Section 2
Executive Summary

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2. EXECUTIVE SUMMARY

2.1 INTRODUCTION

In late 2008 Greenland Minerals and Energy Limited (GMEL) commenced its Prefeasibility Study programme for the development of the Kvanefjeld Multi-Element Project**.** The Kvanefjeld deposit is a large, bulk-tonnage resource of rare earth elements, uranium, zinc and sodium fluoride, located near the southwest tip of Greenland. The deposit is hosted in the Ilimaussaq intrusive complex; the global type-example of an extremely unusual group of alkaline rocks.

Uranium resources at Kvanefjeld were initially defined in work programs conducted by the Danish government through the 1960s and 1970s. However, it has only been recent exploration by GMEL that has redefined Kvanefjeld as a multi-commodity resource of global significance.

The focus of the Prefeasibility Study (the Study or PFS) has been to evaluate the potential for development of a mine and processing plant facilities to treat 10.8 Mt/a of ore to extract rare earth elements (REE) and uranium.

The Interim Report on the Study presents the outcomes achieved to date, which are preliminary only. They represent the current level of metallurgical test work, mine scheduling and process design that has been undertaken to date. Further test work, mine design and engineering is required to add confidence and optimise the project's potential.

The work commissioned by GMEL to date has been carried out by internationally recognised consulting firms covering a wide range of disciplines, and in particular:

- Resource definition and mine plans
  - Coffey Mining, Hellman and Schofield
- Metallurgy and process development
  - AMEC Minproc, ANSTO, SGS Lakefield Oretest, CSIRO, Battery Limits
- Environmental baseline and Environmental Impact Assessment
  - Coffey Environments, Orbicon (Denmark)
- Plant engineering design, infrastructure, capital development
  - AMEC Minproc, NIRAS (Denmark)
- Marketing
  - IMCOA, WNA, MGMT Group.

The Study also draws on extensive historical work conducted by Danish authorities and scientists in the seventies and early eighties, which culminate in the 1983 Pre-Feasibility Study published by Risø.

All costs presented in this report are in US dollars unless otherwise indicated.

2.2 KEY OUTCOMES

The Study has confirmed that the Project has the potential to become a highly profitable world class rare earth and uranium producer. The key outcomes to highlight are:

- A pre-tax ungeared internal rate of return of 24% and a cash payback period of 5 yr to 6 yr, based on long term prices of US$80/lb U_3O_8 and US$13.0/kg RE carbonates. The pre–tax NPV was US$2181 M.
- The Engineering Studies identified a processing route which was selective for uranium extraction and capable of producing a rare earth concentrate. A mine and processing plant capable of treating 10.8 Mtpa is expected to cost US$2.31 billion. Construction is scheduled to commence in 2013 and first production in 2015. The project has a mine life of over 23 yr.
- The recent exploration programs have resulted in a significant increase in resource inventory which now includes rare earths, uranium, zinc and sodium fluoride. The 457 Mt resource is located 7 km from tidewater, with deep fjords running directly to the North Atlantic Ocean. The resource is mostly outcropping and within 300 m of ground surface.
- Mining studies indicate a large open pit with a low waste strip ratio and the highest grades presenting near-surface. Total life of mine production is 239.3 Mt at an average mine grade of 314 ppm U_3O_8 and 1.01% TREO.
- Regional drill holes demonstrate significant potential for new multi-element deposits and further exploration is warranted.
- Rare earth metals and uranium are now widely recognised around the world as strategically important commodities for the future. The market analysis indicates that demand for rare earths and uranium is set to rise strongly over the next 20 yr. In the case of uranium supply there can be little doubt that advancing new production to meet this demand will be a very significant challenge for the industry. Furthermore, the reduction of REO supply from the Chinese market, together with demand growth returning to the historical levels of 8% to 11% pa, will cause REO prices to rise significantly in the longer term.
- Greenland is seen as an emerging mineral province, politically stable and becoming increasingly independent. GMEL is aware of and respects the Greenlandic government stance on uranium exploration and development in Greenland – which is currently a zero tolerance approach to the exploration and exploitation of uranium. Any potential change toward the current stance of zero tolerance is not expected until after a public consultation and review process.
- The future development and success of the Project will hinge on community support. Although Greenland is open to the introduction of new mining and exploration, GMEL is mindful of its need to respect the land, the environment and the wishes of its people. It is, therefore, undertaking all aspects of its work in consideration and consultation with local communities. The major focus will be on the development of an Environmental Impact Assessment (EIA).

2.3 GEOLOGY AND EXPLORATION

2.3.1 Geology of Ilimaussaq Complex

The Ilimaussaq Complex is one of the intrusive complexes of the Gardar igneous province in South Greenland. The layered nature of the complex is attributed to four successive pulses of magma. The first pulse produced an augite syenite, which now forms a marginal shell. This was followed by intrusion of a sheet of peralkaline granite that is mostly preserved in the roof of the complex. The third and fourth stages make up the bulk of the intrusion. Stage four produced the agpaitic lujavrites and kakortokites that formed from volatile-rich alkaline magmas that were extremely enriched in incompatible elements such as rare earth elements, lithium, beryllium, uranium, and high-field-strength elements such as niobium and tantalum.

Black lujavrite is the unit that hosts REEs, uranium, and zinc multi-element mineralisation. Kakortokite units host zirconium mineralisation that also contains REEs and tantalum and niobium. The lujavrite series within the Ilimaussaq Complex is at least 500 m thick and are generally fine-grained and laminated but there are locally some medium to coarse-grained pegmatoidal varieties.

2.3.2 Kvanefjeld Geology

The Kvanefjeld region is located inside the northwest margin of the Ilimaussaq Complex. The region represents a lujavrite-rich area that has been unroofed by erosion. Other rock types that outcrop include basalt, gabbro and sandstone of the Ericsfjord Formation, and augite syenite and naujaite.

Steenstrupine is the dominant host to rare earth elements and uranium in all mineralisation styles. It is a complex sodic phospho-silicate mineral and mineralogical studies suggest that it commonly contains between 0.2 and 1% U_3O_8, and likely hosts greater than 70% of the uranium at Kvanefjeld. Other minerals that are important hosts to REEs include the phosphate mineral vitusite, and to a lesser extent, cerite and monazite. Aside from steenstrupine, uranium is also hosted in unusual sodic silicate minerals that are rich in yttrium, heavy REEs, zirconium and tin. Minor uranium is also hosted in uranothorite and monazite. Zinc is exclusively hosted in sphalerite, which is the dominant sulphide throughout the deposit.

2.3.3 Historical Exploration

The Kvanefjeld deposit was discovered in 1956 during a systematic radiometric reconnaissance survey of the whole Ilimaussaq complex. From 1957 to 1983, exploration proceeded under the direction of Risø, with assistance from the Danish Geological Survey, and concentrated on the northern part of the complex. In 1958, thirty six diamond drill holes for a total of 3728 m were completed at the discovery site on Kvanefjeld, termed the “mine area”. Three further drilling campaigns followed from 1962 to 1977 with a total of 40 additional diamond drill holes for a total of 19 926 m.

In 1979/80 two adits totalling 1020 m in length were completed to provide bulk samples for metallurgical testing and for evaluation of the radiological hazards related to mining. This work was undertaken as part of a prefeasibility study for the Kvanefjeld project. After completion of that prefeasibility study, Risø later terminated the project due to the low uranium market prices and because the Home Rule Authority in Greenland voted against the exploitation of uranium.

Between 1980 and 2000 demand for uranium, tantalum and rare earth metals increased. Danish exploration activity also increased. Companies involved included Carl Nielson A/S, Highwood Resources, Mineral Developments International A/S, and First Development International A/S.

