Preliminary Assessment Technical Report on the Horseshoe and Raven Deposits Hidden Bay Project

Saskatchewan, Canada

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Project No. 2CU005.000

Effective Date: February 15, 2011

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Summary

This Preliminary Assessment Technical Report (“PA”) was compiled by SRK Consulting (Canada) Inc. for UEX Corporation (“UEX”). The purpose of the Technical Report is to describe the results of a preliminary economic assessment conducted on the Horseshoe and Raven deposits of UEX’s Hidden Bay Project.

Kevin Palmer, P.Geo. of Golder Associates Ltd. (“Golder”) conducted the mineral resource estimate for the Horseshoe and Raven deposits. Lawrence Melis, P.Eng of Melis Engineering Ltd. provided metallurgical and mineral processing expertise. Several sections of this report utilize previous Hidden Bay technical reports for information and are referenced, updated and signed off by a current Qualified Person (“QP”).

The reader is advised that the preliminary assessment summarized in this technical report is only intended to provide an initial, high-level review of the project potential. The PA mine plan and economic model include the use of indicated and inferred. The inferred resources are considered to be too speculative to be used in an economic analysis except as allowed for in PA studies. There is no guarantee that inferred resources can be converted to indicated or measured resources and, as such, there is no guarantee that the project economics described herein will be achieved.

The Hidden Bay property is located in the Wollaston Lake area of northern Saskatchewan, Canada, approximately 740 km north of the city of Saskatoon, immediately west of Wollaston Lake. The Hidden Bay property consists of 57,321 hectares (573 km^2) in 43 mineral dispositions. All of these mineral dispositions are owned 100% by UEX Corporation (“UEX”) except for 297 hectares (“ha”) in disposition ML 5424, which is currently owned 76.729% by UEX, 8.525% by ENUSA Industrias Avanzadas, 7.680% by Nordostchweizerische Kraftwerke AG, and 7.066% by Encana. Disposition ML5424 is in the southernmost portions of the Hidden Bay property, near the West Bear deposit, and does not contain any current or historical resources.

The Hidden Bay property is in the eastern Athabasca uranium district, adjacent to, and surrounding several current and past producing uranium deposits on the Rabbit Lake property of Cameco Corporation (“Cameco”), and the McClean Lake property, operated by AREVA Resources Canada Inc. (“AREVA”). The property is accessible year round by Highway 905, a maintained all-weather gravel road, and by maintained access and mine roads to the Rabbit Lake and McClean Lake mining operations, which pass through the property. Infrastructure is well developed in the local area, with two operating uranium ore processing facilities, Rabbit Lake and McClean Lake, located 4 km northeast and 22 km northwest of the Horseshoe and Raven deposits, respectively. The principal hydroelectric transmission lines that service both of these facilities also pass through the property, 3 km to the north of the Horseshoe and Raven deposits.

This technical report has been completed in conformance with the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines referred to in Companion Policy 43-101CP to National Instrument ("NI") 43-101.

Geological Setting

The Hidden Bay property is at the eastern margin of the Athabasca Basin. The property is underlain by flat-lying to shallow dipping Late Proterozoic sandstone of the Athabasca Group to the northwest, which unconformably overlies metamorphosed clastic and chemical meta-sedimentary basement rocks and granitic intrusions of the trans-Hudson orogen, exposed to the east. The property straddles the gradational contact between the Mudjatik Domain of the trans-Hudson orogen to the northwest, composed of granitic gneiss domes and intervening psammitic to pelitic gneiss, and the Wollaston Domain to the southeast. The latter is composed of a basal pelitic gneiss unit that is overlain successively by meta-arkose and a lithologically diverse upper sequence of quartzite with interlayered amphibolite and calcareous meta-arkose termed the Hidden Bay Assemblage. At least two major contractional deformation events and overlapping periods of amphibolite to granulite grade metamorphism are evident in basement rocks in the area and form the main pulses of the 1,820-1,770 Ma Hudsonian orogeny. These events produced two northeast-trending sets of folds with predominantly southeast dipping axial planes, and associated axial planar foliations.

Major faults in the region include northeast-trending reverse faults and north-trending Tabbernor-type sinistral faults, both of which control the distribution of uranium deposits in the district. Northeast-trending faults dip southeast, are generally concordant, and are frequently localized in graphitic gneiss. The dominant structure of this type is the Rabbit Lake Fault, which crosses central parts of the property and has been traced by drilling for over 40 km. Other significant faults in the area include the Collins Bay Fault system, associated with the Collins Bay and Eagle Point deposits on the Rabbit Lake property, and the Telephone Lake and Tent-Seal Faults. These faults are post-metamorphic semi-brittle to brittle shear zones defined by lithified graphite-rich cleaved zones, graphite-matrix breccia, and seams of graphitic or chloritic clay gouge.

Uranium Deposits on the Hidden Bay Property

Uranium deposits and prospects on the Hidden Bay property are of the unconformity type. Three deposits for which National Instrument ("N.I.") 43-101 resources have been estimated occur on the Hidden Bay property: Horseshoe, Raven and West Bear. The Horseshoe and Raven deposits are located in north central portions of the Hidden Bay property. Mineralization at the Horseshoe and Raven deposits comprises shallow dipping zones of hematization with disseminated and veinlet ----- pitchblende-boltwoodite-uranophane that is hosted by folded arkosic quartzite gneiss of the Hidden Bay Assemblage. Mineralization comprises a combination of disseminated pitchblende-chlorite-hematite, and narrower, higher grade nodular and veinlet pitchblende in hematite-clay alteration.

Mineralization occurs in hematitic redox fronts surrounding large, semi-tabular clay alteration zones that are cored by probable faults. Mineralization at the Horseshoe deposit has been defined continuously over a strike length of approximately 800 m and a dip length of up to 300 m, occurring at depths of 100 m to 450 m below surface. At Raven, which lies 0.5 km southwest of Horseshoe, mineralization has been defined over a strike length to date of approximately 910 m at depths below surface of 100 m to 300 m in two dominant, sub-horizontal zones. The deposits are located approximately 5 km south of Cameco's Rabbit Lake operations, and 12 km southeast of AREVA's McClean Lake operations. Both are hosted by competent basement rocks that could be amenable to both open-pit and conventional underground ramp access mining methods. Similar to other basement-hosted deposits in the region, Horseshoe and Raven mineralization comprises pitchblende and other uranium oxides and silicates without potentially deleterious nickel-arsenide minerals that may affect extraction and pose tailings disposal problems.

The West Bear deposit, located in southernmost parts of the Hidden Bay property, is a classic unconformity-hosted uranium deposit which is developed under shallow Athabasca sandstone cover above a conductive graphitic gneiss unit in southern parts of the Hidden Bay property.

West Bear is flat-lying and has been defined by drilling over a strike length of 500 m, in a long, cigar-shaped mineralized zone straddling the unconformity. The mineralization occurs at a vertical depth of between 13 m and 31 m from surface and is one of the shallowest, undeveloped uranium deposits in the prolific Athabasca Basin. The deposit ranges in width from 5 m to 25 m, and in vertical thickness from 0.1 m to more than 10 m. Mineralization occurs in intense clay-hematite alteration where a minor fault system hosted by the underlying graphitic conductor intersects the unconformity. Mineralization comprises sooty to nodular, and locally massive, pitchblende mineralization in clay with associated Ni-Co-As mineralization. This is typical of the style and geochemistry of other unconformity-hosted uranium deposits in the region, including the McClean Lake deposits and Cigar Lake.

In addition to these deposits, a series of prospective exploration targets are also present on the property that include basement-hosted and unconformity-style targets, some of which lie along conductors or fault systems which host uranium deposits on the adjacent McClean Lake and Rabbit Lake properties.

Drilling and Exploration by UEX Corporation

After acquiring the Hidden Bay property in 2002, UEX continued to explore various targets on the Hidden Bay property, utilizing a combination of airborne and ground electromagnetic, magnetic, radiometric resistivity and gravity geophysical methods in more grassroots target areas to identify drilling targets, or direct follow-up drilling in areas where previous drilling had intersected alteration or mineralization.

UEX also initiated a re-evaluation of the Horseshoe and Raven deposits due to rising uranium prices. In 2005, drilling tested mineralization in selected areas of both deposits to test mineralization continuity between the widely spaced historical holes drilled by Gulf Minerals Canada Limited ("Gulf"). The success of that program led to subsequent drilling programs between 2006 and 2009 in which 376 diamond drill holes totalling 119,400 m were drilled at Horseshoe and 243 drill holes totalling 65,600 m were drilled at Raven. These programs not only established continuity of mineralization between the historical Gulf drilling, but expanded the deposit footprints into areas not historically drilled by Gulf for which this drilling forms the basis are reported here.

Metallurgy and Mineral Processing

Metallurgical testing for UEX Corporation's Hidden Bay Project included testwork on both the West Bear deposit and the Horseshoe-Raven deposits. Testwork, completed at SGS Canada Inc.'s Lakefield Research facility in Lakefield, Ontario (SGS Lakefield) under the direction of Melis Engineering Ltd. ("Melis"), started in 2006 on preliminary samples of the West Bear mineralization and was completed in 2009 as a second phase of work on Horseshoe-Raven mineralization. This report focuses on the Horseshoe and Raven deposits.

Based on supporting metallurgical testwork, process recoveries are estimated to be 95%.

Horseshoe-Raven test composites were prepared from assay rejects and from purpose-drilled HQ core. The elemental analyses of the composites showed that the Horseshoe and Raven uranium deposits are relatively low in deleterious elements such as arsenic, molybdenum, selenium, and base metals. Five uranium carriers were identified, uraninite, boltwoodite, uranophane, coffinite and minor amounts of carnotite.

The Horseshoe-Raven composites were categorized as medium in hardness from the perspective of SAG milling, with an average SPI value of 69 minutes. The ball mill Bond Work Indices were all within a tight range of 16.1 to 17.7 kWh/t with an average value of 16.7 kWh/t, showing very little variation across the deposits and characterizing the Horseshoe-Raven mineralization as moderately hard for ball mill grinding.

Leach test results confirmed the Horseshoe-Raven mineralization is easily leached under relatively mild atmospheric leach conditions. Leach extractions of 98% or greater can be achieved for Horseshoe and Raven mineralization under atmospheric leach conditions using a mesh-of-grind K_{80} (80% passing size) of approximately 145 µm, a leach temperature of 50ºC, a free acid concentration of 10 g H_2SO_4/L, representing an acid consumption of 45 kg H_2SO_4/t, an ORP of 500 mV, representing a sodium chlorate consumption of 0.6 kg $NaClO_3$/t, and a leach retention time of 8 to 12 hours. An overall uranium recovery of 95% was used in this study for all the cash flow analysis. Mine optimization work used 96% uranium extraction, prior to finalization of the recovery estimate.

The pregnant leach solution and residue from a Horseshoe bulk leach test were retained to generate waste raffinate and leach residue for waste treatment testing. The specific gravity of the generated tailings was measured at 2.59 t/m^3. The tailings K_{80} was 136 µm and the K_{50} (50% passing size) was 54 µm.

Tailings supernatant aging tests resulted in elevated levels of radium and molybdenum in the supernatant. This was expected, and confirms that, like all uranium tailings supernatant, excess tailings water would be re-used and/or treated in the mill process and waste treatment circuits under normal operating conditions.

The concentrations of uranium (0.015 mg/L), arsenic (0.0067 mg/L), molybdenum (0.0115 mg/L), radium 226 (0.02 Bq/L) and selenium (0.009 mg/L) obtained in treated effluent are below typical regulatory limits set by the provincial and federal governments.

This report assumes that run of mine (“ROM”) material will be trucked to the Rabbit Lake processing facility for treatment. It is assumed that a toll treatment agreement could be reached with Cameco, the owner of the Rabbit Lake plant, which would allow Hidden Bay mineralization to be processed at an average rate of 1,000 tpd. It is also assumed that the Rabbit Lake facility would provide toll tailings deposition for the Hidden Bay ROM material.

West Bear Mineral Resource Estimate

The January 2009 West Bear Resource Estimate was also prepared by K. Palmer, P.Geo., of Golder and the methodology is reported in the Technical report dated February 17, 2009 by Palmer and Fielder. The resource calculation utilized the results from 216 drill holes totalling 6,400 m, which were completed during 2004, 2005 and 2007 sonic drilling programs. The resource estimate was calculated using a minimum cut-off grade of 0.01% U_3O_8 utilizing a geostatistical-block model technique with ordinary kriging methods and Datamine.

The resource reported below reflects the remodelling of the deposit after re-sampling of drill core was undertaken to better define mineralization outlines. The changes in volume, with corresponding decrease in grade with respect to the December 2007 Indicated Mineral Resource, reflect incorporation of lower grade material in the new resource outlines. All the current mineral resources at West Bear are classified as Indicated. Details at different cut-off levels are provided in Table 1.

Table 1: January 2009 Indicated Mineral Resources (Capped) at the West Bear Deposit with Tonnes and Grade at Various U_3O_8 Cut-off Grades

			Grade				Contained Metal			
Cut-off Grade (%U_3O_8)	Tonnes	Density (g/cm^3)	U_3O_8 (%)	Ni (%)	Co (%)	As (%)	U_3O_8 (lbs)	Ni (lbs)	Co (lbs)	As (lbs)
0.01	209,700	1.99	0.358	0.22	0.08	0.22	1,655,000	1,030,000	375,000	1,005,000
0.02	188,100	1.99	0.397	0.24	0.09	0.23	1,646,000	975,000	355,000	974,000
0.03	113,000	2.02	0.645	0.28	0.10	0.32	1,605,000	704,000	254,000	786,000
0.04	85,300	2.03	0.843	0.32	0.11	0.37	1,585,000	600,000	203,000	694,000
0.05	**78,900**	**2.04**	**0.908**	**0.33**	**0.11**	**0.38**	**1,579,000**	**569,000**	**185,000**	**662,000**
0.10	76,100	2.04	0.939	0.33	0.10	0.38	1,574,000	547,000	173,000	640,000
0.15	70,300	2.04	1.005	0.33	0.11	0.39	1,558,000	505,000	165,000	604,000
0.20	63,800	2.04	1.09	0.32	0.11	0.40	1,532,000	453,000	152,000	559,000
0.25	57,300	2.04	1.187	0.31	0.11	0.41	1,500,000	397,000	138,000	514,000
0.30	52,100	2.04	1.279	0.31	0.11	0.42	1,468,000	360,000	127,000	482,000
0.35	47,800	2.04	1.365	0.30	0.11	0.42	1,437,000	319,000	115,000	443,000
0.40	43,600	2.05	1.461	0.31	0.11	0.44	1,403,000	295,000	107,000	418,000

Horseshoe Mineral Resource Estimate

The July 2009 Horseshoe Mineral Resource Estimate was prepared by Kevin Palmer, P.Geo., of Golder and is an update of the September 2008 estimate. The mineral resource estimate was peer reviewed by David Farrow, Pr.Sci.Nat., also of Golder and is summarized in Table 2. The methodology is reported in the Technical report dated September 4, 2009 by Palmer and Fielder.

The mineral resource calculation utilized 376 diamond drill holes (119,400 m from holes HU-001 to HU-358, HS-001 and HO-01 to HO-16) drilled between 2005 and 2009, which test the deposit at 7.5 m to 30 m drill centres. The updated resource comprises 5.120 million tonnes ("Mt") grading 0.203% U_3O_8 in the Indicated category, containing 22.895 Mt of U_3O_8 and 0.287 Mt grading 0.166% U_3O_8 in the Inferred category, containing 1.049 million pounds ("Mlb") of U_3O_8 at a cut-off of 0.05% U_3O_8. The mineral resource estimate was calculated using a minimum cut-off grade of 0.02% U_3O_8 utilizing a geostatistical block-model technique with ordinary kriging methods and the Datamine Studio 3 ("Datamine") software package. Over 95% of the resource is in the Indicated category at a 0.05% U_3O_8 cut-off. At a cut-off of 0.20% U_3O_8, the average grade for the Indicated mineralization is 0.412% U_3O_8 with a tonnage of 1.567 Mt. This may be significant should an economic evaluation recommend an underground mining method for the deposit.

Table 2: July 2009 Indicated and Inferred Mineral Resources (Capped) at the Horseshoe Deposit with Tonnes and Grade at Various U_3O_8 Cut-off Grades

Resource Category	Cut-off Grade (% U_3O_8)	Tonnes	In Situ Grade (% U_3O_8)	Contained Metal (lb U_3O_8)
Indicated	0.02	7,042,400	0.157	24,427,000
	0.05	**5,119,700**	**0.203**	**22,895,000**
	0.10	3,464,800	0.266	20,302,000
	0.15	2,380,800	0.33	17,331,000
	0.20	1,567,000	0.412	14,219,000
	0.25	1,059,900	0.502	11,726,000
	0.30	722,600	0.609	9,696,000
	0.35	529,100	0.713	8,319,000
	0.40	414,600	0.807	7,377,000
Inferred	0.02	444,900	0.122	1,192,000
	0.05	**287,000**	**0.166**	**1,049,000**
	0.10	159,700	0.239	840,000
	0.15	106,800	0.298	702,000
	0.20	79,800	0.34	598,000
	0.25	53,500	0.398	469,000
	0.30	29,300	0.502	324,000
	0.35	15,500	0.665	227,000
	0.40	11,400	0.769	193,000

Raven Mineral Resource Estimate

The July 2009 Raven Mineral Resource Estimate was prepared by Kevin Palmer, P.Geo., of Golder and is an update of the January 2009 estimate. The mineral resource estimate was peer reviewed by David Farrow, Pr.Sci.Nat., also of Golder and is summarized in Table 3. The methodology is reported in the Technical report dated September 4, 2009 by Palmer and Fielder. The mineral resource estimate was based on 243 diamond drill holes (approximately 65,600 m from holes RU-001 to RU-216, and RV-001 to RV-028) drilled between 2005 and 2009, with an approximate drill spacing of 7.5 m to 30 m. The mineral resource was estimated based on a geological model created by UEX which contained 16 mineralized subzones. The geological model was based on clay alteration and a grade cut-off of 0.02% U_3O_8. A 3D block model was created from the geological model which then had grades interpolated into them using the ordinary kriging estimation method. The software that was used to complete the mineral resource estimate was Datamine. During the mineral resource estimate, high grade assay outliers were identified for each subzone and capped accordingly to prevent high grade spreading.

The July 2009 Raven Mineral Resource Estimate contains 5.174 Mt grading 0.107% U_3O_8 in the Indicated category, containing 12.149 Mlb of U_3O_8 and 0.822 Mt grading 0.092% U_3O_8 in the Inferred category, containing 1.666 Mlb of U_3O_8 at a cut-off of 0.05% U_3O_8. At a 0.05% U_3O_8 cut-off, 88% of the tonnes are in the Indicated category.

Details of the July 2009 Raven Mineral Resource Estimate at different cut-off levels are provided in Table 3.

Table 3: July 2009 Indicated and Inferred Mineral Resources (Capped) at the Raven Deposit with Tonnes and Grade at Various U_3O_8 Cut-off Grades

Resource Category	Cut-off Grade (% U_3O_8)	Tonnes	In Situ Grade (% U_3O_8)	Contained Metal (lb U_3O_8)
Indicated	0.02	9,646,100	0.073	15,544,000
	0.05	**5,173,900**	**0.107**	**12,149,000**
	0.10	1,893,400	0.17	7,113,000
	0.15	827,700	0.234	4,274,000
	0.20	424,000	0.294	2,752,000
	0.25	241,500	0.349	1,859,000
	0.30	139,100	0.406	1,244,000
	0.35	80,300	0.467	827,000
	0.40	48,400	0.529	565,000
Inferred	0.02	1,537,600	0.067	2,278,000
	0.05	**822,200**	**0.092**	**1,666,000**
	0.10	176,000	0.186	723,000
	0.15	96,000	0.239	506,000
	0.20	48,500	0.302	323,000
	0.25	25,700	0.37	209,000
	0.30	15,800	0.431	150,000
	0.35	11,700	0.468	121,000
	0.40	8,200	0.509	92,000

Hidden Bay Project – Total Resources

The combined N.I. 43-101 compliant resources for the July 2009 Horseshoe and Raven and the January 2009 N.I. 43-101 compliant resource at the West Bear deposit on the Hidden Bay Project at a cut-off of 0.05% U_3O_8 totals 10.373 Mt and contains 36.623 Mlb U_3O_8 in Indicated Mineral Resource category and 1.109 Mt containing 2.715 Mlb U_3O_8 Inferred Mineral Resource category. A summary of resources at various cut-offs is illustrated in Table 4. It must be noted that the mining of the West Bear deposit is not included in this PA.

Table 4: Total N.I. 43-101 Compliant Indicated and Inferred Mineral Resources (Capped) on the Hidden Bay Project, as of July 2009 at Various Cut-off Grades of % U_3O_8

Resource Category	Cut-off Grade (% U_3O_8)	Tonnes	In Situ Grade (% U_3O_8)	Contained Metal (lb U_3O_8)
Indicated	0.02	16,876,600	0.112	41,617,000
	0.05	**10,372,500**	**0.160**	**36,623,000**
	0.10	5,434,300	0.242	28,989,000
	0.15	3,278,800	0.321	23,163,000
	0.20	2,054,800	0.409	18,503,000
	0.25	1,358,700	0.504	15,085,000
	0.30	913,800	0.616	12,408,000
	0.35	657,200	0.731	10,583,000
	0.40	506,600	0.837	9,345,000
Inferred	0.02	1,982,500	0.079	3,470,000
	0.05	**1,109,200**	**0.111**	**2,715,000**
	0.10	335,700	0.211	1,563,000
	0.15	202,800	0.270	1,208,000
	0.20	128,300	0.326	921,000
	0.25	79,200	0.388	678,000
	0.30	45,100	0.477	474,000
	0.35	27,200	0.580	348,000
	0.4	19600	0.660	285,000

Mine Plan

The Hidden Bay deposits of Horseshoe and Raven are proposed to be developed both as an open pit (“OP”) and underground methods (“UG”). Mining of the Horseshoe and Raven deposits is proposed to produce a total of 2.49 Mt of mill feed and 15.0 Mt of waste over a 7-year mine operating life. Approximately 2.10 Mt of mill feed is planned to be produced from UG mining of the Horseshoe deposit, with 0.39 Mt being produced from OP mining of the Raven deposit. The mill feed is planned to be trucked to Cameco’s Rabbit Lake Facility for processing.

Mine design for the Horseshoe and Raven deposits was initiated with the development of Whittle™ input parameters and UG cut-off grades. These parameters included estimates of metal price (US$60/lb U_3O_8), exchange rate, toll milling and mining costs, mining dilution, mill recovery, and royalties. The resource models for Horseshoe and Raven (as provided by Golder) were based on a 5 m x 5 m x 2.5 m block size. Table 5 summarizes the various input parameters for Whittle™ optimization.

Table 5: Whittle™ Optimization Input Parameters*

Item	Unit	Value 2011
Bulk Density		
Ore	t/m^3	varies in model
Waste	t/m^3	2.48
Overburden	t/m^3	N/A
Metal Prices		
U_3O_8	$US/lb	$60.00
U_3O_8	C$/lb	$63.16
Process Recovery		
U_3O_8	%	96
Site Operating Costs		
Toll milling (includes ore haul cost to mill)	C$/t ore	$70.00
G&A/Sustaining Capital	C$/t ore	$5.00
Incr. Mining Cost	C$/t ore	N/A
Tailings Management Facility	C$/t ore	$35.00
On Site Costs	C$/t ore	**$110.00**
Mining Costs		
Open Pit Ore mining	C$/t mined	$2.70
Open Pit Waste mining - rock	C$/t mined	$2.70
Open Pit Waste mining - overburden	C$/t mined	N/A
Underground mining cost	C$/t mined	N/A
TC/RC		
Refining/Freight/Insurance/ Marketing	C$/lb	N/A
Pit Parameters		
Pit slope angles with ramps		
Overburden	overall °	N/A
Basement Rock	overall °	45
Bench height	m	10
Mining Recovery	%	100
Dilution (@ 0%U_3O_8 grade)	%	10
Production capacity	ore t/yr	1,095,000
Economics		
Exchange rate	C$:US$	1.05
Royalties (% of gross U_3O_8 sales)	%	5.0
Discount Rate	%	10.0
Operating Parameters		
Operating Days	days/yr	365
Shift Schedule	shifts/day	2
Scheduled Shifts	shifts/year	730
Operating Crews	#	4
Energy Cost		
Diesel Fuel Cost	C$/litre	1.00
Electric Power Cost	C$/kWh	0.10

**These parameters were the initial assumptions made to begin the mine planning process. Some of the parameters changed as more detailed work was conducted. For example, the process recovery of U_3O_8 of 96% was used in the optimization and then modified to 95% for the economic analysis as the recovery was finalized by the QP. The processing costs also changed from this preliminary estimate ($70/tonne), done at an assumed head grade of 0.15% U_3O_8, to the final costs estimated using the ROM grade of 0.30% U_3O_8 ($79.20/tonne).*

For the OP at Raven, the model was then used with the Gemcom Whittle - Strategic Mine Planning™ ("Whittle") software to determine the optimal mining shell. Mine planning and scheduling was then conducted on the optimal pit shell with the use of MineSight™ software.

UG mine planning used the input parameters as shown in Table 6 to provide initial mineable shapes.

Table 6: Underground Preliminary Planning Parameters

Item	Unit	Value
Metal Recovery		
U_3O_8 Price	$US/lb U_3O_8	60
Exchange Rate	$C/$US	1.05
U_3O_8 Price	$C/lb U_3O_8	63.16
Payable Metal	% U_3O_8	100
Process Recovery	%	96*
Refining/Freight/Insurance/ Marketing	$C/lb U_3O_8	N/A
Royalties @ 5% NSR	$C/lb U_3O_8	3.03
Net U_3O_8 price	$C/lb U_3O_8	57.60
Opex Estimates		
Mining Cost	$ /t milled	68.0
Toll Processing Cost (including hauling to mill)	$ /t milled	70.0**
G&A/Sustaining capital cost	$ /t milled	5.0
TMF	$ /t milled	35.0
Total Site Cost	$ /t milled	178.0
Cut-off Grade		
Plant feed Cut-off Grade	% U_3O_8	0.14
Dilution	%	10
In-situ Cut-off Grade	% U_3O_8	0.16

**Changed to 95% in the final economic analysis*
***Changed to an average of $79.20/t, processing only, in the final economic analysis*

The estimated mineable mineral resources for both OP and UG are summarized in Table 7 below. The estimated U_3O_8 cut-off grades used are also noted.

Table 7: Hidden Bay - LOM Resource

Deposit	Resource Category	Tonnes (Mt)	Cut-off Grade (U_3O_8%)	Diluted Grade (U_3O_8%)	Contained Metal (Mlb U_3O_8)
Raven	Indicated	0.4	0.10	0.19	1.7
	Inferred	0.0	0.10	0.24	0.0
Horseshoe	Indicated	2.0	0.16	0.32	14.4
	Inferred	0.1	0.16	0.28	0.5
Total	Indicated	2.4	0.15	0.30	16.1
	Inferred	0.1	0.16	0.28	0.5

The current life-of-mine (“LOM”) plan focuses on accessing and milling higher grade material first. As such, the plan commences with UG mining of Horseshoe, followed by the OP at Raven. The maximum total mill feed production from both OP and UG is targeted at 1,000 tpd. Given the relatively small pit size, the maximum daily mined tonnage is targeted at 30,000 t/day total material. The LOM mine production schedule is shown in Table 8.

Table 8: LOM Mine Production Schedule – Horseshoe and Raven Deposits

			YEAR						
Parameter	**Unit**	**Total**	**1**	**2**	**3**	**4**	**5**	**6**	**7**
OPEN PIT MINING - Raven									
O/P total Waste	Mt	15.01	-	-	-	-	-	11.54	3.48
O/P ROM	Mt	0.39	-	-	-	-	-	0.00	0.39
U_3O_8 Grade	U_3O_8 %	0.19	-	-	-	-	-	0.26	0.19
Total ROM mined O/P	Mt	0.39	-	-	-	-	-	0.00	0.39
O/P total Mined	Mlb U_3O_8	1.7						0.0	1.6
O/P Strip Ratio	t:t	38.2						3,958	8.9
UNDERGROUND MINING - Horseshoe									
Development Waste	Mt	0.00							
Horseshoe ROM	Mt	2.10	0.350	0.35	0.35	0.35	0.35	0.35	
U_3O_8 ROM Grade	U_3O_8 %	0.32	0.54	0.39	0.30	0.23	0.23	0.24	
Total Mined lb	Mlb U_3O_8	14.9	4.2	3.0	2.3	1.8	1.8	1.8	
TOTAL ALL DEPOSITS									
Total Waste	Mt	15.01	-	-	-	-	-	11.54	3.48
Total ROM mined	Mt	2.49	0.35	0.35	0.35	0.35	0.35	0.35	0.39
Total Mined grade	U_3O_8 %	0.30	0.54	0.39	0.30	0.23	0.23	0.24	0.19
Total Mined lbs	Mlb U_3O_8	16.6	4.17	3.0	2.3	1.8	1.8	1.8	1.6

Waste Management

Waste rock from the Raven pit is proposed to be deposited in an engineered dump adjacent to the pit. Due to the pit and deposit geometry, the existing road to the Rabbit Lake Facility will require re-routing. A total of 15.0 Mt (or 7.9 Mm^3) of waste will be generated from the Raven pit. It was assumed that 25% of the waste dump would be underlain with a liner to manage potential geochemistry issues. Further testing is required to determine the geochemical characteristics of the waste rock and requirement for a lined facility.

All mill feed is assumed to be processed and all tailings deposited at the Rabbit Lake Facility. No tailings management facility has been considered for this PA. It should be noted that the mined-out Raven pit may make a suitable tailings deposition site for the Rabbit Lake plant. This opportunity has not been factored into the economics of this study but may represent an economic opportunity to UEX in the form of toll tailings storage if the production schedule is modified to mine the open pit first.

Capital and Operating Cost Estimates

Capital ("CAPEX") and operating ("OPEX") cost estimates were based on late-2010 prices and are a combination of first principle calculations, factored costs for similar projects, vendor quotes and estimates based on experience.

It was assumed that open pit mining, due to the small size and short life of the Raven pit when using a metal price of US$60/lb U_3O_8 for mine design would be conducted by a mining contractor. UG mining would be done with an owner-operated fleet. Mineral processing was calculated with a 25% toll treatment mark-up over a base processing cost estimate. A capital cost estimate for an upgrade of the Rabbit Lake plant was conducted to ensure the plant could handle 3,000 tpd comprised of 1,000 tpd from Hidden Bay and 2,000 tpd from other sources. Tables 9 and 10 show a summary of the cost estimates.

Table 9: Unit OPEX Estimate Summary

Operating Factors	Unit (C$)	Unit OPEX Estimate
UG Mining Cost	$/t milled	67.75
OP Mining Cost	$/t mined	2.70
OP Mining Cost	$/t milled	106.68
Combined Mining Cost	$/t milled	73.85
Toll Treatment Cost	$/t milled	79.20
G&A (inc. trucking costs)	$/t milled	11.00
Water Treatment	$/t milled	1.83
Tailings Management	$/t milled	35.00
Average Unit operating Cost	**$/t milled**	**200.88**

Table 10: Capital Cost Estimate Summary

Item	Unit (C$)	Total	Pre-production	Sustaining
Underground Mine	M$	**45.2**	32.4	12.8
Open Pit	M$	**0.2**	0.0	0.2
Rabbit Lake Mill Upgrades	M$	**12.3**	12.3	0.0
Site and Facilities	M$	**18.9**	18.9	0.0
Owner's Costs	M$	**22.0**	22.0	0.0
Closure	M$	**10.0**	0.0	10.0
EPCM (12%)	M$	**6.9**	6.9	0.0
Contingency (25%)	M$	**28.9**	23.1	5.8
Total Capital Cost	**M$**	**144.5**	**115.7**	**28.8**

Economic Analysis

The economic analysis for the project was done using earnings before interest and taxes ("EBIT"). Three cases were run to provide a range of U_3O_8 prices and their affect on the economic results. Case A used a US$60/lb U_3O_8 price to represent potential long-term pricing, Case B used the current spot price of US$70/lb and Case C used a US$80/lb U_3O_8 price. The EBIT analysis shows that the project is very robust for all cases as summarized in Table 11. The break-even U_3O_8 price is US$44/lb.

Table 11: Economic Analysis Results

Parameter	Unit	Case A	Case B	Case C
U_3O_8 Price	US$/lb U_3O_8	60	70	80
Royalty Payments (@10%)	M$	99	115	132
EBIT $NPV_{0\%}$	M$	246	394	542
EBIT $NPV_{5\%}$	M$	163	267	371
EBIT IRR	%	42	55	66
EBIT payback period	Production years	1	1	1

Conclusions

Industry standard mining, process design, construction methods and economic evaluation practices have been used to assess the Horseshoe and Raven deposits. There is adequate geological and other pertinent data available to generate a PA.

Based on current knowledge and assumptions, the results of this study show that the project is economic and should be advanced to the next level of study by conducting the work indicated in the Recommendations section.

Risks

While there are many risks associated with most early-stage mining projects, many of those risks can be mitigated with appropriate information gathering and engineering work. The project does not appear to have any fatal flaws. The main risks associated with the Horseshoe and Raven project are, in summary:

- Geological Interpretation;
- Mineral Resource Classification;
- U_3O_8 price and exchange rate;
- The ability to secure environmental permits;
- The ability to secure an appropriate toll treatment and tailings deposition agreement with a local processing plant;
- The ability to achieve operating and capital cost estimates; and
- The ability to meet dilution and extraction expectations.

Opportunities

The project has many opportunities for improvement, as detailed in Section 23.4, including:

- Expansion of mineable tonnes due to an increase in U_3O_8 price or a reduction in operating costs;;
- Expansion through the discovery of additional resources;
- Increased U_3O_8 price or a stronger American dollar vs. the Canadian dollar;
- Synergies with established local producers to improve costs and efficiencies for all participants;
- The potential use of the Raven pit as a regional toll tailings management site; and
- The inclusion of the West Bear deposit in the overall project mine plan and economics.

Recommendations

There are risks associated with the geological interpretation and mineral resource classification. These should be reviewed prior to preliminary feasibility study ("PFS") being carried out. It is recommended that the project be advanced to a PFS level that includes the West Bear, Horseshoe and Raven deposits. The PFS study would be supported by additional field work and information gathering for geotechnical, environmental, metallurgical and hydrogeological studies. It is also recommended that the project description be compiled and submitted to the government for review and advisement of specific guideline requirements. It is anticipated that the PFS study and associated information gathering will cost $1.0M to 1.5M. Further recommendations details can be found in the Recommendations section of this report.

It is also recommended that additional exploration drilling be conducted to test further geological and geophysical targets in the vicinity of the Horseshoe and Raven deposits as well as targets in other areas of the Hidden Bay property.

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1 Introduction

This Preliminary Assessment Technical Report was compiled by SRK Consulting (Canada) Inc. for UEX Corporation (“UEX”). The purpose of the Technical Report is to describe the results of a preliminary assessment (“PA”) conducted on Horseshoe and Raven deposits of UEX’s Hidden Bay Project.

Several sections of this report utilize previous Hidden Bay technical reports for information and are referenced and signed off by a current Qualified Person (“QP”).

The reader is advised that the preliminary assessment summarized in this technical report is only intended to provide an initial, high-level review of the project potential. The PA mine plan and economic model include the use indicated and inferred resources . The inferred resources are considered to be too speculative to be used in an economic analysis except as allowed for in PA studies. There is no guarantee that inferred resources can be converted to indicated or measured resources and, as such, there is no guarantee that the project economics described herein will be achieved.

The QPs responsible for this report are listed in Table 1.1 along with their responsibilities and site visit dates and descriptions. Each QP in this report takes sole responsibility for their work as outlined in their QP Certificates.

All units in this report are based on the International System of Units (“SI”), except industry standard units, such as troy ounces for the mass of precious metals. All currency values are Canadian Dollars (“C$” or “$”) unless otherwise stated.

This report uses abbreviations and acronyms common within the minerals industry. Explanations are located in Section 26.

Table 1.1: Qualified Persons and Site Visit Information

<table>
<tr><th>Qualified Person</th><th>Responsibility</th><th>Site Visit Date</th><th>Scope of Site Visit</th></tr>
<tr><td>Dino Pilotto, P.Eng.
SRK</td><td>OP Mining, Infrastructure and Waste Management
Report Sections: 16.6,18.7,18.10,21.1.2 and 21.2.2</td><td rowspan="4">Aug 11 to 12, 2010</td><td rowspan="4">- Review drill core
- General site layout and conditions
- Tour of Eagle Point UG mine
- Tour of Rabbit Lake processing facilities
- Tour of McClean Lake Processing facilities</td></tr>
<tr><td>Bruce Murphy, FSAIMM
SRK</td><td>Geotechnical Considerations
Report Section: 18.1</td></tr>
<tr><td>Lawrence Melis, P.Eng.
Melis Engineering</td><td>Metallurgy and Mineral Processing
Sections: 15, 21.1.3 and 21.2.3</td></tr>
<tr><td>Gordon Doerksen P.Eng.
SRK</td><td>UG Mining, Economics, Project Management
Report Sections: Summary, 1,2, 16.7, 18.2 to 18.6, 18.8, 18.9, 19, 21.1.1, 21.1.4, 21.2.1, 21.2.4, 21.2.5, 21.2.6, 22, 23, 24.3, 25, 26, 27 and 28</td></tr>
<tr><td>Kevin Palmer, P.Geo.
Golder</td><td>Mineral Resource Estimates
Report Sections: 3 to 14,16.1 to 16.5, 24.1 and 24.2</td><td>July 23 to 25, 2007 and July 10 to 11, 2008</td><td>- Review drill core
- General site layout and conditions
- Review QA/Qc procedures</td></tr>
<tr><td>Mark Liskowich, P.Geo.</td><td>Environmental Considerations
Report Sections: 20, 21.1.5, 21.1.6 and 21.2.7</td><td>Various</td><td>Several site visits to Rabbit Lake over many years looking at environmental issues</td></tr>
</table>

2 Reliance on Other Experts

Information concerning claim status, ownership and assessment requirements which are presented in Section 3 have been provided to the authors by UEX and have not been independently verified by the authors. However, the authors have no reason to doubt that the title situation is other than which has been presented here.

3 Property Description and Location

The following section was taken directly from UEX's November 12, 2008 N.I. 43-101 report entitled "Technical Report on the Geology of, and Drilling Results from, the Horseshoe and Raven Uranium Deposits, Hidden Bay Property, Northern Saskatchewan" by Rhys et al. (2008).

Minor changes and updates have been made and comments inserted where appropriate.

3.1 Property Location

The Hidden Bay property is located in the Wollaston Lake area of northern Saskatchewan approximately 740 km north of the city of Saskatoon (Figure 3.1), immediately west of Wollaston Lake. The property crosses the boundary between and is located within both the Reindeer and La Ronge mining divisions of northern Saskatchewan. Approximate limits of the property are latitude 57^{o}52'N to 58^{o}27'N (UTM NAD 83 6414000N – 6480000N) and longitude 103^{o}35'W to 104^{o}10'W (UTM NAD 83 552000E – 584000E). Portions of the property occur in 1:50,000 scale topographic map sheets 64L/5, 64L/4, 74I/1 and 74H/16 of the Canadian National Topographic system.

Mineral dispositions are located in the field by corner and boundary claim posts which lie along blazed boundary lines. Post locations and blazed lines for the S-106962 claim, which contains the Horseshoe and Raven deposits, were refurbished and checked by GPS survey by UEX personnel in October 2008. Common boundaries with Cameco's adjacent Rabbit Lake property have been surveyed by Cameco personnel. Claim boundaries in other parts of the Hidden Bay property are defined by unsurveyed corner and boundary claim posts which lie along blazed boundary lines.

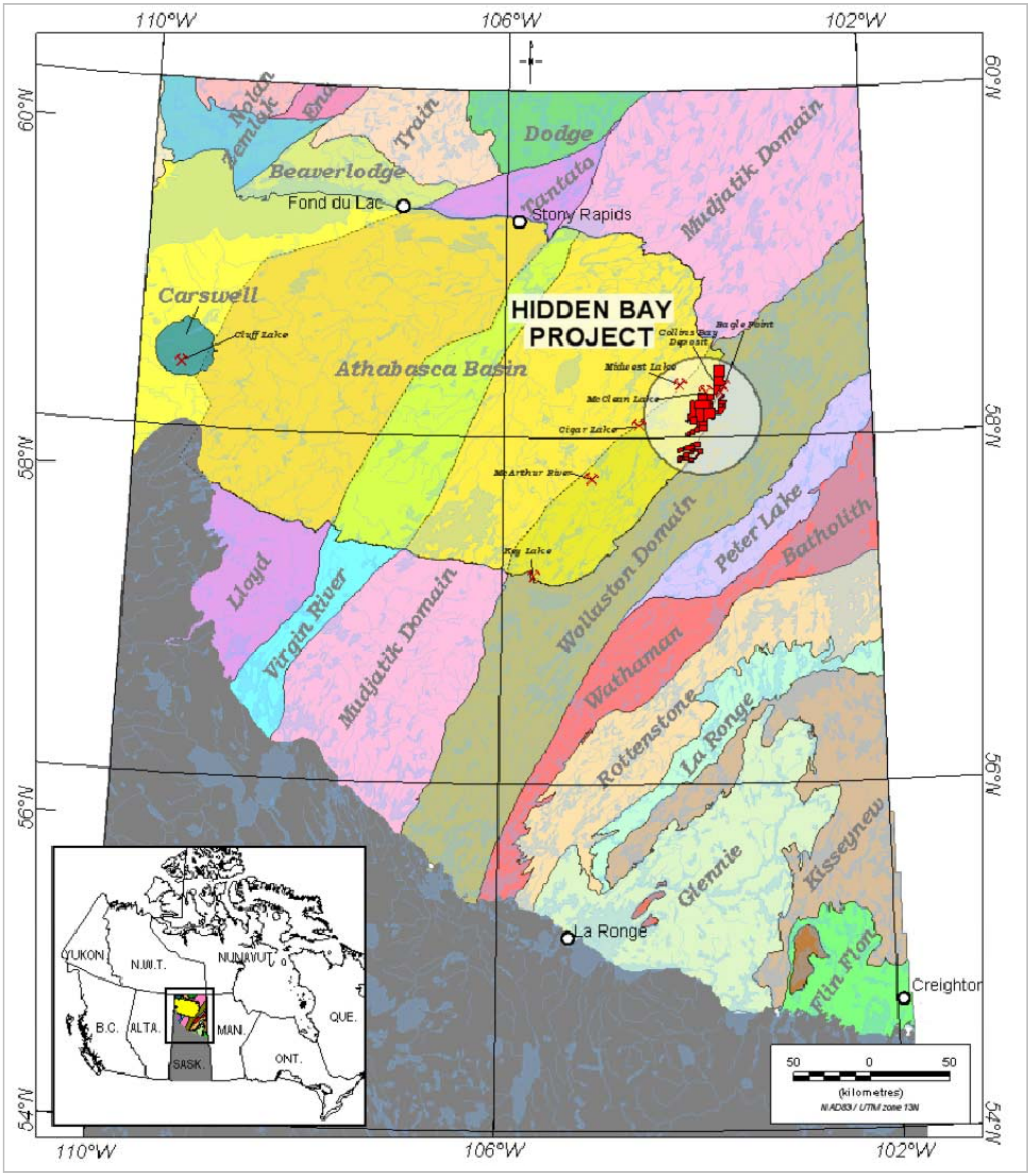


Figure 3.1: Location and Regional Geology of the Hidden Bay Project

3.2 Concession Descriptions and Title

The Hidden Bay property consists of 57,321 ha (573 km^2) in 43 mineral dispositions (Table 3.1; Figure 3.2). These are all owned 100% by UEX except for 297 ha in disposition ML 5424, which is currently owned 76.729% by UEX, 8.525% by ENUSA Industrias Avanzadas, 7.680% by Nordostschweizerische Kraftwerke AG and 7.066% by Encana. Disposition ML 5424 is in southernmost portions of the Hidden Bay property, distal to the Horseshoe and Raven deposits. The Hidden Bay property comprises one contiguous main block totalling 46,376 ha (26 dispositions) and one outlying disposition group to the south in the West Bear area (West Bear and Rhino Claims) totalling 10,945 ha (16 dispositions). The Horseshoe and Raven deposits are in the northern, larger block, entirely within disposition S-106962. The West Bear deposit ("West Bear") is located within the southern block of the Hidden Bay property on mineral claim S-106424 (Figure 3.2).

None of the dispositions are subject to any royalties, back in rights or encumbrances. No mining or waste disposal has occurred on the Hidden Bay property and, consequently, the property is not subject to any liabilities due to previous mining activities.

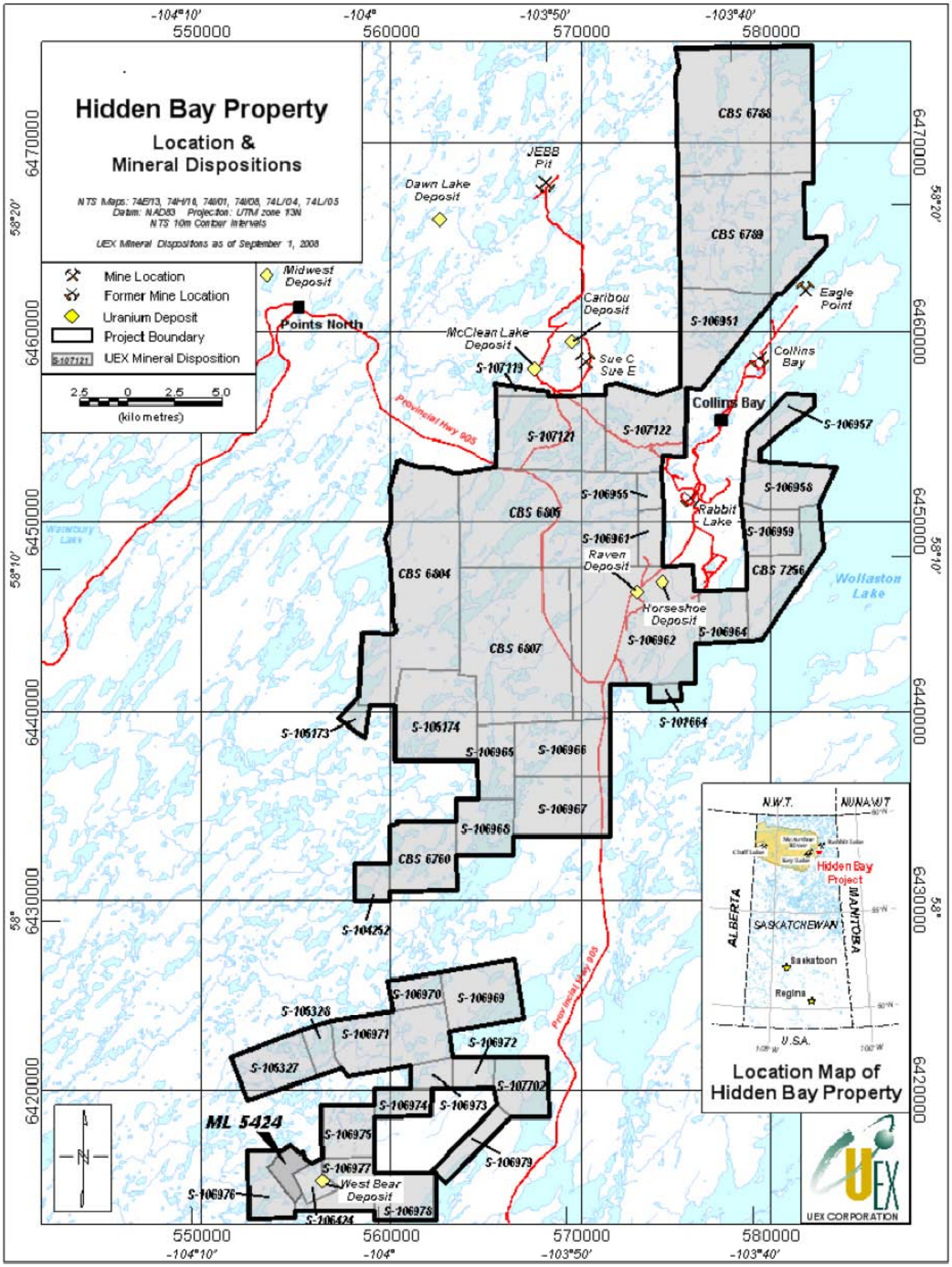


Figure 3.2: Hidden Bay Property , Location and Mineral Dispositions

Table 3.1: List of Mineral Dispositions Comprising the Hidden Bay Property as of February 1, 2011

Claim number	Record Date	Area (ha)	Annual Assessment
CBS 6760	Dec. 1, 1977	1,242	$ 31,050
CBS 6788	Dec. 1, 1977	4,755	$ 118,875
CBS 6789	Dec. 1, 1977	4,125	$ 103,125
CBS 6804	Dec. 1, 1977	4,345	$ 108,625
CBS 6805	Dec. 1, 1977	4,710	$ 117,750
CBS 6807	Dec. 1, 1977	4,510	$ 112,750
CBS 7256	May 8, 1987	1,369	$ 34,225
ML 5424	Mar. 21, 2005	297	$ 22,275
S-101664	Oct. 8, 2004	153	$ 1,836
S-104252	Apr. 11, 1994	380	$ 9,500
S-105173	May 28, 1996	178	$ 4,450
S-105174	May 28, 1996	1,932	$ 48,300
S-105327	Aug. 21, 1995	988	$ 24,700
S-105328	Aug. 21, 1995	332	$ 8,300
S-106424	Dec. 1, 1977	300	$ 7,500
S-106951	Dec. 1, 1977	1,615	$ 40,375
S-106955	Dec. 1, 1977	258	$ 6,450
S-106957	Dec. 1, 1977	529	$ 13,225
S-106958	Dec. 1, 1977	1,050	$ 26,250
S-106959	Dec. 1, 1977	722	$ 18,050
S-106961	Dec. 1, 1977	398	$ 9,950
S-106962	Dec. 1, 1977	4,486	$ 112,150
S-106964	Dec. 1, 1977	713	$ 17,825
S-106965	Feb. 5, 2002	758	$ 9,096
S-106966	Feb. 5, 2002	1,483	$ 17,796
S-106967	Feb. 5, 2002	1622	$ 19,464
S-106968	Feb. 5, 2002	888	$ 10,656
S-106969	Feb. 5, 2002	1,270	$ 15,240
S-106970	Feb. 5, 2002	444	$ 5,328
S-106971	Feb. 5, 2002	1,806	$ 21,672
S-106972	Feb. 5, 2002	361	$ 4,332
S-106973	Feb. 5, 2002	327	$ 3,924
S-106974	Feb. 5, 2002	450	$ 5,400
S-106975	Feb. 5, 2002	770	$ 9,240
S-106976	Feb. 5, 2002	660	$ 7,920
S-106977	Feb. 5, 2002	797	$ 9,564
S-106978	Feb. 5, 2002	800	$ 9,600
S-106979	Feb. 5, 2002	490	$ 5,880
S-107119	Dec. 1, 1977	128	$ 3,200
S-107121	Dec. 1, 1977	2,273	$ 56,825
S-107122	Dec. 1, 1977	1,754	$ 43,850
S-107702	Dec. 30, 2004	853	$ 10,236
Totals		**57,321**	**$ 1,266,759**

Note: Data has been provided by UEX and has not been independently verified by the authors

3.3 Annual Expenditures

Annual expenditures of $12.00 per hectare are required for the first ten years after staking of a claim to retain each disposition. This rate increases to $25.00 per hectare annually after ten years, a rate which currently applies to most of the dispositions comprising the Hidden Bay property.

Required assessment work for each disposition in 2011 is listed in Table 3.1. Total annual assessment expenditure requirements for the entire Hidden Bay property are $1,266,759. Many of the dispositions on the Hidden Bay property have substantial exploration credits that reduce the overall required annual expenditures that are currently required.

3.4 Permits for Exploration, Environmental Issues and Liabilities

Permits for timber removal, work authorization, shore land alteration and road construction are required for most exploration programs from the Saskatchewan Ministry of Environment. Apart from camp permits, fees for these generally total less than $200 per exploration program annually. Camp permit fees are assessed on total man day use per hectare, with a minimum camp size of one hectare assessed. These range from $750 per hectare for more than 500 man days to $175 per hectare for less than 100 man days.

UEX advised Golder and SRK that they have no knowledge of any environmental issues or liabilities on the Hidden Bay property and that they obtained all the permits required to conduct exploration activities on the property for the 2002 to 2010 exploration campaigns were obtained.

4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The following section was taken directly from UEX's November 12, 2008 N.I. 43-101 report entitled "Technical Report on the Geology of, and Drilling Results from, the Horseshoe and Raven Uranium Deposits, Hidden Bay Property, Northern Saskatchewan" by Rhys et al. (2008).

Minor updates and changes have been made and comments inserted where appropriate.

4.1 Accessibility and Infrastructure

The Hidden Bay property is in the eastern Athabasca uranium district, 10 km east of Points North, Landing adjacent to and surrounding several current and past producing uranium deposits on the Rabbit Lake property of Cameco and the McClean Lake property operated by AREVA (Figure 4.1). The property is accessible year round by Highway 905, a maintained all-weather gravel road and by maintained access and mine roads to the Rabbit Lake and McClean Lake mining operations, which pass through the property. Drilling access roads to both the Horseshoe and Raven deposits lie mainly on high ground and are easily accessible year round from Highway 905.

Two airstrips in the area, the Rabbit Lake airstrip and the Points North airstrip, are serviced by several air carriers which provide scheduled flights to major population centers in Saskatchewan for mining operations, fishing and hunting lodges and road maintenance crews.

Float and ski-equipped aircraft can land on most of the larger lakes that are abundant on the property year round. Power and telephone lines to the mine sites link the property area to the Saskatchewan power grid and telephone system. Abundant water is available from the numerous lakes and rivers in the area.

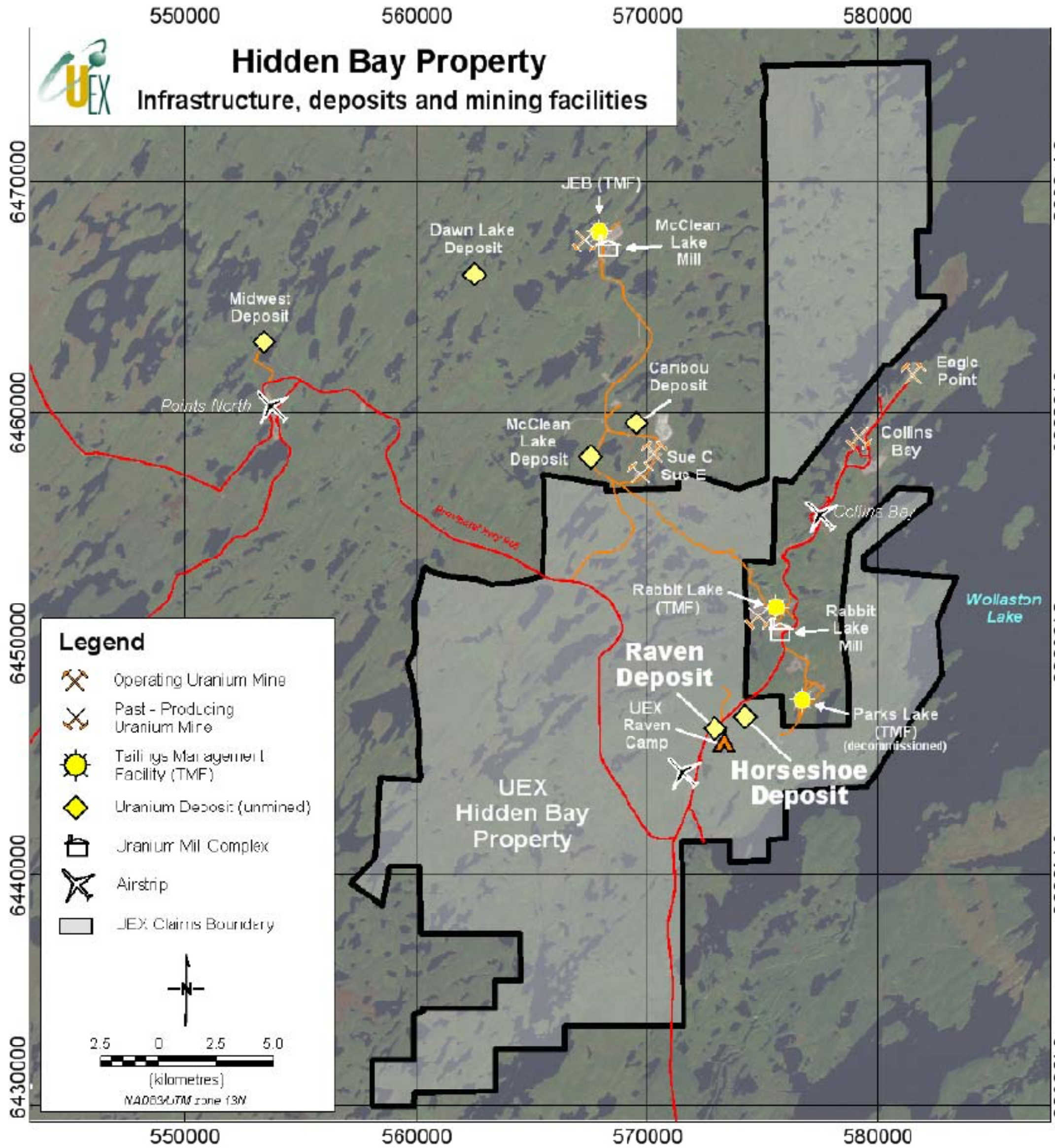


Figure 4.1: Deposits, Infrastructure and Mining Facilities: North and Central Hidden Bay Property

Since 2006, UEX has managed all of its exploration activities in the Hidden Bay area from the Raven Camp, a currently permitted exploration camp which is located 0.8 km south of the Raven deposit (Figure 4.1). This camp is powered by diesel generators. Accommodation in the area is also available at the Points North Landing airstrip to the west.

The Rabbit Lake mill facility, located on the adjacent Rabbit Lake property, is a fully functional uranium ore processing facility owned and operated by Cameco that is located adjacent to the Hidden Bay property 4 km northeast of the Horseshoe and Raven deposits. A second mill facility, the Jeb Mill, operated by AREVA, is located 22 km to the northwest of the Horseshoe and Raven deposits.

Road access along Highway 905 and power transmission lines to the Rabbit Lake and McClean Lake mill facilities pass over central portions of the property near the Horseshoe and Raven deposits.

4.2 Climate, Vegetation and Physiography

The mean daily maximum temperature is 15° C in July, with occasional extremes of up to 30°C. The mean daily minimum temperature is -24°C in winter, with occasional extremes of as low as -45° C. Average annual precipitation is 550 mm, divided equally between rain and snow, occurring regularly throughout the year. The mean annual maximum accumulation is 53 cm (Environment Canada Website, 2008).

Physiography of the Hidden Bay property is typical of Canadian Shield terrain, comprising low rolling hills separated by abundant lakes and areas of muskeg. Relief varies from a base elevation of approximately 396 m above mean sea level (“AMSL”) on Wollaston Lake to the east, to approximately 520 m AMSL near the Rabbit Lake mill site on the adjacent Rabbit Lake property. Hills are typically covered in a mixed boreal jack pine, spruce and aspen forest, separated by low-lying, swampy areas and muskeg fringed by stunted spruce stands.

The geomorphology is dominated by glacial and periglacial sediments that were produced during at least three ice advances (Fortuna, 1984). Outcrop is most common, but not abundant, in southeastern parts of the property underlain by metamorphic rocks outside the Athabasca Basin, particularly near Wollaston Lake and to the north and south of the Horseshoe and Raven deposits. The remainder of the property is mainly covered by glacial sediments. The occurrence of the Horseshoe and Raven deposits beneath a low ridge above adjacent swampy areas allows year round access to drilling roads above the deposits.

5 History

The following section was taken directly from UEX's November 12, 2008 N.I. 43-101 report entitled "Technical Report on the Geology of, and Drilling Results from, the Horseshoe and Raven Uranium Deposits, Hidden Bay Property, Northern Saskatchewan" by Rhys et al. (2008).

Minor updates and changes have been made and comments inserted where appropriate.

5.1 Ownership History

The Hidden Bay property forms part of the original exploration permits acquired by Gulf Minerals Canada Limited ("Gulf") in 1968 during early phases of exploration in the eastern Athabasca Basin. Commencing in 1976, parts of the property were subject to a joint venture agreement between Gulf, Saskatchewan Mining Development Corporation ("SMDC") and Noranda Exploration Company Ltd., with Gulf as operator. In 1983, the interests of Gulf in the property were acquired by Eldorado and, subsequently, with the amalgamation of Eldorado and SMDC in 1988 to form Cameco, full ownership was transferred to Cameco.

In 2002, an agreement was entered into between UEX and Cameco providing for the transfer of the dispositions now comprising the Hidden Bay property which were held by Cameco and Cameco's interest in disposition ML 5424, to UEX following completion of an arrangement proposed by Pioneer Metals Corporation and UEX. According to the agreement between UEX and Cameco, fourteen of Cameco's dispositions were transferred into UEX in their entirety, while five dispositions (CBS-6803, CBS-6806, S-104653, CBS-6802 and CBS-6808) were subdivided by re-staking in January-February 2002 and portions of which were renumbered and incorporated into the Hidden Bay property. Cameco retained the remaining portions of these dispositions that were not included in the Hidden Bay property. These portions cover mine infrastructure and disturbance in their Rabbit Lake property, which lies adjacent to and is partially surrounded by northeastern portions of the Hidden Bay property. Cameco acquired an initial 40% interest in UEX through this transaction (see Pioneer Metals Oct. 24, 2001 news release) and with subsequent dilution currently holds a 21.3% ownership in the company. Additional claims (S-106976 to S-106979) were acquired directly through staking by UEX in 2002.

5.2 Exploration History

5.2.1 Exploration of the Eastern Athabasca Uranium District

The Hidden Bay property occurs within the eastern Athabasca Basin uranium district, which contains several world class uranium deposits. Adjacent properties host seven current and past producing mines and, consequently, the property has been extensively explored since initial discoveries were made in the area in the 1960s. The exploration history outlined below is compiled from several sources, including Jones (1980), Craigie (1971), Andrade (1983a and 1983b), Studer (1984), Ward (1988) and Baudemont et al. (1993).

Attention was first focused on the uranium potential of the region in 1967 when the New Continental Oil Group flew an airborne radiometric survey over the Wollaston Lake area.

Numerous anomalies identified within this survey led New Continental to acquire several exploration permits in the area. These permits were subsequently optioned to British Oil American Company in 1968; the company was later renamed Gulf. Follow-up work consisted of prospecting, mapping and diamond drilling. In October 1968, on the third and last hole of the diamond drilling program, a 50 m section of uranium mineralization was intersected beneath the shore of Rabbit Lake. Between 1969 and 1971, delineation drilling of this discovery in approximately 220 drill holes outlined the Rabbit Lake mineralization on the adjacent Rabbit Lake property.

As a result of the Rabbit Lake discovery, extensive exploration of the eastern Athabasca Basin commenced. Between 1969 and 1980, several deposits, including the Collins Bay zones and Eagle Point on the Rabbit Lake property, the Horseshoe, Raven and West Bear deposits on the Hidden Bay property and the McClean Lake and Sue deposits on the McClean Lake property immediately to the north, were discovered using a variety of geophysical techniques, geochemical methods, prospecting and systematic drilling of prospective targets. Other significant discoveries in the area on adjacent properties include McClean Lake, by Canadian Occidental Petroleum in 1979, Midwest Lake by Esso Minerals in 1978, Dawn Lake by Asamera Inc. in 1978 and the Jeb and Sue deposits on the McClean Lake property between 1985 and 1990 by Total Minatco Ltd.

Gulf commissioned a mill facility and commenced open pit mining at the Rabbit Lake deposit in 1975. After the Rabbit Lake mineral reserves were exhausted in 1984, mining operations moved progressively to the Collins Bay B (1985-1991), D (1995-1996) and A zone (1997) deposits and the Eagle Point deposit (1993-1999). Eldorado acquired the mining assets of Gulf in 1983, which in turn were subsequently acquired by Cameco in 1988, with the creation of that company through the amalgamation of Eldorado and SMDC. Since 1997, the Jeb and Sue deposits on the McClean Lake project, have been exploited by AREVA, formerly named Cogema Resources), the current operator of that project. Total combined production from these deposits and the deposits on the Rabbit Lake property, is more than 200 million lbs U_3O_8 to date (Jefferson et al., 2007).

5.2.2 Property Exploration History Prior to UEX Ownership (Pre-2002)

Due to its proximity to producing mines and the identification of several deposits on the property, the Hidden Bay property has been subject to numerous exploration programs since discovery of the Rabbit Lake deposit in 1969. A review of the details of all of the programs conducted on the area of the property would be too exhaustive to be relevant to this report so, instead, the methods employed significant discoveries made and summary details of the different types of programs that were completed are outlined below.

The reader is referred to compilation reports by Andrade (1983a, 1983b) and Studer (1984) for further details on work completed up until 1983 on the property and references to earlier work. Reports by Studer and Gudjurgis (1985), Studer (1986, 1987 and 1989), Studer and Nimeck (1989), Ogryzlo (1983-1988), Forand and Nimeck (1992), Forand, Nimeck and Wasyluik (1994), Forand (1995 and 1999), Powell (1996) and Foster, Wasyluik and Powell (1997) document work programs conducted between 1983 and 1998 and provide references to further work also conducted during those years.

No exploration was carried out on the property between 1998 and 2002. Exploration since 2002, when UEX acquired the Hidden Bay property, is summarized in Section 9 of this report.

The location and methods of exploration applied on the Hidden Bay property have varied with the differing geological models, exploration priorities and the new technologies developed since discovery of the Rabbit Lake deposit in 1968. Initial exploration programs in the area were based on the basement-hosted Rabbit Lake deposit model, which involved the search for the coincidence of gravity and magnetic lows associated with the large, intense alteration zone and associated faulting at that deposit. These programs employed a multiple parameter search methodology (Whitford, 1971), employing: (i) initial airborne gamma ray spectrometric, electromagnetic, gravity and magnetic surveys conducted in the late 1960s; (ii) ground geological and geophysical checks of the airborne radiometric anomalies; (iii) surface prospecting, scintillometer and geochemical reconnaissance surveys, including radon-in water surveys; and (iv) follow-up overburden and diamond drilling. Most of the Hidden Bay property was subject to these methods during the initial years of exploration, particularly in areas of exposed basement rocks to the southeast, where the potential for basement-hosted Rabbit Lake type deposits was deemed greatest. These methods were used extensively by Gulf up until 1976, when discoveries elsewhere in the Athabasca Basin, particularly the Key Lake deposit, where the spatial association between a string of deposits developed at the intersection between the sub-Athabasca unconformity with graphitic gneiss-hosted faults were recognized. The recognition of the probable genetic role of graphitic gneiss and associated faults in deposit localization shifted the emphasis to the use of ground based electromagnetic ("EM") surveys, such as horizontal loop ("HLEM"), as the principal first pass geophysical survey in target areas. These EM surveys were used to detect the presence of prospective, conductive graphitic lithologies beneath overburden and the Athabasca sandstone. EM surveys still form the principal geophysical exploration tool employed currently, although the technologies currently used differ from the initial programs (e.g. fixed and moving loop) and have led to the targeting of many programs that have ultimately resulted in many new discoveries in the region during follow-up drilling of anomalies.

Prior to the transfer of the Hidden Bay property claims from Cameco to UEX in 2002, more than 1,381 diamond drill holes totalling approximately 205,000 m in cumulative length had been completed on the Hidden Bay property, since commencement of uranium exploration on the property in the early 1970s (Rhys, 2002).

Principal target areas for diamond drilling include systematic drilling of major faults with known associated mineralization, including the Rabbit Lake, Telephone, Seal and Wolf Lake Faults, delineation drilling of deposits (Horseshoe-Raven and West Bear) and concentrated areas of drilling in geologically and geochemically prospective areas (e.g. Vixen Lake-Dragon Lake). Most diamond drilling campaigns have been initially targeted on the basis of ground geophysical surveys and locally, follow-up to reverse circulation drilling anomalies. The reader is referred to Rhys (2002) for further information on the location and quantity of drilling and a review of historical results outside of the immediate vicinity of the Horseshoe and Raven deposits. These exploration programs lead to the discovery of the Horseshoe and Raven deposits and the West Bear deposit by Gulf in the 1970s by follow-up of ground geophysical anomalies and prospecting and for which historical resources were estimated.

Reverse circulation drilling in 929 drill holes (16,818 m total) was also conducted in several programs completed principally between 1976 and 1981 as a grid-based testing of overburden and sandstone covered portions of central and northern parts of the property. These programs aided in the definition of the location and depth of the Athabasca unconformity and allowed evaluation of geological and geochemical environments and located uranium anomalies in overburden and bedrock.

5.2.3 Discovery and Historical Exploration of the Horseshoe and Raven Deposits

The Raven deposit was discovered by Gulf in 1972 during follow-up drilling of an EM conductor located up-ice from a radioactive boulder train in till that was discovered by prospecting (Bagnell, 1978). An EM-16 geophysical survey was subsequently performed over the area and several anomalies were identified. Follow-up drilling located Raven in 1972. Delineation drilling was carried out between 1972 and 1974, during which 22,571 m of diamond drilling were completed on the deposit in 98 drill holes (Bagnell, 1978). During the final year of the Raven drilling, mineralization was intersected several hundred m to the east of the Raven zone on the western flank of a combined gravity and magnetic low similar to that detected over the Raven deposit. This new mineralized area, which was subsequently named the Horseshoe deposit, was tested by drilling 23,173 m in 73 holes completed during 1974 and 1975.

Additional drilling was completed in 1976-1978 to test for mineralization between the deposits and to further delineate the zones. A total of 53,329 m of diamond drilling in 212 holes was completed over the Horseshoe and Raven deposit area by Gulf, which led to the estimation of historical resources.

5.2.4 Discovery and Historical Exploration of the West Bear Deposit

The West Bear deposit was discovered in 1977 by the drilling of a horizontal loop (HLEM – MaxMin II) geophysical conductor defined by ground surveys that directly followed up airborne Very Low Frequency-electromagnetic ("VLF-EM") anomalies (Ogryzlo, 1983). The deposit occurs in an isolated claim group that forms the most southwesterly part of the property, 40 km southwest of the Rabbit Lake deposit.

The deposit was defined by 41 diamond drill holes completed in 1977 (totalling 1,903 m) and 106 reverse circulation drill holes (totalling 3,549 m) completed in 1978-1979 (Ogryzlo, 1983). Reverse circulation drill holes were spaced at 7.6 m (25 ft) intervals along 30.5 m (100-ft) profiles, and alternate with diamond drill holes where they are present. Drilling delineated a 540 m long, sub-horizontal, northeast trending and cigar shaped deposit that straddles the Athabasca unconformity at depths of 10 m 30 m below surface. Widths of the deposit range from 12 m to 52 m in plan view, and the mineralized zone is 1.5 m to 20 m thick..

5.3 Historical Resources

Historical resources on the Hidden Bay property were estimated by Gulf for the Horseshoe, Raven and West Bear deposits. New N.I. 43-101 compliant resources for all three of these deposits have been subsequently reported, and are documented in Lemaitre (2006), Palmer (2007 and 2008), Palmer and Fielder (2009a, 2009b), and in this report (see Section 16 for details).

5.3.1 Historical Resource Estimates at the Horseshoe and Raven Deposits

Gulf estimated resources for both the Horseshoe and Raven deposits in the late 1970s, which were subsequently reported in Healey and Ward (1988) and Eldorado Resources (1986). Resources are summarized in Table 5.1.

The resources are based on drilling results from 212 diamond drill holes in both deposits which were spaced at intervals of 30 m to 80 m on grid lines spaced approximately 61 m (200 ft) apart in mineralized areas using BQ diameter drill core. Based on these resources, total uranium contained in both deposits reported by Healey and Ward (1988) is approximately 23 Mlb (10,387 tonnes) U_3O_8, with most contained in the Horseshoe deposit (59% or approximately 13.6 Mlb U_3O_8). These resources are reported to have been estimated by cross-sectional methods using a cut-off of 0.03% U_3O_8, but no details describing estimation methodology or other parameters are known. Due to the historical nature of these estimations, the need for an updated geological model, uncertainties regarding estimation methodology and uncertainties regarding downhole survey locations and assay quality control, these mineral resources are non-compliant with N.I. 43-101, are not being treated as current and should not be relied upon.

Although the historical Horseshoe and Raven mineral resources are non-compliant, they and the distribution of mineralization outlined by the Gulf drill holes demonstrated that significant mineralizing systems are present at both deposits.

On the basis of the historical drilling results, subsequent definition and step-out drilling in the deposit area was undertaken by UEX which has confirmed the presence of the historical Gulf drilling and in many areas has significantly expanded the footprint of the mineralization. This new drilling information is currently the basis of the N.I. 43-101 mineral resource estimates on the Horseshoe and Raven deposits.

5.3.2 Historical Resource Estimates for the West Bear Deposit

Historical resources at West Bear are documented by Boyd et al. (1980), and are based on the results of the 41 diamond drill holes and 106 reverse circulation drill holes which were drilled between 1977 and 1979. The minimum criterion used for inclusion of drill hole intercepts in the resource model was a minimum intersection of 0.03% U_3O_8 over 1.52 m (5 ft) (Boyd et al., 1980). Mineralized intersections used in the calculation occur in 60 drill holes on 18 sections spaced at 30.5 m, having a vertical thickness of 1.5 m to 19.8 m, and averaging 4.9 m.

Parameters used to calculate the West Bear resource were a cut-off grade of 0.03% U_3O_8 and a constant specific gravity of 2.29, based on the figures used at the Rabbit Lake deposit. Resources estimated by Boyd et al. (1980) are outlined in Table 5.1, and comprise an estimated 130,545 tonnes (1.266 million lbs) U_3O_8 at a grade of 0.44%. This historical mineral resource is non-compliant with N.I. 43-101, is not being treated as current, and should not be relied upon.

Table 5.1: Summary of Historical Mineral Resources Estimated on the Hidden Bay Property*

Deposit	Tonnes	Grade (% U_3O_8)	Cut-off grade (% U_3O_8)
Raven	3,063,000	0.14	0.03
Horseshoe	3,617,000	0.17	0.03
West Bear	131,000	0.44	0.03

**By Gulf Canada Minerals Ltd. (Boyd et al, 1980; Healey and Ward, 1988; Eldorado Resources, 1986)*

These historic mineral resource estimates do not conform with the categories outlined in Sections 1.2 and 1.3 of N.I. 43-101, are not being regarded as current and should not be relied upon.

5.4 Production

No uranium mining has occurred on the Hidden Bay property and no other forms of metallic mineral production are reported.

6 Geological Setting

The following section was taken directly from UEX's November 12, 2008 N.I. 43-101 report entitled "Technical Report on the Geology of, and Drilling Results from, the Horseshoe and Raven Uranium Deposits, Hidden Bay Property, Northern Saskatchewan" by Rhys et al. (2008).

Minor updates and changes have been made and comments inserted where appropriate.

6.1 Regional Geological Setting

The Hidden Bay property is at the eastern margin of the Athabasca Basin. The property is underlain by two dominant lithologic elements: (i) polydeformed metamorphic basement rocks of Proterozoic age, which are overlain by: (ii) flat-lying to shallow dipping, post-metamorphic quartz sandstone of the late Proterozoic Athabasca Group.

Basement rocks in the area are within the Cree Lake zone (Hearne Province) of the Early Proterozoic Trans-Hudson orogenic belt. The Cree Lake zone is composed of Archean gneiss and overlying Early Proterozoic or Archean supracrustal rocks (Bickford et al., 1994), both of which are affected by amphibolite to locally, granulite facies metamorphism. The Cree Lake zone is further subdivided into three transitional lithotectonic domains, of which the Hidden Bay property straddles the gradational boundary between the central and eastern domains, the Mudjatik and Wollaston Domains. The central belt, the Mudjatik Domain, is composed primarily of Archean granitic gneiss, often as domal bodies, which are separated by discontinuous zones of migmatitic, pelitic gneiss and mafic granulite (Lewry and Sibbald, 1980; Sibbald, 1983).

The transition from the Mudjatik to Wollaston lithostructural domains is represented at a regional scale by the rapid increase in the frequency of granite and quartzo-feldspathic gneiss domes in the Mudjatik Domain that profoundly influence the structural style and magnetic signature of the area. At a property scale (Figure 6.1), the boundary is gradational and indistinct. Sibbald (1983) laces the domain boundary along the south side of the Collins Bay Dome from north of the Eagle Point mine to the Rabbit Lake deposit and to the southwest from there, through Lampin Lake along the Rabbit Lake Fault (Figure 6.1). Since the lower pelitic gneisses of the Wollaston Group rocks are continuous with gneiss present west and north of the proposed Wollaston-Mudjatik boundary in the Mudjatik Domain, gneiss sequences on the property that straddle the boundary are collectively described below as basal portions of the Wollaston Group.

