Preliminary Economic Assessment

Bokan Mountain Rare Earth Element Project

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Report to:

PRELIMINARY ECONOMIC ASSESSMENT ON THE BOKAN MOUNTAIN RARE EARTH ELEMENT PROJECT, NEAR KETCHIKAN, ALASKA

EFFECTIVE DATE: JANUARY 10, 2013

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AT THE UCORE RARE METALS INC. OFFICES IN VANCOUVER

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GLOSSARY

UNITS OF MEASURE

above mean sea level	amsl
acre	ac
ampere	A
annum (year)	a
billion	B
billion tonnes	Bt
billion years ago	Ga
British thermal unit	BTU
centimetre	cm
cubic centimetre	cm^3
cubic feet per minute	cfm
cubic feet per second	ft^3/s
cubic foot	ft^3
cubic inch	in^3
cubic metre	m^3
cubic yard	yd^3
counts per second	cps
day	d
days per week	d/wk
days per year (annum)	d/a
dead weight tonnes	DWT
degree	°
degrees Celsius	°C

diameter	ø
dollar (American)	US$
dollar (Canadian)	Cdn$
feet above sea level	fasl
foot	ft
gallon	gal
gallons per minute (US)	gpm
gigajoule	GJ
gigapascal	GPa
gigawatt	GW
gram	g
grams per litre	g/L
grams per tonne	g/t
greater than	>
hectare (10,000 m^2)	ha
hertz	Hz
horsepower	hp
hour	h
hours per day	h/d
hours per week	h/wk
hours per year	h/a
inch	"
kilo (thousand)	k
kilogram	kg
kilograms per cubic metre	kg/m^3
kilograms per hour	kg/h
kilograms per square metre	kg/m^2
kilometre	km
kilometres per hour	km/h
kilopascal	kPa
kilotonne	kt
kilovolt	kV
kilovolt-ampere	kVA
kilovolts	kV
kilowatt	kW
kilowatt hour	kWh
kilowatt hours per tonne (metric ton)	kWh/t
kilowatt hours per year	kWh/a
less than	<
litre	L
litres per minute	L/m
megabytes per second	Mb/s
megapascal	MPa
megavolt-ampere	MVA
megawatt	MW
metre	m

metres above sea level	masl
metres Baltic sea level	mbsl
metres per minute	m/min
metres per second	m/s
metric ton (tonne)	t
microns	µm
milligram	mg
milligrams per litre	mg/L
millilitre	mL
millimetre	mm
million	M
million bank cubic metres	Mbm^3
million bank cubic metres per annum	Mbm^3/a
million tonnes	Mt
minute (plane angle)	'
minute (time)	min
month	mo
ounce	oz
pascal	Pa
centipoise	mPa·s
parts per million	ppm
parts per billion	ppb
percent	%
pound(s)	lb
pounds per square inch	psi
revolutions per minute	rpm
second (plane angle)	"
second (time)	s
specific gravity	SG
square centimetre	cm^2
square foot	ft^2
square inch	in^2
square kilometre	km^2
square metre	m^2
thousand tonnes	kt
three dimensional	3D
three dimensional model	3DM
tonne (1,000 kg)	t
tonnes per day	t/d
tonnes per hour	t/h
tonnes per year	t/a
tonnes seconds per hour metre cubed	ts/hm^3
volt	V
week	wk
weight/weight	w/w

year (annum) .... a

ELEMENTS AND CHEMICAL SYMBOLS

aluminium	Al
calcium	Ca
cerium oxide	Ce_2O_3
cerium	Ce
dysprosium oxide	Dy_2O_3
dysprosium	Dy
erbium oxide	Er_2O_3
erbium	Er
europium oxide	Eu_2O_3
europium	Eu
fluorine	F
gadolinium oxide	Gd_2O_3
gadolinium	Gd
hafnium	Hf
holmium oxide	Ho_2O_3
holmium	Ho
hydrochloric acid	HCl
hydrogen cyanide	HCl
hydrogen fluoride	HF
iron	Fe
lanthanum oxide	La_2O_3
lanthanum	La
lutetium oxide	Lu_2O_3
lutetium	Lu
magnesium	Mg
neodymium oxide	Nd_2O_3
neodymium	Nd
niobium	Nb
nitric acid	HNO_3
phosphorus	P
Phosphorus pentoxide	P_2O_5
potassium	K
praseodymium oxide	Pr_2O_3
praseodymium	Pr
samarium oxide	Sm_2O_3
samarium	Sm
samarium, europium and gadolinium	SEG
scandium	Sc
silicon	Si
sodium	Na
sulphuric acid	H_2SO_4
tantalum	Ta

terbium oxide	Tb_2O_3
terbium	Tb
thorium	Th
thulium oxide	Tm_2O_3
thulium	Tm
titanium	Ti
uranium	U
vanadium	V
ytterbium oxide	Yb_2O_3
ytterbium	Yb
yttrium oxide	Y_2O_3
yttrium	Y
zinc	Zn
zirconium	Zr

ABBREVIATIONS AND ACRONYMS

Activation Laboratories Ltd.	Actlabs
Alaska Department of Fish and Game	ADF&G
Alternative Minimum Tax	AMT
ammonium nitrate fuel oil	ANFO
Aurora Geosciences (Alaska) Ltd	Aurora
Bokan Mountain Project	the Project
Bokan Mountain Property	the Property
Bond abrasion work index	Ai
Bond ball mill work index	Bwi
Bond crusher impact work index	Cwi
Bond rod mill work index	Rwi
CommodasUltrasort	Commodas
cumulative net cash flow	CNCF
Delayed Neutron Counting Uranium	DNC U
Dotson Main East	DME
Dotson Main West	DMW
Dual Energy X-Ray Transmission	DEXRT
Eco-tech Laboratories Ltd	Eco-tech
entangled-photon microscopy	EPM
freight on board	FOB
general and administrative	G&A
geosynthetic clay liner	GCL
global positioning system	GPS
Hazen Research, Inc.	Hazen
heavy rare earth element	HREE
heavy rare earth oxide	HREO
high field strength element	HFSE
high-density polyethylene	HDPE
induced polarization	IP

inductively coupled plasma-mass spectrometry	ICP-MS
inflow design flood	IDF
IntelliMet LLC	IntelliMet
internal rate of return	IRR
Internal Revenue Code	IRC
International Organization for Standardization	ISO
Knight Piésold Ltd.	Knight Piésold
life-of-mine	LOM
light rare earth element	LREE
light rare earth oxide	LREO
liquefied natural gas	LNG
load-haul-dump	LHD
local area network	LAN
Lyntek Incorporated	Lyntek
More Core Diamond Drilling Service, Ltd.	More Core
National Environmental Policy Act	NEPA
National Instrument 43-101	NI 43-101
net cash flow	NCF
net present value	NPV
net smelter return	NSR
ordinary kriging	OK
Precision GeoSurveys Inc.	Precision
preliminary economic assessment	PEA
PricewaterhouseCoopers LLC	PwC
Qualified Persons	QPs
quality assurance/quality control	QA/QC
rare earth element	REE
rare earth oxide	REO
rock quality determination	RQD
run-of-mine	ROM
scanning electron microscope	SEM
solid phase extraction	SPE
Standards Council of Canada	SCC
Stantec Ltd.	Stantec
tailings and mine water management facility	TMWMF
total rare earth element	TREE
total rare earth oxide	TREO
Ucore Rare Metals Inc.	Ucore
United States Forest Service	USFS
US Department of Agriculture	USDA
very high frequency	VHF
voice over internet protocol	VoIP
waste rock management facility	WRMF

1.0 SUMMARY

1.1 INTRODUCTION

The Bokan Mountain Project (the Project), located southwest of Ketchikan, Alaska, will be a 1,500 t/d underground mining operation. Mineralized material will be processed in a material sorting and leaching process plant, coupled with an advanced separation process to directly produce saleable rare earth oxide (REO) concentrates. The Project is 100% owned by Ucore Rare Metals Inc. (Ucore).

In 2011, Ucore commissioned a team of engineering consultants to complete a preliminary economic assessment (PEA) in accordance with National Instrument 43-101 (NI 43-101). The following consultants were commissioned to complete portions of the NI 43-101 technical report:

- Tetra Tech: overall project management; mineral processing and recovery methods; infrastructure; capital and operating cost estimates; and financial analysis
- Aurora Geosciences (Alaska) Ltd (Aurora): geology; mineral resource estimate; and environmental studies, permitting, and social or community impact
- Stantec Inc. (Stantec): mining, including capital and operating costs
- Lyntek Incorporated (Lyntek): mineral processing and metallurgical testing using Dual Energy X-Ray Transmission (DEXRT) mineralized material sorting and magnetic separation circuit design and costs
- IntelliMet LLC (IntelliMet): mineral processing and metallurgical testing with leaching and solid phase extraction (SPE), uranium and thorium removal, leaching and rare earth element (REE) separation design and costs
- Knight Piésold Ltd. (Knight Piésold): tailings and waste rock management, including capital costs.

The metallurgical test work was completed by Hazen Research, Inc. (Hazen), Activation Laboratories Ltd. (Actlabs), SGS Lakefield, and CommodasUltrasort (now known as TOMRA Sorting Solutions).

For the purposes of this study, all currencies are expressed in US dollars, unless otherwise specified.

General information for the Project is summarized in Table 1.1.

Table 1.1 General Project Information

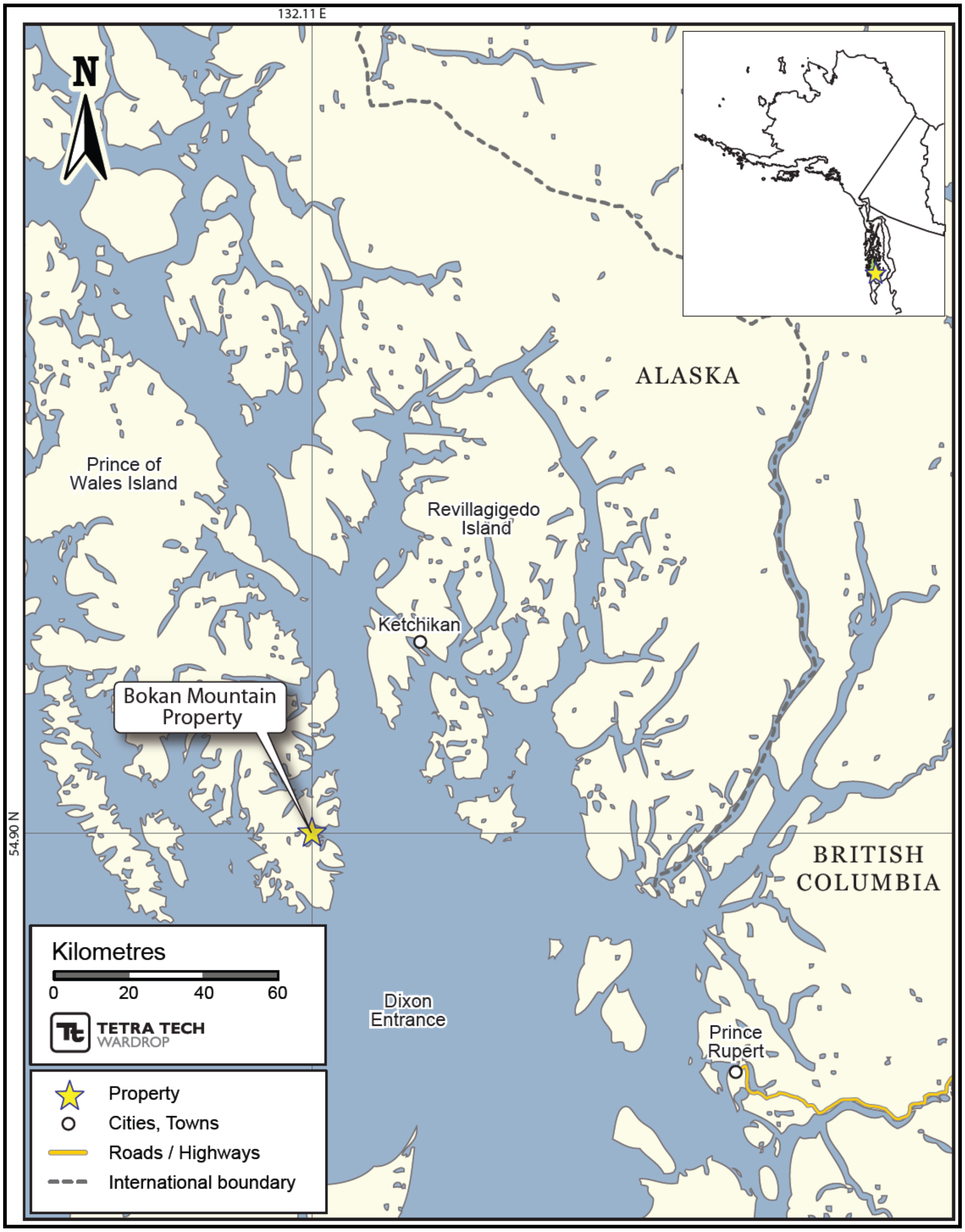
Description	Unit	Amount
Estimated Mineral Resources (Inferred)	Mt	5.228
Life-of-mine (LOM)	years	11
Mine Production Rate	t/d	1,500
Mill Processing Rate (Crushing and X-ray Sorting)	t/d	1,500
Mill Processing rate (Magnetic Separation Sorting)	t/d	750
Mill Processing Rate (Acid Leaching)	t/d	375
Project Capital Cost	US$ million	221
Average Overall Operating Cost	US$/t mined	122.78
Pre-tax Net Present Value (NPV) at 10% Discount Rate	US$ million	577
Pre-tax Payback Period	Years	2.3
Pre-tax Internal Rate of Return (IRR)	%	43

1.2 PROPERTY DESCRIPTION

The Bokan Mountain Property (the Property) is located approximately 60 km southwest of the business and commercial centre of Ketchikan, Alaska (Figure 1.1). The approximate geographic centre of the Property lies at 54° 55’ north latitude and 132° 08’ west longitude. The Property consists of 512 unpatented and un-surveyed lode mining claims covering 38.12 km^2 in T80S, R88E of the Copper River Meridian of the Ketchikan recording district, in southeastern Alaska. There are also four 160 acre MTRSC claims located in sections 29, 30 and 31, T80S, R89E, which are wholly owned by Ucore through its Alaskan subsidiaries.

The Dotson Zone, which is the subject of this study, is located on the southeast flank of Bokan Mountain, between the shore of Kendrick Bay and Bokan Mountain proper. Exploration work on the Dotson Zone has confirmed the presence of continuous REE mineralization over a defined strike length of 2,140 m with an average width of 50 m. The west end of the Dotson Zone trend is 300 masl whilst the east end trends almost into Kendrick Bay at sea level. The Dotson Zone, while probably genetically related to the Ross Adams uranium mine and its associated showings is differentiated both spatially and mineralogically from the uranium deposits. The Dotson Zone is more than 500 m from the Ross Adams mine at its closest point, and contains only minor amounts of uranium and thorium, while being enriched in REEs, as well as in niobium, yttrium, zirconium, hafnium, and tantalum.

Figure 1.1 Property Location Map



1.3 HISTORY

The Dotson Zone has been the subject of limited, small scale prospecting, trenching and sampling over the years by local prospectors and claim holders. The largest amount of field work and associated research was carried out by personnel of the US Bureau of Mines in the mid-to-late 1980s (Warner and Barker 1989). Warner and Barker describes the Dotson Zone as “comprising a system of west-northwest-striking, steeply dipping dikes that can be traced between test pits and outcrops for approximately 2,100 m from tidewater on Kendrick Bay north-westward to the 335 m elevation of Bokan Mountain” (Warner and Barker 1989).

1.4 GEOLOGICAL SETTING

Bokan Mountain lies within the Alexander Terrane of the Canadian-Alaskan Cordillera. The Cordillera is generally considered to be a collage of allochthonous oceanic and pericratonic terranes that were accreted to the western margin of the North American craton during the Mesozoic (Coney et al. 1980; Colpron et al. 2007). The Alexander Terrane is a composite terrane which does not show any evidence of an early relationship with the western margin but instead, geological data suggest that it originates far from western North America (Laurentia) and was only transported into the Cordilleran realm by later tectonic processes (Bradley et al. 2003; Nokleberg et al. 2000). The Alexander Terrane underlies most of southeastern Alaska and parts of western BC and southwestern Yukon. It is an arc terrane composed of late Proterozoic to Triassic mafic to felsic volcanic rocks, terrigenous clastic and carbonate rocks and early Paleozoic and rare Mesozoic granitic rocks. The pre-Devonian rocks were formed in a subduction-related environment. The terrane was amalgamated with another exotic terrane—Wrangellia—some 310 Ma (Gardner et al. 1988) forming the Insular Superterrane which was subsequently accreted onto the western margin of the North American craton by the Cretaceous (approximately 115 to 95 Ma).

Bokan Mountain represents an unusual style of REEs, uranium, niobium, and zirconium mineralization hosted within, or proximal to, a circular Jurassic A-type peralkaline intrusive complex known as the Bokan Mountain Granite. The high-level intrusion contains several concentrically zoned intrusive phases of sodic granites, porphyries and pegmatites. Aegerine and riebeckite are the dominant ferromagnesian minerals.

Intense wall-rock replacement/alteration by albite, chlorite, calcite, fluorite, and hematite accompanied the mineralized material -forming processes and mineralization. Mineralization as veinlets, disseminations or as irregular sublinear masses is hosted within, or closely proximal to, the Bokan Mountain Granite and was derived from magmatic mineralized material fluids. There are several types of uranium-thorium-REE mineralization, the most important of which are:

- irregular structurally-controlled pipes

- shear zone-related pods or lenses (veins)
- pegmatitic or felsic dikes.

The type three felsic dykes or "vein dykes" are the predominant mineralization type in the Dotson Zone, which is the current exploration target on the Property and the subject of this report. These vein dykes form a zone of potentially economic mineralization approximately 50 m wide and 2,100 m long, striking west-northwest and dipping steeply to the north. They range in thickness from a few millimetres to tens of centimetres and contain niobium, tantalum, beryllium, zirconium and REEs (both light and heavy REEs). Niobium and tantalum are present chiefly in euxenite and columbite while REEs are contained in thalenite, bastnaesite, xenotime and monazite.

1.5 Mineral Resources

A multi-season diamond drilling program was carried out on the Property between 2007 and 2010. A total of 5,774 m of drilling was completed in 37 holes on the Dotson structure. During 2009 and 2010, 55 channel samples were cut across veins outcropping in the Dotson Zone. The results of this drilling and sampling were compiled and interpreted, a deposit model was constructed and a mineral resource estimate was calculated. Mineral Resources for the Project were classified in accordance with NI 43-101 requirements. Table 1.2 summarizes the estimated Inferred Mineral Resources published in April 2011.

It should be noted that Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

Table 1.2 Resources at 0.4% TREO Cut-off

Type	Tonnes	LREE (ppm)	HREE (ppm)	TREE (ppm)	LREO (%)	HREO (%)	TREO (%)
Inferred	5,228,200	3,368.77	2,113.50	5,482.25	0.394	0.259	0.653

Note: LREE = light rare earth element; HREE = heavy rare earth element; TREE = total rare earth element; LREO = light rare earth oxide; HREO = heavy rare earth oxide; TREO = total rare earth oxide

1.6 Mineral Processing and Metallurgical Testing

1.6.1 Mineral Processing

The Project will be a 1,500 t/d underground operation with mineralized material processed using x-ray sorting and magnetic separation sorting, acid leaching, solid phase extraction for REE oxide separation.

During full mine production (Years 3-10):

- The process rate of the x-ray sorting circuit is estimated to be 1,500 t/d (540,000 t/a).
- The process rate of the magnetic separation sorting circuit is estimated to be 750 t/d (270,000 t/a).
- The process rate of acid leaching and solid phase extraction is estimated to be 375 t/d (135,000 t/a) with a peak production rate of 9.2 t/d (3,312 t/a).
- The tailing will be sent to the paste plant on site for underground backfill. Constituents such as thorium, uranium, and iron removed from the REE concentrate will be sent to the underground backfill.

1.6.2 METALLURGICAL TESTING

Table 1.3 shows the metallurgical recoveries and concentrate grades for the Project.

Table 1.3 Summary of Metallurgical Test Work Results

	TREE + Y Recovery (%)	Source
X-ray Sorting	94.6	Commodas/Lyntek
Magnetic Separation	95.0	Hazen/Lyntek
HNO_3 Leaching	92.7	IntelliMet
REE Separation	>99.9	IntelliMet

1.7 MINING

The Project will be a 1,500 t/d underground mine operation using a longitudinal blasthole stoping method with paste backfill at an average of 500 t/d per stope using 5.6 m^3 load-haul-dump (LHD) vehicles with remote operating capabilities.

A cut-off grade of 0.4% TREO was used for planning mine production. The annual mine production of mineralized material and waste peak is 540,000 t/a, with an 11-year LOM. A summary of mining details is shown in Table 1.4.

Table 1.4 Summary of Mining Details

Description	Unit	Amount
Mine Life	years	11
Mine Production Rate	t/d	1,500
TREO Produced	t	24,327
Pre-production Capital Cost	US$ million	55.4
Sustaining Capital Cost	US$ million	95.7
Mining Operating Cost	US$/t mined	43.54

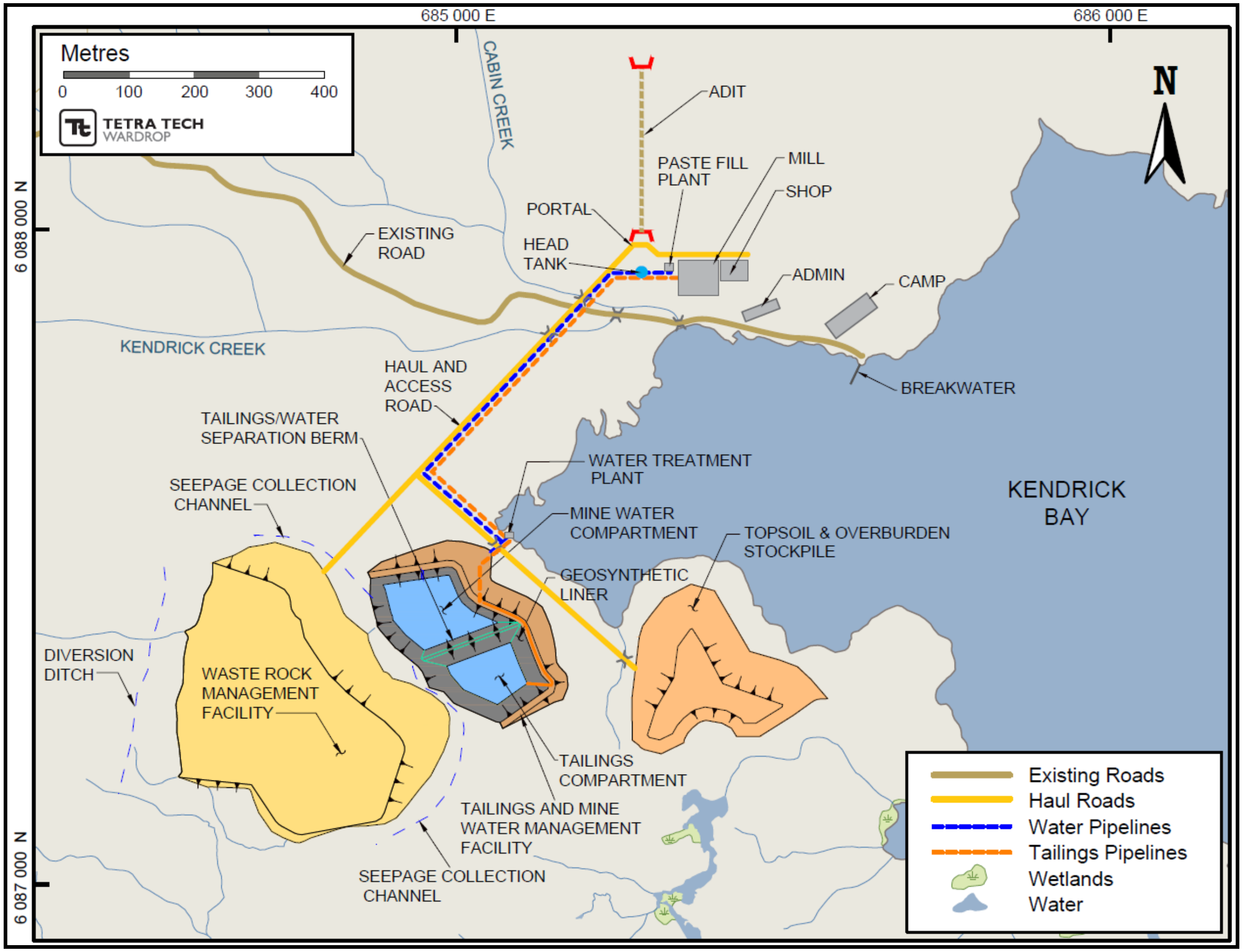
1.8 PROJECT INFRASTRUCTURE

The proposed on-site and off-site infrastructure will include:

- process plant
- permanent camp with administration offices
- emergency vehicle building with vehicle maintenance shop
- liquefied natural gas (LNG) power generators
- standby diesel power generation unit
- main substation and power distribution
- paste backfill plant
- tailings and mine water management facility (TMWMF)
- waste rock management facility (WRMF)
- water treatment facility
- access and on-site roads
- potable and fire water storage and distribution
- sewage treatment facility
- laydown and container storage area.

The general site layout of the Project is provided in Figure 1.2.

Figure 1.2 General Site Layout



1.8.1 WASTE AND WATER MANAGEMENT

The mine plan incorporates the recovery of 5.2 Mt of mineralized material mined over 11 years, in addition to 1.4 Mt of development waste rock. The mineralized material will be transported to a crusher and sorting facility, which will remove 750 t/d of barren rock. The barren and development waste rock will be hauled by truck to the WRMF. The remaining 750 t/d of mineralized material will be processed in the mill, resulting in approximately 10 t/d of concentrate and 740 t/d of tailings.

The milled tailings will be stored on surface in a lined TMWMF during the first year of operations. The tailings in the TMWMF will be re-slurried and used as paste backfill after Year 1, when sufficient space is available in the underground stopes. The tailings produced after Year 1 will be fed directly to the paste plant and placed underground as paste backfill. Backfill is supplemented with waste rock when insufficient tailings are available.

Mine water from the underground workings that are not required for the mill and paste plant will be transported through pipelines to the TMWMF. Any runoff from the WRMF and precipitation on the TMWMF is also collected in the TMWMF. Excess water will be treated, if required, prior to release into Kendrick Bay.

1.9 ENVIRONMENTAL

An environmental review for the Project has been completed. Baseline studies have been initiated to support a pending environmental assessment, which will be conducted by the United States Forest Service (USFS). A complete list of significant environment permits required to operate the Project is provided and discussed in Section 20.0. The development and approval of a comprehensive reclamation plan will also be required before key construction permits are issued. Community consultation has been initiated and will continue throughout the permitting and operating period.

1.10 CAPITAL AND OPERATING COSTS

1.10.1 CAPITAL COST ESTIMATE

An initial capital cost of US$221 million has been estimated for the Project (Table 1.5). This estimate is prepared with a base date of Q3 2012 and does not include any escalation past this date. The quotations used in this estimate were obtained in Q1 to Q3 2012, and are budgetary and non-binding.

All currencies in this section are expressed in US dollars using an exchange rate of US$1.00:Cdn$1.00. The accuracy range of the capital cost estimate is ±35%.

Table 1.5 Capital Cost Summary

Item	Total Cost ($ million)
Direct Capital Costs	
Site Development	6.1
Mine Underground	18.9
Mine Surface Facilities	23.8
Process	62.9
Tailings and Waste Rock Management	10.1
Utilities	3.4
Buildings	3.0
Temporary Facilities	5.2
Plant Mobile Equipment and Miscellaneous	1.4
Subtotal	**134.7**
Indirect Capital Costs	
Indirect Construction Costs	51.1
Owner's Costs	10.9
Contingency	24.5
Subtotal	**86.5**
Total Capital Cost	**221.3**

1.10.2 Operating Cost Estimate

On-site operating costs are estimated to be US$122.78/t mined including mining, processing, general and administrative (G&A), water and waste management and plant services. Table 1.5 summarizes the unit costs, which are based on an annual production rate of 540,000 t.

Table 1.6 Operating Cost Summary

Item	Average Unit Cost (US$/t mined)
Mining	41.69
Processing	54.83
G&A	13.56
Power	11.78
Miscellaneous	0.93
Total Operating Cost	**122.78**

1.11 ECONOMIC ANALYSIS

Tetra Tech conducted an economic evaluation of the Project, incorporating all the relevant capital, operating, working, sustaining costs, and royalties. The evaluation was based on a pre-tax financial model and was calculated in US dollars. For the 11-year mine life and 5,175,889 LOM tonnes mined, the following pre-tax financial parameters were calculated using the base case prices:

- 43% IRR
- 2.3-year payback on US$221 million capital
- US$577 million NPV at a 10% discount value.

Ucore commissioned PricewaterhouseCoopers LLC (PwC) in Vancouver, BC to prepare a tax model for the post-tax economic evaluation of the Project, with the inclusion of applicable federal and state taxes. The post-tax analysis is discussed in Section 22.0. The following post-tax financial parameters were calculated:

- 35% IRR
- 2.5-year payback on US$221 million capital
- US$368 million NPV at a 10% discount rate.

The base case (outlined in Table 1.7) uses the three-year trailing average prices from October 2009 to October 2012.

Table 1.7 Base Case Prices

REO	Base Case Prices (US$/kg)
La_2O_3	48.69
Ce_2O_3	47.21
Pr_2O_3	113.10
Nd_2O_3	126.70
Sm_2O_3	57.74
Eu_2O_3	1,834.94
Gd_2O_3	81.70
Tb_2O_3	1,520.83
Dy_2O_3	845.80
Ho_2O_3	211.39
Er_2O_3	88.20
Tm_2O_3	N/A
Yb_2O_3	102.79
Lu_2O_3	1,036.40
Y_2O_3	80.41

Sensitivity analyses were developed to evaluate the Project economics.

1.12 PROJECT DEVELOPMENT PLAN

The Project will require approximately two years of construction activities to complete. A high-level project development plan and schedule is provided in Section 24.0.

1.13 RECOMMENDATIONS

Based on the work carried out in this PEA and the resultant economic evaluation, this study should be followed by a feasibility study in order to further assess the economic viability of the Project.

Detailed recommendations are provided in Section 26.0 of this report.

2.0 INTRODUCTION

Ucore commissioned a team of engineering consultants to complete this PEA, in accordance with NI 43-101 Standards of Disclosure for Mineral Projects.

The following consultants completed components of this PEA, as listed below:

- Tetra Tech: overall project management, mineral processing and recovery methods, infrastructure, capital and operating cost estimates, and financial analysis
- Aurora: geology, mineral resource estimate, and environmental studies, permitting, and social or community impact
- Stantec: mining, including capital and operating costs
- Lyntek: mineral processing and metallurgical testing using DEXRT sorting and magnetic separation, circuit design and costs
- IntelliMet: mineral processing and metallurgical testing with leaching and SPE, uranium and thorium removal, leaching and REE separation design and costs
- Knight Piésold: tailings and waste rock management, including capital costs.

Hazen, CommodasUltrasort (now TOMRA Sorting) and IntelliMet conducted metallurgical test work.

A summary of Qualified Persons (QPs) responsible for each section of this report is detailed in Table 2.1. The following QPs conducted site visits of the Property:

- Edwin H. Bentzen III, RM SME conducted a site visit on June 14, 2012.
- Les Galbraith, P.Eng. conducted a site visit on April 26, 2012.
- Richard F. Hammen, Ph.D. conducted a site visit on June 14, 2012.
- Ronald James Robinson, P.Geol. conducted site visits between June 12 and October 25, 2011.
- Srikant Annavarapu, RM SME conducted a site visit on November 29, 2011.

Table 2.1 Summary of QPs

Report Section			Company	QP
1.0	Summary		All	Sign off by Section
2.0	Introduction		Tetra Tech	Hassan Ghaffari, P.Eng.
3.0	Reliance on Other Experts		Tetra Tech	Hassan Ghaffari, P.Eng.
4.0	Property Description and Location		Aurora	Ronald James Robinson, P.Geol.
5.0	Accessibility, Climate, Local Resources, Infrastructure, and Physiography		Aurora	Ronald James Robinson, P.Geol.
6.0	History		Aurora	Ronald James Robinson, P.Geol.
7.0	Geological Setting and Mineralization		Aurora	Ronald James Robinson, P.Geol.
8.0	Deposit Types		Aurora	Ronald James Robinson, P.Geol.
9.0	Exploration		Aurora	Ronald James Robinson, P.Geol.
10.0	Drilling		Aurora	Ronald James Robinson, P.Geol.
11.0	Sample Preparation, Analyses, and Security		Aurora	Ronald James Robinson, P.Geol.
12.0	Data Verification		Aurora	Ronald James Robinson, P.Geol.
13.0	Mineral Processing and Metallurgical Testing		-	-
	13.1	Preliminary Hydrometallurgical Test Work – Phase 1	Lyntek	Edwin H. Bentzen III, RM SME
	13.2	Preliminary Hydrometallurgical Test Work – Phase 2	IntelliMet	Richard F. Hammen, Ph.D.
14.0	Mineral Resource Estimates		Aurora	Ronald James Robinson, P.Geol.
15.0	Mineral Reserve Estimates		AMEC	Srikant Annavarapu, RM SME
16.0	Mining Methods		AMEC	Srikant Annavarapu, RM SME
17.0	Recovery Methods		-	-
	17.1	Process Plant Overview	Tetra Tech	Hassan Ghaffari, P.Eng.
	17.2	Crushing and Conveying	Tetra Tech	Hassan Ghaffari, P.Eng.
	17.3	Sorting	Lyntek	Edwin H. Bentzen III, RM SME
	17.4	Tertiary Crushing and Grinding	Tetra Tech	Hassan Ghaffari, P.Eng
	17.5	Acid Leaching	IntelliMet	Richard F. Hammen, Ph.D.
	17.6	Solid Phase Extraction	IntelliMet	Richard F. Hammen, Ph.D.
	17.7	Paste Backfill	AMEC	Srikant Annavarapu, RM SME
18.0	Project Infrastructure		-	-
	18.1	Site Layout	Tetra Tech	Hassan Ghaffari, P.Eng.
	18.2	Power	Tetra Tech	Hassan Ghaffari, P.Eng.
	18.3	Waste Management	Knight Piésold	Les Galbraith, P.Eng.
	18.4	Water Management	Knight Piésold	Les Galbraith, P.Eng.
	18.5	Fresh, Fire, and Potable Water Supply, and Sewage Disposal	Tetra Tech	Hassan Ghaffari, P.Eng.
	18.6	Communication	Tetra Tech	Hassan Ghaffari, P.Eng.
19.0	Market Studies and Contracts		Tetra Tech	Hassan Ghaffari, P.Eng.
20.0	Environmental Studies, Permitting, and Social or Community Impact		Aurora	Ronald James Robinson, P.Geol.

table continues…

Report Section		Company	QP
21.0	Capital and Operating Costs	-	-
	21.1 Capital Costs	-	-
	21.1.1 Summary	Tetra Tech	Hassan Ghaffari, P.Eng.
	21.1.2 Mining	AMEC	Srikant Annavarapu, RM SME
	21.1.3 Process Plant – Mineralized Material and Hydrometallurgical	Tetra Tech	Hassan Ghaffari, P.Eng.
	21.1.4 Tailings and Waste Rock Management Facilities	Knight Piésold	Les Galbraith, P.Eng.
	21.1.5 Sustaining Capital	Tetra Tech	Hassan Ghaffari, P.Eng.
	21.2 Operating Costs	-	-
	21.2.1 Summary	Tetra Tech	Hassan Ghaffari, P.Eng.
	21.2.2 Mining	AMEC	Srikant Annavarapu, RM SME
	21.2.3 Process Plant	Tetra Tech	Hassan Ghaffari, P.Eng.
	21.2.4 G&A Operating Cost Estimate	Tetra Tech	Hassan Ghaffari, P.Eng.
22.0	Economic Analysis	Tetra Tech	Sabry Abdel Hafez, Ph.D., P.Eng.
23.0	Adjacent Properties	Aurora	Ronald James Robinson, P.Geol.
24.0	Other Relevant Data and Information	Tetra Tech	Hassan Ghaffari, P.Eng.
25.0	Interpretation and Conclusions	All	Sign off by Section
26.0	Recommendations	All	Sign off by Section
27.0	References	All	Sign off by Section

Note: AMEC = AMEC E&C Services Inc.

3.0 RELIANCE ON OTHER EXPERTS

Tetra Tech followed standard professional procedures in preparing the contents of this report. Data used in this report has been verified where possible and Tetra Tech has no reason to believe that the data was not collected in a professional manner.

Technical data provided by Ucore for use by Tetra Tech in this PEA is the result of work conducted, supervised, and/or verified by Ucore professional staff or their consultants.

Tetra Tech has not independently verified the legal status or title of the claims or exploration permits, and has not investigated the legality of any of the underlying agreement(s) that may exist concerning the Property.

The QPs who prepared this report relied on information provided by a number of experts who are not QPs.

Mr. Ronald James Robinson, P.Geol., relied upon Ms. Tina M. Sellers of Reeves Amodio LLC, for matters relating to the legal status of mineral claims in Section 4.0.

Mr. Ronald James Robinson, P.Geol, also relied on Mr. Randy MacGillivray of Ucore Rare Metals Inc., for matters relating to the environmental permitting plan in Section 20.0.

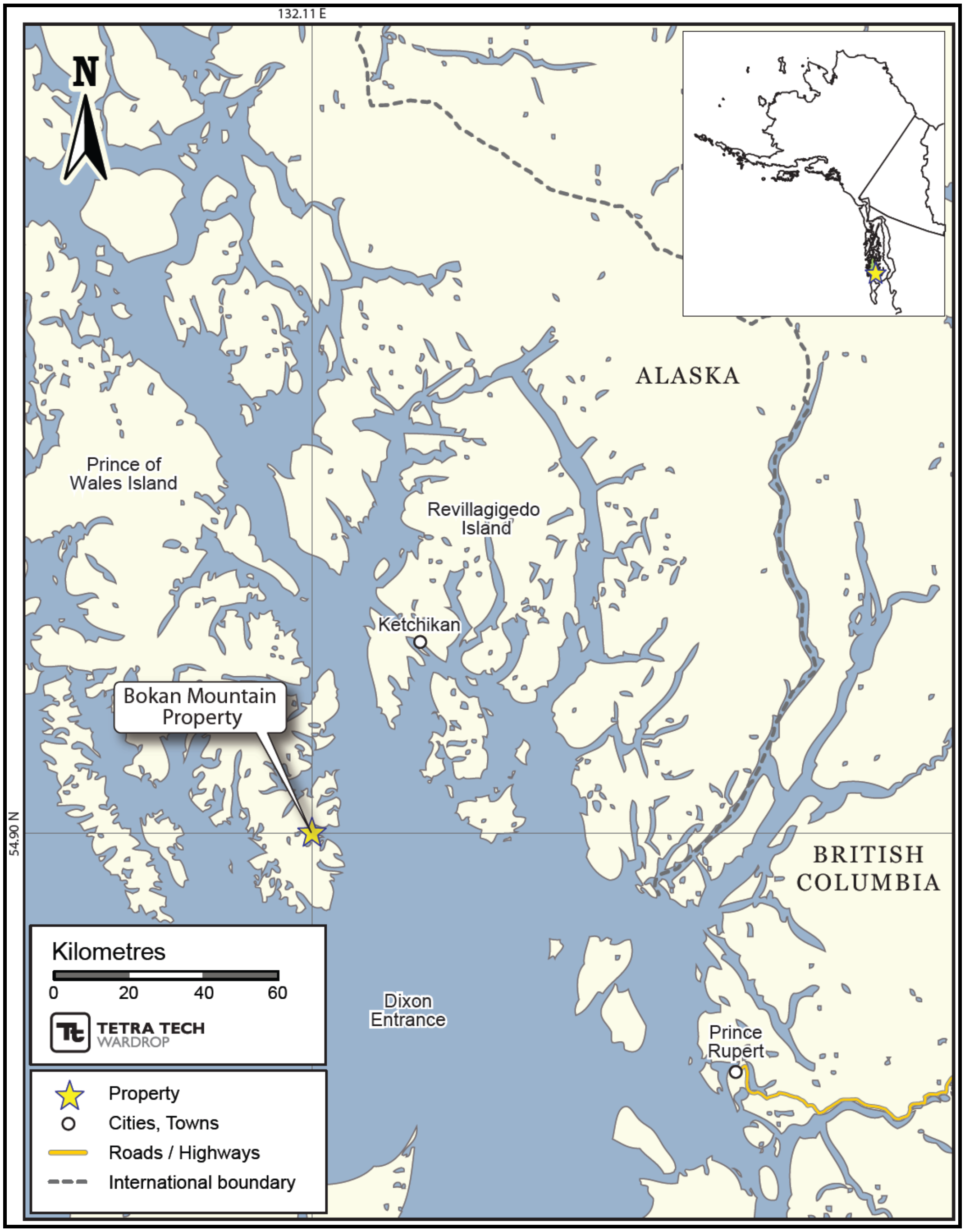
Tetra Tech relied on PwC, who are experts that are not QPs, concerning tax matters relevant to this report. The reliance is based on a letter to UCore entitled “Assistance with the tax portion of the economic analysis prepared by Tetra Tech WEI Inc. (“Tetra Tech”) in connection with the Preliminary Economic Assessment Report (the “Report) on Ucore Rare Metals Inc.’s (“Ucore”) mining project (“Project”)” and dated January 8, 2013. Tetra Tech has relied entirely on this letter for disclosure contained in Section 22.4. Tetra Tech believes that it is reasonable to rely on PwC, based on the assumption that PwC staff have the necessary education, professional designations and relevant experience in tax matters relevant to this study.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION, SIZE, AND TENURE

The Property is located approximately 60 km (37.3 miles) southwest of the business and commercial centre of Ketchikan, Alaska. The approximate geographic centre of the Property lies at 54° 55’ north latitude and 132° 08’ west longitude (Figure 4.1). The Property consists of 512 unpatented and un-surveyed lode mining claims covering 38.12 km^2 (14.72 square miles) in T80S, R88E of the Copper River Meridian of the Ketchikan recording district, in southeastern Alaska. There are also four 160 acre (0.25 square miles) MTRSC claims located in sections 29, 30 and 31, T80S, R89E, which are wholly-owned by Ucore through its Alaskan subsidiaries.

Figure 4.1 Property Location Map



The claims are owned 100% by Ucore, through wholly owned subsidiaries Bokan LLC, Landmark Alaska Limited Partnership, Landmark Alaska LLC, and Landmark Minerals US, Inc. The joint venture agreement between Ucore and the various Landmark entities was superseded by Ucore's purchase of 100% of the outstanding shares of the Landmark companies in 2007. The remaining claims are held by Ucore under the terms of six separate option agreements between Ucore (and its respective subsidiaries) and the underlying owners.

The six agreements are summarized in the following paragraphs, where the option agreements between Ucore, or its US subsidiaries and partners, and other claim holders in the Property area are outlined. Agreements 1 to 6 are illustrated in Figure 4.2. Table 4.1 lists the claims in the Property area.

- Agreements 1, 2 and 3: Three formal option agreements, the first dated February 21, 2007, and amended February 27, 2007, and the other two dated December 7, 2008, between Robert Dotson, Irene Dotson, Gayle Dotson, Derek Dotson and Landmark Alaska Limited Partnership, gives Landmark Alaska Limited Partnership the exclusive right to acquire a 100% interest in 31 lode claims by paying to the vendors $220,000 (all of which has been paid) by July 21, 2007, and an additional $100,000 (which has been paid) upon transfer of the claims to Landmark Alaska Limited Partnership. The vendors retain a 2% net smelter return (NSR) royalty on their claims, which may be purchased for $1 million.
- Agreement 4: A letter agreement dated December 4, 2006, between Mary Anderson, Raymond Anderson, David Anderson, Susan Dotson and Landmark Minerals Inc., gives Landmark Minerals Inc. the exclusive right to acquire a 100% interest in 39 lode claims by paying to the vendors $200,000 (all of which has been paid) within 12 months of signing a formal option agreement. The vendors retain a 2% NSR royalty on their claims, half of which may be purchased for $500,000.
- Agreement 5: A formal option agreement dated April 25, 2008, between Troy Erwin, Anne Erwin, and Landmark Minerals Inc., gives Landmark Minerals Inc. the exclusive right to acquire a 100% interest in 8 lode claims by paying to the vendors $100,000 (all of which has been paid) within 12 months of signing a formal option agreement. The vendors retain a 2% NSR royalty on their claims, half of which may be purchased for $500,000.
- Agreement 6: A formal option agreement dated April 25, 2008, and amended November 2008 between FFI Limited and Landmark Minerals Inc., gives Landmark Minerals Inc. the exclusive right to acquire a 100% interest in the 22 Keg lode claims by paying to the vendors $250,000. A final payment in the amount of $150,000, to complete Landmark Alaska Limited Partnership's obligations under the option agreement, was made on April 7, 2011. This brings the total payments to $250,000, as contemplated by the original agreement. The vendors retain a 4% NSR royalty on their claims, 25% (i.e., the royalty can be reduced from 4 to 3%) of which may be purchased for $1 million.

Mining deeds have been executed further to Agreements 1, 2, 3, 5, and 6. Agreement 4 has been split and the mining deed has been executed with respect to the “S Dotson” portion of the agreement.

Readers are cautioned that reference to all the agreements is required for full disclosure, and these summaries should not be relied on. None of the royalty agreements contain an “Area of Influence” clause; therefore the royalties are only payable on production from the vendor’s claims and not surrounding claims. The Property is located entirely within the Tongass National Forest and administered by the US Department of Agriculture (USDA) Forest Service. Complex policies governing access and exploration can be expected, and permits must be obtained from appropriate authorities prior to conducting exploration work involving ground disturbances.

Aurora has no expertise regarding the legal status of the mineral claims; the QP has relied on an opinion letter supplied by Reeves Amodio LLC.

