PRE-FEASIBILITY STUDY
ON THE AAPPALUTTOQ RUBY PROJECT, GREENLAND
TRUE NORTH GEMS

AAPPALUTTOQ RUBY PROJECT PREFEASIBILITY STUDY
6TH JUNE 2011

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1 EXECUTIVE SUMMARY

EBA, A Tetra Tech Company (EBA) was commissioned by True North Gems Inc. (the "Company") to prepare an independent National Instrument 43-101 (NI 43-101) technical report on the Aappaluttoq Ruby Project, Greenland. The Project is located 20 kilometers southeast of Qeqertarsuatsiaat in western Greenland. The effective date of this report is the 6th June 2011.

The project contemplates an open pit mine, the installation of a process plant, and associated support infrastructure. The operation will be seasonal, producing up to 25,000 tonnes of ore and 430,000 tonnes of waste per year. A rough corundum concentrate will be produced at the minesite and shipped for further processing at the Company's facility in Nuuk. The final product will be a mix of rough ruby/sapphire, along with cut and polished ruby and sapphires.

The Aappaluttoq Ruby Project returns a positive pre-tax Net Present Value (NPV) of $25.7 M at an 8% discount rate and an IRR of 19.1%. The after-tax NPV is $17.5 M at 8% discount and the IRR is 16.5% with an after-tax payback period of 5½ years. Because of the positive economics and low technical risks, it is recommended that further work is completed in aim of establishing an operation at Aappaluttoq.

A draft Social Impact Assessment and draft Environmental Impact Assessment have been submitted along with an application for an Exploitation Permit to the Greenland government on the June 6, 2011. The Company initiated formal consultation with the key stakeholders including the government and local communities in February, 2011.

1.1 Property Description and Location

The Aappaluttoq project is located in southwest Greenland, approximately 150 kilometres south of the capital Nuuk and 20 km southeast of the town of Qeqertarsuatsiaat in the Fiskenæsset mining district. Aappaluttoq is located at 63° 0' 39" north latitude and 50° 19' 11" west longitude.

1.2 Climate Information

Aappaluttoq is located in a maritime influenced polar tundra region. (Kottek, et al. 2006). This is characterized by low daily temperatures, ranging from a low of -10°C in the winter to +10°C in the summer. The amount of sunlight per day varies greatly throughout the year, with long nights in the winter, and long days during the summer. Precipitation remains constant throughout the year, with about 12 precipitation days per month. (World Meteorological Organization 2011) (BBC World News 2011).

1.3 Ownership

The Company has a 100% interest in the Aappaluttoq property, located on Exploration License No. 2008/46, covering 110 square kilometres in three property blocks. The exploration licenses provide exclusive rights to pursue exploration activities in the areas covered by the licenses.

Under Greenland law, there is no ownership of surface rights. For mining, all ownership belongs to the Government of Greenland.

1.4 Geology and Mineralization

The corundum showings at Aappaluttoq typically include occurrences of co-existing red ruby and pink sapphire.

The geology of the Aappaluttoq area is dominated by an intrusive gabbro to leucogabbro sequence of rocks with significant volumes of ultramafic rock. This sequence is intruded into and is structurally juxtaposed against the felsic gneiss basement suite. The Aappaluttoq ultramafic body is internally zoned, with a barren ultramafic core (olivine and lesser pyroxene). It is lensoidal in shape and has a minimum strike length of 170 m and is up to 70 m thick. Gradational alteration is prevalent and evident where ultramafic rocks have been altered to phlogopite. It is in these metasomatic/metamorphic reaction zones between the leucogabbro and ultramafic stratigraphy where the ruby mineralization is mostly concentrated.

1.5 Exploration

Exploration work consists of diamond drilling, mapping, and bulk sample collection. In 2007, 46 drill holes were completed at Aappaluttoq totalling 4,622.1 meters. In 2008, 19 drill holes were drilled totalling 1,834.7 meters. Bulk samples were collected by standard techniques and recovery procedures including cutting with chain saws, chisels, and use of low intensity blasting. Bulk sample sizes increased from 30 t in 2006, to 54 tonnes (28 tonnes rock, plus 26 tonnes of regolith) in 2007, and 160 tonnes (125 tonnes of rock, ~35 tonnes of regolith) in 2008.

1.6 Resource Estimation

The Aappaluttoq open pit mineral resource was estimated in March 2011 by EBA. This initial resource estimate was prepared by EBA from 6,457 m of drilling data and approximately 90 tonnes of bulk samples collected on the property over the last several years and uses recently updated geologic interpretations for the host zone lithology.

Search ellipses for the interpolation profiles are based on geology and observed continuity of the phlogopite host zone. A lower cut-off grade of 1 gram per tonne was selected from evaluation of grade tonnage relationship at several cut off grades. The grade data was interpolated into block models using an inverse distance interpolator with a power of 2 (ID2). The resource is presented in Table 1.

Table 1 Indicated And Inferred Resources (March, 2011)

Category	Volume	Tonnage(1)	Average Grade(2,3)	Average Grade(2,3,4)	Contained Corundum (2,3)	Contained Corundum(2,3,4)
	m3	*t*	*g/t*	*ct/t*	*M.g*	*M.ct*
Indicated	59,110	189,150	313.33	1,566.65	59.27	296.33
Inferred	24,110	77,160	283.46	1,417.28	21.87	109.35

Notes:

(1) Densities are derived from specific gravity measurements of host lithologies and estimated for host zone based on specific gravity of corundum and average grade.

(2) Based on a Total Clean Corundum grades greater than 1.7 mm size fraction from mineralogical lab analysis.

(3) Top cut grade of 7,325 grams per tonne (97.5 percentile), and a lower cut-off grade of 1 gram per tonne.

(4) One gram equals five carats.

1.7 Reserve Estimation

The open pit design is based on a selected shell from a series generated using Gemcom's Whittle 4X software. The pit design uses geotechnical and practical mining parameters which should allow safe and efficient extraction of ore. These parameters are shown in Table 25. The shell selection and pit design is based on Indicated Mineral Resources only. Table 2 below shows the mineral reserve.

Table 2: Aappaluttoq Open Pit Mineral Reserves

	Reserve	**Corundum**	**Contained Corundum**
	t	*g/t*	*M.g*
Proven	-	-	-
Probable	161,700	350	56.6
Total	161,700	350	56.6

The Mineral Reserve contains 95% of the Indicated Mineral Resource in reference to total corundum, and 85% in reference to tonnage.

For the Mineral Reserve, the percentages of gem and near-gem rubies and sapphires contained in that corundum are based on the percentages determined from the analysis of the B1 and B2 bulk samples taken in 2006 and 2007. This information is shown in Table 3. Valuations are only applied to the gem and near-gem portion of the corundum. No value was applied to the non-gem material. The combination of the B1 and B2 were the most representative samples to use in the estimation of gem distribution characteristics as there was increased knowledge of the deposit and greater scrutiny of the samples.

Table 3: Gem Distribution of B1 and B2 Gem and Near Gem Material

Gem Colour	Total weight	Distribution
	g	%
Gem and Near Gem Red (Ruby)	6,467	5.0%
Gem and Near Gem Pink (Sapphire)	44,741	34.3%
Total Gem and Near Gem	51,208	39.3%

1.8 Hydrology

The water balance model suggests that during the proposed mining operation, sufficient water will be available for operation of the mine.

Once the mining operation is completed the lake will be allowed to return to its natural level. Based on an average annual runoff volume calculated as part of the flow routing exercise and the amount of storage required to fill up the lake from the mine operating level, the lake is expected to return to its natural level in approximately 3 years.

1.9 Geotechnical

Geotechnical work on the property has been limited to the geological logging of drill holes. These logs along with core photos were used to calculate the Rock Mass Rating (RMR) which was found to be classified as “good rock”. Acceptable overall slope angles were calculated to between 56° and 65°.

For overall slope stability, the determined static factor of safety is 1.7. This factor of safety is higher than the acceptable static factors of safety of 1.3 commonly adopted in open pits.

1.10 Acid Rock Drainage and Metal Leaching

The initial assessment of the acid rock drainage and metal leaching potential of the Aappaluttoq Ruby Project in Greenland was undertaken in 2009-2010 and was based on discussions and review of drill logs, test data and geological information provided by the Company, and analysis of select core samples from the property.

Based on the results of the preliminary work, a total of 109 core samples, representing sulphide-bearing and non-sulphide intervals of each major lithology at the property, were selected and analysed.

Subaqueous deposition of waste rock and tailings material in the lake will minimize sulphide weathering and reduce potential acid generation to negligible rates in PAG and low-PAG materials by limiting exposure to free oxygen.

Short-term leaching tests indicated that these elements are not likely to be mobilized to any significant extent under the neutral pH drainage conditions that prevail at the project site.

1.11 Development and Operations

The project will involve the development of a mine and process facility at Aappaluttoq along with associated infrastructure and the construction of a corporate office and ruby sorthouse in the Greenland capital of Nuuk.

The ore at Aappaluttoq is extracted using open pit mining methods. Pit optimization, design and production scheduling indicates a high stripping ratio operation which benefits from low mining costs, short ore and waste haul distances, low precipitation environment, competent ground conditions and available local workforce that all combine to give favourable economics over the expected nine year mine life.

The mining operation will be a typical open pit operation. Mining will use conventional blast, load and haul equipment mining up to 450,000 tonnes of total material over an eight month operating season from beginning of April to the end of November each year.

Ore production will increase gradually from approximately 1,000 tonnes in the first year (to satisfy plant commissioning and initial operation) to nearly 25,000 tonnes in the latter years. Increased production is achieved by extending the work season from four months in early years to 8 months in later years.

The major items of rock-moving equipment in the mine are a single 47 tonne excavator and four 35 tonne articulated dump trucks. Table 7 shows a summary of the production schedule and general economics.

Process facilities will include a crushing, screening and jigging circuit to separate and extract the corundum from the host rock. The facility will also have several handpicking stations where operators will be able to extract visibly superior gems. Surveillance and security systems will be put in place to eliminate theft to the maximum extent possible. The concentrate produced (dirty rough concentrate) will contain gem, near-gem and non-gem corundum. Mine production increases will be achieved by extending the operating season from 4 to 8 months, and also by the addition of further mineral processing jigs.

Infrastructure will include a complete camp to accommodate workers, approximately 3 km from the mine, power generation facilities, communication facilities, explosives storage, roads, fuel storage, helipad, and a port close to the camp. A second port will be developed close to the narrow entrance to the fjord to allow goods to be temporarily off loaded and stored if tides and weather conditions do not favour direct shipment to the inner port.

Total manpower at the site during operations is planned to be 63 people over two shifts. During construction manpower is expected to peak at 60 people.

Main access to the site during construction and operations for goods and personnel will be by a four hours boat ride from Nuuk with calls as necessary at nearby villages. Helicopter support will be available; a roughly one hour flight. The dirty rough concentrate will be transported in secure containers by helicopter and boat on an irregular schedule.

The dirty rough concentrate will be taken to the Nuuk facility for initial checking and sorting and will then be taken to a local laboratory to be treated with hydrofluoric acid to remove any host rock still attached to the corundum. The resulting clean rough concentrate will be returned to the sorthouse where it will be graded by size and quality before shipping to either cutting and polishing facilities or to market as rough gems.

1.12 Capital Costs

The capital cost expenditures are divided based on initial and sustaining cost requirements. As of the date of this technical report, the Company has limited infrastructure on site consisting principally of an exploration camp.

The total capital costs for the Aappaluttoq Ruby Project including Nuuk over the life of mine is $40.7 M including $5.0 M contingency (13.8%). This includes sustaining capital and reclamations costs but excludes capital salvage. Table 4 shows the breakdown of these costs. Salvage of equipment and plant was conservatively estimated at $7.1 M.

Table 4: Aappaluttoq Capital Expenditures

Item	Initial	Sustaining	Total
	$'000	*$'000*	*$'000*
Mining	11,062	0	11,062
Processing	6,333	430	6,763
Tailings handling	141	0	141
Camp site, infrastructure & facilities	3,625	0	3,625
Temporary services	69	0	69
Inner port, staging area & fuel	485	0	485
Outer port, barge & staging area	728	0	728
Roads	2,929	0	2,929
Nuuk	280	0	280
Indirects	7,398	2,267	9,665
Contingency	4,389	561	4,950
Total	37,439	3,258	40,696

1.13 Operating Costs

The Aappaluttoq life of mine unit operating costs are shown in Table 5. Operating costs are expected to be $11.1 M per year over the mine life. The estimate includes an overall contingency of 8.5%.

Table 5: Aappaluttoq Operating Expenditures

Item	Annual average	Life of mine total
	$'000	*$'000*
Mining	1,922	17,298
Processing	1,805	16,244
General and admin	1,785	16,068
Nuuk	3,797	34,175
Marketing	440	3,958
Corporate	440	3,958
Contingency	862	7,759
Total	11,051	99,459

1.14 Economic Evaluation

The Aappaluttoq Ruby Project returns a positive pre-tax Net Present Value (NPV) of $25.7 M at an 8% discount rate and an IRR of 19.1%. The after-tax NPV is $17.5 M at 8% discount and the IRR is 16.5% with an after-tax payback period of 5½ years (Table 7).

These figures are based on 100% equity investment in the project by the Company. No financing terms are contemplated in the economics.

All aspects of the mine operation and gem processing and sorting are assumed to be owned and operated by the Company.

Over the nine year mine life, operating costs are $99.5 M and capital costs including sustaining capital are $40.7 M excluding any salvage value. Pricing used for the gems in the economic evaluation is considered to be conservative and is based on three independent valuations obtained for cut and polished stones.

All figures are reported in second quarter 2011 Canadian dollars.

The following provides the basis for the estimate:

- Probable reserves totalling 161.6 kt at an average grade of 350 g/t corundum, containing 56.6 Mg corundum
- A mine life of 9 years
- An overall site recovery rate of 95%
- Polished stone retention of 9.3%
- 100% of ruby sold as polished, 60% of pink sapphire sold as polished

No escalation has been provided for in cost or revenue estimates apart from a modest 2% annual increase in gem value ascribed to a branding factor.

1.15 Sensitivities

The project is sensitive to price, grade, operating costs and capital costs in that order. Table 6 shows the before tax NPV for each sensitivity ranging from -30% to +30%.

Table 6: Aappaluttoq Project Sensitivities

Item	-30%	-10%	-5%	0%	+5%	+10%	+30%
Proportion of Pink Sapphire sold as polished *	-16,495	11,617	18,645	25,673	32,700	39,728	67,840
Grade	-9,143	14,067	19,870	25,673	31,475	37,278	60,488
Ore tonnes	-9,143	14,067	19,870	25,673	31,475	37,278	60,488
Sapphire polish price	-1,718	16,542	21,107	25,673	30,238	34,803	53,063
Ruby polish price	17,337	22,894	24,283	25,673	27,062	28,451	34,008
Operating costs	45,731	32,359	29,016	25,673	22,329	18,986	5,614
Capital costs	36,741	29,362	27,517	25,673	23,828	21,983	14,604

Note:
* base case at 0% change refers to 60% of pink sapphire sold as polished.

Table 7: Aappaluttoq Project Production Schedule And Economics

	Year	0	1	2	3	4	5	6	7	8	9	10	Total
Mining													
Waste	*kt*	0	237	213	426	421	400	335	237	179	168	0	2,616
Ore	*kt*	0	1	7	20	21	22	22	22	23	24	0	162
Total	*kt*	0	238	220	445	442	422	357	259	202	192	0	2,777
Ore Grade													
Ruby	*g/t*	0.0	17.1	88.8	64.2	46.0	47.8	50.4	46.5	42.2	81.6	0.0	55.6
Sapphire	*g/t*	0.0	90.7	470.7	340.3	243.8	253.1	266.9	246.6	223.5	432.6	0.0	294.7
Total Al203	*g/t*	0.0	107.8	559.5	404.6	289.9	300.8	317.3	293.2	265.7	514.2	0.0	350.4
Revenue													
Ruby	*$'000*	0	86	2,870	6,043	4,701	5,173	5,551	5,163	5,110	10,712	0	45,409
Sapphire	*$'000*	0	290	9,636	20,293	15,786	17,373	18,641	17,337	17,160	35,974	0	152,491
Total	*$'000*	0	376	12,506	26,336	20,487	22,546	24,192	22,500	22,270	46,687	0	197,900
Operating Costs													
Site	*$'000*	0	3,415	3,386	6,368	6,361	6,342	6,256	6,038	5,967	5,952	0	50,086
Nuuk	*$'000*	0	2,195	3,361	4,113	3,866	3,908	3,957	3,864	3,836	4,597	0	33,698
Marketing	*$'000*	0	8	250	527	410	451	484	450	445	934	0	3,958
Corporate	*$'000*	0	8	250	527	410	451	484	450	445	934	0	3,958
Contingency @ 8.5%	*$'000*	0	496	639	984	946	950	944	906	891	1,003	0	7,759
Total	*$'000*	0	6,121	7,886	12,519	11,992	12,102	12,125	11,708	11,585	13,420	0	99,459
Capital Costs													
Site	*$'000*	4,447	20,926	430	0	0	0	0	0	0	0	-7,102	18,700
Nuuk	*$'000*	56	224	0	0	0	0	0	0	0	0	0	280
Indirects	*$'000*	1,480	5,918	0	0	0	0	0	0	0	0	2,267	9,665
Contingency 13.8%	*$'000*	878	3,511	107	0	0	0	0	0	0	0	453	4,950
Total	*$'000*	6,860	30,579	537	0	0	0	0	0	0	0	-4,382	33,594
Financial Analysis													
Revenue	*$'000*	0	376	12,506	26,336	20,487	22,546	24,192	22,500	22,270	46,687	0	197,900
Operating	*$'000*	0	-6,121	-7,886	-12,519	-11,992	-12,102	-12,125	-11,708	-11,585	-13,420	0	-99,459
Capital	*$'000*	-6,860	-30,579	-537	0	0	0	0	0	0	0	4,382	-33,594
Pre-tax cashflow	*$'000*	-6,860	-36,324	4,083	13,818	8,495	10,444	12,067	10,791	10,685	33,266	4,382	64,847
Estimated Tax	*$'000*	0	0	0	0	0	0	0	440	3,210	12,309	0	15,959
After-tax cashflow	*$'000*	-6,860	-36,324	4,083	13,818	8,495	10,444	12,067	10,351	7,475	20,958	4,382	48,887

1.16 Conclusions and Recommendations

The results of the Pre-Feasibility Study model for the Aappaluttoq Ruby Project show that this project has good economic potential even under the conditions of a conservative product prices and market.

The metallurgical characteristics of the rock are the most significant technical unknown at this stage.

The market and product prices are economics risks that are unlikely to benefit from further study and analysis. It is reasoned that operations will need to commence before these unknowns can be understood.

It is unlikely that will be significant environment social or environmental impacts on the local or national surroundings.

The Qualified Persons for this report recommend that the Company move to the next stage of the engineering process by undertaking the required work in preparation of commencing production.

A budget of approximately $1.5 M is recommended to complete exploration field programs, engineering and test work including:

- Test work of metallurgical characteristics of the rock for crushing and sorting
- Test work on corundum recovery
- Sourcing and quotations on major equipment items

Further drilling and survey is also recommended to increase the geotechnical understanding which can be done during early stage operations.

2 INTRODUCTION AND TERMS OF REFERENCE

This report summarizes the Pre-Feasibility Study (PFS) of the Aappaluttoq Ruby Project prepared by EBA, a Tetra Tech company ("EBA") of Vancouver, BC; together with several independent professionals and consultants. Geological information, drillhole database, mineral resources and background data presented in this report was supplied to EBA by True North Gems (the "Company"). The mineral resources and reserves used as the basis for economic analysis was prepared and validated by EBA using the data that was supplied in March 2011. This report complies with NI 43-101 Standards. The effective date of this report is 6th June 2011.

John Chow, MAusIMM, MIEAust and Lara Reggin, P.Geo, visited the Aappaluttoq property October 31st to November 5th, 2010 and were accompanied by Andrew Fagan, M.Sc., and Jonathan Clegg, P.Eng, of the Company and Lars Henrick Larson of MT Hojgaard (MTH). The site visit consisted of a tour of the property, the mothballed pilot plant in Qeqertarsuatsiaat, introduction to the property geology and mineralization, review of site infrastructure, Nuuk, and the site history.

Units of measurement used in this report conform to the SI (metric) system. All currencies in this report are in Canadian dollars (CAD) unless otherwise noted. A list of symbols, acronyms and abbreviations is given in Table 8.

Table 8: List Of Symbols, Acronyms and Abbreviations

µ	micro-	kph	kilometre per hour
°C	degree Celsius	kPa	kilopascal
A	ampere	L	litre
a	annum	lb	pounds
amsl	above mean sea level	m	metre
ARD	Acid Rock Drainage	M or M.	mega (million)
CAD	Canadian dollar	Ma	Million years ago
Cal	calorie	min	minute
cfm	cubic feet per minute	mm	millimetre
cm	centimetre	NAG	not acid generating
ct	carat (0.2 grams)	PAG	potentially acid generating
d	day	ppm	part per million
DKK	Danish Krone	ppb	part per billion
ft *or* '	foot	s	second
g	gram	t	metric tonne
G	giga (billion)	tpa	metric tonne per annum
h *or* hr	hour	tpd	metric tonne per day
ha	hectare	USD	United States dollar
HP	horsepower	V	Volt
in *or* "	inch	W	Watt
k	kilo (thousand)	yd	yard
kg	kilogram	yr	year
km	kilometre		

3 RELIANCE ON OTHER EXPERTS

This report has been prepared by EBA in reliance on a large body of data compiled by others, including specific contributions as noted below. Information from third party sources is footnoted, quoted as a report in the text, or referenced. All sources of data included in this report are believed to be reliable by EBA. The information, opinions, estimates, and conclusions contained herein are based on:

- Information available at the time of preparation of this report.
- Assumptions, conditions, and qualifications as set forth in this report.
- Data, reports, and other information supplied by the Company and other third party sources.

For the purpose of this report, EBA has relied on property ownership information provided by the Company and as made publicly available by the Greenland Government. This PFS was completed with the reliance on the following Independent Qualified Professionals (IQP's), who are the authors of this report:

- John Chow, MAusIMM, MIEAust, EBA, a Tetra Tech Company. Responsible for sections 16, 17.9 and 19 and related portions of section 1, 20 and 21.
- Lara Reggin, P.Geo, EBA, a Tetra Tech Company. Responsible for sections 2 through section 17 excluding 17.9, section 18 and related portions of section 1, 20 and 21.

This report also contains contributions from the following experts:

- Lee Groat, PhD, UBC Professor, contributed to the development of the Mineralization and Deposit Type sections
- Andrew Fagan, M.Sc., FGS, Project Geologist True North Gems, completed the Mineralization and Deposit type sections. Provided input and review for mineralization and deposit geology sections.
- Bonnie Weston, P. Geo., G.G., A.J.P., Geologist/Gemologist, True North Gems, provided information regarding the history of the project and of the field programs completed to date.
- Jeffrey Giesbrecht, Ll.B., VP Corporate Development, True North Gems contributed to the tax and marketing sections
- Jonathan Clegg, P.Eng., Project Consulting Engineering, True North Gems, provided input and review to the mine planning and infrastructure.
- N. Eric Fier, CPG, P.Eng. EBA, provided review and input into the mineral resource modelling
- Larry Gilroy. Plant Manager of the 2006 Fiskenæsset pilot plant
- Dr. Rick Lawrence, P.Eng., Partner, Lawrence Consulting Ltd. contributed to the geochemical (ARD/ML) characterization
- Brian Soregaroli, M.Sc. MBA completed the Geochemical (ARD/ML) Characterization
- Karolina Kuzniar, M.Sc. Eng., Project Engineer, MT Højgaard Grønland ApS/MT Højgaard A/S, contributed to areas of health and safety, and closure planning sections
- Lars-Henrik Larsen, B.Sc. Eng., Construction Site Engineer/Geotechnical Engineer, MT Højgaard Grønland ApS/MT Højgaard A/S, Contribution to infrastructure, and logistics sections

- Brian Heerfordt, Building Technician and Estimating Manager, MT Højgaard Grønland ApS/MT Højgaard A/S, contributed to the capital cost estimation
- Nikolaj Piecnik Brandt, B.Sc. Eng., Project Manager, MT Højgaard Grønland ApS/MT Højgaard A/S, contributed to the operating cost estimate
- Kim Ulrik Hansen, B.Sc. Eng. (Hon.), Dept. Manager, MT Højgaard Grønland ApS/MT Højgaard A/S, reviews of all mine planning and infrastructure sections
- Ole Geertz-Hansen, Ph.D., M.Sc., Biologist (Marine and Freshwater), Rambøll Denmark contributed to the baseline study, environment description, Environmental Impact Assessment and monitoring program
- Sesse Bank, M.Sc., Biologist and Geographer, Rambøll Denmark contributed to the environmental description and Environmental Impact Assessment
- Ana Gabriela Factor, Senior Environmental Consultant, Grontmij contributed to the Social Impact Assessment
- Tania Nielsen, M.Sc., Senior Advisor, Grontmij A/S contributed to the Social Impact Assessment
- Heidi Hjørt, M.Sc., Senior Consultant, Grontmij A/S contributed to the Impact and Benefit Plan, and the Monitoring and Evaluation Plan in the Social Impact Assessment.
- Richard Weizenbach, P.Eng, Senior Hydrometallurgist, Wardrop, a Tetra Tech Company contributed to the design and costing of the mineral processing.
- Sonia Aredes, P.Eng, Metallurgist, Wardrop, a Tetra Tech Company contributed to the design and costing of the mineral processing.
- Jake Alexander, M.B.A, Senior Metallurgist and Director India, Wardrop, a Tetra Tech Company contributed to the Nuuk processing plan and site security aspects
- Carlos Chaparro, P.Eng, Senior Geotechnical Engineer, EBA, a Tetra Tech Company contributed to the geotechnical analysis.
- Maria Lau, P.Eng, Hydrotechnical Engineer, EBA, a Tetra Tech Company completed the watershed and site water balance.
- Albert Leung, Hydrotechnical Engineer, P.Eng, P.E, EBA, a Tetra Tech Company completed the hydrotechnical analysis of Lake Ukkaata Qaava.

This PFS was completed with the reliance on the following Reports:

- NI 43-101 Technical Report titled “2008 Report on Field Activities for the Fiskenaesset Ruby Project, Greenland” prepared by Bonnie Weston, P.Geo., G.G., of True North Gems Inc. (Weston, 2008 Report on Field Activities for the Fiskenaesset Ruby Project, Greenland 2008)
- “NI 43-101 Report of Activities for the Fiskenaesset Ruby Project, West Greenland” prepared by Greg Davison, M.Sc., P.Geo., VP Exploration for True North Gems Inc. (Davison 2008)
- “The Aappaluttoq and Sarfaq Ruby Mineralization: A Review of the 2007/2008 Drilling Programs and Geological Interpretation” prepared by Iain Groves of Insight Geology Pty. Ltd. and Catherine Banfield of True North Gems Inc. (Groves and Banfield 2009)

- Draft Social Impact Assessment (SIA) submitted to the Greenland government June 6th 2011 (True North Gems 2011)
- Draft EIA Social Impact Assessment (SIA) submitted to the Greenland government June 6th 2011 (True North Gems 2011)
- REP0001 Infrastructure Facilities Report by MT Højgaard Grønland ApS/MT Højgaard A/S in support to the Exploitation Permit Application (MT Højgaard Grønland ApS/MT Højgaard A/S 2011)
- REP0002 Logistics and Execution Plan Report by MT Højgaard Grønland ApS/MT Højgaard A/S in support to the Exploitation Permit Application (MT Højgaard Grønland ApS/MT Højgaard A/S 2011)
- REP0006 Closure Plan Report by MT Højgaard Grønland ApS/MT Højgaard A/S in support to the Exploitation Permit Application (MT Højgaard Grønland ApS/MT Højgaard A/S n.d.)
- REP0007 Capital Cost Estimate Report by MT Højgaard Grønland ApS/MT Højgaard A/S in support to the Exploitation Permit Application (MT Højgaard Grønland ApS/MT Højgaard A/S n.d.)
- Initial Assessment of Acid Rock Drainage and Metal Leaching Potential Report by Dr. Rick Lawrence, P.Eng., and Brian Soregaroli, M.Sc. (Lawrence, P.Eng and Soregaroli n.d.)

4 PROPERTY LOCATION AND DESCRIPTION

4.1 Regional Location

The Aappaluttoq ruby project is located in southwest Greenland, approximately 150 km south of the capital Nuuk and 20 km southeast of the town of Qeqertarsuatsiaat in the Fiskenæsset mining district. The town of Qeqertarsuatsiaat is home to approximately 240 people and has an all-weather commercial harbour. The town is the main base of operations for the Company. The property is located near the intersection of 63° 00' North latitude and 50° 19' West longitude.

The 3,600 km^2 Fiskenæsset mining district is located on the southwest coast of Greenland. The district measures approximately 60 km × 60 km, and covers portions of three contiguous regional 1:100,000 map sheets: (1) Bjornesund 62 V. 1 Nord; (2) Graedefjord 63 V. 1 Syd; and, (3) Sinarssuk 63 V. 2 Syd. Aappaluttoq is one of several ruby and sapphire prospects identified on the Company's exploration licenses, however this report only addresses the Aappaluttoq prospect and does not include or consider any other prospects.

4.2 Property Agreements and Legislation

4.2.1 Royalties and Taxes

Greenland has an effective income tax rate of 30% for companies with a license under the Minerals Act. All dividend distributions are subject to 37% withholding tax. As the dividends declared are deductible against taxable income, the effective rate of tax will not exceed the withholding tax rate at 37%.

Exploitation (mining) licenses may only be held by Greenlandic registered corporations. Exploration and feasibility expenditures incurred by companies located outside Greenland may be transferred to Greenlandic subsidiaries at the time the exploitation (mining) license is granted.

These expenditures are treated as tax losses and are available immediately upon transfer.

After construction, plant and equipment are deductible through depreciation at a rate of 30% on a declining balance basis while buildings are deductible at a 20 year fixed rate.

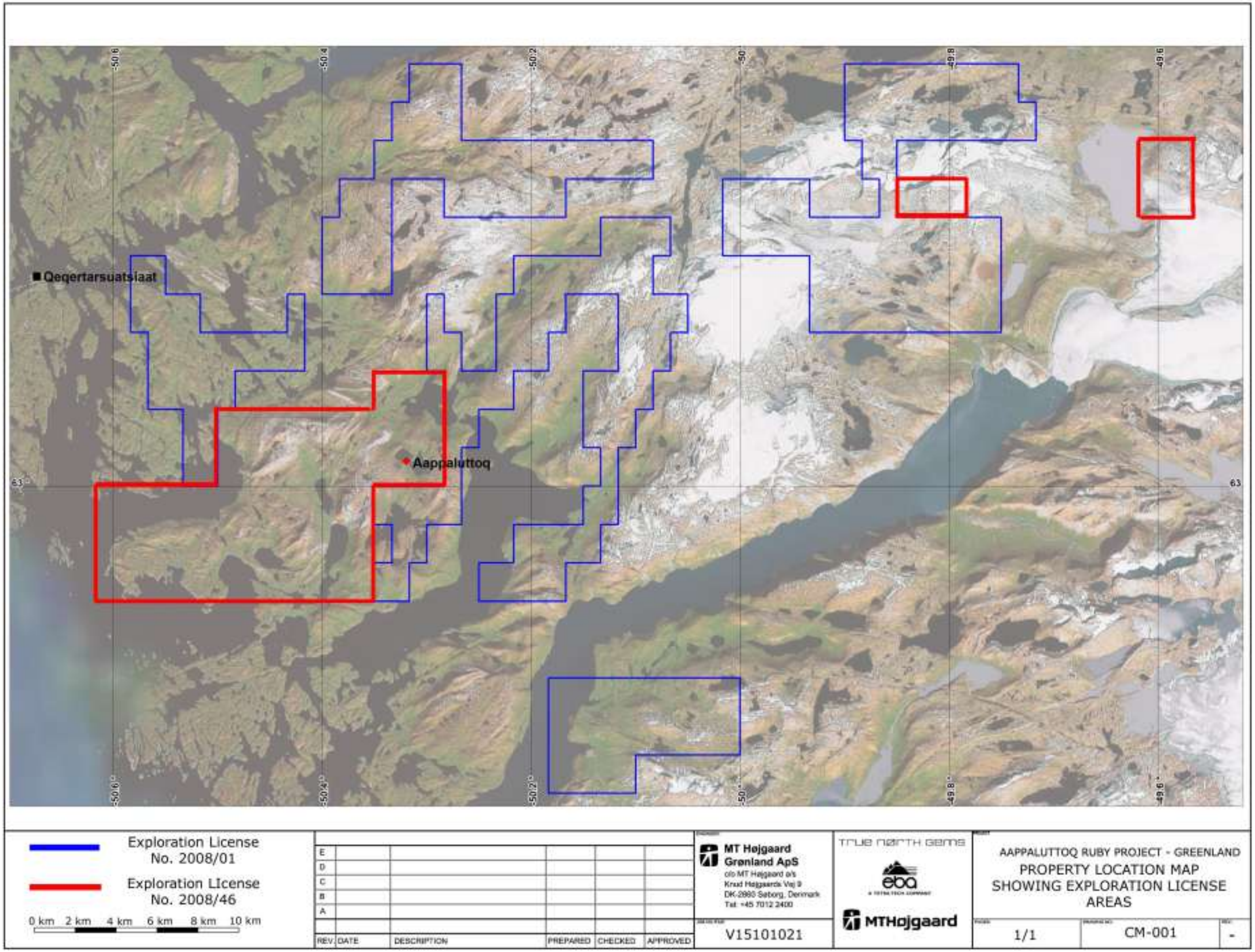
The Greenland fiscal regime also allows additional depreciation limited to 50% of the taxable income in a given year; if applied, this additional depreciation will be the period over which the expenditure can be deducted.

Taxation is further discussed in section 19.25.4 Corporate Tax.

4.2.2 Surface and Property Rights

The Aappaluttoq deposit is located on Exploration License (No. 2008/46 Fiskenæsset), and a portion of the project area is located on Exploration License No. 2008/01, which together covers a total area of 444 km^2. Both Exploration Licenses are registered with Bureau of Minerals and Petroleum (BMP) in the name of the Company (Figure 1). The exploration license expires 31st December 2012.

Figure 1: Property Location Map showing Exploration License areas



The Company's exploration license covers exclusive exploration rights for all mineral resources with the exception of hydrocarbons, oil and gas and radioactive elements.

Licenses are granted for a five year term, and are renewable at the end of the term for another five years provided the terms of the license have been met. After year 10, the license is renewable for three more two-year terms, also provided the terms of the license have been met.

Licenses are subject to application fees DKK 5,000 (CAD 958), granting fees of DKK 36,400 (CAD 6,980), and minimum expenditure requirements. Expenditure requirements include a fixed fee per license, as well as an amount dependent on the size of the license (Table 9). Fees are adjusted annually on the basis of the change of the Danish Consumer Price Index.

Table 9: Expenditure Requirement For Exploration Licenses (2011 Rates)

Amount per license per calendar year:	
Years 1-2	145,600 DKK (CAD 26,354)
Years 3-5	291,200 DKK (CAD 52,707)
Years 6-10	582,500 DKK (CAD 105,433)
Amount per km² per calendar year:	
Years 1-2	1,460 DKK (CAD 264)
Years 3-5	7,280 DKK (CAD 1,318)
Years 6-10	14,600 DKK (CAD 2,643)

Note:
Exploration commitments for years beyond year 10 are specified in the terms of the license.
License 2008/46 is currently in Year 9 and License 2008/01 is in year 4.

Licence 2008/46 was obtained by the Company by satisfying all terms of an option agreement with Brereton Engineering and Developments Ltd. Ongoing commitments from the option agreement include cash payments of CAD 50,000 and CAD 50,000 worth of Company treasury shares annually for each year the Company maintains the exploration licence. Once an exploitation (mining) licence has been obtained, a one-time payment of CAD 500,000 and CAD 500,000 worth of Company treasury shares will be due and all annual payments will cease. There is no royalty payable to Brereton. License 2008/01 was acquired directly by the Company.

The exploitation (mining) permit if granted will be in the name of Kitaa Ruby S.A., a Greenlandic corporation which is a 100% owned subsidiary of the Company.

Surrounding licences are discussed further in section 15 Adjacent Properties.

4.2.3 Environmental Regulations

The Greenlandic Mineral Resources Act of 7 December 2009 (The Mineral Resource Act 2009) forms the main relevant legislation when requiring a permit for exploitation of minerals. The obligations regarding EIA are outlined in part 15: "A license for and approval of [exploitation of minerals] can be granted only when an assessment has been made of the impact on the environment (EIA) of the performance of the activity and a report thereon (EIA report) has been approved by the Greenland Government". A draft EIA was filed with the Greenland Government on June 6, 2011.

The EIA must cover the exploitation period from mine development prior to the mine start until closure of the mine and a subsequent monitoring period. Baseline studies must be performed in the pre-mining phase because the state of the environment must be determined prior to a possible impact from the mining activities. Baseline studies must cover a period of some years before construction starts, so that the environmental variations are incorporated in the baseline description. The number of years needed for baseline studies will depend on the project and the site. The Company has completed baseline studies on the Aappaluttoq property and Lake Ukkaata Qaava.

Currently, the Company has met and maintained all environmental requirements for its exploration licenses (The Mineral Resource Act 2009).

The Company has either completed or is currently undertaking the required studies to produce the EIA for exploitation permitting. Rambøll, a Denmark based company, has been contracted by the Company to complete this work. Rambøll has close to 10,000 employees and has worked on projects in over 75 countries.

Further details on environmental regulations and practices are detailed in section 19.26 Environmental and Permitting.

4.2.4 Socio-Economic Impact Assessment

A Socio-Economic Impact Assessment (SIA) must be submitted with the application for an exploitation license. The main objectives of the SIA process for mineral projects in Greenland are:

- To engage all relevant stakeholders in consultations and public hearings;
- To provide a detailed description and analysis of the social pre-project baseline situation as a basis for development planning, mitigation and future monitoring;
- To provide an assessment based on collected baseline data to identify both positive and negative social impacts at both the local and national level;
- To optimize positive impacts and mitigate negative impacts from the mining activities throughout the project lifetime;
- To develop a Benefit and Impact Plan for implementation of the Impact Benefit Agreement.

A draft SIA was filed with the Greenland Government on June 6, 2011, with work ongoing including public consultations. The SIA covers the construction stage, the operational stage and closure of the proposed mine. The Draft SIA describes the socioeconomic baseline in Greenland and in the main affected areas, evaluates likely socioeconomic impacts related to the project and identifies measures to mitigate negative impacts requiring mitigation. The social impacts have been assessed at two different levels: potential local impacts in Qeqertarsuatsiaat and impacts at regional and national level (Nuuk, Sermersooq Municipality and at national level) to ensure a socially sustainable development of the mine.

Further details on socio-economic impacts are detailed in section 19.26.5.

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

5.1 Accessibility, Local Resources and Infrastructure

The nearby fishing village of Qeqertarsuatsiaat, population 240, lies at the northwest corner of the mining district has no air strip but the full-service commercial harbour is ice-free year round. The Irminger Current, a northern tongue of the warm Gulf Stream, is responsible for moderating the weather and keeping the sea lanes open. Regularly scheduled boat and helicopter service routinely move people, mail, and major supplies along the southwest coast of Greenland, including stops at the village of Qeqertarsuatsiaat. The government of Greenland maintains a medical clinic and an extended care treatment facility for the elderly at Qeqertarsuatsiaat.

The access to the area from the sea is through the fjords Tasiussassuaq and Tasiussaa. There are two possible entrances to Tasiussassuaq from the open sea and both are through narrow "gates" with limited depth. The tidal current is strong and passage has to be adjusted to the tide. The tidal range is about 3 m.

The passage from Tasiussassuaq to the brackish Tasiussaa is through another narrow gate, "the inner gate", about 50 m wide and 5 m deep at low tide and the current is very strong during most of the tidal cycle.

The yearly shipping window, in which it is possible to sail to the site from Nuuk, is expected to take place from beginning of June to December, depending on the ice conditions for the given year. Shipping of equipment and materials to the site has in the past been done by a small tug boat with barge. Outside this period the primary access to the area would be by helicopter from Nuuk.

A temporary camp is already on site. This camp has been used in connection with previous exploration drilling and is located approximately 2 km north of the Aappaluttoq prospect and consists of 18 tents in all:

- 14 accommodation tents
- One office tent
- One kitchen/canteen tent
- One toilet/shower tent
- One core shack tent

The present camp can accommodate up to 28 persons if tents are shared in double occupancy. There are no permanent structures on the site.

5.2 Climate and Physiography

The climate of Qeqertarsuatsiaat is comparable to the climate of Nuuk, both are classified as: “Low Arctic Maritime”. Average monthly temperatures do not exceed 10° Celsius and summer nights are typically frost-free for 60 days a year. Winters lows are typically -8° to -11° Celsius. At Summer Solstice, there are nearly 20 hours of daylight. Precipitation averages around 770 mm per year, mixed rain and snow, with most of the accumulation falling between August and November. Conditions along the coast are often cloudy, with fog to the waterline, especially in the summer months. Conditions are typically dryer and relatively warmer proceeding inland up the fjords, but inland under clear conditions extremely strong katabatic winds can episodically flow off the icecap (Piteraq wind) or descend off of mountain massifs (Foehn wind). Permafrost is discontinuous e.g. in the shade on north facing slopes. Table 10 provides a summary of the climate data available from Nuuk, which is comparable to that of the project site. The nearest meteorological stations are in Nuuk and in Paamiut 120 km to the south of the project area.

Table 10: Climate Data For Nuuk

Month	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Year
Average high (°C)	−4.6	−4.7	−5.1	−1.2	3.1	7	9.9	9.3	6	1.4	−1.3	−3.5	1.4
Average low (°C)	−10.0	−10.7	−10.7	−6.3	−1.7	1.1	3.5	3.5	1.4	−2.7	−5.9	−8.6	−3.9
Precipitation (mm)	40	47	49	47	55	62	87	85	89	66	73	74	774
Avg. precipitation days (≥ 1.0 mm)	9	9	10	9	9	8	10	9	12	10	11	10	116
Sunshine hours	31	84	186	240	186	150	186	124	90	62	30	0	1,369

Note:
(BBC World News 2011) (World Meteorological Organization 2011)

Topographic relief in the Fiskenæsset district ranges from sea level to 1,440 m in elevation. The Peak Summit is “Qaqqatsiaq” (The Big One). Maximum historic tidal variations in the inner harbour of Qeqertarsuatsiaat are less than three metres. The area is host to subarctic vegetation, with till-covered areas blanketed by grasses and ground shrubs (crowberry, birch willow and fen) and flowering plants. No rare or protected plants have been found in the area during baseline studies.

The Aappaluttoq prospect is located on a promontory extending into a lake. The lake has no official or local name, but local people have proposed the name Lake Ukkaata Qaava (meaning "the lake behind the crest") in connection with the present project. The landscape around the lake rises up to about 600 m above mean sea level (amsl), and has been scoured and eroded by glaciers. The slopes are steep and the long eastern shore of the eastern basin and the western and southern shore of the western basins are with 2-300 m high scree. The lake and valley are well hidden from sight from most directions by the steep terrain. The only opening is to the north along the outlet from the lake but the height prevents sight from the fjord and camp area or from any other natural viewpoints.

The lake is split in two main basins by two peninsulas connected by a shallow sill. The prospect is located on the peninsula jutting from the southern shore of the lake.

The lake has a surface area of 0.97 km^2 and a topographic catchment area of 6.2 km^2. Both of the lake basins have maximum depths of more than 50 m. The volume of the lake can be calculated to 19.6 M.m^3 based on preliminary bathymetry.

Wildlife in the region includes migratory birds and a few mammals. Birds encountered within the project area include: Northern wheatear, snow bunting, redpoll, raven, and ptarmigan. No waterbirds in connection with Lake Ukkaata Qaava were observed. Only three mammals are encountered in the region: arctic fox, arctic hare, and caribou.

Photograph 1 shows the local terrain at the project site. The immediate area is the location of the open pit mine. The photograph was taken in early November 2010 and ice build-up can be seen on the shallow portion of the lake, some 20 m in length.

Photograph 1: Aappaluttoq Project Site Showing Local Terrain

6 HISTORY

In 1966, ruby was discovered in outcrop on Ruby Island by the Geological Survey of Greenland and Denmark (GEUS), and that season a total of six ruby locations were discovered in the district.

From 1970 through 1982, reconnaissance exploration for ruby prospects was carried out in the Fiskenæsset area. A magnetic total field aeromagnetic survey over southern west Greenland was completed in 1998 by the GEUS. The geology of the ruby prospects in the Fiskenæsset area was first summarized by Dr. Peter Appel in GEUS Open File 95/11, working on behalf of the Geological Survey of Greenland and Denmark in 1995.

The Company optioned the property in 2004. The discovery of the Aappaluttoq ruby prospect occurred in August 2005. The Company field crews located the ruby prospect along the southern shore of Lake Ukkaata Qaava, about 3.5 km south and along strike from Ruby Island. The discovery was confirmed and validated by collecting a 100 kg sample of outcrop and mantle talus. From 2005 to 2008, the Company undertook a significant amount of exploration work on the prospect, including sample collection, drilling, prospecting, mapping and ground geophysics surveys. In 2009, Iain Groves of Insight Geology Pty Ltd. mapped the local geology and reported on it in the document prepared for the Company titled "The Aappaluttoq and Sarfaq Ruby Mineralization" and dated September 2009 (Groves and Banfield 2009).

A summary of the exploration history is presented in Table 11.

Table 11: History Of The Aappaluttoq Prospect

Year	Activity
2004	The Company obtains the Exploration License (No. 2008/46 Fiskenæsset) registered with Bureau of Minerals and Petroleum (BMP) in the name of True North Gems Inc.
2005	Discovery of the Aappaluttoq prospect
2006	30 tonne bulk sample collected from Aappaluttoq
2007	Three bulk samples ranging from 0.7 to 25 tonnes collected to confirm gemmological criteria, grade distribution and distribution data from the 2006 sample collection
	Diamond drilling in 46 holes totalling 4622.1 m to delineate and identify mineralization
2008	One bulk sample of 125 tonnes was collected to confirm grade distribution and distribution data from previous sampling.
	Diamond drilling in 19 holes totalling 1834.7 m to delineate known ruby mineralization at Aappaluttoq including the Aappaluttoq Deep Zone.

Final sorting of the samples was completed under the supervision of trained gemmologists at the Company headquarters in Vancouver, Canada.

A detailed discussion of the drilling and associated sampling history that occurred on the Aappaluttoq prospect in 2007 and 2008 is presented in section 11.

