NORDMIN
RESOURCE & INDUSTRIAL
ENGINEERING

BENTON
RESOURCES INC.

NI 43-101 TECHNICAL REPORT on the Preliminary Economic Assessment

For the Cape Ray Property; 04, 41, 51 and Windowglass Hill Deposits

Isle aux Morts Area, Newfoundland and Labrador, CANADA

Located at:

58° 55’ 47” W Longitude

47° 45’ 03” N Latitude

PREPARED BY:

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Effective Date : March 7, 2016

Signing Date : April 21,2016

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List of Units and Abbreviations

Abbreviation	Meaning
°C	Degrees Celsius
$ and CAD$	currency of Canada
AA	atomic absorption (assay method to measure metal content)
Ag	silver
Au	gold
BP	Big Pond
BRI	Benton Resources Inc.
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CRFZ	Cape Ray Fault Zone
CRGD	Cape Ray Gold Deposit
CRIC	Cape Ray Igneous Complex
Cu	copper
CPUE	Catch-per-unit-effort
DDH or ddh	diamond drill hole
DFO	Department of Fisheries and Oceans
DMS	Dense Media Separation
EA	Environmental Assessment
EIS	Environmental Impact Statement
el	elevation
EM	electromagnetic
EPR	Environmental Permitting Regulations
g	gram
g/t	grams per tonne
GBC	Grand Bay Complex
GRG	Gravity Recoverable Gold
ha	hectare
HMY	High Mass Yield
IaM	Isle aux Morts
ICP	Inductively coupled plasma (geochemical test method)
IK	Indicator Kriging (block model type)
IP	induced polarization (geological survey method)
JV	joint venture
km	kilometre
kg	kilogram
kg/t	Kilograms per tonne
LCS	Local Coordinate System
LOI	Letter of Intent
LCM	Loose Cubic Metre
m	metre

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Abbreviation	Meaning
mm	millimetre
µm	micrometre
Ma	millions of years
Mt	millions of tonnes
N	north
NN	Nearest neighbour (block model type)
NL	Newfoundland
NSR	net smelter return
NTS	national topographic system
NW	northwest
oz.	Troy Ounces (1 troy oz. = 31.1034 g)
P. Eng.	professional engineer
P. Geo.	professional geoscientist
PABG	Port aux Basques Gneiss
PEA	Preliminary Economic Assessment
Pb	lead
PMA	Particle Mineral Analysis
PMR	Potential Minable Resource
ppm	parts per million
QP	Qualified Person
S	south
S	sulphur
SE	southeast
SG	Specific Gravity
SMU	Smallest mining unit
SW	southwest
t	tonne (metric)
UTM	Universal Transverse Mercator
VLF-EM	very low frequency electromagnetic survey
W	west
WCS	World Coordinate System
WDH	Windowglass Hill
WGHD	Windowglass Hill Deposit
WGHG	Windowglass Hill Granite
WPG	Windsor Point Group
Zn	zinc

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1. Summary

This Preliminary Economic Assessment (PEA) was prepared for Benton Resources and Nordmin Engineering Ltd. for the Cape Ray Gold-Silver Property, located in southwestern Newfoundland, Canada. This report focuses on the deposits within the joint venture agreement known as Cape Ray 04, 41, 51 and Windowglass Hill.

Benton acquired 100% interest in the Cape Ray Property on October 3, 2014 and entered in to a joint venture agreement with Nordmin on January 20, 2015. The property includes historical royalty agreements which are outlined in Section 4 “Property Description and Location”.

There are adjoining deposits known as Iles aux Morts and Big Pond which are held solely by Benton.

This PEA was prepared by independent consultants, Alan Aubut, P.Geo. of Sibley Basin Group Geological Consulting Services Ltd. (Sibley Basin Group), John Folinsbee P.Eng. of Heads Ore Tails Metallurgy and Paul Bonville P.Eng. of PRB Mining Services on behalf of the joint venture agreement between Benton Resources Inc. (Benton) and Nordmin Engineering Ltd. (Nordmin) and has an effective date of March 7, 2016.

This PEA is based on an updated mineral resource estimate completed by Sibley Basin Group Ltd

The Cape Ray Property is located approximately 20 km north east of the town of Channel-Port aux Basques, Newfoundland and is shown below in Figure 1.1.

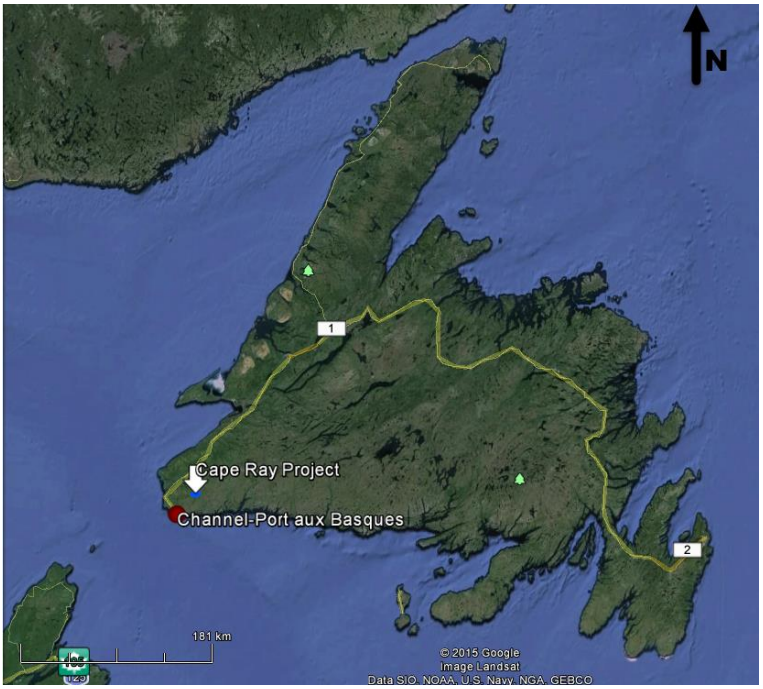

Figure 1.1 Project Location in Newfoundland (Google Earth)

The 04, 41, 51 and Windowglass Hill (WGH) deposits shown in Figure 1.2 are located approximately at 58° 55' 47" W Longitude and 47° 45' 3" N Latitude. The Isles aux Morts River flows through the property and separates WGH deposit from the others and then flows south from the deposits into the ocean.

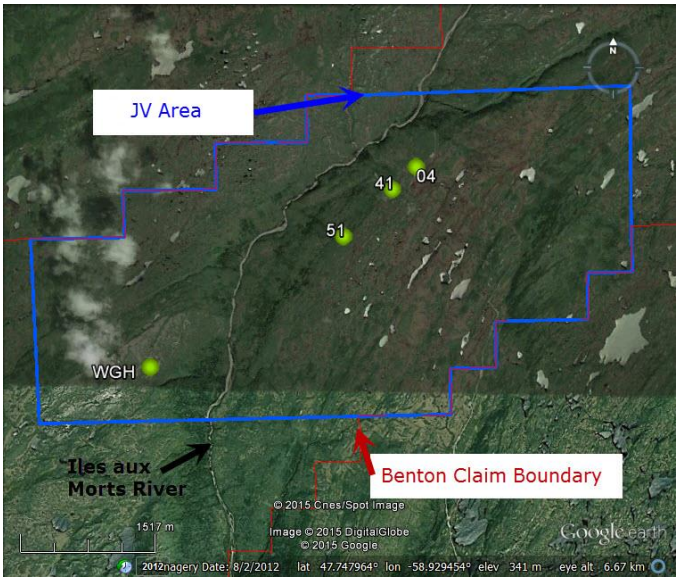

Figure 1.2 Deposit Locations within Claim Boundary (Google Earth)

The Cape Ray Fault Zone is a major structural boundary contained within the licence boundary of the Cape Ray property. It separates the Dunnage Zone to the northwest and the Gander Zone to the southeast. The Fault Zone is approximately 100 km long and up to 1 km wide, extending from Cape Ray in the southwest to Granite Lake to the northeast.

Areas along and adjacent to the southwest portion of the Cape Ray Fault Zone have been subdivided into three major geological domains. From northwest to southeast they include: the Cape Ray Igneous Complex, the Windsor Point Group and the Port aux Basques gneiss. These units are intruded by several pre- to late-tectonic granitoid intrusions. The Cape Ray Igneous Complex comprises mainly large mafic to ultramafic intrusive bodies that are intruded by granitoid rocks. Unconformably overlying the Cape Ray Igneous Complex is the Windsor Point Group, which consists of bimodal volcanics and volcanoclastics with associated sedimentary rocks. The Port aux Basques gneiss is a series of high grade, kyanite-sillimanite-garnet, quartzofeldspathic pelitic and granitic rocks intercalated with hornblende schist or amphibolite.

