GREENLAND
MINERALS AND ENERGY LTD

ASX: GGG

Company Announcement, Friday 4th May, 2012

Kvanefjeld Prefeasibility Study Confirms a Long-Life, Cost Competitive Rare Earth Element - Uranium Project

Key Study Outcomes Include:

- The Prefeasibility Study outlines an initial development scenario with an annual mine throughput of 7.2 Mt, to generate four main products as well as a high-grade zinc sulfide concentrate:
  - *Uranium Oxide – 2.6 Mlbs pa U_3O_8*
  - *Heavy Rare Earth Hydroxide – 4,200 tpa TREO*
  - *Mixed Rare Earth Carbonate – 10,400 tpa TREO*
  - *Light Rare Earth Carbonate – 26,200 tpa TREO*

- Unit costs of production are low; less than US$31/lb U_3O_8 and less than US$8/kg TREO (as contained in the three combined rare earth products). This places the Kvanefjeld Project into the bottom half of the cost curve for uranium producers and it will be one of the lowest cost REE producers worldwide.

- The Kvanefjeld Project generates a pre-tax, ungeared internal rate of return of 32% and a cash payback period less than 4 years, based on long term prices of US$70/lb U_3O_8 and US$41.60/kg TREO. The pre–tax NPV is US$4,631 M (at 10% discount rate).

- Capital costs of an open cut mine, a mineral concentrator and a refining plant, capable of treating 7.2 Mtpa, is estimated to cost US$1.53 Billion (inclusive of US$247 M contingency).

- The Project has an initial mine life of over 33 years, based on the indicated mineral resources established near surface at the Kvanefjeld deposit. Construction is scheduled to commence in 2014 and first production in 2016.

- Highly efficient process flowsheet established drawing on conventional, proven methodologies;
  - *Beneficiation utilising froth flotation achieves high up-grade ratio with dominant REE-uranium minerals concentrated into <15% of ore mass*
  - *Atmospheric leaching of mineral concentrates using sulfuric acid results in >90% extraction of heavy REEs and uranium, with slightly lower LREE extraction. High purity concentrates recovered using solvent extraction.*

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- The Kvanefjeld Mineral Resource contains 619 Mt and 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. Local infrastructure is well established, with the local town of Narsaq within 10kms of the mine and an international airport at Narsarsuaq 30 kms away.

- Mining studies indicate a large open pit with a low waste strip ratio (1.1 tonne of waste for each tonne of ore) in addition to the highest grade material occurring at surface. Total life of mine production is 232.6 Mt at an average mine grade of 341 ppm U_3O_8 and 1.22% TREO.
- The recent exploration programs have resulted in a significant increase in resource inventory, which now includes the new discovery at Zone 2. The Zone 3 resource will be added during Q2 2012. The total resource currently stands at 861Mt, and contains 512 Mlbs U_3O_8, 9.22 Mt TREO and 1.98 Mt Zn (at a 150ppm U_3O_8 cut-off grade). This provides an opportunity to potentially increase the Project mine life to in excess of 60 years.

Introduction

Greenland Minerals and Energy Limited ("GMEL" or "the Company") is pleased to announce the outcomes of a comprehensive Prefeasibility Study (PFS) for the development of the Kvanefjeld Multi-Element Project (rare earth elements, uranium, zinc). The PFS builds upon extensive drilling, research and testwork programs conducted by the Company over the past five years in association with internationally respected research institutions and accredited analytical facilities. The finalization of the PFS marks another key milestone in the progressive advance of the Kvanefjeld Project.

The PFS and also draws on extensive historical work conducted by Danish authorities and scientists in the 1970s and early 1980s, which culminated in an 'historic' prefeasibility study published by Risø National Laboratory (Risø) in 1983. In contrast to the Risø studies that focused solely on the exploitation of uranium, GMEL has evaluated Kvanefjeld for the production of REEs and uranium to access the inherently greater value of a multi commodity resource.

The PFS demonstrates the clear potential for Kvanefjeld to be developed as a long-life, cost effective producer of heavy, light and mixed rare earth concentrates, uranium oxide and zinc. The production profile is of global significance in terms of output capacity, and low production costs. The high upgrade ratio achieved using flotation, the high extraction of uranium and heavy REEs from mineral concentrates using a conventional atmospheric acid leach, and the ability to produce multiple RE products represent key advantages of the Kvanefjeld Project.

Through 2010 and 2011, focused research programs led to important metallurgical breakthroughs. The identification of an effective method to beneficiate the Kvanefjeld ore to generate a low mass, REE-uranium-rich mineral concentrate opened the opportunity to leach both REEs and uranium with conventional acidic solutions under atmospheric conditions; a highly favourable outcome by industry standards. Importantly, this eliminated the need for a whole-of-ore alkaline pressure leach circuit that was considered in the 'Interim Prefeasibility Study', released by GMEL in Q1 2010.

The removal of reagent-consuming silicate minerals through beneficiation allows for the effective use of conventional acidic solutions to leach REEs and uranium from the mineral concentrates. It also allows for significant downsizing of the leach circuits. These key technical developments have led to a simpler flowsheet with lower technical risk and improved capital and operating costs over those released in the 2010 Interim Prefeasibility Study.

Increasing uranium and heavy REE output can be readily achieved through subsequent development phases that future work programs are scoped to address.

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

- **Resource definition and mine plans**
  - SRK Consulting, Coffey Mining
- **Metallurgy and process development**
  - AMEC Minproc, ANSTO, SGS Oretest, CSIRO, ALS AMMTEC, Mintek
- **Environmental Impact Assessment and Social Impact Assessment**
  - Coffey Environments, Orbicon (Denmark), Grontmij (Denmark)
- **Plant engineering design, infrastructure, capital development**
  - AMEC Minproc, NIRAS (Denmark)

Background

Rare Earth and Uranium Markets

Rare earth metals and uranium are now widely recognised around the world as strategically important metals for the future. Market analysis indicates that demand for rare earths and uranium is set to rise over the next 20 years. 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 strong demand growth, particularly in the energy and electronic/optics market sectors will cause REO prices to remain high over the longer term.

Greenland

Greenland is seen as an emerging mineral province, politically stable and seeking to become increasingly financially independent from Denmark. The Company is fully licensed for all of its current development activities and exploration work programs, and is working to complete its Feasibility Studies, inclusive of an environmental and social impact assessment, which is a pre-requisite for obtaining an Exploitation License.

Community support is critically important to the successful future development of the Kvanefjeld Project and the Company is mindful of its need to respect the land, the environment and the wishes of the local people. It is, therefore, undertaking all aspects of its work in consideration and consultation with local communities. The major focus will be on Stakeholder Engagement and the development of an Environmental Impact Assessment (EIA) and Social Impact Assessment (SIA). The finalization of site locations for key infrastructure items will be dependent on ongoing stakeholder engagement.

The Executive Summary of the Kvanefjeld Prefeasibility Study is included within this release.

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Table 1. Statement of Identified Mineral Resources, Kvanefjeld Multi-Element Project

Multi-Element Resources Classification, Tonnage and Grade										Contained Metal				
Cut-off (U_3O_8 ppm)[1]	**Classification**	**M tonnes** Mt	**TREO**[2] ppm	**U_3O_8** ppm	**LREO** ppm	**HREO** ppm	**REO** ppm	**Y_2O_3** ppm	**Zn** ppm	**TREO** Mt	**HREO** Mt	**Y_2O_3** Mt	**U_3O_8** M lbs	**Zn** Mt
Kvanefjeld - March 2011														
150	**Indicated**	437	10929	274	9626	402	10029	900	2212	**4.77**	0.18	0.39	**263**	0.97
150	**Inferred**	182	9763	216	8630	356	8986	776	2134	**1.78**	0.06	0.14	**86**	0.39
150	**Grand Total**	619	10585	257	9333	389	9721	864	2189	**6.55**	0.24	0.53	**350**	1.36
200	**Indicated**	291	11849	325	10452	419	10871	978	2343	**3.45**	0.12	0.28	**208**	0.68
200	**Inferred**	79	11086	275	9932	343	10275	811	2478	**0.88**	0.03	0.06	**48**	0.20
200	**Grand Total**	370	11686	314	10341	403	10743	942	2372	**4.32**	0.15	0.35	**256**	0.88
250	**Indicated**	231	12429	352	10950	443	11389	1041	2363	**0.24**	2.53	2.63	**178**	0.55
250	**Inferred**	41	12204	324	10929	366	11319	886	2598	**0.04**	0.45	0.46	**29**	0.11
250	**Grand Total**	272	12395	347	10947	431	11378	1017	2398	**0.28**	2.98	3.09	**208**	0.65
300	**Indicated**	177	13013	374	11437	469	11906	1107	2414	**2.30**	0.08	0.20	**146**	0.43
300	**Inferred**	24	13120	362	11763	396	12158	962	2671	**0.31**	0.01	0.02	**19**	0.06
300	**Grand Total**	200	13025	373	11475	460	11935	1090	2444	**2.61**	0.09	0.22	**164**	0.49
350	**Indicated**	111	13735	404	12040	503	12543	1192	2487	**1.52**	0.06	0.13	**98**	0.27
350	**Inferred**	12	13729	403	12239	436	12675	1054	2826	**0.16**	0.01	0.01	**10**	0.03
350	**Grand Total**	122	13735	404	12059	497	12556	1179	2519	**1.68**	0.06	0.14	**108**	0.31
Zone 2 - March 2012														
150	**Inferred**	242	11022	304	9729	398	10127	895	2602	**2.67**	0.10	0.22	**162**	0.63
200	**Inferred**	186	11554	344	10223	399	10622	932	2802	**2.15**	0.07	0.17	**141**	0.52
250	**Inferred**	148	11847	375	10480	407	10887	961	2932	**1.75**	0.06	0.14	**123**	0.43
300	**Inferred**	119	12068	400	10671	414	11084	983	3023	**1.44**	0.05	0.12	**105**	0.36
350	**Inferred**	92	12393	422	10967	422	11389	1004	3080	**1.14**	0.04	0.09	**85**	0.28

Project Total

Cut-off (U_3O_8 ppm)[1]	**Classification**	**M tonnes** Mt	**TREO**[2] ppm	**U_3O_8** ppm	**LREO** ppm	**HREO** ppm	**REO** ppm	**Y_2O_3** ppm	**Zn** ppm	**TREO** Mt	**HREO** Mt	**Y_2O_3** Mt	**U_3O_8** M lbs	**Zn** Mt
150	**Indicated**	437	10929	274	9626	402	10029	900	2212	**4.77**	0.18	0.39	**263**	0.97
150	**Inferred**	424	10480	266	9257	380	9636	844	2401	**4.45**	0.16	0.36	**249**	1.02
150	**Grand Total**	861	10708	270	9444	391	9835	873	2305	**9.22**	0.34	0.75	**512**	1.98

[1]There is greater coverage of assays for uranium than other elements owing to historic spectral assays. U_3O_8 has therefore been used to define the cutoff grades to maximise the confidence in the resource calculations.

[2]Total Rare Earth Oxide (TREO) refers to the rare earth elements in the lanthanide series plus yttrium.

Note: Figures quoted may not sum due to rounding.

ABOUT GREENLAND MINERALS AND ENERGY LTD.

Greenland Minerals and Energy Ltd (ASX – GGG) is an exploration and development company focused on developing high-quality mineral projects in Greenland. The Company's flagship project is the Kvanefjeld multi-element deposit (Rare Earth Elements, Uranium, Zinc), that is rapidly emerging as a premier specialty metals project. An interim report on pre-feasibility studies has demonstrated the potential for a large-scale multi-element mining operation. For further information on Greenland Minerals and Energy visit http://www.ggg.gl or contact:

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Greenland Minerals and Energy Ltd will continue to advance the Kvanefjeld project in a manner that is in accord with both Greenlandic Government and local community expectations, and looks forward to being part of continued community discussions on the social and economic benefits associated with the development of the Kvanefjeld Project.

The information in this report that relates to exploration targets, exploration results, geological interpretations, appropriateness of cut-off grades, and reasonable expectation of potential viability of quoted rare earth element, uranium, and zinc resources is based on information compiled by Mr Jeremy Whybrow. Mr Whybrow is a director of the Company and a Member of the Australasian Institute of Mining and Metallurgy (AusIMM). Mr Whybrow has sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined by the 2004 edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves". Mr Whybrow consents to the reporting of this information in the form and context in which it appears.

The geological model and geostatistical estimation for the Kvanefjeld and Zone 2 deposits were prepared by Robin Simpson of SRK Consulting. Mr Simpson is a Member of the Australian Institute of Geoscientists (AIG), and has sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined by the 2004 edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves". Mr Simpson consents to the reporting of information relating to the geological model and geostatistical estimation in the form and context in which it appears.

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PREFEASIBILITY REPORT
Executive Summary
April 2012

KVANEFJELD MULTI-ELEMENT PROJECT

Document No. KV84-PM-RP-0000-0001

SECTION 2
EXECUTIVE SUMMARY

Table of Contents

List of Figures

List of Tables

2 EXECUTIVE SUMMARY

2.1 INTRODUCTION

Greenland Minerals and Energy Limited (the Company) has completed a Prefeasibility Study (the Study) for the development of the Kvanefjeld Multi-Element Project (the Project). The Project is centred on the Northern Ilimaussaq Intrusive Complex (the Complex), and encompasses several large, multi-element deposits. Collectively, these represent one of the world's largest mineral resources of uranium and rare earths.

The Study builds on the extensive drilling, research and testwork programs conducted by the Company over the past five (5) years, and also draws on extensive historical work conducted by Danish authorities and scientists in the 1970s and early 1980s, which culminated in a Pre-Feasibility Study published by Risø National Laboratory (Risø) in 1983.

The focus of the Study has been to evaluate the potential for development of a mine, mineral concentrator and refinery to treat 7.2 Mtpa of ore to extract rare earth elements (REEs), uranium and zinc.

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

- Resource definition and mine plans
  - SRK Consulting, Coffey Mining
- Metallurgy and process development
  - AMEC Minproc, ANSTO, SGS Oretest, CSIRO, ALS AMMTEC, Mintek
- Environmental Impact Assessment and Social Impact Assessment
  - Coffey Environments, Orbicon (Denmark), Grontmij (Denmark)
- Plant engineering design, infrastructure, capital development
  - AMEC Minproc, NIRAS (Denmark)

The Kvanefjeld Project is favourably located in a setting that is both highly accessible, and climatically mild. The Project area is located near the southwest tip of Greenland, in the Erik Aappalaartup Nunaa peninsula within the municipality of Kujalleq (Figure 2.1.1). The town of Narsaq is located at the western end of the peninsula, and is the closest of several towns in the region to the Project area (approximately 10 km). The towns of southern Greenland are serviced by air and ship, with an international airport at Narsarquaq, located approximately 45 km to the east of Narsaq (30 km from the project area). The South Greenland Municipal Council is based in the town of Qaqortoq, located 20 km to the south of Narsaq. The town of Narsaq has a deep water port facility, currently used by local fishermen and also for importing goods. The average temperature in Narsaq across the summer months is approximately 7˚C, and minus 6 ˚C through winter.