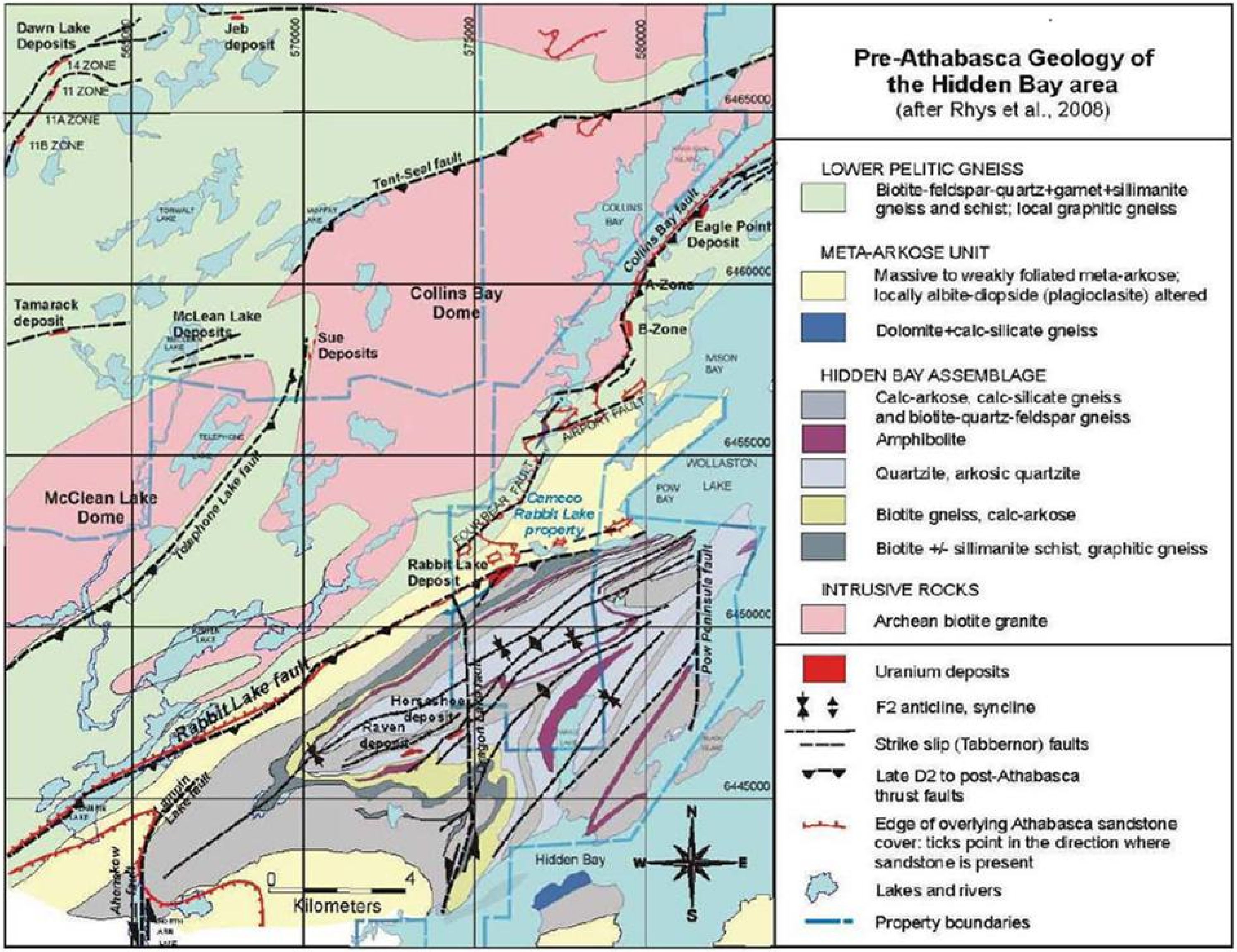


Figure 6.1: Regional Geology of the Horseshoe and Raven Deposits

The age of the Daly Lake and Geike groups, which are probably correlative with the major gneiss sequences of the Wollaston Domain on the Hidden Bay property, is constrained between the 1,920 Ma and 1,880 Ma age of detrital zircons (Yeo and Delaney, 2007) and minimum U-Pb zircon ages of 1,840 Ma and 1,850 Ma of granitic sills and bodies that intrude the sequence in the Hidden Bay area (Annesley et al., 2005). Archean granitic paragneiss units that occur in the western Wollaston and Mudjatik domains yield ages of between -2,550 Ma and -2,700 Ma (Annesley et al., 2005), forming local basement to the Wollaston Supergroup that is exposed in domal antiformal fold cores.

6.1.1 Wollaston Domain Geology on the Hidden Bay Property

Most of the Hidden Bay property is within the Wollaston Domain, which on the property comprises one of the type sequences through the Wollaston Supergroup. The domain is composed of a basal biotite-quartz-feldspar +/- graphite pelitic gneiss unit, which is contiguous with and overlies domes of Archean granitoid gneiss and which is contiguous with pelitic gneiss sequences in the Mudjatik Domain (Wallis, 1971). On the Hidden Bay property, the lower politic gneiss underlies much of the northern and northwestern portions of the property, surrounding the McClean Lake and Collins Bay granitic domes (Figure 6.1). Lowermost portions of the gneiss sequence, generally within a few tens to hundreds of m of the granitic domes, contain graphite-rich pelitic gneiss, along which pre- and post-Athabasca faults which are associated with uranium mineralization are localized. This lower graphitic unit is probably correlative with the Karin Lake Formation that is broadly present in basal portions of the Wollaston Domain regionally (Yeo and Delaney, 2007).

The pelitic gneiss is overlain to the southeast by massive to weakly foliated, grey meta-arkose unit, which near and northeast of the Rabbit Lake deposit is often affected by peak metamorphic albite-pyroxene alteration assemblages termed “plagioclasite” by previous workers (Appleyard, 1984). The meta-arkose unit extends east-northeast through the north-central portions of the Hidden Bay property through Lampin Lake to Pow Bay on Wollaston Lake (Figure 6.1) and is also widespread in southern portions of the property near the West Bear deposit. Discontinuous marble and calc-silicate units occur along the southeastern margins of the meta-arkose unit, at its contact with the Hidden Bay Assemblage to the southeast and form an important host rock to mineralization at the Rabbit Lake uranium deposit; similar, potentially correlative dolomite units occur along the southern shores of Hidden Bay (Wallis, 1971).

Collectively, the lower pelitic gneiss, meta-arkose and potentially the marble units probably form the local manifestation of the Daly River Group, which Yeo and Delaney (2007) define as comprising much of the central and lower portions of the Wollaston Supergroup regionally.

Quartzite with interlayered amphibolite and calcareous meta-arkose which define the Hidden Bay Assemblage of Wallis (1971) and Sibbald (1983) occur to the southeast of the meta-arkose unit in the central Hidden Bay property and is host to the Horseshoe and Raven deposits.

The assemblage is dominated by psammitic gneiss comprising mainly quartzite, quartz-rich meta-arkose and calc-silicate bearing meta-arkose (calc-arkose), but also includes bands of amphibolite and biotite-sillimanite +/- graphite bearing pelitic and semi-pelitic gneiss. These lithologies are described further in Section 6.2, since they are the principal host rocks to the Horseshoe and Raven deposits.

The Hidden Bay Assemblage may be regionally correlative with the uppermost lithologic sequence comprising the Wollaston Supergroup, the Geike River Group, which is extensive through much of the Wollaston Domain (Yeo and Delaney, 2007).

Igneous rocks in the region include probable Archean domes and several generations of granite and pegmatite sills, dykes and stocks that intrude the Wollaston Group. Northern parts of the Hidden Bay property are underlain by the McClean Lake and Collins Bay domes, which mark the transition from the Wollaston to the Mudjatik Domains (Figure 6.1). They are composed of massive, fine- to medium-grained grey biotite granite to tonalite, possibly of more than one phase. Annesley et al. (2005) report Archean U-Pb zircon ages for tonalitic gneiss on the margins of the McClean Lake dome.

6.1.2 Proterozoic Deformation and Metamorphism

Rocks on the Hidden Bay property are affected by at least two significant phases of Hudsonian age syn-metamorphic penetrative deformation, D1 and D2, which are manifested as widespread penetrative tectonic fabrics and folds. Younger features include at one or more generations of phase of open folds (D3, D4) and semi-brittle to brittle faults. Lithologies and foliation trend northeast with predominantly moderate to steep southeast dips, although northwest dips occur in some areas. Although predating uranium mineralization, these phases of deformation have created a complex lithologic architecture which has influenced the distribution of later brittle faults associated with uranium deposits and affect the position and morphology of uranium mineralization. Principal deformation events are as follows.

D1 deformation: The earliest recognizable deformation is manifested by ubiquitous gneissic compositional layering (S1) and a parallel shape fabric defined by alignment of peak metamorphic minerals (Wallis, 1971; Sibbald, 1983). S1 foliation strikes northeast with moderate southeast dips and is parallel to and in part defined by lithologies including compositional layers and granitic leucosomes. S1 is defined by unstrained peak metamorphic minerals, but is also overgrown by porphyroblasts of garnet and cordierite, which contain inclusion trails aligned parallel to S1 (Wallis, 1971; Rhys and Ross, 1999).

These relationships suggest that M1 peak metamorphism was synchronous with, but outlasted, D1 deformation and the formation of S1 foliation (Wallis, 1971). No associated major folds have been identified with this event, however (Sibbald, 1983), although rare rootless F1 folds are locally observable in drill core.

D2 deformation: D2 deformation is manifested by megascopic and minor folds (F2 folds), which have significantly influenced the map patterns of lithologies in the area and by the development of S2 foliation, which is axial planar to F2 folds of S1/gneissosity and lithologies. S2 is inhomogenously developed and varies from an intense foliation that overprints and transposes S1 to a spaced cleavage that is only developed in the hinge zones of F2 folds. Where it is intense, S2 transposes S1. In some units, S2 also forms a spaced crenulation cleavage that is defined by re-oriented domains of S1 and by the alignment of new unstrained metamorphic minerals.

The superpositions of S2 foliation on peak metamorphic mineral assemblages which define S1 and the evidence for new amphibolite-grade mineral growth during S2 suggest that D2 was accompanied by a second pulse of probable amphibolite-grade metamorphism (M2). A mineral lineation (L2) may be developed at the intersection of S1 and S2; it is often parallel to F2 fold axes.

At a regional scale, D2 folds are non-cylindrical and exhibit domal outlines and fold axes that have variable northeast and southwest plunges. Elliptical D2 folds are in part localized around granite domes, but variable fold axis plunges also occur in other areas. The parallelism of L2 elongation lineation with D2 fold axes suggests that significant stretching was accomplished parallel to the fold axes during folding, suggesting that the D2 folds may be sheath-similarly in geometry. The Horseshoe-Raven area is dominated by a series of inclined to upright megascopic D2 folds with southeasterly dipping axial planes that have wavelengths of 0.3 km to 2.0 km and shallow northeast plunging fold axes that form the major map patterns in the Hidden Bay Assemblage (Figure 6.1). At least two generations of late open folds with shallow dipping (F3) and steep (F4), northwesterly trending axial planes also affect lithologies in the area (Rhys and Ross, 1999). F3 folds are open folds with local shallow dipping axial planar cleavage that result in alternating northwest and southeast dips of gneissosity, complicating interpretation of drill core due to repetition of lithologies. Regionally, these folds may contribute to re-orientation of older folds and accentuate the domal map patterns that F2 folds define.

The Mudjatik and Wollaston Domains are affected by amphibolite to locally granulite facies metamorphism that accompanied D1 deformation, defining the main thermotectonic pulse of the Hudsonian orogeny. U-Pb zircon and monazite age dating indicates Hudsonian peak metamorphism occurred between approximately 1,830 Ma and 1,800 Ma in the Wollaston and Mudjatik Domains (Annesley et al., 2005). This metamorphism was accompanied by the intrusion of grey, commonly porphyritic granite sills and by subsequent anatectic K-feldsparquartz-biotite pegmatite sills (Annesley et al., 2005). A second metamorphic pulse may have accompanied D2 deformation between 1,775 Ma and 1,795 Ma.

6.1.3 Post-metamorphic Athabasca Sandstone

The folded Archean to Early Proterozoic metamorphic sequence is uncomfortably overlain by flat-lying to gently inclined quartz-rich sandstone of the Athabasca Group which dips gently to the west, resulting in progressively thicker sandstone westward from the eastern margins of the sandstone cover. The eastern boundary of the basin is erosional, but is in part influenced by post-Athabasca faulting. The sandstone is eroded from eastern and southeastern parts of the Hidden Bay property and is absent from the area of the Horseshoe and Raven deposits where the underlying gneissic basement is exposed.

U-Pb (uranium-lead) dating of apatite cement and dating of tuff units in upper portions of the Athabasca Group, as well as regional constraints on deposition by earlier Hudsonian age granites and Hudsonian deformation that the sub-Athabasca unconformity truncates, suggest progressive deposition of the Athabasca Group between 1769 and 1500 Ma (Ramaekers et al., 2007; Cumming and Krstic, 1992).

Widespread argillic alteration occurs in basement metamorphic rocks beneath the Athabasca sandstone to depths of several tens of metres below the sub-Athabasca unconformity. The alteration is similar in geochemistry, mineralogy and zoning to that observed today in lateritic profiles and consequently has been commonly interpreted as a saprolitic (paleoweathering) profile related to pre-Athabasca erosion of the gneiss sequence (e.g. Hoeve and Sibbald, 1978).

Alternatively, the alteration could be related to the reaction of oxidized diagenetic fluids in the Athabasca sandstone with underlying basement rocks, or a superposition of both processes. Argillic alteration associated with uranium mineralization is superimposed on this alteration.

6.1.4 Regional Faulting and Uranium Deposits

Two dominant, post-metamorphic fault orientations occur in the region (Wallis, 1971; Rhys and Ross, 1999):

a) concordant northeast-trending semi-brittle and brittle reverse faults; and

b) north-south trending, sinistral strike slip faults which represent western splays and parallel structures of the major Tabbernor Fault system.

Both types of faults are spatially associated with uranium deposits in the region.

Northeast-trending, generally graphitic or carbonaceous, reverse faults with moderate to steep southeasterly dips form the dominant fault type in the area. These faults trend sub-parallel or acutely oblique to lithologies and the dominant foliation and are frequently localized along graphitic gneiss units. In basement rocks beneath the Athabasca sandstone, these structures are composed of zones of cataclasis and low temperature semi-brittle (pressure solution) foliation development and clay gouge indicative of variations in structural style during deformation and/or multiple phases of displacement.

Fault fabrics and associated low temperature alteration are superimposed on earlier high temperature metamorphic fabrics.

Deformation style and associated alteration are compatible with retrograde low temperature (<250° C), low pressure conditions during fault activity. Shear fabrics and the reverse displacement of the Athabasca unconformity indicate a dominantly reverse shear sense on these structures with varying strike slip components, depending on fault orientation.

The over-thrusting of basement on to Athabasca sandstone occurred during brittle and, at least in part, during the semi-brittle phase of displacement on these structures since, in the latter case, displacement occurs even where faults lack clay gouge. However, evidence for significant pre-Athabasca, but post-Hudsonian displacement is also apparent on many of these structures where there is no displacement at the unconformity and fault fabrics are overprinted by the paleoweathering profile.

Although regionally extensive and important controlling structures to uranium deposits, post-Athabasca reverse displacement on these structures which offsets the unconformity is not high and generally only reaches a maximum of a few tens of metres on these structures, with the Rabbit Lake Fault having the largest reverse displacement (Rhys and Ross, 1999). Displacement is generally southeast-side up. Northeast trending faults are strongly influenced in their morphology by pre-Athabasca basement geology and are arcuate where they pass around granitic domes and D2 folds, forming favourable structural sites for the formation of uranium deposits.

The most economically significant northeast-trending faults in the Hidden Bay area include:

a) *The Collins Bay Fault*, an arcuate, northeast trending fault which is developed to the northeast of the property, on the adjacent Rabbit Lake property. This fault is a graphitic semi-brittle shear zone up to 15 m wide, often in two to three parallel splays with locally greater than 70 m of reverse displacement that has been traced continuously by drilling for nearly 11 km from 3 km southwest of the Collins Bay B-zone to 2 km northeast of the Eagle Point mine (Figure 6.1). At its southwestern end, the fault terminates in a series of en echelon steps that may represent en echelon linking faults that join the Rabbit Lake Fault zone.

b) *The Rabbit Lake Fault* (Sibbald, 1977) is the dominant and most continuous northeast trending fault in the area, with drilling indicating a minimum 40 km strike length. The Rabbit Lake Fault varies from concordant and localized in graphitic gneiss near the top of the Wollaston lower pelite unit southwest of Lampin Lake, to obliquely crossing lithologies and striking between 005 and 015 °more southeasterly (clockwise) than the lithologic trends near the Rabbit Lake deposit (Figure 6.1), 4 km north of the Horseshoe and Raven deposits. On this structure, at the western margin of the Hidden Bay property, 100 m to 150 m of apparent reverse, southeast side up vertical displacement of the Athabasca sandstone is apparent.

c) *The Telephone Lake Fault* is developed 5 km to 10 km north of the Rabbit Lake Fault in northwestern parts of the Hidden Bay property (Figure 6.1). This fault dips moderately to steeply southeast and is developed primarily in graphitic gneiss units several tens of m above the McClean Lake granite dome. The fault has approximately 60 m to 90 m of reverse displacement distributed over a 20 m to 70 m wide fault zone containing multiple minor faults.

Other significant northeast trending faults include the Tent-Seal Fault, which occurs in northeast parts of the Hidden Bay property along the northern margin of the Collins Bay Dome (Figure 6.1). This structure, which may represent a continuation of displacement along the nearby Telephone Lake Fault, is localized in graphitic gneiss and accommodates several tens of m of reverse displacement.

The second major fault type in the Hidden Bay area comprises north trending, steeply dipping strike-slip faults ("Tabbernor" faults) with dominantly strike slip (sinistral) displacements. The Tabbernor Fault system is a major sinistral north-south trending fault system that is developed to the east of the Athabasca Basin with a strike length of greater than 600 km (Wilcox, 1990). Although the main fault system passes to the east of the property, several branches and parallel faults related to the Tabbernor Fault system extend into the local area. The fault system is a long lived structural feature with early ductile and younger brittle and semi-brittle displacement history and a predominantly sinistral, strike slip shear sense (Elliot, 1994). Fabrics in this structure are post-metamorphic since they deflect and offset metamorphic foliation (Elliot, 1995). Younger brittle faults composed of gouge and cataclasite are superimposed on the ductile fault (Wilcox, 1990).

Several probable Tabbernor-type north trending faults occur in eastern parts of the property, beyond the limits of the Athabasca Basin. These include the Ahenakew, Dragon Lake, Pow Peninsula, Hungry Bay and Otter Bay faults (Wallis, 1971). The faults form topographic lineaments and low swampy areas in many lithologies. Where exposed in outcrop, the faults form steep west-dipping fault zones with clay matrix cataclastic breccias, associated clay-hematitic alteration envelopes, which are surrounded by sets of northwest-trending quartz veinlets. The closest of these Tabbernor Faults to the Horseshoe and Raven deposits is the Dragon Lake Fault, which passes immediately to the east of the Horseshoe deposit. Hoeve and Sibbald (1978) document approximately 200 m of sinistral displacement on the Dragon Lake Fault. The Ahenakew Fault, which also accommodates several hundred m of apparent sinistral displacement, passes 6 km east of the West Bear deposit.

The long history of Tabbernor Faults regionally suggests that these structures existed and potentially were active, at the same time that the northeast trending faults were active. Where drilling and outcrop information is sufficient to trace both fault types in the Hidden Bay property area, the best exposed Tabbernor Faults, the Ahenakew and Dragon Lake Faults, do not cross or displace the northeast trending Rabbit Lake thrust fault.

Instead, both of these faults bend into northeast trending structures where they approach the Rabbit Lake Fault and the meta-arkose unit of the Wollaston Group (Figure 6.1). In the Rabbit Lake mine area, the North-South fault, a northeast trending splay off the Dragon Lake Fault, links it to the Rabbit Lake Fault (Figure 6.1).

Similarly, mapping by Wallis (1971) and drilling indicates that the Ahenakew fault terminates where it intersects the meta-arkose unit in a northeast trending structure, the Lampin Lake fault (Figure 6.1). The Tabbernor Faults may thus feed into the northeast trending faults. Their dominantly sinistral/east side up displacement sense is compatible with the predominantly reverse displacement apparent on the northeast trending structures and suggests that they both were active in response to northwest-southeast directed shortening. These linking points form highly prospective areas for uranium deposits, as illustrated by the Rabbit Lake deposit.

6.2 Local Geology of the Horseshoe and Raven Area

6.2.1 Host Lithologies to the Horseshoe and Raven Deposits

The Horseshoe and Raven deposits are hosted by the Hidden Bay Assemblage, which occurs within a complex northeast-trending D2 synclinorium that sits structurally above and south of the underlying meta-arkose unit of the Daly River subgroup. The synclinorium is cored by quartzite that is succeeded outward concentrically from the core of the folds by other components of the Hidden Bay Assemblage which include a mixed sequence of calc-arkose, additional quartzite, locally graphitic sillimanite-bearing pelitic schist and amphibolite (Figure 6.1). While no Athabasca Sandstone is present above the Horseshoe and Raven deposits since it has been eroded from the local area, sandstone outliers that occur to the southeast of the deposits across Hidden Bay and the local presence of paleoweathering in some drill holes south of the deposit area suggest that the sub-Athabasca unconformity was present just above the current surface. A geological map of the deposits is presented in Figure 6.2 and is based largely on drill hole information that was augmented by geophysical work since outcrop exposure is poor or lacking in most of the deposits area. Descriptions of principal lithologies below are augmented by petrography of representative samples in Ross (2008a), Hubregtse and Duncan (1991) and Quirt (1990).

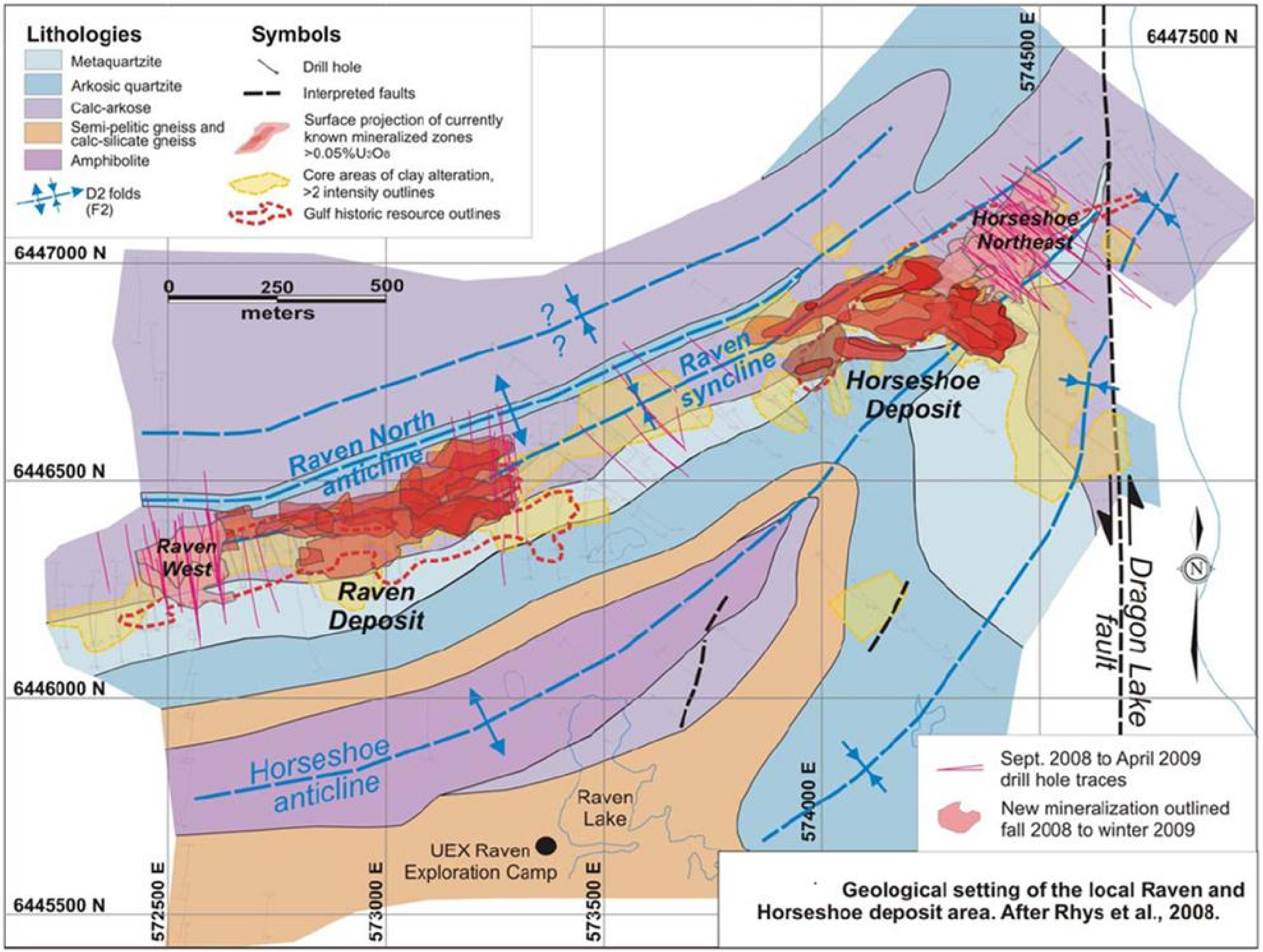


Figure 6.2: Local Geology of the Horseshoe and Raven Deposits

Five dominant lithologic units occur in the deposit area and define a distinct metamorphic stratigraphy. Overall stratigraphy comprises from structurally highest to lowest amphibolites, semi-pelitic and calc-silicate gneiss, arkosic quartzite, quartzite and calc-arkose. In addition, graphite-bearing biotite-quartz-feldspar gneiss is present west and southwest of the deposit area, but is not intersected by any of the drill holes in the immediate area of the deposits.

Photographs of these lithologies can be found in Rhys et al (2008). Principal lithologic units are as follows, listed from structurally lowest to highest in the area of the deposits:

- *a) Amphibolite (drill logging code = AMPH):* This unit occurs as an east-northeast trending lens that, in plan view, reaches a thickness of up to 300 m, which subcrops 300 m to 600 m south of the Raven deposit in the core of the Horseshoe anticline. Amphibolite is dark green grey, massive and coarse-grained and is dominantly comprised of semi-prismatic, interlocking olive green hornblende (50%), intergrown with biotite (10-13%), plagioclase, minor amounts of K-feldspar, accessory apatite and locally up to 10% pyroxene (Ross, 2008a). The distribution of the minerals is irregular, giving the rock a mottled texture. The hornblende crystals range up to 2 mm in length and commonly occur in clots up to 1.5 cm. This rock type is only observed structurally below and south of the Raven deposit.
- *b) Semi-pelitic and calc-silicate gneiss (includes lithocodes SPL0, CALC, CARK and ARKQ):* This lithologically variable unit comprises interlayered semi-pelitic biotite-quartz-feldspar gneiss (code SPL0), calc-silicate (code CALC) and calc-arkosic (CARK) gneiss and local bands of arkosic quartzite gneiss (ARKQ).

  It surrounds the amphibolites in map view (Figure 6.2) and ranges from several tens of m thick adjacent to the amphibolites to more than 270 m in apparent thickness within one hole drilled beneath the Horseshoe deposit (HU-028).

  The unit has a highly variable thickness probably due to folding. Semi-pelitic biotite-quartz-feldspar gneiss predominates, but is often interlayered in its upper portions near the overlying arkosic quartzite unit with pyroxene-amphibole bearing green-grey calc-silicate gneiss that may contain medium- to coarse-grained pale green pyroxene-rich bands and with feldspar-pyroxene-biotite-amphibole bearing fine- to medium-grained, weakly foliated calc-arkose. Bands of arkosic quartzite are often present. Compositionally homogeneous and feldspar porphyroclastic biotite-quartz-feldspar gneiss which occurs locally in this mixed unit has possible myrmekitic intergrowths, suggesting that parts of it may represent metamorphosed, feldspar porphyritic intrusion of intermediate composition (Ross, 2008a).
- *c) Arkosic quartzite (lithocode ARKQ):* This unit is the principal host to mineralization at the Horseshoe deposit and also hosts a significant proportion of the mineralization at Raven.

  This lithology structurally overlies the mixed semi-pelitic and calc-silicate gneiss unit. Arkosic quartzite varies in thickness from 60 m to more than 300 m in apparent thickness at the Horseshoe deposit where it is thickest, averaging approximately 150 m, to typical true thickness of between 40 m and 100 m at Raven. This unit is typically pale grey coloured and varies from massive to locally banded, with banding defined by grain size and local compositional layering that may represent modified relict primary bedding (S0).

The unit varies from fine- to medium-grained, comprising 40% to 65% quartz, 10% to 35% K-feldspar, 10% to 20% plagioclase and typically 3% to 5% biotite when fresh, with local accessory rutile, titanite, pyrite, apatite and zircon (Ross, 2008a).

- *d) Quartzite (lithocode QZIT):* Quartzite lies structurally above the arkosic quartzite and is often gradational through a transition zone over a few m with that unit, in areas characterized by gradational changes in quartz and feldspar content and alternating quartzite and arkosic quartzite layering. It is generally coarser grained than the underlying arkosic quartzite and contains lower total feldspar content.

  Quartzite hosts a significant proportion of mineralization at the Raven deposit and parts of the Horseshoe deposit extend into this lithology. Quartzite has a highly variable thickness and, similarly, the arkosic quartzite is thickest at the Horseshoe deposit, where it generally exceeds 50 m in thickness, ranging locally from 20 m to more than 150 m thick, the latter on both limbs of the Horseshoe anticline in northeastern portions of the deposit. At Raven, the quartzite unit typically ranges from 20 m to 70 m thickness. In both deposits, it is thinnest on the northwest limb of the Raven syncline, where it is often less than 25 m thick and may be tectonically thinned by faulting that is spatially associated with uranium mineralization; it rapidly thickens to the southeast at Horseshoe. Quartzite is generally medium- to coarse-grained and composed of translucent pale grey quartz. The rock varies from weakly foliated with alignment of lenticular quartz grains and biotite and weak compositional layering, to massive textured. Quartzite is characterized by a high quartz content (83% to 88%) and a hard, massive, coarse-grained crystalline texture with crystals up to 8 mm.

  The unit contains up to 10% K-feldspar that is often altered to clay and sericite in or near mineralized areas. Biotite content is typically between 5% and 10%. Disseminated pyrite occurs locally and may be abundant (up to 3%), often associated with biotite or as hairline stringers. Other accessory phases observed are tourmaline, zircon and monazite. The quartzite often contains thin foliation parallel K-feldspar-quartz pegmatite lenses that range from less than one centimetre up to a few tens of centimetres thick.

- *e) Upper calc-arkose (lithocode CARK):* The calc-arkose unit forms the structurally highest portion of the metamorphic stratigraphy in the Horseshoe-Raven deposit area. The unit cores the Raven syncline and is preserved in the upper northwestern portions of the deposits within the synclinal trough, extending from surface to depths of approximately 150 m below surface in both deposit areas.

  The unit is also present further north, in a second synclinal trough across the Raven North anticline (Figure 6.2). Since the unit is only preserved in synclines and its top is eroded, its true thickness is unknown, but is a minimum of approximately 100 m. Mineralization at Horseshoe does not extend into this unit, but it contains a significant proportion of uranium mineralization at the Raven deposit. The calc-arkose unit is typically green-grey in colour and composed of massive to compositionally banded medium- to coarse-grained plagioclase (25-50%), K-feldspar (1-10%), pyroxene (10-25%), biotite (8-10%) and amphibole (2-10%), often with accessory disseminated pyrite or pyrrhotite.

The unit ranges from near massive where pyroxene and plagioclase are most abundant to well foliated where compositional layering and alignment of biotite and amphiboles occur, containing 0.2 cm to 4.0 cm wide pyroxene-plagioclase and biotite rich layers that define a gneissosity. North of the Raven deposit, well banded and layered portions of this unit are locally developed, with alternating pale green pyroxene and pale grey feldspar or dark green amphibole bands. The texture and mineralogy of this upper unit is comparable to some parts of the lower mixed semi-pelitic and calc-silicate gneiss (unit 2), which also contains calc-arkose and calc-silicate components, but which are interlayered with biotite-quartz feldspar gneiss.

In addition to the units described above, two volumetrically minor types of intrusions are also present in the deposits area: granitic pegmatite and fine-grained intermediate dykes. Isolated pegmatite (lithocode PEGM) dykes and/or sills intrude all lithologies in the Horseshoe- Raven area. They are generally less than 5 m thick and form only a minor part of the host lithologies. However, areas of intense pegmatite "segregations" often coincide with areas of significant alteration and/or mineralization. More than one generation of pegmatite dykes are present: early dykes which are affected by D1 strain and transposed into S1 foliation and a late set of shallow dipping planar dykes which are probably late or post D2 in timing as they cut across F2 folds and are unaffected by foliation development or strain. A single, fine-grained biotite-rich intermediate dyke (unit DIAB) that is present in multiple drill holes in northeastern parts of the Horseshoe area is also structurally late, planar and traceable across D2 folds, although does contain internal S2 foliation.

Unit DIAB has been most consistently intersected in the Horseshoe Northeast area, where it is several m thick, dips shallowly to the northwest and is intimately associated with pegmatite dyke that are parallel to it. This unit is overprinted by alteration and associated uranium mineralization.

6.2.2 Structural Setting - Metamorphic Structural Architecture

Lithologies in the Horseshoe and Raven areas outline several significant, upright open D2 (F2) folds in the local area (Figure 6.2). These folds have steep to moderate, southeasterly dipping axial planes and horizontal to shallow northeast plunging fold axes. A D2 timing is indicated since the folds affect both primary lithologic layering as well as lithology parallel S1 penetrative foliation. A spaced, vertical to southeast dipping S2 foliation is axial planar to the folds and locally crenulates older S1 foliation. No older, D1 folds were identified and, if they are present, they are similarly to be isoclinal and difficult to recognize, but could have caused lateral and vertical thickness variations in host lithologies.

Principal folds in the immediate deposit areas include the Horseshoe anticline and adjacent Raven syncline. The Horseshoe anticline is cored by amphibolites south of the Raven deposit and plunges to the northeast, where arkosic quartzite occurs in the hinge area in the Horseshoe deposit (Figure 6.2).

Similarly to other D2 folds in the area, this fold is non-cylindrical and varies in plunge, shallowing to the northeast, where it plunges very shallowly to sub horizontally to the northeast in the Horseshoe deposit area.

The adjacent Raven syncline, with its axial trace 250 m to 550 m northwest of the Horseshoe anticline, has a nearly horizontal fold axis and is cored along its length by arkosic quartzite forming the top of the local metamorphic stratigraphy. Uranium mineralization in both the Horseshoe and Raven deposits is elongate parallel to the trend and plunge of these folds and at Raven preferentially exploits the core of the syncline, while at Horseshoe, mineralization extends between these two folds obliquely crossing the folded sequence.

6.2.3 Post-Hudsonian Faulting in the Horseshoe-Raven Area

Few significant offsets of lithologies occur in the Horseshoe and Raven deposit areas and outside of clay alteration zones associated with uranium mineralization, lithologies are competent and generally lack any significant faulting. The most significant fault in the local area is the Dragon Lake Fault, a north-south trending Tabbernor Fault which passes east of the Horseshoe deposit (Figure 6.2). As discussed above, Hoeve and Sibbald (1978) document approximately 200 m of apparent sinistral displacement on the Dragon Lake Fault, based on displacement of lithologies. Where exposed in outcrop near the Rabbit Lake mine road and observed in core, the Dragon Lake Fault forms a steep west-dipping fault zone. The fault, from surface to depths of approximately 200 m comprises strands of silicified hematitic cataclastic breccias which are separated by variably clay-hematite altered and silicified host rocks. Local clay gouge seams are also present.

Abundant milky white drusy quartz veinlets are common along the trace of the fault in these clay-hematite altered areas and coincide with areas of most intense alteration; these trend northwest in outcrop exposures on the adjacent Rabbit Lake property (Rhys and Ross, 1999), indicating significant hydrothermal fluid flow has occurred along this structure. Alteration and brecciation collectively define a fault and fault damage zone that ranges from several m up to more than 20 m wide, with alteration locally extending tens of m further beyond the fault in some areas. Deeper, southeastern intercepts of the fault immediately to the southeast of the Horseshoe deposit, such as in drill holes HU-233 (329-333 m) and HU-064 (463.5-477.7 m), comprise chlorite-matrix breccias with variable hematite content and with sparse quartz veins. Overall patterns are for decreasing quartz vein density and hematite-illite abundance and for increasing chlorite abundance with depth and to the southeast along the fault. These changes may reflect differences in oxidation state and fluid type down the fault during a significant period of hydrothermal fluid flow along it.

The Dragon Lake Fault may represent a fluid pathway for oxidized hydrothermal fluids possibly originating from the pre-existing Athabasca Sandstone which may have overlain the Horseshoe-Raven area close to the present surface prior to erosion. No mineralization has been intersected on the Dragon Lake Fault to date, but the occurrence of the Rabbit Lake deposit at the intersection between the Rabbit Lake Fault and the North-South fault, a major splay of the Dragon Lake Fault to the north, suggests that this structure has the potential to host or control uranium mineralization.

Uranium mineralization in the Horseshoe and Raven deposits is associated with areas of clay alteration which become locally intense between some mineralized zones. At the Horseshoe deposit, mineralization occurs both above and below a shallow southeast dipping, tabular zone of clay alteration which is locally intense, particularly in northeastern portions of the deposit (Figure 6.2).

The intensity of clay alteration makes identification of potential clay gouge strands, which could occur through this area difficult and it is permissible that a fault zone may be present through the core of these altered areas. Similarly, a steep southeast dipping tabular zone of clay alteration underlies the Raven deposit and, if localized along a fault, may represent the same structure which could control alteration at Horseshoe. Also suggestive of a fault zone are changes in thickness and orientation of lithologies across this structure, including the abrupt thinning of the quartzite unit to typically less than 30 m in both deposits along the southwest dipping northwest limb of the Raven syncline where the clay alteration passes through it and the difficulty in tracing the Horseshoe anticline downward into the mixed calc-arkose/semi-pelitic gneiss beneath the alteration zone, suggesting it is offset. The fault strands now may be overprinted by clay alteration and mineralization, consistent with the timing of other uranium deposits in the region, where mineralization is late in the faulting history. Interaction of oxidized hydrothermal fluids along this potential fault with fluid flow along the adjacent Dragon Lake Fault may have contributed to the formation of hydrothermal fluid cells and to the localization of uranium mineralization in the deposits area (Figure 6.2).

7 Deposit Types

The following section was taken directly from UEX's November 12, 2008 N.I. 43-101 report entitled "Technical Report on the Geology of, and Drilling Results from, the Horseshoe and Raven Uranium Deposits, Hidden Bay Property, Northern Saskatchewan" by Rhys et al. (2008).

Minor changes have been made and comments inserted where appropriate.

The Hidden Bay property is within one of the most prolific uranium producing districts in the world, the eastern Athabasca uranium district. Deposits within the local area, within 0.5 km to 8 km of the property boundaries, have combined production and resources of more than 320 million pounds of U_3O_8 (123,000 tonnes U). Five past or currently producing mines on the adjacent Rabbit Lake property (Rabbit Lake, A-zone, B-zone, D-zone and Eagle Point) have together produced nearly 200 million pounds of U_3O_8 since 1975 and approximately 40 million pounds have also been produced from the Sue and Jeb deposits on the adjacent McClean Lake property (Jefferson et al., 2007). Production continues at both the Rabbit Lake and McClean Lake operations and several deposits nearby are in advanced exploration or permitting phases, including the Midwest Lake deposit located 12 km northwest of the property.

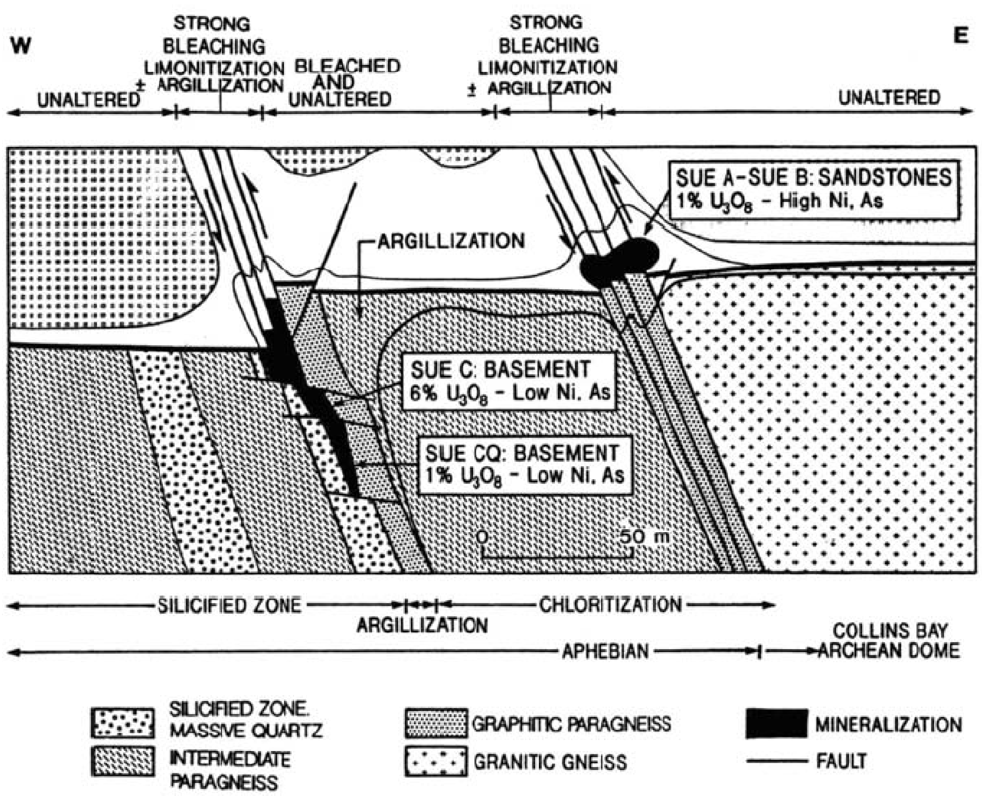


Figure 7.1 Schematic Cross-section through the Sue Zones, McClean Lake Property showing the Unconformity and Basement Styles of Uranium Mineralization that are Common in Unconformity-type Uranium Deposits

Illustrated in Figure 7.1 is a north view [from Baudemont et al., (1993)] showing the spatial association of basement (B-type) and unconformity (A-type) mineralization on parallel mineralized trends and the distribution of associated argillic alteration. Mineralization is developed in graphitic gneiss units that contain concordant faults. Mineralization at the West Bear deposit is of the unconformity A-type, which is comparable to the Sue A-Sue B deposits in the diagram. Mineralization at Horseshoe and Raven is a variant of B-type mineralization, comprising basement-hosted zones of disseminated and veinlet pitchblende-dominant mineralization associated with clay-hematite alteration around a probable fault zone.

These deposits collectively comprise different varieties of the unconformity-associated uranium deposit type described by Jefferson et al. (2007), Ruzicka (1996) and previous workers. All are spatially related to the sub-Athabasca unconformity in the region and are generally interpreted to result from interaction of oxidized diagenetic-hydrothermal fluids with either reduced basement rocks and/or with reduced hydrothermal fluids along faults extending upward toward the unconformity in underlying basement rocks beneath the unconformity (e.g. Hoeve and Quirt, 1985). The common occurrence of mineralization in and associated alteration overprinting Athabasca sandstone indicates post-Athabasca (post 1,700 Ma) timing for uranium mineralization in the region. U-Pb age dates obtained from uraninite mineralization in deposits throughout the Athabasca Basin support a principal phase of mineralization between 1600-1500 Ma with a potential second event between 1,460 Ma and 1,350 Ma and potential later periods of reworking indicated by younger ages (Fayek et al., 2002; Alexandre et al., 2003; Cumming and Krstic, 1992).

Uranium deposits in the area form three different, although commonly spatially related types of unconformity type uranium deposits:

A. Deposits developed at, or just above, the Athabasca unconformity in Athabasca sandstone along the trace of northeast-trending faults. These deposits occur in sandstone in the footwall wedge to graphite-bearing graphitic gneiss overthrust on Athabasca sandstone (e.g. Collins Bay A, B and D-zones), or in gradational drops/humps in the unconformity above graphite-rich lithologies and faults (e.g. Sue A/B, West Bear, McClean Lake; Figure 7.1, right). They are generally associated with non-calcareous graphitic and biotite gneiss. Mineralization occurs in pods and disseminations in intense hematite-clay-chlorite alteration, locally overprinting spatially associated breccias and zones of intense clay alteration that sit directly above mineralization in sandstone. Common structural sites include bends and steps in fault systems, or 5 m to 20 m humps in the unconformity that may reflect the interaction of graphitic shear zones with faults of different orientations. These deposits are characterized by assemblages of Ni and Ni-Co arsenides and sulpharsenides that accompany uranium mineralization.

B. Basement-hosted deposits within or surrounding fault zones in predominantly non-calcareous gneiss. These deposits are exemplified by Eagle Point, Sue C and Sue CQ, which are composed of veins, disseminations and pods that link, or replace faults in or near graphitic-bearing gneiss. Veins frequently occur in extensional fractures that may link individual faults (Sue CQ, Telephone zone; Figure7.1, left), or occur in en echelon steps in faults (Eagle Point).

Unlike deposits of class A, above, these deposits lack arsenide and sulpharsenide minerals in mineralized zones. Mineralization is composed of discrete pitchblende veins, planar replacements of fine-grained nodular pitchblende + clays, or undulating pitchblende/uraninite-bearing redox fronts surrounding clay veins and faults. A variation on this deposit type occurs at Horseshoe and Raven, where mineralization occurs in hematitic redox fronts and veins surrounding large, semi-tabular clay alteration zones that are cored by probable faults. Horseshoe and Raven differ, however, from other basement deposits in the region in that they lack spatially associated graphitic gneiss units or carbonaceous fault zones and are associated with an unconformity.

C. Basement hosted deposits associated with hydrothermal breccias in calcareous gneiss adjacent to northeast-trending faults. The only example of an economic mineralization of this type in the area is the Rabbit Lake deposit, although several local prospects are of similar style and the largest basement hosted unconformity deposits in the Alligator River district of northern Australia are closely comparable. The Rabbit Lake deposit occurs perched above the Rabbit Lake Fault at its intersection with the North-South fault, which is part of the Dragon Lake Tabbernor type fault system. Mineralization occurs on the margins of a large hydrothermal, chlorite-matrix breccia body that affects dolomitic marble and adjacent lithologies and that may have formed during dissolution collapse of the carbonate, forming a highly permeable zone. High grade mineralization is superimposed on the northeastern margins of the breccia and associated silicification/dravitization along the trace of the North-South fault.

Uranium deposits in the district frequently occur in deposit clusters that comprise one or more deposit types. Four major uranium deposits, the Collins Bay zones (Type A deposits) and the Eagle Point mine (Type B), occur along a 5.5 km strike length of the Collins Bay Fault system on the Rabbit Lake property. Other deposit clusters include the Sue, McClean Lake and Dawn Lake deposits, where deposits occur in at least two parallel trends, along which deposits may be strung out along parallel faulted graphite-bearing or calc-silicate units and spaced 100 m to 700 m apart. The position of mineralization may also vary systematically with respect to the Athabasca unconformity across deposit groups in these areas, varying progressively from deposits of Type A developed at, or perched above the Athabasca unconformity, to deposits of Type B, developed in basement rocks 10 m to 200 m below the unconformity that may occur along strike from the unconformity hosted mineralization (e.g. Sue C and Sue A/B; Eagle Point and the Collins Bay zones), accompanied by the disappearance of Ni-As-Co minerals in the basement hosted mineralized zones. The spatial coincidence of unconformity and basement-hosted deposits emphasizes the importance of testing both the unconformity and basement rocks where mineralization has only been historically discovered at the unconformity.

Deposits of all the styles described above are associated with and generally enveloped by, intense zones of argillic alteration that are composed predominantly of illite, chlorite and kaolinite. The influence of alteration extends over a far greater area than the dimensions of the deposits themselves and consequently the tracking of alteration distribution, mineral zonation and associated litho geochemical changes is an important tool in vectoring exploration (Sopuck et al., 1983).

In the Athabasca sandstone, alteration plumes may extend hundreds of m above the unconformity hosted uranium deposits, while in basement rocks alteration is generally more restricted to the vicinity of associated faults. Mineralization frequently occurs at redox fronts marked by zones of hematization, and a change from sulphide to oxide accessory mineral assemblages.

Uranium deposits in the area are generally associated with east and northeast trending, southerly dipping reverse fault zones that are localized within, or cross graphitic gneiss and carbonate/calc-silicate units (Figure 7.1). Mineralization occurs in areas of enhanced structural permeability and/or low stress (dilatancy) along faults including fault junctions (e.g. Rabbit Lake), beneath brecciated sandstone under over-thrust wedges (e.g. Collins Bay zones; McArthur River), at bends and en echelon steps in the faults (e.g. B-zone), and at dilational jogs (e.g. Eagle Point). These structural sites are in turn influenced at a broader scale by the occurrence of pre-Athabasca bends and lobes in the granitic domes and their mantling gneiss units, and folds within the metamorphic sequence, both of which have controlled the distribution, continuity and morphology of the faults.

Mineralization is generally structurally late in the faulting history, and while basement hosted mineralization is frequently localized along or adjacent to faults, both mineralization and its associated alteration may overprint fault rocks. The common position of deposits in fault zones and the morphology and orientation of vein systems suggest that mineralization occurred late during a period of northwest-southeast shortening and fault activity in the region. The occurrence of the Rabbit Lake deposit at the intersection of a northerly trending Dragon Lake Tabbernor-type fault with the northeast trending Rabbit Lake Fault, and the development of clay-hematite alteration with local anomalous radioactivity along the Tabbernor Faults in the local region, suggest that these faults may have also been active during the formation of deposits and contributed to fluid flow and localization of uranium deposits in the district.

8 Mineralization

The following section was taken directly from UEX's November 12, 2008 N.I. 43-101 report entitled "Technical Report on the Geology of, and Drilling Results from, the Horseshoe and Raven Uranium Deposits, Hidden Bay Property, Northern Saskatchewan" by Rhys et al. (2008).

Minor changes have been made and comments inserted where appropriate.

Uranium mineralization in the Horseshoe and Raven deposits occurs along an east-northeast trending zone of illite-Mg-chlorite clay alteration that is developed over at least 2.5 km strike length extending along the southeast flank of the Raven syncline. Along this clay alteration zone mineralization that has been defined (by both current and historical drilling) over strike lengths of approximately 1 km at each deposit, occur as multiple internal mineralized subzones. The two deposits are separated by approximately 0.5 km, laterally between which clay alteration is continuous and often intense, but in which widely spaced historical holes have intersected only anomalous radioactivity; additional drilling is planned in this area to test for additional potential mineralization between the deposits. The clay alteration zone may be cored by and potentially overprint a southeast dipping fault zone, which may have focused fluid flow and controlled the formation of dilatational vein and disseminated replacement style mineralization in the deposits.

Mineralization at the Horseshoe and Raven deposits is entirely hosted by folded arkosic quartzite, quartzite and calc-arkosic gneisses of the Hidden Bay Assemblage and occurs at depths ranging from a few tens of m up to 460 m below surface. The mineralization is locally open at depth. The Athabasca sandstone is eroded from and absent in the area of the deposits, but local sandstone outliers that occur to the southeast of Hidden Bay and sub-Athabasca paleoweathering that is preserved in the near surface in some nearby drill holes suggest that the current surface is just below the elevation of the original sub-Athabasca unconformity in the deposit area, prior to its erosion. Figures 8.1 and 8.2 show the plan and a typical section for mineralization of the Horseshoe deposit and Figures 8.3 and 8.4 are the equivalent figures for the Raven deposit.

Mineralization in each deposit surrounds, or is developed along, the generally southeast dipping clay alteration zone in multiple, generally shallow dipping lenses of disseminated and vein-like pitchblende-uranophane-boltwoodite mineralization that are associated with red-brown hematite alteration. Details regarding the morphology, dimensions and nature of mineralization in each deposit are discussed below.

8.1 Alteration Associated with Uranium Mineralization

The most prominent and continuous feature associated with uranium mineralization in both the Horseshoe and Raven deposits is the continuous, generally southeast dipping zone of clay +/- hematite alteration which extends through and between the deposits. The alteration zone may be manifested as a single, semi-tabular or lobate zone of moderate to steeply dipping alteration, or as multiple lenses and branching lobes of alteration which extend outward often along individual rock units, but which may extend upward or laterally off a narrow more steeply dipping tabular alteration zone that may be centered on a southeast dipping fault. Thickness of clay alteration is variable, but generally ranges from 20 m to 30 m thick depending on geometry. Alteration is developed with variable intensity and is most intense in the very thickest parts of the arkosic quartzite ("ARKQ") unit at Horseshoe and parts of the calc-arkose ("CARK") unit above the quartzite at Raven. In the Raven deposit, alteration locally varies from focused to more broadly distributed zones where patchy, weak to intense clays may affect intervals of quartzite up to 250 m wide.

The alteration zone at Horseshoe becomes progressively more tabular to the northeast, where it dips shallowly to the southeast, while alteration at Raven widens upwards into multiple lobes and shallow dipping zones, but which extend off a master, moderate to steep southeast dipping zone of clay alteration. The alteration zones are overall discordant to lithologies and dip more shallowly to the southeast than F2 fold axes, obliquely crossing F2 fold hinges. The shallower dipping areas of alteration at Horseshoe extend down dip to the east at the northeastern end of the Horseshoe deposit where strong clay alteration may widen up to 175 m in vertical thickness in a broad shallow dipping alteration zone, which extends east and southeast and merges with clay alteration surrounding the northerly trending, steep westerly dipping Dragon Lake Fault.

Clay alteration is composed of pervasive fine-grained pale grey or greenish clay, which preferentially affects feldspars and mafic minerals (biotite, amphibole and pyroxene). Consequently, units with highest feldspar content (e.g. arkosic quartzite, calc-arkose, semi-pelitic gneiss, pegmatite) often are most intensely altered, while quartzite, with its low feldspar content, may exhibits less and more restricted areas of alteration, locally forming a cap to larger areas of alteration beneath it, in the arkosic quartzite in western parts of the Horseshoe deposit. Loss of coherence due to destruction of framework silicates and bleaching or destruction of ferro-magnesium minerals occurs locally where alteration is most intense, where quartz is completely altered to clay, but in most areas, alteration in quartzite and arkosic quartzite retains primary quartz and even altered rocks where feldspars are dominantly clay altered remain competent and have excellent core recoveries during drilling. Within most intensely altered areas, intervals of intense clay often alternate with competent, moderately to strongly altered host rocks in which feldspars and biotite are clay altered and quartz may be pitted. Drusy quartz veins and irregular euhedral quartz-lined vugs occur particularly in areas of less clay altered arkosic quartzite and quartzite at the periphery of the clay alteration zones, possibly reflecting re-deposition of quartz outside the most intense quartz destructive areas of alteration.

To track and model areas of clay alteration, UEX codes relative clay alteration intensity from zero to four, with areas of intense, texturally destructive clay coded four. Areas with clay alteration of intensity two and higher are shown in yellow on cross-sections in Figure 8.4 and Figure 8.2 where "moderate" clay alteration indicates that at least 25% of the core is altered to clay.

Areas of intense clay alteration defined by drilling coincide well with geophysical gravity and resistivity lows. Anomalies that are coincident with clay alteration zones extend beyond areas of closely spaced drilling, outlining several prospective exploration target areas. Resistivity profiles also mirror the morphology of alteration on individual drilling cross-sections, allowing alteration and associated targets to be modelled three dimensionally and greatly enhancing drill targeting. The area of intense clay alteration extends for 2.5 km extending from the Raven deposit trending northeast past the end of recently defined Horseshoe mineralization.

Hematite Alteration

Areas of clay alteration at the Horseshoe and Raven deposits are often enveloped by 2 m to 100 m wide domains of brick red to brown hematite that occur on the margins of clay alteration or separated from the clays by several m of less altered wall rock. Fe-oxides in hematite alteration comprise mainly hematite with varying abundance of more amorphous Fe-oxy-hydroxide species (Ross, 2008b), which collectively are reddish brown to purple in hand sample. These hematite-altered areas are host to, or spatially associated with, much of the uranium mineralization in both deposits. Similarly, the clay alteration, UEX personnel systematically record hematite alteration intensity during drill core logging, which is recorded as a qualitative range from zero to four; areas of hematite of two or greater are shown in cross-sections in Figure 8.2 and Figure 8.4. Hematization generally comprises fine-grained hematite which replaces mafic sites and, to varying degrees, feldspars in gneiss units and is generally accompanied by weak clay or chlorite alteration. The hematization may be patchy, with alternating intensity, or form a more intense pervasive wash throughout the host rock, imparting a pervasive purple-red tint. As clay alteration is generally not intense in hematized areas, the host rock is generally competent, although hematization can also extend into more intensely clay altered areas, tinting the clays.

In the Horseshoe deposit, hematite alteration forms lenses of generally shallow dipping alteration that occur both above and below the main clay alteration zone in the central and eastern Horseshoe deposit and is most abundant above the clay alteration zone in this area where areas of hematization extend up to 100 m above the clay alteration. In the western Horseshoe deposit, as the clay alteration becomes less planar, hematite occurs as lenses mainly developed in arkosic quartzite that surrounds the clay alteration and which coalesce to a 100 m high by 150 m wide broadly hematized area that lies mainly above the clay alteration zone between sections 4500 N and 4600 N. This broader zone of hematization corresponds with the western end of the Horseshoe A zone, extending eastward where it separates into smaller zones that envelop or are spatially associated with the principal areas of uranium mineralization in the eastern Horseshoe deposit.

Up dip to the northwest, hematization is poorly developed or absent up dip to the northwest, tapering and diminishing upward at the base of the calc-arkose unit along the trace of the Raven syncline, although the associated clay alteration locally continues upward as a thin, potentially fault-controlled band.

Similarly to the hematite-altered areas at Horseshoe, hematite alteration at Raven also occurs peripheral to and surrounding the principal clay alteration zone. Hematization often forms a continuous shell to the clay alteration, enveloping and overlapping with it in a broadly tabular southeast dipping zone, particularly in lower parts of the deposit in the arkosic quartzite and underlying semi-pelitic gneiss/arkosic quartzite units. Areas of hematization widen upward into the quartzite unit, particularly in the hangingwall of the clay alteration zone, broadening upward with a geometry that mimics the folded outline of the quartzite on some sections. Uranium mineralization occurs as lenses within these hematitic areas. Hematite alteration extends upward higher than at Horseshoe and may extend to the current surface on some sections in calc-arkose, corresponding with local near-surface development of uranium mineralization.

Outer Alteration

Distal to clay and hematite alteration, host gneiss units are typically fresh, with mafic minerals preserved. However, within a few metres to tens of metres, mafic minerals (biotite in quartzite and arkosic quartzite, pyroxene, amphibole and biotite in calc-arkose and cal-silicate units) are often chlorite altered and incipient chlorite or clay alteration may affect feldspars. In addition, pyrite and locally pyrrhotite may be present, either as primary disseminated minerals locally associated with mafic mineral grains, or as secondary concentrations locally up to two percent disseminated and as stringers within a few metres of hematite alteration zones. These define an outer reduced envelope to the hematite alteration. Drusy quartz veinlets locally occur peripheral to the clay alteration zones in these areas and may contain pyrite and more rarely chalcopyrite, galena and pyrrhotite.

Mineralogical and Geochemical Patterns in Alteration Zones

During drilling, UEX has systematically collected representative samples, approximately every 5 m, for clay mineral analysis using an infrared analytical spectral device (Terraspec unit). Outside of mineralized or highly altered areas where extensive geochemical sampling was not conducted, 10 cm to 15 cm long core intervals from the Terraspec samples were also sent for multi-element geochemical analysis to form complete cross-sectional geochemical and mineralogical profiles on selected sections through and beyond, the Horseshoe deposit. The data was recently reviewed by Halley (2008), augmenting previous work by the authors, Rhys and Ross (1999) and Quirt (1990). Overall patterns determined are as follows:

- Clay minerals within the core of the clay alteration zones at both Horseshoe and Raven proximal to the centre of the clay plume are dominated by assemblages of pale coloured illite and sudoite (Mg-Fe chlorite), with trace dravitic tourmaline (Quirt, 1990). Pale apple green palygorskite and locally talc or serpentine (lizardite) occur locally in some of the more intense clay zones (Raudsepp, 2007). Hematite is locally present but, as discussed above, is generally peripheral to the main clay zones. Overall, mineral assemblages in the clay Alteration zones are consistent with an oxidized and moderately acidic hydrothermal fluid (Halley, 2008).

- In addition to illite and sudoite, mineralized areas near zones of hematization also contain illite, minor amounts of mixed layer illite-smectite and locally kaolinite or smectite (Quirt, 1990; Rhys and Ross, 1999). Carbonate, replacing plagioclase in extremely altered rocks, is also often associated with mineralization in hematized areas peripheral to the main clay zone (Quirt, 1990).
- A zonation in the spectral infrared absorption signature of illite varying from shorter wavelengths in cores of the clay zones near mineralization to longer wavelengths more distally also supports increasingly acidic conditions in the core of the alteration zones (Halley, 2008).
- Geochemically, the clay alteration zones are associated with Mg and K enrichment of the hosting quartzite and arkosic quartzite units, which may be marked in areas of most intense alteration. In addition, geochemical markers which can aid in the mapping of the alteration zone also include enrichment V, V/Sc ratio and Li, the latter which occurs in sudoite, which track the overall footprint of the oxidized alteration zone at Horseshoe (Halley, 2008). As, Bi and Pb also track the core of the alteration zone around the uranium mineralization but are more proximal to the mineralization itself, while anomalous Cu and Mo occur in some areas of hematization mainly above the mineralization in eastern parts of the Horseshoe deposit (Halley, 2008).
- Outer parts of alteration zones are depleted in Ca and Na, associated with plagioclase alteration and depletion (Halley, 2008).
- Outboard of the clay and hematite alteration zones, peripheral alteration is much weaker and comprises darker green more Fe-rich chlorites than in the core of the alteration zone, which are generally restricted to alteration of primary metamorphic mafic minerals. These more Fe-chlorite rich areas may also contain trace kaolinite and local areas of disseminated pyrite, suggesting that they are reduced.

Note that, although forming above-background pathfinders for prospective clay and hematite alteration, the As, Pb, Cu, Bi, Mo and V concentrations in mineralization and wallrocks may not be sufficiently high to form potential disposal or contamination problems.

The mineralogical and geochemical patterns described above will be utilized by UEX in ongoing exploration of the Horseshoe and Raven deposit area. Their significance in the overall evolution of the deposit and its controls are discussed below.

Faults in Alteration Zones: Potential Controls to Uranium Mineralization

Clay alteration may overprint and be focused along a pre- to syn-mineralization, moderate to steep southeast dipping brittle fault zone, which may run along the central axis of the clay alteration zone. As is discussed in Section 7.2.3 above, evidence of a fault coring the clay alteration zone includes abrupt changes in the thickness of the quartzite unit and difficulty in tracing D2 fold hinges across the clay alteration zone, as well as local occurrence of clay gouge seams and focused clay matrix breccia along the up dip projection of the clay alteration zone at Horseshoe.

However, individual fault strands are often not identifiable in clay alteration zones, which could be due to alteration overprinting in the most intensely altered areas, but in areas of weaker clay alteration where primary textures are visible and the host rock more competent, individual fault strands often cannot be identified along the projected fault trace.

If a continuous fault is present, mineralization and alteration may have occurred late during activity of the fault, or exploited an earlier structure, locally healing earlier fault surfaces.

The interpreted position of a controlling fault to both the Horseshoe and Raven deposits is shown in Figure 8.2 and, based on the position of lithologic thickness changes and discordances, alteration intensity and overall morphology of alteration. A discrete, clearly recognizable fault, however, is often not always identifiable at this position. As discussed by Rhys and Ross (1999), discontinuity of potential fault strands could suggest that the fault zone is comprised of individually discontinuous, but en echelon fault surfaces which collectively define a more continuous fault zone.

Geotechnical Considerations

Although extensive, areas of clay alteration often are not associated with any decreases in core recovery during drilling since, in most areas, framework quartz grains in the quartzite and arkosic quartzite are unaffected and retain rock strength. This is supported by initial geotechnical studies, which include rock quality designation (“RQD”) and point load testing studies. Hence, it is anticipated that only areas of most intense alteration (clay intensity of three or four) where broader zones of more friable alteration may consistently affect rock quality and provide problems to ground support during mine development. The most consistently intensely altered areas lie between the BW and A zones in northeastern portions of the Horseshoe deposit, but do not extend into the more competent mineralization and could be potentially avoided during mining, if done by underground development. Friable areas do occur within some higher grade portions of the A zone, but these are closely restricted to the mineralization and the surrounding wallrocks usually become rapidly fresh and competent adjacent to these areas. The alteration intensity recorded during core logging, in conjunction with core recovery data that has also been captured, may consequently provide important engineering constraints on local ground conditions. Few faults were identified during core logging and no discrete corridors of fault development were recognized, apart from potential faulting along the central axis of the clay alteration zone.

8.2 Uranium Mineralization

Uranium mineralization in both the Horseshoe and Raven deposits occurs mainly within zones of hematite alteration which occur peripheral to the zones of clay alteration. Five principal uranium bearing minerals have been identified in the two deposits by Quirt (1990), DiPrisco (2008) and Ross (2008b). The principal and most abundant uranium bearing mineral is uraninite (variety pitchblende - UO_2), which is also generally the paragenetically earliest uranium mineral.

Secondary uranium minerals, which are generally formed here by alteration and remobilization of uranium in uraninite, are comprised of the yellow-green coloured uranium silicates boltwoodite HK $(UO_2)(SiO_4)$-$1.5H_2O$ and uranophane $Ca[(UO_2)SiO_3(OH)]_2$-$2H_2O$, which are locally accompanied by coffinite $U(SiO_4)_{1-x}(OH)_{4x}$ and minor amounts of carnotite $K_2\ (UO_2)_2V_2O_8$-$3H_2O$ and possibly autunite $[Ca(UO_2)(PO_4)(H_2O)_{10-12}]$. There are locally other complex, unidentified U-minerals present, but these are volumetrically minor.

Nickel arsenide and cobalt minerals, which are typically associated with unconformity uranium deposits that occur at the base of the Athabasca sandstone (Type A) are absent at Horseshoe and Raven and the relatively simple pitchblende dominant metallic mineral assemblage at the deposits is typical of other basement-hosted uranium deposits in the region, such as Eagle Point (Quirt, 1990).

Uranium mineralization within mineralized zones occurs with three dominant gradational variations in style, which may either occur together, or occur as the only style within individual drilling intercepts or mineralized lenses:

- a*)* *Disseminated pitchblende-dominant mineralization:* Typically occurring in competent, hematite-rich arkosic quartzite, this style comprises disseminated pitchblende and coffinite grains which replace mafic sites and with increasing abundance, feldspar sites. Chlorite dominant varieties of this alteration may also occur locally, where, instead of hematite, dark green chlorite occurs in the same habit, probably reflecting local variations to more reduced conditions or overprinting alteration. In disseminated mineralization, pitchblende may occur as individual disseminated grains or aggregates, often intergrown with hematite, clays and chlorite. Much of the BE subzone, A2 to A4 subzones and parts of the BW subzones at Horseshoe are composed of this style of mineralization, which is often associated with broad zones of consistent 0.1% to 0.3% U_3O_8 grade that comprise some of the thickest drill intercepts in the Horseshoe deposit. Higher grade areas of this style may also have disseminated boltwoodite and uranophane.
- b) *"Nodular "or redox front style mineralization:* Highest grade of mineralization in both deposits typically occur in this mineralization style, which comprises much of the A1H and A2 subzones at Horseshoe and higher grade portions of the Raven deposit.

  This mineralization typically comprises pervasively disseminated nodules, blebs and lenses of pitchblende which occur either disseminated or as lenses through bands of hematite, or as uraniferous envelopes to lenses and bands of red to pinkish hematite + clay alteration. In the latter case, the mineralization may form along redox fronts, extending outward from the hematite as pervasive grey, fine-grained pitchblende mineralization which diminishes in intensity a few cm from the hematite bands. In some wider drilling intercepts which contain this mineralization style, hematitic bands with associated higher grade uranium mineralization that may be a few tens of centimetres to a few m thick may be separated by several m of relatively unaltered or weakly altered, locally pyrite-bearing wall rock, from additional uraniferous hematite bands, defining alternating high and low grade intervals.

In highest grade areas, where mineralization occurs in hematite, nodules and coarse anhedral clots of dull grey to black U-minerals (pitchblende +/- coffinite) may be present. These clots are often associated with small-scale reduction spots that surround the clots and distinctive pink (hematite) and yellow (uranophane) alteration. Fine-grained U-minerals also occur in micro-fractures within quartz grains (DiPrisco, 2008; Ross 2008b) and interstitial to or intergrown with clays where more pervasively disseminated as envelopes to hematite bands.

Secondary U-minerals, principally uranophane and boltwoodite, are most abundant in higher grade portions of the nodular mineralization and result in characteristic yellow alteration seen in this mineralization style, occurring as irregular veinlets, or disseminated pervasively, often surrounding pitchblende clots, or replacing it in the groundmass. A characteristic pale pinkish colour of oxidized clay altered domains in high grade portions of the nodular mineralized areas at Horseshoe is due to hematite, or more amorphous Fe-hydroxides (Ross, 2008b).

- c) *Veinlet mineralization*: Pitchblende bearing veinlets are locally developed in both deposits. These are most abundant where mineralization is developed in competent, but variably (patchy) hematite altered quartzite. The difference in style with respect to other lithologies probably reflects the more rheologically competent and less permeable nature of the quartzite, which is less susceptible to secondary permeability associated with alteration than other lithologies that contain more disseminated styles (e.g. as seen in the more easily altered arkosic quartzite). Pitchblende veinlets (fracture fillings) in quartzite may occur spaced a few centimetres to tens of centimetres apart and comprises stringers usually less than 3 mm thick of patchy pitchblende + chlorite +/- clay. They generally cut across dominant gneissosity at high angles. Fine-grained disseminated pitchblende may occur interstitial to quartz grains in veinlet envelopes. They may have bleached envelopes in otherwise hematite-altered quartzite. Thicker pitchblende veinlets up to 2 cm thick which are discordant to foliation also occur and were mainly observed at Raven, where they form irregular chains of pitchblende grains and aggregates, often with yellow uranium silicates.

In all mineralization styles, in addition to the coarser-grained U-minerals, primary uraninite often occurs in networks of thin fractures that occur in quartz grains, whereas secondary uranium bearing minerals form tight intergrowths with hydrothermal alteration assemblages that have overprinted the matrix of the host rock (DiPrisco, 2007). In areas of the hematite-rich alteration, aggregates of secondary uranium minerals are intergrown predominately with Fe-oxi-hydroxides and form medium- to very coarse-grained aggregates. Local replacement of micas in the matrix has resulted in extremely fine-grained textures of secondary uranium minerals tightly intergrown with chlorite and Fe-oxi-hydroxides. U-minerals (mainly pitchblende and coffinite) also locally rim sulphide minerals that may occur in fractures or disseminated in the altered groundmass, in both disseminated and nodular textured mineralization (Ross, 2008b). Sulphide content is generally low, typically less than two percent even in high grade samples, consisting dominantly of pyrite, pyrrhotite and locally chalcopyrite, occurring in micro-fractures and disseminated in the mica/clay minerals. Galena and chalcopyrite are also present in trace amounts in micro-fractures and in amorphous U-mineral clots in nodular mineralization.

Precipitation of uranium mineralization may have been directly coupled with hematite formation (Quirt, 1990), occurring at a deposit scale in redox fronts with the mineralization located at the interface between oxidized fluid channel ways in clay alteration zones with illite-sudoite dominant alteration and surrounding reduced wall rock which contains sulphide-bearing assemblages. These patterns also repeat at the local scale; in areas of higher grade nodular style mineralization, the alternating hematite-related higher grade mineralization alternates with adjacent reduced fresher wallrocks, with mineralization often forming higher grade seams at the redox transition.

The deposit scale occurrence of mineralization in hematized fronts surrounding oxidized fluid channel ways is reminiscent in style to the geometry of roll front uranium deposits.

8.3 Horseshoe Deposit: Distribution of Uranium Mineralization

The Horseshoe deposit is of a higher grade than Raven, by contained uranium, and is the larger of the two deposits. Drilling conducted by UEX has defined continuous mineralization in the deposit over a strike length of approximately 800 m. Throughout this area, mineralization occurs in several stacked, linear and shallow dipping, east-northeast plunging zones which follow and are developed peripheral to the main northeast trending, southeast dipping clay alteration zone that passes continuously through and between the deposits. The largest zones of mineralization at Horseshoe occur at depths of between 120 m and 450 m below surface. Mineralization depths increase as the deposit plunges to the northeast, ranging in vertical depth below surface from 130 m to 220 m in the southwestern parts of the A subzone between sections 4540-4650N, to depths of 250 m to 450 m below surface along sections 4690 N4750N. The principal subzones in the southwestern portions of the deposit, the S2, S3 and B West subzones occur at depths of 120 m to 230 m below surface. Principal mineralized subzones at Horseshoe are planar to lenticular in cross-section and in plan view generally elongate in an east-northeast trend (Figure 8.4 and Figure 8.2). The report of Rhys et al. (2008) contains a more comprehensive set of sections through the Horseshoe deposit.

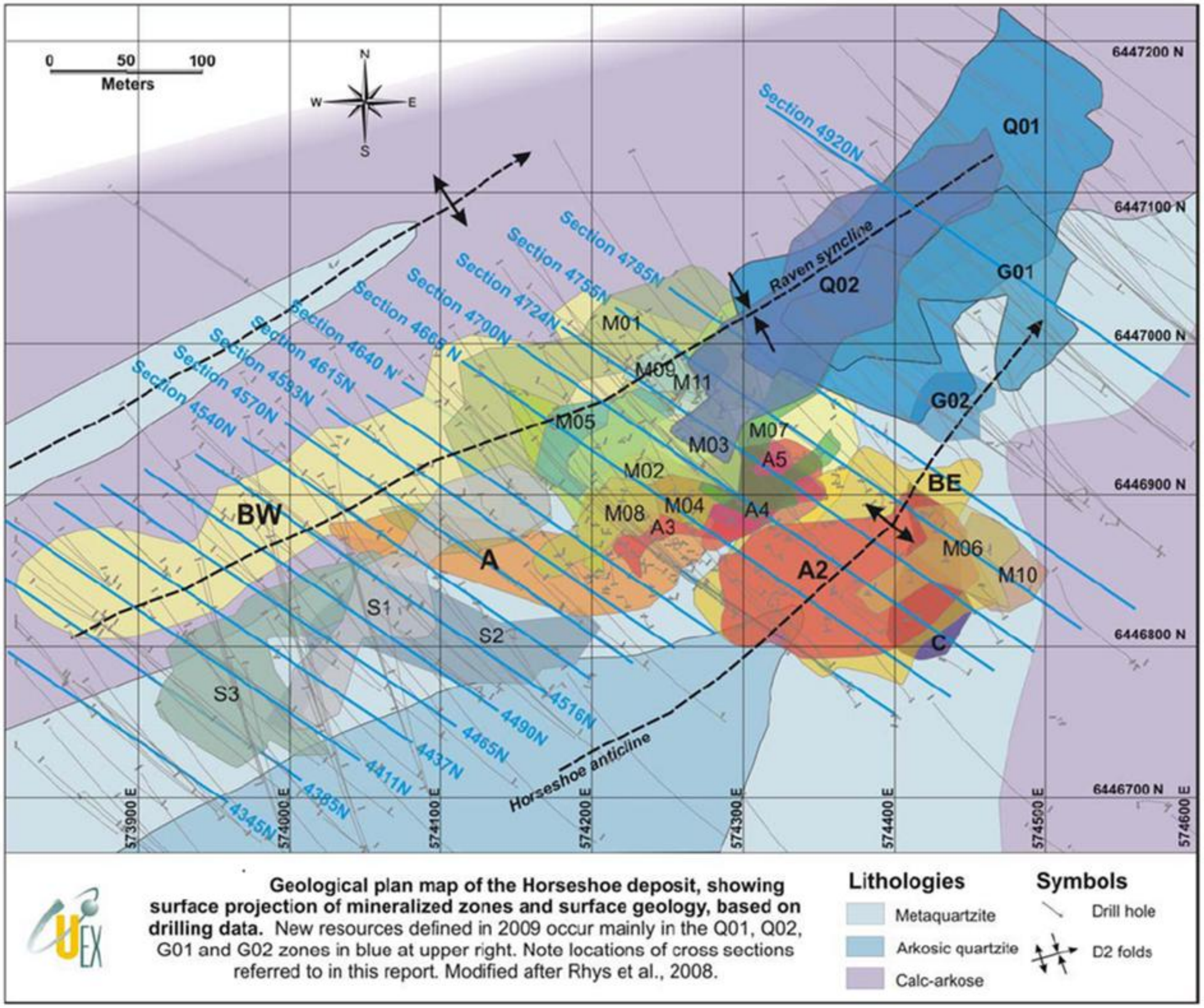


Figure 8.1: Horseshoe Deposit Plan Showing Mineralized Subzones

8.3.1 Geometry and Distribution of Mineralization across the Horseshoe Deposit

The geometry and extent of mineralized zones varies across the Horseshoe deposit. In the western parts of the deposit, between sections 4385 N where mineralization first commences and section 4540 N, mineralization occurs in a series of lenses that are developed mainly in arkosic quartzite within approximately 80 m of the overlying quartzite contact. Several lenses which occur here mimic the geometry of the folded arkosic quartzite unit in the core of the Raven syncline, varying in dip from shallow to the southeast to shallow northwest dipping and surrounding an irregular lobe of clay alteration. Where clay alteration can be traced to depth, it is steeply southeast dipping in this area suggesting that any controlling structure here may dip steeply along the clay alteration zone. This western part of the Horseshoe deposit is comparable in style to the mineralization distribution and setting seen through much of the Raven deposit.

Morphology and extent of the Horseshoe mineralization begins to change between sections 4540 N and 4640 N. In this transitional area, the clay alteration zone associated with mineralization becomes increasingly more focused and tabular and increasingly shallowly dipping. The mineralized zones which dip to the northwest in western parts of the deposit (the S2 and S3 zones) dissipate and mineralized lenses become more consistently shallow southeast-dipping parallel to, or slightly shallower in dip than, the clay alteration zone.

Mineralization occurs both on the fringes above and below the clay alteration zone. It is in this transitional area between the western and eastern parts of the Horseshoe deposit that the A subzones are best developed, occurring above the clay alteration zone and contain the highest grade, with well developed nodular style mineralization.

Central-eastern parts of the Horseshoe deposit, southwest of the Q and G subzones, contain the widest, most extensive and most abundant zones of mineralization. This area coincides with the well developed planar and shallow southeast dipping nature of the clay alteration zone, which cuts obliquely across the folded gneiss sequence. Mineralization occurs in multiple shallow southeast dipping to subhorizontal lenses of mineralization that are developed mainly within 100 m of the hangingwall of the clay alteration zone, but also below it in the B West (“BW”) and C subzones. As with other parts of the deposit, the dominant host rock is arkosic quartzite.

The longer dip length of the mineralized subzones in the eastern part of the Horseshoe deposit results in an overall bend in the dominant trend of the deposit in plan view in that area. The mineralization in the Q and G subzones reappears after a small gap, on the far northeast, forming a southwest plunging zone of mineralization which rises toward surface in the northeast.

The overall changes in mineralization distribution across the deposit may correspond with increasing structural control and intensity of pre-mineralization controlling faulting along the clay alteration zone, as well as an overall shallowing of the controlling clay/fault zone.

This change in orientation could reflect interaction with the nearby steeply dipping and northerly trending Dragon Lake Fault, which lies just to the southeast of sections 4682 E to 4755 E and which has been intersected by recent drilling in that area. The Dragon Lake Fault is enveloped by a broad clay-hematite alteration zone into which the main Horseshoe zone of alteration and potential faulting merges.