In 2001 Rimbal Pty Ltd took up the southern section of the Ilimaussaq intrusion and undertook a detailed channel chip sampling program aimed at exploitation of tantalum, niobium and zirconium within the kakortokite sequence. In 2005, Rimbal took out a second license area covering the northern section of the Ilímaussaq intrusion and sampled from a number of sites within the license.

2.3.3.1 GMEL Exploration

In 2007, the exploration license that covers the northern half of the Ilimaussaq complex was acquired by Greenland Minerals and Energy A/S, a Greenlandic company which GMEL manages, and in which it holds a 61% interest. During the northern summers of 2007 and 2008, GMEL undertook two large exploration campaigns with a focus on resource definition on Kvanefjeld plateau. A smaller campaign was undertaken in 2009 with a focus on drilling to generate geotechnical data and other information relevant to mining, metallurgical and environmental studies. All recent drilling has been undertaken with diamond core drill rigs that were moved between sites with helicopter.

Drilling commenced in early June 2007, with the aim of collecting sufficient data to generate a JORC-compliant multi-element resource estimate. A total of 43 holes were drilled in the 2007 season with BQ size core. The deepest hole reached 398 m and most holes were drilled to a depth of 300 m.

A further large program of drilling was undertaken in the 2008 field season with 76 holes and over 19 000 m of BQ core drilled within the license area. The aims of the 2008 drill program were to: Improve the resource quality by infill drilling; Extend drilling to depth in areas where mineralisation remained open; Conduct close-spaced drilling to show continuity of mineralisation over short intervals; and conduct regional drill holes within the northern Ilmaussaq Complex to improve geological framework and develop new targets.

Bulk Density determinations were undertaken on sections of core before logging using Wet and Dry methods. The lithological units generally used in logging are *anorthosite, basalt, dolerite, foyaite, gabbro, granite, lamprophyre, lujavrite, naujaite, pegmatite, sandstone, syenite, feldspar porphyry*. Many minerals are defined as being of interest and include *steenstrupine, eudialyte, cerite, vitusite, kapustinite, and naujakasite*. Routine colour photography was completed on all drill-core.

At the completion of geological logging, sections of core were then split longitudinally using rotary hand splitters. One half of the split core was packaged and shipped to Genalysis Laboratories, Perth, Western Australia. Each sample was analysed for a standard suite of approximately forty elements including uranium, REEs, yttrium, zirconium, fluorine and tin.

Valuable data was obtained from historical Risø holes by spectral radiometric logging down the open holes and from gaining access to the original core stored at Risø in Denmark. Downhole geophysical interpretation was undertaken using an Auslog gamma spectral tool and logging system. Airborne geophysics was also undertaken with low level helicopter borne spectral radiometric and magnetic surveys.

Ground-based GPS controlled radiometric traversing was completed in 2008 on 6 prospective areas and significant radiometric anomalies were found at three of the prospects, and in-filled with closer spaced lines. Four holes were drilled at the K2 prospect, with initial evaluations indicating the cores featured multi-element mineralisation consistent with that at Kvanefjeld, and five holes were drilled into the K3 target and all holes intersected mineralised lujavrite. Collectively the regional drill holes provided a clearer indication of the enormous resource potential of the overall license area.

A new resource estimate was completed for the Kvanefjeld multi-element deposit when all assays were received from the 2008 drill program.

In the 2009 field season further diamond core drilling was undertaken to obtain geotechnical data relevant to mining studies, sterilise areas of potential infrastructure, and to generate core for ongoing metallurgical test work. A total of 5313 m of core were drilled.

2.4 RESOURCE ESTIMATION

Hellman & Schofield Pty Ltd (H&S) was retained by GMEL to estimate resources for the Kvanefjeld multi-element deposit. Resource estimates were required to be in accordance with the Australasian Code for Reporting of Identified Mineral Resources and Ore Reserves prepared by the Joint Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia, December 2004 Edition (the JORC Code). The elements of primary interest within the deposit include rare earth elements (REEs), uranium, zinc and fluorine.

H&S was provided with the latest drilling data and sectional interpretations of the layered intrusion which had been interpreted by GMEL geologists as being representative of the mineralisation. This data was used by H&S to construct a resource model of the rare earth element - uranium mineralisation at Kvanefjeld.

The resource model has been estimated by Ordinary Kriging (OK) using H&S's proprietary MP3 software, with the search directions consistent with the strike and dip of the mineralisation. The strike is N 36°E with an undulating dip varying from approximately zero to near vertical on the periphery and adjacent to mafic pendants. The aforementioned geometry of the host layered intrusive is consistent with results of spatial analysis of grades in the deposit.

Variables modelled included radiometric and chemical U_3O_8 ppm, REO ppm, TREO ppm (total REO, inclusive of Lutetium and Yttrium), NaF ppm, ThO_2 ppm and Zn ppm. The block model was constructed with block dimensions of 35 mE by 35 mN by 5 mRL. A composite thickness of 1 m down hole was used for input data to the model. Several iterations of the modelling process were undertaken to assess the sensitivity of estimates to variations in estimation parameters. The block grades subsequently were imported into a Micromine three dimensional (3D) model and regularised for use in mine planning software.

The location, quantity and distribution of the data are considered by H&S to be sufficient to allow the classification of the resources as Indicated and Inferred. Estimates of the Kvanefjeld resources for U_3O_8 ppm, REO ppm, TREO ppm, ThO_2 ppm and Zn ppm at 50 ppm U_3O_8 cut-off grades increments, as of May 2009, are presented in Table 2.4.1.

Table 2.4.1
Summary of the Kvanefjeld Resource Estimates – May 2009
(At a Range of Cut-off Grades by 50 ppm Increments)

Cut-off Grade	Indicated Resource Category					Inferred Resource Category				
	Mt	U_3O_8 (ppm)	TREO (ppm)	ThO_2 (ppm)	Zn (ppm)	Mt	U_3O_8 (ppm)	TREO (ppm)	ThO_2 (ppm)	Zn (ppm)
50	458	249	9 895	770	2 021	122	232	10 288	728	2 067
100	435	257	10 138	791	2 068	114	242	10 620	754	2 128
150	365	282	10 629	853	2 166	92	269	11 219	821	2 243
200	276	317	11 253	938	2 264	63	312	12 070	930	2 355
250	207	348	11 955	1 014	2 346	43	355	13 138	1 036	2 455
300	145	379	12 703	1 090	2 433	29	394	14 128	1 132	2 554
350	89	414	13 543	1 174	2 525	20	424	15 006	1 208	2 642
400	46	452	14 495	1 266	2 617	12	461	16 400	1 299	2 754
450	19	494	15 408	1 368	2 704	6	498	17 520	1 389	2 784
500	6	541	16 293	1 477	2 808	2	546	18 488	1 506	2 733

Note: Table 2.4.1 displays the resource estimates in an un-rounded form and does not in any way imply an added level of precision.

The resource estimates were considered Indicated and Inferred due largely to the broad and somewhat irregular drill spacing, and the understanding of the mineralisation controls which is still evolving as GMEL continues evaluation of the project. It is envisaged that with further diamond drilling and detailed core investigation, additional mineralogical analysis, and improved understanding of the deposit's geological setting, it will be possible to apply higher confidence categories to future resource estimates.

H&S is responsible for the resource estimates in conjunction with a Competent Person nominated by GMEL taking responsibility for data quality, geological interpretation, geophysical data manipulation and correction and calibration of probe data using chemical data analysis.

2.5 MINING

Coffey Mining Pty Ltd (Coffey) was retained by GMEL to carry out a Mining Study for inclusion in the Kvanefjeld Prefeasibility Study.

The scope of work that formed the basis of the Mining Study comprised the following tasks:

- Geotechnical assessment
- Hydrological assessment
- Pit optimisation
- Mine design
- Mine production scheduling
- Mine costing.

