Norasa Uranium Project

Definitive Feasibility Study

National Instrument 43-101 Technical Report

Forsys Metals Corp.

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Table of Contents

List of Tables

List of Figures

1. SUMMARY

1.1 PREAMBLE

This National Instrument 43-101 ("NI 43-101") Technical Report describes the Mineral Resource and Reserve estimation and economic analyses to a Definitive Feasibility Study level for the Norasa Uranium Project ("Norasa"), located in the central coastal Erongo Region of Namibia (Figure 1.1). Norasa consists of the Valencia (Mining Licence, ML 149) ("Valencia") and Namibplaas (Exclusive Prospecting Licence, EPL 3638) ("Namibplaas") Uranium Projects. Valencia is held by Valencia Uranium (Pty) Ltd while Namibplaas is held by Dunefield Mining Company (Pty) Ltd. Both of these Namibian companies are wholly owned subsidiaries of Forsys Metals Corp., a Toronto listed company.

This technical report is based on the scientific and technical information as of December 31, 2014. Some of the key highlights include:

- Forsys has completed updated Mineral Reserve and Mineral Resource estimates. Mineral Reserves increased from 79.0 Mlbs U_3O_8 as of October, 2013 to 90.7 Mlbs U_3O_8 as of February, 2015 (up by 14.8%), due to a 16.4% increase in tonnage and only a slight decrease in the average grade. The changes to the Mineral Reserve estimates are primarily due to the addition of 10.7 Mlbs U_3O_8 of Reserves from the Namibplaas deposit, using a 140ppm cut-off grade. See Sections 14 and 15 for more details.

- The operating costs per pound are estimated to average \$32.96/lb U_3O_8 over the first 5 years of production and \$34.72/lb U_3O_8 over the life of the mine. A significant reduction below the 2013 Engineering Cost Study ("ECS") costs of \$34.76 and \$38.20/lb U_3O_8 respectively. A notable achievement given the general trend of escalating costs widespread across the mining industry, particularly in the areas of labour, energy and consumables. See Section 21.3 for more details.

- The economic analysis results in an estimated pre-tax net present value (NPV) at a discount rate of 8% to Forsys of \$622.6 million. Using the initial investment and operating cash flows from inception, the pre-tax internal rate of return (IRR) is estimated to be 32%. See Section 22.3 for more details.

- The Norasa production schedule has been modified to incorporate the additional Mineral Reserves and to include a production rate increase to 11.2 Mtpa per year (up from 8.2 Mtpa in 2010) that is scheduled for 2017. Between 2018 and 2022, an average annual production of 5.3 Mlbs U_3O_8 is forecast. Estimated annual production over the 15 year LoM is estimated to be 5.2 Mlbs U_3O_8. See Section 16 for more details.

- A metallurgical study completed by SGS (South Africa) reported significant improvements when compared to the 2013 ECS:
  - Overall uranium recovery increased to 91.3% from 85.0%
  - Leach acid consumption decreased by 35%
  - Zero iron addition required in leach once steady state operation is achieved due to iron in the ore being sufficient

Figure 1.1 Location of the Norasa Uranium Project.

1.2 MINERAL RESOURCES AND RESERVES

Two NI 43-101 Technical Reports were previously prepared for Norasa in support of Mineral Resources in October 2013 and Mineral Reserves in February 2014. Independent Technical Reports were previously filed for Valencia by the Snowden Group ("Snowden") in 2010 for Resources and Reserves and for Namibplaas by Optiro in 2011 for a maiden Resource.

An updated Mineral Resources and Mineral Reserves Statement for Norasa was completed in February 2015. The Mineral Resources are reported above cut-off grades of 100 ppm and 140 ppm U_3O_8 for Valencia and Namibplaas respectively, and areas have been classified as Measured, Indicated and

Inferred Resources in accordance with the guidelines of the NI 43-101 as listed in Table 1.1. The reporting cut-off grades applied are considered to be higher than the economic cut-off grades to enable a higher-grade feed to maximize production during the life-of-mine.

The estimated Measured and Indicated Mineral Resource for Norasa is 265 Mt at a grade of 197 ppm U_3O_8, which equates to 115 Mlbs of U_3O_8. The estimated Inferred Mineral Resource is 26 Mt at a grade of 200 ppm U_3O_8 for 11 Mlbs of U_3O_8.

Table 1.1 Norasa Mineral Resource (February 2015)

Category	Cut-Off Grades	Tonnes [M]	U_3O_8 [ppm]	U_3O_8 [Mlbs]
Measured				
	Val 60ppm: Nam 100ppm	27	151	9
	Val 100ppm: Nam 140ppm	**16**	**200**	**7**
	Val 140ppm: Nam 180ppm	10	249	6
Indicated				
	Val 60ppm: Nam 100ppm	469	152	157
	Val 100ppm: Nam 140ppm	**249**	**196**	**108**
	Val 140ppm: Nam 180ppm	130	251	72
Measured + Indicated				
	Val 60ppm: Nam 100ppm	496	151	166
	Val 100ppm: Nam 140ppm	**265**	**197**	**115**
	Val 140ppm: Nam 180ppm	140	251	77
Inferred				
	Val 60ppm: Nam 100ppm	50	153	17
	Val 100ppm: Nam 140ppm	**26**	**200**	**11**
	Val 140ppm: Nam 180ppm	13	260	7

Resources are reported inclusive of Reserves.

The Mineral Reserve estimate is summarised in Table 1.2. The total Proven and Probable Norasa Mineral Reserve is 206 Mt at a grade of 200 ppm, which equates to 90.7 Mlbs of U_3O_8. Resources are reported inclusive of Reserves. Mineral Resources that are not Reserves either haven't demonstrated economic viability or don't meet the cut-off grade criteria.

Table 1.2 Norasa Mineral Reserves Estimate (February 2015)

Classification	Tonnes [M]	U_3O_8 [ppm]	U_3O_8 [Mlbs]
Proven	16	200	7.1
Probable	190	200	83.6
Total Reserve	**206**	**200**	**90.7**

Cut-off grades of 100ppm for Valencia and 140ppm Namibplaas

The Reserves come from three deposits, resulting in 3 distinct pits; the Valencia pit, a small satellite pit adjacent to Valencia, and the Namibplaas pit.

1.3 BACKGROUND

AMEC previously conducted a Feasibility Study considering just the Valencia Project in 2008 and has also conducted a number of smaller Engineering Cost Studies on the Valencia Project subsequent to the 2008 Feasibility Study to examine various process plant options to potentially enhance the Project.

This DFS was conducted to update the 2008 Feasibility Study to incorporate the options that indicated a positive outcome on the Project.

Early stage development has commenced at Norasa located in the politically stable country of Namibia, Africa. The Valencia deposit is situated approximately 80km from the coastal town of Swakopmund. It is located 35 km along strike to the world class Rössing Uranium Mine which has been in production for over 30 years and 50 km north of the Langer Heinrich Uranium Mine. Namibplaas is located 7.5km northeast of the Valencia deposit. Norasa's Valencia mine is fully permitted including all environmental approvals as detailed in Table 1.3.

Table 1.3 Norasa Project Permits

Permit	Issued By	Date received
Mining Licence (ML 149)	Ministry of Mines and Energy	23 June 2008
Exclusive Prospect Licence (EPL3638)	Ministry of Mines and Energy	7 Nov 2011
Accessory Works ML 149	Ministry of Mines and Energy	29 May 2009
Environmental Clearance	Ministry of Environment and Tourism	11 April 2013
Exclusive Prospecting Licence (EPL 3638)	Ministry of Mines and Energy	7 Nov 2013
Petroleum Consumer Installation	Ministry of Mines and Energy	12 Oct 2012
Radiation Management Plan (ML 149)	Ministry of Health and Social Services	9 Dec 2010

In August 2008, the Namibian government issued a 25 year Mining Licence for the mine. In the second quarter of 2010 an important access road was completed, joining the Valencia mine site to the main highway. Construction of power infrastructure has commenced in conjunction with NamPower and the Company has entered into a Memorandum of Understanding with NamWater to obtain water required for the mine construction phase and production.

1.4 GEOLOGY AND MINERALISATION

Uranium deposits world-wide have been grouped into 14 major categories of deposit types based on the geological setting of the deposits. The Valencia deposit is an "intrusive type" of uranium deposit that is associated with alaskite intrusives that comprise massive stock-like bodies, dykes of varying thickness, sill like bodies and veins and veinlets, which can be either conformable with or transgressive to the Damara Sequence metasedimentary host rocks. Included in this type are those deposits associated with intrusive rocks including alaskite, granite, pegmatite and monzonites. Major world deposits include Rössing (Namibia), Ilimaussaq (Greenland) and Palabora (South Africa).

Norasa is located in the Damara Orogen which is a Pan African - aged result of a "Wilson Cycle" collision between the Kalahari Craton in the south and the Congo Craton in the north. It comprises a coastal branch along Namibia's north coast into Angola, The Kaoko Belt, and an inland branch (Damara Belt) stretching from the Namibian coast north eastwards through to Zambia. The oblique collision closed an ancient seaway, the Damara Ocean, forcing together a varied collection of depositional environments. The sequence of tectonic and deformational periods which, followed by erosion, produced the strongly-zoned remnants of a continent-continent mountain chain root that we see today.

The Inland Branch (Damara Orogen) has been divided from north to south along NE-SW trending tectono-stratigraphic lineaments. These boundaries divide the Orogen into SW - NE trending Zones.

Metamorphic gradients vary between these zones and are increasing to granulite facies in the Central Zone, toward the more deeply eroded coastal region in the west.

Primary uranium mineralisation of significance is limited to the Central Southern Zone which hosts all the major primary uranium occurrences known today in Namibia. Large volumes of U-bearing leucogranite intrude a limited stratigraphic-range, occasionally cross-cutting into basement but mainly into stratigraphic units directly above and below the Swakop Group contact.

It is at this stratigraphic level where the largest uranium reserves in Namibia are found; the Husab Mine, the Rössing Mine, the Valencia project and the Etango deposit at Goanikontes to name the most significant ones.

The uranium mineralisation throughout Norasa is hosted by either contaminated B and C type (Namibplaas) and or D-type granite only (Valencia). Their appearance ranges from aplitic veins to leukogranitic pegmatites to massive intrusive granites.

In places hydrothermal alteration has overprinted and led to additional enrichment of uranium in form of secondary mineralisation phases. The mineral uraninite is the dominant uranium carrier throughout the deposits. Other primary uranium minerals include carnotite, titanite (including brannerite) and naobate. Late stage alterations of all alakite types can be observed and comprise kaolinisation, illitisation and silicification.

The uranium mineralization at Valencia is hosted only by alaskites that comprise massive stock-like bodies, dykes and sills of varying thickness and veins and veinlets, which can be either conformable with or transgressive to the Damara Sequence metasedimentary host rocks.

Uranium mineralization has been identified over an area of 1,100 m north-south by 500 m east-west. The mineralization dips at approximately 35° to the south and has been identified by DDH drilling to a depth of 499 m below surface (VA26-152).

Uranium mineralization is present as uraninite (UO_2) and secondary uranium minerals, uranophane ($Ca(UO_2)_2SiO_2.7H_2O$) and uranothallite ($Ca_2U(CO_3)_4.10H_2O$). No betafite has been observed at Valencia.

The uraninite is usually fresh with only sporadic, very minor alteration rims. The secondary uranium minerals uranophane and uranothallite occur as yellow coatings on exfoliation planes and joints, where they form specks and tiny flakes on feldspar, quartz, biotite and apatite. Recent studies show that uraninite represents more than 95% of all uranium minerals.

Uranium mineralization predominantly occurs in the finer-grained alaskite and rarely in the coarse pegmatitic phases. A close relationship also exists between the uranium and biotite content, as well as between the degree of uranium mineralization and apatite content. The uranium mineralization is variably distributed through the alaskite intrusions and, a great portion of the mineralization is well disseminated throughout the alaskite with local minor enrichments onto biotite selvages or local accumulations of magnetite.

1.5 MINING METHODS AND RESERVES

An open pit mining operation is proposed for Norasa. Detailed geotechnical studies, pit optimization and design work have defined the Reserves together with appropriate modifying factors.

The life of mine schedule reflects a process plant milling rate of 11.2 Mtpa over the life of mine. Norasa will require a peak mining rate of 74 Mtpa. The average strip ratio for the life of mine is 3.2.

The mining method preferred for the Norasa deposit will be low cost, traditional open pit truck and backhoe operation employing 400t diesel hydraulic excavators, off road 180t haul trucks and 171mm down-the-hole (DTH) hammer diesel drills.

Selective mining for Norasa consists of drilling and blasting on a 10m bench, with loading out in two flitches of equal height, which will nominally be 6.0m high, after allowing for swell from blasting and levelling. The mining selectivity recommended should minimise ore loss and dilution but, at the same time, allow the peak 74Mtpa mining rate to be achieved cost efficiently.

The project supports a 15 year mine life at the proposed production rate.

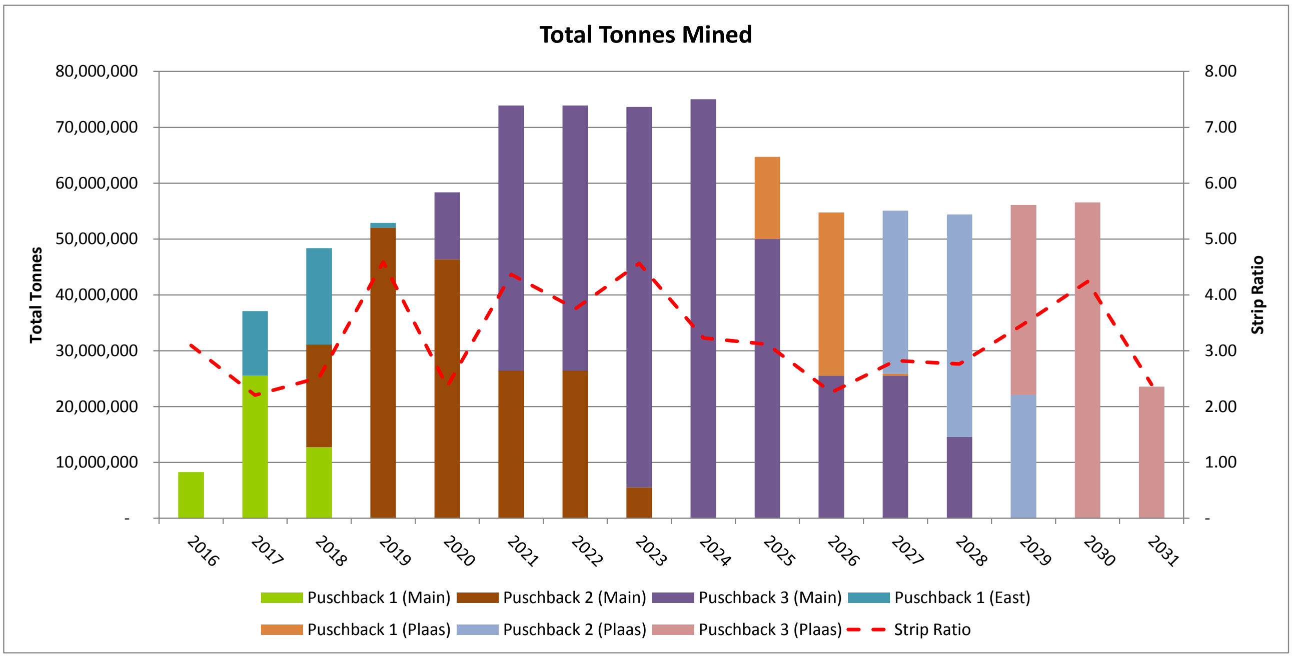


Figure 1.2 Total tonnes mined by area / pushback.

1.6 METALLURGICAL TESTWORK

The Norasa Uranium Project (and formerly Valencia) processing flow sheet has been developed and evaluated in a number of testwork programs and economic studies between 2007 and 2014. As the Project combines the former Valencia Uranium Project with the more recent developed Namibplaas Uranium Project, historic testwork related to both these projects are covered in this section. The flow sheet has evolved over time and the recent 2014 pilot testwork campaign focused on evaluating the impact of some of these changes and the integrated nature of the flow sheet. The objective of this program was to confirm performance of an optimised flow sheet on a feed composite representative of the 2014 mine plan.

The main unit operations included in the flow sheet and covered by the 2014 testwork include sulphuric acid leach, solid-liquid separation and ion exchange recovery of the leached uranium from the pregnant leach solution (PLS).

Composite metallurgical samples were prepared from a series of samples selected by the client within the proposed Valencia life of mine pit. The conclusion reached is the composite is considered indicative of a life of mine average process feed and was prepared to a target uranium grade of 200 ppm U_3O_8 and the resultant solid analysed to between 205 and 216 ppm U_3O_8.

Locked cycle metallurgical pilot testwork was performed on the average feed composite, including acid leach and ion exchange. The main findings are:

- No iron addition is required to leach when recycling barren liquor due to iron leached from the ore and recycled.
- An average leach acid dosage of 12.2 kg/t (as 98% H_2SO_4) is required, excluding grinding media and recovery associated acid consumption and losses.
- Hydrogen peroxide is preferred to MnO_2 as leach oxidant as it is more efficiently utilised and consumes less acid. The required 50% H_2O_2 dosage is 1 kg/t.
- The agitated tank bulk leach uranium extraction averaged 93.2%.
- High IX resin loading of 20 to 30 g U_3O_8/L was obtained.

These results are seen as generally consistent with earlier testwork campaign's findings, taking cognisance of the benefits obtained by changing oxidant and recycling barren liquor. Both these flowsheet improvements contributed to a significant reduction in leach acid consumption. The higher leach extraction and associated overall circuit uranium recovery resulted from improved mixing conditions in the larger scale leach vessels used for pilot testing and a reduction in filter soluble loss due to wash optimisation.

1.7 PROCESS DESIGN

1.7.1 Processing

The proposed process plant flowsheet (Figure 1.3) comprises the following unit operations:

- 2 stage crush & screen with a coarse ore stockpile;
- single SAG mill;
- atmospheric acidic leaching;
- belt filtration;
- continuous ion exchange (CIX);
- solvent extraction (SX) and ammonium diuranate (ADU) precipitation;
- filtration;
- calcination;
- reagent preparation areas.

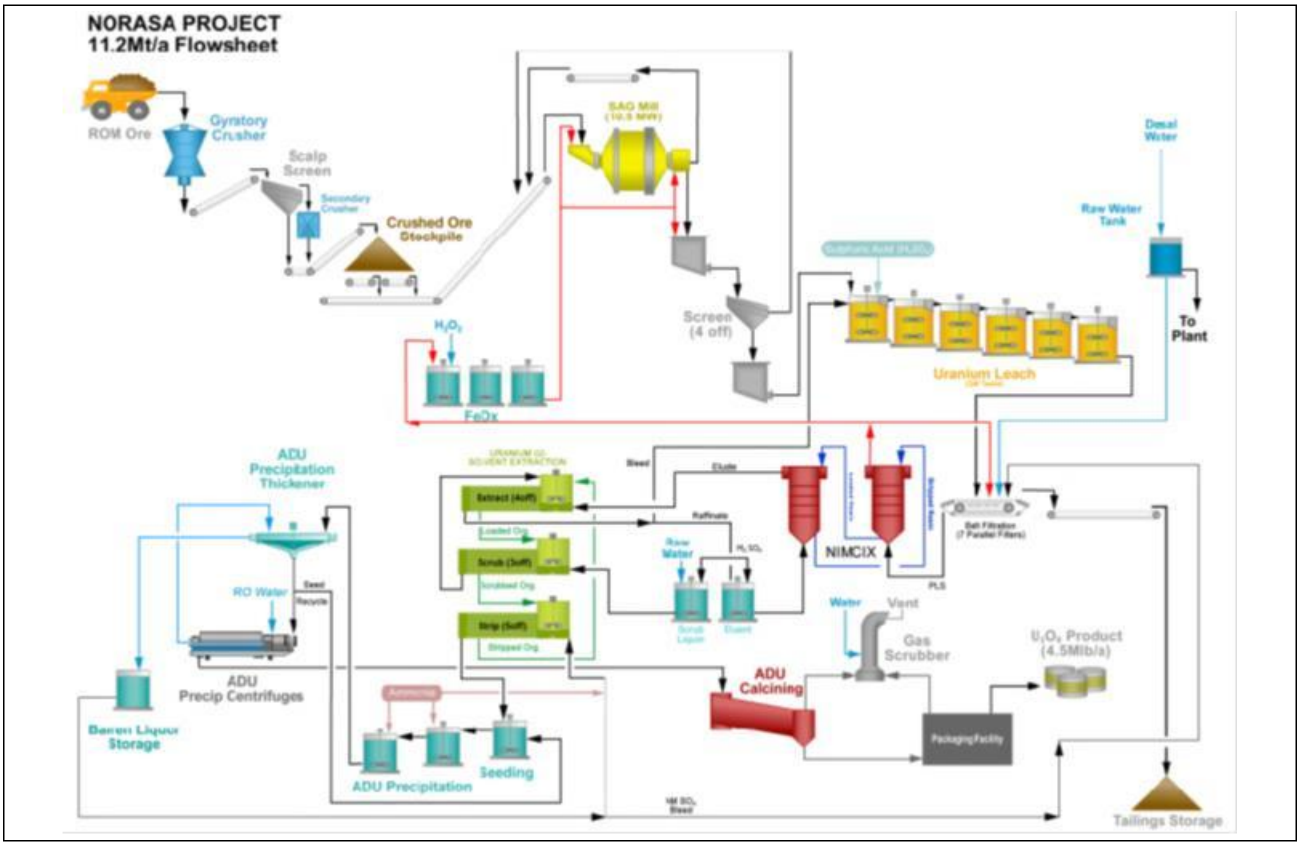


Figure 1.3 Schematic Norasa Flowsheet

The main plant design parameters used in the current study are presented in Table 1.4.

Table 1.4 Plant Parameters

Parameter	Units	Value	Comments
Average feed grade (U_3O_8)	ppm	200	
Plant throughput	Mt/a	11.2	
Plant production at average grade	Mlb U_3O_8/a	4.5	At 238 ppm U_3O_8 – 5.2 Mlb U_3O_8/a
Leach extraction	%	93.2	At 200 ppm U_3O_8 feed grade
Solid/liquid separation recovery	%	98.1	Filtration testwork
IX/SX recovery	%	99.9	
Overall recovery	%	91.3	At 200 ppm U_3O_8 feed grade
Raw water consumption	m^3/t	0.21	
Power consumption	kWh/t	18.0	

1.8 INFRASTRUCTURE

Water

Water will be supplied from the Rössing reservoir with a new 31km pipeline to site. Less than 3.0 Mm^3 of water will be required annually.

Power

Norasa will be supplied power from the national grid, with the nearest take-off point requiring a 26km 220kV line extended to the new substation on the mine site adjacent to the process plant. The installed electrical capacity is approximately 40MW, with the largest demand being the SAG mill at 10.5MW. Construction power supply will be via temporary generator sets on site.

Road
A 26km private industrial gravel road was completed in mid-2010 connecting Norasa to the Trans Kalahari (B2) Highway, which is the main artery from Walvis Bay and Swakopmund to Windhoek and across Southern Africa.

Accommodation
Facilities in the towns of Swakopmund, Walvis Bay and Arandis will support the Norasa operations. A construction camp to house a peak workforce of 800 workers will be constructed on site, and will become an operations camp for the life of mine.

1.9 CAPITAL COSTS

1.9.1 Mining Capital Costs

The majority of the mining capital expenditures were derived from quotations obtained from major equipment suppliers such as Atlas Copco, Liebherr, Sandvik, Hitachi and Caterpillar, with the balance being derived from Forsys in-house cost database and estimates supplied by the Company or AMEC.

Mining capital cost estimates include $94M in preproduction capital and $139M in sustaining capital (including a $11.5M second hand resale credit at the end of life). The capital cost estimate was based on December 2014 quotations and has been completed to an accuracy of ±15%.

1.9.2 Plant and Infrastructure Costs

Comminution, plant and site infrastructure capital costs have been estimated by AMEC, with Forsys estimating some components of infrastructure. Owner's costs to cover corporate, management and administrative costs, as well as capitalised pre-production operating costs, have been supplied by Forsys.

Total plant capital costs are estimated to be $252.3M (including growth allowance and excluding contingency) as at December 31, 2014. There is also an additional $5.5M in deferred first fill expenses. The estimate has been completed to an accuracy of ±15% and includes Direct and Indirect costs, growth provisions (averaging 7.9% of costs or $18.6M) and costs for engineering, procurement and construction management (EPCM) by an independent contractor.

Total infrastructure capital costs are estimated to be $38.5M as at December 31, 2014. Infrastructure includes a water pipeline from Rössing, water reservoir, power line, mine sub-station and regional NamPower upgrade requirements.

1.9.3 Owners and Other Capital Costs

Owner’s costs have been determined by Forsys to be $37.0M, and include:

- Office equipment & IT
- Village construction
- Village operation during construction
- Vehicles (light and busses)
- Tech services
- Mine pre-strip
- Mine dispatch system
- Namibia office
- Recruitment and training

- Sundry overheads
- Tailings (reflected in process plant)

The first 6 months of mining operations are considered the pre-strip phase, which is prior to process plant operation and has been capitalised. This is estimated at US$15.9M.

1.9.4 Total Project Capital Costs

These are summarised in Table 1.5. Pre-production capital costs total $432.8M with additional sustaining capital costs of $139.4M over the LoM. Included in sustaining capital for the years 2018 to 2022 is a primary mining fleet expansion as production levels increase, and for mining equipment replacement. At the end of the LoM, mining equipment that still has an economic life will be sold for an estimated value of $11.5M and will reduce sustaining capital over the LoM. Not currently considered will be sales that are accounted for as operating revenues including receipts from sale of the operations camp and recovery of first fill materials and reagents as they are recovered via operating costs at the end of the project.

Table 1.5 Total Project Capital Costs ($M)

Item	Year -2	Year -1	Year 0	Total
Mining		54.0	40.0	94.0
Process plant / Tailings	12.8	216.9	33.7	263.4
Infrastructure	9.0	29.5		38.5
Owners costs	5.6	30.2	1.2	37.0
Total	**27.4**	**330.6**	**74.8**	**432.8**

1.10 OPERATING COSTS

1.10.1 Mine Operating Costs

The total material movement as derived from the life of mine (LOM) mine production schedule was used to determine the mine equipment requirements over time. A breakdown of the mine operating costs is provided in Table 1.6. Diesel costs are the largest single component of mine operating cost.

Table 1.6 Summary of Life of Mine Operating Costs

Item	US$	US$/t mined	US$/lb U_3O_8
Load and Haul	595,653,275	0.69	7.66
Drill and Blast	395,608,597	0.46	5.09
Secondary and Support	94,118,893	0.11	1.21
Services	91,890,700	0.11	1.18
Labour	145,312,237	0.17	1.87
Total	**1,322,583,702**	**1.53**	**17.00**

1.10.2 Process Operating Costs

The process operating costs reflect operation at a throughput of 11.2Mtpa. The various process plant operating costs are summarised in Table 1.7 using an average feed grade (U_3O_8) of 200ppm.

Table 1.7 Summary of Process Operating Costs			
Item	**US$/a**	**US$/t ore**	**US$/lb U_3O_8**
Reagents	32 965 828	2.94	7.31
Power	19 203 698	1.71	4.26
Consumables (incl. plant air and water)	17 890 537	1.60	3.97
Labour	4 885 454	0.44	1.08
Maintenance	6 264 774	0.56	1.39
Miscellaneous	1 210 000	0.11	0.27
Total	**82 420 291**	**7.36**	**18.28**

1.10.3 Owners and Other Operating Costs

Owner's operating costs are equivalent to $0.75/t crushed, as summarised in Table 1.8

Table 1.8 Summary of Owners Costs			
Item	**US$M/p.a.**	**US$/t Mined**	**% of Cost**
Tailings management	1.7	0.15	20%
Site administration	4.7	0.45	55%
Camp / Village	2.2	0.19	25%
Total	**8.5**	**0.75**	

1.10.4 Total Project Operating Costs

The process operating costs are reduced as the average feed grade increases. Implementing a Radiometric Sort system in 2019 increases the average feed grade (U_3O_8) to 238ppm which reduces the average process operating cost to $16.27/lb U_3O_8 over the LoM. Total operating costs for the Project are $13.18/t ore or $34.72/lb U_3O_8 over the LOM and $12.77/t ore or $32.96/lb U_3O_8 over the first 5 years (Table 1.9).

Table 1.9 Summary of Total Operating Costs					
Item	**Cost ($/t of ore Yr. 1-5)**	**Cost ($/t of ore LoM)**	**Cost ($/lb U_3O_8 Yr. 1-5)**	**Cost ($/lb U_3O_8 LoM)**	**% of Cost**
Mining	5.67	6.42	14.65	16.83	48%
Processing and Infrastructure	6.45	6.14	16.67	16.27	47%
Owners and Other Outlays	0.64	0.62	1.65	1.63	5%
Total	**12.77**	**13.18**	**32.96**	**34.72**	

1.11 PROJECT FINANCIAL MODELLING

1.11.1 Base Case

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

The Base Case model uses the DFS mining and processing schedules and capital and operating costs. The Base Case uranium oxide price used is $65/lb as provided by Forsys, based on a review of forecasts supplied by banking institutions and broking firms. In calculating the potential returns from the project, the fundamental assumptions and parameters are shown in Table 1.10

Table 1.10 Fundamental Assumptions and Parameters of Financial Modelling Analysis

Basis	Project level, pre and post-tax and excluding any debt financing
U_3O_8 prices	Long term contract price of $65/lb U_3O_8
Development period (Months)	24
Mine life (Years)	15.0
Annual throughput	11.2 Mt
Fuel price	$0.80/L, including freight
Sulphuric acid price	$127/t delivered to site
Raw water cost	$3.00/m3
Power cost	$0.0955/kWh
Production rate	Between approximately 4.5 to 5.8 Mlb of U_3O_8 per year
Exchange rates	US$1.00 : A$1.25 : N$11.05 : R11.05 : €0.80 :C$1.10
Tax and royalty rates	37.5% and 3.0%

The key outputs from the financial model based on the above assumptions are reported for the first 5 years of the modelled operation and for the life of mine in Table 1.11.

Table 1.11 Key Financial Model Outputs

	Project	**US$/Share**
Project Economics		
NPV at a Discount Rate of 8% ($M) - (Excl. Tax)	622.6	5.25
- (Incl. Tax)	383.4	3.24
Internal Rate of Return (%) - (Excl. Tax)	32%	
- (Incl. Tax)	26%	
Payback Period from Start of Production (years)	4.4	
Capital Costs ($M)	432.8	
Production	**Life of Mine**	**First 5 Years**
Quantity Ore Treated (Mt)	206.1	66.7
Recoveries (%)	92.4%	92.2%
Uranium sold (Kg U_3O_8)	35,288	11,710
Uranium sold (Mlb U_3O_8)	77.8	25.8
Revenue and Cash Flow		
Average U_3O_8Base Price ($/lb U_3O_8)	65	65
Net Revenue ($M)	5,056.8	1,678.0
Operating cash flow ($M)	1,751.1	440.2
Net cash flow after tax ($M)	1,007.6	161.5
Operating Unit Costs ($/lb produced)		
Mining	16.83	14.65
Processing	16.27	16.67
Owners costs	1.63	1.65
Total Operating Costs ($/lb produced)	**34.72**	**32.96**

1.11.2 Sensitivity Analysis

Revenue

The financial sensitivity analysis in Table 1.12 demonstrates that the economic outcome of the Norasa Project is sensitive to changes in the uranium price. A negative movement of 8% or more from the base case assumption of $65/lb U_3O_8 results in a material decline in Pre and Post-tax NPV (DR 8%). Conversely, the Project would benefit greatly from increases in U_3O_8 prices. It is encouraging that breakeven NPV is at or about $50/Lb U_3O_8 for both pre and post-tax scenarios.

Capital Costs
Increases in the base case capital cost assumptions (excluding working capital) produce less significant changes in the pre and post-tax NPV's.

Operating Costs
Financial performance is sensitive to changes in operating costs. Increases in the base case operating cost assumptions produce adverse changes in the pre and post-tax NPV's but they remain positive at even the highest sensitivity of -30%. Conversely, the Project would benefit from decreases in operating costs.

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

Table 1.12 Sensitivity Analysis (Pre and Post-tax Basis)

Parameter/ Variation	Value		Pre-tax NPV ($M. DR 8%)	Post-tax NPV ($M. DR 8%)
U_3O_8 Price	**U_3O_8 Price ($/lb)**			
-23%	50.00		70.1	17.7
-15%	55.00		254.2	143.5
-8%	60.00		438.4	264.6
Base	65.00		622.6	383.4
8%	70.00		806.7	500.6
15%	75.00		990.9	617.4
23%	80.00		1175.0	733.3
Capital Costs	**Capital Cost ($M)**			
-30%	561.60		562.6	321.9
-20%	518.40		582.6	342.5
-10%	475.20		602.6	363.0
Base	432.00		622.6	383.4
10%	388.80		642.6	406.9
20%	345.60		662.6	436.9
30%	302.40		682.6	466.9
Operating Costs	**Average Operating Costs ($/lb)**			
	First 5 Years	**Life of Mine**		
-30%	42.85	45.14	203.5	122.2
-20%	39.56	41.66	343.2	211.3
-10%	36.26	38.19	482.9	298.0
Base	32.96	34.72	622.6	383.4
10%	29.67	31.25	762.2	473.2
20%	26.37	27.78	901.9	562.5
30%	23.08	24.30	1041.6	651.0

1.12 PROJECT DEVELOPMENT

A project development schedule has been outlined as part of the DFS, indicating completion of engineering design, procurement, transport and construction over a 18 month period following Project approval, with ramp-up to design tonnages after 24 months (Table 1.13). The schedule includes a contingency of 3 months, and is conditional upon obtaining finance to support commencement of construction.

Table 1.13 Project Development Milestones

Task	Period (Months)
Commence early works	-3
Project approval, i.e. project financing	0
Commence site works	4
Commence commissioning (includes 3 month contingency)	16
Commence ramp-up (with contingency)	18
First shipment (with contingency)	20
Ramp-up to design tonnages	24

1.13 PROJECT RISK ASSESSMENT

A range of economic, engineering and other technical risks to the Project have been considered. Those risks assessed as Low, Moderate, Major or as a combination are summarised in Table 1.14.

Table 1.14 Non-Resource / Mining Economic and Technical Risk Assessment

Item	Assessed Risk to Project
U_3O_8 price	Major
Water supply not available	Moderate to Major
Mining equipment under-performance	Moderate
Mine operating costs overrun (sustained increase in labour / materials costs)	Moderate
Capital cost overrun	Low
Operating cost overrun - diesel	Moderate
Operating cost overrun – acid	Moderate to high
Operating cost overrun – power	Moderate

1.14 CONCLUSIONS

Norasa is an advanced stage development project with Mineral Reserves estimated to be 90.7 Mlbs U_3O_8, as of February 2015 (up by 14.8%), due to a 16.4% increase in tonnage and only a slight decrease in the average grade. The changes to the Mineral Reserve estimates are primarily due to the addition of 10.7 Mlbs U_3O_8 of Reserves from Namibplaas.

The Norasa production schedule has been modified to incorporate the additional Mineral Reserves and to include a production rate increase to 11.2M tonnes per year (8.7M tonnes in 2010) that is scheduled for 2017. Between 2018 and 2022, an average annual production of 5.3 Mlbs U_3O_8 is forecast. Estimated annual production over the 15 year LoM is estimated to be 5.2 Mlbs U_3O_8.

A metallurgical study completed by SGS (South Africa) reported significant improvements when compared to the 2013 Engineering Cost Study including an overall uranium recovery increased to 91.3% from 85.0%. Leach acid consumption decreased by 35%. Further, zero iron addition is required in the leach once steady state operation is achieved due to iron in ore being sufficient.

The operating costs per pound are estimated to average \$32.96/lb U_3O_8 over the first 5 years of production and \$34.72/lb U_3O_8 over the life of the mine, a significant reduction below the ECS costs of \$34.76 and \$38.20/lb U_3O_8 respectively. This represents a notable achievement given the general trend of escalating costs widespread across the mining industry, particularly in the areas of labour, energy and consumables.

At the forecast average realized uranium price ($65 /lb) over this 15 year period, it is estimated that Forsys will receive substantial positive net cash flows from its 100% ownership of the project. The economic analysis results in an estimated pre-tax net present value (NPV) at a discount rate of 8% to Forsys of $623.2 million. Using the initial investment and operating cash flows from inception, the pre-tax internal rate of return (IRR) is estimated to be 32%.

The total project estimated life of mine capital costs including construction, expansion and sustaining capital is $611.3 million. The cost to construct including all mining, processing and owners costs is estimated to be $432.0 million.

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

The grade of the deposit is relatively low, throughput is high but this is offset by relatively low Capex and operating costs. The strength of the project is confirmed by the sensitivity analysis which shows it robustness even at low long term contract prices at or about $50/lb U_3O_8.

2. INTRODUCTION

2.1 REPORT RESPONSIBILITIES

In June 2014, AMEC Australia West Pty Ltd (AMEC) was engaged by Forsys Metals Corp (Forsys) to conduct a definitive feasibility study (DFS) on its Norasa Project in Namibia. In January 2015, AMEC merged with another group and is now known as Amec Foster Wheeler. This report shows the findings of this study.

The DFS considers the entire project scope including geology, mining, process plant design, operating cost, capital cost, tailings disposal and project economics.

While the DFS has been issued by AMEC it contains sections from sources other than AMEC.

AMEC were responsible for the following sections:

Section 13: Mineral Processing and Metallurgical Testing
Section 17: Process Design and Development

A number of sections comprised input from both AMEC and Forsys. AMEC contribution to these dual responsibility sections was limited to the process plant design and development and related infrastructure. These sections are as follows:

Section 1: Executive Summary
Section 2: Introduction
Section 4: Property Description and Location
Section 21: Capital and Operating Costs
Section 24: Other Relevant Data and Information
Section 25: Interpretation and Conclusions
Section 26: Recommendations

Forsys was responsible for the following sections:

Section 5: Accessibility, Climate and Infrastructure
Section 6: History
Section 7: Geological Setting and Mineralisation
Section 8: Deposit Types
Section 9: Exploration
Section 10: Drilling
Section 11: Sample Preparation, Analysis and Security
Section 12: Data Verification
Section 14: Mineral Resource Estimate
Section 15: Mineral Reserve Estimate
Section 16: Mining Methods
Section 18: Project Infrastructure
Section 19: Market Studies and Contracts
Section 20: Environmental Studies, Permitting, and Social or Community Projects
Section 22: Economic Analysis

Section 23: Adjacent Properties

This report complies with disclosure and reporting requirements set forth in the Toronto Stock Exchange Manual, Canadian National Instrument 43-101, Companion Policy 43-101CP, and Form 43-101F1. Mineral Resource and Mineral Reserve classifications conform to those adopted by the Ontario Securities Commission on 24 June 2011.

2.2 PRINCIPLE SOURCES OF INFORMATION

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

- Information provided by qualified geologists employed by Forsys regarding the geology, drilling, sampling and other exploration procedures and processes adopted by the Company.
- Metallurgical testwork undertaken by recognised testwork laboratories, notably SGS in Johannesburg, South Africa.
- Information from Forsys personnel in relation to past history and previous studies on the Norasa Project not undertaken by AMEC.
- Snowden and associated entities in the group undertaking field observations, reports and data obtained during field trips in 2007 and 2008.
- Digby Wells and Associates originally prepared Valencia Project Environmental and Social Impact Assessment and Environmental and Social Management Plan in April 2008.
- Water Sciences cc completed groundwater modelling to define groundwater conditions around the Valencia mining area, particularly in the Khan River valley.
- Various published historic, technical and scientific papers and reports.
- Digital exploration data.
- Published information relevant to the Norasa Project area and the region in general.

2.3 PARTICIPANTS

The following qualified persons (QPs) have been involved in compilation of the NI 43-101 report:

AMEC

- Peter Nofal – Technical Director Studies for AMEC. Responsible for Sections 13 and 17 and partially responsible for sections 1, 2, 21, 24, 25 and 26 that relate to the process plant and related infrastructure.

Forsys

- Dag Kullmann – General Manager, Forsys Namibia. Responsible for Section 3, 4, 5, 15, 16, 18, 19, 20, 22 and 23 and partially responsible for sections 1, 2, 21, 24, 25 and 26 that relate to mining and infrastructure.

- Martin Hirsch – Chief Geologist, Forsys Namibia. Responsible for Section 6, 7, 8, 9, 10, 11, 12 and 14 and partially responsible for sections 1, 15.1.1 and 25 that relate to geology,

2.4 QUALIFICATIONS AND EXPERIENCE

AMEC is an international engineering company, with a strong world-wide background in mineral resource engineering. AMEC specialise in resource and mining studies, process design, engineering, cost estimation and feasibility studies for the minerals industry, focusing on gold, base metals, iron ore, mineral sands and uranium, including extensive involvement with the Langer Heinrich and Husab uranium projects in Namibia.

- Peter Nofal, Technical Director Studies for AMEC, is responsible for the process plant related sections of the study. Mr Nofal is a Fellow of the Australasian Institute of Mining and Metallurgy. Mr Nofal has 35 years of operating and management experience in the minerals industry.

- Deon Van Tonder, Process Consultant for AMEC, is responsible for process design of the leach and associated downstream hydrometallurgical plant. He visited the testwork laboratory and interpreted the associated findings in support of the process design. Mr van Tonder is a member of the Australasian Institute of Mining and Metallurgy.

Forsys qualified geologists and engineers managed the geology, drilling, sampling and other exploration procedures and processes adopted, as well as, mine schedules and related capital and operating costs

- Dag Kullmann, General Manager and Mining Manager for Forsys, is responsible for the mine plan, capital and operating costs and owners costs. Mr Kullmann, M.Sc. Mining Engineering from the University of Alberta, a Fellow of the Southern African Institute of Mining and Metallurgy (SAIMM), is the designated Qualified Person responsible for the reporting of Mineral Reserves. Mr Kullmann has sufficient experience in the assessment and application of modifying factors required for the determination of reserves for open pit operations to qualify as a QP under NI 43-101.

- Martin Hirsch, Chief Geologist for Forsys, is responsible for the geology and mineral resource definition. Mr Hirsch, M.Sc in Geology and a Member of the British IMMM, is the designated Qualified Person responsible for the Company's exploration programs and reporting of Mineral Resources. Mr Hirsch has sufficient experience that is relevant to the style and mineralization, type of deposit and the use of radiometrics in resource estimation to qualify as a Qualified Person under NI 43-101.

2.5 INDEPENDENCE

AMEC and their employees are considered independent from Forsys as outlined under Section 1.4 of the Instrument. None of the parties have any material interest in Forsys or related entities or interests. Their relationship with Forsys is solely one of professional association between client and independent consultant. The report was prepared in return for fees based upon agreed commercial rates and the payment of these fees is in no way contingent on the results of the relevant sections.

2.6 PROJECT BACKGROUND

The Norasa Project consists of two separate but geographically close alaskite hosted uranium deposits namely Valencia and Namibplaas.

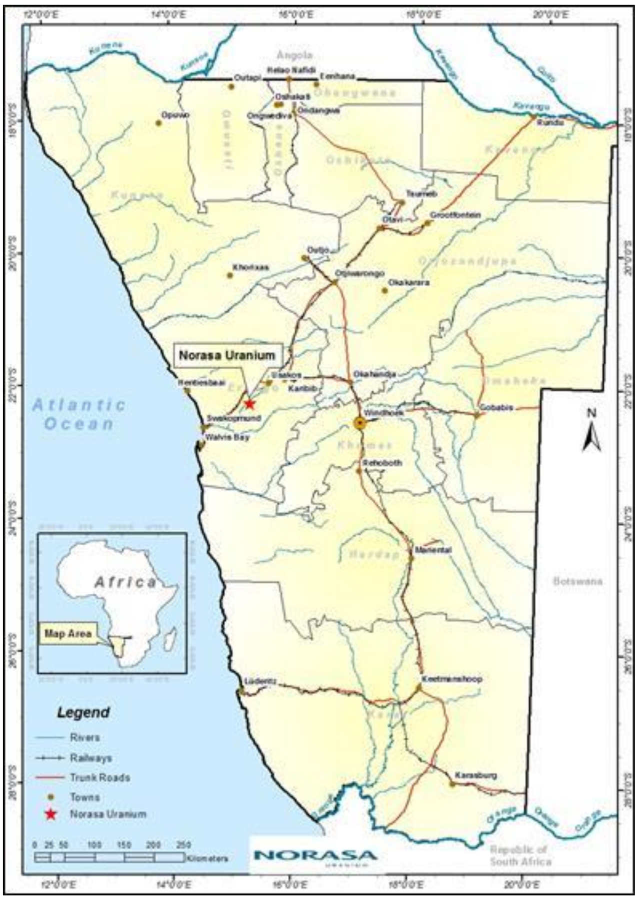


Figure 2.1 Map of Namibia

Figure 2.2 Project Location

AMEC previously conducted a Feasibility Study considering just the Valencia Project in 2008 and has also conducted a number of smaller Engineering Cost Studies on the Valencia Project subsequent to the 2008 Feasibility Study to examine various process plant options to potentially enhance the Project. This DFS was conducted to update the 2008 Feasibility Study to incorporate the options that indicated a positive outcome on the Project.

Early stage development has commenced at the Norasa Project (100% equity) located in the politically stable country of Namibia, Africa. The Valencia deposit is situated approximately 80km from the coastal town of Swakopmund. It is located 35 km along strike to the world class Rössing Uranium Mine which has been in production for over 30 years and 50 km north of the Langer Heinrich Uranium Mine. Norasa's Valencia mine is fully permitted including all environmental approvals. In August 2008, the Namibian government issued a 25 year Mining Licence for the mine. In the second quarter of 2010 an important access road was completed, joining the Valencia mine site to the main highway. Construction of power infrastructure has commenced in conjunction with NamPower and the Company has entered into a Memorandum of Understanding with NamWater to obtain water required for the mine construction phase and production.

2.7 ABBREVIATIONS

A description of technical abbreviations used in this report is provided below.

Abbreviation	Description
%	Percent
ADU	Ammonium diuranate
Ag/AgCl	Silver – silver chloride reference electrode (used for ORP measurement)
AMD	Acid mine drainage
Capex	Capital cost expenditure
COG	Cut-off grade
COS	Crushed ore stockpile
CRM	Certified reference material
DDH	Diamond drill hole
DFS	Definitive Feasibility Study
ECS	Engineering Cost Study
EIA	Environmental impact assessment
EMP	Environmental management plan
EPCM	Engineering, procurement and construction management
EPL	Exclusive Prospecting Licence
FS	Feasibility Study
GFL	Gold Fields Laboratory
GFSA	Gold Fields of South Africa
GPS	Global positioning system
H_2O_2	Hydrogen peroxide or peroxide
H_2SO_4	Sulphuric acid
IRR	Internal rate of return
IX	Ion Exchange
L	Litre
LoM	Life of Mine
MEL	Mechanical equipment list
MET	Ministry of Environment and Tourism
ML	Mining Licence
MME	Ministry of Mines and Energy
µm	Micron (1/1000 of a mm)
mm	Millimetre
MnO_2	Manganese dioxide
MPBIX	Moving packed-bed ion exchange
NIMCIX	Counter-current IX technology developed by NIM (National institute of Metallurgy, now Mintek)
NAD	Namibian dollar
NPV	Net present value
OEM	Original equipment manufacturer
Opex	Operating cost expenditure
ORP	Oxidation Reduction Potential (redox potential)
QA/QC	Quality assurance and quality control

Abbreviation	Description
P_{100}	Size at which 100% of the material passes
P_{80}	Size at which 80% of the material passes
PD	Percussion drilled holes
PFD	Process flow diagram
pH	Measure of hydrogen ion concentration
PLS	Pregnant leach solution
PP	Polypropylene
ppm	Parts per million
PSD	Particle size distribution
Rad Sort	Radiometric sorting
RC	Reverse circulation
RSF	Residue Storage Facility
ROM	Run-of-mine
SAG mill	Semi-Autogenous mill
SG	Specific gravity (density ratio of substance to that of water at 4°C)
SMC	Comminution test developed by Dr Steve Morrell
SRM	Standard reference material
SX	Solvent extraction
SysCAD	Proprietary process modelling software by Kenwalt
TDS	Total dissolved salts
U_3O_8	Uranium oxide
UCS	Unconfined compressive strength
XRD	X-ray diffraction (analytical procedure)
XRF	X-ray fluorescence (analytical procedure)
ZAR	South African Rand

Section 3
Reliance on Other Experts

3. RELIANCE ON OTHER EXPERTS

3.1 OTHER EXPERTS

Other individuals have provided information (both current and from previous work) that has been included in this report, but are not co-authors of this report and hence are not acting as Qualified Persons under NI 43-101 guidelines for this report. They have however the necessary qualifications and experience to provide input and opinions which have been incorporated into this report.

- Dr. Patrick Walker, Xstract Mining Consultants, Perth and formerly of Snowden has provided all the pit slope data analyses and design parameters. His initial involvement was with the Valencia pit designs (Snowden 2009) and visited the site in 2007. He also completed the slope design work for Valencia East and Namibplaas (Xstract 2013, Xstract 2014). This work is summarised in Section 16.1 of this report.
- Werner Moeller, VBKom Consulting Engineers, Windhoek conducted the pit optimisation, design and scheduling work as reported in Section 15. The work was guided by Mr Kullmann, QP for this report.
- Johan Cornelissen, National Environmental Health Consultants, Swakopmund is currently appointed as Valencia's legally appointed environmental consultants and is responsible for reviewing Chapter 20 of this report. He has been on site on several occasions with his last visit on 5 December 2014 to conduct the environmental year end inspection.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 NAMIBIA

Namibia is the 15th largest country in Africa and has a total area of approximately 825,418 km^2. Located on the south-western coast of Africa between Angola, Botswana and South Africa (Figure 2.1), Namibia is home to approximately 2.2 million people from 11 ethnic groups *(Wikipedia, 2011)*. English is the official language with Afrikaans, Oshiwambo, Damara/Nama, Kwangali, and Otjiherero being the predominant languages spoken in homes *(National Planning Commission, 2002).*

The capital city of Namibia is Windhoek. Located in the Khomas administrative region, Windhoek is also the largest urban area in Namibia. There are 12 other administrative regions in Namibia: Caprivi, Erongo, Hardap, Karas, Okavango, Kunene, Ohangwena, Omaheke, Omausati, Oshana, Oshikoto and Otjozondjupa. Norasa is in the Erongo region.

Namibia is very sparsely populated, with a density of 1.7 people per km^2 *(PWC, 2010).* This is partially attributable to the geography of Namibia; along the western coast is the Namib Desert and to the east is the Kalahari Desert. Between these deserts is a central plateau running north to south. In the northwest, the regions of Okavango and Caprivi are the high rainfall areas and support the densest population.

The climate of Namibia is generally arid, with the average rainfall for most areas along the coast less than 50 mm/a *(IIASA, 2011).* To the northeast, the Kavango and Caprivi areas are much wetter, with average annual rainfall close to 600 mm. Rainfall occurs predominantly between November and March.

Over 42% of the land area of Namibia is protected as either National Parks (17%) or conservancies. Permits are required to enter most national parks, and concessions to start new businesses in the area must be granted by the Ministry of Environment and Tourism.

4.2 ERONGO REGION

Named after the Erongo mountain range in the north of the region, Erongo is located on the central western coast of Namibia. The region extends north to the Ugab River, south to the Kuiseb River region, while to the east are the Khomas and Otjozondjupa regions (Figure 2.1).

The primary industries in the Erongo region are agriculture, fishing, mining and manufacture. Swakopmund, the capital city of the region, was established at the mouth of the Swakop River, one of the intermittent rivers in Namibia. Founded in 1892 as the main harbour for German South West Africa, the harbour is shallow and prone to silting. After South West Africa (SWA) was annexed by South Africa in 1915, Swakopmund was transformed into a tourist resort town. With the opening of the Rössing uranium mine 70 km inland from Swakopmund in the 1970s, the town underwent another change and is now a combination tourist and industrial town.

Walvis Bay, 35 km south of Swakopmund by road, is the main commercial harbour for modern Namibia. Each year, the port handles approximately 5 Mt of cargo from 3,000 vessels, and hosts approximately

250,000 containers (*Namport, 2011*). Walvis Bay also has a large fishing industry, with a separate harbour home to the fishing fleet and a strong fish processing industry.

Approximately one-third of the Erongo region falls within national parks. The northern extent of Namib-Naukluft National Park covers the southernmost parts of the region, including the Project area and extending up to Swakopmund and the Swakop River. Along the remaining coast is the Dorob National Park. Another third of the region is part of two conservancies, known as Gaingu and Tsiseb.

4.3 HISTORY

Namibia, formerly the German colony of SWA, was placed under South Africa's administration by the League of Nations after World War I. When the United Nations (UN) succeeded the League of Nations in 1946, it formally requested South Africa place SWA under a trusteeship, but South Africa refused. In 1966, South Africa's mandate to administer SWA was revoked by the UN General Assembly, but South Africa maintained its presence in the country, continuing to treat SWA as the fifth province of South Africa, and further developing the port at Walvis Bay and other infrastructure to support the growing mining industry.

In the same year the South West Africa People's Organisation (SWAPO) launched a war of independence for SWA. In 1968, SWA became known internationally as Namibia when the UN General Assembly changed the territory's name by Resolution 2372 (XXII). In 1988, South Africa agreed to end its administration of Namibia in accordance with the UN peace plan, and Namibia finally gained independence in 1990. Samuel Nujoma, the SWAPO leader, was elected president in 1989 and installed in 1990. He was subsequently elected for a further two 5-year terms.

4.4 NATURAL RESOURCES

Namibia has known deposits of diamonds, copper, uranium, gold, lead, tin, lithium, cadmium, zinc, salt and vanadium, and possible deposits of oil, coal and iron ore. It also has natural gas, hydropower and a significant fishing industry.

4.4.1 Mining

Mining accounts for approximately 8% of Namibia's GDP in 2011, but provides more than 50% of foreign exchange earnings. Namibia is the world's fifth-largest producer of uranium (3,258 t U in 2011 – *World Nuclear Association, 2012, A*) and the producer of lead, zinc, tin, copper, gold, silver and tungsten. Diamonds, various quartzes including amethysts, blue laced agate, chalcedony, and other semi-precious minerals and stones including Mandarin Orange Garnets, tourmaline and dioptase making up the remaining mining income.

The country has a long mining history, with both copper and gold mined in the country from as early as 1850. Diamonds, however, remain Namibia's best known resource, from alluvial mining along the Orange River to surf and shallow offshore mining between Oranjemund to Luderitz and the infamous Skeleton Coast and, increasingly, in deep offshore waters.

Diamond production dropped dramatically in 2009 due to the onset of global financial weakening with an annual production of around 0.9 M carats/a, but bounced back in 2010 with 1.48 M carats produced *(Kaira, 2011).* Production is declining due to a variety of factors including the depletion of onshore diamond deposits and the need for further investment in infrastructure for the offshore deposits.

In 2009, uranium was the only extractive industry which increased production, maintaining Namibia's position as the fourth largest producer of uranium. Growth continued in 2010 but contracted in 2011, with production reducing from 4,496 t to 3,258 t U.

In the Erongo Region, there are a number of major existing and prospective developments, including the Rössing, Langer Heinrich and Trekkopje uranium mines, and the Etango Project. December 2011 saw the awarding of a mining licence to Swakop Uranium for the Husab Project *(Extract Resources, 2011)*.

Namibia's copper and zinc production has reduced dramatically over the years with only the Skorpion and Rosh Pinah zinc mines in operation in 2009. The Otjihase and Matchless Copper mines reopened in early 2011 after being forced to close in 2008.

More recently, the government re-focussed on the gold mining sector in which the Navachab Gold mine is the only producer with 65,000 oz of gold in 2009.

To help promote exploration, the Geological Survey of Namibia has assembled a large database which is widely accessible. As part of its push to attract foreign investment, the Government planned to turn the former South African naval port of Walvis Bay into a free-trade zone. Namdeb had lifted restrictions on prospecting activities in the grounds over which the company previously held exclusive rights. The "Diamond Area 1" has good base metal potential, following development of the Skorpion zinc mine located within the area.

The Namibian Custom Smelter, owned by Dundee Precious Metals Inc., is located at Tsumeb in the northeast of Namibia. Prior to Weatherly Mining International selling the plant, it processed copper from Namibian mines. Now it processes only imported copper ore, including the high arsenic concentrates from Dundee's Bulgarian mines.

4.4.2 Mining Regulations

In Namibia, all mineral rights are vested in the state. Mining in Namibia is regulated by the Minerals (Prospecting and Mining) Act 1992, the Environmental Management Act 2007, and the Minerals Policy of Namibia (2003). Uranium mining is further regulated by the Atomic Energy & Radiation Protection Act 2005, and the associated Atomic Energy Board of Namibia, and the National Radiation Protection Authority.

The Minerals (Prospecting and Mining) Act of 1992 regulates the mining industry in the country. The Mining Rights and Mineral Resources Division in the Directorate of Mining is usually the first contact for investors as it handles all applications for, and allocation of, mineral rights in Namibia. Several types of mining and prospecting licences exist as follows:

- Non-Exclusive Prospecting Licences, valid for 12 months, permit prospecting non-exclusively in any open group not restricted by other mineral rights.
- Reconnaissance Licences allow regional remote sensing techniques, and are valid for 6 months (renewable under special circumstances) and can be made exclusive in some instances.
- Exclusive Prospecting Licences (EPLs) can cover areas not exceeding 1,000 km^2 and are valid for 3 years, with two renewals of 2 years each. Two or more EPLs can be issued for more than one mineral in the same area.

- Mineral Deposit Retention Licences (MDRLs) allow successful prospectors to retain rights to mineral deposits which are uneconomical to exploit immediately. MDRLs are valid for up to 5 years and can be renewed subject to limited work and expenditure obligations.
- Mining Licences (MLs) can be awarded to Namibian citizens and companies registered in Namibia. They are valid for the LOM or an initial 25 years, renewable up to 15 years at a time.

New measures were introduced in 2000 to regulate diamond processing and trading to encourage prospecting in areas no longer required for diamond exploration and mining. The Diamond Act (1999) partially deregulates trading in rough diamonds, and applies up to the limits of Namibia's 220 km exclusive economic zone. The general granting of licences for mining and exploration will continue to take place under the provisions of the Minerals Act. The new legislation, however, outlines the creation of a diamond permit, which is required for individuals wanting to possess, receive or dispose of rough diamonds, unless handling goods as employees or as registered authorised representatives or corporate principles.

Namibia has also signed or ratified several agreements on mining including the 1997 SADC Protocol on Mining.

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

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

4.5 NORASA

Valencia is situated on the farm "Valencia 122", which is located approximately 75km south-west of the town of Usakos in central-west Namibia (Figure 2.2), covers an area of 735.6 ha, and is registered in the name of Valencia Uranium (Pty) Ltd.

ML 149 was converted from EPL 1496 on 27 June 2008 and is valid for 25 years from date of issue by the MME and is renewable.

The perimeter of ML 149 is defined by the coordinates listed in Table 4.1**Error! Reference source not found.** and which are shown in Figure 4.1.

Table 4.1 Boundary coordinates for ML 149

ID	Longitude	Latitude
1	15.23828368	-22.32625528
2	15.25225024	-22.33894483
3	15.22720250	-22.36416441
4	15.21527297	-22.34900956

Namibplaas is located 7.5km northeast of the Valencia deposit on the farm “Namibplaas”. The perimeter of the grant is listed in Table 4.2**Error! Reference source not found.** and shown in Figure 4.1. The total surface area is 1,268.9 ha. The EPL 3638 was renewed for another 2-year period, expiring on 6 November 2015.

Table 4.2 Boundary coordinates for EPL 3638

ID	Longitude	Latitude
1	15.24098096	-22.30127056
2	15.29048686	-22.26535988
3	15.29090991	-22.30992514

(Bessel 1841 Spheroid)

There are no historical environmental liabilities for either the Valencia or Namibplaas properties. There are no royalties payable to any third party in relation to the licences except those to the Namibian Government’s MME as described in Section 22 of this report.

4.5.1 Permits and agreements

The entire Valencia mineral licence area is located on privately held farm land. As required by law, an agreement must be entered into between a mineral licence holder and the landowner to allow exploration activities. In order to progress a project to mine development, a compensation agreement is required to offset the effects of the operation.

In April 2009, Valencia entered into a compensation agreement with the owner of the farm Valencia 122 in relation to Section 52 of the Minerals Act of 1992 granting Valencia unrestricted use of the land on and around ML 149 covering an area of 3,327 hectares. A similar agreement was reached with the owners of the neighbouring farm Bloemhof to the south (for an area of 594ha), for the construction of additional infrastructure and for primary access to the Valencia site.

These agreements have allowed Valencia to fully plan for the necessary infrastructure required to support mining operations. This infrastructure has been approved by the MME as the operation’s Accessory Works and includes *inter alia* the main pit, waste dumps, tailings dump, pipeline, power lines, roads, process plant explosive magazines, etc. The construction camp / cum operations village have also been approved. Environmental clearance was obtained for all operations relating to Valencia, although some of the amendments to the Valencia plan may require addition assessments.

Valencia is fully permitted to allow commencement of construction and mining operations. There are no other requirements for Valencia to commence operations; legal, administrative or environmental.

The Namibplaas mineral licence area is also completely located on private farm land. The majority of the licence (and the entire prospecting area of interest) is on the farm Namibplaas. There is currently an access agreement in place with the landowner of Namibplaas to allow prospecting activities to continue as required. To take the Namibplaas project into the development and then construction phases, an EIA/EMP needs to be completed, a compensation agreement entered into with the landowner and approval received for Accessory Works.

The environmental studies for Namibplaas are underway, with baseline monitoring of groundwater, air quality, noise, vegetation and archaeological studies already completed. This work is being done as part of the Norasa Project and is taking the form of an amendment to the original Valencia EIA/EMP, a process that has been approved by the Ministry of Environment and Tourism.

Table 4.3 Permits received for the Norasa project.

