Title Page

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SUMMARY

This National Instrument 43-101 (NI 43-101) Technical Report describes the Mineral Resource and Reserve estimation and economic analyses to a pre-feasibility 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) and Namibplaas (Exclusive Prospecting Licence, EPL 3638) 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.

A statement of updated Mineral Resources for Norasa was reported in October 2013 and updated Reserves were later reported in February 2014. Independent Technical Reports were previously filed for Valencia by the Snowden Group in 2010 for Resources and Reserves and for Namibplaas by Optiro in 2011 for a maiden Resource.

The Mineral Resources are reported above cut-off grades of 100 ppm and 160 ppm U₃O₈ 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 LoM.

Estimated Measured and Indicated Mineral Resource for Norasa is 237 Mt at a grade of 197 ppm U₃O₈, which equates to 103 Mlbs of U₃O₈. The estimated Inferred Mineral Resource is 50 Mt at a grade of 198 ppm U₃O₈ for 22 Mlbs of U₃O₈.

The Mineral Reserve estimate is summarised in Table 1.2. The total Proven and Probable Norasa Mineral Reserve is 177 Mt at a grade of 202 ppm, which equates to 79 Mlbs of U₃O₈. 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 1.1 Norasa Mineral Resource (October 2013)

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<td>153</td>
<td>9</td>
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<td></td>
<td>Val 100ppm: Nam 160ppm</td>
<td>17</td>
<td>202</td>
<td>7</td>
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<tr>
<td></td>
<td>Val 140ppm: Nam 200ppm</td>
<td>10</td>
<td>253</td>
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<td>Indicated</td>
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<tr>
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<td>Val 140ppm: Nam 200ppm</td>
<td>114</td>
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<td>62</td>
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<tr>
<td>Measured + Indicated</td>
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<td>Val 140ppm: Nam 200ppm</td>
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<td>Val 140ppm: Nam 200ppm</td>
<td>18</td>
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*Resources are reported inclusive of Reserves.*

### Table 1.2 Norasa Mineral Reserves Estimate (February 2014)

<table>
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<tr>
<th>Classification</th>
<th>Tonnes [M]</th>
<th>U₃O₈ [ppm]</th>
<th>U₃O₈ [Mlbs]</th>
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<td>Total Reserve</td>
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<td>79.0</td>
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</table>

*Cut-off grades of 100ppm for Valencia and 160ppm Namibplaas*
The Reserves come from three deposits, resulting in 3 distinct pits; the Valencia pit, a small satellite pit just 500m away from Valencia, and the Namibplaas pit.

Figure 1.1 Location of the Norasa Uranium Project.

1.1 Permitting

On 4 June 2008, Valencia received approval and clearance by the Ministry of Environment and Tourism (MET) on both the Environmental Impact Assessment and the Environmental Management Plan. This covered the development of the mine and associated
infrastructure, including the new access road, pipeline and power line. This permit was renewed for a 3-year period on 11 April 2013 under new legislation.

Valencia was granted its Mining Licence (ML) 149 in respect of the Nuclear Fuel Group of Minerals, effective from 23 June 2008, valid for a period of 25 years and renewable.

As the Valencia Licence area is situated on private land, agreement with the landowner was required prior to mine development. This agreement was signed on 30 April 2009, giving Valencia unrestricted access to 3,327 hectares of the Farm Valencia. In addition, on 8 May 2009 Valencia acquired a servitude within the neighbouring Farm Bloemhof for an area of 594 hectares, granting Valencia the right to construct any works related to the mine development and operation (Accessory Works). Initially, this area will be used for the road main water supply pipeline, a secondary power line and the explosives magazine. In both cases, compensation has been paid making the agreement binding for a period equal to the Mining Licence.

Subsequent to the land owners' agreements, on 29 May 2009, the Ministry of Mines and Energy (MME) granted Valencia approval for the construction of all Accessory Works required for the development of the mine.

Namibplaas currently holds the Exclusive Prospecting Licence (EPL) 3638, which is in good standing and currently under renewal from the MME. MET has also approved the combination of the projects from an environmental perspective and the Valencia EIA / EMP is currently being updated for final clearance to also include Namibplaas. In the mean time, Namibplaas continues to operate with a clearance for prospecting activities.

All legal and permitting requirements have been met for the development of Norasa within the Valencia licence area.

### 1.2 Geology and mineralisation

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 carnottite, titanite (including brannerite) and naobate. Late stage alterations of all alakite types can be observed and comprise kaolinisation, illitisation and silicification.
1.3 Mining

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. Two production / process scenarios were considered with the base case involving the introduction of radiometric sorting after the first couple years of production. As the use of sorting is not committed to in the initial construction period, an alternative production scenario without sorting is also considered. Both scenarios are reviewed in this report, but only the base case is summarised here.

To feed the mills at the design capacity of 11.2 Mtpa after sorting, Norasa will crush at a rate of 14.9 Mtpa, requiring a total peak mining rate of 68 Mtpa. The average strip ratio for the life of mine is 3.0.

![Figure 1.2 Total tonnes mined.](image)

The high mining rate of 68 Mtpa is required to ensure adequate waste stripping to achieve constant plant throughput as the various pits and their relevant pushbacks start up. Figure 1.3 below indicates the total tonnes in relation to areas / pushbacks mined annually.
1.4 Process plant

The proposed processing facility 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) recovery;
- filtration;
- calcination.

Radiometric sorting would be introduced at a later stage (after 2 years of production) once the system is fully evaluated with run-of-mine ore. The system will be designed based on real crusher product to analyse the rock size and grade distributions.

1.5 Infrastructure

Water will be supplied from the Rössing reservoir with a new 31km pipeline to site. Less than 3.0 Mm³ of water will be required annually. 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 30MW, with the largest demand being the SAG mill at 10.5MW.

A 26km private industrial road has been constructed connecting Norasa to the Trans Kalahari (B2) Highway, which is the main artery from Walvis Bay and Swakopmund to Windhoek and across Southern Africa.

Two waste dumps are proposed each for Valencia and Namibplaas as will a single large tailings dump near the process plant.

1.6 Capital and Operating Costs

The mine development is based on an initial annual plant design capacity of 11.2 Mt of ore to the crusher. The plant feed rate is planned to increase to 14.9 Mt after 2 years of operation with the introduction of radiometric sorting.

The project capital cost is US$392.1M with ongoing costs summarised in Table 1.3. These amounts do not include the initial operating costs.
Table 1.3 Capital cost and sustaining capital.

<table>
<thead>
<tr>
<th>Capital Items</th>
<th>Estimated Cost US$ (millions)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Process plant</td>
<td>249.7</td>
</tr>
<tr>
<td>Mining equipment</td>
<td>75.7</td>
</tr>
<tr>
<td>Tailings disposal</td>
<td>8.3</td>
</tr>
<tr>
<td>Bulk water supply</td>
<td>21.2</td>
</tr>
<tr>
<td>Power supply &amp; substation</td>
<td>15.9</td>
</tr>
<tr>
<td>Owner’s buildings and equipment</td>
<td>18.3</td>
</tr>
<tr>
<td>Road / access</td>
<td>3.0</td>
</tr>
<tr>
<td><strong>Project Capital</strong></td>
<td><strong>392.1</strong></td>
</tr>
<tr>
<td>Expansion radiometric sorting</td>
<td>39.9</td>
</tr>
<tr>
<td>Sustaining capital (life-of-mine)</td>
<td>60.3</td>
</tr>
</tbody>
</table>

The life-of-mine average operating cost per run-of-mine (ROM) tonne is US$14.06/t or $38.20/lb. These average unit costs are summarized by major cost items in Table 1.4. The highest unit cost is the mining cost, largely attributed to a stripping ratio of 3.0.

In the first 5 years of production, the average cash cost is only US$34.76/lb.

Table 1.4 Operating cost by item.

<table>
<thead>
<tr>
<th>Operating cost Item</th>
<th>Total Cost ($/t ROM)</th>
<th>Total Cost ($/lb U₃O₈)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mining (ore and waste)*</td>
<td>7.23</td>
<td>19.63</td>
</tr>
<tr>
<td>Process</td>
<td>6.41</td>
<td>17.40</td>
</tr>
<tr>
<td>Tailings disposal</td>
<td>0.05</td>
<td>0.13</td>
</tr>
<tr>
<td>Overheads</td>
<td>0.44</td>
<td>1.20</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>14.06</strong></td>
<td><strong>38.20</strong></td>
</tr>
<tr>
<td>Closure and rehab provision</td>
<td>0.04</td>
<td>0.11</td>
</tr>
<tr>
<td><strong>Total first 5 years of prod.</strong></td>
<td><strong>13.68</strong></td>
<td><strong>34.76</strong></td>
</tr>
</tbody>
</table>

* Includes cost of ROM pad rehandling into the primary crusher

1.7 Economic analysis

A financial model was prepared to assess the economics for Norasa based on the Mineral Reserve and mining schedule to report NPV, payback and IRR. The financial model quantifies the revenues, costs and capital expenditure over a 13-year life of mine. It is believed that these results are accurate to within ±25%, within the constraints of the associated assumptions.

NPVs were calculated on post-tax, uninflated cashflows at discount rates of 0%, 6% and 8% and outcomes shown in Table 1.5. The IRR for the project is 36%. A long-term uranium price of US$68/lb is assumed.

The project has a payback period of 3 years after commencement of production.
<table>
<thead>
<tr>
<th>Discount Rate</th>
<th>NPV (US$ M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0%</td>
<td>851</td>
</tr>
<tr>
<td>6%</td>
<td>491</td>
</tr>
<tr>
<td>8%</td>
<td>410</td>
</tr>
</tbody>
</table>
2 INTRODUCTION

This report has been prepared by Valencia Uranium (Pty) Ltd that has a 100% interest in Valencia, holder of ML 149 and Dunefield Mining Company (Pty) Ltd that has a 100% interest in Namibplaas, holder of EPL 3638. Valencia and Namibplaas are Namibian operating entities, both 100% owned by Forsys Metals Corp. (Forsys), a Canadian mineral exploration company listed on the Toronto Stock Exchange (TSX), reporting in compliance with the disclosure requirements of NI 43-101. Valencia Uranium (Pty) Ltd is the principal operating entity of the projects in Namibia.

Forsys announced the consolidation of its uranium projects, Valencia and Namibplaas in January 2013, the consolidated project being called the Norasa Uranium Project (Norasa) and subsequently reported an updated consolidated Mineral Resource Statement in October 2013 (Forsys Metals 2013).

As the disclosure of this report does not meet the criteria as defined in Section 5.3 (1)(c) of NI 43-101, an independent qualified person is not required to write the Technical Report on Norasa.

The work disclosed within this report is deemed to be of a ±25% accuracy in USD terms and hence to pre-feasibility level. It is noted that Forsys previously disclosed Mineral Resources and Reserves for Valencia with economic analyses to ±15% accuracy in Independent Technical Reports (Snowden 2009b, Snowden 2010). Further, Mineral Resources for Namibplaas have previously been disclosed in an Independent Technical Report (Optiro, 2011). This Mineral Resource was later updated in September 2012 (Forsys Metals 2012) but not subject to a Technical Report.

Forsys announced a Norasa Minerals Reserve Statement (Forsys Metals 2014) in February 2014, this being the reason for the updated Technical Report. This was the first reserve statement for the combined Norasa Project but it is noted that the major component of this reserve is from Valencia, which previously reported reserves in 2010.

The Qualified Persons for the preparation of this report are Mr. Martin Hirsch, Chief Geologist and Mr. Dag Kullmann, Manager Mining and Technical Services, both being permanent employees of Valencia.

Mr Hirsch is responsible for the writing of the exploration and geology related section of this report (Sections: 7 to 12), the Mineral Resource; Section 14 and contributed to Sections 1 to 6 & 25 to 27. Routine site visits were conducted to plan and oversee exploration programs, mapping and logging for geological and geotechnical data collection, monitoring quality control procedures and sample collection and preparation.

Mr Kullmann is responsible for the overall report compilation and supervised the writing of the following sections; 1 to 6, 13, and 15 to 27. Routine site visits were conducted to plan infrastructure logistics and locations such as waste rock dumps, tailings dumps, process plant and mining offices, access road, pipeline, power line and mine sub-station. Visits were also conducted to oversee geotechnical data collection for pit slope and plant foundation investigations.
3 RELIANCE ON OTHER EXPERTS

Comments regarding environmental issues associated with Valencia made by the authors in Section 20 of this report rely on the Environmental Impact Assessment for the Valencia deposit produced by Digby Wells and Associates in April 2008. This information has been reported in earlier Valencia Technical Reports, the most recent being Snowden 2009b, which remains relevant. The authors of this report are not qualified to provide extensive comment on environmental issues associated with Norasa.

Technical work supporting the metallurgy and process plant design aspects of work was conducted by AMEC (overseen by Mr Wayne Galea) as part of the ongoing technical studies. Comments contained in this report on the plant design in Section 17 of this report, but excluding 17.3, were obtained from AMEC’s Engineering Cost Study for Valencia, dated May 2013. This report is available on Sedar and the Forsys Metals website. The authors of this report are not qualified to provide extensive comment on the metallurgy and process plant design.

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, Xtract 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 (Xtract 2013, Xtract 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.
4 PROPERTY DESCRIPTION AND LOCATION

4.1 Valencia and Namibplaas

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 4.1), 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 and which are shown in Figure 4.2.

<table>
<thead>
<tr>
<th>ID</th>
<th>Longitude</th>
<th>Latitude</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>15.23828368</td>
<td>-22.32625528</td>
</tr>
<tr>
<td>2</td>
<td>15.25225024</td>
<td>-22.33894483</td>
</tr>
<tr>
<td>3</td>
<td>15.22720250</td>
<td>-22.36416441</td>
</tr>
<tr>
<td>4</td>
<td>15.21527297</td>
<td>-22.34900956</td>
</tr>
</tbody>
</table>

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 and shown in Figure 4.2. The total surface area is 1,742 ha.

It is noted that EPL 3638 is currently in the renewal process with the MME (Section 71 of the Minerals Act states that an EPL cannot expire during the period in which a renewal is being considered). Exploration licences are only renewed for 2-year periods at a time and the renewal is expected to be granted. The owner, Dunefield Mining Company (Pty) Ltd has been requested by the Ministry to reduce the area of the licence in line with the requirements of the Minerals Act. The reduced area, if so granted in the renewal, will not impact on the Mineral Resource or restrict the company’s ability to develop the project (see also Section 24).

<table>
<thead>
<tr>
<th>ID</th>
<th>Longitude</th>
<th>Latitude</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>15.24122697</td>
<td>-22.30105521</td>
</tr>
<tr>
<td>2</td>
<td>15.29048586</td>
<td>-22.26535988</td>
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<tr>
<td>3</td>
<td>15.30613676</td>
<td>-22.2958070</td>
</tr>
<tr>
<td>4</td>
<td>15.30077411</td>
<td>-22.31163411</td>
</tr>
</tbody>
</table>

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 expect those to the Namibian Government’s MME as described in Section 22.1 of this report.

4.2 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 additional assessments.

Valencia is fully permitted to allow commencement of construction and mining operations. There are no other requirements for Valencia to commence operations; neither legal, administrative nor 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 and noise 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.**

<table>
<thead>
<tr>
<th>Permit</th>
<th>Issued By</th>
<th>Date received</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mining Licence (ML149)</td>
<td>Ministry of Mines and Energy</td>
<td>23 June 2008</td>
</tr>
<tr>
<td>Exclusive Prospect Licence</td>
<td>Ministry of Mines and Energy</td>
<td>7 Nov 2011</td>
</tr>
<tr>
<td>(EPL3638)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Accessory Works ML149</td>
<td>Ministry of Mines and Energy</td>
<td>29 May 2009</td>
</tr>
<tr>
<td>Environmental Clearance</td>
<td>Ministry of Mines and Energy</td>
<td>11 April 2013</td>
</tr>
</tbody>
</table>
Figure 4.1 Location of the Norasa Project relative to other uranium mines.
Figure 4.2 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 1.1). 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 (Figure 4.1).