Figure 4.2 Claims Map

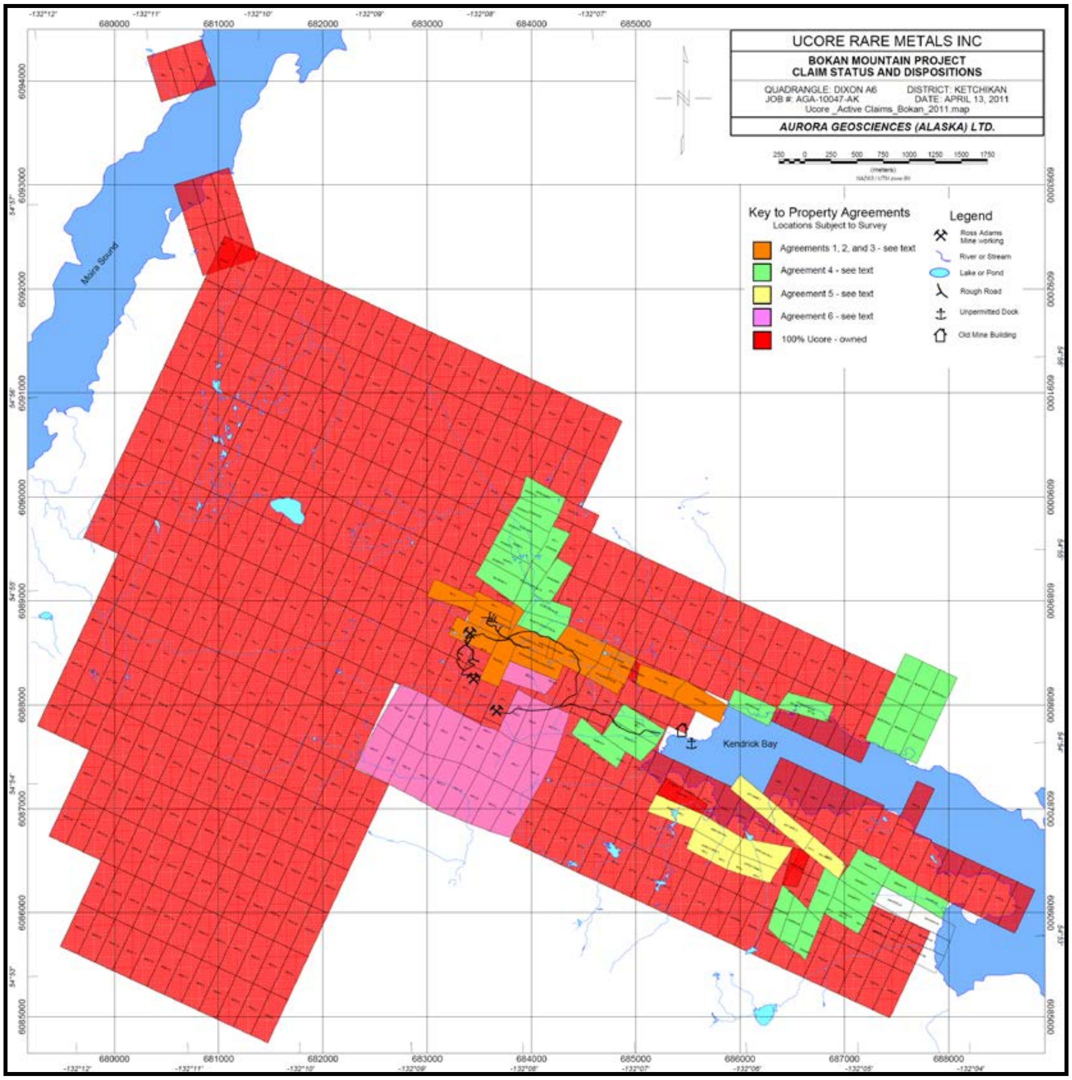


Table 4.1 Claim Name and Ownership Status

BLM Serial No.	Claim Name	BLM Serial No.	Claim Name
Administrative Owner of Record: Bokan LLC			
AA 87921	NEKO 2	AA 89897	NEKO 32
AA 87922	NEKO 4	AA 89898	NEKO 33
AA 87923	NEKO 6	AA 89899	NEKO 34
AA 87924	NEKO 8	AA 89900	NEKO 35
AA 87925	NEKO 10	AA 89901	NEKO 36
AA 87926	NEKO 12	AA 89902	NEKO 37
AA 87927	NEKO 14	AA 89903	NEKO 38

table continues…

BLM Serial No.	Claim Name	BLM Serial No.	Claim Name
AA 87928	NEKO 16	AA 89904	NEKO 39
AA 87929	NEKO 18	AA 89905	NEKO 40
AA 87930	NEKO 20	AA 89906	NEKO 74
AA 87931	NEKO 42	AA 89907	NEKO 76
AA 87932	NEKO 44	AA 89908	NEKO 78
AA 87933	NEKO 46	AA 89909	NEKO 80
AA 87934	NEKO 48	AA 89910	NEKO 82
AA 87935	NEKO 50	AA 89911	NEKO 84
AA 87936	NEKO 52	AA 89912	NEKO 86
AA 87937	NEKO 54	AA 89913	NEKO 88
AA 87938	NEKO 56	AA 89914	NEKO 90
AA 87939	NEKO 58	AA 89915	NEKO 92
AA 87940	NEKO 60	AA 89916	NEKO 93
AA 87941	NEKO 59	AA 89917	NEKO 94
AA 87942	NEKO 57	AA 89918	NEKO 95
AA 87943	NEKO 55	AA 89919	NEKO 96
AA 87945	NEKO 51	AA 89921	NEKO 98
AA 87946	NEKO 49	AA 89922	NEKO 99
AA 87947	NEKO 47	AA 89923	NEKO 100
AA 87948	NEKO 45	AA 89924	NEKO 101
AA 87949	NEKO 43	AA 89925	NEKO 102
AA 87950	NEKO 41	AA 89926	NEKO 103
AA 87951	NEKO 91	AA 89927	NEKO 104
AA 87952	NEKO 89	AA 89928	NEKO 105
AA 87953	NEKO 87	AA 89929	NEKO 106
AA 87954	NEKO 85	AA 89930	NEKO 107
AA 87955	NEKO 83	AA 89931	NEKO 108
AA 87956	NEKO 81	AA 89932	NEKO 109
AA 87957	NEKO 79	AA 89933	NEKO 110
AA 87958	NEKO 77	AA 89934	NEKO 111
AA 87959	NEKO 75	AA 89935	NEKO 112
AA 87960	NEKO 73	AA 89936	NEKO 113
AA 89849	Fred 64	AA 89937	NEKO 114
AA 89850	Fred 65	AA 89938	BUS 1
AA 89851	Fred 66	AA 89939	BUS 2
AA 89852	Fred 67	AA 89940	BUS 3
AA 89853	Fred 68	AA 89941	BUS 4
AA 89854	Fred 69	AA 89942	BUS 5
AA 89855	Fred 70	AA 89943	BUS 6
AA 89856	Fred 71	AA 89944	BUS 7
AA 89857	SON 1	AA 89945	BUS 8

table continues…

BLM Serial No.	Claim Name	BLM Serial No.	Claim Name
AA 89858	SON 2	AA 89946	BUS 9
AA 89859	SON 3	AA 89947	BUS 10
AA 89860	SON 4	AA 89948	BUS 11
AA 89861	SON 5	AA 89949	BUS 12
AA 89862	SON 6	AA 89950	BUS 13
AA 89863	SON 7	AA 89951	BUS 14
AA 89864	SON 8	AA 89952	BUS 15
AA 89865	SON 9	AA 89953	BUS 16
AA 89866	SON 10	AA 89954	BUS 17
AA 89867	SON 11	AA 89955	BUS 18
AA 89868	SON 12	AA 89956	BUS 19
AA 89869	SON 13	AA 89957	BUS 20
AA 89870	SON 14	AA 89958	BUS 21
AA 89871	SON 15	AA 89959	BUS 22
AA 89872	SON 16	AA 89960	BUS 23
AA 89873	SON 17	AA 89961	BUS 25
AA 89874	SON 18	AA 89962	BUS 26
AA 89875	SON 19	AA 89963	BUS 27
AA 89876	SON 20	AA 89964	BUS 28
AA 89877	SON 21	AA 89965	BUS 29
AA 89878	SON 22	AA 89966	BUS 30
AA 89879	SON 23	AA 89967	BUS 31
AA 89880	SON 24	AA 89968	BUS 32
AA 89881	SON 25	AA 89969	BUS 33
AA 89882	NEKO 1	AA 89970	BUS 34
AA 89883	NEKO 3	AA 89971	BUS 35
AA 89884	NEKO 5	AA 89972	BUS 36
AA 89885	NEKO 7	AA 89973	BUS 37
AA 89886	NEKO 9	AA 89974	BUS 38
AA 89887	NEKO 11	AA 89975	BUS 39
AA 89888	NEKO 13	AA 89976	BUS 40
AA 89889	NEKO 15	AA 89977	BUS 41
AA 89890	NEKO 17	AA 89978	BUS 42
AA 89891	NEKO 19	AA 89979	BUS 43
AA 89892	NEKO 21	AA 89980	BUS 44
AA 89893	NEKO 22	AA 89981	BUS 45
AA 89894	NEKO 23	AA 89982	BUS 46
AA 89895	NEKO 24	AA 89983	BUS 47
AA 89896	NEKO 31		
Administrative Owner of Record: Landmark Alaska LP			
AA 87557	KM 1	AA 87632	JET 3

table continues…

BLM Serial No.	Claim Name	BLM Serial No.	Claim Name
AA 87558	KM 2	AA 87633	JET 4
AA 87559	KM 3	AA 87634	JET 5
AA 87560	KM 4	AA 87635	JET 6
AA 87561	KM 5	AA 87636	JET 7
AA 87562	KM 6	AA 87637	JET 8
AA 87563	KM 7	AA 87638	JET 9
AA 87564	KM 8	AA 87639	JET 10
AA 87565	KM 9	AA 87640	JET 11
AA 87566	TATER 1	AA 87641	JET 12
AA 87567	TATER 2	AA 87642	JET 13
AA 87568	TATER 3	AA 87643	JET 14
AA 87569	TATER 4	AA 87644	JET 15
AA 87570	ORCA 1	AA 87645	JET 16
AA 87571	ORCA 2	AA 87646	JET 17
AA 87572	ORCA 3	AA 87647	JET 18
AA 87573	ORCA 4	AA 87648	JET 19
AA 87574	ORCA 5	AA 87649	JET 20
AA 87575	ORCA 6	AA 87650	JET 21
AA 87576	ORCA 7	AA 87651	JET 22
AA 87577	ORCA 8	AA 87652	JET 23
AA 87578	ORCA 9	AA 87653	JET 24
AA 87579	ORCA 10	AA 87654	JET 25
AA 87580	TOT 1	AA 87655	JET 26
AA 87581	TOT 2	AA 87656	JET 27
AA 87582	TOT 3	AA 87657	JET 28
AA 87583	TOT 4	AA 87658	JET 29
AA 87584	KB 3	AA 87659	JET 30
AA 87585	KB 5	AA 87660	JET 31
AA 87586	KB 6	AA 87661	JET 32
AA 87587	KB 7	AA 87662	JET 33
AA 87588	KB 8	AA 87663	JET 34
AA 87589	KB 9	AA 87664	JET 35
AA 87590	KB 10	AA 87665	JET 36
AA 87591	KB 11	AA 87666	JET 37
AA 87592	KB 12	AA 87667	JET 38
AA 87593	KB 13	AA 87668	JET 39
AA 87594	RIO 1	AA 87669	JET 40
AA 87595	RIO 2	AA 87670	JET 41
AA 87596	RIO 3	AA 87671	JET 42
AA 87597	RIO 4	AA 87672	JET 43
AA 87598	RIO 5	AA 87673	JET 44

table continues…

BLM Serial No.	Claim Name	BLM Serial No.	Claim Name
AA 87599	RIO 6	AA 87674	JET 45
AA 87600	RIO 7	AA 87675	JET 46
AA 87601	RIO 8	AA 87676	JET 47
AA 87602	RIO 9	AA 87677	JET 48
AA 87603	RIO 10	AA 87678	JET 49
AA 87604	RIO 11	AA 87679	JET 50
AA 87605	OTTER 1	AA 87680	JET 51
AA 87606	OTTER 2	AA 87681	JET 52
AA 87607	OTTER 3	AA 87682	JET 53
AA 87608	OTTER 4	AA 87683	JET 54
AA 87609	OTTER 5	AA 87684	JET 55
AA 87610	OTTER 6	AA 87685	JET 56
AA 87611	SH 55	AA 87686	JET 57
AA 87612	HOODOO 1	AA 87687	JET 58
AA 87613	HOODOO 2	AA 87688	JET 59
AA 87614	HOODOO 3	AA 87689	JET 60
AA 87615	HOODOO 4	AA 87690	JET 61
AA 87616	HOODOO 5	AA 87691	JET 62
AA 87617	HOODOO 6	AA 87692	JET 63
AA 87618	HOODOO 7	AA 87756	POWDER 1
AA 87619	DARWIN 1	AA 87757	POWDER 2
AA 87620	DARWIN 2	AA 87760	ORCA 11
AA 87621	DARWIN 3	AA 87761	CR 1
AA 87622	DARWIN 4	AA 87762	ARR 8
AA 87623	DARWIN 5	AA 87763	ARR 9
AA 87624	ARR 1	AA 87764	ARR 10
AA 87625	ARR 2	AA 87765	ARR 11
AA 87626	ARR 3	AA 87766	ARR 12
AA 87627	ARR 4	AA 87767	ARR 13
AA 87628	ARR 5	AA 87768	ARR 14
AA 87629	ARR 6	AA 87769	ARR 15
AA 87630	ARR 7	AA 87770	ARR 16
AA 87631	JET 1		
Administrative Owner of Record: Landmark Minerals Inc.			
AA 86892	SH 1	AA 86941	SH 50
AA 86893	SH 2	AA 86942	SH 51
AA 86894	SH 3	AA 86943	SH 52
AA 86895	SH 4	AA 86944	SH 53
AA 86896	SH 5	AA 86945	SH 54
AA 86897	SH 6	AA 86946	SH 58
AA 86898	SH 7	AA 86947	SH 59

table continues…

BLM Serial No.	Claim Name	BLM Serial No.	Claim Name
AA 86899	SH 8	AA 86948	SH 60
AA 86900	SH 9	AA 86949	SH 61
AA 86901	SH 10	AA 86950	SH 62
AA 86902	SH 11	AA 86951	SH 63
AA 86903	SH 12	AA 86952	SH 64
AA 86904	SH 13	AA 86953	SH 65
AA 86905	SH 14	AA 86954	SH 66
AA 86906	SH 15	AA 86955	SH 67
AA 86907	SH 16	AA 86956	SH 68
AA 86908	SH 17	AA 86957	SH 69
AA 86909	SH 18	AA 86958	SH 70
AA 86910	SH 19	AA 86959	SH 71
AA 86911	SH 20	AA 86960	SH 72
AA 86912	SH 21	AA 86961	SH 73
AA 86913	SH 22	AA 86962	SH 74
AA 86914	SH 23	AA 86963	SH 75
AA 86915	SH 24	AA 86964	SH 76
AA 86916	SH 25	AA 86965	SH 77
AA 86917	SH 26	AA 86966	SH 78
AA 86918	SH 27	AA 86967	SH 79
AA 86919	SH 28	AA 86968	SH 80
AA 86920	SH 29	AA 86969	SH 81
AA 86921	SH 30	AA 86970	SH 82
AA 86922	SH 31	AA 86971	SH 83
AA 86923	SH 32	AA 86972	SH 84
AA 86924	SH 33	AA 86973	SH 85
AA 86925	SH 34	AA 86974	SH 86
AA 86926	SH 35	AA 86975	SH 86A
AA 86927	SH 36	AA 86976	SH 87
AA 86928	SH 37	AA 86977	SH 88
AA 86929	SH 38	AA 86978	SH 89
AA 86930	SH 39	AA 86979	SH 90
AA 86931	SH 40	AA 86980	SH 91
AA 86932	SH 41	AA 86981	SH 92
AA 86933	SH 42	AA 86982	SH 93
AA 86934	SH 43	AA 86983	SH 94
AA 86935	SH 44	AA 86984	SH 95
AA 86936	SH 45	AA 86985	SH 96
AA 86937	SH 46	AA 86986	SH 97
AA 86938	SH 47	AA 86987	SH 98
AA 86939	SH 48	AA 86988	SH 99

table continues…

BLM Serial No.	Claim Name	BLM Serial No.	Claim Name
AA 86940	SH 49		
Administrative Owner of Record: Irene & Robert Dotson			
AA 28263	Little Mary Rose #7	AA 28271	Atom Gayle
AA 28264	ATOM IRENE #3 Lode Claim	AA 28278	Fraction Rosemary
AA 28267	ATOM PETE Lode Claim	AA 28279	ATOM ROSE
AA 28269	LITTLE SUE #1	AA 28260	LITTLE RAY #1 Lode Claim
AA 28270	LITTLE RED #2	AA 28261	ATOM MARIETTA #4 Lode Claim
Administrative Owner of Record: Robert Dotson			
AA 87758	KB 56	AA 87759	KB 57
Administrative Owner of Record: Gayle Dotson			
AA 14166	I & L # 3	AA 64598	Geiger I
AA 14167	I & L # 4	AA 64599	Geiger II
AA 14168	I & L # 5	AA 85963	Rascal 1
AA 61364	R.I.D. #1 Claim	AA 85964	Rascal 2
AA 61365	R.I.D. #2 Claim		
Administrative Owner of Record: FFI Limited Agreement #6			
AA 77884	Keg 1	AA 77895	Keg 12
AA 77885	Keg 2	AA 77896	Keg 13
AA 77886	Keg 3	AA 77897	Keg 14
AA 77887	Keg 4	AA 77898	Keg 15
AA 77888	Keg 5	AA 77899	Keg 16
AA 77889	Keg 6	AA 77900	Keg 17
AA 77890	Keg 7	AA 77901	Keg 18
AA 77891	Keg 8	AA 77902	Keg 19
AA 77892	Keg 9	AA 77903	Keg 20
AA 77893	Keg 10	AA 77904	Keg 21
AA 77894	Keg 11	AA 77905	Keg 22
Administrative Owner of Record: Mary Anderson			
AA 28262	ATOM FLORENCE #5 Lode Claim	AA 28273	ATOM MIKE
AA 28265	ATOM JOE #9 Lode Claim	AA 28274	ATOM HARL
AA 28266	ATOM MAXINE #10 Lode Claim	AA 28275	ATOM DON
AA 28268	IRENE D	AA 28276	Atom Del
AA 28272	Little Pete Fraction	AA 28277	Atom Bobby
Administrative Owner of Record: Susan Dotson			
AA 81338	Digger-D#1	AA 81343	Digger-D#6
AA 81339	Digger-D#2	AA 82035	Digger-D #7
AA 81340	Digger-D#3	AA 82036	Digger-D #8
AA 81341	Digger-D#4	AA 82037	Digger-D #9
AA 81342	Digger-D#5	AA 82038	Digger-D #10
Administrative Owner of Record: Troy C. and Anne S. Erwin Agreement #5			
AA 61314	Gray Sea No. 1	AA 61669	Shy Anne #2

table continues…

BLM Serial No.	Claim Name	BLM Serial No.	Claim Name
AA 61315	Gray Sea No. 2	AA 61670	Shy Anne #3
AA 61316	Gray Sea No. 3	AA 61335	Surely Anne #1
AA 61668	Shy Anne #1	AA 61336	Surely Anne #2
Administrative Owner of Record: Susan Streets			
AA 86597	Snicker's No. 1	AA 86600	Snicker's No. 4
AA 86598	Snicker's No. 2	AA 86601	Snicker's No. 5
AA 86599	Snicker's No. 3	AA 86602	Snicker's No. 6
Administrative Owner of Record: Raymond Anderson			
AA 85954	Bear Dog 4	AA 85960	Sand-D1
AA 85955	Bear Dog 5	AA 85961	Sand-D-2
AA 85956	Bear Dog 6	AA 86844	Raymond A
AA 85959	Atom Raymond		
Administrative Owner of Record: David Anderson			
AA 85951	Bear Dog 1	AA 85958	Atom David
AA 85952	Bear Dog 2	AA 85962	Sand-D-3
AA 85953	Bear Dog 3	AA 86843	David A
AA 85957	Marys Creek		

Notes: Claims in orange are Agreements 1, 2 and 3.
Claim in green agre Agreement 4.
Claims in yellow are Agreement 5.
Claims in pink are Agreement 6.
Claims in red are 100% Ucore owned.

4.2 ROYALTIES, PAYMENTS, AND ENCUMBRANCES

To the extent known, the royalties, encumbrances, payments and other agreements pertaining to the claims comprising the Property are fully and completely described in Section 4.1. The QP responsible for this section is not aware of any other legal, social or moral obligations pertaining to the mining claims.

The surface rights are held by the US federal government and are administered by the USFS. This is discussed further in Section 20.0.

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

The Property covers part of a 5 km by 10 km (3.1 miles by 6.2 miles) area on southern Prince of Wales Island, the southern-most major island of the Alexander Archipelago. The Property is centered approximately between the south arm of Moira Sound and the west arm of Kendrick Bay. The area topography ranges from moderate to precipitous. Both Kendrick Bay and Moira Sound are glacial fjords with deep-water marine access to and from Ketchikan and Prince Rupert.

5.1 ACCESSIBILITY

The Property can be accessed via fixed-wing seaplanes and rotary-winged aircraft, or by boat into the West Arm of Kendrick Bay. The Property is located approximately 60 km (35 miles) southeast of Ketchikan, and 133 km (83 miles) northwest of Prince Rupert, BC. There is limited access to portions of the Property from Kendrick Bay by means of four-wheel drive utility vehicles utilizing the old mine haul roads from historic mining operations. The remainder of the Property can be reached by foot or helicopter.

5.2 CLIMATE AND PHYSIOGRAPHY

Southeastern Alaska enjoys a cool-to-moderate temperate climate characterized, by high rainfall and occasionally high winds, due to its low elevation and proximity to the Pacific Ocean. The average yearly temperature in Ketchikan is 7.5°C (45.5°F). Annual precipitation for Ketchikan is estimated at 348 cm (137”), with increased rainfall from September to March. Annette Island, located 25 km (15.5 miles) east of Bokan Mountain, has an average annual temperature of 8°C (46.4°F). with total annual precipitation of 256 cm (101”); Port Alexander, located 215 km (133.6 miles) northwest of Bokan Mountain, has an average annual temperature of 7°C (44.6°F), and total annual precipitation reaches 406 cm (160”). Snowfall is directly related to elevation. The exploration season is generally early May to late September.

Elevations range from sea level to 740 masl (2,423 fasl) at Bokan Mountain, the highest point on the Property. Most of the current exploration areas are less than 300 masl (984 fasl). Vegetation consists of old growth Sitka spruce, Douglas fir, shore pine, western red cedar, hemlock, and red alder rain forest along the shore and in steep valleys. Locally thick underbrush consists of devil’s club and various berries, shrubs, and grasses. Physiography ranges from swampy muskeg in granite saddles on the mountain slopes to open, glacially scoured, alpine granite of Bokan Mountain.

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

The Project area is a remote, uninhabited, roadless (USFS designation) area with year-round access via boat, float plane, barge or helicopter. The only infrastructure consists of about 4 km (2.5 miles) of permitted gravel roads from historic mining operations. There is no electrical infrastructure in the Property area. There is abundant fresh water available at all times of the year.

The nearest road and railhead are located in Prince Rupert, 133 km (82.6 miles) to the southeast.

6.0 HISTORY

6.1 DOTSON ZONE

The Dotson Zone has been the subject of limited, small scale prospecting, trenching and sampling by local prospectors and claim holders. The largest amount of field work and associated research was carried out by personnel from the US Bureau of Mines in the mid-to-late 1980s (Warner and Barker 1989). Warner and Barker describes the Dotson Zone as follows:

> *The prospect comprises a system of west-northwest-striking, steeply dipping dikes that can be traced between test pits and outcrops for approximately 2,100 m from tidewater on Kendrick Bay north-westward to the 335 m elevation of Bokan Mountain. Here they are cut by and probably displaced to the south from the I and L prospect dikes by the north-trending Dotson shear zone. The dikes likely extend southeastward offshore under the West Arm Kendrick Bay (ibid).*

A NI 43-101 compliant resource estimate was published in April 2011 (ibid), which is supported by mapping, trenches, and channel sampling totalling 87 samples from the Dotson dyke veins.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

Bokan Mountain lies within the Alexander Terrane of the Canadian-Alaskan Cordillera. The Cordillera is generally considered to be a collage of allochthonous oceanic and pericratonic terranes that were accreted to the western margin of the North American craton during the Mesozoic (Coney et al. 1980; Colpron et al. 2007). The Alexander Terrane is a composite terrane which does not show any evidence of an early relationship with the western margin but instead, geological data suggest that it originates far from western North America (Laurentia) and was only transported into the Cordilleran realm by later tectonic processes (Bradley et al. 2003; Nokleberg et al. 2000). The Alexander Terrane underlies most of southeastern Alaska and parts of western BC and southwestern Yukon. It is an arc terrane composed of late Proterozoic to Triassic mafic to felsic volcanic rocks, terrigenous clastic and carbonate rocks and early Paleozoic and rare Mesozoic granitic rocks. The pre-Devonian rocks were formed in a subduction-related environment. The terrane was amalgamated with another exotic terrane (i.e. Wrangellia) approximately 310 Ma years ago (Gardner et al. 1988) forming the Insular Superterrane which was subsequently accreted onto the western margin of the North American craton by the Cretaceous (approximately 115 to 95 Ma years ago).

Southern Prince of Wales Island (Figure 7.1) is underlain by igneous, metamorphic, and sedimentary rocks ranging in age from late-Proterozoic to mid-Cretaceous (Gehrels 1992). The oldest exposed rocks are pre-Middle Ordovician arc-related basaltic to andesitic volcanic rocks, volcaniclastic greywacke, and subordinate limestone and rhyolitic volcanic rocks of the Wales metamorphic suite. During middle Cambrian to early Ordovician time, these rocks were metamorphosed to greenschist and locally amphibolite facies and were penetratively deformed (Gehrels and Saleeby 1987) during the Wales Orogeny. The units are stratigraphically overlain by basaltic to rhyolitic flows, tuffs, breccias, and clastic sediments of the Descon Formation accumulated during the Ordovician and possibly as late as Silurian. The sedimentary rocks contain conodonts and graptolites (de Saint-Andre et al. 1983) and were imbricated on southwest-vergent thrust faults continuing into the earliest Devonian (Gehrels and Saleeby 1987). The Descon Formation was intruded by large bodies of middle Ordovician to early Silurian diorite and granodiorite. Both the Descon Formation and the Ordovician-Early Silurian intrusions were emplaced in a subduction-related setting.

Overlying Lower Devonian conglomerates of the Karheen Formation were deposited as part of a regional clastic wedge (Gehrels and Saleeby 1987). Conglomeratic beds, low in the section, were deposited in topographically rugged, subaerial to shallow marine environments, whereas strata high in the section were deposited in more distal marine environments (Gehrels 1992).

Some of the post-Devonian rocks exposed on southern Prince of Wales Island are related to igneous events associated with northeast and northwest strike-slip faults (Gehrels 1992). Rocks include the Jurassic (de Saint-Andre et al. 1983; Armstrong, 1985) Bokan Mountain Granite as well as various felsic to mafic dike swarms. The Bokan Mountain Granite, like other rare Jurassic intrusive bodies in southeastern Alaska (Gehrels and Saleeby 1987; de Saint-Andre et al. 1983; Maas et al. 1995) is associated with an anorogenic extensional tectonic setting. However, middle Cretaceous granodiorite and diorite bodies in the southern part of Prince of Wales Island are probably related to subduction processes (Gehrels and Saleeby 1987).

Figure 7.1 Regional Geology

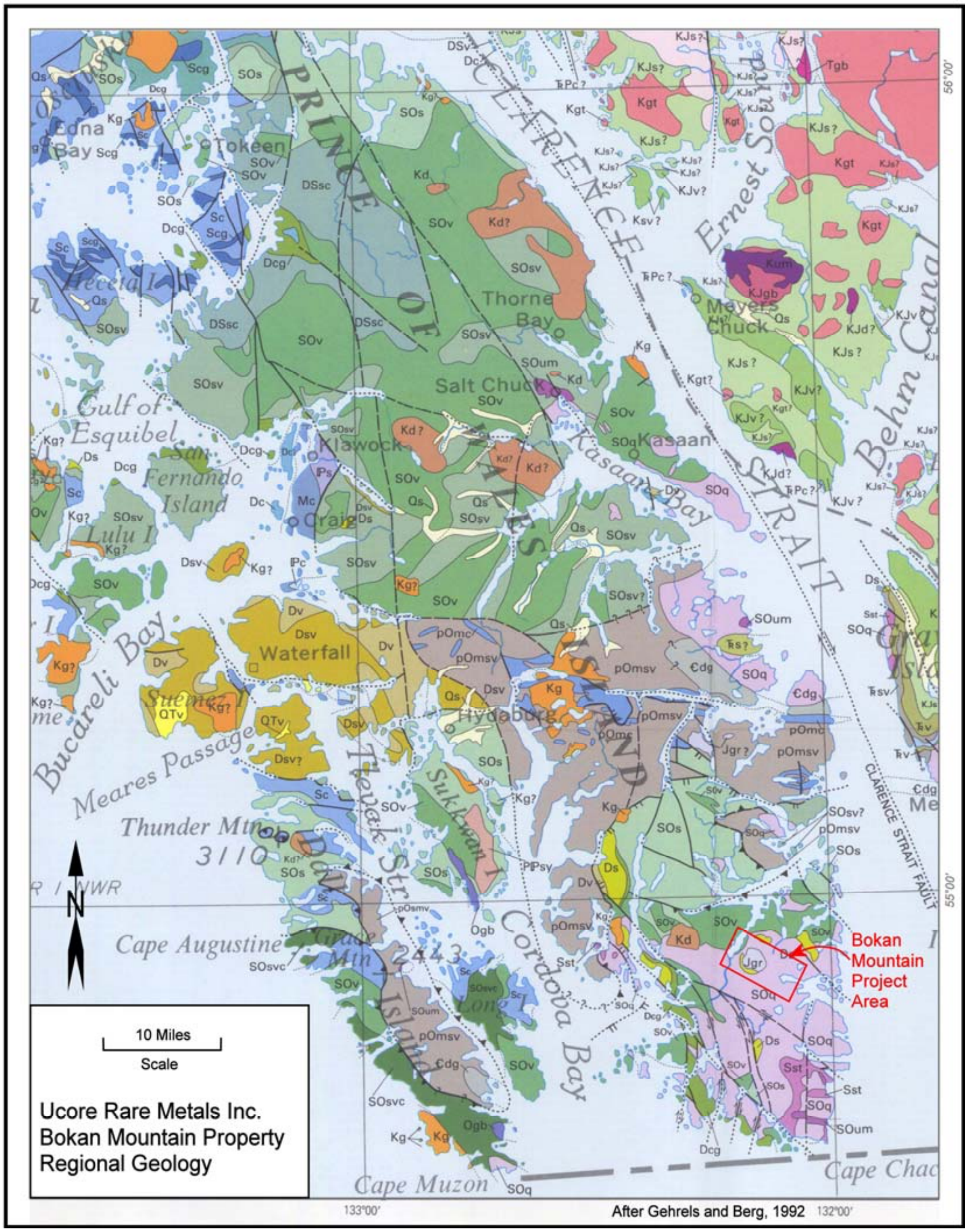
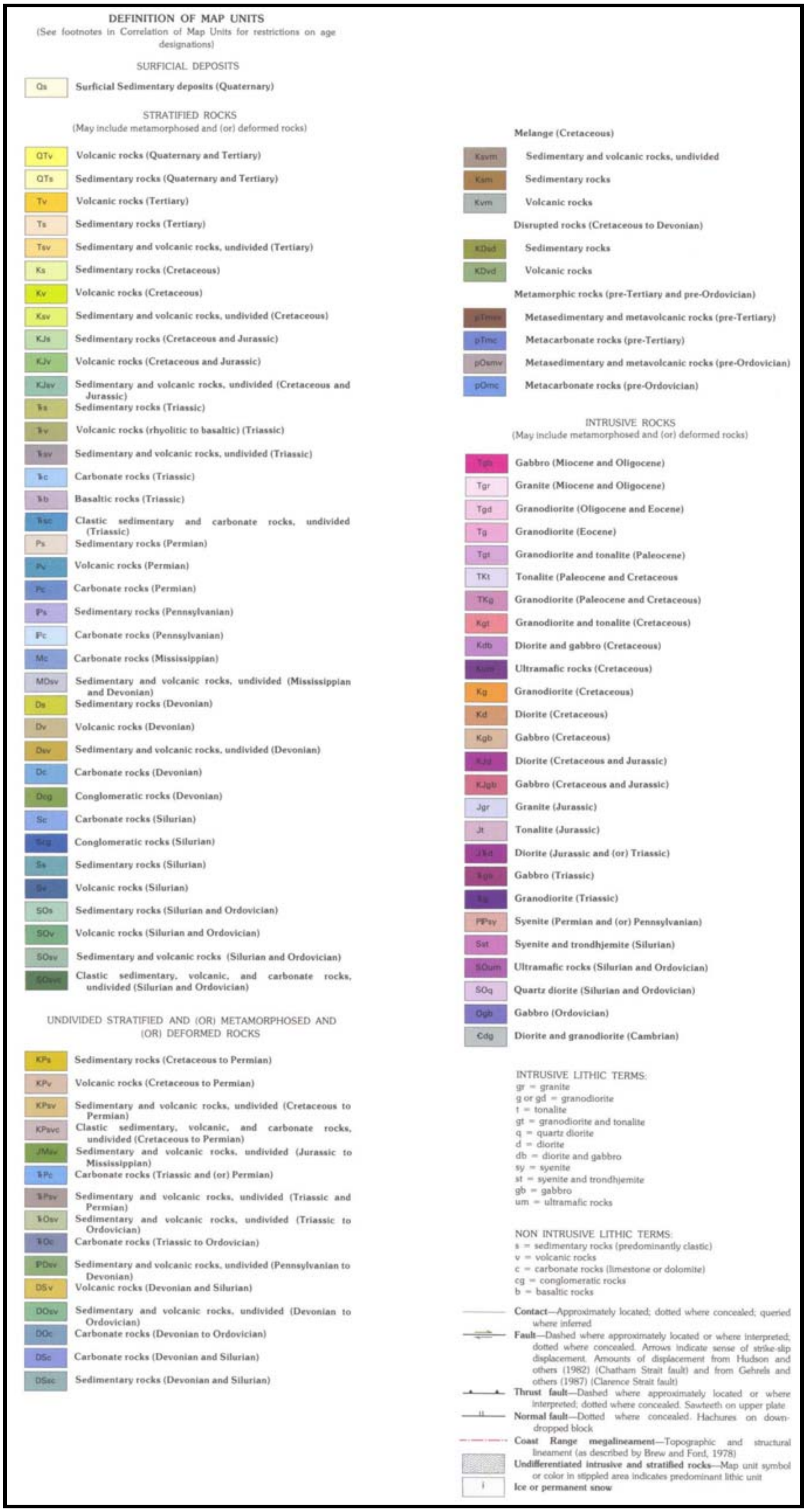


Figure 7.2 Regional Geology Legend



7.2 PROPERTY GEOLOGY

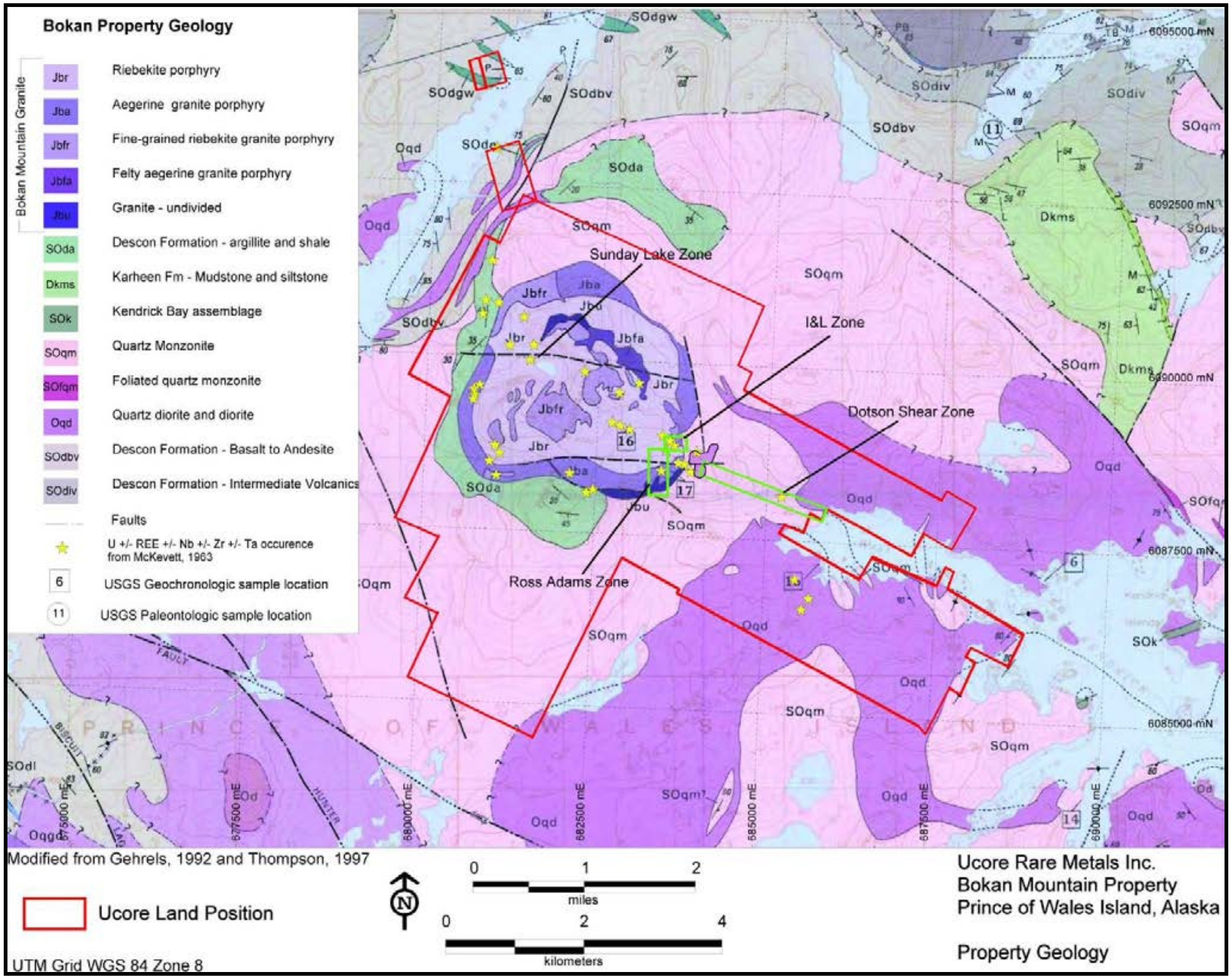
The geology of the Bokan Mountain area including the Dotson Zone has been mapped by McKevett (1963), Gehrels (1992) and Thompson et. al. (1982). Mapping, centered on the Dotson Zone, was conducted by L. Covello in 2009 of Aurora. In 2010, mapping, prospecting and trenching in the immediate area of the Dotson Zone was conducted by Aurora employees P. McDonald, S. Dotson, M. Bezzola, and D. Mac Gerailt. Drilling was performed on the Dotson Zone in 2008, 2009 and 2010 with logging performed by J. Couture, H. Archibald, A. Leather, J. Kriz and M. Bezzola. Figure 7.3 shows the general local geology of the Property.

This section summarizes the geology of the Dotson Zone, based primarily on prospecting, geological mapping, trenching and diamond drilling conducted along the zone during 2010. The area is underlain by the rock units listed in Table 7.1.

Table 7.1 Summary List of Rock Units on the Bokan Mountain Property

Unit (Age)	Summary Description
Overburden (Quaternary - Holocene)	Organic and elluvial soil, boulder till at lower elevations and alluvial deposits at lower elevations in major creeks.
Jbu (Jurassic)	Aplite
Jbr (Jurassic)	Bokan Granite - Aegerine and riebeckite-bearing granite.
MD (Silurian - Ordovician and younger)	Andesitic to basaltic dykes and sills (two series)
Soqm (Silurian - Ordovician)	Quartz monzonite, monzonite, monzodiorite and lesser granite.
Oqd (Ordovician)	Quartz diorite, diorite and gabbro.
JBGr Bokan Granite (Jurassic)	Bokan Granite - Aegerine-bearing granite.
pJQFP Quartz Feldspar Porphyry (pre-Jurassic)	Dikes and small plugs of leucocratic quartz feldspar porphyry
SODa Descon Formation (Silurian - Ordovician)	Andesitic to basaltic dykes and sills (two series), tuffs and metasediments. Adjacent to the Bokan Granite, this unit

Figure 7.3 Property Geology



Rock units in the immediate area of the Property (Table 7.1) are discussed in turn.

OQD

Ordovician quartz diorite, diorite and gabbro occur at the eastern end of the Dotson Zone from Kendrick Bay to below the peak of Dotson Hill, and at the western end of the Dotson Zone in a small outlier. In general, the rocks are dark green to black mottled with white with more mafic phases dark green to black. The rocks are dominantly fine grained (1 to 2 mm) with both coarser and finer phases present. Coarser phases with crystals to 4 mm are more common than the finer phases. The rock is generally massive to very weakly foliated, aphyric to slightly porphyritic with hornblende crystals and aggregates larger than other crystals. Leucocratic components include a mixture of waxy, translucent to opaque green-grey, striated plagioclase (dominant) and rare white to occasionally tan or pink opaque, often lamellar albite or potassium feldspar. Gabbro and diorite phases are entirely composed of plagioclase. The colour index of the rock varies from around 30 to 80% in general with some gabbroic intervals, particularly noted by L. Covello at the western end of the Dotson Zone. Quartz (up to 10%) occurs in anhedral equant blobs or aggregates of anhedral crystals interstitial to other phases. Pyrite is ubiquitous and comprises from 1 to 3%. Hornblende phenocrysts and the dark groundmass of the rock is magnetic indicating that magnetite is common. The magnetic susceptibility of this rock unit varies from 2 to 18 SI units with 6 to 8 being most common. The SI measure of volume magnetic succeptability is defined as H/M, where H is the magnetic field strength measured in amperes per metre, and M is the magnetic dipole moment per unit volume, also measured in amperes per metre. Figure 7.4 shows a typical example of this rock unit from hole LM10-75.

Several styles of alteration are present. Epidote alteration is locally extensive and particularly affects plagioclase. In some areas, pervasive, uniform and intense epidote alteration locally alters leucocratic quartz diorite to a much darker rock resembling diorite or gabbro. Chlorite alteration is present throughout Oqd and dominantly occurs in veins, on fracture surfaces and in the rock unit within a few millimeters of vein contacts. Chlorite alteration is pervasive near fault zones and in other local areas, extending throughout the rock unit. The rock is unfoliated, except with respect to secondary chlorite. Silicification occurs near the contact with SOqm in several intervals in LM10-77. In these intercepts, fine quartz flooding imparts a smoothed or diffuse texture to the rock, particularly on cored surfaces. Calcite veining is also locally common in this rock type, in particular near REE bearing veins.

The contact between Oqd and the overlying Soqm is intercalated on the eastern end of the Dotson Zone where drill intercepts near the contact consist of alternating bands of quartz monzonite and quartz diorite. In some bedrock exposures partially assimilated blocks of fine grained diorite to quartz diorite are found within the quartz monzonite forming agmatite. The contact between Oqd and SOqm is much sharper in the western portion of the Dotson Zone. Gehrels (1992) cites a uranium-lead date of 445 + 5 Ma and a K-Ar date of 439 + 21 Ma for this rock unit (Late Ordovician).

Unit Oqd is cut by two generations of mafic dykes and by an aplite dyke in Hole LM10-86 (18 m), as shown in Figure 7.4.

Figure 7.4 Quartz Diorite (Unit Oqd) – Hole LM10-75

SOqm

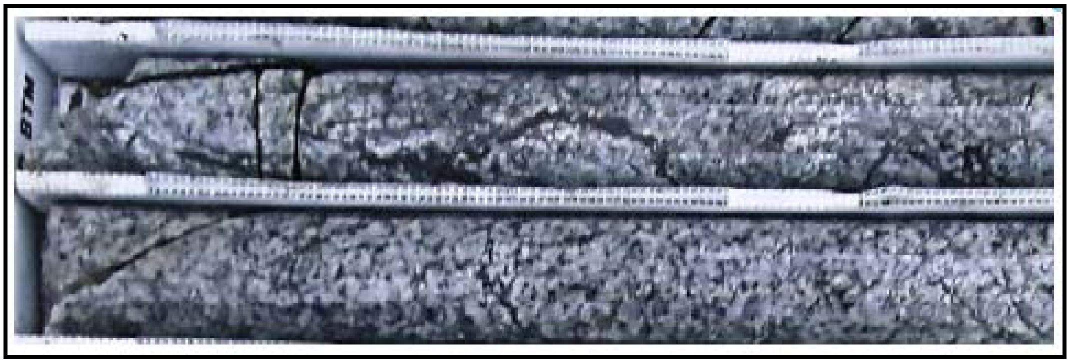
Rock unit SOqm underlies most of the area of the Dotson Zone from the eastern flank of Dotson Hill to the western boundary of the Dotson Zone on the flanks of Bokan Mountain.

In general, this rock unit is greenish light grey to white on fresh surface and light green grey on cored surfaces. The rock is dominantly aphyric and medium grained (1 to 5 mm). Plagioclase and potassium feldspar are the dominant minerals in subequal to plagioclase-dominant proportions. Plagioclase is greenish white, waxy opaque to translucent, dominantly subhedral in striated lathe shaped crystals and, more commonly, in inter-grown crystal aggregates. Plagioclase crystals are commonly rimmed by secondary white, opaque, chalky albite. Potassium feldspar is pink-white, white to tan in colour, medium grained and occurs in the same habit as plagioclase. Hornblende is dark green to black, medium grained, suhedral to euhedral and occurs as disseminated crystals, generally less than 20% of the rock and as little as 5% of some leucocratic units. Quartz comprises 5 to 30% of the rock and occurs as clear glassy to white translucent, subhedral crystals and in crystal aggregates with characteristic conchoidal fracture and iridescent sheen on fracture surfaces. It is also found as light grey interstitial fillings between feldspar. Magnetite is rare and never more than about 1%. Fine grained pyrite is found as disseminations (generally subhedral) or euhedral cubic grains near fractures.

As with Oqd, the rock unit is characterized by extensive epidote and chlorite alteration. In addition, albitization is locally intense and in some instances to the point where primary plagioclase and rock textures are obliterated by white chalky to very fine crystalline white opaque albite. Calcite alteration is locally prevalent near mineralized veins and some chlorite vein sets. Fluorite is rarely observed in this rock unit and generally only near REE mineralized veins. The contact between SOqm and the Bokan

Mountain granite (Jbr) was not observed. Unit SOqm is cut by two generations of mafic dykes and by aplite dykes, as shown in Figure 7.5.

Figure 7.5 Quartz Monzonite (Unit Soqm) from Hole LM10-80



Rock unit MD is not mapped at a regional scale but was noted by McKevett (1963), Gehrels (1992), and Thompson (1997). This rock unit consists of melanocratic, porphyritic to aphyric (diabase) dykes of basaltic through andesitic composition. Rare lamprophyre dykes also occur in drill intersections along the Dotson Zone. There are two generations of these dykes, defined by their alteration mineral assemblage and texture.

Both classes of dyke are characterized by a fine grained to aphanitic dark to medium grey ground mass of quartz, plagioclase and mafic minerals. Most are porphyritic with dominant euhedral plagioclase and lesser hornblende phenocrysts generally 4 to 8 mm. In general, the dykes do not display trachytic texture. Plagioclase phenocrysts typically comprise from 10 to 30% of the rock mass and occur as euhedral crystals to broken crystals up to 4 cm (LM10-80: 76 m). Hornblende phenocrysts are less common except in the case of rare lamprophyre dykes wherein they comprise the majority of phenocrysts. Rare pyroxene (augite?) phenocrysts are found in LM10-79: 75 m, characterized by square phenocryst cross-sections. At LM10-79: 120 m, phenocrysts are sorted throughout the dyke by size; this is the exception rather than the rule. Some of the mafic dykes are aphyric and massive; these may occur in either class. Chill margins are present in some of the wider dykes (greater than 5 m). One wider dyke intersection (LM10-79: 16 m) showed chill margins extending to 10 cm from the dyke edges. In these settings, the rock consists of an aphanitic groundmass (of either class) with grain size increasing towards the center of the dyke, zone symmetrically about the center of the dyke. The vast majority of the dyke intersections are thinner and do not show these features. Lamprophyre dykes are characterized by dominant hornblende phenocrysts with subordinate to absent plagioclase phenocrysts but are otherwise identical to the rest of the dykes in mineralogy, alteration and apparent geometry. In addition, approximately 10% of the dykes in both classes were aphyric very fine grained rocks. Both dyke classes occasionally contain included blocks of country rock and rounded coarser grained dyke rocks to at least 10 cm. Some of the dykes show elevated yttrium levels to in excess of 1,300 ppm. Mineralized veins and aplitic rocks cut both classes of mafic dykes.

The older class of mafic dykes is characterized by chloritic groundmass alteration, by epidote alteration and veining and by more frequent calcite veining, compared to the younger class of dykes. The older dykes appear less “fresh” than the younger dyke class. They are less competent and fracture more readily when struck. These dykes also appear to have a pervasive fabric defined by chlorite.

The younger class of dykes is massive and does not display pervasive chlorite alteration. All lamprophyre dykes are of this class as well. These dykes have very sharp boundaries with surrounding country rocks in contrast to the older dyke class in which occasional examples of gradational contacts may be found.

The mafic dykes do not occur on a regional scale, and occur on a property scale only between LM10-83/84 and LM10-85/86 on the Ross Adams haul road. As shown in Figure 7.6, the older dyke class has highly irregular contacts with the surrounding rocks and the younger dyke class has no known exposed contacts along the Dotson Zone. Gehrels (1990) notes the younger dyke class is steeply dipping and generally about 1 m thick.

Figure 7.6 Mafic Dykes

Note: (a) = older, slightly foliated mafic dyke Intruding quartz monzonite, (b) = younger mafic dyke with both aphyric and porphyritic (white plagioclase) phases.

JBU

Aplite is mapped at the western end of the Dotson Zone and was intersected in holes LM10-85 and LM10-86 (Figure 7.7). An aplite sill outcrops in trench TR10-10 and intrudes mafic dyke and quartz diorite on the nearby upper Ross Adams haul road.

The aplite is grey to tan-white weathering pink, tan to light grey. The rock is fine grained (less than 1 mm) with rare coarse crystalline intervals in wider dykes and sills. The aplite is almost always aphyric with the exception of a few drill intercepts with potassium feldspar phenocrysts to 5 mm. In general, the rock is massive, resistant and equigranular with a characteristic “sugary” texture. Average modal composition includes plagioclase (approximately 40%), quartz (30%), potassium feldspar (20%) and mafic minerals (hornblende, biotite, magnetite, sphene?) (10%). Plagioclase is grey-white, subhedral, less than 1 mm and occurs in masses of inter-grown crystals. Quartz is subhedral, around 1 mm, equant, grey translucent to glassy and occurs in isolated crystals. Potassium feldspar is generally pink-white, subhedral to anhedral and less than 1 mm in size. Mafic minerals occur in isolated crystals, less than 1 mm and uniformly distributed throughout the rock.

The aplite is fresh with little evidence of alteration. The aplite intrudes all other rock units as sills or dykes, generally less than a few metres thick. The mineralized REE veins both cut the aplite and are cut by the aplite.

Figure 7.7 Aplite Intruding Mafic Dyke (Hole LM 10-85)

OVERBURDEN

The Dotson Zone is generally overlain by unconsolidated overburden consisting of immature, very well drained organic soil and eluvium at higher elevations, by boulder till near the Ross Adams Mine and by recent alluvial deposits in Camp Creek and other major drainages.

7.3 DOTSON ZONE

MINERALIZATION

This section describes mineralized rare earth-bearing veins which comprise the principal economic target at the Dotson Zone. The geology of the Dotson Zone as well as detailed mapping results from the 2010 season are shown in Figure 7.9. The rock types and detailed vein properties are described in the following subsections.

VEIN STRUCTURE

This section describes the structure, morphology and distribution of mineralization in the mineralized veins. The Dotson Zone is a steeply dipping, tabular zone of numerous veins and narrow veinlets extending from east of Bokan Mountain to the shore of Kendrick Bay. The Dotson Zone has a defined strike length of 2,140 m, an average width of 50 m and has been traced down dip over a vertical extent of 450 m. On average, the zone strikes 108°, and dips 75 to 85° north.

In detail, there are two separate domains, separated by the Camp Creek Fault Zone which essentially bisects the Dotson Zone. There are small differences between the strike and dip of the veins in each zone and the overall dip of the zone as a whole. Figure 7.8 and Figure 7.10 are stereograms of the poles to veins from the western and eastern portions of the Dotson Zone, respectively.

Figure 7.8 Vein Orientations – West of Camp Creek Fault Zone

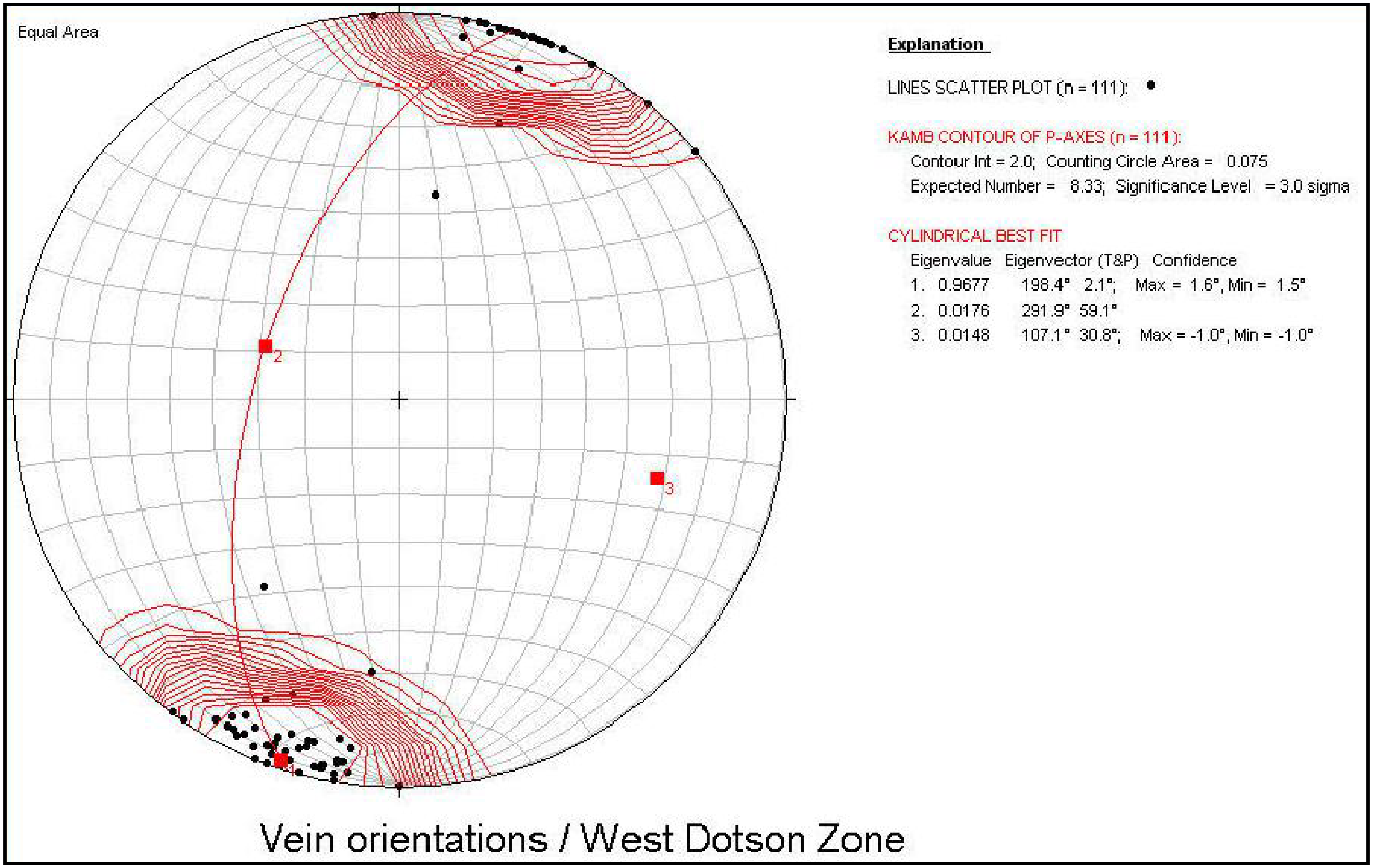


Figure 7.9 Dotson Zone Vein Mapping

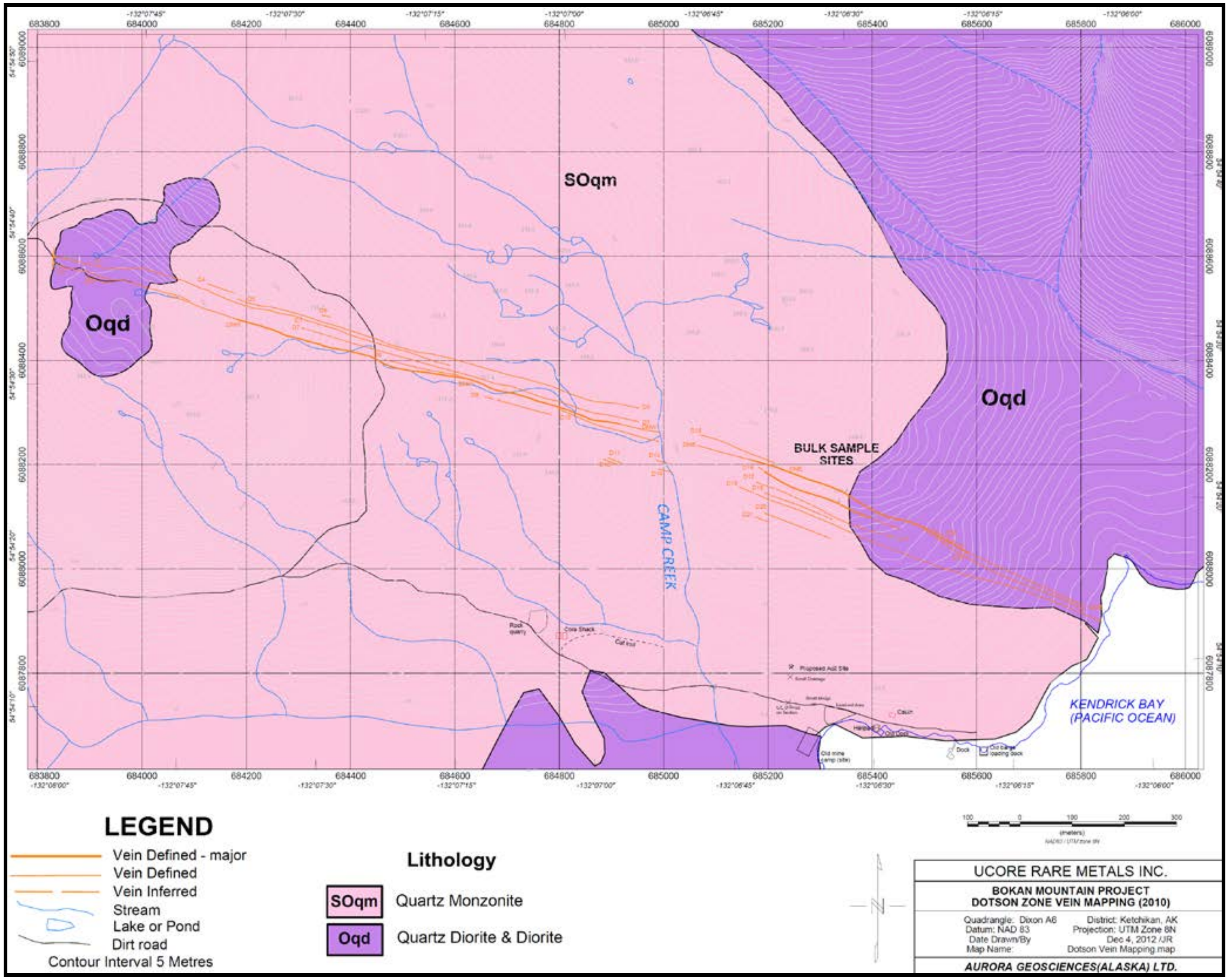
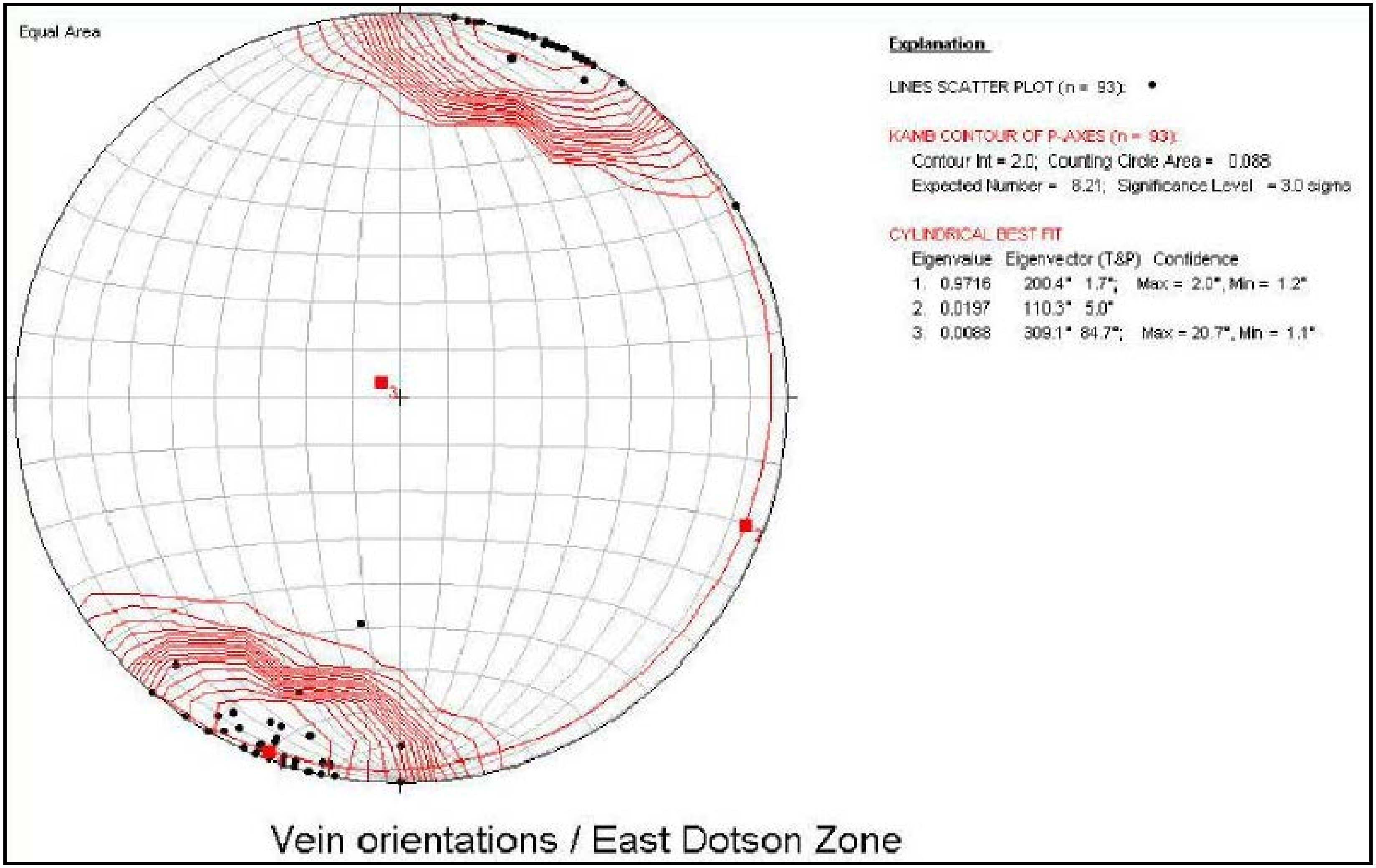


Figure 7.10 Vein Orientations – East of Camp Creek Fault



The mean orientation of veins in the western portion of the Dotson Zone is 108° azimuth and dipping 88° north while the veins in the eastern portion of the Dotson Zone are oriented 110° azimuth and dipping 88° north. The overall dip of the zone defined in drill section is summarized in Table 7.2.