Permit	Issued By	Date received
Mining Licence (ML 149)	Ministry of Mines and Energy	23 June 2008
Exclusive Prospect Licence (EPL3638)	Ministry of Mines and Energy	7 Nov 2011
Accessory Works ML 149	Ministry of Mines and Energy	29 May 2009
Environmental Clearance	Ministry of Environment and Tourism	11 April 2013
Exclusive Prospecting Licence (EPL 3638)	Ministry of Mines and Energy	7 Nov 2013
Petroleum Consumer Installation	Ministry of Mines and Energy	12 Oct 2012
Radiation Management Plan (ML 149)	Ministry of Health and Social Services	9 Dec 2010

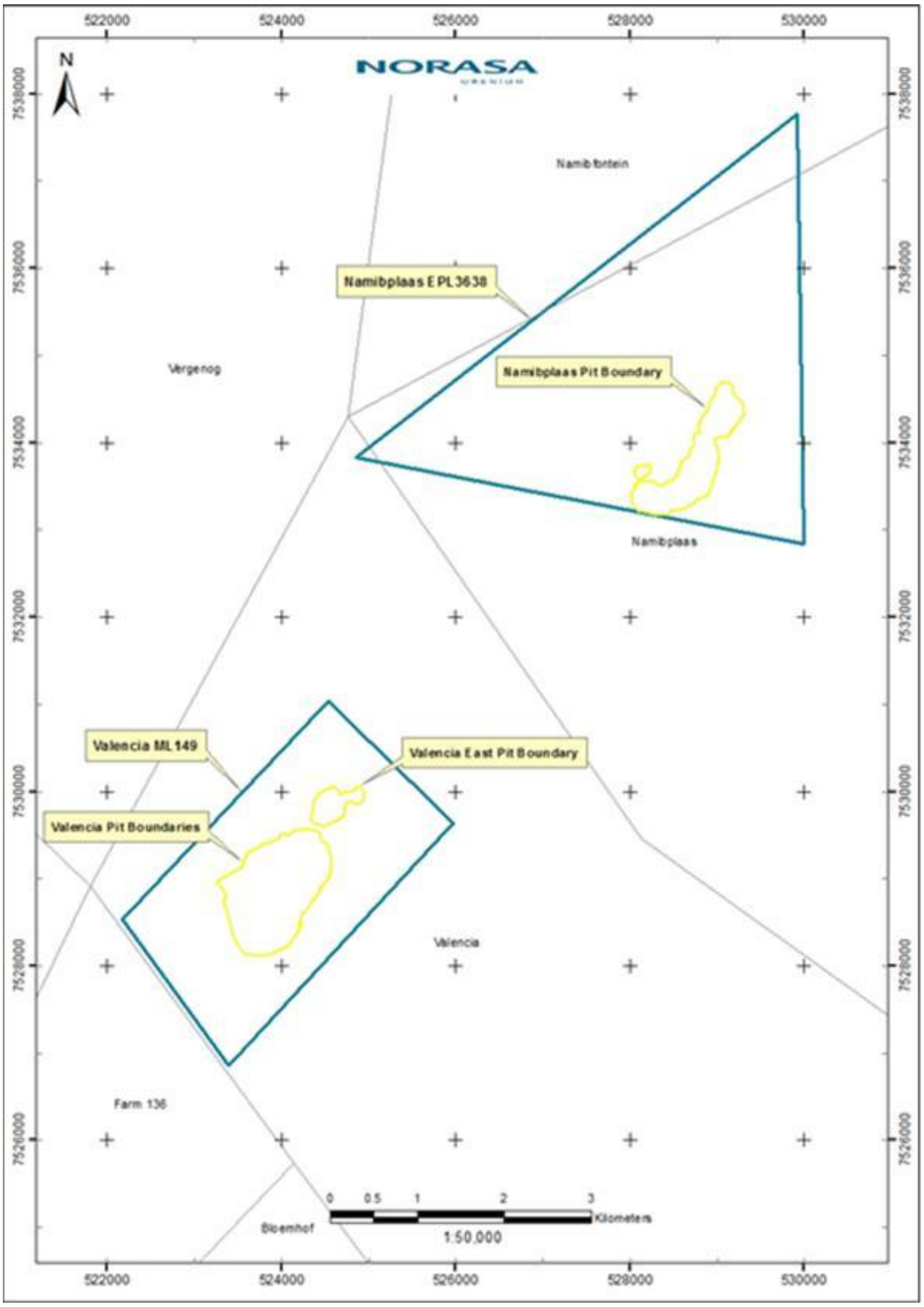


Figure 4.1 Locality map of Valencia ML 149 and Namibplaas EPL 3836.

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

Norasa is accessed from the main east-west B2 Highway, a tarred road linking the coastal towns of Walvis Bay and Swakopmund with the capital city of Windhoek in the interior (Figure 2.1). From the B2 Highway, Valencia constructed an industrial gravel road of 28km to the Norasa site. The turn-off from the highway is located 68km from Swakopmund.

Windhoek and Walvis Bay have international airports with daily flights to many other African and European destinations. Windhoek and Walvis Bay are also linked by rail. The nearest rail siding is located in the town of Arandis, located about 45km by road from the Norasa site.

5.1 CLIMATE

The climate of the project area, and region, is desert with annual rainfall of between 14mm and 150mm mostly in late summer, the period February through to March (Labuschagne, 1979). Rainfall of short duration and a high intensity may occur. Vegetation is sparse with stunted grasses and small trees. The topography is fairly rugged with an average elevation of 725m above mean sea level with approximately a 40m range in elevation around the deposit. Temperatures recorded in the area range between 4°C and 40°C (Berning, 1986). The operating season is 12 months of the year.

Water is mainly found only as sub-flow beneath the streambeds of the larger streams, e.g. the Khan River 4.5km to the north of Valencia. In some cases, dissolved salts render the water non-potable. During the 1973 to 1977 drilling campaign conducted by Trekkopje Exploration, water was extracted from a fountain in the Khan River and was transported over a route of 27km, although the fountain is located only 4.5km in a straight line due to the rugged terrain this route is not negotiable by a vehicle. Potable water was obtained from a borehole at the Valencia farmhouse, which is situated 4.5km to the south-east (Labuschagne, 1979).

5.2 PORT OF WALVIS BAY

Located half way down the coast of Namibia, with direct access to principal shipping routes, Walvis Bay is a natural gateway for international trade[1], located 140km from Norasa.

Walvis Bay is Namibia's largest commercial port, receiving approximately 3,000 vessel calls each year and handling about 5 million tonnes of cargo. It is a sheltered deep-water harbour benefiting from a temperate climate.

The container terminal can accommodate grounds slots for 3,875 containers with provision for 482 reefer container plug points. The terminal can host about 250,000 containers per annum.

The proposed Walvis Bay Port Expansion Project will include the construction of a new container terminal incorporating an additional 40 hectares of land with a quay length of 2,100m. The new container terminal will accommodate a capacity of 650,000 TEU (twenty-foot equivalent unit) per annum and will complement the existing 350,000 TEUs per annum. Upon completion which is anticipated by

[1] http://www.namport.com.na/

2017, the new container terminal will realise a deep water depth of 16m which will be able to accommodate 5,000 TEU vessels enabling large vessels to enter the port of Walvis Bay.

The Project will also increase the port’s bulk and break-bulk handling capacity by freeing up the existing container terminal to become a multi-purpose terminal. This will open up the port for increase scope to accommodate a wide range of additional bulk cargo vessels.

5.3 WATER SUPPLY

AREVA Resources Namibia built the first seawater desalination plant in Southern Africa. Located at Wlotzkasbaken, 30km north of Swakopmund, it was intended to supply all the water that will be required at the Trekkopje mine, located about 40km from there in the desert. However, since the completion of the desalination plant, the Trekkopje operation has been put on care and maintenance for an indefinite period.

Inaugurated on April 2010, the plant was designed to produce 20 Mm^3 of potable water per year using rotary filters, multi-stage ultrafiltration, reverse osmosis, and chemical treatment. The Erongo desalination plant will continue in operation during the mine’s care and maintenance program. Part of the water produced is being sold to the national water distribution company, NamWater, to supply potable water to local industries in the Erongo Region.

The nearest bulk water supply point to Norasa is the Rössing mine reservoirs, located 24km to the WSW. Although this infrastructure belongs to NamWater, it does provide Rössing with their only local water storage facility and is essentially dedicated to the mine. The pipeline supplying these reservoirs extends 55km from the main Swakopmund reservoirs, the main water distribution point in the central coastal area. Valencia has been informed that although the pipeline itself can handle enough water to provide Norasa with its water requirements, in addition to its current customers, the pumping system will need to be upgraded.

5.4 POWER SUPPLY

The central coastal area is supplied with electricity through the national grid by a ring feed connecting the country’s interior region (capital city of Windhoek) and the northern area from where much of the country’s supply is transmitted from. Two main 220kV transmission lines (recently upgraded to meet the growing demand of this coastal area) pass within 10km NW of the Norasa site. The nearest power off-take point that can supply Norasa is the Khan Substation, located 25km to the north. A transmission route of nearly 30km has been laid out by NamPower. Power distribution to the mine is planned to be a 220kV transmission line and could form part of the regional expansion and strengthening of the coastal power supply.

6. HISTORY

This section summarises the ownership history of Valencia and Namibplaas based on the available information. Any missing periods in the ownership history are not considered to be of material significance to the current ownership situation. No information regarding ownership of prospecting licences for Valencia or Namibplaas project areas prior to 1972 is available.

Gold Fields of South Africa Limited (GFSA) was granted the Prospecting Grant M46/3/499 in October 1972. This grant covered portions of the farms Vergernoeg 92 (19,852 ha), Namibfontein 91 (292 ha), Namibplaas 93 (660 ha), Valencia 122 (2,085 ha) and Trekkopje 120 (5,150 ha). In total 28,039 ha was included in the prospecting grant. The grant was valid for a period of two years, and could be renewed if application was made three months prior to its expiry. In June 1973 GFSA ceded the grant to Trekkopje Exploration and Mining Company (Pty) Ltd (Trekkopje Exploration), a wholly owned subsidiary of Gold Fields Mining and Development Ltd. Trekkopje Exploration maintained the grant and renewed it every two years, as the last available information regarding the Prospecting Grant M46/3/499 is a report by Trekkopje Exploration in support of a renewal application dated 20 July 1982 (Bertram, 1982b).

Forsys Metals Corp (Pty) Ltd was incorporated on May 13, 1985 under the Business Corporations Act (Ontario) ("OBCA") in Canada with the primary public listing on the Toronto Securities Exchange (TSX:FSY). Secondary listings include the Frankfurt and Namibia Stock Exchanges. The company is engaged in the business of acquiring, exploring and developing mineral properties, either independently, or through joint ventures with historical acquisitions detailed in Sections 6.1 and 6.2.

6.1 VALENCIA

In 2005, Forsys acquired a 90% interest in Valencia Uranium (Pty) Ltd. and acquired the remaining 10% in 2007.

The status of the Prospecting Grant for the Valencia project between July 1982 and October 1988 is unknown for the purposes of this report; however an approval by the Department of Economic Affairs for a renewal application by Trekkopje Exploration, dated 25 October 1988 was obtained (Dept. Econ. Affairs, 1988). This renewal was for Prospecting Grant M.46/3/1496 and was for a further period of two years starting from 29 November 1988.

Due to the upcoming change in the mining legislation that was promulgated in 1992 (GRN, 1992) and the lack of economic viability of the Valencia project, the Prospecting Grant was considered too large by the MME. It was suggested by the MME that the grant should be reduced in area in order to be accommodated under a "holding-grant" or a MDRL. Prior to finalisation of the legislation the MME suggested a smaller area of 500 ha and included a waiver of any expenditure or work obligation on the condition that:

- Gold Fields Namibia submitted a project prospectus;
- Gold Fields Namibia actively promoted third party interest in the project, and kept the government informed of any progress in this matter (GRN, 1991a).

The MME approved the extension for a further two years of Prospecting Grant M46/3/1496 reckoned from 29 November 1990 and included a waiver of any expenditure or work obligation (GRN, 1991b). It was not indicated whether this was the grant of reduced area.

The status of the prospecting grant between November 1990 and November 1994 is unknown for the purposes of this report.

MDRL 1496 was granted to Tsumeb Corporation Limited (TCL) in November 1994 for a period of five years, which was transferred to Ongopolo Mining Limited (Ongopolo) in March 2000 and again in June 2005 to Tsumeb Exploration Company Limited.

MDRL 1496 was converted by Tsumeb Exploration Company Limited to EPL 1496 on 20 February 2007. Tsumeb Exploration Company Limited changed their name to Valencia Uranium Pty Ltd in November 2007. EPL 1496 was converted to ML 149 on 27 June 2008.

6.1.1 Historical exploration and resource estimates

The Valencia project has had several periods of exploration undertaken by previous owners. The original exploration work was initiated based on the results of a regional airborne radiometric survey conducted by the South African government in 1968. A detailed review of exploration history has been reported in detail in previous Technical Reports on this project, most recently by Snowden, June 2009. These reports are available through the Sedar website (http://www.sedar.com/) listed under the parent company Forsys Metals Corporation.

Historical resource estimates were made in 1979 and 1981. Details were also reported on previously. A feasibility study was completed by GFSA in 1989 and did not result in a favourable economic outcome. There are no records of historical production and no evidence of any mining production has been observed.

6.2 NAMIBPLAAS

Until March 2012, Forsys held a 70% shareholding in Dunefield Mining Company (Pty) Ltd, a Namibian registered company that was incorporated in November 2005. The Company acquired the remaining 30% interest in January 2013.

GFSA drilled 7 diamond drill holes on the property in the late 1970s and early 1980s, NA24-001 to NA24-007, for a total of 1,665.9 metres.

The drilling completed by GSFA was the last phase of exploration undertaken at Namibplaas until Valencia commenced drilling in 2008. No historic estimates of the Namibplaas deposit have been undertaken.

Exclusive Prospecting Licence, EPL 3638, for the Namibplaas project was first granted to Dunefield Mining Company (Pty) Ltd in November of 2006. The exploration licence has been renewed on a number of occasions.

6.3 NORASA URANIUM PROJECT

Forsys announced the consolidation of these 2 uranium projects into a single project, now known as the Norasa Uranium Project.

7. GEOLOGICAL SETTING AND MINERALISATION

7.1 REGIONAL SETTING

Norasa is located in the Damara Orogen which is a Pan African - aged result of a "Wilson Cycle" collision between the Kalahari Craton in the south and the Congo Craton in the north. It comprises a coastal branch along Namibia’s north coast into Angola, The Kaoko Belt, and an inland branch (Damara Belt) stretching from the Namibian coast north eastwards through to Zambia. The oblique collision closed an ancient seaway, the Damara Ocean, forcing together a varied collection of depositional environments. The sequence of tectonic and deformational periods which, followed by erosion, produced the strongly-zoned remnants of a continent-continent mountain chain root that we see today (Kröner, 1984; Brandt, 1985).

The Inland Branch (Damara Orogen) has been divided from north to south along NE-SW trending tectono-stratigraphic lineaments. These boundaries divide the Orogen into SW - NE trending Zones (Figure 7.1).

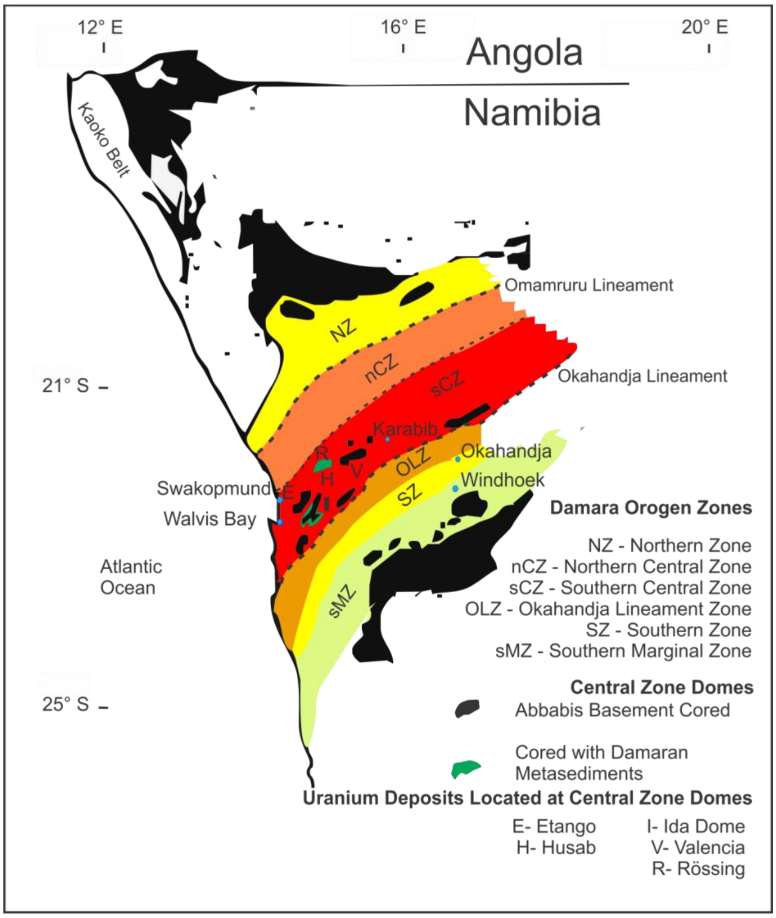


Figure 7.1 The Pan African Damara Orogen of Central, Namibia.

The regional orogeny is divided into a number of tectono-stratigraphic zones, bound by strong lineaments. Valencia (V) is shown in relation to the other three deposits of significance. The deposits occur consistently around prominent, Central Zone dome-like structures. After Miller (1983, 2008) and Kinnaird and Nex (2007); (Freemantle, G., 2010).

Metamorphic gradients vary between these zones and are increasing to granulite facies in the Central Zone, toward the more deeply eroded coastal region in the west (Goscombe, et al 2004).

The main Zones are as follows:

- A Northern Platform (containing the foreland basin of the Orogen)
- The Northern Zone (NZ),
- Central Zone (CZ)
- Okahandja Lineament Zone (OLZ)
- Southern Zone (SZ)
- Southern Marginal Zone (SMZ)
- Southern Foreland (SF)

Primary uranium mineralisation of significance is limited to the Central Southern Zone (sCZ) which hosts all the major primary uranium occurrences known today in Namibia.

Large volumes of U-bearing leucogranite intrude a limited stratigraphic-range, occasionally cross-cutting into basement but mainly into stratigraphic units directly above and below the Nosib - Swakop Group contact (Table 7.1).

Table 7.1 Stratigraphy of the Damara Sequence

SACS (unpublished)				LITHOLOGY
GROUP	SUBGROUP	FORMATION	Maximum thickness (metres)	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.
SWAKOP	KHOMAS	KUISEB	>3 000	
		KARIBIB	1 000	Marble, calc-silicate rock, pelitic and semi-pelitic schist and gneiss, biotite-amphibole schist, quartz schist, migmatite.
		CHUOS	700	Diamictite, calc-silicate rock, pebbly schist, quartzite, ferruginous quartzite, migmatite.
	Discordance			Marble, pelitic schist and gneiss, biotite-hornblende schist, migmatite, calc-silicate rock, quartzite, metaconglomerate.
	UGAB	RÖSSING	200	
	Discordance			
NOSIB		KHAN	1 100	Migmatitic, banded and mottled quartzofeldspathic clinopyroxene-amphibole gneiss, hornblende-biotite schist, biotite schist and gneiss, migmatite, pyroxene-garnet-gneiss, amphibolite, quartzite, metaconglomerate.
		ETUSIS	3 000	Quartzite, metaconglomerate, pelitic and semi-pelitic schist and gneiss, migmatite, quartzofeldspathic clyopyroxene-amphibole gneiss, calc-silicate rock, metarhyolite.
Major unconformity				Gneissic granite, augen gneiss, quartzofeldspathic gneiss, pelitic schist and gneiss, migmatite, quartzite, marble, calc-silicate rock, amphibolite.
ABBABIS COMPLEX				

It is at this stratigraphic level where the largest uranium reserves in Namibia are found; the Husab Mine, the Rössing Mine, the Valencia project and the Etango deposit at Goanikontes to name the most significant ones (Figure 7.1).

7.1.1 Structural Setting

The Central Zone is marked by Dome-and-Basin structures and is separated from the Southern Zone by the Okahandja lineament (Figure 7.1). The Omaruru lineament in turn separates the Central Zone from the Transition Zone in the north. The extensive granite emplacement associated with the domes in the Central Zone is not seen north of the Omaruru lineament.

Several phases of deformation are recognised in the Central Zone and are indicated by fold interference patterns such as that of the Rössing Mountain structure (Smith 1965). The main structural grain is now north-east and is due to an intense (F3) deformation. This was preceded by one or possibly two periods of folding. The early phases of folding produced overturned and recumbent structures and were accompanied by thrusting and shearing. The trend of early fold axial planes was roughly north-westerly. The later north-easterly F3 folds are upright but become overturned to the south-east as the Okahandja lineament is approached.

The basement (Abbabis Complex) has been deformed by ductile shearing in lower metamorphic grade areas and has taken part in the folding in higher grade zones.

A number of later, less intense fold phases occurred after F3 and produced folds oriented between north-east and north-west.

Of particular significance to the emplacement of the uraniferous granites is a post-F3 phase, F4, oriented north-north-east (Corner, 1982) which manifests itself in a prominent north-northeasterly-trending magnetic lineament which is termed the Welwitschia lineament (Figure 7.2). To the east of the Welwitschia lineament, the trend of fold axial planes of structures within the belt is mostly north-east. To the west, however, these structural directions are both north-east and north-north-east, with the latter direction prevailing as the coast is approached.

Corner (1982) considered this north-north-easterly direction to have an important bearing on the emplacement of the uraniferous alaskitic granites since firstly, the currently known occurrences are all located within the vicinity of the Welwitschia lineament and, secondly, the major fold axes of the domes and structures with which these occurrences are associated are parallel to this lineament rather than to the general north-easterly trend of the Central Zone (Figure 7.2).

Dome and basin structures are a feature of the Central Zone but their origin remains controversial. Smith (1965) ascribed them to interference folding whereas Jacob (1974), Sawyer (1978) and Barners and Hambleton-Jones (1978) believed that they have formed as a result of diapiric uprise at about the time of, and following, F3 deformation.

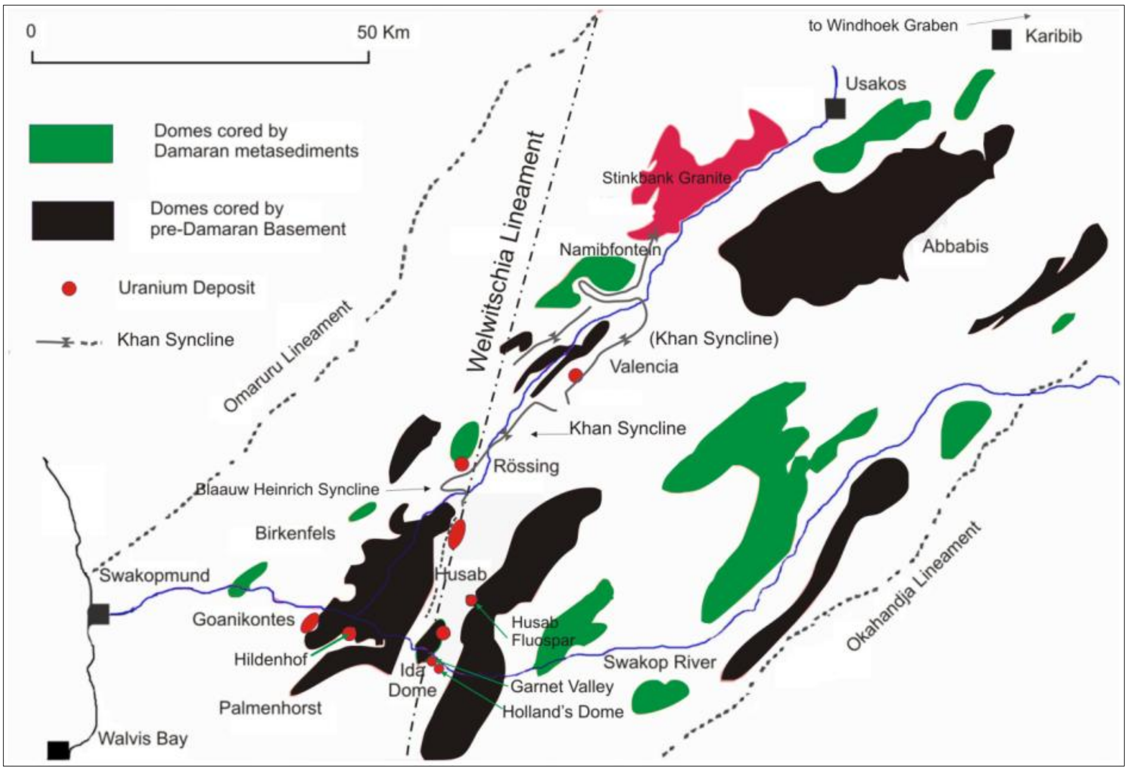


Figure 7.2 Domes of the Central Zone and associated primary uranium deposits.
Compiled after Corner (1981, 1983); Eberle et al. (1995); Anderson and Nash (1997); Basson and Greenway (2004) and Kinnaird and Nex (2007).

7.1.2 Metamorphism

More than one period of metamorphism occurred in the Central Zone (Kröner et al, 1978; Sawyer, 1978). An early metamorphism, dated at 665 + 34 Ma, predated widespread granite intrusion, produced migmatites, and accompanied the early periods of deformation (F1, F2). According to Sawyer (1978), this was followed by another period of metamorphism accompanying the F3 deformation, and was, in turn, followed by intrusion of various granitic rocks whose ages are of the order of 550 Ma.

A late- to post-tectonic thermal event accompanying the F4 deformation, around 470 Ma, in the Central Zone is indicated by Rb-Sr dating of gneisses of the Khan Formation and the Rössing mine alaskites (Kröner et al, 1978), and it is possible that K-Ar biotite ages of 520-450 Ma also reflects this event.

7.1.3 Uraniferous Granites

Economically important uranium mineralisation occurs in the late- to post-tectonic granitic rocks referred to as pegmatite (Smith, 1965), potash granite (Nash, 1971), alaskite (Berning et al, 1976) and Metamorphic Pegmatitic Granite (Cuney, 1980) in the literature. The current, more commonly used terms are now alkaline leuko-granites or alaskites. These granites are found only in the Central Zone and are confined to the areas of highest metamorphic grade. They are situated along the Abbabis Swell and often are associated with a Red Granite Suite. They also occur preferentially in and around anticlinal and dome structures and intrude into the Basement, the Nosib Group and lower Swakop Group, mainly below the prominent marbles of the Karibib Formation.

Kinnaird and Nex (2007) defined six different types of alaskites. They distinguish between 3 pre-tectonic (A, B and C type) and 3 syn- to post-tectonic variations (D, E and F type), of which only the latter ones are enriched in uranium. Table 7.2 below summarises alaskite characteristics in terms of their texture and mineralogy.

Table 7.2 Classification of the Damaran sheeted leucogranites (after Nex and Kinnaird (1995) and Nex et al., 2001).

Types	Color	Texture and Mineralogy
A	Pale pink to white	Irregular to boudinaged, folded by D3, weak S3 foliation at margins, occur in high-strain zones. They are homogeneous, sacharoidal and fine grained.
B	white	Often boudinaged, inhomogeneous with a variable grain size from fine- to pegmatitic and form parallel sided tabular intrusions. Typically garnetiferous, infrequent abundant biotite and tourmaline
C	White to pink	Occasionally boudinaged, emplaced in F3 flexures. Medium to pegmatitic grain size, hypersolvus with interstitial clear quartz, magnetite, ilmenite and tourmaline
D	White	High grey or smoky quartz content, U-enriched and have a medium- to coarse-grain size. Have a granular texture, white feldspar with characteristic smoky quartz frequently visible. They frequently contain topaz that is absent from other types.
E	Variable	They are characterized by the presence of ubiquitous ''oxidation haloes'', highly variable grain size and colour, may have a wide range in modal mineralogy within one sheet.
F	Red	Post-kinematic, have parallel sides and crosscut all preexisting structures. They have coarse-pegmatitic grain size. They contain large perthitic K-feldspars up to 30 cm in size, milky quartz, and interstitial biotite and they are almost albite free.

7.2 PROJECT GEOLOGY

The Valencia and Namibplaas deposits are situated in the same regional structural setting; the so-called Khan Syncline which is underlain by Karibib marbles located just west of the Khan River and east of Valencia and west of the Khanberge.

The structure stretches over 50km from beyond the Rössing Mine in the SW up to close to Usakos in the far NE. The general trend is NE-SW and stretching parallel to the Khan River.

The synclinorium is up to 9km wide (Figure 7.3).

The Valencia and Namibplaas deposits lie adjacent to the tightly folded Abbabis inlier (old "Joly zone" or Valencia North on Valencia, and Area A on Namibplaas).

In both areas, the Damaran sequence is the most complete stratigraphic column and comprises substantially attenuated Etusis, Khan, Rössing, Chuos, Karibib and Kuiseb Formations (Freemantle, 2010) (Table 7.1).

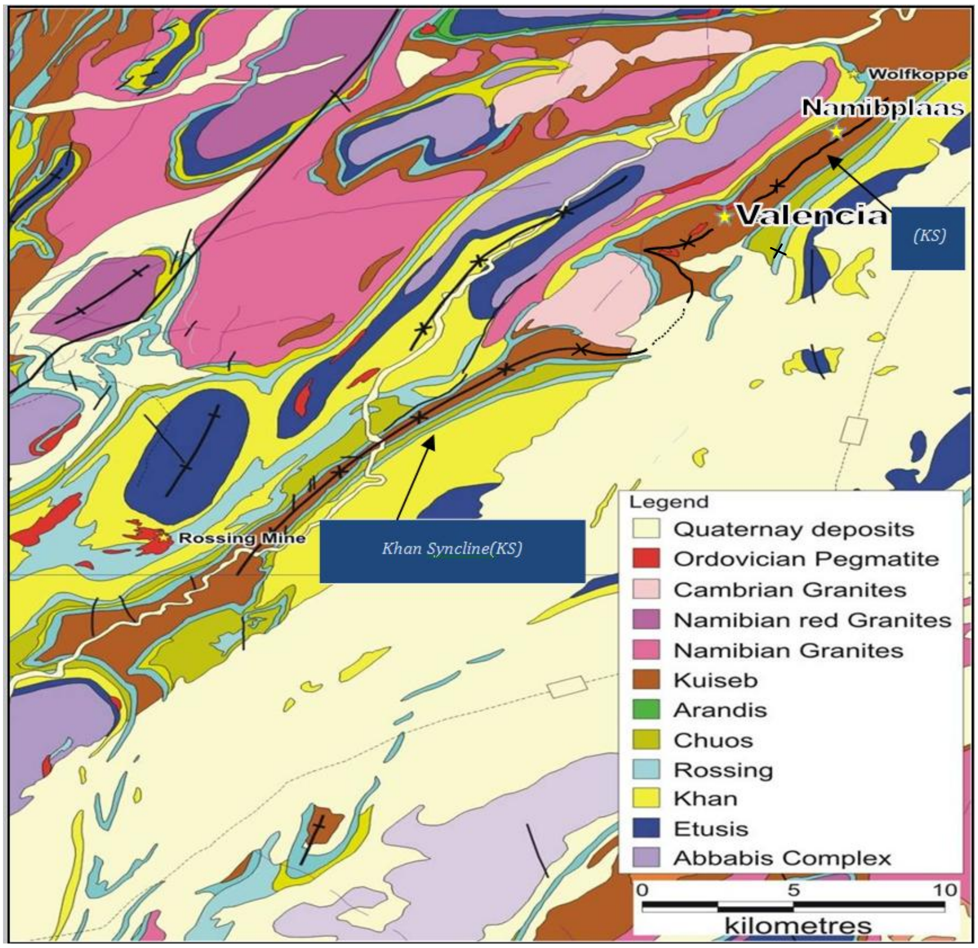


Figure 7.3 Regional Geology (Formations) and position/extend of the Khan Synclinorium.

7.2.1 Valencia

The Valencia deposit is approximately 4km SE of the Khan River Valley and approximately 22km northeast of the Rössing Mine (Figure 7.3).

U-Mineralisation is hosted in leucogranites that have invaded the succession in stock work-like fashion along NNE/SSW structural weakness zones, preferential utilising the fold plane of a characteristic Z shape fold (Figure 7.4).

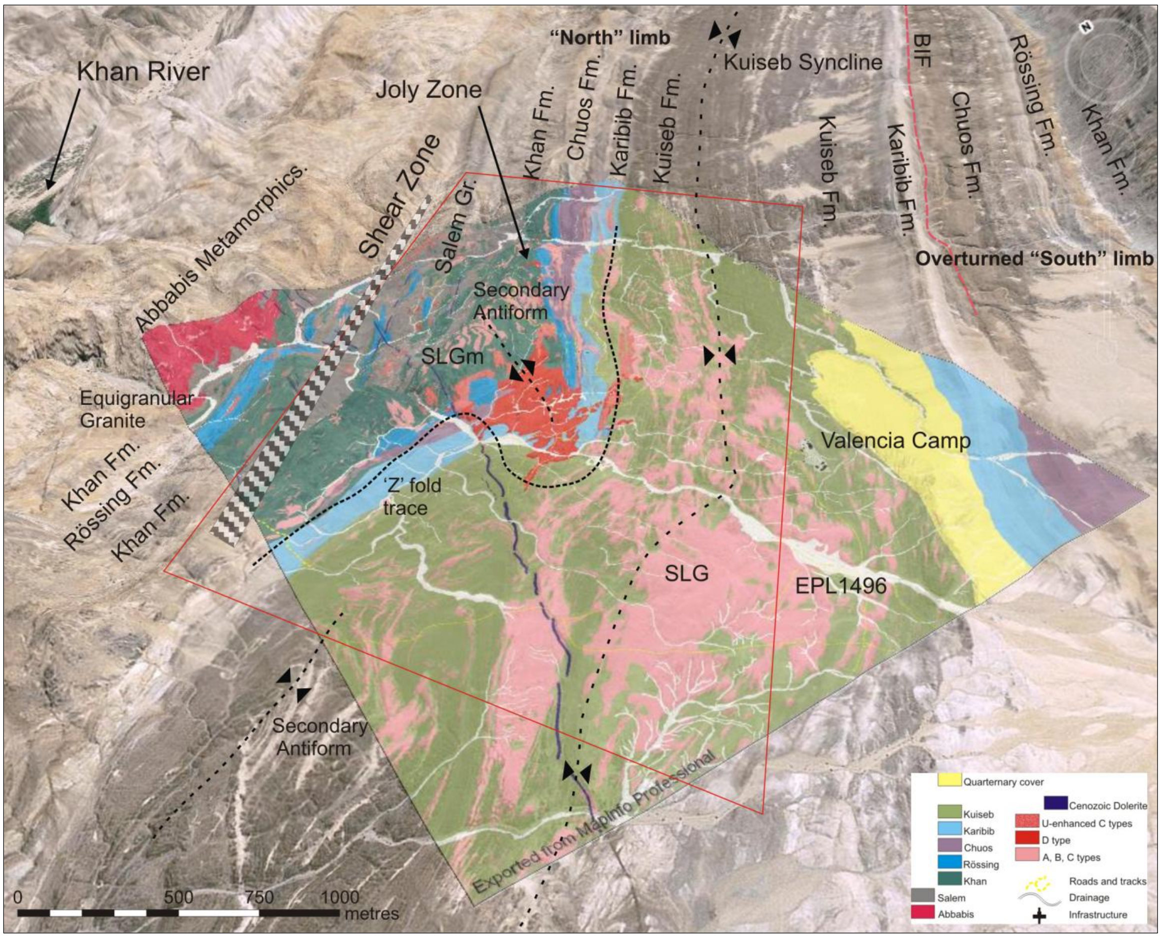


Figure 7.4 Local Geology and structural features of the Valencia area.
Secondary antiform exposing D-type alaskites in its core (unpubl. PhD thesis. Freemantle, G., 2010).

The syncline is a regional scale fold with a core of refolded Kuiseb schists. The fold hinge trending NE-SW. The synform is refolded south of the Rössing deposit where the hinge trend changes to roughly N-S. The eastern limb of the syncline at Valencia is attenuated and a secondary fold provided the weakness pathway for the mineralised intrusives in a saddle-reef style.

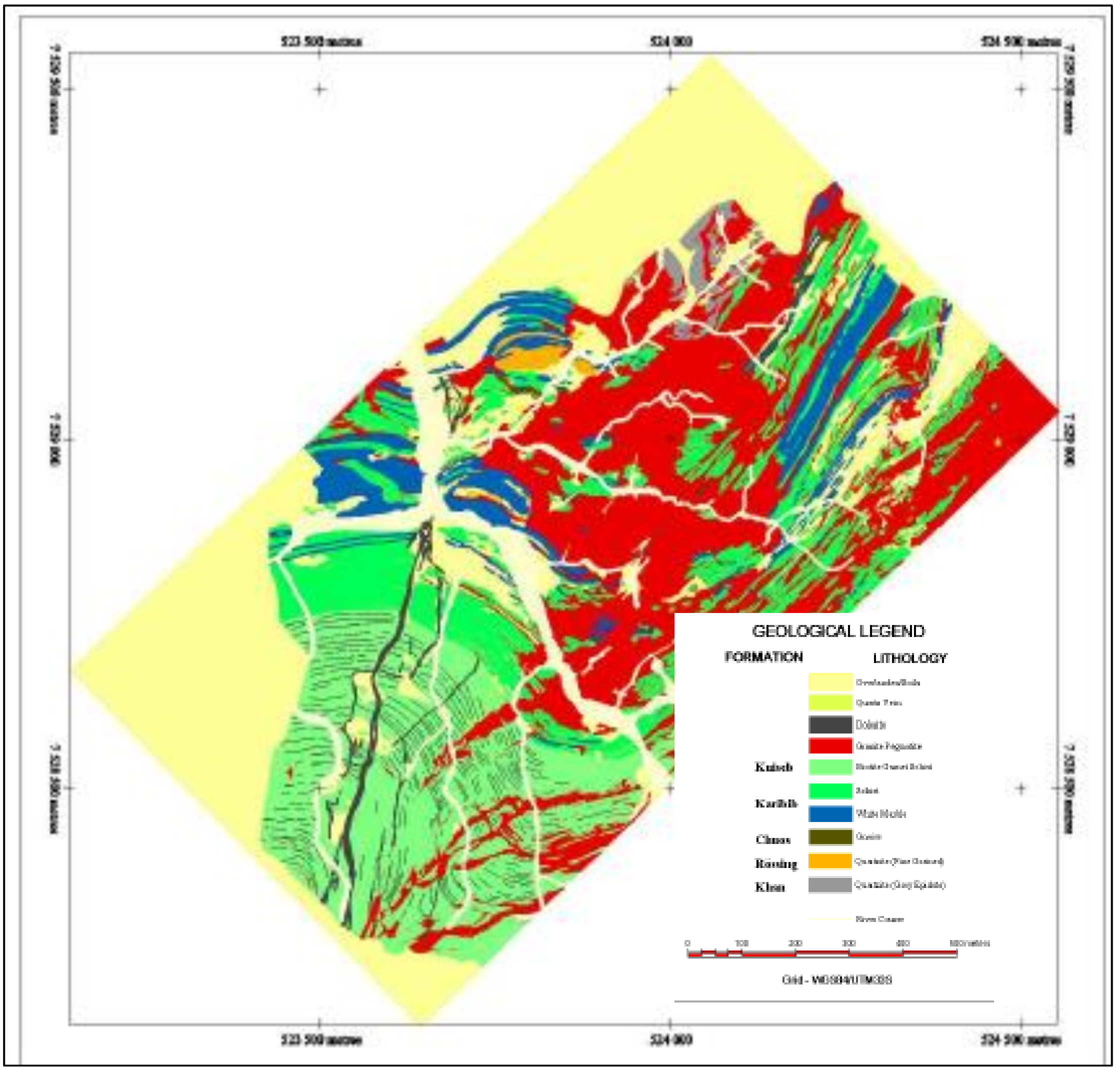


Figure 7.5 Surface Geology map (Goldfields).

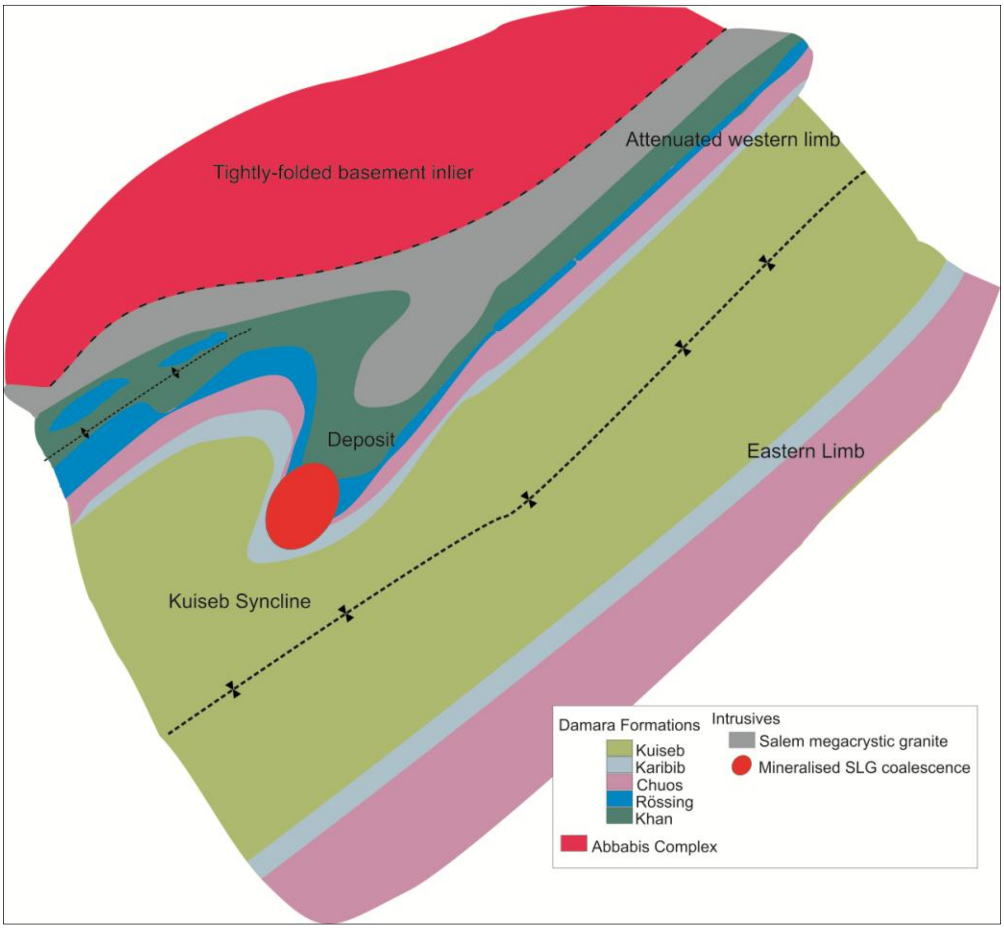


Figure 7.6 Simplified structural schematic of the Khan Syncline around Valencia with D-type alaskite positioning.

7.2.1.1 Lithology

The *Etusis* Formation is represented by a sequence of metamorphosed quartzites and arkoses often showing mylonitic textures on the Northern limb when in contact to Salem Suite granites.

The *Khan* Metasediments follow as well with local mylonitisation still evident. They occur typically as dark grey-blue- to green-coloured banded gneisses. The succession shows upward trends of increasing calcareous and diopsidic tendencies.

The Khan Formation appears better preserved on its Southern limb which is in contrast to the more strained and thinned out deformed Northern limb.

The *Rössing* Formation follows discontinuous and is attenuated in plus/minus 100 m-scale boudins all stretching along strike on the Northern limb.

The calcerous unit comprises two marble units (an upper and a lower) separated by cordierite gneiss between them. The unit shows attenuation as well but in principal occurs continuously along the entire Southern limb. An upper quartzite (pyritic) is present and in direct contact to the overlying Chuos Formation.

The upper marble unit contains serpentine with pelitic gneiss and some schistose units. The lower marble also containing serpentine however carries lenses of grey-coloured marble/limestone with a thin conglomerate layer at its base. The latter forms a good marker in the field.

This sequence closely equates to the upper Rössing package of Nash (1971) whereby the lower garnet – biotite granofels as described for the Rössing Mine stratigraphy being mostly absent at Valencia.

7.2.1.2 Structural Controls and Alaskite Intrusion

Valencia forms the core of an eroded antiform, which plunges to the northeast. The surrounding limbs vary in dip from almost flat to steeply overturned. Isoclinal folding on the south-eastern limb of the antiform, as well as over the central portion of the adjoining synform, which is recumbent with both limbs dipping to the southeast.

The uraniferous alaskite intruded into the north-western limb with the emplacement of alaskite clearly controlled by a younger north-north-westerly to south-south-westerly trending antiformal structure cutting through the older folding at almost right angles (Figure 7.7)

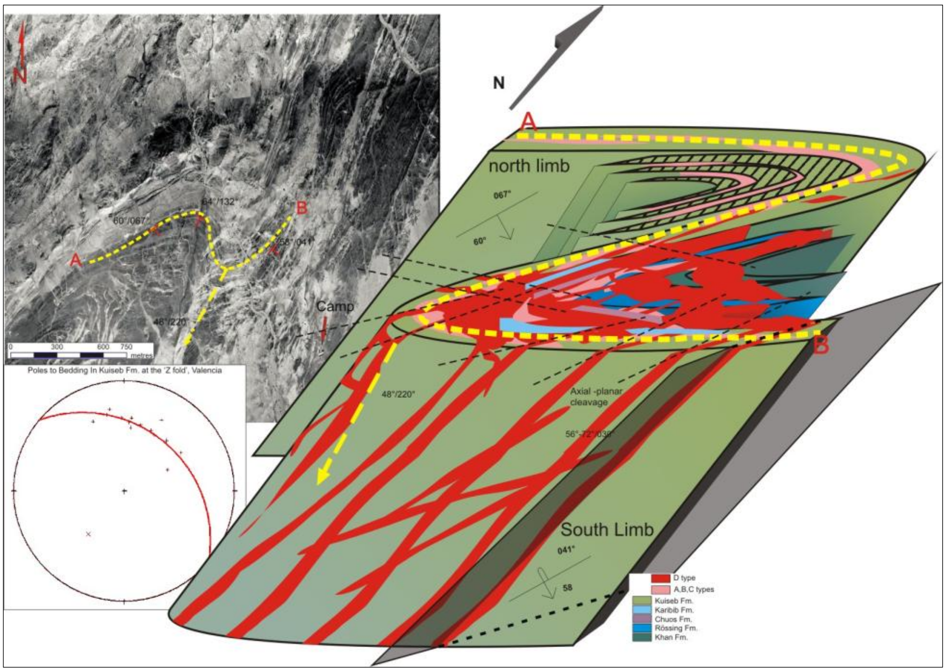


Figure 7.7 Valencia structural feature.
Schematic of the large 'Z' folds into which the majority of mineralised granite has intruded. The attenuation of the marble formations, the Chuos and the Khan has allowed the sheets to invade the Kuiseb. Disharmonic folding has duplicated the Kuiseb and created voids at the hinge into which granite has coalesced in a saddle reef fashion.

The alaskites vary from aplitic, through fine and medium-grained phases to pegmatitic.

At least eight phases of alaskite have been identified based on textural characteristics and uranium content. These phases are interpreted to be separate pulses of intrusion. The different grain size phases are usually all leucocratic, with biotite content often increasing with increasing grain size.

The general composition of the alaskite is quartz and alkali feldspar with or without biotite. Accessory minerals such as tourmaline, apatite, garnet and iron and copper sulphides may become so abundant in places that they form a major constituent of the alaskites (Roesener and Schreuder, 1992).

The conformable nature of relatively thin veins in tight isoclinally folded schist sequences suggests a pre or early syntectonic genesis for these veins, however, the strongly transgressive nature of some dyke-like bodies suggest a separate later syn- to post-tectonic history for some of these bodies (Labuschagne, 1979).

The alaskites contain xenoliths of host rocks in which they were emplaced and these xenoliths range in size from tiny fragments to bodies several tens of metres long. They have conformable relationship with the local structure, with their strikes and dips not varying significantly from those of the country rock.

A number of dolerite dyke-like bodies occur in and around the Valencia deposit. They dip from fairly flat to almost vertical and strike north to south or southwest to northeast. These dykes only rarely exceed a few metres in thickness.

Younger pneumatolitic pegmatites and quartz veins occur throughout the area. They are usually relatively thin and short and do not form conspicuous outcrops.

7.2.1.3 Valencia Satellite

Valencia satellite deposit is an addition to the Valencia deposit which forms geologically an isolated alaskite sheet which is part of the main Valencia mineralisation event. It is less than 1km away from the main deposit NE, along strike and references to Valencia in this report are inclusive of the satellite deposit unless otherwise noted.

In the area, 2 types of alaskite (C and D type) are intruding in sheet like bodies into a sequence of NE striking, steeply SE dipping Khan, Rössing and Chuos Formation rocks. The main intrusion follows the general strike and crosscuts in places to the contact of Rössing Formation lower marble. In places a NNE/SSW linear is indicative of late stage to post-tectonic emplacement.

The marble itself is clearly attenuated along NE-SW with indication of dextral rotation into NNE/SSW isolated marble boudins.

Earlier interpretations considered the Valencia satellite alaskite to be a plain C-type; this was corrected and drilling confirmed the D-type components.

The preferential intrusion path is along structural weakness zones which similar to the main deposit are indicated in steep SSW trending lineaments (Figure 7.8).

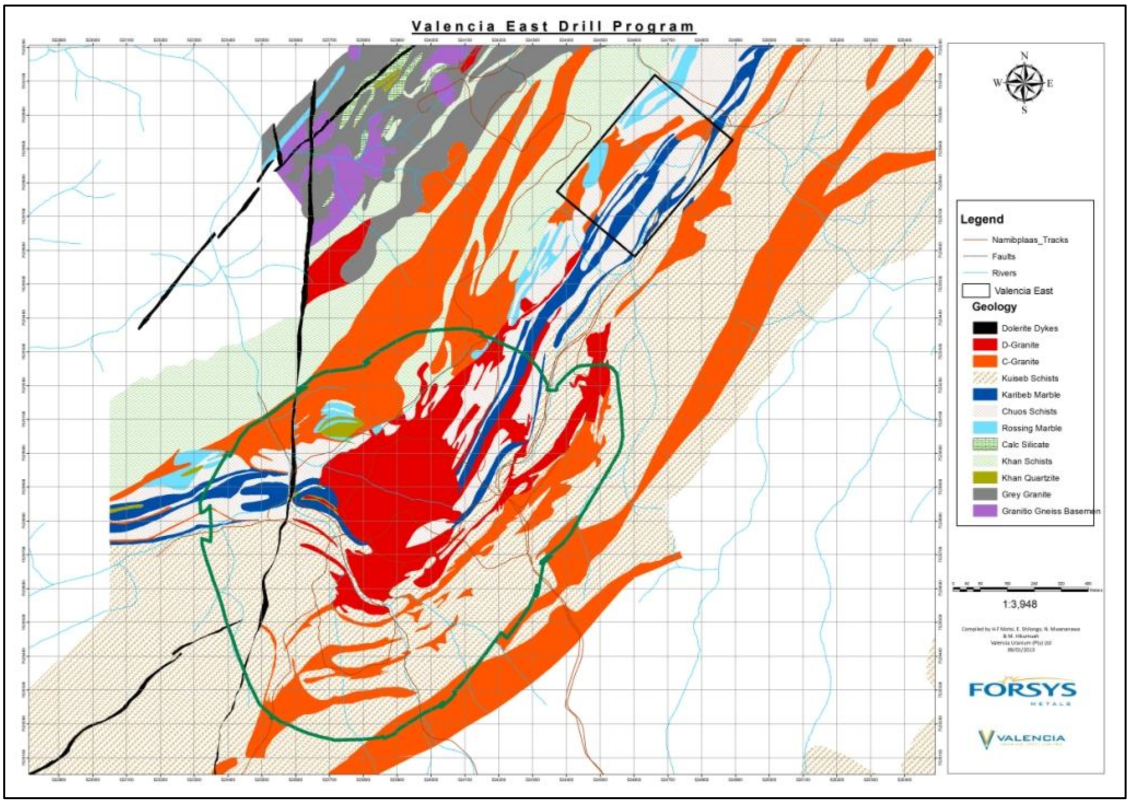


Figure 7.8 Valencia Satellite; combined geology plan of Valencia Satellite & Valencia Main on ML149.

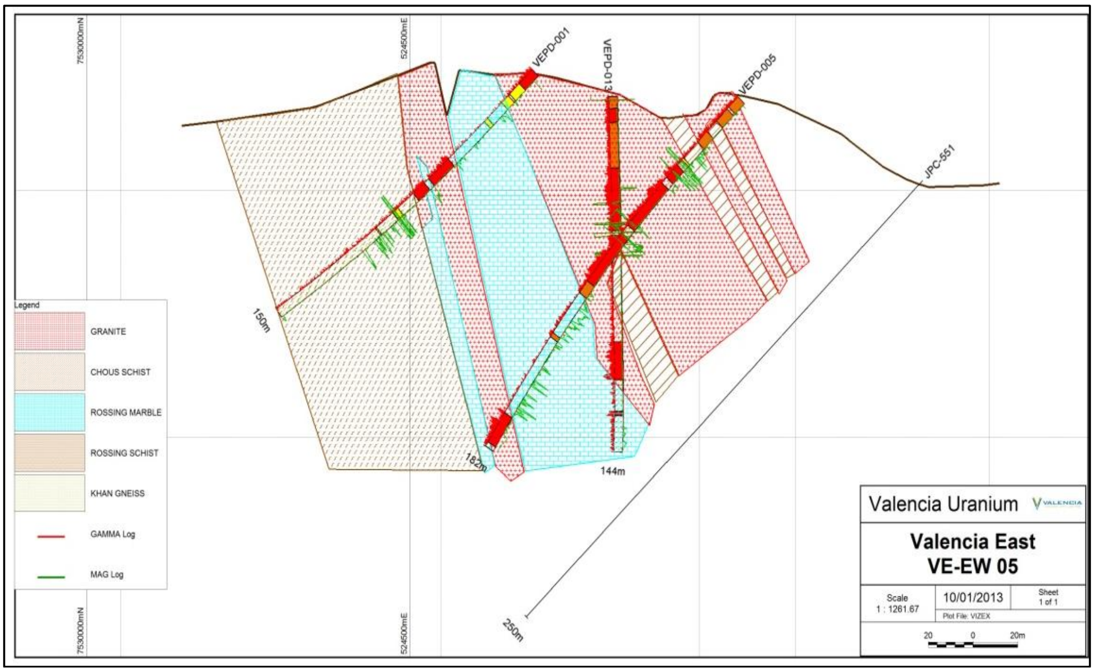


Figure 7.9 Valencia Satellite; section 5 with steep down-dipping strata.

Figure 7.10 provides a general impression of the steep running strata looking from Valencia East westwards towards Valencia Main in the far left (text “SLG” above “Karibib”).

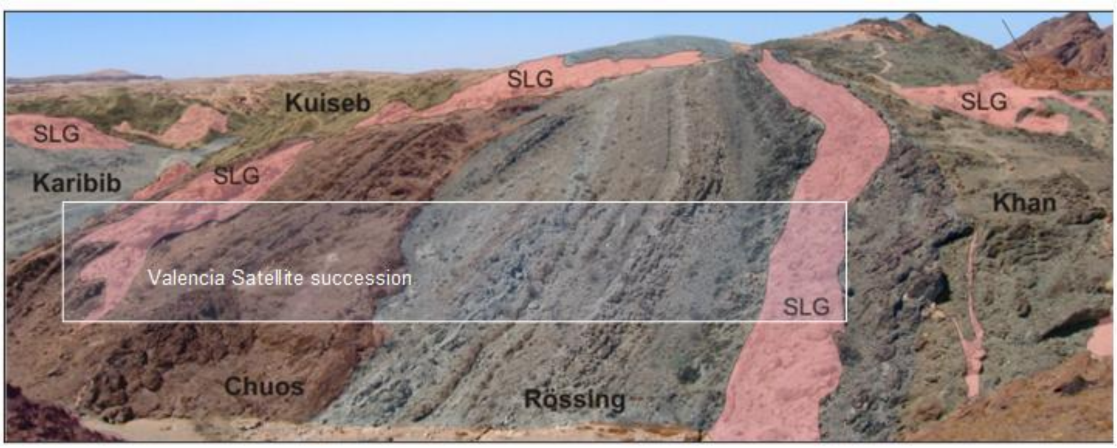


Figure 7.10 Valencia satellite lithostratigraphic succession.

Normal succession (upwards; from right to left) with mineralised SLG along and crosscuts into Rössing Formation.

7.2.2 Namibplaas

Namibplaas in contrast to Valencia is a more rugged area and contains some steeper cliffs and gullies in its northern parts. NE striking strata forming higher elevations in its NW parts with alaskite sheets dipping in subparallel sets to the southeast. The succession being frequently incised by smaller NW-SE running gorges. The eastern edge of the deposit is marked by a smaller NE running Dry River which incises deeper towards the north.

There are two major anomalies, referred to as area A and area B.

The alaskite succession of area B dives under Kuiseb Formation along this river bed which signifies an important tectonic feature in the area.

Airborne uranium anomaly maps, produced by the Namibian Geological Survey in 1997 identified the presence of the two prominent uranium anomalies on farm Namibplaas; one in Abbabis basement and the other in Damaran metasediments.

Ground scintillometer surveys confirmed these 2 anomalies (see Areas A and B in Figure 7.11) but also confirmed significant differences in U/Th ratios between the two.

A detailed ground spectrometer survey revealed high thorium ratios for Area A which led to the exploration activities having moved to Area B for the time being.

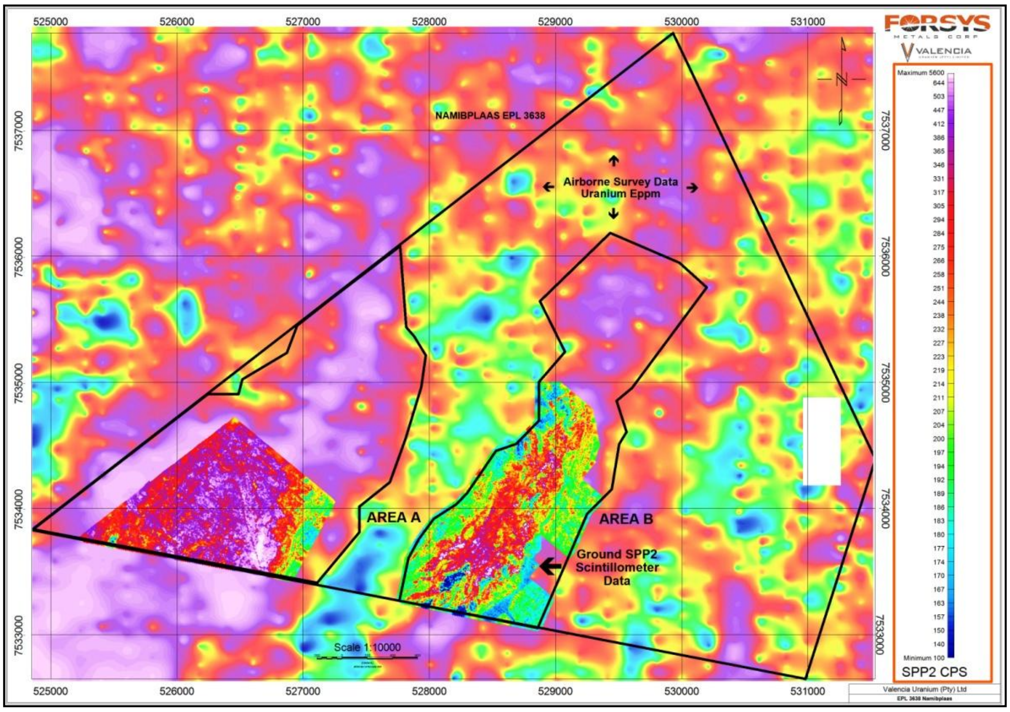


Figure 7.11 Namibplaas ground radiometry overlain onto GRN Airborne radiometric survey map from 1997.

7.2.2.1 Structural Controls and Alaskite Intrusion

The regional structure of the Namibplaas deposit is characterised by the gentle warp of the eastern limb of the Khan anticline which is striking SW-NE in the south-west turning towards N-S in the far north. Dip on foliation and bedding is intermediate remaining at +-45 degrees SE.

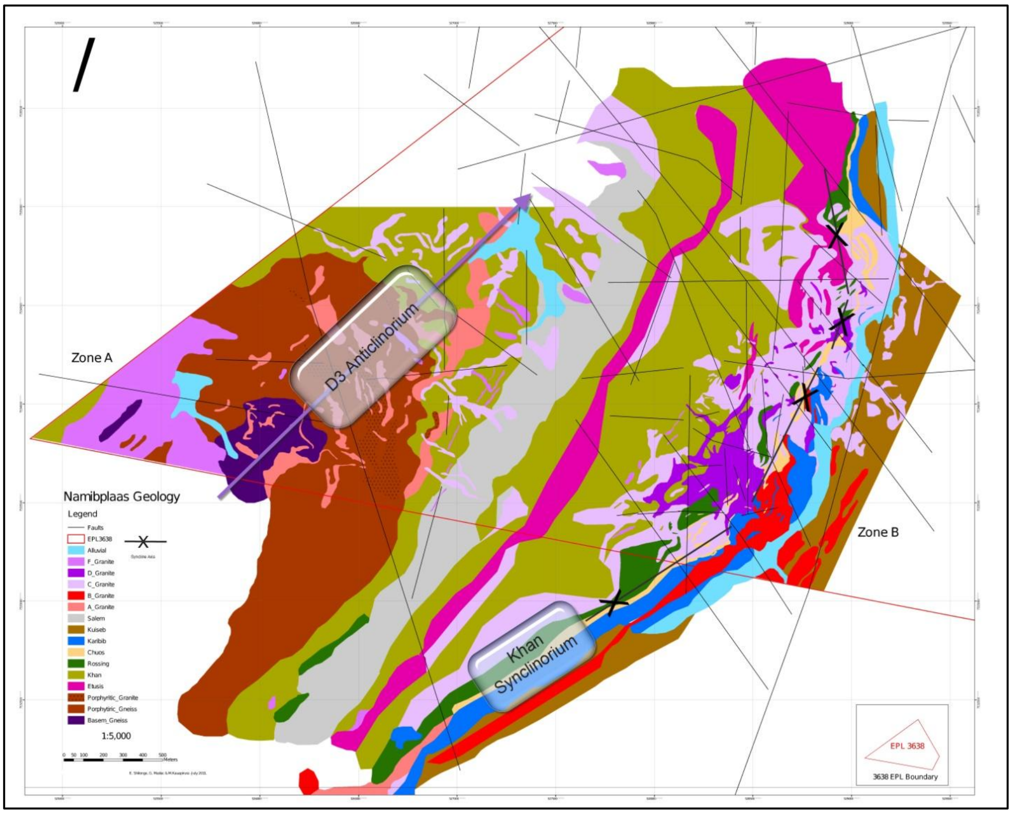


Figure 7.12 Geology of Namibplaas with schematic of structural key features.

The western limb of the antiform is slightly overturned and lies just east of Area A (Zone A, Figure 7.12); the eastern limb of the anticline with slightly folded Khan Syncline lies over Area B (Zone B, Figure 7.12). A major fault separates the Nosib Supergroup formations from the Abbabis complex in Area A. The core of the D3 antiform is made up of Etusis quartzite and forms the elevated crests. Dip and strike of the strata is general NE-SW; overall dip 40-45 degrees to SE-directions; attenuation along strike and down dip is frequently observed (Figure 7.13).