In addition to the close spacing of drill holes which support the shallow dipping orientations of mineralized subzones and higher grade within them, shown in Figure 8.2, an additional verification of the morphology of mineralization is the high core axis angles of banded hematite/pitchblende mineralization in higher grade areas, such as in the A subzone. In these areas, banded mineralization also often cuts across the folded, steeply dipping gneissosity at a high angle. The broad coincidence of hematite alteration and its often high concentration with mineralization also displays similar patterns to the mineralization when modelled, providing an additional geological parameter to support the interpreted distribution of mineralization. These patterns suggest that the vertical to steep orientations of most diamond drill holes cross the shallow-dipping mineralized subzones at a high angle, which is close to true thickness.

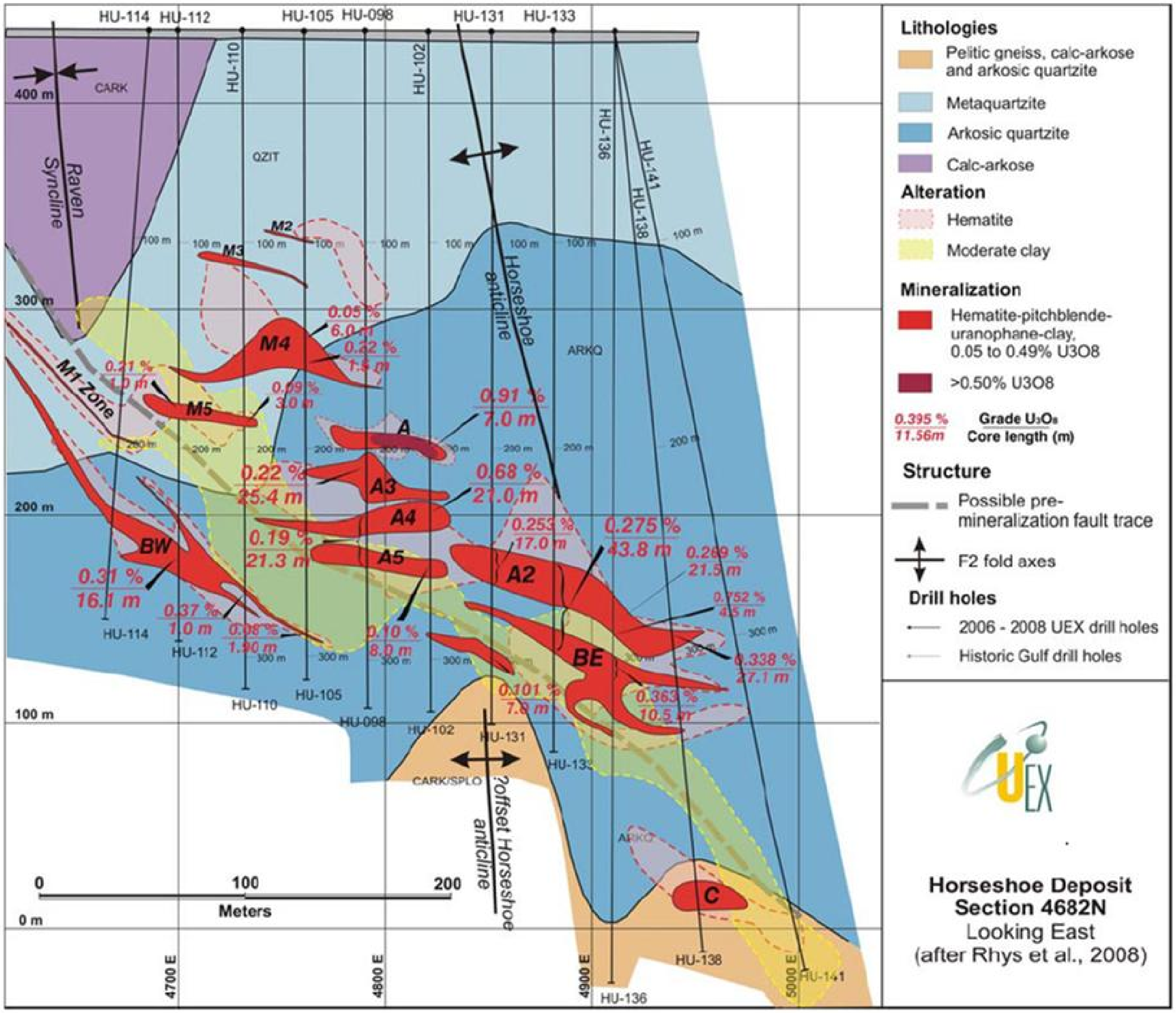


Figure 8.2: Horseshoe Deposit Section 4920N – Looking East

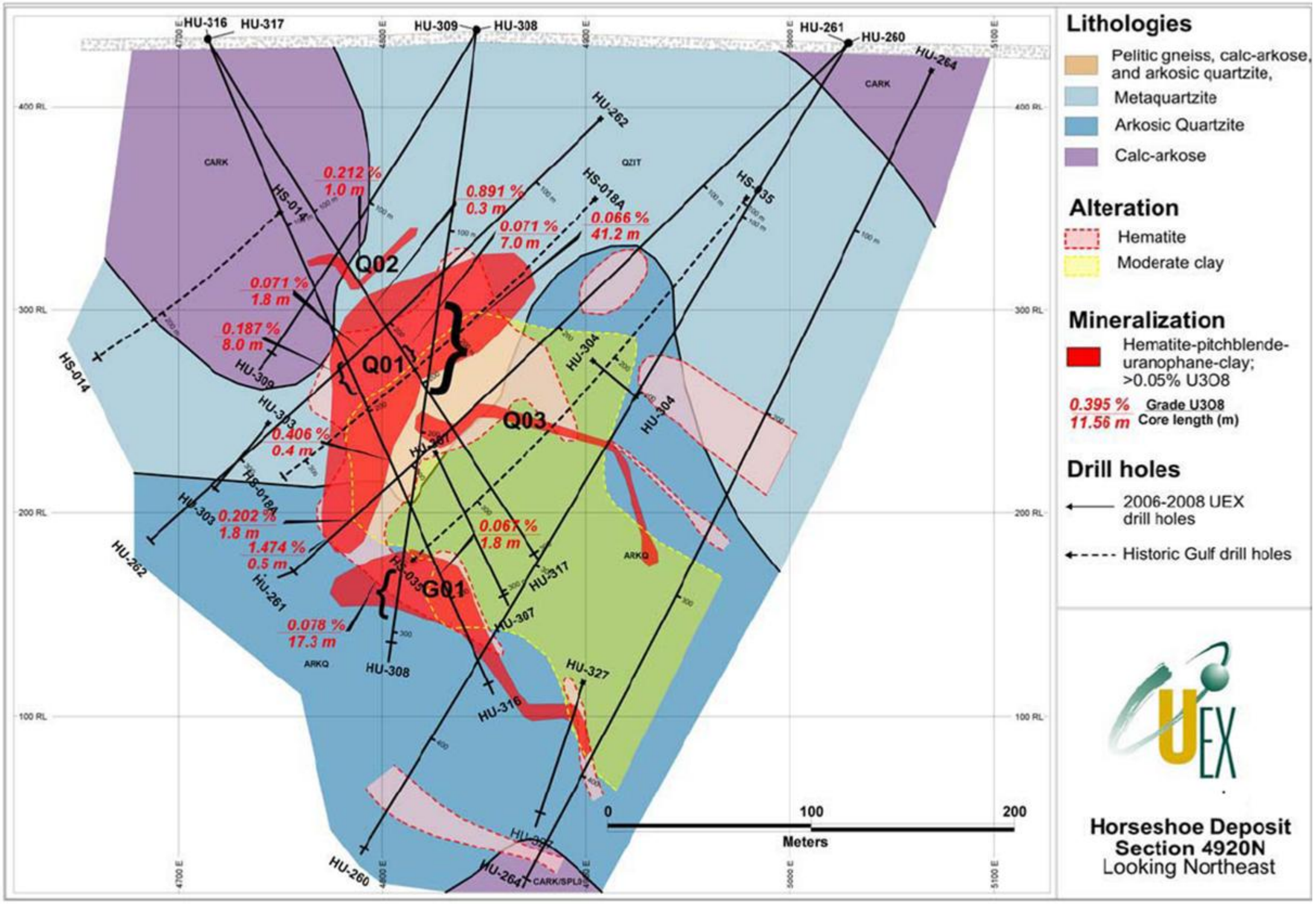


Figure 8.3: Horseshoe Deposit Section 4682N, Looking East

Drilling has bounded the mineralized zones, shown in Figure 8.4 and summarized below. At the eastern end of the deposit, the main mineralized zones defined by drilling terminate at section 4785 N, but historic Gulf drilling indicates that additional mineralization in separate zones is also present to the northeast, which is currently being drill defined.

Principal Mineralized Zones at the Horseshoe Deposit

Wireframe modelling of the Horseshoe deposit has defined twenty-eight individual mineralized subzones, which have been utilized in the Horseshoe resource estimation. The dimensions of these are summarized in Table 8.1. Principal subzones in the Horseshoe deposit are as follows:

a) *The A subzone*: Occurring in central parts of the deposit at depths of 120-180 m below surface above the clay alteration zone, this is the highest grade of the Horseshoe zones, being composed mainly of the higher grade nodular style mineralization. Mineralization is best developed along the southeasterly margin of the zone where it locally rolls from a shallow to a steeper southeasterly dip. A best intersection of 4.54% U_3O_8 over 12.35 m was obtained in this area in hole HU-016. Two or more stacked high grade shallow dipping mineralized lenses can occur internally within the A zone. The A subzone was separated into the A1 and A1H (high grade) subzones for the mineral resource modelling process.

b) *The A2 subzone*: This shallow dipping subzone lies just beneath the northeastern projection of the A zone. This subzone also contains a significant portion of nodular style mineralization.

c) *The B West ("BW") subzone*: This is by volume the largest and most laterally extensive of the mineralized subzones at Horseshoe. Unlike most other subzones, it occurs beneath the clay alteration zone, dipping moderately to shallowly southeast, generally parallel to and immediately below the clay alteration. This subzone is traceable across the entire strike length of the Horseshoe deposit from southwest to northeast. BW is thickest to the northeast, where drill intercepts locally exceed 30 m at grades of 0.5% to 0.6% U_3O_8. Additional parallel, minor subzones may lie above the main BW zone and extend upward into quartzite (e.g. M1 subzone).

d) *The B East ("BE") subzone*: Occurring across (above) the clay alteration zone from the BW zone, this zone is locally linked to it to the east. This is an often thick zone (up to 40 m), which is dominated by the disseminated style of mineralization. BE straddles and often extends above the clay alteration zone and is shallower dipping than the associated clay alteration zone.

e) *The C subzone:* This is the deepest subzone intersected at Horseshoe, lying beneath the clay alteration zone at depths of 420 m to 460 m. It is volumetrically small, but locally contains higher grade intercepts of the nodular style (e.g. hole HU-065, 0.61% U_3O_8 over 17.65 m: intercept on section 4700 N, not shown).

f) *The S subzones*: These subzones form the principal mineralization in western parts of the Horseshoe deposit, which locally exhibit the synclinal morphology of the hosting arkosic quartzite unit. They gradually dissipate where the A subzone begins, between sections 4540 E and4593E.

g) *The A3 to A5 subzones*: These comprise a series of stacked, shallow dipping zones of mixed disseminated and nodular style which occur immediately beneath the northeast end of the A subzone (Figure 8.2).

h) *The M subzones*: Designated M for minor, some of these subsequently were determined to have significant tonnage. These are mainly miscellaneous subzones, most of which are small, that lie above and are separate from the A and B-series subzones in quartzite and arkosic quartzite. The largest, the M01 subzone, is closely spatially associated with the BW zone, occurring immediately above and parallel to that zone over much of its strike length, although often on the opposite side of the clay alteration zone. Other minor zones are developed in quartzite, or occur above the BE zone in arkosic quartzite, where plumes and lenses of hematite alteration extend well above the clay alteration zones. Veinlet and disseminated mineralization styles dominate these subzones.

i) *The G subzone*: Mineralization in the G01 and spatially associated G02 zones occurs in the Horseshoe northeast area, and represents a newly defined portion of the Horseshoe deposit which was not part of previous resources estimates.

The G01 zone lies several tens of m northeast of the BE zone but at a similar position and elevation with respect to the BE mineralization. Highest grade and deepest western parts of the mineralization form a shallow dipping lens, which rapidly steepens to a steep southeast dipping lens to the northeast, extending upward toward mineralization in the Q zones with an overall southerly plunge.

j) *The Q subzone*: The most north-easterly of the mineralized zones at Horseshoe, this is a broad, low grade zone developed in quartzite and underlying arkosic quartzite. The zone has an overall southerly plunge, extending from near surface in the northeast downward toward the G01 zone to the south, where it extends downward from the quartzite into the underlying more arkosic unit.

Table 8.1: Lateral and Down Dip Dimensions and Contained Volume of Mineralized Zones in the Horseshoe Deposit based on Wireframe Modelling of Mineralization

Subzone	Lateral Strike Continuity (m)	Average Dip Length (m)	Volume (m^3)
A	331	55	155,579
A2	170	94	122,697
A3	147	52	41,748
A4	143	48	23,356
A5	161	41	26,582
BW	569	87	535,852
BE	212	127	292,200
C	120	44	42,759
S1	228	50	50,634
S2	240	36	62,249
S3	183	66	79,924
M01	284	81	75,639
M02	90	40	9,245
M03	162	50	21,502
M04	100	118	39,060
M05	160	42	10,158
M06	110	46	17,465
M07	124	22	20,682
M08	90	27	5,680
M09	59	43	3,085
M10	47	68	6,227
M11	57	23	2,131
G01	229	191	449,240
G02	57	77	66,307
Q01	292	164	809,830
Q02	279	83	41,186
Q03	197	95	37,573

The Wireframe model was generated by UEX and has been utilized for the Horseshoe Mineral Resource Estimate

8.4 Raven Deposit: Distribution and Style of Uranium Mineralization

The Raven deposit has been defined since 2005, by drilling for and by UEX, over a strike length of approximately 910 m (Figure 8.4).

Mineralization is developed mainly at consistent depths of between 100 m and 300 m below surface and exhibits no significant plunge, unlike Horseshoe, defining an overall strongly elongate and east-northeast trending zone of mineralization. Minor zones may extend upward to within a few tens of m of surface, but these are not consistently present along the length of the deposit as it is currently defined by drilling. Mineralization is localized along the trace of the Raven syncline, particularly along the southeastern limb of the fold, and is developed extending downward from the base of the folded calc-arkose unit into the underlying quartzite and arkosic quartzite.

Similar to Horseshoe, mineralization at Raven occurs in hematitic altered areas which surround a steep to moderate southeast dipping zone of clay alteration which obliquely crosses the southeastern, dominantly shallow northwest dipping limb of the Raven syncline. Structural position of the mineralization is consequently the same as Horseshoe with respect to the folded metamorphic stratigraphy. The clay alteration zone also shallows in dip to the east through the deposit, although it does not attain the shallow dips of the eastern Horseshoe clay alteration zone.

It may also be controlled by pre- or syn-alteration/mineralization faulting, as evidenced by clay gouge seams up dip from the projection of the principal clay zone. Potential for offset lithologies across the clay zone at Raven is not as pronounced as it is at Horseshoe, with lithologic contacts often showing little or no significant deflection across the trace of the clay zone.

The distribution of mineralization at Raven is more complex in morphology than that observed in the current areas of definition drilling at Horseshoe. In general, there are two general zones of mineralization at Raven, a Lower and an Upper zone (Table 8.2), each of which may be split into subzones. The largest of each of these zones are termed L01 and U01. The L01 Lower subzone extends through the entire defined strike length of the Raven deposit, while the main U01 Upper subzone is best developed in the central portions of the deposit. The U01 Upper zone extends eastward and splits into multiple zones, while dissipating to the southwest.

Table 8.2: Lateral and Down Dip Dimensions and Contained Volume of Mineralized Zones in the Raven Deposit based on Wireframe Modelling of Mineralization

Subzone	Lateral Strike Continuity (m)	Average Dip Length (m)	Volume (m^3)
L01	913	188	2,074,548
L02	215	108	61,905
L03	109	47	7,727
L04	215	79	77,755
L05	67	50	2,294
L06	167	121	32,263
U01	610	140	1,448,800
U02	152	47	44,269
U03	224	85	153,642
U04	116	66	27,838
U05	239	66	55,468
U06	43	47	11,258
U07	49	86	18,399
U08	99	56	31,161
U09	144	56	33,483
U10	443	133	755,247

The Raven L01 Lower subzone generally comprises a tabular, steep to moderate southeast dipping zone of mineralization which occurs along the footwall of, and parallel to the clay alteration zone over vertical dip lengths of 100 m to 200 m. On most sections, it commences in quartzite and passes downward across arkosic quartzite into the upper portions of the mixed semi-pelitic gneiss/calc-arkose sequence. The L01 subzone may occur over widths up to 20 m, but is generally a few m wide, with grades typically between 0.05% and 0.15% U_3O_8 comprised mainly of disseminated and stringer styles of mineralization.

The Raven Upper zone is more complex in geometry. It forms one or more shallow dipping lobes at depths typically between 100 m to 220 m below surface which straddle the quartzite unit, extending both into basal portions of the calc-arkose unit and the upper parts of the underlying calc-arkose. It occurs in the hangingwall of the clay alteration zone. Mineralization is highly variable in grade, with the highest grades occurring between sections 5330E and 5500E in the thickest and most extensive parts of the U01 zone, and between 5630E and 5665E where it splits into multiple zones. In these areas, nodular and veinlet styles of mineralization are locally developed, forming probably sinuous alteration fronts and associated pitchblende +/- U-silicate veinlets that lie along zones of hematization. Multiple sub-zones are developed that are often close enough to model together at various cut-offs and may have complex outlines. Like western parts of the Horseshoe deposit, pods of mineralization in the Raven Upper zone on many sections are approximately stratabound, and therefore vary in orientation around the hinge of the Raven syncline, locally resulting in an overall synclinal form to the mineralization on some sections.

In some areas in the central Raven deposit, the Upper zone may extend downward in two or more lobes which nearly link to the Lower zone below, thus defining an upward widening, semi-circular pattern which in upper portions wraps around and encloses the upper parts of the clay alteration zone. This crudely semi-circular upward facing outline to the mineralization may have represented a large scale upward facing redox front, along which at the leading edge hematization and uranium mineralization may have developed if the front remained stationary for sufficient periods. Internal complexities of mineralization in the U01 Upper zone may have resulted from various advances and retreats of the leading edge of the front, resulting in local overprinting, and variable areas of mineralization depletion and enrichment.

The more complex geometry of the Raven mineralization relative to that seen at Horseshoe may reflect additional factors, including the occurrence of mineralization over a broader range of lithologies that may have influenced mineralization distribution. Lithologic units are thinner here than at Horseshoe, where much of the mineralization is hosted by the substantially thicker arkosic quartzite unit. The steeper dip of the clay zone and potential controlling fault may also have contributed to these patterns, since at Horseshoe the shallower fault dips coincide with more consistent mineralization outlines, while in western parts of that deposit where the clay alteration/fault is steeper, lithologic control becomes increasingly important in influencing the position and orientation of mineralization, as is seen at Raven.

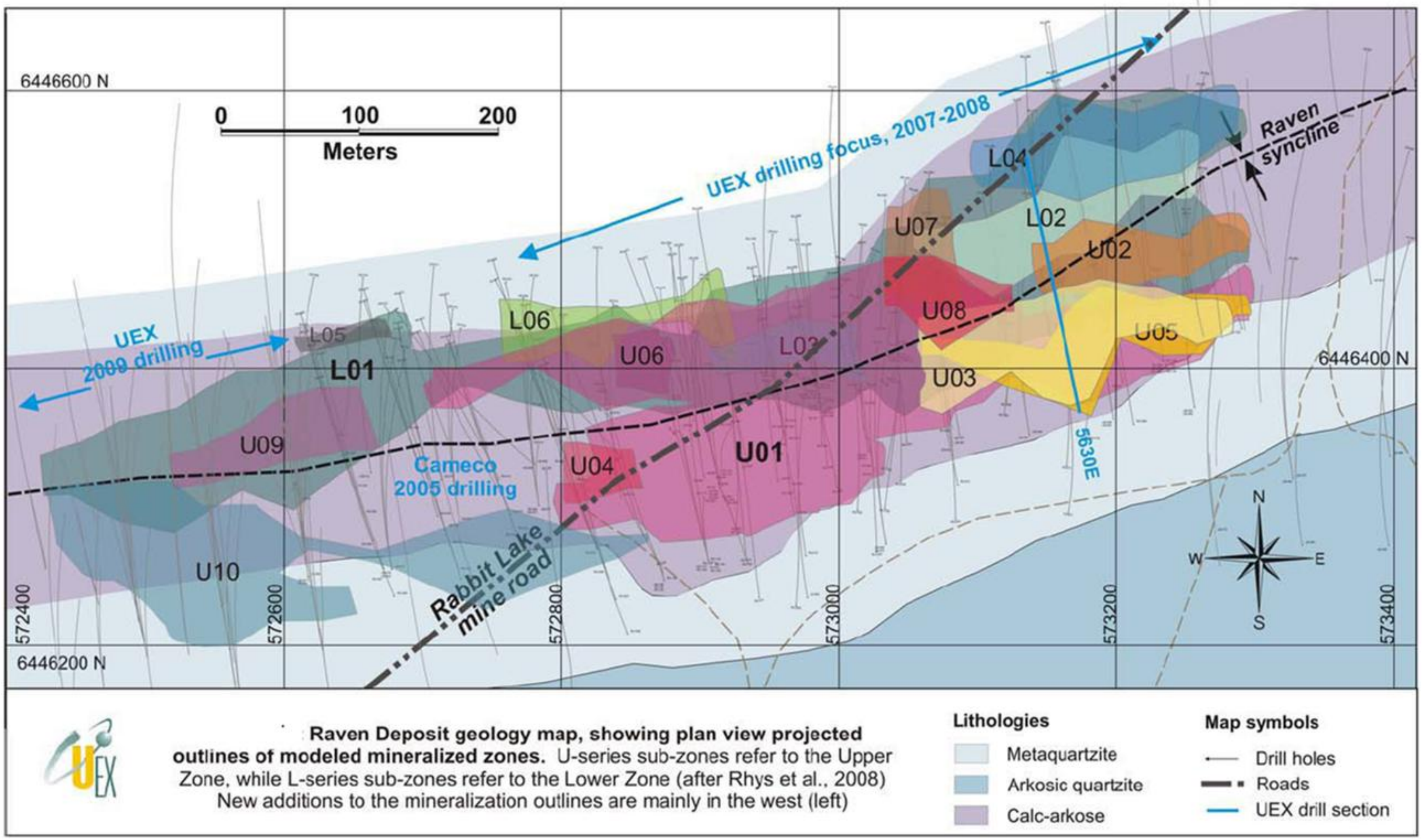


Figure 8.4: Raven Deposit Showing Mineralized Subzones

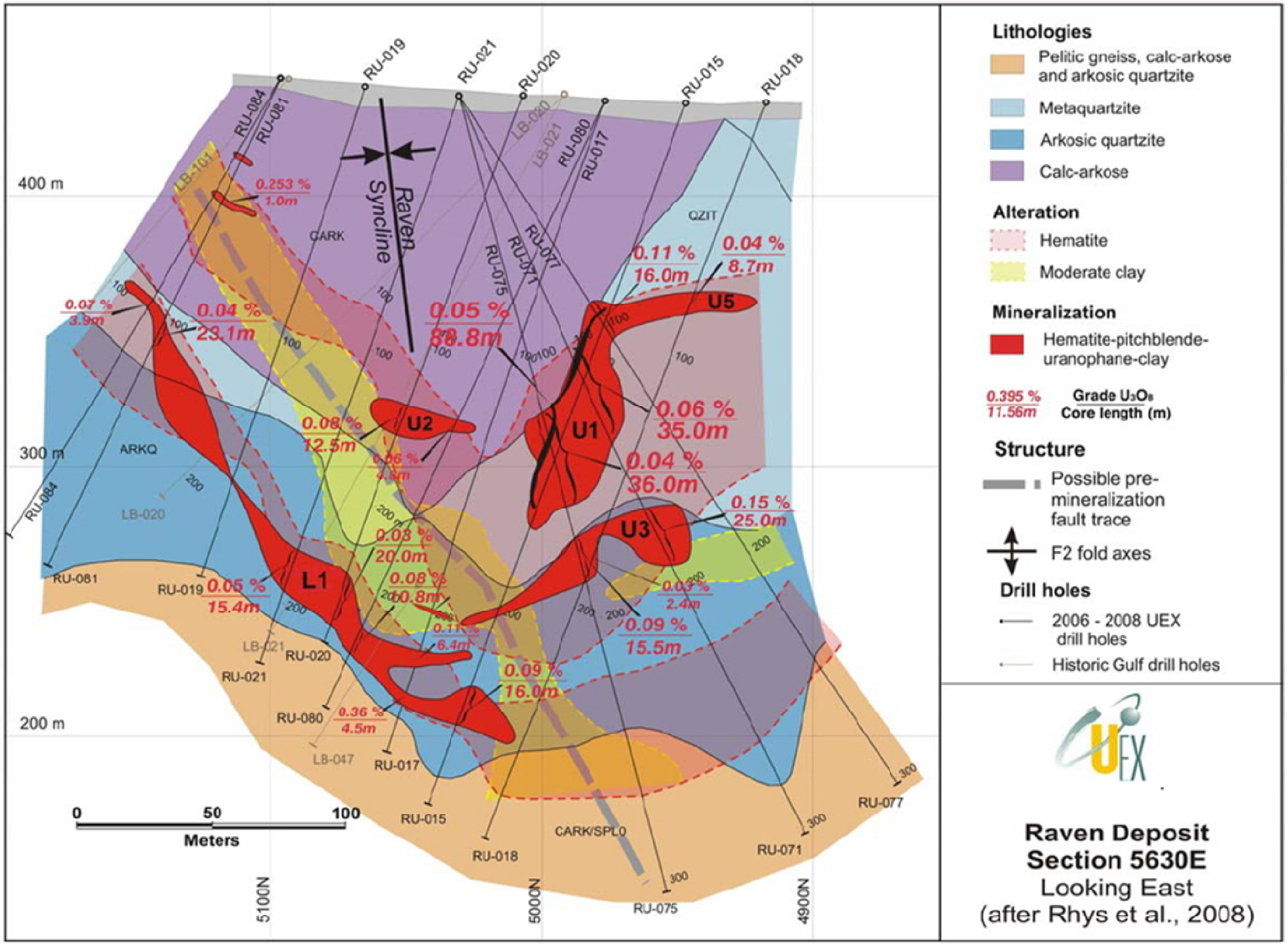


Figure 8.5: Raven Deposit Section 5630E Looking East

Similarly to Horseshoe, mineralization at Raven occurs in hematitic altered areas which surround a steep to moderate southeast dipping zone of clay alteration which obliquely crosses the southeastern, dominantly shallow northwest dipping limb of the Raven syncline. Structural position of the mineralization is consequently the same as Horseshoe with respect to the folded metamorphic stratigraphy. The clay alteration zone also shallows in dip to the eastward through the deposit, although the alteration does not attain the shallow dips of the eastern Horseshoe clay alteration zone. This alteration may also be controlled by pre- or syn-alteration/mineralization faulting, evidence for which includes clay gouge seams up dip from the projection of the principal clay zone. Potential for offset lithologies across the clay zone at Raven is not as pronounced as it is at Horseshoe, with lithologic contacts often showing little or no significant deflection across the trace of the clay zone.

The distribution of mineralization at Raven is more complex in morphology than that observed in the current areas of definition drilling at Horseshoe. In general, there are two general zones of mineralization at Raven, a Lower and an Upper zone, each of which may split into subzones (L- and U- zones in Figure 8.4; largest of each of these subzones are termed L01 and U01). The L01 Lower subzone extends through the entire defined strike length of the Raven deposit, while the main U01 Upper subzone pod is best developed in central portions of the deposit, extending eastward and splitting into multiple zones and dissipating to the southwest.

The Raven Lower zone generally comprises a tabular, steep to moderate southeast dipping zone of mineralization which occurs along the footwall of and parallel to the clay alteration zone over vertical dip lengths of 100 m to 200 m. On most sections, it commences in quartzite and passes downward across arkosic quartzite into the upper portions of the mixed semi-pelitic gneiss/calc-arkose sequence.

The Lower zone may occur over widths up to 20 m, but is generally a few m wide, with grades typically between 0.015% and 0.05% U_3O_8 and consisting of mainly disseminated and stringer mineralization styles.

The Raven Upper zone is more complex in geometry. This zone forms one or more shallow dipping lobes at depths typically between 100 m to 220 m below surface which straddle the quartzite unit, extending both into basal portions of the calc-arkose unit and upper parts of the underlying calc-arkose and occurring in the hangingwall of the clay alteration zone.

Mineralization in the Upper zone is highly variable in grade, with highest grades occurring between sections 5330 E and 5500 E in the thickest and most extensive parts of the U01 zone and between 5630 E and 5665E where it splits into multiple zones. In these areas, nodular and veinlet styles of mineralization are locally developed, forming probably sinuous alteration fronts and associated pitchblende +/- U-silicate veinlets that lie along zones of hematization. Multiple subzones are developed and are often close enough to be joined, which may result in complex outlines. Similarly to the western parts of the Horseshoe deposit, pods of mineralization in the Raven Upper zone on many sections are approximately stratabound and vary in orientation around the hinge of the Raven syncline, locally resulting in an overall synclinal form to the mineralization on some sections.

In some areas in the central Raven deposit, the Upper zone may extend downward in two or more lobes which nearly link to the Lower zone below, defining an upward widening, semi-circular patterns which in upper portions wraps around and encloses the upper parts of the clay alteration zone. This crudely semi-circular upward facing outline to the mineralization may have represented a large scale upward facing redox front, along which at the leading edge hematization and uranium mineralization may have developed if the front remained stationary for sufficient periods. Internal complexities of mineralization in the U01 subzone may have resulted from various advances and retreats of the leading edge of the front, resulting in local overprinting and variable areas of mineralization depletion and enrichment.

The more complex geometry of the Raven mineralization relative to that seen at Horseshoe, may be reflective also of additional factors, including the occurrence of mineralization over a broader range of lithologies that may have influenced mineralization distribution. Lithologic units are thinner here than at Horseshoe, where much of the mineralization is hosted by the substantially thicker arkosic quartzite unit. The steeper dip of the clay zone and potential controlling fault may also have contributed to these pattern, since, at Horseshoe, the shallower fault dips coincide with more consistent mineralization outlines, while in western parts of that deposit where the clay alteration/fault is steeper, lithologic control becomes increasingly important in influencing the position and orientation of mineralization, as is seen at Raven.

9 Exploration

The following section was taken directly from UEX's November 12, 2008 N.I. 43-101 report entitled "Technical Report on the Geology of, and Drilling Results from, the Horseshoe and Raven Uranium Deposits, Hidden Bay Property, Northern Saskatchewan" by Rhys et al. (2008).

Minor changes have been made and comments inserted where appropriate.

Exploration conducted on the Hidden Bay property by UEX as operator and between 2002 and 2005 for UEX by Cameco under the exploration management service agreement has comprised mainly diamond drilling and various geophysical surveys. Diamond drilling in the Horseshoe and Raven area during these periods, which is where by far the bulk of drilling was conducted on the Hidden Bay property, is documented in Sections 10.1.

Other forms of exploration conducted by, or on behalf, of UEX include several types of ground and airborne geophysical surveys, which are summarized below and ground geochemical (soil) surveys, using conventional and partial extraction (MMI) techniques, reconnaissance surveys which were conducted to the south of the Horseshoe and Raven deposits and to the northwest in the Vixen Lake area (Kos, 2004).

Geophysics in the Horseshoe and Raven Deposit Area

Several airborne and ground geophysical surveys that have been conducted since UEX acquired the Hidden Bay property cover all or parts of the Horseshoe and Raven deposit areas. These include:

- a) VTEM airborne electromagnetic surveys which were conducted between 2004 and 2006 over most of the property area by Geotech Ltd. of Aurora, Ontario (Irvine, 2004; Cristall, 2005; Witherly, 2007; Cameron and Eriks, 2008b), and which cover the Horseshoe and Raven areas.
- b) Airborne radiometric and magnetic surveys were conducted in June 2008 by Geo Data Solutions Inc. of Laval, Quebec, which cover much of the Hidden Bay property. More detailed, northwest trending and 50 m spaced flight lines were conducted over the Horseshoe and Raven deposit areas to aid in the identification of magnetic and radiometric patterns that could reflect both near-surface projection of mineralization and/or prospective faults potentially hosting mineralization. Full interpretation of this survey is underway and targets will be integrated into the UEX exploration program when complete.
- c) A RESOLVE airborne electromagnetic and magnetic survey was conducted over selected parts of the property by Fugro Airborne Surveys Corporation of Mississauga, Ontario, including Horseshoe-Raven and West Bear, during 2005 (Cameron and Eriks, 2008a). This outlined in particular the distribution of folded graphitic gneiss, which occurs to the southwest of the Raven deposit, and which could focus faulting that may control uranium mineralization.
- d) A widely spaced ground EM (Moving Loop) survey was conducted across the Horseshoe and Raven area in February – March 2002 by Quantec Geoscience Inc. of Porcupine, Ontario

(Goldak and Powell, 2003). Like the RESOLVE survey, this identified EM targets in the local area mainly associated with graphitic gneiss to the south and west outside of the immediate area of the deposits. One hole was drilled at Raven in 2002 to test whether the folded graphitic gneiss unit was present below the Raven deposit where it might act as a reductant to focusing mineralization along the steeply dipping clay alteration zone (Lemaitre and Herman, 2003). Graphitic gneiss was not intersected, and may lie below the depths tested.

These surveys have provided further insight into the geological setting of the deposits, including identification of the location of potentially controlling faults and folding of favourable host lithologies (e.g. graphitic gneiss and competent quartzite-rich host rocks near faults) that may influence the position of mineralization. Some drilling was conducted in 2004 and 2005 to test these target areas beyond the local area of the Horseshoe and Raven deposits and future drilling is planned at other potentially favourable sites.

In addition to these geophysical surveys, which were mainly of a regional nature, a detailed direct current resistivity (induced polarization) survey was carried out over the Horseshoe and Raven deposits as well as the surrounding area by Peter E. Walcott and Associates Limited between October and December 2006 (Walcott and Walcott, 2008). The survey was conducted along sixteen lines at an azimuth of 160° spaced at 200 m over and extending beyond areas of known uranium mineralization at Horseshoe and Raven. Measurements of apparent resistivity were made along these lines using the pole-dipole technique employing a 100 m dipole, and taking one half to one tenth separation readings at half spacing intervals.

10 Drilling

Section 10.1 was taken directly from UEX's November 12, 2008 N.I. 43-101 report entitled "Technical Report on the Geology of, and Drilling Results from, the Horseshoe and Raven Uranium Deposits, Hidden Bay Property, Northern Saskatchewan" by Rhys *et al.* (2008).

Minor updates and changes have been made and comments inserted where appropriate.

A review of the procedures, described below, by Golder with respect to the core sizes, procedures for logging and recording of core recoveries are considered standard industry practices and provide an acceptable basis for the geological and geotechnical interpretation of the deposits leading to the estimation of mineral resources and economic evaluation of the deposits.

Historically, the Hidden Bay property has been explored by numerous diamond drill holes which were completed by several previous operators, as is summarized in Section 5 of this report and Rhys (2002). Since 2002, when the Hidden Bay property was acquired by UEX, drilling has occurred in several target areas on the property (see Section 5). Drilling has been concentrated in areas for which compliant N.I. 43-101 resources are reported at the Horseshoe, Raven and West Bear deposits. In addition, several outlying target areas have also been tested by significant exploration drilling by, or on behalf of UEX.

10.1 Drilling in the Horseshoe and Raven Area

10.1.1 Historical Drilling by Gulf in the Horseshoe and Raven Area

After initial discovery of the Raven deposit, Gulf drilled a total of 53,329 m in 212 diamond drill holes over the Horseshoe and Raven deposit area between 1972 and 1978. These holes form the basis for the estimation of the pre-N.I. 43-101 historical resources. Drill hole spacing of the Gulf holes is variable across the deposits, but generally varies from 30 m to 90 m and averages approximately 60 m in areas of mineralization. A plan view illustrating the collar locations of the Gulf drill holes is presented in Figure 10.1. Drilling by Gulf returned BQ drill core (36.4 mm diameter). Although the Gulf drill hole collar locations are surveyed and many are still locatable in the field, downhole surveying of drill holes was rudimentary, with many holes only subject to acid tests which provide indications of drill hole dip, but not azimuth. Given these uncertainties and the lack of documentation of analytical methods and laboratory quality controls on uranium analyses, the Gulf drilling data was not used in the Horseshoe Mineral Resource and Raven Mineral Resource estimates, which are reported in Palmer (2008) and Palmer and Fielder (2009a, 2009b).

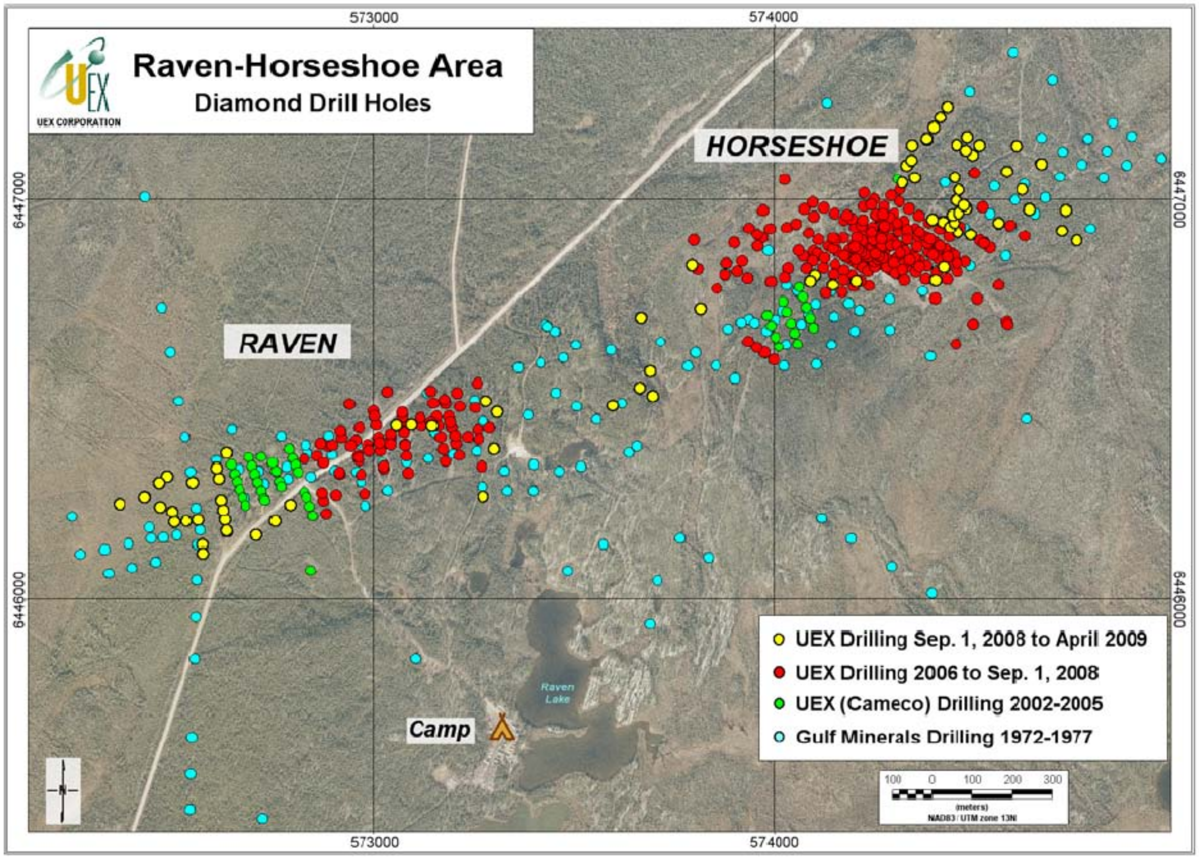


Figure 10.1: Horseshoe and Raven Drill Hole Collars

10.1.2 Drilling in the Horseshoe and Raven Area during 2005

The historical Gulf drilling demonstrated the potential to define significant areas of mineralization at the Horseshoe and Raven deposits, but was too widely spaced to allow confident interpretation of the geometry and extent of mineralized zones. Table 10.1 summarizes the drilling between 2005 and April 2009. In 2005, to test mineralization continuity in parts of the better mineralized areas defined by Gulf, drilling programs were designed in western parts of each of the Horseshoe and Raven deposits with closer spaced drilling. The programs were implemented for UEX by Cameco as geological contractor under the Cameco service agreement and the results are documented in Lemaitre and Herman (2006). The program comprised: (i) 28 diamond drill holes (RV-001 to RV-026) totalling 7,996.3 m in western portions of the Raven deposit on five 50 m spaced cross-sections, with drill holes spaced at 25 m on each section, which test a 200 m strike length of the historical Gulf Raven resource area; and (ii) 16 diamond drill holes (HO-01 to HO-16), totalling 4,815 m, in the western Horseshoe deposit on three cross-sections, with drill holes spaced 25 m apart on each section, which test a 100 m strike length of the historical Gulf Horseshoe resource area.

While re-affirming the presence and location at the Raven deposit, the 2005 drilling program demonstrated the potential for greater continuity and thickness of mineralization in the Horseshoe deposit than was suggested by the historical Gulf drilling results. The drilling also locally intersected wider intercepts of higher grade than had been intersected in the western Horseshoe deposit historically by Gulf. The 2005 Horseshoe drilling included intercepts of:

- 0.55% U_3O_8 over 6.6 m in hole HO-003;
- 0.57% U_3O_8 over 8.7 m and 0.44% U_3O_8 over 6.9 m in hole HO-004;
- 2.82% U_3O_8 over 2.9 m in hole HO-009; and
- 0.48% U_3O_8 over 7.9 m in hole HO-015.

The best intercept in the Raven deposit during this program was 0.46% U_3O_8 over 8.0 m in hole RV-020.

Table 10.1: Summary of Drilling in the Horseshoe and Raven Areas between 2005 and August 2009 by, or on behalf of, UEX

Area	Hole Identifier	Year	Number of Holes	Average Hole Length (m)	Total Length (m)
Horseshoe	HO	2005	16	300.9	4,815
Raven	RV	2005	28	285.6	7,996
Horseshoe	HU	2006-2009	367	320.7	117,713
Raven	RU	2006-2009	226	262.1	59,229
Totals			**637**	**297.9**	**189,753**

10.1.3 2006-2010 Drilling by UEX Corporation

After termination of the Cameco exploration service agreement in 2005, UEX assumed management of all exploration activities on the Hidden Bay property. Since the 2005 drilling only tested short portions of the 1,100 m strike length of the Raven deposit and the 800 m strike length of the Horseshoe deposit as defined by Gulf, UEX proceeded to commence further drill testing of the deposits in 2006, with the drilling programs extending through to 2009 to allow both definition drilling and exploration of the area of the two deposits.

As of April, 2009, 618 surface drill holes had been completed in the Horseshoe and Raven deposit areas since 2005, which represents a total of 184,347 m. These drill holes comprise the basis for the database for the July 2009 Horseshoe and Raven Mineral Resource estimates.

Additional drilling of 19 holes totaling 5,406 m from July to August 2009 was designed to test targets peripheral to the Horseshoe and Raven deposits for possible extensions of mineralization. Results from these holes have not be used to update the July 2009 Horseshoe and Raven Mineral Resource estimates as they are unlikely to have an effect on the resource estimate.

No drilling was carried out in the Horseshoe and Raven deposits areas in 2010.

2006-2010 Drilling at the Horseshoe Deposit

Drilling between June and October 2006 was concentrated in western and central portions of the Horseshoe deposit, further tracing to the east mineralization intersected in the 2005 drilling and testing at 60 m by 30 m spacing areas where some of the best Gulf drill intercepts had occurred. This program, comprising 27 holes (HU-001 to HU-027) and a total of 8,617 m, successfully tracked mineralization eastward from the 2005 drilling and proved mineralization continuity in what is now termed the A and southwestern BW zones. During this program, the most significant drilling intercept to date in the Horseshoe deposit was obtained, with hole HU-016 intersecting 12.35 m grading 4.53% U_3O_8 from 201.50 m to 213.85 m in the Horseshoe A subzone on section 4640N.

Recognition of mineralization continuity and the potential for grades and mineralization thickness in the deposit greater than those identified by Gulf prompted a management decision to conduct definition drilling of the Horseshoe deposit area leading to a new N.I. 43-101 resource estimate.

A systematic drilling program was commenced in January 2007 which extended to April 2009 in which the Horseshoe deposit was drilled off at 15 m to 30 m drill spacing.

Subsequent drilling at Horseshoe comprised:

a) 21,804 m in 63 holes (HU-028 to HU-090) drilled between January and April 2007 which further stepped out to the east at 30 m to 60 m spacing and identified the BE, much of the extent of the BW and the A1-A3 subzones.

b) 30,696 m drilled in 89 holes (holes HU-091 to HU-179) between June and November 2007 which comprised infill drilling to decrease hole spacing to between 15 m and 30 m and additional step-out drilling to extend known zones.

c) 20,371 m drilled in 77 holes (HU-180 to HU-256) between January and April 2008 to test southwestern portions of the Horseshoe deposit, infill between 2005 drill holes in that area and to conduct some peripheral exploration drill holes in projected areas of prospective alteration along strike from mineralized subzones. This is the final phase of drilling that was included in the Horseshoe Mineral Resource Estimate.

d) 4,390 m drilled in 12 holes (HU-257 to HU-268) between June and September 1, 2008 to test exploration targets to the northeast of the main area of resource estimation in an area where historical Gulf drill holes intersected uranium mineralization in widely spaced drill holes.

e) 28,290 m drilled in 90 holes (HU-269 to HU-358) between September 1, 2008 and April 5, 2009 focused mainly in the Horseshoe Northeast area, expanding mineralization there. Ten of the Horseshoe drill holes explored the area between Horseshoe and Raven to the west. This was the final phase of drilling that was included in the July 2009 Horseshoe Mineral Resource estimate.

f) 3,546 m drilled in 9 holes (HU-359 to HU-367) between July and August 2009 to test for the possible continuation of some mineralized zones, including short extensions of mineralization in the Horseshoe Northeast area. The drilling included one deep hole (HU-363) that tested the Dragon Lake Fault on the east side of the Horseshoe deposit.

No drilling was carried out in the Horseshoe deposit area in 2010.

Since most of the ground surface above Horseshoe is elevated and well drained, much of the deposit can be drilled year round, except for southwestern and far southeastern parts of the deposit which are partially under swamp, requiring frozen ground and winter conditions to drill these areas, as was carried out in early 2008. In total, between 2006 and April 2009, 358 diamond drill holes totalling 114,167 m were drilled in the Horseshoe deposit area.

The Horseshoe deposit has presently been drilled by UEX at 15 m to 30 m spacing with locally 7.5 m to 15 m spacing in higher grade areas requiring tighter definition. The UEX drilling programs encountered higher grades, wider intersections, better continuity and an overall greater extent of mineralization at Horseshoe than was outlined by Gulf in the 1970s.

Some of the most significant intercepts received from the 2006-2009 drilling at Horseshoe with grade-thickness product (length multiplied by percent U_3O_8) of greater than 10.0 U_3O_8 % m, include the following:

- 5.43% U_3O_8 over 12.35 m, HU-16 (A zone, section 4640N);
- 0.41% U_3O_8 over 39.0 m, HU-22 (A zone, section 4640 N);
- 0.74% U_3O_8 over 13.40 m, HU-37 (A zone, section 4611N);
- 0.31% U_3O_8 over 65.0 m, HU-43 (A zone, section 4665N);
- 0.58% U_3O_8 over 19.00 m, HU-45 (A zone, section 4593N);
- 0.50% U_3O_8 over 26.60 m, HU-61 (A zone, section 4593N);
- 0.18% U_3O_8 over 60.90 m, HU-63 (A-B zone, section 4755N);
- 0.61% U_3O_8 over 17.65 m, HU-65 (A-B zone, section 4697N);
- 0.83% U_3O_8 over 23.0 m in hole HU-93 (A zone, section 4626N);
- 1.86% U3O8 over 8.3 m in hole HU-99 (A zone, section 4626N);
- 0.28% U_3O_8 over 38.8 m in hole HU-100 (A zone, section 4593N);
- 0.80% U_3O_8 over 22.3 m in hole HU-101 (A zone, section 4611N);
- 0.68% U_3O_8 over 21.0 m in hole HU-102 (A2 zone, section 4682N);
- 0.73% U_3O_8 over 15.4 m in hole HU-113 (BE zone, section 4665N);
- 0.16% U_3O_8 over 65.0 m in hole HU-117 (BE zone, section 4665N);
- 0.22% U_3O_8 over 56.4 m in hole HU-119 (BE zone, section 4740N);

- 0.65% U_3O_8 over 23.1 m in hole HU-126 (A zone, section 4644N);
- 0.64% U_3O_8 over 16.0 m in hole HU-130 (BW zone, section 4724N);
- 0.28% U_3O_8 over 43.8 m in hole HU-133 (BE zone, section 4682N);
- 0.75% U_3O_8 over 31.7 m in hole HU-134 (BW zone, section 4724N);
- 0.47% U_3O_8 over 37.4 m in hole HU-144 (BW zone, section 4724N);
- 1.01% U_3O_8 over 18.2 m in hole HU-156 (A zone, section 4306N);
- 0.567% U_3O_8 over 23.0 m in hole HU-289 (G1 zone, section 4805N); and
- 0.258% U_3O_8 over 41.5 m in hole HU-302, (G1 zone, section 4870N).

Since the drill holes have steep to vertical dips and test shallow dipping subzones, many of these intercepts are close to true thickness.

2006-2010 Drilling at the Raven Deposit

UEX commenced the most recent phase of drilling in the Raven deposit with RU- series drill holes in the latter part of 2006, when 25 holes totalling 6,408 m (holes RU-001 to RU-025) were completed between July and November of that year. The drilling focused on establishing mineralization continuity and extent to the east of the 2005 HO-series drill holes in central parts of the deposit The positive results of that program, which established continuity of several stacked mineralization pods, prompted further drilling with the intent of providing sufficient data for mineral resource estimation. Subsequent drilling from 2007 to 2009 included the following:

- a) Between August and November 2007, 33 drill holes comprising 8,767 m (holes RU-026 to RU-058) were completed which comprised infill drilling between widely spaced sections and step-out drill holes into areas previously defined as mineralized by Gulf, but for which drill spacing was insufficient to confidently establish mineralization continuity.
- b) Between January and April 2008, 18,314 m of drilling in 72 holes (holes RU-059 to RU-130) which continued to expand along 30 m step-out cross-sections along strike, with some infill drilling where necessary to provide a minimum of 30 m drill spacing for resource estimation.
- c) Between June and August 2008, 7,247 m of drilling in 30 holes (holes RU-131 to RU-160), which provided further infill drilling at 15 m to 30 m centres on 30 m spaced cross-sections and step-out holes to the east.

d) Between January and April 2009, 16,633 m of drilling in 56 holes (holes RU-161 to RU-216), consisting mostly of step out drill holes in western parts of the deposit, but also included four infill drill holes and seven holes drilled to test targets east of Raven. This was the final phase of drilling that was included in the July 2009 Raven Mineral Resource estimate.

e) In August 2009, 1,860 m of drilling in 10 holes (RU-217 to RU-226) designed to test for the possible continuation of some mineralized zones, including short extensions of mineralization in the Raven West area. The drilling included six short drill holes that tested a small near surface pod of mineralization south of the current Raven resource, which was previously intersected by several widely spaced Gulf drill holes.

No drilling was carried out in the Raven deposit area in 2010.

To date, the drilling of Raven, including the 2005 drill holes, has defined a 910-m strike length to the Raven deposit, in which mineralization has been defined at 15 m to 30 m drill spacing.

Some of the more significant intercepts with grade-thickness product (length multiplied by percent U_3O_8) of greater than 3.5 % U_3O_8-m include:

- 0.09% U_3O_8 over 40.70 m in hole RU-001 (section 5475E);
- 0.80% U_3O_8 over 2.20 m, 0.08% U_3O_8 over 14.60 m and 0.12% U_3O_8 over 9.00 m in hole RU-002 (section 5475E);
- 0.16% U_3O_8 over 27.0 m in hole RU-004 (section 5475E);
- 0.25% U_3O_8 over 13.30 m in hole RU-005 (section 5535E);
- 0.09% U_3O_8 over 36.20 m and 0.15% U_3O_8 over 8.30 m in hole RU-015 (section 5630E)
- 0.07% U_3O_8 over 20.00 m and 0.06% U_3O_8 over 38.70 m in hole RU-024 (section 5660N);
- 0.10% U_3O_8 over 33.60 m in hole RU-025 (section 5415E);
- 2.98% U_3O_8 over 5.2 m, in hole RU-026 including 7.99% U_3O_8 over 1.5 m (section 5476E);
- 0.13% U_3O_8 over 37.5 m in hole RU-036 (section 5448E);
- 0.18% U_3O_8 over 38.0 m in hole RU-048 (section 5418E);
- 0.16% U_3O_8 over 22.5 m in hole RU-058 (section 5445E);
- 0.09% U_3O_8 over 20.0 m and 0.30% U_3O_8 over 11.0 m in hole RU-071 (section 5630E);
- 0.17% U_3O_8 over 13.5 m and 0.21% U_3O_8 over 8.5 m in hole RU-087 (section 5360E);
- 0.38% U_3O_8 over 37.3 m, including 0.82% U_3O_8 over 9.4 m in hole RU-095 (section 5445E);
- 0.51% U_3O_8 over 7.0 m in hole RU-103 (section 5360E);
- 0.52% U_3O_8 over 19.8 m in hole RU-118 (section 5725E);
- 0.21% U_3O_8 over 24.5 m in hole RU-143 (section 5665E);
- 0.24% U_3O_8 over 24.1 m in hole RU-157 (section 5755E);
- 0.43% U_3O_8 over 18.4 m in hole RU-169 (section 4936E); and
- 0.169% U_3O_8 over 23.0 m in hole RU-179 (section 5613E).

10.1.4 Core Handling, Drill Hole Surveys and Logistical Considerations during the 2005-2010 Drilling Programs

The 2005 to 2008 drilling programs in the Horseshoe and Raven area were performed by Britton Brothers Diamond Drilling Ltd. ("Britton"), of Smithers, B.C., Canada. The winter and summer/fall 2008 drilling programs were completed by Boart Longyear Canada ("Boart") of North Bay, Ontario, following the sale of Britton to Boart in February 2008. The winter and summer-2009 and winter 2010 drilling programs were carried out by Driftwood Diamond Drilling Ltd. ("Driftwood") of Smithers, B.C., Canada. Drill programs were typically run with between two and six rigs operating on a full-time basis during the summer-fall (June to November) and winter (January to April) seasons.

All of the drilling during these programs has been with NQ size core (48 mm core diameter) except for three holes, HU-156, HU-157 and RU-130, which were drilled for metallurgical testing purposes with HQ size core (63.5 mm core diameter).

Drill Hole Field Locations and Surveys

After completion of drilling, the drill hole collar locations are marked in the field with 2 m high wooden pickets, which are visible in all seasons. The pickets are labelled with a permanent aluminum tag with the hole name, dip, azimuth and depth and clearly flagged with high visibility flagging tape.

Proposed hole collars are located in the field by chaining along grid lines from existing collars or located by a hand-held GPS unit. The proposed and completed collars are surveyed internally by UEX personnel with a hand-held Thales ProMark™3 GPS for preliminary interpretations.

Independent checks have been completed on collar locations twice using Tri-City Surveys Ltd. ("Tri-City"), of Kindersley, Saskatchewan. Tri-City used a 5800/Trimble R8 Model 2 hand-held GPS with GNSS. Tri-City also relocated and surveyed the 2005 Cameco drill hole collars. The UEX and Tri-City collar readings are compared and, if any significant differences are noted, the Tri-City reading is re-surveyed; otherwise, it is adopted as the final collar reading. Horseshoe and Raven were drilled on two separate, local project drilling grids.

The Raven grid is rotated approximately 10° clockwise from the UTM WGS 84 (Zone 13) grid north and the Horseshoe grid is rotated approximately 35° anti-clockwise from the UTM WGS 84 (Zone 13) grid north. Surveying, however, is conducted in UTM grids.

Light Detection and Ranging ("LiDAR"), an optical remote sensing technology used primarily for typical digital terrain modelling ("DTM"), was flown over the Horseshoe-Raven and West Bear portions of the Hidden Bay property in August 2007, by LiDAR Services International of Calgary, Alberta. The LiDAR survey was performed to accurately determine the surface landforms in the project areas and forms a cross check to the digital elevations of the surveyed drill hole collars.

A surface DTM was created from the LiDAR and the collars locations were verified in Datamine. Drill hole collars with greater than 1 m elevation difference were reviewed.

Downhole Surveys

Downhole surveys were routinely collected on all holes using the Reflex EZ-Shot® tool at approximately every 25 m to 50 m downhole spacing in the 2006-2009 drilling at Horseshoe and Raven and were also collected during the 2005 drilling program which was managed by Cameco (Lemaitre and Herman, 2006). Reflex EZ-Shot® is an electronic single shot instrument that measures six parameters in one single shot reading azimuth, inclination, magnetic tool face angle, gravity roll angle, magnetic field strength and temperature. These readings are transcribed onto a paper ticket book. Azimuth was recorded in magnetic north and then adjusted to true north with a correction factor of 10.2° of current magnetic declination added to the measured azimuth. This data was then entered in the drill logging database, with corrections if required. On some occasions, the magnetic field was outside of tolerance and, in this case, the measurement was ignored. The error rate where the azimuth had to be removed was 0.57% of all surveys and 0.3% of surveys had transcription errors which were resolved by UEX. Data is exported from the drill logging database and then imported into Datamine, where the drill holes are viewed in plan and section for accuracy.

Drill Core Handling Procedures

At the drill rig, core is removed from the core barrel by the drillers and placed directly in wooden core boxes that are a standard 1.5 m long and a nominal 4.5 m capacity. Individual drill runs are identified with small wooden blocks, where the depth, in metres, is recorded.

Diamond drill core is transported at the end of each drill shift to an enclosed core-handling facility at the Raven camp on the property. In general, the core handling procedures at the drill site are carried out to industry standard.

Core Recovery

Every hole is measured from the start of the hole to the bottom to determine core recovery or block marking errors and for reference m marks. Core recovery is determined by measuring the recovered core length and dividing this by the downhole drilled interval. Core loss is recorded routinely both on the core boxes and during core logging.

UEX has conducted a core loss study over all mineralized domains. Core recoveries through the mineralized subzones in the Horseshoe and Raven deposits are generally very high, with 100% recovery common, even in mineralized intervals. Significant core loss has occurred mainly in the proximal non-mineralized clay alteration haloes to the deposit and in the oxidized zone below the overburden. Up to March 31, 2008, a total of 56.9 m was logged with 0% core recovery, while 4191.95 m were logged with core recoveries from 4% to 99% with the average loss recorded being 30% of the interval drilled.

This equates to 1,248.7 m of core loss over these partial intervals. Adding these figures, the cumulative total core loss was 1305.6 m for the entire UEX drilled RU and HU holes totalling 114,392 m drilled on Horseshoe-Raven up to March 2008, which accounts for 98.9% core recovery. Similar high levels of core recovery are characteristic of the 2005 and 2009 drill holes. Golder has reviewed the core recoveries provided by UEX and has verified these results.

Drill Core Logging

All of the 2006 to 2010 surface holes were geologically logged and sampled by UEX field personnel. All holes were logged in accordance with the UEX legend (Figure 10.2) and geological logging procedure. Geological logging includes the detailed recording of lithology, alteration, mineralization, structure, veining and core recovery. Upon completion of logging a hole, the data is reviewed on a set of working cross-sections for dynamic interpretation of the geology and mineralization. The logging was completed under the guidance of the authors.

Logging data was entered in digitally in to Lagger 3D Exploration ("Lagger") developed by North Face Software on lap top computers. Lagger has the ability to enter and edit drill hole and sample data and has a custom library of UEX geological codes to standardize the logging legend (Figure 10.2).

Principal lithologic units in the Horseshoe and Raven area, QZIT, CARK, ARKQ, SPLO, AMPH and CALC are described in Section 6. Many other units listed below are present on the Hidden Bay property, but not in the vicinity of the deposits.

Codes	UEX name	Description
OB	Overburden	Overburden
CONG	Conglomerate	Conglomerate: maximum grain size >4mm
MDST	Mudstone	Mudstone
SDST	Sandstone	Sandstone: grain size 0.065-4 mm
SLST	Siltstone	Siltstone
UX	Uranium mineralization	Uranium mineralization
CLAY	Clay	Clay alteration: hydrothermal or paleoweathering, protolith uncertain
GOUG	Fault gouge	Fault gouge: unconsolidated cataclasite, clay matrix breccia, precurser lithology is unclear
LOST	Lost core	Lost core
AMPH	Amphibolite	>80% dark green to black amphibole; often massive to crudely banded.
ARKS	Meta-arkose	Massive to weakly foliated or weakly gneissic feldspar > quartz-rich meta-sandstone, with weak to undeveloped gneissic compositional layering. Generally lower biotite content than semipelites
ARKQ	Arkosic Quartzite	Arkosic Quartzite: >30% feldspar, finer grained, more easily altered than the QZIT, specific to Raven Horseshoe area
CALC	Calc-silicate gneiss	Compositionally layered) with amphibole-pyroxene +/- garnet and psammitic (meta-arkosic) layers; may contain dolomite
CARK	Calc-arkose	Arkosic rock with calc-silicate bands (where ARKS>CALC)
DIAB	Diabase	Fine grained mafic dykes with sharp contacts, equigranular, post-metamorphic
DIOR	Diorite	Mafic equigranular, usually medium-grained feldspar with biotite or amphibole-bearing intrusion; usually foliated
DOLO	Dolomite	Grey to cream or pink, usually banded to laminated dolomite-rich unit often with calc-silicate, graphite, or arkosic lamina
GABR	Gabbro	Mafic equigranular, usually medium-grained feldspar + pyroxene +/- amphibole-bearing intrusion; usually foliated
GRAN	Granite	K-feldspar-quartz-biotite granite, massive to foliated; usually medium grained, non-porphyritic; pink to grey
GRGN	Granitic gneiss	Impure granitic gneiss with foliated granitic and other compositional bands
PEGM	Pegmatite	Coarse-grained K-feldspar-quartz-biotite pegmatite; also inludes quartz-dominant pegmatites
PLAG	Plagioclasite	Albite-pyroxene +/- amphibole metasomatic unit after meta-arkose; may contain coarse pyroxene and resemble an intrusion; gradational contacts
PEL0	Pelitic gneiss or schist	Biotite quartz feldspar +/- garnet +/- sillimanite gneiss or schist (>50% biotite for schist) with >25% combined biotite, garnet, and/or sillimanite
PEL1	"	As above, 1-5% graphite
PEL2	"	As above, 5-20% graphite
PEL3	"	As above, >20% graphite
SPL0	Semi-pelitic gneiss	Biotite quartz feldspar gneiss with <25% combined biotite, garnet, sillimanite, often with abundant pegmatitic segregations
SPL1	"	As above, 1-5% graphite
SPL2	"	As above, 5-20% graphite
SPL3	"	As above, >20% graphite
PYRX	Pyroxenite	>80% pyroxene, up to 20% amphibole; often massive to crudely banded. Grains up to 1.5 cm in diameter.
QZIT	Quartzite	Pale grey to white, massive quartz rich meta-sandstone with >80% quartz, and subsidiary feldspar +/- biotite
QZPL	Quartz-rich pelite	Quartz-rich pelite
QV	Quartz Vein	Quartz vein >20cm (+ or - carbonate) NB: Clearly not pegmatoid related

Figure 10.2: Geological Logging Legend Applied to Hidden Bay

The primary purpose of a logging system is to provide a standard process for the geological logging procedures on the Hidden Bay exploration project.

The legend was developed to increase the amount and quality of geological data being collected and allow flexibility with data collection, so geologists can record all the information required without having to record one type of data at the expense of other data. The legend aims to simplify the interpretation of drill hole data and reduce the number of rock codes in the database to a manageable level.

The logging system is broken down into a series or tablets that are used to record the various forms of data required. These tablets include Lithology, Alteration / Paleoweathering, Veining/Structure and Veining/Structure Orientation Data. Each of the individual tablets is treated in isolation such that geologists can refine the data being recorded depending on the types of geological data required for the specific task, e.g. resource definition, grade control, regional exploration.

A core reference library has been established on site and good communication between geologists allow for a consistent approach to geological logging. All core is routinely wet down and digitally photographed as a permanent record of the lithological history, in addition to the geological log, with a Canon Powershot A610 digital camera.

A review by UEX of the historical Cameco logs and scissor holes of the 2005 Cameco drilling indicates that the geological information is complete and of good quality. The Cameco drill holes were logged using a similar legend under the guidance of Roger Lemaitre, P.Geo., from Cameco.

Drill holes completed under the direction of Cameco in 2005 were also re-logged by UEX personnel in summer 2008 to standardize coding and logging data, to perform a second check on sampling intervals and to conduct infill sampling, where necessary.

Geotechnical Logging

All geotechnical logging was completed by, or under the supervision and advice from Golder personnel with the Saskatoon, Saskatchewan and Mississauga, Ontario offices. All selected holes were logged geotechnically in accordance with the UEX Geotechnical Protocol developed by Golder. A selection of holes were logged with RQD, which is the percent of total core length recovered in solid pieces greater than 10 cm in length that correlates with fracture density. Numerous holes were tested for intact rock strength using a rating system based on hammer blows, fracture count per run and detailed total core recovery.

During 2007 and 2008, Golder personnel came to the site and conducted intact rock strength measurements on HQ core using a point load testing machine. Throughout the drill seasons, Golder has also conducted detailed geotechnical assessments of drill core. Logging was completed using the Q rock mass rating system.

In winter 2007/2008, Golder surveyed a series of holes in the Horseshoe area using a downhole televiewer. The aim of this was to determine geotechnical properties directly above the mineralized zones and around the peripheries of the deposit.

Radiometric Probing of Drill Holes

Downhole radiometric probing (gamma logging) with in-hole probing instruments is a routine task undertaken on all holes drilled at the Horseshoe, Raven and West Bear projects. In uranium exploration, probing is integral in accurately detecting gamma radiation downhole which directly correlates to mineralized zones, since these probes are able to quantitatively measure radioactivity caused by the atomic decay of uranium. Through the use of in-house correlation formulas determined from comparing geochemical sampling with probe data, the concentration of uranium in situ can be determined. The probe data is used to determine a uranium equivalent intersection which is used for planning of follow-up drill holes and to correlate intervals in the core boxes to guide geochemical sampling.

A detailed radiation measurement is taken every 10 cm downhole and 10 cm up hole by passing a probe continuously down the drill hole immediately after its completion and measuring in situ radioactivity.

The probes are calibrated before each drill program at the Saskatchewan Research Council's test pit facility in Saskatoon, Saskatchewan. The probing equipment was tested using a known low-grade radioactive source in the field before and after the probing of each hole to ensure that the equipment was functioning properly before and after the in-hole probing occurs. The radiometric logging was performed using a Mount Sopris Model 4MXA/1000 500 m winch, or Model 4MXC/1000 1000 m winch and MGX II Model 5MCA/PMA digital encoder.

A Mount Sopris Modified Triple Gamma Probe consisting of a 2SMA-1000 Sonic Modem section (#3460 or #3461) and 2GHF-1000 Triple Gamma Probe section (#3431 or #3458) was used to probe all holes. Data was acquired using MSLog Version 7.43, a Mount Sopris computer recovery program. Data from the probe is then used to correlate mineralized zones with the drill core and identify zones for sampling and geochemical assay.

A second check is to scan the drill core with a hand-held SPP2 scintillometer or a RS-120/125 super scintillometer.

Detailed radiometric measurements are taken every 10 cm on the core in mineralized zones and recorded on the core and in accordance with standard procedure. At times, there are some discrepancies with the downhole probe interval and the core due to stretch in the winch cable, the counter wheel icing up or a differing zero depth between the core and the probe data.

The detailed radiometric readings from the hand-held scintillometer on the drill core are used as a guide by the geologist for geochemical sampling. The geologist marks the intervals on the individual sample and the sample numbers and location are recorded in drill logs.

Relationship between Sample Length and True Thickness

Since the orientations of drill holes in the deposit vary, and the morphology of mineralized zones has variable orientation across the two deposits, the relationship of geochemical sample length in drill holes to the true thickness of mineralization is also variable. At the Horseshoe deposit, the steep orientation of most drill holes crosses the lens-shaped mineralized zones at or near to true thickness. The 15 m to 30 m spaced drilling density, and geological confidence in the mineralization extent orientation and morphology has enabled 3-dimensional ("3D") wireframe modelling of both deposits which accommodates for variations in sample length to local orientation of drill holes and mineralized zones.

10.2 Drilling on Other Parts of the Hidden Bay Property

Since UEX acquired the Hidden Bay property, drilling as the principal means of exploration has been conducted on several exploration targets in addition to the resource and exploration drilling that is documented here at the Horseshoe and Raven deposits. A review of all of these exploration programs is beyond the scope of this report. However, principal areas targeted by drilling outside the two main deposits, the quantity of drilling, and highlights of the results are outlined briefly below.

The same drill core handling and QA/QC standards are applied to all current drilling on these targets as is applied to drilling in resource areas as is described in other portions of this report.

Table 10.2: Summary of Drilling Conducted by, or for, UEX Corporation, on Exploration Targets within the Hidden Bay Property outside the Horseshoe-Raven Area 2002-2010

Area	Year	No. of Drill Holes	Series	Drilled Length (m)
West Bear Deposit – Sonic Drilling	2004	3	UEX-001 to 003	84
	2005	101	UEX-004 to 101A	2,793
	2007	113	UEX-102 to 214	3,386
Dwyer Dome and West Bear Area Exploration	2002	11	WBE-012 to 022	1,284
	2003	10	WBE-023 to 032	1,345
	2004	15	WBE-033 to 047	1,853
	2005	43	WBE-048 to 091	5,019
	2006	36	WBE-092 to 127	3,958
Telephone	2002	6	SP-142 to 147	1,917
	2003	4	SP-148 to 151	1,055
	2005	6	SP-155 to 160	1,538
	2006	29	SP-161 to 186	2,674
	2007	4	SP-187 to 190	964
	2009	26	SP-191 to 216	7,968
	2010	21	SP-216 to 237	6,531
Shamus	2003	2	SHA-33 to 34	827
	2004	3	SHA-35 to 37	1,331
	2008	5	SHA-38 to 42	1,731
Tent-Seal	2007	13	SEAL-61 to 73	2,928
	2008	25	SEAL-74 to 98	6,583
Kewen Lake	2003	3	SP-152 to 154	731
Rabbit West	2006	9	LMS-106 to 114	1,890
	2007	4	LMS-115 to 118	1,132
	2008	14	LMS-119 to 132	4,252
Vixen Lake	2003	1	VN-01	237
	2004	12	VN-02 to 13	2,256
	2009	4	VU-001 to 004	1,697
Moosippi Lake	2003	1	RW-01	308
	2004	4	RW-02 to 05	652
Wolf Lake	2007	19	WO-114 to 131	3,066

One to two holes were also drilled in several other areas, but only targets for which three or more holes were completed are shown here. Areas are shown in Figure 10.3.

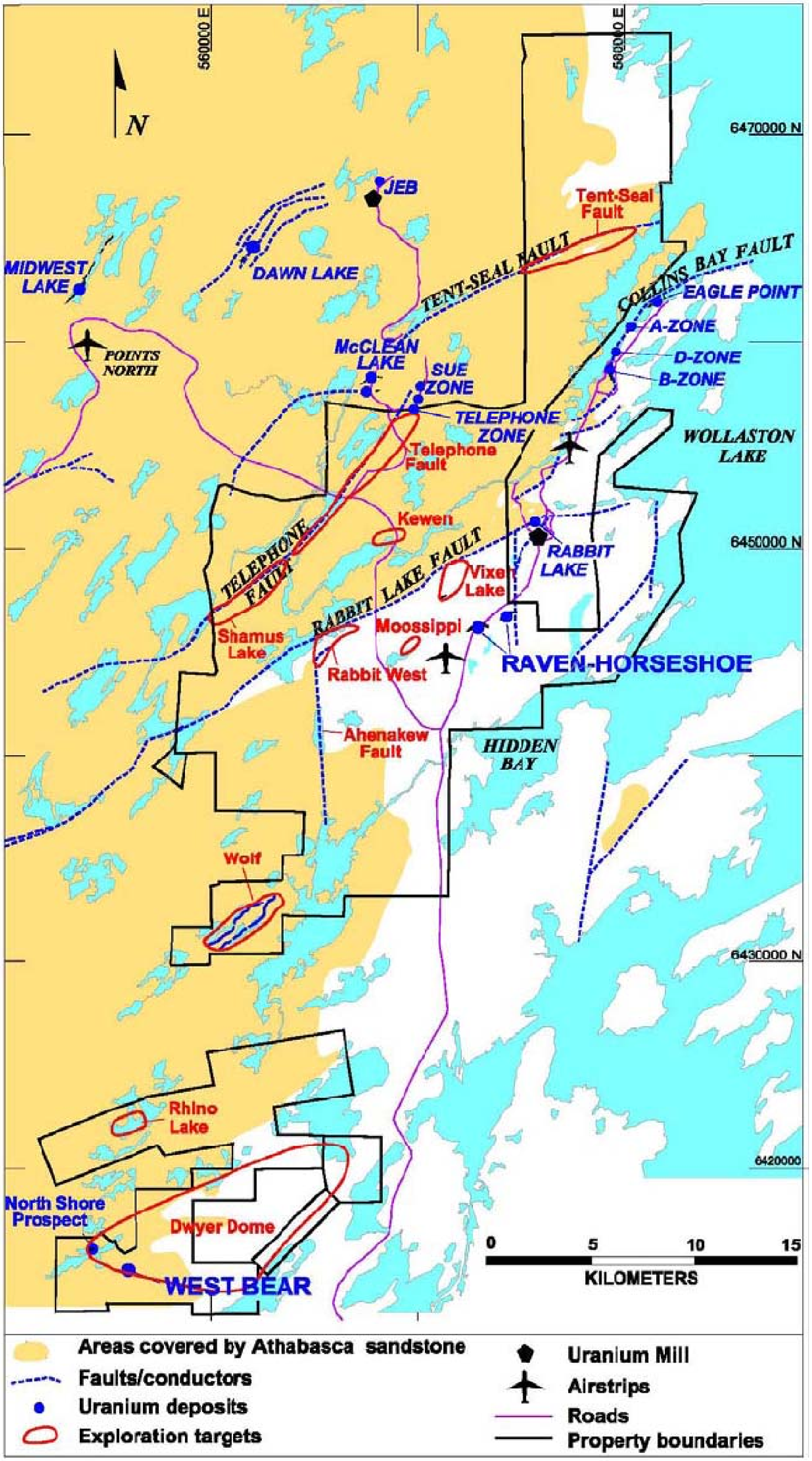


Figure 10.3: Hidden Bay Property Drilling Target Areas 2002-2010

West Bear Deposit

The West Bear deposit, located in the southernmost part of the Hidden Bay property, is a classic unconformity-hosted uranium deposit which is developed under shallow Athabasca sandstone cover above a conductive graphitic gneiss unit. The deposit occurs along a conductive, graphite-bearing gneiss unit where it intersects the overlying Athabasca sandstone along the southern margins of the Dwyer Dome.

West Bear is flat-lying, and forms a northeast-trending mineralized zone that has been defined by drilling over a strike length of 500 m, in a long, cigar-shaped mineralized zone straddling the unconformity. The mineralization occurs at a vertical depth of between 13 m and 31 m from surface and is one of the shallowest undeveloped uranium deposits in the prolific Athabasca Basin. The deposit ranges in width from 5 m to 25 m, and in vertical thickness from 0.1 m to more than 10 m. Mineralization occurs in intense clay-hematite alteration where a minor fault system (West Bear fault) hosted shallow southeast dipping graphitic gneiss intersects the unconformity. The deposit is typical of the unconformity-hosted style of mineralization in the Athabasca Basin that is also exemplified by the McClean Lake and Cigar Lake deposits, and shows the typical association with nickel-cobalt-arsenic mineralization.

Mineralization at West Bear consists of sooty black pitchblende found as disseminations, blebs, and replacement of host rock minerals in both the sandstone and basement rocks. Minor yellow secondary uranium minerals such as uranophane and other gummite minerals are observed as disseminations and blebs in selected drill holes. Higher-grade holes contain intervals of semi-massive pitchblende up to three metres in core length. Pitchblende, sulphides and sulpharsenides of Fe, Ni and Co and Pb (including pyrite, galena, niccolite, gersdorffite, cobaltite, rammelsbergite, and chalcopyrite) are the dominant metallic minerals in the mineralized zone (Fischer, 1981).

Sulphides are paragenetically early, followed by sulpharsenides, arsenides and pitchblende. Nickel-cobalt-arsenic mineralization associated with the sooty pitchblende mineralization is most highly concentrated in eastern portions of the deposit, particularly in lowermost portions of the mineralized zone beneath the unconformity. In these areas, grades range up to 4% nickel. Lemaitre (2006) obtained typical average grades throughout the deposit of 0.34% Ni, 0.11% Co and 0.50% As. Anomalous Ni-Co-As mineralization also occurs in basement graphitic gneiss to the east-southeast of the deposit.

A high-grade core to the West Bear deposit occurs over a strike length of approximately 100 m between sections 1750 E and 1850 E. Within this area, uranium mineralization has the largest widths, highest uranium concentrations and is associated with areas of most intense clay alteration. The resource estimate suggests that approximately 95% of the deposit’s contained uranium, as currently defined, is located within this interval at a 0.05% U_3O_8 cut-off. The best intercepts in this area include:

- 4.927% U_3O_8 over 10.10 m in hole UEX-026 (section 1775E);
- 6.032% U_3O_8 over 10.67 m in hole UEX-206 (section 1762.5E); and
- 4.040% U_3O_8 over 11.41 m in hole UEX-207 (section 1762.5E).

In easternmost portions of the deposit, mineralization splits into multiple, generally lower grade lenses, which range typically in grade from 0.1 to 0.7% U_3O_8.

In 2002, UEX successfully drilled one single diamond drill hole (WBE 017) at the western end of the West Bear deposit. The hole was drilled as part of a larger regional diamond drilling program to test the viability of modern diamond drilling equipment in the West Bear area. Hole WBE-017 intersected uranium mineralization at the sandstone/basement unconformity that averaged 1.79% eU_3O_8 over 11.0 m including 2.34% eU_3O_8 over 7.3 m, significantly higher grade than encountered in proximal Gulf RCD and diamond drill holes. It was noted in the drill log that extremely high core loss was encountered in the sandstone due to very intense hydrothermal alteration resulting in very strong clay alteration and very strong quartz matrix dissolution

In February 2004, UEX initiated a sonic drilling program to test the West Bear deposit with the objective of working towards an updated resource estimate. The drilling program was designed to evaluate core recovery and confirm grades of select Gulf holes within the West Bear deposit. An attempt was made to twin three of Gulf's historic mineralized holes (one RCD hole and two diamond drill holes). A total of 84 m was drilled with only one of the three attempted sonic holes being successfully completed due to drilling difficulties.