7 GEOLOGICAL SETTING

7.1 Regional Geology

The regional geology of the North Atlantic region involves a long and complicated tectonic history. The Fiskenæsset Igneous Complex is located within a poorly understood region of south-western Greenland in an area where the Archean cratons are unaffected by Paleoproterozoic metamorphism and deformation.

The Archean craton of SW Greenland is composed of six Mesoarchean to Neoarchean (ca. 3000-2720 Ma) crustal blocks that display similar structural architecture (Windley and A.A 2009). From South to North these include the Ivittut, Kvanefjord, Bjørnesund, Sermilik, Fiskefjord, and Maniitsoq blocks. The lower region of these blocks was metamorphosed at granulite facies, before being partially retrogressed to amphibolite facies assemblages. The upper region of the blocks was subjected to prograde amphibolite facies metamorphism, which never reached granulite facies. The crustal blocks consist primarily of orthogneisses derived from tonalite-trondhjemite-granodiorite (TTG) protoliths; they contain numerous layers of anorthosite and amphibolite (Friend 2009); (Steenfelt, Garde and Moyen 2005) that are themselves, highly-deformed.

The Fiskenæsset area is situated near the center of the preserved Archean gneiss complex of southern Greenland Figure 2. The area has been well mapped by the Geological Survey of Denmark and Greenland (GEUS) at a scale of 1:20,000. Pidgeon & Kalsbeek (Pidgeon and Kalsbeek 1978) noted that a detailed picture of igneous, metamorphic and structural events has been established for various subareas within the Fiskenæsset area, however the geology of the area as a whole is very complicated and the correlation of geological interpretations of the various subareas is incomplete. This appears to be the case even today, although recent interpretations (Polat, et al. 2010); and others) have assisted in providing insight into the understanding of the regional geology.

The Fiskenæsset anorthosite complex is a layered-cumulate igneous intrusion that occurs in the lower zone of the Bjørnesund structural block. It appears as a sheet-like body concordant with the adjacent orthogneisses and amphibolites (J. Myers 1985). The complex is mid-Archean (2.86Ga) in age, and individual layers in the complex range in width from 2 km to less than a meter. The km-scale fold interference patterns observed are a result of three phases of isoclinal to tight folding (J. Myers 1975). A general overview of the tectonic history of the Fiskenæsset Complex is presented in Figure 3.

Figure 2: Geological Sketch of the Fiskenæsset Area (Pidgeon and Kalsbeek 1978)

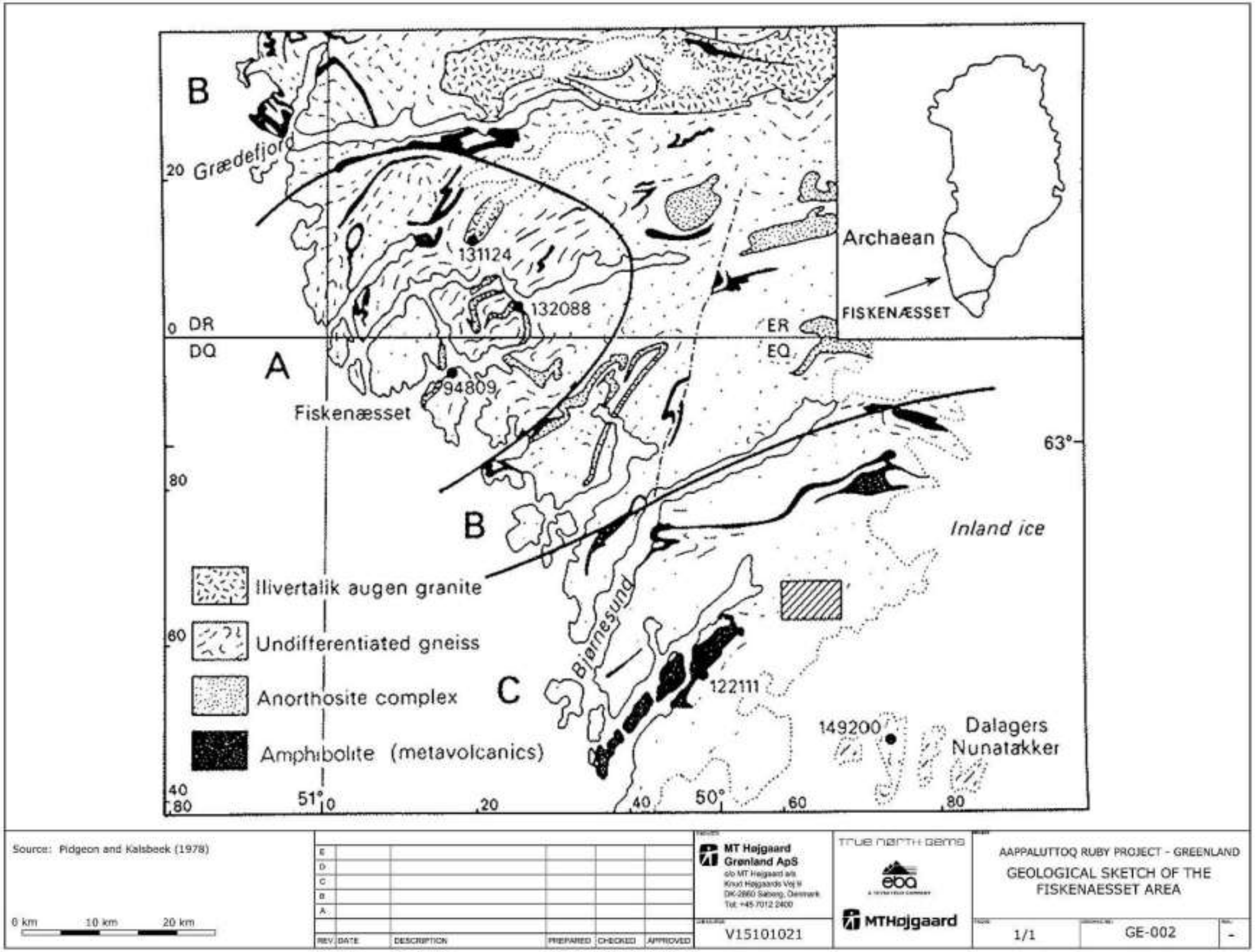
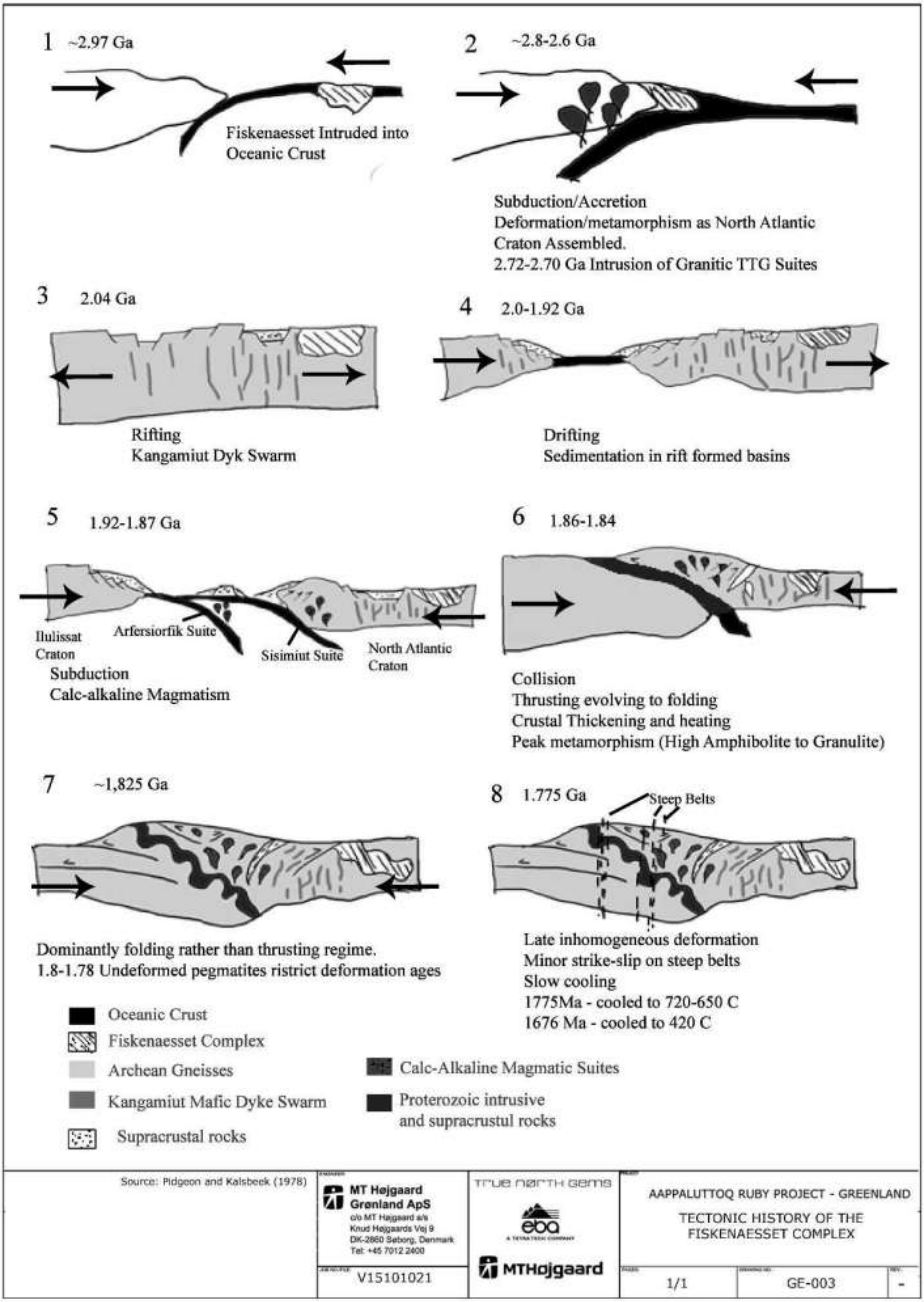


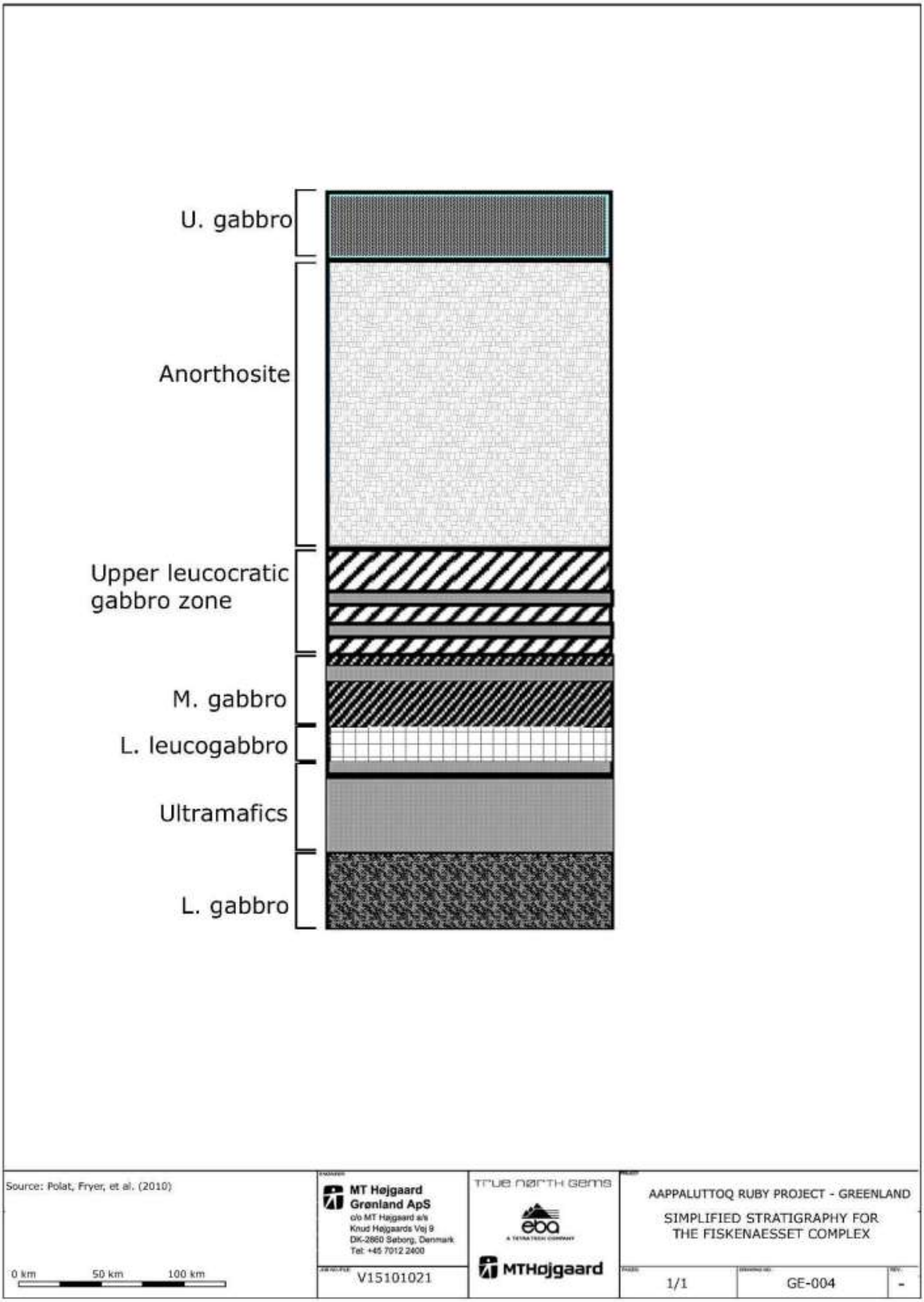
Figure 3: Tectonic History of the Fiskenæsset Complex



The as yet uncharacterized source magma of the Fiskenæsset Complex was metasomatized by slab-derived highly aluminous hydrous melts in the upper mantle/lower crust, creating a hybrid magma rich in aluminum (Polat, et al. 2010). This was then emplaced as multiple sills of semi-molten material crystal mush into the overlying tholeiitic basalt and gabbroic oceanic crust. Geochemical data suggests that the emplacement was at relatively shallow crustal level (Windley and Smith 1974). All rock types, regardless of composition, are characterized by negative Nb anomalies which is consistent with a supra-subduction zone geodynamic setting. The basal amphibolite (remains of the Archean oceanic crust) is now observable as inclusions within the gneiss and as discrete highly deformed belts. This unit is the host of the meta-anorthosites and associated rocks of the main Fiskenaesset Igneous complex. The exact emplacement of the complex is difficult to characterize due to overprinting by later deformation and metamorphic events. However, (Ashwal, et al. 1989) suggested that the emplacement of the complex, subsequent intrusion of TTGs, and high grade metamorphism took place within 70 million years based on a five-point Sm-Nd isochron age of 2860±50 Ma.

The intrusive suite comprises gabbro, ultramafic rocks, leucogabbro, and calcic anorthosite. The anorthosite and various leucogabbros are the dominant rock types present. The complex itself has a thickness of approximately 550 m (J. Myers 1985), and a simplified stratigraphy is provided in Figure 4.

Figure 4: Simplified Stratigraphy for the Fiskenæsset Complex (Polat, et al. 2010)



The cumulate layering within the intrusive complex is generally zoned upwards from mafic to calcic units. The leucogabbros grade into anorthosites with increasing plagioclase content. Seven individual map units are recognized at regional scale. Myers (J. Myers 1985) established a detailed stratigraphy of the complex; from the base moving stratigraphically higher:

1. The Lower Gabbro Unit. A standard gabbro, often strongly deformed and preserved in up to 50 m thick layers;

2. Ultramafic Unit. Comprises graded dunite, peridotite and hornblendite with a total thickness of about 40 m; the unit is usually highly deformed.

3. Lower Leucogabbro Unit. This consists of leucogabbro, gabbro and minor ultramafic layers with a total thickness of 50 m. The lower subunit contains layers rich in spinel and magnetite.

4. Middle Gabbro Unit. This unit is about 40 m thick, comprising gabbro, anorthosite and ultramafic layers. The base of the unit is marked by up to 6 m of hornblende-orthopyroxene-spinel bearing rock ultramafic rock overlain by peridotite. The composition of the Middle Gabbro Unit is distinctly different from other units of the intrusion and this unit appears to represent the influx of a second batch of magma with higher MgO/FeO ratio.

5. Upper Leucogabbro Unit Comprises 60 m of rhythmically layered ultramafics and anorthosite. The cumulate plagioclase clusters are up to 20 cm in diameter. Several chromite-rich sub-layers are found in the anorthosite and in the peridotite layers.

6. Anorthosite Unit. This unit forms a significant interval in the igneous complex –and is about 250 m thick. It shows weak internal stratigraphy with no prominent layering. Cumulate layers can form up to 2 m thick; sub-layers with up to 80% cumulus plagioclase are frequent, as is the presence of chromite bands.

7. Upper Gabbro Unit. This unit is up to 50 m thick consisting of gabbro with layers of peridotite. Where the rocks are strongly deformed, garnet is generally prominent.

The entire area has been subjected to at least one episode of late Archean, granulite facies and retrograde amphibolite facies metamorphism – creating at least 3 stages of regional folding (Henriksen, et al. 2000). Myers (J. Myers 1985) noted that, despite the folding and metamorphism, some parts of the Fiskenæsset complex locally retain their igneous stratigraphy, cumulate textures, mineral grading, layering, channel deposits and cross cutting relationships. The recrystallization of the rock units is directly tied to metamorphic deformation, with igneous minerals best preserved in zones of lowest strain. In the Fiskenæsset Complex, some zones suffered post-tectonic metasomatic alteration and resetting ((Pidgeon and Kalsbeek 1978) and (J. Myers 1985)).

The Fiskenæsset complex was intruded by several generations of quartzo-feldspathic material; these intrusions are widespread in areas of low deformation, they are observed as veins of granitoid composition crosscutting the main mafic and ultramafic layers. Following granitoid emplacement, the complete Fiskenæsset Complex, associated sub-parallel layers of amphibolite and granitoid gneiss were then folded during three major episodes of ductile deformation - F1, F2, F3 (Kalsbeek and Myers 1973). Each deformation event contributed a complex sequence of fold and fabric development, creating complex interference patterns.

The F1 event, folded the sequence into large recumbent isoclinal, nappe-like structures. Subsequently, these structures were refolded (F2 and F3) into folds with steep axial surfaces at high angles to each other. The F2 folds are tight and isoclinal, and imparted the main east-west tectonic grain observed across the region. They are also associated with the development of a strong planar cleavage fabric. The F3 folds have vertical axial surfaces that trend northwest – southeast and plunge to the southeast (J. Myers 1985). F3 folds were contemporaneous with the early stages of the last major metamorphic/metasomatic episode (at amphibolite facies) which led to the alteration of mineral assemblages throughout the region – see Mineralization section, this report, for further details. Zircon ages of 2660 ±20Ma give a maximum date for the end of amphibolite facies metamorphism.

Subsequent to all deformation and metamorphic events, the Fiskenæsset complex was cut by undeformed and unaltered dolerite and tonalitic dykes of early Proterozoic age (J. Myers 1985) and by numerous late stage pegmatities (Pidgeon and Kalsbeek 1978).

Weathering and quaternary ice-sheet development have eroded and exposed the core of the complex, however, it remains one of the best preserved Archean layered mafic igneous complexes in the World.

7.2 Local and Property Geology

Aappaluttoq lies on a small NNE trending peninsula extending into Lake Ukkaata Qaava, which is a lake 50 – 60 m deep. There is significant outcrop, with about 40% of the Aappaluttoq prospect exposed at surface and 60% covered by the lake.

Mapping shows the geology of Aappaluttoq is confined to the lower part of the stratigraphy of the Fiskenæsset complex, namely the Lower Gabbro, Ultramafic, Lower Leucogabbro, and Middle Gabbro sequences, and it is noted that there is a total lack of anorthosite and chromite bodies within this sequence. Table 12 provides a list of lithologies identified on the property. The rock codes provided are used throughout cross sections and resource models discussed in this report.

Table 12: Property Geology

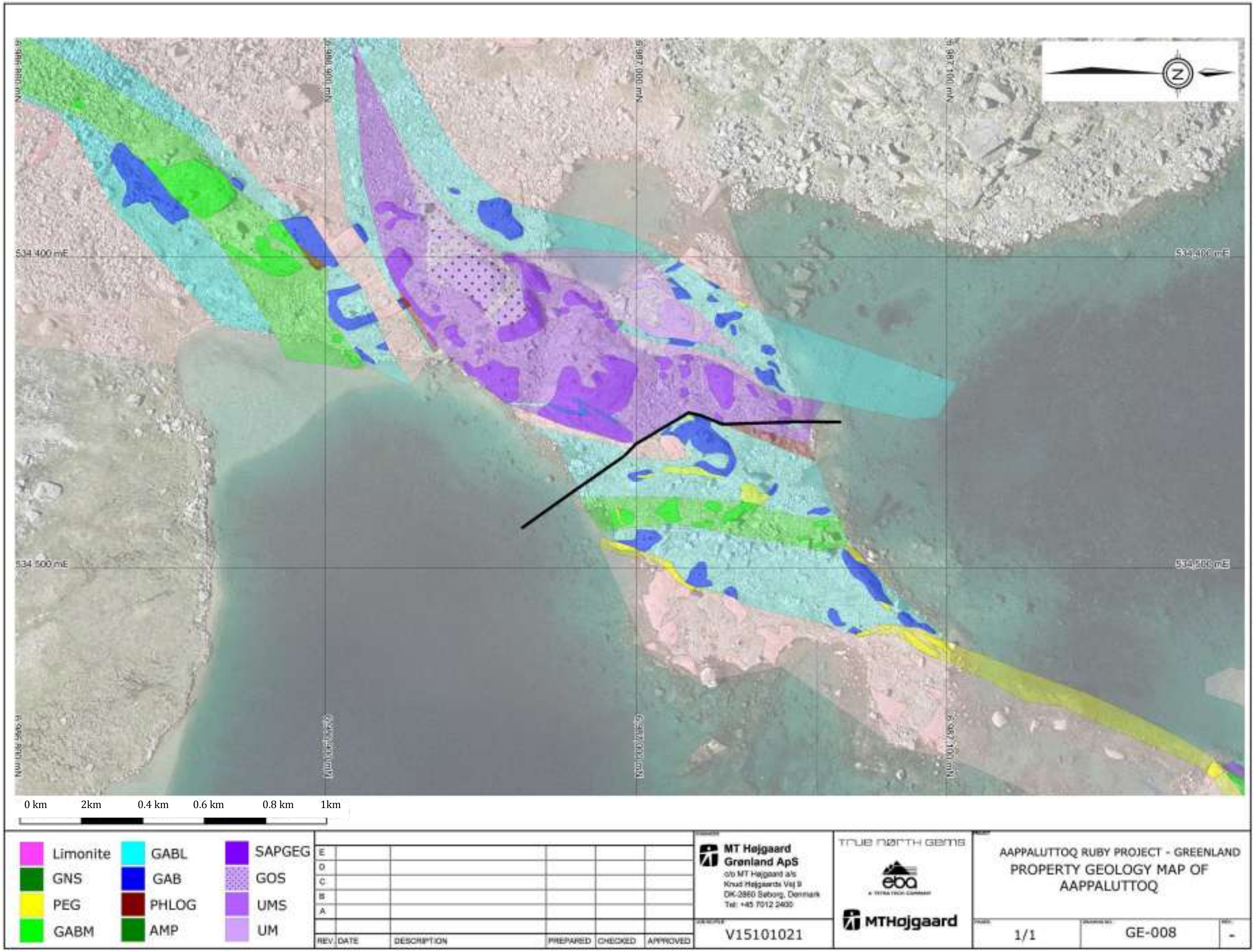
Lithology Type	Rock Code
Phlogopite	PHLOG
Gabbro (Leucocratic and Mafic)	GAB
Sapphirine Gedrite	SAPGED
Pegmatite	PEG
Gneiss, Augen Gneiss	GNS
Ultramafic	UM

The geology of the Aappaluttoq area is dominated by an intrusive gabbro to leucogabbro sequence of rocks with significant volumes of ultramafic rock. This sequence is intruded into and is structurally juxtaposed against the felsic gneiss basement suite. The rocks encountered at Aappaluttoq differ from basement amphibolites (meta-volcanic rocks) in a number of ways; they are less homogenous, do not show a strong cleavage, and they are distinctively granular. The amphibolites are geometrically unrelated to the gabbro intrusive body. The Aappaluttoq ultramafic body is internally zoned, with a barren ultramafic core (olivine and lesser pyroxene). It is lensoidal in shape and has a minimum strike length of 170 m and is up to 70 m thick. Gradational alteration is prevalent and evident where ultramafic rocks have been altered to phlogopite. It is in these metasomatic/metamorphic reaction zones between the leucogabbro and ultramafic stratigraphy where the ruby mineralization is mostly concentrated. Pegmatite dykes are prevalent throughout the property area. (Groves and Banfield 2009)

Structurally, the Aappaluttoq area is very complex with assimilation of the intrusive complex into the gneiss during partial melting, and through fragmentation of the intrusive complex by a series of regional folding events. The stratigraphy observed in the Aappaluttoq area generally trends northeast and dips at 50 to 60° to the northwest. The units dip steeply in the northern part of the area. (Groves and Banfield 2009)

The property geology is presented in Figure 5.

Figure 5: Property Geology Map of Aappaluttoq



8 DEPOSIT TYPE

The geology of primary gem corundum deposits is poorly understood. This is due in part to their geography – the 'classic' localities of Kashmir, Burma and Sri Lanka have been under tight political control for many years, and access to the gem sites by modern scientists is difficult. Another major factor is the secrecy surrounding the geology of these deposits – owners of the localities are often private companies, who do not want the geology to be publically accessible. As a result, much of it remains confidential (Malik 1994). The lack of access to the site, and to geological samples has driven the scientific community to focus on the gemstones themselves – their isotopic signatures (Upton 1999); (Giuliani, Fallick, et al. 2005) and their gemmological characteristics - little discussion on the formation processes or the geological host rocks has occurred until very recently.

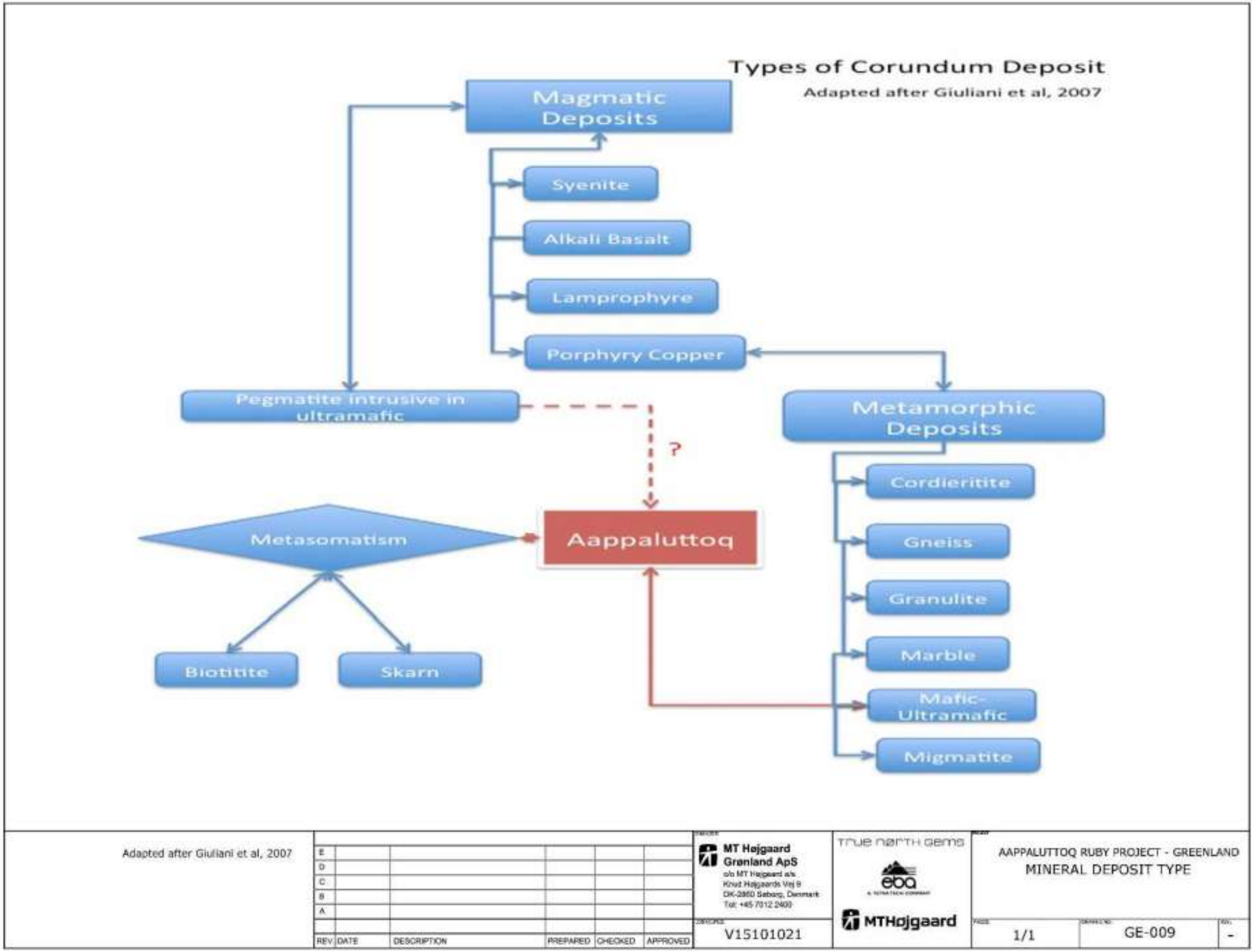
Gem corundum deposits are linked to collisional, rift & subduction zone settings; where fluids are available and the high pressure & temperatures required can easily be generated (Giuliani, Ohnenstetter, et al. 2007). The formation of gem corundum requires a specific set of rare geochemical circumstances, including a low silica content, a correspondingly high aluminum content, and the presence of Cr, Fe, Ti or V as chromophore elements. As such, gem corundum deposits can be formed in a wide range of rock-types including marble (Mogok, Burma – (Kammerling, et al. 1994)), calc-silicate (Beluga, Canada – (Cade, Dipple and Groat 2005)), gneiss (Mysore, India – (Schwartz 1998)),ultramafic (O'Brians, Zimbabwe – (Schreyer, Werding and Abraham 1981)), syenite (Ilmeny-Urals, Russia – (Keivlenko 2003)) desilicated pegmatites (Umba River, Tanzania – (Solesbury 1967)), lamprophyres (Yogo Gulch, USA – (Mychaluk 1995)), basalt (Oberon, Australia – (Sutherland 1996)), granitic skarns (Andranondambo, Madagascar – (Schwartz, Petsch and Kanis 1996)), and others (Keivlenko 2003).

There are numerous classification schemes for gem deposits; but because they are so poorly understood, genetically linking one deposit to another is challenging – this has led to a fairly complex set of overlapping classification criteria. Deposits can be classed by the corundum morphology (Ozerov 1945), geological context (Hughes 1997), lithology of the host-rock (Schwartz 1998), the tectonic process required to form the deposit (Simonet 2000) and others (see (Giuliani, Ohnenstetter, et al. 2007)).

The Company prefers to classify their gem corundum deposits using the methods outlined by Kievlenko in 2003 and Giuliani et al, 2007 (Figure 6). This scheme uses the genetic type of the deposit to classify and sub-classify. This gives several basic categories – metamorphic, metasomatic, igneous, and sedimentary; and then sub-classifies these further depending on the rock-types involved. This simplified geological model does not account for the possibility of multiple formation methods or the interaction of several genetically linked methods. For example, regional metamorphism can lead to localised metasomatism – thus two areas of the classification scheme affect the gem corundum.

Further scientific study needs to occur to better classify and understand the complex and the unique processes involved in the development of gem corundum deposits. This is especially true in regards to retrogressive metamorphism and its effect on the development or 'upgrading' (L. Groat 2011). Figure 6 shows the classification scheme of Guiliani, 2007 – this corresponds with the earlier Kievlenko scheme and is included for illustration purposes. The classification of the Aappaluttoq deposit is shown for reference.

Figure 6: A Simplified Classification Scheme for Hard-rock Corundum Deposits.



9 MINERALIZATION

The Aappaluttoq deposit represents one of the few potentially economic examples of metasomatic ruby formation in a mafic or ultramafic host. The classification of (Keivlenko 2003) and (Giuliani, Ohnenstetter, et al. 2007) – see Deposit Types section of this report – shows the mineralization at Aappaluttoq has similarities to the Rai-Iz massif deposits in the Ural mountains of Russia.

As described in detail elsewhere (Property Geology and Regional Geology – this report), the rocks at Aappaluttoq have very different histories and evolution pathways. The main ore zone is currently comprised of three main rock-types: sapphirine-gedrite, leucocratic gabbro and a phlogopitite; the latter being the most important of these.

The sapphirine-gedrite rocks at Aappaluttoq lie 30 km from the world type-locality for the mineral sapphirine ($[Mg,Al]_8[Al,Si]_6O_{20}$) located within the village of Qetertarsuatsiat (formally known as Fiskenaesset) on the west coast of Greenland. The Fiskenaesset igneous complex contains multiple areas rich in sapphirine-gedrite, many of these were described in detail by (R. Herd 1973) (R. Herd 1969). The authors agree in principle to the established mineralization model for these rocks; metamorphism and metasomatism altered the ultramafic rock by adding silica (SIO_2) and alkalis (K_2O, Na_2O). The Company views this rock as a key indicator for potential corundum mineralization in the Fiskenaesset district, as it forms along the boundary between the ultramafic and the Al+SiO_2 enriched rocks of the gabbroic suite in the Fiskenaesset complex. However, the presence of elevated silica makes this rock an unlikely host for a major volume of free corundum. It is possible that on a local scale, where the metasomatic overprinting has not been as significant, there exists mineralization potential.

The leucocratic gabbro is one of the main rock-forming units within the Fiskenaesset igneous complex. This unit contains a small amount of gem corundum as ruby, but a large amount of pink sapphire. The gabbro is very rich in Al, and is believed to be the unit responsible for releasing the Al to form the corundum throughout the Aappaluttoq deposit.

The phlogopitite is believed to be the host for the majority of the ruby held within Aappaluttoq, although it also holds a minor amount of pink sapphire. This unit is a metasomatic product, formed once the hydrous fluids responsible for the growth of the gem corundum began to crystallize. This rock is only stable at low pressure, low temperatures, thus must have been the last phase to crystallize during the last phases of retrogressive metamorphism. No age dates have been obtained on this unit.

The processes involved in the generation of gem corundum are complex. The presence of corundum dates back to the original building-blocks of the Fiskenaesset igneous complex around 2970Ma; subsequent metamorphic overprinting and metasomatic alteration formed and finally upgraded the deposit into what we see today. For a full history of the regional scale geology and the factors that brought the main ore rocks into being please see the regional geology and property geology section of this report. All of the previous tectonic activity and the generation of the gabbroic sequences are critically important to understanding how and where the corundum mineralisation was produced.

Once all of the current igneous complex rocks were emplaced beside one another, the growth of the corundum crystals could begin. Upper amphibolite to granulite metamorphism (as the various W Greenland continental blocks were accreted and attached) released a large volume of fluid into the igneous complex – the nature of these fluids has not been examined in any great detail, however, they would probably be made of H_2O, $\pm K_2O$ $\pm Na_2O$ $\pm CO_2/CH_4$ depending on localised pressure/temperature and fO_2 conditions. These fluids could move along the contact zones between the various rock-types in the igneous complex (ultramafic/gabbro etc.), creating a zone of alteration in the bedrock. These hot pressurised fluids would facilitate the dissolution of silica from the gabbros into the ultramafic across a large geochemical gradient. The introduction of silica and the presence of the fluid environment caused the ultramafic unit to be altered into sapphrine-gedrite. This metamorphic/metasomatic event would gradually decrease the silica saturation in the gabbro and increase it within the ultramafic; creating a gabbroic environment that is gradually becoming more and more depleted in silica. The primary regional metamorphism ended around 1825Ma, leaving the ultramafic-sapphirine/gedrite-gabbro sequence in place.

A second metamorphic event at temperatures and pressures corresponding to amphibolite facies occurred around 1860Ma-1825Ma. It is this event and the infiltration of new fluids along the previous plane of weakness (between the gabbro and ultramafic) that creates the corundum mineralization.

The source of Al to initially start the mineralization lies within the gabbroic units. It is locked up in primary calcic plagioclase (anorthite); interaction with the metasomatic fluid changes the stability of the feldspars and they are replaced by albite-oligoclase feldspar by the following reaction:

$$Na_{0.05}Ca_{0.95}Al_{1.95}Si_{2.05}0_8 + K_20 + Na_20 + H_20 = Na_{0.90} + Ca_{0.10}Al_{1.05} + Si_{2.95} + O_8 + Al + bringe$$

$$Anorthite + alkali\text{-}rich\ hydrous\ fluid = albite + AL + changed\ fluid$$

This alteration of the feldspars allows free Al to be released. As this element is incompatible, and thus is unstable being in a fluid phase it will oxidise in the hydrous fluid, creating a stable corundum crystal (Al_2O_3). Once the crystals have begun to form, they will act as nucleation centres for other free Al ions within the fluid, allowing the concentration of corundum to occur and creating larger crystals.

The ultramafic unit is rich in Cr, V, Ti and Fe (all chromophore elements that change the colour of corundum) these elements would dissolve into the fluid and leach out of the ultramafic and into the gabbro. The interaction of Cr with the newly formed and forming corundum crystals facilitates the following exchange reaction:

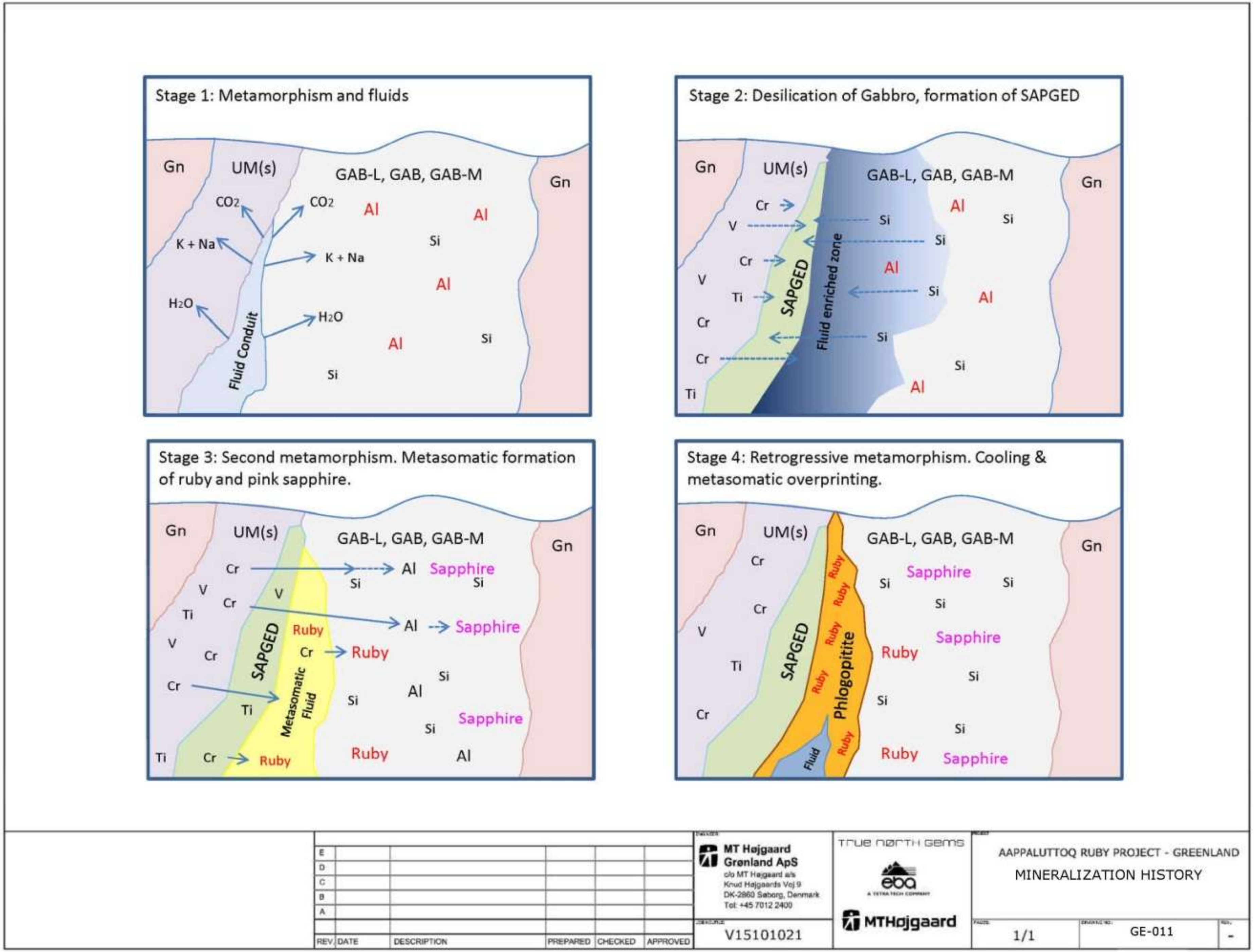
$$Cr^{3+} \rightarrow Al^{3+}$$

This reaction clearly shows the trivalent Cr is substituted into the corundum crystal structure in-place of trivalent Al; turning the crystal red. The more substitutions occur in the crystal the deeper the red colouration becomes – the only limiting factor in this reaction is the availability of Cr^{3+}, this directly corresponds to the Cr concentration in the mineralizing fluids. V and Ti can both substitute in a similar manner and will turn the corundum blue, while Fe substitution will give a yellow hue. Given the high concentration of these elements in the ultramafic unit, there is currently no explanation as to why there is no blue or yellow sapphire observed anywhere in the Aappaluttoq deposit – only red and pink mineralization. One possible explanation is the presence of the sapphrine-gedrite zone between the ultramafic source and the fluid/gabbroic sink – it is possible that all of the Fe, V & Ti were absorbed into the sapphrine unit, and only the Cr (because of its much higher concentration) managed to pass through the SapGed and enter the mineralizing fluid – thus colouring the waiting corundum crystals. The highest concentration of Cr outside the ultramafic is within the phlogopitite unit (the crystallized remains of the fluid phase) where we see a correspondingly high concentration of red ruby. The further the Cr-rich mineralizing fluid travels from the ultramafic the less Cr is held in suspension (it has already substituted into corundum crystals as it passes through the gabbro), thus we see the majority of pink sapphire in the leucocratic gabbro. The main mineralization is along the metasomatic contact boundary and we currently do not know how far corundum mineralization reaches in the leucocratic gabbro zones. Further regional work is required to assess the actual extent of mineralization within the complex.

Retrogressive metamorphism occurred between 1775-1676 Ma at greenschist to lower amphibolite facies (200-600°C, 1-5 kbar). We believe this had a significant effect on the quality and preservation of corundum crystals within the Fiskenaesset igneous complex. This phase of metamorphism was accompanied by a regional folding event – this created the fold patterns and complex geology we see today. The low temperature fluid remobilized the incompatible elements and concentrated them in zones of low pressure (fold hinges, decompressive bends etc.). This allowed any corundum present within the fold to get an influx of chromophore elements and allow them to recrystallize slowly in a fluid environment – the ideal method for creating large, transparent gemstones. The corundum left outside these low strain zones was subjected to shearing and regional metamorphic forces. Many of the corundum surfaces began altering to alumina-silicate minerals such as kyanite. However, once completely formed, this kyanite coat acted as a protective armour for the enclosed corundum crystal, preventing any of the caustic metasomatic fluids from reaching the gem corundum material in the core –thus preserving the corundum crystals. This connection between the kyanite and the corundum has been noted in numerous other corundum localities within the Fiskenaesset igneous district.

The production of corundum by multi-phase metamorphism and metasomatism has been documented elsewhere including (Raith, Rakotondrazafy and Sengupta 2008) who described a similar reaction in rocks from Madagascar. The 'armouring' effect of alumina-silicates is unusual, and has been noted in very few other corundum deposits, most notably at the Revelstoke sapphire deposit in BC, Canada (Dzikowski 2010). The main Aappaluttoq deposit contains folded units at its core – allowing for the potential generation of high quality ruby and pink sapphire crystals in the low pressure zones; it also contains corundum rimmed in kyanite and other alumina-silicates showing the higher strain zones and armouring effect. Further geological structural analysis is required to understand this phenomenon, how it relates to the local geological stress fields and how this can be used for locating future zones of high-quality high-grade corundum mineralization at Aappaluttoq, and throughout the Fiskenaesset complex. (Figure 7).

Figure 7: Mineralization History



The ruby and pink sapphires from Aappaluttoq show several distinctive features and have been analysed in detail by Professor L.A. Groat of the University of British Columbia, a recognised authority on gemstones and gemstone deposits. In his report entitled "A mineralogical and geochemical study of untreated ruby samples from Greenland" he stated that:

"The Aappaluttoq ruby samples, showed average values of 1.14 wt % Cr_2O_3 and 0.45 wt % Fe_2O_3, and one analysis shows 0.34 wt % TiO_2. The ruby sample contained a number of inclusions with irregular outlines. Most of the inclusions were filled with albite and/or hercynite-chromite spinel (sometimes containing Ni). One inclusion showed a core of spinel, approximately 60 × 10 μm in size, which was surrounded by albite feldspar. Other smaller spinel and Fe-oxide inclusions occur in the body of the ruby sample. The Aappaluttoq pink sapphire sample showed much less Cr than in the ruby sample; with microprobe calculated average values of 0.26 wt% Cr_2O_3 and 0.21 wt% Fe_2O_3. This latter sample showed one large inclusion of an Mg-Al-Ca-silicate mineral, most likely an altered amphibole or garnet, and a few smaller inclusions of Ni-bearing hercynite-chromite spinel. A number of grains of cassiterite with minor Cu were also identified." (L. Groat 2005) The Major elemental analysis of rubies and pink sapphires from Aappaluttoq is presented in Table 13.

Table 13: Major Elemental Analysis Of Ruby & Pink Sapphires From Aappaluttoq

	MgO	Al_2O_3	TiO_2	V_2O_3	Cr_2O_3	Fe_2O_3
Aappaluttoq-Red	0.01	96.60	0.34	0.02	0.77	0.59
Aappaluttoq-Red	0.00	96.33	0.01	0.00	1.34	0.40
Aappaluttoq-Red	0.01	97.32	0.02	0.00	1.33	0.36
Average	0.01	96.75	0.12	0.01	1.14	0.45
Aappaluttoq-Pink	0.01	97.79	0.01	0.01	0.21	0.22
Aappaluttoq-Pink	0.00	98.31	0.01	0.00	0.31	0.19
Aappaluttoq-Pink	0.00	97.09	0.01	0.01	0.26	0.23
Average	0.01	97.73	0.01	0.01	0.26	0.21

Note:

All samples $\phi(\rho Z)$ corrected.