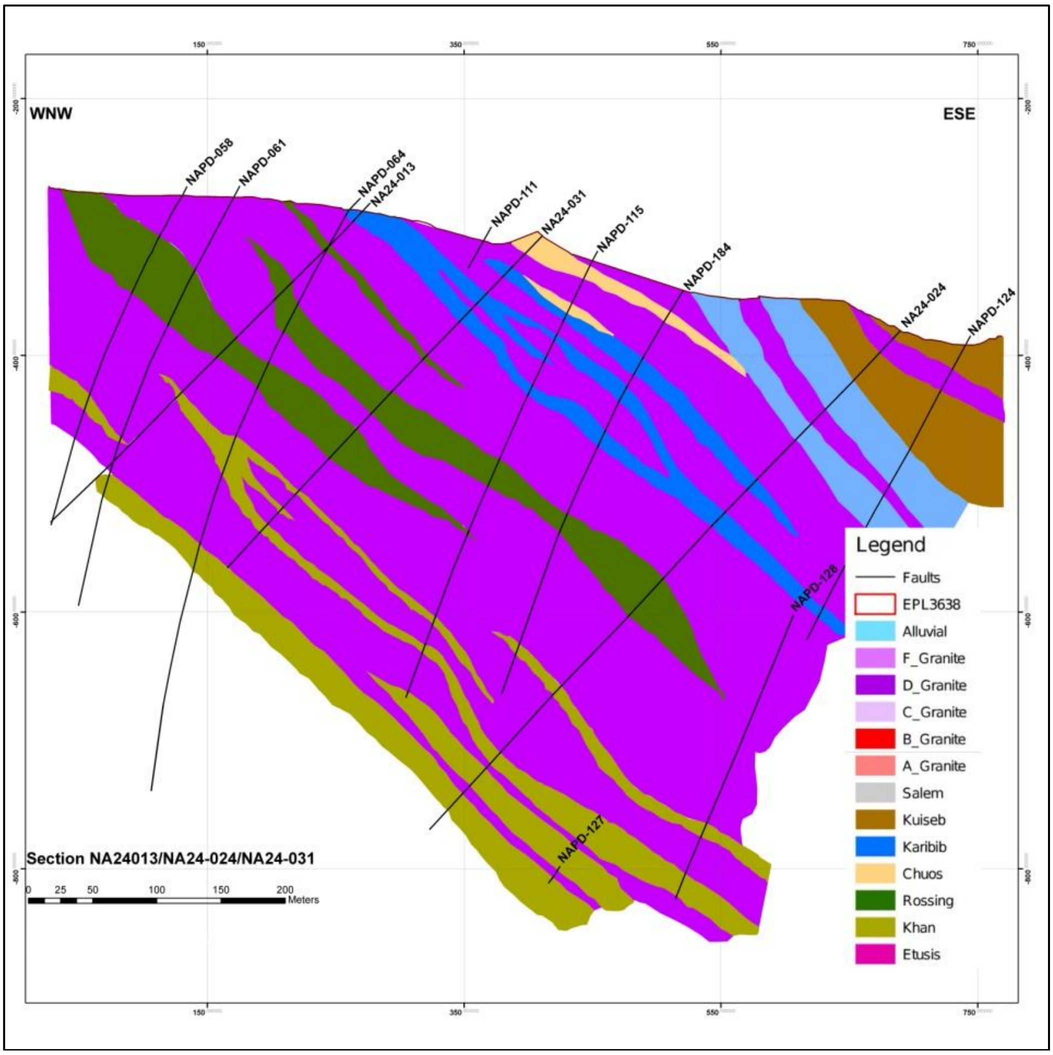


Figure 7.13 WNW/ESE section showing attenuated Damara host rocks surrounded by sheeted alaskite (Laine & Shilongo, 2011).

7.2.2.2 Alaskite intrusions

Area A anomaly in the western portion of the EPL is build up by mylonite, porphyritic granites and basement gneisses, no alkaline leucogranites (alaskites) occur.

Area B anomaly is confined by Damara lithologies with mainly Khan and Kuiseb Formations being the prominent and some attenuated Rössing marbles. Alaskites are common with D-type granite occurring mainly in the south of the anomaly and C and B-type granites dominating the northern part of the anomaly. C-type alaskites often carry embedded lenses of Chuos and Karibib marble (Hinojosa, 2008).

In contrast to the Valencia deposits and general regional classification of alaskites (Nex et al, 2001) are B and C–type alaskites on farm Namibplaas mineralised. It appears though that they constitute contaminated varieties of syn-tectonic D-type alaskites which has absorbed significant amounts of country rock during the anatexis phase. Guy Freemantle (2010) proposed this interpretation from similar observations in other parts of the Damaran Belt.

The A-type alaskite intrudes the Kuiseb and the Karibib formations, the B-type intrudes along the contact between the Karibib and Chuos formations, the C-type tourmaline leucogranite intrudes the Chuos and the Rössing formations, whereas the C-type magnetite leucogranite, occasionally bearing tourmaline-rich, intrudes preferential the Khan.

Most of the leucogranites appear as sills and are concordant to slightly cross-cutting the stratigraphy. General dip is 40 to 50 degrees to the east.

The B type alaskite is more abundant in the south, while the C type is most abundant to the north of the anomaly. In the south, the B type leucogranite can reach over 100m in thickness, while in other parts the leucogranite thicknesses vary from 5m to 40m.

7.2.2.3 Country-rock Lithology

Similar to Valencia the lowermost Damaran unit present at Namibplaas is the Etusis formation.

The Etusis, Khan, Rössing, Chuos, Karibib and Kuiseb formations are present throughout with the Khan formations and the Kuiseb formations bordering the anomaly along the regional SW-NE strike of the country rocks.

The Khan formation forms the surface geological limit on the North Western part of the radiometric anomaly and the Kuiseb on the South Eastern part.

The Rössing, Chuos and Karibib formations occur as elevated, sporadic, discontinuous outcrops along strike within the anomaly. Numerous alaskite sheets intrude the whole metamorphic series.

The Etusis Formation is about 150m thick and is represented by magnetite and diopside bearing quartzites. These quartzites have a highly recrystallized texture and are well bedded. The quartzites are dark grey, fine to medium grained.

The Khan formation is finer-grained and represents sedimentation within a basin environment. The Kahn formation (+/-260m thick) can be subdivided into Ferruginous banded foliated gneiss, a meta-arkose, a biotite banded foliated gneiss, and mottled calc-silicate gneiss. The Khan Gneisses are dark grey to green coloured. These units are fine to medium grained and foliated and bedded. The meta-arkose is pink to buff coloured, medium-grained with magnetite layers along bedding planes, and is characterized by a sugary texture.

The Rössing formation is only 50m thick and is characterized by an impure marble, discontinuous along strike. The marble is white, coarse-grained and crystalline and is associated with a quartzite, porphyroblastic cordierite-biotite schist, a calc-silicate rock and a meta-conglomerate. The quartzite is white to grey, weathered to pink, massive and fine grained. The quartzite is sometimes cut by veins of sulphides. The cordierite schist is grey, fine-to medium-grained, thinly foliated and is banded with felsic bands of quartz, feldspar, cordierite and foliated black biotite. The calc-silicate is green and is medium to coarse-grained.

Above the Rössing formation follows the Chuos diamictite schists (+-70m thick), which are metamorphosed tillites. This unit is characterised by quartzo-feldsparthic dropstones in a grey quartz-mica schistose groundmass and magnetite layers. The Chuos comprises grey, fine grained, thinly-foliated and massive schists.

The Karibib formation (approximately130m thick) comprises three main units; a marble, a calc-silicate rock and a dark grey biotite schist. The marble is grey, medium to coarse grained and occurs in several bands - at least seven packages of the marble alternating with calc-silicate rock are observed. The biotite schist is fine-grained, thinly foliated to massive with black biotite, white feldspar and quartz. The calc-silicate rock is tan to grey, fine grained, massive and banded and is cut by quartz-calcite veins and calcite-filled fractures. The Karibib overlays the Chuos diamictite schist and the contact between the two is seen as a barrier underneath which mineralised sheeted alaskites are observed.

The Kuiseb formation presents the uppermost unit and occurs in grey weathered to brown, medium to coarse-grained, foliated porphyroblastic cordierite bearing augen gneiss.

7.3 MINERALISATION

The uranium mineralisation throughout the prospect is hosted by alkaline pegmatites, called alaskites; some document country rock contamination by having clearly assimilated more iron rich schists, classified as B and C types (Namibplaas). Purer versions are classified D-type granite which usually carry the economic viable amounts of uranium.

Alaskite appearance ranges from small tiny, aplitic veins to massive coarse grained leukogranitic pegmatites to massive intrusive granites. The overall type is pegmatitic.

The mineral uraninite is the dominant uranium carrier throughout the deposits.

Uraninite forms small subhedral to euhedral crystals ranging in size from a few microns to 500 µm although commonly 30-50 µm. It is generally black and resinous. It occurs typically within microcline or plagioclase, or at crystal boundaries between feldspar, quartz, and biotite. It is often surrounded by alteration zones and radial cracks. According to Jacob et al (1983) the uraninite displays a preferential association with biotite and zircon; the latter appearing as inclusions within uraninite grains or as clusters of grains attached to them. Many uraninite crystals are altered in their core to thorite, jarosite.

Other primary uranium minerals include carnotite, titanite (including brannerite) and naobate.

A variety of secondary uranium minerals exits, many of which are brilliantly coloured and fluorescent and include gummite, autunite, saleeite, torbernite, coffinite, uranophane and sklodowskite.

Late stage alterations of all alaskite types can be observed and comprise kaolinisation, illitisation and silicification.

7.3.1 Valencia Mineralisation

Uranium mineralisation at Valencia has been identified over an area of 1,100m north-south by 500m east-west. Towards the NE a separate mineralisation pulse is identified stretching over an area of 600m northeast-southwest by 500m northwest-southeast (Valencia East) The mineralisation in general dips at approximately 35° to 40° to the south and has been identified by DDH drilling to a depth of 380m below surface.

A significant number of holes end in mineralised alaskite supporting the assumption of mineralisation continuing at depth beyond current drillhole cover.

The uranium mineralisation is hosted by alaskites that comprise massive stock-like bodies, dykes of varying thickness and veins and veinlets. No primary uranium is found in the surrounding country rocks. The Valencia granites vary from white to pink in colour; they are medium- to coarse-grained, and homogeneous to inhomogeneous in texture. Mymerkites, perthites and sericitisation of K-feldspar are common. The leucogranites have a high content of alkali-feldspar and are very low in biotite content, whereby a relationship between uranium occurrence and biotite content, as well as apatite seems to exist.

Accessory minerals present are magnetite, garnet, zircon, monazite, apatite and biotite.

The uranium is generally associated with medium-grained homogeneous textured leucogranites that have a high content of smoky-quartz. The uranium mineralisation is present as uraninite (UO_2) and the secondary uranium minerals as uranophane (Ca (UO_2) $2SiO_2.7H_2O$) and uranothallite ($Ca_2U(CO_3)_4.10H_2O$).

The uraninite is usually fresh with only sporadic, very minor alteration rims. The secondary uranium minerals occur as yellow coatings on exfoliation planes and joints. They form specks and tiny flakes on feldspar, quartz, biotite and apatite.

Chemical analyses of alaskites from various sample locations across the deposit identified a fairly consistent composition of uraninite of approximately 87.4% of the total uranium content (Freemantle, G., 2010). The uranium mineralisation predominantly occurs in the finer-grained alaskite and lesser in the coarse pegmatitic phases.

7.3.2 Namibplaas Mineralisation

At Namibplaas mineralisation is confined to syn- to post tectonic leucogranites which are similar in texture and mineralogy to the ones at the Valencia deposit.

In addition to the usual D-type mineralised alaskite Namibplaas has mineralised magnetite rich C-type alaskites. This type is confined to the northern portion of the deposit and quite unique.

Uranium mineralisation remains similar to Valencia and occurs as uraninite (UO_2) mineralisation and the secondary uranium minerals, uranophane ($Ca(UO_2)2SiO_2.7H_2O$) and uranothallite ($Ca_2U(CO_3)_4.10H_2O$. Minor betafite $(U,Ca)(Ti,Ta,Nb)_3O_9$ has been observed.

8. DEPOSIT TYPES

Primary uranium deposits formed by granitic magmas can be classified on two bases: petrologic process of ore formation and their tectonic occurrence.

The processes of ore formation can be subdivided as follows:

1. Syngenetic, orthomagmatic disseminations.
2. High-temperature, late-magmatic deposits.
3. Contact metasomatic deposits, including occurrences of garnetiferous skarns around pegmatite-alaskite bodies; high-temperature vein deposits, commonly associated with quartz-fluorite veins; and autometasomatic deposits, including many of the disseminated and local concentrations in albite-riebeckite granites.
4. Local pegmatites formed by in situ melting of country rocks.

The Norasa deposits of Valencia and Namibplaas all fall into category 2 of high-temperature, late magmatic or “intrusive type” deposits. Latest work by NEX (verbal comms Feb 2015) highlights the pegmatitic tenor and consider moving this type from of pegmatitic/granite to more pegmatitic (category 4). Sub-types at Valencia and Namibplaas resemble a pegmatitic origin, such as the alkaline pegmatites at nearby Rössing and Husab Mine.

The sub-types at Valencia and Namibplaas resemble pegmatite stage deposits, such as the pegmatite-alaskite deposits of nearby Rössing and Husab or similar to the Crocker Well Uranium Project in South Australia.

All of these uranium deposits are associated with alkaline leucogranites that comprise massive stock-like bodies, dykes of varying thickness, sill like bodies or veins and veinlets, which can be either conformable with or transgressive to their host rocks.

The Valencia and Namibplaas deposits form part of these leucogranite-hosted uranium deposits.

Nex et al. (2001) developed a detailed classification and recognized the importance of orogenic timing of the emplacement. He subdivided the alaskites into six granite types based on appearance, structural setting, mineralogy, and petrology; called A, B, C, D, E and F type.

Types A, B and C intruded pre-F3; D, E and F types syn- to post-F3 of which the D type being the most importance one for uranium enrichment.

9. EXPLORATION

9.1 VALENCIA

Valencia Uranium was originally identified from an airborne survey in 1973, and a first detailed exploration campaign was conducted between 1973 and 1983 by Trekkopje Exploration and Mining Company.

Trekkopje Exploration and Mining Company carried out detailed geophysical surveys, surface mapping and drilled 97 diamond drilling holes (DDH) for 24,741m during this period.

Valencia commenced activities in the area in 2005 and started drilling in 2006 adding an initial 44 diamond drill holes over 13,082m with an additional 4 DDH for geotechnical purposes for 292m. A further 5 DDH of 826m were drilled in a granite exposure nearby to the NW. The total DDH drilling program amounted to 38,941m.

There were 148 reverse circulation percussion (RC) drillholes added for 11,111m which were drilled along a tight grid measuring 20m by 20m and drilled to an average depth of 300m across the anomaly.

By 2009, a further 409 percussion drilling (PD) holes of 110,095m, to an average depth of 270m, were drill to hole length ranging from 53m to 401m. In 2012-2013, another 52 PD holes of 6,548m were added into a north eastern satellite body, drilled at an average depth of 126m.

Drilling was carried out by R.A. Longstaff Namibia (Pty), Major Drilling, Erongo Drilling Van Rhyn, Roburgh and Hard Rock Drilling; geological supervision, logging and sampling were conducted by Valencia staff.

9.1.1 Re-logging and assay of historical drillholes

A confirmatory program was conducted to validate the accuracy of the historical 97 holes. This involved a re-survey of all historical drillhole collars and re-logging of drill holes for which no drill hole logs were available.

This also involved sampling and assaying of samples from selected drillholes.

9.1.2 Spectrometer and scintillometer survey

A handheld ground radiometric survey was completed in February 2006 by Forsys geologists. A Pico-Envirotech Spectrometer Model GIS s15 instrument was used to measure gamma ray readings for total counts, uranium, thorium and potassium. Readings were taken at intervals of approximately 1.5m on northwest to southeast oriented lines spaced at 50m apart.

A second handheld ground radiometric survey was completed in December 2007 to January 2008. Readings were taken at intervals of approximately 5m on northwest to southeast oriented lines spaced at 10m apart. The contoured results of the scintillometer survey for uranium are shown in Figure 9.1. The anomalous areas of radioactivity coincide with the outcrops of alaskite and provided targets for drill testing.

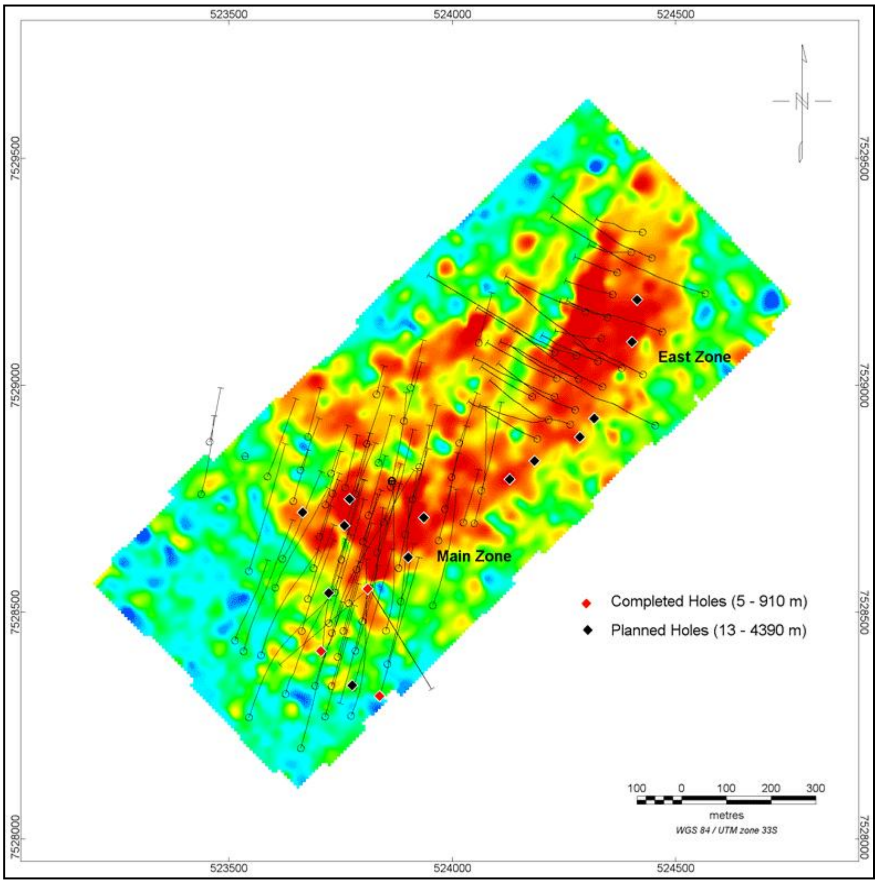


Figure 9.1 Results of ground scintillometer survey with DDH overlain.

9.1.3 DDH drilling

A total of 53 DDH for 14,200m were drilled by Valencia for geotechnical data and infill grade sampling. Of these, 44 holes for 13,082m were used for inclusion in the Mineral Resource and to provide geotechnical data to assist in the open pit slope designs.

Drillhole, VA26-118, was drilled parallel to drillhole VA26-002 at a distance of less than 0.5m from its collar. A comparison of the logs of these two drillholes has indicated a good fit of lithologies and grades down to depths of approximately 270m. The comparison confirmed that the historical drilling information is a reasonable representation of the geology and uranium grades (Snowden 2009a).

The total diamond drilling database consists of 150 holes over 38,941m (Figure 9.2).

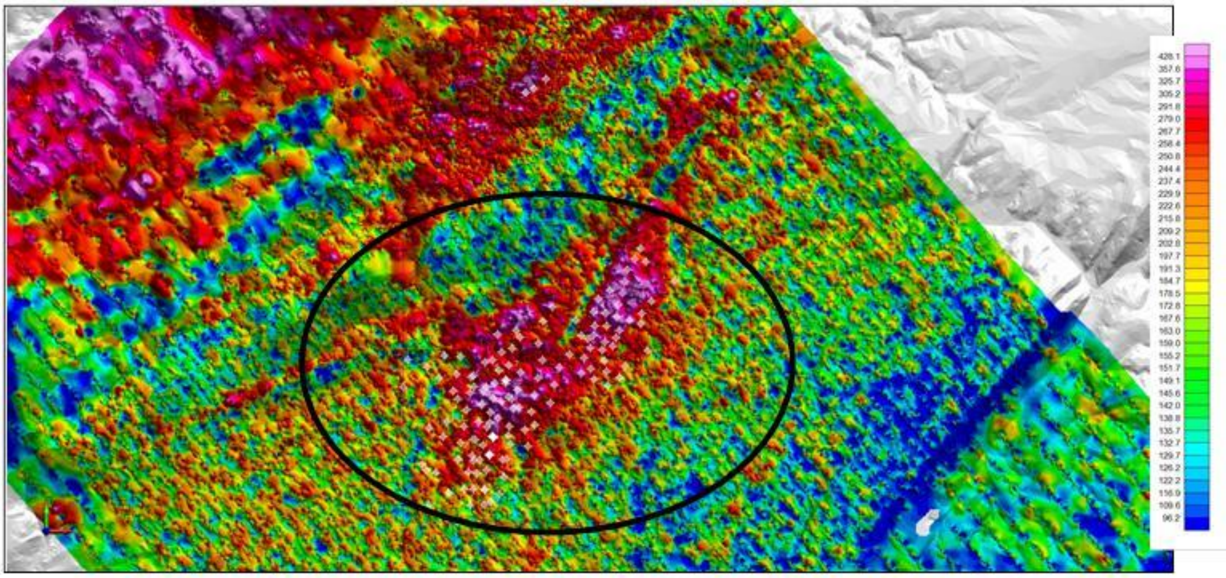


Figure 9.2 Radiometry, ground scintillometer data (cts/sec) of Valencia, showing DD drill collar positions

9.1.4 RC drilling

A total of 148 RC drillholes were completed, for 11,111m on a grid of 20m by 20m over an area of 300m (east to west) and 200m (north to south) down to a depth of 105m below surface.

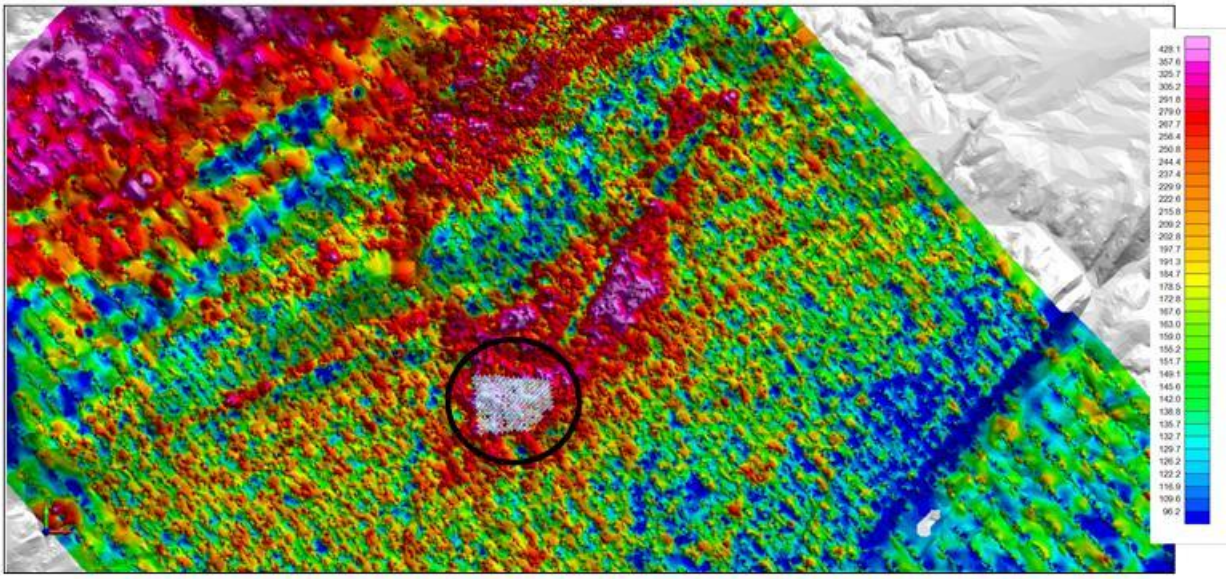


Figure 9.3 Radiometry, ground scintillometer data (cts/sec) of Valencia, showing RC drill collar positions

9.1.5 PD drilling

A total of 409 percussion holes were drilled at Valencia for 110,095m.

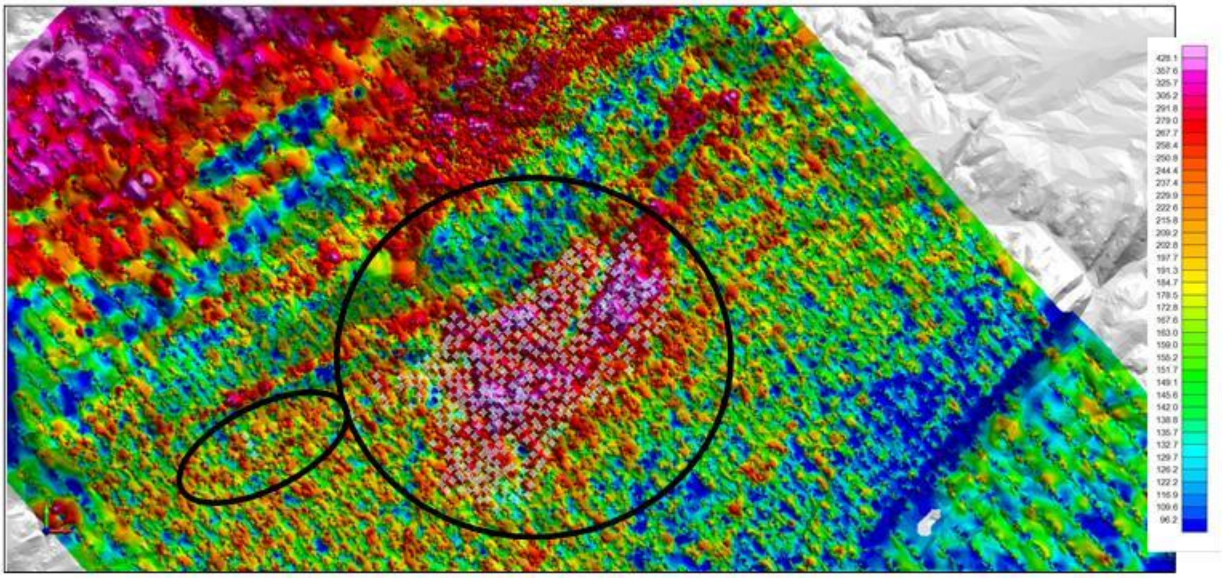


Figure 9.4 Radiometry, ground scintillometer data (cts/sec) of Valencia, showing PD drill collar positions

9.1.6 Digital terrain model

The project area was flown in March 2007 for the compilation of a digital terrain model (DTM), providing contours at 2m intervals and production of digital orthophotos. These were compiled initially from 1:10,000 scale monochrome aerial photographs.

9.2 VALENCIA SATELLITE

In 2012 evaluation work commenced on a smaller anomaly located approximately 600m NE of the north-eastern eastern edge of the main anomaly (Figure 9.5).

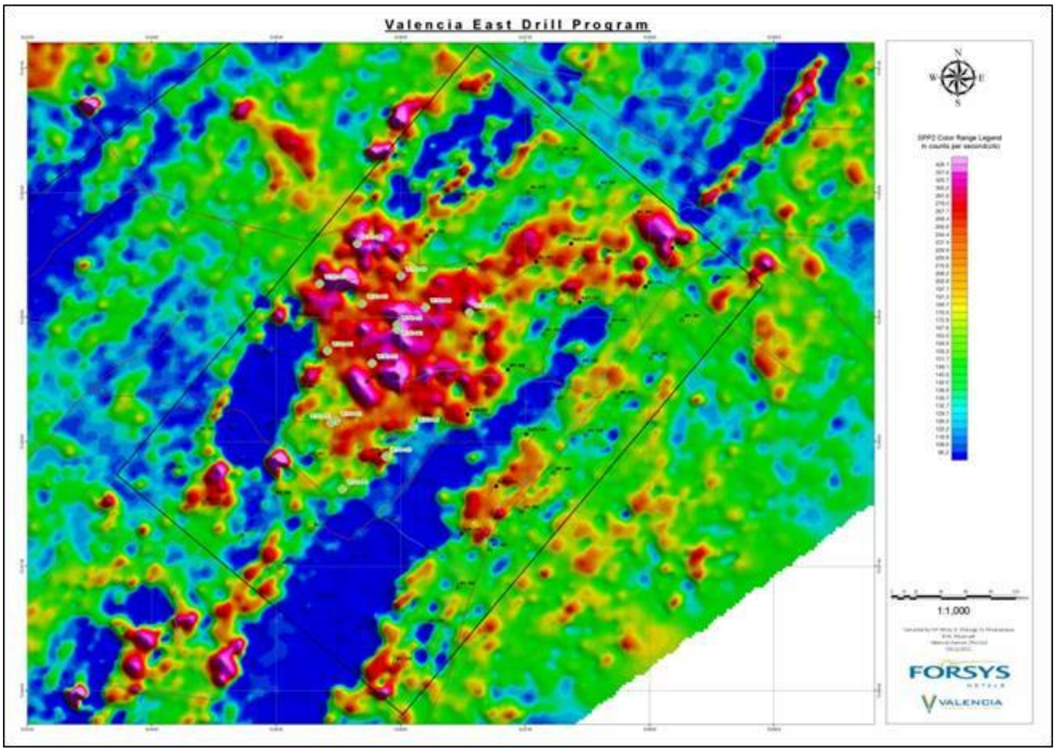


Figure 9.5 Valencia Satellite SPP2 ground radiometric contour map with PD collar positions.

9.2.1 PD drilling

During 2012 and 2013 a total of 52 PD holes were drilled to shallow depths of between 55m to 234m at a total of 6,548 meters.

The drilling objectives succeeded in confirming the existence of D-type alaskite further NE of Valencia with higher grade potential at depth.

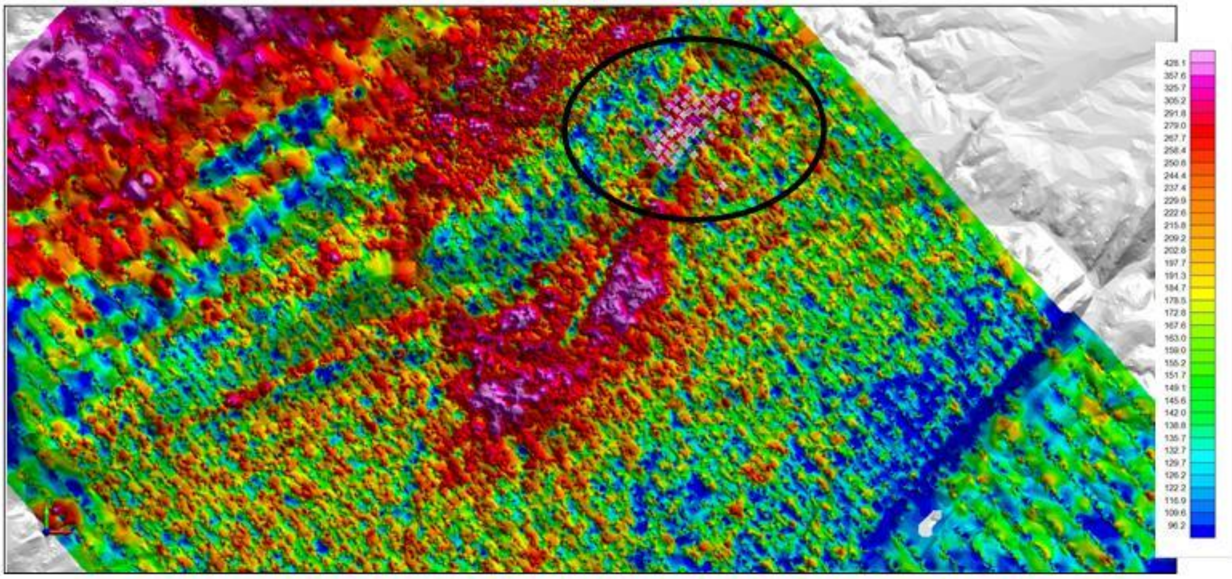


Figure 9.6 Radiometry, ground scintillometer data (cts/sec) of Valencia, showing PD drill collar positions at Valencia Satellite

Drill holes were optimised for geology and generally orientated perpendicular to the geology strike and inclined at 60 degrees towards the NW, some were drilled vertical.

Drilling was carried out by Ferro Drill Namibia. The PD drilling database for Valencia East consists of 52 holes over 6,548m.

9.3 NAMIBPLAAS

At Namibplaas, previous airborne surveys by the Geological Survey of Namibia had outlined two radiometric anomalies which are named A (westernmost) and B (central) (Figure 9.7).

A radiometric ground survey was initiated by Valencia over both zones at 10 meters line spacing and reading positions every 5 meters. The resulting radiometric survey maps are shown as insets in the figure.

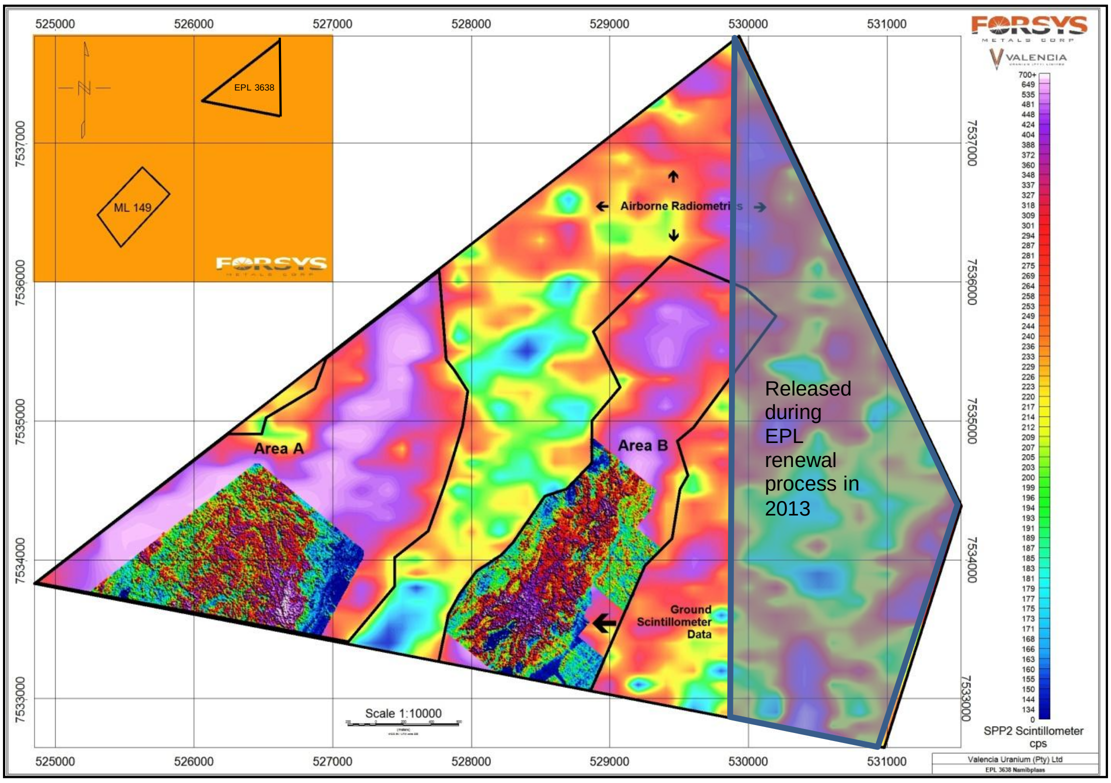


Figure 9.7 Map of EPL 3638 boundary pre/post 2013 EPL renewal, showing the airborne radiometrics in Area A and B (Laine, 2009).

The A anomaly is characterized by a high thorium and uranium ratio, while the B anomaly is high uranium-low thorium. The Namibplaas deposit is located in area B.

The detail ground radiometric survey in area B defined the initial drill target measuring on surface 500m by 1,700m (Figure 9.8).

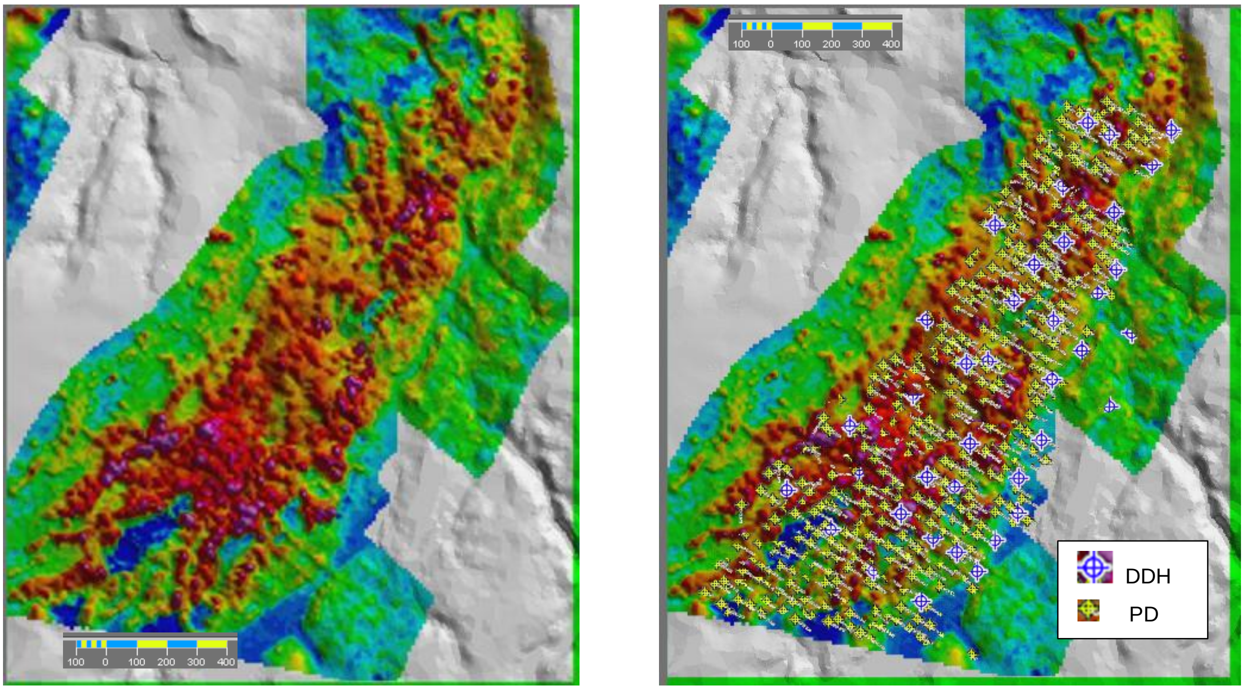


Figure 9.8 Radiometric signature of Area B - without and with drillhole collar

9.3.1 Magnetic signatures

A significant variation to Valencia is the existence of a dominant magnetic signature which runs throughout the entire area of interest. Chuos Formation lithology is easily identified within the core of a South-Easterly dipping, overturned limb of the main NE plunging D3 Khan antiform (Figure 9.9).

The alaskite emplacements are clearly following this contact unit close by.

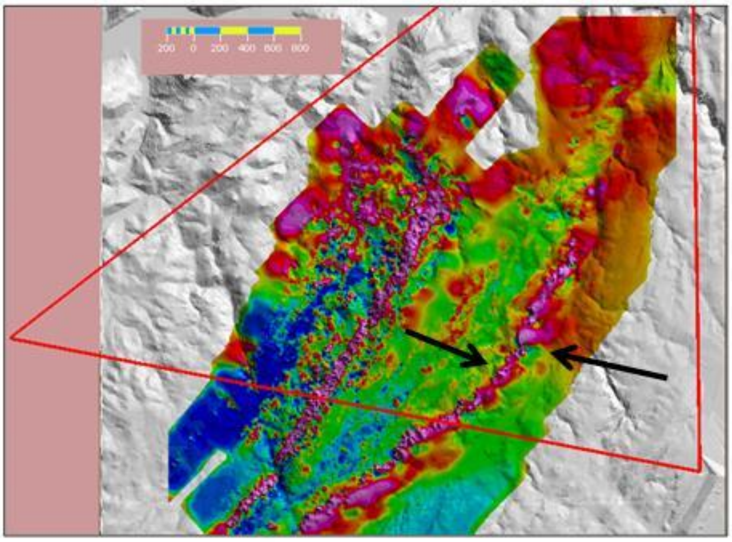


Figure 9.9 Clear magnetic signature (Chuos Fm) along which alaskite is positioned.

9.3.2 DDH drilling

Gold Fields Namibia Ltd drilled seven diamond drill holes before Valencia commenced exploration at Namibplaas in 2008. Drilling began in 2010 with 19 DDH for 1,667m and another 14 DDH in 2011 for 4,668m. In total 40 DDH have been drilled in the area, totalling 9,896m.

Drill core was sampled and chemically analysed for uranium to define the local correlation between the probe data and assayed U_3O_8 values.

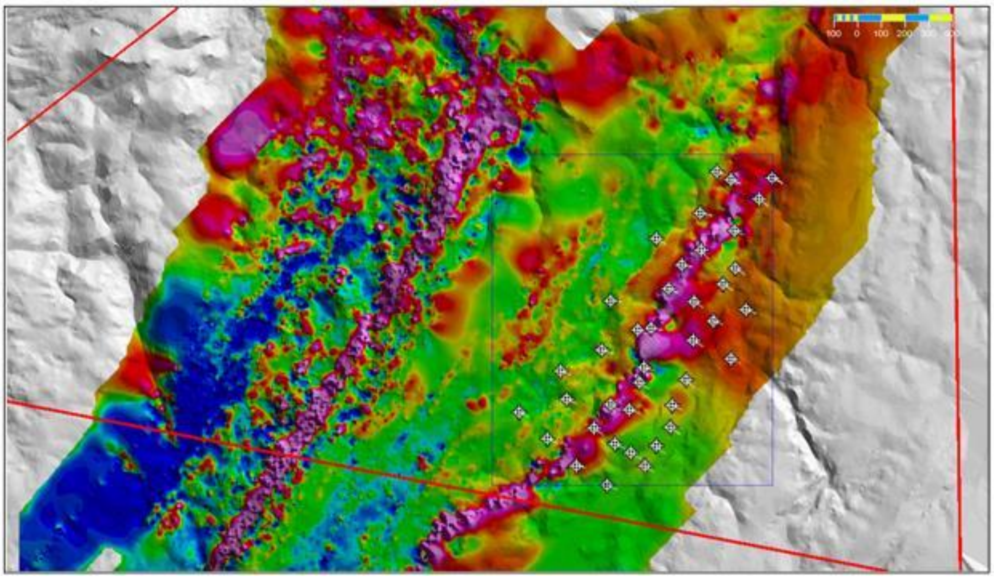


Figure 9.10 Namibplaas magnetic map with all DDH collar display

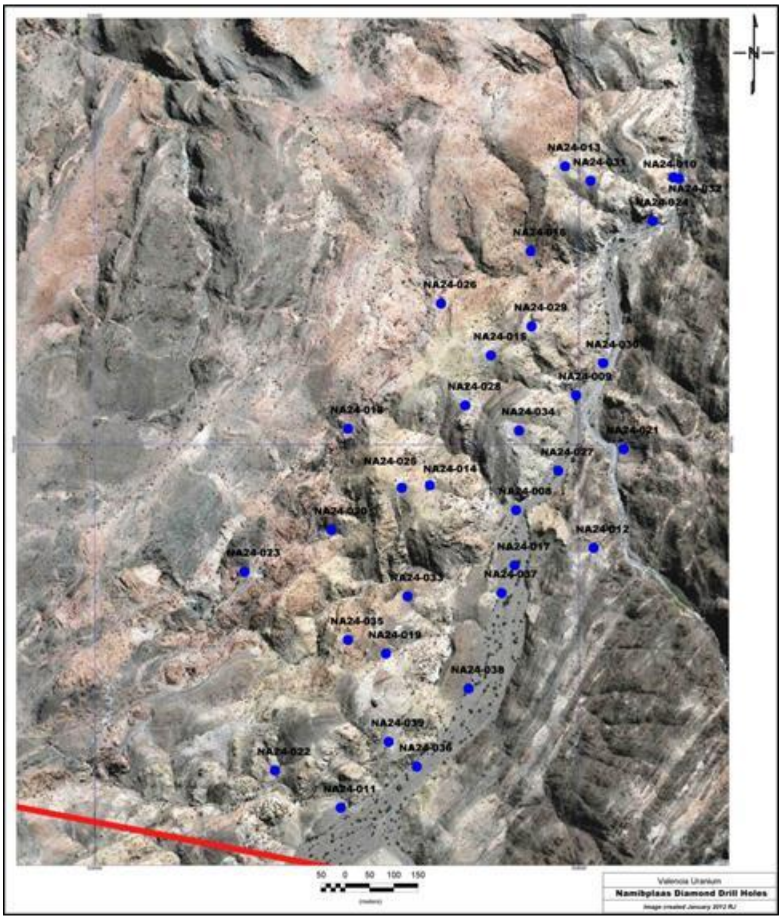


Figure 9.11 VUL DDH collar positions on satellite terrain background clearly depicting the characteristic sheeted layer/dykes of alaskite.

9.3.3 PD drilling

Percussion drilling commenced in 2008 and at the end of 2011, a total of 288 PD holes totalling 63,093m were completed for probe sampling (PD phase 1). This information was used for inclusion in a first Mineral Resource estimation.

Phase 2 PD drilling commenced shortly thereafter in October 2011 and concluded in August 2012 with in-fill drilling the area significantly and adding 242 holes over 47,924m.

The total PD drilling effort at Namibplaas amounts to 530 percussion drill holes over 111,117m.

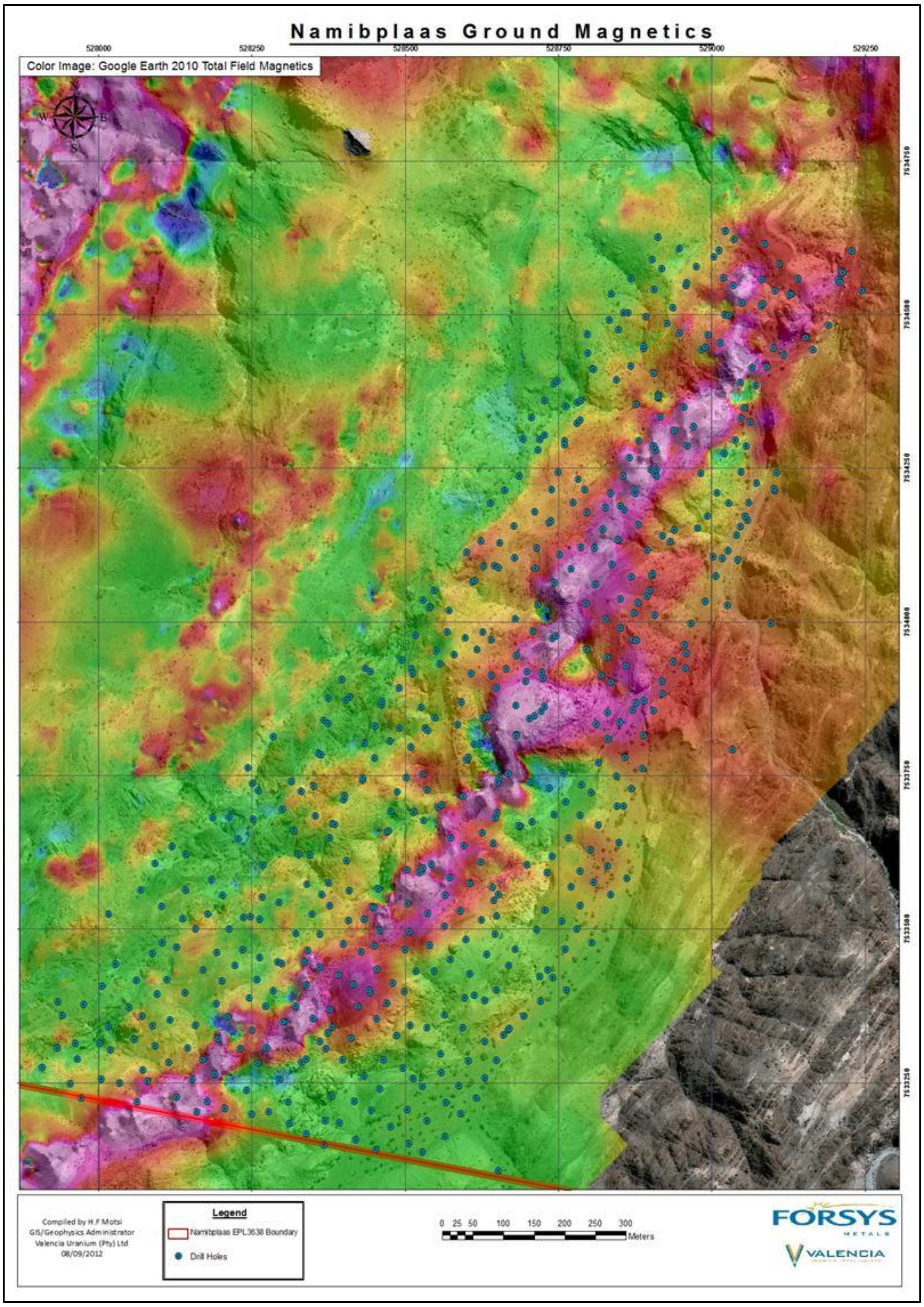


Figure 9.12 Namibplaas drillhole locations superimposed on magnetic map.

The area of higher grades corresponds to the magnetic high shown in the violet coloured zone, trending northeast to southwest.

9.3.4 Channel Sampling

Trenching was carried out in Zone B during 2010. A total of 67 channels over 1,225 meters were cut-sampled and assayed.

This process involved the use of rock saws to cut samples off the rock surfaces which were sent to the laboratory for analysis. The samples cut were 2 centimetres deep and 5 centimetres wide. The lengths of the trenches range from 8 meters to 54 meters.

Figure 9.13 Example of trench sampling at Namibplaas.

9.3.5 Digital Terrain Model

The project area was flown on 28 March 2007 for the compilation of a digital terrain model (DTM), providing contours at 2m intervals and production of digital orthophotos. These were compiled from 1:10,000 scale monochrome aerial photographs.

9.3.6 Geotechnical work

Geotechnical work in support of pit design and optimization work entails structural mapping and optical tele-viewer work which was performed on strategically positioned drillholes.

Optical Tele Viewer logs were processed by Xstrata and the resulting 3D structural dataset used by VBKom to aid latest pit design work.

9.4 NORASA

The “Norasa” term was introduced in late 2012 signifying the merger of the two Forsys projects into a single operation entity. This was justified by the geological similarities between both orebodies. The term “norasa” in the local Damara language sometimes refers to the “combining” or “bringing together of things”.

At Valencia which is positioned in the SW of Norasa, an overturned Khan anticline is deformed into Z shape folds into which the majority of mineralised granite has intruded. Rössing and Karibib marble formations are attenuated with the Chuos and Khan dominating in intact piles such that several sheets invaded up and into the Kuiseb Formation.

Disharmonic folding has duplicated the Kuiseb and created voids at the hinge zones into which granite could amalgamate in sheeted dyke or saddle reef fashion.

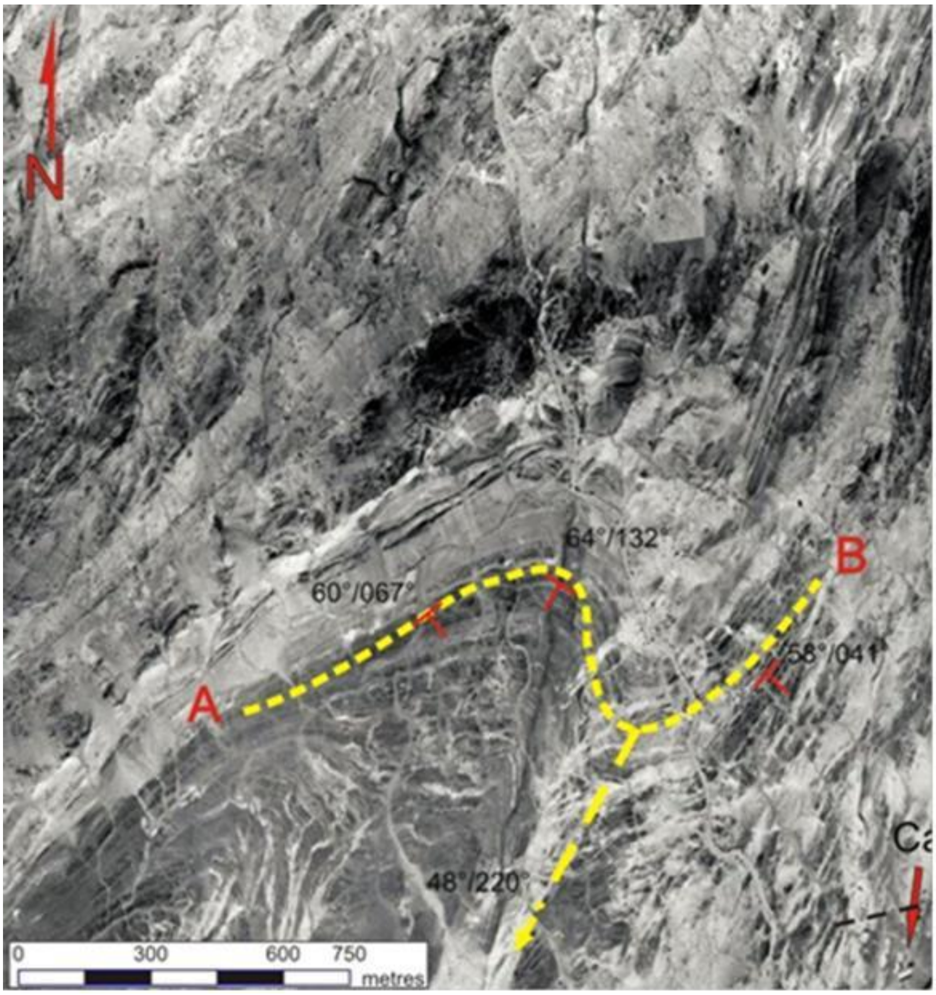


Figure 9.14 Valencia Z fold.

The contrast in local geological features to further SW positioned alaskite bodies outside Norasa (e.g. Rössing, Etango) is most prominent in the absence of a competent, ovoid - shaped dome structure such as the one daylighting at Rössing or Ida Dome. It nevertheless does have a large basement-cored anti-formal inlier towards the NW ("Joly or Valencia West").

The Rössing marble sequence is thinned and is truncated by the leucogranites.

The Valencia deposit is characterised by close proximity to basement. It is structurally-controlled and granite intrusions occur in an anticline close to the basement contact; and an attenuated Khan and Rössing sequence.

All these observations imply that the mineralised granite is sourced from below the Nosib and that emplacement is predetermined by existing structures; in this case a secondary fold on a D3 structure (folded Khan Syncline).

Moving North-East the obvious relationship and possible correlation of Valencia in the SW with Namibplaas in the NE becomes clear when using the basement inlier positioned in the NW and the most eastern high of Karibib marble layer in the SE (Figure 9.15). Along the North-Easterly strike, these markers clearly define the overall framework.

At Valencia the alaskites intrude predominantly along the secondary D3 structure (Z-fold); at Namibplaas alaskites are daylighting again SW of a similar fold feature. The D3 overprint at

Namibplaas is not as predominant as at Valencia but nevertheless is clearly developed and associated with the occurrence of alaskites. This type of occurrence is evident at the north-eastern extent of the Namibplaas deposit where the basement and overlying strata are positioned in an overturned limb with fold axes plunging NE down under Kuiseb Formation rocks, thus displaying another secondary D3 structure.

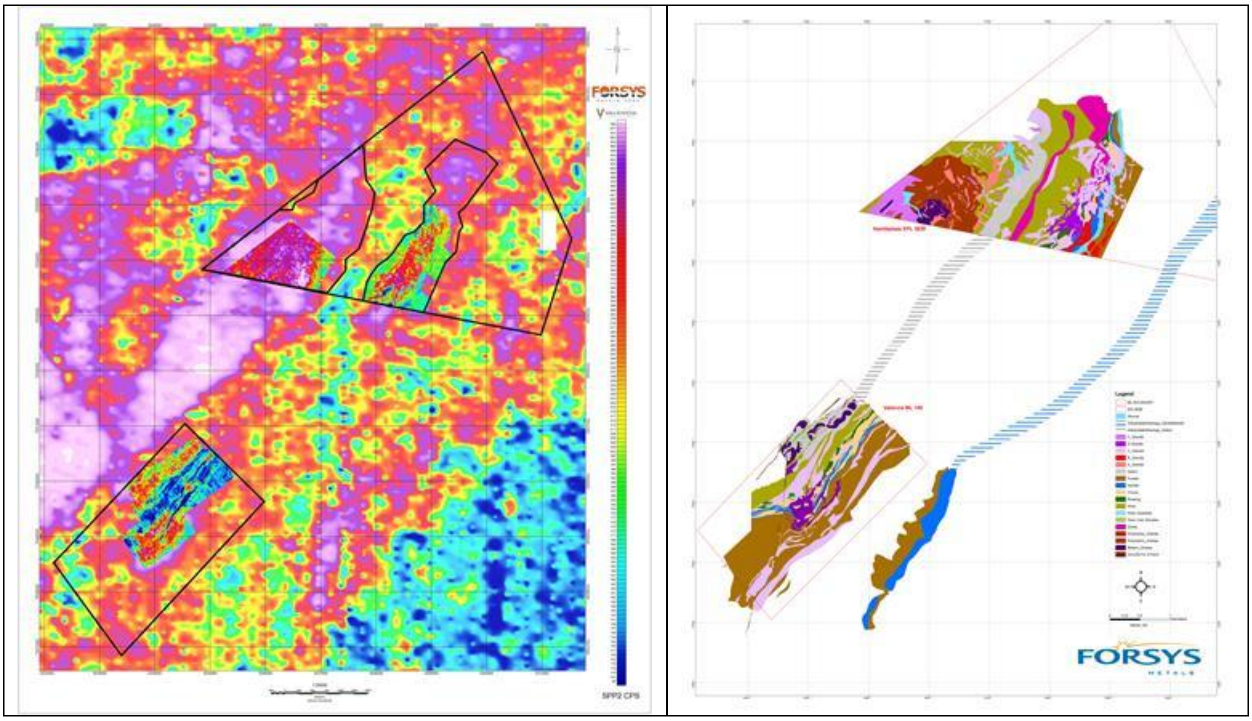


Figure 9.15 Radiometric footprint and geological relationship within Norasa.

The area between the 2 deposits is lacking occurrences of alaskites. Khan, Rössing, Chuos and Kuiseb Formation are running sub-parallel at fairly consistent strike with succession pile generally dipping towards the SE.

At Namibplaas, the succession is entering a D3 feature again and alaskites are reappearing. The Namibplaas D3 feature, despite not being as predominant as the one at Valencia, is nevertheless pointing to similar emplacement characteristics than observed at Valencia which appear to be valid for all mineralised alaskites throughout the project area.

The current emplacement theory is that structural overprinting post D3 is necessary to form pathways for the alaskites; close proximity of Khan-Rössing Formation contact to basement being another critical component. Only once these two components come together, alaskite formation and emplacement appears possible. This perhaps presenting a quiet simplistic view but in principle summarises the most critical observations when encountering mineralised alaskites in the project area.

10. DRILLING

10.1 VALENCIA

10.1.1 DDH drilling

Between 1974 and 1984 Trekkopje Exploration drilled 97 DDH over approximately 25,000m with Valencia adding another 44 DDH over 13,082m during period 2008 to 2009.

The majority of the DDH were drilled at declinations of 45°. Due to the massive and irregular nature of the alaskites, the majority of the sample lengths are the approximate true width of the alaskite bodies. The core was predominantly BXM size (core diameter 41.7mm), with a lesser amount of NXM size core (core diameter 54.5mm) drilled through the first few metres of weathered surface rock.

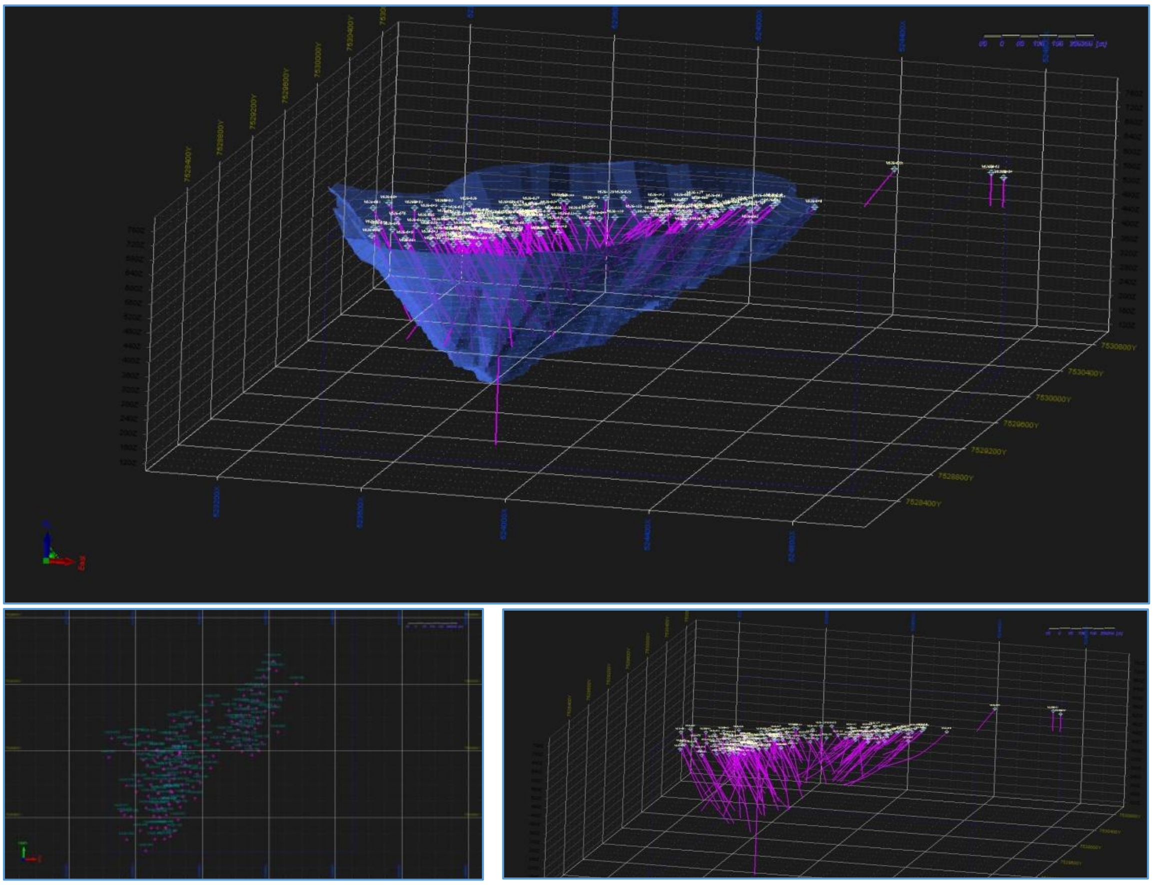


Figure 10.1 DDH drill hole collars VA26-001 to 152 (145 holes).

10.1.2 RC drilling

148 RC drillholes over 11,111m covered an area of 300m by 200m and down to a depth of 105m below surface to achieve increased resource confidence. Figure 10.2 showing position in relation to the Valencia pit.

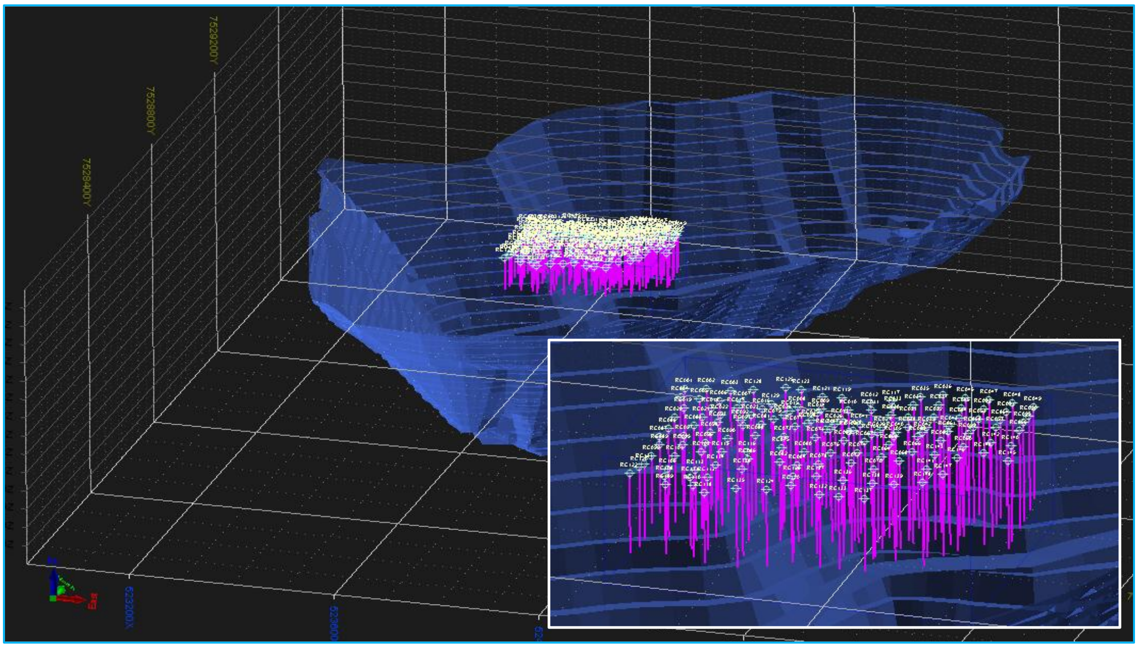


Figure 10.2 RC drill cover in the Valencia pit.