In January 2005, UEX initiated a 101 hole - 2,793 m sonic drilling program on the West Bear deposit, with the objective of determining a N.I. 43-101 compliant resource estimate of the deposit. Cameco implemented the program under an exploration management agreement on the Hidden Bay property with UEX. A total of 97 successfully completed and 4 unsuccessfully completed sonic drill holes were drilled (holes UEX-004 to UEX-101A).

Sonic drilling was carried out on 25 m fences between L19+50E and L21+25E, except for two infill fences in a high grade zone on L17+65E and L17+90E. The spacing of holes along each drill fence was 5 m.

UEX's 2007 winter sonic drilling program included additional infill holes spaced at 5 m intervals on two sections (1762.5 E and 1787.5 E) in the high-grade core of the main deposit area between sections 1750E, 1775 E and 1800 E drilled by Cameco in 2005. These holes were designed to better define the geometry and uranium grades in the higher grade core area of the deposit area where it was identified that expansion of the core areas of the deposit from the 2006 resource calculation were possible. The drilling successfully expanded the area of higher grade mineralization, intersecting up to 6.032% U_3O_8 over 10.67 m in hole UEX-206 on section 1762.5 E and 2.341% U_3O_8 over 7.08 m in hole UEX-197.

In addition, step-out drilling to the east was completed to test the eastern extension of the deposit which was not tested during the 2005 program. A total of 113 additional drill holes totalling 3,386 m were drilled at West Bear during the 2007 program.

Dwyer Dome Targets

Several prospects lie around the Dwyer Dome on the same conductive trend as the West Bear deposit. These include Pebble Hill, North Shore and Blanche Lake, where previously small pods of mineralization had been outlined historically by drilling. Principal targets here are for shallow, unconformity-hosted mineralization like West Bear. UEX tested several of these areas between 2002 and 2006 to follow up on historical results, while simultaneously exploring the area immediately around and east of the West Bear deposit.

During 2002, one drill hole was completed in the Pebble Hill prospect, with hole WBE-16 intersecting a Fe-oxide-clay altered zone in pegmatite was intersected 7.1 m below the Athabasca unconformity, which contains 1.926% U_3O_8 over a 2.2 m interval just below the Athabasca unconformity. This drill hole successfully relocated the historical Pebble Hill mineralization; subsequent drilling suggests that this is close to true thickness, but the lateral extent of this lens is very limited. As a result, in 2003, seven holes (WBE- 23-29) were drilled to define the extent of this mineralization. While these holes intersected anomalous radioactivity and high Ni-Co-As geochemistry, no significant uranium intercepts were encountered, bounding much of this mineralization. In 2006, two holes (186 m) were drilled at the prospect to test for further mineralization to the east and north of known mineralization. A third hole (120 m) tested a prominent conductive feature on the Mitchell-Dwyer Trend to the north. No significant mineralization was intersected and no further work is planned in the Pebble Hill area at this time. In 2006, thirteen holes (1,287 metres) were also drilled to relocate and evaluate the North Shore Prospect on Mitchell Lake northwest of West Bear. The drilling successfully relocated the North Shore Prospect mineralization with four of the holes encountering significant mineralization. For example, hole WBE-117 intersected 0.2 m grading 0.51% U_3O_8 between 43.6 m and 43.8 m depth immediately above the unconformity. True thickness of this intercept and extent of mineralization beyond this drill hole are not known.

Future follow-up drilling is planned to target extensions to the mineralization to the south and east along the Mitchell-Dwyer conductive trend on the northwestern margin of the Dwyer Lake Dome.

Four holes (534 m) were also drilled in 2006 at the Blanche Lake Prospect further to the east to relocate and test for potential extensions of known mineralization. Historical drill hole BC-08 graded 0.21% U_3O_8 over 0.4 m. UEX's 2006 hole WBE-112 intersected 0.13 m grading 0.10% U_3O_8 and although anomalous radioactivity was intersected along the same structure at depth, no other significant mineralization was found.

The lateral extent and true thickness of the mineralization in these intercepts are not known. The Mitchell-Dwyer conductive trend to the east remains highly prospective, particularly those sections associated with an offset caused by the Ahenakew Fault.

Telephone Lake Area

The Telephone Lake area ("Telephone") comprises an along strike continuation of faults and conductors which extend into the Sue deposits area on the adjacent McClean Lake property to the north. The principal target here is the Telephone Lake Fault, a north-northeast trending, southeast dipping reverse graphitic fault zone which is developed along the southeast margin of the McClean Lake Dome. The fault has accommodates approximately 60 m of reverse displacement. Targets here are for Eagle Point style basement mineralization along, and adjacent to the fault in the basement gneiss sequence, and associated unconformity style mineralization where the fault intersects the base of the overlying Athabasca sandstone. Since the mineralization in this area is not yet defined, the true widths and lateral extent of mineralized intervals quoted below for the Telephone Lake area are not yet known.

Prior to UEX acquiring the property, previous operators had drilled approximately 140 holes (SP- and TEL-series) along an approximately 10 km strike length of the fault extending southward from the McClean Lake property boundary, and along several parallel, associated conductors. Several areas of low grade mineralization with associated alteration were intersected along the main fault. Drilling conducted by, or for UEX between 2002 and 2007 further tested this area with 49 drill holes (SP-142 to 151 and SP-155 to 190). Mineralization intersected includes an intercept in hole SP-156, drilled by UEX in 2005 and located at the north end of the Telephone Lake Fault 2.1 km southwest of the Sue E deposit, which returned 4.52% U_3O_8 over its 0.5 m between 189.8 to 190.3 m in basement rocks just beneath the unconformity. Hole SP-176, located 300 m northeast of SP-156, intersected 0.37% U_3O_8 over 0.5 m from 202.4 m to 202.9 m.

Drilling in the southern Telephone area in 2006, 2.6 km to the southwest of SP-156, was intended to test for extensions of mineralization intersected by historical holes SP-32 (0.60% U_3O_8 over 0.9 m) and SP-38 (0.62% U_3O_8 over 0.6 m). Hole SP-166 intersected an approximately 30 m interval containing local disseminated and veinlet-controlled pitchblende in faulted Athabasca sandstone adjacent to faulted basement rocks within the Telephone Lake Fault zone.

Mineralization in this zone was found in two mineralized intersections:

- 0.20% U_3O_8 over 6.80 m from 129.7 to 136.5 m, including subintervals of 0.66% U_3O_8 over 0.5 m, 0.64% U_3O_8 over 0.4 m and 0.57% U_3O_8 over 0.5 m; and
- 0.11% U_3O_8 over 6.50 m from 148.5 to 155.0 m, including 0.64% U_3O_8 over 0.2 m, 0.33% U_3O_8 over 0.2 m and 0.32% U_3O_8 over 0.4 m.

During the summer of 2009, 7,968 m of drilling in 26 drill holes distributed over a 4 km strike length along the Telephone Lake Fault were completed. The drilling targeted areas of known mineralization near the unconformity that included previous intercepts of 0.20% U3O8 over 6.8 m in 2006 drill hole SP-166 and 4.52% U3O8 over 0.5 m in 2005 drill hole SP-156.

Drilling intercepts from the 2009 program with a grade-thickness product of greater than 0.05 and grades of greater than 0.05% U3O8 include the following:

- 0.110% U_3O_8 over 0.5 m in hole SP-191;
- 0.100% U_3O_8 over 2.0 m, and 0.401% U_3O_8 over 1.9 m in hole SP-193;
- 0.277% U_3O_8 over 0.3 m in hole SP-194;
- 0.066% U_3O_8 over 1.1 m, and 0.055% U_3O_8 over 1.0 m in hole SP-196;
- 0.105% U_3O_8 over 1.1 m, and 0.074% U_3O_8 over 2.8 m in hole SP-201;
- 1.527% U_3O_8 over 1.5 m in hole SP-203;
- 0.076% U_3O_8 over 1.6 m in hole SP-207;
- 0.062% U_3O_8 over 1.0 m in hole SP-209;
- 0.120% U_3O_8 over 0.7 m in hole SP-210;
- 0.370% U_3O_8 over 6.5 m, including 1.131% U_3O_8 over 2.0 m, in hole SP-211;
- 0.360% U_3O_8 over 1.0 m in hole SP-212; and
- 0.140% U_3O_8 over 0.4 m, and 0.125% U_3O_8 over 2.7 m in hole SP-213.

Intercepts in drill holes SP-201, 203, 210, 211 and 212 are unconformity-hosted mineralization, while all other intercepts are basement-hosted.

As mineralization was open in many areas, UEX focused its 2010 winter exploration program to follow up on these results. The winter 2010 diamond drilling program consisted of 21 holes totalling 6,531 m designed to test potential downdip continuation of known mineralization, to test along strike for extensions of unconformity mineralization, and to test gaps where widely-spaced sections have geology favourable for basement-hosted mineralization.

Results of principal drilling intercepts obtained during 2010 at Telephone with a grade-thickness product of greater than 0.05 %-m and grades of greater than 0.05% U_3O_8 include the following:

- 0.078% U_3O_8 over 5.0 m, including 0.180% U_3O_8 over 1.0 m in hole SP-217;
- 0.116% U_3O_8 over 4.7m including 0.399% U_3O_8 over 0.9 m in hole SP-222; and
- 0.145% U_3O_8 over 0.5 m in hole SP-229.

The company continues to evaluate this area and it is considered a high priority exploration target for mainly basement-hosted mineralization. The Telephone drilling has highlighted three anomalously mineralized areas that contain a combination of unconformity-hosted and basement-hosted mineralization. Additional mineralized drilling intercepts are also present periodically along the four-kilometre length of the Telephone Lake trend and extend southward into the Shamus Lake area.

Shamus

The Shamus Lake area ("Shamus") is the southwestern continuation of the Telephone Lake area (Figure 10.2) and, like that area, the principal target is the southwestern continuation of the southeast dipping Telephone Lake Fault, which lies along the southeast side of the McClean Lake Dome. The Telephone Lake Fault here splits from a single structure in the Telephone Lake area into several strands on the Shamus grid. The principal target here is either unconformity or basement hosted uranium mineralization, similar to the Eagle Point Mine or the Sue deposits. Prior to UEX acquiring the property, previous operators had drilled holes SHA-001 to SHA-032. These widely spaced drill holes which intersected several areas of low grade mineralization with associated alteration that returned grades ranging from 0.1% to 0.46% U_3O_8 over intervals of several metres, including 0.39% U_3O_8 over 2.2 m in hole SHA-20. The lateral extent and true thickness of the mineralization in these intercepts are not known.

Since UEX acquired the Hidden Bay property, ten holes were drilled in the Shamus area totalling 3,889 m. As with previous drilling, several areas of low grade mineralization and alteration with anomalous radioactivity were intersected both in basement rocks where they are associated with fault strands often marginal to or within pegmatite and adjacent graphitic gneiss, and in the vicinity of the sub-Athabasca unconformity.

Tent-Seal

The principal target in this area is the Tent-Seal Fault, which is an east-northeast trending moderate south-southeast dipping reverse fault zone that is developed in graphitic gneiss. The fault and hosting graphitic gneiss occur along the northerly contact with the Collins Bay Dome (Figure 6.1). Areas of clay alteration with drusy quartz veins and anomalous radioactivity had previously been intersected here along fault strands. The alteration style and drusy quartz veining that was intersected historically are comparable to peripheral alteration adjacent to mineralization at the Eagle Point Mine (Rhys, 2002).

This coupled with the presence of a pod of basement hosted mineralization known to occur along the Tent-Seal Fault on the adjacent McClean Lake property to the west made the Tent-Seal area a prospective exploration target.

In order to follow up on the historical results, and to test previously untested or poorly tested segments of this fault particularly for basement mineralization, UEX drilled 38 diamond drill holes between 2007 and 2008 using a helicopter supported drill in the summer programs.

Much of the drilling was initially focused on a broad right-handed flexure in the fault system where some of the more intense alteration had been previously intersected. Several holes were not completed due to poor drilling. The drilling intersected similar styles of alteration along the fault to what has been intersected historically, with some areas of quartz vein development. Several areas of anomalous radioactivity and low grade mineralization were encountered, for which 2007 geochemical results are available. These include 1.10 m grading 0.248% U_3O_8 from 126.0 m to 127.1 m in hole SEAL-68, and 1.00 m grading 0.206% U_3O_8 from 66.0 m to 67.0 m in hole SEAL-72. The extent and true thickness of the mineralization in these intercepts are not known. Geochemical results from 2008 are still being received, and the area will be fully evaluated by UEX once all data is returned.

Kewen Lake

In 2003, three diamond drill holes totalling 731 m were drilled to test a 600 m long section of the Kewen Lake fault zone in areas where 1990s Cameco drilling previously encountered intense alteration and anomalous geochemistry and radioactivity in the basal Athabasca sandstone above a graphitic conductor. The drilling targeted previously untested basement targets along the fault. However, no significant alteration or radioactivity was encountered in the three holes.

Rabbit West

The Rabbit West target area is situated on, and south of the Rabbit Lake Fault near its intersection with the Lampin Lake fault, the latter which is a northeast trending splay of the Ahenakew fault that links it to the Rabbit Lake Fault (Figures 6.1 and 10.2). The area corresponds with a radiometric high over the project area and fault offsets of magnetic lithologies, forming composite structural-radiometric targets. The radiometric anomaly, defined by airborne surveys and confirmed by historical overburden drilling in this area, terminates up-ice along the Rabbit Lake Fault.

Target areas for mineralization in this area which were tested by UEX's drilling include: 1) the Rabbit Lake Fault itself at the up-ice termination of the broad radiometric anomaly, where only widely spaced holes fully tested the fault and local gaps in drilling of nearly 1 km where the fault was not previously tested; 2) the Lampin Lake and associated faults in the vicinity of the radiometric anomaly; and 3) the area of intersection of the Rabbit Lake and Lampin faults in the radiometric anomaly, where the wedge between the fault surfaces forms a similar structural geometry to the setting of the Rabbit Lake deposit which also occurs in the wedge between a northeast-trending fault and the Rabbit Lake Fault (Rhys, 2002). Between 2006 and 2008, UEX drilled 27 drill holes for 7,274 m over a 3 km strike length in these three areas along and south of the Rabbit Lake Fault. Many holes drilled to the south of the Rabbit Lake Fault intersected minor faults, hematite and weak clay altered pegmatite that is locally brecciated and which contains anomalous radioactivity and uranium mineralization.

Intercepts obtained during the 2006 drilling program include 0.184% U_3O_8 over 0.6 m from 102.2 m to 102.8 m in hole LMS-107, 0.182% U_3O_8 over 0.44 m from 192.46 m to 192.9 m in hole LMS-112, and 0.284% U_3O_8 over 1.16 m from 72.45 to 73.6 m in hole LMS-114. The extent and true thickness of the mineralization in these intercepts is not known.

The 2007 and 2008 drilling delineated the location of the convergence of the Lampin Lake Fault with the Rabbit Lake Fault. The convergence comprises numerous splays and link structures between the two faults and is interpreted to define a steeply dipping chute zone that may host similar mineralization to the Rabbit Lake deposit. One hole (LMS-127) intersected 0.396% U_3O_8 mineralization over 0.4 m and three other holes (LMS-123, LMS-124 and LMS-132) intersected anomalous mineralization greater than 0.020% U_3O_8 over more than 3.3 m.

Vixen Lake

The Vixen Lake area contains an extensive uranium-nickel anomaly and boulder train of glacially transported mineralized material in overburden which was historically identified by Gulf 2.5 km to 4 km southwest of the past-producing Rabbit Lake Uranium deposit. Gravity and soil sampling surveys were performed in the area in 2003 to further evaluate the potential source of these, evaluating the potential for gravitationally low areas of clay alteration and anomalous geochemistry that could be associated with a nearby uranium deposit in areas between or outside historical overburden drilling. Twelve diamond drill holes totalling 2,256 m were drilled in 2004 for UEX under management by Cameco, ten of which encountered strong chlorite ± clay alteration and brittle brecciation similar to the alteration and structures associated with the Rabbit Lake Uranium deposit. Despite the strong alteration encountered, the drill holes did not intersect any significant radioactivity.

Drilling in the summer of 2009 was carried out in the Vixen Lake South area, which lies 1.5 kilometres northwest of the Raven deposit. Four drill holes totalling 1,697 m tested the core of a well-defined, east-northeast trending gravity-resistivity low where historical drilling in shallow holes had identified broad areas of clay alteration similar to the signature of alteration associated with the Horseshoe and Raven deposits. No significant mineralization was intersected.

Wolf Lake

The Wolf Lake area is underlain by a pair of conductive graphitic pelitic gneiss horizons which outline a probable domal D2 fold. Metamorphic lithologies dip shallowly to the south, and graphitic units are remobilized by local post-Athabasca faults beneath a thin cover of Athabasca sandstone. Anomalous uranium mineralization and alteration has been historically intersected in drill holes in several locations along these horizons, including in an S-shaped bend in one structure that may represent a prospective constrictional jog.

Drilling by UEX in 2007 in the Wolf Lake area totalled 3,066 m in 19 drill holes which were focused in three key areas. The drilling followed up, and drilled potential lateral extensions of areas of historical drilling which contained anomalous and low grade intercepts at vertical depths of 40-100 m. Drilling in the southern and central areas failed to intersect any significant mineralization. The northern area identified a clay altered graphitic pelite with significant faults and clay gouge. Intersections include: a) 39.5 m grading 0.036% U_3O_8 from 46.0 m to 85.5 m, including 0.133% U_3O_8 from 64.0 m to 64.3 m and 0.054% U_3O_8 from 76.5 m to 77.4 m in WO-125; b) 1.65 m grading 0.076% U_3O_8 from 101.85 m to 103.5 m in WO-127; c) 2.0 m grading 0.65% U_3O_8 from 53.0 m to 55.0 m in hole WO-130; and d) 0.6 m grading 0.052% U_3O_8 from 77.0 m to 77.6 m in hole WO-131. The target area where these intercepts were obtained is open to the north. The lateral extent and true thickness of the mineralization in these intercepts are not known.

11 Sampling Method and Approach

Section 11.1 was taken directly from UEX's November 12, 2008 N.I. 43-101 report entitled "Technical Report on the Geology of, and Drilling Results from, the Horseshoe and Raven Uranium Deposits, Hidden Bay Property, Northern Saskatchewan" by Rhys et al. (2008).

Minor changes have been made and comments inserted where appropriate.

the following review by Golder of the procedures, for the sampling method and approach used by UEX indicated that they are of an industry standard and provide an acceptable basis for the geological interpretation of the deposits leading to the estimation of mineral resources and economic evaluation of the deposits.

The sampling method and approach at West Bear is very similar to that described below for Horseshoe and Raven and is documented in reports by Palmer and Fielder (2009a, 2009b).

11.1 Horseshoe and Raven

Drill core sampling for geochemical assay is the primary sampling method. A combination of radiometric responses from hand-held scintillometer readings on drill core and recognition of visibly mineralized or altered areas guided sampling. Sampling has been conducted continuously across mineralized intervals within the mineralized zones. Samples were also collected from the non-mineralized core for at least several m above and below mineralized intersections to confirm the location of the mineralization boundaries for each mineralized zone. In the case of multiple zones of mineralization in a hole, the internal non-mineralized section was generally sampled to provide a more continuous profile. In June 2008, UEX implemented a program of sampling weakly and non-mineralized core to clearly bracket mineralization with a nominal 2 m of sampling below 0.02% U_3O_8 and any broad zones of internal waste were sampled.

Re-sampling of holes was conducted at this time where previously sampled intervals were deemed too restricted in extent.

A representative length check on selective sample intervals was conducted on all of the HU and RU holes up until March 31, 2008. A total of 16,756 m of core was sampled representing 24,049 samples averaging 0.7 m in length. Sample intervals range from 0.1 m to 3.0 m with 261 samples or one percent of the total dataset greater or equal to 1.2 m in length. Note this excludes non-routine blanks and standards. Typically, the broader intervals were sampled over areas of low core recovery. An extra 1,635 samples, each approximately 10 cm in length, underwent spectral analysis with PIMA and were assayed with a full multi-element suite to spectrally and geochemically profile the alteration signature of the deposit.

To April 2009, the entire UEX drilled Horseshoe and Raven database includes 46,667 selective sample records and 3,002 systematic sample records (these numbers include routine standards and blanks).

After core logging, all drill core marked for sampling was split longitudinally to obtain a representative half core sample for geochemical analysis. Splitting of core samples was undertaken by employees of UEX at the Raven Camp.

Samples were split dry and not cut, using an electric hydraulic press with a “knife” and “V-block”. The splitter and sample trays were vacuumed clean to prevent contamination between each sample. One half of the core was placed in a clear plastic sample bag and the bag top was rolled down and then securely taped to prevent any sample loss.

Once a sample is split and bagged up, an additional level of quality control was introduced where the radioactivity of the sample was measured by a SPP-2 scintillometer. These samples were then placed in approved pails and then sent to SRC Geoanalytical Laboratory for assaying. The second half was retained for geological documentation and record purposes and remains in the core box. A sample tag with the sample number was stapled into the core box to mark the location of the sample interval. All mineralized sections are kept in permanent wooden racks for easy access and review. After each hole was sampled, the splitting tent was cleaned to prevent hole to hole contamination and to minimize the amount of background radiation from dust.

A small representative portion of drill core has had the second half of the core removed for specific gravity and dry bulk density testing and some intersections have been taken for detailed metallurgical testing. The three HQ holes were bulk sampled for metallurgical testing and, as a result, no remaining core is available.

11.2 Sampling Quality and Representativeness

The sampling methods and approach utilised by UEX at the Horseshoe, Raven and West Bear deposits meet industry standards. The sampling of outlying targets was not reviewed by Golder but is being carried out using the same protocols. There is no drilling, sampling or recovery (core loss) factors that, in Golder’s opinion, could materially impact the accuracy and reliability of the results.

Sample locations and lengths are selected to appropriately represent mineralization distribution, with breaks between sample intervals made between obvious changes in geology or mineralization distribution. As a result, the sampling is considered to consistently represent the appropriate length and quantity of mineralization to determine a representative uranium grade independent of mineralization style. No inherent sampling biases exist in the longitudinal splitting of the core and sample processes are consistent from season to season. It is Golder’s opinion that the samples are of good quality, representative and no material factors that may have resulted in sample biases. The sample data has been verified through correlation of probe, detailed radiometric SPP2 readings and a detailed assay comparison and QA/QC program.

12 Sample Preparation, Analysis and Security

The following section was summarized from UEX's November 12, 2008 N.I. 43-101 report entitled "Technical Report on the Geology of, and Drilling Results from, the Horseshoe and Raven Uranium Deposits, Hidden Bay Property, Northern Saskatchewan" by Rhys et al. (2008).

Minor changes have been made and comments inserted where appropriate.

The sample preparation, analysis and security at West Bear was very similar to that described below for Horseshoe and Raven and is documented in reports by Palmer and Fielder (2009a, 2009b).

Sample preparation procedures have not varied since the initiation of the exploration at Horseshoe, Raven and West Bear in 2005. Quality assurance/quality control ("QA/QC") procedures have improved from laboratory based quality control initially to the implementation of a more in-depth QA/QC protocol. A description of the core handling, sample preparation, security, and sample handling procedures employed by UEX staff while the samples were in their possession has been documented in detail in Section 11 of this report.

All laboratory analyses of drilling samples for UEX, except for select check sampling, were conducted by the Saskatchewan Research Council ("SRC"). The SRC has an ISO/IEC 17025:2005 accredited quality management system (Scope of Accreditation #537), from the Standards Council of Canada (SRC, 2007). SRC's Geoanalytical Laboratory is located at 125-15 Innovation Blvd., Saskatoon, Saskatchewan. The SRC laboratories are accredited by the Canadian Association for Laboratory Accreditation Inc.

Once the samples arrived in Saskatoon, all elements of sample preparation were completed by employees of the Saskatchewan Research Council's Geoanalytical lab. When samples arrived at the lab, no employee, officer, director or associate of UEX, was involved in any aspect of sample preparation and analysis. In Golder's opinion, the sample preparation, security and analytical procedures meet industry standards.

12.1 Shipping and Security

Radioactive samples, mainly drill core, were shipped within Canada in compliance with relevant federal and regulations for transport and handling of radioactive materials. UEX developed a procedure to detail requirements for exploration staff and others to ensure nuclear substances were shipped in compliance with regulatory requirements. The transportation instructions were provided for the shipment of Dangerous Good Class 7, Radioactive Materials.

The samples were held in approved pails and sealed shut with secure lids and met the requirements of the CNSC Packaging and Transport of Nuclear Substances Regulations. Each pail was weighed and the level of the radioactivity was measured in compliance with the transportation of dangerous goods regulations.

The sealed pails were temporarily stored outside the core shacks at the Raven Camp. Once a week, the shipment of radioactive samples was transported by road from the camp directly to SRC's lab in Saskatoon. The pails were shipped in a closed vehicle under the exclusive use rules by UEX's carrier, J.P. Enterprises Inc., based in La Ronge, Saskatchewan. In Golder's opinion, there was little chance of tampering of samples as they were shipped directly to the lab from the camps.

12.2 Geochemical Analyses

Analytical Procedures

The resource data set uses U_3O_8 assay by Inductive Coupled Plasma – Optical Emission Spectrometry ("ICPOES") as the primary analytical method and ICP Total Digestion for lower grade samples (<1,000 ppm U).

On arrival at the SRC laboratory, all samples were received and sorted into their matrix types and received radioactivity levels. The samples were then dried overnight at 80ºC in their original bags and then jaw crushed until more than 60% of the material is less than 2 mm in size. A 100 g sub sample was split using a riffler, which was then ground (either puck and ring grinding mill or an agate grind) until more than 90% is less than 106 µm. The grinding mills were cleaned between sample using steel wool and compressed air or in the case of clay rich samples, silica sand was used. The pulp was transferred to a labelled plastic snap top vial.

The samples were tested using validated procedures by trained personnel. All samples were digested prior to analysis by ICP and fluorimetry. All samples were subjected to multi-suite assay analysis, which included U, Ni, Co, As, Pb by total and partial digestions. During initial phases of exploration, assaying using three separate digestions methods were tested: Boron, Partial and Total. In early winter 2007, routine analysis of Boron was discontinued. Boron analyses exist for 73 holes up to HU-053 and RU-020, and for drill holes completed during the 2005 program which was managed by Cameco.

Total Digestions were performed on an aliquot of sample pulp. The aliquot was digested to dryness on a hotplate in a Teflon beaker using a mixture of concentrated $HF:HNO_3:HClO_4$. The residue was dissolved in dilute HNO_3 (SRC, 2007). Partial digestions were performed in an aliquot of sample pulp. The aliquot was digested in a mixture of concentrated HNO_3: HCl in a hot water bath then diluted to 15 ml with DI water. Fluorimetry was used on low uranium samples (<100 ppm) as a comparison for ICPOES uranium results. Uranium was determined on the partial digestion. An aliquot of digestion solution was pipetted into a 90% Pt 10% Rh dish and evaporated. A NaF/LiK pellet was placed on the dish and fused on a special propane rotary burner and then cooled to room temperature.

The SRC Geoanalytical laboratory reports uranium values in parts per million ("ppm"). In order to convert the uranium values to weight percent U_3O_8, the reported values were divided by a conversion factor of 10,000 (Uppm ÷ 10,000 = U%), and then multiplied by another conversion factor of 1.17924 to convert U% to U_3O_8% (in weight percent).

The reader is referred to the SRC's website (http://www.src.sk.ca/) for more details regarding the analytical techniques and sample handling procedures. SRC Geoanalytical Laboratories U_3O_8 Method Summary (McCready, 2007).

All samples were received and entered into the Laboratory Information Management System ("LIMS"). In the case of uranium assay by ICPOES for UEX, a pulp was already generated from the first phase of preparation and assaying (discussed above). UEX routinely assayed every sample above 1,000 ppm Uranium via ICP Total Digestion with ICPOES Uranium assay. A 1,000 mg of sample was digested for one hour in an HCl: HNO3 acid solution. The totally digested sample solution was then made up to 100 ml and a 10 fold dilution was taken for the analysis by ICPOES. Instruments were calibrated using certified commercial solutions. The instruments used were Perkin Elmer Optima 300DV, Optima 4300DV or Optima 5300DV. The detection limit for U_3O_8 by this method was 0.001%.

SRC management developed quality assurance procedures to ensure that all raw data generated in-house was properly documented, reported and stored to meet confidentiality requirements. All raw data was recorded on internally controlled data forms. Electronically generated data was calculated and stored on computers. All computer generated data was backed up on a daily basis. Access to samples and raw data was restricted to authorized SRC Geoanalytical personnel at all times. All data was verified by key personnel prior to reporting results. Laboratory reports were generated using SRC's LIMS.

Laboratory Audits

Two detailed laboratory audits were completed on the primary laboratory, SRC in Saskatoon, by UEX personnel. A laboratory audit was conducted on September 24, 2007 and a follow-up review on June 5, 2008. The laboratory audit covered all aspects of the sample preparation and analytical process. The review is documented with an appropriate action plan for non-compliance or suggested action items. SRC and UEX established an open relationship where the external QA/QC program and their interpretation of the laboratory's internal QC program are discussed on a regular basis. The laboratory was also visited by Kevin Palmer and Esther Bordet of Golder on July 9, 2008.

Assay Grades vs. Uranium Equivalent Grades

UEX has a sufficient quantity of downhole probe data and geochemical assays to calculate a uranium equivalent grade (eU_3O_8%) value. This was achieved by comparing geochemistry composites from mineralized holes to the same depth corrected probe composites to determine a correlation formula. Calculation of equivalent uranium values from downhole probe data in the absence of geochemical assays is an accepted industry standard procedure.

Prior to examining the relationship between the results obtained from the probe versus actual grade, a number of background steps were preformed on the selected dataset. A number of software applications were written within Mathworks Matlab software to first correct depth, and then extract average probe counts on selective intervals of geochemical composites for comparison (Walcott, pers. comm., 2009).

The probe data was first depth corrected using hand held SPP2 scintillometer measurements. The depth correction was achieved using an autocorrelation function. This function compares the signals and determines where the maximum amount of coherency is. Once this value was obtained, the offset or lag was then used to adjust the probe data to the correct depth as obtained using the SPP2 data and subsequently output to a separate file for use in the second stage. Using the depth corrected probe data, the average counts were then extracted from selected composites used in the 2008 resource estimates, and compared with their respective geochemistry grades.

Using the depth corrected probe data, the average counts were extracted from selective composite intervals, and compared with their respective grades. The average counts were then plotted against U_30_8%. Extreme outliers were removed prior to fitting the data. A number of mathematical functions were utilised to find a best representation for the dataset.. Two separate formulas were used for drill holes within the Horseshoe deposit area due to a limited number a data points, and erratic readings within the lower grade material. A quadratic equation was used for radiometric readings above 1,000 counts per second (cps) and a linear equation was chosen for radiometric readings below 1,000 cps as shown in Equation 1 and 2, respectively. In the Raven deposit area, Equation 3 was used.

Equation 1 *(Horseshoe):* $f(z) = 1.475*10^{-13} z^3 - 1.966*10^{-9} z^2 + 7.255*10^{-5} z + 0.01606$

Equation 2 *(Horseshoe):* $f(z) = 8.011*10^{-5} z + 0$

Equation 3 *(Raven):* $f(z) = 1.455 e^{-(\frac{z-1.259}{1.004})} - 0.5835 e^{-(\frac{z-1.23}{1.495})}$

**where z = average probe counts*

At Horseshoe Northeast, 48 samples composited at 1 m intervals were above the cut-off grade of 0.02% eU_3O_8 and used for the resource calculation. The highest estimated value is 0.13% eU_3O_8 and a windowed area of <0.35% eU_3O_8 was selected for data review (Figure 12.1). In this case, much of the probing data underestimates the geochemistry data, therefore establishing a conservative approach to grade estimation for the lost samples (Figure 12.2).

At Raven West, 112 samples composited at 1 m intervals were above the cut-off grade of 0.02% eU_3O_8 and used for the resource calculation. Only RU-205 contained probe grade composites above the 0.02% eU_3O_8 cut-off. The highest estimated value is 0.28% eU_3O_8 and a windowed area of <0.50% eU_3O_8 was selected for data review (Figure 12.2).

In this case, the geochemistry and probe composites seem to show less scatter than for the Horseshoe data. It is evident from this graph that, at around 0.50% e U_3O_8, the probe grades underestimate the geochemical grades.

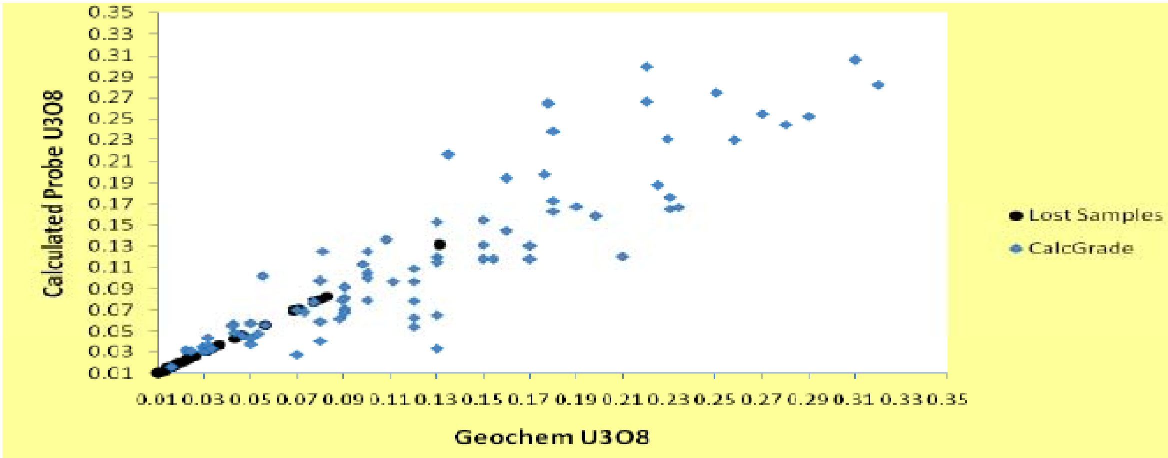


Figure 12.1: Horseshoe Composite Geochemistry U_3O_8vs. Composite Probe Grade eU_3O_8 2008 Drill Holes (<0.35% U_3O_8)

The black circle represents the spread of results calculated from lost holes HU-344 and HU-347. The blue diamonds represent the composites for the 2008 data.

The high degree of scatter in both plots is mainly due to the variability of comparing uranium mineralization from outside the drill hole (probe) to inside the drill hole (geochemistry). The nodular REDOX- and veinlet-styles of mineralization common at Horseshoe and Raven (Rhys, Horn, Baldwin and Eriks, 2008) indicate a high degree of variability in grade and thickness. A much larger dataset may help in producing a tighter fit to the formula. Golder believed that the formulas used here to predict grades for lost intervals of core are adequate to estimate grades in areas of Horseshoe and Raven.

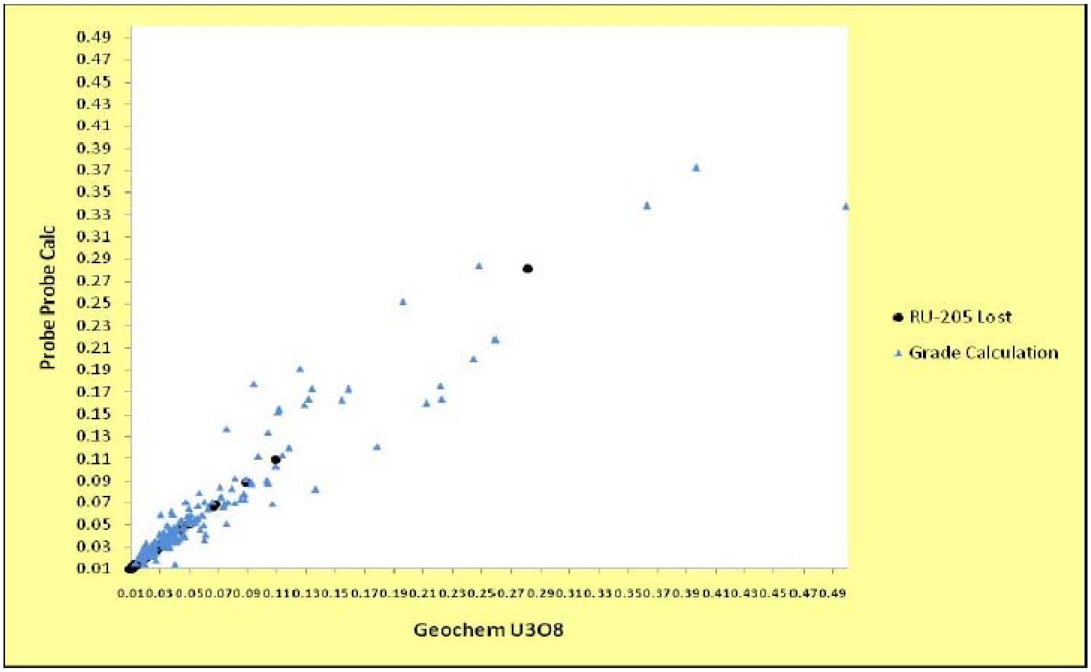


Figure 12.2: Raven Composite Geochemistry U_3O_8 vs. Composite Probe Grade eU_3O_8 2008 Drill Holes (<0.50% U_3O_8)

The black circles represent the spread of results calculated from lost hole RU-205. The blue triangles represent the composites for the 2008 data.

12.3 Dry Bulk Density Samples

In order to obtain bulk density estimates, UEX, under Golder's guidance, took a large selection of samples for dry bulk density measurement. These samples were systematically selected from different mineralized zones and a proportionately valid sample distribution of all rock types and alteration types, including different intensities of clay alteration.

Prior to September 1, 2008 a total of 2,615 samples from 33 holes underwent dry bulk density testing from Horseshoe and Raven. There were 1,845 samples from 33 Horseshoe (HU) holes and 770 samples from four Raven (RU) holes.

A further 1,109 samples, with a particular emphasis on the Raven deposit, underwent dry bulk density testing during the period from September to June 2009, bringing the total number to 3,724 analyses. There are now results for 2,198 samples from 39 Horseshoe (HU) holes and 1,526 samples from 19 Raven (RU) holes with good spatial and lithological spread.

Average dry bulk density for Horseshoe and Raven lithologies is 2.48 g/cm^3. The density statistics by rock type are listed in Table 12.1 and Table 12.2 for Horseshoe and Raven, respectively. A total of 643 samples from 109 holes underwent dry bulk density testing from West Bear.

Table 12.1: Horseshoe Bulk Density (g/cm^3) Statistics Grouped by Lithology

Horseshoe					
Rock	**Count**	**Mean**	**Median**	**Minimum**	**Maximum**
ARKQ	1,455	2.47	2.50	1.45	3.14
CARK	66	2.73	2.75	2.34	2.86
CLAY	12	1.88	1.78	1.33	2.45
DIAB/DIOR	14	2.71	2.73	2.27	2.85
GOUG	2	1.98	1.98	1.75	2.21
PEGM	94	2.37	2.41	1.89	2.65
PEL0	7	2.41	2.38	2.22	2.64
QZIT	450	2.53	2.55	2.02	2.83
SPL0	6	2.57	2.53	2.44	2.75
UX	92	2.49	2.49	1.75	2.95
Total	**2,198**	**2.48**	**2.52**	**1.33**	**3.14**

Table 12.2: Raven Bulk Density (g/cm^3) Statistics Grouped by Lithology

Rock	Count	Mean	Median	Minimum	Maximum
ARKQ/S	301	2.43	2.51	1.11	2.64
BX	10	1.98	1.99	1.74	2.32
CARK	413	2.44	2.42	1.98	2.93
GRAN	17	2.32	2.40	1.64	2.58
PEGM	53	2.41	2.44	1.58	2.89
PEL0	61	2.56	2.62	1.92	2.76
QZIT	632	2.54	2.55	1.44	2.65
SPL0	39	2.50	2.50	2.24	2.67
Total	**1,526**	**2.48**	**2.53**	**1.11**	**2.93**

Analytical Methods

Dry bulk density samples were collected from half split core retained in the core box after geochemical sampling, since the dry bulk density process requires wax coating of the samples, which would affect the geochemical analysis. An approximately 7 cm to 15 cm piece of half split core was submitted for each analysis. Samples were tagged and placed in sample bags on site, then shipped to SRC. Once received by SRC, samples are weighed dry and then covered in an impermeable barrier and then reweighed. The samples are then submersed in room temperature water and reweighed. The dry bulk density is calculated and reported.

As shown in Figure 12.3 below, there is no correlation between grade and dry bulk density. The regression curve is flat. However, above 3% eU_3O_8, there is a small inflection associated with a weak positive correlation between eU_3O_8 grade dry bulk densities.

There is a strong negative correlation with logged proportions of clay in the core and bulk density. Table 12.3 details the uranium grade ranges and specific gravity. Those samples not assayed for uranium are typically sitting distal to mineralization in less altered rock.

Table 12.3: Average Dry Bulk Densities (g/cm^3) by Grade Bins

U_3O_8% Grade Range	Number of Samples	SG Average	U_3O_8% Average
Not assayed	539	2.58	Barren
Assay to 0.05%	1,885	2.47	0.02%
0.05% to 0.1%	385	2.47	0.07%
0.1% to 1%	770	2.45	0.33%
>1%	145	2.48	2.26%
Total	**3,724**	**2.48**	**0.21%**

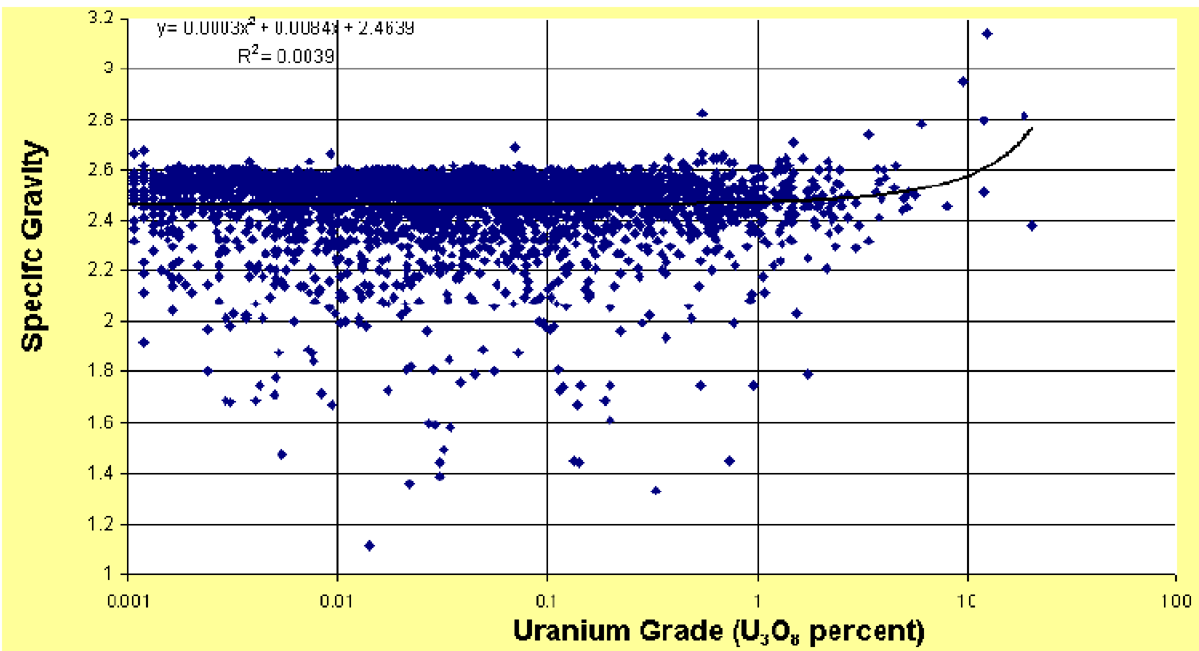


Figure 12.3: Logarithmic Plot of Dry bulk Density vs. Uranium Grade in Corresponding Geochemical Samples

SRC conducted 170 repeat analyses whereby in each batch at least one sample was repeated in every 40 samples. The repeats for this period were completed at a ratio of one repeat to 14 routine samples. All repeats passed the internal QC limit of +/- 0.02 g/cm^3. The sample repeats have a strong positive correlation for both the period prior to September 2008 (Figure 12.4) and the period from September 2008 to June 2009 (Figure 12.5).

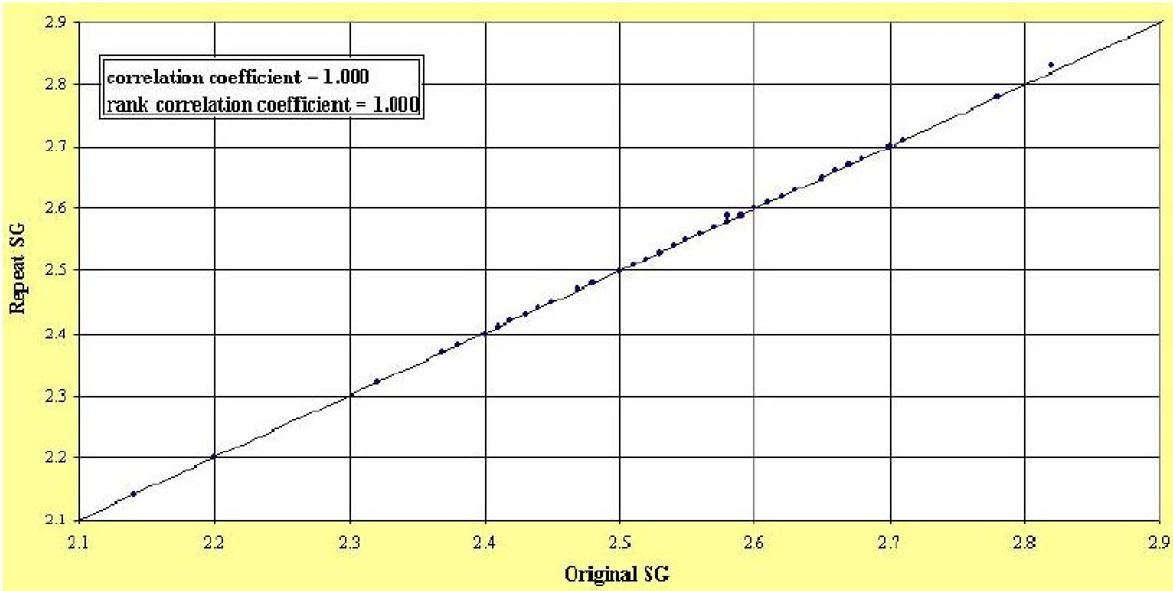


Figure 12.4: Quantile – Quantile Plot of Laboratory Bulk Density Replicates for Batches Submitted for all Seasons prior to September 2008

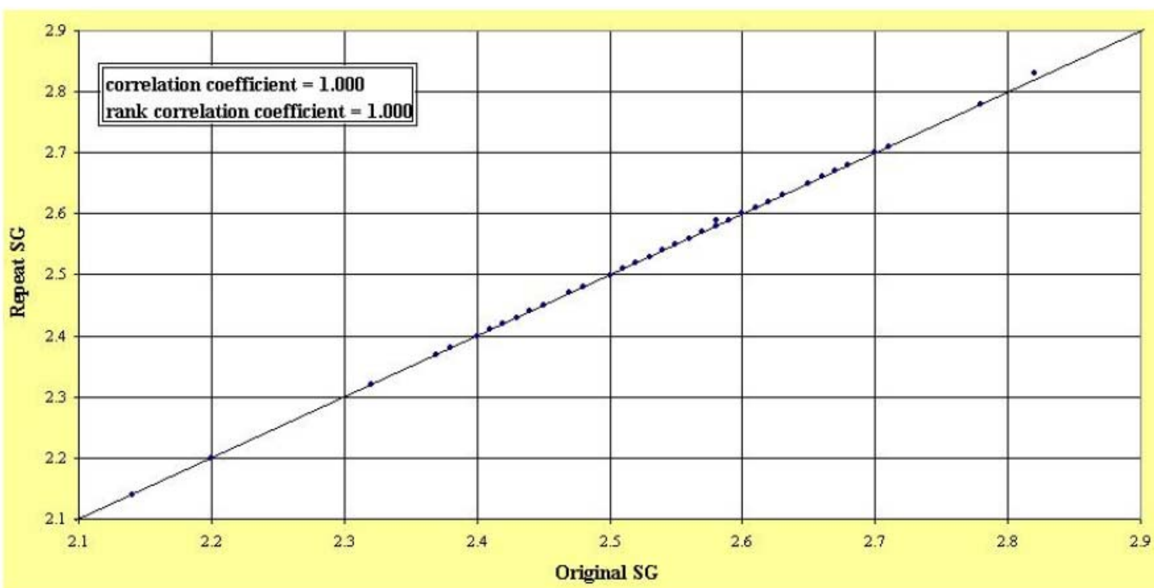


Figure 12.5: Quantile – Quantile Plot of Laboratory Bulk Density Replicates for Batches Submitted between September 2008 and June 2009

As a check, prior to September 2008 a total of 52 samples, or 1 in 50, underwent wet bulk density measurements in parallel with dry bulk density measurement. The average wet density of the selected sample was 2.61 g/cm^3 and the difference between the corresponding dry densities averaging 2.53 g/cm^3 is 2.8%. One known standard, a piece of granite, was used for the wet density measurements and the three results were in the acceptable range of 2.71 g/cm^3 +/- 0.01 g/cm^3.

During the period from September 2008 to June 2009, a total of 51 samples, or 1 in 22, underwent wet density measurements in parallel with the dry bulk density measurement. The average wet density of the selected samples was 2.54 g/cm^3 and the difference between the corresponding dry densities, which average 2.47 g/cm^3, is 2.8%. One known standard, a piece of granite, was used for the wet density measurements and the eleven results were in the acceptable range of 2.71 g/cm^3 +/- 0.01 g/cm^3.

13 Data Verification

Section 13.1 was taken directly from UEX's November 12, 2008 N.I. 43-101 report entitled "Technical Report on the Geology of, and Drilling Results from, the Horseshoe and Raven Uranium Deposits, Hidden Bay Property, Northern Saskatchewan" by Rhys et al. (2008). Minor updates to include the winter 2008/2009 drill program have been made and comments inserted where appropriate.

Rhys et al (2008) provides a full description of the UEX Horseshoe and Raven QA/QC. A review of the UEX QA/QC program by Golder indicated that the program is working and meets industry standards.

The data verification at West Bear was very similar to that described below for Horseshoe and Raven and is documented in reports by Palmer and Fielder (2009a, 2009b).

13.1 QA/QC

As part of UEX's quality improvement programs ("UEX Batch Acceptance Procedure"), a rigorous QA/QC program was implemented during the 2007 summer drilling program and continues to be followed. All drill core samples are submitted to the SRC laboratories in Saskatoon for geochemical analysis. Inserted into each drill core sample batch submitted to SRC are a total of 20 samples for analysis. Sixteen samples are sawed half core drill samples and four QA samples, which include a blank, a duplicate and two standard samples. The standard samples inserted into each batch are a commercially available standard (certified reference material), a blank, a field duplicate and a round robin pulp. Results are documented in Table 13.1 and Table13.2. Most drill holes at both the Horseshoe and Raven deposits that were completed under the management of UEX have been completed under this program. Prior to the implementation of this program, only blank samples were submitted routinely throughout the 2006 and early 2007 drilling programs. Additional QA/QC samples have been taken from the drill holes that were drilled prior to the UEX Batch Acceptance Procedure being implemented to improve the confidence in the earlier sampling. SPP2 radiometric readings have also been compared to the geochemical assays and a good correlation was noted.

UEX's has a full-time data administrator who routinely reviews assay batches returned from the laboratory as per the Batch Acceptance Procedure. The procedure is used to provide a standard process for reviewing QA/QC and accepting batches of geochemical assays from the laboratory on the Horseshoe-Raven exploration project.

Table 13.1: Summary of the Horseshoe and Raven QC Results Reporting period 2005 to September 2008

QA/QC Sample	Number	Outside	Percentage Outside of Tolerance
CG515 Standard ICP	2016	0	0%
Blank (ICP)	1033	6	0.6%
Field Duplicates	228	11	5% (outside of 30% precision)
Lab Replicates (ICP)	1098	0	0%
Lab Replicates (ICPOES)	404	1	0.2%
BL-2 (ICP) Standard	210	0	0
BL-3 (ICP) Standard	180	0	0
BL-4 (ICP) Standard	334	0	0
BL-4A (ICP) Standard	232	0	0
UEX08 (ICP) Standard	9	0	0
BL-1 (ICPOES) Standard	17	0	0
BL-2 (ICPOES) Standard	255	0	0
BL-2A (ICPOES) Standard	159	0	0
BL-3 (ICPOES) Standard	259	0	0
BL-4 (ICPOES) Standard	332	3	1%
BL-4A (ICPOES) Standard	615	0	0
BL-5 (ICPOES) Standard	7	0	0
ICP vs. ICPOES assay compilation	4,575	3	0.1%

Table 13.2: Summary of the Horseshoe and Raven QC Results Reporting Period September 2008 to June 2009 (Baldwin 2009)

QA/QC Sample	Number	Outside	Percentage Outside of Tolerance
CG515 Standard ICP	879	0	0%
Blank (ICP)	261	1	0.4%
Field Duplicates	30	3	10% (outside of 30% precision)
Lab Replicates (ICP)	516	0	0%
Lab Replicates (ICPOES)	116	0	0%
BL-2 (ICP) Standard	5	0	0%
BL-4A (ICP) Standard	520	1	0.2%
UEX08 (ICP) Standard	516	5	1.0%
BL-2 (ICPOES) Standard	16	0	0%
BL-2A (ICPOES) Standard	25	0	0%
BL-3 (ICPOES) Standard	6	0	0%
BL-4A (ICPOES) Standard	251	0	0%
UEX08 (ICPOES) Standard	144	1	0.7%
ICP vs. ICPOES assay compilation	696	4	0.6% (outside 10% precision)

In all cases, results outside of acceptable limits have been followed up through checking results from the batch with the laboratory or having the analysis repeated. In the case of the error repeating, the core was re-split and the new sample submitted for analysis.

Analysis of standards for the period 2005 to September 2008 indicates that results were acceptable (within three standard deviations from the mean) for 100% of 965 standards submitted via U ppm ICP Total Digestion, and 1,641 or 99.8% of the 1,644 standards submitted via the ICPOES U_3O_8 assay technique. Assay comparisons between three different assay techniques revealed a strong positive correlation for U ppm and U_3O_8.

Analysis of standards for the period September 2008 to June 2009 indicates that results were acceptable (within three standard deviations from the mean) for 1913 or 99.6% of 1,920 standards submitted via U ppm ICP Total Digestion and 441 of the 442 standards submitted via the ICPOES U_3O_8 assay technique. Assay comparison between different assay techniques revealed a strong positive correlation for U ppm and U_3O_8.

Laboratory replicates correspond to a pulp analyzed in replicate as part of the laboratory's internal QC measures to ensure reproducibility of assay results over time. Replicates also serve as a validation tool for batches with identified problems in either standards or blanks. The laboratory replicates are found to be in acceptable limits with a correlation coefficient close to one (R2> 0.999) and have very low dispersion for ICP and ICPOES analytical techniques.

13.2 Golder Data Verification

In order to verify that the data in the UEX database was acceptable for the January 2009 West Bear and July 2009 Horseshoe and Raven Mineral Resource Estimates, Golder reviewed the transfer of data from logging through to the final database. The assay data file supplied to Golder was reviewed against assay data obtained directly from SRC, UEX's primary laboratory. The data verification was carried out by Esther Bordet (G.I.T.) and Samuelle Gariepy (G.I.T) under the direction and by Kevin Palmer (P.Geo.), all of Golder. No restrictions were placed on Golder during the data verification process.

Golder data verification for West Bear was very similar to that described below for Horseshoe and Raven and is documented in reports by Palmer and Fielder (2009a, 2009b).

In the database, there are a total of 619 drill holes: 376 for Horseshoe and 243 for Raven. This includes 158 new drill holes which have been added to the database since the completion of the previous estimates for Horseshoe and Raven in January 2009. These include 102 drill holes in Horseshoe drilled in summer 2008 and early 2009, and 56 drill holes in Raven drilled in early 2009.

Drill core results provided by UEX to Golder for the use in the mineral resource estimate included:

- Drill hole collar position data (electronic format);
- Downhole in-hole survey data (hard copy and electronic); and
- Sample assay, sample lithological, drill core recovery and sample bulk density data.

As part of Golder's verification checks for previously reported estimates, Kevin Palmer, P.Geo., and Esther Bordet, G.I.T., of Golder visited the property between July 10 and 11, 2008. Kevin Palmer had previously visited the site from July 23 to 25, 2007.

During these site visits, a selection of drill logs were compared to original stored core samples, logging and sampling procedures were reviewed and 21 Horseshoe and 27 Raven collar positions were independently verified by a hand-held Garmin eTrex GPS. Also, during the site visit, a total of 11 Horseshoe and 5 Raven samples from the remaining half core were collected and later sent to SRC for analysis.

13.3 Logging and Sampling Procedure Review

During Golder's site visit, the logging and sampling procedure were reviewed with the UEX geologist on site and were found to be consistent as those described in Section 11.

The logging and sampling procedure review at West Bear was very similar to that described below for Horseshoe and Raven and is documented in reports by Palmer and Fielder (2009a, 2009b).

13.3.1 Collar Position

During Golder's site visit, 48 drill hole collars were surveyed using a hand-held Garmin eTrex GPS. The surveys were taken when the GPS indicated a minimum of 7 m accuracy. Golder's surveys were then compared to the collar positions in the UEX database. No significant differences were found between the survey collar positions provided by UEX and the GPS surveys complete by Golder.

No significant differences were noted between the GPS readings and the collars in the supplied database as indicated in Table 13.3 and Table 13.4.

As part of the data verification for the 2008 estimate, collar positions from the UEX database were checked against the original Tri-City surveys by selecting randomly approximately 20% of the holes (86 holes) in the Horseshoe and Raven database. The verification of collar positions was conducted by visual checking of the database against original documents supplied by Tri-City. One error was noted in Horseshoe and Raven database, RU-096, out of the 86 collars reviewed. This was corrected prior to the estimate being completed.

Prior to the July 2009 estimate for Horseshoe and Raven, the collar data from UEX database was checked against Tri-City surveys by selecting a random selection of three holes from Horseshoe and three from Raven. No errors were found. In addition, the updated July 2009 collar database was compared to the database used in the previous estimates. The minor differences that were noted were either corrected or were due to new information becoming available.

Table 13.3: Horseshoe Collars, Comparison between Golder GPS and UEX Database

BHID	GPS			Survey			Difference		
	Easting	Northing	Elevation	Easting	Northing	Elevation	Easting	Northing	Elevation
HU-005	574,235	6,446,789	432	574,237	6,446,785	433	-2	4	-1
HU-016	574,298	6,446,822	432	574,297	6,446,821	434	1	1	-2
HU-019	574,270	6,446,917	442	574,270	6,446,914	434	0	3	8
HU-032	574,286	6,446,831	435	574,281	6,446,832	434	5	-1	1
HU-050	574,360	6,446,884	437	574,359	6,446,883	435	1	1	2
HU-051	574,229	6,446,829	434	574,222	6,446,831	433	7	-2	1
HU-053	574,399	6,446,750	432	574,403	6,446,752	428	-4	-2	4
HU-055	574,236	6,446,819	432	574,234	6,446,822	433	2	-3	-1
HU-067	574,423	6,446,880	432	574,428	6,446,877	431	-5	3	1
HU-069	574,430	6,446,802	432	574,432	6,446,802	428	-2	0	4
HU-070	574,109	6,446,902	432	574,111	6,446,900	430	-2	2	2
HU-078	574,540	6,446,883	435	574,541	6,446,881	430	-1	2	5
HU-085	574,385	6,446,872	431	574,387	6,446,870	433	-2	2	-2
HU-086	574,206	6,446,777	433	574,200	6,446,783	433	6	-6	0
HU-097	574,213	6,446,912	441	574,208	6,446,906	434	5	6	7
HU-100	574,179	6,446,861	433	574,177	6,446,861	432	2	0	1
HU-112	574,190	6,446,949	432	574,195	6,446,953	435	-5	-4	-3
HU-188	574,032	6,446,828	432	574,036	6,446,829	429	-4	-1	3
HU-208	574,246	6,446,961	435	574,254	6,446,963	434	-8	-2	1
HU-235	574,102	6,446,957	429	574,100	6,446,958	431	2	-1	-2
HU-239	574,492	6,446,685	431	574,499	6,446,689	426	-7	-4	5

Table 13.4: Raven Collars, Comparison between Golder GPS and UEX Database

BHID	GPS			Survey			Difference		
	Easting	Northing	Elevation	Easting	Northing	Elevation	Easting	Northing	Elevation
RU-001	573,025	6,446,326	438	573,025	6,446,327	441	0	-1	-3
RU-002	573,017	6,446,375	444	573,017	6,446,373	444	0	2	0
RU-005	573,088	6,446,370	440	573,081	6,446,358	438	7	12	20
RU-007	573,075	6,446,388	439	573,078	6,446,387	441	-3	1	-2
RU-009	573,084	6,446,426	440	573,075	6,446,418	445	9	8	-5
RU-010	572,974	6,446,264	437	572,976	6,446,265	439	-2	-1	-2
RU-013	573,083	6,446,312	435	573,085	6,446,316	434	-2	-4	1
RU-016	572,953	6,446,425	455	572,953	6,446,398	450	0	28	5
RU-023	573,195	6,446,428	437	573,194	6,446,430	435	1	-2	2
RU-027	573,067	6,446,457	455	573,071	6,446,456	447	-4	1	8
RU-030	573,015	6,446,397	450	573,014	6,446,391	446	1	6	4
RU-032	573,001	6,446,447	442	573,002	6,446,460	451	-1	-13	-9
RU-036	572,985	6,446,373	449	572,986	6,446,375	446	-1	-2	3
RU-048	572,960	6,446,358	450	572,960	6,446,360	447	0	-2	3
RU-066	573,207	6,446,360	432	573,212	6,446,360	434	-5	0	-2
RU-075	573,157	6,446,464	433	573,157	6,446,458	437	0	6	-4
RU-078	572,916	6,446,419	450	572,916	6,446,421	452	0	-2	-2
RU-084	573,144	6,446,533	435	573,143	6,446,522	442	1	11	-7
RU-087	572,915	6,446,318	449	572,914	6,446,314	447	1	4	2
RU-090	573,173	6,446,503	433	573,176	6,446,500	438	-3	3	-5
RU-109	572,963	6,446,486	454	572,938	6,446,490	456	-2	-4	-2
RU-110	573,233	6,446,403	430	573,234	6,446,405	431	-1	-2	-1
RU-111	572,887	6,446,384	446	572,888	6,446,383	451	-1	1	-5
RU-114	572,902	6,446,265	444	572,905	6,446,262	442	-3	3	2
RU-118	573,258	6,446,418	431	573,260	6,446,424	431	-2	-6	0
RU-122	573,287	6,446,431	437	573,290	6,446,429	432	-3	2	5
RU-128	572,872	6,446,241	438	572,874	6,446,247	444	-2	-6	-6

13.3.2 Downhole Surveys, Collar and Lithology Review

Prior to carrying out the July 2009 estimate, the downhole survey and lithology data were checked against the original survey files and logs and against the 2008 database used for the previous estimates. Golder checked out the validity of the modelling database against lithology log sheets and downhole survey data supplied by UEX in paper and electronic format. No errors were noted in the new data and the minor differences between the old and new databases were due to updated information.

In-hole downhole surveys for the UEX Horseshoe and Raven drill holes included dip and azimuth readings obtained from a Reflex EZ-Shot® downhole survey tool. The digital readings from this instrument are recorded on paper logs and corrected to true north prior to input into the database.

During the verification for the previous estimates a total of 1,208 entries in the survey data file were checked against the paper logs. A total of 19 errors, mainly in bearing, were noted and corrected.

Two entries out of the 1,990 lithology entries checked did not have a lithology recorded. No other transcriptions errors were noted. No significant discrepancies were noted when comparing the core to the drill logs during the site visits.

The July 2009 downhole survey data from UEX database was checked against original survey file by selecting randomly five holes from Horseshoe and three from Raven. The verification of survey data was conducted by visual checking of the database against original documents. Some systematic errors were noted. UEX reviewed all of the entries, including those used in the earlier estimates and corrected the errors.

The lithology data from UEX database was checked against original log by randomly selecting three drill holes at Horseshoe and three at Raven. No errors were found.

13.3.3 Assay and Bulk Densities Databases

The assay data supplied to Golder by UEX consisted of those carried out by Cameco until 2005 and those carried out by UEX from 2006 to 2009. Original assay certificates in electronic format were provided directly to Golder by SRC.

The previous data verification consisted of those carried out by Cameco until 2005 and those carried out by UEX from 2006 to 2008. Four differences were noted out of the 808 Cameco assays, based on a review of the assay certificates supplied to Golder by SRC.

Original assay certificates for the UEX assaying issued by SRC were imported into an Access database and compared to the assay file supplied by UEX. A total of 24,083 U_3O_8 sample values were checked for the Horseshoe and Raven deposits, which represent all of the supplied samples. A total of 1,459 differences were noted, of which 1,251 were due to differences in the sample identifier. The other 208 differences were due to input errors.

Golder also received the original bulk density certificates from SRC to review the Horseshoe and Raven density data file. Two errors were noted among the 2,615 results that were checked, which represent the bulk densities estimated for Horseshoe and Raven.

The July 2009 data verification was carried out on assay values obtained from sampling carried out by UEX from September 2008 to 2009. The 2009 database was checked against the 2008 database and the assays from 2008 to 2009 campaign were checked against the original SRC files.

The 2009 database was compared to the 2008 database. Some differences were noted. These were mainly due to re-sampling or the use of an additional significant figure when converting U to U_3O_8. All the differences were satisfactorily explained. No differences in density were noted.

A total of 12,103 U_3O_8, sample values were checked for the Horseshoe and Raven deposits, which represent all of the summer 2008 and winter 2009 samples. A total of 964 differences were noted. These were primarily due to UEX not using a consistent formula for converting U to U_3O_8.

Golder also received the original bulk density certificates from SRC to review the Horseshoe and Raven density data file. A total of 1,317 values were checked and no error was noted.

13.3.4 Independent Samples

During the site visits in 2007 and 2008, a total of 15 samples were collected from the remaining half core for Horseshoe and Raven and submitted to SRC for assay analysis. These samples were to provide an independent verification of U_3O_8, mineralization on the Horseshoe and Raven deposits. Each sample was analyzed by total digestion ICP Analysis. The assay values for the Golder samples vs. the UEX original samples are provided in Table 13.5. Differences in the assays values are probably due to the sample size difference between the Golder samples and the UEX samples. The Golder samples for Horseshoe and Raven were between 7 cm and 16 cm in length, whereas the UEX samples average was 70 cm. The samples do confirm the presence of U_3O_8 mineralization at Horseshoe and Raven.

Table 13.5: Independent Samples taken by Golder at Horseshoe and Raven

Golder		Original	
Sample ID	U_3O_8 (%)	Sample ID	U_3O_8 (%)
G79037	0.100	87855	2.110
G79038	0.933	65068	0.348
G79040	0.295	69154	0.395
G79041	1.438	62657	0.520
G79042	4.339	89598	7.600
G019190	1.179	2007-901	0.528
G019191	5.742	G-2008-111	1.650
G019192	2.334	G-2008-145	1.880
G019193	2.134	G-2008-73	1.860
G019194	0.011	20071964	0.015
G019195	0.947	2007-1404	0.849
G013038	0.971	2007-1826	0.977
G013039	0.004	2007-1826	0.015
G013040	0.002	2007-397	0.002
G013041	6.732	2007-227	1.780
G013042	0.498	2007-1961	0.238

13.3.5 Conclusion

The Golder data verification indicates that the logging, sampling, shipping, sample security assessment, analytical procedures, inter-laboratory assay validation and validation by different techniques are comparable to industry standard practices.

All the differences noted between the UEX database and Golder's verification were either reconciled or corrected by UEX prior to the use of the databases. The databases are considered acceptable for Mineral Resource estimation of the West Bear, Horseshoe and Raven deposits.

14 Adjacent Properties

The Hidden Bay property occurs in the prolific eastern Athabasca uranium district and deposits on the adjacent Rabbit Lake and McClean Lake properties, which are currently operated by Cameco and AREVA, respectively, have produced more than 200 million pounds of U_3O_8, (Jefferson et al., 2007). As a result, the local area has significant infrastructure, including two currently operating uranium mills of which the closest, Rabbit Lake, is 4 km from the Horseshoe and Raven deposits.

15 Mineral Processing and Metallurgical Testing

The mineral processing and metallurgical testing for Horseshoe and Raven is described below. Mineral processing and metallurgical testing at West Bear is documented in a separate Melis Engineering Ltd. report (2011).

15.1 Horseshoe and Raven Metallurgical Testing – Phase I

15.1.1 Test Composites

The initial testwork on the Horseshoe-Raven deposits, was based on test composites representative of three separate zones or areas of unique grade distribution within known parts of the Horseshoe deposit as established from drilling to the end of 2006. The samples for the metallurgical composites, using assay rejects, were selected with a range of grades which together comprised typical grades of each of the domains/zones selected. This provided the following composites for testing:

- Composite A - representative material from intervals >1.5 m minimum mining width in the Horseshoe A zone,
- Composite B - representative material from intervals >1.5 m minimum mining width in the Horseshoe B zone,
- Composite HU16 - representative material from the high grade HU-16 intersection, and
- Composite Main – a blend of Composite A and Composite B to be used in the initial testing.

Key analyses of these composites are listed in Table 15.1.

Table 15.1: Horseshoe Phase I Testwork – Composites Elemental Analyses

Analyte	Composite A	Composite B	Composite HU16	Composite Main
U_3O_8	0.414	0.297	4.07	0.330
As	0.0048	0.0083	0.0785	0.0063
Co	0.0025	0.0039	0.0278	0.0031
Mo	0.0014	0.0008	0.0012	0.0015
Ni	0.0045	0.0060	0.0175	0.0054
Pb	0.012	0.011	0.086	0.011
Se	<0.0001	<0.0001	<0.0001	<0.0001

The elemental analyses of the composites show that the Horseshoe uranium deposit is relatively low in deleterious elements such as arsenic, molybdenum, selenium, and base metals; hence the Horseshoe mineralization was expected to present minimal processing and environmental difficulties.

Mineralogical examination of core samples, prepared as polished thin sections, was completed to determine the mode of occurrence of uranium in the Horseshoe and Raven deposits. Five uranium carriers were identified: the primary uranium mineral (first deposited) is comprised of uraninite [UO_2]. Secondary uranium minerals, all formed as a result of alteration and remobilization of uranium from the uraninite, are comprised of the uranium silicates boltwoodite [$(H_3O)K(UO_2SiO_4)\cdot H_2O$], uranophane [$Ca[(UO_2)SiO_3(OH)]_2\cdot 5H_2O$] and coffinite [$U(SiO_4)_{1-x}(OH)_{4x}$]; these are accompanied by minor amounts of carnotite [$K_2(UO_2)_2V_2O_8\cdot 1\text{-}3H_2O$].

15.1.2 Leaching Tests

Eight leach tests were completed, five on the Main Composite and one on each test composite, Composite A (representing Zone A), Composite B (representing Zone B) and Composite HU-16 (a high grade composite). Test results are summarized in Table 15.2.

Table 15.2: Horseshoe Phase I Testwork – Summary of Leach Results

Test No.	Composite	% U_3O_8		Weight Loss %	Final Pregnant Solution g U_3O_8/L	% U_3O_8 Extraction		
		Feed	Residue			8 hours	12 hours	24 hours
RH1	Main	0.32	0.008	3.1	1.54	97.6	98.0	98.1
RH2	Main	0.33	0.004	3.1	1.60	98.9	98.8	98.8
RH3	Main	0.32	0.008	-	1.59	97.6	97.8	-
RH4	Main	0.31	0.009	-	1.55	97.5	97.6	-
RH5	Main	0.35	0.009	-	1.73	97.9	98.0	-
RH6	A	0.41	0.008	-	2.02	97.7	98.1	-
RH7	B	0.34	0.014	-	1.62	97.6	97.5	-
RH8	HU-16	5.02	0.046	-	23.05	98.8	99.1	-

The above results show that the uranium in the Horseshoe zone is easily leached under relatively mild atmospheric leach conditions. Leach extractions of 98% can be achieved under the following conditions:

- Grind K_{80} of 90 to 200 µm (both yielded acceptable extractions),
- 12 hour leach retention time,
- Free acid level of 10 g H_2SO_4/L, representing acid additions of approximately 50 kg H_2SO_4/t, and
- A 475 mV redox/potential controlled with $NaClO_3$ at addition rates of 0.5 to 1.0 kg $NaClO_3$/t.

15.1.3 Waste Treatment and Tailings Neutralization

The pregnant leach solution and residues from the eight leach tests were retained to generate waste raffinate and leach residue for waste treatment testing. The pregnant leach solution was contacted in two stages with organic to generate raffinate. The leach residues were re-pulped to 45% solids (w/w) with pH 2 sulphuric acid. Spent regeneration solution was simulated by making up a sodium carbonate solution at pH 9 and spiking it with sodium molybdate to 2.8 g Mo/L.

The combined raffinate and simulated spent regeneration solution, the main liquid waste products produced in a uranium circuit, were neutralized with lime and treated in three stages with intermediate removal of waste precipitates by decanting/filtration of treated liquor. The first stage was at pH 4 with the addition of ferric sulphate (for molybdenum removal) and barium chloride (for radium removal), the second stage (treatment of the supernatant from the first stage) was at progressively increasing pH (5.0, 7.5 and 10.2) with further additions of ferric sulphate and barium chloride, and the third stage (treatment of the second stage supernatant) was at pH 7.5 (adjusted with sulphuric acid) and further additions of barium chloride. The treated water from the third stage was filtered through a Millipore filter to provide treated effluent for analysis.

The resulting treated effluent analyses are summarized in Table 15. 3. Other than molybdenum, all elements are within typical environmental guidelines for treated effluent discharge. Minor adjustments to treatment conditions can reduce molybdenum to the anticipated 0.5 mg/L guideline.

Table 15.3: Horseshoe Phase I Testwork – Treated Effluent Analysis and MMAMC Limits

Parameter	Unit	Treated Effluent	MMAMC Limits
pH	-	7.12	6.0 – 9.5
As	mg/L	0.0043	0.5
Mo	mg/L	1.51	0.5 (typical)
Ni	mg/L	0.013	0.5
Pb	mg/L	0.00077	0.2
Se	mg/L	0.011	Variable, depends on loading
U	mg/L	0.0123	2.5
Zn	mg/L	0.0081	0.5

The re-pulped leach residue was neutralized to pH 4 with lime then the first stage waste precipitate slurry, adjusted to pH 7 with lime, was added to the neutralized residue and the pH increased to 7.5 with further addition of lime. The second stage waste precipitate slurry was then added and the pH increased to 9.5 with lime to provide neutralized tailings slurry for analysis, and to provide treated tailings for supernatant aging tests. Results of key contaminants are summarized in Table 15.4 below.

Table 15.4: Horseshoe Phase I Testwork – Neutralized Tailings Supernatant Aging Tests

Parameter	Unit	Day 1	Day 2	Day 14	Day 30	Day 61
pH	-	7.1	7.54	7.65	7.81	7.91
Ra^{226}	Bq/L	n/a	n/a	n/a	n/a	9.1
As	mg/L	0.0496	0.0383	0.0378	0.0518	0.0565
Cu	mg/L	0.0122	0.0065	0.0028	0.0046	0.0056
Mo	mg/L	54.3	n/a	74.7	80	75.2
Ni	mg/L	0.0264	0.012	0.0111	0.01	0.0093
Pb	mg/L	0.0479	0.0126	0.00164	0.00865	0.00460
Se	mg/L	0.007	0.008	0.007	0.009	0.010
U	mg/L	0.0778	0.114	0.616	0.774	0.709
Zn	mg/L	0.0052	0.0095	0.0045	0.0023	0.003

As expected, molybdenum and residual uranium levels in the tailings supernatant increase upon aging, as does radium, but excess tailings supernatant water would be re-used and/or treated in the mill process and waste treatment circuits under normal operating conditions. The low levels of other contaminants in tailings supernatant confirm that the Horseshoe mineralization is low in deleterious elements.

15.2 Horseshoe and Raven Metallurgical Testing – Phase II

15.2.1 Test Composites

A second phase of testwork, which included comminution tests, confirmation leach tests and further waste treatment tests, was carried out on Horseshoe and Raven composites prepared from purpose-drilled HQ core. The following composites were prepared from the Horseshoe zone for testing:

- Composite AH – a high grade composite from the A zone,
- Composite AL – a low grade composite from the A zone,
- Composite BEH – a high grade composite from the BE zone,
- Composite BEL – a low grade composite from the BE zone, and
- Combined Horseshoe Composite - weighted blend of the four sub-composites.

The zones were defined as per the June 26, 2007 mineralization and lithological map shown in Figure 15.1.