10 EXPLORATION

Exploration has been ongoing at Aappaluttoq since its discovery in 2005. From 2005 until 2010 exploration included the collection of bulk samples, prospecting, surficial mapping, and exploration diamond drilling. Details of the exploration programs from 2005-2006 are described in detail in the "Report of Activities for the Fiskenaesset Ruby Project, West Greenland (Davison 2008).

The work in 2005 and 2006 included surface mapping and sampling of the prospect, and a small 30 t bulk sample was collected in 2006.

The 2007 exploration program focussed on delineation of the prospect, through diamond drilling, and expanded upon bulk sampling programs from previous years.

The 2008 exploration program continued with further diamond drilling and mapping and aimed to define thickness and lateral extent of the mineralization at the Aappaluttoq prospect. Drilling in 2008 tested an area of the mineralization zone to the north and west of previous drilling, and to the east along the projected strike of the Aappaluttoq Deep Zone. This drilling of the Aappaluttoq Deep Zone succeeded in identifying significant concentrations of mineralization and confirming the continuity of mineralization at depth.

Exploration activities were put on hold in late 2008 with the downturn in industry and the tightening of finances.

In 2009, the Company exploration at Aappaluttoq completed a review of drilling and mapping and interpretation of geology and corundum occurrences. Geologists completed re-logging of available core (from 2007 and 2008) for geological and geotechnical data. In addition, a reassessment of drill samples sent for assays and detailed 1:500 mapping occurred over the prospect to tie in new geological interpretations and track mineralization to surface from drill intercepts.

In 2010, a detailed ground magnetic survey was completed at Aappaluttoq to trace the extent of the UM unit in the subsurface.

10.1 Surface Sampling

Surface sampling at Aappaluttoq has consisted primarily of bulk sample collection. Field records for bulk sampling were found to be lacking detail with respect to sample location, characterization and collection methodology. General locations are recorded in field records and precise locations are obvious in the field but have not been surveyed

During the exploration program in 2007, a total of three bulk samples were collected at the Aappaluttoq site. The samples were numbered sequentially, B1, B2, and B3.

Sample B1 was collected from in-situ bedrock across a significant ruby and pink sapphire-bearing interval at Aappaluttoq. It weighed 25.5 t. The sample was collected to confirm grade and distribution data from the 2006 sampling.

Sample B2 was collected to provide an analysis of gemmological criteria that would occur under typical mining conditions. It weighed approximately 27.9 t and was collected from bedrock exposure of ruby and pink sapphire using focused, low intensity blasting and lies adjacent to the B1 bedrock sample.

Sample B3, was a large 741.2 kg corundum bearing boulder. It was shipped to SGS Lakefield for storage for future processing.

There was a fourth sample collected from overlying weathered bedrock in 2007 weighing 25.9 t. The sample was collected from an area directly above ruby-bearing units within the folded and altered ultramafic rocks. The area measured 5 meters by 20 meters.

In 2008, two bulk samples were collected at the Aappaluttoq site. The first was a 125 t sample collected from bedrock. The material was removed in blocks 20-40 cm or larger, along a strike length of nearly 15 m and a depth of up to 2 m. A second sample was collected from regolith overlying mineralization. The sample was approximately 30-40 t (Table 14).

Table 14: Bulk Sample Compilation - Initial Weights (Modified From Weston, 2009)

Sample	Initial Weight
	kg
2006 Aappaluttoq Bulk Sample	30,000
2007 Aappaluttoq Overburden	25,860.9
2007 Aappaluttoq Bulk Sample B1	25,455.8
B1 QC Sample	
2007 Aappaluttoq Bulk Sample B2	27,856.0
B2 QC Sample	
2007 Aappaluttoq Bulk Sample B3	741.2
2008 Aappaluttoq Bulk Sample	125,000
2008 Aappaluttoq Overburden Sample	30,000 – 40,000

10.2 Mapping

In 2006 initial mapping of the property was completed by Megan Ritchie, as part of a Master's thesis at the University of Cambridge in England.

In 2008, detailed re-mapping was conducted by Iain Groves, the Company's Senior Exploration Geologist, over a three day period at 1:500 scale using ortho-rectified aerial photos (WGS84 Zone 22N) as a basemap. Rock-type classifications were based on those defined in Windley (Windley 1971) using standard USGS classifications for gabbroic intrusive rock suites. Locational data was collected using a Garmin 60CSx map GPS. Structural data was collected manually and converted to UTM WGS 84 zone 22N to match the regional datasets. The geological mapping was digitized using MapInfo. A declination of 28° to the west was used to correct the structural measurements. Digitization and development of regional scale geologic maps is ongoing.

10.3 Geotechnical Work

Geotechnical logging, including core recoveries, RQD, fracture density and rock strength was completed during re-logging of 2007 and 2008 drill core in 2009. Sections of core were missing due to whole-core assay sampling, so complete logging of the drill-holes was not possible. In addition structural geology data noting faults and jointing in core was collected by geologists in 2007 prior to sampling and is recorded in their core logs.

11 DRILLING

Diamond drilling was completed at Aappaluttoq in 2007 and 2008. Prior to this no drilling had taken place at the site. Diamond drilling was completed using a small fly rig diamond drill and associated support equipment. All holes have been drilled as NTW thin wall core. Drilling was carried out under the direction of Kluane Drilling based out of Whitehorse, YT. Down-hole drill surveys were not completed, and collar locations were recorded using a handheld GPS unit.

The drillhole orientations are variable, however they have typically been drilled perpendicular to the known geologic trend (across the layered sequence) and reaches beneath Lake Ukkaata Qaava. Typically, the drillholes were drilled from one collar location, with same azimuth and dips of -45°, -60° and -75° to provide a fan of data in cross section.

In 2007, 46 drill holes were completed at Aappaluttoq totalling 4,622.1 m. In 2008, 19 drill holes were drilled totalling 1,834.7 m (see Table 15, Figure 8).

Visible rubies and pink sapphires were identified in drill core. A total of 634 drill core samples from the 2007 and 2008 drilling were selected to be sent to the Saskatchewan Research Council (SRC) for assay.

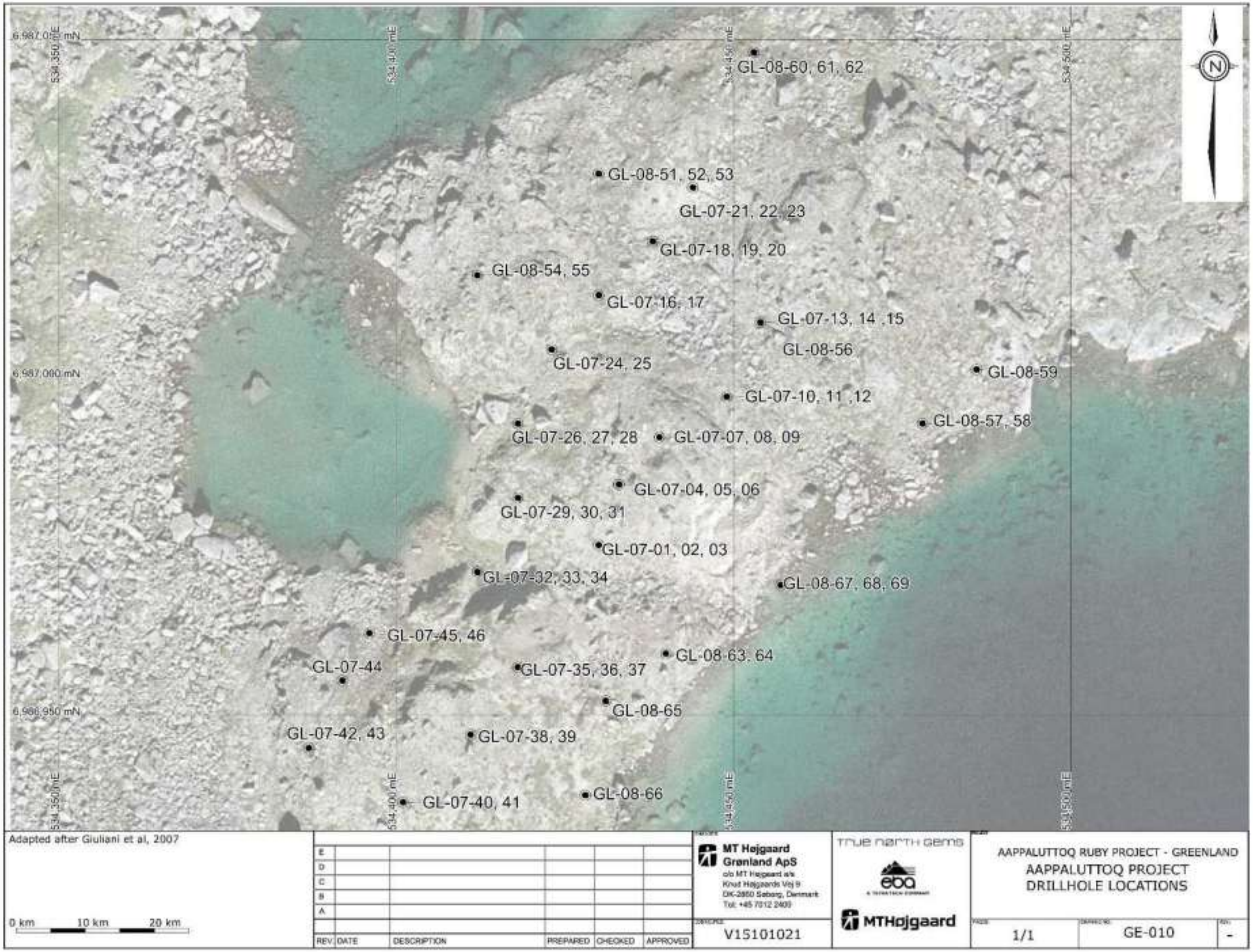
Table 15: Collar Information For 2007-2008 Diamond Drilling (From Weston, 2009)

Hole ID	Easting	Northing	Elevation	Azimuth	Dip	Depth
	m	m	m	°	°	m
GL-07-01	534430	6986975	234	135	-45	50.3
GL-07-02	534430	6986975	234	135	-60	75.6
GL-07-03	534430	6986975	234	135	-75	106.4
GL-07-04	534433	6986984	235	135	-45	53.5
GL-07-05	534433	6986984	235	135	-60	74.7
GL-07-06	534433	6986984	235	135	-72	140.2
GL-07-07	534439	6986991	237	135	-45	68.6
GL-07-08	534439	6986991	237	135	-60	88.4
GL-07-09	534439	6986991	237	135	-75	108.2
GL-07-10	534449	6986997	233	135	-45	58.5
GL-07-11	534449	6986997	233	135	-60	77.7
GL-07-12	534449	6986997	233	135	-75	101.4
GL-07-13	534454	6987008	233	135	-45	50.3
GL-07-14	534454	6987008	233	135	-60	56.3
GL-07-15	534454	6987008	233	135	-75	80.8
GL-07-16	534430	6987012	232	127	-60	108.2
GL-07-17	534430	6987012	232	127	-75	79.3
GL-07-18	534438	6987020	234	127	-60	187.5
GL-07-19	534438	6987020	234	127	-45	50.3
GL-07-20	534438	6987020	234	127	-75	230.7
GL-07-21	534444	6987028	234	127	-45	106.7
GL-07-22	534444	6987028	234	127	-60	108.3
GL-07-23	534444	6987028	234	127	-75	131.9
GL-07-24	534423	6987004	233	125	-60	111.3
GL-07-25	534423	6987004	233	125	-75	140.2
GL-07-26	534418	6986993	230	125	-60	214.9
GL-07-27	534418	6986993	230	125	-75	149.4
GL-07-28	534418	6986993	230	125	-45	50.3

Table 15: Collar Information For 2007-2008 Diamond Drilling (From Weston, 2009)

Hole ID	Easting	Northing	Elevation	Azimuth	Dip	Depth
	m	*m*	*m*	°	°	*m*
GL-07-29	534418	6986982	231	135	-60	109.8
GL-07-30	534418	6986982	231	135	-75	128.0
GL-07-31	534418	6986982	231	135	-45	40.0
GL-07-32	534412	6986971	231	143	-60	122.0
GL-07-33	534412	6986971	231	143	-75	125.0
GL-07-34	534412	6986971	231	143	-45	25.9
GL-07-35	534418	6986957	239	143	-60	155.5
GL-07-36	534418	6986957	239	143	-45	105.2
GL-07-37	534418	6986957	239	143	-75	113.4
GL-07-38	534411	6986947	238	143	-60	97.5
GL-07-39	534411	6986947	238	143	-45	84.9
GL-07-40	534401	6986937	235	119	-45	50.3
GL-07-41	534401	6986937	235	119	-60	71.6
GL-07-42	534387	6986945	234	126	-60	97.5
GL-07-43	534387	6986945	234	126	-75	47.2
GL-07-44	534392	6986955	234	129	-60	100.6
GL-07-45	534396	6986962	230	96	-60	125.0
GL-07-46	534396	6986962	230	96	-75	163.1
GL-08-51	534430	6987030	232	125	-60	50.3
GL-08-52	534430	6987030	232	0	-90	77.7
GL-08-53	534430	6987030	232	125	-75	53.4
GL-08-54	534412	6987015	233.9	125	-60	76.2
GL-08-55	534412	6987015	233.9	125	-75	108.2
GL-08-56	534454	6987008	233.9	125	-45	120.4
GL-08-57	534478	6986993	230	125	-45	62.5
GL-08-58	534478	6986993	230	125	-60	120.4
GL-08-59	534486	6987001	231.6	125	-60	100.6
GL-08-60	534453	6987048	230.8	125	-45	109.8
GL-08-61	534453	6987048	230.8	125	-60	19.8
GL-08-62	534453	6987048	230.8	125	-85	19.8
GL-08-63	534440	6986959	233.5	125	-70	161.6
GL-08-64	534440	6986959	233.5	125	-60	103.7
GL-08-65	534431	6986952	233.5	125	-80	176.8
GL-08-66	534428	6986938	232.1	125	-70	154
GL-08-67	534457	6986969	230	125	-60	102.1
GL-08-68	534457	6986969	230	125	-75	164.6
GL-08-69	534457	6986969	230	125	-45	52.8
					Total	6457.1

Figure 8: Drill Collar Maps



12 SAMPLING METHOD AND APPROACH

Rock and till sample assays from Aappaluttoq have included bulk samples collected on site in 2006, 2007, and 2008, and samples selected from drill core during the 2007 and 2008 drill programs.

12.1 Diamond Drill Core Samples

Logging of drill core was completed on site under the direction of the Company's qualified person during the 2007 field program. However, mineralized core intervals during January and February 2008 were under the supervision of an independent QP. Diamond drill core was logged at the drill site and in more detail at the Aappaluttoq base camp. Sampling of the drill core in 2007 and 2008 targeted intervals displaying visible corundum mineralization and intervals exhibiting alteration assemblages that are consistent with that seen in ruby and pink sapphire occurrences exposed in the surface trenches at Aappaluttoq. Table 16 presents a summary of significant mineralized intercepts from drill core. Significant intervals include areas of continuous and non-continuous sampling within the mineralized solid. In areas of non-continuous sampling within the mineralized zone a grade of zero has been applied to any intervals without assay values.

Table 16: Summary of Significant Mineralized Intercepts

Hole ID	From	To	Grade	Interval	True Thickness
	m	*m*	*g/t*	*m*	*m*
GL-07-07	9.25	10.8	4,463.3	1.55	1.096
GL-07-07	16.5	19.6	432.6	3.1	2.192
GL-07-12	9.3	11.3	1,397.1	2	1.932
GL-07-42	70.1	70.75	665.1	0.65	0.563
GL-08-51	29.9	31.9	786.0	2	1.732
GL-08-56	73.05	78.4	740.3	5.35	3.783
GL-08-58	54.2	60.2	1,704.6	6	5.196
including	34.5	36.5	1,063.3	2	1.732
GL-08-66	77.3	94.9	8,216.00	17.6	16.539
including	77.3	79.3	46,728.30	2	1.879
GL-08-66	12.8	16.2	5,433.3	3.4	3.195
including	13.6	15.2	11,000.6	1.6	1.504
GL-08-66	98.75	111.8	195.9	13.05	12.263
GL-08-67	46	48.5	2,048.8	2.5	2.165

Selective mineralized drill core samples were collected with a maximum length of 1 m, were assigned unique sample numbers, and were sealed in plastic sample bags. Sample bags were placed together in sealed drums for shipping to the Saskatchewan Research Council (SRC) in Canada to undergo assay for determination of total corundum. Chain of custody procedures was put in place to prevent contamination or tampering with the samples, including tamperproof seals on the shipping drums and supervision when opening the drums upon arrival in Canada.

Whole core sampling procedures were utilized for ruby and pink sapphire assay within mineralized zones on the recommendation of the project QP. No representative sample was retained as a record; however, extensive and detailed core photographs were taken prior to shipment. Selective sampling from mineralized zones was undertaken and there is no continuous assay information throughout all core.

In 2008, samples were collected during drilling under the supervision of Greg Davison, P.Geo. Due to financial challenges, the program was shut down and all drill samples remained in Greenland. Samples were securely stored in Qeqertarsuatsiaat in the Company's seacans along with other camp supplies. In 2009, Iain Groves selected approximately 300 of the most relevant and applicable core samples from the 2008 program and sealed them in a small shipping container to be sent with the 2008 bulk sample bags from Greenland to SGS Lakefield.

There are currently a few hundred samples of low-priority core in storage in Qeqertarsuatsiaat . The priority core samples that were shipped were stored at SGS until 2010 when Andrew Fagan opened the sealed storage containers and cut the core samples, retaining ¼ of the core for QA/QC. Mr. Fagan sent the remaining ¾ sample in a sealed container to SRC for gemstone assay and processing.

12.2 Bulk Sample

Ruby-bearing material was collected in contiguous blocks from measured geologic targets sampled normal to texture, foliation, stratigraphy, and structure.

Bulk samples were collected by standard techniques and recovery procedures including cutting with chain saws, chisels, and use of low intensity blasting. Bulk sample sizes increased from 30 t in 2006, to 54 t (28 t rock, plus 26 t of regolith) in 2007, and 160 t (125 t of rock, ~35 t of regolith) in 2008.

Sample B1 from the 2007 program was extracted from in-situ bedrock using diamond chain saws and chisels and processed at the Company's facilities in Qeqertarsuatsiaat. Subsequent bulk samples collected over 2007 and 2008 were extracted using low intensity blasting. The B2 and 2008 samples were collected using spaced charges to ensure grade and width control. Low intensity blasting techniques did not result in a significant amount of stone breakage when compared with chain saw methods. Sample material was packaged in 1 t super sacs labeled, sealed, and stored on site prior to processing.

Following the collection and on-site storage of the 2008 bulk sample there is evidence that some of the storage sacs were sifted through and material removed; compromising the sample and potentially resulting in inaccurate grades. The sample was shipped to SGS in Canada in Fall 2009 for metallurgical testing. None of this material is included in grade, resource or continuity calculations covered in this report.

EBA believes that sampling conducted by the Company was carried out in accordance with NI 43-101 standards and that the samples tested are representative of the material that will be processed from the Aappaluttoq prospect.

13 SAMPLE PREPARATION, ANALYSIS AND SECURITY

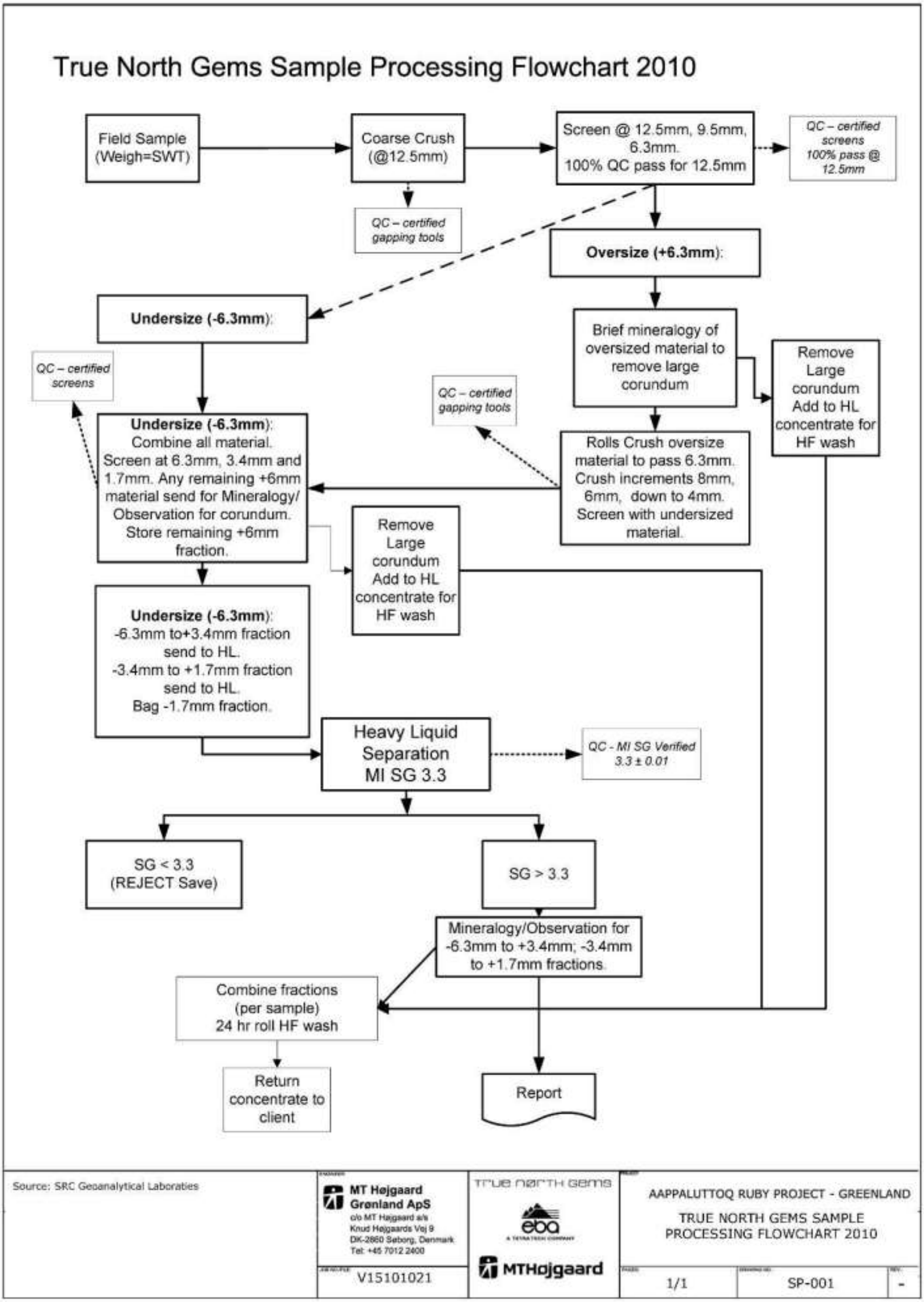
Bulk samples from the 2006 and 2007 exploration programs were processed at the Company's own gravity plant processing facility in Qeqertarsuatsiaat. The only exception from these programs was for sample B2 collected in 2007 which was processed at SGS Minerals Services in Lakefield, Ontario, Canada. Processing of the 2008 bulk sample is currently in progress at SGS Lakefield. Further details of the sample preparation and analysis of the bulk samples is presented in the report on field activities prepared by Weston (Weston 2009).

Diamond drill core from the 2007 and 2008 exploration programs has been processed at the Saskatchewan Research Council (SRC) Laboratories in Saskatoon, SK. Sample preparation in 2007 was under the supervision of the project qualified person (QP), Greg Davison, and the project's independent qualified persons (IQP) at that time, Wardrop Engineering.

EBA, as the IQP of this report, has reviewed the sample preparation procedures and oversaw the laboratory procedures. These procedures have been implemented for processing 2008 material.

The SRC analysis and methodology of the samples from the diamond drill core is presented in the processing flowchart in Figure 9. The diamond drill core samples are run through the assay process in batches. Each batch consists of approximately 60 to 70 samples, and corresponds to one supersac per batch.

Figure 9: Sample Processing Flowchart 2010



The 2007 samples were sent for whole-core processing - a technique widely used in the diamond exploration sector; while the 2008 core samples were cut by the Company's geologist and a ¼ core section retained as a QA/QC representative sample – the remaining ¾ core was processed the same as the 2007.

As no standard procedures exist for assaying corundum from core, procedures were developed by EBA, the Company and engineers from the Saskatchewan Research Council in Saskatoon, SK - an ISO 17025 certified analytical laboratory. This focussed on the recovery of a total corundum weight in grams; only crystals above 1.7 mm (nominally larger than the minimum size of rough gemstone that can be commercially polished) were included and efforts were made to minimise the weight of matrix rock in the final concentrated product. This assay value is referred to as Total Clean Corundum and forms the basis of the assay values for the resource evaluations.

The core samples were crushed in a jaw crusher set at 12.5 mm, products were then screened at 12.5 mm, 9.5 mm and 6.3 mm, and a quick visual analysis of the +6.3 mm material was completed. A lab QC check would ensure 100% of the 12.5 mm material passed through onto the lower screens. If any large corundum crystals were observed then they would be picked out of the material and placed back into the concentrate directly after heavy liquid separation (avoiding further crushing but allowing HF cleaning of the crystals).

The +6.3 mm material was then crushed using high pressure grinding roll (HPGR) at nominal increments of 8 mm, 6 mm and 4 mm and the products were recombined with the -6.3 mm undersize material from the primary jaw crusher. All material was then screened at 6.3 mm, 3.4 mm and 1.7 mm on a standard Tyler mesh. Any remaining +6.3 mm material was sent for visual analysis, mineralogy, documentation and subsequent storage. The -6.3 mm undersize was split into three size fractions: (-6.3 mm +3.4 mm) & (-3.4 mm +1.7 mm) were both sent for heavy liquid separation, the -1.7 mm undersize reject was bagged, weighed, documented and stored.

Heavy Liquid separation was completed in a methyl-iodide liquid set at a specific gravity of 3.3; the consistency of this fluid was constantly checked to ensure the fluid maintained its correct specific gravity throughout the test-work. All material <3.3 SG ''floats" and was collected, washed, weighed and stored. The >3.3 SG material "sinks" was collected and washed to remove the MI liquid, recombined with any large corundum crystals saved as +6.3 mm material and made ready for crystal cleaning. Each size fraction was processed separately to ensure the Company could assess the effects of processing on all the gem material, and allow a direct comparison to the numerous bulk sample grades and distributions collected from the same mineralized locality. QA/QC for checking and maintaining the specific gravity of the heavy liquid in the heavy liquid separation process is done using a hydrometer.

All rough corundum was cleaned using hydrofluoric acid (HF); this acid does not attack the ruby or pink sapphire but dissolves the silicate matrix left attached to the stones. Each sample was inserted into PVC bottles and HF acid added. Each bottle was rolled for 24 hrs, allowing the HF to dissolve most of the available silicate matrix. Some samples required up to three days in the HF bath. The bottles were then emptied, the HF drained and the corundum washed to remove residue. Each sample was weighed as a 'clean corundum' concentrate. Most of the matrix was removed by this process, but a minor amount remained. For the particularly dirty samples a second HF bath was used to assist in matrix removal. Once cleaned, a final 'clean' concentrate weight was recorded and the sample bagged for shipment and assessment. The SRC would produce an assay certificate showing the collective weights of each processing step for each sample. The corundum bearing samples and certificates were then sent to the secure Company lab were then sent to the Company's lab in Vancouver for further gemmological analysis.

The samples were securely shipped in sealed buckets by tracked parcel service between Saskatoon and the Company's laboratory in Vancouver. Upon delivery, each sample was photographed to show the sample number and the 'clean' corundum assay weight written on the bag. A geologist performed a visual estimate on the bags - recording how much of the bag was clean corundum and how much was matrix or non-corundum (despite the HF baths). The senior Company gemmologist then performed a QA/QC check on the recorded lab weights to ensure no mix-up had occurred – then sieved the material on a 1.7 mm Tyler screen. Everything -1.7 mm was bagged and weighed, the remaining +1.7 mm material was split by colour (red or pink) and split again by quality (translucent or opaque). The weight of all fractions was recorded in the central secure assay database. The corundum bags themselves were documented and stored in the Company vault-room.

The following sample preparation and QA/QC procedures was implemented by the Company while working with the samples:

1) QA/QC is performed to confirm the bag number
2) The sample is weighed and compared to the reported weight from the lab. The recorded sample weight is written on the bag
3) A photo of the sample is taken inside a light box with a mm scale visible
4) A visual estimate of the amount sample is recorded and entered into a spreadsheet
5) Check the sieves with the 1.7 mm and 0.85 mm screens
6) Sieve material through the 1.7 mm screen on white paper in a well-lit area
7) Material is sorted based on color, from pink to red
8) Material is sorted based on clarity, from translucent to opaque
9) The sample set is weighed
10) Sample is packaged in small sealed bags. The bag is labelled with weight, original sample number, color, quality,
11) Material is screened through 0.85-1.7 mm. The weight is recorded and sample is bagged
12) Weights and sample information are entered into the master database.

Steps 1-4 described above are carried out by the Company's geological consultant Andrew Fagan, M.Sc. FGS. Steps 5-12 are carried out by geologist and gemmologist, John Mattinson, B.Sc. G.G., the Company's Senior Gemmologist

This database was then transferred to EBA Engineering for internal QA/QC checks and to create a 'factored total corundum' – this figure is a statistical calculation based on the total corundum weight, the visual estimate of the remaining volume of matrix and takes into account the small amount of matrix material still attached to the corundum crystals (i.e. the clean corundum weight included minor matrix). This weight is the final assay weight for the total corundum in the sample and was used to calculate grade in the geological resource model.

13.1 Quality Control

Various levels of QA/QC checks were performed on the core samples in the field, in the assay laboratory and in the Company's gemstone lab. Slightly different procedures were utilized between the 2007 core and the 2008 core. Quality control procedures for the 2007 and 2008 sampling and results available to EBA include documentation of inter-laboratory checks and lab procedures.

Lara Reggin from EBA and Andrew Fagan from the Company completed an inspection of the SRC assay and processing facilities on January 18th and 19th.

In 2007 whole-core processing techniques were used to extract the corundum (as is standard in diamond and gemstone exploration and mineral processing). Field blanks were inserted every 25 samples – these comprised country rock (gneiss) from the base of the drill-holes, taken well away from the nearest mineralization. All but one of these samples passed the QA/QC (i.e. no corundum was recovered); the one fail (sample 144225) returned a value of 0.06 g corundum in the +1.7-3.4 mm size fraction. This sample could represent two things; either the gneiss contains very small amounts of corundum, or a single small corundum grain was caught up inside the crusher between samples. All samples were carefully weighed in and out, all size fractions recorded and industry standard hand-off procedures and chain of custody were followed at all times.

In 2009, the core was sent to SGS Lakefield in a secure sea-can container from Greenland. The seals were unbroken upon arrival in Canada. In 2010 each whole core sample was cut by a Company geologist, so that a ¼ diameter slice was left as QA/QC sample and the remaining ¾ slice was processed in the same fashion as the 2007 core. The ¼ off-cuts were securely sent to the Company laboratory for labeling and storage in the main vault. The remaining core was sent to SRC and processed in exactly the same manner as the 2007 material.

In 2010, the Company produced its own series of blanks, standards and spikes for insertion into the 2008 core material to ensure a more rigorous QA/QC protocol than was completed in 2007. In total 28 samples were prepared, half of these were blanks – comprised of chipped granite material, a felsic volcanic rock and sandstone pebbles. A 500 g batch of natural non-gem quality corundum was acquired by the Company from Indian sources. These crystals were broken up using a hammer so all of the relevant size fractions were included. 14 spikes were produced, each with a specific weight of corundum; these bags were labelled and photographed. Each spike was added to a sample bag alongside a blank sample, thus creating 14 spiked standards, each containing a known weight of corundum. Upon processing, all but 2 of these samples passed QA/QC checks. These two samples were blanks that returned 0.02 g and 0.04 g of corundum respectively – like the 2007 QA/QC sample, the Company believe the 2008 failures were single crystals of corundum that were entrained in the crusher and were transferred between samples.

These were noticed early and the lab was instructed to ensure all of the crusher and jig was cleaned between samples – no further cross-contamination was noted. All of the standards returned corundum weights within 95% of the original spike weight – the slight weight loss was expected and concentrated in the -1.7 mm fraction. All samples were carefully weighed in and out, all size fractions recorded and industry standard hand-off procedures and chain of custody was followed at all times.

Samples from 2007 processed at the lab had the sample number marked on the outside of the bag. Beginning with samples analyzed from the 2008 program, sample bags were labeled with a unique barcode to identify the sample.

Upon receipt of the results from the 2007 program samples it was determined that the sample preparation and processing steps taken by SRC Laboratories were inadequate to clean the corundum completely. QA/QC of the sample database and of the sample concentrates returned to the Company revealed that select samples contained a significant amount of matrix material surrounding the gem material. The recorded weights were determined to include both corundum gem material as well as matrix material. The Company's geologists reviewed all material from 2007 in the +1.7 mm size fraction and identified any samples containing greater than 10% matrix. A total of 33 sample subsets were identified and sent back to SRC for a second round of HF washing. SRC was advised to run future HF washes for as long as was necessary to remove the matrix, generally 24-48 hours.

Quality control samples were collected for both of the bulk samples B1 and B2 samples to test the efficiency of processing. According to (Weston 2009), both of the QC samples for the B1 and B2 bulk samples were collected from tailings material and processed.

Collar locations were determined by handheld GPS units. The collar location reading was taken multiple times and averaged to improve the accuracy of the reported collar location. No differential global positioning system (DGPS) survey or down-hole surveys have been completed to date.

EBA is satisfied that there is minimal risk of contamination and that sample handling is carried out in a sufficient and professional manner by the Company and the laboratories involved in the sample analysis.

13.2 Security

Export permits are required for all gemstone and rock sample material leaving Greenland. All export permits are up to date and have been submitted by the Company's staff and approved by the Greenland Bureau of Mines and Petroleum (BMP). The BMP performs spot check audits on samples leaving Greenland that details tracking, processing and inventory from collection to production of jewellery.

There are a number of security measures in place at the SRC laboratories. Firstly, all staff undergoes criminal checks on a quarterly basis. Staff must have a security key card in order to enter the laboratory area where assay and processing take place. Staff turnover at the lab is very low with no loss of staff in the last eight years.

All samples are numbered, tagged and packaged at the Aappaluttoq base camp. Oversight of all sample preparation and storage is audited by BMP.

Material was shipped in sealed containers with documented chain of custody for all routes of travel. Shipping was completed using secure shipping companies. All chain of custody processes were monitored by William Rohtert, the Company's Qualified Person for the 2006 field program, and J. Gregory Davison, the Company's Qualified Person for the 2007 and 2008 field program, and by Bonnie Weston from 2008 to current.

All products leaving the Vancouver office are listed in transmittal letters and tracked from source to third party facility and on return to source. Every effort is maintained to provide full material balances for each product.

Security procedures passed independent audit in consultation with security experts active in the gemstone industry (Rohtert 2006)

EBA is satisfied that there is minimal risk of tampering and that storage and security is carried out in a sufficient and professional manner by the Company and the laboratories involved in the sample analysis.

14 DATA VERIFICATION

Data was provided by the Company in the form of MS Excel spreadsheets and reviewed by EBA prior to use. Lara Reggin and John Chow of EBA conducted a site visit in November of 2010 during which they reviewed sampling and storage procedures. There were no mineralized sections of core retained on site, as it had all previously been sent for assay, or was in storage in Qeqertarsuatsiaat. In addition a general site tour with the Company's geologists was carried out including a review of significant surface showings on the property and a review of the mineralization and structural controls within the deposit. Representative core samples stored at the Company's Vancouver office were inspected by EBA, and core logs were reviewed and compared to assay results.

Lara Reggin completed a review and tour of the SRC Assay facility in Saskatoon in February 2011 to review assay procedures and reporting procedures. In addition sample handling and records keeping procedures at the Company office in Vancouver was reviewed in detail with Company geologist Andrew Fagan, and Company gemmologist John Mattinson.

In addition to assay certificate verification, EBA verified the database for standard errors including:

- Check for duplicate collars
- Check for twin holes
- Check for overlapping assay intervals
- Check for zero-length assay intervals
- Check for assay spikes
- Check locations for bulk samples.

During the data verification it was noted that there is no down-hole survey data available for drill-holes, and drill-hole collars and bulk sample locations were collected with handheld GPS.

Initially the SRC lab had reported assay data as a blank or a "-" where there was no corundum recovered from the sample, this was clarified with the lab and the database was modified to reflect a zero (0.00 g) value.

EBA is comfortable with the data provided on this project, and did not find any significant issues.

15 ADJACENT PROPERTIES

The Company currently has a renewal application registered with the BMP for a Prospecting License covering the southwest coast of Greenland (an area covering all of onshore Greenland south of latitude 78 N and west of longitude 44 W. A Prospecting License grants the owner nonexclusive rights to explore for mineral resources in the region with the exceptions for areas covered under exploitation or exploration licenses.

The Company also currently has an application registered with the BMP for an Exploration License covering an area of approximately 440 km^2 located near and adjacent to the existing Fiskenæsset and Qaqqatsiaq licenses.

The Qaqqatsiaq license and the newly proposed licenses are both held to cover additional known ruby occurrences within the Fiskenæsset mining district of SW Greenland and establish a secure land tenement position.

There are no similar adjacent properties that have been placed in operation or explored in any manner that would provide information leading to a better understanding of this property.

16 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 Past Testwork Summary

There has been varied metallurgical and gemmological testwork done on the Aappaluttoq project in the past several years. Of these, the field pilot plant work conducted in 2007 by the Company shows that extraction of gem quality corundum material from Aappaluttoq is technically feasible. The following is a list of past work that has been done on extraction of corundum and further cutting/polishing testwork.

- An investigation into optical sorter trials on pre-concentrated crushed samples of ruby ore (SGS, editors 2005a)
- An investigation into bulk sample trials on five individual pre-concentrated samples of ruby ore (SGS, editors 2005b)
- Recovery of rubies from five samples from the Fiskenæsset project (SGS, editors 2006a)
- Plant operations summary report (Gilroy 2007)
- Optical sorting of DMS concentrates for ruby recovery (SGS, editors 2008)
- National Instrument 43-101 Report of Activities For The Fiskenæsset Ruby Project, West Greenland. (Davison 2008)
- Recovery of rubies from the Fiskenæsset Ruby Project (SGS, editors 2009)
- NI 43-101 Report On Field Activities For The Fiskenæsset Ruby Project, Greenland (Weston 2009)
- True North Gems Development Ideas Greenland Project (Gilroy 2009)

All of the testwork reports have been reviewed by the author, and the author is satisfied that the reports are complete and accurately represent the scope and results from the Fiskenæsset samples. Additionally, the author has no reason to believe that any of the testing activities will not withstand scrutiny.

16.2 Current Testwork

SGS Lakefield is currently completing further testwork in relation to extraction of corundum. This work is ongoing and is expected to be complete by the end of quarter 4, 2011 and is expected to allow identification of appropriate final plant equipment and suitable sizing.

16.3 Aappaluttoq 2006 Bulk Sample

Details of processing of the 2006 Aappaluttoq bulk sample have been disclosed in previously published reports (Davison 2008) (Weston, 2008 Report on Field Activities for the Fiskenaesset Ruby Project, Greenland 2008). They are summarised here for completeness. Testwork from the 2006 sample has been used for key parameters to calculate the Mineral Reserve including gem colour distribution, valuations and process plant design criteria.

16.3.1 Rough Dirty Concentrate Extraction and Sorting of 2006 Bulk Sample

The 30 t 2006 Aappaluttoq bulk sample was processed in the village of Qeqertarsuatsiaat using a conventional gravity wash plant producing a dirty jig concentrate of 717 kg and 455 kg of rock and mineral specimens respectively. Of the jig concentrate, 186 kg was further hand cobbed and sorted to produce a secondary concentrate of 61 kg.

A total of 592 kg of material was exported to Vancouver and Germany for further sorting and processing.

27.5 kg of material was removed by hand picking during field operations and sent to Vancouver. The rest was sorted by hand in the lab or during optical sorting tests in Germany. This resulted in a total dirty corundum weight of 191 kg (Figure 10).

At the time optical sorting tests proved inconclusive and are not included in any present production plans. However further testing is planned which if successful could lead to a much more cost effective processing circuit.

Figure 10 shows the mass balance of the 2006 bulk sample dirty rough concentrate production.

Material was first sorted into three size fractions, +1.7 mm, +4.0 mm, and +6.0 mm. It was then sorted by quality into high-quality gem, gem, and near-gem. The high-quality gem and gem components were then separated into red and pink varieties (this colour split was not performed on near-gem material). These sorts resulted in 15 different categories of material.

The laboratory and optical sorting of the 2006 Aappaluttoq final dirty concentrate resulted in the generation of approximately 135 kilograms of gem and near gem rough ruby and pink sapphire. This material was not cleaned or processed in any way beyond initial crushing and jigging in the field (Table 17).

Figure 10 Mass Balance of Dirty Rough Concentrate Production

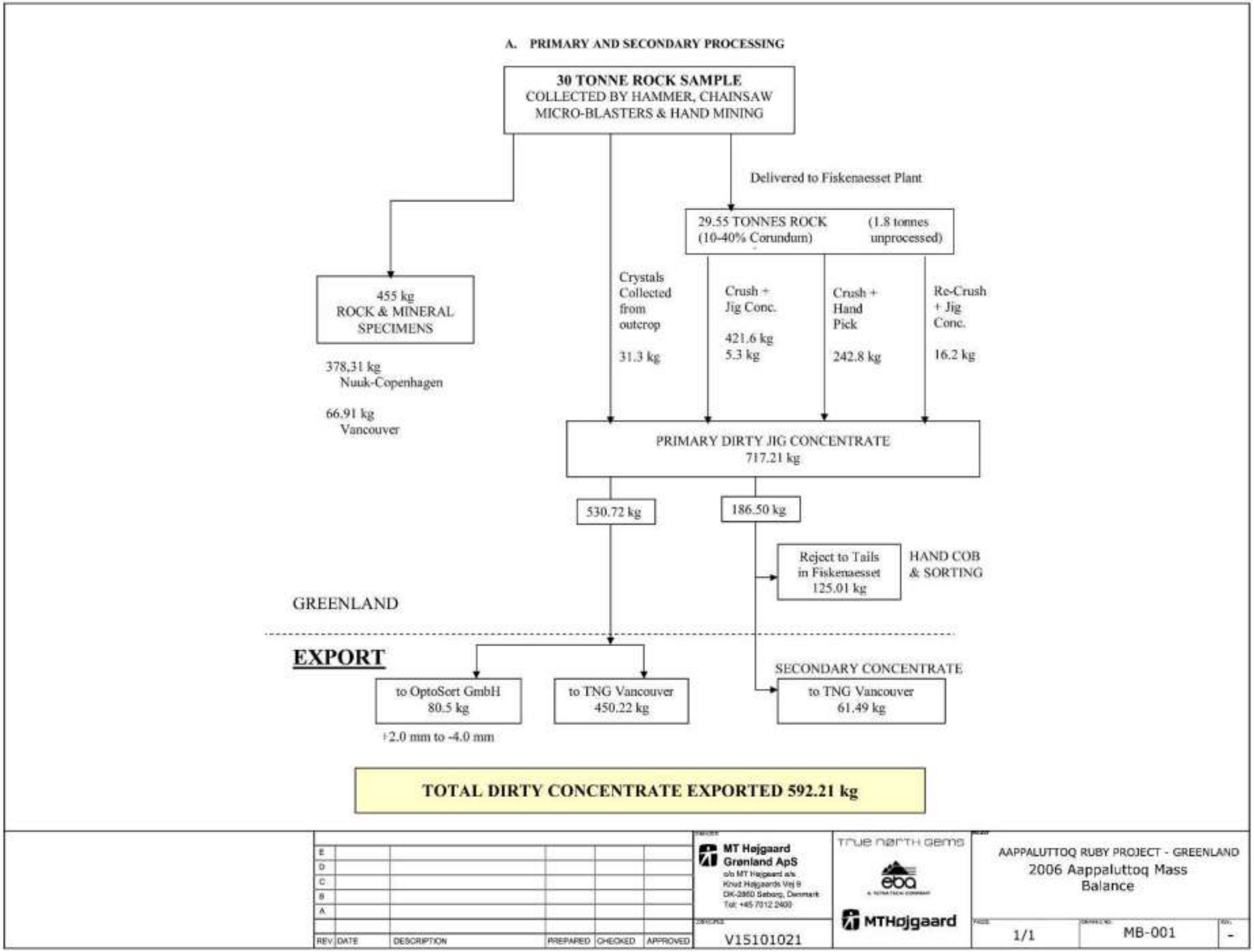


Table 17: Crushing And Jigging Field Data – 2006 Bulk Sample

Sort Method	Size Fraction				Weight
Colour	mixed	+1.7 mm	+3.4 mm	+6.1 mm	Grand Total
	g	g	g	g	g
Field Sort (total)	**27,516.55**				**27,516.55**
Gem (High-Quality)					
Red	222.20				222.2
Pink	627.01				627.01
Lilac	46.76				46.76
Total	895.97				895.97
Gem	10,351.19				10,351.19
Red	3,049.31				3,049.31
Pink	6,661.62				6,661.62
Lilac	640.26				640.26
Total	10,351.19				10,351.19
Near-Gem	14,407.12				14,407.12
Specials	236.71				236.71
Dentes	575.69				575.69
Carving	1049.87				1049.87
Lab Sort (total)		**11,555.47**	**82,531.58**	**43,791.79**	**137,878.84**
Gem (High-Quality)					
Red		345.66	761.97	93.32	1,200.95
Pink		798.71	1,610.54	792.84	3,202.09
Lilac		41.79	118.65	179.13	339.57
Total		1,186.16	2,491.16	1,065.29	4,742.61
Gem					
Red		825.72	5,587.78	3,572.31	9,985.81
Pink		1,747.63	15,285.62	5,085.06	22,118.31
Lilac		50.13	467.02	571.08	1,088.23
Total		2,623.48	21,340.42	9,228.45	33,192.35
Near-Gem					
Tumbled		0	0	3,684.88	3,684.88
Rough		4,665.98	28500.00	16,004.61	49,170.59
Total		4,665.98	28500.00	19,689.49	52,855.47
Non-Gem					
Tumbled		0	0	1,408.56	1,408.56
Rough		3,079.85	30200.00	12400.00	45,679.85
Total		3,079.85	30200.00	13,808.56	47,088.41
Optic Sort (total)		**25,634.13**			**25,634.13**
Gem (High-Quality)					
Red		700.05			700.05
Pink		1,219.62			1,219.62
Lilac		990.24			990.24
Total		2,909.91			2,909.91
Gem					
Red		263.13			263.13
Pink		2,299.88			2,299.88
Lilac		2,640.12			2,640.12
Total		5,203.13			5,203.13
Near-Gem		8,571.09			8,571.09
Non-Gem		8950			8950
Gem (High-Quality)	4,096.07	2,491.16	1,065.29	895.97	8,548.49
Gem	7,826.61	21,340.42	9,228.45	11,163.59	49,559.07
Near-Gem	13,237.07	28,500.00	19,689.49	15,456.99	76,883.55
Non-Gem	12,029.85	30,200.00	13,808.56	0	56,038.41
Grand Total	**27,516.55**	**37,189.6**	**82,531.58**	**43,791.79**	**191,029.52**

16.3.2 Rough Gemstone Splitting and Cleaning of 2006 Bulk Sample

Due to the abundance of gem and near-gem material collected as part of the 2006 sample, it was decided to generate statistically representative splits using a Jones splitter which could be used for cutting and valuation experiments. This would create smaller, more manageable parcels but because they are representative splits, the data generated from them could be applied to the entire sample.