10.1.3 PD drilling

Percussion drilling formed the main tool to explore after DDH and RC provided for the geological guidance and establishment of the correlation coefficient from gamma probing to equivalent uranium conversion. There were 409 drillholes drilled over 110,095m until 2011.

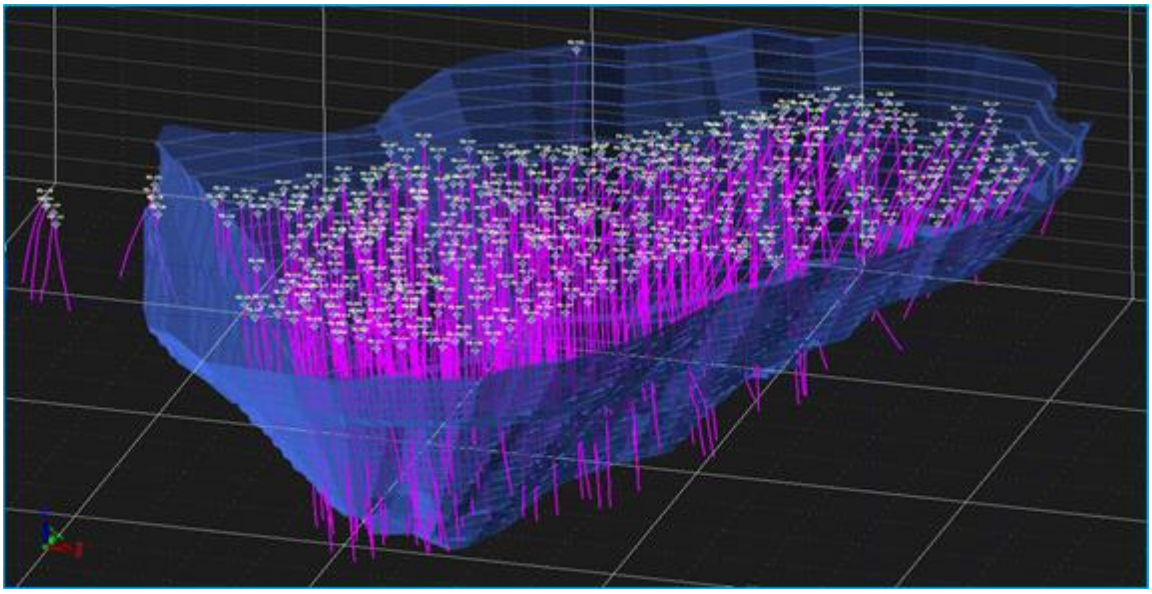


Figure 10.3 Percussion drill holes covering majority of the Valencia pit.

10.2 VALENCIA SATELLITE

During 2012 and 2013 exploration efforts moved to a small area which is close to the main Valencia anomaly. The area is called the Valencia satellite and characterised by a higher grade surface signature measuring 500m by 400m on surface (Figure 10.4).

Valencia decided to drill test the area and test mineralisation at depth. The program succeeded in confirming higher grade mineralisation at depth close to the Valencia main anomaly and incorporated the results into the Valencia resource statement.

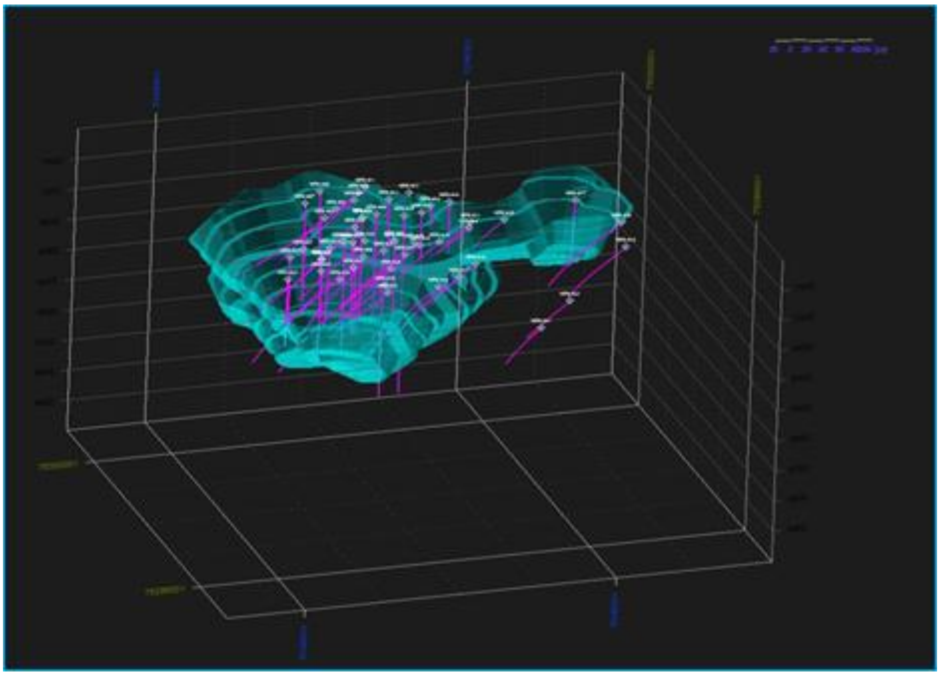


Figure 10.4 Valencia satellite optimised pit with PD drill collars & traces.

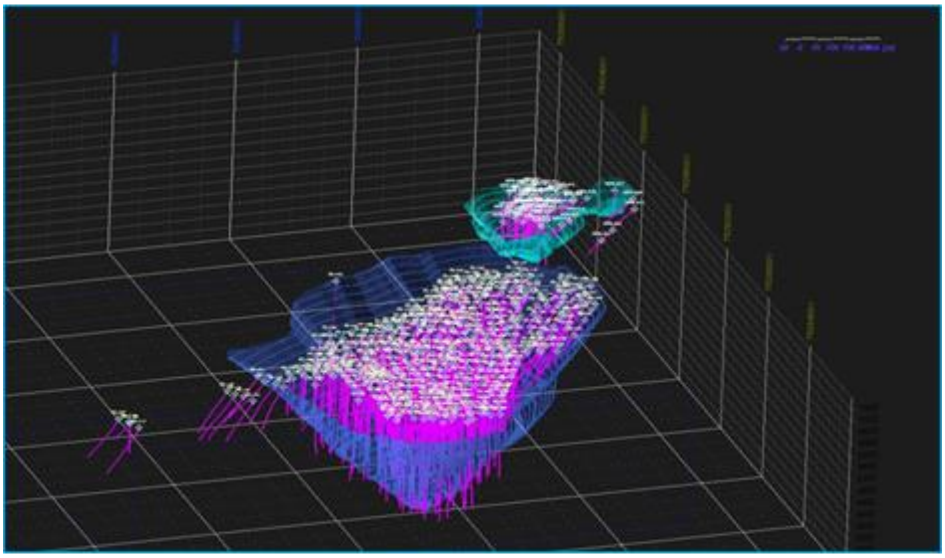


Figure 10.5 Relative position of the satellite pit to Valencia.

There were 52 PD holes drilled to shallow depths.

Geology information was collected from drill chips logged in usual manner as in previous campaigns. Valencia geologists collected drill chips at 1m intervals from the drillrig and laid the chip samples out close to the rig. A representative sample is taken and stored for reference purposes. This simple but effective process has been applied throughout all percussion drilling campaigns (Figure 10.6).

Figure 10.6 Geology chip sample layout and logging (1m intervals).

Grade information is derived from geophysical logging at 0.1m intervals with the gamma readings empirically converted into a U_3O_8 grade. The probing protocol and procedures applied are consistent with applications at previous probing campaigns at Valencia and Namibplaas.

The geophysical probe data is collected at 0.1m intervals downhole and converted into grade thickness (GT) using a correlation coefficient which Dr Laine developed in 2008, and which Snowden confirmed 2009 being acceptable and correct.

Calibration of the probe is undertaken on a daily basis using a fixed source and on a weekly basis, running the probe down a reference drill hole that had been fully sampled down the hole. Allowance is also made for the presence of radon, and holes are re-probed until the results are consistent.

One of 3 available data logger systems (GeoVista) *Calibration source used at daily start up*

Figure 10.7 Probing equipment at Valencia and process of daily calibration.

10.3 NAMIBPLAAS

10.3.1 Diamond drilling

Figure 10.8 below illustrates the DDH collar position over a geologic map background.

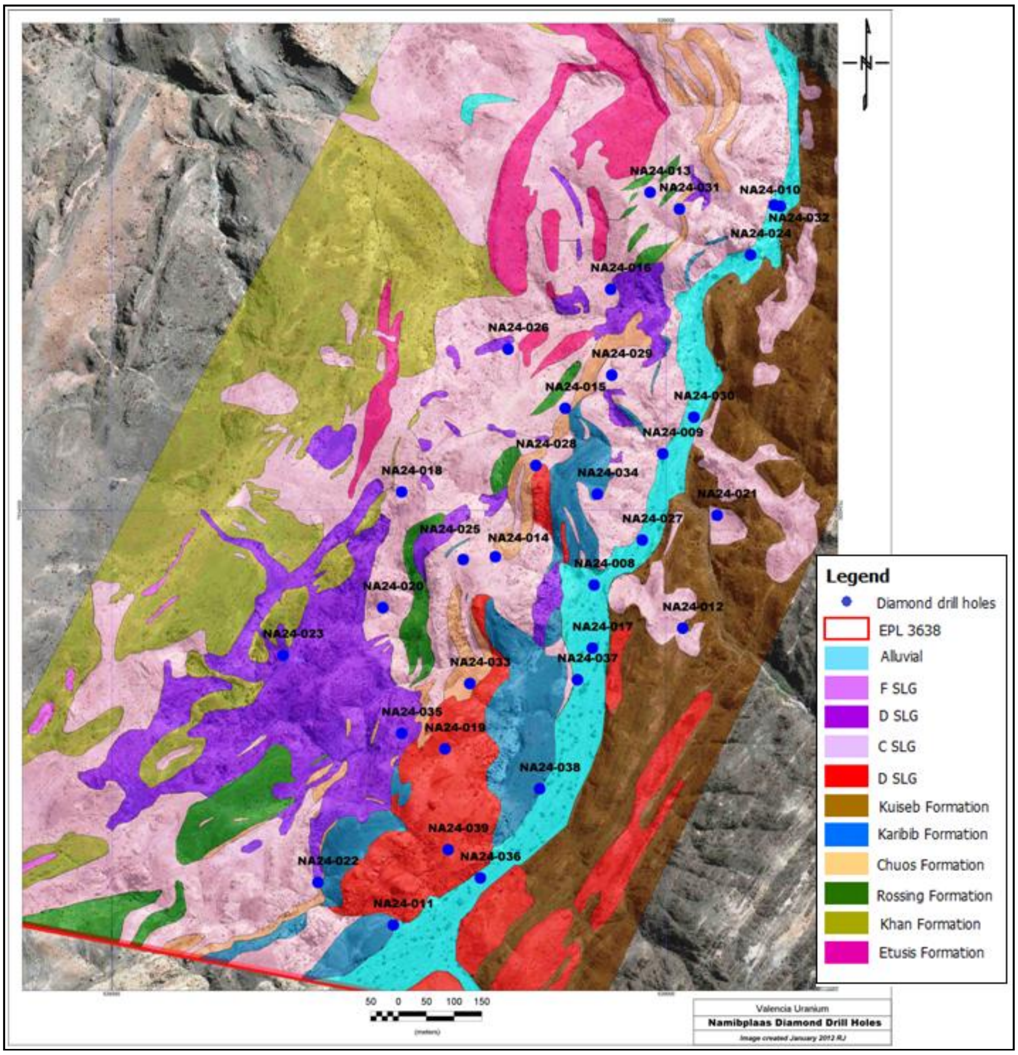


Figure 10.8 Namibplaas DDH positions superimposed onto Geology map.

There were 19 DDH holes drilled (NA24-008 to 026) starting in July 2010 by drilling 4,667m until year-end and continued with another 14 holes (NA24-027 to 040) drilled over 3,561m in 2011. The resulting total DDH dataset, including the historical Goldfields drillholes is covering 9,894m and has been utilised fully for correlation and estimation work (Optiro, 2011).

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

Reference to previous technical reports which are filed on the SEDAR web site, particularly the Snowden 2009 and 2010 reports as well as, the Optiro 2011 report is required as no new samples were analysed. Detailed below is a compressed summary of historical work and various *quotes* from these reports.

11.1 TREKKOPJE EXPLORATION (1974 TO 1984)

Historical assays were validated by probing (with total count GM probe) of 12 holes from the Trekkopje Exploration campaign and using the correlation (see Figure 11.1) which was established from Valencia (DDH drilling by Dr. Laine[2] in 2008).

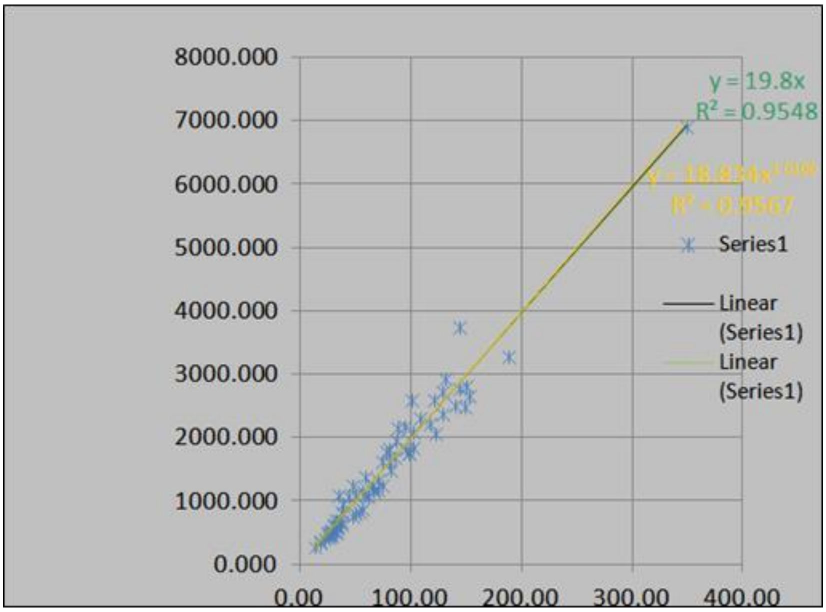


Figure 11.1 Correlation coefficient c/s into equivalent uranium (Laine, R. Dr, 2008).

The correlation curve between historical chemical assays and equivalent U_3O_8 was excellent with a slope of 1 and correlation coefficient of 0.99; 68 mineralized intervals were used in the comparison.

Trekkopje Exploration reported that the samples were transported by rail to the Gold Fields Laboratory Pty Ltd (GFL), Johannesburg, South Africa, in locked steel trunks. The samples were analysed by GFL for U_3O_8 using a fused pellet XRF method.

Analytical quality was monitored by two randomly selected duplicates and one reference material analysis per batch of 15. Check samples were regularly selected at random from the pulverised pulp, for wet chemical assay. It was also not documented whether GFL was certified with a standards association.

It was noted by Labuschagne (1979) that the repeat assays and a comparison of assay values to handheld scintillometer counts, identified some discrepancies in both sample preparation and sample

[2] Dr. Roger Laine was Chief Geologist at Forsys Metals Corp at the time.

labelling. These discrepancies could not be quantified and the conclusion by Labuschagne (1979) was that over a total number of several thousand samples the errors should cancel out and have an insignificant effect on the "viability" of the deposit.

Snowden considered that the sampling, sample preparation, data collection and security measures applied by Trekkopje Exploration were carried out according to the best practice available at the time and that it was done in an acceptable and systematic manner.

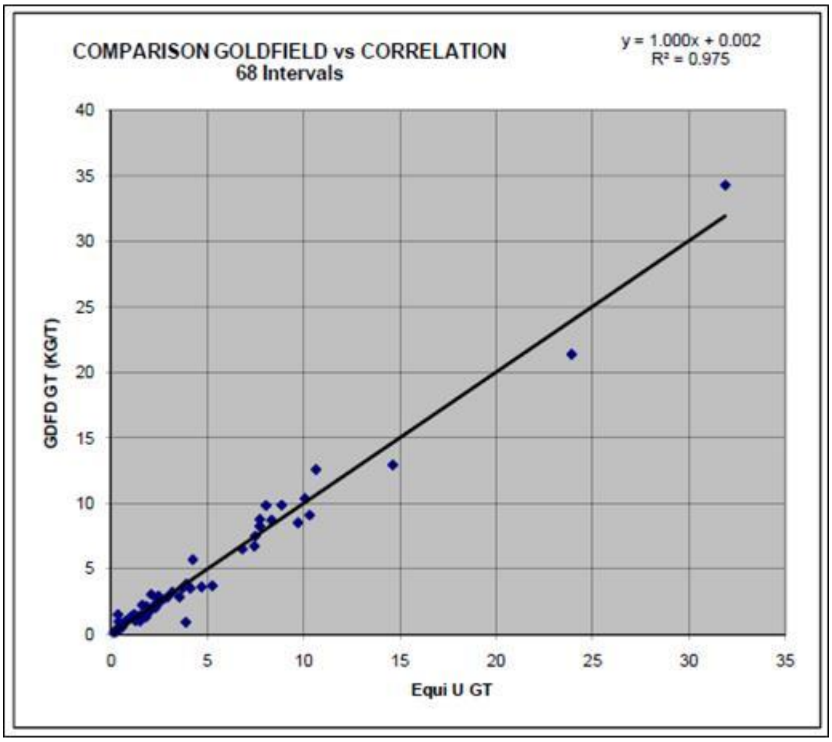


Figure 11.2 Goldfields assays vs Valencia correlation.

11.2 VALENCIA AND NAMIBPLAAS

All DDH half core and RC samples collected by Valencia were assayed at the Setpoint Technology (Setpoint) laboratory in Johannesburg, South Africa. Setpoint is accredited with the South African Accreditation System (SANAS), accreditation number T0223 and is also an ISO17025 accredited laboratory. Setpoint completed the crushing and pulverising of the samples to industry standards. The prepared samples were analysed for U_3O_8 using the XRF pressed pellet method.

Valencia did not include standard reference material (SRM) or blank material in the first submissions to Setpoint. However, SRMs were included in pulps that were returned to the laboratory for re-assay. The Setpoint laboratory included appropriate quality assurance and quality control (QA/QC) procedures during the analysis of the Valencia samples by including certified reference standards (CRM), blanks and duplicates. The protocols for the QA/QC are as follows:

- Commercial CRMs inserted at a frequency of at least one per 20 samples.
- Blanks inserted at a frequency of at least one per 50 samples.
- Duplicates taken at a frequency of at least one per 20 samples.

11.2.1 PD samples

PD drillholes were not physically sampled but were probed with a downhole scintillometer every 0.1m down the hole. Sample risk factors calibration of the scintillometer was undertaken on a regular basis, running it down drillholes that had been fully sampled down the hole. Allowance was also made for the presence of radon with holes being re-probed until the results were consistent. Drill chips were collected for rock type classification at 1m intervals.

11.2.2 Bulk density

Valencia used the results of the bulk density determinations from 200 samples of various rock types using the weight in air and water method. The samples were taken from ten DDH cores.

Valencia further used the results of 2,214 specific gravity determinations conducted by Setpoint using a vacuum pycnometer. These measurements provided an indication of the specific gravity of the minerals and do not account for porosity or voids in the rock. These measurements were therefore not used for estimating the tonnage in this phase of work; however, they have been used to provide a check on the weight in air and water bulk density determination method.

12. DATA VERIFICATION

12.1 VALENCIA

Snowden visited the Valencia project in September 2005, November 2006 and in November 2008. During each visit drill core and core logs were inspected and collar positions verified.

Table 12.1 DDH drillholes verified by Snowden

September 2005 Drillhole	November 2006 Drillhole	November 2008 Drillhole
VA26-011	VA26-115	VA26-135
VA26-017	VA26-116	VA26-138
VA26-036	VA26-125	VA26-140
VA26-058	VA26-127	VA26-142
VA26-059	VA26-130	VA26-145
	VA26-133	VA26-147

Drillhole collar positions were inspected as per table above and positions of their collars were verified by using a Garmin 12 CX handheld global positioning system receiver (GPS) and by comparing these coordinates with the positions depicted on the plans.

12.2 NAMIBPLAAS

Optiro visited the project area in July 2011.

Optiro inspected the position of numerous DDH and PD drill collars with respect to known coordinates. Optiro considers that the collar positions were confirmed within a reasonable distance of the positions on plan.

The cores from three DDH, as listed in Table 12.2, were compared with the drillhole logs. No discrepancies between the drillhole logs and the core were identified.

Optiro also confirmed the radioactive anomalism at the site by carrying out its own scintillometer readings of outcrop (Figure 12.1) and the core that was reviewed.

Table 12.2 Drill core inspected, July 2011

Drillhole		
NA24-008	NA24-011	NA24-036

Figure 12.1 Optiro reading adjacent to drillhole NAPD-077.

Figure 12.2 Optiro reading drillhole NA24-011 ~ 56m downhole depth.

Optiro also re-sampled material from the drillholes presented in Table 12.3 together with the original and re-sampled results. The re-sampled material returned similar values to the original results.

Table 12.3 Optiro resample material

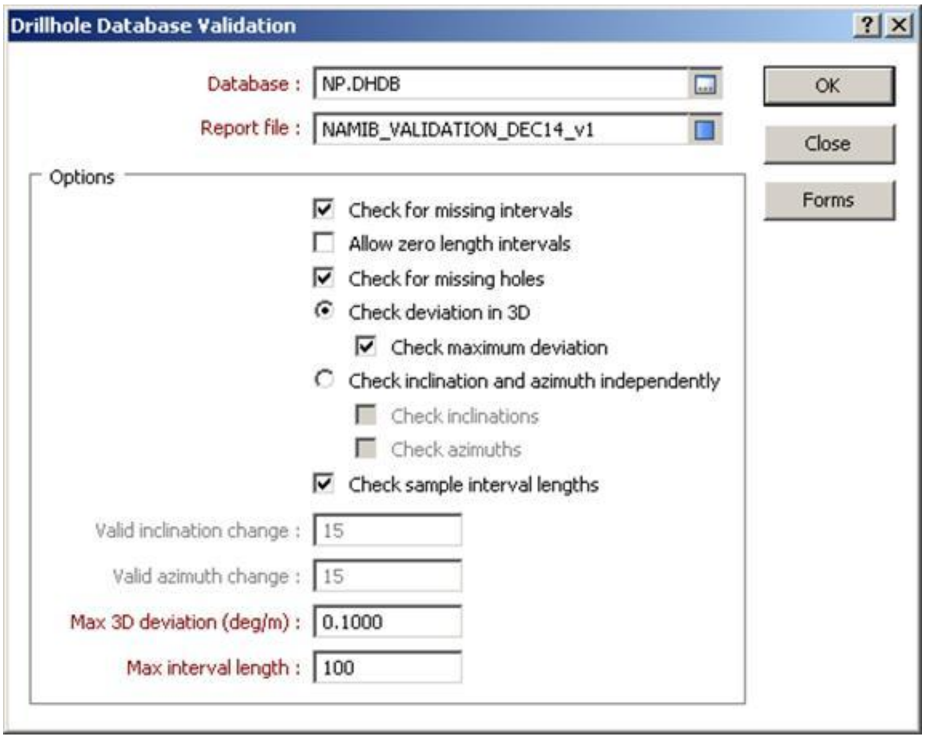
BH ID	SAMP ID	U_3O_8 ppm	U_3O_8 ppm*
8	21002	311	321
8	21011	868	929
8	21012	208	184
8	21016	730	759
11	20751	282	282
12	20203	307	269
12	20204	223	204
16	20873	151	143
16	20874	129	121
25	21486	198	202
25	21487	144	139

*re-assay result

12.3 DRILLHOLE DATABASE VALIDATION

All historical and new drillhole information was transferred from previous Excel and Techbase environments into Micromine's 3D environment. Relevant event tables have been validated in the process using off the shelf mining modelling software MicroMine 2014 (version 15.0.3 build 608, 32bit) database validation tools.

Valencia, Valencia Satellite and Namibplaas's drillhole database validation of subsets (ASSAY, CPS, GAMMA, MAG and GEOLOGY) were assessed individually.



12.4 STATEMENTS REGARDING VERIFICATION

Both, Snowden and Optiro confirmed that the data used for the various mineral estimates is reliable. Snowden (2009) is of the opinion that the data used in the January 2009 Mineral Resource estimates is reliable. Optiro (2011) is of the opinion that the data used in the September 2011 Mineral Resource estimates is reliable.

Further to the above, Valencia has undergone additional database reconciliations and is confident that the data used in this updated Mineral Resource Statement is accurate and reliable.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

Norasa (and formerly Valencia) processing flow sheet has been developed and evaluated in a number of testwork programs and economic studies between 2007 and 2014. As the project combines the former Valencia Uranium Project with the more recent developed Namibplaas Uranium Project, historic testwork related to both these projects are covered in this section. The flow sheet has evolved over time and the recent 2014 pilot testwork campaign focused on evaluating the impact of some of these changes and the integrated nature of the flow sheet. The objective of this program was to confirm performance of an optimised flow sheet on a feed composite representative of the 2014 mine plan.

The main unit operations included in the flow sheet and covered by testwork include comminution, sulphuric acid leach, solid-liquid separation and ion exchange recovery of the leached uranium from the pregnant leach solution (PLS). In the subsequent discussion the following applies:

- Where the term “acid” is used it refers to sulphuric acid, H_2SO_4.
- Pyrolusite has been used as oxidant in leach for many of the acid leach tests. The active ingredient is manganese dioxide or MnO_2, and consumption of oxidant is reported in terms of MnO_2.

The following summary development time-line should provide the reader with perspective on the various test programmes:

- 2006 to 2007: Valencia PFS
- 2009: Valencia DFS
- 2010: First Engineering Cost Study (ECS)
- 2013: Second ECS
- 2014: DFS update, incorporating ECS findings.

13.2 TESTWORK PRIOR TO 2014

Earlier testwork programs are subsequently discussed.

- Historic Testwork (prior to 2008) - Preliminary mineralogical characterisation and metallurgical tests were performed in various test campaigns – initially by Trekkopje Exploration (1979) and by SGS Lakefield (2006/7).
- Mineralogy by SGS Lakefield (2008) - Valencia ore and gangue were characterised by the QEMSCAN system. Further testing included bench-scale comminution and leach bottle roll tests. Additional testwork was subsequently completed to investigate stirred tank bulk leaches and optimise MnO_2 consumption.
- Metallurgical Testing by Mintek (2008/9) - A second phase of testwork evaluated different oxidants in leach.
- Bench Scale Comminution Testwork by SGS Lakefield (2009) - Additional variability testwork was carried out on 27 diamond drill core samples representing the Valencia orebody and mine plan. These included determination of comminution parameters and leach tests.

- Neutralisation Testwork – Mintek (2009) - Testwork was completed on the neutralisation properties of the ore (at different particle size distributions – PSDs) to assess the implications of grinding the ore in recycled barren ion exchange (IX) liquor.
- Radiometric Sorting Bench Testing (2009) - Rock samples from a Valencia test pit were subjected to radiometric sorting in a test rig to predict the cut efficiency of a full scale sorting plant. Uranium recovery and rock rejection efficiency curves against grade were developed.
- Namibplaas Mineralogy – SGS (2011) - SGS conducted a mineralogical evaluation of five Namibplaas diamond drillhole ore samples, with four uranium phases identified: U-oxides, U-silicates, U-Ti-oxides and U-Th-silicates.
- Valencia Variability Leach Tests – SGS (2012/3) - Bench-scale acid leach tests were conducted on 48 diamond drill core samples, which were selected to evaluate orebody variability. The samples provided a range of uranium grades that can be expected to be delivered to the leach tanks and were selected spatially in consideration of the proposed life-of-mine pit shell at the time.
- Leach Tests at P80 = 600 µm Grind – SGS (2013) - Subsequent to the completion of the AMEC Engineering Cost Study Report in May 2013, which identified secondary crush followed by SAG milling as the preferred comminution route, a series of variability tests were conducted by SGS South Africa to evaluate the leaching behaviour at the proposed SAG mill circuit grind size under a set of standard conditions. Acid leach tests were conducted on 20 Valencia samples and 10 Namibplaas samples which were prepared to 80% passing 600 µm. Bottle roll leach conditions comprised 50%(w/w) solids content, pH 1.8, ambient temperature, and redox potential of 480 mV (Ag/AgCl) using MnO_2. Valencia sample grades varied from 71 to 343 ppm U_3O_8 (averaging 199 ppm), while the Namibplaas samples ranged from 139 to 388 ppm U_3O_8 (averaging 212 ppm). The average U_3O_8 grade was designed to align with the average process feed grade of the mine schedule at the time.

13.3 COMMINUTION TESTWORK

The comminution circuit has been designed from interpretation of bench-scale testing of drill core samples sourced from within the proposed Norasa open pit. The following two phases of testing included comminution evaluation:

- Laboratory Testwork to Evaluate the Recovery of Uranium Ore Sources Originating from the Valencia Deposit, O'Connell and Klaas of SGS, Interim Report number MET 08/V66, Feb 2008.
- Laboratory Testwork to Support a Comminution Circuit Options Study, Hobbs and Klaas of SGS, Report number MET 08/460 Rev 3, November 2009.

The testing in 2008 was on composite samples selected by Forsys Metals and included head analysis, and limited JK drop weight, Bond crushing, ball, rod and abrasion index testing. Composite samples described as granite, marble and schist were compared for Bond ball mill, Bond rod mill and SMC properties. Generally moderate rock competency and rod and ball mill grinding outcomes resulted for this first pass evaluation. Only the schist sample is noted to have high rock competency (25.0 Axb) and would justify further testing if deemed to form a significant component of mill feed (e.g. >10% on a continuous monthly basis). It is noted in this program that reference between the composite make up and drillhole origin is not clearly reported for samples that underwent comminution testing.

The latter phase of testing reported in 2009 was more comprehensive as a variability assessment and provided Bond ball mill work index data that better aligned with the P_{80} 600 µm grind size adopted in the BFS flowsheet. In total 27 individual core intervals were tested for SMC (26.5 to 31.5 mm fraction), Bond ball mill work index (212 µm closing screen) and Bond rod mill work index (1,180 µm closing screen). In addition, Bond crushing work index and unconfined compressive strength (UCS) tests were undertaken on intervals representing the three major rock types, namely granite (ore), schist and marble (waste). Leach testing of composite samples at variable grind size (P_{80} 212, 425 and 850 µm) and pulp density (45%, 55% and 65%) provided the uranium leach extraction and reagent consumption data to support the trade-off study. A selection of ten samples also underwent Bond abrasion index testing.

13.4 TESTWORK CAMPAIGN IN 2014 – SGS (PTY) LTD SOUTH AFRICA

Additional ore characterisation and leach testwork was conducted on Valencia composite samples from July to October 2014 by SGS South Africa in Johannesburg.

Subsequent to the earlier testwork in support of the 2008/9 DFS, the follow-on engineering cost studies identified additional test requirements to confirm the associated flow sheet developments. Further deposit and mine plan development also resulted in a requirement to confirm the performance of a representative composite in the optimised flow sheet.

The testwork covered in the 2014 campaign and reasons for these tests are as follows:

1. Preparation of an overall average U_3O_8 grade composite with a particle size distribution (PSD) comparable to that expected from the two stage crush followed by SAG mill comminution configuration. A P_{80} of 600 µm was targeted. A silica crushing/grinding test was performed using the same preparation methodology as that for the composite to determine the non-ore associated iron deportment to the composite sample.
2. Full composite head elemental analysis (by XRF) and metallurgical characterisation by XRD to serve as reference for testing.
3. Leach iron addition optimisation (as ferrous sulphate) by bottle roll batch tests to determine the leach iron requirements and gain a better understanding of the iron chemistry for this ore.
4. Determine the reagent requirements in leach for a closed circuit where IX barren liquor is recycled, while maintaining the optimum leach conditions determined in the bottle roll tests.
5. Packed bed ion exchange was used to close the test circuit and generate barren solution for recycle to leach. At the same time the ion exchange resin performance was confirmed by operating the system to mimic a multi-stage counter-current operation (i.e. simulate NIMCIX).
6. Viscosity testwork on leached slurry to gauge pumping and agitation requirements at different pulp densities.
7. Settling testwork on pre-leach and post-leach slurry to generate clarification and thickening data that could potentially be required for flow sheet improvements at a later stage. Having representative leach slurry available, it was decided to make use of the opportunity to generate this data.
8. Filtration testwork to evaluate the filtration, washing and drying performance of the leached solids at the optimised grind size (P_{80} 600 µm) and PSD.
9. Filter cake characterisation by determining Atterberg limits.

Refer to the SGS Report No. 14-264 Rev 5 (18 November 2014) for full results and the AMEC Report No. 060551-0000-DW00-STY-0001 Rev 0 (Section 5) for interpretation of the testwork and associated flowsheet implications.

13.5 CONCLUSIONS AND RECOMMENDATIONS

13.5.1 Conclusions

A composite metallurgical sample was prepared from a series of samples selected by Forsys within the proposed Valencia life of mine pit. The composite is considered indicative of a life of mine average process feed and was prepared to a target uranium grade of 200 ppm U_3O_8 and the resultant solid analysed to between 205 and 216 ppm U_3O_8.

Locked cycle metallurgical pilot testwork was performed on the average feed composite, including acid leach and ion exchange. The main findings are:

a) No iron addition is required to leach when recycling barren liquor due to iron leached from the ore and recycled.
b) An average leach acid dosage of 12.2 kg/t (as 98% H_2SO_4) is required, excluding grinding media and recovery associated acid consumption and losses.
c) Hydrogen peroxide is preferred to MnO_2 as leach oxidant as it is more efficiently utilised and consumes less acid. The required 50% H_2O_2 dosage is 1 kg/t.
d) The agitated tank bulk leach uranium extraction averaged 93.2%.
e) High IX resin loading of around 20 to 30 g U_3O_8/L was obtained.
f) Leach residue vacuum filtration tests were done, concluding that Magnafloc 333 gives the best filtration performance. The filtration tests showed that the material dewaters and washes at a high rate, but drying takes a significant time. However, at the associated Atterberg liquid limit determined at 23%(w/w), the drying allowance can be limited which gives a reasonable overall filtration flux of around 1,600 $kg/m^2.h$.

These results are seen as generally consistent with earlier testwork campaign's findings, taking cognisance of the benefits obtained by changing oxidant and recycling barren liquor. Both these flowsheet improvements contributed to a significant reduction in leach acid consumption. The higher leach extraction and associated overall circuit uranium recovery resulted from improved mixing conditions in the larger scale leach vessels used for pilot testing and a reduction in filter soluble loss due to wash optimisation.

Although the pilot test composite is representative of the uranium deportment in the Valencia pit as confirmed by comparison of the uranium distributions for the sub-samples and mine block model, the data on the associated gangue composition and variability of the samples and mine block model is limited.

No Namibplaas samples have been tested to date. However, this material is considered geologically similar (coarse grained granitic) to the Norasa mineralisation and is scheduled as a moderate component of mill feed after year 9 in the BFS mine plan (Nil feed until 2025). For these reasons this risk is not considered material to the project economics at this stage of study.

13.5.2 Optimisation Opportunities

The following opportunities have been identified, and warrant further work:

- It is recommended that gangue variability in the orebody be investigated in future work. Some of the important gangue minerals or chemistry include:
  - Carbonates, which affect leach acid consumption
  - Iron containing minerals such as mica and chlorite, which will contribute to leach liquor iron content
  - Chloride, which builds up in the process liquor and affects uranium extraction efficiency.
- Analysis of the multi element chemistry to comminution property relationships as with better defining the schist and carbonate zones within the resource and mine/milling schedule.
- Comminution and leach extraction testing of material from the Namibplaas mineralisation.
- Grinding media consumption test work in acidic pulp to compare and optimise full scale consumption prediction. Testing chrome steel media with chrome content ranging between 5 and 30% is proposed.

- Pilot SAG milling could be beneficial as an optimisation phase and should be considered if further downstream testwork is justified. Optimising the actual SAG mill specific energy and fines screening efficiency may offer improvement potential.
- Purolite's PPA800 resin was used in the pilot testwork. However this resin comes at a price premium compared to Dow's Ambersep 400 resin. There is therefore potential to significantly reduce first fill and resin consumption costs by using the alternative resin, contingent on its performance. Comparative IX performance tests on Ambersep 400 resin to confirm equivalent loading kinetics and capacity to the Purolite PPA800 resin is therefore recommended.
- The high resin loadings obtained in the latest testwork campaign will result in high eluate uranium tenors which are more than double the tenors used for earlier SX extraction testwork. Further SX testwork is therefore recommended at eluate tenors between 5 and 6 g U_3O_8/L, preferably generated from IX elution, to generate updated organic loading isotherm(s). This would provide the opportunity to reduce the size of the SX and associated ADU precipitation circuits.

14. MINERAL RESOURCE ESTIMATES

14.1 VALENCIA

Snowden undertook a first resource estimate in October 2005 with subsequent updates in June 2007, January 2009 and then again in June 2009 incorporating an additional 200 drillholes for 49,562m utilising the full DDH, RC and PD drillhole datasets.

The same methodology was employed for the latest update as for the earlier estimate except that the block size for the ordinary kriging (OK) estimate was reduced from 30m x 30m x 5m to 20m x 20m x 5m.

The uranium mineralisation is associated mainly with the alaskite intrusive and given the complexity of the lithology between drill sections, Snowden elected to use conditional simulation to model the distribution of the alaskite intrusives at the time as opposed to conventional sectional interpretation and wireframing techniques.

Snowden composited the supplied lithology file to 1m intervals based on lithology as coded by Valencia. All composited intervals logged as alaskite intrusive (log codes GRT or BGRT) were flagged with a rockcode (RCODE) of RCODE=GRT.

They were then assigned a numerical code (NCODE) of 1, while all non GRT intervals were assigned NCODE=0. Variography was then undertaken on the numerically coded lithology data. A normal scores transform was used to assist in resolving the directions of maximum continuity. The normal scores variogram was modelled and then back transformed into normal space. The modelled variogram parameters used are presented in Table 14.1. The sill is normalised to 1 and spherical structures were modelled.

Table 14.1 Back-transformed variogram parameters used in simulation of GRT (Snowden, June 2009)

	Direction		Nugget	Structure 1		Structure 2		Structure 3	
				Sill	**Range (m)**	**Sill**	**Range (m)**	**Sill**	**Range (m)**
D1	20	0	0.15	0.59	20	0.15	95	0.11	400
D2	290	-20			20		125		195
D3	290	70			20		80		230

The modelled variogram parameters were used as the basis of the conditional simulation of the GRT lithology. A 5m x 5m x 5m spaced node file was established reflecting the limits of the drilling. The node size was selected to maintain a node file of manageable size with enough resolution to reflect the distribution of the GRT lithologies. The 5m x 5m x 5m spacing reflects the smallest selective mining unit (SMU) perceived achievable when mining the Valencia resource. Fifty simulations were generated and those nodes with a probability of >0.5 of being GRT were then selected to be used as the volume model for estimation.

The sample intervals were composited to 1m intervals commencing at the drillhole collars. The compositing process accounts for length weighting. The composite length was optimised by a statistical analysis of sample intervals. All drilling logged as alaskite was used, with the 1m interval selected to reflect the RC drilling and compositing length of the drill holes which were geophysically probed to determine gamma readings.

Geophysical probe data was collected every 0.1m downhole. The relationship used in the estimate (U_3O_8 = 17.601 gamma + 7.39) was applied to the gamma data from the PD drill holes for the logged GRT 1m composited intervals.

The combined DDH, RC and PD data was used as the basis of the resource estimate. All U_3O_8 assay data from DDH and RC drill holes was used. Gamma derived values were used if there was no U_3O_8 assay value in the database.

Snowden applied a grade cap of 1,000 ppm to the data.

Variography was then undertaken on the combined assay data. A normal scores transform was used to assist in resolving the directions of maximum continuity. The normal scores variogram was modelled and then back transformed into normal space. The modelled variogram parameters are presented in **Error! Reference source not found.**. The sill is normalised to 1 and spherical structures were modelled.

Table 14.2 Back transformed variogram parameters used in estimation of U_3O_8 (Snowden, June 2009)

	Direction		Nugget	Structure 1		Structure 2		Structure 3	
				Sill	Range (m)	Sill	Range (m)	Sill	Range (m)
D1	50	0			20		80		390
D2	320	30	0.12	0.75	20	0.11	40	0.02	175
D3	320	60			20		100		175

A three-dimensional model was created below the topographical surface wireframe by re-blocking the volume model generated by the conditional simulation of the alaskite lithologies to dimensions 20mE x 20mN x 5mRL. Ordinary kriging using Datamine mining software was used to estimate U_3O_8 grades into the 3D block model. Parent cell estimation was carried out and sub-cells were assigned the grade of their parent cell.

The search applied, used the ranges and orientation modelled in the variograms as the basis of the estimate. The minimum number of composites used for the interpolation was ten (10) and the maximum number of samples used was forty-two (42). A bulk density value of 2.63 t/m^3 is used.

The maximum depth of the resource had been constrained to a depth of 380m below topography which is equivalent to an elevation of 325.5m amsl for the lowest row of blocks (block centres).

A total of 100 conditional simulations were run and the 50th percentile of the simulations was selected (P50). Metal content was than reported above a 60 ppm and a 100 ppm U_3O_8 cut-off.

The Mineral Resource utilised anticipated SMU sizes of 10m x 10m x 5m to represent a diluted mining unit.

The ordinary block kriged estimate considered only the alaskite mineralisation; grade information from non-GRT composites (GRT, BGRT) was not reflected.

The relevant Technical Reports outlining the work and results have all been filed on the Sedar website (all by Snowden, January 2009; June 2009; January 2010).

Valencia reviewed the historical resource model early in 2012 and concluded that there were minor shortcomings requiring a review of the applied geostatistical methodology and in particular its application.

The first step involved extending the model downwards by adding blocks from elevation 352.5m amsl down to elevation 302.5m amsl. This was followed by a complete reassessment of the available drillhole data.

Separate drillhole databases were established:

- information derived from laboratory assay data
- information from radiometric gamma probing, which was further separated into the various drill types; diamond drill, reverse circulation and percussion drilling.

The different datasets were composited over weighted length and statistically compared. Intervals length of 0.5m, 1.0m, 1.5m and 2.0m were processed using only alaskite rock coded as GRT, AGRT or AGRTT identifies segments. A 60ppm cut-off was applied to cater for edge effects around alaskite / country rock contacts.

Resulting variogram structures pointed to the 1.0m interval length remaining the most appropriate used for grade interpolation; this confirming earlier work from Snowden in 2009.

The following variogram parameters were established and used for grade interpolation using ordinary kriging (OK).

Table 14.3 Variogram parameter U_3O_8 DD assay

	Direction		Nugget	Structure 1		Structure 2		Structure 3	
				Sill	**Range (m)**	**Sill**	**Range (m)**	**Sill**	**Range (m)**
D1	50	0	0.0428	0.0893	8	0.0484	54	0.057	134
D2	140	0			7		31		141
D3	140	90			3		23		119

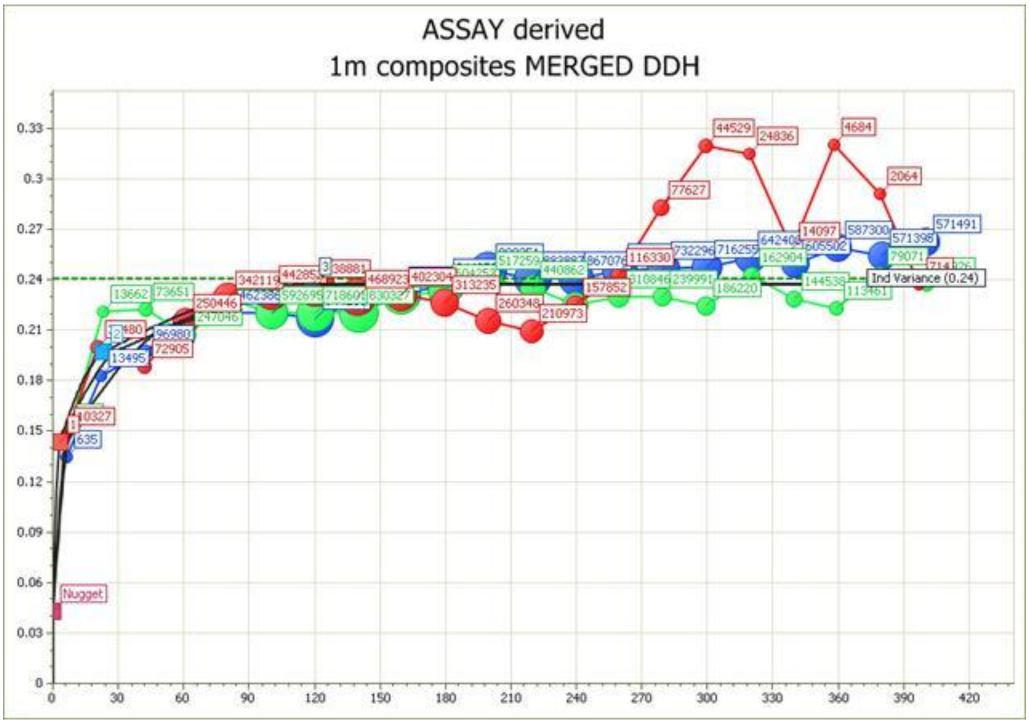


Figure 14.1 Valencia Main assay composite variogram

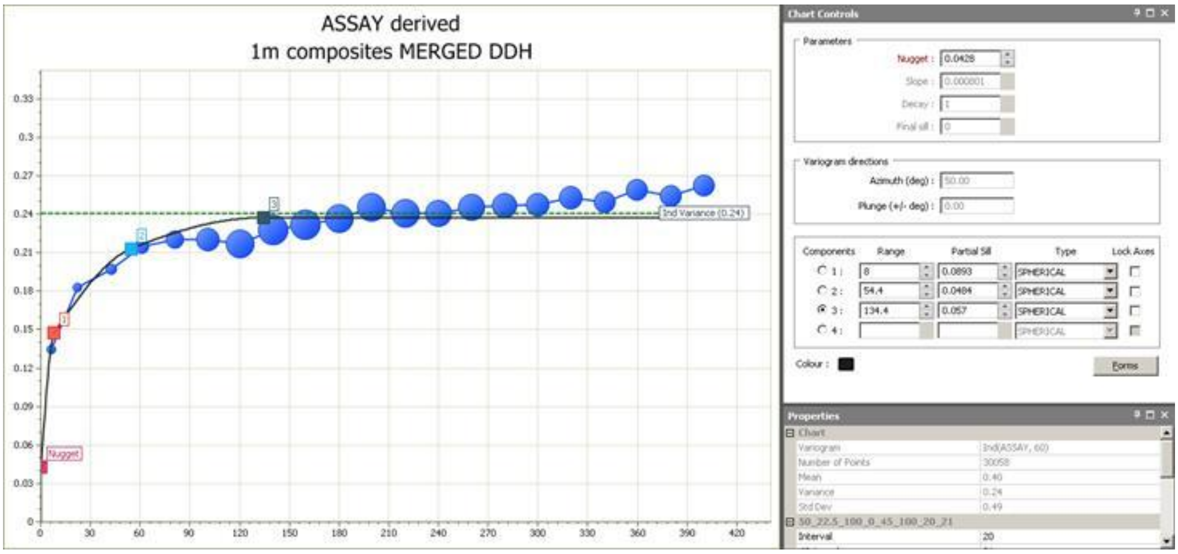


Figure 14.2 Valencia Main assay composite variogram Direction 1

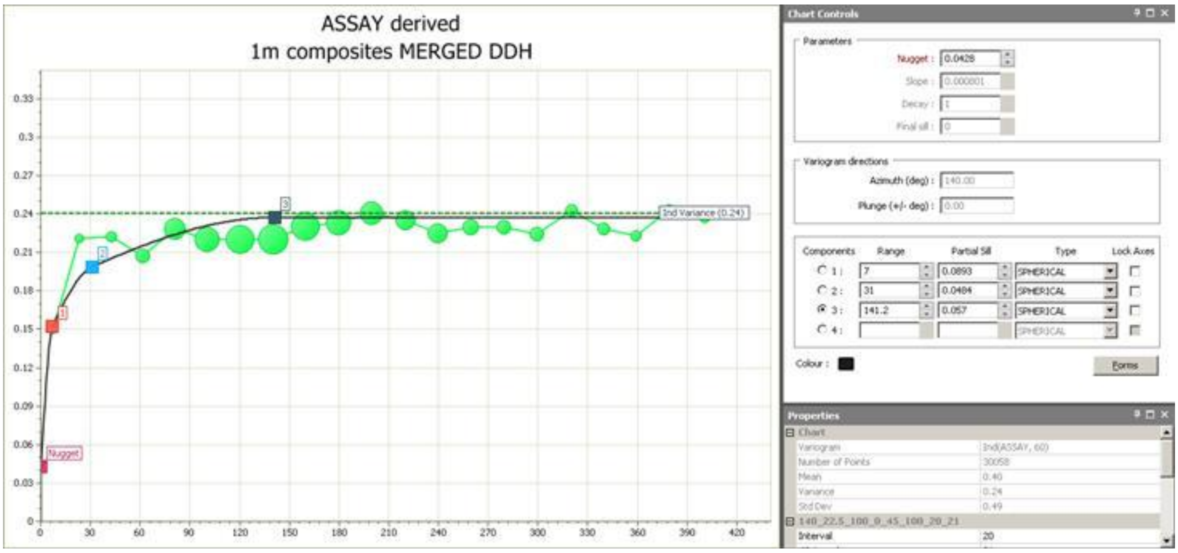


Figure 14.3 Valencia Main assay composite variogram Direction 2

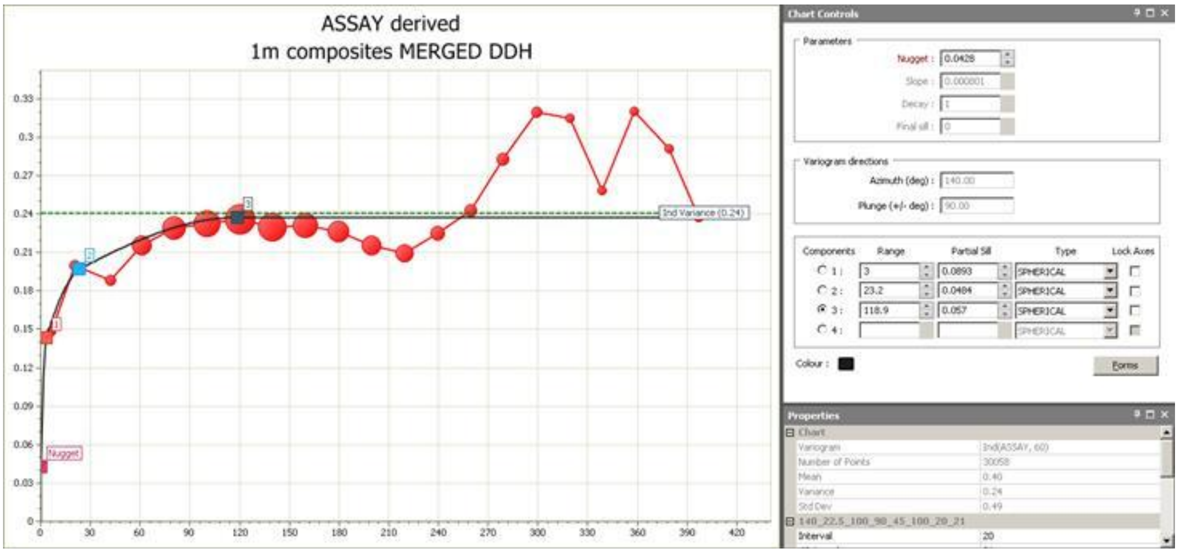


Figure 14.4 Valencia Main assay composite variogram Direction 3

Table 14.4 Variogram parameter eU_3O_8 gamma

	Direction		Nugget	Structure 1		Structure 2		Structure 3	
				Sill	Range (m)	Sill	Range (m)	Sill	Range (m)
D1	30	30	0.19	0.258	8.9	0.112	28.4	0.285	111.5
D2	90	30			5.2		28.8		84.9
D3	210	60			2.8		19.8		109.2

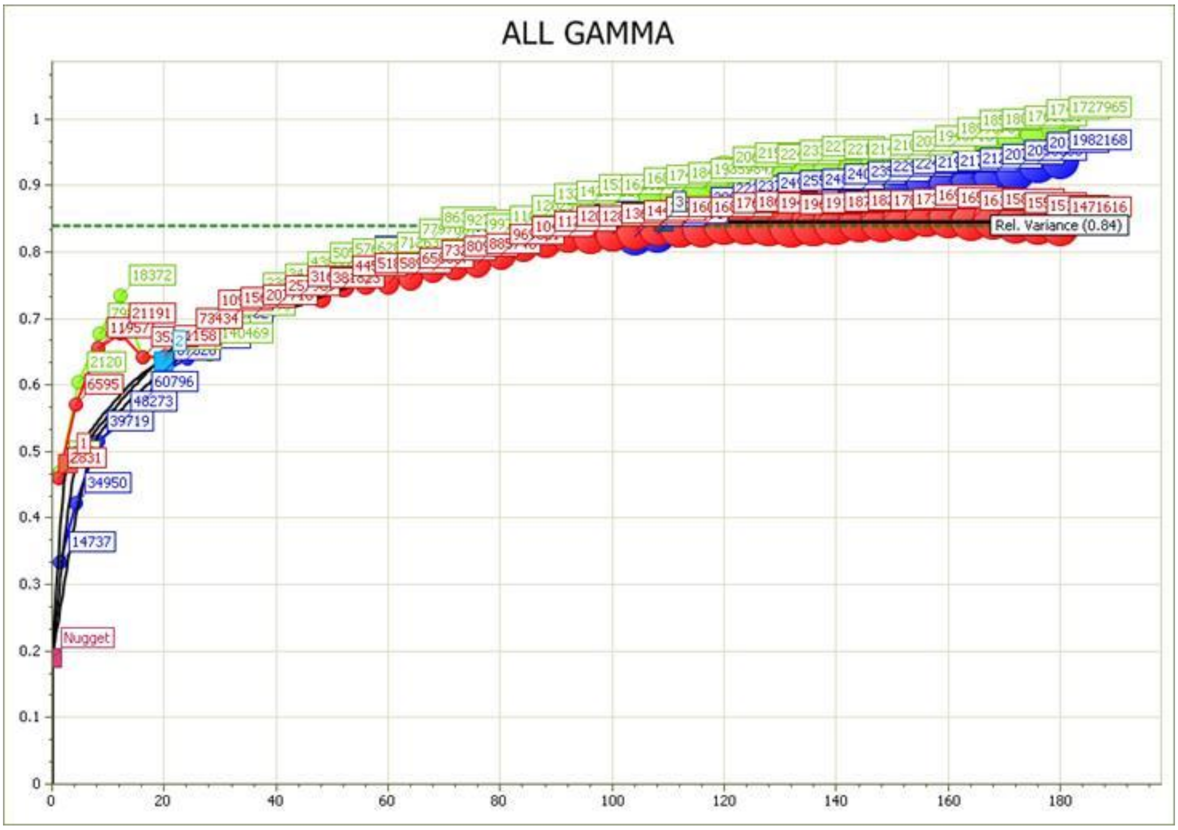


Figure 14.5 Valencia Main gamma derived composite variogram

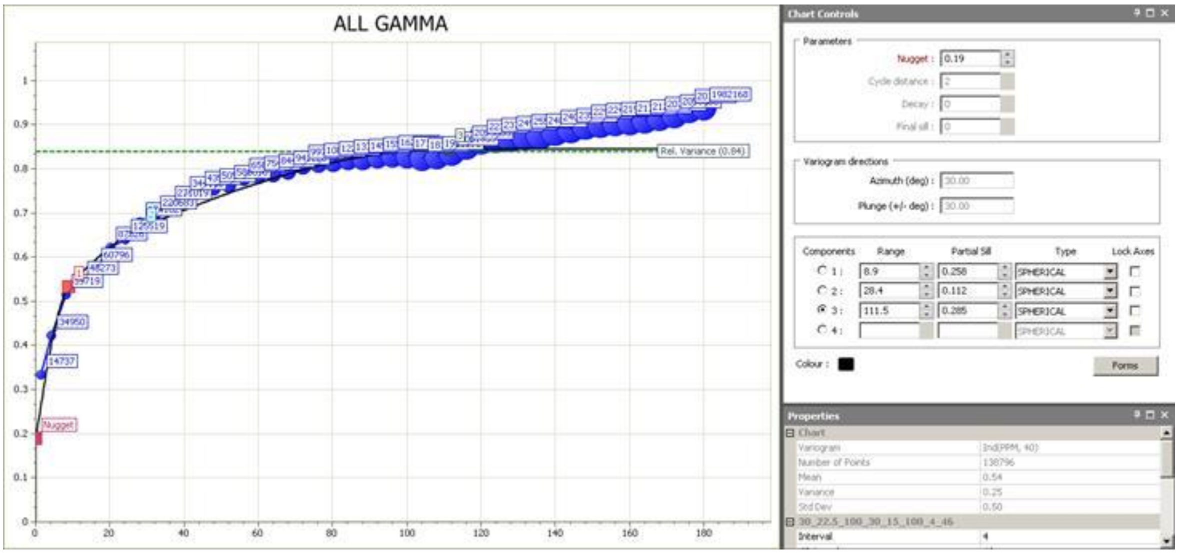


Figure 14.6 Valencia Main gamma derived composite variogram Direction 1

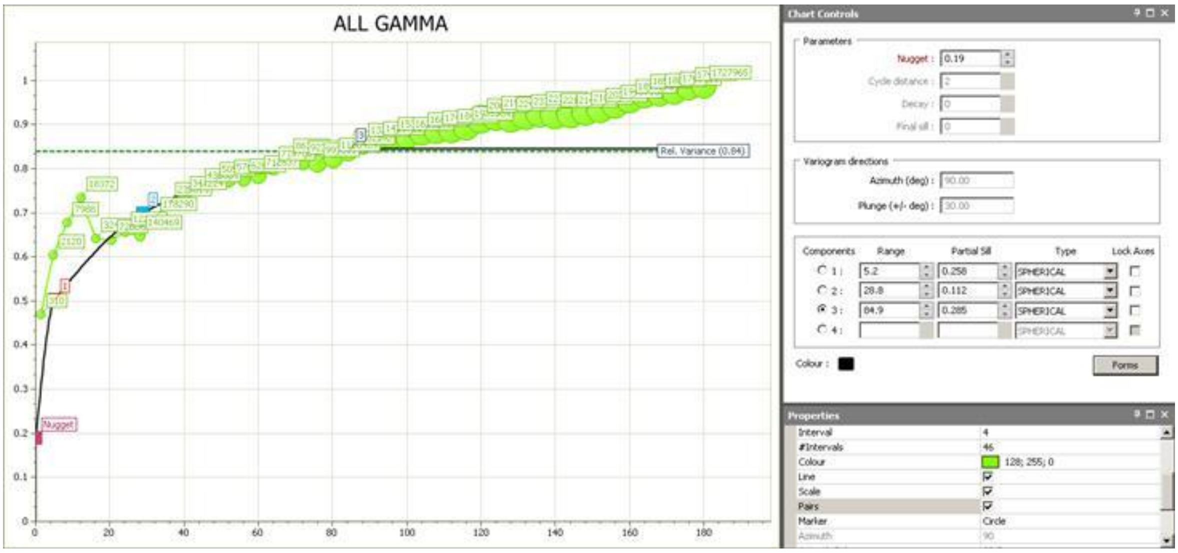


Figure 14.7 Valencia Main gamma derived composite variogram Direction 2

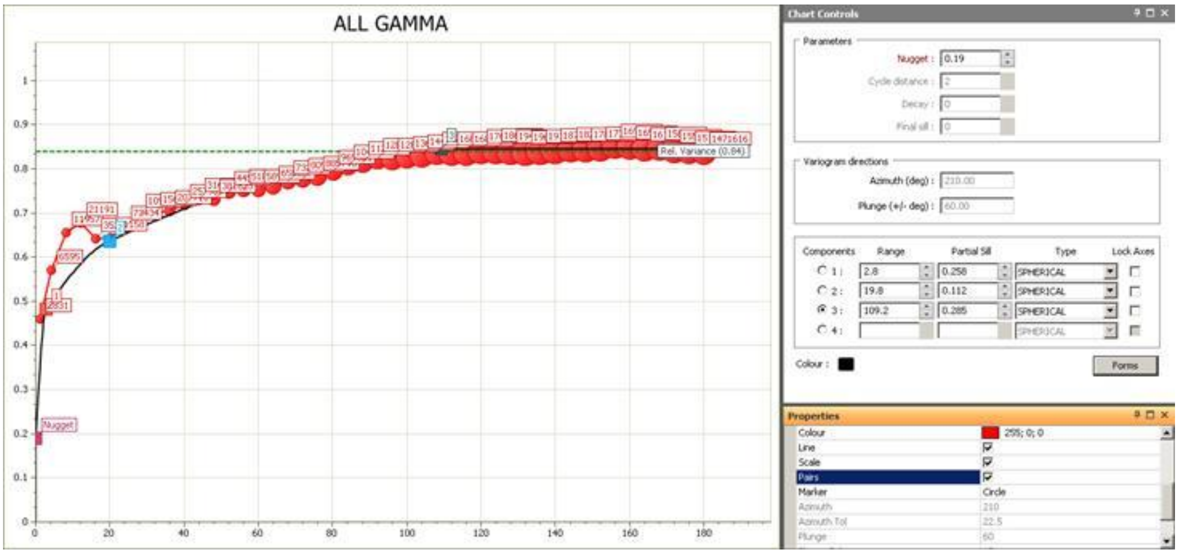


Figure 14.8 Valencia Main gamma derived composite variogram Direction 3

Geologic or alaskite controls at Valencia applied the GRT, AGRT, GRTT composite identifier only and used a 60ppm cut-off. Application of strict search ranges enabled good controls for interpolation into alaskite zones only. Block sizes used reflect SMUs of 10m x 10m x 5m.

Grade interpolation applied ordinary kriging for ASSAY and for GAMMA data in separate evaluation runs which were merged into a GRADE category using relative kriging variance indicator.

Confidence classification and categories were assigned to all blocks using spatial controls considering the number of drillholes within the search and considering their distances to each other. Number of data points furthermore controlled the classification allocation

Confidence level equivalent to Measured, Indicated and Inferred were assigned according to criteria described in Table 14.5 **Error! Reference source not found.** and Table 14.6**Error! Reference source not found.**.

Table 14.5 Search criteria for ASSAY					
Category	**# holes**	**# points**	**Search X**	**Search Y**	**Search Z**
Measured	5	50	60	40	20
Indicated	4	42	60	40	20
Inferred	3	10	120	80	20

Table 14.6 Search criteria for GAMMA					
Category	**# holes**	**# points**	**Search X**	**Search Y**	**Search Z**
Measured	5	50	60	40	20
Indicated	4	42	60	40	20
Inferred	3	10	120	80	40

A 40m perimeter developed in 2014 (Forsys Metals, 2014), which is known as the “D40 domain” (Figure 14.9) was utilised to check for Inferred classed material remaining in areas surrounded by tight drill clusters. Blocks identified inside the pit boundaries were manually flagged for reclassification into Indicated when found within the 40m perimeter of a sampled drillhole.