Hosted by the Cape Ray Fault Zone are the Cape Ray Gold Deposits consisting of three main mineralised zones: the 04, the 41 and the 51 Zones. These occur as quartz veins and vein arrays along a 1.8 km segment of the fault zone at or near the tectonic boundary between the Windsor Point Group and the Port aux Basques gneiss. The gold bearing quartz veins are typically located at or near the southeast limit of a sequence of highly deformed and brecciated graphitic schist. Other veins are present in the structural footwall and represent secondary lodes hosted by more competent lithologies.

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1.1 Resource

This PEA on the indicated and inferred resource estimates were completed for the 04, 41, 51 and Windowglass Hill deposits using a 4.0 g/t cut-off for resources potentially minable using underground mining methods and a 1.0 g/t cut-off for resources potentially minable using open pit mining methods. A summary of the resource estimates by deposit are shown below in Table 1.1 and Table 1.2.

Table 1.1 The Estimated Resources for the Cape Ray 04, 41, 51, and the Windowglass Hill Deposits at a Gold cut-off grade of 4.0 g/t

Deposit	Classification	Tonnes (Mt)	Au g/t	Ag g/t	Cu %	Pb %	Zn %	Contained Au oz.	Contained Ag oz.
04	Measured Resources								
04	Indicated Resources	0.46	6.97	16.93	0.11	0.59	0.54		
04	Meas. & Indicated Resources	0.46	6.97	16.93	0.11	0.59	0.54	103,000	250,000
04	Inferred Resources	0.08	6.41	12.11	0.07	0.42	0.35	16,500	31,000
41	Measured Resources								
41	Indicated Resources	0.13	6.34	20.05	0.17	0.56	0.32		
41	Meas. & Indicated Resources	0.13	6.34	20.05	0.17	0.56	0.32	26,500	84,000
41	Inferred Resources	0.04	5.83	15.02	0.13	0.56	0.38	7,500	19,000
51	Measured Resources								
51	Indicated Resources								
51	Meas. & Indicated Resources								
51	Inferred Resources	0.4	8.39	16.34	0.04	0.14	0.07	108,000	210,000
WGH	Measured Resources								
WGH	Indicated Resources								
WGH	Meas. & Indicated Resources								
WGH	Inferred Resources	0.07	5.63	22.22				12,500	50,000

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Table 1.2 The Estimated Resources for the Cape Ray 04, 41, 51 and Windowglass Hill Deposits at a Gold cut-off grade of 1.0 g/t

Deposit	Classification	Tonnes (Mt)	Au g/t	Ag g/t	Cu %	Pb %	Zn %	Contained Au oz.	Contained Ag oz.
04	Measured Resources								
04	Indicated Resources	1.58	3.45	11.07	0.06	0.34	0.31		
04	Meas. & Indicated Resources	1.58	3.45	11.07	0.06	0.34	0.31	175,000	562,000
04	Inferred Resources	1.10	2.29	4.00	0.03	0.24	0.16	81,000	141,500
41	Measured Resources								
41	Indicated Resources	0.6	2.85	9.58	0.08	0.27	0.16		
41	Meas. & Indicated Resources	0.6	2.85	9.58	0.08	0.27	0.16	55,000	185,000
41	Inferred Resources	0.13	3.18	9.49	0.07	0.32	0.22	13,000	39,500
51	Measured Resources								
51	Indicated Resources								
51	Meas. & Indicated Resources								
51	Inferred Resources	0.88	5.08	11.82	0.04	0.12	0.06	144,000	334,500
WGH	Measured Resources								
WGH	Indicated Resources								
WGH	Meas. & Indicated Resources								
WGH	Inferred Resources	1.80	1.38	8.49				80,000	491,000

A) The mineral resources are estimated in conformance with the Canadian Institute of Mining, Metallurgy and Petroleum – Standards on Mineral Resources and Reserves – Definitions and Guidelines, as referenced in NI 43-101

B) Assay composites were capped at 40 g/t Au.

C) A density value of 2.84 g/cm^3 was used for all deposits.

D) The rounding of tonnes as required by NI 43-101 reporting guidelines may result in apparent differences between tonnes, grade and contained ounces.

E) Mineral resources that are not mineral reserves do not have demonstrated economic viability. This estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing and other relevant issues.

F) The quantities and grades of reported inferred mineral resources are uncertain in nature and further exploration may not result in their upgrading to Indicated or Measured status.

G) Mineral resources are reported unconstrained by any pit shell for surface mining methods or any preliminary underground mining outline for underground methods.

The reader should be cautioned that the PEA is preliminary in nature. It contains inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the results of the PEA will be realized.

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1.2 Preliminary Economic Assessment

The potential minable resource (PMR) for this PEA was estimated at 1,700,MT averaging a combined grade of 4.6 g/t gold and 4.8 g/t silver. Open pit minable resource consists of 840,000 tonnes with anticipated dilution of 10% and mining loss of 5%. The remaining 860,000 minable resource tonnes from underground methods include dilution of 20% in longhole stoping and 10% in cut and fill with mine loss of 5% applied.

Mill recovery rates based on preliminary tests are estimated to be 97% for gold resulting in approximately 250,000 ounces recoverable gold and 45% for silver resulting in approximately 260,000 ounces recoverable silver.

All costs and revenues are in Canadian dollars unless otherwise stated

Table 1.3 Cape Ray combined open-pit and underground mine key economic assumptions and results

	Unit	Value
Total mineralized rock mined	kt	1,700
Gold grade	g/t	4.6
Silver grade	g/t	4.8
AuEq grade	g/t	4.62
Gold recovery	%	97
Silver recovery	%	45
Gold price	US$/oz.	1,200
Silver price	US$/oz.	14
Exchange rate	US$/CAD$	1.25
Payable gold metal	oz.	250,000
Payable silver metal	oz.	260,000
Total net revenue	$m	378
Total capital costs (Project and Sustaining)	$m	84.9
Overall Operating costs (total)	$/t	120.3
Overall Operating costs (AuEq)	US$/oz. AuEq	647.6
Payback period	years	3
Pre-tax cumulative net cash flow	$m	57.6
Pre-tax NPV (7%)	$m	48.4
Pre-tax IRR	%	29
Post-tax NPV (7%)	$m	32.6
Post-tax IRR	%	24

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1.3 Capital and Operating Costs

The Cape Ray Project has been envisioned as a combined open-pit and underground mining operation. Open-pit and underground mining are anticipated to be completed by contract mining companies. The equipment would be supplied by the contractor that is awarded the work.

Grid electrical power would provide the majority of the electrical power to the project over the life of the mine. The work force is expected to come from the Isle aux Morts area for the operation of the Mill. The rest of the workforce will be the responsibility of the contractor.

Costs are based on first principles, recent price quotes and recent projects with similarity in method and quantity.

Table 1.4 Total estimated sustaining and capital expenditures

Expenditures	**$**
Deposit	
Pit 41	560,000
U.G. 04	16,500,000
Windowglass Hill	3,100,000
U.G. 51	9,600,000
Other (41 U.G. / 51 Trench)	3,800,000
Total Sustaining Capital	**33,560,000**
Permitting	2,700,000
Road Work	700,000
Overburden Removal	500,000
Surface Infrastructure – Open Pit/Mill	2,500,000
Material and Waste Pads (3) – Mine & Mill	500,000
Dewatering/Trench/Sumps	300,000
Processing Plant	30,800,000
Tailings	3,500,000
Surface Equipment	800,000
Water Treatment Plants/Testing	2,000,000
Power Distribution to Mill	1,200,000
Working Capital	600,000
Engineering for Capital	800,000
Mine Closure (Paid by Asset Sales)	4,400,000
Total Capital	**51,300,000**
Total of Sustaining and Capital	**84,860,000**

1.4

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1.5 Production and Processing

Operations for the Cape Ray project is planned to have combined Open Pit and Underground Mining. Each deposit could be campaigned separately with use of contractors. The initial estimated mill feed would be 500,000 tonnes from 41 Deposit open pit operations. The underground mining operation at the 04 Deposit could produce approximately 500,000 tonnes.

Once the 41 Deposit mining is completed, there could be a small underground resource below the pit containing the remaining potentially economic PMR. The second open pit operation could produce 330,000 tonnes from the Windowglass Hill deposit. The 51 underground deposit could commence after the 04 Deposit mining is completed and could produce 350,000 tonnes. Section 16 includes a schedule of the mined and processed tonnes.

