TECHNICAL REPORT ON THE DENISON MINES INC. URANIUM PROPERTIES, SASKATCHEWAN, CANADA

PREPARED FOR DENISON MINES INC.

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INTRODUCTION

Roscoe Postle Associates Inc. ("RPA") was retained by Denison Mines Inc. ("Denison") in December 2004 to independently review and audit the Mineral Resources and Mineral Reserves of certain uranium deposits in the Athabasca Basin of northern Saskatchewan in which Denison holds an interest. This technical report was written by RPA in accordance with the requirements of National Instrument 43-101 (NI 43-101), Companion Policy 43-101CP, and Form 43-101F1 of the Ontario Securities Commission (OSC) and Canadian Securities Administrators (CSA).

Denison has a 22.5% interest in the McClean Lake Joint Venture ("the MLJV"). Cogema Resources Inc. ("Cogema") is the operator of the MLJV and owns an interest of 70.0%. Cogema is a wholly owned subsidiary of Cogema S.A., incorporated in France ("Cogema Group"), which in turn is a wholly owned subsidiary of Areva S.A, also incorporated in France.

The MLJV holds mineral claims and leases covering areas that host six uranium deposits including: Sue A, B, D, E, McClean North, and Caribou (collectively referred to as the McClean Lake property). The claims also include the mined-out JEB and Sue C deposits. Ore from these latter deposits are currently being processed from stockpiles.

The MLJV owns a uranium processing facility, the JEB mill, which has a nominal design of 6 million pounds of U_3O_8 per year. It was put into operation in 1999 to process ore of the now mined-out JEB and Sue C deposits. In 2001 the JEB mill received a four-year operating licence that increased its approved annual production capacity from six to eight million pounds U_3O_8. A mill expansion is planned to allow a further increase in annual capacity up to twelve million pounds U_3O_8 by 2006.

Definitive agreements were made in 2002 with the Cigar Lake Joint Venture (Cameco 50%, Cogema 37%) to process its ore at the JEB mill, with the pregnant aqueous solution to be processed at both the Rabbit Lake (Cameco) and MLJV facilities into uranium concentrates.

In 2002, exploration drilling at the MLJV property discovered uranium mineralization at Caribou Lake, located about three kilometres from the Sue C pit. The Caribou deposit occurs in sandstones at the unconformity with the basement rocks.

Denison also owns a 25.17% interest in the Midwest Joint Venture which includes the Midwest uranium deposit (the Midwest property). The latter is located near South McMahon Lake, about 20 kilometres by existing roads from the MLJV processing facilities. Subsequent to completion of a test-mining program in 1988 and 1989, the Midwest property has been under an environmental monitoring and site security surveillance program.

This technical report presents RPA's estimate of Mineral Resources and Mineral Reserves at the MLJV property only. The Midwest Joint Venture property has been reported on under a separate cover.

LAND STATUS

The MLJV surface lease, covering an area of 3,677 hectares, was granted by the Province of Saskatchewan in 1991. This lease was replaced by a new 33-year agreement in 2002. The mineral property consists of two mineral leases covering an area of 980 hectares and ten mineral claims covering an area of 3,250 hectares. The mineral leases are renewable on a 10-year basis; the next expiry date is in April 2006. Title to the mineral claims is secure until 2023.

The MLJV expects that all the leases will be renewed in the normal course, as required, to enable the McClean Lake property to be fully exploited.

EXPLORATION HISTORY

In 1974, Canadian Occidental Petroleum Limited ("Canadian Oxy") commenced uranium exploration in the area between the then known Rabbit Lake deposit and the Midwest property, where previously uraniferous boulder trains had been found. In 1977 a diamond drilling program was carried out in joint venture with Inco Ltd., and one of the 47 drilled holes encountered encouraging uranium mineralization. Extensive exploration work that followed discovered the McClean North deposit in 1979, the McClean South zone in 1980, and the JEB deposit in 1982. In January 1985, after a brief suspension of exploration, Minatco Limited ("Minatco"), a predecessor in title to Cogema, entered into the joint venture with CanadianOxy and Inco Ltd. Exploration resumed and as a result the Sue A deposit was found in 1988, followed by the Sue B and Sue C deposits. The Sue E deposit was discovered in late 1991.

The Caribou Lake pod, discovered in 2002, is not part of an existing mineralization trend and is regarded as a new area of mineralization within the overlying sandstones.

In 1993, the respective owners of McClean Lake properties and the Midwest property combined their interests to make one complementary project for processing ore through a single mill at McClean Lake. In order to accomplish this, a portion of Denison's interest in Midwest was exchanged for an interest in McClean Lake. A number of ownership changes took place between 1993 and 2004. Currently, Cogema is the operator of the joint venture with 70% ownership, and Denison having 22.5% ownership.

GEOLOGY AND MINERALIZATION

The MLJV uranium deposits lie near the eastern margin of the Athabasca basin in the Churchill Structural Province of the Canadian Shield. The bedrock geology of the area consists of Precambrian gneisses unconformably overlain by flat lying unmetamorphosed sandstones and conglomerates of the Athabasca Group. The Precambrian basement complex consists of an overlying Aphebian-aged supracrustal metasedimentary unit infolded into the older Archean gneisses. The younger Helikian-aged Athabasca sandstone was deposited onto this basement complex. The basement surface is marked by a paleoweathered zone with lateritic characteristics referred to as regolith.

Excluding the JEB deposit, which was mined out several years ago and is now used as the Tailings Management Facility, the MLJV deposits are located along two "trends" of mineralization, the McClean trend and the Sue trend. The recently discovered Caribou Lake pod is a singular deposit at this time.

The mineralized zones in the McClean trend occur as sausage shaped pods straddling the unconformity between the Athabasca sandstones and the crystalline basement. The mineralized pods undulate from 37 metres above to 37 metres below the unconformable contact which is on average 160 metres below the topographic surface in this area.

The mineralization is hosted by altered sandstones and Aphebian basement rocks usually altered to clay–rich rocks. A zone of illite alteration forms a mushroom shaped envelope tilted to the north in the McClean North zone. There are 11 discrete pods arranged along two separate but parallel trends (termed the North and South zones) separated by approximately 500 metres. Generally the mineralization in the basement is at the eastern extremity of the combined zone. Uranium mineralization is hosted in hematitically altered clay–rich zones containing massive layers of illite. Uranium occurs as fine–grained coffinite, veinlets and nodules of pitchblende, and massive pitchblende/uraninite. Associated with the uranium are highly variable but generally small amounts of nickel arsenides. Generally, the mineralization located below the unconformity is cleaner than that found in the sandstone.

The deposits of the Sue trend lie along a linear trend on the western flank of the Collins Bay dome. These deposits extend north andsouth along or near a steeply east dipping unit of graphitic gneiss within a 4.2 kilometre long basement conductor. The Sue A and Sue B deposits are located on and above the unconformity which lies 65 to 75 metres below the surface. The bulk of the mineralization occurs in the overlying sandstone. These deposits are typically hosted by massive earthy–red clay extending for about 10 metres above and below the unconformity. The mineralization at Sue A and Sue B is generally associated with niccolite and has an average ratio of Ni+As:U_3O_8 of 3.9:1 and 3.8:1 respectively.

The mined-out Sue C deposit lies 100 metres west of the south end of the Sue A deposit. The strike of the deposit trends south 12 degrees west for 390 metres and occupied a 75 degree east dipping structure. There was a distinct depth gradation to the mineralization of this deposit; with the mineralization subcropping at the unconformity in the northern and central part of the deposit and plunging gently south at the southern portion. The central part of the deposit, occupying a length of 80 to 100 metres, extended downwards from the unconformity for 80 metres and contained approximately 75% of the known reserves.

The Sue E deposit, although discovered in the early 1990s, did not undergo development drilling until 2002. The mineralization has an approximate strike length of 320 metres, with widths varying from 5 to 15 metres, and occurs at 65 to 135 metres below the surface. The style of mineralization and setting is similar to that of the southern part of the Sue C deposit in that it is totally basement-hosted. However the nickel and arsenic in the Sue E deposit are relatively high to the uranium content.

MINERAL RESOURCES

RPA has reviewed the current Cogema resource estimate for the Sue A deposit and has concluded that it conforms to the requirements of NI 43-101 and the definitions set out by the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by the CIM Council on August 20, 2000. Table 1-1 presents a summary of the Sue A Mineral Resources at various block cut-off grades. Based on RPA's review of U_3O_8 prices and mining operating costs, the 0.1% U_3O_8 cut-off grade is reasonable for conversion to Mineral Reserves.

TABLE 1-1 SUE A RESOURCE ESTIMATE

McClean Lake Joint Venture McClean Lake Property, Saskatchewan*

Cut-Off Grade U3O8%	Indicated Resource				
	Tonnes	U_3O_8%	Ni%	As%	U3O8 lbs
0.1%	**39,284**	**1.74**	**3.64**	**4.45**	**1,507,000**
0.2%	38,265	1.78	3.69	4.48	1,502,000
0.3%	37,504	1.81	3.76	4.56	1,491,000
0.4%	33,991	1.96	4.00	4.85	1,469,000
0.5%	31,928	2.06	4.15	5.04	1,450,000

*Denison Mines Inc. holds 22.5% interest in the MLJV and the above Resource

RPA has reviewed the current Cogema resource estimate for the Sue B deposit and after reclassifying portions of the deposit has restated the estimate as summarized in Table 1-2. RPA concludes that this estimate conforms to the requirements of NI 43-101 and the definitions set out by the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by the CIM Council on August 20, 2000. Table 1-2 presents a summary of the Sue B Mineral Resources at various block cut-off grades. Based on RPA's review of U_3O_8 prices and mining operating costs, the 0.1% U_3O_8 cut-off grade is reasonable for conversion to Mineral Reserves.

TABLE 1-2 SUE B RESOURCE ESTIMATE

McClean Lake Joint Venture McClean Lake Property, Saskatchewan*

Cut-Off Grade U_3O_8%	Indicated Resources Tonnes	U_3O_8%	U_3O_8 lbs	Inferred Resources Tonnes	U3O8%	U_3O_8 lbs
0.1%	**72,944**	**0.73%**	**1,174,000**	**12,041**	**0.95%**	**252,000**
0.2%	67,211	0.77%	1,141,000	9,854	1.11%	241,000
0.5%	45,506	0.97%	973,000	5,319	1.70%	199,000
1.0%	22,355	1.40%	690,000	2,758	2.68%	163,000

*Denison Mines Inc. holds 22.5% interest in the MLJV and the above Resources

RPA has re-estimated the resource estimate for the Sue E deposit based on the drilling and sampling database information provided by Cogema. The results of this re-estimation work are summarized in Table 1-3. The new RPA estimate of Indicated Resources represents a substantial increase over previous Indicated Resource estimates for Sue E. RPA has also estimated that there is a significant amount of Inferred Resource associated with the Sue E deposit. RPA concludes that these estimates conform to the requirements of NI 43-101 and the definitions set out by the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by the CIM Council on August 20, 2000. Table 1-3 presents a summary of the Sue E Mineral Resources at various block cut-off grades. Based on RPA's review of U_3O_8 prices and mining operating costs, the 0.1% U_3O_8 cut-off grade is reasonable for conversion to Mineral Reserves.

TABLE 1-3 SUE E RESOURCE ESTIMATE

McClean Lake Joint Venture McClean Lake Property, Saskatchewan*

Cutoff Grade U_3O_8%	Indicated Resources Tonnes	U_3O_8%	U_3O_8 lbs	Inferred Resources Tonnes	U_3O_8%	U_3O_8 lbs
0.1%	**718,285**	**0.78%**	**12,272,000**	**780,261**	**0.69%**	**11,783,000**
0.2%	547,630	0.97%	11,711,000	463,157	1.06%	10,813,000
0.5%	312,278	1.46%	10,024,000	209,737	1.96%	9,044,000
1.0%	151,671	2.23%	7,470,000	109,226	3.12%	7,506,000

*Denison Mines Inc. holds 22.5% interest in the MLJV and the above Resources

RPA has re-estimated the resource estimate for the McClean North deposits based on the drilling and sampling database information provided by Cogema. The results of this re-estimation work are summarized in Table 1-4. The new RPA estimate of Indicated Resources is similar to previous estimates of the McClean North deposit. The cut-off methodology applied by RPA is based on estimates of the economics associated with a Blind Shaft Boring mining method. RPA concludes that these estimates conform to the requirements of NI 43-101 and the definitions set out by the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by the CIM Council on August 20, 2000.

TABLE 1-4 MCCLEAN NORTH RESOURCE ESTIMATE

McClean Lake Joint Venture McClean Lake Property, Saskatchewan*

Indicated Resource

Pod	Tonnes	U_3O_8%	SG	Thickness (m)	U_3O_8 Tonnes	U_3O_8 lbs
Pod 1 East	20,683	9.68	2.39	7.5	2,002	4,414,000
Pod 1 West	8,287	3.77	2.27	8.8	313	690,000
Pod 2	22,154	3.85	2.28	15.0	852	1,879,000
Pod 5	5,804	5.81	2.31	7.6	337	743,000
Total	**56,928**	**6.16**	**2.32**	**10.6**	**3,504**	**7,726,000**

*Denison Mines Inc. holds 22.5% interest in the MLJV and the above Resource

RPA has reviewed the current Cogema resource estimate for the Caribou deposit and has concluded that the resource can be classified as Indicated. RPA concludes that this estimate conforms to the requirements of NI 43-101 and the definitions set out by the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by the CIM Council on August 20, 2000. . Table 1-5 presents a summary of the Caribou Mineral Resources at various block cut-off grades as estimated by Cogema and audited by RPA. Based on RPA's review of U_3O_8 prices and mining operating costs, the 0.1% U_3O_8 (0.85 kg/t U) cut-off grade is reasonable for conversion to Mineral Reserves.

TABLE 1-5 CARIBOU RESOURCE ESTIMATE

McClean Lake Joint Venture McClean Lake Property, Saskatchewan*

Cut-Off Grade		Indicated Resource			
U (kg/t)	**U_3O_8%****	**Tonnes**	**U_3O_8%****	**U_3O_8 lbs****	**U Metal (t)***
0.85	**0.10**	**47,763**	**2.62**	**2,758,816**	**1,063**
3.00	0.35	39,482	3.13	2,724,415	1,049
5.00	0.59	33,945	3.57	2,671,615	1,027
10.00	1.18	24,734	4.58	2,497,409	960
15.00	1.77	19,349	5.45	2,324,796	894
50.00	5.90	5,431	11.11	1,330,221	512

*Denison Mines Inc. holds 22.5% interest in the MLJV and the above Resource
**Restated by RPA for reference to U units used in estimate.

ENVIRONMENTAL STATUS

Roscoe Postle retained SENES Consultants Limited to review the current environmental aspects of the MLJV properties and operations in as far as these aspects could materially affect the potential for mining of the reserves. Briefly, the Sue A deposit is approved for development as an open pit mine. The Sue B deposit is approved for development as an open pit; however, current plans for this operation have been suspended. The Sue E deposit development as an open pit mine has been approved. No material issues have been identified in Environmental Assessment (EA) or EA review. Remote mining methods are being developed and evaluated at the McClean North deposits and the test program is being carried out under appropriate permits. Once the mining methods are established, the mine operating plan will have to be submitted for regulatory review and approvals. The Caribou deposit has not been evaluated and mining plans and environmental assessment has not been completed at this time.

All ore from the MLJV deposits will be processed at the JEB mill, which is being expanded to also process material from the Cigar Lake deposit. The JEB mill has processed all ore from the JEB open pit and is currently processing ores from Sue C pit. Extensive regulatory review has been completed for the management of tailings and waste rock from the MLJV and Midwest Projects. Contaminated waste rock is being disposed of in the disused Sue C pit, and all tailings from the milling of the Cigar, Midwest, and MLJV deposits are disposed of in the JEB tailings disposal facility. This tailings disposal facility can store all future production. Monitoring of the approved disposal facility has demonstrated that the facility is operating as designed.

Effluent treatment facilities are in place to manage all mine and mill effluents from the MLJV Lease. These plants are performing well and meet all regulatory discharge limits.

METALLURGY

The MLJV owns and operates the JEB mill. Operations started in 1999, and the mill has successfully been producing approximately six million pounds of U_3O_8 per year since then from JEB and Sue C ores. Going forward production plans include milling stockpiled Sue C ore, Sue A and E, McClean North and Midwest deposits.

In 2007, Denison plans for the JEB mill to start processing Cigar Lake joint venture (Cameco 50%, Cogema 37%) (CLMC) material concurrently with MLJV deposit ores. CLMC will pay a custom milling fee, and the overall JEB unit milling costs will be reduced by the economies of scale. The custom fee has not been included in this study, but the estimated milling costs reflect the benefits to be obtained through operating at the higher throughput rates that will be realized when Cigar Lake material is processed.

A feasibility study has been completed for custom milling of the Cigar Lake ore ("Cigar Lake Project, 2001 Feasibility Study Supporting Document No. 4A, JEB Mill Expansion", issued by Cogema and Cigar Lake Mining Corporation, April 2001), and the capital costs will be covered by CLMC.

The JEB mill flowsheet includes a grinding circuit designed to reduce the particle size of the ore materials whereupon the ground ore is fed to a leaching circuit where the uranium is leached from the ore in two circuits with sulphuric acid. After leaching the uranium into liquid solution the solids are separated from the solutions in a conventional thickener circuit.

A solvent extraction circuit recovers the uranium from the leach solution using an amine extractant in a kerosene organic solvent to extract the uranium into a pregnant strip solution from which it is precipitated in the form of yellowcake. The precipitate is dried and calcined and then packaged for transportation off site.

In 2004 the JEB processing facility achieved a uranium extraction recovery of over 97.3% .

Thickened tailings from the process are piped and deposited to the JEB tailings disposal pit, using a sub-aerial pervious surround tailings disposal system. The JEB tailings pit contains a dewatering drift and a raise to control the water levels, and a base filter to drain the tailings.

The mill was designed and is operated to meet all environmental and safety regulations. The employee exposure to radiation is well below regulatory limits.

There is an agreement between the MLJV and CLMC to partially custom mill Cigar Lake ores at the JEB mill. The Cigar Lake ore is planned to be ground at the Cigar Lake Mine and transported to the JEB mill as a pulp in specially designed, government approved containers. All of the Cigar Lake ore will be unloaded, stored, and leached at the JEB mill. The pregnant aqueous solution will be further processed at both JEB mill and Rabbit Lake Mill. The product capacity of the JEB mill will be increased from a nominal 6 million pounds of U_3O_8 per year to 12 million pounds of U_3O_8. The JEB mill will require modification and expansion to be able to treat the Cigar Lake ore.

RPA has reviewed the various metallurgical test work results associated with the various deposits under consideration in this report and has concluded that these ores will be amenable to treatment in the JEB processing facilities. Certain process modifications are planned to be implemented to treat those deposits that contain elevated levels of Arsenic. Similarly additional process modifications and additions are planned for the JEB facility in order to recover Nickel and Cobalt values that are prevalent in some of the deposits reviewed in this report.

MINING AND PROCESSING OPERATIONS

Mine development plans for the deposits under review in this report are predominantly based on open pit mining methods and equipment. The mining operations that have been carried out by the MLJV to date have all been based on open pit method, and an existing fleet of mining equipment is in place. For the most part this same equipment fleet will be utilized to develop the new deposits. The only area where other mining methods are currently under consideration is at the McClean North deposit. In this case the mineralized zones are small high grade pods that lie under relatively deep cover. Their development as open pits is unattractive due to the amount of barren waste rock that would have to be excavated in order to access the ore zones. As an alternative, the MLJV is currently carrying out a mining test program where a borehole is being drilled vertically down to the mineralization and hydraulic jet boring and mechanical reaming methods are going to be tested. RPA has reviewed the various studies and analyses developed around these concepts. RPA has chosen to evaluate the economics of the McClean North deposits based on the Blind Shaft Boring method.

MINERAL RESERVES

The Mineral Reserve at Sue A has been calculated based on the RPA resource model and the Denison ultimate pit design, and is summarized in Table 1-6. On the basis of the estimates and forecasts presented, RPA concludes that the Mineral Reserves are consistent with the definitions set out in NI 43-101 and defined by the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by the CIM Council on August 20, 2000.

TABLE 1-6 SUE A PROBABLE RESERVE (AS OF JAN.1, 2005)

McClean Lake Joint Venture McClean Lake Property, Saskatchewan*

Total Material (BCM)	Waste (BCM)	Special Waste (BCM)	Ore (Tonnes)	U_3O_8 Grade (%)	U_3O_8 lbs
947,103	914,568	19,308	31,948	1.99%	1,402,000

*Denison Mines Inc. holds 22.5% interest in the MLJV and the above Reserve

The Mineral Reserve at Sue E has been estimated using an open pit design developed by RPA and based on an economic optimization analysis. The Mineral Reserve estimate is summarized in Table 1-7. On the basis of the estimates and forecasts presented, RPA concludes that the Mineral Reserves are consistent with the definitions set out in NI 43-101 and defined by the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by the CIM Council on August 20, 2000.

TABLE 1-7 SUE E PROBABLE RESERVE (AS OF JAN.1, 2005)
McClean Lake Joint Venture McClean Lake Property, Saskatchewan*

Total Material (BCM)	Waste (BCM)	Special Waste (BCM)	Ore (Tonnes)	U_3O_8 Grade (%)	Nickel Grade (%)	U_3O_8 lbs
5,459,025	5,082,581	114,173	628,077	0.78%	0.53%	10,799,000

*Denison Mines Inc. holds 22.5% interest in the MLJV and the above Reserve

The Mineral Reserve at McClean North has been estimated based on Blind Shaft Boring mining methods reviewed by RPA and based on an economic analysis of cost and revenue estimates. The Mineral Reserve estimate is summarized in Table 1-8. On the basis of the estimates and forecasts presented, RPA concludes that the Mineral Reserves are consistent with the definitions set out in NI 43-101 and defined by the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by the CIM Council on August 20, 2000.

TABLE 1-8 MCCLEAN NORTH PROBABLE RESERVE

McClean Lake Joint Venture McClean Lake Property, Saskatchewan*

Pod	Tonnes Ore	Grade U_3O_8%	U_3O_8 tonnes	U_30_8 lbs
Pod 1**	19,092	8.68	1,657	3,654,000
Pod 2	16,048	3.54	568	1,253,000
Pod 5	3,916	4.85	190	419,000
Total	39,056	6.19	2,416	5,326,000

*Denison Mines Inc. holds 22.5% interest in the MLJV and the above Reserve
**Includes Pod1E and Pod 1W

At year end 2004 the ore stockpile at the JEB mill (consisting primarily of Sue C ore materials) is estimated to contain 268,000 tonnes carrying an average grade of 1.39% U_3O_8, or a total of 8,213,000 lbs U_3O_8. This material is classified as Proven Mineral Reserve consistent with the definitions set out in NI 43-101.

ECONOMIC ANALYSIS

The operating and development projects designed to recover the various Mineral Reserves outlined above are planned to be sequentially developed in order to sustain the ongoing ore processing and uranium production operations at the existing JEB mill facilities. In addition to the Sue A and Sue E open pits and the McClean North blind shaft boring production, the Midwest project is schedule for development with ore production forecast for 20101. The Midwest deposit and associated Mineral Resources and Reserves are described in detail in a report entitled "Technical Report on the Midwest Uranium Deposit and Mineral Resource and Mineral Reserve Estimates Saskatchewan, Canada Prepared for Denison Mines Inc. June 2005". Certain information and data have been extracted from that report for inclusion in the overall MLJV production schedule presented here. In addition, the MLJV plans to process production materials from the Cigar Lake operations. The Cigar Lake processing plans are considered here only to the extent that those quantities impact on the plans for treating the McClean and Midwest ores; however, the cost and revenue forecasts do not include any values corresponding to Cigar Lake production. The mine production schedule is summarized in Table 1-9.

Based on the available mill feed material from the mining schedule, RPA has developed an Operations Cashflow estimate using the combined estimates and projections associated with the various mine development projects and production schedules outlined in the sections above. The production schedule has been developed considering both the various sources of ore feed from the MLJV mines and the projected uranium processing schedule for the Cigar Lake Joint Venture (CLJV) material that is planned to be treated at the JEB facilities. However, the projected revenues, operating costs, and capital costs are only those associated with the MLJV operations excluding any revenue and costs associated with the CLJV.

TABLE 1-9 MCCLEAN LAKE JV AND MIDWEST LAKE JV MINING SCHEDULE

			2005	2006	2007	2008	2009	2010	2011	Total
Sue A *										
Total BCM	BCM	947,103	947,103	-	-					947,103
Overburden	BCM	-	-	-	-					-
Rock Waste	BCM	914,568	914,568	-	-					914,568
Special Waste	BCM	19,308	19,308	-	-					19,308
Ore	BCM	13,227	13,227	-	-					13,227
Tonnes Ore	t	31,948	31,948	-	-					31,948
Grade	U_3O_8 %	1.99%	1.99%	0.00%	0.00%					0.00%
Tonnes U	t	635	635	-	-					24,716
U_3O_8 lbs		1,400,199	1,400,199	-	-					0
										0
McClean North *										0
Tonnes Ore	t	39,056			8,293	8,293	10,259	10,402	1,809	39,056
Grade	U_3O_8 %	6.19%			8.68%	8.68%	4.80%	3.81%	4.85%	6.09%
Tonnes U	t	2,048			610	610	417	336	74	2,048
U_3O_8 lbs		5,325,410			1,587,075	1,587,075	1,084,706	872,966	193,588	5,325,410
Sue E *										
Total BCM	BCM	5,459,025	3,722,321	1,540,580	196,124					5,459,025
Overburden	BCM	-	-	-	-					-
Rock Waste	BCM	5,082,581	3,722,321	1,290,326	69,934					5,082,581
Special Waste	BCM	114,173	-	90,243	23,930					114,173
Ore	BCM	262,272	-	160,012	102,260					262,272
Tonnes Ore	t	628,077	-	382,165	245,911					628,077
U_3O_8 Grade	U_3O_8 %	0.78%	0.00%	0.34%	1.16%					0.66%
Ni Grade	Ni%	0.53%	0.00%	0.27%	0.94%					0.53%
		0.871	0.00%	0.44%	1.55%					
Tonnes U	t	4,151	-	1,102	2,418					3,520
U_3O_8 lbs		10,791,809	-	2,865,429	6,286,026					9,151,454
Ni lbs		7,381,894	-	2,296,230	5,085,663					7,381,894
As lbs		12,062,381	-	3,684,908	8,377,472					
Midwest **										
Total BCM	BCM	44,593,096					22,640,627	21,952,469		44,593,096
Overburden	BCM	2,480,282					2,480,282	-		2,480,282
Rock Waste	BCM	40,457,910					20,137,001	20,320,909		40,457,910
Special Waste	BCM	1,509,656					23,344	1,486,312		1,509,656
Ore	BCM	145,248					-	145,248		145,248
Tonnes Ore	t	345,516					-	345,516		345,516
U_3O_8 Grade	U_3O_8 %	5.47%					-	5.47		4.64%
Ni Grade	Ni %	4.37%					-	4.37		4.37%
Co Grade	Co %	0.33%					-	0.33		0.33%
As Grade	As%	7.20%					-	7.20		
Tonnes U	t	16,013					-	16,013		16,013
U_3O_8 lbs		35,302,555					-	35,302,555		35,302,555
Ni lbs		33,300,163					-	33,300,163		33,300,163
Co lbs		2,544,732					-	2,544,732		2,544,732
As lbs		54,830,580					-	54,830,580		
Total ***										
Total BCM	BCM	50,999,224	4,669,424	1,540,580	196,124	-	22,640,627	21,952,469	-	50,999,224
Overburden	BCM	2,480,282	-	-	-	-	2,480,282	-	-	2,480,282
Rock Waste	BCM	46,455,059	4,636,889	1,290,326	69,934	-	20,137,001	20,320,909	-	46,455,059
Special Waste	BCM	1,643,137	19,308	90,243	23,930	-	23,344	1,486,312	-	1,643,137
Ore	BCM	420,747	13,227	160,012	102,260	-	-	145,248	-	420,747
Tonnes Ore	t	1,044,596	31,948	382,165	254,204	8,293	10,259	355,918	1,809	1,044,596
U Grade	U_3O_8 %	2.29%	1.99%	0.53%	1.40%	8.68%	4.80%	4.61%	4.85%	2.29%
Ni Grade	Ni%	1.77%	0.00%	0.27%	0.91%	0.00%	0.00%	4.24%	0.00%	1.77%
Co Grade	Co%	0.11%	0.00%	0.00%	0.00%	0.00%	0.00%	0.32%	0.00%	0.11%
Tonnes U	t	22,848	635	1,102	3,028	610	417	16,349	74	22,217
U_3O_8 lbs		52,819,974	1,400,199	4,452,503	7,873,101	1,587,075	1,084,706	36,175,521	193,588	52,766,693
Ni lbs		40,682,056	-	2,296,230	5,085,663	-	-	33,300,163	-	40,682,056
Co lbs		2,544,732	-	-	-	-	-	2,544,732	-	2,544,732

* Denison Holds 22.50% Interest in the MLJV Mineral Production.

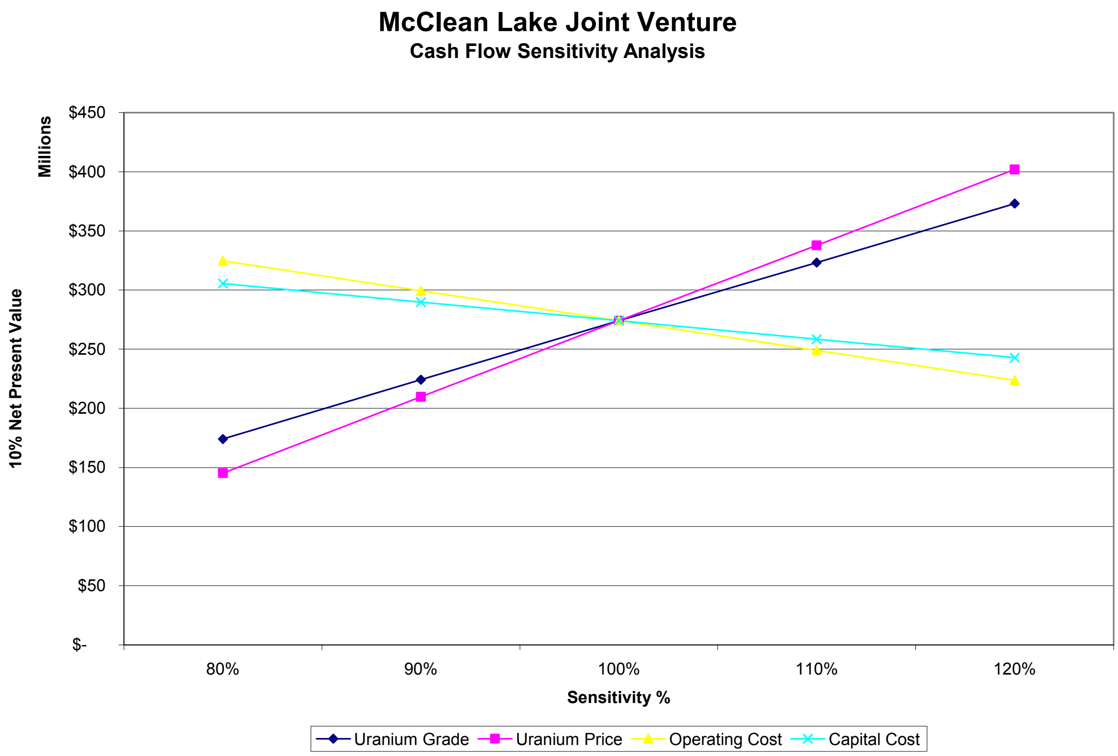
** Denison Holds 25.17% Interest in the Midwest Lake Mineral Production.

Table 1-10 presents a summary of the operating plan and cash flow based on a 16 year operating life. The cash flow is on a pre-income tax basis as the corporate entities involved in the joint venture have different tax pools and tax positions. Since the cash flow represents an ongoing operating entity and there are no net capital investments or negative cash flows in the initial years, an internal rate of return factor cannot be calculated.

SENSITIVITIES

RPA developed a sensitivity analysis for the cash flow estimate presented in Table 1-10 where the impact of changes to uranium grade, capital cost, operating cost, and uranium price was determined. The results of these sensitivities are illustrated in Figure 1-1.

FIGURE 1-1 MCCLEAN AND MIDWEST CASHFLOW NPV SENSITIVITY ANALYSIS



Source: RPA

Table 1-10 MCCLEAN LAKE OPERATIONS CASH FLOW ESTIMATE

	2005	2006	2007	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	Total
Production																	
Recovered U3O8 lbs	5,299	4,010	4,687	3,006	2,513	5,115	4,594	4,053	4,254	4,017	3,545	3,545	3,545	3,545	3,545	3,545	62,816
Recovered Ni lbs	-	-	1,396	644	644	1,856	1,948	1,789	1,878	1,773	1,565	1,565	1,565	1,565	1,565	1,565	21,316
Recovered Co lbs	-	-	-	-	-	93	100	136	143	136	120	120	120	120	120	120	1,324
Revenue																	
Net U3O8 Revenue FOB Minesite	$ 151,287	$ 114,493	$ 133,805	$ 85,812	$ 71,756	$ 146,021	$ 131,157	$ 115,713	$ 121,441	$ 114,694	$ 101,201	$ 101,201	$ 101,201	$ 101,201	$ 101,201	$ 101,201	$ 1,793,383
Net Ni Revenue FOB Minesite	$ -	$ -	$ 3,567	$ 1,645	$ 1,645	$ 4,743	$ 4,977	$ 4,572	$ 4,798	$ 4,532	$ 3,998	$ 3,998	$ 3,998	$ 3,998	$ 3,998	$ 3,998	$ 54,469
Net Co Revenue FOB Minesite	$ -	$ -	$ -	$ -	$ -	$ 641	$ 690	$ 938	$ 993	$ 938	$ 828	$ 828	$ 828	$ 828	$ 828	$ 828	$ 9,166
Total Revenue before Royalty	$ 151,287	$ 114,493	$ 137,372	$ 87,457	$ 73,401	$ 151,406	$ 136,824	$ 121,223	$ 127,232	$ 120,164	$ 106,027	$ 106,027	$ 106,027	$ 106,027	$ 106,027	$ 106,027	$ 1,857,018
Royalties																	
Midwest Royalty	$ -	$ -	$ -	$ -	$ -	$ 642	$ 405	$ 551	$ 583	$ 551	$ 486	$ 486	$ 486	$ 486	$ 486	$ 486	$ 5,645
Net Revenue after Royalty	$ 151,287	$ 114,493	$ 137,372	$ 87,457	$ 73,401	$ 150,764	$ 136,419	$ 120,673	$ 126,649	$ 119,613	$ 105,541	$ 105,541	$ 105,541	$ 105,541	$ 105,541	$ 105,541	$ 1,851,373
Operating Costs																	
Total Mining	$ 19,145	$ 25,459	$ 19,946	$ 16,358	$ 106,821	$ 120,865	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 308,593
Uranium Process	$ 32,006	$ 24,222	$ 28,308	$ 18,154	$ 10,958	$ 22,300	$ 20,030	$ 17,671	$ 18,546	$ 17,515	$ 15,455	$ 15,455	$ 15,455	$ 15,455	$ 15,455	$ 15,455	$ 302,438
Nickel/Cobalt Process	$ -	$ -	$ 1,689	$ 779	$ 779	$ 2,246	$ 2,357	$ 2,165	$ 2,272	$ 2,146	$ 1,893	$ 1,893	$ 1,893	$ 1,893	$ 1,893	$ 1,893	$ 25,792
Total Operating Cost	$ 51,151	$ 49,681	$ 49,943	$ 35,291	$ 118,558	$ 145,410	$ 22,386	$ 19,836	$ 20,818	$ 19,661	$ 17,348	$ 17,348	$ 17,348	$ 17,348	$ 17,348	$ 17,348	$ 636,824
less Capitalized Production Cost	$ -	$ -	$ -	$ -	$ 91,578	$ 64,700	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 156,278
Net Operating Cost	$ 51,151	$ 49,681	$ 49,943	$ 35,291	$ 26,980	$ 80,710	$ 22,386	$ 19,836	$ 20,818	$ 19,661	$ 17,348	$ 17,348	$ 17,348	$ 17,348	$ 17,348	$ 17,348	$ 480,545
Operating Cost/lb U3O8	$ 9.65	$ 12.39	$ 10.66	$ 11.74	$ 10.73	$ 15.78	$ 4.87	$ 4.89	$ 4.89	$ 4.89	$ 4.89	$ 4.89	$ 4.89	$ 4.89	$ 4.89	$ 4.89	$ 7.65
Capital Costs																	
McClean North Project	$ -	$ 18,222	$ -	$ -	$ -	$ -	$ 1,330	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 19,552
Midwest Project Mine Capital	$ -	$ -	$ -	$ -	$ 75,400	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 75,400
Midwest Project Mill Capital	$ -	$ -	$ -	$ -	$ 27,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 27,000
Midwest Pre-Stripping Capital	$ -	$ -	$ -	$ -	$ 91,578	$ 64,700	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 156,278
Total Capital Cost	$ -	$ 18,222	$ -	$ -	$ 193,978	$ 64,700	$ 1,330	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 278,231
Mining Taxes	$ 34,191	$ 25,875	$ 30,547	$ 19,535	$ 5,595	$ 11,538	$ 10,982	$ 24,830	$ 27,923	$ 26,371	$ 23,269	$ 23,269	$ 23,269	$ 23,269	$ 23,269	$ 23,269	$ 357,000
Project Cash Flow (Pre Income Tax)	$ 65,945	$ 20,715	$ 56,883	$ 32,631	-$ 153,152	-$ 6,184	$ 101,721	$ 76,007	$ 77,909	$ 73,580	$ 64,924	$ 64,924	$ 64,924	$ 64,924	$ 64,924	$ 64,924	$ 735,597
10% NPV	$ 274,173																

INTERPRETATION AND CONCLUSIONS

The MLJV projects outlined in this report represent significant economic sources of feed materials for the existing JEB processing facilities and, in conjunction with the Midwest Project described under separate cover, will support an operating life of at least 15 years, producing in the order of 62.8 million pounds of U_3O_8 product. At the US$23.00 per pound uranium price used in the economic analysis in this report, these projects are estimated to produce substantial positive operating cash flows.

Although there is a substantial volume of data and information available for the various deposits under consideration in this report, RPA found that the information provided needed a significant amount of organizing, checking, and clarification. RPA spent a considerable amount of time and effort in the verification and confirmation process in order to confidently develop the estimates of Mineral Resources and Mineral Reserves outlined in this report. While no fatal flaws were uncovered in this process, RPA recommends that the MLJV implement more rigorous controls and procedures in the area of resources and reserves estimation and documentation.

RPA has found that there has been a significant under-estimation of uranium resources and reserves in some of the estimates prepared in the past for the MLJV due to the use of simple grade interpolation methods. RPA has evaluated and used a density weighted grade interpolation methodology that recognizes the importance of the heavy specific gravity associated with high grade uranium minerals. RPA believes that the estimates developed and presented here are better representations of the likely conditions in the deposits and RPA recommends that these methods and procedures be adopted in future Mineral Resource and Reserve estimates for the MLJV.

RPA has found that the resource modeling methodologies used in some of the past estimates based on Uniform Conditioning are difficult to check and confirm by physical examination and validation. RPA believes that the technical and operational staff will find it necessary to have physical representations and interpretations of the geology and mineralization in the deposit in order to effectively manage the mining process. The uniform conditioning methods do not rely upon and do not produce the sort of graphical interpretations necessary for mine planning. While the methodologies may be mathematically correct, they are difficult to use in a practical context. RPA recommends that modeling and estimation programs that will ultimately be employed to support mining operations be carried out using more physically interpretive methods along the lines of the methods used by RPA in developing some of the estimates in this report.

RPA has estimated that the Sue E deposit hosts a significant amount of Inferred Mineral Resource. RPA believes that while this material has not been used in the economic analysis and determination of the Mineral Reserve for Sue E, it does represent potentially economic material. RPA recommends that additional diamond drilling be carried out in order to confirm the presence of this additional mineralization and provide the data necessary to upgrade is classification to Indicated.

2 INTRODUCTION AND TERMS OF REFERENCE

Roscoe Postle Associates Inc. (RPA) was retained by Denison Mines Inc. (Denison) to independently review and audit the Mineral Resources and Mineral Reserves of certain uranium deposits in the Athabasca Basin of northern Saskatchewan in which Denison holds an interest. This technical report was written by RPA in accordance with the requirements of National Instrument 43-101 (NI 43-101), Companion Policy 43-101CP, and Form 43-101F1 of the Ontario Securities Commission (OSC) and Canadian Securities Administrators (CSA).

Denison holds a 22.5% interest in the McClean Joint Venture (MLJV). Cogema Resources Inc., a wholly-owned subsidiary of AREVA, a multinational French government agency, is the operator of the MLJV and holds a 70.0% interest.

The MLJV holds mineral claims and leases covering the areas that host six uranium deposits including: the Sue A, B, D, E, McClean North, and Caribou (all referred to as the McClean Lake property). The claims also include the mined-out JEB and Sue C deposits, ores from which are currently being processed from stockpiles. Reserve reports for the stockpiles are also presented here.

The MLJV owns a uranium processing facility, the JEB Mill, which has a nominal design capacity of 6 million pounds of U_3O_8 per year. It was put into operation in 1999 to process ore from the now mined-out JEB and Sue C deposits. In 2001, the JEB Mill received a four-year operating licence that permits increased annual production from six to eight million pounds U_3O_8. A mill expansion is planned to allow a further increase in annual capacity up to twelve million pounds U_3O_8 by 2006.

Denison also owns a 25.17% interest in the Midwest Joint Venture which includes the Midwest uranium deposit (the Midwest property). The latter is located near South

McMahon Lake, about 20 kilometres by existing roads from the McClean Lake processing facilities. Subsequent to completion of a test-mining program in 1988 and 1989, the Midwest property has been under an environmental monitoring and site security surveillance program.

This technical report presents RPA's estimate of Mineral Resources and Mineral Reserves at the McClean Lake property only. The Midwest property is reported under a separate cover.

The principal technical documents and files related to the McClean Lake uranium deposits are as follows:

- Report on Reserves and Resources of Denison Energy Inc. McClean Lake and Midwest area, Saskatchewan, by William C. Kerr, P.Geo., Joe Spiteri, P.Geo., Gary A Cohoon, P.Geo., H.C. Counsell, P.Eng., and Andrew C. Rickaby. September 15, 2003.

- McClean North Uranium Deposit, Report on Reserves based on Pre-feasibility Study development using Hydraulic Borehole Mining Method, by Denison Energy Inc., Andrew C. Rickaby, William C. Kerr, Gary A. Cohoon. November 29, 2003.

- Denison Mines Annual Information Form for the fiscal year ending December 31, 2004.

Work on this project was completed by RPA Principal Mining Engineer James Hendry, P.Eng., and RPA Consulting Geologist Richard Routledge, MSc., P.Geol.

Mr. Hendry and Mr. Routledge are Qualified Persons in accordance with the requirements of NI 43-101. Mr. Hendry and Mr. Routledge visited the McClean Lake mine site on February 1 and 2, 2005, and the Cogema exploration office in Saskatoon on January 31, 2005 and February 2 to 5, 2005. Mr. Routledge also held further discussions on April 6, 2005 with Cogema resource estimation personnel at their office in Vélizy Cedex near Paris, France. RPA Consulting Geologist Davis Ross, M.Sc., P.Geo. collected additional data and reports from Cogema in Saskatoon from July 19 to 23, 2005.

Technical documents and reports on the property were reviewed at the site and additional information was obtained from the Denison and Cogema personnel. Discussions were held with technical personnel as follows:

Jim Corman, Mine Manager, McClean Lake, Saskatchewan;

Mike Eade, Chief Engineer, McClean Lake, Saskatchewan;

Bill Dodds, Mine Superintendent, McClean Lake, Saskatchewan;

William Kerr, Director, Resource Evaluation, Denison Mines Inc.;

Steve Wilson, Chief Mine Geologist, McClean Lake, Saskatchewan;

Sylvain Eckert, Manager, Mine Products, Cogema Resources Inc., Saskatoon;

Laure Fontaine, Resource Geologist, Service de Reserves, Cogema BUM/DT, France;

Olivier Masset, Resource Geologist, Service de Reserves, Cogema BUM/DT, France;

Julien Conté, Resource Geologist, Service de Reserves, Cogema BUM/DT, France.

RPA would like to acknowledge the co-operation and assistance that has been provided by Denison and Cogema personnel.

3 LIST OF ABBREVIATIONS

In this report, monetary units are Canadian dollars (US$) unless otherwise specified. The metric system (SI) of measurements and units has been used unless otherwise specified. Tables showing abbreviations used in this report are provided below:

TABLE 3-1 STANDARD LIST OF ABBREVIATIONS
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Abbr.	Meaning	Abbr.	Meaning
μ	micro (one-millionth)	km^2	square kilometre
°C	degree Celsius	kPa	kilopascal
°F	degree Fahrenheit	kVA	kilovolt-amperes
μg	microgram	kW	kilowatt
A	ampere	kWh	kilowatt-hour
a	annum	l	liter
CFM	cubic feet per minute	l/s	litres per second
bbl	barrels	m	metre
Btu	British thermal units	M	mega (million)
C$	Canadian dollars	m^2	square metre
cal	calorie	m^3	cubic metre
cm	centimetre	min	minute
cm^2	square centimetre	masl	metres above sea level
d	day	mm	millimeter
dia.	diameter	mph	mile per hour
dmt	dry metric tonne	MVA	megavolt-amperes
dwt	dead-weight ton	MW	Megawatt
ft	foot	MWh	megawatt-hour
ft/s	foot per second	m^3/h	cubic metres per hour
ft^2	square foot	opt, oz/st	ounce per short ton
ft^3	cubic foot	oz	troy ounce (31.1035g)
g	gram	oz/dmt	ounce per dry metric tonne
G	giga (billion)	ppt/ppm/ppb	part per thousand/per million/per billion
gal	Imperial gallon	psia	pound per square inch absolute
g/l	gram per litre	psig	pound per square inch gauge
g/t	gram per tonne	s	second
gpm	Imperial gallons per minute	st	short ton
gr/ft3	grain per cubic foot	stpa	short ton per year
gr/m3	grain per cubic metre	stpd	short ton per day
hr	hour	t	metric tonne
ha	hectare	tpa	metric tonne per year
hp	horsepower	tpd	metric tonne per day
in	inch	US$	United States dollar
in^2	square inch	USg	United States gallon
j	joule	USgpm	US gallon per minute
k	kilo (thousand)	V	volt
kcal	kilocalorie	W	Watt
kg	kilogram	wmt	wet metric tonne
km	kilometre	yd^3	cubic yard
km/h	kilometre per hour	yr	year

TABLE 3-2 SUPPLEMENTARY LIST OF ABBREVIATIONS
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Abbreviation	Meaning
As	Arsenic
Co	Cobalt
Mg	Magnesium
Ni	Nickel
U	Uranium
U_3O_8	Uranium oxide
Ukg/t	Uranium grade in kg/tonne (or ppt)
m.v.	Million years
0_2	Oxygen
e.m.f.	Electromotive force
C.C.D. circuit	Counter current decantation
SAG	Semi autogenous grinding
SX	Solvent extraction
HVAC	Heating, ventilation, air conditioning

4 QUALIFICATIONS

Roscoe Postle Associates Inc. (RPA) is an independent firm of Geological and Mining Consultants based in Toronto with an office in Vancouver. Since its establishment in 1985, RPA has carried out consulting assignments for nearly five hundred clients, including major mining companies, junior mining and exploration companies, financial institutions, governments, law firms and individual investors. Our clients are principally Canadian, American, and European companies.

RPA's business primarily involves providing independent opinions on mineral resources and reserves, technical aspects and economics of mining projects, valuation of mining and exploration properties and scoping, prefeasibility, and feasibility studies. RPA has completed assignments on projects located in all parts of Canada, the United States, Russia, Latin America, Australia, and in other countries in Europe, Africa and Asia.

RPA has completed several hundred assignments related to Mineral Resource or Reserve estimates and audits. RPA has also audited a number of Feasibility Studies and carried out many due diligence and project monitoring assignments for chartered North American and European banks. RPA has participated in a number of Feasibility Studies with Hatch Associates Ltd. (Hatch) and other major international consulting engineering firms.

RPA has extensive experience with uranium deposits including resource and reserve reviews, audits and estimates, QA/QC reviews, database validation assignments for operating mines, and qualifying reports. Details on RPA's qualifications, services, clients, and types of assignments are available on RPA's website (www.rpacan.com).

5 DISCLAIMER

This report has been prepared by RPA for Denison. RPA has not verified the mineral land titles or the status of ownership. RPA has relied on mineral land title information as provided by Denison and Cogema. The information, conclusions, and estimates contained herein are based on:

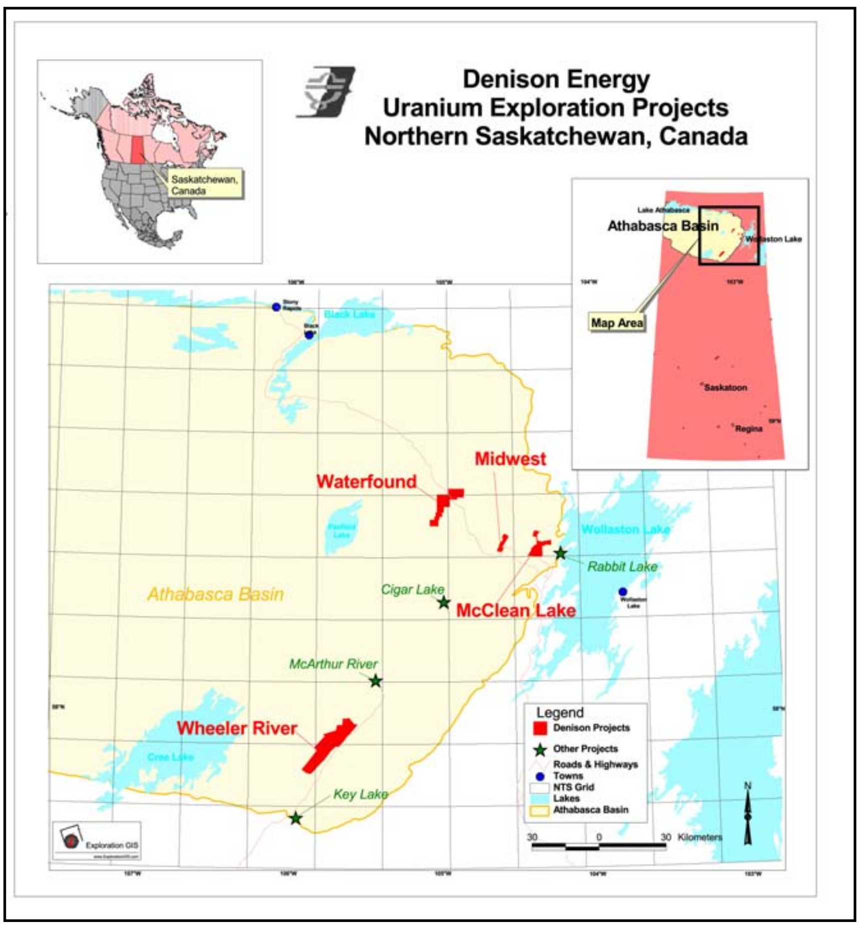
- Information available to RPA at the time of preparation of this report,
- Assumptions, conditions and qualifications as set forth in this report, and
- Data, reports, and opinions supplied by Denison and Cogema and other third party sources.

6 PROPERTY DESCRIPTION AND LOCATION

PROPERTY LOCATION

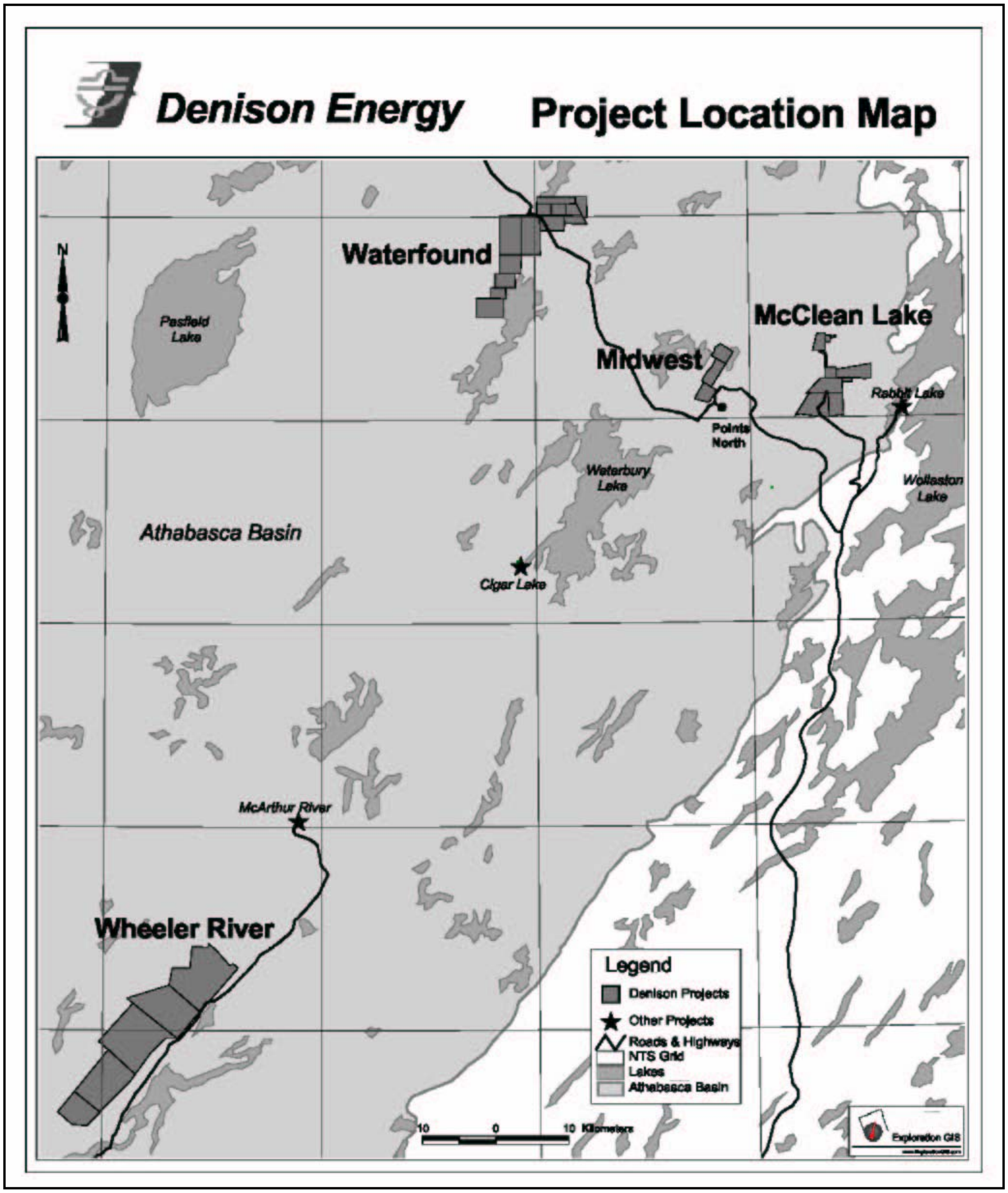
The McClean Lake property is located in northern Saskatchewan at longitude 103º 53'W and latitude 58º 15'N (Figures 6-1 and 6-2). The property, including the JEB mill, is located about 26 kilometres by road west of the Rabbit Lake mine and approximately 750 kilometres by air north of Saskatoon (Figure 6-1).

FIGURE 6-1 LOCATION MAP, DENISON URANIUM PROJECTS, NORTHERN SASKATCHEWAN



Source: Denison Mines Ltd

FIGURE 6-2 DENISON URANIUM PROJECTS IN THE ATHABASCA BASIN



Source: Denison Mines Ltd

CLAIMS STATUS

The McClean Lake property covers an area hosting the Sue A, B, C, D, and E, the McClean North, and the JEB uranium deposits as well as other prospects. Two of these deposits, JEB and Sue C, have been mined-out and the ore, which was stockpiled on surface, is currently being processed. The mined out JEB pit has been converted into the JEB Tailings Management Facility designed to receive tailings from the McClean Lake ores as well as the Midwest Project and Cigar Lake ores. Special low-grade uranium-bearing waste ("special waste[1]") from the McClean Lake and Midwest deposits will be placed in the mined-out Sue C pit. Agreement has been reached for the Cigar Lake special-waste to be deposited in that pit as well.

The JEB Mill consists of a modern mill licensed to produce 8.0 million pounds of uranium concentrate per year, a sulphuric acid plant, warehouses, shops, offices, and living accommodations for site personnel, together with related infrastructure. The JEB Mill is currently operating at a rate of approximately 6 million pounds per year of U_3O_8 to fulfil existing contracts and to optimize stockpile throughput.

All of the surface facilities and the mine sites are located on lands owned by the Province of Saskatchewan. The right to use and occupy the lands was granted in a surface lease agreement with the Province of Saskatchewan. The original surface lease covering an area of approximately 3,677 hectares and granted in 1991 was replaced by a new agreement in 2002 valid for a period of 33 years. Obligations under the surface lease agreement primarily relate to annual reporting regarding the status of the environment, the land development and progress made on northern employment and business development.

The McClean Lake Property consists of two mineral leases covering an area of 980 hectares and ten mineral claims covering an area of 3,250 hectares. The right to mine the McClean Lake deposits was acquired under these mineral leases, as renewed from time to

[1] Special waste is material which is below cut-off (usually about 0.085 %U, 0.1% U_3O_8,) but which does contain uranium mineralization grading greater than 0.025% U and which requires special disposal.

time. The mineral leases are valid for 10 years with the right to renew for successive 10-year periods, provided that the leaseholders are not in default pursuant to the terms of the lease. The terms of the two mineral leases expire in April 2006. It is expected that the leases will be renewed as required to enable the McClean Lake deposits to be fully exploited. Title to the mineral claims is secure until 2023.

The uranium produced from the McClean Lake deposits are subject to Saskatchewan uranium royalties under the terms of Part III of the Crown Mineral Royalty Schedule, 1986 (Saskatchewan), as amended.

ENVIRONMENTAL AND PERMITTING STATUS

The McClean Lake property is subject to decommissioning liabilities. Cogema, the operator, filed a conceptual decommissioning plan with the Saskatchewan government. Financial assurances are in place for the total amount of $35.0 million to cover the estimated costs of this decommissioning work. MLJV has filed an updated decommission plan with the regulatory bodies, with estimated decommissioning costs reduced to $29 million.

The McClean Lake site is operated under various permits, licences, leases and claims granted and renewed from time to time. MLJV reports that currently all are in good standing. On July 25, 2005, the Canadian Nuclear Safety Commission ("CNSC") issued Mine Operating Licence, UMOL – MINE MILL – McCLEAN .02/2009, for a four–year term to May 30, 2009. The Approval to Operate Pollutant Control Facilities 10–2005 was issued on August 26, 2005, by Saskatchewan Environment. This approval expires on August 31, 2010. RPA has viewed documentation supporting the latter two renewals.

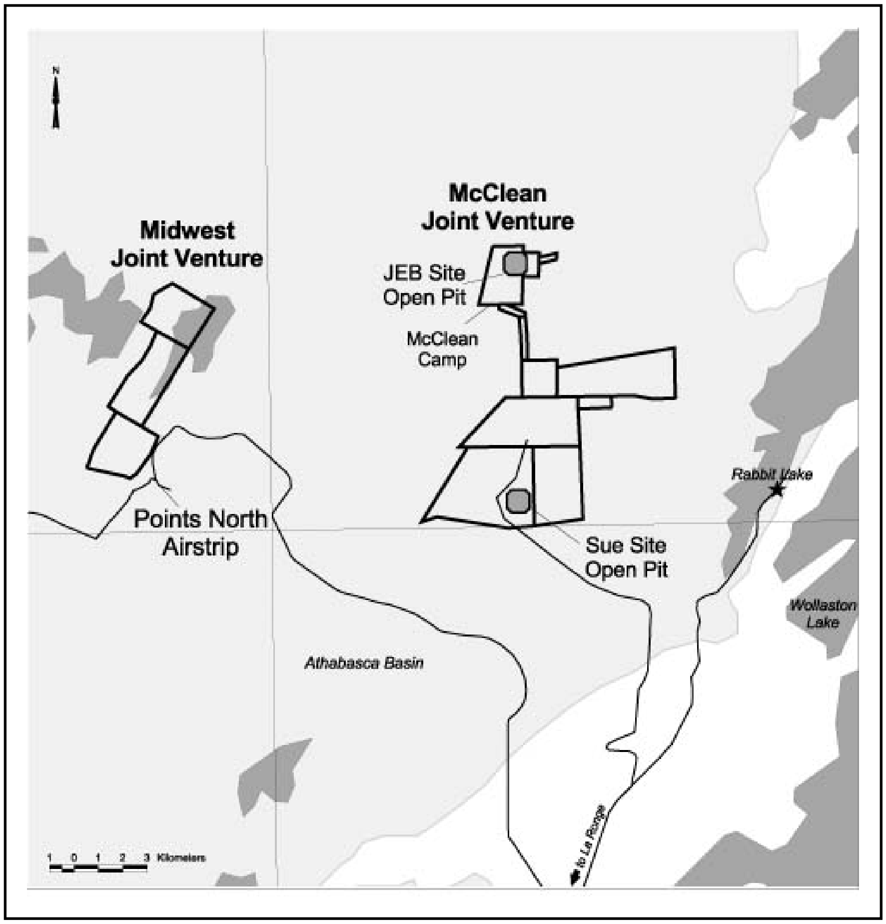
7 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

ACCESSIBILITY

Access to the McClean Lake property sites is by both road and air. Goods are transported to the sites by truck over an all-weather road connecting with the provincial highway system. Air transportation is provided through the Points North airstrip about 25 kilometres from McClean Lake (Figure 7-1).

The nearest permanent community is Wollaston Post, about 50 kilometres from the property on the other side of Wollaston Lake. Workers commute to and from the site by aircraft landing at Points North, then by bus to the site. While at the site, workers reside in permanent camp facilities at McClean Lake. Personnel are recruited from the northern communities and major population centres such as Saskatoon, and normally work one week on and one week off.

FIGURE 7-1 MCCLEAN LAKE AND MIDWEST PROPERTIES



Source: Denison Mines Inc.

CLIMATE

Site activities are carried out all year despite the cold weather during the winter months. The climatological data, temperature and precipitation, have been summarized from data provided by Environment Canada (2003). The mean monthly temperatures are below 0°C for seven months of the year. Annually, mean monthly temperature ranges between -24.3°C and 15.3°C, with extremes as low as –34.2°C, indicating the severity of the winter. The precipitation is relatively heavy for the region (550 millimetres annually with more than half that total falling as rain). The wettest period is from June to September, which accounts for 55% of the total annual precipitation. The mean date of the last frost in spring is June 1st and the mean date of the first frost in the fall is

September 1st, giving a mean annual frost-free period of 86 days. The mean annual temperature is –3.6° C, and the area lies within a zone of discontinuous permafrost.

LOCAL RESOURCES

Water for industrial activities is obtained from Pat Lake, southwest of the JEB Mill, on the McClean Lake Property.

Electric power for the JEB Mill and the Sue Site is obtained from the provincial grid through a switch station at Points North, with stand-by power available as required.

INFRASTRUCTURE

The main facilities and operations at the McClean Lake Property are an open pit mining area (Sue Site) and the JEB Mill located near the previously mined-out JEB pit, which has been converted to the tailings management facility (JEB Site). There are also various supporting facilities for activities such as water treatment, site infrastructure including roads, electricity distribution and the camp facilities. The Sue C pit is mined out, and future mining of the Sue A and B pits has been approved. A 12-kilometre haul road connects the Sue and JEB Sites. The camp facilities are located near the JEB Site. The office and shops for the mill are housed in the mill complex.

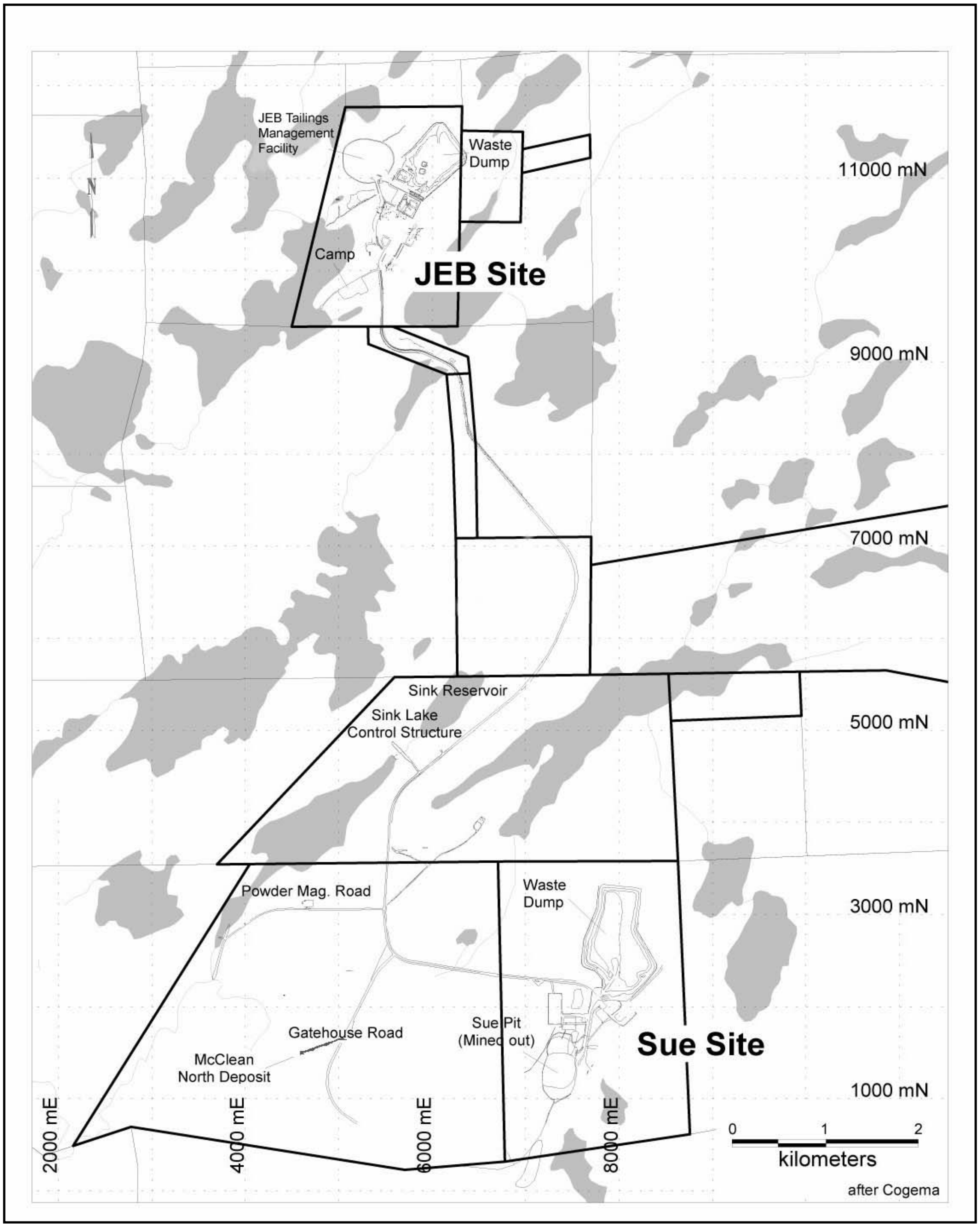
The JEB Mill uses sulphuric acid and hydrogen peroxide leaching and a solvent extraction recovery process to extract and recover the uranium product from the ore. A series of unit processes, or circuits, are directly associated with uranium production. Discharge of treated water is through the JEB Water Treatment Plant, located at the JEB Site. Tailings are discharged through a pipe-in-pipe containment system to the edge of the JEB Tailings Management Facility ("JTMF"), where they are deposited in water in the mined-out JEB pit.

All tailings from the JEB Mill are deposited in the JTMF in the mined-out JEB pit. A facility also has been designed to receive tailings from the processing of the high-grade Midwest and Cigar Lake ores.

PHYSIOGRAPHY

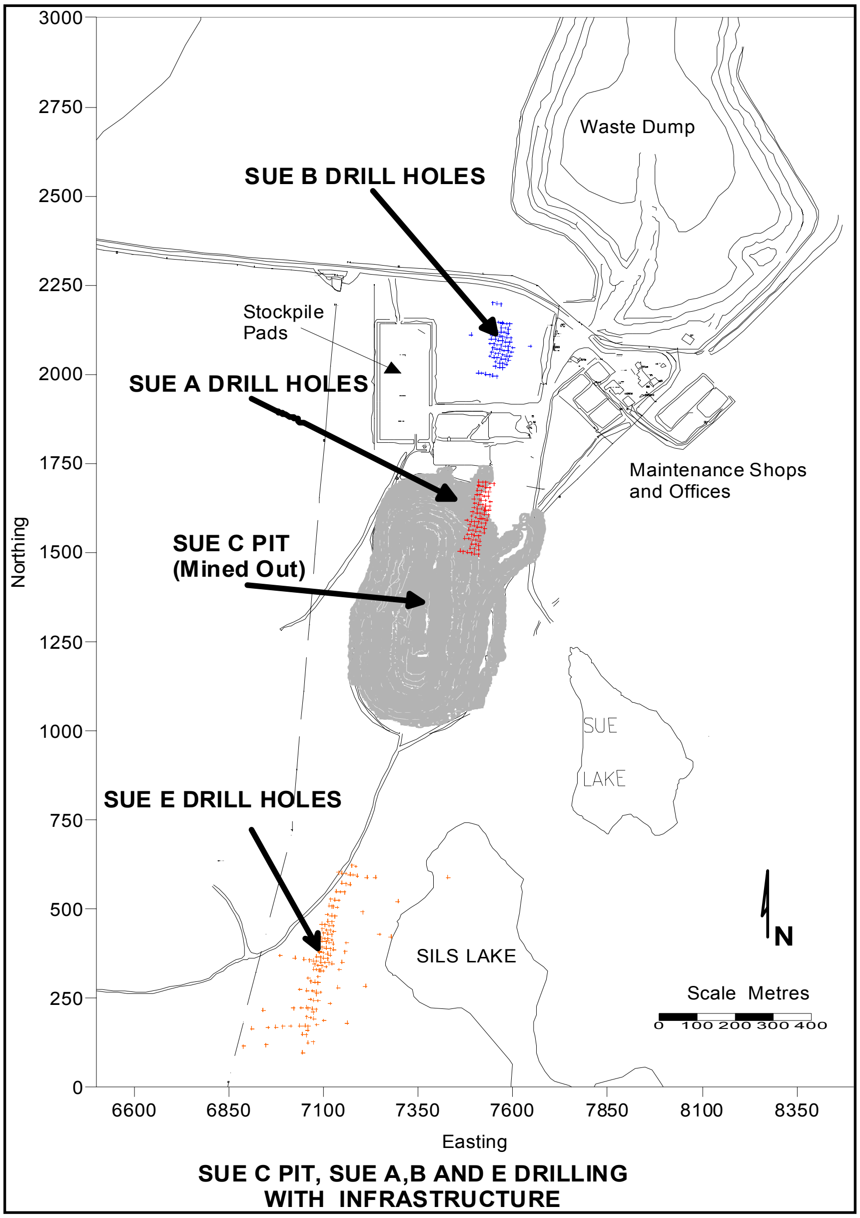
The entire area was glaciated at least three times during the last 150,000 years. The land forms are sandy and gravel moraines, drumlins, and drumlinoids that follow northeast-southwest trends and produce sand and gravel ridges over the largest portion of the area. The maximum relief is 90 metres (450 to 540 metres above sea level). The drainage is typical of relatively flat, recently glaciated regions, with numerous lakes and wetlands covering 25% of the area. Discontinuous muskeg is present throughout the area in topographic depressions and ranges in thickness from one to two metres. The vegetation in the area, rarely more than 10 metres high, consists of jack pine and black spruce with moss as the predominant groundcover.

FIGURE 7-2 JEB AND SUE SITES



Source: Denison Mines Inc.

FIGURE 7-3 SUE SITE, DRILLING, INFRASTRUCTURE AND PHYSIOGRAPHY



8 HISTORY

Canadian Occidental Petroleum Limited ("Canadian Oxy") began exploring for uranium in northern Saskatchewan in 1974. The prospective area was located between the known Rabbit Lake deposit and Midwest Lake where previously uraniferous boulder trains had been found. In April 1977, Canadian Oxy entered into a joint venture agreement ("Wolly Joint Venture") with Inco Limited ("Inco"). During a diamond drilling programme in 1977, one of the 47 holes drilled encountered encouraging uranium mineralization. Over the next two years, extensive exploration work was carried out, including airborne geophysics, electromagnetic surveys, and diamond drilling.

Mineralization was discovered in January 1979, and the follow-up drilling later that year confirmed the existence of a significant unconformity-type uranium deposit (the McClean North deposit). Subsequent exploration resulted in the discovery of the McClean South and JEB deposits in 1980 and 1982, respectively.

In 1984, CanadianOxy and Inco received conditional approval from the regulatory authorities for an underground exploration permit for the McClean deposit. Shortly thereafter, Canadian Oxy and Inco reached a corporate decision to suspend all ongoing field and engineering work on that project.

In January 1985, Minatco Limited ("Minatco"), a predecessor in title to Cogema, entered into the Wolly Joint Venture (predecessor to the McClean Joint Venture) with Canadian Oxy and Inco. From 1985 to 1990, Minatco continued exploration of the McClean Lake Property including airborne and ground geophysical surveys, percussion and diamond drilling. The reconnaissance diamond drilling programme resulted in the discovery of the Sue A deposit in 1988. Further drilling discovered the Sue B and Sue C deposits in the later part of 1988 and 1989, and the Sue E deposit in 1991.

In 1993, the owners of the Midwest Property and the McClean Lake Property agreed to combine their interests and develop two complementary projects. Ownership interests in the respective joint ventures were interchanged with Denison that acquired a 22.5% interest in McClean Lake.

Development of the McClean Lake uranium facility began in March 1995. Construction and commissioning were completed in 1997. The JEB deposit was mined out and the ore stockpiled. In 1999, the JEB Pit was converted into the JEB Tailings Management Facility.

Mining of the Sue C ore body was completed on February 3, 2002, and all of the ore has been stockpiled on surface. The low-grade uranium special waste, from the mining of the JEB and Sue C deposits, was disposed of in the mined-out Sue C pit in such a manner that it could not interfere with the mining of the adjacent Sue A deposit. This work was completed in April 2002. The pit is now being allowed to flood naturally.

In 2002, exploration drilling discovered a pod-like deposit at the western extension of the Sue trend, in the Caribou Lake area, about three km from the Sue C pit. Mineralization occurs in sandstones and is arsenical which makes it distinct from that of the Sue trend.

In October 2003, Denison Energy Inc. issued a NI 43-101 Report on the reserves and resources of the McClean Lake and Midwest areas, with a comment that underground development of the McClean North area was not likely the most economically effective method as originally proposed in a feasibility study by Kilborn in 1990. This was followed by a Denison Energy Inc. report in November 2003 with a resource estimate at the pre-feasibility level assuming development of McClean North using Blind Shaft Boring.

Effective March 8, 2004, Denison became an active business, having acquired the mining and environmental services' business from Denison Energy Inc.

Table 8-1 illustrates the recent production history from the McClean Lake properties:

TABLE 8-1 MCCLEAN LAKE PROPERTIES - PRODUCTION HISTORY

McClean Lake Joint Venture McClean Lake Property, Saskatchewan

	1999	2000	2001	2002	2003	2004
Ore Milled - tonnes x 1,000	23	82	98	122	132	152
Average Grade - % U_3O_8	3.24	3.42	3.10	2.29	2.07	1.86
Production - lbs U_3O_8 x 1,000	1,455	6,015	6,595	6,098	6,028	6,005

9 GEOLOGICAL SETTING

This Section has been taken directly from the 2003 Denison Reserves reports.

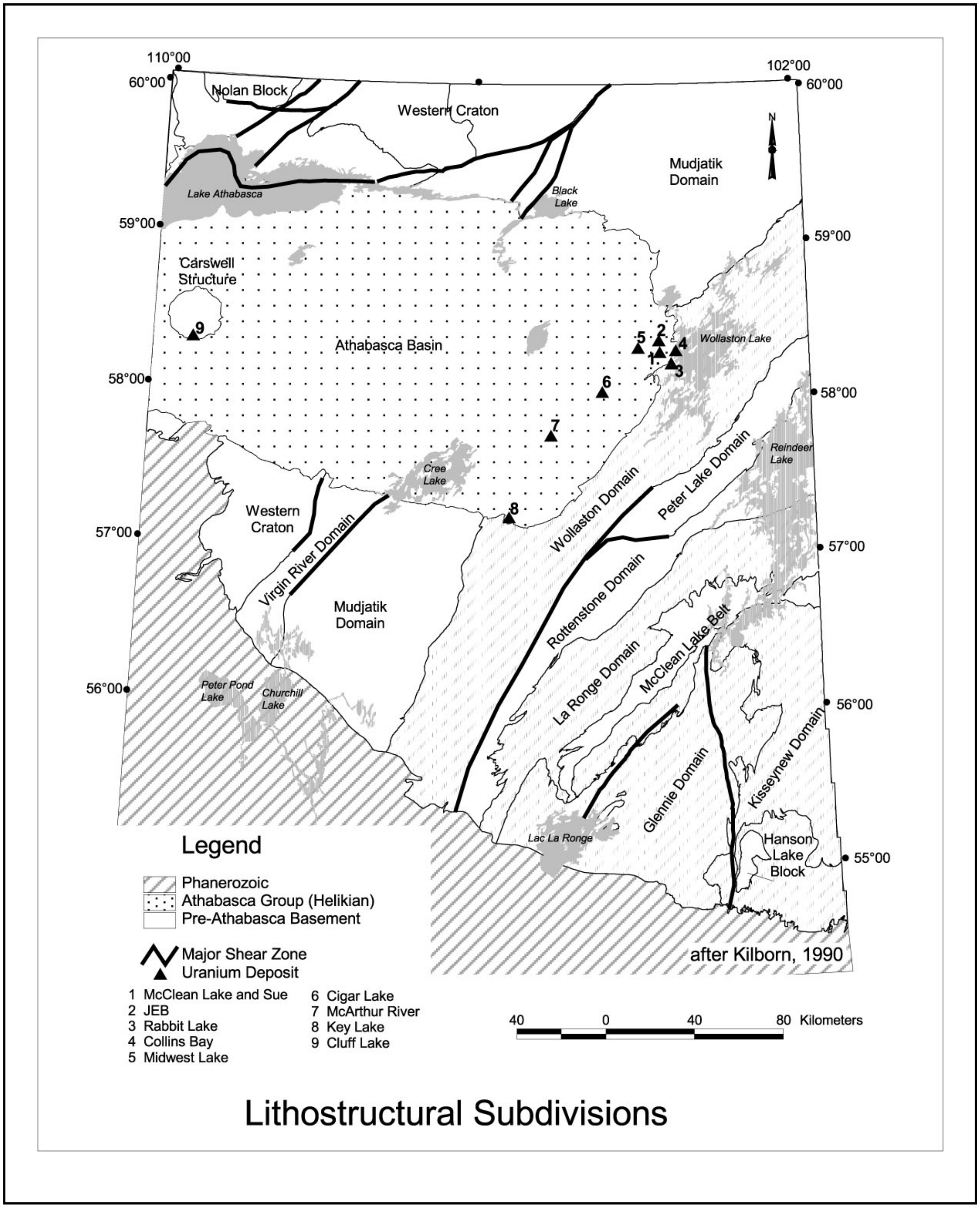
REGIONAL GEOLOGY

The McClean Lake and Midwest uranium deposits lie near the eastern margin of the Athabasca basin in the Churchill Structural Province of the Canadian Shield. The bedrock geology of the area consists of Precambrian gneisses unconformably overlain by flat-lying, unmetamorphosed sandstones and conglomerates of the Athabasca Group. The Midwest property straddles the transition zone between two prominent litho-structural domains within the Precambrian basement, the Mudjatik to the west and the Wollaston to the east, while the McClean Lake Property lies entirely within the Wollaston domain.