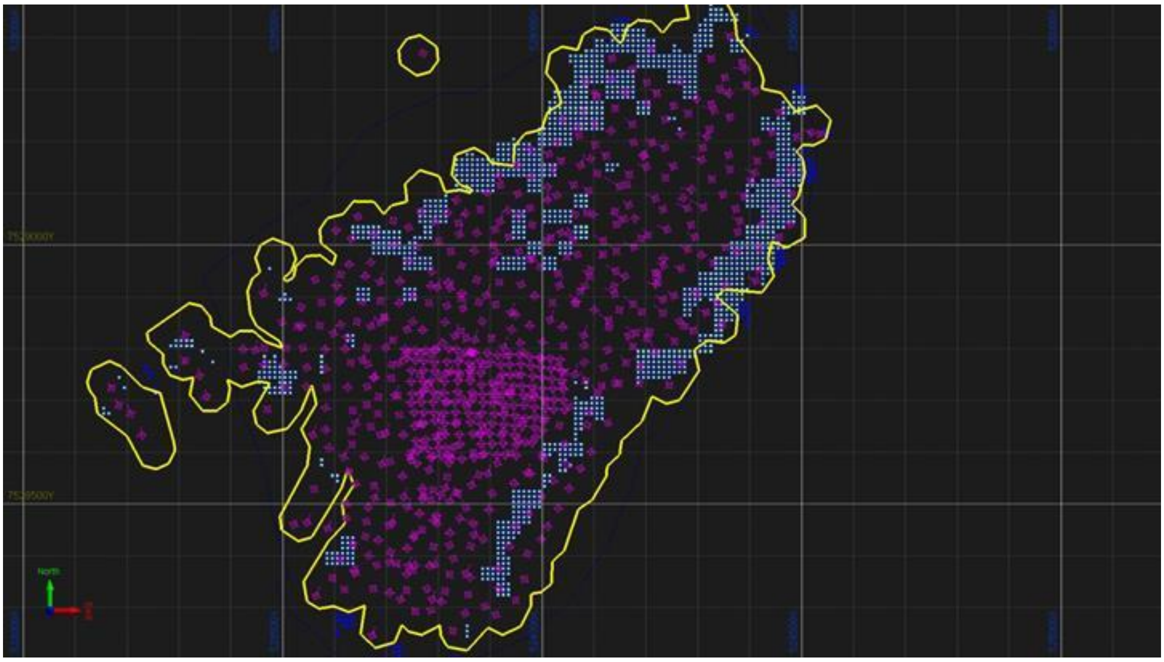


Figure 14.9 Plan view of Valencia deposit at elevation 652.5m.

Categorical model reconciliation followed using logged geology and raw grade data from drill holes; this was done at bench levels and in East / West cross-sections throughout the model in order to visually reconcile estimation and GRT index.

14.1.1 Valencia Satellite

Valencia Satellite is an addition to the main deposit and has been evaluated during 2012 and 2013. A total of 70,270 readings from downhole gamma probing were used to estimate the small high grade orebody.

Gamma raw data was collected by means of down-hole geophysical logging. Gamma probe c/s data was collected at 0.1m intervals by probing upwards at a speed of 6 m/min. Radon effects were monitored and affected holes were re-probed until readings remained consistent.

The correlation coefficient used for conversion from gamma c/s to eU_3O_8ppm grade thickness (GT) over 1m is the same as was used during the Valencia Main PD drilling campaigns (=0.0198) (Laine, R. Dr, 2008).

In contrast to the historical Valencia evaluation a wireframe approach was selected to define the ore envelope(s) which was used to flag resource blocks eligible for the grade interpolation. As such, a block rock type allocation is based on wireframes and drillhole data directly. This approach is considered to be more appropriate than applying simulations for geology.

Variography was undertaken on equivalent uranium data using a lognormal spherical model. Ordinary kriging was then used to interpolate and back transform the uranium grade into the resource model. SMU block size is identical to Valencia Main and is 10m x 10m x 5m. Only data associated with logged alaskite (lithologies: GRT, GRTT and AGRT) was used in the resource estimate.

Visual block reconciliation for rock type was done using sets of host rock wireframes derived from logged drillhole data.

The resource model for the combined Valencia deposit is presented below in Table 14.7.

Table 14.7 Valencia total Mineral Resource (February 2015)

Category	Cut-Off Grades	Tonnes [M]	U_3O_8 [ppm]	U_3O_8 [Mlbs]
Measured				
	60ppm	27	151	9
	100ppm	**16**	**200**	**7**
	140ppm	10	249	6
Indicated				
	60ppm	256	152	86
	100ppm	**157**	**198**	**69**
	140ppm	98	247	53
Measured + Indicated				
	60ppm	283	152	95
	100ppm	**174**	**199**	**76**
	140ppm	108	248	59
Inferred				
	60ppm	20	145	6
	100ppm	**11**	**199**	**5**
	140ppm	7	253	4

14.2 NAMIBPLAAS

Drillhole database QA/QC (see chapter 12.3) re-assigned a total of 1,772 DDH samples of uranium assays and 50,040 radiometric readings of composite downhole information for re-estimating the Namibplaas Mineral Resource. Data is derived from 3 different drilling campaigns, a first diamond drilling phase which was followed by two separate PD drilling phases. Data collected are chemical assays in U_3O_8 and radiometric probing presented as eU_3O_8.

Interpolation made provision for separating chemical assays from the radiometric derived equivalent uranium grades.

U_3O_8 and eU_3O_8were individually modelled with resulting variography used to krige both “sets” into a single model. A final step merged both sets into a derived, or rather selectively merged grade by using relative kriging variances. This process used in earlier years at Rio Tinto’s Rössing Uranium mine, providing excellent reconciliation results of predicted versus mined data.

An earlier estimation by Optiro in 2009 used a combined data set of 2,328 DDH assay samples and 42,124 gamma readings in a single interpolation run.

The methodology of rock type allocation was maintained throughout and remained identical to earlier resource model. In analogy, also only grade from “alaskite” composites is being used.

Generally, the downhole composite length is 1.5 m.

Lithology codes GRTT, GRT and AGRT were coded with LC = 1; with all non GRT intervals being assigned LC code of “0”.

Variography was then undertaken on the numerically coded lithology. LC values of between 0 and 1 were estimated in the block which was considered to reflect the probability of the block being GRT or not, typical LC values range from 0.001 to 1.001. Blocks reflecting a probability of 0.5 and above are than reflected in the final resource.

Ordinary kriging using Micromine mining software was used to estimate U_3O_8 grades into the 3D block model using tight search control. Parent cell estimation was carried out into sub-cells measuring 20m x 20m x 10m and into an SMU of 10m x 10m x 5m. Uranium ASSAY grade and GAMMA grade was assigned from their parent cell into a selectively selected grade based on relative kriging variances.

The lithology GRT probability index was re-assigned at each step into the SMU.

The search used ranges and orientation modelled in the variograms with minimum number of composites used for the interpolation being eight and the maximum number of samples used being 40.

A bulk density of 2.63 was applied, this being the same as was applied to the Valencia estimate.

Table 14.8 Variogram parameter for Grade (ASSAY)

	Direction		Nugget	Structure 1		Structure 2		Structure 3	
				Sill	**Range (m)**	**Sill**	**Range (m)**	**Sill**	**Range (m)**
D1	120	0	141	1954	24.3	2025	51.6	130	118
D2	90	0			24.3		65.9		118
D3	210	90			15.1		67.4		118

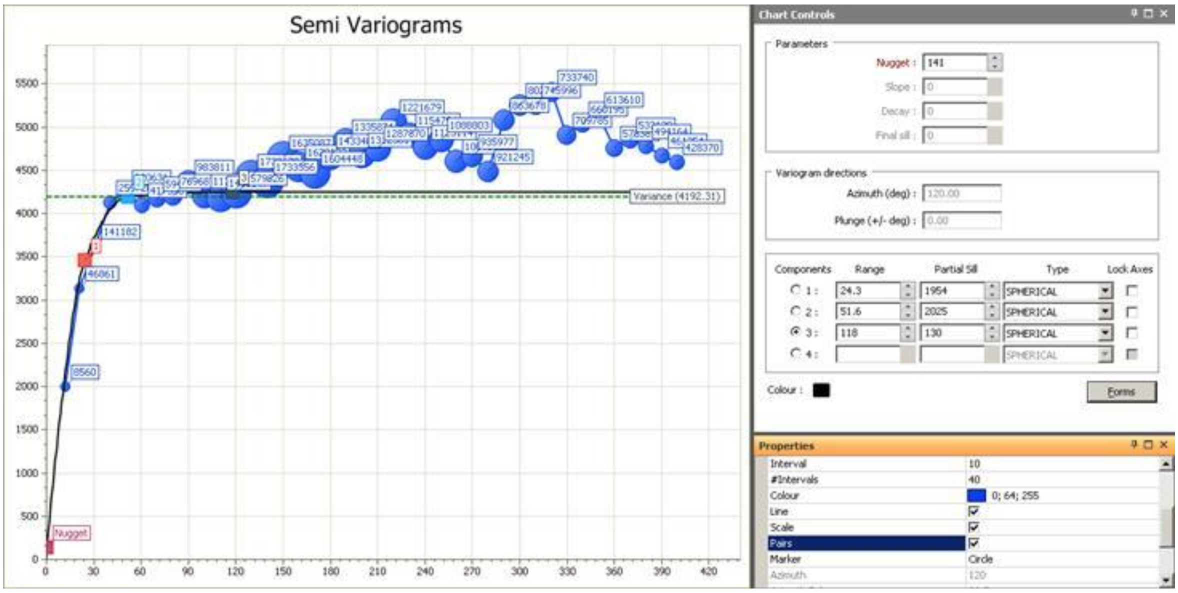


Figure 14.10 Variogram model for assay grade Direction1

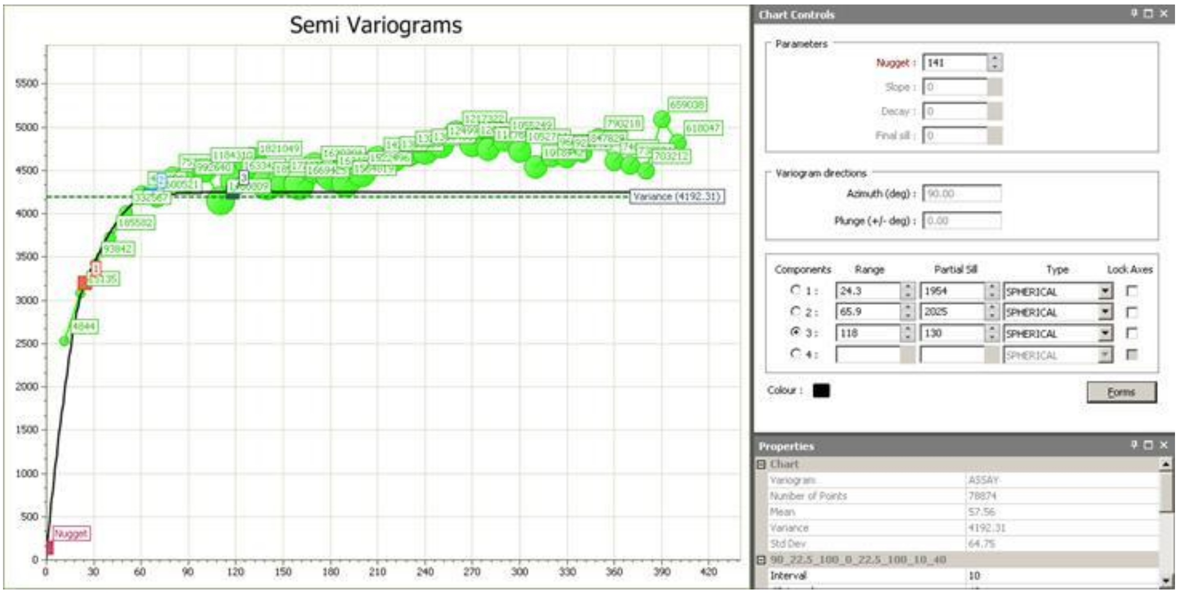


Figure 14.11 Variogram model for assay grade Direction2

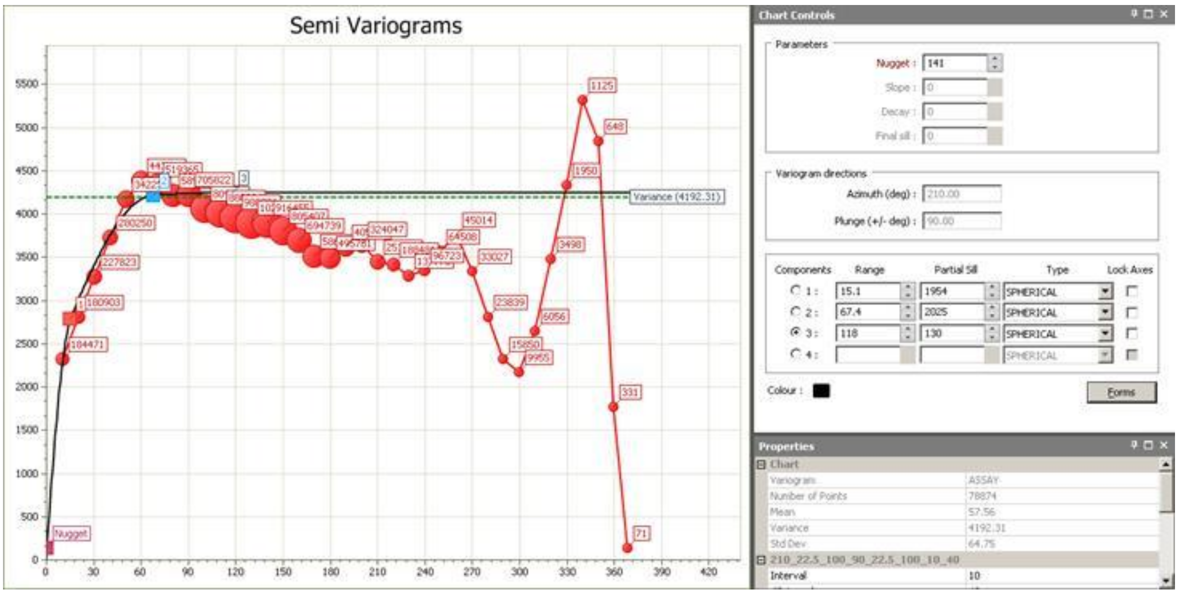


Figure 14.12 Variogram model for assay grade Direction3

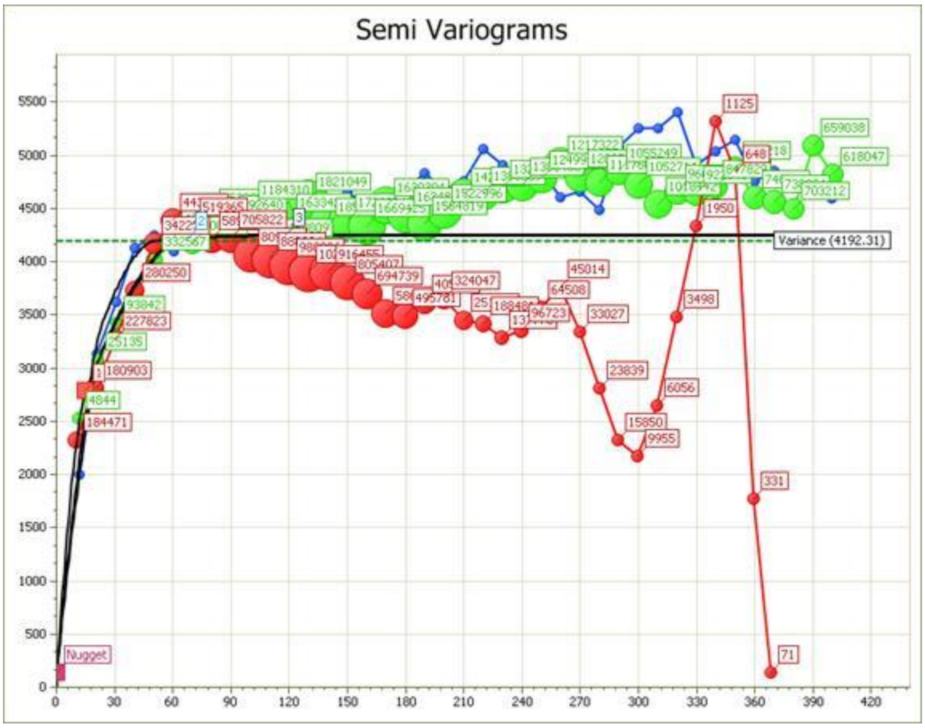


Figure 14.13 Variogram model for assay grade Direction1,2,3.

14.2.1 Evaluation of grade (equivalent uranium) derived by radiometric probing.

Table 14.9 NP Variogram parameter for Grade (GAMMA) log

	Direction		Nugget	Structure 1		Structure 2	
				Sill	Range (m)	Sill	Range (m)
D1	120	0	0.1185	0.0916	40	0.0292	149
D2	90	0			55		181
D3	210	90			50		106

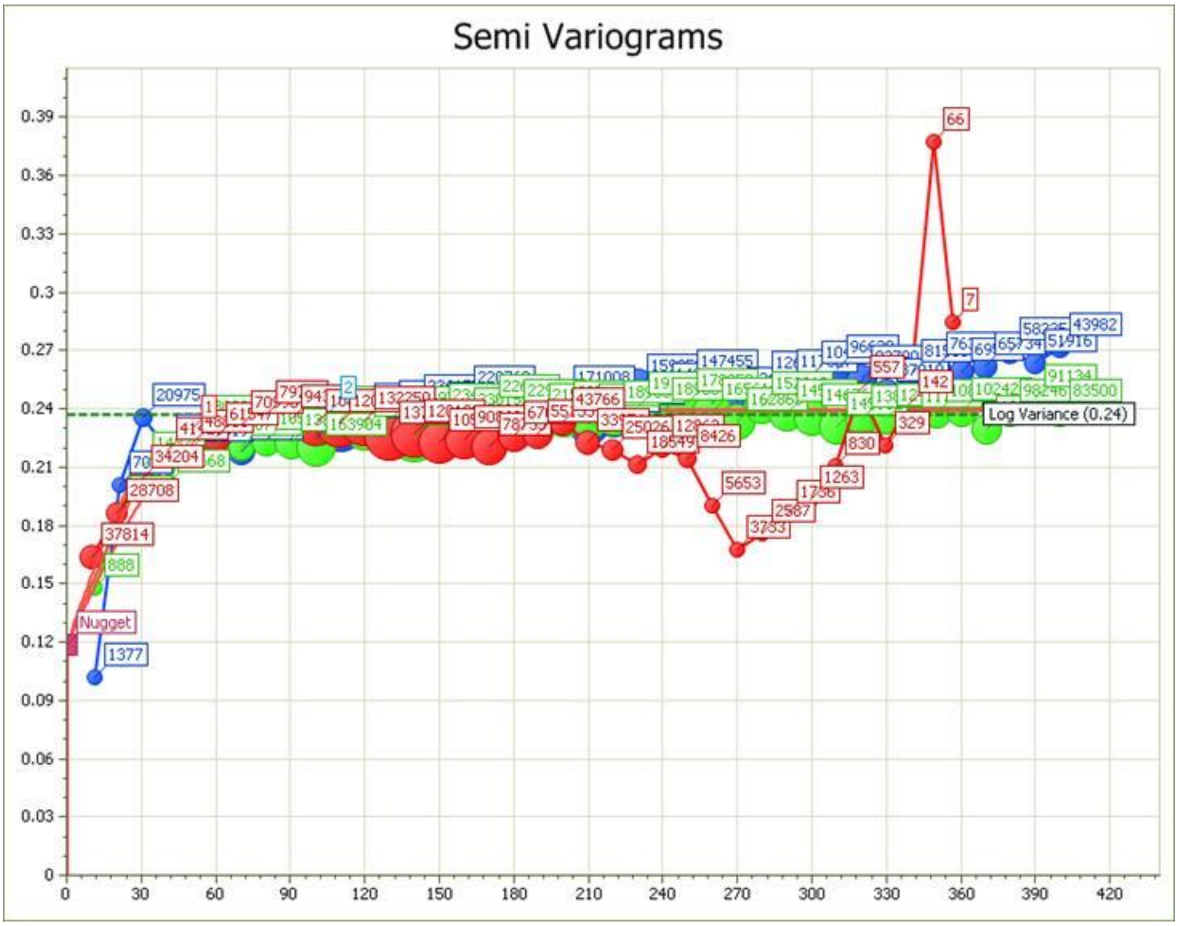


Figure 14.14 NP Variogram model for gamma grade Direction1,2 and 3

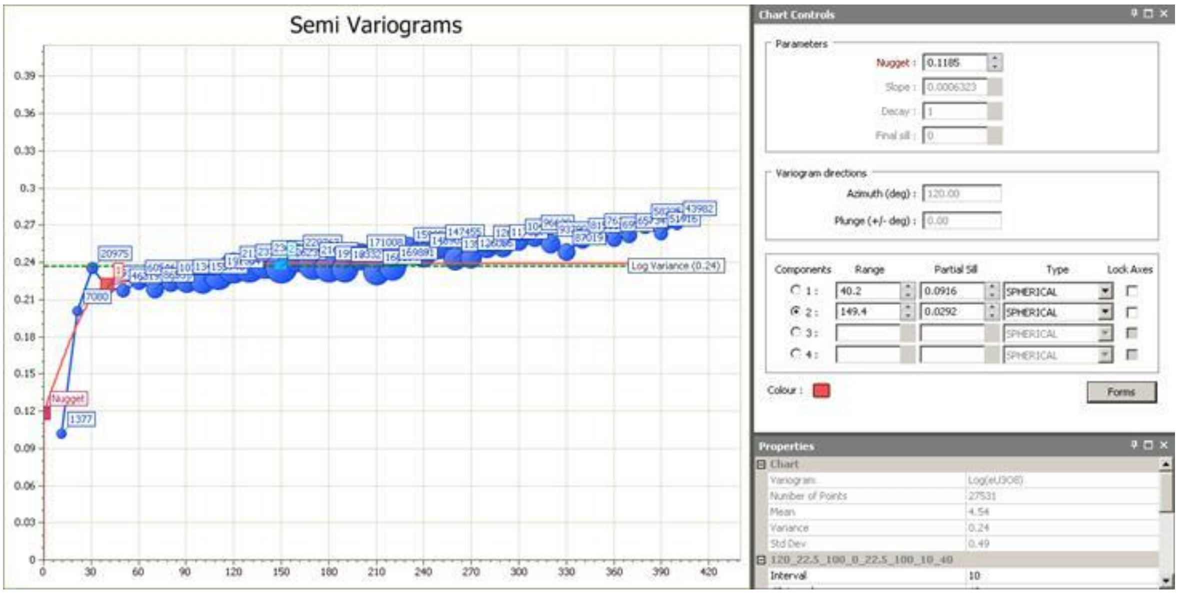


Figure 14.15 NP Variogram model for gamma grade Direction1

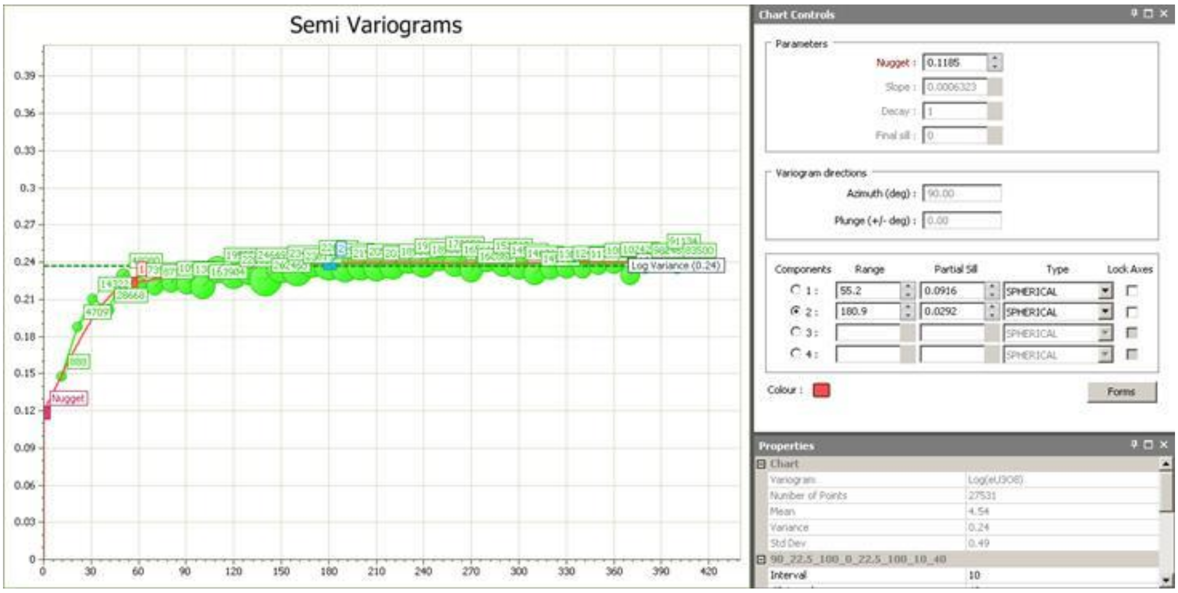


Figure 14.16 NP Variogram model for gamma grade Direction2

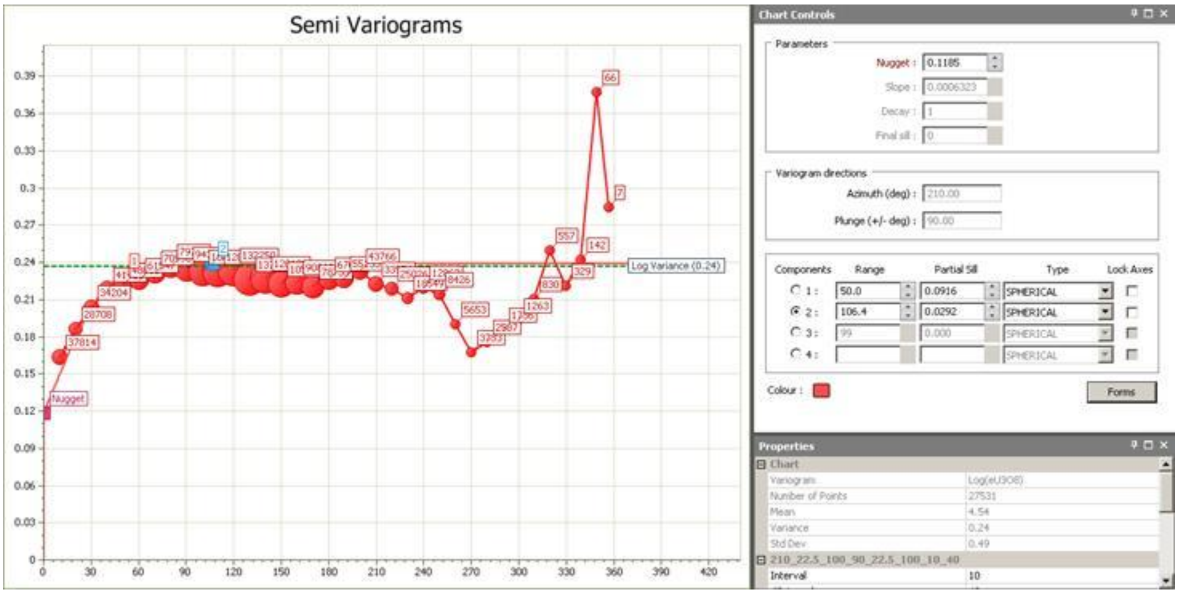


Figure 14.17 NP Variogram model for gamma grade Direction3

Table 14.10 Namibplaas Project Mineral Resource (February 2015)				
Category	**Cut-Off Grades**	**Tonnes [M]**	**U_3O_8 [ppm]**	**U_3O_8 [Mlbs]**
Measured				
	100ppm			
	140ppm			
	180ppm			
Indicated				
	100ppm	213	151	71
	140ppm	**91**	**193**	**39**
	180ppm	32	262	18
Measured + Indicated				
	100ppm	213	151	71
	140ppm	**91**	**193**	**39**
	180ppm	32	262	18
Inferred				
	100ppm	30	159	11
	140ppm	**15**	**201**	**7**
	180ppm	6	267	3

14.3 NORASA

Table 14.11 below is presenting the consolidated Norasa Project resource. Note that Resources are reported inclusive of Reserves.

Table 14.11 Norasa Mineral Resource (February 2015)				
Category	**Cut-Off Grades**	**Tonnes [M]**	**U_3O_8 [ppm]**	**U_3O_8 [Mlbs]**
Measured				
	Val 60ppm: Nam 100ppm	27	151	9
	Val 100ppm: Nam 140ppm	16	200	7
	Val 140ppm: Nam 180ppm	10	249	6
Indicated				
	Val 60ppm: Nam 100ppm	469	152	157
	Val 100ppm: Nam 140ppm	249	196	108
	Val 140ppm: Nam 180ppm	130	251	72
Measured + Indicated				
	Val 60ppm: Nam 100ppm	496	151	166
	Val 100ppm: Nam 140ppm	265	197	115
	Val 140ppm: Nam 180ppm	140	251	77
Inferred				
	Val 60ppm: Nam 100ppm	50	153	17
	Val 100ppm: Nam 140ppm	26	200	11
	Val 140ppm: Nam 180ppm	13	260	7

For the combined resource of both projects, the strategic decision was made to report the individual projects at cut-off grades that would provide a 200ppm average grade. Although Namibplaas still has a considerable amount of material between 100ppm and 140ppm (more than 120Mt for over 30Mlbs), from a strategic perspective this material remains potential upside in the life of mine and economics, and will not considered for in the mine plan, only reported to quantify that additional potential.

15. MINERAL RESERVE ESTIMATES

The Mineral Reserve Estimate was prepared using industry best practise and reported in accordance with the NI 43-101 (OSC 2011) guidelines. The Mineral Reserve is based on pit optimisations using the resource models discussed previously and applying modifying factors such as costs and mining and metallurgical factors determined to be appropriate for the deposits and scale of operation to a feasibility study level of accuracy.

The Mineral Reserve Estimate for Norasa is tabulated in Table 15.1 and has been assigned confidence levels of Proven and Probable Reserve using the guidelines within the NI 43-101. Mineral Resources that are not Minerals Reserves have not demonstrated economic viability, or have not fulfilled the company's strategic criteria of cut-off grade.

The total Proven and Probable Mineral Reserve for Norasa is 206Mt at a grade of 200ppm, which equates to 91Mlbs of U_3O_8.

Table 15.1 Norasa Mineral Reserves Estimate (February 2015)

Classification	Mt	Grade ppm U_3O_8	Mlbs U_3O_8
Proven	16	200	7.1
Probable	190	200	83.6
Total Reserve	**206**	**200**	**90.7**

Cut-off grades of 100 ppm for Valencia and 140 ppm Namibplaas

A breakdown of the Reserves for the individual projects is reported in Table 15.2 and Table 15.3.

Table 15.2 Valencia Reserves Estimate (February 2015)

Classification	Mt	Grade ppm U_3O_8	Mlbs U_3O_8
Proven	16	200	7.1
Probable	139	200	61.3
Total Reserve	**155**	**200**	**68.4**

Cut-off grade of 100 ppm

Table 15.3 Namibplaas Reserves Estimate (February 2015)

Classification	Mt	Grade ppm U_3O_8	Mlbs U_3O_8
Proven	0		0
Probable	51	198	22.3
Total Reserve	**51**	**198**	**22.3**

Cut-off grade of 140 ppm

15.1 PIT OPTIMISATION

The objective of the open pit optimisation process is to determine a generalised open pit shape outline or shell that provides the highest value for a deposit. Pit optimisations were carried out using the Whittle Four-X ("Whittle") pit optimisation software. For a given block model, cost, recovery and slope data, the Whittle software calculates a series of incremental pit shells within which each shell is an optimum for a slightly higher commodity price factor. The Lerchs-Grossman algorithm determines the optimal pit shape for a given set of economic and slope criteria.

The algorithm progressively constructs lists of related blocks that should, or should not, be mined. The final pit shell list defines a pit outline that has the highest possible total value, subject to the required pit slope angles. This outline includes every block that adds value when waste stripping is taken into account and excludes every block that does not add value. The process takes into account all revenues and costs and includes mining and processing parameters. The resulting pit shells are not necessarily practical and do not incorporate ramps, catchment berms etc. It is from analysis of all the nested shells generated in the optimisation process that a single shell will be selected as the guide for a practical ultimate pit design. The Whittle pit shell results are used to assess the sensitivity of the project to changes in the input parameters and also to guide the practical pit design process.

The final pit design defines the Mineral Reserve and subsequently the life-of-mine (LOM) production schedule and associated cashflows can be determined. Hence, the pit optimisation process is the first step in the development of any LOM plan. In addition to defining the ultimate size of the open pit, the pit optimisation process also provides an indication of possible mining push-backs. These intermediate mining stages allow the pit to be developed in a practical and incremental manner, while at the same time targeting high grade ore, and deferring waste stripping.

For the Norasa Project feasibility study pit optimisation process, the most up-to-date information from the AMEC processing study was used to supplement and improve on previous optimisation studies. The input parameters include but are not limited to:

- the NI 43-101 compliant Resource Model;
- modifying factors including mining ore loss and mining dilution for any given block;
- mining operating costs and mining production parameters;
- process recovery, processing costs, mining and processing production rates inclusive of respective ramp-ups;
- geotechnical (pit slope design) parameters;
- metal price and mineral royalties; and
- discount rate.

15.1.1 Loss and dilution

Consideration of the mining method is an essential component of mineral reserves evaluation. This is particularly true when the profitability of a project is conditioned by the ability to mine selectively. Ore loss and dilution assumptions associated with open pit mining were modelled through regularisation. Regularisation involves identifying a smallest mining unit (SMU) block size to mimic the mining selectivity associated with grade control, blasting, excavation, and haulage practices. In the case of the Norasa Uranium Project a range of SMU sizes were modelled in previous studies to test the sensitivity of the resource to the SMU sizes, in particular the bench height.

An analysis was conducted by Omega Geo-Consulting within the RC drilling area (Measured Resource area) of the Valencia main orebody comparing the impact of modelling the orebody to different block sizes. The intention was to define a block size into which the estimated modelled grade would represent a suitably diluted model such that mining studies would no longer have to apply arbitrarily defined losses and dilution. The modelling approach makes use of the material adjacent to the modelled blocks as the diluting material, which is a significant consideration since there is generally no hard boundary between ore and waste. The diluting material is mainly lower grade granite and not a pure, barren waste. From this study, it was recommended that the most suitable SMU for the Valencia and Namibplaas deposit has the dimensions of 10 m (E-W direction) x 10 m (N-S direction) x 5 m.

The SMU is then applied to the geological model weight averaging the uranium grades of all of the modelled resource within an SMU. For the purposes of the study the waste material adjacent to the mineralised zones was assumed to contain no uranium and assigned a U_3O_8 grade of 0 ppm. This new regularised block model is then reported and compared with the undiluted resource to identify where dilution and ore loss occur and the results are summarised in Table 15.4.

Table 15.4 Average dilution and ore losses

Classification	Dilution (%)	Ore Loss (%)
Valencia Main & Satellite Deposit	5.3	5.9
Namibplaas Deposit	8.6	8.2

Note: Reported at a 100ppm cut-off grade

Dilution and ore loss will occur at the interface of mineralised zones and the surrounding waste. The tenor of the grade in the mineralised zones is such that at the cut-off grades under consideration there is minimal internal dilution. In order to maintain the degree of selectivity represented within the model, it is acknowledged that strict control measures will be required. The tight drill pattern (4.7m x 5.3m), efficient blasting practices (for which a blasting company will be contracted), suitable grade control practices for bench mining plans and oversight of loading activities in ore boundary areas, will all help to obtain the modelled ore grade.

Ore loss is defined as ore 'lost' to waste due to operational issues like ore material dispatched erroneously to the waste dumps. Ore losses will be minimised as it is envisaged that the project is operated to world class best practise standards, including:

- A combination of in-advance reverse circulation (RC) grade control drilling and continuous blast hole sampling;
- Computer-aided orebody modelling; and
- An integrated mine wide equipment dispatch system.

15.1.2 Geotechnical input parameters

The pit slope architecture recommended for this study is summarised in Section 16.1 and is based on the recommended design parameters set out by Snowden 2008 and Xstract (2013, 2014), who were appointed to complete a detailed geotechnical analysis of the Norasa Project deposits.

15.1.3 Mining costs

Mining costs applied for the pit optimisation are based on the mining cost model (developed by Snowden 2013), which was compiled from mining suppliers and original equipment manufacturers (OEMs). Costs were developed from first principles and incorporated equipment and personnel costs for all aspects of the mining operation. The costs were generated by pit stage, bench level and material type. From the mine cost model, the weighted average mining cost for material type, by pit location and by bench, was generated. These costs were then approximated by a linear regression relationship by bench level and incorporated into a script that allowed for a mining cost to be calculated for every block in the mining model.

During the pit optimisation runs all blocks, waste and potential ore material, will be mined to the surface of the deposit. Whittle then performs a series of cut-off grade calculations in order to determine whether a block of material should be processed (ore material) or dumped to the waste rock dump. These incremental ore material costs include:

- grade control costs;
- fixed administrative and overhead costs;
- stockpile re-handle costs; and
- the cost differential of a longer or shorter haul.

This differential may be positive or negative depending on the location of the different destinations. The incremental ore mining cost in Table 15.5 are based on a 2km overhaul for the Valencia satellite ore and a 7km overhaul for Namibplaas ore to the Valencia primary crusher tip.

Table 15.5 Average mining costs applied for the pit optimisation exercise

Cost element	Unit	Valencia Main	Valencia Satellite	Namibplaas
Mining cost – reference level	US$/t	1.51	1.40	1.43
Depth factor	US$/t/m	0.0032	0.0032	0.0032
Incremental ore mining cost	US$/t	0.00	0.14	0.65

15.1.4 Processing cost

The processing costs applied for the pit optimisation were based on the initial work that AMEC conducted on the operating costs for the feasibility study. The cost figures were not yet final, but considered sufficient for the pit optimisations.

During the optimisation and scheduling study, no feed grade or acid consumption constraints to the plant was taken into consideration. During the Whittle shell generation the fixed supervision, general and administrative (SG&A) costs for the overall project of US$ 4.66M was added to the processing costs per pit as summarised in Table 15.6.

Table 15.6 Processing unit costs

Cost	Unit	Valencia Main	Valencia Satellite	Namibplaas
Processing cost excl. rad sorting	US$/t ROM ore	**7.20**	**7.20**	**7.20**
Ore mining overhaul distance	km	0	2	7
Incremental ore mining cost	US$/t ROM ore	0.00	0.14	0.65
Tailings dump operation	US$/t ROM ore	0.25	0.25	0.25
Crusher rehandle	US$/t ROM ore	0.10	0.10	0.10
Services	US$/t ROM ore	0.20	0.20	0.20
Rehab & closure cost	US$/t ROM ore	0.06	0.06	0.06
Whittle processing cost (excl. fixed costs)	**US$/t ROM ore**	**7.81**	**7.95**	**8.46**

The base processing unit cost (based on a weighted average 200ppm head grade to the processing facility over the LoM) was further refined by AMEC from the latest metallurgical test work as outlined in Figure 15.1 where the processing costs varies according plant feed grade.

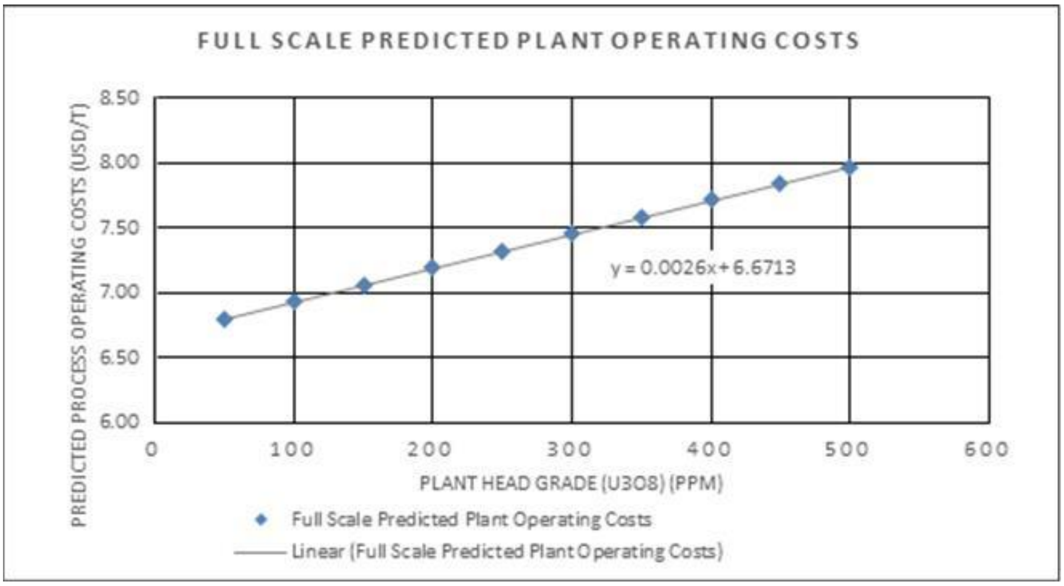


Figure 15.1: Full scale predicted plant operating unit costs.

15.1.5 Processing recoveries and throughput rate

No mill throughput ramp-up was taken into consideration during the optimisation study and the current plant design capacity was set at 11.2 Mtpa excluding radiometric sorting.

Furthermore, there was no recovery ramp-up profile assumed up to steady state. The overall process recovery for Valencia Main, Valencia Satellite and Namibplaas material ore material was updated by AMEC following further metallurgical test work and was assumed to follow the recovery head grade curve depicted in Figure 15.2.

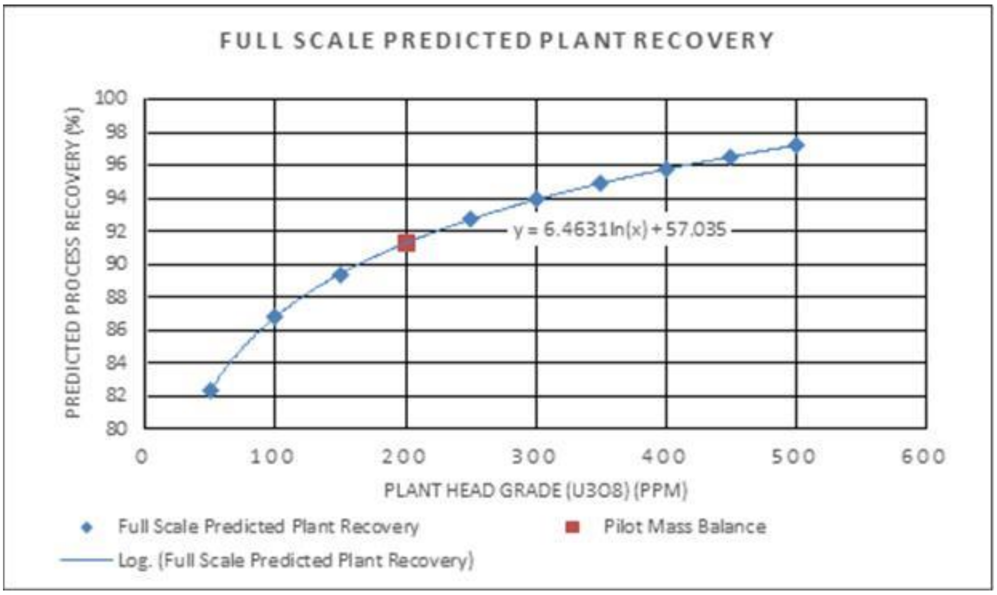


Figure 15.2: Full scale predicted plant recovery.

15.1.6 Uranium price, selling cost and royalties

A constant long term base price of US$65.00/lb of recovered U_3O_8 was assumed for the project. A selling cost of US$0.50/lb was applied which accounts for off-site costs of getting the product to market and include transport, port charges, shipping and insurance.

Royalties are assumed payable at a rate of 3% of net sales (after selling costs have been deducted) as implemented by the Namibian Government based on the market value of uranium sales (Section 22.2).

15.1.7 Cut-off grade calculation

The cut-off grade (COG) is calculated on a breakeven basis and the approach assumes the cost of mining material out of the pit to the waste dump is a sunk cost as it is intrinsic to the mining process, regardless of whether the material is ore or waste. The assessment of whether material is ore or waste occurs once it has been removed from the pit. Similarly capital is a once-off cost that is not applicable to the instantaneous evaluation of a tonne of material to determine its classification.

The economical COG determines whether a tonne of material is ore on the basis that the revenue generated has to be greater than the additional cost of that tonne being processed through the plant. The economical COG is therefore the grade at which the income from the sale of product is equal to or more than the cost of processing.

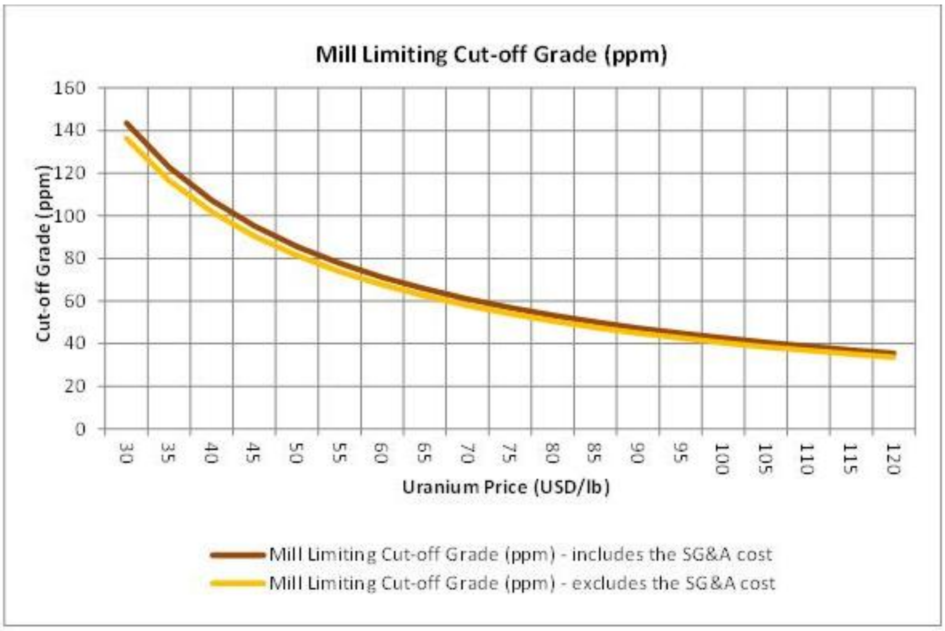


Figure 15.3 Mill limiting cut-off grade (excl. fixed cost portion).

15.2 PIT DESIGNS

The objective of the pit design process was to transform the pit shells obtained from the optimisation into a practical pit, with the inclusion of ramps, bench and berm configurations by taking all the required inputs into account. The practical pit design forms part of a critical input for the scheduling and reserving processes. The Whittle pit optimisation outputs, the design criteria, geotechnical parameters/ constraints and the equipment strategy as well as current world best practice were used as input parameters in order to design the practical final pit. The various pushbacks were based on the interim selected Whittle shells, with full designs including ramp systems. The designs were developed using Geovia's Surpac™ mining software.

Two important considerations for the pit designs were the pushback strategy and the positioning of the access ramps. The optimisation exercise has indicated that there is great value to be generated by this project through an optimum extraction sequence. The starting point of an optimum scheduling sequence is an informed decision regarding pushbacks.

Several pit shells with approximately equal tonnes are usually arbitrarily selected as intermediate phases from the nested pits produced by varying the revenue factor. However, the selection methodology applied during this study identified nested pit shells that are more efficient in terms of stripping ratio and are based on its net value (net revenue less net mining, transport, processing and other costs). The correct selection of a first phase with early access to high grade material while still maintaining a low stripping ratio has the most impact on the project's net present value (NPV) when this mechanism is applied. Subsequent phases may be mined consecutively or concurrently subject to the specific mining rules. The selected interim and ultimate pit shells were used as the basis for the practical pit and pushback designs.

15.2.1 Pit ramps

Sufficient room for manoeuvring needs to be allowed for at all times to promote safety and maintain continuity in the haulage cycle. The width criterion for a haul segment is based on the widest vehicle in use, which was envisioned to be a 180t dump truck with a physical truck maximum operating width of 7.7m.

The haul road design parameters were established taking into consideration the type and size of material hauling equipment that will be used during the operation.

The dimensions of the haul road were based on a 180t haul truck using global standards of good practice. Many of the guidelines specify that the vehicle operating width should be multiplied by a factor of 3.5 for two-lane traffic and 2 for single-lane traffic in order to determine the effective operating width of the haul road and to incorporate the road infrastructure, for example, the safety berm and drainage channel.

15.2.1.1 Haul road gradient

A reduction in haul road grade significantly increases a vehicle's attainable uphill speed. Thus, haulage cycle times, fuel consumption, and stress on mechanical components, which results in increased maintenance costs, can be minimized to some extent by limiting the severity in haul road grades.

A haul road gradient of 1:10 (10% or 5.71º) was selected for Norasa. The selection of the haul road gradient was based on the world best practice for the type of trucks that will be utilized.

15.2.1.2 Haul road width

The equipment study conducted concluded that the 180t dump truck will be used to haul broken rock out of the pit and therefore the road dimensions were based on this type of trucks, taking into considerations global standards of good practice. Designing for anything less than this dimension could create a safety hazard due to a lack of proper clearance. In addition, narrow lanes often create an uncomfortable and unsafe operating environment, resulting in slower traffic and therefore impeding production.

Rules of thumb for determining haulage road lane dimensions vary considerably from one reference source to another. For the purpose of this study, the effective operating width of the haul road for a single-lane was calculated by multiplying the physical truck-operating width by a factor of 2 and a factor of 3.5 for a double-lane haul road.

The haul road width for double-lane haul road was calculated as 30m to cater for effective operating width, safety berm and drainage channel.

15.2.1.3 Ramp position

Ramp positioning within the overall pit design is an integral component of mine design because it influences the stripping ratio on the overall design, the performance of the equipment as well as the operating costs due to direct impact of the ramps on the hauling profiles. The exit positions of the ramps were determined based on the proposed positions of the primary crusher and the waste dump.

15.2.2 Valencia pit

Valencia main pit will have three pushbacks or phases illustrated in Figure 15.4 and the final pit layout in Figure 15.5 *Figure **15.5***. The accesses are designed in such a way that surface hauling distances to the dump, stockpiles and crusher are minimised as much as possible The pit will have dual access established along the final limits to z=690m, and from this point a single ramp will be utilized for both ore and waste hauling going down to pit bottom z=330m. This was done to limit additional waste being mined. Pushback 1 and 2 are also fully designed in accordance with Whittle shells with ramps positioned to minimise ore and waste hauling distances.

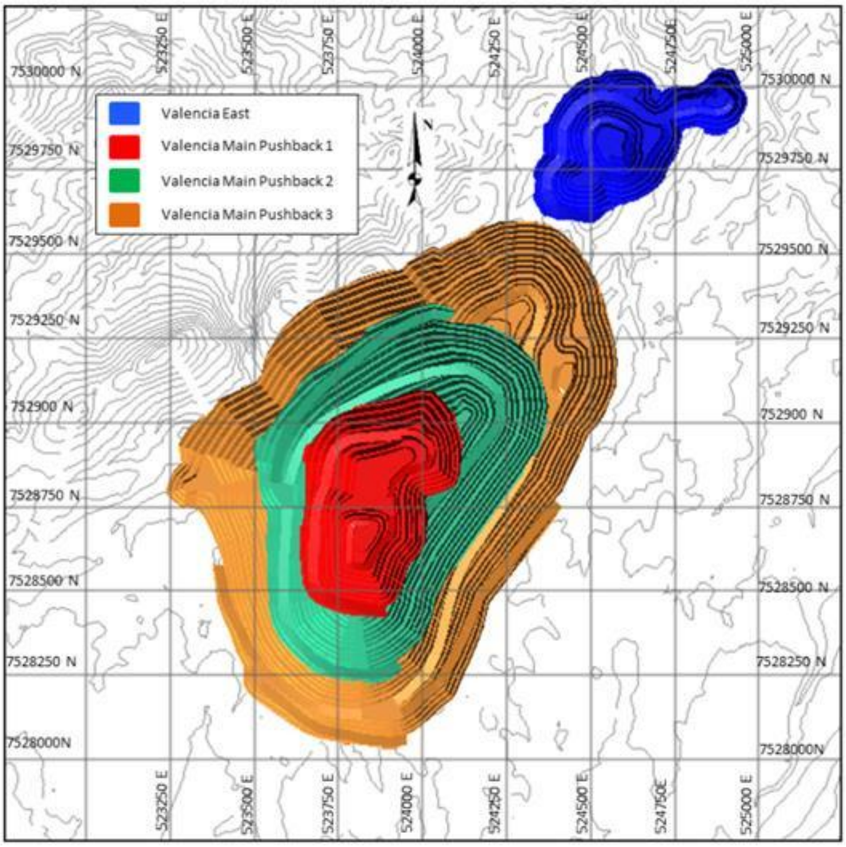


Figure 15.4 Valencia pits with pushback configuration.

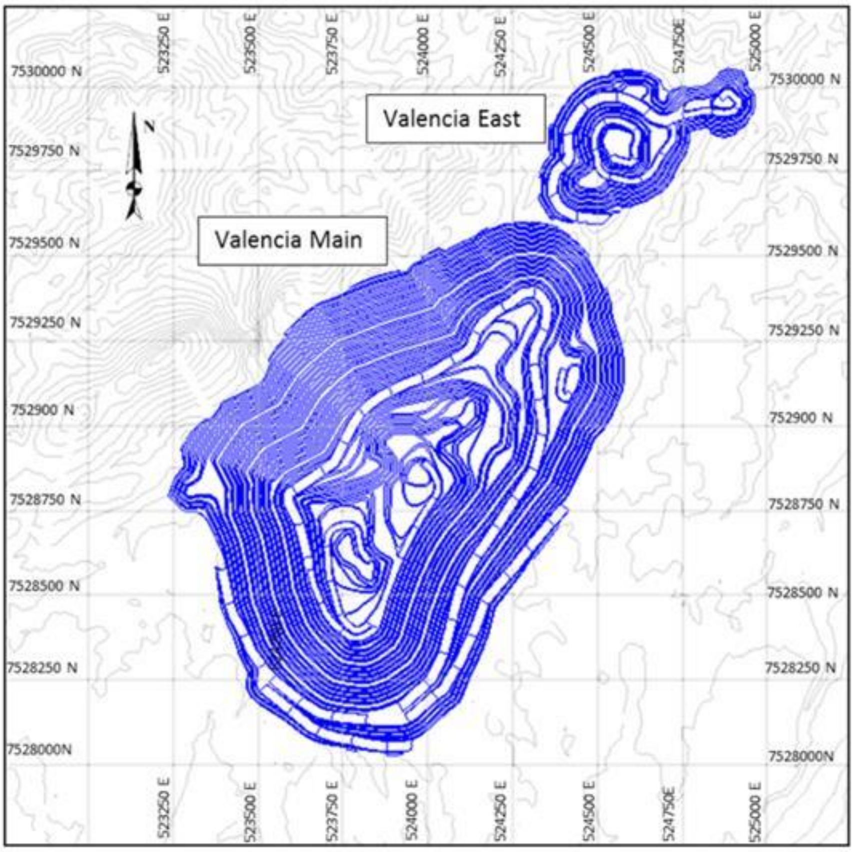


Figure 15.5 Final pit layouts for Valencia pits.

Table 15.7 Valencia main pits dimension				
Description	**Length (m)**	**Width (m)**	**Max depth (m)**	**Area (ha)**
Main	1,660	980	420	131.6

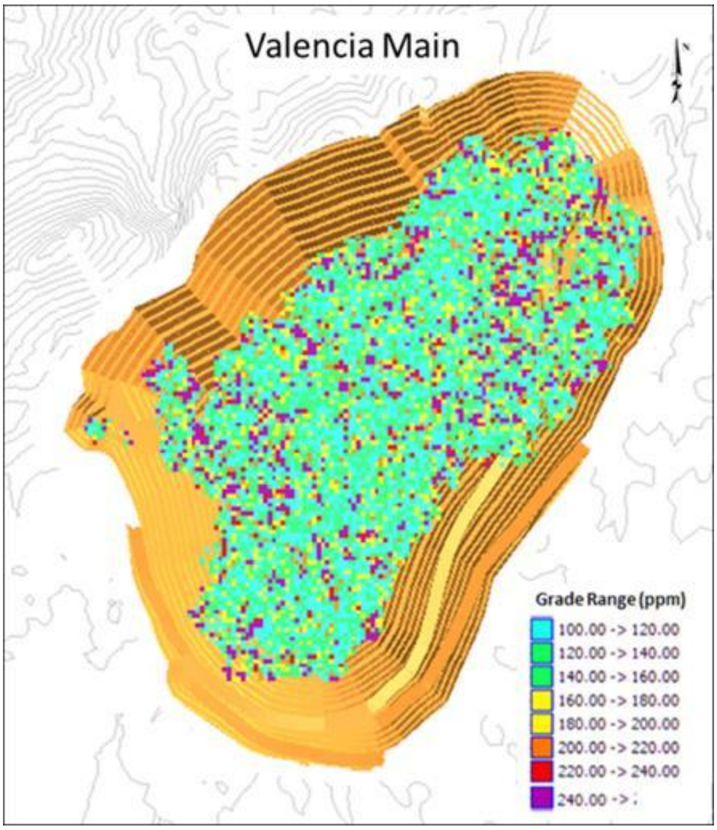


Figure 15.6 Plan view of the orebody within Valencia pit designs.

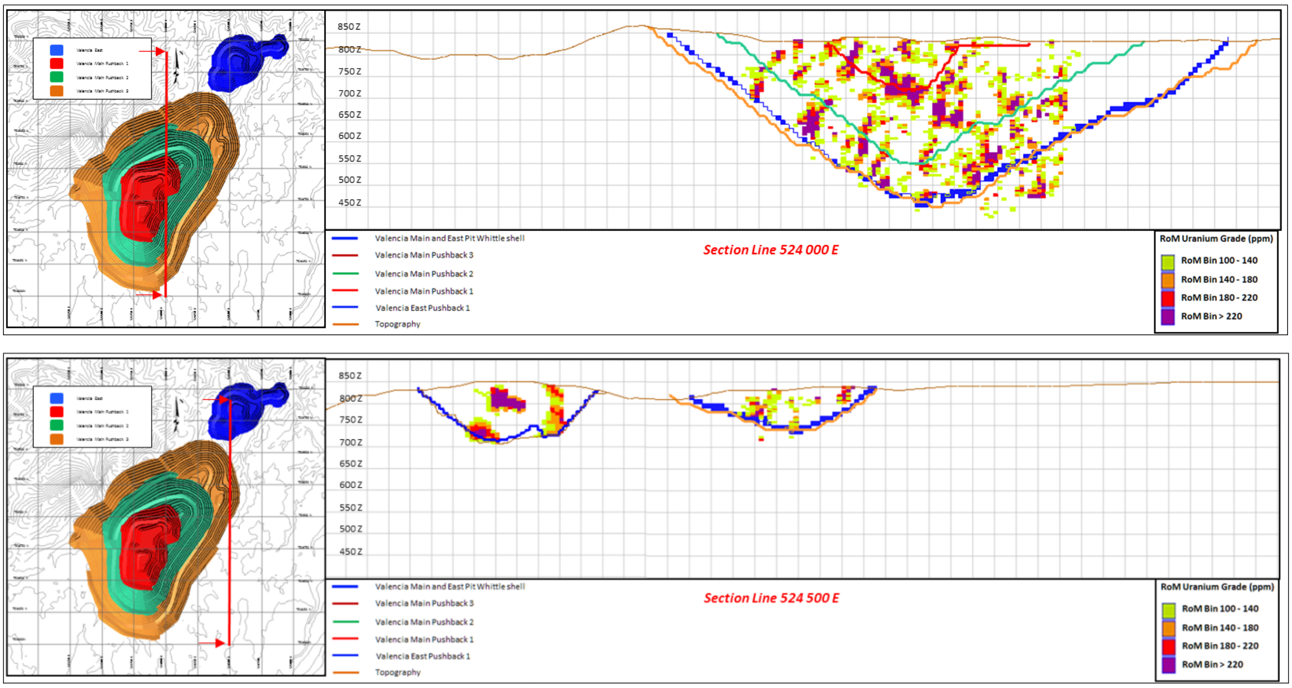


Figure 15.7: Valencia ultimate pit, pushback and optimal Whittle shell

Table 15.7 Total pit inventory of the Valencia orebody					
Parameter	**Unit**	**Pushback 1**	**Pushback 2**	**Pushback 3**	**Final design**
Total ore	Mt	19.4	53.9	75.7	148.9
Total waste	Mt	27.2	121.5	298.9	438.6
Total material mined	Mt	46.6	175.3	365.6	587.4
Stripping ratio		1.4	2.3	3.8	3.0
Average head grade	ppm	205.3	199.3	197.2	199.0
Metal U_3O_8 contained	Mlbs	8.77	23.66	32.89	65.3

15.2.2.1 Valencia satellite

The Valencia satellite pit consists of a small single pushback north-east of the main Valencia pit (Table 15.8**Error! Reference source not found.**). A single ramp system will be developed going down to pit bottom. The ramp is designed to ensure close proximity exit position to the crusher and the dump.

Table 15.8 Valencia satellite pit design dimensions				
Description	**Length (m)**	**Width (m)**	**Max depth (m)**	**Area (ha)**
Valencia satellite pit	712	389	154	22.7

Table 15.9 Total pit inventory of the Valencia satellite orebody		
Parameter	**Unit**	**Final design**
Total ore	Mt	6.2
Total waste	Mt	23.5
Total material mined	Mt	29.7
Stripping ratio		3.8
Average head grade	ppm	229
Metal U_3O_8 contained	Mlbs	3.1

15.2.3 Namibplaas pit

Namibplaas will have three pushbacks or phases illustrated in Figure 15.8*Figure **15.8***, also showing the final pit design. The pit will have a single ramp system going down to form part of a network of ramps leading down to pit bottom in different areas.

The reasoning behind the access strategy is to limit additional waste mining. All three pushbacks form part of the final design boundary and depth.

