp 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	Number 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. The samples assayed at SGS Johannesburg show moderate to good precision with the Genalysis duplicates with 74% of RC field duplicates and 73% of the diamond duplicates 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.

<u>Laboratory Pulp Repeats (Replicates)</u>

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 comparative 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.

<u>Inter-Laboratory and XRF versus ICPMS Comparisons</u>

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%).

<u>Discussion</u>

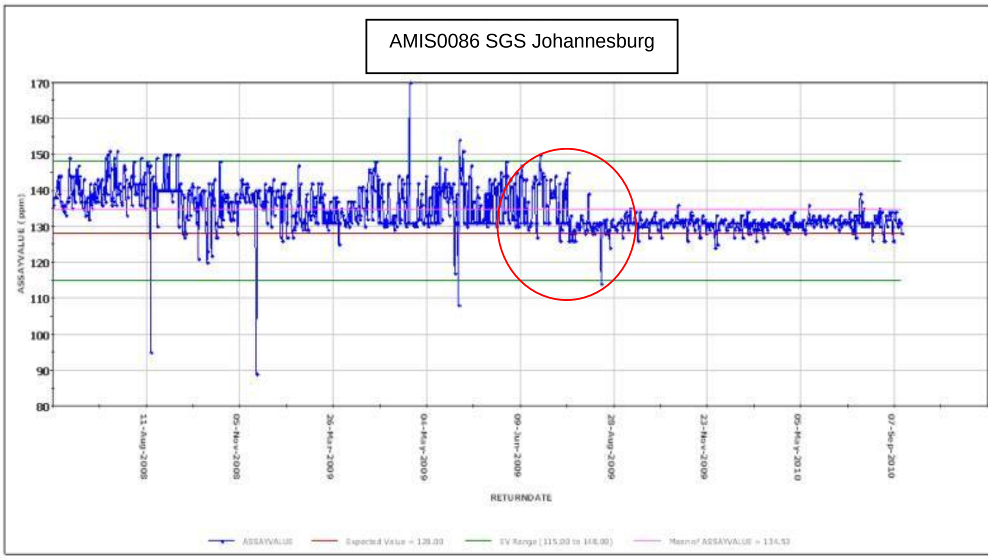
Analyses of the Bannerman inserted standards indicate that the SGS Johannesburg laboratories are reporting a relative bias of between 1% and 8% higher than the expected values for these standards. It is also noted that the SGS internal standards exhibit a bias of 1% to 4%. Genalysis reports a negative 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 however seen 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 and Appendix 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) be regularly sent to Genalysis along with the regular Umpire duplicate samples.
- Intra-Laboratory (i.e. same 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 (Figure 12.2). A total of 40 RC samples from

GARC0362 were placed into plastic bags with numbered security tags attached by the author directly after drilling and splitting in the field. 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.

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.

Figure 12-2 Etango Project

Samples Tagged for Independent Sampling

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 drillhole 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

The metallurgical testwork undertaken since September 2008 on the Etango uranium resource has been focussed on:

1. Developing an understanding of the mineralogy of the Etango resource,
2. Identifying the most practical and cost effective leaching option,
3. Measuring standard comminution parameters to enable simulation and design of an appropriate comminution circuit
4. Investigation of the potential to decrease capital and operating costs with the application of a pre-concentration stage,
5. Define process parameters to enable engineering design for capital and operating cost estimation

Appropriate samples were selected in consultation with the project geologist and metallurgical testwork was initiated to complete the pre-feasibility study objectives.

Metallurgical testwork has been conducted over the period 2007- present (September 2011) in progressive phases supporting ore characterisation, scoping and option studies, PFS development and trade-off/ value engineering studies.

The chronology of the metallurgical testwork program and the related engineering studies is summarised in the following table.

Table 13-1 Chronology of testwork and related engineering studies

PERIOD	ITEM
Jul-08	Full HQ (96mm φ) core samples received in Perth & testwork commenced.
Oct08- Feb09	Bannerman internal investigation of alternative processing options. Development and scoping testwork targeted head grade (beneficiation- radiometric sort; upgrade by size; density based). The coarse beneficiation approaches were not economically successful under the assumed technical conditions (ore characteristics and beneficiation circuit performance), predominantly due to the inability to produce a beneficiation tail stream of low grade to allow discarding, so both concentrate and tails streams required treatment with increased capital and operating costs. Undertook PFS with GRD Minproc on the basis of whole of ore heap leaching and commissioned a metallurgical testwork program to assess the performance of batch centrifugal concentration (gravity concentration) on Etango ore samples. The latter proved unsuccessful as a significant fraction of the contained uranium mineralisation was contained within composite particles of apparently insufficient density or fine size to affect efficient separation by the laboratory scale batch centrifugal concentration equipment employed.
Mar-09	Heap Leach Open Circuit "1M" Column Tests: -12.5mm and HPGR Centre Product. 3x tests at Ammtec March 2009 on Client supply of Whole HQ Core 1. 1m Open Circuit Column Tests: MH8112, MH8113, MH8114. 2. -12.5mm Conventional crush full scale PSD and HPGR Centre Product.

May-09	University of Witwatersrand Mineralogy / QEMSCAN study of samples from Bannerman.
Jun-09	Heap leach column testwork commenced at Ammtec (4x 2m columns. 2 columns with conventional crush to P_{80} = 7.8mm and acid cured feed; 2 columns with HPGR crush to P_{80} = 4mm +agglomeration with acid and binder. Results: Option A: - maximum extraction = 82% for the open circuit column and 78% for the closed circuit after 65 days leaching. Option E: maximum extraction was 93% after 35 days for the open circuit column, & 90% after 65 days in closed circuit leaching
Jul-Sep 2009	Heap Leach testwork at P_{80} of 4.0 mm was selected as the preferred option for further development as a 7m heap. A new Flotation based process option arose. This consisted of two stage conventional crushing, HPGR crushing, grinding, flotation and agitated leaching of the concentrate. This concept compared favourably against Heap leach. Initial scoping test showed promise. Flotation development work was fast tracked in Nov-Dec2009. The follow-up testing showed insufficient recovery to justify further study. The option was rejected as not commercially viable due to the low recovery. Plans to pilot in February 2010 were cancelled.
Sep-Oct 2009	2m HL column development trials at Ammtec. (filled to ~1.86m with ore). 3x3 matrix trials assessed [<12.5 mm final screened ore; <11.2 mm HPGR; <8 mm HPGR] vs. [10,15,20 g/L Free Acid]
Dec 2009	Percolation test results - Heap Leach Percolation Tests for -6.3mm, -9.5mm & -12.5mm crushed ore sizes with varying concentrated acid addition as "binder".
Apr 2010	Agitated Leach variability testwork - 21x drill core samples & 5 composites; P_{80} 300 micron; 45°C; 40% solids; $FeSO_4$ added to ~ 2/3rds of tests; Eh target ~500mV maintained with pyrolusite over 24hrs. Free Acid maintained at 5 gpl.
Jun-Jul 2010	Ammtec Agitated leach testwork - recovery & filterability vs. grind size of P_{80} = 300,400, 500, 600, 700, 800, 1000 micron of Etango Pilot composite ore.
Aug-Nov 2010	Ammtec 190 mm φ Column leach testwork - 1x 7m; 2x 4m in series; 3x 2m closed circuit.

The testwork programs and results are detailed in the following sections.

13.2 Sample Description

Samples were provided as whole HQ[1], ½ NQ1 and ¼ NQ core. NQ core was retained for planned variability testing to follow the current programme of testwork.

Whole HQ core was selected, drilled and supplied specifically for metallurgical testing and formed the basis for the work reported here. Sample identification, lithology and radiometric grade are provided in the sections below for each stage of study.

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

A number of composite samples have been tested throughout the various programs engaged in the pre-feasibility study.

13.2.1 Ore Types

Metallurgical samples comprised a number of material types, while the bulk of resource tonnage was represented in four (4) predominant types. Interrogation of the client's resource data base indicated the following volumetric proportions by main type:

- Type D Alaskite[2] - representing approximately 65% of core examined.
- Type E Alaskite2 - representing approximately 22.5%.
- Chuos[3] Gneiss – glacial sediment representing approximately 6.4% of core and typically found at Type D and Type E boundaries.
- Khan Gneiss - sediment representing approximately 6% of core and also occurring typically at the Type D and Type E boundaries.

Metallurgical testing proceeded on the basis of selecting intervals of core within the above four main ore types, 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 lithologies found little or no significant difference for metallurgical performance[4]. 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 comprising $^2/_3$ Type D and $^1/_3$ Type E and submitted for ICP multi element scan and chemical assay. The results for major analyses are provided below. Full details of multi element analysis are provided in the main report appendix.

Table 13-2

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

[2] The type D & E classification originates from geological assessment of the region by Nex et al. (2001)"*Petrology, geochemistry and uranium mineralisation of post-collisional magmatism around Goanikontes, southern Central Zone, Damaran Orogen, Namibia".* The D & E type alaskites are the principal hosts for uranium mineralisation.

[3] Local formation name.

Table 13-2

Assay Uranium and Potential Organic Co-extracted Species

Analytes	Species	Method	Detection Limit	Unit	Assay
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. Assay values presented in Table 13-2 show uranium as well as species that potentially co-extract with uranium either quantitatively or partially.

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

Table 13-3

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

The assay of potential organic loading retardants is shown above and represents generally low levels. Chloride analysis was extended to include total and water soluble forms.

Table 13-4

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

In the context of solvent extraction with Alamine 336, the species shown in Table 13-4 are typically rejected and again, represent generally low levels. The Iron assay represents the amount of natural total Iron in the ore,

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 Scanning Electron Microscopy (SEM) Analysis

Mineralogical identification and deportment was evaluated via SEM based on core samples selected over 100m of HQ[5] drill hole (GOADH0048).

Consistent with the Atomic Energy Board (1980) ('AEB') report, samples were classified as coarse grained biotite granites (Uraniferous Alaskite) with dominant 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 $(U,Th)SiO_4$ and 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.

The AEB report previously reported acid soluble boltwoodite $K(UO_2)_2(SiO3)_2(OH)_2.5H_2O$ and the refractory double oxide betafite $(Ca,U)_2(Ti,Nb,Ta)2O_6(OH)$.

Uraninite and Uranothorite are tetravalent and as such have a low natural solubility in dilute acid and oxidation to the hexavalent state is essential for economic recovery. Ore mineralogy can be oxide or silicate based. Silicates include Boltwoodite as referenced in the AEB report.

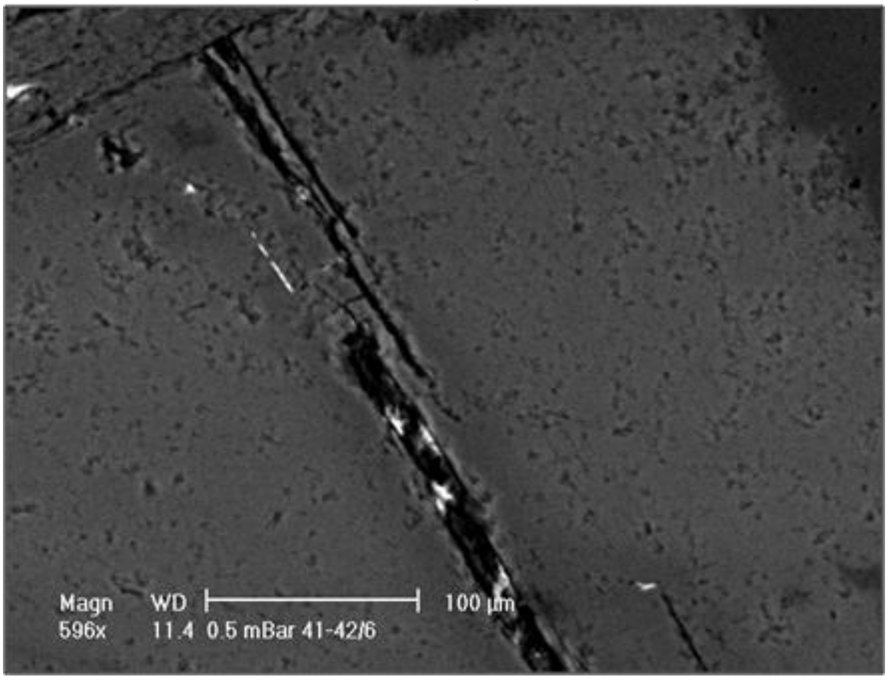
Down-hole mineral deportment is discussed below, based on the initial evaluation of GOADH0048.

<u>GOADH0048 41m to 42m Interval</u>

Uranium mineralisation was identified as uraninite and brannerite. Uraninite occurred as sub 20µm particles in fracture. Fracture was partially in-filled with carbonates, typically calcite and an unidentified cerium oxide.

Figure 13-1

GOADH0048: 41-42 m Sub 20 µm Uraninite in Fracture

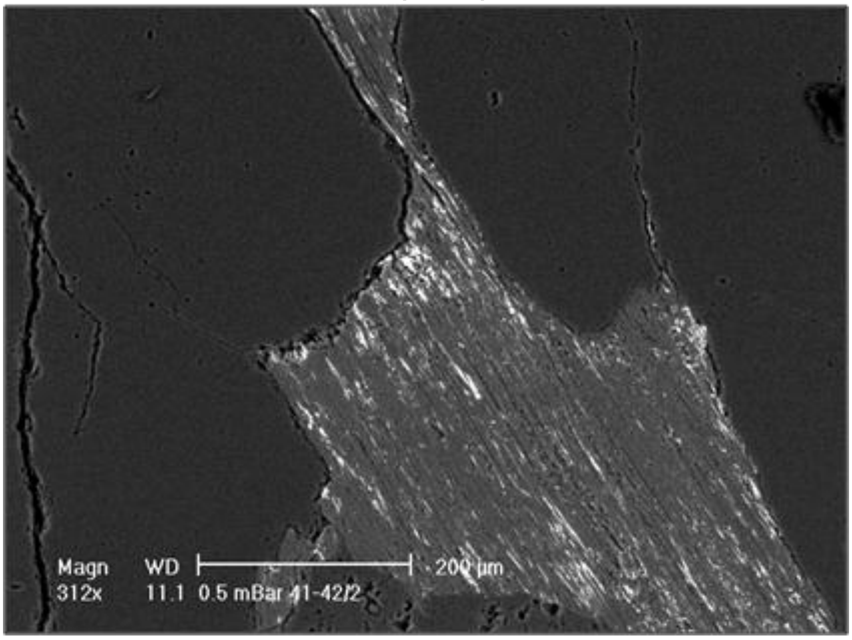


[5] HQ = 96mm drill hole; NQ = 75.7 mm drill hole

Brannerite occurred as 50 to 100µm lenticular grains within basal cleavage planes of phyllosilicate minerals, biotite and chlorite and also with numerous sub 20µm strips within the core of biotite. Brannerite was also identified in minor proportions as 100µm strips in plagioclase feldspar cleavages.

Figure 13-2

GOADH0048: 41-42 m Sub 50 µm Strips Brannerite Within Biotite

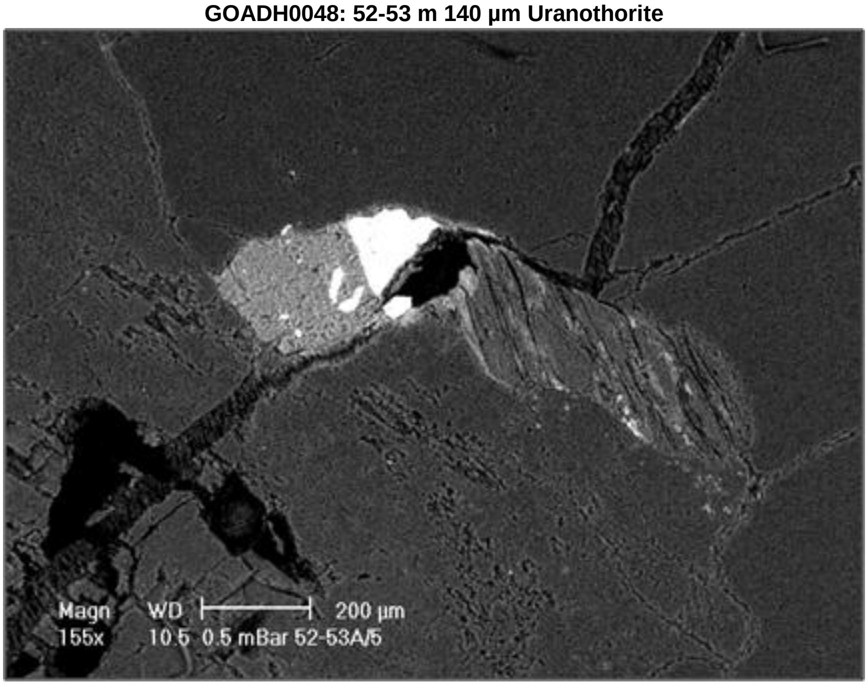


GOADH0048 52-53m Interval

Two thin sections were prepared based on the variation between proportions of plagioclase and microcline feldspar. In contrast to other samples where uranium was identified predominantly within fracture planes, mineralisation was found to occur as discrete grains in granite. Uranium was identified as uranothorite and minor polycrase in addition to uranium and thorium associated with phosphate minerals, monazite and xenotime.

Figure 13-3

GOADH0048: 52-53 m 140 µm Uranothorite



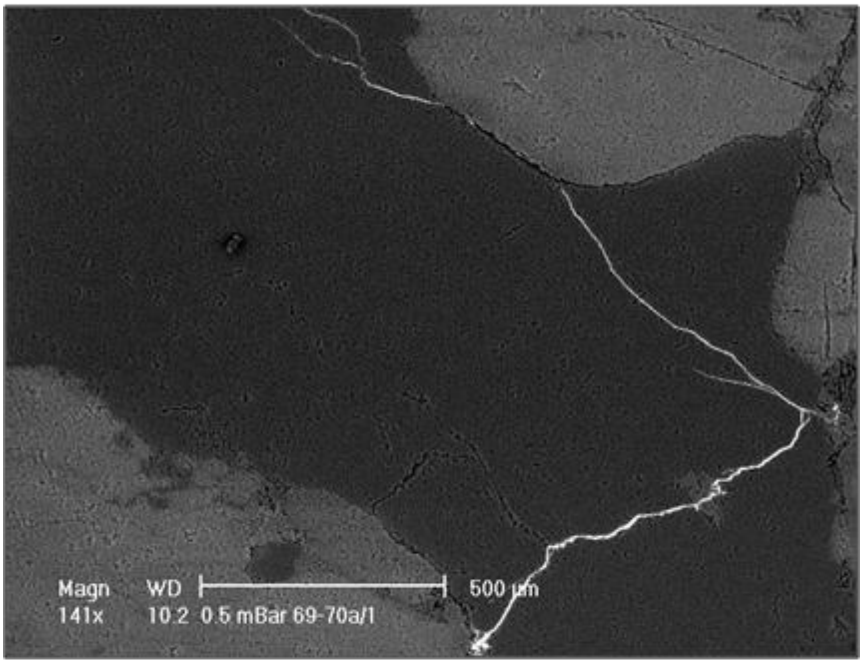
GOADH0048 69-70m Interval

Two thin sections were prepared based on variation in fracture, veins and alteration. Uranium mineralisation was identified as uraninite and uranothorite (assaying approximately 15% UO_2), as well as uraniferous monazite.

Uraninite occurred as variable length 10 - 1,000µm narrow veins through quartz and plagioclase and as narrow 10 - 40µm bands at quartz plagioclase contacts. Uraninite was also identified as narrow 150 - 300µm veins in chlorite and as 100-200µm grains within fracture, partially in-filled with calcite and secondary silicates.

Figure 13-4

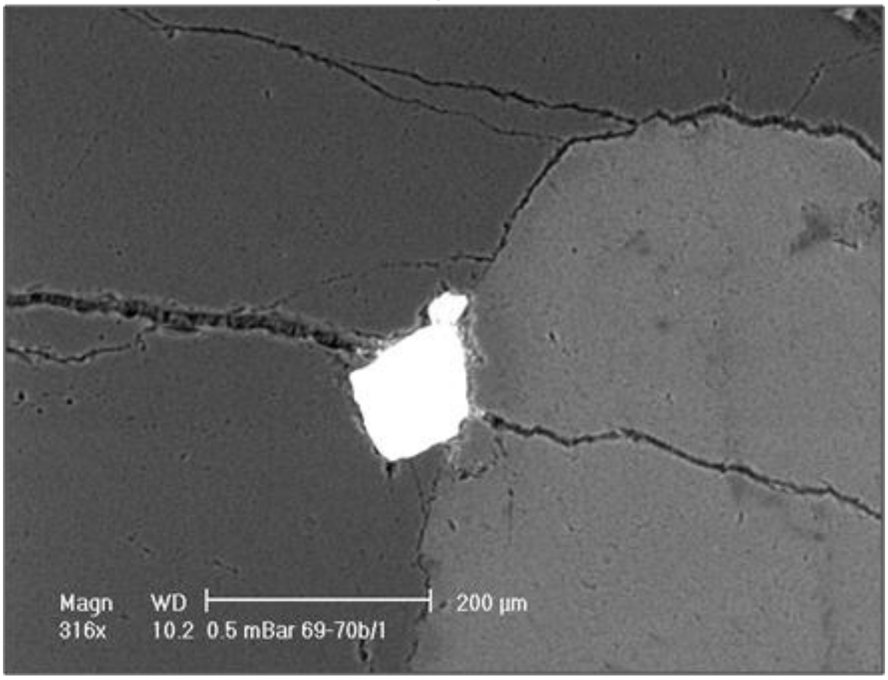
GOADH0048 69-70 m Uraninite Veins in Plagioclase



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 fracture through plagioclase and feldspar, indicating the potential for uranium mineral exposure at coarse size.

Figure 13-5

GOADH0048 69-70 m 100 μm Uranothorite at Contact



13.3.2 QEMSCAN Analysis

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

Uranium Deportment by Mineral Phase

The deportment of uranium associated with each uraniferous mineral phase is shown below, with the dominant mineralisation identified as uraninite and uraniferous silicates.

Uraniferous Silicates where identified as Coffinite and SEM analysis also identified Boltwoodite and Uranothorite. Uraniferous Phosphate mineralisation was identified as Autunite.

Table 13-5

Uranium Deportment by Mineral Phase

Sample Number	**DH-010-2**	**DH-010-5**	**DH-010-7**	**DH-010-7**
Size Fraction (μm)	**-355 μm/+208 μm**	**-208 μm/+90 μm**	**-355 μm/+208 μm**	**-208 μm/+90 μm**
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 the table below and is consistent with the SEM analysis of metallurgical core.

Table 13-6

QEMSCAN Modal Abundance

Sample Number	**DH-010-2**	**DH-010-5**	**DH-010-7**	**DH-010-7**
Size Fraction (µm)	**-355 µm/+208 µm**	**-208 µm/+90 µm**	**-355 µm/+208 µm**	**-208 µm/+90 µm**
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**

Liberation Analysis

SEM analysis indicated that the upper size fraction in the "bimodal distribution" of uranium mineralisation should be liberated at relatively coarse size. QEMSCAN liberation class data is provided below 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 13-7

Uraninite Liberation Class Data

Classification		**Locked**	**Middlings**	**Liberated**	**Total**
Sample Number	**Size Fraction**	**Area <= 30%**	**Area >30% <=80%**	**Area >80%**	
DH-010-2	-355 µm/+208 µm	96.6	1.5	1.9	100.0
DH-010-5	-208 µm/+90 µm	48.9	27.1	23.9	100.0
DH-010-7	-355 µm/+208 µm	99.9	0.0	0.1	100.0
DH-010-7	-208 µm/+90 µm	18.8	64.0	17.2	100.0

The liberation of the upper size fraction in the bimodal distribution of uraninite occurs at a relatively coarse size as seen by comparing the proportions of liberated and middlings fraction appearing within the -355µm/+208µm and -208µm/+90µm fractions.

The same analysis was applied in the case of Uranium Silicates and presented in Table 13-8.

Table 13-8
Uranium Silicate Liberation Class Data

Classification		**Locked**	**Middlings**	**Liberated**	**Total**
Sample Number	**Size Fraction**	**Area <= 30%**	**Area >30% <=80%**	**Area >80%**	
DH-010-2	-355 µm/+208 µm	44.0	22.5	33.5	100.0
DH-010-5	-208 µm/+90 µm	21.0	24.4	54.6	100.0
DH-010-7	-355 µm/+208 µm	99.6	0.4	0.0	100.0
DH-010-7	-208 µm/+90 µm	31.8	2.1	66.1	100.0

Uranium Silicate minerals show greater liberation than Uraninite at the relatively coarse size. As for the Uraninite, the overall liberation is likely to be limited by the bimodal size distribution with a significant proportion of the mineralisation < 25µm grain size.

Table 13-9 provides the same analysis including all uraniferous minerals identified.

Table 13-9
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	-355 µm/+208 µm	60.1	8.4	31.5	100.0
DH-010-5	-208 µm/+90 µm	24.7	21.7	53.7	100.0
DH-010-7	-355 µm/+208 µm	99.6	0.0	0.4	100.0
DH-010-7	-208 µm/+90 µm	24.7	42.2	33.1	100.0

13.3.3 Conclusions

The grain size and deportment of uranium mineralisation along cleavage planes; within and intersected by fracture indicates that ore grade fraction is substantially exposed at coarse size. The granitic nature of the rock, associated with feldspar and biotite, with little evidence of major acid consuming minerals supports the appropriateness of a sulphuric acid leach to extract uranium.

13.4 Comminution Characterisation

The comminution properties of Etango ore were characterised based on selected intervals of whole HQ1 core drilled and supplied specifically for metallurgical testing. Figure 13-6 shows a tray of samples. 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 below:

Preliminary characterisation based on selected intervals abstracted from GOADH0048.

Variability testing utilised intervals derived from GOADH0048, GOADH0058, GOADH0059 and GOADH0060.

High Pressure Grinding Rolls (HPGR) Pilot testwork was performed on selected intervals abstracted from GOADH0062, GOADH0063, GOADH0064, GOADH0065 and GOADH0066.

Figure 13-6

Etango Metallurgical Core Samples

Interval selection was based on 10% dilution. Dilution typically occurred either as Khan and Chuos Gneiss metasediments resident at the edges of ore grade boundaries, or waste grade Type D and Type E Alaskite. Specimens of alaskite and metasediment rock types are shown in Figure 13-7.

Figure 13-7

Drill chips of Etango rock types

13.4.1 Glossary of Abbreviations

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.

13.4.2 Preliminary Characterisation GOADH0048

Approximately 100m of whole HQ[6] core from diamond hole GOADH0048 was used in the preliminary characterisation; preparing 5m and 6m composites of supplied 1m intervals as shown in Table 13-10.

Results from standard Bond suite and JK SMC testing are presented in Section 13.4.3; while results from JK Drop Weight testing performed on composite (Comp-48DWi) are reported below.

Table 13-10

Comminution Test Intervals GOADH0048

Interval From (m)	Interval To (m)	Interval (m)	Composite Number	Test Scope
35	40	5	Comp-48-35-40	UCS, SMC, CWi, RWi, BWi, Ai
61	66	5	Comp-48-61-66	
95	100	5	Comp-48-95-100	
34	101	6	Comp-48DWi	JK Drop Weight Index
34	101	Comp-48DWi Test Residue	-	BWi

<u>JK Drop Weight Test Comp-48 DWi</u>

A single 6m composite was prepared across all intervals and subjected to a full JK Drop Weight test. This single test was conducted for the following purposes:

- To provide first pass comminution data for initial crusher and mill equipment selection.

[6] 96mm diameter Drill Core

- To provide a calibration reference for further JK SMC tests and calculated Drop Weight Index (DWi) values obtained as part of the comminution variability testwork described in Section 13.4.3

The full JK Drop Weight Index test provides a measure of the resistance of the ore to comminution via impact and abrasion and is used for crushing and SAG mill equipment selection.

The results indicate that the ore displays a relatively low resistance to impact breakage, with A*b values in the range 67 – 127 representing soft ore (low impact breakage resistance). Values in the range 56 – 67 are typically taken as representing moderate to soft ore. This implies no difficulty crushing and no issues for SAG mill style impact comminution.

Abrasion resistance testwork measured a t_a value of 0.48. This indicates a medium resistance to abrasion style comminution. Typically values in the range 0.41 – 0.54 represent medium abrasion resistance, with values ranging between 0.54 – 0.65 representing moderate to soft ore.

13.4.3 Comminution Variability

Comminution variability testing was performed on whole HQ test core abstracted from: GOADH0048, GOADH0058, GOADH0059 and GOADH0060 and summarised below for the two main Alaskite types2, Type D and Type E.

The variability was assessed using the following standard tests[7] on a number of ore samples: UCS tests; Bond Crushing Index tests; Bond Abrasion Index tests; JK SAG Mill Comminution SMC tests; Bond Rod Mill Work Index tests, and Bond Ball Mill Work Index tests.

For each test, the resultant data was fitted to a probability distribution that best represents the variability measured.

<u>UCS</u>

UCS testing was used to determine the unconfined compressive strength of rock specimens according to ASTM D2938-95. The test involved the selection of core samples with a length to diameter ratio greater than 2.5:1.

A total of 20 UCS tests were performed and summarised in Figure 13-8, Table 13-11 and Table 13-12, split into the two main lithologies, Type D and Type E Alaskite[8].

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[7] 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.

[8] The D & E type alaskites are the principal hosts for uranium mineralization in EPL3345 Etango prospects including Anomaly A, Oshiveli, Onkelo, Hyena.

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; the plotted data indicates slightly higher UCS values for Type E Alaskite compared to Type D.

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

Figure 13-8

Comminution Variability – Unconfined Compressive Strength

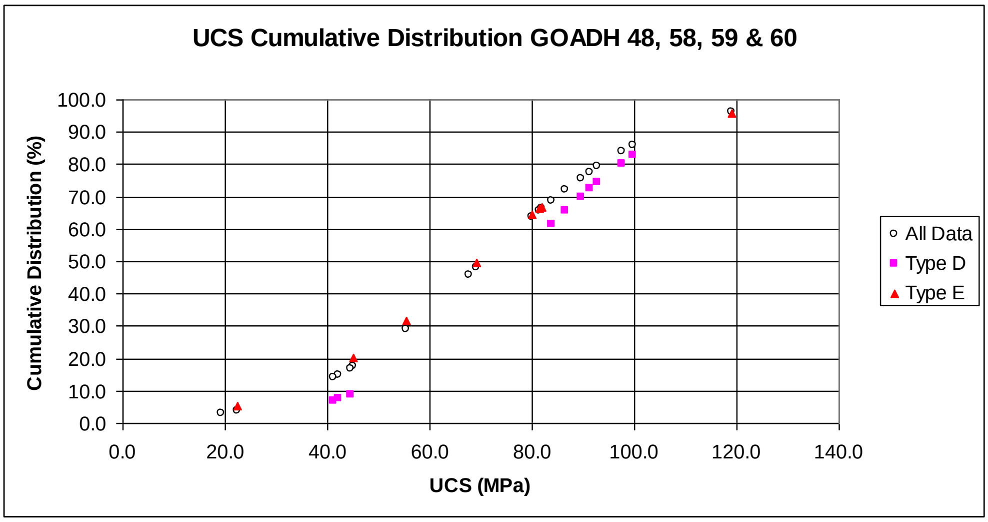


Table 13-11 Comminution Variability UCS Type D Alaskite					
Hole ID	**Interval From (m)**	**Interval To (m)**	**Interval (m)**	**U_3O_{8e} (ppm)**	**UCS (MPa)**
GOADH0048	35.0	100.0	15.0	361	90.3
GOADH0058[9]	14.0	14.6	4.6	464	42.3
GOADH0059	93.0	205.0	3.0	647	92.5
GOADH0060	41.0	92.0	3.0	248	59.5
Average				**400**	**78.3**
Standard Deviation				**191**	**24.1**
Minimum				**194**	**41.1**
Maximum				**840**	**99.7**

Table 13-12 Comminution Variability UCS Type E Alaskite					
Hole ID	**Interval From (m)**	**Interval To (m)**	**Interval (m)**	**U_3O_{8e} (ppm)**	**UCS (MPa)**
GOADH0058	35.0	179.0	5.0	288	57.3
GOADH0059	28.0	173.0	3.0	375	89.4
Average				**320**	**69.3**
Standard Deviation				**127**	**29.0**
Minimum				**167**	**22.3**
Maximum				**467**	**119.0**

[9] GOADH0058 and GOADH0059 both contain Type D and Type E Alaskite.

Bond Crushing Index

A total of 18 Bond Crushing Work Index determinations were performed. Results are summarised in Figure 13-9, Table 13-13 and Table 13-14, split into Type D and Type E lithologies.

Average values of 8.0Kwh/t and 8.3Kwh/t for Type D and Type E respectively are below the typical limit of 10Kwh/t indicative of no issues with crushing performance. Maximum values obtained as 16.4Kwh/t for Type D and 16.5Kwh/t for Type E are also considered indicative of no issues related to performance and also indicate no issues with regard to SAG mill critical size build-up.

Figure 13-9

Comminution Variability – Bond Crushing Index

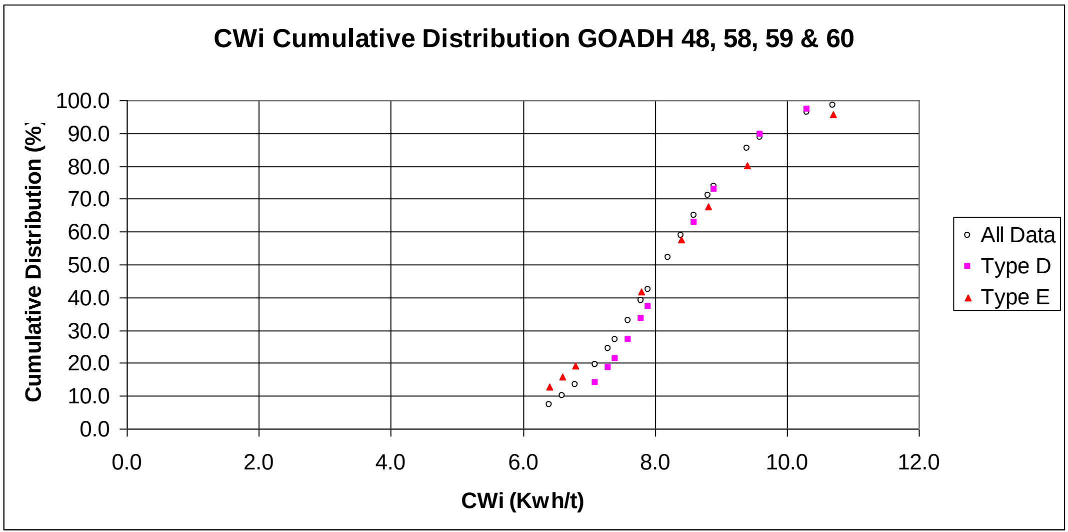


Table 13-13 Comminution Variability Crushing Work Index Type D								
Hole ID	**Interval From (m)**	**Interval To (m)**	**Interval (m)**	**U_3O_{8e} (ppm)**	**Mean CWi (kWh/t)**	**Min CWi (kWh/t)**	**Max CWi (kWh/t)**	**Standard Deviation CWi (kWh/t)**
GOADH0048	35.0	100.0	15.0	361	8.7	5.8	14.1	2.4
GOADH0058	9.0	14.6	5.6	464	7.1	3.5	12.3	3.0
GOADH0059	91.0	207.0	15.0	647	7.6	4.2	13.3	2.3
GOADH0060	39.0	95.3	15.6	248	8.8	4.7	15.4	2.7
Average				**422**	**8.2**	**4.8**	**14.1**	**2.5**
Standard Deviation				**191**	**1.1**	**1.5**	**1.5**	**0.5**
Minimum				**194**	**7.1**	**3.5**	**12.3**	**1.9**
Maximum				**840**	**10.3**	**7.3**	**16.4**	**3.5**

Table 13-14 Comminution Variability Crushing Work Index Type E								
Hole ID	**Interval From (m)**	**Interval To (m)**	**Interval (m)**	**U_3O_{8e} (ppm)**	**Mean CWi (kWh/t)**	**Min CWi (kWh/t)**	**Max CWi (kWh/t)**	**Standard Deviation CWi (kWh/t)**
GOADH0048	35.4	120.3	15.2	280	6.6	4.1	13.6	2.4
GOADH0058	146.0	151.0	5.0	310	9.4	7.2	16.2	3.0
GOADH0059	176.0	181.0	5.0	411	9.3	4.8	13.8	2.5
GOADH0060	169.0	174.0	5.0	215	7.8	5.4	12.7	2.0
Average				**326**	**8.1**	**4.9**	**13.9**	**2.5**
Standard Deviation				**127**	**1.5**	**1.3**	**1.6**	**0.4**
Minimum				**167**	**6.4**	**3.4**	**12.3**	**1.9**
Maximum				**467**	**10.7**	**7.2**	**16.5**	**3.0**

Bond Abrasion Index

A total of 18 Bond Abrasion Index determinations were performed utilising residues from UCS testwork. Test results are summarised in Figure 13-10, Table 13-15 and Table 13-16, split into Type D and Type E Alaskite lithologies.

The distribution of values shown in Figure 13-10 and average abrasion index values of 0.336 for Type D and 0.274 for Type E; show that Type D Alaskite is more abrasive than Type E. The abrasion values measured indicate an ore feed with a moderate abrasion potential.

Figure 13-10

Comminution Variability – Bond Abrasion Index

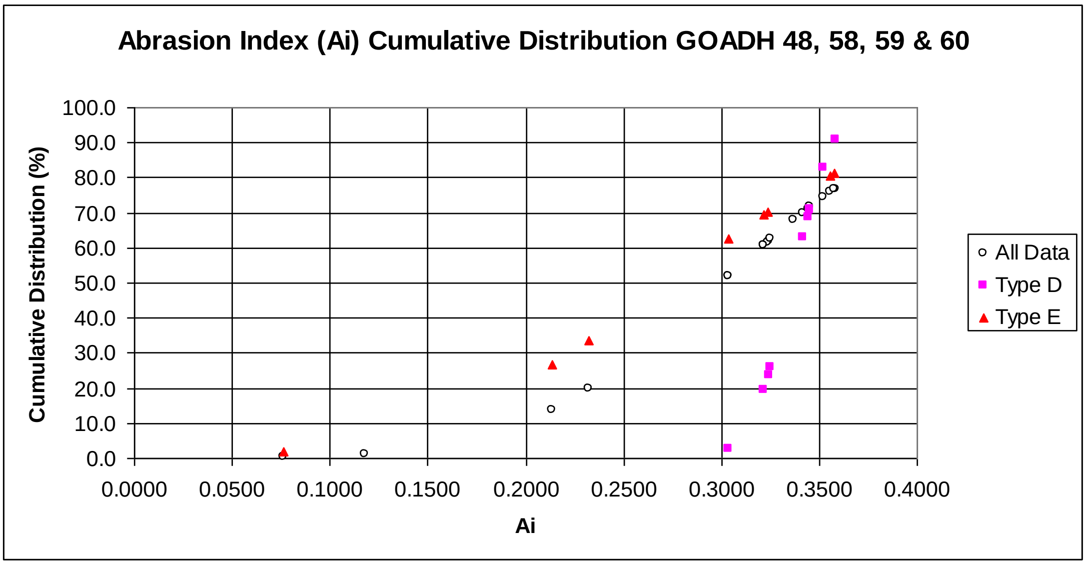


Table 13-15 Comminution Variability Bond Abrasion Index Type D					
Hole ID	**Interval From (m)**	**Interval To (m)**	**Interval (m)**	**U_3O_{8e} (ppm)**	**Ai**
GOADH0048	35.0	100.0	15.0	361	0.341
GOADH0058	9.0	14.6	5.6	464	0.352
GOADH0059	91.0	207.0	15.0	647	0.317
GOADH0060	39.0	95.3	15.6	248	0.345
Average				**422**	**0.336**
Standard Deviation				**191**	**0.017**
Minimum				**194**	**0.304**
Maximum				**840**	**0.358**

Table 13-16 Comminution Variability Bond Abrasion Index Type E					
Hole ID	**Interval From (m)**	**Interval To (m)**	**Interval (m)**	**U_3O_{8e} (ppm)**	**Ai**
GOADH0058	35.4	181.0	25.2	294	0.253
GOADH0059	28.5	174.0	15.0	375	0.309
Average				**324**	**0.274**
Standard Deviation				**127**	**0.093**
Minimum				**167**	**0.076**
Maximum				**467**	**0.358**

Bond Rod Mill Work Index

The Bond Rod Mill Work Index (RWi) variability data is presented below based on a total of 18 tests. The distribution of RWi values appear very consistent with depth and indicate a typically low variability in the global RWi index.

Average values of 12.1Kwh/t for Type D and 12.3Kwh/t in the case of Type E are lower than Ball Mill Work Index (BWi) values presented in Tables 13-19 and 13-20 and as for other SMC testing, indicate no tendency for issues with reduced SAG mill throughput as a result of the build-up of critical size in the mill charge.

Figure 13-11

Comminution Variability – Bond Rod Mill Work Index

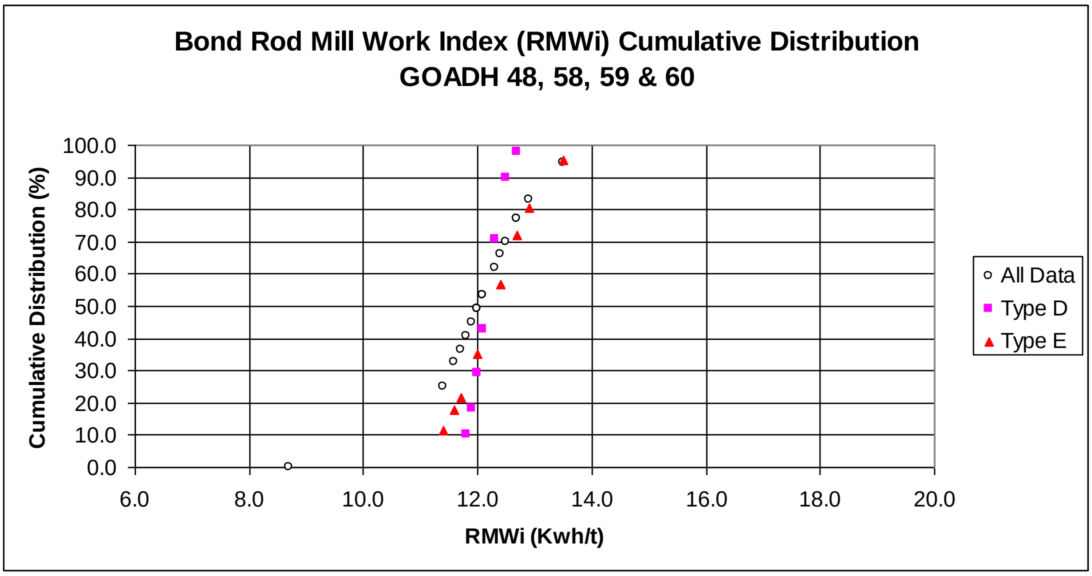


Table 13-17

Comminution Variability Bond Rod Mill Work Index Type D

Hole ID	Interval From (m)	Interval To (m)	Interval (m)	U_3O_{8e} (ppm)	RWi (kWh/t)
GOADH0048	35.0	100.0	15.0	361	12.3
GOADH0058	9.0	14.6	5.6	464	12.3
GOADH0059	91.0	207	15.0	647	12.2
GOADH0060	39.0	95.3	15.6	248	11.9
Average				**422**	**12.1**
Standard Deviation				**191**	**0.3**
Minimum				**194**	**11.8**
Maximum				**840**	**12.7**

Table 13-18

Comminution Variability Bond Rod Mill Work Index Type E

Hole ID	Interval From (m)	Interval To (m)	Interval (m)	U_3O_{8e} (ppm)	RWi (kWh/t)
GOADH0058	35.4	181.0	25.2	294	12.2
GOADH0059	28.5	174.0	15.5	377	12.4
Average				**326**	**12.3**
Standard Deviation				**127**	**0.7**
Minimum				**167**	**11.4**
Maximum				**467**	**13.5**

Bond Ball Mill Work Index

A total of 18 Bond Ball Mill Work Index (BWi) tests were performed. Average and maximum values shown in Table 13-19 and Table 13-20 classify the ore as competent from a fine grinding perspective and the distribution shown in Figure 13-12 indicates that power requirements for milling Type E are slightly higher than in the case of Type D Alaskite. As for other comminution indices, a relatively consistent ball mill comminution response is indicated at depth.

Figure 13-12

Comminution Variability Bond Ball Mill Work Index

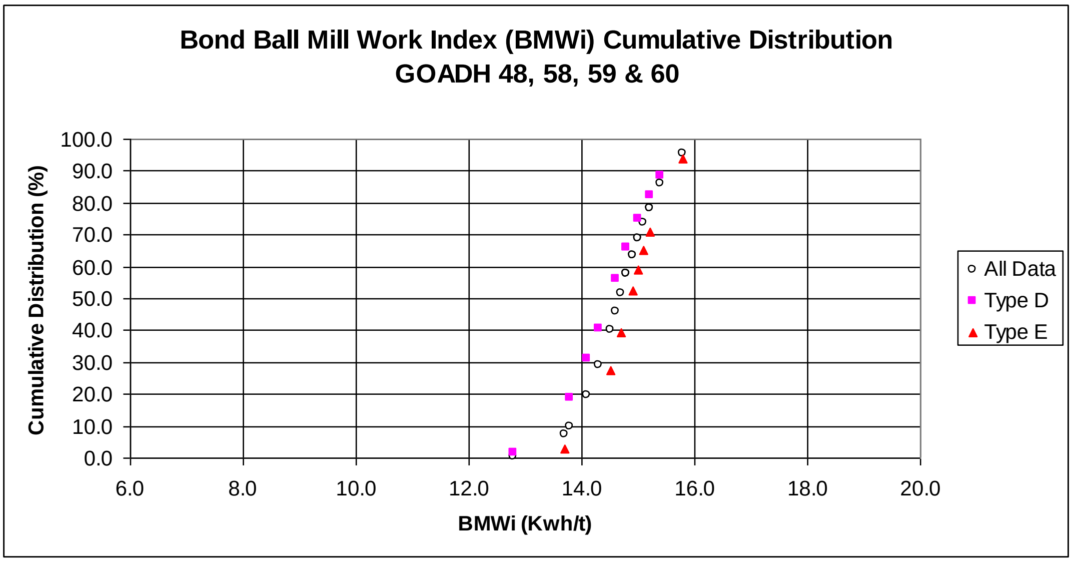


Table 13-19

Comminution Variability Bond Ball Mill Work Index Type D

Hole ID	Interval From (m)	Interval To (m)	Interval (m)	U_3O_{8e} (ppm)	BWi (kWh/t)
GOADH0048	35.0	100.0	15.0	361	14.3
GOADH0058	9.0	14.6	5.6	464	14.8
GOADH0059	91.0	207.0	15.0	647	13.9
GOADH0060	39.0	95.3	15.6	248	15.1
Average				**422**	**14.5**
Standard Deviation				**191**	**0.8**
Minimum				**194**	**12.8**
Maximum				**840**	**15.4**

Table 13-20

Comminution Variability Bond Ball Mill Work Index Type E

Hole ID	Interval From (m)	Interval To (m)	Interval (m)	U_3O_{8e} (ppm)	BWi (kWh/t)
GOADH0058	35.4	181.0	25.2	294	14.9
GOADH0059	28.5	174.0	15.5	377	14.8
Average				**326**	**14.8**
Standard Deviation				**127**	**0.6**
Minimum				**167**	**13.7**
Maximum				**467**	**15.8**

13.4.4 High Pressure Grinding Rolls Pilot Testwork

Parallel testwork programs investigating the mineralogy, agitated leaching and potential to beneficiate Etango ore feed indicated that the ore exhibits a high degree of liberation at coarse size. SEM investigations showed that both of the main material types presented uranium on the natural fracture boundaries within the mineral structure. On this basis a HPGR pilot testwork programme, using the Polysius studded rolls design, was initiated to assess the application of HPGR comminution.

Samples

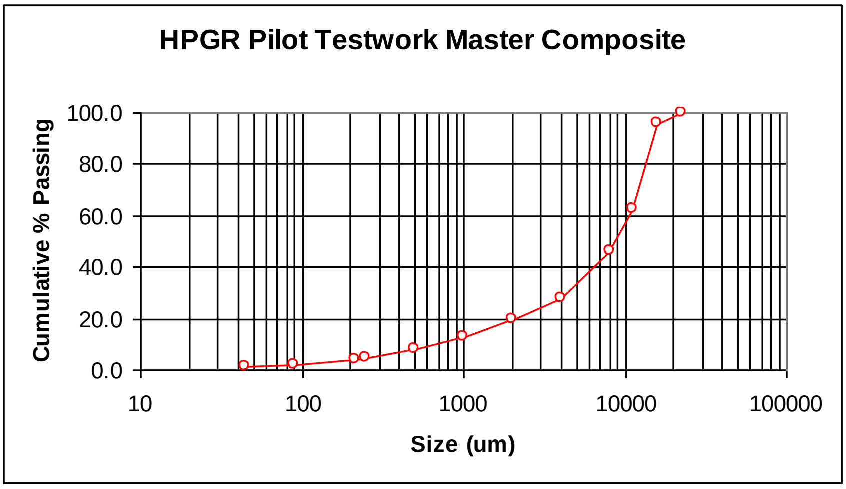
A total of 1,500kg was prepared from whole HQ core specifically selected and drilled for the testwork programme.

Sample Preparation

The master composite was prepared by control crushing to -22.4mm. The prepared composite particle size distribution, with an F_{80} of 13.7mm is shown in Figure 13-13 and this represents the feed to the HPGR.

Figure 13-13

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 13-21.

Table 13-21

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

Open circuit tests were conducted to investigate the effect of two specific pressure settings, roll speeds and moisture levels. Results of the open circuit trials are presented in Table 13-22, and Figure 13-14 to Figure 13-16.

Table 13-22

HPGR Open 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	0.4	0.2
Moisture (%)	6.0	6.0	6.0	3.0
Specific Grinding Force (N/mm^2)	2.90	2.10	2.98	2.99
Operating Gap (excluding zero gap) (mm)	10.0	11.2	10.3	10.5
Operating Gap (including zero gap) (mm)	14.0	15.2	14.3	14.5
Specific Dry Throughput (ts/hm^3)	252.5	253.7	253.2	247.3
Specific Energy (Kwh/t)	1.18	0.88	1.24	1.28
Specific Power (Kws/m^3)	299	224	314	317
Centre Product (% Mass)	59.9	60.8	61.0	59.7
Edge Product (% Mass)	41.1	39.2	39.0	40.3

Figure 13-14

HPGR Open Circuit Trial Reduction Ratio and Pressing Force

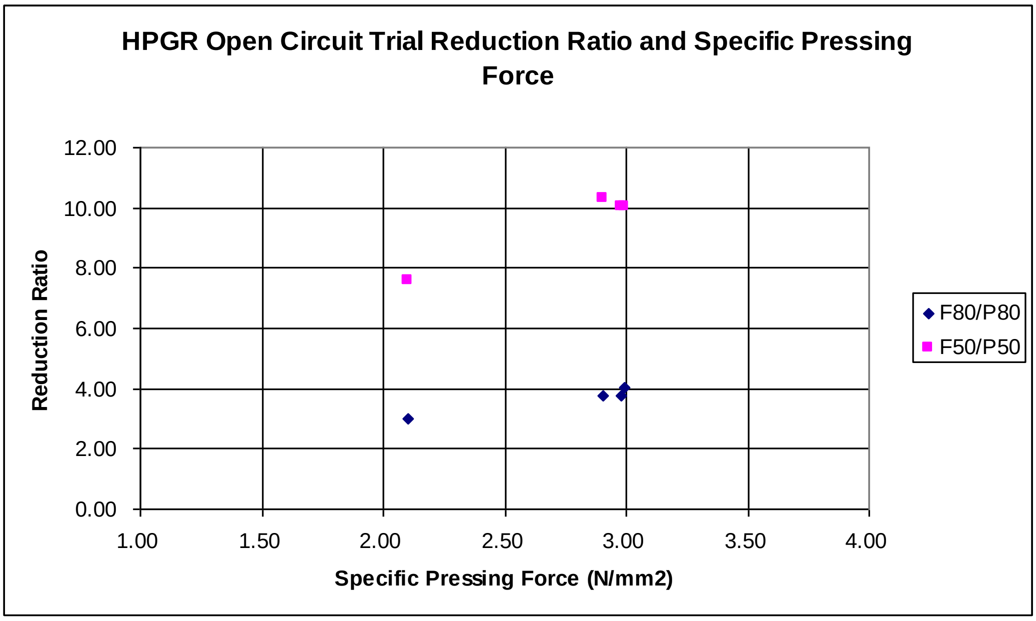


Figure 13-15

HPGR Open Circuit Trial Specific Throughput and Pressing Force

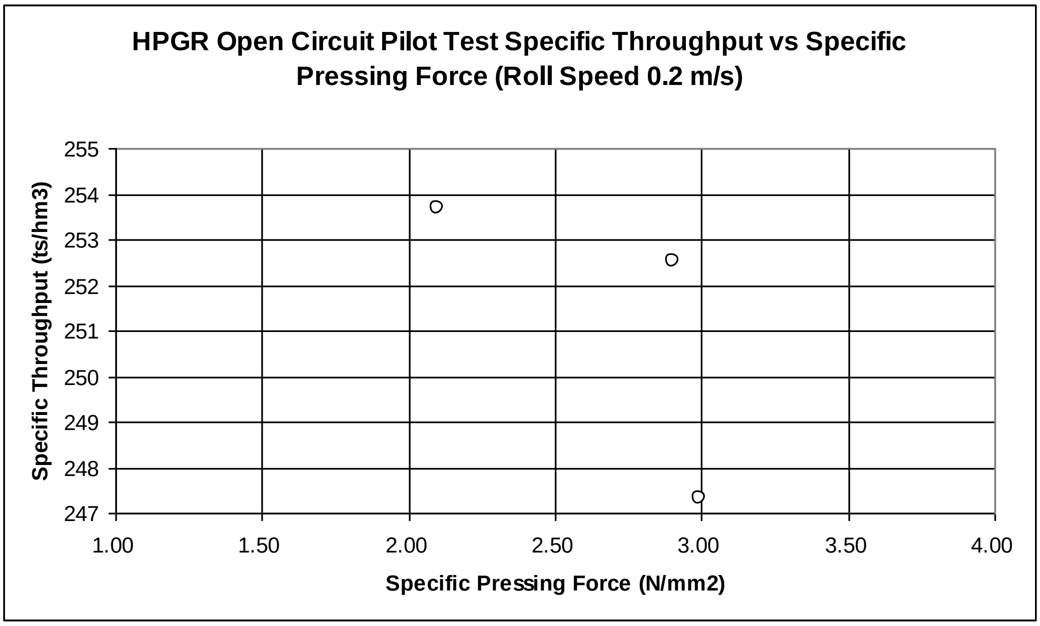
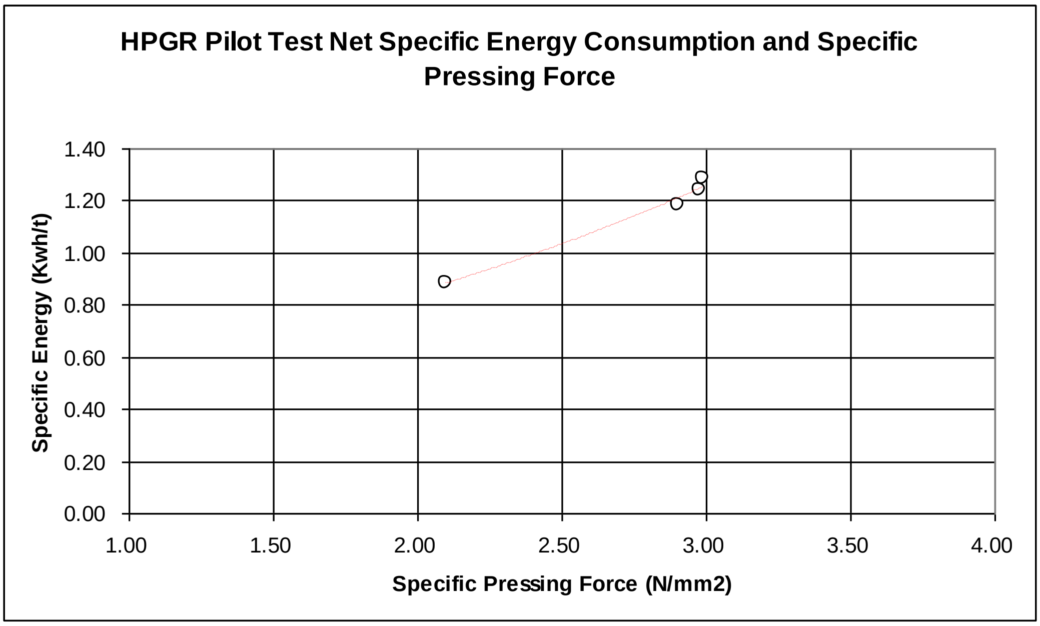


Figure 13-16

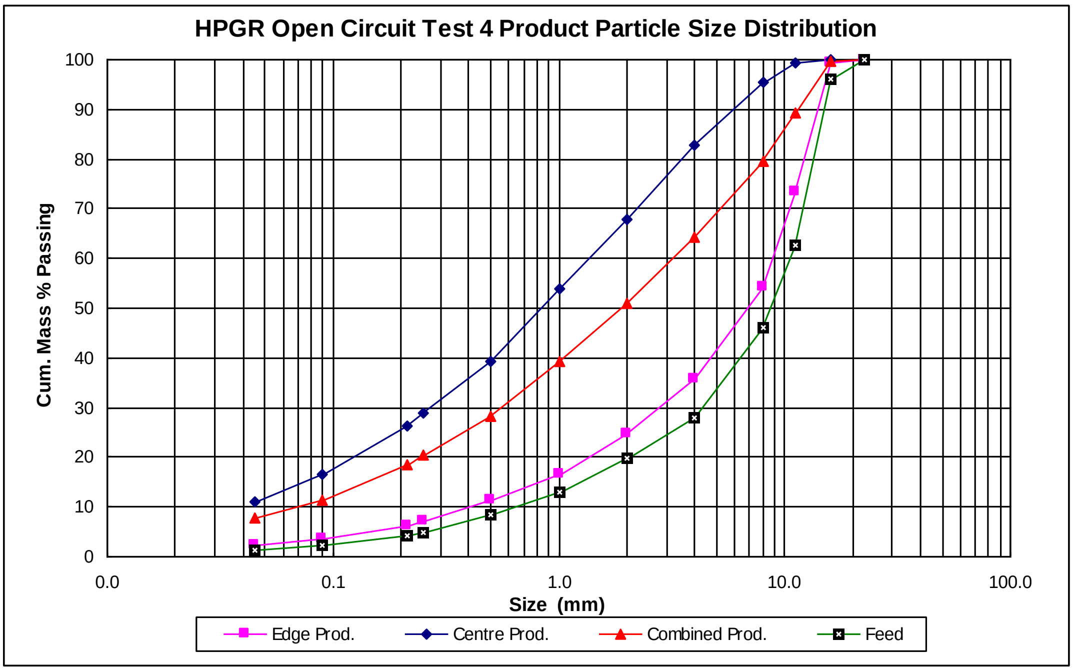
HPGR Open Circuit Trial Specific Energy and Pressing Force



The specific throughput rates and specific energy achieved in open circuit testing indicate 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 0.2 m/s to 0.4 m/s resulted in little change in the specific throughput. Size distributions of the products are shown in Figure 13-17 from Test 4.

Figure 13-17

HPGR Open Circuit Product Particle Size Distribution



HPGR Closed Circuit Trial

Intermittent heap leach bottle roll tests showed improved leaching kinetics and reduced acid consumption rates for HPGR product compared with conventionally crushed feed.

On this basis, initial heap leach investigations were performed on closed circuit HPGR product. The target of the HPGR preparation was a P_{80} size of 4mm, which was chosen as the optimal crush product from the initial heap leach investigations (Section 13.7). The detailed results of the closed circuit HPGR preparation are summarised in Table 13-23.

Table 13-23

HPGR Closed Circuit Pilot Test Data

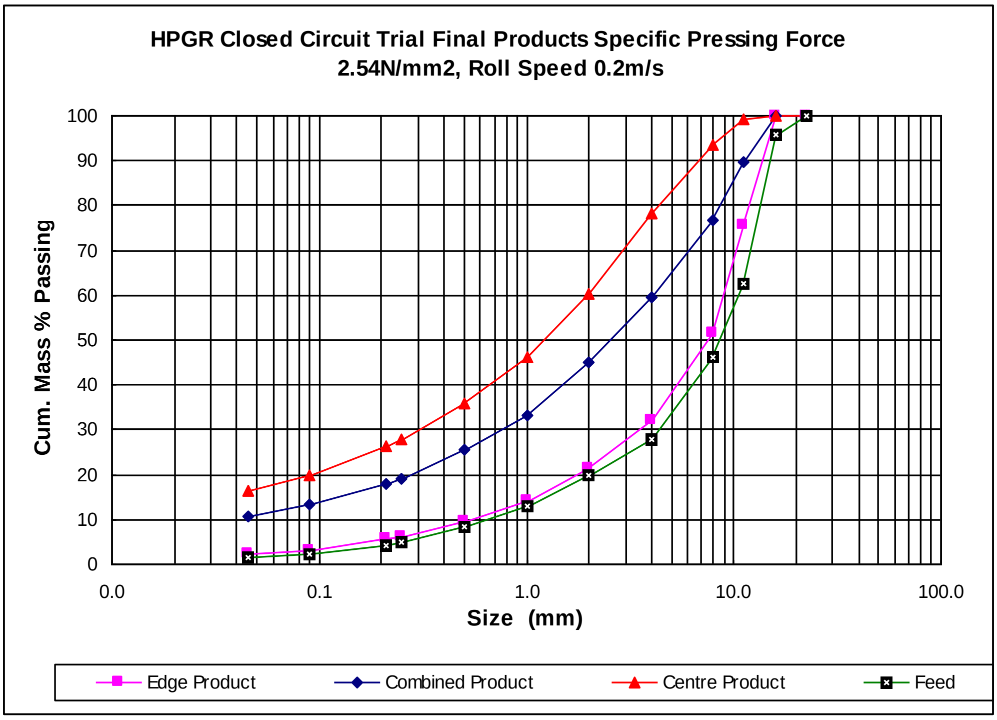
Parameter	Test 1	Test 2	Test 3	Test 4
Roll Diameter (m)	0.5	0.5	0.5	0.5
Roll Length (m)	0.3	0.3	0.3	0.3
Roll Speed (m/s)	0.2	0.2	0.2	0.2
Moisture (%)	3.0	3.0	3.0	3.0
Specific Grinding Force (N/mm^2)	2.55	2.48	2.48	2.54
Operating Gap (including zero gap) (mm)	13.5	13.2	13.2	13.0
Specific Dry Throughput (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
-8 mm in HPGR Discharge (% Mass)	77.1	74.4	74.1	74.6
-8 mm in HPGR Edge Product (% Mass)	53.7	48.8	47.6	50.0
-8 mm in HPGR Centre Product (% Mass)	92.7	91.2	91.9	91.1

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 (4) 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 228.8ts/hm^3 and specific energy of 1.14Kwh/t.

HPGR Product size distributions are shown in Figure 13-18 for final products generated at the fourth test cycle.

Figure 13-18

HPGR Closed Circuit Product Particle Size Distributions



13.4.5 Conclusions

Comminution characterisation demonstrated that Etango ore grade fraction is amenable to conventional crushing and SAG milling and also amenable to HPGR comminution. The ore grade fraction displays a generally low to moderate competency, increasing at finer size. 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

Mineralogical examination via SEM and QEMSCAN reported in Section 13.3 indicated the potential for pre-concentrating the feed. On this basis, the following pre-concentration / beneficiation options were considered and a suite of testwork undertaken.