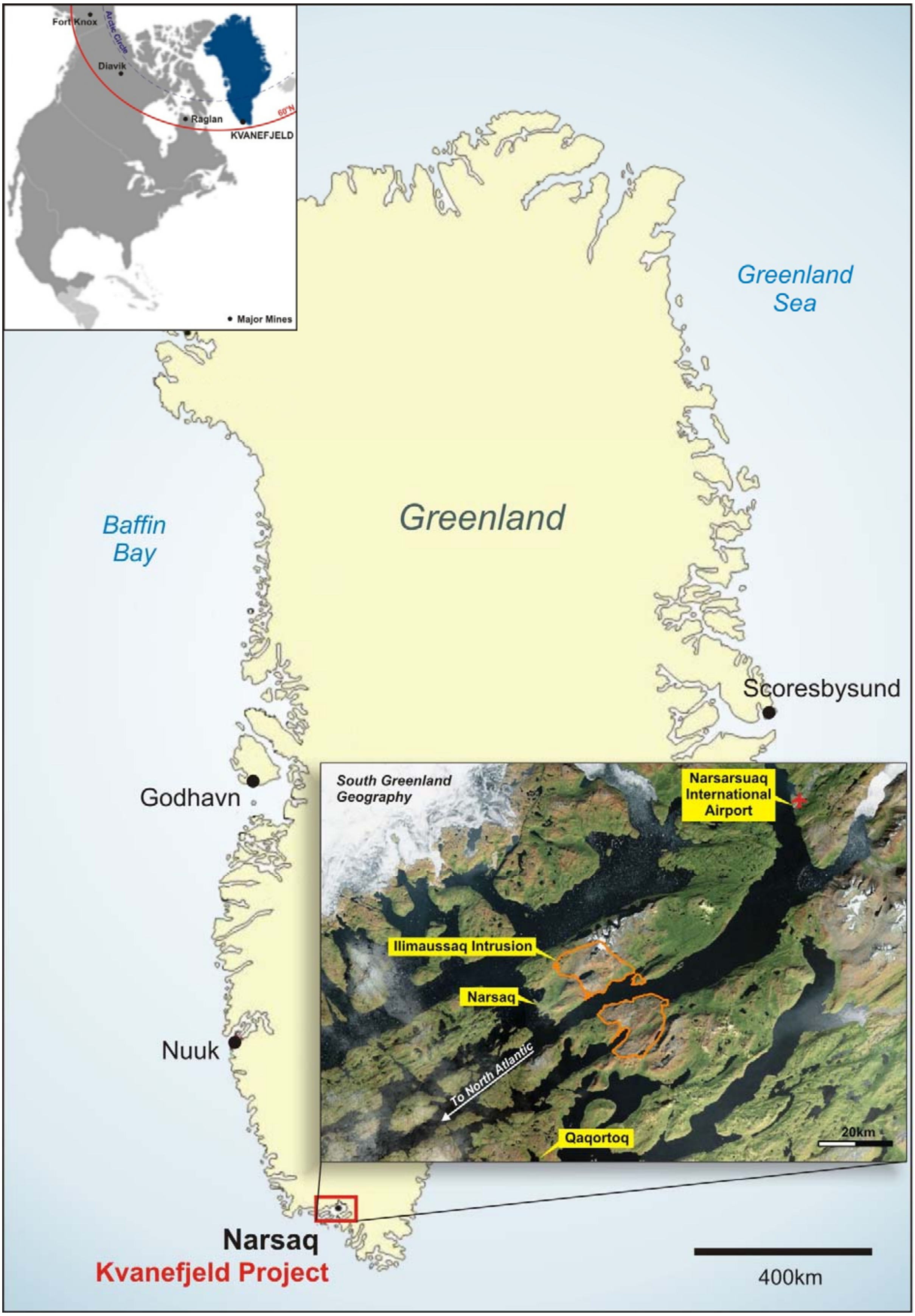

Figure 2.1.1 Site location and project geography

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:

- The Company has identified a processing flowsheet for the Project, based on beneficiation of the mined ore to produce a high grade REE-U mineral concentrate, followed by conventional atmospheric acid leach, solvent extraction and precipitation to separate the uranium and rare earths into high quality, high value products.
- The Company will produce and competitively market a range of products including rare earth hydroxides and carbonates, uranium and zinc. Each product has a ready market and in the case of uranium oxide, heavy rare earth hydroxide and, to a lesser extent, mixed rare earth carbonate, demand is expected to exceed supply in 2015. The processing plant will produce four main products as well as a high grade zinc sulphide concentrate:
    - Uranium oxide – 2.6 Mlbs pa U_3O_8
    - Heavy Rare Earth Hydroxide – 4,200 tpa TREO
    - Mixed Rare Earth Carbonate – 10,400 tpa TREO
    - Light Rare Earth Carbonate – 26,200 tpa TREO
- Unit costs of production are low; less than US$31/lb U_3O_8 and less than US$8/kg TREO (as contained in the three combined rare earth products). This places the Project into the bottom half of the cost curve for uranium producers and it will be one of the lowest cost REE producers worldwide.
- The Kvanefjeld Mineral Resource contains 619 Mt and 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. Local infrastructure is well established, with the local town of Narsaq within 10kms of the mine and an international airport at Narsarsuaq, only 45kms distant.
- Mining studies indicate a large open pit with a low waste strip ratio (1.1 tonne of waste for each tonne of ore) in addition to the highest grade material occurring at surface. Total life of mine production is 232.6 Mt at an average mine grade of 341 ppm U_3O_8 and 1.22% TREO.
- Engineering studies have determined that an open cut mine, a mineral concentrator and a refining plant, capable of treating 7.2 Mtpa, is estimated to cost US$1.53 Billion. Construction is scheduled to commence in 2014 and first production in 2016. The Project has a mine life of over 33 years, based on the higher grade, mineral resources established near surface at the Kvanefjeld deposit.
- The Project generates a pre-tax, ungeared internal rate of return of 32% and a cash payback period of 3 yr to 4 yr, based on long term prices of US$70/lb U_3O_8 and US$41.60/kg TREO. The pre–tax NPV is US$4,631 M.
- The recent exploration programs have resulted in a significant increase in resource inventory, which now includes the new discovery at Zone 2. The Zone 3 resource will be added during Q2 2012. The total resource currently stands at 861Mt, and contains 512 Mlbs U_3O_8, 9.22 Mt TREO and 1.98 Mt Zn (at a 150ppm U_3O_8 cut-off grade). This will potentially increase the Project mine life to in excess of 60 years.
- 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 over the next 20 years. 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 strong demand growth, particularly in the energy and electronic/optics market sectors will cause REO prices to remain high over the longer term.

- Greenland is seen as an emerging mineral province, politically stable and seeking to become increasingly financially independent from Denmark. The Company is fully licensed for all of its current development activities and exploration work programs, and has been actively working with the Government to provide a means by which the Company can complete its Feasibility Studies, inclusive of an environmental and social Impact assessment, which is a pre-requisite for obtaining an Exploitation License.
- Community support is critically important to the successful future development of the Project and the Company is mindful of its need to respect the land, the environment and the wishes of the local people. It is, therefore, undertaking all aspects of its work in consideration and consultation with local communities. The major focus will be on Stakeholder Engagement and the development of an Environmental Impact Assessment (EIA) and Social Impact Assessment (SIA).

2.3 GEOLOGY AND EXPLORATION

2.3.1 Geology of Ilimaussaq Intrusive Complex

The Ilimaussaq Intrusive Complex is one of the most unique geological environments on earth, and is the type-locality for layered peralkaline igneous complexes. Measuring 17 x 8 km, the Complex extends from the Narsaq Peninsula southward across two other peninsulas to straddle the Tunulliarfik and Kangerluarssuk fjords. The Complex is one of several mid-Proterozoic alkaline massifs that were emplaced into rift-related volcano-sedimentary sequences of the Gardar Province and underlying granitic basement. The emplacement of the supracrustal sequences and intrusions of the Gardar Province was strongly controlled by ENE-WSW block faulting that is expressed topographically by the current-day elongate fjords and narrow peninsulas. Rocks of the Gardar Province intruded, and were emplaced onto, granitic basement rocks in a continental rift setting.

The Ilimaussaq Complex is noted for layered syenites with extreme enrichment in sodium and incompatible elements, including actinides, lanthanides, zirconium, tantalum, niobium, phosphorous and fluorine. The Ilimaussaq has an overall peralkaline character, with whole-rock (Na_2O +K_2O) > Al_2O_3 on a molar basis, so that feldspathoids are important mineral constituents of the syenites.

The oldest phases of the Ilimaussaq are augite syenite and alkali granite emplaced at the margins and roof of the intrusion to form a partial shell around the younger and volumetrically dominant sequence of layered nepheline syenites. The province-wide ENE-WSW block faulting sees the deepest portion of the layered syenite sequence exposed on the SE side of the Complex around the Kangerluarssuk fjord where a rhythmically-layered sequence known as the kakortokites occurs. The kakortokites comprise alternating black, white and red sheets that are respectively rich in cumulate of arfvedsonite, nepheline and eudialyte.

Overlying the kakortokites, and central within the Complex, are the naujaites. Naujaite is the most common phase of the exposed portion of the Complex. The naujaites are coarse grained to megacrystic, leucocratic cumulates of alkali feldspar, feldspathoids (especially sodalite), arfvedsonite, aegirine and eudialyte.

The lujavrites are the youngest phase within the Complex and occur as sills and dykes at all levels from the floor to the roof of the Complex. The lujavrites are mesocratic, fine-medium grained syenites. The lujavrites are strongly peralkine to hyper-peralkaline. Lujavrite mineralogy is dominated by alkali feldspars, sodalite, nepheline, analcime, aegirine and arfvedsonite. Flow lamination marked by parallel orientation of the coarser mineral grains is a distinctive feature of the lujavrites.

The lujavrites are further sub-divided into aegirine-rich “green lujavrite”, and arfvedsonite-rich “black lujavrite”. Green lujavrite occurs deeper in the Ilimaussaq Complex, especially to the SE portion of the Complex where deeper paleodepths are exposed. Black lujavrite occurs at higher levels in the Complex and generally features the highest concentrations of uranium, REEs, and zinc. A pegmatoidal lujavrite variety is also documented in areas and is referred to as the medium-coarse (MC) lujavrite.

In the uppermost black lujavrite sections, total REE concentrations can exceed 1.5%, and U_3O_8 can exceed 400 ppm. With increasing depth, grades of REEs and uranium drop to sub-economic levels. The thickness of the mineralized sections can exceed 250m in dome-like structural culminations, and such areas can persist over several square km’s. The mineralized lujavrite sections are mostly preserved in the northern half of the Ilimaussaq Complex. The kakortokites located near the southern margin of the Complex host significant, but low-grade tantalum-niobium-zirconium and REE mineralisation.

2.3.2 Multi-Element Deposits

Several substantial deposits of multi-element mineralisation (REEs, uranium, zinc) are hosted in the lujavrites of the northern Ilimaussaq Complex. A world-class multi-element resource has been established at Kvanefjeld, and substantial new satellite deposits have recently been confirmed at Zones 2 and 3. Geological evidence suggests that Zones 2 and 3 represent outcropping, or near-surface expressions of a mineralised system that extends over several kilometres from Kvanefjeld, and is interconnected at depth.

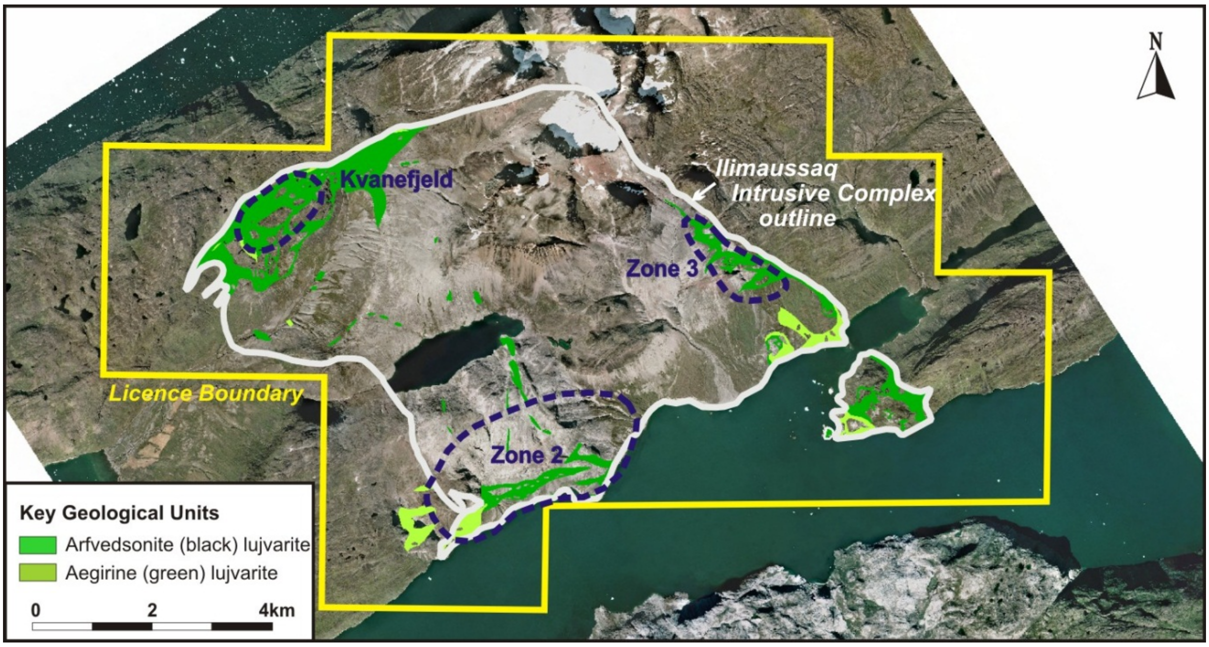

Figure 2.3.1 Location of Kvanefjeld and Zones 2 and 3 Deposits

The Kvanefjeld deposit occurs at the northern end of the Complex where lujavrite has partially intruded into the roof sequence of Gardar Province supracrustals. The Zone 2 and Zone 3 deposits occur as sills within the naujaites at an intermediate level within the Complex. Kvanefjeld has been the subject of extensive drilling, mapping, mineralogical and geochemical studies since the 1960's. Active participants have included the Greenland and Danish geological surveys, university researchers from the broader European community, and mineral explorers such as the Company. Zone 2 and Zone 3 are recent discoveries with drilling undertaken since 2008.

At Kvanefjeld, arfvedsonite-rich black lujavrite outcrops extensively across the plateau region. Recent glaciation has partially unroofed the upper most lujavrites, leaving a mega-breccia of earlier syenite phases and roof basalts entrained within the lujavrite matrix. The mega xenoliths consist of isolated blocks of all sizes randomly distributed through the lujavrite body, as well as chains of rafts that "float" at relatively constant vertical heights within the lujavrite body. Zone 2 is located within the naujaite core of the Ilimaussaq Intrusive Complex, some 6km south of Kvanefjeld, at an intermediate stratigraphic position within the Complex. Two stacked, lenticular sills are emplaced into massive naujaite, and capped by massive naujaite. The lujavrite sequence has a 500m vertical extent like Kvanefjeld, but the base at sea level is lower than at Kvanefjeld. The Zone 2 lujavrite crops out on the cliff face on the north shore of Tunugliarfik, and dips shallowly northwards (Figure 2.3.2). The surface projection of Zone 2 is some 800m x 500m, and is open to the north. Deep diamond drilling has shown that Kvanefjeld and Zone 2 are likely connected at depth.

Kvanefjeld and the satellite lujavrite deposits are divided into two fundamental domains, with an upper uranium-REE enriched domain, and a lower Zr-enriched domain. The transition between the two fundamental domains is both a grade transition and a mineralogical transition. At Kvanefjeld, the upper domain has a mineral assemblage dominated by steenstrupine and the lower domain has a eudialyte-monazite assemblage. The transition from the steenstrupine domain to the eudialite-monazite domain also corresponds to a grade break of approximately 300ppm U_3O_8. The geometry of the domains will see the majority of mine production drawn from the upper steenstrupine-dominated assemblage for at least the first 25 years.

Figure 2.3.2 View over Tunugliarfik Fjord Ilimaussaq Complex

2.3.3 Ore Mineralogy

Steenstrupine is the most important host to both REEs and uranium in the lujavrite-hosted multi-element deposits. It is a complex sodic phospho-silicate mineral. Mineralogical studies suggest that steenstrupine commonly contains between 0.2% and 1% U_3O_8, and likely hosts over half of the uranium contained within the deposits.

Within the black lujavrite, the grain size of the steenstrupine commonly ranges from 75 µm to over 500 µm, whilst in the MC lujavrite, grain size can exceed 1mm. Other minerals that are important hosts to REEs include the phosphate mineral vitusite and, to a lesser extent, britholite and minor monazite. Aside from steenstrupine, uranium is also hosted in unusual zirconium silicate minerals of the lovozerite and eudialyte groups. In these zirconium-silicates a portion of the zirconium is substituted by several hundred ppm each of uranium, yttrium, HREE and tin. Zinc is hosted in the sulphide mineral sphalerite, which is the dominant sulphide throughout the deposit. Lovozerite group minerals coexist with steenstrupine, whereas eudialyte group minerals are present at deeper, lower-grade levels of the deposits.

Table 2.3.1 contains details of the significant ore minerals found in lujavrite.

Table 2.3.1 Significant Ore Minerals			
Mineral	**Family**	**Type**	**Commodities**
Steenstrupine	Cyclo-silicate	Phospho-silicate	U, REE
Britholite	Apatite-Group	Phospho-silicate	U, REE
Phosinaite	Cyclo-silicate	Phospho-silicate	HREE
Vitusite	Phosphate	Na-Phosphate	REE
Xenotime	Phosphate	Phosphate	HREE
Monazite	Phosphate	Phosphate	LREE
Townendite	Cyclo-silicate	Zircono-Silicate	U, HREE, Sn
Eudialyte	Cyclo-silicate	Zircono-Silicate	REE
Catapleiite	Cyclo-silicate	Zircono-Silicate	REE
Kapustinite	Cyclo-silicate	Zircono-Silicate	REE
Cerite	Neso-silicate	Silicate	LREE
Uranothorite	Neso-silicate	Silicate	U
Nacareniobsite	Cyclo-silicate	Silicate	REE, Nb
Sorensite	Ino-silicate	Silicate	Sn, Be
Sphalerite	Sulphide	Sulphide	Zn

In the upper, higher grade portions of Kvanefjeld (>300 ppm U_3O_8) phosphate bearing minerals (e.g. steenstrupine) are the dominant hosts to REEs and uranium, with the zirconium silicates being of secondary importance. However, at greater depth, the zirconium silicates become increasingly important hosts to uranium. The mine schedule established for the Study is focussed in greater than 300 ppm U_3O_8 resource material that dominates the upper level of the Kvanefjeld deposit.

Lujavrite gangue minerals consist of 60% sodium aluminosilicates and 40% sodium-iron pyriboles. The total ore minerals typically comprise 5 – 10% of the rock volume as disseminated grains. Trace components of the rock also include water-soluble sodium silicate and sodium phosphate minerals together with humic material. Uranium, REE, and zinc mineralisation is orthomagmatic, having crystallised from the lujavrite melt along with the silicate gangue.

The typical distribution of minerals in lujavrite is presented in Figure 2.3.3. It is important to note that the five (5) main gangue minerals make up approximately 86% of the mass, hence the majority of the uranium and REEs are locked in a very small quantity of material. Understanding this, and its significance to the potential of beneficiation, has been one of the key technical breakthroughs achieved on the Project.

Table 2.3.2 contains details of the lujavrite gangue mineralogy.