The envisioned process plant includes conventional crushing, grinding, gravity, and cyanide leach. A gold and silver doré would be produced on site. Process reagents would be removed from the plant tailings prior to placement in a tailings management facility.

Table 1.5 Mining Costs

Cash Cost Summary	Cost (C$) millions	C$/tonne
Overall Global Mining Costs	$205	$120
Open Pit and U.G Mine Costs	$160	$94
Processing Costs	$45	$26.4
G & A	$0.8	$0.5

1.6 Conclusions

The project provides significant positive and robust returns. Significant increase in project IRR can be achieved (indicated by sensitivity analysis) through reducing capital expenditures by 10 to 20%. This savings could be realized in part by sourcing a used processing plant and/or used processing equipment. Used processing plant equipment is still available, although the market has contracted and careful due diligence on equipment is required. Savings of up to 20% in the processing plant capital costs may be realized. The potential for custom milling of potentially economic mineralization at processing plants in the region could also improve project returns as capital expenditures for a plant and tailings management area would be significantly reduced.

Table 1.6 Expected project returns

	Pre-Tax	After-Tax
Net Revenue	$378M	$378M
Undiscounted Cash-flow	$88.4M	$63.4M
NPV (7%)	$48.4M	$32.6M
NPV (10%)	$36.7M	$23.6M
IRR	29%	24%
Payback Period	3 years	
Life of Mine	6 years	

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Table 1.7 Project return sensitivity analyses

Variable	Pre-Tax NPV @ 7% ($millions)	IRR%	After-Tax NPV @ 7% ($millions)	IRR %
Gold Price - $1,400 U.S.	90	44	62.3	35
Gold Price - $1,000 U.S.	7	11	3	9
Silver Price - $17.00 U.S.	49	29	33	24
Silver Price - $11.00 U.S.	47.8	29	32	24
Exchange Rate = $1.33	65	35	44.6	29
Exchange Rate = $1.18	33.6	23	22	19
Capital Costs +20%	40.3	24	26.5	19
Capital Costs -20%	54	32	36.3	26
Sustaining Capital Costs +20%	43.3	27	29	22
Sustaining Capital Costs -20%	56.5	36	39	30
Operating Costs +20%	22	18	13.7	15
Operating Costs -20%	75	39	51.7	32

1.7 Recommendations

Based on the positive results of this PEA the recommendations are:

- Additional diamond drilling should be done so existing Inferred Resources may be upgraded to Indicated Resources status as defined by NI 43-101.
- Incorporate oriented core (Geotechnical) with above drilling for rock mechanics engineering.
- Hydrology drilling at a pre-feasibility level.
- Advancing the Cape Ray project toward prefeasibility through additional exploration, environmental permitting and further studies.

It is proposed that a two phase approach be implemented such that results from Phase 1 can be the baseline for Phase 2.

Phase 1 consists of drilling down to the 150 m level and beginning environmental baseline studies. The environmental estimate includes all labor, equipment and materials to carry out hydrology and surface water baseline studies.

Phase 2 includes deeper drilling to provide data for the geological model down to the 300 m level as well as some infrastructure upgrades and continued environmental work. The access road into the property requires some cut and fill work to reduce steep sections and a crushed rock cap to build a passable single lane road.

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A proposed budget for the recommended program is presented in Table 1.8.

Table 1.8 Proposed Budget (in thousands)

Phase1	**Estimated Cost (CAD$)**
Drilling down to 150 m level plus infill drilling	$4,175
Environmental baseline studies, hydrology and oriented core*	$660
Contingency (10%)	$483
Subtotal	**$5,318**
Phase 2	
Drilling down to 300 m level plus infill drilling and oriented core	$7,350
Upgrade to road infrastructure	$560
Updated mineral resource estimate and Prefeasibility study*	$400
Contingency (10%)	$831
Subtotal	**$9,141**
Total	**$14,460**

*Provided by Nordmin Engineering in conjunction with existing JV agreement.

Note: Estimated costs are inclusive of drilling rates, stand by and work rates, camp support, consumables, laboratory charge and supervision.

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2. Introduction

2.1 Terms of Reference

This Preliminary Economic Assessment (PEA) and Technical Report was prepared for Benton Resources and Nordmin Engineering Ltd. to provide a Preliminary Economic Assessment on the mineral resources for the Cape Ray Gold-Silver Property, located in southwestern Newfoundland, Canada. It was prepared by independent consultants: Alan Aubut, P.Geo., of Sibley Basin Group Geological Consulting Services Ltd. (Sibley Basin Group), John Folinsbee P.Eng. of Heads Ore Tails Metallurgy and Paul Bonneville P.Eng. of PRB Mining Services on behalf of the joint venture agreement between Benton Resources Inc. (Benton) and Nordmin Engineering Ltd. (Nordmin). The Cape Ray Gold project deposits of subject for this PEA report include 04, 41, 51 Zone deposits and Windowglass Hill deposit.

This PEA is based on an updated mineral resource estimate completed by Sibley Basin Group Ltd.

As of the date of this Report, BRI is a Canadian junior exploration company and is listed on the Canadian TSX Venture stock exchange (BEX) with a corporate office at:

3250 Hwy 130
Rosslyn, Ontario,
P7K 0B1
Tel: 807-475-7474
Fax: 807-475-7200
This Report is considered effective as of March 7, 2016.

This PEA report is intended to be used by Benton Resources and Nordmin Engineering Ltd. subject to the terms and conditions of their joint venture agreement between Benton Resources Inc. (Benton) and Nordmin Engineering Ltd. (Nordmin). This permits Benton to file this report on SEDAR (www.sedar.com) as an NI 43-101 Technical Report with the Canadian Securities Regulatory. Authorities pursuant to provincial securities legislation. The authors understand that Benton Resources and Nordmin Engineering Ltd may use the report for a variety of corporate purposes including financings. Except for the purposes legislated under provincial securities laws, any other use of this report, by any third party, is at that party's sole risk.

2.2 Sources of Information

This Report is based, in part, on internal company technical reports and maps, published government reports, company letters and memoranda, public information, documented results concerning the project and discussions held with technical personnel from the company regarding all pertinent aspects of the project as listed in Section 27 References of this report. Several sections from reports authored by other consultants have been directly quoted in this report, and are so indicated in the appropriate sections.

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Nordmin has not conducted detailed land status evaluations, but has obtained tenure information from previous technical reports, public documents and statements by BRI regarding property status and legal title to the project.

2.3 Site Visits

Mr. Alan Aubut, P. Geo., of Sibley Basin Group Geological Consulting Services Ltd. (Sibley Basin Group), a qualified person under the terms of NI 43-101, conducted a site visit to the Cape Ray Property on September 22, 2014. A site visit to the Pasadena core shacks was conducted on September 23, 2014.

No other author has visited the property in preparation of this report.

2.4 Units and Currency

Unless otherwise stated:

- All units of measurement in the Report are in the metric system
- All currency amounts in this Report are stated in Canadian dollars ("CAD"), unless otherwise stated
- Gold (Au) and Silver (Ag) assay values are reported in grams per tonne (g/t), unless otherwise stated
- Copper (Cu), Lead (Pb), and Zinc (Zn) assay values are reported in percent (%), unless otherwise stated
- All metal prices are expressed in terms of US dollars ("USD")
- All cost estimates have a base date of the first ("Q1") quarter of 2015.
- Grid coordinates for the Block Models are given in the UTM NAD 27 Zone 21. Maps are either in UTM coordinates or in the latitude/longitude system.

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3. Reliance on Other Experts

This technical report was prepared for Nordmin Engineering Ltd. for the properties included in the joint venture agreement between Nordmin Engineering Ltd. and Benton Resources Inc. Portions of this report rely on information previously reported in technical documents filed on SEDAR and have all been noted in section 27 References.