- Radiometric sorting.
- Scrubbing and screening beneficiation.
- Heavy media separation of coarse (+0.5mm).
- Gravity beneficiation of fines with either a Knelson or Falcon concentrator
- Flotation

A detailed report of this testwork is contained in the December 2010 update to the Pre-Feasibility Study Report

13.5.1 Pre-Concentration Options – Conclusions

The general conclusions drawn from the pre-concentration testwork program are summarised as:

- Traditional scrubbing or screening beneficiation is not likely to add value because the grade of uranium in the coarse fractions of crushed rock is too high to reject outright. Therefore, any mass rejected in this stage will also reject a comparable quantity of uranium to tailings.
- All fundamental testwork focussing on gravity beneficiation concluded that it would only be possible to selectively separate uranium from host gangue if the fresh ore was ground to finer than a 710µm P_{80}. A 710µm P_{80} grinding product appears to be the coarsest product that could provide potentially exploitable uranium liberation from the host gangue.
- Falcon and Knelson concentrators were tested as equipment options that may be able to exploit the gravity concentration potential identified by fundamental laboratory techniques. Whist the upgrading achieved by the Falcon or Knelson concentrators was acceptable, neither apparatus were successful in concentrating enough uranium to the concentrate stream to be viable as a pre-concentration option.
- Both the Falcon and Knelson concentrators required a fine grind (P_{80} = 300µm) to achieve their optimal uranium separation.
- Flotation scavenging of fine gravity tail (Falcon Concentrator) on one sample was successful in achieving a significant mass rejection (96%) at an acceptable uranium loss (6%), however the circuit performance is highly dependent upon a reliable and cost effective flotation stage.
- Flotation requires a fine grind product (sub 400µm) to enable the potential for an acceptable uranium recovery as flotation efficiency decreases at coarser particle sizes.
- On one sample tested, flotation alone generated a comparable result to the flotation scavenging of fine gravity tail with 97% mass rejection and 7% uranium loss.
- Re-testing of the flotation option on a second sample (Pilot Composite) identified significant variability in performance. This variability is deemed a significant risk as it has the potential to lose up to 14% of leachable uranium to the process tail.
- Despite the potential to significantly decrease the mass processed by a downstream leach, flotation will incur a significant operating cost (collector and acid cost of US$2.36/tonne of ore), water burden and grinding requirement (P_{80} of 300µm or less) that would need to be justified against the potential uranium loss introduced by a pre-concentration stage.
- None of the beneficiation options are suitable if heap leaching is the preferred method of downstream uranium leaching.
- Flotation is the only pre-concentration option tested that may suit agitated leaching as the preferred leaching option. However, the primary grind requirement of flotation is

not completely compatible with the agitated leach grind requirements. The latest results suggest that agitated leaching requires a grind product P_{80} of 700-800µm, and flotation requires a grind product P_{80} of 300µm.

13.6 Agitated Leach Testing

Three agitated leaching programs have been initiated at different stages of the project development.

Stage 1:

The first phase of agitated leach testing focused on investigating the effect of temperature and grind size. This phase was completed on a Type D Alaskite composite sample only.

Stage 2:

The second phase of agitated leach testing re-evaluated the effect of grind size on two samples: Type D Alaskite composite and Type E Alaskite composite. For this phase, all leaches were conducted at ambient temperature.

Stage 3:

The third phase of testwork evaluated the variability of leach performance across the resource, and a comprehensive investigation on the effect of grind size. The additional work on grind size was expanded on the indicators from the Stage 1 & 2 programs. Testing on a single composite with grind size as the only variable detailed the size – recovery – time relationship and identified the critical size for optimum recovery.

13.6.1 Samples

The composition of the Type D Alaskite sample obtained from selected intervals of drillhole GOADH0048 is shown in Table 13-24.

Table 13-24

Interval Selection Leach Optimisation Type D

Hole ID	Interval From (m)	Interval To (m)	Metre	U_3O_8 (ppm)
GOADH0048	29	34	5	240
	48	52	4	236
	54	58	4	267
	74	77	3	182
	90	94	4	267
Total			**20**	**241**

The composition of the Type E Alaskite sample is shown in Table 13-25.

Table 13-25

Interval Selection Leach Optimisation Type E

Hole ID	Interval From (m)	Interval To (m)	Metre	U_3O_8 (ppm)
GOADH0029	437.0	452.0	15.0	328
GOADH0030	39.51	55.0	15.5	260
Total			**30.5**	**294**

The composition of the pilot composite sample referred to in Phase 3 is shown in Table 13-26.

Table 13-26

Interval Selection Pilot Composite

Hole ID	**Interval From (m)**	**Interval To (m)**	**Metre**	**kg**	**U_3O_8 (ppm)**
GOADH0076	14	138	124	690	168
GOADH0077	9	168	159	638	147
GOADH0078	20	209	189	1,405	229
GOADH0079	30	193	163	1,258	232
GOADH0080	22	139	117	829	161
GOADH0081	48	203	155	965	289
GOADH0082	46	118	72	589	155
GOADH0083	39	222	183	1,396	242
GOADH0084	51	218	167	1,285	151
GOADH0085	116	188	72	587	89
GOADH0086	73	156	83	678	193
GOADH0087	81	165	84	279	169
GOADH0088	18	207	189	1,228	235
GOADH0089	38	129	91	607	212
GOADH0090	21	148	127	819	292
GOADH0091	41	201	160	1,221	264
GSHDD0005	21	187	166	917	198
Total				**15,392**	**211**

The variability samples referred to in Phase 3 are shown in Table 13-27.

Table 13-27

Variability Samples

Sample ID	**Hole ID**	**Interval From (m)**	**Interval To (m)**	**Metre**	**kg**	**U_3O_8 (ppm)**
V78-1	GOADH0078	20	50	30.0	248.3	349
V78-2	GOADH0078	50	80	30.0	247.7	177
V78-3	GOADH0078	80	114	30.0	250.0	198
V78-4	GOADH0078	114	144	30.0	250.9	240
V78-5	GOADH0078	164	194	30.0	250.2	286
V79-1	GOADH0079	30	60	30.0	247.7	389
V79-2	GOADH0079	60	90	30.0	250.0	335
V79-3	GOADH0079	90	122	30.0	249.2	195
V79-4	GOADH0079	122 164	144 172	30.0	245.3	197
V80-1	GOADH0080	24	56	30.0	246.9	179
V80-2	GOADH0080	108	138	30.0	247.7	184
V81-1	GOADH0081	50 93	68 105	30.0	250.0	313
V81-2	GOADH0081	120	150	30.0	245.3	239
V81-3	GOADH0081	150	180	30.0	245.1	404

V83-1	GOADH0083	41	71	30.0	247.2	375
V83-2	GOADH0083	103	133	30.0	253.2	285
V83-3	GOADH0083	151	181	30.0	253.4	173
V83-4	GOADH0083	190	220	30.0	246.2	158
V84-1	GOADH0084	52	82	30.0	245.4	190
V84-2	GOADH0084	82	112	30.0	247.3	177
V84-3	GOADH0084	133	163	30.0	245.5	139
V84-4	GOADH0084	163	193	30.0	247.4	137
V86-1	GOADH0086	82	112	30.0	245.6	234
V87-1	GOADH0087	83 99 143	89 103 163	30.0	245.9	185
V5-1	GSHDD0005	91	121	30.0	244.7	202

13.6.2 Stage 1 – Initial Tests on Type D Alaskite – July 2008

Phase 1 included a number of kinetic leach tests conducted over a range of grind sizes and temperature, test conditions are described below.

- Testing was conducted on GOADH0048 Composite assaying 151 ± 6 ppm U.
- Primary Grind P80: 1300µm, 1,000µm, 710µm and 425µm
- Temperature: Maintained at: 30ºC, 50ºC, 60ºC and 70ºC
- Water: distilled water
- Solids density – 50% (w/w).
- pH controlled to 1.5.
- Oxidant addition as milled Pyrolusite maintaining +500mV (std calomel).
- Ferrous sulphate addition maintaining a minimum 500ppm Ferric.

The kinetic leach tests were performed for 24 hours with solution samples at: 1, 2, 3, 4, 6, 8, 12 and 24 hours. The following conditions were monitored:

- Leach conditions: Eh, pH and temperature.
- Free acid concentration (g/l).
- Acid consumption rate (kg/t).
- Oxidant consumption rate (kg/t).
- Ferrous Sulphate consumption rate (kg/t).

The results of the leach tests are summarised in Table 13-28. Uranium extraction estimates (based on mass balancing outputs) were calculated from measured solid and liquor assays after 24 hours of leaching.

Table 13-28

Stage 1 Agitated Leach Test Matrix Type D

Grind Size P80 (µm)	Temperature (ºC)	Avg pH	End pt pH	Avg Eh (mV)	End pt mV	H2SO4 consumed (kg/t)	Uranium Extraction (%)
1300	30	1.52	1.7	800	990	0.48	91.7
1300	50	1.54	1.75	769	886	4.45	87.7
1300	60	1.48	1.69	753	910	2.32	89.6
1300	70	1.49	1.85	744	863	3.12	88.8
1000	30	1.58	1.93	688	689	5.00	88.7
1000	50	1.41	1.89	692	662	5.95	87.8
1000	60	1.59	2.26	621	594	3.70	82.4
1000	70	1.62	2.06	613	597	3.12	86.9
710	30	1.57	1.94	666	655	7.89	81.6
710	50	1.55	2.05	662	597	5.52	84.3
710	60	1.54	2.04	617	584	4.90	82.8
710	70	1.48	1.86	572	563	4.19	83.6
425	30	1.06	1.23	663	659	8.72	89.5
425	50	1.3	1.25	647	667	5.36	88.1
425	60	1.45	1.71	579	609	4.31	81.6
425	70	1.5	1.71	547	582	3.96	85
Average		**1.48**	**1.81**	**665**	**694**	**4.56**	**86.2**

Leach extraction profiles for each test are presented in Figures 13-19 to 13-22, grouped by grind size.

Figure 13-19

Agitated Leach Kinetics Type D P80 1300 µm

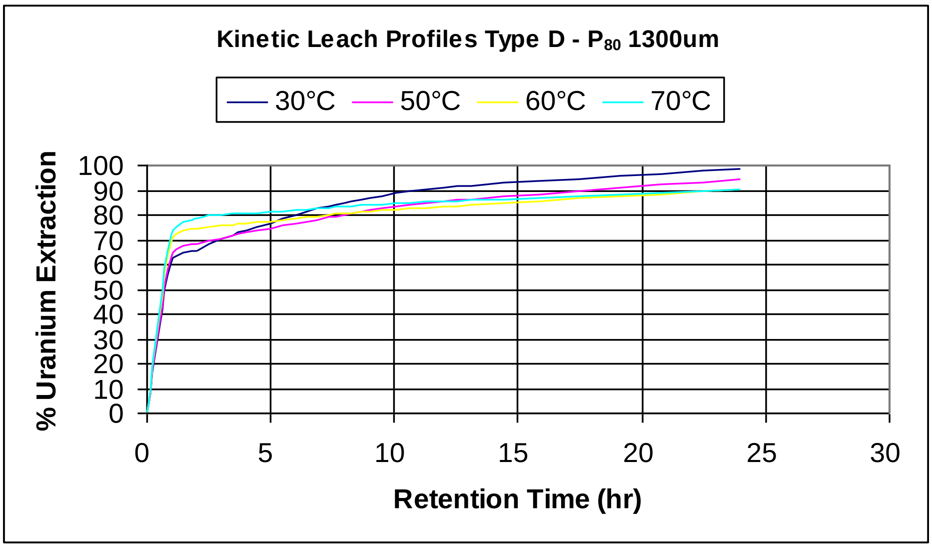


Figure 13-20: Agitated Leach Kinetics Type D P80 1,000 µm

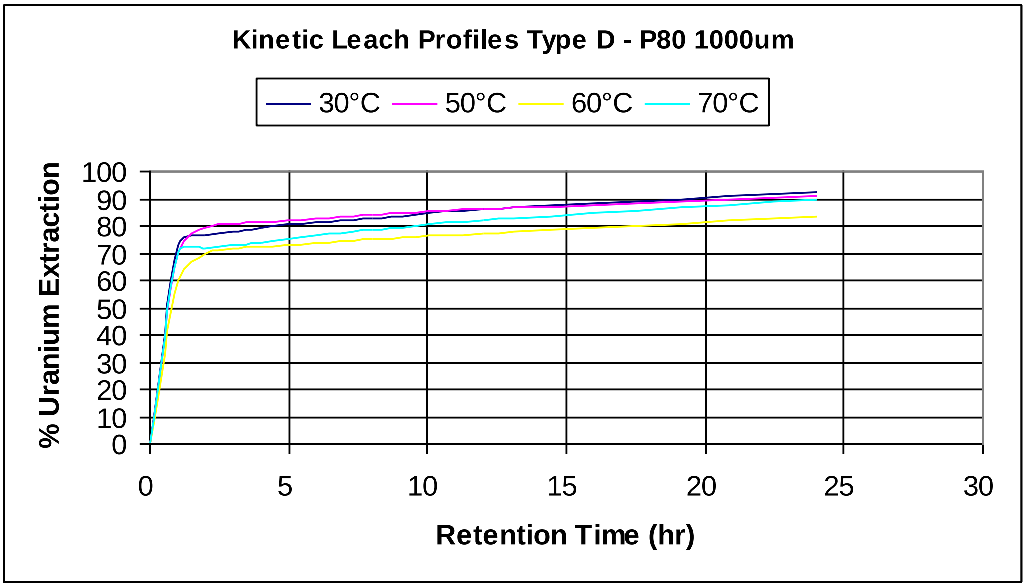


Figure 13-21: Agitated Leach Kinetics Type D P80 710 µm

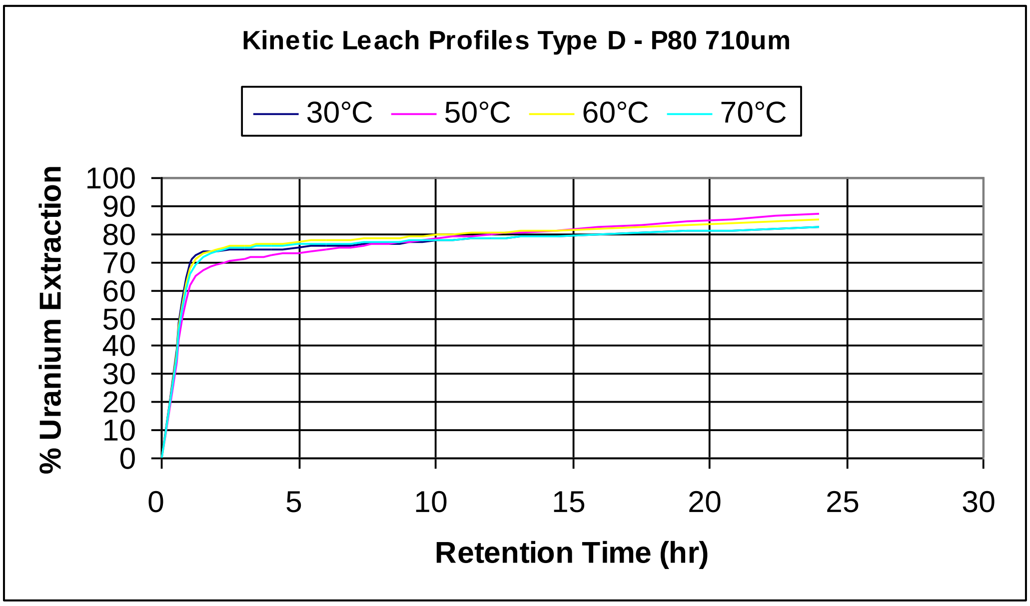
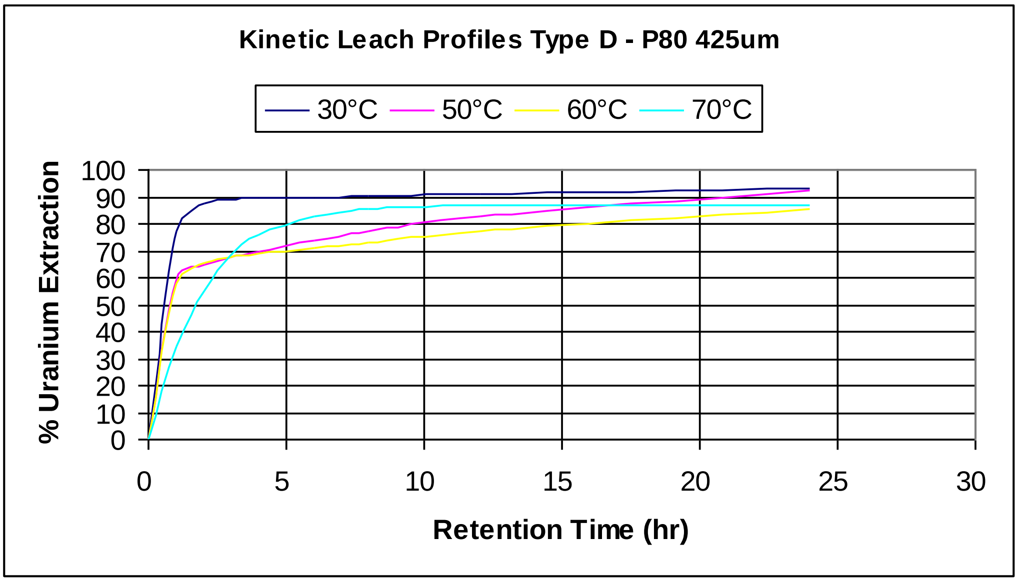


Figure 13-22: Agitated Leach Kinetics Type D P80 425 µm



The results presented above could not identify a significant consistent relationship between uranium extractions and grind size or leach temperature. While natural variability in experimental results and/ or other process variables may mask the subtle relationships it can be concluded that the tested variables have lower relative significance for recovery within the range tested.

13.6.3 Stage 2 – Further Optimisation of Type D and Type E Alaskite – Sep 2008

A second stage of optimisation testing was performed over a range of grind sizes and at ambient temperature. Three (3) tests were performed at a P_{80} size of 1,000µm for both Type D and Type E as a control to compare consistency in final results. Results are presented in the Table 13-29, Figure 13-23, Table 13-30 and Figure 13-24 for each lithology type.

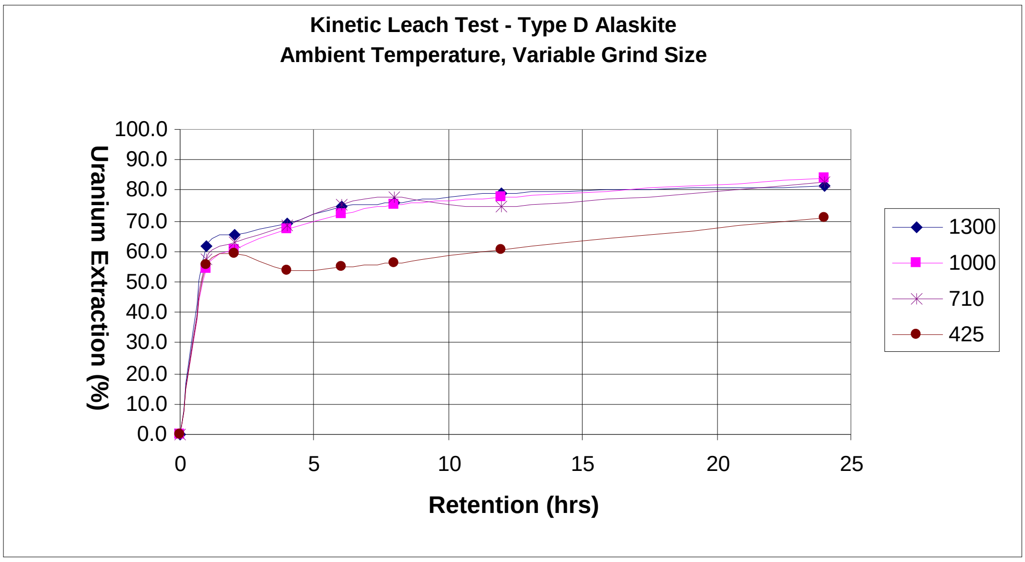
Table 13-29

Stage 2 Leach Optimisation Type D

Grind Size P_{80} (µm)	pH	Eh (mV)	H_2SO_4 (kg/t)	Uranium Extraction (%)
1,300	1.57	487	7.2	81.5
1,000	1.55	509	7.5	85.6
1,000	1.57	506	9.0	81.4
1,000	1.45	476	6.9	85.0
710	1.44	498	8.7	82.9
425	1.52	473	4.3	71.0
Average	**1.52**	**492**	**7.3**	**81.2**

Figure 13-23

Agitated Leach Kinetics Type D Variable Grind Size



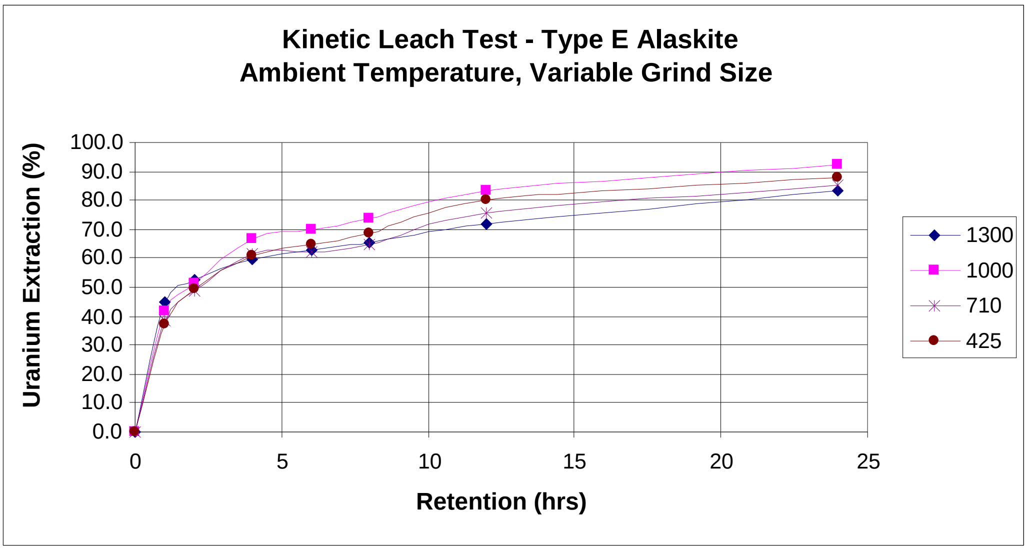
The results for the 1,000µm P_{80} tests show that experimental variability does exist with ultimate calculated extractions varying from 81.4 to 85.6%. Given the observed variability, it is difficult to conclude that grind size exhibits a measureable effect on uranium extraction from single tests conducted at 1,300, 710 and 425µm grind products.

It is noted that the test from a 425µm grind generated an unusual extraction response that is not consistent with the other results.

Table 13-30 Stage 2 Leach Optimisation Type E				
Grind Size P_{80} (µm)	pH	Eh (mV)	H_2SO_4 (kg/t)	Uranium Extraction (%)
1,300	1.48	474	11.6	83.1
1,000	1.53	496	7.7	91.8
1,000	1.59	496	6.9	92.5
1,000	1.60	474	7.0	92.4
710	1.66	513	5.7	85.0
425	1.66	466	6.6	87.9
Average	**1.59**	**487**	**7.6**	**88.8**

Figure 13-24

Agitated Leach Kinetics Type E Variable Grind Size



The variability of the 1,000µm results is much less than seen in the preceding tests with Type D Alaskite composite, however the order of the results is not consistent with results from the other grind sizes. This may be attributable to a masking effect from another variable.

Within the accuracy of the tests, a grind size effect within the range tested could not be concluded from the Stage 1 or Stage 2 testwork programs.

13.6.4 Stage 3 – Leach Response Variability and Effect of Grind Size

The Stage 3 agitated leach program investigated the variability of the response on a range of samples, and defined the effect of grind size via a series of duplicate tests under standard conditions and size by size analysis of results.

Standard conditions for this stage of the testwork program are:

- Primary Grind P_{80}: 300µm for variability program; variable for Effect of Grind Size program
- 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 target free acid concentration of 5g/l was chosen to ensure the uranium extraction vs. grind size relationship was not masked by acidity effects.

Leach Response Variability

21 different samples were subjected to the standard leach conditions following a 300µm P_{80} primary grind. The results have been rigorously analysed and reported separately and the unfiltered data-sets have been presented in Table 13-31 to summarise the final analysis. The data-set presents the results of all tests conducted in the variability program.

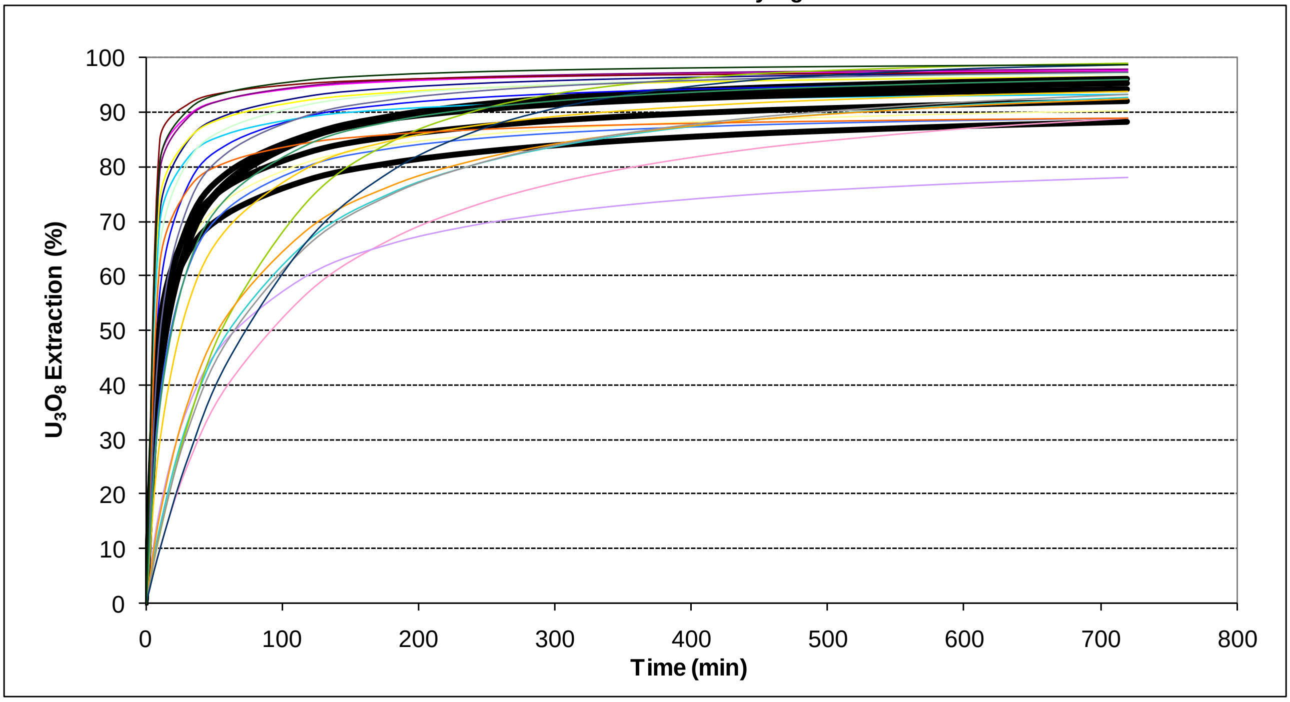
Table 13-31
Unfiltered Agitated Leach Results from Variability Program

Test	**Sample**	**U3O8 Head Grade**		**Reagents**			**Extraction after:**		
		Measured	**Calculated**	**Pyrolusite**	**Acid kg/t ore**		**480**	**600**	**720**
		ppm U3O8		kg/t ore	Added	Cons	min	min	min
JA 1324	V78-2	185	190	0.2	15.0	8.7	96.7	97.0	97.3
JA 1325	V78-3	215	294	0.1	14.7	6.8	97.3	97.5	97.7
JA 1326	V78-4	218	271	0.2	14.9	7.7	95.9	96.3	96.5
JA 1327	PCDrum 1	222	229	0.7	19.9	19.3	86.4	87.4	88.2
JA 1328	V78-5	298	314	0.1	15.1	14.6	97.4	97.6	97.8
JA 1329	V79-1	364	346	0.4	18.3	17.7	97.2	97.4	97.5
JA 1330	PCDrum 8	157	172	0.6	17.5	17.2	94.7	95.5	96.0
JA 1331	V79-2	377	410	1.13	15.1	14.6	94.8	95.2	95.6
JA 1332	V79-3	196	195	0.7	19.3	19.0	92.8	93.1	93.4
JA 1333	PC Drum 25	172	170	0.4	18.1	17.6	90.5	91.3	91.9
JA 1334	V79-4	192	206	0.4	16.1	15.6	96.4	96.8	97.2
JA 1335	V80-1	118	153	0.3	14.8	14.2	89.0	89.8	90.3
JA 1336	PC Drum 32	206	201	0.6	20.4	19.8	93.2	93.9	94.3
JA 1337	V80-2	134	171	0.5	20.6	20.0	84.4	87.1	89.1
JA 1338	V81-2	212	252	0.5	16.9	16.3	75.5	77.0	78.1
JA 1339	PC Drum 47	212	216	0.7	22.5	21.8	94.0	94.8	95.3
JA 1340	V81-3	419	442	0.6	21.7	20.9	87.9	88.5	88.9
JA 1341	V83-1	307	351	0.5	22.0	20.3	89.3	91.3	92.7
JA 1342	V83-3	169	137	2.0	18.6	18.2	97.6	98.6	99.1
JA 1343	V83-2	294	309	1.1	21.2	20.5	92.0	93.0	93.7
JA 1344	V83-4	152	132	0.6	19.1	18.3	89.2	91.1	92.4
JA 1345	V84-1	281	317	0.6	25.1	24.3	88.4	88.7	89.0
JA 1346	V84-2	287	320	1.2	18.6	17.7	96.4	97.0	97.4
JA 1347	V84-3	186	208	1.3	20.1	20.1	89.7	91.8	93.2
JA 1348	V84-4	205	199	1.3	18.3	17.3	96.6	98.1	98.8
JA 1349	V87-1	197	251	1.8	18.4	17.8	94.5	95.4	96.0
JA 1352	V79-1	364	364	3.4	19.2	18.6	98.4	98.6	98.7
Average		235	253	0.8	18.6	17.2	92.4	93.3	93.94
StDev		79	85	0.7	2.7	4.1	5.2	4.8	4.6
Min		118	132	0.1	14.7	6.8	75.5	77.0	78.1
Max		419	442	3.4	25.1	24.3	98.4	98.6	99.1

The uranium extraction curves for all of the tests summarised in Table 13-31 are presented in Figure 13-25.

Figure 13-25

Uranium Extraction Curves for Unfiltered Variability Agitated Leach Tests



The thick black curves in Figure 13-25 represent the extractions of the five pilot composite samples tested.

The relatively consistent performance of all samples and convergence with time gives confidence that, in general, the test process is robust.

The full data set indicates the ore is broadly amenable to the oxidative acid leach system with 26 of 27 results > 85%, 22 of 27 results > 90% and over half > 95% leach extraction after 720 minutes with an overall average of 93.9%.

<u>Effect of Grind Size on Agitated Leach Performance</u>

A 40kg sub-sample was taken from drum number 25 of the Pilot Composite sample, and this sample was used to re-evaluate the effect of primary grind size on agitated leach performance.

The earlier leach tests (Section 13.6.2) identified the difficulty of quantifying the effect of primary grind from a single leach test. As a result each primary grind was tested in quadruplicate to enable statistical analysis. The program has been analysed and reported in detail and the summarised results are presented in Figure 13-26 and Figure 13-27.

The results presented are for all tests and were derived from curve fitting of the general rate equation to the full set of progress sample solids assays.

Figure 13-26

Uranium Extraction as a Function of Grind Product Size P80

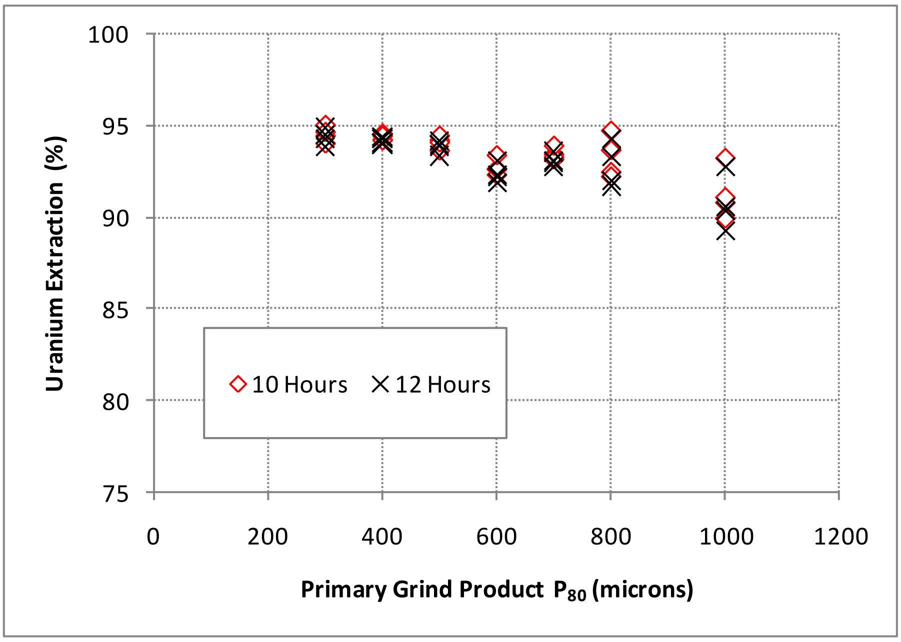
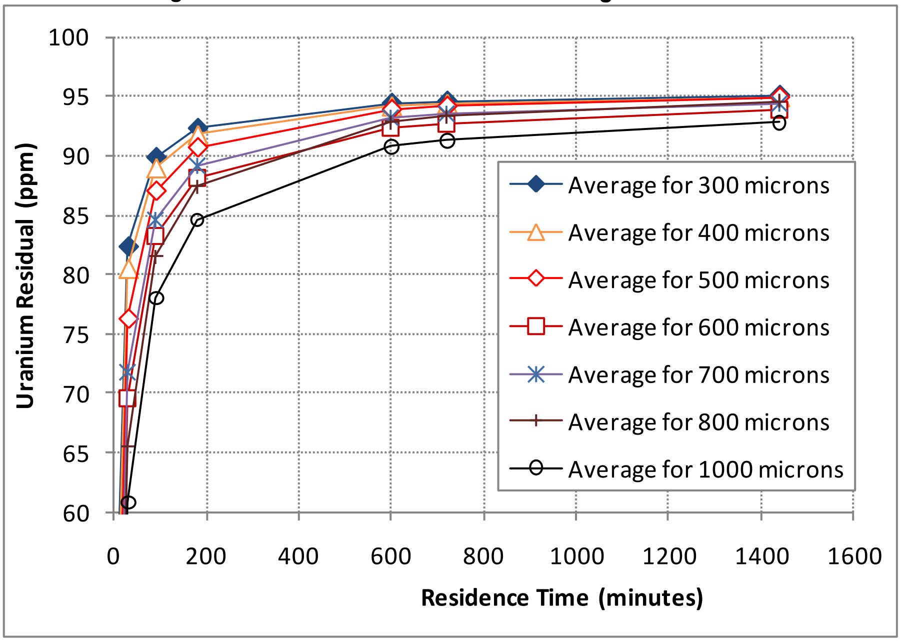


Figure 13-27

Average Uranium Extraction Curves for a Range of Grind Product P80's



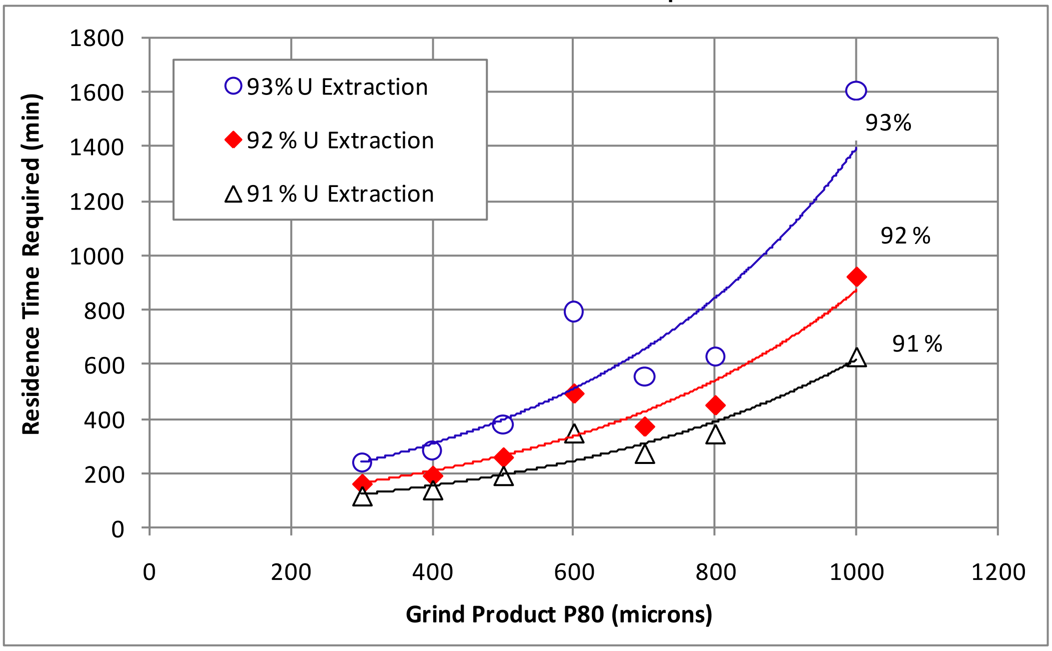
Some data from Figure 13-27 is also presented in Table 13-32.

Table 13-32
Average Uranium Extraction for a Range of Grind Products

Grind Product P_{80} (µm)	**Average Uranium Extraction at:** 10 hours	12 hours	24 hours
300	94.4	94.6	95.1
400	94.2	94.4	95.0
500	93.9	94.1	94.9
600	92.4	92.8	93.8
700	93.1	93.5	94.4
800	92.8	93.3	94.6
1,000	90.7	91.3	92.8

The above data and charts show that whilst leach kinetics consistently slow down as the grind size increases, the overall extraction after 12 hours is only marginally affected (~1%) as the primary grind increases from a 300µm P_{80} to 800µm. Figure 13-28 presents the laboratory leach times required to achieve 91%, 92% and 93% uranium extractions from the various grind products tested.

Figure 13-28
Effect of Grind Size on Residence Time Requirements



Size by size uranium extraction analysis of a 700µm P_{80} and 1,000µm P_{80} test confirmed that uranium extraction starts to decrease for particle sizes greater than 500µm (Figure 13-29 and Figure 13-30).

Figure 13-29

Extraction by Size for 700µm P80 Test

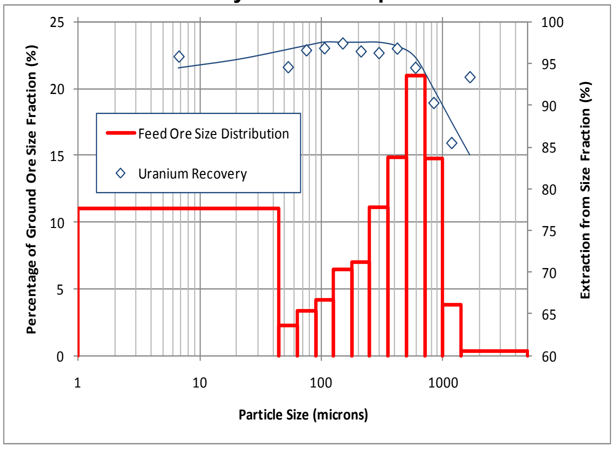


Figure 13-30

Extraction by Size for 1,000µm P80 Test

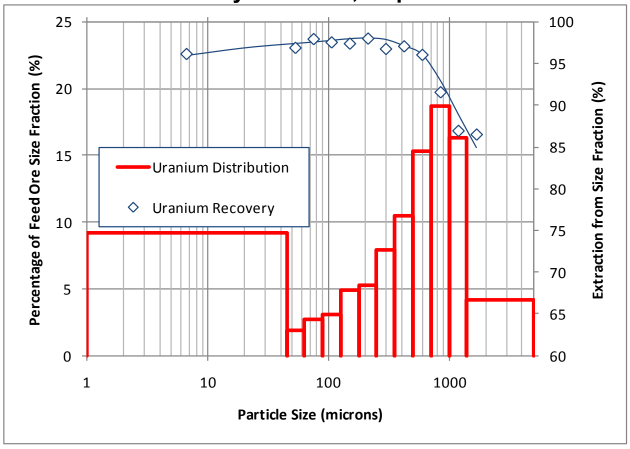


Table 13-33

Summarised Size by Size Uranium Extraction

Size Fraction	**Mass Distribution (%)**		**Uranium Distribution (%)**		**Uranium Extraction (%)**	
	700µm	1,000µm	700µm	1,000µm	700µm	1,000µm
+500µm	39.8	54.6	19.5	30.1	92.5	91.8
+45µm-500µm	49.2	36.2	46.6	40.6	96.6	97.5
-45µm	11.0	9.2	33.9	29.3	95.9	96.2
Overall Recovery					95.6	95.4

The size by size uranium extraction curves for the 700µm and 1,000µm P_{80} grind products (Figure 13-29 and Figure 13-30) are essentially the same, and therefore the variation in overall recovery is due to the quantity of uranium presenting in each size fraction following the primary grind.

13.6.5 Acid Consumption

The agitated leach variability program and effect of grind size on leach performance program provides the most relevant data for estimating the acid consumption of an agitated leach system.

<u>Acid Consumption Variability from Agitated Leach Tests</u>

From the grind versus leach performance program, a chart of grind size versus acid addition/consumption was constructed (Figure 13-31).

Figure 13-31

Acid Addition and Acid Consumption as a Function of Grind Size

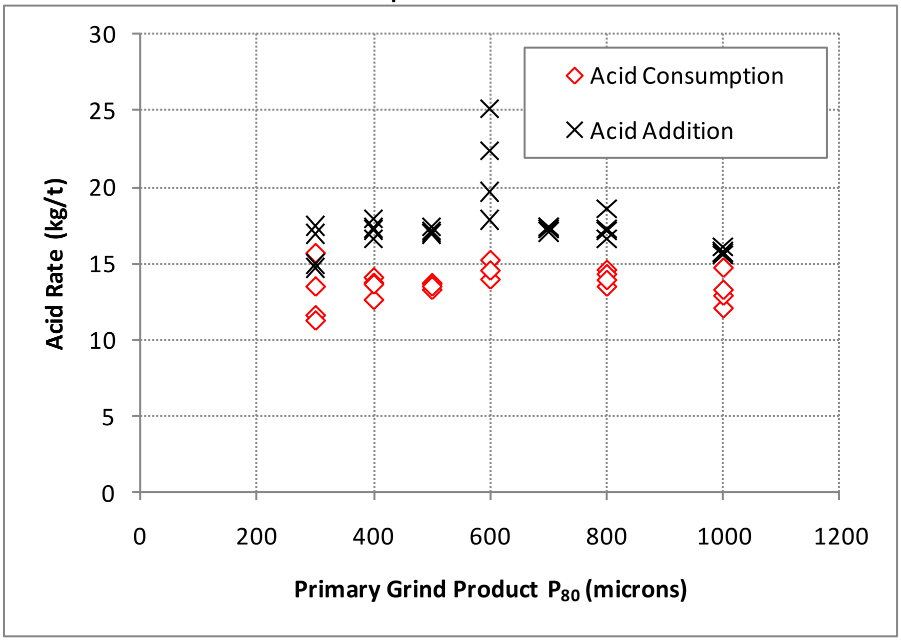


Figure 13-31 suggests that acid consumption is not significantly affected by the primary grind product P_{80}. This means that any acid consumption trends observed from the variability program (conducted using grind product P_{80} = 300µm) will be relevant to an agitated leach system where the grind product P_{80} is within the range of 300 to 1,000µm.

The cumulative acid consumption was calculated for each sampling point of each test in the agitated leach variability program and standardised (consumption at 24 hours equals 100%). The resultant chart is presented in Figure 13-32.

Figure 13-32

Standardised Acid Consumption for all Variability Tests

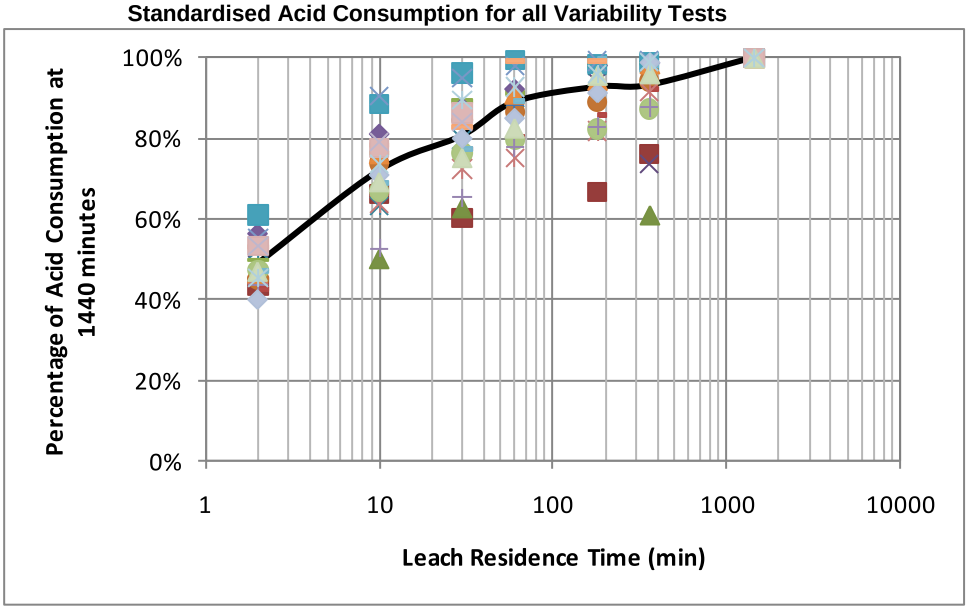


Figure 13-32 suggests that greater than 90% of total acid consumption has been completed within 3 hours of leaching. With probable leach residence times of between 8 and 12 hours, the acid consumption discount from 16.1 kg/t (100% of acid consumption after 24 hours) is not likely to be significant.

13.6.6 Conclusions

Based on the laboratory scale testwork conducted the following conclusions about the agitated leaching system may be drawn:

- On average, >93% uranium extraction can be expected from the agitated Etango ore under mild conditions. The conditions that can deliver this average extraction are:
    - Primary grind finer than a P_{80} of 800μm
    - Mildly acidic conditions: 5 g/L H_2SO_4 in solution which equates to ~16kg/t of acid
    - Pulp Eh of approximately 500 mV (standard calomel), maintaining approximately 500ppm ferric in solution
    - Ambient temperature (~45^{o}C)
- Leach extraction has relatively minor variability across the resource, The best estimate of uranium extraction is the average of the variability program which produced an overall average of 93.9%, with 26 of 27 results > 85%, 22 of 27 results > 90% and over half > 95% leach extraction after 720 minutes.
- Acid consumption is variable across the resource though the average and range of consumption is low. The range of independent measured variables did not support a regression analysis and derivation of a statistical model. The best estimate of acid consumption is the average of the variability program.
- Grind size has a measurable effect on uranium extraction however this is not significant for ore samples where the P_{80} of the grind product is less than 800μm.
- For the chosen grind size, the kinetics of the leach must be considered when engineering a residence time as the grind size does affect the rate of leaching.
- There is no measurable extraction benefit to be gained by leaching at increased temperatures.

13.7 Heap Leach Testing

Based on the findings from previous mineralogy and agitated leach investigations, an evaluation of heap leaching was commissioned with the aim of establishing design criteria for further economic modelling and cost engineering.

The following packages of work were designed to evaluate the potential to heap leach Etango ore feed:

- Sighter testing – comprising intermittent bottle roll tests at -12.5mm and -6.3mm crush sizes.
- Chemical assay and SEM analysis of sighter stage residue size fractions.
- Intermittent bottle roll leach tests and agglomeration testing on Type D and Type E lithologies over a range of crush sizes and free acid concentrations.
- Open circuit column tests – utilising conventional and HPGR generated particle size distributions.
- Agglomeration and percolation testing utilising binder and acid pre-treatment.

- Closed circuit column testing – involving 4 x 4m column tests on conventional and HPGR feed at modelled full scale particle size distribution.
- Batch solvent extraction testwork – performed on closed circuit 4m column PLS.

13.7.1 Samples

Bulk samples were obtained from selected intervals of metallurgical HQ core drilled specifically for the testwork. A master composite was developed based on Holes: GOADH0062, 63, 64, 65 and 66 targeting a 2:1 blend of Type D and Type E Alaskite. Details of the master composite are provided in Table 13-34.

Table 13-34
Interval Selection Heap Leach Master Composite

Lithology	Hole ID	Meters	U_3O_{8e} (ppm)
Type D	GOADH0062	21	367
	GOADH0063	10	434
	GOADH0065	43	391
	GOADH0066	56	289
Total		**130**	**346**
Type E	GOADH0062	4	400
	GOADH0063	27	234
	GOADH0064	6	379
	GOADH0065	29	296
Total		**66**	**284**
Total Type D and Type E		**196**	**305**

13.7.2 Preliminary Intermittent Bottle Roll Tests

A preliminary series of four (4) intermittent bottle roll tests (**IBR**) conducted on sub-samples control crushed to P_{100} sizes at -12.5 mm and -6.3 mm were conducted. For each crush size, a free acid concentration of 5g/L and 15g/L was maintained throughout the leaching period of 30 days to achieve the metal extractions presented below.

No ferrous sulphate or additional oxidant was added in these tests.

<u>Preliminary IBR Leach Results</u>

The uranium extraction and acid consumption curves from the preliminary IBR test are presented in Figure 13-33 and 13-34 respectively.

Figure 13-33

Uranium Extraction Curves for Preliminary IBR Tests

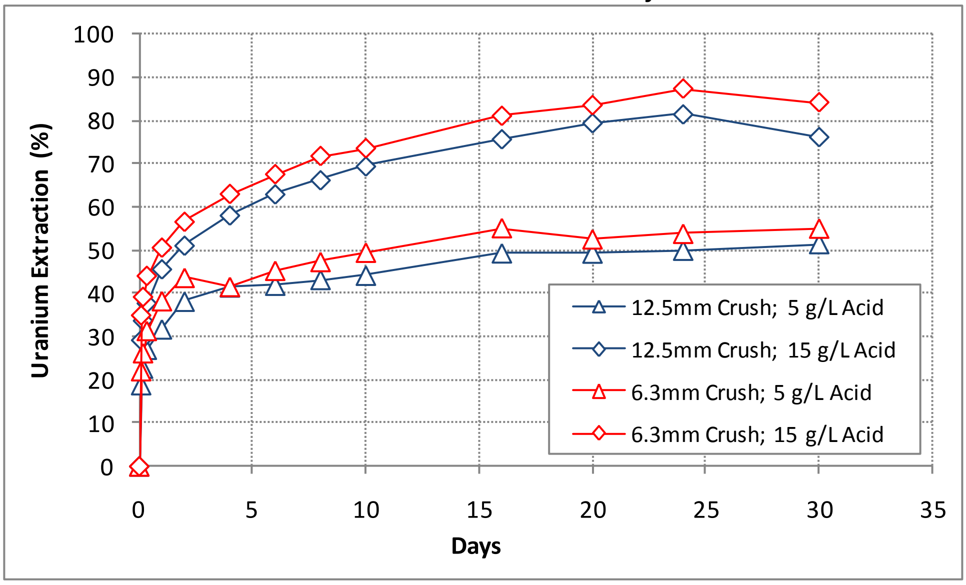
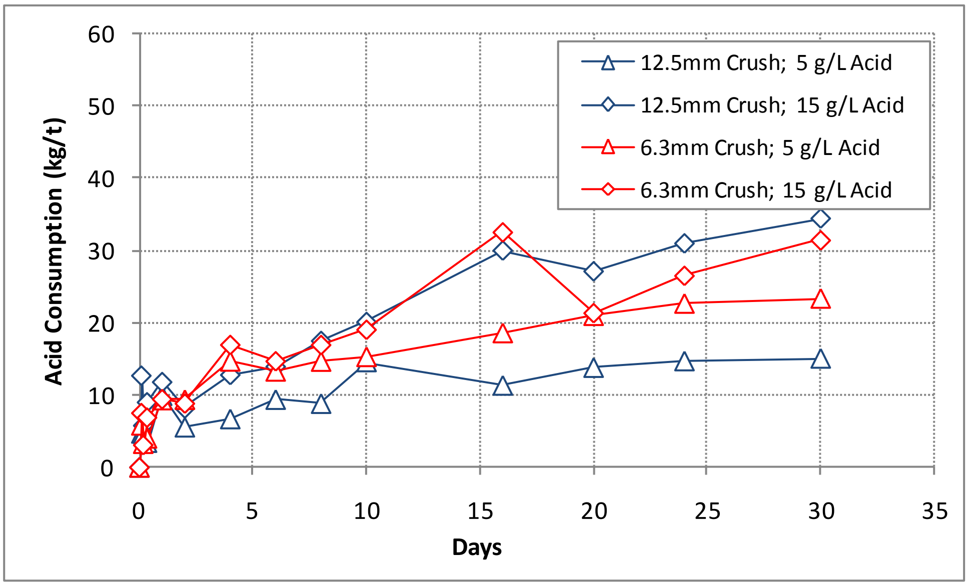


Figure 13-34

Acid Consumption Curves for Preliminary IBR Tests



The key observations drawn from the above analyses are:

1. To achieve maximum uranium extraction, 15 g/L of acid is clearly superior to 5 g/L of acid.
2. The finer crush size appears to increase extraction. This can be concluded from the consistent (yet small) increase in extraction for the 6.3mm crush under each free acid condition.
3. Increased free acid concentration has resulted in an increase in acid consumption
4. Decreasing the crush size does not increase the acid consumption.

Analysis of Preliminary IBR Leach Residues

Head and residue samples from each test were submitted for sizing, and chemical assay of each size fraction. Size by size extraction estimates for the -12.5 mm and -6.3 mm crush sizes are presented in Figure 13-35 and

Figure 13-36 respectively.

Figure 13-35

-12.5 mm Crush Size Residue Size Fraction Extraction

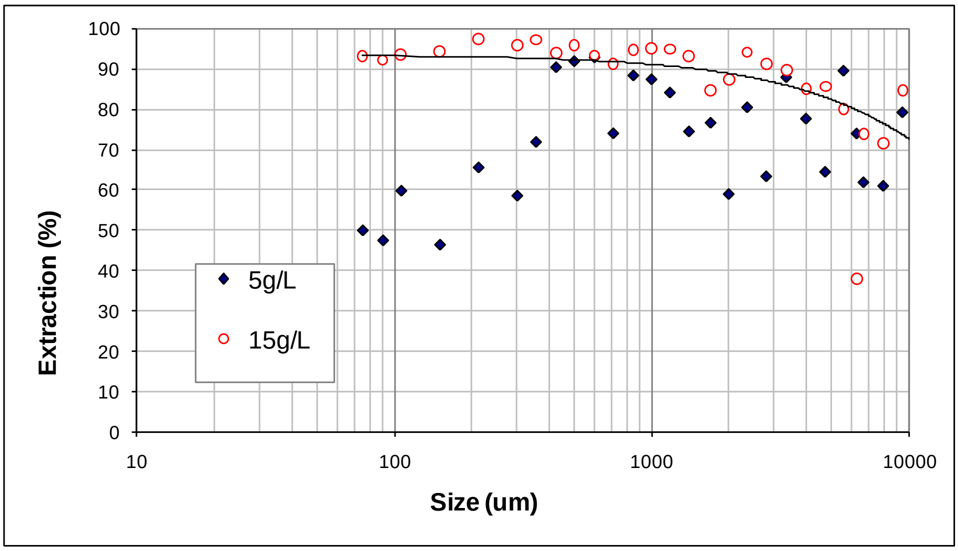
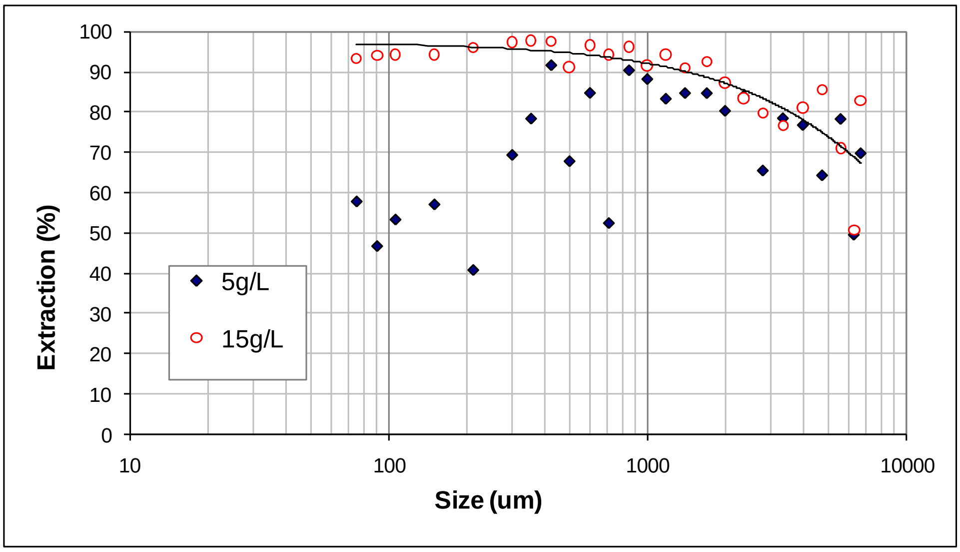


Figure 13-36

-6.3 mm Crush Size Residue Size Fraction Extraction



Figures 13-35 and 13-36 show that the uranium extraction by size trends are consistent for both the -12.5 mm crushed sample and the -6.3mm crushed sample.

13.7.3 Secondary Program of Intermittent Bottle Roll Tests

<u>Test Scope</u>

Following the preliminary IBR program (Section 13.7.2) a secondary program was initiated that tested a range of crush sizes and free acid concentrations. The crushing preparation was also investigated by testing the leach performance of conventional crushed and HPGR prepared samples. Common test conditions maintained through each test were as follows:

- Tests performed in deionised water, ambient temperature.
- Ferrous sulphate added to maintain iron in solution above 800ppm.
- Oxidant as milled Pyrolusite to maintain an Eh above +500mV (versus standard Calomel) and convert the ferrous ions to ferric.

A total of sixteen (16) intermittent bottle roll tests were conducted in this phase of the testwork program. Table 13-35 summarises all conditions tested and the final extractions and acid consumptions from test.

Table 13-35
Intermittent Bottle Roll Test Matrix

Test No.	Ammtec Test No.	Preparation	Crush Size (mm)	P_{80} (mm)	Free Acid (g/L)	Acid Consumption (kg/t)	U Extraction (%)
1	MH8048	Conventional	9.5	8.1	10	21.8	89.3
2	MH8049	Conventional	9.5	8.1	20	27.6	88.4
3	MH8050	Conventional	12.5	10.7	10	16.5	78.5
4	MH8051	Conventional	12.5	10.7	20	26.1	81.3
5	MH8052	Conventional	12.5	10.7	30	29.7	81.9
6	MH8053	Conventional	16.0	14.2	10	18.1	75.5
7	MH8054	Conventional	16.0	14.2	20	25.6	79.5
8	MH8055	Conventional	16.0	14.2	30	20.2	79.7
9	MH8056	Conventional	25.0	22.3	20	26.0	73.1
10	MH8057	Conventional	25.0	22.3	30	28.1	71.5
11	MH8086	HPGR C + E#	16.0	8.5	10	21.5	88.7
12	MH8087	HPGR C + E#	16.0	8.5	20	24.3	82.1
13	MH8088	HPGR C + E#	16.0	8.5	30	28.4	85.0
14	MH8089	HPGR Centre*	16.0	4.0	10	25.9	91.5
15	MH8090	HPGR Centre*	16.0	4.0	20	25.1	90.9
16	MH8091	HPGR Centre*	16.0	4.0	30	23.6	93.9

Notes:

#The P_{80} 8.5mm HPGR ore is the combined centre and edge product from a -16mm open circuit HPGR run

*The P_{80} 4.0mm HPGR ore is the centre product only from a -16mm open circuit HPGR run

The particle size distributions for each of the crushed products are presented in Figure 13-37.

Figure 13-37

Particle Size Distributions for Crushed Products of Secondary IBR Test Program

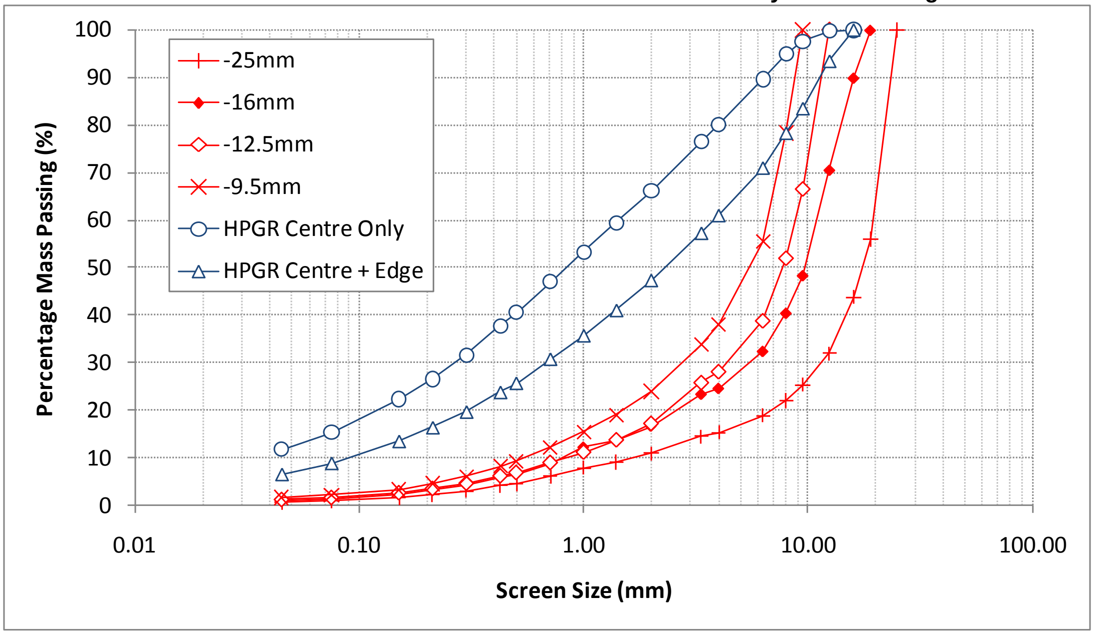


Figure 13-37 shows that the HPGR products contain significantly more fine material than the conventionally crushed material, even for a comparable P_{80} (HPGR Centre + Edge and Conventional -9.5mm). As a result of the variation in particle size distributions, it will be difficult to directly compare the leaching performance of a HPGR product with a comparable conventionally ground product.

All of the IBR uranium leach curves for the conventionally crushed and HPGR crushed ore samples from the Secondary Program are presented in Figure 13-38.