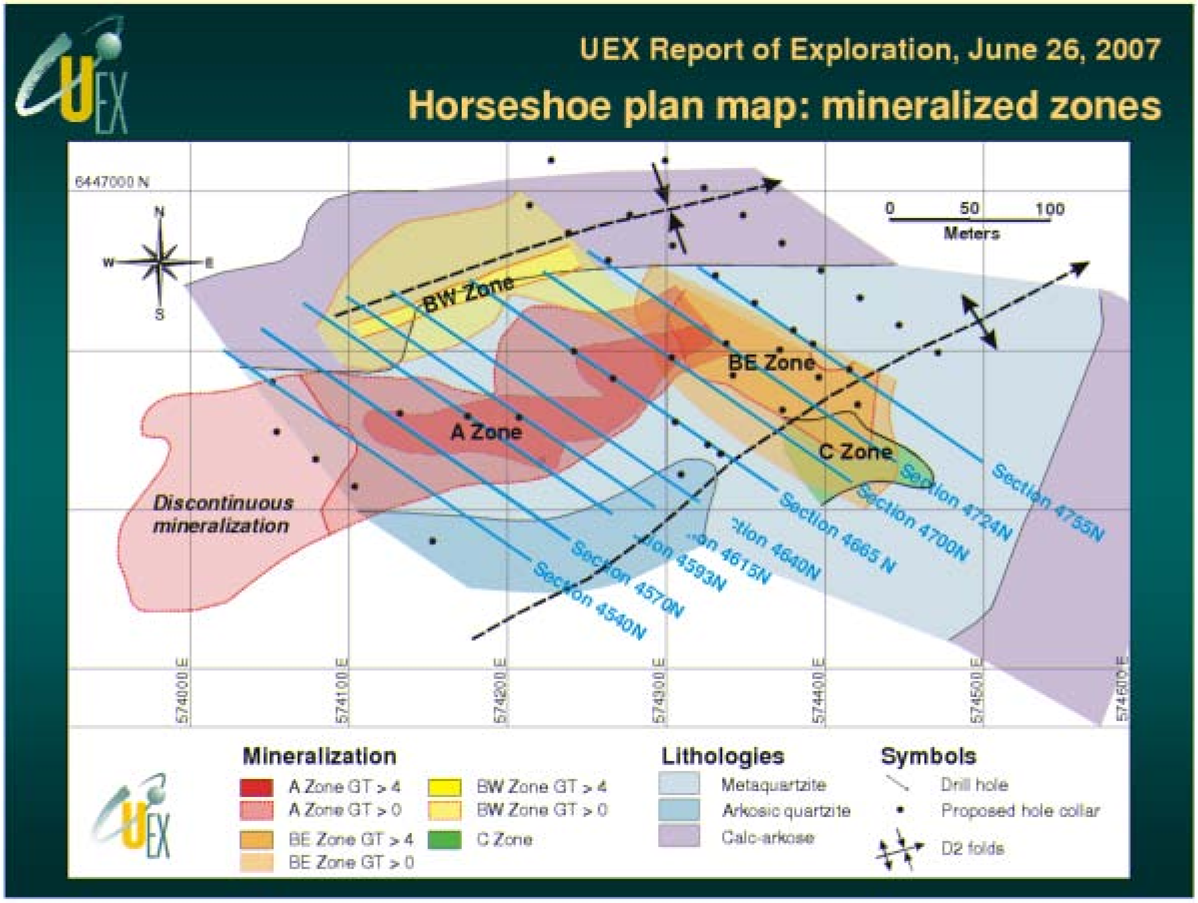


Figure 15.1: UEX Corporation Hidden Bay Project – Horseshoe-Raven Deposits – Phase II Testwork Horseshoe Plan Map Showing Mineralized Zones

A single composite was prepared from the Raven zone for testing:

- Composite RU-130 - representative material from DDH RU-130 in the Raven zone.

Table 15.5 summarizes the more significant assays for the test composites.

Table 15.5: Horseshoe and Raven Phase II Testwork – Summary of Composite Assays

Composite	Assay, %						
	U_3O_8	As	Co	Mo	Ni	Pb	Se
AH	2.18	0.014	0.0065	0.0025	0.0042	0.060	< 0.0030
AL	0.38	0.0052	0.0035	0.0018	0.0036	0.016	< 0.0030
BEH	0.31	0.0055	0.0020	0.0024	0.0042	0.016	< 0.0030
BEL	0.054	< 0.0040	0.0010	0.0016	0.0034	0.006	< 0.0030
RU-130	0.21	< 0.0060	0.0029	0.0025	0.00841	0.008	< 0.0030

The elemental analyses of the composites confirm that the Horseshoe and Raven uranium deposits are relatively low in deleterious elements such as arsenic, molybdenum, selenium and base metals.

15.2.2 Comminution Tests

Nine composites were submitted for ball mill Bond Work Index ("BWI") and two composites were submitted for SPI® determinations. The Horseshoe and Raven composites were categorized as medium in hardness from the perspective of SAG milling, with an average SPI value of 69 minutes. The ball mill BWI indices were all within a tight range of 16.1 kWh/t to 17.7 kWh/t with an average value of 16.7 kWh/t, showing very little variation across the deposits and characterizing the Horseshoe-Raven mineralization as moderately hard for ball mill grinding.

15.2.3 Leaching Tests

Five leach tests were completed, one test each on Phase II Horseshoe composites AH, AL, BEL and BEH, one test on the Raven composite RU130, and one bulk leach test on the overall Horseshoe composite. Test results are summarized in Table 15.6.

Table 15.6: Horseshoe and Raven Phase II Testwork – Summary of Leach Test Results

Test No.	Composite	% U_3O_8		Weight Loss %	Final Pregnant Solution g U_3O_8/L	% U_3O_8 Extraction	
		Feed	Residue			8 hours	12 hours
3H-1	AH	2.26	0.021	3.2	11.18	99.0	99.1
3H-2	AL	0.41	0.004	2.9	1.92	99.2	99.2
3H-3	BEL	0.06	0.004	2.2	0.29	96.1	94.2
3H-4	BEH	0.30	0.004	4.1	1.44	98.5	98.9
3R-5	RU-130	0.21	0.005	4.8	1.00	97.3	97.8
Bulk	Horseshoe	0.48	0.014	N/A	2.71	98.9	97.6

The Phase II leach test results confirmed the leach conditions and leach results in the first phase of testwork, confirming the Horseshoe-Raven mineralization is easily leached under relatively mild atmospheric leach conditions. Leach extractions of 98% or greater can be achieved for Horseshoe and Raven mineralization under the following leach conditions:

- A grind K_{80} of approximately 145 µm;
- A temperature of 50ºC;
- A free acid concentration of 10 g H_2SO_4/L, representing an acid consumption of 45 kg H_2SO_4/t;
- An ORP of 500 mV (Ag/AgCl), representing a sodium chlorate consumption of 0.6 kg $NaClO_3$/t; and
- A retention time of 8 hours to 12 hours.

15.2.4 Tailings and Effluent Treatment

The pregnant leach solution and residue from the bulk leach test were retained to generate waste raffinate and leach residue for waste treatment testing. Treatment conditions were similar to those used in the Phase I treatment test.

The combined raffinate and simulated spent regeneration solution, the main liquid waste products produced in a uranium circuit, were neutralized with lime and treated in three stages with intermediate removal of waste precipitates by decanting/filtration of treated liquor. The first stage was at pH 4 with the addition of ferric sulphate (for molybdenum removal) and barium chloride (for radium removal), the second stage (treatment of the supernatant from the first stage) was at progressively increasing pH (5.0, 7.5 and 10.2) with further additions of ferric sulphate and barium chloride, and the third stage (treatment of the second stage supernatant) was at pH 7.5 (adjusted with sulphuric acid) and further additions of barium chloride. The treated water from the third stage was filtered through a 0.45 micron Millipore filter to provide treated effluent for analysis.

The re-pulped leach residue was neutralized to pH 4 with lime then the first stage waste precipitate slurry, adjusted to pH 7 with lime, was added to the neutralized residue and the pH increased to 7.5 with further addition of lime. The second stage waste precipitate slurry was then added and the pH increased to 9.5 with lime to provide neutralized tailings slurry for analysis, and to provide treated tailings for supernatant aging tests.

The specific gravity of the generated tailings was measured at 2.59 t/m^3. The tailings K_{80} (80% passing size) was 136 µm and the K_{50} (50% passing size) was 54 µm.

From acid base accounting measurements, SGS Lakefield classified the tailings as having an uncertain potential for acid generation due to the relatively low carbonate concentration in the tailings. It should be noted that the sulphide sulphur concentration was <0.01% S, the NP/AP (neutralization potential/acid potential) ratio was 31.6, and the net acid generation of the tailings was <0.1 kg H_2SO_4/tonne, all suggesting that it is improbable the tailings would generate acid.

Tailings supernatant aging tests resulted in elevated levels of radium and molybdenum in the supernatant. This was expected, and confirms that, like all uranium tailings supernatants, excess tailings water would be re-used and/or treated in the mill process and waste treatment circuits under normal operating conditions.

The resulting treated effluent analyses are summarized in Table 15.7. The treated effluent generated from the Horseshoe mineralization met all typical regulatory limits for treated effluent discharge.

Table 15.7: Horseshoe Phase II Testwork – Treated Effluent Analysis and MMAMC Limits

Parameter	Unit	Treated Effluent	MMAMC Limits
pH	-	7.12	6.0 – 9.5
As	mg/L	0.0067	0.5
Cu	mg/L	0.0032	0.3
Mo	mg/L	0.0115	0.5 (typical)
Ni	mg/L	0.0077	0.5
Pb	mg/L	<0.00002	0.2
Se	mg/L	0.009	Variable, depends on loading
U	mg/L	0.015	2.5
Zn	mg/L	0.003	0.5
Ra^{226}	Bq/L	0.02	0.37

16 Mineral Resource and Mineral Reserve Estimates

16.1 Introduction

Uranium deposits on the Hidden Bay property for which historical and more recent N.I. 43-101 mineral resources have been estimated include the West Bear, Horseshoe and Raven deposits.

Resources estimated to N.I. 43-101 standards for the West Bear, Horseshoe and Raven deposits on the Hidden Bay property are documented by Lemaitre (2006), Palmer (2007 and 2008) and Palmer and Fielder (2009a, 2009b).

Based on discussions with UEX, Golder understood that there are no known environmental, permitting, socio-economic, marketing or political issues. The extent to which mining, metallurgical infrastructure or other factors will affect the estimate is also not known at this time.

16.2 Mineral Resource Estimate for the West Bear Deposit

A 2006 N.I. 43-101 compliant Indicated Resource estimate was prepared by Roger Lemaitre, P.Eng., P.Geo., of Cameco Corporation (Lemaitre, 2006). This estimate (first resource estimate) was based on 101 drill holes totalling 2,793 m which were completed during the 2005 sonic drilling program at West Bear. The estimate utilized a cut-off grade of 0.15% U_3O_8 and a grade/thickness parameter of 0.45 m % U_3O_8, outlining an Indicated resource of 45,600 tonnes, grading 1.385% U_3O_8 and totalling 1.391 million pounds U_3O_8. The deposit also contains 0.34% nickel, 0.11% cobalt, and 0.50% arsenic within the same resource outlines. The supporting technical report (Lemaitre, 2006) is dated March 2, 2006 and is available for review at www.sedar.com. Due to subsequent drilling and infill sampling, this resource is no longer current.

Based on the results of the 2007 infill and step-out drilling, a mineral resource estimate by Kevin Palmer, P.Geo., of Golder Burnaby, BC dated December 11, 2007 (second resource estimate) incorporating the results from both the 2005 and 2007 winter sonic drilling programs, outlined an Indicated resource of 73,800 tonnes, grading 1.004% U_3O_8 and totalling 1.614 million pounds of U_3O_8 at West Bear in the high-grade main deposit area. The resource estimate was calculated using a cut-off grade of 0.05% U_3O_8 utilizing a geostatistical-block model technique with ordinary kriging methods and Datamine.

During the calculation of the 2007 resource estimate, it was noted that for many areas in the 2005 drilling, sampling sometimes extended either only to the limits of mineralization, and some areas of anomalous radioactivity extended beyond the limits of sampling.

As a result, additional sampling was undertaken to sample low-grade (0.01 to 0.05% U_3O_8) material not previously sampled during the 2005 and 2007 winter sonic programs, both to better define the limits of mineralization for resource purposes, and to assess the potential distribution of special waste in future preliminary assessments, pre-feasibility and feasibility studies. The January 2009 West Bear resource estimate (third resource estimate) utilized the results from this program.

The January 2009 West Bear resource estimate was also prepared by K. Palmer, P.Geo., of Golder and the methodology is documented in a technical report by Palmer and Fielder (2009) available for review at www.sedar.com. The resource calculation utilized the results from 216 drill holes totalling 6,400 m, which were completed during 2004, 2005 and 2007 sonic drilling programs. The resource estimate was calculated using a minimum cut-off grade of 0.01% U_3O_8 utilizing a geostatistical-block model technique with ordinary kriging methods and Datamine.

Detailed sections on exploratory data analysis, resource block model, interpolation plan, mineral resource classification, mineral resource tabulation and block model verification for the January 2009 West Bear resource estimate are documented in reports by Palmer and Fielder (2009a, 2009b).

The resource reported below reflects the remodelling of the deposit after re-sampling of drill core was undertaken to better define mineralization outlines. The changes in volume, with corresponding decrease in grade with respect to the December 2007 Indicated mineral resource estimate, reflect incorporation of lower grade material in the new resource outlines. All the current mineral resources at West Bear are classified as Indicated. Details at different cut-off levels are provided in Table 16.1.

Table 16.1: **January 2009 Indicated Mineral Resources (Capped) at the West Bear Deposit with Tonnes and Grade at Various U_3O_8 Cut-off Grades**

			Grade				Contained Metal			
Cut-off Grade (%U_3O_8)	Tonnes	Density (g/cm^3)	U_3O_8 (%)	Ni (%)	Co (%)	As (%)	U_3O_8 (lbs)	Ni (lbs)	Co (lbs)	As (lbs)
0.01	209,700	1.99	0.358	0.22	0.08	0.22	1,655,000	1,030,000	375,000	1,005,000
0.02	188,100	1.99	0.397	0.24	0.09	0.23	1,646,000	975,000	355,000	974,000
0.03	113,000	2.02	0.645	0.28	0.10	0.32	1,605,000	704,000	254,000	786,000
0.04	85,300	2.03	0.843	0.32	0.11	0.37	1,585,000	600,000	203,000	694,000
0.05	**78,900**	**2.04**	**0.908**	**0.33**	**0.11**	**0.38**	**1,579,000**	**569,000**	**185,000**	**662,000**
0.10	76,100	2.04	0.939	0.33	0.10	0.38	1,574,000	547,000	173,000	640,000
0.15	70,300	2.04	1.005	0.33	0.11	0.39	1,558,000	505,000	165,000	604,000
0.20	63,800	2.04	1.09	0.32	0.11	0.40	1,532,000	453,000	152,000	559,000
0.25	57,300	2.04	1.187	0.31	0.11	0.41	1,500,000	397,000	138,000	514,000
0.30	52,100	2.04	1.279	0.31	0.11	0.42	1,468,000	360,000	127,000	482,000
0.35	47,800	2.04	1.365	0.30	0.11	0.42	1,437,000	319,000	115,000	443,000
0.40	43,600	2.05	1.461	0.31	0.11	0.44	1,403,000	295,000	107,000	418,000

16.3 Mineral Resource Estimate for the Horseshoe Deposit

The July 2009 Horseshoe Mineral Resource Estimate was prepared by Kevin Palmer, P.Geo., and reviewed by David Farrow, Pr.Sci.Nat., both of Golder, Burnaby, BC. The mineral resource estimation utilized the 376 diamond drill holes (119,400 m from holes HU-001 to HU-358, HS-001 and HO-001 to HO-016) drilled between 2005 and 2009 that are described in preceding sections, which test the deposit at 7.5 m to 30 m drill centres. The mineral resource was estimated using a minimum cut-off grade of 0.02% U_3O_8 utilizing a geostatistical block model technique with ordinary kriging ("OK") methods and Datamine Studio 3.

16.3.1 Exploratory Data Analysis

In order to carry out the evaluation of the property, a digital database for collars, surveys, lithology, density, recoveries and assays, suitable for importing into Datamine was provided in an Excel format by UEX. UEX also provided 28 separate 3D mineralized envelopes which were interpreted to include most of the mineralization above a 0.05% U_3O_8 cut-off on the Horseshoe deposit. However, the subzones, Q01 to Q03 and G01 and G02 on the northeast, are of a lower grade than the areas previously defined and a 0.02% U_3O_8 cut-off was used as a guide when defining the envelopes. Each envelope has been given a numeric and an alphanumeric code (Table 16.2). Envelope A1H contains the higher grade core within A1. This unit was separated out as initial statistic indicating the possibility of more than one population within A1.

Table 16.2: Numeric and Alphanumeric Codes for Horseshoe Mineralized Envelopes

Alphanumeric	A1H	A1	A2	A3	A4	A5	BW	BE	C	S1	S2	S3
Numeric	100	101	102	103	104	105	201	301	401	501	502	503
Alphanumeric	**M01**	**M02**	**M03**	**M04**	**M05**	**M06**	**M07**	**M08**	**M09**	**M10**	**M11**	
Numeric	601	602	603	604	605	606	607	608	609	610	611	
Alphanumeric	**Q01**	**Q02**	**Q03**	**G01**	**G02**							
Numeric	701	702	703	801	802							

Exploratory Data Analysis and Variography were carried out using Supervisor software.

Data

The database is comprised of a total of 376 drill holes and includes Gulf drill holes HO-01 to HO-16, HS-001 and UEX drill holes HU-001 through to HU-358. The Horseshoe database contains 23,100 data entries of % U_3O_8. There are also 2,199 dry bulk density measurements. The mineralized envelopes (all 28 subzones with cut-off grades at or above 0.02% U_3O_8) contain 8,481 data entries of % U_3O_8 and 1,283 bulk density measurements.

Bulk Density

Dry bulk densities were assigned to the individual subzones based on the mean value for that subzone. Subzones that had no values were assigned the mean value of all the mineralized envelopes. Table 16.3 lists the dry bulk densities for the different units.

Table 16.3: Dry bulk Densities for Horseshoe Deposit by Subzone

Subzone	A1H	A1	A2	A3	A4	A5	BW	BE	C	S1	S2	S3
Bulk Density (g/cm^3)	2.497	2.519	2.469	2.486	2.345	2.411	2.510	2.427	2.078	2.564	2.528	2.436
Subzone	**M01**	**M02**	**M03**	**M04**	**M05**	**M06**	**M07**	**M08**	**M09**	**M10**	**M11**	**M12**
Bulk Density (g/cm^3)	2.508	2.507	2.550	2.560	2.464	2.464	2.376	2.464	2.464	2.464	2.464	
Subzone	**G01**	**G02**	**Q01**	**Q02**	**Q03**							
Bulk Density (g/cm^3)	2.549	2.464	2.542	2.540	2.464							

The bulk density for Subzone C is lower than the others due to the highly altered nature of the subzone.

Geological Interpretation

Datamine string files were interpreted around a cut-off of 0.05% U_3O_8 for the majority of the deposit in order to provide an assessment of the mineralization by UEX. However, on the north eastern part of the deposit, it was necessary to reduce the cut-off to 0.02% U_3O_8. These strings were used to create 3D wireframes around the mineralized envelopes. All of the subzones, except for S3, dip to the south and are believed to be related to a pre-mineralization fault zone which has now been overprinted by alteration related to mineralization, and along and peripheral to which replacement and vein style mineralization is developed (Rhys et al., 2008).

The mineralized envelopes are strongly associated with the hematitic alteration halo. UEX generated 3D wireframes from the string files by UEX. These wireframes were subsequently verified for duplicate vertices, duplicate faces and empty faces in Datamine and are illustrated in Figure 16.1.

Golder reviewed the interpretation and verified that they were consistent with UEX's planned geological and mineral interpretation as described above.

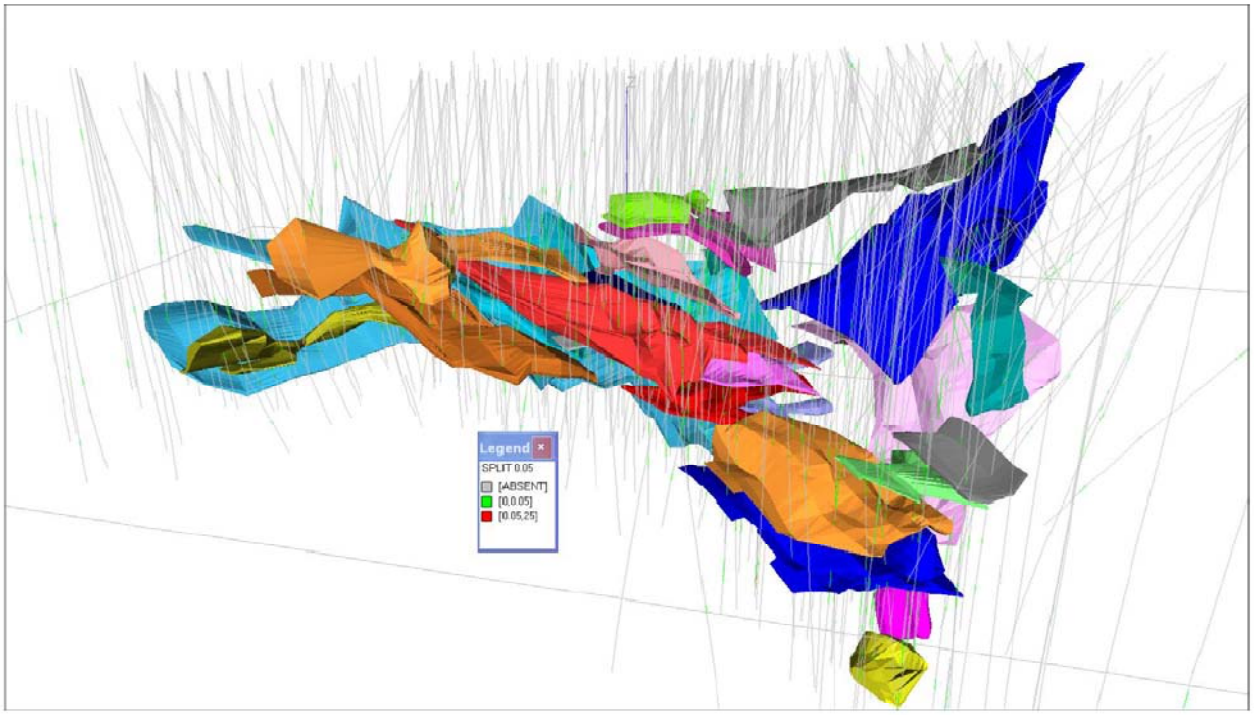


Figure 16.1: **Horseshoe Subzones with Drill Holes, Oblique Section looking North (Legend refers to % U_3O_8 in Drill Holes)**

Assays

Golder carried out a statistical review of the assay files from the 376 drill holes for the Horseshoe deposit . The statistics for the rock type indicate that the lithology coded UX contains the highest grade (Table 16.4). UX is applied to lithologies when the primary rock type has been altered and is no longer identifiable. The mean value for UX is 1.370% U_3O_8 with a median value of 0.392% U_3O_8. The highest grades in an identifiable rock type are found in the Arkosic Quartzite (“ARKQ”) with a mean value of 0.079% U_3O_8 and a median value of 0.008% U_3O_8. Lithologies with less than 10 samples have been removed from the table.

Table 16.4: Horseshoe Statistics for $\%U_3O_8$ by Lithology for Raw Data

Statistic		U_3O_8_PCT	ARKQ	CONG	DIAB	DIOR	GOUG	GRAN	PEGM	PEL0	QZIT	SPL0	UX
Samples		26,226	15,949	24	19	43	113	168	1,116	130	7,427	193	456
Minimum		0	0	0	0.001	0	0	0	0	0	0	0	0.003
Maximum		20.4	17.2	0.106	0.095	0.085	0.553	1.24	5.84	0.79	10.5	0.848	20.4
Mean		**0.069**	**0.079**	**0.019**	**0.02**	**0.008**	**0.022**	**0.043**	**0.044**	**0.038**	**0.027**	**0.034**	**1.37**
Std. Deviation		0.364	0.326	0.027	0.038	0.011	0.075	0.158	0.23	0.111	0.135	0.121	2.614
Coef. Of Var		5.282	4.11	1.447	1.875	1.476	3.447	3.667	5.22	2.917	5.053	3.536	1.908
Variance		0.133	0.106	0.001	0.001	0	0.006	0.025	0.053	0.012	0.018	0.015	6.832
Skewness		20.376	13.89	3.059	3.622	4.277	6.074	5.602	12.965	5.77	48.521	5.609	3.817
Grade at Percentile	10th	0.001	0.001	0.002	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.023
	20th	0.002	0.002	0.004	0.003	0.001	0.001	0.001	0.002	0.003	0.002	0.001	0.054
	30th	0.003	0.003	0.005	0.004	0.002	0.001	0.002	0.003	0.004	0.003	0.001	0.097
	40th	0.005	0.005	0.006	0.005	0.003	0.002	0.004	0.004	0.006	0.005	0.002	0.182
	Median	**0.007**	**0.008**	**0.007**	**0.005**	**0.004**	**0.003**	**0.005**	**0.006**	**0.009**	**0.007**	**0.003**	**0.392**
	60th	0.012	0.013	0.013	0.006	0.005	0.004	0.007	0.01	0.01	0.011	0.004	0.626
	70th	0.02	0.023	0.015	0.011	0.007	0.005	0.01	0.015	0.012	0.017	0.009	1.1
	80th	0.039	0.05	0.022	0.011	0.009	0.009	0.019	0.025	0.027	0.028	0.015	1.74
	90th	0.104	0.146	0.056	0.087	0.018	0.038	0.049	0.06	0.085	0.056	0.04	3.79
	95th	0.262	0.368	0.066	0.092	0.027	0.118	0.15	0.124	0.124	0.097	0.152	6.71
	97.5th	0.545	0.682	0.083	0.092	0.037	0.221	0.544	0.289	0.267	0.152	0.349	9.62
	99th	1.150	1.300	0.106	0.095	0.041	0.532	0.977	0.655	0.775	0.29	0.817	12

The basic statistics for the samples for each subzone are listed in Table 16.5 to Table 16.7.

Table 16.5: Statistics for % U_3O_8 by Main Subzones

	Statistic	%U_3O_8	A1	A1H	A2	A3	A4	A5	BE	BW	C
	Samples	8,481	712	350	443	235	129	116	876	1,859	108
	Minimum	0	0.001	0.001	0.001	0.001	0.005	0.002	0	0	0
	Maximum	20.4	3.45	20.4	3.91	4.12	4.87	0.848	3.87	9.62	2.94
	Mean	**0.202**	**0.135**	**1.492**	**0.282**	**0.281**	**0.302**	**0.138**	**0.203**	**0.235**	**0.233**
	Std. Deviation	0.638	0.239	2.381	0.484	0.466	0.582	0.158	0.282	0.567	0.494
	Coef. Of Var.	**3.163**	**1.775**	**1.596**	**1.714**	**1.659**	**1.924**	**1.138**	**1.381**	**2.412**	**2.117**
	Variance	0.407	0.057	5.671	0.234	0.217	0.338	0.025	0.079	0.321	0.244
	Skewness	11.789	5.761	3.73	3.87	3.898	5.016	2.3891	3.751	7.258	3.666
Grade at Percentile	10th	0.004	0.005	0.013	0.008	0.01	0.026	0.018	0.017	0.005	0.002
	20th	0.01	0.012	0.07	0.024	0.024	0.047	0.035	0.034	0.013	0.009
	30th	0.019	0.023	0.218	0.042	0.045	0.072	0.046	0.054	0.025	0.028
	40th	0.031	0.041	0.443	0.066	0.067	0.092	0.07	0.074	0.043	0.046
	Median	**0.049**	**0.062**	**0.719**	**0.097**	**0.103**	**0.118**	**0.085**	**0.101**	**0.066**	**0.056**
	60th	0.071	0.086	0.964	0.145	0.152	0.159	0.11	0.134	0.1	0.077
	70th	0.111	0.122	1.52	0.262	0.254	0.206	0.126	0.199	0.149	0.114
	80th	0.192	0.186	1.98	0.427	0.417	0.372	0.2	0.323	0.275	0.325
	90th	0.449	0.326	3.8	0.757	0.749	0.648	0.352	0.53	0.577	0.538
	95th	0.816	0.496	5.56	1.07	1.21	1.07	0.424	0.737	0.948	1.44
	97.5	1.42	0.742	8.15	1.53	1.47	1.86	0.636	0.897	1.65	1.75
	99th	2.56	1.2	12	2.47	1.94	2.96	0.736	1.3	2.59	1.92

Table 16.6: Statistics for % U_3O_8 by Minor Subzones

	Coef. Of Var.	M01	M02	M03	M04	M05	M06	M07	M08	M09	M10	M11
	Samples	274	44	110	163	36	49	80	29	46	12	24
	Minimum	0.000	0.010	0.006	0.001	0.019	0.005	0.000	0.019	0.007	0.032	0.005
	Maximum	1.240	0.427	0.424	0.630	0.352	0.828	0.790	1.100	0.282	0.865	0.249
	Mean	**0.102**	**0.087**	**0.075**	**0.056**	**0.075**	**0.126**	**0.102**	**0.128**	**0.069**	**0.191**	**0.059**
	Std. Deviation	0.144	0.081	0.072	0.087	0.063	0.187	0.137	0.170	0.056	0.216	0.053
	Coef. Of Var.	**1.406**	**0.929**	**0.955**	**1.552**	**0.841**	**1.492**	**1.347**	**1.332**	**0.815**	**1.131**	**0.895**
	Variance	0.021	0.007	0.005	0.008	0.004	0.035	0.019	0.029	0.003	0.047	0.003
	Skewness	3.564	2.310	2.268	3.714	2.224	2.493	3.275	3.937	2.021	1.858	2.407
Grade at Percentile	10th	0.008	0.014	0.011	0.004	0.021	0.017	0.002	0.027	0.017	0.035	0.017
	20th	0.017	0.026	0.024	0.005	0.032	0.023	0.007	0.040	0.031	0.038	0.021
	30th	0.029	0.031	0.036	0.013	0.042	0.029	0.031	0.048	0.037	0.045	0.035
	40th	0.043	0.046	0.047	0.019	0.045	0.039	0.055	0.059	0.046	0.046	0.038
	Median	**0.056**	**0.075**	**0.054**	**0.028**	**0.051**	**0.056**	**0.062**	**0.066**	**0.052**	**0.061**	**0.045**
	60th	0.075	0.084	0.069	0.044	0.056	0.063	0.083	0.069	0.060	0.105	0.052
	70th	0.108	0.105	0.081	0.058	0.079	0.098	0.111	0.116	0.065	0.298	0.056
	80th	0.132	0.119	0.103	0.077	0.090	0.141	0.125	0.156	0.100	0.314	0.061
	90th	0.223	0.168	0.146	0.120	0.128	0.370	0.221	0.284	0.123	0.347	0.112
	95th	0.384	0.212	0.256	0.157	0.230	0.625	0.330	0.334	0.176	0.515	0.152
	97.5	0.529	0.348	0.270	0.324	0.238	0.703	0.369	0.660	0.206	0.515	0.152
	99th	0.701	0.427	0.330	0.489	0.238	0.703	0.790	0.660	0.254	0.865	0.249

Table 16.7: Statistics for % U_3O_8 by Northeast Subzones

Coef. Of Var.		G01	G02	Q01	Q02	Q03
Samples		681	83	1,214	81	82
Minimum		0.000	0.000	0.000	0.002	0.004
Maximum		6.010	0.317	3.720	0.399	0.427
Mean		**0.095**	**0.039**	**0.046**	**0.051**	**0.039**
Std. Deviation		0.386	0.057	0.128	0.061	0.051
Coef. Of Var.		**4.077**	**1.445**	**2.748**	**1.202**	**1.305**
Variance		0.149	0.003	0.016	0.004	0.003
Skewness		9.395	2.893	15.773	2.806	4.647
Grade at Percentile	10th	0.001	0.003	0.004	0.007	0.009
	20th	0.002	0.005	0.006	0.012	0.015
	30th	0.004	0.008	0.010	0.020	0.019
	40th	0.006	0.012	0.015	0.023	0.021
	Median	**0.010**	**0.019**	**0.021**	**0.027**	**0.024**
	60th	0.017	0.024	0.028	0.030	0.027
	70th	0.030	0.037	0.040	0.055	0.034
	80th	0.060	0.059	0.057	0.082	0.044
	90th	0.164	0.103	0.090	0.120	0.073
	95th	0.364	0.147	0.137	0.145	0.117
	97.5	0.748	0.176	0.226	0.230	0.168
	99th	1.570	0.275	0.394	0.286	0.213

Subzone A1H has the highest grade with a mean of 1.492% U_3O_8 and a median value of 0.719% U_3O_8. Subzone A4 contains the next highest grades with a mean of 0.302% U_3O_8 and a median value of 0.118% U_3O_8. The histograms of the subzones with well defined histograms indicate that the % U_3O_8 population has a lognormal distribution. There is also the suggestion of more than one population within some of the subzones but they appear to have a significant overlap.

Capping

Capping of sample assays is applied to reduce the impact on the mineral resource estimate of high grade samples that are interpreted as not being part of the lognormal population outliers. Anomalous high grades are cut to the highest grade that would be regarded as being part of that population.

Lognormal histograms and log probability plots were reviewed to establish the capping level for each subzone. A total of 59 samples were cut from all of the subzones, with the most, seven, being cut from G01. The effect of the cutting and the subsequent compositing had the effect of reducing the co-efficient of variation (“CV”) to less than 1.50 for 22 out of the 28 subzones.

The effects of the capping and subsequent compositing are shown in Table 16.8.

Table 16.8: Effect of Capping and Compositing on Coefficient of Variation

Statistic	A1	A1H	A2	A3	A4	A5	BE	BW	C	S1	S2	S3
Uncut CV	1.78	1.6	1.71	1.66	1.92	1.14	1.39	2.41	2.12	3.66	2.48	1.86
Uncut Mean	0.135	1.492	0.282	0.281	0.302	0.138	0.203	0.235	0.233	0.231	0.322	0.267
Cut Mean	0.131	1.437	0.282	0.281	0.282	0.138	0.203	0.230	0.204	0.159	0.313	0.267
Cut CV	1.58	1.44	1.71	1.66	1.60	1.14	1.39	2.22	1.80	1.69	2.22	1.86
No. Cut	4	5	0	0	3	0	0	5	5	6	2	0
Capping Level	1.50	10.50			2.50			5.00	1.50	1.50	6.50	
Composite Cut Mean	0.131	1.437	0.282	0.281	0.282	0.138	0.203	0.230	0.204	0.159	0.313	0.267
Composite Cut CV	1.19	1.14	**1.37**	**1.24**	**1.31**	0.96	1.10	**1.85**	**1.56**	**1.34**	**1.66**	**1.27**
Statistic	**M01**	**M02**	**M03**	**M04**	**M05**	**M06**	**M07**	**M08**	**M09**	**M10**	**M11**	
Uncut Mean	0.102	0.087	0.075	0.056	0.075	0.126	0.102	0.128	0.069	0.191	0.059	
Uncut CV	1.41	0.93	0.96	1.55	0.84	1.49	1.35	1.33	0.82	1.13	0.90	
Cut Mean	0.102	0.087	0.075	0.054	0.075	0.102	0.088	0.128	0.069	0.191	0.059	
Cut CV	1.41	0.93	0.96	1.4	0.84	1.16	0.99	1.33	0.82	1.13	0.88	
No. Cut	0	0	0	6	0	5	4	0	0	0	0	
Capping Level				0.40		0.40	0.30					
Composite Cut Mean	0.103	0.087	0.075	0.054	0.075	0.102	0.088	0.128	0.069	0.191	0.059	
Composite Cut CV	1.04	0.72	0.70	1.18	0.69	1.02	0.81	0.98	0.52	0.84	0.59	
Statistic	**G01**	**G02**	**Q01**	**Q02**	**Q03**							
Uncut Mean	0.095	0.039	0.046	0.051	0.039							
Uncut CV	**4.08**	**1.45**	**2.75**	**1.20**	**1.31**							
Cut Mean	0.085	0.039	0.045	0.049	0.037							
Cut CV	3.32	1.45	2.29	1.10	1.05							
No. Cut	7	0	1	3	2							
Capping Level	2.50		1.50		0.20							
Composite Cut Mean	0.085	0.039	0.045	0.049	0.037							

Composites

Assays were composited to 1.0 m lengths, which is the 80th percentile of the lengths contained within the mineralized envelopes. The minimum composite length allowed is 0.15 m. The compositing method chosen in Datamine is the one whereby all samples are included in one of the composites. This is achieved by adjusting the composite length while keeping the length as close as possible to the 1.0 m.

Compositing was restricted to within individual subzones, based on codes assigned to the drill hole file. Compositing had the effect of reducing the CV in all 28 subzones (Table 16.7).

Spatial Analysis

Variography, using Supervisor software, was completed for % U_3O_8 assay samples for each individual subzone. Downhole variograms were used to determine nugget effect subsequently lognormal variograms were modelled to determine spatial continuity of % U_3O_8. In some of the subzones, it was not possible to develop anisotropic models and, where this was the case, isotropic models were developed. Minor subzones M02, M03, M05, M08, M09, M10 and M11 had insufficient data to establish variograms. In these cases, the modelled variograms obtained from subzone M06 were used. The North East subzones Q02 and Q03 also had insufficient data to establish variograms.

A two-structure spherical model was used to model most of the lognormal variograms. Tables 16.9 to 16.11 summarize the results of the variography.

Table 16.9: Variogram Parameters for Main Subzones

Subzone	Variable	Direction	Azimuth	Dip	Nugget	Sill C_1	Range A_1 (m)	Sill C_2	Range A_2 (m)
A1	U_3O_8	1	105	00	0.00	0.62	23.5	0.38	81.0
	U_3O_8	2	195	-45	0.00	0.62	23.5	0.38	33.5
	U_3O_8	3	015	-45	0.00	0.62	21.0	0.38	40.5
A1H	U_3O_8	1	120	-37	00.0	0.48	27.0	0.52	49.5
	U_3O_8	2	039	13	0.00	0.48	13.0	0.52	22.0
	U_3O_8	3	325	-50	0.00	0.48	6.0	0.52	22.0
A2	U_3O_8	1	090	00	0.00	1.00	41.5		
	U_3O_8	2	180	-10	0.00	1.00	44.5		
	U_3O_8	3	000	-80	0.00	1.00	12.0		
A3	U_3O_8	1	000	90	0.00	0.85	3.5	0.15	20.0
	U_3O_8	2	000	00	0.00	0.85	3.5	0.15	20.0
	U_3O_8	3	270	00	0.00	0.85	3.5	0.15	20.0
A4	U_3O_8	1	000	90	0.00	0.91	3.0	0.09	20.0
	U_3O_8	2	000	00	0.00	0.91	3.0	0.09	20.0
	U_3O_8	3	270	00	0.00	0.91	3.5	0.09	20.0
A5	U_3O_8	1	000	90	0.00	0.74	2.5	0.26	29.0
	U_3O_8	2	000	00	0.00	0.74	2.5	0.26	29.0
	U_3O_8	3	270	00	0.00	0.74	2.5	0.26	29.0
BE	U_3O_8	1	000	90	0.00	0.95	4.0	0.05	30.0
	U_3O_8	2	000	00	0.00	0.95	4.0	0.05	30.0
	U_3O_8	3	270	00	0.00	0.95	4.0	0.05	30.0
BW	U_3O_8	1	135	-30	0.00	0.69	8.0	0.31	63.0
	U_3O_8	2	045	00	0.00	0.69	14.5	0.31	42.0
	U_3O_8	3	315	-60	0.00	0.69	25	0.31	64.0
C	U_3O_8	1	000	90	0.00	0.69	3.0	0.31	13.0
	U_3O_8	2	180	00	0.00	0.69	3.0	0.31	13.0
	U_3O_8	3	090	00	0.00	0.69	3.0	0.31	13.0
S1	U_3O_8	1	207	07-	0.00	0.71	74.5	0.29	77.0
	U_3O_8	2	113	29	0.00	0.71	35.5	0.29	48.0
	U_3O_8	3	310	60	0.00	0.71	3.0	0.29	7.0
S2	U_3O_8	1	055	00-	0.00	0.42	2.0	0.58	13.0
	U_3O_8	2	145	-15	0.00	0.42	3.0	0.58	25.0
	U_3O_8	3	325	-75	0.00	0.42	1.0	0.58	3.5
S3	U_3O_8	1	316	-24	0.10	0.58	89.0	0.32	110.0
	U_3O_8	2	044	60	0.10	0.58	99.0	0.32	118.0
	U_3O_8	3	300	65	0.10	0.58	14.5	0.32	27.0

Table 16.10: Variogram Parameters for Minor Subzones

Subzone	Variable	Direction	Azimuth	Dip	Nugget	Sill C1	Range A_1 (m)	Sill C_2	Range A_2 (m)
M01	U_3O_8	1	140	-40	0.00	0.89	40.0	0.11	89.5
	U_3O_8	2	050	00	0.00	0.89	28.5	0.11	86.0
	U_3O_8	3	320	-50	0.00	0.89	25.0	0.34	61.0
M02	U_3O_8	1	000	90	0.00	0.66	2.0	0.34	31.0
	U_3O_8	2	000	00	0.00	0.66	2.0	0.34	31.0
	U_3O_8	3	270	00	0.00	0.66	2.0	0.34	31.0
M03	U_3O_8	1	000	90	0.00	0.66	2.0	0.34	31.0
	U_3O_8	2	000	00	0.00	0.66	2.0	0.34	31.0
	U_3O_8	3	270	00	0.00	0.66	2.0	0.34	31.0
M04	U_3O_8	1	065	00	0.00	0.64	10.5	0.36	17.5
	U_3O_8	2	335	-15	0.00	0.64	27	0.36	46.0
	U_3O_8	3	335	75	0.00	0.64	3.5	0.36	24.0
M05	U_3O_8	1	000	90	0.00	0.66	2.0	0.34	31.0
	U_3O_8	2	000	00	0.00	0.66	2.0	0.34	31.0
	U_3O_8	3	270	00	0.00	0.66	2.0	0.34	31.0
M06	U_3O_8	1	000	90	0.00	0.66	2.0	0.34	31.0
	U_3O_8	2	000	00	0.00	0.66	2.0	0.34	31.0
	U_3O_8	3	270	00	0.00	0.66	2.0	0.34	31.0
M07	U_3O_8	1	000	90	0.34	0.52	4.0	0.34	30.0
	U_3O_8	2	000	00	0.34	0.52	4.0	0.34	30.0
	U_3O_8	3	270	00	0.34	0.52	4.0	0.34	30.0
M08	U_3O_8	1	000	90	0.00	0.66	2.0	0.34	31.0
	U_3O_8	2	000	00	0.00	0.66	2.0	0.34	31.0
	U_3O_8	3	270	00	0.00	0.66	2.0	0.34	31.0
M09	U_3O_8	1	000	90	0.00	0.66	2.0	0.34	31.0
	U_3O_8	2	000	00	0.00	0.66	2.0	0.34	31.0
	U_3O_8	3	270	00	0.00	0.66	2.0	0.34	31.0
M10	U_3O_8	1	000	90	0.00	0.66	2.0	0.34	31.0
	U_3O_8	2	000	00	0.00	0.66	2.0	0.34	31.0
	U_3O_8	3	270	00	0.00	0.66	2.0	0.34	31.0
M11	U_3O_8	1	000	90	0.00	0.66	2.0	0.34	31.0
	U_3O_8	2	000	00	0.00	0.66	2.0	0.34	31.0
	U_3O_8	3	270	00	0.00	0.66	2.0	0.34	31.0

Table 16.11: Variogram Parameters for Northeast Subzones

Subzone	Variable	Direction	Azimuth	Dip	Nugget	Sill C1	Range A_1 (m)	Sill C_2	Range A_2 (m)
G01	U_3O_8	1	165	-65	0.34	0.32	4.0	0.34	70.0
	U_3O_8	2	075	00	0.34	0.32	10.5	0.34	37.0
	U_3O_8	3	345	-02	0.34	0.32	12.0	0.34	26.0
G02	U_3O_8	1	135	-70	0.37	0.40	4.0	0.23	22.5
	U_3O_8	2	045	00	0.37	0.40	2.5	0.23	10.0
	U_3O_8	3	315	-20	0.37	0.40	2.5	0.23	10.0
Q01	U_3O_8	1	315	-85	0.24	0.47	7.0	0.29	47.0
	U_3O_8	2	045	00	0.24	0.47	29.5	0.29	59.5
	U_3O_8	3	315	05	0.24	0.47	12.5	0.29	25.5
Q02	U_3O_8	1	315	-85	0.13	0.54	7.0	0.33	47.0
	U_3O_8	2	045	00	0.13	0.54	29.5	0.33	59.5
	U_3O_8	3	315	05	0.13	0.54	12.5	0.33	25.5
Q03	U_3O_8	1	315	-85	0.13	0.54	7.0	0.33	47.0
	U_3O_8	2	045	00	0.13	0.54	29.5	0.33	59.5
	U_3O_8	3	315	05	0.13	0.54	12.5	0.33	25.5

Subzone S3 has the largest range (A2, second structure) range of 118.0 m on an azimuth of 044° dipping -06°. A range of between 20 m and 45 m for the second structure appears to be common.

16.3.2 Resource Block Model

Block models were established in Datamine for all subzones. A standard block size of 5.0 m x 5.0 m x 2.5 m (Easting x Northing x Elevation) was used for the interpolation. This was based on the average sample spacing on the property. Sub-cells were allowed in order to improve the fill of the interpreted solids. The minimum cell sizes allowed were 1.0 m for Northing, 1.0 m for Easting and 0.5 m for the Elevation.

16.3.3 Interpolation Plan

The Horseshoe deposit model used the variable anisotropy search model available in Datamine. The dip and dip direction is calculated for each triangle used to make up the wireframe which contains the mineralized drill hole intersections. These two parameters are then interpolated into each block. During the grade interpolation process, the search ranges established during the variography process for each subzone is rotated for each block to match the interpolated dip and dip direction.

At Horseshoe, most of the blocks for U_3O_8, were interpolated during the first pass which was at the range of continuity of the variograms for all subzones except S2, where a search range of 25 m by 25 m by 5 m was used. A second pass at four times and a third at six times the sill range was required to interpolate % U_3O_8, into most of the subzones.

A third pass at eight times the sill range was required for subzone S2 to interpolate grades into all of the blocks. The grade interpolation plan is summarized in Table 16.12. A minimum of four samples and a maximum of 24 samples were used in the first and third pass. The minimum was set to three for the second and third pass. A minimum of two drill holes were used in the first pass and one in the second and third.

Table 16.12: Summary of Horseshoe Grade Interpolation Plan

Model Name	Minmod		
Dimensions	x	y	z
Parent Cell	5.0	5.0	2.5
Minimum sub cell	1.0	1.0	0.5
Model origin	537,300	6,446,400	-100
Total parent cells	450	350	250
Parent discretisation	2	2	1
Estimated attributes	**Attribute**	**Unit**	**Comment**
	OKTU308	%	Capped U_30_8 ordinary kriging
	ID2TU308	%	Capped U_3O_8 inverse distance squared
	NNTU308	%	Capped U_3O_8 nearest neighbour
	OKU308	%	U_30_8 ordinary kriging
	ID2U308	%	U_3O_8 inverse distance squared
	TRDIP	degrees	True dip
	TRDIPDIR	degrees	True dip direction
Assigned attributes	ZONA	Alphanumeric Subzone Code A1H, A1 to A5, BW, BE, C, M01 to M11, S1 to S3, G01 to G02 and Q01 to Q03	
	ZONN	Numeric Subzone Code 100, 101 to 105, 201, 301, 401, 601 to 611, and 501 to 503, 801 to 802 and 701 to 703	
	NSAMU	Number of samples used in interpolation	
	SVOLU	Search neighbourhood volume for U_3O_8	
	VARKU	Kriging Variance for U_3O_8	
	DENSITY	Density was assigned based on mean of samples of samples within subzone. Default of 2.451 g/cm^3 used for subzones with no samples	
	CATEGORY	Numeric Value for mineral resource category 1=Measured, 2=Indicated, 3=Inferred and 4=Exploration potential	
	CATA	Alpha numeric for Resource Categories	
	NSAMPANI	Number of samples used in interpolation TRIP and TRDIPDIR	
	SVOLANI	Search neighbourhood volume for TRDIP and TRDIPDIR	

16.3.4 Mineral Resource Classification

Several factors are considered in the definition of a resource classification:

1. CIM requirements and guidelines
2. Experience with similar deposits
3. Spatial continuity
4. Confidence limit analysis

The search volume was used as a guide to classify the Horseshoe deposit. Blocks interpolated during the first pass would be regarded as Indicated Mineral Resources, containing a minimum of two drill holes within the range of the modelled variograms. On the second pass, one drill hole within four times the range were classified as Inferred Mineral Resources and on the third pass, any blocks remaining within the subzone block model would be classified as Exploration Potential. Only 115 tonnes were interpolated during the third pass and, as this was not regarded as significant, this tonnage has been included in the Inferred Mineral Resources.

16.3.5 Mineral Resource Tabulation

The Indicated Mineral Resources and Inferred Mineral Resources for the Horseshoe deposit capped model are summarized in Table 16.13. The kriged capped values have been used for reporting the mineral resource estimates. No factors have been applied to the U_3O_8, lbs and they represent an in situ value.

Table 16.13: Horseshoe Indicated and Inferred Mineral Resources (Capped) at Various % U_3O_8 Cut-offs (Ordinary Kriged Values)

Category	Cut-off Grade (%U_3O_8)	Tonnes	Grade (%U_3O_8)	Contained U_3O_8 (lb)
Indicated	0.02	7,042,400	0.157	24,427,000
	0.05	**5,119,700**	**0.203**	**22,895,000**
	0.10	3,464,800	0.266	20,302,000
	0.15	2,380,800	0.330	17,331,000
	0.20	1,567,000	0.412	14,219,000
	0.25	1,059,900	0.502	11,726,000
	0.30	722,600	0.609	9,696,000
	0.35	529,100	0.713	8,319,000
	0.40	414,600	0.807	7,377,000
Inferred	0.02	444,900	0.122	1,192,000
	0.05	**287,000**	**0.166**	**1,049,000**
	0.10	159,700	0.239	840,000
	0.15	106,800	0.298	702,000
	0.20	79,800	0.340	598,000
	0.25	53,500	0.398	469,000
	0.30	29,300	0.502	324,000
	0.35	15,500	0.655	227,000
	0.40	11,400	0.769	193,000

16.3.6 Block Model Validation

The Horseshoe grade interpolation plan and model was validated using four methods:

1. Comparison of block model volumes to volumes within solids
2. Visual comparison of colour-coded block model grades with drill hole grades on section and plan plots
3. Comparison of the global mean block grades for ordinary kriging, nearest neighbour and inverse distance squared methods
4. Comparison of block model grades and drill hole grades using swath plots

Block Volume/Solid Volume Comparison

The block model volumes were compared to the original volume within the interpreted mineralized envelopes or subzones provided by UEX. The results are shown by subzone in Table 16.14. Only minor differences were noted which indicates a good translation between the mineralized geometry and the resource block models for each subzone.

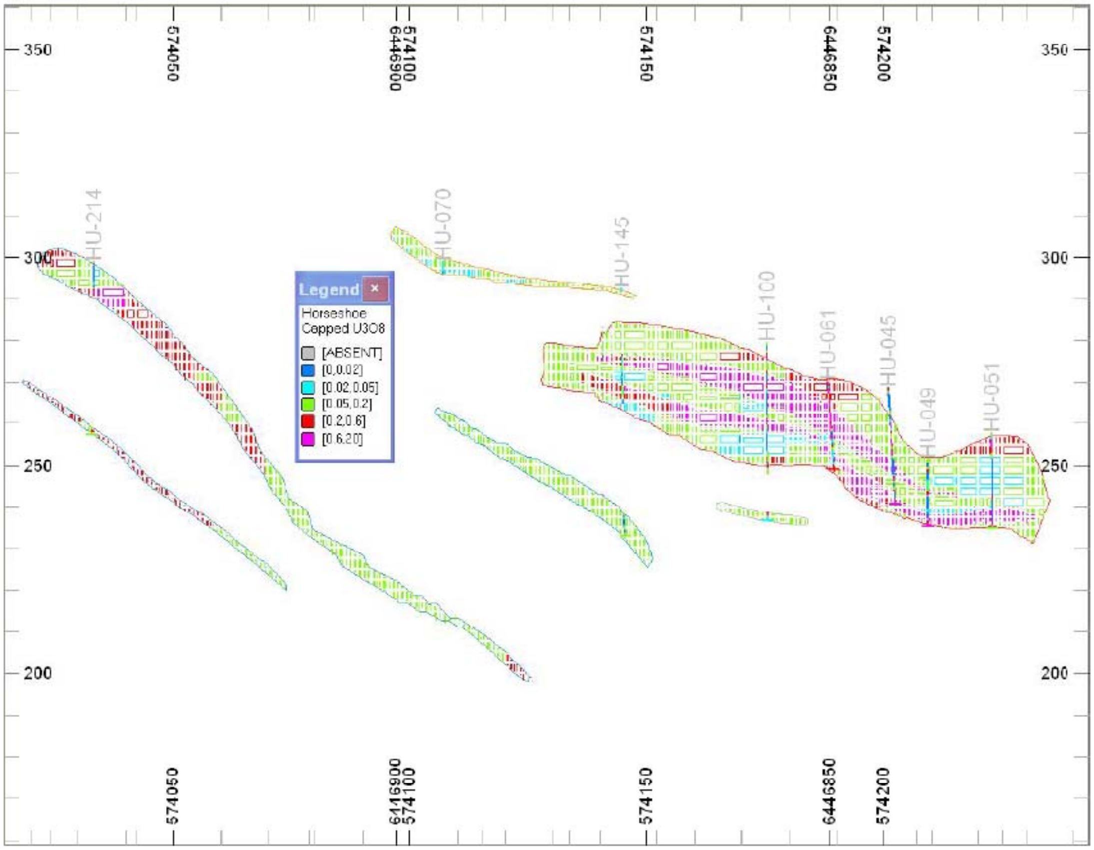
Table 16.14: Comparison of Block Model and Solid Volumes (m^3)

Sub-zone	Model Vol	Solid Vol	%Diff	Sub-zone	Model Vol	Solid Vol	%Diff	Sub-zone	Model Vol	Solid Vol	%Diff
A1H	39,581	39,619	0.1%	**M01**	75,633	75,639	0.0%	**G01**	449,141	449,240	0.0%
A1	155,588	155,579	0.0%	**M02**	9,244	9,245	0.0%	**G02**	66,317	66,307	0.0%
A2	122,682	122,697	0.0%	**M03**	21,483	21,502	0.1%	**Q01**	804,186	809,830	0.7%
A3	41,759	41,748	0.0%	**M04**	39,103	39,060	-0.1%	**Q02**	41,221	41,186	-0.1%
A4	23,368	23,356	0.0%	**M05**	10,168	10,158	-0.1%	**Q03**	37,604	37,573	-0.1%
A5	26,526	26,582	0.2%	**M06**	17,442	17,465	0.1%				
BW	535,762	535,852	0.0%	**M07**	20,627	20,682	0.3%				
BE	292,187	292,200	0.0%	**M08**	5,680	5,680	0.0%				
C	42,753	42,759	0.0%	**M09**	3,080	3,085	0.2%				
S1	50,622	50,634	0.0%	**M10**	6,205	6,227	0.4%				
S2	62,275	62,249	0.0%	**M11**	2,129	2,131	0.1%				
S3	79,872	79,924	0.1%								

Note: Subzone A1 includes A1H Volume

Visual Validation of Sections

The visual comparisons of block model grades with composite grades for the five zones show a reasonable correlation between the values. A review of plans and sections showed no significant discrepancies. Figure 16.2 shows a typical section.



Global Comparisons

Figure 16.2: Horseshoe Dip Section looking East, showing block Model and Drill Holes

Global Comparisons

The global block grade statistics for the ordinary kriging model are compared to the declustered means for each subzone (Table 16.15). Subzones A2, A3, A5, C, S3, M05, M06 and M11 have differences above 10%. Subzone C shows the highest difference with a difference of 34%.

Table 16.15: Comparison of Top Cut Declustered Drill Holes with OK Grades

Subzone	A1	A1H	A2	A3	A4	A5	BE	BW	C	S1	S2	S3
Model Mean	0.131	1.429	0.259	0.286	0.253	0.138	0.172	0.228	0.135	0.153	0.314	0.327
Declust. DH Mean	0.134	1.482	0.295	0.248	0.230	0.121	0.186	0.247	0.205	0.149	0.290	0.369
% Difference	-2	-4	-12	15	10	14	-7	-8	-34	3	8	-11
Subzone	**M01**	**M02**	**M03**	**M04**	**M05**	**M06**	**M07**	**M08**	**M09**	**M10**	**M11**	
Model Mean	0.100	0.072	0.078	0.069	0.076	0.106	0.085	0.114	0.066	0.211	0.059	
Declust. DH Mean	0.104	0.076	0.078	0.063	0.063	0.122	0.078	0.122	0.070	0.204	0.070	
% Difference	-4	-6	-1	10	20	-13	10	-7	-6	4	-16	
Subzone	**G01**	**G02**	**Q01**	**Q02**	**Q03**							
Model Mean	0.106	0.046	0.046	0.049	0.040							
Declust. DH Mean	0.112	0.043	0.045	0.050	0.038							
% Difference	-5	8	4	-3	5							

A further check was carried out on the interpolation where the global ordinary kriged (“OK”) grades were compared to the nearest neighbour (“NN”) and inverse distance squared (“ID^2”) interpolation (Table 16.16). Subzone C shows a greater than 10% difference in both comparisons. The A5 and M08 subzones show a good comparison with the ID2 method, but show a poor (31% and 19%) difference when compared to NN.

Table 16.16:Comparison of Interpolation for Ordinary Kriging

Subzone	**A1**	**A1H**	**A2**	**A3**	**A4**	**A5**	**BE**	**BW**	**C**	**S1**	**S2**	**S3**
OK Model Mean	0.131	1.429	0.259	0.286	0.253	0.138	0.172	0.228	0.135	0.153	0.314	0.327
ID2 Model Mean	0.128	1.331	0.27	0.283	0.253	0.151	0.188	0.229	0.178	0.160	0.314	0.304
%Difference	3	7	-4	1	0	-9	-8	-1	-24	-4	0	7
Subzone	**M01**	**M02**	**M03**	**M04**	**M05**	**M06**	**M07**	**M08**	**M09**	**M10**	**M11**	
OK Model Mean	0.100	0.072	0.078	0.069	0.076	0.106	0.085	0.114	0.066	0.211	0.059	
ID2 Model Mean	0.100	0.076	0.076	0.059	0.078	0.102	0.079	0.119	0.067	0.224	0.058	
%Difference	0	-6	3	17	-3	4	8	-4	-2	-6	2	
Subzone	**G01**	**G02**	**Q01**	**Q02**	**Q03**							
OK Model Mean	0.106	0.046	0.046	0.049	0.040							
ID2 Model Mean	0.100	0.048	0.046	0.051	0.041							
%Difference	6	-3	2	-4	-3							
Subzone	**A1**	**A1H**	**A2**	**A3**	**A4**	**A5**	**BE**	**BW**	**C**	**S1**	**S2**	**S3**
OK Model Mean	0.131	1.429	0.259	0.286	0.253	0.138	0.172	0.228	0.135	0.153	0.314	0.327
NN Model Mean	0.128	1.452	0.265	0.25	0.237	0.111	0.192	0.252	0.185	0.153	0.308	0.317
%Difference	2	-2	-2	14	7	25	-10	-10	-27	1	2	3
Subzone	**M01**	**M02**	**M03**	**M04**	**M05**	**M06**	**M07**	**M08**	**M09**	**M10**	**M11**	
OK Model Mean	0.100	0.072	0.078	0.069	0.076	0.106	0.085	0.114	0.066	0.211	0.059	
NN Model Mean	0.105	0.090	0.083	0.068	0.058	0.120	0.080	0.141	0.066	0.207	0.063	
%Difference	-5	-20	-6	2	31	-12	7	-19	-1	2	-6	
Subzone	**G01**	**G02**	**Q01**	**Q02**	**Q03**							
OK Model Mean	0.106	0.046	0.046	0.049	0.040							
NN Model Mean	0.112	0.049	0.045	0.053	0.039							
%Difference	-5	-6	3	-7	3							

Subzone C is mainly classified as an Inferred Mineral Resource so the difference is within an acceptable range for the classification.

Swath Plots

Swath plots have been generated for OK, ID^2 and NN for the total subzone models. An example of a swath plot is present below (Figure 16.3). This is from one of the lower grade subzones on the northeast.

In general, the swath plots show a good correlation between drill holes, NN, ID^2 and OK values.

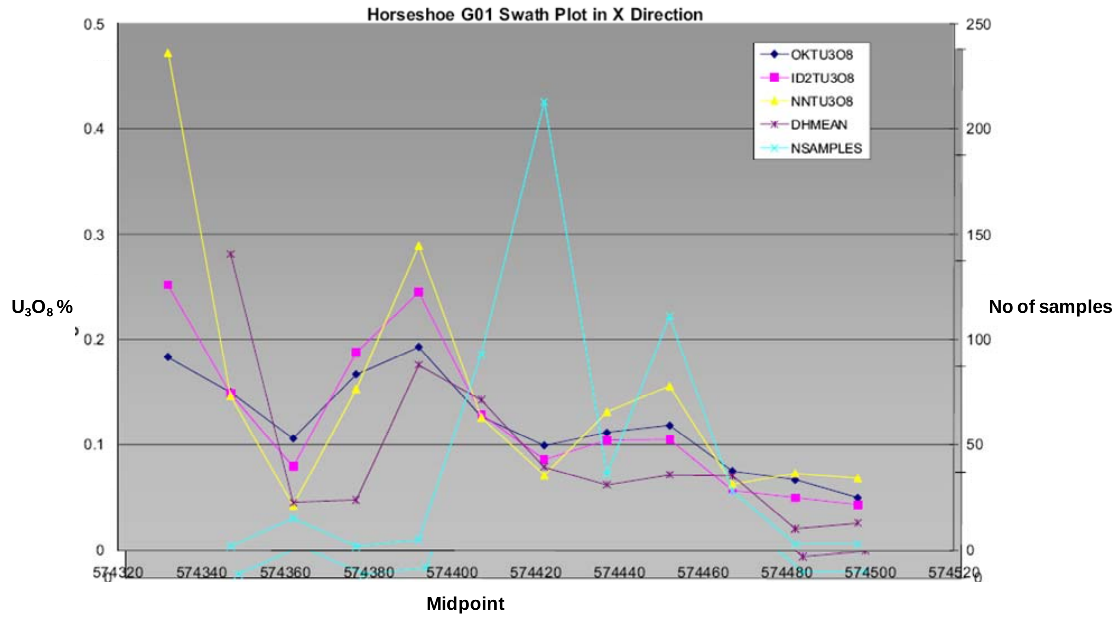


Figure 16.3: % U_3O_8 Swath Plots for G01 Subzones in X Direction

16.4 Mineral Resource Estimate for the Raven Deposit

16.4.1 Exploratory Data Analysis

In order to carry out the evaluation of the Raven deposit, a digital database for collars, surveys, lithology, density, recoveries and assays, suitable for importing into Datamine was provided in an Excel format by UEX. UEX also provided 16 separate 3D mineralized envelopes which were interpreted to include most of the mineralization above a 0.02% U_3O_8, cut-off on the Raven deposit. Each envelope has been given a numeric and an alphanumeric code (Table 16.17).

Table 16.17: Numeric and Alphanumeric Codes for Raven Mineralized Envelopes

Alphanumeric	L01	L02	L03	L04	L05	L06	U01	U02
Numeric	101	102	103	104	105	106	201	202
Alphanumeric	**U03**	**U04**	**U05**	**U06**	**U07**	**U08**	**U09**	**U10**
Numeric	203	204	205	206	207	208	209	210

Exploratory Data Analysis and Variography were carried out using Supervisor software.

Data

The database is comprised of a total of 243 drill holes and includes Gulf drill holes RV-001 to RV-028 and UEX drill holes RU-001 to RU-88 and RU-90 to RU-216.

The Raven database contains 18,100 data entries of % U_3O_8. There are also 1,524 dry bulk density measurements. The mineralized envelopes (all 16 subzones with cut-off grades at or above 0.02% U_3O_8) contain 8,378 data entries of % U_3O_8 and 959 bulk density measurements.

Bulk Density

Dry bulk densities were assigned to the individual subzones based on the mean value for that subzone. Subzones that had no values were assigned the mean value of all the mineralized envelopes (2.448 g/cm3). Table 16.18 lists the dry bulk densities for the different units.

Table 16.18: Dry Bulk Densities for Raven Deposit by Subzone

Subzone	L01	L02	L03	L04	L05	L06	U01	U02
Bulk Density (g/cm^3)	2.420	2.448	2.448	2.523	2.448	2.448	2.509	2.295
Subzone	**U03**	**U04**	**U05**	**U06**	**U07**	**U08**	**U09**	**U10**
Bulk Density (g/cm^3)	2.363	2.448	2.569	2.448	2.448	2.273	2.448	2.524

The bulk density for Subzone U02 and U03 is lower than the other subzones. Some of the samples for this subzone came from intense clay alteration zones. These narrow intensely altered zones are found throughout the deposit.

Geological Interpretation

Datamine string files were interpreted around a cut-off of 0.02% U_3O_8, taking into consideration UEX's knowledge of the geology of the deposit, in order to provide an assessment of the mineralization. These strings were used to create 3D wireframes around the mineralized envelopes. Mineralization is localized along the trace of the Raven syncline, particularly along the southeastern limb of the fold and developed extending downward from the base of the folded calc-arkose unit into the underlying quartzite and arkosic quartzite. The mineralized envelopes are strongly associated with the hematitic alteration halo.

3D wireframes were generated from the string files by UEX. These wireframes were subsequently verified for duplicate vertices, duplicate faces and empty faces in Datamine and are illustrated in Figure 16.4 including the drill hole traces. The red wireframe represents subzone L01, the dark blue U01 and the light blue on the left U10.

Golder reviewed the interpretation and verified that they were consistent with UEX's planned geological and mineral interpretation as described above.

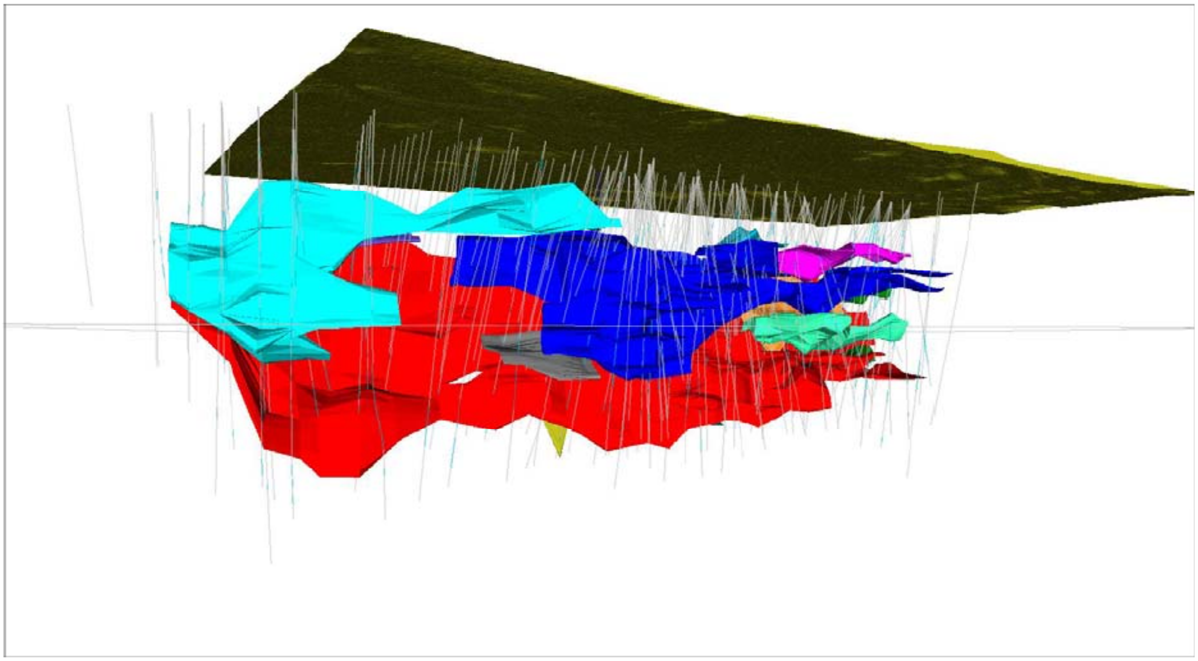


Figure 16.4: **Raven Subzones with Drill Holes, Oblique Section looking North**

Assays

A statistical review of the assay files from the 243 drill holes for the Raven deposit was completed by Golder. Samples have been taken predominantly from three rock types, namely arkosic-quartzite gneiss ("ARKQ"), quartzite ("QZIT") and calc-arkosic gneiss ("CARK"). The statistics for the rock type indicate that the lithology coded CARK contains the highest mean grade (0.054% U_3O_8) and QZIT has the highest median grade (0.009% U_3O_8) (Table 16.19).

Lithologies with less than ten samples have been removed from the table.

Table 16.19: Raven Statistics for % U_3O_8 by Lithology for Raw Data

Statistics		U_30_8_PCT	ARKQ	ARKS	BX	CALC	CARK	CLAY	GRAN	GRGN	PEGM	PEL0	QV	QZIT	SPL0
Samples		19.283	6,861	12	21	18	2,838	14	172	76	1,247	201	12	6.478	1,315
Minimum		0	0	0	0.001	0	0	0.003	0	0	0	0	0	0	0
Maximum		18	2,490	0.039	0.128	0.009	18.8	0.034	0.283	0.897	4.04	0.521	0.002	2.99	0.893
Mean		**0.033**	**0.031**	**0.005**	**0.024**	**0.002**	**0.054**	**0.012**	**0.014**	**0.028**	**0.023**	**0.015**	**0.001**	**0.031**	**0.019**
Std. Deviation		0.163	0.101	0.012	0.033	0.002	0.357	0.01	0.033	0.105	0.117	0.049	0.001	0.095	0.052
Coef. Of Var		**4.952**	**3.206**	**2.424**	**1.414**	**1.039**	**6.634**	**0.826**	**2.291**	**3.761**	**5.047**	**3.228**	**0.886**	**3.009**	**2.786**
Variance		0.027	0.01	0	0.001	0	0.128	0	0.001	0.011	0.014	0.002	0	0.009	0.003
Skewness		58.092	10.211	5.164	2.01	2.108	36.2	1.058	5.723	6.702	18.168	7.894	0.933	12.106	7.316
Grade at Percentile	10th	0.001	0.001	0	0.002	0	0.001	0.003	0.001	0.001	0.001	0	0	0.002	0.001
	20th	0.002	0.002	0.001	0.002	0.001	0.002	0.004	0.001	0.001	0.002	0.001	0	0.003	0.001
	30th	0.003	0.002	0.001	0.002	0.001	0.002	0.006	0.002	0.002	0.002	0.001	0	0.004	0.002
	40th	0.004	0.004	0.001	0.003	0.001	0.004	0.006	0.003	0.003	0.003	0.002	0	0.006	0.003
	Median	**0.006**	**0.006**	**0.002**	**0.006**	**0.001**	**0.005**	**0.007**	**0.004**	**0.003**	**0.004**	**0.003**	**0.001**	**0.009**	**0.004**
	60th	0.01	0.01	0.002	0.008	0.002	0.007	0.007	0.006	0.005	0.006	0.005	0.001	0.014	0.006
	70th	0.016	0.017	0.002	0.035	0.003	0.012	0.016	0.009	0.008	0.008	0.012	0.001	0.021	0.01
	80th	0.028	0.029	0.002	0.041	0.004	0.028	0.018	0.019	0.015	0.014	0.015	0.001	0.033	0.017
	90th	0.062	0.062	0.004	0.067	0.004	0.079	0.025	0.033	0.039	0.03	0.022	0.002	0.066	0.042
	95th	0.122	0.122	0.004	0.089	0.006	0.201	0.028	0.046	0.095	0.065	0.046	0.002	0.117	0.082
	97.5	0.228	0.228	0.039	0.089	0.009	0.425	0.029	0.08	0.215	0.167	0.12	0.002	0.21	0.148
	99th	0.452	0.452	0.039	0.128	0.009	0.923	0.029	0.208	0.477	0.396	0.236	0.002	0.37	0.252

The basic statistics for the samples for each subzone are listed in Table 16.20 and Table 16.21.

Table 16.20: Raven Statistics for %U_3O_8 by Lower Subzones

Statistic		U_3O_8_PCT	L01	L02	L03	L04	L05	L06
Samples		8,378	2,734	101	12	70	4	46
Minimum		0.000	0.000	0.001	0.014	0	0.013	0.002
Maximum		18.800	2.490	0.503	0.092	1.020	1.270	0.323
Mean		**0.072**	**0.065**	**0.035**	**0.035**	**0.057**	**0.228**	**0.039**
Std. Deviation		0.251	0.145	0.062	0.022	0.149	0.532	0.053
Coef. Of Var.		**3.468**	**2.245**	**1.761**	**0.645**	**2.631**	**2.337**	**1.347**
Variance		0.063	0.021	0.004	0.000	0.022	0.283	0.003
Skewness		39.086	7.581	4.729	1.899	4.763	2.940	2.999
Grade at Percentile	10th	0.003	0.002	0.003	0.014	0.001	0.013	0.002
	20th	0.007	0.006	0.005	0.017	0.002	0.013	0.012
	30th	0.012	0.011	0.008	0.018	0.003	0.013	0.017
	40th	0.019	0.018	0.010	0.027	0.005	0.020	0.019
	Median	**0.025**	**0.024**	**0.016**	**0.027**	**0.014**	**0.020**	**0.021**
	60th	0.033	0.034	0.024	0.032	0.022	0.020	0.025
	70th	0.047	0.048	0.028	0.034	0.029	0.022	0.027
	80th	0.077	0.077	0.039	0.045	0.039	0.022	0.046
	90th	0.149	0.151	0.081	0.046	0.095	0.022	0.090
	95th	0.276	0.252	0.121	0.046	0.291	1.270	0.140
	97.5	0.448	0.386	0.156	0.092	0.549	1.270	0.218
	99th	0.849	0.697	0.306	0.092	0.570	1.270	0.252

Table 16.21: Raven Statistics for %U_3O_8 by Upper Subzones

Statistic		U01	U02	U03	U04	U05	U06	U07	U08	U09	U10
Samples		3,647	120	271	29	167	50	46	84	69	928
Minimum		0.000	0.002	0.002	0.005	0.001	0.002	0.003	0.00	0.002	0
Maximum		18.800	4.920	1.320	0.189	0.898	1.120	3.220	0.946	0.452	1.880
Mean		**0.081**	**0.217**	**0.076**	**0.048**	**0.055**	**0.098**	**0.180**	**0.035**	**0.079**	**0.056**
Std. Deviation		0.340	0.487	0.138	0.041	0.098	0.239	0.464	0.069	0.113	0.138
Coef. Of Var.		**4.214**	**2.244**	**1.821**	**0.844**	**1.775**	**2.438**	**2.582**	**1.938**	**1.427**	**2.467**
Variance		0.116	0.237	0.019	0.002	0.010	0.057	0.215	0.005	0.013	0.019
Skewness		36.996	5.599	4.837	1.731	4.339	3.530	5.506	8.470	2.079	6.993
Grade at Percentile	10th	0.005	0.005	0.008	0.015	0.002	0.004	0.003	0.003	0.004	0.003
	20th	0.008	0.010	0.015	0.020	0.004	0.005	0.008	0.004	0.007	0.006
	30th	0.014	0.016	0.021	0.023	0.009	0.009	0.010	0.006	0.017	0.010
	40th	0.020	0.023	0.025	0.027	0.019	0.017	0.013	0.010	0.021	0.016
	Median	**0.026**	**0.035**	**0.030**	**0.035**	**0.024**	**0.021**	**0.029**	**0.015**	**0.031**	**0.022**
	60th	0.035	0.066	0.040	0.036	0.032	0.025	0.038	0.026	0.046	0.027
	70th	0.050	0.149	0.059	0.042	0.044	0.033	0.113	0.032	0.066	0.039
	80th	0.082	0.335	0.095	0.068	0.068	0.038	0.202	0.047	0.110	0.057
	90th	0.156	0.604	0.165	0.103	0.136	0.221	0.470	0.080	0.229	0.113
	95th	0.292	0.923	0.333	0.124	0.224	0.288	0.638	0.115	0.377	0.191
	97.5	0.469	1.120	0.398	0.153	0.307	0.910	0.858	0.121	0.397	0.350
	99th	0.935	2.390	0.764	0.153	0.493	1.120	3.020	0.220	0.452	0.773

Out of the 8,378 samples, 6,381 have been taken from L01 and U01 which represent about 73% of the volume of the deposit. Subzone L05 has the highest grade with a mean of 0.228 % U_3O_8, but this subzone has only been intersected by four samples. The median grades vary from 0.014% U_3O_8 (L04) and 0.035% U_3O_8 (U02 and U04). Subzones L01 has a median grade of 0.024% U_3O_8 and U01 have the same median grade of 0.026% U_3O_8. The histograms of the subzones with well defined histograms indicate that the % U_3O_8 population has a lognormal distribution. There is also the suggestion of more than one population within some of the subzones, but they appear to have a significant overlap.

Capping

Capping of sample assays is applied to reduce the impact on the mineral resource estimate of high grade samples that are interpreted as not being part of the lognormal population outliers. Anomalous high grades are cut to the highest grade that would be regarded as being part of that population. Lognormal histograms and log probability plots were reviewed to establish the capping level for each subzone.

A total of 44 samples were cut from all of the subzones, with the most, nine, being cut from L05. The effect of the cutting and the subsequent compositing had the effect of reducing the CV to less than 1.50 for 8 out of the 16 subzones.

Although the capped CV for U01 is greater than 1.5 (2.28), the log histogram suggests a reasonable log normal distribution for the U_3O_8 assay data. The effects of the capping and subsequent compositing are shown in Table 16.22.

Table 16.22: Raven Effect of Capping and Compositing on Coefficient of Variation

Statistic	L01	L02	L03	L04	L05	L06	U01	U02
Uncut Mean	0.065	0.035	0.035	0.057	0.228	0.039	0.081	0.217
Uncut CV	**2.24**	**1.76**	0.64	**2.63**	**2.34**	**1.35**	**4.21**	**2.24**
Cut Mean	0.064	0.032	0.035	0.050	0.228	0.038	0.078	0.210
Cut CV	2.14	1.38	0.64	2.25	2.34	1.23	2.72	2.05
No.Cut	9	2	0	2	0	3	3	2
Capping Level	1.800	0.220	0.090	0.550	1.270	0.210	4.000	3.200
Composite Cut Mean	0.064	0.032	0.035	0.050	0.228	0.038	0.078	0.210
Composite Cut CV	**1.74**	1.13	0.64	**1.97**	**1.57**	0.74	**2.28**	**1.63**
Statistic	**U03**	**U04**	**U05**	**U06**	**U07**	**U08**	**U09**	**U10**
Uncut Mean	0.076	0.048	0.055	0.098	0.18	0.035	0.079	0.056
Uncut CV	**1.82**	0.84	**1.77**	**2.44**	**2.58**	**1.94**	**1.43**	**2.47**
Cut Mean	0.075	0.048	0.052	0.091	0.14	0.033	0.077	0.055
Cut CV	**1.74**	0.84	**1.51**	**2.32**	**1.69**	1.39	1.38	**2.35**
No. Cut	3	0	5	2	2	2	6	3
Capping Level	1.00	0.19	0.40	0.90	1.10	0.30	0.37	1.30
Composite Cut Mean	0.075	0.048	0.052	0.091	0.14	0.033	0.077	0.055
Composite Cut CV	**1.56**	0.64	1.29	**2.33**	1.38	1.22	1.01	**2.01**

Composites

Assays were composited to 1.0 m lengths, which is the 70th percentile of the lengths contained within the mineralized envelopes. The minimum composite length allowed is 0.15 metre. The compositing method chosen in Datamine is the one whereby all samples are included in one of the composites. This is achieved by adjusting the composite length, while keeping the length as close as possible to the 1.0 metre.