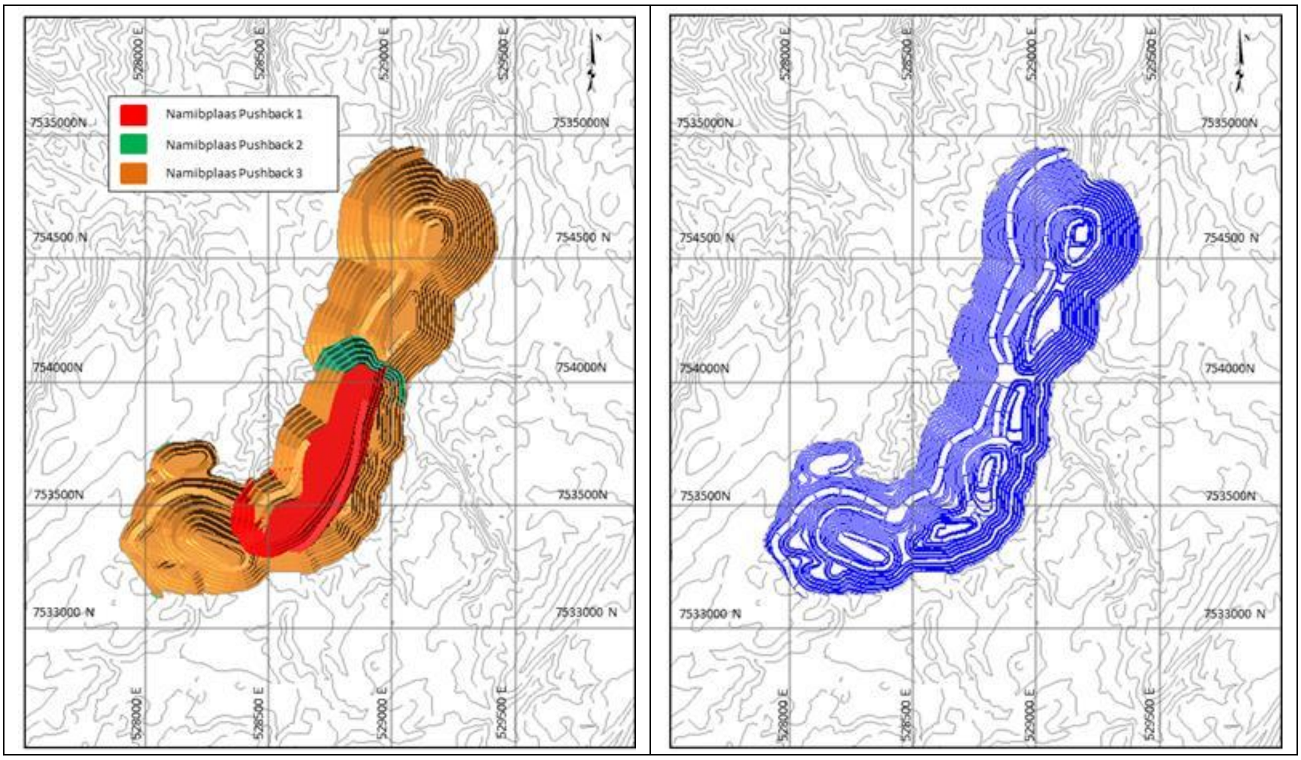


Figure 15.8 Namibplaas pushback configuration and final layout design.

Table 15.10 Namibplaas pit dimensions				
Description	**Length (m)**	**Width (m)**	**Max depth (m)**	**Area (ha)**
Final Design	2,107	655	265	113.4

Figure 15.9*Figure **15.9*** is an illustration of the Namibplaas pit design in conjunction with the ore above 140ppm grade.

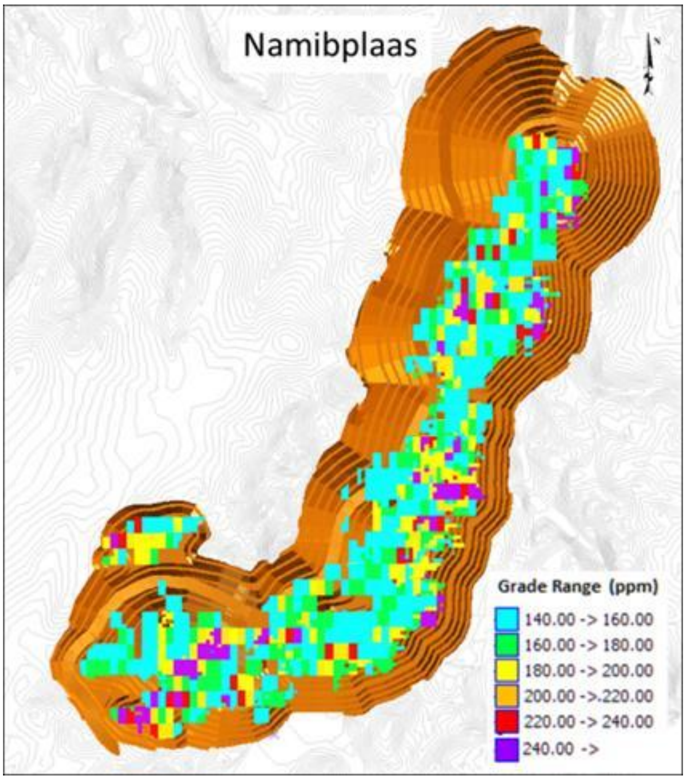


Figure 15.9 Plan view of the orebody within Namibplaas final pit design.

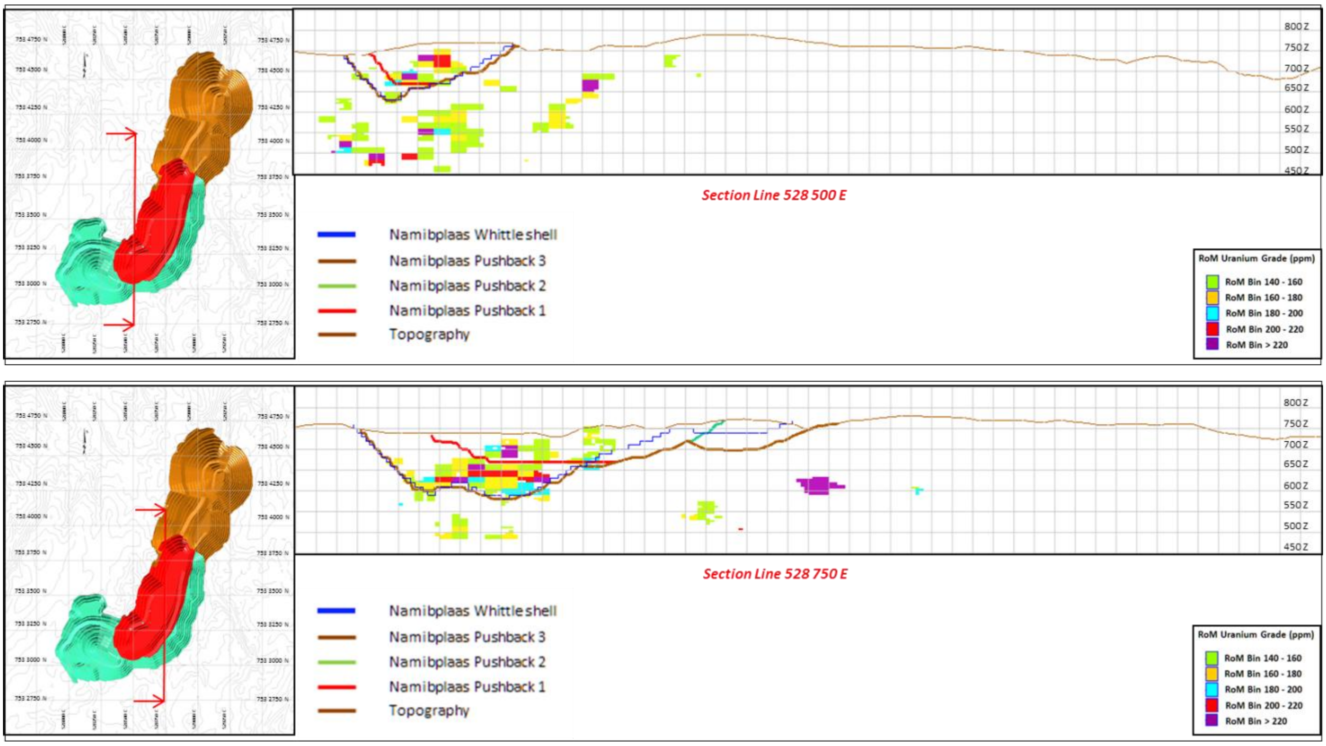


Figure 15.10: Namibplaas ultimate pit, pushback and optimal Whittle shell

Table 15.11 Total pit inventory of the Namibplaas orebody					
Parameter	**Unit**	**Pushback 1**	**Pushback 2**	**Pushback 3**	**Final design**

Total ore	Mt	8.7	24.2	18.2	51.0
Total waste	Mt	35.5	67.0	95.9	198.2
Total material mined	Mt	44.3	91.2	114.1	249.2
Stripping ratio		4.1	2.8	5.3	3.9
Average head grade	ppm	178.0	187.4	222.7	198
Metal U_3O_8 contained	Mlbs	3.4	10.0	8.9	22.3

15.2.4 Compliance with the Whittle Shell

The optimal Whittle shell represents the Shell with the highest NPV but not practical to mine because there are no access ramps. When the optimal shell is converted into a practical pit, the net present value of the resultant pit is expected to be lower because of the extra waste that will have to be mined to make room for access ramps. It is however important that the difference in volumes and overall value is kept at a minimum. Table 15.12 below shows a comparison between the ultimate pit design content and the Whittle shell.

Table 15.12 A comparison between the final pit designs and the optimal Whittle shell

Whittle Shell					
	Ore (Mt)	Waste (Mt)	Total (Mt)	U_3O_8 Mlbs	Grade ppm
Valencia Main	149.7	397.5	547.2	65.9	200
Valencia Sat	6.0	19.5	25.6	3.1	231
Namibplaas	52.9	217.9	270.6	24.3	209
Ultimate Pit Design					
	Ore (Mt)	Waste (Mt)	Total (Mt)	U_3O_8 Mlbs	Grade ppm
Valencia Main	148.9	438.6	587.4	65.3	199
Valencia Sat	6.2	23.5	29.7	3.1	229
Namibplaas	51.0	198.2	249.2	22.3	198

15.3 PRODUCTION SCHEDULE

In order to select the best suited production schedule for the combination of the various Norasa pits, various production schedules were created. Production scheduling was performed on the basis of the following requirements:

- maximised overall NPV for the project;
- plant feed tonnages and grade;
- stockpile movement and inventory;
- waste tonnes and total material mined;
- number of active benches and vertical advance; and

The basis of these schedules has consistently focused on the base case production options of a plant throughput rate based on two scenarios:

  - 11.2 Mtpa for scenarios incorporating non-radiometric sorting: and
  - 14.9 Mtpa with the radiometric sorting of the product (this tonnage is crushed and sent to the radiometric scan of which 11.2 Mtpa will be sent to the process plant at a higher grade).

The schedule was developed using Geovia's Minesched™ scheduling software and analysed primarily on relative project value and the cost of mining capital whilst taking cognisance of unit mining operating costs and vertical pit advance rates.

15.3.1 Scheduling assumptions and parameters

For each of the developed production schedules, Geovia's mine scheduling software was used as the primary scheduling tool. By adopting a rules-based iterative approach through Minesched, it allows for schedule of scenarios to fit specific requirements. Thus the best and most practical schedule can be chosen.

Although different options are assessed, it is still easy to accommodate major changes to mine design and geological data. Changes made at one point (or many points) will flow through the model according to the rules that are set.

Minesched develops schedules based on the following:

- a set of user defined objectives such as material movement, fleet capacity's, etc.
- a block-by-block approach as opposed to a bench-by-bench scheduling approach, consequently the software does not need to utilise bench averaging. Instead the true grades and true strip ratios are reported in any reporting period;
- any schedule generated by Minesched adheres to a number of rules that can be imposed by the user;
- furthermore, capacity constraints and targets can be used to control the tonnes, volumes content being mined for flagged material types; and
- the block size was chosen to fit into increments of the required mining blocks and bench heights. This is important since Minesched consolidates the blocks to fit the mining blocks as specified for the schedule.

The scheduling process and assumptions taken for each production schedule are described as follows:

- determine an appropriate production schedule which provides the optimal value whilst balancing practical mining constraints, particularly bench turnover rates and capital expenditure during ramp-up period; and
- the strip ratio is allowed to freely fluctuate in order to meet all the defined scheduling strategies.

15.3.2 Selected production schedules

The following production schedules were selected using the results from the Minesched simulations and incorporating the following factors:

- relatively high NPV;
- a low amount of additional capital expenditure over other schedules in maximising NPV;
- a practicality (vertical sinking and active benches).

The production scheduling model extracts material from each practical pit following a mining sequence, and mining rates for each pushback determined by the required stripping ratio. The model allows for mining from different benches, but a dynamic objective maximises the bench number and thus prioritises mining from the top benches.

15.3.3 Base case - 14.9 Mtpa process rate incorporating radiometric sorting

This scenario incorporates a rapid but steady production build-up over 4 years to a maximum of 74 Mtpa (Figure 15.11). Low grade and marginal material, although not considered ore as they fall below the strategic cut-off grade, are tracked as they represent additional profitable material that could be treated at a later stage.

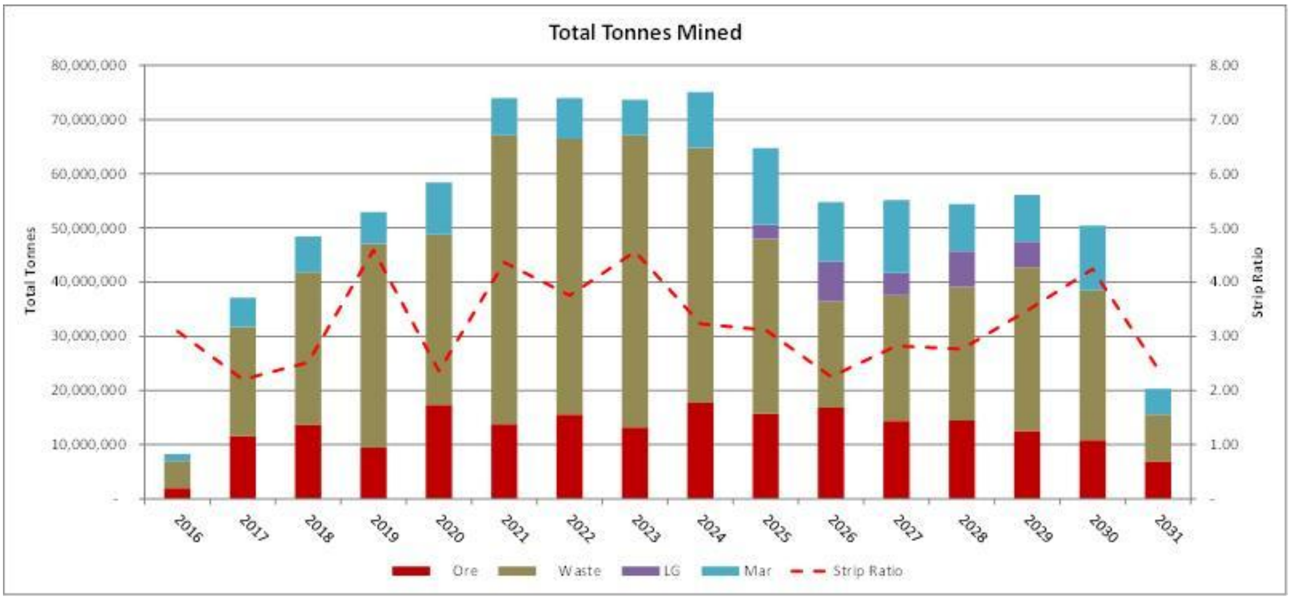


Figure 15.11 Total tonnes mined according to material type.

The mining rates are required to ensure adequate waste stripping to achieve constant plant throughput as the various pits and their relevant pushbacks start up. Figure 15.12 indicates the total tonnes in relation to areas / pushbacks mined annually, and also gives insight of the pushback release strategies.

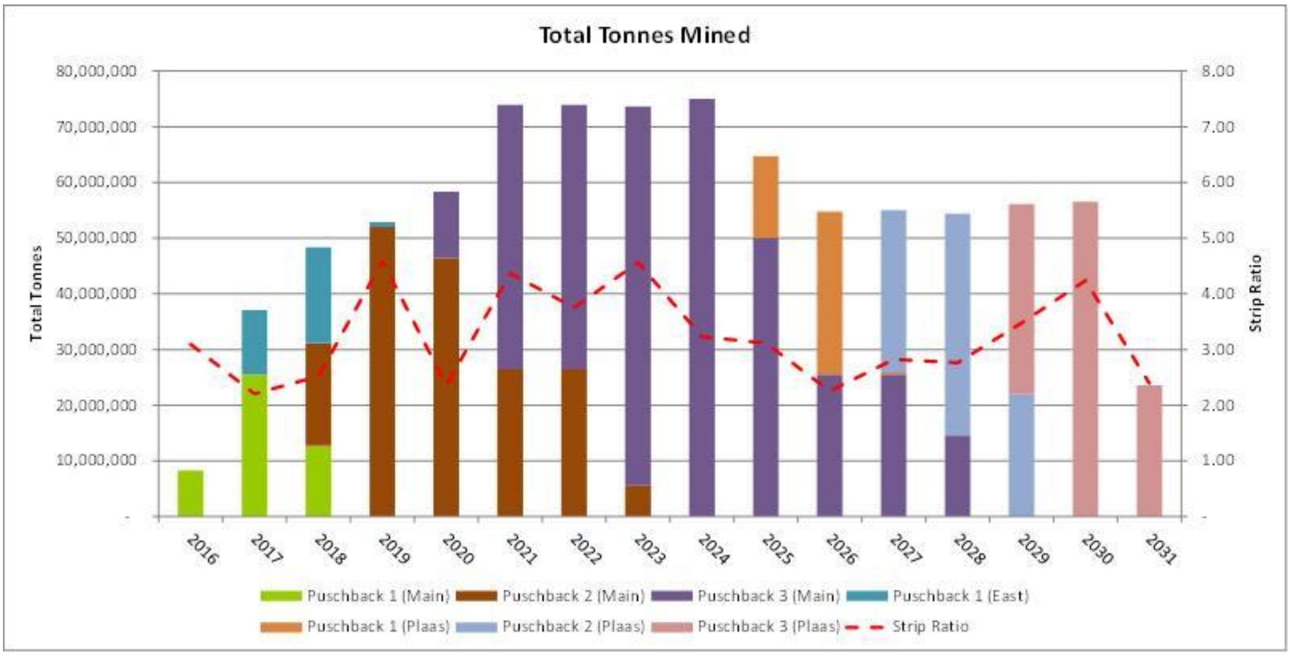


Figure 15.12 Total tonnes mined according to mining location and pushbacks.

Figure 15.13 illustrates the annual tonnes fed to the crusher. The target feed to the plant was 14.9 Mt per annum, which will be achieved in year 4 and continue for the life of mine. The plant feed balance is maintained with the high grade stockpile, whose material balance is illustrated in Figure 15.14.

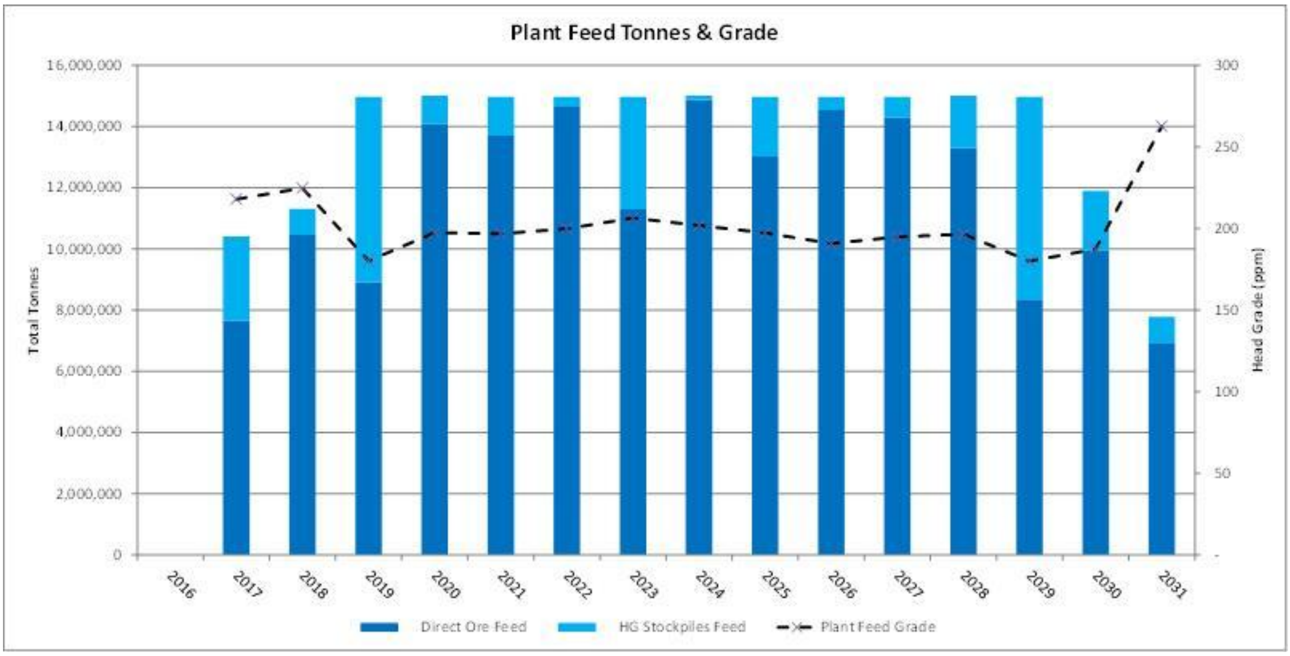


Figure 15.13 Plant feed tonnes and grade.

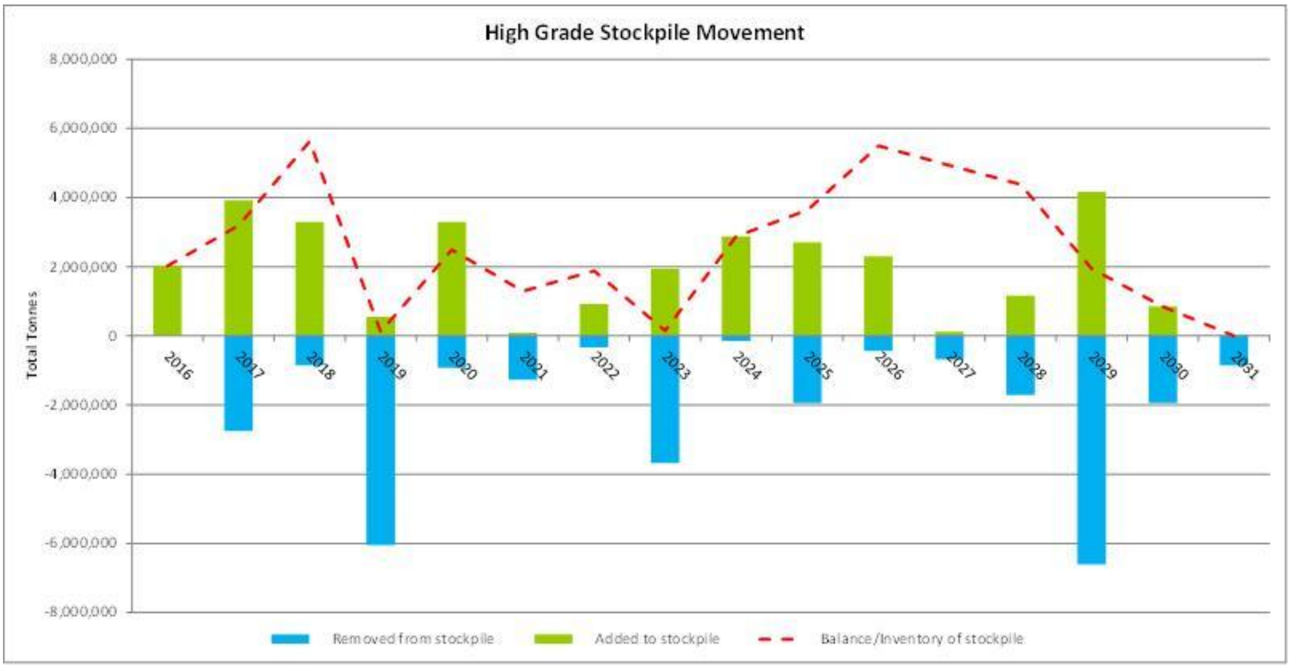


Figure 15.14 High grade stockpile movement.

16. MINING METHODS

The resource models reported in Section 14 were used as the basis for the mining study.

The mining study included the development of appropriate pit slope design parameters based on kinematic failure mechanisms and empirical slope design stability analysis. These parameters, together with process recoveries, operating costs and revenue factors, were used to define an optimal pit outline.

Practical pit designs were created using the identified optimal pit shells which in turn were used to develop a life of mine production schedule for use in financial modelling. Mining dilution (other than that implicit in the orebody model) was not included in the estimate because it was not considered material to the style of mineralisation. Resource models in network-vein style deposits contain an implied amount of dilution related to the block size and SMU size and it is not appropriate to attempt to model dilution beyond this scale.

Mining will take place by conventional drill, blast, load and haul methods. All hydraulic excavators will be configured as backhoe for loading from the top of benches. Due to the nature of the orebodies and mineralisation, selective mining practices will be implemented in specified areas where ore and waste loading needs to be more closely managed with suitable in-pit supervision and management systems. It is estimated that the 10m *in situ* benches will expand and bulk to about 12m after blasting and levelling. These benches will then be loaded in two flitches to maintain suitable levels of ore / waste separation as will be defined by the bench mining models. Areas of bulk mining (being ore or waste) will be mined with the same type of equipment to maintain flexibility of the fleet allocation. The option of configuring one of the excavators as a face shovel will be investigated at a later stage.

The bench heights will essentially dictate the blasthole pattern. The 10m benches will be drilled with 171mm diameter holes on a 4.7m x 5.3m burden and spacing, which will both manage the fragmentation and provide suitable bench model resolution to ensure that selective mining can be implemented.

16.1 GEOTECHNICAL CHARACTERISTICS OF THE PROJECT SITE

Norasa mineralisation consists of the uranium bearing alaskite that comprises massive stock work orebodies, dykes of varying thickness, veins and veinlets which can be either conformable with, or transgressive to the meta-sedimentary host rocks which consist of the psammitic Norib group and the calcareous politic granites and marbles. Mineralisation tends to follow the dominant foliation within the rock mass.

Several phases of deformation have been recognised within the rock at Valencia and Namibplaas. The emplacement of the alaskite appears to have been controlled by a NNE to SSW trending antiform folding with bedding and foliation dipping between SW and SE at moderate to steep angles.

Geophysical interpretation from aerial photography suggests that the pit areas are characterised by pit scale and smaller faulting predominantly in the schists into which the mineralised alaskites have been intruded. The majority of the interpreted structures cross cut the stratigraphy at a high angle and have been classified as axial plane faulting. The dip and dip direction of these interpreted structures is not known although they are likely to have dip angles steeper than 60 degrees. Faulting and shearing parallel to the stratigraphy is also thought to be likely based experience at Rössing Uranium. A number of thin dolerite dykes also cut the Valencia pit area and are likely to have intruded into existing fault structures (Snowden 2008).

Geotechnical rock mass classification of the major units in the vicinity of the proposed pits (schists and granites) indicates the rock mass is generally of good quality with average intact rock strengths of 80 to 150 MPa respectively.

The climate of the project area is desert with annual rainfall between 15 and 150mm. Topography is rugged with an average elevation of about 725m amsl with a range of approximately 40m around the deposits. The depth to the base of the weathering over the site is minimal.

Water is mainly found as sub-flow beneath the stream beds of the larger water courses such as the Khan River located 3km to the north-west of the deposits. Water for various exploration programs has been obtained within 2km of the deposits. Standing water levels in the drill holes suggests the natural groundwater level to be 700m at Valencia. At Namibplaas however, the ground surface is much more undulating and hence the water level is also more erratically located on an elevation basis. Measurements have determined that groundwater is not more than 15m below surface and generally follows the topography, except for near surface in the main riverbed that transects the deposit.

16.1.1 Rock mass structure

The structural models for Valencia and Namibplaas were developed from geological plans, surface outcrop mapping, geophysical mapping, aerial photography, oriented core drilling and downhole optical tele-viewer photography. The pits have been split into fault block domains based on the dominant faulting in the pits.

16.1.2 Slope stability

Pit slopes will be developed predominantly in granite and schist rock types at Namibplaas. The distribution of these major rock types across the deposit is complex and for the purposes of slope design the schist rock mass has been used to assess stability given the foliated nature of this rock type which will potentially have a significant effect of slope stability.

Given the strength and quality of the rock mass at Namibplaas, slope stability will be entirely structurally controlled. Based on a review of the structural mapping data, Xstract (2014) considers the principal failure mechanisms will be planar sliding on foliation and foliation parallel faulting and wedge failure of steeply dipping joint structures. There is also a potential for toppling to develop on foliation and foliation parallel faults on the eastern slopes of the pit.

There is also the potential for complex failure mechanisms to develop on the east wall of the pit where this slope is intersected by the interpreted major fault zone. The extent of any failure will be controlled by the width of the fault zone (35m to 65m has been suggested) and the rock mass quality within this zone. Current evidence suggests however, that the quality and character of the rock mass in the potential fault zone is very similar to elsewhere on the east wall and hence significant slope instability through the potential fault zone is not indicated.

The Type 1 cross cutting faults are unlikely to significantly impact slope stability.

16.1.3 Stability analyses

Kinematic stability analyses have been undertaken for the major pit slope orientations based on their interaction with the interpreted foliation and joint set orientations. Planar, wedge and toppling analyses have all been undertaken. The analyses have been undertaken on the following basis:

- Foliation and fault structures are assumed to be continuous on an inter-ramp scale.
- Joint structures are assumed to be continuous on a batter scale.
- Frictional strength of both foliation and joints has been estimated at 35° based on the results of direct shear testing carried out.
- A probability of failure (PoF) for batter slopes of between 20% and 30% is considered appropriate.
- A probability of failure for inter-ramp slopes of less than 5% is considered appropriate.
- Batter heights of 10m and 20m have been assumed.

16.1.4 Recommended slope design parameters

Recommended slope geometries are detailed in Table 16.1 to Table 16.3. The various slope design elements are defined in Figure 16.1.

Table 16.1 Recommended slope design parameters – Valencia

Pit Wall Sector	Inter ramp angle (º)	Bench stack angle (º)	Batter angle(º)	Batter height (m)	Berm width (m)	Overall slope angle (º) *
Footwall	42	45	55	20	8	42
Hangingwall	55	58	70	20	7	52+
SW Wall	55	58	70	20	7	52+
NE Wall	51	54	65	20	7	51

* based on a 300m high slope
\+ 12m wide catch berms @ 60m, 140m and 220m below ground surface

Table 16.2 Recommended slope design parameters – Valencia satellite

Pit Wall Sector	Inter ramp angle (º)	Bench stack angle (º)	Batter angle(º)	Batter height (m)	Berm width (m)	Overall slope angle (º) *
Footwall – Main Pit	53	56	70	20	8	51
Footwall – NE Ext.	42	45	55	20	8	41
Hangingwall	51	54	65	20	7	48
West Wall	54	58	70	20	7	51

* based on a 160m high slope which includes a 15m wide geotechnical catch berm @ 80m below ground surface

Table 16.3 Recommended slope design parameters – Namibplaas						
Pit Wall Sector	**Inter ramp angle (º)**	**Bench stack angle (º)**	**Batter angle(º)**	**Batter height (m)**	**Berm width (m)**	**Overall slope angle (º) ***
Footwall	39	41	50	20	8	40
Hangingwall	53	56	70	20	8	51+
South End Wall	49	52	65	20	8	48+

* based on a 300m high slope
+ includes a 15m wide geotechnical catch berm @ 100m and 200m below ground surface

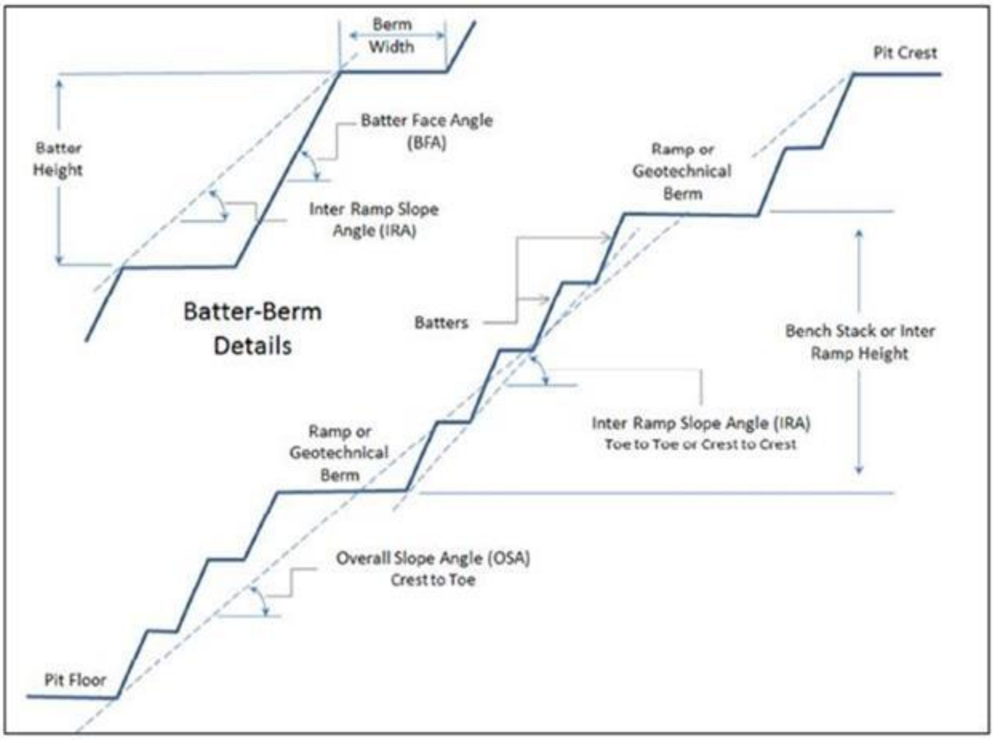


Figure 16.1 Definition of slope design elements.

The overall slope angles include a 12 to 15m wide geotechnical catch berm at various bench stack intervals. The position of this geotechnical berm are adjusted, or even eliminated, depending on the position of the haulage ramp(s) in the pit design.

16.2 MINING EQUIPMENT

The mine design was based on a conventional excavator and truck mining operation. To suitably manage the selective mining requirements near ore / waste boundaries, nominal 22m3 buckets are considered the largest that should be considered. The typical class of machine would include 400t class excavators combined with 180t off-highway haul trucks. The following fleet complement is proposed to manage up to 74 Mtpa mining rate:

- excavators: five 400t class backhoe configuration with 22m^3 buckets;
- haul trucks: building up to 37 units of 180t class;
- bench drills: 14 units capable of drilling 171mm blastholes;
- pre-split drills: two track rigs to drill perimeter holes;
- utility excavator: two 90t class;
- track dozer: four 60t units;
- wheel dozer: two 40t unit;

- FEL: two 15m³ unit;
- grader: three 25t units;
- water bowser: two 55m³ units;
- diesel bowser: two 35m³ units;
- generators, trailer mounted for lighting and pumping;
- others, including low-bed, mobile crane, forklift, tyre handler / tool carrier, lube and service vehicles, utility excavator with rock breaker and various LDVs.

For the purpose of the financial evaluation prepared for this report, mining has been based on an owner operator scenario. Associated capital costs include transport of equipment, establishment of office, warehouse and workshop and personnel related issues. Mining equipment purchases have been scheduled in accordance with the scheduled production build-up.

16.3 MINE WORKSHOP

The heavy equipment workshop will initially consist of a 6 bay facility to service and maintenance of all mining equipment. Five bays will be dedicated to tyre equipment and one reinforced bay for track type equipment. A stores area will be required for maintenance and service parts and equipment as well as a store area for all the rotary equipment.

The GET will be stored outside in a fenced in area. There will be a separate tyre bay with overhead roofing to protect the tyres against the sun. The welding bay is usually a separate section of the workshop. A small machine shop is also planned for small jobs that do not warrant sending work externally.

The bays will be equipped with 20t and 30t overhead cranes to remove major components such as engine transmission and axles. A mobile crane is also dedicated to the mining maintenance area.

Electrically driven steam cleaners rated at 80 amps single phase would also be required with one high pressure water pump to clean machines externally. The fuel and oil installation will have transfer pumps.

The complex will also have four offices including a small room for meetings and training. There will also be a change house.

Service and maintenance vehicles and equipment will consist of:

- 120t low-bed to move drill rigs, track dozers and utility excavators
- 90t mobile crane
- Tyre handler
- Fork lifts – 5t and 16t rated
- Service trucks, lube trucks and crane truck
- Mobile generator
- Firefighting trailer

16.4 MINE OPERATIONS STAFFING

Mine personnel estimates include both operating and salaried-staff personnel. Operating personnel are estimated as the number of people required to operate trucks, loading equipment, and support equipment to achieve the production schedule. Mine staffing has been based on the people required for supervision and support of mine production.

There are numerous forms of shift schedules in the surface mining industry, all with their associated merits and shortfalls. After analysing several shift schedules, it has been determined that a 2 panel full calendar shift schedule with 2 nominally twelve hour shifts is the most effective for Norasa. All personnel will be housed in an operations camp (referred to as the Valencia Village) during their working periods. The Village will also be available for contractors. For shift workers, they will spend 2 weeks on site being provided room and board and then be shuttled to their home locations for their off periods. Other staff will have accommodation available for the week days and then travel to their homes for the weekends.

Contracted services will include tyre management and a full blasting service.

The following summarises the 562 mining operation staffing for peak operations:

- equipment operators 390
- maintenance 91
- services & contractors 34
- mining management & technical 47

17. PROCESS DESIGN AND DEVELOPMENT

17.1 INTRODUCTION

The DFS flowsheet, shown in Figure 17.1, is an acid leach uranium flowsheet comprising of comminution, leaching, filtration, ion exchange, solvent extraction and ADU precipitation and processing areas.

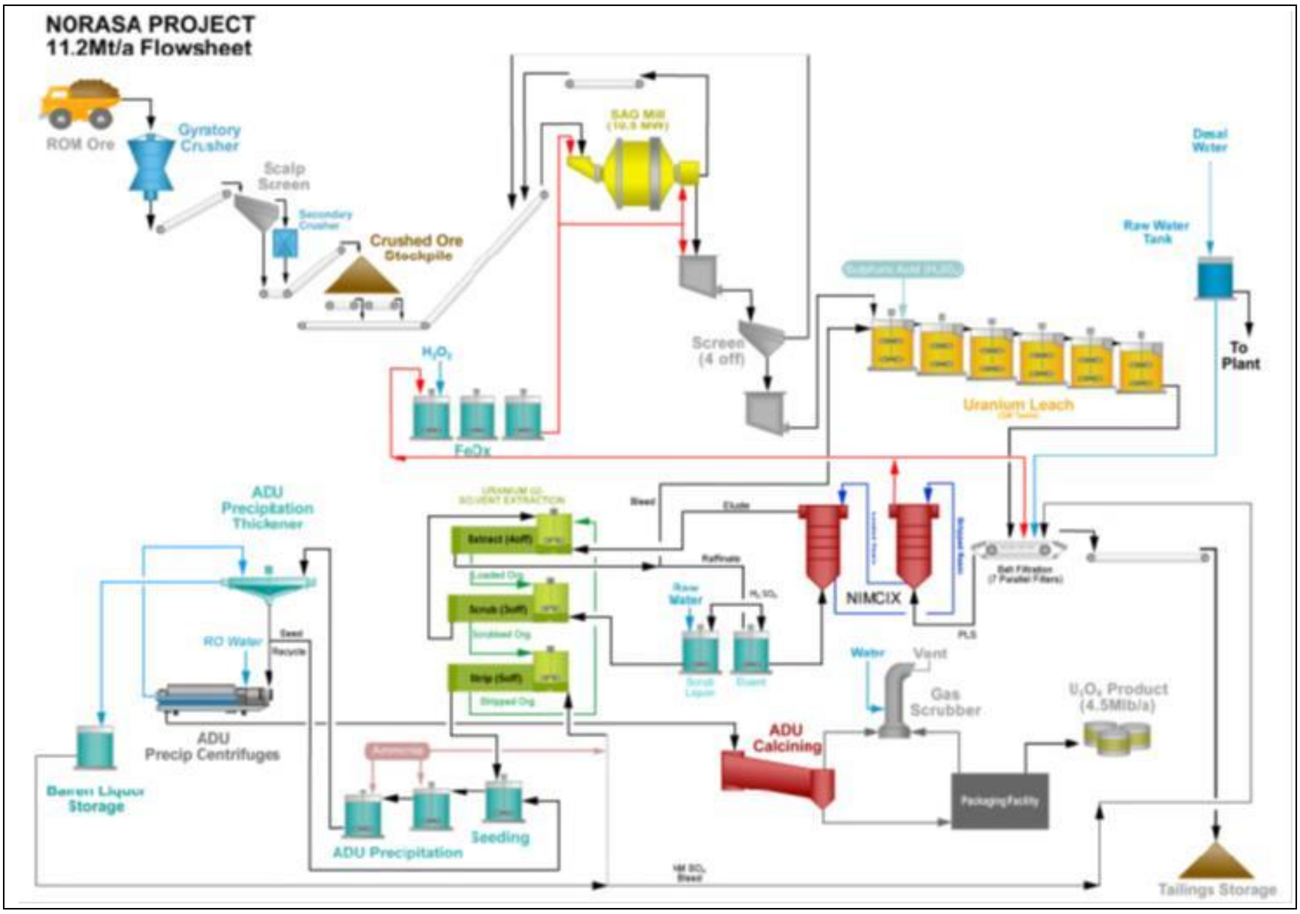


Figure 17.1 Schematic Norasa Flowsheet

17.2 PLANT PARAMETERS

The main plant design parameters are presented in Table 17.1 .The full rationale for plant design and flowsheet selection is provided in the AMEC Report No. 060551-0000-DW00-STY-0001 Rev 0 – Section 4.

Table 17.1 Plant Parameters			
Parameter	**Units**	**Value**	**Comments**
Average feed grade (U_3O_8)	ppm	200	
Plant throughput	Mt/a	11.2	
Plant production at average grade	Mlb U_3O_8/a	4.5	At 238 ppm U_3O_8 – 5.2 Mlb U_3O_8/a
Leach extraction	%	93.2	At 200 ppm U_3O_8 feed grade
Solid/liquid separation recovery	%	98.1	Filtration testwork
IX/SX recovery	%	99.9	
Overall recovery	%	91.3	At 200 ppm U_3O_8 feed grade
Raw water consumption	m^3/t	0.21	
Power consumption	kWh/t	18.0	

17.2.1 Feed Grade

The 2015 mine block model uranium distribution gives an average uranium content of 200ppm U_3O_8 for material above the cut-off 100ppm U_3O_8 level. Although radiometric sorting (Rad Sort) is not considered in the present study, the design allows space for incorporation of a Rad Sort facility. Should further Rad Sort testwork confirm the economic benefit of such an installation, this would potentially be installed at some stage after plant start-up.

17.2.2 Plant Throughput

A process plant capacity of 11.2 Mt/a has been determined by the predicted capability of a single train SAG milling circuit governed by the maximum diameter (28') known to be operating in an acidic grinding duty.

17.2.3 Leach Extraction

The 10 hour leach extractions for the 2014 SGS pilot tests were calculated from the residue analysis and plotted against head grade. A natural log function was fitted to the data and showed an improvement of around 2% on the earlier extraction-grade curve. The analysis having only three basis points for the pilot data may not be statistically valid, however demonstrates the pilot leach extraction results were above that which would be predicted by the bottle roll variability data.

From the pilot data fit, the 10 hour leach extraction at a grade of 200ppm U_3O_8 is estimated at 93.2%.

17.2.4 Downstream Uranium Recovery

The 2014 vacuum filtration tests showed that an overall recovery of close to 100% could be attained assuming three-stage washing with an overall wash ratio in excess of 0.6 m^3/t. For the design, the filtration wash curves were used to simulate two-stage washing with an overall wash ratio of 0.4 m^3/t. This gave a filter uranium recovery of 98.1% (which is in agreement with the recovery predicted in the testwork at this wash ratio).

The IX and SX technologies employed downstream of filtration are very efficient, plus the closed nature of the circuit minimises losses, with the only loss possible being through the ammonium sulphate bleed reporting to the back-end of the belt filters. The associated overall recovery is estimated to be in excess of 99.9%.

17.2.5 Total Recovery

The overall circuit uranium recovery is the combination of leach extraction and filtration plus downstream recovery, and at 200ppm U_3O_8 this gives 0.932 x 0.981 x 0.999 = 0.913 or 91.3%. Using a similar approach, based on the leach extraction variation with grade correlation

17.2.6 Process Description

The ore processing flowsheet selected for the Project's DFS comprises the following unit operations: primary and secondary crushing, SAG milling, sulphuric acid leaching, belt filtration, ion exchange (IX) and solvent extraction (SX), ammonium diuranate (ADU) precipitation and thickening, calcining of the ADU cake and packaging of the U_3O_8 product, tailings storage, reagent storage, make-up and distribution, and utilities (water and air) storage and distribution. The text that follows describes each of these areas in detail.

17.2.6.1 Crushing

The production objective is to feed ore directly to the primary crusher and minimise stockpiling of run-of-mine ore. Provision for stockpile rehandling is included in the layout to enable an improved feed blend control if necessary. In either case, ore is delivered to the gyratory crusher feed bin by dump trucks of 180t capacity.

The crushing circuit is two staged with an open circuit secondary crushing configuration designed to generate a combined product of 80% passing 32 mm for SAG milling.

The gyratory crusher feed bin is positioned below ground above the crusher and includes a two sided dumping access. There is no sizing grizzly ahead of the primary crusher. The primary crusher is a 60" x 89" gyratory operating at an average open side setting of 165 mm. The operational crusher setup will be such that a product 80% passing size of 120 mm is targeted and a top size of 350 mm achieved. The primary crusher has a rock top size limit of 1,200 mm.

The crushed ore gravitates into a 400t capacity bin, which discharges onto an apron feeder that regulates the feed rate to downstream. The crushed product reports via conveyor belt to a vibrating double deck scalping screen. Oversize from the screen (+38 mm) material gravitates to a secondary cone crusher. It is anticipated that the secondary crusher will operate with a closed side setting of approximately 37 mm. Undersize from the scalping screen (-38 mm) combines with crushed ore from the secondary crusher (P_{100} of 58 mm), which is then conveyed to the crushed ore stockpile (COS).

Dust suppression sprays (water only) and forced extraction points are strategically located around the primary and secondary crushing circuits to minimise dust emission during the tipping and crushing operations. The forced extraction ducts transport the dust-laden air to dust scrubbers where the solids are removed by spray water in a tower prior to venting the air to atmosphere.

17.2.6.2 Milling

Ore from the crushing circuit is stockpiled on a crushed ore stockpile which provides 24 hour live surge capacity between the crushing and milling circuits. The air quality in the COS reclaim tunnel is maintained by displacing air via a pressurising ventilation fan. Dust extraction is also performed at the reclaim points and transfer points.

Dust-laden water from the primary, secondary and milling area dust extraction scrubbers' is combined and clarified by a pin-bed clarifier prior to recycle of the reclaimed water to the various scrubbers. The pin-bed clarifier underflow slurry is pumped to the milling area fines screen feed distribution box. Raw water is used as make-up for the nett requirement of the scrubbing water system and dust suppression sprays.

Mill feed is reclaimed from the crushed ore stockpile by two variable speed apron feeders that discharge onto the COS reclaim conveyor. This conveyor transfers material to the SAG mill feed conveyor. Ore from the mill feed conveyor is charged via a removable mill feed chute into the SAG mill which operates in closed circuit with up to four double deck screens. The SAG mill has variable speed capability to manage power draw, charge weight and recirculating load. The mill feed rate is governed to maintain a mill weight set point and or limit at a downstream constraint.

IX barren solution is added to the mill inlet to achieve a milling solids content of around 70% (w/w). Acidic IX barren solution is used around the grinding circuit for solids slurrying and as spray-water to facilitate screening of fines. The acid consuming capacity of the ore will neutralise most of the acid in the barren liquor and materials of construction are appropriately selected to allow for the acidic conditions. Around 20 to 30% of the uranium will be leached in the milling circuit.

The SAG mill pulp discharges via a 12 mm aperture trommel screen, with oversize pebbles being recycled via two conveyors to the mill feed conveyor. Some barren IX spray-water is applied to the screen for cleaning oversize and to facilitate screening. The undersize (pulp) gravitates into the mill discharge hopper.

Grinding media (i.e. high Cr steel balls) is automatically charged to the mill via the mill feed conveyor as needs dictate. A bulk grinding media container unloader with associated chute is included to allow transfer of the grinding media into the SAG mill grinding media storage bunker and enable grinding media addition via the automatic SAG mill ball loader.

Barren IX solution is added to the mill discharge hopper to dilute the mill fines screen feed pulp solids content to 60.5% (w/w). The pulp is then pumped onto four double deck vibrating screens via a distributor. The solids are screened on the upper deck at 3.35 mm aperture and the lower deck at 1.1 mm aperture, targeting lower deck undersize product size distribution of 80% less than 600 µm. The oversize is returned to the SAG mill feed conveyor via the same two conveyors used for pebble return. Barren IX solution is used as screen spray water to clean oversize and in so doing maintain high screen efficiency.

The ground slurry (screen undersize) discharge into hoppers which pump directly to the leach feed splitter box. The slurry is 48% (w/w) solids by mass. All four screen undersize hopper pumps are driven by variable speed motors.

Spillage generated in the milling area circuit gravitates to a conventional spillage pump arrangement and is pumped back to the mill circuit via the mill discharge sump. Allowance is made for skid-steer loader drive in to remove coarse and excess solids from the area.

17.2.6.3 Leaching

The ground slurry is mixed in the leach feed splitter box and then sampled using a combination of a primary and secondary sampling system. The leach circuit is split into two trains, each with six agitated tanks, to minimise impact of the hydraulic level drop across the train as the slurry cascades by gravity from the first to the last tank. The leach splitter box incorporates underflow and overflow weirs to ensure mixing and equal distribution of the slurry between the two leach trains.

For each train, the leach feed reports to the first in a series of six mechanically agitated tanks. The tanks are interconnected by overflow launders and the slurry cascades from the first to the last tank by gravity. The tanks are arranged in a triangular fashion to allow bypass of a single tank for maintenance, via bypass launders. The mean leach residence time is 10 hours. The outlet from the last tank in each train gravitates to a pencil tank from where the slurry is pumped to two three/four-way filter feed distributors.

The leach reagents, namely sulphuric acid and hydrogen peroxide, are added in the first three or four leach tanks, with the bulk of the reagent addition expected to be into the first tank to initiate leaching by adjusting the pH and ORP to 1.7 and 470 mV (Ag/AgCl) respectively.

In the leaching circuit, uranium is solubilised by the oxidation of tetravalent uranium (U^{4+}, which is insoluble in acidic solutions) to hexavalent uranium (U^{6+}, which is soluble in acidic solutions) in the presence of ferric ions:

$$UO_2(s) + 2Fe^{3+}(aq) \rightarrow 2Fe^{2+}(aq) + UO_2^{2+}(aq)$$

The ferric ions are supplied by recycling oxidised IX barren liquor to the grinding circuit and by oxidising leached ferrous ions in the leach circuit by hydrogen peroxide:

$$2Fe^{2+}(aq) + H_2O_2(aq) + 2H^+(aq) \rightarrow 2Fe^{3+}(aq) + 2H_2O$$

The ferrous-ferric couple is important to facilitate fast uranium oxidation kinetics by acting as an electron transfer agent. Acid is consumed in the ferrous-ferric oxidation reaction and by gangue in the ore. The pH needs to be maintained below 2 by acid addition to prevent ferric precipitation with associated impairment of uranium oxidation and leaching.

Spillage generated in the leach feed area gravitates to a conventional spillage pump arrangement and is pumped back into the feed splitter box. Similarly, spillage generated in the rest of the leaching area gravitates to a conventional spillage pump arrangement and is also pumped back into the leaching circuit via the feed splitter box.

Acidic barren IX liquor is recycled to grinding and leaching to dilute the slurry to 48% (w/w) solids content. By recycling this liquor, the residual acid is utilised and the need for iron addition eliminated. Prior to recycling the barren IX liquor, the bulk of the ferrous iron is oxidised to ferric iron by contact with hydrogen peroxide. The ferrous oxidation circuit comprises three mechanically agitated tanks, with the solution cascading by gravity via overflow launders from tank to tank. The oxidised solution is pumped from the final ferrous oxidation tank to the grinding circuit and leach as required.

17.2.6.4 Filtration and Clarification

The relatively coarse grind makes belt filtration a very attractive solid-liquid separation methodology. The filtration circuit comprises seven 150 m² belt filters, with the intention of having one standby and six in duty at any time.

The filter feed gravitates from the two distributors to six individual feed boxes feeding six 150 m² belt filters. Flocculent is added at a predetermined volume to assist filter duty. Each belt filter is divided into four zones, namely the formation zone, wash zone 1, wash zone 2 and dry/neutralise zone. The filtrate from the formation zone and the first wash zone is combined in a receiver per filter and pumped as the filtrate product to clarification. The filtrate from the second wash zone and the dry zone combines in the second filtrate receiver per filter and is returned as a recycle solution to the first wash zone. This is to increase the wash efficiency on the filter. Wash solution in the form of IX barren is fed to the second wash zone and the filter cloth spray system. Wash water in the form of process water is added to the final wash zone to make up an overall wash ratio of 0.5. Milk of lime is added at the start of the dry zone to neutralise the residue.

The clarifier underflow slurry is returned to the first wash zone as this contains pregnant solution. The “dry” filter cake contains 15 to 17% (w/w) moisture. This is deposited directly on the residue conveyor and reports via a second conveyor to the dry tailings disposal facility.

The vacuum for the filters is supplied via a vacuum pump per filter on a common manifold. These vacuum pumps (7 off) require water that is kept cool by a common water cooling system, incorporating a hot well and cooling tower.

The filtrate is clarified in two pin bed clarifiers operating in parallel. The filtrate from all filters is combined in the clarifier feed tank from where it is pumped by individual pumps to the feed wells of the two clarifiers. Each pin bed clarifier periodically back-washes, with the wash liquor being returned to the feed tank. Coagulant is dosed to the feed solution to assist in clarification and removal of polymeric silica. The clarifier underflow is returned to the leach discharge pencil tank(s). The clarified solution is the feed to the ion exchange plant and reports to the pregnant leach solution (PLS) pond.

17.2.6.5 Residue Stacking

The filtered and washed residue will be transferred to a mobile boom stacker arrangement via tripper and movable conveyors. The residue is stacked in the residue storage facility (RSF). Seepage from the RSF is collected in a pond from where it is returned by pumping to the barren IX solution pond.

17.2.6.6 Ion Exchange

The design of the IX circuit for the Project utilises NIMCIX technology for sorption and elution. NIMCIX is a counter-current ion exchange technology where the resin-solution contact is facilitated by fluidising the resin in a column by solution up-flow. The resin is retained in the column by gravity, with the column having a flare at the top with overflow weir. Internal mixing and/or migration of resin is prevented by perforated plates (trays) which divides the column into stages. Flow of solution through the NIMCIX contactor is semi-continuous with intermittent transfer of resin slugs.

17.2.6.7 Solvent Extraction

The IX eluate is transferred into the SX feed tank which provides surge to minimise feed flow rate fluctuation to SX. The feed tank also allows mixing of spent scrub liquor, recovered aqueous (from crud treatment) and spillage with the eluate providing a reasonably constant SX feed composition.

Whereas IX uses a solid extractant, SX uses a liquid extractant which is contained in an organic phase. This facilitates continuous counter-current contact of the aqueous and organic phases. A range of specialised equipment has been developed to achieve this counter-current liquid-liquid extraction, with the mixer-settler being the most common. A mixer-settler comprises of a mix tank where the aqueous and organic phases are intimately mixed to form an emulsion which then flows into a settler where the phases separate. Multi-stage counter-current contact is achieved by connecting mixer-settlers in series with the aqueous and organic streams flowing in opposite directions from unit to unit. The hydraulics are simplified by employing pumper-mixers in the mix tanks which both mix the phases and provide the head for flow between stages.

The organic solvent used in the SX circuit comprises three distinct reagents; a tertiary amine extractant, a kerosene diluent and an alcohol based third phase inhibitor. Mixed in the correct ratios, these three reagents constitute the SX organic.

17.2.6.8 ADU Precipitation and Thickening

In this area, the uranium is precipitated from the loaded strip liquor and the resulting precipitate is thickened to dewater the solids for subsequent washing.

Ammonium diuranate (ADU) is precipitated from the loaded aqueous using anhydrous ammonia gas:

Ammonium sulphate is produced during precipitation. Precipitation takes place in four mechanically agitated tanks in series. Temperature control during ADU precipitation is important. If it is too high or too low, the sulphate content and the physical size of the ADU particles can be adversely affected, with resulting poor solid-liquid separation and washing properties.

The ADU slurry gravitates from the precipitation tanks to the feed box of a thickener and is diluted with centrate from the first washing centrifuge before being fed to the ADU thickener. Thickener underflow is pumped to a mechanically agitated centrifuge feed tank where the slurry is diluted to the required feed solids concentration using centrate from a second washing centrifuge and demineralised water. Thickener overflow gravitates to an overflow collection tank and is pumped through a polishing filter before being recycled back to the strip circuit in SX. The polishing filter is periodically backwashed, with the wash slurry being returned to the thickener. Excess barren ammonium sulphate solution is bled to the filtration circuit and combined with the filtered residue for disposal.

17.2.6.9 ADU Washing and Calcining

In the ADU washing and centrifuge circuit, the ADU precipitate is washed and dewatered for calcination.

Thickened ADU slurry from the ADU thickener is pumped to the first of two washing centrifuges. Centrate from this washing centrifuge gravitates back to the ADU thickener feed box as dilution for the incoming ADU slurry. The cake from the first washing centrifuge discharges through a chute into a mechanically agitated re-pulping tank and is diluted with demineralised water ahead of the second washing centrifuge. Centrate from this centrifuge is recycled as dilution for the feed to the first washing centrifuge. ADU cake from the second washing centrifuge discharges into a feeder, which delivers the ADU cake to a porcupine dryer where the bulk of the moisture is removed. The dried powder flows by gravity into a rotary calciner located below the dryer where the ADU thermally decomposes to uranium oxide (U_3O_8). The calcined product gravitates to the product hopper in the product packing area.

The off-gas from the dryer and calciner is cooled and treated in a bag filter and scrubber for particulate and ammonia removal. Recovered dust is slurried and returned to the ADU thickener.

17.2.6.10 Product Packing

During day shift, U_3O_8 will be packaged into 210 L drums. To accomplish this, a lined drum will be positioned in the packaging capsule below the product hopper. A rotary valve will be activated to fill the drum with a set amount of U_3O_8. When the drum is full, it will be conveyed to the encapsulated lidding station where the drum lid, seal and clamping ring will be fitted. The lidded drum will then be pushed along the roller conveyor, cleaned, weighed, labelled and stacked in the product storage area adjacent to the packaging room. Once sufficient product drums have been loaded into a standard shipping container, the container is stored on site until sufficient containers are available to make up a shipment for delivery to contracted refineries.

18. PROJECT INFRASTRUCTURE

18.1 INFRASTRUCTURE AND SERVICES

Infrastructure design and planning was undertaken by Valencia in conjunction with relevant national authorities. Commentary on key areas is provided below.

18.1.1 Bulk water supply

Norasa has received NamWater's (Namibia's national bulk water utility) assurance of a supply of water during the construction phase of the project. This will require a 31km temporary pipeline extending from the Rössing reservoir to the construction site. Valencia will design and construct this temporary pipeline with a 300 m^3/day capacity required to service the construction camp and for construction activities. This pipeline is to be installed adjacent to the completed access road. Production from the Norasa will require construction of a permanent 31km main pipeline (replacing the temporary line used during mine construction) linking Norasa to the Rössing reservoir. The Company is working with NamWater, who is responsible for the tendering and construction of this water pipeline.

It was announced in late September 2014 that the Namibian government intends to buy Areva Namibia's seawater desalination plant, located north of Swakopmund. Areva is currently operating the plant on a limited scale to provide water to local mining operations through NamWater. The plant was originally designed to produce 20M m^3/annum of potable water, providing sufficient capacity for the existing mines in the region (requiring less than half this volume) with spare capacity for newcomers on a first-come first-served basis. At this stage, only about half the Reverse Osmosis modules have been installed. The overall plant capacity can be expanded.

Previously, the government appointed the National Desalination Task Force to proceed with a tender process for its own desalination plant just north of Swakopmund. By mid-2012, three competitive proposals were received, but the process has been shelved as the Areva deal was being pursued.

Some of the existing infrastructure may require upgrade to cater for Norasa and the expansion plans of other operations. Norasa has requested a water allocation of 3M m^3 annually to cater for initial operating requirements and planned expansions.

18.1.2 Power supply

The nearest power off-take point that can supply Norasa is the Khan Substation, located at Ebony, 26km north of the project site. However, the direct route is very rugged through the Khan Valley and tributaries, and an indirect transmission route of nearly 30km has been laid out by NamPower.

The Khan Substation has recently been upgrading and expanded. NamPower carried the cost of the new substation although a new bay for Norasa will be at the mine's expense, as will be the cost of the transmission line to mine.

Power distribution to the mine is planned to be a 220kV transmission line. At an installed capacity of approximately 35MW and a mine draw of about 32MW, two 40 MVA transformers would be installed, one of which would be maintained as a backup unit. This study assumed that the mine would have to carry the cost of establishing the substation.

A decision on standby power generators will be taken closer to the time. The generators would be connected to a synchronization and load control panel to operate the generator sets. This control panel will consist of a switchboard arranged for automatic synchronizing of the generator sets, which would include motorized circuit breakers to synchronize the generator sets to a common bus bar. We would include a bus coupler to split or combine the common bus bar to give flexibility to synchronizing or power sharing. A centralised diesel power station has not been allowed for at this time.

18.1.3 Roads

The preferred route to access the mine was determined to be across the Khan River, using tributary valleys. This route links the mine to the B2 highway, 12km northeast of Rössing. The total length of this new road is approximately 26km.

The crossing of the Khan River was designed with low-water culvert structures with concrete drifts between them. The system was designed such that in the event of exceptionally large flood events, water will wash over the road, leaving it temporarily impassable (matter of hours), but undamaged. During such times, alternate routes are available for personnel transport. Roadside drainage systems have been catered for in the design.

Construction of the industrial grade gravel road was completed in mid-2010. Some of the internal service roads were also constructed. As part of the initial pit establishment, an allowance was made for 7km of road.

In order to minimise dust generation and reduce water consumption for dust suppression, it is planned that permanent and long term road be sealed using a bitumen or polymer based binder. This will also improve tyre equipment travel efficiency and extend tyre life. On pit ramps, it will also impact on the ramp speed of loaded haul trucks. The monitoring and maintenance of the sealed road will be done by the supplying contractor, who will also provide his own road maintenance equipment.

18.1.4 Buildings and other

A central office and administrative complex for plant, mining and mine management personnel will be constructed. Change house facilities for mining and plant will however be separate as will workshop and stores buildings.

A mining workshop with fuel station will be constructed as required for the size and number of equipment units.

It is envisaged that a regional and independent assay laboratory will be setup to service the needs of the site. Allowance was however made for a sample preparation facility on site.

A construction camp will be established near the site during the project construction phase. The camp will be serviced with the main supplies coming from the coastal town of Swakopmund.

Security fencing will be installed to control access and prevent intrusion by wildlife. Security personnel will regulate access points and patrol boundaries.

A multi-channel, two-way radio communications system will be installed on site as an official form of communication. Cellular phone communication is also currently available at many areas of the project site (including the current camp location).

18.1.5 Mine Accommodation

All mine personnel will be housed in the mine's operations camp, commonly referred to as the Valencia Village, which will be the construction camp upgraded into a life-of-mine accommodation facility. Due to limitations in infrastructure in the towns near Norasa and the already increasing number of people migrating to Namibia's central coastal area, the decision was taken to provide employees with accommodation on site. The Village is restricted to employees only while they are on their work rotations; shift personnel staying for 2 weeks at a time while weekday employees will go home over the weekends. On-site transport is provided, as is bussing to and from nearby towns as people report and leave for their cycles.

The construction camp is sized to accommodate 800 people for the duration of construction. The camp will consist of accommodation suitable for the various levels of personnel, kitchen and dining rooms, cold rooms, recreation rooms, laundry and security. The camp will be fully serviced by a contracted service provider who will provide three meals daily for all site personnel, laundry service, cleaning and housekeeping, maintenance of all site facilities, pest control and waste management. The facility will be converted to an operations camp suitable for at least 450 people at any one time, with the service contract ongoing. The camp / Village will be located about 8km from the main Valencia operations site.