Figure 13-38

IBR Uranium Leach Curves for Conventionally Crushed Ore – Secondary Program

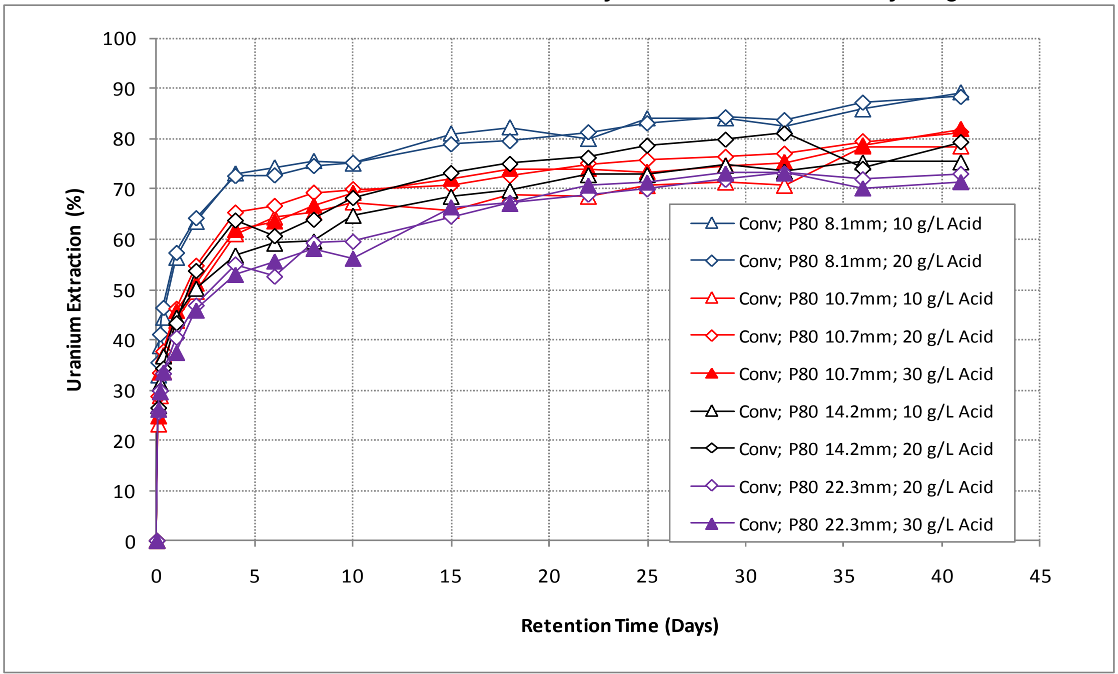


Figure 13-39

IBR Uranium Leach Curves for HPGR Crushed Ore – Secondary Program

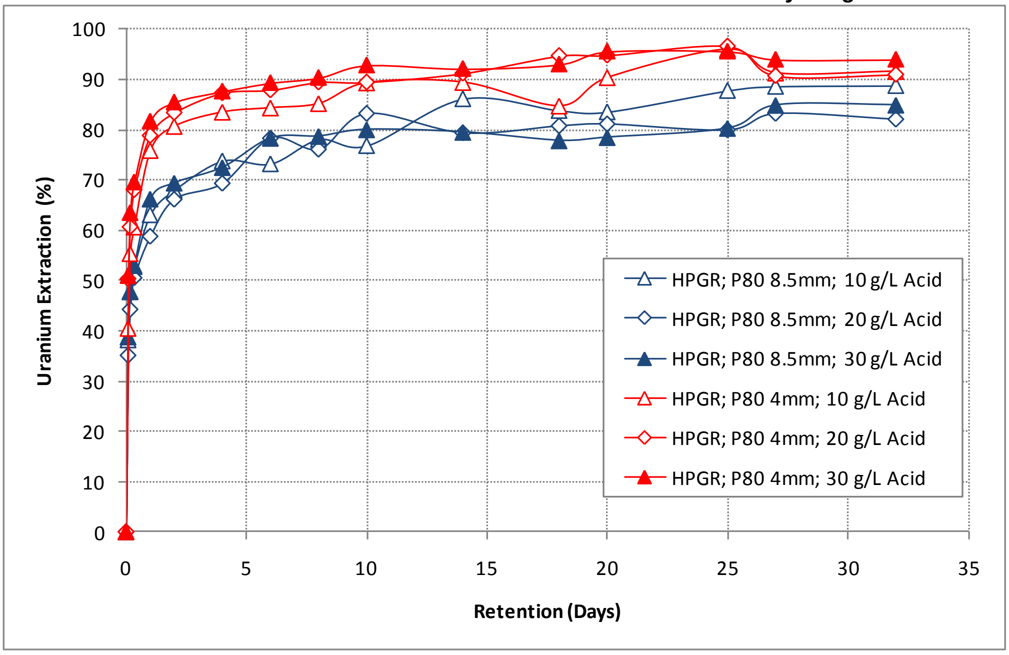


Figure 13-38 and Figure 13-39 demonstrate two significant trends. Specifically:

1. Varying the acid concentration from 10 g/L to 30 g/L does not result in a significant increase in uranium extraction
2. Uranium extraction consistently increases as the crush product becomes finer. Maximum uranium extractions were achieved from the finest crush product (HPGR Centre only, P_{80} = 4mm)

In the previous section (Section 13.7.2), it was concluded that acid consumption increased as free acidity increased. A similar trend was observed in the Secondary program however it is not a strong trend. In a leaching scenario like this one, the best measure of acid efficiency is the relationship between acid consumption and uranium extraction. Figure 13-40 presents acid consumption against uranium extraction.

Figure 13-40

IBR Acid Consumption Data for Conventionally Crushed Ore – Secondary Program

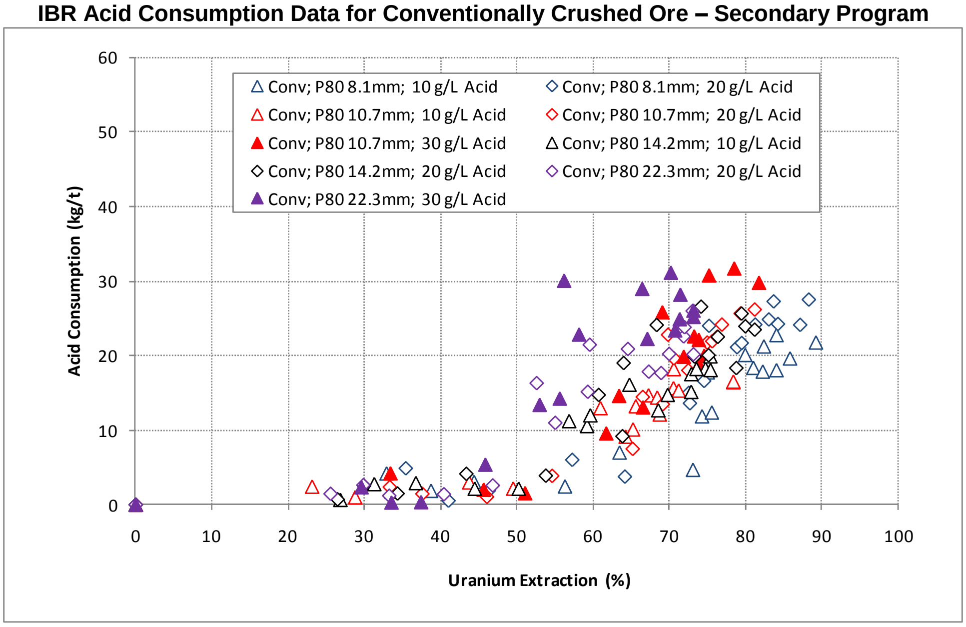


Figure 13-40 shows that the efficiency of acid consumption increases as the particle size distribution becomes finer and the uranium extraction increases.

Generally, the HPGR products are finer than any of the conventionally crushed products so an unbiased comparison of the subsequent leachability of the relative crushing techniques is not possible because a particle size effect has been previously demonstrated (Section 13.6.4 and Section 13.7.2). The two most similar particle size distributions from different crushing techniques are:

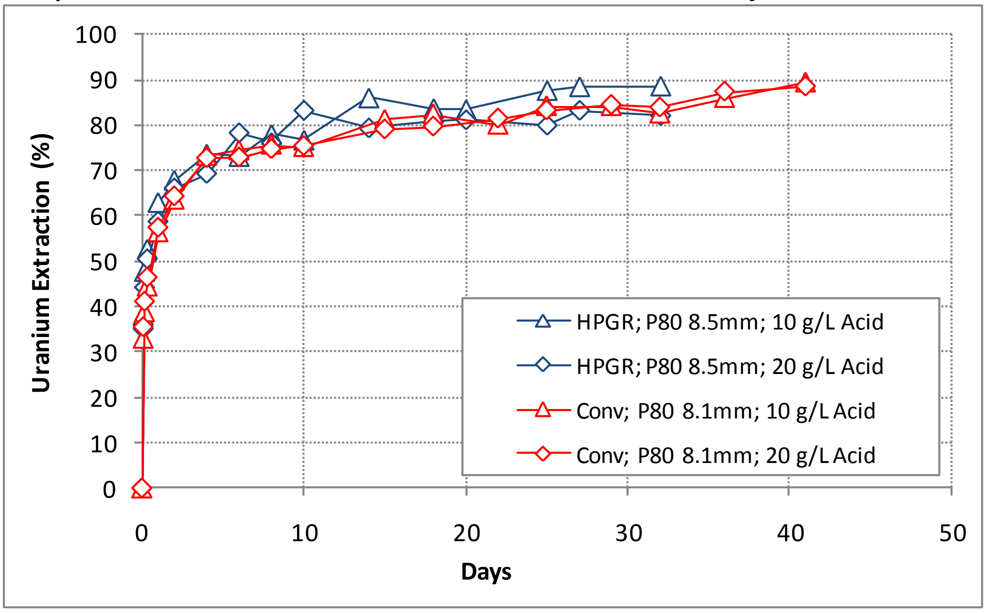
a) The combined centre and edge product from the HPGR crush – P80 = 8.5mm, and

b) The -9.5mm conventionally crushed ore sample - P_{80} = 8.1mm

Figure 13-41 compares the IBR uranium extraction curves for these two samples under similar chemical conditions.

Figure 13-41

Comparison of IBR Uranium Extractions for HPGR and Conventionally Crushed Ore



Once again it is noted that the sample prepared by HPGR is Type E Alaskite only and the conventionally crushed ore is a 2:1 blend of Type D and Type E Alaskite ore. Despite this and the differences in particle size distribution, Figure 13-41 shows that there is very little difference between the HPGR and conventionally crushed ore samples in the IBR test under comparable conditions. Therefore, the selection of crushing circuit design should be based on the relative cost and operability of the crushing options designed to deliver the required particle size distribution, and should not be based on the potential for downstream extraction benefits.

13.7.4 Preliminary Open Circuit Column Tests

Parallel to the secondary program of intermittent bottle roll tests, a series of three (3) open circuit column tests (103mm diameter x 1m) were performed as follows:

- Conventional crushing -12.5mm crush size (P_{80} = 10.7mm) 15g/L free acid
- Conventional crushing -12.5mm crush size (P_{80} = 10.7mm) 25g/L free acid

- HPGR preparation closed circuit centre only product (P_{80} = 4.0mm) 15g/L free acid

Common conditions for the three column tests were:

- No binder added to the agglomeration stage
- 12.5 L/m^2/hour lixiviant irrigation rate (2,500 mL/day)
- Irrigation duration – 48 days
- Open circuit – fresh lixiviant added for the duration of the test
- 26.4 g of ferrous sulphate (heptahydrate) added each day to ensure that iron in solution is greater than 500g/L
- 1.5g of pyrolusite to oxidise ferrous in solution to ferric

Because of the significant variation in particle size and its effect on uranium extraction, the leach performances of the HPGR product and conventionally crushed product cannot be directly compared, although the results are presented on the same chart for convenience.

The uranium extraction curves are presented in Figure 13-42 and acid consumption versus uranium extraction curves are presented in Figure 13-43

Figure 13-42

Open Circuit Column Uranium Extraction

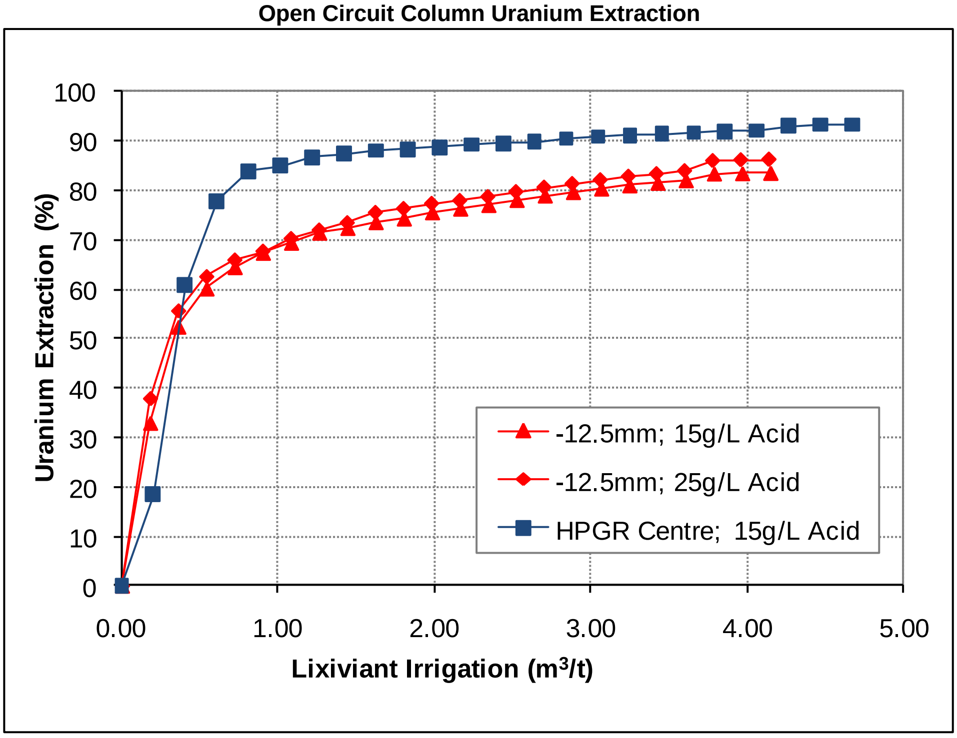
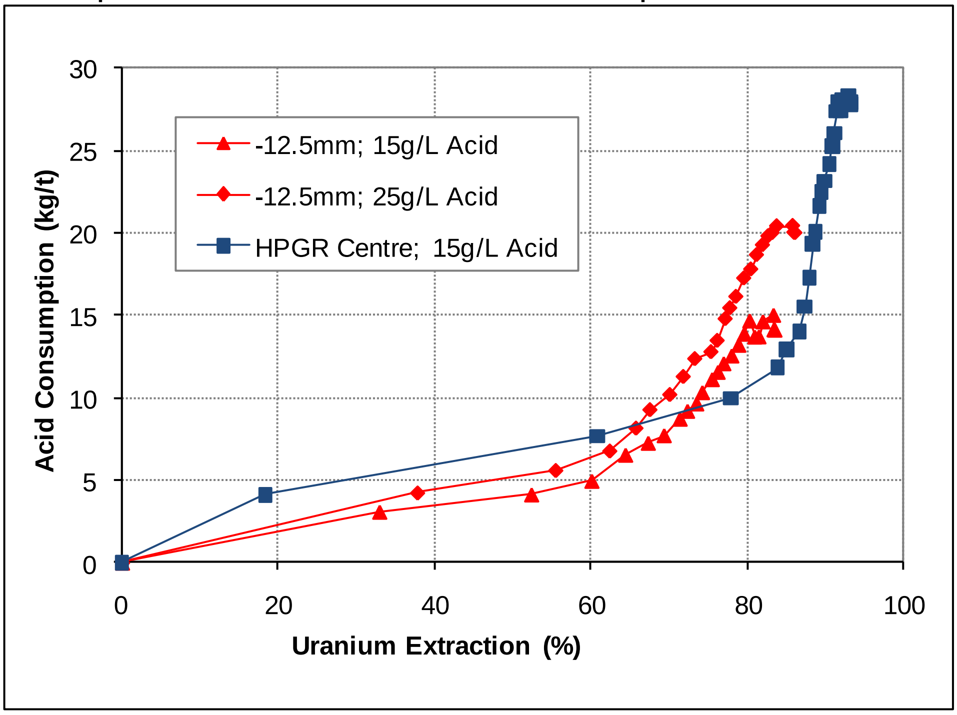


Figure 13-43
Open Circuit Column Test Cumulative Acid Consumption and Extraction



The results in Figure 13-42 and Figure 13-43 are consistent with the IBR test program results in that:

- The finest product (HPGR Centre) resulted in the highest uranium extraction.
- Increasing the free acidity of the lixiviant from 15 g/L to 25 g/L did not result in a significant increase in uranium extraction.

Unlike the IBR tests, the increase in free acidity did result in a consistent increase in acid consumption for a given uranium extraction.

Additional comments that can be made from observation of this testwork program are:

- No slumping was observed in the columns that tested the conventionally crushed ore to -12.5mm. Therefore, no binder is required in the agglomeration of particle size distributions coarser than tested in this series.
- 6% slumping was observed in the column that tested the HPGR sample (P_{80} = 4.0mm). The competency of the fine HPGR product would likely benefit from agglomeration with a binder.
- 12.5 L/m^2/h irrigation rates did not create any physical problems with liquor percolation through either the fine HPGR product or the coarse conventionally crushed product in the column

13.7.5 Agglomeration and Percolation

<u>Sighter Stage Tests</u>

A number of sighter trials were initiated using NALCO anionic binders, Product Numbers 82296 and 82295, although results of testing were generally not reproducible.

Initial tests were performed with and without acid agglomeration. It was decided that any future agglomeration would include acid as a portion of the agglomeration liquor. Including acid in the agglomeration process would reduce the risk of developing an acid profile down the height of the heap which could reduce the effectiveness of leaching for an On/Off Pad design.

Agglomerate preparation to date has proved that the final results are very sensitive to sample mixing, wetting and reagent dilution. This is because the typically sandy nature of the ore lacks sufficient clays or other soft fine fractions that makes the agglomeration preparation stage more forgiving.

Magnafloc 351 Non-Ionic Binder Tests

Successful agglomeration trials were performed using non-ionic Magnafloc 351 binder. Testing in this phase also used acid addition in the agglomeration liquor that would contribute to acid requirement in the leaching stage.

Results of final testing are presented below.

Figure 13-44

Agglomeration and Percolation Tests MF351

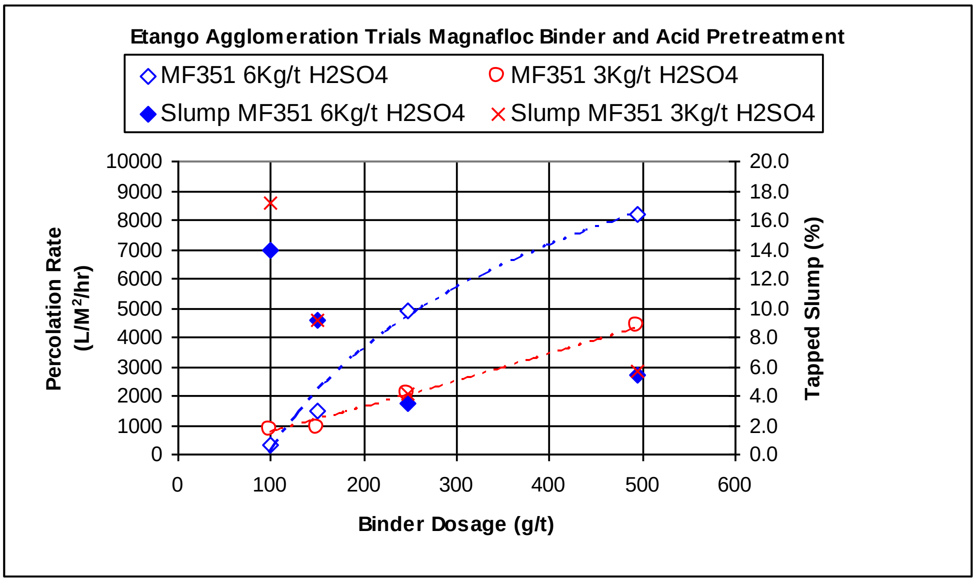


Table 13-36 **Agglomeration and Percolation MF351 Non-Ionic Binder**				
Test Number	**13**	**14**	**15**	**16**
Ammtec Number	**MH8194**	**MH8195**	**MH8196**	**MH8197**
Starting Sample Moisture (%)	2.00	2.00	2.00	2.00
MF351 Stock (%)	0.3	0.6	0.3	0.6
MF351 Dosage (g/t)	**248**	**495**	**248**	**495**
100% H_2SO_4 (kg/t)	**2.97**	**2.97**	**5.95**	**5.95**
Solution Added % of Dry Solids	12.1	12.4	12.0	12.3
Agglomerate Cure (days)	3	3	3	3
Drained Solution	Clear	Clear	Clear	Clear
Drained Solution pH	0.96	0.98	0.94	0.96
Auto Slumpage (%)	0.0	0.0	0.0	0.0
Tapped Slumpage (%)	**4.1**	**5.7**	**3.4**	**5.5**
Percolation Rate (L/m^2/h)	**2,096**	**4,415**	**4,927**	**8,178**

The results indicate that agglomeration liquor which uses 250g/t Magnafloc 351 binder and 6 kg/t of acid will result in a competent heap that will resist slumping. All percolation rates measured were acceptable and the results plotted in Figure 13-44 also indicate that percolation rate will increase with increasing binder addition rates.

Figure 13-45 shows typical agglomerate formed during the tests.

Figure 13-45

Typical Agglomerate 250 g/t MF351 and 6 kg/t Acid Addition

13.7.6 Secondary Program of Four Metre Column Tests

Samples

A composite of approximately 875kg was prepared from the leftover samples of the previous IBR program (Section 13.7.3) on samples crushed to: -50mm, -25mm and -16mm. The resultant composite was stage crushing to -16mm.

Testwork Scope

The bulk composite was homogenised and split in half to allow preparation of four (4) samples as follows:

- Two (2) samples prepared by control crushing -12.5mm to achieve a P_{80} size of 8mm and representing a conventional crusher PSD.
  This procedure was as used for previous conventionally crushed products used in previous open circuit column tests (Section 13.7.4).
- Two (2) samples prepared by locked cycle HPGR crushing to achieve a P_{80} size of 4mm.

Ore was agglomerated and four (4) 190mm x 4m columns were set up and operated using synthetic process water per Table 13-37.

Table 13-37

4m Column Test Sample Description and Conditions

Sample Details				**Agglomeration**		**Leach Conditions**	
Crush Method / Test Description	**P_{80} (mm)**	**Test Identifier**	**Head Grade U (ppm)**	**Acid (kg/t)**	**MF351 (g/t)**	**Irrigation Rate ($L/m^2/hr$)**	**Lixiviant Acidity (g/L)**
Conventional Open Circuit	8	A1	198	8.0	-	15	20
HPGR Open Circuit	4	E1	214	5.9	250	15	20
Conventional Closed Circuit	8	A2	202	8.0	-	15	20
HPGR Closed Circuit	4	E2	206	5.9	250	15	20

For each ore preparation (Conventional and HPGR), a pair of columns was set-up to test:

- An open circuit configuration where the column was irrigated with fresh lixiviant
- A closed circuit configuration where:
  - The column was irrigated with fresh lixiviant for the first four days of operation,
  - Column discharge collected was contacted with Alamine 336 (batch solvent extraction) to create a raffinate solution assaying approximately 50ppm uranium
  - Re-acidified and re-oxidised raffinate was re-introduced to the column

Common conditions for all column tests were:

- Target irrigation rate of 15 $L/m^2/hr$
- The fresh lixiviant / raffinate was re-acidified to 20 g/L
- Ferrous sulphate was added as required to ensure that the column feed contained greater than 500ppm of iron in solution
- 30% (w/w) peroxide (H_2O_2) was added as required to ensure that the pulp potential of lixiviant was greater than 500mV (standard Calomel) such that ferrous was predominantly converted to ferric.

<u>Testwork Results</u>

Open and closed circuit leach profiles for the conventional and HPGR prepared samples are presented in Figure 13-46 and Figure 13-47 respectively.

Figure 13-46

4m Column Uranium Extractions - Conventional Preparation

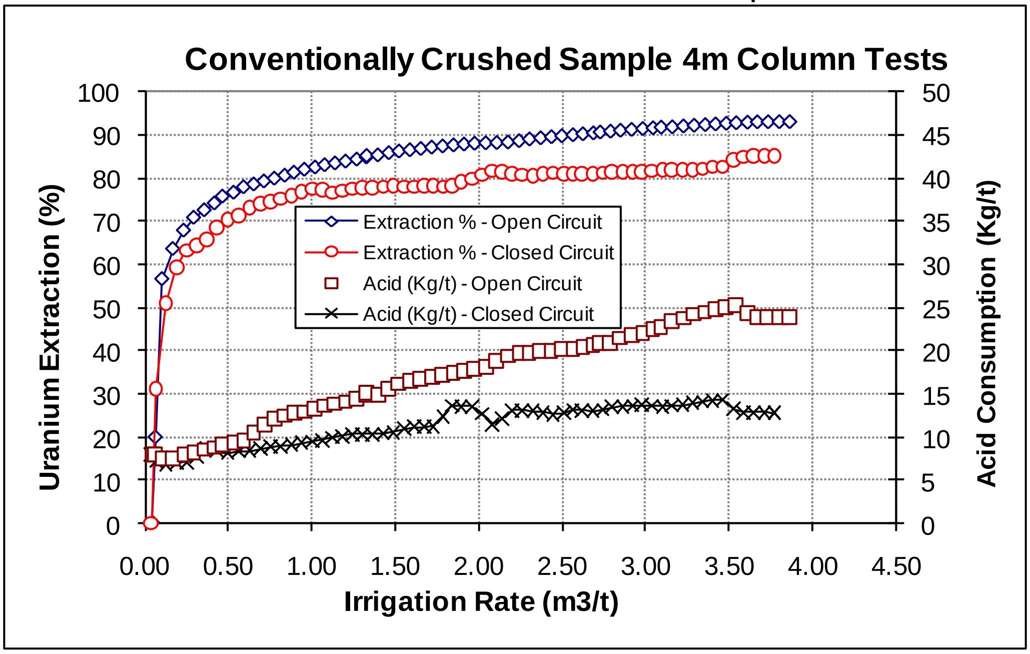


Figure 13-47

4m Column Uranium Extractions - HPGR Preparation

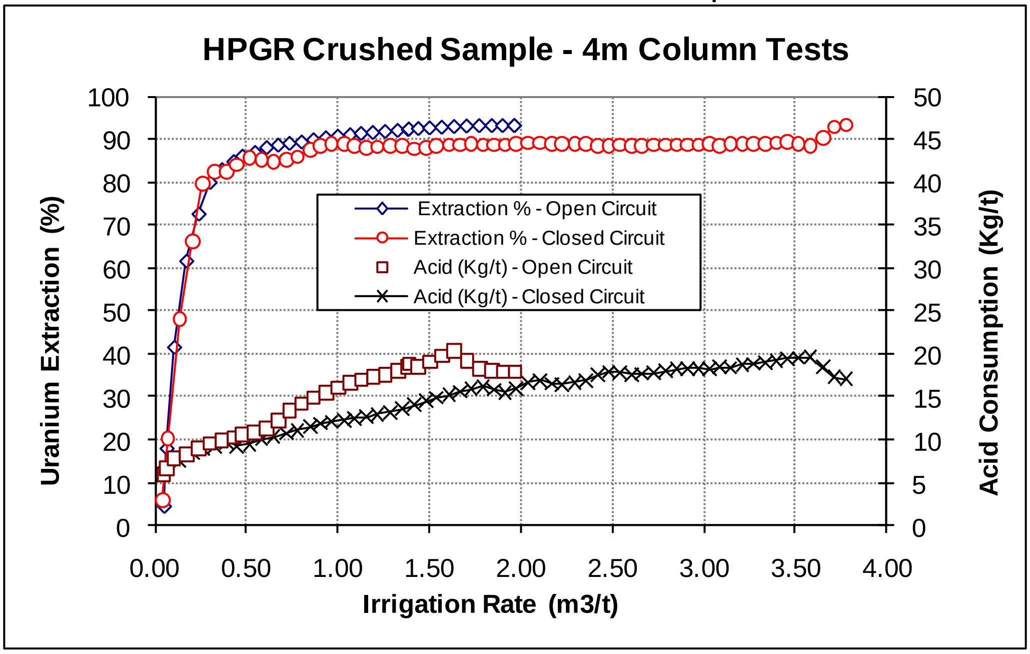


Figure 13-46 and Figure 13-47 show that the kinetics is only marginally slowed, if at all, by closing the column with raffinate that has passed through a solvent extraction stage.

Acid consumption continues to increase linearly throughout the duration of all leach tests, even once the uranium extraction levels off. This suggests that the dissolution of uranium is not contributing significantly to the acid consumption, and that the gangue mineralisation is dominating this operating cost.

As a guide to acid efficiency, the uranium extractions versus acid consumption relationships are charted below in Figure 13-48. There was no apparent difference between the open and closed circuit relationships for each crush product, so all of the data from each crush product is presented as a single series.

Figure 13-48

4m Column Uranium Extractions versus Acid Consumption

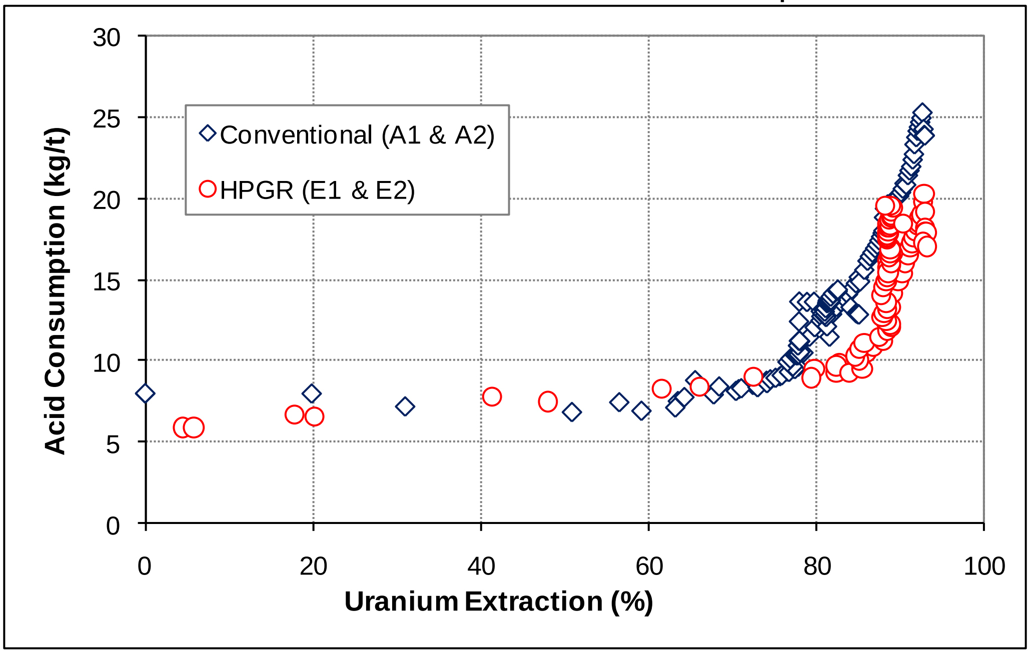


Figure 13-48 shows that the HPGR prepared ore is marginally more efficient however this is most likely due to the increased uranium extraction that is achieved from the finer leach feed.

Column residues were sampled over each metre from top to bottom, head assaying and assay by size deportments were determined for each metre section sampled. Average results are shown in Table 13-38 for each column test.

Table 13-38

4m Column Residue Assays and Conditions

Description	Identifier	U (ppm)	V (ppm)	Th (ppm)	% Moisture	% Slump
Conventional Open Circuit	**A1**	14	9	23	3.7	0.3
HPGR Open Circuit	**E1**	15	6	24	8.3	0.8
Conventional Closed Circuit	**A2**	31	7	17	5.1	0.0
HPGR Closed Circuit	**E2**	14	8	15	9.0	1.0

13.7.7 Variable Testing – Two Metre Column Program

A suite of open circuit 190mm x 2m column tests were commissioned to provide data for assessing the following variables ahead of a planned large diameter column test during the DFS stage:

- Variation in free acid concentration.
- Variation in crush size

A separate composite was prepared for this stage of testing and comprised a mixture of ½ NQ[2] as well as whole HQ core as shown in Table 13-39.

Table 13-39 Heap Leach Variable Test Composite Details				
Hole Number	**Type**	**Metres**	**Mass (kg)**	**U_3O_8e (ppm)**
GOADH0029	½ NQ[2]	68.3	153.7	206
GOADH0030	½ NQ[2]	40.0	90.0	223
GOADH0033	½ NQ[2]	180.5	406.0	213
GOADH0039	½ NQ[2]	121.3	272.9	279
GOADH0062	HQ	2.0	16.7	356
GOADH0063	HQ	2.3	18.6	451
GOADH0064	HQ	1.2	9.4	479
GOADH0065	HQ	1.2	9.3	375
Total		**416.8**	**976.6**	**242**

Testwork Scope

A total of nine (9) 190mm diameter x 2m columns were commissioned to test the effect of variable crush products and lixiviant acidity. All tests were conducted under the following common target conditions:

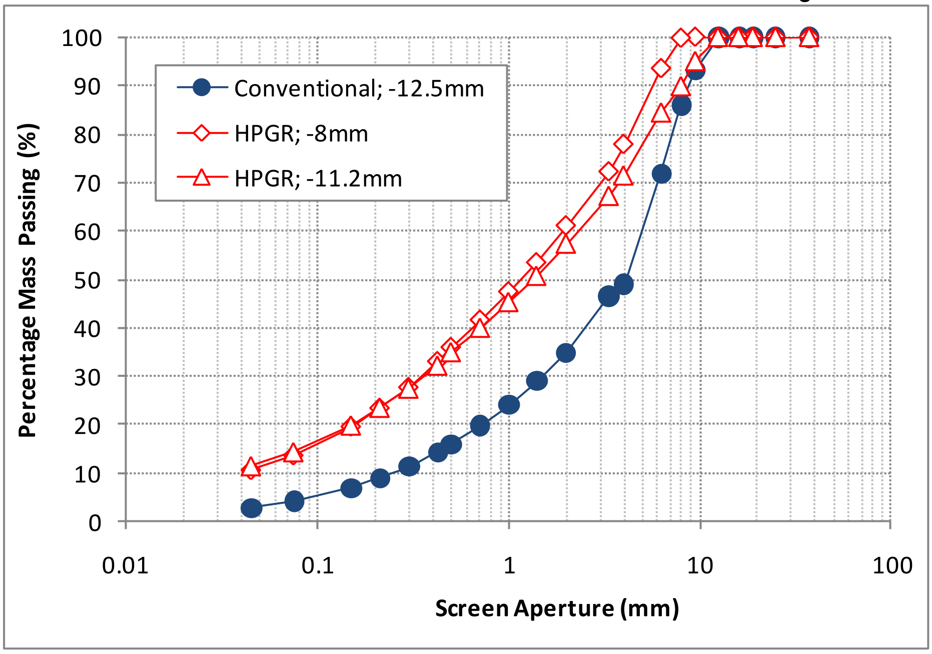
- Irrigation Rate – 15L/m^2/hr.
- Acid agglomeration equivalent to 30% of expected final leach acid consumption.
- Binder agglomeration of HPGR samples using 250g/t MF351 non-ionic binder.
- 30% (w/w) peroxide as oxidant maintaining an Eh of +500Mv (std cal).
- Ferrous Sulphate addition to maintain ferric above 500ppm.

The variables tested and general test descriptions are presented in Table 13-40.

Table 13-40 2m Column – Variability Program Conditions							
Sample Details				**Agglomeration**		**Leach Conditions**	
Crush Method / Test Description	**P_{80} (mm)**	**Test Identifier**	**Back-Calc Head Grade U (ppm)**	**Acid (kg/t)**	**MF351 (g/t)**	**Irrigation Rate (L/m^2/hr)**	**Lixiviant Acidity (g/L)**
Conventional (-12.5mm)	7.3	A1	193	4.8	-	15	20
Conventional (-12.5mm)	7.3	A2	174	4.8	-	15	15
Conventional (-12.5mm)	7.3	A3	187	4.8	-	15	10
HPGR (-11.2mm)	5.1	E1	166	3.6	250	15	20
HPGR (-11.2mm)	5.1	E2	188	3.7	250	15	15
HPGR (-11.2mm)	5.1	E3	202	3.6	250	15	10
HPGR (-8.0mm)	4.3	E4	177	3.5	250	15	20
HPGR (-8.0mm)	4.3	E5	216	3.5	250	15	15
HPGR (-8.0mm)	4.3	E6	200	3.5	250	15	10

Figure 13-49

Particle Size Distributions of Crushed Products for 2m Column Program



Testwork Results

The overall reagent consumption and uranium extractions for the variability program are summarised in Table 13-41. Figure 13-50 and Figure 13-51 below.

Table 13-41

Heap Leach Variable Test Samples and Conditions Day 48

Sample				Agglomeration		Extraction & Consumption @ Day 48			
Identifier	Head Grade U (ppm)	P_{100} (mm)	P_{80} (mm)	Acid (kg/t)	MF351 (g/t)	$FeSO_4$ (kg/t)	H_2O_2 (kg/t)	Extraction (%)	Acid (kg/t)
A1	193	12.5	7.3	4.8	-	12.6	1.9	86.4	24.8
A2	174	12.5	7.3	4.8	-	12.9	2.0	84.4	23.1
A3	187	12.5	7.3	4.8	-	12.8	2.5	79.4	17.0
E1	166	8.0	4.3	3.5	250	13.4	2.1	94.4	30.5
E2	188	8.0	4.3	3.5	250	14.4	2.2	94.5	34.5
E3	202	8.0	4.3	3.5	250	13.9	2.8	94.2	24.8
E4	177	11.2	5.1	3.5	250	12.5	1.9	89.7	30.1
E5	216	11.2	5.1	3.5	250	13.5	2.0	91.2	27.7
E6	200	11.2	5.1	3.5	250	13.2	2.6	91.1	24.8

Figure 13-50

2m Column Variability Program – Uranium Extraction Curves

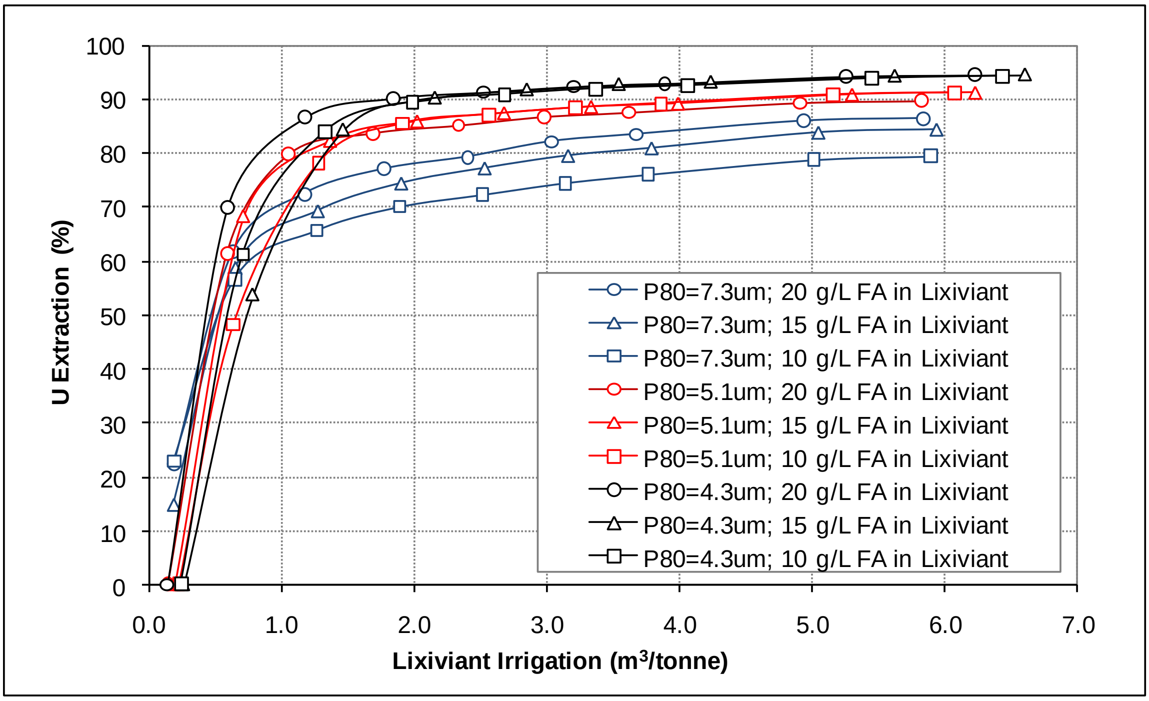


Figure 13-51

2m Column Variability Program – Acid Consumption against Uranium Extraction

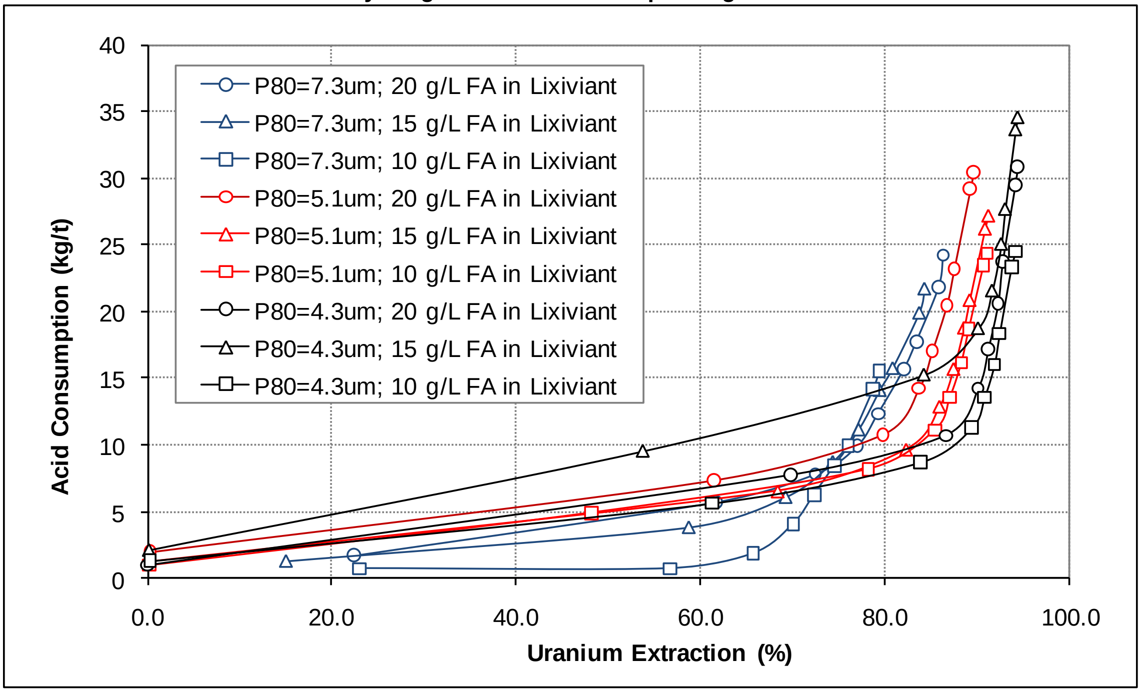


Figure 13-50 once again demonstrates the increase in extraction as the crush product becomes finer. For the finer crushed products, the variation in extraction is negligible as the free acidity in the lixiviant ranges from 10-20 g/t.

The earliest IBR program suggested that efficiency of acid consumption against uranium extraction decreased as the lixiviant free acidity increased, however this has not been clearly observed in any of the column leaching tests. It is logical however to decrease the risk of inefficient acid use by setting the minimum free acidity required to achieve acceptable uranium extraction.

Whilst 10 g/L free acid in lixiviant appears to be sufficient, a conservative approach would be to engineer for, and use 15 g/L free acid in lixiviant for all future tests.

The addition of binder to the finer HPGR products does not appear to have any detrimental effect on extractions, and therefore it is recommended that this practise should be included in engineering and any future heap leach testwork programs as it appears to improve percolation and competency (Section 13.7.5).

13.7.8 Conclusions

A summary of conclusions from the heap leach work to date are:

- Crush size is a major driver for maximising uranium extractions. The finest crush sizes have been achieved with HPGR preparation and they have consistently resulted in the highest extractions for each program that compared the effect of crush product size on extraction.
- The competency of the fine crush product (HPGR crushed to <11.2mm) was improved by agglomeration with acid (3.5-5 kg/t) and MF351 non-ionic binder (250g/t). Extraction was not affected by acid and MF351 in agglomeration therefore this should become minimum standard practise when heap leaching finely crushed ore.
- Heap leach testing has shown that Etango ore grade fraction is amenable to heap leaching with high extraction above 80% possible via either conventional or HPGR comminution. Recent cost benefit analysis tracking incremental uranium extraction and continued acid consumption has indicated HPGR preparation is preferred over conventional 3 stage crushing preparation.
- The testwork indicated faster leach kinetics in the case of HPGR preparation, likely as a result of a greater level of liberation and faster initial kinetics on finer fraction.
- On the basis of adopting an On/Off heap leach design acid pre-treatment has been adopted to mitigate the potential for developing an acid profile through the heap. Pre-treatment with approximately 40% of full consumption is indicated as optimal at this time.
- HPGR prepared feed will require binder agglomeration, while conventionally crushed feed at a P_{80} size of 8mm will likely not require binder agglomeration; testing to date shows that addition of non-ionic binder Magnafloc 351 at 250g/t provides an acceptable percolation rate with reduced slump.

13.7.9 Expanded Heap Leach Testwork Program Aug-Nov 2010

The preferred configuration for an operational heap leach is:

- 7 metre high on/off pad sized for approximate 50 day cycle
- Two stage leach configuration where fresh ore is irrigated by intermediate liquor solution (ILS) from partially leached ore
- Spent ore to be stacked separately on a >46 metre high Ripios heap

This configuration introduced a number of process risks that had not been studied at this point. An additional testwork program was therefore undertaken to answer the following parameters:

1. Irrigation behaviour (permeability & stability) and uranium extraction in a 7 metre heap; performance relative to the previously tested smaller columns (i.e. size scale-up).
2. Geotechnical limits of a 7 metre heap
3. Geotechnical assessment (stability) of spent ore stacked to 40 metres on the Ripios heap.
4. Kinetic effects of irrigating fresh ore with ILS compared to fresh lixiviant.
5. Repeatability of column test results and robustness of process performance.
6. Comparison of pyrolusite and hydrogen peroxide as oxidants for the heap leach.

The testwork program conducted from August to November 2010 included the following:

1. A 7 metre column to assess the leaching performance of a column built to design height
2. Geotechnical testwork to assess:
    a. the permeability / percolation of freshly agglomerated ore under load equivalent to a 7 metre height
    b. the permeability / percolation of spent ore under load equivalent to a 7 metre height
    c. the competency and stability of a 40 metre heap constructed with spent ore
3. 2 x 4 metre columns in series to assess the effect of a two-stage leaching configuration on initial leaching kinetics
4. Duplicate 2 metre columns (closed circuit) testing the effect of pyrolusite as the oxidant and the reliability of column test method – 2 columns testing the same conditions
5. Control 2 metre column (closed circuit) that will re-establish the baseline performance of a 2 metre column using standard conditions derived from the earlier programs.
6. 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 testwork results and conclusions are summarised below.

<u>Test work Results</u>

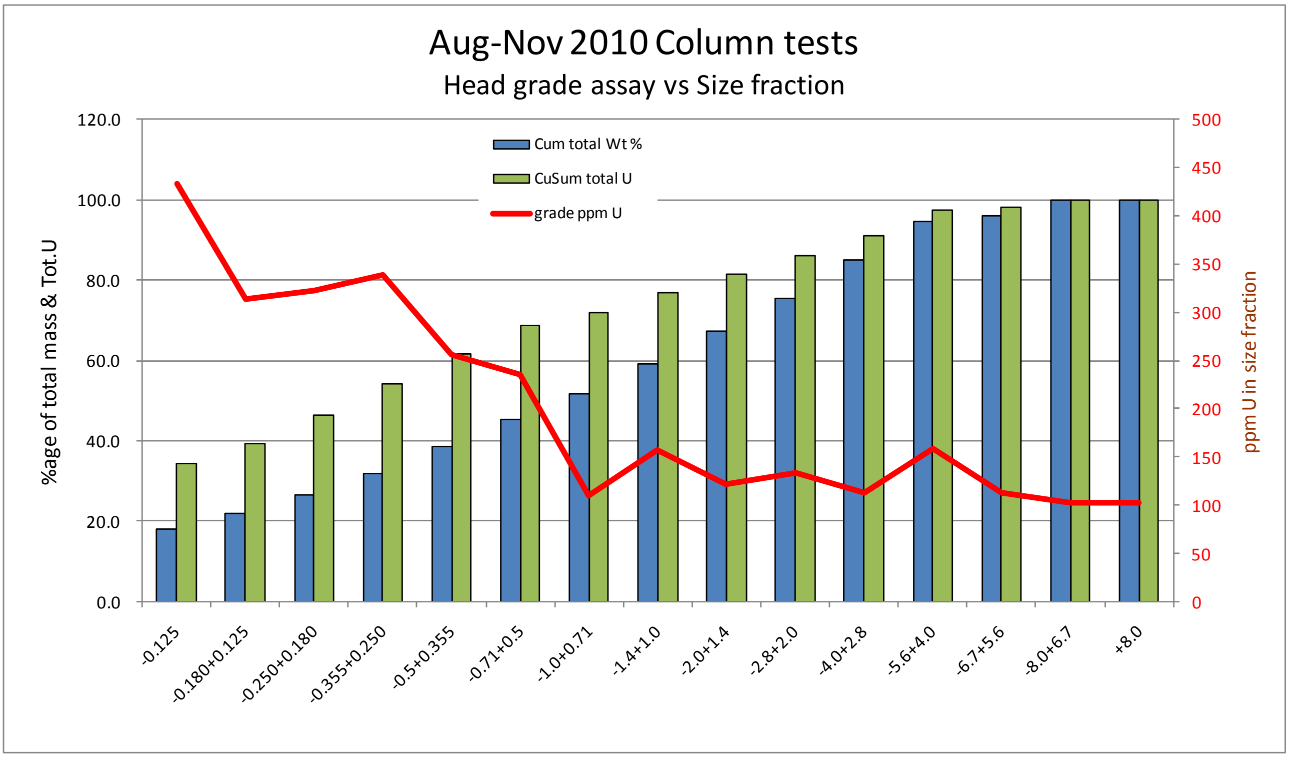
(1) Column Tests

Six 190 mm diameter column tests were conducted at Ammtec between Aug – Nov 2010.

The process parameters were as follows:

1. The feed ore for all columns (designated "Pilot Comp 25") had been crushed to an 8 mm P_{100} (3.375 mm P_{80}) by HPGR. The resultant size and grade distribution is shown in Figure 13-52.

Figure 13-52 Aug-Nov 2010 Column Tests



2. The ore was agglomerated in a lab scale mixing drum with binder flocculant M351 plus 6kg/ t H2SO4 dilution to 25 vol. % to aid distribution. The agglomerated ore was "cured" for 3-4 days while held pending charging to the columns.
3. The process parameters for the column are summarised in Table 13-42 below.

Table 13-42 Column Test Process Parameters

COLUMN	A	B	C	D	E	F
Height	7m	4m	4m	2m	2m	2m
Diameter	190 mm	190 mm	190 mm	190 mm	190 mm	190 mm
Arrangement	open circuit - fresh solution added continuously every day.	open circuit - fresh solution added continuously every day.	1st 24d with Col.B product +acid. Fresh soln thereafter	1st 4d with fresh lixiviant; Thereafter= recycled raff adj to 20gpl acid	1st 4d with fresh lixiviant; Thereafter= recycled raff adj to 20gpl acid	1st 4d with fresh lixiviant; Thereafter= recycled raff adj to 20gpl acid
Dry wt of ore charge kgs	283.2	156.3	156.3	83.0	78.1	78.1
Agglom.acid: % of dry ore	0.61%	0.61%	0.65%	0.61%	0.61%	0.61%
Agglom.curing time (days)	3.00	3.00	3.00	3.00	3.00	4.00
Agglomate "Moisture" as charged	11.50%	11.42%	11.48%	11.41%	11.40%	9.83%
Irrigation rate ltrs/hr/sq.m	14.99	14.988	15.116	14.846	14.957	14.953
Irrig. rate ltrs/ hr/ t ore	1.5	2.7	2.7	5.1	5.4	5.4
Leachant: gpl H_2SO_4	20	20	20	20	20	20.7
Oxidant *1 gm of 30% H2O2 ~= 1.30 gms MnO2*	Daily Constant rate of 1.57 gms 30%H2O2/ ltr irrigation	Daily Constant rate of 1.57 gms 30%H2O2/ltr irrigation	Daily Constant rate of 0.565 gms 30%H2O2/ltr irrigation	Daily Constant rate of 0.595 gms pyrolusite /ltr irrigation	Daily Constant rate of 0.595 gms pyrolusite /ltr irrigation	Daily Constant rate of 0.392 gms 30%H2O2/ltr irrigation
FeSO4.7H2O Addition	Daily addition of 0.53gpl Fe to irrigation liquor	Daily addition of 0.528 gpl Fe to irrigation liquor	zero until Day25 (Fe supplied from Col.B liq.out) From Day25, 0.541 gpl Fe added to fresh leachant.	Daily addition of 0.528 gpl Fe for 1st 4 days. Fe supplied thereafter by the internal recycle of raffinate.	Daily addition of 0.528 gpl Fe for 1st 4 days. Fe supplied thereafter by the internal recycle of raffinate.	Daily addition of 0.528 gpl Fe for 1st 4 days. Fe supplied thereafter by the internal recycle of raffinate.
Irrigation Cycle durations (Days)						
Irrigation (acidic, oxidative) - days	37.0	29.0	37.0	40.0	39.0	29.0
Irrigation (acidic only) - days	-	-	-	36.0#		
1st Drain - days	5.0	5.0	5.0	5.0	5.0	5.0
Rinse - days	8.0	9.0	8.0	11d @20L/d** 9d @30L/d 1d @40L/d	8.0	8.0
Final Drain - days	7.0	6.0	8.0	TBA	7.0	6.0

Column D was irrigated with 20gpl leachant for an additional 36 days to assess acid consumption with extended contact
** Column D was trialled at 2,3 & 4x std irrigation rates during rinse cycles to confirm high bed permeability.

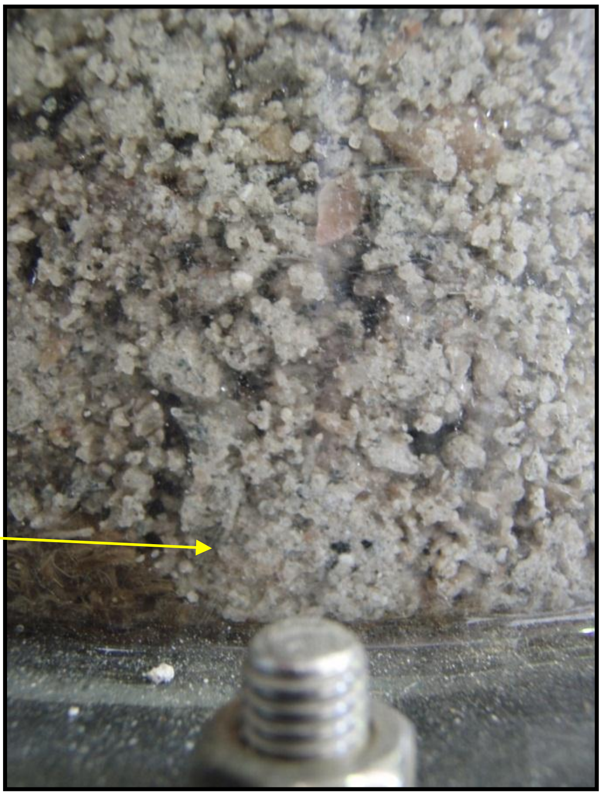
(1)(a) Hydraulic Performance

The hydraulic performance of the columns was as follows: -

1. All columns had liquor break through within 1 day of commencing irrigation.
2. Steady state was typically achieved with ~3 days. Liquor retention was based on Cumulative volume in – volume out.
3. Free liquor in the form of a saturated bed/ "water table" or rivulets was not observed at any time (All test columns had transparent walls). The agglomerated ore structure was retained in all columns throughout the trials (Figure 13-53).

Figure 13-53 Column A (7m tall) after 37days leach, 5d drain, 7d rinse, 9d drain.

Structure of the ore bed was visually uniform over the full height of the column.



The ore bed at base of column was visually open and permeable.

← The top 1m of the column was detached and poured in trays – robust agglomerate structure retained after handling.

4. The measured bed slump in all columns after 40-50 days was < 1.5%.
5. Drainage of the 7m column was >80% complete within 3 days. The measured moisture content of each section of bed following 7 days drainage is shown below.

Table 13-43 Drained residue moisture

Vertical profile down column A	Wt % Moisture
6-7 m (top)	8.0
5-6 m	11.0
4-5 m	10.7
3-4 m	11.6
2-3 m	11.9
1-2 m	11.8
0-1 m (bott)	12.6
Weighted Average: Whole Column residue	11.2

(1)(b) Uranium Extraction and Acid Consumption

The metallurgical performance was consistent across all columns. The performance characteristics were as follows:

1. Uranium extraction is rapid without a "long tail" i.e. 90% approach to end point achieved within 6-11days, the taller columns taking longer reflecting the higher tonnage treated per unit volume of irrigation liquor.
2. The end point was consistent for all columns at ~94% extraction (End points of the 5 tests ranged from 93.3% to 94.8%, averaging 93.9%) (Figure 13-54).

Figure 13-54 Progressive uranium extraction

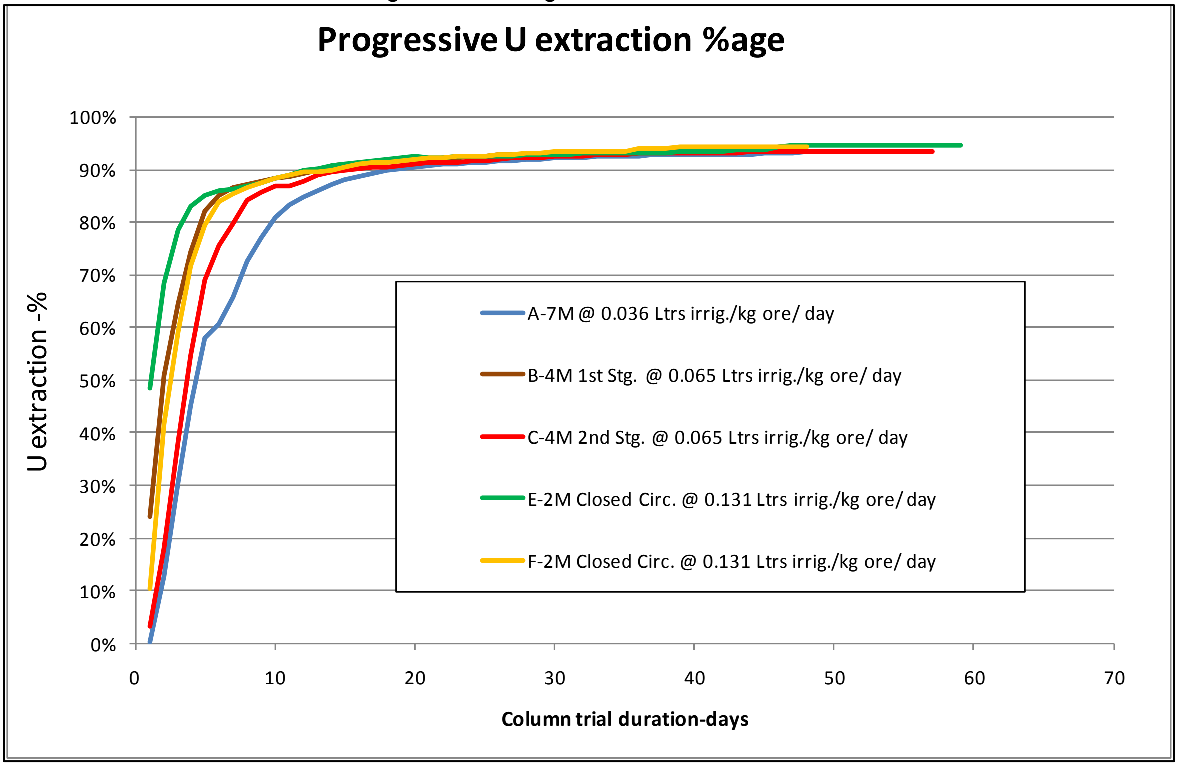


Figure 13-55 Acid Consumption: cumulative gms

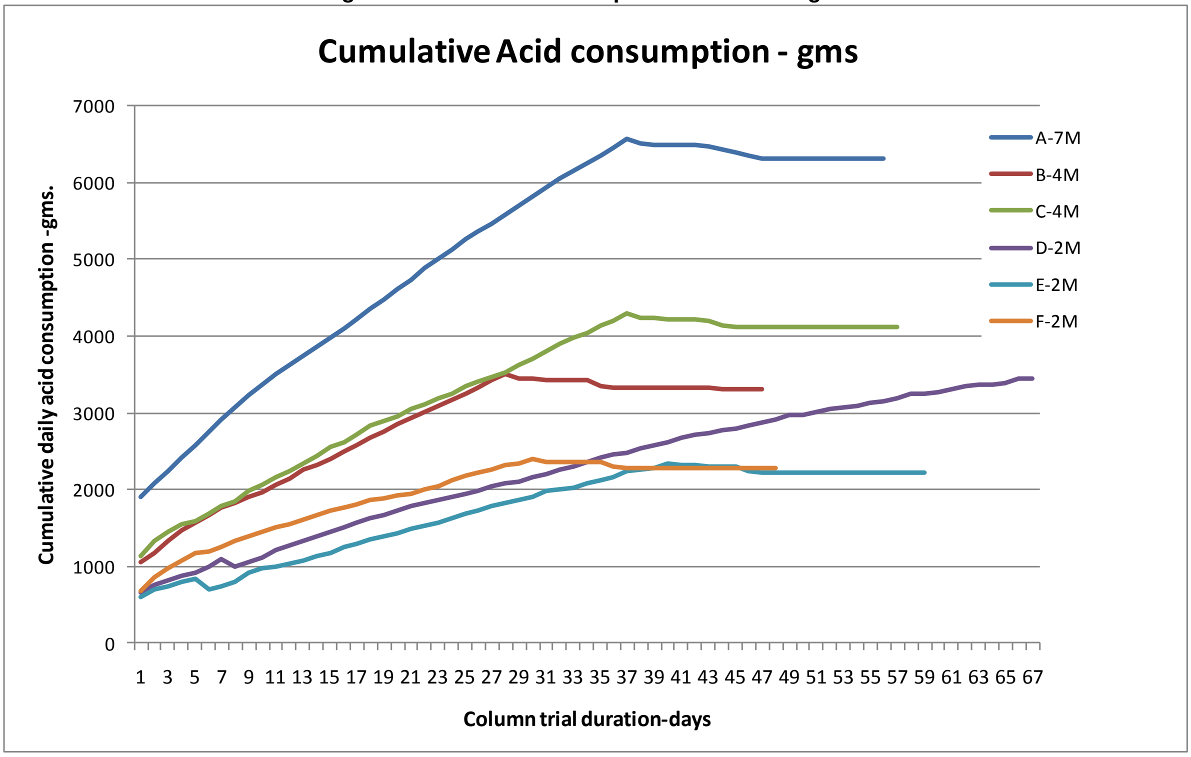
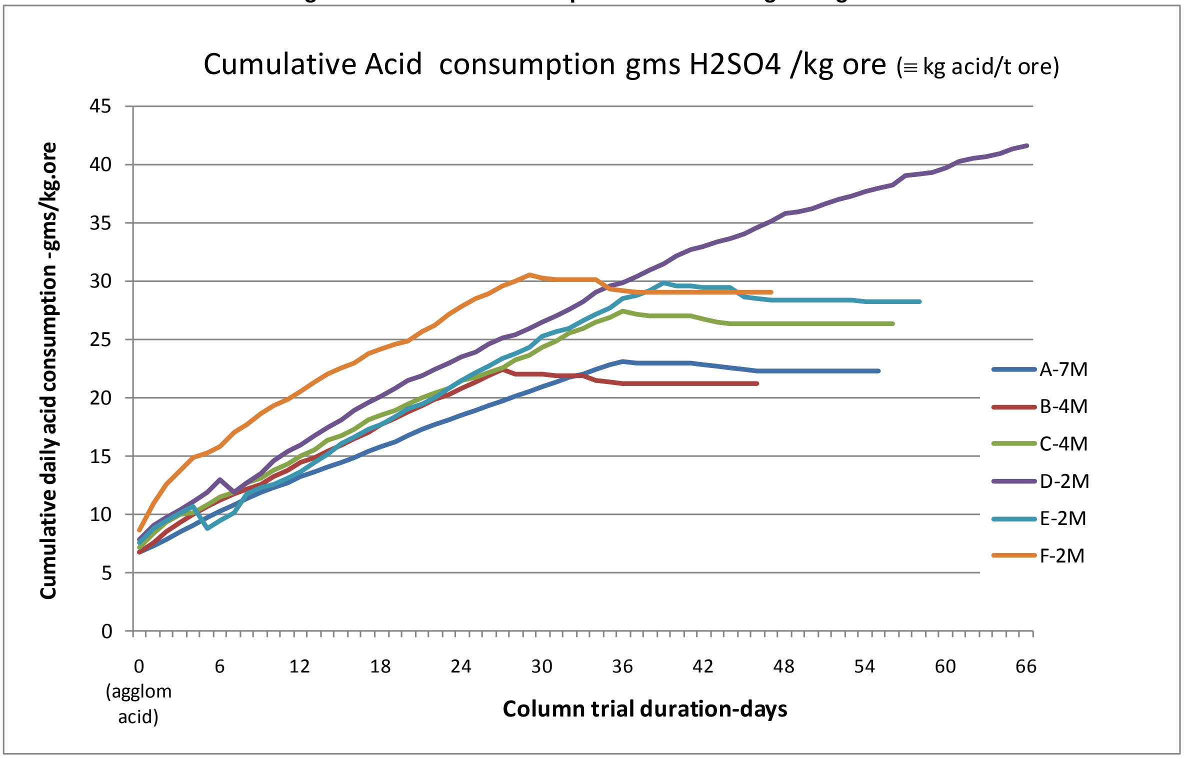


Figure 13-56 Acid Consumption: cumulative gms/ kg ore



All columns used the same irrigation rate of 15l/m^2/hr and feed liquor acid concentration of 20 gpl H2SO4).

The acid consumption trends shown above (graph (a.) cumulative gms), are in order of column height. This is expected as the taller columns increase the time that irrigation liquor remains in contact time with ore. The relationship can be clearly seen in Figure 13-57 showing average Free Acid (gpl) in column liquor out versus column height.