Compositing was restricted to within individual subzones, based on codes assigned to the drill hole file. Compositing the drill holes has reduced the number of samples all of the subzones. Compositing had the effect of reducing the CV in 13 out of the 16 subzones.

Spatial Analysis

Variography, using Supervisor software, was completed for % U_3O_8 assay samples for each individual subzone and for the top cut U_3O_8 assay samples in Subzones L01 and U01. No differences were noted in the variograms of the uncut and cut data.

Downhole variograms were used to determine nugget effect subsequently lognormal variograms were modelled to determine spatial continuity of % U_3O_8. In some of the subzones, it was not possible to develop anisotropic models and, where this was the case, isotropic models were developed. Subzones L02 to L06, U04 and U06 to U09 had insufficient data to establish variograms. In these cases, the modelled variograms obtained from subzone U03 were used.

A two-structure spherical model was used to model most of the lognormal variograms. Table 16.23 summarizes the results of the variography.

Table 16.23: Variogram Parameters for Lower and Upper Subzones

Subzone	Variable	Direction	Azimuth	Dip	Nugget	Sill C_1	Range A_1 (m)	Sill C_2	Range A_2 (m)
L01	U_3O_8	1	165	-65	0.24	0.54	7.5	0.22	20.0
	U_3O_8	2	075	00	0.24	0.54	45.0	0.22	65.5
	U_3O_8	3	345	-25	0.24	0.54	8.5	0.22	23.0
U01	U_3O_8	1	136	-72	0.19	0.40	11.5	0.41	63.0
	U_3O_8	2	077	10	0.19	0.40	21.5	0.41	31.5
	U_3O_8	3	350	-15	0.19	0.40	9.5	0.41	18.0
U02	U_3O_8	1	000	00	0.00	0.84	1.5	0.16	5.5
	U_3O_8	2	090	00	0.00	0.84	1.5	0.16	5.5
	U_3O_8	3	000	90	0.00	0.84	1.5	0.16	5.5
U03	U_3O_8	1	340	-55	0.35	0.32	20.5	0.33	30.0
	U_3O_8	2	070	00	0.35	0.32	8.0	0.33	19.5
	U_3O_8	3	340	35	0.35	0.32	12.0	0.33	28.0
U05	U_3O_8	1	085	00	0.00	1.00	33.0		
	U_3O_8	2	175	00	0.00	1.00	33.0		
	U_3O_8	3	000	90	0.00	1.00	33.0		
U10	U_3O_8	1	090	-55	0.21	0.42	26.5	0.37	45.0
	U_3O_8	2	090	35	0.21	0.42	16.5	0.37	84.0
	U_3O_8	3	000	00	0.21	0.42	16.5	0.37	26.0

Subzone L01 has the largest range (A2, second structure) range of 65.5 m on an azimuth of 075° dipping 0°. This is the approximate strike of the subzone. The largest range for U01 is similar but in the dip direction. The modelled variograms were reviewed by UEX and the directions and ranges agree with their geological understanding of the two major subzones: L01 and U01.

16.4.2 Resource Block Model

Block models were established in Datamine for all subzones. All of the modelled wireframes are below the overburden and there was no need to cut block model below the topography.

A standard block size of 5.0 m x 5.0 m x 2.5 m (Easting x Northing x Elevation) was used for the interpolation. This was based on the average sample spacing on the property. Sub-celling was allowed in order to improve the fill of the interpreted solids. The minimum cell sizes allowed were 1.0 m for Northing, 1.0 m for Easting and 0.5 m for the Elevation.

16.4.3 Interpolation Plan

The Raven deposit model used the variable anisotropy search model available in Datamine. The dip and dip direction is calculated for each triangle used to make up the wireframe which contains the mineralized drill hole intersections. These two parameters are then interpolated into each block. During the grade interpolation process, the search ranges established during the variography process for each subzone is rotated for each block to match the interpolated dip and dip direction.

Most of the blocks for all of the capped and uncapped U_3O_8 were interpolated during the first pass, which was at the range of continuity of the variograms for all subzones except U02, where an isotropic search range of 15 m was used. A second pass at four times and a third at six or ten times the sill range was required to interpolate % U_3O_8 in all of the subzones. The grade interpolation plan is summarized in Table 16.24. A minimum of four samples and a maximum of 24 samples were used in the first and a minimum of 3 samples and a maximum of 24 samples in the second and third pass. A minimum of two drill holes were used in the first pass and one in the second and third.

Table 16.24: Summary of Grade Interpolation Plan

Model Name	**Minmod**		
Dimensions	x	y	z
Parent Cell	5	5	2.5
Minimum sub cell	1	1	0.5
Model origin	572,410	6,446,200	95
Total parent cells	220	150	200
Parent discretisation	2	2	1
Estimated attributes	**Attribute**	**Unit**	**Comment**
	OKTU308	%	Capped U_30_8 ordinary kriging
	ID2TU308	%	Capped U_3O_8 inverse distance squared
	NNTU308	%	Capped U_3O_8 nearest neighbour
	OKU308	%	U_30_8 ordinary kriging
	ID2U308	%	U_3O_8 inverse distance squared
	TRDIP	degrees	True dip
	TRDIPDIR	degrees	True dip direction
Assigned attributes	ZONA	Alphanumeric Subzone Code L01 to L06, U01 to U10	
	ZONN	Numeric Subzone Code 106, 201 to 210	
	NSAMU	Number of samples used in interpolation	
	SVOLU	Search neighbourhood volume for U_3O_8	
	VARKU	Kriging Variance for U_3O_8	
	DENSITY	Density was assigned based on mean of samples of samples within subzone. Default of 2.448 g/cm^3 used for subzones with no samples	
	CATEGORY	Numeric Value for Mineral Resource Category 1=Measured, 2=Indicated, 3=Inferred and 4=Exploration potential	
	CATA	Alpha numeric for Resource Categories	
	NSAMPANI	Number of samples used in interpolation TRDIP and TRDIPDIR	
	SVOLANI	Search neighbourhood volume for TRDIP and TRDIPDIR	

16.4.4 Mineral Resource Classification

Several factors are considered in the definition of a resource classification:

- CIM requirements and guidelines;
- Experience with similar deposits;
- Spatial continuity; and
- Confidence limit analysis.

The search volume was used as a guide to classify the Raven deposit. Blocks interpolated during the first pass would be regarded as Indicated Mineral Resources, containing a minimum of two drill holes within the range of the modelled variograms. U02 used an isotropic range of 15 m.

On the second pass, one drill hole within four times the range were classified as Inferred Mineral Resources and on the third pass, any blocks remaining within the subzone block model would be classified as Exploration Potential. Only 550 t were interpolated during the third pass and, as this was not regarded as significant, this tonnage has been included in the Inferred Mineral Resources.

16.4.5 Mineral Resource Tabulation

The Indicated and Inferred Mineral Resources for the capped model are summarized Table 16.25. The capped ordinary kriged values have been used for reporting the mineral resource estimates. No factors have been applied to the U_3O_8 lbs and they represent an in situ value.

Table 16.25: Raven indicated and Inferred Mineral Resources (Capped) at Various % U_3O_8 Cut-offs (Ordinary Kriged Values)

Category	Cut-off Grade (% U_3O_8)	Tonnes	Grade (% U_3O_8)	Contained U_3O_8 (lb)
Indicated	0.02	9,646,100	0.073	15,544,000
	0.05	**5,173,900**	**0.107**	**12,149,000**
	0.10	1,893,400	0.17	7,113,000
	0.15	827,700	0.234	4,274,000
	0.20	424,000	0.294	2,752,000
	0.25	241,500	0.349	1,859,000
	0.30	139,100	0.406	1,244,000
	0.35	80,300	0.467	827,000
	0.40	48,400	0.529	565,000
Inferred	0.02	1,537,600	0.067	2,278,000
	0.05	**822,200**	**0.092**	**1,666,000**
	0.10	176,000	0.186	723,000
	0.15	96,000	0.239	506,000
	0.20	48,500	0.302	323,000
	0.25	25,700	0.37	209,000
	0.30	15,800	0.431	150,000
	0.35	11,700	0.468	121,000
	0.40	8,200	0.509	92,000

A cut-off grade of 0.05% U_3O_8 results in 5,173,900 tonnes at an average grade of 0.107% U_3O_8, giving 12,149,000 lbs U_3O_8 in the Indicated Mineral Resource category and 822,200 tonnes at an average grade of 0.092% U_3O_8, giving 1,666,000 lbs U_3O_8 in the Inferred Mineral Resource category.

16.4.6 Block Model Validation

The Raven deposit grade interpolation plan and model was validated using four methods:

- Comparison of block model volumes to volumes within solids
- Visual comparison of colour-coded block model grades with drill hole grades on section and plan plots
- Comparison of the declustered drill hole grades to ordinary kriged block grades as well as global mean block grades for ordinary kriging, nearest neighbour and inverse distance squared methods
- Comparison of block model grades and drill hole grades using swath plots

Block Volume/Solid Volume Comparison

The block model volumes were compared to the original volume within the interpreted mineralized envelopes or subzone provided by UEX. The results are shown by subzone in Table 16.26. Only minor differences were noted, which indicates a good translation between the mineralized geometry and the resource block models for each subzone.

Table 16.26: **Comparison of Block Model and Solid Volumes (m^3)**

Subzone	Model Volume	Solid Volume	%Diff	Subzone	Model Volume	Solid Volume	%Diff
L01	2,074,611	2,074,548	0.0%	**U03**	153,602	153,642	0.0%
L02	61,920	61,905	0.0%	**U04**	27,855	27,838	-0.1%
L03	7,701	7,727	0.3%	**U05**	55,441	55,468	0.0%
L04	77,743	77,755	0.0%	**U06**	11,230	11,258	0.2%
L05	2,294	2,294	0.0%	**U07**	18,347	18,399	0.3%
L06	32,335	32,263	-0.2%	**U08**	31,168	31,161	0.0%
U01	1,449,027	1,448,800	0.0%	**U09**	33,511	33,483	-0.1%
U02	44,303	44,269	-0.1%	**U10**	755,224	755,247	0.0%

Visual Validation of Sections

The visual comparisons of block model grades with composite grades for the subzones show a reasonable correlation between the values (Figure 16.5). No significant discrepancies were apparent from the sections and plans reviewed.

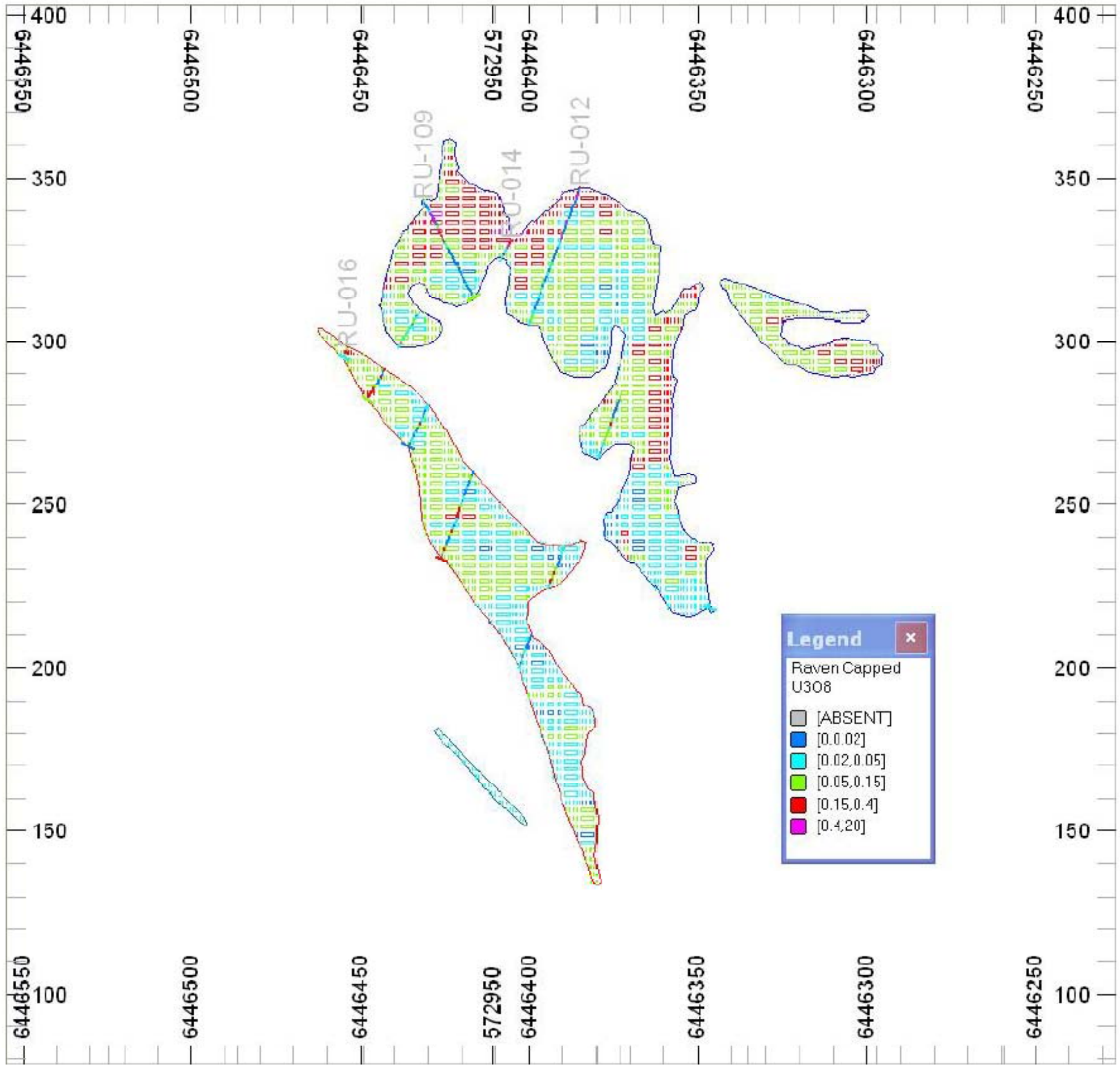


Figure 16.5: **Dip Section looking East, showing Block Model and Drill Holes**

Global Comparisons

The global block grade statistics for the ordinary kriging model are compared to the declustered means for each subzone (Table 16.27). Subzones L03, L05 U01, U02, U01, U04, U06 and U07 have differences above 10%. Subzone U07 shows the highest difference with a difference of 29%.

Table 16.27: Comparison of top Cut Declustered Drill Holes with Ordinary Kriged Grades (%U_3O_8)

Subzone	L01	L02	L03	L04	L05	L06	U01	U02
Model Mean	0.066	0.037	0.045	0.075	0.247	0.040	0.078	0.152
Declust. DH Mean	0.063	0.038	0.039	0.070	0.317	0.039	0.067	0.166
% Difference	5	-2	**15**	7	**-22**	2	**17**	-9
Subzone	**U03**	**U04**	**U05**	**U06**	**U07**	**U08**	**U09**	**U10**
Model Mean	0.064	0.054	0.058	0.06	0.158	0.043	0.076	0.056
Declust. DH Mean	0.067	0.048	0.061	0.078	0.224	0.040	0.078	0.055
% Difference	-4	**12**	-6	**-24**	**-29**	8	-2	2

A further check was carried out on the interpolation where the global OK grades were compared to the NN and ID^2 interpolation (Table 16.28). Subzones L03, U04, U06 and U07 show a greater than 10% difference with the NN and ID^2 method.

Table 16.28: Comparison of Interpolation for Top Cut Ordinary Kriging (%U_3O_8)

Subzone	L01	L02	L03	L04	L05	L06	U01	U02
OK Model Mean	0.066	0.037	0.045	0.075	0.247	0.040	0.078	0.152
ID^2 Model Mean	0.065	0.036	0.039	0.07	0.245	0.042	0.080	0.171
% Difference	2	3	**16**	7	1	-4	-1	**-11**
Subzone	**U03**	**U04**	**U05**	**U06**	**U07**	**U08**	**U09**	**U10**
OK Model Mean	0.064	0.054	0.058	0.060	0.158	0.043	0.076	0.056
ID^2 Model Mean	0.066	0.047	0.051	0.072	0.203	0.043	0.074	0.057
% Difference	-3	**14**	**12**	**-17**	**-22**	1	3	-1
Subzone	**L01**	**L02**	**L03**	**L04**	**L05**	**L06**	**U01**	**U02**
OK Model Mean	0.066	0.037	0.045	0.075	0.247	0.040	0.078	0.152
NN Model Mean	0.066	0.038	0.040	0.086	0.263	0.042	0.071	0.165
% Difference	1	-2	**12**	**-13**	-6	-4	11	-8
Subzone	**U03**	**U04**	**U05**	**U06**	**U07**	**U08**	**U09**	**U10**
OK Model Mean	0.064	0.054	0.058	0.06	0.158	0.043	0.076	0.056
NN Model Mean	0.070	0.047	0.061	0.088	0.178	0.039	0.077	0.056
% Difference	-9	**15**	-5	**-32**	**-11**	**11**	-1	0

At the 0.05% U_3O_8 cut-off, 100% of L03, 85% of U04, 20% of U06 and 65% of U07 subzone tonnes are in the Inferred Mineral Resource category. These differences are regarded as being within an acceptable range for the classification most of the subzones. U06 contains only 5,500 tonnes at this cut-off and is therefore not regarded as a significant risk.

Swath Plots

Swath plots have been generated for OK, ID^2 and NN for the total subzone models for Subzone U10. An example of a swath plot is present below (Figure 16.6).

This swath plots show a reasonable correlation between the drill hole, NN, ID^2 and OK grades. The plot below indicates a good correlation between the drill hole grades and the various interpolation methods.

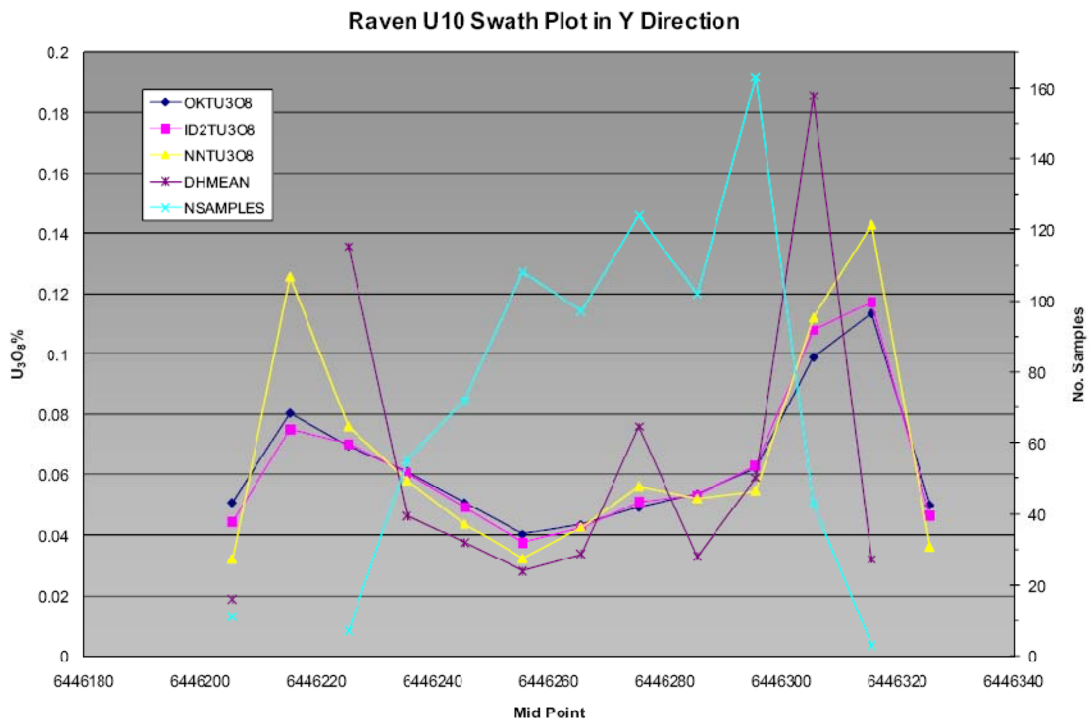


Figure 16.6: %U_3O_8 Swath Plots for U10 Subzone in Y Direction

16.5 Hidden Bay Mineral Resources

The total Indicated and Inferred Mineral Resources for the Hidden Bay Property are summarized in Table 16.29.

Although a lower cut-off grade (0.04% U_3O_8) has been recommended for the West Bear Property, a cut-off of 0.05% is recommended for the entire Hidden Bay Property as the majority of the tonnes are defined within Horseshoe and Raven.

The combined January 2009 N.I. 43-101 compliant resource for the West Bear deposit, and the July 2009 N.I. 43-101 compliant resource at the Horseshoe and Raven deposits on the Hidden Bay Project at a cut-off of 0.05% U_3O_8 total 10.373 million tonnes which contain 36.623 million pounds U_3O_8 in the Indicated Mineral Resource category and 1.109 million tonnes containing 2.715 million pounds U_3O_8 in the Inferred Mineral Resource category.

The pounds of U_3O_8 are raw pounds and have had no mining or milling factors applied to them.

Table 16.29: Total NI 43-101 Compliant Indicated and Inferred Mineral Resources (Capped) on the Hidden Bay Project, as of July 2009 at Various Cut-off Grades of %U_3O_8

Category	Cut-off Grade (% U_3O_8)	Tonnes	Grade (% U_3O_8)	Contained U_3O_8 (lb)
Indicated	0.02	16,876,600	0.112	41,617,000
	0.05	**10,372,500**	**0.16**	**36,623,000**
	0.10	5,434,300	0.242	28,989,000
	0.15	3,278,800	0.321	23,163,000
	0.20	2,054,800	0.409	18,503,000
	0.25	1,358,700	0.504	15,085,000
	0.30	913,800	0.616	12,408,000
	0.35	657,200	0.731	10,583,000
	0.40	506,600	0.837	9,345,000
Inferred	0.02	1,982,500	0.079	3,470,000
	0.05	**1,109,200**	**0.111**	**2,715,000**
	0.10	335,700	0.211	1,563,000
	0.15	202,800	0.27	1,208,000
	0.20	128,300	0.326	921,000
	0.25	79,200	0.388	678,000
	0.30	45,100	0.477	474,000
	0.35	27,200	0.58	348,000
	0.40	19,600	0.66	285,000

16.6 Mineral Reserve

No mineral reserves can be defined at the current time for the Hidden Bay project. By Canadian Institute of Mining definition, reserves can only be declared after a minimum of a preliminary feasibility study ("PFS") has been completed. The level of detail and accuracy of this report does not meet, nor is it intended to meet PFS standards.

16.7 Mineral Resources Extracted in the LOM Plan

Resources from the Horseshoe and Raven deposits used in the LOM mine plan were estimated to total 2.49 Mt grading 0.30% U_3O_8 from both OP and UG (see Section 18 for details). The West Bear deposit has not been incorporated into the LOM plan for this study.

17 Other Relevant Data and Information

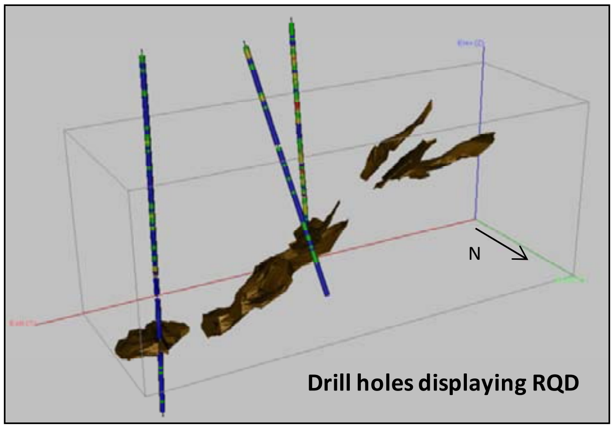
No other significant information concerning the Horseshoe and Raven deposits and their local area is considered relevant to the report at this time.

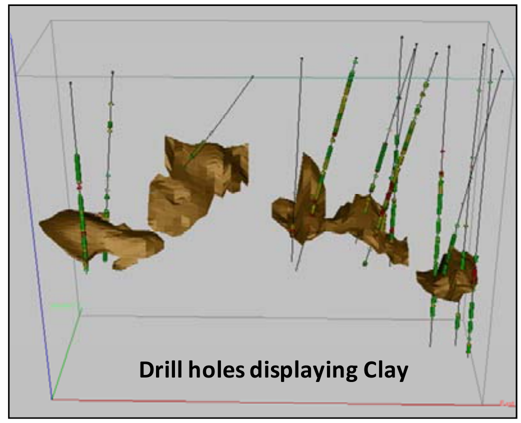
18 Mine Plan and Schedule

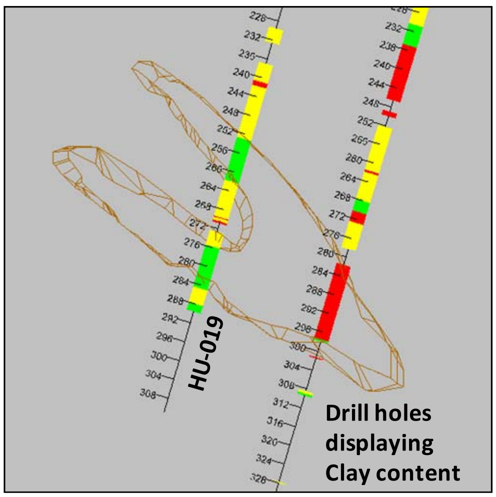
The preliminary mining investigation revealed that the Horseshoe deposit would be amenable to underground mining methods and the Raven deposit suitable for open pit (“OP”) mining. Raven also has the potential to support an underground (“UG”) operation, under the open pit, but it would need higher than the Case A U_3O_8 price of US$60/lb to be economic. For this study, therefore, all UG plans are for the Horseshoe deposit and all OP plans are for Raven.

18.1 Geotechnical Considerations

The rock mass in and around the underground mineralized zones of the Horseshoe deposit were evaluated through a review of the available Total Core Recovery (TCR), Rock Quality Designation (RQD) and clay logging values encountered in the drill holes (Figure 18.1). This process consisted of the geotechnical evaluation of each separate preliminary mining shape, which was based on a 0.16% U_3O_8 cut-off, for the likely rock mass conditions within and in immediate proximity to the to the mining shape. These core logging findings were then cross-referenced to the detailed core photos to further evaluate the likely rock mass conditions that would be encountered in each potential mining area.







TCR/RQD and IRS Colour Legend		
Lower Bound	Upper Bound	Colour
0 or R0	20 or R1	Red
20 or R1	40 or R2	Brown
40 or R2	60 or R3	Yellow
60 or R3	80 or R4	Green
80 or R4	100 or R5+	Blue

Clay Content	Colour
<2	Blank
3	Green
3	Yellow
4	Red

Figure 18.1: Example evaluation of the Stope 201, Showing RQD values and Clay Alteration Levels in Isometric and Section Views

Based on the geometry and dip of the mineralized zone and the assessment of the rock mass quality, the likely mining method was assessed, the support requirements estimated, as well as the likely level of extraction that could likely be achieved

18.2 Mining Context

SRK received the geological data and the block model (using capped OK grades) supplied by Golder and the same as those used for the Horseshoe and Raven deposit mineral resource estimates. The deposits were evaluated for open pit and underground mining methods. SRK determined that Raven deposit would be mined by open pit methods and the Horseshoe deposit by underground methods.

The main geological and geotechnical contextual considerations for the Horseshoe deposit are:

- Deposit Geometry:
  – The size of deposit is about 200 m by 600 m;
  – The deposit dip varies from 5° to 45°;
  – Depth from surface varies from 120 m to 420 m;

  - The mineralization is somewhat irregular or "poddy" when defined using grade shells
  - The deposit varies in thickness up to about 10 m with horizontal width from 5 m to 25 m.
- Geotechnical Characteristics:
  - The rock mass conditions are somewhat unknown, however, it was assumed that unsupported spans of 7 m and extraction ratios of 70% for Room and Pillar and 95% for Drift and Fill mining methods could be achieved.
- Mineralization:
  - The Horseshoe mineable mineralization starts about 100 m below surface and extends down to 450 m below surface.
- Mineralized material grades:
  - About 70% of all resources are below a U_3O_8 cut-off grade of 0.16% and average grade of 0.07% U_3O_8;
  - The remaining 30 % of resources at above 0.16% of U_3O_8 cut-off grade have a U_3O_8 average grade of 0.34%.
- Groundwater
  - The groundwater conditions are not well defined for the Horseshoe deposit. It was thought that the deposit would experience groundwater inflow but it was assumed that it would be manageable with conventional mine dewatering techniques.

18.3 Underground Mining Method Selection

The choice of mining method was determined after taking into consideration all of the known contextual factors of the Horseshoe deposit. The main factors for determining an appropriate mining method for the Horseshoe deposit were: the irregular geometry of the mineralization with varying thicknesses and a 25° average dip angle, that makes the caving and sub-level open stoping mining methods unsuitable for the deposit.

Three mining methods were initially proposed to be appropriate based on the mineral deposit size, geometry and preliminary geotechnical assessment:

- Cut and Fill ("C&F") mining method to be used where the mineral deposit thickness and dip would not allow taking more than one cut at the same elevation;
- Drift and Fill ("D&F") mining method to be used for the wider portion of the mineral deposit which require more than one cut at the same elevation and where the grades of mineralized material would require a higher extraction rate;
- Room and Pillar ("R&P") mining method would be used for low grade mineralization, where D&F mining would be uneconomic.

Considering the value of the mineralized material and mining cost, which are about 35% of total on-site costs, it would be more cost effective to increase mining extraction by using more expensive D&F method with cemented backfill for most of the deposit.

18.3.1 Description of Drift and Fill Mining Method

The D&F method is used in mineral deposits that are irregular in shape and have relatively high values as almost 100% of the mineralized material may be extracted. D&F mining has proved to be most useful in the extraction of areas that are not thick enough for efficient long hole stoping.

D&F is a development-style mining method in which parallel drifts are mined in a primary secondary sequence. After the primary drift is mined, it is backfilled with cemented fill so that a drift may be mined alongside the backfilled drift exposing stable fill walls due to the cement content of the fill. Using primary-secondary stoping sequence would reduce the cement consumption as secondary stopes would require less cement or no cement at all. Various layouts may be adopted along the same theme.

An average stope size of 5.0 m wide and 4.5 m high was assumed for mining productivities and cost estimation in this study.

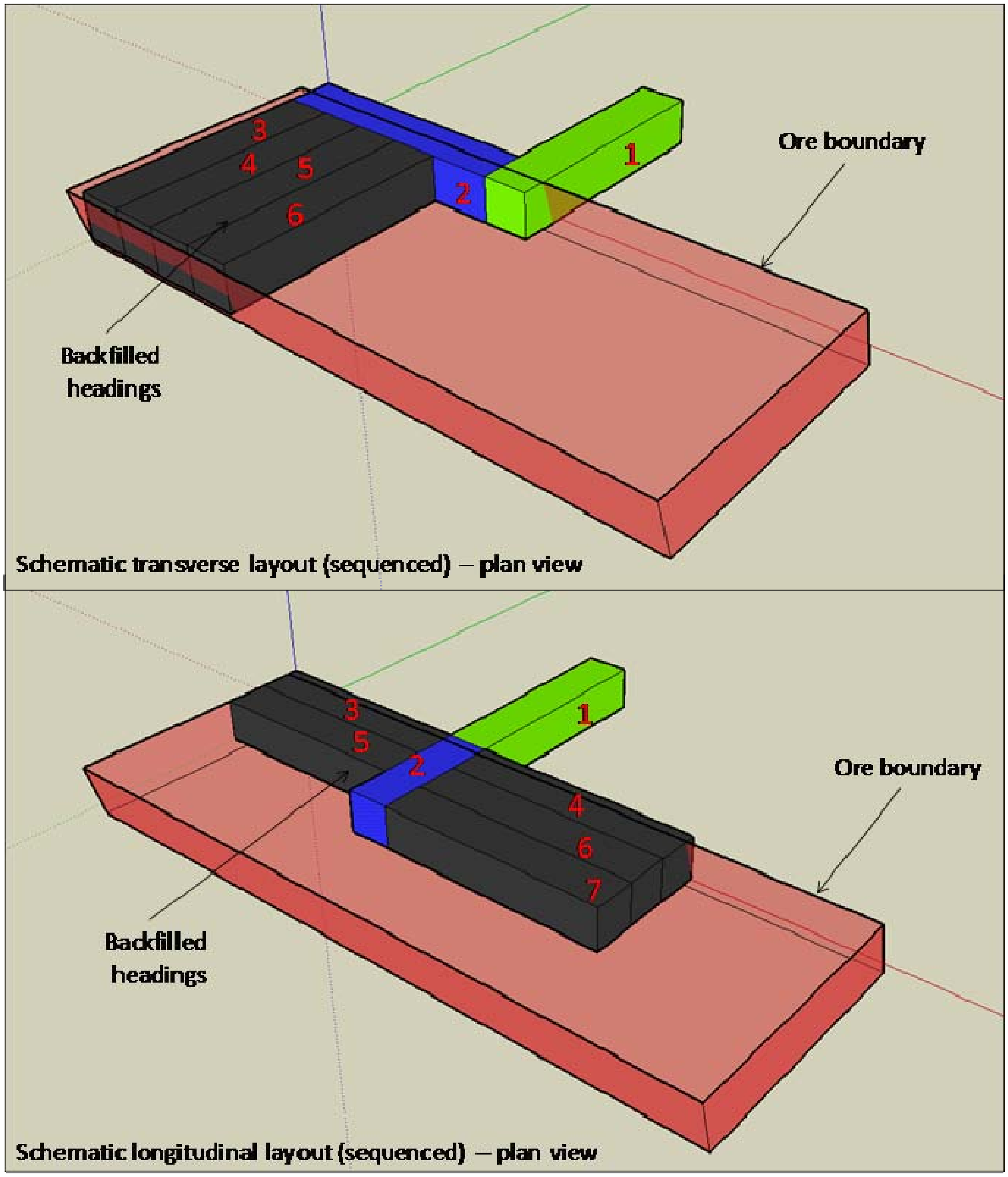


Figure 18.2: Drift and Fill Mining Method

18.3.2 Description of Room and Pillar Mining Method

The R&P mining method is an open stoping method that utilizes un-mined rock as pillars to support a series of rooms or small stopes around the pillars. The method normally is designed with pillars in a checkerboard pattern. The location of the pillars can be under survey control or done in a more random manner depending on the geotechnical requirements. The method is selective and zones of low grade can be left as pillars. Pillars can sometimes be mined on retreat to help improve the overall mining extraction.

The R&P method is normally quite productive, very flexible, and requires minimal access development before production starts.

An average stope size of 7.0 m by 7.0 m with pillar size of 5.0 m by 5.0 m was assumed for mining extraction, productivities and cost estimation.

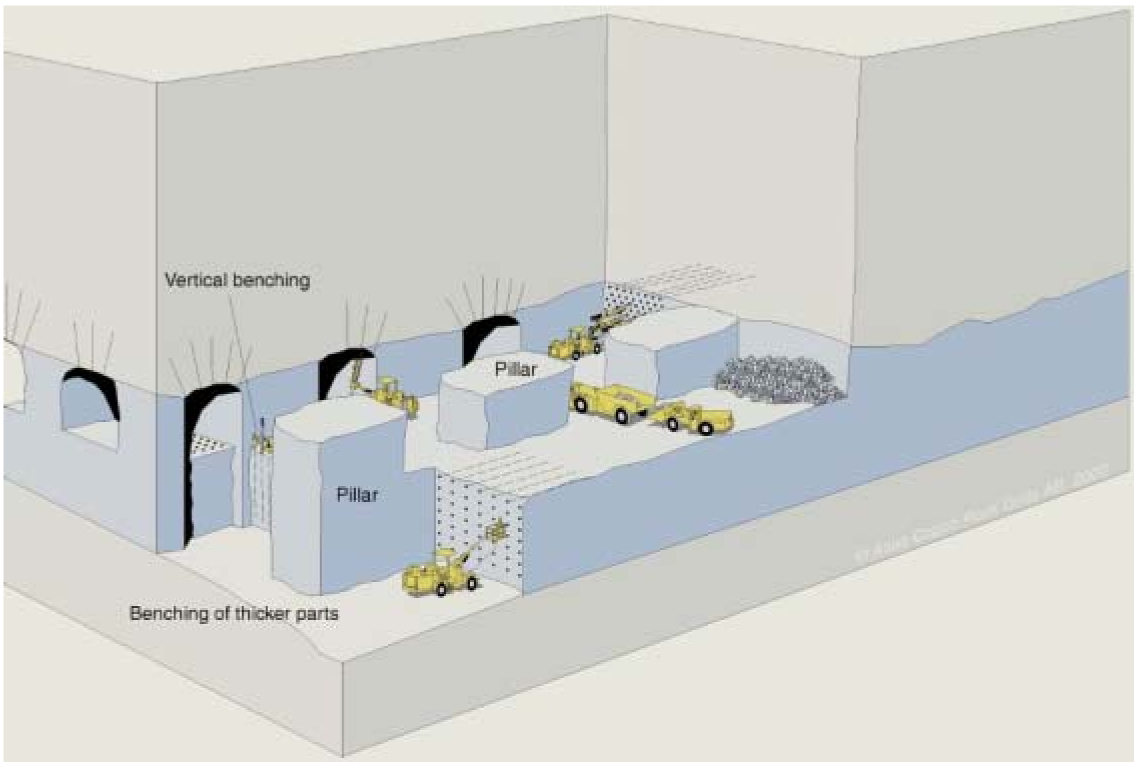


Figure 18.3: Room and Pillar Mining Method (from Atlas Copco)

18.4 Mining Inventory

18.4.1 Cut-off Criteria

Golder provided SRK with a mineral resource block model. SRK then applied mining and economic parameters to generate the mining inventory (an estimate of the quantity and grade to be mined during the life of the mine). A preliminary estimate of total on-site costs of $178 per tonne of mined material, which included mining operating cost of $68/t, toll processing cost of $70/t, tailings management cost of $35/t, and G&A cost of $5/t, were used to determine a cut-off grade.

Table 18.1 outlines the parameters used in the cut-off grade calculation.

Table 18.1: Cut-Off Grade Calculation

Item	Unit	Value
Metal Recovery		
U_3O_8 Price	\$US/lb U_3O_8	60
Exchange Rate	\$C/\$US	1.05
U_3O_8 Price	\$C/lb U_3O_8	63.16
Payable Metal	% U_3O_8	100
Process Recovery	%	96
Refining/Freight/Insurance/ Marketing	\$C/lb U_3O_8	N/A
Royalties @ 5% NSR	\$C/lb U_3O_8	3.03
Net U_3O_8 price	\$C/lb U_3O_8	57.60
Opex Estimates		
Mining Cost	$ /t milled	68.0
Toll Processing Cost (including hauling to mill)	$ /t milled	70.0
G&A/Sustaining capital cost	$ /t milled	5.0
TMF	$ /t milled	35.0
Total Site Cost	$ /t milled	178.0
Cut-off Grade		
Plant feed Cut-off Grade	% U_3O_8	0.14
Dilution	%	10
In-situ Cut-off Grade	% U_3O_8	0.16

A 0.16% U_3O_8 cut-off grade was applied for design of mining shape outlines.

18.4.2 Mining Shapes

The Gemcom / Surpac mine planning software was used for analysis of the mineral deposit geometry and grades. The sections of the deposit wireframes and block model were created at 10 m intervals. The mining shapes were outlined based on 0.16% U_3O_8 cut-off grade criteria and minimum mineable thickness of 2 m.

The mining shapes used for mining inventory estimation, underground mine design and schedule are shown on the figure below.

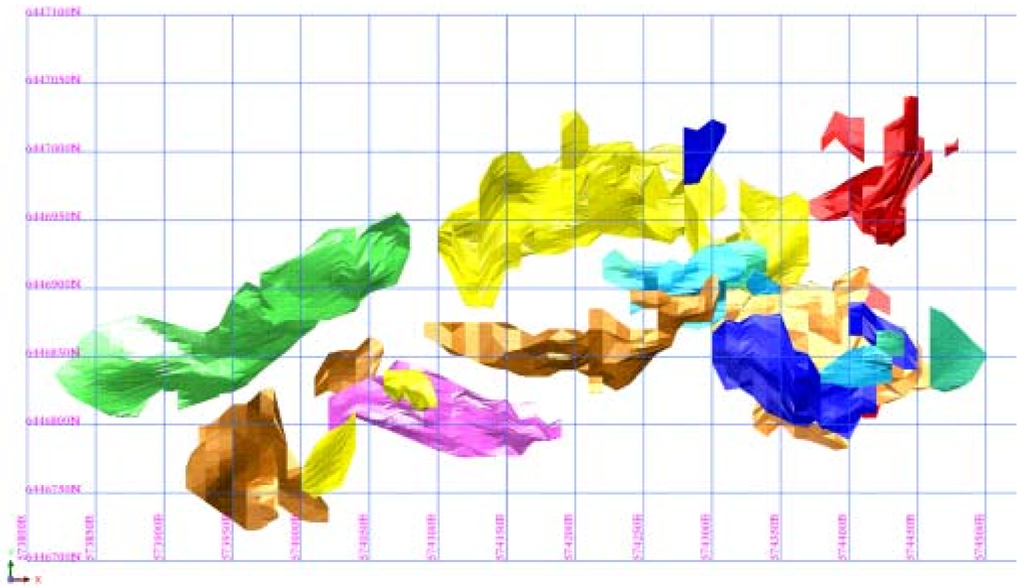


Figure 18.4: Mining Shapes

18.4.3 Dilution and Recovery

Dilution and recovery factors were applied to each individual mining block based on the various mining thicknesses, mineral deposit dip and mining method to be used. Dilution was defined as the ratio of waste to mineralized material. Two sources of dilution would be expected in the mine: internal and external dilution.

Internal dilution derives from material with grades that are less than a cut-off grade that falls within a designed stope boundary (i.e. it will be drilled and blasted within the stope during mining). In selective mining, in many cases where less than cut-off grade zones are encountered, they may be left as pillars, or mined as waste due to the highly flexible nature of D&F and R&P mining.

External dilution derives from low or zero grade material from beyond the stope design boundaries due to blasting overbreak, adverse geological structure, failure within zones of weak rock, and when mucking on the top of backfill material. External dilution is almost always generated and an allowance is always made for it during the reserve estimation process.

A recovery factor was assumed for each stope to account for losses in the permanent pillars, left on the floor, or gets mixed into the backfill floor and is left behind.

18.4.4 Mining Inventory

The following assumptions were used for mining inventory estimates:

- Mineralized material inside the mining shapes with grades above 0.1% U_3O_8 would be considered as incremental and would be sent to the mill;
- Mineralized material with grades below 0.1% U_3O_8 inside the mining shapes would be left in pillars due to selective mining methods, or mined as waste and not be sent to the mill;
- 95% extraction for Drift and Fill and 80% extraction for Room and Pillar mining method;
- Depend on geometry and size of mining shapes 10% to 25% external dilution with 0 grades due to waste dilution and dilution from backfill.

The summary of mining inventory estimate is shown in the table below. The tonnages and grades were extracted from the block model and dilution and recovery factors applied depend on mining shape and thickness.

Table 18.2: Mining Inventory @ 0.16% U_3O_8 Cut-off

Mining Block	In-situ Resources			Dilution %	Recovery %	Total Diluted Recovered		
	Tonnes Kt	U_3O_8 %	U_3O_8 Klb			Tonnes Kt	U_3O_8 %	U_3O_8 Klb
100	154	0.99	3,347	10%	95%	161	0.90	3,179
102	181	0.35	1,417	10%	95%	190	0.32	1,344
103	73	0.35	565	10%	95%	76	0.32	537
104	40	0.30	263	15%	95%	44	0.26	249
201a	251	0.36	2,005	10%	95%	262	0.33	1,896
201b	565	0.27	3,348	10%	90%	560	0.24	2,997
301	328	0.23	1,662	10%	75%	271	0.21	1,249
401	21	0.25	117	10%	95%	22	0.23	111
501	32	0.31	220	15%	95%	35	0.27	211
502	119	0.38	991	15%	95%	130	0.33	935
503	128	0.43	1,204	10%	95%	134	0.39	1,149
601	11	0.22	51	25%	95%	12	0.17	48
606	6	0.24	32	25%	95%	7	0.19	30
610	12	0.22	56	25%	95%	14	0.18	54
801	204	0.25	1,141	10%	80%	180	0.23	915
Total	**2,125**	**0.35**	**16,421**	**11%**	**89%**	**2,097**	**0.32**	**14,906**
Including								
Indicated	**2,049**	**0.35**	**15,904**			**2,022**	**0.32**	**14,436**
Inferred	**77**	**0.31**	**517**			**75**	**0.28**	**470**

18.5 Conceptual Mine Design

18.5.1 Mine Access

The access to the mining blocks via decline developed at a -15% gradient was considered as the most appropriate method based on the geometry of the deposit, depth and location of mineralized zones, proposed mining method, and mine life.

It would provide early access to mineralized material and ability to start production at earlier stage, ability to follow the mineralized zone down-dip, provide multiple accesses to the mining blocks through the cross-cuts, reduce initial capital cost and provide opportunity to carry out delineation drilling of the resource.

The main decline would be used for haulage, access for personnel, equipment, materials, and services. It would also be utilized as a major intake airway.

The size of the decline was selected according to the mobile equipment size, required clearances, and ventilation requirements during development and production. It was estimated that a 4.6 m wide by 4.6 m high decline would be satisfactory for a 20 t truck and ventilation requirements for 1,000 t/d production rate Figure 18.5). A 25 m ramp curve radius was assumed for convenience to drive a mobile drill jumbo. The general arrangement of the decline is shown in Figure 18.6 and Figure 18.7.

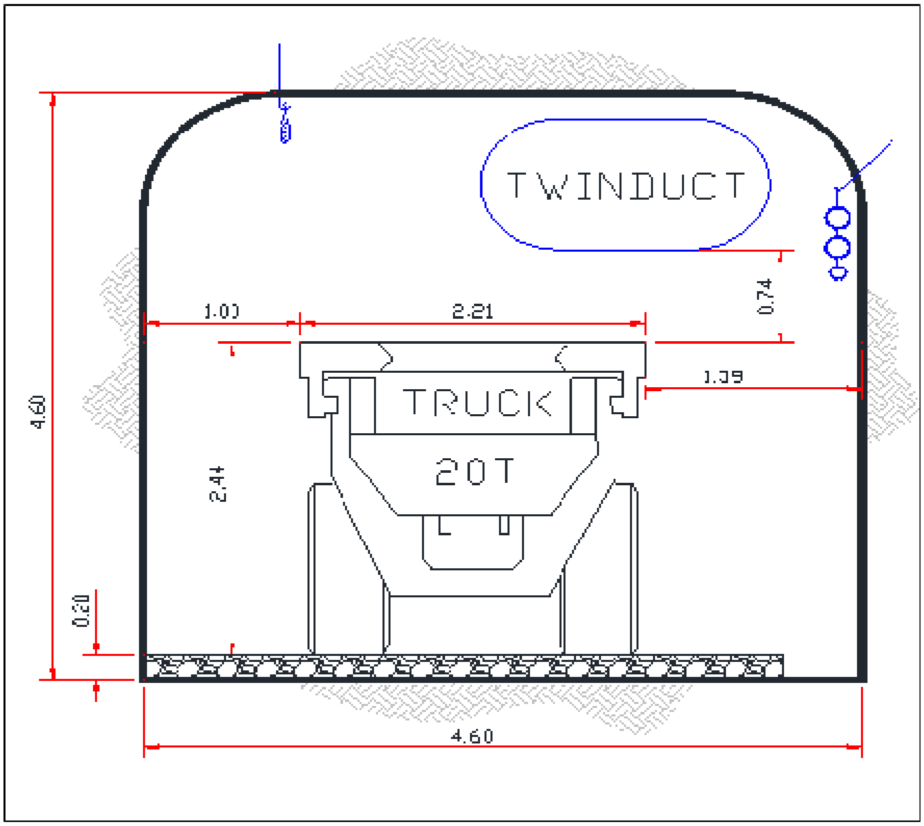


Figure 18.5: Proposed Decline Cross Section

Re-muck bays were planned to be developed every 150 m along the decline to allow efficient use of the drilling equipment and would hold two rounds of development muck. The re-muck bays would be of a similar size as the decline and would be typically 15 m long. After they are no longer used for development, the bays would be used for equipment storage, pump stations, drill bays, refuge chambers, etc.

Passing bays are proposed to be mined periodically in the ramp, in zones of competent ground, to allow for the trucks to pass each other.

Installation of 2.4 m fully grouted resin rebar bolts on the back and the walls of the ramp on 1.2 m x 1.2 m pattern, 100% mesh coverage and an allowance of 50 mm of shotcrete for 15% of the total length of the ramp was assumed for ground support.

The return ventilation raise was planned to be carried down with the main ramp to provide a primary ventilation circuit. This would allow the return airway to stay in close proximity to the main ramp development face and would provide favourable ventilation conditions for ramp development.

The 120 m of the top portion of the ventilation raise from the surface to the 300 m elevation was planned to be developed using raise-borer method. The lower portions of the raise between sub-levels are short (from 15 m to 30 m long) and would be developed using drop raises technique or other conventional methods of raise development.

The ventilation raise would have a man-way equipped with ladders and platforms to provide an auxiliary exit from the mine in case of emergency.

Ventilation access drifts would be developed to connect the level development and ramp to the ventilation raises. Those drifts would be 15 m to 40 m long and in some cases could be developed at -15% gradients to reduce length of the raise.

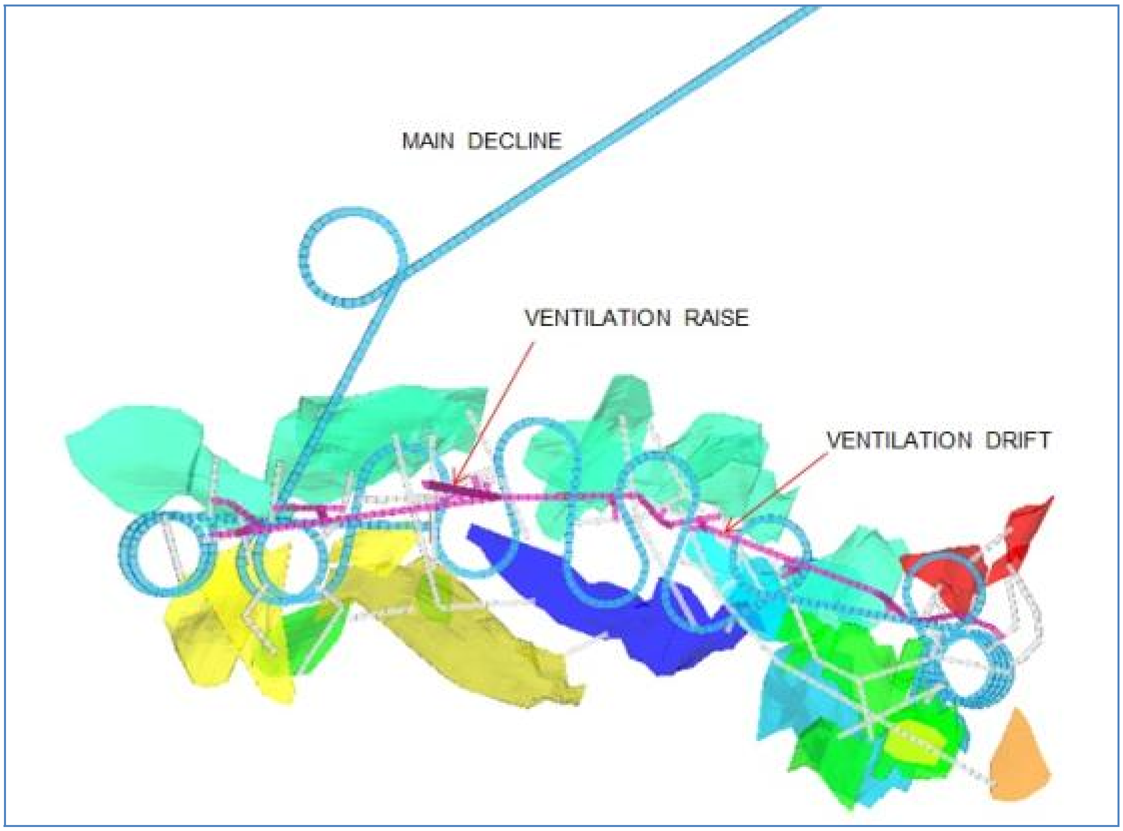


Figure 18.6: Underground Access Plan View

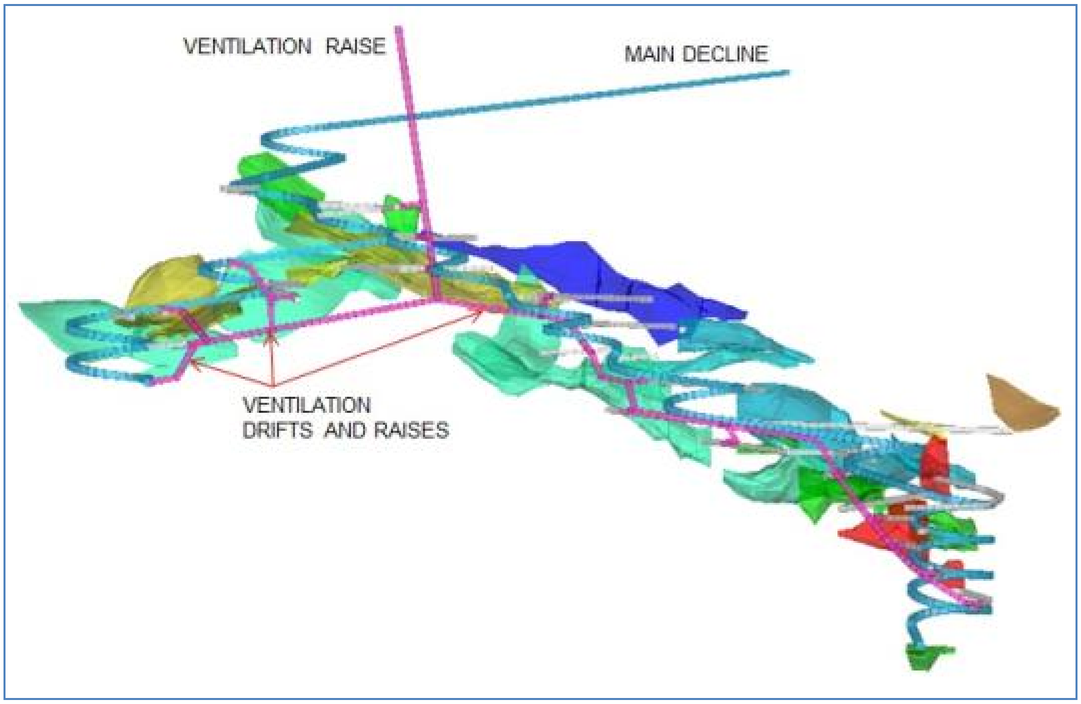


Figure 18.7: Underground Access Isometric View

18.6 Unit Operations

18.6.1 Stoping

All mining are proposed to be carried out by electric-hydraulic, two-boom jumbos, taking 3.85 m rounds with 45 mm diameter drill holes. The holes would be loaded with ANFO from a pneumatic loader and blasting initiated with NONEL caps.

Mucking would be carried out by diesel LHDs loading underground trucks. The trucks are planned to be loaded near the mining face.

Ground support rock bolts would be installed by mechanized rockbolters, and in some cases, installed manually from the blasted rock muck pile, using jacklegs and stopers. Fully grouted 2.4 m long resin rebars would be installed on 1.5 m square pattern.

Pipelines, ventilation ducts and power cables are planned to be installed as the heading advances to maintain services near the working face.

18.6.2 Haulage

The waste rock from the development headings is planned to be mucked by LHDs directly to the trucks or to remuck bays located up to 150 m from the face. The waste rock would be hauled by the 20 t trucks to the waste dump on surface during the pre-production period. When underground mine production commences, it is proposed to use mine waste rock from development as stope backfill.

The broken mineralized material from the stopes is planned to be mucked by stope LHDs and loaded directly onto underground trucks. The trucks would= carry mineralized material from the mine directly to the stockpile on surface. For the haulage cycle time estimation, it was assumed that the stockpile will be located on the surface, within 200 m of the portal.

The same trucks would carry backfill material from the surface to mined-out stopes. Coordination of the trucks will be necessary to ensure safe and efficient haulage.

Two different sizes of stope LHD (2.0 m^3 and 3.7 m^3) were selected to satisfy different stope size requirements.

18.6.3 Backfill

The D&F mining method is designed to use cemented rock fill (“CRF”). CRF consists of waste rock mixed with cement slurry to improve the bond strength between the rock fragments. Waste rock would be sourced from the open pit operation or existing waste stockpiles at Rabbit Lake and would be supplemented with underground development waste when available. Using waste rock as a backfill material would reduce the environmental impact on surface.

The rock would be screened, then mixed with cement slurry and placed in the mined out stopes underground. Cement slurry concentration was assumed to be approximately 55% by weight (1.2:1 water : cement ratio).

The same trucks used to haul mineralized material to the surface stockpile would be used to bring CRF underground into mined-out stopes. Back-hauling backfill would offer an efficient, cost-effective system.

One of the advantages of CRF would be a high strength to cement content ratio. A cement binder content of 4% has been assumed for CRF and used for backfill cost estimation purpose. The primary-secondary stoping sequence would reduce the cement consumption as secondary stopes would require less cement or no cement at all. For backfill cost estimation purposes it was assumed that 75 % of D&F stopes would require cemented backfill.

Backfill testing would be required for future studies to define the optimum cement binder content, backfill strength and backfill curing time to obtain required strength.

18.6.4 Mine Services

Ventilation

The design basis of the ventilation system at Hidden Bay underground operation was to adequately dilute exhaust gases produced by underground diesel equipment and minimize radon daughter exposure.

Air volume was calculated on a factor of 0.06 m^3/s per installed kW of diesel engine power. The kW rating of each piece of underground equipment was determined and then utilization factors, representing the diesel equipment in use at any time, applied to estimate the amount of air required. Ventilation losses were included at 20% of the total ventilation requirements. Table 18.3 lists the air requirements for full production with the total of 119 m^3/s flow rate.

Table 18.3: **Ventilation Requirements at Full Production**

Description	Quantity	Diesel (kW)	Utilization (%)	Utilized (kW)	Air Volume (m^3/s)
Jumbo (2-boom)	2	74	10	15	0.9
Rockbolter	2	55	20	22	1.3
LHD, 3.7 m3	2	201	80	240	14.4
LHD, 2.0 m3	1	71	80	57	3.4
Truck, 20 t	5	240	80	960	57.6
Grader	1	149	30	45	2.7
ANFO Loader	2	93	30	56	3.4
Cassette Carrier	2	112	50	112	6.7
Mechanics Truck	1	93	25	23	1.4
Scissor Lift	1	112	30	34	2.0
Supervisor Vehicle	3	93	20	56	3.4
Electrician Vehicle	1	93	30	28	1.7
Sub-Total				**1,647**	**99**
Losses	**20%**				**20**
Total Air Requirements					**119**

The required air volume estimated based on diesel equipment was the same as estimated based on radiation produced. A 2 hours break between production shifts would provide time for ventilation to clear the fumes from blasting operations and reduce radiation.

Air velocity in the main ramp was restricted to a range of 0.25 m/s to 6 m/s. This range was used to determine the size of development.

The main exhaust fan would be installed on surface at the collar of the ventilation raise. An exhaust ventilation system is the most efficient method of diluting and removing radon progeny from the underground workings. The dedicated exhaust drifts will be required to connect each of the production drifts to the exhaust ventilation raise. During production, fresh air was designed to be downcast through the main ramp and exhaust up-cast through the ventilation raise.

The ventilation system design was modelled using Ventsim Mine Ventilation Simulation Software (Ventsim). This software allows input parameters including resistance, k-factor (friction factor), length, area, perimeter, and fixed quantities (volume) of air. The ventilation circuits during the initial production and in Phase 2 of production are presented in Figure 18.8 and Figure 18.9, respectively.

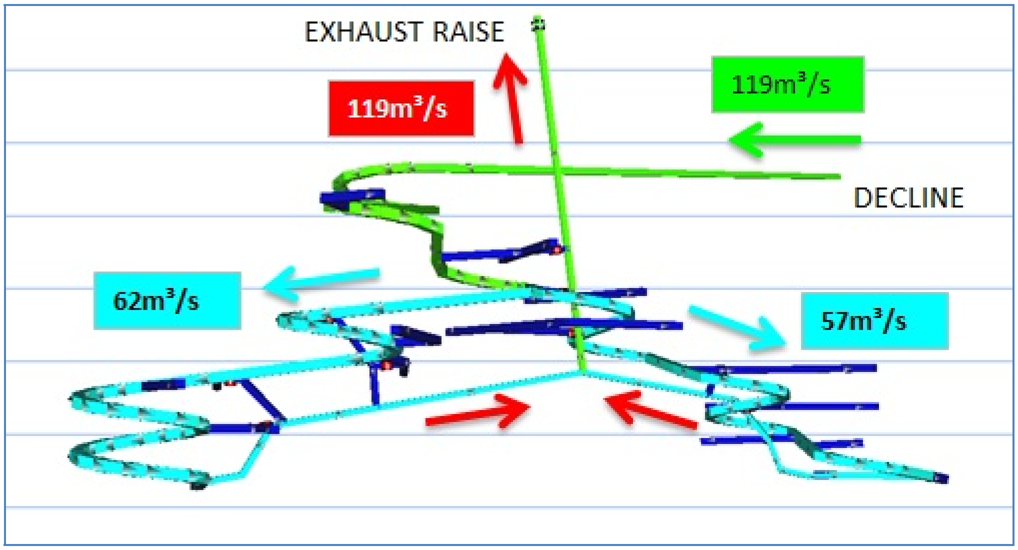


Figure 18.8: Ventilation Circuit at Start Production

Figure 18.9: Ventilation Circuit in Phase 2 of Production

A number of auxiliary ventilation fans would provide ventilation of the development headings and production stopes. Ventilation regulators, doors, and bulkheads would be also used to control airflow in the mine.

Ventilation of Headings during Development

An air flow of 24 m^3/s would be required for a 20 t truck, a 3.7 m^3 LHD, and a two-boom jumbo working in a development heading.

Table 18.4: Ventilation Requirements for Development Heading

Description	Quantity	Diesel (kW)	Utilization (%)	Utilized (kW)	Air Volume (m^3/s)
LHD, 3.7 m3	1	150	100	150	9
Truck, 20t	1	240	100	240	14
Jumbo, two-boom	1	74	10	7	1
Total					**24**

The requirements for auxiliary ventilation were estimated for the 1,200 m long development heading, as the longest decline development distance required by mine development program. The auxiliary ventilation fans and ventilation ducts would be used to provide required amount of air at the development face. Only resistance of the duct was considered to calculate the pressure loss and power requirements as the resistance of the heading is negligible by comparison. Using Atkinson's equation for air flow in ducts, two 55 kW auxiliary fans with 1.5 m diameter ventilation duct, or twinduct of equivalent size, would be required for the decline development.

Table 18.5: Atkinson Equation for Air Flow in Ventilation Ducts

Duct Diameter (m)	Duct Area (m^2)	Duct Perimeter (m)	Air Volume (m^3/s)	Duct Air Velocity (m/s)	Friction Factor (kg/m^3)	Duct Length (m)	Pressure Loss (kPa)	Power Required (kW)	Fan Power (kW)
1.0	0.79	3.14	24	30.4	0.003	1,200	13.1	313	417
1.2	1.13	3.77	24	21.1	0.003	1,200	5.3	126	168
1.3	1.33	4.08	24	18.0	0.003	1,200	3.5	84	112
1.4	1.54	4.40	24	15.5	0.003	1,200	2.4	58	78
1.5	1.77	4.71	24	13.5	0.003	1,200	1.7	41	55

The same type of auxiliary fan with 1.2 m diameter duct is assumed for ventilation of other development headings and production stopes with average length of 200 m.

Mine Air Heating

Heating of the intake air would be required during the winter months to prevent water freezing underground and to provide acceptable conditions for underground workers and equipment. Mine air would be heated to +2 C by a direct-fired propane heater located at the portal. A parallel portal and drift would house the propane heater infrastructure. The air in the propane heater drift will be heated and blended with the cold air entering through the main decline, allowing vehicles and personnel to enter the mine without the use of double air lock doors.

It was estimated that approximately 1.33 million litres of propane per year would be required to heat the mine air during five months from November to March.

Underground Electrical Power Distribution System

The major electrical power consumption in the mine would be from the following:

- Main and auxiliary ventilation fans;
- Drilling equipment;
- Mine dewatering pumps;
- Air compressors; and
- Maintenance shop.

High voltage cable would enter the mine via the decline and be distributed to electrical sub-stations located near production stopes. The power cables would be suspended from the back of development headings. All equipment and cables would be fully protected to prevent electrical hazards to personnel.

High voltage power would be delivered at 4.16 kV and reduced to 600 V at electrical sub-stations. All power would be three-phase. Lighting and convenience receptacles would be single phase 120 V power.

The following list of equipment would require power usage for underground mine.

Table 18.6: Power System Requirements for Underground Mine

Description	Quantity	Unit (kW)	Load Factor (%)	Utilization (%)	Power Consumption (kW/yr)
Surface					
Maintenance Shop	1	200	80	30	350,400
Surface Misc. (office, lighting, etc.)	1	50	80	30	146,000
Main Ventilation Fan	1	300	80	100	2,102,400
Underground					
Jumbo, two-boom	2	135	95	60	1,123,500
Rockbolter	2	70	95	60	582,500
Exploration Drill	1	75	95	50	156,000
Portable Compressor	2	100	80	30	350,400
Portable Welder	1	34	80	10	23,800
Auxiliary Fan	8	55	80	90	2,775,000
Refuge Chamber	2	5	80	100	70,000
Main Dewatering Pump	1	100	85	50	372,300
Portable Pump	3	15	85	30	55,500
Sub-total					8,108,000
Miscellaneous Power Allowance	10%				811,000
Total Power					**8,919,000**

Underground Communication System

A leaky feeder communication system would be used as the primary communication system for mine and surface operations. Telephones will be located at key infrastructure locations such as the electrical sub-stations, refuge stations, and main sump.

Key personnel (such as mobile mechanics, crew leaders, and shift bosses) and mobile equipment operators (such as loader, truck, and utility vehicle operators) would be supplied with an underground radio for contact with the leaky feeder network.

Compressed Air

The mobile drilling equipment such as jumbos, rockbolters, and scissor lifts with ANFO loaders would be equipped with their own compressors. No reticulated compressed air system was envisioned to be required.

Two portable compressors would be required to satisfy compressed air consumption for miscellaneous underground operations, such as: jackleg and stoper drilling, secondary pumping with pneumatic pumps.

Explosives Storage and Handling

Explosives would be stored on surface in permanent magazines. Detonation supplies (NONEL, electrical caps, detonating cords, etc.) would be stored in a separate magazine.

Underground powder and cap magazines would be prepared underground. Day boxes would be used as temporary storage for daily explosive consumption.

Ammonium nitrate (AN) and fuel oil (FO) would be used as the major explosive for mine development and production. Packaged emulsion would be used as a primer and for loading lifter holes in the development headings. Smooth blasting techniques may be used as required in main access development headings, with the use of trim powder for loading the perimeter holes.

During the decline development, blasting in the development headings would be done at any time during the shift when the face is loaded and ready for blast. All personnel underground would be required to be in a designated Safe Work Area during blasting. During production period, a central blast system would be used to initiate blasts for all loaded development headings and production stopes at the end of the shift. All blasting in the mine would be development-style blasting. No large scale blasts (“mass blasts”) would be undertaken.

Fuel Storage and Distribution

Haulage trucks, LHDs, and all auxiliary vehicles would be fuelled at fuel stations on surface. The fuel/lube cassette will be used for refuelling and lubrication of drills and rock bolters.

An average fuel consumption rate of approximately 3,800 l/d is estimated for the period of full production as shown in Table 18.7.

Table 18.7: Underground Mining Fuel Consumption

Description	Quantity	Consumption (l/hr)	Load Factor (%)	Utilization (%)	Total Fuel (l/day)
LHD, 3.7 m3	2	40	75	66	570
Truck, 20 t	5	51	75	77	2,150
Jumbo, two-boom	2	22	75	10	50
Rockbolter	2	18	75	25	100
Grader	1	36	75	30	120
ANFO Loader	2	22	75	30	140
Cassette Carrier	2	27	75	50	270
Mechanics truck	1	22	75	25	60
Scissor Lift	1	27	75	30	90
Supervisor Vehicle	3	18	75	25	150
Electrician Vehicle	1	22	75	30	70
Forklift	1	16	75	25	40
Total					**3,800**

Mine Dewatering

The main sources of water inflow to the underground mine are anticipated to be from groundwater and drilling operations. There was no information available at the time of writing on the local hydrogeological conditions. A hydrogeological study is required to estimate underground water inflow rates.

The main sump would typically be a two-bay design to allow suspended solids to settle out of the water before pumping. It would be usually located at the bottom level of the mine.

Water was planned to be pumped from the main sump by a high-pressure pump through a 6" diameter steel pipe located in the main ramp to the final tailing pump box on surface. Pumping may require multiple pumping stages, depending on depth and flow. Each sump would be equipped with two high-head submersible pumps – one for operation and one on standby.

Old remuck bays, planned to be located every 150 m along the access ramp, would be utilized as temporary sumps during main access ramp development.

Transportation of Personnel and Materials Underground

All mine supplies and personnel would access the underground workings via the main access decline.

Two Maclean CS-3 Carriers with personnel cassettes would be used to shuttle men from surface to the underground workings and back during shift changes. Supervisors, engineers, geologists, and surveyors would use diesel-powered Toyota trucks as transportation underground. Mechanics and electricians would use the mechanics' truck and maintenance service vehicles.

A Maclean CS-3 boom deck with a Hiab 095 10-t crane would be used to move supplies, drill parts, and other consumables from surface to active underground workings.

Underground Construction and Mine Maintenance

A mine service crew will perform the following:

- Mine maintenance and construction work;
- Ground support control and scaling;
- Road checking and maintenance;
- Construction of ventilation doors, bulkheads, and concrete work;
- Mine dewatering.
- Safety work.

An underground grader and scissor lift will be utilized to maintain the main declines and active work areas.

Equipment Maintenance

Mobile underground equipment would be maintained in a mechanical shop located on the surface. Some small maintenance and emergency repairs would be performed underground. A mechanics truck would be used to perform emergency repairs underground.

Major rebuild work would be conducted off site.

A maintenance supervisor would provide a daily maintenance work schedule, ensure the availability of spare parts and supplies, and provide management and supervision to maintenance crews. He would also provide training for the maintenance workforce.

A maintenance planner would schedule maintenance and repair work, as well as provide statistics of equipment availability, utilization and life cycle. A computerized maintenance system is recommended to facilitate planning.

The equipment operators would provide equipment inspection at the beginning of the shift and perform small maintenance and repairs as required.

Mine Safety

The portable refuge stations would be provided in the main underground work areas. The refuge stations were designed to be equipped with compressed air (medical grade), potable water, and first aid equipment; they will also be supplied with a fixed telephone line and emergency lighting. The refuge chambers can be sealed to prevent the entry of gases.

The portable refuge chambers would be move as the working areas advance, eliminating the need to build permanent refuge stations.

Fire extinguishers would be provided and maintained in accordance with regulations and best practices at the underground electrical installations, pump stations, fuelling stations, and wherever a fire hazard exists. Every vehicle would carry at least one fire extinguisher of adequate size and proper type. All underground mobile vehicles would be equipped with automatic fire suppression systems.

A mine-wide stench gas warning system would be installed at the main intake raise to alert underground workers in the event of an emergency.

The main access decline would provide primary access and the ventilation raises with dedicated manway would be equipped with ladders and platforms providing the secondary exit in case of emergency.

Radon Monitoring

Continuous radon progeny monitoring devices would be used in the all mining working places. Local lamp-boxes would be installed to alert personnel of action levels to be taken. All regulatory measures as related to radioactive dusts would be adhered to in the mine operations planning and training

18.6.5 Mine Equipment

Criteria used in the selection of underground mining equipment include:

- Mining Method;
- Mineral deposit geometry and dimensions;
- Mine production rate;
- Ventilation requirements;
- Operating and capital cost.

Table 18.8 lists underground mobile equipment required for 1,000 t/d mine production rate.

Table 18.8: Underground Mobile Equipment List

Equipment	Type	Quantity
Drilling Equipment		
Development Jumbo (2 boom)	Tamrock DD320-26C	1
Production Jumbo (2 boom)	Tamrock DD320-26C	1
Rockbolter	Tamrock DS 310	2
Loading & Hauling Equipment		
Production / Development LHD, 3.7 m^3	LH 307	2
Production LHD, 2.0 m^3	LH 203	1
Haulage Truck, 20 t	TH 320	5
Service Vehicles		
Grader	GR 12 H	1
ANFO Loader	Toyota HZJ79	2
Cassette Carrier	Maclean CS-3	2
Personnel Cassette	Maclean Personnel	2
Boom Cassette	Maclean Flat Deck	1
Fuel / Lube Cassette	Maclean Fuel / Lube	1
Mechanics Truck	Maclean MT-3	1
Scissor Lift	Maclean SL-3	1
Supervisor/Engineering Vehicle	Toyota HZJ79	3
Electrician Vehicle - Scissor Lift	Toyota HZJ79	1
Shotcrete Sprayer	Maclean SS-3	1
Transmixer	Maclean TM-3	1
Forklift	Toyota	1

The equipment list was developed based on the scheduled quantities of work and estimated from first principle cycle times and productivities (83% operational efficiency was used accounting for 50 min of usable time in one operating hour). Some other efficiency factors such as: 80% efficiency for the second boom on the drill jumbo, fill factors for LHD and trucks, additional time for travel, setup and teardown were used in cycle time estimations. The number of operating units was calculated based on 88% shift efficiency (shift change, lunch break, and equipment inspection time were excluded from the shift hours) and then converted to a fleet size by accounting for 80% equipment mechanical availability.

Stationary equipment was selected and would be installed and used for the following:

- Primary and auxiliary ventilation;
- Compressed air;
- Mine water management;
- Underground electrical;
- Communication;
- Mine safety;

- Explosives storage; and
- Engineering equipment.

18.6.6 Personnel

The mining employees at the Hidden Bay underground operation were divided into two categories: salaried personnel, and hourly labour.

The personnel requirement estimates were based on the following:

- A 1,000 t/d production rate; and
- A crew rotation of two 10-h shifts per day with two crews working on site and one crew off.

A mining contractor would begin work in the pre-production development stage of the mine life to allow time for the Owner to recruit staff for the project. The labour and personnel requirements described in this section do not include pre-production development, which would be performed by the contractor.

Salaried personnel requirements, including engineering, technical, and supervisory staff, are listed in Table 18.9.

Table 18.9: Technical and Supervisory Staff

	Quantity
Staff Mine Operation	
Mine Superintendent	1
Mine Captain	1
Mine Supervisor/Shift Boss	3
Senior Mining Engineer	1
Mine Ventilation/Project Engineer	1
Geotechnical Engineer	1
Senior Geologist	1
Geologist Technician / Sampler	3
Mine Rescue/Safety Officer	1
Surveyor / Mine Technical	2
Surveyor Helper	1
Total Operating Staff	**16**
Staff Mine Maintenance	
Maintenance Superintendent	1
Maintenance Planner	1
Mechanical / Electrical Foreman	1
Maintenance Supervisor/Shift Boss	3
Total Mine Maintenance Staff	**6**
Total Mining Staff	**22**

Hourly personnel were estimated based on production and development rates, operation productivities, and maintenance requirements. Personnel productivities were estimated for all main activities by developing cycle times for each operation.

Hourly labour requirements at full production are listed in Table 18.10.

Table 18.10: Hourly Labour

Labour Description	Personnel per Shift	Personnel per Day	Total Payroll
HOURLY MINE LABOUR			
Production / Development			
Jumbo Operator	2	4	6
Ground Support	4	8	12
Blaster	1	2	3
Blaster Helper	1	2	3
Haulage			
Scoop-Loader Operator	2	4	6
Truck Drivers	5	10	15
Exploration			
Diamond Driller	1	1	1
Diamond Drill Helper	1	1	1
Mine Services & Safety			
Service Crew	1	2	3
Grader Operator	1	1	1
Utility Vehicle Operator/Nipper	1	2	3
General Labourer	1	2	3
Sub-total Mine Operating	**21**	**39**	**57**
MINE MAINTENANCE			
HD Mechanic	2	4	6
Electrician	2	4	6
Welder	1	1	1
Tireman	1	1	1
Instrument Man	1	1	1
HD Mechanic Apprentice	1	1	1
Millwright	1	1	1
Sub-total Mine Maintenance	**8**	**11**	**14**
Total Mine Operating	**29**	**50**	**71**

18.7 Open Pit Mine Plan

18.7.1 Whittle™ Open Pit Optimization

The 3D mineral resource block model (based on a 5 m x 5 m x 2.5 m block size) as provided by Golder was used as the basis for deriving the economic pit limit for the Raven deposit. Mine design for the Hidden Bay deposits was initiated with the development of Whittle™ input parameters. These parameters included estimates of:

- Metal price (US$60/lb U_3O_8);
- Exchange rate;
- Toll milling and mining costs;
- Mining dilution;
- Geotechnical pit slope parameters;
- Mill recovery; and
- Royalties.

Table 18.11 summarizes the various input parameters.