Table 7.2 Dip of the Dotson Veins Observed Along the Structure

Baseline Station	Holes	Dip (°)
2020	85 to 86	82 to 86 north
1890	63 to 64	76 north
1820	83 to 84	77 to 81 north
1700	49 to 50	77 north
1630	72 to 73	75 to 86 north
1520	81	85 to -87 north
1430	47 to 48	75 to 88 north
1380	79 to 80	81 to 84 north
1330	51 to 52	85 to 88 north
1230	78	81 to 86 north
1110	53 to 54	77 to 82 north
1000	82	78 to 86 north
790	70 to 71	87 north to 86 south
700	36 to 37	87 to 89 north
580	77	86 to 89 north
525	55 to 56	83 to 90 north
435	76	84 to 87 north
350	57 to 58	82 north to 87 south
210	74 to 75	89 north to 87 south
35	68 to 69	88 to 90 south

There is a small flexure in the zone centered about 500 m west of Kendrick Bay where the dip of the zone (west of the flexure point) changes from steeply north-dipping to steeply south-dipping and continues in this orientation to the eastern end of the zone. This flexure occurs in an area where the overall width of the zone flares and veins appear more numerous and closely-spaced.

The Dotson Zone consists of a network or array of at least 25 sub-parallel veins. The REE mineralization occurs in individual veins, rarely up to 2 m wide and in vein arrays comprised of numerous thin parallel veins over intervals of several metres. The Dotson Main vein is the strongest single vein in the zone and tends to define the axis of mineralization in the drill sections. It extends for a strike length of 1,900 m and is up to 3 m wide in places. It has been subdivided into the Dotson Main West (DMW) and the Dotson Main East (DME) veins (shown on maps and drill sections) but appears to be a single continuous structure, slightly offset by the Camp Creek Fault. Other major veins of significant strike extent and width include the D1, D5, D7, D16, D19 and D23 veins.

The constituent veins in the Dotson Zone are remarkably planar and show little deviation over tens of metres or more. The veins rarely join in the horizontal plane and junctions are more often observed on the vertical plane in some outcrops. Figure 7.11 illustrates the typical form of observed vein junctions.

Figure 7.11 Vein Junctions on a Sloped Outcrop Near LM10-79

7.3.1 VEIN DESCRIPTIONS

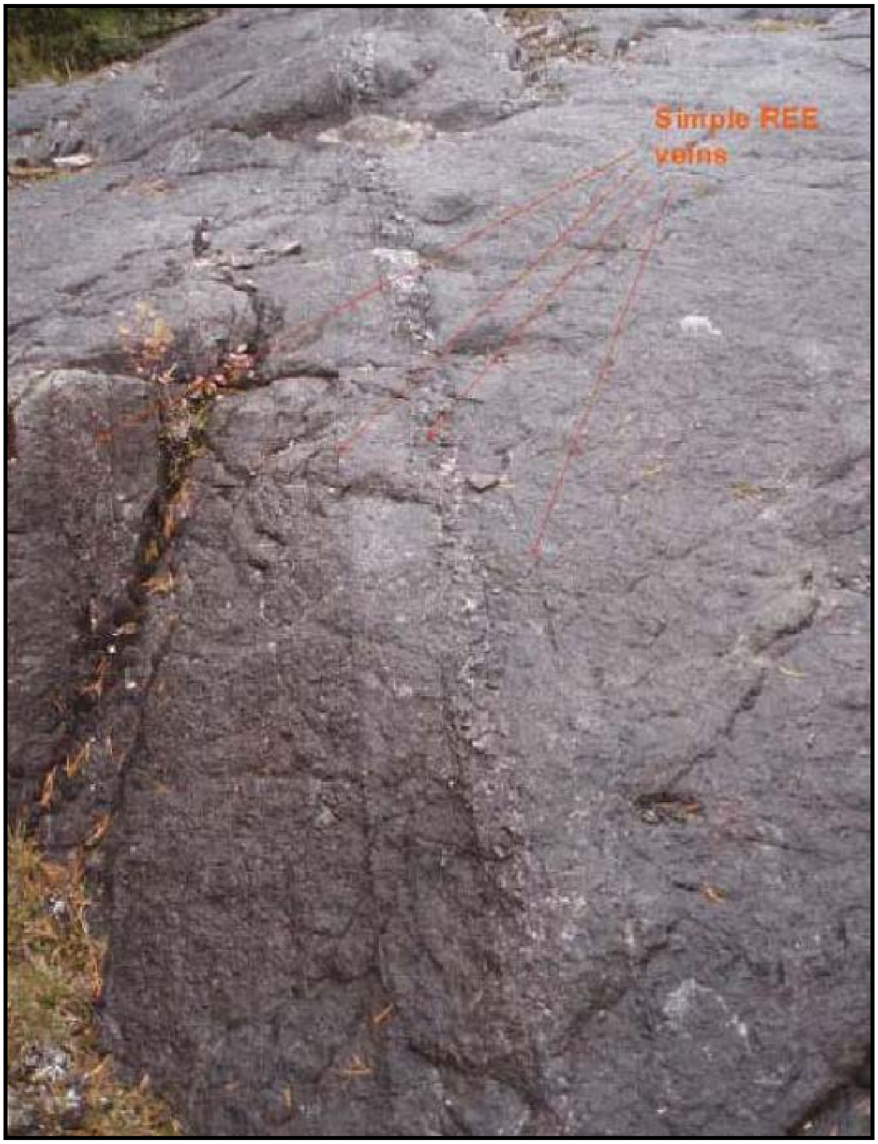
The REE-bearing veins in the Dotson Zone are fine to very coarse crystalline, simple to zoned mixtures of quartz, albite, chlorite, magnetite, calcite and a large variety of rare earth minerals including bastnasite, kainosite, allanite, and other exotic species

(T. Mariano 2010, pers. comm.). All REE-bearing veins show very sharp, straight and unaltered contacts with the surrounding wall rock. There are three types of mineralized veins found in the Dotson Zone described herein.

SIMPLE VEINS

The majority of the veins found along the Dotson Zone are simple veins consisting of a homogeneous mixture of equigranular minerals with little or no evident zonation. An example is shown in Figure 7.12. Virtually all veins less than 4 cm thick are simple veins and wider veins at the east and western ends of the zone are also simple veins. The vein mineralogy and orientation of the simple veins is not significantly different from that of the zoned mineralized veins described in the subsequent section except that the more exotic minerals are absent and they contain fewer pseudomorphs.

Figure 7.12 Simple REE-bearing Veins Near LM10-33



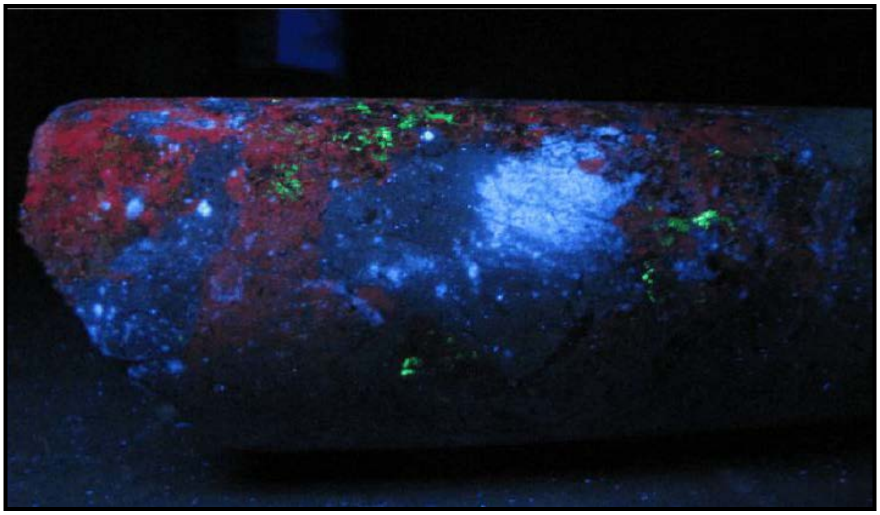
ZONED VEINS

Veins wider than about 10 cm in the centre of the Dotson Zone are commonly zoned with variations in mineralogy bilaterally symmetrical with respect to the vein centre line (Figure 7.13). The zoning consists of dark, aphanitic to very fine crystalline dark minerals adjacent to the vein walls grading or changing abruptly to leucocratic coarse crystalline minerals in the core of the veins. The vein selvage adjacent to the wall rock often consists of a mixture of magnetite, chlorite, aegerine (western end of the Dotson Zone) grading into dark rare earth-bearing minerals (allanite?) through to a core of quartz, albite and coarse rare earth-bearing minerals. As shown in Figure 7.13 and Figure 7.14, significant brecciation or comminution of the constituent minerals is absent, aside from some settling features. In the core of the veins, accessory minerals can include pyrite, galena, chalcopyrite and willemite (Figure 7.14).

Figure 7.13 Zoned REE-bearing Veins Near LM10-52

Figure 7.14 Detail of Zoned Vein Showing Sharp, Straight Margins, Lack of Wall Rock Alteration, Ingrowing Marginal Minerals and the Euhedral Central Vein Minerals

Figure 7.15 Core from Centre of a Zoned Vein in Filtered Ultraviolet Light



Note: Willemite fluoresces bright green, albite fluoresces dull.

CHLORITE-CALCITE VEINS

Chlorite and calcite veins are common alteration features in the quartz monzonite rock unit. A unique set of veins comprised of contorted black to very dark green-grey chlorite with fine to medium crystalline, often contorted calcite are found peripheral to the mineralized rare earth veins, either in parallel lateral veins or at the distal ends of an individual mineralized vein. These veins carry elevated but below economic concentrations of vanadium and zirconium (as measured with x-ray fluorescence).

Figure 7.16 illustrates typical fine crystalline calcite veining near the termination of a REE-bearing vein. In a few rare instances, calcite veins were observed grading into weak, thin REE-bearing veins (Figure 7.20). Some of the calcite veins cross-cut REE-bearing quartz veins. Finally, a few apparently late stage, high level calcite veins including rhombs in void spaces were intersected in the west central Dotson Zone (LM10-81).

Figure 7.16 Fine Calcite Veining Near the Termination of a REE-bearing Quartz Vein

Note: This intersection is beneath a thin weak simple REE-bearing vein exposed in a nearby trench (LM10-80).

Figure 7.17 White Calcite Vein (Right) Grades into Darker REE-bearing Quartz Vein Tip (Near "212" mark) (LM10-80)

DISSEMINATED REE MINERALIZATION IN QUARTZ MONZONITE

A single 30 cm occurrence of disseminated REE-bearing minerals in quartz monzonite was intersected at 135 m in DDH LM10-80. The interval consisted of albitized quartz monzonite containing 25% quartz and disseminated bastnasite(?) or pseudomorphs thereof together with disseminated calcite. This material returned 0.558% TREO (54% HREO).

DETAILED VEIN DESCRIPTIONS

The following subsections are a detailed description of the veins observed in the Dotson Zone during mapping and diamond drilling in 2010. The vein numbers refer to the veins as shown in Figure 7.17.

DMW

The DMW vein system is located at the central-west area of the Dotson Trend. Most of the vein system is well exposed with abundant outcrops. Two areas within the zone are poorly exposed: the western portion is forested and is mostly underneath deep overburden, and a 150 m section west of the road is located partially under overburden and partially under a creek bed. The vein's radiometric signature is lost under overburden both to the east and to the west. The vein system comprises one to seven distinct veins along its strike. It is estimated to continue over a length of 975 m; 36 stations were recorded. The average thickness is 34 cm, with a maximum observed thickness of 85 cm and a minimum observed thickness of 7 cm. The strike is approximately 102°. The strike of the veins varies slightly along the length of the vein system. The western portion (640 m long) has a strike which averages 107°, and the measurements of the eastern section have an average strike of 111° with variable but steep dip. The foliation noted in most outcrops is essentially vertical to sub-vertical with steep northerly dips.

The vein averages a maximum reading of approximately 3,200 cps. Typically, it has a coarse-grained core of 1 to 3 cm quartz and plagioclase crystals. Zones of quartz and plagioclase with pink and black mineral inclusions are typically located on the edge of the core of the vein, with centimetre-scale selvages of a black, metallic mineral. Most of the veins near the center of the vein system have black and pink lath-like minerals with their principal axes aligned perpendicular to strike. Generally, contacts are sharp and straight. The thickest veins are located on the western portion of the system, and thin out towards the eastern portion. The apparent thinness of the veins in the eastern portion of the system could be due to the fact that there is more overburden there and the entire vein system was not uncovered (i.e. only one or two of several veins were observed there).

D1

The vein is located on the west side of the Dotson Trend, north of the main zone (DMW). There is good exposure (much of it previously uncovered) on its easternmost outcrops, but the western side of the vein is located under forest with poor exposure or radiometric detection. The vein can be traced along much of the strike but has been estimated to continue between certain outcrops because of outcrop location, vein thickness, and strike. The vein system comprises one vein. The vein is estimated to continue over a length of 470 m; 10 stations were recorded. The average thickness is 60 cm, with a maximum observed thickness of 120 cm and a minimum observed thickness of 20 cm. The strike of the vein is approximately 110°. The west side of the vein is cut off by a fault, which is exposed. The east side of the vein is not traceable through overburden.

The vein averages a maximum reading of approximately 2,000 cps. Typically, it has a pegmatitic core with finer crystals near edges. Quartz crystals in the middle zone are up to 3 cm in size. Several outcrops contain up to 1 cm-long fibrous crystals of a black metallic mineral. Most outcrops have dark fine- to medium-grained banding at the edges. Contacts are generally planar but several outcrops have wavy contacts.

D1a

The vein system is located in the west-central zone of the Dotson Trend, approximately 8 m south of D8, 9 m north of D9, and 30 m north of the DMW. The vein system is located in an open area with good exposure, and it is traceable over most of its strike. The vein system is not traceable to the west where the radiometric signature is lost under a cat road. It is possible that it is the same vein system as D1 above, which is on strike west of the cat road. To the east, the radiometric signal is lost under overburden. The vein's estimated trace is extended 40 m east because of its agreement with the drill section. The vein system comprises one to four distinct veins along its strike. It is estimated to continue over a length of 245 m; 7 stations were recorded. The average thickness is 7 cm, with a maximum observed thickness of 15 cm and a minimum observed thickness of 1 cm. The strike is approximately 110°.

The vein averages a maximum reading of approximately 1,050 cps. Typically, it has a coarse-grained core of quartz, plagioclase, and aegirine crystals. The veins have thin, medium-grained, centimetre-scale selvage of a black mineral.

D2

The vein system is located on the west side of the Dotson Trend, approximately 6 m north of the DMW. There is good exposure along the strike of the vein and it is traceable with radiometrics along much of its length. The vein is not traceable to the east. However, it is possible that it joins the main vein system to the east of the easternmost outcrop. The vein system comprises one to five distinct veins along its strike. It is estimated to continue over a length of 270 m; 9 stations were recorded. The average thickness is 63 cm, with a maximum observed thickness of 162 cm and a minimum observed thickness of 4 cm. The strike is approximately 295° with a sub-vertical dip between 80 and 90°.

The vein system averages a maximum reading of approximately 2,300 cps. Typically, it has a pegmatitic core (with quartz crystals of up to 2 cm) grading to medium grain size near the edges. Large euhedral quartz and plagioclase crystals are dominant in the core zone. Inclusions of a medium-grained, earthy, pink mineral are concentrated between the core and outer medium- to fine-grained banding of a black, metallic mineral.

D3

The vein is located on the west side of the Dotson Trend, approximately 8 m south of D2. The vein wasexposed by trenching through approximately 20 cm of overburden. It is possible that this vein is connected to the DMW. Both D3 and the DMW are located between 5 to 10 m south of D2 and have similar strikes, but there is a zone of muskeg and a creek between the two zones and they are not traceable radiometrically.

The vein system outcrops only once at station PM10-229. From tracing its radiometric signature, it is estimated to be continuous over a length of at least 46 m. The vein's thickness is 30 cm. The observed strike as well as the strike traced by radiometrics is approximately 102°.

The vein averages a maximum reading of approximately 5,700 cps. It is dominantly medium-grained quartz and albite with centimetre-scale banding of a fine- to medium-grained black metallic mineral.

D4

The vein is located on the west side of the Dotson Trend, approximately 14 m north of D2. The vein is traceable with radiometrics along much of its estimated length. It is located in a densely forested part of the trend and is not traceable to the west. The vein system outcrops twice. From tracing its radiometric signature, it is estimated to be continuous over a length of 54 m. The vein's thickness is 5 cm. The observed strike is approximately 100°.

The vein averages a maximum reading of approximately 1,300 cps. It is coarse-grained quartz and plagioclase with smaller crystals of an earthy pink mineral. Banding of a fine-to-medium-grained black metallic mineral is concentrated along the edges.

D5

The vein system is located on the west side of the Dotson Trend, approximately 7 m north of D1. The vein system is located in a forested area with poor exposure, but there is sufficient exposure and radiometric signature along the strike of the vein to trace it. It is not traceable to the west, and the radiometric signature is lost underneath a cat road to the east. The vein system at D7 is on strike with D5 and could be the same vein. The vein system comprises one to four distinct veins along its strike. It is estimated to continue over a length of 73 m; 4 stations were recorded. The average thickness is 26 cm, with a maximum observed thickness of 45 cm and a minimum observed thickness of 17 cm. The strike is approximately 110°.

The vein averages a maximum reading of approximately 2,300 cps. Typically, it has a coarse-grained core of less than 1 cm quartz and plagioclase crystals grading to medium grain size near the edges. A medium-grained black metallic mineral banding along the edges is wavy and irregular in some outcrops.

D5

The vein system is located in the west-central area of the Dotson Trend, approximately 8 m north of D8 and 40 m north of the DMW. The east side of the vein system is located in an unforested area with good exposure, and its west side is covered in overburden. The vein's radiometric signature is lost under a cat road to the west. To the east it is also lost under overburden in a steep, forested area by Camp Creek. The vein system comprises one to eight distinct veins along its strike. It is estimated to continue over a length of 525 m; 10 stations were recorded. The average thickness is 17 cm, with a maximum observed thickness of 39 cm and a minimum observed thickness of 9 cm. The strike is approximately 110°.

The vein averages a maximum reading of approximately 1,500 cps. Typically, it has a coarse-grained core of less than 1 cm quartz and plagioclase crystals grading to a medium grain size near the edges. Dark green, medium-grained aegirine selvage is observed in one outcrop. In others the selvage is dark, dominantly composed of a black, metallic mineral, with wavy contacts.

D6

The vein system is located on the west side of the Dotson Trend, approximately 14 m north of D7 and 60 m west of the road. The vein system is located in a forested area with poor exposure, and it is not traceable to the east or west. The vein system comprises one vein observed at one station in outcrop. It is possible that it continues along strike, but was not observed. It is also possible that the vein folds or pinches out. The thickness is 10 cm. The strike was not recorded due to poor visibility in outcrop.

The vein's maximum reading is 2,300 cps. The core of the vein is coarse-grained quartz and plagioclase, grading out to finer grain sizes close to the edges, with more inclusions of pink and black minerals. The edges of the vein are almost entirely composed of a fine- to medium-grained black mineral.

D7

The vein system is located on the west-central area of the Dotson Trend, approximately 10 m south of D8 and 15 m north of the DMW. The vein system is located in a forested area with poor exposure, but there is sufficient exposure and radiometric signature along the strike of the vein to trace it. The vein's radiometric signature is lost underneath overburden to both the east and west. The vein system comprises one to two distinct veins along its strike. It is estimated to continue over a length of 460 m; seven stations were recorded. The average thickness is 5 cm, with a maximum observed thickness of 9 cm and a minimum observed thickness of 3 cm. The strike is approximately 283° with a vertical to sub-vertical (90 to 87°) dip.

The vein reads approximately 1,100 cps. Typically, it has a coarse-grained core of euhedral quartz and plagioclase grading to medium grain size near the edges. Medium-grained black metallic mineral banding along the edges is thin (less than 1 cm). Some zones have 0.3 mm grains of a subhedral, earthy pink mineral.

D8

The vein system is located in the west-central area of the Dotson Trend, approximately 5 m north of the DMW. The vein system is located in an open area with good exposure. The vein's radiometric signature is lost underneath overburden to both the east and west.

The vein system comprises two to ten distinct veins along its strike. It is estimated to continue over a length of 140 m; six stations were recorded. The average thickness is 32 cm, with a maximum observed thickness of 63 cm and a minimum observed thickness of 18 cm. The strike is approximately 110°.

The vein averages a maximum reading of approximately 2,500 cps. Typically, it has a coarse-grained core of quartz and plagioclase crystals grading to a finer grain size near the edges. Medium-grained subhedral-to-euhedral pink mineral grains are located in the central zone, some are lath-like. Black metallic mineral banding along edges is present in most outcrops.

D9

The vein system is located in the west-central area of the Dotson Trend, approximately 25 m south of the DMW. The vein system is located in an open area with good exposure. The vein's radiometric signature is lost underneath overburden to both the east and west.

The vein system comprises one vein observed in one outcrop. It is estimated to continue over a length of 60 m, from correlation with drill results. Its thickness is 16 cm and strike is 113°.

The vein averages a maximum reading of approximately 1,800 cps. It has a 2 cm-wide core of euhedral coarse-grained quartz, plagioclase, and pink mineral. Selvage is 3 cm-thick banding of a medium-grained black metallic mineral.

D10

The vein system is located in the west-central area of the Dotson Trend, approximately 10 m south of the DMW. The western section of the vein system is located in an open area with good exposure; the eastern section is forested and has poor exposure and is not traceable. The vein's radiometric signature is lost underneath overburden to both the east and west.

The vein system comprises one to two distinct veins along its strike. It is estimated to continue over a length of 200 m; five stations were recorded. The average thickness is 13 cm, with a maximum observed thickness of 32 cm and a minimum observed thickness of 3 cm. The strike is approximately 108°.

The vein averages a maximum reading of approximately 2,100 cps. Typically, it has a coarse-grained core of quartz and plagioclase crystals grading to smaller grain size near the edges, which are medium-grained plagioclase and a black metallic mineral, generally about 1 cm thick. In one outcrop, the core of the vein is composed dominantly of a medium-grained black metallic mineral.

D11

The vein system is located on the west side of the Dotson Trend, approximately 70 m south of the DMW and 95 m west of Camp Creek. It is located in an open area with good exposure, but it is not traceable through overburden to the east and west as it is small and the radiometric signature is low.

D12

The vein system comprises two veins observed at one station in outcrop. The total thickness is 2 cm (each vein is 1 cm thick). The strike is 107°.

The vein's maximum reading is 450 cps. The cores of the veins are coarse-grained crystalline quartz, with a fine-grained black metallic mineral selvage about 2 mm thick.

The vein system is located in the western section of the Dotson Trend, approximately 75 m south of the DMW and 95 m west of Camp Creek. It is located in an open area with good exposure, but it is not traceable through overburden to the east and west as it is small and the radiometric signature is low.

The vein system comprises one vein observed at one station in outcrop. The thickness is 1 cm and the strike is 108°.

The vein's maximum reading is 500 cps. The core of the vein is coarse-grained crystalline quartz, with fine-grained black metallic mineral selvage about 2 mm thick.

D13

The vein system is located in the middle of the Dotson Trend on the west side of the canyon in Camp Creek, approximately 55 m south of the DMW. The exposure along the creek is good but both sides are steep and heavily forested with deep overburden. The vein is not traceable to the east or west. The vein system comprises two veins observed at one station. The total thickness is 6 cm. The face on which it outcrops is vertical, so the strike is not known. The vein has a dip of 90°.

The maximum reading is on the southernmost vein and is 2,200 cps. The cores of the veins are coarse-grained crystalline quartz, with a fine-grained black metallic mineral selvage about 2 mm thick. Mineralogy is difficult to discern because of pervasive weathering in the creek.

D14

The vein system is located in the middle of the Dotson Trend on the west side of the canyon in Camp Creek, approximately 70 m south of the DMW. The exposure along the creek is good but both sides are steep and heavily forested with deep overburden. The vein is not traceable to the east or west. The vein system comprises one vein observed at one station. The thickness is 5 cm. The face on which it outcrops is vertical, so the strike is not known. The vein has a dip of 90°.

The maximum reading is on the southernmost vein and is 1,500 cps. The vein is dominantly coarse-grained crystalline quartz, with a fine-grained black metallic mineral banding about 2 mm thick in the middle of the vein. Mineralogy is difficult to discern because of pervasive weathering in the creek.

D15

The vein system is located on the central area of the Dotson Trend, approximately 20 m north of D18 and 85 m east of Camp Creek. The vein system is located in a forested area with poor exposure, many of the outcrops were dug out. The vein's radiometric signature is lost underneath overburden to both the east and west. The vein system comprises one to six distinct veins along its strike. It is estimated to continue over a length of 145 m; 5 stations were recorded. The average thickness is 9 cm, with a maximum observed thickness of 13 cm and a minimum observed thickness of 6 cm. The strike is approximately 107°.

The vein averages a maximum reading of approximately 1,500 cps. Typically, it has a coarse-grained crystalline quartz core grading to smaller grain size near the edges with a higher concentration of black metallic mineral and pink mineral. Sub-centimetre scale stringers were observed in some outcrops, and some outcrops in the western section are albitized.

DME

The vein system is located in the central area of the Dotson Trend, approximately 20 m south of D17 and 85 m east of Camp Creek. The vein system is located in a forested area with poor exposure; many of the outcrops were dug out. The vein's radiometric signature is lost underneath overburden to both the east and west. The vein system comprises one to two distinct veins along its strike. It is estimated to continue over a length of 90 m; 5 stations were recorded. The average thickness is 4 cm, with a maximum observed thickness of 5 m and a minimum observed thickness of 2 cm. The strike is approximately 108°.

The vein averages a maximum reading of approximately 1,450 cps. Typically, it is medium-grained quartz and plagioclase with a medium- to fine-grained black metallic mineral in approximately equal amounts.

D16

The vein system is located in the central area of the Dotson Trend, approximately 20 m north of the DME. The vein system is located in a lightly forested area with medium exposure. The vein's radiometric signature is lost underneath overburden to both the east and west. The vein system comprises one to five distinct veins along its strike. It is estimated to continue over a length of 235 m; 7 stations were recorded. The average thickness is 8 cm, with a maximum observed thickness of 15 m and a minimum observed thickness of 4 cm. The strike is approximately 110°.

The vein averages a maximum reading of approximately 1,600 cps. Typically, it has a coarse-grained quartz core grading to smaller grain size near the edges with a higher concentration of black metallic mineral and pink mineral. Quartz grain size is coarse but sub-centimetre, which is distinctly different than many of the veins seen in outcrop, especially in the central area of the Dotson Trend.

D17

The vein system is located in the central area of the Dotson Trend, approximately 20 m south of the DME. The area in which it is exposed is mostly forested with some good exposure. The vein's radiometric signature is lost underneath overburden to both the east and west. The vein system comprises one to two distinct veins along its strike. It is estimated to continue over a length of 170 m; 7 stations were recorded. The average thickness is 5 cm, with a maximum observed thickness of 11 m and a minimum observed thickness of 2 cm. The strike is approximately 288° with a vertical to sub-vertical (85°) dip.

The vein averages a maximum reading of approximately 1,300 cps. Typically, it has a medium-grained quartz and plagioclase core grading to smaller grain size near the edges with a higher concentration of black metallic mineral. Contacts are generally sharp and some outcrops have mm-scale inclusions of an earthy, pink mineral.

D18

The vein system is located in the central area of the Dotson Trend, approximately 30 m south of the DME. The area in which it is exposed is mostly forested, but with variable exposure. The vein's radiometric signature is lost underneath overburden to both the east and west.

The vein system comprises one to three distinct veins along its strike. It is estimated to continue over a length of 25 m; 2 stations were recorded. The total thickness of the vein system is 8 cm. The strike of the vein at station PM10-150 is 150°. The area along strike of this vein was traversed but the vein was not found, although radiometrics above overburden suggest that the vein follows a strike of approximately 105°. The vein probably folds slightly at PM10-150 and continues along the general strike of the vein system.

The vein averages a maximum reading of approximately 1,300 cps. Typically, it has a coarse-grained quartz and plagioclase core grading to smaller grain size near the edges with 1 cm-thick fine-to-medium- grained black mineral selvage.

D19

The vein system is located on the east side of the Dotson Trend, approximately 40 m south of the DME. The east side of the vein system is located in an unforested area with good exposure, and its west side is covered in overburden with fewer outcrops and is hard to trace. The vein's radiometric signature is lost under deep overburden to both the west, and it is lost under boulders on the beach to the east.

The vein system comprises one to three distinct veins along its strike. It is estimated to continue over a length of 735 m; 10 stations were recorded. The average thickness is 7 cm, with a maximum observed thickness of 15 cm and a minimum observed thickness of 3 cm. The strike is approximately 111°.

The vein averages a maximum reading of approximately 1,600 cps. Typically, it is medium-to-coarse-grained. The core is dominantly quartz with plagioclase, a black, metallic mineral and a pink, earthy mineral more concentrated towards the edges.

D19 (DME)

The vein system is located in the central area of the Dotson Trend 65 m east of Camp Creek. The exposure in the area is poor and is heavily forested with deep overburden. The vein is not traceable to the east or west. The vein system comprises one vein observed at one station. The thickness is 7 cm. Strike and dip were not observed.

The maximum reading is 1,300 cps. The vein is dominantly coarse-grained crystalline quartz, with a fine- grained black metallic mineral selvage less than 1 cm thick.

D20a

The vein system is located on the east side of the Dotson Trend, approximately 65 m south of the DME and 165 m east of Camp Creek. It is located in an open area with good exposure, but it is not traceable through overburden to the east and west as it is small and the radiometric signature is low. The vein system comprises two veins 22 cm apart observed at one station in outcrop. The thicknesses of the veins are 0.2 and 0.5 cm and the strike is 115°.

The vein's maximum reading is 300 cps. The core of the vein is coarse-grained crystalline quartz, with a fine-grained black metallic mineral selvage about 2 mm thick.

D20

The vein system is located on the central-east area of the Dotson Trend. Most of the vein system is well exposed with abundant outcrops. However, the western side of the system is under overburden. The vein's radiometric signature is lost under overburden both to the east and to the west. The vein system comprises one to seven distinct veins along its strike. It is estimated to continue over a length of 210 m; 8 stations were recorded. The average thickness is 24 cm, with a maximum observed thickness of 44 cm and a minimum observed thickness of 9 cm. The strike is approximately 107°.

The vein averages a maximum reading of approximately 3,100 cps. Typically, it has a coarse-grained core of 1 to 2 cm quartz crystals in a finer-grained quartz-albite groundmass. Zones of quartz and plagioclase with pink and black mineral inclusions are typically located on the edge of the core of the vein, with centimetre scale selvage of a black, metallic mineral.

D21

The vein system is located on the east side of the Dotson Trend, approximately 80 m south of the DME and 150 m east of Camp Creek. It is located in an open area with good exposure, but it is not traceable through overburden to the east and west as it is small and the radiometric signature is low. The vein system comprises one vein observed at one station in outcrop. The thickness of the vein is 1 cm and the strike is 115°.

The vein's maximum reading is 400 cps. The vein is coarse-grained crystalline quartz with minor plagioclase.

D22

The vein system is located in the east area of the Dotson Trend, approximately 7 m north of D26 and 55 m north of D23. The vein system is located in a forested area with medium exposure. The vein's radiometric signature is lost underneath overburden to both the east and west.

The vein system comprises one vein along its strike. It is estimated to continue over a length of 140 m; 3 stations were recorded. The average thickness is 20 cm, with a maximum observed thickness of 30 cm and a minimum observed thickness of 11 cm. The strike is approximately 109°.

The vein averages a maximum reading of approximately 5,000 cps. It has a coarse-grained, dominantly quartz and plagioclase core, with minor amounts of a black, metallic mineral and a pink, earthy mineral. Fine-grained centimetre-scale black banding is located in two outcrops, on contacts and within the middle part of the vein.

D23

The vein system is located on the east side of the Dotson Trend, approximately 4 m south of D26 and 55 m north of D23. It is located in a forested area and is not traceable through overburden to the east and west as it is small and the radiometric signature is low.

The vein system comprises two veins 1.5 cm apart observed at 1 station in outcrop. The thicknesses of the veins are 1 and 2 cm and the strike is 112°.

The vein's maximum reading is 800 cps. The vein is medium-grained, dominantly quartz and plagioclase with some pink inclusions. It has a millimetre-scale fine-grained black selvage.

D24

The vein was not observed in outcrop. It is projected to surface from observations in drill section D26 (DME). The vein system is located in the east area of the Dotson Trend, approximately 50 m north of D23. The vein system is located in a forested area with medium exposure. The vein's radiometric signature is lost underneath overburden to both the east and west. The eastern extent and location of the vein was estimated from correlation with drill sections.

The vein system comprises one vein along its strike observed at four stations. It is estimated to continue over a length of 140 m; 4 stations were recorded. The average thickness is 13 cm, with a maximum observed thickness of 30 m and a minimum observed thickness of 4 cm. The strike is approximately 293° with a vertical to sub-vertical dip (87°).

The vein averages a maximum reading of approximately 2,500 cps. It is medium-grained, dominantly quartz and plagioclase with minor amounts of a black, metallic mineral and a pink, earthy mineral. In some spots, the vein has a 1 cm-thick zone in the core of coarse-grained quartz.

8.0 DEPOSIT TYPES

Bokan Mountain hosts a new significant REE mineralization resource. The host granitic-syenitic intrusion was emplaced at shallow depths forming a ring dike complex. It is one of several known felsic intrusive centres in southeastern Alaska which contain traces of radioactive or rare earth elements (Eakins 1975). The host granitic rocks are peralkaline, i.e. they are undersaturated in alumina. These magmatic deposits are related to an enrichment of certain incompatible trace elements, particularly in REEs and high field strength elements (HFSEs) during late stages of emplacement and crystallization of the magma (Bohse et al. 1974). The petrological and geochemical characteristics of the peralkaline granites suggest that they developed from a highly differentiated magma. The concentration of the REE and HFSE appears to increase proportionally with increasing differentiation. Thompson et al. (1980, 1982) and Thompson (1988, 1997) invoked the following multi-stage model for the origin of the Ross Adams deposit of the Bokan Mountain complex:

- emplacement of a sodium-rich oxidized magma
- low initial calcium and titanium contents prevented the formation of early accessory minerals
- development of a separate volatile phase with HFSE and REE content
- rapid degassing of the magma chamber, resulting in a silica-saturated magma and a volatile phase emplaced in a zone of structural weakness
- precipitation of the REE and HFSE minerals.

Bokan Mountain has few characteristics in common with well-studied rare earth deposits and mines and therefore the following information is not necessarily indicative of mineralization on the Property and in the Dotson Zone. NI 43-101 requires the presentation of a deposit model that may be used to aid in the economic assessment of the Property, and therefore data on known deposits are provided solely for NI 43-101 compliance and are not intended to suggest that such deposits will be found at Bokan Mountain.

The REEs and HFSEs are both too large and too highly charged to participate in common fractional crystallization phases. Therefore, during differentiation/fractional crystallization, they are concentrated increasingly in late siliceous melts. It is understood that the peralkaline granites, such as those found in the Bokan Intrusive Complex, are products of differentiation of partial melts from an undepleted, metasomatically enriched mantle (Marks et al. 2003; Halama et al. 2004).

There is little agreement, however, as to why the REEs and yttrium are concentrated in these intrusions

Examples of such deposits enriched in zirconium, niobium, yttrium and REEs include Strange Lake in Québec-Labrador, Canada, Ilimaussaq in Greenland, the Amis Complex in Namibia, the Pilanesberg Complex in South Africa, the Khaldzan-Buregtey Granitoid Complex in Mongolia, and the Lovozero Massif in Russia, which is the largest perkaline intrusion in the world, but is preferentially enriched in LREEs (Sheard 2010).

According to Warner and Barker (1989), the REE tenor at Bokan Mountain generally increases from shear- and fracture-hosted mineralization to mineralized pegmatites to dyke-hosted occurrences. Dykes and veins host resources of beryllium, niobium-tantalum, zirconium, yttrium and REE. Particular economic significance is attached to the observation that the Bokan Mountain deposits are relatively enriched in the HREEs (also referred to as the yttrium subgroup, elements 64-71, gadolinium to lutetium and yttrium).

A number of minerals have been identified as the sources of rare metals such as yttrium and REE at Bokan Mountain, e.g. iimoriite-(Y), synchysite-(Y), kainosite, apatite, bastnaesite, fergusonite, monazite, samarskite, euxenite and gadolinite (Keyser 2010). Positive recognition of some of the rare minerals, and *in situ* analysis of their chemistry requires polished samples and use of electron microscope/microprobe (scanning electron microscope (SEM)/entangled-photon microscopy (EPM)) technology. Some of the coarser/more abundant phases could be identified in the labelled rock off-cuts using modern models of SEM with energy-dispersive analyzers and the ability to image grains without prior carbon coating. Some of the minerals are very rare, e.g. iimoriite (Fleischer 1973; Marty 2004), which is known only from a pegmatite in Japan, a REE-enriched talc-chlorite deposit in the French Pyrenees, and the Bokan Mountain complex.

Marginal quartz-albite dykes radiate out for more than 6 km from the Bokan Mountain peralkaline granite, and contain additional rare REE host phases in exotic mineral assemblages. The Dotson Zone is a zone of emplacement of dykes and veins enriched in REEs and HFSEs. The REE occur in thalenite and its alteration product (tengerite), bastnaesite, synchysite, xenotime and monazite. Niobium occurs in euxenite-polycrase minerals and in columbite. Other minerals include zircon, aegirine, barite, galena, native silver and sphalerite. Near the complex, the dykes average up to 2 m in width, are pegmatitic, and are relatively rich in HREE. Further from the complex, the dykes are thinner but more widespread and finer-grained, with high niobium, yttrium, zirconium and LREEs (Warner and Mardock 1987).

The emphasis at Bokan Mountian is on the REEs but there are no suitable conceptual models at present which completely explain the mineralization observed on the Property.

9.0 EXPLORATION

Between 2007 and 2010, Ucore conducted airborne geophysical surveys, prospecting, geological mapping, trenching, channel sampling and diamond drilling. These programs and the pertinent results are summarized in the following two sections.

9.1 AIRBORNE GEOPHYSICS

Precision GeoSurveys Inc. (Precision) of Vancouver, BC completed airborne total magnetic field and radiometric surveys over the Property during 2009 and 2010. The objective of these surveys was to map radiogenic rocks on the Property with the radiometric survey in order to directly locate uranium and rare earth mineralization, and to map structure and stratigraphy indirectly with the total magnetic field survey. The surveys are described in Precision (2010).

9.2 GROUND GEOPHYSICS

Ground total magnetic field and radiometric surveys were performed by employees of Aurora on the Dotson Zones during 2009 to map these zones prior to drilling. An induced polarization (IP) survey was run by Aurora at Sunday Lake to attempt to discern controls on mineralization intersected in drillholes at this site.

10.0 DRILLING

The drilling discussed in this section was conducted in the summers of 2008, 2009, and 2010. A total of 5,774 m of core have been drilled to date in the Dotson Zone. In 2008, 196 m of core were drilled in 4 holes. In 2009, 2,364 m of core were drilled in 20 holes. In 2010, 3,214 m of core were drilled in 13 holes.

10.1 2008 DRILLING

In 2008, a total of 196 m were drilled in 4 holes in the Dotson Zone. The drill contractor was DDC, The Directional Drilling Company, and the QPs supervising the drilling were Mike Power, P.Geoph. P.Geol. M.Sc. and Jim Robinson, F.G.A.C. P.Geol. of Aurora. Two drills were employed in 2008; a vintage Winkie drill for shallower holes on smaller drill pads, and a DMW-45 for longer holes. Drilling operations were conducted between May 23 and August 21, 2008. All drill moves and drill support was provided by a Bell 206L Long Ranger helicopter supplied by Pathfinder Aviation. A summary of the 2008 diamond drilling can be found in Table 10.1.

10.2 2009 DRILLING

In the 2009 exploration season, the focus of exploration shifted from uranium to rare earths. The confirmation of elevated rare earth values on the Dotson structure expanded the scope of the Project by several kilometres southeast.

The drill contractor for the 2009 season was More Core Diamond Drilling Service, Ltd. (More Core) of Hyder, Alaska. More Core used a Hydracore 2000 to drill NQ core. The QPs for the program were Mike Power, P.Geoph., P.Geol., M.Sc., Jim Robinson, F.G.A.C. P.Geol., Gary Vivian, P.Geol., M.S., Dave White, P.Geol., and Louis Covello, all of Aurora. Drilling operations were conducted between September 8 and October 24, 2009. This proved to be rather too late in the season as productivity decreased due to severe weather conditions and reduced daylight hours later in the season.

Drill production totalled 2,384 m in 20 holes in the Dotson Zone. The drillhole information is detailed in Table 10.1.

A plan map showing all drill collars and traces for the drilling completed on the Dotson Zone up to the end of 2010 is located in Section 10.3.

10.3 2010 Drilling

The drill program in 2010 in the Dotson Zone consisted of 13 diamond drillholes totalling 3,214 m. The same drill contractor and drill as in the 2009 program was used. The QPs were Mike Power and Jim Robinson. Figure 10.1 is a plot of drillhole traces together with TREO values in bar graph format. Figure 10.2 and Figure 10.3 are long sections of the west and east halves of the Dotson Zone facing north. A drillhole summary table (Table 10.1) can be found at the end of this section.