Figure 13-57 Column Product Liquor Free Acid vs column height.

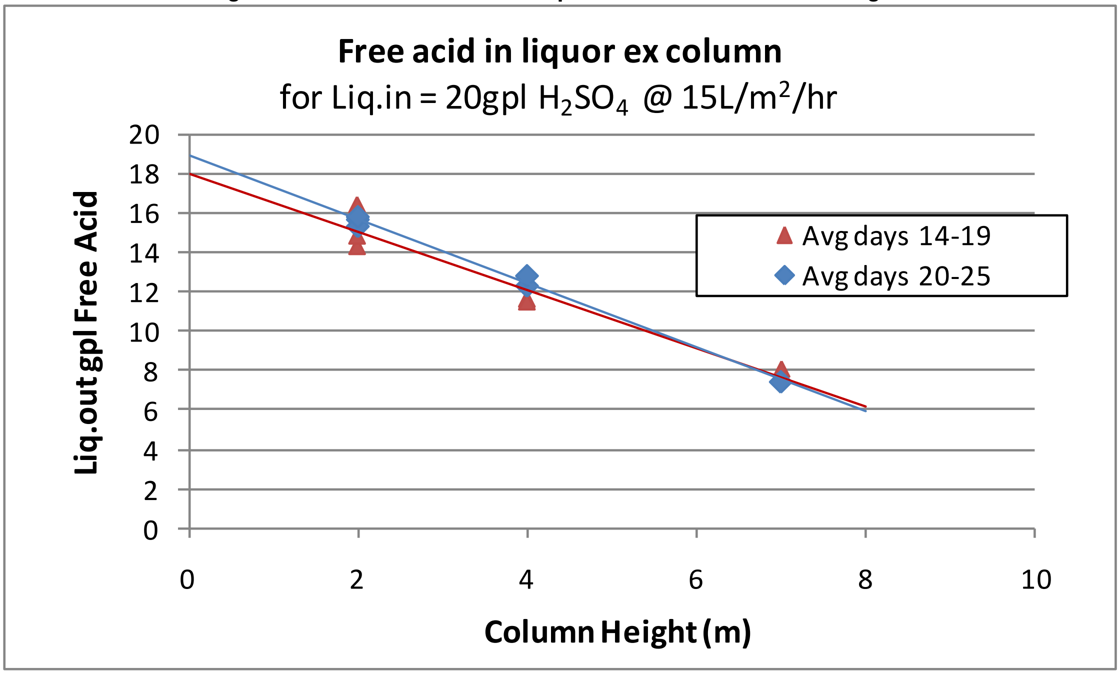
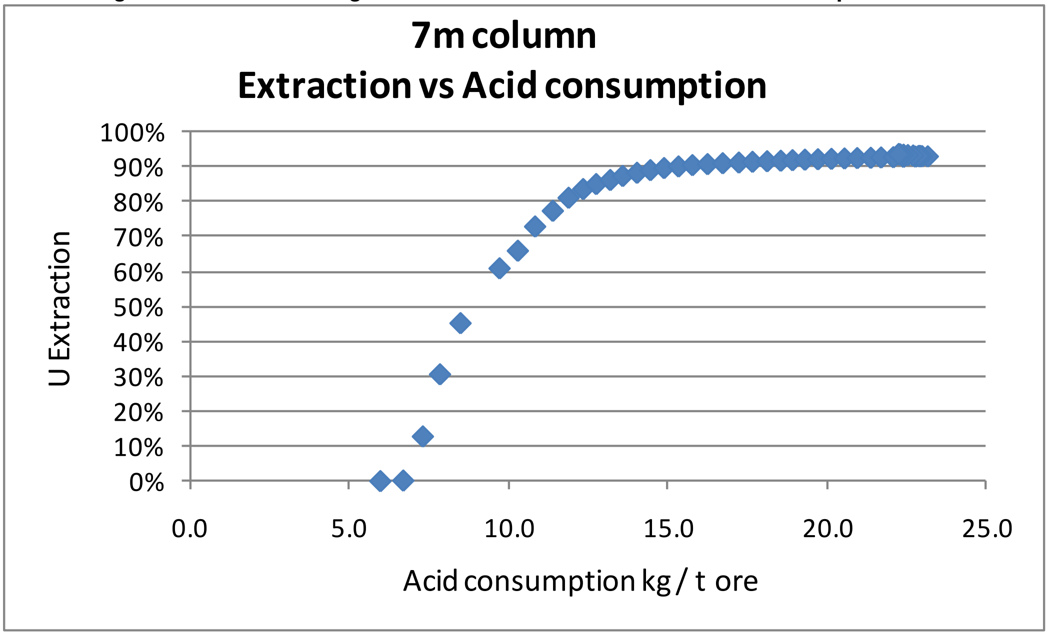


Figure 13-56 shows cumulative acid consumption per kg ore. The series order is reversed as acid consumption rates in the lower sections of taller columns slow due to lower free acid after passing through the upper section of the column.

The acid consumption trend for Column F is partially offset from the other 2m columns D & E. This may be attributable to Column F having a ~ 50% higher liquor retention at steady state than Columns D and E.

The contrasting uranium extraction (asymptotic) and acid consumption (~linear) results in diminishing returns of uranium for increasing acid consumption as seen in Figure 13-58.

Figure 13-58 Diminishing uranium returns for increased acid consumption



The relationship will underpin ongoing value optimisation adjustments with varying uranium market price and varying acid cost.

(1)(c) Iron Reagent

Iron is required in the leach liquor to maintain the Fe^{3+}/ Fe^{2+} redox couple for the oxidative acid leaching of uranium. Accordingly, new iron was added to the Leach feed liquor throughout the open circuit trials. New iron was also added for the first 4 days of closed circuit trials after which the recirculating load of iron in pregnant liquor and SX raffinate met process requirements.

New iron was added as $FeSO_4.7H_2O$ to maintain 0.5 gpl Fe.in Leach feed liquor.

All column trials showed a net production of iron through iron extraction from ore.

The extraction trend for column A is shown in Figure 13-59.

Column C did not require additional iron as the leachant was reacidified, iron bearing product liquor from Column B.

Columns D, E & F were self sufficient in Fe after 4 days when closed circuit recirculation of SX treated product liquor commenced.

Provided a minimum of 0.5 gpl Fe is maintained in raffinate recirculated to the leach stage, no new iron should be required for plant operations.

A net extraction of iron from ore is expected under plant conditions. An iron removal stage may be required to control the recirculating load.

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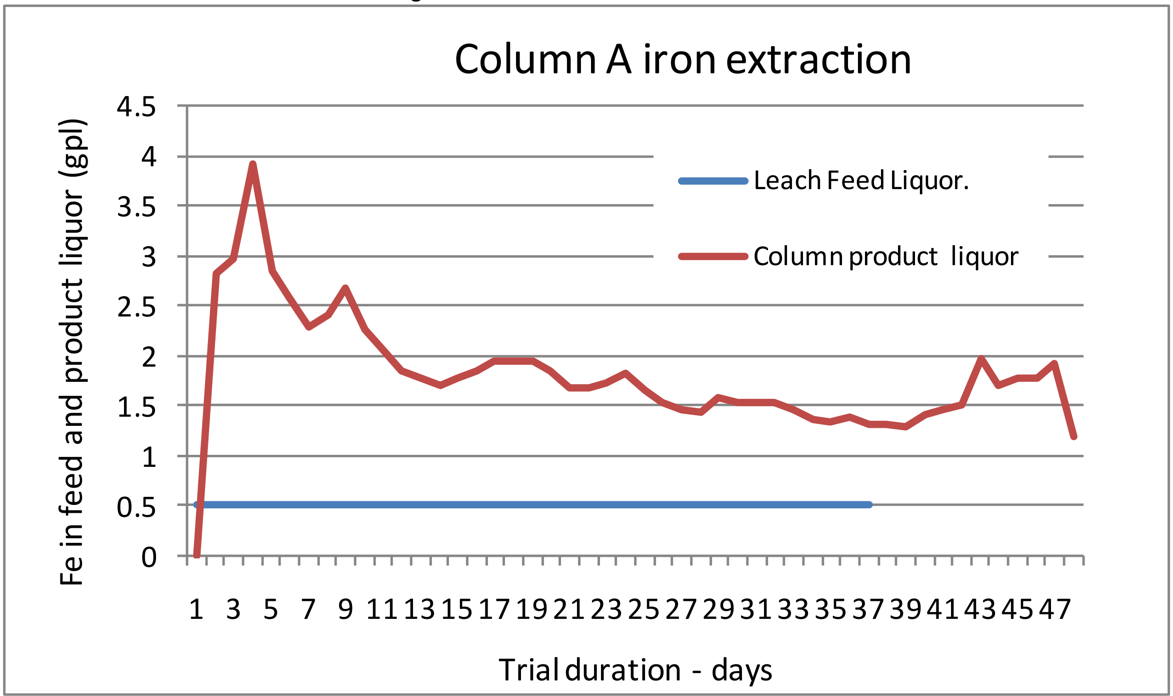
Figure 13-59 Iron extraction from ore



(1)(d) Oxidant Reagent

Oxidising conditions are required during leaching to convert U^{4+} mineralisation to the more soluble U^{6+} form. This may be achieved by: - (a) direct addition of Fe^{3+} (in lieu of oxidising the recirculating Fe^{2+}); (b) Hydrogen peroxide addition; (c) Pyrolusite addition.

Pyrolustite (MnO_2) is the industry standard due to cost and handling advantages, however, the by-product MnSO4 accumulates as a recirculating load in process liquors which is most often controlled through a bleed stream with an associated water loss and containment cost.

Hydrogen peroxide does not generate by-products however it has a higher supply and handling costs and requires stringent safety controls.

Stoichiometrically 1.3 kg of 30vol% H_2O_2 equates to 1.0 kg of MnO_2

Pyrolustite and hydrogen peroxide were assessed as oxidants during the column trials.

Oxidation potential was monitored as Eh (mV) measurements on both feed on product solutions. A target range of 400-500 mV is considered suitable to support the uranium leaching process.

The daily oxidant addition rates are shown in Table 13-44.

Table 13-44 Oxidant Addition Rates

Column	A	B	C	D	E	F
Oxidant	30vol% H_2O_2	30vol% H_2O_2	30vol% H_2O_2	Pyrolusite (MnO_2)	Pyrolusite (MnO_2)	30vol% H_2O_2
Average gms/day added to Leachant #	16.1	16.0	5.8	5.7	5.5	4.0
Average mgs/day/ kg ore to D27	56.8	102.3	37.3	68.4	71.0	51.4
All expressed as equiv 30vol% H_2O_2 **	56.8	102.3	37.3	88.9	92.2	51.4
# All columns irrigated at the same daily volumetric rate ** Stoichiometrically, 1.3gm of 30% H_2O_2 = 1.0 gms MnO_2						

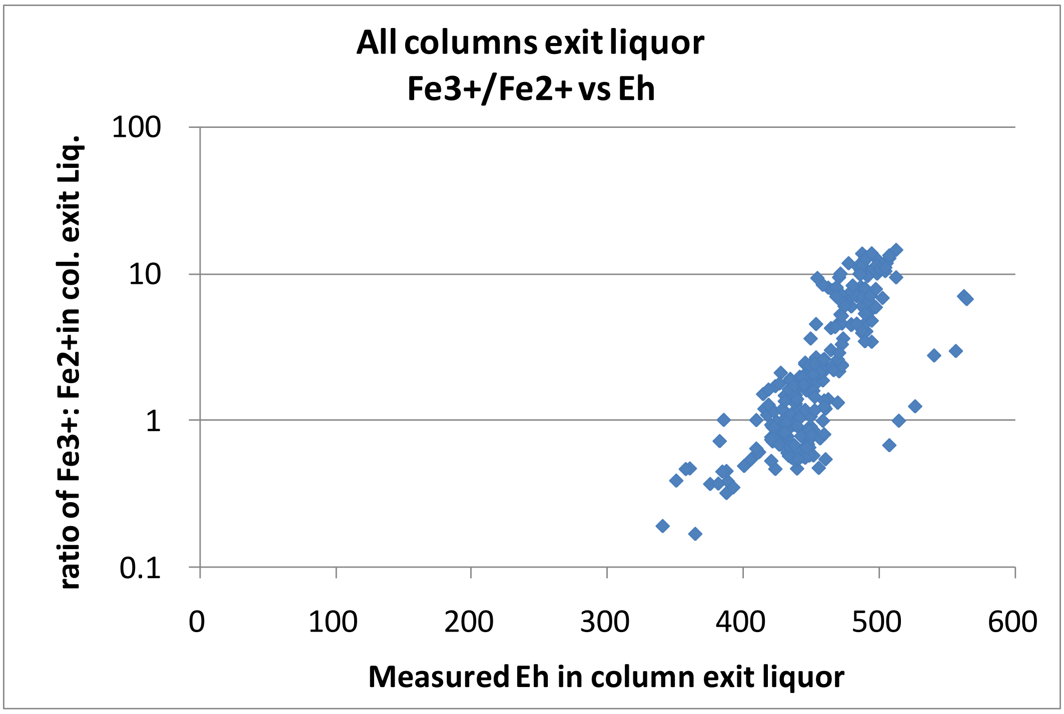
The resultant Eh (mV) for the leach feed liquor and column product liquors are shown in Table 13-45.

Table 13-45 Average Eh during process cycles

	Column A	Column B	Column C	Column D	Column E	Column F
	Eh mV	Eh mV	Eh mV	Eh mV	Eh mV	Eh mV
Leach cycle Liquor in.	582	585	551	544	552	523
Leach cycle Liquor	424	433	444	471	472	475
Water Rinse cycle out	442	446	426		482	446
Acid only irrigation from day 40 – 75				437		

All column product liquor Eh results are shown in Figure 13-60.

Figure 13-60 Eh and Fe3+:Fe2+ results for all daily product liquor from all columns



The results show no correlation with the oxidant addition rates. There were no notable trends in the Eh of Liquor output throughout the trials. It is noteworthy that an average Eh of 437 mV was maintained for acid only (20 gpl) irrigation of column D from day 40 – 75.

The implication is that addition of oxidant may not be required to maintain the target Eh. The current hypothesis is that magnetite in the ore is generating (& buffering) the required Eh when the acid is acidified.

(2) Geotechnical Evaluation

(2)(a) Geomechanical testing of the agglomerated feed ore and residue for Heap Leaching

Bannerman commissioned Golder Associates to perform laboratory geomechanical testing on agglomerated composite ore being the feedstock for heap leach column testwork at Ammtec over the period Aug-Nov 2010. Agglomeration of the ore was proposed to achieve the required bed permeability to support target irrigation rates for the heap leach process.

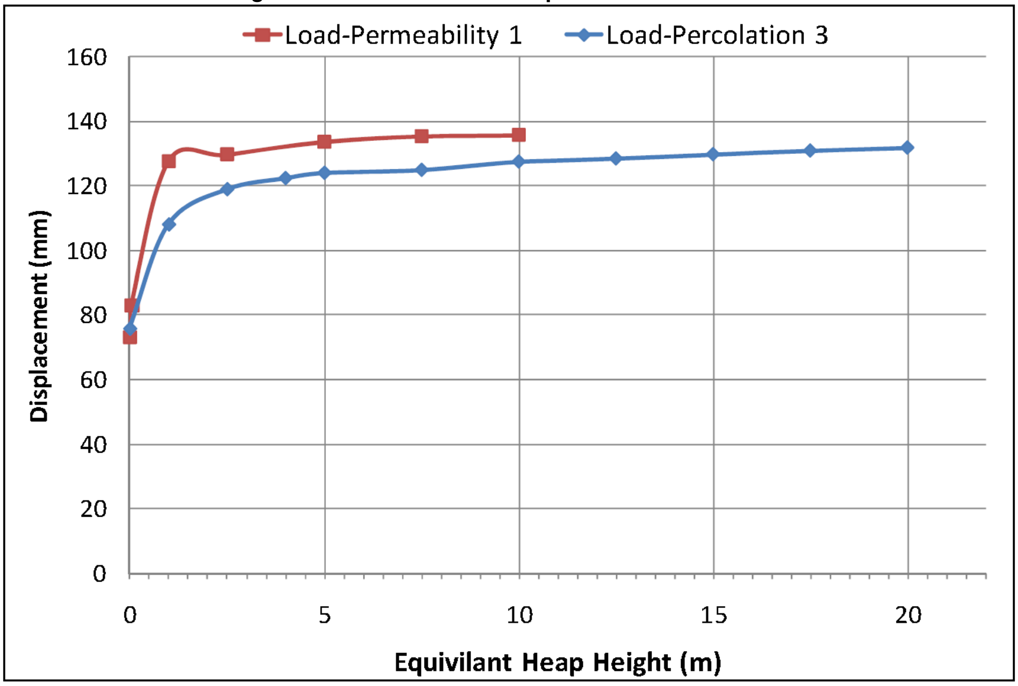
Golders (20) tested and reported the load-permeability and load-percolation rate relationships for the agglomerated feed ore and the final residue (bottom 1m) of the 7m tall Column A, being the two extremes in material structure.

The conclusions and recommendations arising from the study are 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 ~ 80 kPa as demonstrated by the load displacement curve in Figure 13-61. This is typical of sandy materials which have low compressibility. Consistent with this behaviour, the permeability of the heap materilal does not significantly decrease with further load.
- The results of the heap leach work on the feed ore indicate that the target of a 7 m high heap to pass leach liquor liquor at a percolation rate of 15L/m^2/hr is achievable.
- The load percolation tests on "undisturbed" heap leach residue indicated that an application rate of 160[10] L/m^2/hr was achievable before ponding occurred .
- The "disturbed" heap leach residue test involved removal of the residue from the 190 mm diameter column by hand, placement of it into a 258 mm diameter mould, distribution and tamping by hand, then placement of mesh and piston on the surface to carry out the test. The "disturbed residue commenced ponding at an application rate of 5 L/m^2/hr
- It was recommended that additional studies be undertaken during subsequent stages of the study to assess the sensitivity of the results to the quality of the agglomerates.

[10] Confirmed as one hundered and sixty litres / m^2/hr

Figure 13-61 Load versus Displacement for feed ore



(2)(b) Stability Analysis for Residue Storage

Bannerman commissioned Golder Associates to conduct geotechnical stability analysis for the proposed residue (ripios) facility. The residue material is to be removed from on-off heap leach pads and placed within a lined residue storage facility to a height of ~ 23metres.

The testwork was conducted using residue product from the Ammtec 7m column trial, of Aug-Oct 2010.

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

The preliminary results 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 residue can be formed at a maximum batter of 2.5H:1V (~22^o).
- The underdrainage system should be designed to maintain a phreatic level that can be managed to 10% of heap height.
- It is recommended that slope configuration also facilitate closure and long term stability of the final landform.
- The design should also consider the geotechnical stability of the foundation below the residue storage facility.
- Retaining regular benches on the outer slope is not recommended as it will concentrate flow of water and lead to erosion.
- Surface water and run off adjacent to the facility needs to be controlled.

13.8 Future Work Plan

The future work plan for Etango is designed to improve the understanding of the engineering design criteria, and mitigate any perceived technical risks of the project. Key programs that have been initiated include:

- Investigation of processing techniques to decrease operational acid consumption
- Investigation of effect of waste rock on heap leach performance
- Waste rock classification
- Development of solvent extraction technical knowledge
- Updating process model
- Large scale piloting of leach and solvent extraction

13.8.1 Investigation of processing techniques to decrease operational acid consumption

Acid consumption is recognised as significant contributor to the project operating cost and a range of small scale column tests have been committed to with the aim of finding operating conditions that will decrease acid consumption. Specifically, a range of column testwork programs have been initiated in ALS Ammtec (Perth, Australia) and Bureau Veritas (Swakopmund, Namibia) to test a range of liquor acidities, liquor irrigation rates and agglomeration acid dosages to define their respective effect on uranium extraction and acid consumption.

These programs will result in an optimised acid addition and liquor irrigation strategy that will ensure the most efficient use of acid to achieve the most profitable uranium extraction.

13.8.2 Investigation of effect of waste rock on heap leach performance

To date, the majority of leaching testwork has been conducted on composite samples that are representative of the anticipated average grade lithology over the life of the mine. However, it is recognised that significant variations in ore dilution will likely occur on a daily basis during operation and this may affect the efficiency of the leaching process with respect to uranium extraction, deleterious element extraction and reagent consumption.

To understand this effect, two programs have been initiated:

1. Waste rock classification – this program will define the mineralogy of waste rock that will report to the heap leach process

2. Column testing on ranges of waste rock dilution – this program will test a range of dilution percentages for a range of waste rock types in a standard column leach test. Variations in uranium extraction, reagent consumptions and deleterious elements will be measured such that they can be tested on the process model of the processing facility.

13.8.3 Development of Solvent Extraction Technical Knowledge

To date the solvent extraction circuit performance has been based upon mineralogy assumption and process performance assumptions generated from the extensive experience of experts engaged by Bannerman (AMEC Minproc, Bateman Engineering). These assumptions have been incorporated into the process model which has been used to predict overall process performance and define the capacities of the various processes.

Now that the leaching performance and likely pregnant liquor solution (PLS) has been suitably defined, Bateman Engineering has been engaged to design and execute a laboratory solvent extraction program that will measure isotherms for the solvent extraction, scrubbing and stripping stages of the process using the best estimates of feed pregnant liquor solution (PLS) composition as the starting point.

The data generated from this program will be used to update the process model of the operation, therefore obtain more accurate predictions of operational performance, liquor compositions and process capacity requirements.

13.8.4 Update Process Model (MetSim model)

As with most new hydrometallurgical developments, an accurate process simulation is a critical component of the design process because the resultant mass balance is used to define equipment capacities required and potentially identify the build-up of problematic elements that will need to be managed.

All relevant data generated from the aforementioned leaching and solvent extraction testwork programs will be used to update the process model of the proposed operation, the improved process model will be used to verify, or improve the definition of capital and operating costs for the project.

13.8.5 Large scale piloting of leach and solvent extraction

The majority of testwork has been focused on defining each of the states of the process independently, however it is recognised that the successful integration of the unit processes has been calculated via the process model generated by AMEC Minproc (MetSim model).

A large scale piloting campaign is planned to demonstrate the integrated performance of the leaching and solvent extraction components of the flowsheet in a continuous manner. This program will include the following components:

- Full height heap leach CRIB's that will configured as per the design criteria i.e. continuous integration of initial leaching stage, intermediate leaching stage, drainage and washing stages to generate PLS, ILS and rinse product
- Solvent extraction using PLS generated from leaching stage
- Scrubbing, and stripping to generate barren liquor for return to the leaching stage

The continuous integration of the leaching and solvent extraction components of the flowsheet will provide the final validation of the process model and demonstrate the operational performance in a continuous manner.

14 MINERAL RESOURCE ESTIMATES

14.1 Etango Project Mineral Resource

The October 2010 Resource update (Table 14-1) represents an incremental increase in the Etango Ordinary Kriged (OK) resource endowment; a previous estimate was completed in March 2010 utilising Uniform Conditioning (UC).

Until such time as SMU issues are resolved with current Multiple Indicator Kriging (MIK) trial model, the recommended Resource model has reverted back to the OK estimate, as recent MIK studies have indicated that the UC methodology used in the previous estimate was not handling the SMU modelling and dilution in an optimal manner. The alternative is to eventually utilise the MIK SMU model for reporting purposes.

This estimate includes 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
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

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 lbs to kg = x 2.20462

14.1.1 Introduction

In August 2010, Coffey Mining was requested to undertake a Resource update of the Etango Project. This document details the steps taken in preparing the October 2010 Ordinary Kriged estimate.

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

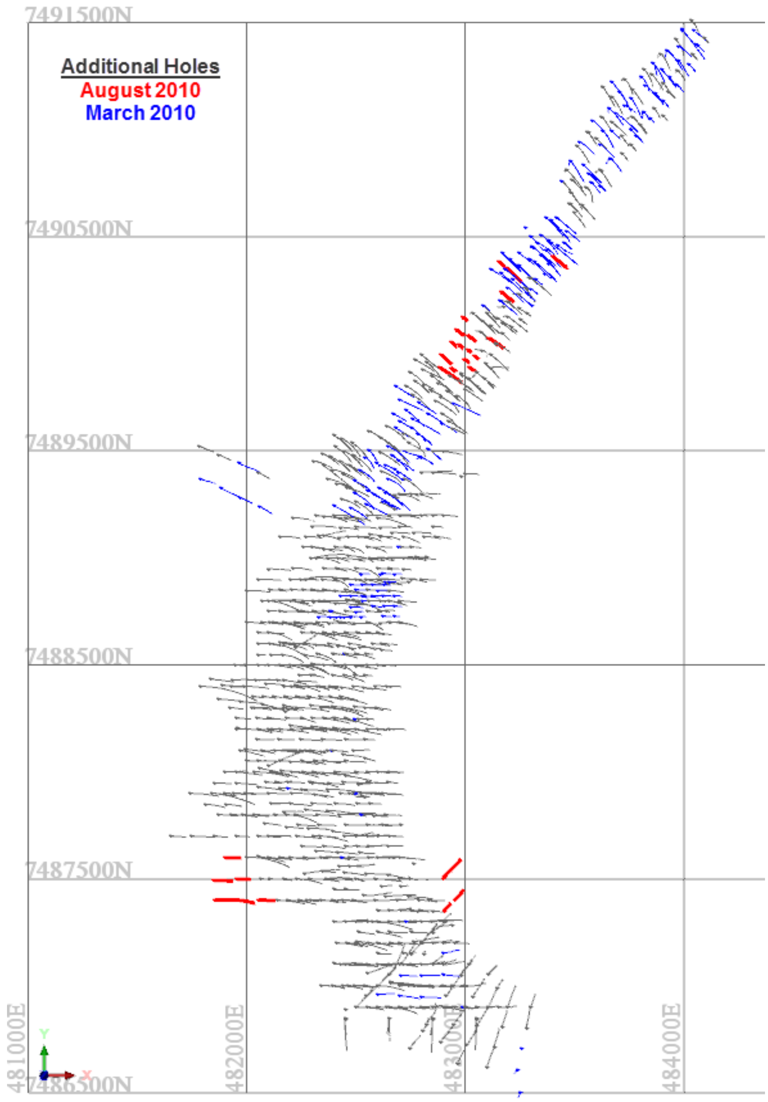
This chapter concentrates of the estimate methodology undertaken. The QA/QC, geology, sampling and drilling procedures are discussed in detail elsewhere in this document.

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. Ordinary Kriging (OK) was used as the method for estimating the resource. The OK model is currently being run in conjunction with parallel MIK trial estimates. The OK model is the subject of this report.

The Qualified Person responsible for the Etango Project resource estimate is Mr Neil Inwood (Principal Resource Consultant) who is employed with the consultancy Coffey Mining. The details of the resource estimations are summarised later in this section.

Figure 14-1
Etango Uranium Project
Plan View of Drilling Locations



14.1.3 Resource Database and Validation

Database

The drillhole database in the direct area of the Etango resource used for the October 2010 resource estimate consists of 913 RC and 145 diamond drill holes for 246,950m. For the October 2010 resource update, only drillholes drilled by Bannerman have been used in the estimate. Figure 14-1 displays the location of the drillholes used in the estimate and highlights the additional holes used for the October 2010 update.

The drillholes were drilled typically at 60° to the east (UTM grid) with a drill spacing ranging from 25m by 50m, to 50m by 50m and 50m by 100m.

A total of 58,065 chemical (93%) and radiometric (7%) assays were used in the estimate. A density value of 2.64t/m3 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 for crushing, pulverisation and chemical analysis. SGS Johannesburg conducts the analyses and is a SANAA accredited laboratory (T0169). Samples are analysed by pressed pellet X-ray fluorescence ('XRF') for U3O8, Nb, Th and 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 U3O8). Intervals which were not sampled internal to mineralised zones were treated as null values (i.e. no samples), affecting 156 1m intervals.

Validation

The October 2010 drillhole 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) 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, with typically little coherent mineralisation occurring in the surrounding sediments. The alaskite contacts were therefore considered at the time of 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 drillholes, with grades typically falling below 30ppm to 20ppm U_3O_8 away from the logged alaskite contacts.

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.

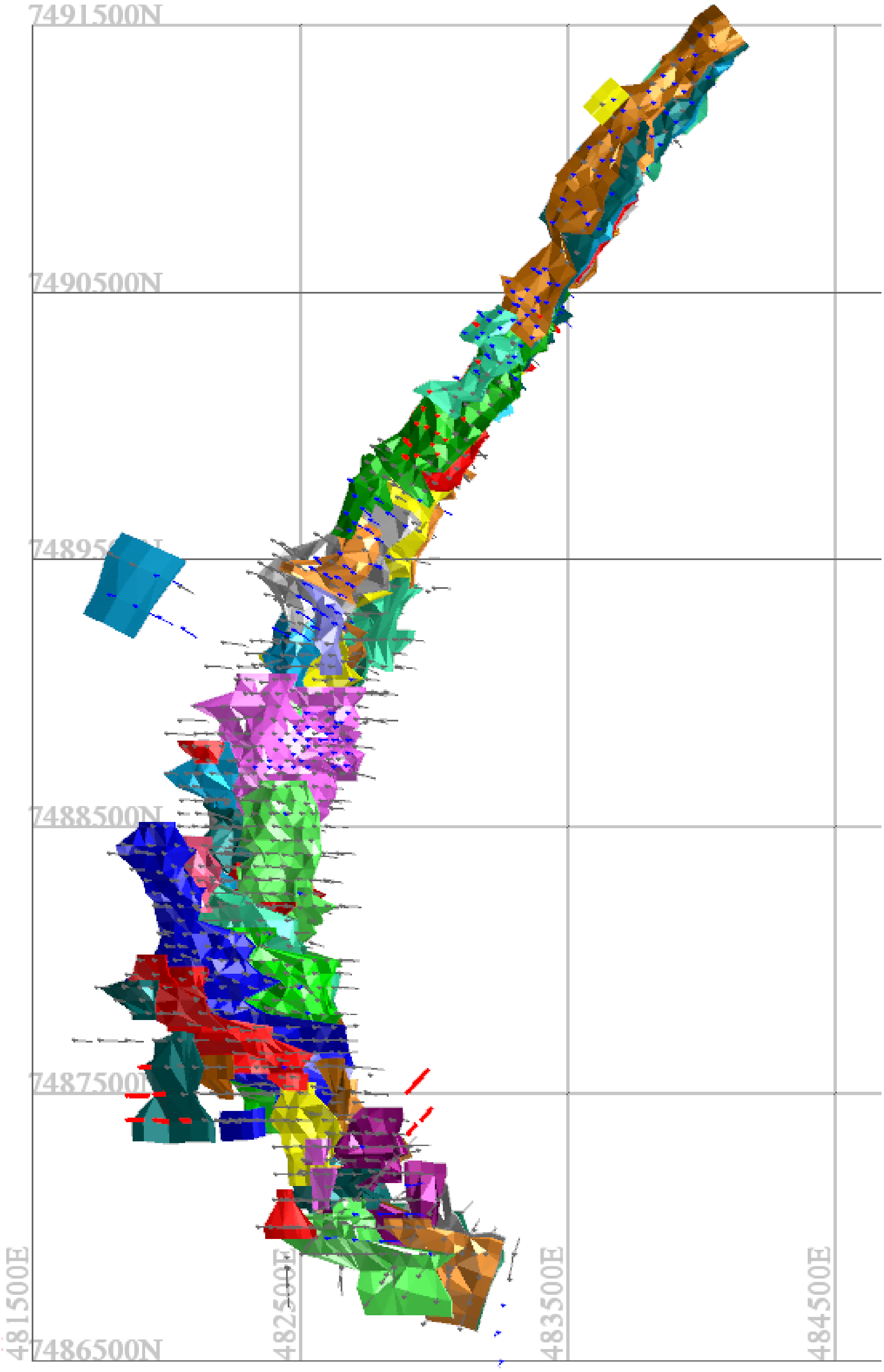
The mineralisation constraints were generated based upon sectional interpretation and three dimensional analyses of the available drilling data. The mineralised zones were modelled as 68 distinct zones (comprising 110 validated 3D shapes ranging from 3m to 135m thick – averaging 20m thick) with strike trends to the south-east, north and north-east 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 150m to 1,400m.

<u>Weathering Profile</u>

The pedolith mainly consists of <1m of transported sands. In places minor calcrete or gypcrete is encountered within the transported sand and where present it 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).

Figure 14-2
Etango Uranium Project
Modelled Mineralised Zones



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 eU3O8 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 eU3O8
    - 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 Tables 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
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
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 a relatively low coefficient of variations (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:

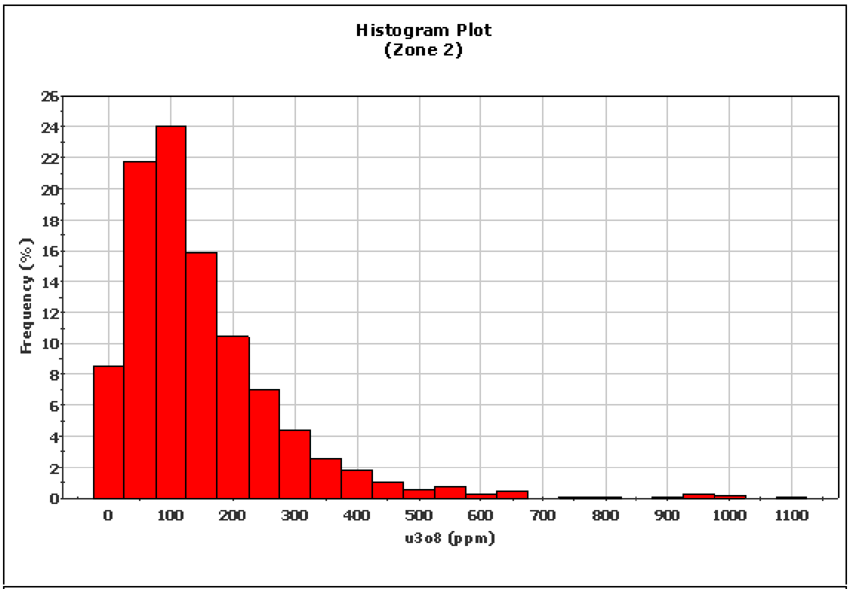
- Detailed review of histogram and probability plots, with significant breaks in populations used to interpret possible outliers;
- Detailed review of spatial distribution plots; and
- Ranking of the composite data and the investigation of the influence of individual composites on the mean and standard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

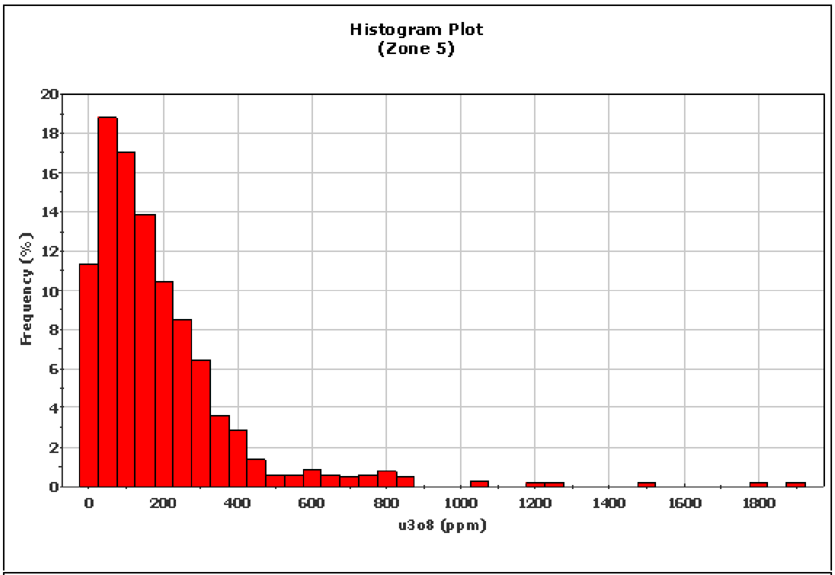
(Zone 2)

	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 drillholes 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

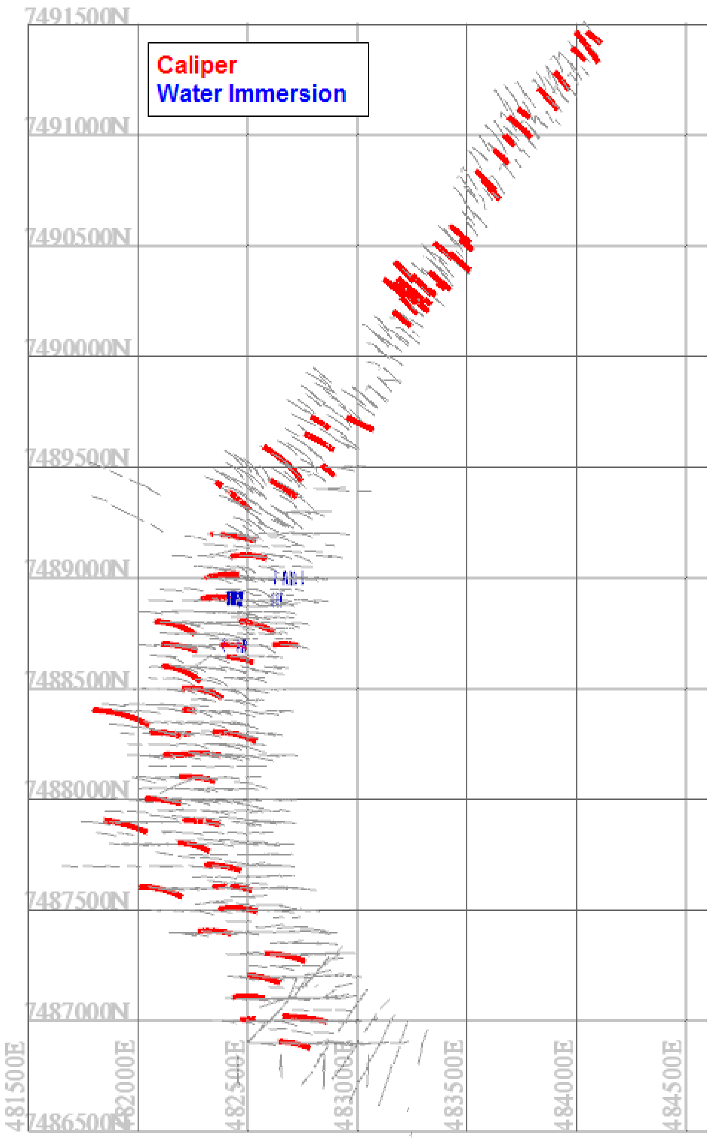


Table 14-3 Summary Statistics for Bulk Density Data

(Calliper and Water Immersion) (t/m^3)

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 meta-sedimentary units. For the mineralised zones, the bulk density measurements averaged $2.64t/m^3$. Based upon the water immersion and calliper readings, the Chuos, Khan and Etusis units had average bulk density values of $2.71t/m^3$, $2.86t/m^3$ and $2.81t/m^3$ 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

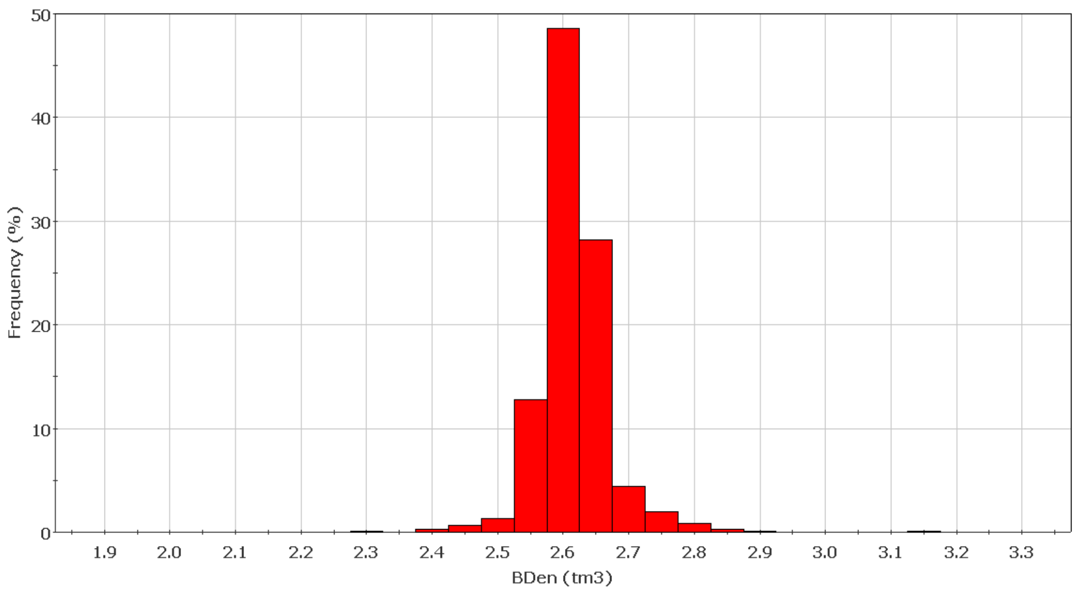
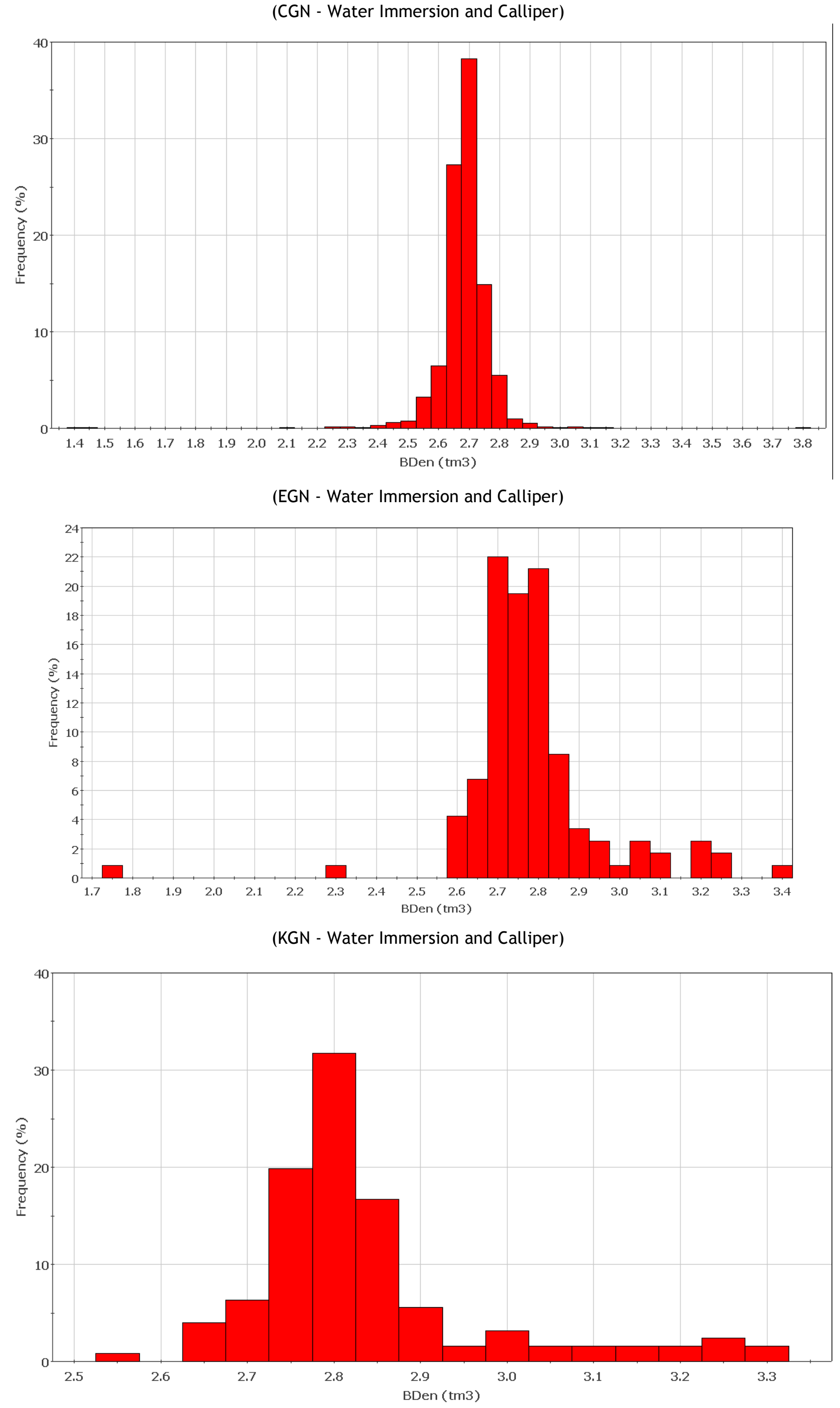


Figure 14-6 Histogram Plot of Bulk Density Readings from the Meta-Sediments



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 the U_3O_8 3m composites for 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 used for the October 2010 resource update was calculated based upon key domains, being 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		
	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 30 to 60m in the major axis. This has typically resulted in most of the zones having 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.

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

Figure 14-7 Zone 2 Variogram Plot

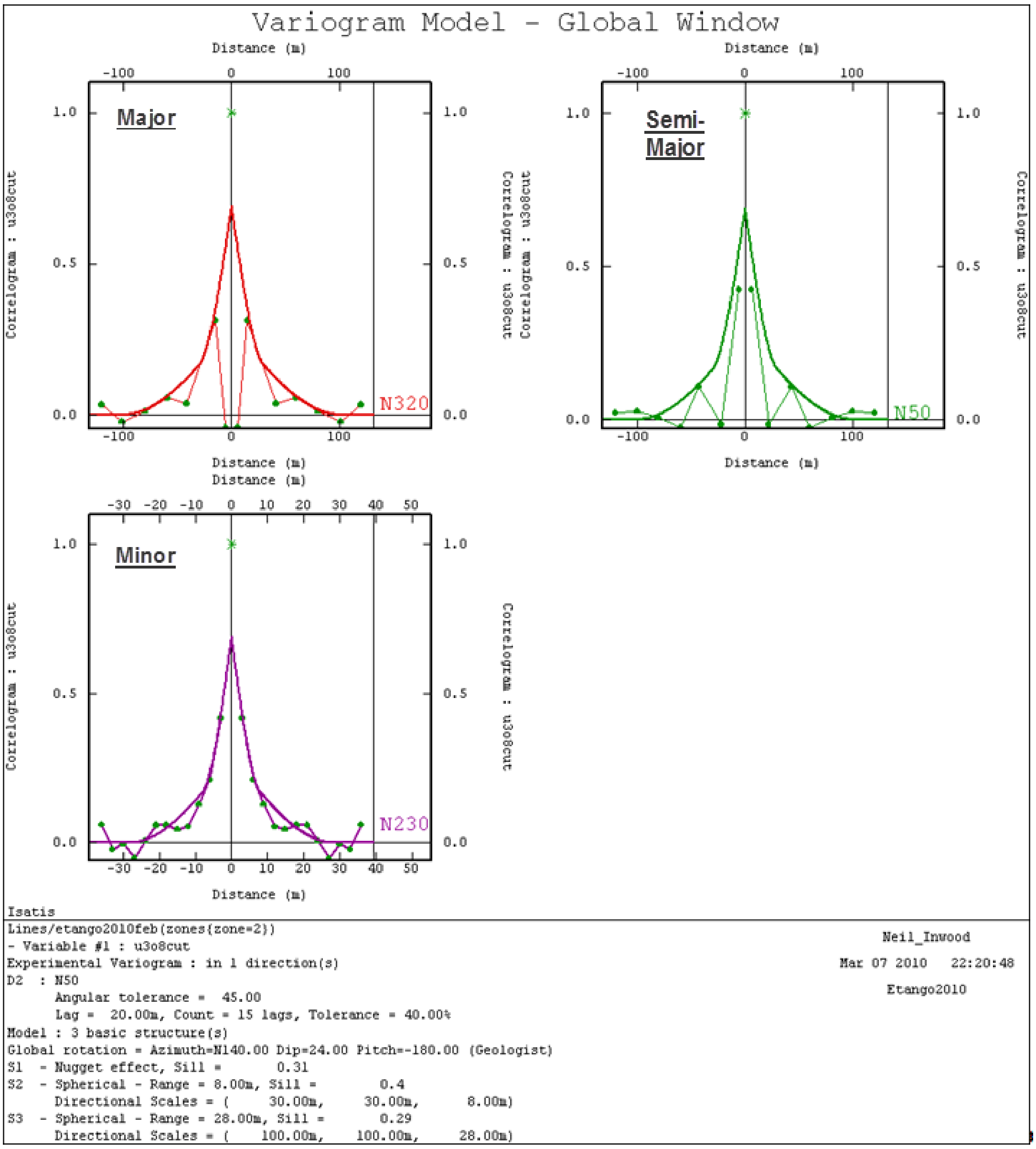
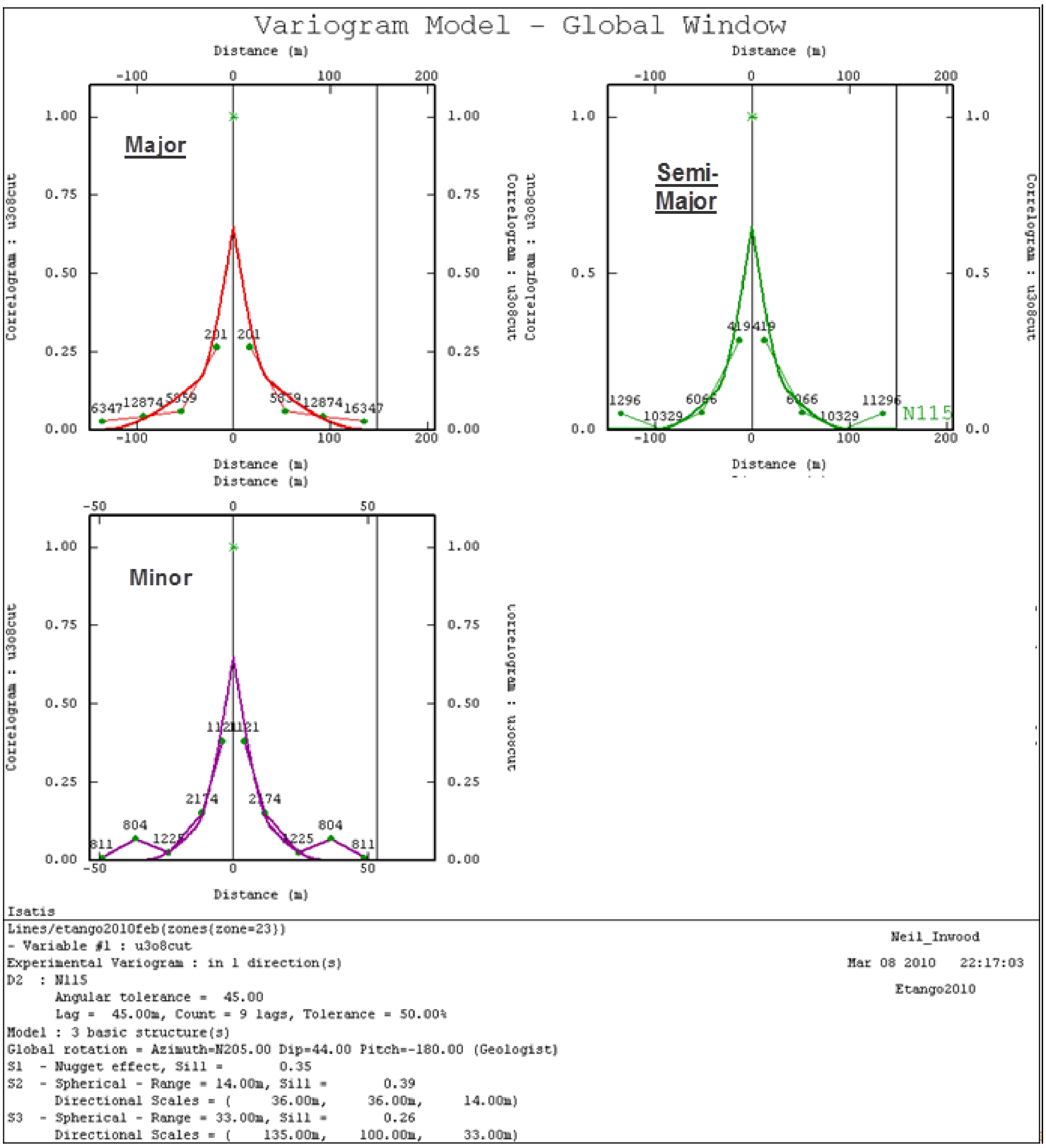


Figure 14-8 Zone 23 Variogram Plot



14.1.7 Block Model Construction

A block model was created using Surpac mining software with a parent cell size of 25m (Easting) by 25m (Northing) by 10m (RL) which was sub-blocked to 6.25m (Easting) by 6.25m (Northing) by 1.25m (RL). No rotation was applied to the block model. The block model parameters are summarised below 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	6.25

14.1.8 Grade Estimation

<u>OK Estimate</u>

Grade was estimated into the block models using Ordinary Block Kriging (OK) for U_3O_8 using Surpac mining software. Sample neighbourhood testing was conducted, also using Surpac to determine an appropriate search strategy for the OK estimation. The 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, drillhole data clustering was not a significant problem, and similar sample selection criteria were used for all mineralised zones. The sample search was orientated the same as for the variogram and search ellipse orientations above in Table 14-5. The resulting staged sample search strategy is summarised in Table 14-7, below.

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**	**Max / 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 5 points in the x-dimension, 5 points in the y-direction and 5 points in the z-dimension was used for block estimates.

<u>Validation</u>

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 17-8).

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

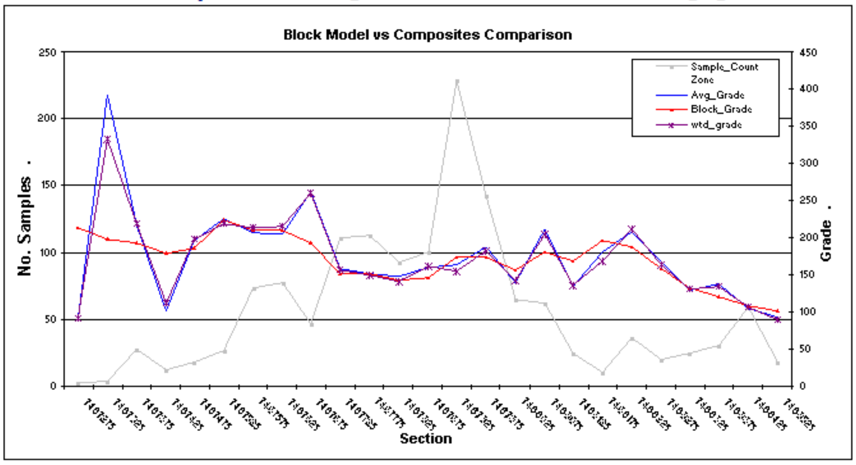
The bulk density values used for the resource model were based upon the data analysed in Section 14.1.5. A value of 2.64t/m³ was used for all material within the modelled alaskite bodies. The same value was coded into all modelled mineralised zones. Bulk densities of 2.70t/m³, 2.86t/m³ and 2.80t/m³ were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

Figure 14-9 Validation Plot Examples

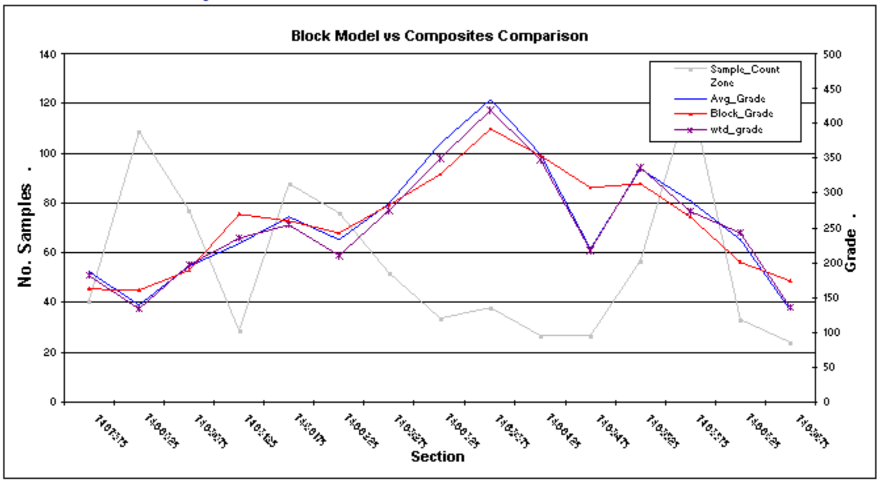
Spacing (m): 50 Orientation: y

Zone *2* *Composite Variable* u3o8_cut *Block Model Variable* u3o8_cut_ok



Spacing (m): 50 Orientation: y

Zone *14* *Composite Variable* u3o8_cut *Block Model Variable* u3o8_cut_ok



Spacing (m): 50 Orientation: y

Zone *23* *Composite Variable* u3o8_cut *Block Model Variable* u3o8_cut_ok

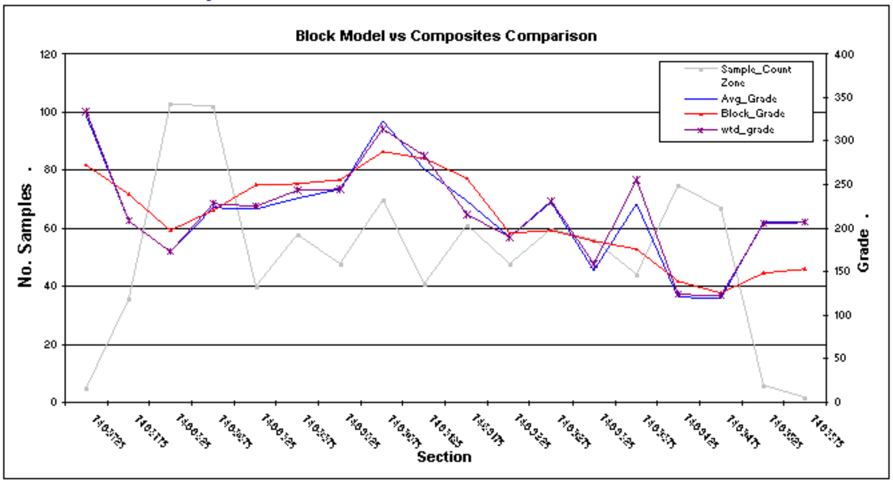


Table 14-8 OK Block Estimates Versus 3m Composite Data Comparison

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%
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%

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 ("CNI43") 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 CNI43 categorisation guidelines.

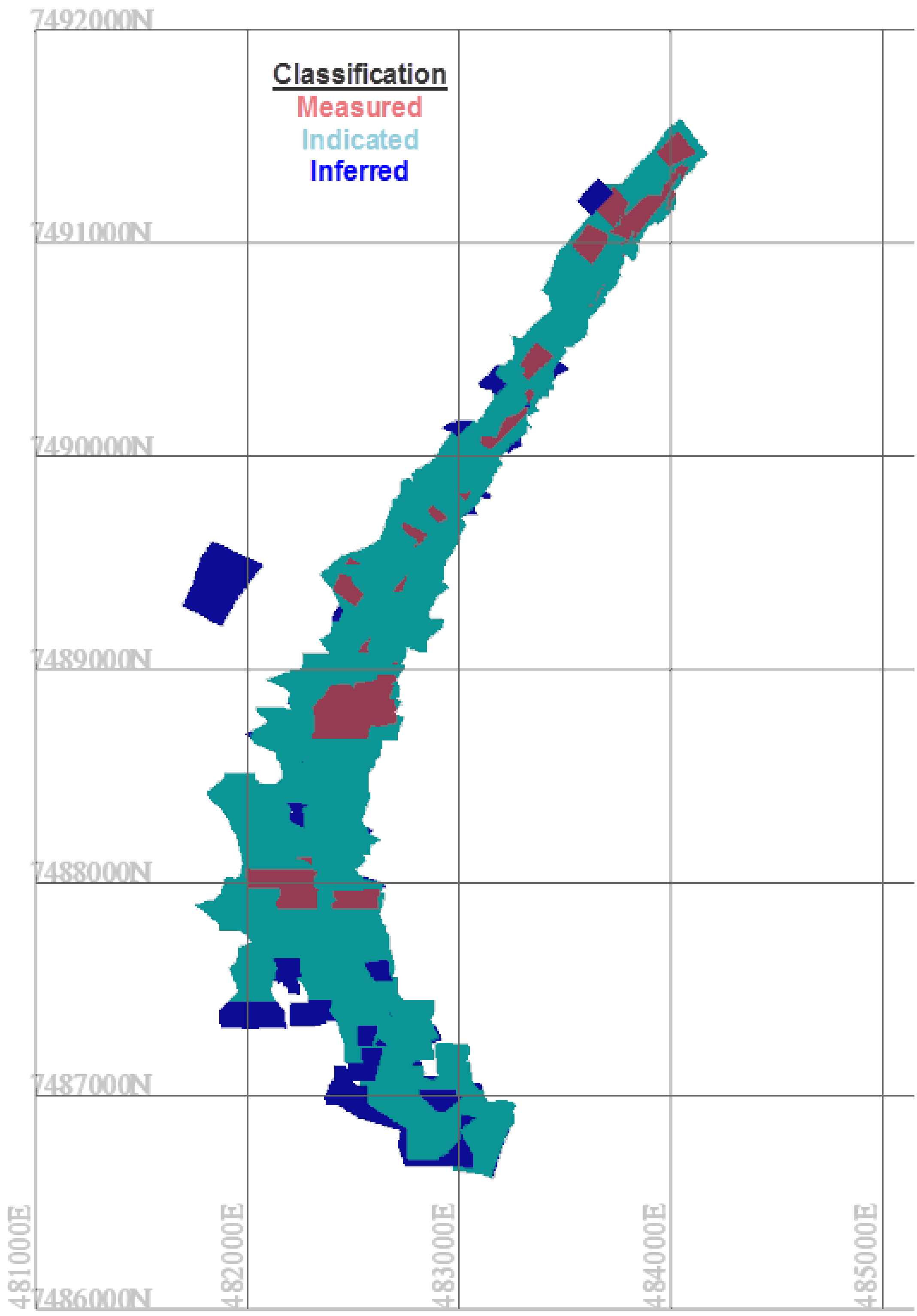
Criteria for Resource Categorisation

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.

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 drillhole collars surveyed by GPS surveyed and most drillholes 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 drillhole 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



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 by 50m drillhole 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 by 50m to 50m by 100m drillhole coverage.

Inferred Resource

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

14.1.10 Etango Grade Tonnage Reporting

The reported OK resources for the Etango Project reported above various cut-offs is 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. The availability of suitable water and power supplies may be key factors in any future mining studies.

Table 14-10
Etango Deposit, Etango Project, Namibia
October 2010 Resource Estimate
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

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 lbs to kg = x 2.20462

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 increased 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 U3O8).
- 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, 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 extents.

Apart from the OK model being reported here, recent MIK testwork has indicated that the UC derived SMU model is likely to be producing optimistic grade-tonnage results due, in part, to the tightly constrained OK domains used as a basis for the estimate. It is considered that the UC based SMU model is not currently reflecting dilution issues adequately. It is recommended that the MIK methodology be used for future recoverable resource estimates. Varied SMU parameters are currently being discussed between Bannerman and Coffey Mining.

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 uranium deposits in Namibia. The Ondjamba and Hyena deposits are located within Bannerman's Etango Project Area (within EPL 3345) which is 31km east of the major town of Swakopmund and 47km northeast of the port town of Walvis Bay. The Ondjamba deposit is located approximately 1km along strike to the south east of the Etango deposit. The Hyena deposit is located approximately 1km to the south of the Etango deposit, Figure 9-1.

Neil Inwood from Coffey Mining visited the Etango Project Area and surrounding areas on several occasions between 2007 and 2011.

The resource estimation study included a review of the available drillhole 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 Deposit Geology

At Ondjamba, uranium occurrences are located along the southern flank of the Palmenhorst Dome. The Palmenhorst Dome consists of pre-Damara basement, with a core of reddish leucocratic gneiss (quartz, microcline and accessory plagioclase biotite) that is commonly referred to as the ‘red granite gneiss’. The central gneiss is surrounded by migmatites and other basement rock types.

Uranium mineralisation at Ondjamba and Hyena is mainly located in the post-F3 alaskite granites. Minor uranium mineralisation is also found in the metasedimentary sequences close to the alaskite contacts. The major mineralised alaskite bodies are associated with the lower part of the Khan Formation and occur within 400m of the contact between the Etusis and Khan Formations.

The alaskites consist mainly of quartz and feldspar with minor, but variable, accessory minerals. Accessory minerals include ilmenite, biotite, apatite, topaz, garnet, tourmaline, uraninite, betafite, zircon, and monazite. Quartz varies in colour from colourless through smoky to almost black (indicating the presence of higher grade uranium mineralisation).