A crusher feed target of 10.8 Mt/a formed the basis of the study and based on an average waste to ore strip ratio of 0.8:1, this throughput equates to an average annual total material movement of 22 Mt. The study is based on owner mining, with the mining fleet leased. It was assumed that maintenance of all mobile equipment will be carried out by the original equipment manufacturer (OEM) as part of its supply and maintain contract.

Mining capital and operating cost estimates are presented in Sections 2.11 and 2.12.

2.5.1 Mining Fleet

At this stage of the Project, a standard drill/blast/truck/shovel operation would be considered the lowest operating risk mining method, both in terms of cost and productivity, and thus has been selected as the base case for the Mining Study.

The equipment selection was based on an earlier mining study which considered a 4.2 Mt/a process plant throughput rate. The 4.2 Mt/a study indicated that the mining fleet would most likely consist of 180 t sized hydraulic excavators and 100t capacity off highway dump trucks and standard open-cut drilling and auxiliary equipment. Due to time constraints larger equipment sizing was not considered for the larger scale project of 10.8 Mt/a. However, it is recommended that a larger loading unit (200 t to 250 t) and particularly larger dump trucks (150t) are further investigated in the next phase of the Project.

2.5.2 Manning Levels

Based on mining equipment proposed and the nature, complexity and location of the project, it is estimated that the mining workforce directly involved with the earthmoving component would consist of 48 management and supervision personnel, 146 operators, 32 maintenance and servicing personnel with blast and mine service crew estimated at 10, for a total of 236 employees.

2.5.3 Pit Optimisation

The Whittle Four-X optimisation software was used for the pit optimisation work, utilising the May 2009 Hellman and Schofield resource model.

Based on the Indicated Resources only, and using the shell selection criteria of the maximum undiscounted cash flow, the optimum pit shell contained some 219 Mt of ore at 300 ppm U_3O_8. Some 183 Mt of waste are contained within the pit shell, giving a strip ratio of 0.8:1

2.5.4 Mine Design

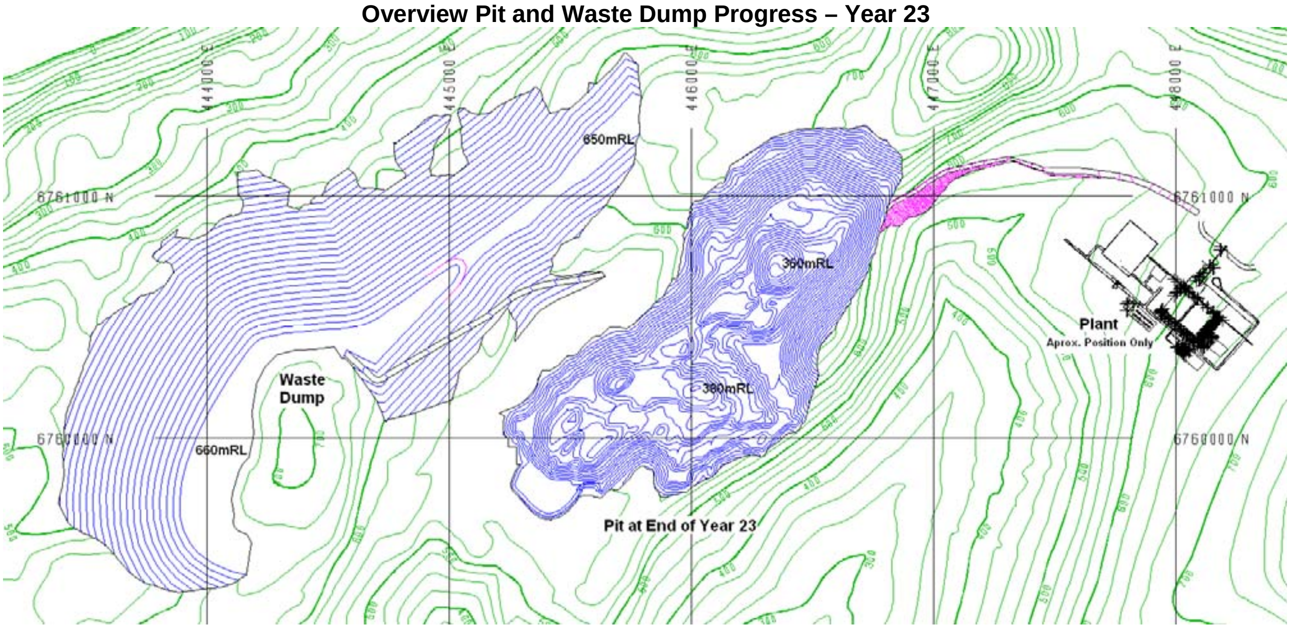
The final pit design was based on the shell as described above. Table 2.5.1 provides a summary of the material breakdown as contained within the pit.

Table 2.5.1
Summary Materials Breakdown by Pit Stage

Pit Stage	Total Material	Waste	Strip Ratio	Mill Feed				
				Tonnes	Grade			
					U_3O_8	REE	Zn	ThO_2
	(Mt)	(Mt)	(w:o)	(Mt)	(ppm)			
Stage 1	9.6	2.9	0.4	6.6	363	11 678	2 607	1 054
Stage 2	39.5	16.2	0.7	23.3	377	11 111	2 208	1 088
Stage 3	18.5	8.3	0.8	10.2	374	12 105	2 282	1 082
Stage 4	116.2	51.8	0.8	64.4	336	10 263	2 215	988
Stage 5	140.8	55.7	0.7	85.1	284	9 307	2 131	860
Stage 6	98.5	48.8	1.0	49.7	267	9 254	2 097	815
Total	**423.0**	**183.7**	**0.8**	**239.3**	**314**	**10 085**	**2 185**	**933**

An overview of the final pit and waste dump at the end of year 23 is presented in Figure 2.5.1.

Figure 2.5.1
Overview Pit and Waste Dump Progress – Year 23



2.6 METALLURGY

2.6.1 Introduction

Metallurgical testwork and flow sheet development for the Kvanefjeld resource has been undertaken in two distinct stages. A staged program of metallurgical development for the project was undertaken by Risø laboratory in Denmark from the mid 1960s through to the mid 1980s and focussed on recovery of uranium only. A second phase of testwork in Australia was managed by GMEL in 2008 and 2009 and evaluated recovery of REE and uranium.

The mineralogy of the Kvanefjeld ore types is complex and unique. Mineralisation is primarily associated with lujavrite, an unusual igneous rock type. Steenstrupine, a complex sodic phospho-silicate mineral, is the dominant host to REE and uranium. Aside from steenstrupine, uranium is also hosted in unusual sodic silicate minerals that are rich in yttrium, heavy REEs, zirconium and tin. Minor uranium is also hosted in uranothorite and monazite. Zinc is exclusively hosted in sphalerite, which is the dominant sulphide throughout the deposit. Studies by Risø and GMEL have shown that mineralogical variations significantly affect metallurgical processing characteristics. Carbonate pressure leaching (CPL) tests have shown favourable results from the steenstrupine-rich black lujavrite, and less favourable results from samples characterised by the eudialyte-monazite association. Work is ongoing to increase the geological and mineralogical understanding of lujavrites at Kvanefjeld, as well as understanding how the different minerals behave metallurgically.

Conventional atmospheric sulphuric acid leaching is not a feasible treatment method for Kvanefjeld ore due to the very high acid consumption as a result of the alkaline nature of the deposit. Sulphation roasting of ore at a temperature of 700^{o}C in the presence of sulphur dioxide and air was extensively tested and piloted by Risø in the 1970’s. This process option was rejected due to low and variable uranium recoveries and high reagent consumptions and operating costs. More recently GMEL has undertaken a program of sulphation digest testwork on flotation concentrate. This procedure has involved mixing flotation concentrate with additions of concentrate sulphuric acid, and then baking the samples at temperatures of 180^{o}C to 240^{o}C followed water leaching. Tests resulted in high reagent consumptions, practical difficulties in leaching, requiring very low density slurry, and low overall uranium recovery.