Table 18.11: Whittle™ Optimization Parameters

Item	Unit	Value
Bulk Density		
Mill Feed	t/m^3	varies in model
Waste	t/m^3	2.48
Overburden	t/m^3	N/A
Metal Prices		
U_3O_8	$US/lb	$60.00
U_3O_8	C$/lb	$63.16
Process Recovery		
U_3O_8	%	96
Site Operating Costs		
Toll milling (includes ore haul cost to mill)	C$/t ore	$70.00
G&A/Sustaining Capital	C$/t ore	$5.00
Incr. Mining Cost	C$/t ore	N/A
TMF	C$/t ore	$35.00
On Site Cost	C$/t ore	**$110.00**
Mining Costs		
Open Pit Mineralized Zonemining	C$/t mined	$2.70
Open Pit Waste mining - rock	C$/t mined	$2.70
Open Pit Waste mining - overburden	C$/t mined	N/A
Underground mining cost	C$/t mined	N/A
TC/RC		
Refining/Freight/Insurance/ Marketing	C$/lb	N/A
Pit Parameters		
Pit slope angles with ramps		
Overburden	overall °	N/A
Basement Rock	overall °	45
Bench height	m	10
Mining Recovery	%	100
Dilution (@ 0% U_3O_8 grade)	%	10
Production capacity	ore t/yr	1,095,000
Economics		
Exchange rate	C$:US$	1.05
Royalties (% of gross U_3O_8 sales)	%	5.0
Discount Rate	%	10.0
Operating Parameters		
Operating Days	days/yr	365
Shift Schedule	shifts/day	2
Scheduled Shifts	shifts/year	730
Operating Crews	#	4
Energy Cost		
Diesel Fuel Cost	C$/litre	1.00
Electric Power Cost	C$/kWh	0.10

**These parameters were the initial assumptions made to begin the mine planning process. Some of the parameters changed as more detailed work was conducted. For example, the process recovery of U_3O_8 of 96% was used in the optimization and then modified to 95% for the economic analysis as the recovery was finalized by the QP. The processing costs also changed from this preliminary estimate, done at an assumed head grade of 0.16% U_3O_8, to the final costs estimated using the ROM grade of 0.30% U_3O_8.*

18.7.2 Economic Open Pit Limit

The ultimate economic pit limit was based on a Whittle ™ pit optimization evaluation of the resources in the Raven model. This evaluation included the aforementioned Whittle™ parameters. The economic pit limit included indicated and inferred mineral resources (there are no measured resources in the model).

18.7.3 Open Pit Cut-off Grade

The base case economic parameters mentioned above were used to calculate cut-off grade for the Raven deposit (see Table 18.12 below). The incremental cut-off grade incorporates all operating costs except mining. This cut-off is applied to material contained within an economic pit shell where the decision to mine a given block was determined by the Whittle™ optimization. This incremental cut-off of 0.096 % U_3O_8 was applied to all of the open pit mineral resource estimates that follow.

Table 18.12: Cut-off Grade Calculations*

Parameter	Unit	RAVEN PIT	
Revenue, smelting & refining		**Resource COG**	**Incr. Resource COG**
U_3O_8 price	US$/lb U_3O_8	60.00	60.00
Exchange rate	C$/US$	1.05	1.05
U_3O_8price	C$/lb U_3O_8	63.16	63.16
Payable metal	% U_3O_8	100.00%	100.00%
TC/RC/Transport	C$/lb U_3O_8	0.00	0.00
Royalties @ 5% NSR	C$/lb U_3O_8	3.16	3.16
Net U_3O_8 price	C$/lb U_3O_8	60.00	60.00
Opex estimates			
Mining cost	C$/t mined	2.70	0.00
Strip Ratio	t:t	40.00	0.00
Mining Cost	C$/t milled	110.70	0.00
Processing Cost	C$/t milled	70.00	70.00
G&A/Sustaining capital cost	C$/t milled	5.00	5.00
TMF	C$/t milled	35.00	35.00
Site Cost	C$/tonne milled	220.70	110.00
Recovery and Dilution			
Recovered U_3O_8 grade	% U_3O_8	0.17	0.08
Process Recovery	average %	96%	96%
Plant feed U_3O_8 grade	diluted % U_3O_8	0.17	0.09
Dilution	%	10%	10%
Cut-off Grade			
In-situ cut-off U_3O_8grade	% U_3O_8	0.193	0.096

**These parameters were the initial assumptions made to begin the mine planning process. Some of the parameters changed as more detailed work was conducted. For example, the process recovery of U_3O_8 of 96% was used in the optimization and then modified to 95% for the economic analysis as the recovery was finalized by the QP. The processing costs also changed from this preliminary estimate, done at an assumed head grade of 0.16% U_3O_8, to the final costs estimated using the ROM grade of 0.30% U_3O_8.*

18.7.4 Optimization Parameters and Results

The geotechnical parameters as well as mining, milling, G&A and power costs summarized in Table 18.12 above along with the estimated projected topography as of late 2010 were used as the starting point for the pit optimization.

A series of Whittle™ pit shells were generated based on varying revenue factors. The results were analyzed with pit shells chosen as the basis for further design work and preliminary stage designs.

The resources within the various pit shells were generated from the following 3-D block model items:

- Block centroid coordinates;
- U_3O_8 grade;
- Class (indicated, inferred);
- Rock code;
- Topography percentage;
- Specific gravity.

The results of the Whittle™ pit optimization evaluation for varying revenue factors values (Whittle™ shell 6 is revenue factor 1.0) are summarized in Table 18.13 for indicated and inferred resources.

Table 18.13: Whittle Pit Optimization Results

Final Pit	Revenue Factor	Mine Life	Ore Diluted (tonnes)	Diluted Grade U_3O_8 (%)	Contained Metal U_3O_8 (Mlb)	Waste (tonnes)	Strip Ratio	Total (tonnes)	Total CF (C$)	NPV Best $ disc	NPV Worst $ disc
1	0.90	1.5	334,476	0.20	1.5	14,046,947	42.00	14,381,423	10,585,231	8,102,456	8,102,456
2	0.92	1.5	345,579	0.20	1.5	14,360,057	41.55	14,705,636	10,803,338	8,242,276	8,238,996
3	0.94	1.5	356,444	0.20	1.6	14,493,933	40.66	14,850,377	10,943,479	8,315,013	8,310,607
4	0.96	1.5	360,737	0.20	1.6	14,554,441	40.35	14,915,178	10,978,350	8,325,891	8,321,030
5	0.98	1.5	374,165	0.20	1.6	14,859,106	39.71	15,233,271	11,049,821	8,327,200	8,314,203
6	**1.00**	**1.5**	**381,156**	**0.20**	**1.6**	**15,025,721**	**39.42**	**15,406,877**	**11,050,366**	**8,296,109**	**8,276,800**
7	1.02	1.5	382,063	0.20	1.6	15,036,016	39.35	15,418,079	11,049,166	8,290,981	8,271,643
8	1.04	1.5	385,551	0.19	1.7	15,110,702	39.19	15,496,253	11,020,620	8,250,560	8,229,254
9	1.06	1.5	392,162	0.19	1.7	15,219,214	38.81	15,611,376	10,948,299	8,158,293	8,134,768
10	1.08	1.8	487,746	0.19	2.0	19,426,822	39.83	19,914,568	9,444,520	5,969,941	5,923,983
11	1.10	2.6	697,501	0.18	2.8	28,039,014	40.20	28,736,515	5,124,588	1,797,780	-2,668,330
12	1.12	2.6	698,008	0.18	2.8	28,049,687	40.19	28,747,696	5,116,007	1,791,020	-2,678,493
13	1.14	2.6	703,965	0.18	2.8	28,228,283	40.10	28,932,248	4,975,877	1,680,796	-2,848,105
14	1.16	2.7	708,756	0.18	2.8	28,448,368	40.14	29,157,124	4,806,897	1,548,336	-3,035,282
15	1.18	2.7	718,631	0.18	2.8	29,052,241	40.43	29,770,872	4,355,717	1,197,161	-3,450,638
16	1.20	2.7	723,390	0.18	2.9	29,382,161	40.62	30,105,552	4,102,758	1,001,904	-3,674,719
17	1.22	2.8	725,971	0.18	2.9	29,445,239	40.56	30,171,210	4,014,466	933,996	-3,757,930
18	1.24	2.8	726,998	0.18	2.9	29,519,470	40.60	30,246,467	3,953,835	887,434	-3,811,154

To better determine the optimum Whittle shell on which to base the pit phasing and scheduling, and to gain a better understanding of the deposit, the shells were analyzed in a preliminary schedule. The schedule assumed a maximum milling capacity of 3,000 tonnes/day. No stockpiles were used in the analysis and no capital costs were added. Both best case (mine out pit 1, the smallest pit, and then mine out each subsequent pit shell from the top down, before starting the next pit shell) and a worst case (mine each bench completely before starting next bench) scenarios were analyzed. The shells were each scheduled at varying revenue factors (0.9 through to 1.24 of base case) to produce a series of nested pit with the NPV results shown in Table 18.13 above.

Based on the analysis of the Whittle pit shells and preliminary schedule, Whittle pit shell 6 was chosen as the base case shell for further pit phasing and scheduling, which is revenue factor 1 shell and maximizes total cash flow. Table 18.14 below summarizes the tonnages and grades contained within the shell limits (using the incremental cut-off grade of 0.096% U_3O_8, and a dilution factor of 10%).

Table 18.14: Open Pit Resources Extracted in LOM Plan by Classification

Deposit	Resource Category	Tonnes (Mt)	Cut-off Grade (U_3O_8%)	Diluted Grade (U_3O_8%)	Contained Metal U_3O_8 (Mlb)
Raven	Indicated	0.39	0.096	0.19	1.65
	Inferred	0.00	0.096	0.24	0.01
	Sub-total	**0.39**	**0.096**	**0.19**	**1.66**

A typical section (looking west) is shown in Figure 18.10 with existing ground, chosen Whittle shell, and block model outlines shown.

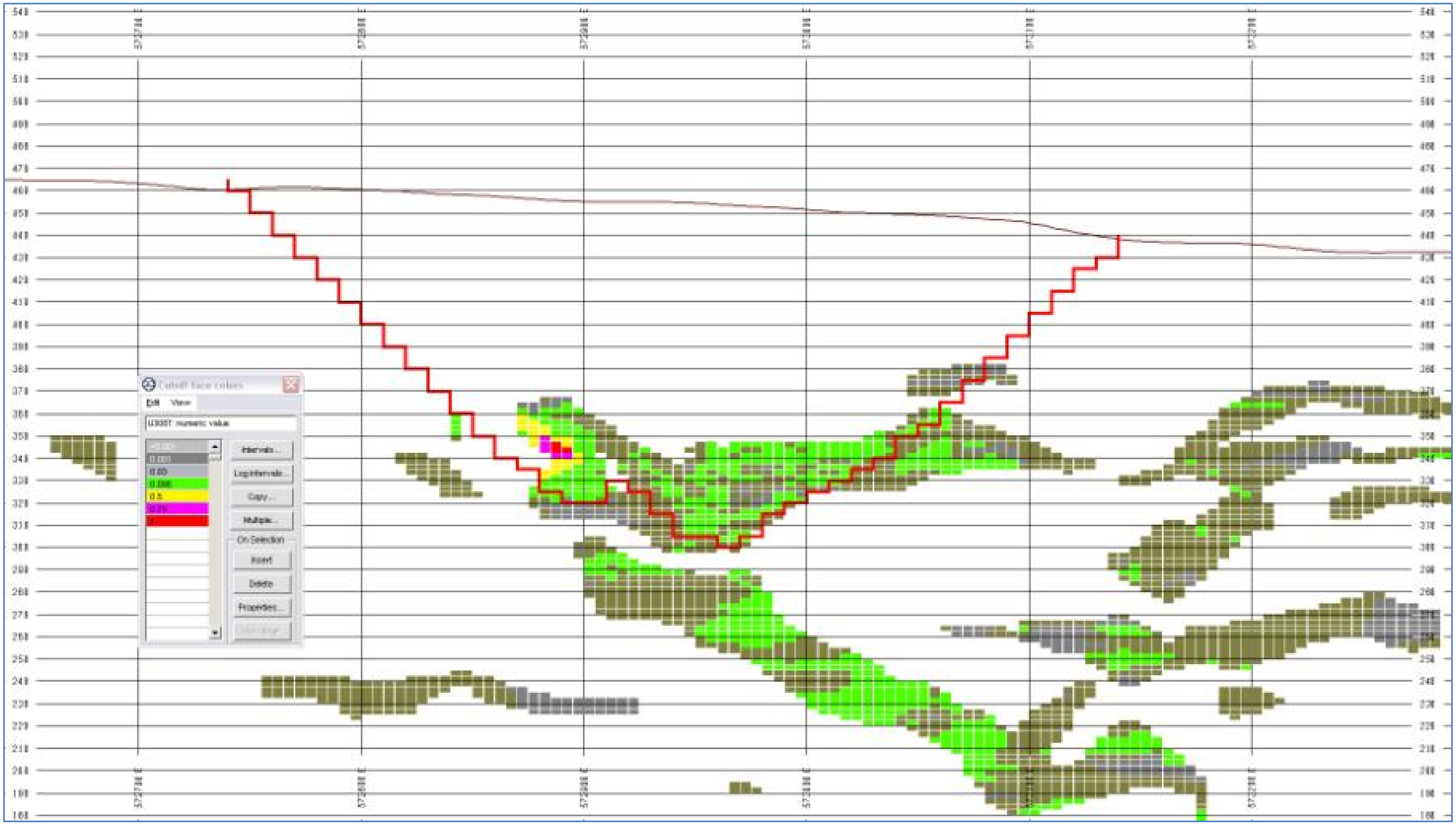


Figure 18.10: Typical Section (Looking North) Showing and Pit Shell

18.7.5 Mine Design

Mine planning for the OP pit Raven deposit was conducted using a combination of Mintec Inc., MineSight™ software and Whittle™ software. The base 3-D resource block model (as provided by Golder), along with subsequent economic and technical input parameters were imported into Whittle™. The production scheduling was undertaken with the use of MineSight™ software.

A preliminary pit design was then based on the Whittle shell analysis (Section 18.7.4). Preliminary waste dumps were then designed to account for the material produced from the OP mining of the Raven deposit.

Whittle pit shell 6 was chosen as the base case pit design for the Raven deposit. Figure 18.11 below illustrates the proposed Raven pit.

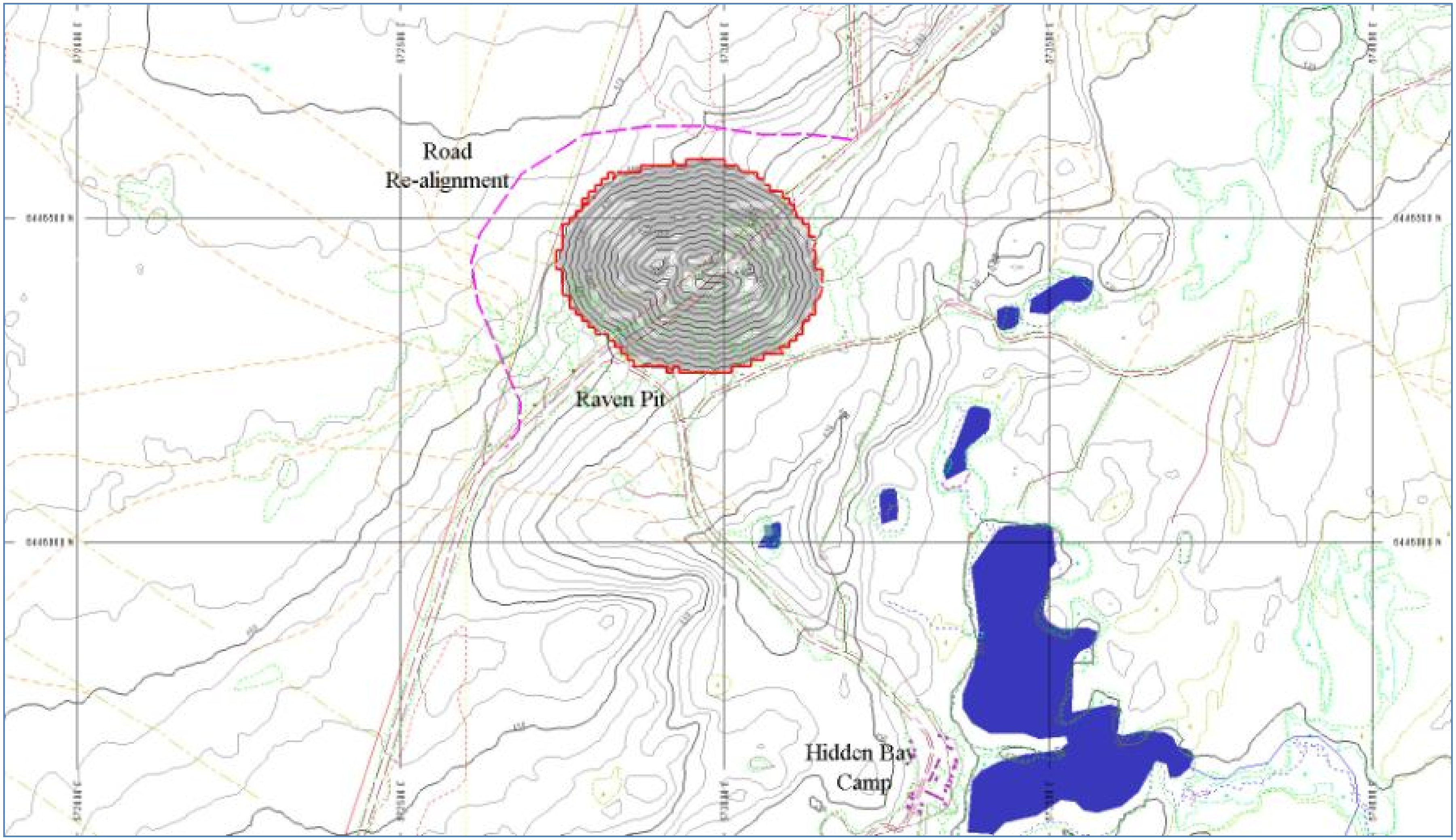


Figure 18.11: Raven Pit Plan View

Mine Operation

Given the short mine life, the open pit mining activities for the Raven pit were assumed to be undertaken by a mining contractor as the basis for this preliminary assessment. The unit rate used in the Whittle™ optimization was $2.70 per tonne of material mined for pit and dump operations, road maintenance, mine supervision and technical services. The mining unit rate was calculated based on equipment required to achieve a maximum mining rate of 30 ktpd. Mining costs were developed from first principles for similar sized operations, along with local labour, fuel and power costs, along with an estimate for contractor profit margin.

Equipment

The major mining equipment requirements are indicated in Table 18.15 and are based on similar sized open pit operations. The proposed plant processing rate of 1,000 tpd along with a maximum mining rate of 30 ktpd was used to estimate the mining equipment fleet needed. The fleet has an estimated maximum capacity of 30,000 tpd total material, which will be sufficient for the life-of-mine plan.

Table 18.15: Mining Equipment

Equipment Type	No. of units
D9R-class Dozer	2
Diesel, 6.5-cu-m Front Shovel	2
6.5-cu-m Wheel Loader	1
55-t Haul Truck	8
14H-class Grader	2
165mm dia. Rotary, Crawler Drill	4
115mm dia. Hydraulic Track Drill	1
Water Truck, Lube Truck, Low-bed	1
Service Trucks, Pickups, Crew Van	1

Unit Operations

The 165mm diameter drill performs the majority of the production drilling in the mine. The hydraulic drill with a 115mm diameter bit is to be used for secondary blasting requirements and may be used on the tighter spaced patterns required for pit development blasts. The main loading and haulage fleet consists of 55 t haul trucks, which are loaded primarily with the diesel 6.5 m^3 front shovel or the 6.5 m^3 wheel loader, depending on pit conditions. As pit conditions dictate, the track D9 dozers are used to rip and push material to the excavators, as well as maintaining the waste dumps.

The additional equipment listed in Table 18.16 will be used to maintain and build access roads, and to meet various site facility requirements.

The work schedule is based on two twelve hour shifts, seven days a week, 365 days per year.

18.8 Underground Development Schedule

18.8.1 Mine Access Development

The mine development is divided into two periods: pre-production development (prior to mine production) and ongoing development (during production).

The objective of the pre-production development was to provide access to higher-grade areas and prepare enough resources to support the mine production rate during the period of time in which access to the lower levels would be established.

Pre-production development was scheduled to:

- Provide access for trackless equipment;
- Provide ventilation and emergency egress;
- Establish material handling systems;
- Install mining services (power distribution, communications, explosives storage, water supply, mine dewatering);
- Provide optimum sub-level development in advance of start-up to develop sufficient mineral resources to support the mine production rate.

The initial mine development was planned to the 240 m elevation, that would provide access to multiple mining blocks containing resources to support production for about one year. Having multiple stopes would provide an opportunity to have enough production faces to support the mine production rate and flexibility on the sequencing of production and backfill. The main decline would continue to be developed to the bottom of the mine to the 20 m elevation.

It was assumed that all underground pre-production development would be performed by a contractor. Approximately one month would be required for mobilization of mining equipment and crews to the site and establish the required services. The jumbo crew would develop a portal and start developing decline.

The development schedule was developed based on estimated cycle times for jumbo development and best practises of North American contractors. Development cycle times are shown in the table below.

Table 18.16: Development Cycle Times

Item	Unit	Ramp (4.6 m X 4.6 m)
Design Criteria		
Width	m	4.6
Height	m	4.6
Gradient	%	15
Summary Cycle Times		
Drilling	Hrs	4.1
Blasting	Hrs	1.8
Re-Entry	Hrs	0.5
Mucking	Hrs	2.4
Support	Hrs	4.3
Services	Hrs	1.1
Secondary Mucking	Hrs	4.3
Trucking	Hrs	4.3
Single Heading		
Critical Path Cycle Time	Hrs	14.0
Advance Per Shift	m	2.5
Advance Per Day	m	5.0

It was assumed that the advance rate of the main decline development would be approximately 150 m/month per single heading. In the first month when the decline development starts from the portal, the advance rate would be at 50% of average or 75 m/month. When multiple headings are available, the advance rate per jumbo crew would increase to 220 m to 250 m per month.

The second jumbo crew would start working when first crew would advance decline development to 1,100 m and multiple headings would be available for both crews.

The ventilation raise providing access to surface was planned to be done by a raiseboring crew. It was assumed that a raiseboring crew would drill approximately 500 m per month of pilot holes, and ream to the 4.0 m diameter at an advance rate of 100 m per month. The ventilation raise between sub-levels would be developed by drop raising techniques.

It was estimated that all pre-production development would be completed in twelve months. All waste development in the pre-production period was assumed as capital development. During mine production, the decline development, ventilation drifts and raises were considered as capital development, but crosscuts were included in operating costs. All capital development would be completed by the end of year 3.

Table 18.17: Pre-production and Capital Development Schedule

				Pre-production Quarters				Production Years		
Summary	**Length (m)**	**Advance (m/d)**	**Duration (days)**	**Q1**	**Q2**	**Q3**	**Q4**	**1**	**2**	**3**
Crew Mobilization			15	1						
Setup Services			15	1						
Decline	3,470	5.0		225	450	470	420	750	750	405
Crosscuts	280	5.0				60	220			
Ventilation Drifts	1,090	5.0				30	380	320	280	80
Remuck Bay	330	5.0		15	45	45	45	75	75	30
Total Advance				240	495	605	1,065	1,145	1,105	515
Pilot Hole						120				
Ream							120			
Drop Raise							75	120	110	70
Total Raise							195	120	110	70

The main access ramp would continue to be developed to the bottom of the mine. The mine access development would be ahead of production for not less than six months to insure stope scheduling.

18.9 Underground Production Schedule

18.9.1 Mine Production Rate

The optimum mine production rate was theoretically estimated by applying Taylor's formula, as shown below:

$$\text{Optimum Production Rate} = \frac{5 \text{ x } (\text{Expected Reserves})^{3/4}}{(\text{Production Days per year})}$$

Taylor's formula was used for preliminary estimation of mine production rate assuming that mine will operate 350 days a year.

Table 18.18: Mine Production Rate

Item	Tonnes / Day	Tonnes / Year	Years
Expected Mineable Resources 2,100,000 t			
Taylor's Formula	800	280,000	7.5
Proposed Production Rate	1,000	350,000	6.0

The underground mine production rate of 1,000 t/d was considered appropriate due to the high degree of mechanization, potential high productivities of selected stoping methods. Based on the availability of several deposits and ability to have production from different mining blocks, SRK considers the 1,000t/d to be achievable.

18.9.2 Production Schedule

The criteria used for scheduling of underground mine production at the Horseshoe deposit were as follows:

- Target mining blocks with higher grade mineralization in the early stages of mine life to improve project economics;
- Production sequence of the mining blocks would be from the top down;
- An average annual production rate of 350,000 t was scheduled, including mineralized material from development and stopes;
- The mine will operate two 10-h shifts per day, 350 d/a;
- Provide enough production faces to support a daily mine production rate of 1,000 t/d.

The cycle times and productivities were estimated from first principles for average production stope size. The tonnage in an average production stope size of 5 mW x 4.5 mH with advance of 3.85 m per cycle would yield about 225 tonnes per round, therefore, 1,000 tpd of production would require approximately 4.5 full cycles per day. Productivity of one jumbo crew could be up to 1,000 t/day. Assuming that multiple production faces would be available all the time, two jumbo crews would fully satisfy the production and development schedule.

The mine production by year is shown in Table 18.19.

Table 18.19: Underground Production Schedule

Parameter	Unit	Year						Total
		1	2	3	4	5	6	
Tonnes	Kt	350	350	350	350	350	347	2,097
Grade	U_3O_8%	0.54	0.39	0.30	0.23	0.23	0.24	0.32
Contained U_3O_8	U_3O_8 Klb	4,173	3,033	2,335	1,788	1,768	1,808	14,906

Phase 1 and Phase 2 of production period are shown on the figures below.

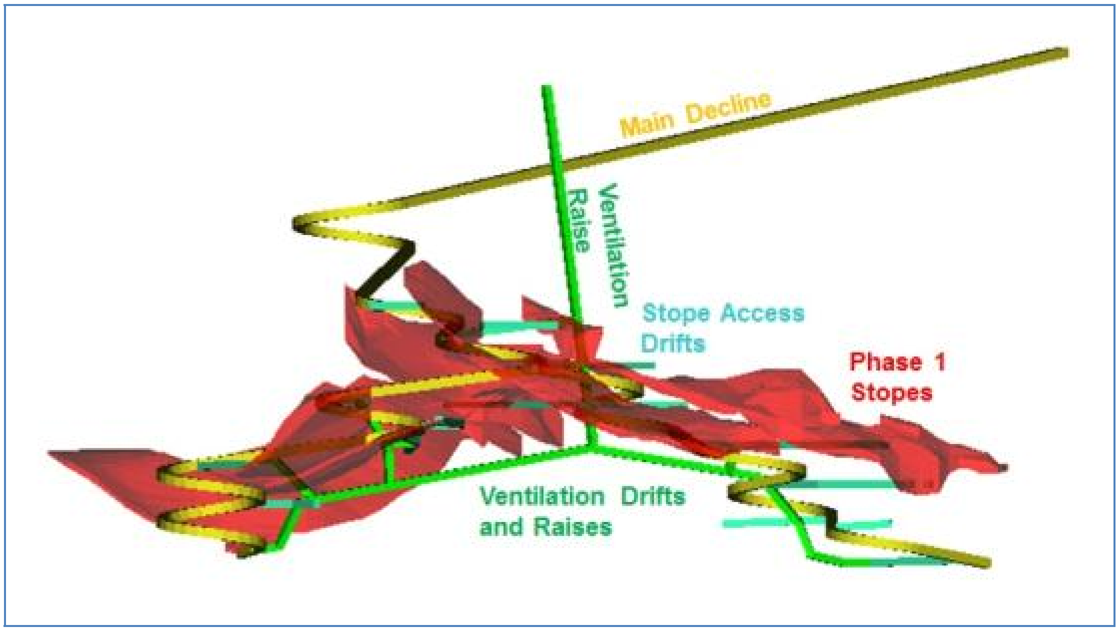


Figure 18.12: Mine Production. Phase 1

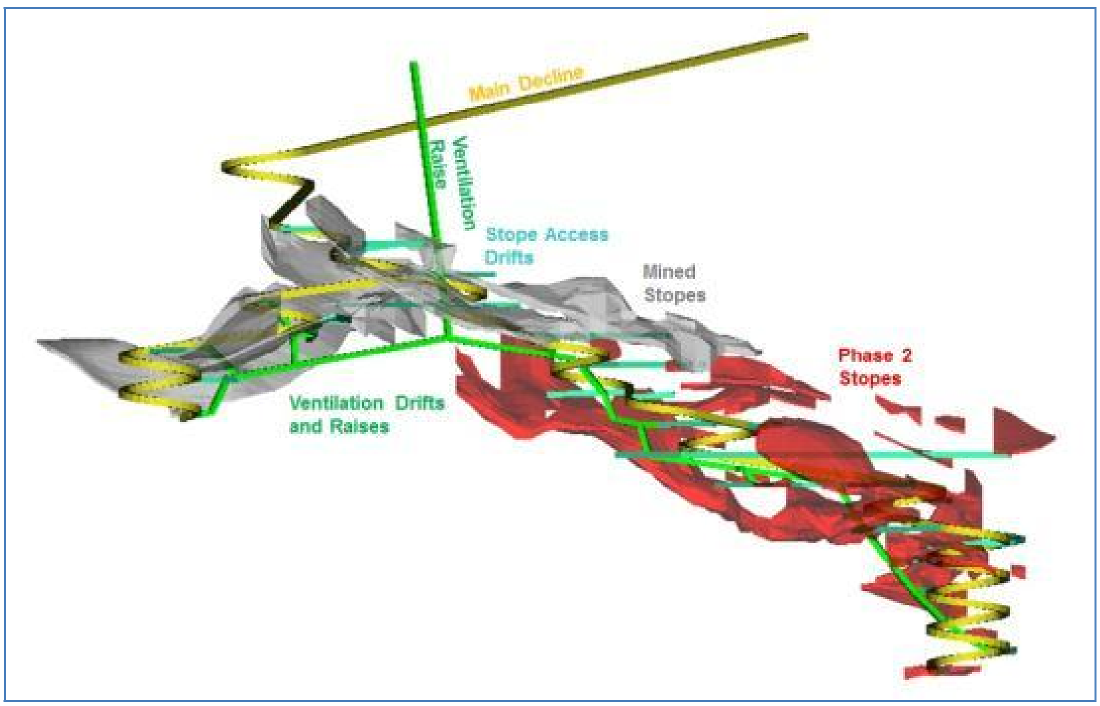


Figure 18.13: Mine Production. Phase 2

18.10 Production Schedule

Mine Sequence/Staging – Open Pit and Underground

The current life-of-mine ("LOM") plan focuses on accessing and milling higher grade material first. As such, the plan commences with UG mining of Horseshoe, followed by the OP at Raven. Given the size of the Raven pit, no pit staging has been considered, rather, the pit is mined as one stage. The latter years of underground mill feed from Horseshoe will be supplemented with the OP feed from the Raven pit in order to attain the scheduled mill throughput.

The overall site plan configuration is illustrated in Figure 18.14 below, that includes the proposed waste dump location.

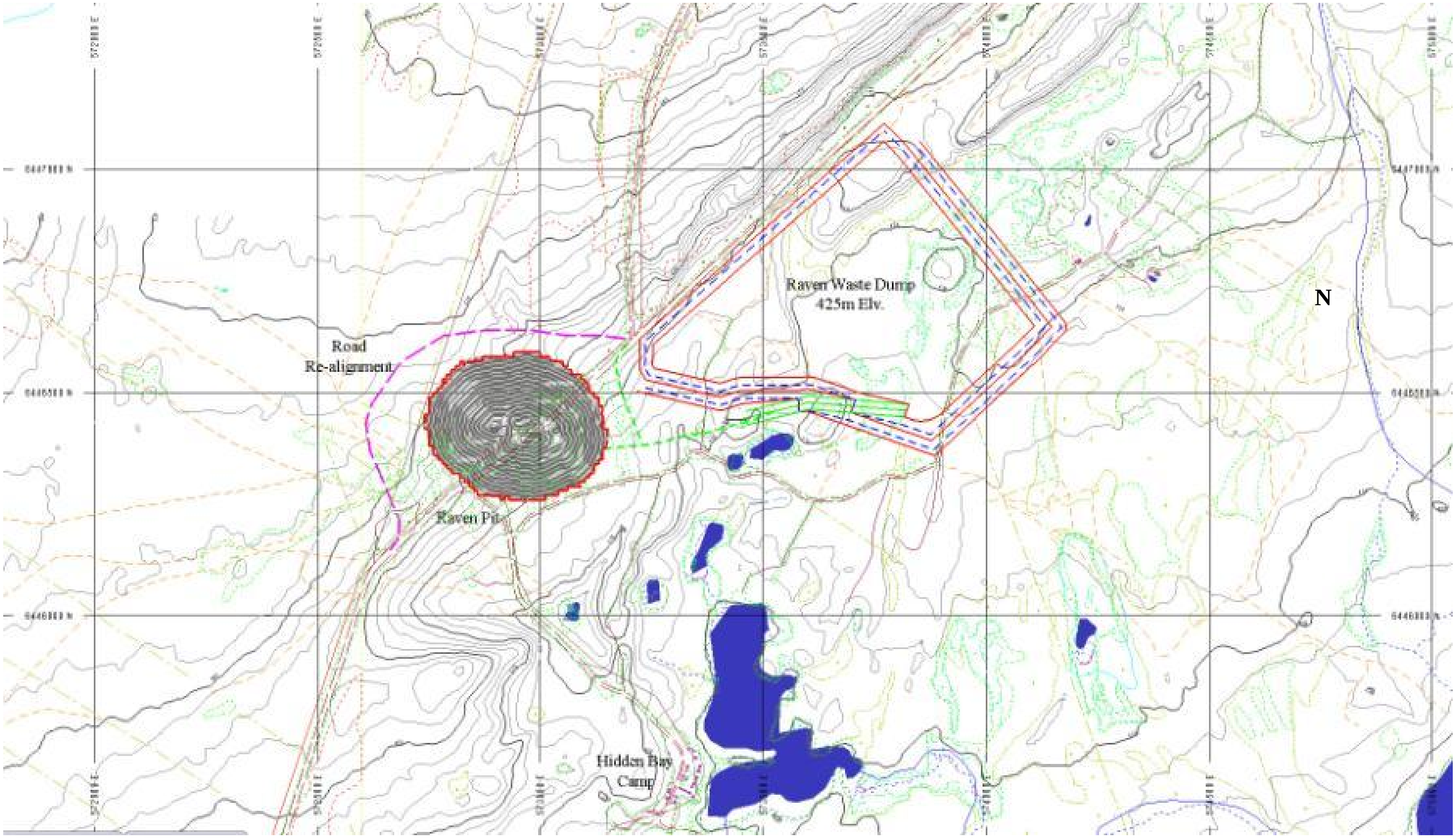


Figure 18.14: Hidden Bay Overall Site Plan Configuration

The tonnages and associated grades and metal recoveries of the Hidden Bay deposits are summarized in Table 18.20.

Table 18.20: OP and UG Tonnages and Grades

Deposit	Resource Category	Tonnes (Mt)	Cut-off Grade (U_3O_8%)	Diluted Grade (U_3O_8%)	Contained Metal U_3O_8 (Mlb)
Raven	Indicated	0.4	0.10	0.19	1.7
	Inferred	0.0	0.10	0.24	0.0
Horseshoe	Indicated	2.0	0.16	0.32	14.4
	Inferred	0.1	0.16	0.28	0.5
Total	**Indicated**	**2.4**	**0.15**	**0.30**	**16.1**
	Inferred	**0.1**	**0.16**	**0.28**	**0.5**

Mine Production Schedule – Open Pit and Underground

The production schedule for Hidden Bay includes both the open pit deposit at Raven and the underground deposit at Horseshoe.

Plant feed is planned at 1,000 t/d for a net yearly production of 0.35 Mt of ore. In the latter production years, the mill feed from the Horseshoe underground mine will be supplemented with the mill feed from the Raven pit in order to maintain the scheduled mill throughput.

The maximum planned total material movement from the Raven pit is approximately 30,000 t/d. The average total open pit mining rate is planned to be 21,000 t/d. The maximum underground production rate is 1,000 t/d of mill feed. Only indicated and inferred resources were used in the LOM plan as there were no measured resources in the resource model, however, inferred resources represent only 3% of the material mined and processed.

Table 18.21 is a summary of total material movement by year for the mine production schedule.

Table 18.21: Production Schedule – Hidden Bay Deposits

			Year						
Parameter	**Unit**	**Total**	**1**	**2**	**3**	**4**	**5**	**6**	**7**
OPEN PIT MINING - Raven									
O/P total Waste	Mt	15.01						11.54	3.48
O/P ROM ore	Mt	0.39						0.00	0.39
U_3O_8 Grade ore	U_3O_8 %	0.19						0.26	0.19
Total ore mined O/P	Mt	0.39						0.00	0.39
Total Mined lbs O/P	Mlb U_3O_8	1.7						0.0	1.6
Strip Ratio	t:t	38.2						3958.9	8.9
UNDERGROUND MINING - Horseshoe									
Horseshoe total waste	Mt	0.00							
Horseshoe ROM ore	Mt	2.10	0.350	0.35	0.35	0.35	0.35	0.35	
U_3O_8 Grade ore	U_3O_8 %	0.32	0.54	0.39	0.30	0.23	0.23	0.24	
Total Mined lbs	Mlb U_3O_8	14.9	4.2	3.0	2.3	1.8	1.8	1.8	
TOTAL ALL DEPOSITS									
Total Waste	Mt	15.01						11.54	3.48
Total ore mined	Mt	2.49	0.35	0.35	0.35	0.35	0.35	0.35	0.39
Total Mined grade	U_3O_8 %	0.30	0.54	0.39	0.30	0.23	0.23	0.24	0.19
Total Mined lbs	Mlb U_3O_8	16.6	4.17	3.0	2.3	1.8	1.8	1.8	1.6

The Hidden Bay Raven open pit and Horseshoe underground will produce a total of 2.49 Mt of mill feed and 15.0 Mt of waste rock over a 7-year mine operating life. The mine schedule focuses on achieving the required plant feed production rate and mining of higher grade material early in schedule. All mill feed is assumed to be hauled to the Rabbit Lake Processing Facility for processing. Figure 18.15 summarizes mined ore production schedule by period and area.

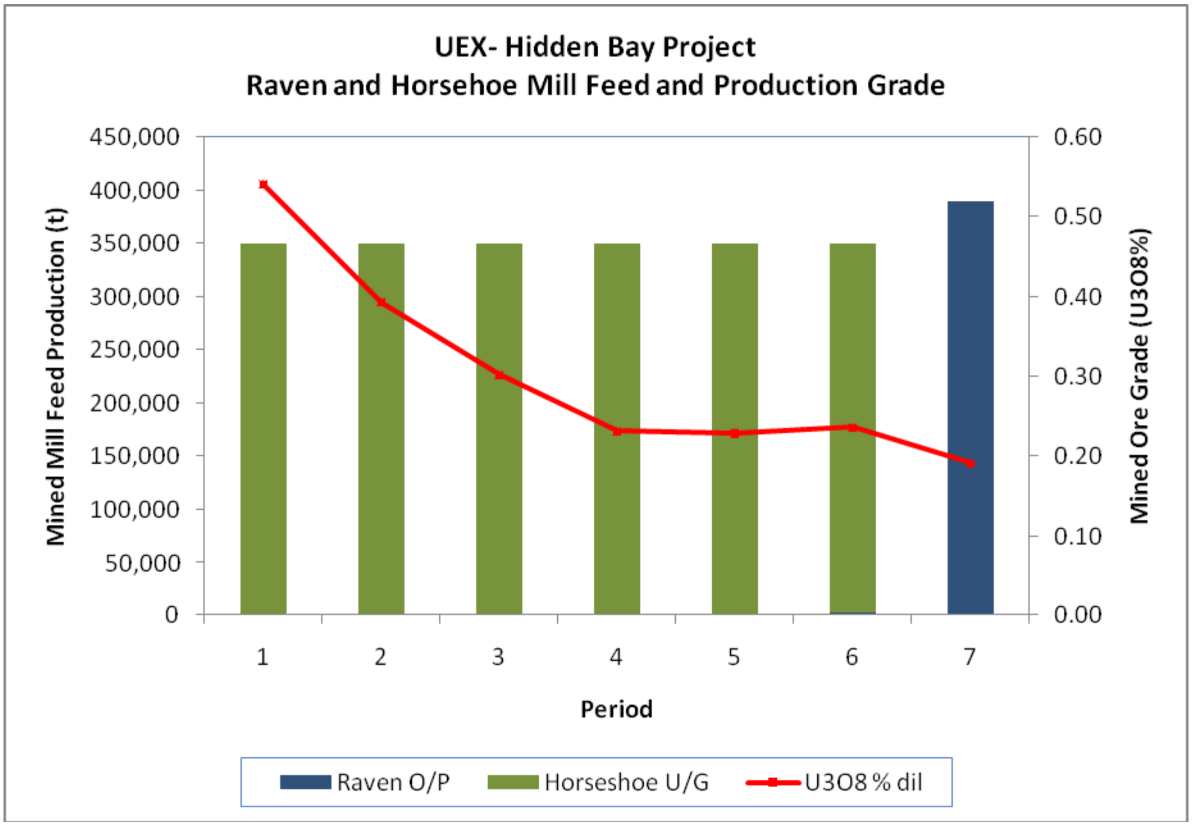


Figure 18.15: Period Tonnages and U_3O_8Grade

Open Pit/Underground Development

Year 1-5: Development of the Hidden Bay deposits commences with mining of the Horseshoe deposit via underground methods. Maximum underground production is targeted at 1,000 tpd. All mill feed is scheduled to be hauled to the Rabbit Lake Facility for processing. U_3O_8 grades range from 0.54% to 0.23%.

Year 6: Underground production ends at Horseshoe. The Raven open pit production commences (primarily pre-stripping). Overall mill feed grade is 0.24%. Total open pit production of 11.5 Mt of material.

Year 7: Open pit mining of Raven continues and is completed by year end. Mined grade of 0.19% U_3O_8. Total mill feed of 1,000 tpd maintained with a strip ratio of 8.9:1.

Waste Management Facilities

Waste Rock Dumps

The waste rock dump is to be located adjacent to the final pit limits to the north west of the proposed Raven pit (see Figure 18.14 above).

The waste rock dump will be built in a series of lifts in a "bottom-up" approach. The dump will be constructed by placing material at its natural angle of repose (approximately 1.5H:1V) with safety berms spaced at regular intervals.

A total of 15.0 Mt of waste rock is planned to be mined from the Raven open pit.

Table 18.22 below summarizes the waste rock by U_3O_8% grade bin. Material with a U_3O_8 content of greater than 0.03% and less than the cut-off grade of 0.096% has been classified as "special" waste. For the Raven pit this amounts to 0.29 Mt, however capital costs for a waste dump liner have been assumed for 25% of the waste generated. Chemical waste characterization has not been conducted on the Hidden Bay deposits. Once this characterization is completed (future studies) the waste rock material will likely be segregated into two types; benign waste and; special waste/PAG rock. The special waste/PAG rock dump will require a liner in order to capture any run-off.

Table 18.22: Raven Pit Waste Rock Classification

Pit	Total Rock (Mt)	Total U_3O_8 %	Grade Bin 0.03-0.096% U_3O_8			Grade Bin 0.00-0.03% U_3O_8			Grade Bin 0% U_3O_8		
			Waste (Mt)	U_3O_8 (%)	U_3O_8 (Mlb)	Waste (Mt)	U_3O_8 %	U_3O_8 (Mlb)	Waste (Mt)	U_3O_8 %	U_3O_8 (Mlb)
Raven	15.0	0.001	0.29	0.057	0.37	0.07	0.022	0.03	14.7	0.000	0.00

All mill feed is to be trucked to the Rabbit Lake Facility, as such, no tailings management facility has been included in this PA. However, the cost of tailings disposal has been captured in the economic parameters.

19 Market, Contracts and Taxes

19.1 Market

This study assumes that U_3O_8 produced from the Hidden Bay deposits will be treated along with mill feed from other Cameco operations and retained and marketed by Cameco as part of their combined plant Rabbit Lake plant production.

19.2 Contracts

SRK is not aware of any significant existing contracts that might affect the project. UEX has recently commenced preliminary discussions with Cameco on the potential toll treatment and tailings deposition of the Hidden Bay deposits. Cameco has shown general interest in the project and SRK sees no reason why further discussions between Cameco and UEX will not continue to their mutual benefit.

19.3 Taxes

The economic analysis performed of this study does not include taxes. This section is included for reference and is extracted from CostMine (InfoMine USA, Inc).

19.3.1 Federal Taxes

Income Tax

Effective January 1, 2008 the tax rate was set at 19.5%. Income earned outside Canada is taxed at the full 33%. The general tax rate is to be simultaneously reduced such that it will be 15% in 2011.

In general, the tax regulations shown here apply equally to domestic or foreign firms operating in Canada. Alternatively, a foreign firm can establish a Canadian corporation to conduct business in Canada. In general terms, federal taxable income for a mining company is defined as mining revenue less the following deductions:

- Operating costs;
- Capital cost allowance (CCA);
- Resource allowance;
- Canadian exploration expense (CEE);
- Cumulative Canadian development expense (CCDE) ;
- Interest expense ; and
- Crown royalties and provincial mining taxes paid.

19.3.2 Provincial Taxes

Corporate Income Tax

Saskatchewan's corporate tax is levied as a percentage of the share of a corporation's federally defined taxable income that is allocated to the Province. Saskatchewan's general tax rate on corporate taxable income is 12%.

Capital Tax

Saskatchewan resource producers are subject to a capital tax surcharge.

Capital Tax Surcharge

Resource producers are subject to a capital tax surcharge equal to the difference between 3.6% of the corporation's value of Saskatchewan resource sales and the existing capital tax liability. This tax applies only to natural gas, oil, potash, coal, and uranium producers. If sales are less than $100 million the producer can deduct $2.5 million from sales before determining the capital surcharge due.

The capital tax charged to qualifying potash, uranium, coal, oil and natural gas producers was reduced from 3.6% of value of Saskatchewan resource sales to 3.0% on July 1, 2008.

If sales in a given year are less than $100 million the producer can deduct $2.5 million from sales before determining the capital surcharge due.

Uranium Royalty Structure

In 2001, Saskatchewan introduced a new uranium royalty structure governing all uranium production in the province. The structure is divided into two separate royalties, a Basic Royalty and a Tiered Royalty, that are computed separately. The two sums calculated are added to determine the total royalties due. The royalties are calculated for all operations of each firm rather than on a project by project basis. Following are the main features of the royalty structure.

Basic Royalty

The basic uranium royalty rate is equal to 5% of the value of gross uranium sales less transportation costs from mine to point of sale. The Saskatchewan Resource Credit, which is one percent of gross sales, is credited against the Basic Royalty thus the effective rate is 4%.

Tiered Royalty

The Tiered Royalty is based upon unit gross sales values, with progressively higher royalty rates for increasing, realized uranium prices per pound. There is a minimum below which no royalty applies. The royalty is determined for sales falling within each of the increments defined. The rates are 6, 10, and 15%. When combined with the Basic Royalty, the total rate is 4, 10, 14, and 19% of gross revenues for sales in the price bracket of each tier (see Table 19.1). These prices are indexed for inflation from 1998. The index for 2005 was 1.15261.

Table 19.1: Saskatchewan Uranium Royalty Structure

Price Bracket	U_3O_8 Price Range ($/kg)	Tax Rate (%)	Total Tax Rate[1] (%)
Tier One	30-45	6	10
Tier Two	45-60	10	14
Tier Three	>60	15	19

[1] The total tax includes both Basic and Tiered royalties

A cumulative capital recovery bank is established for each producer in which a prescribed investment allowance is calculated in $/kilogram capacity. The bank is used to reduce the amount of revenues subject to the Tiered Royalty. The prescribed amount for new mills is $80/kg, mill expansions $50/kg, surface mines $45/kg, and underground mines $60/kg U_3O_8 capacity. These amounts are indexed for inflation.

No more than 50% of the capital recovery bank total may be used to reduce revenues in a given year. After deduction of any capital recovery bank amounts, small producers, producing less than 2 million pounds per annum, are allowed a credit of $750,000, indexed for inflation from 1998. This additional deduction compensates for the increase in royalties under the new structure. Producers who had not applied for this credit prior to March 31, 2002 are not eligible for this additional deduction.

The royalty structure is undergoing review at the time of writing this report and it is unknown what the future structure will be, although it is expected that the structure will improve for lower grade, new projects like Hidden Bay.

Provincial Sales Tax

The PST in Saskatchewan is 5%. The tax basis does not include the federal GST of 8% and is applied to the original purchase price.

Property Tax

In Saskatchewan, property values are updated every four years. The Saskatchewan Assessment Management Agency (SAMA) conducts a full revaluation of all properties in the province to co-ordinate with a new base date. The 2009 property assessment system will move to a more flexible, results-focused market value assessment system which is essentially the same in all other Canadian provinces.

Three generally accepted appraisal techniques used to value property in a market value assessment system include the cost approach, the sales comparison approach and the property income (rental) approach. Of these, the only method currently allowed in Saskatchewan for valuing commercial properties is the cost approach which will continue to be used in smaller municipalities and for specific property types.

20 Environmental and Social Considerations

The environmental assessment and permitting framework for uranium mining in Canada is well established. In Saskatchewan the process consists of a two tiered system, whereby the proposed project undergoes an environmental assessment (EA) phase involving departments from both the federal and provincial governments. Following a successful environmental assessment the operation undergoes a construction and operating licensing/permitting phase. The project is then regulated through all phases (construction, operation, closure and post closure) by both federal and provincial departments and agencies.

20.1 Environmental Assessment

20.1.1 Provincial Requirements

In the Province of Saskatchewan the Environmental Assessment Act is administered by the Ministry of Environment (MOE). The level of assessment for mining projects is dependent on the specific characteristics of each individual project. The Ministry follows the following process to determine which level of assessment will be required.

In Saskatchewan, the proponent of a project that is considered to be a “development” pursuant to Section 2(d) of the provincial Environmental Assessment Act is required to conduct an environmental impact assessment (EIA) of the proposed project and prepare and submit an environmental impact statement (EIS) to the Minister of Environment.

Section 2(d) of the Environmental Assessment Act reads:

“development” means any project, operation or activity or any alteration or expansion of any project, operation or activity which is likely to:

Have an effect on any unique, rare or endangered feature of the environment;

Substantially utilize any provincial resource and in so doing pre-empt the use, or potential use, of that resource for any other purpose;

Cause the emission of any pollutants or create by-products, residual or waste products which require handling and disposal in a manner that is not regulated by any other Act or regulation;

Cause widespread public concern because of potential environmental changes;

Involve a new technology that is concerned with resource utilization and that may induce significant environmental change; or

Have a significant impact on the environment or necessitate a further development which is likely to have a significant impact on the environment;

The Hidden Bay Project as it is currently defined meets the Provinces definition of a “development” and will therefore be required to conduct a Provincial EIA.

20.1.2 Federal Requirements

A federal environmental assessment is required for a proposed project if it meets the definition of a “project” as defined in the Canadian Environmental Assessment Act (CEAA) and a federal authority has certain decision making responsibilities. The CEAA is triggered when a federal authority:

- Is the proponent
- Provides financial assistance to a proponent in order to advance the project
- Has an ownership position in the land or grants an interest in the land allowing the project to proceed
- Has a regulatory responsibility for a component of the project (CEAA, 2007)

Every uranium development in Canada by law, requires regulatory decisions by the Canadian Nuclear Safety Commission (CNSC), a federal agency mandated to control the health, safety and environmental consequences of nuclear activity in Canada, and therefore triggers CEAA and a federal assessment in all cases.

Once a federal assessment is triggered the CEAA then determines what level of an EA the project will require. The CEAA defines four possible levels of assessment, Screening Level, Comprehensive Study, assessment by Mediator or a Review Panel. It is anticipated, based on the current scope of the Hidden Bay Project, that a federal Comprehensive Study assessment would be required.

In addition to the federally legislated requirements defining the need for an environmental assessment the federal government introduced the Major Projects Management Office (MPMO) in 2007. The MPMO role is to provide a management and coordinating role for major resource development projects in Canada. The authority and mandate of the office is provided through a Committee comprised of Deputy Ministers from federal departments typically identified as “responsible authorities” in the conduct of a federal environmental assessment. The MPMO has no legislative authority.

Saskatchewan and Canada honour a cooperation agreement which harmonizes the two assessment processes to run concurrently under a single administrative process. This process is typically administered jointly by Saskatchewan’s Assessment Branch and the CEAA regional office located in Winnipeg, Manitoba.

20.2 Licensing and Permitting

In the event environmental assessment approvals by both the provincial and federal governments are granted, the project will be allowed to proceed to the second tier of environmental approvals. This requires the proponent to obtain a variety of approvals and permits again from both levels of Government.

The federal licensing process is much more onerous than the provincial process and requires the submission of detailed engineering design packages as well as detailed Management Plans for all facets of the operation as part of their licensing process.

20.3 Assessment Schedule and Estimated Costs

Based on the scope of the project it is expected that it will be required to undergo a Comprehensive Study Assessment in conjunction with a Provincial assessment. Using previous assessments of similar projects for comparison, it is estimated that the environmental assessment of the Hidden Bay project will require approximately 48 months from submission of the Project Description to receipt of environmental assessment approvals to proceed with the project as shown in Figure 20.1. It must be noted that this timeline represents the best case timing. Actual timing may vary considerably.

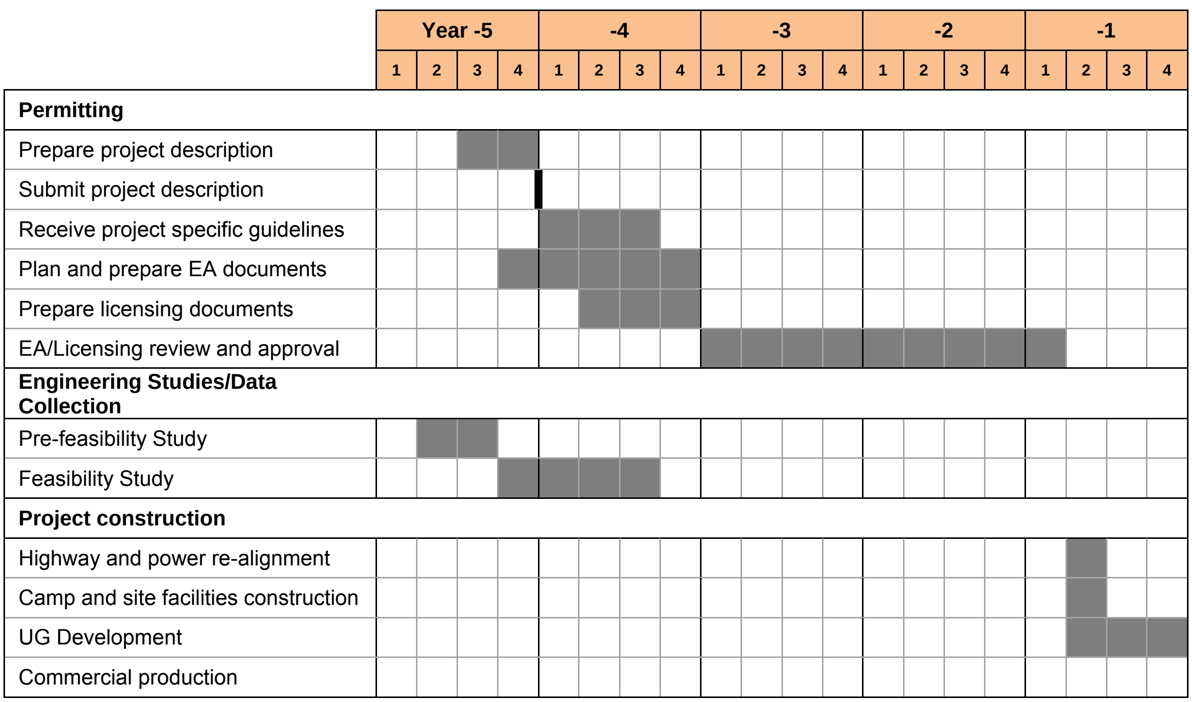


Figure 20.1: Conceptual EA and Licensing Schedule

Costs associated with completing an Environmental Assessment are a function of the complexity of the project, the level of Assessment the project must undergo and the commodity involved. When uranium is the commodity the Canadian Nuclear Safety Commission (CNSC) is the lead federal agency involved in the Assessment. This Agency operates on a cost recovery basis which allows the agency to bill the proponent for each hour their staff dedicates to the Assessment process which complicates the ability to accurately estimate the total costs of an assessment.

A reasonable estimate of the total costs associated with completing an Environmental Assessment for this project is approximately $4 million.

20.4 Environmental Considerations

The main environmental considerations associated with this project are centered around the management of the various waste streams associated with the project. The dominant and/or potentially more problematic of these waste streams are tailings, waste rock and mine water.

20.4.1 Tailings

The current scope of the project involves the milling of all ore at one of two nearby uranium mills, Cameco's Rabbit Lake processing facility or AREVA's McClean Lake processing facility. Under a toll milling arrangement all tailings would be managed off site at tailings management facilities currently approved to receive uranium tailings. The proponent would however be obligated to demonstrate that the geochemical characteristics of the tailings generated from processing the Hidden Bay ores would not produce contaminant source terms with concentrations higher than those concentrations the chosen toll mill was designed to contain.

20.4.2 Waste Rock

The development of this project has the potential to generate three types of waste rock, clean waste rock, potentially acid generating and/or metal leaching (PAG/ML) waste rock and/or special waste rock. Previous studies of the deposits have assumed that all waste rock generated would be classified as PAG and therefore would be managed to address any potential environmental problems associated with the generation of acid and/or metal leaching.

Following a waste rock geochemical characterization program, to be completed at the next level of engineering, a waste rock management plan will be developed. This plan will segregate the waste rock into two categories, clean waste rock and PAG/ML and/or special waste. For the purposes of this study special waste has been defined as waste containing 0.03 % U^3O_8 to 0.10 % U_3O_8 as shown in Table 18.22. The next phase of engineering will require the design of a lined pad to contain the special waste and PAG/ML. All seepage and runoff collected on this pad will require treatment prior to discharge to the environment.

The remaining clean waste rock will be stockpiled on an unlined pad.

20.4.3 Mine Water

All mine water collected in both the underground and open pit mines will likely require treatment to remove radionuclides and other potential contaminants of concern prior to being able to discharge this water to the environment.

Careful consideration to the discharge location of the treated effluent is recommended. Discharge to larger water bodies is typically preferred by the regulatory and scientific communities while recent public consultations with residents of northern Saskatchewan suggest their preference would be to discharge treated effluents into smaller water bodies which are not as heavily utilized for traditional, commercial and recreational pursuits. In addition any effluent reporting to Hidden Bay of Wollaston Lake would initiate concerns of a cumulative nature. Cameco’s Rabbit Lake Operation currently discharges treated effluent into Hidden Bay.

Based on the existing data there were no environmental fatal flaws identified with this proposed project.

20.5 Social Considerations

Significant efforts have been expended by the Saskatchewan government and the uranium mining industry since the early 1990s to solicit and incorporate Traditional Knowledge, concerns and desires of northern Saskatchewan Residents (both aboriginal and non-aboriginal) into the environmental assessment process. There are a number of well established forums and committees in existence, in Saskatchewan, mandated to facilitate consultation between the proponents of proposed uranium developments and stakeholder groups. The main stakeholder groups the Hidden Bay project will be required to interact with throughout all phases of the project life (Environmental Assessment, operations, closure and post closure) will be:

- Hatchet Lake First Nations;
- Fond du lac First Nation;
- Black Lake First Nation;
- Community of Stony Rapids;
- Community of Uranium City;
- Prince Albert Grande Council; and
- Northern Saskatchewan Environmental Quality Committee

Previous assessments involving the above stakeholder groups have shown the fundamental areas of concern involve the development and implementation of a robust environmental management plan throughout operations, coupled with a closure plan that ensures very low risk of long term environmental impacts. With respect to the Hidden Bay project the most sensitive valued ecosystem component would be Wollaston Lake.

The main stakeholder groups involved with this project's assessment will be looking for assurance that Wollaston Lake would not be adversely impacted in the short or long term as a result of the development of this project.

From a Socio-economic perspective, many if not all of these communities and political entities have interests in limited partnerships and other business ventures established to take advantage of the economic opportunities associated with northern Saskatchewan's mining industry.

These stakeholder groups would be looking for opportunities to enter into contractual arrangements to maximize the involvement of these businesses with the project in the event the project gains environmental assessment approvals to proceed.

20.6 Conceptual Decommissioning and Reclamation Plan

Based on the current scope of the project a robust, while economically realistic, closure plan can be developed for the Hidden Bay Project. The goals of this closure plan would be to:

- Restrict or eliminate the migration of all potential contaminants of concern from all sources on the mine site;
- Restrict or eliminate all potential radiological exposures to animal or humans;
- Restrict or eliminate all potential public safety risks associated with the decommissioned and reclaimed mine site; and
- Return the property, to the extent possible, to pre-mining conditions.

To meet the above goals all mine infrastructure would be removed. All clean waste rock piles would be contoured to a stable slope angle. The piles and disturbed areas would be scarified, vegetated and if necessary fertilized on a regular frequency until such time that the vegetation established itself. All PAG/ML waste would be covered with an appropriate cover designed to limit infiltration and encourage runoff. The underground and open pit mines would be allowed to flood. No water would be allowed to discharge to the environment from either of these facilities without meeting the site specific closure requirements which would be developed in consultations with the regulatory community and stakeholder groups prior to implementation of the decommissioning and reclamation plan.

Depending on the results of the geochemical characterization of waste rock it is possible that there are limited volumes of PAG/ML associated with this project. In which case it is possible the PAG/ML rock could be placed at the bottom of the open pit or in the underground workings as a viable closure option for this material. The water treatment plant would remain operational until such time that all mine water sources were below or met the site specific decommissioning and reclamation criteria.

21 Operating and Capital Cost Estimation

21.1 Operating Cost Estimate

The LOM average operating cost estimate is shown in Table 21.1.

Table 21.1: Unit OPEX Estimate Summary

Operating Factors	Unit	Unit OPEX Estimate
UG Mining Cost	$/t milled	67.75
OP Mining Cost	$/t mined	2.70
OP Mining Cost	$/t milled	106.68
Combined Mining Cost	$/t milled	73.85
Toll Treatment Cost	$/t milled	79.20
G&A (includes trucking of mill feed)	$/t milled	11.00
Water Treatment	$/t milled	1.83
Tailings Management	$/t milled	35.00
Average Unit operating Cost	**$/t milled**	**200.88**

21.1.1 Underground Mining Operating Cost Estimate

The underground mining operating cost was estimated for the owner operating scenario. The underground mining operating costs were calculated for each cost category such as development, production, haulage, maintenance, mine services, and labour required for the 1,000 t/d operating mine. The cost was estimated using a combination of first principles calculations, experience and factored costs. A contingency factor was not included in the cost estimation, but was applied in the financial model.

Table 21.2 shows the input data for cost estimation that were assumed in this study.

Table 21.2: Operating Cost Input Data

Operating Factors	Unit	Quantity
Underground Production		
Mine Days	d/a	365
Nominal Mining Rate	t/d	1,000
Average Mining Rate	t/a	365,000
Rock Characteristics		
In Situ Density of Mineralization	t/m^3	2.49
In Situ Density Waste	t/m^3	2.48
Swell Factor	%	50
Loose Density of Mineralization	t/m^3	1.66
Loose Density Waste	t/m^3	1.65
Shift Data		
Working Days per Week	ea.	7
Shifts per Day	ea.	2
Shift Length	hr	10
Shift Change	hr	0.5
Lunch Break	hr	0. 5
Equip Inspection	hr	0.25
Subtotal Non-productive	hr	1.25
Usable Time per Shift	hr	8.75
Shift Efficiency	%	88
Usable Minutes per Hour	min	50
Hour Efficiency (50 min/h)	%	83
Effective Work Time per Shift	hr	7.3

Productivities, equipment operating hours, labour, and supply requirements were estimated for each type of underground operation. Supply costs were based on estimated supply consumption and recent Canadian supplier's prices for drill and steel supplies, explosives, ground support, and services supplies, and were included in development and stoping costs. Maintenance consumables, such as parts, tyres, etc., as well as fuel ($1.0 /L) and power ($0.1 /kWh) were included in equipment operating costs and are part of mine development, stoping, haulage, or services costs.

The stope production cost was estimated based on estimated cycle times for each operation, supplies and equipment requirements for an average stope size. It was assumed that 90% of total mineralized material would be mined by C&F and D&F methods and 10% - by R&P.

The haulage cost of $3.98 per tonne was estimated for 20 t trucks based on an average haulage distance of 1,750 m on ramp and 200 m on surface.

The mine services cost was estimated based on equipment working time and materials supply required for ventilation, air heating, compressed air, transportation of personnel and materials, mine and road maintenance, mine dewatering, and underground construction.

The mine maintenance cost was estimated based on required maintenance equipment, tools and supplies for maintenance shop. Maintenance consumables, such as parts, tyres, etc., were included in equipment operating costs and are part of mine development, stoping, haulage, or services costs. The maintenance labour costs were included in the overall underground mine labour costs.

Mine safety and training costs were estimated based on the number of underground mine personnel and required personal protective equipment, first-aid and safety supplies, mine rescue, and safety training.

The underground labour requirements and rates used for determining the overall mining cost were based on experience for similar operations of this size, and were divided into salaried and hourly personnel. The labour costs include overtime and shift premiums, leave pay, bonuses, pension and superannuation benefits, insurance coverage and educational assistance.

Table 21.3: Average Salaried Personnel Cost

Staff Description	Quantity	Base Salary ($/year)	Loaded Salary ($/year)	Total Cost ($/year)
Mine Superintendent	1	110,000	165,000	165,000
Mine Captain	1	100,000	150,000	150,000
Mine Supervisor/Shift Boss	3	70,000	105,000	315,000
Senior Mining Engineer	1	90,000	135,000	135,000
Mine Ventilation/Project Engineer	1	70,000	105,000	105,000
Geotechnical Engineer	1	80,000	120,000	120,000
Senior Geologist	1	90,000	135,000	135,000
Geological Technician	3	65,000	97,500	292,500
Surveyor / Mine Technician	2	65,000	97,500	195,000
Surveyor Helper	1	50,000	75,000	75,000
Mine Rescue/Safety/Training Officer	1	70,000	105,000	105,000
Sub-total Mine Operating Staff	16			1,792,500
Maintenance Superintendent	1	100,000	150,000	150,000
Mechanical/Electrical G. Foreman	1	80,000	120,000	120,000
Maintenance Planner	1	70,000	105,000	105,000
Maintenance Supervisor/Shift Boss	3	70,000	105,000	315,000
Sub-total Mine Maintenance Staff	6			690,000
Total Mining Staff	**22**			**2,482,500**

Total hourly labour requirements were estimated to achieve the daily mining production rate based on 2 shifts at 10 h/d with 3 crews.

Table 21.4: Average Hourly Labour Cost

Staff Description	Quantity	Base Salary ($/hr)	Loaded Salary ($/year)	Total Cost ($/year)
Jumbo Operator	6	35.00	164,150	984,900
Ground Support/Services	12	32.00	150,080	1,800,960
Blaster	3	30.00	140,700	422,100
Blaster Helper	3	25.00	117,250	351,750
LHD Operator	6	30.00	140,700	844,200
Truck Driver	15	28.00	131,320	1,969,800
Diamond Driller	1	30.00	107,200	107,200
Diamond Drill Helper	1	25.00	89,333	89,333
Utility Vehicle Operator/Nipper	3	25.00	89,333	268,000
General Labourer	3	22.00	78,613	235,840
Grader Operator/Road Maintenance	1	25.00	89,333	89,333
Service Crew	3	25.00	89,333	268,000
Sub-Total Mine Operating Labour	57			7,431,417
HD Mechanic	6	35.00	164,150	984,900
Electrician	3	35.00	164,150	492,450
Tireman/Instrument Man	2	25.00	89,333	178,666
Welder	1	30.00	107,200	107,200
HD Mechanic Apprentice	1	20.00	71,467	71,467
Millwright	1	25.00	89,333	89,333
Sub-Total Mine Maintenance Labour	14			1,924,017
Total Mine Labour	**71**			**9,355,433**

Summary of total underground mining operating cost is shown in Table 21.5.

Table 21.5: Underground Mining Operating Cost Estimate Summary

Cost Distribution	Estimate ($/t)
Stope Development (Waste Crosscuts)	2.87
Stope Production	13.31
Stope Backfill	6.09
Sub-total Stoping Cost	**22.27**
Truck Haulage	**3.98**
Services	5.66
Maintenance	1.03
Exploration	0.23
Mine Safety, Training, Mine Rescue	0.38
Sub-total Mine Services Cost	**7.29**
Miscellaneous (5%)	**1.68**
Labour Operating	20.36
Labour Maintenance	5.27
Salaried Personnel	6.80
Sub-total Mine Labour	**32.43**
Total Mining Operating Cost per Tonne	**$67.65**

21.1.2 Open Pit Mining Operating Cost Estimate

Open pit mining costs were estimated on a simplified first principles basis and were based on manufacturer's input and SRK experience. Local labour rates and fuel prices (C$1.00/litre) were taken into account. A contract mining scenario forms the basis of the estimate.

The unit rate used in the Whittle™ optimization was $2.70 per tonne of material mined for pit and dump operations, road maintenance, mine supervision and technical services. The mining unit rate was calculated based on equipment required to achieve a maximum mining rate of 30 ktpd. Mining costs were developed from first principles for similar sized operations, along with local labour, fuel and power costs, along with an estimate of 10% for contractor profit margin. Table 21.6 summarizes the open pit OPEX estimate by function.

Table 21.6: Operating Cost Estimate by Function

Function	C$/t mined
Drilling	0.47
Blasting	0.40
Loading	0.31
Hauling	0.73
Roads/Dumps/Support	0.40
General Mine/Maintenance	0.11
Supervision/Technical	0.27
Total Open Pit OPEX	**2.70**

21.1.3 Toll Treatment OPEX Estimate

Toll treatment costs assumed a combined (Hidden Bay and Cameco) mill throughput of 3,000 tpd. A first principles basis was used of the treatment costs and then an additional 25% toll charge was added to the estimated actual processing costs. For an average head grade of 0.30% U_3O_8 and a processing rate of 3,000 t/day through the Rabbit Lake mill, the mill operating cost was estimated at $79.20/t milled. The following subsections provide a more detailed explanation of the processing cost estimation.

<u>Campaign Milling Operating Cost Estimates</u>

An initial estimate of operating costs for campaign milling of Hidden Bay mineralization through the Rabbit Lake mill, based on a mill throughput of 3,000 t/day and 0.15% U_3O_8 head grade, were provided in earlier Melis memoranda (December 3, 2010 and December 9, 2010). Subsequent evaluations of the deposit led to changes in both annual tonnage and head grades, necessitating revisions to operating cost estimates.

The key bases of the revised estimates presented in this memorandum are:

- Annual tonnages and feed grades as supplied by SRK (see table 21.7) .
- 95% net uranium recovery.

- A total production of 16.6 Mlbs U_3O_8/a from the Rabbit Lake Mill, comprised of a constant amount from Cigar Lake mineralization, the feed from Raven-Horseshoe mineralization and the remainder from Eagle Point ore (or equivalent).
- Battery limits are after the trucking of Hidden Bay mineralization to the Rabbit Lake mill, discharge of the tailings to the tailings management facility, and storage of the yellowcake produced in drums at the Rabbit Lake site.
- Processing requirements based on metallurgical characteristics of the Raven-Horseshoe mineralization as determined in previous Melis testwork.
- The operating cost estimate only includes costs directly applicable to the operation and maintenance of the Rabbit Lake mill, it does not include other site costs or G & A costs, nor does it include the costs associated with the tailings management facility.
- A contingency estimated at 10%.
- A toll milling charge of 25% (this remains to be negotiated with Cameco Corporation).
- Operating costs are estimated in first quarter 2011 Canadian dollars.

Table 21.7: UEX Corporation Hidden Bay Project Processing of Raven-Horseshoe Mineralization at the Rabbit Lake Mill Annual Feed Tonnage and Grade as Supplied by SRK

Year of Operation		Sum	1	2	3	4	5	6	7	8
Mill Feed	tonnes	2.5	350,000	350,000	350,000	350,000	350,000	350,000	350,000	40,000
Mill Feed Grade	%U_3O_8	0.3	0.54	0.39	0.3	0.23	0.23	0.24	0.19	0.19

The average process operating costs for the Raven-Horseshoe mineralization fed to the Rabbit Lake mill at an average grade of 0.30% U_3O_8 were estimated at $24,636,000 per year, equivalent to $79.20 per tonne of mill feed or $12.61/lb U_3O_8 produced. The annual, and average, estimated costs are listed below in Table 21.8.

Table 21.8: Summary of Estimated Rabbit Lake Mill Operating Costs for Processing Raven Horseshoe Mineralization by Year of Operation Including a Toll Milling Fee of 25%

Year of Operation	Feed Rate, Tonnes/a	Feed Grade, % U_3O_8	Production, lbs U_3O_8/a	Estimated Costs for Milling Raven-Horseshoe at Rabbit Lake Mill		
				$/a	$/tonne	$/lb(1) U_3O_8
1	350,000	0.54	3,958,000	38,290,000	109.40	9.67
2	350,000	0.39	2,859,000	31,558,000	90.17	11.04
3	350,000	0.30	2,199,000	27,622,000	78.92	12.56
4	350,000	0.23	1,686,000	24,597,000	70.28	14.59
5	350,000	0.23	1,686,000	24,597,000	70.28	14.59
6	350,000	0.24	1,759,000	25,028,000	71.51	14.23
7	350,000	0.19	1,393,000	22,881,000	65.37	16.43
8	40,000	0.19	159,000	2,680,000	67.00	16.84
Average	**311,250**	**0.30**	**1,962,000**	**24,636,000**	**79.20**	**12.61**

(1) Pound produced

The relationship between the Raven-Horseshoe feed grade and the estimated milling costs described in terms of dollars per pound produced is shown below in Figure 21.1.

The relationship between the Raven-Horseshoe feed grade and the estimated milling costs described in terms of dollars per tonne is shown below in Figure 21.2.

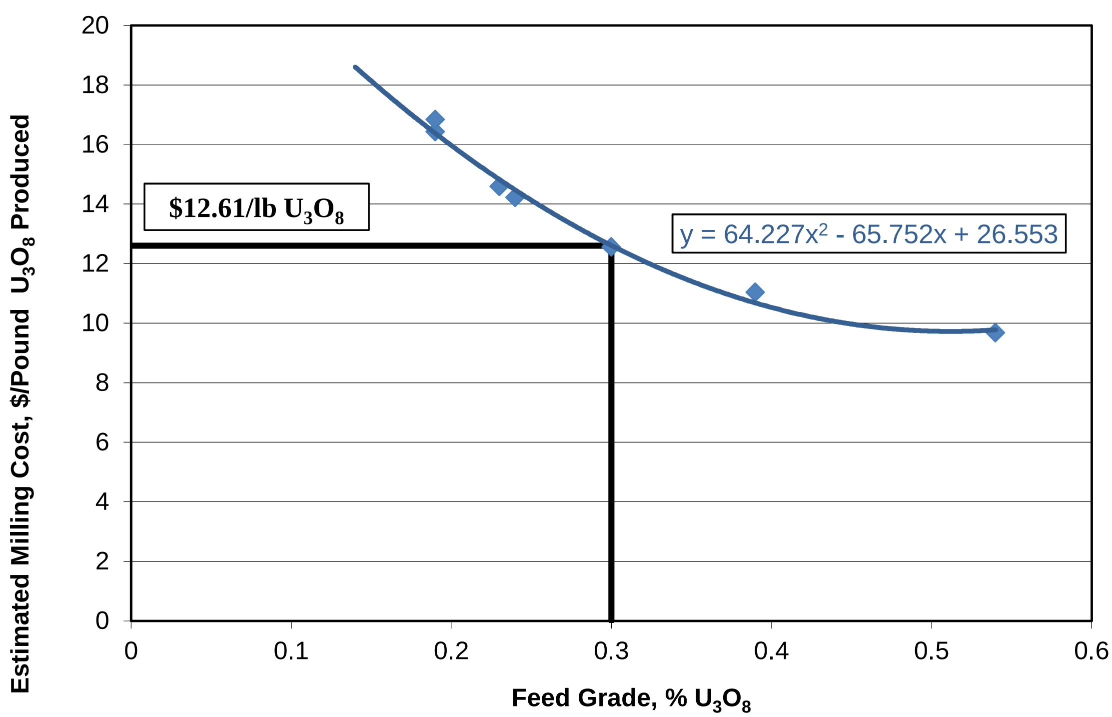


Figure 21.1: Summary of Estimated Horseshoe-Raven Mill Operating Costs ($/lb U_3O_8) Including a Toll Milling Fee of 25% at Different Horseshoe-Raven Feed Rates with Specified Co-Milling

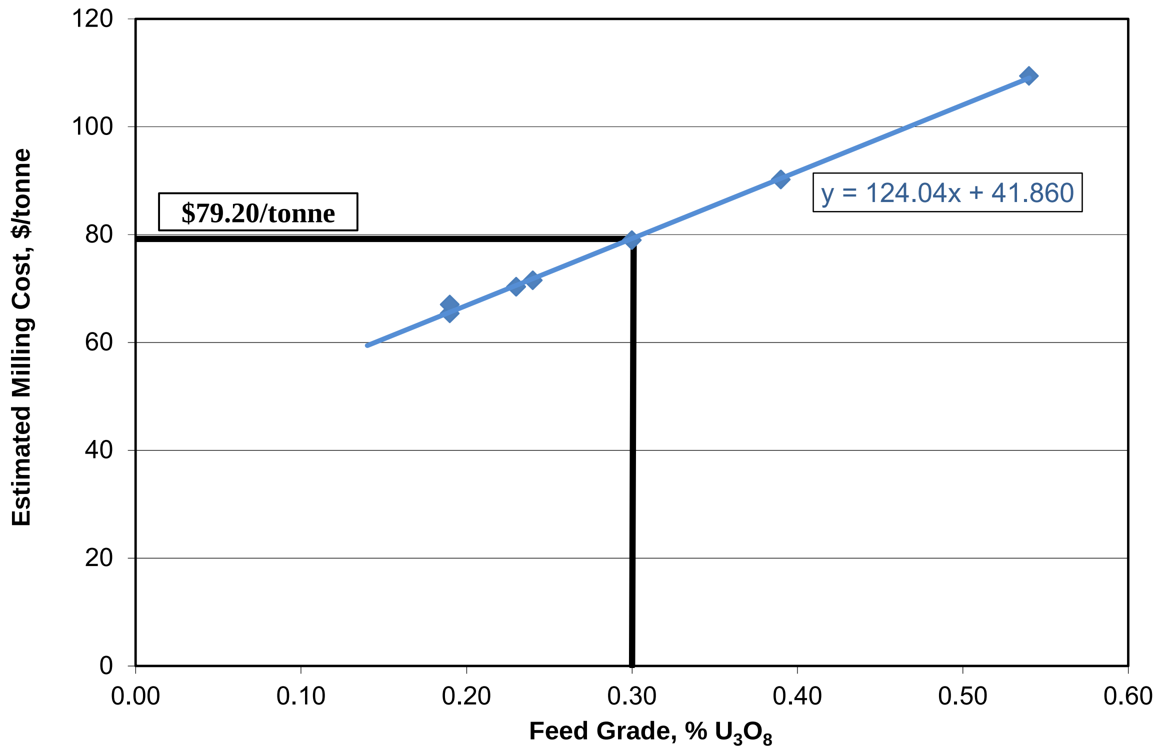


Figure 21.2: Summary of Estimated Horseshoe-Raven Mill Operating Costs ($/tonne milled) Including a Toll Milling Fee of 25% at Different Horseshoe-Raven Feed Rates with Specified Co-Milling

21.1.4 General and Administration OPEX Estimate

General and Administration costs were estimated based on other northern operations and factored as appropriate. A contract trucking cost of $1.10/t was assumed to transport mineralized material to the Rabbit Lake mill, a distance of 5 km.