Table 2.3.2 Lujavrite Gangue Mineralogy

Mineral	Type	Class	Comments
Aregirine	Pyroxene	Na-Fe Silicate	
Arfvedsonite	Amphibole	Na-Fe Silicate	Trace refractory Li, F
Neptunite	Inosilicate	Ti-Na-Fe-Mn Silicate	Trace refractory Li, F
Naujakasite	Phyllosilicate	Na-Fe-Mn Alumino-silicate	Trace refractory Li, F
Biotite	Phyllosilicate	K-Fe Alumino-silicate	Trace refractory Li, F
Microcline	Feldspar (Na+K)	Alumino-silicate	
Albite	Feldspar (Na)	Alumino-silicate	
Nepheline	Feldspathoid (Na+K)	Alumino-silicate	Acid susceptible
Sodalite	Feldspathoid (Na)	Alumino-silicate	Acid susceptible
Analcime	Zeolite (Na)	Hydrated Alumino-silicate	Acid susceptible
Natrolite	Zeolite (Na)	Hydrated Alumino-silicate	Acid susceptible
Ussingite	Zeolite (Na)	Hydrated Alumino-silicate	Acid susceptible

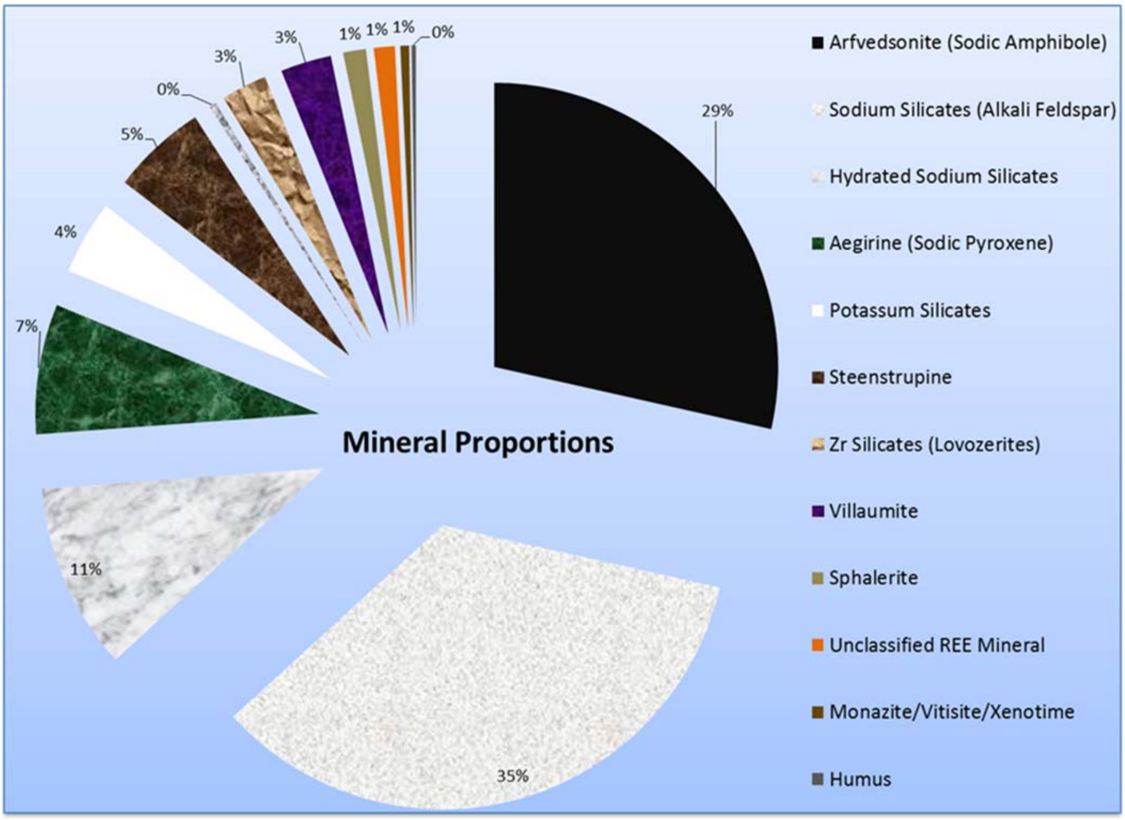

Figure 2.3.3 Mineral Proportions of Lujavrite

2.3.4 Mineral Resources

The current mineral resources were estimated by SRK Consulting (Australasia) Pty Ltd, and categorised in accordance with the JORC Code (2004). The latest Kvanefjeld mineral resource estimate was publicly released in March 2011, and was based on drilling completed to the end of the 2010 field season. The Zone 2 estimate was publicly released in March 2012, and was based on drilling completed to the end of the 2011 field season.

The Kvanefjeld deposit has a total resource of 619 Mt, and is characterised by thick, mostly sub-horizontal slabs of lujavrite. The highest grades occur near surface, with grades of REEs, uranium and zinc decreasing with depth. Features of the Kvanefjeld resource include:

Kvanefjeld - global resource:

- 619 Mt containing 350 Mlbs U_3O_8, 6.6 Mt TREO, 3 Blbs zinc
- REE resource inventory includes 240,000 t HREO, 530,000 t Y_2O_3
- 437 Mt of resources in the Indicated category

Kvanefjeld - higher grade upper section:

- 120 Mt @ 400 ppm U_3O_8, 497 ppm HREO, 1100 ppm Y_2O_6, 1.4% TREO, 0.25% Zn

A further 240 Mt of inferred resources have been established at Zone 2. Zone 2 features many similarities to the Kvanefjeld deposit, including a higher grade upper section.

Zone 2 - higher grade upper lens:

- 119 Mt @ 400 ppm U_3O_8, 414 ppm HREO, 940 ppm Y_2O_3, 1.2% TREO, 0.3% Zn

The mineral resources have been delineated by diamond core drilling from surface. The majority of grade information is based on chemical assaying of half-core, although 15% of the Kvanefjeld assay data is based on historical spectral assays. Chemical assays were performed by NATA-certified laboratories in Australia, using multi-acid-digest and ICP-OES and ICP-MS instruments. For the samples with only spectral assay data, REE grades were estimated from uranium grades using linear models, based on linear regression parameters calculated from the more extensive set of chemical assays.

The resource definition drilling has been by diamond coring from surface, either NQ or BQ diameter, with HQ diameter holes utilised for geotechnical assessments and metallurgical sampling. Drill hole spacing is approximately 70m x 70m over the northeast of Kvanefjeld, widening to 140m x 140m in the southwest. Zone 2 has a wider hole spacing of between 150m and 300m. Drill hole locations are partially constrained by locally rugged topography, which inhibits drilling on an exact grid spacing. The majority of holes are oriented vertical, or near vertical, to achieve intercepts that are close to true thickness given the sub-horizontal orientation of the lujavrite sills. Recovery is generally 100%, or close to 100%.

Kvanefjeld has a long exploration history, with 65 holes drilled by Danish institutions in the period

1958-1977, and 156 holes completed by the Company in the period 2007-2010, as summarised in Table 2.3.3. Drilling completed at Kvanefjeld by the end of 2010 totalled 45,000m of drill core and 23,000 assays.

Zone 2 has a recent exploration history, with all but one hole drilled by the Company in the 2010/2011 field campaigns. The discovery hole was drilled by the Company near the close of the 2008 field season. A total of 23 holes, 10,351m of core and 4,600 assay samples have been completed to date at Zone 2.

Table 2.3.3 Summary of Kvanefjeld Drilling

Drill Program	Holes	Metres
GMEL resource definition (2007 -2010)	130	31,436
GMEL Geotechnical (2009)	12	1,870
GMEL Metallurgic (2009)	14	2,254
Historical (1958-1977)	65	9,830

Mineral resources were estimated using industry best-practice geological modelling techniques to constrain the lujavrite volume together with geostatistical modelling to constrain the distribution of grades within the lujavrite volumes. Leapfrog software was used to model lujavrite and sub-domain volumes directly from drill hole intercepts using a 3D-splining technique. Isatis and Surpac software was used to model the distribution of grades using Ordinary Kriging. Mineral Resources are summarised over a series of uranium cut-off grades in Table 2.3.4.

The kriging quality parameter generated during Ordinary Kriging was the primary consideration in classifying the estimation confidence of the mineral resources. For the Project, there was a case to be made for classifying a portion of the mineral resource as Measured had U_3O_8 been the only metal of interest. Given the material importance of REE grades to the feasibility study, an Indicated classification was applied instead due to the proportion of samples that only had spectral uranium assays (approximately 15%). Zone 2 was classified as Inferred, due to the wider drilling spacing performed to date.

Simple geochemical indices, such as the Hf/Yb ratio, can be used to define the grade-breaks between the sub-layers with better resolution than uranium and total REE grades alone. The current resource estimation methodology utilises 3D models of the Hf/Yb ratio to constrain the block model into the geochemical/mineralogical sub-domains.

Mineral resources that are utilised within the Study are from the Kvanefjeld deposit, and only include resource tonnes in the Indicated category. With further drilling the confidence in inferred high-grade resources at Zone 2 and 3 can be improved, and then factored into the mining schedule to improve both grades and life of mine.

Kvanefjeld is the world's largest JORC Code compliant REE-U mineral resource. There are two dominant types of rare earth deposits, those associated with carbonatites (Mountain Pass, Mt Weld), and those associated with peralkaline igneous complexes (Thor Lake, Strange Lake, Kvanefjeld).

Rare earths are typically described as heavy or light. The heavy rare earths (HREs) include dysprosium, terbium, yttrium and europium. The light rare earths (LREs) are lanthanum, cerium, praseodymium, neodymium and samarium.

LREs nearly always occur in much greater abundance than HREs but it is generally the deposits that are associated with peralkaline complexes that are enriched with lucrative HREs.

Accordingly, the REE resource at Kvanefjeld is not just extremely large, but also contains a favourable mix of REEs, with a relative enrichment of the HREs. Kvanefjeld is also strongly enriched in yttrium, and combined they account for 14% of the rare earth resource at Kvanefjeld.

The value of the Kvanefjeld deposit is therefore not just a function of the overall grade and tonnage of the resource but also a function of relatively high proportion of HREs contained within the orebody. The HREs are relatively scarce and, in the face of demand for an ever increasing range of applications for the constituent metals, their value has significantly increased.

Table 2.3.4 Multi-Element Resources Classification, Tonnage and Grade

Multi-Element Resources Classification, Tonnage and Grade										Contained Metal				
Cut-off	**Classification**	**M tonnes**	**TREO**[2]	**U_3O_8**	**LREO**	**HREO**	**REO**	**Y_2O_3**	**Zn**	**TREO**	**HREO**	**Y_2O_3**	**U_3O_8**	**Zn**
(U_3O_8 ppm)[1]		Mt	ppm	ppm	ppm	ppm	ppm	ppm	ppm	Mt	Mt	Mt	M lbs	Mt
Kvanefjeld - March 2011														
150	**Indicated**	437	10929	274	9626	402	10029	900	2212	**4.77**	0.18	0.39	**263**	0.97
150	**Inferred**	182	9763	216	8630	356	8986	776	2134	**1.78**	0.06	0.14	**86**	0.39
150	**Grand Total**	619	10585	257	9333	389	9721	864	2189	**6.55**	0.24	0.53	**350**	1.36
200	**Indicated**	291	11849	325	10452	419	10871	978	2343	**3.45**	0.12	0.28	**208**	0.68
200	**Inferred**	79	11086	275	9932	343	10275	811	2478	**0.88**	0.03	0.06	**48**	0.20
200	**Grand Total**	370	11686	314	10341	403	10743	942	2372	**4.32**	0.15	0.35	**256**	0.88
250	**Indicated**	231	12429	352	10950	443	11389	1041	2363	**0.24**	2.53	2.63	**178**	0.55
250	**Inferred**	41	12204	324	10929	366	11319	886	2598	**0.04**	0.45	0.46	**29**	0.11
250	**Grand Total**	272	12395	347	10947	431	11378	1017	2398	**0.28**	2.98	3.09	**208**	0.65
300	**Indicated**	177	13013	374	11437	469	11906	1107	2414	**2.30**	0.08	0.20	**146**	0.43
300	**Inferred**	24	13120	362	11763	396	12158	962	2671	**0.31**	0.01	0.02	**19**	0.06
300	**Grand Total**	200	13025	373	11475	460	11935	1090	2444	**2.61**	0.09	0.22	**164**	0.49
350	**Indicated**	111	13735	404	12040	503	12543	1192	2487	**1.52**	0.06	0.13	**98**	0.27
350	**Inferred**	12	13729	403	12239	436	12675	1054	2826	**0.16**	0.01	0.01	**10**	0.03
350	**Grand Total**	122	13735	404	12059	497	12556	1179	2519	**1.68**	0.06	0.14	**108**	0.31
Zone 2 - March 2012														
150	**Inferred**	242	11022	304	9729	398	10127	895	2602	**2.67**	0.10	0.22	**162**	0.63
200	**Inferred**	186	11554	344	10223	399	10622	932	2802	**2.15**	0.07	0.17	**141**	0.52
250	**Inferred**	148	11847	375	10480	407	10887	961	2932	**1.75**	0.06	0.14	**123**	0.43
300	**Inferred**	119	12068	400	10671	414	11084	983	3023	**1.44**	0.05	0.12	**105**	0.36
350	**Inferred**	92	12393	422	10967	422	11389	1004	3080	**1.14**	0.04	0.09	**85**	0.28
Project Total														
Cut-off	**Classification**	**M tonnes**	**TREO**[2]	**U_3O_8**	**LREO**	**HREO**	**REO**	**Y_2O_3**	**Zn**	**TREO**	**HREO**	**Y_2O_3**	**U_3O_8**	**Zn**
(U_3O_8 ppm)[1]		Mt	ppm	ppm	ppm	ppm	ppm	ppm	ppm	Mt	Mt	Mt	M lbs	Mt
150	**Indicated**	437	10929	274	9626	402	10029	900	2212	**4.77**	0.18	0.39	**263**	0.97
150	**Inferred**	424	10480	266	9257	380	9636	844	2401	**4.45**	0.16	0.36	**249**	1.02
150	**Grand Total**	861	10708	270	9444	391	9835	873	2305	**9.22**	0.34	0.75	**512**	1.98

1 There is greater coverage of assays for uranium than other elements owing to historic spectral assays. U_3O_8 has therefore been used to define the cutoff grades to maximise the confidence in the resource calculations

[2]Total Rare Earth Oxide (TREO) refers to the rare earth elements in the lanthanide series plus yttrium.

Note: Figures quoted may not sum due to rounding.

2.4 MINING

Mining studies indicate the suitability of a large, open pit mine at Kvanefjeld. The mine will have a low waste strip ratio and, as the highest grades present near-surface, will generate higher grade run of mine ore in the early years of production. The total life of mine production from the mine is 232.6 Mt at an average mine grade of 341 ppm U_3O_8 and 1.22% TREO.

Coffey Mining Pty Ltd (Coffey) was retained by the Company to carry out the mining study, based on a mill throughput of 7.2Mtpa, as part of an engineering update to the pre-feasibility study that the Company undertook on the Project in 2011.

Coffey completed a similar exercise for the Company on a mill throughput of 10.8Mtpa in 2009. The 2009 study formed the basis of the current mine study, however, data was updated to reflect current costs, market conditions and a reduced mill throughput of 7.2Mtpa.

The scope of work that formed the basis of the mine study comprised the following tasks:-

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

With a crusher feed target of 7.2Mtpa and an average waste to ore strip ratio of 1.1:1, the average total material movement from the mine is 14.5Mtpa.

The mining study is based on owner mining, with the mining fleet being leased. It is assumed that the maintenance of all mobile equipment will be carried out by the original equipment manufacturer (OEM) as part of their supply and maintain contract.

2.4.1 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. Therefore this configuration has been selected as the base case for the mining study.

Equipment selection has been based on the 2009 mining study which considered a 10.8Mtpa process plant throughput rate. The 2009 mining study indicated that the mining fleet would most likely consist of 120t and 200t capacity hydraulic excavators and 50t to 100t capacity off-highway dump trucks together with standard open-cut drilling and auxiliary equipment.

2.4.2 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, 103 - 116 operators, 27 - 28 maintenance and service personnel with blast and mine service crew estimated at 6. This is a total of 184 - 198 employees.

2.4.3 Pit Optimisation

The Whittle Four-X optimisation software was used for pit optimisation purposes, utilising the May 2011 SRK resource model.

Based on the Indicated Resources only, and selecting the shell that produced the maximum undiscounted cash flow, the optimum pit shell contained some 238 Mt of ore at over 300 ppm U_3O_8.

Some 251 Mt of waste are contained within the pit shell, giving a strip ratio of 1.1:1.

2.4.4 Mine Design

The final pit design was based on the shell as described above. Table 2.4.1 provides a summary of the material breakdown as contained within the pit.

Table 2.4.1 Summary Materials Breakdown by Pit Stage

Pit Stage	Total Material	Waste	Strip Ratio	Mill Feed				
				Tonnes	Grade			
					U_3O_8	REE	Zn	TREO
	(Mt)	(Mt)	(w:o)	(Mt)	ppm			
Stage 1	9.1	3.6	0.7	5.5	356	10,814	2,550	11,837
Stage 2	38.7	16.6	0.8	22.1	391	12,297	2,273	13,400
Stage 3	17.6	4.6	0.4	12.9	348	12,083	2,145	13,239
Stage 4	82.4	24.4	0.4	58.1	368	11,694	2,496	12,777
Stage 5	166.8	84.3	1.0	82.5	333	10,986	2,295	12,011
Stage 6	165.9	114.4	2.2	51.5	298	10,163	2,235	11,048
Total	**480.5**	**247.9**	**1.1**	**232.6**	**341**	**11,162**	**2,328**	**12,185**

As previously discussed in section 2.3.3 Ore Mineralogy, in the upper, higher grade portions of Kvanefjeld (>300 ppm U_3O_8) the phosphate bearing minerals (e.g. steenstrupine) are the dominant hosts to REEs and uranium. The mine development is planned in six main stages, with the majority of ore scheduled from the greater than 300 ppm U_3O_8 resource material that dominates the upper level of the Kvanefjeld deposit.