These domains are the result of the Hudsonian Orogeny in which an intense thermo-tectonic period remobilized the Archean age rocks and led to intensive folding of the overlying Aphebian-age supracrustal metasedimentary units. The Mudjatik domain represents the orogenic core and comprises non-linear, felsic, granitoid to gneissic rocks surrounded by subordinate thin gneissic supracrustal units. These rocks, which have reached granulite-facies metamorphic grades, usually occur as broad domal features. The adjacent Wollaston domain consists of a steeply dipping isoclinally-folded sequence of Aphebian gneissic rocks with a distinct northeast lineal structural trend. The basement surface is marked by a paleo-weathered zone with lateritic characteristics referred to as regolith.

The sedimentary rocks of the Athabasca Basin unconformably overlie the metamorphic basement. The basin is deep, closed and elliptically shaped. The sedimentary rocks in the basin are fluvial sandstones and conglomerates with minor shales and dolomites.

FIGURE 9-1 GEOLOGY OF NORTHERN SASKATCHEWAN



Source: Denison Mines Inc.

The area is cut by a major northeast-striking fault system of Hudsonian Age. The faults occur predominantly in the basement rocks but often extend up into the Athabasca

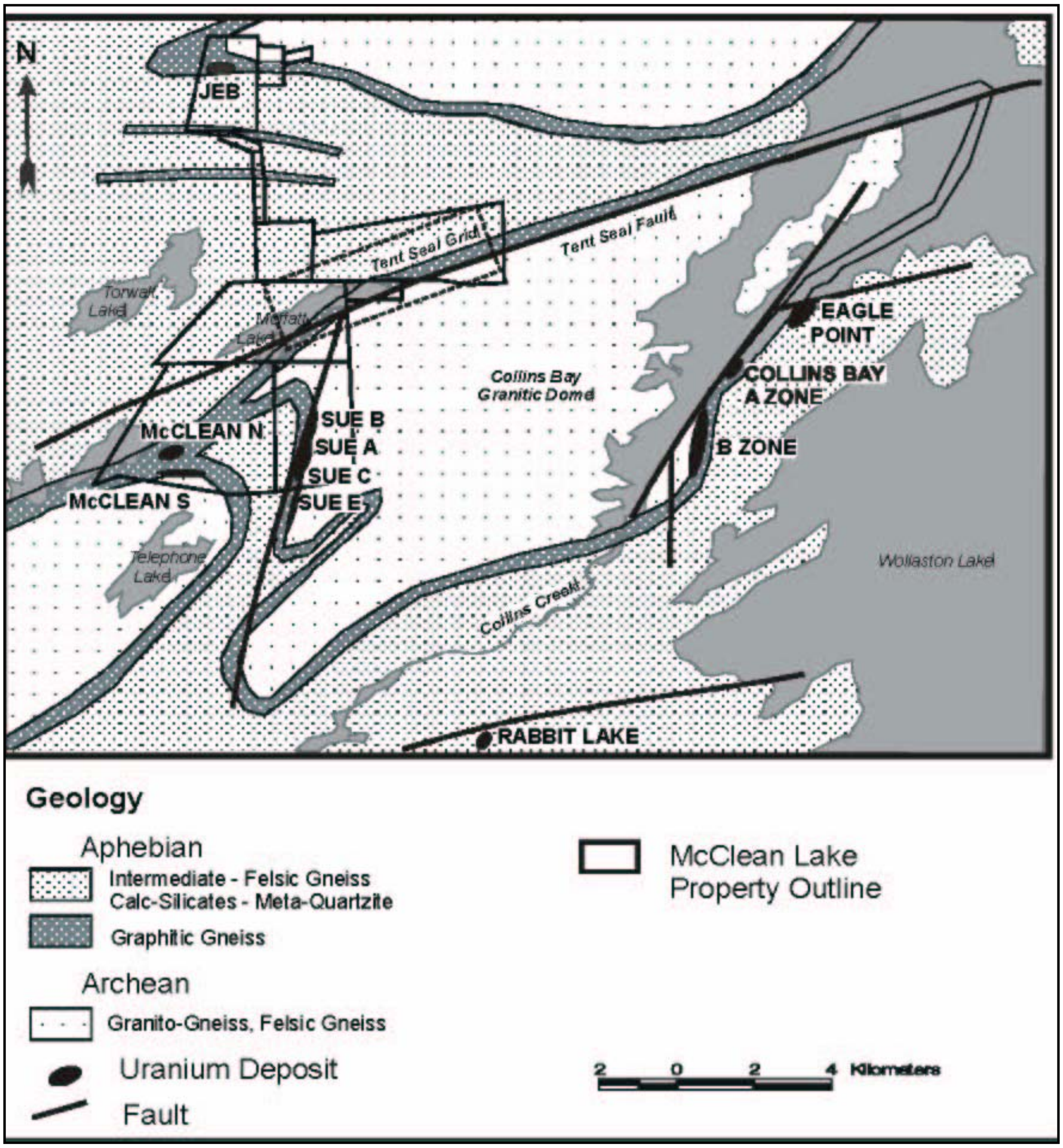
Group due to several periods of post-depositional movement. Diabase sills and dykes are frequently associated with the faulting. The diabase dykes are often mineralized as exhibited in holes 192 and 487 at Midwest.

LOCAL AND PROPERTY GEOLOGY

PRE-ATHABASCA FORMATION - MCCLEAN LAKE AREA

The pre-Athabasca or basement geology underlying the McClean Lake area is composed of a thin cover of Lower Aphebian gneissic rocks, believed to be 200 to 300-metres thick, lying on Archean granitoid gneisses. Geophysical evidence suggests that approximately one half of the McClean Lake area is underlain by these felsic granitoids. The rocks occur as domal masses and range from foliated granitoids in the core to more gneissic rocks on the margins and in many instances are wrinkles or bulges of much larger features (Figure 9-2). Complex folding has produced thin arcuate antiforms in the Archean granitoids surrounded by narrow synforms of lower Aphebian pelitic gneisses containing a graphitic unit that is highly significant with regards to uranium exploration. The lower member of the Aphebian cover displays a continuous stratigraphic succession of predominantly metapelitic gneisses containing a dominant graphitic member. All of the known significant uranium mineralization on the McClean property is directly associated with that graphitic member.

FIGURE 9-2 BASEMENT GEOLOGY OF THE MCCLEAN LAKE PROPERTY AND AREA (AFTER KILBORN, 1990)



Source: Kilborn

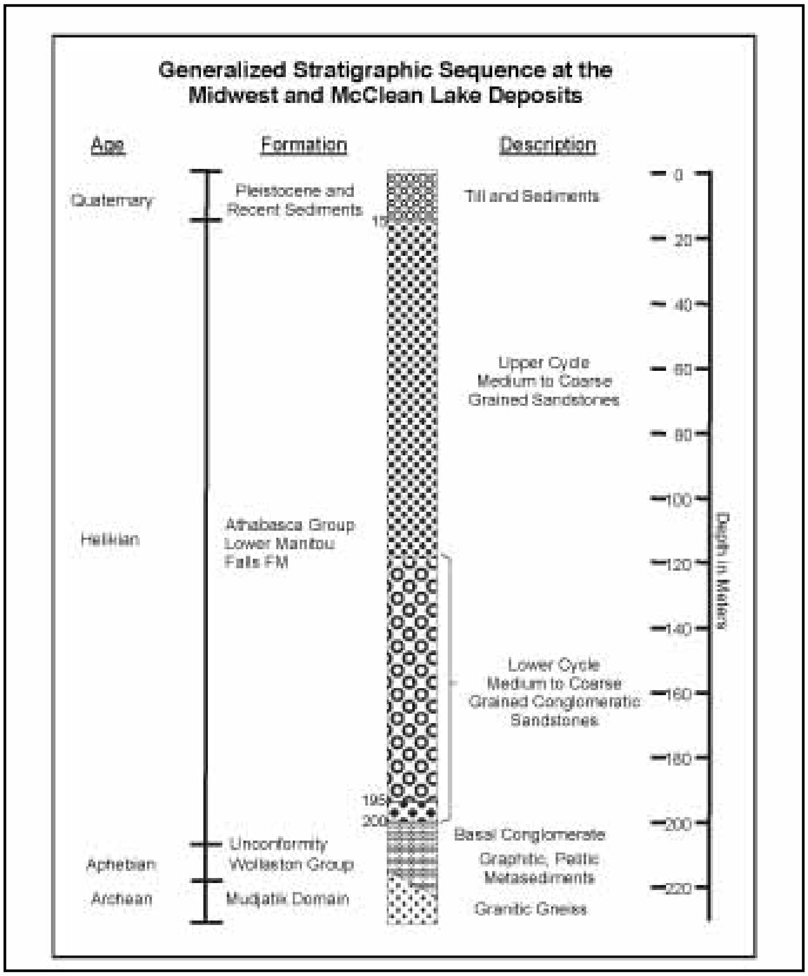
ATHABASCA FORMATION - MCCLEAN LAKE PROPERTY

Figure 9-3 illustrates the generalized stratigraphic sequence in the McClean Lake Property.

The unconformity at the base of the Athabasca Sandstone contains a tropical paleo-weathering profile. The regolith varies from a few metres to over 30-metres thick, the thickness being highly dependent on the composition of the parent rock as well as basement structures. The regolith is often completely destroyed by hydrothermal alteration in the zones of mineralization.

The Athabasca Sandstone unit covers the whole area of the Property. It is represented by up to 200 metres of the Manitou Falls formation, a non-marine fluviatile sandstone with conglomeratic lenses in the basal B member. These sandstones were deposited on alluvial fans and in braided streams and typically show abundant cross-bedding, coarser and finer units, and a general horizontal layering. The Athabasca thickens westward into the basin.

FIGURE 9-3 TABLE OF FORMATIONS



Source: Denison Mines Inc.

QUATERNARY GEOLOGY

The surficial deposits are of Quaternary age and consist largely of a Pleistocene drumlinized till plain resting directly on the sandstone bedrock. The till is locally overlain by sediments consisting of glacio-fluvial sands and gravels, and recent alluvial sands and silts. The till generally is two to four-metres thick, but reaches as much as 15 metres under gently undulating drumlins that add up to 30 metres to the local relief.

STRUCTURE

The structural geology of the pre-Athabasca rocks is highly complex, having undergone at least three major deformational episodes of folding during the Hudsonian orogeny. Many of the faults exhibit several superimposed periods of activity with both horizontal and vertical movements being evident. Some fault sets were reactivated following Athabasca sedimentation and provided channel-ways for hydrothermal solutions and the loci for uranium deposition. Horizontal shear cleavage has been identified at the unconformity horizon and is best expressed in the highly altered environment of the uranium deposits. These shear structures appear to be related to and control the alteration.

The McClean North and South deposits are controlled by a zone of strong east-west faulting and fracturing coincident with the basement graphitic gneisses. These faults dip about 70° south and exhibit a combination of normal and reverse offsets which create basement highs of a few metres. There are also steeply-dipping northeast and northwest-trending fracture sets which show both vertical and lateral displacement.

The favourable graphitic gneiss, which hosts or is immediately below the Sue deposits, is in fault contact to the east with feldspathic gneisses and granitoid rocks, whereas to the west it is gradational with intermediate gneissic units.

At the Sue deposits combinations of normal and reverse faults which parallel the east-dipping foliation in the graphitic gneisses have resulted in basement relief of 10 to 20 metres. Reverse faulting stepped the unconformity down to the west. The Sue A and B deposits occur along the western flank of a basement horst which has 8 to 10 metres of relief. Northeasterly and northwesterly striking faults offset and modify the major north-south structural controls, creating conditions which limit, or significantly control, the extent of mineralization along the trend.

ALTERATION

The following description of alteration associated with unconformity-type uranium deposits was largely taken from Quirt, 2003 by Denison:

The two main types of ore paragenesis in the Athabasca basin are dictated by form of fluid interaction and can be separated by deposit location:

(1) sandstone hosted egress-type (Midwest) involving mixing of the oxidized sandstone brine with relatively reduced fluids issuing from the basement into the sandstone, and

(2) basement hosted ingress-type (Sue C and E) involving fluid-rock reactions between oxidising sandstone brine entering basement fault zones and the wall rock. Both types of mineralization and associated host-rock alteration occurred at sites of basement-sandstone fluid interaction where a spatially-stable redox gradient/front was present.

The dominant ore location can occur in the sandstone directly above the unconformity (McClean Lake Property), straddling the unconformity (Midwest), or perched high above the unconformity (certain zones at both McClean Lake and Midwest). Similarly, in some deposit areas, there is a plunge to the mineralized pods from sandstone-hosted to basement-hosted within deposit–scale strike lengths (McClean Lake trend, Sue trend).

Most sandstone hosted deposits display dominant desilicification features and coincident abundant accumulations of clay minerals and detrital minerals like zircon and tourmaline. Around basement hosted deposits, however, the host rock alteration is dominantly chloritic with restricted illite at the expense of biotite, cordierite and garnet as at Sue C.

Illite is often characteristic of the core of the altered and mineralized zone. Complex redox-controlled reactions and acid-base reactions resulted in precipitation of massive pitchblende with associated hematite accumulation and varying amounts of base and other metallic mineralization at sites of fluid-fluid and fluid rock interaction. The geochemical signatures of the individual unconformity-type deposits do vary significantly. Sandstone hosted deposits, such as Midwest, predominantly demonstrate subequal U+Ni+Co+As mineralization, while the basement hosted deposits of the Sue trend are predominantly U+V.

Kilborn (1990) describes the alteration at the McClean Lake deposits as follows:

At the McClean North and South deposits, alteration is extensive above and below the mineralization, being largely controlled by the zone of east-west faulting. Argillic (clay) alteration with some hematitic and chloritic alteration envelopes the mineralization and extends upwards along fractures for several tens of metres where it is ultimately capped by silicified sandstones. Alteration of the basement rocks below the mineralization consists of bleaching, chloritization, argillization, and hematization. Transverse to the mineralized trend, the alteration diminishes very rapidly and rocks are frequently fresh within a few metres of mineralization. At Sue A, the deposit lies on and immediately above the unconformity in an envelope of massive earthy-red clay. Argillic alteration extends almost to the sandstone subcrop along fault zones, leaving only scattered sections of silicification in the cap rock. At Sue B, the mineralization is likewise hosted by massive earthy-red clay, while the upper zone displays remnant silicification. The sandstone between the upper and lower zones is lightly silicified. The vein type Sue C deposit is intimately associated with clay alteration and argillization of the basement. The Sue E deposit is likewise basement hosted and has limited basement alteration outside of the mineralization.

10 DEPOSIT TYPES

The following description of unconformity-type uranium deposits was adapted from Quirt (2003) by Denison:

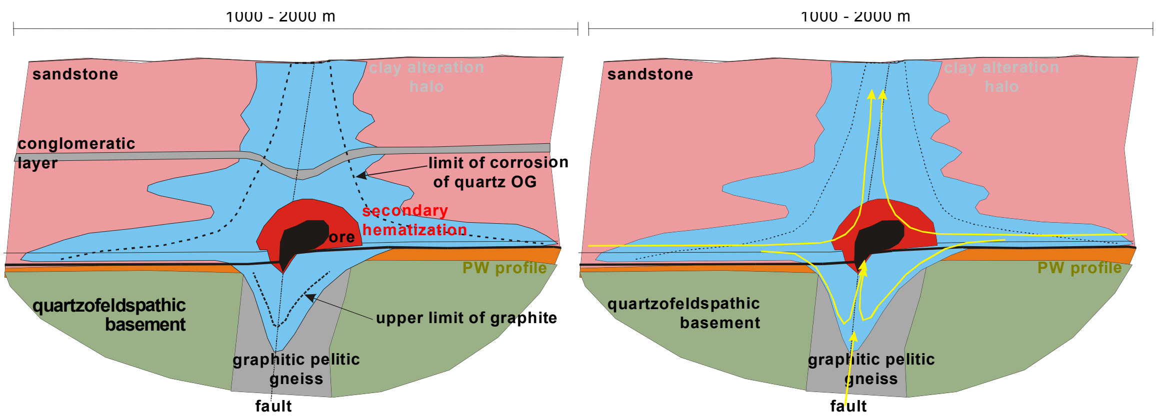
Unconformity-type uranium deposits are very high-grade and high-tonnage relative to other types of uranium deposits, and the Athabasca-hosted deposits in Saskatchewan currently account for over 34% of world-wide uranium production. A model of unconformity-type uranium deposits is illustrated in Figure 10-1. According to Quirt (2003), there are two main types of ore paragenesis that are dictated by form of fluid interaction and can be separated by deposit location:

1) Sandstone-hosted egress-type (e.g. Cigar lake, Cluff D, McArthur River, Collins Bay, Midwest) involving mixing of the oxidized sandstone brine with relatively reduced fluids issuing from the basement into the sandstone, and

2) Basement-hosted ingress-type (e.g., Rabbit Lake, Eagle Point, Sue C, Claude, and Cluff Lake N) involving fluid-rock reactions between oxidising sandstone brine entering basement fault zones and the wall rock.

For the sandstone-hosted deposits, fluid-fluid interactions best explain the presence of massive and fracture mineralization; while for basement hosted deposits, fluid-rock interactions best explain the presence of fracture filling mineralization. Both types of mineralization and associated host-rock alteration occurred at sites of basement-sandstone fluid interaction where a spatially-stable redox gradient/front was present. Without sufficient ore reaction constituents and/or the presence of a stable redox front, a barren host-rock alteration halo formed without significant mineralization.

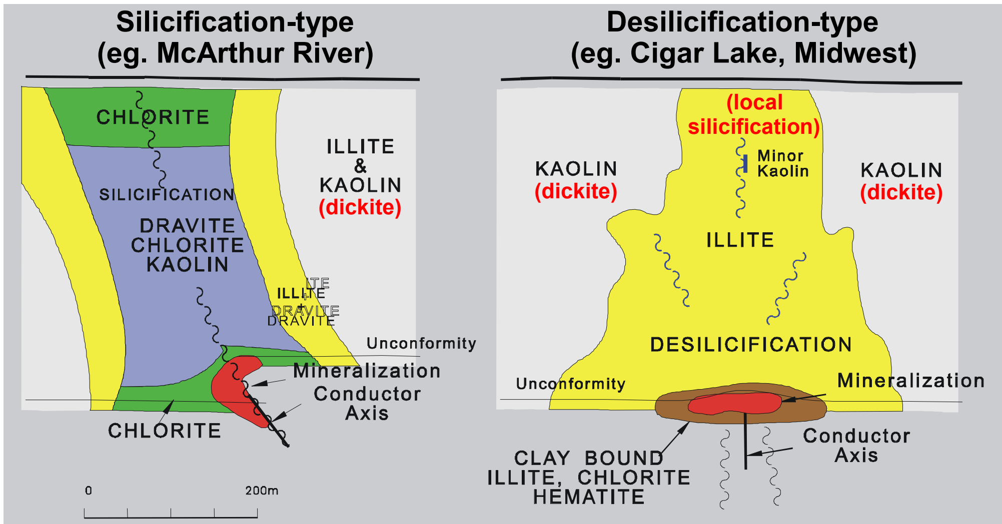
FIGURE 10-1 CONCEPTUAL MODEL OF EGRESS TYPE, SANDSTONE-HOSTED MINERALIZATION, AFTER HOEVE AND QUIRT 1984, 1987.



Conceptual model of egress-type sandstone-hosted mineralization;
left: mineralization and alteration features, right: fluid flow (Hoeve and Quirt, 1984, 1987).

The prevailing hydrological conditions controlled the location of fluid interaction relative to the unconformity, with either egress-type or ingress-type deposits. For the egress-type deposits, the location relative to the unconformity of the fluid mixing and the redox front were variable and controlled by the hydrological environment. The dominant ore location can occur in the sandstone directly above the unconformity (Key Lake, Midwest) or perched high above the unconformity (McClean Lake, Cigar Lake). The basement-hosted fluid rock interactions show less variation in location relative to the unconformity. Similarly, in some deposit areas, there is a plunge to the mineralized pods (e.g., McClean Lake trend, Sue trend). Other deposit areas do not exhibit this feature (Midwest, Cigar Lake).

FIGURE 10-2 VARIATION IN EGRESS-TYPE SANDSTONE-HOSTED, HOST ROCK ALTERATION FEATURES, AFTER QUIRT, 2003 AND WASYLIUK, 2000



11 SUE A

PROPERTY GEOLOGY

The Sue trend lies on the west flank of the Collins Bay granitoid dome and is hosted by a north-south segment of a regionally extensive, steeply-dipping thin band of graphitic gneiss within the Wollaston Domain. The Sue A deposit is immediately north of the Sue C open pit where the subsidiary splay structure that hosted the Sue C deposit veers to the northeast and rejoins the principal Sue north-south shear structure and graphite unit. The latter hosts the Sue D and E deposits to the south and the Sue B to the north. The Sue trend of graphic gneisses, from Sue E to north of Sue B, is 2.5 km long. Further north, the graphitic units follow the Collins Bay granite contact and swing west in the “Sue nose area”. The favourable trend thence continues southwest, extending an additional 2 km to the Caribou deposit site that is located approximately 1.8 km northwest of Sue A.

The Sue A deposit is approximately 175 m long, 10 m to 30 m wide, and 3 m to 9 m thick for a 0.1% U_3O_8 grade envelope. Uranium mineralization is hosted mostly in red, earthy clay-altered sandstone at the angular unconformity and for 10 m above it, with minor mineralization extending up to 10 m below it. The deposit dips gently west parallel to the sandstone-basement contact and lies at depths of 59 m to 78 m.

The Sue A deposit occurs along the western flank of a basement horst which has 8 m to 10 m of relief (Kerr et al. 2003). Movements along normal and reverse faults that parallel the east-dipping gneiss foliation have resulted in sandstone-basement contact displacements of 10 m to 20 m.

Argillic alteration extends above the mineralization in the sandstones, with chlorite developed in the basement and hematite prevalent near the unconformity where the paleo-weathering profile has been destroyed.

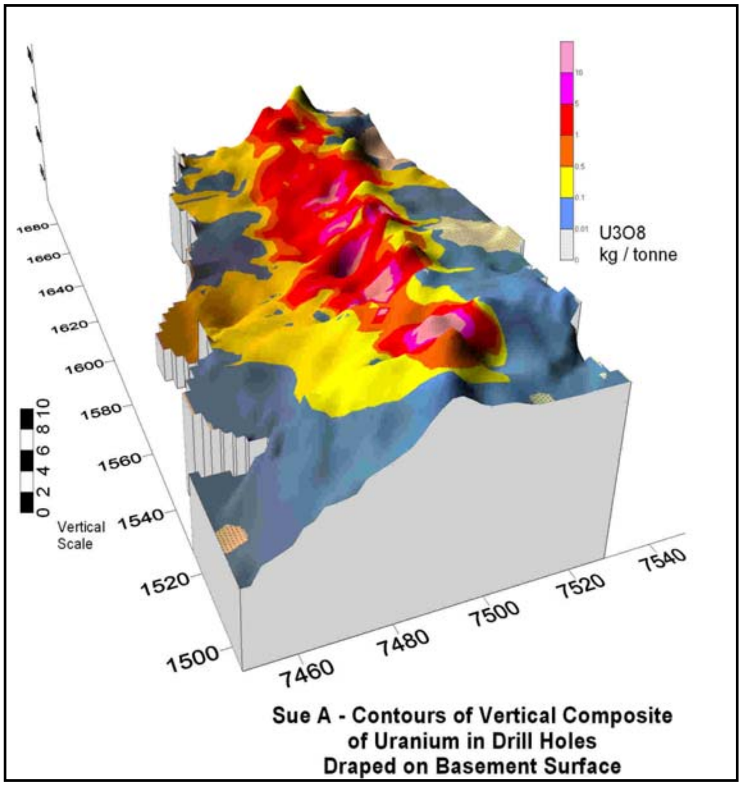
DEPOSIT TYPE

In contrast to the adjacent basement-hosted Sue C deposit, the Sue A is a sandstone-unconformity-hosted egress-style deposit similar to other sandstone-hosted deposits, such as Cigar lake, Cluff D, McArthur River, Collins Bay, and Midwest. Deposit genesis at Sue A likely involved mixing of oxidized sandstone brines with relatively reduced fluids issuing along fault zones from the basement into the sandstone, where a spatially-stable redox gradient/front was present. The prevailing hydrological conditions controlled the location of fluid interaction relative to the Athabaska-basement unconformity. The dominant uranium mineralization is just above the unconformity, and there is limited variability in the position of the uranium mineralization relative to the unconformity. There is a plunge to the mineralized pods in the Sue trend.

MINERALIZATION

The mineralization at Sue A consists primarily of uranium oxides (uraninite and pitchblende) with a suite of nickel-cobalt arsenides (primarily niccolite) in a hematitic clay matrix. Nickel, cobalt, and arsenic grades are generally low <<1%. Typically, high-grade mineralization is surrounded by a thin low-grade envelope (0.05% to 0.5% U_3O_8), which is one to two metres thick and consists of massive clay and strongly argillized sandstone.

FIGURE 11-1 SUE A, U_3O_8 CONTOURS IN DRILLING, DRAPED ON BASEMENT



Source: Denison Mines Inc.

EXPLORATION

The Sue A deposit was diamond drill tested in two campaigns by Minatco. In 1988 20 holes totalling 2,050 m were completed, and 5,840 m were drilled in 60 holes in 1989, bringing the total for Sue A to 80 holes and 7,890 m.

DRILLING

Delineation diamond drilling at Sue A was primarily NQ (47.6 mm), with most holes penetrating 15 m to 25 m or more into the basement. In general, holes were collared on 12.5 m sections and spaced at 10 m on section (Figure 11-2).

Drill hole collars were surveyed for local grid coordinates and elevation. Coordinates were subsequently converted to UTM coordinates by Cogema. Downhole deviation was measured by Sperry-Sun multishot instrumentation.

RPA received two drill hole databases for Sue A, one from Cogema and the other from Denison. The former contains drill holes for northern Sue C and covers exploration holes between Sue C and Sue A. These have been excluded from the Sue A resource estimation database. In addition to lithology, assay, collar headers, and survey tables as in the Cogema database, the Denison database contains mineralization wireframes and topographic, overburden, and unconformity and pit surfaces as well as composites used for resource estimation. The Cogema database contains only a pit wireframe.

The digital resource database for Sue A contains 81 diamond drill holes totalling 8,005.8 m. The drilling covers an area of approximately one hectare. Holes were collared on 12.5 m sections and spaced at 10 m on section. The axis of the drill grid is approximately N12ºE. There are 28 "S" series holes and 53 "CS" series holes. RPA notes that the hole numbers in the digital databases have an additional "1" appended to the original number.

Seventy-nine of the holes (7,770.8 m) were drilled at -90º dip (vertical) and two holes (345 m) at the south extremity of the drilling area were drilled at -60º to the north-northwest. Holes lengths (depths) range from 84 m to 134 m with the angled holes up to 184 m. Drill cross sections are shown in Figures 11-3 to 11-6.

All holes record downhole deviation surveys from one measurement to eight taken generally below the casing (8 m to 31 m, and possibly 72 m) and then at a nominal 15 m interval to the toe. Intervals in the inclined holes are a nominal 30 m. Two holes were surveyed only at the toe (CS051 at 134 m and CS591 at 92 m). For two other holes (S1511 and S1581), there is a large distance of 57 m to 77 m between the last two readings which show no deviation and are probably dummy entries.

RPA checked for excessive deviation as an indication of drilling problems. Forty-one holes showed possibly excessive (0.065º/m, i.e. 2º/100 ft.) dip change. Of these, 38 occurred on the first reading (14 m to 18 m generally) and presumably below the casing where the drill string may be deflected more than usual on penetration of bedrock. The remaining three dip changes, larger than usual, are in three holes, S1511, S1601 and CS111.

The Denison (Kerr et al, 2003) digital database contains 2,308 assay records (893 m), of which 2,143 have at least one analysis for one of U_3O_8%, Ni%, As%, and Co% (848.8 m). Some 374 assays (150 m) are ≥0.1% U_3O_8 and approximately 430 U_3O_8 analyses for 176 m fall within the Sue A deposit envelope at 0.1% U_3O_8 as outlined by Kerr et al. (2003). Within the sampling intervals, there are some 965 non-sampled intervals and of these approximately 265 are within or near the 0.1% envelope indicating that sampling has been somewhat discontinuous and guided predominantly by radiometrics. Because the data units were not labeled in the Cogema database provided, RPA believes that the values posted are in parts per thousand.

Core sample intervals vary from 0.05 m to 1.0 m within the resource, with most samples (>90%) taken at 0.25 and 0.50 m intervals. As noted in the data verification section, some of the short intervals appear to be artifacts of database manipulation and do not agree with original sampling and assaying intervals as reflected in the Cogema database.

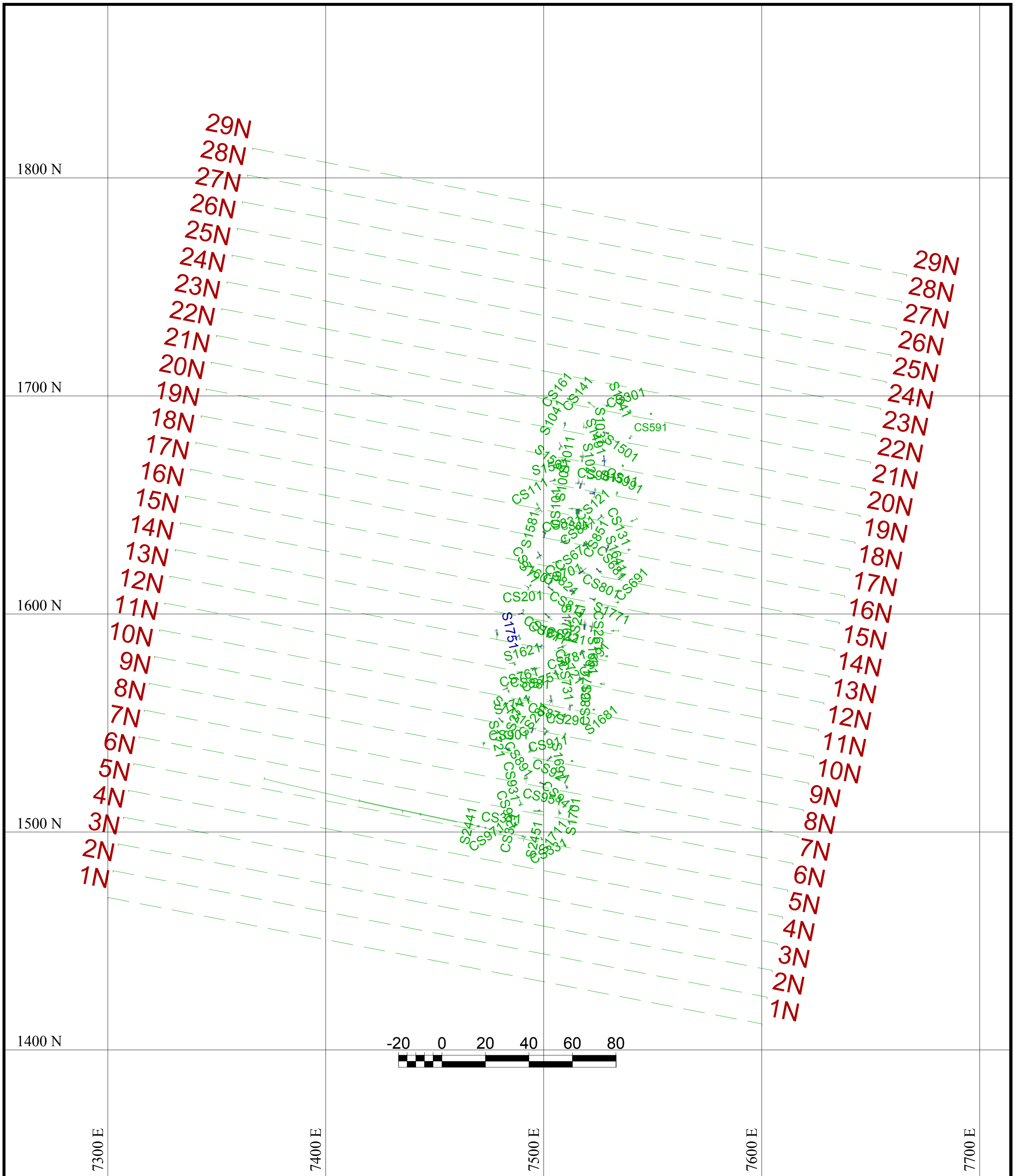


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Figure 11-2
Denison Mines Limited
Saskatchewan Projects
Sue A Deposit
Drill Hole Location Plan

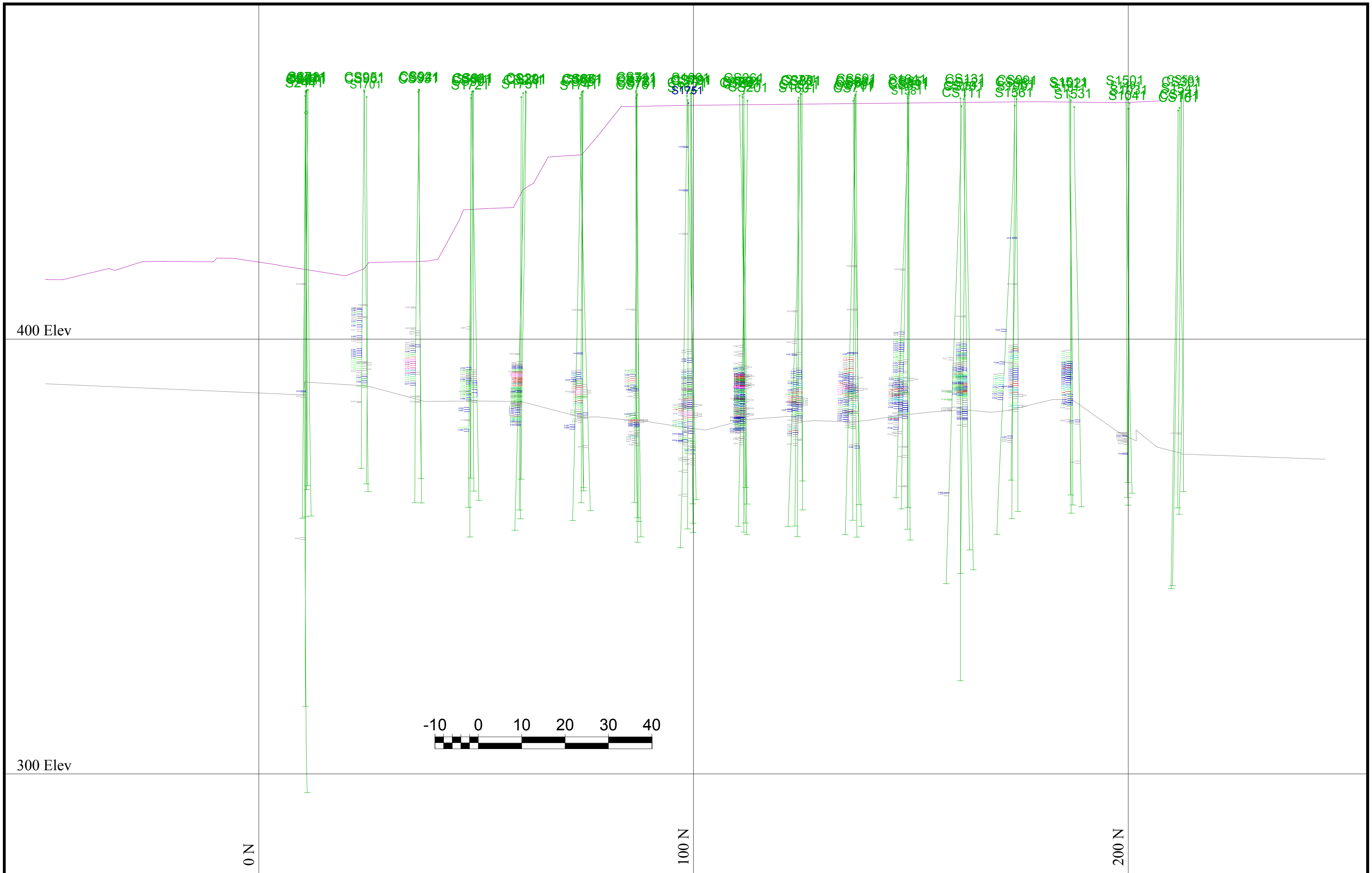

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Figure 11-3
Denison Mines Limited
Saskatchewan Projects
Sue A Deposit
Composite Drill Hole Longitudinal Section

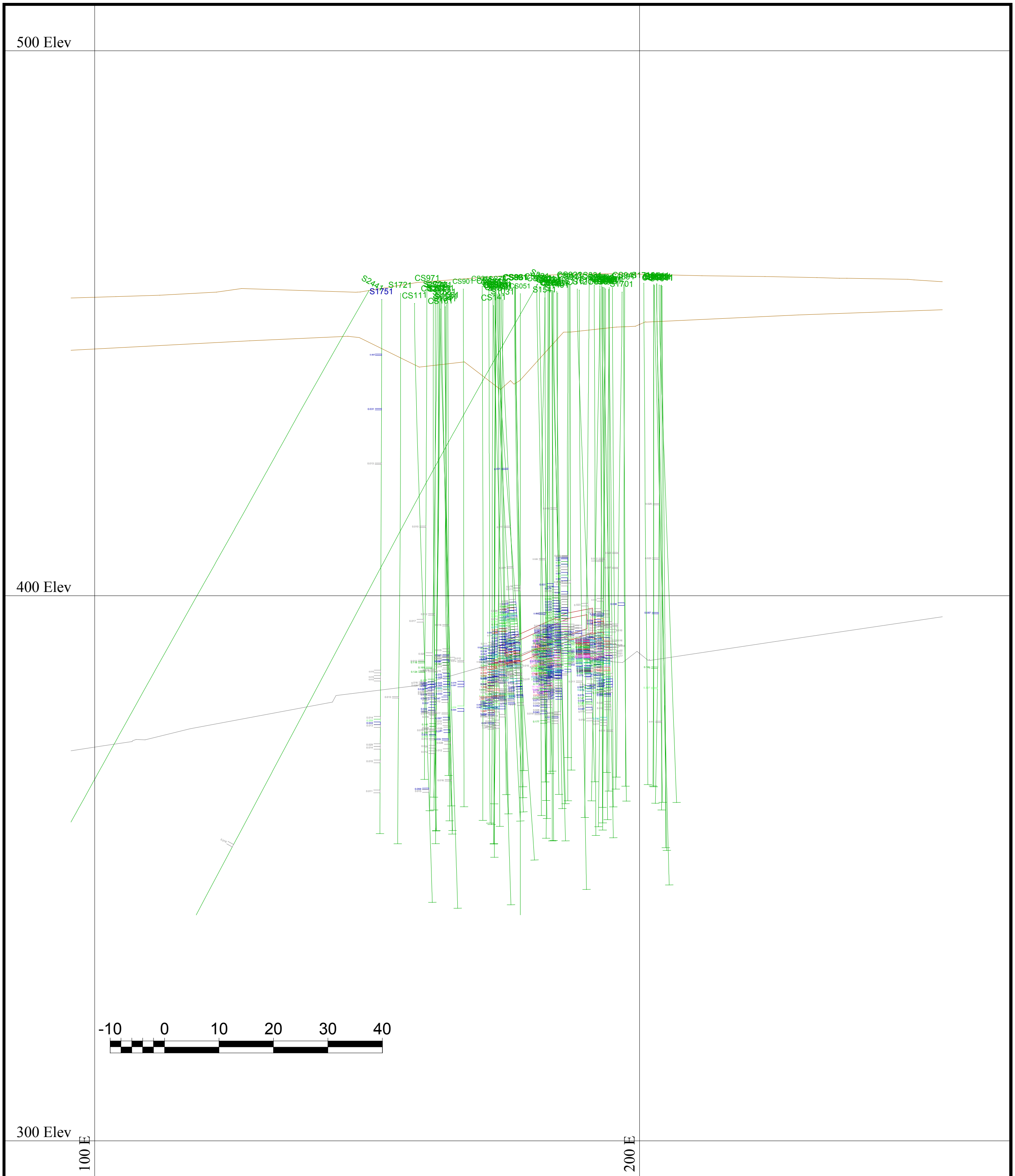

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Figure 11-4
Denison Mines Limited
Saskatchewan Projects
Sue A Deposit
Composite Drill Hole Cross Section

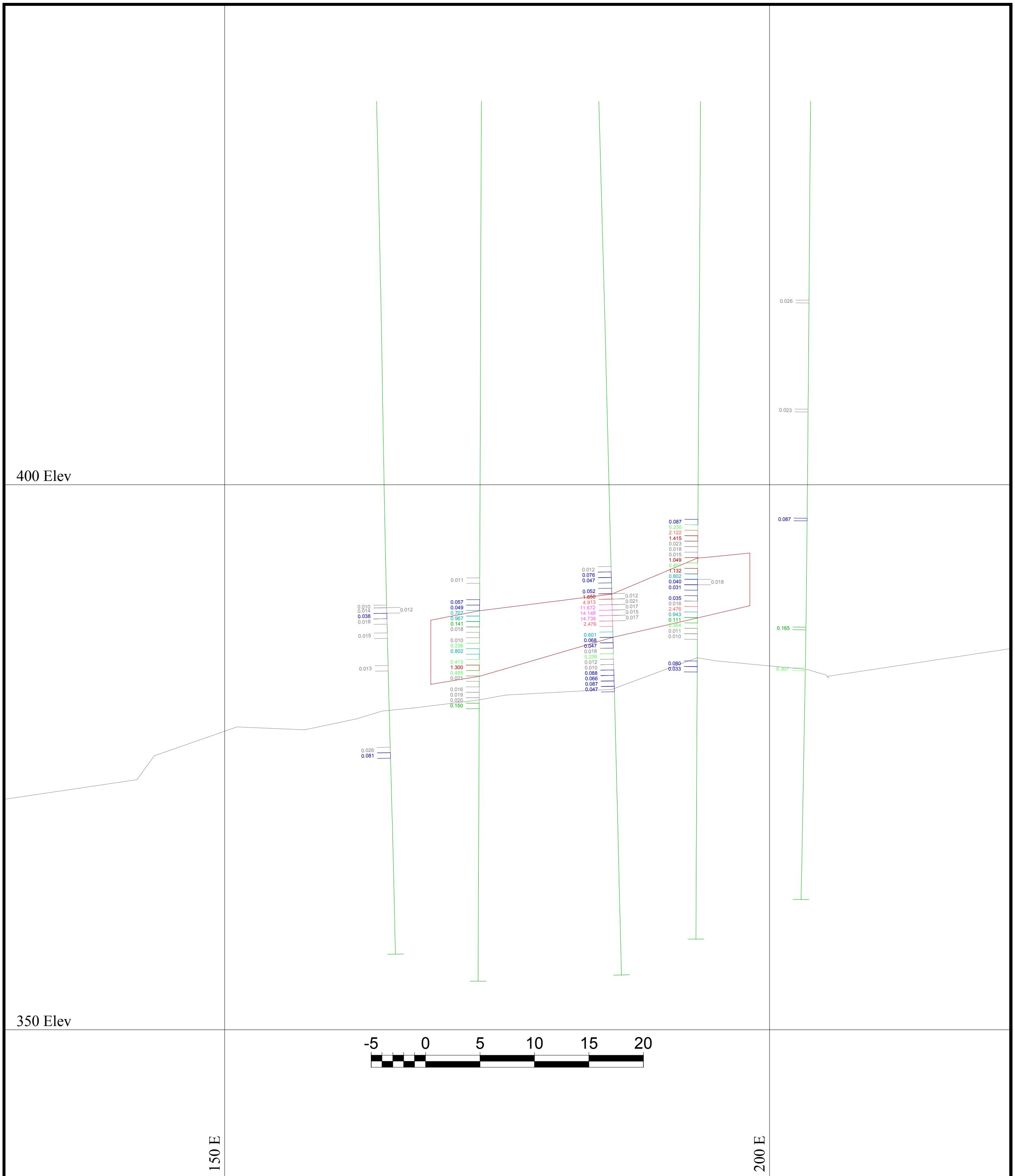

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Figure 11-5
Denison Mines Limited
Saskatchewan Projects
Sue A Deposit
Vertical Cross Section 11N

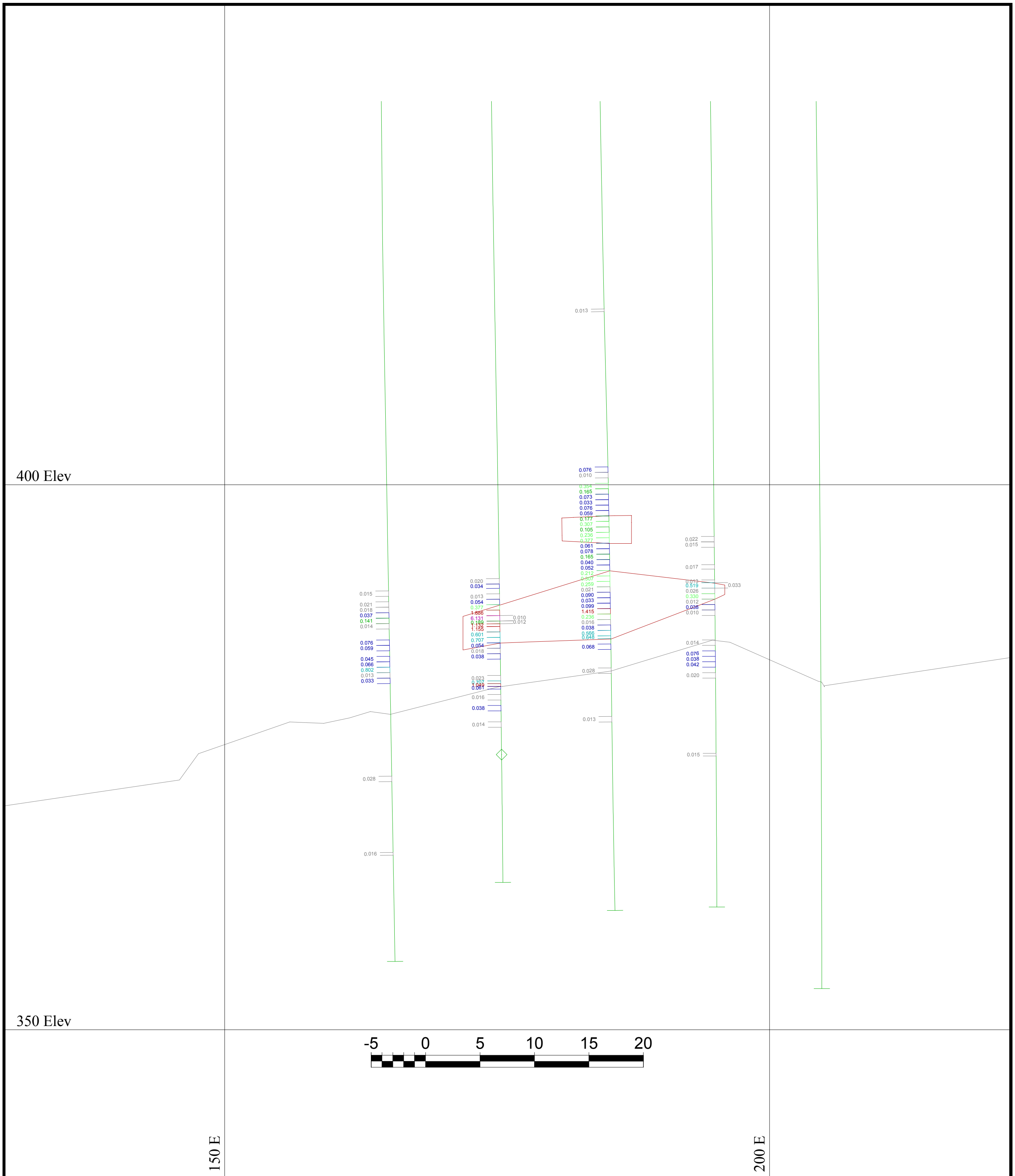

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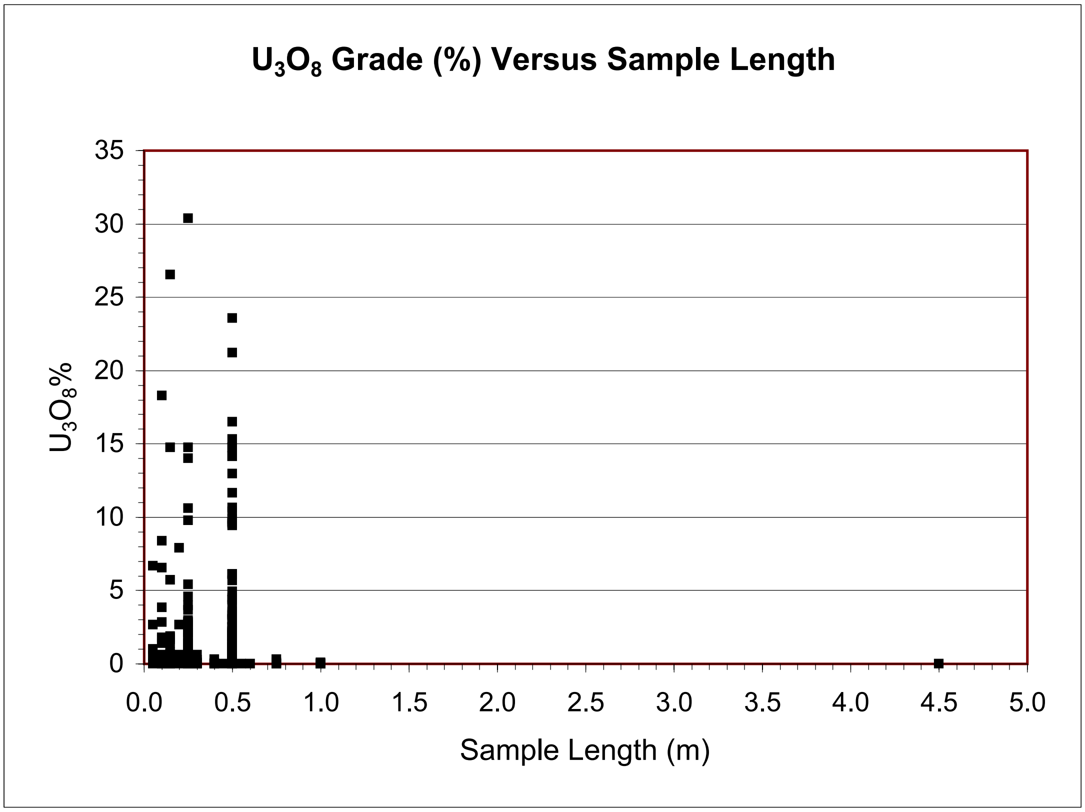
Figure 11-6
Denison Mines Limited
Saskatchewan Projects
Sue A Deposit
Vertical Cross Section 12N

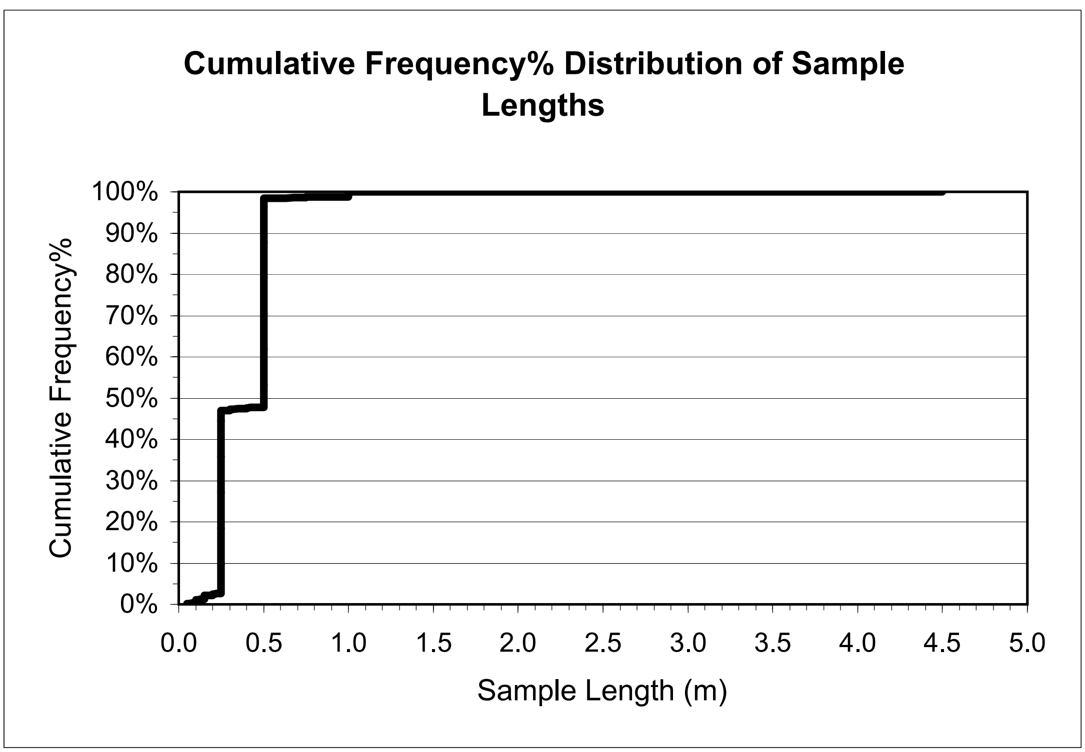
SAMPLING METHOD AND APPROACH

Core sampling is the primary sampling method. Handheld scintillometer readings on core guided sampling and provided for sampling on the basis of radiometric responses (uranium grade) where necessary. Sampling was relatively continuous for mineralized and waste intervals within the mineralized zone, but above the zone in sandstone only mineralized intervals were analyzed.

Sampling was standardized at 0.25 m and 0.5 m intervals, and approximately 90% of the core samples assayed had this length. Sampling is relatively grade independent, although the intervals longer than 0.5 m (<5%) were in very low grade/waste material (Figure 11-7). The longer intervals are not within the resource wireframe.

Figure 11-7 Sue A Sample Length Statistics
Denison Mines Ltd. Saskatchewan Uranium Projects





SAMPLE PREPARATION, ANALYSES AND SECURITY

Chemical analyses of core samples for U_3O_8 in % or ppm were performed on behalf of Minatco by Barringer Laboratories (Alberta) Ltd. in Calgary during 1988 and 1989. Approximately 60% of the samples were also analyzed for Cu (%), Ni (%), Co (%), Pb (%), Mo (%), V (%), and As (%). U_3O_8, originally in ppm, is recorded in percent to three decimal places in the digital database. In some cases, values <0.0005% are incorrectly rounded to 0.001%.

DATA VERIFICATION

RPA cross-checked digital databases received from Denison (Kerr et al, 2003) and Cogema. The Dension database was derived from a Cogema database. Elements entered in the two databases differ and units are not the same. U_3O_8, Ni, As, Co, Cu, V, Pb, and Mo, as reported originally by Barringer Laboratories, are restated in parts per thousand in the Cogema database.

The Denison database has the $U_3O_{8,}$ Ni, As, Co in percent as originally analyzed, and also has empty columns for S%, Fe%, and core loss. RPA notes that the base metal units had been misconverted in the Denison assay and composites tables of the Gemcom database. RPA corrected the units in the Denison digital database.

Cross referencing the sampling intervals in the two databases disclosed that the Denison database has some adjacent short intervals less than the standardized sample intervals of 0.25 m and 0.5 m. These combine to the standardized interval widths and also have identical grades posted. In the Cogema database, these intervals are represented by a single standard interval that agrees with the original analytical certificates and Minatco summary assay logs printed in 1988 and 1989. RPA believes that the short intervals in the Denison database are an artifact of wireframe modeling. RPA clearly identified only nine such intervals in the database, four of which grade

>0.1% U_3O_8. RPA notes that similar "artificial" sub-intervals also exist in the Denison database for the McClean North project.

RPA cross-referenced 229 analyses in six drill holes from the Barringer laboratory certificates to Minatco summary logs of assay intervals (1988-1989), and then to the Denison and Cogema resource digital databases. In hole CS81 of the Denison database RPA found that the analytical values have been posted one sample interval (0.5 m) higher with respect to the Minatco summary sheets. The Cogema database correctly positions the analyses and sample intervals in this hole. RPA has not assessed the impact of this error on Denison's 2003 resource estimate (audit), but it is expected to be minimal in terms of overall tonnes and grade. However, any mine planning based on the Denison work will have to adjust for the +0.5 m shift in elevation of the wireframe near hole CS81.

MINERAL PROCESSING AND METALLURGICAL TESTING

Leach tests were conducted on composited Sue A drill core by Cogema's Service d'Etudes de Procédés et Analysis in 1998. Leach performance was similar to JEB and Sue C ores being processed at that time. Extraction was 98% in 8 hours, with overall recovery expected at 97% (Kerr et al., 2003). Sue A ore is expected to perform well in the existing JEB processing circuit and uranium recoveries are estimated to be consistent with current experience.

MINERAL RESOURCE ESTIMATES

Resources for Sue A have been estimated by Cogema (Demange, 1998 and Eckert, 2005) and independently by Kerr et al. (2003).

COGEMA ESTIMATES

1998 Cogema Resource Estimate

The basis and methodology for the 1998 estimate is summarized as follows:

1) Model utilized Minatco geologic interpretation and a mineralization envelope (not provided to RPA) at 0.1% U_3O_8.

2) Units for resource estimation are kg/t U_3O_8 or ppt ($^o/_{oo}$).

3) Analyses/assays composited to 3 m.

TABLE 11-1 STATISTICS OF COGEMA 3 M COMPOSITES (1998)
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Number of 3 m Composites	178
Mean Grade (U_3O_8 kg/t)	5.26
Maximum Grade (U_3O_8 kg/t)	82.66
Variance (U_3O_8 kg/t)	162.81

Adapted from Demange (1998)

4) Sue A Model origin: X=7443.0; Y=1490.0; Z=410.0; rotation12º

5) Semi-variograms created as nested spherical models, no nugget introduced (no graphics presented with report):

Sue A	Spherical 1	Spherical 2
Sills	10.08	143.28
EW range (m)	2.0	7.0
NS Range (m)	2.0	8.0
Vertical Range (m)	3.0	7.0

6) Search distance: 27 blocks i.e. 3 blocks EW, 3 blocks NS and 3 blocks vertically (rotated model grid). This is 15 m EW x 37.5 m x 9 m vertical.

7) Linear (ordinary) kriging for 10 m x 12.5 m x 3 m blocks (e.g. hole spacing). The 3 m bench height is consistent with past mining at Sue C and is practical since mining Sue A will involve access from the Sue C pit and a push back of the north pit wall.

8) Density=1/(0.452-(0.00326*%U_3O_8)).

9) Precision on the estimate (volume and estimation) of the total resources stated as Sue A: ±11% and Sue B: ±19%.

10) Recoverable resources were estimated by uniform conditioning based on selective mining blocks of 2.5 m x 6.5 m x 3 m (U_3O_8 kg/t variance 80.63)

Table 11-2 states the Cogema Recoverable Resource Estimates as reported in 1998:

TABLE 11-2 1998 COGEMA RESOURCE ESTIMATE FOR SUE A
Denison Mines Inc. McClean Lake Project, Saskatchewan

U_3O_8 Cut-Off Grade	Tonnes	U_3O_8%	U_3O_8 (lbs x1,000)
0.1%	81,672	0.88	1,576
0.2%	61,947	1.11	1,514
0.3%	50,517	1.30	1,453
0.4%	42,830	1.48	1,395
0.5%	37,192	1.60	1,340

Cogema did not classify the resources to CIM or other standards.

Cogema Resource Update April 2005

Cogema (Eckert, 2005) re-estimated the resources for comparison to the 1998 estimate and for use in mine planning of operations expected to resume in the Sue C pit in summer 2005. The 1998 model methodology is fundamentally similar except for:

- Block model elevation origin shifted to 1 m lower in the new 2005 model, starting at 412 RL, versus the 1998 model.
- The working unit was changed to U kg/t in contrast to U_3O_8 kg/t in 1998.
- Indicator kriging used to define mineral envelope in 2005 versus Minatco geologic interpretation and wireframe in 1998.
- Block sizes differ being 10 m x 12.5 m x 3 m in 2005 versus 5 m x 12.5 m x 3 m in 1998. Figures 11-8 to 11-11 show cross sections and plans of the 2005 block model with an outline of >0.1% U_3O_8 mineralization provided by Denison.
- New variography and a small nugget (16%) introduced to the ordinary kriging profiles have a small impact on grade re-estimation.

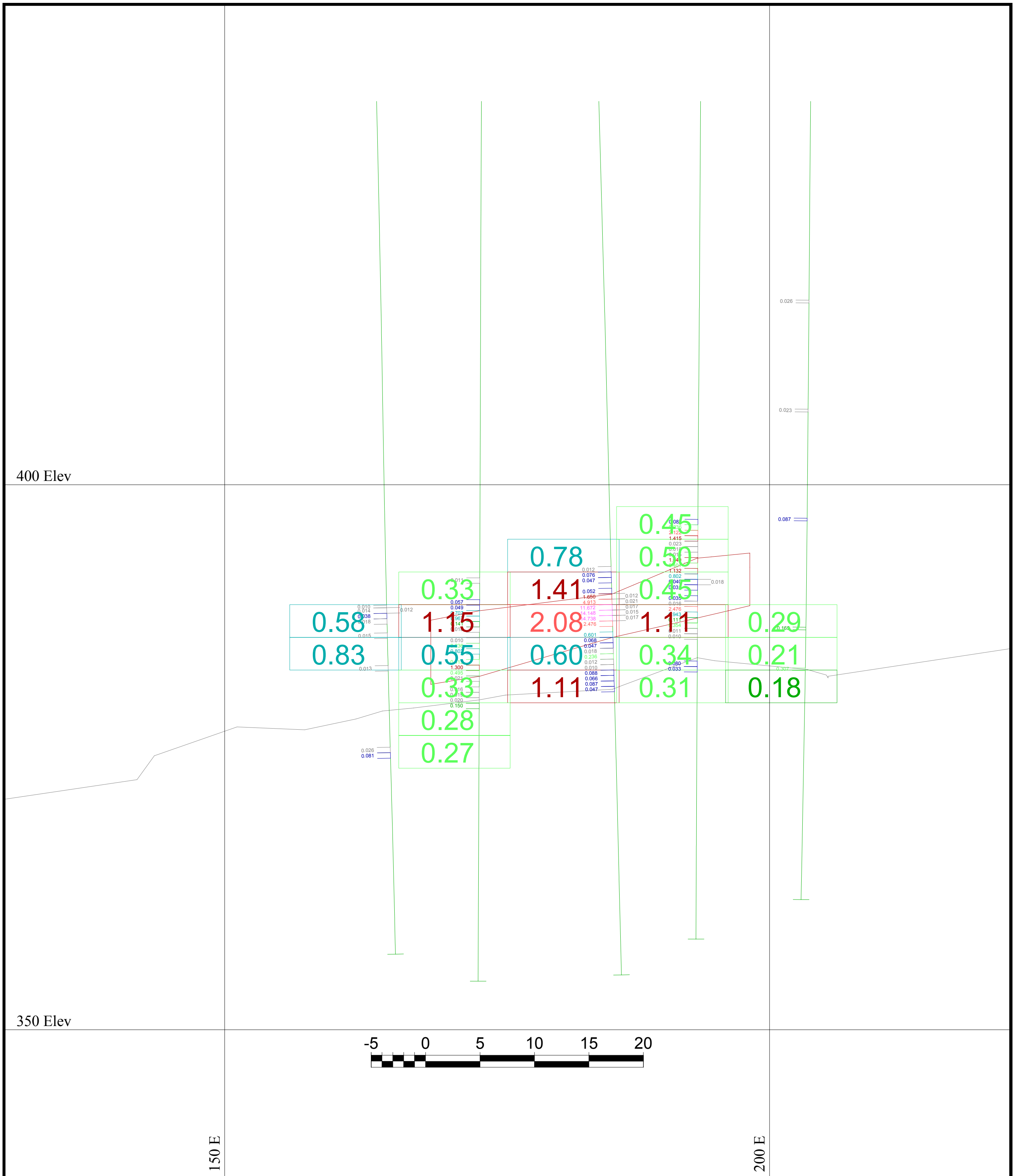


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Figure 11-8
Denison Mines Limited
Saskatchewan Projects
Sue A Deposit
Block Model Vertical Cross Section 11N

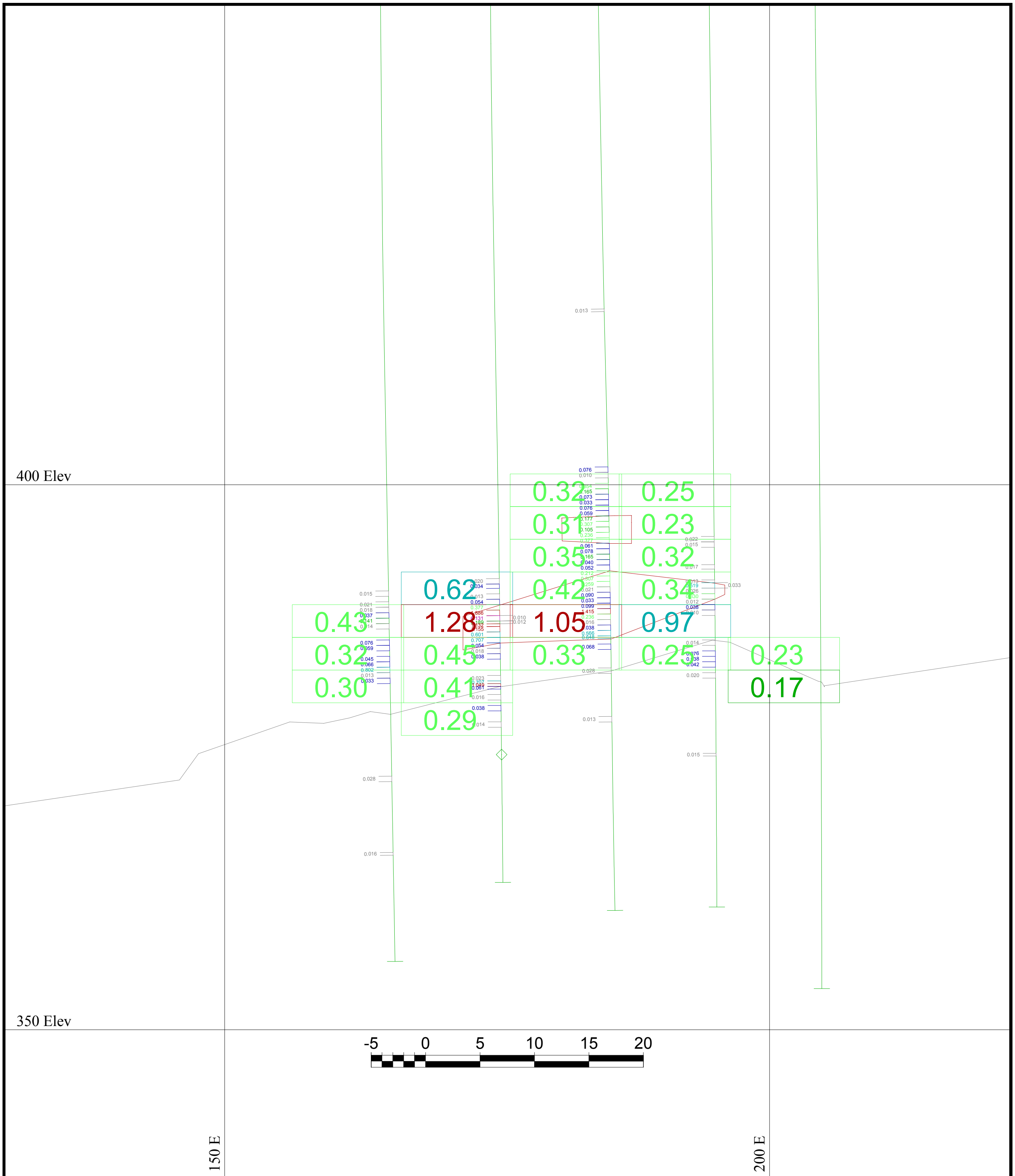

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Figure 11-9
Denison Mines Limited
Saskatchewan Projects
Sue A Deposit
Block Model Vertical Cross Section 12N

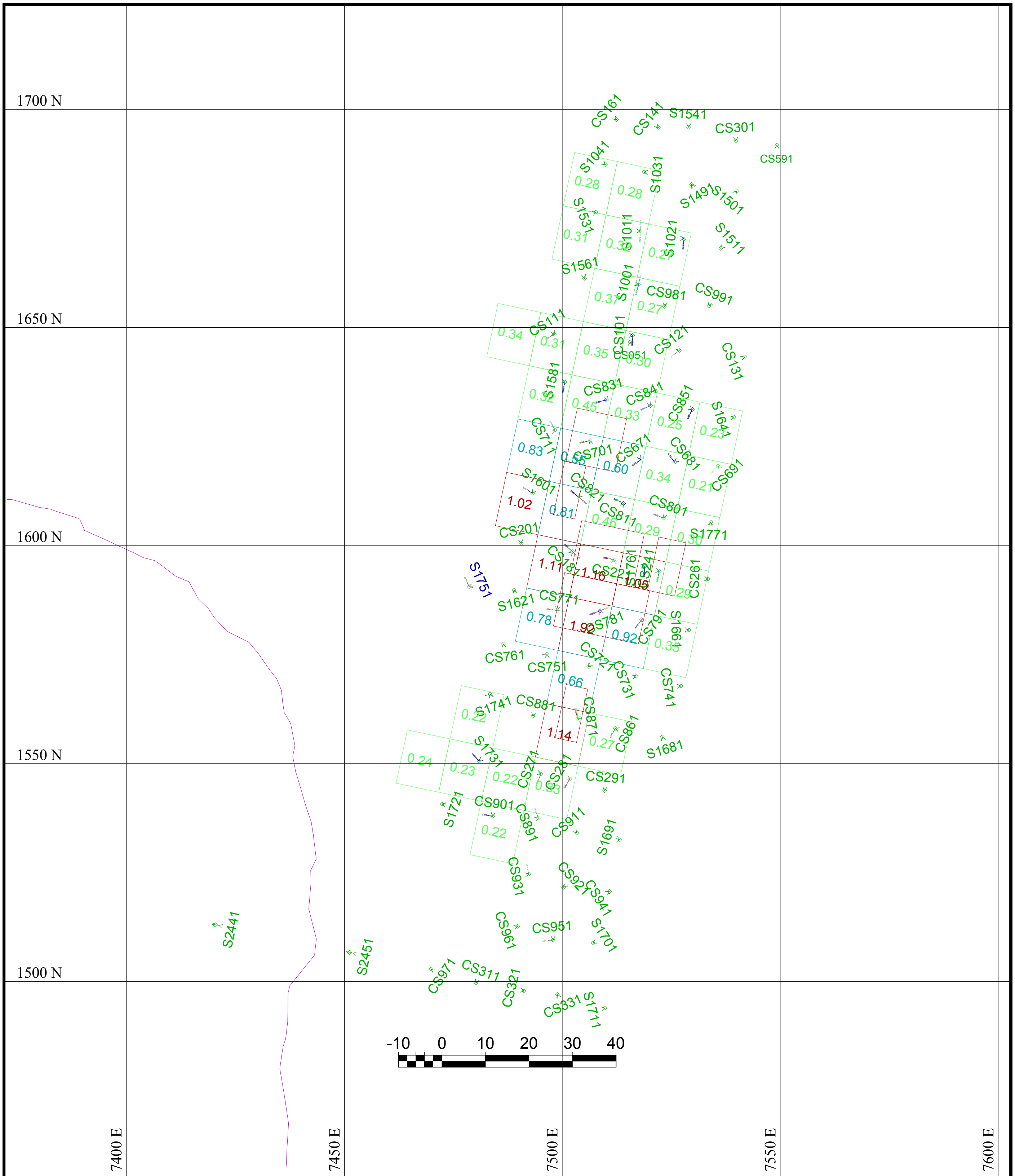

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Figure 11-10
Denison Mines Limited
Saskatchewan Projects
Sue A Deposit
Block Model Plan 383 m RL

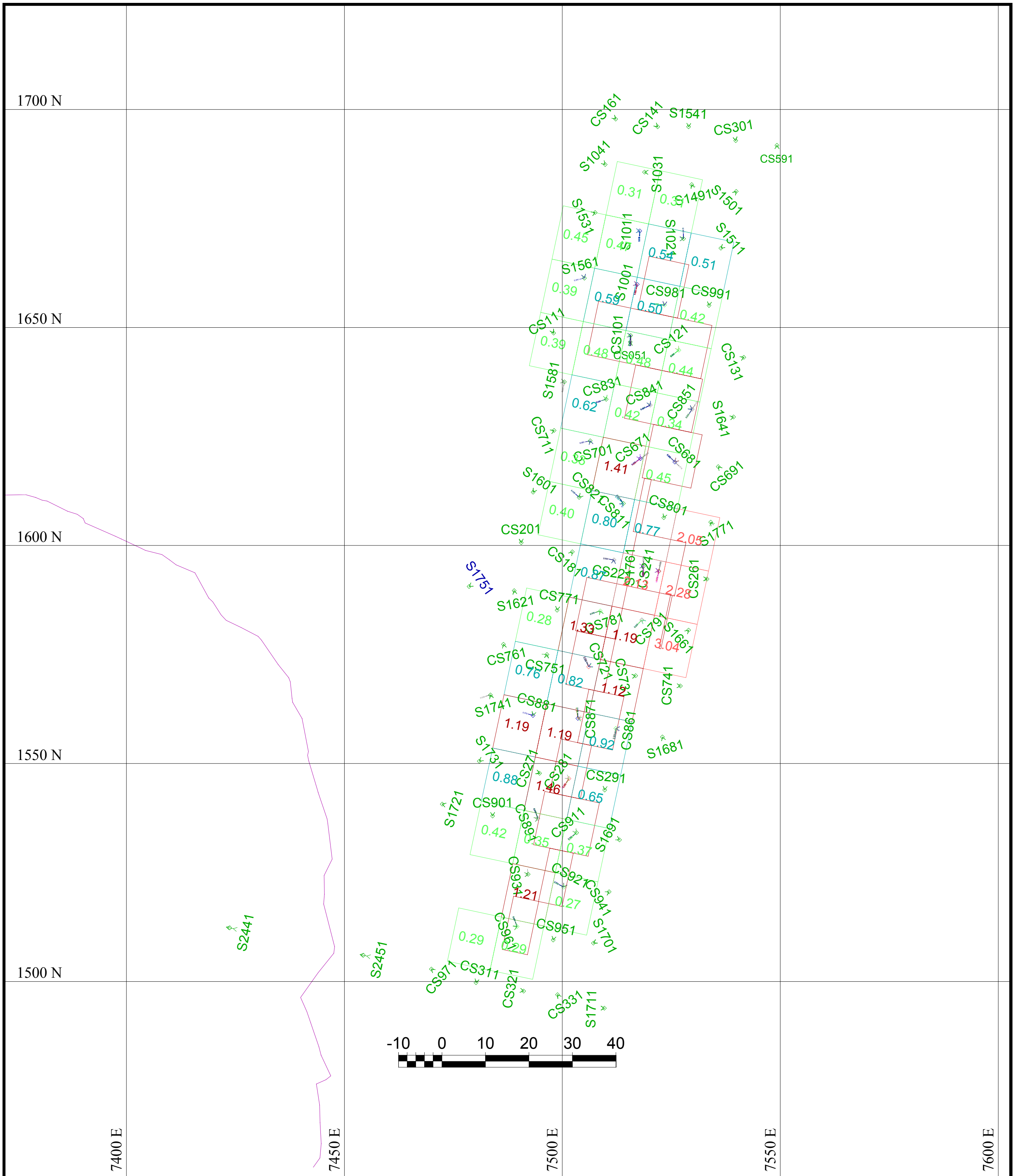

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Figure 11-11
Denison Mines Limited
Saskatchewan Projects
Sue A Deposit
Block Model Plan 389 m RL

2005 Methodology

Assays (analyses) were composited to 3 m down hole.

TABLE 11-3 STATISTICS OF COGEMA 3 M COMPOSITES (2005)
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Number of 3 m Composites	153	
	U kg/t	**U_3O_8 kg/t**
Mean Grade	4.97	5.86
Minimum Grade	0.1	0.12
Maximum Grade	64.35	75.88
Variance	125.7	148.23

The composites differ in number and somewhat in statistics compared to 1998. The difference likely arises from the use of a wireframe in their generation in 1998, which may have caused composites to incorporate higher waste locally.

A selective mining unit (SMU) was taken at a block size of 2.5 m x 2.5 m x 3 m vertical (18.75 m^3). In lieu of wireframing, indicator kriging (IK) of composites was utilized to interpolate the probability of mineralization occurrence in the SMU blocks; for example,. blocks were assigned a probability of 1 if containing grade above a cut-off of $1.0^{o}/_{oo}$ U or 0 if not. In this manner, 2,965 blocks (±55,600 m^3) were populated from 42,635 blocks (±799,410 m^3) contained in the model volume. The IK probabilities in the SMU blocks were imported to larger blocks of 10 m x 12.5 m x 3 m (375 m^3) within the model and used to assign a proportion of "ore" to each large block and thereby create a percent model.