The gem and near-gem material was split into 8 equal parts, the first two were kept separate as individual samples and the remaining 6 were recombined. The first 1/8th split contained 16,188 g of material and the second 1/8th split contained 16,387 g. The remaining six splits were recombined into the 6/8th working split.

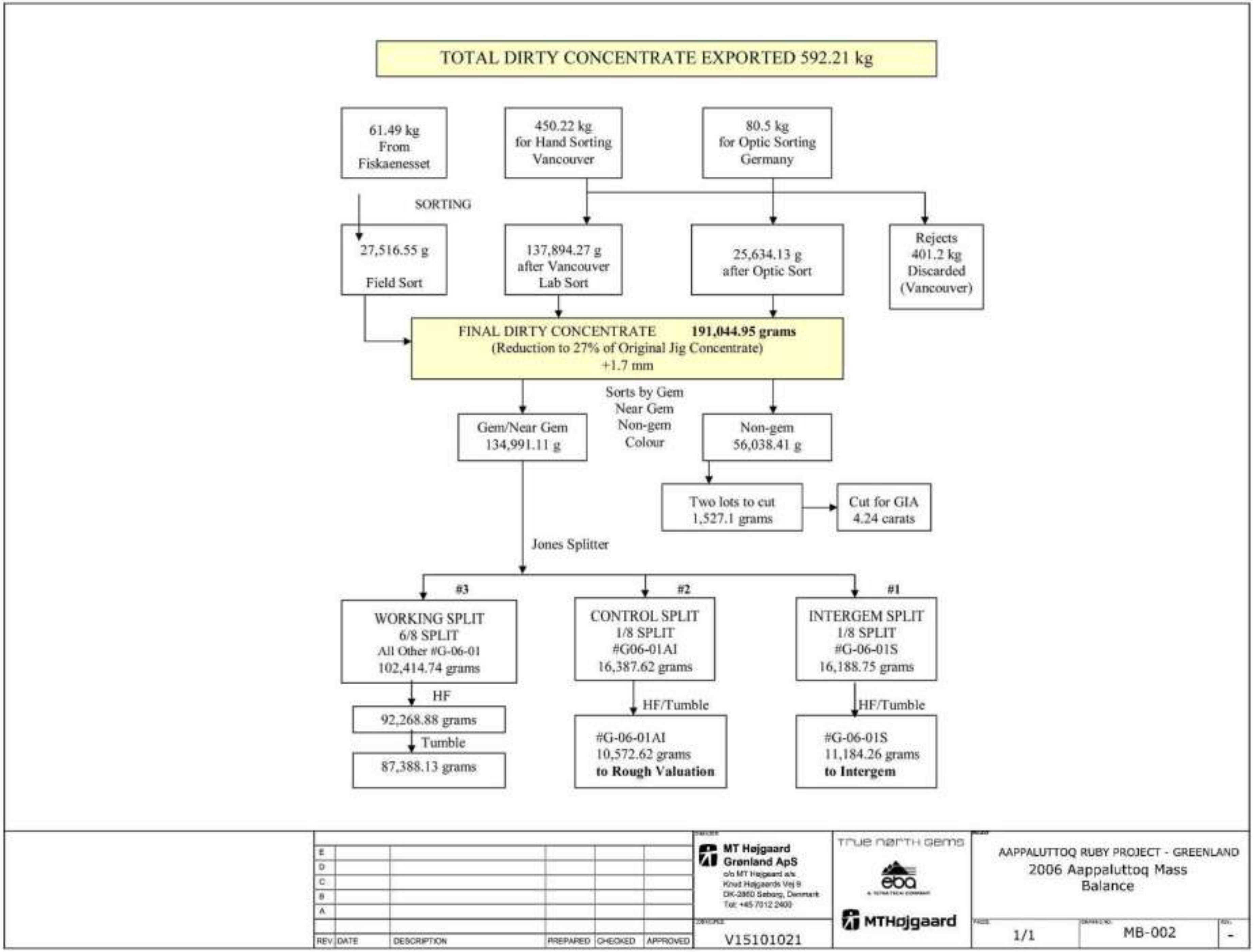
All of the 1/8th splits of rough material were considered 'dirty' due to the presence of matrix still attached to many of the crystals. Material within each +4.0 mm and +6.0 mm bag was further sorted into rough material that was either naturally clean or dirty rough material that needed further cleaning. The material needing further cleaning was sent to Saskatchewan Research Council (SRC) for HF cleaning and when returned was tumbled in the Company's lab using vibratory tumblers to remove the white Calcium Fluoride (CaF) residue coating some of the stones. All +1.7 mm material was sent to SRC for HF cleaning and was tumbled upon return. No additional work was done to try and clean the dirty material, so it was shipped out for cutting 'as is'.

Final total weight of the first 1/8th split was 11,184.26 g resulting in the total loss from cleaning of approximately 31%.

Material from the second 1/8th split which weighed 16,265 g was sent to SRC for HF cleaning and subsequently tumbled upon return with the final weight at 10,572 g. The loss from cleaning was approximately 35%.

Figure 11 shows the mass balance of the 2006 bulk sample gem and rough gem production.

Figure 11: Mass Balance of the 2006 Bulk Sample Gem and Rough Gem Production



16.3.3 Cutting and Valuations of 2006 Bulk Sample

16.3.3.1 First 1/8th Split of 2006 Bulk Sample

In order to provide an assessment of the value and potential use of the Aappaluttoq ruby and pink sapphire material, manufacturing testwork was conducted on the first 1/8th split which was polished and valued as a finished parcel. In order to ensure chain of custody, and address issues of security, sample tracking and logistics, restrictions were placed on the methodology of the manufacturing experiment such that all of the material was to be processed through a single cutting factory with the capability to cut the volume of material in a reasonable amount of time.

Intergem Exports, a cutting facility in Sri Lanka, was identified as the supplier of polishing services for the parcel. The material was shipped to Intergem Exports on April 16, 2007. Return shipments were received in increments on June 20, July 6, July 24, August 17, and October 1, 2007.

Yield was calculated by dividing the final finished carat weight of each lot by the initial shipping weight. The overall yield for the entire shipment is 9.3%. This is comprised of 0.8% yield for faceted stones and 8.5% yield for cabochons.

No pre-forming or physical trimming of the marketable ruby and pink sapphire concentrate was completed after cleaning. The parcel of marketable clean rough represents a natural product. Like all other gem samples that the Company has extracted, no value added processes or treatments such as heat treatment or flux filling were applied to this first 1/8th split, and therefore, it is representative of a clean marketable product that would be expected from any potential mining operation at the Aappaluttoq prospect.

The faceted and cabochon gemstones from the first 1/8th split of the 2006 sample were submitted for valuation with Mr. John Mattinson, B.Sc., G.G. Mr. Mattinson has been involved in the gemstone and jewellery business since 1984. Mr. Mattinson was considered an independent person at the time, although has subsequently become the Company's Senior Gemmologist.

The value attributed to 9,327 faceted rubies and pink sapphires weighing 471.83 carats is USD 26,188 (average US 56/carat). The value attributed to 15,970 ruby and pink sapphire cabochons weighing 4,752.27 carats is USD 97,079 (average USD 20/carat). The combined total is USD 123,266 for the entire 5,224.10 carat sample (Mattinson 2007).

For verification purposes, two sample lots, #3 and #17, from the 2006 sample were sent to two other independent facilities for valuation. The purpose of this exercise was to compare the results of all three valuations for consistency. All values reported are US wholesale.

Mr. Duncan Parker, FGA, FCGmA, of the Harold Weinstein Ltd. Gemmological Laboratory performed the first valuation audit.

Mr. Stuart Robertson, G.G., is Research Director for Gemworld International, Inc. Gemworld International is a research and consulting firm that monitors relevant developments regarding the worldwide diamond and coloured gemstone trade. Mr. Robertson performed the second valuation audit.

Table 18 below shows a summary of the audits performed. Results from Parker show an increase in value, from those reported by Mattinson. Results from Robertson show similar values to those reported by Mattinson for faceted stones but slightly higher values for cabochons.

Table 18: Summary Of Valuation Audit Of Faceted And Cabochon Gemstones

Valuer:			John Mattinson		Duncan Parker		Stuart Robertson	
Lot #	Material	Parcel Weight	Parcel Value	Value	Parcel Value	Value	Parcel Value	Value
		ct	*$*	*$/ct*	*$*	*$/ct*	*$*	*$/ct*
Lot 3	Faceted Ruby	3.37	201	59	404	120	487	55
	Faceted Pink Sapphire	4.53	112	25	407	90	141	31
	Cabochon Ruby	341.95	6,949	20	20,517	60	8,438	25
	Cabochon Pink Sapphire	168.35	1,684	10	8,417	50	2,716	16
Lot 17	Faceted Ruby	2.48	153	62	496	200	146	59
	Faceted Pink Sapphire	0.44	36	82	88	200	19	44
	Cabochon Ruby	84.47	1,689	20	5,068	60	1,520	18
	Cabochon Pink Sapphire	3.98	60	15	159	40	100	25
Total		609.57	10,884		35,556		13,567	

16.3.3.2 Second 1/8th Split of 2006 Bulk Sample

The second 1/8th split was shipped to the Company's Bangkok office in December 2007 where it was re-sorted into lots consistent with typical Governmental rough valuation procedure and protocol. Material was kept in the same size (+1.7 mm, +4.0 mm, and +6.0 mm) and clarity (transparent and translucent) classes as it was sent, but it was further sorted by local gemstone sorters into different colours (red, red/pink, and pink) and reject (due to poor colour, broken or uncuttable stones) categories.

The re-sorted material was then made available for "rough" valuation purposes.

The second 1/8th split was submitted for rough gemstone valuation in January 2007. After resorting in Bangkok, 8.1 kg of material was considered amenable to cutting and polishing and only that material was valued. The material was independently valued by Mr. Anura Wijemanne of Anura Wijemanne Associates of Sri Lanka. Mr. Wijemanne is a well-regarded coloured stone trader that specializes in the sale and purchase, as well as cutting and polishing of rough coloured gem material, especially ruby. The balance of sample, 2.5 kg, was discarded as waste material.

The final valuation for the 8.1 kg of clean rough corundum from the second 1/8th split was USD 4,167. This averages USD 517/kg for ruby and pink sapphire rough, ranging from USD 175/kg for low quality rough to USD 2,875/kg for high quality rough. (Weston, 2008 Report on Field Activities for the Fiskenaesset Ruby Project, Greenland 2008)

16.4 Site Mineral Processing Summary

The site processing plant will consist of crushing, screening and jigs to produce a rough ruby concentrate from the mined ore. The nominal throughput of the initial base case plant is 78 tpd. Production is limited by the capacity of the mineral jigs. This method of extraction was successfully demonstrated in the large–scale field pilot plant operation conducted by the Company at Qeqertarsuatsiaat in 2007.

The overall objective of the plant layout and design will be to permit the introduction of progressive degrees of automation including optical sorting, heavy liquid or dense media sorting and magnetic separation as these processes are tested and determined to be appropriate.

16.4.1 Site Crushing Plant Design

The crushing circuit is a three stage system which will reduce the nominally 300-500 mm size of the mined rock to minus 10 mm. The circuit is dry apart from water used for dust suppression.

Mine haul trucks will dump ore directly into a grizzly covered hopper or on a stockpile at the processing plant; a front end loader will load the ore from the stockpile into a grizzly covered hopper as required. Oversize rocks occasionally caught on the grizzly will be cracked into smaller pieces with a mounted rock breaker. Ore in the hopper is moved along by a 1,020 mm × 3,650 mm vibrating grizzly feeder into the primary jaw crusher. This 610 mm × 915 mm primary crusher will reduce the size of the rocks down to nominal minus 90 mm. The 90 mm undersize from the vibrating grizzly will bypass the primary jaw crusher and directly feed the conveyor.

Crushed ore is conveyed to a 1,000 mm × 6,000 mm hand-picker belt equipped with four hand-picking stations.

The crushed ore is then conveyed to a 37 m^3 crushing screen feed bin feeding into the vibrating screen. The course fraction above about +22 mm size is conveyed to the secondary crushing stage, a 250 mm × 760 mm secondary jaw crusher which will crush this fraction to a nominal minus 25 mm. Secondary crushed ore is then conveyed to the second 1,000 mm × 5,000 mm hand-picker belt equipped with four hand-picking stations. After hand picking the remaining crushed rock is conveyed back into the crushing screen feed conveyor.

The hand-picking stations will use trained operators to visually pick and re-move large sized crystals into 60 kg capacity self-draining security bins. These bins will join the other security bins for transport directly to Nuuk.

The middle size fraction from the double deck crushing screen, between +10 mm and -22 mm size, will be conveyed to a 10 m3 surge bin feeding the third stage cone crusher. A 2,260 × 1,850 × 21,000 mm, 75 kW cone crusher will nominally crush to minus 10 mm. After tertiary crushing the rocks are conveyed back into the crushing screen feed conveyor.

The -10 mm undersize from the double deck vibrating screen falls into a 10 m^3 capacity crushed product bin. Crushed product, all now at -10 mm size, is conveyed to the concentration side of the process plant where wet screening and jigging of the material is carried out.

16.4.2 Site Screening and Jigging Plant Design

Concentration plant feed is withdrawn from a 37 m^3 sizing screen feed bin via a feed conveyor at a set mass flow rate into a log washer which simply mixes water with the crushed rock to form slurry. The slurry is then fed onto the first wet-sizing screen which separates the fines, and the coarse material is sent to a second log washer which readjusts pulp density and then to the second wet screen followed by multiple jigging units. This equipment will isolate the denser corundum concentrate from the waste rock.

The first log washer discharges the mixture of crushed ore and water, slurry, into the top side of a wet screen that is continuously sluiced with water. The sizing screens separate the finest fraction from the coarsest. The finest fraction, -1.7 mm rock, is sluiced directly into the tailings tank.

The coarser fractions will feed a second log washer to readjust pulp density followed by a second wet screen and then feed into separate sets of jigs where the heavier corundum gem mineral is concentrated out of the gangue material.

The slurry will feed into a second wet screen which will separate the solids in three sizes. The top separator will have a screen size of 6.3 mm. The undersize will then directly feed into the next vibrating screen. The +6.3 mm oversize will be gravity fed into two 610 mm × 915 mm duplex jigs. This process is repeated for the 3.4 mm and 1.7 mm size fractions. The lighter reject portion from all the jigs is all sluiced with water into the jig tailings tank where it is mixed with the slurry from the Sizing Screen Undersize Tank. All of the tailings slurry is then pumped to the tailings discharge point deep under the surface of the lake.

The corundum concentrate collected within each jig is batch discharged into self-draining 60 kg capacity security bins. At the end of each shift, these bins are sealed under security supervision and stored in an onsite vault for transport to the Nuuk sorting facility.

16.4.3 Operating Schedule and Availability

The plant will operate between four and 8 months per year. The initial capacity is 78 tpd which is dependent on the capacities of mineral jigs. Once further testwork and commissioning is completed, an expansion will be completed in year 3 (2014) to increase capacity to 117 tpd.

The plant is assumed to have an availability of 85% and utilization of 85% which is below industry standards. This allows contingency for training and the small size of the operation.

16.4.4 Tailings Disposal

See section 19.9.1 Tailings for details on tailings disposal.

16.4.5 Power

The total equipment rated power is 445 kW. The actual draw during operations is expected to be 267 kW which is 60% of the total equipment rated power. Table 19 and Table 20 show the major equipment and power requirements.

Power will be provided by onsite diesel generators.

Table 19: Crushing Process Equipment List

Area	Description	size	Power
			kw
E01-AIC-001	CRUSHING PLANT AIR COMPRESSOR	50 m^3/h	7
E01-AID-001	CRUSHING PLANT AIR DRYER	25 m^3/h	0.5
E01-AIR-001	CRUSHING PLANT AIR RECEIVER	2 m^3	
E01-BIN-001	CRUSHING SCREEN FEED BIN	37 m^3	
E01-BIN-002	CONE CRUSHER FEED BIN	10 m^3	
E01-BIN-003	JAW CRUSHER No.2 FEED BIN	10 m^3	
E01-BIN-004	CRUSHED PRODUCT BIN	10 m^3	
E01-CNV-001	JAW CRUSHER No.1 DISCHARGE CONVEYOR	15,000 L × 457 mm W	4
E01-CNV-002	HAND PICK CONVEYOR No.1	6,000 L × 1,000 mm W	4
E01-CNV-003	JAW CRUSHER No.1 DISCHARGE TRANSFER CONVEYOR	23,000 L × 457 mm W	4
E01-CNV-004	CONE CRUSHER BIN FEED CONVEYOR	19,000 L × 457 mm W	4
E01-CNV-005	CONE CRUSHER DISCHARGE CONVEYOR	22,000 L × 457 mm W	4
E01-CNV-006	JAW CRUSHER No.2 FEED CONVEYOR	12,000 L × 457 mm W	4
E01-CNV-007	JAW CRUSHER No.2 DISCHARGE CONVEYOR	12,000 L × 457 mm W	4
E01-CNV-008	HAND PICK CONVEYOR No.2	6,000 L × 1,000 mm W	4
E01-CNV-010	SIZING SCREEN FEED CONVEYOR No.1	38,000 L × 457 mm W	4
E01-CNV-011	CRUSHER DISCHARGE TRANSFER CONVEYOR	23,000 L × 457 mm W	
E01-CNV-012	CRUSHER DISCHARGE CONVEYOR	25,000 L × 457 mm W	
E01-CNV-013	CRUSHING SCREEN FEED CONVEYOR	30,000 L × 457 mm W	
E01-CRU-001	JAW CRUSHER No.1	2,600 × 1,880 × 2,390 mm	75
E01-CRU-002	JAW CRUSHER No.2	1,800 × 1,600 × 1,320 mm	30
E01-CRU-003	CONE CRUSHER	2,260 × 1,850 × 2,100 mm	75
E01-FDR-001	STATIONARY VIBRATING GRIZZLY FEEDER	1,020 × 3,650 mm	15
E01-LUB-001	JAW CRUSHER LUBE UNIT No.1		
E01-LUB-002	JAW CRUSHER LUBE UNIT No.2		
E01-LUB-003	CONE CRUSHER LUBE UNIT		
E01-PSU-001	CRUSHING AREA SUMP PUMP		
E01-SCB-001	JAW CRUSHER No.1 DISCHARGE CONVEYOR WEIGHTOMETER		
E01-SCN-001	CRUSHING SCREEN	1,525 W × 4,628 mm L	11
E01-SSW-001	SAFETY SHOWER AND EYE WASH STATION No.1		
E01-SSW-002	SAFETY SHOWER AND EYE WASH STATION No.2		
E01-SSW-003	SAFETY SHOWER AND EYE WASH STATION No.3		
E01-SYS-001	CRUSHING PLANT DUST COLLECTION SYSTEM		
E01 Total			249.5

Table 20: Concentration Process Equipment List

Area	Description	size	Power
			kw
E02-AGI-001	SIZING SCREEN UNDERSIZE TAILINGS TANK AGITATOR	1,200 mm Ø	40
E02-AGI-002	JIG TAILINGS TANK AGITATOR	1,200 mm Ø	40
E02-BIN-001	SIZING SCREEN FEED BIN	37 m^3	
E02-CNV-001	SIZING SCREEN FEED CONVEYOR No.2	12,000 L × 457 mm W	4
E02-EQP-001	JIG No.1	1,070 mm cell	2.2
E02-EQP-002	JIG No.2	1,070 mm cell	2.2
E02-EQP-003	JIG No.3	1,070 mm cell	2.2
E02-EQP-004	JIG No.4	1,070 mm cell	2.2
E02-EQP-005	JIG No.5	1,070 mm cell	2.2
E02-EQP-006	JIG No.6	1,070 mm cell	2.2
E02-PSL-001	SIZING SCREEN UNDERSIZE TAILINGS TANK PUMP No.1	200 kPa; 50 m^3/h	10
E02-PSL-002	SIZING SCREEN UNDERSIZE TAILINGS TANK PUMP No.2	200 kPa; 50 m^3/h	10
E02-PSL-003	JIG TAILINGS TANK PUMP No.1	500 kPa; 50 m^3/h	20
E02-PSL-004	JIG TAILINGS TANK PUMP No.2	500 kPa; 50 m^3/h	20
E02-PSU-001	CONCENTRATION AREA SUMP PUMP		2
E02-SCB-001	SIZING SCREEN FEED CONVEYOR WEIGHTOMETER		
E02-SCN-001	SIZING SCREEN No.1	1,528 W × 4,000 mm L	15
E02-SCN-002	SIZING SCREEN No.2	1,528 W × 4,000 mm L	15
E02-SSW-001	SAFETY SHOWER AND EYE WASH STATION No.1		
E02-SSW-002	SAFETY SHOWER AND EYE WASH STATION No.2		
E02-SSW-003	SAFETY SHOWER AND EYE WASH STATION No.3		
E02-TNK-001	SIZING SCREEN UNDERSIZE TAILINGS TANK	3,600 Ø × 3600 mm H	
E02-TNK-002	JIG TAILINGS TANK	3,610 Ø × 3610 mm H	
E02-WAS-001	LOG WASHER No.1	870 × 3,144 × 750 mm	3
E02-WAS-002	LOG WASHER No.2	870 × 3,144 × 750 mm	3
E02 Total			195.2

16.4.6 Water

The sources of water for the processing of the ore will be the nearby lake.

- The crushing stage is assumed to be dry crushing with minimal water used for dust suppression.
- Fresh water will be required for the concentrating plant, screen and jigs.
- The main plant will also require potable water facilities which are considered to be a small quantity and not included in the estimate of water usage.

Total process water consumption is estimated to be 10 kL per ore tonne.

17 MINERAL RESOURCES AND RESERVES ESTIMATES

17.1 History

There has been no previous mineral resources or reserves reported for the Aappaluttoq project.

17.2 Geologic Modelling

EBA has reviewed updated three dimensional geological interpretations of the mineralized zones (prepared primarily by the Company) and extensive project data collected since 2006. The geological model was based on observations and structural measurements collected from surface outcrop as well as from drill logs. Drill density within the deposit is at a consistent collar spacing of 8 m centers with fans of drill holes from each collar location at -45°, -60° and -75° dips giving good control to the geological model. Holes were drilled at an average azimuth of approximately 133° intercepting the mineralized zone perpendicular to the strike direction of the host lithology.

The Aappaluttoq geological model for the mineralized host zone was constructed as a solid model. Drill hole data was imported into Gemcom GEMS software (v6.3) and interpretive 3D rings linking correlated host zone rock, notably the phlogopite and contact zone between the phlogopite and leucocratic gabbro units, were digitized on 2 to 4 m spaced cross-sections. Each interpretive ring was then linked with the corresponding ring on the adjacent cross sections to create a 3D wireframe solid for the mineralized (host) zone. Corundum assay values from drill core samples were used to provide added control on the wireframe extents.

The geological model for the non-mineralized (waste rock) lithologies was constructed through interpolation of data from the drill-holes directly into the Rock Type model within the Block model. Interpretation of the lithological data was accompanied by assigning a numerical lithology code to each waste rock type and used an ID calculation between logged intervals.

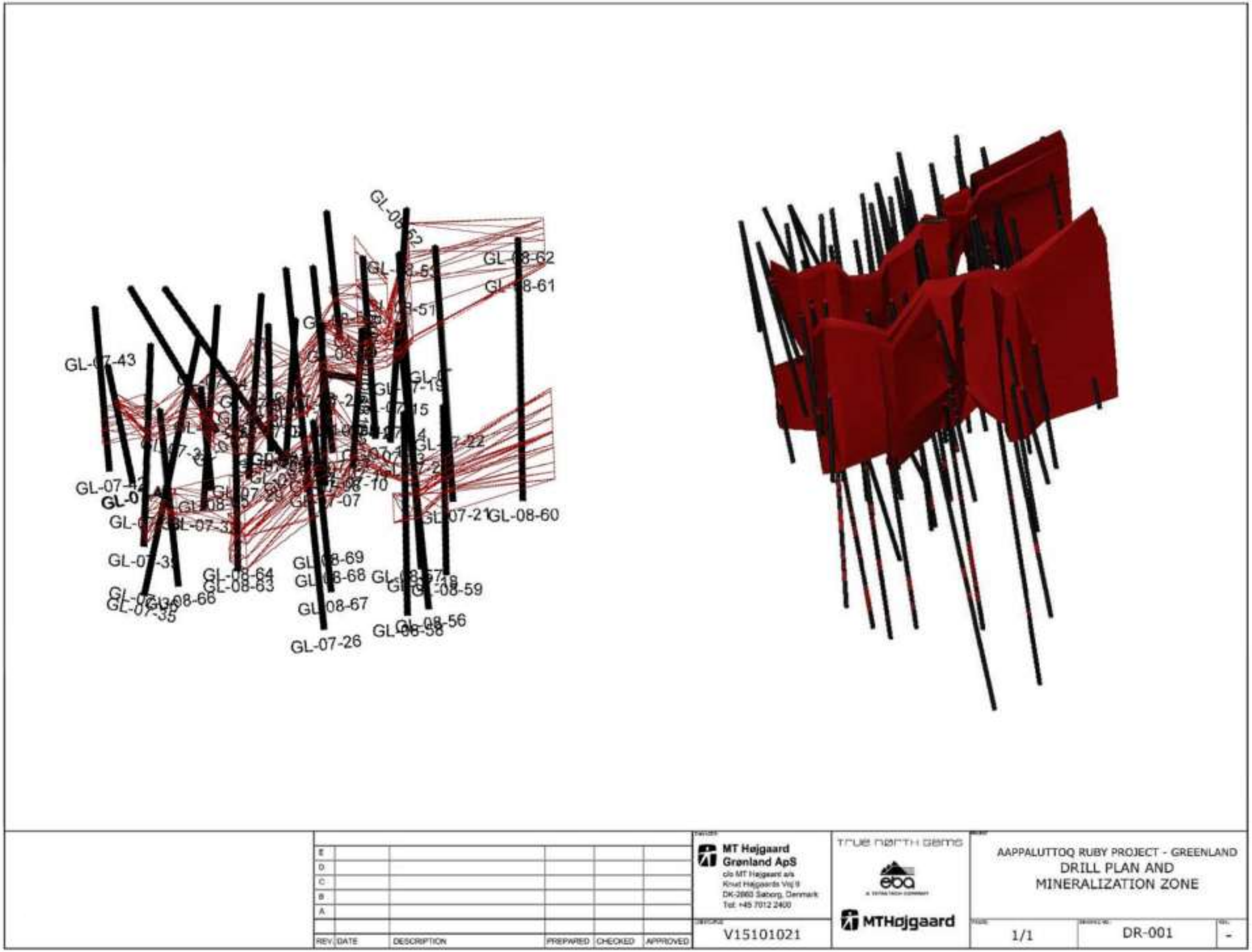
17.3 Block Modelling

The block model was created by EBA using Gemcom GEMS software (v6.3). The block model consists of blocks measuring 1 × 1 × 1 m. Extents of the block model in x, y, z directions are 295 m, 280 m, and 150 m, respectively. The model is oriented to azimuth 50 and the block dimensions are orthogonal with the trend of the mineralization. The model is shown in Figure 12 along with the geological drilling.

The block size of 1 m was chosen based on the geometry of the host zone lithology, which is as narrow as 3 m in some places, and the block model takes in to consideration potential mining methods for subsequent mine planning. Assay data for total raw corundum were utilized for interpolation into the block models on a geological model basis. A rock code of 200 was assigned within the mineralized zone and interpolation of the grade data was constrained to this rock code. Only data from inside mineralized solid were used for block model interpolation. The cut-off depth of 83 m for the mineral resource categories were defined within each zone based on density of sample data and drill-hole logs including lithology and the consistency of grade data within the geological models. Resource tonnages were derived using rock volumes reported from the block model and representative specific gravity measurements for each lithology. The SG values of the 2008 drill core samples (Series 143) were calculated prior to crushing and processing. The SG values of the 2007 drill core samples (Series 144) were not calculated prior to crushing and processing 2007 core assays do not appear to have S.G's calculated on the core prior to crushing and processing. Additional sampling of the 2007 drill core was undertaken during the re-logging process for ARD and SG work.

The block model was constructed using GRADE basis proximate data; interpolated values included corundum (g/t).

Figure 12: Drill Plan and Mineralization



17.4 Geostatistics

Variography and other geostatistical methods were used by EBA for the purpose of defining the search parameters for the geological and grade quality block models. Geostatistics were used to assess top cut off value of 7,325 g/t (97.5 percentile of assay data) for the block model. Variography interpretation was not successful for this data set, and appeared to be a reflection of the data distribution, indicating that the preferential direction for mineralization was parallel to drill hole azimuth. This is likely a result of the tight spacing of drill holes, with limited assay sample data within the drill holes.

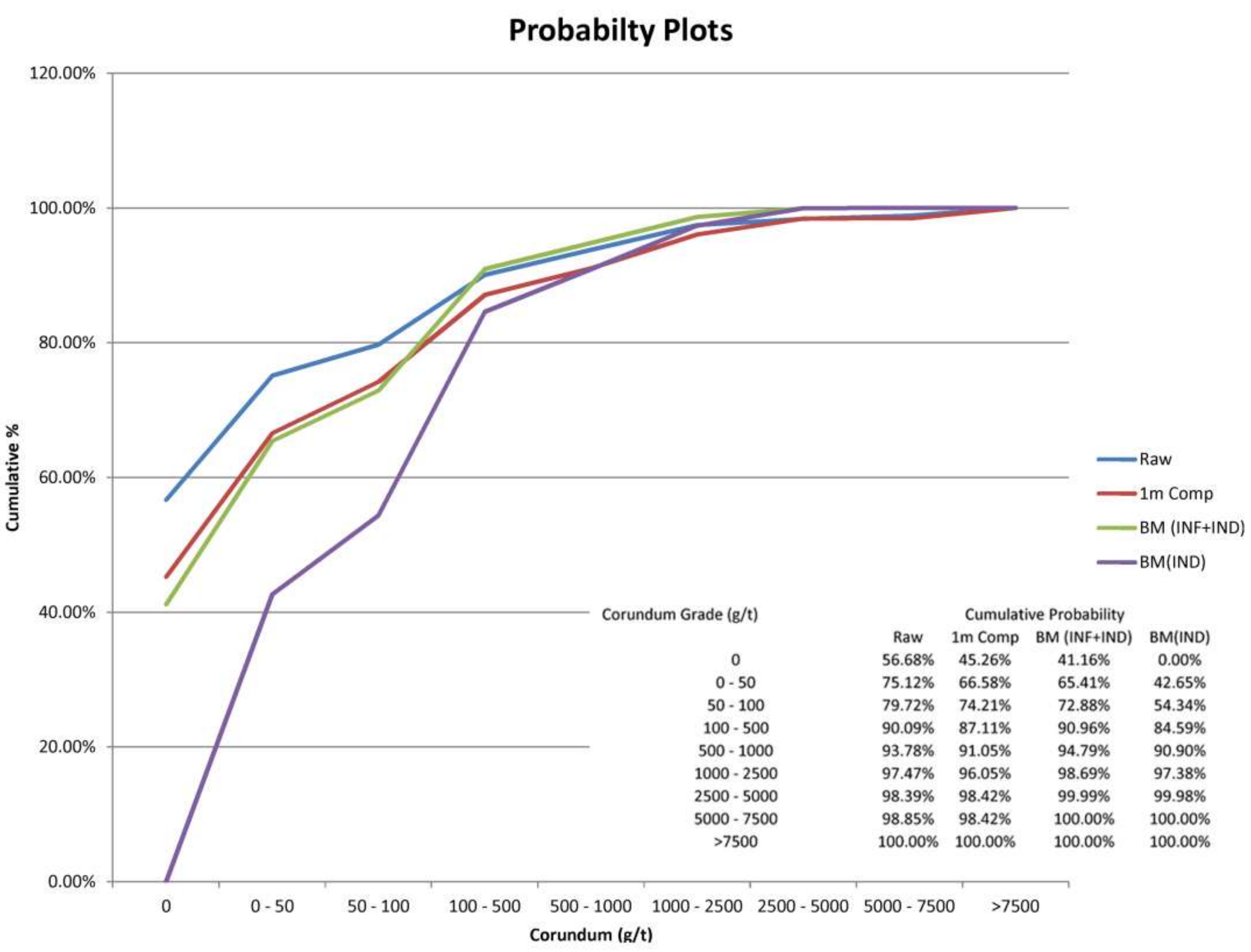
Statistics were compiled for raw data, composites, and block model grades (Table 21).

Table 21 Aappaluttoq Block Model Statistics Summary

	Total Number	Mean Grade	Maximum Grade	Minimum Grade	Standard Deviation	Variance
		g/t	*g/t*	*g/t*		
Raw Assays	497	534.56	72,812	0	4,184	17,505,590
Composites	380	537.80	57,162	0	3,353	11,240,837
Indicated Blocks	73,489	302.15	6,663	0	618	381,380
Inferred Blocks	48,225	166.41	5,325	0	412	169,334

Probability plots for raw, composite and block model data were plotted and compared to ensure similar distribution of data and to identify any errors or outliers (Figure 13) The plot shows consistent data distribution for raw, composite and total block model data, with a smoothing of data from raw to composite, and from composite to block model as would be expected.

Figure 13: Probability Plots



Corundum Grade (g/t)	Cumulative Probability			
	Raw	1m Comp	BM (INF+IND)	BM(IND)
0	56.68%	45.26%	41.16%	0.00%
0 - 50	75.12%	66.58%	65.41%	42.65%
50 - 100	79.72%	74.21%	72.88%	54.34%
100 - 500	90.09%	87.11%	90.96%	84.59%
500 - 1000	93.78%	91.05%	94.79%	90.90%
1000 - 2500	97.47%	96.05%	98.69%	97.38%
2500 - 5000	98.39%	98.42%	99.99%	99.98%
5000 - 7500	98.85%	98.42%	100.00%	100.00%
>7500	100.00%	100.00%	100.00%	100.00%

REV.	DATE	DESCRIPTION	PREPARED	CHECKED	APPROVED
E					
D					
C					
B					
A					

MT Højgaard Grønland ApS
c/o MT Højgaard a/s
Knud Højgaards Vej 9
DK-2860 Søborg, Denmark
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V15101021

TRUE NORTH GEMS

eba
A TETRA TECH COMPANY

MTHøjgaard

AAPPALUTTOQ RUBY PROJECT - GREENLAND
PROBABILTY PLOTS

1/1

PR-001

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17.5 Composites

Assay samples were composited on a 1.0 m down-hole composite interval within geological boundaries of the mineralized zone. During compositing process residual samples were retained regardless of length (i.e. remaining samples less than the 1 m composite length occurring at the floor of the solid). Composites were not cut.

17.6 Search Parameters

Search ellipses for the interpolation profiles are based on geology and observed continuity of the phlogopite host zone. A lower cut-off grade of 1 g/t was selected from evaluation of grade tonnage relationship at several cut off grades. The grade data was interpolated into block models using an inverse distance interpolator with a power of 2 (ID2).

The search ellipse radii used in the interpolation profile are presented in Table 22. The parameters defining the search ellipse are as follows: x Direction is along strike, y Direction is thickness, and z Direction is along depth of the mineralized zone. Further there is an axial adjustment for strike and dip of the mineralized zone of 10° around the x axis.

Table 22: Summary Of Search Ellipse Parameter Radii

Classification	x Direction	y Direction	z Direction
	m	*m*	*m*
Indicated	10	5	20
Inferred	16	5	30

Search parameters and sample interpolation restrictions included a minimum of 2 samples and a maximum of 6 samples per ellipsoid. High grade was capped at 7,325 g/t, which corresponds to the 97.5th percentile value of drill core assays.

17.7 Classifications

The resource for the Aappaluttoq Property has been estimated as Indicated and Inferred resources in accordance with CIM Estimation of Mineral Resources and Mineral Reserves Best Practises Guidelines. The resource estimation is prepared in accordance with NI 43-101 guidelines. The resources for the property have not been classified as measured due to a degree of reduced confidence in the geological database; specifically the lack of down-hole survey data, incomplete sampling throughout the mineralized zone, and lack of oriented down-hole structural data. It is recommended that future drill programs practice continuous sampling throughout the host zone intercepts with samples on either side to provide constraints on the mineralized solid.

17.8 Mineral Resource

This initial resource estimate was prepared by EBA from 6,457 m of drilling data and approximately 90 t of bulk samples collected on the property over the last several years and uses recently updated geologic interpretations for the host zone lithology. Due to limited sampling data and the distribution of sample locations resource evaluation and resource modeling was based on geology. Resource modeling was based on the knowledge that corundum mineralization was immediately tied to phlogopite and also with the gabbro unit. This understanding was derived from sampling that was conducted throughout the phlogopite, gabbro and units directly adjacent on the contact margins. Furthermore, from visual inspection of the diamond drill core in the field and in the lab it was noted that corundum was present visually in the phlogopite and gabbro units, and not present visually in the other units.

The mineral resource estimate comprises the integration of mineralization volumes, density, petrology and Total Clean Corundum content data obtained from diamond drilling and bulk sampling. Mineral resources are presented in Table 23.

Table 23: Indicated And Inferred Resources

Category	Volume	Tonnage(1)	Average Grade(2,3)	Average Grade(2,3,4)	Contained Corundum (2,3)	Contained Corundum(2,3,4)
	m^3	*t*	*g/t*	*ct/t*	*M.g*	*M.ct*
Indicated	59,110	189,150	313.33	1,566.65	59.27	296.33
Inferred	24,110	77,160	283.46	1,417.28	21.87	109.35

Notes:

(1) Densities are derived from specific gravity measurements of host lithologies and estimated for host zone based on specific gravity of corundum and average grade.

(2) Based on a Total Clean Corundum grades greater than 1.7 mm size fraction from mineralogical lab analysis.

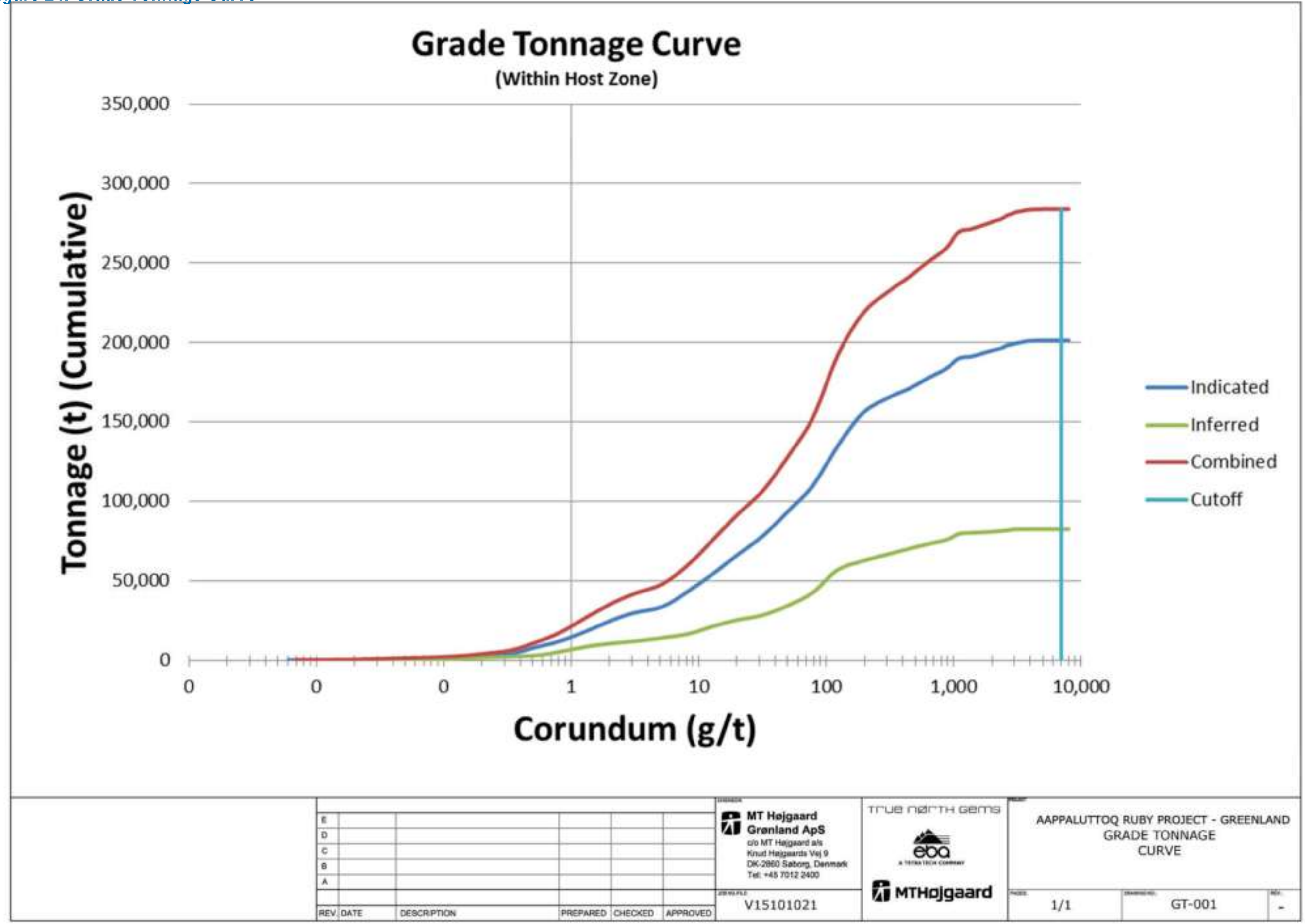
(3) Top cut grade of 7,325 grams per tonne (97.5 percentile), and a lower cut-off grade of 1 gram per tonne.

(4) One gram equals five carats.

Volumes and tonnages presented in the above table are rounded. Densities are derived from specific gravity measurements of host rock lithologies and are estimated for the mineralized zone based on specific gravity of corundum and average grade. This resource is based on Total Clean Corundum grades greater than 1.7 mm size fraction from mineralogical lab analysis. The top cut-off grade of 7,325 g/t (97.5 percentile), and the lower cut-off grade of 1 g/t, are based on a review of the grade tonnage curves (Figure 14) one gram equals five carats.

Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Figure 14: Grade Tonnage Curve



Bulk samples taken at surface were used to provide discrete data points within the database; however the influence of this data is limited by the same parameters used for all other samples in the data base. Tonnage from the bulk samples has not been subtracted out of the resource estimates.

All tonnages cited are for mineralized lithologies only.

Corundum resource grades are estimated for greater than +1.7 mm size fraction recovered after core sample preparation. This is nominally larger than the minimum size of gemstone rough that can be commercially polished, and which is consistent with the minimum sieve size used by the Company in evaluations reported on January 16 and 17, 2008. No distinction has been made in the calculation for variations in quality (gem, near-gem and non-gem), size or colour of the recovered corundum. No work was done on the resource to evaluate the consistency or distribution of gem or near-gem quality, and non-gem corundum within the mineralized zone intersected by drilling because of the small volume of corundum recovered from each of the drill core intercepts.

However, the colour and quality distribution analysis of corundum and valuation of the sorted rough ruby and pink sapphire was completed on the larger bulk samples from surface. The variations of the colour and quality of corundum greatly influences the value of the Total Clean Corundum recovered. The value of sorted and recovered ruby and pink sapphire from the surface bulk samples are discussed in section 16.3.3 Cutting and Valuations of 2006 Bulk Sample.

Table 24 presents a summary of grade distribution per lithology type for 634 drill core samples collected. The majority (80%) of sampling was conducted in the phlogopite unit and displays a grade distribution of 0-72,812.4 g/t, averaging 401.1 g/t. The 17 samples collected in the gabbro display a grade distribution of 0-20,993.0 g/t, averaging 2683.7 g/t. The 77 samples collected from the SAPGED unit display a grade distribution of 0-48,067.5 g/t, averaging 1192.4 g/t. There are 345 samples with zero grade.

Table 24: Summary Of Grade Distribution By Lithology Type

Lithology	Number of Samples	Zero Grade	Min of Grade	Max of Grade	Average of Grade
		#	g/t	g/t	g/t
GAB	17	7	0	20,993.0	2,683.7
GNS	4	2	0	468.0	215.6
PARG	6	0	29.3	918.3	320.6
PEG	7	5	0	13.6	3.1
PHLOG	511	285	0	72,812.4	401.1
SAPGED	77	39	0	48,067.5	1,192.4
UM	12	7	0	9,914.9	1,069.9
Grand Total	**634**	**345**	**0**	**72,812.4**	**564.7**

17.9 Mineral Reserves

The open pit design is based on a selected shell from a series generated using Gemcom's Whittle 4X software. The pit design uses geotechnical and practical mining parameters which should allow safe and efficient extraction of ore. The shell selection and pit design is based on Indicated Mineral Resources only. Table 25 below shows the mineral reserve.

Table 25: Aappaluttoq Open Pit Mineral Reserves

	Reserve	Corundum	Contained Corundum
	t	g/t	M.g
Proven	-	-	-
Probable	161,700	350	56.6
Total	161,700	350	56.6

The Mineral Reserves contains 95% of the Mineral Resource in reference to total corundum, and 85% in reference to tonnage.

The Whittle 4X parameters were initially based on the economic parameters of the project. As these parameters did not produce a wide range of shells, the parameters were magnified to produce a wide range of pit shells. A number of shells were evaluated and the final selection was based trade-off between economics, mine life and risk reduction. A smaller shell could result in slightly better economics, but shorter mine life and increased risk.

Key assumptions used for the economic analysis and Mineral Reserve are shown in Table 26 below.