2.6.2 Carbonate Pressure Leaching and Uranium Recovery

The front end process option selected for this Study is carbonate pressure leaching of whole ore feed. The CPL process was extensively tested at Risø including programs with a continuous 2 t/h pilot pipe reactor. Although uranium extraction was shown to be variable with rock type and alteration, Risø concluded that uranium extractions of around 85% would be feasible on the dominant rock types using the CPL process. More recently batch testing has been undertaken in Australia to confirm the earlier findings. Some key conclusions from CPL programs undertaken include:

- CPL processing of whole ore at a temperature of 260^{o}C, and using a reagent scheme of 120 g/L $NaHCO_3$ and 20 g/L Na_2CO_3, is a highly selective leaching technique for extraction of uranium. Gangue dissolution is very low.
- The CPL process results in low reagent consumption and a relatively clean leach solution suitable for downstream uranium recovery. The CPL technique provides a good initial separation of uranium from the REE and thorium. REE and thorium nearly all report to CPL residue.
- Although the chemistry during leaching is not well understood, Risø pilot testing suggests that soluble fluorides, silicates and sulphates are converted to stable precipitates during leaching. The precipitation chemistry will need to be confirmed in later studies.

CPL processing for extraction of uranium has been selected as the front end circuit for this study on the basis of selectivity, uranium recovery and potential economics.

2.6.3 CPL Residue Leaching and REE Recovery

Testwork has recently been undertaken to evaluate extraction of rare earths from the CPL residue. The aim of the recent work has been to produce a leach solution suitable for downstream RE recovery by precipitation of a bulk rare earth carbonate product. The testwork findings are based on a preliminary program of work and performance should improve with further optimisation

Beneficiation of CPL residue by flotation has been successful using the same alkyl hydroxamate collector as used in previous ore flotation testing. The REE flotation response in preliminary testing has been shown to be similar, albeit slightly inferior, to that of whole ore flotation.

Hydrochloric acid leaching tests have been undertaken on CPL residue flotation concentrate at low but controlled acid addition. The tests under atmospheric conditions have shown that leaching at pH 2.5 rapidly solubilises approximately 15% to 30% of the REE at a potentially economic acid consumption. The leach conditions have proven to be selective against uranium and thorium, and other non REE minerals as well as gangue minerals. The selective leaching results in a leach solution that should be suitable for downstream purification and RE recovery.

The results also showed that the extraction of REEs with more concentrated levels of hydrochloric acid were reasonable and mostly in the range of 50% to 70%. However, acid consumption was high, in the range 250 kg/t to 300 kg/t of residue.

ANSTO has reviewed the preliminary testwork data and based on its experience has proposed a flow sheet for production of bulk rare earth carbonates from CPL leach solution. The key steps in the REE recovery are:

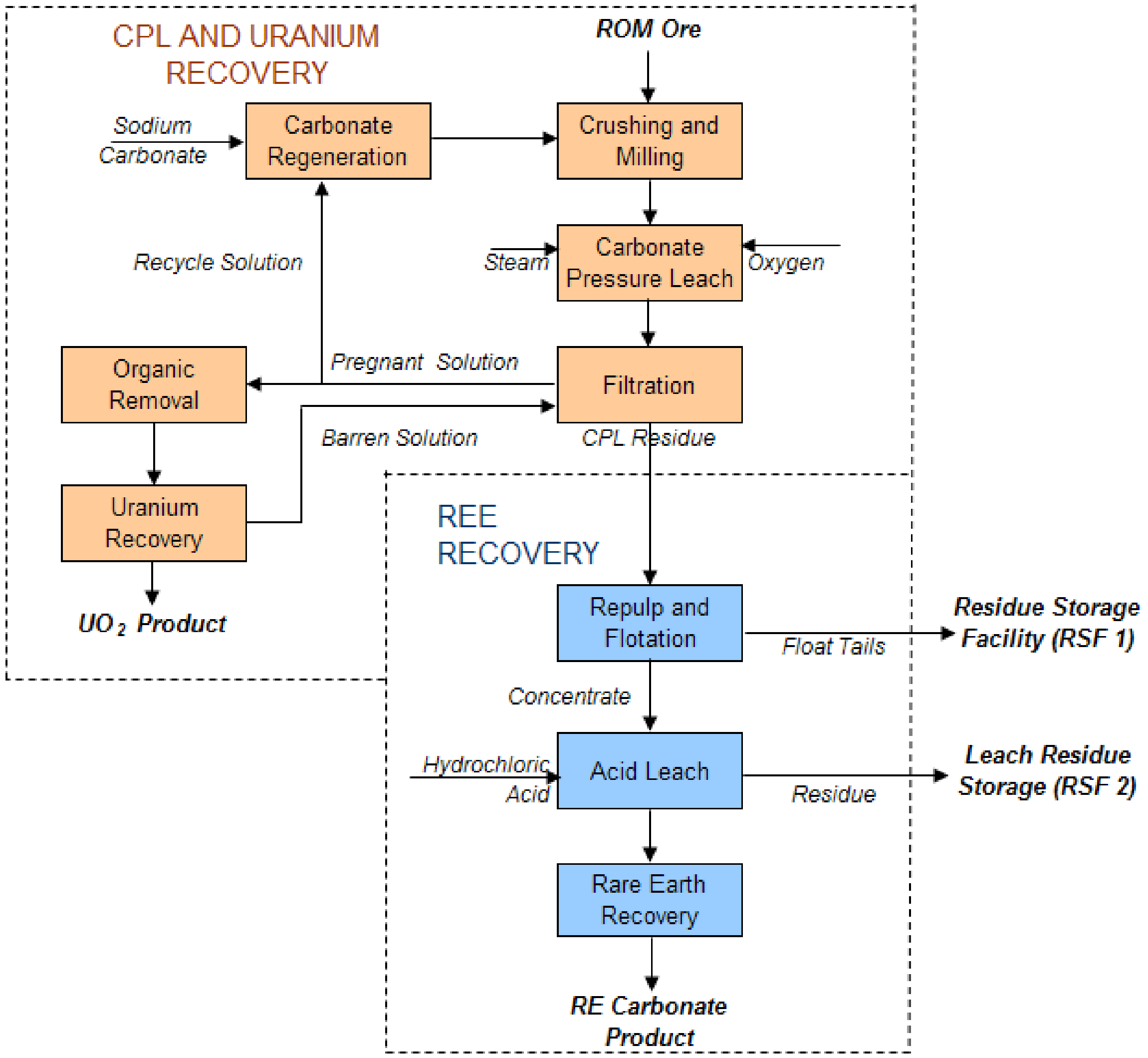
- Partial neutralisation of leach solution with caustic at pH 4.2, and subsequent filtration to precipitate and remove impurities Al, Fe, Si.
- Precipitation of zinc from leach solution using sodium hydrogen sulphide (NaHS) and subsequent filtration to produce a zinc sulphide by-product.
- Neutralisation of leach solution at pH 7.5 and subsequent filtration to recover REE hydroxides to an impure filter cake.
- Selective digestion of filter cake with dilute hydrochloric acid at pH 4.5 to solubilise REE, and subsequent filtration to remove remaining Al, Mn and Si in residue.
- Carbonate precipitation of REE in digest solution using sodium carbonate, and subsequent filtration drying and packaging of the RE carbonate product.

Hydroxide precipitate may be an alternative final product negating the need for the last two steps. Metallurgical testwork for recovery of REE from leach solutions has not been tested to date, and will be evaluated in later stages of the project.