Results of Cogema nested spherical model variography on composites in 2005 is as follows:

Parameter	Spherical 1	Spherical 2
Nugget	18.45 (16%)	
Sills	6.35 (5.5%)	90.2 (78.4%)
EW range (m)	4.5	7.0
NS Range (m)	4.5	6.16
Vertical Range (m)	3.0	7.0

Ordinary kriging interpolated grade to the 10 m x 12.5 m x 3 m blocks. Interpolation search was expanded in a series of passes to populate all blocks. Approximately 75% of the blocks were estimated from ≥5 composites.

TABLE 11-4 STATISTICS OF COGEMA BLOCKS (2005)
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Number of Blocks	296	
	U kg/t	U_3O_8 kg/t*
Mean Grade	4.92	5.80
Minimum Grade	0.2	0.24
Maximum Grade	22.7	26.8
Variance	16.8	19.8

*U kg/t converted to U_3O_8 for reference

Bulk density was calculated from the formula used for the 1998 estimate. The same formula was used for other McClean Lake project resources: density=1/(0.452-(0.00326*%U_3O_8)).

After applying the percent model, the Cogema 2005 global resource model is estimated at 123, 555 tonnes averaging 5.13 U kg/t (0.60% U_3O_8). The global grade does not differ significantly from the average of the composites, which is 4.97 U kg/t, and the average of all blocks, which is 4.92 U kg/t.

Uniform conditioning was utilized to estimate recoverable resources at various cut-off grades for a selective mining unit (SMU) of a 2.5 m x 6 m x 3 m vertical block, i.e., at a volume change of support of 8.3:1. The change of support coefficients for uniform conditioning are as follows:

- composites/SMU r_v=0.75
- composites/panels r_v=0.51
- SMU/panel r_v/r_v=0.68

TABLE 11-5 2005 COGEMA RESOURCE ESTIMATE FOR SUE A
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

U kg/t Cut-Off Grade	Tonnes	U kg/t	U_3O_8 (lbs x1,000)
>0 $^o/_{oo}$ *	123,555	5.1 $^o/_{oo}$	1,648
1.0 $^o/_{oo}$	87,396	7.0 $^o/_{oo}$	1,602
3.0 $^o/_{oo}$	53,907	10.0 $^o/_{oo}$	1,441
4.0 $^o/_{oo}$	44,919	12.0 $^o/_{oo}$	1,360
5.0 $^o/_{oo}$	38,146	13.0 $^o/_{oo}$	1,281
7.5 $^o/_{oo}$	26,626	16.0 $^o/_{oo}$	1,097
10.0 $^o/_{oo}$	19,343	18.0 $^o/_{oo}$	933

*Global model

The 2005 cut-off grades are approximately 18% higher than the Cogema 1998 cut-offs at nearly comparable values (e.g. 0.1% U_3O_8 versus 1.0 $^o/_{oo}$ U); however, the tonnes and lbs of U_3O_8 are reasonably similar and mutually support the two estimates (Figure 11-12). Cogema did not classify the resources to CIM or any other regulatory standard.

DENISON AUDIT AND INDEPENDENT ESTIMATE OF 2003

Denison carried out an independent resource estimation for Sue A as part of its audit of Cogema resources (Kerr et al., 2003). A cut-off grade of 0.1% U_3O_8 was used to contour and wireframe the Sue A deposit in 3D space. The contours were interpreted on 12.5 m cross-sections and one inclined isopac section. The contours were digitized and then a wireframe was constructed by extrusion between cross-sections.

The model was rotated 12º east. Resource blocks were 4 m grid east-west, 4 m grid north-south, and 2 m vertical. Assay/analysis intervals were composited to 2 m within the wireframes to produce 100 composites. Block grades were interpolated by inverse distance cubed (ID^3) since no grade subdomains were used. Therefore Kerr et al. (2003) believed that further constraint was needed on clustered, uncut high grade composites.

Kerr at al. used a calculated SG for bulk density derived by Kilborn (1990) below:

$$SG=1/(0.452-(.00326*(\%U_3O_8+\%Ni+\%As)))$$

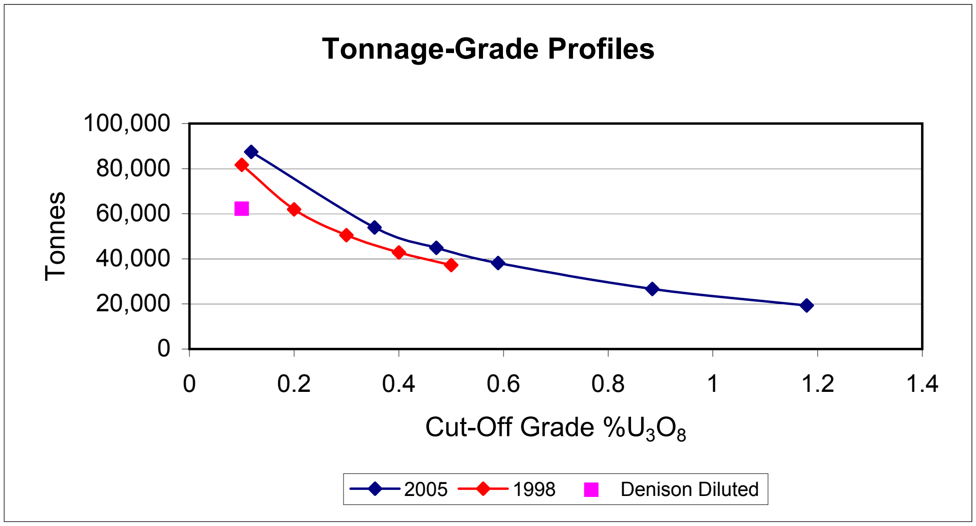
This is basically the same as the uranium-only Cogema formula for low to negligible nickel and arsenic values. The Kerr et al. SG increases, however, with respect to the Cogema calculated value, when these other metals are present in significant amounts. RPA notes that where uranium mineralization is absent or grades are low, the Kerr et al. and Cogema formulas calculate bulk density at 2.21 t/m^3. This appears to be low for sandstone.

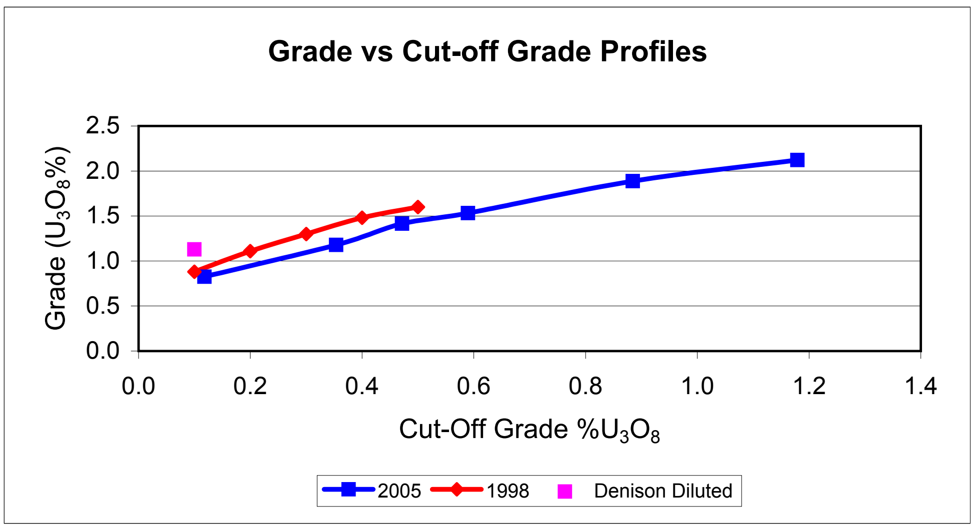
Volume to tonnage conversion used a 2.37 t/m^3 average bulk density; no density model was created. This density appears to be close to the average SG of 2.39 calculated for assays.

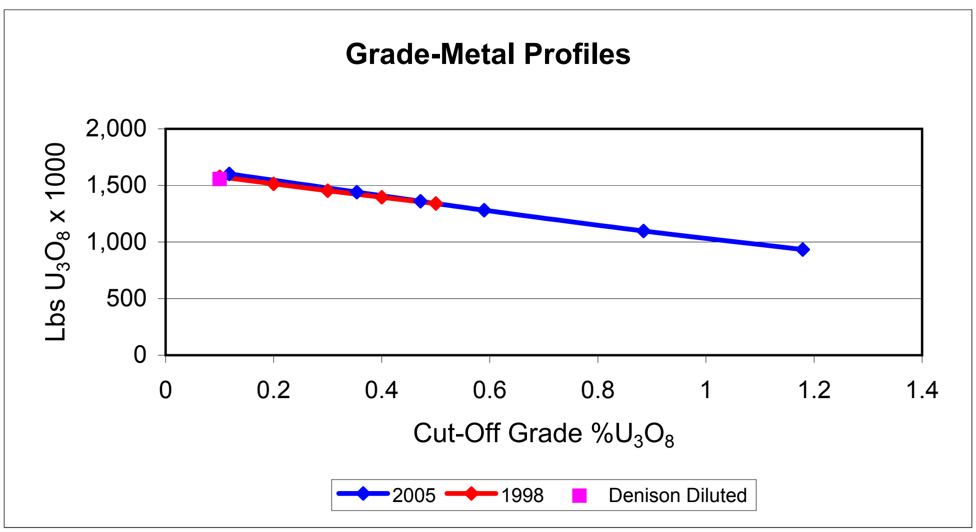
Dilution from overbreak was calculated by creating a 1 m shell around the main resource wireframe and estimating the tonnes and grade within the shell.

Denison estimated the in-place mineralization at 40,215 tonnes grading 1.72% U_3O_8, 3.86% Ni, and 4.84% As. Including dilution of 22,033 tonnes at 0.04% U_3O_8, 0.35% Ni, and 0.36% As, the in-place diluted resource was estimated at 62,248 tonnes at 1.13% U_3O_8, 2.62% Ni, and 3.24% As. The latter represents 1,556 kilo pounds of U_3O_8 and is close to the contained metal estimates of Cogema for a comparable cut-off grade. RPA notes that the tonnages, for a density of 2.37 t/m^3, are consistent with the volume of the Gemcom solids.

Figure 11-12 Resource Estimates Tonnage-Grade-Metal Profiles
Denison Mines Ltd. Sue A Deposit, Saskatchewan







RESOURCE AUDIT AND MODEL VALIDATION

Since Cogema has not interpreted a constraining mineralization envelope or wireframe in their resource modeling work, RPA reviewed the Kerr et al. (2003) uranium mineralization wireframe constructed using a 0.1% U_3O_8. A sectional extrusion method was used. Some boundaries of the wireframe do not appear to "snap" directly to the assay intervals (i.e., they may have an offset in the intervals position as discussed under Data Verification). Nevertheless, the wireframe spatially honours the mineralization reasonably well, has been generated in a conventional manner, and may be somewhat conservative. In RPA's opinion, only minor improvement could be made to the wireframe model and this would have little impact in terms of overall resource estimation error. RPA has accepted the Kerr et al (2003) wireframe as a reasonable representation of the in situ mineralization for the purpose of auditing the Sue A resource.

In order to validate the model and the previous resource estimates, RPA carried out an independent check estimate as follows:

- RPA carried out statistical analysis of the raw analyses contained in the wireframe.
- Raw analyses were composited down hole within the wireframe to 3 m intervals by length and SG weighting. The 3 m length is the same as Cogema's, but larger than the 2 m used by Kerr et al (2003). SG/bulk density was calculated using the Kerr et al (2003) formula that includes U_3O_8, Ni, and As.
- A 3-D block model was constructed with block (panel) dimensions of 10 m x 12.5 m x 3 m, the same as in the previous estimates.
- U_3O_8, Ni, and As grades were interpolated to resource blocks by inverse distance squared (ID^2) based on a 15 m x 15 m x 15 m search and block grade population criteria of two composites minimum and six composites maximum.
- Statistics for raw analyses, composites, and blocks were compared to validate the block model grade interpolation.
- The RPA check estimate was compared to the previous estimates.

STATISTICS

Table 11-6 shows statistics for raw analyses, and 2 m composites within the wireframe for the Kerr et al. (2003) drill hole database. Figures 11-13 and 11-14 show cumulative frequency log probability plots of all U_3O_8 analyses, and analyses within the Kerr et al. (2003) 0.1% U_3O_8 wireframe, respectively.

Statistics for raw analyses, RPA 3 m composites and the RPA block model are shown in Table 11-7. The number of raw analyses within the wireframe differs between the databases (Table 11-6 versus Table 11-7) because of split intervals. The difference in number of composites between the tables arises from the different lengths, i.e. 2 m versus 3 m.

With respect to the Kerr et al. (2003) estimate, the check estimate by RPA at a 0.1% U_3O_8 cut-off is 2.3% lower in tonnes, 1.2% higher in U_3O_8 grade, and 1.2% lower in contained metal. These differences are small and well within the variance expected from use of different composite lengths and ID interpolation power. The RPA estimate therefore confirms the Kerr et al. (2003) estimate. Compared to the Cogema 1998 estimate at the same cut-off grades, the RPA (and Kerr et al, 2003 audit estimate) check estimate reports half the tonnes (-52%) and twice (+98%) the grade at a 0.1% U_3O_8. The contained U_3O_8, however, only varies from -4% to +8% with the increase in a cut-off grade and is not significantly different given the difference in the modelling and interpolation approaches.

Table 11-8 summarizes the RPA Sue A Resource Estimate.

Table 11-6 Sue A Deposit Statistics
Denison Mines Ltd. Saskatchewan Uranium Projects

All Database Stats*	Length (m)	%U_3O_8	%Ni	%As	%Co	SG_OLD	SG-COMP
Count (N)	2,308	2,308	2,308	2,308	2,308	2,308	2,308
Sum	893	-	-	-	-	-	-
Minimum	0.05	0.00	0.00	0.00	0.00	0.00	2.21
Median	0.50	0.00	0.01	0.01	0.00	2.21	2.21
Maximum	4.50	30.36	33.01	37.15	8.70	4.51	4.51
Range	4.00	30.36	33.01	37.15	8.70	2.30	2.30
Mean	0.39	0.31	0.74	0.92	0.04	2.25	2.25
Len. Wtd Mean	-	0.31	0.83	1.03	0.05	2.25	2.26
SG-Len. Wtd Mean	-	0.38	1.03	1.29	0.06	-	-
Variance	0.03	2.97	6.73	11.01	0.18	0.03	0.03
Standard Deviation	0.17	1.72	2.59	3.32	0.42	0.17	0.16
Coefficient of Variation	0.44	5.52	3.51	3.61	10.38	0.08	0.07
25th Percentile	0.25	0.00	0.00	0.00	0.00	2.21	2.21
75th Percentile	0.50	0.03	0.29	0.27	0.00	2.22	2.22
90th Percentile	0.50	0.31	1.50	1.87	0.01	2.27	2.27
95th Percentile	0.50	0.98	3.94	4.78	0.04	2.39	2.39
97th Percentile	0.50	2.12	6.40	7.87	0.08	2.54	2.54
98th Percentile	0.50	3.30	8.47	11.18	0.13	2.70	2.70
99th Percentile	1.00	9.54	14.96	17.80	0.73	3.08	3.08
99.5th Percentile	1.00	14.15	19.25	25.36	1.96	3.44	3.43

Stats U_3O_8>=0.1%*	Length (m)	%U_3O_8	%Ni	%As	%Co	SG_OLD	SG-COMP
Count (N)	374	374	374	374	374	374	374
Sum	150	-	-	-	-	-	-
Minimum	0.05	0.10	0.00	0.00	0.00	2.21	2.21
Median	0.50	0.47	1.25	1.69	0.01	2.27	2.27
Maximum	0.75	30.36	33.01	37.15	8.70	4.51	4.51
Range	0.25	29.89	33.01	37.15	8.70	2.24	2.24
Mean	0.40	1.87	3.32	4.18	0.13	2.41	2.41
Len. Wtd Mean	-	1.75	3.47	4.46	0.16	2.42	2.42
SG-Len. Wtd Mean	-	2.08	4.27	5.45	0.19	-	-
Variance	0.02	15.46	27.41	41.56	0.62	0.13	0.12
Standard Deviation	0.14	3.93	5.24	6.45	0.79	0.35	0.35
Coefficient of Variation	0.35	2.10	1.57	1.54	6.00	0.15	0.14
25th Percentile	0.25	0.22	0.33	0.43	0.00	2.23	2.23
75th Percentile	0.50	1.42	3.90	4.85	0.02	2.42	2.42
90th Percentile	0.50	4.44	8.88	11.15	0.07	2.77	2.79
95th Percentile	0.50	10.03	15.14	17.69	0.24	3.09	3.09
97th Percentile	0.50	14.15	18.65	24.34	0.75	3.43	3.43
98th Percentile	0.50	14.74	21.89	26.55	1.11	3.77	3.65
99th Percentile	0.50	19.07	25.45	30.44	3.60	4.08	4.05
99.5th Percentile	0.50	23.98	28.16	36.14	7.26	4.15	4.13

Stats All-Ore Zone*	Length (m)	%U_3O_8	%Ni	%As	%Co	SG_OLD	SG-COMP
Count (N)	430	430	430	430	430	430	430
Sum	176	-	-	-	-	-	-
Minimum	0.05	0.00	0.00	0.00	0.00	2.21	2.21
Median	0.50	0.28	0.74	1.11	0.01	2.25	2.25
Maximum	1.00	30.36	33.01	37.15	8.70	4.51	4.51
Range	0.50	30.08	33.01	37.15	8.70	2.26	2.26
Mean	0.41	1.57	2.79	3.52	0.11	2.38	2.38
Len. Wtd Mean	-	1.43	2.90	3.74	0.13	2.39	2.38
SG-Len. Wtd Mean	-	1.73	3.64	4.65	0.16	-	-
Variance	0.02	13.90	24.73	37.63	0.54	0.11	0.11
Standard Deviation	0.14	3.73	4.97	6.13	0.73	0.33	0.33
Coefficient of Variation	0.35	2.38	1.78	1.74	6.59	0.14	0.14
25th Percentile	0.25	0.09	0.19	0.21	0.00	2.22	2.22
75th Percentile	0.50	0.99	2.87	3.67	0.02	2.35	2.35
90th Percentile	0.50	3.43	8.09	9.73	0.05	2.66	2.65
95th Percentile	0.50	9.65	13.99	17.16	0.11	3.06	3.06
97th Percentile	0.50	14.05	18.27	22.26	0.45	3.33	3.29
98th Percentile	0.50	14.74	20.31	25.32	0.92	3.51	3.44
99th Percentile	0.50	17.76	23.54	29.29	3.14	4.03	3.98
99.5th Percentile	0.71	23.24	25.57	35.34	6.64	4.14	4.11

Composites Statistics*				
Statistic	Length (m)	Ni%	As%	Calc. SG
Count (N)	99	99	99	99
Sum (m)	167.41	-	-	-
Minimum	0.49	0.00	0.00	2.21
Median	2.00	1.20	1.83	2.27
Maximum	2.00	19.87	26.46	3.23
Range	1.51	19.87	26.46	1.02
Mean	1.69	3.15	4.07	2.38
Len. Wtd Mean	-	3.25	4.17	2.38
Variance	0.29	20.64	34.71	0.07
Standard Deviation	0.53	4.54	5.89	0.26
Coefficient Variation	0.32	1.44	1.45	0.11
25th Percentile	1.49	0.48	0.42	2.23
75th Percentile	2.00	3.79	5.01	2.40
90th Percentile	2.00	8.88	11.02	2.65
95th Percentile	2.00	16.38	18.78	3.05
97th Percentile	2.00	16.71	21.08	3.19
98th Percentile	2.00	16.84	21.81	3.22
99th Percentile	2.00	18.00	23.15	3.23
99.5th Percentile	2.00	18.94	24.81	3.23

*from Kerr et al. (2003) drill hole database

Table 11-7 Statistics for Wireframe Raw Analyses and RPA 3 m Composit
Denison Mines Ltd. Sue A Uranium Deposit, Saskatchewan

Statistics of Raw Analyses for Sue A Wireframe (Cogema database)

Statistic	Length (m)	U308 ppt	Ni ppt	As ppt	Co ppt	Calc. SG
Count (N)	424	424	424	424	424	424
Sum (m)	176.85	-	-	-	-	-
Minimum	0.05	0.02	0.03	0.02	0.00	2.21
Median	0.50	2.83	7.42	11.12	0.06	2.25
Maximum	1.00	303.60	330.10	371.50	87.00	4.51
Range	0.95	303.58	330.07	371.48	87.00	2.30
Mean	0.42	15.71	28.16	35.57	1.13	2.38
L Wtd Mean	-	14.17	28.91	37.26	1.33	2.38
SG-L Wtd Mean	-	17.47	36.36	46.41	1.63	-
Variance	0.02	1,409	2,501	3,806	54.44	0.11
Standard Deviation	0.14	37.53	50.01	61.70	7.38	0.34
Coefficient of Variation	0.34	2.39	1.78	1.73	6.55	0.14
25th Percentile	0.25	0.85	1.90	2.14	0.02	2.22
75th Percentile	0.50	9.90	28.87	36.85	0.16	2.35
90th Percentile	0.50	35.02	81.29	98.86	0.51	2.67
95th Percentile	0.50	96.62	143.29	171.88	1.11	3.07
97th Percentile	0.50	140.67	183.71	226.29	4.82	3.35
98th Percentile	0.50	147.39	205.18	254.18	9.26	3.51
99th Percentile	0.50	178.68	235.54	294.40	32.11	4.03
99.5th Percentile	0.72	233.09	255.77	354.93	67.34	4.14

Statistics of RPA 3 m Composites for Sue A Wireframe

Statistic	Length (m)	U308 ppt	Ni ppt	As ppt		Calc. SG
Count (N)	66	66	66	66	-	66
Sum (m)	158.87	-	-	-	-	-
Minimum	0.47	0.87	0.29	0.22	-	2.21
Median	3.00	5.11	14.09	19.11	-	2.29
Maximum	3.00	92.58	195.52	247.86	-	3.21
Range	2.53	91.71	195.23	247.64	-	1.00
Mean	2.41	16.55	35.00	44.20	-	2.40
SG Wtd Mean	-	18.23	39.32	50.32	-	2.40
Variance	0.76	654	2,229	3,603	-	0.07
Standard Deviation	0.87	25.57	47.22	60.03	-	0.27
Coefficient of Variation	0.36	1.55	1.35	1.36	-	0.11
25th Percentile	2.06	2.54	5.45	6.08	-	2.23
75th Percentile	3.00	14.73	38.39	49.17	-	2.41
90th Percentile	3.00	69.00	109.15	144.37	-	2.90
95th Percentile	3.00	84.68	147.77	187.68	-	3.02
97th Percentile	3.00	87.30	150.29	209.47	-	3.12
98th Percentile	3.00	87.88	160.26	213.03	-	3.17
99th Percentile	3.00	89.70	175.60	226.29	-	3.20
99.5th Percentile	3.00	91.14	185.56	237.07	-	3.20

Statistics of RPA Block Model Blocks for Sue A Wireframe

Statistic	U308 ppt	Ni ppt	As ppt	SG
Count (N)	152	152	152	152
Minimum	1.04	1.92	1.97	2.22
Median	9.71	27.96	35.41	2.37
Maximum	62.19	165.29	170.70	3.03
Range	61.15	163.37	168.73	0.81
Mean	16.15	35.00	43.91	2.40
Tonnage Wtd Mean	17.34	36.37	44.45	2.41
Variance	212	964	1,394	0.03
Standard Deviation	14.56	31.04	37.34	0.16
Coefficient of Variation	0.90	0.89	0.85	0.07
25th Percentile	5.16	10.05	13.09	2.28
75th Percentile	24.72	51.60	61.53	2.48
90th Percentile	38.51	76.48	100.82	2.61
95th Percentile	47.79	98.85	122.87	2.71
97th Percentile	51.66	104.63	131.06	2.74
98th Percentile	53.05	112.25	139.37	2.80
99th Percentile	57.61	139.39	157.43	2.91
99.5th Percentile	61.49	160.55	169.81	3.02

FIGURE 11-13
Denison Mines Ltd. Sue A Deposit, Saskatchewan
Cumlative Frequency % Log Probability Plot U_3O_8%

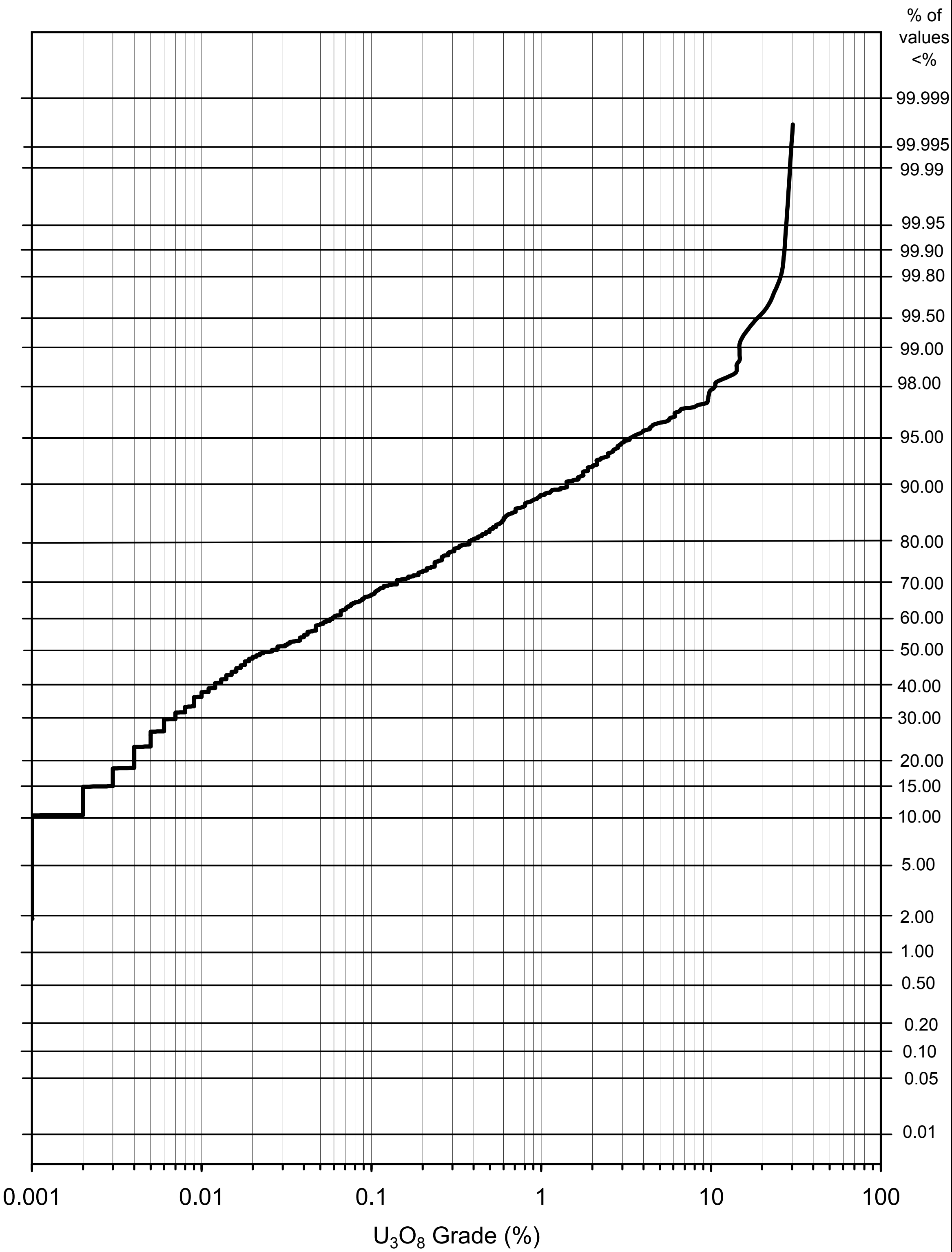
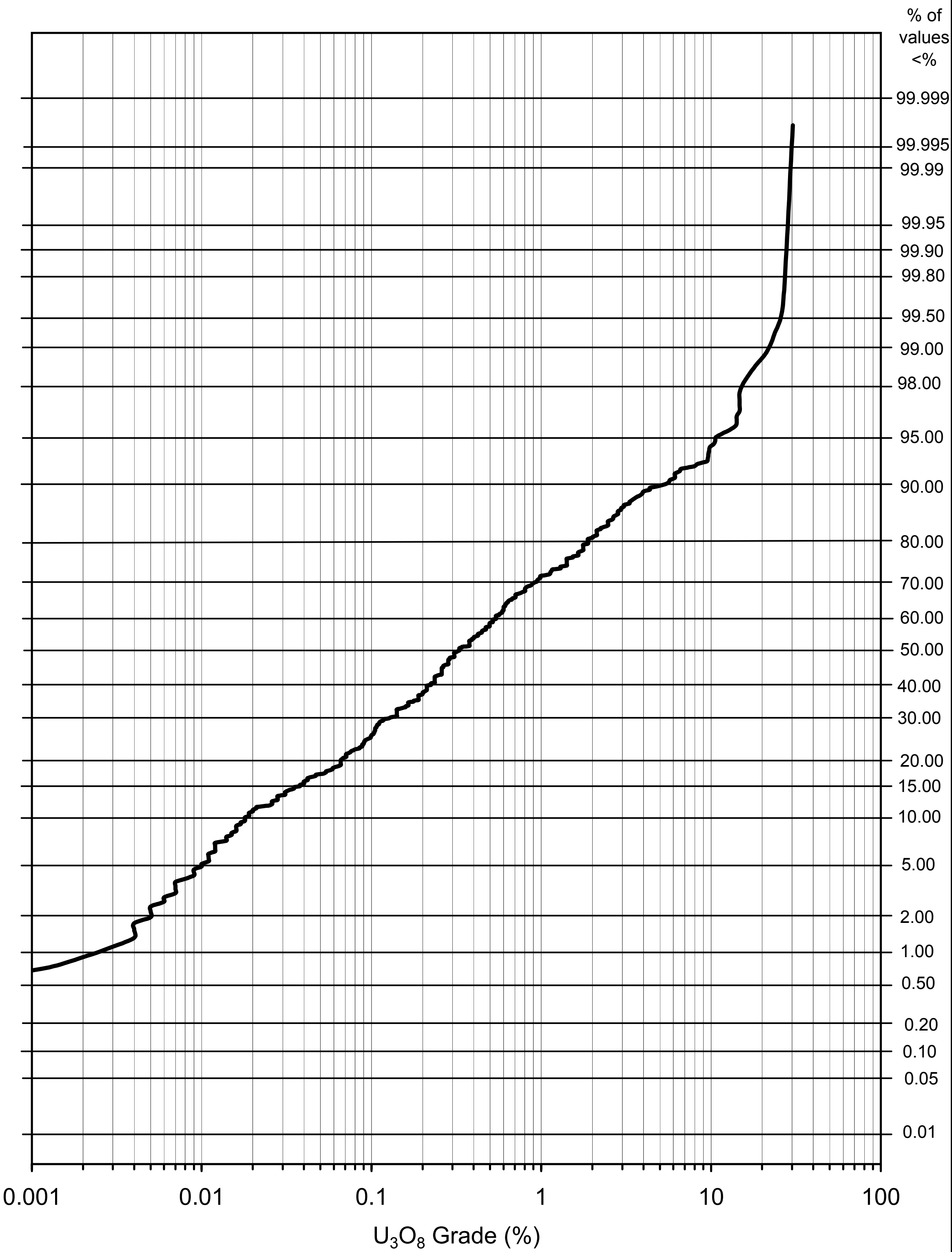


FIGURE 11-14 Kerr et al (2003) Analyses Within Wireframe
Cumlative Frequency % Log Probability Plot U_3O_8%
Denison Mines Ltd. Sue A Deposit, Saskatchewan



For the purpose of reserve definition and Whittle open pit optimization and design, RPA prefers a constrained wireframe model and therefore has used the RPA estimate. Based on RPA's review of U_3O_8 prices and mining operating costs, the 0.1% U_3O_8 cut-off grade is reasonable for conversion to Mineral Reserves.

TABLE 11-8 SUE A RESOURCE ESTIMATE

McClean Lake Joint Venture McClean Lake Property, Saskatchewan*

Cut-Off Grade U3O8%	Indicated Resource				
	Tonnes	U_3O_8%	Ni%	As%	U3O8 lbs
0.1%	**39,284**	**1.74**	**3.64**	**4.45**	**1,507,000**
0.2%	38,265	1.78	3.69	4.48	1,502,000
0.3%	37,504	1.81	3.76	4.56	1,491,000
0.4%	33,991	1.96	4.00	4.85	1,469,000
0.5%	31,928	2.06	4.15	5.04	1,450,000

*Denison Mines Inc. holds 22.5% interest in the MLJV and the above Resource

INTERPRETATION AND CONCLUSIONS

The Cogema database is reasonable for resource estimation. The unconstrained deposit modelling, large panel ordinary kriging, and uniform conditioning approach results in generally higher tonnage and lower grades but similar contained U_3O_8 pounds compared to the constrained wireframe and inverse distance check methods.

The significant differences between the kriging/uniform conditioning and inverse distance estimation approaches are the spatial distribution of in situ metal (constraining wireframe versus no wireframe and indicator modelling) and the resource grade differences that will impact on the economics related to mining cost/lb U_3O_8 extracted.

For the purpose of reserve definition and Whittle open pit optimization and design, RPA prefers a constrained wireframe model and therefore has used the RPA estimate. RPA has not added dilution to the resource estimate as was done by Kerr et al. (2003). The wireframe model includes internal dilution and the Kerr et al. (2003)1 m envelope introduces in excess of 50% dilution that is not supported in open pit mining practice where radiometric probing of blast holes and bench faces provides dilution control.

MINERAL RESERVE ESTIMATES

MINING

The Sue A deposit is scheduled to be mined in 2005. As the current Sue C ore stockpile declines through the ongoing draw down of feed materials for the JEB mill, ore materials from Sue A pit first, and then subsequently Sue E pit, will be mined and delivered to the mill stockpile to sustain operations. The Sue A and Sue E mining plans are extensions of the mining operations already carried out at the JEB and Sue C pits.

Open pit mining operations will start in 2005 with the stripping of overburden and waste rock materials. RPA has developed a revised resource block model for the Sue A deposit. The RPA deposit model is similar to the model developed previously by Denison in 2003, and has been used as the basis for reporting the estimated quantities of ore materials to be recovered from the Sue A pit based on the Denison pit design developed in support of the 2003 Technical Report. RPA has not added dilution to the resources since mining and the pit design are relatively insensitive to dilution. In RPA's opinion, these pit design limits are reasonable and appropriate for the mining and recovery of the Sue A deposit.

The Mineral Reserve at Sue A has been calculated based on the RPA resource model and the Denison ultimate pit design, and is summarized in Table 11-9 below. On the basis of the estimates and forecasts presented, RPA concludes that the Mineral Reserves are consistent with the definitions set out in NI 43-101 and defined by the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by the CIM Council on August 20, 2000.

TABLE 11-9 SUE A PROBABLE RESERVE (AS OF JAN.1, 2005)

McClean Lake Joint Venture McClean Lake Property, Saskatchewan*

Total Material (BCM)	Waste (BCM)	Special Waste (BCM)	Ore (Tonnes)*	U_3O_8 Grade (%)	U_3O_8 lbs*
947,103	914,568	19,308	31,948	1.99%	1,402,000

*Denison Mines Inc. holds 22.5% interest in the MLJV and the above Reserve

The proposed Sue A pit limit is shown on the upper right corner of the existing Sue C open pit in the Figure 11-15.

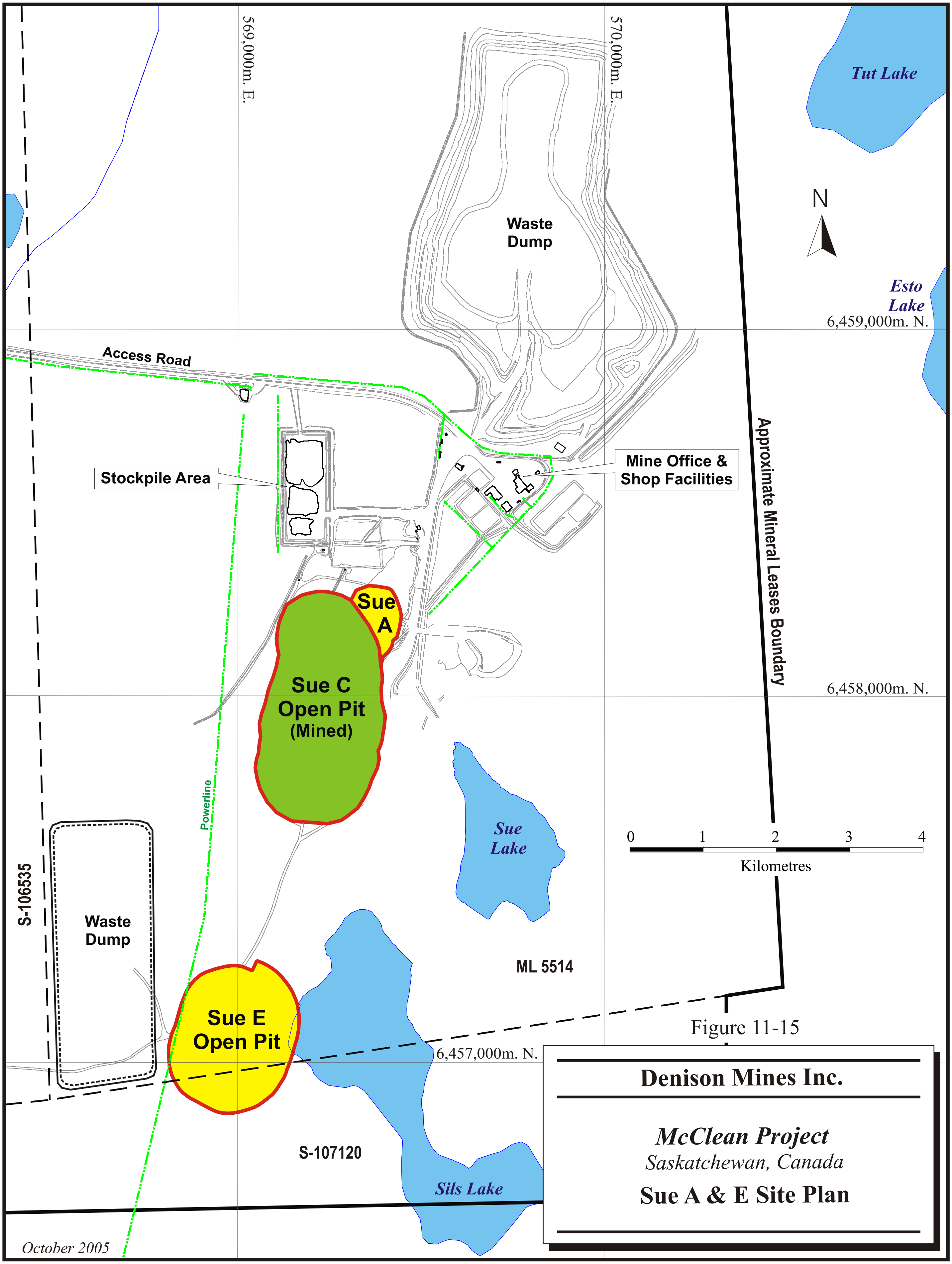


Figure 11-15

Denison Mines Inc.

McClean Project
Saskatchewan, Canada

Sue A & E Site Plan

CUT-OFF GRADE

The MLJV has historically operated at the Sue C deposit based on an ore cut-off grade of 0.1% U_3O_8 to define ore scheduled for processing at the JEB mill facility versus discard material. RPA reviewed this cut-off grade against the current economic factors, including operating costs, metallurgical recovery, and revenue criteria, and determined that it represents a reasonable cut-off level. For the purposes of this analysis, RPA has applied the MLJV cut-off policy in calculating Mineral Reserves.

MINE PRODUCTION PLAN AND SCHEDULE

In the Sue A deposit, the initial mining activity will primarily involve the excavation of waste materials, both barren and contaminated waste, above the ore zones. As mining progresses downward, the proportion of waste mined will decrease and the quantities of ore recovered will increase. The Sue A pit is approximately 80 metres deep and is located on the north east flank of the existing Sue C pit. Access into the Sue A pit will be achieved via the existing Sue C pit ramp. Mining operations will be carried out using 12 metre high benches.

MINE SITE PLAN

A general arrangement plan for the mine site providing for access roads, waste dump facility, special waste stockpile area, and ore blending yard is illustrated in Figure 11-15. The waste rock material mined in the course of developing the Sue A pit will be dumped in the nearby Sue C waste dump, while special waste materials will be dumped into the Sue C pit. Ore materials will be truck hauled out of the pit and hauled to the JEB mill site for deposition in the ore stockpile yard. The arrangement will help manage site drainage and facilitate ditching and settlement pond design.

The Sue A pit mining operations are expected to be completed over a period of approximately nine months and the ore materials mined will be delivered to the existing Sue C stockpile adjacent to the JEB mill for subsequent feeding into the processing facilities.

Special waste material has been designated as waste rock containing a minimum of 0.03% U_3O_8 and a maximum of 0.10% U_3O_8. Above this level the material is classified as ore. When special waste is encountered in the course of mining, it will be mined selectively from the barren waste material and deposited directly into the existing Sue C open pit for disposal below the water table in order to prevent oxidation.

MINING EQUIPMENT FLEET

The mining operations at Sue A will be carried out using the existing equipment fleets at the McClean Lake operating site. There are existing equipment maintenance, office, and site management facilities existing at the Sue C mine site. These same facilities will be used to support the mining operations developed at Sue A.

Operating cost estimates for the Sue A operations are based on the most recent operating cost experience at the Sue C operations (2002). The equipment is listed in Table 11-10 below. The mine working schedule is based on two twelve hour shifts per day, 350 operating days per year.

TABLE 11-10 MOBILE EQUIPMENT FLEET
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Description	Quantity
O&K RH120 Shovel	1
Hitachi 1100 Backhoe	1
Cat 777 Haul Trucks	5
Cat D9 Dozer	2
Cat 16G Grader	1
Water Truck	1
Pickup Trucks	6
Fuel Truck	1
Service Truck	1
Subtotal	**19**

CAPITAL COSTS

No substantial capital costs are expected to be incurred for the development of the Sue A deposit as open pit mining operations since the site has already been developed (the Sue C deposit has been mined) and the equipment and support facilities already exist. The Sue A mining plan will be an extension of the previous operations that were carried out at Sue C.

OPERATING COSTS

Operating costs have been estimated based on the previous actual operating experience at the McClean Lake Sue C mine operations. Mining costs have been estimated based on the actual production and operating experience at the MLJV during the mining of the Sue C open pit. They are forecast to average approximately $4.15 per tonne of material excavated.

12 SUE B

PROPERTY GEOLOGY

The Sue B deposit is located 350 metres north of the Sue A deposit and, like Sue A, occurs along the western flank of a basement horst which has 8 m to 10 m of relief (Kerr et al. 2003). The basement horst at Sue B is more prominent than at Sue A, and the basement is at a slightly shallower depth of 60 m to 98 m. The Sue B mineralization is largely fault and fracture controlled and, similarly to Sue A, straddles the unconformity in a "lower" lens. Additional movement that occurred along the normal and reverse faults parallel to the east-dipping gneiss foliation likely resulted in fracturing of the sandstones in the upper Athabaska stratigraphic column and introduction of uranium mineralization that is perched high in the sandstones as an "upper" lens.

The Sue B upper sandstone-hosted lens is approximately 17 m to 43 m deep, whereas the lower lens dips gently west parallel to the sandstone-basement contact and lies at depths of approximately 52 m to 82 m. The lenses are separated vertically by 20 m to 35 m of waste. The upper lens is approximately 60 m long and up to 30 m wide, averaging 10 m wide. The lower lens is 30 m to 90 m long and 2 m to 20 m wide. Thickness of the upper lens ranges up to 19 m and the lower lens up to 8 m.

Two drill holes in the deposit, S112 and S128, are weakly to strongly mineralized ($\geq$0.1% U_3O_8) over 23 m to 47 m intervals and indicate the two lenses are connected by vertically extensive mineralization along feeder faults. Because the drilling is mostly vertical, the horizontal limits of the mineralized subvertical faults are not well delineated except for adjacent barren holes on section.

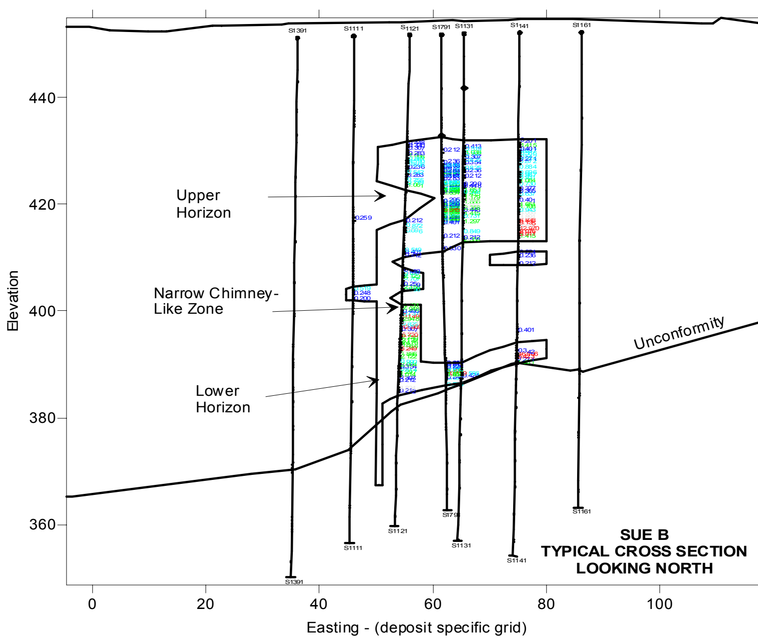
DEPOSIT TYPE

Similar to Sue A, the Sue B is an unconformity-type uranium deposit of the sandstone-unconformity-hosted egress-style in contrast to the basement-hosted Sue C, Sue E, and Sue D deposits to the south. The trend from sandstone-hosted to basement-hosted deposits indicates a south plunge to the uranium mineralization. Other similar style sandstone-hosted deposits in the camp are Cigar Lake, Cluff D, McArthur River, Collins Bay, and Midwest. Deposit genesis at Sue B likely involved mixing of oxidized sandstone brines with relatively reduced fluids issuing along the basement structure into the sandstones where a spatially-stable redox gradient/front was present. The prevailing hydrological conditions controlled the location of fluid interaction relative to the Athabaska-basement unconformity.

MINERALIZATION

The mineralization at Sue B consists primarily of uranium oxides (uraninite and pitchblende), with a suite of nickel-cobalt arsenides (primarily niccolite) hosted in massive, earthly red clays. The sandstone-hosted upper lens contains remnant silicification. U_3O_8 grades in the database range up to 16.5%. Nickel, arsenic, and cobalt grades are variable, with nickel ranging up to 32% and arsenic up to 43%.

FIGURE 12-1 SUE B TYPICAL CROSS SECTION LOOKING NORTH



Source: Denison Mines Inc.

EXPLORATION

The Sue B deposit has been explored only by diamond drilling. CanOxy initially tested the deposit with three holes totalling 348 m prior to 1988. Minatco drilled 19 holes (1,788 m) in 1988 and 48 holes (4,664.5 m) in 1989 bringing the total exploration drilling on the deposit to 6,452.5 m in 70 holes. An additional 587 m in five holes explored the trend between Sue A and Sue B in 1989.

DRILLING

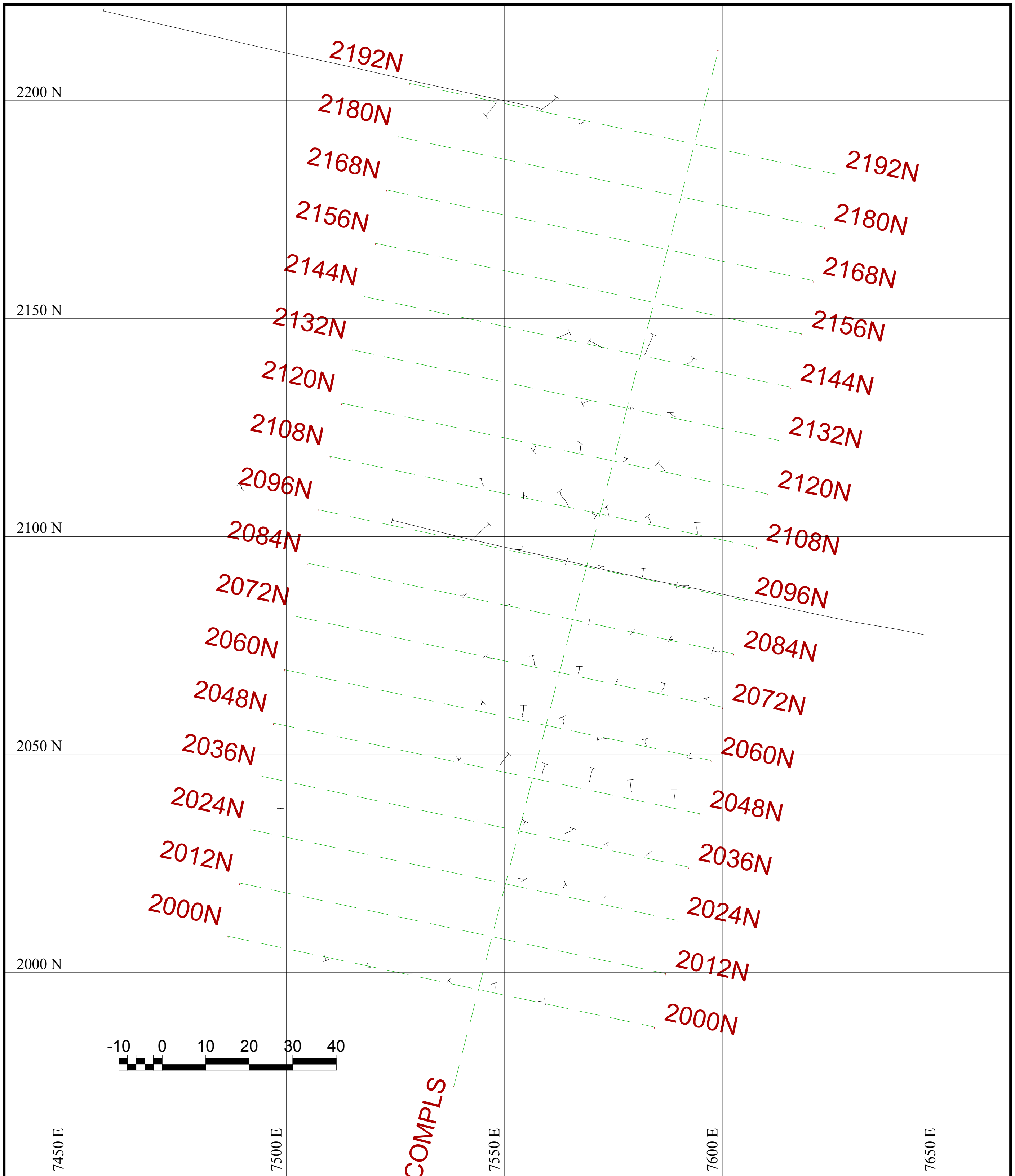
RPA received two drill hole databases for Sue B, one from Cogema and the other from Denison. RPA worked with the Cogema database. This database contains data tables for lithology, analyses, collar locations, down hole surveys, and a wireframe of the Cogema pit. The Denison database also contains mineralization wireframes and topographic, overburden, and unconformity and pit surfaces as well as composites used for resource estimation.

The Cogema digital resource database specific for Sue B contains 71 diamond drill holes totalling 7,094.8 m (Figure 12-2). There are 50 “S” series holes and 21 “CS” series holes. The drilling covers an area of approximately three hectares. The axis of the drill grid is approximately N11ºE. RPA notes that the hole numbers in the digital databases have an additional “1” appended to the original number.

Delineation diamond drilling at Sue B was primarily NQ (47.6 mm), with most holes penetrating 15 m to 25 m or more into the basement. In general, holes were collared on 12.5 m sections with 10 m step outs on section. The section spacing for one tier of holes at the south end of the deposit is 25 m, and, at the north end, the last tier of holes is 50 m north of the resource area. Sixty-nine of the holes (m) were drilled at -90º dip (vertical), and two holes (402 m) in the center and on the northernmost section were drilled at -50º and -60º to the north-northwest. Hole lengths (depths) range from 80 m to 212 m. Drill cross sections are shown in Figures 12-3 to 12-6.

Drill hole collars were surveyed for local grid coordinates and elevation. All holes are vertical, except for two holes that are inclined at -50º and -60º to the north-northwest. Down hole deviation was measured by Sperry-Sun multishot instrumentation. All holes record down hole deviation surveys from one measurement to 10 taken generally below the casing (6 m to 28 m, and possibly 46 m), then at nominal 15 m to 18 m intervals, with a few at 30 m, to the toe. Intervals in the inclined holes are a nominal of 30 m. Six holes have down hole survey records for only the toe and five of these (CS343, 345, 347 and 358, and S108) show no deviation and are likely unsurveyed. This results in some uncertainty with respect to their intercept locations. As determined from the existing deviations for vertical holes at Sue B, a lateral displacement at the toe is potentially up to 5.5 m or half the distance between holes (and half the resource block size).

RPA checked for excessive deviation as an indication of erroneous readings or drilling problems. Seven holes showed possibly excessive dip change (0.065º/m, i.e., 2º/100 ft.) not related to the expected higher deflections below the casing. The traces of these holes were reviewed on screen and they do not appear unreasonable.



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GEOLOGICAL AND MINING CONSULTANTS
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Figure 12-2
Denison Mines Limited
Saskatchewan Projects
Sue B Deposit
Diamond Drill Hole Location Plan

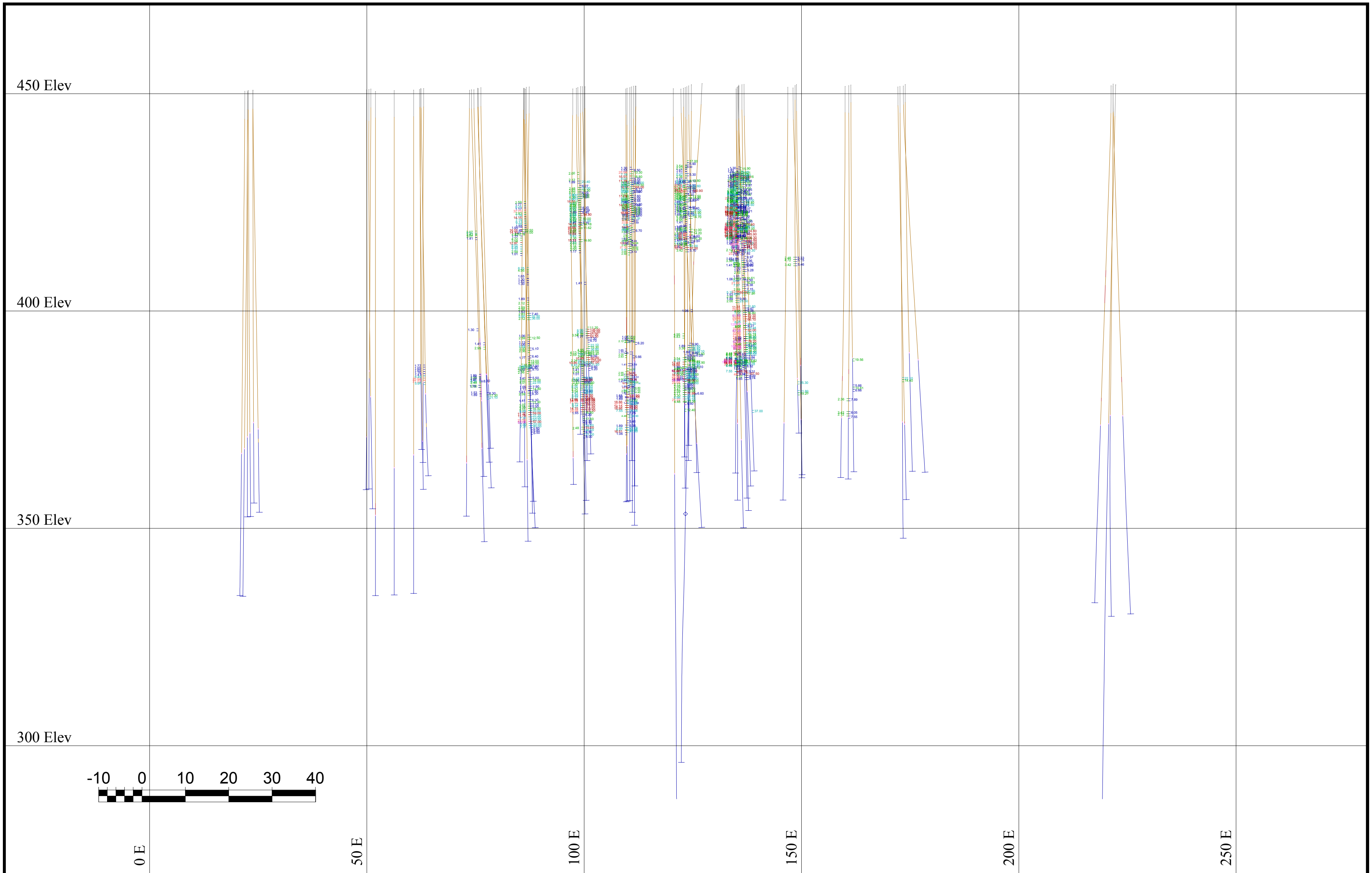

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Figure 12-3
Denison Mines Limited
Saskatchewan Projects
Sue B Deposit
Composite Vertcal Longitudinal Section

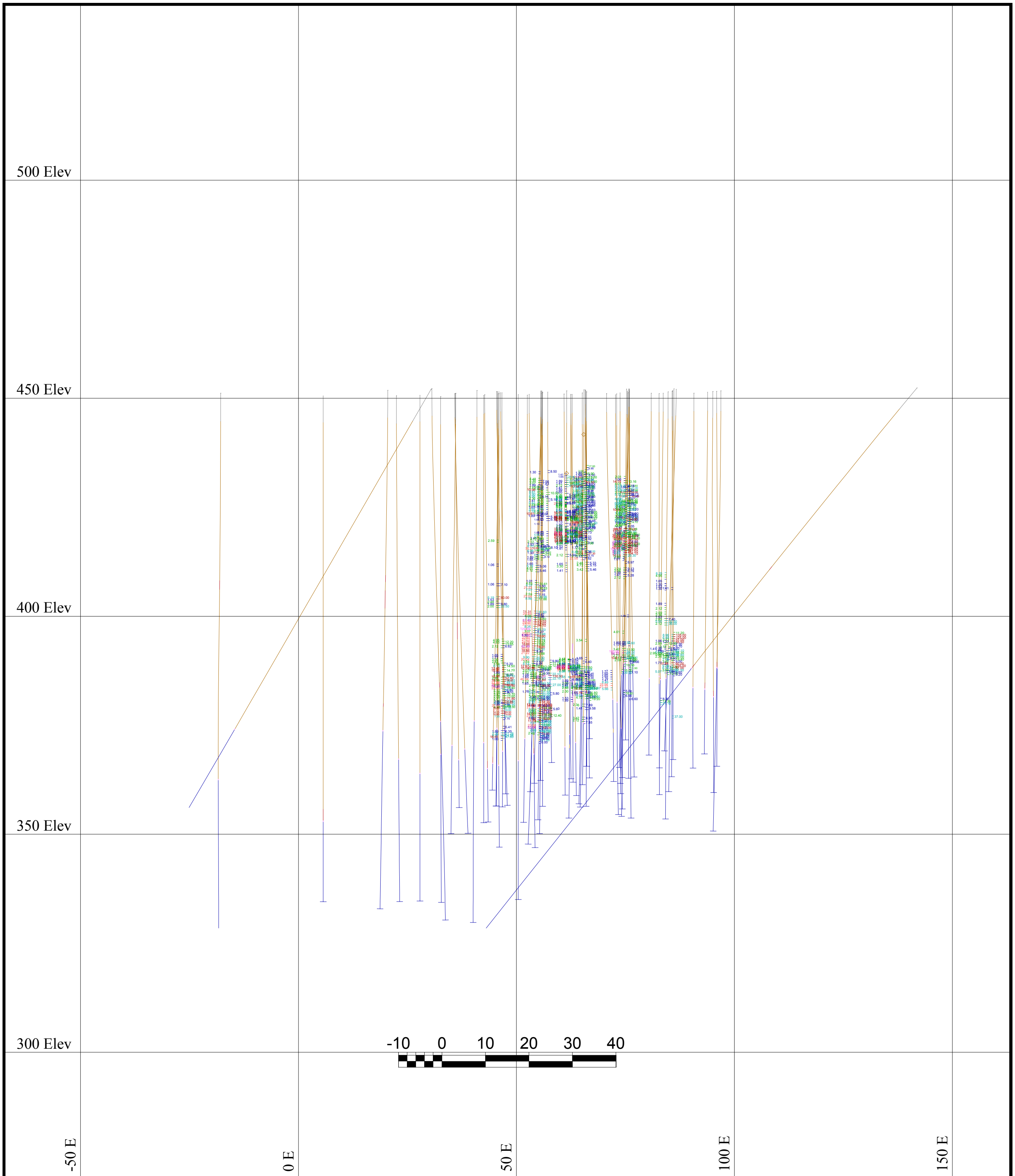

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Figure 12-4
Denison Mines Limited
Saskatchewan Projects
Sue B Deposit
Composite Vectical Cross Section

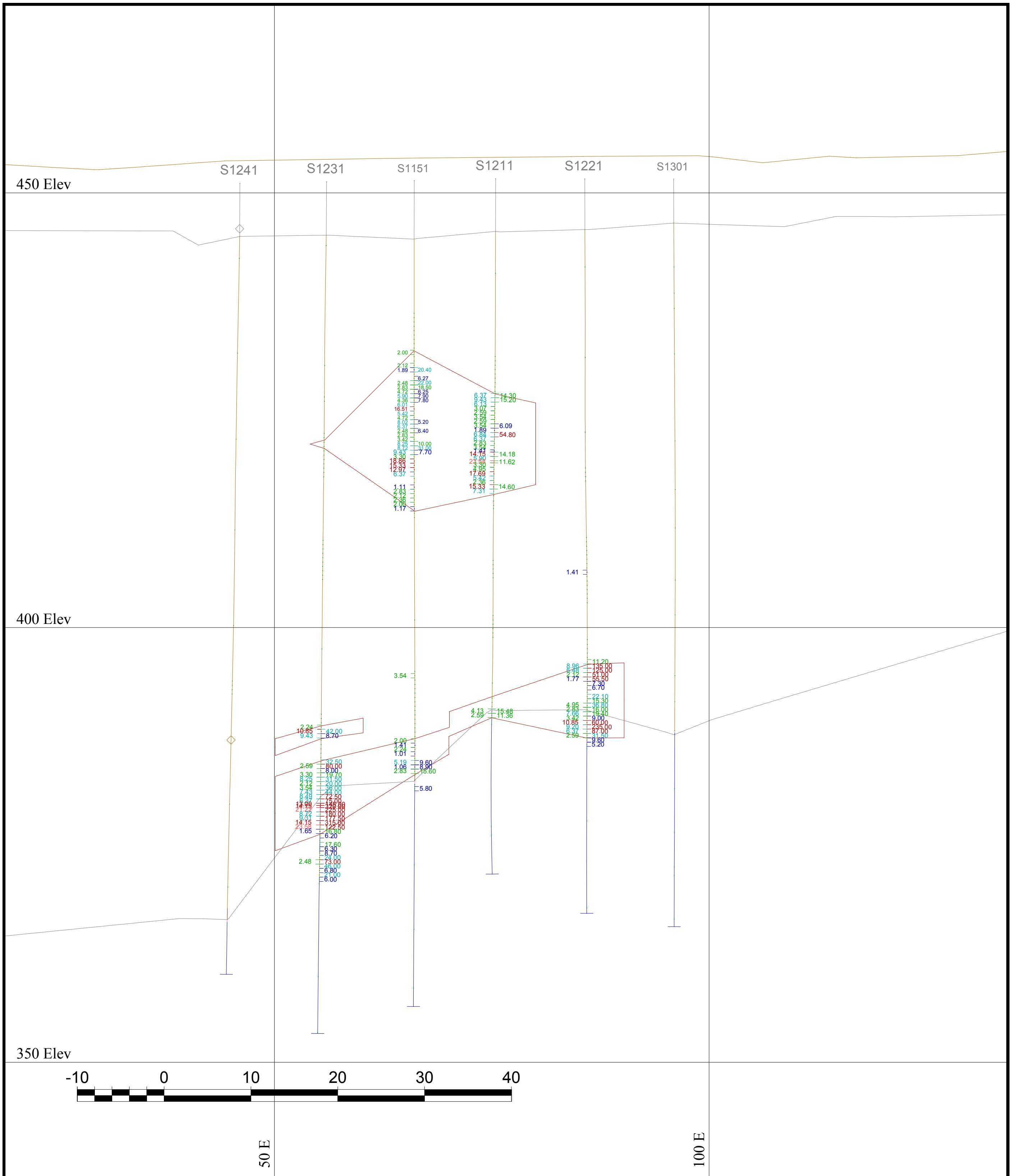

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Figure 12-5
Denison Mines Limited
Saskatchewan Projects
Sue B Deposit
Vectical Cross Section 2072N

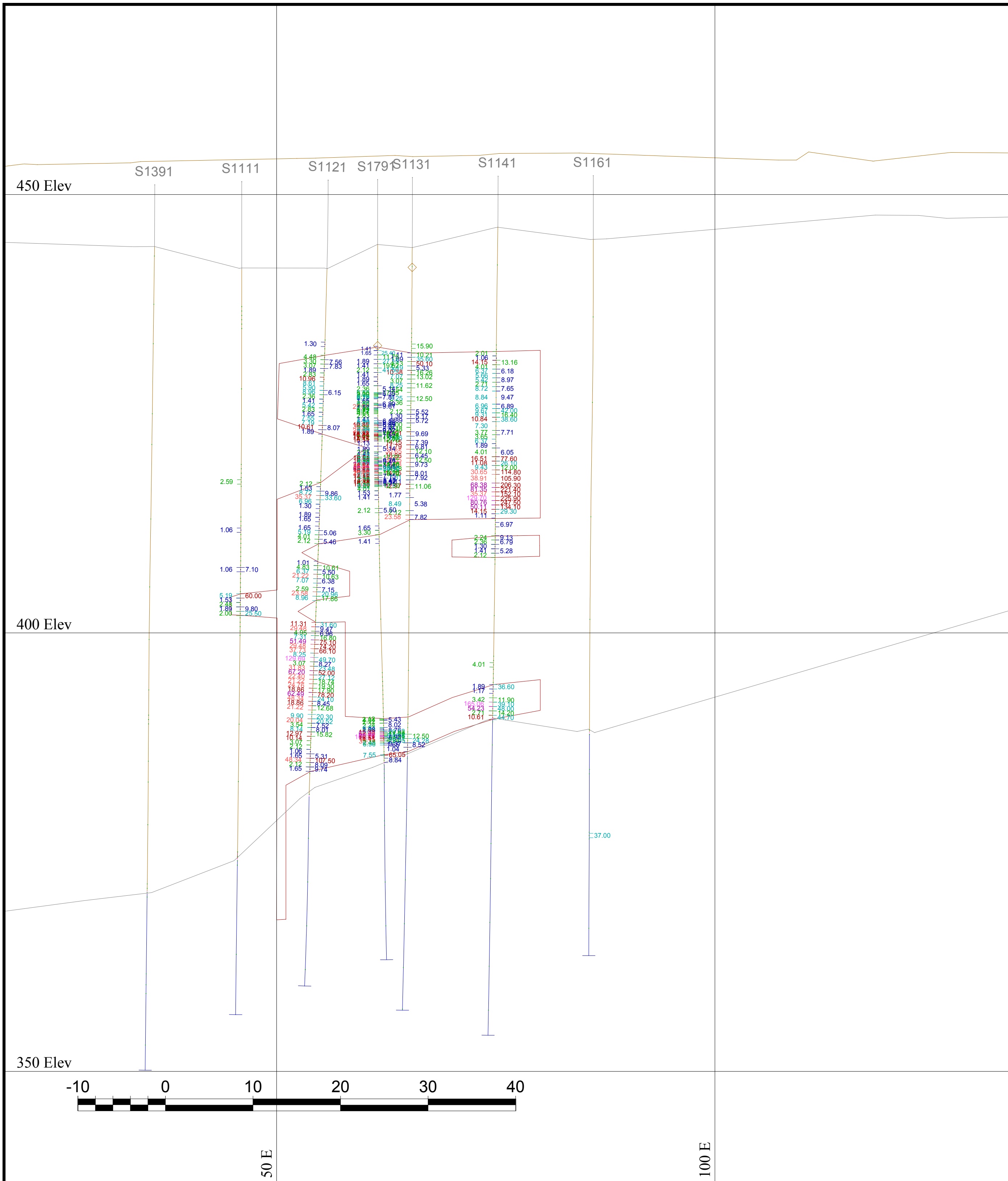

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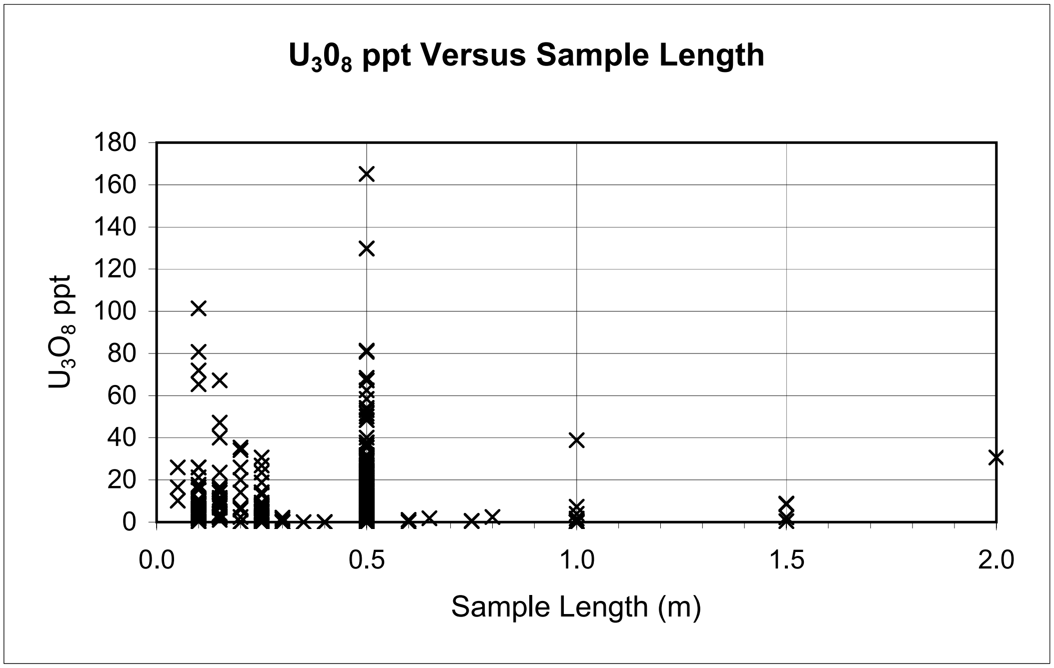
Figure 12-6
Denison Mines Limited
Saskatchewan Projects
Sue B Deposit
Vectical Cross Section 2108N

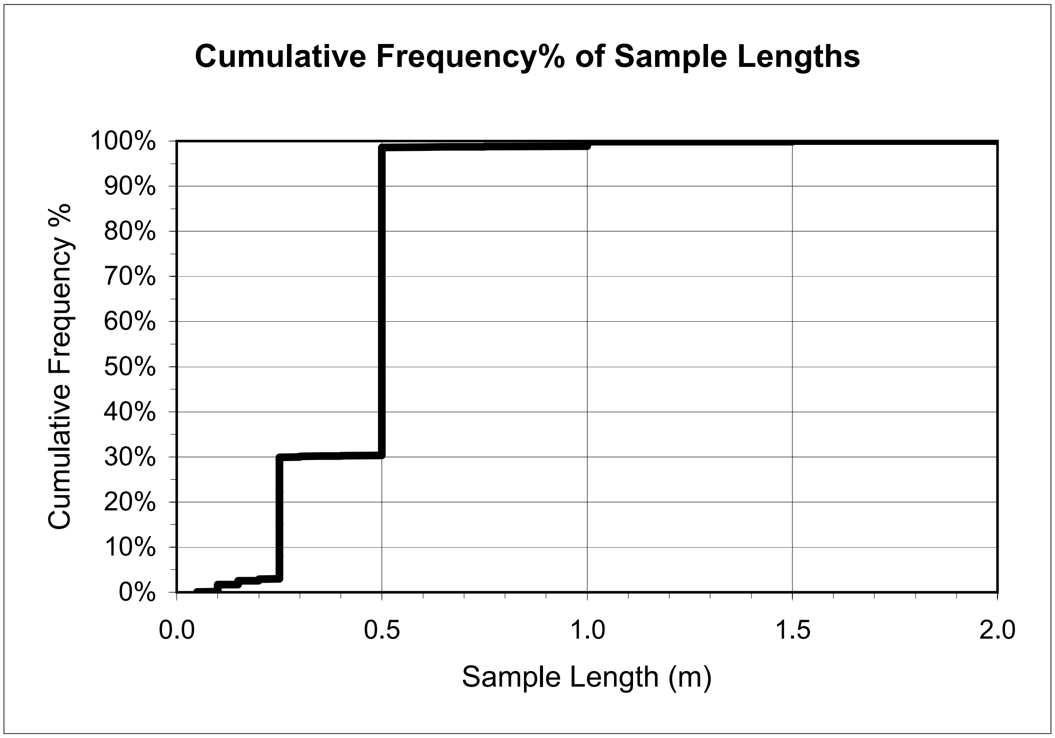
SAMPLING METHOD AND APPROACH

Core sampling is the primary sampling method. Handheld scintillometer readings on core guided sampling and provided for sampling on the basis of radiometric responses (uranium grade), where necessary. Sampling was relatively continuous for mineralized and waste intervals within the well mineralized zones in the sandstones, but only individual samples were taken between the perched and unconformity zones and in the basement. This discontinuous sampling was likely guided by radiometrics.

Sampling was standardized at 0.5 m and 0.25 m intervals. Approximately 95% of the core samples assayed had these lengths, with the 0.5 m intervals accounting for approximately 68%. Sampling is relatively grade independent, although the intervals longer than 0.5 m (<2%) tend to be in low grade/waste material, and short intervals of 0.05 m to 0.2 m tend to be in higher U_3O_8 grades (Figure 12-7).

Figure 12-7 Sample Length Statistics
Denison Mines Ltd. Sue B Uranium Deposit, Saskatchewan





SAMPLE PREPARATION, ANALYSES AND SECURITY

Sample preparation was similar to that described for Sue A and the McClean Lake project in general (See the respective sections in Sue A and McClean North). Chemical analyses of core samples for U_3O_8 in % or ppm were performed on behalf of Minatco by Barringer Laboratories (Alberta) Ltd. in Calgary during late 1988 and 1989. Approximately 70% of the samples were also analyzed for Cu (%), Ni (%), Co (%), Pb (%), Mo (%), V (%), and As (%). U_3O_8, originally in ppm or percent, is recorded in parts per thousand to two decimal places in the digital database. The other metals were reported in percent and have also been converted to ppt in the Cogema database.

DATA VERIFICATION

RPA cross-referenced Barringer analysis certificates with 1989 Minatco analysis summary logs and then with the digital database. The database does not contain sample numbers, so at first the summary logs with sample numbers and intervals were checked and then the logs referenced to the digital database.

RPA examined 205 U_3O_8 analyses (7% of database) and 189 Ni and Co analyses in drill holes CS46, CS47, and CS48. The summary logs convert all analyses to percent to five decimal places on some logs and round to four decimals places in others. RPA found three rounding/input errors in very low grade entries in the summary logs and six in the digital database for the three holes examined. These errors are not significant in terms of resource estimation.

Gemcom software check routines validated the database structure; with no errors reported other than legitimate missing sample intervals.

In cases where unanalyzed intervals occur in uranium mineralization grading generally $\geq .05\%$ U_3O_8, the missing interval has been assigned the length-weighted average of the analyses above and below the interval. RPA identified 20 such intervals in the database. One of these missing intervals, occurring between elevated and low grade, was assigned the low grade analysis from the next interval down hole.

RPA notes that uranium units in the Cogema block model are in $\%U_3O_8$ in contrast to ppt units in the drill hole database.

MINERAL PROCESSING AND METALLURGICAL TESTING

Leaching test work was conducted on composited Sue B drill core by Cogema's Service d'Etudes de Procédés et Analysis in 1998. Leach performance was similar to JEB and Sue C ores being processed at that time. Extraction was 98% in 8 hours, with overall recovery expected at 97% (Kerr et al., 2003).

MINERAL RESOURCE ESTIMATES

Resources for Sue B have been estimated by Cogema (Demange, 1998) and independently by Kerr et al. (2003).

COGEMA 1998 RESOURCE ESTIMATE

The basis and methodology for the 1998 estimate is the same as for the 1998 Sue A estimate, and is summarized in point form as follows:

11) The model utilized Minatco geologic interpretation and a mineralization envelope (not provided) at 0.1% U_3O_8.

12) Units for resource estimation are kg/t U_3O_8 or ppt ($^{o}/_{oo}$).

13) Analyses/assays were composited to 3 m.

TABLE 12-1 STATISTICS OF COGEMA 3 M COMPOSITES (1998) FOR SUE B
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Number of 3 m Composites	205
Mean Grade (U_3O_8 kg/t)	4.84
Maximum Grade (U_3O_8 kg/t)	51.87
Variance (U_3O_8 kg/t)	61.52

Adapted from Demange (1998)

14) Sue B Model origin: X=7507.0; Y=2012.0; Z=436.0; rotation12º

15) Semi-variograms were created as nested spherical models, no nugget was introduced (no graphics presented with the report):

Sills	18.63	42.10
EW range (m)	2.0	10.0
NS Range (m)	2.0	18.2
Vertical Range (m)	3.0	10.0

16) Search distance: 27 blocks, namely, 3 blocks EW, 3 blocks NS and 3 blocks vertically (rotated model grid). This is 15 m EW x 37.5 m x 9 m vertical.

17) Linear (ordinary) kriging for 10 m x 12.5 m x 3 m blocks (e.g. hole spacing).

18) Density=1/(0.452-(0.00326*%U_3O_8))

19) Precision on the estimate (volume and estimation) of the total resources stated as ±19%.