TETRA TECH

ucore. RARE METALS

Figure 10.1 Plan Map of the Dotson Zone Showing Drillhole Collars and Traces for Holes Drilled in 2008, 2009, and 2010

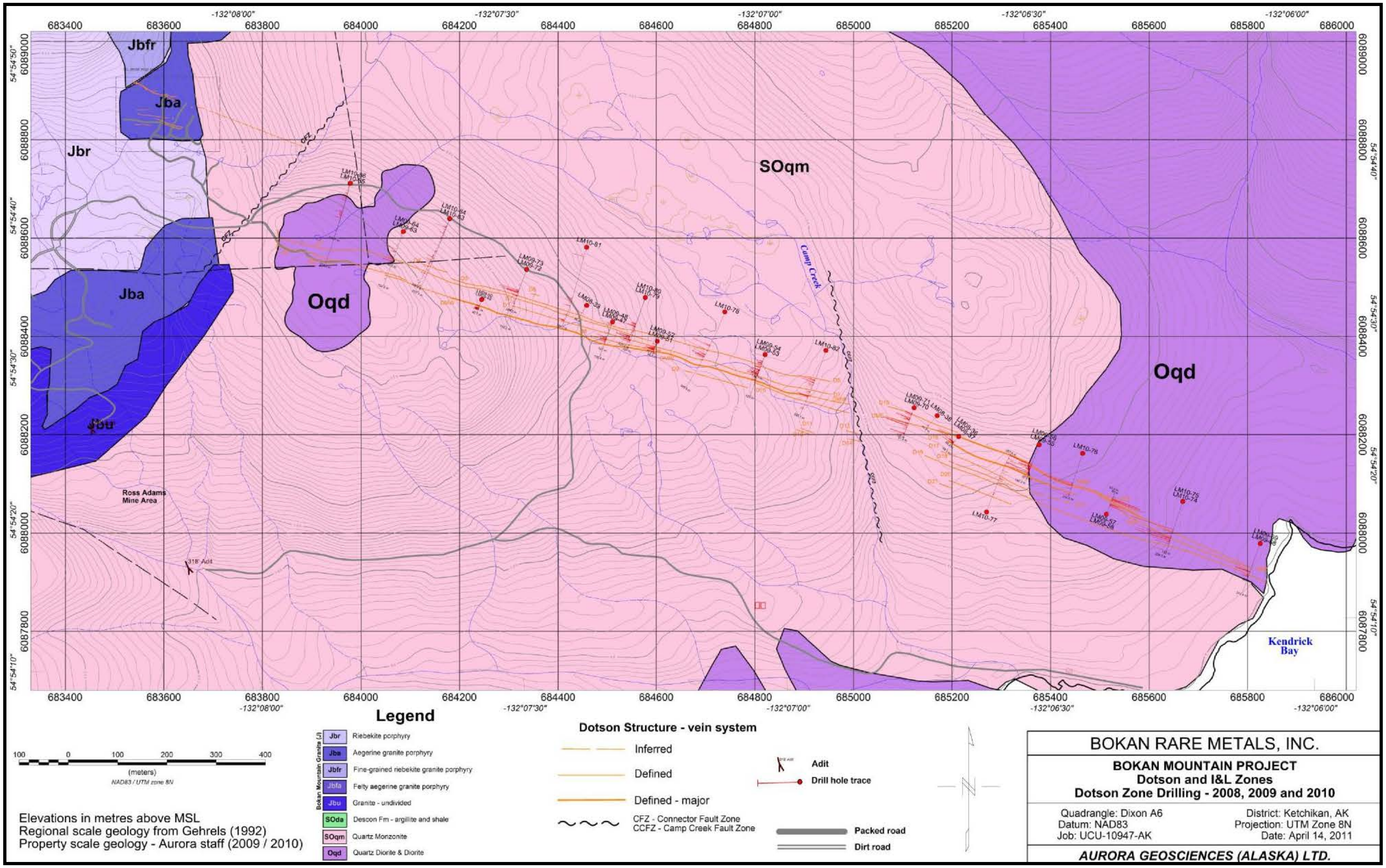


Figure 10.2 Long Section of Dotson Zone Looking North – West Half

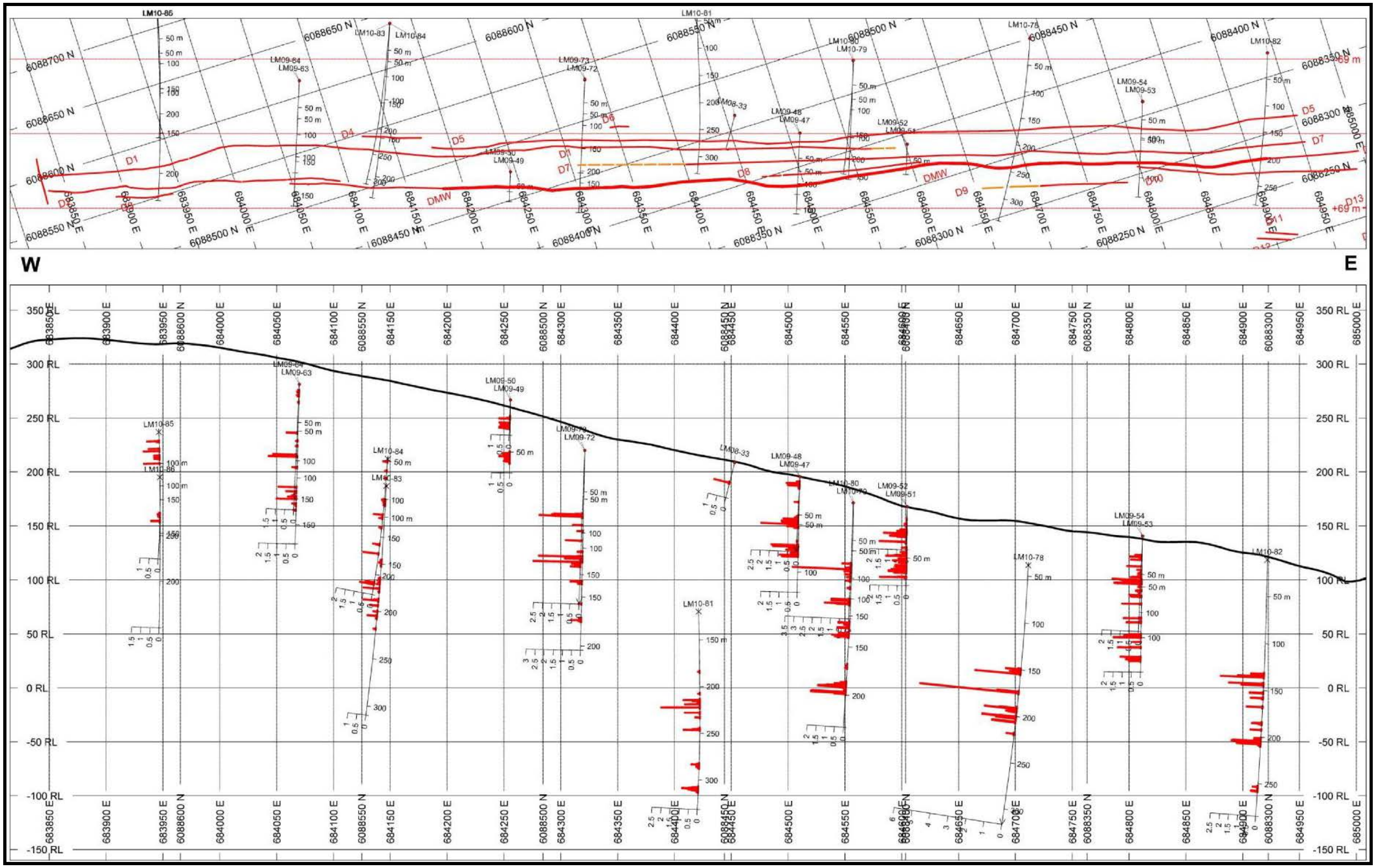


Figure 10.3 Long Section of Dotzon Zone Looking North – East Half

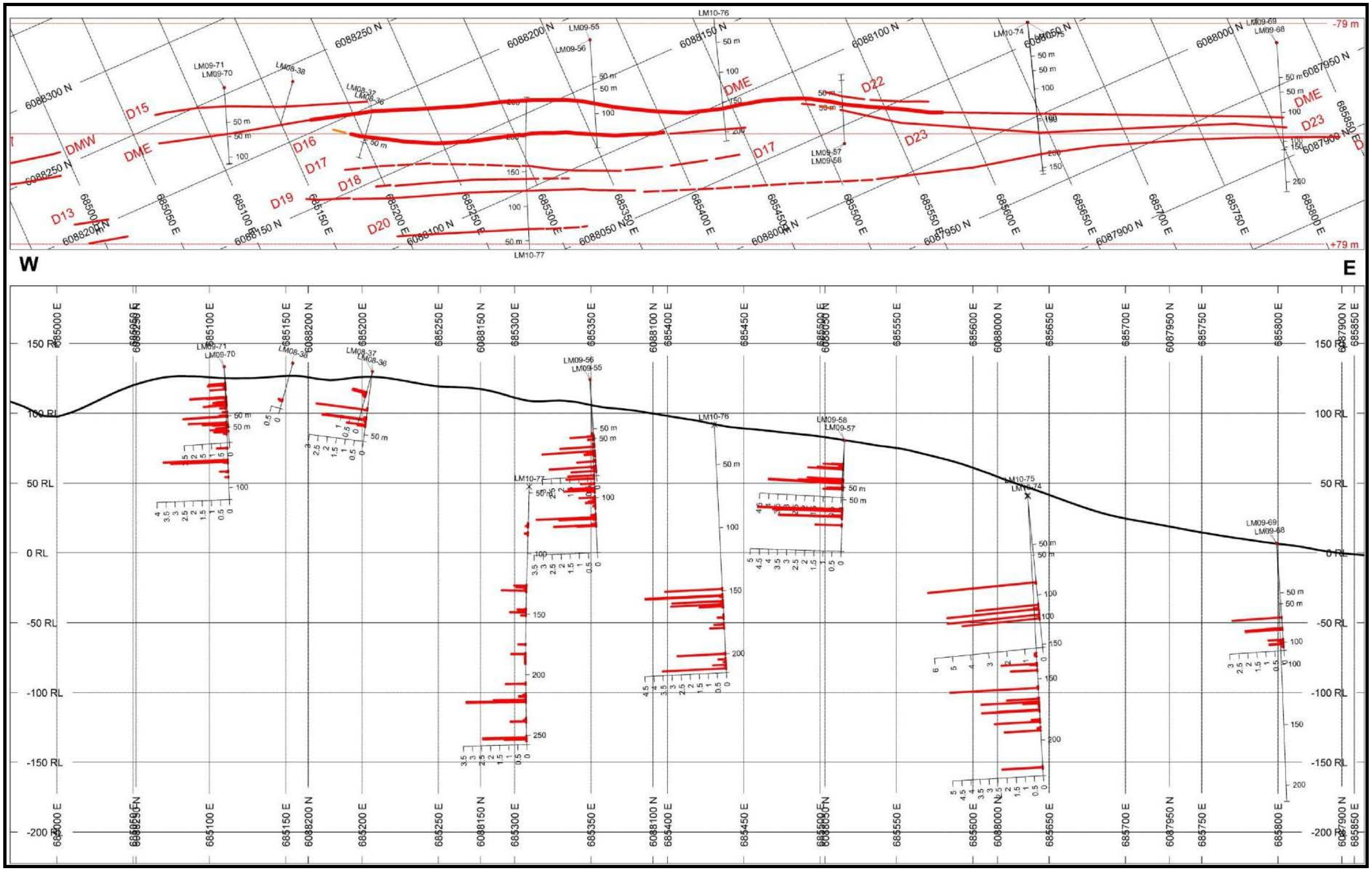


Table 10.1 Summary of Diamond Drilling for the Dotson Zones, 2008 to 2010

Hole No.	Zone Pad	Pad	Provisional			DGPS-Final			Azimuth (°)	Inclination (°)	TD (ft)	TD (m)	Core Size	Start	Finish
			UTME	UTMN	Z (masl)	UTME	UTMN	Z (masl)							
2008 Diamond Drilling															
LM08-33	Dotson	A-29	684457.9	6088461.6	209.0	-	-	-	240	-54	151.00	46.03	Winkie	18-Jul-08	23-Jul-08
LM08-36	Dotson	A-30	685213.8	6088195.4	130.1	-	-	-	220	-45	161.00	49.07	Winkie	23-Jul-08	27-Jul-08
LM08-37	Dotson	A-30	685213.8	6088195.4	130.1	-	-	-	220	-65	181.00	55.17	Winkie	27-Jul-08	28-Jul-08
LM08-38	Dotson	A-31	685170.2	6088237.9	136.0	-	-	-	220	-45	150.50	45.87	Winkie	28-Jul-08	1-Aug-08
2009 Diamond Drilling															
LM09-47	Dotson	09-01	684512	6088433	194	684509.9	6088430.1	195.8	200	-45	347.01	105.77	NQ	8-Sep-09	10-Sep-09
LM09-48	Dotson	09-01	684512	6088433	194	684509.9	6088430.1	195.8	200	-62	397.01	121.01	NQ	10-Sep-09	11-Sep-09
LM09-49	Dotson	09-02	684245	6088473	266	684244.5	6088474.4	267.0	200	-50	139.99	42.67	NQ	12-Sep-09	13-Sep-09
LM09-50	Dotson	09-02	684245	6088473	266	684244.5	6088474.4	267.0	200	-75	229.99	70.10	NQ	13-Sep-09	14-Sep-09
LM09-51	Dotson	09-03	684602	6088393	168	684601.9	6088395.5	168.2	200	-55	160.01	48.77	NQ	14-Sep-09	15-Sep-09
LM09-52	Dotson	09-03	684602	6088393	168	684601.9	6088395.5	168.2	200	-73	250.00	76.20	NQ	15-Sep-09	15-Sep-09
LM09-53	Dotson	09-04	684820	6088361	140	684820.8	6088365.0	140.7	200	-45	410.01	124.97	NQ	16-Sep-09	18-Sep-09
LM09-54	Dotson	09-04	684820	6088361	140	684820.8	6088365.0	140.7	200	-70	439.99	134.11	NQ	18-Sep-09	19-Sep-09
LM09-55	Dotson	09-05	685366	6088170	125	685377.6	6088182.2	124.7	220	-45	320.01	97.54	NQ	20-Sep-09	21-Sep-09
LM09-56	Dotson	09-05	685366	6088170	125	685377.6	6088182.2	124.7	220	-58	479.99	146.30	NQ	22-Sep-09	23-Sep-09
LM09-57	Dotson	09-06	685512	6088041	117	685513.6	6088030.9	79.3	21	-42	200.00	60.96	NQ	23-Sep-09	24-Sep-09
LM09-58	Dotson	09-06	685512	6088041	117	685513.6	6088030.9	79.3	21	-58	307.09	93.60	NQ	24-Sep-09	25-Sep-09
LM09-63	Dotson	09-09	684092	6088604	288	684084.1	6088615.3	280.1	200	-45	540.12	164.63	NQ	1-Oct-09	3-Oct-09
LM09-64	Dotson	09-09	684092	6088604	288	684084.1	6088615.3	280.1	200	-60	460.01	140.21	NQ	3-Oct-09	4-Oct-09
LM09-68	Dotson	09-11	685828	6087981	15	685828.2	6087981.0	6.9	200	-43	355.74	108.43	NQ	10-Oct-09	11-Oct-09
LM09-69	Dotson	09-11	685828	6087981	15	685828.2	6087981.0	6.9	200	-60	700.16	213.41	NQ/BQ	11-Oct-09	12-Oct-09
LM09-70	Dotson	09-12	685121	6088251	132	685121.8	6088256.7	136.2	200	-45	300.07	91.46	NQ	14-Oct-09	16-Oct-09
LM09-71	Dotson	09-12	685121	6088251	132	685121.8	6088256.7	136.2	200	-60	420.01	128.02	NQ	16-Oct-09	17-Oct-09
LM09-72	Dotson	09-13	684336	6088536	220	684342.0	6088530.0	219.7	200	-65	670.01	204.22	NQ	17-Oct-09	21-Oct-09
LM09-73	Dotson	09-13	684336	6088536	220	684342.0	6088530.0	219.7	200	-50	629.99	192.02	NQ	21-Oct-09	24-Oct-09

table continues…

Hole No.	Zone Pad	Pad	Provisional			DGPS-Final			Azimuth (°)	Inclination (°)	TD (ft)	TD (m)	Core Size	Start	Finish
			UTME	UTMN	Z (masl)	UTME	UTMN	Z (masl)							
2010 Diamond Drilling															
LM10-74	Dotson	10-03	685680	6088065	40	685668.1	6088064.2	41.9	198	-45	502.00	153.05	NQ2	3-Jul-10	7-Jul-10
LM10-75	Dotson	10-03	685680	6088065	40	685668.1	6088064.2	41.9	198	-60	752.00	229.27	NQ2	7-Jul-10	11-Jul-10
LM10-76	Dotson	10-04	685470	6088160	108	685465.1	6088161.8	108.3	198	-64	705.00	214.94	NQ2	11-Jul-10	17-Jul-10
LM10-77	Dotson	10-05	685282	6088040	85	685270.1	6088042.7	86.5	18	-60	845.00	257.62	NQ2	18-Jul-10	22-Jul-10
LM10-78	Dotson	10-06	684740	6088450	147	684739.0	6088449.7	147.0	198	-60	1115.00	339.94	NQ2	23-Jul-10	30-Jul-10
LM10-79	Dotson	10-07	684580	6088475	172	684577.5	6088478.9	171.4	198	-45	505.00	153.96	NQ2	30-Jul-10	1-Aug-10
LM10-80	Dotson	10-07	684580	6088475	172	684577.5	6088478.9	171.4	198	-63	765.00	233.23	NQ2	1-Aug-10	7-Aug-10
LM10-81	Dotson	10-08	684460	6088580	175	684457.8	6088581.3	173.9	198	-60	1088.00	331.71	NQ2	7-Aug-10	13-Aug-10
LM10-82	Dotson	10-02	684950	6088370	125	684944.1	6088371.4	128.4	198	-60	935.00	285.06	NQ2	14-Aug-10	18-Aug-10
LM10-83	Dotson	10-09	684210	6088630	245	684179.9	6088639.5	245.2	198	-60	1005.00	306.40	NQ2	18-Aug-10	24-Aug-10
LM10-84	Dotson	10-09	684210	6088630	245	684179.9	6088639.5	245.2	198	-45	745.00	227.13	NQ2	24-Aug-10	28-Aug-10
LM10-85	Dotson	10-01	684000	6088715	276	683978.0	6088711.5	276.6	198	-45	765.00	233.23	NQ2	29-Aug-10	2-Sep-10
LM10-86	Dotson	10-01	684000	6088715	276	683978.5	6088712.0	276.6	198	-60	815.00	248.48	NQ2	2-Sep-10	5-Sep-10

Note: All depths in metres, locations NAD83 Zone 8N UTM.
DGPS elevations interpolated from digital topography to agree with DT for elevation reconciliation.
Azimuths shown are UTM azimuths (Mag decl 18).

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Aurora conducted surface selected sampling, and channel and drill core sampling from 2008 through 2010. The sampling was conducted by geologists working under the supervision of a Project Manager who was a QP as defined by NI 43-101. Appropriate quality assurance/quality control (QA/QC), handling, and chain of custody procedures were followed throughout the sampling programs. Drillholes and trench logs were compiled in separate spreadsheet files and include all data collected. Separate digital assay and location databases were also compiled to consolidate the analytical and topographic survey information for verification, security and analysis.

This section describes the sample methods and approach, sample handling, security and analytical procedures employed in 2008 through 2010 inclusive.

11.1 SAMPLE METHODS AND APPROACH

11.1.1 DRILLHOLE SAMPLING

Aurora conducted drill core sampling in 2008, 2009 and 2010. Drill core handling and sampling consisted of the following procedures:

- Driller's footage blocks were converted from feet to metres and core boxes were labelled with metal tags recording the hole and interval contained therein.
- Core was reassembled followed by calculation of rock quality determination (RQD) and recovery from the measured reconstructed core.
- Physical properties of the core including magnetic susceptibility and radiometrics were recorded during the 2008 through 2010 programs. For each hole during the 2010 program, density measurements were taken on approximately six specimens of mineralized and wall rock. These were taken with a TMI MD-300S digital density meter capable of recording density to +0.001 g/cm^3 using the water immersion method. A sample of core from 2 to 6 cm long was weighed in air and then again while immersed in water. The specific gravity was calculated by the instrument and recorded in the digital drill log.
- Core logging, including petrology, structure, alteration, and mineralization, were recorded with logging intervals set at any change in lithology,

alteration, structure or mineralization. Structural measurements were scribed on the core in china marker. In 2009 and 2010, a Niton x-ray fluorescence gun was used to make in situ measurements of REE concentrations on the core faces. Anomalous measurements were scribed on the core in china marker. All data was entered into a standardized logging spreadsheet for each hole.

- Sample intervals were laid out according to the following rules and procedures:
  - No sample interval was longer than 1.0 m.
  - Sample intervals did not cross logging intervals.
  - Shoulder samples in barren rock were taken above and below each sampled mineralized interval.
  - All samples were recorded in three-tag sample books. One tag was submitted to the lab with the sample, a second was stapled in the core box at the top of the sampled interval and the third was left in the sample book. The hole number, sample interval and type of sample were written in the sample book.
  - A blank, duplicate and standard sample was inserted into every run of 20 samples. Sample tags for all samples were stapled in sequence in the core box.
- Core was photographed wet during 2008 through 2010. Lithological logging and sample intervals and the location of density samples were marked with a standard colored flagging protocol.
- Core intervals laid out for sampling were split lengthwise and one half of the split core was taken as a sample. Duplicates were taken by quartering the half-core sample and submitting the separate quarters as individual samples.
- Core boxes were banded, separated into sampled and barren batches and shipped to Ketchikan for storage.
- Logging was performed digitally using a single standardized spreadsheet incorporating all measurements for each hole. Drill core and channel sample information was entered into Aurora's secured assay database.

11.1.2 Channel Sampling

Shallow trenches were excavated by hand and the exposed rock was channel sampled in 2009 and 2010. Trenching and sampling were conducted according to the following specifications:

- Location Datum: NAD83 Zone 8N UTM metric coordinates.
- Trench Location: The origin of each trench was recorded with differentially corrected global positioning system (GPS) coordinates. Each location reading was averaged at least 30 times using the Annette Island beacon.

- Channel Sampling: Each trench interval was excavated to bedrock and cleaned. A metric survey tape was laid out along the trench, commencing at the origin and sample intervals are described as distance from the origin (as in a diamond drillhole). Some intervals could not be excavated because of the depth of overburden. Parallel diamond saw cuts were made 5 to 8 cm apart along the exposures. Samples were moiled from the rock between the cuts, collected in a sheet and transferred to sample bags
- Station Records: Interval in meters from the trench origin, lithology, structure, alteration, economic mineralization, and sample intervals were recorded for each trench. In 2010, photographs were also taken of the trenches prior to sampling.
- Sample Marking: Sample intervals were marked with metal tags at the start of each interval, inserted into the remnant diamond saw cuts.

11.1.3 RECONNAISSANCE SAMPLING

Aurora and other consulting geologists took selected rock specimens during prospecting and mapping to determine the tenor of mineralization. From 2008 through 2009, these samples were submitted in separate shipments to the laboratory that analyzed the core or trench samples. In 2010, these samples were submitted to the Stewart Group laboratory in Stewart, BC as well as to Actlabs. Blanks, duplicates and standards were not inserted into the surface rock sample shipments; these samples were not included in Aurora's assay database and they were not used in the resource calculation.

11.1.4 ORIENTATION GEOCHEM SURVEY – 2009

The orientation geochem survey of 2009 consisted of orientation soil and stream sediment concentrate sampling over the Dotson Zone and reconnaissance soil and stream sediment concentrate sampling over peripheral targets. The geochemical surveys were conducted using the following specifications:

- Soil samples were collected from pits dug with mattocks. The C-horizon (mineral soil) was sampled wherever present. Stream sediment samples were collected from the centre of the creek bed behind rocks or other catchments. Approximately 2.3 kg of sample were collected in plastic sample bags. Samples were neither weighed nor volume sized.
- Samples were described including color, source, slope aspect and the presence of water.
- Samples were panned by hand in nearby streams and reduced to approximately 200 g of concentrate. The concentrate was stored in cloth sample bags.
- Samples were air-dried.

- Samples were dry-sieved to a sand (125 to 250 µm) and silt (-125 µm) fraction. Samples were transferred to Niton sample holders (pucks) for analysis.
- Samples were analyzed using the Niton XRF analyzer, taking three repeat measurements and rotating the puck between measurements. Final results for each sample are an average of the three measurements. Only measurements for lanthanum, cerium, vanadium, and zirconium are plotted as the Niton results are accurate only for this suite of elements.

11.2 2008 SAMPLING

Core, channel and surface rock samples collected during 2008 program were analyzed by Actlabs in Ancaster, Ontario. At the time of the program, this lab was certified to International Organization for Standardization (ISO) 17025 and to CAN-P-1579 by the Standards Council of Canada (SCC).

Core and channel samples were shipped by air charter and commercial air carrier to the laboratory in sealed containers. Sample shipment manifests accompanied each shipment and were also emailed to the laboratory when the shipment was consigned to the first carrier. Sample shipments were checked on arrival at the laboratory to ensure that all samples were received. Prior to shipment from the field, blanks, duplicates and a uranium standard were inserted into the sample stream with a frequency of one each per batch of 20 samples.

Drill core and rock samples were analyzed using Actlabs' preparation procedure RX-1 and analytical procedure 4B2. The preparation and analytical procedures were as follows:

- Samples were weighed on arrival.
- Samples were then crushed to the extent that 75% of the sample material passed through a 2 mm mesh.
- The fine crush was split with a riffle splitter to extract a 250 g sample.
- The subsample was pulverized with a puck pulverizer using a hardened steel facing to the point where 95% of the sample passed through a 105 µm mesh.
- Uranium concentration was determined with the delay neutron counter technique.
- A subsample was subjected to lithium metaborate/tetraborate fusion followed by weak nitric acid digestion and inductively coupled plasma-mass spectrometry (ICP-MS) to determine the concentration of 52 elements including all REEs, thorium, niobium, zircon and uranium.
- Sample results were delivered in digital spread sheet compilation and in signed digital (.pdf) format.

The sample pulps and coarse rejects from the 2008 program are stored at Ucore's facility in Ketchikan, Alaska.

11.3 2009 SAMPLING

Core, channel and surface rock samples collected during 2009 program were analyzed by Eco-tech Laboratories Ltd (Eco-tech) in Kamloops, BC. At the time of the of the 2009 program, this lab was certified to ISO 9000:2008. Samples were prepared at a satellite preparation lab in Stewart, BC and the pulps were sent to Kamloops for final analysis.

Core and channel samples were shipped by air and marine commercial carrier to the laboratory in sealed containers. Sample shipment manifests accompanied each shipment and were also emailed to the laboratory when the shipment was consigned to the first carrier. Sample shipments were checked on arrival at the laboratory to ensure that all samples were received. Prior to shipment from the field, blanks and duplicates were inserted into the sample stream with a frequency of one each per batch of 20 samples. A standard was added to the duplicates and blanks during the latter half of the drill program.

Drill core, channel samples and surface rock samples were analyzed using Eco-tech's preparation procedure BRC-11c and analytical procedure BWRMS-11. The preparation and analytical procedures were as follows:

- Samples were weighed on arrival.
- Samples were then crushed to the extent that 70% of the sample material passed through a 2 mm mesh.
- The fine crush was split with a riffle splitter to extract a 250 g sample.
- The subsample was pulverized on a ring mill pulverizer to the point where 95% of the sample passed through a 100 µm mesh.
- A subsample was subjected to lithium metaborate fusion followed by hydrogen fluoride and nitric acid digestion. The digestion was further cut with hydrogen cyanide (5%) and nitric acid (2%) and analyzed with ICP-MS to determine the concentration of 44 elements including all REEs, thorium, niobium, zircon and uranium.
- Over-limit assays were re-analyzed by repeating the sampling and lithium metaborate fusion followed by digestion, an additional dilution and ICP-MS.
- Sample results were delivered in digital spread sheet compilation and in signed digital (.pdf) format.

The sample pulps and coarse rejects from the 2009 program are stored at Ucore's warehouse in Ketchikan, Alaska.

11.4 2010 SAMPLING

Core and channel samples collected during the 2010 program were analyzed by Actlabs. Samples were shipped to their preparation laboratory in Stewart, BC and the analysis was conducted at their facility in Ancaster, Ontario. At the time of the program, this lab was certified to ISO 17025 and to CAN-P-1579 by the SCC.

Core and channel samples were shipped by air and marine commercial carrier to the laboratory in sealed containers. Sample shipment manifests accompanied each shipment and were also emailed to the laboratory when the shipment was consigned to the first carrier. Sample shipments were checked on arrival at the laboratory to ensure that all samples were received. Prior to shipment from the field, blanks, duplicates and one of two rare earth standards described in Section 12.0 were inserted into the sample stream with a frequency of one each per batch of 20 samples.

Drill core and rock samples were analyzed using Actlabs' preparation procedure RX-1 and analytical procedure Code 8 - REE. The preparation and analytical procedures were as follows:

- Samples were weighed on arrival.
- Samples were then crushed to the extent that 75% of the sample material passed through a 2 mm mesh.
- The fine crush was split with a riffle splitter to extract a 250 g sample.
- The subsample was pulverized with a puck pulverizer using a hardened steel facing to the point where 95% of the sample passed through a 75 µm mesh.
- A subsample was subjected to lithium metaborate/tetraborate fusion followed by weak nitric acid digestion and ICP-MS to determine the concentration of 10 major oxides and 45 trace elements including all REEs, thorium, niobium, zircon and uranium. Only samples with whole rock analyses between 98% and 101% were reported by the laboratory, as samples reporting whole rock analyses outside this range may have been incompletely fused.
- Sample results were delivered in digital spread sheet compilation and in signed digital (.pdf) format.

The sample pulps and coarse rejects from the 2010 program are stored at Ucore's warehouse in Ketchikan, Alaska.

11.5 CHECK ANALYSES

Check assays were performed to verify the absence of any significant laboratory bias in the analytical results. ALS Group laboratories was selected as the check laboratory for the 2008 through 2010 sample sets and analyzed each batch of check samples using a common set of procedures. The samples were prepared using ALS code PREP31 and analyzed using codes ME-MS81 and ME-MS81h (for over-limit samples). The preparation and analytical procedures were as follows:

- Samples were weighed on arrival.
- Samples were then crushed to the extent that 70% of the sample material passed through a 2 mm mesh.
- The fine crush was split with a riffle splitter to extract a 200 g sample.
- The subsample was pulverized with a puck pulverizer to the point where 85% of the sample passed through a 75 µm mesh.
- A minimum 1 g sample was split from the pulp and subjected to lithium borate fusion, acid digestion and analyzed with ICP-MS to determine the concentration of 38 elements including all rare earth elements, thorium, niobium, zircon and uranium. Over-limit samples were reanalyzed by re-sampling the pulp, performing a second lithium borate fusion and acid digestion with additional dilution, and finally reanalyzing the sample with ICP-MS.
- Total oxides and loss on ignition were recorded and provided with the trace element analytical data.
- Sample results were delivered in digital spread sheet compilation and in signed digital (.pdf) format.

After the completion of the check analysis, remaining pulps were sent to Ucore's warehouse storage facility in Ketchikan, Alaska.

For the 2008 sample set, Actlabs extracted and conveyed the samples directly from their storage facility in Ancaster, Ontario to the ALS Group laboratories in Vancouver, BC. The 2009 and 2010 samples were extracted by Aurora staff from the sample storage facility in Ketchikan, Alaska and transported to the ALS Group laboratories in Whitehorse, Yukon. Table 11.1 shows the number of check samples assayed compared to the total number of samples taken in each year of the program.

Table 11.1 Check Assay Program from 2008 to 2010

Year	Check Pulps	Total Samples
2008	30	385
2009	27	862
2010	58	1,163

12.0 DATA VERIFICATION

This section describes measures taken to verify the accuracy of the assay, location and density data summarized in this report. Appendix B contains full descriptions of assay quality verification for 2007 through 2009; this section summarizes those results.

12.1 SITE SUPERVISION

The 2008, 2009 and 2010 drill programs were supervised directly on-site by one of the authors of the Aurora report (Mike Power), with another author (Jim Robinson) providing relief. Both authors supervised the 2009 drill program on-site for various periods and the program was supervised by a Professional Geologist at all times on Aurora's staff. All logging, sampling and core handling operations were performed by Aurora staff, or employees of Ucore acting under the on-site Project Manager. Jim Barker, CPG, supervised most of the soil and stream sediment sampling and some of the surface grab sampling program which was undertaken between 2008 and 2010.

12.2 LOCATION VERIFICATION

The locations of drillhole collars and of the origin of trenches were surveyed with a Trimble Geo-XT differential GPS receiver. Corrections were supplied by a stationary differential GPS correction beacon on Annette Island, 39 km east southeast of the peak of Bokan Mountain. Positions were averaged at least 30 times and the quoted horizontal accuracy of the final readings was less than a metre in all cases. Drillhole and trench elevations were interpolated from a digital elevation model constructed from photogrammetric mapping by Eagle Mapping Services of Anchorage, Alaska. In 2010, the locations of all 2009 drillholes on the Dotson Zone were re-surveyed as a check survey. No significant discrepancies between the original and re-surveyed locations were discovered.

12.3 ASSAY DATABASE VERIFICATION

A digital database of all drill core and channel samples was assembled by M. Power prior to the resource calculation. The elements, lanthanum, cerium, praseodymium, neodymium, samarium, europium, gadolinium, terbium, dysprosium, holmium, erbium, thulium, ytterbium, lutetium, yttrium, niobium, zirconium, thorium, and uranium were certified in this database. Results from elements of interest in the laboratory digital data were manually checked against the results reported in the assay certificates prior to entering the data into the database. There were no errors detected in the 2008 and 2010 data. A pair of duplicate over-limit assay certificates was discovered when auditing the 2009 database. The laboratory was contacted and the correct certificate was identified and entered into the database. The digital assay database is secured with access available only to the authors of this report at the time of writing. A copy of the assay database is compiled in Appendix C.

Following entry into the database, the analytical data were also checked against the assay books and the logging spreadsheets to trap errors such as switched or omitted blanks, duplicates or standards. These were rectified during entry and were not logged. Additional errors came to light when the data were analyzed in detail and these are noted together with corrective action in the assay database log.

12.4 VERIFICATION OF FIELD QUALITY CONTROL DATA

Field QA/QC measures included the insertion of inferred or verified blank rock material into the sample stream to check for sample preparation contamination, the insertion of duplicates to measure sample variance and the insertion of appropriate certified standards to ensure that the laboratory analyses were accurate. The protocols changed from 2007 to 2010 as exploration work on the Property changed focus from uranium to rare earth exploration and from reconnaissance to definition drilling.

2008

Assay QA/QC measures taking during the 2008 program included the insertion of blanks, duplicates and standards into the sample stream. Blank material consisted of ornamental gravel purchased from a local hardware store. Duplicates were submitted in sequential pairs and consisted of quartered core. Table 12.1 summarizes the QA/QC sample frequency for the 2008 exploration program.

Table 12.1 QA/QC Measures from 2008

	Total No.	Value (%)
Total samples	385	100.0
Blanks	34	8.8
Duplicates	35	9.1
Standards	19	4.9

The mean concentrations of the elements of interest in the blank material and their standard deviations were calculated from the entire set of blank analyses less four clear outliers. Time series of the analyses for the element of interest are plotted in Appendix B. These graphs show the sequential variation of the blank analyses during the program, expressed in terms of standard deviations from the mean blank analysis. A failed blank analysis is a single analysis more than ± three standard deviations from the mean or two or more sequential analyses ± two standard deviations from the mean. In the 2008 data set, one blank failed in uranium, a second failed in niobium and a third failed in lanathum, cerium, praseodymium, and gadolinium. The standards following these blanks met certified means in these elements and the sample lots were passed. There is no evidence of significant sample contamination in the data.

Reference standards were only inserted after the completion of the fourth hole in the 2008 program and were used throughout, thereafter. Standard MEG-U-1 certified by MEG Labs of Reno, Nevada was used as the field standard. This standard is certified for uranium only. Three sequential field standards failed dramatically, returning uranium analyses twice the certified value. In the same analyses, the concentrations of all other elements of interest were twice the mean concentration of all other reference sample analyses. The Delayed Neutron Counting Uranium (DNC U) analysis returned a value half that of the ICP-MS analysis so the problem is clearly with the ICP-MS analysis. There was no discrepancy between uranium analyzed by ICP-MS and by DNC in all of the samples within these batches and the problem was clearly confined to the standard measurements. Discussions with the lab failed to resolve the problem but with no evidence of contamination in the nearby blanks and no discrepancy between ICP-MS and uranium analyses in the other samples, the batches were accepted. There were no other problems with the standards.

2009

Assay QA/QC measures taking during the 2009 program included the insertion of blanks, duplicates and standards into the sample stream. Blank material consisted of ornamental gravel purchased from a local hardware store. Duplicates were submitted in sequential pairs and consisted of quartered core. Duplicates channel samples were not taken. Table 12.2 summarizes the QAQC sample frequency for the 2009 exploration program.

Table 12.2 QA/QC Measures from 2009

	Total No.	Value (%)
Total	863	100.0
Blanks	41	4.8
Duplicates	44	5.1
Standards	11	1.3

The mean concentrations of the elements of interest in the blank material and their standard deviations were calculated from the entire set of blank analyses less three outliers. Three samples were removed from the data set prior to this calculation when examination revealed likely sampling errors (anomalous values with missing preceding or following sample numbers in the first two cases, and an obvious duplicate pair in the third). Time series of the analyses for the element of interest are plotted in Appendix B. These graphs show the sequential variation of the blank analyses during the program, expressed in terms of standard deviations from the mean blank analysis. A failed blank analysis is a single analysis more than ± three standard deviations from the mean or two or more sequential analyses ± two standard deviations from the mean. In the 2009 program, one sample failed in lanthanum and cerium but all others analyses passed. There is no evidence of significant sample contamination in the data.

Reference standards were inserted systematically only after the completion of the 19th of 27 holes. Standard MEG-U-1 certified by MEG Labs of Reno, Nevada was used as the field standard. This standard is certified for uranium only. A systematic upward drift in uranium concentration was noted in the standard sample analyses, culminating in the failure of the last two samples at the limit of the 95% confidence interval about the certified standard mean. There was no comparable drift in the concentrations of the REEs in the same data set.

2010

Assay QA/QC measures taking during the 2010 program included the insertion of blanks, duplicates and standards into the sample stream. Blank material consisted of phyllitic schist collected from an exposure bordering Whipple Creek near Ketchikan. Duplicates were submitted in sequential pairs and consisted of quartered core. Duplicates channel samples were not taken. Table 12.3 summarizes the QA/QC sample frequency for the 2010 exploration program.

Table 12.3 QA/QC Measures from 2010

	Total No.	Value (%)
Total	1,158	100.0
Blanks	65	5.6
Duplicates	22	1.9
Standards	65	5.6

Mean concentrations of the elements of interest in the blank material and provisional standard deviations were calculated from five assays of this material by Stewart Group laboratories in Stewart, BC performed prior to the start of the 2010 program. Final analysis consisted of calculating the mean concentrations of the elements of interest and their standard deviations from the entire set of blank analyses less three outliers. Time series of the analyses for the element of interest are plotted in

Appendix B. These graphs show the sequential variation of the blank analyses during the program, expressed in terms of standard deviations from the mean blank analysis. A failed blank analysis is a single analysis more than ± three standard deviations from the mean or two or more sequential analyses ± two standard deviations from the mean. In the 2010 program, one blank failed all REE's but passed uranium, thorium, and niobium. The standard immediately preceding this sample passed as did the following blank and standard. No action was taken regarding the sample lot. Zirconium and niobium values were erratic in all the blanks and this may reflect natural variation in zircon content in the blank material. There is no evidence of significant sample contamination in the data.

The 2010 program had two sets of standards (REO-A and REO-B) prepared by CDN Resource Laboratories Ltd. located in Langley, BC from coarse sample rejects. Standard REO-A has a TREO concentration of approximately 1.57% while REO-B has a TREO concentration of about 0.175%. The standards were certified by Dr. Barry Smee, P.Geo., of Smee & Associates Consulting Ltd. located in North Vancouver, BC. One of the two standards was randomly selected for each sample insertion and the standards were inserted into every batch of 20 samples.

Niobium analyses were highly erratic and appear to be depressed by approximately two standard deviations from the certified standard means. The differences between the certified niobium standards and the measured value of these standards are summarized in Table 12.4.

Table 12.4 Certified Niobium Standards versus Measured Value of These Standards

	REO-A		REO-B	
	Mean	Standard Deviation	Mean	Standard Deviation
Certificate	1,459.0	75.5	209.0	11.0
Measured	1,301.0	173.3	187.9	27.1
Difference (%)	-10.8	-	-10.1	-

The means of the standards agreed within the bounds of apparent measurement error but the field standards were consistently lower and show large apparent measurement error. The depression of niobium results may be due to the phosphorus pentoxide content; REO-A contains about 0.1% phosphorus pentoxide and all of the REO-A niobium analysis are significantly depressed. Actlabs advised Aurora that elevated phosphorus can severely degrade the accuracy of niobium analyses.

In addition to the problem with niobium, results for REO-A show slightly elevated cerium and lutetium, and slightly depressed dysprosium, gadolinium, and holmium while the results for REO-B show slightly elevated yttrium and erbium. These errors are minor and no significant discrepancies between the certified standards and the analyses of the field standards are present with the exception of the aforementioned problem with niobium. The field standard analyses indicate that there is no significant problem with the accuracy of the laboratory with respect to REEs, thorium and uranium but that niobiummay be depressed by phosphorus in some samples.

12.5 VERIFICATION OF LABORATORY QUALITY CONTROL DATA

The laboratories to which samples were submitted in 2007 through 2010 have internal QA/QC programs involving the analysis of standards, blanks, duplicates and resplits to ensure accurate data is delivered to their clients. A summary by year is described in the following sections.

2007

ALS Minerals submitted their internal QA/QC data in .pdf format but not in spreadsheet format. This prevented systematic analysis of their results. Aurora inspected the laboratory QA/QC reports and noted that the standards for certificate VA 07124990 showed evidence of slight contamination with REE analyses greater than the upper limit of the 95% confidence interval. There were no other significant problems noted.

2008

Cerium and neodymium for standards DNC-1 and MAG-1 reported analyses slightly and systematically lower than their certified values. There were no other problems noted with the data.

2009

There were no systematically significant discrepancies between the standard measurements and the published standards for the Eco-tech ICP-MS standard runs. A systematic error in the standards run for over-limit assay QA/QC was detected however. In the first four batches OKA2 cerium values were reported which were one tenth the certified standard mean for cerium. The laboratory attributed this to a reporting and transcription error and the results for the remaining elements analyzed showed no discrepancies with respect to subsequent and published values.

2010

Actlabs analyzed a suite of over 12 standards for each batch of samples. Standards W-2a, NCS DC70009, OREAS 100a (fusion) and JR-1 were tabulated and monitored as part of the company QA/QC program. These laboratory standards spanned the REE series and included thorium, uranium, zirconium, and niobium.

Statistically significant responses greater than two standard deviations relative to the certified standard means are summarized in Table 12.5.

Table 12.5 Elements with Responses Greater Than Two Standard Deviations

Standard	High	Low
W2a	La	Y, Th
NCS DC70009	La	None
OREAS 100a	La, Sm, Ho, Yb	Gd, Dy, Er, Lu
JR-1	None	None

The observed failures in the first three standards are small and no statistically significant errors were found for standard JR-1.

12.6 VERIFICATION OF LABORATORY RESULTS BY EXTERNAL ASSAY

ALS Group laboratories was selected to run check assays from the 2008 through 2010 program to further verify the accuracy of the results. The check assay program is described in Section 11.5 and the results are summarized in the following subsections. The data was analyzed by linear regression of the check assay results (y) against the sampling laboratory assay results (x). If the results agreed perfectly, the calculated regression slope would be 1.00 and the intercept would be 0.00. Variations in the slope above or below 1.00 indicate check assay values greater than or less than the original assays.

2008

ALS returned over-limit analyses for dysprosium, erbium, thorium, and uranium, and the verification is consequently limited to concentrations less than 0.5% (5,000 ppm). There are significant differences of +13% and -14% between the ALS Group analyses for zirconium and niobium and the corresponding Actlabs analyses. These are attributed entirely to ALS Group laboratories as ALS Group check assays for zirconium and niobium failed to meet certified standard analyses for these elements whereas the Actlabs analyses did meet certified standards for zirconium and niobium. There is also a very large difference of 35% in the europium response between the ALS Group and Actlabs analyses. In the check assay program, ALS Group laboratories consistently reported europium check analyses which were significantly higher than the original europium analyses, regardless of laboratory or year. Consequently, the europium anomaly discrepancy is attributed to ALS Group laboratories and is not considered significant. The remainder of the elements show variations of 6% or less between the laboratories with no consistent pattern.

2009

The results for cerium, praseodymium, samarium, gadolinium, dysprosium, erbium, yttrium, and thorium are suspect because of the over limit analyses and no significance can be attached any apparent discrepancy in the analyses. The over limit responses limit the correlation range to analyte concentrations less than 0.5% (5,000 ppm). In general there are wide discrepancies between the ALS Group laboratories and the Eco-tech laboratories results for any element; these are reflected in the range between the upper and lower bounds of the slopes and intercepts in the regression analysis.

The results indicate that the two laboratories are in agreement within 6% for 6 analytes; that there are significant differences between the laboratories for 3 of the elements and that no conclusion can be made concerning the remaining 8 analytes because of over limit ALS Group analyses. There are significant discrepancies between europium (+18% higher in the ALS Group laboratories data), ytterium (-21%) and lutetium (-25%). These results suggest that Eco-tech may have over reported ytterium and lutetium by around 20%. ALS Group europium assays are elevated with respect to the other laboratories in all of the check samples and consequently the europium discrepancy is not considered significant.

2010

ALS Group laboratories returned over-limit assays in cerium, samarium, dysprosium, and gadolinium and the verification is consequently limited to concentrations less than 0.5% (5,000 ppm) for these elements. The overall results indicate that the two laboratories are in agreement within 5% on 15 of the 17 elements examined. There were significant discrepancies between the ALS Group laboratories and the Actlabs results in samarium and gadolinium. Given that ALS Group returned over-limit assays for both these elements, thereby confining the range of the analysis, these findings are not considered significant. It cannot be concluded that there is a significant difference between the results from Actlabs and ALS Group laboratories for the suite of level samples analyzed.

12.7 ESTIMATES OF PRECISION

Field duplicate variance is the sum of sampling, splitting and analytical error and provides the best measure of overall assay precision. Field duplicate variance was analyzed using the method of Thompson and Howarth (1978). This procedure yields estimates of sample variance in percent for a given concentration. The precision of an analysis for a given analyte is the concentration at which the sample variance is 100%. The precision for each element is summarized by year in Table 12.6, drawn from the summaries in Appendix B.

Table 12.6 Precision of Analysis from 2008 to 2010 (in ppm)

Analyte	2008	2009	2010
La	50	100	300
Ce	50	200	400
Pr	10	50	100
Nd	50	50	200
Sm	10	10	50
Eu	10	10	10
Gd	50	50	100
Tb	10	10	50
Dy	10	50	100
Ho	10	10	50
Er	10	50	50
Tm	10	10	10
Yb	10	50	50
Lu	10	10	10
Y	50	200	300
Zr	300	200	100
Nb	50	10	100
Th	50	100	100
U	50	10	50

The 2010 program sampling was much more aggressive than that of the preceding years, and included the sampling of numerous intersections of inhomogeneous stringer mineralization. The deterioration in precision in 2010 likely reflects this in part.

12.8 Verification of Density Measurement

During the 2010 program, a total of 79 density measurements were made on samples of mineralized vein material in the Dotson Zone and on adjacent wall rock and distal country rock samples. The average density of the vein material (2.77 ± 0.12 g/cm^3) and the country rock (2.77 ± 0.12 g/cm^3) are consistent with tabulated values for this material reported elsewhere (Paul et. al. 2009; Telford et.al. 1990).

The density measurements were verified by submitting a subsample of 10 of the original density measurement samples to ALS Group laboratories in Vancouver, BC for check density measurements both with and without a paraffin coating (Code OA-GRA09A) (see Appendix D).

The check values returned by ALS Group fall within the density measurements taken in the field. Therefore it can be concluded that the water immersion density measurements in the data base are reliable and suitable for use in a resource calculation.

12.9 SUMMARY

The horizontal locations of drillholes and trenches appear to be accurately located to within ±1 m and their elevations are interpolated from a digital terrain model constructed with high resolution air photography. The location data is sufficiently accurate to support a resource model but will require resurveying to centimeter accuracy prior to planning development workings.

The samples collected from drillholes and trenches were systematically extracted by trained personnel working under fully qualified supervisory staff. The samples were secured and conveyed in an appropriate chain of custody. QA/QC procedures changed as the program evolved from reconnaissance to advanced exploration and the focus changed from uranium to REE exploration. Consequently, the frequency of insertion of QA/QC materials was initially too low and the standards employed were inappropriate for an advanced REE exploration program until 2010. Despite these limitations, it can be concluded from analysis of the blanks that the sample stream is free of any significant contamination; that the laboratories demonstrated that their REE analyses were accurate using analyses of certified standards; that field standards independently verified the accuracy of the laboratory assays for some or all of the analyses of interest, depending upon the year; and that the laboratory results agree between laboratories within the bounds of variance. Finally, analysis of the field duplicates indicates that the sample precision is sufficiently high to support the use of the samples in a resource calculation.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 PRELIMINARY HYDROMETALLURGICAL TEST WORK – PHASE 1

As part of the development of the mineral resource in September 2011, Ucore contracted Hazen to conduct testing in three main phases:

- Mineralogical characterization of Dotson Shear Zone composite
- Preliminary physical separation tests
- Hydrometallurgical preliminary laboratory investigation.

The metallurgical testing programs are listed in Section 27.

In the same period of 2011, Ucore contracted Commodas to conduct a preliminary sorting investigation and bench top amenability tests. In December 2011, Commodas completed the DEXRT sorting tests following the initial bench top amenability testing.

In September 2011, Hazen conducted preliminary test work to study sample mineralogy and determine process technology based on the response of the prepared samples. These samples originated from the Dotson Shear Zone, Bokan Mountain Project, Alaska. The samples were tested and the solid densities of the samples are shown in Table 13.1

Table 13.1 Solid Densities of Samples

Hazen	Client ID	Material	Density (g/cm^3)	Assay (ppm) TREE (LREE+HREE)
52714-296*	H026154	Drill Core	2.61	3,120
52714-321**	H030982	Drill Core	2.97	70
52714-324	H030987	Minus 1.7 mm	2.84	1,710
52874-1	Barren	Rock	2.79	340
52874-1	Barren	Minus 150 µm	2.79	430
52714	High Grade Composite	Minus 1.7 mm	2.75	8,700
52714	High Grade Composite	Minus 1.7 mm	2.74	8,600
52874-2	Mineralized	Rock	2.83	29,700
52874-2	Mineralized	Minus 150 µm	2.84	30,100

Note: * ATT bulk density measurement
** LREE and HREE assays provided by Ucore. All other analyses were done by Actlabs.

Table 13.2 summarizes the Bond ball mill work index (BWi), Bond rod mill work index (RWi), Bond crusher impact work index (CWi) and Bond abrasion work index (Ai) tested results for these samples.

Table 13.2 Sample Identification and BWi, RWi , CWi, and Ai Results

Hazen	Client ID	Value (kWh/t)			Ai (g)
		BWi	RWi	CWi	
52874-1	Barren	16.6	20.1	19.65	0.4741
52874-2	Mineralized	10.9	14.1	13.74	0.5958
53005-1	Zone 28 Sorter Mineralized Composite	16.1	nd	nd	nd
53105-1	1:1:1 Zones 25, 26, 28 Sorter Mineralized Composite	14.6	nd	nd	nd

The chemical analytical work was conducted primarily by Actlabs in Ancaster, Ontario, Canada, and the control analyses were performed at Hazen. The test results are shown in Table 13.3, which presents bulk chemical assay results for the composite of core sections.

Table 13.3 Head Assay for Composite Mineralized Material

Analyte Element	Analysis (%)	Analyte Oxide	Analysis (%)
Y	0.30	Y_2O_3	0.38
LREE	0.70	LREO	0.82
HREE*	0.18	HREO*	0.21
HREE**	0.48	HREO**	0.59
TREE***	1.18	TREE***	1.41
La	0.13	La_2O_3	0.15
Ce	0.31	Ce_2O_3	0.36
Pr	0.04	Pr2O3	0.05
Nd	0.17	Nd_2O_3	0.20
Sm	0.047	Sm_2O_3	0.054
Eu	0.005	Eu_2O_3	0.005
Gd	0.046	Gd_2O_3	0.0052
Tb	0.009	Tb_2O_3	0.010
Dy	0.0055	Dy_2O_3	0.063
Ho	0.010	Ho_2O_3	0.012
Er	0.028	Fr_2O_3	0.032
Tm	0.004	Tm_2O_3	0.004
Yb	0.021	Yb_2O_3	0.024
Lu	0.003	Lu_2O_3	0.03
Al	5.4	Al_2O_3	10.2
Be	0.012	-	-

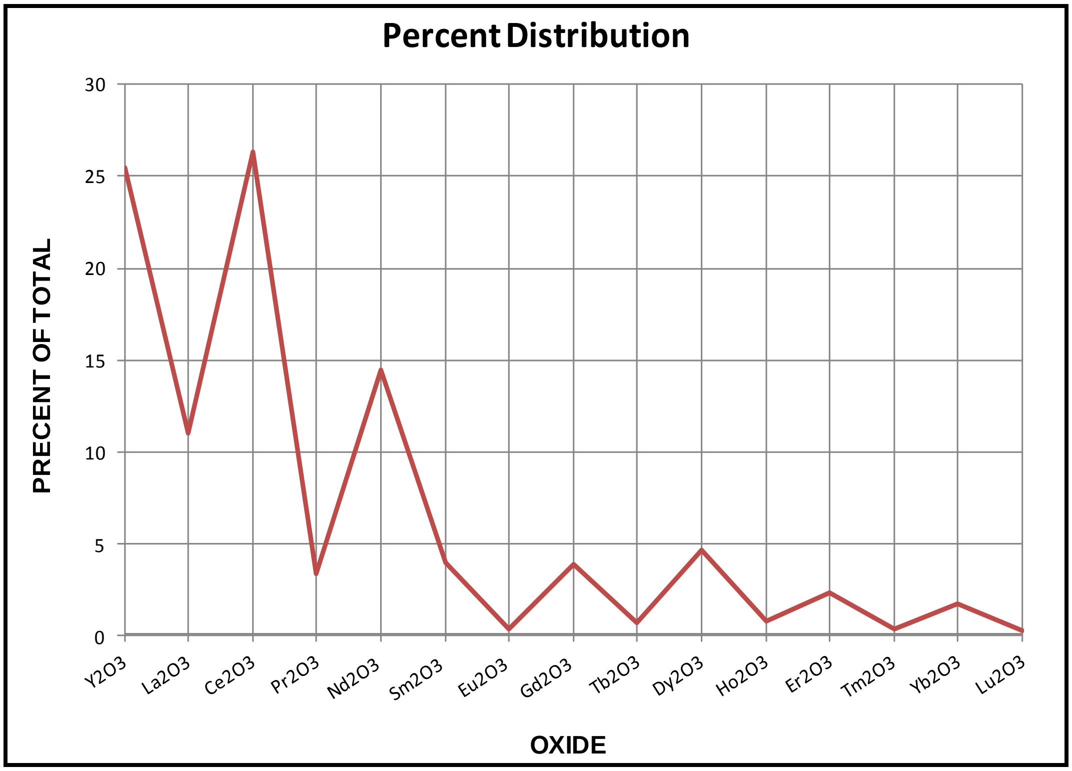
table continues…

Analyte Element	Analysis (%)	Analyte Oxide	Analysis (%)
Ca	2.0	CaO	2.8
-	-	Co_2****	1.31
F	0.31	-	-
Fe	3.0	Fe_2O_3	4.3
Hf	0.008	-	-
K	1.5	K_2O	1.9
Mg	0.9	MgO	1.5
Na	3.1	Na_2O	4.2
Nb	0.092	Nb_2O_5	0.131
Sc	0.0008	-	-
Si	33.2	SiO_2	70.9
Ta	0.007	-	-
Th	0.037	ThO_2	0.042
U	0.016	U_3O_8	0.028
Zr	0.44	ZrO_2	0.059

Notes: *Excluding yttrium
**Including yttrium
***Total rare earth elements or oxides including yttrium
****Hazen assay

Figure 13.1 presents a graphical representation of the distribution of the REOs (+yttrium) in the sample showing the high abundance of HREOs in the sample.

Figure 13.1 Percent Distribution of Each REO versusTREO+Y



The mineralization reportedly consists of a complex, relatively fine-grained assemblage of heavy and light rare earth-containing minerals, some with high yttrium content, as well as niobium-, zirconium-, and thorium-bearing minerals. Of particular interest was the mode of occurrence of rare earth-containing minerals and an assessment of potential rare earth recoverability in a metallurgical process. Because of the broad range of objectives, this section specifically summarizes mineralogical characterization of Dotson Shear Zone composite.

Hazen conducted quantitative mineralogical characterization of the REE occurrence in September 2011. The mineralogical investigation consisted of detailed QEMSCAN analyses of separate screen size fractions between 1.7 mm to 20 µm to characterize the REE mineralization and associated gangue constituents. Assay concentration and distribution of REE are expressed as metals and as REO. Mineralogical analyses revealed that the REE mineralization is complex, consisting of several REE mineral species but also REE–yttrium-bearing gangue minerals, i.e. not actual REE mineral species. Rather than establishing the exact identity of each REE mineral in the classical sense, emphasis was on metallurgical oriented issues such as grain size, intergrowth textures, and other features providing guidance for selection of physical upgrading options of the REE values. The results of the QEMSCAN analyses are summarized below, and the mineral abundance of the mineralized material is shown in Table 13.4 and Table 13.5.