Table 3.1 Sections for which each Author takes responsibility for

Section	Title	Qualified Person	Company
01	Summary	Alan Aubut, P.Geo	Sibley Basin Group
02	Introduction	Alan Aubut, P.Geo	Sibley Basin Group
03	Reliance on Other Experts	Alan Aubut, P.Geo	Sibley Basin Group
04	Property Description and Location	Alan Aubut, P.Geo	Sibley Basin Group
05	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Alan Aubut, P.Geo	Sibley Basin Group
06	History	Alan Aubut, P.Geo	Sibley Basin Group
07	Geological Settings and Mineralization	Alan Aubut, P.Geo	Sibley Basin Group
08	Deposit Types	Alan Aubut, P.Geo	Sibley Basin Group
09	Exploration	Alan Aubut, P.Geo	Sibley Basin Group
10	Drilling	Alan Aubut, P.Geo	Sibley Basin Group
11	Sample Preparation, Analyses and Security	Alan Aubut, P.Geo	Sibley Basin Group
12	Data Verification	Alan Aubut, P.Geo	Sibley Basin Group
13	Mineral Processing and Metallurgical Testing	John Folinsbee P.Eng.	Heads Ore Tails Metallurgical Consulting
14	Mineral Resource Estimate	Alan Aubut, P.Geo	Sibley Basin Group
15	Mineral Reserve Estimate	Paul Bonneville, P.Eng.	PRB Mining Services
16	Mining Methods	Paul Bonneville, P.Eng.	PRB Mining Services
17	Recovery Methods	John Folinsbee, P.Eng.	Heads Ore Tails Metallurgical Consulting
18	Project Infrastructure	Paul Bonneville, P.Eng.	PRB Mining Services
19	Market Studies and Contracts	Paul Bonneville, P.Eng.	PRB Mining Services
20	Environmental Studies, Permitting and Social or Community Impact	Alan Aubut, P.Geo	Sibley Basin Group
21	Capital and Operating Costs	Paul Bonneville, P.Eng.	PRB Mining Services
22	Economic Analysis	Paul Bonneville, P.Eng.	PRB Mining Services
23	Adjacent Properties	Alan Aubut, P.Geo	Sibley Basin Group
24	Other Relevant Data and Information	Alan Aubut, P.Geo	Sibley Basin Group
25	Interpretation and Conclusion	Alan Aubut, P.Geo	Sibley Basin Group
26	Recommendation	Alan Aubut, P.Geo	Sibley Basin Group
27	References		

An independent verification of land title and tenure was not performed. The authors are relying on Benton's legal council to have verified the legality of any underlying agreement(s) that may exist concerning the licences or other agreement(s) between third parties.

Benton has reviewed a draft copy of this report for factual errors. The authors are relying on their historical and current knowledge of the Property. Any changes made as a result of the review did not involve any alteration to the conclusions made, therefore the statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this report.

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4. Property Description and Location

4.1 Description of Area and Location

The Cape Ray Property is located approximately 25 km north east of the town of Channel-Port aux Basques, Newfoundland and is shown below in Figure 4.1.

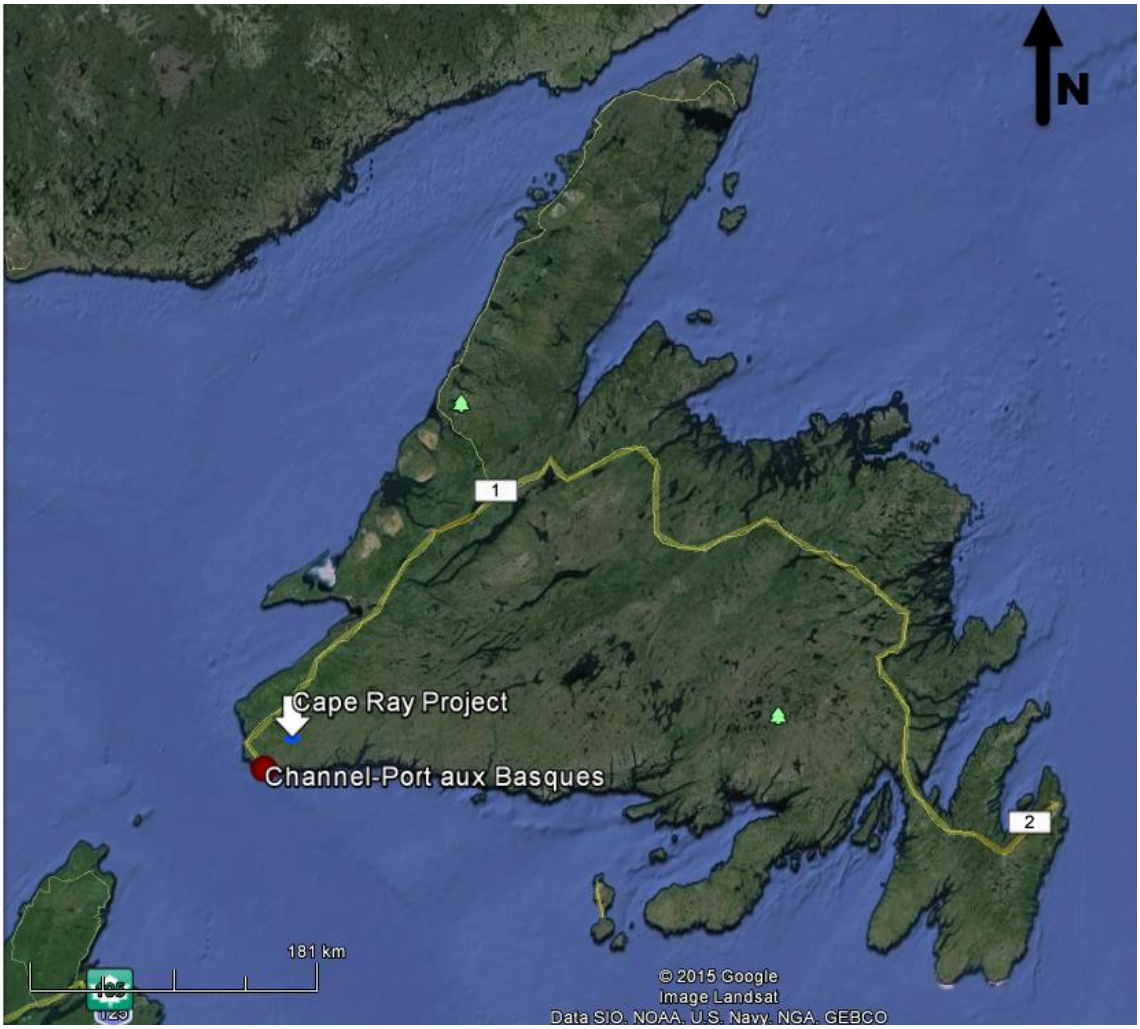

Figure 4.1 Project Location in Newfoundland (Google Earth)

The property covers portions of 1:50,000 NTS map sheets 11O/10, 11O/11, and 11O/15. Figure 4.2 shows the NTS maps in the area of the Cape Ray Project.

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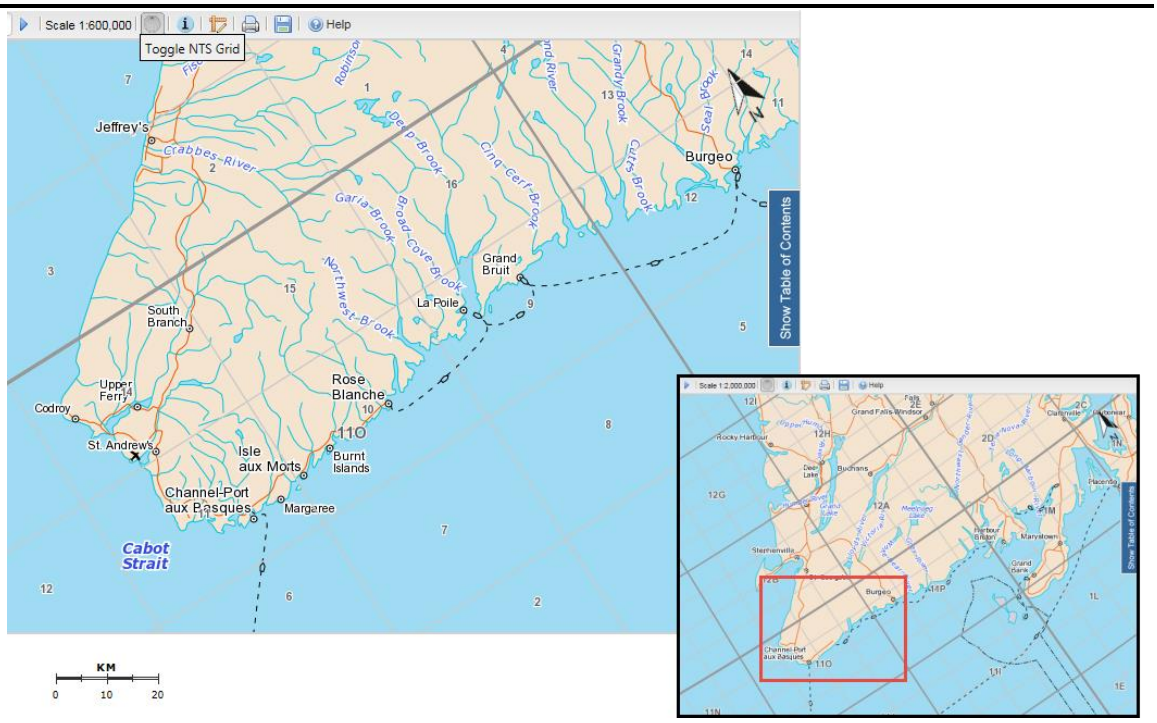

Figure 4.2 NTS Map Numbers in the Cape Ray Project Area (NRCAN)

The property is host to the 04, 41, 51 and Windowglass Hill (WGH) deposits shown in Figure 4.3. They are located approximately at 58° 55' 47" W Longitude and 47° 45' 3" N Latitude (UTM Zone 21 NAD 27 355300 m E, 5290200 m N). The Isles aux Morts River flows through the property separating the WGH deposit from the other deposits then flows south into the ocean.