20) Recoverable resources were estimated by uniform conditioning based on selective mining blocks of 2.5 m x 6.5 m x 3 m (U_3O_8 kg/t variance 34.19)

Table 12-2 states the Cogema Recoverable Resource Estimates as reported in 1998:

TABLE 12-2 1998 COGEMA RESOURCE ESTIMATE FOR SUE B
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

U_3O_8 Cut-Off Grade	Tonnes	U_3O_8%	U_3O_8 (lbs x1,000)
0.1%	128,683	0.61	1,726
0.2%	101,680	0.73	1,638
0.3%	81,048	0.85	1,526
0.4%	65,300	0.98	1,406
0.5%	53,137	1.10	1,291

Cogema did not classify the resources to CIM or other standards.

DENISON AUDIT AND INDEPENDENT ESTIMATE 2003

Denison carried out independent resource estimation for Sue B as part of its audit of Cogema resources (Kerr et al., 2003). A “loose” cut-off grade of 0.1% U_3O_8 was used to contour and wireframe the Sue B deposit in 3D space. The contours were interpreted on 12.5 m cross-sections. The contours were digitized and then a wireframe constructed by extrusion between cross-sections.

The model was rotated 12º east. Resource blocks were 4 m grid east-west, 4 m grid north-south, and 3 m vertical. Block horizontal width is large compared to the width of the wireframe. Assay/analysis intervals were composited to 2 m within the wireframes to produce 100 composites. Block grades were interpolated by inverse distance cubed (ID^3) to constraint the smearing of high grades, similar to the interpolation used for Sue A. Block model cross sections and plans were shown in Figures 12-8 to 12-11.

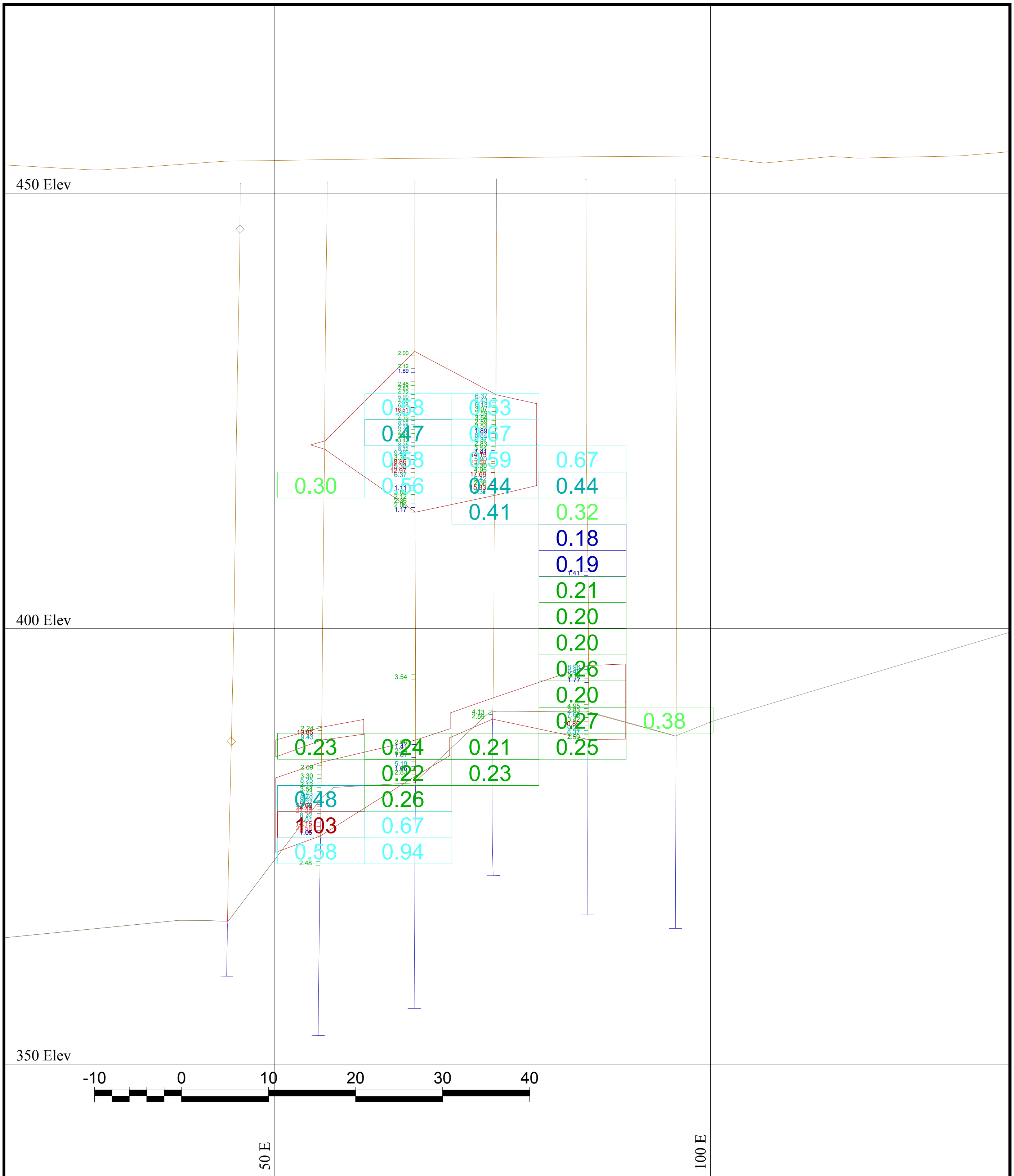


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Figure 12-8
Denison Mines Limited
Saskatchewan Projects
Sue B Deposit
Block Model Cross Section 2072N

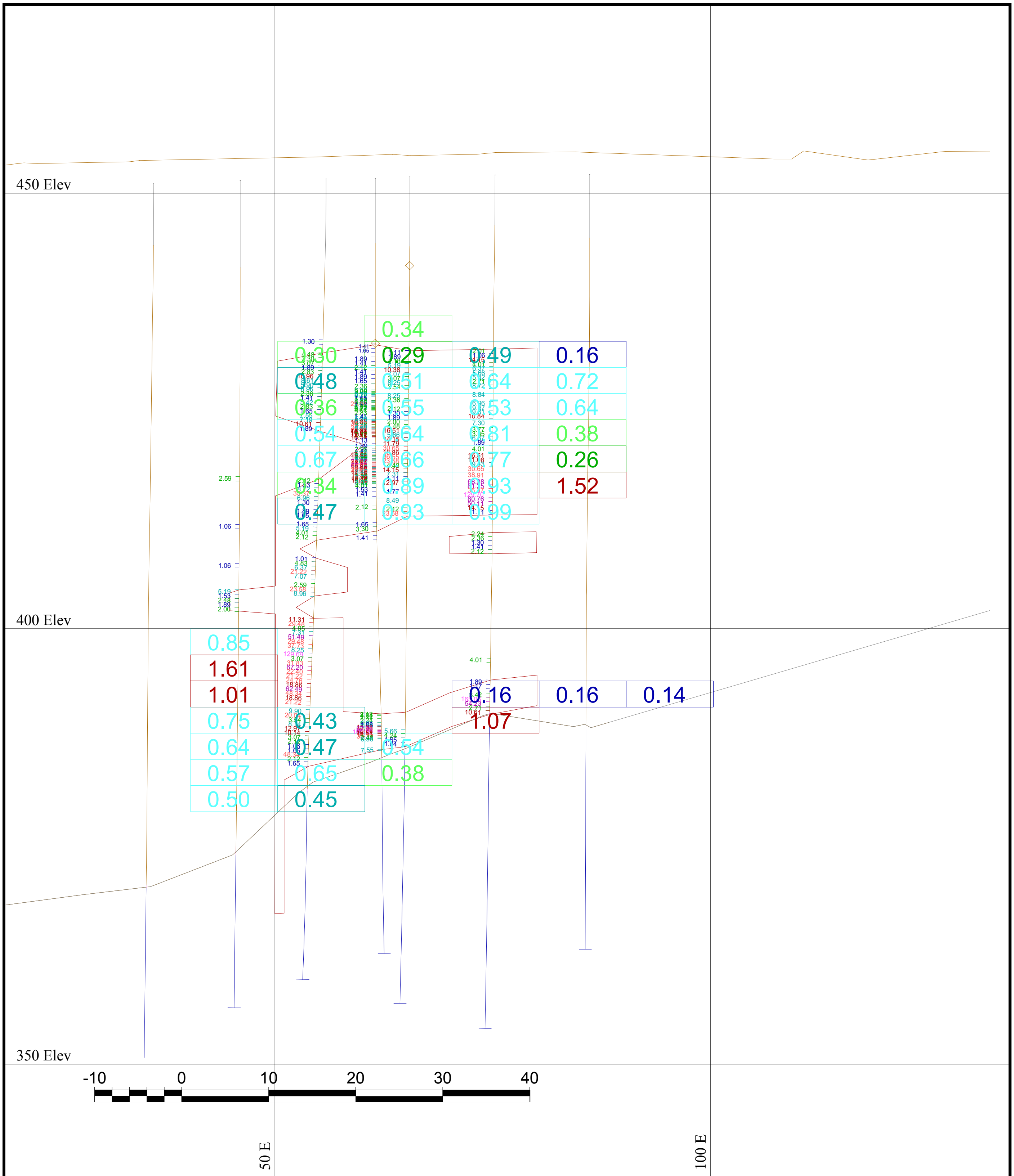

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Figure 12-9
Denison Mines Limited
Saskatchewan Projects
Sue B Deposit
Block Model Cross Section 2108N

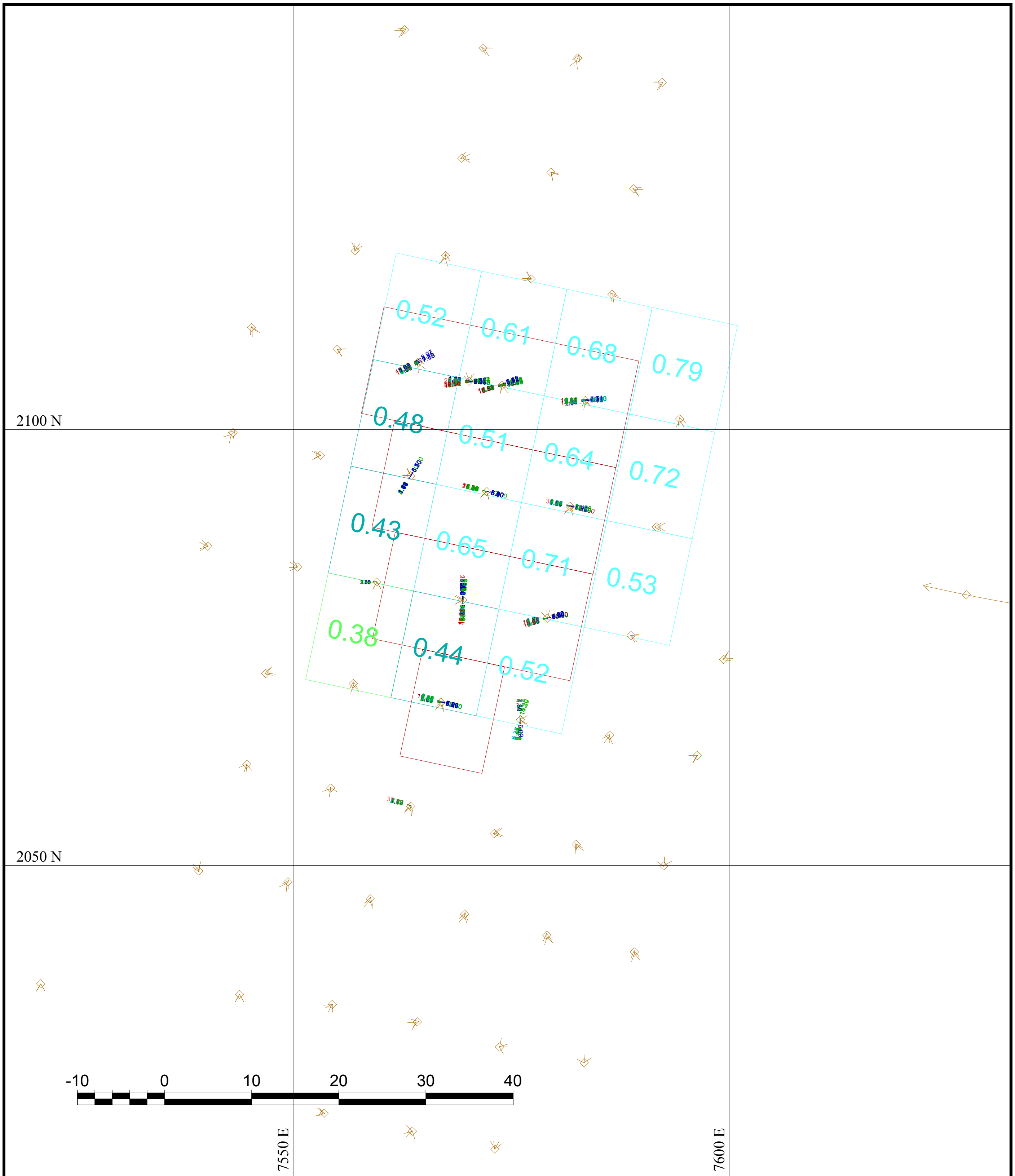

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Figure 12-10
Denison Mines Limited
Saskatchewan Projects
Sue B Deposit
Elev. 427 Block Model Plan

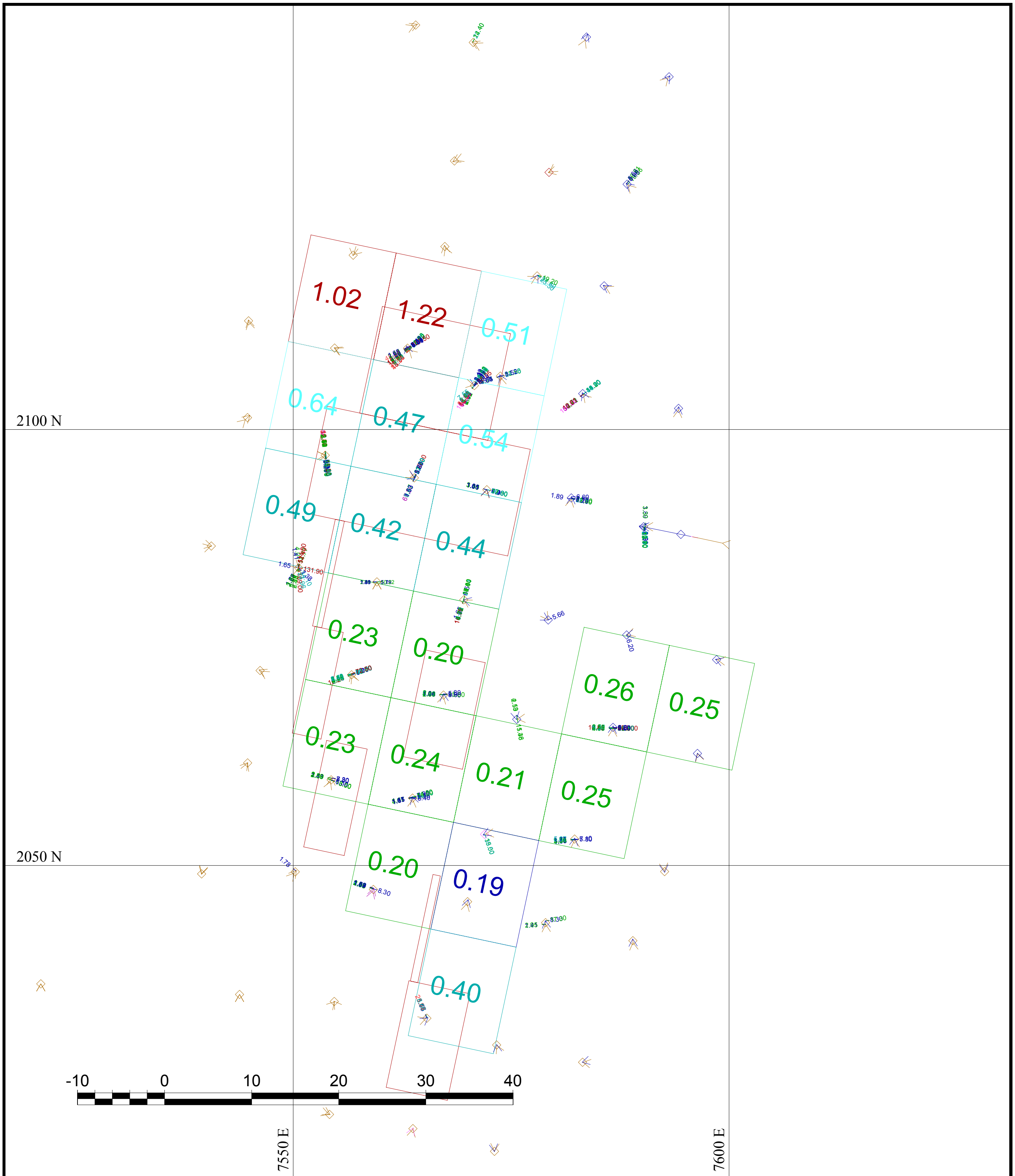

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Figure 12-11
Denison Mines Limited
Saskatchewan Projects
Sue B Deposit
Elev. 385 Block Model Plan

Kerr at al. used a calculated SG for bulk density derived by Kilborn (1990) below:

$$SG=1/(0.452-(.00326*\%U_3O_8))$$

This is the same as the Cogema formula but differs from the Sue A formula which incorporates Ni and As analyses. For low to negligible nickel and arsenic values, the calculated SG is similar. RPA notes that where uranium mineralization is absent or grades are low, the Kerr et al. and Cogema formulas calculate bulk density at 2.21 t/m^3. This density may represent the massive clay-altered unconformity mineralization well but appears to be low for sandstone and sandstone-hosted mineralization.

Volume to tonnage conversion used an average of 2.22 t/m^3 average bulk density; no density model was created. This density appears to be close to the average calculated SG for assays of 2.39.

Denison estimated in-place mineralization contained in the two lenses as 88,225 tonnes grading 0.742 % U_3O_8.

The 72% dilution from overbreak on the walls of the deposit was calculated by creating a 1.5 m shell around the main resource wireframe and estimating the tonnes and grade within the shell.

Including dilution of 63,299 tonnes at 0.031% U_3O_8, Denison estimated the in-place diluted resource at 151,524 tonnes grading 0.445% U_3O_8 (Table 12-3). The latter represents 1,487 kilo pounds of contained U_3O_8 and is 14% lower than the Cogema estimate for a 0.1% U_3O_8 cut-off. RPA notes that the Denison tonnages, for an average bulk density of 2.22 t/m^3, are consistent with the volume of the Denison Gemcom solids.

TABLE 12-3 ESTIMATE OF SUE B IN-PLACE MINERALIZATION (KERR ET AL. 2003)
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Lens	Tonnes	U3O8%	U3O8 tonnes	U3O8 lbs (x 1000)
Undiluted In-Place Mineralization				
Upper	55,053	0.704	388	854
Lower	33,172	0.806	267	589
Total	88,225	0.742	655	1,443
Dilution				
Upper	29,681	0.029	9	19
Lower	33,618	0.032	11	24
Total	63,299	0.031	20	43
Diluted In-Place Mineralization				
Upper	84,734	0.468	397	874
Lower	66,790	0.416	278	613
Total	151,524	0.445	674	1,487

Kerr et al. (2003) classified the resources as Measured and Indicated.

Table 12-4 compares the Denison (Kerr et al, 2003) and Cogema (Demange, 1998) resource estimates for a 0.1% U_3O_8 cut-off grade.

TABLE 12-4 COMPARISON OF DENISON AND COGEMA RESOURCE ESTIMATES FOR 0.1% U_3O_8 CUT-OFF GRADE
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Estimate	Tonnes	U_3O_8%	U_3O_8 lbs (x 1000)
Cogema	128,683	0.610	1,726
Denison (1)	88,225	0.742	1,443
Denison (2)	151,524	0.445	1,487
Variance: Denison versus Cogema			
Denison (1)	-31.4%	21.6%	-16.4%
Denison (2)	17.7%	-27.0%	-13.8%

(1) Undiluted in-place mineralization; (2) Diluted in-place mineralization

With respect to the Cogema kriging/uniform conditioning model, the Denison ID^3 estimate results in substantial differences in tonnes and grade for both undiluted and diluted resources. The metal content in the undiluted estimate is 16% lower than the corresponding Cogema estimate. This implies there may be fundamental differences in the data used or errors in estimation.

RESOURCE AUDIT AND MODEL VALIDATION

In the absence of Cogema wireframes, RPA reviewed the Kerr et al. (2003) uranium mineralization wireframe constructed at 0.1% U_3O_8. A sectional extrusion method was used. In RPA's opinion, only minor improvement could be made to the wireframe model by more rigorous wireframe integration between sections and this would have little impact in terms of overall resource estimation error. RPA has accepted the Kerr et al (2003) wireframe as a reasonable representation of the in situ mineralization for the purpose of auditing the Sue B resource. RPA notes that portions of the wireframe trace mineralization vertically along three drill holes. The holes appear to have followed mineralization in subvertical feeder faults between the upper and lower zones, and these likely have small widths in contrast to the width of the wireframe. There is no continuity between sections for this mineralization. In addition, the resource grade interpolation in these vertical "chimney" areas of the resource is supported by assays from a single hole. RPA has therefore reclassified the Kerr et al. (2003) undiluted in place resource lying between the upper and lower zones in Sue B to Inferred Resources. Table 12-5 presents a summary of the Sue B Mineral Resources at various block cut-off grades. Based on RPA's review of U_3O_8 prices and mining operating costs, the 0.1% U_3O_8 cut-off grade is reasonable for conversion to Mineral Reserves.

The Sue B deposit is not included in current MLJV mine plans; consequently, RPA has not evaluated the economic potential of this deposit and has not estimated Mineral Reserves.

TABLE 12-5 RPA RECLASSIFICATION OF DENISON 2003 RESOURCES FOR SUE B DEPOSIT

McClean Lake Joint Venture McClean Lake Property, Saskatchewan*

U_3O_8%	Indicated Resource			Inferred Resource		
Cut-Off	Tonnes	U_3O_8%	U_3O_8 Lbs	Tonnes	U3O8%	U_3O_8 Lbs
0.10	**72,944**	**0.73%**	**1,174,000**	**12,041**	**0.95%**	**252,000**
0.20	67,211	0.77%	1,141,000	9,854	1.11%	241,000
0.50	45,506	0.97%	973,000	5,319	1.70%	199,000
1.00	22,355	1.40%	690,000	2,758	2.68%	163,000

*Denison Mines Inc. holds 22.5% of the MLJV and the above resources

INTERPRETATION AND CONCLUSIONS

The Cogema database is suitable for resource estimation. The unconstrained deposit modelling, large panel ordinary kriging, and uniform conditioning approach results in generally higher tonnage and lower grades. In contrast to the Sue A estimates, there is a larger disparity (14% to 16%) in contained U_3O_8 pounds compared to the constrained wireframe and inverse distance check methods.

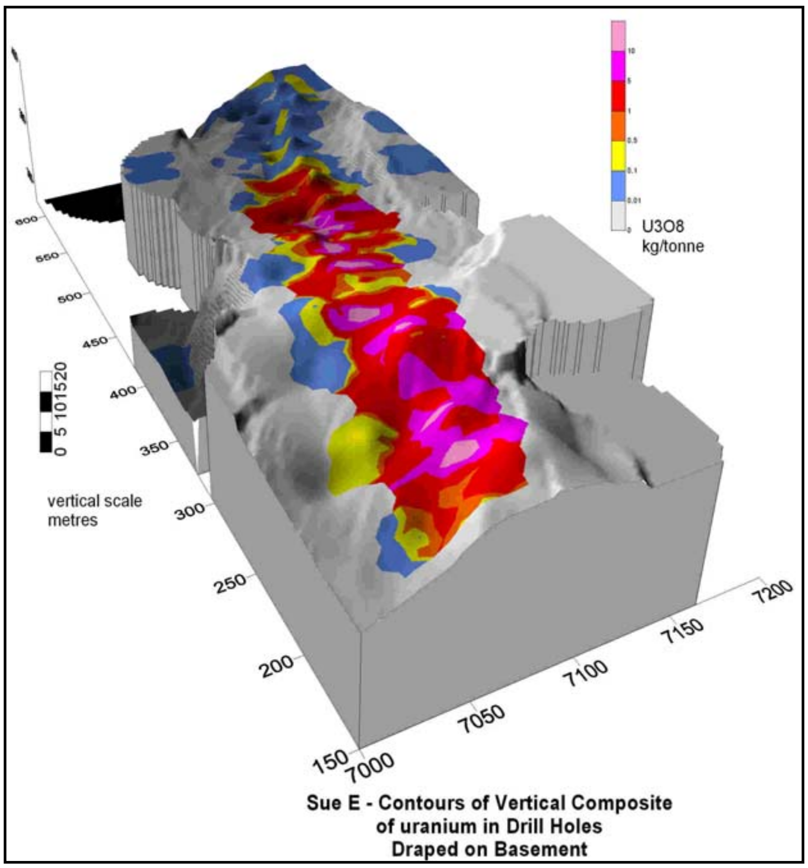
The significant differences between the kriging/uniform conditioning and inverse distance estimation approaches will be in the spatial distribution of in situ metal and the overall estimate of resource grade. RPA has accepted the Kerr et al. (2003) estimate but has revised classification of the resources.

13 SUE E

MINERALIZATION

The Sue E deposit, although discovered in the early 1990s, did not undergo development drilling until 2001. The mineralization has a strike length of approximately 320 metres, with horizontal widths varying from 4 to 15 metres, and occurs from 50 m to 200 m below the surface (Figure 13-1). The style of mineralization and setting is similar to that of the southern part of the Sue C deposit, that is, totally basement hosted, narrow, steeply-dipping vein-type, and with relatively clean mineralogy. However, Sue E does contain more Ni and As than Sue C. The U_3O_8 to As ratio is 1.03, and the U_3O_8 to Ni ratio is 1.57.

FIGURE 13-1 SUE E VERTICAL COMPOSITE OF URANIUM ASSAYS DRAPED ON BASEMENT, MCCLEAN NORTH AND SOUTH



DRILLING

At Sue E, a total of 135 diamond drill holes have been cored for a total of 23,757 metres. Drill spacing was at staggered 10 metre centres on 12.5 metre lines.

SAMPLE PREPARATION, ANALYSES AND SECURITY

Sample preparation was similar to that described for Sue A, Sue B, and the McClean Lake project in general (see the respective sections under Sue A and McClean North).

DATA VERIFICATION

RPA cross-referenced Barringer Laboratories (Minatco drilling) and Saskatchewan Research Council (SRC) analysis certificates for four drill holes with the Sue E digital database. No data entry errors were found. RPA compiled and analyzed data from reference standard analyses and pulp duplicates used for SRC in-house quality assurance and quality control (QA/QC). The precision for duplicate pulp analyses is acceptable, and for the most part the accuracy of the analyses, for the five reference standard used, is within industry acceptability as shown is the graphs below (Figure 13-2).

FIGURE 13-2 ANALYSES OF SRC QA/QC SAMPLES

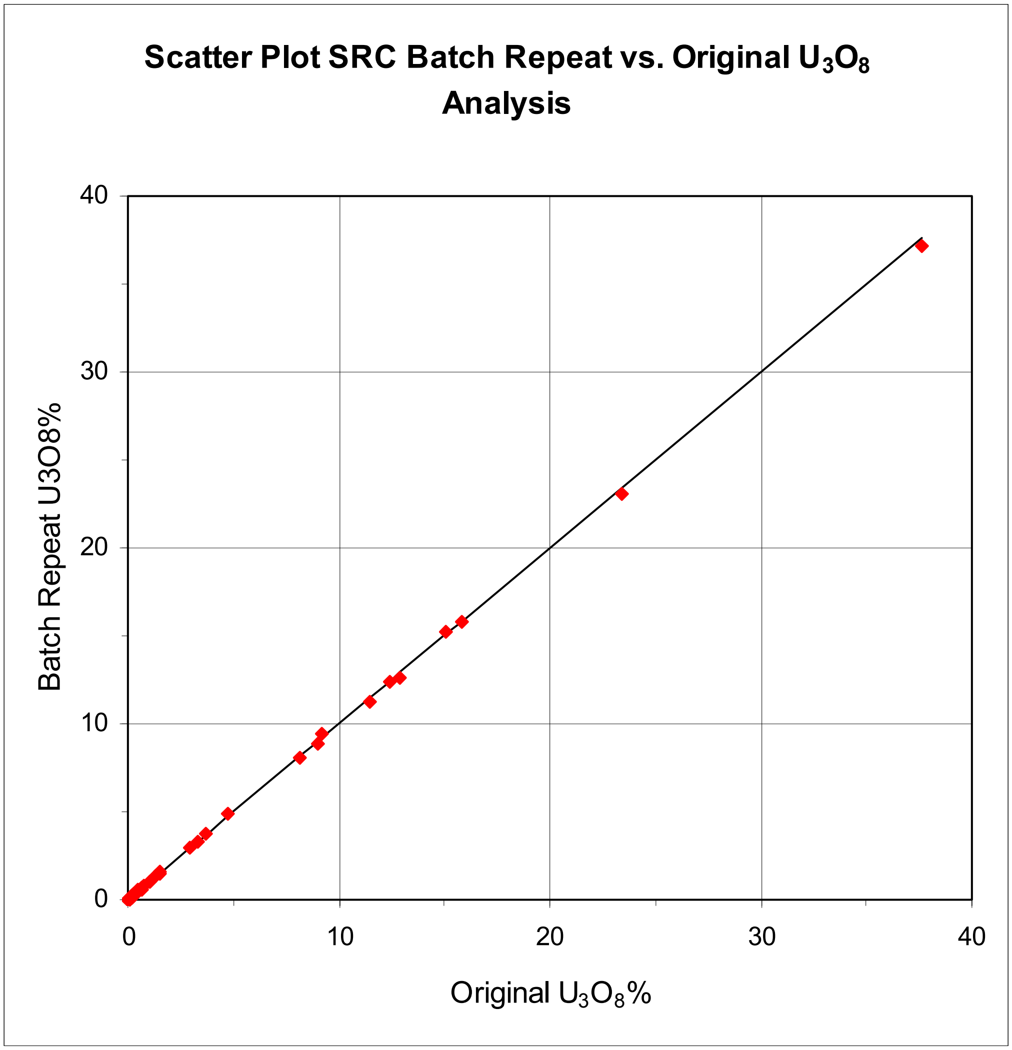
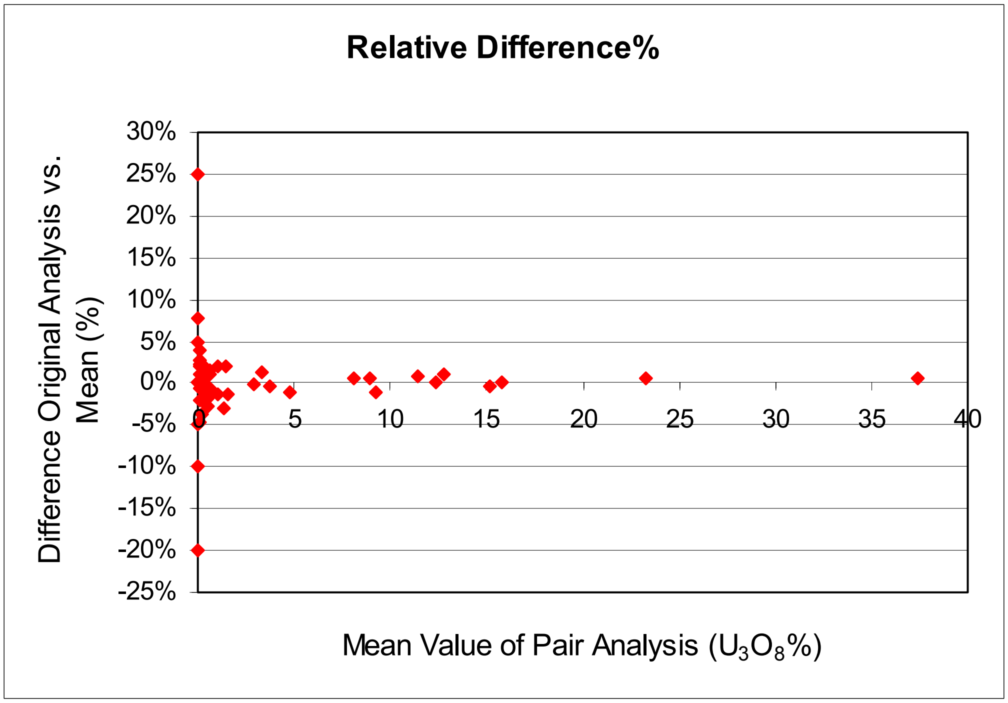


Figure 13-2 (continuation)



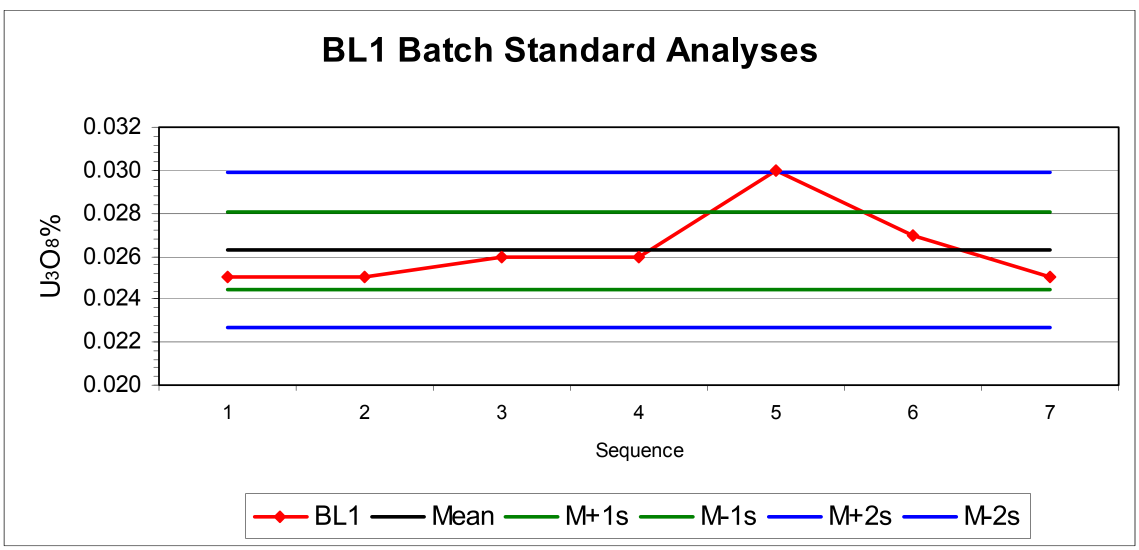
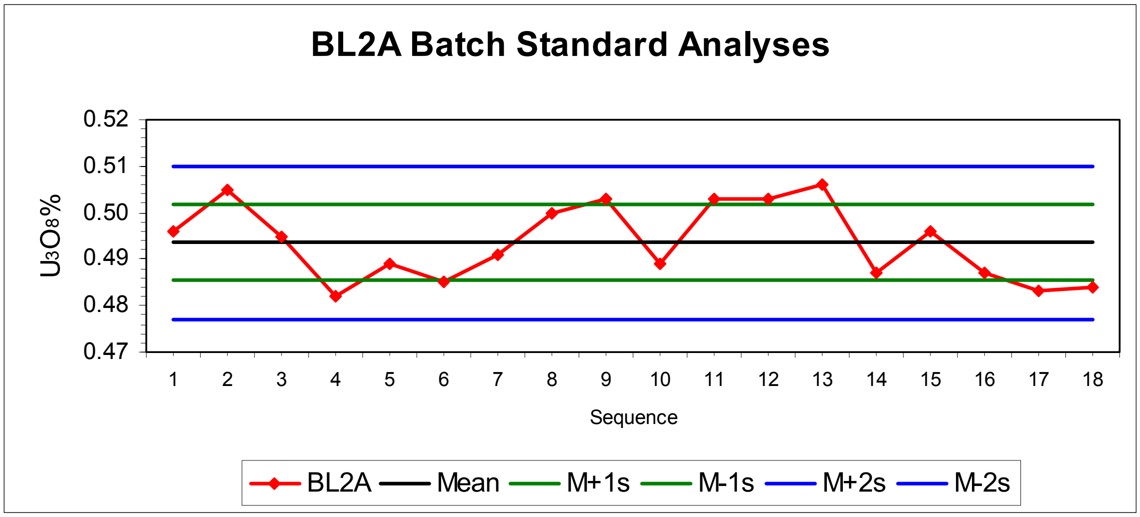


Figure 13-2 (continuation)



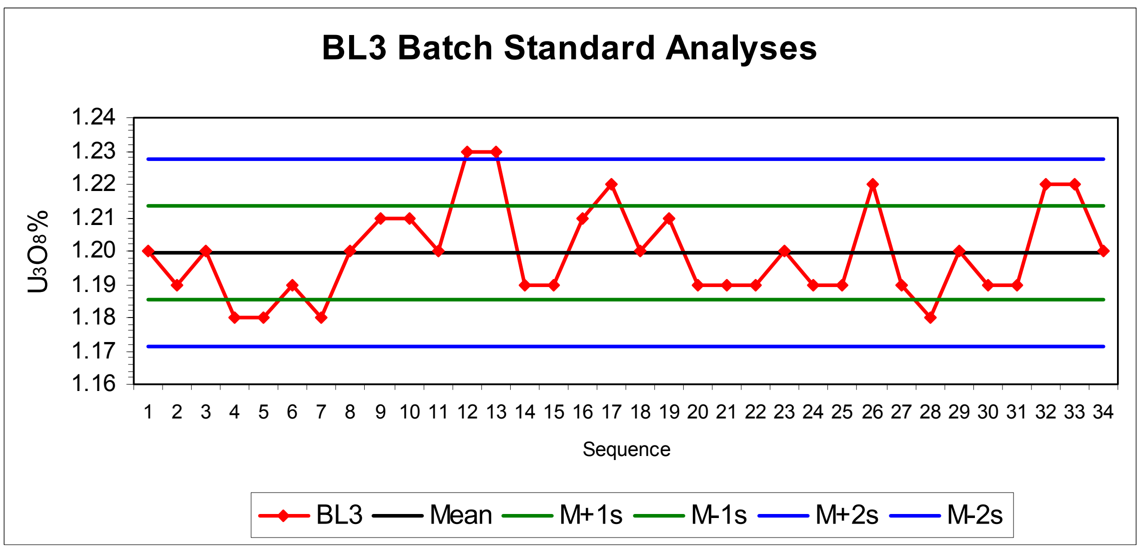
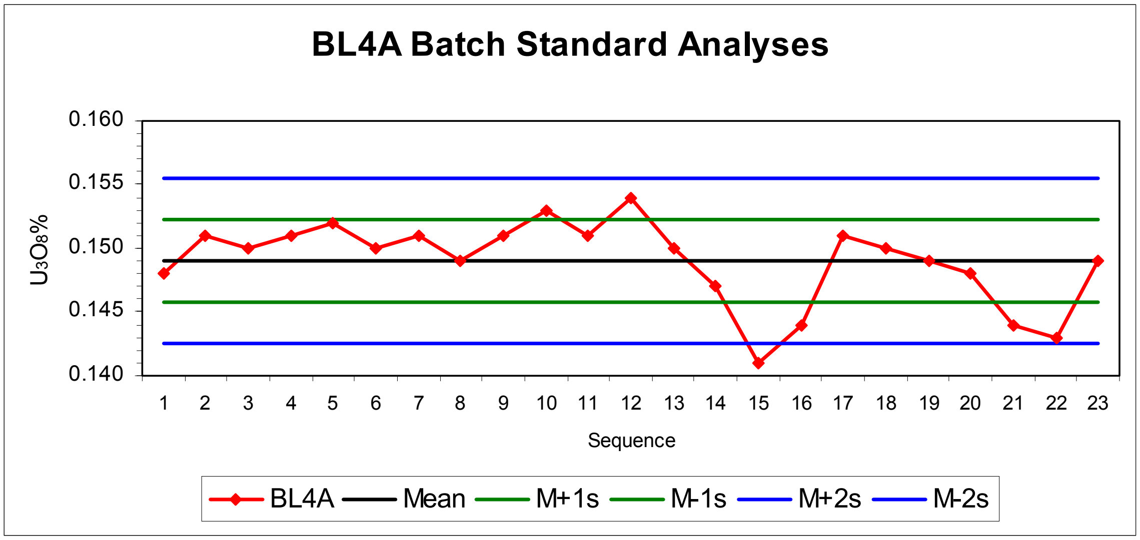
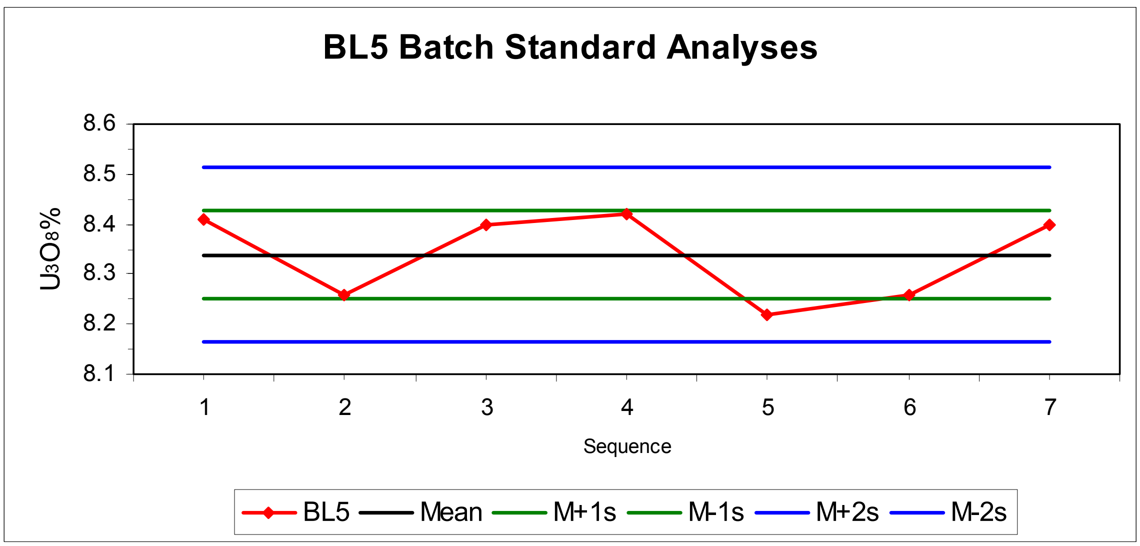


Figure 13-2 (continuation)





MINERAL PROCESSING AND METALLURGICAL TESTING

Metallurgical tests were carried out by SEPA in 2003 on a drill core composite prepared by the mine site geological staff to represent the orebody ("Sue E Leach Data", Cogema Internal Report, September 3, 2003). The core was drilled in 2000, and the sample assayed 1.07% U_3O_8 and 1.38% arsenic.

A 97% recovery was obtained from the SEPA test work. Optimum leaching recovery can be achieved by leaching for 9 hours with hydrogen peroxide oxidant or for 24 hours with oxygen at 2 bar pressure. The Sue E deposit is scheduled to be milled at the rate of about 125,000 tonnes per year concurrently with the Cigar Lake ore. To achieve the required milling rate, it will be necessary to complete the leaching plant expansion prior to milling the Cigar Lake ore.

The SEPA test work also showed that the majority of arsenic in the ore was dissolved during uranium leaching and that very little of the iron was leached. This soluble arsenic will increase the consumption and cost of ferric sulphate. On-site production of ferric sulphate will be required before the milling of the Sue E deposits starts.

The ferric sulphate cost estimation is shown below:

- 1.28 lb of arsenic precipitated for each lb U_3O_8 milled;
- 3 lb iron contained in ferric sulphate per lb arsenic;
- 3.84 lb iron in ferric sulphate required per lb U_3O_8 milled;
- Ferric sulphate cost per lb U_3O_8 is $1.08.

MINERAL RESOURCE ESTIMATES

Denison previously estimated the Mineral Resource at the Sue E deposit (Kerr et al., 2003) and the results are summarized in Table 13-1 below.

TABLE 13-1 ESTIMATE OF SUE E IN-PLACE MINERALIZATION (KERR ET AL. 2003)
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Tonnes	U_3O_8%	Ni%	As%	U_3O_8 tonnes	U_3O_8 (000 lbs)
Undiluted In-Place Mineralization					
227,826	1.479	0.836	1.411	3,370	7,429
Dilution (117.6%)					
267,868	0.063	0.257	0.359	169	372
Diluted In-Place Mineralization					
495,694	0.714	0.523	0.843	3,539	7,803

RPA has carried out an update of the Mineral Resource estimate for the Sue-E deposit based on drill hole data collected by Minatco and Cogema. The estimate was prepared using a block model constrained by a mineralized envelope. The mineralized envelope was developed using geological information and indicator kriging. Grade interpolation for U_3O_8, As, and Ni was done using Ordinary Kriging (OK), with different variogram models being applied to the northern region, where the direction of major continuity is sub-horizontal, and the southern region, where the direction of major continuity dips steeply.

The grade interpolation for U_3O_8 was based on 3m bench composites of both chemical assays and, where chemical assays were unavailable, equivalent U_3O_8 grades (e- U_3O_8) calculated from radiometric logs. The grade interpolation of As and Ni was based solely on chemical assays, with a default low-grade background value used where chemical assay information was unavailable.

DRILL HOLE DATABASE

The main drill hole database was provided to RPA by Denison in an ASCII dump file from the SERMINE software system used by Cogema. This file was translated into a set of CSV files containing collar coordinates, downhole survey information, chemical assays, and radiometric logs. This first database contained information for exactly 100 drill holes:

- 43 drilled prior to 2001 by Minatco, with hole IDs in the S426 to S494 range
- 57 drilled in 2001 by Cogema, with hole IDs in the S595 to S682 range

Subsequent review of other documents provided by Denison made it clear that there were additional drill holes that might contain relevant data for the Sue E resource estimation. Cogema was able to provide two supplementary ASCII data files from the SERMINE software system, one containing data for eight of the early Minatco drill holes (S199, S413, S415, S417, S419, S421, S423, and S424) and another containing data for five of the Cogema holes drilled in 2001 but not included in the first compilation (S665 through S669). These two supplementary databases were translated to CSV files and integrated into the project database.

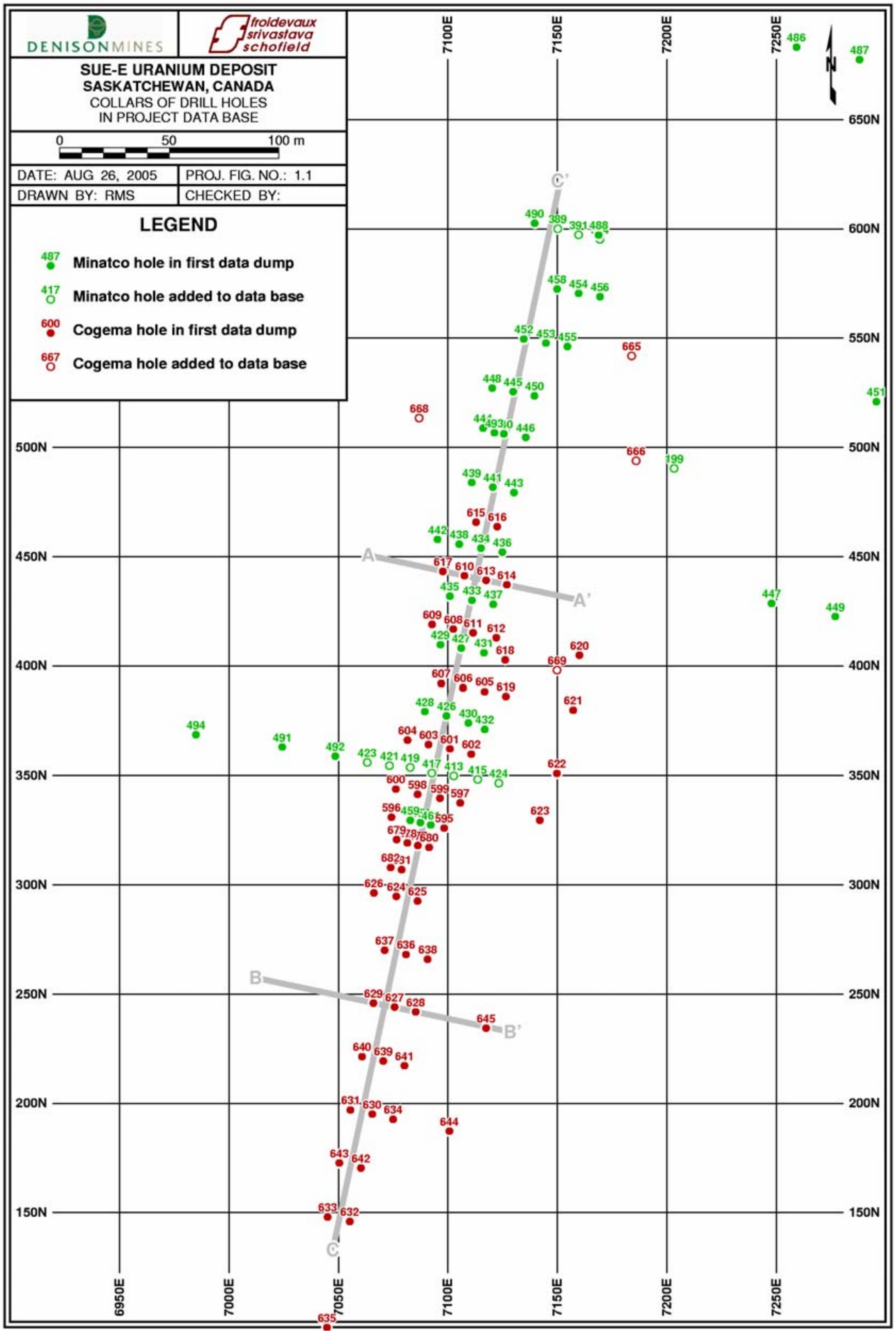
Corman's 1992 report on Sue E resources contains appendices listing drill hole data that were used at that time to estimate Sue E resources but that no longer appear in the Cogema data files. The collar, survey, and assay data from holes S389, S390, S391, and S394 were keypunched directly from the listings in Appendix A of Corman's report and integrated into the project database.

In summary, the drill hole database used by RPA contains data from a total of 117 drill holes:

- 55 drilled by Minatco prior to 2001: S199, S389–S391, S394, S413, S415, S417, S419, S421, S423, S424, S426–S459, S461, S486–S488, and S490–S494; and
- 62 drilled by Cogema in 2001: S595–S645, S665–S669, and S677–S682.

The collars of these holes are shown in Figure 13-3, with the Minatco drill holes shown in green and the Cogema drill holes shown in red.

FIGURE 13-3 DRILL HOLE COLLAR LOCATIONS



RPA is aware that there are additional Sue E drill holes, S806–S810, that were drilled in 2005 but for which no assay data has been provided. All of these are west-dipping holes drilled to intersect potential deep targets and would not provide any additional assay information that would impinge on the Sue-E block model developed for the report.

RPA recommends that a single comprehensive Sue E drill hole database be compiled to include all drill holes that are relevant to the Sue E deposit, including recent deep drilling.

UNITS AND CHECK OF DATABASE INTEGRITY

The ASCII data files from the SERMINE software system do not explicitly report the units that have been used for recording the chemical assays.

An inspection of the ASCII data files reveals that As and Ni values range into the several hundreds for the pre-2001 drill holes and into the several hundreds of thousands for the 2001 drill holes. RPA assumes therefore that the As and Ni grades were recorded in kg/t (or parts per thousand) for the pre-2001 drill holes and in g/t (or parts per million) for the 2001 drill holes. The project database has been checked against all the available tabulations of As and Ni data in Corman's 1992 report (where the units are clearly labelled as %) and against the spreadsheet files provided by the assay lab (where the units are also clearly labelled as %). Each of the few thousand sample intervals where such checks were possible confirmed that the units used for As and Ni assays in the ASCII data files were kg/t for the pre-2001 holes and g/t for the 2001 holes.

An inspection of the ASCII data files reveals that the values used to report uranium grades range into the several hundreds for all the drill holes, both the pre-2001 drill holes and the 2001 drill holes. RPA assumes therefore that these were recorded in kg/t, but there is no supporting documentation to clarify whether these assays report directly the uranium grade or the uranium oxide grade. The project database has been checked against all the available tabulations of uranium data in Corman's 1992 report (where the units are clearly labelled as % U_3O_8) and against the spreadsheet files provided by the assay lab (where the units are also clearly labelled as % U_3O_8). Each of the few thousand sample intervals where such checks were possible confirmed that the units used for uranium assays in the ASCII data files were kg U_3O_8/t for all the drill holes.

In addition to helping to resolve the issue of units, the checks of the project database against the laboratories spreadsheets and against the available tabulations in old reports also provide good insight into data base integrity. In the few thousand sample intervals where U_3O_8, As, and Ni assays were checked, there was no difference (apart from unit conversion and rounding) between the value recorded in the electronic database and the value reported in these other documents and files.

Though there is no supporting documentation for the ASCII dump files from the SERMINE software system, RPA is confident that the project database compiled from these ASCII files correctly reflects the assays as originally reported by the lab. RPA recommends that the assay certificates (or lab spreadsheets, if no certificate was provided) for all of the Sue E drill holes be gathered and collated into a single collection. Such a compilation of original paper (or electronic) records would greatly simplify the task of confirming the correctness of any electronic databases that might be created for the Sue E project in the future.

SPECIFIC GRAVITY

The only directly tested dry density data available for Sue E come from samples obtained in drill hole S457. In this hole, the core from the upper 60 m of the basement was used for dry density measurements. 116 intervals, most of them roughly 0.1 m to 0.2 m in length, were selected for dry density determinations.

RPA has performed a statistical analysis of the relationship between dry density and the intensity of U_3O_8, As, and Ni mineralization. The results are shown in Figures 13-4 and 13-5.

FIGURE 13-4 DRY DENSITY VERSUS (%U_3O_8+%AS+%NI)

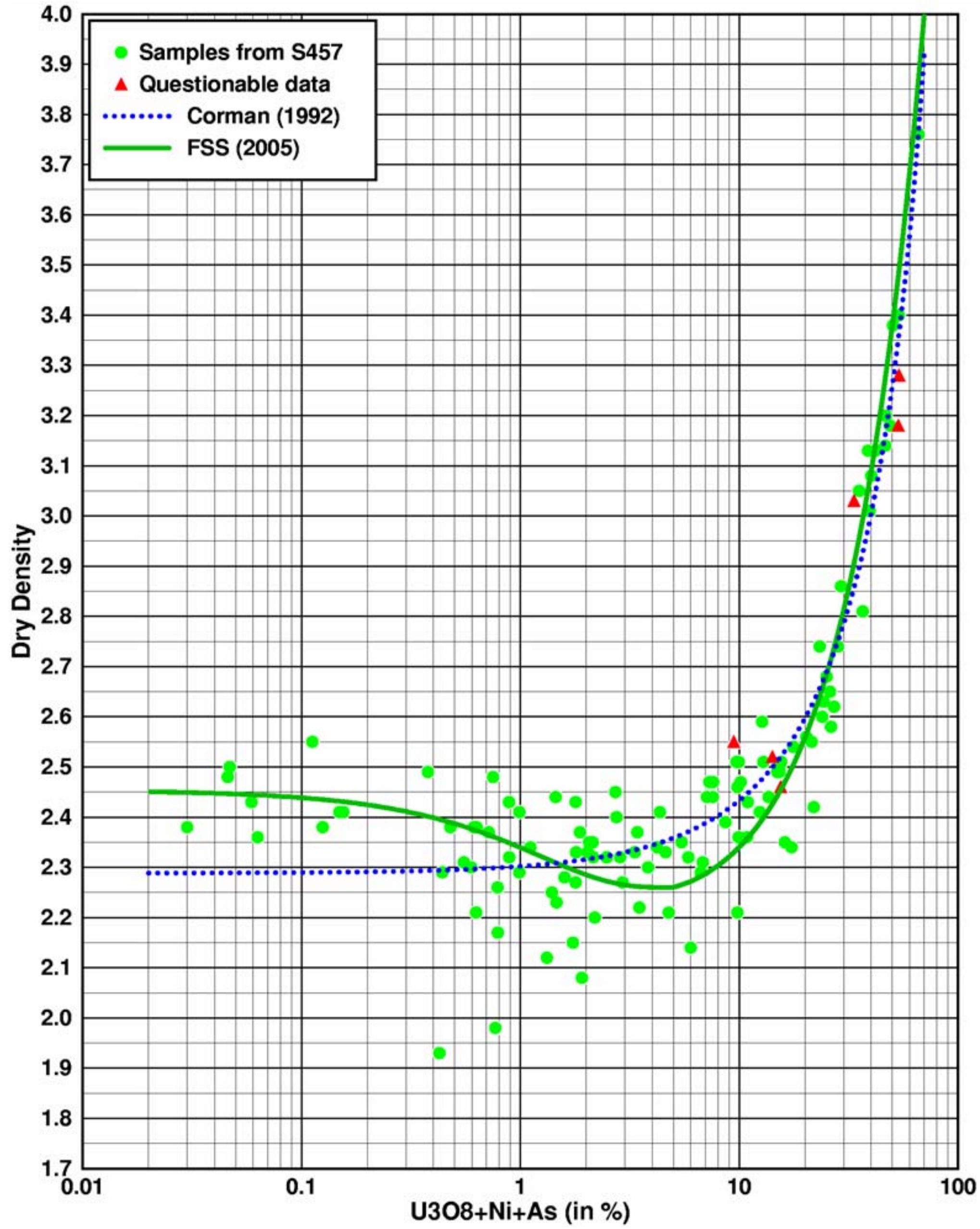
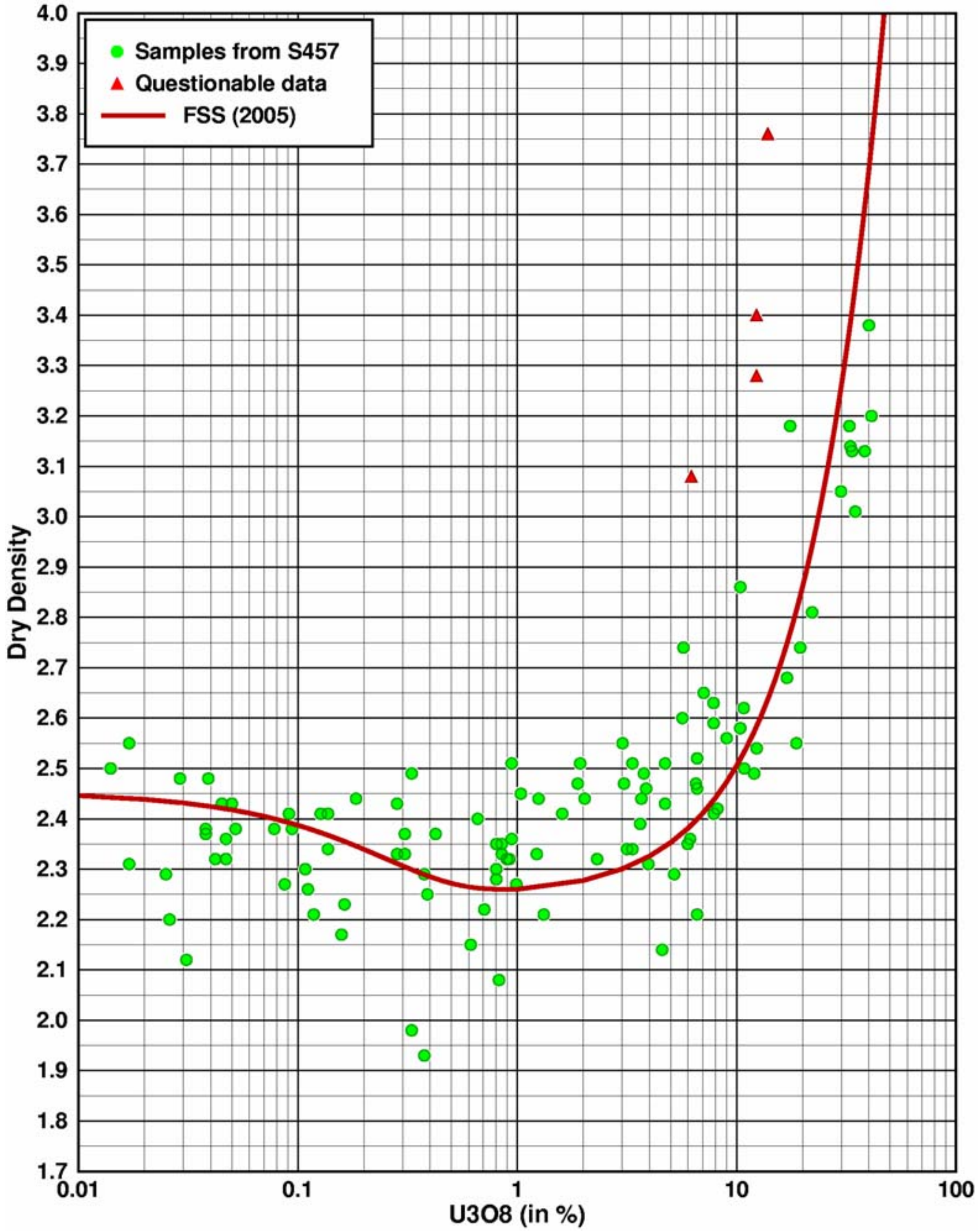


FIGURE 13-5 DRY DENSITY VERSUS % U_3O_8



When U_3O_8, As, and Ni assays are all available, the dry density can be predicted from the sum of these three using the following equations:

If $(\%U_3O_8 + \%As + \%Ni) > 5$:

$$\textit{Predicted dry density} = 2.26 + 0.01161 \times \left| (\%U_3O_8 + \%As + \%Ni) - 5.0 \right|^{1.2}$$

If $(\%U_3O_8 + \%As + \%Ni) < 5$:

$$\textit{Predicted dry density} = 2.26 + 0.00031 \times \left| (\%U_3O_8 + \%As + \%Ni) - 5.0 \right|^{4.0}$$

When As and Ni assays are not available, the dry density can be predicted from the U_3O_8 grade alone using the following equations:

If $\%\ U_3O_8 > 1$:

$$\textit{Predicted dry density} = 2.26 + 0.01758 \times \left| \%\ U_3O_8 - 1.0 \right|^{1.2}$$

If $\%\ U_3O_8 < 1$:

$$\textit{Predicted dry density} = 2.26 + 0.19375 \times \left| \%\ U_3O_8 - 1.0 \right|^{4.0}$$

These equations are similar to the one proposed by Corman (1992) and used in subsequent resource estimation studies. By separating the prediction into two parts, one for weak mineralization and the other for strong mineralization, the equations used in this study better capture the tendency for specific gravity to decrease slightly with increasing grade (for low grades) and then to increase noticeably when the intensity of mineralization exceeds a certain threshold. The same type of relationship has been noted elsewhere in northern Saskatchewan: in the Gaertner and Deilmann deposits, or at Key Lake, where the same kind of two-part regression formula was also used for resource estimation purposes. At Key Lake, the reason for this type of dry density relationship was understood to be the result of the mineralization being structurally controlled. With low to moderate amounts of mineralization, the decreasing density reflects the fact that mineralized rock tends to have been more permeable, probably due to shearing or fracturing, which often goes hand-in-hand with slightly higher porosity. But as the intensity of mineralization increases, the high specific gravities of the nickel-arsenides and the pitchblende overwhelm the density–paleo-porosity relationship.

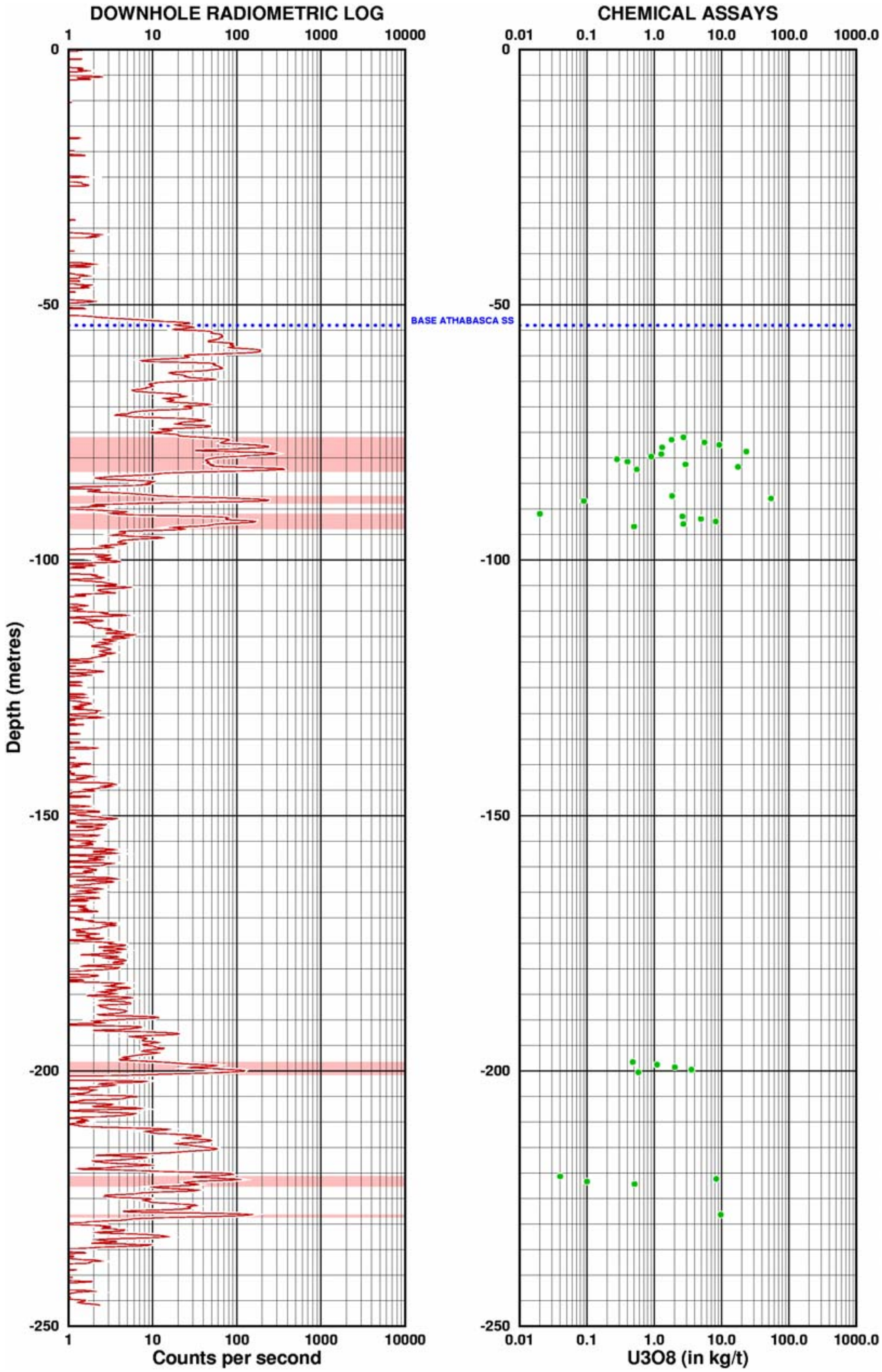
RPA notes that all of the available dry density data come from basement rock in a single drill hole. It is recommended that additional dry density measurements be made using core from other drill holes and that some mineralized samples from the base of the Athabasca sandstone be included. Currently, the block model assumes that mineralization does not extend into the sandstone. The drill hole data do, however, occasionally show some low-grade mineralization near the base of the sandstone that may prove to be economic. If future block models of Sue E resources extend the potentially economic mineralization into the sandstone, it will be important to have a dry density prediction formula that is customized to sandstone and can distinguish it from the basement.

RADIOMETRIC DATA AND EQUIVALENT U_3O_8 ASSAYS

For the vast majority of drill holes, there are two radiometric logs available: the gamma-ray counts per second (CPS) from a downhole probe and the CPS from a handheld scintillometer. In 2001, the decision to chemically assay certain intervals was based on the handheld scintillometer. Unfortunately, the correlation between CPS and U_3O_8 grade is not as good for a handheld scintillometer as it is for a downhole probe. With the gamma-ray counts responding primarily to radon, the correlation between CPS and U_3O_8 deteriorates when radon is not in equilibrium with uranium, as is often the case when core is brought to the surface and some radon escapes.

In the 2001 drilling, there are therefore many intervals that carry moderate levels of uranium mineralization (based on an examination of the downhole radiometric log) but that were not chemically assayed because they did not respond well on the handheld scintillometer. Figure 13-6 shows an example from hole S647.

FIGURE 13-6 RADIOMETRIC LOG AND CHEMICAL ASSAYS IN S647



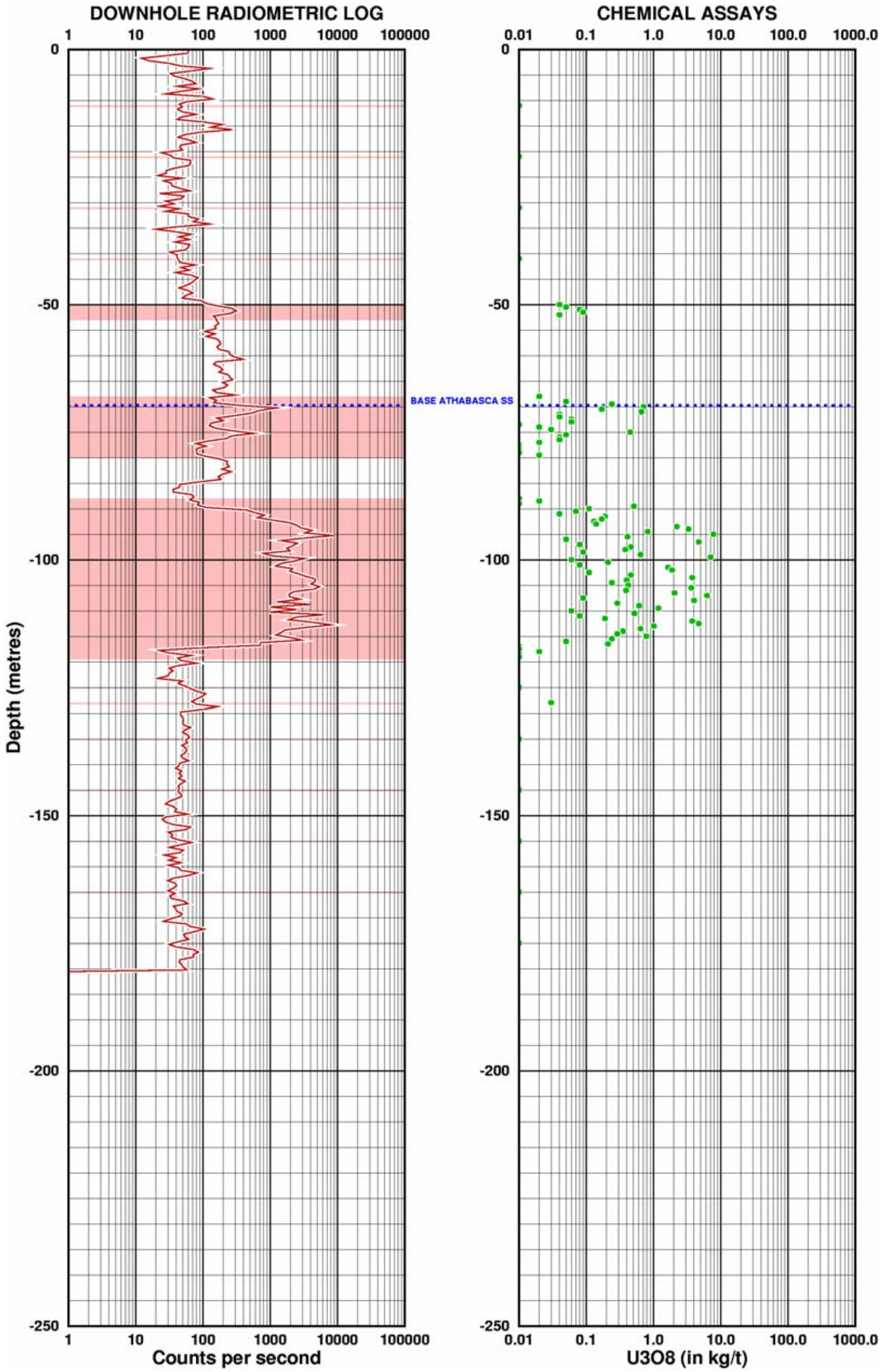
Immediately below the Athabasca unconformity at about 54 m down the hole, the downhole probe was recording 50–100+ counts per second. But there are no chemical assays until about 75 m down the hole, where the counts per second again reach 100. Judging by the chemical assays reported for the 75 m to 95 m interval, it is likely that the U_3O_8 grades in the 55 m to 60 m interval are running 0.1% to 1.0%, which would likely be ore grade material.

The same kind of problem does not occur so often in the Minatco holes because the sampling protocol in the early 1990s was to systematically assay:

- Every 10 m down the hole (regardless of the radiometric response);
- The interval immediately beneath the Athabasca unconformity (regardless of the radiometric response); and
- All other intervals with a strong radiometric response.

An example of the chemical assaying and radiometric response in a Minatco hole is shown in Figure 13-7. In S429, the 75 m to 80m interval does not have a strong radiometric response, but since it lies just beneath the Athabasca unconformity, it is assayed anyway.

FIGURE 13-7 RADIOMETRIC LOG AND CHEMICAL ASSAYS IN S429



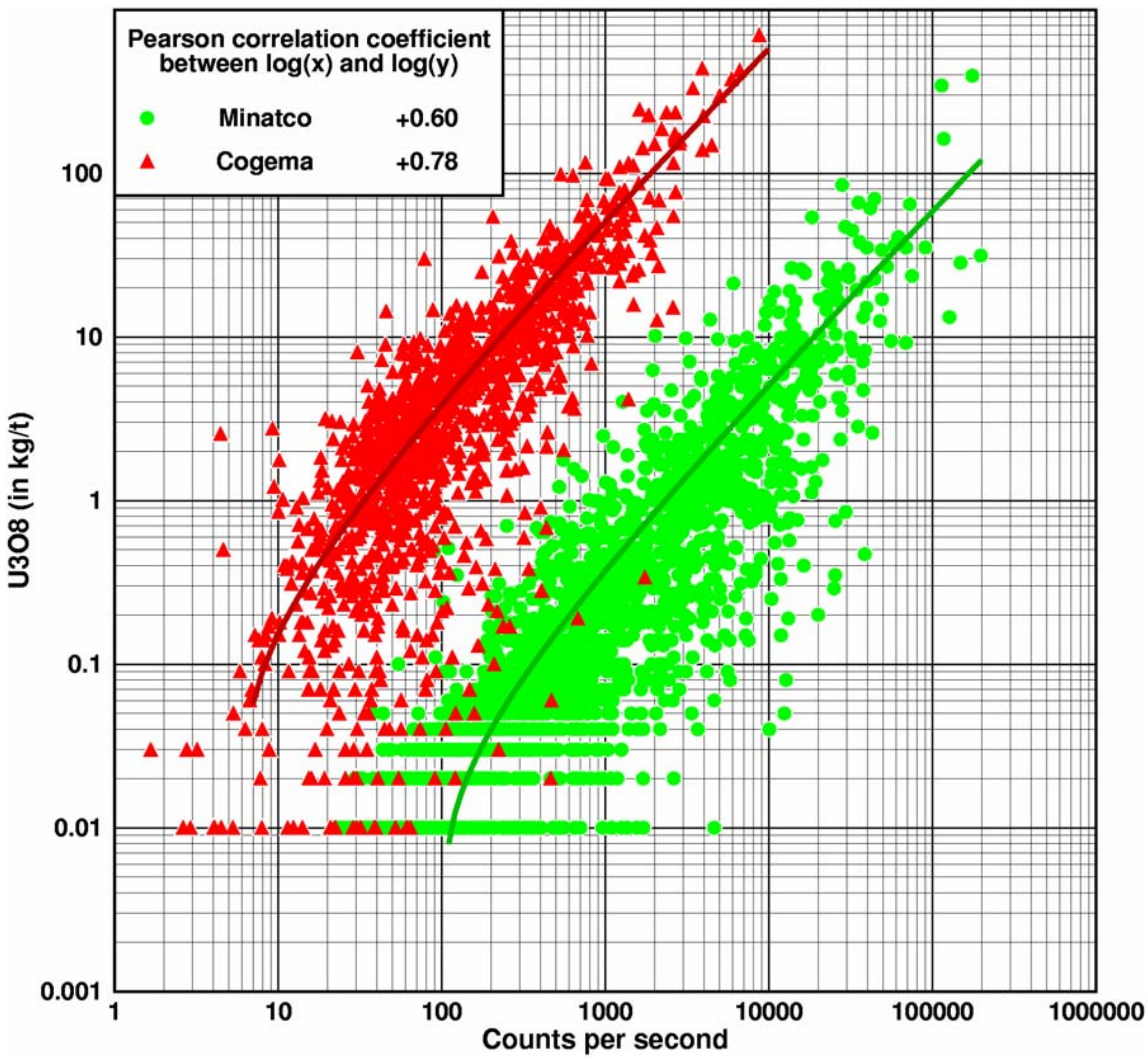
The Minatco holes therefore provide more chemical assays than the Cogema holes, and rarely miss interesting or anomalous uranium mineralization.

RPA has analyzed the downhole radiometric logs and the chemical assays and concludes that they offer an excellent correlation allowing U_3O_8 grades to be predicted with a high degree of reliability when there is no chemical assay available. CIM guidelines on resource estimation for uranium support the use of "equivalent" assays derived from radiometric logs:

"When the precision of Equivalent Assay data has been demonstrated, the Equivalent Assay data may be merged with chemical assay data from drill core in the database for a MRMR estimate. Data from non-core drill holes may provide a considerable portion of the database; however, in order to satisfy QA/QC of radiometric data, and provide geological information for deposit interpretation, core drilling is also required."