Figure 2.4.1 shows a long section through the Kvanefjeld resource model, with drill strings coloured by REO grade. The model generally follows the lujavrite contact. The northern half features zones of black lujavrite over 200 m thick that outcrop at surface. To the south, the lujavrite forms a series of thinner lenses. Highest REO, uranium and zinc grades occur together in the upper parts of the deposit. Grades begin to decrease below 200 m.

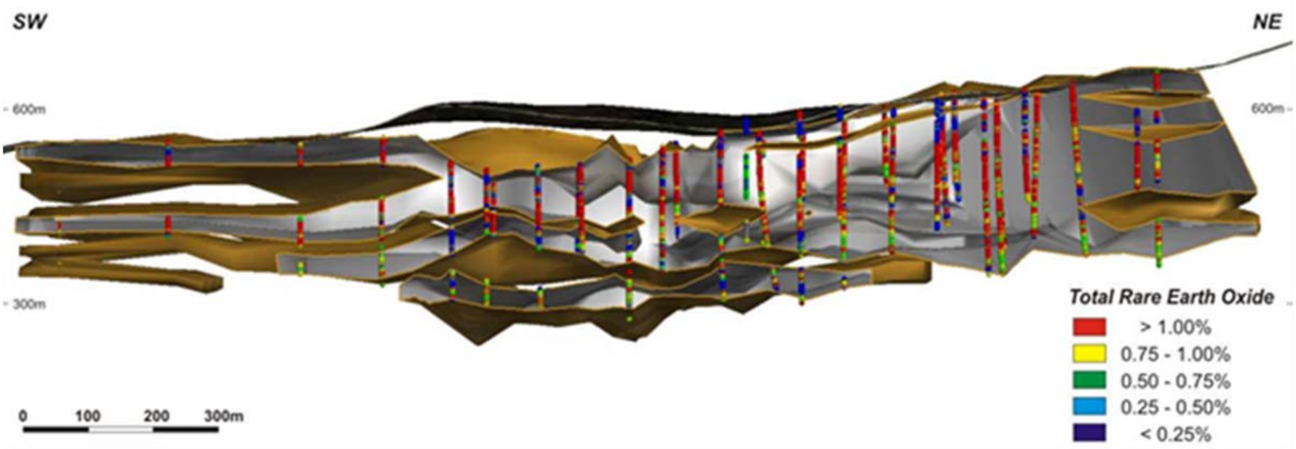

Figure 2.4.1 Long Section through Kvanefjeld Resource Model

The fact that Kvanefjeld is essentially a plateau, with the orebody outcropping at surface and the highest grade material occurring in the upper zones, means that the waste material moved per tonne of ore (strip ratio) is extremely low. The strip ratio is only 0.7 tonne waste per 1 tonne ore over the first 25 years of mine life, and as a consequence the mining costs are very economic.

An overview of the final pit and waste dump at the end of year 33 is presented in Figure 2.4.1 below.

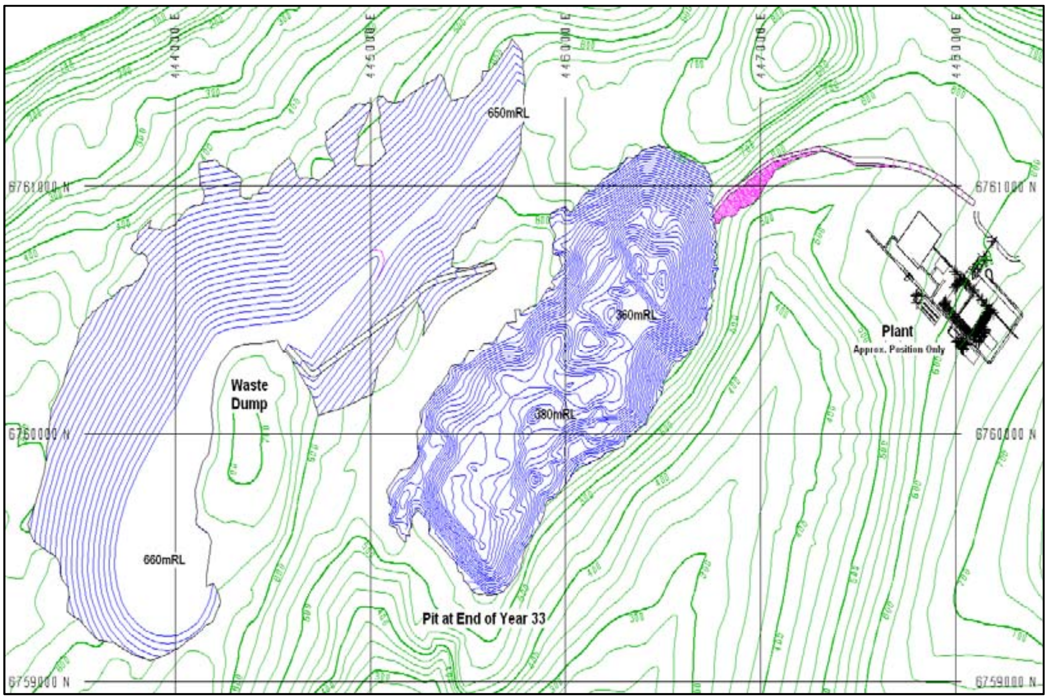

Figure 2.4.2 Overview – Final Pit and Waste Dump – Year 33

2.5 METALLURGY

2.5.1 Introduction

Metallurgical testwork and flow sheet development for the Kvanefjeld resource has been undertaken in two distinct stages, over several decades.

A rigorous program of metallurgical development for the project was undertaken by Risø laboratory in Denmark from the mid-1960s through to the mid-1980s. Risø's development programme included pilot plant testwork and was focused on the recovery of uranium only.

Since 2008, the Company has conducted an extensive metallurgical development testwork program, focused on the recovery of both uranium and REEs. This work has been managed directly by an in-house metallurgical team. A priority has been placed on taking a mineral driven approach. This commenced by establishing a comprehensive understanding of the minerals that make up the lujavrite-hosted resources. A focus was then placed on isolating the economic minerals from the uneconomic gangue minerals to create REE-uranium rich mineral concentrates. Once achieved, the optimal method to leach both REEs and uranium was identified.

2.5.2 Beneficiation Laboratory Testwork

As discussed in section 2.3.3 Mineralogy, REEs and uranium are hosted in both phosphate and non-phosphate bearing minerals. Through the upper levels of the Kvanefjeld deposit as delineated by material approximately >300 ppm U_3O_8, phosphate-bearing minerals are the dominant hosts to REEs and uranium. These include steenstrupine, vitusite and britholite. In lower grade material, which occurs deeper, zirconium silicates become increasingly important hosts to uranium and heavy REEs, and monazite increasingly hosts light REEs. Zinc is hosted throughout the deposit by sphalerite.

REE and uranium bearing phosphate minerals can be concentrated into ten percent (10%) of the original mass with high recoveries. This enriched, so called, "REP" (Rare Earth Phosphate) concentrate has been shown to contain greater than 10% REO and 2000 ppm U_3O_8. The typical recoveries of the zinc and REP rougher flotation stage are presented in Table 2.5.1.

Table 2.5.1 Summary of Flotation rougher performance

Ore sample	Zinc sulphide rougher scavenger		Rare earth phosphate rougher scavenger				
	Recovery		Recovery				
	Mass (%)	Zinc (%)	Mass (%)	Ce (%)	P (%)	Y (%)	U (%)
Ore Domain A	1	80	21	89	88	78	66
Risø composite*	1	80	25	89	89	77	67

*sampled from remaining stockpiles of ore recovered by Risø from the mine adit

Residual uranium and heavy REEs are hosted in Na-Zr silicate minerals of the lovozerite group that are not collected during flotation. The low mass, high grade nature of the REP concentrate clearly justifies pursuit as the primary focus of further downstream process development. Within the framework of the study further evaluation of the lovozerite group minerals has not been pursued.

The Company plans to conduct further work into recovering the residual uranium-REE minerals, as a later stage in the projects development. This will provide a means to increase both uranium and heavy REE output.

Locked cycle and pilot plant testwork has confirmed the performance of the batch tests and provides a high level of confidence in the process. A simple flotation circuit is envisaged, as shown in Figure 2.5.1.

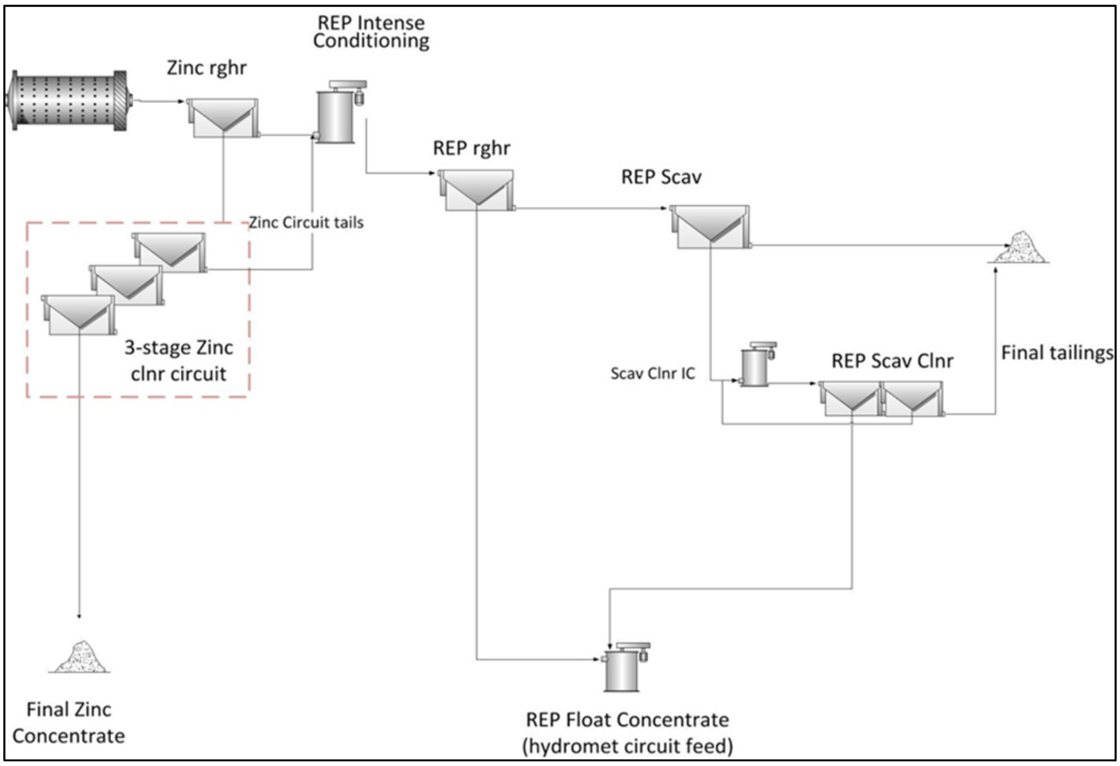

Figure 2.5.1 Zinc sulphide and Rare Earth Phosphate (REP) flotation circuit

2.5.3 Hydrometallurgical Testwork

Laboratory testwork has demonstrated that atmospheric acid leaching of the REP concentrate is a cost effective and efficient way of extracting uranium and REEs from the concentrate. Atmospheric leaching at elevated temperature (up to 95°C) can leach the uranium and REEs with acceptable acid consumption levels and a low level of impurities.

Acid baking (wet and dry concentrate up to 330°C) and high pressure acid leaching (up to 260°C) were also investigated as alternative methods for extraction. Acid baking was successful in achieving a higher light rare earth extraction than atmospheric acid leach, although uranium and HREE was slightly lower. High pressure acid leaching provided no extraction benefit over atmospheric leaching.

A counter current leaching stage is envisaged for the processing plant design, which consists of both strong and weak acid leaching stages, similar to those successfully commercialised in the zinc industry.

Leach results show that, across a range of ore types, atmospheric leaching produces high recoveries of uranium and HREEs. The leach extractions for LREEs are slightly lower but consistent with industry standards.

2.5.4 Flowsheet Selection

Following initial exploratory testwork programs in the first half of 2011 it was possible to identify metallurgical processes which showed the greatest promise. As part of selecting the metallurgical flowsheet it was necessary to focus the development efforts on a list of seven flowsheet contenders. The contending flowsheets considered for final evaluation were:

Base case	"Whole of Ore" Carbonate Pressure Leaching (CPL) followed by flotation and re-leach
Flowsheet 1	The "Whole of Ore" base case flowsheet optimised
Flowsheet 2	Flotation followed by sodium carbonate pressure leaching
Flowsheet 3	Flotation followed by potassium carbonate pressure leaching
Flowsheet 4	Flotation followed by atmospheric leaching in hydrochloric acid
Flowsheet 5	Flotation followed by atmospheric leaching in sulphuric acid
Flowsheet 6	Flotation followed by acid bake and ion exchange

Testwork programs were conducted to determine the metallurgical performance of each flowsheet at laboratory bench scale. The results of the testwork programs were then used to develop a process design for each option.

From the process design, capital and operating costs were estimated for the purpose of ranking the alternatives in terms of a financial metric. A risk analysis was completed for each flowsheet. The risk analysis was combined with the financial metric to develop an understanding of the upside and downside for each flowsheet.

Flowsheet 5, flotation followed by atmospheric leaching in sulphuric acid, was shown to have the lowest risk. It also provides a number of advantages which include:

- Simple processing equipment allowing the flowsheet to be developed at reasonable cost and within an acceptable timeframe;
- Having the lowest exposure to energy costs and carbon emissions as significant energy is generated from sulphuric acid production;
- Uranium is recovered in an industry conventional manner using solvent extraction;
- Higher HREE extraction than LREE, enhancing recovery of the more valuable REE components;
- Low capital cost due to downsizing of the hydrometallurgical plant;
- Having the fewest health and safety issues; and
- Having the lowest risk of radionuclide contamination of RE products.

The successful application of phosphate flotation as a means of beneficiating the ore and generating a low mass, high grade mineral concentrate has resulted in the vast majority of acid consuming minerals being rejected to the flotation residues. This in turn has provided the opportunity for the application of a simple, atmospheric acid leach process to economically extract the uranium and REEs from the concentrate. The uranium can then be recovered via conventional solvent extraction techniques and the REEs can be subsequently separated into three distinct, high quality, radionuclide free products. This is one of the key technical and economic advantages of the Project.

As previously discussed REEs and uranium are present in two major mineral groups, rare earth phosphates and sodium zirconium silicates. Initial beneficiation flowsheets considered sodium zirconium silicate mineral recovery as the third step in sequential flotation, following on from the zinc sulphides and phosphate minerals . At present most of the sodium zirconium silicates do not float with the phosphate containing minerals and therefore report to the flotation tailings.

An extensive literature search and collaboration with three reagent suppliers has identified a range of potential reagents for sighter testwork. The testwork program is ongoing. Future testwork will focus on detailed mineralogy of the minerals, further reagent screening tests, and alternative unit processes.

In addition, alkaline leaching tests on flotation tailings are showing promising results. Atmospheric carbonate leaching has indicated that nearly half of the uranium and heavy rare earths present can be leached from the tailings. Further follow up work is planned to increase the efficiency of this extraction as an alternative metallurgical treatment to concentration.

2.6 PROCESS PLANT

A preliminary design of the processing plant has been completed for the recovery of uranium oxide, ZnS concentrate and three separate REO products. The selected flowsheet comprises;

- an initial flotation circuit to concentrate zinc and rare earth phosphate (REP) minerals:
- atmospheric leaching of the REP concentrate with sulphuric acid in stirred tanks:
- solvent extraction of uranium from the sulphate leach solution;
- various stages of impurity removal:
- precipitation of REEs from the sulphate solution; and
- processing of leach residues to produce a range of REO products.

The leach residues are treated to recover the bulk of the REEs into a chloride solution. Following impurity removal, the HREEs are separated from the LREEs (and other impurities) by solvent extraction. Uranium and thorium are also removed from the REE products at this stage.

The solvent extraction step produces a high purity heavy rare earth hydroxide product. A light rare earth carbonate product is then precipitated from the chloride solution following an aluminium removal step. Finally a mixed rare earth carbonate product is produced.

A simplified block-flow diagram for the flowsheet is provided in Figure 2.6.1.