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 only 4.5km in a straight line and due to the rugged terrain was 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 trade¹, 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 deepwater 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 per annum and will complement the existing 350,000 TEUs per annum. Upon completion which is anticipated by 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 consumed at the Trekkopje mine, located about 40km from there in the

¹ http://www.namport.com.na/
desert. However, since the completion of the desal 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 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 as 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 (Norasa).
7 GEOLOGICAL SETTING AND MINERALIZATION

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.](image)

*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 Kinnaid 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

<table>
<thead>
<tr>
<th>SACS (unpublished)</th>
<th>LITHOLOGY</th>
</tr>
</thead>
<tbody>
<tr>
<td>GROUP SUBGROUP FORMATION</td>
<td>Maximum thickness (metres)</td>
</tr>
<tr>
<td>SWAKOP KHOMAS KUISEB</td>
<td>Pelitic and semi-pelitic schist and gneiss, migmatite, calc-silicate rock, quartzite.</td>
</tr>
<tr>
<td></td>
<td>Tinkas Member; pelitic and semi-pelitic schist, calc-silicate rock, marble, para-amphibolite.</td>
</tr>
<tr>
<td></td>
<td>Marble, calc-silicate rock, pelitic and semi-pelitic schist and gneiss, biotite-amphibole schist, quartz schist, migmatite.</td>
</tr>
<tr>
<td></td>
<td>Diamictite, calc-silicate rock, pebbly schist, quartzite, ferruginous quartzite, migmatite.</td>
</tr>
<tr>
<td></td>
<td>Discordance</td>
</tr>
<tr>
<td></td>
<td>200</td>
</tr>
<tr>
<td></td>
<td>UGAB RÖSSING Discordance</td>
</tr>
<tr>
<td>NOSIB KHAN</td>
<td>1 100</td>
</tr>
<tr>
<td></td>
<td>Migmatitic, banded and mottled quartzofeldspathic clinopyroxene-amphibole gneiss, hornblende-biotite schist, biotite schist and gneiss, migmatite, pyroxene-garnet-gneiss, amphibolite, quartzite, metagranulite.</td>
</tr>
<tr>
<td>ETUSIS</td>
<td>3 000</td>
</tr>
<tr>
<td></td>
<td>Quartzite, metagranulite, pelitic and semi-pelitic schist and gneiss, migmatite, quartzofeldspathic clinopyroxene-amphibole gneiss, calc-silicate rock, melaphyllite.</td>
</tr>
<tr>
<td>Major unconformity</td>
<td>Gneissic granite, augen gneiss, quartzofeldspathic gneiss, pelitic schist and gneiss, migmatite, quartzite, marble, calc-silicate rock, amphibolite.</td>
</tr>
</tbody>
</table>

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).

![Figure 7.2 Domes of the Central Zone and associated primary uranium deposits.](image_url)

Figure 7.2 Domes of the Central Zone and associated primary uranium deposits.


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.
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 leuco-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 Ababias 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>
<thead>
<tr>
<th>Types</th>
<th>Color</th>
<th>Texture and Mineralogy</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>Pale pink to white</td>
<td>Irregular to boudinaged, folded by D3, weak S3 foliation at margins, occur in high-strain zones. They are homogeneous, saharoidal and fine grained.</td>
</tr>
<tr>
<td>B</td>
<td>white</td>
<td>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</td>
</tr>
<tr>
<td>C</td>
<td>White to pink</td>
<td>Occasionally boudinaged, emplaced in F3 flexures. Medium to pegmatitic grain size, hypsersolvus with interstitial clear quartz, magnetite, ilmenite and tourmaline</td>
</tr>
<tr>
<td>D</td>
<td>White</td>
<td>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.</td>
</tr>
<tr>
<td>E</td>
<td>Variable</td>
<td>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.</td>
</tr>
<tr>
<td>F</td>
<td>Red</td>
<td>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.</td>
</tr>
</tbody>
</table>
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 is stretching 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).

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

The Valencia and Namibplaas deposits lie adjacent to the tightly folded Abbabis inlier (old Filoly zone-oor 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).

7.2.1 Valencia

The Valencia deposit is approximately 4km SE of the Khan River Valley located 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).
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.
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 calcereous 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 i biotite granofels as described for the Rössing Mine stratigraphy being mostly absent at Valencia.
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 7.6).

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.

**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 7.7)

![Figure 7.7 Valencia Satellite; combined geology plan of Valencia Satellite & Valencia Main on ML149.](image-url)
7.2.2. Namibplaas

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

Ground scintillometer surveys confirmed these 2 anomalies (see Areas A and B in Figure 7.10) 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.
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 7.10 Namibplaas ground radiometry overlain onto GRN Airborne radiometric survey map from 1997.

Figure 7.11 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 7.11); the eastern limb of the anticline with slightly folded Khan Syncline lies over Area B (Zone B, Figure 7.11). 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 7.12).

![Figure 7.12 WNW/ESE section showing attenuated Damara host rocks surrounded by sheeted alaskite (Laine & Shilongo, 2011).](image)

**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 C1 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.

**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 diamicite schists (+/-70m thick), which are metamorphosed tillites. This unit is characterised by quartzofeldspathic 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 (+130m 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 diamicite 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 gray 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 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. 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 resinos. 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 carnitite, titanite (including brannerite) and naobate.

A variety of secondary uranium minerals exists, 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 (UO₂) and the secondary uranium minerals as uranophane (Ca (UO₂) 2SiO₂.7H₂O) and uranothallite (Ca₂U(CO₃)₄.10H₂O).

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.

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 quiet unique.

Uranium mineralisation remains similar to Valencia and occurs as uraninite (\(\text{UO}_2\)) mineralisation and the secondary uranium minerals, uranophane (\(\text{Ca}(\text{UO}_2)2\text{SiO}_2.7\text{H}_2\text{O}\)) and uranothallite (\(\text{Ca}_2\text{U}(\text{CO}_3)_4.10\text{H}_2\text{O}\)).

Minor betafite (\(\text{U},\text{Ca})(\text{Ti},\text{Ta},\text{Nb})_3\text{O}_9\)) 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.

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.

Included in this type are those deposits associated with intrusive rocks including alaskite, granite, pegmatite and monzonites. 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) of approximately 25,000m of in this period.

Valencia commenced activities in the area in 2005 and started drilling in 2006 adding an initial 44 diamond drill holes over 12,832m to the 97 historical diamond drillholes over 24,790m.

Until 2009 a further 148 reverse circulation percussion (RC) drillholes were added which were drilled along a tight grid measuring 20m by 20m and drilled to an average depth of 300m across the anomaly.

Drilling was carried out by R.A. Longstaff Namibia (Pty), Major Drilling, Erongo Drilling Van Rhyn, Roburgh and Hard Rock Drilling with logging and sampling 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 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 (see Table 9.1).

<table>
<thead>
<tr>
<th>Hole</th>
<th>Re-log / Re-assay</th>
</tr>
</thead>
<tbody>
<tr>
<td>VA26-088</td>
<td>Re-log and re-assay</td>
</tr>
<tr>
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<td>Re-log and re-assay</td>
</tr>
<tr>
<td>VA26-093</td>
<td>Re-log and re-assay</td>
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<tr>
<td>VA26-096</td>
<td>Re-log and re-assay</td>
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</tr>
<tr>
<td>VA26-104</td>
<td>Re-log and re-assay</td>
</tr>
</tbody>
</table>

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 44 diamond drill holes (DDH) for 13,083m were drilled by Valencia for geotechnical data and infill grade sampling (Table 9.2). This information was 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).

Table 9.2 DDH drilled by Valencia.

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<th>Elevation</th>
<th>Depth</th>
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</table>

The total diamond drilling database (historical and drilled by Valencia) consisting of 144 holes over 37,760m.

### 9.1.4. RC drilling

A total of 148 reverse circulation (RC) drillholes were completed, for 11,101m 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. The information from the RC drilling was used for Mineral Resource estimation providing infill information with a high degree of confidence in the grade estimate.

### 9.1.5. PD drilling

A total of 410 percussion drilling (PD) drillholes, including 5 re-drilled holes at Valencia for 109,160m.

### 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 9.2).

![Figure 9.2 Valencia Satellite SPP2 ground radiometric contour map with PD collar positions.](image)

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 (max) at a total of 6,548 meters (Table 9.3).

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

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.

Table 9.3 PD drilled at Valencia Satellite.

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Drilling was carried out by Ferro Drill Namibia.

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 9.3).

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 Figure 9.3.
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 9.4).

Figure 9.4 Map of EPL 3638 Zone B after Hinojosa (2008).
9.3.1. DDH drilling

Gold Fields Namibia Ltd drilled seven diamond drill holes before Valencia commenced exploration at Namibplaas in 2008 (Table 9.4).

Table 9.4 Collars of Gold Fields boreholes.

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Valencia drilled additional 19 DDH in 2010 and 14 in 2011 (Table 9.5).

Table 9.5 Collars of Valencia diamond drill holes (Figure 9.5).

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In total 40 DDH, totalling 9,894m (33 new holes for 8,229m and 7 historic holes for 1,665m) have been completed and provide information for grade interpolation. Drill core was sampled and chemically analysed for uranium to define the local correlation between the probe data and assayed U₃O₈ values.

9.3.2. PD drilling

PD 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 in-fill drilling the area significantly by adding 242 holes over 47,924 m.

The total PD drilling effort amounts to 530 percussion drill holes over 111,017m (Figure 9.6).
The area of higher grades corresponds to the magnetic high shown in the violet coloured zone, trending northeast to southwest.

9.3.3. 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.
9.3.4. 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.

9.3.5. 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. OTV logs were processed by Xstrata and the resulting 3D structural dataset used by VBKom to aid latest pit design work.
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 12,832m 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).](image)

10.1.2 RC drilling

148 RC drillholes over 11,101m covered on 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.
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 410 drillholes drilled until 2011.

10.2 Valencia satellite

During 2012 and 2013 exploration efforts moved to a small area which is close to the main Valencia anomaly and which was not included in previous property evaluations; the project 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 decided to incorporate results into the combined resource statement.

52 PD holes were drilled to shallow depths; for coordinates see Table 9.3.

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).
Grade information is derived from geophysical logging at 0.1m intervals with the gamma readings empirically converted into a $\text{U}_3\text{O}_8$ 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.

10.3 Namibplaas

10.3.1 Diamond drilling

Figure 10.8 below outline DDH collar position using various displays.
19 DDH holes were 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 2009a 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 establish from Valencia DDH drilling by Dr. Laine² in 2008).

The correlation curve between historical chemical assays and equivalent U₃O₈ 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 U₃O₈ 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 scintilometer counts, identified some discrepancies in both sample preparation and sample 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.

The author (Snowden) considers 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.

---

² Dr. Roger Laine was Chief Geologist at Forsys Metals Corp at the time.
11.2 Valencia

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 U3O8 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 (QAQC) procedures during the analysis of the Valencia samples by including certified reference standards (CRM), blanks and duplicates. The protocols for the QAQC 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

rPD 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).

Table 12.1 DDH drillholes verified by Snowden.

<table>
<thead>
<tr>
<th>September 2005</th>
<th>November 2006</th>
<th>November 2008</th>
</tr>
</thead>
<tbody>
<tr>
<td>Drillhole</td>
<td>Drillhole</td>
<td>Drillhole</td>
</tr>
<tr>
<td>VA26-011</td>
<td>VA26-115</td>
<td>VA26-135</td>
</tr>
<tr>
<td>VA26-017</td>
<td>VA26-116</td>
<td>VA26-138</td>
</tr>
<tr>
<td>VA26-036</td>
<td>VA26-125</td>
<td>VA26-140</td>
</tr>
<tr>
<td>VA26-058</td>
<td>VA26-127</td>
<td>VA26-142</td>
</tr>
<tr>
<td>VA26-059</td>
<td>VA26-130</td>
<td>VA26-145</td>
</tr>
<tr>
<td></td>
<td>VA26-133</td>
<td>VA26-147</td>
</tr>
</tbody>
</table>

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.

<table>
<thead>
<tr>
<th>Drillhole</th>
</tr>
</thead>
<tbody>
<tr>
<td>NA24-008</td>
</tr>
<tr>
<td>NA24-011</td>
</tr>
<tr>
<td>NA24-036</td>
</tr>
</tbody>
</table>

Figure 12.1 Optiro reading adjacent to drillhole NAPD-077.
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.

<table>
<thead>
<tr>
<th>BH ID</th>
<th>SAMP ID</th>
<th>U₃O₈ ppm</th>
<th>U₃O₈ ppm*</th>
</tr>
</thead>
<tbody>
<tr>
<td>8</td>
<td>21002</td>
<td>311</td>
<td>321</td>
</tr>
<tr>
<td>8</td>
<td>21011</td>
<td>868</td>
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<tr>
<td>8</td>
<td>21012</td>
<td>208</td>
<td>184</td>
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<tr>
<td>8</td>
<td>21016</td>
<td>730</td>
<td>759</td>
</tr>
<tr>
<td>11</td>
<td>20751</td>
<td>282</td>
<td>282</td>
</tr>
<tr>
<td>12</td>
<td>20203</td>
<td>307</td>
<td>269</td>
</tr>
<tr>
<td>12</td>
<td>20204</td>
<td>223</td>
<td>204</td>
</tr>
<tr>
<td>16</td>
<td>20873</td>
<td>151</td>
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<td>20874</td>
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<tr>
<td>25</td>
<td>21487</td>
<td>144</td>
<td>139</td>
</tr>
</tbody>
</table>

*re-assay result

12.3 Statements regarding verification

Both, Snowden and Optiro confirmed that the data used for the various mineral estimates is reliable. Snowden (2009a) 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.

Valencia is confident that the data used in the 2012 and 2013 mineral resource updates is accurate and reliable.
13 MINERAL PROCESSING AND METALLURGICAL TESTING

Mineralogical and metallurgical leach testwork was carried out by SGS Lakefield Research (later SGS South Africa, “SGS”) and Mintek over a number of work programmes. Historic work was documented in the Technical Report of June 2009. More recent and pertinent work performed at various laboratories is discussed below.

13.1 Mineralogy

13.1.1. Valencia

Mineralogical evaluations and uranium deportment analyses (SGS, 2008b) were also conducted in order to understand the mineral composition of the various samples across the orebody. Characterisation of ore and gangue minerals were based on observations from polished thin sections based on the QEMSCAN system. The uranium deportment analysis included:

- specifying the uranium phases;
- uranium distribution / deportment;
- grain size distribution;
- liberation;
- association analyses of uranium phases.

The following points summarise the key finding of this work:

- 87% of the uranium is hosted in uraninite (ranging between 69% and 96% for the samples submitted.
- U-silicates consist of the following:
  - mostly coffinite and uranophane (also very amenable to acid leaching);
  - lesser amounts of uraniferous leucoxene / brannerite (less amenable to acid leaching).
- No betafite was reported (betafite is not amenable to acid dissolution).
- The uraninite grain size was widely variable, but generally less than 150 micron in size. There were however several samples containing particles greater than 150 microns. This indicates that some uraninite ‘nugget effect’ can occur during sampling.
- In general, the uranium silicates are less than 30 microns in size.
- Clearly liberation is dependent on grain size and grind. The high uranium extraction achieved on most of these samples indicates that the bulk of the uranium is exposed to the acid solution.
13.1.2. Namibplaas

SGS (2011a) conducted a general mineralogical and uranium deportment analysis on five uranium ore samples from the Namibplaas Project. These were preliminary only to determine if the nature of the uranium mineralisation is similar to that of Valencia.

The uranium-phases present in the samples were grouped into four groups: U-oxides, U-silicates, U-Ti-oxides and U-Th-silicates:

- The U-oxide minerals are mainly uraninite (UO$_2$) and possibly also pitchblende, a more hydrated and lower grade version of uraninite.
- The U-Ti-oxides are complex minerals, including betafite and low amounts of brannerite. The U-Ti-oxides do not readily release uranium with acid, even in an oxidized state and cannot be leached at atmospheric pressure with sulphuric leach. Note that the U-Ti-oxide content of the samples may be slightly over estimated due to the difficulty in distinguishing some Ti-rich minerals.
- This predominant U-silicate phase is Ca-poor and is most probably similar to coffinite. In some cases a U-oxide phase is being replaced by a U-silicate phase.
- Thorite and uranothorite and/or thorogummite, are present in significant amounts in some of the samples and may be a major carrier of uranium. Uranothorite is acid leachable.
- Other uranium-bearing minerals present in significant amounts are monazite and zircon. The uranium in these two minerals is not leachable.

Liberation analyses were done on the material crushed to ~80%-850μm, and grinding finer normally leads to better liberation (Table 13.1). Note that the liberation characteristics do not directly correlate with leachability, since locked grains that are exposed will still be leachable.

Uranium oxides (uraninite and pitchblende) and U-silicates (uranophane and coffinite) and uranothorite can be leached using acidic or alkaline media, provided that the uranium minerals are exposed or the gangue is sufficiently permeable to allow access of the leach medium. It is therefore important to quantify the exposure characteristics of the uranium minerals. Exposure analyses were done for all the uranium phases grouped together. Note that the sample was crushed to ~80%-850μm.
Variability leach tests – Valencia samples

In the latter part of 2012, Valencia commissioned SGS (2013a) to conduct a series of acid leach tests on a range of Valencia ore samples to better understand the variability of the leach results under a standard set of conditions as previously defined in 2010.

Of the potential samples identified, 48 were chosen for leach tests. The samples were representative of the orebody as a whole, with a range of grades that can be expected to be delivered to the leach tanks and taken from across the proposed pit at the time.

Samples were to be taken from the coarse reject material from the original half-core that was sent for assay from previous diamond drill programmes. These samples had been stored in a manner that preserved the integrity of the samples. A complete list of samples was available such that they could be identified for suitability in terms of grade and available mass for the planned testwork. The following criteria were used for selection:

- All samples are included within the Mineral Resource.
- All samples are taken from mineralized areas that are at least 5m wide (by core length), and in general more than 10m.
- There are no samples adjacent to each other that are of similar grade.
- Samples were selected to provide a reasonable distribution by grade and downhole depth. An average grade of approximately 200 ppm was achieved from all samples. All DD holes were drilled generally between 40 and 50 degrees.
- The samples represent all stages of mining.