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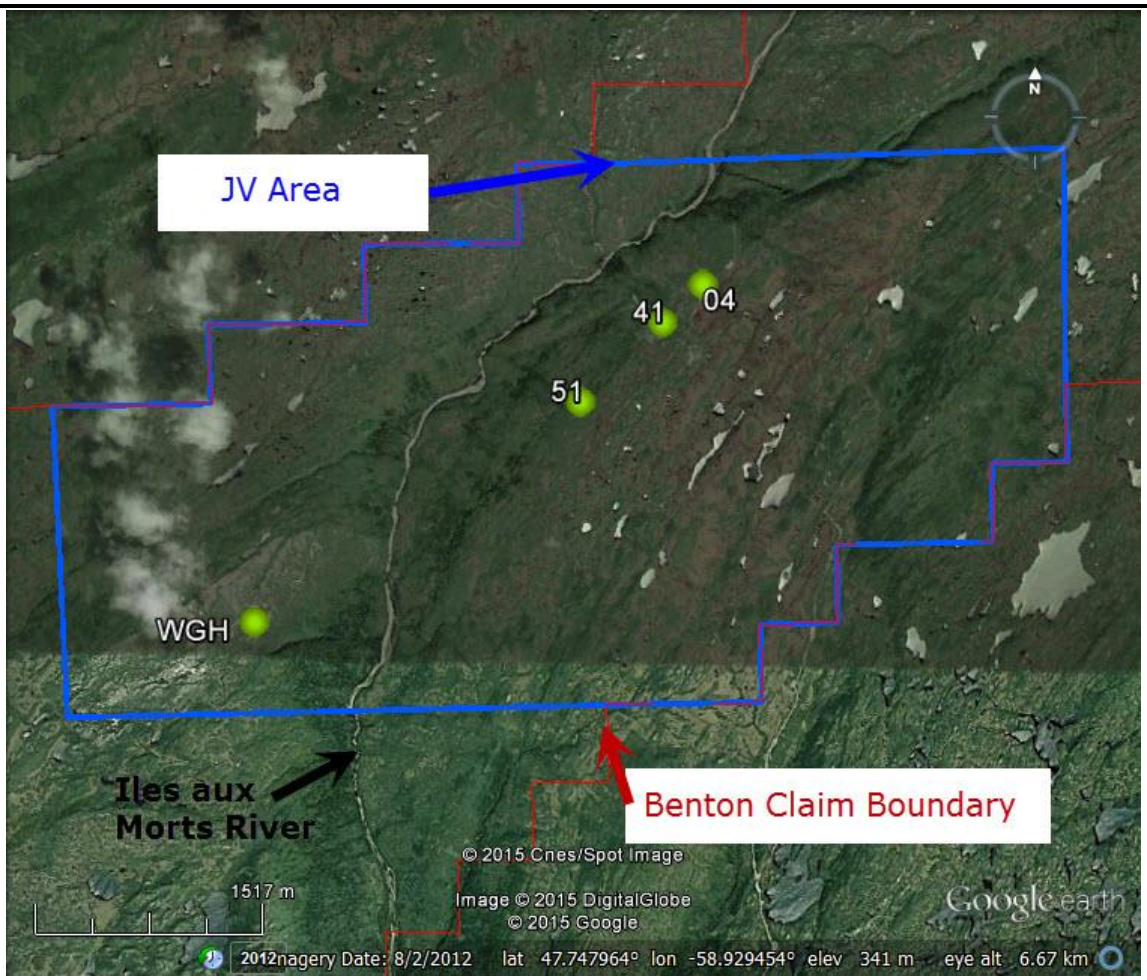

Figure 4.3 Deposit Locations within Claim Boundary (Google Earth)

4.2 Land Tenure

As of December 11, 2015 Benton is the holder of 1169 claims in Newfoundland and Labrador. In the Cape Ray area Benton holds approximately 355 claims.

Benton acquired 100% interest in the Cape Ray Property on October 3, 2014 and entered into a joint venture agreement covering a portion of the property with Nordmin Engineering on January 20, 2015. The claims in the Cape Ray area cover the deposits which are a part of the Nordmin joint venture (JV) as shown in Figure 4.4 below.

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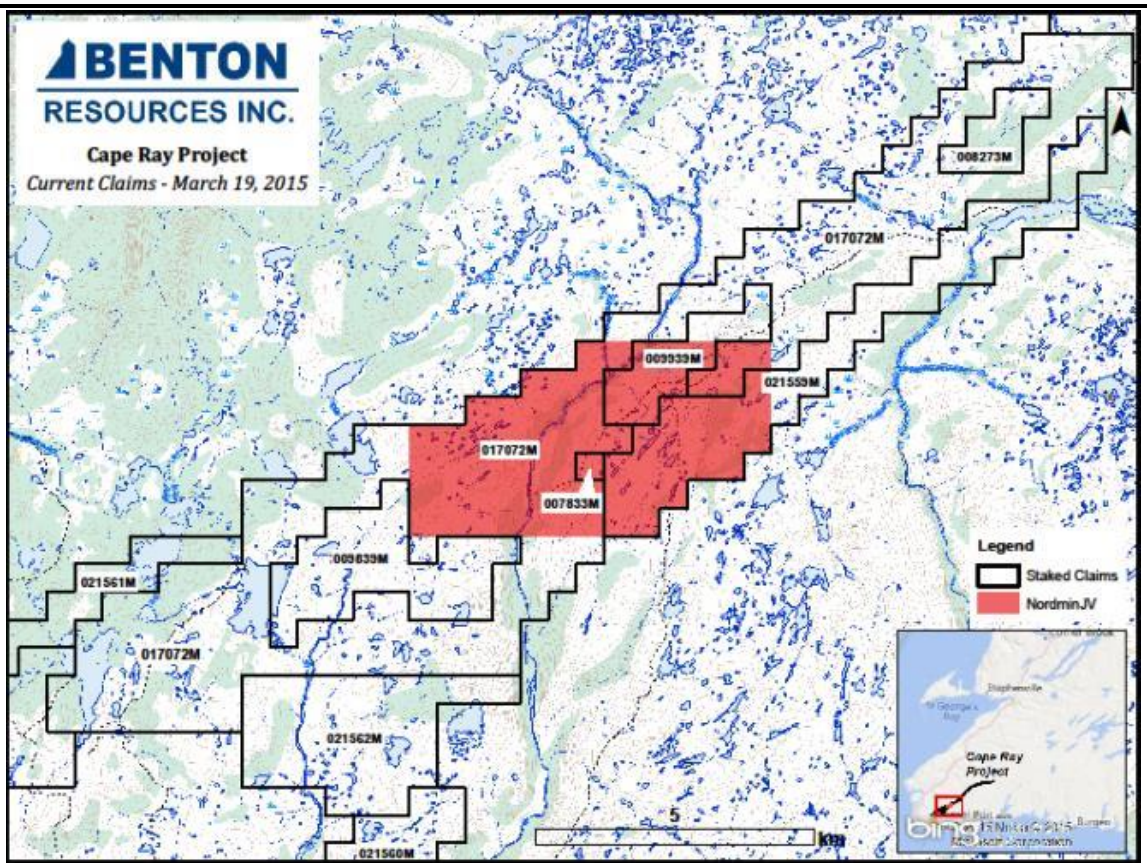

Figure 4.4 Nordmin-Benton JV Zone within Cape Ray Claims

A listing of the Cape Ray claims and their associated royalties are below in Table 4.1.