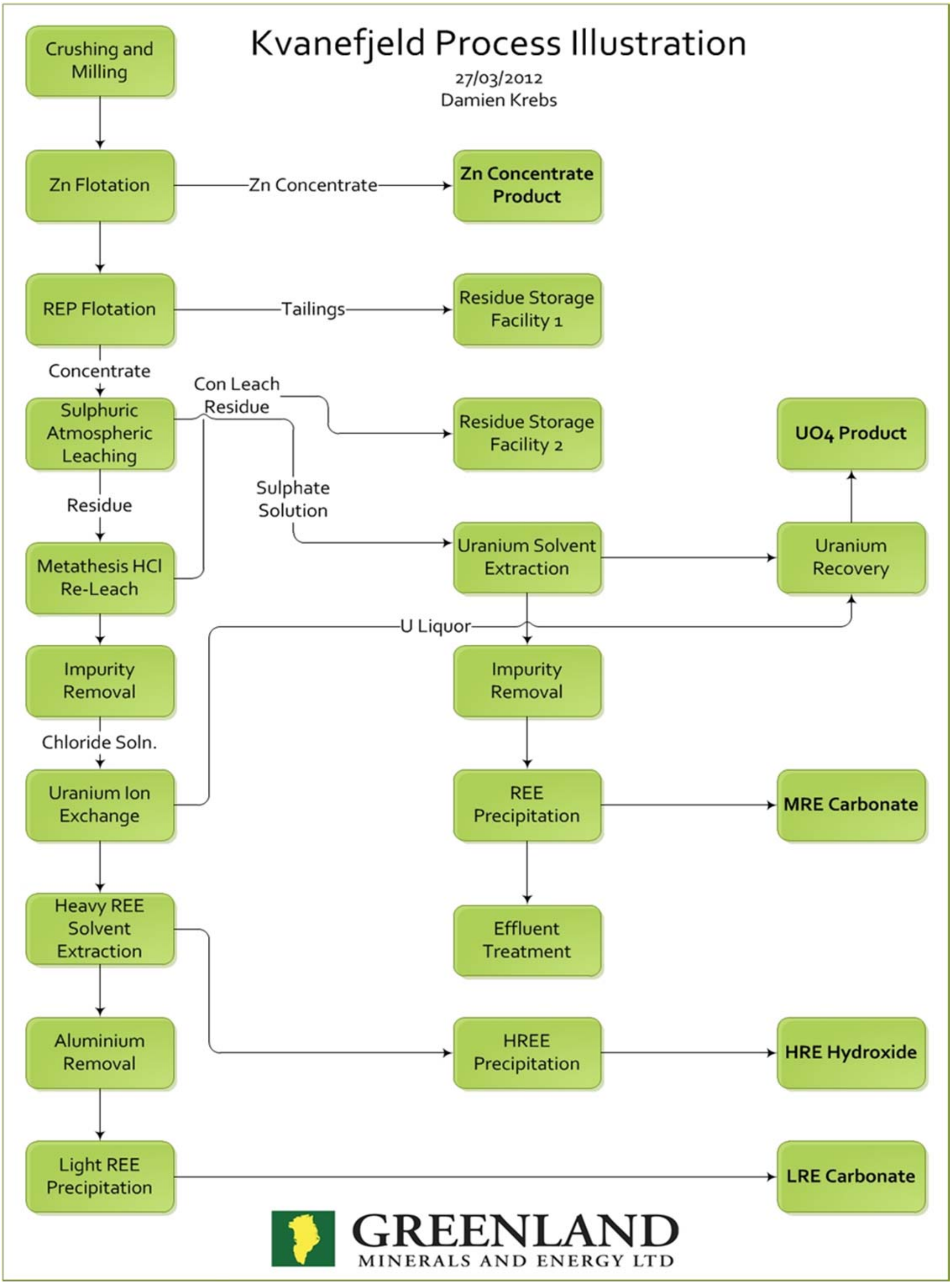

Figure 2.6.1 Block-flow Diagram

The process plant has been designed to a Pre-Feasibility level including a detailed mass and energy balance constructed using IDEAS software. The process model was used to determine recoveries and production rates, and to provide the basis for estimating operating and capital costs.

Table 2.6.1 highlights the major design inputs for the process plant based on the average over the first six (6) years of operation. The processing plant has been designed for a throughput of 7.2Mtpa of ore, based on the design capacity used in the 2011 mining and engineering studies performed by Coffey Mining and AMEC Minproc respectively.

Table 2.6.1 Plant Design Parameters

Plant Throughput	t/a Ore	7,200,000
ROM Ore U_3O_8 Grade	ppm	364
Uranium Production	t U_3O_8/a	1,185
ROM TREO Grade	% REO	1.27
Total REE Plant Production	t TREO/a	40,800
Light REE Production	t LREO/a	35,200
Heavy REE Production	t HREO/a	5,600

2.6.1 The Concentrator

2.6.1.1 Comminution

A conventional crushing and milling circuit, similar to that widely used throughout the mineral processing industry, has been chosen. Run of mine ore is tipped directly into a gyratory crusher. The crushed ore is then stockpiled, reclaimed and fed to a SAG mill where the ore is ground. The SAG mill operates in closed circuit with a ball mill in order to achieve a slurry product with grind size of 80% passing 75 um.

2.6.1.2 Beneficiation

Zinc is removed from the ore slurry via froth flotation to produce a high grade concentrate for sale. The tails from the zinc flotation circuit are then thickened before undergoing high intensity conditioning. During high intensity conditioning flotation reagents are added to cause the phosphatic minerals to become selectively hydrophobic.

Rare Earth Phosphate flotation occurs within tank cells with forced air addition to control froth rate. The first stage consists of short residence time rougher flotation with the concentrate produced reporting to the final concentrate thickener. The grade of this concentrate is high enough that cleaning of the concentrate is not warranted. The tails from the rougher flotation stage proceed to the scavenger flotation in which flotation occurs for an additional 10 minutes. The scavenger concentrate is not of suitable grade to be considered final concentrate. Therefore cleaning of the concentrate is required.

The flotation tailings are thickened in a high rate thickener with the overflow water recovered for use as process water. The thickened tailing slurry is pumped to the Residue Storage Facility 1 (RSF1).

As well as providing storage for flotation tailings, RSF1 decant water is recycled for use in the Concentrator.

2.6.2 The Refinery

2.6.2.1 Acid Leach

The REP concentrate slurry is pumped via an overland slurry pipeline to the Refinery Site, located to the east of the Concentrator Plant Site.

REP concentrate is atmospherically leached in a counter current leaching circuit. The counter current leaching circuit offers the most efficient use of acid. Fresh REP concentrate is initially leached in weak acid, at 90°C. The discharge is then diverted to a thickener which separates the liquor from the solids. The liquor is forwarded to the uranium recovery circuit while the solids are recycled to the strong acid leach circuit.

The strong acid leach stage operates at 95°C and 32% solids with an extended residence time to control silica precipitation. The leached slurry is thickened and filtered.

During the sulphuric acid leach, uranium and rare earths are initially leached from the concentrate into solution. Nearly all of the light rare earths precipitate from solution as sodium-rare earth-double sulphates. Most of the heavy rare earths also precipitate from solution as sodium-rare earth-double sulphates. Essentially the rare earth elements report to the solids residue while the uranium reports to the liquor. This separates the rare earth elements from most of the contaminants. A modest amount of the uranium and other contaminants such as aluminium also form stable sulphate precipitates in the leach therefore reporting to the leach residue.

2.6.2.2 Uranium Recovery

The leach solution is sent to the uranium extraction stage. Here solvent extraction is used to selectively extract the uranium from the aqueous liquor into an organic phase. The organic and aqueous phases are intimately mixed and then allowed to separate in a settling chamber. The separated organic from the extraction stage is called the loaded organic, while the aqueous phase, which contains the rare earths and impurities, is called the raffinate. The raffinate is forwarded to the impurities removal circuit while the loaded organic progresses to the uranium stripping stage.

Uranium solvent extraction strip solution is diverted into a series of continuously stirred reactor tanks. Here the uranium is recovered from solution as a sodium di-uranate (SDU) precipitate. Caustic is dosed into the reactors at ambient temperature to cause the uranium to precipitate from solution.

SDU repulped solids are then releached with sulphuric acid. The sulphuric acid re-leach solids are separated from the solution in a filter. The filtrate solution is then treated with hydrogen peroxide solution, which causes the uranium to precipitate from solution as uranium oxide ($UO_4.xH_2O$) solids. The uranium oxide product solids are filtered on a horizontal pressure filter. The solids are dried and stored in 200 L drums. The drums are sealed and prepared for export as the final uranium product.

2.6.2.3 REE Recovery From Acid Leach Solution

The raffinate from uranium solvent extraction contains impurities as well as REEs. The liquor is neutralised with limestone to precipitate the majority of the contained iron, thorium, aluminium and silica impurities. The precipitate is thickened, filtered, washed, repulped and pumped to the Effluent Treatment plant. Zinc is then selectively precipitated from the liquor as zinc sulphide using sodium sulphide solution. The contained REEs are precipitated from the purified liquor exiting zinc precipitation by raising the pH with dilute sodium carbonate solution. The REE precipitate is thickened, filtered, washed to produce a final product which contains a high proportion of heavy rare earth elements. This is one of two HREE products produced by the process with this product termed the Mixed Rare Earth (MRE) Carbonate Product.

2.6.2.4 REE Recovery From Acid Leach Residue

Filter cake from the acid leach residue filter is contacted with strong caustic solution to cause the conversion of sodium-REE-double sulphates into REE hydroxide. This then renders the REE hydroxides amenable to re-leaching with hydrochloric acid.

Solids from the metathesis stage are selectively re-leached using hydrochloric acid to minimise aluminium and iron re-dissolution. The re-leach solids are separated from the solution with the resultant REE rich chloride liquor advanced to the iron removal area. Here iron, thorium and some of the aluminium are removed from the chloride liquor via precipitation with lime.

Sodium sulphide is used to selectively remove base metal contaminants from the chloride liquor prior to uranium and rare earth element recovery. Ion exchange is used to remove uranium from the chloride liquor prior to the recovery of REEs. This recovers uranium to produce additional product and prevents contamination of rare earth products. The uranium recovered here combines with the uranium recovered in the solvent extraction stage, prior to SDU precipitation.

Solvent extraction is then used to preferentially recover the heavy rare earth elements from the chloride liquor to produce a separate high value product. The HREEs are recovered from the solvent extraction strip solution by precipitation with caustic. The HREE form a hydroxide precipitate, termed the Heavy Rare Earth (HRE) Hydroxide Product, which are thickened and then dewatered by filtration in a horizontal plant and frame pressure filter. Washed filter cake of 45% moisture is produced and discharged into bulka bags for export sales.

Light rare earths are then precipitated from the chloride liquor as carbonate after aluminium removal. The Light Rare Earth (LRE) Carbonate product is produced in this circuit.

2.6.2.5 Reagent Supply

Sulphuric acid will be generated onsite from raw sulphur to produce concentrated 98% sulphuric acid and steam for use in the hydrometallurgical plant. The capacity of the sulphuric acid plant is 1070 tonnes per day at 100% basis.

The process of HCl production commences with a chlor-alkali plant. A saturated sodium chloride solution is produced by dissolving salt in a recirculating chloride solution. This solution is decomposed in banks of electrolytic cells to produce chlorine gas, hydrogen gas and NaOH in the form of a 32% w/w solution. The NaOH production capacity of the chlor-alkali plant is 323 t/day (100% NaOH basis) and will consume approximately 25 MW (drawn power), which represents approximately 40% of the total electrical power demand.

2.6.2.6 Effluent Treatment

Waste streams generated within the hydrometallurgical plant are pumped to an effluent treatment plant for neutralisation, metal precipitation and clarification. The streams are mixed and reacted with hydrated lime slurry and barium chloride. Neutralised slurry is pumped to a thickener and solids are thickened prior to pumping to the residue storage facility (RSF 2), which is a double-lined and totally contained storage facility sited adjacent to the Refinery.

2.7 RESIDUE MANAGEMENT

2.7.1 Introduction

The concentration and treatment of ore through the processing plant will generate two residue streams which the Company intends to store for possible future re-processing or for rehabilitation after mine closure. The flotation residue stream from the Concentrator will be stored at Taseq (RSF1) as this has been identified by the Company as the optimum location to safely contain this over the life of mine. The smaller residue stream generated from the Refinery will be stored at a location east of the Nakalak range in a lined facility near to the Refinery and well away from the township of Narsaq. Both streams will be pumped to their respective RSF as a slurry. These two locations have been selected by the Company after numerous investigations, workshops and site visits.

The Company has engaged a number of consultants since 2009 to investigate options for residue storage for the Project. Coffey Mining completed a preliminary study in 2009 which focused primarily on Taseq as the main storage location. AMEC was then engaged to further develop the RSF concept in 2010 and 2011. AMEC Environment & Infrastructure subsequently contributed to a desk top scoping study, issued during July 2011, by identifying a series of potential site options.

In August 2011 a technical team visited Narsaq to view the environs of the Project site and assess potential RSF sites. A site visit report was subsequently issued which described the site conditions at the respective potential RSF sites. A SWOT analysis was then carried out on each of the RSF locations which identified the strengths, weaknesses, opportunities and threats of each option.

2.7.2 Residue Storage Facility Site Selection

The identification of potential sites for the Project’s two residue storage facilities (RSF1 and RSF 2), has been focused on the Company’s concession area, at sites adjacent to the Kvanefjeld mineral deposit and the proposed plant sites. In 2011 AMEC identified seven potential RSF sites which were subsequently assessed with respect to the following social, environmental and technical characteristics:

- Geotechnical factors
- Impact on the natural environment
- Impact on social environment/position of the local communities.
- Area requirements and topography
- Distance from a likely plant site location and accessibility

The location of the sites are summarised as follows:

- Site A: Taseq basin area
- Site B: South of the open pit, east of Narsaq town
- Site C: Central valley site, east of Nakalak range
- Site D: Natural basin east of Nakalak
- Site E: Valley site, west of Mt Naajarsuit
- Site F: Sahannguit Fjord, northwest of Ipiutaq town
- Site G: Valley site, east of Nakalak range.

Figure 2.7.1 shows the location of RSF1 and RSF2 and the alternative options that were assessed.

Figure 2.7.1 Residue Storage Facility Location Options

The assessment of these sites concluded that site option F, located to the south of the project area within the environs of the Sahannguit Fjord, immediately north of the Ipiutaq farm and, site option B, located within the Narsaq Valley south of the proposed open pit, were not considered to be acceptable options from either an environmental or engineering point of view. These were excluded from any further study.

A preliminary environmental analysis suggested that site options A and G exhibited minimal potential for environmental impact. However, a RSF located at site G would require a large embankment in order to store concentrator flotation residue for the mine life., Furthermore, given the distance from Kvanefjeld to site G, consideration was given to the substantial pumping and pipeline costs that would be required, hence this option was excluded from further study

Options C, D and E were also highlighted as being favourable with respect to the environment, however these three options would all be required to store the concentrator flotation residue for the life of mine, and all would require significant volumes of rockfill to construct the embankment to the height required.

Site option A, Taseq, being a natural, impermeable basin, requires the least volume of rockfill to form a suitable embankment, and has the capacity to contain the concentrator flotation residue for the life of mine. As the residues from the concentrator contain lower levels of radionuclides than that found naturally in the Kvanefjeld, and as the ore has not been chemically processed at this stage it is considered suitable for storing in Taseq. The residue will be covered by water hence radon emissions will be safely managed. Consequently Taseq was identified as the most favourable option for the permanent storage of concentrator flotation residues.

2.7.3 RSF Residue Storage Facility Design

The preferred residue storage concept involves the development of two separate RSF's. The residues will be sub-aqueously discharged into respective RSF's via a series of open ended discharge points around the perimeter in order to ensure 100% water cover at all times. Sub-aqueous deposition has two key advantages. It mitigates radon gas release and eliminates dust generation.

2.7.3.1 Residue Storage Facility 1

The primary RSF (RSF1) will be Taseq and will be designed to store residue material generated from the upfront concentration step. Taseq can adequately store the entire volume of the concentrate residue over the 33 mine life. It is also located within 3km of the concentrator and at a similar elevation permitting the residue to be pumped as slurry. Preliminary analysis also suggests that Taseq pre-deposition works and final embankment works are significantly less than other sites.

The second RSF (RSF2) will be a smaller facility and will store the residues generated from the Refinery. This will be an impermeable double lined RSF due to the nature of the residues that will be stored. RSF2 will be collocated with the Refinery to minimize pumping requirements. RSF D, in the options study, has been selected as the most suitable location for this RSF due to its proximity to the proposed plant location and similar elevation.

The RSF1 embankment will be formed on the western "rim" of Taseq outlet. The confining embankment will be constructed using approved mine waste or natural rock quarried from adjacent slopes. Taseq water that may be displaced by residue will be reclaimed back to the concentrator process plant.

Figure 2.7.2 Location of RSF1

2.7.3.2 Residue Storage Facility 2

Option D, the natural basin east of the Nakalak range, has been chosen as the preferred location for RSF2. This location allows RSF2 to be located alongside the proposed processing plant/refinery and at a similar elevation. The refinery will produce a residue slurry with 22% w/w solids content suitable for pumping to RSF2. Figure 2.7.3 shows the expected area that RSF2 will cover after the 33yr mine life.