Note that some initial acid leach tests were conducted on these samples, but some irregularities were identified and could not be suitably explained from the limited number of tests. The results are therefore not reported here. Further leach tests were conducted subsequently and noted below in Section 13.3.
The grade of samples ranged from 44 to 486 ppm U₃O₈. There appears to be a relationship of recovery as a function of grade (Figure 13.3). This is also noted in the summary graphs of the leach kinetics. Figure 13.4 illustrates the average recovery with time for all 48 samples where an average recovery of over 90% can be achieved in 12 hours. However, if the dataset is divided into the 24 lower grade samples and 24 high grade samples, there is a clear variation in recovery confirmation the relationship with grade (Figure 13.5).

![Figure 13.3](image1.png)

**Figure 13.3** Uranium recovery (%) as a function of the head grade.

![Figure 13.4](image2.png)

**Figure 13.4** Weighted average recovery at various sampling periods.

![Figure 13.5](image3.png)

**Figure 13.5** Comparison of recoveries of the higher and lower grade samples.
The average acid consumption for all tests after 12 hours of leach time was 15.4 kg/t and did not appear to have any significant dependence on grade.

13.3 Leach tests at SAG mill grind conditions

Subsequent to the completion of the AMEC Engineering Cost Study Report in May 2013, a series of tests were conducted by SGS South Africa (2013c) to determine the impact of the leach characteristics under a set of standard conditions but the finer grind of SAG milling. A total of 30 samples were tested, 20 from Valencia and 10 from Namibplaas. The samples were each blended and milled to 80% -600 μm. Slurries at 50% solids were prepared in bottles with sulphuric acid added to decrease the pH to 1.8. The pH was maintained at 1.8 and the redox potential maintained above 480 mV using MnO2. Tests were conducted at ambient temperatures.

The Valencia samples ranged in grade from 71 to 343 ppm U3O8 (averaging 199 ppm), while the Namibplaas samples ranged from 139 to 388 ppm (averaging 212 ppm).

The results of the 24 hour leach tests are summarised in Figure 13.6 for both sets of samples separately. The leach recovery of the Namibplaas samples are lower as expected from the mineralogical work reported in Section 13.1.2.

![Figure 13.6 Weighted average recovery for SAG mill conditions.](image)

It should be noted that the above tests do not take into account the pre-leach benefits obtained by milling in acid that is proposed for the SAG option reported by AMEC. Also for consideration is that the leach kinetics is enhanced during the milling process as heat is generated as a direct result of the milling.

The average acid consumption for Valencia and Namibplaas were 13.3 kg/t and 10.3 kg/t respectively.

13.4 Comminution testwork

The need was identified to embark on a further testwork program (SGS, 2008a) on a set of properly selected samples to confirm and gain confidence in previous testwork results. The testwork program included testwork to determine comminution parameters on a number of samples to evaluate variability across the ore body. Leach tests were also performed on these samples to evaluate leach liberation characteristics and determine variability in reagent consumption. Samples were selected such that they were spread across the ore body and represent the ore to be mined throughout the life of mine. The testwork program was designed such that the selected grind size of 80% passing 850 μm, (P80 of 850 μm), could be confirmed or a new optimum grind size could be specified. The selected grind size in conjunction with the comminution test results was used to trade off three stage crushing and rod milling with single stage crushing and SAG milling.

The aims of the testwork program were:

- to better understand ore and dilution rock variability with respect to comminution properties, acid demand and grind sensitivity
• to generate improved confidence in the selection of comminution circuit design criteria and parameters used to assign the value of an ore block in the mining model or the process operating cost model

• to support selection of the optimal comminution circuit for the project.

A total of 27 diamond drill core samples were selected. The samples were selected such that they were representative of the material to be mined over the total life of mine of approximately 15 years. A total of 8 drill core samples were selected from the area to be mined in the first six months, which will be processed during the commissioning and ramp-up phase. A total of 11 drill core samples were selected from the area that will be mined from six months to 5 years of operation, which represents the payback phase, and a total of 8 drill core samples were selected from the area to be mined beyond 5 years.

In addition, samples were selected of the three major lithologies at Valencia (granite, marble and schist) for UCS and crushability tests. A composite sample was made up from additional material and used for the first three variable grind leach tests.

The following conclusions were noted:

• The Uranium grade of the samples ranged from 110 ppm to 243 ppm.

• The Crushability work index ranged from 7.7 kWh/t to 13.1 kWh/t.

• The Bond Abrasion Index ranged from 0.1098 to 0.3098.

• The Bond Ball Mill Work Indices varied from 13.7 kWh/t to 15.8 kWh/t.

• The Bond Rod Mill Work Indices varied from 10.2 kWh/t to 10.7 kWh/t.

• The Effect of Grind appraisal indicated that the sample was relatively insensitive to grind, with the only notable difference being an increase in the rate of uranium dissolution. This leads to savings on acid consumption on the finer grind, but economic considerations will need to be taken into account to determine the optimum grind size, not leach performance.

• The slurry density testwork indicated that the lower slurry density yielded faster initial kinetics. However, at 65% solids the slurry appeared viscous and may prove difficult to handle on a bulk scale.

• The stirred tank reactor tests yielded faster dissolution kinetics than the equivalent bottle roll tests, however the ultimate dissolutions were similar but had higher acid consumptions for the stirred tank reactor testwork.

• The bulk leach testwork indicated similar results to that obtained in the bench scale work with dissolutions of 90% being obtained for both the 850 &m and the 212 &m material.

13.5 Radiometric sorting bench testing

As part of the ongoing metallurgical test programs for evaluating various process options, Valencia blasted 9 mini-bulk sample pits from across the orebody. Approximately 130t of blasted rock was produced from each pit. This would provide the best possible run-of-mine representative samples in terms of rock size and grade distributions. Although the pits were drilled with hand-held, compressed air equipment with small diameter, short blast holes, it did provide raw material that did not come from small diameter diamond core. The blasted ore was then loaded and hauled to a commercial crushing facility to produce a -40mm product (approximately 80% at -40mm). Although on a small scale, it did provide the best representation of production material that could also be used for radiometric sorting testwork.

Three pits of various average grade blasted rock (86, 163 and 267 ppm U₃O₈) were identified as suitable for evaluating the sorting potential by radiometric systems. The larger rocks were then collected (all about +30mm in the smallest dimension) for analyses in a radiometric test rig. Radiation levels were measured and correlated to subsequent chemical assay results. More than 1,400 radiometric measurements were conducted. These results were then used to predict the cut efficiency full scale sorting.
The results of the analyses illustrate the upgrade potential for different feed grades at various sorting cut-points. Figure 13.7 is an example of the upgrade when a cut-point of 120 ppm (an equivalent setting based on radiometric signature). Assuming this cut-point, with a feed grade of 200 ppm, 35% of the rock is rejected with an associated loss of 12% of the uranium. This results in a 33% upgrade of the plant feed to 267 ppm while the 'waste' rock has an average grade of 71 ppm.

Figure 13.7 Radiometric sort performance for Valencia rock samples.

It must be born in mind that only rock samples can be practically sorted. The smaller the size fraction of rock that is to be sorted, the lower the throughput of the sorting plant. The cost-benefit is thus reduced. A full analysis is required prior to introduction on a large scale.
14 MINERAL RESOURCE ESTIMATES

The following should be read in conjunction with the various Technical Reports from Snowden and Optiro released between January 2009 and September 2011; which are available on the Sedar website (see also Section 6.1.1)

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 drill holes 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.

In summary; U₃O₈ 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).

<table>
<thead>
<tr>
<th>Direction</th>
<th>Structure 1</th>
<th>Structure 2</th>
<th>Structure 3</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Nugget</td>
<td>Sill</td>
<td>Range (m)</td>
</tr>
<tr>
<td>1) 0°→020°</td>
<td>20</td>
<td>95</td>
<td>400</td>
</tr>
<tr>
<td>2) -20°→290°</td>
<td>0.15</td>
<td>0.59</td>
<td>20</td>
</tr>
<tr>
<td>3) 70°→290°</td>
<td>20</td>
<td>80</td>
<td>230</td>
</tr>
</tbody>
</table>

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 drill hole 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 (U₃O₈ = 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$_3$O$_8$ assay data from DDH and RC drill holes was used. Gamma derived values were used if there wasn’t a U$_3$O$_8$ 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 Table 14.2. The sill is normalised to 1 and spherical structures were modelled.

Table 14.2 Back transformed variogram parameters used in estimation of U$_3$O$_8$ (Snowden, June 2009).

<table>
<thead>
<tr>
<th>Direction</th>
<th>Nugget</th>
<th>Structure 1</th>
<th>Structure 2</th>
<th>Structure 3</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Sill (m)</td>
<td>Range (m)</td>
<td>Sill (m)</td>
</tr>
<tr>
<td>1) 00° - 050°</td>
<td>20</td>
<td>80</td>
<td>390</td>
<td></td>
</tr>
<tr>
<td>2) 30° - 320°</td>
<td>0.12</td>
<td>0.75</td>
<td>20</td>
<td>0.11</td>
</tr>
<tr>
<td>3) -60° - 320°</td>
<td>20</td>
<td>100</td>
<td>175</td>
<td></td>
</tr>
</tbody>
</table>

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$_3$O$_8$ 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$_3$O$_8$ cut-off.

The Mineral Resource utilised anticipated SMU sizes of 10m x 10m x 5m.

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 been filed on the Sedar website (Snowden January 2009; Snowden June 2009 & Snowden 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 study activities for a common application across all deposits are detailed below.

The internal study commenced in 2012 and involved extending the model downwards by adding blocks from elevation 352.5m amsl down to elevation 302.5m amsl (block centre) (Figure 14.1).
Figure 14.1 P50 resource model looking north showing “opportunity” for expanding the model downwards.

The process of extending entailed adding an empty block model with corresponding block dimensions to the existing P50 model and merging the two together.

In total 10 rows (levels) of 5m each across the entire P50 were added to the bottom without performing any changes to the P50 blocks at this stage.

Figure 14.2 shows the resulting block model looking north with additional blocks providing “room” for previously “un-utilised drillhole data.

No rock type allocations were done for the additional blocks. A GRT equivalent flag was used from grade information at a \( \text{U}_3\text{O}_8 +50\) ppm threshold.

Blocks in the extended portion are classified inferred unless found to be positioned within a 40m distance from the next drillhole data point.

Grade interpolation into the bottom block model commenced excluding the upper P50 blocks and utilising all available drillhole data. Ordinary kriging was used applying the same geostatistical parameter as were used during Snowden’s P50 generation in 2010.

Software used for the update was MicroMine, version 11.
A second change involved a categorical block model reconciliation using wireframe based domains and flagged resource blocks which were classified as "inferred" but are positioned in close proximity (max 40m) to drillhole data.

Respective blocks were re-assign to indicated if found "empty" (no-category) or if found inferred.

The 40m range of influence was utilised as the principle guideline for this classification review and was used in alignment to the grade variogram parameter developed by Snowden in 2009 (Table 14.2, structure 2). Figure 14.3 illustrates the extend in 2D by showing in colour "light blue" affected blocks, clearly visible positioned inside of the dense drill cover which is depicted by the yellow D40 perimeter line.

![Figure 14.3 Plan view of Valencia deposit at elevation 652.5m.](image)

The third and final change commended at the end of 2013 and involved analyses and review of the applied proportioning methodology with focus on its application and after peer review, the outcomes were adopted.

Excluding "non-GRT" grade information in the evaluation process plus applying block proportioning using indicator kriging or simulation techniques is a common approach in alaskite type deposits, however, a grade proportioning at SMU level is not seen as a correct application anymore.

At Valencia it is evident that one side of the "proportional grade" was lost in the process by maintaining the proportioning down to the smallest block units.

The use of alaskite proportioning is a technique which was developed "next door" at Rio Tinto's Rössing mine a number of years ago and has since then spread and evolved further but principals remain the same that care must be taken when approaching SMU sizes. (MFH)³, ⁶

With reference to Valencia the application of proportioning at 20m x 20m x 5m (parent block) and continuing this into SMU sizes of 10m x 10m x 5m is erroneous. Grade interpolation throughout the process is always interpolated into a full block, irrespective of its size. Reporting a proportional metal content from a non-proportional alaskite (at SMU level) is then subsequently suppressing some of the contained metal.

³ (MFH) Martin Hirsch, co-author of this report and Chief Geologist at Valencia since 2011, worked prior at Rössing for almost 13 years, key roles in Structural Geology, Resource Geology and last role as Chief Geologist.
**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 down-hole gamma probing were used to estimate the small high grade ore body.

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 \( \text{eU}_3\text{O}_8 \)ppm 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.

OK 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 14.3.

### Table 14.3: Valencia Total Mineral Resource (October 2013).

<table>
<thead>
<tr>
<th>Category</th>
<th>Cut-Off Grades</th>
<th>Tonnes [M]</th>
<th>( \text{U}_3\text{O}_8 ) [ppm]</th>
<th>( \text{U}_3\text{O}_8 ) [Mlbs]</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Measured</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>60ppm</td>
<td>27</td>
<td>153</td>
<td>9</td>
<td></td>
</tr>
<tr>
<td>100ppm</td>
<td>17</td>
<td>202</td>
<td>7</td>
<td></td>
</tr>
<tr>
<td>140ppm</td>
<td>10</td>
<td>253</td>
<td>6</td>
<td></td>
</tr>
<tr>
<td><strong>Indicated</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>60ppm</td>
<td>258</td>
<td>153</td>
<td>87</td>
<td></td>
</tr>
<tr>
<td>100ppm</td>
<td>160</td>
<td>199</td>
<td>70</td>
<td></td>
</tr>
<tr>
<td>140ppm</td>
<td>100</td>
<td>248</td>
<td>55</td>
<td></td>
</tr>
<tr>
<td><strong>Measured + Indicated</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>60ppm</td>
<td>286</td>
<td>153</td>
<td>97</td>
<td></td>
</tr>
<tr>
<td>100ppm</td>
<td>177</td>
<td>199</td>
<td>78</td>
<td></td>
</tr>
<tr>
<td>140ppm</td>
<td>111</td>
<td>248</td>
<td>60</td>
<td></td>
</tr>
<tr>
<td><strong>Inferred</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>60ppm</td>
<td>31</td>
<td>165</td>
<td>11</td>
<td></td>
</tr>
<tr>
<td>100ppm</td>
<td>20</td>
<td>214</td>
<td>10</td>
<td></td>
</tr>
<tr>
<td>140ppm</td>
<td>12</td>
<td>281</td>
<td>7</td>
<td></td>
</tr>
</tbody>
</table>

### 14.2 Namibplaas

The following is a partial excerpt of Optiro’s 2011 Technical Report which is available for download from the SEDAR website.

A total of 2,328 DDH sample uranium assays and 42,124 readings from down-hole geophysical logging of PD drillholes were used to estimate the Namibplaas Mineral Resource. Only samples associated with logged alaskite were used in the resource estimate.
Lithology was composited into 1.5m intervals based on lithology codes with all composited intervals logged as alaskite (log codes GRTT, GRT or AGRT) than flagged as CODE=GRT. They were then assigned a numerical code (NLITH) of 1, while all non GRT intervals were assigned NLITH=0.

Variography was then undertaken on the numerically coded lithology. NLITH values of between 0 and 1 were estimated in the block which was considered to reflect the probability of the block being GRT, in context of resource reporting this probability being used as an alaskite proportion.

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 used the ranges and orientation modelled in the variograms. The minimum number of composites used for the interpolation was ten and the maximum number of samples used was 42.

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

### Table 14.4 Variogram parameter for Grade.

<table>
<thead>
<tr>
<th>Direction</th>
<th>Nugget</th>
<th>Structure1</th>
<th>Structure2</th>
<th>Structure3</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Sill</td>
<td>Range (m)</td>
<td>Sill</td>
</tr>
<tr>
<td>-05--&gt;201</td>
<td>0.14</td>
<td>0.46</td>
<td>27</td>
<td>1.08</td>
</tr>
<tr>
<td>29--&gt;288</td>
<td></td>
<td>0.17</td>
<td>27</td>
<td>0.23</td>
</tr>
<tr>
<td>-60--&gt;300</td>
<td></td>
<td>20</td>
<td>72.5</td>
<td>130.5</td>
</tr>
</tbody>
</table>

Reporting is on full SMU ore blocks.

### Table 14.5: Namibplaas Project Mineral Resource (October 2013).

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Measured</td>
<td>100ppm</td>
<td>161</td>
<td>152</td>
<td>54</td>
</tr>
<tr>
<td></td>
<td>160ppm</td>
<td>160</td>
<td>191</td>
<td>25</td>
</tr>
<tr>
<td></td>
<td>200ppm</td>
<td>14</td>
<td>246</td>
<td>8</td>
</tr>
<tr>
<td>Indicated</td>
<td>100ppm</td>
<td>161</td>
<td>152</td>
<td>54</td>
</tr>
<tr>
<td></td>
<td>160ppm</td>
<td>160</td>
<td>191</td>
<td>25</td>
</tr>
<tr>
<td></td>
<td>200ppm</td>
<td>14</td>
<td>246</td>
<td>8</td>
</tr>
<tr>
<td>Measured + Indicated</td>
<td>100ppm</td>
<td>161</td>
<td>152</td>
<td>54</td>
</tr>
<tr>
<td></td>
<td>160ppm</td>
<td>160</td>
<td>191</td>
<td>25</td>
</tr>
<tr>
<td></td>
<td>200ppm</td>
<td>14</td>
<td>246</td>
<td>8</td>
</tr>
<tr>
<td>Inferred</td>
<td>100ppm</td>
<td>74</td>
<td>152</td>
<td>25</td>
</tr>
<tr>
<td></td>
<td>160ppm</td>
<td>30</td>
<td>188</td>
<td>12</td>
</tr>
<tr>
<td></td>
<td>200ppm</td>
<td>6</td>
<td>245</td>
<td>3</td>
</tr>
</tbody>
</table>

### 14.3 NORASA

Table 14.6 below is presenting the combined project resource. Note that Resources are reported inclusive of Reserves.
Table 14.6: Norasa Mineral Resource (October 2013).