Table 4.1 Cape Ray Claims held by Benton Resources Inc. as of December 11, 2015

Licence #	Known Deposit	# of Claims	Area (km^2)	Royalty
017072M	Windowglass Hill (WGH) and 51	183	45.70	(a) & (b)
021560M	-	14	3.50	none
020641M	-	10	2.50	none
021561M	-	16	4.00	none
021559M	-	44	10.99	none
007833M	-	1	0.25	none
021562M	-	34	8.49	none
008273M	Isle aux Morts (IaM)	7	1.75	(c)
009839M	Big Pond (BP)	26	6.49	(c)
009939M	04 and 41	12	3.00	(c)
022680M	-	8	2.00	none

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a) 1.75% net smelter returns royalty ("NSR") held by Alexander J. Turpin pursuant to the terms of an agreement dated June 25, 2002, as amended February 27, 2003 and April 11, 2008. The agreement between Alexander J. Turpin, Cornerstone Resources Inc. and Cornerstone Capital Resources Inc., of which 1.0% NSR can be repurchased for $1,000,000 reducing such royalty to a 0.75% NSR. The agreement which royalty applies to Licences 14479M, 17072M, 9338M, 9339M and 9340M covering 229 claims, all as described in the foregoing agreements.
b) 0.25% net smelter returns royalty ("NSR") held by Cornerstone Capital Resources Inc. and Cornerstone Resources Inc. (collectively the "Royalty Holder") pursuant to the terms of an agreement dated December 19, 2012, as amended June 26, 2013, between the Royalty Holders and Benton, which royalty applies to Licence 017072M, as described in the foregoing agreement.
c) Sliding scale net smelter returns royalty ("NSR") held by Tenacity Gold Mining Company Ltd. pursuant to the terms of an agreement dated October 7, 2013 with Benton Resources Inc.
    I. 3% NSR when the quarterly average gold price is less than US$2,000 per ounce (no buy-down right);
    II. 4% NSR when the quarterly average gold price is equal to or greater than US$2,000 per ounce but less than US$3,000 per ounce with the right to buy-down the royalty from 4% to 3% for Cdn$500,000; and
    III. 5% NSR when the quarterly average gold price is equal to or greater than US$3,000 per ounce with the right to buy-down the royalty from 5% to 4% for CAD$500,000; On Licences 7833M, 8273M, 9839M and 9939M as described in Schedule C of the foregoing agreement.

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4.2.1 Nordmin Benton Joint Venture Agreement

On January 20, 2015, Benton Resources entered into a joint venture agreement with Nordmin Engineering, pursuant to which Nordmin will earn a 50% interest in the Cape Ray Project through a series of expenditures and services to be provided. Summarized below are the highlights of the Agreement.

Under the terms of the Agreement, Benton will form a 100% wholly owned subsidiary, ("SubCo B" for illustrative purposes) and transfer 100% of Benton's property rights in the Project to SubCo B. Nordmin may then earn up to 50% of the Project held by SubCo B by completing a series of work commitments and project milestones which is well underway advancing the Project towards production as follows:

- Benton will lead the exploration effort for the project which will be funded by Nordmin up to the completion of the Feasibility Study. This includes any infill drilling to allow the Projects to be brought up to National Instrument 43-101 status ("NI 43-101"). The necessity for further exploration will be determined and approved by the management committee;
- Nordmin will, earn a 5% interest, make a $250,000 cash payment in two tranches to SubCo B which will go towards the current exploration program;
- Nordmin will, earn a further 15% interest, fund and provide the services required to complete NI 43-101 resource estimates for the 04, 41, 51, and Windowglass Hill deposits, produce a resource model, a PEA, complete a detailed assessment of the Geology, Mining, Metallurgy, Environmental, Engineering, Construction, Economics and Schedule for the Project;
- Nordmin will, earn a further 10% interest, complete the Environmental Assessment and Impact reviews, and secure the necessary permits for a mine, mill and related plant in order for the Project to move forward;
- Nordmin will, earn a further 10%, complete a feasibility study for the Project (detailing and advancing all of the same issues within the PEA);
- Nordmin will, earn a further 10% interest, complete detailed design of the mine, mill and related plant and work to arrange a minimum of 50% of the Project financing;
- Nordmin will assume operatorship of the Project upon signing.

4.3 Environmental Liabilities and Other Risks

Mr. John Sferrazza, Environmental Scientist and Director of Environmental Services for Nordmin visited the site on September 5th and 6th 2014, and also on several occasions in 2015 during the trenching and exploration program. No obvious environmental liabilities were noted during the visit other than the flooded underground workings from the previous owner. Benton is also not aware of any current liabilities.

At the time of the visit, water was pooled in front of the adit from the previous mine workings and was reporting to a series of collection ditches and holding ponds prior to discharging to the Isle aux Morts River. Water samples were taken from the portal mouth as well as from the

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collection ditches and ponds. More detailed water analysis is presented in Section 20 and show that and show that the current water quality meets Newfoundland and Labrador Regulation 65/03 wastewater quality criteria as well as most water quality criteria established in the Canadian Water Quality Guidelines for the protection of Aquatic Life. Further water quality surveys of the surrounding watersheds were conducted in 2015 as part of the environmental monitoring program conducted during the exploration efforts. Should Nordmin and Benton proceed with developing the mineral resource on the property, environmental baseline studies will be undertaken for environmental assessment, permitting and closure plan requirements.

Environmental risks include the site's proximity to the Isle aux Morts River which is a Scheduled Salmon River by the Newfoundland and Labrador Department of Natural Resources.

4.4 Permits Required

Permits that will potentially be required for exploration work include license to operate a temporary work camp in accordance with Section 10 of the Lands Act (1991) issued by the Department of Municipal and Intergovernmental Affairs, a Surface Lease and Mineral Exploration Approval both issued by the Newfoundland Department of Natural Resources, Mineral Development Division. A Water Use Licence may also be required from the Newfoundland Department of the Environment and Conservation, Water Resources Division, as well as a Certificate of Approval for Septic System for water use and disposal for project site facilities.

Permits that will potentially be required for the construction and development of a mining and production facility are outlined in the Table below including and Federal Approvals.

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Table 4.2 Summary of Anticipated Applicable Acts

Level of Government	Approval Potentially Required	Responsible Agency	Legislation/Act
Federal	Authorization under Section 35 or 36 of the Fisheries Act	Department of Fisheries and Oceans and Environment Canada	• Fisheries Act
		Transport Canada	• Navigation Protection Act • Transportation of Dangerous Goods Act
	Environmental Assessment, Compliance Standards; Permits may be required	Environment Canada	• Migratory Birds Convention Act • Canadian Environmental Protection Act • Canadian Environmental Assessment Act 2012 • Species at Risk Act • Federal Policy on Wetland Conservation
	Explosives User Magazine Licence	Natural Resources Canada	• Explosives Act
	Compliance Standards; Permit may be required	Engineering Services Division, Service Newfoundland	• National Fire Code • National Building Code
Provincial	Surface Lease – Areas to be developed for mining should be held under a Surface Lease.	Mineral Development Division, Department of Natural Resources	• Mining Act
	Mineral Exploration Approval – An application for approval must contain a detailed plan and description of the exploration activities	Mineral Development Division, Department of Natural Resources	• Mineral Act and Mineral Regulations
	Environmental Assessment Water Use License – Water withdrawals for use at temporary camps or during construction and operation. Certificate of Approval for Water withdrawal system of > 4,500 L/d. Water supply for construction or operations activities.	Department of Environment and Conservation	• Water Resources Act • Environmental Protection Act • Environmental Assessment Regulations
	Certificate of Approval for Septic System > 4,546 L/d and for drilled wells. Certificate of Approval for Storage and Handling of Gasoline and Associated Products Fuel cache Permit	Engineering Services Division, Service Newfoundland	• Environmental Protection Act
	Quarry Permit – to dig, excavate and extract quarry borrow material for construction purposes	Mineral Lands Division, Department of Natural Resources	• Quarry Materials Act and Regulations
	Cutting Permit Operating Permit Permit to Burn	Forestry Branch, Department of Natural Resources	• Forestry Act and Regulations
	Certificate of Approval for installation of a water supply and Sewage system.	Water Resources Division, Department of Environment and Conservation.	• Health and Community Services Act and Regulations
	Permit for Storage, Handling, Use or Sale of Flammable and Combustible Liquids	Engineering Services Division, Service Newfoundland	• Fire Protection Act
Municipal	Approval for Waste Disposal – the operation of a waste disposal site may require municipal approval. License to operate a temporary work camp.	Community Council	• Municipal Plan and Development Regulations

The specific approvals required will be known once the Environmental Assessment (EA) process has been completed. It is premature to prepare a precisely detailed approvals list until each aspect of the project is better defined.