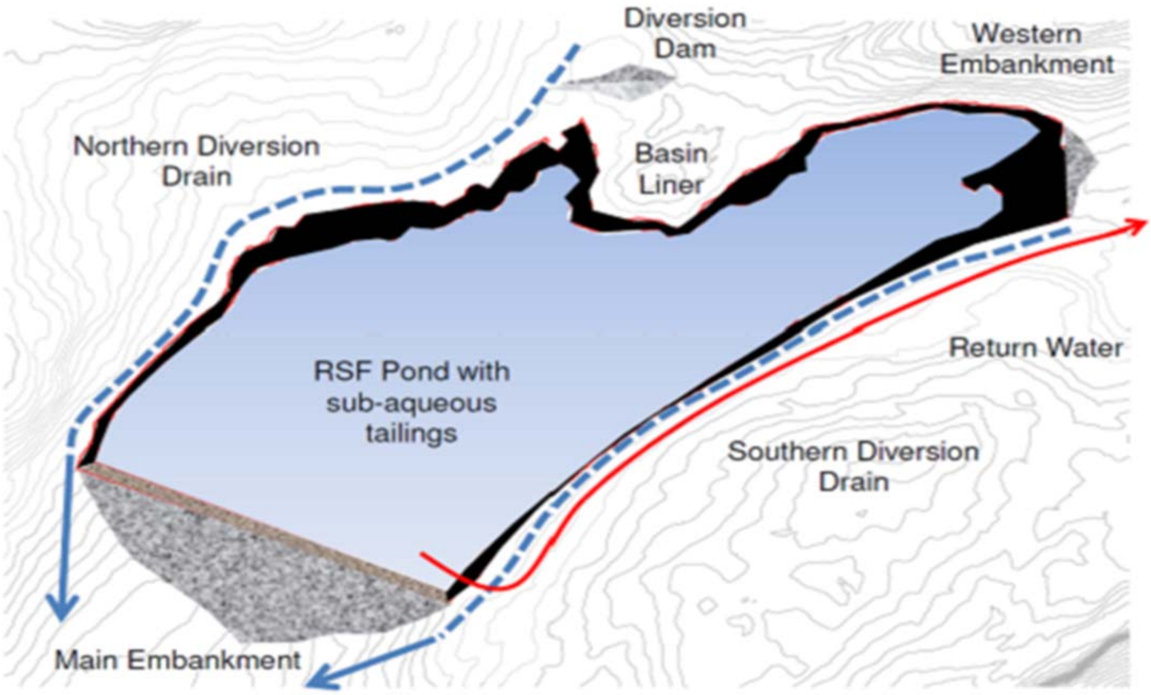

Figure 2.7.3 RSF2 Final Capacity

Due to the nature of the refinery residue, allowance has been made to fully line RSF2 so as to create an impermeable barrier. The initial embankment height for RSF2 will be in the order of 20m with subsequent lifts of between 1m and 5m. The final embankment height is expected to be in the order of 62m.

It is important to note that the residues generated from both the Concentrator and the Refinery still contain elements that may have a significant commercial value in the future. Both Residue Storage Facilities have been designed to safely store the residues and whilst long term closure plans have been provided for in the design and cost estimates there may be potential to recover the residues for further processing at a later stage. The options chosen for the location of RSF1 and RSF2, whilst preferred, are not the only options available for the Project and further investigations and design work is planned for in the next phase of studies.

2.8 ENVIRONMENTAL AND SOCIAL

2.8.1 Environmental and social impact assessments

The successful completion of an Environmental Impact Assessment (EIA) and a Social Impact Assessment (SIA) are necessary pre-requisites for an application for an Exploitation License in Greenland.

The Company commenced its EIA and SIA at the beginning of 2011. When completed, these assessments will be reviewed by the Government of Greenland through the office of the Bureau of Minerals and Petroleum (BMP). The BMP will be supported in its review by the Danish National Environmental Research Institute (NERI).

The Terms of Reference for both the EIA and SIA each were approved by the BMP in 2011. All scopes of work for studies forming part of the EIA or SIA are issued to BMP for approval prior to work commencing. Through this process the Company helps to ensure that work on the EIA and SIA is progressing to the satisfaction of the Greenlandic government.

The scoping phase of the both the EIA and SIA have been completed. The assessments are now primarily focussed on the collection of baseline data. The EIA and SIA will document the results of the baseline studies, the potential impacts of the Project and will identify mitigation and management measures to reduce or, where possible, eliminate the impact of the Project on the social and physical environment. A number of plans will also form part of the EIA and SIA. These will include a Benefit and Impact Plan, a Monitoring and Evaluation Plan and an Environmental Management Plan.

The Company is also conducting an extensive and thorough stakeholder engagement process. This process has been designed to ensure that all potential issues regarding the Project are identified at an early stage, thereby allowing for the issues to be effectively integrated into planning and impact assessments.

The Company has given several presentations to the local communities at town hall meetings since commencing its exploration and development studies. Two Community "Open Days" were held in 2010 and 2011. In addition, four stakeholder workshops were held during 2011 in Qaqortoq, Narsaq and Nuuk.

The key environmental management issues for the Project were identified through the scoping phase, including stakeholder engagement. These issues are summarised below:

- Discharges to water (surface waters, fjords and groundwater) including stormwater runoff from disturbed areas (such as the waste rock dump) and discharges from the open pits, process plant and the RSF;
- Alkaline drainage;
- Potential for contaminants to enter the food chain (e.g. fluorine, heavy metals and uranium);
- Conservation of biodiversity (terrestrial and aquatic), including the presence/absence of rare and/or threatened species;
- Atmospheric emissions such as radon gas, dust, combustion products and other gaseous emissions;
- Radiation from radioactive sources within the project area;
- General waste management;
- Tailings design, location and management to minimize environmental risks during operation, during decommissioning and after closure; and
- Rehabilitation of areas disturbed by the project.

As with the identification of environmental management issues, the key social management issues for the Project have been identified through the scoping phase and stakeholder consultation. These issues (both positive and negative) are summarised below:

- Alienation from land required for the Project components and its ancillary infrastructure;

- Impact on the amenity of Narsaq and surrounding settlements as a result of dust, noise and light emissions from the Project area;
- The impact of the Project on the water supply for Narsaq township and surrounding settlements;
- The impact of the Project on subsistence, artisanal and commercial fishing and hunting (including fish spawning and nursery areas and seal pupping areas);
- The impact of the Project on cultural heritage and archaeological sites, including sacred and spiritual places, traditional fishing or hunting campsites, traditional trails and burial grounds;
- The impact of the Project on transportation infrastructure, incremental traffic flows (air, land and sea) and transportation risks;
- The impact of the Project of the project on local social 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 Greenland economic sectors that drive economic growth; and
- Improvements in the nation's balance of trade, infrastructure development.

These issues have all been identified from an early stage. As a result studies are well underway to better understand and plan for mitigation and management.

2.8.2 Baseline studies

Risø conducted environmental baseline studies of the local area in the 1970's and 1980's. A Preliminary Environmental Impact Statement was issued in 1990.

The Company has been undertaking annual environmental baseline studies since the Project was acquired in 2007. The scope of these studies has included:

- Biological sampling of soil, water, and sediment from lakes, marine and terrestrial locations;
- Archaeology surveys;
- Hydrological monitoring;
- Monitoring of climate and air quality, including dust;
- Radiation sampling;
- Geochemical characterisation of waste rock and tailings;
- Air emissions modelling;
- Noise;
- Hydrocarbon spills;
- Local land use;
- Drinking water; and
- Taseq risk assessment.

A number of social impact studies are also currently underway. These include studies into:

- Traditional living conditions in South Greenland;
- Local Land Use;
- The potential impact of the Project on health outcomes; and
- Opportunities created by the Project and the need for planned coordination of infrastructure development.

The environmental and social issues identified for the Project will be managed in an appropriate manner in conjunction with stakeholder consultation to minimise and avoid adverse impacts to the land and local communities. The Company is committed to operating to the highest levels of environmental standards at all stages of the exploration, development, mining and rehabilitation processes.

2.9 INFRASTRUCTURE

This study presents a new concept for the location of processing facilities and infrastructure for the Project. One of the primary drivers for a revised plant site location was the potential impact that operations may have on the local township of Narsaq and the surrounding environment. To minimise potential negative impacts on Narsaq the Company considered that providing adequate buffer between the Refinery component of the processing plant and the town itself was fundamental to developing a long term, sustainable project. The development of a concentration stage in the processing of the ore to produce a concentrate facilitated this philosophy such that the Refinery could be decoupled from the concentrator. This concept has now been realised with the Mine and Concentrator located at Kvanefjeld and the Refinery located east of the Nakalak range. The project will be serviced by a new port located at Ipiutaq.

These changes have come about through numerous site location studies undertaken by the Company. In the interim PFS ("IPFS") in 2010, all of the processing facilities were to be located near the mine site at the top of the Narsaq valley. Subsequently, workshops and site visits highlighted the risks associated with locating the processing facilities in the Narsaq Valley. A key risk was the significant social impact on the town of Narsaq which would be transformed into a mining town. In addition, the prevailing wind patterns within the Narsaq valley may leave the town of Narsaq susceptible to noise and potential emissions from the plant.

In order to mitigate these issues the Company sought alternative locations for the Refinery east of the Nakalaq range. This general location would allow for a much larger buffer zone between the Refinery and the town of Narsaq thereby minimizing any negative social and environmental impacts on the town.

A number of alternatives have been examined and a preferred location for the Refinery has been identified east of the Nakalaq range approximately 18km to the east of Narsaq.

The proposed project site layout for the Study is shown in Figure 2.9.1

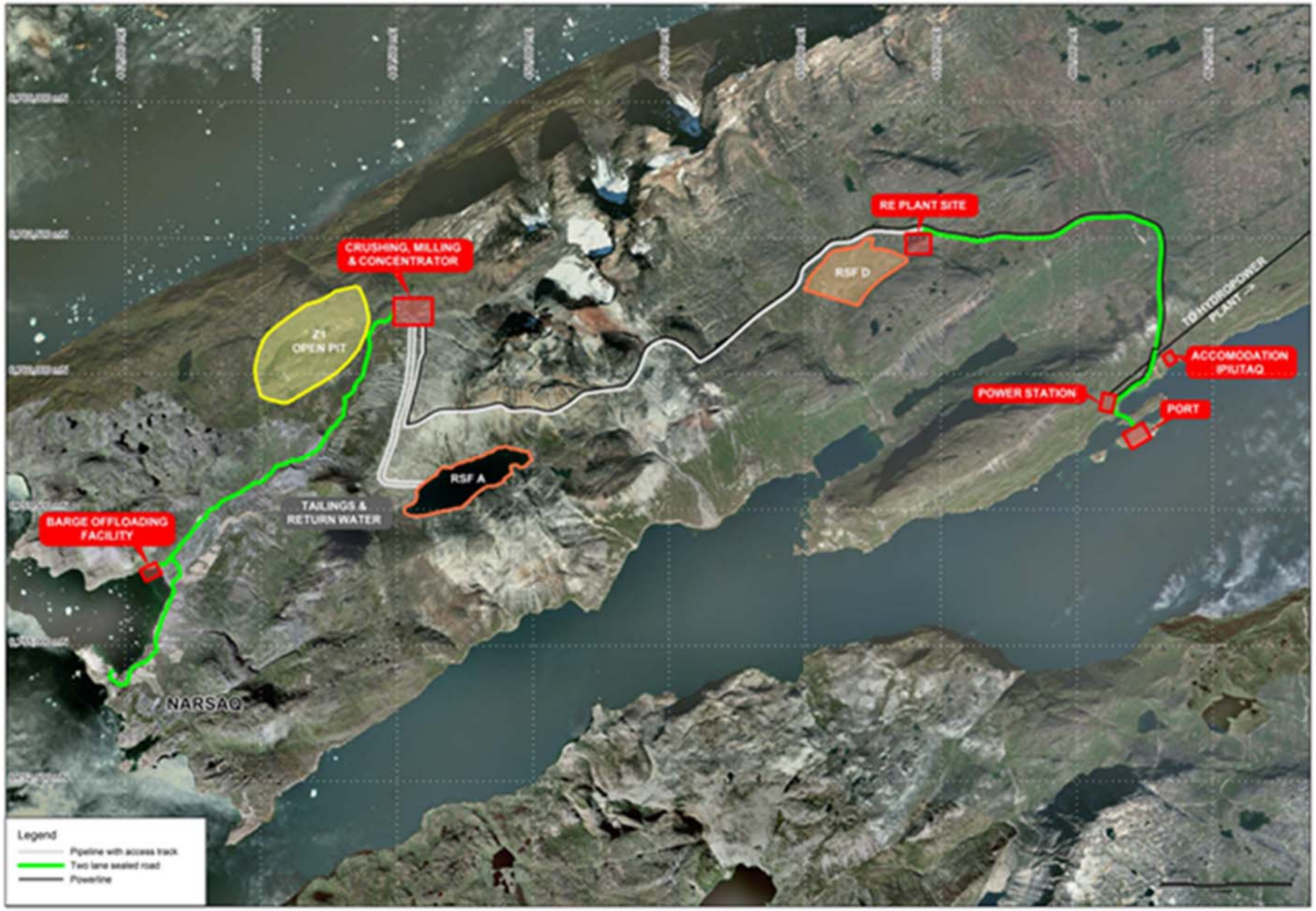

Figure 2.9.1 Proposed Project Layout

The new site layout breaks the project into three main sections connected by process infrastructure and services.

The first section, the crushing, milling and flotation circuit (the Concentrator), will be co-located at the Kvanefjeld mine site at the top of the Narsaq Valley. Concentrate will be pumped via an insulated and traced pipeline from the Concentrator, along the northern shore of Taseq and the southern edge of the Nakalaq range to the Refinery. A four wheel drive maintenance track will run alongside the concentrate pipeline to facilitate pipeline maintenance access.

The mine and concentrator will be accessed by a two lane, sealed road from the town of Narsaq and will be serviced from a barge facility located in Ilua Bay. Barges will be loaded with spares, reagents and consumables at the main port which is to be located at Ipiutaq.

The second section, the Refinery is located east of the Nakalaq range 500m above sea level and will be accessed by a two lane, sealed road from Ipiutaq.

The third section is located at the Ipiutaq where a new port will be constructed together with a HFO power station and an accommodation village for fly in – fly out (FIFO) employees. All of these facilities will be built and operated by third party suppliers.

Power will be supplied to the mine, concentrator and refinery via 33kV overhead power lines from a hydropower station and the supplementary HFO power station at Ipiutaq. The power plants and power distribution network will be built and operated by a third party power supplier.

The capital impact of relocating processing facilities and infrastructure away from the Narsaq Valley will increase the project capital expenditure by approximately US$60M. The Company considers this investment necessary to reduce the potential for adverse impacts on the township and community of Narsaq.

2.9.1 Water Supply

As discussed, the Mine and Concentrator, the Refinery, the Port, Power Station and Village are spread over three locations with raw water required at each location. Due to the distance between the locations and the freezing temperatures experienced in the area, it has been decided to source water locally at each of the sites.

A Hydrology Status Report, issued by the Company's environmental consultants Orbicon, presents preliminary hydrological information collected in the vicinity of each location. The report indicates that there is ample water available at each location. A detailed study of annual flow variations is required to quantify the water balance on a seasonal basis. Seasonal fluctuations in river flows will require raw water storage facilities at each location. This aspect of the project will be further developed in the next phase of engineering design.

2.9.2 Third Party Facilities and infrastructure

The ownership and operation of certain facilities and infrastructure are not consistent with the Company's core business. As a result the Company has made the decision to use third party suppliers to provide power, to own and operate the port at Ipiutaq and to own and operate the accommodation village for employees on FIFO rosters.

2.9.2.1 Accommodation Village

Most of the personnel to staff the management, operations and maintenance functions of the Project will be accommodated in a custom built village to be located near the proposed port at Ipiutaq. The scope of this new village is comprehensively described in an Accommodation report prepared for the Company by NIRAS.

The cost of the accommodation village was estimated by NIRAS and subsequently updated by the Company's technical team to meet the requirements of the Study. This includes allowance for 2000 workers at the peak of construction activities. The operational phase will see 118 rooms for management staff and 640 rooms for operators and technicians.

The accommodation village will be built and operated by a third party and is not included in the capital estimate. Payment to a third party supplier has been included as an on-going operating expense.

2.9.2.2 Harbour Facilities

It is proposed to locate the new harbour facilities for the Project at Ipiutaq. The 2011 NIRAS report contains the scope and cost for the new harbour, including provision for bulk liquids storage, a covered storage area and container yard. A smaller barge facility is also planned for the northern shore of Ilua Bay to service the mine and concentrator facilities.

The harbour at Ipiutaq and barge facility at Ilua Bays will be built and operated by a third party provider and is not included in the capital estimate. Payment to a third party supplier has been included as an on-going operating expense. The third party provider will also make provision for a tug boat, barges and two ferries. Power and potable water will be provided at the fence line of the harbour facilities to be reticulated within those facilities by the third party provider.

2.9.2.3 Power Supply

The project overall energy requirements have been determined to be in the order of 58MW. A new hydroelectric power station and a HFO power station will be established to meet this load.

The hydroelectric power station will be built at Johan Dahl Land, approximately 60 km East North-East of the Kvanefjeld mine site. The Company engaged NIRAS to prepare a study into the supply of hydroelectric power. NIRAS estimate that 45MW could be generated by the hydro-electric power plant however annual utilisation would be 77% based on the water available. This results in an available power of approximately 35MW. This would provide approximately 60% of the required 58MW needed for the project. This utilisation factor could be improved to 90% by tapping into the available water capacity in nearby Lake Hullet. Improving utilisation would increase the available power to approximately 40MW. The remaining 18MW will be provided by the HFO power plant.

The HFO power station will be built at site of the proposed port at Ipiutaq. A new 132kV overhead power line with access track will be built from the hydroelectric plant to the HFO power plant. A new 33kV overhead power line will then be reticulated from the HFO power station to the processing plant/refinery, concentrator and mine. There will be a maintenance track along all of the 33kV line.

The power stations and distribution network will be built and operated by a third party provider and is not included in the capital estimate. Payment to a third party supplier has been included as an on-going operating expense.