21.1.5 Water Treatment OPEX Estimate

Water treatment costs were calculated based on treating 150,000 m^3/yr, a volume pro-rated from other sites in the region. The total treatment cost was estimated to be $570K/yr with a unit cost of 3.80/m^3. The following units costs were used:

- Reagents: $0.55/$m^3$
- Power: $0.20/$m^3$;
- Labour (4 operators): $2.38/$m^3$;
- Maintenance: $0.67/$m^3$.

21.1.6 Tailings Deposition OPEX Estimate

A tailings deposition cost of $35.00/t was assumed based on SRK's general knowledge.

21.2 Capital Cost Estimate

The summarized capital cost estimate for the project is shown in Table 21.9. Capital costs were taken as of Q4 2010. The basis for estimate of the capital costs included a combination of vendor quotes, factored estimates from similar projects and cost estimation reference documents. EPCM costs were assumed to be 12% of the capital cost (excluding UG equipment which would be handled by the owner). A contingency of 25% was applied to all capital costs.

Table 21.9: Capital Cost Estimate Summary

Item	Unit	Total	Pre-production	Sustaining
Underground Mine	M$	**45.2**	32.4	12.8
Open Pit	M$	**0.2**	0.0	0.2
Rabbit Lake Mill Upgrades	M$	**12.3**	12.3	0.0
Site and Facilities	M$	**18.9**	18.9	0.0
Owner's Costs	M$	**22.0**	22.0	0.0
Closure	M$	**10.0**	0.0	10.0
EPCM (12%)	M$	**6.9**	6.9	0.0
Contingency (25%)	M$	**28.9**	23.1	5.8
Total Capital Cost	**M$**	**144.5**	**115.7**	**28.8**

21.2.1 Underground Capital Cost Estimate

The underground mining capital cost estimate was based on the following:

- Underground mining equipment list;
- In-house database;
- Western Mining estimation references;
- Budget quotes obtained by SRK from development contractors and equipment manufacturers;
- Preliminary project development plan.

Mining capital was divided into equipment capital cost and mine development cost categories. The summary of underground mining capital cost by year is shown in Table 21.10.

Table 21.10: Summary of Underground Mining Capital Costs

Item	Year -1 (M$)	Year 1 (M$)	Year 2 (M$)	Year 3 (M$)	Year 4 (M$)	Year 5 (M$)	Year 6 (M$)
Equipment Capital Cost	17.35	1.69	1.88	0.98	0.98	0.98	0.98
Development Capital Cost	15.06	2.20	2.03	1.11			
Total Underground Mining Capital Cost	**32.41**	**3.89**	**3.91**	**2.09**	**0.98**	**0.98**	**0.98**

UG Equipment Capital Cost

The purchase of a permanent mining equipment fleet would be required as the underground mining operation will be performed by the owner.

It was assumed that for 6 years mine life it would not require to replace the major equipment fleet as most of underground equipment has the same life; however, annual purchasing of spare parts in amount of 5% of equipment cost was assumed to provide equipment maintenance. An additional 4% of equipment cost was applied to cover expenses for delivering equipment to the site.

The unit prices for equipment are shown in Table 21.11 were used for equipment capital cost.

Table 21.11: Underground Mining Equipment Unit Cost Estimates

Equipment	Unit	Quantity	Unit Cost (K$)
Drilling Equipment			
Jumbo (2 boom)	ea.	2	998.5
Rockbolter	ea.	2	807.0
Jackleg	ea.	8	5.3
Stoper	ea.	8	4.5
Loading & Hauling Equipment			
LHD, 3.7 m^3 (6.7 t)	ea.	2	625.0
LHD, 2.0 m^3 (4.0 t)	ea.	1	420.0
Haulage Truck, 20 t	ea.	5	700.0
Service Vehicles			
Grader	ea.	1	330.0
ANFO Loader	ea.	2	220.0
Cassette Carrier	ea.	2	275.0
Personnel Cassette	ea.	2	70.0
Boom Cassette	ea.	1	85.0
Fuel / Lube Cassette	ea.	1	120.0
Mechanics Truck	ea.	1	305.0
Scissor Lift	ea.	1	305.0
Supervisor/Engineering Vehicle	ea.	3	85.0
Electrician Vehicle - Scissor Lift	ea.	1	115.0
Shotcrete Sprayer	ea.	1	650.0
Transmixer	ea.	1	350.0
Bulldozer	ea.	1	350.0
Forklift	ea.	1	100.0
Primary Ventilation Fan	ea.	1	250.0
Primary Ventilation Fan Accessories	ea.	1	50.0
Fan Foundation and Installation	ea.	1	25.0
Auxiliary Ventilation Fan	ea.	8	30.0
Mine Air Heaters	ea.	1	500.0

Compressed Air			
Portable Compressor	ea.	2	70.0
Mine Water Management			
Main Dewatering Pump	ea.	2	55.0
Portable Pump	ea.	4	8.0
Mine Electrical			
Power Line	ea.	1	200.0
750kVA Portable Substation	ea.	4	115.0
Distribution and Safety Switches	ea.	1	300.0
Surface Substation and Misc.	ea.	1	120.0
Communication System			
Leaky Feeder System	ea.	1	150.0
Mine Safety			
Portable Refuge Station	ea.	2	80.0
Gas Monitoring System	ea.	1	10.0
Mine Rescue Equipment	ea.	1	50.0
First Aid Equipment	ea.	1	20.0
Cap Lamps	ea.	102	0.3
Cap Lamp Charger	ea.	3	15.0
Personal Protective Equipment	ea.	102	0.4
Fire Extinguishers	ea.	30	0.1
Sanitary Unit	ea.	4	5.0
Sanitary Pumping Tank System	ea.	1	5.0
Stench Gas System	ea.	1	22.0
Foam Generator	ea.	1	25.0
Mine Engineering Equipment			
Survey Equipment	ea.		75.0
PC, Printers, Network, Software	ea.		50.0
Mine Design Software	ea.		50.0
Geology Department Software	ea.		50.0
Miscellaneous			
Mining Tools	ea.	1	30.0
Surface Repair Shop	ea.	1	200.0
Explosives Storage			
Underground Powder Magazines	ea.	1	25.0
Underground Cap Magazines	ea.	1	20.0
ANFO Kettle	ea.	1	10.0

Purchasing of additional trucks in year 1 and 2 would be required as the haulage distance increases. Underground mining equipment capital cost by year is summarized in Table 21.12.

Table 21.12: Underground Mining Equipment Capital Cost Summary

	Year							
Item	**-1 (M$)**	**1 (M$)**	**2 (M$)**	**3 (M$)**	**4 (M$)**	**5 (M$)**	**6 (M$)**	**Total (M$)**
Drilling Equipment	3.69	0.02	0.02	0.02	0.02	0.02	0.02	3.81
Loading and Hauling Equipment	4.47	0.7	0.7					5.87
Service Vehicles	4.10							4.10
Ventilation	1.07							1.07
Compressed Air	0.14							0.14
Mine Water Management	0.26							0.26
Underground Electrical	1.08							1.08
Communication	0.15							0.15
Safety	0.43		0.11	0.11	0.11	0.11	0.11	0.98
Underground Engineering Equipment	0.28	0.02	0.02	0.02	0.02	0.02	0.02	0.37
Underground Miscellaneous	0.23							0.23
Underground Explosives Storage		0.06						0.06
Spare Parts (5%)	0.8	0.8	0.8	0.8	0.8	0.8	0.8	5.68
Freight (4%)	0.7	0.1	0.1					0.95
Total Underground Mining Equipment Capital Cost	**17.35**	**1.69**	**1.88**	**0.98**	**0.98**	**0.98**	**0.98**	**24.85**

UG Development Capital Cost

All development in pre-production period is included in capital costs. During mine production, the decline development, raise development, ventilation drifts and underground infrastructure were considered as capital costs, but stope crosscuts development was included in the mining operating cost.

It was assumed that all development in pre-production period would be done by contractor, so the contractor rates were used accordingly. It was assumed that contractor would demobilize from the site when owner start production. Then all jumbo development would be performed by owner but raise development – by contractor. The estimated owner costs per m of development excluding labour were applied for all jumbo development headings and the contractor rates were used for raise development.

Underground mine development capital costs is shown in Table 21.13.

Table 21.13: Underground Development Capital Cost

Development	Unit	Unit Cost (K$)	Year -1 (M$)	1 (M$)	2 (M$)	3 (M$)	Total (M$)
Mobilization	ea	200.0	0.20				0.20
Setup Services	ea	100.0	0.10				0.10
Decline Portal	ea	250.0	0.25				0.25
Main Access Decline	m	5.4	8.45	1.20	1.20	0.65	11.51
Remuck Bays	m	4.3	0.65	0.10	0.10	0.04	0.78
Ventilation Drift	m	4.9	1.99	0.46	0.40	0.12	2.97
Crosscut	m	5.4	1.51				1.51
Ventilation Raise Collar	ea	50.0	0.05				0.05
Ventilation Raise Pilot Hole	m	0.9	0.11				0.11
Ventilation Raise Ream	m	3.0	0.36				0.36
VR Ground Support/Manway	m	1.6	0.19				0.19
Drop Raise	m	3.0	0.23	0.36	0.33	0.21	1.13
Main Sump	ea	200.0	0.20			0.10	0.30
Powder Magazine	ea	100.0	0.10				0.10
Cap Magazine	ea	25.0	0.03				0.03
Underground Miscellaneous		100.0	0.10				0.10
Mechanical Shop	ea	200.0	0.20				0.20
Mine Dry	ea	150.0	0.15				0.15
Surface Miscellaneous		200.0	0.20				0.20
Site Tear-down	ea	30.0		0.03			0.03
Demobilization	ea	50.0		0.05			0.05
Total Development Capital Cost			**15.06**	**2.20**	**2.03**	**1.11**	**20.41**

21.2.2 Open Pit Capital Cost Estimate

Open pit capital costs were estimated to be $200K and include only pumps and pipeline for dewatering. No capital cost for the open pit major equipment has been included in the estimate since the Raven pit is proposed to be mined under a contract mining scenario.

21.2.3 Rabbit Lake Mill Upgrades

The operating costs for toll treatment were estimated based on a total of 3,000 tpd supplied to the Rabbit Lake Mill. It was assumed that a 1,000 tpd average would be supplied from the Hidden Bay project. In order for the Rabbit Lake facility to treat 3,000 tpd, it was estimated that several upgrades would have to be made. These upgrades would include increased grinding capacity, leaching capacity and counter-current decantation thickener capacity, with the main assumption that high rate feed wells can be installed to increase thickener unit rates.

The capital cost estimate for the Rabbit Lake mill upgrade (excluding EPCM and contingency) was calculated to be \$12.3M including:

- Labour: \$3.9M;
- Equipment: \$6.3M;
- Mobilization, demobilization and rentals \$1.0M;
- Contractor fee: \$0.5M; and
- Capital spares: \$0.5M.

If co-milling is the processing option used in order to fully utilize the approved Rabbit Lake mill throughput of 16,600,000 lbs U_3O_8 per year, as assumed in this report, other capital modifications to the mill circuit may be required to handle the extra solution flow through the solvent extraction circuit and the effluent treatment circuit, and may also be required for the additional uranium throughput. This potential additional capital cost will need to be reviewed in future discussions with Cameco Corporation regarding toll milling of the Raven-Horseshoe mineralization at the Rabbit Lake processing facility, as production planning takes place

21.2.4 Site Facilities and Infrastructure

The site facilities and infrastructure CAPEX estimate was based on factored costs from other similar installations and assumes new equipment. Construction and installation costs are included.

Table 21.14: Site Facilities and Infrastructure CAPEX Estimate

Description	Quantity	Units	Unit Price ($)	TOTAL COST (M$)
Surface Prep				
Site Roads	3	km	$200,000	0.60
Site Grading & Preparation	20,000	m2	$15	0.30
Subtotal Surface Preparation				0.90
Camp				
Kitchen	1	11' x 52' Unit	$300,000	0.30
Office	2	12' x 60' Unit	$100,000	0.20
Dry	2	12' x 64' Unit	$300,000	0.60
Bunkhouse Unit (8 Man)	10	12' x 64' Unit	$250,000	2.50
Recreation Hall	1	24' x 52'	$200,000	0.20
Camp Entrance & Hallway	400	m2	$400	0.16
Septic System	1	lot	$400,000	0.40
Water Well & Treatment System	1	lot	$150,000	0.15
Potable water system	1	lot	$30,000	0.03
Subtotal Camp				4.54
Buildings				
First-aid, Mine Rescue, inc ambulance	1	24' x 52'	$500,000	0.50
Warehouse	1	40' x 82'	$400,000	0.40
Maintenance Shop	1	20m x 30 m	$1,000,000	1.00
Compressor/Generator & Elec. Building	1	2 x Seacans	$50,000	0.05
Maintenance Shop Tools & Equipment	1	lot	$100,000	0.10
Subtotal Buildings				2.05
Environmental				
Water Treatment Facility	1	lot	$5,000,000	5.00
Water Treatment Pond	1	lot	$500,000	0.50
Water Treatment Operating Cost Yr-1	1	lot	$570,000	0.57
Liner for special waste dump	100,000	m^2	$14	1.40
Subtotal Environmental				7.47
Highway Diversion				
Highway 905 and power line diversion	1	km	$1,000,000	1.00
Subtotal Highway Diversion				1.00
Other				
Site power line	1	lot	$250,000	0.25
Site Infrastructure & Distribution	1	lot	$1,000,000	1.00
Spare Generator (600kva)	1	lot	$400,000	0.40
Fuel Farm	1	lot	$100,000	0.10
Loader/Forklift	2	Each	$300,000	0.60
Pickup Trucks	5	Each	$50,000	0.25
Telecommunications	1	lot	$75,000	0.08
Office Furnishings & Equipment	1	lot	$50,000	0.05
Engineering Equipment	1	lot	$100,000	0.10
Eng & Geol Software	1	lot	$110,000	0.11
Subtotal Other				2.94
SUB-TOTAL				**18.90**

21.2.5 Owner's Costs

Owner's capital costs were estimated to include resource definition drilling, engineering studies, field data collection, laboratory testwork, environmental and permitting initiatives and general overhead costs. A breakdown of the Owner's costs, excluding EPCM and contingency, are shown in Table 21.15.

Table 21.15: Owner's CAPEX Estimate

			Year				
Item	**Unit**	**Total**	**-5**	**-4**	**-3**	**-2**	**-1**
Environmental and permitting	M$	**4.0**	0.5	1.5	1.0	1.0	0.0
Engineering studies and design	M$	**6.0**	1.0	2.0	1.0	1.0	1.0
Resource upgrading and data collection	M$	**6.0**	2.5	0.5	0.3	2.5	0.3
Other	M$	**6.0**	1.0	1.0	1.0	1.0	2.0
Total Owner's CAPEX	**M$**	**22.0**	**5.0**	**5.0**	**3.3**	**5.5**	**3.3**

Estimates for environmental and permitting costs were based on the experience of SRK's Environmental QP, Mark Liskowich,. These costs assume a smooth permitting process with no unexpected issues. Most of the $4.0M estimate will be used for consultant's fees in establishing the project description, environmental assessment and licensing documents. It also includes on-going consultation with the government and other stakeholders.

Engineering study costs field data collection for metallurgical, geotechnical, hydrogeological and soils mechanics information. This field program would likely cost $0.3M to $0.6M and makes up part of the $1.0M allocated for a pre-feasibility study in year -5. In Year -4 the $2.0M budgeted is for more field work and a full feasibility study. The $1.0M per year in Years -3,-2 and -1 are for design work, assistance with permitting and ongoing engineering value adding studies.

Resource upgrading programs have been estimated to cost a total of $6.0M for the five preproduction years and encompass, resource definition drilling, resource expansion drilling, drilling for engineering field work and condemnation drilling.

Other owner's costs are for project overheads, insurance, salaries, head office charges, etc. A nominal $1.0M per year was assumed for each of years -5 to -2 and $2.0M in Year -5 as the project construction begins.

21.2.6 Working Capital

Working capital was assumed to be the equivalent of 4 months operating costs in the first production year or $18.8M. The working capital costs were recovered in the final production year yielding a net zero cost (undiscounted).

21.2.7 Closure

The preliminary closure cost estimate is $10M and does not include the backfilling of waste rock back into the pit. It assumed that waste dumps would be covered and vegetated. The UG portal would be sealed and the pit would be allowed to flood. Tailings closure costs were not included as they are deemed to be the responsibility of the toll treatment facility.

22 Economic Analysis

The economic analysis described in this report provides only a preliminary overview of the project economics based on broad, factored assumptions. The mineral resources used in the LOM plan and economic analysis include Inferred mineral resources. Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them to be categorized as mineral reserves, and there is no certainty that the inferred resources will be upgraded to a higher resource category. Based on this, there is no certainty that the results of this preliminary assessment will be realized.

22.1.1 Assumptions

Simplified earnings before interest and taxes ("EBIT") analyses were performed based on three product prices, US$60/lb U_3O_8, US$70/lb U_3O_8 and US$80/lb U_3O_8. For each case, the mineable tonnes and mill feed tonnes were held constant and the U_3O_8 price was varied only in the economic model.

For all cases, the Whittle optimization was performed using a U_3O_8 price of US$60/lb as discussed in the mining section.

Common economic model assumptions for all cases include:

- 5% discount rate ("DR") for net present value ("NPV") calculation;
- 100% equity financing assumed;
- C$1.05 to US$1.00
- Exclusion of all duties and taxes;
- 10% NSR provincial royalty;
- 95% process recovery;
- 100% payable U_3O_8 and no off-site costs (the toll treatment facility would carry these costs); and
- Five years of permitting and construction prior to production

22.1.2 Economic Analysis Results

Based on the Case A U_3O_8 price of US$60/lb, the EBIT internal rate of return ("IRR") was 442% and the EBIT net present value at a 5% discount rate ("$NPV_{5\%}$") was $163M.

The Case B economic analysis yielded an EBIT IRR of 55% and an EBIT $NPV_{5\%}$ of $267M results, at US$70/lb.

The Case C economic analysis yielded an EBIT IRR of 66% and an EBITDA $NPV_{5\%}$ of $371M results, at US$80/lb. See Table 22.1 for a summary of the economic results.

Table 22.1: Economic Results

Parameter	Unit	Case A	Case B	Case C
U_3O_8 Price	US$/lb U_3O_8	60	70	80
Royalty Payments (@10%)	M$	99	115	132
EBIT $NPV_{0\%}$	M$	246	394	542
EBIT $NPV_{5\%}$	M$	163	267	371
EBIT IRR	%	42	55	66
EBIT payback period	Production years	1	1	1

The break-even U_3O_8 price for the project is US$44/lb. The simplified EBIT economic analyses are shown in Tables 22.3 to 22.5.

Table 22.2: EBIT Model Elements Common to All Cases

				Year												
		UNIT	**TOTAL**	**-5**	**-4**	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**
MATERIAL SCHEDULE																
OP Mining	OP Waste	Mt	**15**											11.54	3.48	
	OP Ore	Mt	**0.4**											-	0.39	
	Grade	%	**0.19**											0.26	0.19	
	Total OP Mined lb	Mlb U_3O_8	**1.63**											-	1.63	
UG Mining	Ore	t	**2.1**						0.35	0.35	0.35	0.35	0.35	0.35		
	Grade	%	**0.32**						0.54	0.39	0.3	0.23	0.23	0.24		
	Total Mined lb	Mlb U_3O_8	**14.89**						4.17	3.01	2.31	1.77	1.77	1.85		
Total Mining	**Total Ore**	**Mt**	**2.5**						**0.35**	**0.35**	**0.35**	**0.35**	**0.35**	**0.35**	**0.39**	
	Grade	**%**	**0.3**						**0.54**	**0.39**	**0.3**	**0.23**	**0.23**	**0.24**	**0.19**	
	Total Mined lb	**Mlb U_3O_8**	**16.53**						**4.17**	**3.01**	**2.31**	**1.77**	**1.77**	**1.85**	**1.63**	
Processing	Daily Mill Feed	Ore t/day	**1,000**						1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000
	Mill feed	Mt	**2.5**						0.35	0.35	0.35	0.35	0.35	0.35	0.35	0.04
	Mill head grade	% U_3O_8	**0.3**						0.54	0.39	0.3	0.23	0.23	0.24	0.19	0.19
		Mlb U_3O_8	**16.53**						4.17	3.01	2.31	1.77	1.77	1.85	1.47	0.17
PLANT RECOVERY																
Recovery	Mill recovery	%	**95%**						95%	95%	95%	95%	95%	95%	95%	95%
Metal Production	U_3O_8 from mill	Mlb U_3O_8	**15.86**						3.96	2.86	2.2	1.69	1.69	1.76	1.39	0.16
OPERATING COST																
	UG mining cost	M$	**142**						23.7	23.7	23.7	23.7	23.7	23.7	0	0
	OP mining cost	M$	**42**						-	-	-	-	-	31.16	10.45	-
	Processing Opex	M$	**197**						21	21	21	21	21	21	21	2.4
	G&A	M$	**27**						3.85	3.85	3.85	3.85	3.85	3.85	3.85	0.44
	Water Treatment	M$	**5**						0.57	0.57	0.57	0.57	0.57	0.57	0.57	0.57
	Tailings	M$	**87**						12.3	12.3	12.3	12.3	12.3	12.3	12.3	1.4
	TOTAL OPEX	**M$**	**500**						**61**	**61**	**61**	**61**	**61**	**93**	**48**	**5**
	UG Mining Unit OPEX	$/t ore	**67.75**						67.75	67.75	67.75	67.75	67.75	67.75	67.75	67.75
	OP Mining Unit OPEX	$/t mined	**2.70**						2.70	2.70	2.70	2.7	2.7	2.7	2.7	2.7
	Processing Opex	$/t milled	**79.2**						79.2	79.2	79.2	79.2	79.2	79.2	79.2	79.2
	G&A	$/t milled	**11**						11	11	11	11	11	11	11	11
	Water Treatment	$/t milled	**1.83**						1.63	1.63	1.63	1.63	1.63	1.63	1.63	14.25
	Tailings	$/t milled	**35**						35	35	35	35	35	35	35	35
	Unit OPEX	**$/t milled**	**200.88**						194.58	194.58	194.58	194.58	194.58	283.6	156.68	139.45
	Unit OPEX	**$/lb U_3O_8**	**31.86**						17.2	23.82	30.97	40.39	40.39	56.42	39.37	35.04
CAPITAL COST																
Underground	UG Mine Development	M$	**20.4**					15.1	2.2	2	1.1					
	UG Mine Equipment	M$	**24.8**					17.4	1.7	1.9	1	1	1	1		
Open Pit	OP Mine Misc./Pumping	M$	**0.2**											0.2		
Process	Process Plant	M$	**12.3**				6.1	6.1								
Infrastructure	General Site (roads/surface prep)	M$	**0.9**				0.9									
	Camp and facilities	M$	**4.5**				4.5									
	Buildings	M$	**2.1**				2.1									
	Water treatment plant and ARD control	M$	**7.5**				6.9	0.6								
	Highway 905 diversion (inc. power line)	M$	**1**				1									
	Other site facilities	M$	**2.9**				2.9									
Owners Costs	Environmental and permitting	M$	**4**	0.5	1.5	1	1									
	Engineering studies and design	M$	**6**	1	2	1	1	1								
	Resource upgrading and data collection	M$	**6**	2.5	0.5	0.3	2.5	0.3								
	Other	M$	**6**	1	1	1	1	2								
EPCM	EPCM	M$	**6.9**				3.7	3.3								
Closure	Closure	M$	**10**													10
Working	Working capital	M$	**-**						20.4							-20.4
	Capital cost w/o contingency	**M$**	**115.6**	**5**	**5**	**3.3**	**33.6**	**45.6**	**22.3**	**3.9**	**2.1**	**1**	**1**	**1.2**	**-**	**-10.4**
Contingency	Contingency	%	**25%**	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%
	Contingency	M$	**28.9**	1.3	1.3	0.8	8.4	11.4	5.6	1	0.5	0.2	0.2	0.3		2.5
	TOTAL CAPITAL COST	**M$**	**144.5**	**6.3**	**6.3**	**4.1**	**42.1**	**57.1**	**27.9**	**4.9**	**2.6**	**1.2**	**1.2**	**1.5**	**-**	**-7.9**

Table 22.3: Case A EBIT Model Results (US$60/lb U_3O_8)

				Year												
		UNIT	**TOTAL**	**-5**	**-4**	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**
NET SMELTER RETURN																
Metal Production	U_3O_8 from mill	Mlb U_3O_8	**15.70**						3.96	2.86	2.20	1.69	1.69	1.76	1.39	0.16
Metal Prices	U_3O_8 Price	C$/lb	**63**						63	63	63	63	63	63	63	63
Payable Metal	Payable U_3O_8	%	**100**						100	100	100	100	100	100	100	100
	Payable U_3O_8	Mlb U_3O_8	**15.70**						3.96	2.86	2.20	1.69	1.69	1.76	1.39	0.16
	Gross Income pre-royalties	**M$**	**989**						**249**	**180**	**139**	**106**	**106**	**111**	**88**	**10**
Rate: 10%	Royalty	M$	**99**						25	18	14	11	11	11	9	1
	Gross Income From Mining	**M$**	**890**						**224**	**162**	**125**	**96**	**96**	**100**	**79**	**9**
NET OPERATING INCOME																
	NET OPERATING INCOME	**M$**	**390**						**156**	**94**	**57**	**27**	**27**	**0**	**24**	**3**
EBIT																
Discount rate: 5%	**EBIT**	**M$**	**246**	**-6**	**-6**	**-4**	**-42**	**-57**	**131**	**89**	**54**	**26**	**26**	**-1**	**24**	**11**
	Annual Discounted EBIT	M$	**163**	-6	-6	-4	-36	-47	103	66	38	18	17	-1	14	6
	Cum. EBIT	M$		-6	-13	-17	-59	-116	15	104	158	185	211	210	234	246
	Cum. Discounted EBIT	M$		-6	-12	-16	-52	-99	4	70	108	126	143	142	157	163

Table 22.4: Case B EBIT Model Results (US$70/lb U_3O_8)

				Year												
		UNIT	TOTAL	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8
NET SMELTER RETURN																
Metal Production	U_3O_8 from mill	Mlb U_3O_8	**15.70**						3.96	2.86	2.20	1.69	1.69	1.76	1.39	0.16
Metal Prices	U_3O_8 Price	$/lb	**74**						74	74	74	74	74	74	74	74
Payable Metal	Payable U_3O_8	%	**100**						100	100	100	100	100	100	100	100
	Payable U_3O_8	Mlb U_3O_8	**15.70**						3.96	2.86	2.20	1.69	1.69	1.76	1.39	0.16
	Gross Income pre-royalties	**M$**	**1,154**						**291**	**210**	**162**	**124**	**124**	**129**	**102**	**12**
Rate: 10%	Royalty	M$	**115**						29	21	16	12	12	13	10	1
	Gross Income From Mining	**M$**	**1,039**						**262**	**189**	**145**	**112**	**112**	**116**	**92**	**11**
NET OPERATING INCOME																
	NET OPERATING INCOME	**M$**	**538**						**194**	**121**	**77**	**43**	**43**	**17**	**37**	**5**
EBIT																
Discount rate: 5%	**EBIT**	**M$**	**394**	**-6**	**-6**	**-4**	**-42**	**-57**	**168**	**116**	**75**	**42**	**42**	**16**	**37**	**13**
	Annual Discounted EBIT	M$	**267**	-6	-6	-4	-36	-47	132	87	53	29	27	10	22	7
	Cum. EBIT	M$		-6	-13	-17	-59	-116	53	169	244	286	328	344	381	394
	Cum. Discounted EBIT	M$		-6	-12	-16	-52	-99	33	119	173	201	228	238	260	267

Table 22.5: Case C EBIT Model Results (US$80/lb U_3O_8)

		UNIT	TOTAL	Year -5	-4	-3	-2	-1	1	2	3	4	5	6	7	8
NET SMELTER RETURN																
Metal Production	U_3O_8 from mill	Mlb U_3O_8	**15.70**						3.96	2.86	2.20	1.69	1.69	1.76	1.39	0.16
Metal Prices	U_3O_8 Price	$/lb	**84**						84	84	84	84	84	84	84	84
Payable Metal	Payable U_3O_8	%	**100**						100	100	100	100	100	100	100	100
	Payable U_3O_8	Mlb U_3O_8	**15.70**						3.96	2.86	2.20	1.69	1.69	1.76	1.39	0.16
	Gross Income pre-royalties	**M$**	**1,319**						**333**	**240**	**185**	**142**	**142**	**148**	**117**	**13**
Rate: 10%	Royalty	**M$**	**132**						33	24	18	14	14	15	12	1
	Gross Income From Mining	**M$**	**1,187**						**299**	**216**	**166**	**127**	**127**	**133**	**105**	**12**
NET OPERATING INCOME																
	NET OPERATING INCOME	**M$**	**687**						**231**	**148**	**98**	**59**	**59**	**34**	**50**	**6**
EBIT																
Discount rate: 5%	**EBIT**	**M$**	**452**	**-6**	**-6**	**-4**	**-42**	**-57**	**206**	**143**	**96**	**58**	**58**	**32**	**50**	**14**
	Annual Discounted EBIT	M$	**371**	-6	-6	-4	-36	-47	161	107	68	39	37	20	29	8
	Cum. EBIT	M$		-6	-13	-17	-59	-116	90	233	329	387	445	477	528	542
	Cum. Discounted EBIT	M$		-6	-12	-16	-52	-99	62	169	237	276	314	333	363	371

22.1.3 Sensitivity Analysis

Sensitivity analyses were done for all cases by individually modifying the capital cost, operating cost, metal price and grade up and down by 20% to show the sensitivity of the EBIT $NPV_{5\%}$. The results show that the project is very robust. Like most mining projects, the project is most sensitive to most sensitive to commodity price and mill feed grade. For Case A, a 20% increase in U_3O_8 price leads to a 77% increase in pre-tax $NPV_{5\%}$ from $163M to $288M. A change in grade by 20% has a similar effect on $NPV_{5\%}$. The converse occurs if the metal price or mill feed grade drops by 20%, the pre-tax $NPV_{5\%}$ drops from $163M to $38M.

Operating costs are the next most sensitive parameter. A 20% increase in operating costs in Case A reduces the $NPV_{5\%}$ by $69M from $163M to $94M or -42%. In Case B, because of its higher base value, a 20% increase in operating costs only reduces the $NPV_{5\%}$ by 26%.

All cases are not particularly sensitive to capital as the total amount of capital compared to earnings. For Case A, a 20% increase capital costs results in a $25M (15%) drop in $NPV_{5\%}$.

A summary of the sensitivity analysis is shown in Table 22.6 and Figures 22.1 to 22.3.

Table 22.6: Sensitivity Analysis Results

Case	Variable	EBIT $NPV_{5\%}$ (M$)		
		-20% Variance	0% Variance	20% Variance
Case A	Capital Cost	187	163	138
	Operating Cost	232	163	94
	Metal Price	38	163	288
	Grade	38	163	288
Case B	Capital Cost	291	267	242
	Operating Cost	336	267	198
	Metal Price	121	267	413
	Grade	121	267	413
Case C	Capital Cost	396	371	346
	Operating Cost	440	371	302
	Metal Price	205	371	537
	Grade	205	371	537

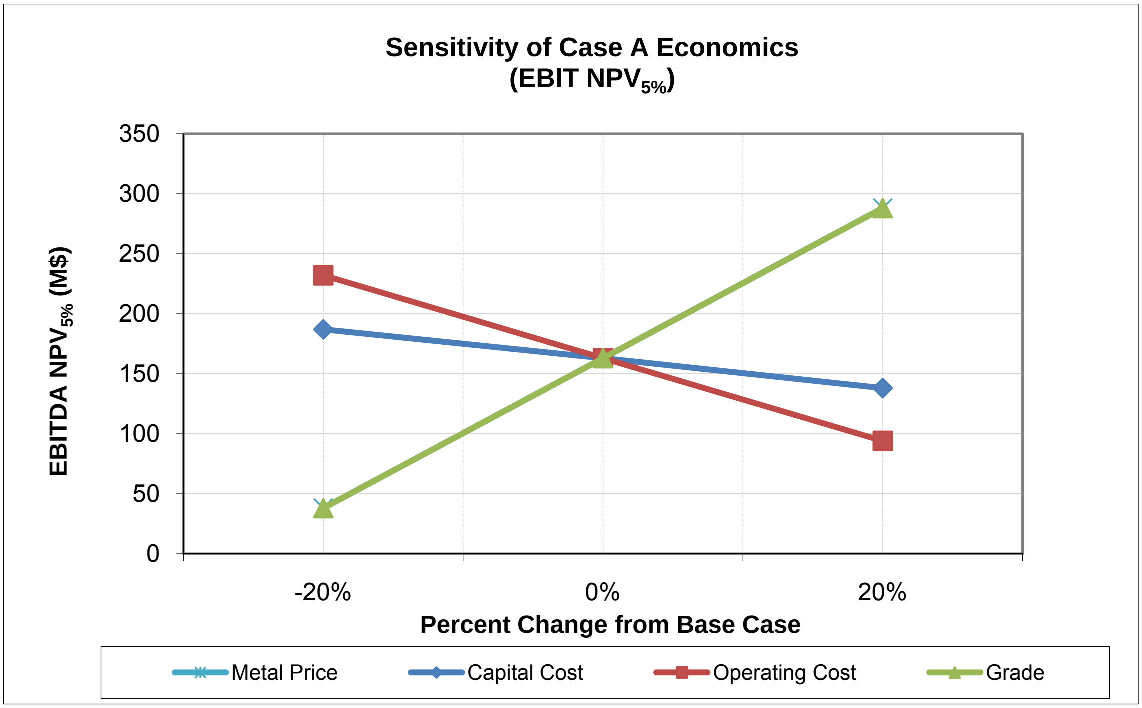


Figure 22.1: Case A Sensitivity Results

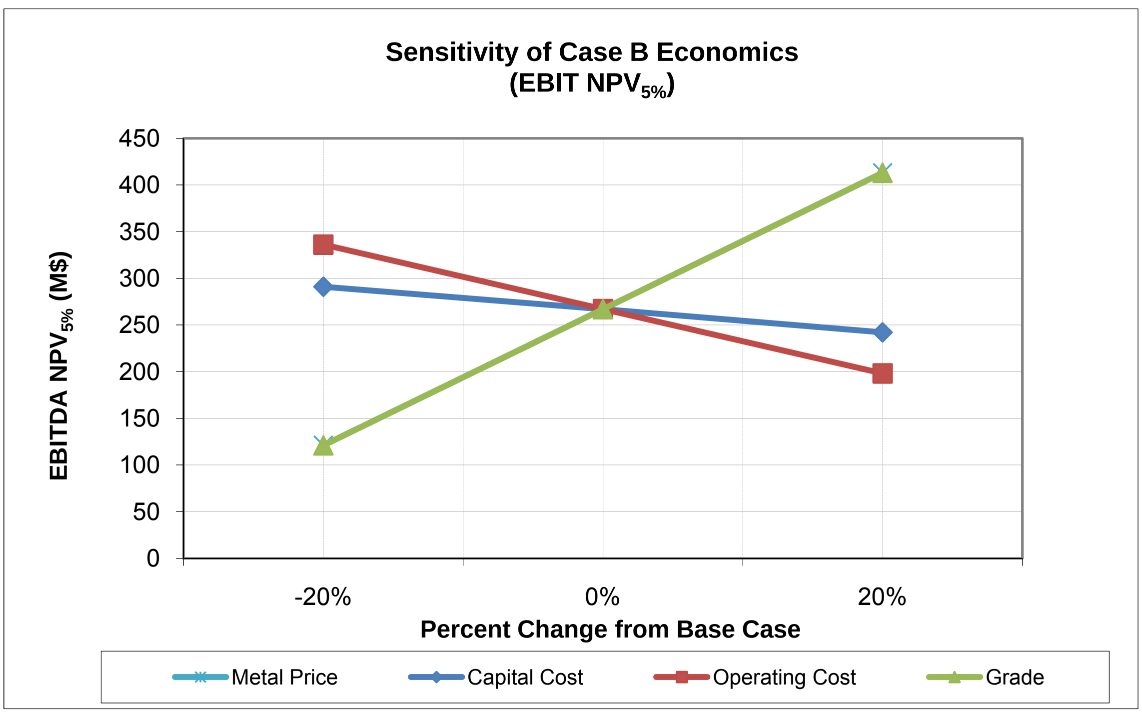


Figure 22.2: Case B Sensitivity Results

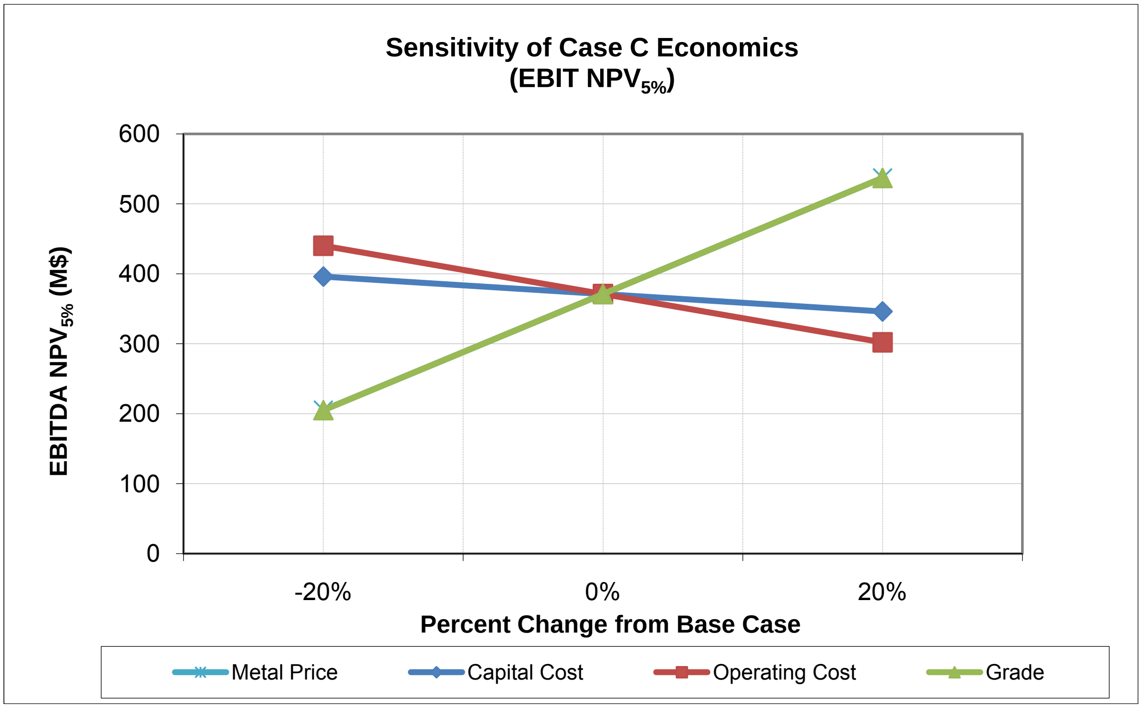


Figure 22.3: Case C Sensitivity Results

22.2 Payback Period

The payback period for all cases is occurs in year 1 of production. Figures 22.4 to 22.6 show the annual and cumulative EBIT results for Case A, B and C, respectively.

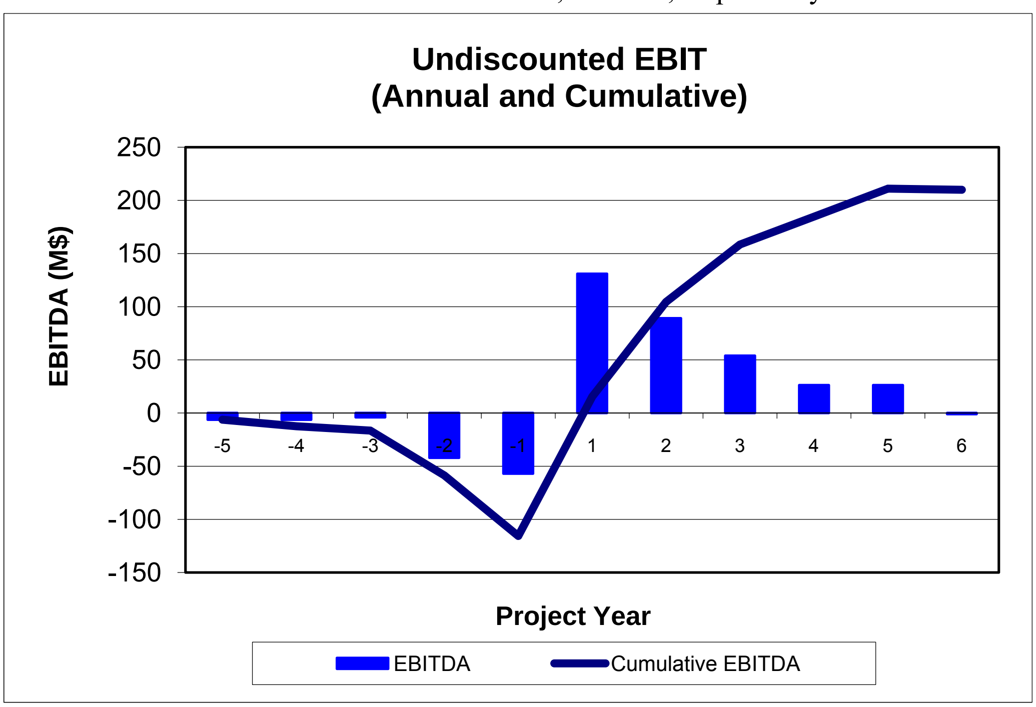


Figure 22.4: Case A Annual and Cumulative Undiscounted EBIT

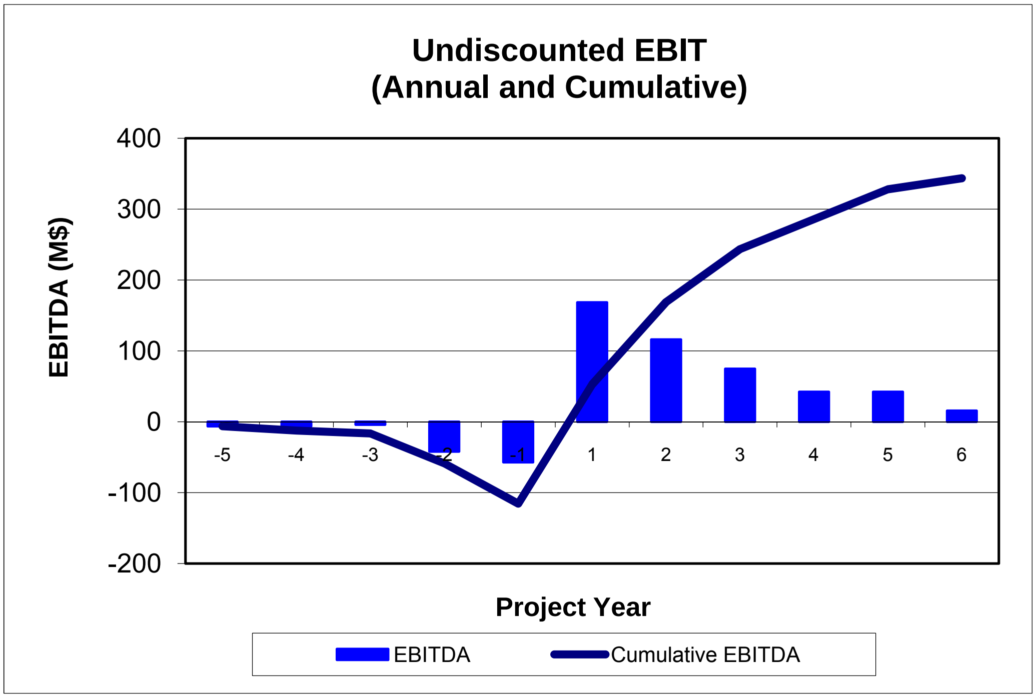


Figure 22.5: Case B Annual and Cumulative Undiscounted EBIT

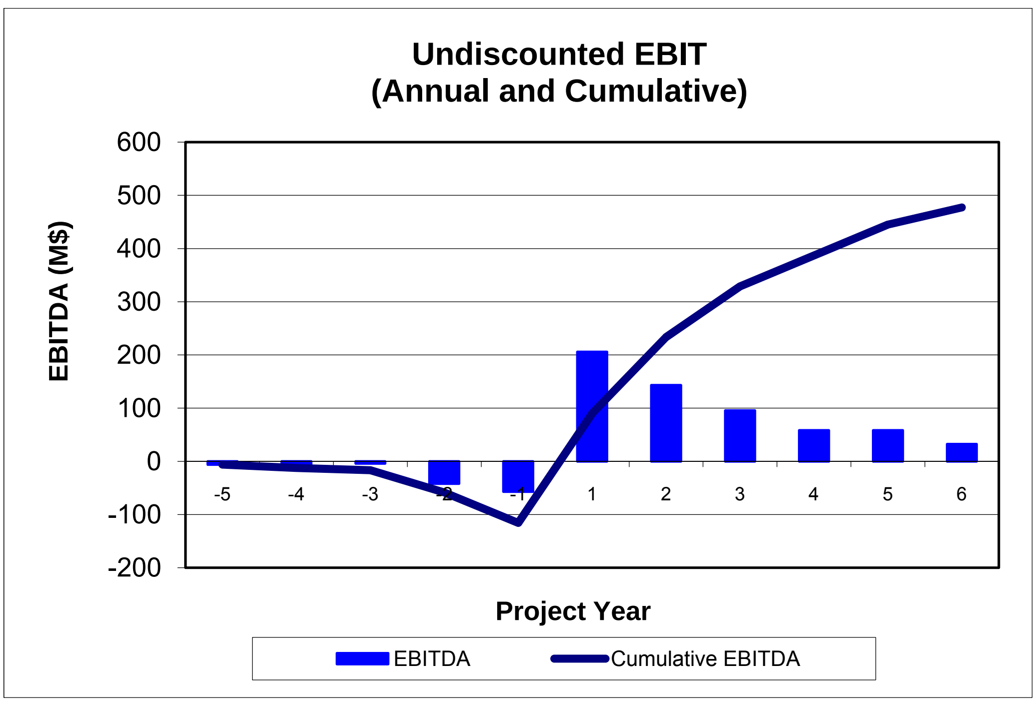


Figure 22.6: Case C Annual and Cumulative Undiscounted EBIT

22.3 Mine Life

The mine production life at the Hidden Bay project encompassing the Horseshoe and Raven deposits, based on the assumptions made in this study, is 7 years. There are a number of potential factors that could extend the mine life that have not been included in this report.

The Hidden Bay deposits will still contain resources, particularly at depth, after the proposed Raven pit is mined. The PA Whittle™ optimization analysis and subsequent mine design were conducted using a US$60/lb U_3O_8 price. If a higher price is used, then, all things being equal, the optimal Whittle™ shell would expand and the total mill feed and mine life would increase. Conversely, a drop in U_3O_8 price below US$60/lb will make the pit smaller.

There is the potential for further exploration discoveries and/or expansions of current known resources along known mineralized trends, including the West Bear deposit.

23 Interpretation and Conclusions

23.1 Resource Estimation

Golder was retained by UEX to complete updated mineral resource estimates for the Horseshoe and Raven deposits on UEX's Hidden Bay Project. Golder visited the project site as part of this initial undertaking, where the core logging and sampling methods were reviewed. Subsequent to the visit, the UEX QA/QC program and the drill hole sample database used to estimate the mineral resources were reviewed for the initial estimates and subsequent updates.

UEX has a formal QA/QC with a more rigorous program being implemented in July 2007 during the summer drilling program that continues to be followed. During the drill hole sampling process, 16 routine and four QA samples, which include a blank, a duplicate and 2 standard samples, are submitted for every 20 samples. The latter include a commercially available standard (certified reference material), a blank, a field duplicate and a round robin pulp. Most drill holes, which were completed under the management of UEX at both the Horseshoe and Raven deposits, utilized this program. Prior to the summer of 2007, blank samples had also been submitted throughout the 2006 and early 2007 drilling program.

The Golder data verification indicated that the logging, sampling, shipping, sample security assessment, analytical procedures, inter-laboratory assay validation and validation by different techniques are comparable to industry standard practices. All of the differences noted between the UEX databases and Golder's verification were either reconciled or corrected by UEX prior to the use of the database. The databases are considered acceptable for mineral resource estimation of the Horseshoe and Raven deposits.

The geological interpretation of the Horseshoe and Raven deposits were developed by UEX's geologists. Golder reviewed this geological interpretation and concluded that it is consistent with the data and the actual understanding of the deposits.

3D regular block models were constructed in Datamine and NN, ID^2 and OK used to interpolate block U_3O_8 grades. The OK interpolated capped grades have been used for reporting. The mineral resource classification criteria were based on the number and spatial distribution of samples used to estimate U_3O_8 grades. A variable bulk density was assigned to the subzones based on the mean of the samples lying within each subzone in the Horseshoe and Raven deposits. Subzones that had no data were assigned the overall mean value of the subzones for each deposit. The density values were assigned to each block based on the subzone.

The July 2009 Horseshoe Mineral Resource Estimate at a cut-off grade of 0.05% U_3O_8 results in 5,119,700 t at an average grade of 0.203% U_3O_8, yielding 22,895,000 lb U_3O_8 in the Indicated Mineral Resource category and 287,000 t at an average grade of 0.166% U_3O_8, yielding 1,049,000 lb U_3O_8 in the Inferred Mineral Resource category.

The July 2009 Raven Mineral Resource Estimate contains 5,173,900 Mt grading 0.107% U_3O_8 in the Indicated category, containing 12,149,000 Mlb of U_3O_8 and 822,200 Mt grading 0.092 % U_3O_8 in the Inferred category, containing 1,666,000 Mlb of U_3O_8 at a cut-off of 0.05% U_3O_8. At a 0.05% U_3O_8 cut-off, 88% of the tonnes are in the Indicated category.

The combined July 2009 N.I. 43-101 compliant resources for Horseshoe and Raven deposits and the January 2009 N.I. 43-101 compliant resources for the West Bear deposit on the Hidden Bay Project at a cut-off of 0.05% U_3O_8total 10.373 Mt which contain 36.623 Mlb U_3O_8 in the Indicated Mineral Resource category and 1.109 Mt containing 2.715 Mlb U_3O_8 in the Inferred Mineral Resource category. A summary of resources at various cut-offs is illustrated in Tables 23.1.

Table 23.1: Total NI 43-101 Compliant Indicated and Inferred Mineral Resources (Capped) on the Hidden Bay Project as of July 2009 at Various Cut-off Grades of % U_3O_8

Category	Cut-off	Tonnes	U3O8 (%)	U3O8 (lbs)
Indicated	0.02	16,876,600	0.112	41,617,000
	0.05	**10,372,500**	**0.16**	**36,623,000**
	0.10	5,434,300	0.242	28,989,000
	0.15	3,278,800	0.321	23,163,000
	0.20	2,054,800	0.409	18,503,000
	0.25	1,358,700	0.504	15,085,000
	0.30	913,800	0.616	12,408,000
	0.35	657,200	0.731	10,583,000
	0.40	506,600	0.837	9,345,000
Inferred	0.02	1,982,500	0.079	3,470,000
	0.05	**1,109,200**	**0.111**	**2,715,000**
	0.10	335,700	0.211	1,563,000
	0.15	202,800	0.27	1,208,000
	0.20	128,300	0.326	921,000
	0.25	79,200	0.388	678,000
	0.30	45,100	0.477	474,000
	0.35	27,200	0.58	348,000
	0.40	19,600	0.66	285,000

23.2 Mining Conclusions and Interpretations

Industry standard mining, process design, construction methods and economic evaluation practices have been used to assess the Horseshoe and Raven deposits. There is adequate geological and other pertinent data available to generate a PA.

Based on current knowledge and assumptions, the results of this study show that the project is economic and should be advanced to the next level of study by conducting the work indicated in the recommendations section.

As with almost all mining ventures, there are a large number of risks and opportunities that can affect the outcome of the project. Most of these risks and opportunities are based on a lack of scientific information (test results, drill results, etc.) or the lack of control over external drivers (metal price, exchange rates, etc.). The following section identifies the most significant potential risks and opportunities currently identified for the Hidden Bay project, many of which are common to mining projects at this early stage of study.

Subsequent higher-level engineering studies will needed to further refine these risks and opportunities, identify new ones and define mitigation or opportunity implementation plans.

The reader is reminded that this is only a preliminary study using factored costs and containing inferred resources. Subsequent studies may or may not verify the findings of this PA. While a significant amount of information is still required to do a complete assessment, at this point the project does not appear to have any fatal flaws.

The study met it its original objective of providing a preliminary review of the potential economic viability of the Horseshoe and Raven deposits.

23.3 Risks

Tables 23.2 and 23.3 highlight the main risks as currently understood for the project.

Table 23.2: Internal Project Risks

Risk	Explanation	Potential Impact	Possible Risk Mitigation
Geological Interpretation	Interpretation based on drill holes may be different to reality.	Tonnages and grade may differ from what is estimated in the model.	UEX has a put a significant effort into understanding the geology at Raven and Horseshoe. However, using a range of techniques could quantify the level of risk.
Mineral Resource Classification	The method used may either overestimate or underestimate the level of classification applied to the mineral resources.	Pre-Feasibility or Feasibility could be carried out on resources that do not meet the required level of confidence. This may result in erroneous decisions on the project.	The preliminary assessment defines annual productions and this combined with conditional simulation could be used to alternate method in order to assess the level of risk.
Permit Acquisition	The ability to secure a mining permit.	Failure to secure a mining permit would stop the project.	The development of close relationship with the local communities and government along with a thorough ESIA and a project design that gives appropriate consideration to the environment and local people is required before mining would be permitted.
Process Costs and Recoveries	The toll processing costs were calculated based on a total 3,000 tpd feed to the Rabbit Lake Mill from a combination of hidden Bay and other Cameco sources.	A reduction in mill throughput would add to processing costs and project economics could be negatively impacted. A 50% increase in processing costs would reduce the EBIT $NPV_{5\%}$ by \$68M or about 42% (Case A).	Risk mitigation could be realized with a fixed cost toll processing contract.
	A reduction in uranium recovery would have a negative impact on the project economics.	A process recovery of 95% was assumed in the economic analysis based on preliminary tests. A reduction in recovery to 90% would lower the EBIT $NPV_{5\%}$ by \$33M.	Further testwork would confirm recoveries.
CAPEX and OPEX	The ability to achieve the estimated CAPEX and OPEX costs is an important element of the project economics	An increase in OPEX of 20% would reduce the EBIT $NPV_{5\%}$ by \$69M, assuming tonnages are held constant. An increase in CAPEX of 20% has a \$25M negative impact on EBIT $NPV_{5\%}$.	Further cost accuracy with the next level of study as well as the active investigation of potential cost-reduction measures
Development Schedule	The development schedule (in particular, permitting) may be aggressive and the project could be delayed depending on several factors.	A change in schedule would alter the project economics by having a longer lead time until production.	Further project definition would be needed before a more reliable permitting timeline could be estimated.
Mining Dilution and Extraction	The ability to extract mineralized material with minimum dilution is a potential risk.	Waste dilution would add to treatment costs per lb of U_3O_8 and have a negative impact on project economics. A reduction in mill head grade of 20% would lead to a drop in EBIT $NPV_{5\%}$ of \$125M.	Selective mining methods have been chosen for the UG deposits which will help mitigate dilution unless the mineralized zones are very erratic. Further geological definition of the deposits could help verify geologic modelling and improve mining confidence.
Water Management and Geochemistry	It has been assumed that the 25% of the waste dumps need to be lined.	If geochemical testing of the waste rock indicates that all of the waste dumps will have to be lined the capital cost of the project would increase by \$4.2M.	Adequate testing of tailings and waste materials needs to be done to determine the liner requirements. A water treatment plant is included already in the CAPEX and OPEX estimates.

Table 23.3: External Project Risks

Risk	Explanation	Potential Outcome	Possible Risk Mitigation
Uranium price	Uranium price has a significant impact on the economic viability of the project.	A 20% drop in U_3O_8 price reduces the project EBIT $NPV_{5\%}$ by $125M assuming the mineable tonnes are not changed.	Future strong demand for uranium through its increased acceptance as a “clean” energy source may improve prices in the long term.
Finance	The project will require an estimated $116M in pre-production capital	Failure to secure funding could slow the project or stop its development altogether	Continued value-adding field work including additional resource development and technical studies as well as developing a financing plan if the project continues to develop are needed
Toll Treatment	The project relies on the use of the Rabbit Lake processing facility for the treatment of Hidden Bay mineralized material.	The inability of the company to toll treat at Rabbit Lake would mean that toll treatment at McClean Lake would have to be pursued or a stand-alone processing facility built.	The construction of a processing facility by UEX for the Hidden Bay deposits would add considerably to the CAPEX and the permitting timeline and uncertainties
Toll Tailings Deposition	The project relies on the use of the Rabbit Lake processing facility for the treatment of Hidden Bay mineralized material. It is likely that Rabbit lake does not currently have permitted tailings storage space available.	The inability of the company to deposit tailings at Rabbit Lake would mean that the option of storage in the Raven pit would have to be pursued.	The use of the Raven pit for tailings storage would likely add cost, complexity and potentially delay to the permitting of the project, although OPEX would potentially reduce significantly.

23.4 Opportunities

Table 23.4: Project Opportunities

Opportunity	Explanation	Potential Benefit
Tailings Storage in the Raven Pit	There is a shortage of tailings storage volume in the region and the use of the mined-out Raven pit could provide a minimum of 4 to 5 Mm^3 of tailings storage and potentially much more.	This study assumes a tailings deposition cost of $35/t milled by using Cameco's facilities. The use of the Raven pit to store tailings, and elimination of the toll tailings deposition fee, could significantly reduce the tailings deposition costs, potentially up to $50M. This OPEX reduction would lower the cut-off grade of the pit and lead to its significant expansion (double the current size). The expansion of the Raven pit to accept more tailings could further increase the economic benefit of the project by providing a regional toll facility for tailings deposition.
West Bear	The inclusion of the West Bear deposit into the Horseshoe and Raven LOM plans.	The addition of the West Bear with Horseshoe and Raven plans may create a longer mine life, a more flexible mining schedule and cost savings when compared to analysing the deposits independently. The West Bear Pre-feasibility Study completed by Golder in 2010 estimated a Probable Mineral Reserve estimate of 1.5Mlb of U_3O_8 grading 0.94% U_3O_8 at a cut-off of 0.18% U_3O_8 which represents 96% of the mineral resource. The high conversion rate reflects the near-surface nature of the West Bear mineralization which is amenable to shallow, open-pit mining. The Study presents the base-case scenario uranium price of C$77.73/lb U_3O_8 , resulting in a NPV of $23.4 million and an Internal Rate of Return of 118%. The West Bear project economics may improve due to the sharing of facilities and infrastructure with Horseshoe-Raven.
Uranium Prices	Uranium prices have the biggest single impact on the project economics.	An increase in uranium value from the prices assumed in this study would have a significant economic benefit. In Case A, a 20% increase in U_3O_8 price increases the EBIT $NPV_{5\%}$ of about $125M or 77%. Uranium prices are thought by some to be at the lower end of their cycle.
	An increase in U_3O_8 prices to US$80/lb, all other things being equal, would lower the cut-off grade and enable substantially more mineralized material to be considered economic.	A simple, ***preliminary*** exercise was conducted to estimate the potential increase in mineable tonnes and NPV if a price of US$80/lb U_3O_8 was used in mine optimization and the economic model instead of the US$60/lb in Case A. The exercise showed: - an increase in mineable tonnes from 2.5Mt to 4.8Mt - A reduction in mined grade from 0.3% to 0.2% U_3O_8 - An increase in contained metal from 16.5 Mlb U_3O_8 to 23.6 Mlb U_3O_8 - A potential increase in EBIT $NPV_{0\%}$ of about $78M.
Exploration Potential	Favourable exploration potential in the area could increase resources and might have a positive impact on the project mineral resources	Increased resources would lead to potentially better project economics if they could be converted to reserves in the future. More tonnes available to mine would potentially increase project revenues, without necessarily adding capital cost.
Resource Grade	An increase in resource grade of uranium would have a significant impact on the economic viability of the project.	A 20% increase in uranium grade, while maintaining tonnes, increases the Case A EBIT $NPV_{5\%}$ by $125M.
Potential Synergies with Local Producers	The proximity of the Cameco's Rabbit Lake and AREVA's McClean Lake facilities makes for the potential of working together to improve economics for all the operations.	Cost saving could potentially be realized from shared camp complexes, spare parts, optimized combined production schedules, tailings storage, etc.

24 Recommendations

24.1 Interpretation Risk

During the review of the Horseshoe and Raven Datamine 3D block model, comparisons between different estimation methods (nearest neighbour and inverse distance power against kriging interpolation method) were completed. This review noted that out of a total of 43 mineralized subzones, 13 of the subzones had a difference in interpolated grade of greater than 15% when compared to nearest neighbour, inverse distance models or the declustered mean. Five of these subzones show a discrepancy in more than one of the comparisons. These five subzones make up only a small portion of the resource. This may be due to the geological interpretation.

In order to quantify the risk due to interpretation, a single mineralized envelope should be constructed to contain the majority of samples with an assay of greater than 0.02% U_3O_8 for Raven and Horseshoe and the mineral resources re-estimated. The internal low grade clay alteration at Raven should also be modelled so that the data within the alteration can be uniquely coded.

The estimated cost of evaluating the risk in the current interpretation would be approximately $80,000.

24.2 Mineral Resource Classification Risk

The completion of this preliminary assessment provides annual rates for production which when combined with a conditional simulation exercise could provide an alternate method for classifying the resources. Prior to the commencement of a pre-Feasibility or Feasibility it is recommend that risk be analyzed.

The estimated cost of evaluating the risk in the current classification would be approximately $40,000.

24.3 Mining and Exploration

During the review of the Horseshoe Datamine 3D block model, comparisons between different estimation methods (nearest neighbour and inverse distance power against kriging interpolation method) were completed. This review noted that out of a total of 43 mineralized subzones, 13 of the subzones had a difference in interpolated grade of greater than 15% when compared to nearest neighbour, inverse distance models or the declustered mean. This may be due to the geological interpretation.

In order to quantify the risk due to interpretation, a single mineralized envelope should be constructed to contain the majority of samples with an assay of greater than 0.02% U_3O_8 for Horseshoe and Raven and the mineral resources re-estimated. The internal low grade clay alteration at Raven should also be modelled so that the data within the alteration can be uniquely coded.

The estimated cost of evaluating the risk in the modelling method would be approximately $80,000.

24.3.1 Preliminary Assessment, Pre-Feasibility and Feasibility Studies

A combined engineering study is recommended to assess the viability of the Hidden Bay project when all of the main deposits are considered: West Bear, Horseshoe and Raven. The study could be done as a PA to quickly assess the combined benefits of these deposits or as a higher-level pre-feasibility study that could potentially show reserves. A combined PA would likely cost $100K while a PFS, along with further engineering and metallurgical data collection and testing would likely cost between $1.0M to 1.5M and would involve geotechnical and hydrogeological drilling as well as soils testing.

24.3.2 Exploration

The footprint of the Horseshoe and Raven deposits was successfully expanded by definition drilling in the winter of 2008/2009. Drilling has now tested the area of the previous historical outline defined by Gulf. Given the successful results from drilling the Horseshoe and Raven deposits over the last several years, UEX intends to carry out an aggressive drilling program in 2011 to test additional geological and geophysical targets in the area. These outlying exploration targets include areas with resistivity and gravity anomalies similar to those at the Horseshoe and Raven deposits, suggesting the possibility of new zones of clay alteration which may be associated with uranium mineralization. This drill program will also test structural targets where projections of known faults may extend across potentially favourable lithologies that are host to uranium mineralization.

In addition to the drilling program in the Horseshoe and Raven deposit areas, further exploration drilling is planned for the Shamus Lake area in the northwestern part of the Hidden Bay property. This area, which lies just south of and along strike from UEX's Telephone Lake area and the Sue Deposits on the adjacent McClean Lake mine property operated by AREVA Resources Canada Inc., has the potential for the discovery of Sue C or Eagle Point style mineralization along the Telephone Lake Fault. The Telephone Lake Fault, which lies along the southeast side of the McClean Lake Dome, splits from a single structure in the Telephone Lake area into several strands on the Shamus grid.

Previous drilling in Shamus with widely spaced drill holes intersected several areas of low grade mineralization with associated alteration that returned grades ranging from 0.1% to 0.46% U_3O_8 over intervals of several metres, including 0.39% U_3O_8 over 2.2 m in hole SHA-20. Mineralization was intersected both in basement rocks, where they are associated with fault strands often marginal to or within pegmatite and adjacent graphitic gneiss, and in the vicinity of the sub-Athabasca unconformity.

The company continues to evaluate Shamus as there are still numerous untested targets within the area which may host unconformity- or basement-hosted uranium mineralization, similar to the Eagle Point Mine or the Sue deposits.

Further exploration drilling is planned for 2011 in the Shamus Lake area to test target areas identified by recent geophysical data which are down dip from several previously intersected areas of uranium mineralization encountered near the Athabasca unconformity and in underlying basement rocks.

In total, approximately 12,000 m of drilling are proposed during the winter of 2011 to test targets in the vicinity of the Horseshoe and Raven deposits as well as in the Shamus Lake area. At established all-in costs of drilling, on-site camp/accommodation, transportation, assaying/sampling, salaries/contractors fees, supplies, expediting and management, based on UEX's ongoing exploration in the area, this equates to a cost of approximately $2.0 million.

Infill holes to upgrade Inferred portions of the Horseshoe and Raven resources to Indicated status could also be considered, but since resources are dominantly in the Indicated category and most Inferred resources are in lower grade zones, such additional drilling is considered low priority. The drilling and attempted upgrade of inferred resources increase in importance as U_3O_8 price goes up and the mining areas expand, pushing into zones of inferred resources.

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26 Abbreviations and Acronyms

Distance	
µm	micron (micrometre)
mm	millimetre
cm	centimetre
m	metre
km	km
” or in	inch
’ or ft	foot
Area	
ac	acre
ha	hectare
Time	
s	second
m or min	minute
h or hr	hour
d	day
y or yr	year
Volume	
l	litre
usg	US gallon
lcm	loose cubic metre
bcm	bank cubic metre
Mbcm	million bcm
Mass	
kg	kilogram
g	gram
t	metric tonne
Kt	kilotonne
lb	pound
Mt	megatonne
oz	troy ounce
wmt	wet metric tonne
dmt	dry metric tonne
Pressure	
psi	pounds per square inch
Pa	pascal
kPa	kilopascal
MPa	megapascal
Elements and Compounds	
Au	gold
Ag	silver
As	arsenic
Cu	copper
Fe	iron
Mo	molybdenum
Pb	lead
S	sulphur
U_3O_8	triuranium octoxide a constituent of “yellowcake”
U	uranium
Zn	zinc
Electricity	
kW	kilowatt
kWh	kilowatt hour
V	volt
W	watt
Ω	ohm
A	ampere

Unit Prefixes	
µ	micro (one millionth)
m	milli (one thousandth)
c	centi (one hundredth)
d	deci (one tenth)
k or K	kilo (one thousand)
M	Mega (one million)
G	Giga (one trillion)
Temperature	
^oC	degree Celsius (Centigrade)
^oF	degree Fahrenheit
Misc.	
Btu or BTU	British Thermal Unit
Ø	diameter
r	radius
hp	horsepower
s.g.	specific gravity
masl	metres above sea level
elev	elevation above sea level
Rates and Ratios	
p or /	per
mph	miles per hour
cfm	cubic feet per minute
usgpm	United States gallon per minute
tph	tonnes per hour
tpd	tonnes per day
mtpa	million tonnes per annum
ppm	parts per million
ppb	parts per billion
Acronyms	
SRK	SRK Consulting (Canada) Inc.
CIM	Canadian Institute of Mining
NI 43-101	National Instrument 43-101
ABA	acid- base accounting
AP	acid potential
NP	neutralization potential
ML/ARD	metal leaching/ acid rock drainage
PAG	potentially acid generating
non-PAG	non-potentially acid generating
RC	reverse circulation
DD / DDH	diamond drill / diamond drill hole
IP	induced polarization
HL	heap leach
COG	cut off grade
NSR	net smelter return
NPV	net present value
LOM	life of mine
EBIT	earnings before interest and taxes
IRR	internal rate of return
DR	discount rate
PA	preliminary assessment
PFS	preliminary feasibility study
FS	feasibility study
Conversion Factors	
1 tonne	2,204.6 lb
1 troy ounce	31.1035

27 Date and Signature Page

This technical report was written by the Qualified Persons listed below. The effective date of this technical report is February 15, 2011.

Qualified Person	Signature	Date
Bruce Murphy, FSAIMM	Original signed	February 23, 2011
Dino Pilotto, P.Eng.	Original signed	February 23, 2011
Gordon Doerksen, P.Eng	Original signed	February 23, 2011
Kevin Palmer, P.Geo.	Original signed	February 23, 2011
Lawrence Melis, P.Eng.	Original signed	February 23, 2011
Mark Liskowich, P.Geo.	Original signed	February 23, 2011

Reviewed By

Chris Elliott, AusIMM

All data used as source material plus the text, tables, figures, and attachments of this document have been reviewed and prepared in accordance with generally accepted professional engineering and environmental practices.

28 Certificates of Qualified Persons

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CERTIFICATE OF QUALIFIED PERSON

Gordon Doerksen, P.Eng.

I, Gordon Doerksen, am a Professional Engineer, employed as a Principal Consultant - Mining with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "Preliminary Assessment Technical Report on the Horseshoe and Raven Deposits, Hidden Bay Project, Saskatchewan, Canada" submitted on February 23, 2011 with an effective date of February 15, 2011.

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia. I graduated with a BS (Mining) degree from Montana College of Mineral Science and Technology in May 1990.

I have been involved in mining since 1985 and have practised my profession continuously since 1990. I have been involved in mining operations, mine engineering and consulting covering a wide range of mineral commodities in Africa, South America, North America and Asia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I visited the Hidden Bay Project site in August 2010.

I am responsible for the Summary and Sections 1, 2, 18.2 to 18.6, 18.8, 18.9, 19, 21.1.1, 21.1.4, 21.2.1, 21.2.4, 21.2.5, 21.2.6, 22, 23, 24.3, 25, 26, 27 and 28 of the "Preliminary Assessment Technical Report on the Horseshoe and Raven Deposits, Hidden Bay Project, Saskatchewan, Canada" submitted on February 22, 2011 with an effective date of February 15, 2011.
.

I am independent of UEX Corporation as independence is described by Section 1.4 of NI 43-101.

I have not previously been involved with the Hidden Bay Project.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.
As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

ORIGINAL SIGNED AND STAMPED

Gordon Doerksen, P.Eng. Dated: February 23, 2011

QP Certificate Doerksen_Hidden Bay PA_20110222.doc

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CERTIFICATE OF QUALIFIED PERSON

Bruce Murphy, FSAIMM.

I, Bruce Murphy, am a fellow of the South African Institute of Mining and Metallurgy, employed as a Principal Consultant - Mining Rock Mechanics with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "Preliminary Assessment Technical Report on the Horseshoe and Raven Deposits, Hidden Bay Project, Saskatchewan, Canada" submitted on February 23, 2011 with an effective date of February 15, 2011.

I graduated with a B.Sc. Honours (Mining and Exploration Geology) degree and a M.Sc. Engineering (Mining) degree from the University of the Witwatersrand, Johannesburg in 1989 and 1996 respectively.

I have been involved in mining since 1989 and have practised my profession continuously since then. I have been involved in mining operations, mine rock engineering and consulting covering a wide range of mineral commodities in Africa, South America, North America and Asia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I visited the Hidden Bay Project site in August 2010.

I am responsible for Section 18.1 of the "Preliminary Assessment Technical Report on the Horseshoe and Raven Deposits, Hidden Bay Project, Saskatchewan, Canada" submitted on February 22, 2011 with an effective date of February 15, 2011.

I am independent of UEX Corporation as independence is described by Section 1.4 of NI 43-101.

I have not previously been involved with the Hidden Bay Project.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.
As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

ORIGINAL SIGNED

Bruce Murphy, FSAIMM

Dated: February 23, 2011

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CERTIFICATE OF QUALIFIED PERSON

Dino Pilotto, P.Eng.

I, Dino Pilotto, am a Professional Engineer, employed as a Principal Consultant - Mining with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled “Preliminary Assessment Technical Report on the Horseshoe and Raven Deposits, Hidden Bay Project, Saskatchewan, Canada” submitted on February 23, 2011 with an effective date of February 15, 2011.

I am a member of the Association of Professional Engineers and Geoscientists of Saskatchewan and Alberta. I graduated with a B.A.Sc. (Mining & Mineral Process Engineering) from the University of British Columbia in May 1987.

I have practiced my profession continuously since June 1987. I have been involved with mining operations, mine engineering and consulting covering a variety of commodities at locations in North America, South America, Eastern Europe, and Africa.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I have visited the Hidden Bay Project site in August 2010.

I am responsible for open pit mine engineering aspects of Sections 16.6, 18.7, 18.10, 21.1.2, and 21.2.2 of “the “Preliminary Assessment Technical Report on the Horseshoe and Raven Deposits, Hidden Bay Project, Saskatchewan, Canada” submitted on February 22, 2011 with an effective date of February 15, 2011.

I am independent of UEX Corporation as independence is described by Section 1.4 of NI 43-101.

I have not had prior involvement with the Hidden Bay Project.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.
As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

ORIGINAL SIGNED AND STAMPED

Dino Pilotto, P.Eng. Dated: February 23, 2011

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CERTIFICATE OF QUALIFIED PERSON

Mark Liskowich, P.Geo

I, Mark Liskowich, am a Professional Geologist, employed as a Principal Consultant – Geo-environment with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "Preliminary Assessment Technical Report on the Horseshoe and Raven Deposits, Hidden Bay Project, Saskatchewan, Canada" submitted on February 23, 2011 with an effective date of February 15, 2011.

I am a member of the Association of Professional Engineers and Geoscientists of Saskatchewan. I graduated with a B.Sc. (Geology) degree from the University of Regina in May 1989.

I have practised my profession within the mineral exploration, mining industry since 1989. I have been directly involved, professionally in the environmental and social management of mineral exploration and mining projects covering a wide range of commodities since 1989 with both the public and private sector. My areas of expertise are environmental management, environmental auditing, project permitting, licensing, public and regulatory consultation.

On the basis of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I did not visit the Hidden Bay Project site specifically for this report, however I have visited both the McClean Lake and Rabbit Lake mines/mills on several different occasions since 1992 and as recent as 2010. I have also had several recent visits to advanced exploration properties in the Athabasca Basin very similar to the Hidden Bay Project.

I am responsible for the Environmental Considerations (Chapter 20, 21.1.5, 21.1.6, 21.2.7) of the "Preliminary Assessment Technical Report on the Horseshoe and Raven Deposits, Hidden Bay Project, Saskatchewan, Canada" submitted on February 23, 2011 with an effective date of February 15, 2011.

I am independent of UEX Corporation, as independence is described by Section 1.4 of NI 43-101.
I have not previously been involved with the Hidden Bay Project.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

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CERTIFICATE OF QUALIFIED PERSON

Kevin Palmer, P.Geo.

I, Kevin Palmer, am a Professional Geologist, residing in Nanaimo, British Columbia, Canada.

This certificate applies to the technical report titled "Preliminary Assessment Technical Report on the Horseshoe and Raven Deposits, Hidden Bay Project, Saskatchewan, Canada" submitted on February 23, 2011 with an effective date of February 15, 2011.

I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (Licence #30020). I am also a member in good standing of The South African Council for Natural Science Professions (License #400320/04).

I graduated from University of the Witwatersrand, Johannesburg, South Africa with a B.Sc. (Honours) Geology in 1984.

I have practised my profession continuously since graduation.

My relevant experience with respect to Horseshoe and Raven Deposits includes over 21 years in exploration, mining geology and grade estimation in Canada and southern Africa. Over the last 4 years, I have carried out mineral resource estimates following CIM guidelines on a number of projects including the West Bear, Horseshoe and Raven Uranium Deposits in Northern Saskatchewan, Canada.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (N.I. 43-101).

I visited the Hidden Bay Project during the periods July 23 to 25, 2007 and July 10 to 11, 2008.

I am responsible for Sections 3 to 14, 16.1 to 16.5, 24.1 and 24.2 of the "Preliminary Assessment Technical Report on the Horseshoe and Raven Deposits, Hidden Bay Project, Saskatchewan, Canada" submitted on February 23, 2011 with an effective date of February 15, 2011.

I am independent of UEX Corporation as independence is described by Section 1.4 of N.I. 43-101.

I have previously carried out Mineral Resource estimates on the Horseshoe, Raven and West Bear Deposits on the Hidden Bay Property. All of the results of these estimates are contained in previously filed technical reports.

I have read National Instrument 43-101 and the sections for which I am responsible in this Technical Report have been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the sections of this Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

ORIGINAL SIGNED AND STAMPED

MELIS ENGINEERING LTD.

CONSENT OF QUALIFIED PERSON

TO: British Columbia Securities Commission
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TECHNICAL REPORT: *National Instrument 43-101 – Standards of Disclosure for Mineral Projects* ("**NI 43-101**") technical report entitled, "Preliminary Assessment Technical Report on the Horseshoe and Raven Deposits, Hidden Bay Project, Saskatchewan, Canada" dated February 23, 2011 (the "**Technical Report**").

I, Lawrence Melis, P.Eng., hereby grant this consent, pursuant to section 8.3 of NI 43-101.

I herby certify as follows:

I am a "Qualified Person" responsible for preparing Sections 15, 21.1.3 and 21.2.3 of the Technical Report and I hereby consent to the public filing of the Technical Report, specifically Sections 15, 21.1.3 and 21.2.3, with the above mentioned securities regulatory authorities. I further hereby consent to the use of extracts from, or a summary of, Sections 15, 21.1.3 and 21.2.3 of the Technical Report in written disclosure filed by the Issuer, including in a news release and material change report to disclose the information that is described in the Sections 15, 21.1.3 and 21.2.3 of the Technical Report.

I hereby confirm that I have read the written disclosure being filed and that it fairly and accurately represents the metallurgical information in Sections 15, 21.1.3 and 21.2.3 of the Technical Report that supports the disclosure.

Dated: February 23, 2011

____
Lawrence Melis, P.Eng.

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