Table 13.4 Summary of Mineral Abundance

Mineral	QEMSCAN Analysis Mass %*
REE silicates	4.7
REE carbonates	0.9
REE phosphates	0.34
REE oxides	0.6
Oxide gangue	1.8
CFA** gangue	2.2
Silicate gangue	88.2
Misc gangue	0.3
Unidentified	0.6
Total	**100**

Notes: *Mass% represents the grams of the particular mineral in 100 g of mineralized material.
**Calcite, fluorite, and apatite

Table 13.5 Mineral Abundance (Detailed Mineral List)

Mineral	Formulae	QEMSCAN Analysis Mass %[a]
REE silicates	-	4.6
Allanite	(Ce,Ca,Y)2(Al,Fe3+)3(SiO4)3(OH)	1.3
Zircon	ZrSiO4	1.8
Cerite	$(La,Ce,Ca)_9(Mg,Fe^{3+})(SiO4)_6[SiO_3(OH)](OH)_3$	0.1
Kainosite	Ca2(Y,Ce)2Si4O12(CO3)•(H2O)	0.2
REE–Ca–Y silicate	-	0.2
Limoriite	Y2(SiO4)(CO3)	0.4
Y–Fe–Ca–REE silicate	-	0.2
Th(U) mineral	ThSiO4	0.1
REE phosphates		
Monazite	Ce,La,Nd,Th)PO4	0.2
Xenotime	YPO4	0.1
REE carbonates	-	-
Bastnäsite	(Ce,La,Y)(CO3)F	0.3
Synchysite **I**	CaCe(CO3)2F – CaY(CO3)2F – Ca(Ce,La)2(CO3)3F	0.5
REE oxides	-	-
Pyrochloreb	(Na,Ca)2Nb2O6(OH,F)	0.2
Fergusonite	(Ce,La,Y)NbO4	0.3
Ti–Y–Fe–REE oxide	-	0.02
Silicate gangue	-	-
Quartz	SiO2	34.6

table continues…

Mineral	Formulae	QEMSCAN Analysis Mass %[a]
Feldspar	$NaAlSi_3O_8$ – $KAlSi_3O_8$	37.3
Mica and chlorite	$K(Mg,Fe^{2+})_3[AlSi_3O_{10}(OH,F)_2]$ – $(Mg,Fe^{2+})_5Al(Si_3Al)O_{10}(OH)_8$	8.4
Amphibole and pyroxene	$NaFe^{3+}Si_2O_6$ – $Ca_2(Mg,Fe^{2+})_5Si_8O_{22}(OH)_2$	6.8
Epidote	$Ca_2(Fe^{3+},Al)_3(SiO_4)_3(OH)$	0.8
Fluorite	CaF_2 0.5	0.5
Carbonate	$CaCO_3$	1.9
Apatite	$Ca_5(PO_4)_3(OH,F,Cl)$	0.1
Oxide gangue	-	-
Pyrophanite	$MnTiO_3$	0.01
Ti oxide and Fe–Ti oxide	TiO_2 and $FeTiO_3$	0.1
Fe oxide and Fe hydroxide	Fe_2O_3 – $FeOOH$	1.7
Sulphides	-	-
Pyrite	FeS_2	0.2
Sphalerite	ZnS	0.2
Galena	PbS	0.1
Chalcopyrite	$CuFeS_2$	0.02
Misc. minerals		0.7
Unidentified		0.7
	Total	**100**

Notes: [a] Mass% represents the grams of the particular mineral in 100 g of mineralized material.
[b] Pyrochlore proper is not a REE mineral but was found to contain REE and also includes yttropyrochlore and plumbopyrochlore.

Yttrium and REEs are contained in a variety of silicates, carbonates (bastnäsite and synchysite), oxides (pyrochlore, fergusonite, and an as yet-unidentified titanium-yttrium-iron-zinc oxide), and phosphates (monazite and xenotime). The main REE silicates are yttrium-calcium-bearing. This group contains minerals like iimoriite, kainosite (both mixed silicate–carbonate), cerite, and other as yet-unidentified calcium-yttrium–REE-bearing silicates. Other yttrium- and REE-bearing minerals are zircon (porous in appearance and likely partially hydrated), allanite (epidote group mineral), and thorite. The concentration of individual REE minerals varies from 0.5% (synchysite) to 0.02% (unidentified titanium-iron-calcium-REE silicate). Zircon is the REE-bearing mineral with the highest concentration (1.8%), followed by allanite (1.3%). When grouping the REE minerals into silicates, carbonates, phosphates, and oxides, the silicate group has the highest concentration (mainly due to zircon and allanite).

The main gangue minerals are quartz and feldspar (predominantly albite with minor K-feldspar). Other REE-free silicates are amphiboles and pyroxenes, mica, chlorite (clinochlore), and REE-free epidote. Calcite is the only gangue carbonate observed.

Fluorite and probably apatite contain fluorine in the mineralized material. Beryllium deportment in the mineralized material was not tracked, but traces of genthelvite ($Zn4Be3(SiO4)3S$) were observed.

The total sulphide (pyrite, sphalerite, galena, and chalcopyrite) concentration is 0.5%.

The composite mineralized material contains 35% quartz and 37% feldspar. Both minerals are well-liberated, where 81% of the quartz and 74% of the feldspar occurs in liberated particles (across the size range of minus 600 to minus 20 µm). If the liberated quartz particles were removed by means of physical separation, the calculated losses for selected REE (yttrium, cerium, neodymium, gadolinium, dysprosium, and ytterbium), zirconium, and niobium would be between 7% (dysprodium, as the element with the lowest loss) and 12% (niobium, as the element with the highest loss). If liberated feldspar particles (greater than 70%) only were removed, the calculated losses would be lower: between 2% for selected REE and zirconium and 3% for niobium. If liberated quartz and feldspar were removed together quantitatively by physical means, the calculated losses for REE are approximately 10% and 14% and 15% for zirconium and niobium respectively.

The possible removal of magnetic mineral fractions (to minimize acid consumption during leaching and to further reduce mass) and associated REE losses were theoretically evaluated based on QEMSCAN data. Particles in the plus 38 µm fraction that contain more than 50 area percentage of amphibole, pyroxene, chlorite, biotite, epidote, allanite, iron oxides, and iron hydroxides were quantified. These particles represent 7% of the total sample mass and contain 2% of the total REE minerals present in the sample.

If liberated REE minerals (i.e., greater than 70 area percentage, averaged across the size ranges greater than 38 µm) were to be separated, the theoretical recovery of REE, zirconium, and niobium would be only approximately 30%. This indicates that a separation of REE minerals as the only means of upgrading REE would not be beneficial.

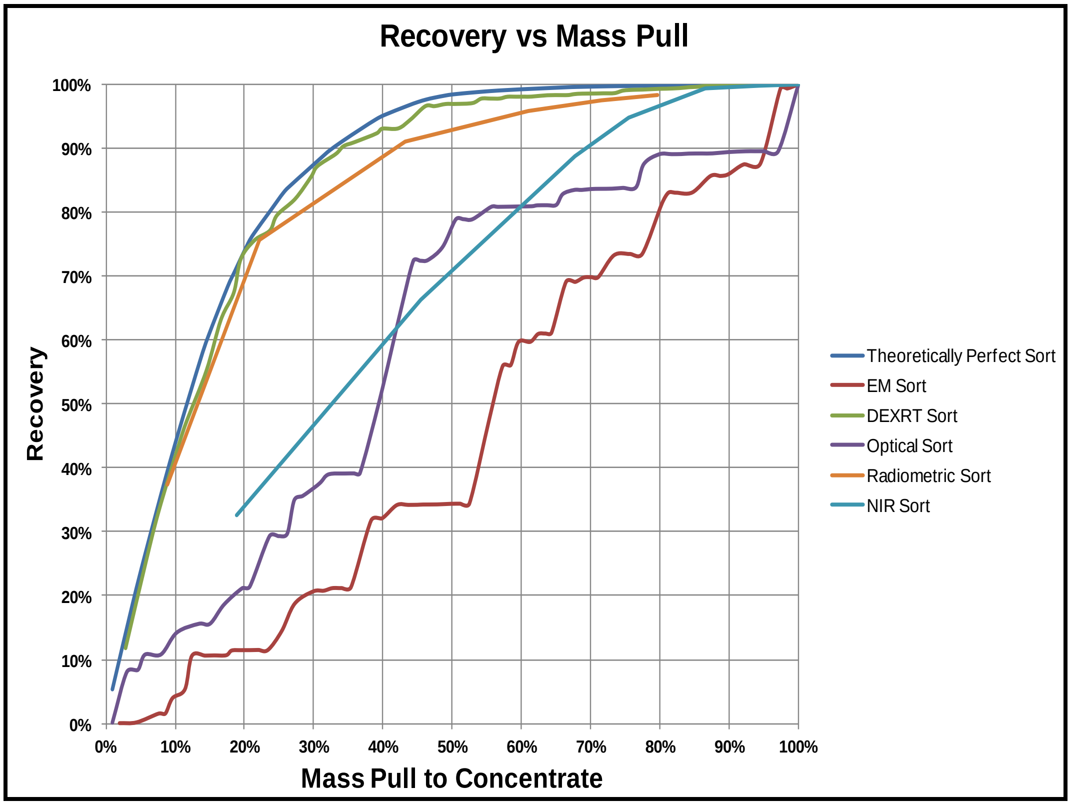
At a density of 2.67 g/cm^3 (based on QEMSCAN data), 40% of the total sample mass falls into the light category. This portion consists of 21% quartz and 18% feldspar. This theoretical density fraction contains 1.6% of the total concentration of REE minerals in the sample. The minus 38 µm fraction, likely inseparable by physical means on an industrial scale, contains approximately 30% of the total REE minerals.

Overall, the mineralizations are related to the peralkaline ring dike complex known as the Bokan Intrusive Complex. Variable concentrations of LREEs, HREEs, zirconium, beryllium, tantalum, and niobium are associated with uranium mineralizations. Ucore's focus is "on dysprosium and HREEs" (McKenzie 2011.

In September 2011, Commodas conducted a preliminary sorting investigation, which included visible spectrum optical sorting, x-ray transmission sorting, conductivity/magnetic susceptibility sorting, radiometric sorting, and near infrared sorting. The bench top sorting studies determine the characteristics that separate REE minerals on the basis of their radioactivity, conductivity, magnetic susceptibility, color, texture, X-ray fluorescence spectroscopy features, near infra-red characteristics, DEXRT features and UV fluorescence.

Figure 13.2 shows the results of the preliminary sorting tests. The graph shows the relationship of the recovery of the total lanthanoids+yttrium as a function of mass pull for each of the sorting characteristics in comparison to the theoretically perfect recovery curve.

Figure 13.2 Recovery Curves for Lanthanoids+Yttrium for a "Perfect Sort", conductivity/magnetic susceptibility (EM), Dual Energy X-Ray Transmission (DEXRT), Optical, Radiometric (RM) and Near Infrared (NIR) Sort



The theoretically perfect recovery curve for this sample was obtained by ordering the rocks in descending grade, where grade refers to the sum of the grades of the lanthanoids and yttrium.

The tests showed that both x-ray transmission sorting and radiometric sorting appeared to have the best potential for sorting by grade as the recovery curves are almost a perfect match to the theoretically perfect recovery curve. For example, the x-ray transmission sorting results in Figure 13.2 show a recovery of approximately 98% of the total lanthanoids and yttrium in 61% of the mass. In other words, 39% of the mass could be rejected from this set of rocks by x-ray transmission sorting with a loss of only 2% of the total lanthanoids and yttrium. At a 50% reject of the material the recovery of lanthanoids and yttrium would be 97%, and at a 60% reject of the material the recovery of lanthanoids and yttrium would still be 93%.

In December 2011, Commodas completed the x-ray transmission tests, following the preliminary sorting investigation test. The products from the Commodas were sent to Hazen for weighing, crushing, splitting and chemical analyses of all the fractions generated.

The estimated rates of feed for the tests were as follows:

- for ½" to 1½" estimated throughput of minimum 8 t/h
- for 1½" to 2½" estimated throughput of minimum 25 t/h
- for 2½" to 3" estimated throughput of minimum 40 t/h.

Table 13.6 shows a summary of the results of the sorting for each zone. The plus 3" fraction of the material was not treated in the sorting studies, as the particles were too thick to produce a difference in data between mineralized material and waste rocks. It is interesting to note that mass pull to concentrate in each of the tests was almost proportional to the TREE+Y in the sorter feed material. If each zone is taken individually, only zone 28 has sorting results that are similar to those obtained during previous bench top sorting. For example, Table 13.8 shows that if the zone 28 sample is sorted and the -0.25" fraction is added to the sorter concentrate, then 56% of the mass will end up in the zone 28 concentrate with 94% of TREE+Y contained in this concentrate. By contrast, zone 25 was sorted with the same parameters and resulted in a TREE+Y recovery of only 82% in a mass pull of 21% to concentrate. Similarly, when zone 26 was treated with the same parameters the result was a recovery of 75% of the TREE+Y into 22% of the mass pull.

Table 13.6 Recoveries and Mass Pulls by Zone

Sample Label	Mass (kg)	Mass (%)	TREE+Y La-Lu+Y ppm	TREE+Y La-Lu+Y Total %
Zone 28				
Mineralized Material -3" and -0.25"	662.71	56	14,747	94
Waste	528.3	44	1,191	6
Total	**1,191**	**100**	**8,734**	**100**
Zone 25				
Mineralized Material -3" and -0.25"	164.6	21	16,487	82
Waste	607.2	79	508	18
Total	**771.8**	**100**	**3,916**	**100**

table continues…

Sample Label	Mass (kg)	Mass (%)	TREE+Y La-Lu+Y ppm	TREE+Y La-Lu+Y Total %
Zone 26				
Mineralized Material -3" and -0.25"	210.1	22	13,066	75
Waste	724.3	78	1,739	25
Total	**934**	**100**	**4,286**	**100**

Table 13.7 shows that the full-scale sorting results approach the results obtained during the bench top tests and the difference if the -0.50" material or the +0.25" material were taken directly to sorter concentrate and then the size fractions for the three zones were sorted together. The result for not sorting the minus 0.50" material would be a mass pull of 41% to concentrate with a TREE+Y recovery of 91% against a mass pull of 33% and a TREE+Y Recovery of 87% if only the minus 0.25" material was not treated in a sorter. Consequently, it can be inferred that when zones 28, 26 and 25 were mined, crushed screened and sorted together, then high recoveries could be obtained by only sorting the minus 3" by plus 0.50" material.

Table 13.7 Recoveries and Mass Pulls when Zones 28, 26 and 25 are Mixed Together

Sample Label	Mass (%)	TREE+Y La-Lu+Y ppm	TREE+Y La-Lu+Y Total %
Mixed Mineralized Material Composite from Zones 28, 26 & 25			
-3"- Mineralized Material and -0.25"	33	14,740	87
Waste	67	1,135	13
Total	**100**	**5,645**	**100**
-3"- Mineralized Material and -0.5"	41	12,499	91
Waste	59	903	9
Total	**100**	**5,645**	**100**

Note: The mass pulls of a blend are based on a ratio of 1:1:1 ration of feed to the sorter.

Three separate samples of Bokan Mountain material were studied to examine the ability of magnetic separation to separate and recover the minerals containing the TREE+Y values. The samples consisted of a composite of selected core (Phase 2, Composite Sample 1, Hazen Test 3475-149), the Zone 28-Sorter Mineralized Concentrate (Hazen Test 3503-109) and the blend of the sorter-mineralized-material composite of Zone 25, 26 and 28 (Hazen Test 3540-74). The results are summarized in Table 13.8.

Table 13.8 Summary of Magnetic Separation Tests

Product	Phase 2, Composite Sample 1			Zone 28 Sorter Mineralized Composite			Zone 25/26/28 Sorter Mineralized Composite		
	Weight, %	TREE+Y %	% Distribution TREE+Y	Weight, %	TREE+Y %	% Distribution TREE+Y	Weight, %	TREE+Y %	% Distribution TREE+Y
Analyzed Head	-	1.13	-	-	1.59	-	-	-	-
Calculated Head	100.0	1.14	100.0	100.0	1.51	100.0	100.0	1.59	100.0
Total Magnetic	57.1	1.90	95.2	78.9	1.61	83.5	63.8	2.08	83.6
Total Middlings	N/A	N/A	N/A	2.3	4.14	6.3	6.1	1.21	4.6
Total Non-Magnetic	42.9	0.126	4.8	18.8	0.83	10.2	30.1	0.62	11.8

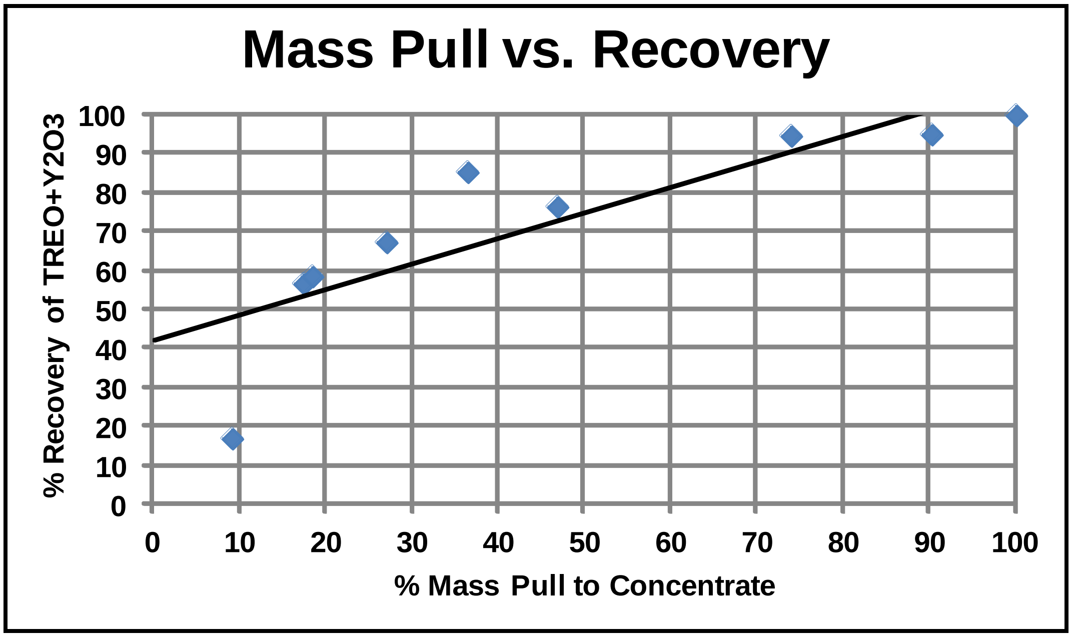
Eight scoping froth-flotation tests were conducted on a sample Identified as the High Grade Composite, (Phase 2, Composite Sample 1). The tests investigated collector reagents, two grind sizes, as well as caustic, acid and natural pH conditions. The flotation test results are summarized in Table 13.9.

Table 13.9 Summary of Flotation Tests

Test	Product	Weight (%)	Analysis (%) TERO+Y2O3	Distribution (%) TERO+Y2O3	Reagent	Grind ~P_{80} µm	Conditioning pH
Head Assay		-	1.35	100.0	-	-	-
3503 -15	Con	90.2	1.38	95.1	SM 15	38	6–9
	Tails	9.8	0.66	4.9	-	-	-
3503 -16	Con	17.4	4.06	56.8	FA-1	38	~8.5
	Tails	82.6	0.65	43.2	-	-	-
3503 -17	Con	73.9	1.71	94.7	727	38	~8.5
	Tails	26.1	0.27	5.3	-	-	-
3503 -18	Con	36.4	3.48	85.4	Linoleic Acid	38	~8.5
	Tails	63.6	0.34	14.6	-	-	-
3503 -105	Con	46.8	2.30	76.5	Saponified FA-1	38	8–10
	Tails	53.2	0.62	23.5	-	-	-
3503 -106	Con	18.4	3.97	58.7	Linoleic Acid	75	8–9
	Tails	81.6	0.63	41.3	-	-	-
3503 -107	Con	9.1	2.56	17.1	Aero 825	38	3–8
	Tails	90.9	1.24	82.9	-	-	-
3503 -108	Con	27.0	3.47	67.4	Linoleic acid	75	8–10
	Tails	73.0	0.62	32.6	-	-	-

The flotation tests were scoping in nature and did not attempt to optimize the grade or the recovery of the TERO+Y2O3 values. An examination of the relationship between the mass pull to the flotation concentrate-froth and the recovery of values demonstrates that there was an upgrading and some concentration of the values by froth flotation. Figure 13.3 depicts the relationship of the mass pull to froth vs. the recovery of the values. The straight line is a linear trend-line demonstrating that the relationship is not a 1:1, and some upgrading occurred during the tests.. However the studies of magnetic separation were demonstrating superior results to flotation and therefore further flotation studies were placed on hold. Additional froth flotation studies may be initiated when additional samples are available. In the future flotation studies more detail will have to be paid to the conditions of pH, reagent types, activator/depressant reagents, as well as temperature.

Figure 13.3 Flotation Results, Mass Pull versus Percent Recovery of Values



In October 2010, Ucore asked Hazen to investigate the preliminary dissolution of REEs of the Bokan Mountain deposit on a laboratory scale, as Phase 1 of the scoping work.

The original hydrometallurgical plan included concentrated acid attack of mineralized material or concentrate at high temperature followed by a water or H_2SO_4 leach, caustic cracking of mineralized material or concentrate followed by water leaching, and calcining of the mineralized material or concentrate followed by dissolution in hydrochloric acid or sulphuric acid. In this work, only concentrated H_2SO_4 attack of the mineralized material was done. No hydrometallurgical processing of concentrate was done because an insufficient amount of concentrate was available during the program. Chemical attack of minerals is expected to be similar for mineralized material and concentrate, but acid consumption would be expected to be lower for a concentrate than for whole mineralized material. Because the initial dissolution from acid attack gave extractions in the mid-80th percentile, other methods, which were thought to be more expensive, were not explored. In addition to being expensive, caustic cracking was also expected to produce competing reactions with finely ground quartz in the mineralized material.

A series of H_2SO_4 bake-leaching scoping experiments was performed to recover REEs from the Dotson Shear Zone mineralized material composite sample from Bokan Mountain. Acid was added to the bake in dosages varying from 485 to 1,230 kg acid/t mineralized material. The acid bake time was 1 hour at 225°C. Mineralized material particle sizes were 17, 40, and 130 µm. Because of the limited amount of mineralized material sample available, the work was done in a 100 mL twin-screw steel pug reactor fabricated at Hazen. After acid baking, the slurry was leached in water at room temperature.

The mineralized material composite analyses are shown in Table 13.10 and the total REEs amount to 8,850 ppm. The analyses of solids for the hydrometallurgical work were primarily done by Actlabs in Ancaster, Ontario, Canada.

Table 13.10 Head Analyses of the Composite Minerialized Material Sample

Analyte	Analysis (ppm)
F^-	2,500
Na	29,200
Mg	8,700
Al	53,220
Si	330.300
P	480
K	15,000
Ca	19,200
Sc	8
Ti	2,220
Fe	44,700
Zn	1,086
Y	3,022
Zr	4,070
Nb	905
La	1,308
Ce	3,166
Pr	411
Nd	1,708
Sm	455
Eu	44
Gd	452
Tb	87
Dy	556
Ho	106
Er	285
Tm	39
Yb	209
Lu	25
Hf	71
Th	360
LREE (La-Sm)	7,050
HREE (Eu-Lu)	1,800

table continues…

Analyte	Analysis (ppm)
HREE+Y	4,830
REE	8,850
REE+Y	11,870

Considering all the experiments, extractions ranged from 79 to 95% for REEs, 87 to 97% for LREEs, and 80 to 88% for HREEs.

The REE extractions from this mineralized material sample were in the mid-80th percentile with some variation caused by different particle sizes and acid additions to the acid bake. Results are summarized in Table 13.11. Figure 13.4 shows that decreasing the 80% passing particle size (P_{80}) from 130 to 40 or 17 µm and increasing the acid addition from 485 to 745 kg/t mineralized material increased REE extractions from the mid-80th percentile to above 90%. Keeping the particle size at 130 µm and increasing the acid addition to 1,200 kg/t did not increase extraction. Also, this set of experiments showed that the acid consumption increased as acid dosage increased. The acid consumption for the finer particle size experiments may explain the better leaching. Filtration after leaching was normal. Experiments with various acid dosage, particle size, acid bake time and temperature, leach time, and mineralogical analysis of the residues are necessary to understand the effect of each parameter and find favorable acid bake-leaching conditions for recovering REEs directly from concentrate or mineralized material.

Table 13.11 Summary of H_2SO_4 Bake-Leaching Experiments

Experiment Host Rock, Hazen	Head 52547	3386-47 52547	3386-48b 52547	3386-49 52547	3386-55 52547	3386-56 52547	3386-59 52547
P_{80}, µm	-	130	130	130	40	17	130
Acid Bake Acid/Host Rock Addition Ratio (kg/t)		631	1,234	485	745	745	785
Consumption		310	850	400	530	560	350
Temperature (°C)		225	225	225	225	225	225
Time (h)		1	1	1	1	1	1
Water Leach Temperature (°C)		Ambient	Ambient	Ambient	Ambient	Ambient	Ambient
Time (h)		1	1	1	1	1	1
Pulp Density (% Solids)		13	12	22	12	14	21
Filtration Characteristics		Normal	Normal	Normal	Normal	Normal	Normal

table continues…

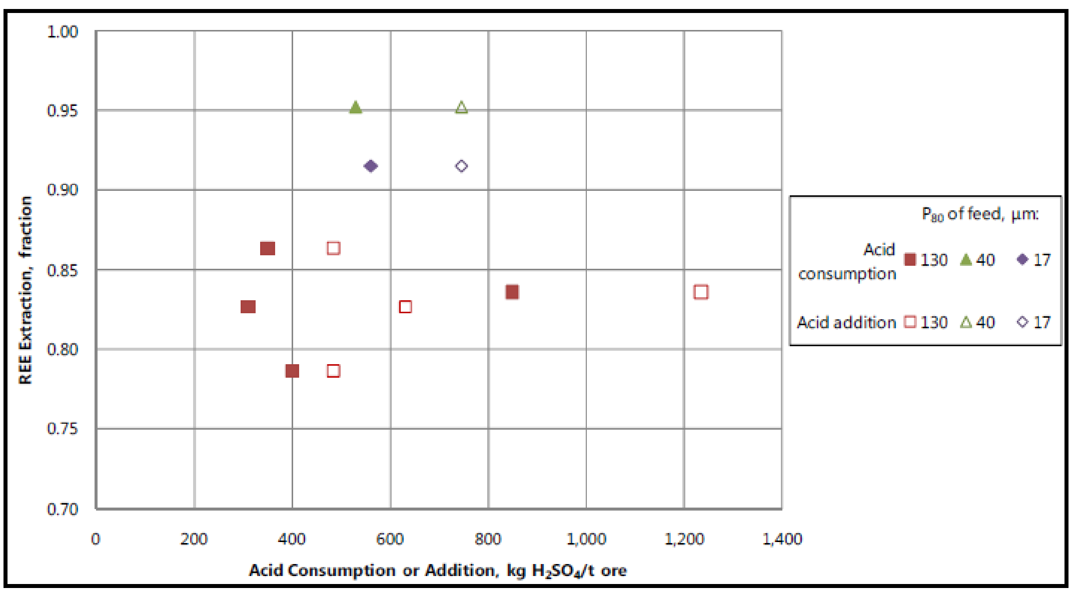
Analyte	Average Analysis, ppm	Extraction, Fraction (Solids Basis)					
LREE (La-Sm)	7,050	0.83	0.84	0.78	0.97	0.94	0.87
HREE (Eu-Lu)	1,800	0.83	0.83	0.80	0.88	0.83	0.84
HREE+Y	4,830	0.83	0.84	0.80	0.88	0.84	0.84
REE	8,850	0.83	0.84	0.79	0.95	0.92	0.86
REE+Y	11,870	0.83	0.84	0.79	0.94	0.90	0.86
Element	**Mass Balances, Out/In, Fraction**						
Ce	-	0.83	0.70	0.76	0.85	0.89	0.88
Y		0.81	0.64	0.72	0.85	0.94	0.87

Notes: Acid addition is the acid added to the experiment. Acid consumption is the amount of acid chemically consumed and does not include evaporation.

It should be noted that the mass balances were less than desired, introducing uncertainty into the reported extraction values. The probable causes of the less-than-desirable mass balances are the difficulty of removing solids from the reactor after the experiments and the small quantity of host rock used. Not removing all of the solids would bias the extractions high. Calculations of distribution of elements into the liquor gave REE extractions that were 2 to 8% points lower than the solids-basis extractions.

Mass extractions vary between 10 and 23%. The highest zirconium extraction is 31% and is achieved using a 40 µm particle size and 530 kg acid/t mineralized material acid consumption. About 50 to 90% of the iron dissolves and 4 to 15% of the silicon dissolves. Thorium extraction varies between 77 and 91%.

Figure 13.4 REE Extraction as a Function of Acid Consumption



Experiments 3386-49 and 3386-59 were performed under the same acid bake–leach conditions and resulted in REE extractions of 79 and 86%, giving an indication of the variation possible in this small reactor.

13.2 Preliminary Hydrometallurgical Test Work – Phase 2

Following the results of the Phase 1 studies, a new direction of leaching and extraction of the REEs was investigated. Although other companies are following the sulphuric acid-bake process, the environmental problems associated with air emissions and disposal of wastes were thought to be detrimental to the project due. IntelliMet of Montana proposed a change to nitric acid for the extraction of the REEs. The change was a result of preliminary leaching tests by IntelliMet and the potential for recovery of the nitric acid via diffusion dialysis. The diffusion dialysis process is a currently available off-the-shelf technology and does not result in the waste products associated with the use of sulphuric acid.

During the Phase 2 hydrometallurgical studies, samples of material collected from the deposit before and after DEXRT concentration were tested. There was insufficient material produced by the magnetic separation studies on DEXRT sorted material to conduct the following described leaching studies.

A sample of mineralized material was collected by Ucore from the deposit designated as Zone 25. The sample was determined to be higher grade than the average for the deposit, but in line with the expected grade of material produced by the magnetic separation of the DEXRT sorting process on average grade material. The sample was designated as “Zone 25-Selected” for identification in laboratory studies.

A sample of the Zone 25-Selected material ground in the laboratory to a particle size of 191 µm (80% passing 191 µm) and leached by the two different techniques; the sulphuric acid bake process developed by Hazen which is followed by a room temperature leach, and a 90°C nitric acid leaching process. The nitric leach procedure consists of two counter-current steps to utilize the full strength of the acid and increase extraction. After mechanical beneficiation, the mineralized material is first exposed to partially depleted acid to produce the process leach solution containing the leached REEs. The partially leached mineralized material is then exposed to fresh nitric acid in a second leaching step and then the slurry is filtered and washed with the solids sent to the paste backfill system. The partially depleted nitric acid from this second leach proceeds on to be the leach solution for the new mineralized material concentrate.

This leach procedure was modelled and tested by both Hazen and Intellimet. Table 13.12 presents the results of the nitric acid leaching process with a comparison to the sulphuric acid bake extractions.

Table 13.12 Acid Leaching of Zone 25-Selected Sample

Testing Lab	IntelliMet	IntelliMet	Hazen
Test No.	CH-5-784	CH-5-849	3477-27
Feed Material	Z-25 Selected	Z-25 Selected	Z-25 Selected
Assay Laboratory	Actlabs	Actlabs	Hazen
Acid	H_2SO_4	HNO_3	HNO_3
Acid concentrate (v/v)	10%	30%	30%
Grind (µm)	191	191	191
Solids (%)	20	20	20
Feed Grade (ppm)			
LREE	28,890	28,890	28,200
HREE+Y	13,160	13,160	12,910
TREE+Y	42,050	42,050	41,110
Y	7,732	7,732	7,700
Extraction (%)			
LREE	87	97	93
HREE+Y	79	83	75
TREE+Y	85	93	87

The hot nitric acid leaching process at both the laboratories showed an improvement over the sulphuric acid-bake procedure. Although the Hazen test products were analyzed internally at Hazen and the IntelliMet products were analyzed by ActLabs in Canada, the results of the tests show the superior extraction of the nitric acid procedure on this sample.

A sample of mineralized material produced by the DEXRT sorting process on a bulk sample collected from the area of the deposit identified as Zone 28. The material was a blend of the sorted mineralized material blended with the minus 14" material that was not sorted. The sample was designated as "Zone 28-Selected" for identification in laboratory studies. The sample is lower grade than the expected mill feed after sorting and magnetic separation.

The material was processed by crushing and grinding in the laboratory to four particle sizes (80% passing the micron size). The ground samples were then subjected to a standard hot nitric acid leach for 8 hours at 90°C. Table 13.13 summarizes the results of the four tests.

Table 13.13 Effect of Particle Size on the Leaching of Zone 28-Sorted Composite

Testing Lab	Hazen	IntelliMet	Hazen	IntelliMet
Test No.	3477-25	CH-5-5122	3477-33	CH-5-1272F
Feed Material	Z-28 Sorted	Z-28 Sorted	Z-28 Sorted	Z-28 Sorted
Assay Laboratory	Hazen	Actlabs	Hazen	Actlabs
Acid	HNO_3	HNO_3	HNO_3	HNO_3
Acid concentrate (v/v)	30%	30%	30%	30%
Grind (µm)	2,000	190	40	21.8
Solids (%)	20	20	20	40
Feed Grade (ppm)				
LREE	11,280	11,104	11,280	11,104
HREE+Y	4,239	4,077	4,239	4,077
TREE+Y	15,519	15,181	15,519	15,181
Y	2,500	2,492	2,500	2,492
Extraction (%)				
LREE	81	90	85	78
HREE+Y	59	67	62	58
TREE+Y	75	84	79	72

The results of the four tests illustrate the effect of particle size on the extraction of the rare earth elements at similar conditions (the test at 21.8-microns was at a higher pulp density during leaching). The selection of the final design grind prior to leaching will be investigated in the next phase of laboratory and pilot plant studies planned in the year 2013 and the plant design adjusted accordingly.

Additional laboratory and pilot plant studies are planned on a fresh set of samples collected from the deposit that will include leaching of material after DEXRT sorting and high gradient magnetic separation.

Experimental Procedure and Results for Solid Phase Extraction

The projections for an SPE operating facility to process and separate purified elements from Bokan leach solution is based on laboratory testing at bench scale. The two phase class separation was first tested and optimized with 12.5 mL and 25 mL (Column 1 and 2 respectively), and a 100 mL and 250 mL column, respectively. Column 1 takes in an acidic mixed REE solution containing substantial iron, uranium, and thorium impurities, and binds the iron, uranium, and thorium selectively while passing the mixed REEs. The nuisance metal depleted solution is then passed to Column 2. After rinsing, Column 1 is eluted with an ammonium citrate solution to remove the iron, uranium and thorium in a concentrated solution, rinsed, and then the next cycle begins. Column 2 receives the nuisance metal depleted feed from Column 1, and binds most of the iron, and all uranium, thorium, and REEs. The REEs are then stripped from the column with a weak acid (acetic acid stream), while the nuisance metals remain bound. These are eluted with acidic leach solution to form the feed for Column 1. Column 2 is then regenerated with base. Typical solution analyses (determined by ICP analysis) are shown in Table 13.14, for the two main output streams of the Class separation process, namely the ammonium citrate waste stream and the acetic acid mixed REE eluate stream.

Table 13.14 ICP Analysis on Typical Outputs from Class Separation Process at Bench Scale

	Ammonium Citrate Waste	Acetic Acid Eluate
Aluminum	33	50
Alk. Earths	-1	528
Iron	371	3
Rads (U+Th)	19	1
Manganese	0	59
Zirconium	63	1
TREE + Y	17	1,387

Note: Metal quantities in ppm

This process was run with the larger sized columns for multiple cycles to produce gallon quantities of acetic acid eluate stream. This could then be subjected to oxalic acid precipitation to provide a white, purified mixed REE solid, with the only significant impurity being calcium, and notably iron and the radioactive elements removed. These results are shown in Table 13.15.

Table 13.15 Solids Analysis of Precipitate Generated from Mixed REE Solution Produced by SPE

	Bulk Solution (ppt)	Purified Solution (ppt)
Iron (% in Fe_2O_3)	14.87	0.02
Thorium (% Th)	0.415	0.0027
Uranium (% U)	0.172	0.0018
REO+Y_2O_3 (%)	21.9	32.65
Other* (%)	15.11	6.415
Loss on Ignition (LOI) (%)	45.51	58.98
Other: *(Al_2O_3+CaO+SiO_2+TiO_2+Na_2O+MnO+MgO+K_2O+P_2O_5)		

Note: As compared to solid obtained by direct base precipitation of leach solution.

The next step of processing is to pass the acetic acid eluate through a series of four columns, which progressively deplete the rare earth elements, beginning with the heavy rare earths (terbium, dysprosium and heavier), followed by SEG+Y elements in the second column (samarium, europium, gadolinium, and yttrium), followed by neodymium+praseodymium, followed by cerium+lanthanum. These columns are then eluted with weak acid (0.4 M nitric acid) through a second column. The slow elution, followed by binding/release events on the second “amplifier” column results significant separation of the bound element fractions in the column. These fractions from the four column sets are recombined to form crude subclass solutions for precipitation.

Table 13.16 shows the percentage recovery of rare earths into one of the second column in some bench scale testing of the SEG binding column. Note that predictably, at higher loadings, less of the total metal is bound to the column. But also notable is the fact that the drop in recoveries at higher loadings is more rapid for lighter elements than heavier. This enable relative concentration of heavier elements in the column in preference to lights.

Table 13.16 Recovery of REE Subclasses During Various Loadings of SEG Concentrator Column in Second Depletion Step from Acetic Acid Mixed REE Stream

Bed Volume Load	Ce+La (%)	Pr+Nd (%)	Y (%)	Sm+Eu+Gd (%)	Tb+Dy (%)
3	99.62	99.44	98.72	99.46	96.11
4	99.85	99.86	99.75	100.01	97.54
6	99.57	99.77	99.73	99.92	98.20
8	98.43	99.49	98.21	99.84	98.45
12	90.86	96.44	92.91	99.18	98.72
24	43.29	67.33	58.70	90.45	95.39
32	34.80	59.53	58.66	86.86	95.08

Note: Metal quantities in flowthrough and eluate measured by ICP analysis

Table 13.17 shows analysis on the flowthrough samples during the acid elution of the heavy elements (terbium, dysprosium and heavier) concentrator column. These samples were collected subsequent to being passed through the amplifier column to increase separation of the elements. The samples show cuts that are enriched in cerium+lanthanum followed by neodymium+praseodymium+yttrium, followed by SEG+heavier elements. This demonstrates that the metal bound in the SEG binding column, already enriched in SEG elements relative to other REEs, can be further split into enriched columns using the weak acid elution/amplifier column effect. These results are also illustrated in Figure 13.5.

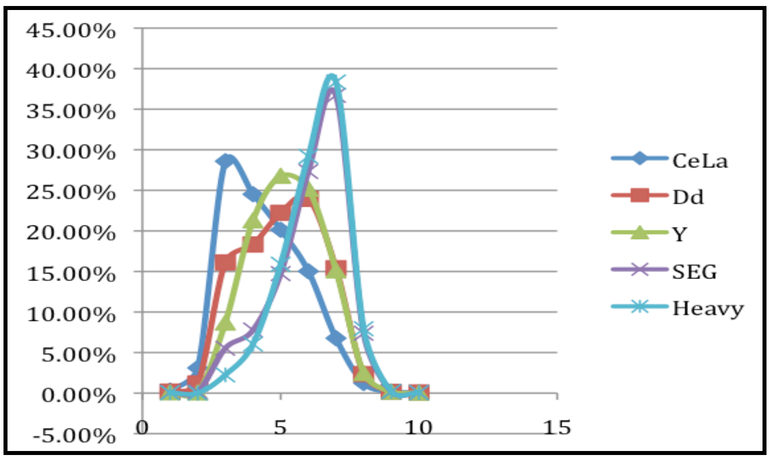
Subsequent separation of the crude subclasses into individual elements is projected based on bench scale testing of SPE processes with synthetic solutions designed to model solution conditions anticipated in the Bokan SPE separation process.

Table 13.17 Flowthrough Samples Measured During Weak Acid Elution of Heavy Concentrator Followed by Amplifier Column

Sample	Collected As	Ce/La (%)	Dd (%)	Y (%)	SEG (%)	Heavy (%)
1	Recycled Acid	0.32	0.19	0.10	0.10	0.05
2	Ce/La solution	3.11	1.20	0.08	0.19	0.01
3	Didymium rich	28.59	16.12	8.70	5.53	2.25
4	Didymium rich	24.52	18.32	21.30	7.78	6.05
5	Didymium rich	20.14	22.23	26.79	14.74	15.89
6	Raffinate	15.03	23.95	25.19	27.33	29.18
7	SEG/Heavy Rich	6.77	15.40	15.13	36.71	38.35
8	SEG/Heavy Rich	1.25	2.34	2.47	7.38	7.94
9	Recycled Acid	0.19	0.16	0.18	0.18	0.21
10	Recycled Acid	0.10	0.09	0.06	0.07	0.07

Notes: Dd = didymium
Represented as percent of total mass recovered of each subclass, as measured by ICP

Figure 13.5 Graphical Representation of Table 13.8 Data



Note: Showing separation of REE subclass peaks

14.0 MINERAL RESOURCE ESTIMATES

In November 2010, Aurora was commissioned to produce a model and resource estimate for the Dotson Zone on the Property. After completion of the estimate, a NI 43-101 compliant resource estimate was published in April 2011. The effective date of the published resource estimate is March 7, 2011.

Mr. Ronald James Robinson, P.Geol. of Aurora, used Surpac™ modelling and resource calculating software, v. 6.1.4, from Gemcom Software International Inc., of Vancouver, BC.

This section will include explanations of the process used to create the 3D model of the Dotson deposit and calculate and report on the resources present.

The first step in the resource calculation was to create a Surpac™ database for the Dotson Zone containing all parameters, definitions and calculations required for modelling and calculating the resource. The Surpac™ database definition is described in Table 14.1.

Table 14.1 Surpac Definition Database

Field	Type	Nulls	Length	Number Decimals	Low Bound	High Bound	Case	Valid Entries	Physical or Virtual
Assay									
hole_id	character	N	12	-	-	-	upper	-	physical
samp_id	character	V	10	-	-	-	upper	-	physical
depth_from	real	N	7	2	0	9999	-	-	physical
depth_to	real	N	7	2	0	9999	-	-	physical
x_from	real	N	11	3	-	-	-	-	calculated
x_to	real	N	11	3	-	-	-	-	calculated
y_from	real	N	11	3	-	-	-	-	calculated
y_to	real	N	11	3	-	-	-	-	calculated
z_from	real	N	11	3	-	-	-	-	calculated
z_to	real	N	11	3	-	-	-	-	calculated
la	real	V	10	2	0	99999	-	-	physical
ce	real	V	10	2	0	99999	-	-	physical
pr	real	V	10	2	0	99999	-	-	physical
nd	real	V	10	2	0	99999	-	-	physical
sm	real	V	10	2	0	99999	-	-	physical
eu	real	V	10	2	0	99999	-	-	physical
gd	real	V	10	2	0	99999	-	-	physical
tb	real	V	10	2	0	99999	-	-	physical
dy	real	V	10	2	0	99999	-	-	physical
ho	real	V	10	2	0	99999	-	-	physical
er	real	V	10	2	0	99999	-	-	physical
tm	real	V	10	2	0	99999	-	-	physical
yb	real	V	10	2	0	99999	-	-	physical

table continues…

Field	Type	Nulls	Length	Number Decimals	Low Bound	High Bound	Case	Valid Entries	Physical or Virtual
lu	real	V	10	2	0	99999	-	-	physical
y	real	V	10	2	0	99999	-	-	physical
nb	real	V	10	2	0	99999	-	-	physical
zr	real	V	10	2	0	99999	-	-	physical
th	real	V	10	2	0	99999	-	-	physical
u	real	V	10	2	0	99999	-	-	physical
lree	real	V	10	2	0	99999	-	-	physical
hree	real	V	10	2	0	99999	-	-	physical
tree	real	V	10	2	0	999999	-	-	physical
ta	real	V	10	2	0	999	-	-	physical
hf	real	V	10	2	0	999	-	-	physical
vein	character	V	10	-	-	-	mixed	-	physical
year	datetime	V	24	-	-	-	-	-	physical
Collar									
hole_id	character	N	12	-	-	-	upper	-	physical
hole_path	character	V	8	-	-	-	mixed	linear;curved;vertical;linear;curved;vertical	physical
max_depth	real	N	11	3	0	9999	-	-	physical
type	character	V	10	-	-	-	mixed	-	physical
x	real	N	11	3	-999999	9999999	-	-	physical
y	real	N	11	3	-999999	9999999	-	-	physical
year	character	V	10	-	-	-	mixed	-	physical
z	real	N	11	3	-999999	9999999	-	-	physical
Comp									
depth_from	real	N	11	3	0	9999	-	-	physical
depth_to	real	N	11	3	0	9999	-	-	physical
hole_id	character	N	12	-	-	-	upper	-	physical

table continues…

Field	Type	Nulls	Length	Number Decimals	Low Bound	High Bound	Case	Valid Entries	Physical or Virtual
p3	character	V	10	-	-	-	mixed	-	physical
samp_id	character	V	10	-	-	-	mixed	-	physical
x_from	real	N	11	3	-	-	-	-	calculated
x_to	real	N	11	3	-	-	-	-	calculated
y_from	real	N	11	3	-	-	-	-	calculated
y_to	real	N	11	3	-	-	-	-	calculated
z_from	real	N	11	3	-	-	-	-	calculated
z_to	real	N	11	3	-	-	-	-	calculated
Comp3									
depth_from	real	N	7	2	0	9999	-	-	physical
depth_to	real	N	7	2	0	9999	-	-	physical
hole_id	character	N	12	-	-	-	upper	-	physical
p3	character	V	10	-	-	-	mixed	-	physical
samp_id	character	V	10	-	-	-	upper	-	physical
x_from	real	N	11	3	-	-	-	-	calculated
x_to	real	N	11	3	-	-	-	-	calculated
y_from	real	N	11	3	-	-	-	-	calculated
y_to	real	N	11	3	-	-	-	-	calculated
z_from	real	N	11	3	-	-	-	-	calculated
z_to	real	N	11	3	-	-	-	-	calculated
Comp4									
depth_from	real	N	7	2	0	9999	-	-	physical
depth_to	real	N	7	2	0	9999	-	-	physical
hole_id	character	N	12	-	-	-	upper	-	physical
point4	character	V	10	-	-	-	mixed	-	physical
samp_id	character	V	10	-	-	-	upper	-	physical

table continues…

Field	Type	Nulls	Length	Number Decimals	Low Bound	High Bound	Case	Valid Entries	Physical or Virtual
x_from	real	N	11	3	-	-	-	-	calculated
x_to	real	N	11	3	-	-	-	-	calculated
y_from	real	N	11	3	-	-	-	-	calculated
y_to	real	N	11	3	-	-	-	-	calculated
z_from	real	N	11	3	-	-	-	-	calculated
z_to	real	N	11	3	-	-	-	-	calculated
Comp5									
depth_from	real	N	7	2	0	9999	-	-	physical
depth_to	real	N	7	2	0	9999	-	-	physical
hole_id	character	N	12	-	-	-	upper	-	physical
point5	character	V	10	-	-	-	mixed	-	physical
samp_id	character	V	10	-	-	-	upper	-	physical
x_from	real	N	11	3	-	-	-	-	calculated
x_to	real	N	11	3	-	-	-	-	calculated
y_from	real	N	11	3	-	-	-	-	calculated
y_to	real	N	11	3	-	-	-	-	calculated
z_from	real	N	11	3	-	-	-	-	calculated
z_to	real	N	11	3	-	-	-	-	calculated
Geology									
depth_from	real	N	7	2	0	9999	-	-	physical
depth_to	real	N	7	2	0	9999	-	-	physical
hole_id	character	N	12	-	-	-	upper	-	physical
litho	character	V	10	-	-	-	mixed	-	physical
samp_id	character	V	10	-	-	-	upper	-	physical
x_from	real	N	11	3	-	-	-	-	calculated
x_to	real	N	11	3	-	-	-	-	calculated

table continues…

Field	Type	Nulls	Length	Number Decimals	Low Bound	High Bound	Case	Valid Entries	Physical or Virtual
y_from	real	N	11	3	-	-	-	-	calculated
y_to	real	N	11	3	-	-	-	-	calculated
z_from	real	N	11	3	-	-	-	-	calculated
z_to	real	N	11	3	-	-	-	-	calculated
Intersect3									
depth_from	real	N	7	2	0	9999	-	-	physical
depth_to	real	N	7	2	0	9999	-	-	physical
hole_id	character	N	12	-	-	-	upper	-	physical
intersection	character	V	10	-	-	-	mixed	31;32;33	physical
samp_id	character	V	10	-	-	-	upper	-	physical
x_from	real	N	11	3	-	-	-	-	calculated
x_to	real	N	11	3	-	-	-	-	calculated
y_from	real	N	11	3	-	-	-	-	calculated
y_to	real	N	11	3	-	-	-	-	calculated
z_from	real	N	11	3	-	-	-	-	calculated
z_to	real	N	11	3	-	-	-	-	calculated
Intersect4									
depth_from	real	N	7	2	0	9999	-	-	physical
depth_to	real	N	7	2	0	9999	-	-	physical
hole_id	character	N	12	-	-	-	upper	-	physical
intersection	character	V	10	-	-	-	mixed	41;42;43	physical
samp_id	character	V	10	-	-	-	upper	-	physical
x_from	real	N	11	3	-	-	-	-	calculated
x_to	real	N	11	3	-	-	-	-	calculated
y_from	real	N	11	3	-	-	-	-	calculated
y_to	real	N	11	3	-	-	-	-	calculated

table continues…

Field	Type	Nulls	Length	Number Decimals	Low Bound	High Bound	Case	Valid Entries	Physical or Virtual
z_from	real	N	11	3	-	-	-	-	calculated
z_to	real	N	11	3	-	-	-	-	calculated
Intersect5									
depth_from	real	N	7	2	0	9999	-	-	physical
depth_to	real	N	7	2	0	9999	-	-	physical
hole_id	character	N	12	-	-	-	upper	-	physical
intersection	character	V	10	-	-	-	mixed	51;52;53	physical
samp_id	character	V	10	-	-	-	upper	-	physical
x_from	real	N	11	3	-	-	-	-	calculated
x_to	real	N	11	3	-	-	-	-	calculated
y_from	real	N	11	3	-	-	-	-	calculated
y_to	real	N	11	3	-	-	-	-	calculated
z_from	real	N	11	3	-	-	-	-	calculated
z_to	real	N	11	3	-	-	-	-	calculated
Rad									
depth_to	real	N	7	2	0	9999	-	-	physical
hole_id	character	N	12	-	-	-	upper	-	physical
rad	real	V	10	2	0	9999	-	-	physical
x_to	real	N	11	3	-	-	-	-	calculated
y_to	real	N	11	3	-	-	-	-	calculated
z_to	real	N	11	3	-	-	-	-	calculated
Styles									
code	character	V	20	-	-	-	mixed	-	physical
field_name	character	N	18	-	-	-	lower	-	physical
from_value	character	V	23	-	-	-	mixed	-	physical
graphics_colour	character	N	32	-	-	-	mixed	-	physical

table continues…

Field	Type	Nulls	Length	Number Decimals	Low Bound	High Bound	Case	Valid Entries	Physical or Virtual
graphics_pattern	character	N	5	-	-	-	mixed	-	physical
line_colour	character	N	32	-	-	-	mixed	-	physical
line_style	character	N	32	-	-	-	mixed	-	physical
line_weight	integer	N	4	-	1	9	-	-	physical
marker_size	real	N	4	2	0.01	9	-	-	physical
marker_style	character	N	16	-	-	-	mixed	-	physical
plotting_colour	character	N	32	-	-	-	mixed	-	physical
plotting_pattern	character	N	16	-	-	-	mixed	-	physical
style_type	character	N	1	-	-	-	upper	-	physical
table_name	character	N	18	-	-	-	lower	-	physical
to_value	character	V	23	-	-	-	mixed	-	physical
Survey									
azimuth	real	N	6	2	0	360	-	-	physical
depth	real	N	7	2	0	9999	-	-	physical
dip	real	N	6	2	-90	90	-	-	physical
hole_id	character	N	12	-	-	-	upper	-	physical
x	real	N	11	3	-	-	-	-	calculated
y	real	N	11	3	-	-	-	-	calculated
z	real	N	11	3	-	-	-	-	calculated
Translation									
code	character	N	6	-	-	-	mixed	-	physical
description	character	V	32	-	-	-	mixed	-	physical
field_name	character	N	18	-	-	-	mixed	-	physical
num_equiv	real	N	8	2	-999999	9999999	-	-	physical
table_name	character	N	18	-	-	-	mixed	-	physical

The next phase of the modelling process involved importing drillhole and channel sample data from the Project drillhole database into the Surpac™ database. The data in the Project database had been subject to a rigorous verification and QA/QC process as described in Section 12.0. When importing data into a Surpac™ database, the software performs several quality checks on the data. These checks include checking for duplicate sample numbers, duplicate intervals, overlaps of sample intervals, positive sample length, and samples beyond the end of the hole.

After the data was imported, the drillholes were rotated and translated into a grid coordinate system to facilitate modelling of the structures parallel to the zone. The base point for the rotation was the south end of trench TR10-11 (UTM NAD 83-7:683830.1E, 6088598.2N) and it was given grid coordinates 1000, 1000. The holes were rotated clockwise 340.7°.