2.7 PROCESS PLANT

AMEC Minproc has completed a preliminary design of the processing plant for recovery of both uranium and REEs from Kvanefjeld ore. The selected process is based on CPL technology for removal of uranium from feed ore, and dilute hydrochloric acid leaching for extraction of REE from CPL residue. A simplified block flow diagram of the proposed processing facility is presented in Figure 2.7.1.

Figure 2.7.1
Simplified Block Flow Diagram



Key plant design production parameters are presented in Table 2.7.1.

Table 2.7.1
Plant Design Parameters

Plant Throughput	t/a	10 800 000
Overall Uranium Recovery	% U	83.8
Uranium Production	t U_3O_8/a	3 895
Overall REE Plant Recovery	% REO	34
REE Plant Production, Nominal	t REO/a	43 729

2.7.1 CPL Processing and Uranium Extraction

The ore is considered to be of moderate hardness. The comminution circuit selected for the process is a conventional Semi-Autogenous primary mill, ball mill and pebble crushing (SABC) circuit, as used widely throughout the mineral processing industry. The target product grind size selected is 80% passing 75 microns based on recent testwork and previous Risø experience. A thickener has been included in the mill circuit to achieve a CPL feed density of 60% solids.

The CPL process has been selected for extracting uranium due to its selectivity for uranium over gangue minerals. The high selectivity results in relatively low reagent consumption, and also provides a good separation of uranium from REE and thorium. A wide body of data is available for the CPL process from both early Risø work, and more recent testwork initiated by GMEL. The process entails leaching at 260^{o}C in an autoclave employing a mixture of 120 g/L sodium bicarbonate and 20 g/L sodium carbonate as lixiviant. Oxygen is added during leaching to assist in solubilising uranium.

Three identical CPL autoclave trains are included in the preliminary design and serviced by a common steam boiler and oxygen plant. Horizontal multi-compartment agitated autoclaves have been selected similar in design to those used in pressurised leaching applications in the nickel and gold industry.

Belt filters have been chosen for residue filtration based on recent preliminary testwork and requirements of the water balance, so as to maximise uranium recovery and minimise reagent losses. The filters employ counter-current washing of the filter cake with recycled liquor in the first two wash stages, and fresh water in the third wash stage.

A portion of the leach solution is pumped to the uranium recovery circuit while the remainder is recycled to the milling circuit. The circuit has been designed such that carbonate is the sole reagent that is needed to make-up for the carbonate and bicarbonate losses. The majority of the make-up carbonate is added as a dry powder onto the SAG mill feed conveyor. Bicarbonate is regenerated by reacting carbon dioxide sourced from the power station off-gas with the carbonate in mill thickener underflow and overflow streams.

The selected process for recovery of uranium from leach solution is direct precipitation of uranium dioxide (UO_2) from leach solution by hydrogen reduction. The circuit comprises of a single six compartment horizontal hydrogen reduction autoclave, thickener, filter, dryer and drumming plant. The autoclave operates at 150^{o}C. Prior to the reduction process leach solution is passed through an organic removal circuit for removal of traces of solubilised humus present in the ore.

The overall uranium recovery in the CPL and uranium extraction process is calculated to be 83.8% based on an extraction of 85% in CPL as determined in Risø studies.

2.7.2 CPL Residue Leaching and REE Recovery

The REE recovery flow sheet has been developed based on limited REE extraction testwork undertaken by GMEL, as well as a number of reasonable assumptions for future metallurgical improvements, and flow sheet recommendations by Australian Nuclear Science and Technology Organisation (ANSTO).

Testwork has shown that beneficiation using a conventional phosphate flotation collector can be used to upgrade the REE minerals in the CPL residue. Rougher flotation testwork has shown that the best separation of REE over gangue was achieved at a mass pull of 25%, and this mass pull was therefore selected for the design of the rougher circuit. Assumptions have been made for cleaning of rougher concentrate since no cleaning testwork has yet been undertaken. Rougher flotation is conducted in a series of five rougher flotation cells. Following a polishing grind in a ball mill the rougher concentrate is cleaned with a target mass pull of 40% of rougher feed giving an overall mass pull of 10%. Concentrate is then thickened prior to leaching.

Recent hydrochloric leaching test work has shown that mild acid leaching of the flotation concentrate solubilises a portion of the contained REE. Although REE extractions of over 70% can be achieved with high acid additions, mild leach conditions are necessary to limit the extent of gangue reactivity in order to achieve economic acid consumption. It has been assumed in the plant design that extractions will be significantly improved over those obtained in preliminary test results. The plant design involves leaching of repulped concentrate at pH 2.5 using hydrochloric acid followed by filtration and washing of the leach residue.

Standard unit operations are then used to progressively purify the rare earth leach solution and ultimately produce a bulk rare earth carbonate product. The flow sheet selected for this engineering study was suggested by ANSTO, and reviewed by AMEC Minproc and agreed for use in the study without further development. The key processing steps are:

- Partial neutralisation of leach solution with caustic soda to precipitate Al and Fe
- Removal of Zn by precipitation with sodium hydrosulphide
- Precipitation of rare earth hydroxide (REH) using caustic soda to reject Mn
- Releaching of REH in weak hydrochloric acid solution to selectively dissolve REE
- Precipitation of REC using soda ash.

Further testwork will be required to confirm the suitability of this flow sheet for REE recovery.

2.7.3 Plant Services and Water Balance

Waste streams from both the CPL and REE plants are sent to the effluent treatment area for neutralisation, heavy metal precipitation and solids containment within a dedicated residue storage facility. Surplus treated liquor is discharged to the ocean.

Raw water make up is minimised by use of recirculation and inclusion of a number of separate process water circuits. The three main types of process water are: organic-free process water, acidic process water, and neutral process water.

Due to the large quantity of hydrochloric acid consumed by the REE plant, a chlor-alkali plant has been incorporated into the process design for on-site acid production. This has the added benefit of producing a caustic soda by-product, another major REE plant reagent. On site hydrogen and oxygen generation plants are also included in the design.

Power generation is provisionally based on use of heavy fuel oil fired multiple reciprocating machines. The option of implementing hydro-power for Kvanefjeld is to be assessed in future studies for the project.

2.8 TAILINGS MANAGEMENT

The Tailings Management section discusses a potential process for managing the tailings, or residues, arising from the Kvanefjeld Multi-Element Project processing plant, including the design, operating and closure concepts. The Tailing Management process is still in the conceptual stage. The concepts discussed are based on earlier designs by Risø (1983), with additional options to be reviewed in the next phase of the Project's development.

The two sources of tailings that will require management are as follows:

- The major source is produced by the CPL plant for which a permanent storage solution (residue storage facility 1 or RSF 1) will be required
- The second source of tailings, from the REE plant, is potentially recoverable for further processing and is proposed that it will be placed initially in residue storage facility 2 (RSF 2).

For RSF 1, the tailings storage concept involves pumping tailings from the plant via a fully welded 300 mm pipeline and discharging below water into Lake Taseq. To store the estimated volume of tailings generated over the life of mine, an embankment will be constructed at the outlet using mine waste or rock quarried from adjacent slopes. The lake water displaced by tailings will be treated and released. The key advantages of subaqueous tailings storage include mitigation of radon gas release and mitigation of dust generation. Based on the information provided, there is sufficient storage in RSF 1 for the design life of mine production.

For RSF 2, the concept involves pumping slurry from the REE plant effluent treatment unit into RSF 2, with reclaim water returned to effluent treatment. The REE rich material in RSF 2 would be stockpiled for potential future reprocessing and recovery of additional REE product. Once reprocessed RSF 2 material would be trucked to the mine and backfilled into the mined out pits as part of the mine closure plan.

Several alternatives for these tailing management concepts have been identified and these will be reviewed as the project develops.