Table 26: Key Economic Parameters

Item	Units	Value
Mining productivity	t/d	3,300
Diesel price	$/L	1.08
Process productivity	t/d	117
Site recovery rate	%	95
Ruby rough price	$/gr	0.39
Sapphire rough price	$/gr	0.39
Ruby polish price	$/ct	32.34
Sapphire polish price	$/ct	25.60
Acid washing upgrade	%	65
Polishing retention	%	9.3
Discount rate	%	8

For the Mineral Reserve, the percentages of gem and near-gem rubies and sapphires contained in that corundum are based on the percentages determined from the analysis of the B1 and B2 bulk samples taken in 2006 and 2007. This information is shown in Table 27. The combination of the B1 and B2 were the most representative samples to use in the estimation of gem distribution characteristics as there was increased knowledge of the deposit and greater scrutiny of the samples. These samples are detailed in sections 10.1 Surface Sampling and 12.2 Bulk Sample. Valuations are only applied to the gem and near-gem portion of the corundum. No value was applied to the non-gem material.

Table 27: Gem Distribution Characteristics of B1 and B2 Samples

Category	Colour	B1 sample		B2 sample		Combined	
		Total weight	Distribution	Total weight	Distribution	Total weight	Distribution
		g	%	g	%	g	%
Gem	Red	744	2.6%	426	0.4%	1,170	0.9%
	Pink	3,669	13.0%	8,035	7.9%	11,704	9.0%
Near-Gem	Red	3,049	10.8%	2,248	2.2%	5,297	4.1%
	Pink	4,356	15.4%	28,681	28.1%	33,037	25.3%
Non-Gem	Red	1,711	6.1%	12,513	12.3%	14,224	10.9%
	Pink	14,701	52.1%	50,217	49.2%	64,918	49.8%
Total	Red	5,504	19.5%	15,187	14.9%	20,691	15.9%
	Pink	22,725	80.5%	86,933	85.1%	109,658	84.1%
Gem	Both	4,413	15.6%	8,461	8.3%	12,874	9.9%
Near-Gem	Both	7,405	26.2%	30,929	30.3%	38,334	29.4%
Non-Gem	Both	16,412	58.1%	62,730	61.4%	79,142	60.7%
Total	Both	28,229	100.0%	102,120	100.0%	130,349	100.0%

18 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information applicable for this PFS.

19 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

19.1 Hydrology Assessment

A preliminary hydrological assessment for the Aappaluttoq Ruby Project site has been completed. The scope of this work component includes a review of background information relevant to the site hydrology and a preliminary site water balance analysis

19.1.1 Watershed Description

The Aappaluttoq ruby prospect is situated on a peninsula protruding from the southern shore of Lake Ukkaata Qaava (formerly called Lake Katrina), located about 20 km southwest of the town of Qeqertarsuatsiaat. The lake is split into two main basins by two peninsulas connected by a shallow sill. Since the prospect reaches down beneath the lake surface, drawing down the lake by about 10 m during mining operation is proposed. Once the mining operations are completed, the lake will be allowed to return to its natural level, covering the processed materials entering the lake by at least 10 m of water.

The total catchment area of Lake Ukkaata Qaava is about 6.2 km^2. The lake surface is 230.6 m above sea level, and its surface area is about 1 km^2. The upper basin, located in the southern portion of the lake, has a surface area of about 0.6 km^2, and has a maximum depth of about 45 m. The deeper lower basin, located in the northern portion of the lake, has a surface area of about 0.4 km^2, and has a maximum depth of about 50 m.

Outflow from Lake Ukkaata Qaava is through a channel in the north of the lower basin. The watershed elevation varies from approximately 600 m near the headwaters to 230 m near the outlet of the lake. The average watershed elevation is about 300 m above sea level.

The monthly outflow from the lake were calculated with peak outflows occurring between May and July and between September and November, likely due to ice thaw and snowmelt in the spring and due to rain or rain on snow events in late summer and autumn. Minimum outflow occurs during the winter months usually from December to April, while the lake is covered by ice.

Photograph 1 shows a shallow water scene from Lake Ukkaata Qaava. The green flora is a type of moss. No fish life was found in the lake.

Photograph 2: Lake Ukkaata Qaava

19.1.2 Site Water Balance

For a more complete understanding of the site hydrology, or in this case, the water balance of the lake prior to and during mining operations, a monthly water balance model has been developed. Major input to the model includes the storage-elevation relationship of the lake, runoff, evaporation and monthly inflow pattern. The net inflow to the lake was routed through the lake to determine the resulting lake level by using the established stage-discharge relationship of the outlet channel for the existing condition and assuming a reasonably sized outlet structure during the operation of the mine.

The water balance model suggests that during the proposed mining operation, sufficient water will be available for the operation of the mine.

Once the mining operation is completed, the plan is to allow the lake to return to its natural level. Based on an average annual runoff volume calculated as part of the flow routing exercise and the amount of storage required to fill up the lake from the mine operating level, such process will likely be completed in 3 years.

19.2 Open Pit Geotechnical Analysis and Slope Angles

Geotechnical work on the property has been limited to the geological logging of drill holes. These logs along with core photos were used to calculate the Rock Mass Rating (RMR) which is a standard means of measuring rock strength in the mining industry.

Table 28 indicates the rock mass is classified as "good rock". Additionally, the RMR values estimated from the GSI are very similar to the RMR estimated from the sum of the rock mass parameters.

Table 28: Summary Of RMR Statistics

Rock Mass Characterization	Mean	Minimum	Maximum	Standard Deviation
RMR 1976	76	40	92	8.7
RMR 1976 (estimated from GSI)	70	30	95	13.5

19.2.1 Slope Angles

Berm stability of the pit walls was assessed using the kinematic stability of the structural geology with respect to the proposed pit wall orientations. This approach was chosen since the rock mass at the site consists of homogeneous strong, massive, gabbro and ultramafic rocks with a well-defined foliation, and hence stability is governed by structural-controlled modes of failures. Bench face angles, where rock structure is the primary failure control, depend on the wall orientation.

Figure 15 shows the orientation of foliations. Figure 16 shows the output of the stereographic projection plots for the stability of the southeast and northwest pit walls. The figures illustrate that batter face angles steeper than 65° will become potentially unstable for sliding on the SE wall, and toppling becomes significant at angles steeper than 70° on the NW wall. The lack of additional structural information does not allow assessment of the kinematic stability of the NE and SW pit walls. The effects on stability due to foliation orientation will be less on these two walls in comparison to the SE and NW pit walls so this does not represent a shortcoming in the pit design.

19.2.2 Overall Slope Stability

Even though the stability of the pit walls is governed by the rock structure, the likelihood of generating large-scale (deep-seated) circular failure through the rock mass was assessed. Geotechnical characteristics of the rock mass, the height of the pit walls, blasting practice and the stress conditions at the toe of the slope can all impact the stability of the overall pit wall.

Stability analysis was performed for a vertical section perpendicular to the planed pit wall. The analysed cross section has a height of 65 m and an overall angle of 65°. A graphical representation of the analysis is included in Figure 17.

The determined static factor of safety is 1.7. This factor of safety is higher than the acceptable static factors of safety of 1.3 commonly adopted in open pits.

Figure 15: Foliation Orientations

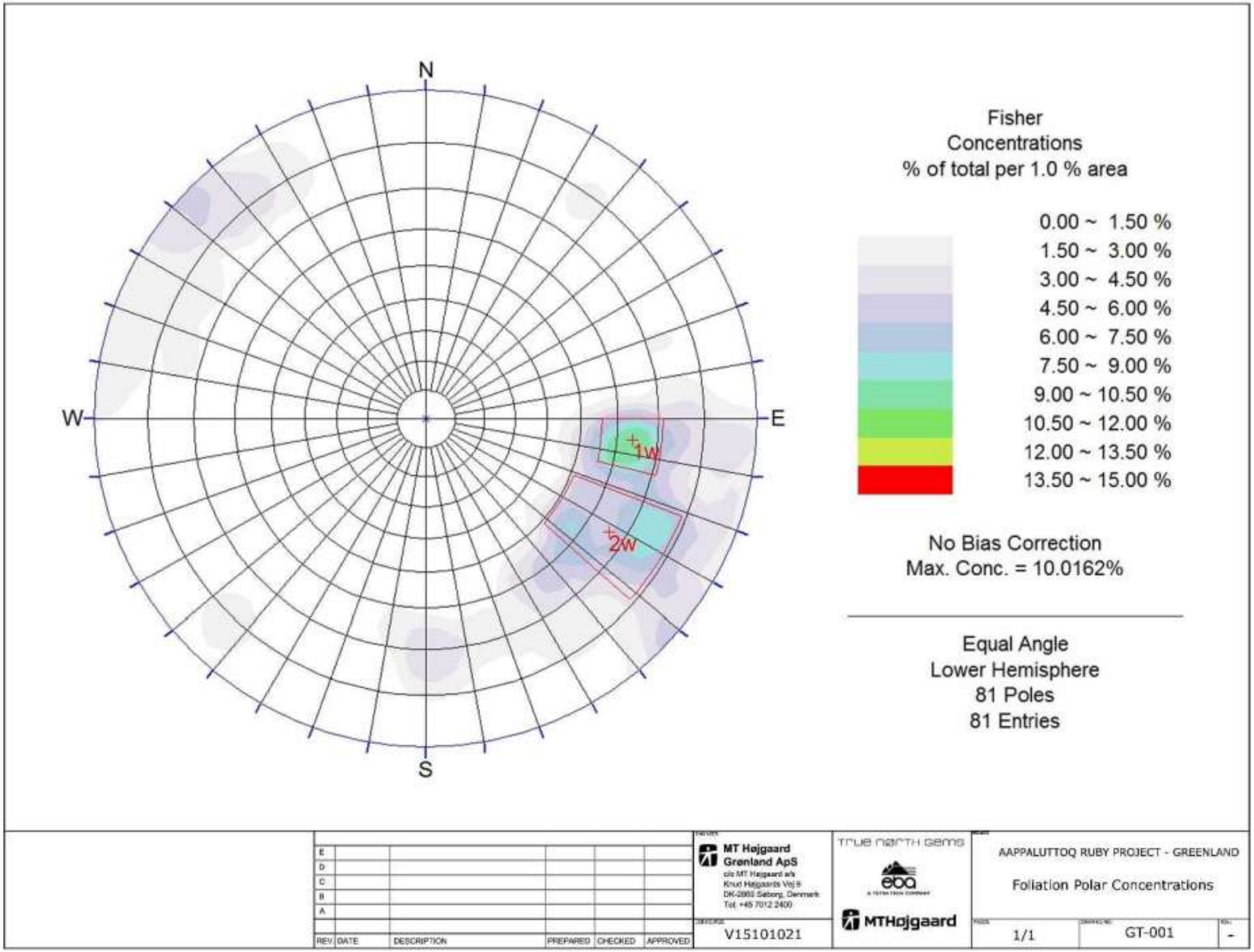


Figure 16: Stereographic Stability Projections

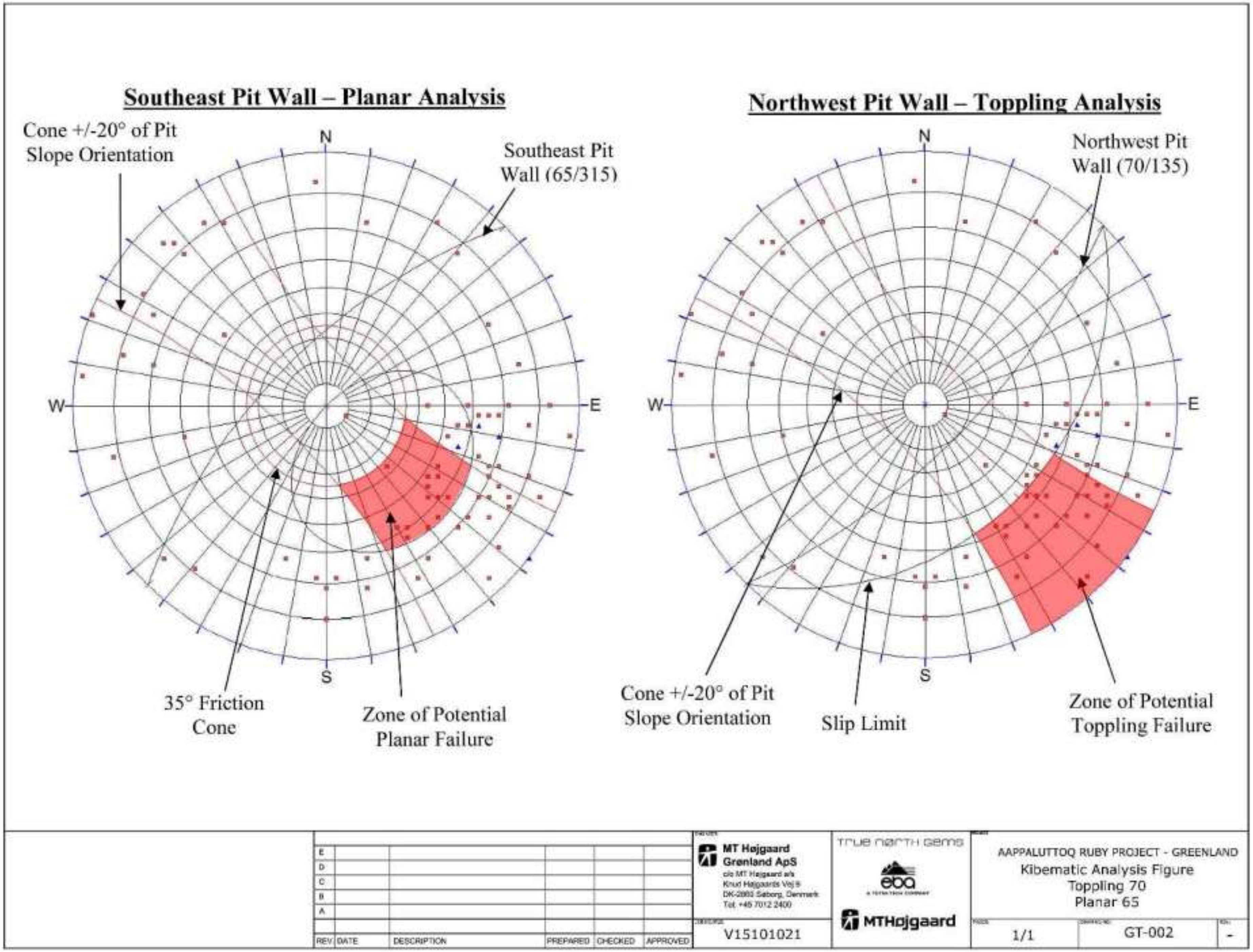
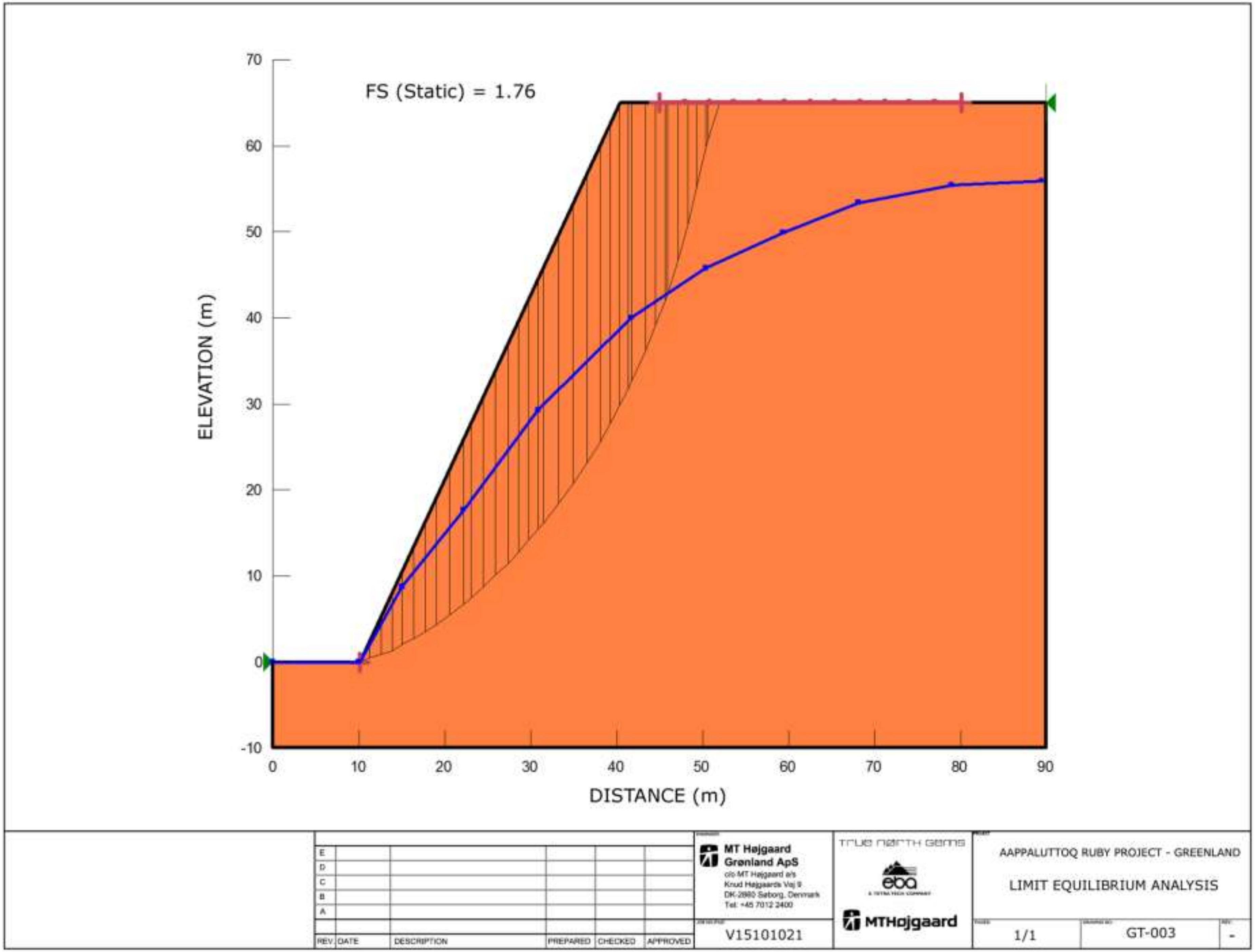


Figure 17: Slope Stability Analysis



19.2.3 Catch Berm Widths

The berm width consists of a required catch width and an "overbreak distance." The purpose of the catch bench is to arrest potentially hazardous rock falls and contain any spillage from the benches above. The back-break distance is incorporated into the design to account for reduction of the bench width due to blasting effects and scaling.

The required catch width was adopted from the Oregon Department of Transportation (2001) Rockfall Catchment Area Design Guide (Pierson, Gullixson and Chassie 2001). The required width to contain 95% of the rock fall impacting the bench from a 2V:1H, 24 m high bench is 4.5 m. It should be noted that the required catch bench width to restrain the falling rocks from rolling downslope after these impact on the bench is larger. A containment value of 95% is used as the area is a mining operation with low exposure to people and equipment.

The overbreak distance depends on the blasting effects and scaling work after blasting and mucking. Scaling is to remove loose rocks resulting from blasting as well as potential localized unstable rocks formed at the bench crest once the bench face is exposed. The localized unstable rocks at the bench crest are formed due to the variation of the dip angles of the rock fabric. For this purpose a minimum back-break allowance of 2 m and a minimum bench width of 6.5 m is recommended.

19.2.4 Design Criteria

Based on the information found by the above geotechnical analysis, the optimum bench and overall pit angles at prefeasibility level are presented in Table 29 for a 65 m pit wall divided into 24 m high benches.

Table 29: Bench Face And Pit Wall Heights And Angles

Wall	Bench Face			Overall Pit Wall	
	Height *m*	Width *m*	Angle *degrees*	Height *m*	Angle *degrees*
SE Wall	24	6.5	65	65	56
NW Wall	24	6.5	70	65	61
NE Wall	24	6.5	75	65	65
SW Wall	24	6.5	75	65	65

Notes:
Overall Pit angle at locations where ramp is located would be approximately 4 degrees flatter.
Overall angle refer to angle measured from the toe of the slope to the topmost crest.

19.3 Ore Definition, Dilution and Recovery

As it is expected that it will be difficult to differentiate between ore and waste, any material with grade will be mined and processed. For mine planning purposes, this is assumed to be any material with a grade of greater than 1 g/t. With other mining commodities, grade control drilling would allow the measurement of grades. Grade control drilling is not expected to be effective for corundum as it is impractical to assay and determine grades. This has impacts on all aspects of mine planning.

As measurement of insitu grades is impractical in a mining environment, all mineralized material will be mined and processed. This method introduces significant dilution and positive ore recovery. The life of mine economic cutoff grade is back-calculated to be 178.5 g/t which indicates an economic dilution of 214%. This is different from mining dilution which is unavoidable dilution caused by mining method.

For mine planning purposes, the geological block model was modified to use whole block analysis, which dilutes the contained grade through the whole block. This is a standard mine planning procedure.

The normal concept of a Smallest Mining Unit (SMU) is not applicable due to the dilution concepts above.

19.4 Open Pit Optimization

Optimizations were run using Gemcom's Whittle 4X software. Whittle 4X creates a series of Lerchs Grossman (LG) pit shells at variations of a base commodity price. The selection of one or more of these shells, taking into account practical constraints, creates an optimal way to mine the deposit. Typically, small variations from the optimal case do not influence economics, which is true for this deposit.

Optimizations for the open pit were not run in the standard manner of assigning costs per tonne. The Whittle 4X parameters and resulting optimisations were initially based on the economic parameters of the project. As these parameters did not produce a wide range of shells, the parameters were magnified to produce a wide range of pit shells. A number of shells were evaluated and the final selection was based trade-off between economics, mine life and risk reduction. A smaller shell could result in slightly better economics, but shorter mine life and increased risk.

19.5 Open Pit Mine Design

The open pit design was based on the generated Whittle 4X shell and geotechnical design parameters. The approximate dimensions of the final pit are 200 m × 160 m to a depth of 78 m (Figure 18).

The roads and haul ramps are designed with a total width of 7 m to give sufficient width for single lane 35 t capacity articulated trucks. Single lane ramps are viable due to the shallow pit depth and low number of trucks. The last bench has steeper wall angles as the mining at this time will proceed quickly and the pit walls will not need to stable long term. Open pit parameters are shown in Figure 19 and summarized below:

- Bench height of 24 m
- Mining flitch height of 4 m
- Berm width of 6.5 m
- Minimum mining width of 25 m
- Single lane ramp width of 7 m
- Typical centerline ramp gradient of 12.5%

Safety bunds along ramps within the pit will be 1 m high to provide adequate safety protection for haul trucks and auxiliary equipment during mining. Standard berms will be half the diameter of the largest vehicle wheel, Volvo A35 haul trucks.

The pit will be developed with no major pushbacks. A detailed staged design was found to be unnecessary to meet ore feed requirements. High grade areas will be targeted during operations. The overall stripping ratio is 16:1. Figure 20 shows the Mineral Resource in relation to the pit design.

Figure 18: Open Pit Design

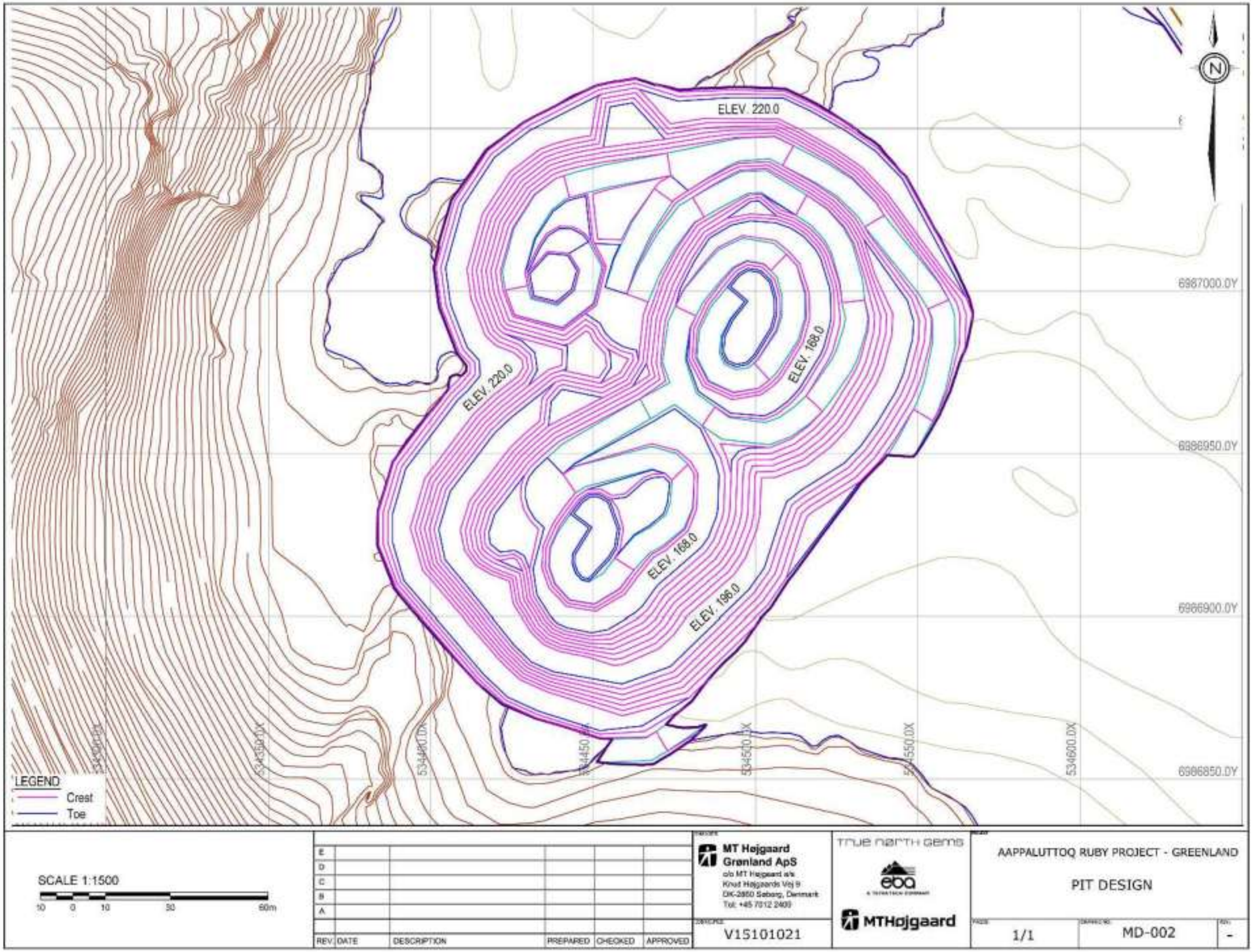


Figure 19: Typical Cross Section of Pit Wall

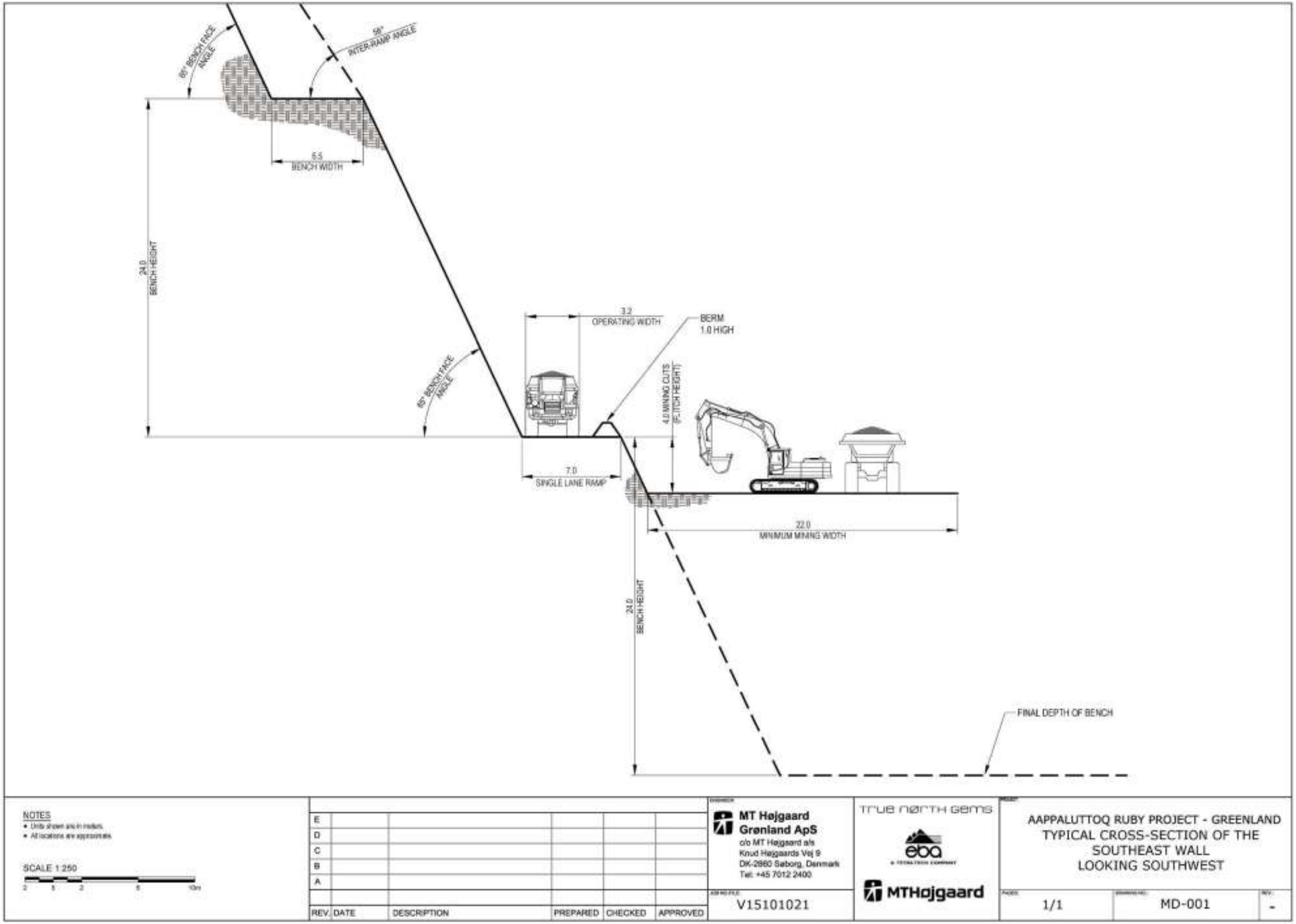
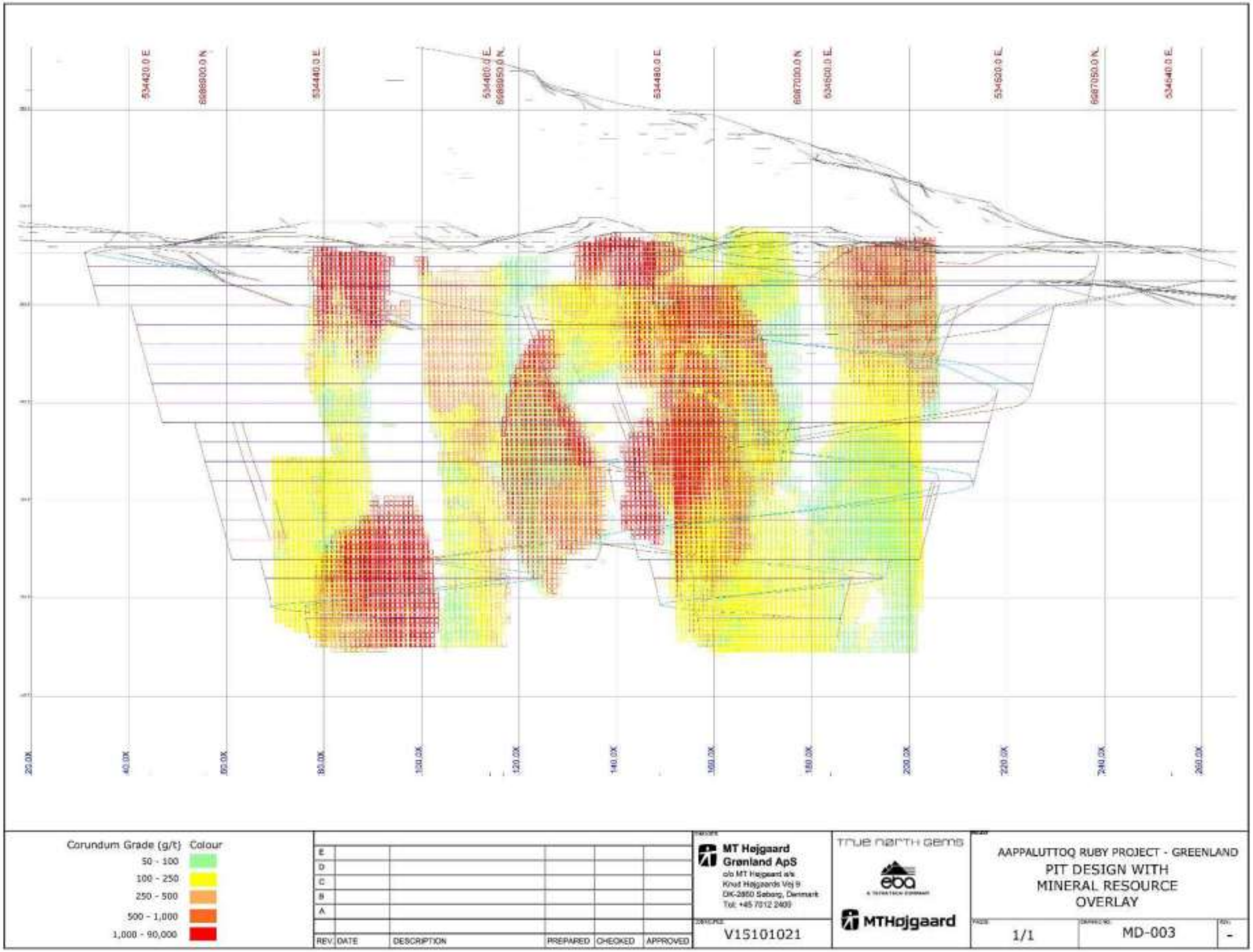


Figure 20: Pit Design Against Mineral Resources



19.6 Open Pit Equipment Selection

Equipment selection can have a substantial effect on the operating costs and production rates of an operation. Due to the remote location, equipment needs to be oversized to allow for production catchup in the advent of extended down time.

One 47 t excavator will be required during production which should have a backhoe configuration to provide for greater flexibility.

Haulage will be carried out by four 35 t capacity dump trucks.

Production drilling will be provided by a Sandvik Panterra 1500 rig capable of drilling up to 140 mm diameter holes. This drill is also able to drill inclined pre-split holes.

Other mobile fleet equipment includes a 25 t tracked dozer, 14 t grader, 19 t wheel loader, lighting plants, water truck, fleet maintenance vehicles and a Manitou. Table 30 shows the complete mobile fleet.

Table 30: Mobile Fleet

Item	No.	Model
Excavator	1	Hitachi ZX470
Articulated dump truck	4	Volvo A35
Tracked dozer	1	CAT D8
Articulated drill	1	Sandvik DX 1500
Grader	1	Cat 14M
Manitou	1	Manitou MT 732 - Attachments: Forks, blade, bucket, jib, ANFO cage
ANFO mixer	1	Volvo A25E with customised body
Water truck with water cannon	1	20,000 L prime mover
Light vehicle - mining	1	Toyota Hilux, 4 person cab
Light vehicle – explosives	1	Toyota Hilux with wooden carry compartments
Fuel truck	1	5,000 L diesel tank, 600 L lube tank & air compressor
Service truck	1	4 t crane, air compressor
Water standpipe & pump	1	25 HP
In-pit pumps	3	3 HP, max 24.4 m head
Pit dewatering pump	1	75 HP, max 76 m head

19.7 Open Pit Mining Operations

The mining operation is assumed to be owner managed and maintained. The Company will hire all operators, maintenance and technical staff required. Some specialized work may need to be outsourced to contractors or consultants as and when needed. Section 19.19 Labour Requirements details the labour requirements for the operations.

The operations will be day shift only, and no lighting plants will be required for the mining operations.

19.7.1 Rock Breakage

The host rock is classified as hard, competent rock and will require drill and blast for rock breakage. Drilling in ore zones will be done using a tracked drill rig with holes 4.8 m deep and 76 mm in diameter. This drill pattern will have a burden and spacing of 2.5 m by 3 m respectively. A standard ANFO mixture will be used as the blasting agent, with standard industry NONEL detonators and an emulsion booster. Blasts in ore will range in size depending on operational requirements, but typically be 1,500 m^3.

Waste material will be drilled with an articulated, tracked drill rig with holes up to 9.6 m deep and 102 mm in diameter. This drill pattern will have a burden and spacing of 3 m by 3.5 m respectively. This includes a subdrill of 1.6 m. Standard ANFO will be used along with standard industry NONEL detonators and an emulsion booster. Blasts will range in size depending on operational requirements, but typically be 5,000 m^3.

Blasting will occur as required to maintain production and is expected to occur several times per week. Explosives and detonators will be stored in an onsite magazine. Blasting will be carried out under the supervision of a certified and trained operator during daylight only.

During blasting, the pit and a surrounding area of 250 m will be evacuated for safety reasons. After firing, blasts will be inspected for misfires. Misfires will be rectified under the supervision of an experienced and certified operator prior to allowing workers back within the pit.

Walls of the pit will be pre-split to improve geotechnical stability with a spacing of 0.8 m. Ramps will be drilled and blasted to grade, which is typically 12.5%.

19.7.2 Load and Haul

After blasting, waste material will be loaded by a 47 t, 2.1 m^3 bucket excavator into 35 t capacity articulated dump trucks. The waste material will either be used for construction activities or taken to the lake for sub aqueous disposal. The excavator will load 4 m benches into trucks with an average payload of 32 t. Trucks will be under loaded to improve mechanical reliability and to allow for a 12.5% grade of pit ramps.

When possible, ore will be direct tipped into the primary jaw crusher feed hopper. It is expected that approximately 60% of material will require re-handling. ROM stockpiles will be located at the process area. They will be used to temporarily hold ore material which the wheel loader will load into the primary jaw crusher. Blasted stocks in the pit will ideally be sufficient for four weeks production.

19.7.3 Ancillary Operations

Roads will be maintained by a 14 t grader. Roads and pit ramps will be maintained to minimise equipment damage and maximise operator safety. Haul roads will have 2-3% cross-fall to effectively shed water from rainfall or groundwater. A water truck will be used when required for dust suppression on haul roads.

19.7.4 Dewatering

Each time a bench is developed in the pit, a 6 m deep, 10 m × 10 m sump will be blasted into waste. The sump will be used to dewater the pit area with water being pumped to a storage pond at the pit crest. From there, it will be further pumped by a diesel pump into the lake or used for dust suppression. The pond will be regularly monitored for contaminants and if necessary will be circuited through an oil-water separator for cleaning.

Small portable diesel powered "puddle jumper" pumps will be used to pump water into the sump from areas of the pit subject to water build-up.

Detailed engineering, pump sizing and water flow analysis has not been completed.

19.7.5 Pit Ramps and Mine Roads

The ex-pit mine road will be approximately 0.15 km long. It will be constructed in order to transport ore material from the mine site (Aappaluttoq deposit) to the process facilities. The road will be constructed as a single-lane gravel paved road.

Pit ramps will have a typical centerline gradient of 12.5%. The ramps will be single lane which is possible as there will be sufficient pull outs and a small truck fleet.

The proposed alignments of the access and mine roads are shown in overall site layout, Figure 26, located in section 19.5 Open Pit Mine Design.

The dominant traffic will be haul trucks (articulated haulers) up to 35 t and light vehicles for transport of personnel. The maximum permitted axle load is 20 t and maximum total vehicle load 61 t. If special vehicles with heavy loads are required, the situation will have to be assessed.

The road pit ramps will be established as a 5.0 metre wide single-lane gravel paved road. The total width of the road inclusive of shoulders and safety bund will typically be 7.0 metres.

Speed limits will be 30 km/h for heavy equipment including haul trucks and 40 km/h for other vehicles.

19.7.6 Grade Control

The ore zones are generally steeply dipping. Grade control drilling will not be done for reasons outlined in section 19.3 Ore Definition, Dilution and Recovery. Visual control, mapping and face samples will be used by the geologist to define ore contacts.

19.8 Site Schedule

19.8.1 Work season

The mining operation will be a typical open pit operation. Mining will use conventional blast, load and haul equipment mining up to 450,000 t of total material over an eight month operating season from beginning of April to the end of November each year.

Ore production will increase gradually from approximately 1,000 t in the first year (to satisfy plant commissioning and initial operation) to nearly 25,000 t in the latter years. Increased production is achieved by extending the work season from four months in early years to 8 months in later years (Table 31).

Manpower is expected to remain constant over the operating years at 15.5 persons; the "half" person being the lead maintenance hand who will also supervise the process plant maintenance.

Table 31: Open Pit Working Schedule

	Year	0	1	2	3	4	5	6	7	8	9
Ore tonnes	*t*	0	1,103	6,923	19,769	21,044	21,875	21,820	21,530	23,056	24,484
Waste tonnes	*t*	0	236,572	213,180	425,625	420,830	399,989	335,284	237,051	179,330	167,672
Total tonnes	*t*	0	237,674	220,103	445,394	441,873	421,864	357,104	258,581	202,386	192,156
Work season	*weeks*	0	18	18	35	35	35	35	35	35	35
Days per year per crew	*days*	0	84	84	161	161	161	161	161	162	163
Scheduled hours per crew	*hrs*	0	1,008	1,008	1,956	1,956	1,956	1,956	1,956	1,956	1,956

19.8.2 Process Capacity

The plant will operate between 4 to 8 months per year. The initial capacity is 78 tpd which is dependent on the capacities of the mineral jigs. Once further testwork and commissioning is completed, an expansion will be completed in year 3 (2014) to increase capacity to 117 tpd. Table 32 shows details of processing over the life of mine.

The plant is assumed to have an availability of 85% and a utilization of 85%, which is below industry standards. This allows a contingency for training and to account for the dynamics of a small size operation.

The operations will stop during the last half-hour of each shift. The operators will leave and the supervisor, process engineers and security will seal and transport the security bins containing rough concentrate to the vault using a floor crane.

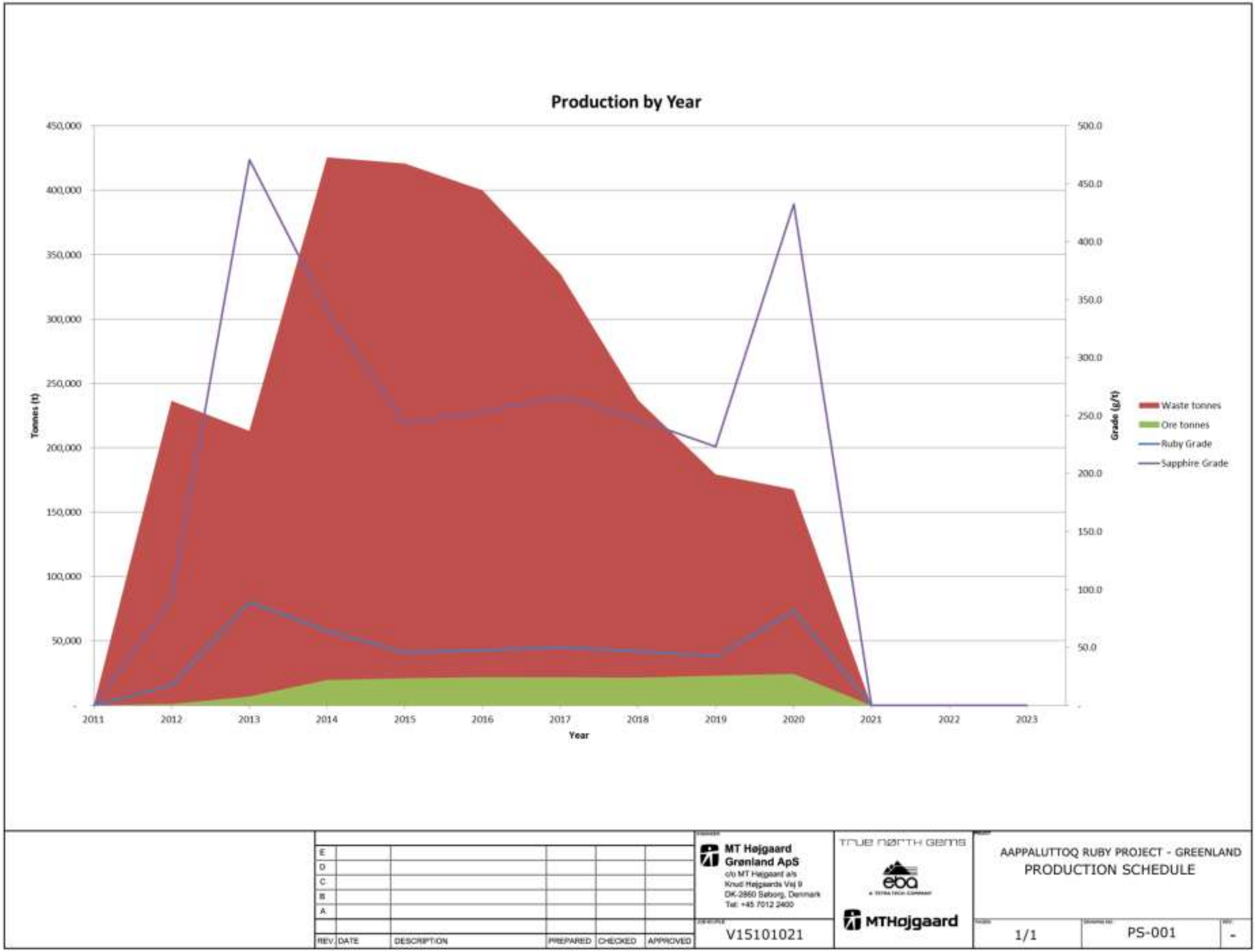
Table 32: Ore Feed Requirements

	Year	0	1	2	3	4	5	6	7	8	9
Work season	*week*	0	18	18	35	35	35	35	35	35	35
Days per year per crew	*day*	0	63	63	122	122	122	122	122	122	122
Schedule hours per crew	*hr*	0	756	756	1,464	1,464	1,464	1,464	1,464	1,464	1,464
Crews	#	2	2	2	2	2	2	2	2	2	2
Process capacity											
Process capacity	*tpa*	0	1,229	9,828	28,548	28,548	28,548	28,548	28,548	28,548	28,548
Ore feed	*tpa*	0	1,103	6,923	19,769	21,044	21,875	21,820	21,530	23,056	24,484
Process capacity	*tpd*	0	10	78	117	117	117	117	117	117	117
Ore feed	*tpd*	0	9	55	81	86	90	89	88	94	100
Material shipped to Nuuk											
Ruby	*kg*	0	18	584	1,206	920	992	1,044	952	924	1,898
Sapphire	*kg*	0	95	3,096	6,392	4,875	5,259	5,533	5,045	4,895	10,061
Matrix material	*kg*	0	73	2,392	4,939	3,766	4,064	4,275	3,898	3,782	7,774
Total	*kg*	0	186	6,072	12,536	9,561	10,315	10,851	9,894	9,601	19,734
Final recovered gem & near gem											
Rough ruby	*kg*	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Rough sapphire	*kg*	0.0	15.5	505.2	1,043.1	795.6	858.3	902.9	823.3	798.9	1,642.0
Total rough	*kg*	0.0	15.5	505.2	1,043.1	795.6	858.3	902.9	823.3	798.9	1,642.0
Polish ruby	*ct '000*	0.0	2.6	85.3	176.1	134.3	144.9	152.4	139.0	134.9	277.2
Polish sapphire	*ct '000*	0.0	10.9	354.0	730.9	557.4	601.4	632.6	576.8	559.8	1,150.5
Total polished	*ct '000*	0.0	13.5	439.3	906.9	691.7	746.3	785.0	715.8	694.6	1,427.6

19.8.3 Material Movement Schedule

Mining rates are within the capacity of the mining fleet. In 2017 and 2018, an additional truck is required in order to meet the increase in ore feed requirements. Ore grades are significantly higher at the start and end of operations. Figure 21 shows the mining schedule and grades over the mine life.