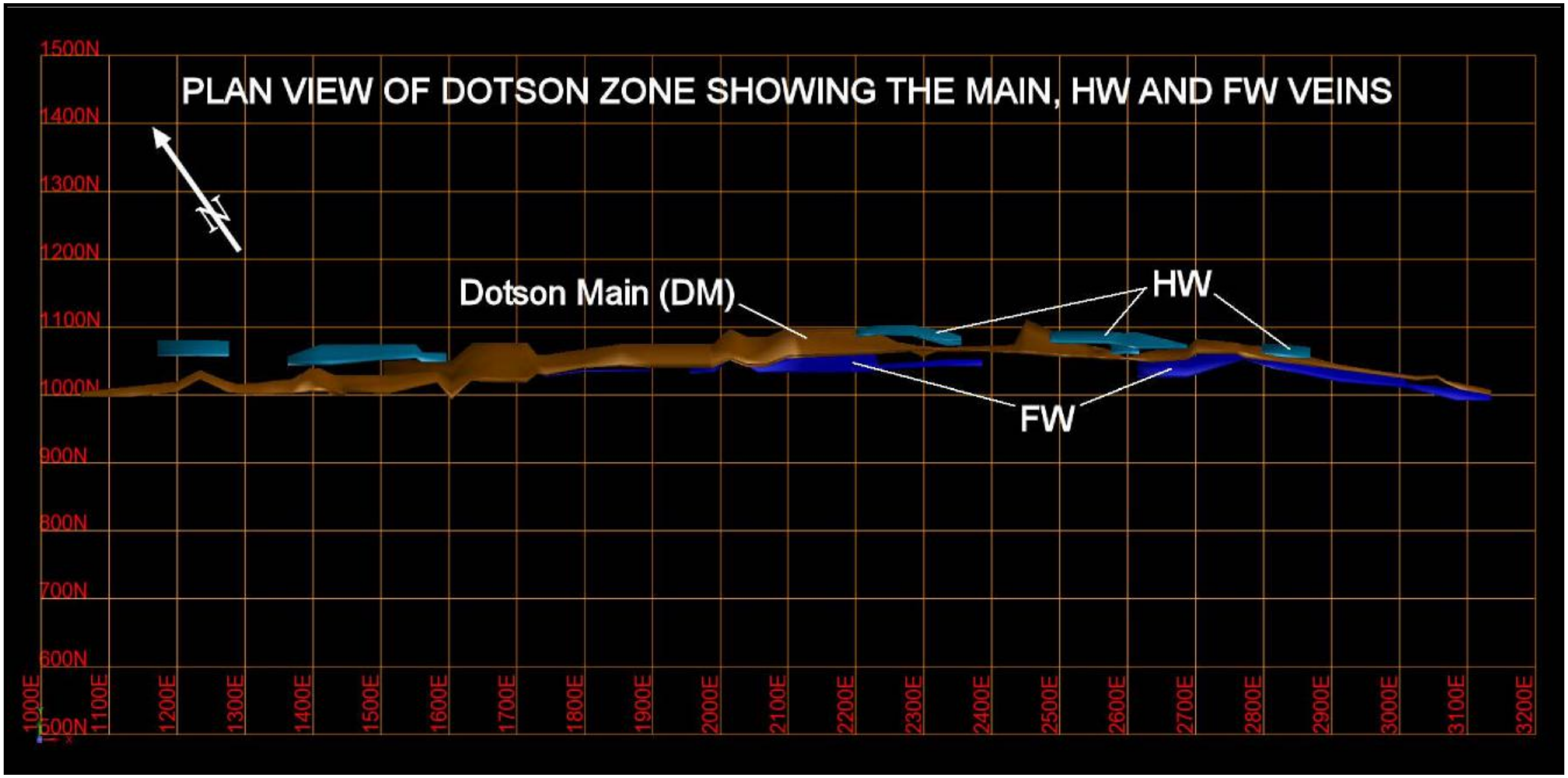
The assay data was composited to allow for statistical analysis using the Surpac™ composite downhole function with the following parameters:

- composite length: 0.5 m
- composite length determined by: fixed length
- minimum percent of sample to be included: 75
- dilute negative samples: yes.

The composited TREE values were then displayed and manually edited to produce composite values with a minimum true width of 1.5 m and a minimum value of 0.40% TREE.

Cross-sections were then digitized to include those composites meeting the minimum parameters and the sectional polygons were triangulated to create 3D models of the various structures. In the Dotson Zone, it was apparent that there were three vein structures which could be modelled. The veins in the Dotson Zone are very straight and planar down dip and quite linear along strike with only minor flexures, as can be seen in Figure 14.1. The 3D models were inspected and modified to ensure that they matched all field observations and mapping. Intersections, between drillholes and the 3D models, were calculated to produce intersection tables in the database. Assay values were then composited within each structure within the Dotson Zone. Simple statistical analyses were performed on each element, in each zone and structure, to determine top cut limits. As no mining or reconciliation studies between forecast grade and recovered product have been performed on the deposit, and no comparisons can be drawn to other producing rare earth mines, the 95^{th}% confidence level for each element was used as the top cut. The 95^{th}% confidence is calculated using the formula CL=mean+(1.96*SD). The composited assay values for each element in each zone and structure were subjected to variogram analysis to determine appropriate search ellipsoids and variogram models for block model estimation.

Figure 14.1 Plan View of Dotson Zone Showing Main, Hanging Wall (HW) and Foot Wall (FW) Structures



The search ellipsoids for each element in each structure in the Dotson Zone were the same within the margins of modeling error, so the same ellipsoid was used for all estimations. The ellipsoid is defined as follows: bearing=0, pitch=85, dip=0, major/semi-major anisotropy ratio=1, and major/minor ratio=5.

In the Dotson Zone, variograms were modeled for each element in the Dotson Main structure and for several elements in the FW and HW zones. There was little difference between the variograms for a particular element between the different structures. As such, those variograms calculated for the Dotson Main structure were used for the FW and HW structures as well.

Separate block models were constructed for the Dotson and the I&L Zones. The table definitions for the two zones are listed in the Table 14.2 (Dotson Zone) and Table 14.3 (I&L Zone). The Dotson Zone minimum block size is 1 m x 1 m x 1 m. An attempt was made to utilize 0.5 m blocks due to the narrow nature of the veins, but the extensive length and depth of the deposit caused the block model and associated file to become unmanageably large.

Once the block models were created, and the variography and isotropy determined, the block models were estimated using ordinary kriging (OK). The three structures in the Dotson Zone were estimated separately, constrained by the appropriate 3D model for each estimation. This prevented the kriging routine from using intersections from adjacent veins when calculating individual block grades.

During the 2010 program, 79 density measurements were taken on samples of mineralized vein material in the Dotson Zone and on adjacent wall rock and distal country rock samples. The average density of the vein material (2.77 + 0.12 g/cm^3) and the country rock (2.77 + 0.12 g/cm^3) are consistent with tabulated values for this material reported elsewhere (Paul et. al. 2009; Telford et.al. 1990). The density readings were analyzed for possible correlations between density and depth, easting, rock type, assay grade or vein location (HW, DM, or FW). No correlations of any statistical significance were found and a simple average of the density measurements was used. The statistical study for the density data is found in Appendix D.

The resource estimates were reported from the block model. Mineral resource estimates for the Dotson Zone are reported in Table 14.3. Although the drill density and control in the Dotson Zone would have allowed a significant percentage of the material to be classed as Indicated, these resources were downgraded to the Inferred class at the request of Ucore.

The rationale for this decision was that with six cut-off grades and two resource classes, the information would be too confusing for Ucore’s shareholders. At the time, the Project was in a fairly early stage of exploration, and it was felt that the need to understand the numbers outweighed the desire for a higher confidence-level resource.

The various definitions and formulae used to calculate the oxide percentages from the elemental concentrations, and light, heavy and total rare earth element and oxide concentrations from the respective REE and REO values are illustrated in Table 14.2.

Table 14.2 Dotson Block Model Definition

Type	Y	X	Z
Minimum Coordinates	950	1,000	-300
Maximum Coordinates	1,206	3,176	404
User Block Size	4	4	4
Minimum Block Size	1	1	1
Rotation	0	0	0
Total Blocks	1,921,218	-	-
Storage Efficiency %	99.51	-	-

Attribute Name	Type	Decimals	Background	Description
Aniso_dist	Real	3	-99	-
Ave_dist	Real	3	-99	-
Ce	Real	2	0	-
Ce_2O_3	Calculated	-	-	ce*0.000117128
Dy	Real	2	0	-
Dy_2O_3	Calculated	-	-	dy*0.000114769
Er	Real	2	0	-
Er_2O_3	Calculated	-	-	er*0.000114348
Eu	Real	2	0	-
Eu_2O_3	Calculated	-	-	eu*0.000115793
Gd	Real	2	0	-
Gd_2O_3	Calculated	-	-	gd*0.000115262
Hf	Real	2	0	-
Ho	Real	2	0	-
Ho_2O_3	Calculated	-	-	ho*0.000114551
HREE	Calculated	-	-	eu+gd+tb+dy+ho+er+tm+yb+lu+y
HREO	Calculated	-	-	eu2o3+gd2o3+tb2o3+dy2o3+ho2o3+ er2o3+tm2o3+yb2o3+lu2o3+y2o3
La	Real	2	0	-
La_2O_3	Calculated	-	-	la*0.000117277
LREE	Calculated	-	-	la+ce+pr+nd+sm
LREO	Calculated	-	-	la2o3+ce2o3+pr2o3+nd2o3+sm2o3
Lu	Real	2	0	-
Lu_2O_3	Calculated	-	-	lu*0.000113716
Nb	Real	2	0	-
Nd	Real	2	0	-

table continues…

Attribute Name	Type	Decimals	Background	Description
Nd_2O_3	Calculated	-	-	nd*0.000116638
No_samples	Integer	-	-99	-
Pr	Real	2	0	-
Pr_2O_3	Calculated	-	-	pr*0.000117032
SG	Float	2	2.77	-
Sm	Real	2	0	-
Sm_2O_3	Calculated	-	-	sm*0.000115961
Ta	Real	2	0	-
Tb	Real	2	0	-
Tb_2o_3	Calculated	-	-	tb*0.000115101
Th	Real	2	0	-
Tm	Real	2	0	-
Tm_2O_3	Calculated	-	-	tm*0.000114206
TREE	Real	2	0	-
Treecalc	Calculated	-	-	lree+hree
TREO	Calculated	-	-	lreo+hreo
U	Real	2	0	-
Variance	Real	3	-99	-
Y	Real	2	0	-
Y_2O_3	Calculated	-	-	y*0.000126994
Yb	Real	2	0	-
Yb_2O_3	Calculated	-	-	yb*0.000113869
Zr	Real	2	0	-
Block Model Summary 1/1			-	-

Source: Gemcom Software International (April 18, 2011)

Notes: Block Model Summary bm\point4.mdl

Dotson veins digitized at 0.40% TREE

Table 14.3 Resource Estimate for Dotson Zone, Bokan Mountain Property, Alaska (All Resources Classified as Inferred)

TREO Cut-off	Tonnes	LREE (ppm)	HREE (ppm)	TREE (ppm)	La (ppm)	Ce (ppm)	Pr (ppm)	Nd (ppm)	Sm (ppm)	Eu (ppm)	Gd (ppm)	Tb (ppm)	Dy (ppm)	Ho (ppm)	Er (ppm)	Tm (ppm)	Yb (ppm)	Lu (ppm)	Y (ppm)
0.20	6,579,100	2,947.66	1,938.28	4,885.94	469.82	1,425.63	175.95	692.69	183.57	18.53	187.42	34.08	211.85	41.43	115.73	15.68	87.26	10.30	1,216.00
0.30	6,054,200	3,125.27	2,015.84	5,141.10	503.00	1,508.78	183.25	736.25	193.98	19.50	195.40	35.44	220.14	42.99	119.77	16.21	90.24	10.61	1,265.54
0.40	**5,228,200**	**3,368.77**	**2,113.50**	**5,482.25**	**552.09**	**1,620.87**	**195.26**	**792.53**	**208.02**	**20.84**	**207.65**	**37.47**	**231.58**	**44.99**	**124.47**	**16.73**	**92.78**	**10.80**	**1,326.17**
0.50	3,637,200	3,931.47	2,320.19	6,251.66	656.46	1,883.34	221.00	931.26	239.41	23.71	231.31	41.17	253.56	48.99	134.35	17.98	99.58	11.40	1,458.14
0.60	2,465,200	4,338.38	2,647.04	6,985.42	742.68	2,058.56	238.18	1031.69	267.28	26.59	259.02	46.22	286.89	55.71	153.59	20.66	115.64	13.15	1,669.56
0.70	1,531,200	5,105.96	2,857.25	7,963.22	906.03	2,411.54	275.67	1206.27	306.46	29.62	286.25	50.75	312.93	60.26	164.52	22.13	119.93	13.93	1,796.93
0.80	1,005,700	5,763.34	3,070.29	8,833.62	1,015.51	2,719.76	310.05	1369.47	348.55	33.53	318.21	55.70	337.87	64.20	172.65	22.99	123.59	14.22	1,927.32

table continues…

TREO Cut-off	Tonnes	LREO (%)	HREO (%)	TREO (%)	La_2O_3 (%)	Ce_2O_3 (%)	Pr_2O_3 (%)	Nd_2O_3 (%)	Sm_2O_3 (%)	Eu_2O_3 (%)	Gd_2O_3 (%)	Tb_2O_3 (%)	Dy_2O_3 (%)	Ho_2O_3 (%)	Er_2O_3 (%)	Tm_2O_3 (%)	Yb_2O_3 (%)	Lu_2O_3 (%)	Y_2O_3 (%)
0.20	6,579,100	0.345	0.237	0.582	0.055	0.167	0.021	0.081	0.021	0.002	0.022	0.004	0.024	0.005	0.013	0.002	0.010	0.001	0.154
0.30	6,054,200	0.366	0.247	0.612	0.059	0.177	0.021	0.086	0.022	0.002	0.023	0.004	0.025	0.005	0.014	0.002	0.010	0.001	0.161
0.40	**5,228,200**	**0.394**	**0.259**	**0.653**	**0.065**	**0.190**	**0.023**	**0.092**	**0.024**	**0.002**	**0.024**	**0.004**	**0.027**	**0.005**	**0.014**	**0.002**	**0.011**	**0.001**	**0.168**
0.50	3,637,200	0.460	0.284	0.744	0.077	0.221	0.026	0.109	0.028	0.003	0.027	0.005	0.029	0.006	0.015	0.002	0.011	0.001	0.185
0.60	2,465,200	0.507	0.324	0.832	0.087	0.241	0.028	0.120	0.031	0.003	0.030	0.005	0.033	0.006	0.018	0.002	0.013	0.001	0.212
0.70	1,531,200	0.597	0.350	0.947	0.106	0.282	0.032	0.141	0.036	0.003	0.033	0.006	0.036	0.007	0.019	0.003	0.014	0.002	0.228
0.80	1,005,700	0.674	0.376	1.050	0.119	0.319	0.036	0.160	0.040	0.004	0.037	0.006	0.039	0.007	0.020	0.003	0.014	0.002	0.245

table continues…

TREO Cut-off	Tonnes	Nb (ppm)	Zr (ppm)	Th (ppm)	U (ppm)
0.20	6,579,100	322.83	1,678.56	72.97	57.96
0.30	6,054,200	345.23	1,785.93	77.37	61.85
0.40	**5,228,200**	**380.67**	**1,836.90**	**80.52**	**66.23**
0.50	3,637,200	444.88	1,874.55	87.50	73.65
0.60	2,465,200	496.01	2,207.29	98.30	88.45
0.70	1,531,200	577.86	2,569.71	113.92	108.69
0.80	1,005,700	628.65	2,692.68	120.31	119.67

Notes: Dotson All Zones Ordinary Kriging Estimation Method 0.40% Digitizing Cut-off

Constraints used (Dotson)

a. INSIDE 3DM SOLIDS/ALL_ZONES_POINT 4

b. NOT ABOVE DTM topo/topo_rotated1.dtm Object ID 4 Trisolation ID 1

Constraint Expression: a and b

Keep blocks partially in the constraint: True

15.0 MINERAL RESERVE ESTIMATES

A mineral reserve is the economically mineable part of a Measured or Indicated Mineral Resource. A mineral reserve has not been estimated for the Project as part of this PEA.

16.0 MINING METHODS

16.1 OVERVIEW

Stantec developed a mine plan based on 1,500 t/d mining rate for 360 operational days each year, for a throughput of 540,000 t/a. The mine plan involves underground mining using a blasthole stoping mining method. The underground mine will be progressively developed over 11 years, 2 of which will be devoted to pre-production (Year -2 and Year -1). Material processing activities will begin in Year 1.

Underground development is the underground excavation and construction activitiy required to access the minable material. Development activity will begin with the excavation of 319 and 3,289 linear metres in Year -2 and Year -1, respectively. Activities will increase to 4,763 linear metres in Year 1, and generally decrease over the following eight years to 690 linear metres in Year 9.

Further details of the proposed mining method for the Project are provided in Stantec's mine report located in Appendix E. All information in this section is summarized from the report.

16.2 MINE PLAN

Using Taylor's Formula, Stantec estimated the daily production requirements for the Project resource and established a mining rate of approximately 1,500 t/d. Assuming a 360 d/a operation, Stantec evaluated three different mining methods for the Project: shrinkage stoping, overhand cut-and-fill, and blasthole stoping with paste backfill. For the purposes of this study, Stantec selected blasthole stoping with paste backfill, based on its economic advantages and the geometry of the host rock body.

16.2.1 MINING METHOD SELECTION

Stantec evaluated several mining methods, including shrinkage stoping, overhand cut-and-fill, and blasthole stoping with paste backfill. The mining method selection and design are based on the following observations:

- The mining method must be appropriate for a high-angle, narrow-vein deposit.
- The host rock grade material is contained within the vein, and the host rock will be waste.

- Paste backfill will be the preferred backfill method.
- Both the host rock and the waste are generally competent.
- Host rock and waste are easily differentiated visually.

The shrinkage stoping method requires broken host rock to remain in the stopes as a work platform until the stope is mined bottom-up to the crown pillar, therefore deferring revenue from the blasted host rock. Mining can be very selective with this method; this minimizes dilution, but working areas are isolated and difficult to mechanize. Additionally, work areas are difficult to ventilate and generally yield low productivity rates.

Overhand cut-and-fill using mill tailings as the main component in the paste backfill is advantageous because of the selectivity in mining. However, this method is costly due to increased host rock and waste development, slower production rate, and increased backfilling costs associated with construction requirements to backfill each drift.

The blasthole stoping method with paste backfill can be performed in numerous ways. The restrictions imposed by the geotechnical recommendations eliminate many of the variations that employ multiple levels and result in large, unsupported openings between the production phase and the backfill phase. Blasthole stoping relies on relatively long drillholes, which gives higher productivity and lower operating costs than overhand cut-and-fill or shrinkage stoping. The longer holes result in less selectivity, so host rock loss and dilution are increased within the planned stope boundaries. Increasing the vertical interval between levels decreases the development and production costs, but increases the potential dilution. The degree of sinuosity in the vein boundary and the accuracy with which it is located in advance of mining have a direct effect on the dilution that results in production. Backfilling the exhausted stopes with either paste backfill/tailings or waste rock is required for overall stability. The backfill must include cement for primary stopes where the secondary stope is adjacent and mined subsequently.

Based on this inspection and review of the geometry of the host rock body, Stantec determined that blasthole stoping is the most reasonable mining method for this conceptual level of study. Further evaluation should be performed in any studies going forward to ensure optimization of underground mining.

16.2.2 Geotechnical Parameters

Ucore did not provide Stantec with any geotechnical information. All geotechnical assumptions are based on a visual inspection of drill core and outcropping veins conducted in November 2011 by Stantec's geotechnical QP. The rock mass can be classified as "Fair" to "Good" with little evidence of large-scale alteration around the mineralized zone.

Stantec developed anticipated drift dimensions of 4.5 m wide by 4.5 m high for all waste development, and then developed ground support packages. Host rock drifts will be 3.5 m wide by 3.5 m high. The drift dimensions were established by determining the smallest possible excavation to accommodate the required mining equipment to sustain the production rate.

Stantec recommends two ground support packages: one for ground classified as “Good”, and another for ground classified as “Fair”. For estimating purposes, an average of both packages was used to calculate the development costs. These packages include waste development and host rock development. Table 16.1 presents the ground support package for each development drift size for “Good” ground conditions.

In order to minimize potential dilution because of the undulations in the veins, Stantec has restricted the the level spacing to 20 m sill-to-sill. The drilling accuracy declines as the level spacing increases, and typically, 20 m is a good starting point for a conceptual mine design. The stope mining segment length is set at 50 m for production cycle calculations and paste backfill requirements.

The current design leaves a 20 m high crown pillar in place between surface and the uppermost mining level. This crown pillar is conceptual; further geotechnical information will be required to identify the optimal thickness of the crown pillar.

Sill pillars (20 m high) will be used between the mining zones, where required. The schedule assumes the pillars will be fully recovered.

Stantec inspected the proposed portal location and found suitable for the mine plan.

Table 16.1 Development Ground Support Package

Type	Height (m)	Width (m)	Bolt Type	Bolt Length (m)	Bolt Spacing (m)	Wire Mesh (Yes/No)	Shotcrete	Shotcrete Thickness (mm)	"Good" Ground Assumption (%)	Classification
Main Access Decline	4.5	4.5	Resin Rebar Bolt #7	1.8	1.8	No	Yes	50	70	Main Development
Internal Ramps	4.5	4.5	Resin Rebar Bolt #7	1.8	1.8	No	-	-	70	Main Development
Footwall Drifts	4.5	4.5	Resin Rebar Bolt #7	1.8	1.8	No	-	-	70	Main Development
Level Access Drives	4.5	4.5	SS-39 Split Sets	1.8	1.8	Yes	-	-	70	Secondary Development
Host Rock Drive Access	4.5	4.5	SS-39 Split Sets	1.8	1.8	Yes	-	-	-	Secondary Development
Host Rock Drive_1	3.5	3.5	SS-39 Split Sets	1.5	1.5	Yes	-	-	-	Cross-cut
Host Rock Drive_2	3.5	4.5	SS-39 Split Sets	1.5	1.5	Yes	-	-	-	Cross-cut
Host Rock Drive_3	3.5	6.0	SS-39 Split Sets	1.5	1.5	Yes	-	-	-	Cross-cut

16.2.3 MINING METHOD DESCRIPTION

Two variations of the blasthole stoping method are generally utilized to maximize resource extraction: transverse and longitudinal. For this study, transverse stopes (perpendicular to strike) were not considered due to the narrow width of the host rock body. One size of longitudinal stopes (parallel to strike) was developed and used for evaluation: 20 m high (sill to sill). The width of the longitudinal stopes will up to 3.5, depending on the width of the vein. The total length for a stoping area between accesses to the footwall drift will be 300 m. The stope segment length will 50 m. Further work should be performed to optimize the methods and designs of the stopes specific to each area of the resource in the next level of study.

Each stope includes a top cut (or drill drift) at the top of the stope and an undercut (or extraction drift) at the bottom of the stope. Top cuts and undercuts are developed by driving a host rock access drift from the footwall drift to the host rock body and then host rock drifts are driven parallel to the footwall drift. In the areas where there are multiple vein sets and a primary/secondary stoping arrangement exists, stopes are mined from the hanging wall to the footwall. Once the extraction drift and drill drift are completed, a slot raise of 3.0 m by 2.0 m is developed between the two drifts. The drill plan for the slot raise is a general diamond formation utilizing 152 mm diameter reamer holes in the burn cut. Production blastholes (63.5 mm diameter) are drilled vertically from the drill drift with a 2.0 m burden and 0.7 m spacing pattern. Blastholes are loaded with ammonium nitrate fuel oil (ANFO) explosives, leaving 1.5 m available for stemming. Figure 16.1 illustrates a typical production drill and blast plan for a 3.5 m wide stope. Figure 16.2 shows an isometric view of a 50 m stope segment with planned drillholes for a typical 3.5 m wide stope.

A blast fragmentation prediction was prepared for the drill and blast plan for a typical 3.5 m wide stope utilizing the Kuz-Ram Model. The results predict approximately 80% of the material blasted will be within 0 to 0.20 m in side length. Table 16.1 presents the results for the blast fragmentation prediction. Figure 16.3 illustrates the fraction passing results in a histogram format.

Production LHD vehicles with 5.6 m^3 capacity buckets will muck the blasted host rock from the extraction level into haul trucks or dump the host rock into a host rock pass depending on stope location. The LHDs will be equipped with remote control capabilities allowing the LHD to muck inside open stopes while the operator remains in a safe location outside the stope. The majority of the production mucking will be performed remotely.

Figure 16.1 Cross Section of Typical 3.5 m Wide Stope Drill and Blast Design

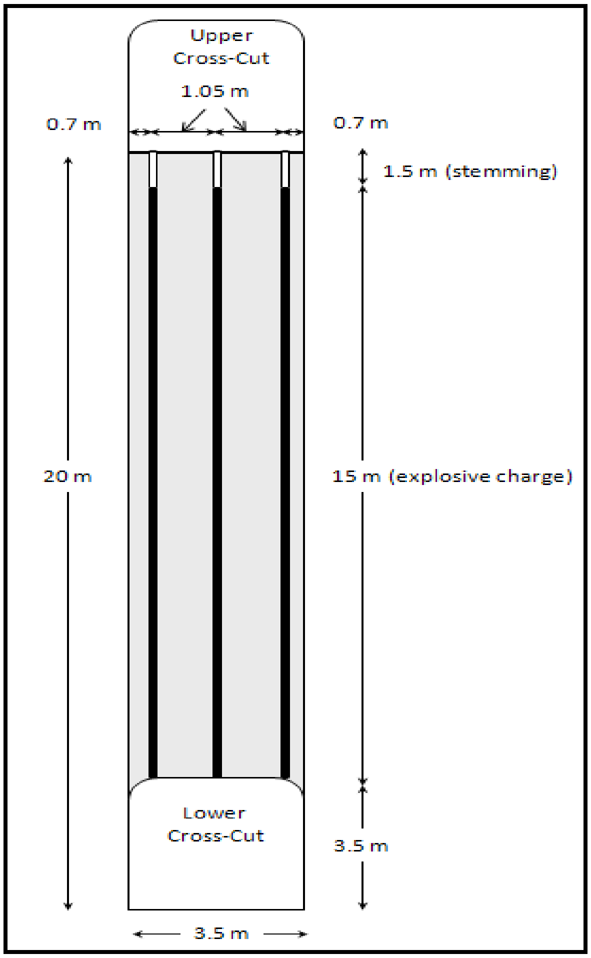


Figure 16.2 Isometric View of Typical 3.5 m Wide Stope Drill Plan

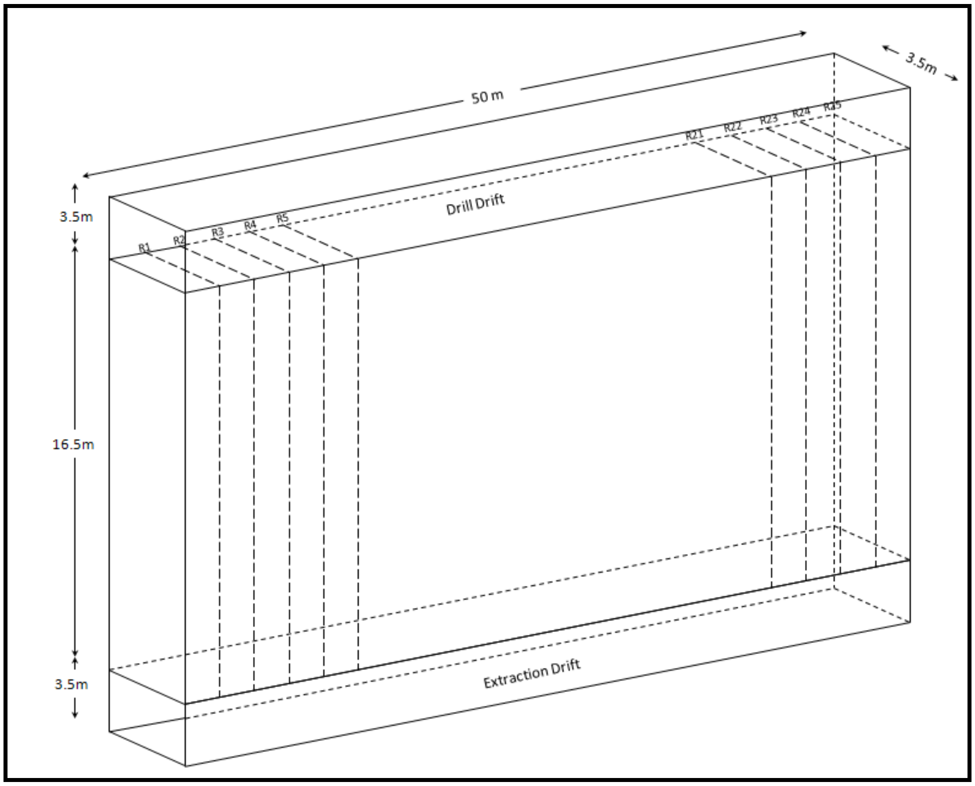
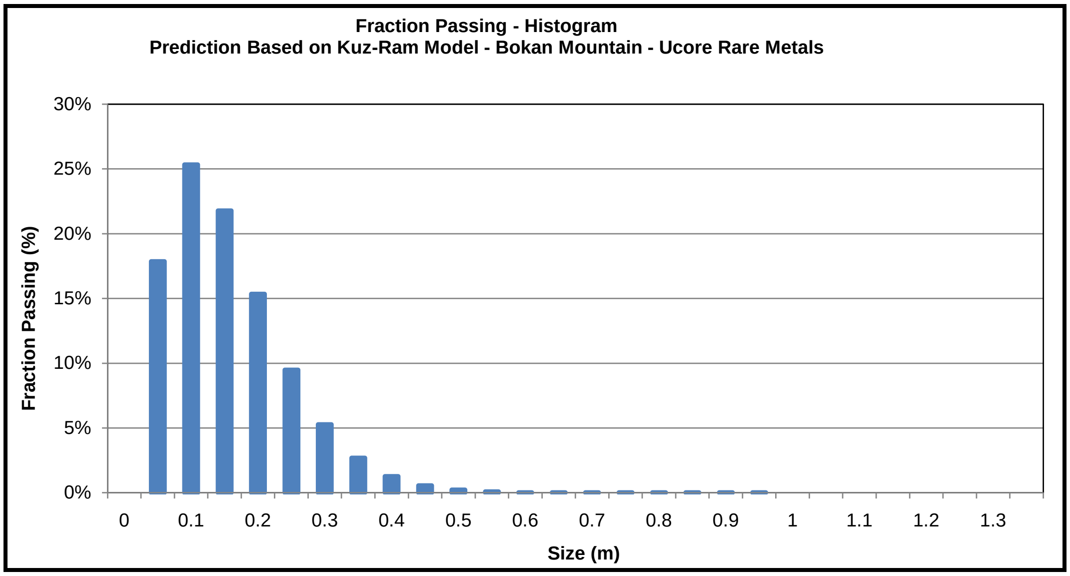


Table 16.2 Blast Fragmentation Prediction (Kuz-Ram Model)

Percent Passing (%)	Fraction Passing (%)	Size (m)	Size (ft)
0.0	0.0	0.00	0.00
17.9	17.9	0.05	0.16
43.2	25.3	0.10	0.33
65.0	21.8	0.15	0.49
80.3	15.4	0.20	0.66
89.8	9.5	0.25	0.82
95.1	5.3	0.30	0.98
97.8	2.7	0.35	1.15
99.1	1.3	0.40	1.31
99.6	0.6	0.45	1.48
99.9	0.2	0.50	1.64
99.9	0.1	0.55	1.80
100.0	0.0	0.60	1.97

Note: Stope is 3.5 m wide

Figure 16.3 Blast Fragmentation Estimates



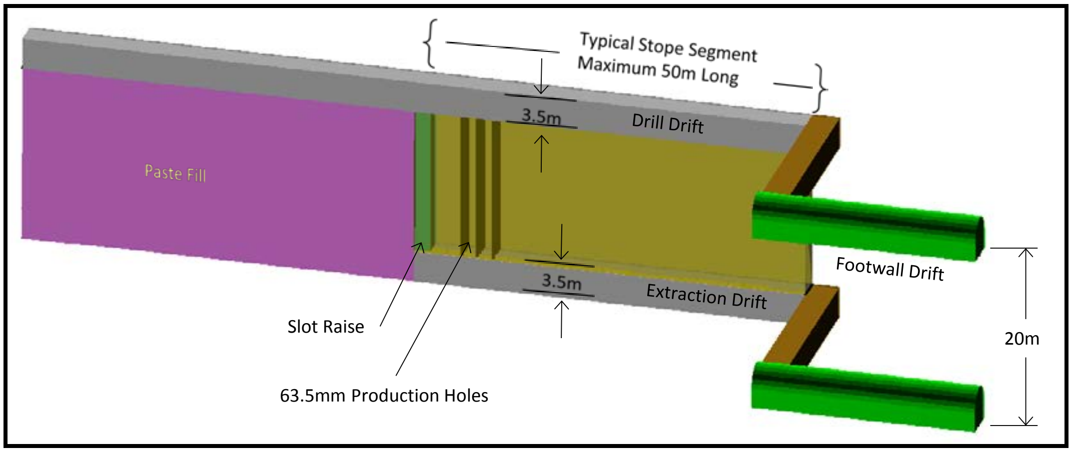
Stantec selected paste backfill as the preferred backfill method. This method allows the disposal of mill tailings underground with only a small tailings impoundment to temporarily store tailings in Year 1, for backfilling in Year 2. Further work is necessary to evaluate the planned grind size and tailings material used to produce a component paste backfill that meets necessary strength requirements for underground mining. Further work is also necessary to project an overall mass balance between the material mined and resulting tailing available for paste backfill.

Waste rock can supplement the paste backfill in areas where backfill strength is not critical to the mining sequence. This option was not considered in establishing operating costs associated with backfill truck selection. The paste backfill mix design was not altered for areas that require underhand mining method. Further work is required to properly evaluate the paste fill mix and associated backfill cost.

After a 50 m stope segment is mined and cleaned out, a bulkhead will be built in the extraction drift before and backfilling commences. Backfill will be delivered to the stopes through a series of boreholes and pipes connected to the paste plant on surface. Each stope will be filled to within 3.5 m of the stope back, except in areas where underhand mining is planned. The remaining opening will be used as the extraction level for the stope above. After the backfill sets for approximately 14 days, slotting will begin on the following stope segment. If the stope has already been mined, the stope above will be mined.

Figure 16.4 provides an isometric view of a typical 20 m high stope segment.

Figure 16.4 Typical 20 m High Blasthole Stope



Previous similar projects suggests that an individual blasthole stope can produce an average of 500 t/d of total tonnes from the stope which includes mineralized material and planned dilution. This average includes critical development requirements, production drilling, production loading and blasting, production mucking, paste backfilling, and delay allowances. Based on this average, a 540,000 t/a operation would require approximately three active stopes to be in production at any given time. The number of active stopes required to sustain production is minimal; however, additional engineering and design effort will be required to confirm that the geometry of the deposit and the designed stopes support this assumption.

16.3 DEVELOPMENT PLAN

General assumptions related to lateral and vertical development rates and dimensions include.

- All waste development will be divided into two categories: main development and secondary development.
- All waste development drifts are 4.5 m wide by 4.5 m high.
- All ramps will have a maximum grade of 15%.
- A 12% miscellaneous development factor is applied to all development meters to account for development that is not included in a conceptual design, such as sumps, mine load center drifts, etc.
- A 10% overbreak factor is applied to all waste development tonnes.
- All development performance rates are considered single-heading rates.
- Development rates for the contractor are the same for the owner.

Table 16.3 presents each development type, dimensions, and performance rate utilized in the development schedule.

Table 16.3 Development Details

Dimensions	Type	Rate	Unit
4.5 m wide by 4.5 m high	Main Development – Access	5.7	m/d
4.5 m wide by 4.5 m high	Main Development	6.7	m/d
4.5 m wide by 4.5 m high	Secondary Development	6.6	m/d
3.5 m wide by 3.5 m high	Host Rock Development	7.7	m/d
4.5 m wide by 3.5 m high	Host Rock Development	7.1	m/d

16.3.1 MAIN ACCESS DECLINE

The underground mine will be accessed by a decline that will be located from the portal near the proposed plant location to a location on the footwall side of the Dotson Vein structure at the -20.0 m elevation on the eastern side of the deposit on strike. Ucore provided the portal location. The decline will be 4.5 m wide, 4.5 m high, and 289 m long. An additional 12% of total decline metres has been added as an allowance to account for muck bays and miscellaneous development requirements. The costs associated with the main access decline and associated infrastructure are included in the preproduction costs provided in Appendix E.

16.3.2 DEVELOPMENT LAYOUT

The mine development design for Bokan Mountain consists of footwall drives developed approximately 20 m from the host rock body and driven parallel to the strike. The footwall drives are on 20 m level spacing and act as the main connection between mining zones for material handling and ventilation. The footwall drives are designed to end 300 m from the end extents of the host rock body in an attempt to reduce waste development. Internal ramps designed at 25 m radius and ±15% grade connect the upper and lower levels of the host rock body. Also included in the overall conceptual development design are level access drives, crosscut access drives, and ventilation access drives. Table 16.4 presents a development summary by development type and location which includes total meters and tonnes.

Figure 16.5 presents an isometric view of the mine layout. Figure 16.6 and Figure 16.7 provides a plan view and long section view of the mine design with stope and host rock crosscut shapes.

Table 16.4 Development Summary

Type	Name/Location	Development (m)	Miscellaneous Development Allowance (m)	Total Development (m)	Total Tonnes
Access Decline	Main Decline	285	34	319	19,695
Internal Ramp	Lower East Ramp	1,664	200	1,864	114,992
	Central Ramp	735	88	823	50,793
	Lower West Ramp	1,132	136	1,268	78,228
	West Ramp	1,176	141	1,317	81,269
	Upper West Ramp	843	101	944	58,256
Level Access Drives	Lower East Zone	429	51	480	29,646
	Lower West Zone	552	66	618	38,146
	Central Zone	241	29	270	16,655
	Upper West Zone	260	31	291	117,411
	Upper West Zone - Middle	168	20	188	11,610
	Upper West Zone – Left	50	6	56	3,325
Footwall Drives	Lower East Zone	2,762	331	3,093	190,871
	Lower West Zone	290	35	325	20,041
	Central Zone	3,299	396	3,695	227,981
	Upper West Zone	1,699	204	1,903	117,411
	Upper West Zone – Middle	348	42	390	24,049
Cross Cut Access	Lower East Zone	435	52	487	30,061
	Lower West Zone	307	37	344	21,216
	Central Zone	540	65	605	37,317
	Upper West Zone	400	48	448	27,642
	Upper West Zone – Middle	120	14	134	8,293
	Upper West Zone – Left	60	7	67	4,146
Vent Raise Access	Lower East Zone Ramp	125	15	140	8,638
	Central Ramp	170	20	190	11,748
	Upper West Ramp	492	59	551	34,000
	Footwall Drive	104	12	116	7,187
Host Rock Pass Access	Central Zone	304	36	340	21,008
	Total	**18,990**	**2,279**	**21,269**	**1,411,636**

Figure 16.5 Long Section View of Mine Layout (Looking Northwest)

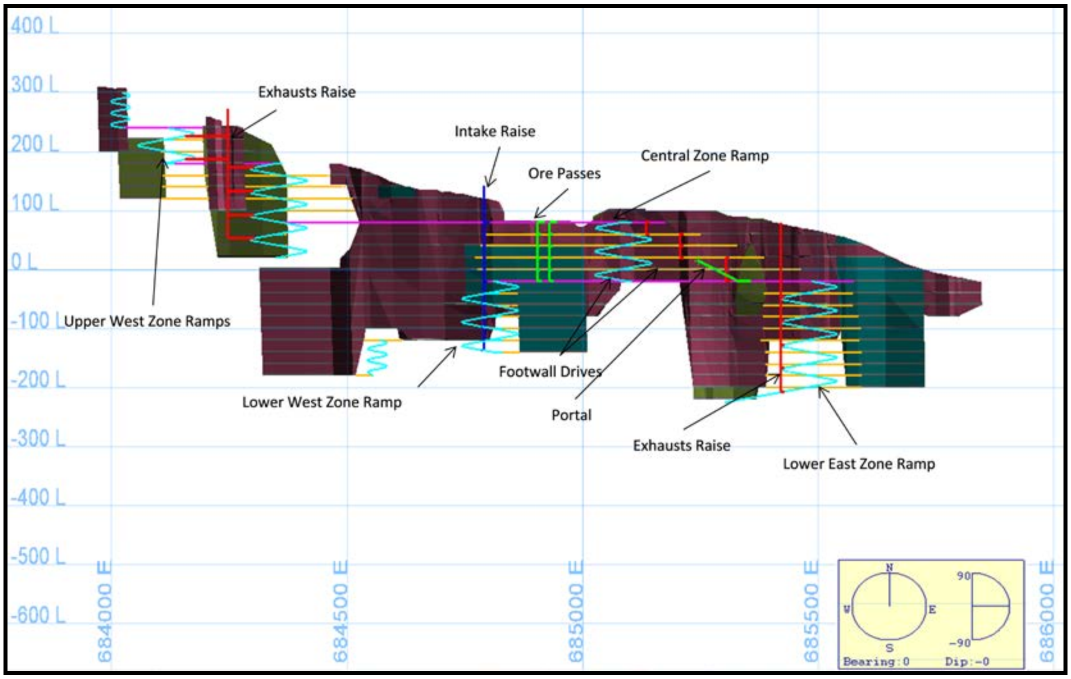


Figure 16.6 Plan View of Mine Layout

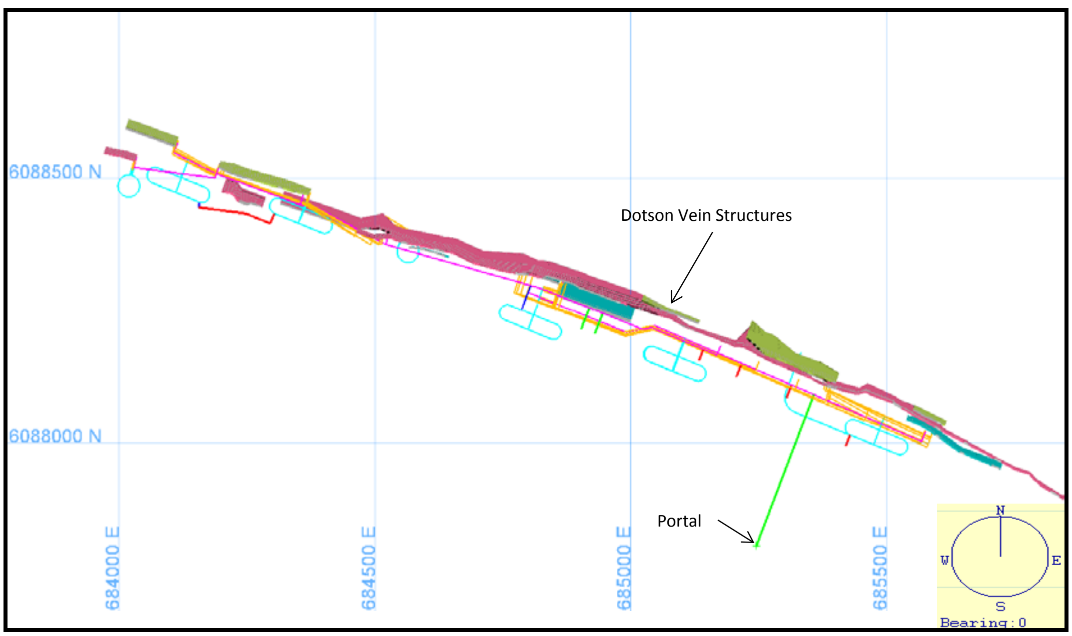
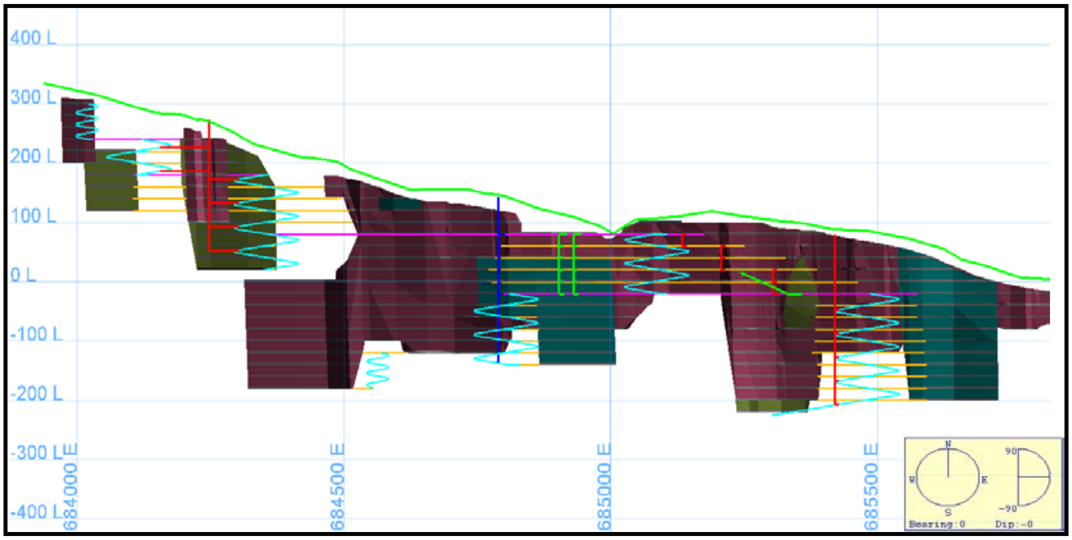


Figure 16.7 Long Section View of Mine Layout (Looking Northeast)



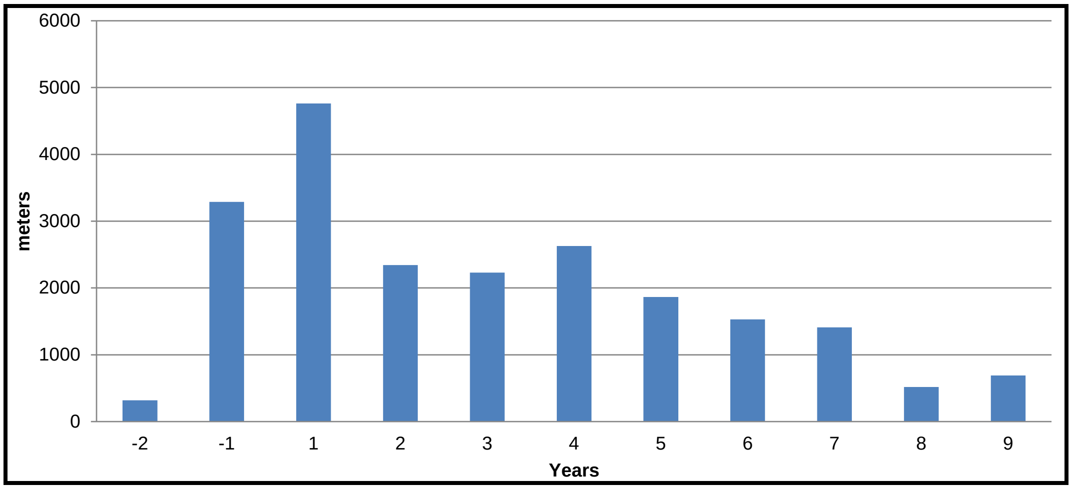
16.4 DEVELOPMENT SCHEDULE

The waste development schedule includes all lateral waste development broken down by category and crews. All scheduling is completed to an annual basis level of detail. There are 21,269 m of lateral development scheduled over the course of 11 years which includes 2 years of preproduction. The required development is high in the first couple of years in order to provide access to the required mining zones. The schedule is crew-based and structured as one main waste development crew supported by host rock development crews when required. As the mine ramps up in production, waste development is less critical, allowing for a reduced amount of development crews to the point where there is only one development crew required for both host rock and waste for the last few years of the schedule. Table 16.5 presents the development metres scheduled by crew on an annual basis. Figure 16.8 illustrates total development metres per year.

Table 16.5 Waste Development Schedule by Crew

Crew Unit	Main Development (m)	Secondary Development (m)	Total (m)	Year										
				-2	-1	1	2	3	4	5	6	7	8	9
				(m)										
CA	1,613	61	1,674	319	1,674	-	-	-	-	-	-	-	-	-
CB	1,333	282	1,615	-	1,615	-	-	-	-	-	-	-	-	-
CA	2,022	351	2,373	-	-	2,373	-	-	-	-	-	-	-	-
OD_1	1,322	1,068	2,390	-	-	2,390	-	-	-	-	-	-	-	-
OD_1	1,559	337	1,896	-	-	-	1,896	-	-	-	-	-	-	-
OD_1	1,257	282	1,539	-	-	-	-	1,539	-	-	-	-	-	-
OD_3	300	147	447	-	-	-	447	-	-	-	-	-	-	-
OD_3	652	39	691	-	-	-	-	691	-	-	-	-	-	-
OD_1	1,758	364	2,122	-	-	-	-	-	2,122	-	-	-	-	-
OD_3	-	506	506	-	-	-	-	-	506	-	-	-	-	-
OD_1	1,646	218	1,864	-	-	-	-	-	-	1,864	-	-	-	-
OD_2	1,280	252	1,532	-	-	-	-	-	-	-	1,532	-	-	-
OD_2	1,184	227	1,411	-	-	-	-	-	-	-	-	1,411	-	-
OD_2	243	276	519	-	-	-	-	-	-	-	-	-	519	-
OD_2	184	506	690	-	-	-	-	-	-	-	-	-		690
Total	**16,353**	**4,916**	**21,269**	**319**	**3,289**	**4,763**	**2,343**	**2,230**	**2,628**	**1,864**	**1,532**	**1,411**	**519**	**690**

Figure 16.8 Annual Development Schedule



16.5 MINE INFRASTRUCTURE

The conceptual design for the Bokan Mountain mine is straightforward and does not require a large amount of mine infrastructure. The following list shows items included in the mine infrastructure design at a conceptual level:

- host rock flow system
- ventilation
- mine services.

Further work is required to better identify opportunities for improvements or efficiencies for the next level of study.

16.5.1 HOST ROCK FLOW SYSTEM

Production host rock will be loaded at the extraction level in the stopes using 5.6 m^3 LHDs. The majority of this loading will be performed remotely. The LHDs will transport the host rock to a muck bay or load the host rock directly into an underground haul truck. All host rock and waste will be hauled from underground to a specified location on surface.

There will be two host rock passes included in the system located in the western part of the Central Zone. These host rock passes will be utilized for a majority of the host rock mined from the Central Zone and from the Upper West Zone. Each host rock pass will be equipped with a grizzly on top and reinforced open drawpoint at the bottom. When the host rock passes are in use, an LHD will be required to load the haul trucks from the bottom of the host rock pass.

16.5.2 VENTILATION

Ventilation is provided for the underground operation by a system of ventilation fans, ducting, drifts, and raises. The first step in the ventilation design was to establish the required quantity of fresh air. Using the standard convention of 0.059 m^3/s (125 cfm) per operating horsepower, Stantec calculated required ventilation from a list of all of the underground fleet during peak production, the rated horsepower, and the anticipated utilization. The calculation also includes an allocation of 0.07 m^3/s (150 cfm) per person. The results determined that, at peak operation, the Project requires approximately 232 m^3/s (491,500 cfm) of fresh air. Table 16.6 summarizes the required ventilation calculation.

Table 16.6 Maximum Ventilation Requirements Summary

Main Equipment	Quantity (Max. No. of Equipment in Peak)	Operating Factor (%)	Unit Horsepower (hp/unit)	Unit Ventilation (cms/unit)	Total Ventilation (cms)
LHD, 5.6 m^3, Diesel	5	85	250	15	63
Haulage Truck, 32.6 t, Diesel	4	85	400	24	80
Drill Jumbo, 2-Boom, Electro/ Hydraulic Drills	3	15	78	5	2
Rock Bolter, Single Boom, Electro/Hydraulic Drill with Screen Handler	2	20	173	10	4
Drill Jumbo, Double Boom, Electro/Hydraulic Drill (Rock Bolting)	1	15	173	10	2
Production Drill - Electro/ Hydraulic	2	15	100	6	2
Explosives Truck/Jumbo, Diesel, ANFO	1	15	147	9	1
Scissor Lift Truck, Diesel	3	20	147	9	5
Underground Personnel Carrier – 8 People	7	40	128	8	21
Boom Truck, Diesel	1	40	147	9	3
Cassette Carrier	2	40	149	9	7
Telehandler, Diesel, Underground	1	75	141	8	6
Underground Road Grader	1	75	158	9	7
Total	35	-	-	-	-
Fixed Allowances	-	-	-	-	-
Personnel Underground	80	100	0	0.07	6
Subtotal Mobile Equipment and Fixed Allowances	-	-	-	-	209
Miscellaneous Allowance	10%	-	-	-	21
Total Ventilation Requirement 230					
Ventilation requirements assessed per brake hp at = 0.1 m^3/s					
Daily Production Rate 1,500 t/d $(m^3/s)/t/d$ = 0.15					

Stantec included the following assumptions in an effort to design the mine ventilation system while maintaining controllable air velocities.