2.9 ENVIRONMENTAL

A comprehensive environmental study and report has been prepared by Coffey Natural Systems Pty Ltd on behalf of GMEL entitled "Preliminary Project Strategy – Kvanefjeld Multi-Element Project", September 2009.

In discussing environmental issues, the Coffey Natural Systems Report evaluates existing information and regulatory requirements, identifies relevant environmental, social and permitting issues and outlines a process that will lead to the development of a strategic environmental and social framework for the Kvanefjeld Multi-Element Project.

The main environmental issues that impact on the Project include:

- Water contamination (including surface waters, fjords and groundwater) due to discharges from open pit, waste rock dumps, process plant and the RSF (eg fluorine, uranium and heavy metals).

- Effects of atmospheric emissions such as radon gas, dust, combustion products and other gaseous emissions. Emissions can be carried over a long distance by the wind, with implications for the environment and for human health.
- Accumulation of contaminants in the food chain, with the sensitive receptors identified as sheep, Arctic char (Salvelinus alpines) and commercial fish species.
- Management of biodiversity issues (terrestrial and aquatic), including the presence/absence of rare and/or threatened species.
- Effects of radiation from radioactive sources within the project area.
- General waste management.
- Rehabilitation of areas disturbed by the project and decommissioning of the tailings management system.

The main social issues (both positive and negative) identified for the Project were summarised as follows:

- Effects of land alienation from existing uses that will be required by the project components and ancillary infrastructure
- Effects on the amenity of Narsaq and surrounding settlements resulting from dust, noise and light emissions from the project area
- Effects of the project on the water supply for Narsaq township and surrounding settlements
- Effects on subsistence, artisanal and commercial fishing and hunting (including fish spawning and nursery areas and seal pupping areas)
- Effects on cultural heritage and archaeological sites (including sacred and spiritual places, traditional fishing or hunting campsites, traditional trails and burial grounds)
- Effects on transportation infrastructure and incremental traffic flows (air, land and sea) and transportation risks
- Effects of the project on local infrastructure, health, education and other government services
- Opportunities for training, employment and business development during construction and operations
- Monetary (such as taxes and royalties) and other benefits (such as improved sanitation and health services) associated with the project
- Economic multipliers associated with the project, as well as backward and forward economic linkages within the Greenland economic sectors that drive economic growth
- Improvements in the nation's balance of trade, infrastructure development, and commercial, employment and educational opportunities.

A range of base line studies covering the impact and potential management of these environmental and social issues has already been commissioned by GMEL, with further studies to be carried out in the future.

Stakeholder consultation will be essential to the success of the Project and GMEL has already initiated some stakeholder consultation, including discussions with the Greenlandic Bureau of Mines and

Petroleum (the BMP), Danish National Environmental Research Institute (NERI) and the community, and will continue to do so as the Project progresses.

2.10 INFRASTRUCTURE

The infrastructure requirements for the Kvanefjeld Project are primarily based on a recent infrastructure report by NIRAS, and historical reports by Risø. AMEC Minproc has supplemented the information from these sources with in-house information, and information from GMEL.

The project location is relatively remote approximately 10 km north-northeast of the town of Narsaq in south-east Greenland. The infrastructure to be installed as part of the project comprises of staff accommodation, dedicated new harbour facilities with off-plant storage requirements, new roads, power transmission, water supply, and recycle and treatment systems.

2.10.1 Regional Communities and Current Infrastructure

The district and municipality of Narsaq has a total population of more than 2000, of whom between 1600 and 1700 live in the town itself. The majority of inhabitants are employed in fishing or sheep-farming. Narsaq is a port of call for the Arctic Umiaq Line, a passenger and freight shipping line. The sea connection provided by Arctic Umiaq is a lifeline for the entire western Greenland.

Narsaq Heliport operates year-round, linking Narsaq with Qaqortoq on the shores of Labrador Sea and also linking Narsaq with the nearby Narsarsuaq Airport. Narsarsuaq is a small community with approximately 200 people mainly employed at the airport or the associated hotel. Narsarsuaq has air connections to Nuuk (capital of Greenland) and to Copenhagen with Boeing 757s. Nuuk has a population of over 17 000.

2.10.2 Accommodation

It is predicted that a total 709 personnel will be required for the project operation and approximately 222 of these personnel will be recruited locally from within the Narsaq municipality. The remaining project personnel will be accommodated on a temporary fly-in/fly-out (FIFO) basis in a custom built village to be located between the existing town of Narsaq and the process plant.

An accommodation village will be provided with an access road off a new road connecting the mine and plant to the harbour. The village will be supplied with power (from the process plant power station), water and sewage treatment. A 2000 m^2 activity centre is envisaged, with an indoor swimming pool, sauna, gymnasium, meeting rooms, canteen and internet connections.

2.10.3 Harbour Facilities

Dedicated new port facilities will be installed at Ilua Bay for the Kvanefjeld Project. The new port will handle materials and equipment for the construction of the mine and plant, and for the operational phase the ongoing import of fuel, reagents and consumables, and export of products. The new facilities will be designed to handle vessels similar to those currently serving Narsaq port.

The port is designed with a 130 m quay front with conveyors for bulk cargo, and a mobile crane for containers. Adjacent to the quay, an area will be prepared for container stacking and covered bulk storage for both imports and exports.

2.10.4 Other Transport Facilities

A new 7 m wide road, approximately 13 km long, will be built to connect the harbour at Ilua Bay, the process plant, the mine and accommodation village. The new road will follow an existing gravel road along the Narsaq River. The new road will be for all imports and exports transport between port, plant and mine, as well as ore transportation from the mine to the plant. Heavy fuel oil (HFO) supplied from the port to the power plant sited at the process plant will be pumped through a buried fuel line following the new road route.

Personnel will generally commute by bus between the accommodation village and the work sites at the mine and plant. The existing heliport at Narsaq is considered to require an extension to passenger facilities, but the airport at Narsarsuaq is considered adequate to handle additional passenger loads resulting from the Kvanefjeld construction and operation. Additional flights between Narsarsuaq and Nuuk, Reykjavik and Copenhagen may be necessary for the increased volume of passengers.

2.10.5 Water Supply

The process plant can access raw water from the mine area, Lake Taseq catchment area, and water resulting from tailings displacement in Lake Taseq. This engineering study assumes that mine water is not required for processing and that excess water is discharged to the environment without any contact with the process plant.

Water will be recovered from Lake Taseq, which will perform the dual function of tailings and water reservoir. It is estimated that a proportion of the recycle decant water will require basic water treatment and filtration for use in the process plant. A diversion and segregation of a portion of the precipitation run-off into the lake is required to satisfy the demand for uncontaminated water.

A preliminary water balance indicates that there will be a net discharge of treated water to the Taseq and Narssaq River systems and/or the Bredefjord. This discharge to the rivers will be in addition to an environmental discharge of chloride containing treated process plant effluent to the ocean.

2.10.6 Hydropower

Although the engineering studies are based on importing and burning HFO to meet the total project energy requirements, Greenland is well suited for hydro electric power development from both a topographical and a hydrological point viewpoint. An existing hydropower facility is located in the area north of Narsarsuaq. Potential areas for future hydropower are considered in this study.

2.11 CAPITAL COSTS

The engineering study technical and capital cost estimating scope was split between contributing parties to the study as follows:

- Mine and tailings – Coffey Mining Pty Ltd (Coffey)
- Process plant, plant infrastructure and overall cost estimate – AMEC Minproc

- Area and regional infrastructure – NIRAS Greenland A/S

Costs are estimated in United State dollars as at second quarter 2009 and are judged to have an accuracy of not less than ±35%, ie in line with “order-of-magnitude” estimates. Capital costs relate to the entire project, excluding owner’s costs and contingencies for scope changes and project risks. Costs presented are for a 10.8 Mt/a throughput.