Figure 21: Production Schedule



19.9 Tailings and Waste Rock Strategy

The objective of the mine tailings and waste rock plan is to minimise environmental and visual impacts. The major causes of environmental impacts normally encountered in mines are Acid Rock Drainage (ARD), Metal Leaching (ML) and reactive chemical issues.

The mine plan aims to minimise these impacts by using a sub-aqueous disposal technique which has been proven internationally. Minimising these impacts at Aappaluttoq is made straightforward as both the tailings and waste rock are generally benign in terms of ARD and ML. In addition, the processing methodology is purely mechanical and does not rely on any foreign chemicals.

Tailings from the process operations and a majority of mining waste rock will be deposited into the nearby lake, Lake Ukkaata Qaava. This is an ideal environmental solution as the lake is not fish bearing, and subaqueous disposal suppresses any potential acid and metal leaching and minimises visual impacts on mine closure.

19.9.1 Tailings

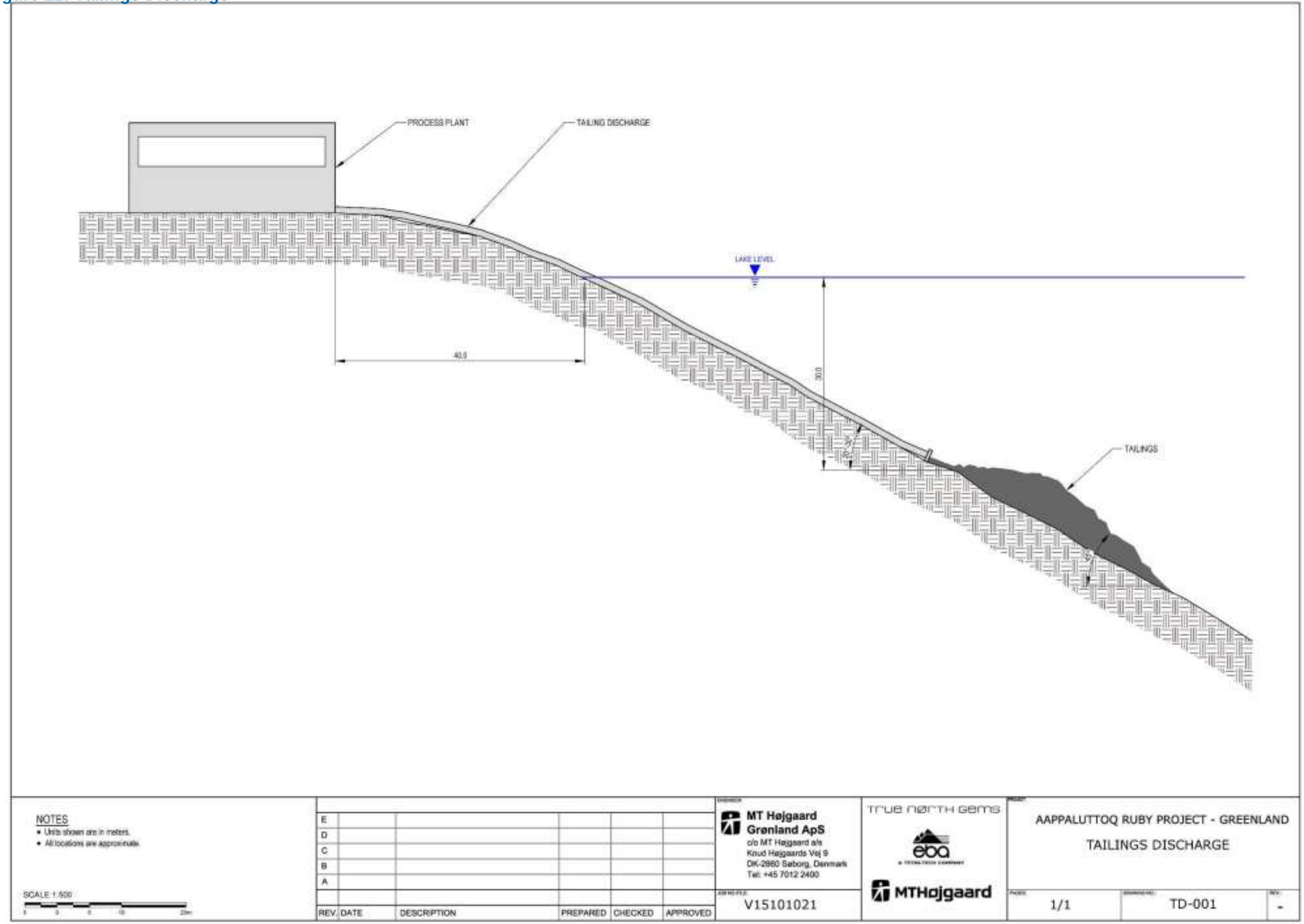
Processing operations will generate tailings solids at a rate ranging from 78 tpd up to 120 tpd. Ten kL of water will also be dis-charged for every tonne of tailings solids. Therefore, the approximate daily volume of tailing solids discharged into the lake will range from 25 m^3 to 65 m^3, and the daily volume of the liquid fraction of the tailings will range from 800 m^3 to 1,200 m^3, corresponding to between 0.01 and 0.02% lake volume of the East Basin, and 4% and 8% inflow volume of the lake. The volume of inflow into the East Basin has not been determined, although topographic estimates indicate that approximately 85% of the lake catchment is connected to the East Basin.

The tailings effluent is proposed to be discharged into the East Basin of Lake Ukkaata Qaava. The deep spot in the East Basin, approximately 50 m deep and located on the eastern side of the basin, has the steepest sloping bed along the southeast shoreline, which makes the best location for tailings discharge so as to minimise accumulation of tailings solids along the tailing effluent's path of descent. The steepest slope along the bank is 20-25˚. However, the tailings solids will be mainly gravels and the angle of repose for gravels is 35-40˚. Thus the gravels could potentially settle out of the tailings stream under water and pile up on the slope until the gravel pile becomes steeper than the angle of repose and starts slumping again.

The current proposed plan is to discharge the tailings effluent underwater with the pipe outlet deeper than the pynocline. Discharging the tailings effluent at 30 m or deeper would be acceptable, assuming the pynocline would form and be maintained at depths no deeper than 30 m, such that containment of dissolved contaminants or leachate metals, if any, in the lower part of the water body during the summer operation period would be achieved (Figure 22).

With regard to the water quality in the lake, the physical setup of Lake Ukkaata Qaava is such that the East Basin acts as a primary settling pond for tailings solids with a shallow surface flow into the West Basin. The West Basin then acts as a secondary settling pond. Tests show that the gravels are relatively clean and will generate minimal amount of ARD and ML, hence the water quality at the outlet will not be significantly affected. Also, from the perspective of suspended solids concentration, water quality will not deteriorate significantly due to the long residence time (~3.7 years) in which most tailings solids settle out of the water column.

Figure 22: Tailings Discharge



19.9.2 Waste Rock

After appropriate screening, non-ARD/ML waste rock from the initial stages of mining operations will be used for the construction of roads, building pads and two flood protection dykes. The dykes will allow pit expansion and protect the pit from the risk of flooding.

Waste material in excess of that required for construction will be backfilled into Lake Ukkaata Qaava. The risk of material slumping may require a dozer to push each load off, rather than direct dumping into the lake (short dump and push). This will be further evaluated during mining operations and may vary depending on how blasting is carried out.

Backfill will progress from the eastern basin side of the lake which should eliminate any risk of sediments polluting the downstream watershed.

Depending on the construction quality of the rock some non-ARD/ML material may be stockpiled on the surface for use in future earthworks such as roads and port maintenance.

19.9.3 Final Waste Dump

The total amount of waste material which will be deposited into Lake Ukkaata Qaava is 2.7 Mt (1.3 M.m^3). This represents 10.2% of the after-mining total volume of the east basin (12.7 M.m^3).

19.9.4 Stability

There are no reasons to expect that there will be stability issues with the mine waste after remediation. During operations, there may be occasional slumping caused when mine waste is dumped into water. These issues will be managed on site by careful monitoring to avoid safety issues.

19.9.5 Rock Types

Country host rock, regionally mapped as gneiss, represents a substantial proportion of the rock in the mine plan. The ARD and ML characteristics are assumed to be similar to gneiss sampled as part of the ARD/ML studies, but this is yet to be confirmed. This is a conservative approach as the gneiss material has a low threshold of 0.1% sulphide to be classified at PAG material. See section 19.10 PAG/NAG Differentiation for further details of ARD/ML issues.

Figure 23 and Figure 24 show the individual rock types that are being mined on a yearly schedule. Figure 25 shows the rock types in categories of PAG/ML, NAG/ML and country rock.

Figure 23: Rock Type by Year (Bar Graph)

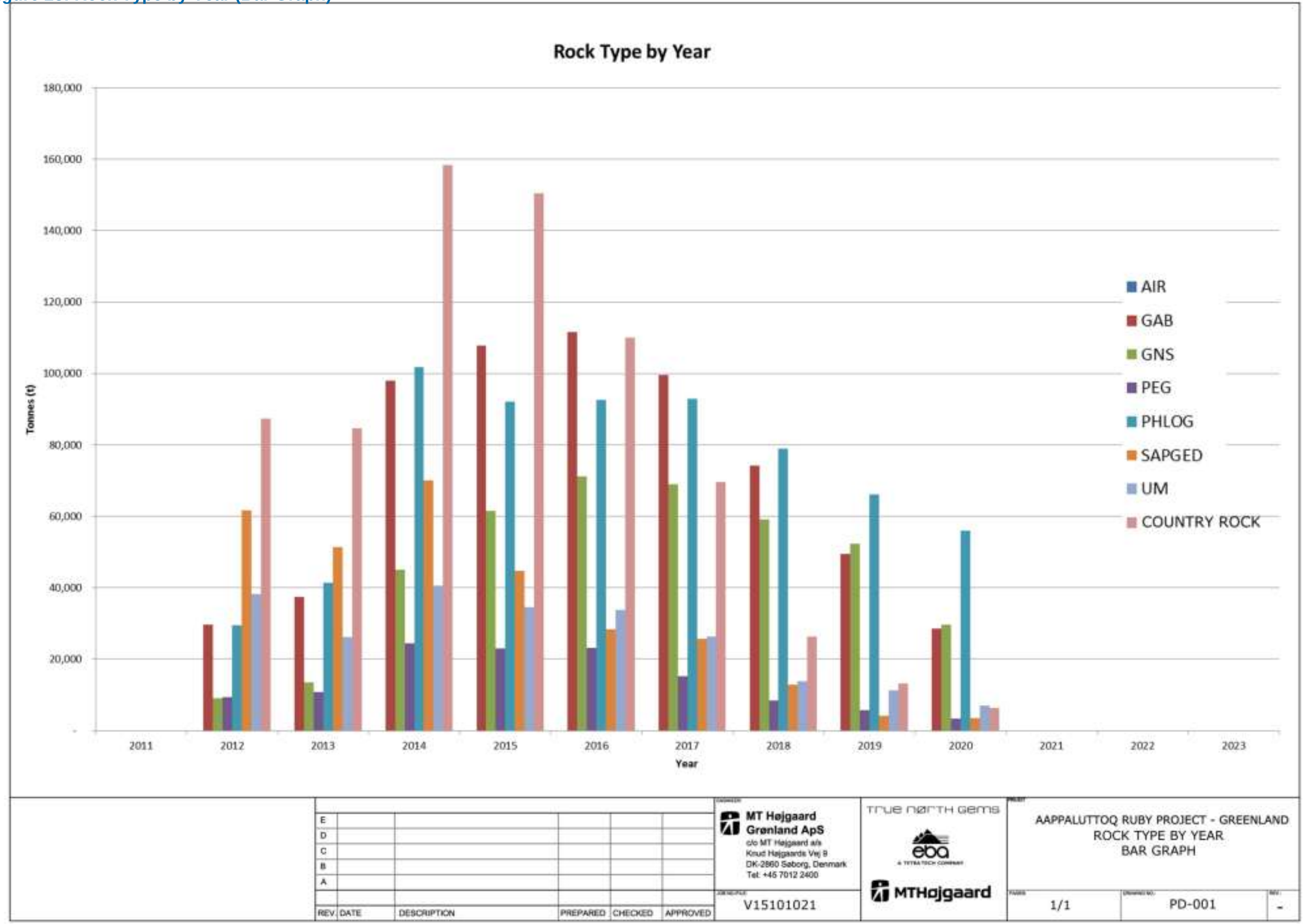


Figure 24: Rock Type by Year (Line Graph)

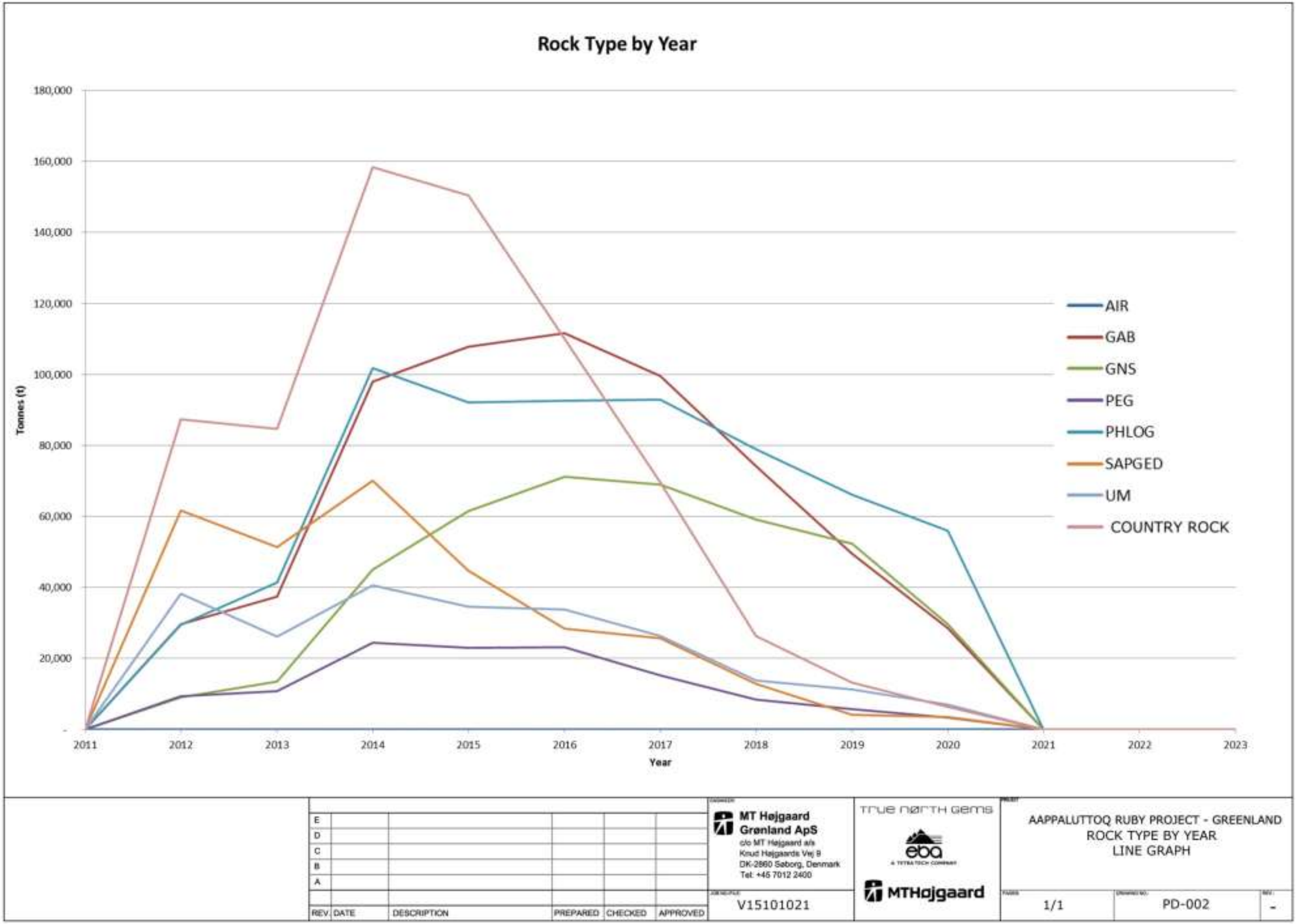
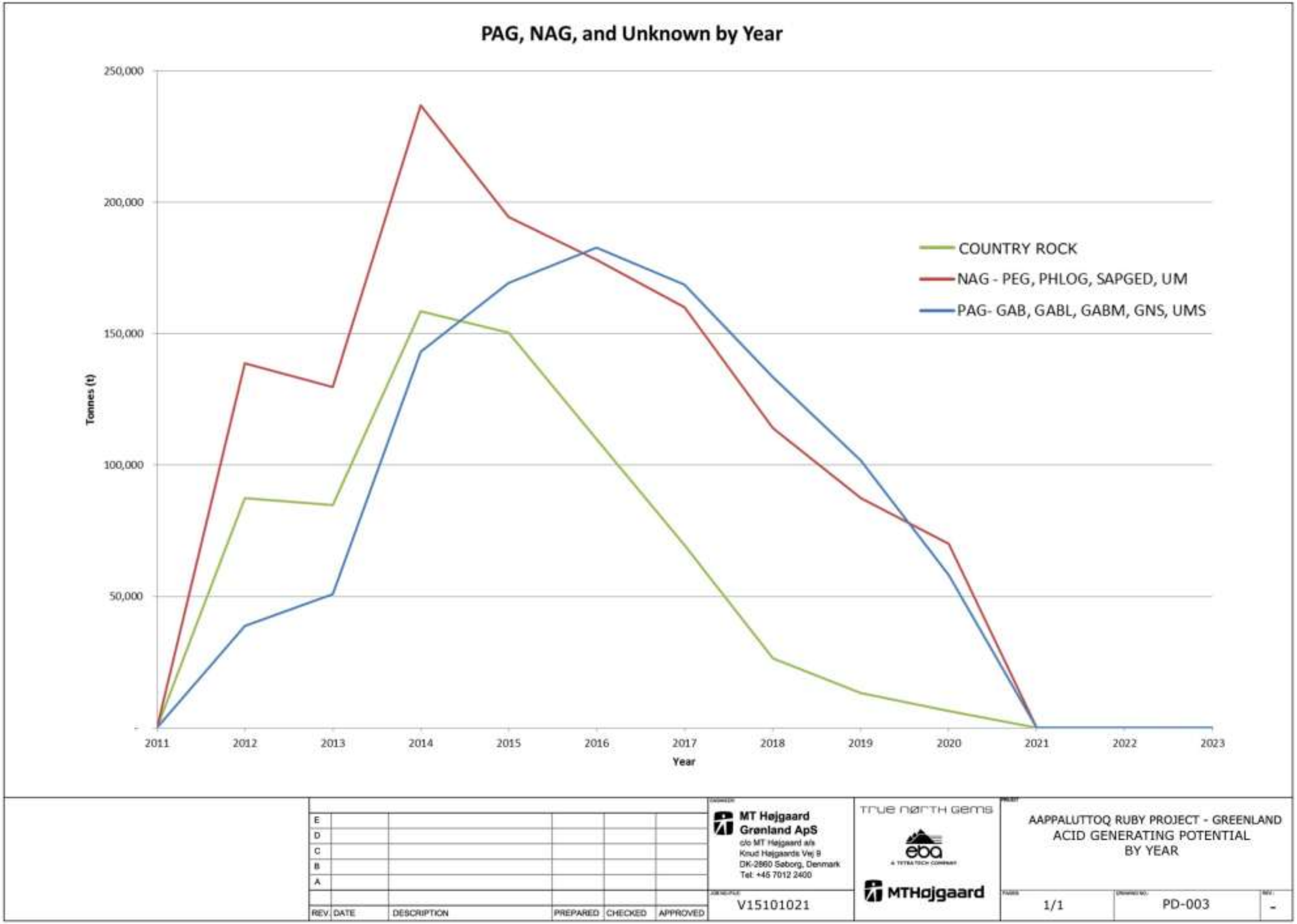


Figure 25: ARD by Year



19.10 PAG/NAG Differentiation

The initial assessment of the acid rock drainage (ARD) and metal leaching (ML) potential of the Aappaluttoq Ruby Project in Greenland is based on discussions and review of drill logs, test data and geological information provided by the Company, and analysis of select core samples from the property by Maxxam Analytics (formerly Cantest Ltd.) in Burnaby, BC, Canada.

A total of 109 core samples, representing sulphide-bearing and non-sulphide intervals of each major lithology at the property, were selected and put through an analytical program that included:

- Acid-Base Accounting (ABA) using the Modified ABA NP method (Price 2009)
- Total sulphur and sulphate-sulphur speciation analysis to determine sulphide-sulphur content, by the difference between the two
- Elemental (trace metals) analysis by aqua regia digestion followed by ICP-MS scan
- Whole Rock analysis by lithium metaborate fusion followed by x-ray fluorescence (XRF) spectroscopy
- Mineralogical assessment of 6 samples, conducted by Vancouver Petrographics, and
- Short-term leaching tests on 20 select samples, using shake-flash extraction in distilled water followed by ICP-MS scan.

Table 33 shows the classification of each material type which has been encountered at Aappaluttoq.

Table 33: PAG/NAG Material Types at Aappaluttoq

Classification	Lithology
Non-Acid Generating (NAG)	Pegmatite (PEG) Phlogopite (PHLOG) (Main Ore, Tailings) Sapphirine/Gedrite (SAPGED) Overburden Non-sulphide Ultramafics (UM)
Low Potential for Acid Generation (Low-PAG)	Gabbro (GAB) (Secondary Ore) Leucocratic Gabbro (GABL) (Secondary Ore)
Potentially Acid Generating (PAG)	Mafic Gabbro (GABM) Gneiss (GNS) Sulphide-bearing Ultramafics (UMS)

19.10.1 NAG Material Types

NAG lithologies generally do not pose a concern with regard to acid generation, as sulphide minerals are typically not associated with them. Metal leaching from these materials is expected to be minimal.

From the perspective of ARD and metal leaching, these lithologies are generally appropriate for infrastructure use.

Although likely to be a relatively uncommon occurrence for these rock types, field screening should be undertaken to ensure that any sub-units containing greater than 0.3% sulphides are managed by sub-aqueous disposal.

19.10.2 Low potential for Acid Generation

Low-PAG lithologies generally pose a low concern with regard to acid generation, as sulphide concentrations are typically low.

Acid generation is a potential issue when sulphides are present in higher concentrations, but this appears to be the exception for these rock types.

Metal leaching from these materials is expected to be minimal.

From the perspective of ARD and metal leaching, these rock types may be appropriate for use in construction, provided that field screening is undertaken to ensure that any sub-units containing greater than 0.3% sulphides are managed by sub-aqueous disposal.

19.10.3 Potentially Acid Generating

The acid generating potential of PAG lithologies is related directly to the presence of sulphides, which these rock types appear to possess in variable but generally higher amounts. Acid generation is a potential issue when sulphides are present in higher concentrations.

Metal leaching from these materials is expected to be minimal.

From the perspective of ARD and metal leaching, these rock types are not generally recommended for use in construction, but if necessary they may be used provided that field screening is undertaken to ensure that any sub-units containing greater than 0.3% sulphides (0.1% in the case of gneiss) are managed by sub-aqueous disposal.

19.10.4 PAG/NAG Conclusions

The principal conclusions resulting from this assessment are as follows:

- Lithological units that are not generally of concern with respect to PAG, include pegmatite (PEG) and phlogopite (PHLOG), which represent the ore and tailings, sapphirine/gedrite (SAPGED), overburden (OVB) and non-sulphide ultramafics (UM). Sulphides do not appear to be generally associated with these rock types.
- A few lithological units, such as mafic gabbro (GABM), gneiss (GNS) and sulphide-bearing ultramafics (UMS) may be of concern with respect to potential acid generation. These concerns are directly related to sulphide concentrations in the relative absence of adequate neutralizing potential.
- Subaqueous deposition of waste rock and tailings material in the lake will minimize sulphide weathering and reduce potential acid generation to negligible rates in PAG and low-PAG materials by limiting exposure to free oxygen.
- Short-term leaching tests indicated that these elements are not likely to be mobilized to any significant extent under the neutral pH drainage conditions that prevail at the project site.

Other conclusions of note:

- Chemical-weathering reactions will be hindered by the low summer temperatures and frozen winter conditions at the property.

- NAG materials may be used for infrastructure construction if needed for this purpose, provided that a field screening protocol is undertaken to ensure that any sub-units that contain greater than 0.3% sulphide are managed by sub-aqueous disposal.
- Kinetic test work and modeling of potential acid generation and pit water quality are not warranted at this stage in the mine development process given the generally low potential for acid generation and metal leaching.
- The open pit will be flooded, post-closure, and the subaqueous environment will limit ongoing oxidation of sulphides exposed in the pit walls to negligible rates. The extent of fresh rock exposed after flooding of the pit is expected to be negligible, at most only a few metres wide along the west and southwest edge of the open pit, with a small area (less than 100 m^2) at the north end of the pit. As such, this exposed rock does not represent a significant concern for acid generation or metal leaching.

19.10.5 PAG Management

Potential Acid Generating material will need to be managed to reduce as far as possible likelihood of environmental impacts to the area. The management plan will include:

- A field screening protocol should be developed and implemented to ensure that any materials that contain greater than 0.3% sulphide are managed by sub-aqueous disposal
- Gneiss will generally be submerged due to the pervasive presence of pyrite. Any gneiss material with greater than 0.1% sulphides will be treated as PAG material
- Low-PAG materials of most rock types, other than gneiss, may be used for infrastructure construction provided that a field screening protocol ensures that any sub-units contain less than 0.3% sulphide (and 0.1% sulphide in the case of gneiss).
- PAG materials would be identified and managed. These materials will be submerged and not used for infrastructure construction.
- Open pit water quality will be monitored during mine operation to determine what, if any, mitigating measures might be required.

19.11 Overall Site Layout

The overall site consists of the four main areas, the operations area, camp area, explosive storage and outer port. Figure 26 shows the overall site layout.

19.11.1 Operations Area

The operations area contains the processing plant, main workshop and open pit mine. This area will be mostly active during dayshift. The process plant area will be illuminated at all times during the operational season.

19.11.2 Camp Area

The camp area includes the accommodation, helipad, inner port and fuel storage areas. This area will be active at all times during operation.

19.11.3 Explosive Storage

The explosive storage area will be separate for safety reasons. The area will only be active when storing or retrieving detonators and explosives. The area will be illuminated at all times during the operational season. No explosives will be stored on site in the off season.

19.12 Infrastructure

Infrastructure items were designed by MTH and reviewed by John Chow who is satisfied for their inclusion into the PFS. MTH has a history of civil works in sub-arctic locations including Greenland.

19.12.1 Design Standards

Eurocodes with Greenlandic and/or Danish annexes are outlined below as well as various building and technical regulations form the basis for the design standard for this study:

General:

- "Greenland Parliament Act No. 7 of December 7, 2009, on mineral resources and mineral resource activities (the Mineral Resources Act)"
- (Inatsisartutlov nr. 7 af 7. December 2009 om mineralske råstoffer og ak-tiviteter af betydning herfor)

Buildings (accommodation, offices, workshop):

- Greenlandic Building Regulations (BR06)
- (Bygningsreglement 2006)

Steel structures:

- Eurocode EN1993 1-1 to 1-10

Timber structures:

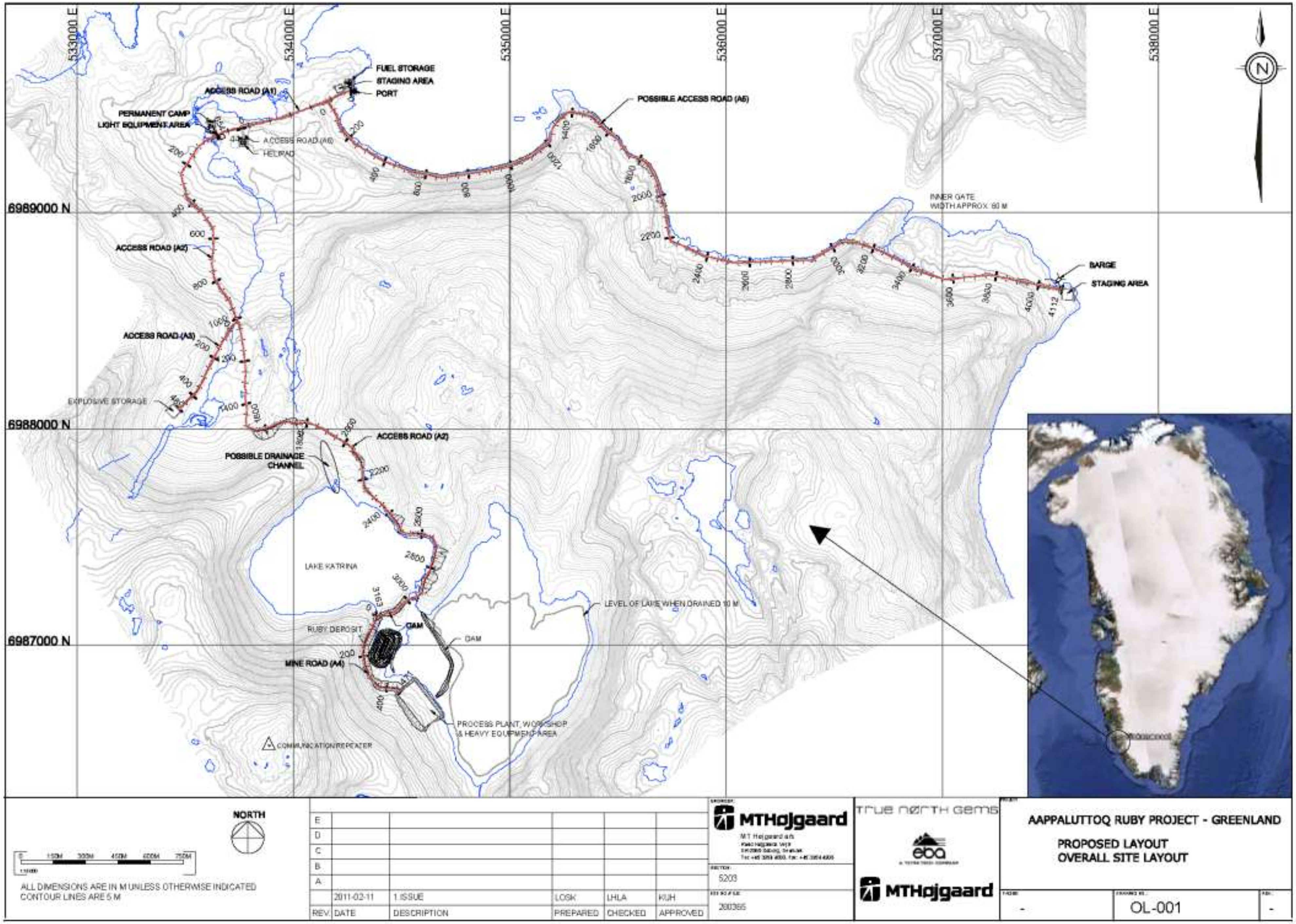
- Eurocode EN 1995-1-1

Foundation:

- Eurocode EN 1997-1

Detailed design has yet to be finalized and will likely result in changes to account for terrain, practical limitations and budget quotations.

Figure 26: Overall Site Layout



19.12.2 Construction Camp

A temporary camp is already on site (Photograph 3). This camp has been used in connection with previous exploration drilling. The existing camp is located approximately 2 km north of the Aappaluttoq deposit and consists of 18 tents in all:

- 14 accommodation tents
- One office tent
- One kitchen/canteen tent
- One toilet/shower tent
- One core shack tent

Photograph 3: Current exploration camp

The camp can accommodate up to 28 persons if tents are shared in double occupancy. In preparation for the construction work, the camp will be refurbished. Additional accommodation for the construction period will be container accommodation for approximately 30 persons requiring:

- 14 × 20' double occupancy containers
- Three 20' single occupancy containers
- Four 20' kitchen/canteen containers
- One 20' reefer container
- One 20' freezer container
- Two 20' storage containers
- Three 20' office containers (one will be placed at the workshop)
- One 20' container for water treatment

- One 20' container for waste management (incinerator)
- One 20’ sewage container

The containers will be placed on a gravel pad and supported with 8" × 8" wooden beams and/or concrete blocks. The containers will be secured with anchor plates or similar in the gravel pad to prevent uplifting.

The present camp is equipped with two 25 kW generators for power supply, one tank for purification of drinking water and one tank for water treatment. One 25 kW generator will be added for power supply. The generators will be placed in such a way that noise will not be an issue for camp residents.

Heating will be supplied by electrical power from the overall power supply.

Water supply will be taken from a nearby water source separate from the operations watershed.

In addition to the expansion of the present camp for the construction phase, additional water treatment such as chlorination and UV treatment will be implemented. A sewage container and an incinerator will be added for treatment of sewage water and sanitation as well as solid waste management.

The camp site will be equipped with fire extinguishers. In case of large fires, fire hoses will be available for pumping water from the nearby lake.

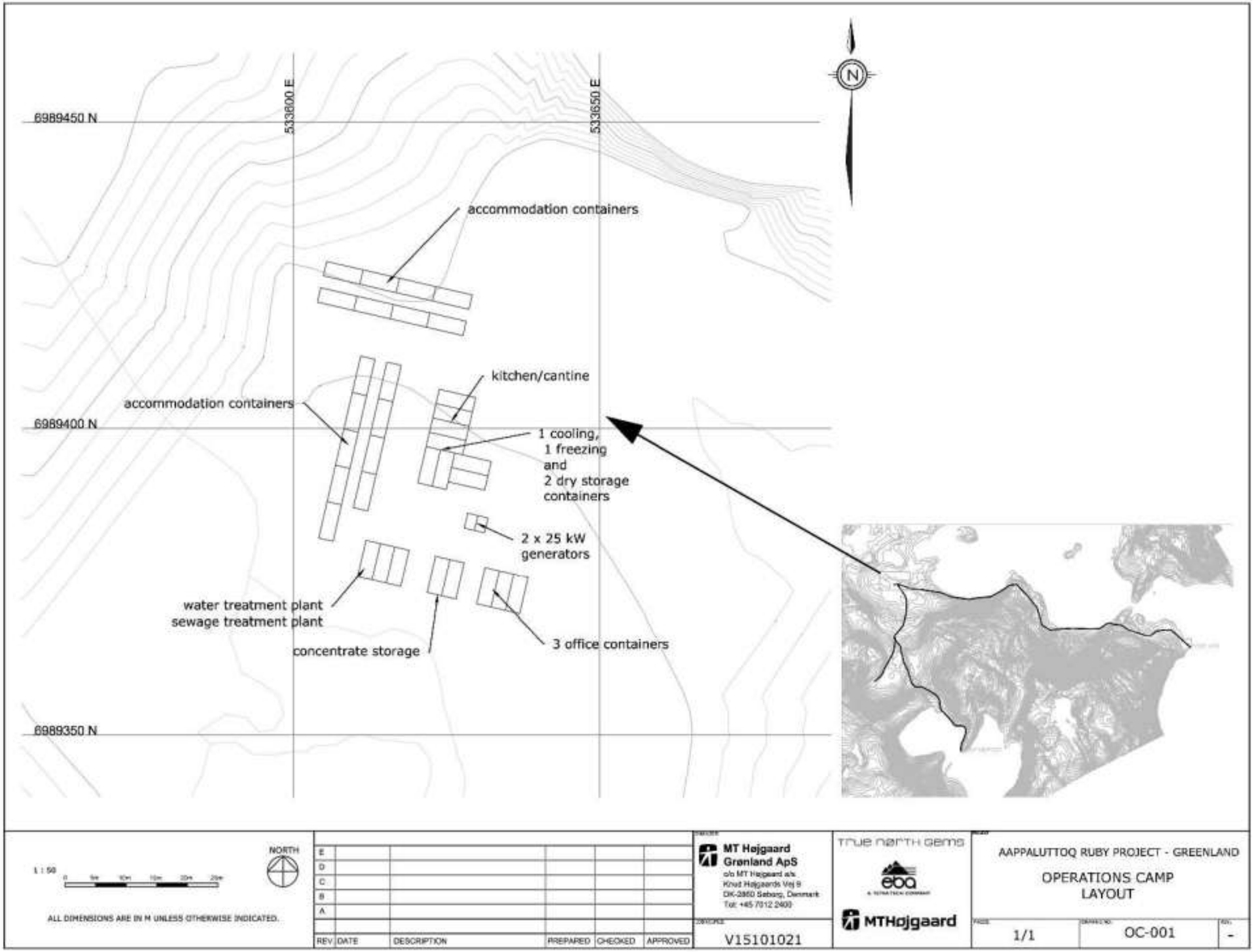
19.12.3 Operations Camp

The container camp installed for construction will be refurbished and used as the camp during the mining operations. The container camp accommodation will be for approximately 40 persons and will consist of:

- 21 × 20' double occupancy containers incl. toilet/shower facilities
- Three 20' single occupancy containers incl. toilet/shower facilities
- Five 20' kitchen/canteen containers
- Four 20' office containers (one will be placed at the workshop)
- One 20' container for water treatment
- One 20' container for waste management (incinerator)
- One 20' container for sewage
- Two 20' containers for first aid room and storage
- Four 20' containers for recreation

An overview of the permanent operations camp and construction principles is shown in Figure 27.

Figure 27: Operations Camp



19.12.4 Workshop

A workshop will be established in connection with the process and mining facilities. The workshop will be able to maintain and repair all types of vehicles, from ATVs to haul trucks and excavators. An interior bay will allow for hot work to be done.

The workshop will be constructed from eight 20' containers forming walls and a wooden roof structure. The workshop will be equipped with prefabricated drive through fuel berms for collection of oil residue and to minimize oil spills. Waste oil will be collected in barrels and either used for heating or shipped to Nuuk for destruction.

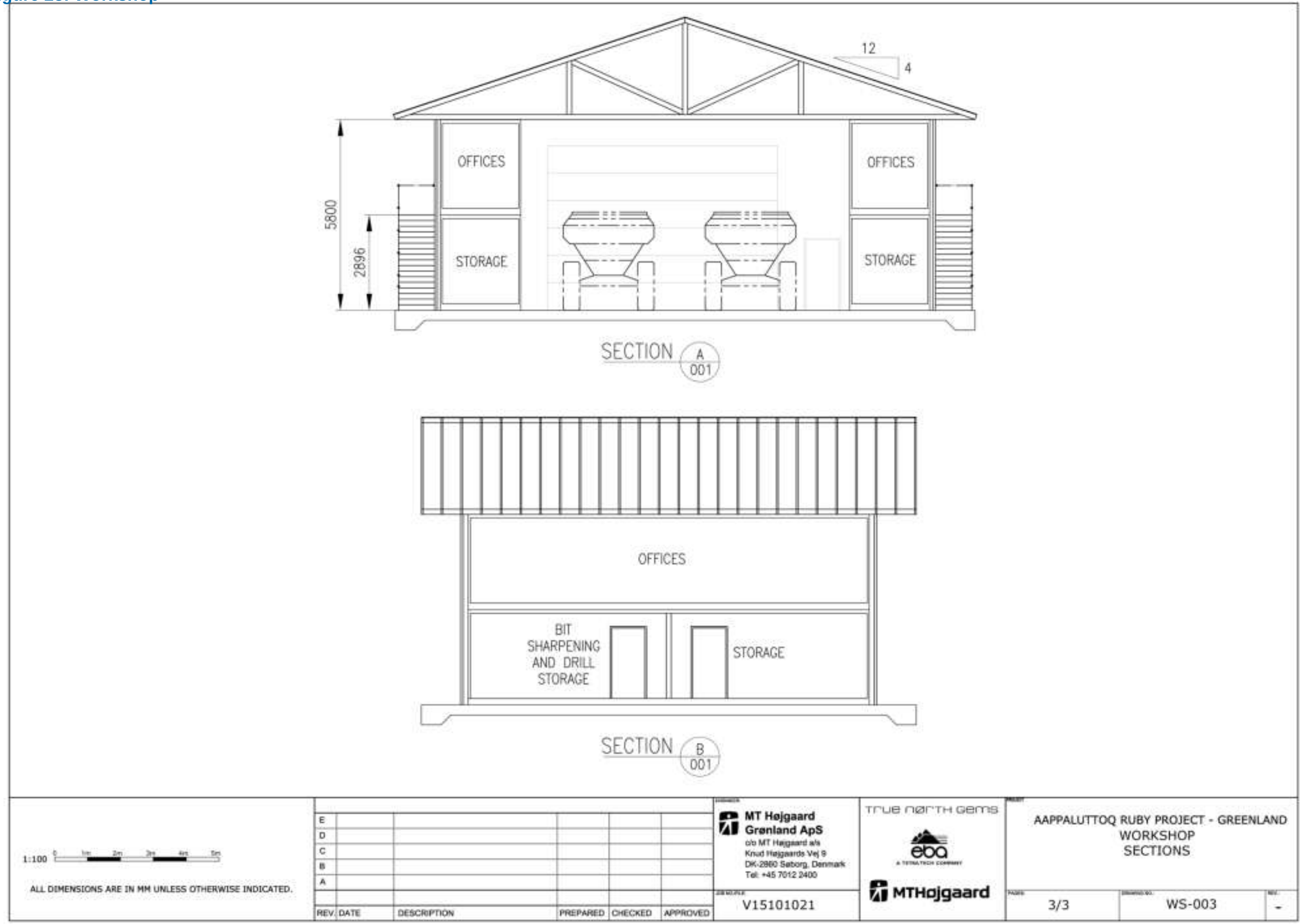
The floor will be constructed of I-Trac plates or similar.

A 50 kW power generator will be connected to the workshop for power supply.

One 20' container will serve as office and emergency facility for the workshop. Further, one container will be attached to the workshop for storage of oil and lubricants.

An overview of the workshop is shown in Figure 28.

Figure 28: Workshop



19.12.5 Process Building

All process equipment is placed in the process building except for the primary and secondary crushers.

The process is described in section 16.4 Site Mineral Processing Summary.

The building is 78 m long, 33 m wide and approximately 14 m high. The building will be constructed of structural steel to support the process equipment as well as the siding and roof system. The primary construction will be a system of beams and columns in a module of approximately 6 m × 6 m. The slope of the roof will be approximately 18°.

The siding will be constructed with pre-insulated wall panels, and the roof will be constructed with a roof system of trapezoid steel sheets, insulation and asphalt board on top. The building will be founded on concrete spot footings of 5 m^3. The floor will be made of reinforced concrete.

The building will be heated with a minimum design temperature of 15° C and supplied with ventilation and electrical installations.

19.12.6 Communications

The main communication link will be a satellite telecommunications system comprising the following components:

- Fibre optic communications system
- Internet, telephony and television capabilities
- Satellite earth station in a heated enclosure suitable for outdoor arctic installation
- Communications equipment including all routers, switches, Ethernet bridges, controllers, security firewall, modems and other system requirements
- Cabinets, cabinet wiring and equipment mounting

Further, a 3-channel radio system for communications during operations will be on site. Vehicles will have a fixed radio and a number of handheld units will be available to personnel.

19.13 Access Roads

The road from the inner port to the camp site will be approximately 500 m in length. The road continues from the camp site to the Aappaluttoq deposit, a length of approximately 3.6 km. A connecting road to the explosives storage on this alignment will have a length of approximately 500 m. These roads will be constructed as single-lane gravel paved roads.

A possible future access road from the inner port to the outer port will have a length of 3.5 km.

The proposed alignments of the access and mine roads are shown in the overall site layout, Figure 26.

The design standards used for the roads are:

- Greenland Home Rule, "Village roads in Greenland, directions for execution"
- (Grønlands Hjemmestyre, Bygdeveje i Grønland, vejledning i udførelse, juli 1984)

- Department of the US Army Technical Manual, "Arctic and Subarctic Construction, Runway and Road Design", October 1954

Roads were designed by MTH and reviewed by John Chow who is satisfied for their inclusion into the PFS. MTH has a history of civil works in sub-arctic locations including Greenland.

19.13.1 Logistics

Traffic on access roads is estimated to be low with an average of approximately 10 vehicles per day in both directions over the year.

The dominant traffic on access roads will be light vehicles for transport of personnel and trucks with fuel and supplies. The maximum permitted axle load is 15 t and maximum total vehicle load is 45 t. If special vehicles to transport heavy equipment are required, the adequacy of the roads will have to be assessed.

The maximum speed according to Greenland traffic rules is 60 km/h. With the planned standards for the roads, the average speed will be lower, approximately 30 km/h.

In the spring and early summer, thaw water tends to build up between the non-frozen layers and the frozen layers. This might weaken the road, and heavy traffic should be reduced in this period. Culverts such as steel pipes or similar will be placed perpendicularly in the road structure where considered needed in order to provide drainage and to minimise flood damage on the road.

Ongoing maintenance of the access roads will be carried out on a regular basis to ensure safe operations.

Based on the risk of having the road washed away during periods of extreme weather and heavy snow falls all work sites will have sufficient fuel storage capacity for at least two weeks, in order to maintain productivity.

It will be a requirement that all vehicles will be equipped with radio and in regular contact with either plant site and/or port site. This will also allow drivers to be warned of oncoming traffic and pull their vehicles over in plenty of time to allow the other vehicle to pass safely. Furthermore, all vehicles will be equipped with an emergency kit in case of incidents.