- main access/drift maximum ventilation velocity = 7.1 m/s (1,400 ft/min)
- main intake and exhaust raise velocity = 20.32 m/s (4,000 ft/min)

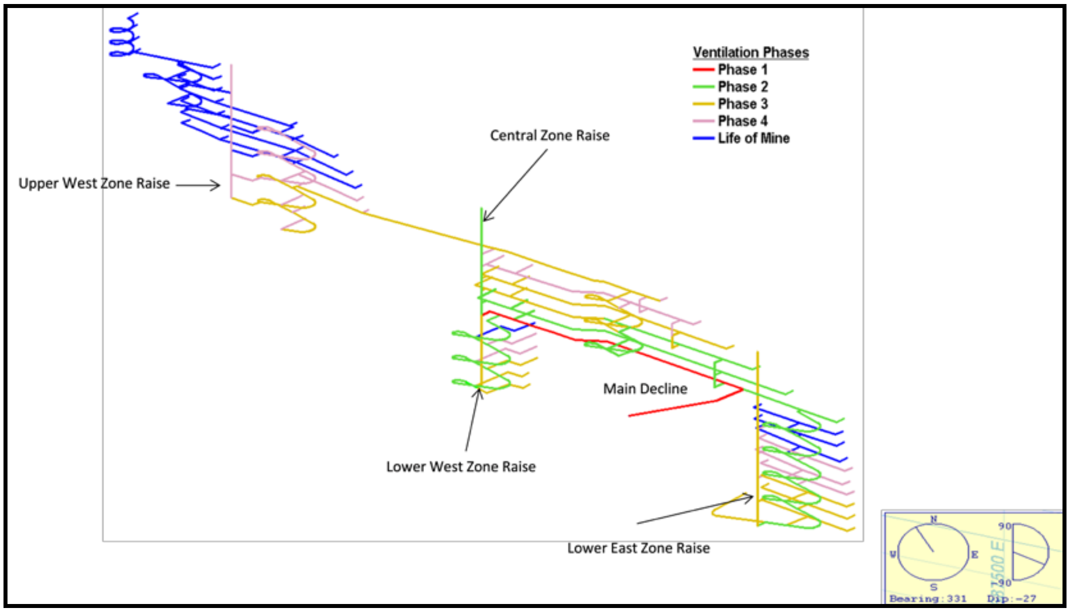
The resulting breakdown of main airways to support ventilation is shown in Table 16.7.

Table 16.7 Main Airways

Description	Size	Design Capacity (cms)
Intake		
Main Access Decline	4.5 m wide by 4.5 m high	1,346
Central Zone Raise	3.0 m diameter	1,600
	Total Intake	**2,946**
Exhaust		
Lower East Zone Raise	3.0 m diameter	1,600
Upper West Zone Raise	3.0 m diameter	1,600
	Total Exhaust	**3,200**

Stantec then divided the ventilation plan into separate phases. Each phase is defined by the distinct operating requirements and equipment arrangement of the ventilation system. Figure 16.9 illustrates the different ventilation phases over the LOM. During the first phase, the access decline will be driven and a drift will be brought over to the base of the Central Zone raise. This arrangement will use a series of ventilation fans with flexible bag ducting to bring fresh air to the work area.

Figure 16.9 LOM Ventilation Plan (Looking Northwest)



The completion of the Central Zone is the first step of the second phase. The raise is used for exhaust, and will have a vane axial exhaust fan installed at the collar. Fresh air will be drawn down the decline, circulated through the work area ramps, and over to the exhaust raise, establishing a primary ventilation loop. From this loop, air will be brought into the production work areas using vane axial fans and bag ducting. Included in this phase is development down to the base of the lower east raise and initiation of the lower west drop raises.

The third phase includes development of two intake and one exhaust raises. The exhaust fan will be relocated from the Central Zone raise over to the lower east raise, with fresh air coming in through both the decline and the Central Zone raise. The additional airflow from this arrangement will now be able to support production in both the Central and Lower East Zones. Included in this phase is development over to the Upper West Zone and the base of the Upper West Zone raise.

The fourth and final phase begins with the completion of the upper west vent raise. This raise will be used as a second exhaust with a vane axial exhaust fan installed at the collar. The final arrangement will include two intakes (i.e. the central raise and the decline), and two exhaust raises (i.e. upper west and lower east), to carry 491,500 cfm to support total ventilation requirement at maximum production levels.

An important aspect of the mine ventilation plan is the requirement for mine air heating, particularly at the decline portal. Ensuring ventilation air is above freezing is critical to prevent freezing of the installed utility lines, and to maintain desired production levels in the work areas. Heating will be provided by package type indirect heaters, installed at the portal and lower east raise collar. When the air temperature approaches freezing, the heaters will come online at low speed, drawing diesel fuel from the onboard tank. As the air temperature continues to drop, the units will switch to high speed, generating the additional heat required.

The ventilation distribution patterns will be adjusted throughout production. As areas are mined out, the air flow rates and routing can be modified by bulkheading and adjusting the ventilation fans.

16.6 MINE SERVICES

Stantec made the following assumptions related to underground mine services supporting a 540,000 t/a operation. These mine services are not engineered or designed; the assumptions described here represent infrastructure specifications developed for other properties with similar levels of production and similar mining methods. The costs associated are included in the preproduction capital and/or sustaining capital costs.

Compressed Air Plant

Ucore requested no compressed air plant to be included in the scope for the underground mine design. Stantec assumed the required compressed air will be available when required.

Main Pumps and Sumps

An allocation is provided to include excavation and construction of main pumps and sumps. Very little hydrological information is available at this time. Additional work and hydrological data is necessary to identify the passive resistive inflows present to reasonably identify pumping requirements. This study assumes passive resistive inflows of 31 L/s (500 g/m) and estimates service water requirements at 12 L/s (200 g/m).

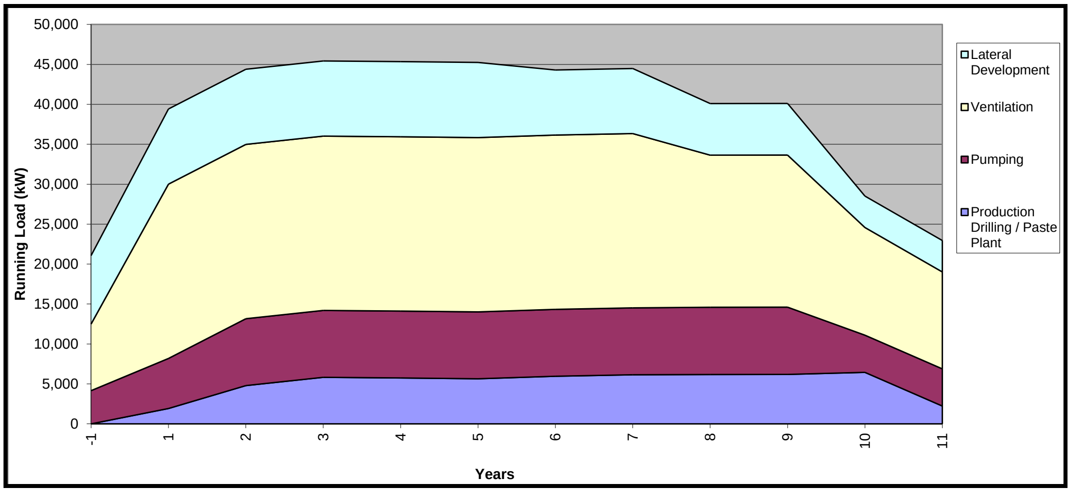
Electrical Distribution

Main power is delivered underground at 13.8 kV via the main access decline from power generators located on surface. Allowances for electrical equipment and distribution are included in the capital costs. Total operating power underground is estimated with peak operating loads of 3 MW and diversified operating loads of 2 MW. Table 16.8 and Figure 16.10 provide the estimated peak operating loads and diversified operating loads by year.

Table 16.8 Peak and Diversified Operating Loads by Year

Summary	Year											
	-1	1	2	3	4	5	6	7	8	9	10	11
Diversified Running (kW)	878	1,643	1,350	1,893	1,390	1,885	1,846	1,854	1,671	1,671	1,188	956
Peak (kW)	1,369	2,695	2,870	2,940	2,940	2,940	2,835	2,835	2,615	2,615	1,805	1,679
Daily (kWH)	21,067	39,421	44,396	45,432	45,351	45,249	44, 3-tr.2	44,488	40,101	40,114	28,517	22,953

Figure 16.10 Peak and Diversified Operating Loads by Year



Communications

A leaky feeder system will be installed throughout the underground; a main control and dispatch will be located on surface.

Paste Plant

The paste backfill plant will be located on surface. Paste backfill will be delivered underground through a network of dedicated steel lined boreholes to each mining level.

The plant has been sized to support the production rate. No engineering or design work was performed on the paste backfill plant. Capital and operating costs, obtained from other projects of similar size, are included in the cost estimates. Additional engineering and design work is required at the next level of study.

16.7 MINE EQUIPMENT

The list of equipment permanent that supports the mine plan is divided into mobile equipment and fixed equipment. Mobile equipment is essentially defined as rubber-tired equipment; and fixed equipment is defined as permanently located equipment. Table 16.9 and Table 16.10 summarizes mobile and fixed equipment requirements, respectively, for the Project.

Table 16.9 Mobile Equipment Requirements

Name/Model	Type	Total Quantity	Initial Quantity
Atlas Copco ST-1030	LHD, 5.6 m^3, Diesel	5	4
Production Loader - Tele Remote Equipment	Production Loader - Tele Remote Equipment	2	1
Atlas Copco MT 436B	Haulage Truck, 32.6 Tonne, Diesel	4	2
Atlas Copco Boomer 282 - Jumbo	Drill Jumbo, 2-Boom, Electro/Hydraulic Drills	3	3
Atlas Copco Boltec MC - Rockbolter - 1 Boom	Rock Bolter, Single Boom, Electro/Hydraulic Drill with Screen Handler	2	2
Atlas Copco Boltec MC - Rockbolter - 2 Boom	Drill Jumbo, Double Boom, Electro/Hydraulic Drill (Rock Bolting)	1	1
Schwing WP1250X	Concrete Pump - Diesel - Trailer Mount	2	2
Shotcrete Technologies	Shotcrete Machine	1	1
Atlas Copco Simba 1254 - Ring Drill	Production Drill - Electro/Hydraulic	2	1
MacLean AC-3 ANFO Charger	Explosives Truck/Jumbo, Diesel, ANFO	1	1
Maclean Mine-Mate SL-3	Scissor Lift Truck, Diesel	3	3
Toyota PC	Underground Personnel Carrier - 8 Person	7	5
MacLean Mine-Mate BC-3	Boom Truck, Diesel	1	1
MacLean CS-3	Cassette Carrier	2	2
MacLean CS-3	Fuel/Lube Cassette	1	1
MacLean CS-3	Flatbed Cassette	1	1
TL1255	Telehandler, Diesel, Underground	1	1
Cat 12M	Underground Road Grader	1	1
Lincoln Air Vantage 500	Diesel Welder - Trailer	1	1
Portable Concrete Mixer - 1/2 yd	Portable Concrete Mixer - 1/2 yd	1	1
	Total Quantity	**42**	**35**

Table 16.10 Fixed Equipment Requirements

Area	Item	Quantity
Surface Facilities	Service Water Pump	1
	Service Water Tank - 10,000 L/2,642 gal	1
Ventilation	Main Fans - Surface - Axial - 78"	2
	Package Mine Air Heater	2
	Drift Fans - 125 HP - 60"	3
	Drift Fans - 40 HP - 36”	8
	Air Door - Set of 2 - Pneumatic with Man Door	-
Water Management	Drift Pumps - Electric - 58 HP	2
	Drift Pumps - Electric - 24 HP	3
	Drift Pumps - Pneumatic Wilden T1510	3
	Tsurumi LH322W 30 HP Submersible - Contact Water	-
	500 gal Tank on Skid	6
	Miller Contact Water Pump Skid	6

table continues…

Area	Item	Quantity
Material Handling	Grizzly	2
	Grizzly Monorail Crane - 5 Ton - 20 ft Lift	2
Safety and Miscellaneous	Portable Refuge Chamber - 20 Man	5
	Stench System	1
	Self-contained Self Rescuers - Initial Purchase	220
	Mine Rescue Gear-1 Man	10
	Mine Rescue Gear Tester	1
	Miscellaneous Mine Rescue Supplies	-
	Mine Lamp	165
	Mine Lamp Charger - 30 Position	6
	Miscellaneous First Aid Supplies - Underground Storage	3
	Emergency Escape Hoist + Boom + 600M Rope + Cage	1
	Sanitary Facilities	5
	Sanitary Facilities Service Unit for Flat Bed Truck	1
	Hilti Drill	3
	Jack Leg Drill	3
	Total Station Survey Equipment	1
Electrical	Main Substation	1
	Underground Substation	1
	Mine Load Centres	6
	Main Vent Fan Substation	2
	Development Mine Load Centre	6
	Leaky Feeder Radio	1

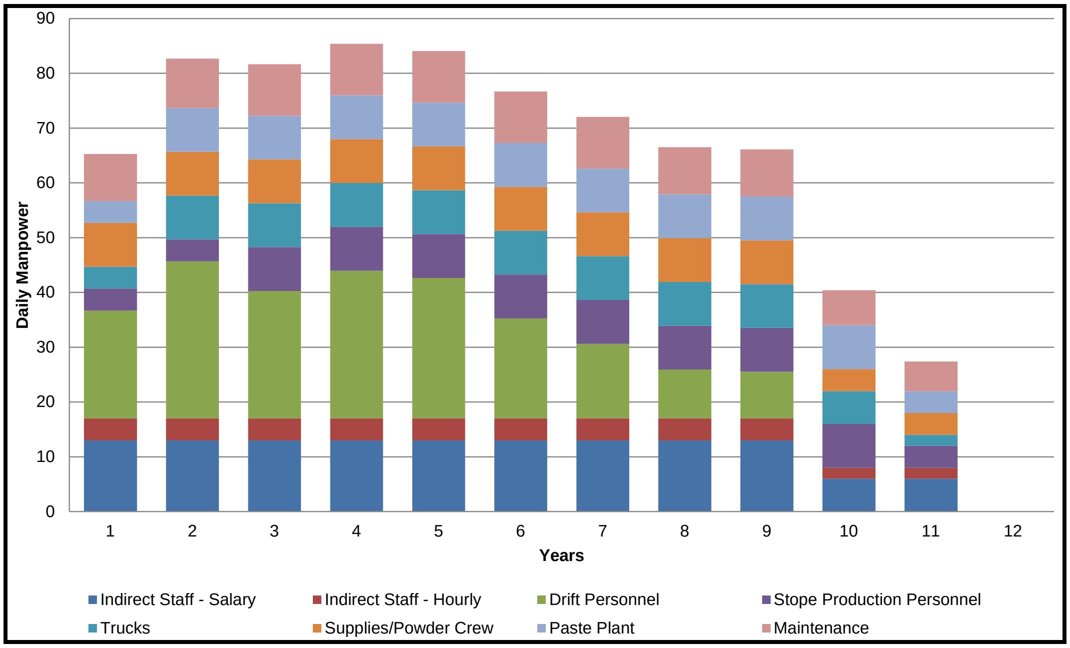
16.8 MANPOWER

Stantec estimated manpower requirements for the underground mining and support activities associated with the Project. The manpower requirements are representative numbers only, based on project experience using similar mining methods and similar production rates. These manpower requirements are used in the calculation of direct manpower requirements based on the production schedule. The assumptions used for the manpower calculations are:

- 360 working days per year
- crew rotation is two weeks on two weeks off
- two 11-hour shifts per day.

Figure 16.11 presents the total daily manpower requirements for the LOM.

Figure 16.11 Total Daily Manpower Requirements



The total manpower requirements on- and off-site are representative numbers only based on project experience using similar mining methods and similar production rates. Table 16.11 presents a required manpower breakdown by position.

Table 16.11 Total Required Manpower

Position	Total Quantity
Salaried	
Mine Manager	1
Mine Superintendent	1
Mine Foreman	4
Maintenance General Foreman	2
Maintenance Foreman - Mechanical	2
Chief Mining Engineer	1
Senior Mining Engineer	3
Senior Surveyor	1
Chief Geologist	1
Senior Geologist	1
Safety Inspector/Trainer	2
Total Salaried Personnel	19
Hourly	
Secretary (Indirect)	1
Surveyors (Indirect)	2

table continues…

Position	Total Quantity
Samplers (Indirect)	2
Paste Plant Operators	12
Explosives Crew	6
Supply Crew	6
Mechanics/Electricians – Mobile Equipment	9
Mechanics/Electricians – Fixed Equipment	3
Grader Operator	2
Mine Services and Construction	9
Direct Mining Personnel	47
Total Hourly Personnel	**99**
Total Personnel Required	**118**

17.0 RECOVERY METHODS

17.1 PROCESS PLANT OVERVIEW

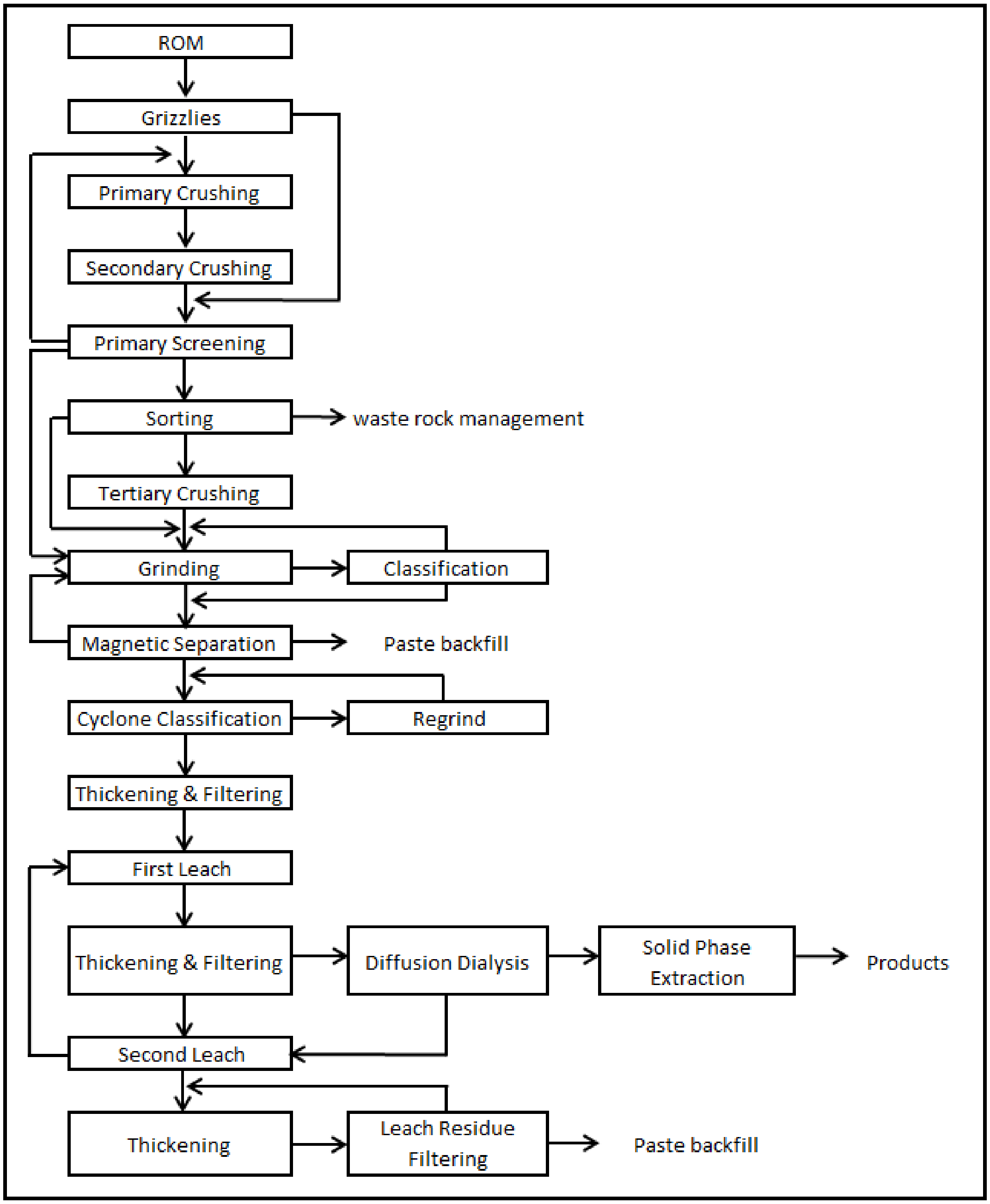
The process plant comprises the following major circuits and their nominal process rates:

- primary crushing (1,500 t/d)
- secondary crushing (1,500 t/d)
- screening to remove the minus 1/4" (6.3 mm) fraction
- x-ray mineralized material sorting (1,125 t/d)
- tertiary crushing (750 t/d)
- rod mill grinding (750 t/d)
- magnetic separation (750 t/d)
- tower mill re-grinding (375 t/d)
- leaching (375 t/d)
- SPE (9.2 t/d).

The primary crushing facility will be located near the underground portal to minimize the haulage cycle time of the underground trucks. The majority of process equipment items will be located inside the process facility.

A simplified process flowsheet is shown as Figure 17.1. Detailed process flowsheets are provided in Appendix F.

Figure 17.1 Simplified Process Flowsheet



17.2 CRUSHING AND CONVEYING

17.2.1 PRIMARY CRUSHING FACILITY

The primary crushing facility will be located near the portal entrance, to reduce the cycle times for the mine trucks. The primary crushing circuit will consist of typical crushing equipment, including a jaw crusher, grizzlies, and conveyor belt. This facility will crush mine rock to minus 6”.

17.2.2 SECONDARY CRUSHING AND SORTING FACILITY

The secondary crushing and sorting facility will be located in the mill building. The secondary crushing circuit will consist of typical secondary crushing equipment, including a cone crusher and screens. The material will be crushed to minus 2-1/2” and screened into four size ranges to feed the sorters. The minus 1/4” material will bypass the sorters and go directly to grinding. Each of the other size fractions will be fed to individual sorters.

17.3 SORTING

The sorting will be carried out by dual energy x-ray sorters. Three size ranges will be fed to individual sorters:

- 1/4 to 5/8”
- 5/8 to 1-1/2”
- 1-1/2 to 2-1/2”.

The minus 1/4" material will bypass the sorters and be conveyed to the tertiary crushing and grinding circuit. Overall the screening and sorting will reject approximately 50% of the crushed material, as waste. The waste material will be hauled to the waste rock management area, or trucked underground and used as backfill. The concentrated material, along with the minus 1/4" material, will be conveyed to the fine crushing and grinding area.

17.4 TERTIARY CRUSHING AND GRINDING

After separation of the barren rock from the mineralized material in the sorting process, the concentrate will be stage-crushed to pass 1/4" and placed into a fine material storage bin. The crushed material will be withdrawn from the bin by feeders at a controlled rate and fed to a dry grinding rod mill with air classification to return material greater than the plus 30-mesh (600 µm) back to the mill. The air-classified minus 30-mesh material will be fed to a series of dry screens to make separations at 50-mesh (300 µm), 100-mesh (150 µm) and 200-mesh (75 µm). The minus 200-

mesh fine fraction will be sent to the final fine grinding circuit for reduction to minus 40 µm prior to leaching.

The three screen fractions between -30+50, -50+100 and -100+200 mesh will be treated on dry belted-roll magnetic separators equipped with permanent magnets. The magnetic separators will produce three products: a magnetic concentrate, a middling and a non-magnetic tailing. The middling fraction will return to the rod mill feed for further size reduction. The non-magnetic tailing will be combined with the leach circuit tailings for return to the underground operations for back fill, or to the temporary tailings management area. The magnetic concentrates will be combined, slurried and sent to the fine grinding circuit, along with the minus 200-mesh from the rod mill grinding circuit, for reduction to minus 40 µm for leaching.

The final fine grinding circuit will be comprised of a tower mill, or ball mill, operating in closed circuit with a classifying cyclone. The cyclone overflow of minus 40 µm will report to a thickener to increase the slurry density to at least 40% solid. The thickened slurry will then proceed to the acid leaching process.

17.5 ACID LEACHING

The material will be leached in two stages with nitric acid at 90°C with an eight-hour retention. The slurry will then be filtered and washed and the solids sent to the paste backfill plant to be used as cemented backfill for filling mined out areas underground. The pregnant solution will be treated with diffusion dialysis to reclaim the unconsumed nitric acid to the leach circuit. The pregnant solution will then be treated by SPE, which is a process developed by IntelliMet in conjunction with Ucore.

17.6 SOLID PHASE EXTRACTION

The element separation process involves four progressive steps:

1. removal of nuisance elements
2. segregation of elements first into impure element subclasses
3. separation into purified element subclasses
4. separation into individual elements.

REMOVAL OF NUISANCE ELEMENTS

The rare earth loaded nitric acid solution obtained from the leach process is injected into a two column process to remove nuisance elements in the leach solution, such as iron, uranium, thorium, and zirconium. The rare earths are recovered as a mixed rare earth stream in acetic acid. The first column in the sequence binds nuisance elements from the acidic rare earth bearing solution, depleting nuisance elements from the rare earth stream. The iron, uranium, thorium, and zirconium and other nuisance elements are then eluted from the first column with ammonium citrate which produces a waste stream. The metals are removed from the ammonium citrate waste stream via ammonium carbonate precipitation, producing a metal carbonate waste product, and the ammonium citrate can be reused as an eluant. The waste is added to the mill tailings to be placed underground as cemented paste backfill. The flowthrough from the first column is injected into the second column, which binds the rare earths and other metals in solution. The rare earths are then eluted from the column using a weak acid eluant (acetic acid), providing a stream of rare earths and calcium, and nearly free of other contaminants. The residual metals bound to the column are then eluted with strong acid, provided by fresh leach solution, combining the bound metal with metal in the leach solution while not incurring dilution, and the eluate is injected into the first column to restart the cycle.

Subclass Separation

The acetic acid stream is run through four progressive columns, which deplete the rare earths by subclass, and produces five outputs:

- The first output contains the heavy rare earths terbium, dysprosium and heavier.
- The second output contains samarium, europium and gadolinium (SEG) and yttrium.
- The third output contains neodymium and praseodymium (didymium).
- The final columns produce cerium and lanthanum.
- This leaves a stream containing calcium with nearly all of the rare earths removed.

Calcium is removed from solution by sulphuric acid gypsum precipitation to regenerate the acetic acid which is recycled. The precipitate is added to the underground backfill. The segregation between the four columns is not definitive. The effect of this subclass separation step is to reduce the number of steps required to split given crude mixed oxide product to elements, since there is already an enhanced ratio towards one portion of the elements. After loading the column, the columns are eluted with dilute acid through a secondary (amplifier) column, which splits the elements into further fractions. These fractions from the four columns are combined and precipitated with calcium oxide to produce crude mixed metal oxide products (heavy, SEG+yttrium, didymium, and cerium+lanthanum).

SEPARATION INTO PURIFIED ELEMENT SUBCLASSES

Crude Subclass Splitting Process

Each of the four crude oxide products are dissolved in nitric acid to form metal nitrate salts, and are inserted into a series of individual column processes to refine the product. The goal of these steps is to segregate the mixed oxide products into fractions of appropriate content to enter the “element separation hubs.”

Dysprosium/Terbium Mix Production Hub

Solutions containing predominantly dysprosium and terbium are inserted into this column process. During loading, the early flowthrough is enriched in ultraheavy elements holmium, erbium, thulium, ytterbium, lutetium, and SEG elements. It is depleted in dysprosium/terbium. This flowthrough is passed into the next column in the sequence until nearly all dysprosium/terbium is removed. The late flowthrough of each column is richer in dysprosium/terbium and is recycled. The column is then eluted with nitric acid to provide a mixed dysprosium/terbium nitrate stream. After a base regeneration of the column with ammonia, the cycle is repeated. The final column in the sequence produces an early flowthrough that predominantly contains the ultraheavy REEs.

SEG Elimination Hub

Inputs containing SEG elements, heavy elements, and yttrium, but free of neodymium and lighter, are injected into a series of columns. The early flowthrough is enriched in SEG elements and yttrium, and is passed onto the next column in the stream. The late flowthrough is recycled as raffinate. The column is eluted after each cycle with nitric acid to generate a heavy element (terbium and heavier) fraction. The eluted fraction from the first column in the series is directed to the dysprosium/terbium mix production hub, and the early flowthrough from the last column in the series is directed to the SEG purification hub.

Ultraheavy Production Hub

Three columns split a mixture containing predominantly ultraheavy elements (holmium, erbium, thulium, ytterbium, lutetium) from dysprosium/terbium and SEG impurities. The feed is loaded into a column with a high affinity for ultraheavy elements. The flowthrough is depleted in ultraheavy elements and contains dysprosium and lighter. After rinsing, the column is eluted with nitric acid to provide a concentrate containing the ultraheavy elements as mixed metal nitrate solution (holmium, erbium, thulium, ytterbium, and lutetium). This solution is then precipitated with ammonia to provide a mixed ultraheavy metal oxide concentrate, and the residual ammonium nitrate supernatent goes to recycling/acid regeneration.

Yttrium Purification Hub

Solutions containing yttrium with neodymium and praseodymium as the predominant impurity are passed through a series of columns where the yttrium is bound and neodymium/praseodymium are displaced. Yttrium is progressively removed from the neodymium/praseodymium through a series of columns to provide a purified mixed didymium concentrate. Each column is eluted with nitric acid, and the eluate is precipitated with ammonia to provide a purified yttrium oxide product.

SEG Purification Hub

The mixed metal solution containing predominantly SEG elements (samarium, europium and gadolinium), but with a minority of lighter and heavier elements, is put into a ladder sequence of columns. Fresh solution is loaded into a column in the middle of the sequence, and the early flowthrough is enriched in SEG and lighter elements, and is passed to a column lower on the sequence. The late flowthrough is recycled as raffinate. The column is then rinsed, and eluted with nitric acid to provide a mixed metal nitrate solution, that is neutralized with ammonia and provides feed for the next higher column on the ladder. The eluate at the top column in the sequence contains a solution of terbium, dysprosium, holmium, erbium, thulium, ytterbium, and lutetium, which is fed into the dysprosium/terbium purification hub. The flowthrough from the lowest column in the sequence contains SEG elements and yttrium and lighter elements. This is then passed through a series of columns which bind SEG elements in preference to lighter elements. The nitric acid eluate from the first column in this sequence contains purified SEG elements, which then proceeds to europium recovery. The flowthrough from the last column in the sequence contains an yttrium and lighter mix which proceeds to the yttrium purification hub. Acid eluates are neutralized with ammonia and added to feed into a higher column in the sequence.

Europium Extraction Hub

The mixture containing SEG with a small impurity of heavier elements is fed into a ladder of columns. The columns bind europium in preference to samarium and gadolinium. The flowthrough is enriched in samarium and gadolinium, and is passed as feed to a lower column in the sequence. The bound fraction is enriched in europium and heavy elements, and is eluted with nitric acid, neutralized with ammonia, and fed into the next higher column in the sequence. After ammonia neutralization, the column begins its next cycle of operation. The flowthrough from the lowest column in the sequence produces a purified mixture of samarium and gadolinium. The highest column in the sequence produces a europium nitrate product with a heavy element impurity (terbium+).

SEPARATION INTO INDIVIDUAL ELEMENTS

Dysprosium/Terbium Splitting Hub

The mixed metal solution containing predominantly dysprosium and terbium is fed into a ladder of columns. The columns are loaded and dysprosium displaces the terbium. The flowthrough solution is enriched in terbium and is fed to a column lower on the sequence. After rinse, the column is eluted, and the dysprosium nitrate solution is fed into a column higher on the sequence. At either end of the sequence, the purified dysprosium nitrate and terbium nitrate products are purified of ultraheavy element impurities (holmium, erbium, thulium, ytterbium, and lutetium), to purified solutions which are precipitated with ammonia to provide dysprosium oxide and terbium oxide. The mixed ultraheavy metal oxide products are returned to the ultraheavy production hub.

Europium Purification Hub

The mixture containing europium with a heavy element impurity (terbium, dysprosium, holmium, erbium, thulium, ytterbium, and lutetium) is injected into a ladder of columns which bind the heavy elements in preference to europium. Europium flowthroughs advance to a lower column in the sequence, while nitric acid eluates of the heavy elements are neutralized and fed to a higher column in the sequence. The acid eluate from the highest column is a mixed heavy metal nitrate solution (terbium+) which is fed into the dysprosium/terbium mix production hub, and the flowthrough from the lowest column in the sequence produces a purified europium solution, which is precipitated with ammonia to provide a europium oxide product.

Gadolinium/Samarium Splitting Hub

The mixed gadolinium/samarium nitrate feed is passed into a column ladder in which columns bind samarium in preference to gadolinium. The gadolinium enriched flowthrough is passed to column lower on the sequence, while the samarium nitrate containing nitric acid eluant is passed to a column higher on the sequence. The flowthrough from the lowest column contains a purified gadolinium solution which is precipitated with ammonia to produce a gadolinium oxide product, and the acid eluant from the highest column on the sequence contains a purified samarium solution, which is precipitated with ammonia to produce a purified samarium oxide product.

Neodymium/Praseodymium Splitting Hub

The mixed neodymium/praseodymium solution is fed into a ladder of columns which bind neodymium in preference to praseodymium. The column is loaded, and the flowthrough contains a solution enriched in praseodymium over neodymium, which is passed as feed to a lower column in the sequence. The eluate contains an enriched neodymium solution, which is neutralized and fed to the next higher column on the sequence. The eluate from the highest column in the sequence contains a purified neodymium solution, which is precipitated with ammonia to produce a purified neodymium oxide product. The flowthrough from the lowest column in the sequence contains praseodymium with a cerium impurity, which is passed through a sequence of columns which bind praseodymium in preference to cerium. The eluted fraction from these columns contains a purified praseodymium solution, which is precipitated with ammonia to produce a purified praseodymium oxide product. The final flowthrough from this sequence contains a purified cerium solution, which is precipitated with ammonia to produce a cerium oxide product.

Cerium/Lanthanum Splitting Hub

The mixed cerium/lanthanum containing solution is passed into a series of ladder of columns which bind cerium in preference to lanthanum. The flowthrough is enriched in lanthanum over cerium and forms the feed for the next lower column in the sequence. The acid eluate contains cerium enriched over lanthanum and after neutralization forms the feed for the next higher column in the sequence. The eluate from the highest column in the sequence provides a purified cerium solution, which is precipitated with ammonia to provide a purified cerium oxide product. The flowthrough from the lowest column the sequence provides a purified lanthanum solution, which can be precipitated with ammonia to provide a purified lanthanum oxide product.

17.7 PASTE BACKFILL

The backfill assumptions in this report are based on experiences in other projects under similar conditions.

Leached residue will be filtered to remove and recover leach liquor and then slurried with mine water and treated in a deep-cone-paste thickener to increase the pulp density to a consistency suitable for backfill. Cement will then be added, and the tailings will be pumped underground to fill mined out areas. When the mine cannot accept backfill it will be directed to the lined TMWMF. The tailings will be held in the tailings management area until the mine requires backfill, after which it will be sent to the backfill plant. This is necessary as the mill will produce less tailing material than the mine requires for fill. The shortfall will be made up with mine waste rock.

18.0 PROJECT INFRASTRUCTURE

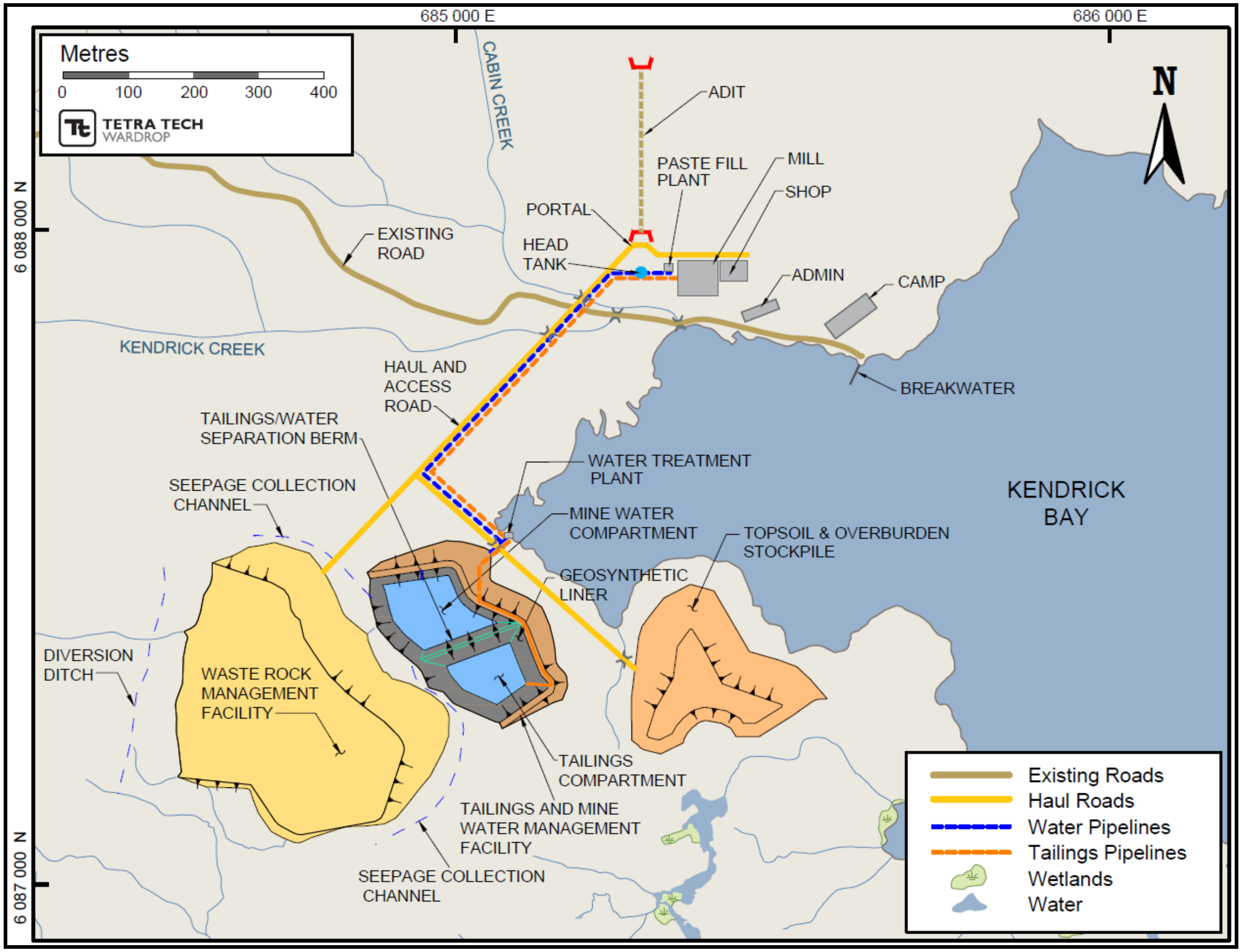
18.1 SITE LAYOUT

The proposed on-site and off-site infrastructure for the Project will include:

- a process plant
- a permanent camp with administration offices
- an emergency vehicle building with vehicle maintenance shop
- a storage laydown area
- power generation units (through LNG)
- a main substation and power distribution
- a potable and fire water storage and distribution
- plant and camp sewage treatment facilities
- a laydown and container storage yard
- a paste backfill plant
- a TMWMF
- a WRMF
- a water treatment facility
- access and site roads.

The general site layout of the Project is provided in Figure 18.1.

Figure 18.1 General Site Layout



18.1.1 PROCESS PLANT

The process plant will comprise the following areas:

- crushing and grinding
- DEXRT sorting
- magnetic separation
- leaching
- REE solid phase extraction
- reagents handling
- assay and metallurgical laboratory.

18.1.2 SITE ACCESS AND SITE ROADS

The Property has immediate deep water access to the Pacific Ocean. There is currently an exploration barge camp in operation with an on-site beachhead landing for barge landing, right by the existing gravel road. The beach head is located approximately 150 m south of the proposed portal. During the construction and operating stage, supplies will be received via transport barge and moved to the plant site for storage. Concentrate will be shipped out by the returning barge in a similar fashion. The Property is approximately 140 km (87.5 miles) from Prince Rupert, BC, a major sea and land transportation hub.

There is currently unscheduled float plane service between the Property and Ketchikan, Alaska, which takes about 20 minutes each way. The Ketchikan International Airport provides daily commercial flights to and from Seattle, WA and other cities in Alaska. During the construction and operation phase, the float plane or a boat service service will be expanded to provide regular, scheduled service.

As shown in Figure 18.1, there is an existing gravel road within the Property which is currently maintained and utilized for exploration activities. A new 1.2 km (0.75 miles) access road with crossings over Kendrick Creek will be constructed to allow vehicle access to the TMWMF and WRMF, as well as the water treatment facility.

18.2 POWER

The average electrical demand is estimated to be 4.0 MW for this mine, based on the current equipment list.

The power will be supplied by three 2 MW LNG generators. Two of the three units are expected to operate full-time; the third generator will be available as a stand-by unit.

A step-up transformer will increase the system voltage to 13.8 kV to facilitate more efficient power transmission to the other site facilities. (The transmission poles will carry both power and communication lines.) Power for the administrative offices and mine camp will be provided from this overhead 13.8 kV supply. Transformers at each location will reduce the voltage levels to utilization levels. Motor control centres (MCCs) and power distribution centres at each facility will manage and control power requirements.

A skid-mounted electrical house with a transformer will be installed at the TMWMF. The electrical house will be self-contained with switchgear, MCC, lighting, heating, communications, etc. Power to the electrical house will be supplied from the 13.8 kV line.

Emergency back-up generators with automatic transfer switches will supply the Project with back-up power.

A waste heat recovery system will collect waste heat generated by the power plant and transfer the heat to the process plant and other buildings for heating, via the glycol circulation system. Only non-toxic glycol, such as propylene glycol, will be used.

18.3 WASTE MANAGEMENT

Knight Piésold has developed a conceptual plan for the surface component of the waste rock, tailings and water management facilities. The WRMF and the TMWMF will be located approximately 600 m southwest of the mill.

The WRMF is sized to store a total of 3.6 Mt of barren and development rock over the 11-year mine life. This is the total quantity of waste rock produced by the mine, the excess rockfill from TMWMF construction, less the material required for underground backfill. The overall slope of the waste dump is assumed to be 2H:1V. The waste rock is assumed to be metal leaching, and runoff is routed to the TMWMF using seepage collection ditches for treatment before release into Kendrick Bay. An underdrain system will be installed at the WRMF foundation in order to minimize infiltration of runoff into the subsurface.

The TMWMF has been sized to accommodate 150,000 t of tailings solids in addition to the supernatant pond, one month of mine dewatering, the inflow design flood (IDF) and 2 m of freeboard for wind set-up and wave run-up. The total storage capacity of the TMWMF is 231,000 m^3 and 2 m of freeboard. The storage requirements decrease after Year 1 as the tailings will be recovered and used for paste backfill in the underground stopes.

The TMWMF basin will be partially excavated into bedrock using drill and blast techniques, and the blasted rock will be used as embankment construction material. The basin floor will be established at an elevation of 5 masl. The upstream and downstream slopes of the impoundment will be constructed at 2.5H:1V, with a maximum embankment height of 20 m.

It is anticipated that an engineered liner system will be required over the full basin floor area and embankments of the TMWMF to minimize seepage of potential contaminants into the subsurface. The impoundment will be lined with 80 mil high-density polyethylene (HDPE) liner that keys into the embankment crest, and is underlain by a geosynthetic clay liner (GCL) and bedding layer. The GCL minimizes the risk of puncturing the HDPE by the foundation materials and provides a secondary seepage barrier. The water that accumulates in the impoundment will be treated in a water treatment plant before it is released into Kendrick Bay.

Topsoil from the TMWMF and WRMF footprint will be stripped and stockpiled for use during reclamation. The topsoil stockpile is assumed to have side slopes of 4H:1V.

The waste rock in the WRMF will be re-graded and vegetated at the end of the mine life. The contact water will be collected in the TMWMF and treated before release into Kendrick Bay until water quality is acceptable. The TMWMF and water treatment plant will be decommissioned and removed upon confirmation of acceptable water quality. The TMWMF footprint will be reclaimed and re-seeded as appropriate.

Knight Piésold's full tailings and waste rock management report is provided Appendix G.

18.4 WATER MANAGEMENT

There are four types of water sources at the Property that will require management:

- groundwater reporting to the underground mine
- precipitation that will contact material associated with engineered facilities for the Dotson Ridge project (interim tailings pond, WRMF, and the mill area)
- small streams which will be diverted around the footprint of the facilities
- stormwater runoff from surface disturbance areas (roads, camp, topsoil stockpiles, etc.).

The groundwater reporting to the underground mine and the contact precipitation will be collected, treated, and recycled for use in the milling and paste backfill program. It is expected, based on the preliminary assessment, that excess water will need to be released back into the environment. This expectation requires that the PEA include capital and operating costs for a state-of-the-art water treatment plant. The water treatment plant has been sized to treat at an average rate of approximately 30 L/s (480 gpm). The need and type of water treatment facility will be determined during the feasibility study stage.

Small streams in the area proposed for development of the WRMF, the interim tailings pond, and the camp and mill processing area, will be diverted around the footprint of these facilities. This PEA includes costs for these diversions.

Stormwater runoff from roadways, the personnel camp, topsoil stockpiles and any other non-contact areas, will be treated by USFS-approved best management practices.

18.5 Fresh, Fire, and Potable Water Supply, and Sewage Disposal

The fresh and fire water will be primarily required for start-up and emergency purposes, gland seal water, reagent, flotation cleaning stages, and process water makeup. The gland and seal water will be pumped and distributed to the slurry pumps from the fire-fresh tank.

Fresh water will be supplied from the diversion system that diverts water from areas around the WRMF.

Potable water will be supplied from water wells. A potable water tank and hydro-chlorination system will be provided.

The sewage treatment plant will be a pre-packaged rotating biological contactor. The plant will be manufactured off-site and containerized for simple connection to the collection system on site. Once treated, the sewage treatment plant effluent will be discharged into the environment in accordance with the Alaskan regulations.

18.6 Communication

On-site communication systems will include a voice over internet protocol (VoIP) telephone system, a local area network (LAN) with wired and wireless access points, hand-held very high frequency (VHF) radios, underground leaky feeder systems for the underground mine and a satellite television system for the accommodations.

Off-site communications will be achieved with a satellite-base system, which will be utilized during the construction phase and the operating phase of the Project.

19.0 MARKET STUDIES AND CONTRACTS

There were no market studies conducted and no contracts reached between Ucore and refiners at the time that this PEA was completed.

Due to inceasing demand in the US and diminishing supplies from China, it is expected that the final REO products will be sold within the US.

Three-year trailing average of China freight on board (FOB) prices from October 2009 to October 2012 were used to establish prices for the REOs, except for holmium, lutetium, ytterbium, and erbium oxides, two-year trailing averages were used due to limited Chinese market data. Individual REO+yttrium oxide trailing averages are provided in Table 19.1.

Table 19.1 Oxide Prices Used in the Economic Analysis (Base Case)

REO	Metal-Pages Three-year Trailing FOB China Price (October Mean) (US$/kg)	Composition by Head Grade (%)
Ce_2O_3	47.21	28.37
Dy_2O_3	845.80	4.23
Eu_2O_3	1,834.94	0.36
Gd_2O_3	81.70	3.75
La_2O_3	48.69	9.33
Nd_2O_3	126.70	13.70
Pr_2O_3	113.10	3.52
Sm_2O_3	57.74	3.64
Tb_2O_3	1,520.83	0.67
Y_2O_3	80.41	27.01
Ho_2O_3	211.39*	0.84
Lu_2O_3	1,036.40*	0.19
Yb_2O_3	102.79*	1.75
Er_2O_3	88.20*	2.33
Tm_2O_3	N/A**	0.31
	Total	**100.00**

Notes: *Asian-metals two-year trailing averages
**Chinese market data is not available at this time. A trivial price is assumed ($0).
REE oxide prices are transaction based, therefore prices vary as determined by supply/demand, purity of oxide, and quantity.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 PHYSIOGRAPHY

The Project site falls within Alaska's Alexander Archipelago Temperate/Coastal (Southeast) Ecoregion (ADEC 1999; USGS 2011) and comprises subalpine, old growth western hemlock-spruce forest, marine intertidal, and marine subtidal habitats (USGS 1995). Elevations on Ucore's claim block range from sea level to 740 masl at Bokan Mountain, the highest point on the Property. The Dotson Ridge dike system, which is the focus of current development efforts, is less than 300 masl.

20.2 VEGETATION AND WILDLIFE

The subalpine habitat is characterized by barren rock and plants that have adapted to the colder and windier environment. Subalpine vegetation includes mountain hemlock, yellow cedar, and the dwarf form of the shore pine. Wildlife in the subalpine zone includes Sitka black-tailed deer, black bear, ptarmigan, and songbirds such as thrush and dark-eyed junco.

The old-growth habitat is characteristically dominated by western hemlock and Sitka-spruce. Other common vegetation includes red alder, western red cedar, Devil's club, and a variety of ferns and berries. The old growth forest supports a variety of songbirds as well as mammals such as the Sitka black-tailed deer, black bear, and mink. Kendrick Creek drains through the forested area and is joined by its tributaries Mine Fork Creek and Cabin Creek and discharges to the head of the West Arm of Kendrick Bay.

The marine intertidal zone is bounded by the low and high tides and occupies approximately 27 acres at the head of the West Arm of Kendrick Bay. The intertidal vegetative community includes many species of plants and algae including rockweed, eelgrass, sugar kelp, and bull kelp. Marine and estuarine invertebrates are common and include a variety of clams, crabs, and starfish, as well as chitons, worms, amphipods, isopods, and sea cucumbers. Birds using the inter-tidal zone include gulls, shorebirds, waterfowl, crows, belted kingfisher, and bald eagle. Mink are present in the intertidal zone and black bear and Sitka black-tailed deer inhabit the perimeter of this zone. Many of these plant and animal species extend into the contiguous subtidal waters. Wildlife in this zone includes sea otter, harbor seal, gulls, and loons and a variety of fish and invertebrates (Tetra Tech 2010).

20.3 FISH HABITAT

Four streams within the Bokan Mountain claim block are designated as important for the spawning or rearing of anadromous fish under Alaska Statute 41.14.870. Three of the streams are located west of Bokan Mountain and are not affected by the proposed mine development project. One drainage, Kendrick Creek, is within the area of the proposed project and is designated for the presence of coho, chum, and pink salmon. Although andromonous salmon species may use the stream for spawning and rearing, useable habitat is limited to the lower portion of Kendrick Creek where this habitat is only of moderate quality at best (Tetra Tech 2010). During the 2011 and 2012 field seasons, the Alaska Department of Fish and Game (ADF&G) conducted fish habitat studies in the Kendrick Bay Area and filed a preliminary report. There are no provisional findings or recommendations that will impact the mine plan as envisaged.

20.4 THREATENED OR ENDANGERED SPECIES

Based on review of current lists from the US Fish and Wildlife Service, the NOAA National Marine Fisheries Service, and the ADF&G, terrestrial threatened and endangered wildlife or plant species are not known to be present in the area (Tetra Tech 2010).

20.5 PERMITTING

The Property lies within the Tongass National Forest, and as such is on federal land, administered by the US Forest Service (USFS). A plan of operations will be developed from the pending feasibility study and submitted to the USFS to start the required National Environmental Policy Act (NEPA) review process. The successful completion of the environmental review process results in the issuance of a record of decision, which is an approval required to construct and operate the mine.

Table 20.1 lists the major permits that are required to be obtained for the construction and operation of the mine.

Table 20.1 Major Permits Required for Mine Construction and Operation

Agency	Permit(s)
Alaska Department of Environmental Conservation	Air Quality Permit Oil Discharge Prevention and Contingency Plan Solid Waste Disposal Permit Storm Water Pollution Prevention Plan
ADF&G	Title 16 Fish Habitat Permit
Alaska Department of Natural Resources	Tideland Lease Permit Water Rights Permit Dam Safety Permit Cultural Resource Clearance
Bureau of Alcohol, Tobacco, and Firearms	Explosives Permit
US Coast Guard	Facility Response Plan
US Army Corps of Engineers	404 Certification – Waters of the US
Environmental Protection Agency	Spill Prevention, Containment, and Countermeasures Plan
USFS	Record of Decision – Plan of Operations Approval
Federal Aviation Authority	Notice of Intent – Landing Area

20.6 COMMUNITY CONSULTATION

The purpose of this program is to ensure that all potentially affected persons businesses, and communities have a full understanding of the Project and access to continuing engagement, training programs, and employment and business opportunities. As an example, discussions have been initiated with local fishing organizations and Alaska tribal associations to address potential issues and incorporate planning strategies into the Project development. It is recognized that the Project area is located within the traditional territory of the Kasaan.