<table>
<thead>
<tr>
<th>Category</th>
<th>Cut-Off Grades</th>
<th>Tonnes [M]</th>
<th>U$<em>{3}$O$</em>{8}$ ppm</th>
<th>U$<em>{3}$O$</em>{8}$ [Mlbs]</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Measured</strong></td>
<td>Val 60ppm: Nam 100ppm</td>
<td>27</td>
<td>153</td>
<td>9</td>
</tr>
<tr>
<td></td>
<td>Val 100ppm: Nam 160ppm</td>
<td>17</td>
<td>202</td>
<td>7</td>
</tr>
<tr>
<td></td>
<td>Val 140ppm: Nam 200ppm</td>
<td>10</td>
<td>253</td>
<td>6</td>
</tr>
<tr>
<td><strong>Indicated</strong></td>
<td>Val 60ppm: Nam 100ppm</td>
<td>419</td>
<td>153</td>
<td>141</td>
</tr>
<tr>
<td></td>
<td>Val 100ppm: Nam 160ppm</td>
<td>221</td>
<td>197</td>
<td>96</td>
</tr>
<tr>
<td></td>
<td>Val 140ppm: Nam 200ppm</td>
<td>114</td>
<td>248</td>
<td>62</td>
</tr>
<tr>
<td><strong>Measured + Indicated</strong></td>
<td>Val 60ppm: Nam 100ppm</td>
<td>447</td>
<td>153</td>
<td>150</td>
</tr>
<tr>
<td></td>
<td>Val 100ppm: Nam 160ppm</td>
<td>237</td>
<td>197</td>
<td>103</td>
</tr>
<tr>
<td></td>
<td>Val 140ppm: Nam 200ppm</td>
<td>125</td>
<td>248</td>
<td>68</td>
</tr>
<tr>
<td><strong>Inferred</strong></td>
<td>Val 60ppm: Nam 100ppm</td>
<td>105</td>
<td>156</td>
<td>36</td>
</tr>
<tr>
<td></td>
<td>Val 100ppm: Nam 160ppm</td>
<td>50</td>
<td>198</td>
<td>22</td>
</tr>
<tr>
<td></td>
<td>Val 140ppm: Nam 200ppm</td>
<td>18</td>
<td>269</td>
<td>10</td>
</tr>
</tbody>
</table>
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 pre-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 fulfilled the company’s strategic criteria of cut-off grade.

The total Proven and Probable Mineral Reserve for Norasa is 177 Mt at a grade of 202 ppm, which equates to 79 Mlbs of U₃O₈.

Table 15.1: Norasa Mineral Reserves Estimate (February 2014).

<table>
<thead>
<tr>
<th>Classification</th>
<th>Mt</th>
<th>Grade ppm U₃O₈</th>
<th>Mlbs U₃O₈</th>
</tr>
</thead>
<tbody>
<tr>
<td>Proven</td>
<td>16</td>
<td>203</td>
<td>7.3</td>
</tr>
<tr>
<td>Probable</td>
<td>161</td>
<td>202</td>
<td>71.7</td>
</tr>
<tr>
<td>Total Reserve</td>
<td>177</td>
<td>202</td>
<td>79.0</td>
</tr>
</tbody>
</table>

Cut-off grades of 100 ppm for Valencia and 160 ppm Namibplaas

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

Table 15.2: Valencia Reserves Estimate (February 2014).

<table>
<thead>
<tr>
<th>Classification</th>
<th>Mt</th>
<th>Grade ppm U₃O₈</th>
<th>Mlbs U₃O₈</th>
</tr>
</thead>
<tbody>
<tr>
<td>Proven</td>
<td>16</td>
<td>203</td>
<td>7.3</td>
</tr>
<tr>
<td>Probable</td>
<td>135</td>
<td>201</td>
<td>60.1</td>
</tr>
<tr>
<td>Total Reserve</td>
<td>152</td>
<td>201</td>
<td>67.4</td>
</tr>
</tbody>
</table>

Cut-off grade of 100 ppm

Table 15.3: Namibplaas Reserves Estimate (February 2014).

<table>
<thead>
<tr>
<th>Classification</th>
<th>Mt</th>
<th>Grade ppm U₃O₈</th>
<th>Mlbs U₃O₈</th>
</tr>
</thead>
<tbody>
<tr>
<td>Proven</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Probable</td>
<td>25</td>
<td>206</td>
<td>11.6</td>
</tr>
<tr>
<td>Total Reserve</td>
<td>25</td>
<td>206</td>
<td>11.6</td>
</tr>
</tbody>
</table>

Cut-off grade of 160 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 Pre-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 reserve evaluation. This is particularly true when the profitability of a project is conditioned by the ability to mine selectively. Linear estimation methods such as Ordinary and Simple Kriging commonly fail to provide unbiased estimates of recovered ore and metal tonnage after cut-off, which means that a mining project can be exposed to undue risk.

Non-linear estimation techniques can then be used to generate a mineralisation geometry that is defined both by probability of occurrence and spatial distribution. The result is often considered more robust as the orebody definition is based on the 3D distribution of grade data, rather than on a deterministic manual interpretation along 2D sections.

The current resource models can be used for mine planning without additional dilution consideration:

- The use of the multiple indicator kriging (MIK) method emulating a SMU is a recoverable model where edge dilution and mining loss would only normally be accounted for.
- Internal dilution is adequately catered for with the inclusion of coherent runs of mineralised and sub-mineralised material.
- A soft domain approach has been used in the MIK process, which has tempered the tonnage model.

Ore loss is defined as ore dostoto waste due to operational issues like ore material dispatched erroneously to the waste dumps. The MIK grade estimation methodology does adequately apply for ore losses and it is envisaged that the project is operated to world class best practice 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 Xtract (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 15.4 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 15.4 Average mining costs applied for the pit optimisation exercise. |
|-----------------|-----|----------------|----------------|
| Cost element     | Unit| Valencia Main | Valencia Satellite | Namibplaas |
| Mining cost Ő reference level | US$/t | 1.55 | 1.55 | 1.55 |
| Depth factor     | US$/t/m | 0.0034 | 0.0034 | 0.0034 |
| Incremental ore mining cost | US$/t | 0.0 | 0.0629 | 0.2596 |

15.1.4. Processing cost

The processing costs applied for the pit optimisation are based on the AMEC processing plant trade-off study performed in May 2013.

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$ 4.57M was added to the processing costs per pit as summarised in Table 15.5.
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 was no recovery ramp-up profile assumed up to steady state. The overall process recovery for the Valencia ore material was assumed to be 85% and is not grade dependent while the Namibplaas ore has an overall processing recovery of 84%.

15.1.6. Uranium price, selling cost and royalties

A constant long term base price of US$65.00/lb of recovered U₃O₈ 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 22.1).

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.

Table 15.5 Processing unit costs.

<table>
<thead>
<tr>
<th>Cost</th>
<th>Unit</th>
<th>Valencia Main</th>
<th>Valencia Satellite</th>
<th>Namibplaas</th>
</tr>
</thead>
<tbody>
<tr>
<td>Processing cost excl rad sorting</td>
<td>US$/t ROM ore</td>
<td>7.84</td>
<td>7.84</td>
<td>7.84</td>
</tr>
<tr>
<td>Ore mining overhaul distance</td>
<td>km</td>
<td>0</td>
<td>2</td>
<td>7</td>
</tr>
<tr>
<td>Incremental ore mining cost</td>
<td>US$/t ROM ore</td>
<td>0</td>
<td>0.06</td>
<td>0.26</td>
</tr>
<tr>
<td>Tailings dump operation</td>
<td>US$/t ROM ore</td>
<td>0.06</td>
<td>0.06</td>
<td>0.06</td>
</tr>
<tr>
<td>Crusher rehandle</td>
<td>US$/t ROM ore</td>
<td>0.1</td>
<td>0.1</td>
<td>0.1</td>
</tr>
<tr>
<td>Services</td>
<td>US$/t ROM ore</td>
<td>0.20</td>
<td>0.20</td>
<td>0.20</td>
</tr>
<tr>
<td>Rehab &amp; closure cost</td>
<td>US$/t ROM ore</td>
<td>0.06</td>
<td>0.06</td>
<td>0.06</td>
</tr>
<tr>
<td>Whittle processing cost (excl. fixed costs)</td>
<td>US$/t ROM ore</td>
<td>8.26</td>
<td>8.32</td>
<td>8.52</td>
</tr>
</tbody>
</table>
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, but no designs have been performed. The designs were developed using Geovia’s Surpac mining software.

Figure 15.1 Mill limiting cut-off grade (excl fixed cost portion).

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 150t dump truck with a physical truck operating width of 6.6m.

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 150t Caterpillar 785 dump 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 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. The haul road gradient and width are discussed below.

**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.

**Haul road width**

The equipment study conducted concluded that the 150t 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 will 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 on 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 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.

**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 15.2 and the final pit layout in Figure 15.3. 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=660, and from this point a single ramp will be utilized for both ore and waste hauling going down to pit bottom z=300. This was done as to limit additional waste being mined. Pushback 1 and 2 are represented by Whittle shells and will still have to be designed as part of the next level of study.
Figure 15.2 Valencia pits with pushback configuration.

Figure 15.3 Final pit layouts for Valencia pits.

Table 15.6 Valencia main pits pushback dimensions.

<table>
<thead>
<tr>
<th>Description</th>
<th>Length (m)</th>
<th>Width (m)</th>
<th>Max depth (m)</th>
<th>Area (ha)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pushback 1</td>
<td>910</td>
<td>496</td>
<td>230</td>
<td>36</td>
</tr>
<tr>
<td>Pushback 2</td>
<td>1,110</td>
<td>770</td>
<td>300</td>
<td>62</td>
</tr>
<tr>
<td>Final Design</td>
<td>1,510</td>
<td>990</td>
<td>434</td>
<td>121</td>
</tr>
</tbody>
</table>
Figure 15.4 Isometric view of the orebody within Valencia pit designs (view NNE).

Table 15.7 Total pit inventory of the Valencia orebody.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Unit</th>
<th>Pushback 1</th>
<th>Pushback 2</th>
<th>Pushback 3</th>
<th>Final design</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total ore</td>
<td>Mt</td>
<td>27.5</td>
<td>34.7</td>
<td>83.7</td>
<td>145.9</td>
</tr>
<tr>
<td>Total waste</td>
<td>Mt</td>
<td>33.4</td>
<td>79.7</td>
<td>275.8</td>
<td>388.9</td>
</tr>
<tr>
<td>Total material mined</td>
<td>Mt</td>
<td>61.7</td>
<td>114.2</td>
<td>358.9</td>
<td>534.8</td>
</tr>
<tr>
<td>Stripping ratio</td>
<td></td>
<td>1.2</td>
<td>2.3</td>
<td>3.3</td>
<td>2.7</td>
</tr>
<tr>
<td>Average head grade</td>
<td>ppm</td>
<td>216.4</td>
<td>209.3</td>
<td>191.4</td>
<td>200.5</td>
</tr>
<tr>
<td>Metal U3O8 contained</td>
<td>M lbs</td>
<td>13.3</td>
<td>16.2</td>
<td>34.8</td>
<td>64.3</td>
</tr>
</tbody>
</table>

Valencia satellite

The Valencia satellite pit consists of a small single pushback north east of the main Valencia pit (Table 15.8). 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.

<table>
<thead>
<tr>
<th>Description</th>
<th>Length (m)</th>
<th>Width (m)</th>
<th>Max depth (m)</th>
<th>Area (ha)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Valencia satellite pit</td>
<td>690</td>
<td>370</td>
<td>160</td>
<td>16.4</td>
</tr>
</tbody>
</table>
Table 15.9 Total pit inventory of the Valencia satellite orebody.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Unit</th>
<th>Final design</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total ore</td>
<td>Mt</td>
<td>5.85</td>
</tr>
<tr>
<td>Total waste</td>
<td>Mt</td>
<td>21.88</td>
</tr>
<tr>
<td>Total material mined</td>
<td>Mt</td>
<td>27.74</td>
</tr>
<tr>
<td>Stripping ratio</td>
<td></td>
<td>3.7</td>
</tr>
<tr>
<td>Average head grade</td>
<td>ppm</td>
<td>244</td>
</tr>
<tr>
<td>Metal U3O8 contained</td>
<td>M lbs</td>
<td>3.1</td>
</tr>
</tbody>
</table>

15.2.3. Namibplaas pit

Namibplaas will have three pushbacks or phases illustrated in Figure 15.5 as well as a small pit to the north of the Main Namibplaas pit forming part of pushback 1. It also shows 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 15.5 Namibplaas pushback configuration and final layout design.](image)

Table 15.10 Namibplaas pushback dimensions.

<table>
<thead>
<tr>
<th>Description</th>
<th>Length (m)</th>
<th>Width (m)</th>
<th>Max depth (m)</th>
<th>Area (ha)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pushback 1</td>
<td>578</td>
<td>484</td>
<td>180</td>
<td>30.9</td>
</tr>
<tr>
<td>Pushback 2</td>
<td>845</td>
<td>440</td>
<td>230</td>
<td>31.7</td>
</tr>
<tr>
<td>Final Design</td>
<td>474</td>
<td>368</td>
<td>240</td>
<td>19.5</td>
</tr>
</tbody>
</table>

Figure 15.6 is an illustration of the Namibplaas pit design in conjunction with the ore above 200 ppm grade.
Figure 15.6 Isometric view of the orebody within Namibplaas final pit design. (view N)

Table 15.11 Total pit inventory of the Namibplaas orebody.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Unit</th>
<th>Pushback 1</th>
<th>Pushback 2</th>
<th>Pushback 3</th>
<th>Final design</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total ore</td>
<td>Mt</td>
<td>7.46</td>
<td>11.50</td>
<td>6.53</td>
<td>25.5</td>
</tr>
<tr>
<td>Total waste</td>
<td>Mt</td>
<td>34.3</td>
<td>53.0</td>
<td>34.1</td>
<td>121.4</td>
</tr>
<tr>
<td>Total material mined</td>
<td>Mt</td>
<td>42.1</td>
<td>64.3</td>
<td>40.5</td>
<td>146.9</td>
</tr>
<tr>
<td>Stripping ratio</td>
<td></td>
<td>4.64</td>
<td>4.61</td>
<td>5.23</td>
<td>4.77</td>
</tr>
<tr>
<td>Average head grade</td>
<td>ppm</td>
<td>249.7</td>
<td>207.0</td>
<td>228.6</td>
<td>206</td>
</tr>
<tr>
<td>Metal U₃O₈ contained</td>
<td>M lbs</td>
<td>2.9</td>
<td>4.9</td>
<td>3.8</td>
<td>11.6</td>
</tr>
</tbody>
</table>

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.

<table>
<thead>
<tr>
<th></th>
<th>Whittle Shell</th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Ore</td>
<td>Waste</td>
<td>Total</td>
<td>U₃O₈ Mlbs</td>
<td>SR</td>
</tr>
<tr>
<td>Valencia Main</td>
<td>146.99</td>
<td>356.17</td>
<td>503.16</td>
<td>65.07</td>
<td>2.42</td>
</tr>
<tr>
<td>Valencia East</td>
<td>5.84</td>
<td>19.60</td>
<td>25.45</td>
<td>3.11</td>
<td>3.35</td>
</tr>
<tr>
<td>Namibplaas</td>
<td>25.57</td>
<td>98.59</td>
<td>124.16</td>
<td>11.65</td>
<td>3.86</td>
</tr>
<tr>
<td>Ultimate Pit Design</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Ore</td>
<td>Waste</td>
<td>Total</td>
<td>U₃O₈ Mlbs</td>
<td>SR</td>
</tr>
<tr>
<td>Valencia Main</td>
<td>145.94</td>
<td>388.92</td>
<td>534.86</td>
<td>64.33</td>
<td>2.66</td>
</tr>
<tr>
<td>Valencia East</td>
<td>5.85</td>
<td>21.88</td>
<td>27.74</td>
<td>3.09</td>
<td>3.74</td>
</tr>
<tr>
<td>Namibplaas</td>
<td>25.46</td>
<td>121.40</td>
<td>146.86</td>
<td>11.58</td>
<td>4.77</td>
</tr>
</tbody>
</table>

### 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;
- amount 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 main process 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 allow 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, 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 68 Mtpa (Figure 15.7). Low grade and marginal material, although not considered ore as they fall below the strategic cut-off grade, are tracked as this represents additional profitable material that could be treated at a later stage.

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 15.8 indicates the total tonnes in relation to areas / pushbacks mined annually and also gives insight of the pushback release strategies.

Figure 15.7 Total tonnes mined according to material type.
Figure 15.8 Total tonnes mined according to mining location and pushbacks.

Figure 15.9 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 15.10.