Figure 13-8 shows the correlation between counts per second and % U_3O_8. Minatco and Cogema used different downhole radiometric probes, so the radiometric response for a particular drill hole interval depends on the type of probe used. In Figure 13-8, the data for Minatco holes are shown in green and the data for the Cogema holes are shown in red. Also shown are the regression lines that can be used to predict the U_3O_8 grade from counts per second:

FIGURE 13-8 COUNTS PER SECOND VS. % U_3O_8 ON ASSAY INTERVALS



For the Minatco holes:

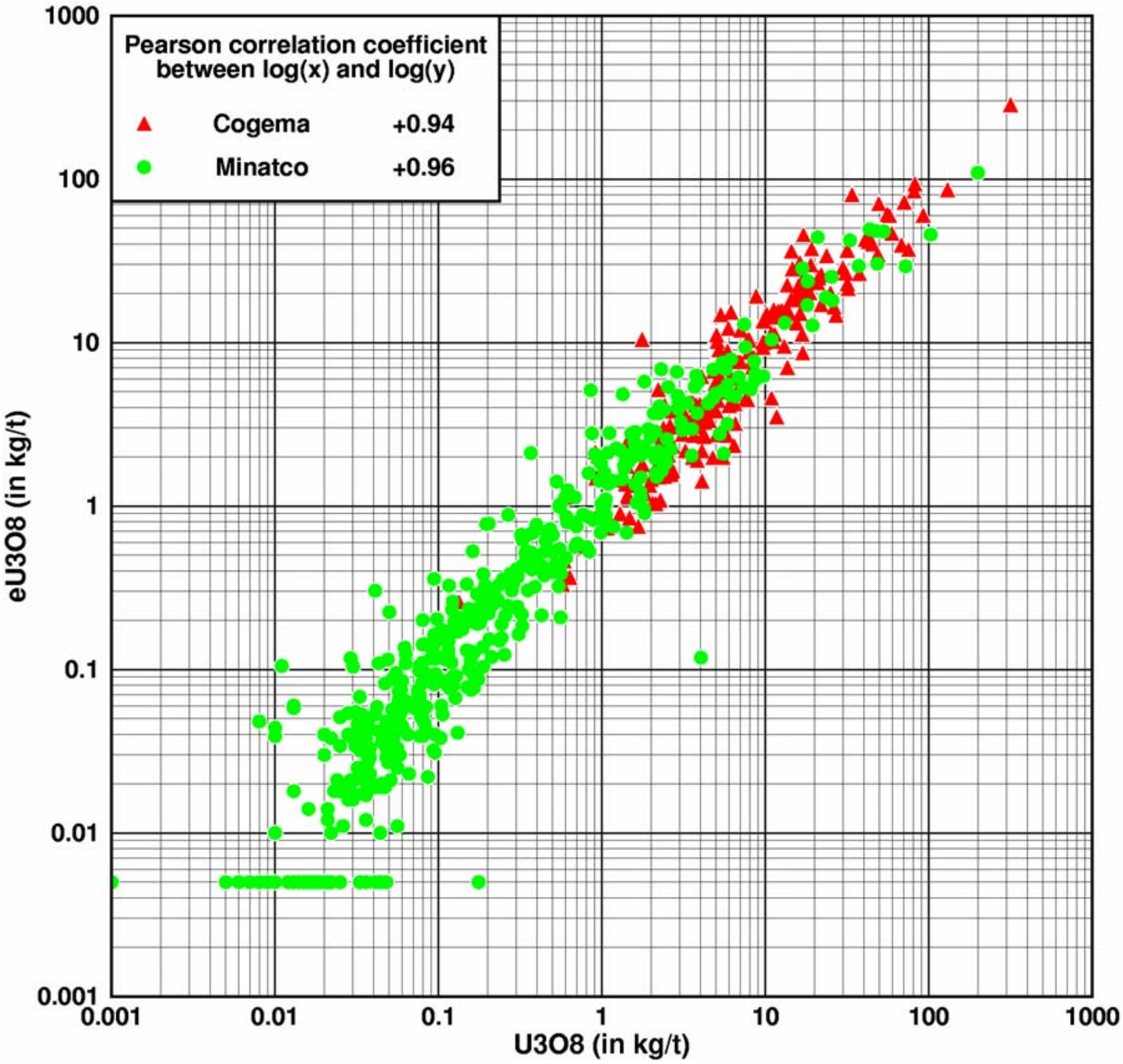
$$\log_{10}[\mathrm{U_3O_8}(\text{in kg/t})] = \sqrt{[\log_{10}(\mathrm{CPS}) - 0.77]^2 - 1.5876} - 2.27$$

For the Cogema holes:

$$\log_{10}[\mathrm{U_3O_8}(\text{in kg/t})] = \sqrt{[\log_{10}(\mathrm{CPS}) + 0.51]^2 - 1.7424} - 1.55$$

Figure 13-9 demonstrates the precision of these e-U_3O_8 assays by comparing the e-U_3O_8 to the chemical assay of U_3O_8 for 3 m composites. Within every 3 m composite interval that contained chemical assays, the average of the U_3O_8 values (i.e. the chemical assays) was compared to the average of the e-U_3O_8 values (i.e. the grades predicted from the radiometric log). The correlation coefficient is very high: 0.96 for the Minatco holes and 0.94 for the Cogema holes. This very strong correlation confirms that e-U_3O_8 assays derived from the downhole radiometric logs are very reliable. The chemical assay database was therefore supplemented with e-U_3O_8 assays wherever chemical assays were unavailable. These additional e-U_3O_8 assays never overlap with existing chemical assays. For any interval where a chemical assay exists, the chemical assay takes precedence; the only intervals in which e-U_3O_8 assays were created were those where no chemical assay was available.

FIGURE 13-9 E-U_3O_8 VERSUS U_3O_8 ON 3 M COMPOSITE INTERVALS



In the 116 drill holes in the project database, a total of 3,189 metres were chemically assayed. The e-U_3O_8 assays add a considerable amount of information to the assay database: 10,304 metres contain e-U_3O_8 assays. The vast majority of these additional e-U_3O_8 assays, however, are virtually barren: 86% are below 0.01% U_3O_8.

Nevertheless, there are a considerable number of the supplementary e-U_3O_8 assays that have significant grades. The chemical assays contain a total of 883 metres with grades above 0.1% U_3O_8; the e-U_3O_8 assays provide additional 194 metres with grades above 0.1% U_3O_8, i.e., an increase of 22%. But at a higher cut-off of 1% U_3O_8, the chemical assays contain a total of 245 metres above this threshold, while the e-U_3O_8 assays provide only 8 metres, an increase of barely 3%. The main effect of adding the e-U_3O_8 assays is therefore to significantly increase the number of assays with moderate levels of uranium mineralization.

RPA recommends that the data from downhole radiometric logs be converted to e-U_3O_8 assays at the earliest possible opportunity. Having e-U_3O_8 assays on hand will accomplish a number of objectives:

1. Since a handheld scintillometer can underrepresent the intensity of uranium mineralization, especially if the core is at surface for some time before being scanned, the e-U_3O_8 assays give a second (and more reliable) prediction of which core intervals need to be chemically assayed.

2. The very good correlation between U_3O_8 and e-U_3O_8 values makes the e-U_3O_8 assays useful for QA/QC of laboratory assays. Any batch for which the e-U_3O_8 values differ from the U_3O_8 assays by more than an order of magnitude should be considered for re-assaying.

3. If chemical assays cannot be obtained for certain intervals, the e-U_3O_8 assays provide a very reliable "backup" value that will lead to more accurate resource predictions than the assumption that all unassayed intervals are barren.

Even if the time required to process the raw data from the downhole radiometric log means that core has already been sent to the lab before the downhole log information is available, the calculation of e-U_3O_8 assays still serves the second and third objectives above.

COMPOSITING

The vast majority of the chemical assays were done for 0.5 m sample intervals. Though the sample interval for the Minatco radiometric logs was also 0.5 m, the sample interval for the Cogema radiometric logs was 0.1 m. The assay database was therefore composited prior to statistical data analysis and grade interpolation. Three metre bench composites (corresponding to the height of the blocks in the block model) were created.

The assays within each composite interval are both length-weighted and density-weighted to calculate the average grade for the composite. For the sample intervals with chemical assays, the density was calculated using the formula based on the sum of the U_3O_8, arsenic, and nickel grades. For the sample intervals with only e-U_3O_8, the density was calculated using the formula based on U_3O_8 alone.

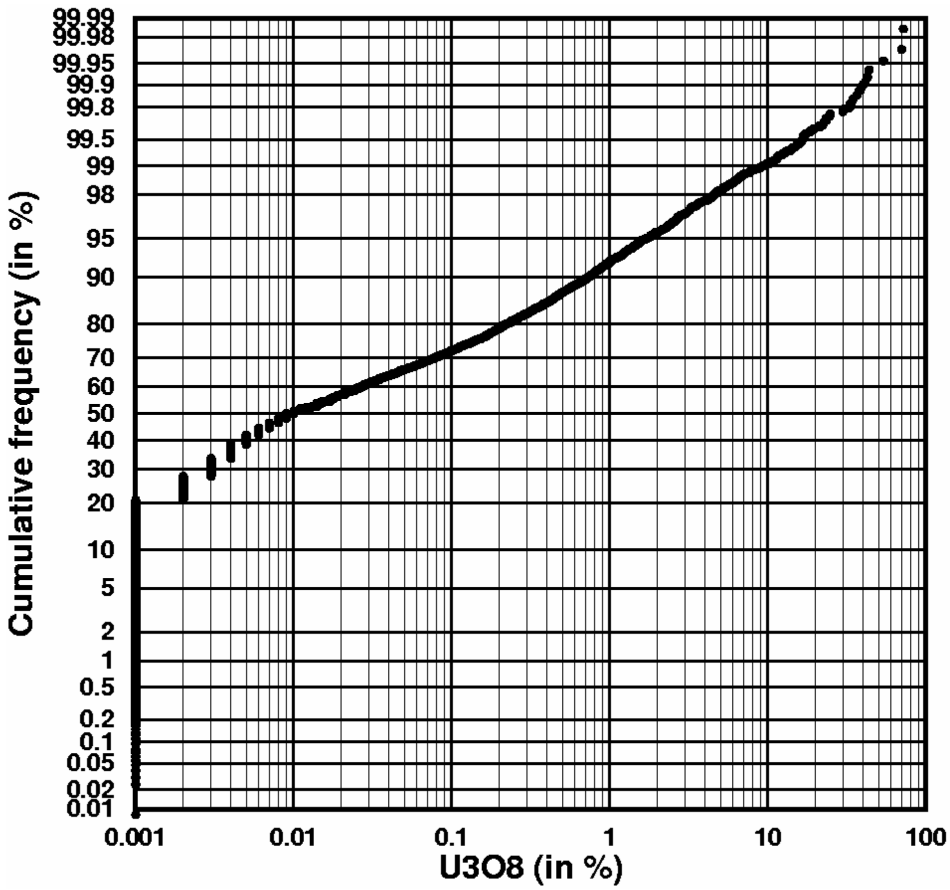
Each composite was assigned a lithology code, either sandstone or basement, depending on whether its midpoint fell above or below the Athabasca unconformity, as recorded in the geological logs. Each composite was also assigned a mineralized/ unmineralized indicator: 0 if the composite grade was less than 0.01% U_3O_8 and 1 if the composite grade was greater than 0.01% U_3O_8.

Composite intervals with less than 0.5 metres of assay information were not included in the composite file. A total of 4,563 3m bench composites were created, 719 in the sandstone and 3,844 in the basement.

CAPPING OF HIGH GRADE URANIUM ASSAYS

Based on the cumulative probability plot of the U_3O_8 assays (Figure 13-10), which shows a break around 30% in the high grade tail of the distribution, U_3O_8 assay values were capped at 30% prior to compositing. A total of 19 assays were affected by this high-grade capping. The effect of this capping on the average grade of the U_3O_8 assays is to reduce the average grade by 5%.

FIGURE 13-10 CUMULATIVE PROBABILITY PLOT OF U_3O_8 ASSAYS



No capping was done of arsenic and nickel assays since the high grade tails of their distribution did not show any breaks or discontinuities.

DATA ANALYSIS

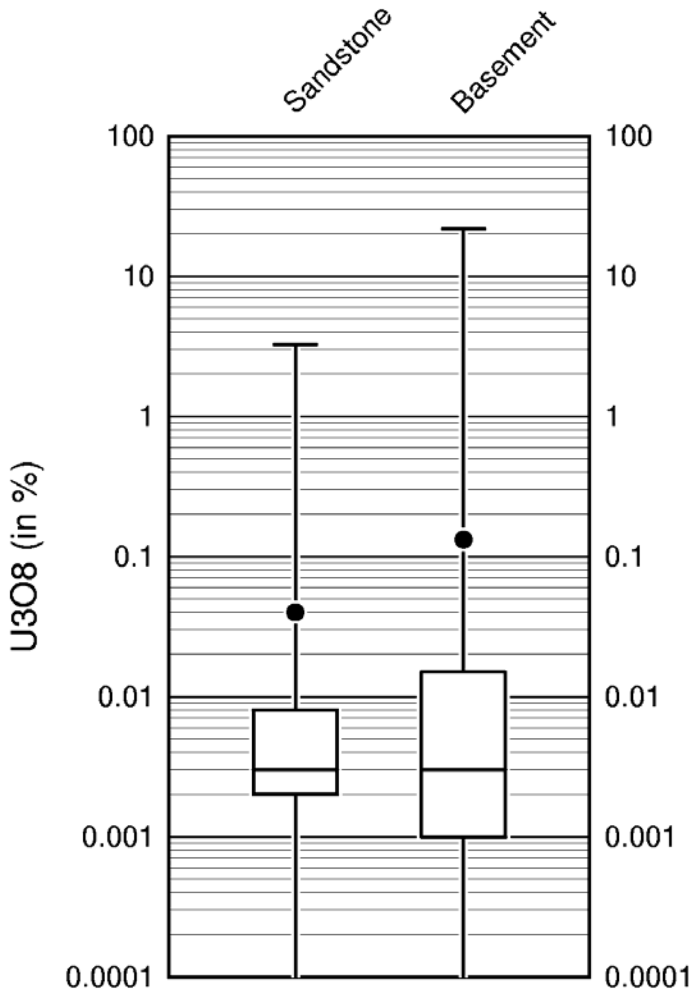
COMPOSITE GRADE DISTRIBUTIONS

Figure 13-11 shows boxplots and summary statistics of the U_3O_8 grades for the 3 m bench composites. The mineralization in the sandstone is generally very weak; 80% of the composites have grades below 0.01% U_3O_8. The composites that do show some moderate uranium mineralization in the sandstone are rarely correlatable from hole to hole, suggesting that the holes are located along steeply dipping structures, such as fractures, that extend into stronger mineralization in the basement below.

In the basement, the uranium grade distribution is extremely skewed, with a coefficient of variation well above 6, even on composites of capped assays. The mean is several orders of magnitude higher than the median, with a handful of extremely high-grade composites having a profound influence on the mean.

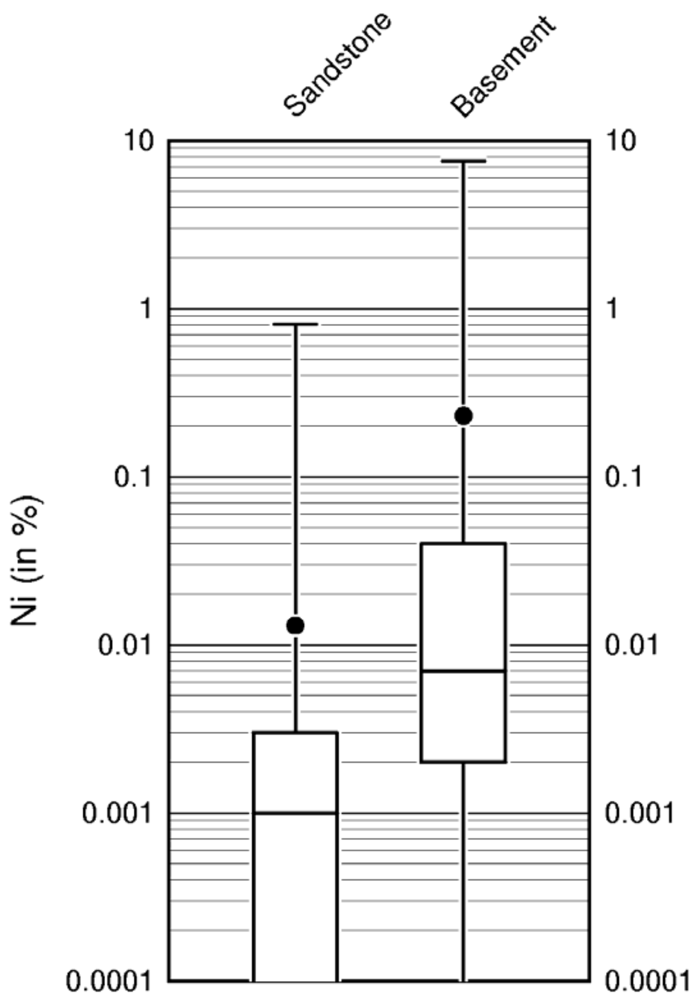
Figures 13-12 and 13-13 show the grade distributions of nickel and arsenic. As with the uranium mineralization, the sandstone generally has very little or no nickel-arsenide mineralization. In the basement, the Ni and As grade distributions are strongly skewed, with coefficients of variation about 3.

FIGURE 13-11 BOXPLOTS AND STATISTICS FOR U_3O_8 COMPOSITES



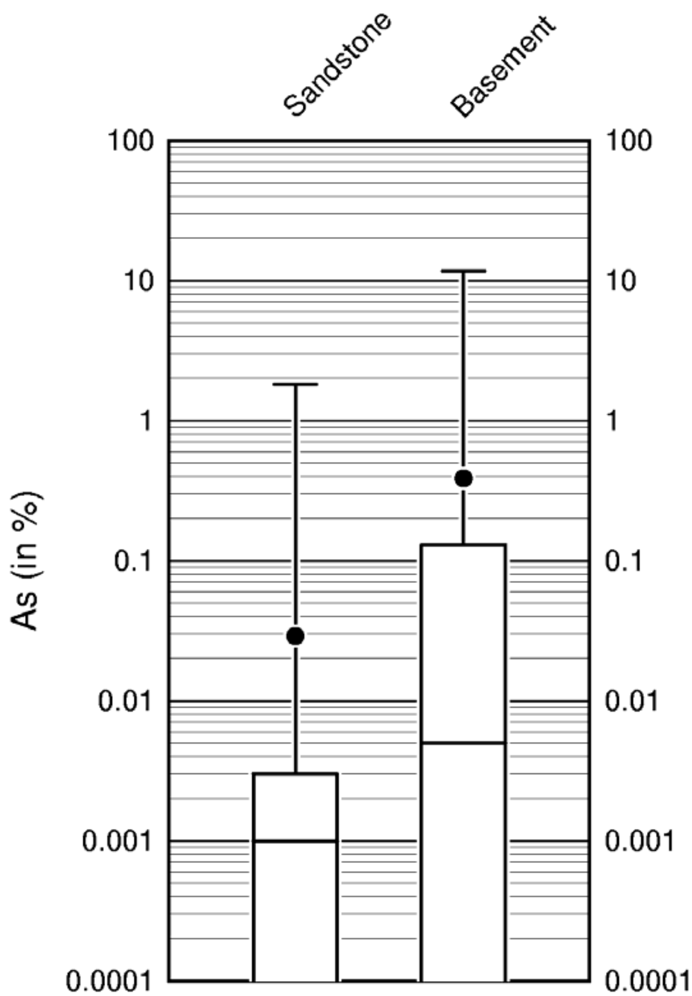
	Sandstone	Basement	
Number	719	3844	Number
Mean	0.040	0.132	Mean
Standard deviation	0.186	0.858	Standard deviation
Maximum	3.248	21.716	Maximum
75th percentile	0.008	0.015	75th percentile
Median	0.003	0.003	Median
25th percentile	0.002	0.001	25th percentile
Minimum	0.000	0.000	Minimum

FIGURE 13-12 BOXPLOTS AND STATISTICS FOR NI COMPOSITES



	Sandstone	Basement	
Number	315	1510	Number
Mean	0.013	0.230	Mean
Standard deviation	0.081	0.748	Standard deviation
Maximum	0.808	7.568	Maximum
75th percentile	0.003	0.040	75th percentile
Median	0.001	0.007	Median
25th percentile	0.000	0.002	25th percentile
Minimum	0.000	0.000	Minimum

FIGURE 13-13 BOXPLOTS AND STATISTICS FOR ARSENIC COMPOSITES



	Sandstone	Basement	
Number	315	1510	Number
Mean	0.029	0.387	Mean
Standard deviation	0.152	1.164	Standard deviation
Maximum	1.802	11.666	Maximum
75th percentile	0.003	0.130	75th percentile
Median	0.001	0.005	Median
25th percentile	0.000	0.000	25th percentile
Minimum	0.000	0.000	Minimum

SPATIAL ANALYSIS AND VARIOGRAMS

Figure 13-14 shows a longitudinal cross-section looking east along the C-C' section line from Figure 13-3. On the right-hand side of this section, the uranium mineralization lies entirely in the basement, closely following the Athabasca unconformity. North of 375N, significant mineralization never extends more than 40 m to 50m below the unconformity. On the left-hand side of this section, the uranium mineralization is more erratic, runs deeper into the basement, and occasionally extends into the base of the Athabasca sandstone. These observations suggest that the uranium mineralization is controlled predominantly by subvertical faults and fractures in the south and by the unconformity in the north.

Figure 13-15 shows a typical cross-section through the northern part of the deposit (the A-A' section from Figure 13-3), and Figure 13-16 shows a typical cross-section through the southern part of the deposit (the B-B' section).

In the north (Figure 13-15), the mineralization clearly follows the Athabasca unconformity, with the geometry of the mineralized envelope dipping gently to the west immediately below the place where a major NNE-trending fault system causes the unconformity to stair-step down to the west.

In the south (Figure 13-16), the mineralization dips steeply to the east; it extends much more deeply into the basement (and slightly into the basal sandstone) and is more erratic than on the A-A' section from Figure 13-15. The attitude of the mineralization is subparallel to major structural discontinuities and/or to the boundaries of graphitic units within the basement gneiss.

Figure 13-17 shows the experimental variograms of U_3O_8 north of 375N; Figure 13-18 shows the experimental variograms of U_3O_8 south of 375N. In the north, the direction of maximum continuity is subhorizontal, with a range of nearly 25 m in directions subparallel to the Athabasca unconformity and less than 10 m in the vertical direction. In the south, the range of spatial correlation of the U_3O_8 grades is shorter than in the north.

FIGURE 13-14 U_3O_8 COMPOSITES ON LONGITUDINAL SECTION C–C´

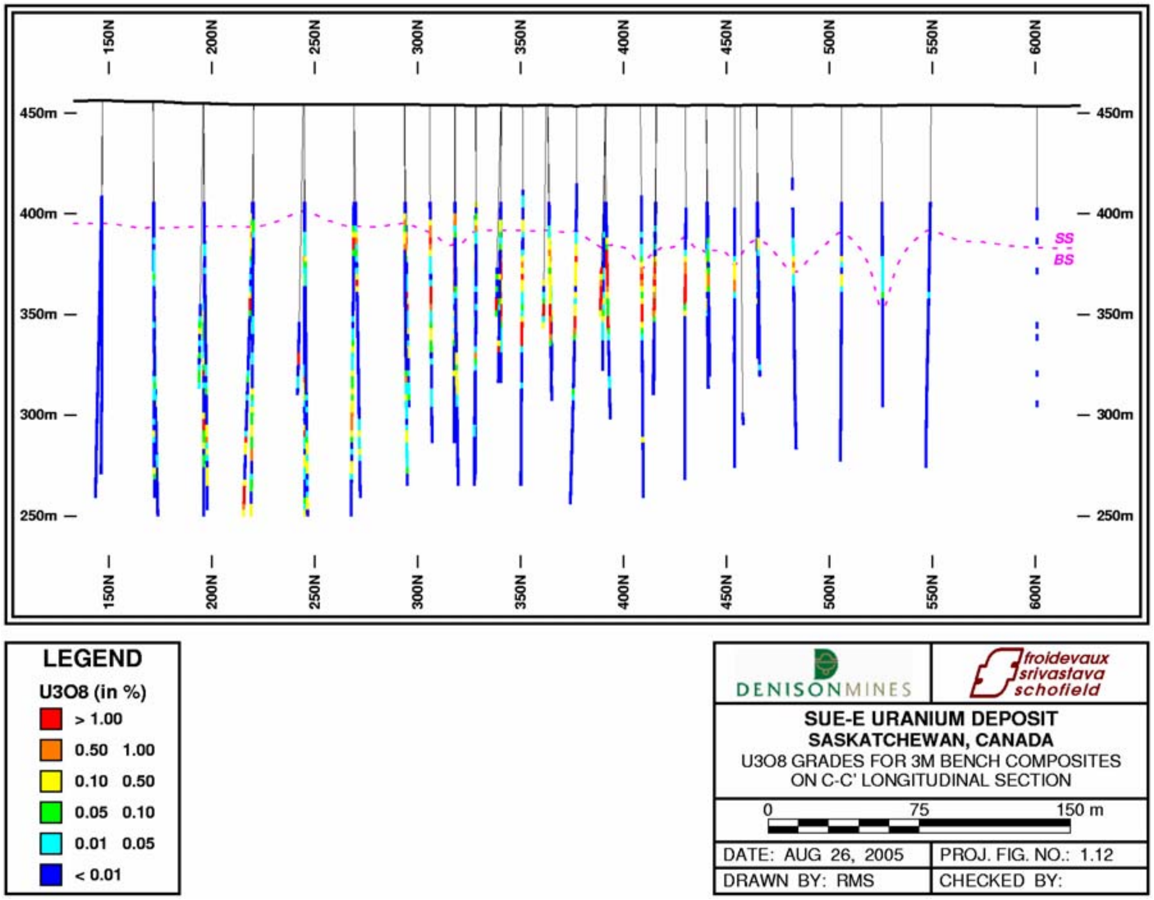


FIGURE 13-15 U_3O_8 COMPOSITES ON CROSS-SECTION A–A´

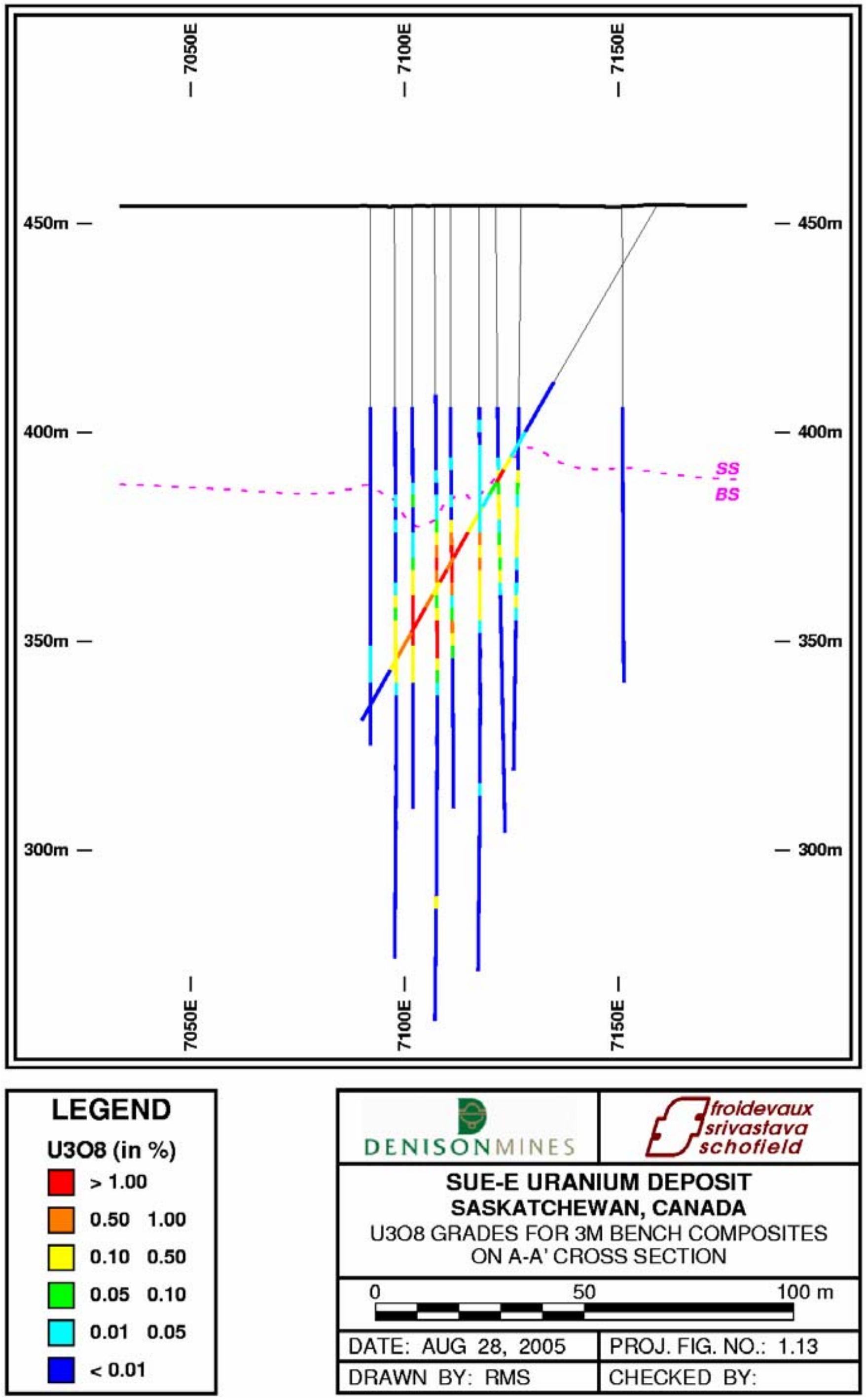


FIGURE 13-16 U_3O_8 COMPOSITES ON CROSS-SECTION B–B´

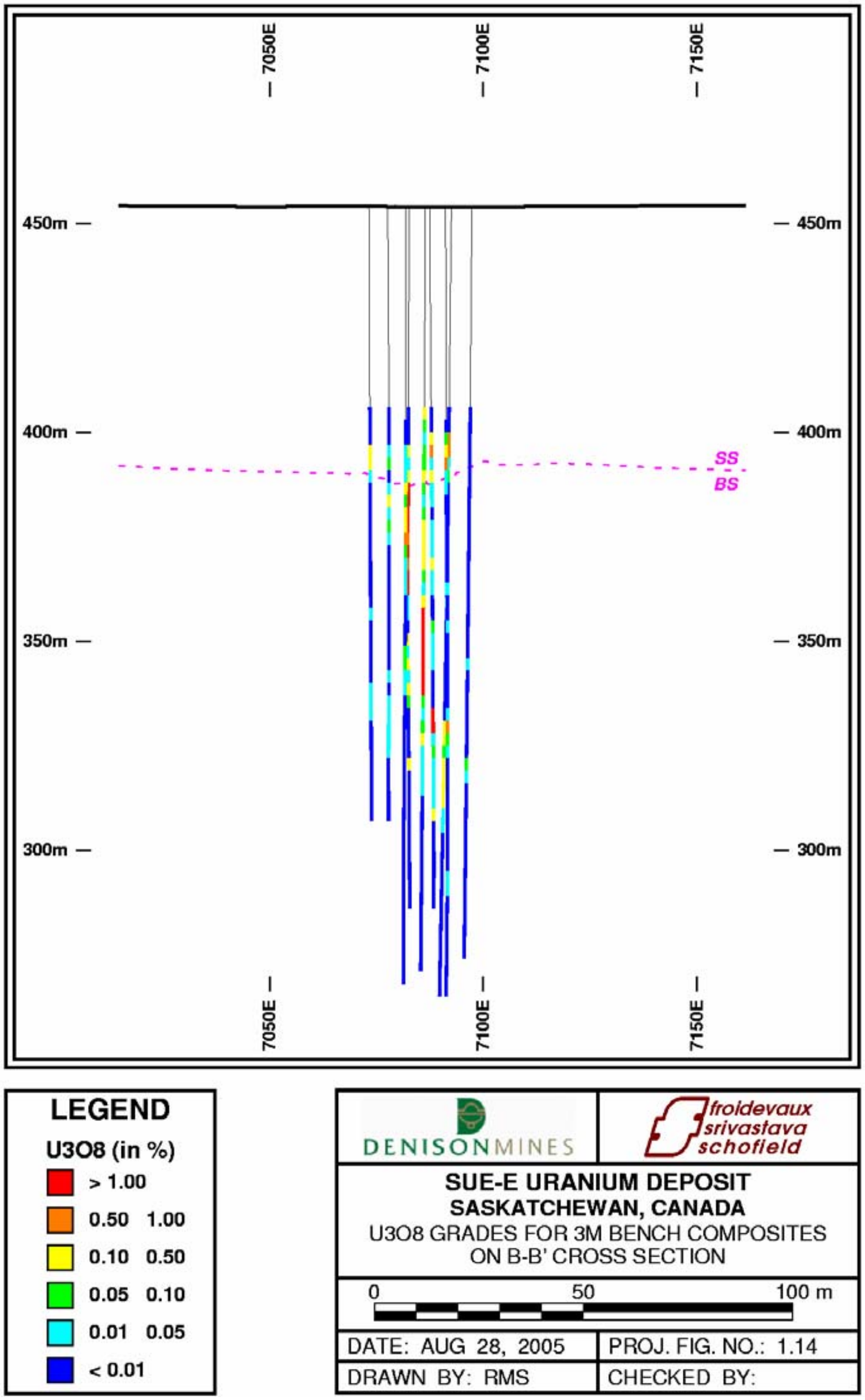
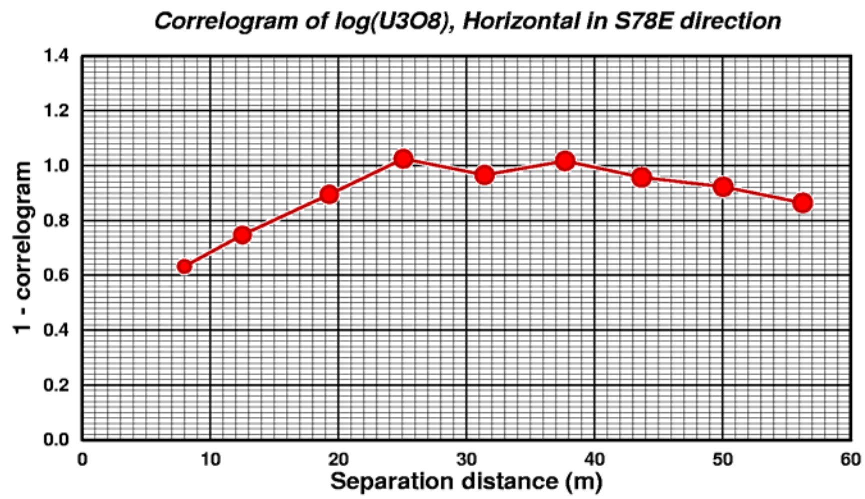
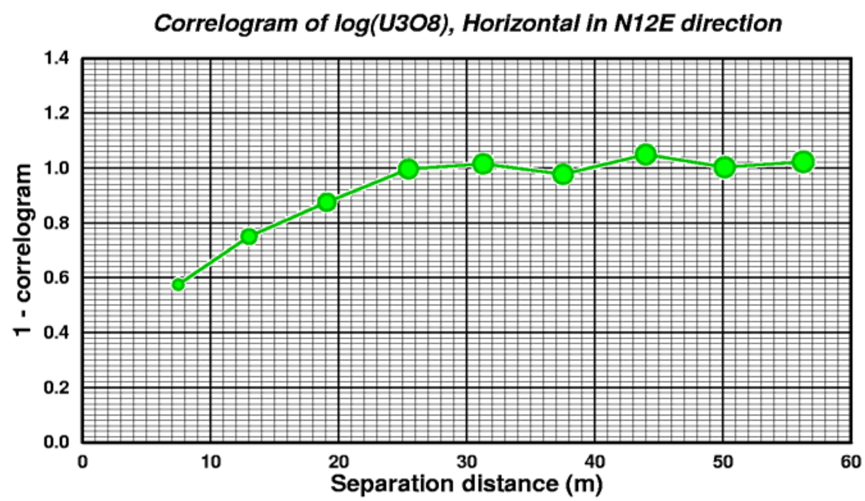


FIGURE 13-17 EXPERIMENTAL VARIOGRAMS OF U_3O_8, NORTH OF 375N





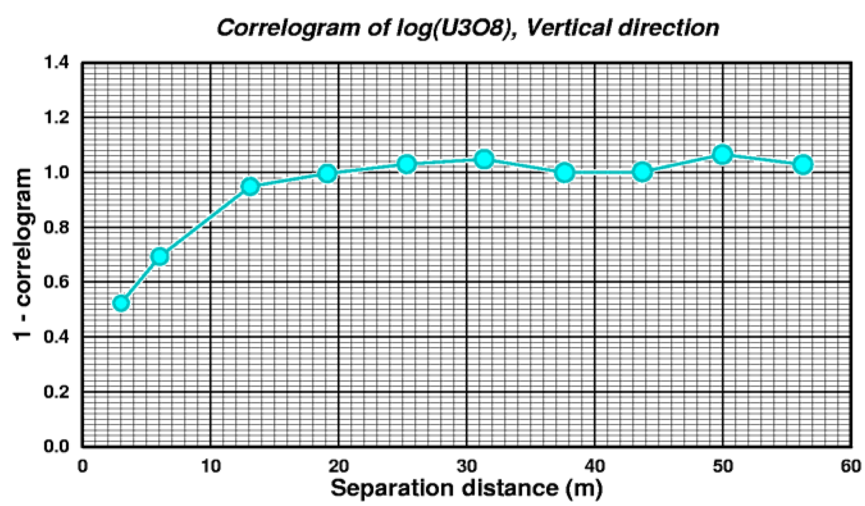
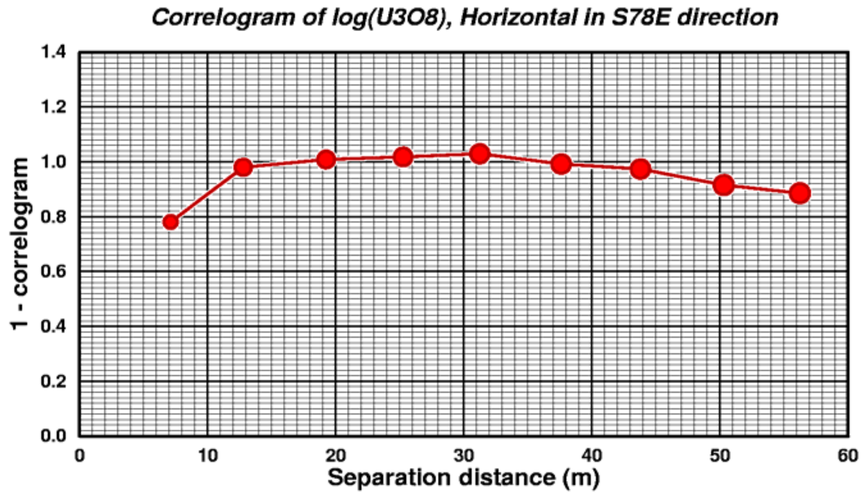
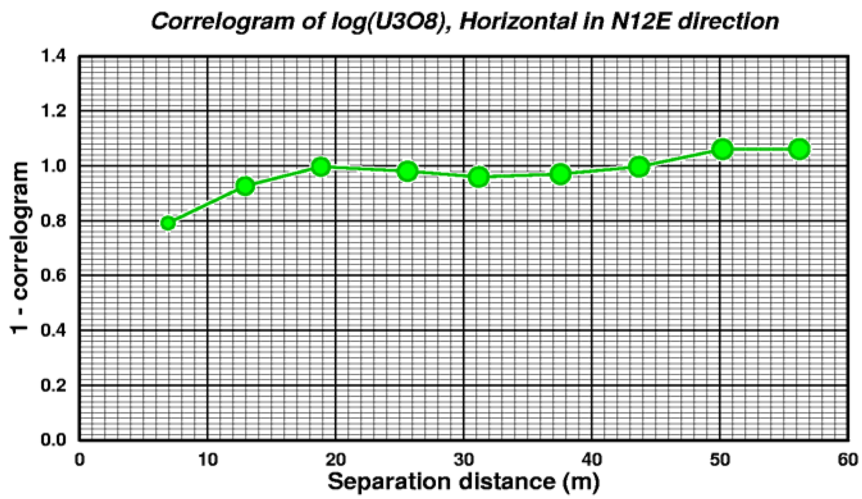
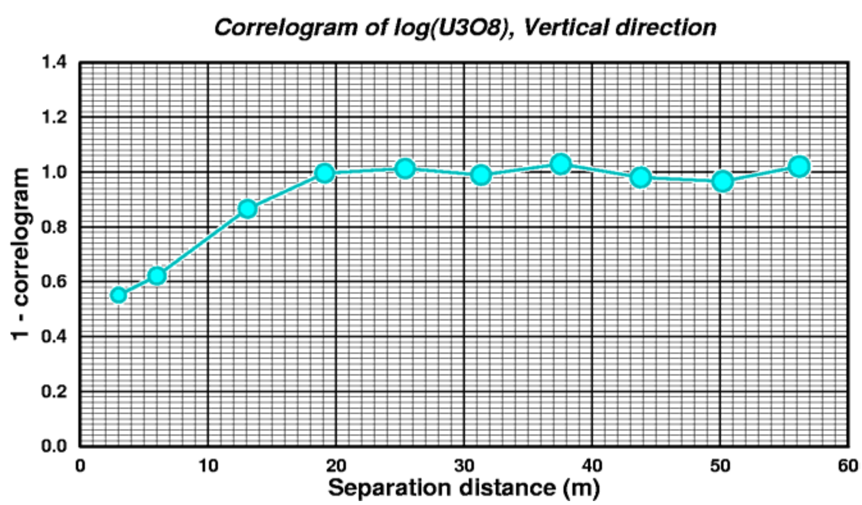


FIGURE 13-18 EXPERIMENTAL VARIOGRAMS OF U_3O_8, SOUTH OF 375N







The longest ranges are in the vertical direction and along the strike of the mineralization (N12ºE), where they reach about 15 m; in the across-strike direction (N78ºW), it is difficult to see a range of correlation, even with a drill hole spacing that is often less than 10 m in this direction.

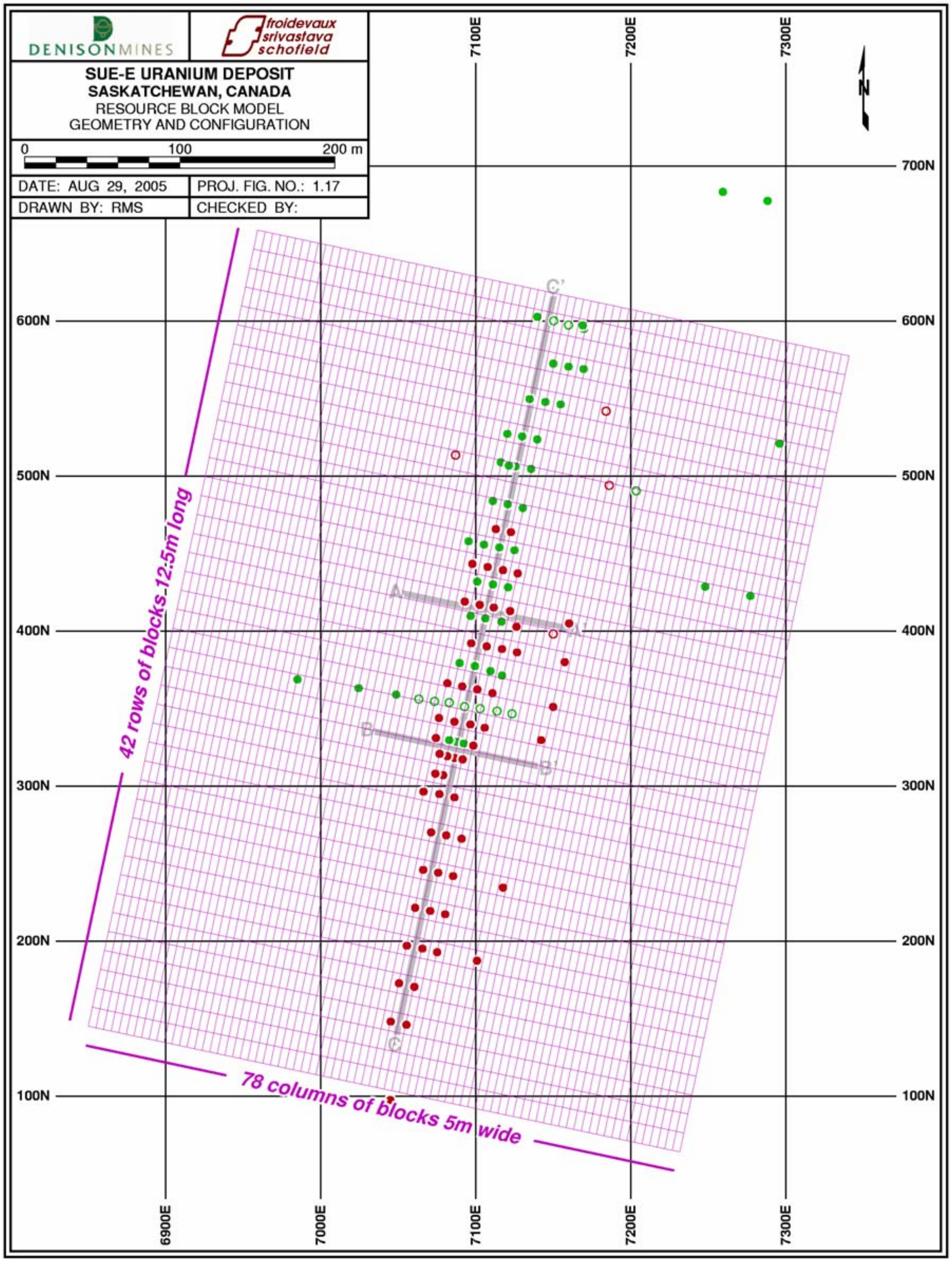
BLOCK MODEL GEOMETRY

Figure 13-19 shows the configuration of the block model in plan view. The blocks are 12.5 m along the strike of the deposit (N12ºE), 5 m across the strike of the deposit (N78ºW), and 3 m high in the vertical direction. The origin (bottom southwest corner) of the model is at 6850E, 145N, 250 m. The block model has 78 columns across the strike of the deposit, 42 rows along the strike of the deposit, and 70 levels from top to bottom.

A 12.5 m ×5 m × 3 m block will contain roughly 450–550 tonnes of rock. Based on the current and recent production practice at the other Sue deposits and on a review of past mining planning studies for Sue E, this volume of rock corresponds well to the selective mining unit, the smallest volume of material that can effectively be segregated as ore or waste during the open pit mining operation.

The block model is considerably wider, and slightly longer than needed purely for resource estimation purposes. The additional rows and columns were included so that the crest of an earlier ultimate pit design, the one referred to as “Design 17” by Cogema, would fit inside the block model region.

FIGURE 13-19 BLOCK MODEL CONFIGURATION



MINERALIZED ENVELOPE

A mineralized envelope was developed to delineate the mineralized core of the deposit and to separate this core from the surrounding waste. The mineralized core was defined using indicator kriging, firstly to map out in 3D the regions that had a high probability of being mineralized and secondly to manually refine this region to incorporate geological constraints such as the Athabasca unconformity, the major faults, and the interpreted contacts of graphitic units.

The indicator kriging (IK) was done using the 0/1 indicators defined on the 3 m composites, using simple kriging with an isotropic spherical variogram model with a range of 15 m in all directions. For each block in the resource block model, this IK produces a value between 0 and 1 that can be interpreted as a probability that the block has mineralization above 0.01% U_3O_8.

The blocks with a greater than 50% chance of being at least weakly mineralized were then reviewed in section and manually edited to: (1) remove all blocks above the Athabasca unconformity, and (2) to be consistent with RPA's manual interpretation of ore geometry in regions where the interpretation was following either an observed fault or an observed graphitic contact and was well constrained by two or more drill holes.

The result of this two-step procedure is a mineralized envelope defined as a list of blocks that fall within the mineralized core of the deposit. The 3 m composites within this envelope were used to estimate grades inside the envelope. All blocks outside this envelope were assumed to be barren.

RESOURCE ESTIMATION PROCEDURE AND PARAMETERS

INTERPOLATION METHODOLOGY

Ordinary kriging (OK) was used to calculate weights for composites that fall within the search neighbourhood. These OK weights were multiplied by the length of the composite and by the calculated dry density, and then renormalized to sum to one. The weighted average of the nearby composite U_3O_8 grades is then calculated and used as the estimate of the average grade of the entire 12.5 m × 5 m × 3 m resource block.

The same OK weights were used to calculate an average specific gravity for the block, using the calculated dry density of the nearby composites that fall within the search neighbourhood.

Many of the composites used in the resource estimation contain only e-U_3O_8 values and have no chemical assays. For the estimation of arsenic and nickel grades, any nearby composites that lacked chemical assays were assigned default background grades: 0.095% Ni and 0.089% As. These background default values were calculated by taking all low-grade (<0.01% U_3O_8) composites within the mineralized envelope that had chemical information and calculating their average nickel and arsenic grades. Once these background default values had been assigned, the same composites that formed the basis for the U_3O_8 estimation could also be used for nickel and arsenic estimation, and the OK weights used for U_3O_8 and SG could also be used for nickel and arsenic interpolation.

Resource blocks were deemed to be inside the mineralized envelope if they were included in the list of blocks discussed above. Since the mineralized envelope is not explicitly represented as a wireframed solid, there is no partial blocking or subblocking required; all blocks are deemed to be either entirely inside the mineralized envelope or entirely outside.

SEARCH STRATEGY

The search neighbourhood for each block is defined by an ellipsoid centered at the centre of the block. The lengths of the ellipsoid's axes are the same as the ranges of the variogram model. Composites that fall beyond the search ellipsoid are not used in the estimation. Within the search ellipsoid, a maximum of two composites from each drill hole are used.

VARIOGRAM MODEL PARAMETERS

Tables 13-2 and 13-3 list the variogram model parameters used in the resource estimation. With different patterns of spatial continuity in the north and south, the deposit was split into two domains, with the variogram model in the northern region having good continuity in subhorizontal directions parallel to the unconformity and the variogram model in the southern region having good continuity in the vertical direction and along the strike of the deposit.

TABLE 13-2 VARIOGRAM MODEL PARAMETERS, NORTH OF 375N
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

	Direction	Nugget effect	Type	Range
Major	N12ºE, dipping 0º	0.45	Exponential	25m
Intermediate	N78ºW, dipping 0º	0.45	Exponential	25m
Minor	Vertical	0.45	Exponential	10m

TABLE 13-3 VARIOGRAM MODEL PARAMETERS, SOUTH OF 375N
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

	Direction	Nugget effect	Type	Range
Major	S78ºE, dipping -80ºE	0.58	Exponential	18m
Intermediate	Vertical	0.58	Exponential	18m
Minor	S78ºE, inclined +10ºE	0.58	Exponential	6m

All variogram models are single-structure exponential models that have their final sill at 1.0; the contribution of the exponential structure is therefore 1.0 minus the nugget effect.

RESOURCE CLASSIFICATION

Resources were classified into "indicated" and "inferred" categories using the following criteria:

- A block that had at least one nearby composite within 10 m of its centre, and that had composites from at least two different drill holes in its search neighbourhood was classified as part of the "indicated" resource.

- All blocks within the mineralized envelope that were not classified as "indicated" were classified as "inferred".

No blocks were classified as "measured". With the extreme skewness of the U_3O_8 grade distribution, the significant nugget effect, and the short ranges of correlation, RPA is of the opinion that none of the resource currently meets the CIM definition of a "measured" resource since the continuity of mineralization cannot be "demonstrated" in the sense required by the CIM guidelines.

With at least one sample and two different drill holes within the range of the variogram, the "indicated" resource meets the CIM definition of a region in which the sampling is sufficient for geological and grade continuity to be reasonably assumed.

With at least one sample within the range of the variogram, and falling inside a mineralized envelope that has been tightly constrained by geologic and statistical considerations, the "inferred" resource meets the CIM definition of a region within which the grades and geology are likely continuous.

Figures 13-20 through 13-22 show longitudinal and cross sections through the classified resource block model for the same sections shown earlier in Figures 13-14 through 13-16.

FIGURE 13-20 CLASSIFIED RESOURCE BLOCK MODEL, SECTION C-C´

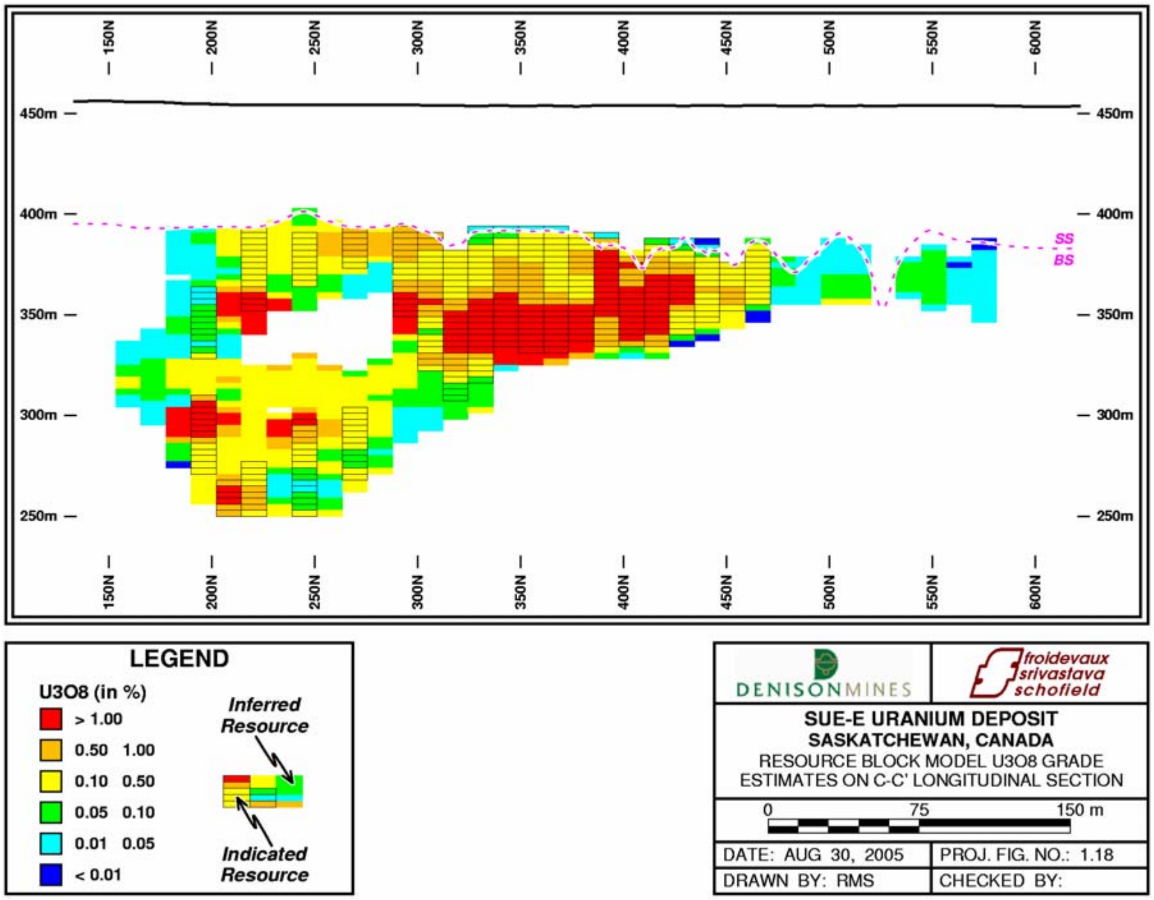


FIGURE 13-21 CLASSIFIED RESOURCE BLOCK MODEL, SECTION A-A´

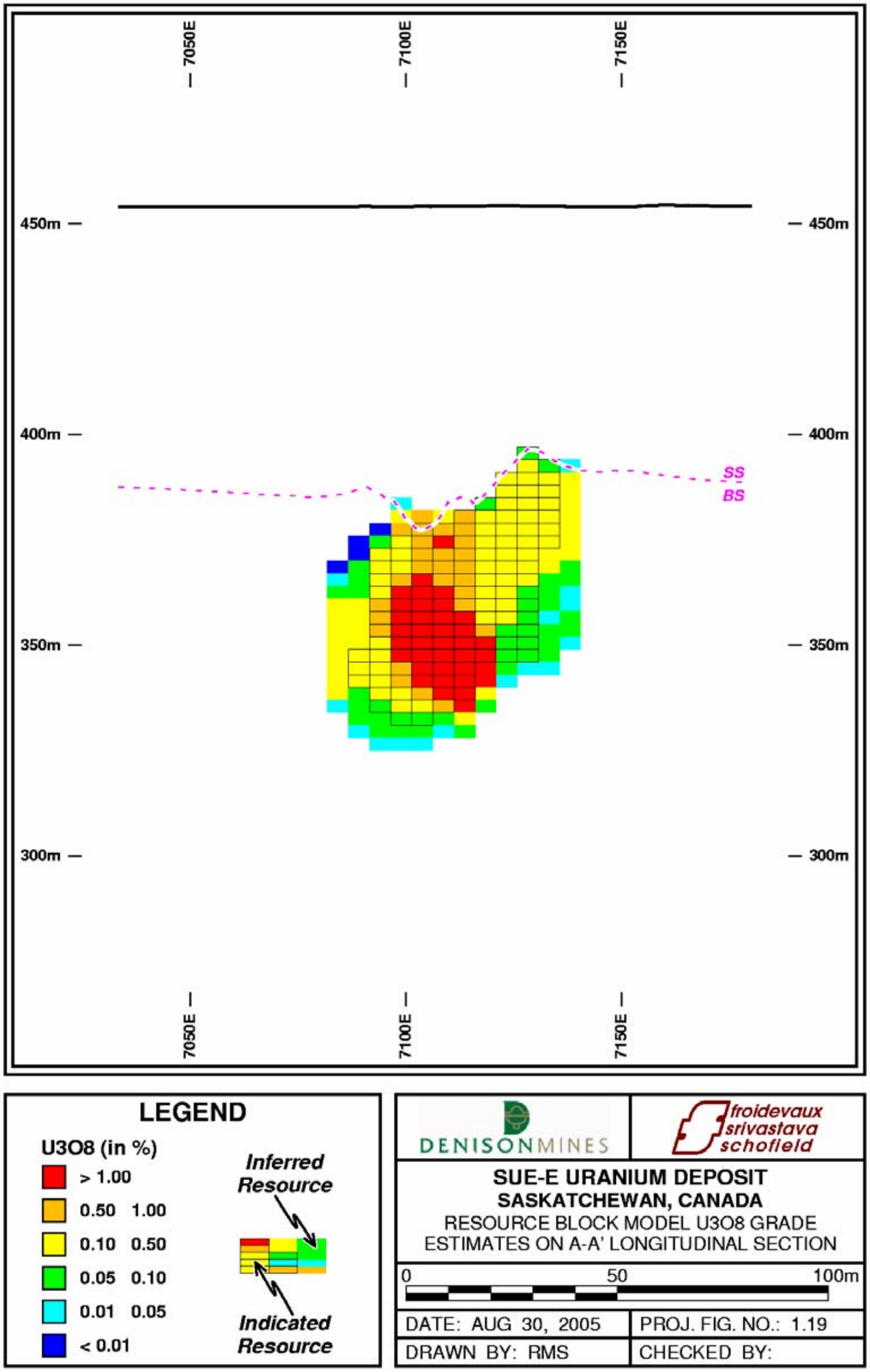
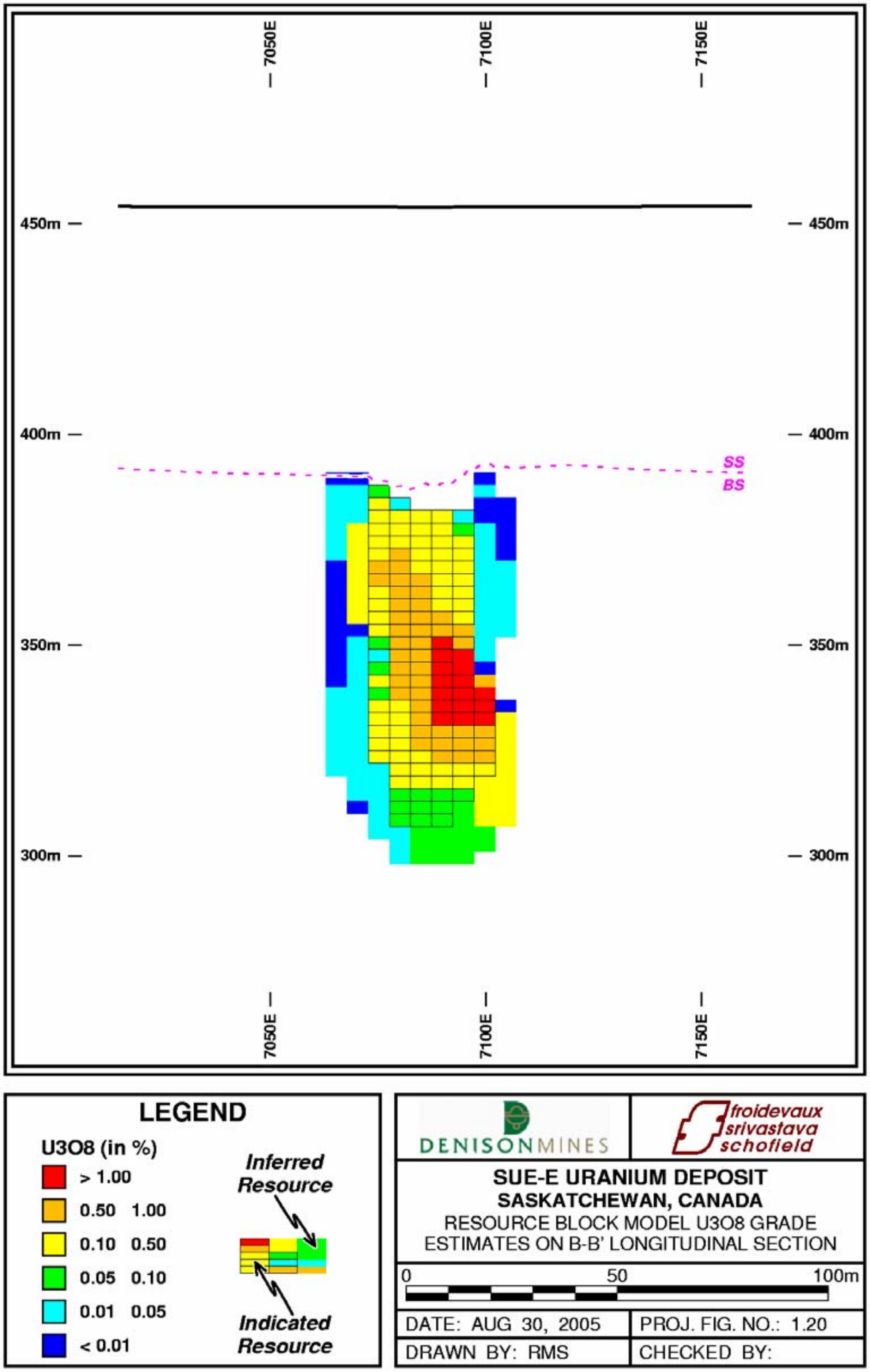


FIGURE 13-22 CLASSIFIED RESOURCE BLOCK MODEL, SECTION B-B´



RESOURCE SUMMARY

Table 13-4 summarizes the estimated Sue E Indicated Resource for various block cut-offs from 0.1% U_3O_8 to 1.0% U_3O_8. Based on RPA's review of U_3O_8 prices and mining operating costs, the 0.1% U_3O_8 cut-off grade is reasonable for conversion to Mineral Reserves. Table 13-5 a summary of the estimated Inferred Resource.

TABLE 13-4 SUE E ESTIMATED INDICATED RESOURCES
McClean Lake Joint Venture McClean Lake Property, Saskatchewan*

Cut-Off	Tonnage*	Grade		Contained Metal*			
(in %U_3O_8)	(in t)	(in %U_3O_8)	(in %U)	(in kg U_3O_8)	(in kg U)	%As	%Ni
0.10	**718,285**	**0.775**	**0.657**	**5,565,942**	**4,719,919**	**0.801**	**0.487**
0.20	547,630	0.970	0.823	5,313,165	4,505,564	0.996	0.603
0.50	312,278	1.456	1.235	4,547,207	3,856,031	1.477	0.894
1.00	151,671	2.234	1.894	3,387,606	2,872,690	2.087	1.282

*Denison Mines Inc. holds 22.5% interest in the MLJV and the above Resources

TABLE 13-5 SUE E ESTIMATED INFERRED RESOURCES
McClean Lake Joint Venture McClean Lake Property, Saskatchewan*

Cut-Off	Tonnage*	Grade		Contained Metal*			
(in %U3O8)	(in t)	(in %U3O8)	(in %U)	(in kg U3O8)	(in kg U)	%As	%Ni
0.10	**780,261**	**0.685**	**0.581**	**5,343,412**	**4,531,213**	**0.916**	**0.575**
0.20	463,157	1.059	0.898	4,904,993	4,159,434	1.386	0.867
0.50	209,737	1.956	1.659	4,103,244	3,479,551	2.149	1.219
1.00	109,226	3.117	2.644	3,405,090	2,887,516	2.960	1.678

*Denison Mines Inc. holds 22.5% interest in the MLJV and the above Resources

There is considerable tonnage and metal contained in the estimated inferred resource. Several factors contribute to this:

- Sparse drilling (relative to the range of the variogram), especially in the southern domain;

- The need for "indicated" resource to have at least two different drill holes within the search;

- Some of the extremely high-grade samples occur in the deep southern region and are not supported in the samples from surrounding drill holes.

With the current drilling, these estimates of the inferred resource are considerably less reliable than the estimates of the indicated resources. But with significant additional potential resources in the inferred category, more definition drilling is warranted, particularly in the deep southern extensions of the deposit. RPA also notes that an earlier recommendation to drill some east-dipping holes from the west has not yet been implemented. The prevailing view of the deposit has been of steeply east-dipping structures. While this study supports this as a general observation about the mineralization in the south, it also suggests that the mineralization in the north may generally dip to the west. If this interpretation is correct, east-dipping holes would help to better delineate the ore outlines in the north.

ADDITIONAL RESOURCE POTENTIAL

In addition to the resources that further drilling might be able to graduate from "inferred" to "indicated", there is some additional resource potential in the basal sandstone. This study, in keeping with previous studies, has clipped the mineralized envelope to the Athabasca unconformity. The current drilling contains several significant (>1% U_3O_8) showings in the basal sandstone; most of these are in the south. There are currently too few of these, and they are too far apart to make any reliable estimation of uranium resources in the sandstone. If additional drilling continues to encounter moderate to occasionally strong uranium mineralization near the base of the Athabasca, RPA recommends that an attempt be made to model this additional geologic domain. If the showings remain erratic and difficult to correlate from hole to hole, then RPA recommends that the regions with such showings be delineated. When the open pit reaches these levels, more detailed mapping and in-pit sampling can be used to determine whether there are pods of mineralization in the basal sandstone that can be effectively segregated as ore.

MINERAL RESERVE ESTIMATES

MINING

CURRENT MINE DEVELOPMENT PLAN

The Sue E deposit is currently planned to be developed and mined starting in 2005 in order to supplement and continue to supply mill feed materials for the JEB mill. As the current Sue C ore stockpile declines through the ongoing draw down of feed materials for the JEB mill ore materials from Sue E pit will be mined and delivered to the mill stockpile to sustain operations. The Sue E mining plans are extensions of the mining operations already carried out at the JEB and Sue C pits.

As outlined above, RPA has developed a resource block model for the Sue E deposit. This model has been used as the basis for the open pit economic optimization analysis using the Whittle 4X software program. The open pit economics have been evaluated using the Indicated class resources only. The significant quantities of Inferred class material indicate that there may be some opportunity to further optimize the mine development if this material can be upgraded to Indicated class through further drilling. The operating cost factors that have been used in this analysis are based on actual operating experience at the MLJV and on the use of the existing mine equipment fleet and operating practices.

The key input parameters to the economic analysis are summarized in Table 13-6. The uranium price of US$23.50 per lb. U_3O_8 used in this calculation is based on price quotations current in March 2005; at a US/Cdn exchange rate of $0.81, it amounts to C$29.00 per lb. U_3O_8. Downstream transportation, handling, and selling costs have been deducted resulting in a net back to the mine site value of C$26.39. The uranium recovery factors reflect the current expectation based on metallurgical test results. The pit slope angles are based on the MLJV actual operating experience at the nearby Sue C open pit.

In addition, RPA evaluated the potential value associated with nickel metal in the deposit. Unfortunately, no cobalt values were available in the assay database;

consequently, they have not been considered even though cobalt is likely present. In the open pit economic analysis, the nickel and cobalt values were not found to result in a significant change in the pit shape or overall design. If the nickel and cobalt values are recovered, then they will serve to improve the overall economic performance of the project; however they will not materially alter the mining plan. These nickel values were assessed based on a price of US$ 3.75 per lb. of nickel.

TABLE 13-6 SUE E OPTIMIZATION PARAMETERS
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

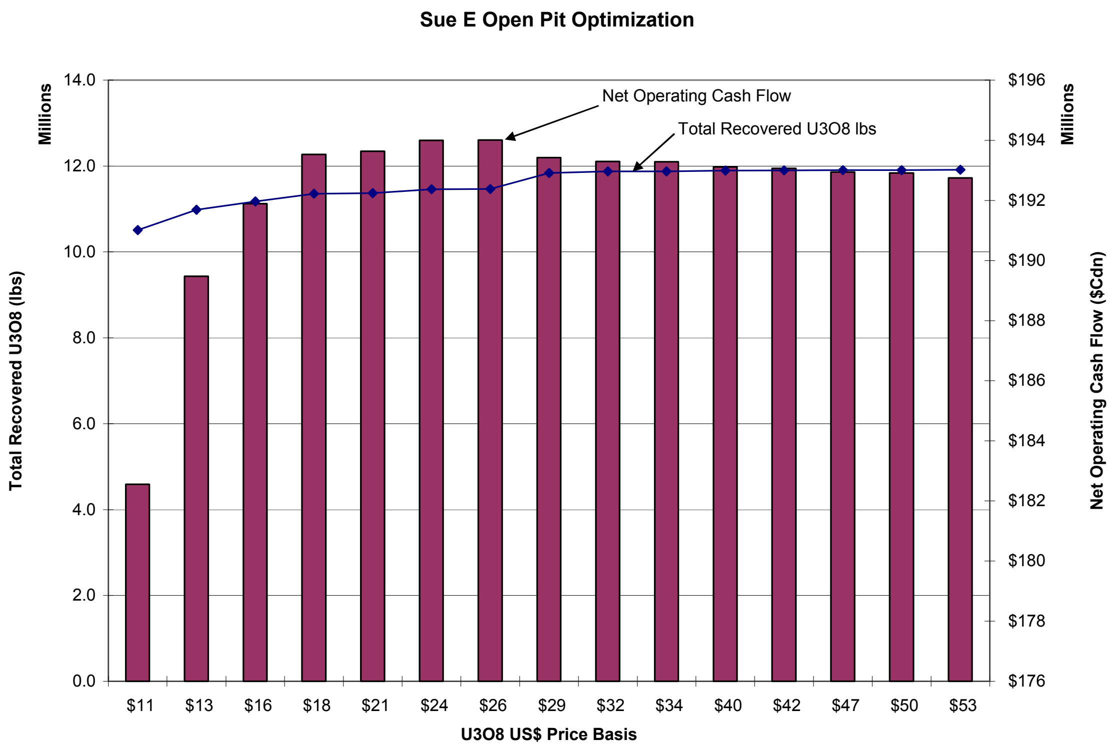
Item		Unit	Value
Operating Cost			
	Waste Mining	$/t mined	$4.10
	Ore Mining	$/t mined	$4.10
	Processing	$/lb U3O8	$5.08
	Ni & Co Processing	$/lb Ni	$ 0.50
	Admin & Overhead	$/t lb U3O8	$1.52
Process Recovery			
	U_3O_8 Recovery	%	95%
	Nickel Recovery	%	54%
Revenue Parameters			
	Net U_3O_8 NSR (after offsite charges)	$C/lb U_3O_8	$ 26.39
	Ni NSR/lb Ni in Precip.	$C/lb Ni	$ 3.00
Pit Slope Angles			
	Overburden	degrees	28º
	Sandstone	degrees	42º

Source: RPA

The optimization results show the Sue E pit economics to be robust, with pounds of U_3O_8 recovered being relatively insensitive to changes in metal price. The results are illustrated graphically in Figures 13-23 and 13-24. Over a range of uranium prices from a low of C$11 through C$29 (260% variance), the recovered lbs of U_3O_8 only changed by 13%. Beyond the C$29 price level there is no increase in the recovered uranium as the bulk of the known deposit has been mined out. Additional runs of the pit economics model were carried out to test the impact of the incremental value contributed by the nickel content. It was found that the nickel did not materially change the shape of the pit.

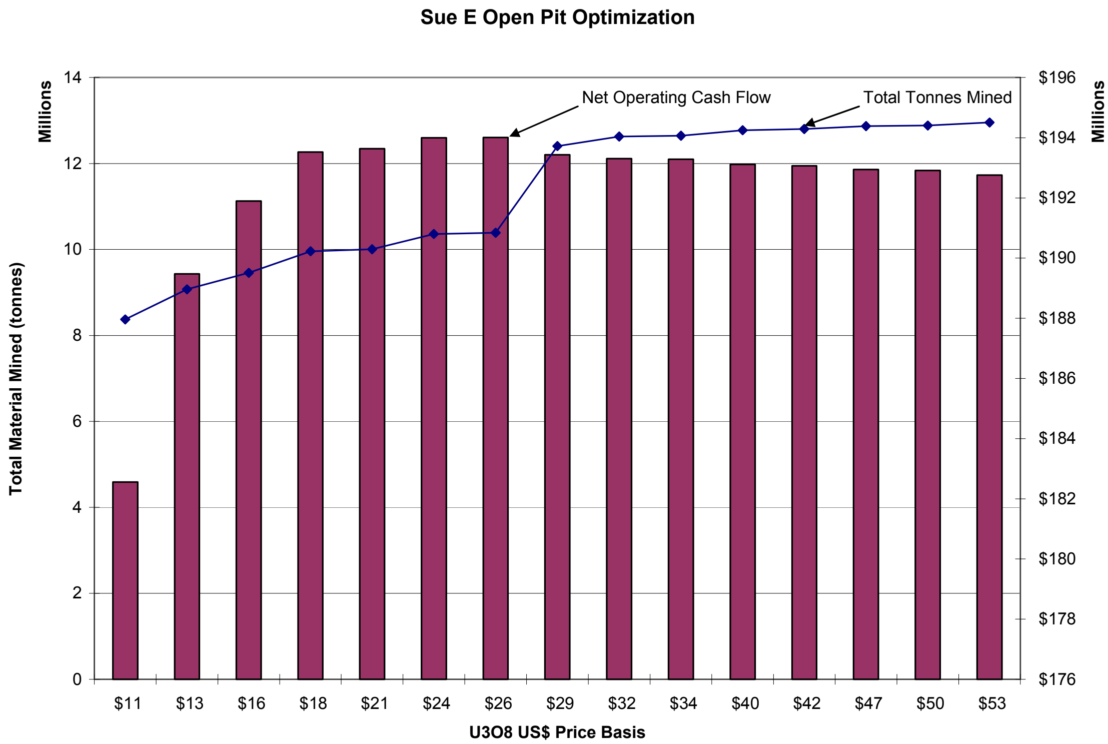
Figure 13-24 shows the impact on pit size in terms of total excavation material over the same series of pit shells. The range of prices used to test the economic limits ranged from C$11 to C$53 per lb. The total pit excavation volume increases from approximately 10 million tonnes of material mined at the lower range of uranium prices up to 12 million tonnes of material mined when the price rises beyond C$26. In general the economic pit size is relatively insensitive to uranium price.

FIGURE 13-23 OPEN PIT OPTIMIZATION – U_3O_8 MINED



Source: RPA

FIGURE 13-24 OPEN PIT OPTIMIZATION – PIT SIZE



Source: RPA

CUT-OFF GRADE

The MLJV has historically operated at the Sue C deposit based on an ore cut-off grade of 0.1% U_3O_8 to define ore scheduled for processing at the JEB mill facility versus discard material. RPA reviewed this cut-off grade against the current economic factors including operating costs, metallurgical recovery, and revenue criteria, and determined that it represents a reasonable cut-off level. For the purposes of this analysis, RPA has applied the MLJV cut-off policy in developing the pit optimization and mine development plans.

MINE DESIGN

The optimum pit shell developed in the RPA Whittle analysis has been used as the basis for developing an ultimate pit limit design, incorporating catchment berms and a haulage ramp for access to the bottom of the pit. The pit ramp has been designed at 12%, suitable for conventional mechanical drive haulage trucks. The pit slopes have been designed based on the pit slope experience at the Sue C pit. The ultimate pit design is illustrated in Figure 13-25.

The final recoverable resources estimated to be within the detailed pit design limits are summarized in Table 13-7. The resource grade model is smoothed and incorporates internal dilution as a feature inherent in the indicator kriging process used to outline blocks with the probability of containing $U_3O_8\% \geq 0.01\%$ cut-off and then construct a wireframe enclosing these blocks as described previously. As such RPA believes the resource model carries sufficient dilution and no additional dilution was added to convert the in-pit resource blocks to reserves.

On the basis of the estimates and forecasts presented, RPA concludes that the Mineral Reserves are consistent with the definitions set out in NI 43-101 and defined by the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by the CIM Council on August 20, 2000.