As previously highlighted the Company has conducted an extensive and thorough stakeholder engagement process as part of establishing the Terms of Reference for the EIA and SIA. Inclusive to this process was the establishment of a “Project Brief” which described in general terms the Project’s infrastructure requirements. The process was designed to ensure that all potential issues regarding the Project are identified at an early stage, thereby allowing for the issues to be effectively integrated into planning and impact assessments.

As a result of the new options developed for the location of infrastructure required for the Project the Company will being updating the “Project Brief” and will be holding further key stakeholder engagement meetings and presentations to the local communities in 2012.

2.10 CAPITAL COST

2.10.1 Capital Cost Summary

The detail of the project capital cost estimate, inclusive of mine infrastructure, process plant, residue storage facilities and area/regional infrastructure is set out in Table 2.10.1. The estimate is based on a mine and plant capacity of 7.2Mtpa.

The capital cost estimate is current as of the first quarter 2012, and is presented in US$.

The capital cost presented here is exclusive of:

- the cost of the mining fleet, which will be leased and is therefore covered under the Project operating costs;
- the cost of the power supply, harbour and accommodation village infrastructure, which will be supplied and operated by third party operators via Build Own Operate (Transfer) “BOO(T)” contracts, and is covered under the Project operating costs; and
- owner’s costs which are detailed in the Financial Evaluation.

Table 2.10.1 Capital Cost Estimate: Mine and Processing Plant		
Area No	**Area Title**	**US$M**
Direct Costs	Mine prestrip, infrastructure and equipment	15.1
	Grinding and Flotation Plant (Concentrator)	264.2
	Hydrometallurgical Process Plant (Refinery)	534.2
	Subtotal Process Plant	**813.4**
	Infrastructure – Plant	121.5
	Infrastructure – Area + Regional	36.7
	Subtotal – Infrastructure	**158.2**
	First Fill Reagents and Consumables	46.6
	Ocean Freight	22.8
	First Fill Spares	22.6
	Mobilisation and Demobilisation	28.9
	Vendor's Representatives	4.0
	Commissioning Assistance	14.7
	Subtotal – Miscellaneous	**139.7**
	Total Direct Cost	**1,111.2**
Indirect Costs	Temporary Facilities	11.4
	Engineering, Procurement and Construction Management	164.7
	Total Indirect Costs	**176.1**
	Total Project Costs (Net)	**1,287.3**
	Contingency (Growth Allowance)	247.2
	Total Project Costs (Overall)	**1,534.6**

2.10.2 Sustaining Capital

Sustaining capital expenditures for increases in the capacity of residue storage facilities will be required over the life of mine. These costs are summarised under Residue Management.

Additional and replacement mining fleet capital will be required for the Project. The timing of the replacement of all capital items was matched to the useful life of the equipment concerned, as recommended by the relevant equipment suppliers. As with the initial mining fleet, any additional mining fleet is leased and renewed on a scheduled basis, and is included in the mining costs.

Sustaining capital costs for plant and infrastructure have been incorporated into the plant maintenance cost estimates.

2.10.3 Estimate Structure

2.10.3.1 Direct Costs

Direct costs are those expenditures that include supply of equipment and materials, the cost of freight to site and construction labour. These are the costs to build the project and exclude other costs described below.

2.10.3.2 Indirect Costs

Indirect costs are those expenditures including the establishment of construction facilities, engineering, procurement and construction management (EPCM) services, and supervision of commissioning of the plant and owners costs.

2.10.3.3 Contingencies (Growth Allowance)

Growth allowances have been assigned to each element of the capital expenditure forecast on the basis of the level of uncertainty associated with the estimate for that element, having regard to the fact that estimates have been completed to a preliminary feasibility level.

Contingency provisions make allowance for the following risks:

- Minimal design input suitable for estimates of this accuracy;
- Preliminary scope definition;
- Quantity survey errors and omissions;
- Rework;
- Gross vs. net quantities;
- Material and labour rate accuracy;
- Equipment budget costing; and
- Incorrect “bulks” factor application.

An accuracy provision/contingency of approximately 20% has been calculated and is considered to be appropriate for this level of study.

2.10.4 Estimate Cost Basis

The capital cost estimates for the selected process flowsheet for the Project has been developed from:

- Studies commissioned by the company and performed by independent third parties;
- Budget quotations; and
- In-house cost information.

The following third party independent studies were utilised:

- AMEC Minproc’s Updated 7.2 Mt/a Rare Earths and Uranium Engineering Study;
- AMEC Minproc’s Plant and Tailings Storage Facility Location Study;
- AMEC Minproc’s March 2012 RSF Addendum Memorandum;
- NIRAS Greenland A/S’s Harbour Study Report;
- NIRAS Greenland A/S’s Accommodation Study Report;
- NIRAS Greenland A/S’s Hydropower Study Report;
- NIRAS Greenland A/S’s Logistics Study Report; and
- Coffey Mining Pty Ltd’s Mining Study Report.

2.10.5 Modularisation

The capital cost estimates developed by the third party studies have assumed that the plant construction will be entirely via "stick-building" techniques. A plant modularisation study performed by AMEC Minproc (7.2 Mt/a Rare Earths and Uranium Engineering Study) has indicated that plant modularisation will be feasible for the Project. In this study AMEC estimated that a capital cost saving in the order of US$30M will result from a modularised approach. This saving has been included in the capital cost estimate.

2.11 OPERATING COST SUMMARY

2.11.1 Operating Cost Summary

The total operating costs for the selected flowsheet for the Project during the first 6 years of production are summarised in Table 2.11.1. Costs are inclusive of mining, process plant, area and regional infrastructure, and BOO(T) costs.

Table 2.11.1 Operating Cost Summary

	Proportion of Cost (%)	Annual Cost (US$'000/a)	Unit Cost
			US$/t Ore
Mining and Haulage	15.2	59,898	8.3
Labour	6.6	26,408	3.7
Power	15.1	60,324	8.4
Reagents	39.4	157,090	21.8
Consumables	5.9	23,337	3.2
Maintenance Materials	9.2	36,837	5.1
General & Administration	8.6	34,443	4.8
Total	**100**	**398,336**	**55.32**

2.11.2 Unit Costs

Four products are produced with the average production for first six years represented below:

- Uranium oxide – 2.6 Mlbs pa U_3O_8 equivalent
- Heavy Rare Earth Hydroxide – 4,200 tpa TREO equivalent.
- Mixed Rare Earth Carbonate – 10,400 tpa TREO equivalent
- Light Rare Earth Carbonate – 26,200 tpa TREO equivalent

The process plant has a capacity of greater than the nominal production rates stated above. The capacity takes into account years of peak production which is used to size the plant equipment.

Unit operating costs have been calculated for each of these products, as shown in Figure 2.11.1, and as summarised in Table 2.11.2.

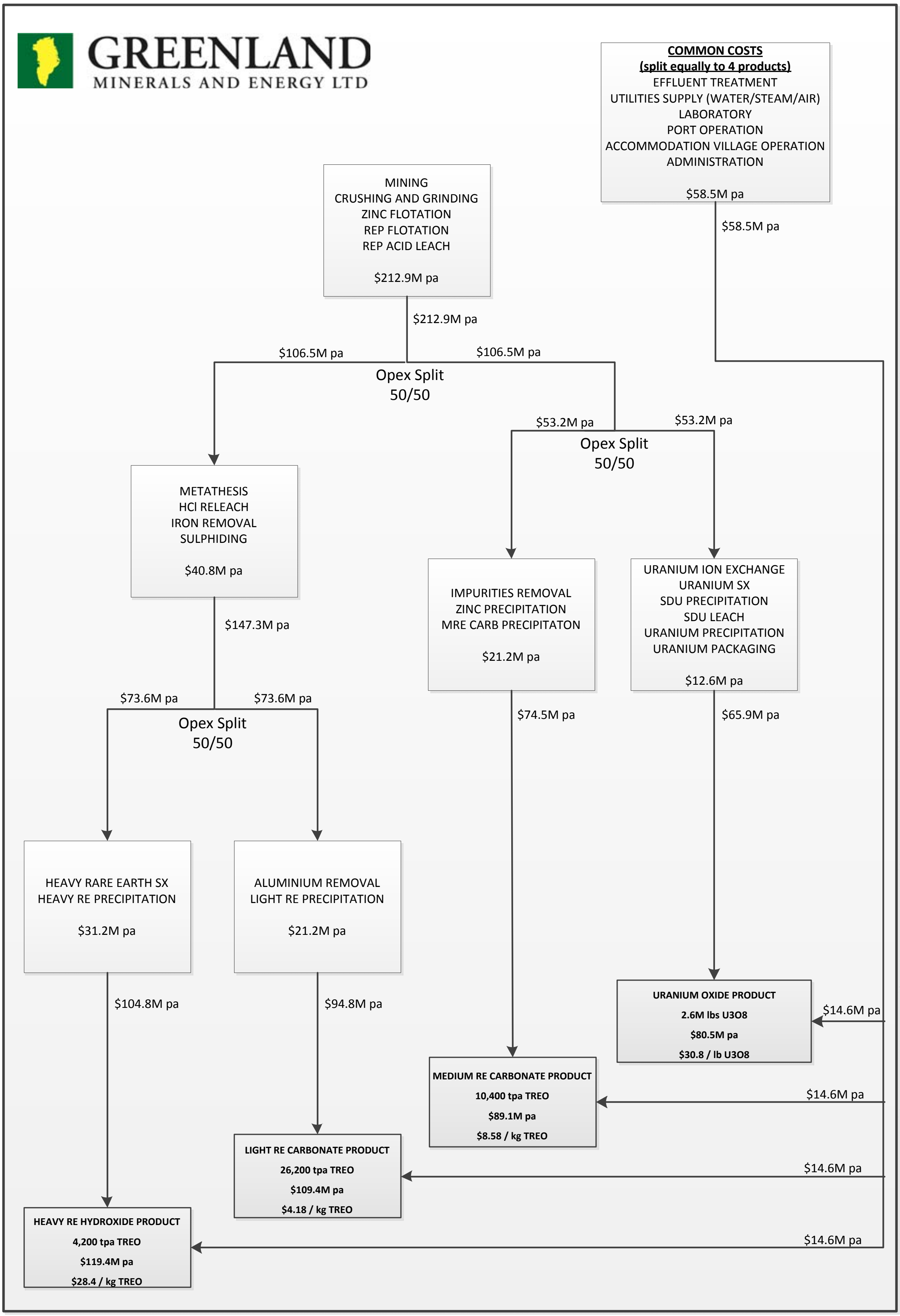

Figure 2.11.1 Unit Operating Cost per Product Allocation

Table 2.11.2 Unit Operating Cost Per Product Summary

	Unit Costs				
	Uranium (US$/lb U_3O_8)	**Heavy RE Hydroxide (US$/kg TREO)**	**Light RE Carbonate (US$/kg TREO)**	**Medium RE Carbonate (US$/kg TREO)**	**Combined RE Products (US$/kg TREO)**
Mining and Haulage	5.73	3.56	0.57	1.44	1.10
Labour	2.53	1.57	0.25	0.64	0.49
Power	3.20	6.46	0.63	0.80	1.27
Reagents	11.03	10.43	1.78	3.63	3.15
Consumables	2.20	1.31	0.24	0.56	0.43
Maintenance Materials	2.82	2.99	0.38	0.68	0.72
General and Administration	3.30	2.05	0.33	0.83	0.63
Total	**30.80**	**28.36**	**4.18**	**8.58**	**7.79**

Note: *The unit costs in presented in this table will vary with ore head grade. The numbers presented here are based on process plant design feed grade of 1.27%wt TREO and 364 ppm U_3O_8. These numbers represent the expected average feed grades for the first 6 years of plant operation.*

2.11.3 BOO(T) Costs

Included in the operating costs are Build Own Operate Transfer, referred to as "BOO(T)", costs for the supply of power, and the operation of the harbour and accommodation village, as summarised in Table 2.11.3. In a typical BOO(T) arrangement, a third party investor constructs and operates a facility that provides a service or services for the exclusive use of a customer. In return, the customer agrees to purchase the services of the facility on terms that provide the investor with recovery of capital and operating costs and a profit margin. The customer may also retain the right to buy the facility at a future date.

Table 2.11.3 Operating Cost Summary – BOO(T) component

	% of Total OPEX	**Annual Cost (US$'000/a)**
Power Supply	10.5	41,680
Port	2.0	7,910
Accommodation Village	1.5	6,150
Total	**14.0**	**55,740**

2.11.4 Estimate Cost Basis

Operating costs were developed with contributions from the following parties:

- Process plant, plant infrastructure and minor area and regional infrastructure – the Company with input from AMEC (Kvanefjeld Multi Element Project – 7.2 Mt/a Rare Earths and Uranium Engineering Study, July, 2011);

- Major area and regional infrastructure – NIRAS Greenland A/S (Kvanefjeld Multi-Element Project - Harbour Study Report, December 2011; Kvanefjeld Multi-Element Project – Accommodation Study Report, December 2011; and Kvanefjeld Multi-Element Project Energy Supply Study Report, March 2011); and

- Mining – Coffey Mining Pty Ltd (Kvanefjeld REE and Uranium Project – Mining Study, August 2011).

2.11.5 Operating Cost Estimation Methods

2.11.5.1 Mining Costs

The operating costs associated with mining and haulage of Kvanefjeld ore have been developed for the project by Coffey Mining, and are fully detailed in their Kvanefjeld REE and Uranium Project – Mining Study report.

The following table summarises the total operating costs estimated by Coffey over a 33 year mine life.

Table 2.11.4 Mining Operating Cost Summary

Cost Item	Total (US$)	US$/t	% Split
DRILLING	185,086,456	0.39	10.1%
BLASTING	128,604,996	0.27	7.0%
LOAD	100,653,793	0.21	5.5%
HAUL	612,106,528	1.27	33.5%
MONTHLY MAINTENACE FEE, MOB., MISC CAPITAL COST ETC	135,022,311	0.28	7.4%
MAJOR ANCILLARY	333,406,256	0.69	18.3%
MINOR ANCILLARY	74,743,812	0.16	4.1%
INDIRECT COSTS	255,456,313	0.53	14.0%
TOTAL	**1,825,080,465**	**3.80**	**100%**

In addition to the above costs supplied by Coffey, further allowances have been included in the operating cost estimate presented in Table 2.11.4 for the following:

- accommodation and messing costs (US$1,200,000 pa);
- flights for FIFO labour force (US$3,100,000 pa); and
- Maintenance costs associated with mining infrastructure (buildings, workshops, etc.) (US$300,000 pa).

2.11.5.2 Labour/Site Manning

Labour complements for management, operations and maintenance have been estimated for the mine by Coffey and the process plant and infrastructure by AMEC, NIRAS, and the Company.

2.11.5.3 Labour Rates

Rates (US$/man/year) have been based on AMEC'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.

2.11.5.4 Power

Electrical power consumption have been calculated across the process plant and estimated for the infrastructure. Power costs are based on across-the-fence supply from a third party BOO(T) operator using a combination of Hydro-electric power and HFO fired equipment. HFO will be supplied to the BOOT operator at cost based on a benchmark oil price of US$80/barrel. Diesel powered equipment is used for mining.

2.11.5.5 Reagents

Consumption volumes for reagents have been calculated based on the mass and energy balances developed by the Company using the IDEAS® process simulation software package. Unit prices for key reagents such as sodium carbonate, sodium bicarbonate, sodium chloride and flotation reagents have been based on budget quotations from reliable suppliers. In-house price information has been used for other reagents. All reagents and consumables include an estimate of freight cost from the source of supply to site. Two freight costs have been used, containerised freight for minor reagents, and bulk freight for major reagents.

2.11.5.6 Maintenance

Maintenance costs are factored from direct capital costs based on benchmarks derived from other similar projects. Maintenance includes allowance for maintenance spares and any specialised contract labour and expenditure on sustaining capital.

2.11.5.7 Consumables

Consumables, other than reagents, include steel balls for milling, steel mill liners, filter cloths, laboratory samples and HFO for steam generation. Grinding media and liner consumptions are based on wear rates as supplied by AMEC. Steam consumption was estimated by the IDEAS® software from mass and energy balances. Relevant operating experience involving similar size plants and ore types was used to develop estimates for other consumables.

2.11.5.8 General and administration

Allowances based on AMEC experience, for general freight costs (excluding reagents), transport (FIFO) costs for personnel not recruited locally in Narsaq, recruitment, training, insurance and administration costs. Harbour and accommodation village operating costs are based on third party BOO(T) operator providing these services.

2.12 MARKETING

The Company will produce and competitively market a range of products including rare earth hydroxides and carbonates, uranium and zinc. Each product has a ready market and in the case of uranium oxide, heavy rare earth hydroxide and, to a lesser extent, mixed rare earth carbonate, demand is expected to exceed supply in 2015.

The processing plant will produce four main products as well as a high grade ZnS concentrate:

• Uranium oxide	2.6 Mlbs pa	U_3O_8
• Light Rare Earth Carbonate	26,200 t pa	TREO
• Mixed Rare Earth Carbonate	10,400 t pa	TREO
• Heavy Rare Earth Hydroxide	4,200 t pa	TREO

2.12.1 Uranium Oxide

The majority of uranium mined today, referred to as primary uranium production, is used in the generation of nuclear power. The outlook for nuclear power continues to improve as a result of the rapid growth in energy consumption in developing countries like China and India and an increasing requirement in countries across the globe for sovereignty over energy supplies.