Figure 15.9 Plant feed tonnes and grade.
15.3.4. Alternate case - 11.2 Mtpa process rate without radiometric sorting

This scenario incorporates a steady mining ramp up over 4 years to a maximum average annual production rate of 51 million tonnes (Figure 15.11). The various colours represent waste material as well ore material at different grades. The red line represents the strip ratio per period.

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 – non rad sort.

Figure 15.13 illustrates the annual tonnes fed to the crusher. The target feed to the plant was 11.2 Mt per annum, which was achieved in year 2 and maintained until the end of the life of mine. The plant feed balance is maintained with the high grade stockpile, whose material balance is illustrated in Figure 15.14. The stockpile serves as a buffer when there was a shortage in high grade ore as direct feed to the plant. The minimum stockpile size as shown below was 0.2 Mt but average around 2 Mt.

Figure 15.13 Plant feed tonnes and grade – non rad sort.
Figure 15.14 High grade stockpile movement – non rad sort.
16 MINING METHODS

The October 2013 resource models 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 4m 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 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, but 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.

<table>
<thead>
<tr>
<th>Pit Wall Sector</th>
<th>Inter ramp angle (°)</th>
<th>Bench stack angle (°)</th>
<th>Batter angle(°)</th>
<th>Batter height (m)</th>
<th>Berm width (m)</th>
<th>Overall slope angle (°) *</th>
</tr>
</thead>
<tbody>
<tr>
<td>Footwall</td>
<td>42</td>
<td>45</td>
<td>55</td>
<td>20</td>
<td>8</td>
<td>42</td>
</tr>
<tr>
<td>Hangingwall</td>
<td>55</td>
<td>58</td>
<td>70</td>
<td>20</td>
<td>7</td>
<td>52*</td>
</tr>
<tr>
<td>SW Wall</td>
<td>55</td>
<td>58</td>
<td>70</td>
<td>20</td>
<td>7</td>
<td>52*</td>
</tr>
<tr>
<td>NE Wall</td>
<td>51</td>
<td>54</td>
<td>65</td>
<td>20</td>
<td>7</td>
<td>51</td>
</tr>
</tbody>
</table>

* 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.

<table>
<thead>
<tr>
<th>Pit Wall Sector</th>
<th>Inter ramp angle (°)</th>
<th>Bench stack angle (°)</th>
<th>Batter angle(°)</th>
<th>Batter height (m)</th>
<th>Berm width (m)</th>
<th>Overall slope angle (°) *</th>
</tr>
</thead>
<tbody>
<tr>
<td>Footwall î Main Pit</td>
<td>53</td>
<td>56</td>
<td>70</td>
<td>20</td>
<td>8</td>
<td>51</td>
</tr>
<tr>
<td>Footwall î NE Ext.</td>
<td>42</td>
<td>45</td>
<td>55</td>
<td>20</td>
<td>8</td>
<td>41</td>
</tr>
<tr>
<td>Hangingwall</td>
<td>51</td>
<td>54</td>
<td>65</td>
<td>20</td>
<td>7</td>
<td>48</td>
</tr>
<tr>
<td>West Wall</td>
<td>54</td>
<td>58</td>
<td>70</td>
<td>20</td>
<td>7</td>
<td>51</td>
</tr>
</tbody>
</table>

* 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.

<table>
<thead>
<tr>
<th>Pit Wall Sector</th>
<th>Inter ramp angle (°)</th>
<th>Bench stack angle (°)</th>
<th>Batter angle(°)</th>
<th>Batter height (m)</th>
<th>Berm width (m)</th>
<th>Overall slope angle (°) *</th>
</tr>
</thead>
<tbody>
<tr>
<td>Footwall</td>
<td>39</td>
<td>41</td>
<td>50</td>
<td>20</td>
<td>8</td>
<td>40</td>
</tr>
<tr>
<td>Hangingwall</td>
<td>53</td>
<td>56</td>
<td>70</td>
<td>20</td>
<td>8</td>
<td>51*</td>
</tr>
<tr>
<td>South End Wall</td>
<td>49</td>
<td>52</td>
<td>65</td>
<td>20</td>
<td>8</td>
<td>48*</td>
</tr>
</tbody>
</table>

* based on a 300m high slope
* includes a 15m wide geotechnical catch berm @ 100m and 200m below ground surface
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 20m$^3$ buckets are considered the largest that should be considered. The typical class of machine would include 350t class excavators combined with 150t off-highway haul trucks. The following fleet complement is proposed to manage up to 68 Mtpa mining rate:

- excavators: four 350t class backhoe configuration with 20m$^3$ buckets;
- haul trucks: building up to 28 units of 150t class;
- bench drills: ten units capable of drilling 7 holes;
- pre-split drills: two units to drill;
- utility excavator: two 85t class;
- track dozer: four 60t units;
- wheel dozer: two 40t unit;
- FEL: one 15m$^3$ unit;
- grader: two 25t units;
- water bowser: three 55m$^3$ units;
- diesel bowser: two 35m$^3$ units;
- generators, trailer mounted for lighting and pumping;
- other, including lowbed, crane, forklift, tyre handler / tool carrier, service vehicles, rock breaker;
- 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 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 down-the-hole full blasting service.

The following summarises the 635 mining operation staffing for peak operations:

- equipment operators 369
- maintenance 195
- services & contractors 34
- mining management 39
17 RECOVERY METHODS

In January 2010, AMEC completed an Engineering Cost Study (ECS) on the Valencia Uranium Project the purpose of which was to update the 2008 DFS completed by AMEC South Africa. With the recent evaluation of additional mineral deposits close to the existing Valencia Main deposit, the reserves have now been significantly increased and warranted a review of the proposed plant design. Recent reviews of the 2008 DFS and ECS have identified alternative comminution flowsheet options which may potentially improve the Project value.

During 2009, additional testwork was performed with the aim of reducing the reagent consumptions and confirming the process flowsheet. This review included an ECS incorporating the latest improved testwork results, the aim being to further reduce operating and capital costs. The exercise employed as its starting point the DFS undertaken by AMEC South Africa incorporating the staged crush – rod mill circuit without radiometric sorting.

A two-stage crush plus single stage semi-autogenous grinding mill (SAG mill) using acidic filtrate at a grind of P80 ~600 μm delivered the highest overall value.

AMEC did not include the radiometric sorting as part of the process flow sheet as this would be treated as a modular retro fit to the plant at a later stage. Valencia has been dealing directly with the technology supplier relating to testwork and implementation. The final process design will make provision for the addition of the radiometric sorting following an initial evaluation period.

17.1 Processing facility

The proposed processing facility comprises the following unit operations:

- crushing, sorting, screening and stockpiles;
- milling;
- leaching;
- belt filtration;
- continuous ion exchange (CIX);
- solvent extraction (SX) and ammonium diuranate (ADU) recovery;
- filtration;
- calcination.

Radiometric sorting would be introduced at a later stage. The system will have to be designed based on real crusher product rock size distribution.

A process flow diagram is provided in Figure 17.1.

17.2 Crushing and comminution circuit

The crushing circuit comprises of primary crushing, followed by scalp screening and open circuit secondary crushing of the oversize. The crushed ore reports to the coarse ore stockpile from where it is reclaimed and fed to the SAG mill.

17.2.1 Primary crushing

The design tonnage is based on 14.9 Mtpa to be crushed via the primary crusher, subject to radiometric sorting for the elimination of a portion of barren / low grade material, thereby increasing the head grade and decreasing the volume for further treatment.

The feed size to the primary crusher is limited to a top size of 1,200 mm. The selected crusher is a gyratory crusher with a feed capacity of 3,600 tph. While this is in excess of the annual crushing tonnage requirements, it has been sized at this capacity due to mining requirements. The mining schedule requires that two truckloads of 150t be able to be crushed in five minutes. Note that the ROM bin will have two-sided truck tipping.
Once radiometric is implemented, the primary crusher product is fed via a conical stockpile to the sorting plant with a maximum rock size of 260mm.

### 17.2.2. Secondary crushing

The secondary crushing plant consists of a single secondary cone crusher in open circuit with three tertiary cone crushers in closed circuit as well as a sizing screen for classification. The F80 of the feed to this plant is 120mm. The final crusher product (classification screen underflow) is minus 32mm. The secondary crusher is sized to handle the design tonnage plus build an empty stockpile to full capacity with a specified time.

### 17.2.3. Stockpiles

While various stockpile options were reviewed, current designs have been based on conical configurations. With a height of 40m and base width of 85m, the live capacity of the stockpile is approximately 33,600t with an associated dead capacity of 95,700t. Additional area is available for dozing capacity to cater for times of extended crusher downtime for crusher liner change.

The stockpile base consists of 3 vibrating feeders onto a single conveyor.

### 17.2.4. Milling

The SAG mill is fed from the coarse ore stockpile. The mill feed conveyor has a straight approach to the SAG mill, to avoid maintenance issues associated with having feed conveyors over one of the twin pinion motor drives.

Slurry discharges via the trommel screen with the oversize material conveyed back to the mill feed conveyor. Undersize discharges into the mill discharge sump. To limit mill personnel interaction with the acidic mill discharge stream, the slurry is pumped to a classification screen near the ground-based section of the SAG mill feed conveyor. The screen oversize material is returned to the SAG mill feed conveyor and the screen undersize will report to the leach circuit.

This circuit incorporates grinding 1,400 t/h solids (at 70% w/w density) in acid liquor in a SAG mill with an 8.4m diameter and 5.64m grinding length. As the return of water that is acidic, the intention is to manufacture the mill of LDX 2101 stainless steel. Due to the acid consuming nature of the solids, most of the acid in the liquor used for grinding will quickly be consumed, with an expected mill discharge slurry pH of between 3 and 5. By returning acid liquor to the mill and not neutralising the liquor with lime or limestone, the added cost of these reagents is avoided offering a significant operating cost saving.

Grinding in acid liquor has been commercially applied at other operations.

### 17.3 Radiometric sorting

The radiometric sorting plant is designed to sort barren (or very low grade) material from uranium containing material. For the purpose of this study, it has been assumed that all rock of +32mm would be sorted, accounting for 69% of the crusher product. The grade of ore by size fraction indicates that the fines show an upgrade of about 10%.

The plant is designed for a 36% reject rate of sort feed, with an associated 13% loss of metal. Hence, with a 31% sort plant bypass, the overall reject rate will be 24.8% and 8.6% metal loss (Figure 17.2).

The sorter plant consists of four fine ore and five medium ore sorters in parallel. These are fed from the underflow of a vibrating screen at size -150mm. The screen oversize (+150mm) reports to two coarse ore sorters in parallel. The rejects from all the sorters could report to either waste dumps or the tailings disposal facility. The accepted ore portion reports to the secondary crushing plant.

It must be noted that as a result of an improved grade in the ore and the fact that it is difficult to design a radiometric sorter for a greenfields plant, the radiometric sorting plant will not be installed earlier than 2018. A cost benefit analysis (to determine a grade cut-point, offsetting ore rejection with metal loss) will be conducted during the initial production stage in order to assess the design parameters using real production data.
17.4 Leaching circuit

The design of the leach circuit for the Norasa Uranium includes an ambient sulphuric acid leach at a density of 48% solids and using MnO$_2$ (as pyrolusite) as the oxidant to oxidise ferrous iron to ferric iron. Alternative oxidants were evaluated and the density of the leach solution could vary from 48% to 65% based on the oxidant selected. A final decision on which oxidant will be used may still change.

The current layout caters for eleven leach tanks with a feed volumetric flow rate of 2030 m$^3$/h from 2,049m$^3$ capacity measuring 13.23m diameter and a 14.9m operating height. The total leach time is 10.1 hours.

17.5 Filtration and clarification

Belt filtration was considered as an alternative to conventional sand/slime separation and CCD washing. A high level comparison of these options was completed, using only belt filters for the leach tails washing and dewatering. Although slightly more expensive than the CCD options, the belt filters came out as the best technical solution.

The belt filters offer a much lower solution loss to tails than CCD, with the belt filter tails discharged as filter cake with a moisture content of approximately 20% (80% solids). The tails is then discharged and stored on a dry tails storage facility. The filters also allow for a smaller volume of solution to the IX circuit as a result of a smaller wash ratio required for washing. The uranium tenor in the solution to the IX will also be higher.

The belt filter installation consists of nine 149m$^2$ units. These are fed via gravity from the leach tail distribution box. Filtration rate is 1,040 kg/m$^2$h. Overall filter efficiency is expected to be 99.6%.

The final filtrate solution is fed to a single hopper clarifier. The solids content is reduced to approximately 20 ppm which is considered to be tolerable for the ion exchange process.

17.6 Ion exchange circuit

Because Norasa has a fairly low head grade (resulting in a fairly low uranium tenor in the pregnant leach solution (PLS)) an ion exchange circuit was selected for the initial upgrade of the PLS to a level that can be extracted by solvent extraction. A NIMCIX continuous ion exchange design was previously selected. NIMCIX (fluidised bed IX) was compared to an up-flow moving packed bed (MPBIX) configuration with resin-retaining screens. The MPBIX technology is widely used in the uranium industry and has approximately three times the up-flow velocity and so smaller columns are required, however, this option also requires an additional clarification stage upstream. MPBIX has the advantage of reduced resin inventory over the NIMCIX circuit.

Five loading columns and 5 elution columns are required.

17.7 Solvent extraction circuit

Conventional tertiary amine extraction with ammonia/ammonium sulphate stripping was selected for the Norasa flowsheet. Four extraction, three scrub and four strip stages were considered for the SX plant design.

17.8 Calcining

Conventional yellow cake precipitation with ammonia followed by centrifuge washing and calcination was selected for the base case Norasa flowsheet.

Industry standard values were used for the design of the ADU precipitation plant.
Figure 17.1 Plant block flow diagram.
Valencia Uranium (Pty) Ltd / Dunefield Mining Company (Pty) Ltd:
Norasa Uranium Project
NI 43-101 Technical Report

Figure 17.2 Radiometric sorting plant mass balance.
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

Valencia 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 m³/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.

In late 2011, the National Desalination Task Force (NDTF) announced it is proceeding with the tender process for a second desalination plant at Mile 6 (just north of Swakopmund). Tenders were closed at the end of June 2012 and three competitive proposals were received by the NDTF. The awarding of the tender has been delayed as government continues to negotiate with Areva for a medium term off-take agreement from their desalination plant, which currently has sufficient spare capacity to support the coastal mining industry over the near future. Once timing and commercial terms of this interim off-take have been finalised, the development of the new plant can be appropriately scheduled. There is currently a short term arrangement in place where the Areva plant is supplying water to NamWater pending the final agreement on the medium term agreement.

The initial plant capacity of the Mile 6 desalination plant will be 20 Mm³/annum, 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. The plant should be fully operational in 2 years of the awarding of the construction tender.

Most of this infrastructure will require upgrade to cater for Norasa and the expansion plans of other operations. Norasa has requested a water allocation of 3 Mm³ 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 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 30MW and a mine draw of about 24MW, 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.

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. 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.
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. Many of the internal service roads were also constructed.

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).

18.2 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 177 million dry tonnes of fine tailings over the LoM. The capacity of the planned facility is for 195 Mt (Figure 18.1).
- Two waste rock dumps storing collectively 411 million dry tonnes of open pit material for the Valencia pits. The capacities are 380 Mt for the Main Dump and 34 Mt in the smaller Dump A site, which is a small valley fill area on the NW side of the main pit (Figure 18.1).
- Two waste rock dumps storing collectively 122 million dry tonnes of open pit material for the Namibplaas pit. The capacities are 49 Mt in the smaller NP South Dump and 127 Mt in the NP North Dump. It is noted that there is substantially more capacity in the dumps than currently required in the mine plan, but the Namibplaas deposit also holds substantial potential for pit expansion (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.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value/Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>LoM</td>
<td>14 years</td>
</tr>
<tr>
<td>Total fine tailings*</td>
<td>177 million dry tonnes</td>
</tr>
<tr>
<td>Total rock waste Valencia</td>
<td>411 million dry tonnes</td>
</tr>
<tr>
<td>Total rock waste Namibplaas</td>
<td>122 million dry tonnes</td>
</tr>
<tr>
<td>Average <em>in situ</em> dry density of fine tailings</td>
<td>1.40 t/m³</td>
</tr>
<tr>
<td>Average <em>in situ</em> dry density of waste rock</td>
<td>2.00 t/m³</td>
</tr>
<tr>
<td>Solid particle specific gravity</td>
<td>2.63 t/m³</td>
</tr>
</tbody>
</table>

* includes the coarse reject material from the radiometric sorting, which may also be placed in the waste rock dumps.

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 are to be constructed using the open waste material, which is truck 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.

<table>
<thead>
<tr>
<th>Item</th>
<th>Capacity (m³)</th>
<th>Area (m²)</th>
<th>Elevation (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Tailings Dump</td>
<td>139.5 million</td>
<td>5.10 million</td>
<td>820</td>
</tr>
<tr>
<td>Valencia Dump Main</td>
<td>189.6 million</td>
<td>4.11 million</td>
<td>795</td>
</tr>
<tr>
<td>Valencia Dump A</td>
<td>17.1 million</td>
<td>0.51 million</td>
<td>720</td>
</tr>
<tr>
<td>NP Dump South</td>
<td>24.4 million</td>
<td>0.74 million</td>
<td>750</td>
</tr>
<tr>
<td>NP Dump North</td>
<td>63.4 million</td>
<td>1.53 million</td>
<td>760</td>
</tr>
</tbody>
</table>

No consideration is taken at this time to backfill mined out areas of the pits instead of 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%.