18.1.5.1 Construction Phase

During the construction phase, the camp will need to be powered by generator until the site is connected to the national grid. There will be four 250kVA generators powering the camp. A medium voltage transmission line will be extended to the construction site to supply the contractors with sufficient power (additional mobile generation capacity will be installed at the construction site for localised needs). Once grid power is established, the camp (Village) will already be connected and the generators will remain in place as back-up capacity.

A temporary pipeline will need be extended from the Rössing reservoir to site, from which the camp will be supplied. A small reservoir and water disinfection plant will ensure potable water for the camp and construction site. Sewage treatment and septic tanks will be in place for the life of mine.

18.2 TAILINGS DISPOSAL

Norasa will develop a dewatered tailings dump and not a slurry tailings dam. After the uranium has been leached from the ore, the silt tailings will be thickened and dewatered with the use of belt filters, before being conveyed to the dump.

A proposed dump stacking system is required to collect the tailings at the plant and convey it to the proposed tailings stacking dump. The conveyor capacity is based on 1,650tph with a bulk density of 1,450kg/m³. The conveyor from the plant to the base of the dump will be approximately 560m long and will the run up a 12 degree ramp formed by the mine using waste material generated during the pre-strip period (1.6Mt). The height to the top of the dump will be 50m and the conveyor from the plant to the top of the dump will be a single flight system with a drive house situated at the base of the dump. The single flight conveyor will eliminate the spillage associated with a transfer point.

At the top of the dump, the conveyor will feed in two directions, firstly onto a dump "Finger Stacker" and alternatively onto a small throw out conveyor running at right angles to the finger stacker. The stacker will initially be equipped with a 14m boom which will be extended at a later date to 24m. The throw out conveyor will eventually be replaced with a full additional finger stacker system again running at right angles to the first unit. Using a single stacking system will require a movement of the stacker to extend the feed conveyor. This will require the bypassing of the stacker onto the throw out conveyor; the movement will take 2 days until the boom is extended to 24m. The additional stacking system will make the movements less frequent.

The total system will consist of:

- A single 900m long conveyor from plant to top of dump (+70m).
- A combined drive house at the base of the dump with the necessary drives and take up system.
- A transfer tower at the top of the dump allowing for a feed onto a stacking system and an alternative throw out conveyor.
- A take up system to the stacker conveyor complete with 50m of belt storage and 300m of modules, belt and idlers.
- A finger stacker complete with 14m counterweighted boom which will be extended to 24m with the addition of another 10m of boom and additional counterweight.
- The finger stacker will be advanced on a rail bed using its own hydraulic power pack.

A dedicated track dozer is planned for the tailings dump.

18.3 WASTE DUMPS

In 2009, Epoch Resources (Pty) Ltd prepared an assessment of a residue disposal facility for Valencia. This work has now been expanded to include Namibplaas and the figures below now cover the requirements for Norasa as a whole.

The residue disposal facility for Norasa comprises the following installations:

- A "dry" fine tailings dump storing 206 million dry tonnes of fine tailings over the LoM.
- The Valencia waste rock dump stores 462 million dry tonnes of open pit waste from both of Valencia's pits. The dump is divided in the middle by the primary ore crushing station and stockpile area (Figure 18.1).
- Two waste rock dumps at Namibplaas store collectively 198 million dry tonnes of pit waste. The current plans assume two separate dumps generated from 2 ramp systems (Figure 18.2).

The design criteria associated with the Norasa dumps are summarised in Table 18.1.

Table 18.1 Norasa waste and tailings dump design criteria.

Parameter	Value/Description
LoM	15 years
Total fine tailings	206 million dry tonnes
Total rock waste Valencia	462 million dry tonnes
Total rock waste Namibplaas	198 million dry tonnes
Average *in situ* dry density of fine tailings	1.50 t/m^3
Average *in situ* dry density of waste rock	2.00 t/m^3
Solid particle specific gravity	2.63 t/m^3

The sites for the rock dumps and tailings dump were thus identified taking into account surface and mineral rights areas, as well additional infrastructure and environmental constraints imposed on the dumps. In the case of the waste dump sites, proximity from the open pit was a major factor so as to limit overhaul distances and associated operating costs.

The tailings dump is to be constructed using the fine tailings which has been through a filter press, producing a "dry" filter cake tailings, and deposited off an advancing conveyor system. To control erosion and dust, the final exposed faces of the dump are to be flattened to slopes of 1V:4H (14.0 degrees) using dozers and cladded with waste rock sourced from the open pit.

The waste dumps will be constructed using the open pit waste, which is trucked and tipped to achieve the required dump configuration and profile. The final exposed faces of the dumps are to be flattened to slopes of 1V:3H (18.4 degrees) using dozers.

A summary of the physical parameters of the RDF is provided in Table 18.2.

Table 18.2: Summary of Norasa RDF parameters

Item	Capacity (m^3)	Area (m^2)	Elevation (m)
Tailings Dump	140 million	5.10 million	820
Valencia Dump	230 million	4.39 million	814
NP Dump South	51.1 million	1.53 million	760
NP Dump North	49.6 million	1.20 million	760

These designs take no consideration, at this time, of backfilling mined out areas of the pits instead of disposing on the designated waste dumps. The orebodies are currently open-ended at depth and planned backfilling at this time could potentially sterilise future expansion options of the operations.

The general principle of separating “clean & dirty” storm water is to be applied. Storm water arising and draining off the tailing dump is to be captured in a storm water dam for reuse as make up water in the process plant. The storm water dam is designed to store the volume of water arising from a 1:50 year 24hr storm over the tailings dump operational footprint area.

In terms of acid mine drainage (AMD) the main uranium mineral is uraninite, or a uranium oxide with the secondary mineralisation in the form of oxides or carbonates of uranium that in general indicates that the system should tend to be alkaline in nature. Results of an analysis which had been conducted on a batch of leached residue (Epoch Resources 2008) indicated that the generation and management of AMD should not be a significant problem as:

- The sample has a neutralising capability, i.e. it does not appear to be net acid generating, with a net neutralising potential of 19.7 and a neutralising over acid potential ratio of 64.2.
- The net acid generation test also confirmed that the sample was non-acid forming.
- The mineralogical tests indicated that the sample major constituents are quartz, microcline, plagioclase and biotite. Pyrite levels were less than 3%.

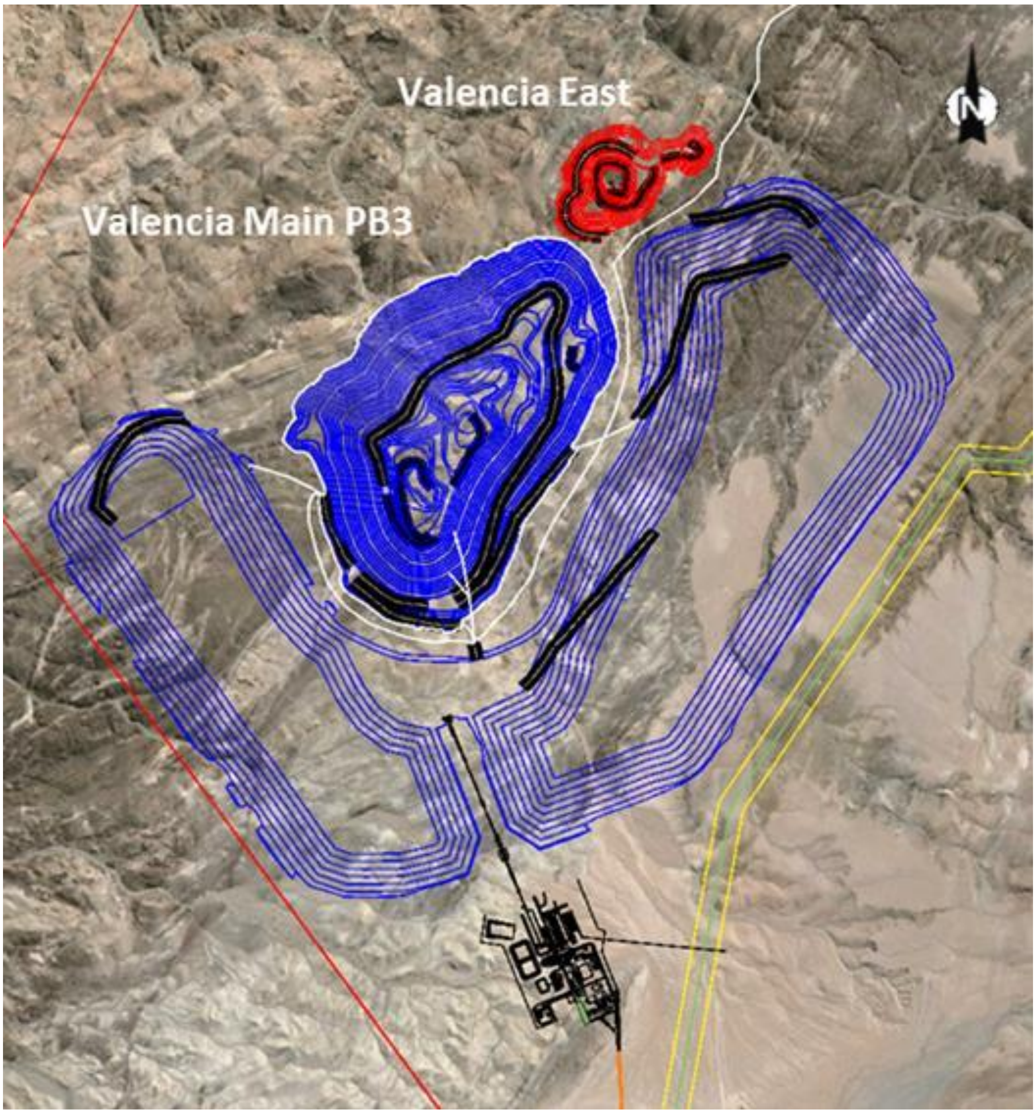


Figure 18.1 Proposed Valencia pit and waste dump locations with the process plant.

Proposed mine roads shown in white, process plant in black to the south.

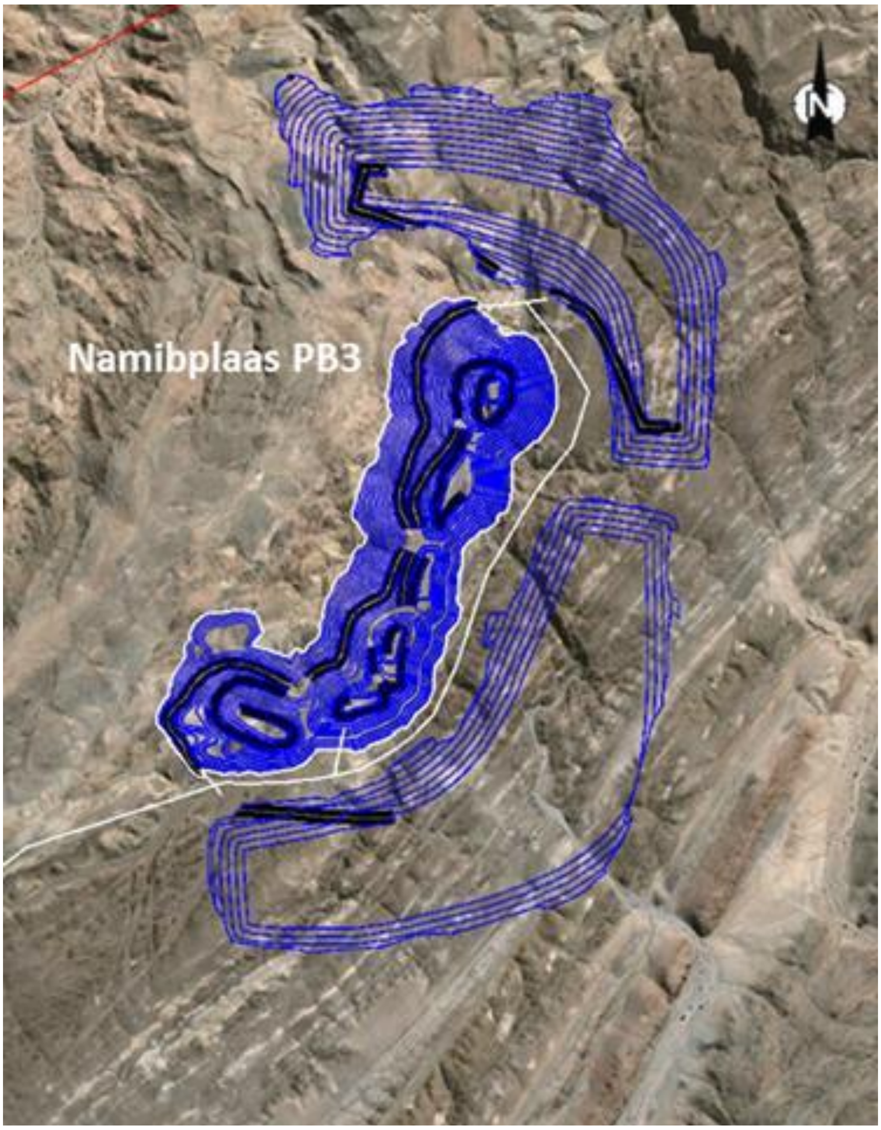


Figure 18.2 Proposed Namibplaas waste dump locations around the pit.

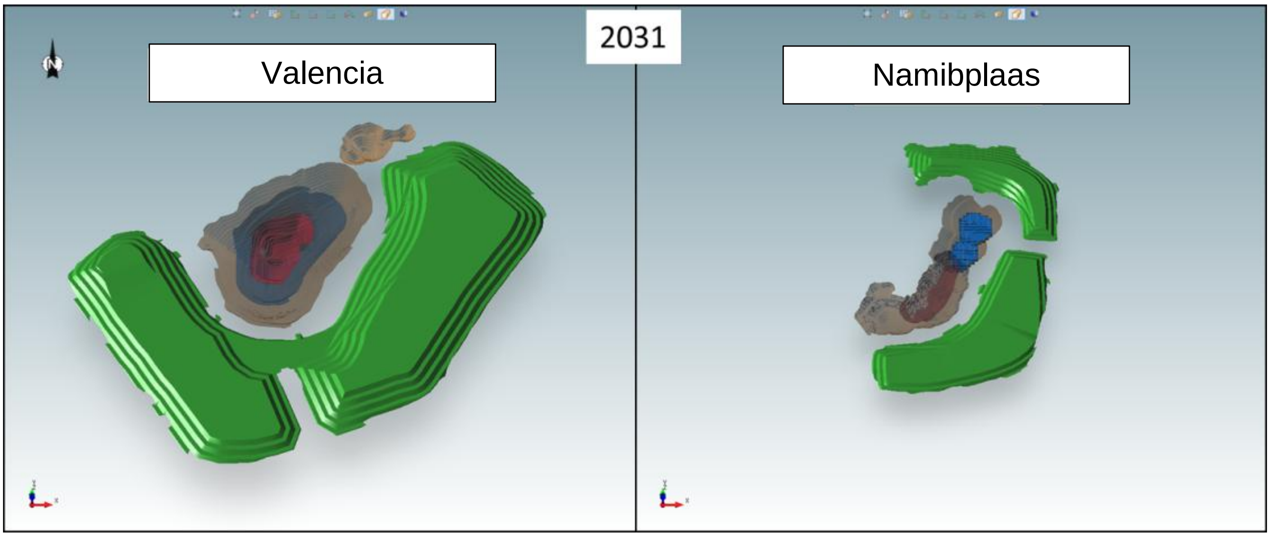


Figure 18.3 Oblique view of 3D schematic pit and dump layout at end of mine.

18.3.1 Open Pit Backfilling of Waste

At this stage, no allowance has been made to progressively backfill pits, or portions thereof, as the orebodies remain open at depth. Unless it is absolutely certain that a pit will not expend beyond a certain depth or laterally, will backfilling of pits be considered. This then becomes an operational decision that will be taken during operations.

The advantages of pit backfilling are:

- dumps are potentially smaller, in footprint and / or in height;
- truck hauling distances may be reduced if the backfilling area is adjacent to current mining.

Due to the ore deposit and pit configuration, backfilling potential is likely to be limited to the following:

- The Valencia satellite pit is mined out early and the adjacent main pit could have its waste hauled to backfill instead of onto the dump.
- The Namibplaas pit is mined progressively from the south to the north. Waste material mined in the latter years could be hauled to the southern pit area for backfilling. The need to preserve pit ramps could limit the extent to which backfilling can take place.

The total backfilling potential is not likely to exceed 20% of the overall waste tonnage.

19. MARKET STUDIES AND CONTRACTS

19.1 PRODUCT SPECIFICATIONS

The processed product from the Norasa Project will be uranium oxide (U_3O_8), known as 'yellow cake', contained in standard drums each holding up to 450kg of U_3O_8 depending on the density of the final product. Yellow cake is inert and mildly radioactive, emitting alpha radiation which is absorbed by the drum. It is non-toxic and would be dangerous to humans only if ingested in quantity. A range of regulations governs the transport of the drums, including Namibian and international transportation regulations.

19.2 PRODUCT SHIPPING AND CONVERSION

19.2.1 Shipping

The drums of processed yellowcake will be packed into sea containers at the mine site and transported by road to the port of Walvis Bay. Drums of yellow cake have been exported from Namibia through Walvis Bay for approximately 38 years, the material being sourced from Rössing and, in recent years, also from the Langer Heinrich operation.

Specialist shipping agents exist for yellow cake and other nuclear materials, located in Europe and the USA. Consistent with standard practice, Forsys expects to pay for all shipping and transport to the conversion facility, and then for the weighing, sampling and assaying at the converter.

19.2.2 Conversion

The drums of yellow cake will be shipped to one of three or four established conversion facilities throughout the world, with the primary ones located in France (Areva / Comurhex), US (Honeywell / Converdyn) and Canada (Cameco / Port Hope / Blind River). At the conversion facility, the U_3O_8 is converted into a gas (uranium hexafluoride, UF6), placed in canisters and either stored, sold or shipped to an enrichment facility.

Title to the yellow cake typically passes from the producer to the buyer upon delivery to the conversion facility. The producer receives a credit to its metal account at the conversion facility for the majority of the delivered quantity soon after delivery, with the balance determined after weighing, sampling and assaying. Sale of the final determined quantity of uranium occurs in accordance with the producer’s relevant sales contracts.

All conversion facilities have pre-set specifications for yellow cake. Before signing up with a particular conversion facility, sample quantities will be sent to each conversion facility for analysis and acceptance. Ultimately a contract will be negotiated between the producer and each of the conversion facilities utilised. The contract covers the procedures for weighing, sampling and assaying of the yellow cake, and the terms for storage, as well as the details of surcharges for impurities. There is typically a free storage period with additional charges for longer term storage.

Testwork carried out on the Norasa ore to date does not indicate that the final yellow cake product will contain above-standard levels of impurities which would typically attract penalty surcharges at the relevant conversion facilities.

19.3 SALES AND MARKETING

19.3.1 Sales and Marketing Strategy

Forsys expects to form an in-house sales and marketing function to administer the Norasa Project's uranium sales arrangements and revenues. This function will be supported by specialist uranium marketing groups as required.

Cost allowances for in-house and external marketing services have been allowed in the operating cost estimates for the Project.

The yellow cake sold from the Project will be sold under a mix of spot (short term sales and delivery), medium term (1-2 years to delivery) and long term (3+ years to delivery) sales contracts. Initial marketing efforts are expected to involve the negotiation of sales contracts with 'ramp up' features allowing for some flexibility in the development timetable as production and sales volumes increase with the establishment of stable operations.

The buyers of the U_3O_8 product from the Norasa Project will largely comprise nuclear power utilities in various nations which generate power using nuclear facilities including China, South Korea, USA, Japan, France, UAE, Saudi Arabia, UK, Finland, Sweden, Spain and Russia. In addition to nuclear power utilities, sales are expected to occur to nuclear fuel brokers and potentially other producers seeking to build inventories for their own contractual obligations or investment purposes.

19.3.2 Sales and Marketing Costs

Estimated sales-related costs incorporated in the DFS total \$0.61/lb U_3O_8, comprising freight, shipping, insurance, sales and marketing and an allowance for conversion impurities.

19.4 URANIUM DEMAND AND SUPPLY FORECASTS

Extensive studies and analyses of the global nuclear power and uranium markets are frequently published by industry analysts and capital markets institutions. The following subsections provide an overview of recent views regarding the global uranium market and associated price forecasts.

19.4.1 Uranium Market

Uranium oxide is used, primarily, in the generation of electricity within nuclear power facilities. Based on data from the World Nuclear Association, total uranium consumption in 2014 was approximately 145Mlb U_3O_8 and total uranium production was approximately 132Mlb U_3O_8 (2013). The supply deficit is presently filled from secondary supplies including the sale of US Government inventories.

On 1 January 2015 the World Nuclear Association reported there are 437 nuclear power plants operating worldwide, with 70 nuclear reactors under construction, 183 reactors planned worldwide and 311 reactors proposed and those in operation currently produce 15% of the world's electricity generation. The low operating cost of nuclear power generation and the increasing concern for the environment and climate change are driving a nuclear renaissance. With the only significant commercial use for uranium being fuel for nuclear reactors, it may follow that the nuclear renaissance will have a significant influence on future uranium demand and price.

Following the natural disasters in Japan in March 2011 and the resultant operating issues with the Fukushima Daiichi nuclear power facility, uranium spot and long-term contract prices weakened and remained depressed to the date of this report. However, the clean nature of nuclear power for base load electricity generation remains a key alternative and growth area for the world's industrialised and fast-developing nations. This fact is expected by numerous analysts to drive higher future uranium prices.

19.4.2 Uranium Price Forecasts

Spot prices and long-term contract prices were approximately $36/lb and $49/lb respectively, at the end of 2014. Various banking institutions and broking firms prepare periodic forecasts of future uranium spot and long term contract prices, which have been used by Forsys in establishing its price expectations.

Forecast spot prices from the above sources over the next 5 years presently range from approximately $60 up to $70/lb U_3O_8 and the range of forecast long term contract prices is slightly higher.

The economic assessment within the DFS utilises a base case uranium price, stated in (real) December 2014 dollars, of $65/lb U_3O_8. Sensitivity analyses have been run at various prices either side of the base case price.

19.5 CONTRACTS

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

The next stage of the Project requires Forsys board approval and obtaining of finance for Project development, at which point it will become necessary to negotiate a number of fundamental agreements and contracts, including:

- EPCM contract for Project construction, including early engineering activities.
- Uranium sales contracts (short, medium and long-term).
- Supply contracts with NamPower and NamWater for provision of power and water to site.
- Supply and service contracts are expected for major reagent supplies, in particular sulphuric acid and the various reagents for the SX process.
- Fuel supply contract that will include supply and maintenance of fuel storage and dispensing equipment.

The particulars of the relevant contracts will be prepared as and when the Project is developed.

20. ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY PROJECTS

The first integrated EIA/EMP report prepared by Digby Wells and Associates (Pty) Limited (Johannesburg) assessed probable impacts associated with all construction, operation, decommissioning and closure activities related to the Valencia Uranium (Pty) Limited (Valencia Uranium or the Company) project. It also incorporated recommendations for the mitigation, monitoring and management aspects for the construction of the mine, related infrastructure, operations/processing and land rehabilitation.

The EIA/EMP report was submitted to the Ministry of Environment and Tourism in April 2008 and Valencia was granted Environmental Clearance for the project to proceed on 4 June 2008.

Subsequently National Environmental Health Consultants applied for a new clearance, under updated legislation - Environmental Management Act, 2007, which became effective only in early 2012. The Clearance was renewed by the Environmental Commissioner until April 2016, and is renewable for 3-year periods.

The 2008 EIA/EMP is currently being expanded by National Environmental Health Consultants to include the Namibplaas Project, resulting in a single Clearance for Norasa Uranium.

A description of some of the key issues is outlined below.

20.1 CLIMATE

Climatic conditions affect long- and short-term patterns of plant and animal distribution, as well as the pollution dispersion. Consequently, climatic conditions have an important influence on potential impacts resulting from mining.

Namibia's climate is defined as hyper-arid, arid or semi-arid and the Country is classified as the second most arid country in Africa. Namibia has a stable climate which is temperate and subtropical climate characterised by hot and dry conditions with little rainfall along the coast.

The long-term regional rainfall varies from 50 to 100mm, with rainfall steadily decreasing westwards. Rainfall records recorded at Rössing (situated 35km south-west of the Norasa Project) indicate a mean annual rainfall of between 30 and 35mm. The upper reaches of the Khan River catchment, by comparison, receive an average of 400mm per annum.

Rain falls mainly in late summer and autumn, with only very light showers occasionally recorded during the winter time. Rainfall is highly variable during summer months, and generally occurs in the form of heavy thunderstorms of short duration. Flash floods in the Khan River, and even smaller drainage channels, occur frequently and can be extremely powerful forces of erosion.

Inland evaporation rates in the Namib Desert are extremely high with the gross annual potential evaporation of approximately 3,150mm.

The mean maximum temperature for the hottest month (February) is approximately 32°C and mean minimum temperatures for the coldest month (August) are between 10-12°C. The diurnal temperature range is considerable with a maximum of 39°C and a minimum of 6°C recorded over a 2-year monitoring period.

Winds in the Namib Desert are influenced mainly be two high-pressure systems: the Sub-continental high and the South Atlantic high. The strongest winds are north-easterly or east winds, which may blow for up to 50 days a year, between April to September, and are strong, dry, hot winds with monthly average wind speeds of 27 km/hour. Hourly wind speed averages can reach up to 43 km/hour.

Strong south-westerly and north-easterly winds have important implications for the erosion of stockpiles, spoil heaps and tailings as well as the transport of pollutants from the mine and processing sites. Wind speed and wind direction must therefore be considered for the effective management and monitoring of environmental plans for Norasa.

20.2 AIR QUALITY

Baseline Air Quality Survey – Dust was conducted during 2012 and will be included in the new updated combined EIA/EMP Report. Baseline dust levels are generally low due to the pebbly/rocky nature of the surface topography. Increased fugitive dust levels, containing inhalable and respirable dust particles will probably occur with possible dispersion of radionuclides. Dust suppression will be implemented on all haul roads, material transfer points, crushers and in the pit. The tailings dump will be progressively covered with waste rock to minimise windblown dust.

20.3 HYDROLOGY

The site is characterised by three non-perennial tributaries of the Khan River. Some of these drainage lines will be affected by the proposed mining activities, including the establishment of the open pit, plant infrastructure and tailings dam.

The precipitation in the area is characterised by late summer and autumn rainfall thunderstorms of high intensity and short duration. With such rainfall characteristics, it is anticipated that storm water management on site will be of importance, not only to ensure the safety of people at the mine, but also prevent discharge of polluted water from containment facilities due to a lack of capacity.

20.4 GEOHYDROLOGY

Consultant to re-evaluate the inclusion of Namibplaas to existing Valencia Uranium Project was appointed and will form part of the updated EIA/EMP - Norasa Uranium Project. A hydrocensus was done in a 30km radius around the site. There are numerous groundwater abstraction points, for which use, yield, quality and depth information has been recorded. Most uses are for low volumes such as domestic use and cattle/game watering. Rössing Uranium Mine is the exception and abstracts 600m^3/day from the Khan River aquifer. Modelling in the area where Norasa could extract water from the Khan aquifer shows inflows in the region of 1,348 m^3/day.

Groundwater is found in a fractured rock aquifer and the Khan River primary aquifer. All the aquifers have high levels of dissolved salts (predominantly Cl ions) which signify that the groundwater at the Norasa Site is of a poor quality and not suitable for human consumption without treatment.

The fractured aquifer has an extremely low permeability and storativity and thus a low water yield. Recharge is low due to the arid nature of the area in which the mine is located.

Due to the low yields from local aquifers it is planned to use desalinated sea water during operation. A new 30km pipeline will be needed from Rössing's reservoir for this purpose. Thus, during operation, groundwater abstraction rates will be low and limited to site de-watering activities.

The proposed pit will lower the water table in the fractured rock aquifer as it descends below the water table. It is likely that the pit will remain a water sink after closure due to the high evaporation rate. The impact of this dewatering will be limited by the low permeability of the surrounding rock.

Water quality impacts from the plant, pit and tailings areas should be limited due to the low recharge rate, the fact that tailings are placed dry, and because the pit acts as a water sink. Management to contain chemicals and process fluids is required.

20.5 FLORA AND FAUNA

A consultant to re-evaluate the inclusion of Namibplaas to existing Valencia Uranium Project was appointed and will form part of the updated EIA/EMP - Norasa Uranium Project. The hyper-arid conditions support a sensitive ecosystem. Vegetation of the Norasa area of interest falls within the semi-desert and savannah transition zone. Annuals are the dominating plants in the middle and driest part of the Namib Desert with stipagrostis occurring widely on the Aeolian plains east of the Site. A number of endemic and near-endemic species have been recorded on site, including aloe, commiphora and the elephant's foot.

A total of 21 mammal species are known to occur within the mine site and surrounding area. A total of 31 bird species were observed and at least 76 species of reptiles are known, reported and/or expected to occur in the area.

Conservation and protection fauna and flora in the surrounding area during construction and operation has begun by establishing a sanctuary area. Fauna and Flora monitoring include the annual monitoring of vegetation on affected sites, surrounding areas and in the established sanctuary. The Khan River valley vegetation will also be included in the monitoring programme to determine if there are any effects from the abstraction of groundwater.

20.6 SOCIAL ENVIRONMENT

The Erongo Region is sparsely populated, particularly in the rural areas. Low intensity farming and the Rössing Mine are the only significant sources of employment in the vicinity of Norasa. The availability of water is regarded as a key potential constraint to development in the area. In a more regional context, other mining operations, tourism and fishing industries in Swakopmund and Walvis Bay also generate a significant number of jobs.

There will be a positive impact due to employment and business creation opportunities during the construction and operational phases, as well as potential skills transfer to Namibians. Potential negative impacts may arise during decommissioning and closure due to loss of employment and retrenchment. Development of employment and business opportunities for the directly affected communities will help minimise or avoid negative impacts associated with mining through social management plans and monitoring.

20.7 ENVIRONMENTAL LIABILITY AND CLOSURE PLANNING

Based on the proposed mine development, initial consideration has been given to the decommissioning and closure of the mine and the associated closure objectives.

Waste dumps will be re-profiled and tailings facilities will be rehabilitated using a capping of waste rock. These would then be covered with pre-stripped ‘soils’ containing seeds. Appropriate storm water control measures would be established to minimise soil erosion.

Rehabilitation will include the dismantling and removal of infrastructure (including decontamination of soils, etc.) taking cognisance of potential radiation issues. The time-period for post-closure monitoring and reporting is still to be determined.

The value of financial provision required for mine closure will be updated annually and contributions to the fund adjusted accordingly. The mine plans to progressively rehabilitate areas as work is completed. This will include the early closure of the Valencia satellite pit. Portions of the tailings dump will also be profiled and capped as filling is completed. The rehab is expected to commence, and run continuously, from year 4 of operations.

20.8 WASTE HANDLING

Waste will be characterised and sorted at source according to type, generation rate and disposal methods. Adequate receptacles will be provided at the source of generation. All waste will be handled in accordance with its class (hazardous or general) and all personnel collecting, handling, transporting or disposing of waste will be trained in the correct procedures for dealing with the respective waste types. Waste will be contained in appropriately labelled containers (skips, bins, drums) that will specify the waste class. The containers will be appropriately designed to store liquid, solid, hazardous or general waste and different waste types or classes will not be mixed.

Waste will be collected from the generation source and taken to a central sorting and storage yard. Here waste will be further sorted into the respective class and type and allocated for further re-use, recycling or disposal. Should it not be possible to separate hazardous and general waste, the entire load will be classed as hazardous.

Hazardous waste storage areas will be clearly demarcated and marked and will have the necessary precautionary measures.

20.9 LEGAL, POLICY AND PERMITTING REQUIREMENTS

A legal, policy and permitting requirement assessment was completed as part of the EIA/EMP, focusing on the requirements of Namibian legislation.

In addition to local legislative requirements, the Norasa Uranium aims to implement an effective Environmental Management System in compliance with international best practice.

A list of some of the International Best Practice Standards and Guidelines include:

- International Finance Corporation Policy on Social and Environmental Sustainability;
- World Bank Health and Safety Guidelines;
- Equator Principles;
- World Bank Group Air Quality Standards (1998);
- International Council on Mining and Metals: Sustainable Development Framework;
- International Atomic Energy Agency, Vienna: 2000: Monitoring and Surveillance of the Residue from the Mining and Milling of Uranium and Thorium.

20.10 HEALTH AND SAFETY CONSIDERATIONS

Due to the nature of the project and the relevant health and safety considerations associated with uranium mining, health and safety considerations are currently being addressed prior to implementation of the project by National Environmental Health Consultants and will form part of the new updated EIA/EMP - Norasa Uranium Project.

21. CAPITAL AND OPERATING COSTS

21.1 INTRODUCTION

The project consists of the development of a new uranium mine and associated infrastructure, approximately 80 km from the coastal town of Swakopmund, Namibia.

Capital and operating Costs have been determined by:

- Forsys: mining associated infrastructure; equipment OEM costs, power, water, owner's infrastructure.
- VBKom: scheduled mining capital and operating costs.
- AMEC: process plant and related infrastructure costs.

All costs are quoted in US$ as of December 31, 2014.

Equipment is quoted in the currency of source of quote and converted to US$ as per currency of this estimate. Generally, bulk material costs are quoted in N$ or ZAR and similarly converted to US$ in the estimate. Where budget prices were obtained in currencies other than US$, the following exchange rates in Table 21.1 have been used.

Table 21.1 Exchange Rates

Currencies	Rate	Source
US Dollar to Namibian Dollar	US$1.00 = N$11.00	Forsys
US Dollar to South African Rand	US$1.00 = ZAR11.00	Forsys
US Dollar to Australian Dollar	US$1.00 = A$1.25	Forsys
US Dollar to Euro	US$1.00 = EUR0.80	Quoted rates

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

21.2 TOTAL CAPITAL COST

Major project infrastructure capital is based on previous costs derived from quotes and budget estimates obtained directly from suppliers and service providers. This includes power supply and the site substation, bulk water supply pipeline and tailings disposal systems. Provision has been made for inflation of these items from the original estimates; formal quotes based on new investigations will be sought for the feasibility study. System optimisations will also be evaluated during the next phase of work. The construction of the main access road was completed in mid-2010, but allowance has been made for further upgrades for construction.

The initial mining period will consist of establishing pit working benches and mining face. This work will also be used to supply rock / aggregate for various construction activities. Although a substantial amount of high grade ore is available on and near surface, a short 'pre-strip' period will be required to ensure that sufficient pit face is established to meet the rapidly increasing plant feed requirements.

The overall initial construction capital and mining equipment build-up is summarised in Table 21.2. A value is also provided for the initial operating costs.

Table 21.2 Total Project Capital Costs	
Capital Items	**Est. Cost US$/M**
Process plant (and first fill)	257.8
Mining equipment	94.0
Tailings disposal	5.6
Bulk water supply	20.8
Power supply & substation	17.7
Owner's & other outlays	37.0
Project Capital	**432.8**
Sustaining capital (life-of-mine)	139.4
Radiometric Sorting Upgrade	39.9

The initial project capital costs do not include an allowance for a Rad Sort facility which has been estimated to cost $39.9 million and is scheduled to be implemented in year 3 of production. The initial design of the crushing circuit takes consideration of retrofitting radiometric sorting. Radiometric sorting would utilise spare capacity in the primary crusher to allow feeding at a higher feed rate and rejecting a portion of the primary crushed feed on the basis of low radiometric counts on individual particles. The net effect is to maintain a similar feed rate to the secondary crushing and downstream circuits, but at a higher head grade due to the rejection of barren feed material.

The initial mining costs (pre-strip period) have been capitalised in Owners Costs.

21.2.1 Process Plant Capital Cost

The capital cost estimate for an 11.2 Mt/a process plant and site infrastructure was developed based on an engineering, procurement and construction management (EPCM) contracting strategy. The plant design is based on tank leaching with final treatment via an ion exchange and solvent extraction plant, through to final product packaging.

The plant area includes: crushing, milling, screening, leaching, filtration, ion exchange, solvent extraction, product drying and packaging, acid storage, various reagents, plant infrastructure, spares, first fills, commissioning and temporary facilities as defined in the scope of work set out herein. The estimate excludes the residue stacking system including movable conveyors and radial stacker.

All costs are presented in United States dollars (US$) as at 31 December 2014.

The estimate for the Class 3 feasibility study (FS) estimate scope of work is: US$257.8M (including growth allowance but excluding contingency).

The capital cost estimate is structured to encompass the following major categories.

- Direct capital costs include expenditures incurred during the implementation of the project for the construction of the process plant and infrastructure complete with all associated plant and equipment. The costs include permanent materials and equipment, freight to site, contractors' construction labour, equipment, supervision, overheads and profit, temporary construction facilities, construction mobile equipment, accommodation of construction labour and contractor mobilisation and demobilisation.

- Indirect costs are those expenditures covering temporary construction facilities, engineering, procurement and construction management (EPCM) services and owner's costs.
- Growth is the increase in quantities and cost that may occur as design develops from the early stages of a study when the estimate is prepared based on preliminary design quantity forecasts, to construction completion when final quantities become available. These allowances included in an estimate are generally categorised as:
  - Design allowance
  - Quantity allowance
  - Cost growth allowance.

Assessment is made of these allowances for inclusion in the estimate based on the engineering maturity and knowledge at the time when the estimate quantities were developed. The total growth allowance was 7.9%, or $18.6m.

The estimated total costs are summarised in Table 21.3.

Table 21.3 Total Costs Summarised by Area

Area Description	Total Cost (US$M)	Area Description	Total Cost (US$M)
Buildings	3.3	Hydrogen Peroxide	0.3
Common Area	12.3	Ferrous Sulphate	0.4
Crushing	23.2	Flocculant and Coagulant	1.3
Milling	42.5	Emergency Generators and	
Leaching	27.4	Diesel Handling	1.6
Filtration	24.2	Organic Make-Up	0.2
Residue Stacking	0.2	Sodium Carbonate	0.1
Ion Exchange	27.6	Sodium Hydroxide	0.8
Solvent Extraction	12.0	Ammonia	0.2
ADU Precipitation and Thickening	3.8	Mobilisation and Demobilisation	2.9
ADU Washing and Calcining	9.2	Temporary Facilities	1.4
Product Packing	3.1	Heavy Lift Cranage	0.7
Water Systems	7.1	Commissioning Assistance	0.5
Air Systems	1.1	First Fills and Consumables	10.0
Sulphuric Acid	2.0	Spares	5.4
Lime	1.0	Vendor Representatives	1.0
		Direct Costs – Subtotals	**226.8**
		EPCM	25.5
		Direct Costs and Indirect Costs (EPCM) – Subtotals	**252.3**
		Contingency	Excluded
		Deferred First Fill	5.5
		Total	**257.8**

The estimate is presented in US$ million and has a base date of 31 December 2014.

The estimate has been prepared in accordance with and conforms to the requirements for a Class 3 definitive feasibility study (DFS) estimate in terms of Amec's estimating procedures. The targeted accuracy of this FS estimate is ±15%. For a detailed discussion of the build-up of the process plant capital costs refer to AMEC Report No. 060551-0000-DW00-STY-0001 Rev 0 (Section 8).

The main estimating activities undertaken to achieve the targeted accuracy include the following:

- level of engineering appropriate for a Class 3 estimate
- multiple and/or quotes sourced for equipment and bulk materials supply
- detailed material take-offs (MTOs) prepared for most bulk materials
- labour rates based on information received from contractors and industry agreements
- labour productivity calculations based on information from contractors currently active in the region
- indirect construction costs, including temporary facilities and construction support, calculated in detail.

21.2.1.1 Estimate Basis

The estimate has been assembled in conformance with a formal work breakdown structure comprising cost categories, plant areas, facilities and work disciplines.

The estimate quantities have been based on the Feasibility Study designs detailed elsewhere in this report including:

- Process flow diagrams (PFDs).
- Latest mechanical equipment list – Rev D.
- Latest electrical equipment list.
- General arrangement layout drawings.
- Material take-offs for earthworks, concrete works, structural steelwork, platework and electrical installation disciplines.
- The piping and instrumentation disciplines have been factored based on historical information specifically related to similar uranium process plants.
- Buildings as per general arrangement drawings. Cost based on building areas.
- Temporary services and facilities based on information from Forsys, including making use of existing exploration camp in the mine lease area.
- Project and Owners contingency is excluded from this estimate.

21.2.2 Mine Capital Cost Estimate

Mining capital cost estimates were estimated primarily by Forsys with VBKom providing the mining schedule on which to base the mining equipment schedule. Associated infrastructure includes offices, change house, stores (and store laydown areas), the heavy equipment workshop, waste handling facilities, fuel station and scrap metal yard. The main workshop consists of bays for maintenance, service, welding and tyres, as well as oil stores and air and water services. Local fuel suppliers have agreed to provide fuel storage and dispensing equipment free of charge as part of a supply contract and will install and maintain all equipment while the mine will prepare the site for installation with earthworks and civil engineering.

It is assumed that a full blasting service will be contracted to a local service provider. Part of the site establishment cost will include the explosives storage and handling facilities. The site will contain three 100t overhead bulk storage silo and three 300 case accessories magazines. Mobile equipment will be provided by the service provider.

Mining capital cost estimates include $94M in preproduction capital and $132M in sustaining capital (including a $11.5M resale credit at the end of life). A list of equipment and estimated capital acquisition costs is detailed in Table 21.4.

Table 21.4 Mining Equipment and Costs		
Equipment type	**Capacity / Rating**	**Cost year 1&2 (US$)**
Haul Truck	180 t	$ 42.0
Hydraulic Excavator	400 t	$ 21.9
Blasthole Rigs	171 mm holes	$ 7.2
Secondary Drill	Trim holes	$ 0.7
Wheel Loader	12 m^3	$ 5.0
Track Dozer	60 t	$ 4.5
Grader	200 kW	$ 1.8
Water Bowser	55 m^3	$ 1.3
Diesel Bowser	35 m^3	$ 1.4
Wheel Dozer	50 t	$ 1.2
Support and Service Equipment	Sum	$ 3.6
Workshop, Offices & Infrastructure	Sum	$ 3.3
Total		$ 94.0

21.2.2.1 Mobile Mining Equipment

The primary mobile mining equipment used in the operations includes haul trucks, excavators, production drills and secondary drills. Support equipment consists of tracked and tyred dozers, graders, utility excavator, water and fuel bowsers and rehandle front end loader. The equipment sizing was based firstly on the need to maintain a selective mining capability near ore / waste transition zones. This dictated the size of the primary excavator and all other equipment was sized to match. A schedule of fleet build-up and replacement was compiled in accordance with the proposed production schedule over the life-of-mine.

Requests for equipment supply proposals were sent to the Namibian offices of major mining equipment manufacturers. Cost estimates for the major mobile equipment (including specified additional options) include all manufacture, shipment, assembly, and commissioning costs. Excluded from the quotation were any local taxes. Costs of the equipment were quoted in South African Rands or Namibian Dollars, but all foreign currency components and exchange rates were noted. Proposals were assessed for completeness and competitiveness by Forsys and reviewed by VBKom. At least two suppliers for all primary mining equipment were received and assessed.

The purchase price for large support equipment was based on one or two quotes for which includes low-bed truck and trailer, mobile crane, tyre handler, service and lube trucks and forklift. Minor equipment costs were partly obtained by quotes or database estimates. Small equipment costs were estimated.

21.2.2.2 Site Infrastructure

As part of the enquiry process, market prices for key consumables were requested from Namibian and South African suppliers. Enquiry documents were assembled for mine tyres, diesel and lubricants, and explosives.

Based on mine plan requirements, surface roads within the mine for the initial production period were estimated. These estimates were based on the design width of surface roads. Large fill volumes will be made up from initial waste mining material; an abundant volume of road dressing / wear course is available at Norasa, at the site of the future tailings dump. A cost for road establishment is included in the mining infrastructure costs. An allowance has been included for the road network establishment required for operations. Additional road construction and maintenance activities required (after the mine operation is established) are covered in the operating costs.

Site infrastructure required by mining operations includes offices, ablutions, workshops, wash bays, warehouses, fuel farm, lube storage, explosives plants and magazines. Costs for the equipping and furnishing of offices, store rooms and workshop are estimated.

21.2.2.3 Capitalised Mine Operating Costs

Detailed engineering costs include an allowance for the completion of works associated with the design of the open pit and completion of supply contracts make up part of the capitalized operating costs.

The remainder of the capitalised operating costs for the mine include initial production prior to plant start-up and initial staffing.

21.2.3 Tailings Dump

The tailings dump and disposal facility is described in Section 18.2. Establishment of the dump requires:

- clearing of the initial dump area by dozing and storing gravels and calcrete for future rehabilitation,
- starter ramp for the conveyor, which includes bringing initial blasted mine waste and then surfaced with compacted gravel and calcrete obtained from the tails dump area,
- up- and downstream water diversion,
- lined storm water dam.

A conveyor and stacker system is designed to handle tailings at a rate of 1,650t/hr and is estimated in Table 21.5.

Table 21.5 Tailings dump establishment

Expense Areas	Cost (US$)
Engineering	$ 0.32
Mechanicals	$ 1.74
Structural	$ 2.10
Electrical Control & Instrumentation	$ 0.33
Civils	$ 0.11
Shipping, Construct & Commission	$ 0.28
Ground Preparatory Works	$ 0.70
Total	**$ 5.58**

21.2.4 Owner's Capital Cost Estimate

The owner's cost estimate takes account several initial cost items related to the owner's project team during construction and items required for commencement of operations. The operational items cover the costs until first production. Included in the Owner's Costs are:

- Namibian office support,
- initial recruitment and training,

- construction camp (which will become the operations Village),
- operation of the camp during the construction period,
- light vehicle used for construction and then operations, including busses,
- technical services equipment and software (supplement current items)
- office equipment and IT (supplement current items),
- mine dispatch system, including equipment performance monitoring,
- mine pre-strip, and
- sundry expenses.

Table 21.6 Owner's capital costs

Operational items	Cost (US$)
Namibia office	$ 1.8
Recruitment and training	$ 1.4
Camp construction	$ 7.5
Camp operation	$ 5.0
Vehicles	$ 1.0
Technical services	$ 0.9
Office equip & IT	$ 0.7
Mine dispatch system	$ 2.0
Mine pre-strip	$ 15.9
Sundry overheads	$ 0.8
Total	**$ 37.0**

21.2.5 Sustaining Capital Costs

There is limited sustaining capital required for the plant and infrastructure, the majority being required for build-up and replacement of the mining fleet and other mobile equipment (Table 21.7).

Table 21.7 Total Sustaining Capital Costs

Expense Areas	Cost (US$)
Mining and Plant Equipment	132.1
Service Infrastructure	0.7
Additions to Tailings Management Facilities	2.4
Replacement Workshop Equipment & Vehicles	3.2
Replacement Owner Technical Equipment	1.0
Total	**$ 139.4**

21.2.6 Closure and Rehabilitation Capital Costs

A detailed closure plan was initially developed for the Valencia Project. This is now being updated as part of the amended Norasa Environmental Management Plan, due for submission later in 2015 for Environmental Clearance. High-level consideration has been given to the closure requirements. Forsys intends to set aside a total of $20M for this purpose.

Given the desert environment, scant flora and fauna and poor quality of the existing groundwater, combined with the low acid and metal generating potential of run-off and seepage, this closure cost is considered reasonable.

It is also noted that Norasa is planning a program of continuous rehabilitation of mining areas that can be closed. Starting in year 4, Norasa plans to begin closing areas including the Valencia satellite pit and portions of the tailings dump. Over the course of the mine life, an expenditure of US$6.5M is planned.

21.3 OPERATING COSTS

Operating costs were determined based on estimates from first principles and contractor / supplier estimates, and are considered to be estimated to ±15% with effective date 31 December 2014. All costs are presented in United States dollars (US$).

The life-of-mine average cost per run-of-mine (ROM) tonne is US$13.10/t or $34.72/lb. These average unit costs are summarized by major cost items in Table 21.8. The highest unit cost is the mining cost, largely attributed to a stripping ratio of 3:2.

Table 21.8 Summary of Total Operating Costs

Item	Cost ($/t of ore Yr. 1-5)	Cost ($/t of ore LoM)	Cost ($/lb U_3O_8 Yr. 1-5)	Cost ($/lb U_3O_8 LoM)	% of Cost
Mining	5.67	6.35	14.65	16.83	48%
Processing and Infrastructure	6.45	6.14	16.67	16.27	47%
Owners Outlays	0.64	0.61	1.65	1.63	5%
Total	12.77	13.10	32.96	34.72	

The operating costs for LoM and first 5 years of production include the introduction of a Rad Sort facility. While radiometric sorting is not included in the base case flowsheet, provision has been made and it will be implemented in year 3 of production. The initial design of the crushing circuit takes consideration of retrofitting radiometric sorting. Radiometric sorting would utilise spare capacity in the primary crusher to allow feeding at a higher feed rate and rejecting a portion of the primary crushed feed on the basis of low radiometric counts on individual particles. The net effect is to maintain a similar feed rate to the secondary crushing and downstream circuits, but at a higher head grade due to the rejection of barren and low grade feed material. The additional cost to include Rad Sort is estimated at $0.40 per tonne of ore crushed which equates to an additional cost of $0.70/lb of U_3O_8 produced for the first 5 years and $0.95/lb over LoM.

If Rad Sort was not introduced, the operating costs would increase to $36.50 for LoM and to $34.55 for the first 5 years. The impact on Capex, Opex and NPV is noted in Table 21.9.

Table 21.9 Comparison of Rad Sort to Non Rad Sort

Metric	Rad Sort	Non Rad Sort	Variance
Capex ($M) - Construction	432.8	424.0	-9.6
Capex ($M) - LoM	612.1	570.9	-42

Opex - first 5 years ($/lb U_3O_8)	32.96	34.55	1.59
Opex - LoM ($/lb U_3O_8)	34.72	36.50	1.78
NPV - 8%DR pre-tax ($M)	626.1	528.0	-68.8
NPV - 8%DR post-tax ($M)	383.4	329.0	-42.9

21.3.1 Process Plant

The average life-of-mine (LOM) process operating cost has been estimated for the design ore type assaying 200 ppm U_3O_8, based on testwork data, material costs and rates derived from data supplied by manufacturers, contractors and consultants.

21.3.1.1 Process Operating Cost Summary

The process operating costs for the Norasa Project generally reflect the first 2 years of operation at a head grade of 200 ppm U_3O_8 and a throughput of 11.2 Mt/a. The accuracy of the operating cost estimates is ±15% and reflects the plant operating at design capacity.

A breakdown of the process plant operating costs is illustrated in Table 21.10 including details of total annual cost, cost per tonne of ore crushed and cost per pound of U_3O_8 produced (assuming a head grade of 200 ppm U_3O_8).

Table 21.10 Summary of Process Operating Costs

Item	US$/a	US$/t ore	US$/lb U_3O_8
Reagents	32 965 828	2.94	7.31
Power	19 203 698	1.71	4.26
Consumables (inc. plant air and water)	17 890 537	1.60	3.97
Labour	4 885 454	0.44	1.08
Maintenance	6 264 774	0.56	1.39
Miscellaneous	1 210 000	0.11	0.27
Total	**82 420 291**	**7.36**	**18.28**

For a detailed discussion of the build-up of the process plant operating costs refer to AMEC Report No. 060551-0000-DW00-STY-0001 Rev 0 (Section 7).

21.3.2 Mine Operating Costs

21.3.2.1 Summary

The mining study considers that mining equipment will be purchased, controlled and maintained by the owner. Tyre management, explosives supply and blasting service and fuel supply contracts are included in the study. The scope of works for the explosives contract includes a bulk plant, magazines, and transport and manufacture of bulk explosives. This is referred to as a full blasting service. Fuel supply includes on-site depot for fuel and lubes, transport and management. The supplier will also provide, install and maintain the fuel storage and dispensing equipment as part of the supply contract.

Table 21.11 Summary of mining operating costs by activity		
Details	**Total Cost ($/t mined)**	**Total Cost ($/lb U_3O_8)**
Loading	0.16	1.74
Hauling	0.53	5.92
Drilling	0.16	1.81
Blasting	0.29	3.27
Secondary equipment	0.08	0.89
Support equipment	0.01	0.10
Technical services & maintenance	0.11	1.18
Total labour	0.17	1.87
Sub-Total	1.51	16.78
Rom pad rehandling	0.02	0.22
Total	**1.53**	**17.00**

21.3.2.2 Major Consumables

The fuel price was determined from a proposal submission, assuming a 5 year supply contract for at least 1.2M litres monthly. Applicable rebates, taxes, tariffs, duties, and bulk purchase discounts were applied. A diesel price of $0.80/L was adopted for the Study. Lubricant costs were part of the proposal.

Explosives costs were based on proposals for two contract options; down-hole service and a full blasting contract. Blasting costs were based on a bulk emulsion (maximise fragmentation without excessive heave). Bulk explosives prices are based on constituent costs of ammonium nitrate and fuel costs. An average bulk explosives price of US$792/t (delivered) was adopted for the Study. A full blasting service by a local contractor is proposed.

Local mining labour and mining administration labour costs were based on current market related costs to companies for the Namibian mining industry. An industry remuneration survey conducted by PwC was used as a guide for defining labour rates. The costs of non-Namibian labour is based on current foreign rates accounting for on-costs and adjusted to US$ at the DFS exchange rates.

Tyre costs are based on submissions from a tyre supplier, who can also provide a complete tyre management system for operations. The service is planned in the Study.

There is no electrical power planned for the pit operations. Any power required within the pit will be provided through diesel generation for items such as lighting in loading areas, high traffic areas and pit water pumping. Minimal power is required for the mining operation within the workshops and office areas. There is about 200kW of installed power at the mining offices and workshop.

21.3.2.3 Other Consumables

The cost of ground engaging tools (GET) and wear items (buckets, bowls, tracks, etc) are based on OEM proposals and suggested part life. Workshop consumables were estimated.

21.3.2.4 Equipment Ownership Costs and Operating Assumptions

Ownership costs (less insurance) are included as equipment capital costs.

The following operating assumptions were used for equipment:

- Loading parameters, including fragmentation, material density, moisture content and material swell.
- Geological information with rock and rock mass properties.
- Equipment availability based on industry standard figures.
- Bucket fill factors based on industry experience and local parameters.
- Excavator load cycle times based on performance data and industry standards.
- Truck cycle times based on an industry standard computer mine truck modelling package (TALPAC). Haul route were defined for each ore block mined.
- Truck hauling parameters (rolling resistance, speeds, and spot/dump times) based on industry standards.
- Fuel burn rates as supplied by published industry figures for specified working conditions.

21.3.2.5 Drill and Blast

Drill and blast costs were developed by VBKom and Forsys. Blasting consultants and explosives suppliers derived powder factors based on the results of the Kuz-Ram fragmentation analysis, geotechnical assessment, crusher requirements and the mine's selective mining philosophy. Drill production rates are 23m/hr for 171mm production hole and 25m/hr for 140mm trim hole and are consistent with industry standards.

21.3.3 Mining Management

The costs of mine administration include mine supervision, technical services and administration. The majority costs for grade control are carried as capital items, with the operating costs covering the manning and the ongoing operating costs of gamma logging.

21.3.4 Support Services

Support services includes any non-technical support, such as security, medical, IT, training, human resources, finance, administration, and commercial. Costs for operating these departments have been estimated.

Within this area have been included items such as the following:

- The road maintenance contractor who will establish and maintain the sealed roads (bitumen or polymer based solutions. The contractor will be responsible for his own equipment and not be reliant on the mining fleet for support.
- Concurrent rehabilitation will commence from year 4 and continue through the life of mine. Although the activities will be overseen by the HSE department, the work will include the use of mining equipment and the costs will be reflected as a rehab cost.
- Pit dewatering take place in 2 areas; direct water diversion within the pit, to a sump and pumped out (or into the water truck for dust suppression) and dewatering from boreholes adjacent to the pit.

21.3.5 Tailings Operation

The main activities associated with the tailings operation are the conveyor / stacker system and dozing of tailings over the dump edge and levelling. Ground preparation for the expanding dump is also required on an ongoing basis as only a portion of the initial dump area was cleared in the initial start-up period. The cost breakdown is provided in Table 21.12. Note that the application of radiometric sorting will reduce the cost per pound as a portion of the plant feed is rejected after the primary crusher.

Table 21.12 Summary of tailings operating cost		
Details	**Total Cost ($/t ore)**	**Total Cost ($/lb U_3O_8)**
Conveyor maintenance	0.02	0.05
Power	0.03	0.08
Dozing, clearing, ground prep	0.07	0.19
Labour	0.02	0.05
Total	**0.14**	**0.37**

21.3.6 Owner's Operating Costs

Owner's operating costs consists of site administration and the Village operation. The Village is operated by a services contractor as described in Section 18.1.5. The cost of operating the busses has also been included in the Village costs. This includes transport on site and to the nearest town at the end of employee's shift cycles. The costs estimated by Forsys total $6.9M annually, equivalent to $1.32/lb U_3O_8 produced, as summarised in Table 21.13.

Table 21.13 Summary of owners annual costs		
Details	**Total Cost US$M /a**	**US$/lb U_3O_8**
General administration	4.7	0.90
Village	2.2	0.42
Total	**6.9**	**1.32**

22. ECONOMIC ANALYSIS

22.1 INTRODUCTION

This section describes the financial model developed by Forsys Metals.

Model inputs have been derived from the mining and plant feed schedules, metallurgical parameters and capital and operating costs identified earlier in this report. These have been reviewed by AMEC and are in accordance with their relevant sections.

22.2 FINANCIAL MODEL INPUTS AND ASSUMPTIONS

The financial model has been created in Excel. Mining and processing data, and capital and operating cost estimates have been inserted into the financial model to enable the calculation of an internal rate of return (IRR) and a net present value (NPV) based on the indicative production and cash flow forecasts.

22.2.1 Basis of Financial Model

The scope of the financial model has been restricted to the project level excluding the effects of financing. The model reports both pre-tax and post-tax outcomes. The financial model outputs reflect the results of the project at the Forsys Namibia level allowing for an appropriate level of allocated administrative and corporate costs from the various ownership entities.

The financial model reflects the equity cash flows of the Norasa Project without any debt financing. The sensitivity analysis has been undertaken on a pre-tax and pro-tax basis. All revenue and cost estimates are expressed in US$ and are based on real December 2014 quarter values. Accordingly, no inflation assumption has been incorporated.

The key assumptions incorporated into the financial model for the DFS analysis are described in further detail as follows.

22.2.2 Production Physicals

The calculation of annual uranium oxide output is based on the mining and processing schedules which set out the appropriate parameters for these activities. Only Proven and Probable Mineral Reserves have been considered. The financial model allows for the variation in all key assumptions including mining rate, waste/ore stripping ratios, ore grades and metallurgical recovery (base estimated at 91.3%).

Annual production is summarised in Figure 15.12. Further detail was available in the relevant worksheets of the financial model.

22.2.3 Working Capital

A working capital build-up and delay between production and cash revenue receipts of 45 days has been assumed and a delay in payment for COGS of inventory and other accounts payable of 60 days and 45 days respectively.

22.2.4 Revenue

For the Base Case, uranium output is sold at a long-term contract price of $65/lb of U_3O_8 (Section 1.8). Sensitivities have also been run at different price assumptions. Net revenue has been calculated after deducting royalties and an allowance of $0.61/lb for the estimated marketing, freight and conversion-related costs prior to sale at the relevant conversion facility.

22.2.5 Royalties and Export Levy

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

22.2.6 Taxes

An overview of the fiscal system in Namibia, outlining the principal taxes and duties expected to be payable by the project, is as follows. Taxation of the parent company, and/or individual investors is not considered in this overview.

The rate of corporate income tax payable by mining companies is 37.5%, payable on taxable profits with a capital deductions regime allowing the deduction of pre-production and other capital expenditure over a three year period.

A royalty of 3.0% of gross sales is expected to be applicable. Value Added Tax (VAT) may be chargeable on sales and paid on purchases within Namibia. Where applicable, the VAT rate is 15%, although certain items are zero rated for VAT. Uranium produced by the Project will be exported, and will therefore not be subject to VAT.

22.2.7 Foreign Exchange Rates

Capital and operating items in foreign currency were converted to US$ using assumed long term exchange rates based on available economic research from various investment and banking institutions including Rand Merchant Bank and Investec. The base case assumption is that US$1.00 equals A$1.25; N$11.00; ZAR11.00 and €0.80.

22.2.8 Rad Sort

The operating costs for LoM and first 5 years of production include the introduction of a Rad Sort facility. While radiometric sorting is not included in the base case flowsheet, provision has been made and it will be implemented in year 3 of production. The initial design of the crushing circuit takes consideration of retrofitting radiometric sorting. Radiometric sorting would utilise spare capacity in the primary crusher to allow feeding at a higher feed rate and rejecting a portion of the primary crushed feed on the basis of low radiometric counts on individual particles. The net effect is to maintain a similar feed rate to the secondary crushing and downstream circuits, but at a higher head grade due to the rejection of barren feed material. The additional cost to include Rad Sort is estimated at $0.40 per tonne of ore crushed which equates to an additional cost of $0.70/lb of U_3O_8 produced.

22.2.9 Operating Costs

Operating costs have been estimated for each of the key functions of the project, and are detailed in the financial model.

Operating costs include all on-site costs and related overheads. As noted above, costs associated with the marketing, freighting and conversion of final product are modelled as deductions from revenue in accordance with industry and accounting practice.

22.2.10 Capital Costs

Capital costs are set out on the capital costs worksheet of the financial model. Each of the key capital cost components is set out in detail within this worksheet.

The financial model for the purposes of this report does calculate a pro-forma accounting profit and, as a result, there is non-cash depreciation or depletion calculations for capital expenditure.

The cash operating surplus comprises net revenue less annual operating costs. Estimates of annual net cash flow are derived after deducting capital expenditure and allowances for working capital from the relevant period's cash operating surplus.

22.2.11 Financial Parameters

Net Present Value (NPV)
Project NPVs are calculated on both annual pre-tax and post-tax net cash flows. 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 annual discount rate of 10% has been assumed.

Internal Rate of Return (IRR)
The various IRRs for the project are calculated using the annual pre-tax and post-tax net cash flows.

Payback Period
The payback period is defined as the period of time in which the cumulative undiscounted pre-tax or post-tax net cash flows ultimately becomes positive. At this point, the project will have paid back the initial development and working capital costs.

22.3 FINANCIAL MODEL OUTCOMES

The base case cash flow is shown in Table 22.1. In calculating the potential returns from the project, the fundamental assumptions shown in Table 22.2 have been made.