TABLE 13-7 SUE E PROBABLE RESERVE (AS OF JAN.1, 2005)
McClean Lake Joint Venture McClean Lake Property, Saskatchewan*

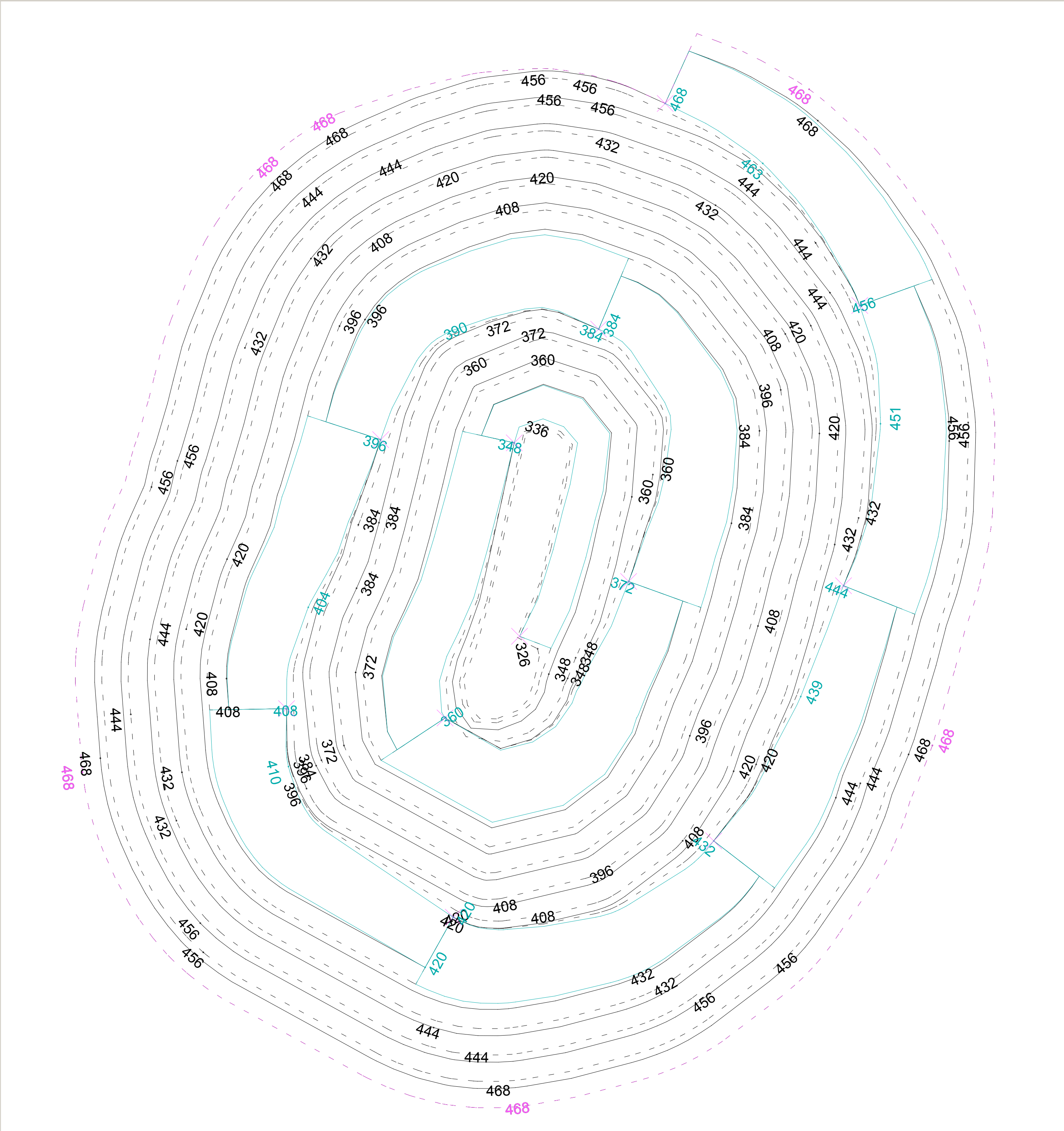
Total Material (BCM)	Waste (BCM)	Special Waste (BCM)	Ore (Tonnes)*	U_3O_8 Grade (%)*	Nickel Grade (%)*
5,459,025	5,082,581	114,173	628,077	0.78%	0.53%

*Denison Mines Inc. holds 22.5% interest in the MLJV and the above Reserve

MINE PRODUCTION PLAN AND SCHEDULE

The initial mining activity will primarily involve the excavation of waste materials, both barren and contaminated waste, above the ore zones. As mining progresses downward, the proportion of waste mined will decrease and the quantities of ore recovered will increase.

The Sue E pit is located to the south of the Sue C pit. It does not intersect the Sue C pit and will be entirely independent and separate from it. The Sue E pit is approximately 140 metres deep. Mining operations will be carried out using 12 metre high benches.



RPA

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GEOLOGICAL AND MINING CONSULTANTS

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Figure 13-25

SUE E PROJECT

PIT DESIGN

MINE SITE PLAN

A general arrangement plan for the mine site providing for access roads, waste dump facility, special waste stockpile area, and ore blending yard is illustrated in Figure 13-26.

The Sue E pit mining operations are expected to be completed over a period of approximately 3 years with primarily waste mining operations carried out during the first two years. In the last year of operation, virtually all of the ore materials to be recovered will be mined and delivered to the stockpile area adjacent to the JEB mill for subsequent feeding into the processing facilities.

Special waste material has been designated as waste rock containing a minimum of 0.03% U_3O_8 and a maximum of 0.10% U_3O_8. Above this level the material is classified as ore. When special waste is encountered in the course of mining, it will be mined selectively from the barren waste material and deposited directly into the existing Sue C open pit for disposal below the water table in order to prevent oxidation.

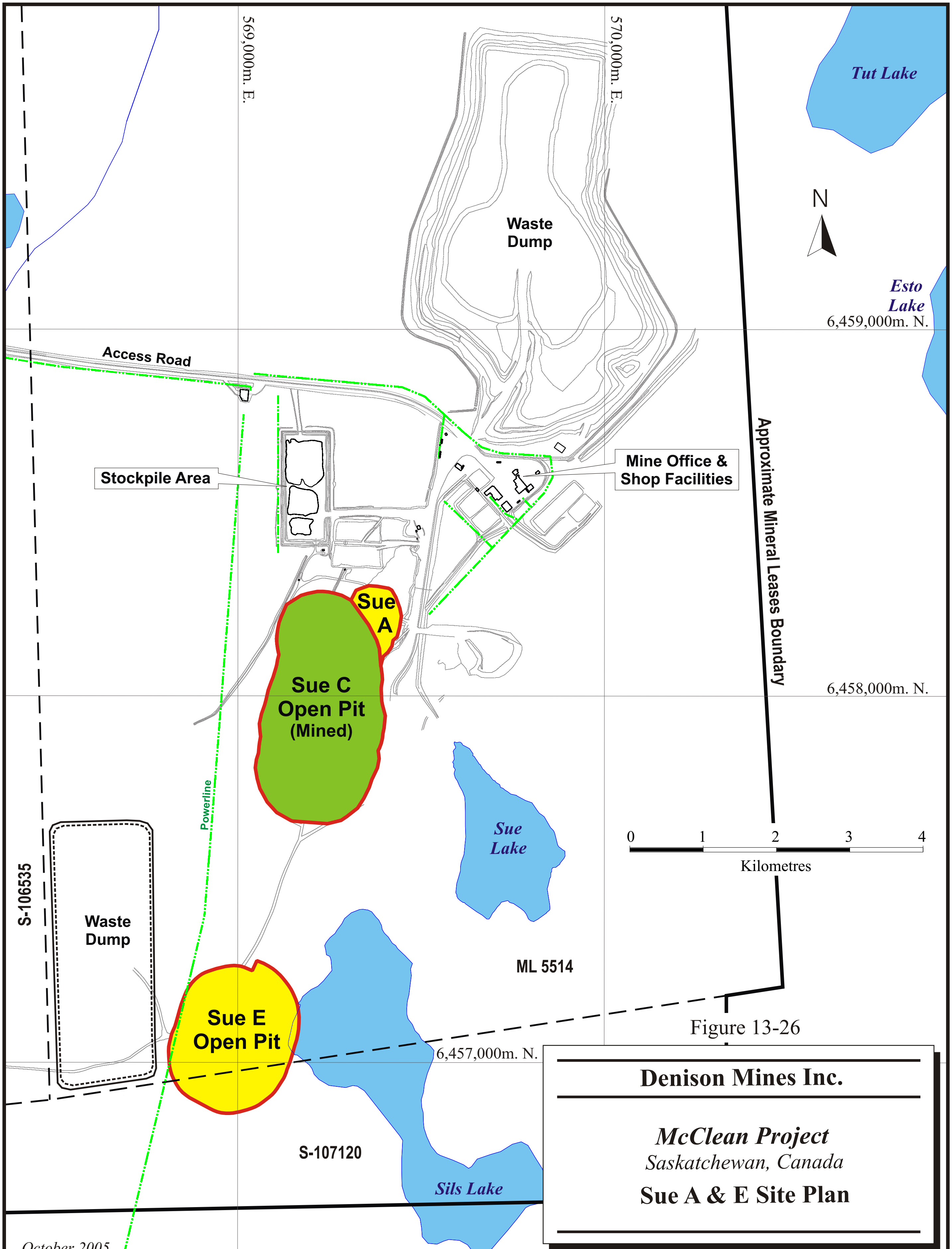


Figure 13-26

Denison Mines Inc.

McClean Project

Saskatchewan, Canada

Sue A & E Site Plan

MINING EQUIPMENT FLEET

The mining operations at Sue E, as in the case of Sue A, will be carried out using the existing equipment fleets at the McClean Lake operating site. There are existing equipment maintenance, office, and site management facilities available at the Sue C mine site. These same facilities will be used to support the mining operations developed at Sue E.

Operating cost estimates for the Sue E operations are based on the most recent operating cost experience at the Sue C operations (2002). The equipment is listed in Table 13-8 below. The mine working schedule is based on two twelve hour shifts per day, 350 operating days per year.

TABLE 13-8 MOBILE EQUIPMENT FLEET
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Description	Quantity
O&K RH120 Shovel	1
Hitachi 1100 Backhoe	1
Cat 777 Haul Trucks	5
Cat D9 Dozer	2
Cat 16G Grader	1
Water Truck	1
Pickup Trucks	6
Fuel Truck	1
Service Truck	1
Subtotal	**19**

CAPITAL COSTS

No substantial capital costs are expected to be incurred for the development of the Sue E deposit as open pit mining operations since the site has already been developed (the Sue C deposit has been mined) and the equipment and support facilities already exist. The Sue E mining plan will be an extension of the previous operations that were carried out at Sue C.

OPERATING COSTS

MINING

Operating costs have been estimated based on the previous actual operating experience at the McClean Lake Sue C mine operations. Mining costs have been estimated based on the actual production and operating experience at the MLJV during the mining of the Sue C open pit. They are forecast to average approximately $4.15 per tonne of material excavated.

POTENTIAL NICKEL AND COBALT RECOVERY

The Sue E deposit is estimated to carry an average nickel content of 0.553%, and represents a potential source of additional economic value for that deposit. RPA understands that the MLJV is currently undertaking a detailed technical review of the nickel and cobalt recovery technology, including process design and an evaluation of product marketing options. Among other issues to be addressed prior to the final production decision, RPA recommends that potential recovery of nickel and cobalt from the Sue E ores be considered as well.

14 MCCLEAN NORTH AND SOUTH

PROPERTY GEOLOGY

Within the McClean project area, the basement geology under the Athabaska sandstones is characterized by a dome and basin setting in which large Archean granitoid domes alternate with Aphebian metasedimentary rocks. The McClean North and South deposits are situated between two Archean basement domes and are aligned along two trends within a linear belt of graphitic gneisses. These east-northeast trending gneisses may represent a splay off the west extension of the Tent Seal fault that forms the north contact of the Collins Bay dome with Aphebian intermediate to felsic gneiss, calc-silicates, and quartzites. The Sue uranium deposits lie on a north-trending segment of the graphitic gneisses at the west contact with the Collins Bay dome, approximately three kilometres to the east. The JEB deposit and Cogema mill facilities are nine kilometres north.

The McClean North and South mineralized trends strike N70°E to EW and are approximately 500 m apart. Uranium deposits occur along the trends as 11 elongated pods straddling the Athabasca sandstone-basement contact (Figure 14-1). The uranium mineralization is hosted in altered sandstone and basement rocks and are surrounded by a clay alteration halo that includes chlorite and hematite. The illite clay alteration extends upwards along fractures in the sandstones for tens of metres where it is capped by silicified sandstones (Kilborn, 1990). In the basement footwall of the mineralization, alteration consists of bleaching, chloritization, argillization and hematization.

The hanging wall sandstones are typically 150 m to 160 m thick and are covered by 1 m to 10 m of glacial overburden. Beneath the sandstones, the regolith varies from 15 m to 45 m thick, but it is invariably destroyed in the zones of uranium mineralization.

Uranium mineralization in the North trend pods occurs over vertical widths of typically 10 m to 20 meters. In cross-section the pods are flat, lenticular to oval shaped bodies with thicknesses from 7 m to 15 m. The higher grade portions of the pods undulate from 13 m above to 12 m below the sandstone–basement contact which is, on average, 160 m below the surface at approximately the 275 m elevation.

DEPOSIT TYPE

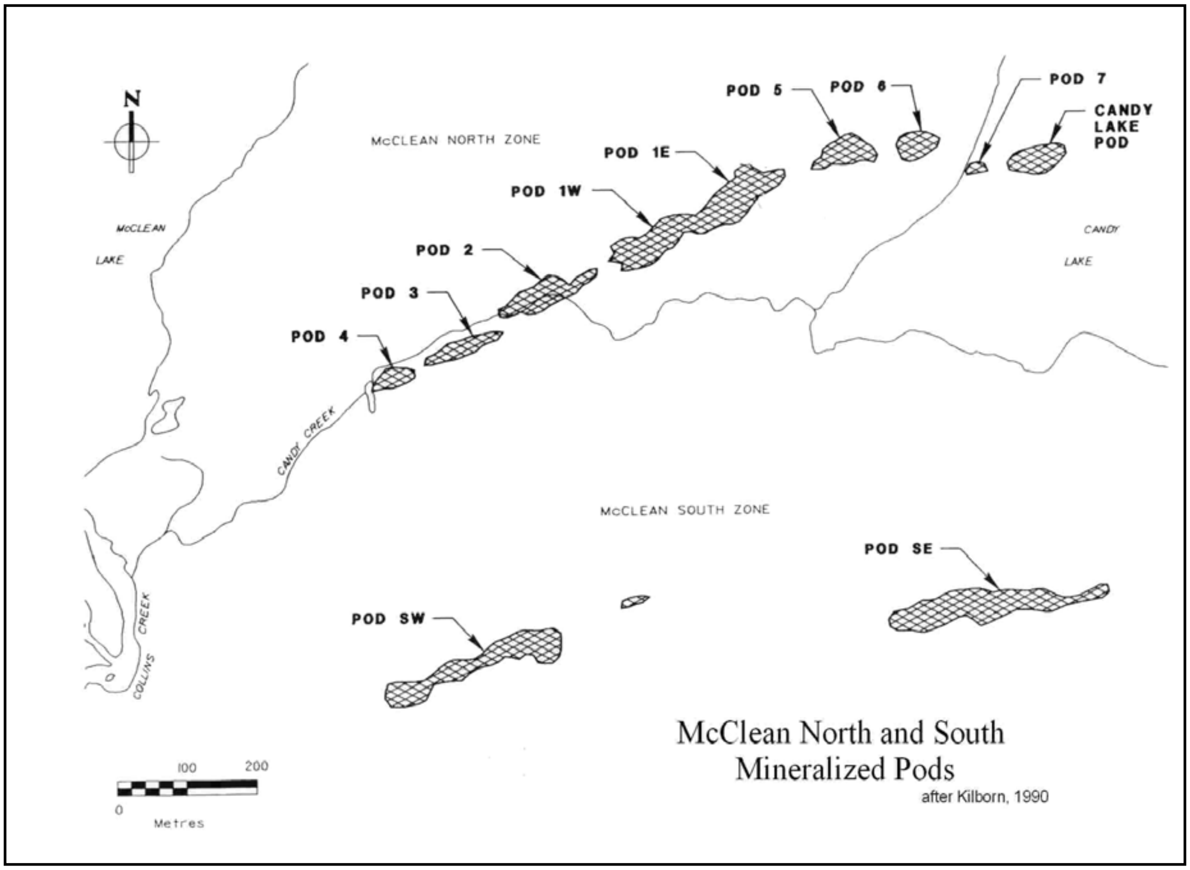
The McClean North deposits are egress type, unconformity-related uranium (nickel-cobalt-arsenic) deposits.

MINERALIZATION

Uranium mineralization is hosted in hematite-altered clay-rich zones containing massive layers of illite. In the McClean North trend, the illite forms a mushroom–shaped envelope tilted to the north. Uranium occurs as fine-grained coffinite veinlets and nodules of pitchblende, and as masses of pitchblende/uraninite. Deposition appears to be controlled by a zone of strong east-west faulting and fracturing that is coincident with the basement graphitic gneisses. Alteration is extensive above and below the mineralization, being largely controlled by the east-west faulting.

Associated with the uranium are highly variable but generally small amounts of nickel arsenides. The McClean North has a U_3O_8 to arsenic and nickel ratio of 0.20 and 0.11 respectively, while the McClean South ratios are 0.57 and 0.31 respectively. Generally, the mineralization located below the unconformity has less arsenic and nickel than that found in the sandstone.

FIGURE 14-1 MCCLEAN NORTH AND MCCLEAN SOUTH MINERALIZED TRENDS



EXPLORATION

Uranium mineralization at McClean North was discovered in January 1979 following extensive airborne electromagnetic surveying and drilling in the McClean Lake area by the “Wolly Joint Venture” partners, CanOxy and Inco Limited. The McClean South trend was discovered in 1980. Minatco Limited entered the joint venture in 1985, and from 1985 to 1990, the company funded airborne and ground geophysics, percussion and reconnaissance diamond drilling on the McClean Lake property, and delineation diamond drilling on the McClean North deposits. Delineation drilling ended in April 30, 1990. By this time, some 81,810 m in 416 holes had been completed on the North and South trends. Minatco accounted for 113 holes totalling 22,123 m, and CanOxy and Inco - for 303 holes totalling 59,687 m (Rickaby et al., 2003).

DRILLING

Delineation diamond drilling at McClean North was primarily NQ (47.6 mm) with most holes penetrating 25 m to 30 m into the basement. In general, holes were collared on 15 m sections and spaced at 7.5 m along the section. Fill-in drilling in high grade areas, e.g. Pod 1E, reduced the drill hole pattern to 7.5 m by 7.5 m and resulted in holes clustered in the higher grade portion of the pods.

Drill hole collars were surveyed for local grid coordinates and elevation. Coordinates were subsequently converted to UTM coordinates by Cogema. Down hole deviation was measured by Sperry-Sun multishot instrumentation in holes drilled later than 1986, i.e. Minatco holes. Prior to 1986, acid dip tests were done, as well as some Tropari azimuth and dip surveys. Deviation of holes was minimal at generally <2º (Kilborn, 1990). Rickaby et al. (2003) notes that a ±2º deviation in an unsurveyed 150 m hole can result in a horizontal variation of up to 10 m.

In the resource pod areas, there are 10 holes that lack down hole surveys. This results in some uncertainty with respect to intercept locations. The northern boundary of Pod 2 has one unsurveyed hole; the southwest area of Pod 1W is uncertain due to three unsurveyed holes; the northeast and southeast corners, as well as the eastern margin of Pod 1E, are uncertain because of four unsurveyed holes.

SAMPLING METHOD AND APPROACH

A Century Geophysical Model 9067 gamma probe was utilized for down hole radiometric readings as a guide for later core sampling. Drill core was transported from the collar site in standard 1.5 m wooden core boxes to an enclosed facility for geotechnical and geologic logging and sampling. RQD (rock quality designation) measurements were taken and then geologic logging recorded lithology, alteration, mineralization, structure, fracturing and density, and core recovery. Uranium mineralization, mineral boundaries and high grade segments were identified in core using the down hole probe gamma logs and by scanning with a handheld scintillometer.

Sample intervals were standardized at 0.5 m, with the length reduced to 0.25 m at high grade mineralization contacts. Shorter intervals, generally in high grade, make up <5% of the assay database. CanOxy sampling was commonly at 0.3 m to 0.31 m (1 ft.) or 27% of the assay database. One metre samples were taken in the hanging wall and footwall of the mineralization, and 0.5 m character samples were taken in various sandstone and basement rock units. Faults and alteration were also character sampled.

Core was split, with one half bagged for chemical assay and the other returned to the core box for storage at the Wolly joint venture exploration camp. Laboratory rejects were returned to Minatco for storage at the camp.

SAMPLE PREPARATION, ANALYSES AND SECURITY

Samples collected from 1979 to1982 were shipped to Inco's J. Roy Gordon Research Laboratory in Sheridan Park, Mississauga. Minatco as operator of the Wolly Joint Venture had all samples (1985^{+}) prepared and analyzed by Barringer Magenta Laboratories (Alberta) Ltd. in Calgary, AB (Barringer). This also included samples collected from Minatco drilling of the Sue deposits.

Barringer's analytical protocol was:

- Dry core
- Crush core to –4 mm (5 mesh).
- Crush sample reduction to 500 g by Jones Riffle splitter.
- Ring pulverize 500 g to -147 µm (100 mesh).
- Reduce/split pulp to 500 mg (0.5 g) for analysis.

Mineralization, fault, and alteration character samples were analyzed for U_3O_8, Ni, Co, As, Cu, V, Mo, and Pb. In unmineralized sandstone character samples, only U_3O_8 was determined. At Barringer, pulps were completely digested by a multi acid nitric-perchloric-hydroflluoric mix, and Ni, Co, V, Mo, and Pb were determined by atomic absorption spectrophotometry (AA). U_3O_8 was analyzed by fluorimetry and arsenic by colorimetry. Results exceeding 5% U_3O_8 were re-analyzed using a 1 g pulp aliquot; the sample was digested as previously described and then analyzed volumetrically for U_3O_8.

No protocol description is available for the analytical work done at Inco's J. Roy Gordon Research Laboratory before 1980. Samples were analyzed by X-Ray Fluorescence (XRF). No As or Ni analyses are available for the 1980 drilling.

Kilborn (1990) reports the following analytical quality assurance/quality control (QA/QC) work:

- Batch control samples were routinely inserted and analyzed by Barringer.
- Minatco periodically submitted duplicate samples for U_3O_8 analysis at Barringer and pulps for check analysis at other laboratories. Kilborn reports that variability in U_3O_8 grade is within 10% for grades U_3O_8 >0.10%.
- The Inco laboratory routinely carried out internal (batch) QAQC. Results are unavailable.
- Inco XRF-analyzed samples (271) from the 1979 and earlier drilling programs were re-analyzed by XRF at XRAL Laboratories in Don Mills, Ontario. Kilborn reports that the results showed variations within the limits of the analytical method sensitivity. The largest variation was found with low grade samples. The check analysis program confirmed reliability of the Inco lab, and all further analyses were done by Inco until Minatco assumed operatorship of the Wolly joint venture.

Denison comments that since 1990 the majority of samples have been assayed at the Saskatchewan Research Council Laboratories ("SRC") in Saskatoon. SRC analyses for uranium using the fluorimetric method using a Jarrel Ash Fluorimeter with a detection limit of 0.2 ppm U. Base metals are analyzed using ICP methods using a Perkin Elmer Optima 3000 DV. SRC includes standards and blanks interspersed amongst samples.

DATA VERIFICATION

Rickaby et al. (2003) compared original analytical reports (Inco) for U_3O_8 with the digital database for 1975 series holes C175 and C183. Hardcopy drill logs and computer-generated sample results for U_3O_8 were compared to the database for 1980 series holes holes 2036 and 2071. Barringer certificates for U_3O_8, Ni, and As were compared to the database entries for 1988 series holes MC36 and MC64. Discrepancies observed between original analytical data and drill logs with respect to the resource digital database were:

- One analysis in hole 2071 was recorded in the drill log as 0.029% U_3O_8 versus 0.027% U_3O_8 in the database. The sample interval is remote from mineralized pods and has no impact on resource estimation.

- Analyses less than the detection limit of 0.01% U_3O_8 are entered in the database as 0.01% U_3O_8, which appears to have been Minatco's convention at that time for other projects as well. Again this has no impact on resource estimation.

- In numerous instances sample intervals actually analyzed are entered in the database as two or more intervals with the same grades. RPA has noted this in other databases, e.g. Sue A. While this impacts on raw analyses statistics, it has little impact once analyses are composited for resource grade interpolation.

RPA obtained three drill hole databases, one used by Denison (Kerr et al. and Rickaby et al. 2003) and two from Cogema. Coordinates for the Dension database are local grid, whereas the Cogema data are converted to UTM. RPA imported all the three databases into Gemcom software to validate entries using software routines and to desurvey the analytical intervals to be used for compositing. The initial database received from Cogema had problems with exporting/importing uranium chemical assays, since values were mixed hole to hole. At RPA's request, Cogema provided its current database in Microsoft Access format. This database has been partially verified by Cogema exploration personnel.

RPA notes that the Cogema drill hole database for McClean North has 498 holes compared to the Kerr et al. database of 363 holes. RPA further notes that the length of holes differs in 238 holes, and for 139 of these, the difference exceeds 3 m. RPA compared hole collar surveys in the two databases and found that there is no simple grid conversion between collar data (multiple drill grid orientations) and that some of the data appear to have been corrected. RPA therefore accepted the current Cogema database for use in developing the current resource estimate. Cogema is currently resurveying drill holes and verifying sampling data in the McClean North area to determine the effectiveness and economics of blind shaft boring mining. There are apparently some survey location and assay data problems that are being addressed (pers. comm. S. Eckert, Cogema), but Dension advises (pers. comm. Wm. Kerr, Denison) that in the pod resource area data was verified by Dension for its pre-feasibility development work and reserve reporting in 2003 (Rickaby et al., 2003).

RPA compared the number of drill holes contained in the databases specifically for the pods. RPA notes that one hole used in the previous Cogema resource estimate for Pod 5 lacks assays in the current Cogema database. Consequently this hole was not used in RPA's estimate.

The database has a number of blank analysis fields that are available in the other databases, but these missing data are not in the area of the pod resources. Otherwise, the header, survey, and assay files for the current Cogema database validated in Gemcom without the need for corrections.

RPA obtained analysis assay certificates for U_3O_8 for five holes (MC23 to MC27) and checked 158 results against database entries. The chemical analyses for U_3O_8 are reported as total ppm or percent. RPA notes that some entries were rounded to 0 although results are reported to one decimal place ppm and that values below detection limit of 0.2 ppm are entered at the detection limit instead of a lower value of half the detection limit (0.1 ppm) or zero as is general industry practice. Of the results referenced to the database, RPA found only one error in hole MC25 where the value 233.4 ppm was entered as 233.0. While this is consistent with rounding in another part of the database as stated above, it is inconsistent within the series of analysis entries for that hole. These errors and practices are minor and affect an analytical level that does not impact on resource estimation.

RPA cross-referenced 252 analyses, from digital assay drill logs for holes MC93 and MC95 to MC99, with the resource database and found no errors.

MINERAL PROCESSING AND METALLURGICAL TESTING

Orteck carried out metallurgical test work on samples from the McClean deposits in 1989 (Ref 11).

Orteck received core samples from four pod areas in the McClean deposit.

Deposit	Number of Individual Core Samples
McClean Pod 1W	136
McClean Pod 1E	89
McClean Pod 2	117
McClean Pod 5	66

Ortech combined portions of these core composites to provide two process test feed composites called McClean 1, McClean2. Assayed grades for these composites are close to the grades calculated from the weights and grades of the individual core samples.

TABLE 14-1 ORTECK METALLURGICAL TESTWORK ON MCCLEAN CORE SAMPLES
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Analysis	McClean 1	McClean 2
U_3O_8	1.44 / 1.5	187 / 2.03
As	0.42 / 0.40	0.32 / 0.25
Ni	0.08 / 0.16	0.07 / 0.12

The testwork established:

- Leaching extraction was between 98 and 99%.

- Leaching time was short, about 6 hours and consumption of oxidizing agent was low.
- A fine grind was needed.
- There were no problems with settlement or solvent extraction tests.

It is expected that this ore will have the same milling characteristics as Sue C ore, the overall recovery will be 98%, and there will be low ferric sulphate consumption.

MINERAL RESOURCE ESTIMATES

PREVIOUS ESTIMATES

A feasibility study carried out in 1990 contemplated mining of the McClean North deposits by underground mining methods (Kilborn, 1990). That feasibility has not been updated to reflect 2003 costs and practices. However, preliminary analysis indicates that other mining methods would be more economically appropriate. RPA has independently estimated resources and reserves based on exploitation of portions of Pod 1, Pod 2 and Pod 5, using blind shaft boring.

Cogema (Demange, 1998) prepared a resource (historic reserves) estimate that utilized 2-D block modelling and ordinary kriging to estimate mean values for thickness and grade-thickness as well as sensitivities to mining selectivity and dilution (Demange, 1998). The estimate is based on 15 m x 7.5 m blocks, 2 m vertical mining width, minimum waste pillar of 2 m, and footwall and hanging wall dilution of 0.5 m. Table 14-2 lists the 1998 estimated resources for a 0.3% U_3O_8 cut-off grade.

TABLE 14-2 COGEMA MCCLEAN NORTH RESOURCE ESTIMATE (1998)
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Pod	Tonnes	U3O8%	U_3O_8 Tonnes	U_3O_8 (lbs x 1000)
Pod 1	130,348	2.50	3,282	7,235
Pod2	41,763	2.49	1,041	2,294
Pod 5	25,234	2.10	536	1,182
Total	192,394	2.53	4,859	10,712

Kerr et al. (2003) estimated resources and reserves for these pods under the assumption of mining by blind shaft boring (Table 14-3). This estimate was based on a 2% U_3O_8 cut-off grade and polygonal weighting of drill hole composites within a mineralization wireframe.

TABLE 14-3 MCCLEAN NORTH RESOURCES (KERR ET AL. 2003)
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Pod	Volume (m3)	Specfic Gravity	Tonnes	Thickness (m)	U_3O_8%	U_3O_8 (lbs x 1,000)
Pod 1E	6,621	2.42	16,022	6.6	10.42	3,680
Pod 2	7,540	2.30	17,342	8.2	4.87	1,861
Pod 5	2,274	2.31	5,253	5.1	5.90	683
TOTAL	**16,435**	**2.35**	**38,617**	**6.6**	**7.31**	**6,224**

Cogema prepared a resource estimate in 2003 that utilized 2-D block modelling, ordinary kriging, and uniform conditioning. Table 14-4 lists the estimated resources.

TABLE 14-4 COGEMA MCCLEAN NORTH RESOURCE ESTIMATE (2003)
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Pod	Tonnes	U_3O_8%	SG	Thickness (m)	U_3O_8 Tonnes	U_3O_8 (lbs x 1000)
Pod 1 East	21,478	6.87	2.20	8.9	1,476	3,253
Pod 1 West	9,180	2.57	2.26	10.9	236	521
Pod2	14,643	3.78	2.25	8.7	553	1,219
Pod 5	4,284	4.33	2.28	8.3	185	279
Total	**49,585**	**4.94**	**2.23**	**9.2**	**2,450**	**5,272**

Cogema is currently re-estimating resources for McClean North, but results are not available as of the date of this report.

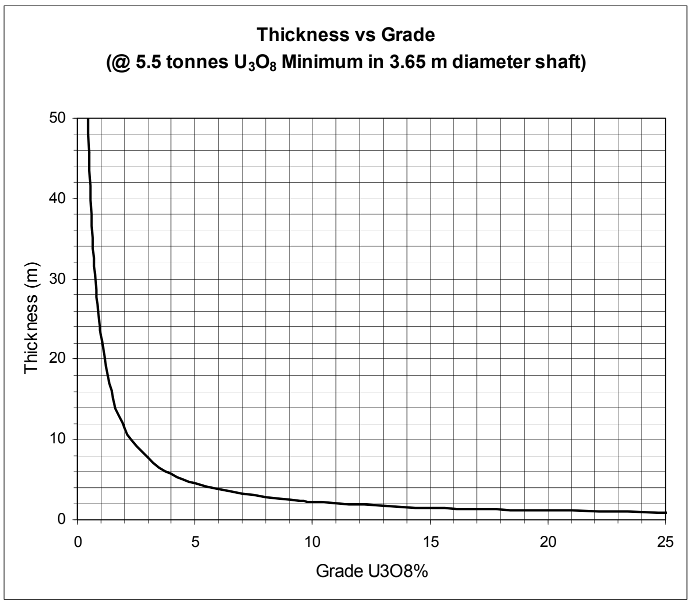
RPA ESTIMATE

The geological model RPA used for the McClean North deposit is consistent with the models previously utilized by Kerr et al. (2003) and Cogema as described above. Within the overall trend of the McClean North mineralization, eight pods with "higher" grade mineralization have been identified by diamond drilling. Portions of Pods 2, 1, and 5, in the sequence from west to east, contain high grade over widths that have potential to support mining by blind shaft boring. RPA has estimated resources for these pods. Pod 1 is subdivided into east and west segments.

DRILLING AND RESOURCE DEFINITION

Resource definition work carried out by Cogema for the McClean Lake North deposit has identified three of the pods shown in Figure 14-1 that may be suitable for exploitation using borehole mining methods.

RPA's definition of the McClean North resources is based on a cut-off U_3O_8 content determined from the current U_3O_8 price and a preliminary estimate of the operating cost per bore hole. In order for a block model cell to be classified as resource, the total recoverable uranium value available must exceed the total expected cost of recovery. The economics of each model cell is a function of its estimated grade and thickness. For the McClean North Blind Shaft Boring plan, RPA has determined that the average operating cost per hole drilled and reamed at 3.65 m diameter is expected to be about \$335,000 based on a consistent depth below surface of 165 meters for these deposits. Using an average U_3O_8 price of \$29.00 Cdn per lb., the minimum cut-off is estimated to be 5.5 tonnes of recoverable U_3O_8 per hole drilled and is equivalent to a grade-thickness of 24 U_3O_8%-m, assuming a mineral zone rock density of 2.3 t/m^3, as illustrated in the graph below.



Minimum thickness=5.5t U_3O_8/(U_3O_8%*(3.65/2)^2*3.14593*2.3t/m^3
…..24 U_3O_8%-m (GT) is area above curve

The RPA resource estimate is based on 97 drill holes and 2, 010 U_3O_8 chemical assays contained in broad areas of mineralization in the pod models that RPA defined by a minimum contour of 0.1% U_3O_8/3 m or a grade-thickness (GT) ≥ 0.3 U_3O_8%-m. Many of the sample intervals were less than the "standard" of 0.5 m and averaged 0.3 m to 0.4 m. The resource areas contained within a GT contour of ≥12 U_3O_8%-m, as an incremental cut-off, are defined by 54 drill holes (36 in Pod 1, 11 in Pod 2, and 7 in Pod 5) which collectively encompass 1,454 individual U_3O_8 chemical assays

Both exploration and delineation drilling utilized mostly vertical holes. Initial exploration drilling tended to be carried out on line intervals of 15 m and 20 m to 30 m step-outs. More detailed drilling on 12.5 m to 15 m sections and 5 m to 10 m step outs has been completed within pods 1, 2 and 5. In the resource areas, holes with higher grade-widths are clustered at a closer spacing. In RPA's opinion, the detailed hole spacing, in the resource areas of material potentially exploitable by blind shaft boring, warrants classification as Indicated Resources.

Pod 1

Pod 1 is delineated by 58 holes and constrained by some 36 holes outside its boundary (Figure 14-2). The pod mineralized area (≥ 0.3 GT contour) is 240 m long by 20 m to 40 m wide with elongation to N65ºE. Two higher grade areas, where GT is ≥12 U_3O_8%-m, have been defined within the pod outline as Pod 1 East and Pod 1 West. These pods are 60 m by 40 m and 65 m by 20 m, respectively, and contain the resources.

Pod 2

Pod 2 is delineated by 24 holes and constrained by some 14 holes outside its boundary (Figure 14-3). The pod mineralized area (≥ 0.3 GT contour) is 135 m long by 20 m wide with elongation to N65ºE. Two higher grade areas, where GT is ≥12 U_3O_8%-m, have been delineated within the pod outline. The eastern area is 45 m by 20 m, and the western area, delineated by two holes, is 30 m by 7 m. These areas contain the resources.

Pod 5

Pod 1 is delineated by 16 holes and constrained by some 16 holes outside its boundary (Figure 14-4). The pod mineralized area (≥ 0.3 GT contour) is 35 m long by 25 m wide with elongation to N80ºE. The higher grade resource area, where GT is ≥12 U_3O_8%-m, has been defined at approximately 40 m by up to 23 m.

RESOURCE ESTIMATION METHODOLOGY

The estimate was carried out by 2-D block modelling with inverse distance cubed (ID^3) interpolation of drill hole composites spanning the vertical thickness of the pod.

Drill holes that delineate the pods were extracted from the database. The potentially economic uranium mineralization was correlated on longitudinal and cross sections and in plan to define the plan two-dimensional boundaries of the pods. Discrete intercepts of mineralization in the footwall below the pod's main mineralization horizon - small satellite or stacked pods - were not correlated or used to estimate additional resources. Uranium chemical assays were composited at a minimum grade of 0.1% U_3O_8 to define the pod thickness. This grade is consistent with resource minimum grades used by Cogema at other deposits. The plan outline of drill hole composites, for which grade-thickness (GT) $\geq$ 0.3 U_3O_8%-m (0.3 GT), was then digitized to delineate the pod. This effectively provides a contour of mineralization grading 0.1% U_3O_8 (2.2 lbs) over 3 m vertically.

Statistics and cumulative frequency%-log probability plots for raw U_3O_8 assays (Figures 14-5 to 14-7) in the pod, including statistics for composites, were carried out to examine grade distributions, the need for grade capping and validation of the modelling (Tables 14-5 to 14-7). The cumulative frequency%-log probability plots show lognormal grade distribution up to sharp inflection points at 50% U_3O_8 to 90% U_3O_8. Depending on the pod, 98.5% to 99.7% of the chemical assays follow this lognormal distribution. Higher grades above the inflection point may represent an outlier population and/or lack of data in this range; however, these high grades are not random outliers since they all occur in a few specific holes in the core of the pod. Consequently grades were not capped in agreement with common practice in the camp.

RPA estimates that the costs of blind shaft boring and processing dictate that recovery of 5.5 tonnes of U_3O_8 per bore hole is breakeven at current uranium prices. This represents a cut-off GT of 24 U_3O_8%-m (24 GT). An additional plan contour within the pod outline, containing the drill hole composites with GTs $\geq$24 GT, was established by RPA at a GT of 12 U_3O_8%-m (12 GT) as a minimum incremental value to delineate the resource area with potential for mining by blind shaft boring.

ID^3 interpolation of grade x SG x thickness, SG x thickness, and thickness was carried out for composites within each pod using a "soft boundary" between the 0.3 GT and 12 GT outlines. ID^3 was selected in order to constrain the influence of composites with very high GT, and angular (10º) declustering was used to decluster the close-spaced hole composites in the high grade areas. This 2-D computer method approximates the manual estimation method of grade-thickness or "metal accumulation" contouring.

2-D search ellipses in plan were based on hole spacing and the overall dimensions of the pods and resource areas within the pods. Ellipse anisotropy was tailored to pod length-width ratio with orientation parallel to the long axis trend of the pod (N65ºE to N80ºE). Ellipses were 18 m x 6 m for Pod 2 and 16 m by 8 m for pods 1 and 5. Block cell dimensions were selected at 3 m x 3 m to approximate the proposed blind shaft boring diameter of 3.65 m. The block model was not rotated since the pods' trends are only 10º to 25º off east-west.

The resource estimate is constrained to resources lying within the 12 GT contour and includes only blocks ≥ 24 GT (Figures 14-8 to 14-10).

FIGURE 14-2 LOCATION OF POD 1 DRILL INTERCEPTS
Denison Mines Ltd. McClean North Underground Uranium Deposit

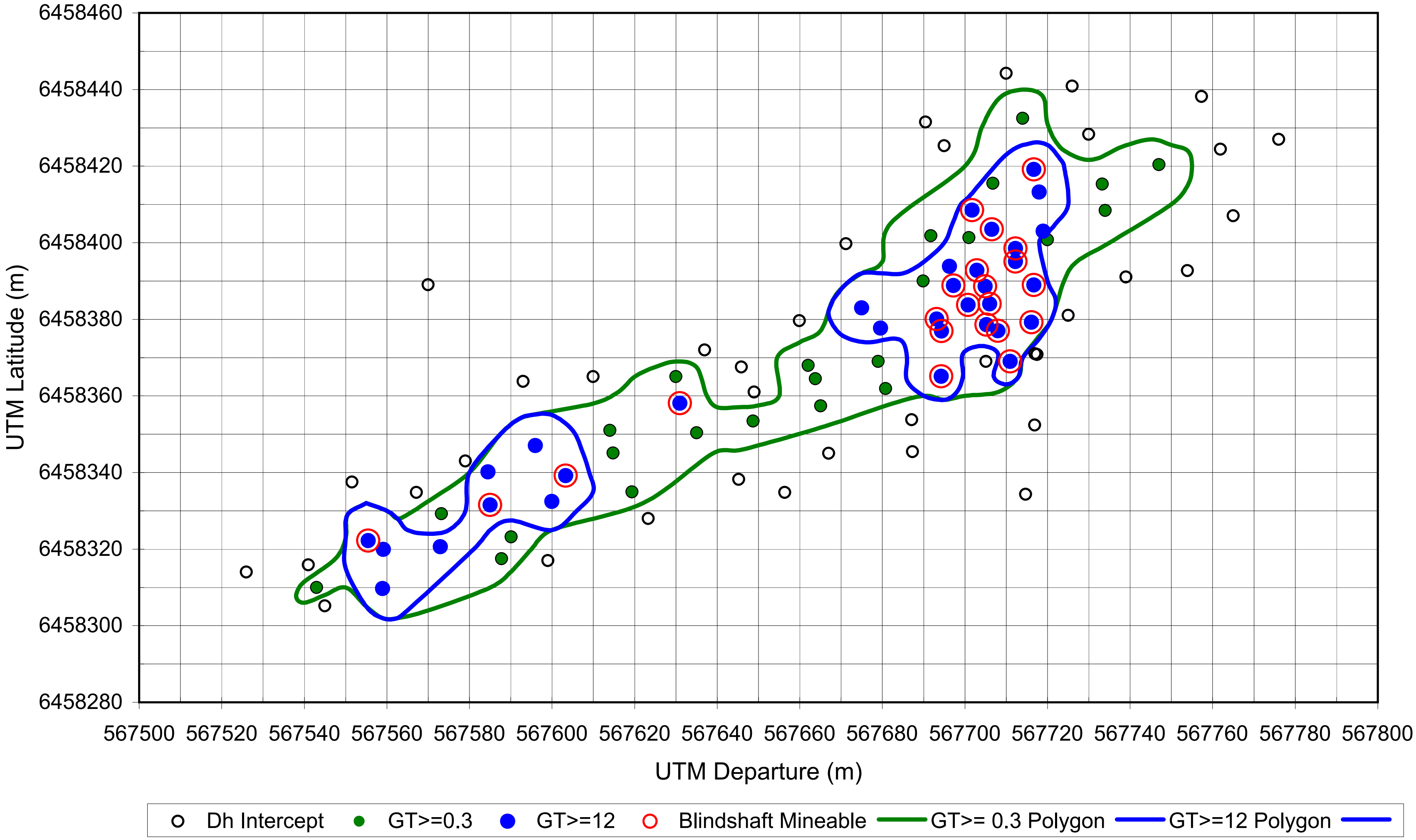


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FIGURE 14-3 LOCATION OF POD 2 DRILL INTERCEPTS
Denison Mines Ltd. McClean Lake North Underground Uranium Deposits

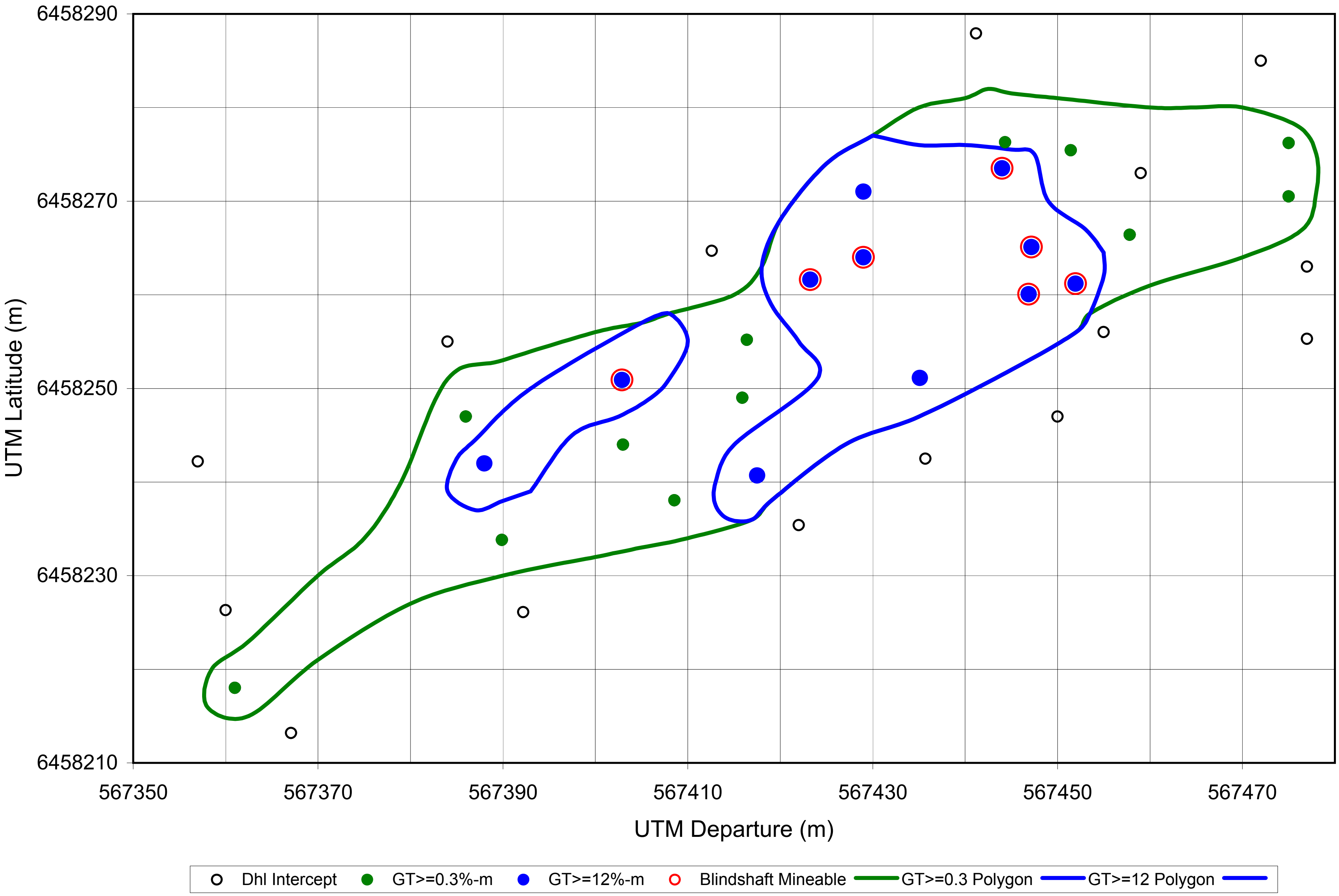


FIGURE 14-4 LOCATION OF POD 5 DRILL INTERCEPTS
Denison Mine Ltd. McClean Lake North Underground Uranium Deposit

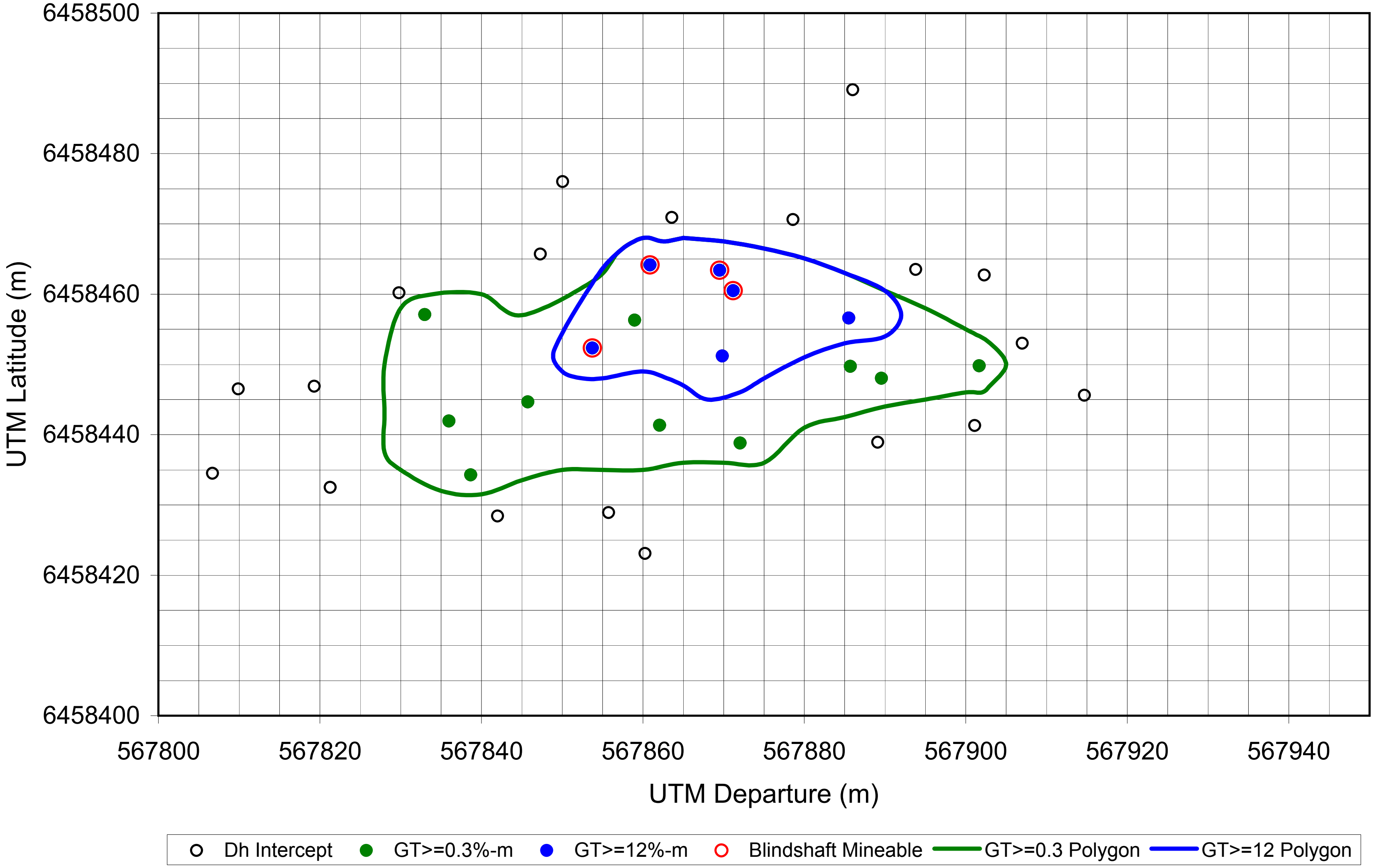


FIGURE 14-5
DENISON MINES LTD. MCCLEAN LAKE NORTH POD 1
Cumlative Frequency% Log Probability Plot of Assays

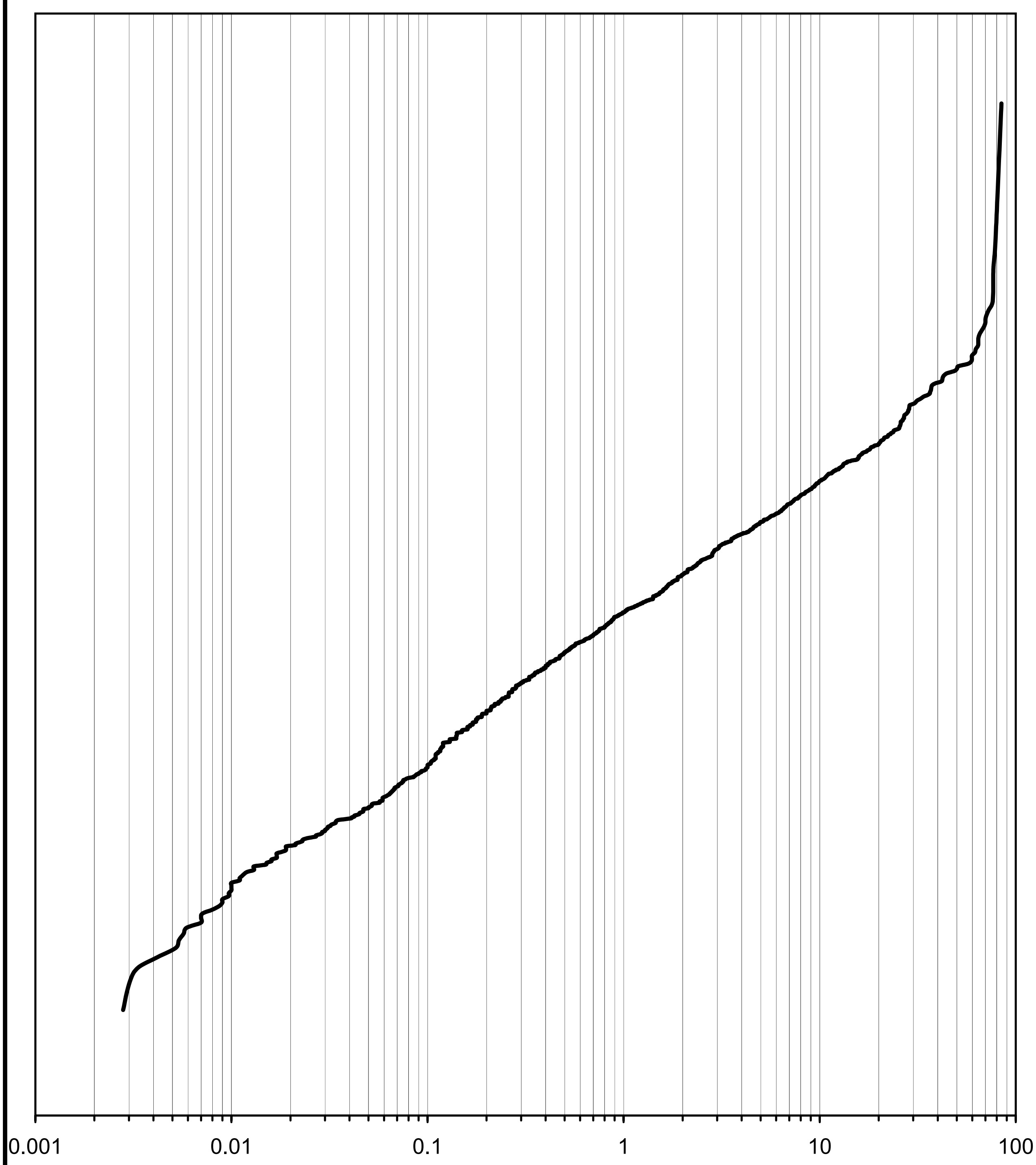


FIGURE 14-6
DENISON MINES LTD. MCCLEAN LAKE NORTH POD 2
Cumlative Frequency% Log Probability Plot of Assays

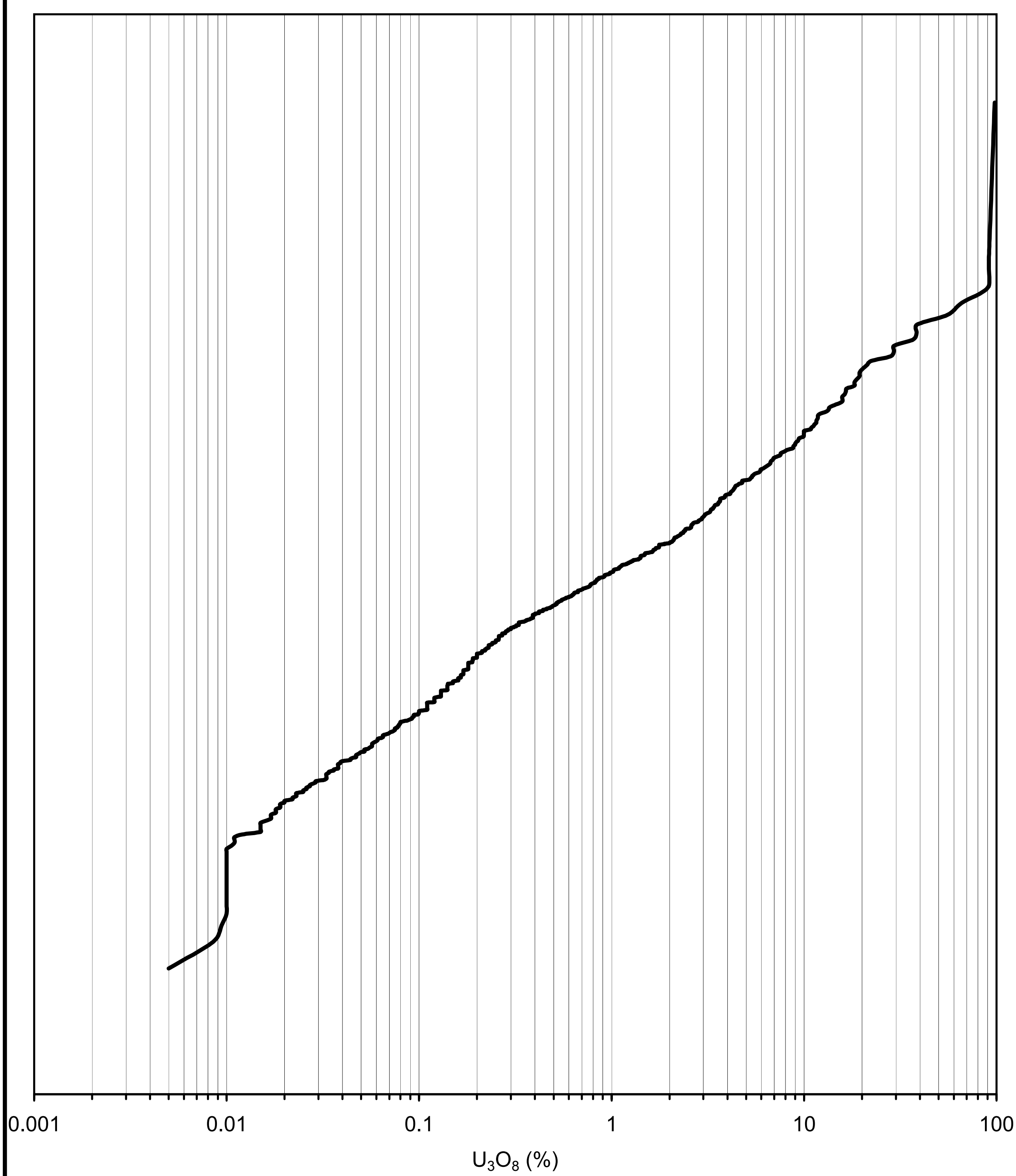


FIGURE 14-7
DENISON MINES LTD. MCCLEAN LAKE NORTH POD 5
Cumlative Frequency% Log Probability Plot of Assays

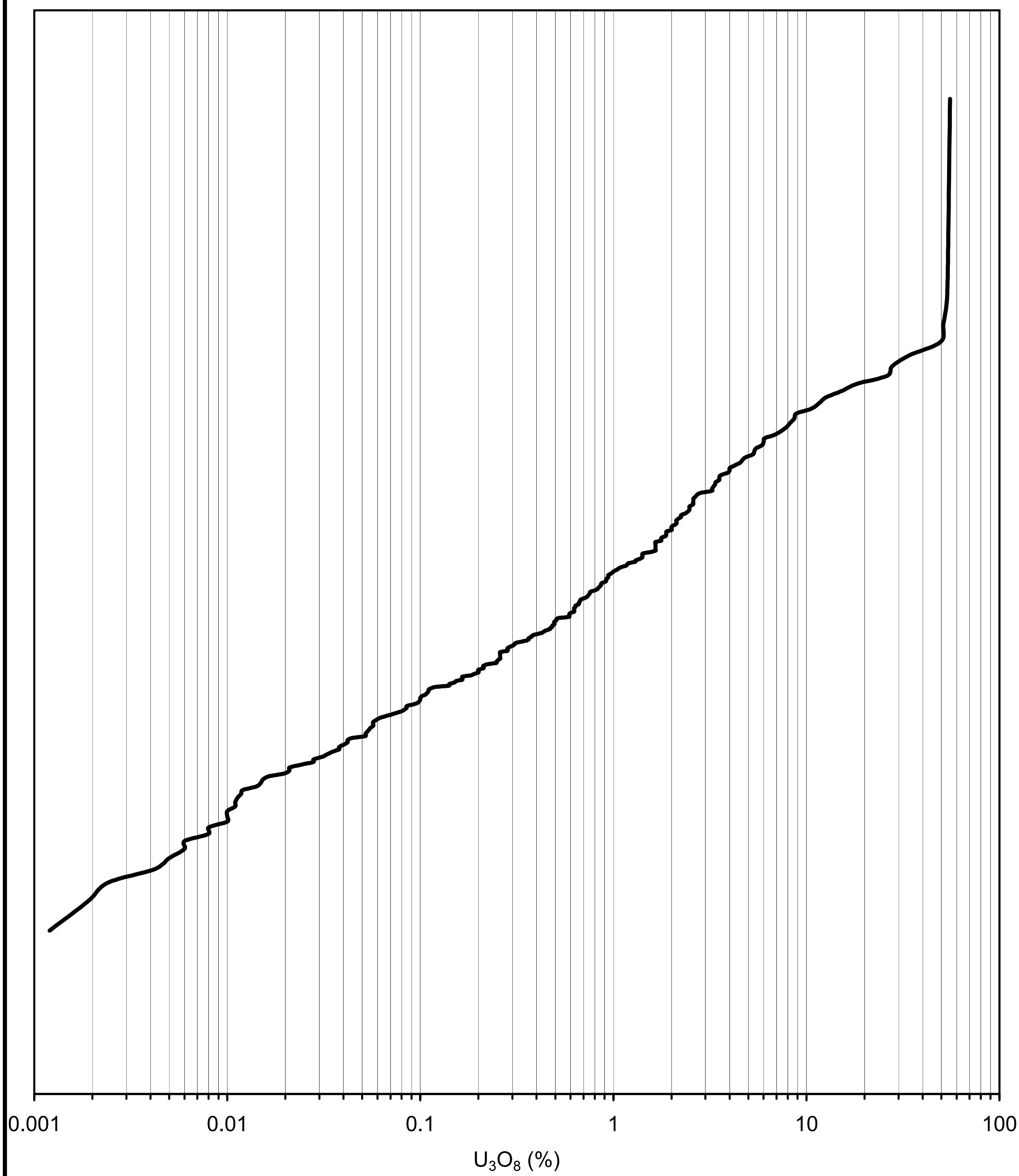


TABLE 14-5 SUMMARY STATISTICS FOR POD 1 RESOURCE ASSAYS AND ZONE INTERCEPT COMPOSITES
Denison Mines Ltd. McClean Lake North Undergound Pod Deposits, Saskatchewan

	Assays in GT>=0.3%-m Polygor				Intercept Composites for GT>=0.3%-m Polygons			
	Length (m)	U3O8%	GT (%-m)	SG	Length (m)	U3O8%	GT (%-m)	SG
Count	1,175	1,175	1,175	1,175	57	57	57	57
Sum	399.3	-	-	-	396.5	-	-	-
Minimum	0.05	0.00	0.00	2.21	0.61	0.14	0.35	2.21
Median	0.31	0.81	0.25	2.23	6.70	2.06	14.53	2.25
Maximum	1.83	84.50	38.40	5.66	16.00	36.25	331	3.00
Arithmetic Mean	0.34	4.79	1.38	2.32	6.96	4.57	33.03	2.29
Length Weighted Mean	-	4.05	-	2.30	-	4.75	-	2.30
SGxLength Weighted Mean	-	5.22	-	-	-	5.15	-	-
Variance	0.02	124	10.53	0.10	13.65	39.02	3,563	0.02
Standard Deviation	0.15	11.13	3.25	0.32	3.70	6.25	59.69	0.13
Coefficient of Variation	0.43	2.33	2.36	0.14	0.53	1.37	1.81	0.05
90th Percentile	0.50	12.50	3.81	2.43	12.31	11.10	51.78	2.40
95th Percentile	0.50	25.83	6.67	2.72	13.48	16.25	119.20	2.51
98th Percentile	0.61	43.93	11.48	3.24	14.00	21.03	271.9	2.61
99th Percentile	0.64	64.43	19.02	4.13	14.90	28.04	307.9	2.79

	Assays in GT>=12%-m Polygor				Intercept Composites for GT>=12%-m Polygons			
Count	860	860	860	860	36	36	36	36
Sum	282.3	-	-	-	279.3	-	-	-
Minimum	0.05	0.00	0.00	2.21	2.14	0.48	1.44	2.22
Median	0.30	1.32	0.39	2.23	8.08	4.25	30.57	2.28
Maximum	1.83	84.50	38.40	5.66	16.00	36.25	331	3.00
Arithmetic Mean	0.33	6.20	1.77	2.35	7.76	6.32	49.48	2.33
Length Weighted Mean	-	5.38	-	2.33	-	6.38	-	2.33
SGxLength Weighted Mean	-	6.94	-	-	-	6.87	-	-
Variance	0	160.1	13.66	0.14	13.95	46.73	4,940	0.02
Standard Deviation	0.16	12.65	3.70	0.37	3.74	6.84	70.28	0.14
Coefficient of Variation	0.47	2.04	2.09	0.16	0.48	1.08	1.42	0.06
90th Percentile	0.50	18.11	4.95	2.54	13.06	11.80	95	2.42
95th Percentile	0.50	28.71	8.25	2.79	13.82	17.47	179	2.53
98th Percentile	0.61	59.66	15.08	3.88	14.62	25.99	302	2.73
99th Percentile	0.76	68.61	19.77	4.38	15.31	31.12	317	2.86

TABLE 14-6 SUMMARY STATISTICS FOR POD 2 RESOURCE ASSAYS AND ZONE INTERCEPT COMPOSITES
Denison Mines Ltd. McClean Lake North Undergound Pod Deposits, Saskatchewan

	Assays in GT>=0.3%-m Polygor				Intercept Composites for GT>=0.3%-m Polygor			
	Length (m)	U3O8%	GT (%-m)	SG	Length (m)	U3O8%	GT (%-m)	SG
Count	635	635	635	635	24	24	24	24
Sum	225.2	-	-	-	225.6	-	-	-
Minimum	0.12	0.01	0.00	2.21	1.22	0.17	0.25	2.21
Median	0.31	0.39	0.14	2.22	9.44	1.36	10.73	2.23
Maximum	1.53	98.00	18.62	7.55	27.44	17.76	184.03	2.54
Arithmetic Mean	0.35	2.89	0.87	2.28	9.40	2.45	27.69	2.25
Length Weighted Mean	-	2.45	-	2.26	-	2.95	-	2.26
SGxLength Weighted Mean	-	3.06	-	-	-	3.06	-	-
Variance	0.02	72.20	4.36	0.12	46.20	13.35	1,827	0.00
Standard Deviation	0.14	8.50	2.09	0.34	6.80	3.65	42.75	0.07
Coefficient of Variation	0.38	2.94	2.40	0.15	0.72	1.49	1.54	0.03
90th Percentile	0.50	6.79	2.37	2.32	16.69	4.98	84.65	2.30
95th Percentile	0.61	11.80	3.63	2.42	19.61	6.57	88.30	2.32
98th Percentile	0.61	20.42	8.90	2.60	24.07	12.73	140	2.44
99th Percentile	0.72	37.86	11.01	3.04	25.75	15.25	162	2.49

	Assays in GT>=12%-m Polygor				Intercept Composites for GT>=12%-m Polygor			
Count	451	451	451	451	11	11	11	11
Sum	155.3	-	-	-	155.0	-	-	-
Minimum	0.12	0.01	0.00	2.21	5.80	1.14	13.92	2.23
Median	0.31	0.55	0.17	2.22	13.00	2.22	28.78	2.25
Maximum	1.53	98.00	18.62	7.55	27.44	17.76	184.03	2.54
Arithmetic Mean	0.34	3.68	1.10	2.30	14.09	4.33	55.10	2.29
Length Weighted Mean	-	3.19	-	2.28	-	3.91	-	2.28
SGxLength Weighted Mean	-	4.04	-	-	-	4.04	-	-
Variance	0.02	98.56	5.86	0.16	35.36	23.08	2,659	0.01
Standard Deviation	0.13	9.93	2.42	0.40	5.95	4.80	51.56	0.09
Coefficient of Variation	0.37	2.70	2.21	0.18	0.42	1.11	0.94	0.04
90th Percentile	0.50	9.48	3.00	2.37	20.11	6.82	88.61	2.33
95th Percentile	0.50	15.80	4.82	2.50	23.78	12.29	136.32	2.43
98th Percentile	0.61	29.30	10.61	2.81	25.97	15.57	164.94	2.49
99th Percentile	0.70	47.00	11.90	3.38	26.71	16.67	174.49	2.52

TABLE 14-7 SUMMARY STATISTICS FOR POD 5 RESOURCE ASSAYS AND ZONE INTERCEPT COMPOSITES
Denison Mines Ltd. McClean Lake North Undergound Pod Deposits, Saskatchewan

	Assays in GT>=0.3%-m Polygor				Intercept Composites for GT>=0.3%-m Polygor			
	Length (m)	U3O8%	GT (%-m)	SG	Length (m)	U3O8%	GT (%-m)	SG
Count	250	250	250	250	16	16	16	16
Sum	112.2	-	-	-	112.2	-	-	-
Minimum	0.00	0.00	0.00	2.21	2.00	0.22	0.65	2.21
Median	0.50	0.59	0.22	2.22	6.45	1.23	7.74	2.23
Maximum	4.45	55.41	26.82	3.69	13.00	10.86	107.46	2.40
Arithmetic Mean	0.45	2.61	0.99	2.26	7.01	2.60	18.41	2.26
Length Weighted Mean	-	2.21	-	2.26	-	2.62	-	2.25
SGxLength Weighted Mean	-	2.70	-	-	-	2.70	-	-
Variance	0.12	56.06	8.25	0.03	12.51	9.95	728	0.00
Standard Deviation	0.34	7.49	2.87	0.18	3.54	3.15	26.99	0.06
Coefficient of Variation	0.76	2.87	2.90	0.08	0.50	1.21	1.47	0.02
90th Percentile	0.50	4.67	1.80	2.29	12.27	6.60	38.90	2.32
95th Percentile	0.50	8.76	4.20	2.36	12.66	9.43	63.53	2.37
98th Percentile	1.11	27.83	8.52	2.76	12.86	10.29	89.89	2.39
99th Percentile	1.53	50.41	13.13	3.48	12.93	10.58	98.68	2.39

	Assays in GT>=12%-m Polygor				Intercept Composites for GT>=12%-m Polygor			
Count	143	143	143	143	7	7	7	7
Sum	54.4	-	-	-	54.4	-	-	-
Minimum	0.05	0.00	0.00	2.21	4.00	1.36	9.39	2.23
Median	0.50	0.86	0.25	2.23	6.90	3.69	25.51	2.27
Maximum	0.60	55.41	26.82	3.69	13.00	10.86	107.46	2.40
Arithmetic Mean	0.38	3.92	1.45	2.29	7.77	4.89	36.32	2.29
Length Weighted Mean	-	3.82	-	2.29	-	4.67	-	2.29
SGxLength Weighted Mean	-	4.76	-	-	-	4.76	-	-
Variance	0.03	92.98	13.72	0.06	12.35	13.29	1,145	0.004
Standard Deviation	0.17	9.64	3.70	0.24	3.51	3.65	33.84	0.06
Coefficient of Variation	0.44	2.46	2.55	0.10	0.45	0.75	0.93	0.03
90th Percentile	0.50	8.18	3.37	2.35	12.40	9.72	72.31	2.38
95th Percentile	0.50	17.98	6.89	2.54	12.70	10.29	89.89	2.39
98th Percentile	0.50	49.79	12.61	3.45	12.88	10.63	100.43	2.39
99th Percentile	0.50	52.65	20.69	3.57	12.94	10.75	103.95	2.40

SPECIFIC GRAVITY AND BULK DENSITY

Where specific gravity (bulk density) is a function of grade, grade must be weighted by SG as well as length during chemical assays compositing. Block SG is used for block volume to block tonnes conversion, and SG is also utilized when interpolating block grades. RPA employed calculated SG weighting for compositing and for block grade interpolation. The SG was calculated using the following grade-based formula:

$$\text{Density} = 1/(0.452-0.00326 \text{ x } (U_3O_8\%))$$

This calculated SG method is the same as the one used in the past by Cogema (Demange, 1998), Kilborn and Kerr et al., 2003. Bulk density is assumed to be equal to specific gravity. RPA applied the above formula to generate SG for raw chemical assays of core and then compared these to SG estimates with those in the earlier Denison-Cogema estimate. The two SG estimates agree well for lower to medium grades, but the formula above has a higher range for higher grades.

FIGURE 14-8 2-D BLOCK MODEL OF POD 1
Denison Mines Ltd. McClean North Underground Uranium Deposits

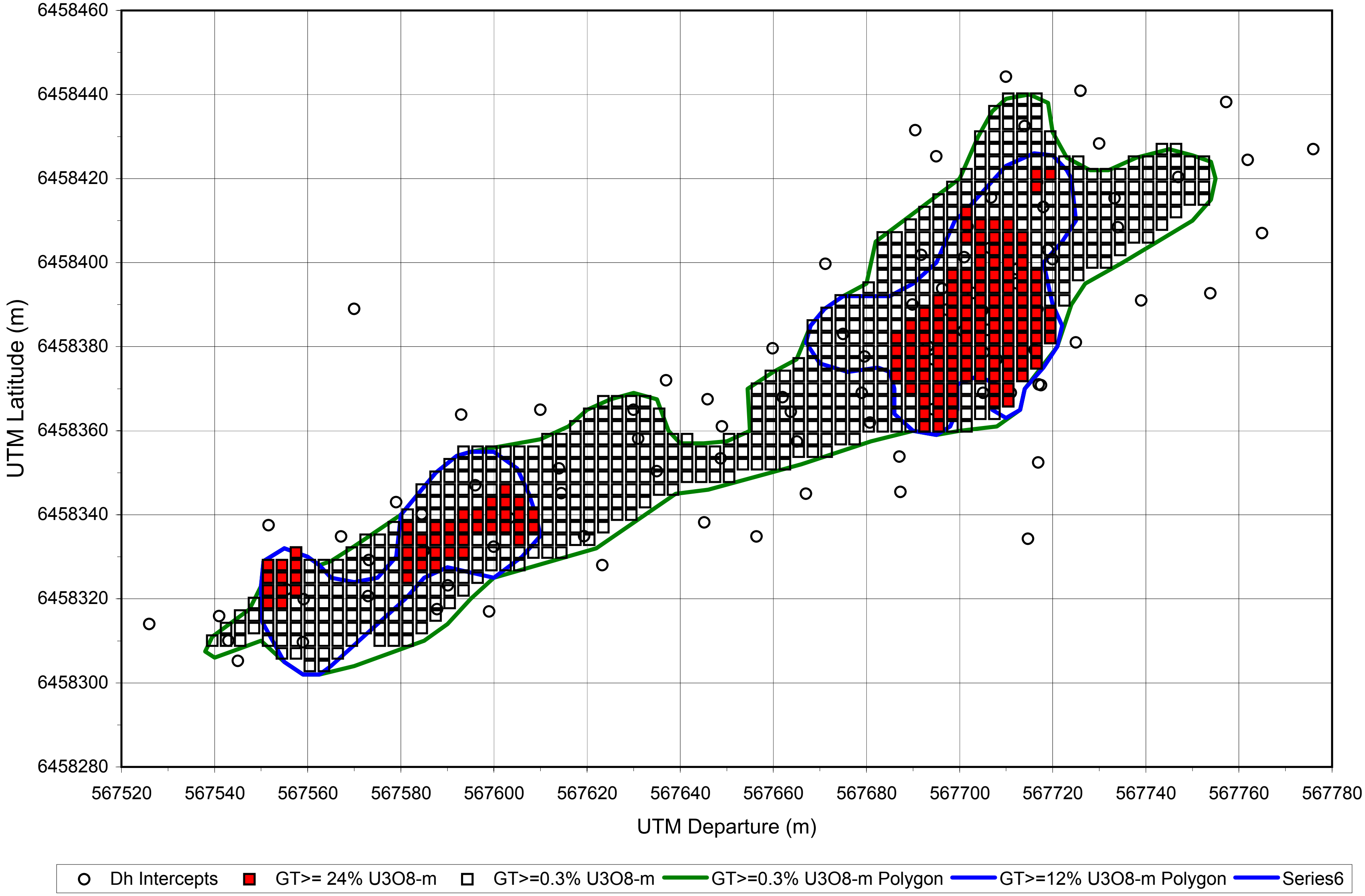


FIGURE 14-9 2-D BLOCK MODEL OF POD 2
Denison Mines Ltd. McClean North Undergroung Uranium Deposits

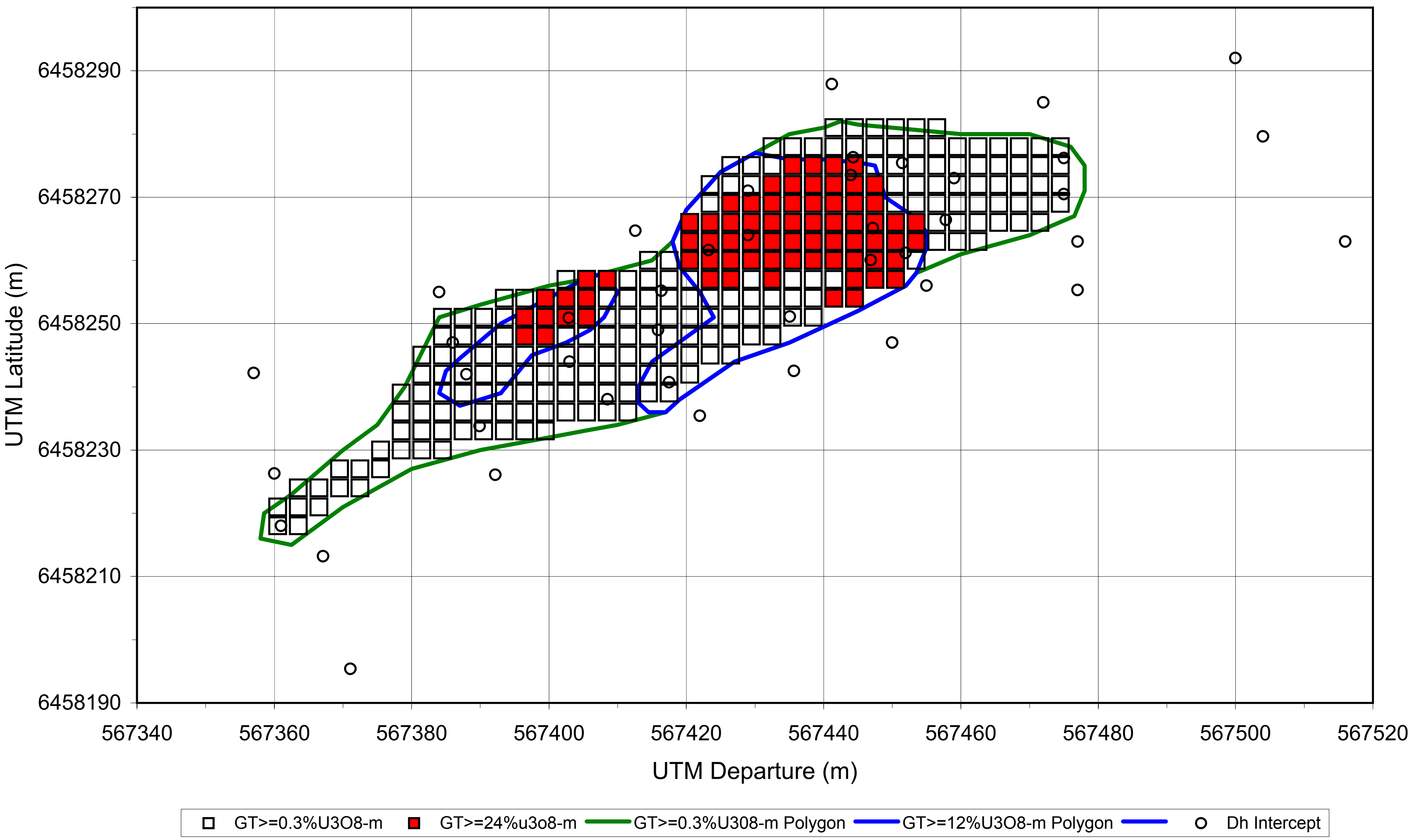
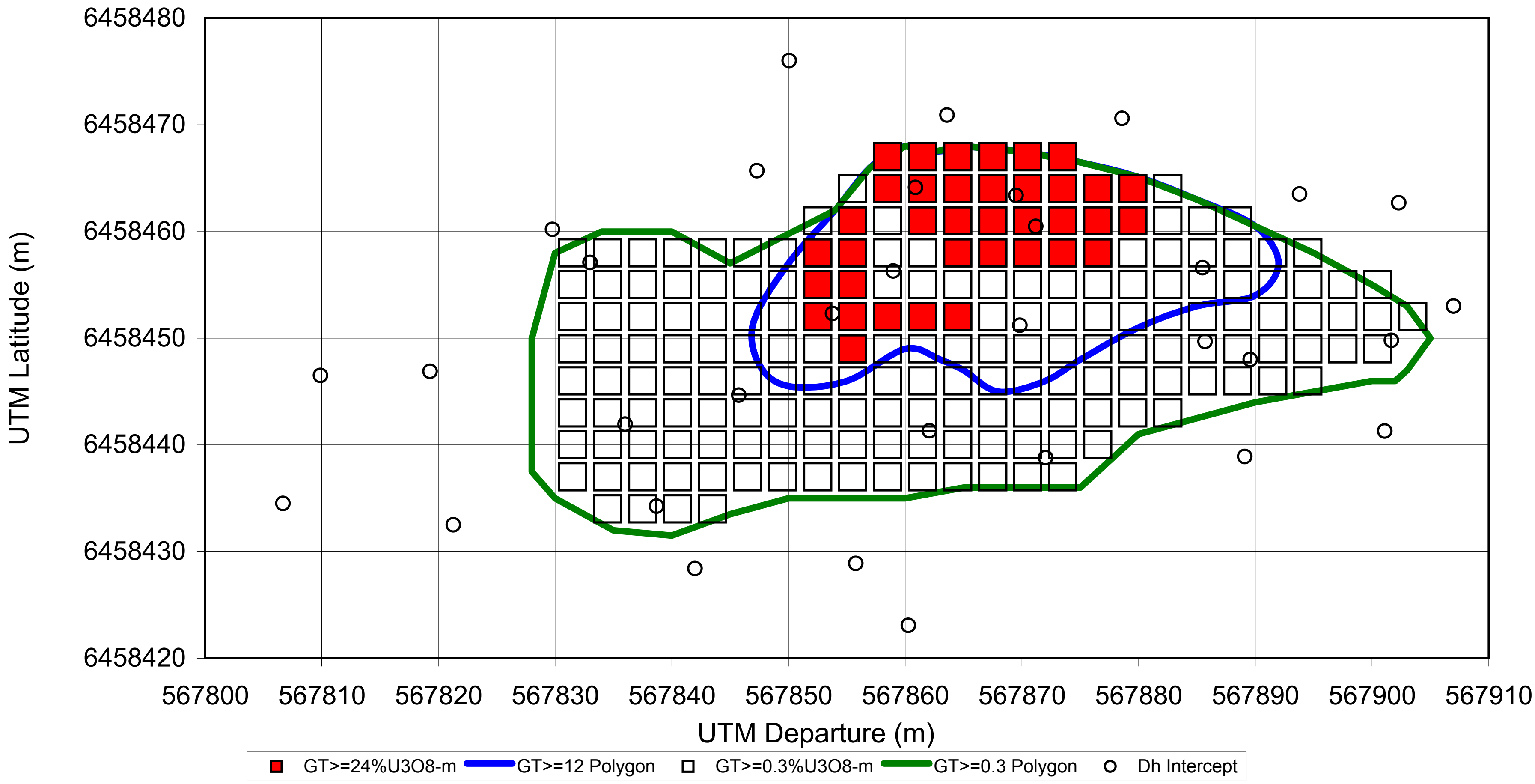


Figure 14-10 2-D BLOCK MODEL OF POD 5
Denison Mines Ltd. McClean North Underground Uranium Deposits



This is consistent with the expected limited grade range for Cogema block SGs and inherent smoothing by the model. The calculated SG is parabolic, and the rate of SG increase elevates with grade; however at 100% U3O8, the calculated SG is only 7.94 versus a theoretical pure pitchblende SG of ±10. This builds in some conservatism in the SG weighting the very high "outlier" chemical assays.