19.13.2 Design Criteria and Authorities' Approval

The road is established as a privately financed road and is solely used in connection with the mining activities at Aappaluttoq. It is, however, expected that the Greenland authorities will require that general Greenland traffic rules apply. Hence the road is expected to be designed and constructed according to Greenland road regulations and standards.

Further, it should be expected that the Greenland authorities will demand that the width of the road is kept as narrow as possible and that contact with and damages to the existing vegetation be kept to a minimum.

The proposed alignment of the access road is based on the topography provided by the Company and MTH's site visit.

19.13.3 Overall Alignment

In general, the road will be established as a 3.5 m wide single-lane gravel paved road. The total width of the road inclusive of shoulders will typically be 5.5 m. Pull-outs will be provided to allow vehicles to pass.

Drainage crossings, varying in size from very small to small, are located along the various alignments of the access roads. Crossings will be constructed with culverts such as steel pipes or similar.

19.13.4 Construction Principles

In general, the access roads will be constructed on the existing ground, either on exposed bedrock, bedrock covered by vegetation/moraine or local sediments consisting of mostly sand.

The construction design is based on normal road building practice in subarctic areas.

In general, 0.75 m thick layer of 0-100 mm screened quarry run will be placed as sub-base directly on bedrock/moraine or sediments. A top layer of 0.05 m of 0-8 mm screened quarry run will be laid out as a closing layer. The material will be compacted with dozer or similar equipment. Where more filling is required below the road structure (0.8 m thickness in all), 0-500 mm quarry run will be used.

Side slopes will be constructed with an angle of 1:1.5 (vertical:horizontal).

The road will be constructed with a gradient of 3% perpendicular to the alignment in order to drain off surface water.

The grade of the access road will be a maximum of 15%.

The work comprises:

- Preparation of the existing ground surface, e.g. removal of large stones etc.
- Laying out of 0.75 m sub-base
- Excavation, primary levelling and building up of embankments at ravine crossings and culverts with locally excavated soil materials or quarry run
- Laying out of finishing top layer of 0.05 m screened quarry run 0-8 mm

In general, the road structure will be constructed with quarry run from suitable mine waste or an established quarry or from borrow pits containing sand and gravel. Berms or large boulders will be placed as crash barriers on embankments and stretches on high slopes.

19.14 Transportation and Freight Facilities

Port and helipad items were designed by MTH and reviewed by John Chow who is satisfied for their inclusion into the PFS. MTH has a history of civil works in sub-arctic locations including Greenland.

Design standards are the same as those presented in section 19.12.1 Design Standards.

19.14.1 Freight Facilities

In order to be able to unload equipment for construction and mining operations and to receive service vessels carrying supplies and fuel, a pier head inside the inner gate in the bottom of the fjord, also called the "inner port", will be constructed. Furthermore, a barge including mooring facilities will be established just outside the inner gate, functioning as a pier head to allow unloading of materials before further distribution, in case tide conditions or load sizes do not permit immediate access to the inner port. This pier head is called the "outer port".

The inner port will be placed in the Tasiusarsuaq fjord inside the inner gate close to the location of the present and future camp site. The narrow straits into the Tasiusarsuaq fjord do not allow access for larger bulk carriers; it is, however, considered that smaller vessels including tug barges can access the narrow straits which has also been confirmed by Nuuk based shipping company "Masik".

These locations are shown in the overall site layout Figure 26.

Sea ice conditions in the winter period could cause access problems even with the quite strong current at the proposed port location. The sea ice will typically be present in the area from December to May. Access in this period will be carried out with ice-reinforced vessels if required. However, construction and operation activities are not planned to be year-round and hence sea ice is not expected to be a significant problem.

The sailing route from Nuuk to the site will pass the village Qeqertarsuatsiaat, thus offering the local community a chance to become involved and creating possibilities of a social and economic benefit for the village.

In terms of shipping, Royal Arctic Line (RAL) has scheduled routes to Qeqertarsuatsiaat which might also serve as a support port.

19.14.2 Outer Port Facilities

The outer port will be constructed as a pier construction consisting of a 15 m × 21 m barge. The barge will be anchored in four corners in order to locate the barge horizontally, but still allowing vertical movement due to the tide. Towards the shore, the corners of the barge will be anchored to bollards placed onshore. The bollards will be made from prefabricated bollard blocks.

In the opposite corners, the barge will be secured by anchors.

The barge will be equipped with a winch in each corner in order to be able to adjust the anchor lines continuously.

The barge will be equipped with extra bollards for mooring of supply and service vessels as well as for security reasons.

Fendering will be provided by use of used mining truck tires or similar.

The barge front will be fitted with one rescue ladder and one rescue post with life ring and equipment.

Access from the barge to the shore will be provided by a ramp of wooden beams or similar.

In connection with the port site, a staging area of approximately 50 m × 50 m will be constructed. The staging area will be used for storage of larger equipment and supplies until further distribution to the camp site and mine site. The staging area will have power and lighting and a small office enclosure.

19.14.3 Inner Port

The inner port will be constructed as a pier head consisting of a sheet pile wall filled with quarry materials. The sheet pile wall will be designed for a lifetime of 10 years.

Towards land, a dike will be constructed by use of suitable quarry run laid out with dozer or similar. The pier will be constructed as an embankment with a slope gradient of 1:1.5 (vertical:horizontal) and a crown width of 5 m.

In connection with the port site a staging area of approximately 50 m × 50 m will be constructed. Like the laydown area at the outer port, this laydown area will have power from a portable generator and be equipped with lights and a small office enclosure.

19.14.4 Heliport

A heliport will be constructed in connection with the camp facilities. In order to allow a large helicopter such as a Sikorsky S-61, the heliport will be constructed as a gravel pad with an approach and take-off area with a diameter of 30 m and a safety zone with a diameter of 45 m. The surface of the gravel pad in the approach and take-off area as well as in the safety zone will be compressed and sealed to avoid gravel to be sucked up in the helicopters slipstream. A 2 m wide shoulder will be established around the safety zone to avoid reduction of the safety zone in case of slope erosion. The approach and take-off area will be marked with an "H" which will be painted in white. A windsock will be mounted for the helicopter's guidance.

Warning signs prohibiting unauthorized entry during helicopter operations will be mounted.

Fire fighting equipment as well as life saving equipment will be placed at the heliport.

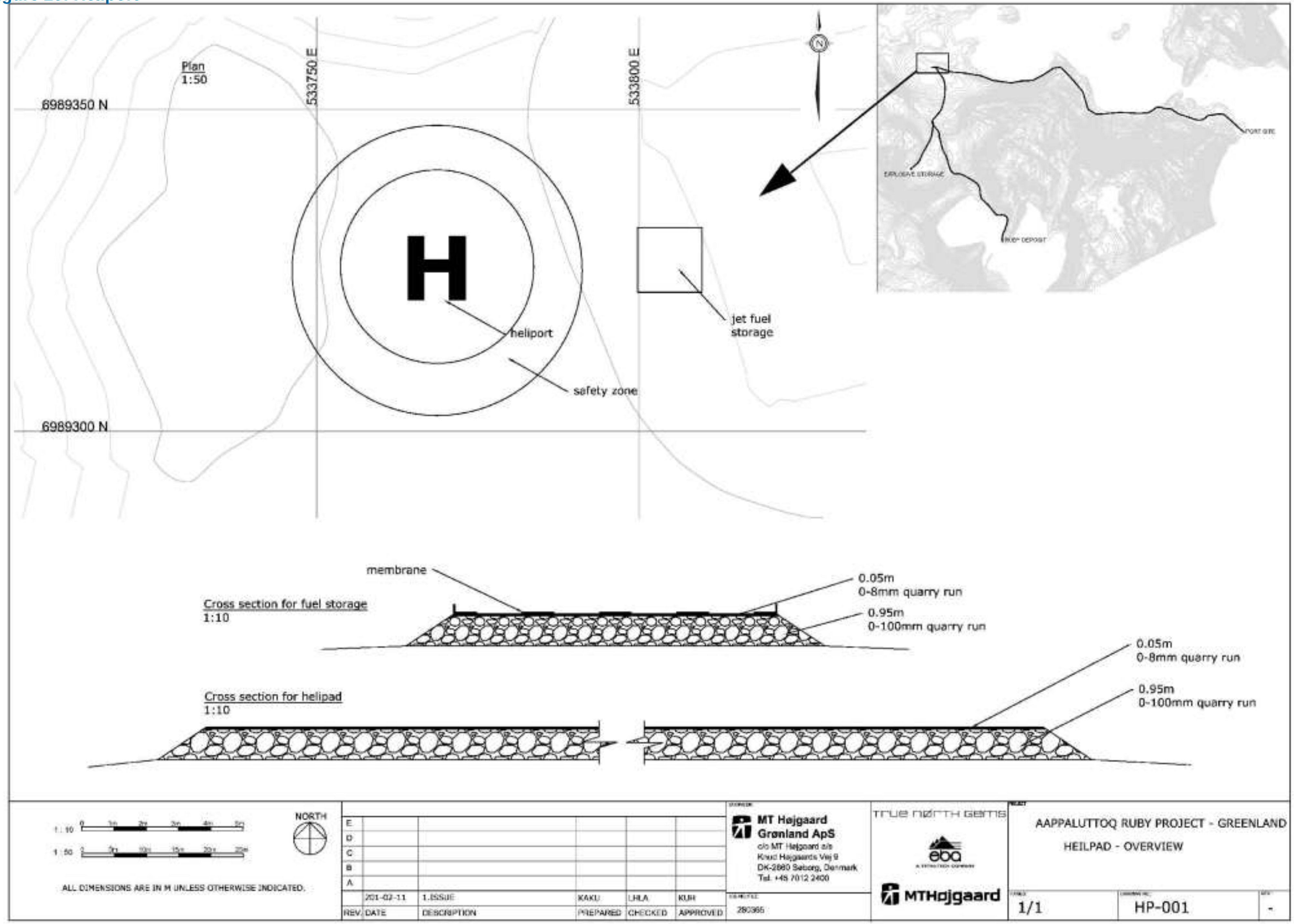
As it is not the intention to fly at night or in wintertime, the heliport will not be equipped with night lights or landing lights. During emergencies, light can be provided by either vehicles or a mobile lighting plant.

A fuel storage area with a capacity of holding one hundred fuel drums will be established at the heliport, see section 19.16.4 Heliport Fuel Storage. Fuel will be supplied by the helicopter company. Refuelling equipment will not be established in connection with the fuel storage. The helicopter crew will bring their own refuelling equipment; in this way the helicopter crew will be responsible for the fuel quality.

The heliport will be constructed according to “BL 3-8 Bestemmelser om etablering og drift af helikopterflyvepladser” (Statens Luftfartsvaesen 2008) and the requirements of the Danish Transport Authority (previously the Civil Aviation Administration Denmark).

Location of the heliport as well as construction details are shown in Figure 29.

Figure 29: Heliport



19.15 Power Generation

In the construction phase one 25 kW generator will be added to the existing two 25 kW generators for power supply. When the construction phase is over and the operations phase commences the present tent camp will be dismantled and shipped to another destination and one of the generators will also be dismantled and shipped to another destination.

During operations, power will thus consist of:

- Two 25 kW generators for the operations camp
- Two 500 kW generators at the crusher facilities
- Two 150 kW generators at the process facilities
- One additional 50 kW generator

19.15.1 Power Consumption

The total equipment power at the site processing plant is 445 kW. The actual load during operations is expected to be 267 kW which is 60% of the total equipment power.

The workshop and office building will consume approximately another 75 kW with a similar load for items such as lighting and security equipment.

A total of 816 kWh will be consumed on average per year over the mine life (Table 34).

Table 34: Site Power Consumption

Area	Units	Average annual consumption
Mining	*kW*	130,900
Processing	*kW*	623,750
G&A	*kW*	61,271
Total	*kW*	815,921

19.16 Fuel Storage

Fuel storage facilities with a fuel capacity of 150,000 litres will be placed close to the inner port allowing direct delivery from a supply ship or barge. Fuel will be delivered by a hose to the fuel tanks. Delivery to the power supply at the process and camp facilities will be carried out by tank truck via the access road from port site.

Furthermore, a 10,000 L tank will be placed at the process facilities. The annual fuel consumption is estimated to be 620,000 L for the site operation. Fuel will be supplied by tugboat and barge equipped with fuel containers which can transport 100,000 L of fuel. It is estimated that fuel will be supplied approximately every four to six weeks.

The design standards applicable are:

- Greenland Home Rule, "Technical regulations for flammable liquids"
- (Grønlands Hjemmestyre, Tekniske forskrifter for brandfarlige væsker)

19.16.1 Fuel Consumption

Fuel consumption is based on the equipment quantity, efficiency factor, working hours and estimated hourly consumptions. Gasoline for ATV's is included in the diesel fuel costs.

Table 35 shows site fuel consumption which includes fuel for power generation and explosives usage.

Table 35: Site Fuel Consumption

Area	Units	Average annual consumption
Mining	*L*	294,410
Processing	*L*	246,095
G&A	*L*	70,551
Total	*L*	611,056

19.16.2 Port Fuel Storage Facilities

Fuel tanks at the port site will consist of two 75,000 L collapsible bladder tanks, i.e. a total capacity of 150,000 L. Although it would have been possible to maximize capacity so as to meet requirements for the maximum annual fuel consumption, the chosen solution is considered beneficial in order to reduce the risk of theft and leakage to the environment.

In an area of 25 m × 40 m, vegetation will be removed and the area will be prepared either by filling or blasting and levelled. A gravel pad of 0.95 m of quarry run 0-100 mm and 0.05 m of 0-8 mm quarry run will be laid out with a gradient of 2 % for drain-off of surface water.

Each bladder tank will be installed in a prefabricated fuel foam berm covered with membrane liner. Each berm will be 13.2 m × 13.2 m × 0.5 m with a capacity of 15% in excess of the tank volume. The tanks will be interconnected and attached to a filling station which will be installed in a 10' container.

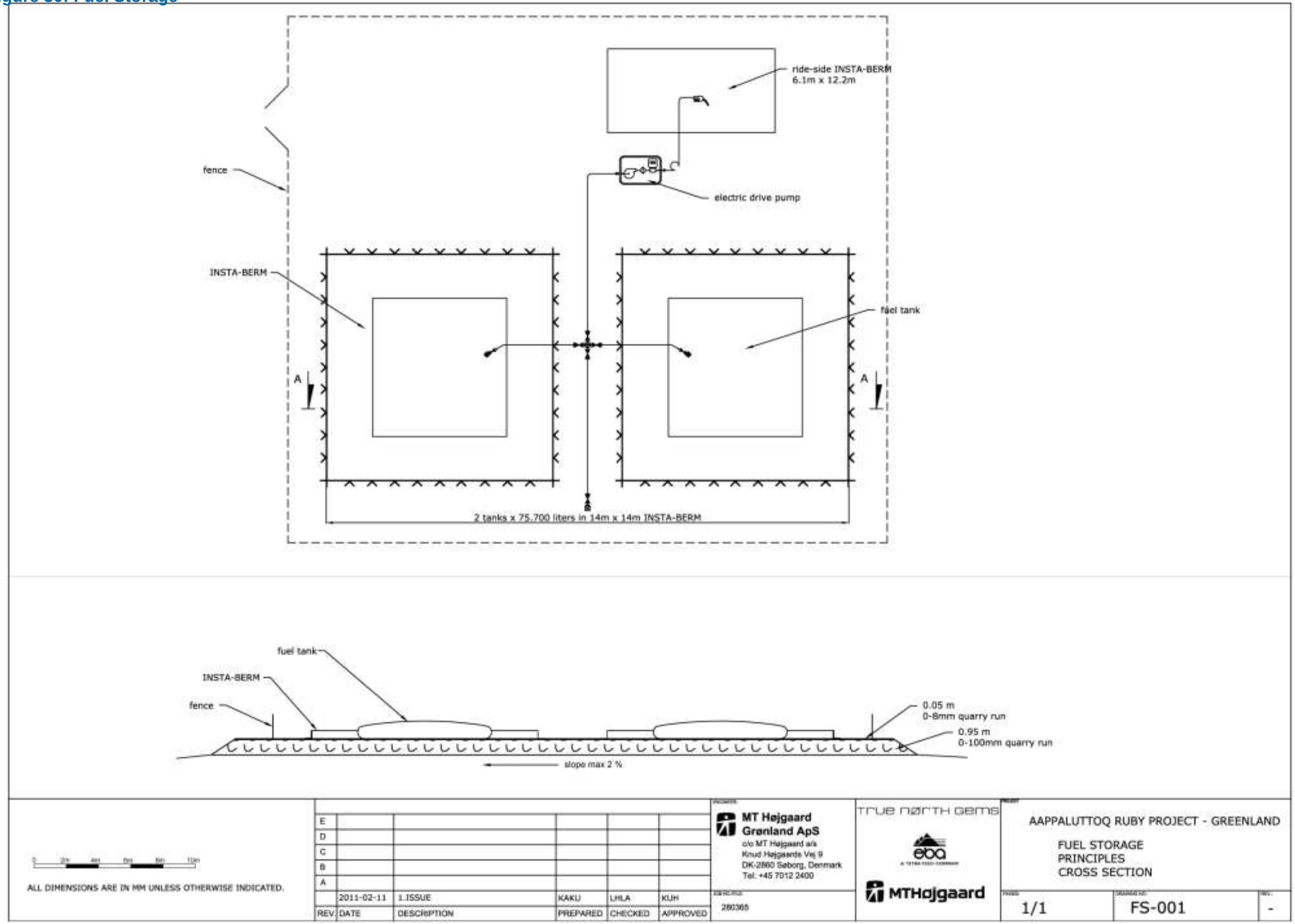
The fuel storage area will be fenced in with a 1.8 m high fence. Warning signs will mounted on the fence.

The tanks will be secured with straps attached to prefabricated concrete blocks or similar to prevent the tanks from lifting, when the tanks are emptied at the end of the season.

The filling station is based on a complete unit, delivered as a standard 10 ft (3.05 m) container solution. The filling station will be placed on wood beams and/or concrete blocks and placed in a prefabricated berm in order to protect the surroundings against incidental fuel leakage. Furthermore, a drive-through berm will be placed where filling activities are carried out; this is also to protect against leaks and for collection of fuel spill.

The general layout of the fuel storage area is shown in Figure 30.

Figure 30: Fuel Storage



19.16.3 Process Fuel Storage Facilities

The fuel storage at the process facilities will consist of one steel tank containing 10,000 L of fuel. The steel tank will be placed inside a berm with the dimensions 7.5 m × 5.0 m × 0.5 m and a capacity of 15% besides the tank volume.

Before placing the tank, an area of 10 m × 10 m will be prepared, either by blasting or filling. A layer of 0.5 m of sand or 0-8 mm quarry run will be laid out and a prefabricated fuel foam berm will be placed. A membrane liner will be installed for leak protection.

The tank will be secured with straps attached to prefabricated concrete blocks or similar to prevent the tank from lifting. The tank will be equipped with filling pump.

19.16.4 Heliport Fuel Storage

At the heliport, a fuel storage area with a capacity of holding one hundred 210 litre fuel drums will be established. The barrels will be placed inside a prefabricated berm similar to the berm constructed at port and plant site. An area of 10 m × 10 m will be cleared and levelled. This fuel storage area will primarily contain jet fuel. When filling the helicopter, the fuel barrel will be placed in a special berm to prevent leakage to the environment during filling.

19.17 Explosives Storage

In order to be able to extract 5,000 t of ore per year and up to 450,000 t of waste rock, explosives on the site are essential. The explosives storage will be placed approximately 1.0 km northwest of the mine and process site. For location of the explosives storage facilities, see the overall site layout plan (Figure 26).

The design of the storage area is calculated from material movements and blasting parameters. Explosives include packaged explosives, ANFO and detonators.

Explosives will be stored in a 20' locked and vented container, reinforced for storage of explosives, and detonators in a separate 20' locked and vented container, reinforced for storage of detonators. The containers will be authorized according to current regulations. The annual quantity of explosives consumed will vary between 20 t to 40 t.

During mobilization and construction of access and mine roads, explosives will be stored in reinforced containers at a temporary location placed at the outer port just outside the inner gate approximately 3.0 km from the camp site and the inner port. The reinforced containers containing detonators and the containers with ANFO will be placed at the port site area as well. The various containers will be placed according to the current regulations for storage of explosives.

The explosives, detonator and ANFO containers will be moved once the permanent explosives storage facilities have been prepared and constructed and the access road established.

The storage area will consist of a shelter which will be blasted into the bedrock in order to protect camp and process facilities as well as an access road with minor traffic. The storage area will be placed approximately 1 km from the camp site in a sheltered direction and 300 m from the access road in a sheltered direction.

The containers for explosives, detonators and ANFO will be placed on wooden beams or similar and on a levelled plateau of 0-8 mm quarry run. The plateau will be laid with a gradient of 2% to drain off surface water.

The detonator container will be placed at a minimum distance of 10 m from the explosives container and sheltered from this by an embankment of quarry run or similar.

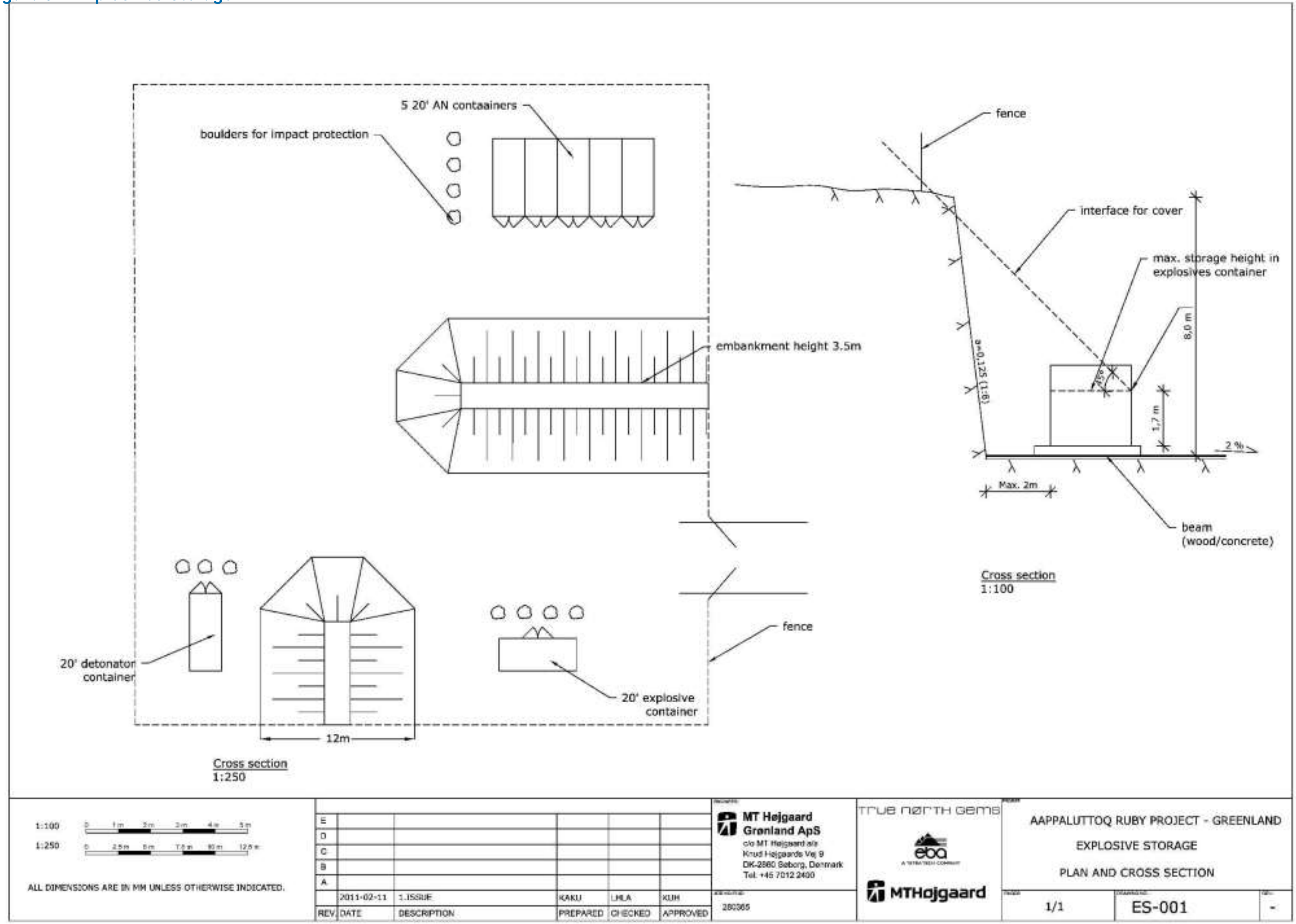
The entire explosives storage will be fenced with a 1.8 m high fence. On top of this barbed wire will be placed. The fence will be placed on the top of the insitu rock. The fence and containers will be equipped with lightning protection and security lighting.

Warning signs will be attached to the fence and the gate will be kept locked.

A general plan and cross sections of the explosives area with notes are shown in Figure 31.

Infrastructure items were designed in conjunction by MTH and John Chow.

Figure 31: Explosives Storage



19.18 Site Logistics

All equipment and goods used in connection with the mobilisation and construction are expected to be shipped from Aalborg (Denmark) to Nuuk by Royal Arctic Line (RAL) or from Halifax (Canada) to Nuuk by Eimskip.

Spare parts, equipment and supplies will be unloaded in Nuuk before further transport to the site. In Nuuk, large goods will be loaded onto a supply vessel and/or barge, while small goods and supplies will be loaded onto a small service boat, e.g. the boat used for crew change.

On arrival at the site, the large goods will be unloaded at the staging area at the outer port site until the tide allows further distribution to the inner port site. Small goods and supplies will be transported by service boat and unloaded directly at the inner port site, depending on the tide. Transport with the service boat will, as far as possible, be planned according to tide variations in order to gain access to the inner port site.

19.19 Labour Requirements

The operations between site and Nuuk will employ a total of 85 people. At any one time, 59 staff will be working, with the other staff on R&R. When available, personnel will be sourced from Greenland and Denmark. Senior mine personnel who require extensive experience or professional designation may be sourced internationally. Further details of labour for each section are detailed below.

There will be preference to hire local labour from areas such as Nuuk and Qeqertarsuatsiaat. A small number of foreign workers will be required as it is not expected that people with the required experience will be available in Greenland. This includes staff requiring a professional designation for the Company to be compliant, such as engineering, geology and accounting. As operations progress, experienced national staff may become available and will replace expatriates when possible.

19.19.1 Site Mining Labour

The mining operation will be owner managed and maintained. The Company will hire all operators, maintenance crew and technical staff required. Some specialized work may need to be outsourced to contractors or consultants on an as and when needed basis.

The mining operations will employ a single work crew of 15 people. Three positions will be occupied by expatriates if experienced and qualified Greenlandic persons are not available. The crew will work a 12 hour shift, 2 weeks on, 1 week off roster.

19.19.2 Site Processing Labour

The processing operations will employ two work crews of 13 people each. One position will be occupied by an expatriate if experienced Greenlandic persons are not available. The work roster will be 1 week on, 1 week off.

All technical services will be provided by external consultants.

19.19.3 Site General and Administrative Labour

The operations will be supported by a 23 general and administrative staff. The work roster will be 1 week on, 1 week off. Management staff will work a roster as required by operations.

19.19.4 Nuuk Labour

A total of 21 staff will be employed at the Nuuk operations. Personnel engaged in corporate matters in support of the mine operations will work on a regular 40 hours week throughout the year.

The Security Supervisor will work a regular shift, the security officers will provide 24 hour rotating coverage.

The maintenance and janitorial staff will be contracted and be on an off-shift basis.

19.19.5 Training and Skill Development

One of the most significant benefits expected from the project is related to training and skills development opportunities. Specific on-the-job training will be provided for all operators at the mine, processing plant and sorting house. Some of the training is specific for the gem industry, but most of the new skills can be applied to the mining industry in general, improving significantly the employability of all workers involved in the operation, particularly the unskilled ones. The possibility of providing 5 trainee positions for students from Greenland School of Minerals and Petroleum, the Food Service and preparation school and other vocations such as electricians and plumbers will have a significant impact on the vocational training sector.

19.19.6 Construction Labour

Approximately 60 persons will work on the site during the construction phase with personnel assumed to be 75% Greenlandic and 25% Danish. The workdays will be 12 hours per day, 7 days a week. In general, personnel will work on site for two weeks and have one week off site. Some personnel might work on site for four weeks and then have two weeks off site, depending on their home country, e.g. Denmark.

The camp manager will also function as the medical responsible person on site.

The construction crew will be divided into four groups where each group will be working on access roads, the camp and port sites as mentioned above.

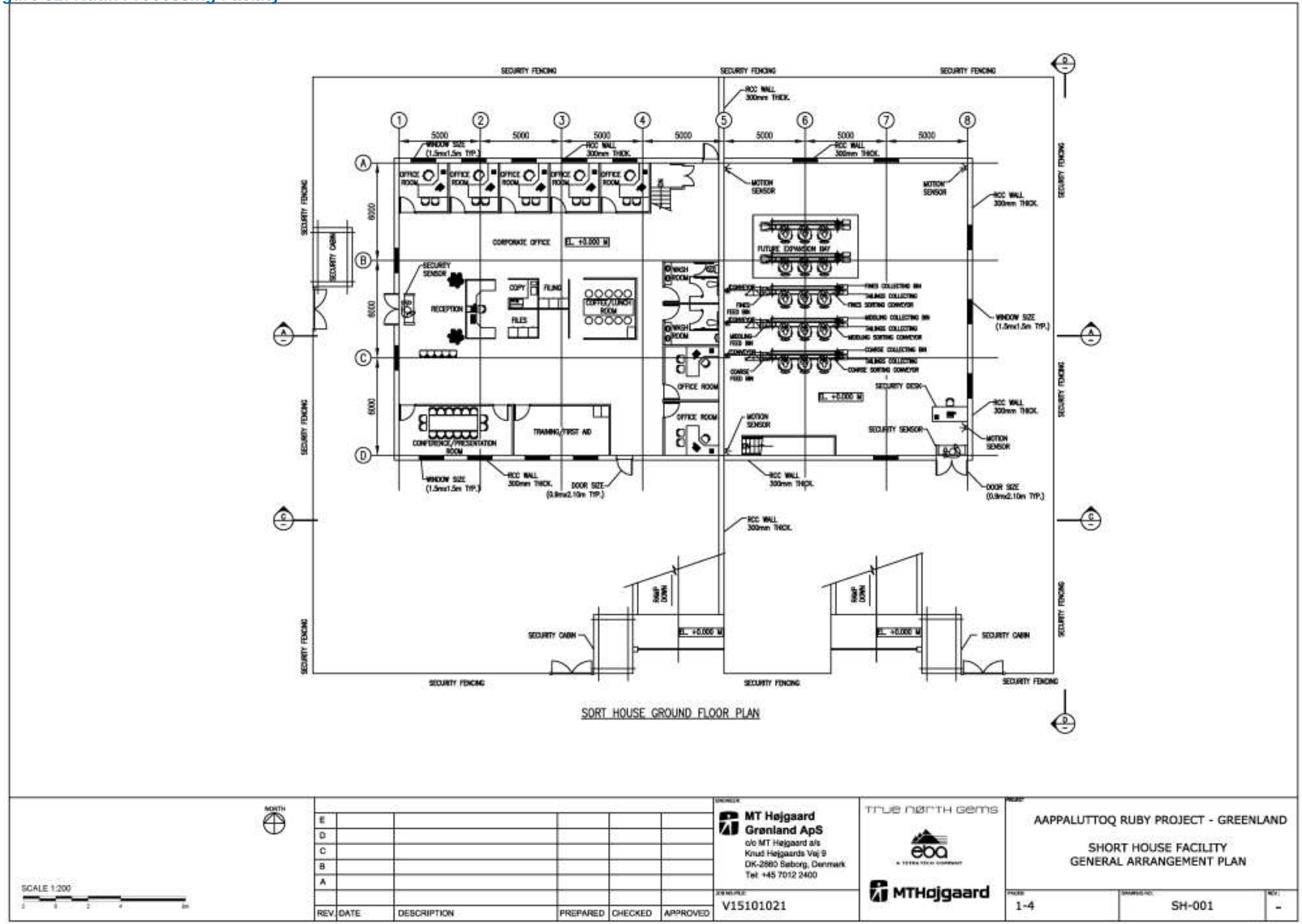
19.20 Nuuk Processing Facility

The Company will establish a facility in Nuuk which will form the center of its Greenlandic operations. This facility will house the corporate base of operations in Greenland and also be used for support of the mine site operations.

At the same location, but physically separated from the corporate offices, will be the sort house facilities where dirty rough concentrate is received from site, dispatched for cleaning, received from cleaning, sorted and subsequently shipped to markets (Figure 32).

This facility will initially be a small leased facility, with a purpose-built facility (either owned or leased) constructed once economics are established.

Figure 32: Nuuk Processing Facility



19.20.1 Corporate Base

The senior corporate representative of the Company in Greenland will be based at the facility in Nuuk and from there be the point of contact with the various levels of government and regulatory authorities as necessary for the Company to support its operations.

19.20.2 Mine Support

The construction and operation of the Aappaluttoq mine will require the transportation of personnel, supplies and spares to and from the site. This will be organized from the Nuuk office in coordination with the mine and any logistics personnel located elsewhere, such as in Qeqertarsuatsiaat. In addition, international transportation of major items of equipment and of expatriate personnel will also be handled from here.

The staff in Nuuk will purchase supplies for the camp and spare parts for equipment as requested by personnel at the site. Contracts for construction and supply of services will also be tendered through this office.

Personnel resources for operations will be arranged in conjunction with the supervisory staff at the mine. Training of new staff for work on site will be arranged from the Nuuk office.

The accounting department will be responsible for making payments to suppliers and will ensure workers payroll requirements are properly completed. Local reporting of taxes will also be handled through this office.

19.20.3 Nuuk Concentrate Handling

Dirty rough concentrate from site will be shipped by boat or helicopter in secure containers on an irregular schedule. They will be received at Nuuk and transported by secure carrier from the dock or heliport to the Nuuk facility.

The dirty rough concentrate received at the Nuuk facility will be checked to ensure that the shipment matches all documentation from the mine. The dirty rough concentrate is then cleaned in a process using hydrofluoric acid. This process requires the supervision of a highly trained chemist as the hydrofluoric acid (HF) is dangerous if not handled carefully and consequently. The Company will contract the work to a qualified laboratory in Greenland to ensure the operation is completed in the safest possible manner.

Accordingly, the dirty rough concentrate will be placed in appropriate sealed containers, weighed and logged, and then be dispatched by secure carrier to the treatment laboratory.

The HF acid treatment will remove unwanted silicate material remaining on the ruby concentrate and after rinsing and drying will leave a clean ruby concentrate which will be returned, again by secure carrier, back to the Nuuk facility for sorting.

The cleaned concentrate will be received and checked to ensure the shipment is complete. From there the concentrate will be sorted into gem/non-gem categories, and graded for size and quality. Initially, sorting will be by hand but tests are being undertaken to determine if optical sorting can be effectively employed. The gem quality material will be stored in the vault prior to export. Non-gem material and tailings will be stored prior to disposal.

19.20.4 Nuuk Facilities

The Company will either lease a facility in Nuuk or if a suitable facility is not available will develop one. It would be a strong preference that these facilities are in close proximity to the facility in which the hydrofluoric acid treatment takes place. A possible layout option of the Nuuk facilities is shown Figure 32 at the end of this section.

The building will preferably be in a secure fenced compound and access will only be gained by special permit. The compound will be covered continuously by security cameras that will be monitored in both on-site and off-site locations.

The building will be approximately 24 × 45 m long and have a full underground basement area which can be accessed by vehicles using ramps. The office area (approximately 24 × 20 m) which will house the corporate and mine support staff will be completely separate from the sort house area for security reasons.

The building walls will be constructed of concrete to protect against attempts to gain unauthorized access. The roof will be of steel truss construction. Special secure windows will be used in both office and sort house facilities. Windows will be as large as practical to provide optimum conditions for workers.

Under the office area will be a basement designed to receive secure carriers bringing dirty rough concentrate which has been delivered from the minesite. In that section of the basement the concentrate will be checked against dispatch records from the mine. Once the integrity of the shipment has been confirmed it will be logged into dirty rough concentrate inventory prior to being sent to the facility at which the hydrofluoric acid treatment takes place.

Acid treatment will take approximately 24 hours and the dry, clean, rough concentrate will be returned by secure carrier. The carrier will access the basement area below the sort house by ramp and the concentrate unloaded and checked.

The concentrate will then be transferred to the upper level and sorted into gem, non-gem and tailings in different size fractions. The gem quality material will then be stored in a large secure vault prior to shipping to market. Shipments will be carefully planned and will take place at irregular intervals.

Non-gem and tailings material will be stored in the basement for eventual shipment off site.

Both sort house and office areas as well as the basement areas will be equipped with several levels of security including HD security cameras monitored on and off-site, motion sensors and various biometric scanning devices. The security system will also be linked to the local police detachment. Access to the sort house will be restricted to sort house workers, senior management and security personnel.

19.21 Security

Theft of rubies is not only of concern in terms of an economic loss for the Company but it is also an issue in terms of a loss of potential tax that will not be collected by Greenland, reducing the benefit of the mine for all Greenlanders. Hence, security measures will be put in place and security protocols developed at the mine, process plant, during transport and at the sorting house in Nuuk such as fencing, security cameras and periodic security checks.

Staff will be required pass security clearance before employment. For the safety of all personnel, only persons without criminal records will be employed. The camp will be alcohol and drug free, and this policy will be strictly enforced.

Detailed security has been planned and costed for the PFS. Details have been excluded from this report as a precaution.

19.22 Ruby and Pink Sapphire Product

19.22.1 Physical characteristics

Ruby and pink sapphires are family members of the mineral corundum. A very small amount of chromium in the mineral corundum produces a pink colour: pink sapphire. A slight increase in chromium produces red corundum known as ruby. The pink colour graduates to red but there is no industry standard division between pink sapphire and ruby. The other family members of the corundum family are known as sapphires, which occur in many other colours. Ruby and pink sapphire crystals in the natural state are called "rough" and look like red pebbles. Rough has a wide range of quality, from non-gem (opaque, with many visible impurities) to gem (transparent) quality.

19.22.2 Market and uses

Anything other than the gem or near-gem portion of rough (which in bulk samples taken at Aappaluttoq represent about 1/3 of the corundum) has little or no value. Of the large rubies seen at Aappaluttoq and in the Fiskenæsset area, most large "rubies" are actually crystal masses consisting of more than one crystal in a conglomeration. These large masses generally contain very little material that is actually clear, gem or near-gem quality ruby.

Ruby and pink sapphire are predominately used in gem stones in jewellery. There was some use industrial utilising the natural hardness of corundum but have now been replaced by synthetic materials. Synthetic rubies are used in manufacturing lasers, natural stones being unsuitable due to their internal crystal flaws.

19.22.3 Expected size and value

The majority of corundum obtained from the Aappaluttoq mine is expected to be small and of low value with bulk samples returning approximately 90% (by weight) of rough smaller than 6.3 mm. It is expected that any larger gem quality rubies or pink sapphires found will be relatively easy to sell, but the market for smaller material is not well understood and will require extensive research and test marketing.

A Greenlandic exploitation (mining) permit is required before the Company can begins sales or be able to enter into agreements to sell ruby and pink sapphire. Once the required permits are granted, the Company will be able to cut and polish pink sapphire and ruby from Aappaluttoq.

No large gem quality rubies have yet been found at Aappaluttoq. So far the largest, finest polished ruby produced by the Company is a 0.69 carat round stone with an estimated wholesale value of $2,100 USD. Until Aappaluttoq has been mined for a period of time it is impossible to assess the likelihood of finding large, high-valued gem rubies, which could significantly increase the value of the deposit.

There is very little information available on prices for rough ruby. There can be a large differential between value of rough and polished rubies because even an experienced polisher will not know with certainty what can be polished from an individual rough crystal. Most rough contains inclusions and fractures that are not necessarily apparent, but can cause the stone to break on the polishing wheel. Prices for rough are lower because purchasers of rough take this risk.

Prices used in the economic model were based on three completely independent valuations of gem and near-gem stones obtained from bulk samples, which had been cut and polished. Two of the valuations were generally similar, a third was considerably higher. The prices used in the economic model were based on the higher of the two low values. A fourth valuation was obtained for rough gem and near-gem stones. This has been used for a portion of the sales.

Historical prices are impossible to present in any objective or practical way. This is typical of products that are by nature individual and value subjective.

From the valuations of the 2006 bulk sample, an average rough and polished price for ruby and sapphire was developed. Table 1 shows these prices and cutting costs. The cutting costs are based off $0.20 per stone for facetted stones and $0.25 per stone for cabochons.

Table 36: Ruby And Sapphire Prices And Cutting Costs

Item	Units	Value
Ruby rough price	$/g	0.39
Sapphire rough price	$/g	0.39
Ruby cut and polished price	$/ct	32.34
Sapphire cut and polished price	$/ct	25.60
Ruby cut and polish cost	$/ct	1.18
Sapphire cut and polish cost	$/ct	1.03

19.22.4 Product Branding

A marketing advantage of the rubies and pink sapphires produced in the Aappaluttoq mine will be their potential branding as Greenlandic. Today, most rubies originate in Burma and some countries have legal restrictions on the import of rubies and pink sapphires from Burma. Therefore, purchasers of Greenlandic ruby wishing to import into those countries may require documentation to prove that that the rubies are from Greenland and not Burma. The Company will be able to supply this documentation to purchasers of rough and polished stones who buy directly from the Company.

A 2% per year price escalation for product branding has been included in the project economics. Table 37 below demonstrates the escalation in product prices.

Table 37: Product Price Escalation

	year	0	1	2	3	4	5	6	7	8	9	Average
Escalation	%	0	2	2	2	2	2	2	2	2	2	1.8
Ruby rough	$/g	0.39	0.40	0.41	0.42	0.43	0.44	0.44	0.45	0.46	0.47	0.43
Sapphire rough	$/g	0.39	0.40	0.41	0.42	0.43	0.44	0.44	0.45	0.46	0.47	0.43
Ruby polished	$/ct	32.34	32.99	33.65	34.32	35.01	35.71	36.42	37.15	37.89	38.65	35.41
Sapphire polished	$/ct	30.60	30.60	30.60	30.60	30.60	30.60	30.60	30.60	30.60	30.60	27.75
Indexed value	%	100.0	102.0	104.0	106.1	108.2	110.4	112.6	114.9	117.2	119.5	-

19.23 Operating Costs

Total operating costs over the life of the project amount to $101 million equating to $540 per tonne of ore mined (Table 38). Costs are based on a combination of historical quotes, similar projects, cost estimation guides, engineering calculations and experience. These costs may be sensitive to operation size and cannot be scaled without further cost estimation work.

Table 38: Operating Costs

Item	Annual average	Life of mine total
	$'000	*$'000*
Mining	1,922	17,298
Processing	1,805	16,244
General and admin	1,785	16,068
Nuuk	3,797	34,175
Marketing	440	3,958
Corporate	440	3,958
Contingency	862	7,759
Total	11,051	99,459

The operating cost estimate includes all recurrent costs for payroll, service contractors, consultants, camp operations, maintenance parts and supplies, consumables, supplies, freight charges, transportation of personnel etc. to operate all facilities as described in this feasibility study. Operating expenses are defined as any recurrent expenditure which can be expensed in the tax year in which it occurs.

All work activities on site are based on a 12 hour working day, with a seven day working week. The operation will run between 4 to 8 months per year. One crew in the mining operations will work a 2 week on, 1 week off roster. Other personnel will work a 1 week on, 1 week off roster with two crews rostered. Over eight months, this equates to 1,956 hours per year for mining operators, and 1,464 hours for other staff.

The working schedule for personnel working in the Nuuk facilities will be weekdays (Monday to Friday), eight hours a day, 40 hours a week (1,920 hours per year).

19.24 Capital Costs

The total capital costs for the Aappaluttoq Ruby Project including Nuuk over the life of mine is $40.7 M excluding any salvage value (Table 39). This includes sustaining capital and reclamations costs but excludes capital salvage.

Table 39: Capital Costs

Item	Initial	Sustaining	Total
	$'000	*$'000*	*$'000*
Mining	11,062	0	11,062
Processing	6,333	430	6,763
Tailings handling	141	0	141
Camp site, infrastructure & facilities	3,625	0	3,625
Temporary services	69	0	69
Inner port, staging area & fuel	485	0	485
Outer port, barge & staging area	728	0	728
Roads	2,929	0	2,929
Nuuk	280	0	280
Indirects	7,398	2,267	9,665
Contingency 13.8%	4,389	561	4,950
Total	37,439	3,258	40,696

Capital costs are considered to be within a ±25% accuracy range of actual costs. All costs are expressed in USD, with no allowance for escalation (inflation), interest during construction or taxes.

Costs are based on a combination of historical quotes, similar projects, cost estimation guides and engineering experience. These costs may be sensitive to operation size and cannot be scaled without further cost estimation work.

19.25 Economic Analysis

The mine plan base case has a 9 year mine life that recovers a majority of the indicated resources. This scenario provides for a balance of economic return and mine life. A longer mine life will allow the Company to conduct further exploration while producing. It is possible to return a slightly better economic result with a shorter mine life.

The Aappaluttoq Ruby Project returns a positive pre-tax Net Present Value (NPV) of $25.7 M at an 8% discount rate and an IRR of 19.1%. The after-tax NPV is $17.5 M at 8% discount and the IRR is 16.5% with an after-tax payback period of 5½ years. Total estimated corporate taxes payable is $16 M. (Table 42, p 144)

Discussions with project funding agents are in preliminary stages. For the purpose of this application, 100% equity funding is assumed.

Key assumptions used in the analysis are shown in Table 40 below.

Table 40: Key Economic Parameters

Item	Units	Value
Mining productivity	*t/d*	3,300
Diesel price	*$/L*	1.08
Process productivity	*t/d*	117
Site recovery rate	*%*	95
Ruby rough price	*$/gr*	0.39
Sapphire rough price	*$/gr*	0.39
Ruby polish price	*$/ct*	32.34
Sapphire polish price	*$/ct*	25.60
Acid washing upgrade	*%*	65
Polishing retention	*%*	9.3
Discount rate	*%*	8

19.25.1 Product Revenue

The mining schedule provides an annual corundum production figure based on the block model grades, the mining productivity rate and various other physical factors. A recovery factor of 95% is applied to the grade to arrive at the estimated corundum recovered.

The economic analysis assumes that all ruby production will be sold as polished gemstones, and that 60% of the pink sapphire production will be sold as polished gemstones and 40% sold as rough.