![Diagram of proposed dump locations](image)

**Figure 18.1 Proposed Valencia waste and tailings dump locations. (view NNE)**

![Diagram of proposed dump locations](image)

**Figure 18.2 Proposed Namibplaas waste dump locations. (view North)**

### 18.3 Product shipping

Contract with the customers for uranium shipping is not uniform, however most mines are responsible for the delivery of their product to the various uranium conversion facilities. If Norasa has adopted this approach to the sale of product; local transport, port charges (both sides), shipment and transport to the final destination and included the appropriate assumptions in the financial model. The cost of packaging has already been considered in the financial models under process cost miscellaneous items.

Product will be packaged in UN approved metal drums (manufactured locally in Namibia). These drums will contain between 350 to 400kg of uranium oxide and approximately 46 drums will be placed into a 20 foot shipping container. It is estimated that each container will handle 18 tonnes of product (40,000 pounds of U₃O₈).
Final product is generally not shipped directly to the mine’s customer, but rather to a customer-designated uranium conversion facility. Commercial facilities are located in North America or Europe. There are no unduly onerous requirements for the transport of uranium oxide in Namibia. Hence, Valencia proposes to transport the product containers by truck to the Port of Walvis Bay. The total road distance is approximately 140km.
19 MARKET STUDIES AND CONTRACTS

19.1 Worldwide Uranium Supply and Demand

The uranium market supply and demand fundamentals remain strong and support the need for additional uranium supply to come into production over the next few years. The result of the Fukushima disaster in Japan is that the previously predicted supply-demand deficit that was expected to occur in 2014 has been pushed back to 2016. The forecast demand for uranium in higher nuclear growth nations such as China, the Kingdom of Saudi Arabia, South Korea, India, Russia and the United Arab Emirates is expected to remain strong and supportive of strengthening of the uranium price over the medium and longer term.

Demand is relatively stable and predictable whilst future mining supply is in jeopardy. Global primary mine production currently supply 87% of demand for uranium. The balance of demand is supplied from secondary sources such as remaining excess commercial inventories, reprocessing of spent fuel, inventories held by governments and the blending of recycled highly-enriched uranium (HEU) from nuclear weapons programs. By far, the most significant of the secondary supplies was the approximate 24 Mlbs per year being provided from the HEU blending program. This is equivalent to 14% of current global demand for uranium. The HEU program terminated at the end of 2013. It is likely that the supply gap created by this termination will need to be replaced from new primary mine production in coming years.

One of the most respected uranium analysts, Dundee Capital Markets released a comprehensive report in February 2014. They noted that supply is a huge issue with not much supply growth over the next 5 years except for Cigar Lake (which is delayed), Husab (which is delayed) and Imouraren (which is very delayed). They report that the trends are clear; expecting 4 new mines in 2014 ìLost Creek (small), Nichols Ranch (small), Goliad (small) and Cigar Lake. More than counteracting this growth they reported that the following mines are closing ìLa Sal, Zarechnoye, Beaver, Pandorá, Daneros. Further, there are a number of deferred projects with a loss of up to 70 Mlbs annually, e.g. Olympic Dam, Trekkopje, Cameco double U, Langer Heinrich Stage 4, Kazakhstan. Dundee also mention no HEU supply ìlosing 24 Mlbs and unexpected disruptions ìRanger leach tank leak, Rössing leach tank leak, Cigar Lake mine and mill issues, Areva Niger negotiations with Government and the Areva Niger bombings.

They concluded:

- forecast a deficit of 7 Mlbs by 2016, and 16 Mlbs by 2020;
- forecast demand increasing from 169 Mlbs in 2013 to 223 Mlbs;
- forecast supply increasing from 198 Mlbs in 2013 to 207 Mlbs.

They predict a long-term uranium price of $65/lb. Other analysts including Cantor Fitzgerald and Raymond James are predicting $70/lb for contract prices and the consensus long-term price from 2016 is $68/lb (BMO Capital Markets ìFebruary 2014b).

On 1 February 2014 the World Nuclear Association reported there are 434 nuclear power plants operating worldwide, with 70 nuclear reactors under construction, 173 reactors planned worldwide and 310 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 follows that the nuclear renaissance will have a significant influence on future uranium demand and price.

Namibia is a major source of uranium, being the fifth largest producer in the world in 2013 from the established Rössing and Langer Heinrich uranium mines.

19.2 Marketing of Uranium

Uranium is typically sold by producers to utilities and operators of nuclear generating facilities principally through long-term contracts under which deliveries of U3O8 occur over a period greater than one year under agreements which include various pricing formulas and delivery terms. Historically, annual sales volumes in the spot market, where deliveries
normally occur within twelve months, have accounted for 10 to 15% of annual U₃O₈ sales volumes.

The long-term contract price for uranium negotiated between suppliers and utilities fluctuates within a narrow range, taking into consideration the need to ensure that producers require a certain price in order to support continued exploration for and development of new uranium deposits and earn a reasonable return on their investment. The long-term price of uranium was US$72.00 on December 31, 2006 (source: UxC Consulting) and rose to $95.00 on December 31, 2007 declining gradually to $70.00 on December 31, 2008 and further declining to $50.00 on December 2013 for reasons outlined above.

Valencia has not yet entered into any off-take contracts for the sale of uranium from the Norasa Project, but it is anticipated that uranium from Norasa will be sold under long-term contracts to one or more utilities.
ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The 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 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, under updated legislation in the Environmental Management Act, 2007, which became effective only in early 2012, the Clearance was renewed by the Environmental Commissioner to April 2016, and is renewable for 3-year periods. The EIA/EMP is being expanded 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.

20.2 Air Quality

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

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 600m³/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 m³/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 dewatering 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

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.

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 reuse, 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;

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.
21 CAPITAL AND OPERATING COSTS

Capital and operating acquisition and processing costs were updated during the first half of 2013, a period when the exchange rate between the Namibian and US Dollars fluctuated between N$8.5 to 10.0 to US$1.0. Since most of the costs originate in US$ or European based currencies and reporting is US$ based, the impact of the rate fluctuations on the overall project economics is not significant. Operating costs are deemed to be ±25% accuracy in US$ for the process plant and infrastructure. Mine operational and capital costs are at ±20%.

21.1 Capital costs

Run of mine production was based on an initial annual plant design capacity of 11.2M tonnes of ore derived from open pit operations to the crusher. The plant feed rate is planned to increase to 14.9 Mtpa after 2 years of operation with the introduction of radiometric sorting. This time period should allow sufficient opportunity to fully evaluate the crusher product from a rock size and grade distribution perspective to ensure adequate design and installation of the sorting system. A production level of up to 5.5 million pounds of product (uranium oxide) was modelled.

The majority of these costs are US$ based with some in N$. All costs have been converted to US Dollar costs for the purposes of this report, using an exchange rate of US$1.00 = N$10.00.

The capital cost estimates for the 11.2 Mt/a mill rate process plant and site infrastructure was developed on an area-by-area, discipline-by-discipline basis utilising equipment costs from vendors or the AMEC database from previous similar studies and projects. Original feasibility ECS prices were updated to 2013 pricing. Similarly, other direct costs have been largely estimated from preliminary quantities and rates, factored from equipment costs or derived from recent projects. Indirect costs assume an EPCM implementation strategy with a high level of owner involvement during project implementation phase.

Most of the implementation capital will be spent during a two year period, towards the end of which, mining will commence with the establishment of benches, initial waste stripping and ore stockpile generation.

21.1.1. Direct Costs

Direct capital costs including: mining equipment, bulk water supply, power supply, crushing, milling, leach, filtration and clarification, barren neutralisation, solvent extraction, ion exchange, ADU precipitation and thickening, product drying and packaging, ponds, services, reagents, plant & mining infrastructure, spares, first fills, commissioning and temporary facilities. The costs include permanent materials and equipment, freight to site, construction labour and equipment (including contractors' supervision, overheads and profit), temporary construction facilities, construction mobile equipment, and contractor mobilisation and demobilisation.

21.1.2. Indirect Costs

Indirect capital costs are the expenditures related to the engineering design, procurement, project management, site construction management and commissioning supervision by the EPCM contractor. Indirect costs also provide for consultants required to supplement design engineering and construction. The indirect costs include appropriate allowances for the EPCM contractor's overhead contribution.


The accuracy provisions reflect the level of definition available relating to the scope of work, process design, conceptual engineering design and cost data at the time of the capital estimate development and make appropriate allowances for uncertain elements of cost, for estimating errors and omission in quantification, thereby reducing the risk of cost variation within the required accuracy level.

The accuracy provisions are an integral component of the capital cost estimate and must be considered as part of the overall costs necessary for implementation of the project. This allowance is not intended to cover contingency issues such as change of scope, abnormal
or inclement weather, acts of God, industrial disturbances, foreign currency rates of exchange variations or escalation. Provision for these major undefined issues is excluded from this study.

The accuracy provisions were assessed at discipline level on a line-by-line basis to reflect the level of accuracy of material take-offs and design detail available at the time of the estimate and assessed to be 11.5% overall.

Elements allowed for in the accuracy provisions are:

- variance in material take-off quantities due to omissions and lack of detail definition;
- unforeseen variances in materials, bulk quantities and labour rates and difficulty level of the work required; and
- status of equipment costing (escalated budget pricing).

### Table 21.1 Process plant cost estimate.

<table>
<thead>
<tr>
<th>Description</th>
<th>Amount (US$M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Buildings</td>
<td>4.2</td>
</tr>
<tr>
<td>Process plant</td>
<td>186.3</td>
</tr>
<tr>
<td>Services and utilities</td>
<td>6.1</td>
</tr>
<tr>
<td>Reagents</td>
<td>8.8</td>
</tr>
<tr>
<td>Contractor mobilisation and demobilisation</td>
<td>4.4</td>
</tr>
<tr>
<td><strong>Direct Construction Costs</strong></td>
<td><strong>209.8</strong></td>
</tr>
<tr>
<td>First fills and spares</td>
<td>9.4</td>
</tr>
<tr>
<td>Temporary services and facilities</td>
<td>3.2</td>
</tr>
<tr>
<td><strong>Total Costs</strong></td>
<td><strong>222.4</strong></td>
</tr>
<tr>
<td>EPCM allowance</td>
<td>27.4</td>
</tr>
<tr>
<td><strong>Total Costs (Excluding Contingency)</strong></td>
<td><strong>249.7</strong></td>
</tr>
</tbody>
</table>

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-stripping 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. This figure does not include the initial operating costs.
Table 21.2 Total capital cost estimates.

<table>
<thead>
<tr>
<th>Capital Items</th>
<th>Estimated Cost US$ (millions)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Process plant</td>
<td>249.7</td>
</tr>
<tr>
<td>Mining equipment</td>
<td>75.7</td>
</tr>
<tr>
<td>Tailings disposal</td>
<td>8.3</td>
</tr>
<tr>
<td>Bulk water supply</td>
<td>21.2</td>
</tr>
<tr>
<td>Power supply &amp; substation</td>
<td>15.9</td>
</tr>
<tr>
<td>Owner’s buildings and equipment</td>
<td>18.3</td>
</tr>
<tr>
<td>Road / access</td>
<td>3.0</td>
</tr>
<tr>
<td><strong>Project Capital</strong></td>
<td><strong>392.1</strong></td>
</tr>
<tr>
<td>Expansion in radiometric sorting</td>
<td>39.9</td>
</tr>
<tr>
<td>Sustaining capital (life-of-mine)</td>
<td>60.3</td>
</tr>
</tbody>
</table>

21.2 Operating costs

Operating costs were determined based on estimates from first principles and contractor / supplier estimates, and are considered to be estimated to ±25% or better in some areas. Opportunities to reduce costs will be pursued as part of the feasibility study.

A summary of the key unit costs driving the OPEX is presented in Table 21.3.

Table 21.3 Key items in operating costs.

<table>
<thead>
<tr>
<th>Item</th>
<th>Units</th>
<th>Value</th>
<th>Comments</th>
</tr>
</thead>
<tbody>
<tr>
<td>Diesel</td>
<td>US$/litre</td>
<td>1.11</td>
<td></td>
</tr>
<tr>
<td>Power</td>
<td>US¢/kWh</td>
<td>10.56</td>
<td></td>
</tr>
<tr>
<td>Water</td>
<td>US$/m³</td>
<td>3.0</td>
<td></td>
</tr>
<tr>
<td>Sulphuric acid</td>
<td>US$/t</td>
<td>120</td>
<td>98% w/w H₂SO₄</td>
</tr>
<tr>
<td>Ammonia</td>
<td>US$/t</td>
<td>1,100</td>
<td></td>
</tr>
<tr>
<td>Pyrolusite</td>
<td>US$/t</td>
<td>271</td>
<td>38% w/w MnO₂</td>
</tr>
<tr>
<td>Ferrous sulphate</td>
<td>US$/t</td>
<td>360</td>
<td>90% w/w FeSO₄.7H₂O</td>
</tr>
<tr>
<td>SAG mill grinding media</td>
<td>US$/t</td>
<td>1,250</td>
<td></td>
</tr>
</tbody>
</table>

The base case is proposed with implementation of radiometric sorting after the first 2 years of production. This allows time to evaluate the run-of-mine plant feed material for particle size and grade distribution by rock size. The sorting circuit can then be designed to suit Norasa’s specific mineralisation, orebody and mining practices. The implementation of radiometric sorting results in a total mining rate of 68 Mtpa as indicated in the proposed schedule in Figure 15.7. The average life-of-mine unit costs are presented in Table 21.4 to Table 21.6. These costs are compared to a scenario in which radiometric sorting is not implemented, be it for economic or strategic reasons, where the peak mining rate is 51 Mtpa.
Table 21.4 Summary of plant operating costs (LoM).

<table>
<thead>
<tr>
<th>Details</th>
<th>Total Cost ($/t ROM)</th>
<th>Total Cost ($/lb U₃O₈)</th>
<th>Total Cost ($/t ROM)</th>
<th>Total Cost ($/lb U₃O₈)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Radiometric Sorting</td>
<td>No Radiometric Sorting</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Labour</td>
<td>0.33</td>
<td>0.89</td>
<td>0.42</td>
<td>1.06</td>
</tr>
<tr>
<td>Power</td>
<td>1.33</td>
<td>3.61</td>
<td>1.70</td>
<td>4.28</td>
</tr>
<tr>
<td>Reagents</td>
<td>2.74</td>
<td>7.45</td>
<td>3.51</td>
<td>8.85</td>
</tr>
<tr>
<td>Water</td>
<td>0.51</td>
<td>1.38</td>
<td>0.65</td>
<td>1.64</td>
</tr>
<tr>
<td>Consumables</td>
<td>0.69</td>
<td>1.87</td>
<td>0.88</td>
<td>2.22</td>
</tr>
<tr>
<td>Maintenance</td>
<td>0.34</td>
<td>0.91</td>
<td>0.43</td>
<td>1.08</td>
</tr>
<tr>
<td>Miscellaneous</td>
<td>0.06</td>
<td>0.17</td>
<td>0.08</td>
<td>0.20</td>
</tr>
<tr>
<td>Radiometric Sorting*</td>
<td>0.35</td>
<td>0.95</td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>6.35</strong></td>
<td><strong>17.23</strong></td>
<td><strong>7.68</strong></td>
<td><strong>19.33</strong></td>
</tr>
</tbody>
</table>

*Radiometric sorting is treated as a stand-alone module for costing purposes. Power, labour, maintenance, etc associated with rad sort are all inclusive into this line item.

Snowden Mining Industry Consultants (2013) developed a spread-sheet based mining cost model that has been used for estimating mining costs for a number of their studies. This model is appropriate for pre-feasibility level studies and pit optimisation exercises. Although the model is only suitable for life of mine average estimates and does not consider individual mining periods, incremental costs estimates can be produced and applied to mine cash-flow models over time. In mid 2013, Valencia obtained cost information from OEM suppliers to refine the cost model for operating conditions and environment appropriate for Norasa.

Table 21.5 Summary of mining operating costs by activity.

<table>
<thead>
<tr>
<th>Details</th>
<th>Total Cost ($/t mined)</th>
<th>Total Cost ($/lb U₃O₈)</th>
<th>Total Cost ($/t mined)</th>
<th>Total Cost ($/lb U₃O₈)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Radiometric Sorting</td>
<td>No Radiometric Sorting</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Loading</td>
<td>0.20</td>
<td>2.12</td>
<td>0.20</td>
<td>2.00</td>
</tr>
<tr>
<td>Hauling</td>
<td>0.71</td>
<td>7.63</td>
<td>0.71</td>
<td>7.11</td>
</tr>
<tr>
<td>Drilling</td>
<td>0.20</td>
<td>2.17</td>
<td>0.20</td>
<td>2.00</td>
</tr>
<tr>
<td>Blasting</td>
<td>0.34</td>
<td>3.63</td>
<td>0.34</td>
<td>3.40</td>
</tr>
<tr>
<td>Ancillary equipment</td>
<td>0.16</td>
<td>1.74</td>
<td>0.15</td>
<td>1.50</td>
</tr>
<tr>
<td>Overheads &amp; services</td>
<td>0.20</td>
<td>2.18</td>
<td>0.19</td>
<td>1.90</td>
</tr>
<tr>
<td><strong>Sub-Total</strong></td>
<td><strong>1.80</strong></td>
<td><strong>19.47</strong></td>
<td><strong>1.79</strong></td>
<td><strong>17.92</strong></td>
</tr>
<tr>
<td>Rom pad rehandling</td>
<td>0.16</td>
<td>0.15</td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>19.63</strong></td>
<td><strong>18.07</strong></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

The life-of-mine average cost per run-of-mine (ROM) tonne is US$14.06/t or $38.20/lb. These average unit costs are summarized by major cost items in Table 21.6. The highest unit cost is the mining cost, largely attributed to a stripping ratio of 3.0.