The dominant primary uranium mineral is uraninite (UO_2) but minor betafite $(Ca,U)_2(Ti,Nb,Ta)_2O_6(OH)$ is also present. The primary uranium mineralisation occurs as disseminations within rock fractures, at crystal interfaces, and as inclusion within other minerals. Secondary uranium minerals such as autunite $Ca(UO_2)_2(PO_4)_2$•10-12H2O and uranophane $Ca(UO_2)2(SiO_3OH)_2$•$5H_2O$ occur as replacement of the primary minerals or as coatings along fractures. The uraninite is commonly associated with chloritised biotite in the alaskites within the lower Khan Formation and with ilmenite and magnetite within foliated alaskites.

14.2.2 Resource Database

Ondjamba

The drillhole database in the vicinity of the estimation consists of 125 RC drill holes totalling 22,231m.

The drillholes were drilled typically at 60° to the north (UTM grid) with a drill spacing ranging from 100m by 100m to 200m by 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 drillhole database in the vicinity of the estimation consists of 148 RC and 4 diamond drill holes totalling 15,262m. Of those drillholes, 47 RC and 3 diamond drill holes totalling 9,061m were directly used for the deposit model.

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

14.2.3 QAQC, Density & Sampling

The Bannerman QAQC data was reviewed; standards, blanks and field duplicates, and showed acceptable levels of precision and accuracy, no laboratory derived QAQC information was supplied.

Only a limited number of bulk density determinations, from the three diamond holes at Hyena were supplied. A density value of 2.64t/m^3 was used for the mineralised zones after comparison with the nearby Etango deposit.

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

14.2.4 Geological Modelling

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

The mineralisation constraints were generated based upon sectional interpretation and three dimensional 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 drillhole intersections on two consecutive sections and for which a strong geological continuity could not be established, were typically not estimated.

<u>Ondjamba</u>

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.

<u>Hyena</u>

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 W-E trend. Three domains exhibit a southerly dip from -30° to -40° to the south, with domain 3 exhibiting a near vertical W-E trend (Figure 14-14). Individual zones were modelled from 150m to 1,750m long. Figure 14-14 shows a typical sectional interpretation.

14.2.5 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.

Three dimensional block models were constructed for the purposes of grade estimation for each deposit. A parent block size of 25m N by 25m E by 10m RL 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 3.125m N by 3.125m E by 1.25m RL in order to achieve appropriate volume definition of the mineralisation.

Ordinary Kriging ('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; 120m x 80m x 40m for Hyena and 240m x 160m x 80m 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.

Figure 14-11 Ondjamba Mineralised Zones and Drilling

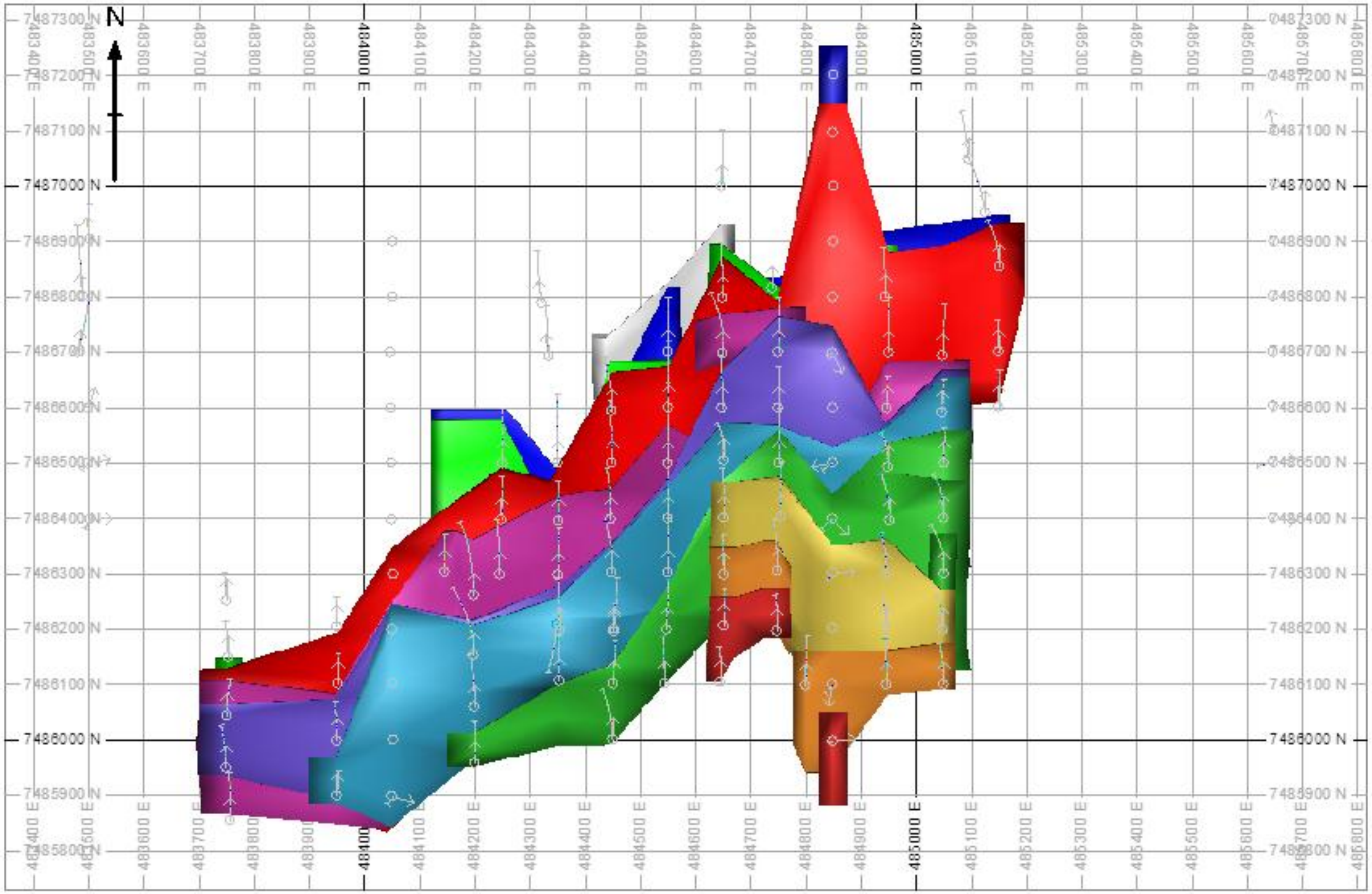


Figure 14-12 Ondjamba South-North Sectional Interpretation (484,850mE)

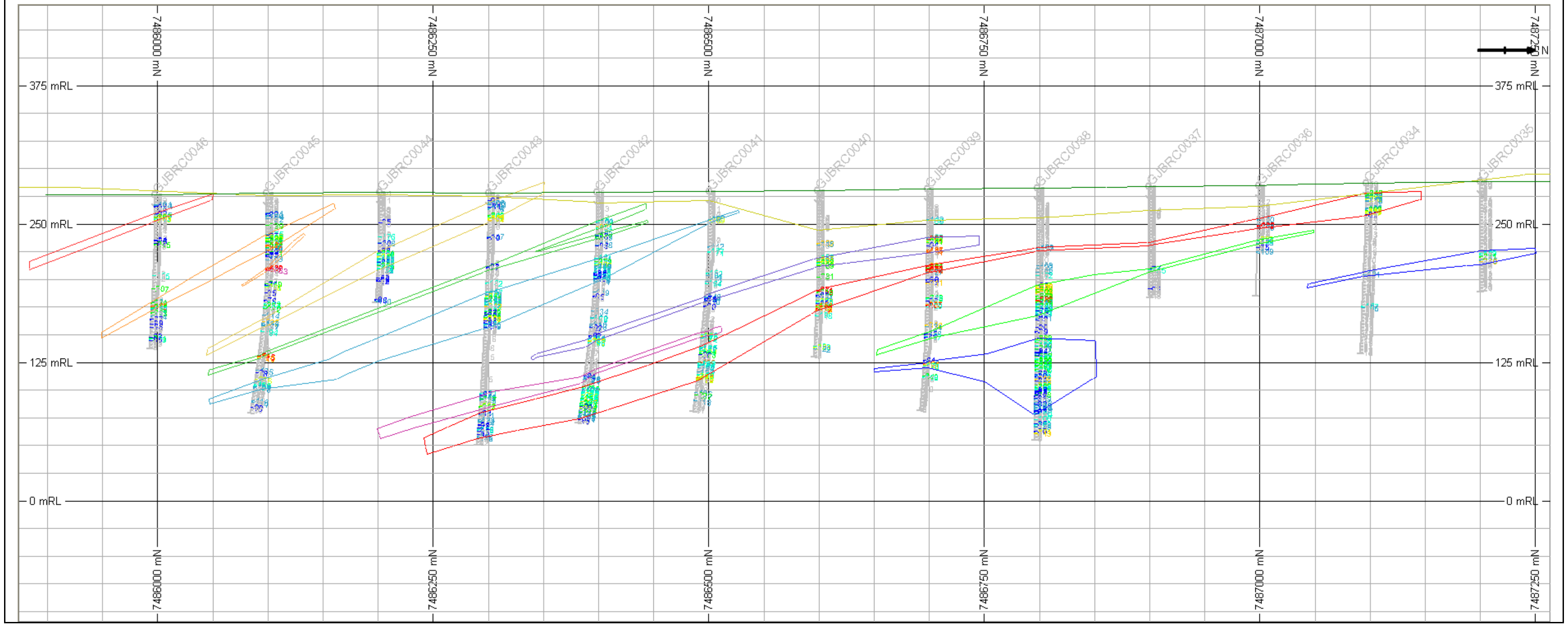


Figure 14-13 Hyena Mineralised Zones and Drilling

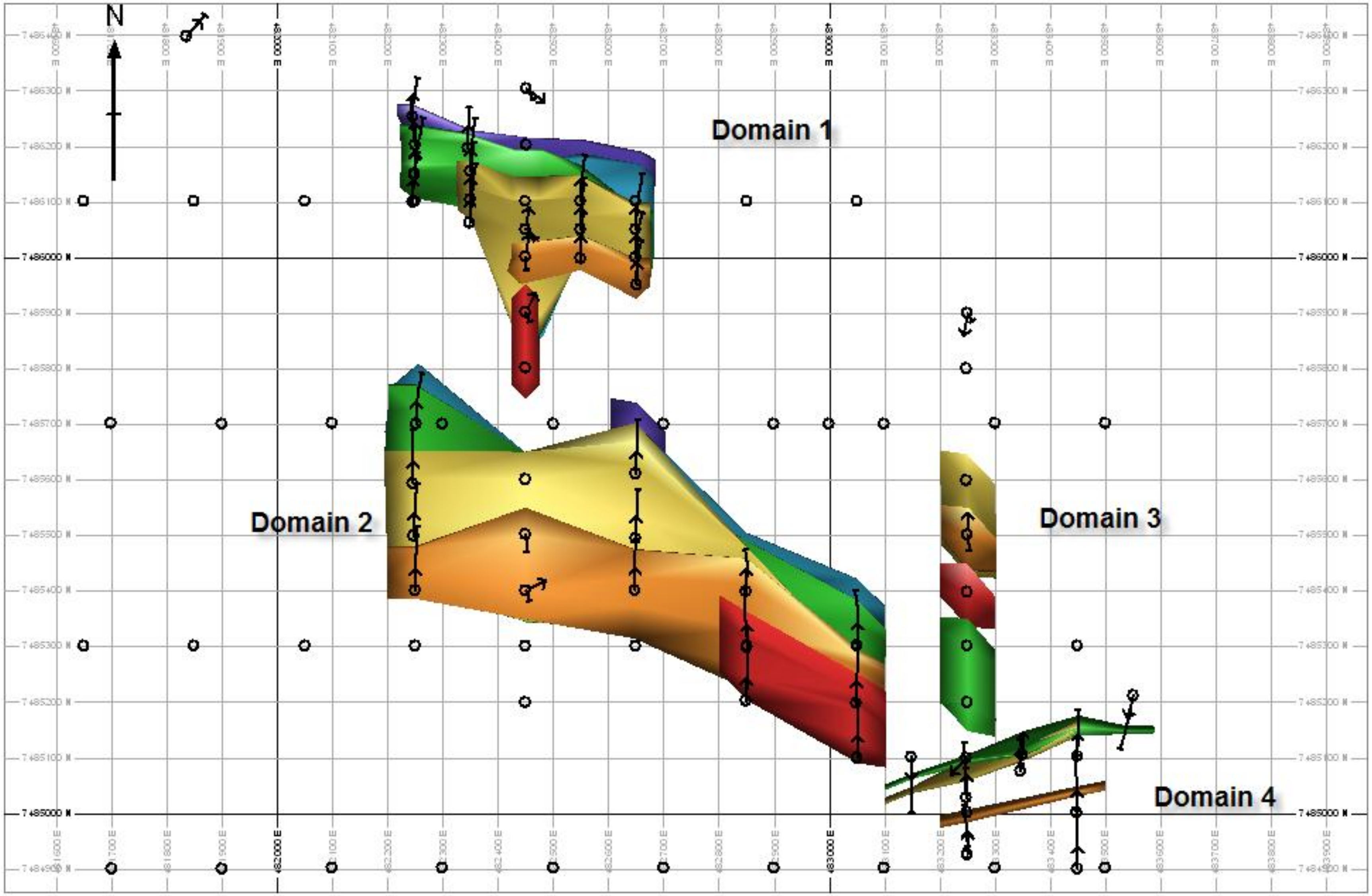
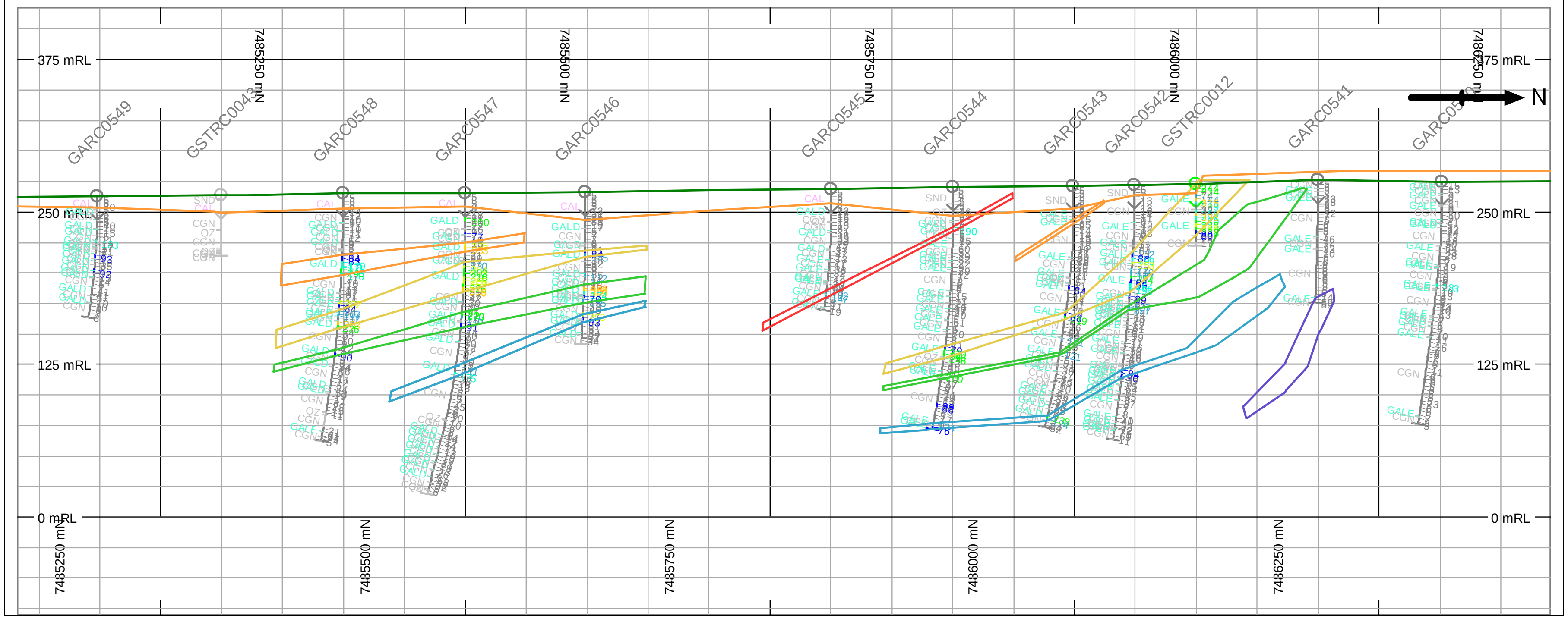


Figure 14-14 Hyena South-North Sectional Interpretation (482,450mE)



14.2.6 Ondjamba and Hyena Resources

Categorisation of the grade estimate was undertaken on the basis of the criteria laid out in the Instrument. The Resource was classified as Inferred using the criteria determined during the validation of the grade estimates, with detailed consideration of the Instrument categorisation guidelines.

Blocks were classified as Inferred considering issues such as geological and grade continuity and within a nominal 100m by 100m drillhole spacing. Blocks not classified as Inferred were left as Unclassified. Two zones at Ondjamba and five zones at Hyena were not classified where drillhole 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

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

Lower Cut	Tonnes Above Cut-off (Mt)	U_3O_8 (ppm)	Contained U_3O_8 (M lb)
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.

Table 14-12

Hyena Deposit, Etango Project, Namibia - October 2010 Resource Estimate

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

Lower Cut	Tonnes Above Cut-off (Mt)	U_3O_8 (ppm)	Contained U_3O_8 (M lb)
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.

14.3 Combined Mineral Resources

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

Etango Project Mineral Resource Estimate October 2010

Table 14-13

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**

Etango Project Mineral Resource Estimate October 2010

Table 14-14

Total estimate reported at a range of cut-off grades

	Measured Resources				Indicated Resources				Inferred Resources			
Cut-off Grade (ppm U_3O_8)	**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)	(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

A formal Mineral Reserve estimate has not yet been published for the Etango Project.

16 MINING METHODS

16.1 PFS Update (December 2010) Overview

As part of the PFS Update completed in December 2010, Bannerman and its technical consultants modelled the Etango Project as a conventional hard rock open pit operation, with drilling, blasting, loading and truck hauling. The ultimate pit dimensions for the Etango deposit pit were approximately 6km long by 1km wide, with a maximum depth of approximately 400 metres below surface. Owner mining was assumed for mine planning and costing purposes.

The estimated processed tonnage in the PFS Update, drawn only from Measured and Indicated mineral resources, totalled 292Mt at an average grade of 195ppm U_3O_8, representing approximately 87% of the total Measured and Indicated resource of the Etango deposit. The newly identified satellite deposits, Ondjamba and Hyena, were not included in the analysis and offer the potential for mining flexibility and mine life extensions.

16.2 Mining Overview

The PFS Update identified that a conventional owner-operated open pit mining operation could be economically viable. The mining schedule involved mining of the open pit over a 15 year period providing a run-of-mine feed of 15 million tonnes per annum (**Mtpa**) whilst building a low grade stockpile over the first 15 years to be treated over the following 5 years, for a total mine life of 20 years. The maximum material movement was approximately 110Mtpa between Years 2 and 8.

The PFS Update mining study covered the following scope of work:

- Geotechnical review;
- Mine costing;
- Pit optimisation;
- Mine design; and
- Mine production scheduling.

The mining study was based on the October 2010 Resource (as reported in Section 14) and versions were run both with and without Inferred Resources.

16.3 Geotechnical and Hydrogeological Review

The geotechnical stability modelling work was based on nine orientated drill holes located within the Anomaly A area of the Etango deposit, with the assessments of the core including rock mass quality, rock discontinuity orientations, and discontinuity characteristics. In general, results of the analysis indicate that intact rock strength and rock mass quality will provide strength well above of the stress levels that the pit slope walls will generate. The recommended inter-ramp pit wall slopes range from 42^0 (weathered material) to 55^0 (granite), assuming drained slopes.

Whilst limited hydrogeological work has been undertaken, Coffey Mining believes that, based on existing information, groundwater is not expected to be a problem.

16.4 Mining Method and Equipment Selection

The PFS Update was based on owner mining of a long life mine with large annual material movement.

The Etango deposit varies in geometry significantly, with zones of bulk waste, bulk ore (generally to the east, south and central areas of the deposit), and zones of narrow ore (generally to the north and west of the deposit). The potential for excessive dilution in the narrow areas requires allowance for two different mine equipment configurations, one for selective and one for bulk (non-selective) mining.

Whilst for both bulk and selective mining areas drilling and blasting would be performed on benches between 5m and 10m high, the selective areas would be mined with 250t to 300t sized hydraulic excavators and bulk areas would be mined with 450t to 550t sized hydraulic excavators. The loading equipment selected is capable of loading 220t capacity off highway dump trucks, with this size of unit being suitable for meeting the 110Mtpa mining rates envisaged in the mining schedule. For all areas, standard open-cut drilling and auxiliary equipment will be required.

16.5 Optimisation and Design

The resource model was blocked to the regular cell size of 25mE (width) x 25mN (along strike) x 10mRL (vertical) for ease of data manipulation and to better reflect the pit slope and ramp requirements.

Staged development of the deposit is driven by the desire to maximise the grade of the initial plant feed, minimise waste pre-stripping and the requirement for consistent total material movement. The pit was modelled as mined in 9 stages. A large low grade stockpile is developed during the mine life, with higher grade ore (above 150ppm) being fed to the crusher preferentially.

The material breakdown for the final pit design is shown below (Table 16-1).

Table 16-1
Material Breakdown for Final Pit Design
PFS Update (December 2010)

	Total Mt	Waste Mt	Strip Ratio w : o	Mill Feed Mt	U3O8 ppm
Total	1,361	1,058	3.5*	303	**192**

* Increases to 3.7 with the exclusion of in-pit Inferred mineral resources.

Waste dumps have been designed to accommodate the life of mine waste production, taking into consideration environmental constraints. An opportunity for dumping up to 350Mt of waste (approximately 25% of the total waste) back into the open pit was identified, but this was not included in the PFS Update and is an opportunity being investigated as part of the DFS.

16.6 Mining Schedule

The mining schedule was developed on a quarterly basis for the pre-production period and the first two production years, and annually thereafter. The schedule is based on bench by bench mining of the quantities calculated within the individual pit stages.

The following table (Table 16-2) shows the key mining physicals over the duration of the mine life.

Table 16-2

Key Mining Physicals

PFS Update (December 2010)

Period		Total	Waste	Strip Ratio	Mill Feed Mined		Mill Feed On Stockpile (Cum.)		Mill Feed Processed	
					Tonnes	Grade	Tonnes	Grade	Tonnes	Grade
		(Mt)	(Mt)	(w:o)	(Mt)	(ppm)	(Mt)	(ppm)	(Mt)	(ppm)
Pre-Production	Qtr -2	8.6	7.5	6.7	1.1	150.3	1.1	150.3	0.0	0.0
	Qtr -1	8.7	7.1	4.3	1.7	155.1	2.8	153.2	0.0	0.0
	Total	17.4	14.6	5.2	2.8	153.2			0.0	0.0
Year 1	Qtr 1	22.7	19.8	6.8	2.9	170.1	2.9	116.8	2.8	207.7
	Qtr 2	25.5	20.4	4.1	5.0	183.8	4.5	118.2	3.4	214.6
	Qtr 3	20.0	15.0	3.0	5.0	195.5	5.9	119.4	3.6	224.0
	Qtr 4	22.4	17.5	3.6	4.9	206.0	7.1	120.1	3.7	231.9
	Total	90.5	72.6	4.1	17.9	191.0			13.5	220.5
Year 2	Qtr 1	22.0	17.0	3.4	5.0	207.0	8.4	120.4	3.8	234.9
	Qtr 2	22.0	16.9	3.4	5.0	205.5	9.6	120.7	3.8	233.2
	Qtr 3	22.5	17.4	3.4	5.1	205.1	11.0	121.1	3.7	234.5
	Qtr 4	21.9	16.7	3.2	5.2	204.9	12.5	121.6	3.7	236.5
	Total	88.4	68.1	3.3	20.3	205.6			15.0	234.8
Year 3		104.0	80.9	3.5	23.1	193.6	20.6	121.3	15.0	232.9
Year 4		110.3	89.0	4.2	21.4	193.2	26.9	121.4	15.0	223.5
Year 5		110.5	90.4	4.5	20.0	205.3	31.9	121.5	15.0	233.3
Year 6		110.8	88.5	4.0	22.3	201.4	39.3	120.3	15.0	243.4
Year 7		111.0	88.9	4.0	22.2	195.6	46.4	119.9	15.0	232.9
Year 8		109.6	87.7	4.0	21.9	189.6	53.3	120.1	15.0	221.0
Year 9		101.1	78.1	3.4	23.0	189.5	61.3	120.2	15.0	225.6
Year 10		96.3	76.3	3.8	20.0	207.1	66.3	120.7	15.0	234.0
Year 11		102.4	77.7	3.1	24.7	176.6	76.0	121.1	15.0	210.8
Year 12		81.6	56.6	2.3	25.0	179.8	85.9	121.1	15.0	218.8
Year 13		80.0	57.1	2.5	22.9	182.2	93.8	121.0	15.0	214.3
Year 14		39.8	27.9	2.3	12.0	184.5	90.7	121.1	15.0	171.4
Year 15		7.4	4.2	1.3	3.3	200.9	79.0	121.1	15.0	138.4
Year 16		0.0	0.0	0.0	0.0	0.0	64.0	121.1	15.0	121.1
Year 17		0.0	0.0	0.0	0.0	0.0	48.9	121.1	15.0	121.1
Year 18		0.0	0.0	0.0	0.0	0.0	34.0	121.1	15.0	121.1
Year 19		0.0	0.0	0.0	0.0	0.0	19.0	121.1	15.0	121.1
Year 20		0.0	0.0	0.0	0.0	0.0	3.9	121.1	15.0	121.1
Year 21		0.0	0.0	0.0	0.0	0.0	0.0	0.0	3.9	121.1
Total		**1,361.2**	**1,058.5**	**3.5**	**302.7**	**192.0**			**302.7**	**192.0**

The schedule involves mining of the open pit over a 15 year period, in order to meet a crusher feed target of 15Mtpa, producing approximately 3,000t of U_3O_8 per annum for the first 15 years of the mine life and 1,650t of U_3O_8 per annum for a further 5 years, during which the low grade stockpile is treated. The average waste to ore strip ratio is 3.7:1 (or 3.5:1 inclusive of in-pit inferred resources, representing approximately 3.6% of mined mineralisation), which resulted in a total material movement of 90Mtpa for Years 1 and 2 and approximately 110Mtpa for the next 6 years, reducing to between 80-100Mtpa for the following five years of operation.

16.7 Mining Operating and Capital Costs

The operating mining costs for owner mining, excluding mine equipment ownership costs, were estimated in PFS Update at approximately US$1.75/t of material mined (waste and ore).

An initial capital expenditure for mining equipment of approximately US$64 million was estimated for the start-up fleet, in addition to initial capitalised mining costs of US$33 million.

Sustaining capital of US$55 million between Years 2 to 6 was required to allow for an increased total material movement, and mine equipment replacement capital over the life of the mine (predominantly from years 7 to 13) was estimated at approximately US$132 million.

17 RECOVERY METHODS

17.1 PFS Update (December 2010) Overview

As part of the PFS Update completed in December 2010, and the metallurgical testwork described in Section 13, Bannerman and its technical consultants gained considerable understanding about the mineralogy of the Etango deposit, in particular:

- Over 90% of the mineralisation is contained within the alaskite host rock;
- The geological sequence excludes the high acid consuming carbonate (marble) formations prevalent in other areas of the region;
- The predominantly uraninite (UO_2) mineralisation is located at crystal interfaces and as inclusions with other minerals; and
- No clay is evident in the deposit.

Work in 2010 demonstrated the technical and economic advantages of the application of the heap leaching processing option. This option uses less sulphuric acid, water and electricity than tank leach processing.

17.2 Heap Leaching

In the PFS Update, Bannerman modelled a 15Mt per annum heap leaching process incorporating a three stage crushing circuit, with the third stage comprising high pressure grinding rolls. Crushed material is agglomerated and stockpiled onto an on-off heap leach pad using the type of stacking and reclaiming equipment currently employed at a number of large copper heap leach operations in South America. The heaped ore will then be percolated with dilute sulphuric acid to leach the uranium minerals into solution. This solution will be collected for further processing in standard solvent extraction, precipitation and calcination circuits before the U_3O_8 is packaged in drums for containerised export through the nearby Walvis Bay deep-water port.

Bannerman achieved consistent recoveries of ~90% and acid consumption of 10-15kg/tonne over a 15 day period in column tests up to 7 metres in height. To allow for scale-up factors and other losses, the PFS Update has assumed a design metallurgical recovery of 85% on a large scale heap leach pad over a 52 day on/off cycle. An increase/decrease of 1% in the assumed metallurgical recovery rate has the effect of decreasing/increasing operating costs by approximately US$0.50/lb U_3O_8.

17.3 Process Plant Description

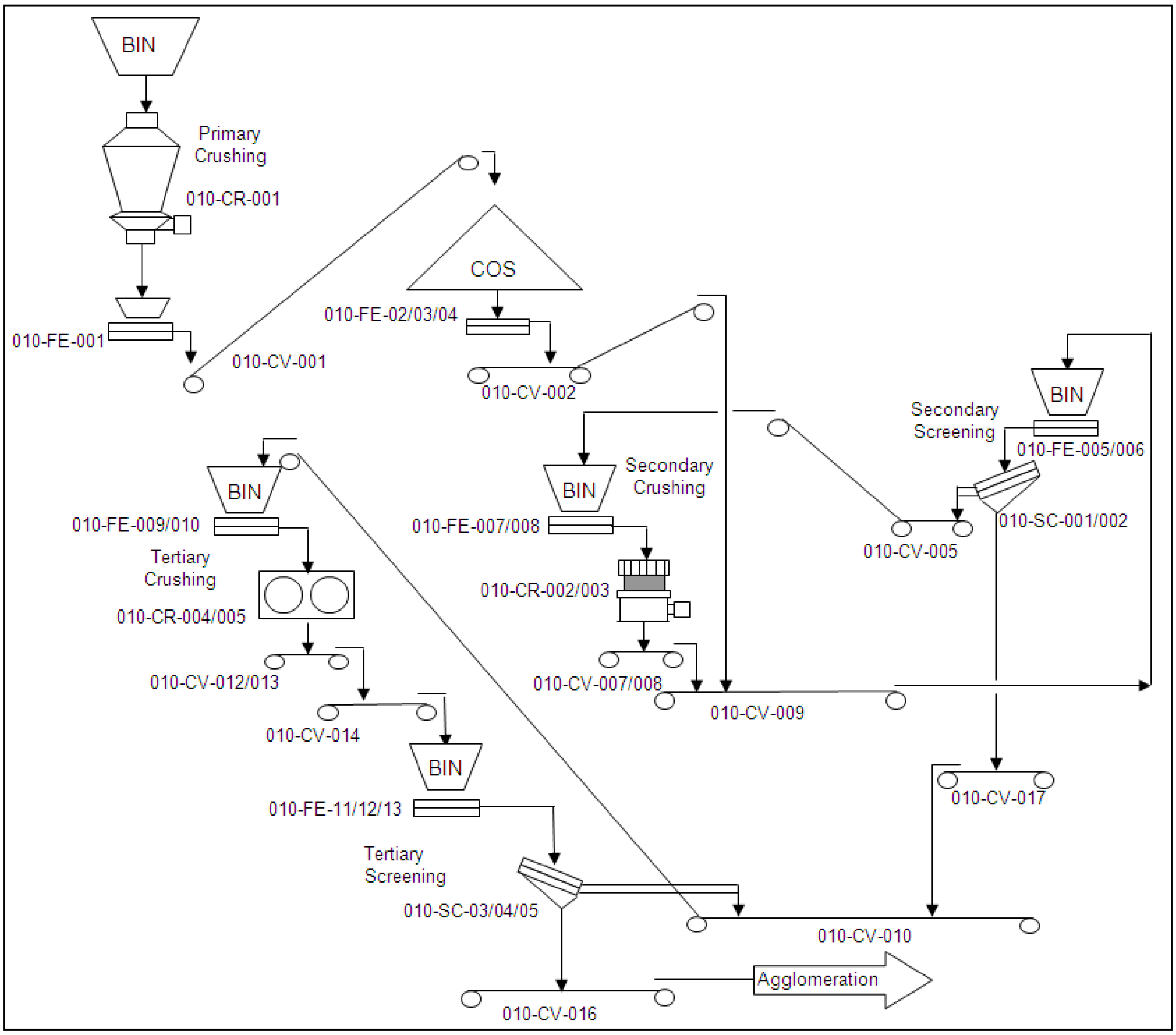
Process development activities conducted by Independent Metallurgical Operations, AMEC Minproc and Bannerman from 2007 to late 2010 evaluated a range of processing approaches including agitated leaching, heap leaching, and several beneficiation techniques. Review of the PFS in early 2010 concluded that both Heap Leach and Agitated Tank Leach options were technically viable however more detailed development of both was required to allow clear differentiation and selection of the preferred extraction process for the process plant. Consequently, the PFS Update considered both options for the extraction stage. From this work, heap leach was identified as having the potential to deliver the best economic outcomes.

17.3.1 Crushing

The crushing circuit is direct fed from 220 - 250 t haul trucks. During periods when direct tipping is not available, a Front End Loader feed is assumed. The ROM bin feeds directly into the gyratory primary crusher. The crushed ore is transferred to a coarse ore stockpile. This is followed by two secondary crushers operating in closed circuit and two HPGR tertiary crushers delivering crushed product at a target P80 product size of 4.0 mm.

The crushing circuit is summarised in Figure 17-1 below.

Figure 17-1
Heap Leach Crushing Circuit



Two agglomerating drums receive crushed ore, water, sulphuric acid and binder and agglomerated ore is then transferred to the heap leach stacking system with a design capacity of approximately 2,200t/h. Heaps are stacked to a maximum height of 7m.

Leached residue is transferred from the heap leach by conveyor to the leach residue stacking system. The leach residue pad is constructed using several layers to provide an impervious layer, so preventing leakage of liquids to the environment though the floor of the pad.

17.3.2 Leaching

Leach pads are constructed and operated in a set of 54 modules, each module being equivalent to 1 stacking day. The modules are rotated and application of leach solution through dripper systems solubilises the uranium producing the pregnant liquor solution (**PLS**), which is then pumped to the solvent extraction circuit for uranium recovery.

17.3.3 Solvent Extraction

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

17.3.4 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**), then thickened whilst barren liquor is clarified to remove suspended ADU solids.

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

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

18 PROJECT INFRASTRUCTURE

18.1 PFS Update (December 2010) Overview

The Etango Project is located approximately 41km by road from the town of Swakopmund. Power and water are proposed to be supplied from the well established national infrastructure. The PFS Update estimates incorporate provision of high-voltage power lines and reticulation systems, desalinated water supply with pumping and storage facilities, and access roads.

18.2 Power

The Namibian power utility, NamPower, has confirmed its ability to provide power to the Etango Project and has offered a 30MVA supply. NamPower is currently upgrading supply and distribution capacity in the region.

As published by NamPower in 2009, it can currently generate 393 MW of electricity, reducing to 250 MW during the dry season due to non-operation of the run-of-river Ruacana hydroelectric power station. Peak power demand in Namibia (2009) was 550 MW. The country has to import 20% of its electricity from Eskom, the South African power utility and 40% from other countries in the region. However, in recent times, domestic demand for electricity in South Africa has exceeded supply resulting in shortages in the Southern African Development Community (SADC) region. Over the next 4 years the proposed development of new uranium mines, desalination plant and the expansion plans of existing facilities is likely to increase demand by 300 MW in the Erongo region alone. Therefore, Namibia has recognised the need to increase its power generation capacity as well as reduce dependency on South Africa. Power prices are expected to rise significantly to fund this additional generation capacity and to offset increases in supply tariffs, with an expectation that by 2012 the power tariffs would need to increase by 50% to meet costs. In May 2010, NamPower requested a 35% hike in the power tariffs but was granted a rise of 18% instead by the industry's regulator.

NamPower is considering the following alternatives to increase power generation capacity:

- Combined cycle gas-fired power station (Kudu Gas) – 400 MW to 800 MW (earliest 2014).
- Coal-fired power station at Walvis Bay – 400 MW.
- Diesel peaking station at Walvis Bay – 50 MW.
- Lower Orange River Small Hydro Stations – 108 MW.
- Renewable energy such as wind and solar – < 40 MW.
- Baynes Hydro-Power Station – 360 MW to 550 MW (50-50 split between Namibia and Angola).

Some of the above projects are being advanced through the commencement of Environmental and Social Impact Assessments.

The Ruacana hydro-power station has been fitted with a fourth 80 MW turbine to increase peak demand capacity from 2010. The 400 kV Caprivi Link Interconnector has also been commissioned and will import power from Zimbabwe's Hwange Power Station.

18.3 Water

The water requirement for the Etango Project was estimated in the PFS Update as 2.6-3.2 million cubic metres (2.6-3.2GL) per annum.

The Namibian state-owned water utility, NamWater, is expanding capacity in the Erongo region to meet increasing demand from mining and other industries. Bannerman, along with other members of the Erongo Mining Water Users' Group, is examining the technical and financing aspects of the installation of a second desalination plant on the coast to the north of Swakopmund. Opportunities also exist to secure water from third party operators in the region.

NamWater is proposing to establish a Sea Water Reverse Osmosis (**SWRO**) desalination plant on the Atlantic coastline. The proposed project consists of a desalination plant (including a sea water intake structure and a brine disposal system), a 20,000 m^3 storage reservoir on site, a pipeline to the existing Omdel Swakopmund pipeline and a new 44 km long distribution line.

For the PFS Update, Bannerman assumed its proportionate share of capital cost contributions to the SWRO desalination plant and associated infrastructure (approximately US$62 million), plus an operating cost for water consumption of US$1.08/$m^3$ based on the tariff rate advised in NamWater correspondences on predicted operating costs.

18.4 Roads

Bulk consumables are proposed to be transported to the Etango site via the existing railway network to a location approximately 25km from the site, after which they will be transported to the site via existing and new access roads.

The C28 is a main road between Swakopmund, routed south of Etango to the Langer Heinrich mine and ultimately to Windhoek. It is largely unsealed, though intermittent stretches of blacktop have been added as 'overtaking zones' for safety reasons.

The project includes the construction of a spur road from the C28 to the mine site for use during the construction phase and this will be the main access route to the mine during operations. It is envisaged that this road will follow the separate, but parallel services route for the power line and water pipeline.

18.5 Rail

A rail siding is to be established on the existing rail line from Walvis Bay to Swakopmund. This siding will be equipped to handle multi-modal transport units. Road haulage will be used from the rail siding to site. A multi-modal approach has the advantage of lower capital cost compared to constructing a rail spur directly to the mine site, and lower environmental and social costs compared with utilising road haulage for the entire route to and from the Walvis Bay port.

19 MARKET STUDIES AND CONTRACTS

19.1 Product Specifications

The processed product from the Etango Project will be uranium oxide or "yellow cake", in standard drums each holding 300-420kg of U_3O_8. The drums will be packed in sea containers for shipment, subject to certain transport restrictions. Drums of uranium oxide have been exported from Namibia for over 30 years, through Walvis Bay deep water port, located approximately 67km from Etango.

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 Namibian and international regulations govern the transport of the drums.

The drums of uranium oxide are shipped to one of 3-4 established conversion facilities throughout the world, with the primary ones located in France (Areva/Comurhex), US (Honeywell/Converdyn) and Canada (Cameco/Blind River). At the conversion facility, the uranium oxide is converted into a gas uranium hexafluoride, UF_6 gas and then shipped to an enrichment facility. At the enrichment facility, the UF_6 gas is enriched through various processes to increase the naturally-occurring incidence of U-235 atoms in the material from 0.7% to 3.5-5.0% such that the enriched material can then be prepared by a fuel rod/bundle manufacturer for a nuclear power utility to load into a reactor.

Title to the uranium oxide typically passes from the miner to the buyer upon delivery to the conversion facility. The miner receives a credit to its metal account at the conversion facility for the vast 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 miner's relevant sale 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 Bannerman 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 deleterious mineral content. There is typically a free storage period with additional charges for longer term storage.

The specifications for the conversion facilities are similar but not identical because the process at each of the primary conversion plants is different, and because the governing regulations are different between countries. In all cases, there is a maximum allowable percentage for certain elements, particularly heavy metals with a financial penalty for higher contents and then absolute maxima above which the yellow cake shipment is rejected.

19.2 Shipping

Regular container services operate from Walvis Bay in Namibia to Europe, Asia and the US. Such services have operated for over 30 years for the Rössing uranium operation majority-owned and operated by Rio Tinto plc, and in more recent years have also transported uranium oxide from the Langer-Heinrich operation owned by a subsidiary of Paladin Energy Limited.

Specialist shipping agents, for yellow cake and other nuclear materials are located in Europe and the USA. Bannerman expects to deliver yellow cake in drums to the container terminal at Walvis Bay, and to utilise the services of a local Namibian shipping agent for this purpose.

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. Bannerman has adopted an allowance of US$1.40/lb to cover the costs of on-land and sea transport, conversion facility weighing, sampling and assaying, conversion facility surcharges (if any), and general sales and marketing expenditure, whether in-house and/or external.

19.3 Sales and Marketing Costs

The key markets for the uranium to be produced from the Etango Project are expected to be Asia, Europe and the USA. Discussions have been held with various end-user and other potential buyers. Contracts are likely to be entered into progressively once the DFS for the Etango Project is finalised and a development decision is made.

Bannerman expects to form an in-house sales and marketing function to administer the Etango Project's sales arrangements and revenues. Bannerman has also been approached by specialist uranium marketing groups and other uranium producers seeking to market the Etango Project's uranium production.

19.4 Uranium Market and Prices

Currently there is no formal single established market for uranium or uranium oxides. A number of brokers specialise in offering prices and volumes for buyers and sellers. However, most volumes are transacted directly between producers and end-user utilities under long term sales contracts. These contracts range from 3-10 years in term and prices can be fixed or calculated by reference to prevailing spot and term contract prices (by reference to two established industry price reporting organisations) at time of delivery. Floor and ceiling prices may also be specified.

For financial modelling purposes, all production from the Etango Project is assumed sold at a long term contract price of US$75/lb U_3O_8, with sensitivities run at various alternative prices.

The current long term contract price for U_3O_8 is around US$65/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.

19.5 Contracts

The engineering, procurement, construction and management (**EPCM**) of the Etango Project is expected to be executed by an experienced and capable engineering contractor. Under the EPCM contractor, various sub-contractors will be used to execute major work areas such as the civil, concrete, piping and electrical aspects of the site.

It is also envisaged that maintenance and repair (MARC) type contracts will be used to manage the supply and servicing of the major pieces of mobile mining equipment. Supply and service contracts are expected for major reagent supplies, in particular sulphuric acid and the various reagents for the solvent extraction process.

The above assumptions have been utilised in the estimation of capital and operating costs in Section 21. The particulars of the relevant contracts will be prepared as part of the DFS.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

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. A Mining Licence is required before mining may commence.

20.1 Environmental Studies

Bannerman completed an initial Environmental and Social Impact Assessment (**ESIA**) on the Etango Project in 2009, and applied for an Environmental Clearance. In April 2010, the Namibian Ministry of Environment and Tourism (**MET**) issued Bannerman with an Environmental Clearance based on the 2009 ESIA. The ESIA was based on open pit mining and heap leach processing focused on the Anomaly A area within the Etango deposit.

In July 2011, Bannerman received a separate Environmental Clearance from the MET for the linear infrastructure proposed to service the Etango Project. The linear infrastructure comprises an access road and telecommunications, power and water services.

An updated ESIA and accompanying Environmental and Social Management Plan (**ESMP**), incorporating the now expanded resource area and site layout refinements, is scheduled to be lodged for public comment in late 2011 or early 2012, following which it will be submitted to the Namibian Ministry of Environment and Tourism for approval.

The ESIA must determine the potential positive and negative environmental impacts of the Project. These findings will be used to finalise the ESMP for the construction and operation of the Etango mine facility. The assessment and issue of the Environmental Clearance is conducted in accordance with Namibia's Environmental Assessment Policy (1994) published by the Directorate of Environmental Affairs within the MET.

All existing baseline studies and impacts assessments are also being expanded to include the pit expansion into the Oshiveli and Onkelo deposits.

20.2 Mining Licence

Bannerman submitted its initial mining licence application for the Etango Project to the Namibian Ministry of Mines and Energy in December 2009. The application was at that time 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.

Upon receipt of an updated Environmental Clearance for development of the Etango Project (refer Section 20.1 above), Bannerman will lodge supplementary information with the Ministry of Mines and Energy in further support of the existing Etango mining licence application.

21 CAPITAL AND OPERATING COSTS

21.1 PFS Update (December 2010) Overview

Capital and operating costs for the Etango Project PFS Update are summarised in the following sections. A summary economic analysis is set out in Section 22.

21.2 Capital Costs

21.2.1 Pre-production Capital Costs

Capital expenditure estimates for the Etango Project heap leach operation include all on-site items for the processing plant, heap leach pads and stacker and reclaim equipment, storage facilities for depleted heap leach material, administration and service facilities, consumables storage, mining infrastructure and pre-production waste rock stripping activities, as well as access and site roads.

Off-site items in the capital costing include water pipelines, high-voltage power lines and related equipment.

The capital cost estimates are based on owner mining and exclude working capital and financing charges but include all mining establishment, waste pre-stripping and EPCM (engineering, procurement, construction and management) costs.

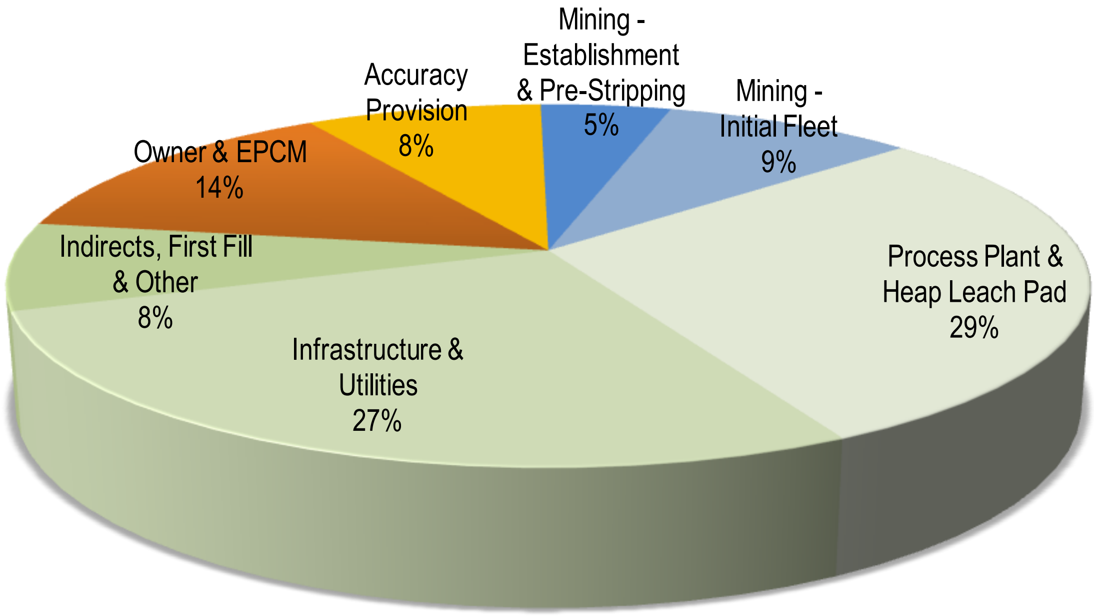
Accuracy provisions have been separately assessed for the process plant, construction directs and indirects, and the EPCM contract, equating to an average allowance of approximately 14% on these items. No contingency allowance has been included in Table 21-1.

Table 21-1
Pre-production Capital Cost Estimate
PFS Update (December 2010)

Pre-production Capital Cost Estimate	**Heap Leaching** (US$ million)
Mining (including pre-stripping, excluding initial mining fleet)	33
Processing plant and associated heap leach pad construction	203
Infrastructure and utilities	191
Indirect & other costs	55
Owner & EPCM costs	96
Accuracy provision	60
Total initial capital expenditure	**638**
Initial mining fleet	64
Total (excluding working capital and financing charges)	**702**

The breakdown of the estimated pre-production capital costs is shown graphically below:

Figure 21-1
PFS Update – Pie Chart of Estimated Pre-Production Capital Costs



21.2.2 Sustaining Capital Costs

Estimated sustaining capital over the life of the operation comprises US$272 million in mining fleet additions, US$77 million in general sustaining capital allowances, and US$33 million in rehabilitation and closure costs.

21.3 Operating Costs

Operating cost estimates in the PFS Update average US$42/lb U_3O_8 for the life-of-mine. Given the relatively shallow nature of the open pit mine and the uniform leaching characteristics of the deposit, the Project has a flat operating cost profile.

Operating costs are defined as direct operating costs including mining, processing, on-site and off-site infrastructure and utilities, and general and administrative costs, as follows:

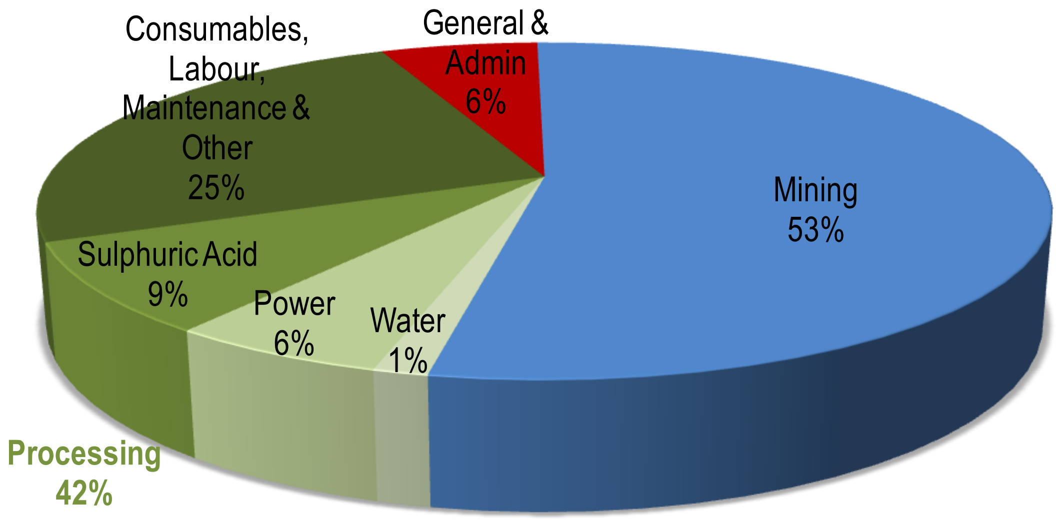
Table 21-2
Operating Cost Estimate
PFS Update (December 2010)

Operating Cost Estimate	**Heap Leaching**	
	(US$/tonne processed)	(US$/lb U_3O_8)
Mining	8.24	22.60
Processing	6.28	17.23
General & administrative	0.94	2.58
Total	**15.46**	**42.41**

Operating costs do not include Government royalties or freight and selling costs, which are treated as deductions from revenue in the economic analysis for the Project.

The breakdown of the estimated operating costs is shown graphically below:

Figure 21-2
PFS Update – Pie Chart of Estimated Operating Costs



Mining costs include mining supervision, equipment, consumable, labour, drill and blast, and maintenance costs. Refer Section 16.7.

Processing costs include labour, power, water, reagents and consumables, maintenance and other processing-related costs.

With the inclusion of in-pit Inferred mineral resources, the operating cost estimate reduces slightly to US$42.23/lb U_3O_8 produced.

22 ECONOMIC ANALYSIS

The following sections summarise the results of the PFS Update which are now subject to review as part of the DFS presently underway.

22.1 Basis of Economic Analysis

The Etango Project is owned 100% by Bannerman Namibia which in turn is owned 80% by Bannerman. The economic analysis reflects the results of the project at the Bannerman Namibia level excluding the effects of financing. The financial model results also do not reflect the potential dilution, as may occur under the existing shareholder agreement, of the minority 20% shareholding in Bannerman Namibia to a 2% net revenue royalty.

All revenue and cost estimates are expressed in United States Dollars (US$) and are based on real 2010 values.

The key assumptions incorporated into the financial model for the PFS analysis are described in further detail below.

22.1.1 Revenue

Final uranium output is assumed to be sold at a base case long term contract price of US$75/lb U_3O_8. Sensitivities have also been run at different price assumptions. Net revenue has been calculated after deducting royalties and an allowance for the estimated marketing, freight and conversion assaying costs prior to sale at the relevant conversion facility.

22.1.2 Royalties

The Namibian Government royalty on uranium mining is 3% of gross revenue. Along with additional allowances for off-site shipping, marketing and other sales-related costs, royalties have been deducted from gross sales revenue and are not included in the operating cost estimate.

The economic analysis assumes that the Vendor royalty (refer Section 4.6.2) does not come into existence.

22.1.3 Capital and Operating Costs

The estimated capital and operating costs are discussed further in Section 21.

22.1.4 Working Capital

A working capital build-up and delay between production and cash revenue receipts of 4 months has been assumed to simulate the estimated timeframe of the uranium oxide sales process.

22.1.5 Net Present Value (NPV)

The Project's NPV is calculated based on the annual net cashflows. The financial model is configured such that a range of discount rates can be applied. For the purposes of the base case evaluation, a real discount rate of 10% has been assumed.

22.1.6 Internal Rate of Return (IRR)

The IRR's for the Project are calculated using the annual net cashflows.

22.1.7 Payback Period

The payback period is defined as the period of time in which the cumulative undiscounted net cashflow ultimately becomes positive.

22.1.8 Tax

After-tax results have been calculated based on the Namibian tax legislation currently applying. The corporate tax rate for mining companies in Namibia is presently 37.5% of taxable profits, with a capital deductions regime allowing the deduction of pre-production and other capital expenditure over a three year period.

22.2 Key Assumptions

Based on the various assumptions and matters set out above, the economic evaluation of the Etango project PFS Update (December 2010) has been prepared on the base case of an open pit mining operation with a nominal 15Mtpa heap leach operation.

In calculating the potential returns from the project, the following fundamental assumptions were made in the PFS Update:

Table 22-1
Fundamental Assumptions of Financial Modelling Analysis

Basis	Project level and pre-debt financing.
U_3O_8 prices	Long term contract price assumed at US$75/lb U_3O_8.
Development period	2-3 years.
Mine life	20 years, based on the October 2010 mineral resource estimate.
Annual throughput	15 million tonnes of ore per year.
Fuel price	US$0.81 per litre, plus US$100,000 p.a. management fee.
Sulphuric acid price	US$100/tonne delivered to site.
Raw water cost	US$1.08 per cubic metre plus capital allocation for SWRO plant and associated infrastructure to Etango.
Power cost	US$0.08 per kilowatt hour.
Production rate	Approximately 5-7 million pounds of U_3O_8 per year.
Exchange rates	US$1.00 : A$1.19 : N$7.80 : R7.80 : €0.80

22.3 Economic Assessment

The key outputs from the financial model based on the above assumptions are tabulated below.

Based on the results of the feasibility study update, life-of-mine production from the Measured and Indicated resources of the primary Etango deposit is estimated as 106Mlbs U_3O_8 using heap leach processing.

The mining schedule in the PFS Update provided for approximately 20 years of production at an average of 5-7Mlbs U_3O_8 per annum. Bannerman expects additional material to be included in the ultimate open pit mine design based on the addition of Inferred resources at the Ondjamba and Hyena satellite deposits.

The key economic assessment statistics for the Etango Project from the PFS Update are tabulated below:

Table 22-2
Key Project Economic Assessment Statistics
PFS Update (December 2010)

Total material processed* (M tonnes)	292 Mt
Annual throughput (M tonnes ore)	15 Mt
Average head grade (ppm U_3O_8)	195
Stripping ratio (waste/ore) (*Reduces to 3.5:1 with the inclusion of in-pit Inferred mineral resources)	3.7* : 1.0
Metallurgical recovery	85%
Life of mine production* (Mlbs U_3O_8)	106 Mlbs
Annual production (Mlbs U_3O_8)	5-7 Mlbs
Mine life	20 years
Initial capital cost (real 2010 dollars), excluding mining fleet and working capital	US$638 million
Life of mine operating cost (US$/lb U_3O_8)	US$42
Assumed uranium price (US$/lb U_3O_8)	US$75
Payback period at US$75/lb U_3O_8 (pre/post tax)	6 years / 6 years
IRR at US$75/lb U_3O_8 (pre/post tax)	15.1% / 12.3%
$NPV_{10\%}$ at US$75/lb U_3O_8 (pre/post tax)	US$276 million / US$99 million

* From Measured and Indicated mineral resources only. An additional 10.9Mt of in-pit Inferred mineral resources, giving rise to an additional 2.5Mlbs of U_3O_8 production over the life-of-mine, has not been included in the analysis in this Section.

Summary statistics from the annual cashflow model are contained on the following page, reflecting the results of the PFS Update based on only the Measured and Indicated mineral resources of the Etango deposit. The addition of in-pit Inferred mineral resources, representing approximately 3.7% additional mineralised material, improves the cashflows slightly.

Table 22-3
Summary Annual Production and Cashflow Statistics - PFS Update (December 2010)

ETANGO PRELIMINARY FEASIBILITY STUDY UPDATE (December 2010)	TOTAL	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	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24
Physicals																									
Mining, Milling and Production (Mt)																									
Ore	302.7			2.8	17.9	20.3	23.1	21.4	20.0	22.3	22.2	21.9	23.0	20.0	24.7	25.0	22.9	12.0	3.3						
Waste	1,058.5			14.6	72.6	68.1	80.9	89.0	90.4	88.5	88.9	87.7	78.1	76.3	77.7	56.6	57.1	27.9	4.2						
Total Material Mined	1,361.2			17.4	90.5	88.4	104.0	110.3	110.5	110.8	111.0	109.6	101.1	96.3	102.4	81.6	80.0	39.8	7.4						
Grade (ppm)	191.8			153	189	206	194	193	205	201	196	190	189	207	177	180	182	185	201						
Ore Feed (Measured and Indicated resources only)	291.8				13.5	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	8.1	
Grade (ppm) (Measured and Indicated resources only)	194.6				220	235	233	223	233	243	233	221	226	234	211	219	214	171	138	121	121	121	121	121	
Recovery (%)					85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	
Production U_3O_8 (000t)	48.3				2.5	3.0	3.0	2.9	3.0	3.1	3.0	2.8	2.9	3.0	2.7	2.8	2.7	2.2	1.8	1.5	1.5	1.5	1.5	0.8	
Production U_3O_8 (mlbs)	106.4				5.6	6.6	6.6	6.3	6.6	6.9	6.5	6.2	6.3	6.6	5.9	6.2	6.0	4.8	3.9	3.4	3.4	3.4	3.4	1.8	
Revenue (US$m)																									
Price (US$/lb)	75				75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75
Sales (mlbs)	106.4				3.5	6.5	6.6	6.4	6.5	6.8	6.6	6.3	6.3	6.5	6.1	6.1	6.1	5.2	4.2	3.6	3.4	3.4	3.4	2.4	0.6
Gross Revenue	7,982.5				259.9	489.5	493.2	478.5	485.2	506.8	498.6	474.0	472.7	488.2	460.9	456.2	456.0	391.8	315.0	267.5	255.7	255.2	255.2	176.6	45.7
Royalties and Conversion Costs	399.1				13.0	24.5	24.7	23.9	24.3	25.3	24.9	23.7	23.6	24.4	23.0	22.8	22.8	19.6	15.8	13.4	12.8	12.8	12.8	8.8	2.3
Net Revenue	7,583.3				246.9	465.0	468.5	454.6	460.9	481.5	473.6	450.3	449.1	463.8	437.9	433.4	433.2	372.2	299.3	254.2	242.9	242.4	242.5	167.7	43.4
Operating Expenditure (US$m)																									
Mining	2,405.0				140.9	139.4	172.5	185.2	190.2	191.2	201.9	191.5	183.4	171.4	176.1	150.2	162.3	89.3	18.5	9.0	9.0	9.0	9.0	4.8	
Processing	1,827.3				86.4	93.9	94.0	94.0	93.9	94.0	93.8	93.8	94.0	94.0	93.8	94.0	94.0	93.8	93.9	93.9	94.0	93.7	93.9	50.4	
Infrastructure	7.0				0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.2	
Owners Costs	275.1				14.1	14.1	14.1	14.1	14.1	14.1	14.1	14.1	14.1	14.1	14.1	14.1	14.1	14.1	14.1	14.1	14.1	14.1	14.1	7.6	
Total Operating Expenditure	4,514.3				241.7	247.8	280.9	293.6	298.5	299.7	310.1	299.8	291.8	279.8	284.4	258.6	270.8	197.6	126.8	117.3	117.5	117.1	117.4	63.0	
Pre-Tax Cash Flow	**3,069.0**				**5.2**	**217.2**	**187.6**	**161.0**	**162.4**	**181.8**	**163.5**	**150.5**	**157.3**	**183.9**	**153.5**	**174.8**	**162.4**	**174.7**	**172.5**	**136.8**	**125.4**	**125.2**	**125.1**	**104.7**	**43.4**
Tax	718.3										37.5	54.9	56.8	64.2	48.2	53.0	49.1	58.9	61.8	50.6	46.5	46.6	46.5	38.9	4.7
Post-Tax Cash Flow	**2,350.7**				**5.2**	**217.2**	**187.6**	**161.0**	**162.4**	**181.8**	**125.9**	**95.6**	**100.5**	**119.8**	**105.3**	**121.8**	**113.4**	**115.8**	**110.7**	**86.2**	**78.9**	**78.7**	**78.6**	**65.8**	**38.7**
Capital Expenditure (USm)																									
Mining Direct Costs	368.2			96.5	84.3	27.6	9.0		18.8		2.4	6.8	5.3	23.5	43.1	30.9	17.7	1.3	1.1						
Processing Direct Costs	203.3		81.3	122.0																					
Infrastructure Direct Costs	147.7		34.9	52.4			60.4																		
Owners Direct Costs	43.3	6.0	8.2	29.1																					
External Infrastructure	104.0	0.2	49.2	54.6																					
Sustaining Capital	16.7				0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	
Spares, Ocean Freight, First Fills and Commissioning	25.7		10.3	15.4																					
Mobilisation and Demobilisation	4.6		1.8	2.7																					
EPCM & PCM for Turnkey Packages & Accuracy Provision	113.7	11.3	45.5	57.0																					
Rehabilitation	33.0						0.5			0.5			0.5			0.5			0.5			0.5			30.0
Construction Camp and Facilities	23.4		9.4	14.0																					
Owners Contingency																									
Total Capital Expenditure	1,083.6	17.5	240.6	443.7	85.1	28.4	70.7	0.8	19.6	1.3	3.2	7.6	6.6	24.3	43.9	32.2	18.5	2.1	2.4	0.8	0.8	1.3	0.8	0.8	30.0
Post Capital Expenditure Cash Flow	1,267.1	(17.5)	(240.6)	(443.7)	(80.0)	188.9	116.9	160.1	142.8	180.5	122.7	88.0	93.8	95.5	61.3	89.6	94.8	113.6	108.2	85.4	78.0	77.3	77.7	65.0	8.7

22.4 Sensitivity Analyses

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

The Etango 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.

22.4.1 Sensitivity to Changes in U_3O_8 Prices

The Etango Project is most sensitive to changes in uranium prices. Positive movements of 10% and 20% from the base case assumption of US$75/lb U_3O_8 produce significant changes in the post-tax NPV from US$99 million to US$289 million and US$475 million respectively, the latter with a post-tax IRR of 20.0%. Likewise, negative movements of 10% and 20% from the base case assumption of US$75/lb U_3O_8 result in the post-tax NPV reducing from US$99 million to US$(99) million and US$(317) million respectively. A 20% increase in the U_3O_8 price reduces the payback period by 2 years (to 4 years) and a 20% decrease in the U_3O_8 price increases the payback by 9 years (to 15 years).

Should higher prices than the base case assumption be available to the Project, then the economics become immediately and significantly more attractive.

22.4.2 Sensitivity to Changes in Total Operating Costs

Given the large annual throughput of the project, the financials are also very sensitive to changes in total operating costs.

Operating cost reductions of 10% and 20% from the base case assumptions result in the post-tax NPV increasing from US$99 million to US$219 million and US$337 million respectively, the latter with a post-tax IRR of 17.4%. Increases of 10% and 20% in the base case cost assumptions produce significant adverse changes in the post-tax NPV from US$99 million to US$(25) million and US$(154) million respectively, the latter with a post-tax IRR of 6.8%. A 20% decrease in total operating costs reduces the payback period by 1 year (to 5 years) and a 20% increase in total operating costs increases the payback period by 4 years (to 10 years).

22.4.3 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.