The percentages of gem and near-gem rubies and sapphires contained in that corundum are based on the percentages determined from the analysis of the B1 and B2 bulk samples taken in 2006 and 2007. This information is shown in Table 41. The combination of the B1 and B2 were the most representative samples to use in the estimation of gem distribution characteristics as there was increased knowledge of the deposit and greater scrutiny of the samples. These samples are detailed in sections 10.1 Surface Sampling and 12.2 Bulk Sample.

Table 41: Gem Distribution Characteristics of B1 and B2 samples

		total weight	Distribution	Total weight	Distribution	Total weight	Distribution
		g	%	g	%	g	%
Gem	Red	744	2.6%	426	0.4%	1,170	0.9%
	Pink	3,669	13.0%	8,035	7.9%	11,704	9.0%
Near-Gem	Red	3,049	10.8%	2,248	2.2%	5,297	4.1%
	Pink	4,356	15.4%	28,681	28.1%	33,037	25.3%
Non-Gem	Red	1,711	6.1%	12,513	12.3%	14,224	10.9%
	Pink	14,701	52.1%	50,217	49.2%	64,918	49.8%
Total	Red	5,504	19.5%	15,187	14.9%	20,691	15.9%
	Pink	22,725	80.5%	86,933	85.1%	109,658	84.1%
Gem	Both	4,413	15.6%	8,461	8.3%	12,874	9.9%
Near-Gem	Both	7,405	26.2%	30,929	30.3%	38,334	29.4%
Non-Gem	Both	16,412	58.1%	62,730	61.4%	79,142	60.7%
Total	Both	28,229	100.0%	102,120	100.0%	130,349	100.0%

Revenue has been estimated for the recovered gem and near-gem rubies and sapphires based on the pricing discussed in section 19.22.3 Expected size and value above. A small incremental price increase (2% per annum) has been allowed for market acceptance of the product as discussed in section 19.22.4 Product Branding. This is not a price escalation factor but a result of marketing activities by the Company, as has been experienced (to a much larger degree) by other companies in similar operations.

No revenue has been estimated for the non-gem corundum although it is anticipated that a market can be found for this material.

19.25.2 Project Cashflow

The project becomes cashflow positive in year 2 and remains positive for the remainder of the mine life (Table 42, p 144).

19.25.3 Project NPV and IRR

The Aappaluttoq Ruby Project returns a pre-tax NPV of $26.1 M at an 8% discount rate and an IRR of 19.3%. The after-tax NPV is $16.7 M and the IRR is 16.2%.

Payback of capital is achieved after 5½ years (Table 42, p 144).

Table 42: Project Economics

	Year	0	1	2	3	4	5	6	7	8	9	10	Total
Mining													
Waste	*kt*	0	237	213	426	421	400	335	237	179	168	0	2,616
Ore	*kt*	0	1	7	20	21	22	22	22	23	24	0	162
Total	*kt*	0	238	220	445	442	422	357	259	202	192	0	2,777
Ore Grade													
Ruby	*g/t*	0.0	17.1	88.8	64.2	46.0	47.8	50.4	46.5	42.2	81.6	0.0	55.6
Sapphire	*g/t*	0.0	90.7	470.7	340.3	243.8	253.1	266.9	246.6	223.5	432.6	0.0	294.7
Total Al203	*g/t*	0.0	107.8	559.5	404.6	289.9	300.8	317.3	293.2	265.7	514.2	0.0	350.4
Revenue													
Ruby	*$'000*	0	86	2,870	6,043	4,701	5,173	5,551	5,163	5,110	10,712	0	45,409
Sapphire	*$'000*	0	290	9,636	20,293	15,786	17,373	18,641	17,337	17,160	35,974	0	152,491
Total	*$'000*	0	376	12,506	26,336	20,487	22,546	24,192	22,500	22,270	46,687	0	197,900
Operating Costs													
Site	*$'000*	0	3,415	3,386	6,368	6,361	6,342	6,256	6,038	5,967	5,952	0	50,086
Nuuk	*$'000*	0	2,195	3,361	4,113	3,866	3,908	3,957	3,864	3,836	4,597	0	33,698
Marketing	*$'000*	0	8	250	527	410	451	484	450	445	934	0	3,958
Corporate	*$'000*	0	8	250	527	410	451	484	450	445	934	0	3,958
Contingency @ 8.5%	*$'000*	0	496	639	984	946	950	944	906	891	1,003	0	7,759
Total	*$'000*	0	6,121	7,886	12,519	11,992	12,102	12,125	11,708	11,585	13,420	0	99,459
Capital Costs													
Site	*$'000*	4,447	20,926	430	0	0	0	0	0	0	0	-7,102	18,700
Nuuk	*$'000*	56	224	0	0	0	0	0	0	0	0	0	280
Indirects	*$'000*	1,480	5,918	0	0	0	0	0	0	0	0	2,267	9,665
Contingency 13.8%	*$'000*	878	3,511	107	0	0	0	0	0	0	0	453	4,950
Total	*$'000*	6,860	30,579	537	0	0	0	0	0	0	0	-4,382	33,594
Financial Analysis													
Revenue	*$'000*	0	376	12,506	26,336	20,487	22,546	24,192	22,500	22,270	46,687	0	197,900
Operating	*$'000*	0	-6,121	-7,886	-12,519	-11,992	-12,102	-12,125	-11,708	-11,585	-13,420	0	-99,459
Capital	*$'000*	-6,860	-30,579	-537	0	0	0	0	0	0	0	4,382	-33,594
Pre-tax cashflow	*$'000*	-6,860	-36,324	4,083	13,818	8,495	10,444	12,067	10,791	10,685	33,266	4,382	64,847
Estimated Tax	*$'000*	0	0	0	0	0	0	0	440	3,210	12,309	0	15,959
After-tax cashflow	*$'000*	-6,860	-36,324	4,083	13,818	8,495	10,444	12,067	10,351	7,475	20,958	4,382	48,887

19.25.4 Corporate Tax

Corporate tax for companies with a license under the Mineral Act is levied at a flat rate of 30%. The taxable income is determined on the basis of the profit shown in the statutory annual report, adjusted to comply with the prevailing tax provisions.

Greenlandic companies are to withhold a dividend tax corresponding to the personal tax in the municipality of registration. For companies with permits under the Mineral Resources Act, the present dividend tax rate is 37%. Because dividends are allowed as a deductible expense, the effective rate of taxation in Greenland for mineral companies is lower.

Companies with exploration or utilization permits under the Mineral Resources Act are entitled to carry forward tax losses without timing limitations.

If the taxpayer has calculated a tax profit, it is possible to make further tax amortization of 50% of the profit. The amortization may freely be deducted from the balance in one of the said three amortization/depreciations groups, however, provided that the remaining balance of the group remains positive or DKK zero.

Companies with utilization under the Mineral Resources Act may, when computing their taxable income, deduct any change in the provisions made to ensure that a closure plan can be carried out in a financially viable way.

Taxation law, both in Greenland, Denmark and Canada are complex subjects and this section serves as a general summary of some of these tax implications. This section has been reviewed, but not verified by John Chow, MAusIMM, MIEAust. Further discussions with the Greenlandic government and expert tax advice will be needed up to and into operations.

19.25.5 Sensitivities

The project is sensitive to the proportion of sapphire sold as polished material, then price, grade/ore tonnes, operating costs and capital costs in this order (Table 43). This information is also presented in Figure 33.

Table 43: Project Sensitivities

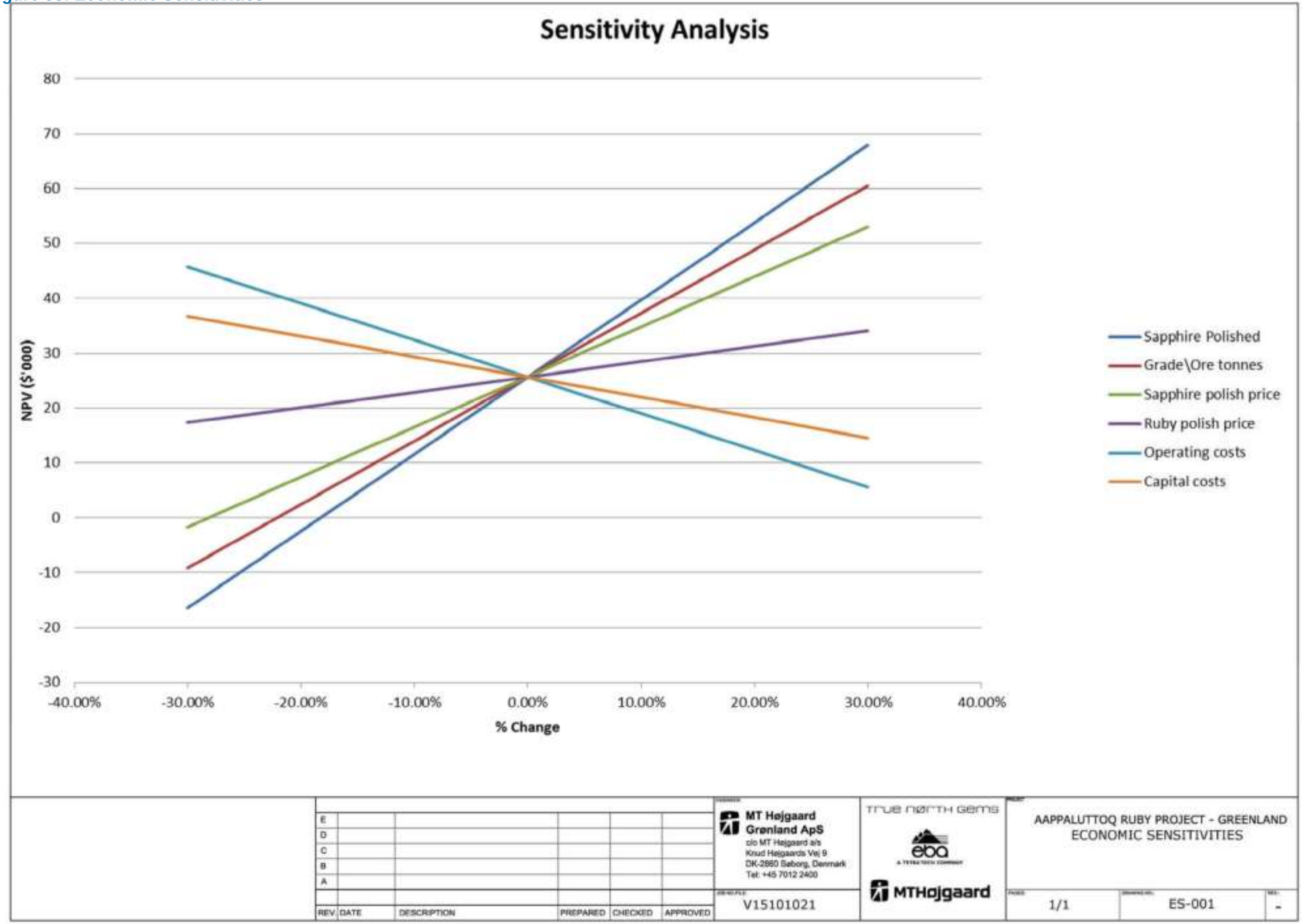
Item	-30%	-10%	-5%	0%	+5%	+10%	+30%
Proportion of Pink Sapphire sold as polished	-16,495	11,617	18,645	25,673	32,700	39,728	67,840
Grade	-9,143	14,067	19,870	25,673	31,475	37,278	60,488
Ore tonnes	-9,143	14,067	19,870	25,673	31,475	37,278	60,488
Sapphire polish price	-1,718	16,542	21,107	25,673	30,238	34,803	53,063
Ruby polish price	17,337	22,894	24,283	25,673	27,062	28,451	34,008
Operating costs	45,731	32,359	29,016	25,673	22,329	18,986	5,614
Capital costs	36,741	29,362	27,517	25,673	23,828	21,983	14,604

Note:
* base case at 0% change refers to 60% of pink sapphire sold as polished.

The greatest risk for the project is product price. The market as explained in section 19.22.2 Market and uses, does not act like a traditional mining market like iron ore or gold.

Further work or research cannot reduce this risk without the actual mining, processing and sales of the Greenland ruby and sapphire material.

Figure 33: Economic Sensitivities



19.26 Environmental and Permitting

The Company is proceeding with its Environmental Impact Assessment (EIA) which is a required component of the Company's Exploitation Permit, and a draft EIA was filed with the Greenland Government on June 6, 2011.

The purpose of the EIA is three-fold:

- The formal and legislative purpose is to provide the decision-makers with a proper basis for their decisions on the licence and the conditions that may apply, and
- To provide a basis for public consultations. The local knowledge and priorities are taken into account and can be incorporated in the project
- A third, but equally important purpose is to provide inputs during the feasibility and design phases in order to mitigate negative impacts in an early stage of the project development.

The EIA consider the effects of planned activities (e.g. building of roads, storage of tailings) and their intentional (e.g. lowering the lake water level) and unintentional effects (e.g. leakage of heavy metals). Unplanned and unlikely activities/accidents (e.g. ship wreckage) are not considered.

The EIA process is underway and this section serves to summarise the process. John Chow, MAusIMM, MIEAust has reviewed this information and found it suitable for inclusion into the PFS. It must be highlighted that both the process is ongoing, and that the EIA process for mining is relatively new in Greenland.

The first step of the assessment is to describe and rate the impacts of the project activities during construction and operation and after closure of the mine according to the intensity and reversibility, the geographical reach and to the duration of the effects.

The descriptions and ratings of the many different impacts can be condensed to more simple statements on the environmental impacts. Techniques range from simple score-cards over GIS-tools of varying complexity to comprehensive simulation models. But all techniques are based on knowledge of sensitivity and importance or coverage of the different types of environmental features considered..

19.26.1 Baseline Studies and Data Sources

The environmental base line studies have been implemented with the principal objectives of delivering data for the unperturbed environment prior to the mining activities. The data provide a necessary background for the impact assessment and the studies are a mandatory requirement for acquisition of an Exploitation License.

Another important objective of the baseline studies has been to determine the state of the environment regarding heavy metals and other harmful substances. This part of the study is used as a frame of reference for the monitoring activities during the lifetime of the mine and after closure.

The Aappaluttoq ruby prospect reaches down beneath the surface of the Lake Ukkaata Qaava and a third objective was to study the lake hydrology and basic hydrological features in case a lowering of the water table would be necessary. These studies have been carried out during 2007, 2008 and 2009.

Local knowledge by residents and users of the area is an important source of information and this has been obtained through consultations with residents from Qeqertarsuatsiaat.

Archive search and archaeological fieldwork has been carried out in 2010 by the Greenland National Museum and Archives.

Other data sources are publications from the Greenland Institute of Natural Resources, the National Environmental Research Institute (NERI), Danish Meteorological Institute, and the scientific literature in general.

19.26.2 Environmental Management Plan

The Environmental Management Plan (EMP) deals with all aspects of the mining operation including the production and shipment of the rubies. The EMP is worked out to minimize negative impacts of the mining operation during construction phase, operation phase and decommissioning phase.

The EMP addresses the issues that arise from the present Environmental Impact Assessment (EIA) and ensures that management actions are clearly defined and implemented throughout the above mentioned phases. The EMP includes instructions regarding documentation of environmental performance during the activities and regarding measures to be taken in order to meet changes in the project and unforeseen events. The EMP puts into practice a system that meets the requirements for registration under ISO 14001.

A senior manager will be appointed as Environmental Health and Safety (EHS) Manager responsible and accountable for all environmental and compliance aspects of the operation together with health and safety aspects. The EHS Manager will be responsible for the maintenance and update of the EMP together with all monitoring and sampling requirements for environmental purposes and for all actions required to maintain compliance.

19.26.3 Environmental Monitoring and Control Systems

An environmental monitoring program focused on pollution has been proposed for the mining phase including:

- The fresh water system, comprising the mine pit water, the two basins of the lake Ukkaata Qaava, and the stream leading from the lake to the brackish fjord Tasiussaq. In addition, samples for references are sampled in the tributaries to the lake and in freshwater systems not connected to the Ukkaata Qaava-system
- Lichens in the terrestrial area on fixed positions around the mining area and infrastructure facilities
- Marine biota including macroalgae, mussels and fish in Tasiussaq, Tasiussarssuaq and reference areas close to the open sea
- Sediment samples in the lake and in the fjord

19.26.4 Analyses and Sampling Frequencies

The sampling frequency varies from two weeks for the pit water to annually sampling of the marine biota. The monitoring program is described in the EIA report section 9.

After mine closure and restoration of the lake water level, a monitoring pro-gram is proposed with the above program as a starting point. A ten-year program is suggested but the actual monitoring parameters and sampling frequency will be agreed with BMP based on the monitoring data collected during the mining phase. Further considerations are found in the EIA report section 10.

19.26.5 Draft EIA Conclusions

Lowering the lake will leave 1/3 of the lake dry and barren. Thus, the mine pit and the process plant will have large esthetic impact on landscape around the lake and pit, but it is only visible from a short distance due to the topography. Most of this impact is neutralized when the lake level is restored and the mine pit is covered with water as part of the closure plan. The most visible and long lasting effect on the landscape will be from the quarry used for construction materials before opening the mine pit and from the roads and staging areas which will be visible for an extended period after closure.

The lake itself is deep, very clear, with little vegetation, and no fish or birdlife. The lowering of the surface level, building dikes, depositing waste rock and tailings will reduce clarity and change the lake dynamic. The water residence time is more than one year in each lake basin and no significant amount of suspended matter is expected to reach the outlet.

The potential for acid generation and for metal leaching from the deposited material has been examined. The conclusion is that the tailings have no acid generating potential. 30% of the waste rock is potentially acid generating but pose no threat if disposed under water where the low oxygen availability limits acid generation to negligible rates.

Short term leaching tests suggest that only very minor metal leaching should be expected from these materials. Monitoring of runoff and lake water is recommended, but mitigating measures are not expected to be necessary. The lake is expected to slowly restore its dynamic and clarity when the mine closes. The surface level will be restored within 2-3 years of re-filling the outlet trench.

No rare or protected plants have been found in the area. The mine pit, camp, staging areas, roads and other infrastructure may occupy an area of up to 120,000 m^2 (12 ha), now mostly covered by vegetation. Compared to the vast extent of these vegetation types, this is insignificant and the impact is regarded as small and local, although the rehabilitation of the vegetation in areas use for roads and other barren areas is very slow.

The three terrestrial mammals, caribou, fox, and hare are found abundant in the area throughout the year and are hunted in season. The nearest caribou calving area is about 60 km away, and will not be disturbed. The loss of grazing area due to the project is regarded as insignificant.

Few birds are found in the project area, but ptarmigan are common and hunted by local hunters. White tailed eagle is nesting north of the inner gate and by the shore 9 km southwest of the mine. Common eider and razorbill are abundant during the summer season In Tasiussarssuaq, And 20 km south of the area is the shallow fjord area Ikkattoq which is designated as a Ramsar area, i.e. an area of international importance due to the many nesting and moulting waterbirds and to nesting white tailed eagle. White tailed eagle is sensitive to helicopter traffic during breeding season. Helicopter traffic to the outer port should be minimized and flying north and east of the Inner gate should be avoided completely. The Ikkatoq area should also be avoided, but is not within the usual transport corridors. If these rules are observed only minor local impacts on the wildlife are expected during the construction and operation of the mine.

19.27 Social Considerations

19.27.1 Social Impact Assessment

The Company is required to prepare a Social Impact Assessment (SIA). The objective of the SIA is to identify, analyze and monitor the social impacts and benefits associated to the proposed mining project. The Aappaluttoq Ruby project is small and the potential social negative effects are expected to be limited, and considerable positive effects are expected. The draft SIA was filed with the Greenland government on June 6, 2011.

The SIA covers the construction stage, the operational stage and closure of the proposed mine. The draft SIA describes the socioeconomic baseline in Greenland and in the main affected areas, evaluates likely socioeconomic impacts related to the project and identifies measures to mitigate negative impacts requiring mitigation. The social impacts have been assessed at two different levels: potential local impacts in Qeqertarsuatsiaat and impacts at regional and national level (Nuuk, Sermersooq Municipality and at national level) to ensure a socially sustainable development of the mine.

To conduct the SIA, the Company has brought in the expertise of the company Grontmij | Carl Bro, to work along with Greenlandic authorities, experts, institutions, local people and groups of interest such as fishermen, hunters, artisans, and others. By participating in the SIA activities, local people are able to raise subjects that are of particular interest, in connection with the following activities of the SIA process:

- identify the main issues and concerns that need to be addressed in the SIA
- conduct individual and group interviews to establish a social baseline study
- propose what can be done to optimize the opportunities created by the mining project development and minimize any potential negative impacts that come with it (Impact and Benefit Plan)
- organize public hearing to discuss the data collected for the final SIA report
- provide input to the development of the impact and benefit agreement

The SIA study is expected to be finalized by late 2011.

The SIA process is underway and this section serves to summarise the process. John Chow, MAusIMM, MIEAust has reviewed this information and found it suitable for inclusion into the PFS. It must be highlighted that the process is ongoing, and that the SIA process for mining is relatively new in Greenland.

19.27.2 Expected Benefits to Greenland

The ruby mining project in Aappaluttoq will contribute to the Greenlandic economy through employment and through the payment of corporate tax by the Company's Greenlandic company, Kitaa Ruby A/S. A central computerized tracking system will contain all relevant information about the rubies mined and sold and all the information will be accessible for audits by the Greenlandic authorities.

The project will benefit the local and national Greenlandic population by creating jobs and business opportunities. During the construction phase employment is expected to peak at approximately 60. During the operations phase approximately 63 people will be required seasonally at the Aappaluttoq site and 14 to 21 people in Nuuk once full production is reached. There will be a need for all skill levels, and priority for employment will be given to Greenland nationals. The Company will provide training and capacity building for the mining, processing and promotion of this new Greenlandic product.

There will be a need for local service providers and suppliers such as barge, boat and helicopter charters, construction contractors, equipment suppliers, fuel merchants, mechanical and electrical parts dealers, expeditors, food wholesalers, among others. Support business for the mine is expected to be split between Qeqertarsuatsiaat and Nuuk. The Company will engage Greenlandic service and suppliers to the maximum extent possible during construction and operation of the mine.

The encouragement of local lapidary shops and artisanal jewellery design by providing supplies and workshops to improve people's techniques will be a prime objective of the Company. Through increased awareness of Greenland and Greenland gemstones because of international marketing, the Company believes that local jewellery producers will see increased demand for their products. Depending on the participation of the Greenland government, a possible additional benefit of the project will be the growth of Greenlandic expertise in gemstones to create structures, procedures and processes that will allow other miners, including small-scale gemstone mining operations in Greenland, to develop their business.

19.27.3 Employment

During the construction phase, the Company expects to utilize a local lead contractor such as MTHojgaard to manage local contractors. Employment is expected to peak at approximately 60 people who will be employed for road construction, facility construction, assembly of the processing plant and preparation of the mine itself. The Company will ensure that local participation is maximized to ensure that mining and construction skills and infrastructure are created locally in partnership with the community.

During operations, the Company estimates a site workforce of 63 people seasonally at Aappaluttoq and 14 to 21 in Nuuk once full production is reached. After a period of training, the Company expects that all positions can be occupied by Greenlanders with few exceptions. For Aappaluttoq operations, the Company initially believes for a majority of staff, a rotation of one week at the site and one week off (7 shifts of 12 hours each) will be optimal, but will make efforts to be flexible. Staff involved with the mining will work a two weeks at the site and one off (14 shifts of 12 hours each).

Because the Company will have a very small operation, personnel will work on various different aspects of the operation so that, for example, road maintenance can be undertaken during periods when persons and equipment are available from process or mining operations.

Most of these jobs will be seasonal, and the length of the operating season will depend on factors such as weather and ruby production.

Education and training, in particular:

- on the job training for all the operators at mine front, processing plant and sorting house will improve significantly the employability of all workers involved in the operations
- trainee positions for students from the Greenland School for Minerals and Petroleum and other vocational schools

19.28 Mine Closure and Decommissioning

The closure period is anticipated to last 3 months. The handling of closure and decommissioning comprises:

- Buildings, mobile plant and fixed plant will be salvaged
- Organic material is burned on site
- Concrete is crushed, spread and covered with soil
- Steel and other metals are shipped out on a bulk carrier for sale elsewhere
- All other demolished materials are removed from the site
- Road materials is loosened and spread
- The dike is excavated to 2 m below normal level of the lake
- Drainage channel is blocked with approximately 2 m^3 of concrete and general mine waste

It is estimated that 50 t of industrial waste will need to be disposed of. The waste is shipped out to Denmark but it might be possible to negotiate with the nearby towns.

19.28.1 Waste Dump Remediation

At the end of the operation, the drainage channels will be blocked and the lake will be allowed to refill to its natural level. This will mean that all waste material apart from those used for roads will be covered in a minimum of 10 m of water. Subaqueous deposition of waste rock and tailings material in the lake will minimize sulphide weathering and reduce potential acid generation to negligible rates by limiting exposure to free oxygen.

Figure 34 Figure 35 and Figure 36 show the before, during and after mine design with the surface of Lake Ukkaata Qaava. No waste rock or tailings piles will be visible after reclamation.

Figure 34: Before Mining

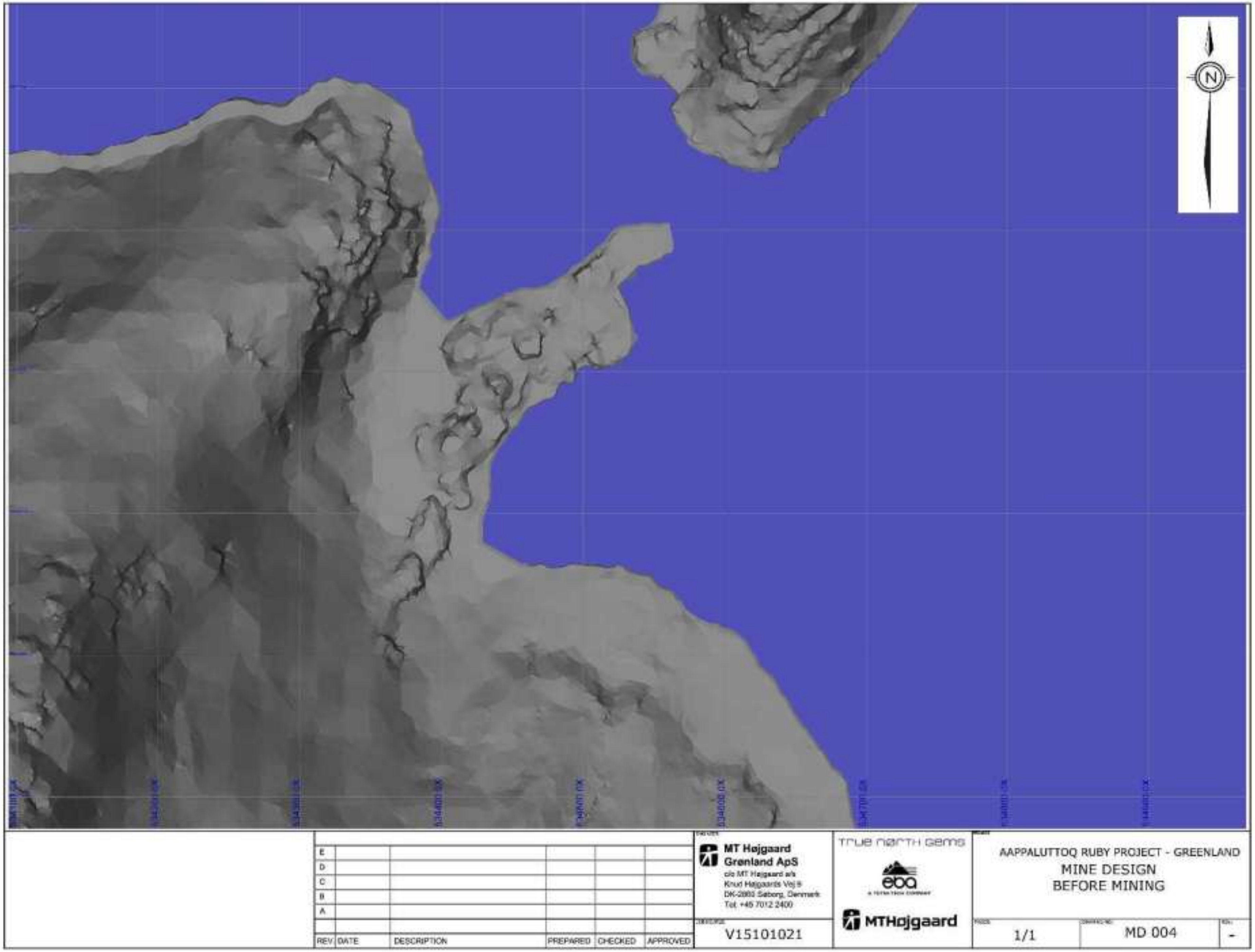


Figure 35: During Mining

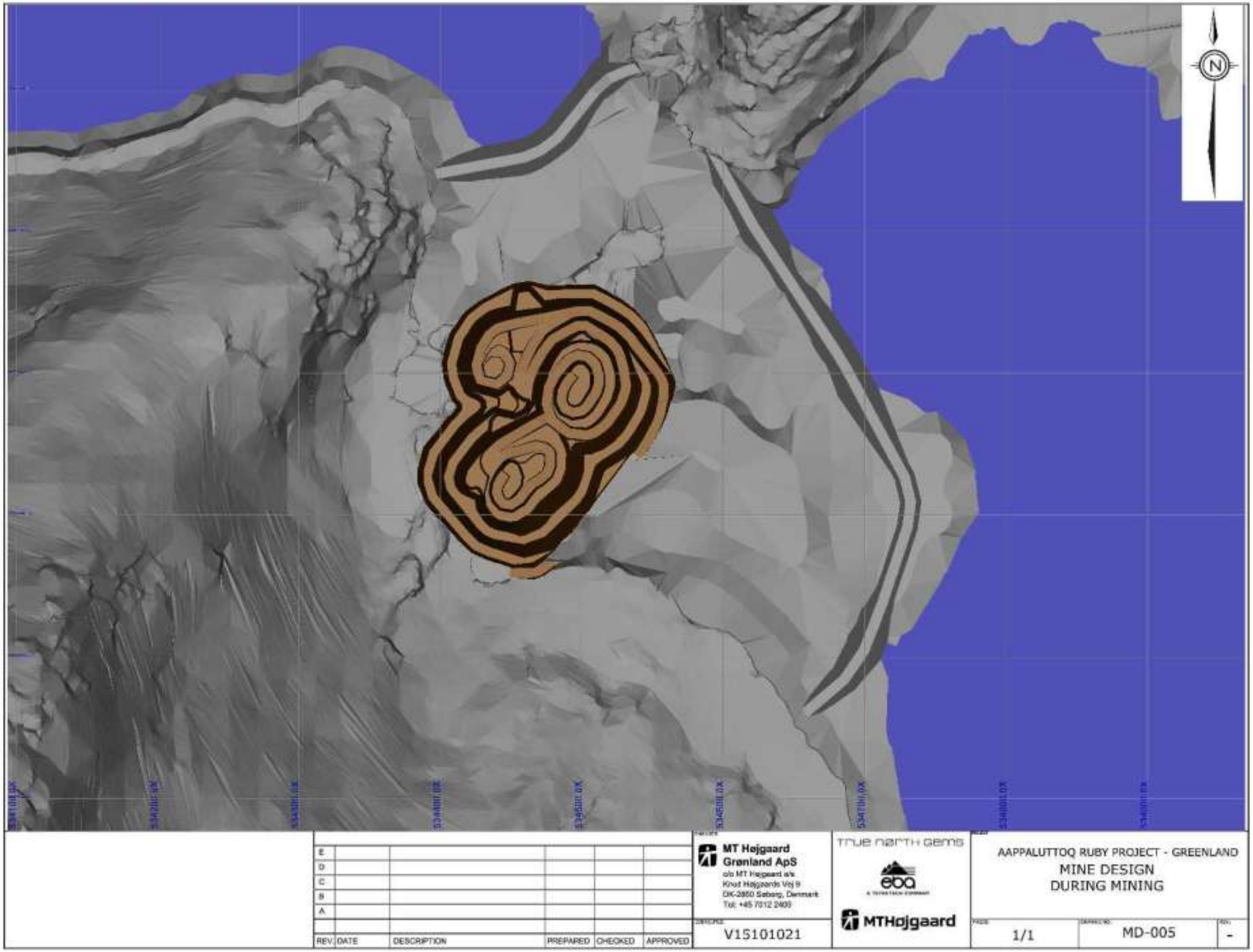
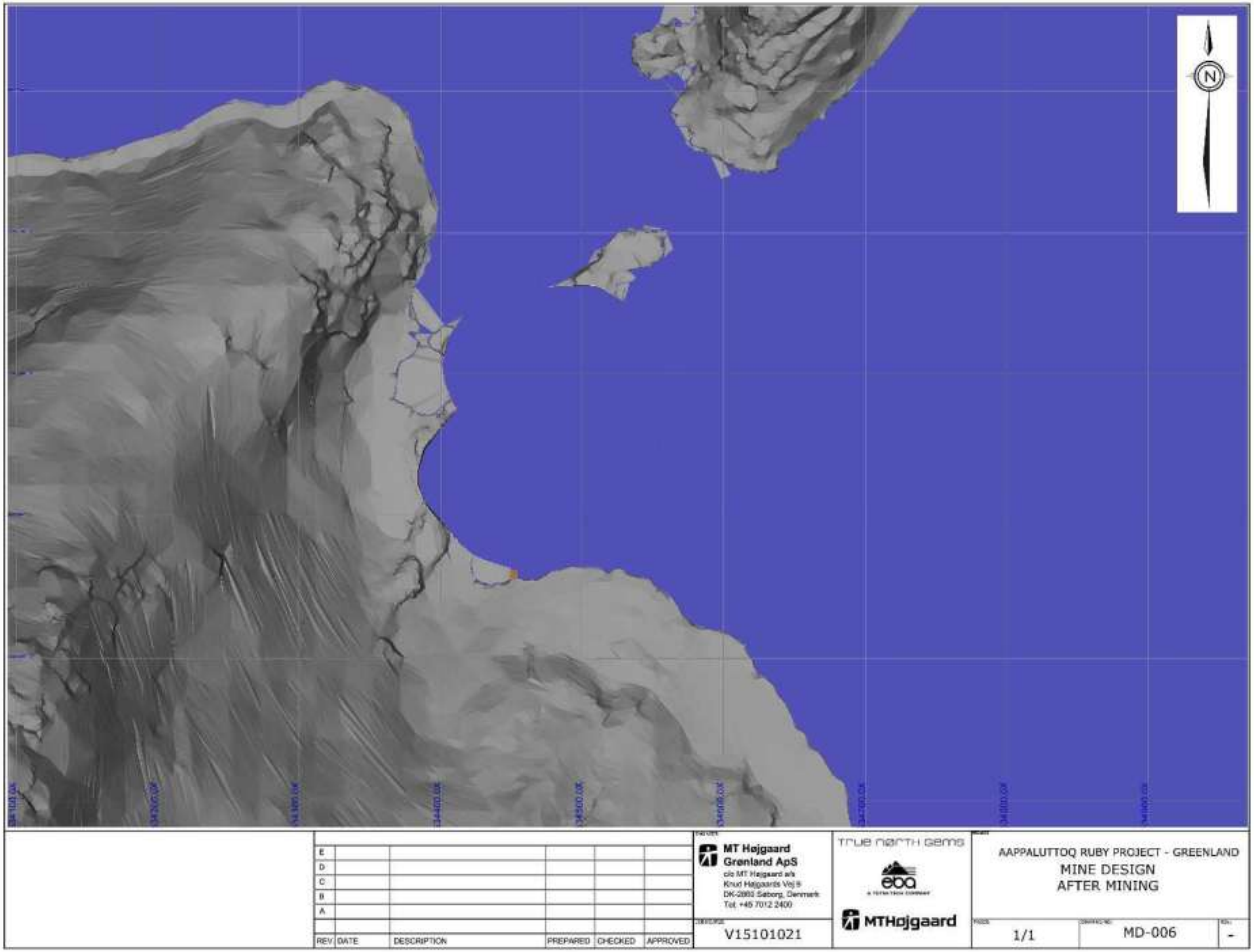


Figure 36: After Mining



20 INTERPRETATION AND CONCLUSIONS

The results of the Pre-Feasibility Study model for the Aappaluttoq Ruby Project show that this project has good economic potential even under the conditions of a conservative product prices and market.

The metallurgical characteristics of the rock are the most significant technical unknown at this stage.

The market and product prices are economics risks that are unlikely to benefit from further study and analysis. It is reasoned that operations will need to commence before these unknowns can be understood.

It is unlikely that will be significant environment social or environmental impacts on the local or national surroundings.

The Qualified Persons for this report recommend that the Company move to the next stage of the engineering process by the required work in preparation of commencing production.

A budget of approximately $1.5 M is recommended to complete exploration field programs, engineering and testwork including:

- Testwork of metallurgical characteristics of the rock for crushing and sorting
- Testwork on corundum recovery
- Sourcing and quotations on major equipment items

Further drilling and survey is also recommended to increase the geotechnical understanding which can be done during early stage operations.

21 RECOMMENDATIONS

21.1 Mineral Resource Recommendations

EBA recommends that the Company maintain a comprehensive complete database of QA/QC procedures for future audits and reviews.

Future drill programs should include downhole survey data collection for all boreholes, as well as survey of drill collar locations. It is recommended that continuous sampling through mineralized intervals be completed for any new sampling, preferably with sampling extending to 1 m beyond the mineralized zone to capture any bleeding of mineralization beyond the visible host zone.

Any future bulk sample or surface sample data should be surveyed and be mapped and documented during collection.

21.2 Mineral Processing Recommendations

To verify the wet screening equipment recommendation it is recommended to perform a screening test on representative sample material.

To verify the mineral jig equipment recommendation and to determine the exact tapered wedge bar bedding screen panels required to make the desired products from the three feed streams proposed (fine, medium, and course size fractions), it is recommended to perform a screening test on representative sample material at a suitable laboratory testing site. Mineral Jigs can have a wide range of performance capability and there is potential to select smaller units for the duty envisioned for this corundum concentration process.

There is a concern that expected performance of the current simple Mineral Jig selection may be too sensitive to the skill applied by the Concentration Operator running the plant. Skilled Mineral Jig operators may not be available for the continuous staffing needs at the Aappaluttoq project location. Alternative jigs, more advanced versions that are fully PLC controlled are recommended to be considered in further project study and development. Jigs which create and modulate the 'jig pulse' utilizing air are a modern development of the older jigging technology applied in the 2007 Fiskenæsset pilot plant. The advantage of using low pressure air to create the pulsing action is the superior control it provides over the pulse characteristics (shape, amplitude, and frequency). Industry experience is that pulse characteristics and equipment condition are the two key factors that directly influence the efficiency of the mineral separation in the jigging process. The Company is recommended to investigate the potential advantages such modern jigs may have for this project.

The current engineering study equipment selections for the crushing plant have been made without the benefit of a solid inventory of technical parameters which define the rock characteristics to be encountered as regards the Aappaluttoq ore. Work Index, "crushability", and Abrasion Index, "wear profile" are two key pieces of science that need strong definition and confidence for successful Crusher selection. It is recommended to complete additional tests on a selection of ore to generate a basket of data that give definition to the characteristics of the rock mechanics for the Aappaluttoq ore. This will enable optimisation of the Crushing flow-sheet which may reduce equipment requirements.

21.3 Tailings Disposal Recommendations

An analysis of the lake water demand is recommended for further refinement of the existing model as definition of the layout and arrangement of the tailings discharge pipeline is completed. Reasonably continuous circulating flow at high velocity will likely be required to motivate the movement of the course 10 mm size fraction of the rocks along the pipe to the lake discharge point. Downward sloping pipe layout without pockets may be helpful to the reduced occurrence of blockages during the on and off operation of this process plant.

Further monitoring studies are required to ensure water quality of Lake Ukkaata Qaava during the operations such that an accurate prediction of water quality in the lake can be made. The two types of studies recommended are a further field study and a laboratory study.

21.4 Geotechnical Recommendations

Further geotechnical information should be collected from any future geology drilling and during construction, including structural information. Existence of faults should be reviewed in the field to gain an understanding, if present, of their location, orientation, inclination, persistence and geotechnical characteristics.

Collection of rock samples for geotechnical laboratory testing. In particular, direct shear testing, point load testing and unconfined compression strength.

Refine the kinematic analysis based on future geotechnical information to minimize waste stripping.

21.5 Hydrological Recommendations

To further improve the overall understanding of the hydrology of the project site and to assist with a more detailed hydrological study, it is recommended the following:

- Continuation of discharge measurements and data-logging at the outlet of Lake Ukkaata Qaava
- Discharge measurements and data-logging at streams flowing into Lake Ukkaata Qaava.

21.6 Mining Operation Recommendations

The following items will increase the confidence of the mine plan:

- Once any further geotechnical and Mineral Resource work is complete, the mine plan may be further optimised by adjusting the pit design.
- Work to receive capital quotations and operating costs for the planned equipment fleet will allow the Company to receive a competitive price which will increase confidence and may increase the project economics.

21.7 Site Planning and Infrastructure Recommendations

Discussions with the Greenland authorities should continue in regards to the use of soft shelled structures for accommodation and operational buildings. These building may offer an economic, environmental and ergonomic advantage for the operation.

Testing of PAG/NAG characteristics of material used for infrastructure construction.

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23 DATE AND SIGNATURE PAGE

Authors Name	Signature	Date
Lara Reggin, P.Geo.	"*Lara Reggin*"	6th June 2011
John Chow, MAusIMM, MIEAust	"*John Chow*"	6th June 2011

24 STATEMENT OF QUALIFICATION

John Chow, MAusIMM, MIEAust

I, John Chow, MAusIMM, MIEAust, as an author of this report entitled "NI 43-101 Technical Report Pre-Feasibility Study On The Aappaluttoq Ruby Project, Greenland", (the "Technical Report") prepared for True North Gems and dated 6th June 2011 do hereby certify that:

I am a Senior Mining Engineer with EBA, A Tetra Tech Company. My office address is Suite 900, 1066 W. Hastings Street, Vancouver, British Columbia, Canada V6E 3X2.

I am a graduate of the Western Australian School of Mines, in 2002, with a Bachelor's degree in Mining Engineering and a Bachelor's degree in Commerce.

I am registered as a Member of the Australian Institute of Mining and Metallurgy (206194) and Engineers Australia (3642893). I have worked as a mining engineer for 8 years since my graduation. My relevant experience for the purpose of the Technical Report is:

- Review and report as a consultant on a number of mine development projects for operations and reporting requirements, including:
  - Yellowknife Gold Project Pre-feasibility report author.
  - Cadia East Open Pit Scoping Study, Pre-feasibility and Feasibility report author.
  - Reserve Update for the Santa Elena Open Pit report author and Qualified Person.
  - Preliminary Assessment for the Santa Elena and Cruz de Mayo Expansion Project report author and Qualified Person.
- Processing and crush experience from:
  - Processing operator at Jundee Mill and Nimary Mill, Newmont Australia.
  - In-pit crush convey trade off study, Newcrest Australia.
  - Organization of aggregate crushing for concrete and road construction at Jundee, Newmont Australia and Lumwana, Lumwana, Lumwana Mining Company.
- Long term planning work at operations including:
  - Resource and Reserve calculations for Cadia Valley Operations, Newcrest; Lumwana, Lumwana Mining Company.
  - Pit optimizations, budgets and open pit mine designs for Cadia Valley Operations, Newcrest; Lumwana, Lumwana Mining Company; Jundee, Newmont.

I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

I have visited the Aappaluttoq property between the 2nd and 4th of November 2010, and 2 additional days in Nuuk which will serve as the site for post-processing work.

I am responsible for sections 16, 17.9 and 19 and related portions of section 1, 20 and 21.

I am independent of True North Gems applying the test set out in Section 1.4 of National Instrument 43-101.

I have had no prior involvement with the property that is the subject of the Technical Report.

I have read National Instrument 43-101, and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

To the best of my knowledge, information, and belief, as of the date of this certificate the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated 6th June 2011

John Chow, MAusIMM, IEAust

Lara Reggin, P. Geo

I, Lara Reggin., P. Geo., as an author of this report entitled "NI 43-101 Technical Report Pre-Feasibility Study On The Aappaluttoq Ruby Project, Greenland", (the "Technical Report") prepared for True North Gems and dated 6th June 2011 do hereby certify that:

I am a Senior Project Geologist and Project Director for EBA Engineering Consultants Ltd. My office address is 9th Floor, 1066 West Hastings Street Vancouver, B.C.

I am a graduate of the University of British Columbia in 1995 with a Bachelor of Science degree in Geological Sciences.

I am registered as a Certified Professional Geologist registered with the Association of Professional Engineers and Geoscientists of the Province of British Columbia (Reg.# 28236). I have worked as a geologist for a total of 15 years since my graduation. My relevant experience for the purpose of the Technical Report is:

- Review and report as a geologist and consultant on numerous exploration and mining projects within Canada for due diligence, operations and regulatory requirements, including:
  - Geotechnical, Preliminary Assessment and Prefeasibility reports for the Yellowknife Gold Project, NWT from January 2005 to 2009.
  - Technical Report on the Courageous Lake Deposit, NWT.
  - Technical Report on the Border Coal Deposit, Saskachewan.
  - Geotechnical Preliminary Assessment of the DO27 Project, NWT
  - Technical Report on the Bonanza Ledge Gold Deposit, Wells, B.C.
- Mine geologist with duties including reserves and grade control at operational mine sites including:
  - Echo Bay Mines Lupin Gold Mine, Nunavut, Canada;
  - Echo Bay Mines, Ulu advanced exploration project (gold), Northwest Territories;
  - Battle Mountain Gold's Golden Giant Gold Mine, Ontario, Canada.
- Exploration and consulting geologist with duties including drilling, deposit definition and bulk sample collection at exploration sites including:
  - BHP Billiton, Windy Lake and High Lake properties, NT, Nunavut, Canada1995;
  - Jericho Diamond project, bulk sample of kimberlite pipe, Nunavut, Canada, 1999;

    Echo Bay Mines, Finger Lake and Sky properties, NT, Canada, 1994.

I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

I have visited the Aappaluttoq property between the 2nd and 4th of November 2010, and 2 additional days in Nuuk which will serve as the site for post-processing work.

I am responsible for sections 2 through Section 17 excluding 17.9, section 18 and related portions of section 1, 20 and 21.

I am independent of True North Gems applying the test set out in Section 1.4 of National Instrument 43-101.

I have had no prior involvement with the property that is the subject of the Technical Report.

I have read National Instrument 43-101F1, and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

To the best of my knowledge, information, and belief, as of the date of the report, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated the 6th June 2011

L. Reggin, P. Geo.