20.7 RECLAMATION

Mine closure and reclamation plans will be necessary to address the closure of all project components. Mine closure plans are developed in accordance with state and federal criteria and approval. The closure plan must be presented as a part of the plan of operations submitted to the USFS for review. Financial security must be posted once the record of decision has been obtained from the USFS.

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

21.1.1 SUMMARY

The capital cost for the Project is estimated at $221.2 million and is separated into direct and indirect capital costs. Direct capital is estimated to be $134.7 million and indirect capital is estimated at $86.5 million, with an accurancy range of ±35%.

This estimate includes provisions for both the mining and mineral process. Infrastructure and project facilities include the process plant, ancillary buildings and accommodations, roadways, material conveyance, tailings and waste management facilities and related site development costs.

The capital cost estimate is expressed in US dollars unless otherwise noted. An exchange rate of US$1.00:Cdn$1.00 is used, unless otherwise noted.

The total capital cost breakdown is shown in Table 21.1.

Table 21.1 Initial Capital Cost

Item	Cost (US$)
Direct Capital Costs	
Site Development	6,053,102
Mine Underground	18,900,000
Mine Surface Facilities	23,800,000
Process	62,919,481
Tailings and Waste Rock Management	10,085,000
Utilities	3,360,798
Buildings	2,975,935
Temporary Facilities	5,170,840
Plant Mobile Equipment and Miscellaneous	1,409,975
Subtotal	**134,675,131**
Indirect Capital Costs	
Indirect Construction Costs	51,056,285
Owner's Costs	10,923,051
Contingency	24,541,000
Subtotal	**86,520,336**
Total Cost	**221,195,467**

Capital cost exclusions include:

- environmental permitting costs
- exploration activities
- working capital to cover pre-operation costs due to construction delays
- cost of financing
- royalties and taxes.

A summary of the capital costs is provided in Appendix H.

21.1.2 MINING

The total mine capital costs for the mining portion of this Project are estimated to be $151 million, which comprises $55.4 million in initial (start-up) capital, and $95.6 million in sustaining capital over a period of 11 years. Pricing for mine development and major production equipment was provided by Stantec, based on direct quotes and experience from other similar projects, factored for operation size and location.

21.1.3 PROCESSING PLANT – MINERALIZED MATERIAL AND HYDROMETALLURGICAL

The capital costs for the initial construction of the processing plant (provided in Table 21.2) are estimated to be $62.9 million. These costs are based on typical process requirements for processing REE-baring material in crushing, grinding, x-ray and magnetic sorting circuits of an equivalent model for a 1,500 t/d operation. The estimate includes a hydrometallurgical circuit utilizing nitric acid leaching with a 375 t/d leaching circuit.

The estimate provides for concentrate storage, modular accommodation units, load out for concentrate shipment, material transportation from mine to mill and assay lab facility. The estimate also includes respective unit operations in the mineral processing circuit. Inputs for process plant capital costs are based on metallurgical information and equipment specs provided by Lyntek and IntelliMet.

Table 21.2 Processing Plant Capital Costs

Processing Facility	Cost ($)
Mill Building	7,981,333
Crushing	391,196
Grinding	11,653,969
Leaching	32,142,697
Recovery	6,817,371
Miscellaneous	3,932,915
Total Cost	**62,919,481**

21.1.4 Tailings and Waste Rock Management Facilities

On a preliminary basis, capital cost expenditure in the order of $10.1 million is estimated for the construction of the TMWMF and WRMF, as provided by Knight Piésold.

21.1.5 Sustaining Capital

Mining Equipment

A sustaining capital cost of $95.7 million for additions, replacements, and re-builds of mining equipment has been estimated to match the annual production schedule tonnages and unit operating hours of the mining fleet. The cost was provided by Stantec and is included in the financial model for the Project.

Tailings and Waste Rock Management Facilities

A sustaining capital of $644,000 for the tailings and waste rock management facilities was provided by Knight Piésold and is included in the financial model for the Project.

Process Plant, Equipment and Site Infrastructure

Sustaining capital for the process plant, equipment and other site infrastructures was estimated at 3% per year of the direct capital spend extended over the 11-year operating plant and mine life.

21.2 Operating Costs

21.2.1 Summary

The total operating cost for the mine and processing over the LOM is estimated at $636 million which equates to $122.78/t mined. The breakdown of operating costs is shown in Table 21.3.

An exchange rate of US$1.00:Cdn$1.00 is used, unless otherwise noted.

A summary of the operating costs is provided in Appendix I.

Table 21.3 Summary of Operating Costs

Item	Average Unit Cost ($/t mined)	Total LOM Cost ($ million)
Mining	41.69	216.0
Processing	54.83	284.0
G&A	13.56	70.2
Power	11.78	61.0
Miscellaneous	0.93	4.8
Total Cost	**122.78**	**636.0**

21.2.2 MINING

The total mine operating cost is estimated at $215,700,000 which equates to $41.69/t of material mined or $166.74/t of material leached. The mine operating costs includes mine engineering supervision, mine operations supervision, accommodations and travel for mine technical, operations, and maintenance staff and light duty vehicle operating costs.

Detailed mining manpower requirements and personnel list are discussed in Section 16.0.

21.2.3 PROCESSING PLANT

SUMMARY

The total process plant operating cost is estimated at $284 million which equates to $54.83/t of material mined. Table 21.4 outlines the operating cost of running the process operations.

Table 21.4 Process Plant Operating Costs

Operating Cost – Area	Total Annual Cost During Full Production ($)	Average Unit Cost ($/t mined)
Process Labour	3,709,580	6.87
Process Supplies	12,216,000	22.62
G&A Labour	1,302,075	2.41
G&A Supplies	6,020,775	11.15
Power Supply	6,358,902	11.78
Total Cost	**29,607,332**	**54.83**

LABOUR

Tetra Tech prepared and estimated the labour details based on the required personnel for a 375 t/d acid leaching plant with mineralized material sorting. Base rates from other similar Tetra Tech projects in the area were considered for the purposes of this PEA. An additional burden of 35% is considered for health benefits, pension, overtime, training, etc.

Labour costs are based on a required manpower and rates to operate the plant. The suggested scheme employs 30 persons for the plant operations and 9 persons for mill maintenance, for a total of 39 people for the mineral processing. The total annual cost is $3.71 million.

REAGENTS AND SUPPLIES

The nitric acid and supply types and consumptions are based on the requirements for 375 t/d operations of the leaching circuit. The total costs for reagents and supplies are $12.2 million per year or $101.46/t.

POWER

The power costs include costs of fuel (LNG) and power plant operating and maintenance costs. Tetra Tech estimated the electrical power cost at $0.18/kWh based on other comparable projects. Three LNG power generating units are proposed for the Project. Consistent with industry, the operating cost was developed based on the LNG consumption, which was provided by a LNG gen set manufacturer. The total costs per annum are $6.36 million.

21.2.4 G&A OPERATING COST ESTIMATE

G&A costs are the costs that do not relate directly to the mining or processing operating costs. This includes:

- personnel – general manager and staffing in accounting, purchasing, and environmental departments, as well as G&A
- various employees in surface services and head office in Ketchikan, Alaska
- G&A expenses including insurance, various administrative supplies, medical services, legal services, human resources related expenses, travelling, road maintenance, as well as external assaying and testing.

The G&A expenses are estimated at approximately $7.32 million.

22.0 ECONOMIC ANALYSIS

A PEA should not be considered a prefeasibility or feasibility study, as the economics and technical viability of the Project have not been demonstrated at this time. The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Furthermore, there is no certainty that the conclusions or results as reported in the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Tetra Tech conducted an economic evaluation of the Project, incorporating all the relevant capital, operating, working, sustaining costs, and royalties. The evaluation was based on a pre-tax financial model, and was calculated in US dollars. For the 11-year mine life and 5,175,889 LOM tonnes mined, the following pre-tax financial parameters were calculated using the base case prices:

- 43% IRR
- 2.3-year payback on US$221 million capital
- US$577 million NPV at a 10% discount value.

Ucore commissioned PwC in Vancouver, BC to prepare a tax model for the post-tax economic evaluation of the Project, with the inclusion of applicable federal and state taxes. The post-tax analysis is discussed in Section 22.4.

The following post-tax financial parameters were calculated:

- 35% IRR
- 2.5-year payback on US$221 million capital
- US$368 million NPV at a 10% discount rate.

The base case (outlined in Table 22.1) uses the three-year trailing average prices from October 2009 to October 2012.

Table 22.1 Base Case Prices

REO	Base Case Prices (US$/kg)
La_2O_3	48.69
Ce_2O_3	47.21
Pr_2O_3	113.10
Nd_2O_3	126.70
Sm_2O_3	57.74
Eu_2O_3	1,834.94
Gd_2O_3	81.70
Tb_2O_3	1,520.83
Dy_2O_3	845.80
Ho_2O_3	211.39
Er_2O_3	88.20
Tm_2O_3	N/A
Yb_2O_3	102.79
Lu_2O_3	1,036.40
Y_2O_3	80.41

Sensitivity analyses were developed to evaluate the Project economics.

22.1 PRE-TAX MODEL

22.1.1 REO PRODUCTION IN FINANCIAL MODEL

The revenues projected in the cash flow model were based on the average REO values indicated in Table 22.2.

Table 22.2 REO Production

	LOM
Total tonnes mined ('000)	5,176
Mine life (years)	11
Average Grade (%)	
Ce_2O_3	0.134
Dy_2O_3	0.020
Er_2O_3	0.011
Eu_2O_3	0.002
Gd_2O_3	0.018
Ho_2O_3	0.004
La_2O_3	0.044

table continues…

	LOM
Lu_2O_3	0.001
Nd_2O_3	0.065
Pr_2O_3	0.017
Sm_2O_3	0.017
Tb_2O_3	0.003
Tm_2O_3	0.001
Y_2O_3	0.128
Yb_2O_3	0.008
TREO	0.473
Total Production (kg)	
Ce_2O_3	6,096,947
Dy_2O_3	889,121
Er_2O_3	388,222
Eu_2O_3	75,537
Gd_2O_3	778,320
Ho_2O_3	149,377
La_2O_3	2,009,622
Lu_2O_3	25,126
Nd_2O_3	2,940,686
Pr_2O_3	758,162
Sm_2O_3	771,549
Tb_2O_3	133,350
Tm_2O_3	48,958
Y_2O_3	4,782,634
Yb_2O_3	256,441
TREO	20,104,052

22.1.2 *FINANCIAL EVALUATIONS – NPV AND IRR*

The production schedule has been incorporated into the 100% equity pre-tax financial model to develop annual recovered REO production from the relationships of tonnage processed, head grades, and recoveries.

REO revenues were calculated based on market prices. Unit operating costs for mining, processing, site services, G&A, and off-site charges (transportation, insurance and royalties) areas were applied to annual mined tonnages to determine the overall operating cost, which was deducted from the revenues to derive annual operating cash flow (net revenue).

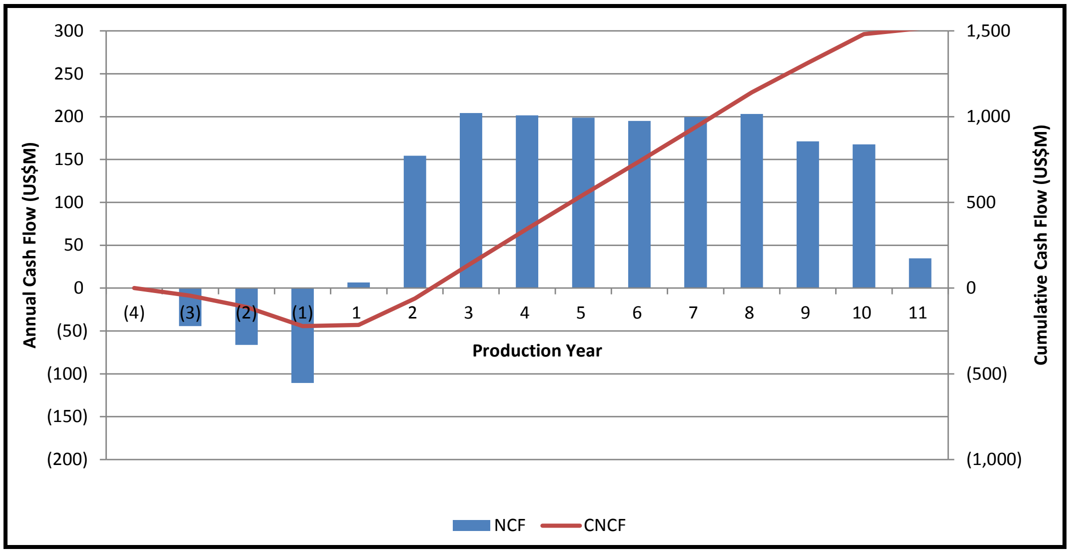
Initial and sustaining capital costs have been incorporated on a year-by-year basis over the LOM, and then deducted from the net revenue to determine the net cash flow before taxes. Initial capital expenditures include costs accumulated prior to first production of REO; sustaining capital includes expenditures for mining and processing additions, replacement of equipment, tailings embankment construction, water treatment, and environmental/closure costs.

The financial analysis used three months of the on-site operating cost as working capital cost. The working capital is recovered at the end of the mine life.

Reclamation costs were estimated at US$4 million.

The undiscounted annual cash flows are illustrated in Figure 22.1.

Figure 22.1 Pre-tax Undiscounted Annual and Cumulative Cash Flow



Notes: NCF = net cash flow; CNCF = cumulative net cash flow

22.2 REO PRICE SCENARIOS

Tetra Tech evaluated the base case, as well as two additional REO price scenarios. For the base case, Tetra Tech used a three-year trailing average of China FOB prices from October 2009 to October 2012 to establish prices for the REOs, except holmium, lutetium, ytterbium and erbium oxides, where two-year trailing averages were used due to limited Chinese market data. The two alternate cases are based on pricing REOs based on a six-month trailing average and a three-month trailing average.

The pre-tax financial model was established on a 100% equity basis, excluding debt financing and loan interest charges. The financial outcomes have been tabulated for NPV, IRR and payback of capital. Discount rates of 8%, 10% and 12% were applied to all scenarios. The results are presented in Table 22.3.

Table 22.3 Summary of Pre-tax NPV, IRR and Payback by REO Price Scenario

	Unit	Base Case Three-year Trailing Average	Alternate Case 1 Six-month Trailing Average	Alternate Case 2 Three-month Trailing Average
REO				
La_2O_3	US$/kg	48.69	20.85	18.42
Ce_2O_3	US$/kg	47.21	21.38	19.23
Pr_2O_3	US$/kg	113.10	110.00	103.08
Nd_2O_3	US$/kg	126.70	108.96	101.58
Sm_2O_3	US$/kg	57.74	71.79	61.42
Eu_2O_3	US$/kg	1,834.94	2,185.00	2010.00
Gd_2O_3	US$/kg	81.70	99.42	96.35
Tb_2O_3	US$/kg	1,520.83	1,907.12	1,840.38
Dy_2O_3	US$/kg	845.80	1,009.42	948.08
Ho_2O_3	US$/kg	211.39	107.25	107.05
Er_2O_3	US$/kg	88.20	153.61	140.08
Tm_2O_3	US$/kg	N/A	N/A	N/A
Yb_2O_3	US$/kg	102.79	124.07	110.51
Lu_2O_3	US$/kg	1,036.40	1,420.79	1,427.56
Y_2O_3	US$/kg	80.41	100.75	85.12
Economic Results				
NCF	US$ million	1,516	1,620	1,379
NPV (at 8%)	US$ million	697	747	623
NPV (at 10%)	US$ million	577	620	513
NPV (at 12%)	US$ million	478	515	423
IRR	%	43	44	40
Payback	years	2.3	2.2	2.4

22.3 SENSITIVITY ANALYSIS

Sensitivity analyses were carried out on the following parameters:

- REO price
- initial capital expenditure
- on-site operating costs.

The analyses are presented graphically as financial outcomes in terms of NPV and IRR. The Project NPV is most sensitive to REO price followed by operating costs and initial capital. The Project IRR is most sensitive to REO price followed by initial capital and operating costs. The NPV and IRR sensitivities are shown in Figure 22.2 and Figure 22.3.

Figure 22.2 Sensitivity Analysis of Base Case Pre-tax NPV at 10% Discount Rate

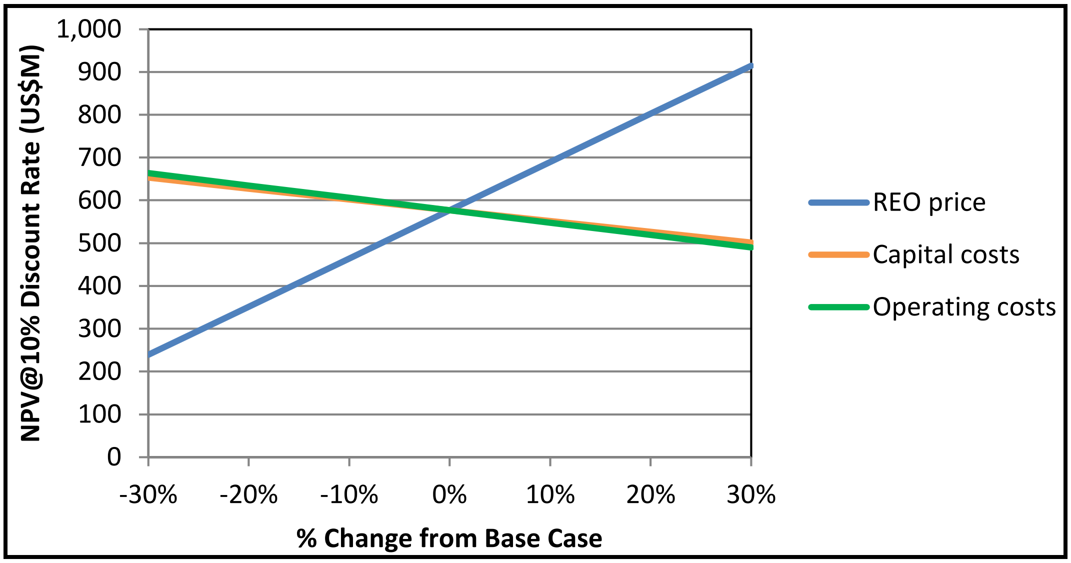
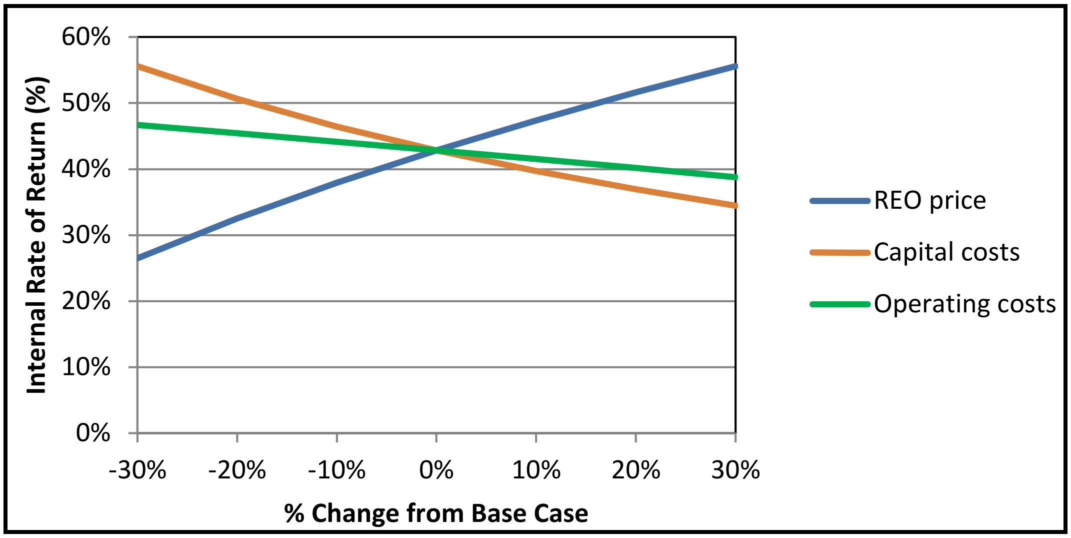


Figure 22.3 Sensitivity Analysis of Pre-tax Base Case IRR



22.4 POST-TAX FINANCIAL ANALYSIS

Ucore commissioned PwC in Vancouver, BC to prepare a tax model for the post-tax economic evaluation of the Project, which included applicable federal and state taxes.

The following sections outline the currently applicable general tax regime.

22.4.1 US FEDERAL AND STATE TAXATION REGIME

For US federal income tax purposes, in accordance with the Internal Revenue Code (IRC), a taxpayer is required to calculate taxes under both the regular corporate tax system and the Alternative Minimum Tax (AMT) system and pay whichever method results in the higher amount of taxes. The statutory US federal income tax rate is 35% and the tax rate under AMT is 20%. The maximum Alaska state income tax rate is 9.4%. As state taxes are deductible for federal purposes, the combined statutory income tax rate for the Project will be 41.1% of taxable income.

Net operating losses generated in a given year may be carried forward for 20 years and applied to taxable income when it arises, or carried back two years and applied against taxable income from the Project in those years. The IRC also provides certain deductions to incentivize investment by mining companies, including depletion and development expenditures. The tax rates for US federal and Alaska state income are provided in Table 22.4 and Table 22.5, respectively.

Table 22.4 US Federal Tax Rate

Taxable Income (US$)	Tax Rate (%)
0 - 50,000	15
50,000 - 75,000	25
75,000 - 100,000	34
100,000 - 335,000	39
335,000 - 10,000,000	34
10,000,000 - 15,000,000	35
15,000,000 - 18,333,333	38
18,333,333 +	35

Table 22.5 Alaska State Income Tax Rate

Taxable Income (US$)	Tax Rate (%)
0 - 10,000	1
10,000 - 20,000	2
20,000 - 30,000	3
30,000 - 40,000	4
40,000 - 50,000	5
50,000 - 60,000	6
60,000 - 70,000	7
70,000 - 80,000	8
80,000 - 90,000	9
90,000 +	9.4

22.4.2 DEPLETION

Generally speaking, depletion, like depreciation, is a form of cost recovery. Just as the owner of a business asset is allowed to recover the cost of an asset over its useful life, a mining company is allowed to recover the cost of mineral property. Depletion is taken over the period that mineral is being extracted.

For federal income tax purposes, two forms of depletion are allowed: cost depletion and percentage depletion. The taxpayer is required to use the method that will result in the greatest deduction.

22.4.3 COST DEPLETION

Cost depletion is the process of charging a portion of the property cost to the production cost. This is based upon the premise that the value of a reserve declines in proportion with the quantity of minerals extracted and removed. The first step of this method is to determine the number of units (as of the beginning of each year), which comprise the deposit. The unit can be any measure of production such as tonnes of ore, barrels of oil, board feet of timber, etc. The taxpayer must be consistent from year to year in the type of unit being calculated to ensure uniformity. The second step takes the cost or adjusted basis of the property, which pertains to the deposit and divides this basis by the total number of units to obtain the depletion cost per unit. The depletion cost per unit is multiplied by the total units sold during the year to arrive at cost depletion.

22.4.4 PERCENTAGE DEPLETION

Under the percentage depletion method, a flat percentage of adjusted gross income from the activity is used to calculate the depletion allowance. The deduction for depletion cannot exceed 50% of the adjusted taxable income from the activity. This limitation is computed without regard to the depletion allowance. The amount of the deduction allowable under percentage depletion is not limited by the basis of the property. Thus, even though the basis of the property is reduced by the amount of depletion taken, if the basis becomes zero, the depletion based on the percentage of adjusted gross income may continue. However, if cost depletion in a given taxation year yields a higher deduction, it must be used to calculate the amount deducted.

22.4.5 PRODUCTION TAXES

Alaska collects taxes relating to mineral production. Production taxes include a mining licence tax and an Alaska state royalty. The State of Alaska has no state sales tax.

The Alaska mining licence tax is assessed on mining net income and royalties received in connection with properties and activities in Alaska. Except for sand and gravel operations, new mining operations are exempt from the mining license tax for a period of 3.5 years from the commencement of initial production. The maximum mining licence tax rate is 7% on net income over $100,000.

All holders of Alaska state mining locations must pay a production royalty on all revenues received from minerals produced on state land. The production royalty requirement applies to all revenues received from minerals produced from a state mining claim or mining lease during each calendar year. The Alaska state royalty is calculated at 3% of net income from mining operations located on Alaska state lands (not applicable as Ucore's mineral properties are located on federal land).

There is no provision in the legislation to carry losses forward to offset future profits in the state royalty, mining licence or severance tax calculation. At the base case REO prices used for this study, total estimated taxes payable are US$499 million over the 11-year LOM. The components of the various taxes that will be payable are shown in Table 22.6.

Table 22.6 Components of the Various Taxes

Tax Component	LOM Amount (US$ million)
Federal Income Tax	334.72
Alaska State Tax	164.28
Total Taxes	**499.00**

Base case post-tax financial results are summarized in Table 22.7.

Table 22.7 Summary of Post-tax Financial Results

Description	Value
Net Cash Flow (US$ million)	1,017
Discounted Cash Flow NPV (US$ million) at 8%	450
Discounted Cash Flow NPV (US$ million) at 10%	368
Discounted Cash Flow NPV (US$ million) at 12%	300
Payback (years from start of mill operations)	2.5
IRR (%)	35

22.5 ROYALTIES

The royalty consists of US$2.00/t mined.

22.6 SMELTER TERMS

Smelter terms are not applicable for the Project, because the REOs are sold directly to the market without smelting.

22.7 TRANSPORTATION LOGISTICS

Transportation costs of US$100.00/t REO were included in the economic analysis.

22.7.1 INSURANCE

An insurance rate of 0.15% was applied to the provisional invoice value of REO.

23.0 ADJACENT PROPERTIES

There are no material properties adjacent to the Property that is the subject of this PEA report.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT DEVELOPMENT PLAN

In order to achieve the schedule, the long-lead process equipment will need to be identified at the beginning of the feasibility study stage. The critical path of the Project is through the supply and delivery of this equipment.

The early start date is driven by the civil construction work. To achieve this schedule, several construction packages will need to be issued as unit rate packages. The unit rate packages will include rough grading, concrete and structural steel buildings, and interior steel platforms.

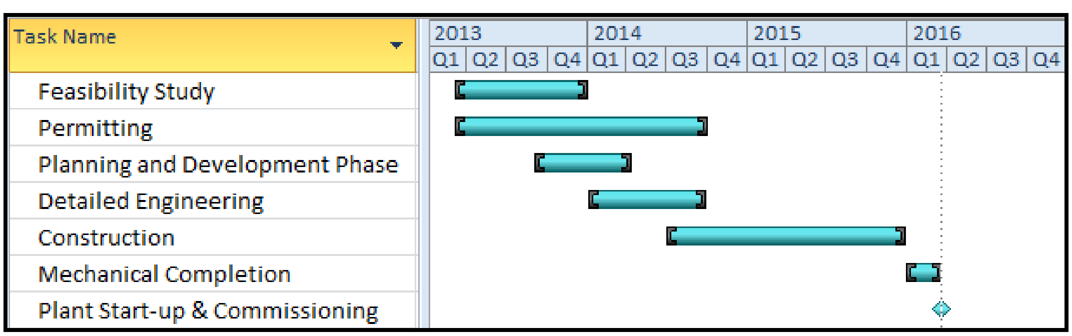
Temporary construction facilities will be mobilized in summer 2014, including the batch plant and aggregate plant. Site preparation, grading, and road construction will commence immediately upon receipt of permits and approvals. Modular construction will be utilized wherever practical in order to reduce field construction.

The concrete for the main process building, truck shop, and powerhouse building foundations will be poured in the summer 2014 to allow the buildings to be erected. Once the buildings are erected, the concrete inside the buildings (including equipment supports) can be poured in a controlled environment through the winter.

Electrical and mechanical installation contracts will be bid lump sum to qualified contractors. A start-up and commissioning period has been allowed at the completion of construction in order to complete mechanical check out and acceptance and commissioning of the facilities.

A summary schedule for the Project is shown in Figure 24.1.

Figure 24.1 Project Summary Schedule



25.0 INTERPRETATION AND CONCLUSIONS

25.1 GEOLOGY

Bokan Mountain represents an unusual style of REEs, uranium, niobium, and zirconium mineralization hosted within, or proximal to, a circular Jurassic A-type peralkaline intrusive complex known as the Bokan Mountain Granite. The high-level intrusion contains several concentrically zoned intrusive phases of sodic granites, porphyries and pegmatites. Aegerine and riebeckite are the dominant ferromagnesian minerals.

Intense wall-rock replacement/alteration by albite, chlorite, calcite, fluorite, and hematite accompanied the ore-forming processes and mineralization. Mineralization as veinlets, disseminations or as irregular sublinear masses is hosted within, or closely proximal to, the Bokan Mountain Granite and was derived from magmatic ore fluids. There are several types of uranium-thorium-REE mineralization, the most important of which are: (1) irregular structurally-controlled pipes, (2) shear zone-related pods or lenses (veins), and (3) pegmatitic or felsic dikes. The type three felsic dykes or “vein Dykes” are the predominant mineralization type in the Dotson Zone, which is the current exploration target on the property and the subject of this report. These vein dykes form a zone of potentially economic mineralization approximately 50 m wide and 2,100 m long, striking west-northwest and dipping steeply to the north. The vein dykes range in thickness from a few millimetres to tens or hundreds of centimetres and contain important concentrations of thorium, niobium, tantalum, beryllium, zirconium and REEs (both light and heavy REEs). Niobium and tantalum are present chiefly in euxenite and columbite while REEs are contained in thalenite, bastnaesite, xenotime and monazite.

Exploration work on the Dotson Zone has confirmed the presence of continuous REE mineralization over a defined strike length of 2,140 m with an average width of 50 m. The west end of the Dotson Zone trend is 300 masl while the east end trends almost into Kendrick Bay at sea level. The zone remains open on strike to the west and at depth. Mapping, prospecting, trenching and drilling defined 25 veins, dykes and vein arrays within the Dotson Zone. The veins generally dip steeply north. Most veins are less than 1.5 m wide but at least 5 of the veins have section widths up to 3 m and grades averaging around 1.1% TREOs. The vein system is characterized by a high proportion of H REEs (europium to lutetium and yttrium), averaging about 40% of the TREOs.

A multi-season diamond drilling program was carried out on the Property between 2007 and 2010. A total of 5,774 m of drilling was completed in 37 holes on the Dotson structure. During 2009 and 2010, 55 channel samples were cut across veins outcropping in the Dotson zone. The results of this drilling and sampling were compiled and interpreted, a deposit model was constructed and a mineral resource estimate was calculated. Mineral resources for the Project were classified in accordance with NI 43-101 definitions. Table 25.1 summarizes the estimated Inferred Mineral Resources published in April 2011.

Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

Table 25.1 Resources at 0.4% TREO Cut-off

Type	Tonnes	LREE (ppm)	HREE (ppm)	TREE (ppm)	LREO (%)	HREO (%)	TREO (%)
Inferred	5,228,200	3,368.77	2,113.50	5,482.25	0.394	0.259	0.653

25.2 MINING

The Project envisages the mining of 540,000 t/a of a significant REO resource using underground mining methods with proven mining technologies. The emplacement of tailings as backfill into the excavated voids adds to the stability of the underground excavations and allows for the suitable disposal of the tailings. The design of the underground mining has been completed to the scoping level and additional design work will need to be undertaken using updated geological/geotechnical and economic data as more information is generated at the Project site.

The following risks have been identified with the mine plans evaluated for the underground mining at Bokan Mountain:

- Mine designs and associated work was prepared at a conceptual level, which requires additional engineering and design to ensure the level of design is commensurate to any financial decisions that Ucore is considering in relation to the deposit.
- Lateral development rates may be considered aggressive. Slower development rates will lengthen the pre-production schedule and defer revenue, affecting the overall economics.
- All ground support packages utilized for the Project are only based on visual inspection of the drill core and ore outcrop at site. Further geotechnical work is required to better identify a proper ground support package.
- The 20 m crown pillar currently estimated for the Project should be designed using adequate geotechnical data to ensure stability of the surface.

- Portal costs are based on past projects of similar size. There is no design completed at this level of study.
- The selected location of the ventilation raises assumes adequate geotechnical conditions. Further geotechnical information is required to verify this assumption.
- Stope shapes in the current design follow the exact contour of the Dotson Vein structures adjusted to a 3.5 m minimum mining width. Further detail design for stopes would require adjusting the shapes to a more contiguous shape. This adjustment would affect the planned dilution associated with the stopes.
- Paste plant is based on a larger project and scaled down to fit the paste backfill requirement. An independent study focusing on all aspects of utilizing paste for backfill is recommended.
- Paste mix design requires further test work associated with expected tailings properties. The mix design could change and have an effect on paste backfill costs.
- Paste backfill mix design was not altered for extra strength for the stopes that will require underhand mining method. The paste backfill in these stopes will require extra strength which will have an effect on the paste backfill cost.
- Planned and unplanned dilution are included in the overall stope shape. No extra dilution is included in the production schedule. Further work is required to optimize the exact stope dimensions that represent the vein width per zone.
- The production schedule assumes all backfilling is accomplished by paste backfill and enough tails is available to meet the requirement. No allowances are made in the truck haulage selection, quantity required, or operating costs for backhauling waste material.
- The pre-production period is scheduled for two years in duration. If the preproduction schedule is not met then this will affect the production ramp up period as well.
- Production build up does not consider a significant learning curve from an operational readiness standpoint. Blasthole and paste backfill operational planning is complex both from a technical and logistical standpoint, which presents risk related to meeting the schedules. Risk of higher upfront costs may be incurred to ensure operational readiness.
- All costs presented herein are preliminary with very little engineering performed (±40%).
- Equipment availability is current a risk being realized by all mine operators. Significant lead times are currently required for new mobile equipment from vendors and suppliers.

The following opportunities have been identified with the mine plans evaluated for the underground mining at Bokan Mountain:

- Greater economic opportunity exists by adjusting minimum mining widths based on specific vein widths in each area of the Dotson Vein structure and by identifying areas where two or more vein structures can be mined simultaneously by taking a wider stope.
- Employment of the resueing mining technique in the narrow areas of the vein will decrease the planned dilution associated with stopes and increase the overall grade.
- The portion of the mineral resource estimate used in the mine plan assumes a recovery of 90%. Further evaluation may allow for a higher mining recovery, perhaps to as high as 95%.
- Economic opportunities exist with identifying potential alternative backfill methods. If mill tailings are limited, waste rock can be used to backfill the stopes. Also, a slurry mix of tails can still be pumped into the stopes without the need of a paste plant, lowering backfill costs.
- Installing an additional decline at the upper west end of the ore body could provide a reduction in underground truck haulage costs and the possibility of bringing more mining areas online sooner.

25.3 Waste and Water Management

The waste and water management concept for the Project includes a lined TMWMF for water management and temporary storage of 150,000 t of tailings, and a WRMF for storage of 3.6 Mt of waste rock. The tailings in the TMWMF are re-slurried and used as underground paste backfill after Year 1. Quantities and preliminary capital costs were developed to provide initial costing for the Project financial model and this PEA report.

25.4 Mineral Processing

The REE processing facility to treat 1,500 t/d of mined material was designed based on the results of the metallurgical test programs conducted at Commodas, Hazen and IntelliMet.

The process design includes primary and secondary crushing, screening, DEXRT sorting and magnetic separation circuits, as well and tertiary crushing, a rod mill and a tower mill grinding circuits.

A leaching circuit will produce an upgraded concentrate for further treatment, with the waste sent to the paste backfill plant to be used as cemented backfill for filling mined out areas underground.

A solid phase extraction circuit will separate the bulk REE concentrate into individual REOs as sellable products.

25.5 ENVIRONMENTAL

The necessary steps required to permit, construct, and operate the rare earth mine, from an environmental perspective are well understood. Consultation with both state and federal regulators, as well as community stakeholders, has been initiated and will be ongoing throughout the permitting and operating period of the Project. Baseline environmental studies have been initiated and will continue throughout 2013 as the feasibility study progresses and the Project description is further defined. A plan of operations will be developed from the pending feasibility study and submitted to the USFS to start the required NEPA review process. The successful completion of the environmental review process results in the issuance of a record of decision, which is an approval required to construct and operate the mine.

26.0 RECOMMENDATIONS

26.1 GEOLOGY

Ucore should retain a structural mapping consultant for a two-to-three week mapping program. There have been no definitive conclusions made regarding the offset distance and direction on the camp creek fault or what happens to the mineralized veins at the east and west ends of the Dotson Zone. Careful structural mapping should be able to shed some light on the structural processes controlling the offsets and termination of the mineralized structure. The estimated cost of this mapping project is $25,000 including the cost of a field report.

Ucore should continue to collect bulk density data for material from the mineralized veins and the various host rock material in the footwall and hanging wall of the mineralization along the Dotson structure. This is required in order to get a better determination of the densities of the different potential mill feed material. More samples should be sent to independent laboratories for confirmatory testing. The costs for internal testing while logging and teching the core are nominal and the costs for the external, confirmatory testing should amount to no more than $5,000.

The drill results from the 2011 diamond drill program need to be incorporated into a new deposit model and mineral resource estimate, and the results of this new modelling reported. The drilling has been complete for some time, and it is necessary to analyze the data and update the deposit model. The cost for re-modelling the deposit, re-calculating the resource estimate and reporting the results is estimated at no more than $60,000.

Ucore should continue to work with their metallurgical consultants on treatment of material from other parts of the deposit. Ucore should take more material from several test pits in the centre and western portions of the deposit The data from this new testing should be compared with the results from the bulk sample material removed in the summer of 2011 to ensure that the nature of the mineralization is fairly homogeneous across the Property. It is estimated that the cost of carrying out the quarrying, shipment of the samples and lab testing of the excavated material will be approximately $135,000.

26.2 MINING

This scoping-level study for the development of an underground mine at Bokan Mountain indicates that the mine can produce about 1,500 t/d of mineralized material and waste over an 11-year LOM at $43.54/t mined with a pre-production capital requirement of $55.4 million. These scoping-level estimates are within reasonable limits for the Project area; Stantec recommends further investigations to improve the estimates in the next level of study.

The following recommendations are made for evaluation in the next phase of investigations:

- Geotechnical/hydrogeological assessments: Further geotechnical data collection and assessment will be required to better design the stopes and for identification of appropriate ground support packages. The data will also be used for the design of an adequately sized surface crown pillar and for the optimal locations of infrastructure such as ventilation raises. Hydrological assessments of water inflows into the mine due to the high precipitation in the area will also be required. The estimated cost for this work is $500,000, including drilling of additional geotechnical holes.
- Production rate: After the resource categories have been upgraded and the tonnage and distribution of mineral resources is identified with greater confidence, trade-off studies should be undertaken to determine optimal mine production rate as part of prefeasibility or final feasibility study targeting higher grade areas early in the mine life. The estimated cost for this work is $50,000.
- Key equipment selection: Trade-off studies need to be undertaken to select appropriately sized key fixed and mobile equipment. The design of excavations will need to be reviewed based on the selected equipment. The estimated cost for this work is $75,000.
- Backfill study: An independent study focusing on all aspects of utilizing paste for backfill is recommended. Paste mix design requires further test work associated with expected tailings properties. The estimated cost for this work is $100,000 including testing and mix design.
- Ventilation: Model the ventilation network of the mine at various stages to optimize the size of the main ventilation drifts and raises, optimize main fan and auxiliary fan configurations such as centrifugal versus vane-axial fans on surface; determine strategy to maintain flexibility of ventilation system over the mine life. The estimated cost for this work is $100,000 including fan selection.
- Overall risk assessment: Conduct risk assessment sessions for the mine development and operation plan to characterize key risks and prioritize follow-up engineering and other work to mitigate. The estimated cost for this work is $50,000.

The total cost to complete the recommended mining work is $875,000.

26.3 Waste and Water Management

The waste and water management concept presented herein has the potential for optimization during subsequent studies as the geotechnical conditions, design basis and operating criteria are further refined. Knight Piésold makes the following recommendations:

- Optimization of the waste and water management facilities layout. Knight Piésold recommends that optimization of the waste and water management facilities locations be completed to ensure that the best option from an environmental, engineering and economic perspective is carried forward.
- Geochemical characterization of waste rock and tailings. Knight Piésold recommends that the geochemical characterization of waste rock and tailings be confirmed and refined. The geochemical characteristics are required to determine water quality and water treatment requirements.
- Development of an updated water balance with refined input values. A preliminary water balance has been completed as part of this study that indicated a water surplus. Knight Piésold recommends updating the water balance with refined input values. The current estimate for mine dewatering involves a high degree of uncertainty. A reduction of mine dewatering could require substantial make-up water demand from the TMWMF.
- Geotechnical site investigation. Knight Piésold recommends completing a geotechnical site investigation. This investigation would characterize the foundation conditions for the impoundment, with a focus on determining the depth and geotechnical characterization of the overburden in the area. Soil characterization would be determined through in situ testing as well as laboratory index testing of collected samples. Rock core would be logged with a focus on rock mass quality, and rock core samples would be collected for laboratory strength testing. Hydraulic conditions would be determined through the use of in situ hydraulic testing.

The data collected during the geotechnical site investigation would form the basis for the feasibility design of the waste and water management facilities. The cost to complete the site investigations for the surface component of waste and water management is estimated to be $1,500,000.

26.4 Infrastructure Foundation Studies and Surface Geotechnical Drilling

Further geotechnical drillings for infrastructure foundation studies are recommended for the proposed plant site location. The estimated cost for this work is $1,000,000.

26.5 MINERAL PROCESSING AND METALLURGICAL TESTING

At an estimated cost of $2,000,000, the following additional laboratory studies are recommended:

- New samples from the deposit should be collected and submitted for DEXRT sorting.
  - The mineralized material from this new phase of DEXRT testing should be subjected to additional laboratory testing to optimize the size reduction methods as well as the magnetic separation process.
  - Froth flotation techniques will be re-examined.
- After the laboratory studies have optimized the methods for rejecting the barren waste from the minerals containing the rare-earth-elements of interest, the process should be conducted on a large enough scale to produce sufficient upgraded material for additional physical and chemical processing.
- A detailed program should be developed for a pilot plant that will produce sufficient quantities of pregnant liquor to prove the SPE process on a continuous basis of loading, stripping and re-loading of the various values, and to produce sufficient quantities of finished products for market evaluation and sales values.

At an estimated cost of $2,000,000, the following pilot plant studies are recommended:

- A large bulk sample of material should be obtained from the deposit that represents the material that will be mined in the early years of operations.
- The sample needs to be crushed and screened to the sized suitable for DEXRT processing in a commercial scale unit.
- A commercial laboratory will be selected to conduct duplicate tests of the physical beneficiation laboratory studies to confirm the new sample respond to the process previously developed, and will produce a final pregnant solution for confirmation of the hydrometallurgical separation process.
- The purified pregnant solutions will then be treated to separate the various values into separate products for market evaluations.

26.6 ENVIRONMENTAL

Existing environmental baseline characterization studies are ongoing. Significant technical studies and modelling are required to better understand both the hydrology and the geochemistry of the deposit and the associated process streams. Funding will also be required to support the permitting effort at both the state and federal levels. It is anticipated that further environmental studies and capacity funding will be required to successfully obtain the permits required to construct and operate the rare earth project at an approximate cost of $1,500,000.

26.7 Construction Schedule and Methods

Opportunities to reduce the construction timeline, such as early mobilization and modulization of the process plant, should be investigated during next phase of the study. The opportunity of optimizing the cash flow by expediting the construction schedule and building process plant modules offsite should be evaluated further as part of the next phase of study, with an estimated budget of $2,500,000.

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APPENDIX A

Certificates of Qualified Persons

Edwin H. Bentzen III, RM SME

I, Edwin H. Bentzen III, RM SME, of Lakewood, Colorado, do hereby certify:

- I am a Project Manager with Lyntek Incorporated with a business address at 1550 Dover Street, Lakewood, Colorado, 80215.
- This certificate applies to the technical report entitled "Preliminary Economic Assessment on the Bokan Mountain Rare Earth Element Project, near Ketchikan, Alaska", dated January 10, 2013 (the "Technical Report").
- I am a graduate of the Mackey School of Mines, University of Nevada, Reno, Nevada, (BSc. Metallurgical Engineering, 1967). I am a registered member in good standing of the Society of Mining, Metallurgy and Exploration, Inc. License #229580RM, the American Chemical Society, and the Canadian Institute of Mining. I have been practicing my profession as an engineer for over 45 years in the discipline of extractive mineral beneficiation. I have been employed in consulting engineering research and process development since graduation from the University of Nevada in 1967. I have worked on over 100 projects, including three recent rare earth mineral projects and have authored or co-authored over 25 professional papers/presentations at national and international meetings. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was June 14, 2012 for one day.
- I am responsible for Sections 1.6, 13.1, 17.3, 25.4, 26.5, and 27.0 (mineral processing and metallurgical testing section only) of the Technical Report.
- I am independent of Ucore Rare Metals Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 10th day of January, 2013 at Lakewood, Colorado

Original document signed and sealed by
Edwin H. Bentzen III, RM SME

Edwin H. Bentzen III, RM SME
Project Manager
Lyntek Incorporated

HASSAN GHAFFARI, P.ENG.

I, Hassan Ghaffari, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Manager of Metallurgy with Tetra Tech WEI Inc. with a business address at 800-555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled "Preliminary Economic Assessment on the Bokan Mountain Rare Earth Element Project, near Ketchikan, Alaska", dated January 10, 2013 (the "Technical Report").
- I am a graduate of the University of Tehran (M.A.Sc., Mining Engineering, 1998) and the University of British Columbia (M.A.Sc., Mineral Process Engineering, 2004). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #30408. My relevant experience with respect to mineral process engineering includes 22 years of experience in mining and plant operation, project studies, management, and engineering. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I did not complete a personal inspection of the Property.
- I am responsible for Sections 1.1, 1.8 (except 1.8.1), 1.10, 1.12, 1.13, 2.0, 3.0, 17.1, 17.2, 17.4, 18.1, 18.2, 18.5, 18.6, 19.0, 21.1.1, 21.1.3, 21.1.5, 21.2.1, 21.2.3, 21.2.4, 24.0, 26.4, and 26.7 of the Technical Report.
- I am independent of Ucore Rare Metals Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 10th day of January, 2013 at Vancouver, British Columbia

Original document signed and sealed by
Hassan Ghaffari, P.Eng.

Hassan Ghaffari, P.Eng.
Manager of Metallurgy
Tetra Tech WEI Inc.

LES GALBRAITH, P.ENG.

I, Les Galbraith, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Specialist Engineer/Project Manager with Knight Piésold Ltd. with a business address at 1400-750 West Pender Street, Vancouver, BC, V6C 2T8.
- This certificate applies to the technical report entitled “Preliminary Economic Assessment on the Bokan Mountain Rare Earth Element Project, near Ketchikan, Alaska”, dated January 10, 2013 (the “Technical Report”).
- I am a graduate of the University of British Columbia, (B.A.Sc., 1995). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #25493. My relevant experience is 17 years of experience in providing civil and geotechnical engineering support to mining and hydroelectric projects. I am a “Qualified Person” for the purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was April 26, 2012 for one day.
- I am responsible for Sections 1.8.1, 18.3, 18.4, 21.1.4, 25.3, 26.3, and 27.0 (waste and water management section only) of the Technical Report.
- I am independent of Ucore Rare Metals Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 10th day of January, 2013 at Vancouver, British Columbia

Original document signed and sealed by
Les Galbraith, P.Eng.

Les Galbraith, P.Eng.
Specialist Engineer/Project Manager
Knight Piésold Ltd.

Richard F. Hammen, Ph.D.

I, Richard F. Hammen, Ph.D., of Missoula, Montana, do hereby certify:

- I am the CEO with IntelliMet LLC with a business address at 4200 Fox Farm Road, Missoula, Montana, 59802.
- This certificate applies to the technical report entitled "Preliminary Economic Assessment on the Bokan Mountain Rare Earth Element Project, near Ketchikan, Alaska", dated January 10, 2013 (the "Technical Report").
- I am a graduate of Stanford University (Bachelor of Science, Chemistry,1966) and of the University of Wisconsin (Ph.D. Chemistry, 1973). I am a registered member in good standing of the Society of Mining, Metallurgy and Exploration, Inc. License # 4184323RM. I am also a member in good standing of the Canadian Institute of Mining. My relevant experience is the invention, patenting, development, and testing of Solid Phase Extraction (SPE) units used for Rare Earth Element and other metal separations. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was June 14, 2012 for one day.
- I am responsible for Sections 1.6, 13.2, 17.5, 17.6, 25.4, and 26.5 of the Technical Report.
- I am independent of Ucore Rare Metals Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 10th day of January, 2013 at Missoula, Montana

Original document signed and sealed by
Richard F. Hammen, Ph.D.

Richard F. Hammen, Ph.D.
CEO
IntelliMet LLC

RONALD JAMES ROBINSON, P.GEOL.

I, Ronald James Robinson, P.Geol., of Yellowknife, Northwest Territories, do hereby certify:

- I am a Senior Resource Geologist with Aurora Geosciences Ltd. with a business address at 3506 McDonald Drive, Yellowknife, Northwest Territories, X1A 2H1.
- This certificate applies to the technical report entitled "Preliminary Economic Assessment on the Bokan Mountain Rare Earth Element Project, near Ketchikan, Alaska", dated January 10, 2013 (the "Technical Report").
- I am a graduate of the University of British Columbia (B.Sc., 1985). I am a member in good standing of the Association of Professional Engineers and Geoscientists of Northwest Territories and Nunavut, License #1662. I have been employed in my profession by various mining and consulting companies since my graduation. I have produced and supervised the production of mineral resource estimates and mineral reserve documents on numerous deposits and deposit types for the past 20 years. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was between June 12 and October 25, 2011.
- I am responsible for Sections 1.2 to 1.5, 1.9, 4.0 to 12.0, 14.0, 20.0, 23.0, 25.1, 25.5, 26.1, 26.6, and 27.0 (environmental and geology sections only) of the Technical Report.
- I am independent of Ucore Rare Metals Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 10th day of January, 2013 at Yellowknife, Northwest Territories

Original document signed and sealed by
Ronald James Robinson, P.Geol.

Ronald James Robinson, P.Geol.
Senior Resource Geologist
Aurora Geosciences Ltd.

SABRY ABDEL HAFEZ, PH.D., P.ENG.

I, Sabry Abdel Hafez, Ph.D., P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Senior Mining Engineer with Tetra Tech WEI Inc. with a business address at 800-555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled "Preliminary Economic Assessment on the Bokan Mountain Rare Earth Element Project, near Ketchikan, Alaska", dated January 10, 2013 (the "Technical Report").
- I am a graduate of Assiut University, (B.Sc Mining Engineering, 1991; M.Sc. in Mining Engineering, 1996; Ph.D. in Mineral Economics, 2000). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #34975. My relevant experience is in mine evaluation. I have more than 19 years of experience in the evaluation of mining projects, advanced financial analysis, and mine planning and optimization. My capabilities range from the conventional mine planning and evaluation to the advanced simulation-based techniques that incorporate both market and geological uncertainties. I have been involved in the technical studies of several base metals, gold, coal, and aggregate mining projects in Canada and abroad. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I did not complete a personal inspection of the Property.
- I am responsible for Sections 1.11 and 22.0 of the Technical Report.
- I am independent of Ucore Rare Metals Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 10th day of January, 2013 at Vancouver, British Columbia

Original document signed and sealed by
Sabry Abdel Hafez, Ph.D., P.Eng.

Sabry Abdel Hafez, Ph.D., P.Eng.
Senior Mining Engineer
Tetra Tech WEI Inc.

Srikant Annavarapu, RM SME

I, Srikant Annavarapu, RM SME, of Mesa, Arizona, do hereby certify:

- I am a Principal Mining Engineer with AMEC E&C Services Inc. with a business address at 1640 S. Stapley Drive, Suite 241, Mesa, Arizona, 85204. I was formerly employed as an Engineering Consultant with Stantec Inc. for completing the mining work.
- This certificate applies to the technical report entitled “Preliminary Economic Assessment on the Bokan Mountain Rare Earth Element Project, near Ketchikan, Alaska”, dated January 10, 2013 (the “Technical Report”).
- I am a graduate of the Indian Institute of Technology, Kharagpur, India (B.Tech. degree in mining engineering, 1980) and the University of Arizona (M.S. degree in mining and geological engineering, 1998). I am a Professional Engineer in Arizona (#36554) and a registered member of the Society of Mining, Metallurgical and Exploration, Inc. I have practiced my profession for 28 years during which time I have been involved in the design of underground mining projects, including mine design, ground stabilization instrumentation, monitoring, and analysis, for various mines. I am a “Qualified Person” for the purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was November 29, 2011.
- I am responsible for Sections 1.7, 15.0, 16.0, 17.7, 21.1.2, 21.2.2, 25.2, and 26.2 of the Technical Report.
- I am independent of Ucore Rare Metals Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 10th day of January, 2013 at Mesa, Arizona

Original document signed and sealed by
Srikant Annavarapu, RM SME

Srikant Annavarapu, RM SME
Principal Mining Engineer
AMEC E&C Services Inc.