Increases of 10% and 20% in the base case capital cost assumptions produce adverse changes in the post-tax NPV from US$99 million to US$36 million and US$(28) million respectively, the latter with a post-tax IRR of 9.6%. Likewise, capital cost reductions of 10% and 20% from the base case assumptions result in the post-tax NPV increasing from US$99 million to US$161 million and US$222 million respectively, the latter with a post-tax IRR of 15.9%. A 10% decrease in capital costs reduces the payback period by less than one year (to remain at 6 years) and a 10% increase in capital costs increases the payback period by 1 year (to 7 years).

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.

23.1 Rössing Mine

The Rössing Mine is controlled by Rössing Uranium Limited which in turn is owned by Rio Tinto (69%), the Government of Iran (15%), the Industrial Development Corporation of South Africa (10%), the Namibian Government (3%), and private ownership (3%). The mine is the third largest uranium mine in the world, and is the largest granite-hosted uranium mine in the world, and is located approximately 13km from the north-eastern boundary of EPL 3345. Production commenced in 1976. In 2009, Rössing completed a feasibility study into an expansion of the mine and a program to extend the mine life to 2023 and beyond (Aurecon, 2010).

Uranium mineralisation at Rössing is associated with syn/post-D3 alaskites (Basson and Greenway, 2004) which have preferentially intruded into pyroxene-hornblende gneiss and biotite-amphibole schist units of the Khan Formation in the northern ore zone, and into biotite-amphibole schist/lower marble/lower biotite-cordierite gneiss of the Rössing Formation in the central ore zone (Roesener and Schreuder, 1997). The main primary uranium mineral is magmatic uraninite (Basson and Greenway, 2004).

The alaskites range in size from small quartzo-feldspathic lenses to large intrusive bodies, with the bulk of the economic mineralisation being contained in alaskite on the northern limb of the 'mine' synclinorium (Roesener and Schreuder, 1997).

The Rössing style of mineralisation is identical 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.

In 2009, Rössing mined 54.5Mt of rock and produced 9.3Mlbs (4,150 tonnes) of U_3O_8.

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 October 2010, Paladin estimated the remaining total mineral resources at the Langer Heinrich Mine to be 142.8Mt at 550ppm U_3O_8 for 173Mlbs of U_3O_8, at a 250ppm U_3O_8 cut-off grade. The remaining mineral reserves were estimated at 110.2Mt at 550ppm U_3O_8 for

134Mlbs of U_3O_8, at a 250ppm U_3O_8 cut-off grade, approximately 10% of this was in existing stockpiles.

23.3 Husab (Rössing South) Project

The Husab project is owned by a subsidiary of Extract Resources Ltd (**Extract**). It consists of two EPL's with a total area of 637km² and is located between Bannerman's tenements (EPL 3345 and EPL 3346).

The 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 of 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, at the project, 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 Improvement Review

Following release of the PFS Update in December 2010, Bannerman commenced a review of the Etango Project utilising in-house and external technical consultants. The work identified opportunities to enhance the Project economics, incorporating potential operating cost reductions of up to US$4/lb U_3O_8 equating to a potential reduction in estimated operating costs to US$38/lb U_3O_8 produced:

- Investigation of an increase in the processing throughput from 15Mtpa to 20Mtpa. This opportunity arose due to the mining schedule presently incorporating stockpiling of lower grade mineralisation in the earlier years of the operation. This stockpiled material can be processed through an expanded crushing and heap leaching facility for no additional mining cost as it is already fully costed for mining prior to stockpiling. In addition, avoidance of the need to delineate low grade material for stockpiling is expected to create opportunities for improved mining efficiency. The mine life is not expected to reduce materially because presently only approximately 60% of the total resource estimate has been subjected to the PFS Update, and further resources are expected to come into the ultimate mine designs.

- The nature and scale of the envisaged large open pit mining operation created opportunities to favourably impact on the operating costs, including:

  - Scheduling had indicated that some of the waste rock could be back-filled into the mined pit voids; and

  - Optimising the site layout of the processing infrastructure, waste dumps and heap leach residue pad to gain maximum benefit from minimising the materials handling costs of the location and layout of the crushing, heap leaching and extraction circuits; and

- Investigating the co-disposal of washed heap leach residue within the waste rock dumps. The relatively inert and coarse characteristics of the depleted heap leached material make this a potentially viable approach leading to savings in operating and capital costs.

The above opportunities formed part of the initial work being pursued within the DFS.

24.2 Definitive Feasibility Study (DFS)

Bannerman commenced the Etango Project DFS at the start of the June 2011 quarter, with the initial phase of work focused on incorporating the key project improvements previously identified to the PFS Update.

Key changes in the project configuration stemming from the improvement work include a 33% increase in the plant throughput, simplification of the mining approach and revisions to the project layout. These changes are expected to result in production estimates increasing by approximately 20% from 5-7Mlbs U_3O_8 per annum in the PFS Update to 6-8Mlbs U_3O_8

per annum. A reduction in estimated average life-of-mine operating costs from US$42/lb to US$38/lb U_3O_8 is targeted.

Bannerman expects to continue to progress the DFS while carefully managing its expenditure rates, with the objective of completing the DFS by the end of March 2012.

24.2.1 Mining

The DFS is planned to be based on a 33% increased plant throughput scenario of 20Mtpa resulting in an approximately 20% increased annual uranium oxide production of 6-8Mlbs compared with the PFS Update. The increased plant throughput captures an opportunity based on the previous approach of stockpiling approximately 5 million tonnes of low grade material annually, coupled with simplifying the mining schedule to minimise the material movement costs.

The status of the key mining activities at the date of this report is as follows:

- A review of the mining dilution model has confirmed the robustness of the PFS Update estimates;
- Work on optimising the pit and mining equipment configuration is ongoing;
- A review of the layout of the project site has highlighted potential operating cost savings associated with prioritising the location of the waste dumps over that of the processing infrastructure;
- Mine scheduling based on the simplified single-product approach has confirmed the opportunity to reduce the material movement in the early years of the mine life with likely commensurate benefits in the operating and capital costs during this period;
- An initial review of the maintenance support capabilities of a number of equipment suppliers has been completed to consider the relative merits of conducting in-house or outsourced maintenance; and
- Investigation of the technical and economic viability of co-disposal of washed heap leach residue within the mined waste rock dumps is ongoing. The coarse nature of the leach residue material makes this a potentially viable approach.

24.2.2 Processing

The status of key processing activities at the date of this report was:

- A review of the heap leaching parameters, including pad dimensions, acid concentration and acid addition rates, was well advanced and indicated that it may be advantageous to reduce the height of the heap leach pad;
- Testwork also confirmed that little or no additional oxidation agents are required for efficient leaching; and
- Metallurgical testing has been substantially relocated from Perth, Australia, to Swakopmund, Namibia. This change will not only improve sample logistics but will also

enable ongoing cost-effective testing for the establishment of a large-scale field test program envisaged for early 2012.

24.2.3 Engineering

Engineering activities are currently focused on the design and layout changes required to give effect to the increase in plant throughput to 20Mtpa.

24.2.4 Environmental and Social Impact Assessment

The hydrogeological model for the Etango Project will be re-run once the revised plant and site layout has been completed.

A geo-chemical testwork program to evaluate the opportunity of co-disposing the washed heap leach residue with the mine waste rock has commenced.

25 INTERPRETATION AND CONCLUSIONS

25.1 Geology and Resources

The region of EPL 3345 currently represents the most significant asset for Bannerman due to the advanced nature of exploration and the identified Measured, Indicated and Inferred Mineral Resources in the Anomaly A, Oshiveli and Onkelo deposits at Etango and the nearby Ondjamba and Hyena satellite deposits. Bannerman's other assets include the EPL 3346 project, although this is not currently considered to be a material asset of the Company.

The Etango Project hosts a significant uranium resource and represents an advanced exploration project which is now undergoing feasibility studies. The western flank of the Palmenhorst Dome represents a prospective strike length of over 15km which incorporates the Anomaly A, Oshiveli, Onkelo, Ondjamba and Hyena deposits. The eastern flank of the Palmenhorst Dome is also highly prospective, as are other soil and sand covered areas in the southern portion of EPL 3345, to the north of Etango and elsewhere in the vicinity.

EPL 3345 is located within the highly prospective southern Central Zone of the Damara Orogenic Belt. Currently 12 historic uranium anomalies have been identified over the EPL 3345 area, some of which correspond to radiometric anomalies associated with the Rössingberg Dome and the Palmenhorst Dome. EPL 3346 is considered prospective for primary and calcrete hosted uranium mineralisation, although most of the current work is focussed upon the Etango Project within EPL 3345.

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.

25.2 Mining

Coffey Mining recommends the following further detailed geotechnical assessment to be undertaken during the DFS:

- Drilling of alaskite contact to recover fault gouge to better characterise shear strength of the material with remoulded triaxial testing as well as remoulded direct shear testing.
- To run stability analysis utilising future findings from a further detailed hydro-geological study.
- Geotechnical evaluation of the blasting and 'diggability' of the rock.

25.3 Metallurgical

Metallurgical testwork has been completed on drill core samples to further define the comminution and leaching characteristics of the Alaskite mineralisation of the Anomaly A, Oshiveli and Onkelo deposits.

The testwork indicates that the ore properties support 3 Stage crushing with HPGR to a nominal 4mm product size with the agglomerated ore stacked to 7m on an acid heap leach.

Chemical analysis confirms that the samples analysed are characterised by low levels of impurity elements.

Further column testwork is proposed to provide additional detailed ore performance parameters for ongoing process and general engineering studies.

25.4 Geotechnical and Hydrology

A transient hydrogeological model has been developed.

Objectives of the further investigatory work will include:

- Construct groundwater monitoring facilities around the Heap Leach Pads and liquor ponds; and
- Groundwater investigatory drilling and bore construction to further investigate pit hydrogeology as each stage of mining proceeds. For the first stage, five bores should be allowed with depths of penetration at least to the base of Stage 1 mining.

Further hydrogeological studies be undertaken to investigate the paleodrainage aquifer and continue with monitoring of the groundwater monitoring network to ensure a good background of base-line (pre-mining) data.

25.5 Project Development

Bannerman is commencing the DFS activities and updating the approved ESIA in 2011.

26 RECOMMENDATIONS

Bannerman has committed to a Definitive Feasibility Study (DFS) to demonstrate the economic potential of the Etango Project. The study will include the following activities.

26.1 Resource Definition and Modelling

Bannerman is planning further exploration efforts to define the full extents of the mineralisation around the Etango, Ondjamba and Hyena deposits. Most of the immediate Etango Project area has now been intensively delineated by resource definition, infill, metallurgical bulk sample and geotechnical drilling programmes and the only drilling rigs currently in operation in the project area are those on periodic exploration work programs.

It is recommended that a Multiple-Indicator-Kriged resource model be considered to emulate any selective mining scenarios for the Etango Project. It is also recommended that sample recovery be routinely recorded for RC drilling samples.

26.2 Mining Studies

The mining work during the DFS will be the development of the final feasibility study mine plan and associated capital and operating cost estimates.

26.3 Geotechnical and Hydrology

Additional development of the transient hydrogeological model and detailed geotechnical assessment as detailed in Section 25.2 to be undertaken during the DFS.

26.4 Metallurgical Testwork

Additional column testwork on drill core to further refine the reagent consumptions and improve confidence in the ore characteristics determined to date and to provide information required for continued process and general engineering development.

During 2011-12, Bannerman intends to install a number 45t crib test units, on site at Etango, to provide additional metallurgical, engineering and operational information.

27 REFERENCES

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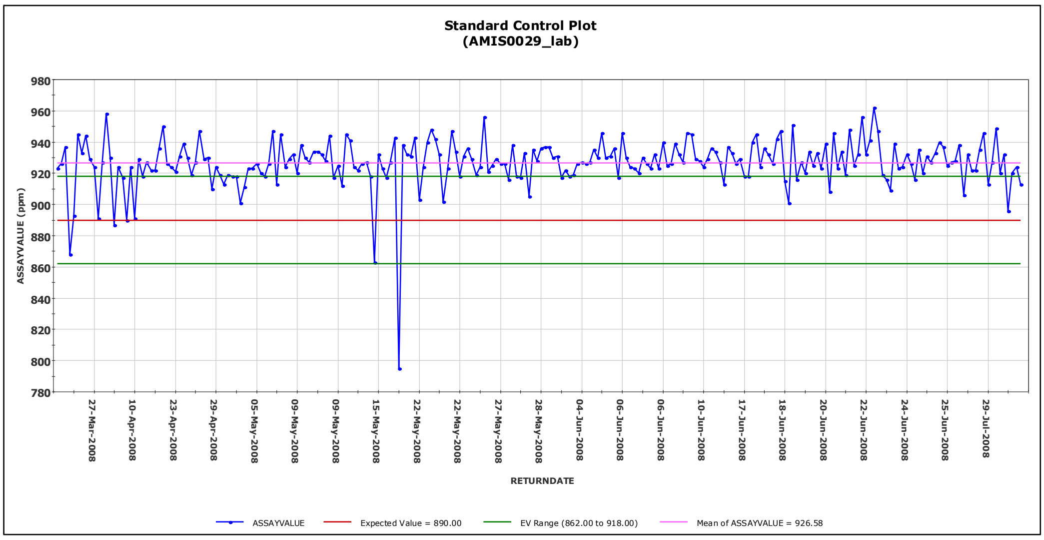
Appendix 1
QAQC Plots

Appendix 1
QAQC Plots

Bannerman Supplied Standards – SGS Johannesburg

Lab Standard - SGS Johannesburg (AMIS0029_lab)

Standard:	AMIS0029	No of Analyses:	238
Element:	ASSAYVALUE	Minimum:	795.00
Units:	ppm	Maximum:	962.00
Detection Limit:	5	Mean:	926.58
Expected Value (EV):	890.00	Std Deviation:	15.83
E.V. Range:	862.00 to 918.00	% in Tolerance	18.91 %
		% Bias	4.11 %
		% RSD	1.71 %



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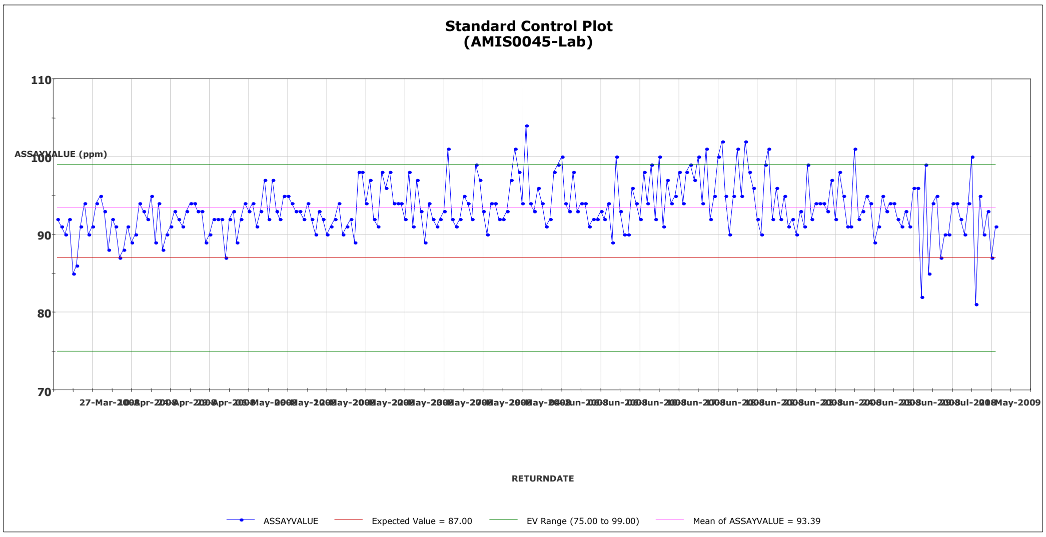
Standard - SGS Johannesburg (AMIS0045-Lab)

Standard:	AMIS0045	No of Analyses:	241
Element:	ASSAYVALUE	Minimum:	81.00
Units:	ppm	Maximum:	104.00
Detection Limit:	5	Mean:	93.39
Expected Value (EV):	87.00	Std Deviation:	3.47
E.V. Range:	75.00 to 99.00	% in Tolerance	93.78 %
		% Bias	7.34 %
		% RSD	3.72 %



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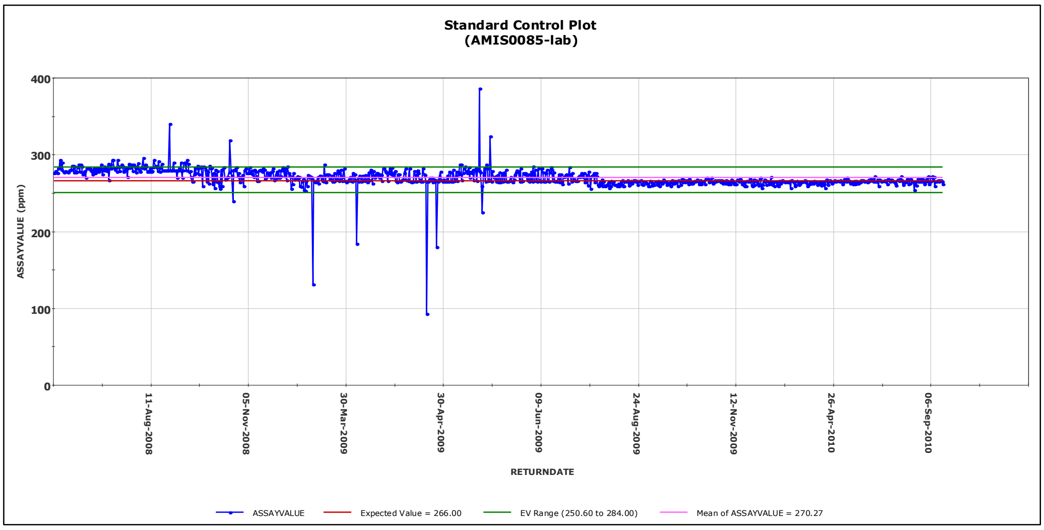
QAQC Plots

Standard - SGS Johannesburg (AMIS0085-lab)

Standard:	AMIS0085	No of Analyses:	912
Element:	ASSAYVALUE	Minimum:	93.00
Units:	ppm	Maximum:	386.00
Detection Limit:	10	Mean:	270.27
Expected Value (EV):	266.00	Std Deviation:	12.88
E.V. Range:	250.60 to 284.00	% in Tolerance	93.42 %
		% Bias	1.60 %
		% RSD	4.76 %



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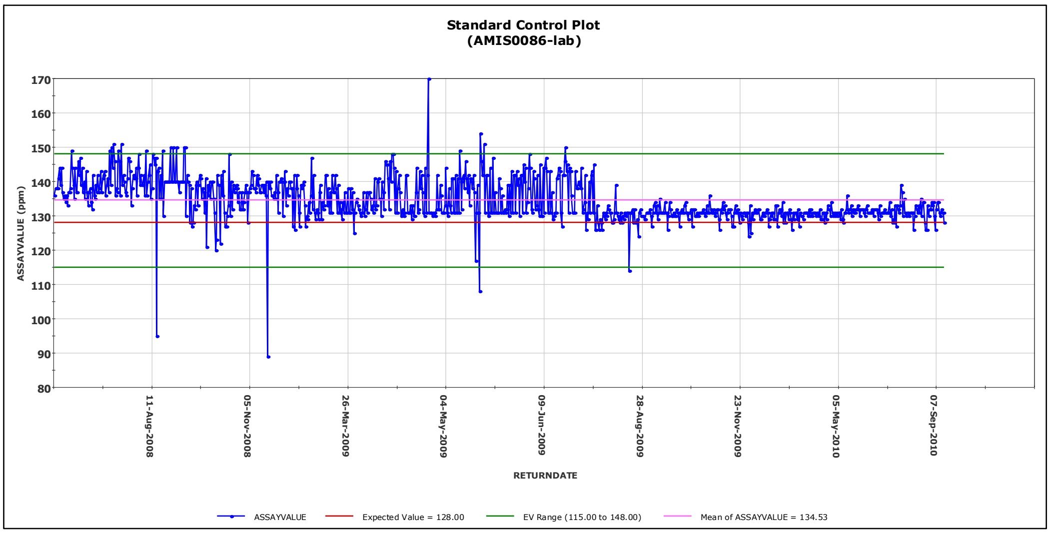
Standard SGS Johannesburg (AMIS0086-lab)

Standard:	AMIS0086	No of Analyses:	908
Element:	ASSAYVALUE	Minimum:	89.00
Units:	ppm	Maximum:	170.00
Detection Limit:	10	Mean:	134.53
Expected Value (EV):	128.00	Std Deviation:	6.15
E.V. Range:	115.00 to 148.00	% in Tolerance	97.58 %
		% Bias	5.10 %
		% RSD	4.57 %



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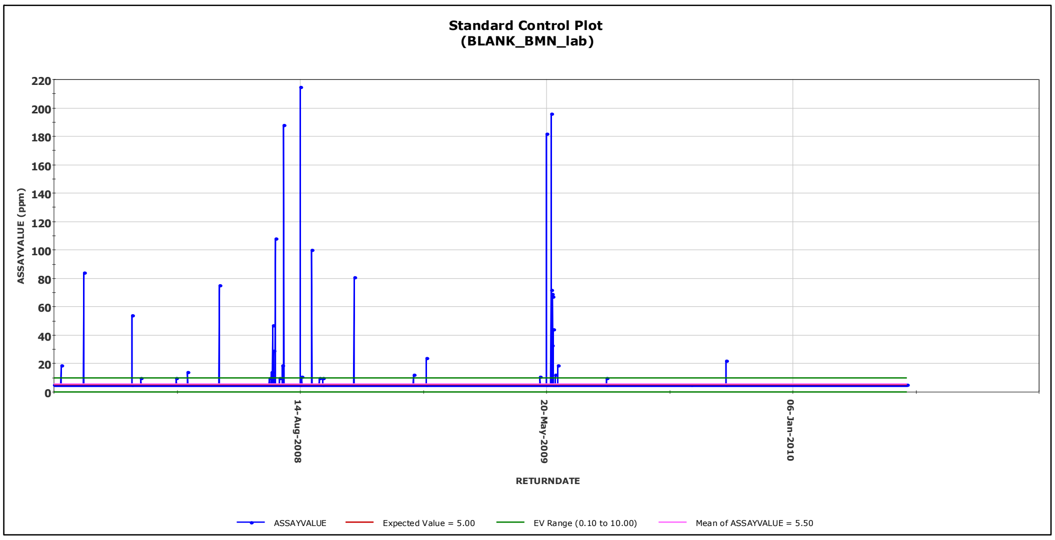
Appendix 1
QAQC Plots

Lab Standard SGS Johannesburg (BLANK_BMN_lab)

Standard: Element: Units: Detection Limit: Expected Value (EV): E.V. Range:	BLANK_BMN ASSAYVALUE ppm 10 5.00 0.10 to 10.00		No of Analyses: Minimum: Maximum: Mean: Std Deviation: % in Tolerance % Bias % RSD	3463 5.00 215.00 5.50 7.64 99.19 % 9.91 % 139.09 %



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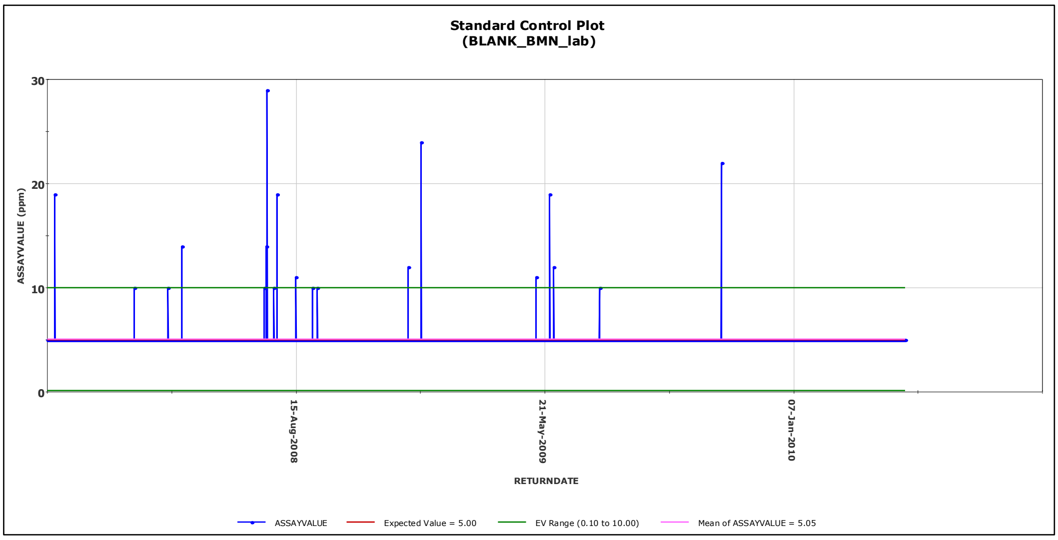
Lab Standard Waste Rock (BLANK_BMN_lab)

Standard: Element: Units: Detection Limit: Expected Value (EV): E.V. Range:	BLANK_BMN ASSAYVALUE ppm 10 5.00 0.10 to 10.00		No of Analyses: Minimum: Maximum: Mean: Std Deviation: % in Tolerance % Bias % RSD	3447 5.00 29.00 5.05 0.82 99.65 % 1.05 % 16.21 %



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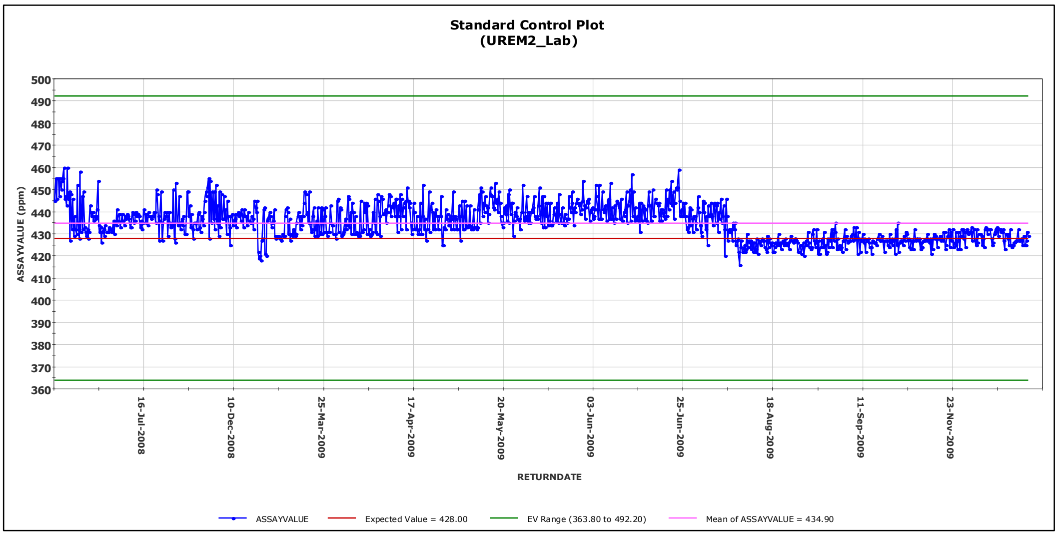
Appendix 1
QAQC Plots

SGS Johannesburg Internal Standards

Lab Standard - SGS Johannesburg (UREM2_Lab)

Standard:	UREM2	No of Analyses:	1084
Element:	ASSAYVALUE	Minimum:	416.00
Units:	ppm	Maximum:	460.00
Detection Limit:	5	Mean:	434.90
Expected Value (EV):	428.00	Std Deviation:	7.86
E.V. Range:	363.80 to 492.20	% in Tolerance	100.00 %
		% Bias	1.61 %
		% RSD	1.81 %



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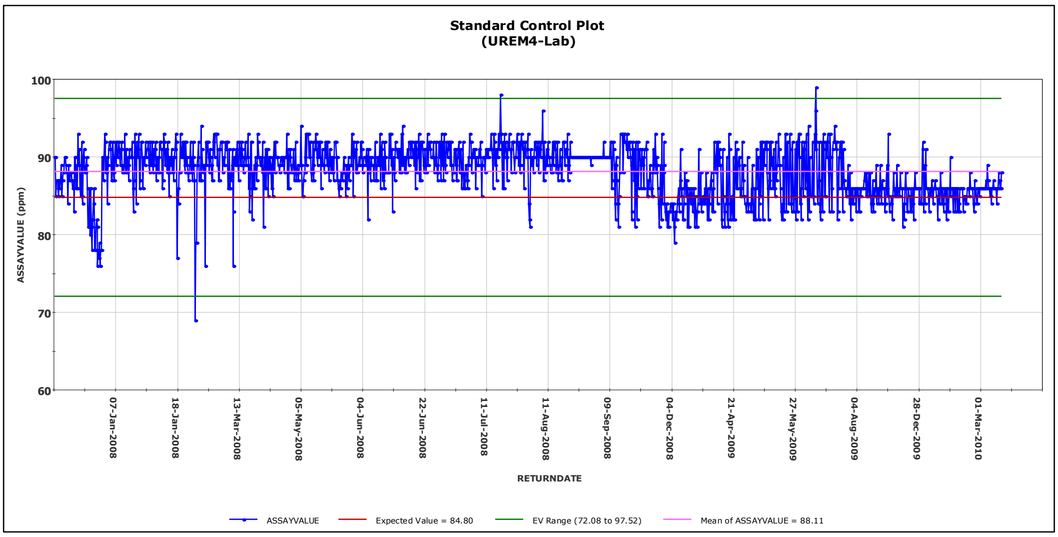
Lab Standard - SGS Johannesburg (UREM4-Lab)

Standard:	UREM4	No of Analyses:	1534
Element:	ASSAYVALUE	Minimum:	69.00
Units:	ppm	Maximum:	99.00
Detection Limit:	10	Mean:	88.11
Expected Value (EV):	84.80	Std Deviation:	3.30
E.V. Range:	72.08 to 97.52	% in Tolerance	99.80 %
		% Bias	3.90 %
		% RSD	3.75 %



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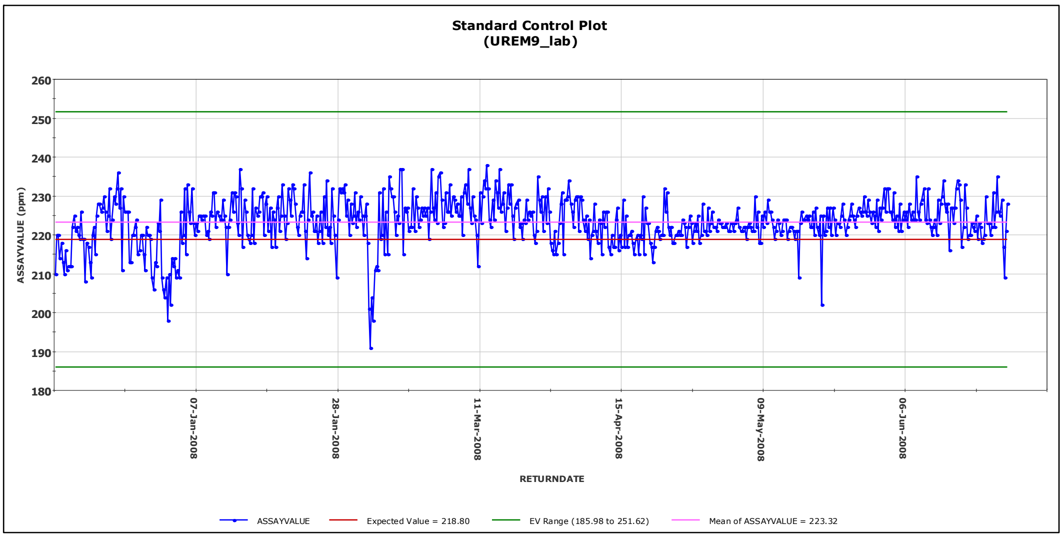
Appendix 1
QAQC Plots

Lab Standard - SGS Johannesburg (UREM9_lab)

Standard:	UREM9	No of Analyses:	672
Element:	ASSAYVALUE	Minimum:	191.00
Units:	ppm	Maximum:	238.00
Detection Limit:	10	Mean:	223.32
Expected Value (EV):	218.80	Std Deviation:	6.16
E.V. Range:	185.98 to 251.62	% in Tolerance	100.00 %
		% Bias	2.07 %
		% RSD	2.76 %



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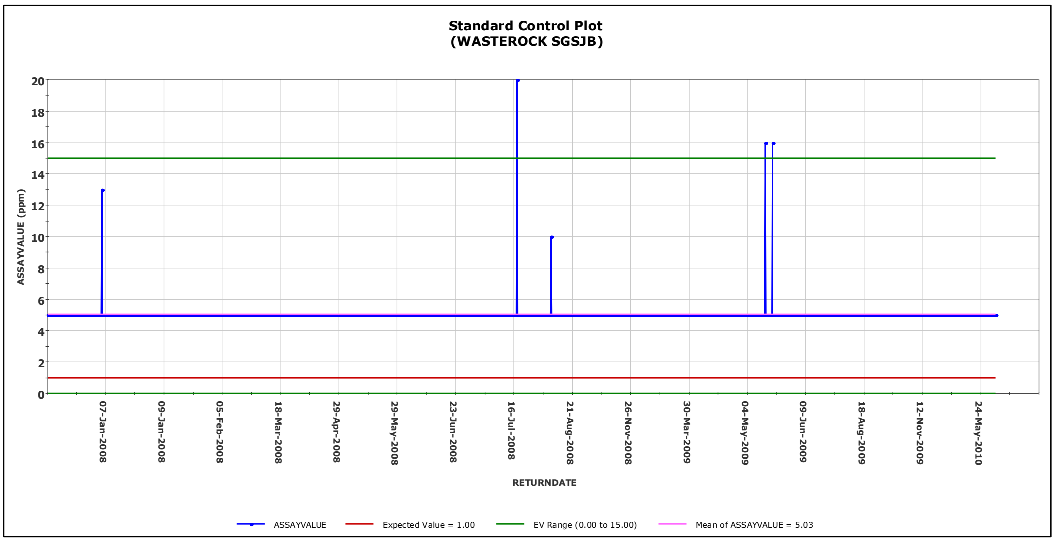
Lab Standard Waste Rock (WASTEROCK SGSJB)

Standard:	WASTE ROCK	No of Analyses:	1626
Element:	ASSAYVALUE	Minimum:	5.00
Units:		Maximum:	20.00
Detection Limit:	10	Mean:	5.03
Expected Value (EV):	1.00	Std Deviation:	0.58
E.V. Range:	0.00 to 15.00	% in Tolerance	99.82 %
		% Bias	403.08 %
		% RSD	11.61 %



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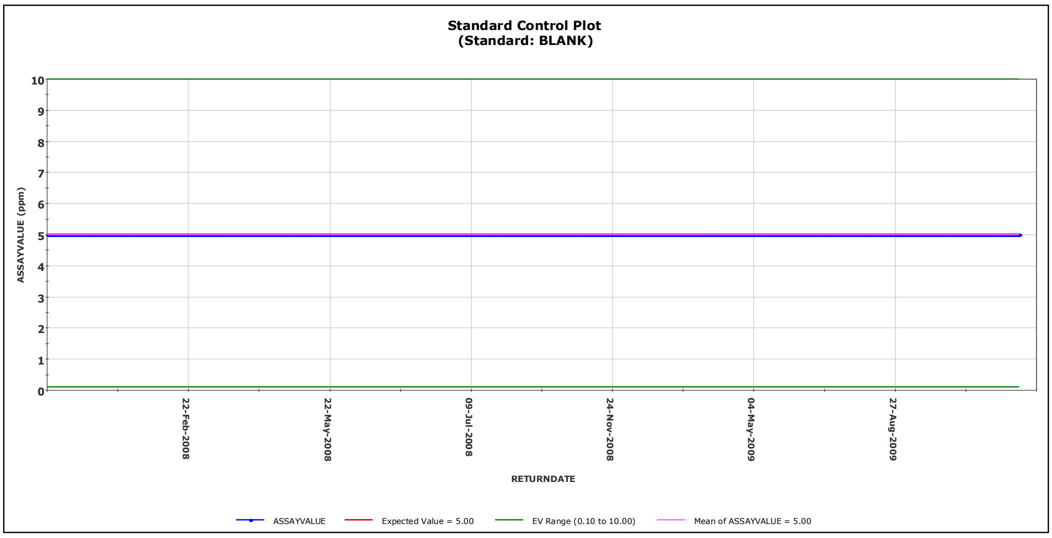
Appendix 1
QAQC Plots

Lab Standard SGS Johannesburg
(Standard: BLANK)

Standard:	BLANK		No of Analyses:	6877
Element:	ASSAYVALUE		Minimum:	5.00
Units:	ppm		Maximum:	5.00
Detection Limit:	10		Mean:	5.00
Expected Value (EV):	5.00		Std Deviation:	0.00
E.V. Range:	0.10 to 10.00		% in Tolerance	100.00 %
			% Bias	0.00 %
			% RSD	0.00 %



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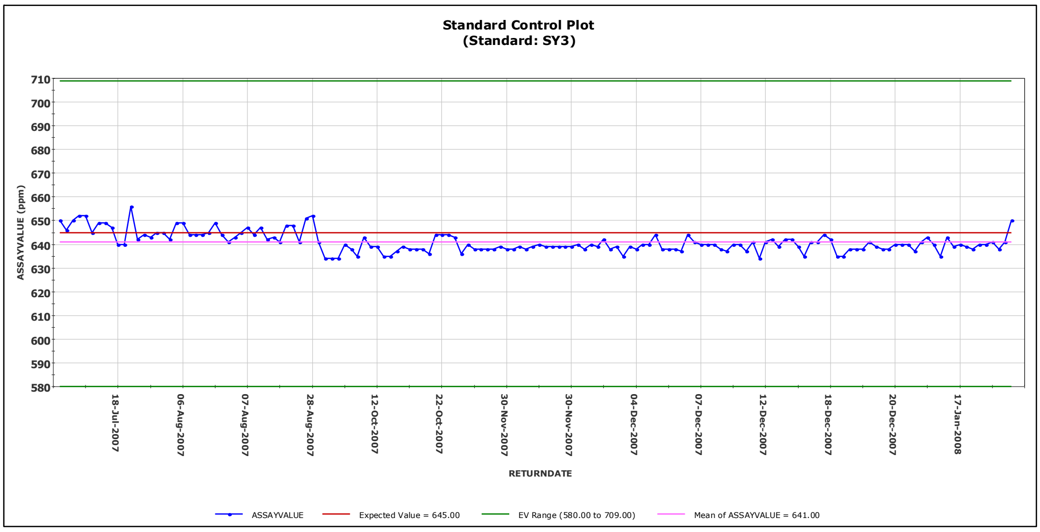
Appendix 1
QAQC Plots

SGS Perth Internal Standards

Lab Standard - SGS Perth (Standard: SY3)

Standard: Element: Units: Detection Limit: Expected Value (EV): E.V. Range:	SY3 ASSAYVALUE ppm 2 645.00 580.00 to 709.00	No of Analyses: Minimum: Maximum: Mean: Std Deviation: % in Tolerance % Bias % RSD	148 634.00 656.00 641.00 4.23 100.00 % -0.62 % 0.66 %



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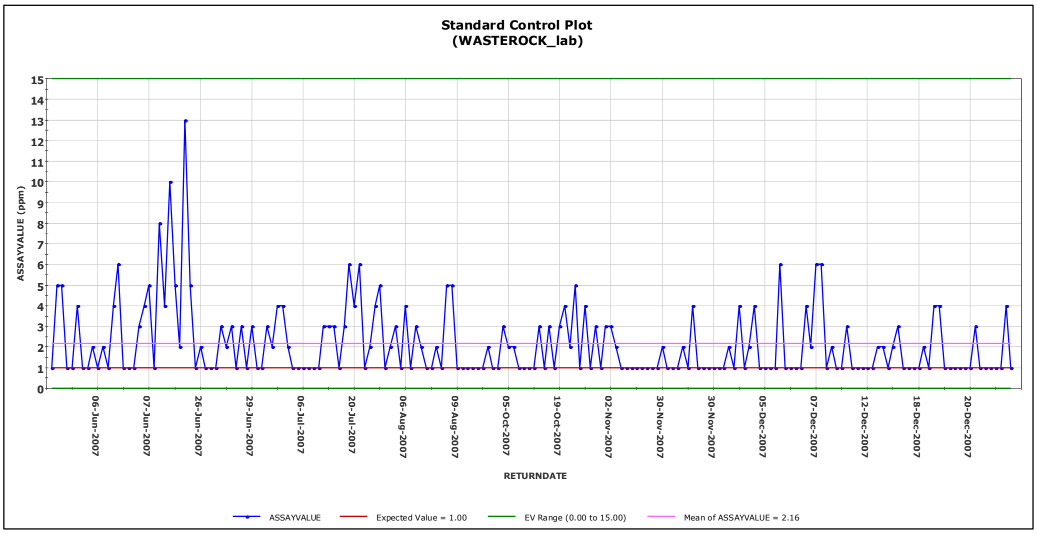
Lab Standard SGS Perth (WASTEROCK_lab)

Standard: Element: Units: Detection Limit: Expected Value (EV): E.V. Range:	WASTE ROCK ASSAYVALUE 10 1.00 0.00 to 15.00	No of Analyses: Minimum: Maximum: Mean: Std Deviation: % in Tolerance % Bias % RSD	188 1.00 13.00 2.16 1.77 100.00 % 115.96 % 82.12 %



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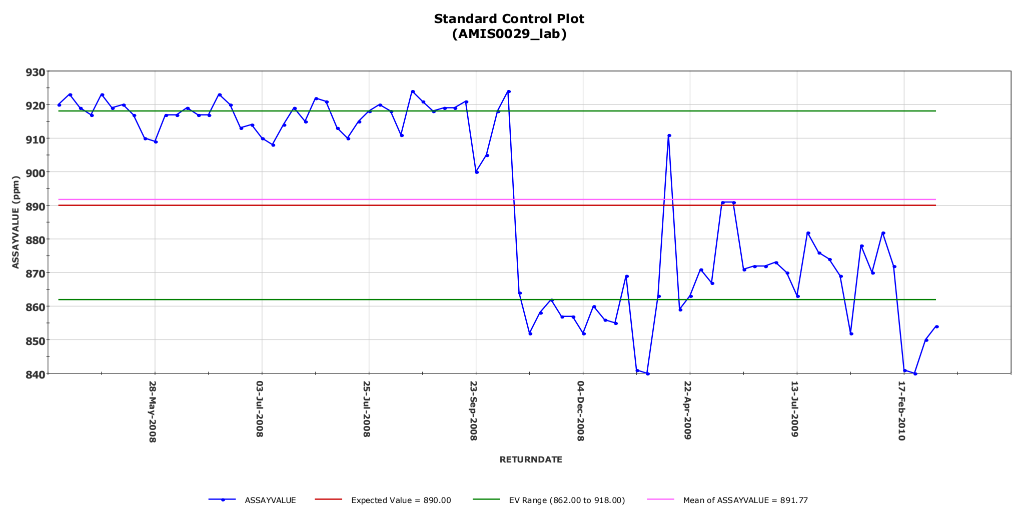
Appendix 1
QAQC Plots

Genalysis Perth Internal Standards

Lab Standard - Genalysis Perth (AMIS0029_lab)

Standard:	AMIS0029		No of Analyses:	83
Element:	ASSAYVALUE		Minimum:	840.00
Units:	ppm		Maximum:	924.00
Detection Limit:	5		Mean:	891.77
Expected Value (EV):	890.00		Std Deviation:	27.98
E.V. Range:	862.00 to 918.00		% in Tolerance	57.83 %
			% Bias	0.20 %
			% RSD	3.14 %



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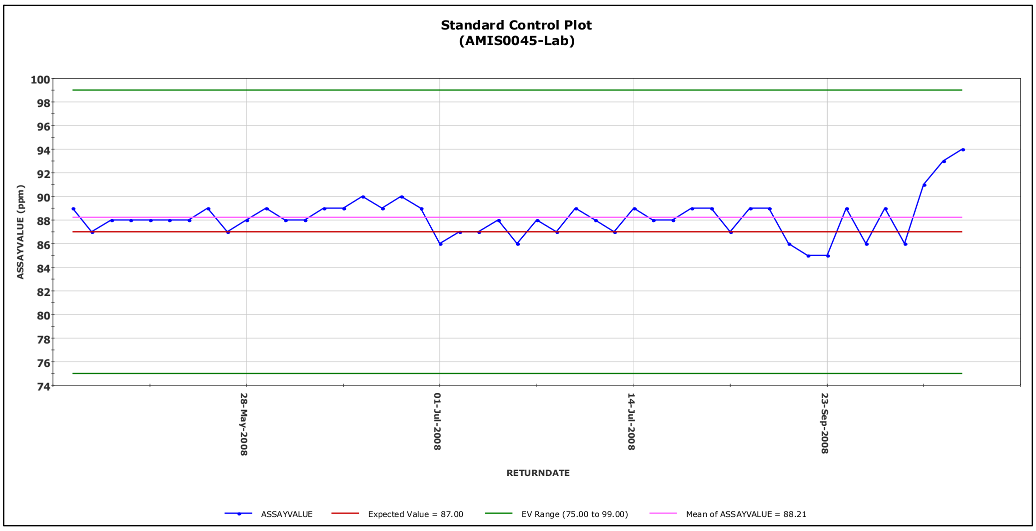
Lab Standard - Genalysis Perth (AMIS0045-Lab)

Standard:	AMIS0045		No of Analyses:	47
Element:	ASSAYVALUE		Minimum:	85.00
Units:	ppm		Maximum:	94.00
Detection Limit:	5		Mean:	88.21
Expected Value (EV):	87.00		Std Deviation:	1.70
E.V. Range:	75.00 to 99.00		% in Tolerance	100.00 %
			% Bias	1.39 %
			% RSD	1.93 %



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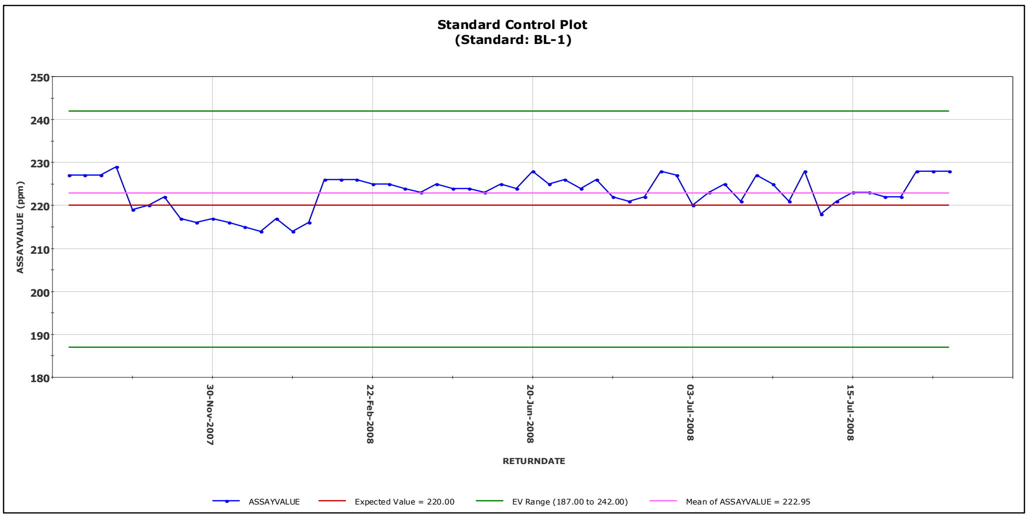
QAQC Plots

Lab Standard Genalysis Perth
(Standard: BL-1)

Standard:	BL-1	No of Analyses:	56
Element:	ASSAYVALUE	Minimum:	214.00
Units:	ppm	Maximum:	229.00
Detection Limit:	10	Mean:	222.95
Expected Value (EV):	220.00	Std Deviation:	4.02
E.V. Range:	187.00 to 242.00	% in Tolerance	100.00 %
		% Bias	1.34 %
		% RSD	1.80 %



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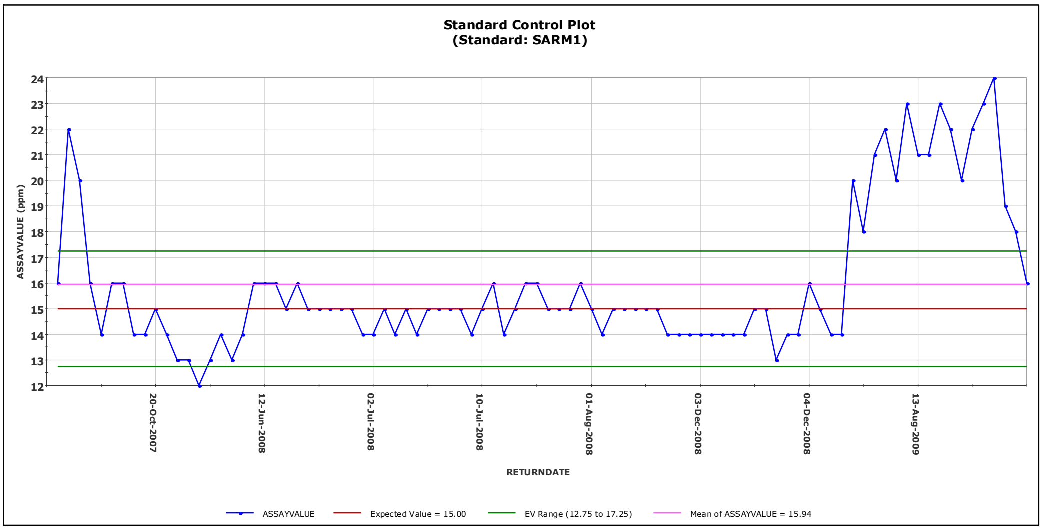
Lab Standard Genalysis Perth
(Standard: SARM1)

Standard:	SARM1	No of Analyses:	90
Element:	ASSAYVALUE	Minimum:	12.00
Units:		Maximum:	24.00
Detection Limit:	5	Mean:	15.94
Expected Value (EV):	15.00	Std Deviation:	2.79
E.V. Range:	12.75 to 17.25	% in Tolerance	78.89 %
		% Bias	6.30 %
		% RSD	17.48 %

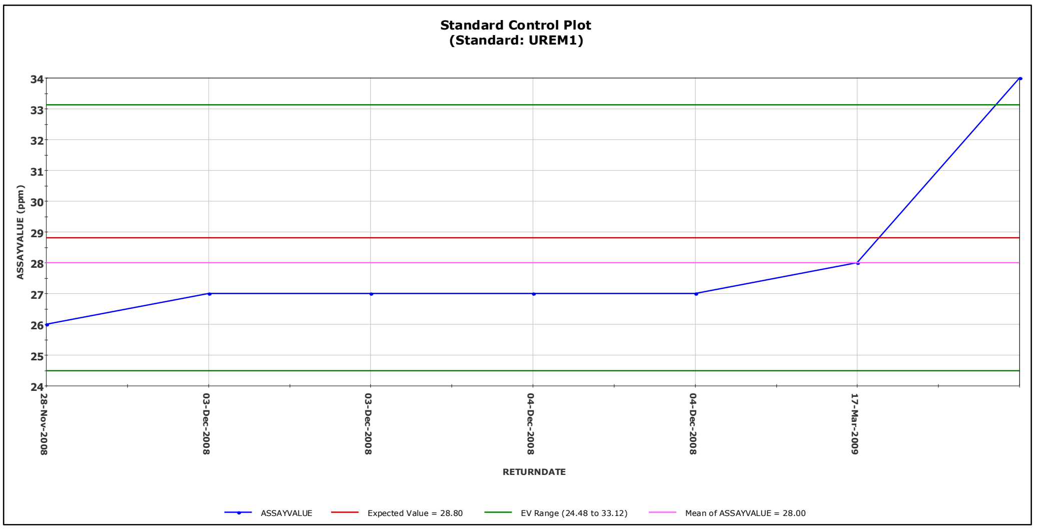


Printed: 10-Dec-2010 11:40:06 Data Refreshed: 10-Dec-2010 10:57:21 Page 1

Appendix 1
QAQC Plots

Lab Standard - Genalysis Perth
(Standard: UREM1)

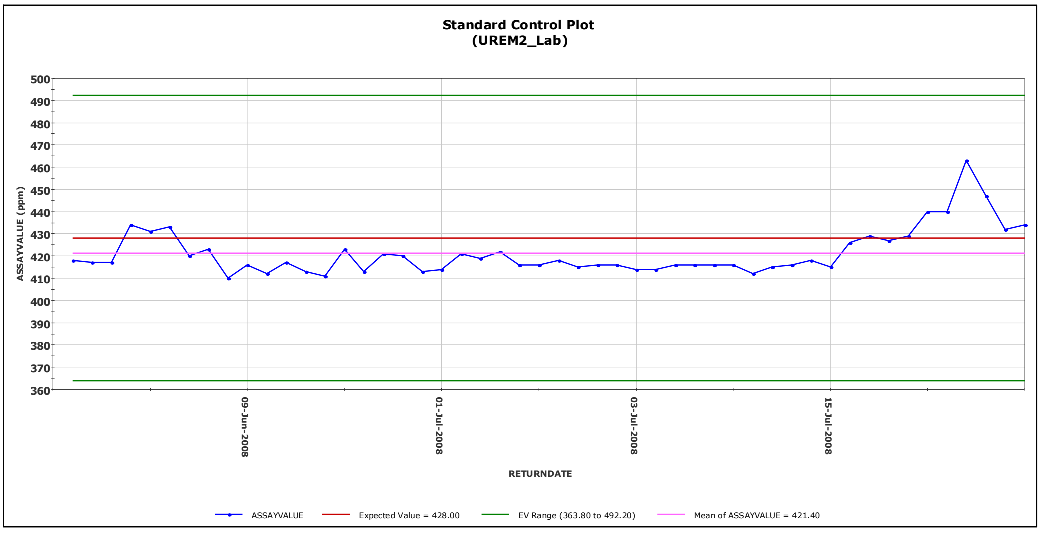
Standard:	UREM1	No of Analyses:	7
Element:	ASSAYVALUE	Minimum:	26.00
Units:	ppm	Maximum:	34.00
Detection Limit:	5	Mean:	28.00
Expected Value (EV):	28.80	Std Deviation:	2.51
E.V. Range:	24.48 to 33.12	% in Tolerance	85.71 %
		% Bias	-2.78 %
		% RSD	8.95 %



Printed: 09-Dec-2010 13:54:05 Data Imported: 08-Dec-2010 13:54:24 Page 1

Lab Standard - Genalysis Perth
(UREM2_Lab)

Standard:	UREM2	No of Analyses:	50
Element:	ASSAYVALUE	Minimum:	410.00
Units:	ppm	Maximum:	463.00
Detection Limit:	5	Mean:	421.40
Expected Value (EV):	428.00	Std Deviation:	10.21
E.V. Range:	363.80 to 492.20	% in Tolerance	100.00 %
		% Bias	-1.54 %
		% RSD	2.42 %

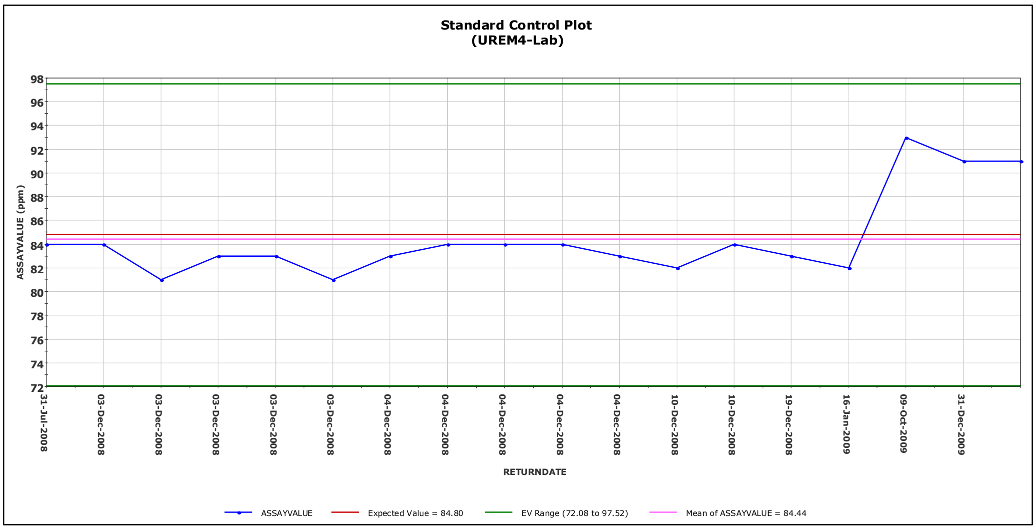


Printed: 09-Dec-2010 14:11:49 Data Imported: 08-Dec-2010 13:54:24 Page 1

Appendix 1
QAQC Plots

Lab Standard - Genalysis Perth
(UREM4-Lab)

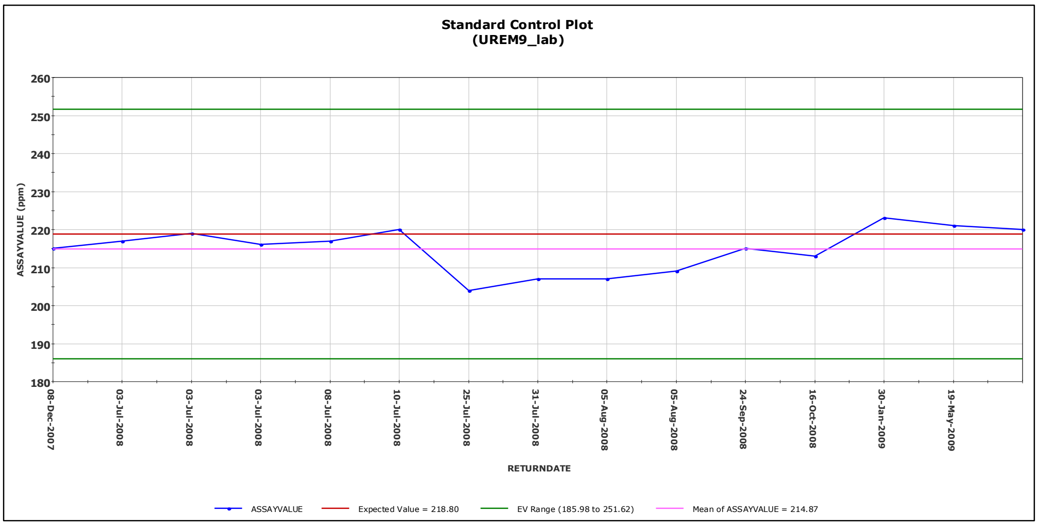
Standard:	UREM4		No of Analyses:	18
Element:	ASSAYVALUE		Minimum:	81.00
Units:	ppm		Maximum:	93.00
Detection Limit:	10		Mean:	84.44
Expected Value (EV):	84.80		Std Deviation:	3.39
E.V. Range:	72.08 to 97.52		% in Tolerance	100.00 %
			% Bias	-0.42 %
			% RSD	4.01 %



Printed: 09-Dec-2010 14:26:32 Data Imported: 08-Dec-2010 13:54:24 Page 1

Lab Standard - Genalysis Perth
(UREM9_lab)

Standard:	UREM9		No of Analyses:	15
Element:	ASSAYVALUE		Minimum:	204.00
Units:	ppm		Maximum:	223.00
Detection Limit:	10		Mean:	214.87
Expected Value (EV):	218.80		Std Deviation:	5.56
E.V. Range:	185.98 to 251.62		% in Tolerance	100.00 %
			% Bias	-1.80 %
			% RSD	2.59 %

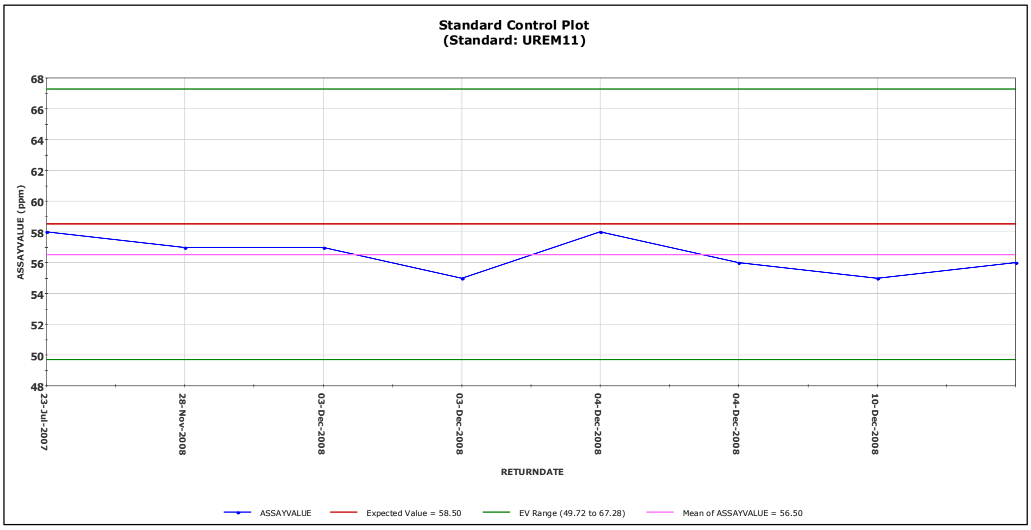


Printed: 09-Dec-2010 14:36:46 Data Imported: 08-Dec-2010 13:54:24 Page 1

Appendix 1
QAQC Plots

Lab Standard - Genalysis Perth
(Standard: UREM11)

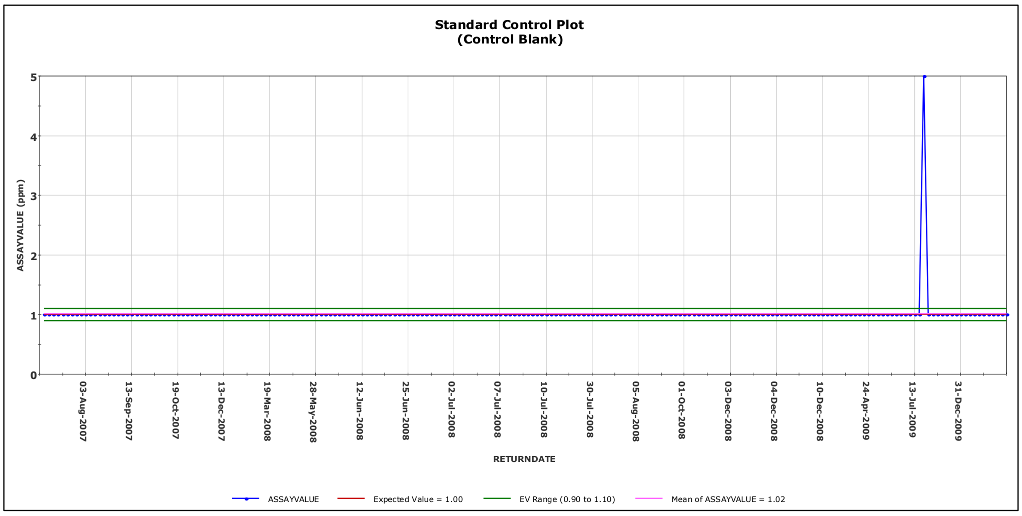
Standard: Element: Units: Detection Limit: Expected Value (EV): E.V. Range:	UREM11 ASSAYVALUE ppm 5 58.50 49.72 to 67.28		No of Analyses: Minimum: Maximum: Mean: Std Deviation: % in Tolerance % Bias % RSD	8 55.00 58.00 56.50 1.12 100.00 % -3.42 % 1.98 %



Printed: 09-Dec-2010 14:47:48 Data Imported: 08-Dec-2010 13:54:24 Page 1

Lab Standard Genalysis Perth
(Control Blank)

Standard: Element: Units: Detection Limit: Expected Value (EV): E.V. Range:	CONTROL BLANK ASSAYVALUE ppm 1.00 0.90 to 1.10		No of Analyses: Minimum: Maximum: Mean: Std Deviation: % in Tolerance % Bias % RSD	210 1.00 5.00 1.02 0.28 99.52 % 1.90 % 27.02 %



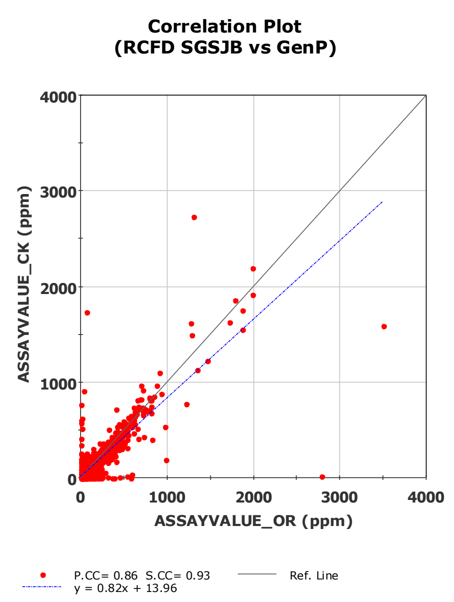
Printed: 17-Dec-2010 09:41:11 Data Refreshed: 17-Dec-2010 09:39:18 Page 1