5. Accessibility, Climate, Local Resources and Infrastructure

5.1 Accessibility

The Cape Ray Project is located approximately 25 km northeast of the coastal town of Channel-Port aux Basques, Newfoundland (population: 4,170 – 2011 census), and 20 km north of the coastal community of Isle aux Morts, Newfoundland (population: 600 – 2011 census) as shown in Figure 5.1.

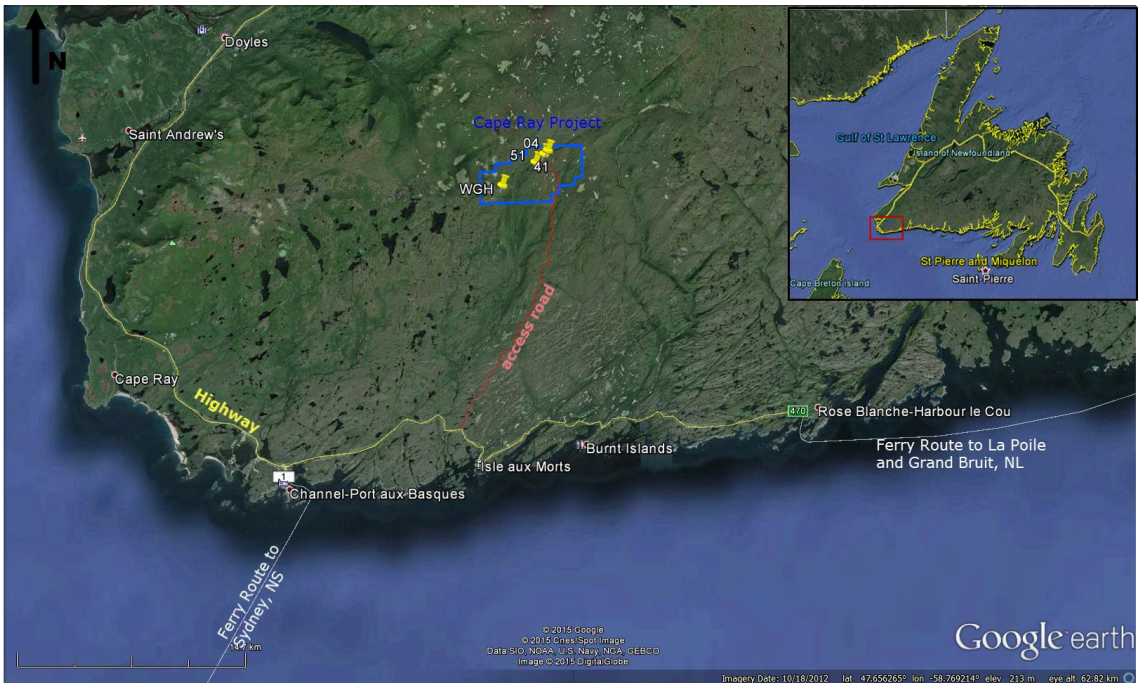

Figure 5.1 Project Location with Nearby Ferry Service. (Google Earth)

Port aux Basques is one of the ports which the Marine Atlantic ferry service provides year-round service to and connects the island of Newfoundland with mainland Canada at North Sydney, Nova Scotia as illustrated in Figure 5.2.

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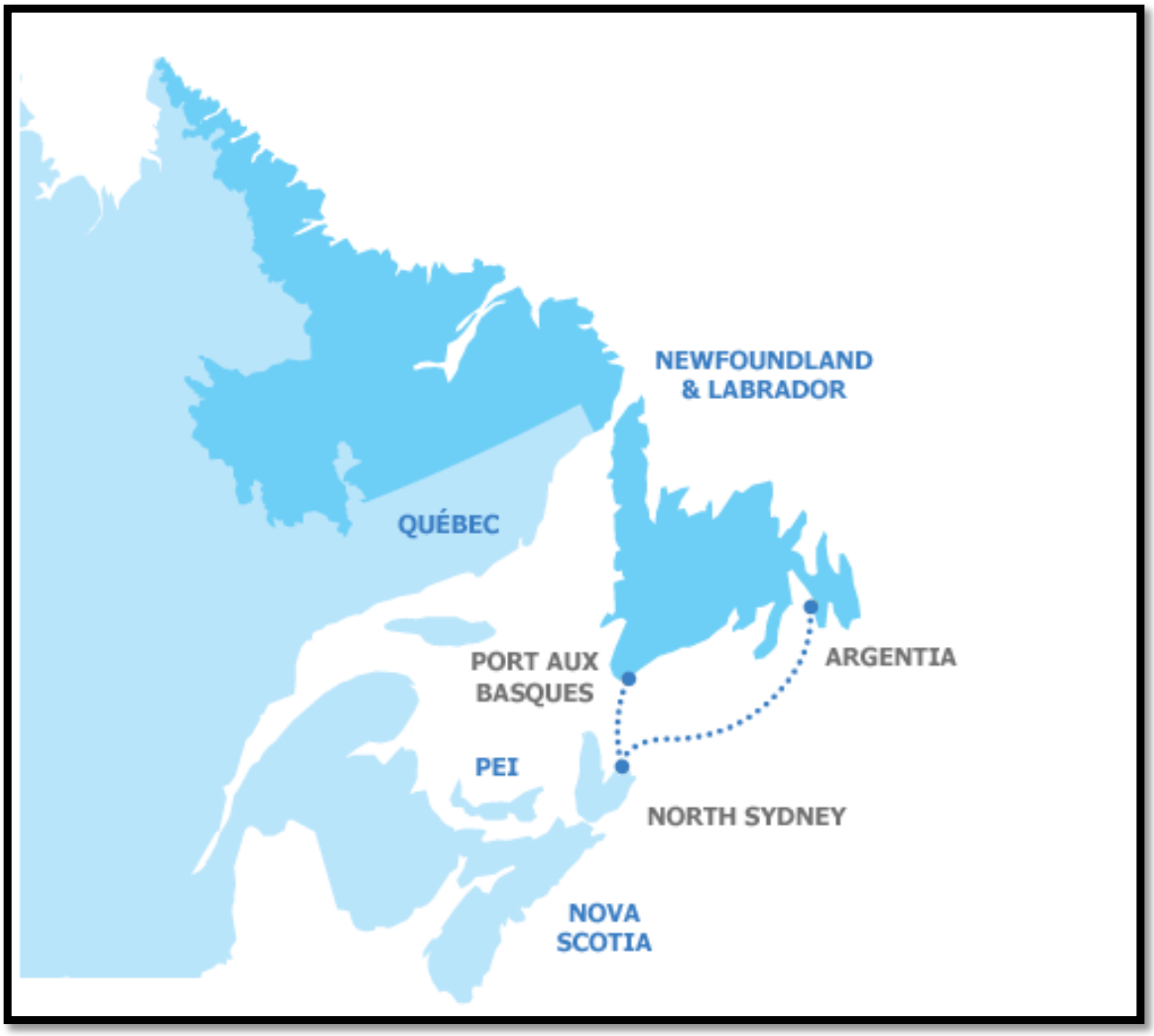

Figure 5.2 Marine Atlantic Ferry Routes. Marine-Atlantic Sailing Routes. Retrieved March 15, 2015, from www.marineatlantic.ca/routes.asp

Direct road access to the Cape Ray East Property is via a 22 km long, rough gravel road that extends north-northeast from Route 470 to provide access to the deposits. The gravel access road was originally constructed in 1989 with minor road upgrades in 2003. The road begins 2.5 km west of the community of Isle aux Mort as shown in Figure 5.3.