In the first 5 years of production, the average cash cost is only US$34.76/lb.
### Table 21.6 Operating cost (average unit price, Life of Mine).

<table>
<thead>
<tr>
<th>Operating cost Item</th>
<th>Total Cost ($/t ROM)</th>
<th>Total Cost ($/lb U₃O₈)</th>
<th>Total Cost ($/t ROM)</th>
<th>Total Cost ($/lb U₃O₈)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Radiometric Sorting</td>
<td>No Radiometric Sorting</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Mining (ore and waste)</td>
<td>7.23</td>
<td>19.63</td>
<td>7.17</td>
<td>18.07</td>
</tr>
<tr>
<td>Process</td>
<td>6.41</td>
<td>17.40</td>
<td>7.73</td>
<td>19.33</td>
</tr>
<tr>
<td>Tailings disposal</td>
<td>0.05</td>
<td>0.13</td>
<td>0.06</td>
<td>0.15</td>
</tr>
<tr>
<td>Overheads</td>
<td>0.44</td>
<td>1.20</td>
<td>0.58</td>
<td>1.45</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>14.06</strong></td>
<td><strong>38.20</strong></td>
<td><strong>15.48</strong></td>
<td><strong>39.01</strong></td>
</tr>
<tr>
<td>Closure and rehab provision</td>
<td>0.04</td>
<td>0.11</td>
<td>0.04</td>
<td>0.10</td>
</tr>
<tr>
<td><strong>Total first 5 years of prod.</strong></td>
<td><strong>13.68</strong></td>
<td><strong>34.76</strong></td>
<td><strong>14.65</strong></td>
<td><strong>35.05</strong></td>
</tr>
</tbody>
</table>

*Includes cost of ROM pad rehandling into the primary crusher.
**22 ECONOMIC ANALYSIS**

A financial model was prepared to assess the economics for Norasa based on the Mineral Reserve and mining schedule to report NPV, payback and IRR. The financial model quantifies the revenues, costs and capital expenditure over a 13-year life of mine. It is believed that these results are accurate to within ±25%, within the constraints of the associated assumptions. However, there exists additional opportunities to improve the project economics as the full value if not currently being realized.

NPVs were calculated on post-tax, uninflated cashflows at discount rates of 0%, 6% and 8%. The derived NPVs are summarised in Table 22.1. The IRR for the project is 36%. This demonstrates that the extraction of the reported Mineral Reserves could reasonably be justified. A long-term uranium price of US$68/lb is assumed.

The project has a payback period of 3 years after commencement of production.

<table>
<thead>
<tr>
<th>Discount Rate</th>
<th>NPV (US$ M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0%</td>
<td>851</td>
</tr>
<tr>
<td>6%</td>
<td>491</td>
</tr>
<tr>
<td>8%</td>
<td>410</td>
</tr>
</tbody>
</table>

The cashflow summary includes all capital costs to establish the project (Table 21.2). The cashflow shows a net positive cash flow of US$851M and a payback period of 3 years.

### 22.1 Royalties and taxes

The following taxation regime has been applied:

- State royalties of 3% on revenue (Government Gazette No. 4236, 1 April 2009, Government Notice No. 45) in accordance with the Minerals Act, Section 114(1)(c). The Act (Section 114(3)(b) makes provision for a deduction from the Royalty payable, the transportation cost (freight, road, rail, sea), harbour and storage fees, handling charges and insurance fees.
- Company taxation rate of 37.5%.
- Customs, duties and company or withholding taxes have not been considered at this time.
- An Export Levy has been proposed by the Minister of Finance during her 2014 / 15 Budget Speech in February 2014⁴. Although the rates on various goods have not yet been gazetted, it is believed that uranium concentrate (U₃O₈) will attract a tax of 0.2% on the value exported.
- All exploration costs (capital) can be carried forward from the year of expenditure until written off against profits.
- Development expenditures can be carried forward to no more than one-third of the cost annually for three years, starting during the first year that the mine commences. Assessed losses can be carried forward indefinitely. For simplicity, development capital has been deducted during the year of expenditure, with losses carried forward.

A value added tax (VAT) of 15% on domestic goods and services and 16.5% on imported goods and services. An exemption for imported project capital goods and services can be applied for with the Ministry of Finance for approval of zero rated VAT exemption. A refund on the 15% VAT on domestic goods and services is expected to be approved and the expected refund cycle period is estimated to be four months. The Customs Duties and Fees have various rates for the various categories of items imported into Namibia. An

---

exemption from the customs duties and fees for imported project capital goods and services can be applied for.

The costs of product handling, shipping and insurance to its final destination is deductible from the Mineral Royalty. It is estimated that the cost of product handling and logistics is US$0.60/lb. At this stage, 100% of this cost is refundable from, and to the limit of, the Royalty paid.

22.2 Financial model

The financial modelling is based on the following production assumptions:

- construction to commence in early 2015,
- plant commissioning by mid 2016,
- mining to commence in early 2016 to prepare pit benches and open mining faces,
- the mining rate steadily increase to 68 Mtpa in 2021,
- radiometric sorting is introduced in mid 2018, after 2 years of plant operation,
- nameplate plant capacity of obtained in 2019.

Economic parameters include:

- metal price of US$68/lb throughout the life of mine,
- as of end 2013, deferred exploration expenditures of US$32.9M for Valencia and US$7.3M for Namibplaas are capital deductible and included in the model.

The production schedule is described in Section 15.3 and forms the basis for the economic model with consideration of the above development schedule. The resulting annual uranium production is given in Figure 22.1. A summary of the main input parameters of production, capital and operating costs and taxation is provided in Table 22.2.

For comparison, a similar model was developed for a scenario without implementing radiometric sorting (Table 22.3); a similar project NPV was reported. This comparison illustrates that the decision to implement radiometric sorting will likely be a strategic one, dictated by the economics at the time. It is noted that radiometric sorting is a modular system and can be retro fitted in stages a unit at a time depending on throughput characteristics.

![Figure 22.1 Planned uranium production for the two production scenarios.](image-url)
## Table 22.2 Valencia economic model with radiometric sorting.

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</tr>
</thead>
<tbody>
<tr>
<td><strong>Revenue (US$ '000)</strong></td>
<td>15,000</td>
<td>152,233</td>
<td>197,564</td>
<td>27,255</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>392,053</td>
<td>4,437,703</td>
</tr>
<tr>
<td><strong>Operating costs (US$ '000)</strong></td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>900</td>
<td>101,637</td>
</tr>
<tr>
<td><strong>Taxes (US$ '000)</strong></td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>392,053</td>
<td>4,437,703</td>
</tr>
<tr>
<td><strong>Cumulative cash flow (US$ '000)</strong></td>
<td>-15,000</td>
<td>-167,233</td>
<td>-295,126</td>
<td>-393,990</td>
<td>-479,016</td>
<td>-572,867</td>
<td>-657,863</td>
<td>-751,875</td>
<td>-849,002</td>
<td>-915,743</td>
<td>-984,484</td>
<td>-1,075,971</td>
<td>-1,275,690</td>
<td>835,538</td>
</tr>
</tbody>
</table>

NPV @ 0% (US$ '000): 850,538
NPV @ 6% (US$ '000): 491,263
NPV @ 8% (US$ '000): 410,365

Post-tax IRR: 36.0%
Table 22.3 Valencia economic model without radiometric sorting.

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</thead>
<tbody>
<tr>
<td>Ore mined (000 tonnes)</td>
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<td></td>
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</tr>
<tr>
<td>Valencia Main</td>
<td>5,802</td>
<td>9,545</td>
<td>13,341</td>
<td>5,463</td>
<td>13,038</td>
<td>11,187</td>
<td>11,201</td>
<td>8,299</td>
<td>10,796</td>
<td>13,430</td>
<td>43,838</td>
<td>145,940</td>
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<td></td>
</tr>
<tr>
<td>Valencia Sat</td>
<td>0</td>
<td>741</td>
<td>1,770</td>
<td>3,344</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>5,865</td>
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<tr>
<td>Namibplaas</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>25,498</td>
<td></td>
<td>25,498</td>
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<tr>
<td>Waste mined (000 tonnes)</td>
<td>12,694</td>
<td>21,827</td>
<td>26,400</td>
<td>40,410</td>
<td>37,259</td>
<td>39,371</td>
<td>39,576</td>
<td>41,993</td>
<td>39,465</td>
<td>36,947</td>
<td>196,707</td>
<td>532,652</td>
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</tr>
<tr>
<td>Strip ratio</td>
<td>0.00</td>
<td>0.00</td>
<td>2.19</td>
<td>2.12</td>
<td>1.75</td>
<td>4.59</td>
<td>2.86</td>
<td>3.52</td>
<td>3.53</td>
<td>5.06</td>
<td>3.66</td>
<td>2.75</td>
<td>2.84</td>
<td>3.00</td>
</tr>
<tr>
<td>Ore crushed (000 tonnes)</td>
<td>4,663</td>
<td>11,202</td>
<td>11,200</td>
<td>11,201</td>
<td>11,213</td>
<td>11,200</td>
<td>11,200</td>
<td>11,200</td>
<td>11,213</td>
<td>11,200</td>
<td>11,213</td>
<td>71,766</td>
<td>177,293</td>
<td></td>
</tr>
<tr>
<td>Ore milled (000 tonnes)</td>
<td>4,663</td>
<td>11,202</td>
<td>11,200</td>
<td>11,201</td>
<td>11,213</td>
<td>11,200</td>
<td>11,200</td>
<td>11,200</td>
<td>11,213</td>
<td>11,200</td>
<td>11,213</td>
<td>71,766</td>
<td>177,293</td>
<td></td>
</tr>
<tr>
<td>Stockpiles (000 tonnes)</td>
<td></td>
<td></td>
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<td></td>
<td></td>
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</tr>
<tr>
<td>Ore</td>
<td>1,139</td>
<td>224</td>
<td>4,135</td>
<td>1,740</td>
<td>3,548</td>
<td>3,535</td>
<td>3,536</td>
<td>635</td>
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<td>2,431</td>
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<td>Low grade</td>
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<td>0</td>
<td>0</td>
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<td>0</td>
<td>16,660</td>
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<tr>
<td>Marginal</td>
<td>2,728</td>
<td>6,757</td>
<td>11,319</td>
<td>15,038</td>
<td>20,287</td>
<td>24,779</td>
<td>28,759</td>
<td>32,913</td>
<td>36,592</td>
<td>45,594</td>
<td>85,723</td>
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<td></td>
<td></td>
</tr>
<tr>
<td>Product sold (000 U3O8)</td>
<td>1,789</td>
<td>4,544</td>
<td>5,114</td>
<td>4,755</td>
<td>4,481</td>
<td>4,445</td>
<td>4,605</td>
<td>4,749</td>
<td>4,381</td>
<td>4,208</td>
<td>27,284</td>
<td>70,355</td>
<td></td>
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</tr>
<tr>
<td>Revenue (US$ '000)</td>
<td>121,625</td>
<td>309,003</td>
<td>347,756</td>
<td>323,313</td>
<td>304,702</td>
<td>302,233</td>
<td>313,159</td>
<td>322,961</td>
<td>297,904</td>
<td>286,155</td>
<td>1,855,315</td>
<td>4,784,125</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Capital (US$ '000)</td>
<td>15,000</td>
<td>152,233</td>
<td>193,936</td>
<td>18,177</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>379,345</td>
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<tr>
<td>Operating costs (US$ '000)</td>
<td>27,998</td>
<td>50,653</td>
<td>68,983</td>
<td>77,806</td>
<td>83,736</td>
<td>83,749</td>
<td>87,547</td>
<td>85,694</td>
<td>90,694</td>
<td>97,261</td>
<td>507,570</td>
<td>1,261,091</td>
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<td></td>
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<tr>
<td>Taxes (US$ '000)</td>
<td>3,649</td>
<td>9,270</td>
<td>10,433</td>
<td>9,690</td>
<td>9,141</td>
<td>9,067</td>
<td>9,395</td>
<td>9,689</td>
<td>8,937</td>
<td>8,585</td>
<td>55,659</td>
<td>143,524</td>
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</tr>
<tr>
<td>Export Levy</td>
<td>243</td>
<td>618</td>
<td>696</td>
<td>647</td>
<td>609</td>
<td>604</td>
<td>626</td>
<td>646</td>
<td>596</td>
<td>572</td>
<td>3,711</td>
<td>9,568</td>
<td></td>
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</tr>
<tr>
<td>Company tax</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>24,749</td>
<td>41,295</td>
<td>42,099</td>
<td>44,919</td>
<td>48,379</td>
<td>51,313</td>
<td>251,312</td>
<td>510,395</td>
<td></td>
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</tr>
<tr>
<td>Post-tax cash flow (US$ '000)</td>
<td>-15,000</td>
<td>-152,233</td>
<td>-125,862</td>
<td>166,817</td>
<td>181,283</td>
<td>103,813</td>
<td>68,513</td>
<td>71,257</td>
<td>78,150</td>
<td>83,389</td>
<td>50,380</td>
<td>44,203</td>
<td>376,763</td>
<td>910,552</td>
</tr>
<tr>
<td>Cumulative cash flow (US$ '000)</td>
<td>-15,000</td>
<td>-167,233</td>
<td>-293,115</td>
<td>-127,298</td>
<td>33,985</td>
<td>137,796</td>
<td>206,311</td>
<td>277,668</td>
<td>365,816</td>
<td>439,207</td>
<td>489,587</td>
<td>533,789</td>
<td>910,552</td>
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<tr>
<td>NPV @ 0% (US$ '000)</td>
<td>925,552</td>
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<tr>
<td>NPV @ 6% (US$ '000)</td>
<td>497,138</td>
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<tr>
<td>NPV @ 8% (US$ '000)</td>
<td>407,459</td>
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<tr>
<td>Post-tax IRR</td>
<td>34.9%</td>
<td></td>
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</table>
22.3 Sensitivity analyses

A series of sensitivity analyses has been carried out on the major parameters of Norasa. Uranium price, site operating costs and capital expenditure, were all varied by ±15%. In each case the cash flows post-tax NPVs at 8% real per annum were evaluated. The following star chart (Figure 22.2) shows the results of the sensitivity analysis on the major parameters for the case including radiometric sorting.

![Sensitivity chart showing impact of changes in revenue, opex, and capex on NPV](image.png)

Figure 22.2 Sensitivity of variables to NPV (US$) at a discount rate of 8%.

22.4 Upside

A stockpile of low grade material at Namibplaas with 17 Mt at 127 ppm has been built up by the end of the mine life. Although this material did not meet the initial strategic criteria for reserves, it is payable and could be treated at the end of the mine life, producing an additional 3.6 Mlbs of saleable product.

Additional upside also exists in the 50 Mt of Inferred material within and around all pits, which is not considered in the pit optimisation process or the financial modelling. Potential for inclusion of this material will only be confirmed once operations further evaluate these areas with ongoing drilling programs.
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 company 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 uraniumiferous 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 (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 U₃O₈ per year. Table 23.3 provides recent information on the operation until 2012.

**Table 23.1: Mineral Resource Estimate – Rössing (end 2012).**

<table>
<thead>
<tr>
<th>Classification</th>
<th>Mt</th>
<th>Grade % U₃O₈</th>
<th>Mlbs U₃O₈</th>
</tr>
</thead>
<tbody>
<tr>
<td>Measured Resources</td>
<td>15</td>
<td>0.026</td>
<td>8.6</td>
</tr>
<tr>
<td>Indicated Resources</td>
<td>148</td>
<td>0.024</td>
<td>78.3</td>
</tr>
<tr>
<td>Measured &amp; Indicated</td>
<td>163</td>
<td>0.024</td>
<td>86.9</td>
</tr>
<tr>
<td>Inferred Resources</td>
<td>173</td>
<td>0.026</td>
<td>99.2</td>
</tr>
</tbody>
</table>

Note that resources are quoted exclusive of reserves. Cut-off grade of 0.010% U₃O₈.

**Table 23.2: Ore Reserves Estimate – Rössing (end 2012).**

<table>
<thead>
<tr>
<th>Classification</th>
<th>Mt</th>
<th>Grade % U₃O₈</th>
<th>Mlbs U₃O₈</th>
</tr>
</thead>
<tbody>
<tr>
<td>Proved Ore Reserve</td>
<td>29</td>
<td>0.031</td>
<td>19.8</td>
</tr>
<tr>
<td>Probable Ore Reserve</td>
<td>102</td>
<td>0.035</td>
<td>78.7</td>
</tr>
<tr>
<td>Total Ore Reserve</td>
<td>131</td>
<td>0.034</td>
<td>98.5</td>
</tr>
</tbody>
</table>

Cut-off grade of 0.010% U₃O₈.