2.11.1 Mining

Mining set up capital costs have been included for clearing, mine infrastructure, explosives magazines and office setup, and total US$10.3 M. Mining fleet costs have been included in the operating cost model , based on the assumption that the mining fleet is to be leased for the purposes of the Mining Study.

2.11.2 Process Plant and Infrastructure

The project capital cost estimate for process plant and infrastructure as prepared by AMEC Minproc is summarised in Table 2.11.1

Table 2.11.1
Capital Cost Estimate: 10.8 Mt/a Process Plant

Area No	Area Title	US$’000
Direct Costs		
	CPL Process Plant	683 112
	REE Process Plant	321 520
	Subtotal Process Plant	**1 004 632**
	Infrastructure – Plant	246 333
	Infrastructure – Area	91 730
	Infrastructure – Regional	94 063
	Subtotal – Infrastructure	**432 126**
	Process Plant – Miscellaneous	**107 387**
	Total Direct Cost	**1 544 145**
Indirect Costs		
	Temporary Facilities	123 532
	Engineering, Procurement and Construction Management	245 162
	Total Indirect Costs	**368 694**
	Total Project Costs (Net)	**1 912 839**
	Contingency (Growth Allowance)	382 568
	Total Project Costs (Overall)	**2 295 407**

2.12 OPERATING COSTS

Mine operating costs have been estimated by Coffey Mining, and operating costs relating to the process plant and infrastructure have been estimated by AMEC Minproc.

2.12.1 Mining Costs

A breakdown of the life of mine operating costs for the 23 yr mine life as developed by Coffey is presented in Table 2.12.1.

Table 2.12.1
Cost Summary by Activity

Cost Item	Total ($)	$/tonne moved	% Split
Drilling	168 314 165	0.40	10%
Blasting	111 207 714	0.26	7%
Load	94 872 598	0.22	6%
Haul	548 768 646	1.30	34%
Monthly Maintenance Fee – Mob, Misc Capital Cost ,etc	124 295 985	0.29	8%
Major Ancillary	324 982 862	0.77	20%
Minor Ancillary	49 045 108	0.12	3%
Indirect Costs	199 120 353	0.47	12%
TOTAL	**1 620 607 433**	**3.83**	**100%**

The mean operating cost over the 23 yr mine life is calculated to be $81.03 M/a or $7.50/t of ore treated.

2.12.2 Process Plant and Infrastructure Costs

The following summarises the methods used by AMEC Minproc for the processing plant and infrastructure operating cost estimate:

- Labour : Labour rates (US$/man/year) have been based on AMEC Minproc's Australian database for management, professional and supervisory staff, supplemented with base salary estimates for skilled, unskilled and shift workers extracted from information supplied by Niras for Greenland conditions.
- Energy: Electrical power and thermal energy (steam) consumptions have been calculated across the process plant and estimated for the infrastructure. Energy costs are based on the use of owner-operated HFO fired equipment, related to a benchmark oil price of US$80/barrel.
- Reagents: Consumptions of reagents are based on the mass balances undertaken by AMEC Minproc . Unit rates for key reagents such as sodium carbonate, sodium bicarbonate, sodium chloride and flotation reagents have been based on budget quotations from reliable suppliers, with in-house information for other reagents. All reagents and consumables include an estimate of freight cost from the source of supply to site.
- Maintenance: Maintenance costs are factored from direct capital costs based on benchmarks derived from other similar projects. Maintenance includes for limited expenditure on sustaining capital, maintenance spares and any specialised contract labour.
- Consumables: Consumables, other than reagents, include steel balls for milling, mill liner steel, filter cloths and lab samples. Grinding media and liner consumptions are based on wear rates factored from the Bond abrasion index using AMEC Minproc's in-house calculations. Relevant operating experience involving similar size plants and ore types was also incorporated.

- General and administration: Allowances based on AMEC Minproc experience, for general freight costs (excluding reagents), transport (FIFO) costs for personnel not recruited locally in Narsaq, recruitment, training, insurance and administration costs.

Operating costs are presented separately for the CPL and REE circuits. All infrastructure costs are allocated to CPL processing and uranium production as presented in Table 2.12.2, while REE costs shown in Table 2.12.3 are incremental production costs for recovery of REEs.

Table 2.12.2
10.8 Mt/a Ore Throughput Operating Cost Summary – CPL Circuit

	Proportion of Cost (%)	Annual Cost ($'000/a)	Unit Cost	
			$/t Ore	$/lb U_3O_8
Labour	8.6	21 838	2.02	2.54
Energy – Power and Thermal	37.6	95 417	8.83	11.11
Reagents	24.2	61 536	5.70	7.16
Consumables	9.8	24 953	2.31	2.91
Maintenance Materials	14.7	37 318	3.46	4.34
General and Administration	5.1	13 016	1.21	1.52
Cost of Sales (Shipping)	0.1	264	0.02	0.03
Total	**100**	**254 343**	**23.55**	**29.61**

Note: The unit cost in $/lb U_3O_8 will vary with head grade, and that presented in Table 12.3.1 is based on process plant design feed grade of 365 ppm uranium.

Table 2.12.3
10.8 Mt/a Ore Throughput Operating Cost Summary – REE Circuit

	Proportion of Cost (%)	Annual Cost ($'000/a)	Unit Cost	
			$/t Ore	$/kg REO
Labour	6.0	8 856	0.82	0.20
Energy – Power and Thermal	19.4	28 577	2.64	0.65
Reagents	51.1	75 154	6.96	1.72
Consumables	2.0	2 971	0.28	0.07
Maintenance Materials	12.1	17 781	1.65	0.41
General and Administration	1.6	2 395	0.22	0.05
Cost of Sales (Shipping)	7.7	11 353	1.05	0.26
Total	**100**	**147 087**	**13.62**	**3.36**

Note: The unit cost in $/kg REO will vary with head grade, and that presented in Table 12.3.2 is based on process design feed grade of 1.19% REO equivalent.

2.13 MARKETS

2.13.1 Uranium Oxide

The majority of U_3O_8 is used in the generation of nuclear power. The outlook for nuclear power continues to improve as a result of: the need for increased sovereign security over energy supplies; environmental concerns; and continued improvement in the performance of nuclear reactors. Development of adequate new uranium production to produce the nuclear fuels required to meet increasing demand for nuclear power is likely to be a significant challenge for the industry.

In a recently released report the World Nuclear Association (WNA) indicates, in its Reference scenario, that if 80% of the mine capacity that is currently under development comes into production on schedule, the uranium market should be largely in balance until 2015/6. Beyond 2016 the market will require production from mines that have not yet received development approval in order to meet the level of demand forecast in this scenario. By 2020, mine production from mines that have been publicly announced but not proved to be commercially viable will be needed to meet forecast demand. Requirements for uranium are forecast to increase from approximately 65 000 t U/a in 2009 to over 100 000 t U/a in 2030, a 55% increase. A structural gap is looming between supply and demand which is expected to remain significant for the medium to long term. The long term fundamentals appear extremely positive for uranium producers.

The increase in the price for uranium since 2003 has encouraged some increase in the level of exploration and planning for future mine development. It is noted however that bringing new production on stream takes a considerable time, and is typically characterised by significant delays. Most of the mines currently under development were discovered in the 1970s and 1980s.

Uranium prices, both spot and long term, have experienced an extraordinary boom/bust cycle since the beginning of 2005. However, consistent with a market largely in balance, long term contract prices over the next 5 yr are expected to remain in the range of US$65 to US$75/lb. As demand for uranium increases uranium prices will also need to be high enough to encourage the development of new primary production capacity. Without confidence in higher prices then investment in exploration, and in new primary production capacity, will be deferred or cancelled, in turn putting upward pressure on prices. Higher prices will be required before investors develop the confidence to make long term investments in the primary production of uranium.