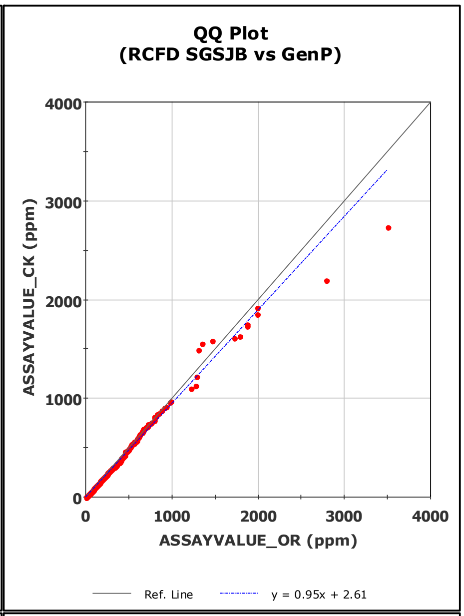
Appendix 1
QAQC Plots

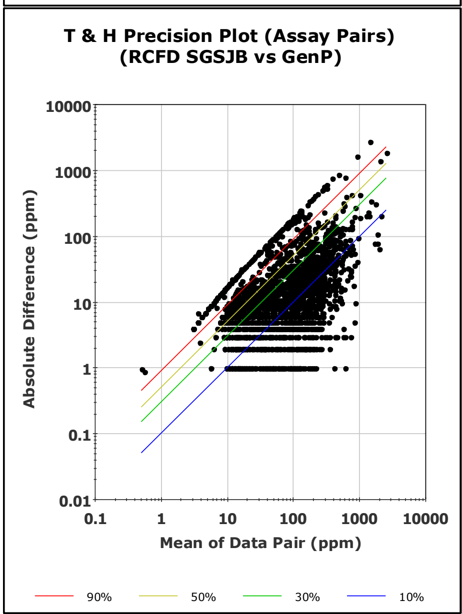
Duplicates Data

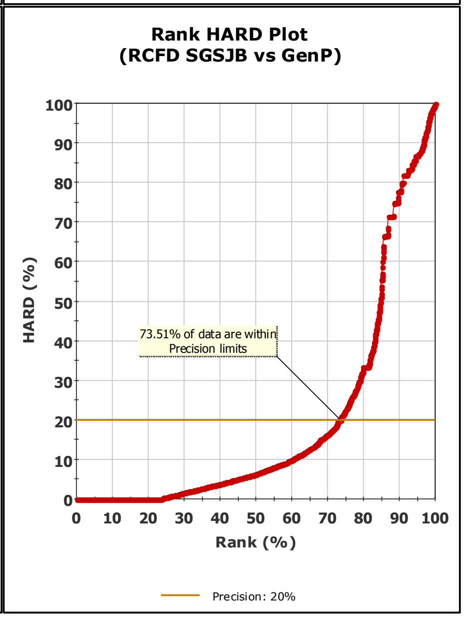
Summary
(RCFD SGSJB vs GenP)

	ASSAYVALUE_OR	ASSAYVALUE_CK	Units			Result
No. Pairs:	3,175	3,175			Pearson CC:	0.86
Minimum:	0.01	1.00	ppm		Spearman CC:	0.93
Maximum:	3,495.00	2,735.00	ppm		Mean HARD:	19.63
Mean:	91.23	88.95	ppm		Median HARD:	6.38
Median	27.00	25.00	ppm			
Std. Deviation:	177.86	169.17	ppm		Mean HRD:	-0.55
Coefficient of Variation:	1.95	1.90			Median HRD	0.00









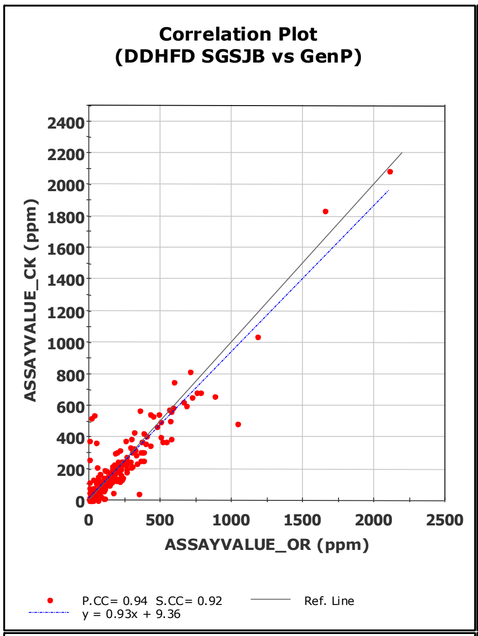
Printed: 14-Dec-2010 16:33:28 Data Refreshed: 14-Dec-2010 16:11:01 Page 1

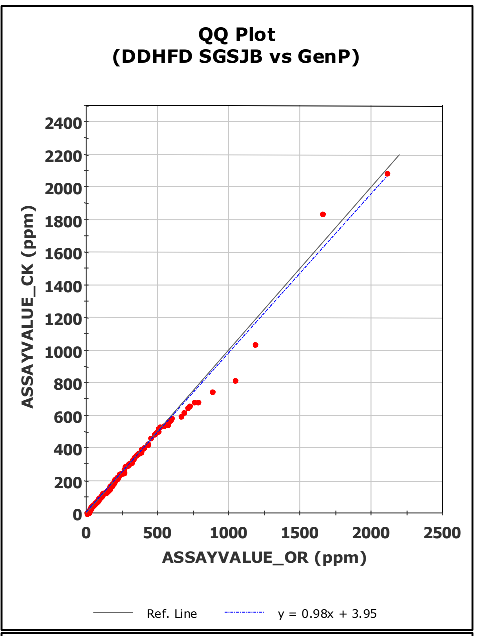
Duplicates Data

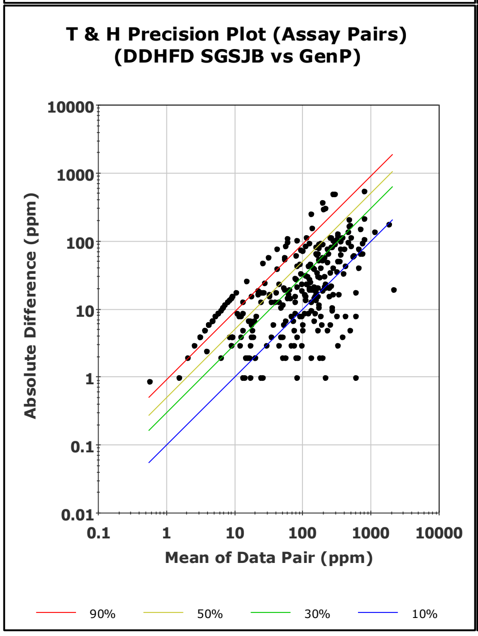
Appendix 1
QAQC Plots

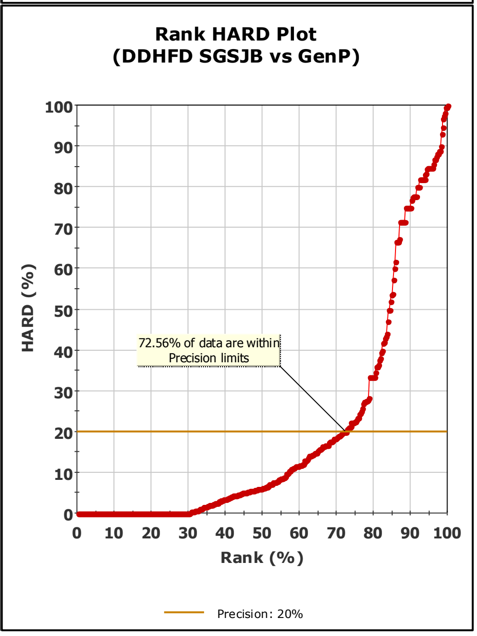
Summary
(DDHFD SGSJB vs GenP)

	ASSAYVALUE_OR	ASSAYVALUE_CK	Units			Result
No. Pairs:	430	430			Pearson CC:	0.94
Minimum:	0.10	1.00	ppm		Spearman CC:	0.92
Maximum:	2,106.00	2,086.00	ppm		Mean HARD:	19.68
Mean:	108.07	109.47	ppm		Median HARD:	6.25
Median	19.50	21.00	ppm			
Std. Deviation:	205.33	201.36	ppm		Mean HRD:	-4.46
Coefficient of Variation:	1.90	1.84			Median HRD	0.00









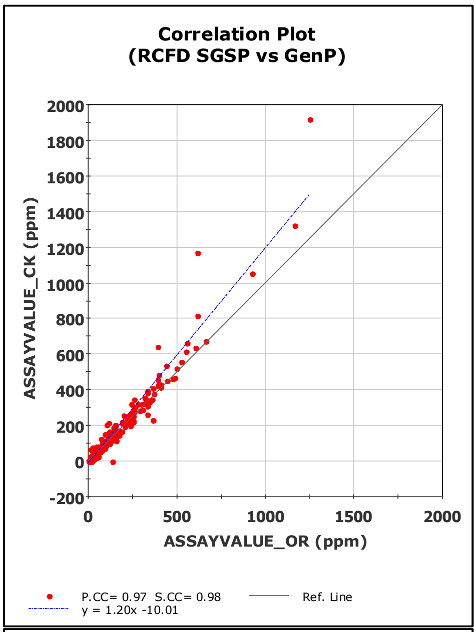
Printed: 14-Dec-2010 16:37:31 Data Refreshed: 14-Dec-2010 16:11:01 Page 1

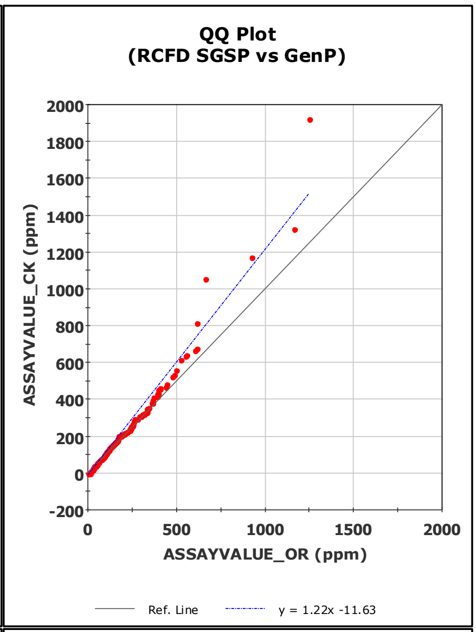
Duplicates Data

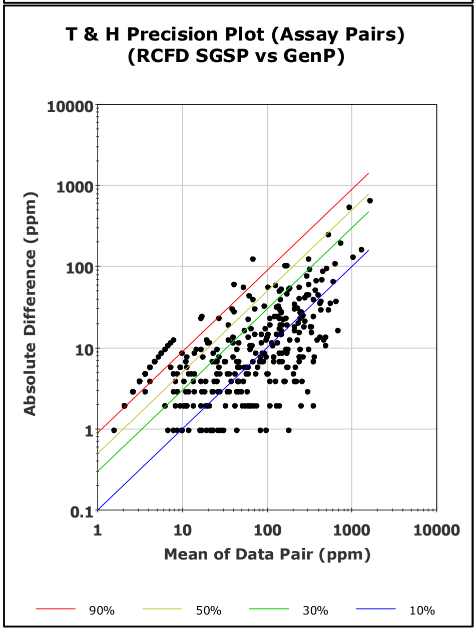
Appendix 1
QAQC Plots

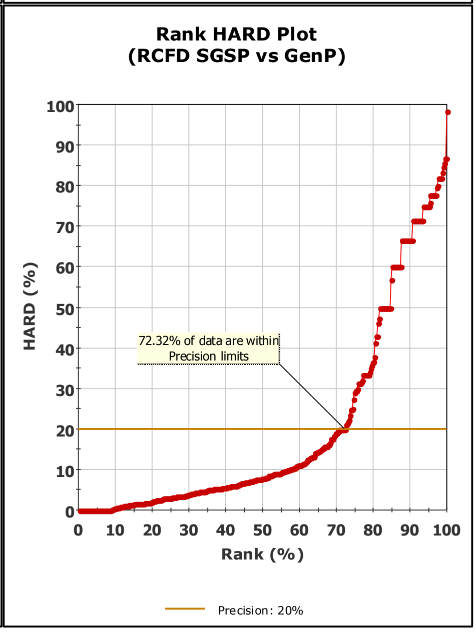
Summary
(RCFD SGSP vs GenP)

	ASSAYVALUE_OR	ASSAYVALUE_CK	Units			Result
No. Pairs:	401	401			Pearson CC:	0.97
Minimum:	1.00	1.00	ppm		Spearman CC:	0.98
Maximum:	1,250.00	1,923.00	ppm		Mean HARD:	20.09
Mean:	99.33	109.58	ppm		Median HARD:	7.87
Median	37.00	40.00	ppm			
Std. Deviation:	152.17	187.99	ppm		Mean HRD:	10.42
Coefficient of Variation:	1.53	1.72			Median HRD	0.81









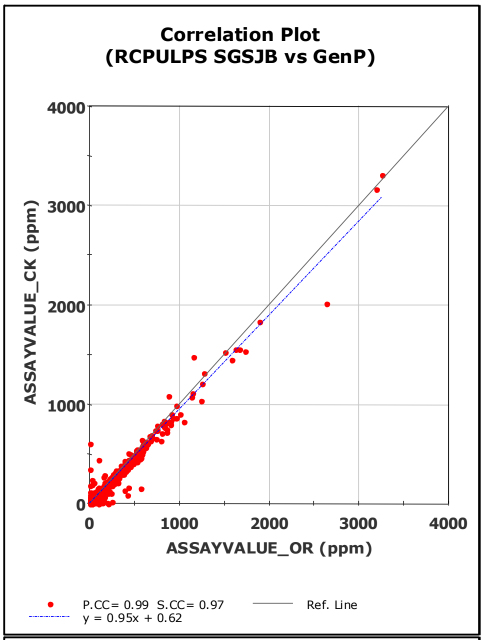
Printed: 14-Dec-2010 16:35:43 Data Refreshed: 14-Dec-2010 16:11:01 Page 1

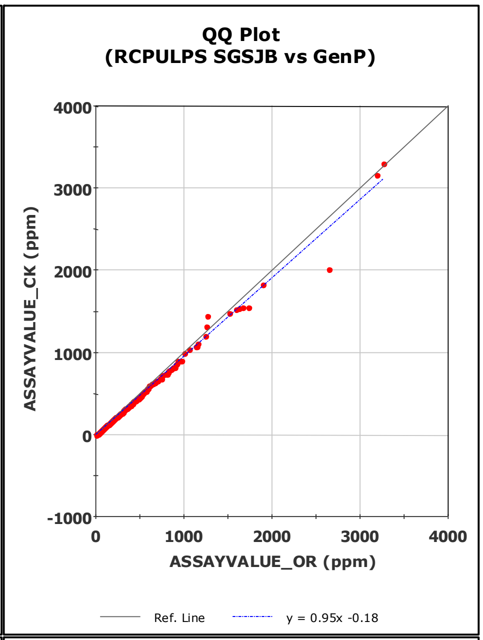
Duplicates Data

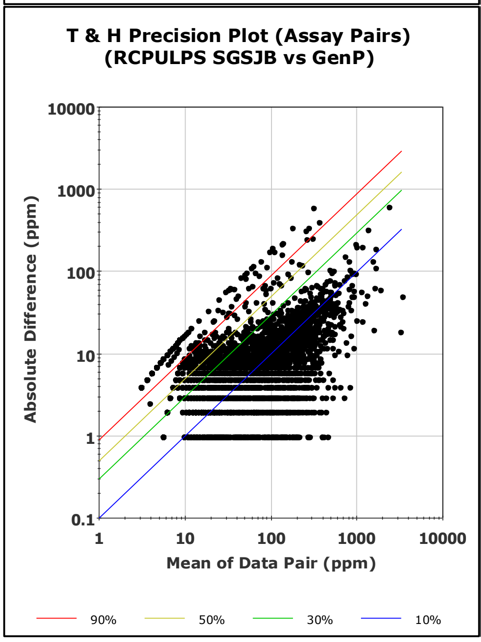
Appendix 1
QAQC Plots

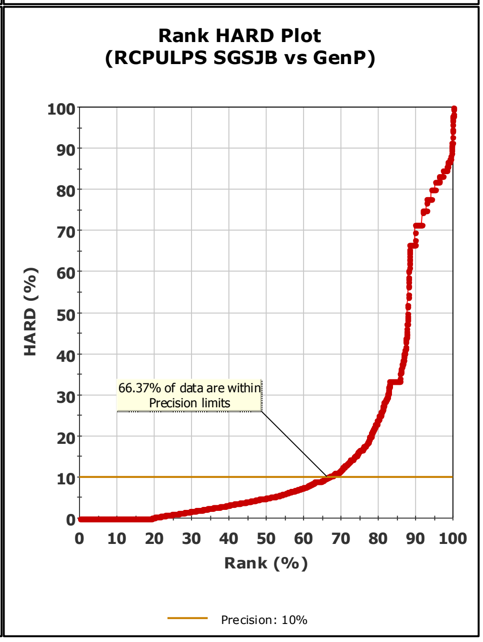
Summary
(RCPULPS SGSJB vs GenP)

	ASSAYVALUE_OR	ASSAYVALUE_CK	Units			Result
No. Pairs:	4,606	4,606			Pearson CC:	0.99
Minimum:	0.01	1.00	ppm		Spearman CC:	0.97
Maximum:	3,258.00	3,309.00	ppm		Mean HARD:	16.17
Mean:	81.32	77.47	ppm		Median HARD:	5.03
Median	25.00	22.00	ppm			
Std. Deviation:	158.46	151.61	ppm		Mean HRD:	-0.84
Coefficient of Variation:	1.95	1.96			Median HRD	0.70









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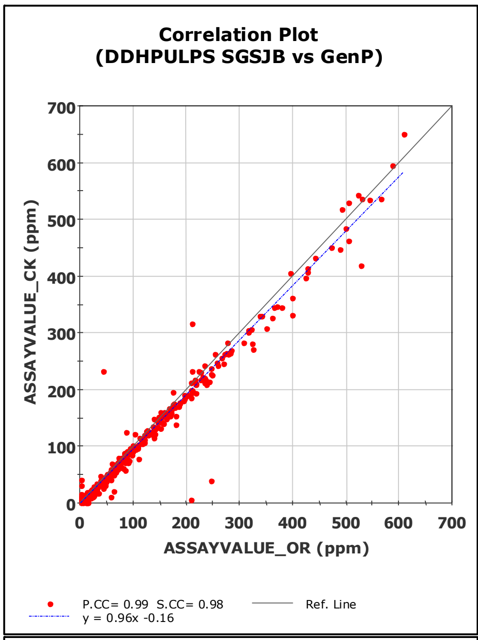
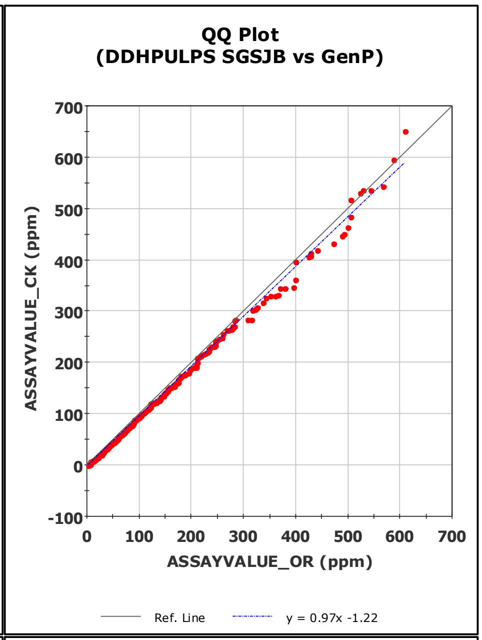
Duplicates Data

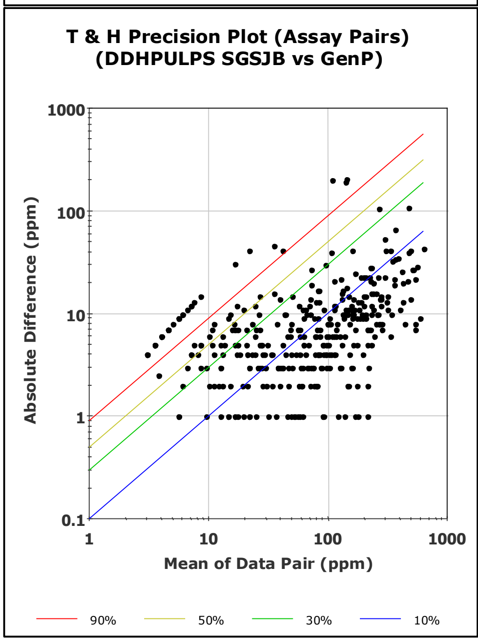
Appendix 1
QAQC Plots

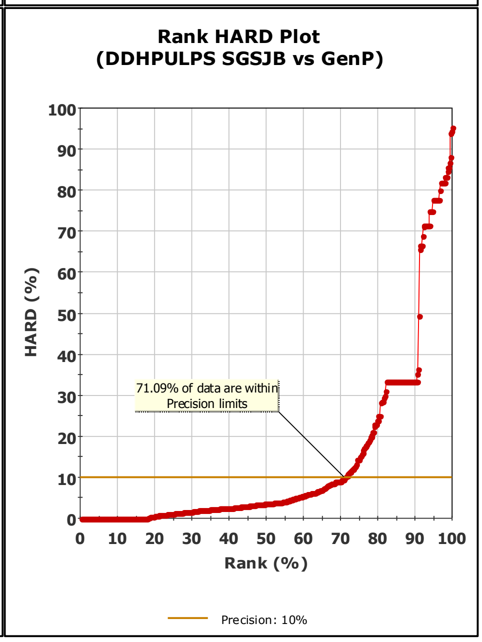
Summary
(DDHPULPS SGSJB vs GenP)

	ASSAYVALUE_OR	ASSAYVALUE_CK	Units			Result
No. Pairs:	512	512			Pearson CC:	0.99
Minimum:	1.00	1.00	ppm		Spearman CC:	0.98
Maximum:	609.00	652.00	ppm		Mean HARD:	14.03
Mean:	86.48	82.56	ppm		Median HARD:	3.67
Median	35.50	32.00	ppm			
Std. Deviation:	117.76	114.22	ppm		Mean HRD:	0.77
Coefficient of Variation:	1.36	1.38			Median HRD	2.09









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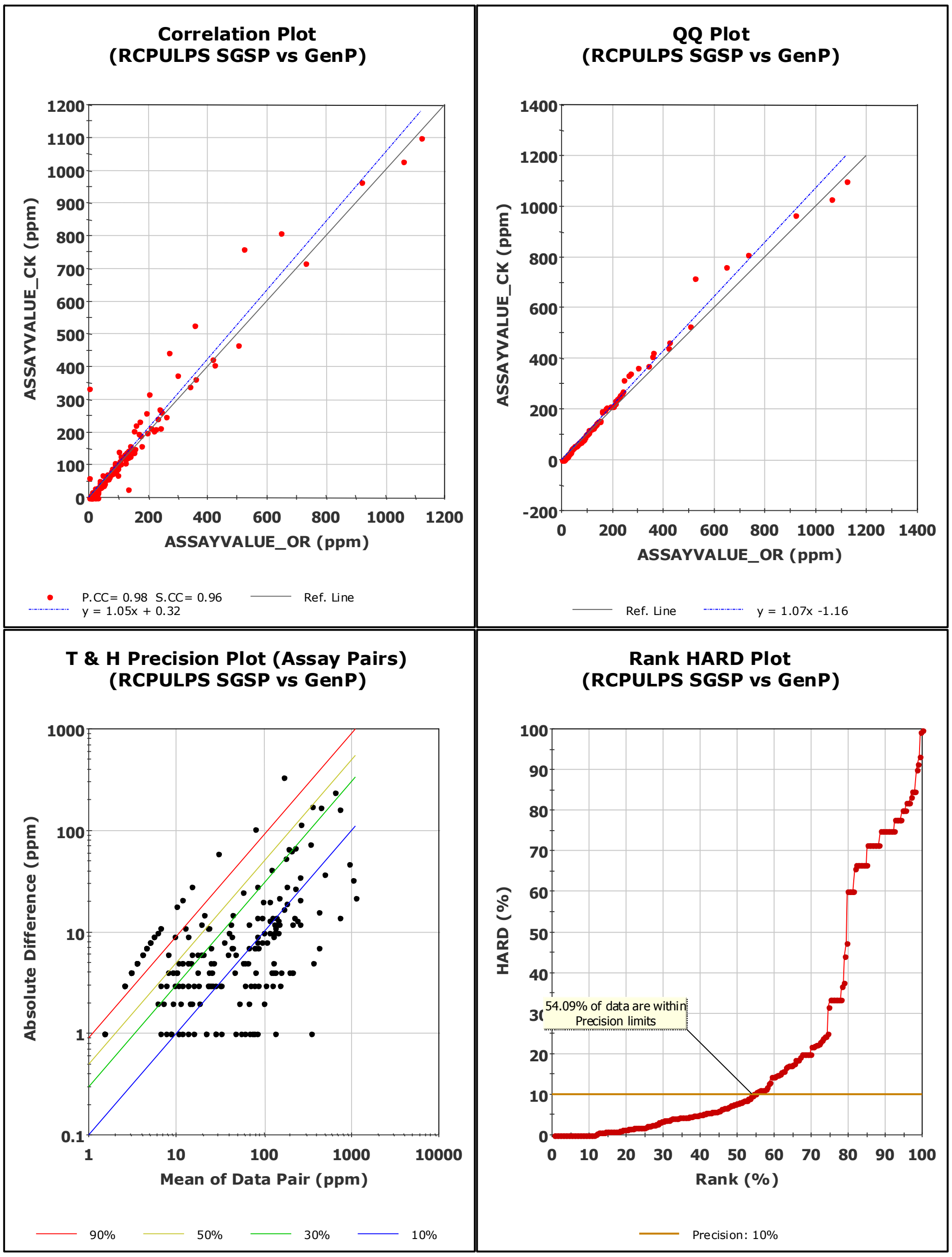
Duplicates Data

Appendix 1
QAQC Plots

Summary
(RCPULPS SGSP vs GenP)

	ASSAYVALUE_OR	ASSAYVALUE_CK	Units			Result
No. Pairs:	257	257			Pearson CC:	0.98
Minimum:	0.10	1.00	ppm		Spearman CC:	0.96
Maximum:	1,120.00	1,098.00	ppm		Mean HARD:	22.51
Mean:	75.46	79.84	ppm		Median HARD:	8.00
Median	20.00	17.00	ppm			
Std. Deviation:	146.52	158.19	ppm		Mean HRD:	16.21
Coefficient of Variation:	1.94	1.98			Median HRD	2.52



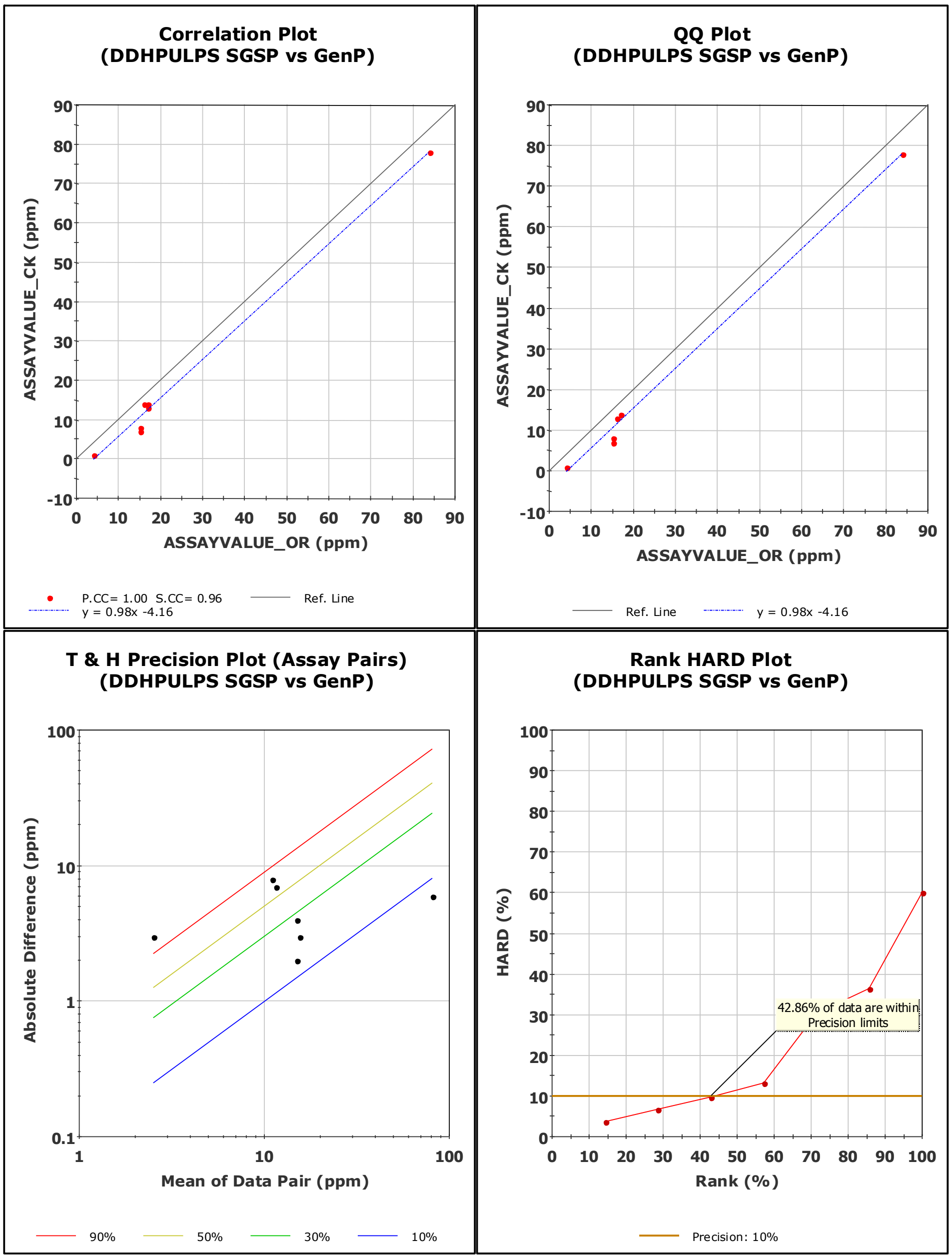
Printed: 14-Dec-2010 16:43:54 Data Refreshed: 14-Dec-2010 16:11:01 Page 1

Duplicates Data

Appendix 1
QAQC Plots

Summary
(DDHPULPS SGSP vs GenP)

	ASSAYVALUE_OR	ASSAYVALUE_CK	Units			Result
No. Pairs:	7	7			Pearson CC:	1.00
Minimum:	4.00	1.00	ppm		Spearman CC:	0.96
Maximum:	84.00	78.00	ppm		Mean HARD:	22.88
Mean:	24.00	19.29	ppm		Median HARD:	13.33
Median	16.00	13.00	ppm			
Std. Deviation:	24.85	24.36	ppm		Mean HRD:	22.88
Coefficient of Variation:	1.04	1.26			Median HRD	13.33



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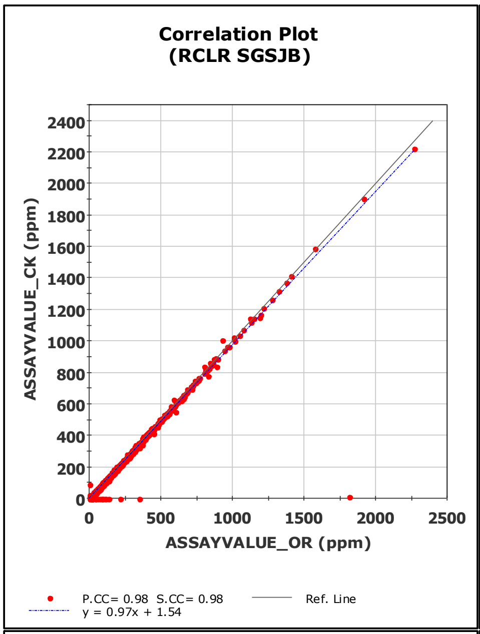
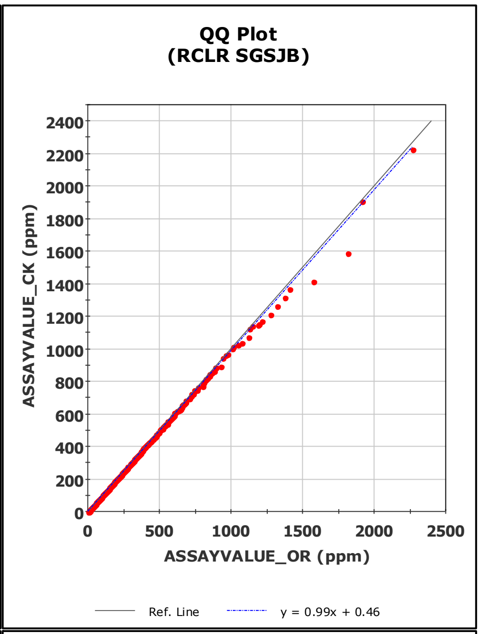
Duplicates Data

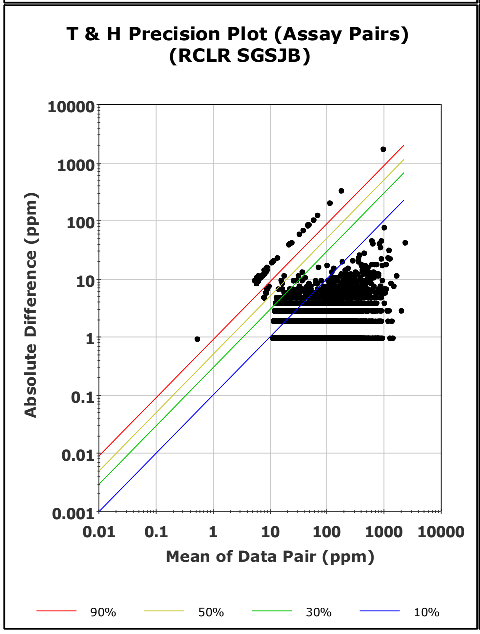
Appendix 1
QAQC Plots

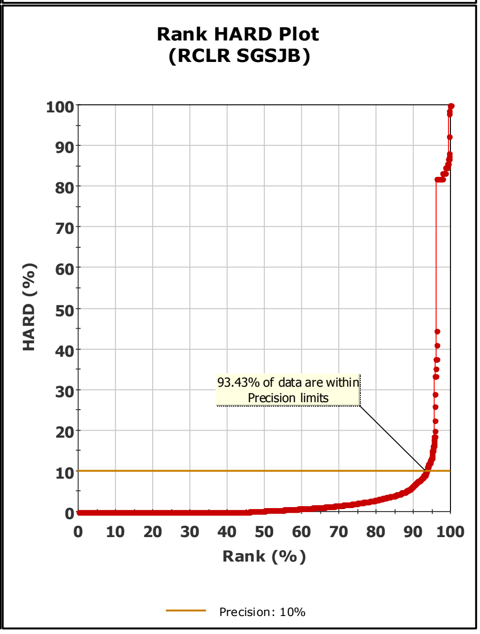
Summary
(RCLR SGSJB)

	ASSAYVALUE_OR	ASSAYVALUE_CK	Units			Result
No. Pairs:	6,243	6,243			Pearson CC:	0.98
Minimum:	0.01	0.01	ppm		Spearman CC:	0.98
Maximum:	2,269.00	2,225.00	ppm		Mean HARD:	4.89
Mean:	73.87	73.33	ppm		Median HARD:	0.35
Median	20.00	20.00	ppm			
Std. Deviation:	137.90	136.08	ppm		Mean HRD:	0.43
Coefficient of Variation:	1.87	1.86			Median HRD	0.00









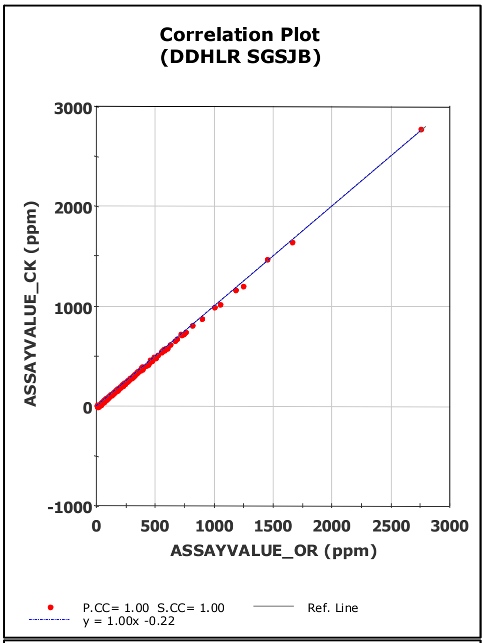
Printed: 15-Dec-2010 09:18:54   Data Imported: 10-Dec-2010 16:09:12   Page 1

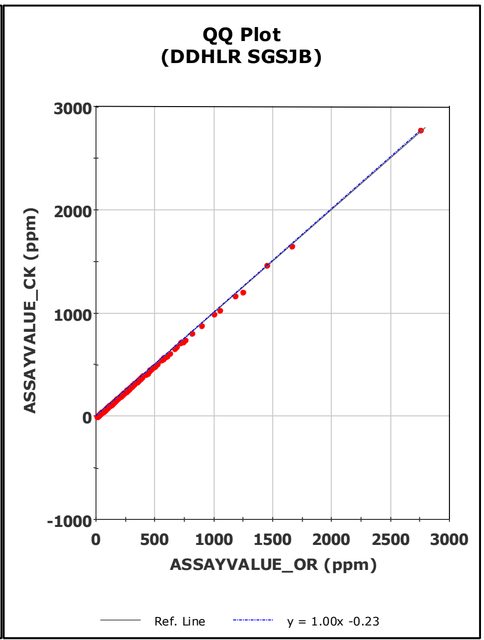
Duplicates Data

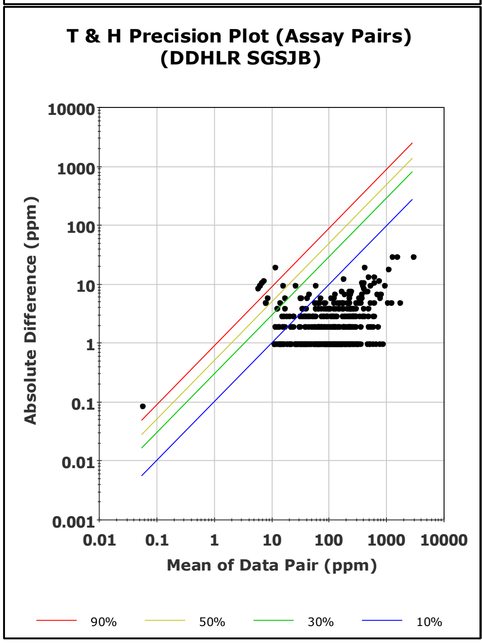
Appendix 1
QAQC Plots

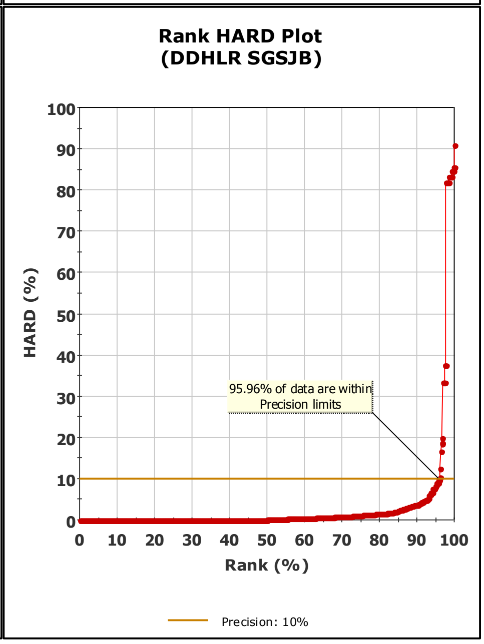
Summary
(DDHLR SGSJB)

	ASSAYVALUE_OR	ASSAYVALUE_CK	Units			Result
No. Pairs:	842	842			Pearson CC:	1.00
Minimum:	0.10	0.01	ppm		Spearman CC:	1.00
Maximum:	2,750.00	2,780.00	ppm		Mean HARD:	3.25
Mean:	101.99	101.95	ppm		Median HARD:	0.15
Median	31.00	31.00	ppm			
Std. Deviation:	190.61	190.99	ppm		Mean HRD:	0.20
Coefficient of Variation:	1.87	1.87			Median HRD	0.00









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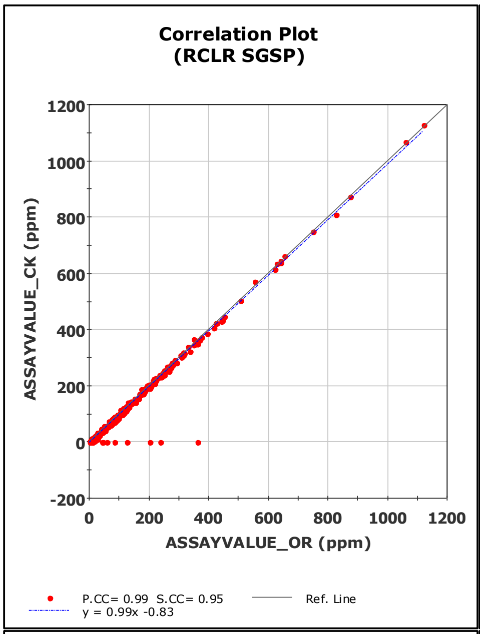
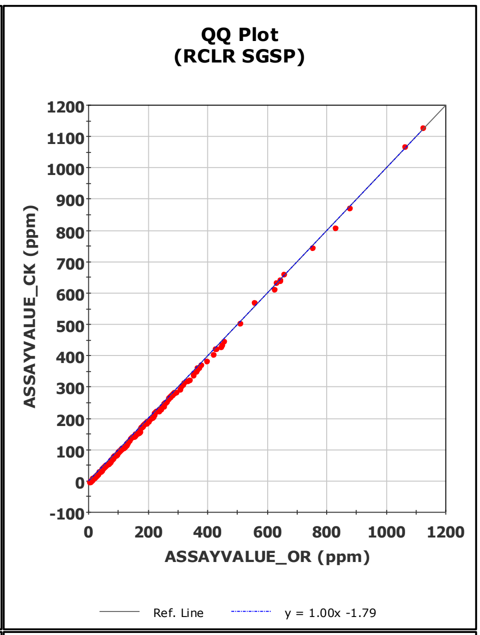
Duplicates Data

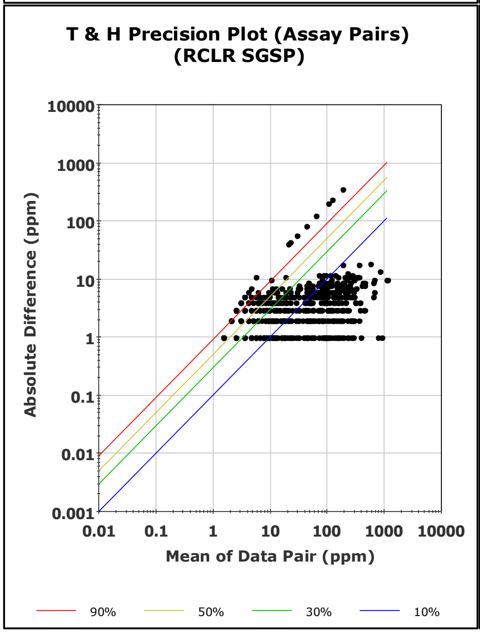
Appendix 1
QAQC Plots

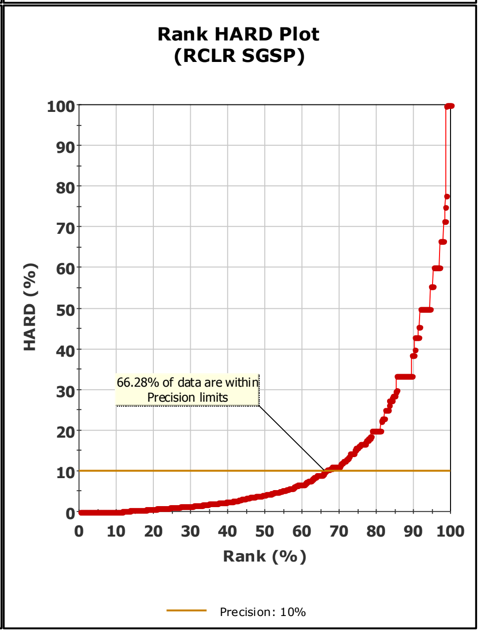
Summary
(RCLR SGSP)

	ASSAYVALUE_OR	ASSAYVALUE_CK	Units			Result
No. Pairs:	682	682			Pearson CC:	0.99
Minimum:	0.01	0.01	ppm		Spearman CC:	0.95
Maximum:	1,120.00	1,130.00	ppm		Mean HARD:	12.76
Mean:	80.49	78.78	ppm		Median HARD:	4.28
Median	26.00	22.00	ppm			
Std. Deviation:	128.26	128.42	ppm		Mean HRD:	4.03
Coefficient of Variation:	1.59	1.63			Median HRD	0.00









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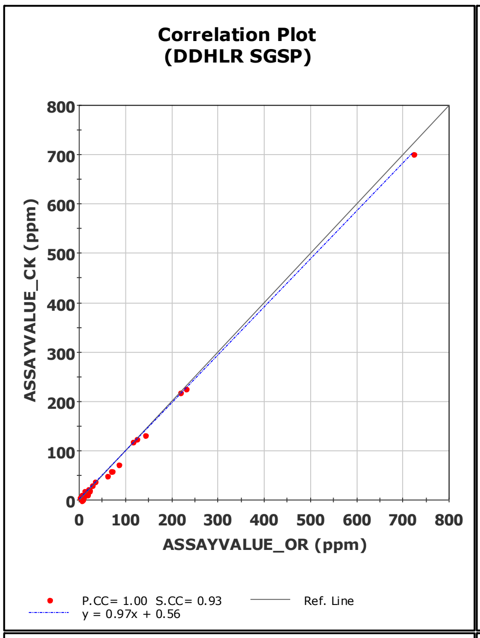
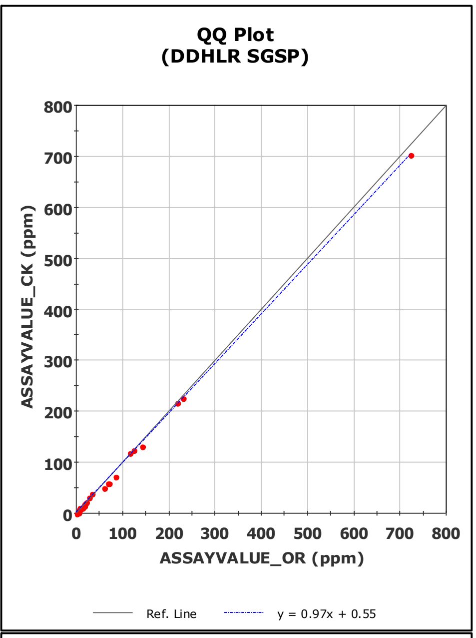
Duplicates Data

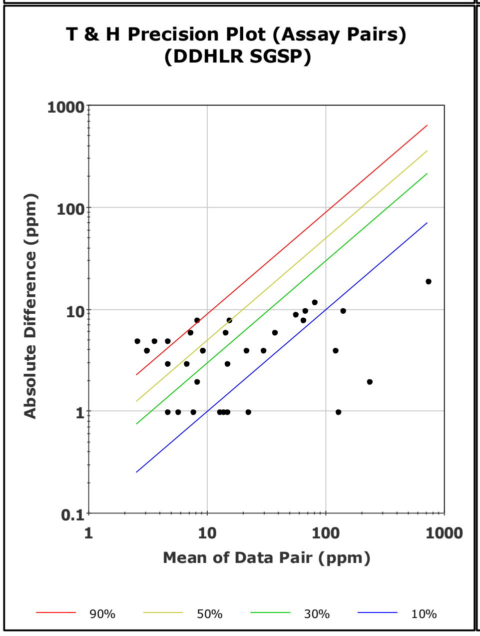
Appendix 1
QAQC Plots

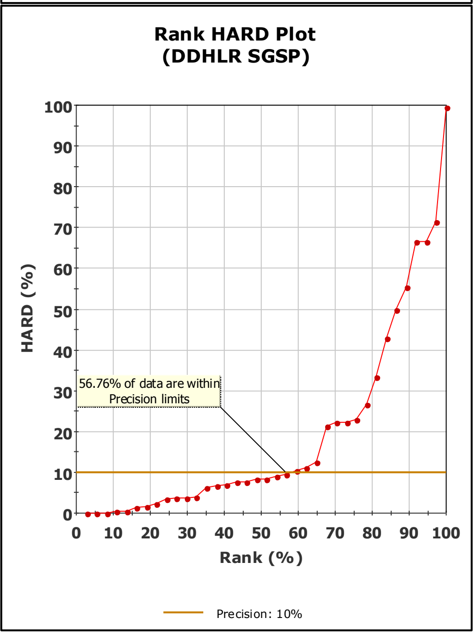
Summary
(DDHLR SGSP)

	ASSAYVALUE_OR	ASSAYVALUE_CK	Units			Result
No. Pairs:	37	37			Pearson CC:	1.00
Minimum:	1.00	0.01	ppm		Spearman CC:	0.93
Maximum:	722.00	703.00	ppm		Mean HARD:	19.65
Mean:	57.24	56.27	ppm		Median HARD:	8.33
Median	13.00	13.00	ppm			
Std. Deviation:	124.49	121.25	ppm		Mean HRD:	0.48
Coefficient of Variation:	2.17	2.15			Median HRD	0.00









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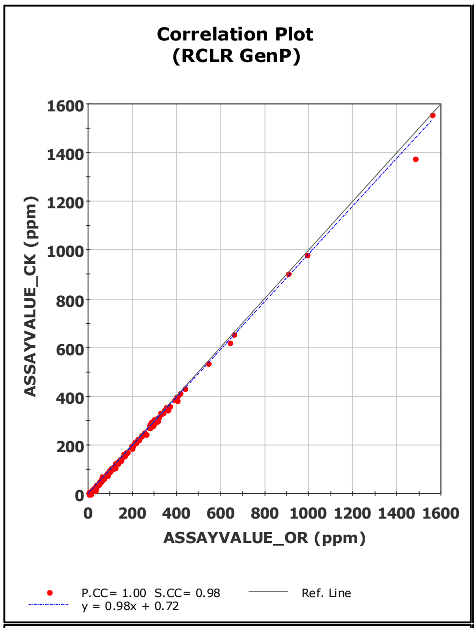
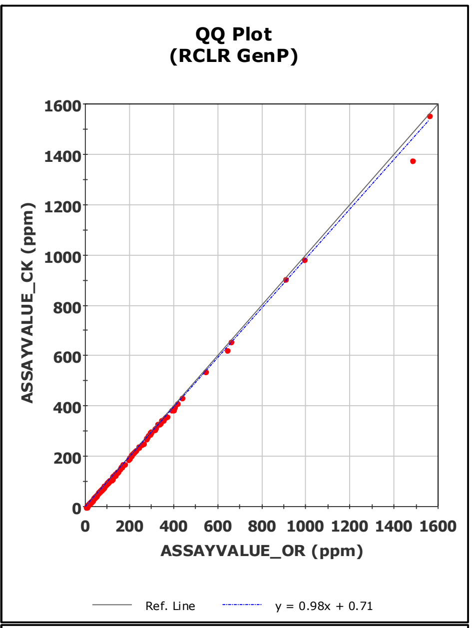
Duplicates Data

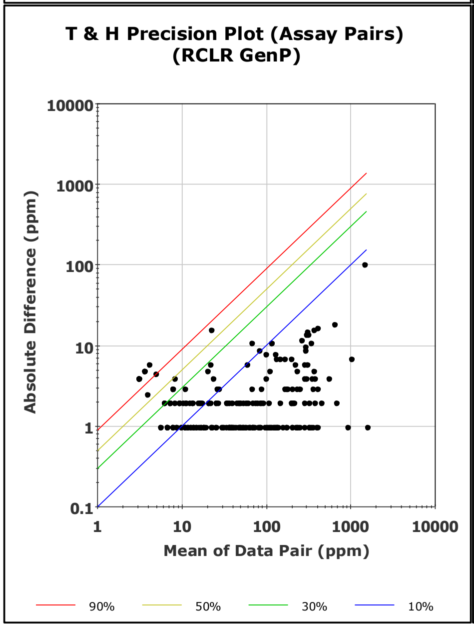
Appendix 1
QAQC Plots

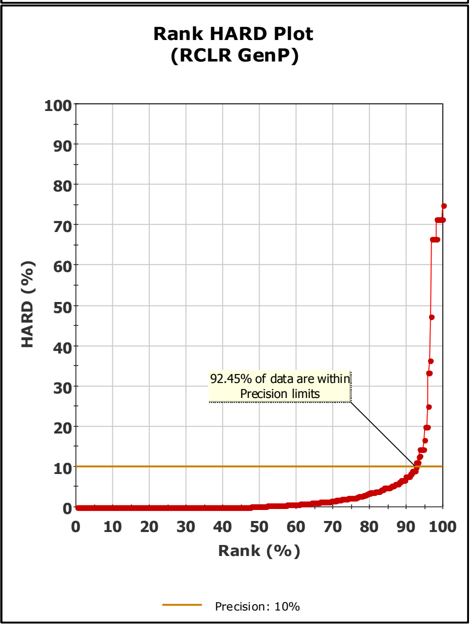
Summary
(RCLR GenP)

	ASSAYVALUE_OR	ASSAYVALUE_CK	Units			Result
No. Pairs:	477	477			Pearson CC:	1.00
Minimum:	1.00	1.00	ppm		Spearman CC:	0.98
Maximum:	1,556.00	1,557.00	ppm		Mean HARD:	4.30
Mean:	72.45	72.03	ppm		Median HARD:	0.23
Median	16.00	15.00	ppm			
Std. Deviation:	149.19	146.94	ppm		Mean HRD:	1.18
Coefficient of Variation:	2.06	2.04			Median HRD	0.00









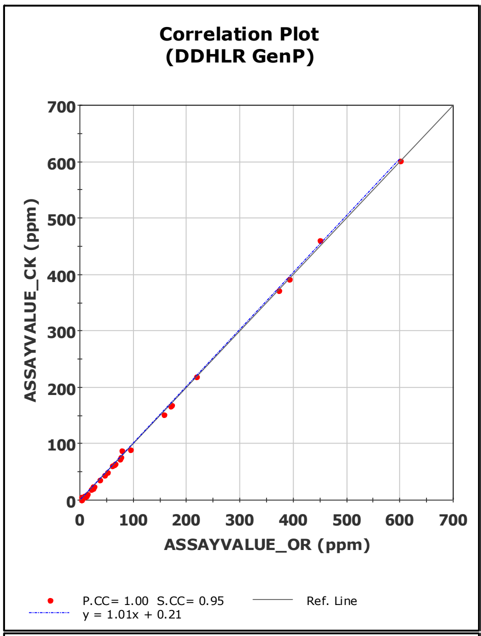
Printed: 15-Dec-2010 09:57:37 Data Imported: 10-Dec-2010 16:09:12 Page 1

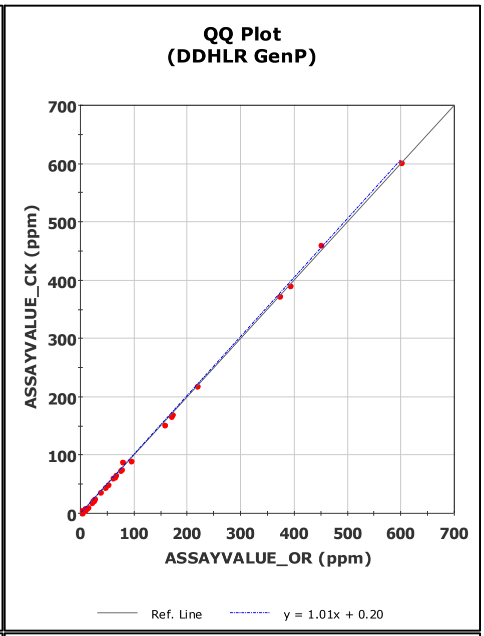
Duplicates Data

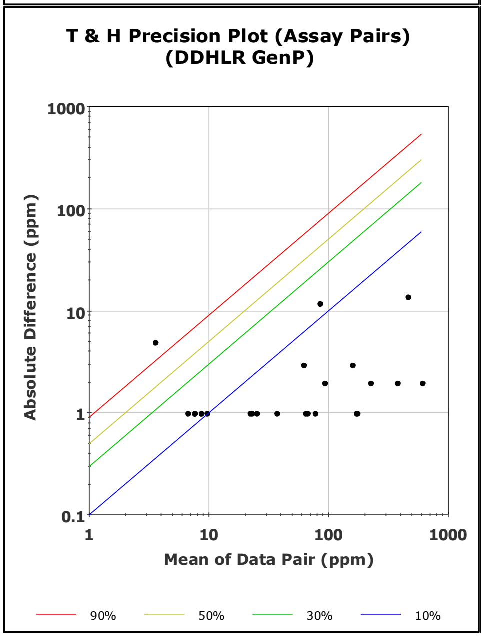
Appendix 1
QAQC Plots

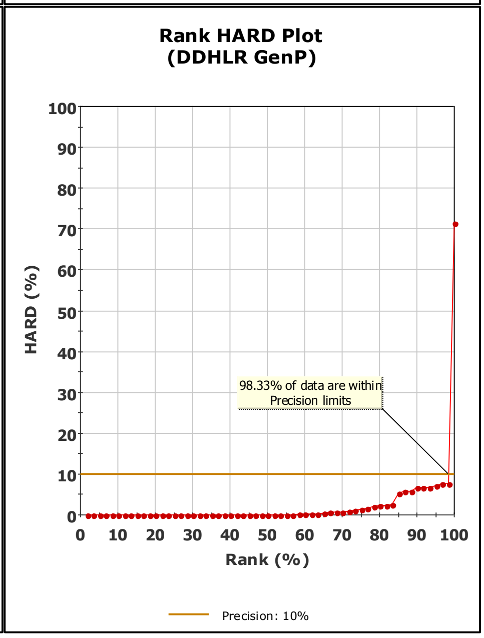
Summary
(DDHLR GenP)

	ASSAYVALUE_OR	ASSAYVALUE_CK	Units			Result
No. Pairs:	60	60			Pearson CC:	1.00
Minimum:	1.00	1.00	ppm		Spearman CC:	0.95
Maximum:	600.00	602.00	ppm		Mean HARD:	2.48
Mean:	56.98	57.64	ppm		Median HARD:	0.00
Median	8.00	7.50	ppm			
Std. Deviation:	117.92	118.89	ppm		Mean HRD:	-1.70
Coefficient of Variation:	2.07	2.06			Median HRD	0.00









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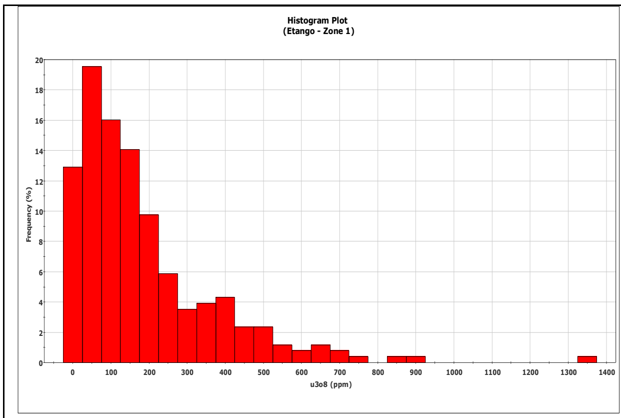
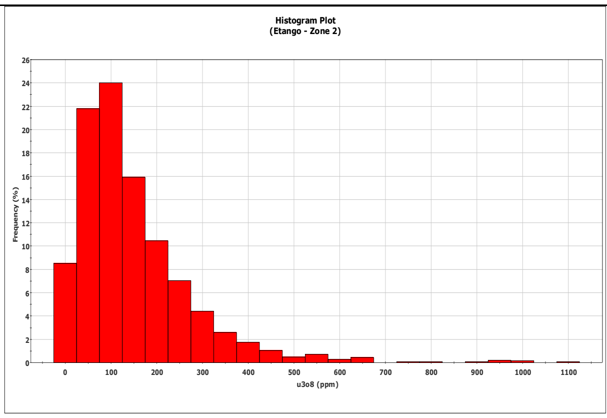
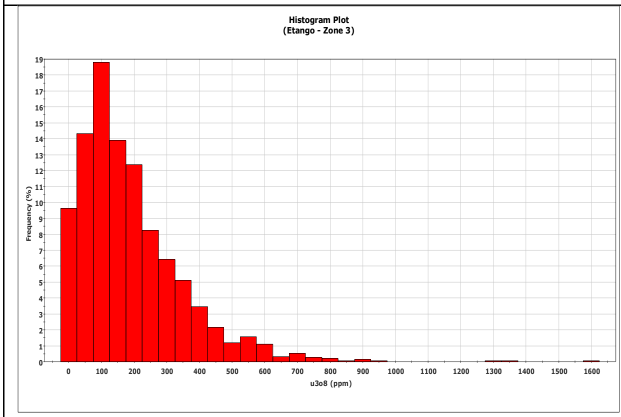
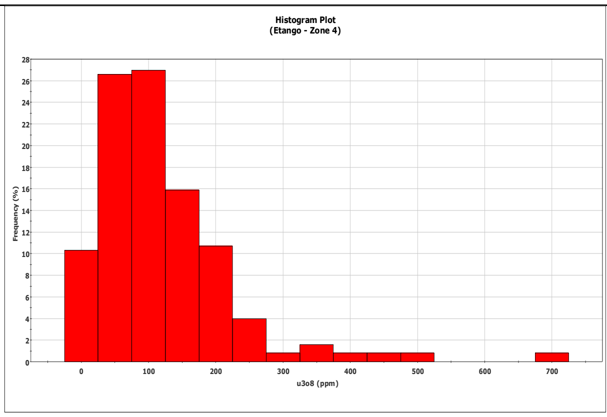
Appendix 2 Composite Statistics

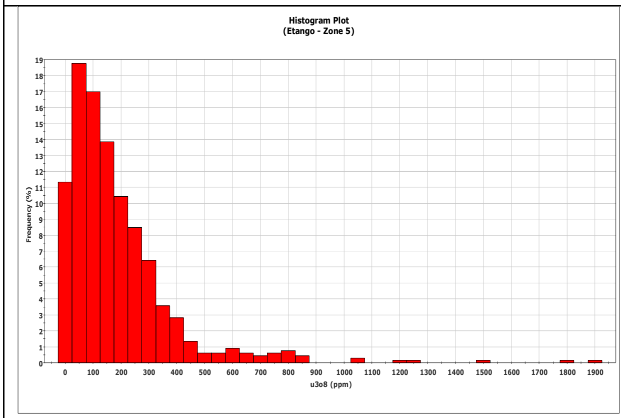
Appendix 2
Composite Statistics
Etango Project