Table 23.3: Performance data of the Rössing Uranium Mine<sup>6</sup>.

<table>
<thead>
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<th></th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>The employees of Rössing Uranium</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Number of employees</td>
<td>1,528</td>
<td>1,637</td>
<td>1,592</td>
<td>1,415</td>
<td>1,307</td>
<td>1,175</td>
<td>939</td>
<td>860</td>
</tr>
<tr>
<td><strong>Production</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>U&lt;sub&gt;2&lt;/sub&gt;O&lt;sub&gt;5&lt;/sub&gt; produced (tonnes)</td>
<td>2,699</td>
<td>2,148</td>
<td>3,628</td>
<td>4,150</td>
<td>4,108</td>
<td>3,046</td>
<td>3,617</td>
<td>3,711</td>
</tr>
<tr>
<td>Ore processed (1000 tonnes)</td>
<td>12,127</td>
<td>10,729</td>
<td>22,598</td>
<td>12,633</td>
<td>12,858</td>
<td>12,613</td>
<td>12,008</td>
<td>12,027</td>
</tr>
<tr>
<td>Waste rock removed (1000 tonnes)</td>
<td>31,737</td>
<td>39,913</td>
<td>41,955</td>
<td>38,755</td>
<td>33,899</td>
<td>21,396</td>
<td>16,835</td>
<td>7,483</td>
</tr>
<tr>
<td>Ratio of ore processed to waste rock removed</td>
<td>0.38</td>
<td>0.27</td>
<td>0.28</td>
<td>0.33</td>
<td>0.38</td>
<td>0.59</td>
<td>0.71</td>
<td>1.61</td>
</tr>
<tr>
<td><strong>Health, safety and environment</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>No. of cases of personal annual radiation exposure above 20 mSv</td>
<td>1</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>New cases of occupational pneumoniconiosis</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>1</td>
<td>1</td>
<td>0</td>
</tr>
<tr>
<td>New cases of occupational dermatitis</td>
<td>3</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>New cases of occupational hearing loss</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>New cases of occupational chronic bronchitis</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Lost-time injury incident rate</td>
<td>0.49</td>
<td>0.81</td>
<td>0.89</td>
<td>0.73</td>
<td>0.91</td>
<td>0.71</td>
<td>0.35</td>
<td>0.89</td>
</tr>
<tr>
<td>No. of lost-time Injuries</td>
<td>4</td>
<td>11</td>
<td>14</td>
<td>6</td>
<td>8</td>
<td>9</td>
<td>6</td>
<td>8</td>
</tr>
<tr>
<td>Fresh water consumption (1000 m&lt;sup&gt;3&lt;/sup&gt;)</td>
<td>3,103</td>
<td>3,060</td>
<td>2,870</td>
<td>3,131</td>
<td>3,700</td>
<td>3,000</td>
<td>3,315</td>
<td>3,170</td>
</tr>
<tr>
<td>Fresh water per tonne of ore processed (m&lt;sup&gt;3&lt;/sup&gt;/t)</td>
<td>0.26</td>
<td>0.29</td>
<td>0.25</td>
<td>0.25</td>
<td>0.29</td>
<td>0.26</td>
<td>0.28</td>
<td>0.27</td>
</tr>
<tr>
<td>Ratio of fresh water to total water</td>
<td>0.38</td>
<td>0.39</td>
<td>0.31</td>
<td>0.33</td>
<td>0.36</td>
<td>0.32</td>
<td>0.35</td>
<td>0.37</td>
</tr>
<tr>
<td>Seepage water collected (1000 m&lt;sup&gt;3&lt;/sup&gt;)</td>
<td>2,387</td>
<td>2,349</td>
<td>2,680</td>
<td>2,879</td>
<td>2,740</td>
<td>3,050</td>
<td>2,736</td>
<td>2,018</td>
</tr>
<tr>
<td>Energy use on site (1000 GJ)</td>
<td>1,852</td>
<td>1,897</td>
<td>1,996</td>
<td>2,168</td>
<td>1,812</td>
<td>1,534</td>
<td>1,366</td>
<td>1152</td>
</tr>
<tr>
<td>Energy use per tonne of ore processed (MJ/t)</td>
<td>153.03</td>
<td>182.90</td>
<td>172.1</td>
<td>140.9</td>
<td>121.6</td>
<td>113.7</td>
<td>95.6</td>
<td></td>
</tr>
<tr>
<td>CO&lt;sub&gt;2&lt;/sub&gt; emission (Kt CO&lt;sub&gt;2&lt;/sub&gt; equivalent)</td>
<td>211.6</td>
<td>208.08</td>
<td>221.0</td>
<td>243.2</td>
<td>222.6</td>
<td>197.0</td>
<td>181.2</td>
<td>161.0</td>
</tr>
<tr>
<td>CO2 emission per unit of production (CO&lt;sub&gt;2&lt;/sub&gt; t/t U)</td>
<td>78.41</td>
<td>97.37</td>
<td>60.70</td>
<td>58.60</td>
<td>54.20</td>
<td>64.7</td>
<td>50.1</td>
<td>43.4</td>
</tr>
<tr>
<td>Source dust levels at Fine-crushing Plant (mg/m&lt;sup&gt;3&lt;/sup&gt;)</td>
<td>2.35</td>
<td>2.55</td>
<td>4.02</td>
<td>2.33</td>
<td>1.52</td>
<td>0.93</td>
<td>1.49</td>
<td>1.12</td>
</tr>
<tr>
<td><strong>Product and customers</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Uranium spot market price (US$/lb)</td>
<td>48.70</td>
<td>56.75</td>
<td>46.00</td>
<td>46.00</td>
<td>61.00</td>
<td>72.00</td>
<td>36.25</td>
<td>36.25</td>
</tr>
</tbody>
</table>

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 deepwater 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.

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 carnitite, 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 carnitite occurs as thin films lining cavities and fracture planes and as grain coatings and disseminations in the calcretised sediments.

<sup>6</sup> www.rossing.com
Mineralisation is near surface, 1m to 30m thick and is 50m to 1,100m wide depending on
the width of the palaeovalley.

Following calcritisation 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. 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 23.4: Mineral Resource Estimate – Langer Heinrich (June 2013).

<table>
<thead>
<tr>
<th>Classification</th>
<th>Mt</th>
<th>Grade % U3O8</th>
<th>t U3O8</th>
<th>Mlbs U3O8</th>
</tr>
</thead>
<tbody>
<tr>
<td>Measured Resources</td>
<td>25.3</td>
<td>0.055</td>
<td>13,851</td>
<td>30.54</td>
</tr>
<tr>
<td>Indicated Resources</td>
<td>70.1</td>
<td>0.055</td>
<td>38,729</td>
<td>85.38</td>
</tr>
<tr>
<td>Measured &amp; Indicated</td>
<td>95.5</td>
<td>0.055</td>
<td>52,580</td>
<td>115.92</td>
</tr>
<tr>
<td>Stockpiles</td>
<td>28.6</td>
<td>0.042</td>
<td>11,932</td>
<td>26.31</td>
</tr>
<tr>
<td>Inferred Resources</td>
<td>17.8</td>
<td>0.06</td>
<td>10,335</td>
<td>22.9</td>
</tr>
</tbody>
</table>

Note that resources are quoted inclusive of reserves.

Table 23.5: Ore Reserve Estimate – Langer Heinrich (June 2013).

<table>
<thead>
<tr>
<th>Classification</th>
<th>Mt</th>
<th>Grade % U3O8</th>
<th>t U3O8</th>
<th>Mlbs U3O8</th>
</tr>
</thead>
<tbody>
<tr>
<td>Proved Ore Reserve</td>
<td>20.0</td>
<td>0.055</td>
<td>11,093</td>
<td>24.46</td>
</tr>
<tr>
<td>Probable Ore Reserve</td>
<td>59.4</td>
<td>0.057</td>
<td>33,616</td>
<td>74.11</td>
</tr>
<tr>
<td>Stockpiles</td>
<td>28.6</td>
<td>0.042</td>
<td>11,932</td>
<td>26.31</td>
</tr>
<tr>
<td>Total Ore Reserve</td>
<td>108.1</td>
<td>0.052</td>
<td>56,642</td>
<td>124.87</td>
</tr>
</tbody>
</table>

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.

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 UO4.

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. 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 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 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.

<table>
<thead>
<tr>
<th>Classification</th>
<th>Mt</th>
<th>Grade ppm U₃O₈</th>
<th>Mlbs U₃O₈</th>
</tr>
</thead>
<tbody>
<tr>
<td>Measured Resources</td>
<td>74</td>
<td>510</td>
<td>84</td>
</tr>
<tr>
<td>Indicated Resources</td>
<td>281</td>
<td>440</td>
<td>274</td>
</tr>
<tr>
<td>Measured &amp; Indicated</td>
<td>355</td>
<td>455</td>
<td>358</td>
</tr>
<tr>
<td>Inferred Resources</td>
<td>175</td>
<td>340</td>
<td>130</td>
</tr>
</tbody>
</table>

Note that resources are quoted inclusive of reserves.

<table>
<thead>
<tr>
<th>Classification</th>
<th>Mt</th>
<th>Grade ppm U₃O₈</th>
<th>Mlbs U₃O₈</th>
</tr>
</thead>
<tbody>
<tr>
<td>Proved Ore Reserve</td>
<td>62.7</td>
<td>569</td>
<td>78.7</td>
</tr>
<tr>
<td>Probable Ore Reserve</td>
<td>217.3</td>
<td>504</td>
<td>241.2</td>
</tr>
<tr>
<td>Total Ore Reserve</td>
<td>280.0</td>
<td>518</td>
<td>319.9</td>
</tr>
</tbody>
</table>

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

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6 Extract Resources, 7 June 2011, ASX Media Release, Husab Project Established as the 4th Largest Uranium Deposit in the World.
7 Extract Resources, 10 August 2011, ASX Media Release, 37% Increase in Reserves at Husab
solvent extraction process upgrading and refinement. The uranium is then precipitated and calcined to produce the final product of uranium oxide. However, the exact details of this process are not provided in the text.

http://swakopuranium.com/
24 OTHER RELEVANT DATA AND INFORMATION

The Namibplaas EPL 3638 was due to expire on 6 November 2013 (previous renewal was granted for 2 years as guided by the Minerals Act). A renewal application was submitted to the MME on 6 August 2013. The Ministry had subsequently requested Dunefield to propose a reduced for the EPL area in accordance with Section 72(2) of the Minerals Act. The total surface area of 1,742 ha was reduced to 1,269 ha. The new area is shown in Figure 24.1 with co-ordinates listed in Table 24.1.

![Figure 24.1 Revised EPL area (Renewal application August 2013).](image)

**Table 24.1 EPL corner beacon coordinates in Lat/Long.**

<table>
<thead>
<tr>
<th>Beacon</th>
<th>Lat</th>
<th>Long</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>-22.301055</td>
<td>15.241229</td>
</tr>
<tr>
<td>B</td>
<td>-22.265362</td>
<td>15.290484</td>
</tr>
<tr>
<td>C</td>
<td>-22.309980</td>
<td>15.291275</td>
</tr>
</tbody>
</table>

The impact in this possible change in area in relation to the Valencia project is shown on a map in Figure 24.2. Confirmation of the new co-ordinates and EPL area will only be known once Dunefield has received confirmation of the renewal. Note that this is expected at any time, but has not yet been received as of the writing of this section of the report.

The reduced area, if so granted in the renewal, will not impact on the Mineral Resource and Reserve or restrict the company’s ability to develop the project.
Figure 24.2 EPL 3638 location and its position in relation to Valencia.
25 INTERPRETATION AND CONCLUSIONS

25.1 Testwork & Plant Design

The ECS completed by AMEC in May 2013 is based on interpolating testwork that was conducted under various leach conditions, but no tests had been specifically conducted for the preferred two-stage crush and SAG milling. Subsequent to the completion of the ECS report, additional testwork confirmed that higher recoveries could be achieved, supporting the results from other work. These higher recoveries were subsequently used in the financial modelling.

The estimate for costs associated with the revised site location and layout is based on the original design work conducted in 2009. No cost estimate adjustment has been made for new layouts relating to the SAG milling option.

25.2 Geology and Mineral Resource

Across the Norasa deposits, significant quantities of uranium bearing alaskite sheets are occurring either as dykes or sills, ranging in size from several centimetres to +100m in thickness. They are generally situated in and around anticlinal and dome structures and intrude into the basement, the Nosib Group and lower Swakop Group, along the prominent marbles of the Rössing and Karibib Formations. They also occur along the Abbabis Swell and often are associated with a Red Granite Suite. These occurrences provide an upside potential as none have been assessed in detail to date due their complexity and therefore have been excluded in the current assessment of Norasa.

The Resource evaluation is based on nearly a decade of evaluation work in the Damaran hosted alaskites with extensive drilling and sampling work having matured to a level of understanding suitable for feasibility work.

Further mineral resource upside exists in the possible conversion of approximately 50 million tonnes of inferred material.

Norasa is positioned in line with other significant primary uranium producers, such as the emerging Husab Mine and Rio Tinto’s operating Rössing Uranium Mine, which has been in production since 1976.

25.3 Mineral Reserve

The Norasa uranium deposit has demonstrated economic viability via conventional open pit mining methods using truck and shovel equipment. Limitations in the metallurgical testing are not deemed material to the project’s viability but future work is likely to lead to some change the project’s value, either slightly higher or lower.

The prescribed cut-off grades of 100 ppm at Valencia and 160 ppm at Namibplaas are higher than the marginal cut-off grade of approximately 70 ppm (this being dependent on the uranium price). As such, the project would have a higher NPV if all potentially economic material were treated in a suitably designed plant. In addition, Inferred material both inside and outside the pits (22 Mlbs of U₃O₈), has the potential to add, at least in part, to the Reserves once operations are underway. The potential economic benefit from the Inferred material has not been considered at this time.

25.4 Economics

Several major infrastructure items have not been reviewed since the original estimations were reported in early 2010, however all major capital costs have been escalated to 2013 dollars. Some external circumstances and site designs have changed, which will impact on the concepts behind the infrastructure strategy. Key items are electricity supply, source of bulk water, waste dump capacities and surface water management.

Closure and rehabilitation requirements will have to be reassessed for the larger Norasa Project.

Norasa is presented assuming that radiometric sorting is implemented to upgrade the ore feed into the leaching circuit. Although there is risk associated with this system as it remains untested at full scale, the risk is greatly mitigated as it is planned to be
implemented after an initial evaluation period once the main process plant is in full production and the overall project economics and viability are not dependent on the radiometric sorting being implemented.

25.5 Summary

Norasa represents an advanced stage, permitted uranium project with strong economics supporting a viable mining operation.
26 RECOMMENDATIONS

The report satisfies the level of work required, at a level of accuracy consistent with a pre-feasibility study. It is recommended that these studies be upgraded to feasibility study compliance to achieve a ±15% accuracy), and dependant on the level of funding, complete documentation that satisfies the requirements for a bankable feasibility study. To achieve this, the following steps are recommended:

- review the current status of all metallurgical testwork and propose any additional work that may be required to ensure that the plant design is fully supported;
- confirm some aspects of the preferred process flowsheet and upgrade the plant design to the required level of accuracy;
- upgrade the pit designs for the interim stages. Ramp and haul roads could also be optimised for dump and crusher locations. Haul truck routes should be modelled by time period to ensure the truck fleet and cost is appropriately reflected;
- review and update the design and costing of infrastructure, especially for bulk water and electricity supply;
- fix a base date for costing and economic parameters and ensure they remain consistent for all areas of study;
- the Valencia Environmental Impact Assessment (EIA) that was completed in 2008, remains valid for the Valencia project alone. This EIA report will be updated to include the Namibplaas project into a single Norasa EIA/EMP. It is proposed that any design changes on Valencia that impact the current EIA should be included into the updated assessments. Although Valencia can commence construction and production with the current EIA, the assessment should be updated to cater for certain medium and long-term changes in strategy, particularly the addition of the satellite pit and expansion and change in configuration of waste dumps;
- the current rocktype allocation uses different modelling techniques and is derived from conditional simulation at Valencia, complex wireframing at Valencia’s satellite orebody and by indicator kriging at Namibplaas. In order to align all three models into a future planning resource, an alignment of allocation method is recommended;
- ensure that there are no gaps or overlaps in the studies being conducted by various consultants.

The Resource models presented in this report are deemed fit for feasibility level work.
### REFERENCES

<table>
<thead>
<tr>
<th>Author</th>
<th>Title</th>
</tr>
</thead>
<tbody>
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</tr>
<tr>
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</tr>
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</tr>
</tbody>
</table>
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28 **DATE AND SIGNATURE**

**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 23 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 overall report compilation and supervised the writing of the following sections: 1 to 6, 13, and 15 to 27.
- I have visited the site on a regular basis, totalling more than one hundred occasions. My last visit to Norasa was on Friday, 21 March 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.

Signed and dated this 25th day of March 2014 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 27 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 section of this report (sections: 7 to 12) and the Mineral Resource section 14.
- 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 27 February 2014 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.

Signed and dated this 25th day of March 2014 as Swakopmund, Namibia.

[Signature]

Martin Hirsch, MIMMM