2.13.1.1 Uranium Marketing and Pricing Assumptions

Sales of U_3O_8 are predominantly undertaken on a long term contract basis. Current data suggests a relatively small surplus over the next 5 yr and this is supported by the general consensus on long term contract price forecasts of around US$65 to US$75/lb U_3O_8. The forecasts are reflecting a market in relative equilibrium with little or no upward pressure on prices.

In GMEL's view the apparent relative equilibrium in the market over the next few years masks pressures that are likely to build in the medium to long term and the long term. GMEL considers that the price for uranium will range between US$70 to US$90/lb. The Base Case financial evaluation assumes a mid-point of $80/lb. The price assumption of US$80/lb by 2015 assumes an annual growth of only 5%. Over the previous five years U_3O_8 long term contract prices have escalated at over 21% pa.

2.13.2 Rare Earths

The review of the REO market has been based on two reports: "Marketing Options for the Kvanefjeld Rare Earths Project" (September 2009), undertaken by Industrial Minerals Company of Australia Pty Ltd (IMCOA) and "Rare Earths: Worldwide Markets, Applications, Technologies" by BCC Research (June 2009).

Rare Earth Elements (REEs) are a group of specialty metals with unique physical, chemical and light-emitting properties. The REE industry is seeing dramatic increases in REE demand, owing to an increasing range of technological applications, and growing recognition of the role REEs have to play in reducing the impact of global warming and climate change. The unique properties of REEs make them critical materials to many emerging technologies which are becoming increasingly commonplace in today's society.

Rare earths are characterised as being "light" or "heavy". Heavy rare earths are relatively less abundant than light rare earths but are more valuable. Rare earths are sold as mineral concentrates, as mixed carbonate concentrates and as separated rare earth oxides.

China dominates the world market which was estimated to be worth US$1.3 billion in 2008. It is noted that:

- China has more than 40% of the world's resources
- China supplies more than 92% of the world's requirements
- China consumes more than 60% of the world's production.

China is currently depleting its rare earth resource inventory at a rate that is significantly out of proportion to its resource base, and the Chinese Government has started to take steps to address the situation. Over the past 3 or 4 yr, the government has taken a number of steps including imposition of quotas and taxes and, prohibition of foreign investment, to help ensure that Chinese reserves of rare earths are developed primarily for the benefit of Chinese domestic manufacturing industries. In particular the Government has adopted policies to encourage rare earth producers to go downstream and add value. The effect of these steps will ultimately be a reduction in the volume of rare earths produced in China that will be exported to the world.

Until the global financial crisis in 2008, the market for rare earths had grown of the order of 8% to 11% pa in the prior decade (the exception being 2001/2). It is expected that growth in demand will return to these levels as the global economy recovers (IMCOA 2009).

China's intent to develop its resources for the prime benefit of its own manufacturing industry will constrain the supply available for international consumers, thereby tending to create upward pressure on prices. In an environment where demand is forecast to increase substantially the upward pressure on prices could be significant.

2.13.2.1 Rare Earth Pricing Assumptions

As noted above, IMCOA has forecast an 8% to 11% pa growth in demand up to 2014. Despite this strong underlying demand growth IMCOA forecasts that short to medium term rare earth carbonate prices in China will remain relatively stable, ranging between US$5/kg and US$7/kg (FOB China).

Prices for rare earth carbonates are determined by reference to the prices for the rare earth elements contained in the concentrate. The mix of rare earth elements contained in the Kvanefjeld orebody should produce a carbonate concentrate that will attract a premium of approximately US$1/kg when compared with the bench mark - Baotou carbonate (IMCOA).

Based on the short to medium term forecast for rare earth carbonate prices in China, IMCOA's view is that, for the purposes of the Study, GMEL should use a price range from US$7.50/kg REO (US$3250/t carbonate FIS) to US$10/kg REO (US$4500/t carbonate FIS).

IMCOA's price forecast shows only a modest overall price growth [less than 5% compound annual growth rate (CAGR)] over the period to 2014. GMEL is of the view that the price forecast by IMCOA is conservative and that future REO prices will be significantly higher.

BCC Research are estimating that prices for all rare earths will increase at between 20% and 30% CAGR over the same period. The BCC Research forecast is based on the view that recent Chinese actions [reduction of mine production and export quotas, and the imposition of an export tax on selected rare earths] will continue to have a strong upward influence on prices against a backdrop of steady demand growth,

GMEL concurs with BCC Research's view that the reduction of supply from the Chinese market, together with demand growth returning to historical levels, will cause REO carbonate prices to return to pre "global crisis" levels in 2010 and to increase steadily thereafter.

Given the above, GMEL expects that by 2015, prices will fall within the range US$7.50/kg to US$18.50/kg, tending towards the upper end of this range. The base case evaluation uses US$13/kg, which is the midpoint of this range.

2.14 FINANCIAL EVALUATION

The GMEL Financial Model is a discounted unleveraged cash flow model (DCF) of the Kvanefjeld project which has been built in Excel. The model describes a Base Case and has the capability to evaluate the impact of variations in key inputs on financial metrics for the Kvanefjeld project. The model uses net present value (NPV), internal rate of return (IRR) and payback period as its evaluation metrics.

The following assumptions about the project form the basis of the model:

- the capacity of the project is 10.8 Mtpa
- the capital cost of the project has been estimated at US$2.31 billion
- construction is scheduled to commence in 2013
- production is scheduled to commence in 2015
- the project has a life of 23 yr.

2.14.1 Base Case Summary

The financial outcomes of the model for the Base Case indicate a robust project that will generate over US$8.9 billion in free cash flow over its operating life. The outcomes are presented in Table 2.14.1:

Table 2.14.1
Base Case Financial Outcomes

Parameter	Pre-tax	Post tax
NPV	US$2 181 M	US$1 282 M
IRR	24%	19%
Capital Payback Period	5 to 6 yr	6 to 7 yr

In terms of modelling the net cost of producing either U_3O_8 concentrates or REO carbonates the philosophy for allocating costs in the model, recognising that U_3O_8 extraction precedes rare earths extraction in the processing plant flow sheet, results in all costs to the point at which the U_3O_8 and REE streams can be separated being allocated to the Carbonate Pressure Leach (CPL) Circuit costs. This upwardly skews the apparent cost of producing U_3O_8.

However, over the life of the project, the average cost of producing a pound of U_3O_8, is exceeded by the co-product credit that is earned from the sale of REO carbonates. Put another way, the cost of producing a pound of U_3O_8 is negative.

If the U_3O_8 concentrates were considered to be a by-product of REO carbonate production, rather than the other way around, then the cost of producing REO carbonates would be calculated, net of credits from the sale of the U_3O_8 concentrates. As with the example given above, the by-product credit that is earned from the sale of U_3O_8 concentrates exceeds the average cost of producing REO carbonates, hence the cost of producing a kilogram of REO carbonate is negative.

For a project of this size and complexity these are very encouraging preliminary results as there is considerable potential to add value to the project and optimise its value.

2.14.2 Sensitivities

Sensitivity analyses show that the two inputs to the model that have the most significant impact on the Project's financial results are;

- the price for U_3O_8 produced
- the price for RE carbonate concentrate produced.

The project is also sensitive to the capital and operating costs of the carbonate pressure leach circuit and a relatively modest reduction in operating or capital costs in this circuit has the potential to significantly improve the Project's financial parameters.

The movements in revenue over the life of mine production, given constant ore throughput, also identify the uranium head grade as a key sensitivity for the project.

Figure 2.14.1 shows the sensitivity of the project's NPV to changes in product prices and changes in capital and operating costs. The steeper the slope of the line the more sensitive the project NPV is to changes in the input variable.

Figure 2.14.1

Input Sensitivities