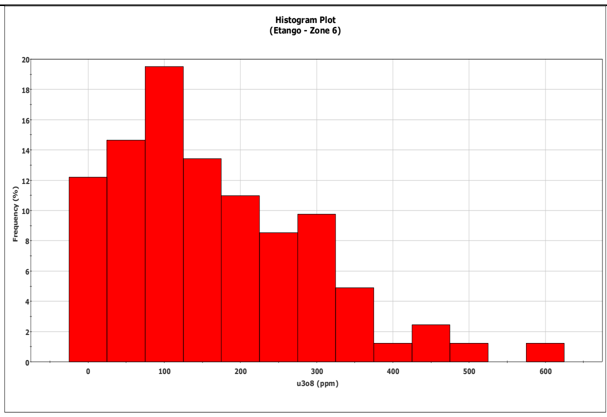


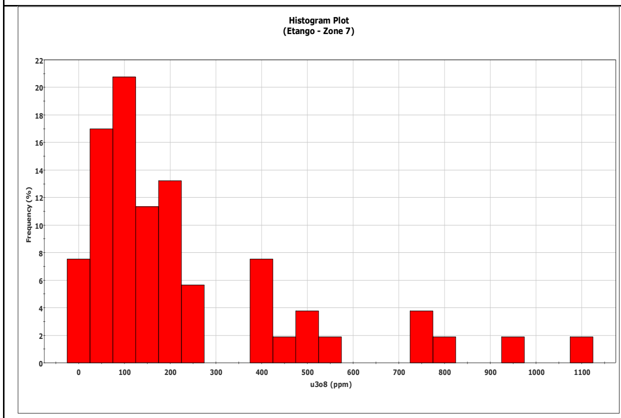


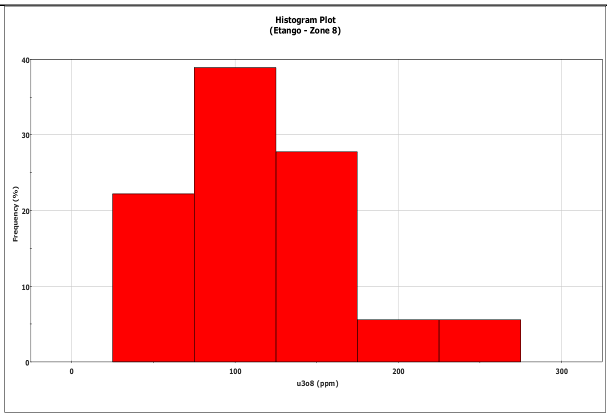




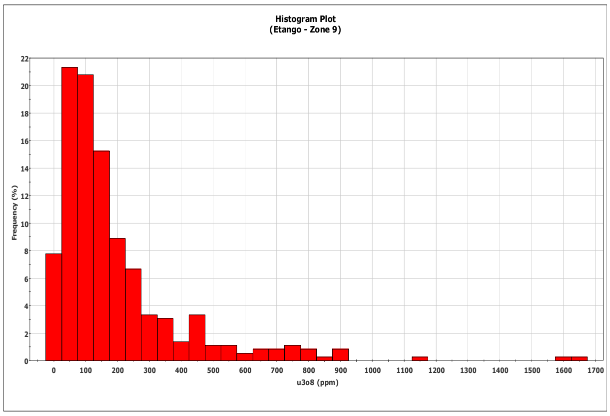


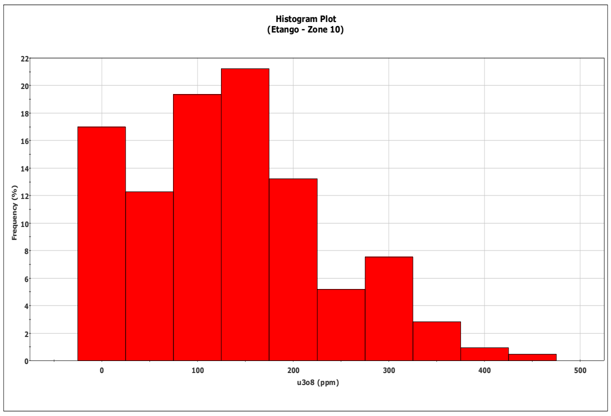


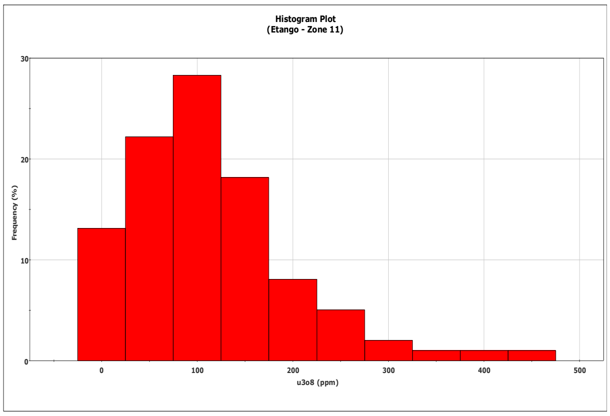


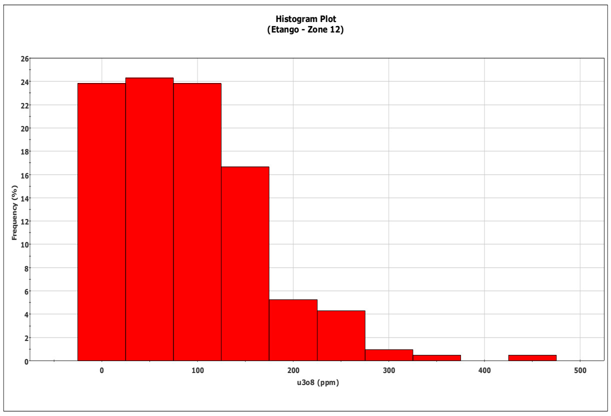


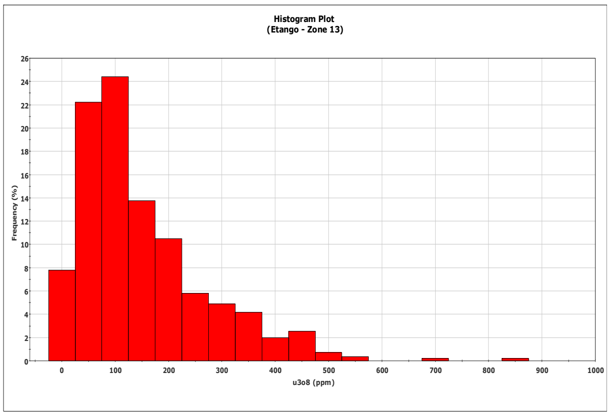
Appendix 2
Composite Statistics
Etango Project

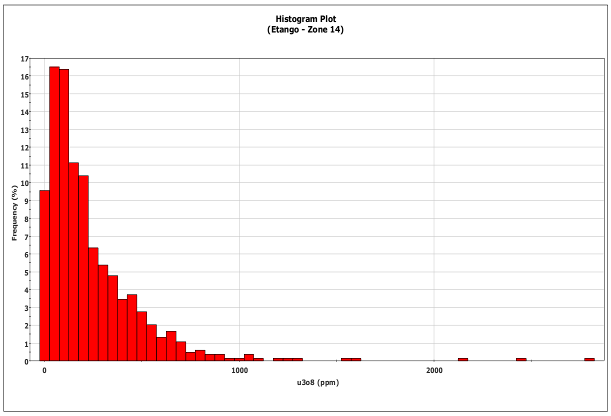


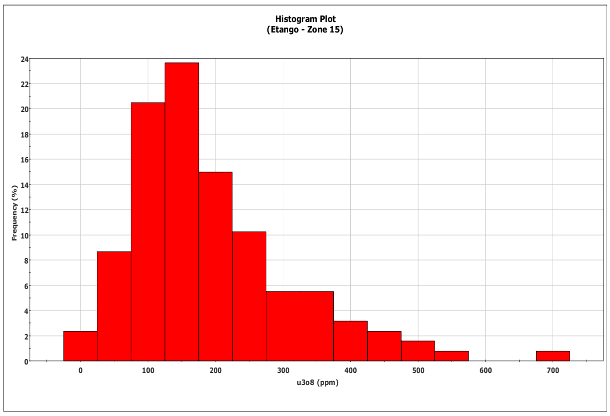


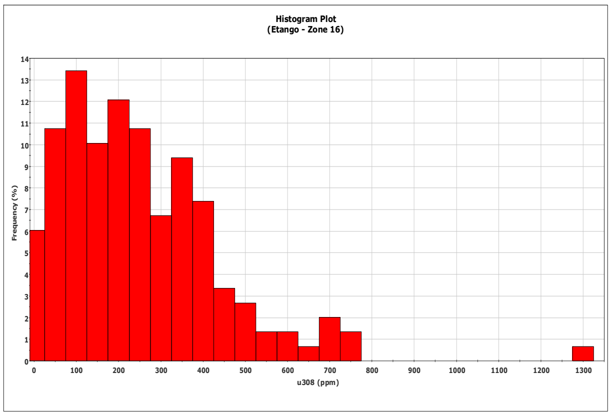




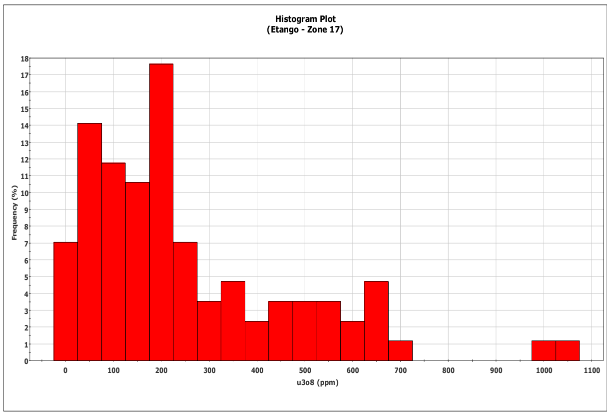


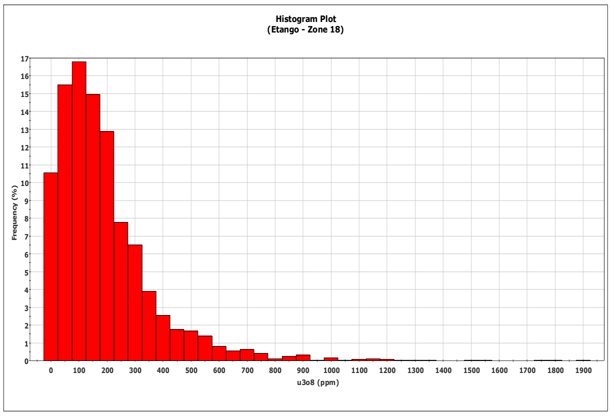


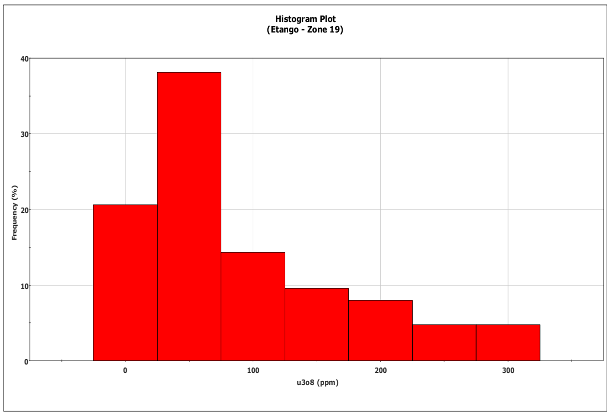


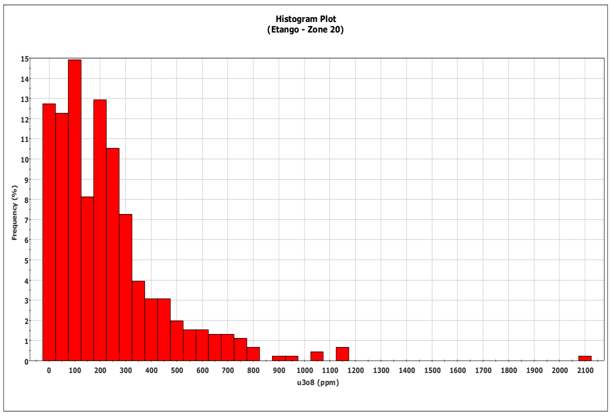


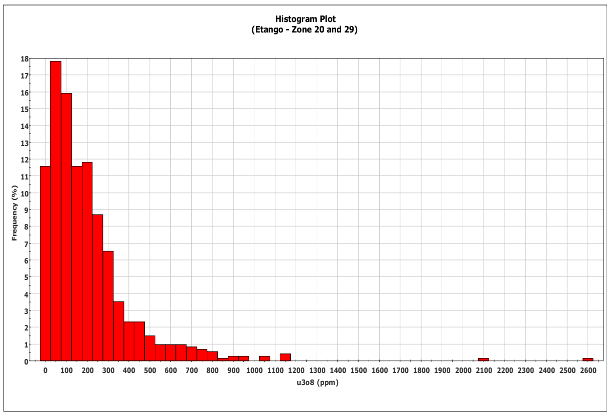
Appendix 2
Composite Statistics
Etango Project

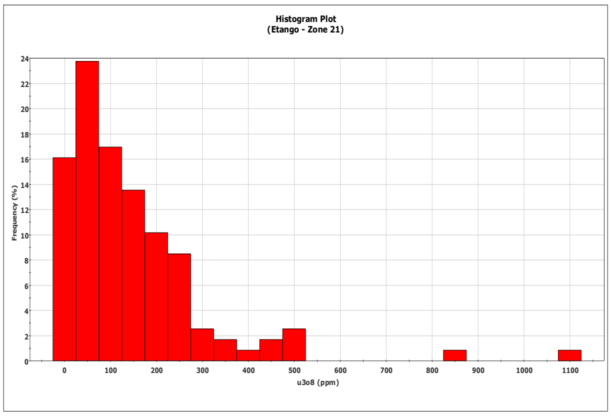


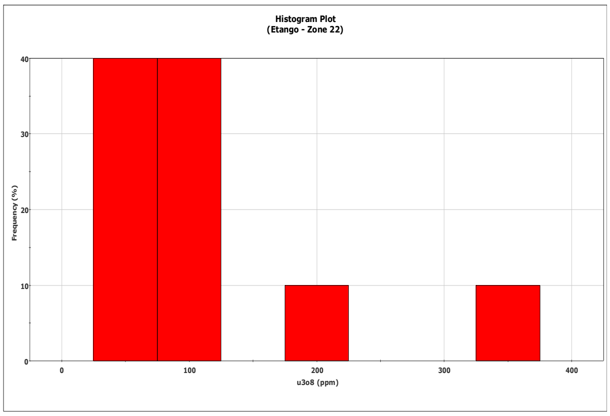


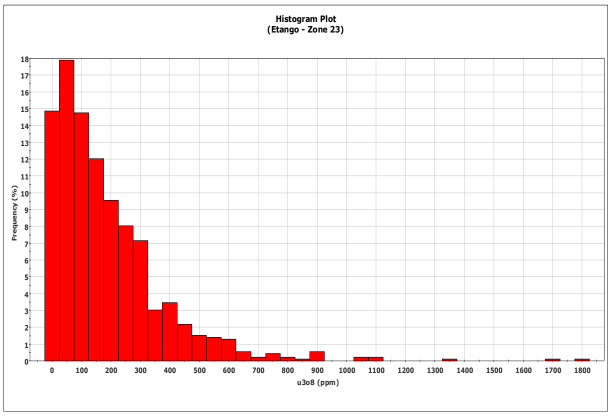




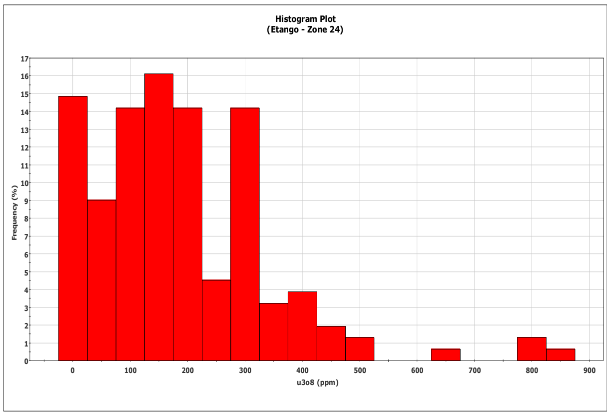


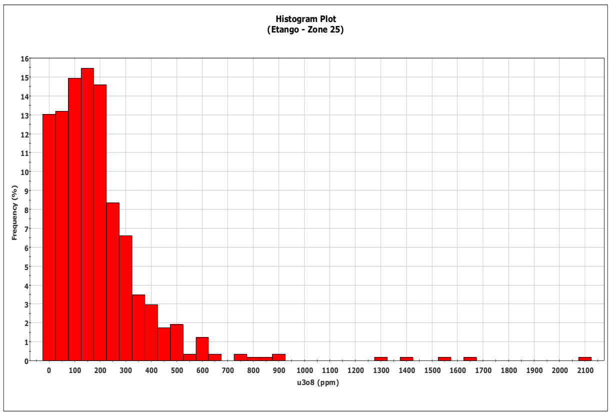


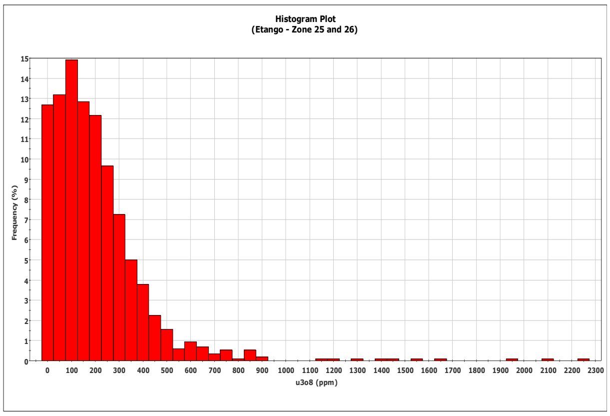


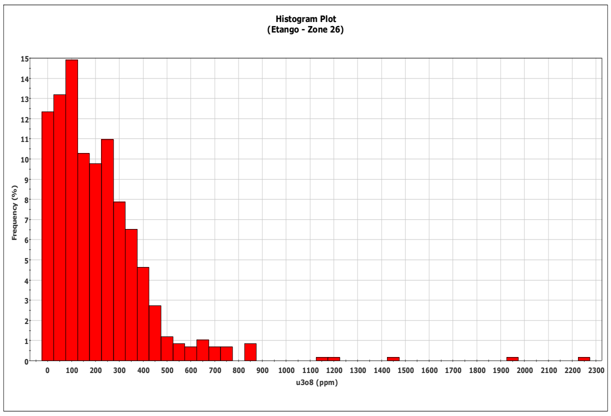


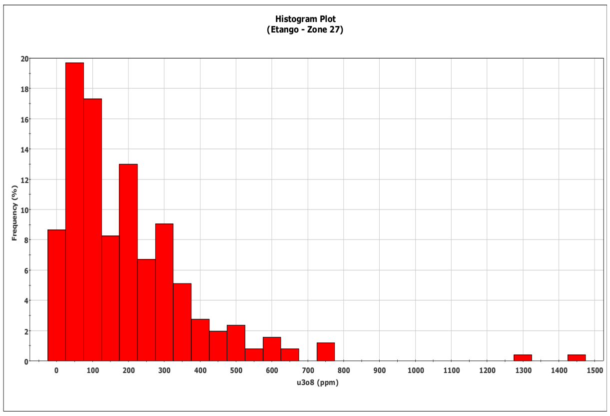
Appendix 2
Composite Statistics
Etango Project

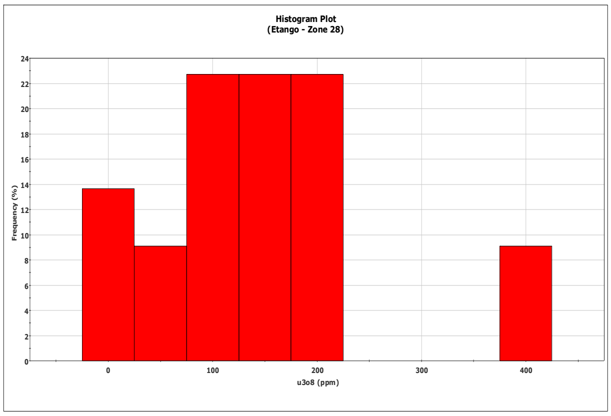


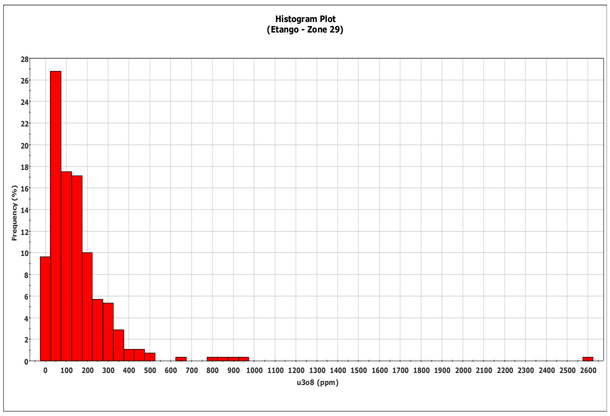


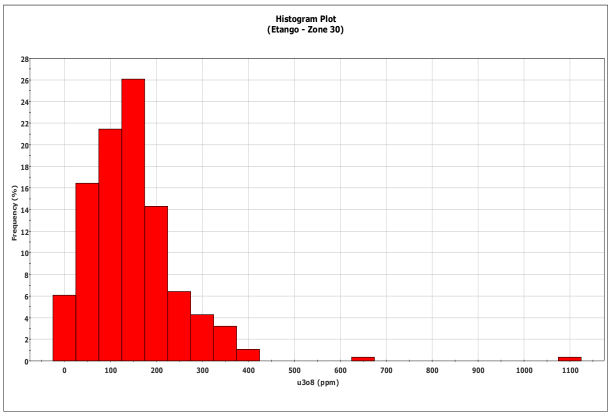




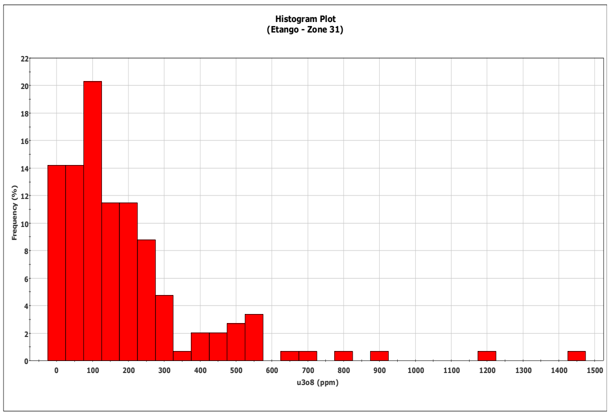


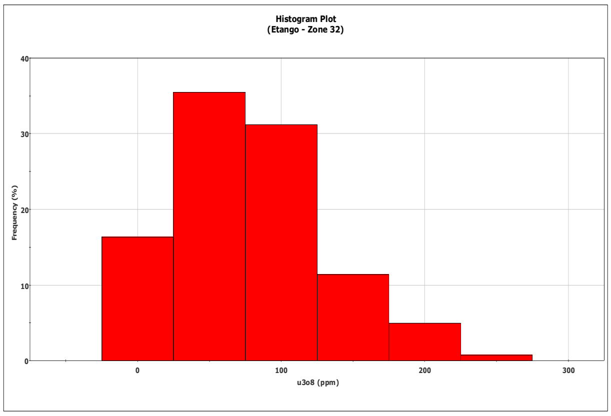


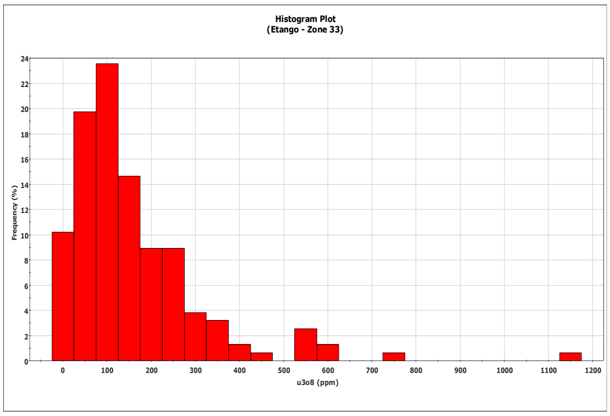


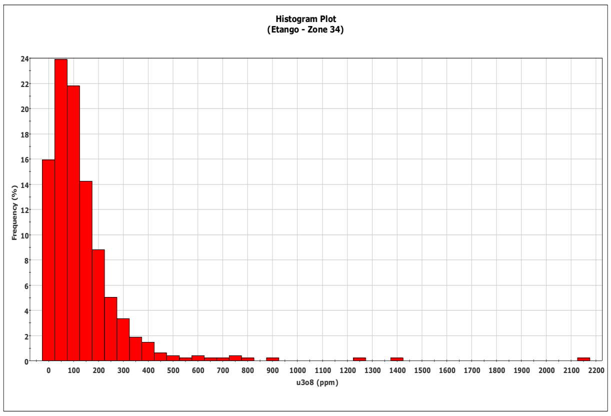


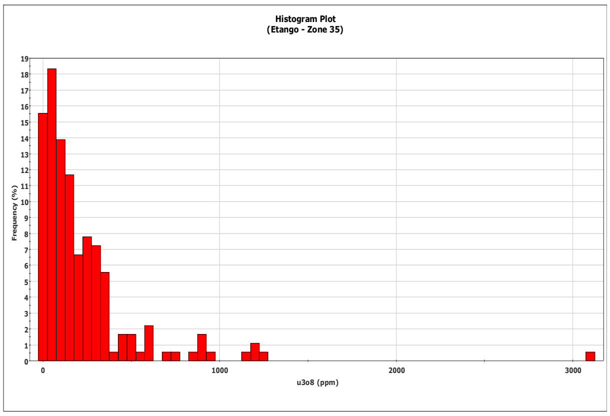
Appendix 2
Composite Statistics
Etango Project

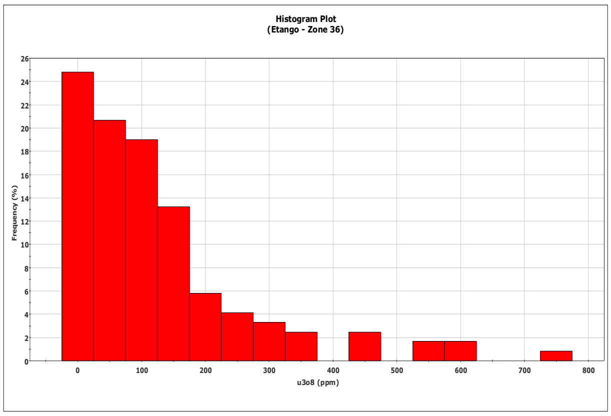


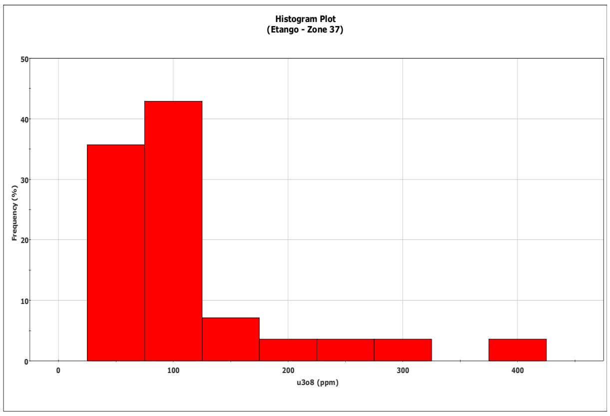


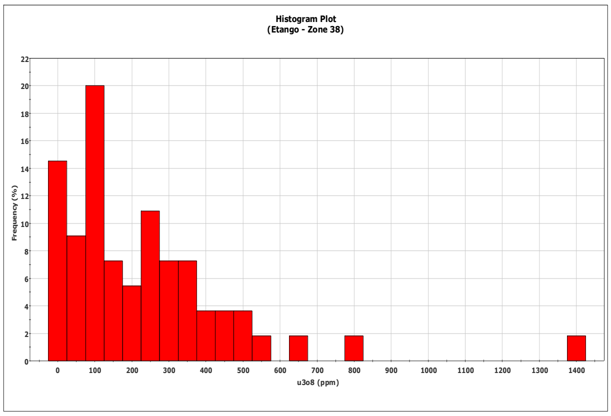




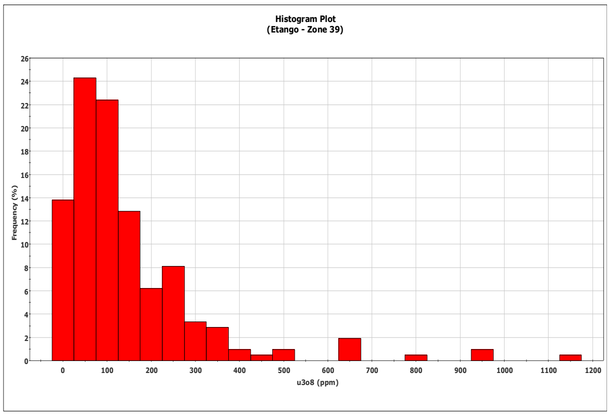


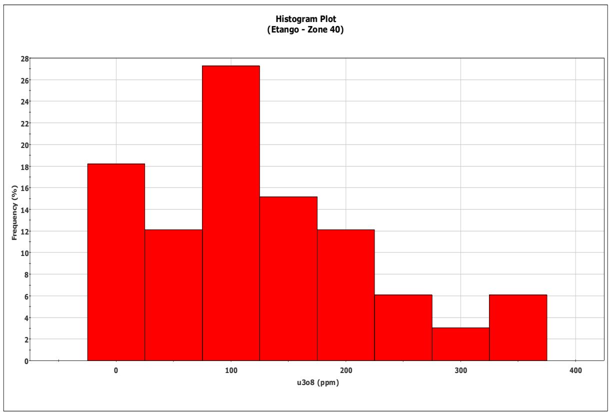


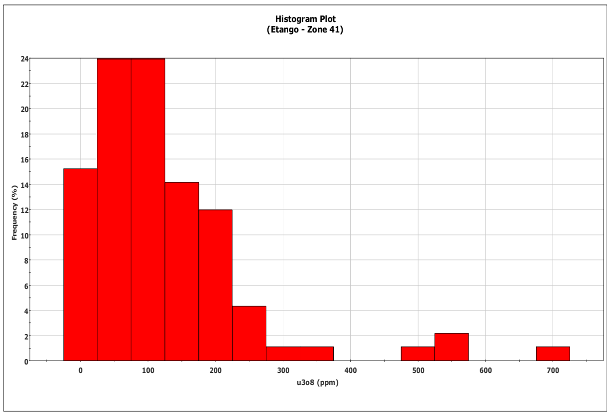


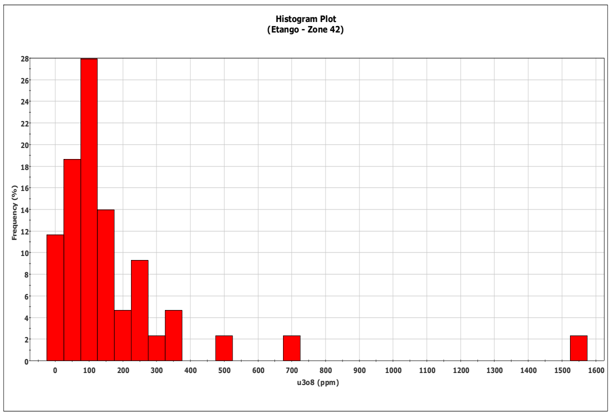


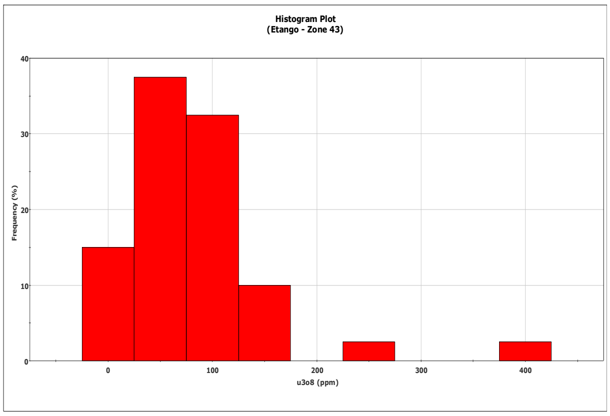
Appendix 2
Composite Statistics
Etango Project

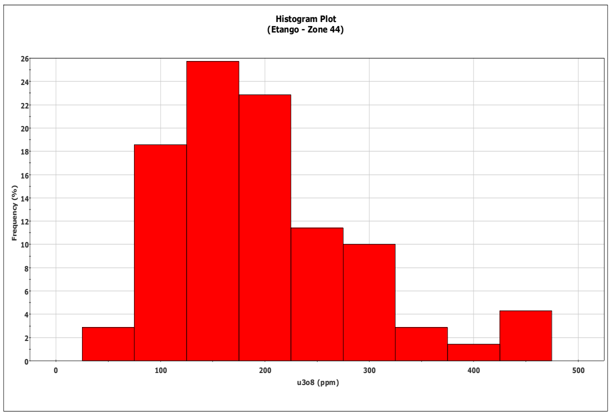


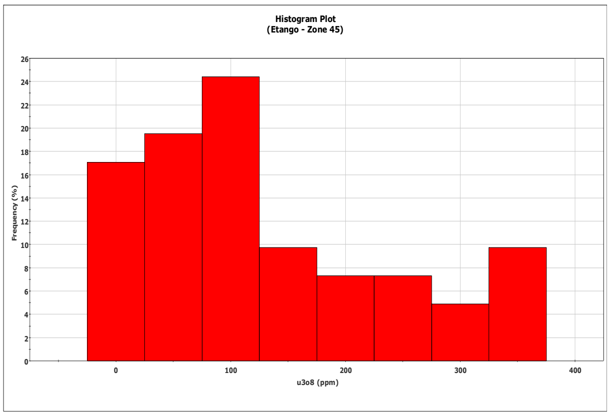


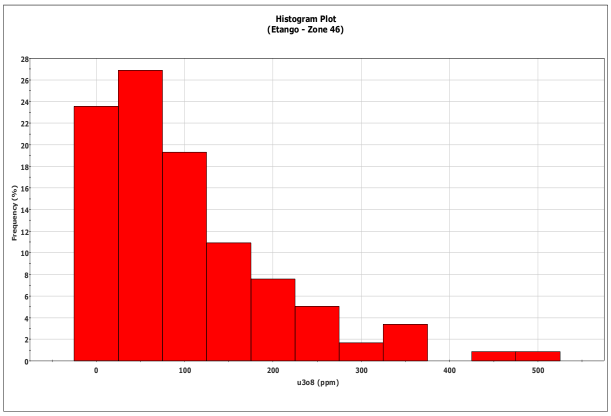




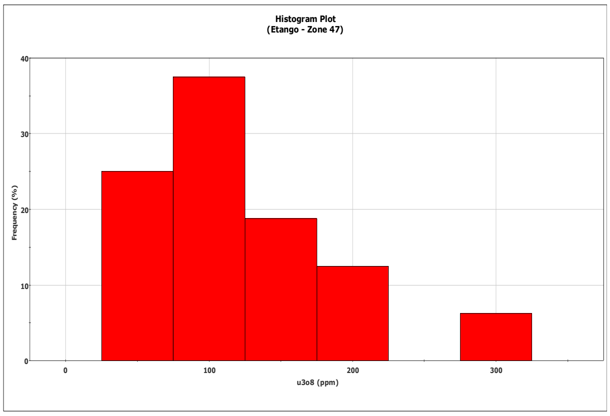


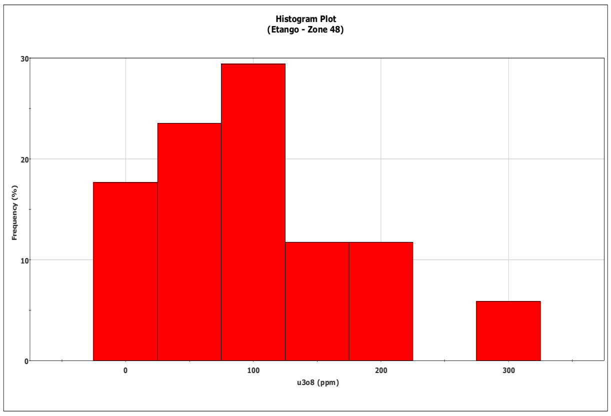


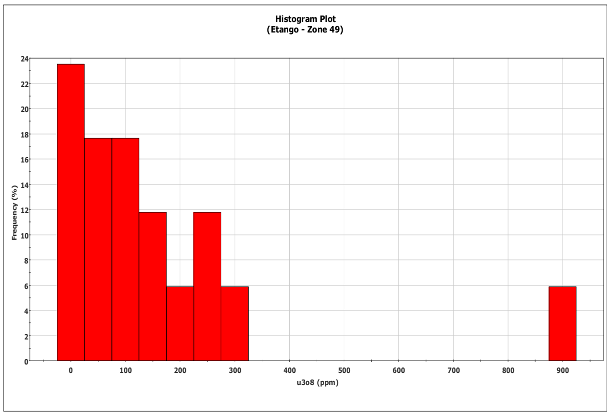


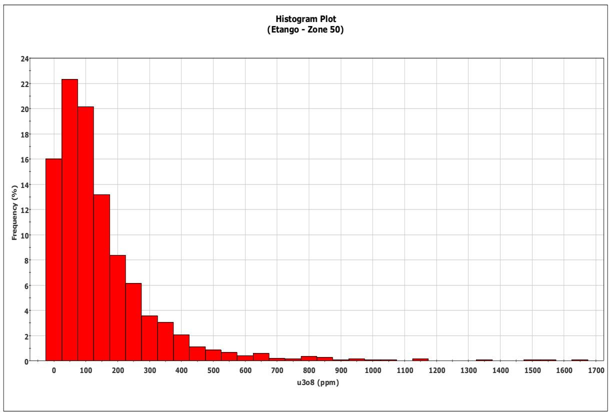


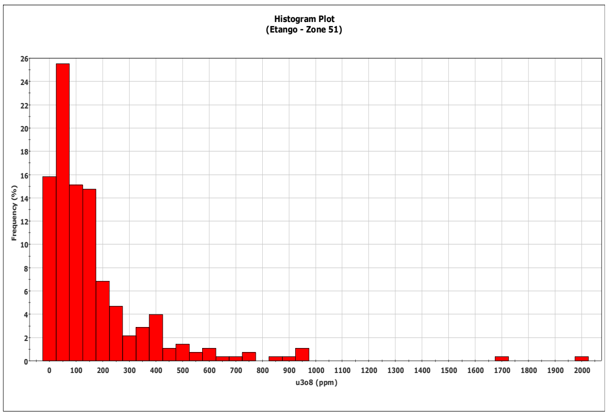
Appendix 2
Composite Statistics
Etango Project

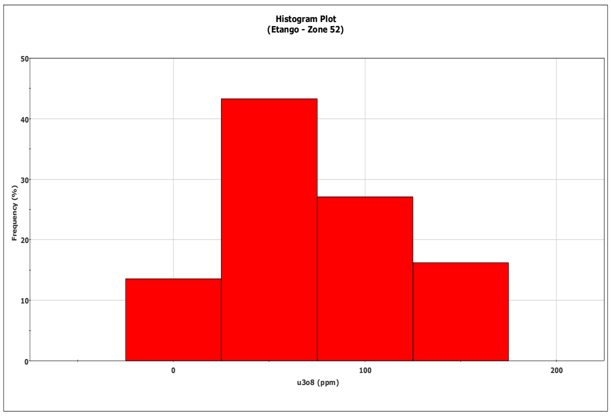


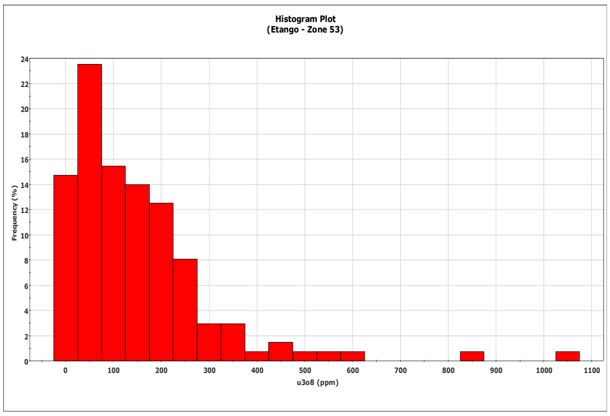


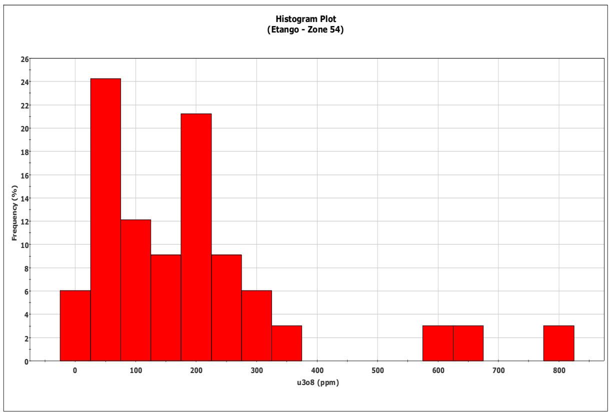




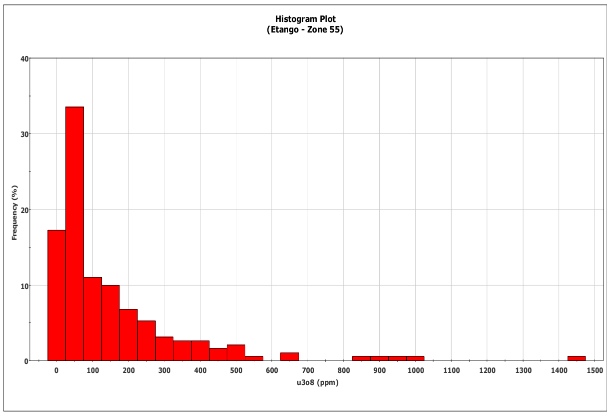


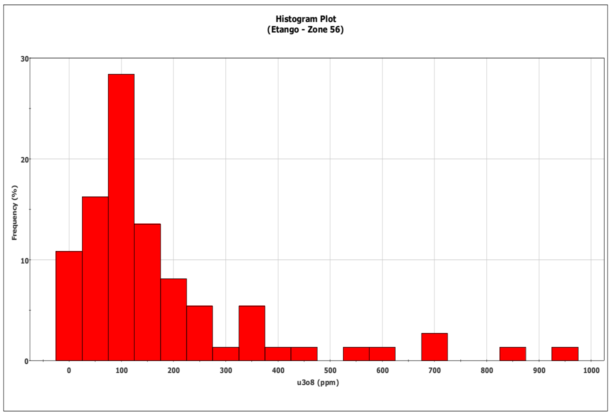


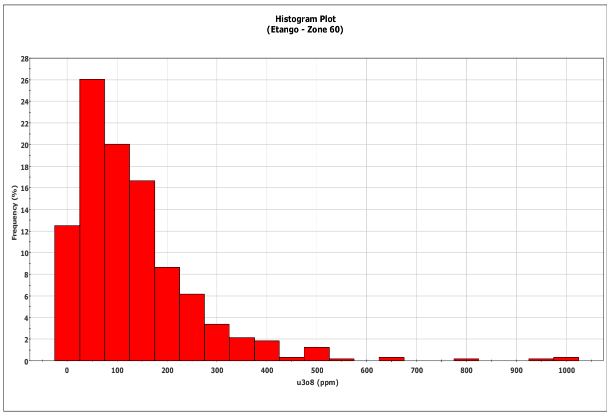


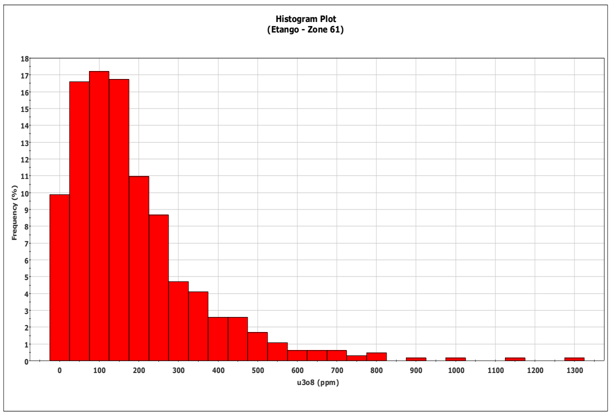


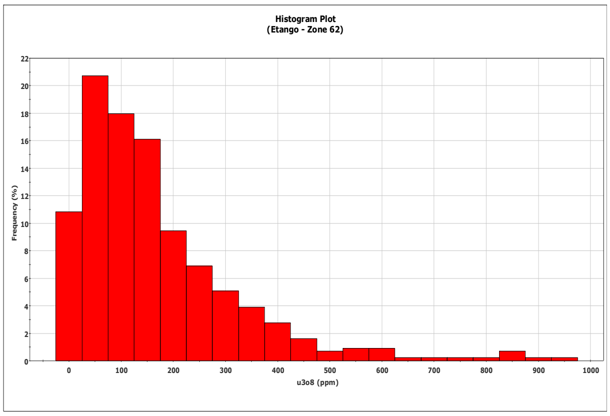
Appendix 2
Composite Statistics
Etango Project

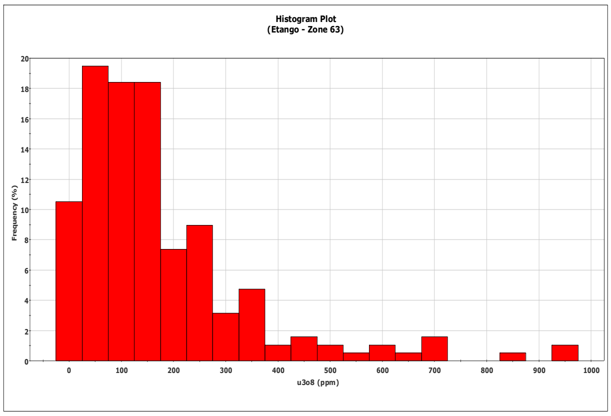


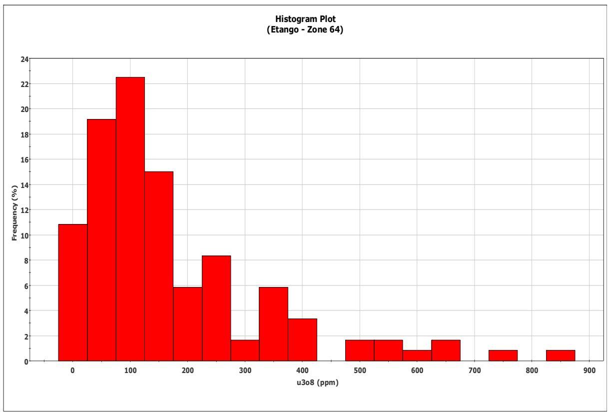


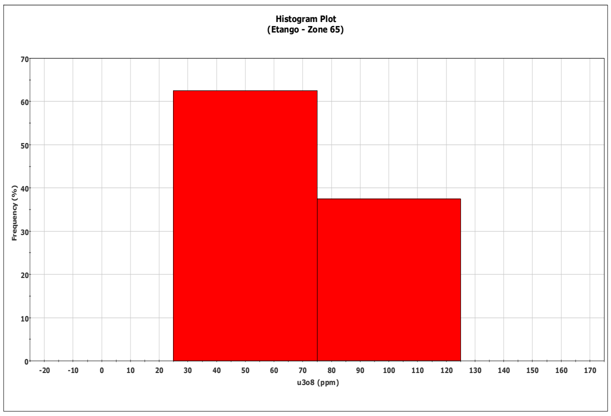




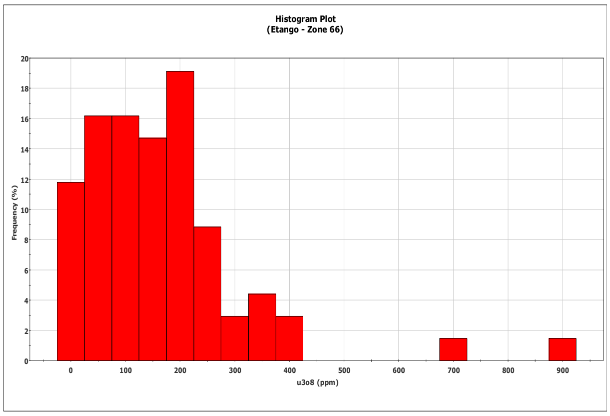


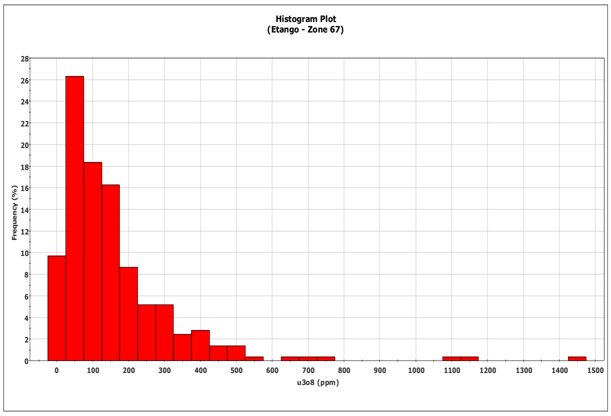


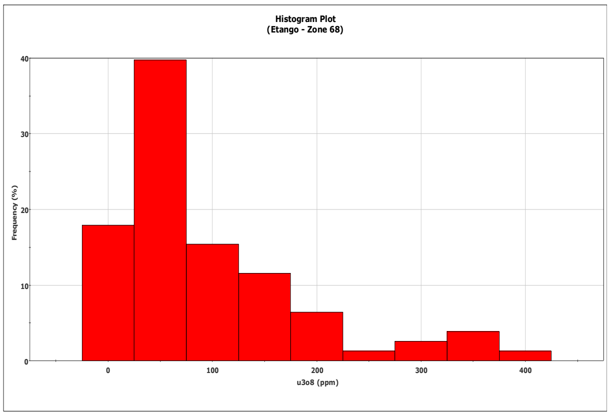


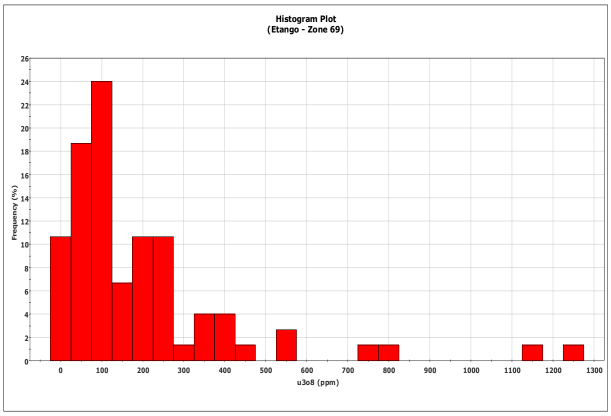


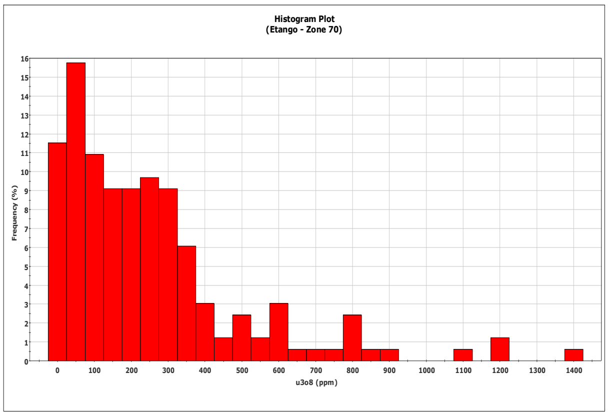
Appendix 2
Composite Statistics
Etango Project



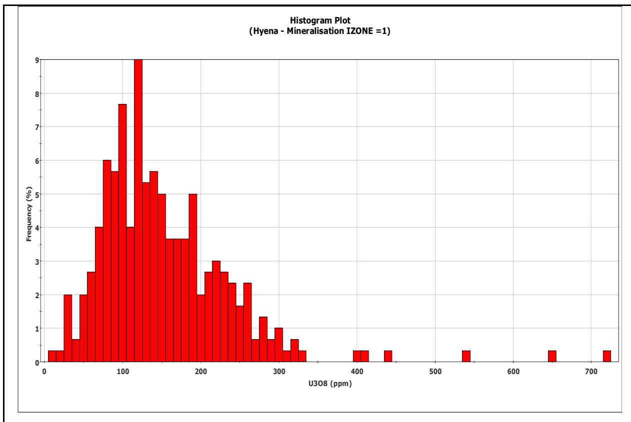


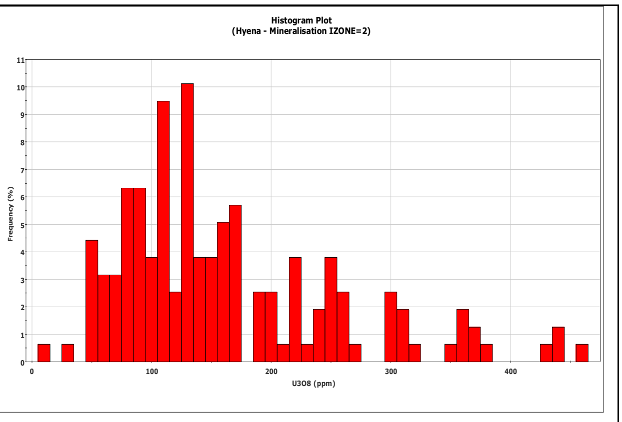


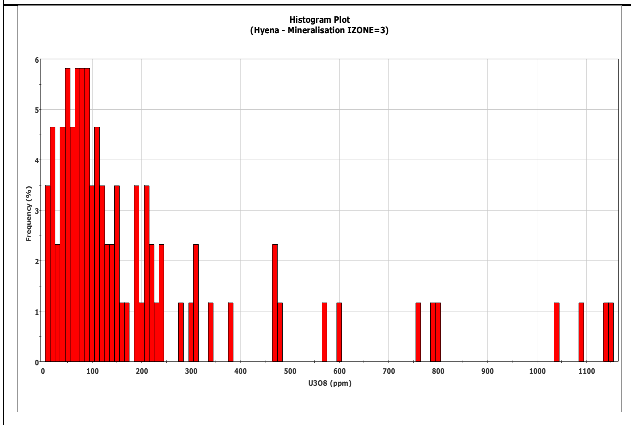


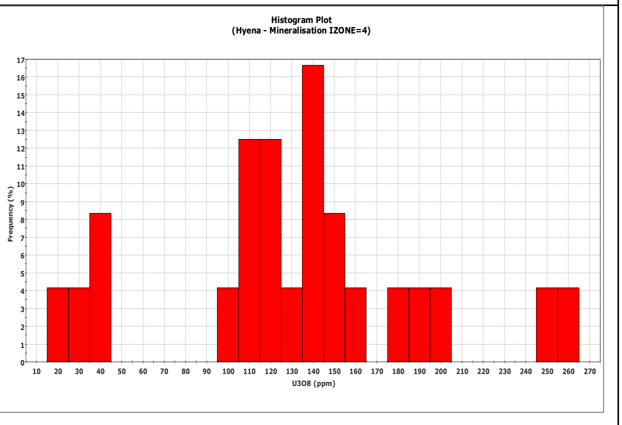


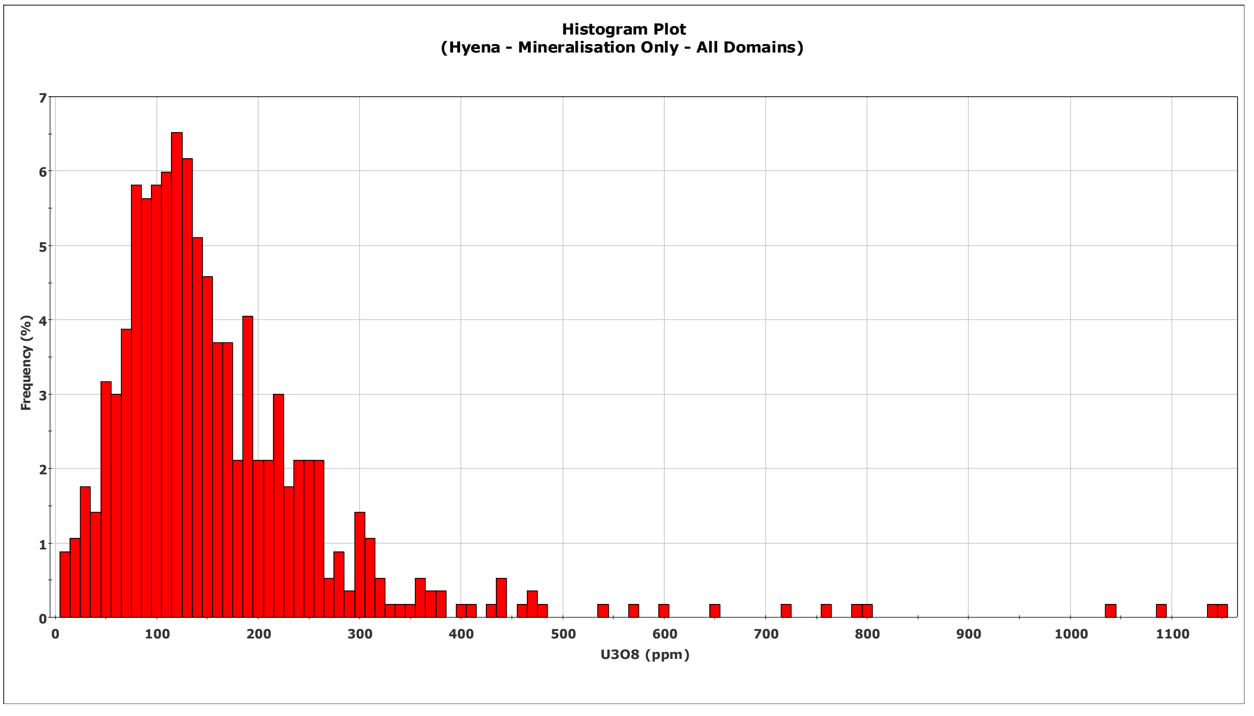
Appendix 2
Composite Statistics
Hyena Project



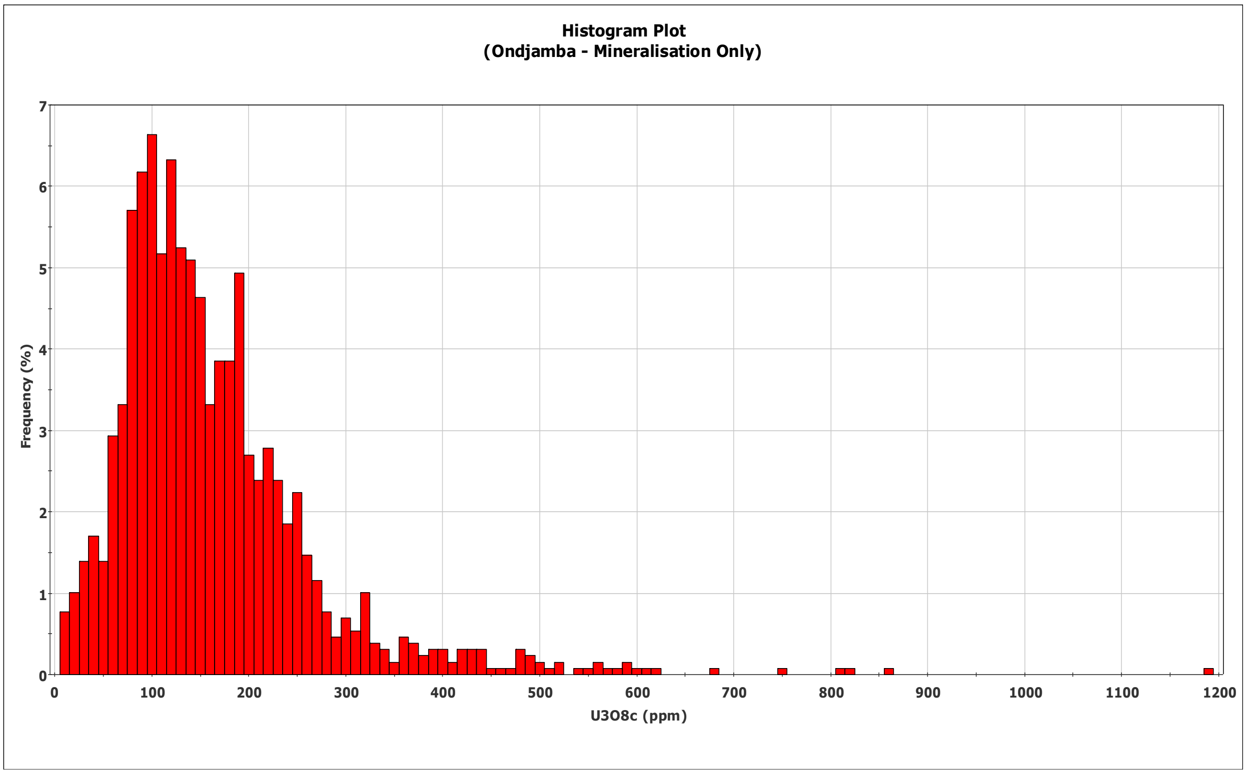








Appendix 2
Composite Statistics
Onjamba Project



Appendix 3
Certificates

Certificate of Qualified Person

As an author of the report entitled "*Etango Uranium Project, Namibia, National Instrument 43.101 Technical Document*" dated 28 September 2011, on the Etango Project property of Bannerman Resources Limited (the "Study"), I hereby state:

1. My name is Neil Andrew Inwood 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 Fellow of the AusIMM (210871).
3. I am a graduate of Curtin University of Technology in Western Australia with a BSc in Geology in 1993 and a PGrad Dip in Hydro-Geology in 1994. In 2007 I graduated from the University of Western Australia with a MSc in Geology, and from Edith Cowan University with a Post Graduate Certificate in Geostatistics.
4. I have practiced my profession continuously since 1994.
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 several times from 2007 until 2011. I have performed consulting services and reviewed files and data supplied by Bannerman Resources between July 2007 and September 2011.
7. I contributed to and am responsible for Sections 12, 14 and 26.1 of the Study and the associated text in the summary, conclusions and recommendations.
8. As of the effective date of the Study, to the best of my knowledge, information and belief, the parts of the Study for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Study not misleading.
9. I am independent of Bannerman Resources pursuant to section 1.4 of the Instrument.
10. I have read the National Instrument and Form 43-101F1 (the "Form") and the Study has been prepared in compliance with the Instrument and the Form.
11. I do not have nor do I expect to receive a direct or indirect interest in the Etango Project property of Bannerman Resources, and I do not beneficially own, directly or indirectly, any securities of Bannerman Resources or any associate or affiliate of such company.

Dated at Perth, Western Australia, on 28 September 2011.

[signed]

Neil Inwood
Principal Resource Geologist

BSc (Geology)
MSc (Geology)
Post Grad Cert Geostatistics

Certificate of Qualified Person

As an author of the report entitled "*Etango Uranium Project, Namibia, National Instrument 43.101 Technical Document*" dated 28 September 2011, on the Etango Project property of Bannerman Resources Limited (the "Study"), I hereby state:

1. My name is John R Turney and I am a Project Director with Bannerman Resources Ltd of Level 1, 513 Hay Street, Subiaco, WA, 6008, Australia.
2. I am a practising Process Engineer and a Fellow of the AusIMM (103053).
3. I am a graduate of Monash University, in Victoria, Australia with a BE (Chem) in 1974. In 1979 I graduated from Colorado School of Mines, United States of America with an MSc in Metallurgical Engineering.
4. I have practiced my profession continuously since 1974.
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 contributed to and am responsible for the generation of test program and engineering studies and the resulting associated text in the summary, conclusions and recommendations.
7. I contributed to and am responsible for all sections of the Study and the associated text in the summary, conclusions and recommendations.
8. As of the effective date of the Study, to the best of my knowledge, information and belief, the parts of the Study for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Study not misleading.
9. I am an employee of Bannerman Resources and am therefore not independent pursuant to Section 1.4 of the Instrument.
10. I have read the National Instrument and Form 43-101F1 (the "Form") and the Study has been prepared in compliance with the Instrument and the Form.
11. I do not have nor do I expect to receive a direct or indirect interest in the Etango Project property of Bannerman Resources, and I do not beneficially own, directly or indirectly, any securities of Bannerman Resources or any associate or affiliate of such company.

Dated at Perth, Western Australia, on 28 September 2011.

[signed]

John Turney — BSc (Chem)

Project Director — MSc (Metallurgical Engineering)

Certificate of Qualified Person

As an author of the report entitled "*Etango Uranium Project, Namibia, National Instrument 43.101 Technical Document*" dated 28 September 2011, on the Etango Project property of Bannerman Resources Limited (the "Study"), I hereby state:

1. My name is Kieron David Munro and I am a consultant to, and previously Head of Geology of, Bannerman Resources Limited of Suite 18, Level 1, 513 Hay Street, Subiaco, WA, 6008.
2. I am a practising Geologist and a Member of the Australian Institute of Geoscientists (AIG) (Member No. 2153).
3. I am a graduate of The University of Western Australia and hold both Bachelor (BSc) and Master (MSc) Degrees in Geology.
4. I have practised my profession continuously since 1979. I have over 30 years' international experience in geological exploration, project development and mining operations across a wide range of precious and base metals, iron ore, vanadium, chromite, uranium and industrial minerals.
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 conducted personal inspections of the Etango Project property between June 2009 and September 2011.
7. I contributed to and am responsible for sections 1 to 12 and 23 of the Study and the associated text in the summary, conclusions and recommendations.
8. As of the effective date of the Study, to the best of my knowledge, information and belief, the parts of the Study for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Study not misleading.
9. I am formerly Head of Geology for Bannerman Resources and am therefore not independent pursuant to section 1.4 of the Instrument.
10. I have read the National Instrument and Form 43-101F1 (the "Form") and the Study has been prepared in compliance with the Instrument and the Form.
11. I do not have nor do I expect to receive a direct or indirect interest in the Etango Project property of Bannerman Resources, and I do not beneficially own, directly or indirectly, any securities of Bannerman Resources or any associate or affiliate of such company.

Dated at Perth, Western Australia, on 28 September 2011.

[signed]

Kieron Munro BSc, MSc (Geol)
Consultant MAIG