Table 22.1 Project Cash Flow Projections ($M)

	Total / Avg	Year -1	Year 0	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
Mining, Milling and Production (Mt)																			
Ore	206.1	2.0	11.6	13.7	9.5	17.4	13.8	15.6	13.2	17.7	15.7	16.8	14.4	14.5	12.5	10.8	6.9		
Waste	660.5	6.3	25.5	34.6	43.4	41.0	60.1	58.4	60.4	57.3	49.0	37.9	40.7	40.0	43.6	45.8	16.6		
Total material mined	**866.7**	**8.3**	**37.1**	**48.4**	**52.9**	**58.4**	**73.9**	**73.9**	**73.6**	**75.0**	**64.7**	**54.8**	**55.1**	**54.4**	**56.1**	**56.6**	**23.6**		
Feed grade (U3O8 ppm)	238			218	225	220	241	240	244	252	246	241	233	238	240	220	229	321	
Strip ratio	3.2		3.1	2.2	2.5	4.6	2.4	4.4	3.8	4.6	3.2	3.1	2.3	2.8	2.8				
Recovery	92.4%			91.8%	92.0%	91.9%	92.5%	92.5%	92.6%	92.9%	92.7%	92.5%	92.3%	92.4%	92.5%	91.9%	92.2%	94.9%	
U3O8 M/lb	77.8		-	4.6	5.2	5.0	5.5	5.5	5.6	5.8	5.7	5.5	5.3	5.4	5.5	5.0	4.2	3.9	-
Cash Flows (US$/m)																			
Revenue																			
Sales	$5,056.8			$298.7	$335.5	$325.5	$360.4	$357.9	$364.3	$377.0	$369.2	$358.9	$346.3	$354.2	$358.5	$325.5	$270.0	$255.0	
Royalties and conversion costs	$104.2			$6.2	$6.9	$6.7	$7.4	$7.4	$7.5	$7.8	$7.6	$7.4	$7.1	$7.3	$7.4	$6.7	$5.6	$5.3	
Net revenue	**$4,952.5**			**$292.6**	**$328.6**	**$318.8**	**$352.9**	**$350.5**	**$356.8**	**$369.2**	**$361.6**	**$351.5**	**$339.1**	**$346.9**	**$351.1**	**$318.8**	**$264.4**	**$249.8**	
Operting Expenditure																			
Mining	$1,309.0			$52.0	$67.2	$75.7	$82.5	$100.8	$105.9	$109.9	$112.1	$99.8	$92.6	$90.5	$96.1	$90.6	$87.1	$46.3	
Processing	$1,265.7			$77.2	$83.9	$89.6	$89.9	$89.6	$89.6	$89.6	$89.9	$89.6	$89.6	$89.6	$89.9	$89.6	$71.2	$46.6	
Owners costs	$194.0			$12.2	$12.8	$13.0	$13.4	$13.3	$13.4	$13.5	$13.4	$13.3	$13.2	$13.3	$13.3	$13.0	$11.9	$10.9	
Working capital	$0.0			-$3.6	-$1.9	-$5.3	$2.2	-$5.5	-$0.7	$0.4	-$1.6	$2.4	$0.5	$1.6	-$1.2	-$2.3	-$0.3	$17.2	-$1.8
Total operating expenditure	**$2,768.6**			**$137.8**	**$162.0**	**$173.1**	**$187.9**	**$198.3**	**$208.2**	**$213.4**	**$213.7**	**$205.1**	**$196.0**	**$194.9**	**$198.2**	**$191.0**	**$169.9**	**$120.9**	**-$1.8**
Operating cash flow before tax	$2,183.9			$154.7	$166.6	$145.7	$165.0	$152.2	$148.5	$155.8	$147.8	$146.4	$143.2	$151.9	$153.0	$127.8	$94.5	$128.8	$1.8
Tax	$564.2						$4.4	$52.1	$46.0	$55.6	$51.4	$53.8	$50.9	$56.1	$56.8	$47.0	$35.3	$54.7	
Operating cash flow after tax	**$1,619.7**			**$154.7**	**$166.6**	**$145.7**	**$160.7**	**$100.1**	**$102.5**	**$100.2**	**$96.5**	**$92.7**	**$92.3**	**$95.8**	**$96.1**	**$80.8**	**$59.2**	**$74.1**	**$1.8**
Capital Expenditure																			
Process Plant	$257.8	$12.6	$211.9	$33.3															
Tailings	$5.6	$0.2	$5.0	$0.4															
Mining fleet	$94.0		$54.0	$40.0															
Infrastructure	$38.5	$9.0	$29.5																
Owners costs	$37.0	$5.6	$30.2	$1.2															
Expansion / Sustaining capital	$179.3				$78.5	$4.6	$40.2	$7.8	$25.1	$7.9	$9.2	$5.4	$7.9	$3.8	$0.2	$0.1			-$11.5
Total capital expenditure	**$612.1**	**$27.4**	**$330.6**	**$74.8**	**$78.5**	**$4.6**	**$40.2**	**$7.8**	**$25.1**	**$7.9**	**$9.2**	**$5.4**	**$7.9**	**$3.8**	**$0.2**	**$0.1**			**-$11.5**
Net Cash Flow	**$1,007.6**	**-$27.4**	**-$330.6**	**$79.9**	**$88.1**	**$141.1**	**$120.5**	**$92.3**	**$77.4**	**$92.3**	**$87.2**	**$87.2**	**$84.3**	**$92.0**	**$95.9**	**$80.7**	**$59.2**	**$74.1**	**$13.3**
Cumulative Cash Flow Forecast		-$27.4	-$358.0	-$278.1	-$190.0	-$48.9	$71.6	$163.9	$241.3	$333.6	$420.8	$508.1	$592.4	$684.4	$780.3	$861.0	$920.2	$994.3	$1,007.6

Table 22.2 Fundamental Assumptions and Parameters of Financial Modelling Analysis	
Basis	Project level, pre and post-tax and excluding any debt financing
U_3O_8 prices	Long term contract price of \$65/lb U_3O_8
Development period (Months)	24
Mine life (Years)	15.0
Annual throughput	11.2 Mt
Fuel price	\$0.80/L, including freight
Sulphuric acid price	\$127/t delivered to site
Raw water cost	\$3.00/m3
Power cost	\$0.0955/kWh
Production rate	Between approximately 4.5 to 5.8 Mlb of U_3O_8 per year
Exchange rates	US\$1.00 : A\$1.25 : N\$11.05 : R11.05 : €0.80 :C\$1.10
Tax and royalty rates	37.5% and 3%

The key outputs from the financial model based on the above assumptions are reported for the first 5 years of the modelled operation and for the life of mine in Table 22.3.

Table 22.3 Key Financial Model Outputs		
	Project	**US\$/Share**
Project Economics		
NPV at a Discount Rate of 8% (\$M) - (Excl. Tax)	623.2	622.6
- (Incl. Tax)	383.9	383.4
Internal Rate of Return (%) - (Excl. Tax)	32%	32%
- (Incl. Tax)	26%	26%
Payback Period from Start of Production (years)	4.4	4.4
Production	**Life of Mine**	**First 5 Years**
Quantity Ore Treated (Mt)	206.1	66.7
Recoveries (%)	92.4%	92.2%
Uranium sold (Kg U_3O_8)	35,288	11,710
Uranium sold (Mlb U_3O_8)	77.8	25.8
Revenue and Cash Flow		
Average U_3O_8Base Price (\$/lb U_3O_8)	65	65
Net Revenue (\$M)	5,056.8	1,678.0
Operating cash flow (\$M)	1,751.1	440.2
Net cash flow after tax (\$M)	1,007.6	161.5
Operating Unit Costs (\$/lb produced)		
Mining	16.83	14.65
Processing	16.27	16.67
Owners costs	1.63	1.65
Total Operating Costs (\$/lb produced)	**34.72**	**34.72**

Based on the above, at a throughput rate of 11.2Mtpa, the Project is modelled to produce between 5.0 to 5.8Mlb U_3O_8 per year. The average cash operating cost in the first 5 years is estimated at \$32.96/lb U_3O_8 and over the life of mine is estimated at \$34.72/lb U_3O_8.

22.4 FINANCIAL MODEL SENSITIVITY ANALYSIS

Sensitivity analyses have been undertaken on key parameters within the financial model to assess the impact of changes upon project pre-tax and post-tax cash flows, NPV and IRR.

In assessing the sensitivity of the project returns, each of the parameters has been varied independently of the others. Accordingly, combined positive or negative variations in any of these parameters may have a more marked effect on the forecast positive or negative economics of the project than will the individual variations considered.

The convention adopted in this analysis is that negative sensitivities are adjustments that reduce project economics or value (for example, increased capital or operating costs) and, correspondingly, positive sensitivities are adjustments that improve project economics and value.

Table 22.4 presents the results of the sensitivity analysis.

Table 22.4 Sensitivity Analysis (Pre and Post-tax Basis)

Parameter/ Variation	**Value**		**Pre-tax NPV ($M. DR 8%)**	**Post-tax NPV ($M. DR 8%)**
U_3O_8 Price	**U_3O_8 Price ($/lb)**			
-23%	50.00		70.1	17.7
-15%	55.00		254.2	143.5
-8%	60.00		438.4	264.6
Base	65.00		622.6	383.4
8%	70.00		806.7	500.6
15%	75.00		990.9	617.4
23%	80.00		1175.0	733.3
Capital Costs	**Capital Cost ($M)**			
-30%	562.40		562.6	321.9
-20%	519.20		582.6	342.5
-10%	476.00		602.6	363.0
Base	432.80		622.6	383.4
10%	389.60		642.6	406.9
20%	346.40		662.6	436.9
30%	303.20		682.6	466.9
Operating Costs	**Average Operating Costs ($/lb)**			
	First 5 Years	**Life of Mine**		
-30%	42.85	45.14	203.5	122.2
-20%	39.56	41.66	343.2	211.3
-10%	36.26	38.19	482.9	298.0
Base	32.96	34.72	622.6	383.4
10%	29.67	31.25	762.2	473.2
20%	26.37	27.78	901.9	562.5
30%	23.08	24.30	1041.6	651.0

Figures 22.1 and 22.2 show the sensitivity results on the Project’s NPV and IRR to changes in U_3O_8 prices, capital costs and operating costs in graphical form.

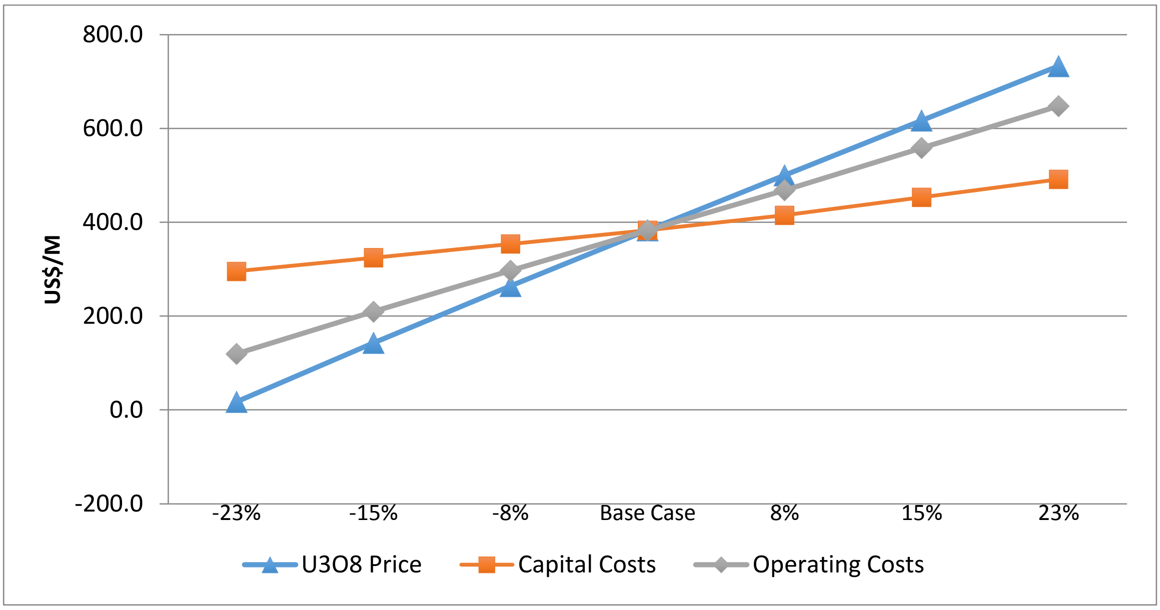


Figure 22.1 Project NPV (Post-tax Basis)

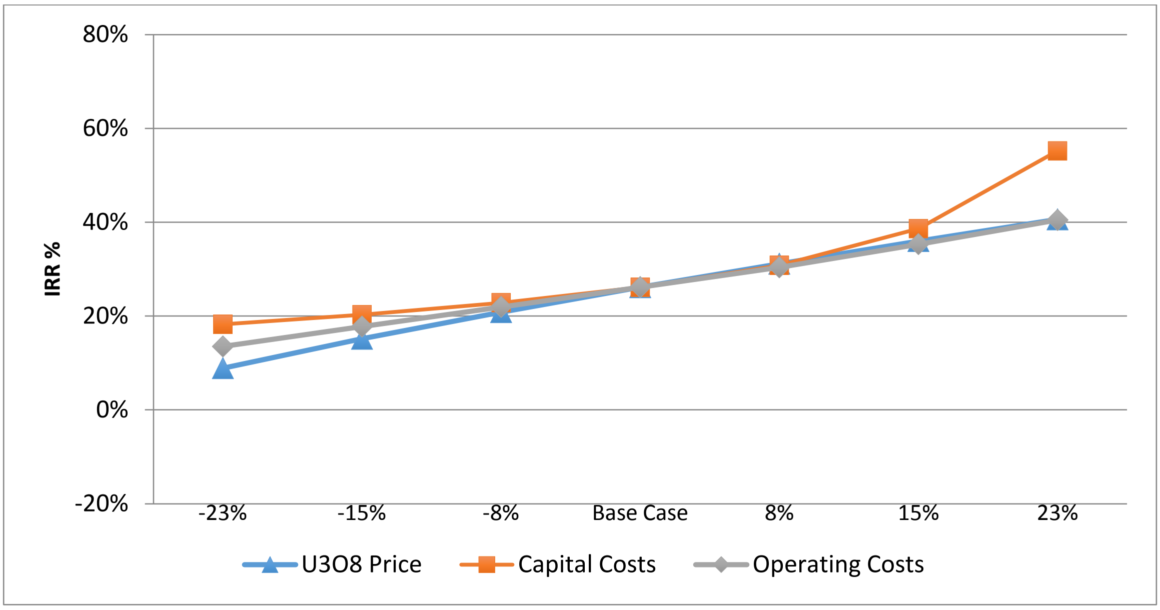


Figure 22.2 Project IRR (Post-tax Basis)

22.4.1 Relative Sensitivities

The financial sensitivity analysis demonstrates that the economic performance of the Norasa Project is most sensitive to changes in the uranium price, followed by operating costs. This is unsurprising given the large scale and relatively modest grade of the deposit. The uranium price has the greatest bearing upon cash operating margins. Each component is discussed briefly below.

Sensitivity to Changes in U_3O_8 Prices

As noted, the Norasa Project is most sensitive to changes in uranium prices. However, negative movements of up to 23% from the base case assumption of $65/lb U_3O_8 result in the pre and post-tax NPV remaining positive. Likewise, positive movements of up to 23% from the base case assumption of

\$65/lb U_3O_8 produce significant changes in the pre-tax NPV from \$623.2M to \$1175.0M and post-tax NPV from \$383.9M to \$733.3M.

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

Sensitivity to Changes in Total Operating Costs
As noted above, given the large annual throughput of the project, the financial performance is also sensitive to changes in total operating costs.

Increases of 10%, 20% and 30% in the base case cost assumptions produce adverse changes in the post-tax NPV’s but in all cases the NPV remains positive.

Likewise, similar cost reductions from the base case assumptions result in the pre-tax and post-tax NPV materially increasing.

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

The Project is least sensitive to the sensitivities in the base case capital cost assumptions. NPV changes at a similar percentage to the nominated percentage changes. For example, capital cost reductions of 10% and 20% from the base case assumptions result in the post-tax NPV increasing from \$383.4M to \$406.9M and \$436.9M respectively.

22.5 CAUTIONARY STATEMENT

The results of the economic analysis are based on forward-looking information that are subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Forward-looking information includes commodity prices and exchange rates; the proposed mine production plan; projected plant head grade and recovery rates; uncertainties and risks regarding the estimated capital and operating costs; uncertainties and risks regarding cost estimates and completion schedule for the proposed Project infrastructure, including the need to obtain additional permits and governmental approvals.

23. ADJACENT PROPERTIES

The information presented below for the various operations was obtained from company websites and company produced information documents. Notice should be taken of the ownership of the various companies described and the reporting obligations due to the nature of that ownership:

- Rössing Uranium's major shareholder (at 69%) is Rio Tinto Group, a dual-listed company traded on both the London Stock Exchange and the Australian Securities Exchange
- Langer Heinrich Uranium was wholly owned by Paladin Energy Ltd as of early 2014. Paladin Energy Ltd is listed on the Australian Securities Exchange, the Toronto Stock Exchange and the Namibian Stock Exchange. On 20 January 2014 Paladin announced its intention to sell a 25% joint-venture equity stake in Langer Heinrich to China Uranium Corporation Limited, a wholly owned subsidiary of China National Nuclear Corporation (CNNC), a Chinese nuclear utility. The transaction had not yet been finalised as of the date of this report.
- The Husab Uranium Mine (Swakop Uranium) is owned by Taurus Minerals Limited of Hong Kong, which has a 90% stake in the company, and the Namibian state-owned company Epangelo (10%). Taurus is an entity owned by China General Nuclear Power Company (CGNPC) Uranium Resources Co Ltd and the China-Africa Development Fund. Until April 2012, Swakop Uranium was 100% owned by Extract Resources, an Australian company listed on the Australian Securities Exchange, the Toronto Stock Exchange.

The most recent information from the Husab project (subsequent to the change of ownership in April 2012) may not conform to any reporting standards and hence should be considered with that in mind. The Qualified Persons of this report state for all project reported in this section that:

- the operational information is all publicly disclosed by the owners or operators of those operations;
- the Qualified Persons have been unable to verify the information;
- the information is not necessarily indicative of the mineralisation on the Valencia property.

23.1 RÖSSING URANIUM MINE

Information on the Rössing Uranium Mine is included as it situated approximately 35km to the southwest of the Norasa Project.

The Rössing uranium deposit occurs in a highly deformed zone in which uraniferous alaskites were intruded into deformed metasedimentary rocks of the Khan and Rössing Formations. The alaskitic rocks range from small quartzo-feldspathic lenses, to large intrusive and replacement bodies varying widely in texture, size and emplacement habit (Roesener and Schreuder, 1992).

Mining is done by blasting, loading and hauling in 180 tonne haul trucks from the main open pit, referred to as the SJ Pit, before the uranium-bearing rock is processed to produce uranium oxide. The open pit currently measures 3km long, 1.5km wide and is 390m deep. Additional satellite orebodies have been identified and are included within the mines ore reserves.

Blasted ore is hauled to surface to be crushed and then ground with rod mills. A combined leaching and oxidation process takes place in an acidic solution in atmospheric leach tanks. Cyclones, rotoscoops and counter current decantation separate the uranium bearing solution from the granular tailings, which is pumped in slurry form to the tailings dam. Continuous ion exchange and solvent extraction purify and concentrate the uranium solution (OK liquor) and the barren solution is returned to the circuit. The uranium is precipitated out of solution as uranium diuranate, thickened and calcined (roasted) into the final product of uranium oxide.

Based on the prevailing business climate, the mine closure has been pushed back to 2023, as proposed in the current life-of-mine plan. The expansion of the current main SJ pit and development of the Z20 satellite deposit gives rise to the JORC compliant Resources and Reserves[3] (Table 23.1 and Table 23.2 respectively).

The Rössing Uranium Mine commenced production in 1976 and since then has produced up to 4,000 tonnes of U_3O_8 per year. Table 23.1 and Table 23.2 provide recent information on the operation until 2012.

Table 23.1 Mineral Resource Estimate – Rössing (end 2012)

Classification	**Mt**	**Grade % U_3O_8**	**Mlbs U_3O_8**
Measured Resources	15	0.026	8.6
Indicated Resources	148	0.024	78.3
Measured & Indicated	**163**	**0.024**	**86.9**
Inferred Resources	173	**0.026**	99.2

Note that resources are quoted exclusive of reserves. Cut-off grade of 0.010% U_3O_8.

Table 23.2 Ore Reserves Estimate – Rössing (end 2012)

Classification	**Mt**	**Grade % U_3O_8**	**Mlbs U_3O_8**
Proved Ore Reserve	29	0.031	19.8
Probable Ore Reserve	102	0.035	78.7
Total Ore Reserve	**131**	**0.034**	**98.5**

Cut-off grade of 0.010% U_3O_8.

23.2 LANGER HEINRICH URANIUM MINE

The Langer Heinrich Mine (LHM) is located on the western side of central Namibia, Southern Africa. It lies 80km east of the major deep-water port at Walvis Bay and the coastal town of Swakopmund.

When production commenced, LHM was the first conventional uranium mining and processing operation to be brought online in over a decade. Paladin was able to deliver the project on schedule and within the original budget of US$92M despite the significant cost pressures experienced by the mining industry during the 20-month construction term.

The mine has subsequently completed two expansions and is currently producing at a rate of 5.2 Mlb/yr.

[3] http://www.riotinto.com/annualreport2012/production_reserves_and_operations/index.html

Uranium mineralisation at Langer Heinrich is associated with the calcretisation of valley-fill fluvial sediments in an extensive tertiary palaeodrainage system. Calcrete is a chemically precipitated limestone that forms under arid to semi-arid climatic conditions.

Uranium mineralisation occurs as carnotite, an oxidised uranium and vanadium secondary mineral. The deposit extends over a 15km length in 7 higher grade pods within a lower grade mineralised envelope. The carnotite occurs as thin films lining cavities and fracture planes and as grain coatings and disseminations in the calcretised sediments. Mineralisation is near surface, 1m to 30m thick and is 50m to 1,100m wide depending on the width of the palaeovalley.

Following calcretisation and uranium deposition, portions of the host sediments were eroded as a result of uplift and rejuvenated river flows. The present day Gawib River has dissected and modified both the calcrete and associated mineralisation. In places, this prevailing ephemeral drainage system has blanketed the deposit with up to 8m of river sands and scree.

These Mineral Resources and Ore Reserves are quoted at a cut-off grade of 0.025% and conform to both the JORC (2004) and NI 43-101 guidelines[4]. The resources and reserves have been depleted for mining to end June 2013. These reserves form the basis of the continuing life-of-mine plan for the Project. The revised mine plan allows a project life in excess of 20 years, based on a processing feed rate of 3.45 Mt/a.

Table 23.3 Mineral Resource Estimate – Langer Heinrich (June 2013)

Classification	Mt	Grade % U_3O_8	t U_3O_8	Mlbs U_3O_8
Measured Resources	25.3	0.055	13,851	30.54
Indicated Resources	70.1	0.055	38,729	85.38
Measured & Indicated	**95.5**	**0.055**	**52,580**	**115.92**
Stockpiles	28.6	0.042	11,932	26.31
Inferred Resources	**17.8**	**0.06**	**10,335**	**22.9**

Note that resources are quoted inclusive of reserves.

Table 23.4 Ore Reserve Estimate – Langer Heinrich (June 2013)

Classification	Mt	Grade % U_3O_8	t U_3O_8	Mlbs U_3O_8
Proved Ore Reserve	20.0	0.055	11,093	24.46
Probable Ore Reserve	59.4	0.057	33,616	74.11
Stockpiles	28.6	0.042	11,932	26.31
Total Ore Reserve	**108.1**	**0.052**	**56,642**	**124.87**

With the uranium being present as a coating on the sediments, it is not necessary to grind the material finer, but only to remove the surface layer from the individual grains. As a consequence, the process employs crushing and scrubbing to break down agglomerates into individual grains and to remove the uranium minerals from the grain surfaces.

[4] http://www.paladinenergy.com.au/

Cyclones and screens are then employed to separate the high grade fines (leach feed) from barren discard material. Typically the barren solids will contain 40-50% of the solids mass but only 5-10% of the uranium in the ROM feed.

After thickening, the leach feed slurry is conditioned with sodium carbonate and bi-carbonate, heated and pumped to the leach circuit. After leaching and heat recuperation, the slurry is fed through a Counter Current Decantation (CCD) circuit in which the high grade uranium solution is removed from the solids. This solution undergoes further clarification before being pumped through both fixed bed and continuous NIMCIX ion exchange vessels where the uranium is recovered onto resin. Uranium is stripped from the resin and precipitated as Sodium Diuranate (SDU) then redissolved using sulphuric acid before being re-precipitated with hydrogen peroxide. This product is dewatered, dried and drummed as UO_4.

23.3 HUSAB URANIUM MINE

The Husab Project is located within the central Damara Orogenic Belt (DOB) in a zone characterised by basement domes, regional folding, faulting, and late Damaran intrusive rocks.[5]

The Husab Project is dominated by a series of north-northeast to northeast trending regional-scale antiforms and synforms, which make up the main structural architecture of the entire Central Zone of the Damara. These meta-sedimentary folds or dome-like structures of the DOB are cored by gneissic and metasedimentary rocks of the Abbabis Formation. The basement rocks are covered to the northeast and south by stranded cover sequences of flat-lying calcrete and alluvial deposits, which are associated with a broad northeast trending valley marginal to the Khan River.

The Husab prospect represents a 15 kilometre target zone, most of which is covered by the Namib Desert (eolian sand and gravels) with the prospective target zone defined by the magnetic trend that can be verified in outcrop and then traced beneath the desert sands. Extract have confirmed the potential of the prospective stratigraphic trend, defined by the magnetic data, to host uraniferous granite (alaskite). Drilling completed to date at Husab has followed a zone of uraniferous alaskites that crop out at the northern end of EPL 3138 (Zone 1) and trend southwards under cover for a distance of at least nine kilometres. The mineralised alaskites are associated with calc-silicate, metasediments, gneiss and biotite schist lithologies of the Khan, Rössing and Chuos Formations. The Rössing Formation is the dominant host into which the passive uraniferous granites have intruded.

The following Mineral Resources and Ore Reserves[6] are quoted at a cut-off grade of 100 ppm and conform to both the JORC (2004) and NI 43-101 guidelines for a 20 year mine life.

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[5] Extract Resources, 7 June 2011, ASX Media Release, Husab Project Established as the 4th Largest Uranium Deposit in the World.

[6] Extract Resources, 10 August 2011, ASX Media Release, 37% Increase in Reserves at Husab

Table 23.5 Mineral Resource Estimate – Husab (August 2011)			
Classification	**Mt**	**Grade ppm U_3O_8**	**Mlbs U_3O_8**
Measured Resources	74	510	84
Indicated Resources	281	440	274
Measured & Indicated	**355**	**455**	**358**
Inferred Resources	175	340	130

Note that resources are quoted inclusive of reserves.

Table 23.6 Ore Reserves Estimate – Husab (June 2011)			
Classification	**Mt**	**Grade ppm U_3O_8**	**Mlbs U_3O_8**
Proved Ore Reserve	62.7	569	78.7
Probable Ore Reserve	217.3	504	241.2
Total Ore Reserve	**280.0**	**518**	**319.9**

Construction commenced in November 2012 and first production is expected towards the end of 2015. The mine will be an open-pit operation utilising diesel and electric powered shovels to load 327-tonne haul trucks. The haul trucks will use a trolley assist system to haul rock out of the pit. The planned production rate is 15 Mtpa ore treated at a life-of-mine average strip ratio of 6.2.

Proposed comminution will consist of crushing and milling (SAG and ball) to feed atmospheric leach tanks using sulphuric acid and pyrolusite to extract the uranium. Counter current decantation using the NIMCIX technology and conventional solvent is used to separate the solids and the liquids with the tailings slurry pumps to the tailings dam. The uranium rich solution passes through continuous ion-exchange and conventional solvent extraction process upgrading and refinement. The uranium is then precipitated and calcined to produce the final product of uranium oxide.[7]

[7] http://swakopuranium.com/

24. OTHER RELEVANT DATA AND INFORMATION

The Minerals (Prospecting and Mining) Act, 1992 (Act 33 of 1992) of Namibia clearly defines the procedure required for an exploration project to be granted a mining licence and begin construction and operations. The Valencia Project has all of the required permits and agreements in place to begin at any time.

Namibplaas however is still an exploration project and none of the requirements are in place. This feasibility study is the first hurdle as it demonstrates the project viability. A formal application for a mining licence must be made to the Ministry of Mines and Energy (MME) with all the supporting documentation as stipulated. The next step will be to obtain an Environmental Clearance from the Ministry of Environment and Tourism.

The current Namibplaas exploration licence expires in November 2015, giving Forsys sufficient time to compile and lodge a mining licence application with MME.

As the land on which the project is situated is privately held, a compensation agreement must be entered into with the land owner for the loss of use of his land. These negotiations can take several months, or longer.

MME must also grant each prospective mining company approval for their Accessory Works to cover all construction that will take place for the project. This includes all buildings, roads, pipelines, power lines, dumps, pits and any other excavations and structures.

Water use licences will be required for any water discharged, diverted or extracted. This includes pit dewatering.

Although these application processes for Namibplaas can take several months, or longer, these do not impact the commencement of construction or operations as Valencia is fully permitted and the Norasa Project can commence upon funding without delay.

25. INTERPRETATION AND CONCLUSIONS

25.1 PROJECT RISKS

25.1.1 Introduction

A range of economic, engineering and other technical risks to the Project have been considered. Those risks assessed as Low, Low to Moderate, Moderate to High, High or Major are summarised in Table 25.1 in order of magnitude, and are discussed by discipline in the remainder of this section.

Table 25.1 Non-Resource / Mining Economic and Technical Risk Assessment

Item	Assessed Risk to Project
U_3O_8 price	Major
Operating cost overrun – acid	Moderate to high
Operating cost overrun – diesel	Moderate
Mining equipment under-performance	Moderate
Mine operating costs overrun (sustained increase in labour / materials costs)	Moderate
Timely availability of water supply	Moderate
Operating cost over-run	Moderate
Acid leaching under-performance	Moderate
Reserve under performs	Low to Moderate
Scheduling and production time not achieved	Low to Moderate
Adequacy of power supply	Low to Moderate
Qualified personnel availability	Low to Moderate
Ramp-up schedule over-run	Low to Moderate
Capital cost over-run	Low to Moderate
Mobile equipment poor availability	Low
Late completion of mine pioneering works	Low
Failure to achieve plate throughput	Low
Project construction delays	Low
Foreign exchange variation	Low
Permits refused or seriously delayed	Low
Royalty and Tax rate increases	Low

25.1.2 Geological Interpretation

While the reported mineral reserves are considered to be robust, these remain estimates and there are underlying uncertainties relating to interpretation of drill results and the geology, continuity and grade of the mineral deposits. Such risks are typical of all mining projects, and the level of risk is judged to be no more than **Low to Moderate**, given the general continuity in geometry and grade of the deposit.

25.1.3 Mining Risk Assessment

The risk associated with the resource estimate extends into the mining study, in terms of potential inaccuracies in deposit geometry, continuity and grade. These uncertainties will be reduced during grade control drilling prior to mining.

Other mining risks include:

- Late completion of pioneering works, however, the schedule allows for excess ore production in the first year
- Poorer than expected equipment performance and/or availability which would lead to failure to meet the production schedule and increased unit costs
- Sustained unbudgeted increase in labour / materials / consumables costs.

Again, such risks are typical of all mining projects, and the **Low to Moderate** levels of risk is not unusual.

25.1.4 Price of U_3O_8

A long-term contract price of $65/lb U_3O_8 has been utilised in the DFS, which is the lower end of the average analyst current long-term price predictions of $65-70/lb.

Exposure to significantly lower prices for U_3O_8 would be a **Major** risk to the project. Lower than modelled prices for U_3O_8 would reduce modelled operating cash flows and could cause the deferral of a development decision or the suspension of operations.

Conversely, higher than modelled U_3O_8 prices would have a significant positive impact on cash operating margins, as there would be minimal additional costs.

Forsys intends to seek a strategic partnership with an established industry end-user such that specified quantities of future production can be sold at minimum prices consistent with the $65/lb minimum price.

25.1.5 Foreign Exchange Rate Exposure

The perceived risk of exchange rate exposure is considered relatively **Low** due to the fact that the vast majority of capital expenditure is in the SADC countries. A number of banks are predicting a significant improvement in the strength of the US$ over the next few years, especially compared to the Southern African currencies.

25.1.6 Capital Cost Overrun

As for any major mining project, there is a significant risk of capital cost overruns resulting from a range of factors, primarily sudden and unpredicted increases in equipment, materials or labour capital costs. Additional risk lies in uncertainty regarding site geotechnical conditions, although no obvious issues were identified from preliminary examination.

Engineering has been taken to a level appropriate for a DFS, and an accuracy growth allowance of 8.2% made for expected, unidentified additional costs once detailed process engineering has been undertaken. These provisions are based on AMEC's experience with similar project, but there is no certainty that such provisions are sufficient. A further project contingency in the plant and infrastructure capital cost estimate to allow for other unexpected engineering or cost issues has not been included by Forsys in the base case financial model.

No provision has been made for outside contingencies such as abnormal weather impacts or delays, unforeseen environmental or social constraints, schedule impacts from late delivery of critical equipment items, or unforeseen changes in legislation. The absence of a project contingency in the base case financial model increases project risk but the low level of sensitivity as detailed in Section 22.4 keep capital cost overruns to **Low to Moderate**.

25.1.7 Operating cost overrun

Diesel costs are the highest single component of the operating cost constituting 12.8% of the total. Consequently, increases in oil prices will impact significantly on project economics, and, given the volatility of oil prices in recent years, this is considered a **Moderate** risk area.

The base case annual consumption of sulphuric acid is approximately 153,000tpa. At the assumed delivered price of $127/t, this represents the highest process operating cost item, and some 10.6% of total operating costs. Project economics are sensitive to changes in acid price, which constitutes a **Moderate to High** risk to the Project.

The future cost of electricity supplied by NamPower is uncertain. The 2014 price of 9.55c/kWh has been applied to determine operating costs, but there is pressure on NamPower to increase the tariff above CPI for the next 4 to 5 years, which has not been accounted for. Electricity costs account for 7.7% of the total operating cost and this is a **Low to Moderate** risk to the Project.

25.1.8 Process

A summary of the unit operations are: primary and secondary crushing, SAG milling, sulphuric acid leaching, belt filtration, ion exchange (IX) and solvent extraction (SX), ammonium diuranate (ADU) precipitation and thickening, calcining of the ADU cake and packaging of the U_3O_8 product, tailings storage, reagent storage, make-up and distribution, and utilities (water and air) storage and distribution.

A significant amount of metallurgical testwork has been carried out for the base case acid leach option and the results are generally consistent.

Low

Risks were identified relating to the following design aspects and should be considered in more depth during detailed engineering design.

- Additional raw water consumption for dust control in the crushing and stockpile area
- Variable SAG mill throughput as a result of SAG mill feed PSD control
- Residue transport: the sand-sliding tendency of the material noted during the Atterberg tests warrant conveyor design to be reviewed.
- Raw water consumption increase due to higher evaporation losses and seasonal effects: recommendation is to develop a fully integrated dynamic water balance covering the overall operation (process, mining, exploration etc.) considering seasonal variation.

25.1.9 Utility Supply

Adequate and timely supply of water and electricity are fundamental to all activities in the construction and operation of the mine. NamPower and NamWater have a track record of supplying utilities across the country, but specific risks should be considered further by Forsys, since the implications of late or reduced supply could be very significant.

Electricity Supply
The Khan substation has recently been upgraded and expanded. NamPower met the cost of the new substation although a new bay for Norasa will be at the mine’s expense, as will be the cost of the transmission line to the mine. Power distribution to the mine is planned to be a 220kV transmission line as part of a regional expansion and strengthening of the coastal power supply using the Norasa line as

stage one of a ring feed. At an installed capacity of approximately 35MW and a mine draw of about 31MW, two 40 MVA transformers would be installed, one of which would be maintained as a backup unit. It is assumed that the Company would have to carry the cost of establishing the substation. In mid-October 2014, NamPower announced a power purchasing agreement with the Zimbabwe Power Company for the supply of 80MW of electricity for the next 15 years. This supplements its own generation capacity and other imports required to maintain power supply to customers. Standby power generators of up to 10 MVA are being considered, but a decision on the capacity will be taken closer to the time. The generators will be connected to a synchronization and load control panel to operate the generator sets.

A significant risk would be failure of a transformer during commissioning or ramp-up. Generally, arrangements can be made to share and swap spare or extra capacity, but delays would certainly occur. The risk is judged to be **Low to Moderate**.

Water Supply

Norasa has received NamWater's (Namibia's national bulk water utility) assurance of a supply of water during the construction phase of the project. This will require a 31km temporary pipeline extending from the Rössing reservoir to the construction site. Norasa will design and construct this temporary pipeline with a 300 m^3/day capacity required to service the construction camp and for construction activities.

Production from Norasa will require construction of a permanent 31km main pipeline (replacing the temporary line used during mine construction) linking Norasa to the Rössing reservoir. The Company is working with NamWater, who is responsible for the tendering and construction of this water pipeline.

The Namibian government announced in September 2014 that they intend to buy Areva's seawater desalination plant, located north of Swakopmund. Areva is currently operating the plant on a limited scale to provide water to local mining operations through NamWater. The plant was originally designed to produce 20 Mm3/annum of potable water, providing sufficient capacity for the existing mines in the region (requiring less than half this volume) with spare capacity for newcomers on a first-come first-served basis. At this stage, only about half the RO modules have been installed. The overall plant capacity can be expanded.

Most of the water supply infrastructure will require an upgrade to cater for Norasa and the expansion plans of other operations. Norasa has requested a water allocation of 3 Mm^3 annually for its operating requirements. The delivery of water supplies within the proposed timeframe is also of concern given the number of stakeholders in the equation and the importance of an adequate and affordable water supply to the Project. The risk is judged to be **Moderate.**

25.1.10 Regulation

Namibia is very supportive of mining as can be seen from the history of diamond and uranium mining; the Rössing uranium mine has been in continuous operation for over 35 years. The issues of title to land, permitting, licences, access over public land and possible legal challenges to any of title, right to mine or right to access the licensed mining or EPL areas are all regarded as manageable and a **Low** risk.

Permitting

There is currently no reason to believe that the additional permits required to enable development of the Norasa Project will not be obtained in due course, and the level of risk is considered **Low**.

Royalties and Taxes

An amendment in December 2008 to the Act has provided the Minister for Mines and Energy with the effective discretion to set the mineral royalty for all commodities for all mining projects, including nuclear fuels, at any level. The 2006-gazetted Government royalty on nuclear fuels in Namibia is 3%. The risk of changes to royalties (and the corporate tax rate) cannot be discounted in any jurisdiction, but, given Namibia's commitment to development of the mining industry, it is considered **Low**.

25.1.11 Labour and training

Southern Africa, including Namibia, has a long history of mining developments and operations, and there is a good skill base, including in the Erongo Region. However, the development of the Husab Mine and possible expansions at Rössing and Langer Heinrich will put considerable pressure on the pool of skilled and semi-skilled employees. Namibian legislation such as the Affirmative Action (Employment) Act 1998 and anticipated NEEEF (New Equitable Economic Empowerment Framework) initiatives makes this more than simply a financial issue to be solved by importing labour.

The risk of not being able to identify suitably trained personnel in any of the positions from unskilled to senior management is regarded as **Low to Moderate**. Forsys has every intention of contributing to the operation of technical institutions to train semi-skilled and unskilled workers, establishing training regimes and HR policies and processes that negate the potential risks.

Industrial action is a part of the labour landscape in Africa, and so is to be expected from time to time in the life of an operating mine. The democratic governance and comparative political stability of the country are counters to the possibility of long-term, debilitating industrial action.

25.1.12 Schedule delays

Project Execution Schedule Delays
The current schedule has been built up from first principles including standard engineering design times, quoted supplier delivery times, historical installation times and industry standard float. The project area is not prone to excessive adverse weather conditions and is serviced by excellent existing infrastructure; however, the study is unable to predict international resource activity during the procurement and construction period, which can have a significant impact on the supply chain and product delivery times.

The 3 month early engineering period will allow a review of the long lead items list prevalent at the time, which will mitigate some of the risk. The risk of excessive and costly delays to project construction are considered **Low to Moderate.**

Ramp-up Delays
The risk in prolonged ramp-up to full production is considered **Low to Moderate**, and mitigated to some extent since the ramp-up schedule for the process plant of 6 months is not aggressive.

26. RECOMMENDATIONS

26.1 ENVIRONMENTAL

Completion of all environmental and social specialist studies for Namibplaas and compile the updated EIA / EMP Report for Norasa for submission to MET for clearance. The required activities are currently underway and the report should be completed by the time a mining licence application is due to be lodged.

26.2 PROCESS PLANT

The following opportunities have been identified, and warrant further work:

- Testing and evaluation of radiometric or other sorting technology to increase the resource base at a lower cut-off grade or to increase feed grades.
- Although not included for this study, future design development should consider the merits of acoustic mill monitoring, online sizing of the new feed from each reclaim feeder and ensuring a secondary mill load measuring system is allowed for or provided.
- Leach peroxide consumption: The utilisation of peroxide in leach and associated ferrous oxidation circuits is dependent on accurate control and stable operation. During the locked cycle testwork the utilisation was not optimised and the peroxide consumption indicated by the testwork is therefore seen as conservative. Under stable operation it is foreseen that there is an opportunity to reduce the peroxide consumption.
- Reduced SX and ADU precipitation equipment sizes: The higher IX resin loading presents the opportunity of downsizing the subsequent SX and precipitation circuits should this be confirmed by additional testwork.
- Potential moisture reduction of belt filter residue cake using alternate dewatering reagents.
- Further layout and circuit design optimisation during the next stage of development.

27. REFERENCES

Author	Title
Amec, 2010	Engineering Cost Study, Valencia Uranium, Document No. 60210-00000-23-002-001, January 2010.
Amec, 2013	Engineering Cost Study, Valencia Uranium, Document No. 60491-0000-1100-RPT-0001_1, May 2013.
Anderson, H., Nash, C., 1997	Integrated Lithostructural Mapping of the Rossing Area, Namibia using High Resolution Aeromagnetic, 28, 185–191.
Barnes, J.F.H., Hambleton-Jones, B.B., 1978	A review of the Geological and Geochemical Setting of Uraniferous Granites in the Damara Orogen. Unpublished Lecture to the International Atomic Energy Agency Working Group Meeting, Nancy, France.
Basson, I.J. Greenway, G., 2004	The Rössing Uranium Deposit: a product of late-kinematic localization of uraniferous granites in the Central Zone of the Damara Orogen, Namibia. Journal of African Earth Sciences. 38, 413–435.
Berning, J., Cooke, R., Hiemstra, S.A., and Hoffman,1976	The Rössing uranium deposit, South West Africa. Economic Geology. 71, 351-368.
Berning, J. 1986	The Rössing Uranium Deposit, South West Africa/ Namibia. Mineral Deposits of Southern Africa, pg 1819-1832. Vol II. Geol. Soc. S.Afr.
Bertram, N.G.E. 1982a.	Annual Report on prospecting activities for the period ending December 1981. Trekkopje Exploration and Mining Company (Pty) Limited, Prospecting Grant M46/3/449. February, 1982.
Bertram, N.G.E. 1982b.	Report in support of the application for renewal of Prospecting Grant M46/3/449. Trekkopje Exploration and Mining Company (Pty) Limited, Prospecting Grant M46/3/449. 20 July, 1982.
BMO, 2014a	Global Mining Research, Q1/14 Commodity Canvas, Uranium, BMO Capital Markets, 13 January 2014.
BMO, 2014b	Street Consensus Uranium Price, BMO Capital Markets, February 2014.
Brandt, R. (1985)	Preliminary Report on the Stratigraphy of the Damara Sequence and the Geology and Geochemistry of Damaran Granites in an Area between Walvis Bay and Karibib. Communications of the Geological Survey of Namibia 1. 31-43.
Business Report, 2011	Namibia decision sparks confusion, Business Report. 16 May 2011, Retrieved 8 Dec 2011 from http://www.iol.co.za/business/international/namibia-decision-sparks-confusion-1.1069734.
Corner, B. (1981)	'An Interpretation of the Aeromagnetic Data Covering Portions of the Damara Orogenic Belt with Special Reference to the Occurrence of Uraniferous Granite', PhD thesis, University of the Witwatersrand, Johannesburg, South Africa.
Corner, B. 1982	An interpretation of the aeromagnetic data covering portion of the Damara Orogenic Belt with special reference to the occurrence of uraniferous granite. NUCOR PER- 95, 115 pp.
Corner, B. (1983)	An Interpretation of the Aeromagnetic Data Covering the Western Portion of the Damara Orogen in South West Africa/Namibia. Special Publications of the Geological Society of South Africa, 11. pp 339-354
Cuney, 1980a	Preliminary Results on the Petrology and the Fluid Inclusions of the Rössing Uraniferous Alaskites. Trans. Geol. Soc. S. Afr., 83, 39-46.
Cuney, 1980b	Petrography Geochemistry and Fluid Inclusion Studies of the Rössing Alaskites and Surrounding Gneisses. Nancy, Centre de Recherches sur la Geologie de l'uranium.: 58.
Dept. Econ. Affairs. 1988	Application for extension of Prospecting Grant M.46/3/1496: Trekkopje Exploration and Mining. Letter from Department of Economic Affairs to The Chief Geologist, Gold Fields Prospecting, Windhoek. Dated 25 October 1988.

Digby Wells & Associates, 2008	Environmental Impact Assessment (EIA) and Environmental Management Plan (EMP) for the Valencia Uranium Project, April 2008.
Dundee, 2014	Uranium Sector Insights: Supply Crisis Brewing, Dundee Capital Markets, 3 February 2014.
Eberle, D., Andritzky, G., Wackerle, R., 1995	The New Magnetic Data Set of Namibia: Its Contributions to the Understanding of Crustal Evolution and Regional Distribution of Mineralization. Communications of the Geological Survey of Namibia 10, 141–150.
Epangelo Mining Company, 2010	Vision, Mission Statement, Retrieved 8 Dec 2011 from http://www.epangelomining.com/?page_id=47
Epoch Resources, 2008	Design and development methodology associated with the dry tailings dump at Valencia. Project No: 103-005, August 2008
Extract Resources 2011	ASX/ Media Release: Mining License Issued for Husab Uranium Project. Retrieved 7 Dec 2011 from http://clients2.weblink.com.au/clients/extractresources2/article.asp?asx=EXT&view=6569736
Financial C-Bonds Information, 2011	Moody's assigns first-time ratings of Baa3/stable to Namibia, 23 Sep 2011, Retrieved 17 Nov 2011 from http://www.cbonds.info/eng/news/index.phtml/params/id/522467
Forsys Metals 2012	News Release, Updated Namibplaas Resource, Forsys Metals Corp, 27 September 2012
Forsys Metals 2013	News Release, Forsys Provides Updated Mineral Resource Estimate for its Norasa Uranium Project, Forsys Metals Corp, 7 October 2013.
Forsys Metals 2014	Norasa Uranium NI 43-101 Technical Report, 27 March 1014.
Freemantle, G. 2010	Petrographic, Mineralogical and Geochemical Characterisation of Granite-hosted Uranium Deposits of the Central Damara Orogen, Namibia. Ph D Thesis. University of the Witwatersrand, Johannesburg.
Goscombe, B.D., Gray, D. and Hand, M. 2004	Variation in Metamorphic Style along the Northern Margin of the Damara Orogen, Namibia. Journal of Petrology. 45. No. 6. pp 1261-1295.
GRD Minproc, 2009	Final Report for the Comminution Circuit Trade-off Study for the Valencia Uranium Project, Rev B, 5 October 2009.
GRN, 1991a	Renewal application with waiver of work obligation: Prospecting Grant M46/3/1496 (Valencia) – Namib Mines Limited. Letter from Mining Commissioner, Ministry of Mines and Energy, Republic of Namibia, to The Chief Geologist, Gold Fields Namibia, Windhoek. Dated 1 March October 1991.
GRN, 1991b	Application for the extension and reduction and waiver of work obligation of Prospecting Grant M46/3/1496 – Namib Mines Limited. Letter from Mining Commissioner, Ministry of Mines and Energy, Republic of Namibia, to The Chief Geologist, Gold Fields Namibia, Windhoek. Dated 16 May 1991.
GRN, 1992	Promulgation of Minerals (Prospecting and Mining) Act, 1992 (Act 33 of 1992), of the National Assembly. Government Gazette of the Republic of Namibia, 31 December 1992.
GRN, 2007	Promulgation of Environmental Management Act, 2007 (Act 7 of 2007), of the Parliament. Government Gazette of the Republic of Namibia, Notice No. 232, 27 December 2007.
GRN, 2009	Determination of rates of royalties leviable in respect of certain groups of minerals: Minerals (Prospecting and Mining) Act, 1992. Government Gazette of the Republic of Namibia, Notice No. 45, 1 April 2009.
GRN, 2011	The New Equitable Economic Empowerment Framework (NEEEF), Page 8.
GRN, 2014	Statement for the 2014/2015 Budget, Ministry of Finance, Government of Namibia, 19 February 2014.
Hinojosa, D., 2008	Annual Technical Report, Namibplaas, unpublished.
IAASA, 2011	(International Institute for Applied Systems Analysis. Country Briefs: Namibia. Retrieved 2 Dec 2011 from http://www.iiasa.ac.at/Research/POP/pde/briefs/na-geo.html
IMF, 2010	International Monetary Fund. Namibia: 2010 Article IV Consultation—Staff Report; Staff Supplement; Public Information Notice on the Executive Board Discussion. IMF Country Report No. 10/269. Retrieved 6 Dec 2011 from http://www.imf.org/external/pubs/ft/scr/2010/cr10269.pdf
Jacob, R.E., Corner, B., and Brynard, H.J. (1986)	The Regional geological and structural setting of the uraniferous granitic provinces of Southern Africa. Minerals deposits of Southern Africa, 2, 1807 – 1818.

Jacob, R.E. 1974	The radioactive mineralisation in part of the central Damara Belt, South West Africa, and its possible origin. Report PIN-234(BR), Pelindaba, Atomic Energy Board, South Africa.
Jacob, R.E., Snowden, P.A., Bunting, F.J.L., 1983	Geology and Structural Development of the Tumas Basement Dome and its Cover Rocks. In: Miller, R.McG. (Ed.), Evolution of the Damara Orogen of South West Africa. Special Publication of the Geological Society of South Africa 11.
JORC, 2004	Australasian Joint Ore Reserves Committee. Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, December 2004.
Kaira, C, 2011	Namibia Diamond Production Increases 57% to 1.48 Million Carats in 2010. Bloomberg. 9 Feb 2011, Retrieved 7 Dec 2011 from http://www.bloomberg.com/news/2011-02-09/namibia-diamond-production-increases-57-to-1-48-million-carats-in-2010.html
Kinnaird, J.A., Nex, P.A.M. 2007	A review of geological controls on uranium mineralisation in sheeted leucogranites within the Damara Orogen, Namibia. Applied Earth Science, 116, 2
Kröner, A., 1984	Dome Structures and Basement Reactivation in the Pan-African Damaran Belt of Namibia. In: Kröner, A., Greiling, R. (Eds.), Precambrian Tectonics Illustrated. Schweizerbarth,Stuttgart, pp. 191–206.
Kröner, A., Halpern, M., Jacob, R.E., 1978.	Rb–Sr Geochronology in Favour of Polymetamorphism in the Pan-African Damara Belt of Namibia (South West Africa). Geologische Rundschau 67, 688–706.
Labuschagne, A. N. 1979	The Anomaly 26 Uranium Prospect and its Environment – Valencia 122. Trekkopje Exploration and Mining Company (Pty) Limited, Prospecting Grant M46/3/449. January, 1979.
Laine, R. 2008	Comparison Between Uranium Assays From Core Samples Done by Goldfields in the 1970s with Chemistry / Gamma Correlation Established with Recent Diamond Drilling by Valencia. Valencia Uranium (Pty) Ltd. Unpublished, internal note. September 2008.
Laine, R., & Shilongo, E. 2011	Geology of Namibplaas and Mineralisation, Namibia. Internal report, Valencia Uranium (Pty) Ltd. Unpublished.
Miller, R.McG. 1983	The pan-African Damara Orogen of South west Africa/Namibia. Evolution of the Damara Orogen of South West Africa/Namibia.11, 431- 515.
Miller, R. McG. 2008	The Geology of Namibia, Vols. 1 & 2. Geological Survey of Namibia Special Publication.
Mintek, 2008	Laboratory Testwork to support a Definitive Feasibility Study for the Valencia Uranium Project, 12 December 2008
Namibia, 2010	The Constitution of the Republic of Namibia.
Namport,2011	Welcome to the Port of Walvis Bay. Retrieved 2 Dec 2011 from http://www.namport.com.na/content/show.php?m=4
National Planning Commission, 2002	National Planning Commission, Government of Namibia, Namibia – Census Indicators - 2001, 1991. Retrieved 30 Nov 2011 from http://www.npc.gov.na/census/index.htm
NID, IPPR, 2011	Namibian Institute for Democracy and the Institute for Public Policy Research (2011). The Constitution in the 21st Century: Perspectives on the context and future of Namibia's supreme Law.
Nash, C.R. 1971	Metamorphic Petrology of the SJ Area, Swakopmund District, South West Africa. Bulletin of the Precambrian Research Unit, University of Cape Town 9.
Nex, P.A.M., & Kinnaird, J.A., 1995	Granites and their mineralization in the Swakop River area around Goanikontes, Namibia. Communications of the Geological Survey of Namibia. 10.
Nex, P.A.M. 1997	Tectono-Metamorphic Setting and Evolution of Granitic Sheets in the Goanikontes Area, Namibia. Ph.D. Thesis (Unpubl.). National University of Ireland.
Nex, P. A.M., Kinnaird J.A., Oliver G.J.H. 2001	Petrology, geochemistry and uranium mineralisation of post-collisional magmatism around Goanikontes, southern Central Zone, Damaran Orogen, Namibia. Journal of African Earth Sciences, 33, 481–502.
Office of the President, 2004	Republic of Namibia (2004), Namibia Vision 2030: Policy framework for long-term national development (Summary) page 15 and 25
Office of the President, 2007	National Planning Commission (2007). An analysis of the Economic Challenges of Namibia and How the Donor Community should Assist, Country Paper for the International Conference on Development Cooperation with Middle Income Countries (MICs), Madrid, Spain, http://www.npc.gov.na/publications/EconomicChallengesofNamibia.pdf

OMEGA Geo-Consulting, 2008a	Valencia RC-Drilled Zone: Study of 39 Percussion holes, 13 October 2008
OMEGA Geo-Consulting, 2008b	Valencia RC-Drilled Zone: Global Resource Estimates, 18 July 2008
Ontario Securities Commission, 2011	NI 43-101 Standards of Disclosure for Mineral Projects, Form 43-101F1 Technical Report and Related Consequential Amendments, OSC Bulletin Volume 34, Issue 25, June 24, 2011.
Optiro 2011	Technical Report on the Namibplaas Deposit, Namibia – September 2011, Optiro Pty Limited, 17 October 2011.
PWC, 2010	PricewaterhouseCoopers, Namibia: 2010 and beyond. Retrieved on 30th Nov 2011 from http://www.pwc.com/en_NA/na/assets/pdf/pwc_namibia-2010-and-beyond.pdf
Roesener, H. and Schreuder, C.P. 1992	Uranium. The Mineral Resources of Namibia, pg 7.1-1 to 7.1-55. Published by the Ministry of Mines and Energy, and the Geological Survey, Republic of Namibia.
Sawyer, E.W., 1978	Damaran Structural and Metamorphic Geology of an Area Southeast of Walvis Bay, S.W. Africa/Namibia. M.Sc. Thesis (Unpubl.). University of Cape Town.
SGS Lakefield Research, 2008a	Laboratory Testwork to Evaluate the Recovery of Uranium from Ore Sources Originating from the Valencia Deposit. Mineralogical Report No. MET 08/V66, 1 February 2008.
SGS Lakefield Research, 2008b	Uranium Deportment Analyses and General Mineralogical Characterisation of a Composite Ore Sample, its Leach Residues and Twelve Variability Ore Samples from the Valencia Deposit in Namibia. Mineralogical Report No. MIN1207/208, 26 February 2008.
SGS Lakefield Research, 2009	Laboratory Testwork to Support a Comminution Circuit Options Study. Report No. MET 08/460 Rev3, 23 November 2009.
SGS South Africa, 2011a	General Mineralogical Characterisation and Uranium Deportment Analyses of Five Uranium Ore Samples from the Namibplaas Project, Mineralogical Report: Min 0411/084, 20 June 2011.
SGS South Africa, 2011b	X-ray Diffraction Analysis of Fifteen Uranium-bearing Samples from Namibplaas, Mineralogical Report: Min 0411/084, 19 October 2011.
SGS South Africa, 2012	Uranium Deportment Analysis on Six Uranium-ore Leach Residue Samples from the Namibplaas Project, Mineralogical Report: MIN0112/006 Rev1, 27 February 2012.
SGS South Africa, 2013a	Testwork Conducted on Valencia Samples, Metmin Report A22, 1 February 2013.
SGS South Africa, 2013b	Optimisation Testwork on the Namibplaas Composite, Metmin Report No: 12/243, 28 February 2013.
SGS South Africa, 2013c	Leach Testwork Conducted on Valencia and Namibplaas Samples, Metmin Report 13/713, 30 December 2013.
Shilongo, E. 2011	A Stratigraphical, Petrographical and Geochemical Characterization of the Namibplaas Sheeted Leucograntes and Associated Uranium Mineralalization. A thesis research proposal: Degree of master of science in geology(Unpublished). Department of Geology, the University of Namibia.
Smith, D. A. M 1965	The Geology of the area around the Khan and Swakop Rivers in Namibia. Memoir 3, Geological Survey, Department of Mines, South Africa.
Snowden 2008	Valencia Pit Slope Design Guidelines, Project No. 7037, Memorandum, 13 November 2008.
Snowden 2009a	Valencia Uranium (Pty) Ltd: Valencia Project, Namibia; Technical Report, Project 7480, January 2009.
Snowden 2009b	Valencia Uranium (Pty) Ltd: Valencia Project, Namibia; Technical Report, Project 7516, June 2009.
Snowden 2009c	Valencia Uranium (Pty) Ltd. Pre-split Blasting at Valencia, Memorandum, Project: 7516, 23 July 2009.
Snowden 2010	Valencia Uranium (Pty) Ltd: Addendum to June 2009 Technical Report, Project 696, January 2010.
Snowden 2013	Forsys Metals: Valencia Mine Cost Model, Project No. AU4094, October 2013.
UltraSort, 2009	Personal Communication, Data Analysis of Valencia Rock Samples for Radiometric Sorting Efficiency, November 2009.

VBKom, 2014	Norasa Project Pre-Feasibility Study – Mining Technical Report, Report No 2014-103-01, VBKom Namibia Consulting Engineers (Pty) Ltd, March 2014.
Wikipedia, 2011	Demographics of Namibia. Retrieved 30 Nov 2011 from http://en.wikipedia.org/wiki/Demographics_of_Namibia
World Economic Forum, 2011	The Global Competitiveness Report 2011-2012: Country Profile Highlights. Retrieved 7th Dec 2011 from http://www3.weforum.org/docs/WEF_GCR_CountryProfilHighlights_2011-12.pdf
World Nuclear Association, 2012	World Uranium Mining. Retrieved 5 June 2012 from http://www.world-nuclear.org/info/inf23.html
Xstract Mining Consultants, 2013	Valencia East Pit Slope Design Parameters, Project Ref: P1897, 25 October 2013.
Xstract Mining Consultants, 2014	Namibplaas Uranium Geotechnical Assessment, Project Ref: P1897, January 2014.

28. DATE AND SIGNATURES

Certificate of Qualified Person

As a reviewer of the report Technical Report titled Norasa Uranium Project, Definitive Feasibility Study, NI 43-101 Technical Report and dated 16 March 2015, by Forsys Metals Corp., I hereby state:

- My name is Peter Nofal and I am a Technical Director Studies with the firm of Amec Foster Wheeler of Level 7, 197 St Georges Terrace, Perth, WA, 6000, Australia.
- I am a chemical engineer and a Fellow of the AusIMM (207660).
- I am a graduate of the University of the Witwatersrand in South Africa with a BSc (Eng) in 1982. In 1992 I graduated from the University of South Africa with a BCom (Hons), majoring in business economics.
- I have practiced my profession continuously since 1982.
- I am a "qualified person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
- I have not visited the Norasa Uranium Project.
- I am responsible for reviewing and approving all AMEC prepared sections of the Norasa Uranium Report (Sections 13 and 17 and partially responsible for sections 1, 2, 21, 24, 25 and 26 that relate to the process plant and related infrastructure).
- I am independent of Forsys Metals Corp. pursuant to section 1.4 of the Instrument.
- I have not had any involvement with the Norasa Uranium Project property prior to my engagement as a reviewer of the AMEC prepared sections, the results of which form part of the Forsys Report.
- I have read the Instrument and Form 43-101F1 (the "Form") and the parts of the Forsys Report for which I am responsible have been prepared in compliance with the Instrument and the Form.
- As of the effective date of the Forsys Report, to the best of my knowledge, information and belief, the parts of the Forsys Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Forsys Report not misleading.

Dated at Perth, Western Australia, on 16 March 2015.

{Signature covered}

Peter Nofal
Technical Director Studies,
AMEC Australia Pty Ltd

Certificate of Author

I, Dag Kullmann, M.Sc., FSAIMM do hereby certify that:

- I am the General Manager and Manager Mining and Technical Services for Valencia Uranium, Makarios Centre, Cottage Avenue, PO Box 4437, Vineta, Swakopmund, Namibia.
- I graduated with a M.Sc. Mining Engineering from the University of Alberta in 1989.
- I am a Fellow of the Southern African Institute of Mining and Metallurgy.
- I have worked as a mining engineer for 25 years since my graduation in 1989. I have been involved in mine design and reserve estimation for a range of commodities throughout Africa.
- I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, work experience and professional affiliation, I fulfil the requirements to be a “qualified person” for the purposes of NI 43-101.
- I am responsible for the supervising and writing of the following Section 3, 5, 15, 16, 18, 19, 20, 22 and 23 and partially responsible for Sections 1, 2, 4, 21, 24, 25 and 26 that relate to mining and infrastructure.
- I have visited the site on a regular basis, totalling more than one hundred occasions. My last visit to Norasa was on Wednesday, 17 December 2014 to inspect existing infrastructure.
- I have been directly involved with the project under review since February 2007.
- I am not independent of the issuer as defined by Section 1.5 of NI 43-101. As the disclosure of this report does not meet the criteria as defined in Section 5.3 (1)(c) of the NI 43-101, an independent qualified person is not required to write the Technical Report on the Norasa Project.
- I have read NI 43-101, and the parts of the Technical Report that I am responsible for have been prepared in accordance with NI 43-101.
- As at the effective date of the Technical Report and to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible for contain all scientific and technical information that is required to be disclosed to make the report not misleading.
- I consent to the public filing of the Technical Report titled Norasa Uranium Project, Definitive Feasibility Study, NI 43-101 Technical Report and dated 16 March 2015, by Forsys Metals Corp.

Signed and dated this 16th day of March 2015 as Swakopmund, Namibia.

Dag Kullmann, FSAIMM

Certificate of Author

I, Martin Hirsch, M.Sc., MIMMM, do hereby certify that:

- I am the Chief Geologist for Valencia Uranium, Makarios Centre, Cottage Avenue, PO Box 4437, Vineta, Swakopmund, Namibia.
- I graduated with a M.Sc. Geology (Diplom Geologe) from the Johann Wolfgang Goethe University of Frankfurt am Main, Germany in 1986.
- I am a Member of the Institute of Materials, Minerals and Mining.
- I have worked as a geologist for 28 years since my graduation in 1986. I have been involved in uranium exploration and mining in Namibia.
- I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, work experience and professional affiliation, I fulfil the requirements to be a “qualified person” for the purposes of NI 43-101.
- I am responsible for the supervising and writing of the geology sections including Sections 6, 7, 8, 9, 10, 11, 12 and 14 and partially responsible for sections 1, 15.1.1 and 25 that relate to geology,
- I have visited the site on a weekly basis (for up to 3 days at a time) during the exploration program. My last visit was on 9 March 2015 for the day.
- I have been directly involved with the project under review since August 2012.
- I am not independent of the issuer as defined by Section 1.5 of NI 43-101. As per Section 5.3.2 of the NI 43-101, an independent qualified person is not required to write the Technical Report on the Norasa Project.
- I have read NI 43-101, and the parts of the Technical Report that I am responsible for have been prepared in accordance with NI 43-101.
- As at the effective date of the Technical Report and to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible for contain all scientific and technical information that is required to be disclosed to make the report not misleading.
- I consent to the public filing of the Technical Report titled Norasa Uranium Project, Definitive Feasibility Study, NI 43-101 Technical Report and dated 16 March 2015, by Forsys Metals Corp.

Signed and dated this 16th day of March 2015 as Swakopmund, Namibia.

Martin Hirsch, MIMMM