RESOURCE MODEL

RPA estimated undiluted in situ resources for Pod 1 East, Pod 1 West, Pod 2, and Pod 5 as described above (Table 14-8). No estimate of resources was made for the other pods on the north and south McClean trends since these are lower grade and not candidates for blind shaft boring at this time. The 3 m x 3 m block model cells used for each of the pods were classified either as waste or resource on the basis of their total U_3O_8 content. Model cells ≥24 GT were then aggregated and are reported as the blind shaft boring extractable resource for each pod. Minimum vertical thickness of 3 m is implied by the overall pod GT contour of 0.3 U_3O_8%-m, i.e., 0.1% U_3O_8 over three metres. In the resource area at ≥24 GT thickness generally exceeds three metres.

TABLE 14-8 MCCLEAN NORTH INDICATED RESOURCE ESTIMATE (JUNE 2005)

(Based on a 5.5 Tonne U_3O_8/Block Cut-Off for Blind Shaft Boring)

McClean Lake Joint Venture McClean Lake Property, Saskatchewan*

Pod	Tonnes*	U_3O_8%	SG	Thickness (m)	U_3O_8 Tonnes*	U_3O_8 (lbs x 1000)*
Pod 1 East	20,683	9.68	2.39	7.5	2,002	4,414
Pod 1 West	8,287	3.77	2.27	8.8	313	690
Pod2	22,154	3.85	2.28	15.0	852	1,879
Pod 5	5,804	5.81	2.31	7.6	337	743
Total	**56,928**	**6.16**	**2.32**	**10.6**	**3,504**	**7,726**

*Denison Mines Inc. holds 22.5% interest in the MLJV and the above Resource

MODEL VALIDATION

The modeled grade of the resource areas (GT≥12) of Pod 5 was compared to its respective composite average grade. As part of the validation process grade models were developed using both hard and soft boundary interpolation in order to check the grade interpolation process. The composite average grade and thickness, and average SG from statistical analysis, were compared to block model results. Tonnage calculated from plan area, average composite thickness and average SG were compared as well. The block average grade is somewhat lower than the composite average grade due to soft boundary interpolation and the effect of declustering as expected; however, the hard boundary bock model grade is only slightly lower and validates the grade modelling. RPA also used Gemcom software to confirm reasonableness of the estimate by ID^2 interpolation.

MINERAL RESERVE ESTIMATES

MINING

BOREHOLE METHOD EVALUATION HISTORY

Denison and Cogema have evaluated two alternative methods of borehole mining for some of their deposits at the McClean Lake and Midwest Lake properties in northern Saskatchewan. The two alternatives developed are Blind Shaft Boring and Hydraulic Borehole Mining. At present the focus for application of these methods is at the McClean North and Caribou deposits.

OCTOBER 2001 REPORT

In 2001 the Midwest Joint Venture evaluated the Blind Shaft and Hydraulic borehole mining methods in a study reported in a Cogema Resources Inc. – Midwest Project Report dated October 2001. This report presents a set of estimates and designs based on work carried out by Cogema, Golder Associates, Layne Christensen, and Zeni Drilling and develops a comparison of the two options. The report concludes that the Hydraulic Borehole Mining method offers the greatest potential for further improvements. It also notes that the Blind Shaft Boring method could continue to be developed in parallel as a potential alternative. The report credits the Blind Shaft Boring method as being a proven technology to provide better cavity stability characteristics, whereas the Hydraulic

Borehole method is judged to be more flexible in deployment, cheaper in operation, and holds the potential for higher recovery of the resource.

DECEMBER 2003 REPORT

In December 2003 Denison Energy Inc. developed a detailed capital and operating cost estimate for a mining development for the McClean North deposits using the Hydraulic borehole mining method. The analysis is reported in a Denison Energy Inc. report entitled "McClean North Uranium Deposit Report on Reserves based on Pre-Feasibility Study Development Using Hydraulic Borehole Mining Method" and dated November 29, 2003. The Pre-Feasibility Study was developed for Denison Energy Inc. by Layne Christensen Canada Limited and Golder Associates Limited.

In summary, the mining plan presented in Denison's report for the McClean North deposits outlines development of three individual deposits or "pods" at the property. The hydraulic mining method described in the study involves a series of steps including:

- Delineation drilling to define the orebody.
- Drilling of 61 cm diameter access holes from ground surface through the deposit.
- Directional survey of the drill hole.
- Deployment of an expandable reamer in the ore zone to cut a 1.5 meter diameter cavity.
- Deployment of a water jet cutter using an "air shroud" to expand the mining cavity in the ore to a four meter diameter.
- Extraction of the ore materials from the bottom of the hole using "airlift" technology.
- Cavity monitoring of the completed excavation.
- Backfilling of the ore cavity with cemented fill.

In the process of mining the cavities, the drill fluids are to be processed in a surface plant in order to separate and recover the solid ore materials for subsequent loading and hauling to the processing plant.

The total in situ resource of the three pods was stated to be 33,676 tonnes containing a total of 5.4 million pounds of U_3O_8 at an average grade of 7.2% U_3O_8. A mining recovery factor of 85% was applied to these in situ resources along with a dilution factor averaging 30.5%. As a result the estimated blind shaft boring extractable material totalled 37,342 tonnes carrying an average diluted grade of 5.5% U_3O_8 for all the three pods.

DECEMBER 2004 COST STUDY

In December 2004 a Blind Shaft Boring cost analysis was carried out by Golder in order to provide a cost comparison for this method against the 2003 analysis described above. In this study the Blind Shaft Boring estimate was developed based on previous quotations and the estimates provided by Zeni Drilling Company (Zeni), a large diameter shaft boring contractor based in the U.S. The most recent project proposal and cost estimate developed by Zeni was completed in 2001 as part of the Midwest project analysis described above.

The development plan outlined by Zeni is based on a 1.5 meter diameter primary shaft to be bored from surface through the deposits. An expandable reamer head is deployed in the ore zone to cut cavities with a maximum diameter of 3.65 meters. In this analysis the Blind Shaft Boring mining method was estimated to produce ore materials at approximately half the rate projected for the Hydraulic Borehole method in the 2003 study. With the capital and operating costs being similar, the cost per pound of recovered U_3O_8 with Blind Shaft Boring was estimated to be approximately twice as high as that with the Hydraulic Borehole method. On this basis MLJV decided to focus their development efforts on the pilot testing of the Hydraulic Borehole mining method.

ONGOING TESTING AND PILOT PROGRAM

As part of the ongoing pilot testing program, a series of laboratory procedures have been set up to test ore material cutability, and water jet and air shroud performance. Once the laboratory scale tests are completed, a field test of a drilling rig, reamer and jet boring equipment, airlift systems, solid separation and ore recovery systems is planned.

MINERAL RESERVE EVALUATION

Notwithstanding the above-described pilot testing program being undertaken by MLJV, it is RPA's opinion that the technology and equipment for the Hydraulic Borehole mining method are currently under development and that the cost projections for this method cannot be relied upon at this time in making an assessment of the economics of mining the McClean North deposits. While there is a high probability of successful development of the technology and equipment for this method and ultimately it may prove to be of significant economic benefit over the Blind Shaft Boring method, a reliable estimate of its performance and cost factors cannot be made at this time. Given the technical development issues remaining to be resolved, RPA has elected to evaluate the economics of the McClean North deposits on the basis of the Blind Shaft Boring method.

BLIND SHAFT BORING MINING PLAN

Mineralization at McClean North, as elsewhere on the Property, occurs in very discreet pods and structures. Contacts between ore and waste are expected to be near vertical and extremely sharp over distances of less than the width of the mined cavity. Consequently, the economics of development is sensitive to minimizing the proportion of boreholes which occur partially or wholly within waste.

Due to the geometry and limited size of the pods, approximately half of the mined cavities will occur on or in proximity to the periphery of the pods. With the present amount of definition drilling, the ore-waste contact locations can only be estimated with a precision which is greater than the diameter of the mined cavities. Consequently, without additional definition of the contacts, a number of the boreholes will likely fall outside of the actual ore zone and, similarly, a portion of the currently defined ore zones may prove to be below cutoff value. It is anticipated that borehole placement and planning will be carried out based on the results of delineation drilling in combination with the performance of adjacent production bore holes. The capital and operating cost estimates have incorporated an allowance for small-diameter delineation drill holes prior to production mining drilling in order to refine the definition of the limits of the ore materials. It is assumed that 25% of the production holes will require an advance delineation hole to be implemented prior to carrying out production boring. In the reserve estimate a number of boreholes have been assumed to straddle the actual ore-waste contact resulting in a certain amount of dilution of the ore materials with adjacent lower grade material. On this basis it has been estimated that approximately 10% dilution of the primary ore material will occur. This dilution is forecast to be incurred with a U_3O_8 grade of 0%. A mining recovery factor of 65% has been applied to allow for the material lost around the periphery of the reamed cavities. Table 14-9 below summarizes the resource estimate for the three pods.

TABLE 14-9 PROBABLE RESERVE ESTIMATE FOR MCCLEAN NORTH BASED ON BLIND SHAFT BORING

McClean Lake Joint Venture McClean Lake Property, Saskatchewan*

Pod	Tonnes*	Grade U_3O_8%	Contained U_3O_8 (tonnes)*	Contained U_3O_8 (lbs)*
Pod 1**	19,092	8.68	1,657	3,653,716
Pod 2	16,048	3.54	568	1,252,586
Pod 5	3,916	4.85	190	419,108
Total	39,056	6.19	2,416	5,325,410

*Denison Mines Inc. holds 22.5% interest in the MLJV and the above reserve
**Includes Pod1E and Pod 1W

A production schedule has been developed based on an estimate of the time required to drill and ream one blind shaft using the Zeni penetration and production rates. The primary access shaft for each bore hole will be a 1.5 meter diameter, drilled from ground surface through to the bottom of the mineralized pod. The penetration rate for drilling through the approximately 165 meter sandstone interval over the unconformity and the approximately 7 meters of ore material is estimated to average 3 meters per hour. This productivity rate is reduced by 60% to allow for operational delays, including shift change, lunch breaks, operational delays, weather etc. Once the primary bore hole is established, the reaming phase of the cycle begins and the production rate for this operation is estimated to be 5 m^3 per hour. This rate is then reduced by 50% to allow for operational delays. In addition to the deration of the drilling rates, specific delay times have been provided for during the mining cycle including: Initial Setup time – 4hrs; Reamer Setup time 12 hrs; Trip Out at end of reaming 12 hrs; Cavity Survey time 6 hrs; Backfilling time 26.5 hrs. Overall these factors result in an average time requirement of 7.6 days to complete one bore hole.

Table 14-10 outlines a production schedule for the McClean North deposits based on the Blind Shaft Boring mining method. The total recoverable ore for the three pods identified is 39,056 tonnes at an average grade of 6.19% U_3O_8. The production schedule calls for a period of just over three years.

TABLE 14-10 BOREHOLE MINING SCHEDULE – 3.65 METRE CAVITIES
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Pod	Production	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Pod 1	Ore Tonnes	t	8,293	8,293	2,506	-	-	19,092
	Ore Grade	U_3O_8 %	8.68	8.68	8.68	-	-	8.68
	Days		350	350	106	-	-	806
Pod 2	Ore Tonnes	t		-	7,753	8,295	-	16,048
	Ore Grade	U_3O_8 %		3.54	3.54	3.54	-	3.54
	Days		-	-	244	261	-	506
Pod 5	Ore Tonnes	t			-	2,107	1,809	3,916
	Ore Grade	U_3O_8 %			4.85	4.85	4.85	4.85
	Days				-	89	76	165
Total	**Ore Tonnes**	**t**	**8,293**	**8,293**	**10,259**	**10,402**	**1,809**	**39,056**
	Ore Grade	**U_3O_8 %**	**8.68**	**8.68**	**4.80**	**3.81**	**4.85**	**6.19**
	Days		**350**	**350**	**350**	**350**	**76**	**1,476**

The ore drilled out by the reamer bit will be mixed with the drilling fluids and pumped to the surface, where the slurry will be directed to a solids separation plant located adjacent to the drilling site. This plant will include classifying screens, pumps, and conveyors. The recovered solids fraction will be deposited into bins for subsequent loading and hauling by conventional trucks. The liquid portion of the stream will be redirected back to the drill for re-use in the drilling process. The waste materials recovered in the course of drilling through the sandstone will be stockpiled and re-used for backfilling of the completed cavities. The ore materials will be stockpiled and periodically hauled by truck to the JEB processing facility.

CAPITAL COST ESTIMATE

Capital costs to be incurred under the McClean North Blind Shaft Boring project have been estimated based on the Zeni project proposal plus the project design and evaluation work presented by Golder and Denison. Table 14-11 gives a summary of the expected capital costs.

TABLE 14-11 BLIND SHAFT BORING CAPITAL COST ESTIMATE
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Capital Cost Estimate	Year 0	Year 1	Year 2	Year 3	Year 5	Total
Delineation Drilling	$630,000	$0	$0	$0	$0	$630,000
Shaft Boring Rig	$6,025,800	$0	$0	$0	$0	$6,025,800
Mobilization	$4,457,000	$0	$0	$0	$0	$4,457,000
Site Set-Up	$752,000	$0	$0	$0	$0	$752,000
Instrumentation	$3,138,800	$0	$0	$0	$0	$3,138,800
Solids Separation System	$1,609,300	$0	$0	$0	$0	$1,609,300
Demobilization	$0	$0	$0	$0	$1,330,000	$1,330,000
Total Capital Cost	**$18,222,200**	**$0**	**$0**	**$0**	**$1,330,000**	**$19,552,229**

OPERATING COST

Operating costs have been estimated based on the Zeni project proposal plus the project design work by Golder and Denison. Power supply is assumed to be supplied by a moveable diesel generator set rented and operated by project personnel. Drill hole casing and piping materials have been provided for, including collar casing through the near surface overburden materials. The sandstone portions of the shafts are expected to be drilled in stable, competent materials, where no casing requirements are anticipated.

Production drilling consumables have been estimated to include provisions for drill and reamer bit replacement, drilling fluid consumption, fuel and lubes, maintenance parts and freight, and support equipment. The abandonment costs provide for cemented backfill for each borehole shaft.

The labour cost includes provisions for a project management team with Manager, Engineer, Administrator and Drilling Superintendent. The operating crews are expected to comprise ten people from the drilling contractor (Zeni) as well as separation plant operators and support equipment operators. A maintenance crew of electricians, mechanics and welders are also provided for. In total the project team is expected to include 38 people.

Administration and Overhead charges include allocations from the McClean administration departments supporting the project as well as camp costs for housing project personnel and miscellaneous offsite costs.

Haulage costs for loading and trucking of the recovered ore material to the JEB processing facility have been included at a rate of $1.00 per tonne hauled. Processing charges for treating and recovering the U_3O_8 product have been provided for at a rate of $5.00 per pound of contained U_3O_8.

The overall operating costs are projected to average $14.06 per pound of U_3O_8 produced for the life of the Blind Shaft Boring project. Table 14-12 presents a summary of the operating cost estimate.

TABLE 14-12 OPERATING COST ESTIMATE
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Operating Costs	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Power Supply	$ 388	$ 388	$ 357	$ 358	$ 87	$ 1,578
Conductor Casing	$ 1,579	$ 1,579	$ 1,453	$ 1,457	$ 354	$ 6,422
Production Drilling Consumables	$ 3,814	$ 3,814	$ 3,728	$ 3,731	$ 837	$ 15,924
Abandonment	$ 765	$ 765	$ 704	$ 706	$ 172	$ 3,112
Labour Cost	$ 3,301	$ 3,301	$ 3,301	$ 3,311	$ 741	$ 13,955
Administration & Overhead Costs	$ 1,510	$ 1,510	$ 1,490	$ 1,393	$ 339	$ 6,242
Ore Haulage	$ 8	$ 8	$ 10	$ 10	$ 2	$ 38
Ore Processing	$ 7,777	$ 7,777	$ 5,315	$ 4,278	$ 949	$ 26,096
Total Operating Cost	**$ 19,142**	**$ 19,142**	**$ 16,358**	**$ 15,244**	**$ 3,481**	**$ 73,367**
Operating Cost/lb U_3O_8	**$ 12.31**	**$ 12.31**	**$ 15.39**	**$ 17.82**	**$ 18.35**	**$ 14.06**

BLIND SHAFT BORING PROJECT ECONOMICS

Based on the current estimate of long term U_3O_8 prices at US$23.00 per pound the McClean North Blind Shaft Boring project is expected to generate an attractive rate of return overall. Therefore RPA classifies the recoverable resources outlined above as a Mineral Reserve consistent with the CIM definitions. Table 14-13 provides a summary of the estimated cash flow based on the capital and operating cost projections outlined above.

TABLE 14-13 PROJECT CASH FLOW
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

		Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Recovered U3O8 lbs	(000 Lbs)		1,555	1,555	1,063	856	190	5,219
U3O8 Price	US$/lb		$23.20	$23.20	$23.20	$23.20	$23.20	
U3O8 Price	Cdn$/lb		$29.00	$29.00	$29.00	$29.00	$29.00	
Gross Product Revenue	(000 Cdn$)		$45,105	$45,105	$30,827	$24,810	$5,502	$151,348
Total Operating Cost	(000 Cdn$)		$19,142	$19,142	$16,358	$15,243	$3,481	$73,366
Total Capital Cost	(000 Cdn$)	$18,222	$0	$0	$0	$0	$1,330	$19,552
Net Project Cash Flow	(000 Cdn$)	($18,222)	$25,962	$25,962	$14,469	$9,567	$691	$58,430

MINERAL RESERVES

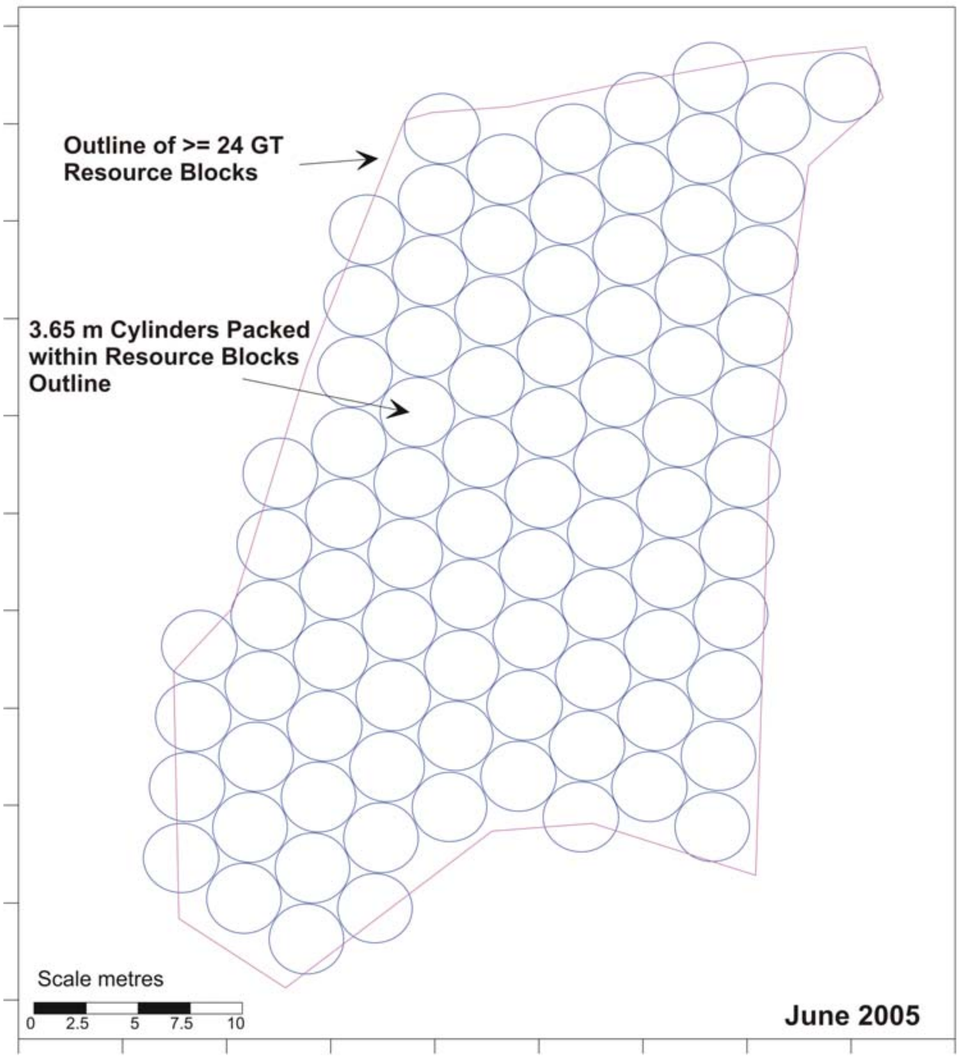
The reserve estimate for McClean North is based on an estimate that approximately 66% of the in-situ resource mineralization in the three pods can be extracted with approximately 159 bore holes with a reamed diameter through ore of 3.65 metres as illustrated in Figure 14-11.

Additional dilution tonnage will originate from material above and below the ore zones, from boreholes which extend beyond the zones, and as a result of incidental re-drilling of backfilled material due to hole deviation. The grade of this dilution has been assumed to be zero. Dilution was modelled by extruding a cylindrical solid out from the axis of each blind shaft hole and accumulating waste tonnage on the margins of the excavated reserve tonnage. The waste and excavated reserve tonnes were averaged, taking into account bulk density, to produce diluted recoverable reserve. Dilution averaged 7% to 8%.

In RPA's opinion, the reserve estimate can be classified as Probable based on the existing drill hole spacing and the geological characteristics of the deposits.

FIGURE 14-11 PLAN VIEW OF CONCEPTUAL PLACEMENT OF BOREHOLES WITHIN A POD

(adapted from Kerr at al, 2003)



15 CARIBOU DEPOSIT

PROPERTY GEOLOGY

The Caribou uranium deposit is located under Caribou Lake, approximately two kilometres NW of the Sue deposits, and is centered at UTM coordinates 569042 mE and 6459440 mN (NAD 83).

The Caribou deposit is approximately 85 m long, 25 m wide, and 10 m to 25 m thick. It has been thickened at its centre by reverse fault repetition along an EW fault, near its intersection with the NE trending faults, and as a result of fault control on upward movement of uranium bearing hydrothermal fluids at this location (Tessier, 2003).

DEPOSIT TYPE

Caribou is an unconformity and sandstone-hosted egress-type deposit, similar to such deposits as Cigar Lake, Cluff D, McArthur River, Collins Bay, and Midwest Lake. Current knowledge suggests that these are formed by the mixing of oxidized sandstone brine with relatively reduced fluids issuing from the basement into the sandstones via faults.

MINERALIZATION

The Caribou mineralization consists primarily of uranium oxides (uraninite and pitchblende) with a suite of nickel-cobalt arsenides (niccolite), sulphides, and sulpharsenides in a clay altered matrix within the sandstones and fault breccias in the basement.

The mineralization is concentrated along the Athabaska sandstone-basement unconformity and at the contact between the “A” and “B” members of the Manitou Falls member of the Athabaska Group, particularly where permeable fanglomerate is present.

Structural controls of the uranium mineralization above the unconformity in the Athabaska rocks consist of NE (030º azimuth) trending, 65º to 70º east-dipping faults that host medium to high-grade pitchblende veins and low-grade replacement mineralization. These faults also control low and medium grade uranium mineralization in the basement adjacent to the unconformity. They form a parallel set separated by 5 m to 10 m and show reverse and normal displacements (<5 m) that cut the mineralization but also host massive to semi massive pitchblende veins. The veins are typically brecciated due to reactivation or protracted movement along the structures; however, replacement and open-space filling textures are also evident.

EXPLORATION

The Caribou deposit was discovered during a winter drilling program in 2002. Eleven holes (2,850 m) were drilled in June and July 2002, and drilling completed in 2003.

DRILLING

The drill hole digital database contains records for 44 NQ (47.6 mm core diameter) diamond drill holes totalling 7,022.3 m. Apart from one hole inclined at -60º to the northwest that lacks assays, all holes are vertical and have intersected the Caribou deposit in an area 88 m by 82 m or approximately 0.72 Ha. Holes were drilled on nominal 12.5 m sections at a spacing of ±5 m (Figures 15-1 to 15-3).

Drilling was carried out in 2002 (1,269 m) and 2003 (5,753.3 m). Radiometric probing was done through the drill string using a Mont Sopris 2500 logger and a natural gamma probe. Mineralized intervals >1000 cps were re-measured with a STD 27 or STD 27 HF high flux probe.

There are some 752 chemical assays over 366.1 m in the database. Of these, 539 are total digestion uranium analyses and 386 partial uranium digestion analyses (non-blank intervals). Assay lengths range from 0.01 m to 4.1 m, and average 0.5 m. The Cogema database has results for U, Ni, As, Co, Cu, Mo, Pb, Ag, V, Zn in ppm, and Fe_2O_3 in %. Uranium assays range from 0 kg/t to 350 kg/t, with the specific density-length weighted average being 14.9 kg/t.

RPA examined and compared databases obtained in Paris and Saskatoon. In terms of sample intervals, these databases are identical, except that several blank intervals in the former are omitted from the latter. RPA has used a corrected version of the Cogema database.

Analyses based on both total (U_t_ppm: 78% of records) and partial (U_p_ppm: 51% of records) digestion are entered in the database as separate columns with the total digestion data predominating. There is incomplete data (95) for U_3O_8 ppm that are check analyses performed by Loring Laboratories in Calgary.

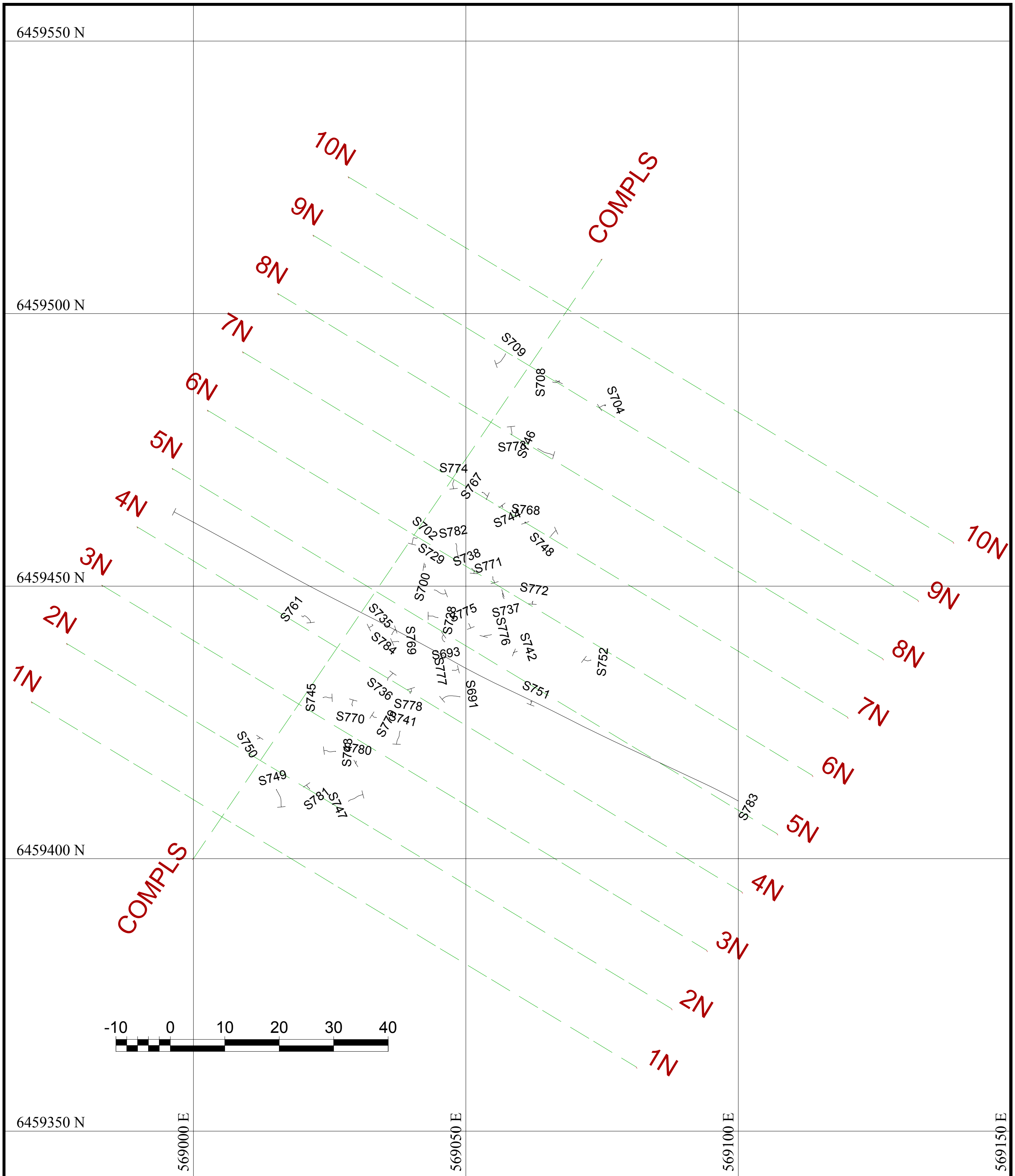


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Figure 15-1
Denison Mines Limited
Saskatchewan Projects
Caribou Urarium Deposit
Drill Hole Location Plan

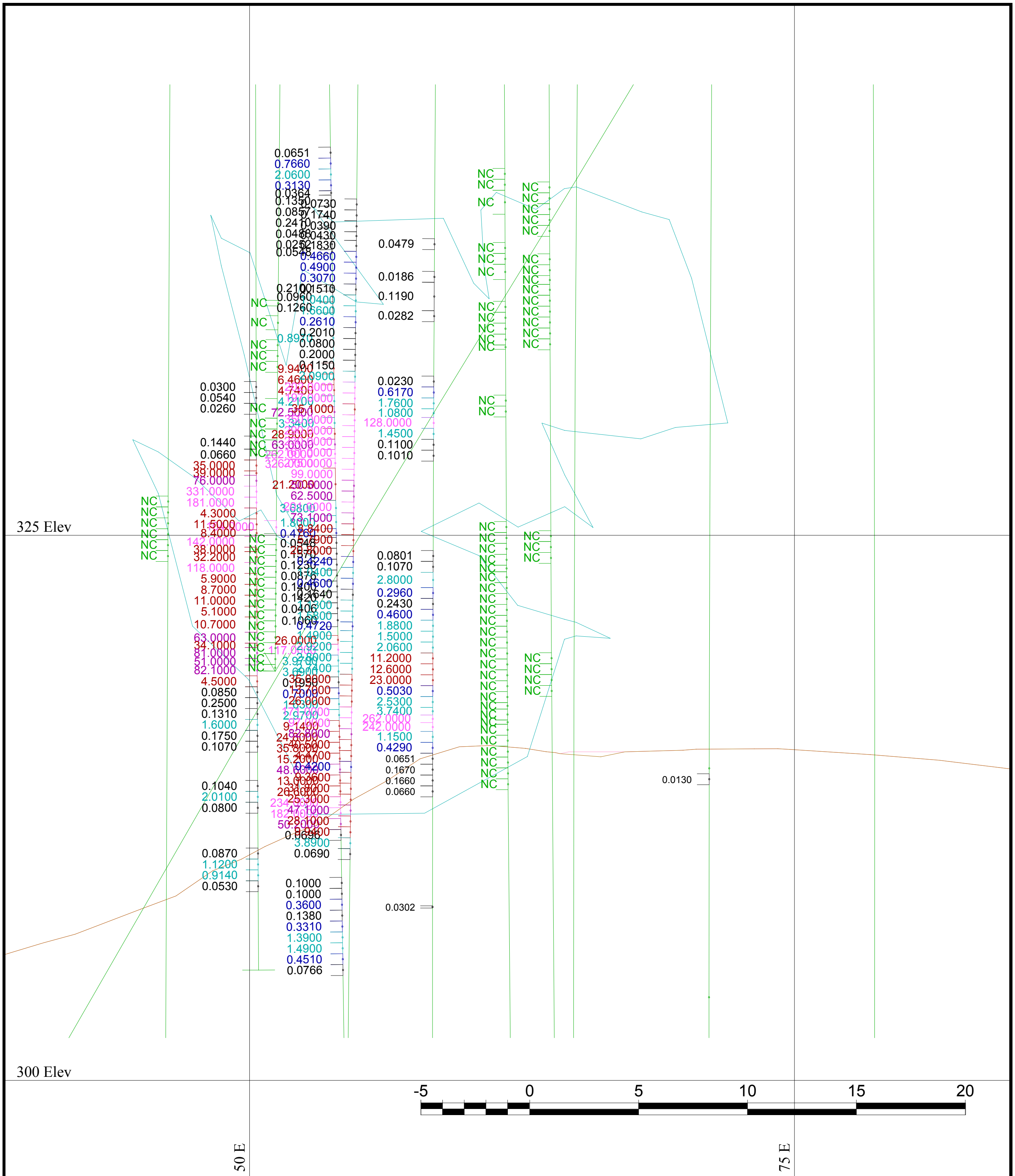

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Figure 15-2
Denison Mines Limited
Saskatchewan Projects
Caribou Urarium Deposit
Cross Section 5N

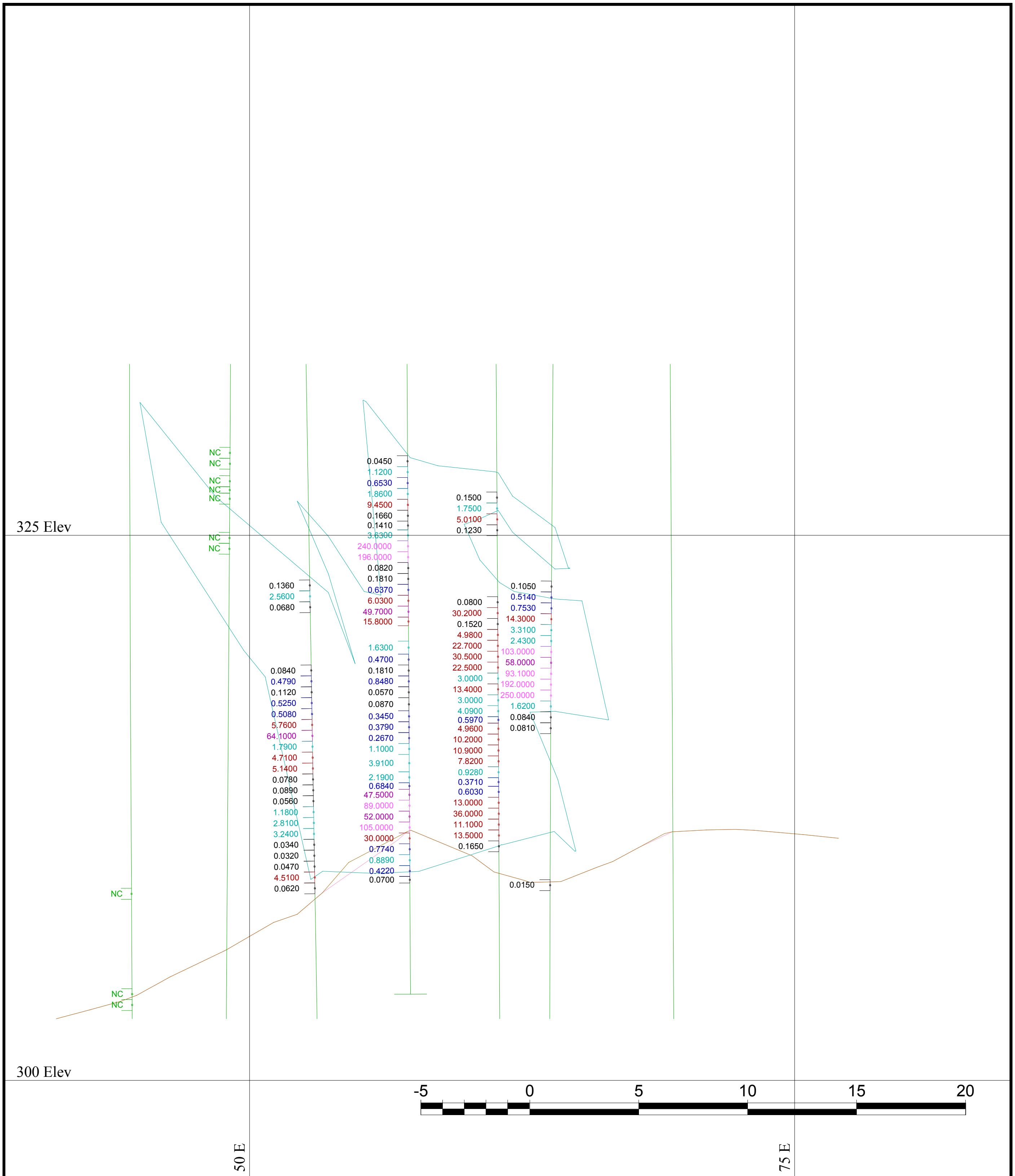

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Figure 15-3

Denison Mines Limited

Saskatchewan Projects

Caribou Urarium Deposit

Cross Section 6N

Original analytical results are recorded in the database; internal laboratory repeats are not averaged with original. The following ten holes in the database have no to only a few assays: S729, S735, S741, S745, S746, S768, S769, S770, S775, and S778.

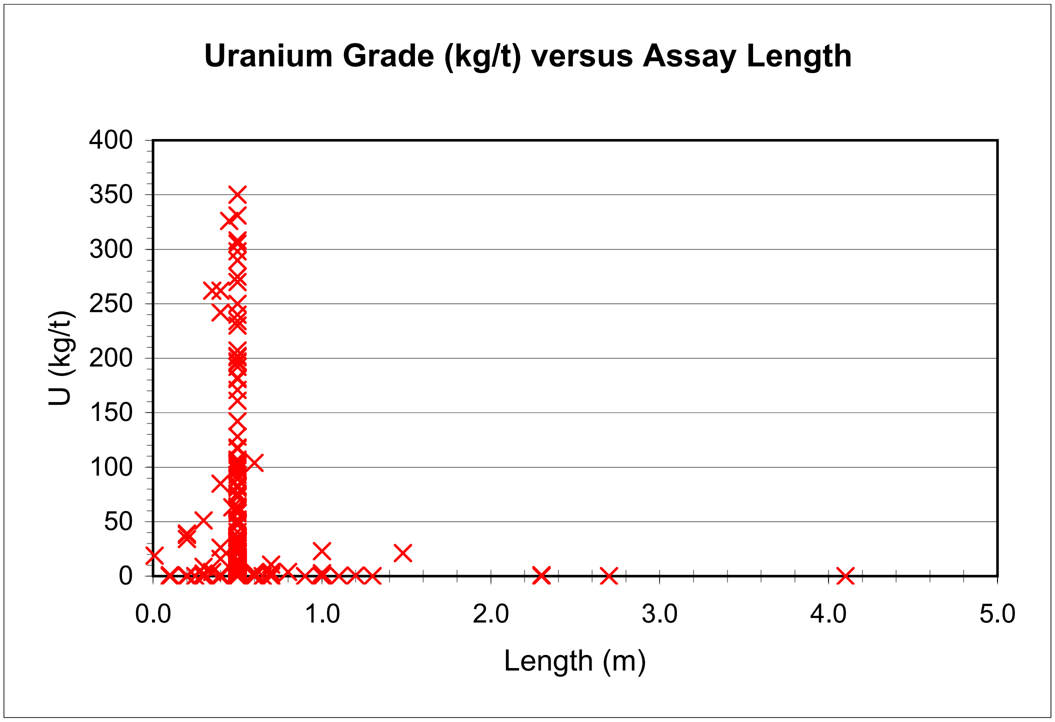
Drill hole collars were initially positioned using exploration grid coordinates but were located by GPS by Cogema personnel after hole completion. Down hole surveys were done by Reflex E-Z Shot single shot digital probe, except for the one inclined hole where Sperry Sun single shot was used. Dip and azimuth readings were taken generally at 15 m to 18 m down hole below the casing, and then at ±30 m intervals. Deviations for the first seven holes were measured at ±50 m intervals. Only minor flattening of dip was recorded for the holes. RPA examined the dips for an excessive rate of inclination that could indicate drilling problems and found that flattening >2º/30 m occurred in several holes but always at the first reading below the casing, where higher deviation is expected.

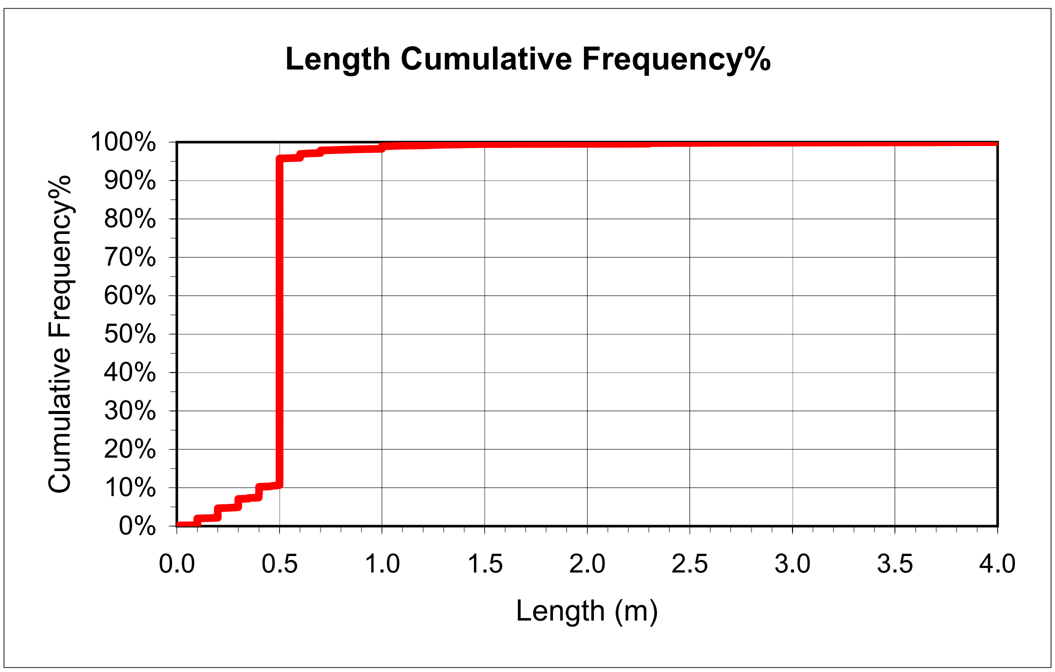
SAMPLING METHOD AND APROACH

Core sampling is the primary sampling method. Down hole radiometric logging was carried out but not used to guide sampling or substitute for chemical analyses. Handheld scintillometer (SPP2, GMT) readings on core were used to guide sampling and to provide for sampling on the basis of radiometric responses (uranium grade) where necessary. Sampling was relatively continuous for mineralized and waste intervals within the mineralized zone, but elsewhere only mineralized intervals were analyzed. Higher grade core was segregated from low grade and analyzed in separate batches.

Sampling was standardized to 0.5 m intervals, and approximately 85% of the core samples assayed met this requirement. Sampling is relatively grade independent, although the intervals longer than 0.5 m (<5%) were in very low grade/waste core (Figure 15-4). The longer intervals are not within the Cogema resource wireframe. As such, raw sample support is relatively regular and could be used for interpolation without compositing.

Figure 15-4 Caribou Deposit Assay Length Statistics
Denison Mines Ltd. Saskatchewan Uranium Properties





SAMPLE PREPARATION, ANALYSES AND SECURITY

Core samples were analyzed by the Saskatchewan Research Council's Geoanalytical Laboratories SRC (SRC). The SRC is an ISO/IEC 17025 accredited mineral laboratory and has been analyzing samples for uranium for more than 30 years.

For 2002 core sample analyses, 11 elements were analyzed by HNO_3/HCl acid digestion and ICP: Cu, Ni, Pb, Zn, Co, Mo, Ag, As, Bi, V, and Th. Uranium was analyzed by tri-acid HF/HNO_3/$HClO_4$ total digestion and fluorescence. Au, Pt, and Pd were analyzed by fire assay/ICP. Elements were reported in ppm, except for Au, Pt, and Pd, which were in ppb. Additional ICP analyses based on tri-acid total digestion were performed for Al_2O_3%, Fe_2O_3%, MgO%, and K_2O%. C% and S% were determined by LECO furnace, with LOI% based on weight loss after heating to 1,000º C.

In 2003 analyses were done for 47 elements by tri-acid total digestion and ICP. Uranium, nickelб and cobalt are reported in ppm; As was not analyzed in this package. A 125 g pulp was digested by gently heating in a mixture of HF/HNO_3/$HClO_4$ acids until dry, and the residue was dissolved in dilute HNO_3 for ICP analysis. Detection limits for As, Co, Niб and U are 0.2 ppm, 0.1 ppm, 0.1 ppm, and 0.5 ppm, respectively.

Element	Detection Limit
As	0.2 ppm
Co	0.1 ppm
Ni	0.1 ppm
U	0.5 ppm

From the review of SRC analysis certificates and the drill hole database, RPA notes that the laboratory sample includes the drill hole number and coding for sandstone or basement. The numbering is in sequence downhole. This sample numbering system is not good practice in terms of security, particularly in cases when a stock market-listed junior mining company holds an interest in the property, since third parties can reference drill holes, geologic units, and approximate depth of sampling from the sample tags.

DATA VERIFICATION

The Cogema resource reports do not describe data verification or sampling and analysis quality assurance/quality control. RPA obtained and reviewed SRC analytical reports for two drill holes from the 2002 campaign and 11 holes (in part) from the 2003 series. These holes were selected because they contained higher grades, whereas the rest of the SRC reports RPA obtained were mostly for lower grade core. Results were compared to the Cogema resource database obtained from Cogema in Paris that had sample numbers (note this database is essentially the same as provided by Cogema Saskatoon). Some 221 records (41% of database) were checked for uranium and no errors were found; spot checks were also done for nickel without errors noted.

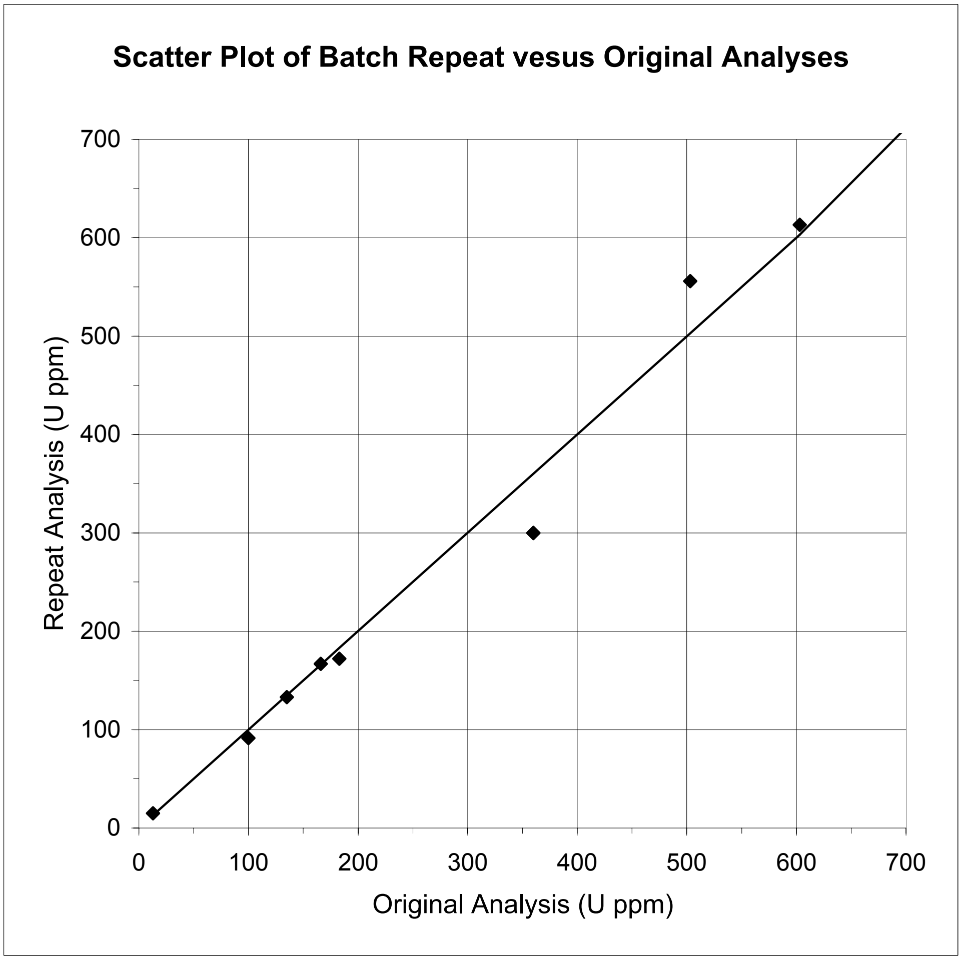
RPA conducted Gemcom data entry checks on the Cogema database and found no errors with the exception of non-analyzed intervals. Ten holes, as previously discussed, have no uranium analyses.

QUALITY ASSURANCE/QUALITY CONTROL

RPA examined SRC internal laboratory repeat analyses (11) for uranium on pulps and compared the data to assess the laboratory precision for uranium (Figure 15-5). The repeats show some minor bias to lower grade; however, the relative precision is ± 10% with most (9 of 11) repeats within 5%. Reference samples (10) in SRC analytical batches examined by RPA varied between 6 ppm and 8 ppm, indicating that the analytical calibration is consistent. No contamination was detected at least for 41% of the analytical data base examined.

RPA compared the Loring Laboratories' U_3O_8 check analyses in the database to the SRC total uranium analyses after conversion of the ppm units to % uranium (Figure 15-6). Checks were performed for samples generally containing >0.5% U_3O_8. While correlation is very good, the check analyses are biased high for >10% U. Relative precision is good with 93 of the 95 samples within 10%, 86 (75%) of the samples being within the ±5% envelope.

Figure 15-5 Pulp Analytical Precision
Denison Mines Ltd. Caribou Deposit, Saskatchewan



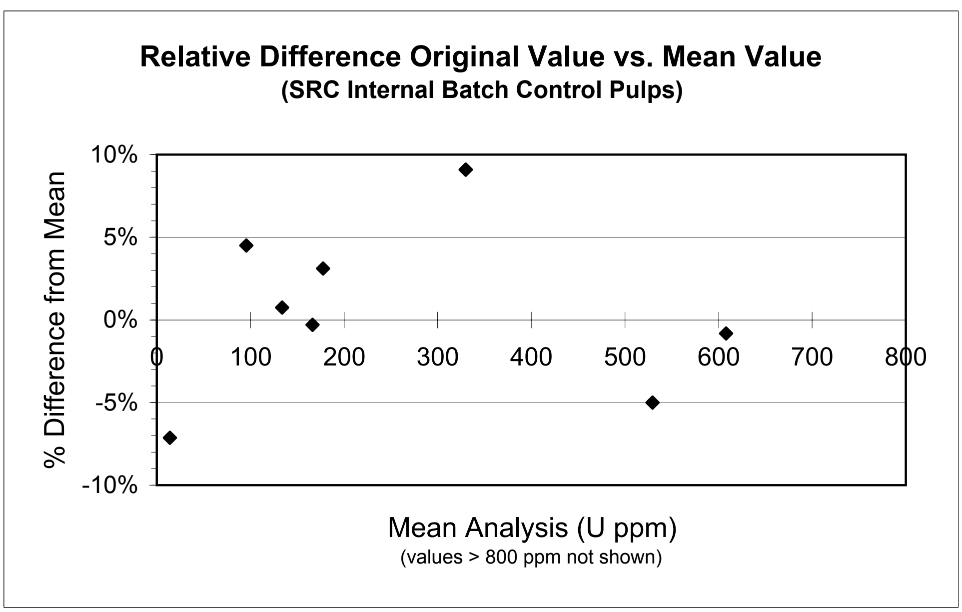
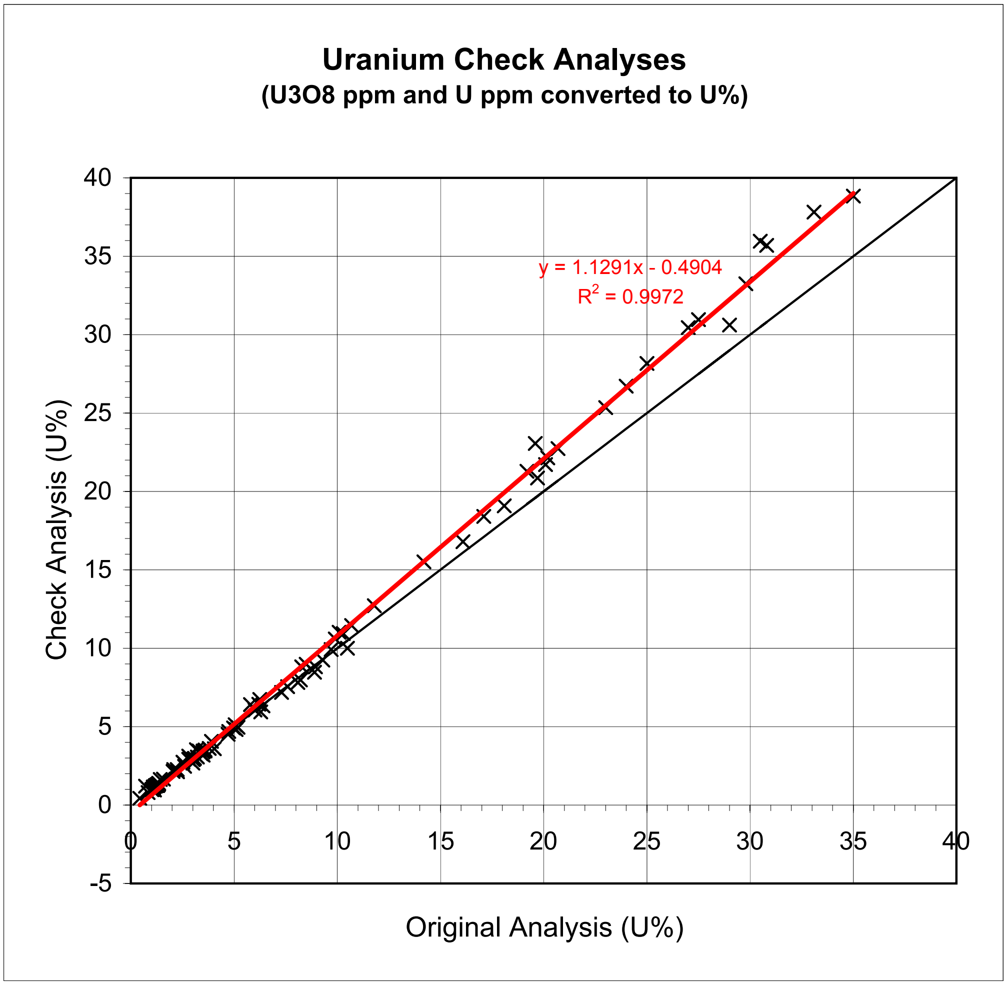
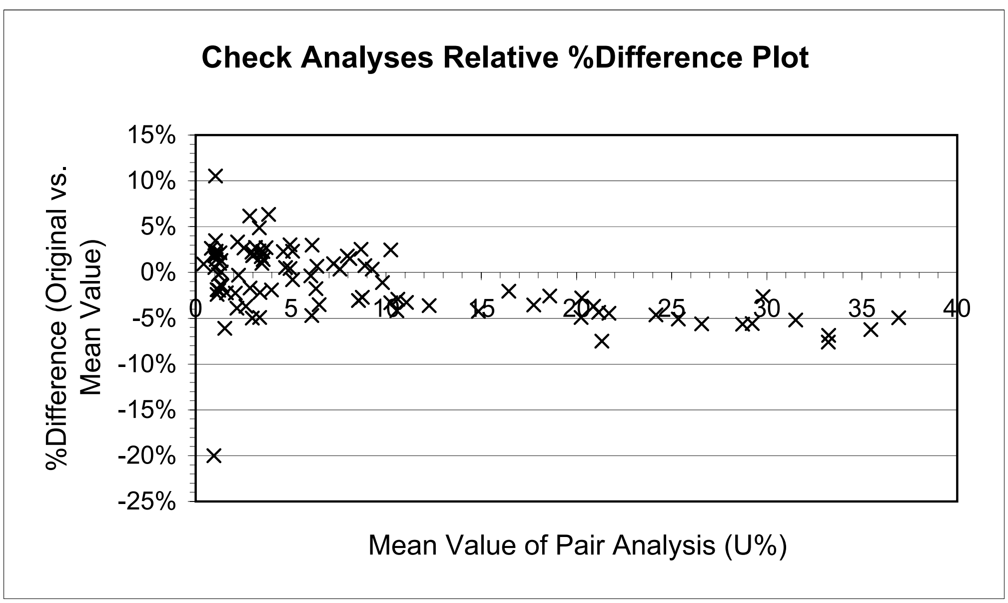


Figure 15-6 Precision on Uranium Check Analyses
Denison Mines Ltd. Caribou Deposit, Saskatchewan





Cogema checked a limited number (±8) of high grade Caribou samples by neutron activation (NA) at SRC's Slowpoke Reactor. All NA samples were lower than the SRC chemical analyses by up to 1.6% in contrast to Loring reporting higher. As a routine check on laboratory results, Cogema compared the chemical analyses with equivalent uranium (eU) generated from bore hole probe logs.

From this limited QA/QC data available, there is no reason to doubt the SRC laboratory analytical precision and accuracy, and RPA considers that the analyses for uranium in the Caribou drill hole database are reasonable for resource estimation.

MINERAL RESOURCE ESTIMATE

Cogema estimated the Caribou deposit mineral resources based on 3D computer block modeling utilizing in-house Sermine software and conventional ordinary kriging interpolation. The estimate relies entirely on drill hole data. Grade, rock, and density (SG) models were constructed and block interpolations carried out for SG x grade (uranium kilograms per tonne, or U kg/t) and SG. Although significant nickel (>1.5% on average) and cobalt has been analyzed from core samples and recorded in the drill hole database, Cogema has not estimated resources for these metals. Table 15-1 presents the Cogema resource estimate for various cut-off grades of U kg/t and equivalent U_3O_8%.

TABLE 15-1 CARIBOU RESOURCE ESTIMATE
McClean Lake Joint Venture McClean Lake Property, Saskatchewan*

Cut-Off Grade		Indicated Resource			
U (kg/t)	U_3O_8%**	Tonnes	U_3O_8%**	U_3O_8 lbs**	U Metal (t)
0.85	**0.10**	**47,763**	**2.62**	**2,758,816**	**1,063**
3.00	0.35	39,482	3.13	2,724,415	1,049
5.00	0.59	33,945	3.57	2,671,615	1,027
10.00	1.18	24,734	4.58	2,497,409	960
15.00	1.77	19,349	5.45	2,324,796	894
50.00	5.90	5,431	11.11	1,330,221	512

*Denison Mines Inc. holds 22.5% interest in the MLJV and the above Resource
**Restated by RPA for reference to U units used in estimate

MINERALIZATION WIREFRAME

Cogema (Comte and Demange, 2004) constructed the uranium mineralization envelope wireframe guided by the interpretation and cross-sections from Tessier (2003). Cogema does not report a cut-off grade criterion for the outline, and the latter appears to encompass all mineralization drill intercepts.

The wireframe is irregular, cross-section to section and level to level, with steep southeast dipping apophyses extending from the main body locally, embayments in the wireframe and linear detached portions. Given the pitchblende and nickel arsenide depositional controls exerted by steep faults, the subhorizontal unconformity, and permeable sandstones, and fault displacement of the deposit itself, the complexity of mineralization outline and difficulty in establishing continuity of outline section to section are not unexpected and are more or less typical of these deposits, particularly where drilling is by mostly vertical holes. However, this complexity impacts on grade interpolation; samples separated by modeled embayments of waste, and in semi-isolated extensions, cannot be distinguished in the interpolation search, and grade is carried over resulting in a global averaging and poor grade resolution locally.

BULK DENSITY

Bulk density information for grade weighting and volume conversion to tonnage has been derived from 324 specific gravity (SG) tests on core samples from 11 holes and applied as a calculated SG. Cogema employs various SG test methods, depending on quality of the core (competent and intact, broken, porous, etc.), from simple core weight/volume based on diameter and length measurement to water displacement and immersion tests.

Calculated SG was used for resource estimation. It was determined by linear regression of variables U, Co+Ni, Cu, Pb, Al_2O_3, gangue and U alone, on measured SGs, and various combinations of variables. This produced several formulas. The formulas derived are:

$SG=1/(0.36938-(0.00068xU^{o}/oo))$

$SG=1/(0.38535-(0.00058x(Ni\ ppm+Co\ ppm))-(0.00041xU^{o}/oo)-(0.0007xAl_2O_3\ ppm))$

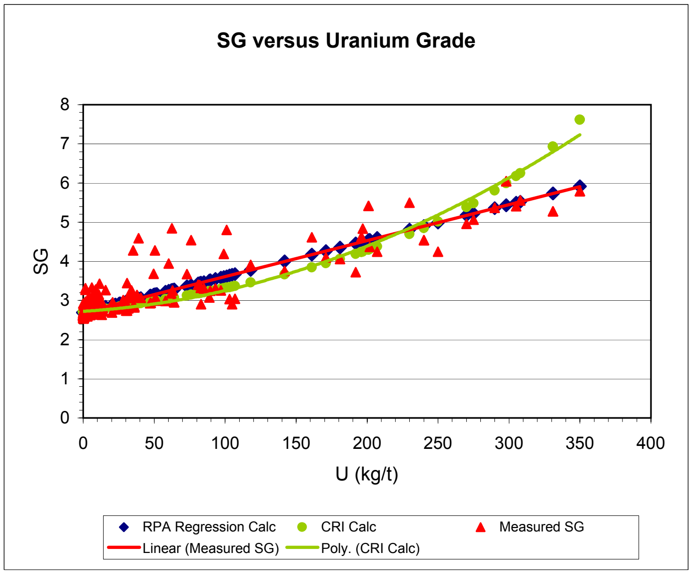
Cogema does not state which formula was used for calculated SG in its report; however, it appears that the formula applied depended on the availability of analyses in the database record by record. The U alone formula produces a very small range in SG and a small effect on weighted average, i.e. length weighted assays are 14.5 kg/t versus 14.9 kg/t for density and length weighting.

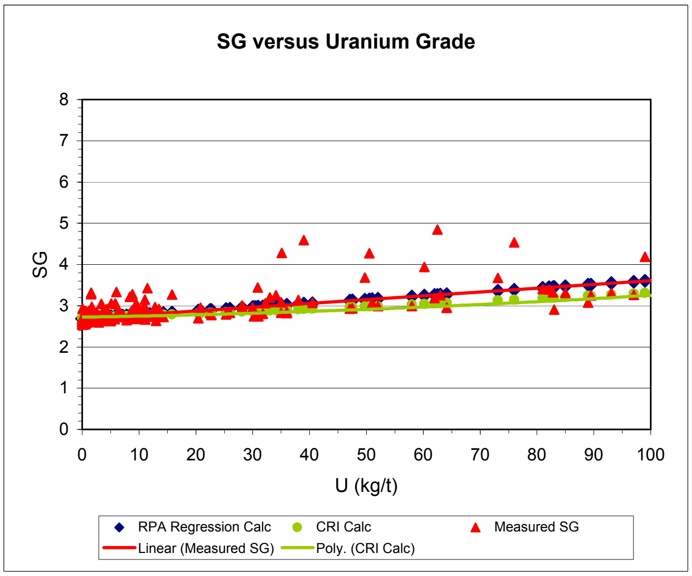
RPA used original uranium grade and SG test sample data provided by Cogema to construct a simple SG versus uranium (kg/t or $^{o}/oo$) linear regression formula (Figure 15-7).

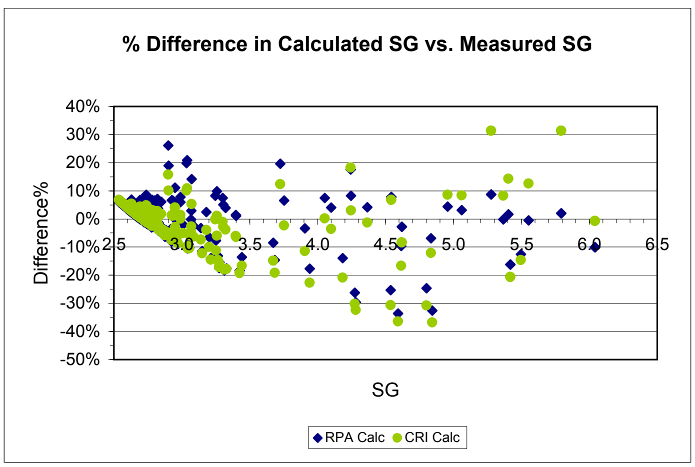
$SG=(0.0092*U^{o}/oo)+2.6901$

The RPA derived formula (U only) was compared to Cogema for the test results. Both produce scatter with respect to actual measurements and thereby introduce some uncertainty when SG weighting composites and converting block volumes to tonnes, particularly in the grade range of 30 U kg/t to 80 U kg/t, where SG tends to be consistently higher than calculated.

Figure 15-7 SG versus Uranium Grade
Denison Mines Ltd. Caribou Uranium Deposit, Saskatchewan







COMPOSITES

Cogema prepared composites of 1 m length, which is equivalent to block height. RPA was not provided with the composites and generated these independently within the Cogema wireframe for the purpose of auditing the Cogema estimate. RPA density (SG) and length weighted the assay grades to prepare the composites.

VARIOGRAPHY

Cogema reports that variography on raw composites was not interpretable. RPA confirmed that variography for 1 m composites and raw assays is not interpretable except for downhole/vertically. RPA was able to derive reasonable profiles on 0.85 U kg/t indicator data for horizontal strike and horizontal cross strike directions. Cogema performed variography by Gaussian transform (Z scores) on UxSG and SG (calculated) and then back transformed the results. Cogema utilized a UxSG nested spherical global model at 30º Az. with 0.05 nugget, 10 m x 5 m x 3 m range at 0.13 sill and 20 m x 8 m x 5 m at 0.81 sill. Cogema developed a SG single spherical global model with the parameters: 30º Az.; 0.05 nugget, 10.5 m x 5.2 m x 3.5 m range at 0.95 sill.

BLOCK MODEL

Cogema created block models of density, rock types and UxSG. The Cogema block model origin is X=568,800.927; Y=6,459,430.0; Z=290 (NAD 83 UTM coordinate system) and consists of X=23,964 blocks, Y=1,024 blocks and Z=256 blocks. The model, and blocks, are rotated to the NE; 45º Az. This departs somewhat from the deposit trend of 30º Az. Block dimensions are 4 m x 4 m x 1m vertical and are consistent with the Caribou drill hole spacing and the selective mining unit (SMU) typical of benching and blast hole spacing and burden used at the Sue C open pit.

BLOCK MODEL INTERPOLATION

A grade x density model interpolated using ordinary kriging (OK). Similarly a density was interpolated using OK. The grade density interpolation was based on a nested spherical model oriented at 30º Az with the following parameters: 0.05 nugget; 10 m x 5 m x 3 m ranges at a 0.17985.66 sill and 20 m x 8 m x 5 m ranges at a 14978.89 sill.

The density interpolation used a spherical model at 30º Az. with 0.0707 nugget; 10.5 m x 5.2 m x 3.5 m ranges and a 0.12409 sill.

Grade x density kriging parameters were as follows:

- Four quadrants, a minimum of 10 samples and a maximum of 20 samples per quadrant.
- Search distances: X=20 m, Y=8 m, Z=4 m.
- Minimum distance between two selected samples: 1 m.

density interpolation kriging parameters were as follows:

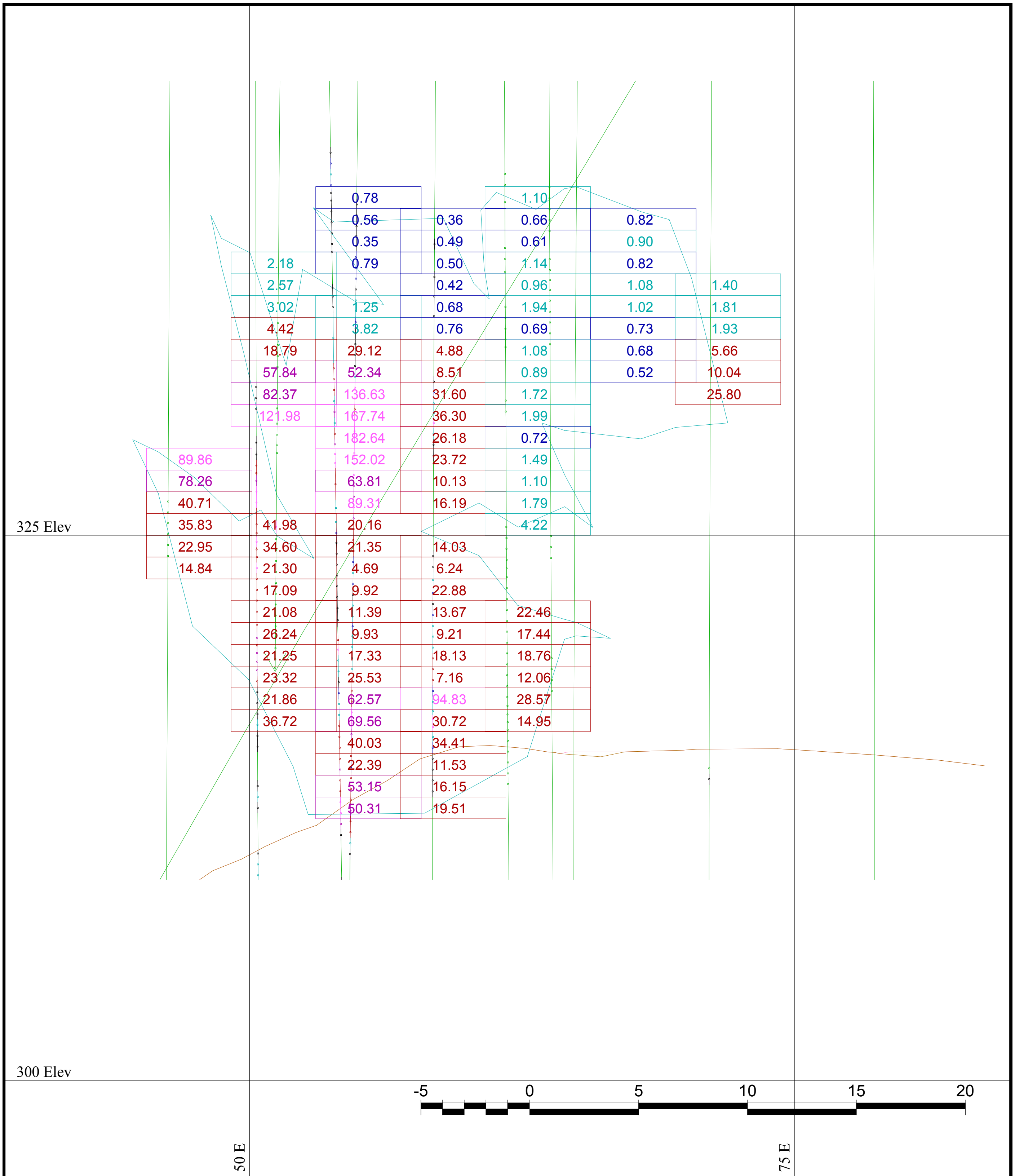
- Four quadrants, a minimum of 6 samples and a maximum of 20 samples per quadrant.
- Search distances: X=15 m, Y=10 m, Z=4 m.
- Minimum distance between two selected samples: 1 m.

Two interpolation passes were performed: the primary searches as above and the secondary - at 2 x the primary search ellipse dimensions.

Figures 15-8 and 15-9 show blocks on cross sections 5N and 6N, respectively, in the core area of the deposit. Figure 15-10 shows the block model in plan on 320 RL.

RESOURCE CLASSIFICATION

Cogema has not classified resources. Based on the close spacing of drill holes, mineralization continuity, and taking into account the complexity of geologic interpretation and the wireframe, RPA classifies the resource as Indicated Resources under CIM definitions (2000).