Other factors likely to contribute to the underlying increase in demand for nuclear energy include environmental concerns over CO_2 emissions and continual improvement in the safety and efficiency of new nuclear reactor designs.

However, developing adequate new uranium production to meet growing demand for nuclear fuels is likely to be a significant challenge for the industry.

The World Nuclear Association (WNA), in its 2011 report on supply and demand, forecast that 80% of the uranium mine capacity currently under development, or planned for development, must come into production by 2015 for the uranium market to remain in balance.

Given the mining industry's poor track record in meeting development schedules, and given the disruption to development plans caused by the accident in 2011 at the Fukushima plant in Japan, it is likely that the schedule and capacity forecasts for new mine production will prove optimistic. As a result, the industry faces a major challenge in being able to meet demand for primary uranium beyond 2015.

Significant new mine production is required thereafter to meet steadily rising demand. The WNA has forecast that primary uranium production will have to increase by approximately 33,000tpa, a 59% increase in mine output from 2011 levels, to meet forecast reactor demand in 2030. A structural deficit is looming in the uranium market, a deficit which is expected to remain significant for the medium to long term, and will put upward pressure on uranium prices.

Uranium prices, both spot and long term, have experienced a boom/bust cycle since the beginning of 2005. However, consistent with a market largely in balance, long term contract prices over the next 5 yrs are forecast by many analysts to remain in the range of US$65 to US$75/lb.

Given expected tightness in the market post 2015, the Company will be well placed to offer potential customers an attractive proposition. Because of the size of the resource and the life of the mine the Company will be in a position to make secure long term commitments for significant quantities of uranium. Further, as uranium is a by-product from the Company's project, customers will have the additional comfort of knowing that security of uranium supply from the Company will not be affected by falling uranium prices.

Given these factors it is the Company's position that there is little downside to marketing its uranium production at the prices forecast, and considerable upside for higher prices.

2.12.1.1 Pricing Assumptions

In the medium to long term, uranium prices will need to be high enough to support the development of new mine production capacity. This new mine capacity will be needed to meet increasing demand. If prices are too low, investment in exploration and new primary production capacity will be deferred or cancelled. Prices higher than current levels will be required before investors develop the confidence to make long term investments in the primary production of uranium.

Sales of U_3O_8 are predominantly undertaken on a long term contract basis. Current data suggests a relatively small surplus over the next 3 yrs and this is supported by the general consensus on long term contract price forecasts which is around US$65 to US$75/lb U_3O_8. The forecasts reflect a market in relative equilibrium with little or no upward pressure on prices.

The most significant event for pricing during 2011 was the accident at the Fukushima plant. Prior to the accident, spot prices had gradually recovered from the lows of 2010 and were above $70/lb. In the immediate aftermath of the accident prices fell rapidly to just above $50/lb were they have remained.

In the Company'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. However, the Company takes the conservative view that a long term contract price in the range of US$65/lb to US$75/lb will be required before investors develop the confidence to make long term investments in the primary production of uranium.

2.12.2 Rare Earths

Rare Earth Elements (REEs) are a group of specialty metals with unique physical, chemical and light-emitting properties. REEs are relatively common in the earth's crust but rarely occur in commercial concentrations, the most significant known occurrences are located in China.

REEs are commonly characterised as being "light" or "heavy". Heavy rare earths elements (HREE) are less abundant than light rare earths elements (LREE). The relative scarcity of HREEs, along with strong industrial demand results in the HREEs being of considerably higher value than LREEs. In the initial phase of the supply chain, rare earths are sold as mineral concentrates and separated rare earth oxides.

The predominant applications for refined REEs are permanent magnets, phosphors, metal alloys, batteries, catalysts, ceramics and in glass polishing.

2.12.2.1 Supply

China has supplied over 90% of the world's REEs since 2006. It has, however, signalled its intention to support domestic industry by limiting the volume of REEs available for export. There was a significant reduction in export quotas in 2010 when, for the first time, export quotas were insufficient to meet rest of world demand. This development sent shockwaves through the REE industry and precipitated the massive price increases seen in 2010 and 2011.

There are two major REE projects in advanced stages of development. Molycorp's Mountain Pass project in California and Lynas' Mt Weld/LAMP project in Australia/Malaysia are both finalising construction activities and should be producing in 2012. Both projects will be producing predominantly LREE.

There is no new source of HREE on the development horizon, other than the Project.

2.12.2.2 Demand

Since the mid-2000s there have been significant increases in REE demand as a result of:

- An ever increasing range of technological applications for REE; and
- A growing recognition of the role that the unique properties of REEs have to play in reducing global warming and the impacts of climate change.

The diverse properties of different REEs make them critical materials to many emerging technologies which are becoming increasingly commonplace.

BCC has forecast that the market for REO will grow by a compound annual growth rate of over 10% through to 2016. The major source of growth is forecast to be the energy sector, primarily driven by increased use of hybrid electric vehicles, and the electronics sector.

2.12.2.3 Pricing assumptions

Supply demand projections show that, in general terms, the markets for elements falling within the basket of light rare earths (LREE) will be well supplied or, in the case of lanthanum and cerium, oversupplied through to 2016. In contrast projections for elements considered to fall in to the heavy rare earth (HREE) basket are forecast to be in deficit. This is an industry consensus view.

The Company referred to two independent and respected sources of price forecasts together with other market sourced information when formulating its forecasts which are set out in the following table.

Table 2.12.1 Price Forecasts

	Company View	**Roskill***	**BCC**	**Actual Mar-23**
		2015	2016	
La	**$10**	$28	$6	$28.00
Ce	**$5**	$13	$4	$27.00
Pr	**$100**	$100	$19	$135.00
Nd	**$100**	$100	$130	$135.00
Eu	**$1,100**	$1,100	$4,350	$3,020.00
Tb	**$1,100**	$1,100	$3,650	$2,220.00
Dy	**$900**	$900	$2,170	$1,170.00
Y	**$50**	$50	$275	$120.00

*midpoint of range quoted

The Company has adopted a conservative approach to its price forecasts opting for the lower end of the range of prices suggested by Roskill and BCC.

Using the price forecasts in Table 2.12.1, the average “basket” value of rare earth product from the

Project is US$41.61 per kg. The Company has assumed that it will be paid for 60% of the contained rare earth value in each kilo. This is equivalent to an average sales price of US$24.96 per kg, which can be broken down as follows for the three separate products:

- Heavy Rare Earth Hydroxide US$82.20/kg
- Mixed Rare Earth Carbonate US$26.10/kg
- Light Rare Earth Carbonate US$15.60/kg

The US Department of Energy (DoE) has developed a classification for REEs based supply risk and significance in existing and developing clean energy applications. The DoE has identified 5 "critical" REEs being those both at high supply risk and of significant importance in existing and developing clean energy applications. Those 5 are neodymium, dysprosium, yttrium, europium and terbium.

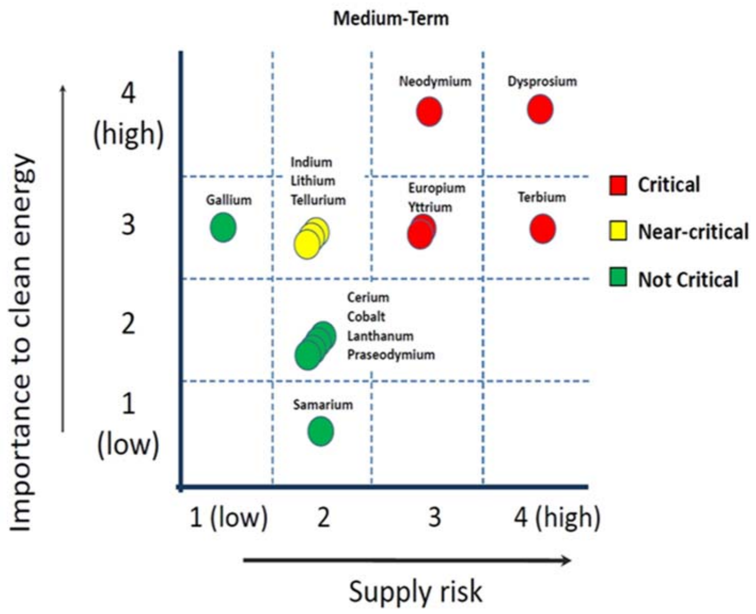

Figure 2.12.1 US Department of Energy – Critical Elements Matrix

The Company will be a globally significant, long term supplier of these critical rare earths.

The Project will produce:

1 A heavy rare earth hydroxide comprised almost entirely of REE identified as critical by the DoE. This product will be in high demand. The Company will be the only new supplier of this type of material when the project comes into operation.

2 A mixed rare earth carbonate comprising a mixture of heavy and light REE. This product will have a ready market as access to the contained heavy REE and neodymium will be attractive to customers.

3 A light rare earth carbonate comprised almost entirely of light REE. This is a more competitive market but the Company will be able to effectively compete for customers in the market because it has a range of products that its competitors do not, and, having

several revenue streams, the Company is less sensitive to fluctuations in prices for individual products than its competitors.

2.13 FINANCIAL EVALUATION

The Financial Model is a discounted unleveraged cash flow model (DCF) of the Project which has been built in Microsoft 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 Project. The model uses net present value (NPV), internal rate of return (IRR) and payback period as its evaluation metrics.

The **key financial results** from modelling the base case are provided in Table 2.13.1.

Table 2.13.1 Key Financial Results

	Discount Rate - 10%
Ungeared project return (pre-tax)	
NPV	US$4,631 M
IRR	32%
Payback period (undiscounted cashflows)	
From commencement of construction	5-6 years
From commencement of operations	3-4 years
Ungeared project return (post-tax)	
NPV	US$2,947M
IRR	26%

This is a very favourable outcome and demonstrates the robustness of the Project.

It reflects:

- The large scale and quality of the resource;
- The ease of mining;
- The attractive location of the Project;
- The relatively simple flowsheet which utilises conventional technology; and
- The increasing demand and value expected for the various products.

2.13.1 Capital and Operating Costs

The capital cost for the project is US**$1.534Bn** and the annual operating costs are **US$398.34M.**

The capital estimate comprises:

Direct costs	US$1,111M
Indirect costs	US$ 176M
Accuracy provision/contingency	US$ 247M
TOTAL	**US$1,534M**

Annual and unit costs by product are set out in Table 2.13.2

Table 2.13.2 Operating Costs by Product

	Total Annual Cost, US$M	Unit cost US$
Uranium	80.47	**30.80/lb**
Heavy rare earth hydroxide	119.38	**28.36/kg**
Mixed rare earth carbonate	89.07	**8.58/kg**
Light rare earth carbonate	109.37	**4.18/kg**

The Company will be a very low cost, competitive producer of each of its products

2.13.2 Base Case

The base case of the financial evaluation has been mainly expressed as a pre-tax evaluation as the likely tax regime has not been settled. The model, however, allows the impact of tax on the Project to be calculated and, for the purposes of this evaluation, a flat tax rate of 32% (the profits tax rate for companies in Greenland) has been used.

The model includes carried forward tax losses of approximately US$40M.

The NPV of the Base Case has been calculated on a pre-tax basis. The after tax NPV is US$2,947M.

2.13.3 Base Case Parameters

The Project is scheduled to commence construction in 2014. The construction programme is expected to take approximately 2 years, with mining operations, and hence revenues, commencing in 2016.

At the proposed mine and plant capacity of 7.2Mt/a, the project has a mine life of 33 years based on the present mine plan.

The key assumptions that underpin the project’s ability to achieve the financial performance set out in the Base Case are:

- The financial outcomes represent 100% of the Project and ignore ownership and financing structure;
- The financial outcomes are stated in un-escalated real dollars and are presented in US$ unless otherwise noted;
- Capital and operating cost estimates were prepared in real 2012 dollars. It is assumed that these remain constant in real terms;
- Technical and economic estimates are prepared to a tolerance of ±25% unless otherwise stated;
- There are no mining royalties in Greenland and no allowance for royalties has been made in the financial estimates;
- Prices for the products sold, UO_4, heavy rare earth hydroxide, mixed rare earth carbonate and light rare earth carbonate are based on estimates generated for long term contracts commencing from first production in 2016. Product prices modelled are based on the prices for the equivalent quantities of U_3O_8 and the mix of individual rare earth elements contained in the three rare earth products;
- The excess sodium hydroxide and chlorine produced by the chlor-alkali plant in the REE plant is converted into sodium hypochlorite (100% w/w) and sold at a discount of 50% to the quoted market price; and
- The zinc sulphide produced is sold at the current market price for bulk zinc concentrate. The revenue from zinc sales is, in the first instance, credited against the cost of operating the zinc recovery circuit. Surplus “credit” is then deducted from the operating costs of the processing plant.

The key project statistics for the Base Case are set out in Table 2.13.3.

Table 2.13.3 Key Project Statistics and Financial Metrics		
Project mine life	Years	33
Construction period	Years	2
Capital cost	US$M	1,535
LOM production		
Mine production	Mt	230.8
U_3O_8 production	Mlb	78.3
RE production	Kt	1,254.30
LOM sales revenue		
U_3O_8	US$M	5,484.60
RE products	Tonnes	31,230.80
Cumulative free cash flow	US$M	22,693
Annual production *		
Uranium	Mlb	2,613
Heavy rare earth hydroxide	Tonnes	4,209
Mixed rare earth carbonate	Tonnes	10,381
Light rare earth carbonate	Tonnes	26,165
* Years 1 - 6		
Annual revenue		
Uranium	US$M	183
Heavy rare earth hydroxide	US$M	346
Mixed rare earth carbonate	US$M	271
Light rare earth carbonate	US$M	409

Sensitivity analyses were completed on assumptions in the model relating to:

- Product prices;
- Operating costs; and
- Capital costs.

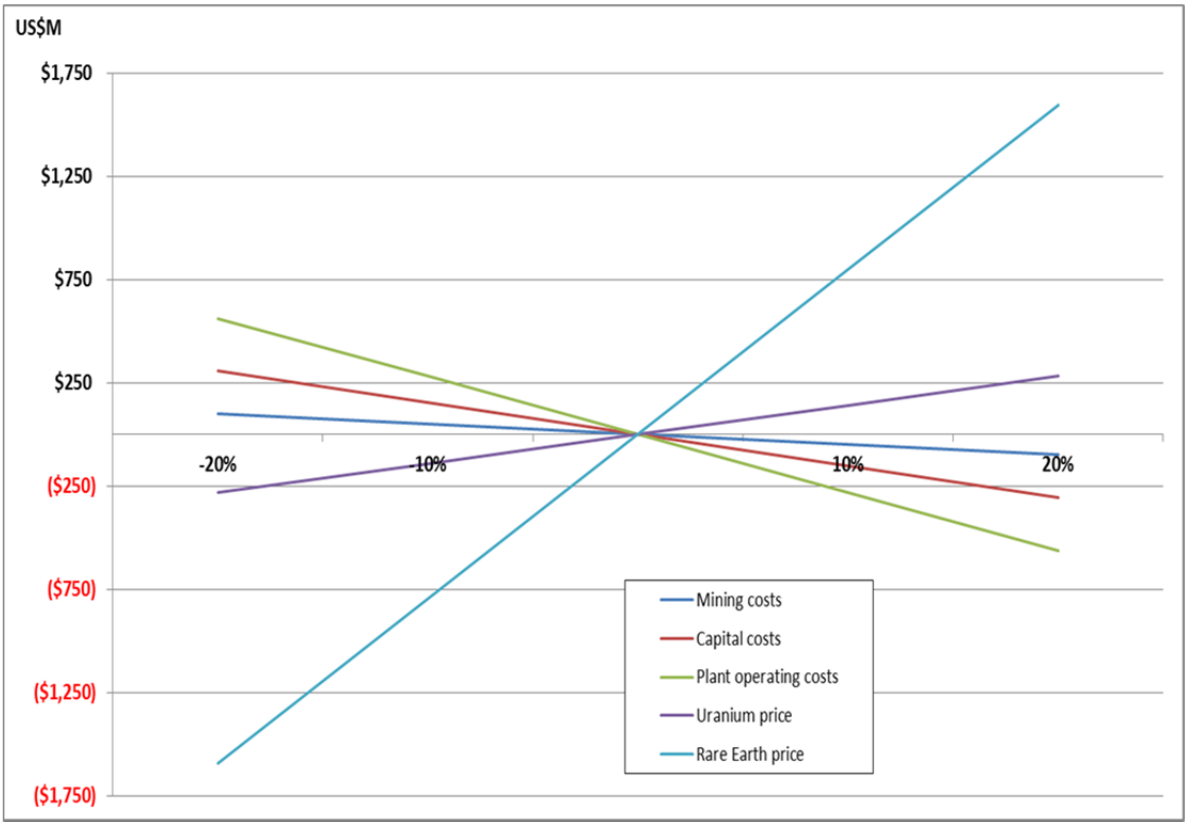

Figure 2.13.1 Input Sensitivities

Figure 2.13.1 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 variable.

The Project is most sensitive to changes in the price for rare earths. The next most significant sensitivity is Refinery operating costs. Project value is least sensitive to mining operating costs.

Table 2.13.4 shows the impact on after tax Project NPV of a 10% reduction for each of the key inputs.

Table 2.13.4 Impact on after tax Project NPV

Vector	Impact of a 10% reduction US$M	Resulting Project NPV US$M
Mining costs	50	4,681
Capital expenditure	154	4,785
Operating costs	281	4,912
Uranium price	-141	4,490
Rare earth price	-796	3,835