RPA

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Figure 15-8
Denison Mines Limited
Saskatchewan Projects
Caribou Urarium Deposit
Block Model Cross Section 5N

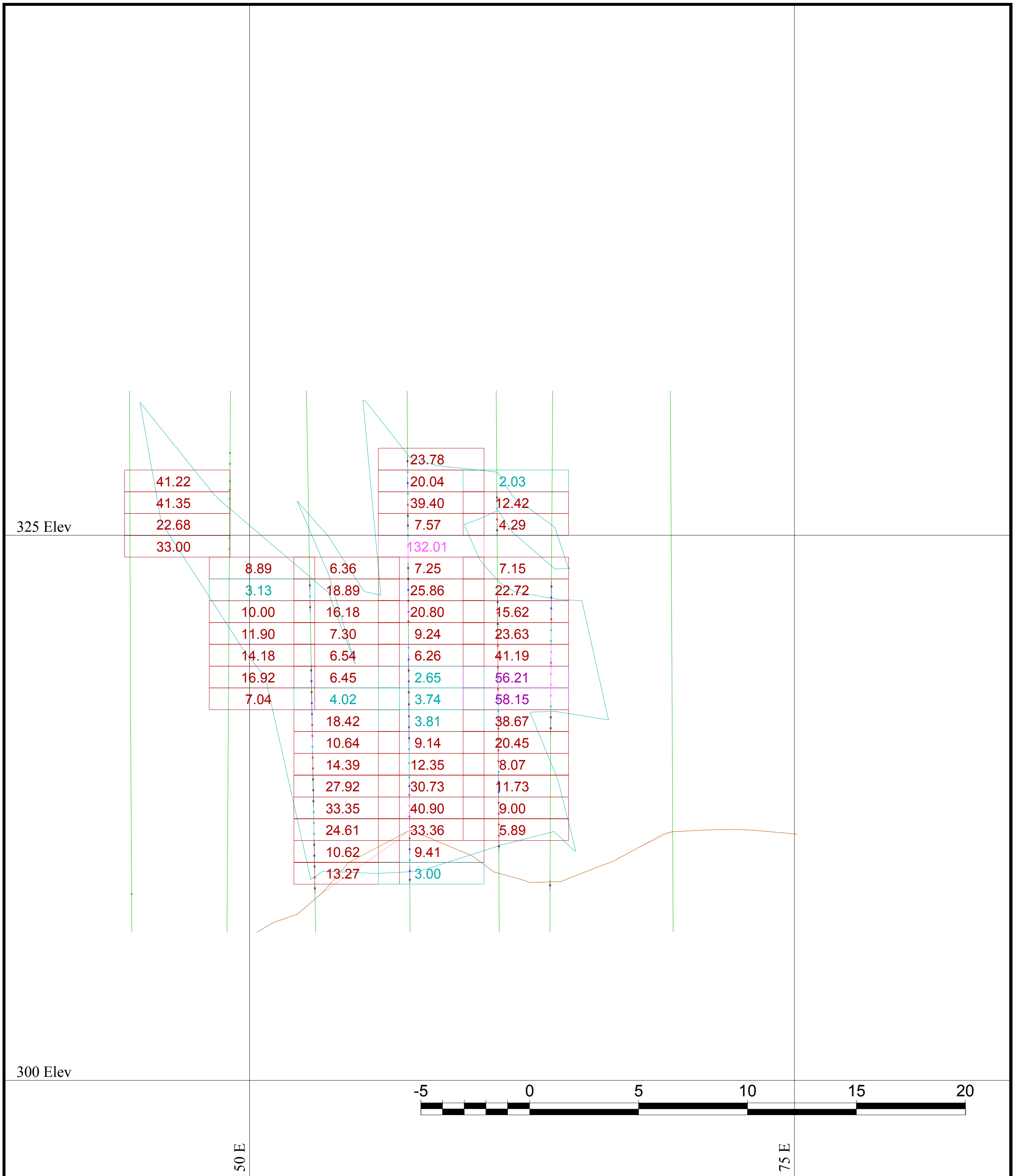

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Figure 15-9
Denison Mines Limited
Saskatchewan Projects
Caribou Urarium Deposit
Block Model Cross Section 6N

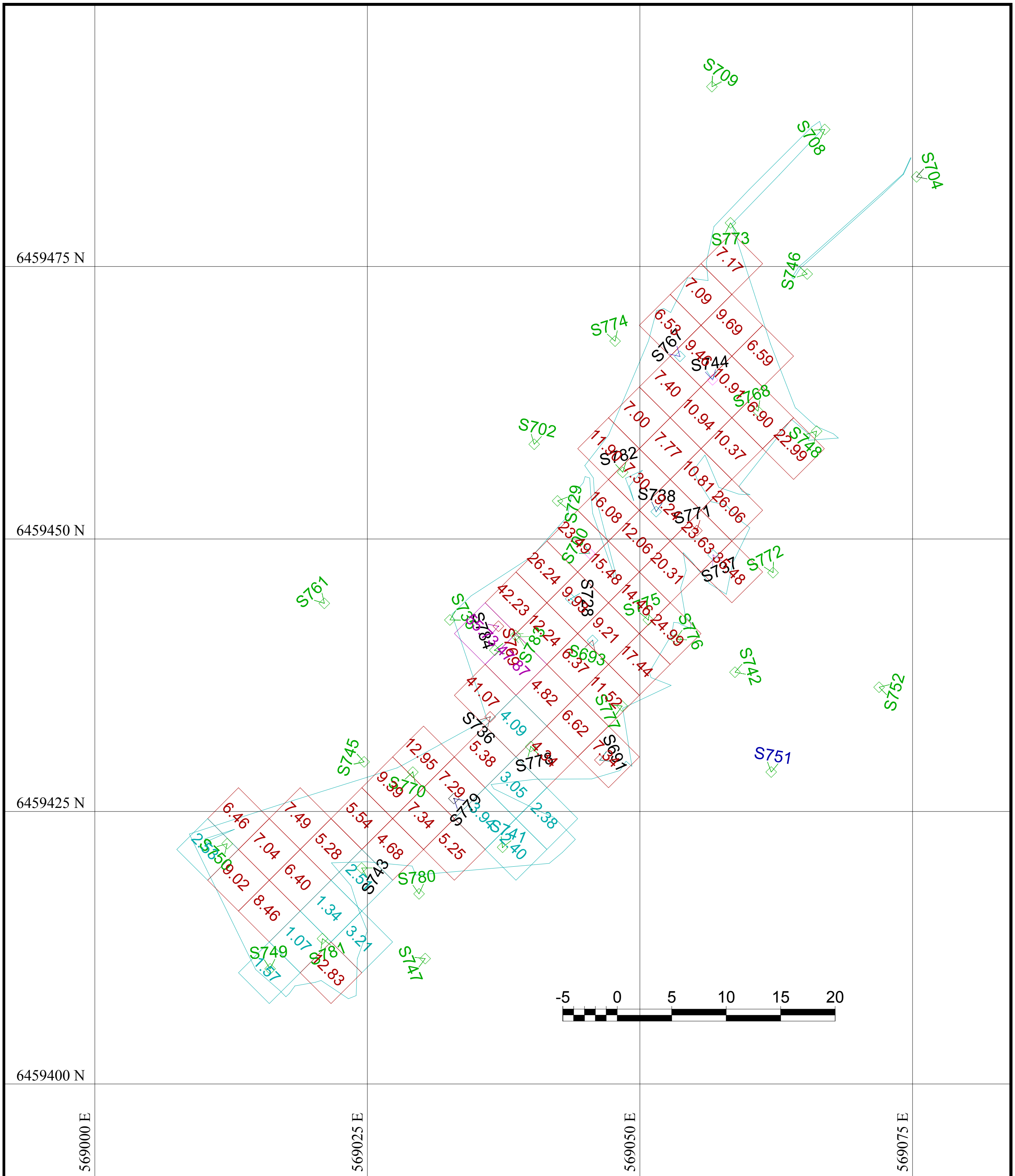

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Figure 15-10
Denison Mines Limited
Saskatchewan Projects
Caribou Urarium Deposit
320 RL Plan - Drill Holes and Blocks

MODEL VALIDATION

Cogema does not report validation of the model by other estimation methods.

RPA checked the tonnage and grade values reported by Cogema using Gemcom software reporting on the Cogema grade, density, and percentage block model values. The reported results duplicated the Cogema reported resources (Table 15-2) to within ±0.54%. The Cogema wireframe volume as calculated by Gemcom software is 17,883.5 m^3. Assuming an average bulk density of 2.85 m^3/t calculated from assay and composite averages, the global tonnage of the wireframe is approximately 50,680 tonnes. At zero cut-off within the wireframe, global tonnes and grade reported for the Cogema model are 50,581 tonnes grading 21.09 U kg/t, which is very close to the tonnage indicated from the wireframe volume.

RPA extracted assays within the Cogema solid and prepared density and length weighted 1 m down hole composites within the solid. Average grades of the assays, composites, and blocks were compared to the Cogema reported resource grades at zero cut-off as follows:

Cut-Off Grade U kg/t	Block Average U kg/t	Cogema Resource Report U kg/t	Cogema Composites U kg/t	RPA Composites U kg/t	Assays U kg/t
0	18.62	21.09	17.99	17.95	18.28

RPA carried out a preliminary ID^2 check estimate using the Cogema generated solid, RPA generated composites, and Cogema modeling parameters and search distances. The Cogema density model was also retained. For the same blocks reported by Cogema above the 0.85 U kg/t cut-off grade, the ID^2 model grade and metal content are approximately 9% higher. With the 0.85 U kg/t cut-off grade applied to the ID^2 model blocks, the tonnes are lower by approximately 11%, and the grade is higher by 23% for a net increase in metal content of less than 10% (Table 15-2). RPA does not regard this level of difference as significant.

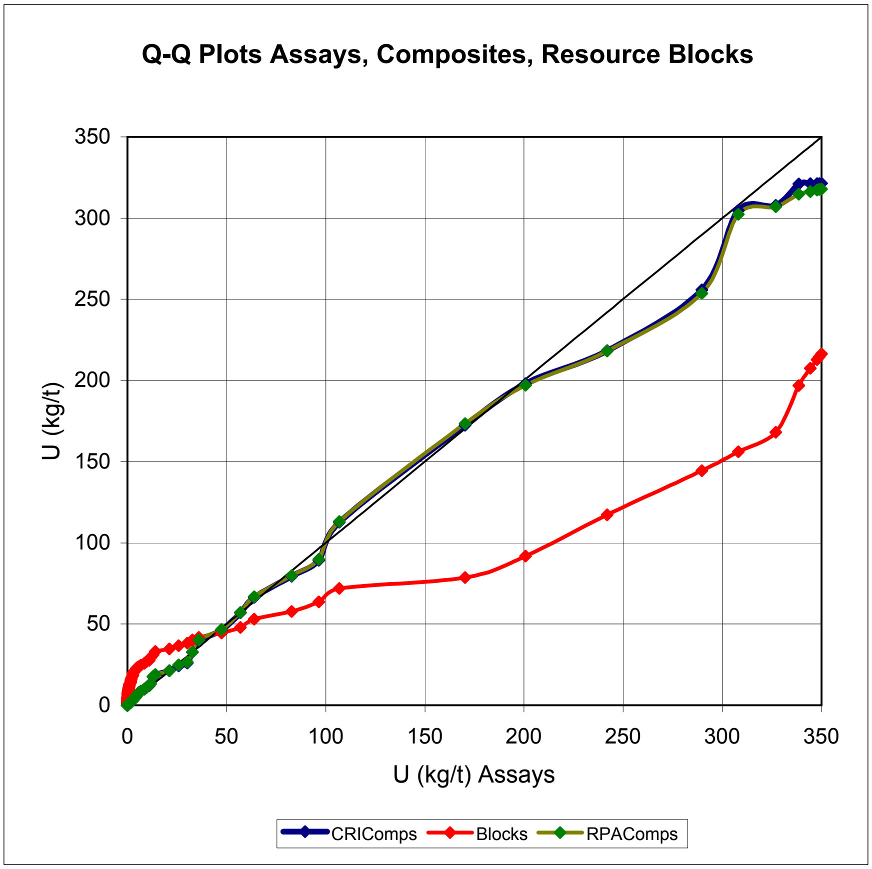
RPA notes, however, that there are a few high grade Cogema block values that are not supported by the comparable ID^2 interpolated blocks, or the surrounding composites. RPA concludes that either the Cogema estimate relied on additional drill hole data that is lacking in the RPA databases or there may be a problem with the Cogema kriging weights, multiple pass interpolation, or software itself. This does not appear to have a significant impact on the overall average grade of the deposit. In RPA's opinion, the Cogema estimate is acceptable for further work.

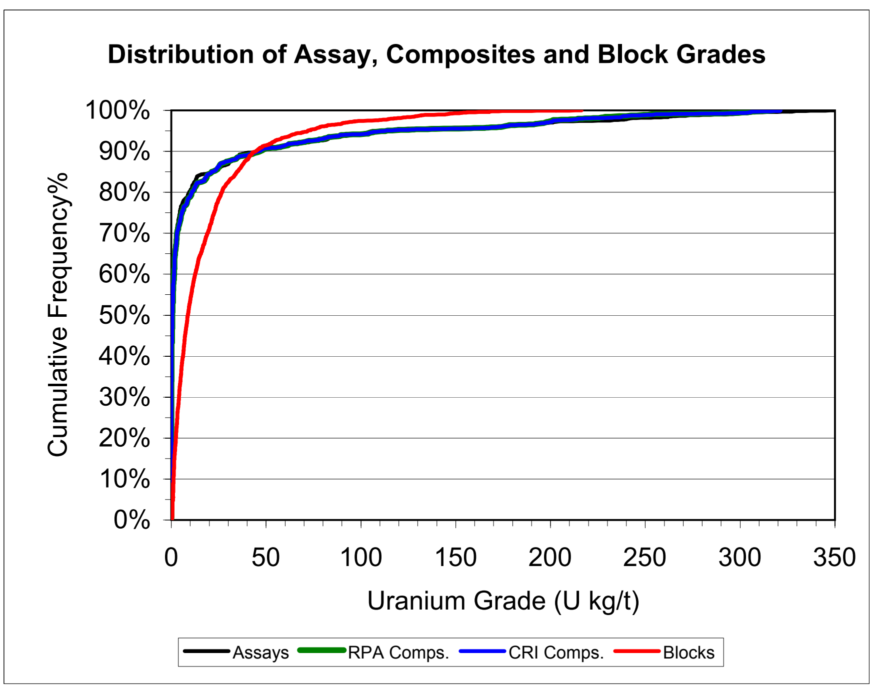
TABLE 15-2 VALIDATION OF COGEMA OK INTERPOLATION BY ID2

McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Model	Cut-Off Grade (U kg/t)	Tonnes	Grade U (kg/t)	U Metal (t)	ID^2	ID^2 U Metal (t)
Cogema Report Base Case	0.85	47,763	22.26	1,063	-	-
Cogema Blocks @ COG	0.85	47,588	22.38	1,065	24.45	1,164
ID^2 Blocks @ COG	0.85	42,584	24.11	1,027	27.35	1,165
Cogema Blocks @COG	Differences%vs. Base Case	-0.37%	0.54%	0.17%	9.84%	9.44%
ID^2 Blocks @ COG		-10.84%	8.31%	-3.43%	22.87%	9.54%

Figure 15-11 Distribution Assay, Composite and Block Grad in Cogema Resource Wireframe





16 ADJACENT PROPERTIES

The Sue D deposit has been identified south of the Sue A, B, C, and E deposits. It is not planned for development at this time. RPA has not completed a review of this deposit and it is not included in this report.

The property immediately surrounding the McClean Property, on three sides, was part of the Wolly Joint Venture which received considerable exploration effort. The McClean Property was carved out of portions of the Wolly Joint Venture properties by the participants.

The property south of the McClean Property is held by Cameco. The Sue E deposit extends onto this property.

17 OTHER RELEVANT DATA AND INFORMATION

MCCLEAN LAKE JOINT VENTURE OPERATION PLAN

The operating and development projects designed to recover the various Mineral Reserves outlined in the sections above are planned to be sequentially developed in order to sustain the ongoing ore processing and uranium production operations at the existing JEB mill facilities. In addition to the Sue A and Sue E open pits and the McClean North blind shaft boring production, the MLJV plans to develop the Midwest project. The Midwest deposit and associated Mineral Resources and Reserves are described in detail in a report entitled "Technical Report on the Midwest Uranium Deposit and Mineral Resource and Mineral Reserve Estimates Saskatchewan, Canada Prepared for Denison Mines Inc. June 2005". Certain information and data have been extracted from that report for inclusion in the overall MLJV production schedule. In addition, the MLJV plans to process production materials from the Cigar Lake operations. The Cigar Lake processing plans are considered here only to the extent that those quantities impact on the plans for treating the McClean and Midwest ores; however, the cost and revenue forecasts do not include any values representing Cigar Lake production.

CURRENT MILL METALLURGY

The McClean Joint Venture owns and operates the JEB mill. Operations started in 1999, and the mill has successfully been producing approximately 6 million pounds of U_3O_8 per year from the JEB and Sue C ores. Production plans include milling stockpiled Sue C ore, Sue A and E, and Midwest deposits.

In 2007, Denison plans for the JEB mill to start processing Cigar Lake joint venture (Cameco 50%, Cogema 37%) (CLMC) ore concurrently with McClean Lake deposit ores. CLMC will pay a custom milling fee, and the McClean Lake milling costs will be reduced by the economies of scale. The custom fee has not been included in this study, but the estimated milling costs reflect the benefits obtained through operating at higher throughput rates that will be realized when Cigar Lake material is processed.

A feasibility study has been completed for custom milling of the Cigar Lake ore ("Cigar Lake Project, 2001 Feasibility Study Supporting Document No. 4A, JEB Mill Expansion", issued by Cogema and Cigar Lake Mining Corporation, April 2001), and the capital costs will be covered by CLMC.

During 2004, MLJV reported that 152,092 tonnes of Sue C stockpile ore were processed in the JEB mill at a grade of 1.86% U_3O_8, producing over 6 million pounds of U_3O_8 calcined yellow cake. The uranium recovery was 97.3%.

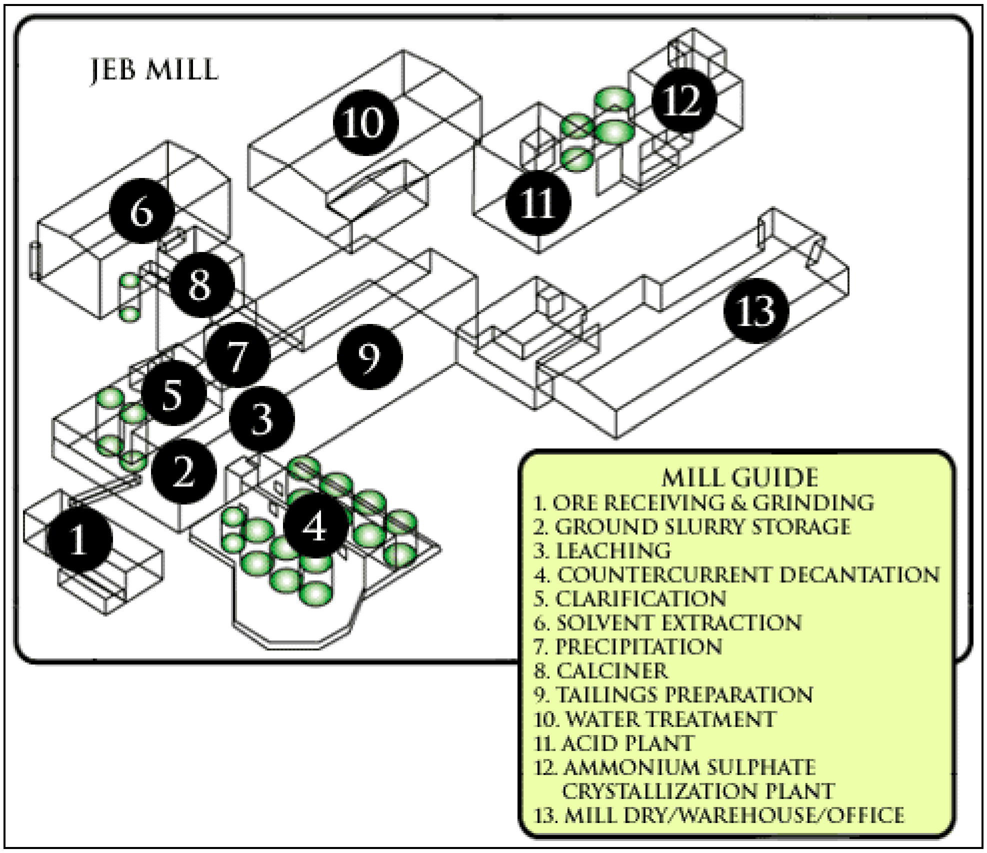
In 2005, MLJV plans to continue processing of Sue C stockpile ore, increasing the treatment rate to 165,000 tonnes per year but at a lower head grade of 1.68% U_3O_8. This will result in slightly higher mill losses at a recovery rate of 97%.

Thus far, the JEB mill has processed ores from the JEB pit and Sue C ores. Over the last five years, the operation of the mill has improved showing a consistent reduction in unit operating costs.

Figure 17-1 illustrates a simplified schematic of the JEB Mill. The main unit operations in the process are:

- Grinding with SAG and ball mill
- Leaching
- Counter current decantation (C.C.D)
- Pregnant solution clarification
- Solvent extraction
- Yellow cake precipitation
- Ammonium sulphate crystallizer
- Tailings neutralization and disposal
- Water treatment.

FIGURE 17-1 JEB MILL SCHEMATIC



Source: Denison

The ore is stockpiled near the mill and fed into the SAG mill. The grinding circuit is oversized, and MLJV reports that it was only required to operate 44.9% of the possible time in 2004. The ground slurried ore is stored in pachucas for continuous feed to the leaching circuit.

Uranium is leached from the ore in two circuits with sulphuric acid. Hydrogen peroxide is used as the oxidizing agent. The first circuit operates at room temperature and the second at 50ºC, and both circuits operate at atmospheric pressure. In 2004, the extraction was over 98% and the total mill sulphuric acid consumption was 94.8 kg/tonne. The total retention time is approximately 8 hours.

The solids are separated from the uranium-containing solutions after leaching in a conventional 6-stage thickener C.C.D circuit and MLJV reported that the system was 99.20% efficient in 2004. The overflow from the first thickener is clarified in a sand filter.

The solvent extraction circuit employs conventional technology and uses an amine extractant in a kerosene organic solvent. The circuit comprises uranium extraction, arsenic scrub, water wash, uranium stripping with ammonia, ammonia scrub and organic regeneration stages. The raffinate is partially recirculated to the C.C.D. circuit, and an exceptional 99.98% of the uranium was recovered in 2004.

Pregnant strip solution contains uranium extracted in the previous SX circuit, and also a significant amount of molybdenum. The molybdenum would precipitate with the uranium if not removed. Molybdenum concentration in the strip solution is high enough to exceed reject limits imposed by the uranium conversion facilities that treat yellowcake produced at McClean Lake. However, the carbon adsorption columns remove approximately 75% of the molybdenum from the pregnant strip solution and reduce the amount of molybdenum carried over to the precipitation circuit to below penalty limits.

Excess ammonium sulphate is produced when ammonia is used in the solvent extraction and yellowcake precipitation circuits. An ammonium sulphate extraction circuit is required to remove this excess. In addition, the ammonium sulphate extraction circuit evaporates any water containing ammonium sulphate when this has been added to the two circuits. The ammonium sulphate is sold as a fertilizer.

Yellowcake in the pregnant strip solution is precipitated with ammonia and the solids are separated from the liquid in thickeners and centrifuges. The precipitate is then dried and calcined in a rotary multiple hearth.

The calcined yellowcake is discharged to an automatic packaging capsule into 210 litre drums.

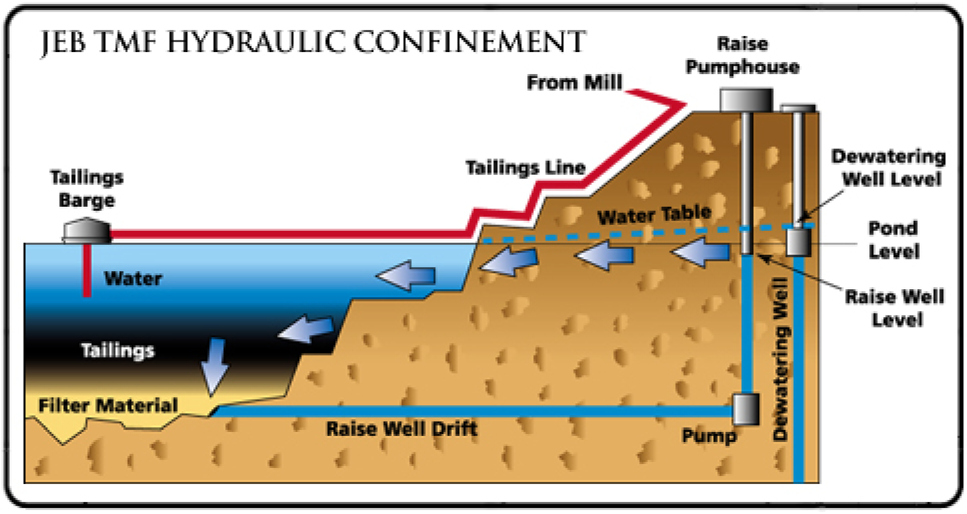
The tailings and part of the raffinate are mixed together with lime, barium chloride and ferric sulphate. The lime neutralizes acid, and the barium chloride and ferric sulphate precipitate radium and arsenic.

Thickened tailings received from the tailings thickener are deposited to the JEB tailings disposal pit, using a sub-aerial pervious surround tailings disposal system. The JEB tailings pit contains a dewatering drift and raise to control the water levels, and a base filter to drain the tailings. (Figure 17-2) The surrounding sandstone has a higher permeability than the consolidated tailings, thereby allowing ground water to flow around the deposited tailings mass. During plant operations, a hydraulic gradient is created in the filter layer under the tailings by pumping water through the drift and raising it with a submersible raise water pump. The water level in the tailings pit is controlled by pumping water from the pit with barge-mounted vertical turbine pumps to the reclaim water tank.

Water pumped from the surface of the tailings pit and the tailings thickener overflow is treated in a three-stage water treatment plant. Each stage contains a mixing tank and a clarifier. Lime, barium chloride and ferric sulphate are added to each stage and further precipitate radium and arsenic. The overflow from the last thickener flows into monitoring ponds and the water is not discharged until assays confirm that the water conforms to all regulations.

The mill was designed and is operated to meet all environmental and safety regulations. The employee exposure to radiation is well below the limits.

FIGURE 17-2 JEB TAILINGS MANAGEMENT FACILITY



Source: Denison

JEB MILL EXPANSION

There is an agreement between the MLJV and CLMC to partially custom mill Cigar Lake ores at the JEB mill.

The Cigar Lake ore will be ground at the Cigar Lake Mine and transported to the JEB mill as a pulp in specially designed, government approved containers. All of the Cigar Lake ore will be unloaded, stored and leached at the JEB mill. The pregnant aqueous solution will be further processed at both JEB mill and Rabbit Lake Mill. The capacity of the JEB mill will be increased from a nominal 6 million pounds of U_3O_8 per year to 12 million pounds of U_3O_8.

The JEB mill will require modification and expansion to enable it to treat the Cigar Lake ore and the flow sheet of the mill after the expansion is illustrated below in Figure 17-3.

The most important changes that will be required are:

- A slurry unloading and storage system

- Pregnant aqueous solution storage and load-out system
- A cyclone counter current circuit
- An extra pregnant aqueous sand filter and storage tanks
- Two extra molybdenum adsorption columns
- Expansion of the ammonium sulphate plant
- An extra ammonium sulphate storage bin
- Some extra reagent preparation and storage equipment
- An oxygen plant.

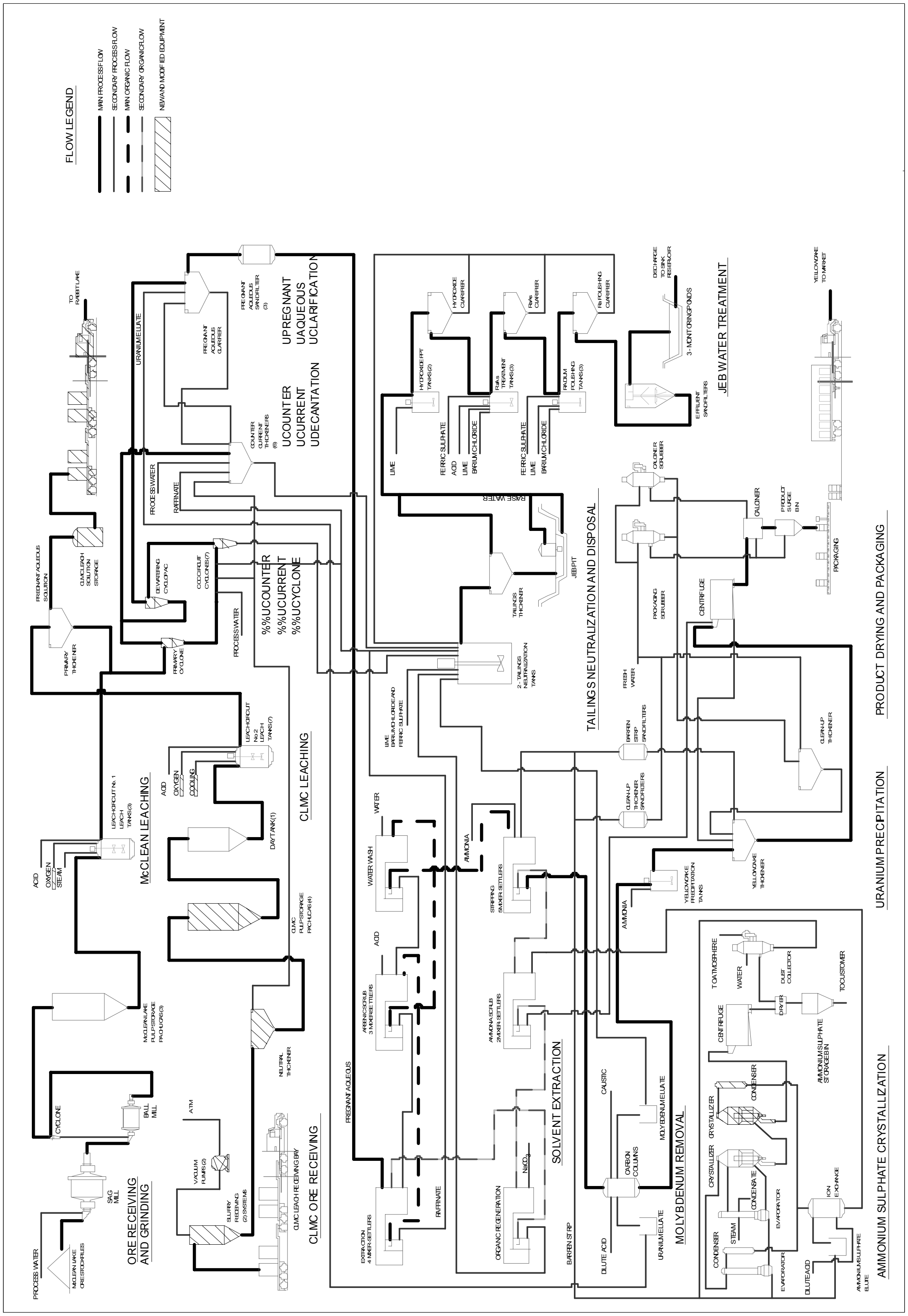
An engineering company, AMEC E&C Services Ltd. of Saskatoon (AMEC), completed a report “JEB Mill Expansion Project” in August 2002. This report includes flow sheets, general arrangement drawings, site plan drawings, and HVAC schematics, as well as design criteria and capital cost estimates. All the capital costs to adapt the plant to process Cigar Lake ore will be paid for by CLMC and so do not affect this study. The only mill-related capital cost to Midwest will be to expand the leaching circuit.

A key consideration for the McClean Lake and Midwest project economics is the requirement for the expanded mill to possess sufficient capacity to meet milling schedules.

Prior to the AMEC report, Cogema completed a study “Cigar Lake Project, 2001 Feasibility Study, Supporting Document No. 4A JEB Mill Expansion”. Included in this study are detailed calculations of the current capacity and required expansion for each unit process in the mill. Also, the capacity has been taken into account of all ancillary facilities such as water distribution, water treatment, electrical distribution, camp accommodation and sulphuric acid plant. These calculations have been checked by RPA and demonstrate that a reasonable safety margin has been included in each step of the milling.

The expansion includes an oxygen plant. The hydrogen peroxide currently used will be replaced with oxygen to reduce operating costs. The leaching pressure will be increased to 2 Bar from atmospheric, and the current leaching tanks are designed for this pressure.

FIGURE 17-3 JEB MILL SIMPLIFIED FLOW SHEET AFTER EXPANSION



PROCESS OPERATING COSTS

Operating costs for processing McClean Lake ores in the JEB Mill have been estimated by RPA for the years 2005 to 2017. These estimates were based on the production schedule developed by RPA and on the cost estimates provided by Cogema ("McClean Site Operating Expenses by Nature", Table 2, February 3, 2005).

The production schedule has been checked by RPA to ensure it conforms with the current and expanded mill capacities. The Cogema cost estimates were checked against actual costs for previous years, projected reagent consumptions, and production rates. The JEB Mill will also be processing Cigar Lake ore simultaneously with McClean Lake or Midwest ore starting from 2007, and the economies of scale will greatly reduce the unit milling costs. The extra cost of ferric sulphate required to precipitate arsenic has been applied to Sue A, Sue E, and Midwest ores, as described earlier in this report.

Cogema used a 2004 cost base with no inflation applied. The detailed operating cost estimations are shown in Table 17-1. The final costs include onsite and offsite support costs and an administration cost of 3% of the direct mill cost. A credit for the ammonium sulphate sold has also been taken into account. The total unit milling costs are estimated by RPA as $6.04 per lb from 2005 to 2009.

Operating supplies for the JEB Mill are generally purchased by Denison through annual supply contracts. At present, some commodities such as lime, caustic soda and ammonia are experiencing unusually high prices with significant increases expected in 2005. Reagent and operating supplies' costs account for between 8% and 25% of the total milling costs, depending on the milling rate.

The price of steel has risen sharply during 2004. The quantity of grinding balls and drums purchased will have an insignificant effect on overall costs. Propane costs are not expected to be significantly higher in 2005.

The increase in the price of chemicals will raise the total estimated milling cost by about \$0.10 per lb U_3O_8, or from 1.2 to 2.3% of the total milling cost.

TABLE 17-1 MCCLEAN LAKE ORE MILLING COST ESTIMATE

(2005 to 2009)

McClean Lake Joint Venture McClean Lake Property, Saskatchewan

	2005	2006	2007	2008	2009
Direct Milling Costs					
Personnel	$ 9,222	$ 9,339	$ 9,691	$ 9,913	$ 9,913
Reagents	$ 4,492	$ 2,896	$ 5,661	$ 6,101	$ 6,494
Maintenance Supplies	$ 3,179	3,,200	$ 3,168	$ 3,136	$ 3,105
Propane	$ 3,620	$ 2,963	$ 4,000	$ 4,325	$ 3,383
Contracts	$ 547	$ 300	$ 297	$ 294	$ 291
Utilities	$ 1,332	$ 906	$ 1,534	$ 1,656	$ 1,298
Total	$ 22,392	$ 19,604	$ 24,351	$ 25,125	$ 24,184
Direct Cost ($/lb)	$ 4	$ 5	$ 2	$ 2	$ 3
Support Costs Total					
Personnel	$ 4,185	$ 4,185	$ 4,185	$ 4,185	$ 4,185
Supplies	$ 1,356	$ 1,406	$ 1,406	$ 1,406	$ 1,392
Contract	$ 6,113	$ 6,132	$ 5,176	$ 5,011	$ 5,011
Utilities	$ 335	$ 360	$ 360	$ 320	$ 320
Taxes, fees, insurance	$ 2,363	$ 2,363	$ 2,363	$ 2,363	$ 2,363
Off-site allocation	$ 1,135	$ 1,005	$ 909	$ 972	$ 941
	$ 15,487	$ 15,451	$ 14,399	$ 14,257	$ 14,212
Onsite Manpower					
Mine	55	55	16		
Mill	118	120	124	123	123
Support	61	61	61	61	61
Total	234	236	201	184	184
Distributed Support Costs					
Mine	$ 4,925	$ 4,852	$ 1,141		
Mill	$ 10,562	$ 10,599	$ 12,750	$ 14,257	$ 14,212
Direct & Support	$ 32,954	$ 30,203	$ 37,101	$ 39,382	$ 38,396
JEB Output - U_3O_8 Pounds per Year					
Cigar Lake (lbs 000)			7,235	8,800	5,012
McClean Lake (lbs 000)	5,939	3,621	3,214	3,214	2,424
Total (lbs 000)	5,939	3,621	10,449	12,014	7,436
Mill Cost Allocation to McClean Ores	$ 32,954	$ 30,203	$ 11,412	$ 10,536	$ 12,516
Extra Charge Ferric Sulphate		$ 2,653	$ 3,471	$ 3,471	$ 2,621
Ammonium Sulphate Credit	$ (154)	$ (94)	$ (84)	$ (84)	$ (63)
Admin Cost 3% Direct Cost	$ 672	$ 588	$ 225	$ 201	$ 236
McClean Lake Milling Cost	**$ 33,472**	**$ 33,350**	**$ 15,024**	**$ 14,124**	**$ 15,310**
Milling Cost ($/lb)	**$ 5.64**	**$ 9.21**	**$ 4.67**	**$ 4.39**	**$ 6.32**

Source: Cogema

Total milling cost for McClean ores, 2005 to 2009, is $111,280,000 or $6.04 per lb.

PROCESS CAPITAL COST ESTIMATES

The present McClean Lake leaching circuit in the JEB mill will be modified from its existing configuration into two separate single stage leach circuits to allow CLMC ore to be leached separately from McClean Lake ores. It should be noted that additional leach capacity will be required for Midwest ores and the leaching circuit will be expanded from 3 to 8 tanks.

The cost of this expansion was not included in the Cigar Lake Feasibility Study as it will be paid for by Midwest. Preliminary estimates by Cogema based on flow sheets and general layout drawings indicate a capital cost of $17 million.

The preliminary capital cost of the ferric sulphate plant is estimated by SEPA at $3.0 million.

ENVIRONMENTAL CONSIDERATIONS

RPA has retained SENES Consultants Limited to address Environmental Considerations for this 43-101 review of McClean Lake and Midwest Projects that could materially affect the potential for mining of the reserves. This section of the report summarises SENES findings.

The McClean Lake deposits under review and summary comments on their history and status include the following.

- The JEB orebody: mined from 1995-1997.
- McClean Lake Underground: mining deferred until remote mining method has been developed.
- Sue ore bodies including A, B, C, and E zones.

- Sue C: open pit mined from 1997 – 2002.
- Sue A: Mining Proposed for summer 2005 as open pit.
- Sue B: Approved project as an open pit, with mining deferred until remote mining method has been developed.
- Sue E: Approval expected in 2005 as an open pit mine. No material issues have been identified in Environmental Assessment (EA) or EA review. Mining is proposed for 2005-2007.
- McClean North: EA has not been submitted. Project is deferred until remote mining method has been developed

MILLING AND TAILINGS MANAGEMENT

All ore from the McClean Lake deposits will be processed at the JEB mill which has recently been expanded to also process ores from the Cigar Lake deposit. The JEB mill has processed all ore from the JEB open pit and is currently processing ores from Sue C pit.

Extensive regulatory review has been completed for the management of tailings and waste rock from the McClean and Midwest Projects. Contaminated waste rock is being disposed of in the disused Sue C pit and all tailings from the milling of the Cigar, Midwest and McClean deposits are disposed of in the JEB tailings disposal facility. This tailings disposal facility can store all future production. Monitoring of the approved disposal facility has demonstrated that the facility is operating as designed.

Effluent treatment facilities are in place to manage all mine and mill effluents from the McClean Lease. These plants are performing well and meet all regulatory discharge limits.

PERMITTING AND APPROVALS

All uranium mining projects in Saskatchewan are to undergo environmental assessments under the Canadian Environmental Assessment Agency (CEA) and require Provincial Environmental Impact Statements (EIS). The CEA process is coordinated with the Province of Saskatchewan so that the EAs will meet both Federal and Provincial requirements.

Prior to the enactment of CEA, environmental permitting of the uranium mines was subject to the Environmental Assessment and Review Process Guideline Order. Under this order, a joint Federal/Provincial Panel was established to review the Uranium Mine developments in Northern Saskatchewan. This Panel approved the mining and milling of McClean North (underground mine), the Sue A, B and C open pits, the Midwest Mine (underground jet boring), and the JEB open pit mine. Although all projects were approved, Cogema has only recently obtained a CNSC licence for JEB and the Sue A, B and C open pits.

In November 2004, a CEA screening report was filed for the mining of Sue E. This report was reviewed by the regulators, and comments were received with no material issues raised. Cogema prepared a response document (filed in February 2005), to address all issues raised. Cogema expected that the licence application for Sue E would go before the CNSC Board for approval in the late spring of 2005, with the approval to be received by the fall of 2005.

At this time, there is no definitive schedule for licensing of the McClean underground or Caribou deposits. Cogema are conducting testing program on remote mining techniques in 2005/2006. It is hoped this test work will demonstrate that remote mining is a cost effective method for mining of all the deep deposits such as Caribou, Sue B, and Midwest. Should this not prove successful, near term mining of the deposits is unlikely. Remote mining techniques are projected to have minimal environmental issues and are likely readily permittable.

ECONOMIC ANALYSIS

PROJECT CAPITAL COST ESTIMATES

The capital cost estimates for each of the development projects have been outlined in the respective sections above and are presented here in summary form with the emphasis on timing. As noted at the outset of this section Midwest Project costs have been included here as part of the overall MLJV plans for the JEB processing facility. The costs and data included here have been extracted from the report entitled "Technical Report on the Midwest Uranium Deposit and Mineral Resource and Mineral Reserve Estimates Saskatchewan, Canada Prepared for Denison Mines Inc. June 2005".

Table 17-2 presents a summary capital cost schedule by project area.

TABLE 17-2 CAPITAL PROJECT SUMMARY
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

	2006	2007	2008	2009	2010	2011	Total
McClean North Project	$ 18.20	$ -	$ -	$ -	$ -	$ 1.30	$ 19.50
Midwest Project Mine Capital	$ -	$ -	$ -	$ 75.40	$ -	$ -	$ 75.40
Midwest Project Mill Capital	$ -	$ -	$ -	$ 27.00	$ -	$ -	$ 27.00
Midwest Pre-Stripping	$ -	$ -	$ -	$ 91.60	$ 64.70	$ -	$ 156.30
Total Capital Cost	$ 18.20	$ -	$ -	$ 194.00	$ 64.70	$ 1.30	$ 278.20

The McClean North Project capital spending is forecast to be incurred in 2006 providing the opportunity to start production operations in 2007. The Midwest Project is forecast to start in 2009 to enable production and ore delivery to the JEB mill in 2010.

No capital costs are forecast to be incurred for the development of Sue A or the Sue E pit at this time.

PROJECT OPERATING COST ESTIMATES

The operating cost estimates for each of the development projects have been outlined in the respective sections above and are presented here in summary form by operating area.

Mining costs have been estimated based on the mining method, and physical conditions associated with each of the mine development project. The open pit mining costs have been forecast based on a rate of $4.10 per tonne moved in the Sue A and Sue E mines. These two developments are nearby and closely related to the completed Sue C open pit and operating conditions are expected to be consistent with that experience. The $4.10 operating cost rate is based on the actual operating experience during the period when Sue C was mined.

The McClean North Project operating costs have been developed based on contract shaft drilling proposal and project design work by Golder and Denison. Administration and overhead allocations including camp costs for housing project personnel and miscellaneous offsite costs have been included. Ore haulage costs for loading and trucking of the recovered ore material to the JEB processing facility have been included at a rate of $1.00 per tonne hauled.

The Midwest Project operating costs are outlined in the report entitled "Technical Report on the Midwest Uranium Deposit and Mineral Resource and Mineral Reserve Estimates Saskatchewan, Canada Prepared for Denison Mines Inc. June 2005".

URANIUM MARKETS AND PRICES

RPA is not expert in the area of uranium markets and price forecasting. The following briefly summarizes the current market situation paraphrasing a number of descriptions and commentaries published by various firms and organizations which do follow the uranium market trends.

The only significant current commercial use for uranium in the world is to fuel nuclear power plants for the generation of electricity. Nuclear power currently contributes 16% of the world's power requirements, and there is a general anticipation that nuclear power capacity will increase slightly, with developing countries showing the highest growth rate. The world's power reactors require about 180 Million lb. of uranium oxide (U_3O_8) concentrate each year. Forecasts generally expect the demand for nuclear fuel to increase at a somewhat lower rate than the increase in power generation due to a trend of improving generation efficiency.

Primary production (uranium produced from mining of uranium deposits) in 2003 was estimated at 93 Million lb. of U_3O_8, representing about 55% of annual uranium consumption. Secondary supply (from inventories built up historically, de-enrichment from nuclear weapons, and spent fuel reprocessing) provided 45% of fuel requirements in 2003. The current expectation is that inventories will be exhausted over the next few years, resulting in a potential supply gap emerging which will need to be filled through additional mine supply.

Spot market volume for U_3O_8 over the last ten years has averaged about 11% of demand. Spot prices have been steadily increasing since the low of US$7.10 per pound at December 31, 2000 to US$26.25 at the end of April, 2005. Long term prices are usually at a US$1.00-2.00 premium to the spot price at the time of entering into the contract and are escalated throughout the life of the contract. Currently, the spread between the long-term price and the spot price is about US$3.00 with long term prices quoted at US$28.00-29.00.

URANIUM SUPPLY

In 2003, eight mining companies accounted for approximately 80% of worldwide mine production (Table 17-3). Geographically, the two leading uranium producing countries (Canada and Australia) account for 47% of production, with one region, Saskatchewan (Canada), accounting for about 30% of production.

TABLE 17-3 MAJOR URANIUM PRODUCERS (>4 MLBS/YEAR)
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Company	Country	2003 Production (000,000 lb.)
Cameco	Canada	18.5
Cogema	France	12.3
ERA	Australia	11.2
KazAtomProm	Kazakhstan	8.4
Priargunsky	Russia	7.3
WMC	Australia	7.0
Rössing	Namibia	5.3
Navoi	Uzbekistan	4.6

Source: World Nuclear Association

Production is concentrated in a relatively few large mines, with McArthur River, Ranger, and Olympic Dam being the largest (Table 17-4). Reliability of supply is a large concern for uranium consumers and has become more topical in the last year due to recent operational problems at several large uranium mines. In April 2003, there was a flood at the McArthur River mine that curtailed production for three months. Olympic Dam had a fire in the SX circuit that reduced production. Rio Tinto announced that the Rössing Mine is uneconomic at current uranium prices and is likely to close in 2007. Low prices for numerous years have stifled investment in the uranium industry.

TABLE 17-4 LARGEST WESTERN WORLD URANIUM MINES BY 2003 PRODUCTION

McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Mine	Country	Main Owner	Type	Production (Million lbs.)	Percent
McArthur River + Key Lake	Canada	Cameco	Underground	15.2	16.3
Ranger	Australia	ERA (Rio Tinto 68%)	Open pit	11.2	12.0
Olympic Dam	Australia	WMC	By-product	7.0	7.5
McClean Lake	Canada	Cogema	Open pit	6.0	6.5
Rabbit Lake	Canada	Cameco	Underground	5.9	6.4
Rössing	Namibia	Rio Tinto (69%)	Open pit	5.3	5.7
Akouta	Niger	Cogema/Onarem	Underground	5.2	5.6
Arlit	Niger	Cogema/Onarem	Open pit	2.9	3.1
Vaal River	South Africa	Anglogold/Nufcor	By-product	2.0	2.1
Beverley	Australia	Heathgate	ISL	1.6	1.7
Top Ten				**62.3**	**66.9**

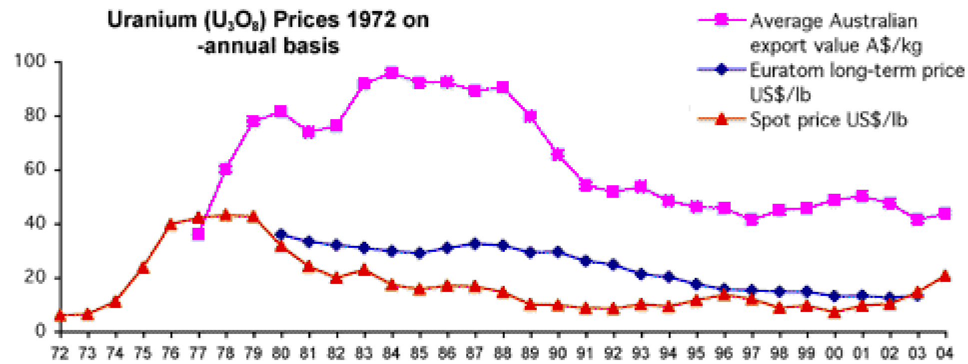
Source: World Nuclear Association

As a result of these factors, new mines are required to meet the medium and long-term market requirements. However, apart from the scheduled Cigar Lake start-up, no large new mine is being proposed for development and several mines are scheduled to close as their reserves are depleted. Significant uncovered demand exists in the market from 2005 onward and Denison expects the tightening supply-demand balance to put upward pressure on uranium prices.

URANIUM PRICES

Figure 17-4 illustrates the spot market price for U_3O_8 over the past three decades. The spot market represented only about 11% of uranium demand in 2002, in line with the level in recent years (see also Section 20 Contracts). The spot price increased during 2001 from US$ 7.10 to US$9.60 and to US$10.20 at December 31, 2002. Spot prices in to June 2003 increased to the US$10.90 to US$11.00 range.

FIGURE 17-4 URANIUM SPOT PRICE HISTORY



URANIUM PRICE ASSUMPTIONS

A price of US$23.20 per pound U_3O_8 has been used for the estimation of reserves and resources in this report. RPA is of the opinion that this estimate is consistent with various independent forecasts of supply and demand fundamentals and price projections (outlined above). Current spot market prices are in the range of US$26.25 per pound U_3O_8. RPA has used a US-Cdn exchange rate based on current levels of US$0.81:C$1. Denison currently operates with a policy of selling most of its production under long-term contracts. Generally, long term contract prices reflect the spot market at the time of negotiation, with a premium for delivery guarantees. Long term contract terms are confidential, and RPA is not aware of the terms of Denison's existing contracts.

TAXES AND ROYALTIES

PROVINCIAL ROYALTIES AND TAXES

MWJV pays royalties to the Province of Saskatchewan on the sale of uranium extracted from ore bodies in the province under the terms of Part III of the Crown Mineral Royalty Schedule, 1986 (Saskatchewan) ("Schedule") as amended.

Two royalties are payable: the basic royalty and the tiered royalty. The basic royalty is calculated as 5% of the gross sales revenue from uranium, reduced by the Saskatchewan Resource Tax Credit. This credit is equal to 1% of the gross sales revenue and results in a net royalty of 4%. The tiered royalty is based on uranium price and is payable as a percentage of revenue, after applying credits for capital recovery. The capital credit allowance for an open pit mine is specified as $45 per kg of annual capacity, and on that basis amounts to a credit of $61 million. Table 17-5 sets out the tiered royalty rates to be applied, based on prevailing uranium prices.

TABLE 17-5 SASKATCHEWAN TIERED ROYALTY SCHEDULE
McClean Lake Joint Venture McClean Lake Property, Saskatchewan

Average $/kg U_3O_8 *	Royalty %
Up to $30	0
$30 to $45	6
$45 to $60	10
more than $60	15

*1999 bracket value to be indexed annually

Source: Denison Mines Inc.

Royalties are calculated based on gross revenue value less shipping and transport costs.

The Province of Saskatchewan levies a mining tax based on net profit with a rate of 5% up to a cumulative unit sales of one million metric tonnes of all minerals. Above that level the tax rate increases to 10%. Net profit is based on the gross value of mineral sales less all direct operating costs, current exploration and pre-production expenses, depreciation, reclamation, and decommissioning costs and losses from prior years. In 2002, the province introduced a 10 year holiday for new gold and base metal mines.

MLJV is subject to capital tax on paid-up capital (as defined in provincial tax legislation) in respect of its operations in Saskatchewan. It currently pays a rate of 0.6% on paid up capital in excess of $15 million. In addition, a resource corporation in Saskatchewan pays a corporate surcharge of 3.6% of the gross sales of uranium where the amount calculated exceeds the regular capital tax described above. The provincial resource surcharge is based on gross revenues less the transportation component of any revenue.

A provincial resource surcharge of 3.6% on uranium production less $0.09 per lb. for the estimated shipping component has been used in the RPA economic analysis to reflect the combined effect of capital tax and provincial resource surcharge.

OTHER ROYALTIES

At the Midwest Project two royalties, with identical terms, are payable on 20% of the production from MWJV, declining to 12.5% after payout (revenue equal to capital, operating costs and royalties). Denison is responsible for 5.5% and Cogema 14.5% (declining after payout). Each of the royalties has the following terms.

Payments under each royalty are calculated to be:

- 1% of revenue on first 800,000 pounds of U_3O_8 production
- 1.75% of revenue on following 700,000 pounds of U_3O_8 production
- 2% of revenue on balance of U_3O_8 production

MCCLEAN LAKE OPERATIONS CASHFLOW

RPA has developed a mine production schedule incorporating all of the identified Mineral Reserves outlined in this report as well as the Midwest Project Mineral Reserves. Table 17-6 summarizes the mining schedule.

The year end 2004 ore stockpile at the JEB mill (consisting primarily of Sue C ore materials) is estimated to contain 268,000 tonnes carrying an average grade of 1.39% U_3O_8. This material is classified as Proven Mineral Reserve.

Based on the available mill feed material from the mining schedule, RPA has developed an Operations Cashflow estimate that includes the combined estimates and projections associated with the various mine development projects and production schedules outlined in the sections above. The production schedule has been developed considering both the various sources of ore feed from the MLJV mines and the projected uranium processing schedule for the Cigar Lake Joint Venture (CLJV) material that is planned to be treated at the JEB facilities. However, the projected revenues, operating costs, and capital costs are only those associated with the MLJV operations, excluding any revenue and costs associated with the CLJV.

Table 17-7 presents a summary of the operating plan and cash flow based on a 16-year operating life. The cash flow is on a pre-income tax basis as the corporate entities involved in the joint venture have different tax pools and tax positions. Since the cash flow represents an ongoing operating entity and there are no net capital investments or negative cash flows in the initial years, an internal rate of return factor cannot be calculated.

TABLE 17-6 MCCLEAN LAKE JV AND MIDWEST LAKE JV MINING SCHEDULE

			2005	2006	2007	2008	2009	2010	2011	Total
Sue A *										
Total BCM	BCM	947,103	947,103	-	-					947,103
Overburden	BCM	-	-	-	-					-
Rock Waste	BCM	914,568	914,568	-	-					914,568
Special Waste	BCM	19,308	19,308	-	-					19,308
Ore	BCM	13,227	13,227	-	-					13,227
Tonnes Ore	t	31,948	31,948	-	-					31,948
Grade	U_3O_8 %	1.99%	1.99%	0.00%	0.00%					0.00%
Tonnes U	t	635	635	-	-					24,716
U_3O_8 lbs		1,400,199	1,400,199	-	-					0
										0
McClean North *										0
Tonnes Ore	t	39,056			8,293	8,293	10,259	10,402	1,809	39,056
Grade	U_3O_8 %	6.19%			8.68%	8.68%	4.80%	3.81%	4.85%	6.09%
Tonnes U	t	2,048			610	610	417	336	74	2,048
U_3O_8 lbs		5,325,410			1,587,075	1,587,075	1,084,706	872,966	193,588	5,325,410
Sue E *										
Total BCM	BCM	5,459,025	3,722,321	1,540,580	196,124					5,459,025
Overburden	BCM	-	-	-	-					-
Rock Waste	BCM	5,082,581	3,722,321	1,290,326	69,934					5,082,581
Special Waste	BCM	114,173	-	90,243	23,930					114,173
Ore	BCM	262,272	-	160,012	102,260					262,272
Tonnes Ore	t	628,077	-	382,165	245,911					628,077
U_3O_8 Grade	U_3O_8 %	0.78%	0.00%	0.34%	1.16%					0.66%
Ni Grade	Ni%	0.53%	0.00%	0.27%	0.94%					0.53%
		0.871	0.00%	0.44%	1.55%					
Tonnes U	t	4,151	-	1,102	2,418					3,520
U_3O_8 lbs		10,791,809	-	2,865,429	6,286,026					9,151,454
Ni lbs		7,381,894	-	2,296,230	5,085,663					7,381,894
As lbs		12,062,381	-	3,684,908	8,377,472					
Midwest **										
Total BCM	BCM	44,593,096					22,640,627	21,952,469		44,593,096
Overburden	BCM	2,480,282					2,480,282	-		2,480,282
Rock Waste	BCM	40,457,910					20,137,001	20,320,909		40,457,910
Special Waste	BCM	1,509,656					23,344	1,486,312		1,509,656
Ore	BCM	145,248					-	145,248		145,248
Tonnes Ore	t	345,516					-	345,516		345,516
U_3O_8 Grade	U_3O_8 %	5.47%					-	5.47		4.64%
Ni Grade	Ni %	4.37%					-	4.37		4.37%
Co Grade	Co %	0.33%					-	0.33		0.33%
As Grade	As%	7.20%					-	7.20		
Tonnes U	t	16,013					-	16,013		16,013
U_3O_8 lbs		35,302,555					-	35,302,555		35,302,555
Ni lbs		33,300,163					-	33,300,163		33,300,163
Co lbs		2,544,732					-	2,544,732		2,544,732
As lbs		54,830,580					-	54,830,580		
Total ***										
Total BCM	BCM	50,999,224	4,669,424	1,540,580	196,124	-	22,640,627	21,952,469	-	50,999,224
Overburden	BCM	2,480,282	-	-	-	-	2,480,282	-	-	2,480,282
Rock Waste	BCM	46,455,059	4,636,889	1,290,326	69,934	-	20,137,001	20,320,909	-	46,455,059
Special Waste	BCM	1,643,137	19,308	90,243	23,930	-	23,344	1,486,312	-	1,643,137
Ore	BCM	420,747	13,227	160,012	102,260	-	-	145,248	-	420,747
Tonnes Ore	t	1,044,596	31,948	382,165	254,204	8,293	10,259	355,918	1,809	1,044,596
U Grade	U_3O_8 %	2.29%	1.99%	0.53%	1.40%	8.68%	4.80%	4.61%	4.85%	2.29%
Ni Grade	Ni%	1.77%	0.00%	0.27%	0.91%	0.00%	0.00%	4.24%	0.00%	1.77%
Co Grade	Co%	0.11%	0.00%	0.00%	0.00%	0.00%	0.00%	0.32%	0.00%	0.11%
Tonnes U	t	22,848	635	1,102	3,028	610	417	16,349	74	22,217
U_3O_8 lbs		52,819,974	1,400,199	4,452,503	7,873,101	1,587,075	1,084,706	36,175,521	193,588	52,766,693
Ni lbs		40,682,056	-	2,296,230	5,085,663	-	-	33,300,163	-	40,682,056
Co lbs		2,544,732	-	-	-	-	-	2,544,732	-	2,544,732

* Denison Holds 22.50% Interest in the MLJV Mineral Production.

** Denison Holds 25.17% Interest in the Midwest Lake Mineral Production.

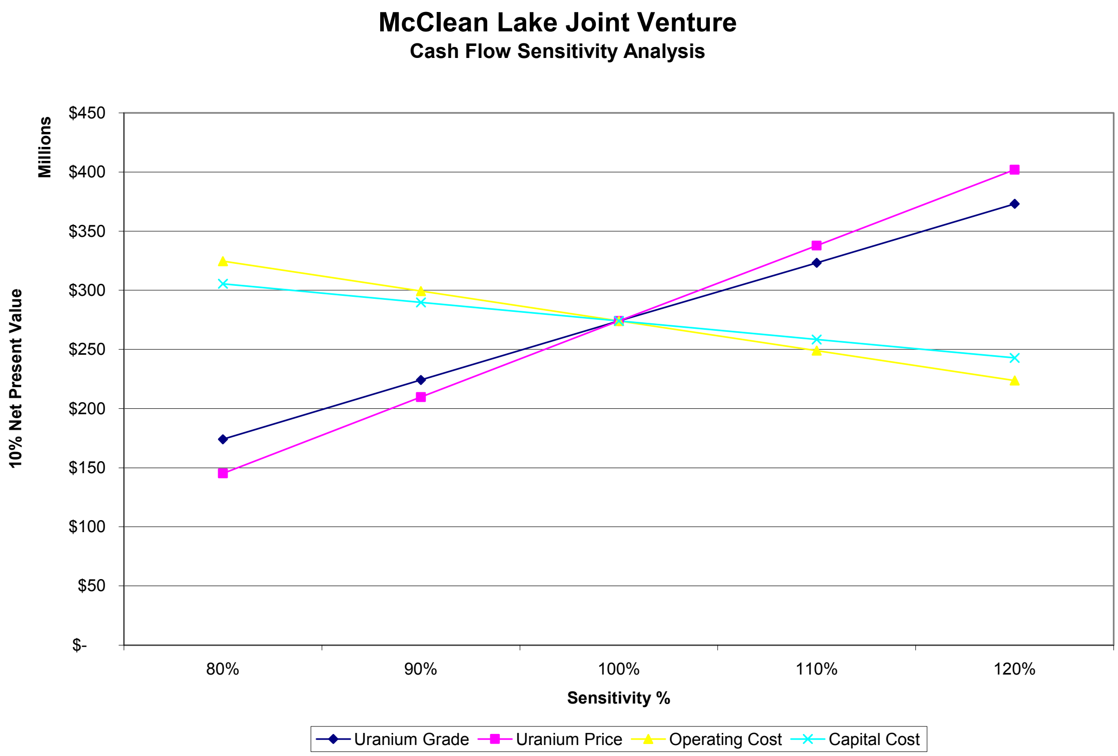
Table 17-7 MCCLEAN LAKE OPERATIONS CASH FLOW ESTIMATE

	2005	2006	2007	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	Total
Production																	
Recovered U3O8 lbs	5,299	4,010	4,687	3,006	2,513	5,115	4,594	4,053	4,254	4,017	3,545	3,545	3,545	3,545	3,545	3,545	62,816
Recovered Ni lbs	-	-	1,396	644	644	1,856	1,948	1,789	1,878	1,773	1,565	1,565	1,565	1,565	1,565	1,565	21,316
Recovered Co lbs	-	-	-	-	-	93	100	136	143	136	120	120	120	120	120	120	1,324
Revenue																	
Net U3O8 Revenue FOB Minesite	$ 151,287	$ 114,493	$ 133,805	$ 85,812	$ 71,756	$ 146,021	$ 131,157	$ 115,713	$ 121,441	$ 114,694	$ 101,201	$ 101,201	$ 101,201	$ 101,201	$ 101,201	$ 101,201	$ 1,793,383
Net Ni Revenue FOB Minesite	$ -	$ -	$ 3,567	$ 1,645	$ 1,645	$ 4,743	$ 4,977	$ 4,572	$ 4,798	$ 4,532	$ 3,998	$ 3,998	$ 3,998	$ 3,998	$ 3,998	$ 3,998	$ 54,469
Net Co Revenue FOB Minesite	$ -	$ -	$ -	$ -	$ -	$ 641	$ 690	$ 938	$ 993	$ 938	$ 828	$ 828	$ 828	$ 828	$ 828	$ 828	$ 9,166
Total Revenue before Royalty	$ 151,287	$ 114,493	$ 137,372	$ 87,457	$ 73,401	$ 151,406	$ 136,824	$ 121,223	$ 127,232	$ 120,164	$ 106,027	$ 106,027	$ 106,027	$ 106,027	$ 106,027	$ 106,027	$ 1,857,018
Royalties																	
Midwest Royalty	$ -	$ -	$ -	$ -	$ -	$ 642	$ 405	$ 551	$ 583	$ 551	$ 486	$ 486	$ 486	$ 486	$ 486	$ 486	$ 5,645
Net Revenue after Royalty	$ 151,287	$ 114,493	$ 137,372	$ 87,457	$ 73,401	$ 150,764	$ 136,419	$ 120,673	$ 126,649	$ 119,613	$ 105,541	$ 105,541	$ 105,541	$ 105,541	$ 105,541	$ 105,541	$ 1,851,373
Operating Costs																	
Total Mining	$ 19,145	$ 25,459	$ 19,946	$ 16,358	$ 106,821	$ 120,865	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 308,593
Uranium Process	$ 32,006	$ 24,222	$ 28,308	$ 18,154	$ 10,958	$ 22,300	$ 20,030	$ 17,671	$ 18,546	$ 17,515	$ 15,455	$ 15,455	$ 15,455	$ 15,455	$ 15,455	$ 15,455	$ 302,438
Nickel/Cobalt Process	$ -	$ -	$ 1,689	$ 779	$ 779	$ 2,246	$ 2,357	$ 2,165	$ 2,272	$ 2,146	$ 1,893	$ 1,893	$ 1,893	$ 1,893	$ 1,893	$ 1,893	$ 25,792
Total Operating Cost	$ 51,151	$ 49,681	$ 49,943	$ 35,291	$ 118,558	$ 145,410	$ 22,386	$ 19,836	$ 20,818	$ 19,661	$ 17,348	$ 17,348	$ 17,348	$ 17,348	$ 17,348	$ 17,348	$ 636,824
less Capitalized Production Cost	$ -	$ -	$ -	$ -	$ 91,578	$ 64,700	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 156,278
Net Operating Cost	$ 51,151	$ 49,681	$ 49,943	$ 35,291	$ 26,980	$ 80,710	$ 22,386	$ 19,836	$ 20,818	$ 19,661	$ 17,348	$ 17,348	$ 17,348	$ 17,348	$ 17,348	$ 17,348	$ 480,545
Operating Cost/lb U3O8	$ 9.65	$ 12.39	$ 10.66	$ 11.74	$ 10.73	$ 15.78	$ 4.87	$ 4.89	$ 4.89	$ 4.89	$ 4.89	$ 4.89	$ 4.89	$ 4.89	$ 4.89	$ 4.89	$ 7.65
Capital Costs																	
McClean North Project	$ -	$ 18,222	$ -	$ -	$ -	$ -	$ 1,330	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 19,552
Midwest Project Mine Capital	$ -	$ -	$ -	$ -	$ 75,400	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 75,400
Midwest Project Mill Capital	$ -	$ -	$ -	$ -	$ 27,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 27,000
Midwest Pre-Stripping Capital	$ -	$ -	$ -	$ -	$ 91,578	$ 64,700	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 156,278
Total Capital Cost	$ -	$ 18,222	$ -	$ -	$ 193,978	$ 64,700	$ 1,330	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 278,231
Mining Taxes	$ 34,191	$ 25,875	$ 30,547	$ 19,535	$ 5,595	$ 11,538	$ 10,982	$ 24,830	$ 27,923	$ 26,371	$ 23,269	$ 23,269	$ 23,269	$ 23,269	$ 23,269	$ 23,269	$ 357,000
Project Cash Flow (Pre Income Tax)	$ 65,945	$ 20,715	$ 56,883	$ 32,631	-$ 153,152	-$ 6,184	$ 101,721	$ 76,007	$ 77,909	$ 73,580	$ 64,924	$ 64,924	$ 64,924	$ 64,924	$ 64,924	$ 64,924	$ 735,597
10% NPV	$ 274,173																

SENSITIVITIES

RPA developed a sensitivity analysis for the cash flow estimate presented in Table 17-7 where the impact of changes to uranium grade, capital cost, operating cost, and metal prices (uranium, nickel, and cobalt) was determined. The results of these sensitivities are illustrated in Figure 17-5.

FIGURE 17-5 MCCLEAN AND MIDWEST CASHFLOW NPV SENSITIVITY ANALYSIS



Source: RPA

EXPLORATION POTENTIAL

RPA has not investigated the exploration potential of the various properties outlined in this report; however, it is noted that there remains some areas of opportunity based on the information available. The Sue E deposit as estimated by RPA hosts a significant Inferred Resource as outlined in section 13 above. Several factors contribute to this:

- Sparse drilling (relative to the range of the variogram), especially in the southern domain;

- The need for "indicated" resource to have at least two different drill holes within the search;
- Some of the extremely high-grade samples occur in the deep southern region and are not supported in the samples from surrounding drill holes.

With the current drilling, these estimates of the inferred resource are considerably less reliable than the estimates of the indicated resources. But with considerable additional potential resources in the inferred category, additional definition drilling is warranted, particularly in the deep southern extensions of the deposit. RPA also notes that an earlier recommendation to drill some east-dipping holes from the west has not yet been implemented. The prevailing view of the deposit has been of steeply east-dipping structures. While this study supports this as a general observation about the mineralization in the south, it also suggests that the mineralization in the north may generally dip to the west. If this interpretation is correct, east-dipping holes would help to better delineate the ore outlines in the north.

In addition to the resources that further drilling might be able to graduate from "inferred" to "indicated", there is some additional resource potential in the basal sandstone. This study, in keeping with previous studies, has clipped the mineralized envelope to the Athabasca unconformity. The current drilling contains several significant (>1%U3O8) showings in the basal sandstone; most of these are in the south. There are currently too few of these, and they are too far apart to make any reliable estimation of uranium resources in the sandstone. If additional drilling continues to encounter moderate to occasionally strong uranium mineralization near the base of the Athabasca, RPA recommends that an attempt be made to model this additional geologic domain. If the showings remain erratic and difficult to correlate hole to hole, then RPA recommends that the regions with such showings be delineated. When the open pit reaches these levels, more detailed mapping and in-pit sampling can be used to determine whether there are pods of mineralization in the basal sandstone that can be effectively segregated as ore.

The McClean North and South areas offer potential to identify further resources beyond those estimated in this report.

18 INTERPRETATION AND CONCLUSIONS

The MLJV projects outlined in this report represent significant economic sources of feed materials for the existing JEB processing facilities and, in conjunction with the Midwest Project described under separate cover, will support an operating life of at least 15 years producing in the order of 62.8 million pounds of U_3O_8 product. At the $23.00 per pound uranium price used in the economic analysis in this report, these projects are estimated to produce substantial positive operating cash flows.

Although there is a substantial volume of data and information available for the various deposits outlined in this report, RPA found that the information presented needed a significant amount of organizing, checking, and clarification. RPA spent a considerable amount of time and effort in the verification and confirmation process in order to confidently develop the estimates of Mineral Resources and Mineral Reserves outlined in this report. While no fatal flaws were uncovered in this process, RPA recommends that the MLJV implement more rigorous controls and procedures in the area of resource and reserves to ensure that all estimates are supported by detailed and explicit documentation. Due to the complex nature of the estimating process, clear documentation and preservation of the supporting data and analysis is critical to being able to understand the estimate.

RPA has found that there has been a significant under-estimation of uranium resources and reserves in some of the estimates prepared in the past for the MLJV due to the use of simple grade interpolation methods. RPA has evaluated and used a density weighted grade interpolation methodology that recognizes the impact of the heavy specific gravity with high grade uranium minerals. RPA believes that the estimates developed and presented here are better representations of the likely conditions in the deposits and RPA recommends that these methods and procedures be adopted in future Mineral Resource and Reserve estimates for the MLJV.

RPA has found that the resource modeling methodologies used in some of the past estimates based on Uniform Conditioning results in estimates that are difficult to check and confirm by physical examination and validation. RPA believes that the technical and operational staff will find it necessary to have physical representations and interpretations of the geology and mineralization in the deposit in order to effectively manage the mining process. The uniform conditioning methods do not rely upon and do not produce these sorts of products. While the methodologies may be mathematically correct, they are difficult to use in a practical context. RPA recommends that modeling and estimation programs that will ultimately be employed to support mining operations be carried out using more physically interpretive methods along the lines of the methods used by RPA in developing some of the estimates in this report.

RPA has estimated that the Sue E deposit hosts a significant amount of Inferred Mineral Resource. RPA believes that while this material has not been used in the economic analysis and determination of the Mineral Reserve for Sue E, it does represent potentially economic material. RPA recommends that additional diamond drilling be carried out in order to confirm the presence of additional mineralization.

RPA has estimated the Mineral Reserve at the McClean North deposits based on a Blind Shaft Boring mining method. RPA understands that the test program for the Hydraulic Borehole mining method is ongoing and ultimately this method may prove to be advantageous; however, at this time RPA believes that the Blind Shaft Boring method represents a method that utilizes existing proven technology and can recover portions of the McClean North deposits economically.

19 RECOMMENDATIONS

In the course of completing the review program outlined in this report, RPA has found that the MLJV has substantial and valuable mineral assets in the various deposits that have been identified and evaluated. Those assets have been the subject of extensive programs involving drilling, sampling, assaying, testing, and have produced an extensive amount of documentation and reports which in total represents a substantial investment. RPA found that documents were not easy to access and that not all of the known information could be retrieved. RPA recommends that the MLJV undertake a program of cataloguing and filing of the basic data to ensure ready and complete access to data in the future. In addition, RPA recommends that the MLJV implement rigorous controls and procedures in the area of mineral resource and reserve estimation designed to ensure that all estimates are supported by detailed and explicit documentation.

In some deposits RPA has found that past estimates appear to have underestimated uranium resources due to the use of simple grade interpolation methods as opposed to a density weighted grade interpolation methodology recognizing the impact of high specific gravities associated with high grade uranium mineralization. RPA recommends that the MLJV adopt density weighting methods for future Mineral Resource estimates.

RPA recommends that modeling and estimation techniques used to support mining operations be carried out using physical interpretations of geology to guide and control the estimates in order to facilitate reconciliation and monitoring during the mining phase.

RPA recommends that additional infill diamond drilling be carried out in the deeper and southern areas of the Sue E deposit in order to confirm the Inferred class mineralization interpreted by RPA and to upgrade this material to the Indicated category.

RPA recommends that the MLJV continue to pursue the Hydraulic Borehole Mining, and Blind Shaft Boring test program for the McClean North deposits, and potentially for application at the Caribou deposit.

RPA recommends that additional dry density measurements be made using core from other drill holes and that some mineralized samples from the base of the Athabasca sandstone be included, in order to expand the data base supporting the density values used in the Mineral Resource estimates.

RPA recommends that the MLJV advance the implementation of process modifications to treat elevated levels of Arsenic in ores being produced from some deposits. Similarly, RPA recommends that the MLJV pursue the evaluation of additional process modifications, and additions will be required to recover and realize the value from the Nickel and Cobalt that are contained in some of the deposits under consideration.

RPA recommends that the MLJV periodically update the economic evaluations of the deposits discussed in this report as additional deposit information becomes available through drilling and/or experience in the mine operation, as well as updating cost factors and uranium pricing levels.

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21 SIGNATURE PAGE

This report titled "Technical Report on the Denison Mines Inc. Uranium Properties, Saskatchewan Prepared for Denison Mines Inc." and dated February 16, 2006, was prepared and signed by the following authors:

(Signed & Sealed)

Dated at Toronto, Ontario
February 16, 2006

James W. Hendry, P. Eng.
Principal
Roscoe Postle Associates Inc.
Consulting Engineer

(Signed & Sealed)

Dated at Toronto, Ontario
February 16, 2006

Richard E. Routledge, M. Sc., P. Geol.
Roscoe Postle Associates Inc.
Consulting Geologist

22 CERTIFICATE OF QUALIFICATIONS

JAMES W. HENDRY

As an author of this report entitled "Technical Report on the Denison Mines Inc. Uranium Properties, Saskatchewan Prepared for Denison Mines Inc." (the Report) and on behalf of Denison Mines Inc. (Denison), I hereby make the following statements:

A. My name is James W. Hendry and I am Principal Mining Engineer with Roscoe Postle Associates Inc. My office address is 55 University Avenue, Suite 501, Toronto, Ontario M5J 2H7.

B. I am a Qualified Person for the purposes of National Instrument 43-101 of the Canadian Securities Administrators. I have received the following degree in Mining Engineering:

- B.Sc. in Mining Engineering from Queen's University, Kingston, Ontario.

C. I am registered as a Professional Engineer in the Province of Ontario. I am also a Member of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM).

D. My contributions to the Report are based on my personal review of technical reports provided by the Issuer, on discussions with the Issuer's representatives, and on information available in public files. I have been practising as a professional mining engineer for over 31 years. My relevant experience for the purpose of the Technical Report is:

- Review and evaluation of numerous open pit gold mine and other operations in Canada, the United States, Latin America, Russia and Southeast Asia.

- Due diligence review of ore body block modeling, preparation of open pit optimization, mine design, capital and operating cost forecasts for a nickel mine development in Brazil.

- Due diligence review of ore body block modeling, preparation of open pit optimization, mine design, capital and operating cost forecasts for base metal mines in Canada and copper-gold mines in Argentina and Peru.

E. I visited the McClean Lake operations on February 1 and 2, 2005 and the Cogema Resources Inc. offices in Saskatoon, Saskatchewan on January 31 and from February 2 to 4, 2005.

F. I am responsible for the overall preparation of the Report, including:

- Interpretation and Conclusions
- Recommendations

G. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

H. I am independent of Denison Mines Inc. applying the tests set out in section 1.5 of National Instrument 43-101. I have had no prior involvement with Denison Mines Inc. or with its properties in Saskatchewan.

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

(Signed & Sealed)

Dated at Toronto, Ontario
February 16, 2006

James W. Hendry, P.Eng.

RICHARD E. ROUTLEDGE

As an author of this report entitled "Technical Report on the Denison Mines Inc. Uranium Properties, Saskatchewan Prepared for Denison Mines Inc." (the Report) and on behalf of Denison Mines Inc. (Denison), I hereby make the following statements:

A. My name is Richard E. Routledge and I am a Consulting Geologist employed by Roscoe Postle Associates Inc. (RPA). My office address is Suite 501, 55 University Avenue, Toronto, Ontario M5J 2H7. I am a Qualified Person for the purposes of National Instrument 43-101 of the Canadian Securities Administrators.

B. I have received the following degrees:
   - B.Sc. (Major Geology) 1971 – Sir George Williams (now Concordia) University, Montreal, Quebec
   - M.Sc. (Applied Mineral Exploration) 1973 - McGill University, Montreal, Quebec

C. I am licensed as a Professional Geologist in the Northwest Territories and have applied for registration in the Association of Professional Geoscientists of Ontario. I am a Member of the Canadian Institute of Mining, Metallurgy and Petroleum, Toronto Branch.

D. I am a Qualified Person for the purposes of National Instrument 43-101.

E. This Report is based on my personal review of information provided by Denison, on discussions with Denison and Cogema Resources Inc. personnel, and on information available in public files. My relevant experience for the purpose of the Report is:
   - Staff Geologist and Associate, Derry, Michener and Booth Consulting Geologists and Engineers, 1973 to 1985
   - Evaluations and Special Projects Geologist with Teck Explorations Ltd., 1985 to 1992
   - Vice President Exploration, Greater Lenora Resources Ltd., 1993-1994
   - Consulting Geologist with RPA from 1994 to present

F. I have been practicing continuously as a professional geologist for 32 years.

G. I visited the McClean Lake operations on February 1 and 2, 2005 and the Cogema Resources Inc. offices in Saskatoon, Saskatchewan on January 31 and from February 2 to 4, 2005.

H. I am responsible for sections 7 to 16 of this Report as wells parts of:

   - Sections 1 to 5

- Other Relevant Data and Information
- Interpretation and Conclusions
- Recommendations

I. I am not aware of any material fact or material change with respect to the subject matter of the Report, which is not reflected in the Report, the omission to disclose which makes the Report misleading.

J. I am independent of Denison Mines Inc. applying the tests set out in section 1.5 of National Instrument 43-101. I have no prior involvement with the Denison Mines Inc. properties that are the subject of the Report.

K. I have read National Instrument 43-101 and National Instrument 43-101F1 and this Report has been prepared in compliance with both of these Instruments.

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

(Signed & Sealed)

Dated at Toronto, Ontario
February 16, 2006

Richard E. Routledge, M. Sc., P. Geol.