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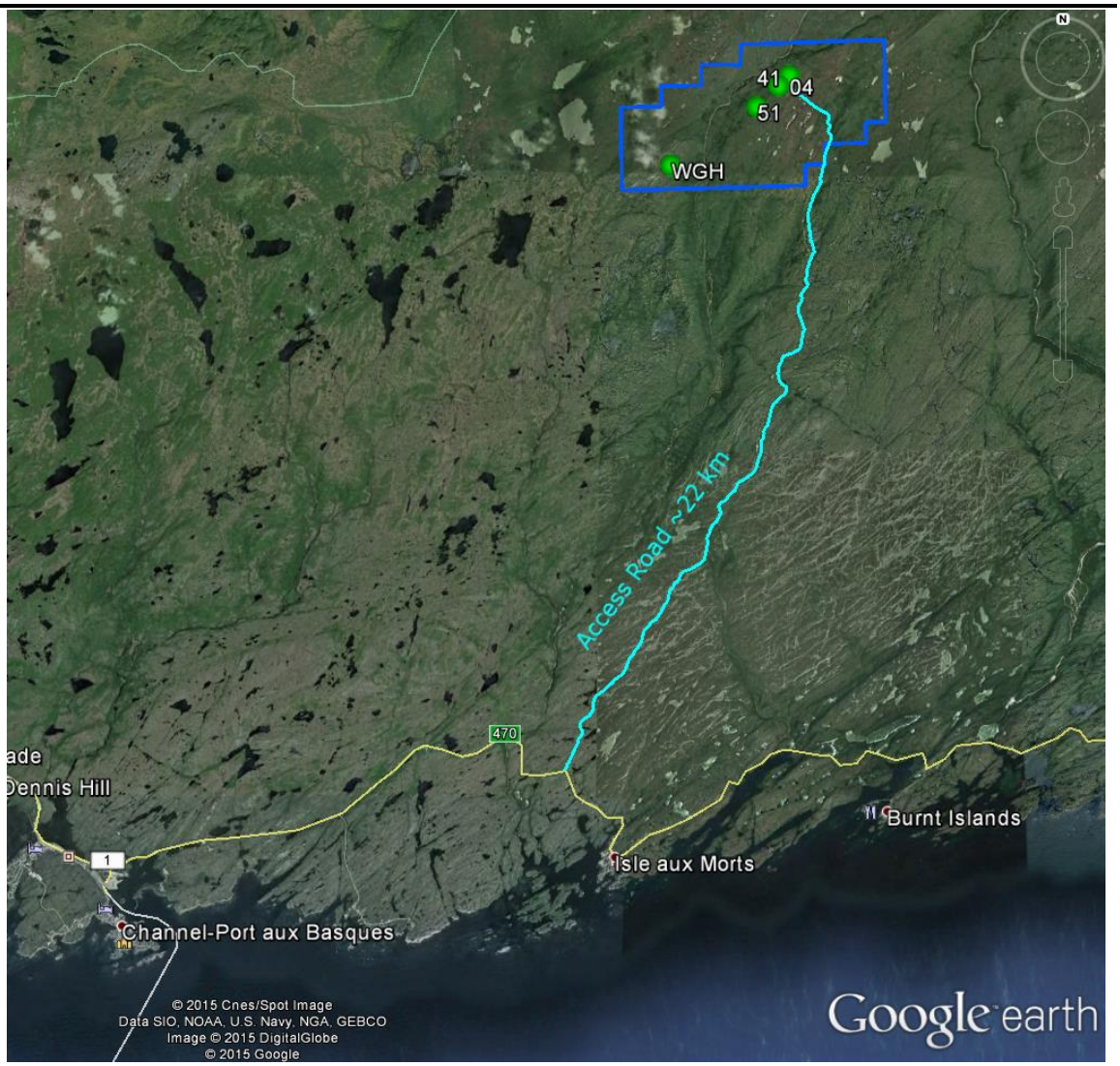

Figure 5.3 Site Access Road. Google Earth.

The access road is currently used for diamond drill and personnel access to the site during the summer months. A series of ATV trails provide access to the other deposits on the property. Route 470 (La Poile Highway) is a coastal, paved regional highway that begins at the Port aux Basques ferry terminal at the intersection with the Trans-Canada Highway (Route 1) and ends approximately 40 km to the East in the community of Rose Blanche-Harbour Le Cou (population 547 – 2006 census) (see Figure 5.1).

Access to the more remote Windowglass Hill deposit on the west side of the Isle aux Morts River (see Figure 5.4) currently requires either the use of a helicopter or acquisition of provincial and federal government permits for temporarily fording the Isle aux Morts River. These permits include an “Environmental Permit to alter a Body of Water”; issued by the provincial Department of Environment and Conservation, Water Resources Management Division, and a “Letter of Advice” issued by the federal Department of Fisheries and Oceans (DFO). Accessibility to certain areas of the Cape Ray Property can be difficult due to the varied terrain and in some instances

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helicopter is the most practical method of transportation of equipment and personnel to drill sites or for other surface exploration activities. Helicopter service is available from commercial bases in the Deer Lake-Pasadena area, approximately 180 km (100 nautical miles) to the northeast.

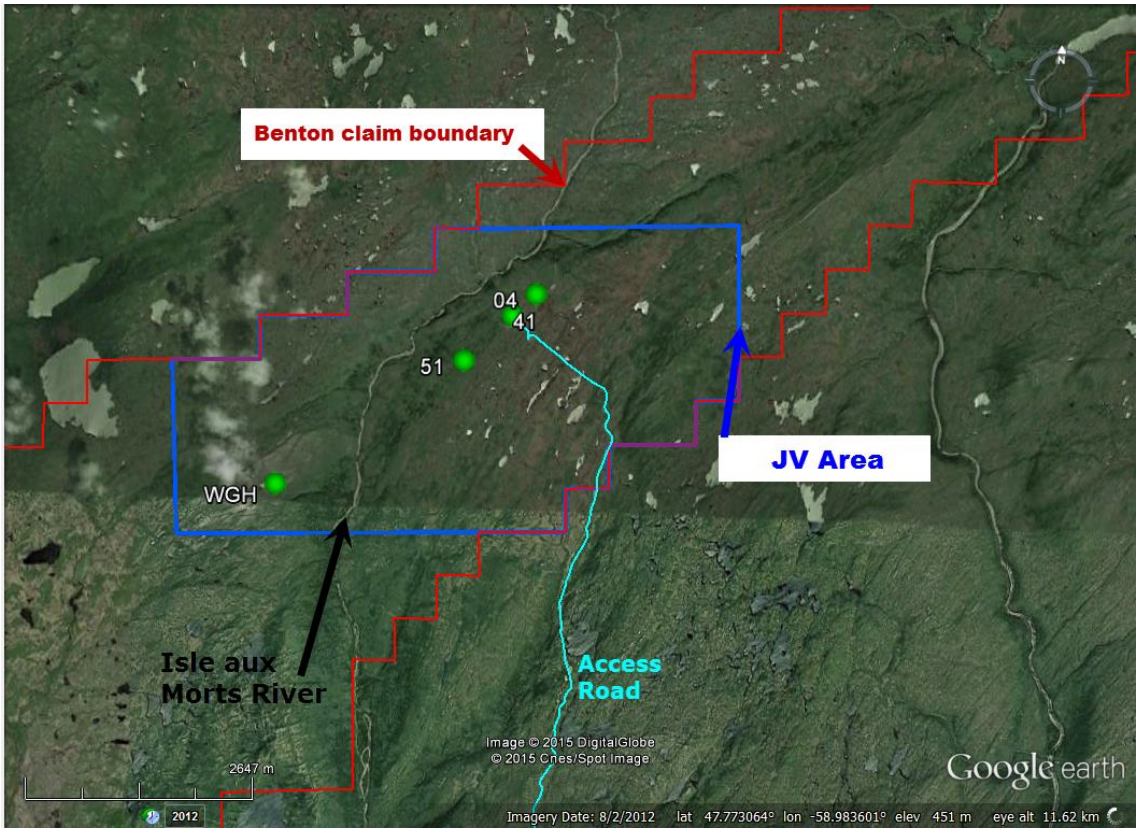

Figure 5.4 Deposit Locations within Claim Boundary. Google Earth.

The communities of Corner Brook (19,886 population - 2011 census), Deer Lake (3,977 population - 2011 census), Pasadena (3,352 population - 2011 census), and Stephenville (6,719 population - 2011 census) combine to provide a wide range of services and logistical support capabilities and are shown below in Figure 5.5. These include daily scheduled flights by air carriers (Air Canada and Provincial Airlines are year-round, and seasonal service via WestJet) from Deer Lake or Stephenville to transportation centres such as Halifax, NS, Toronto, ON, and St. John's NL, as well as regional carrier service to smaller centres such as Goose Bay and Gander, NL. Charter fixed wing aircraft and helicopter services are available through local flight bases in Pasadena and Deer Lake, and the Trans-Canada Highway connects these communities with the rest of Canada via the Marine Atlantic ferry service between Port aux Basques, NL and North Sydney, NS.

Figure 5.5 Communities Servicing the Cape Ray Project. Google Earth.

5.2 Climate

The climate of southwestern coastal Newfoundland is strongly affected by its proximity to the Atlantic Ocean (Cabot Strait), which exerts a moderating effect with respect to temperature extremes. However, dramatic seasonal variations occur, with winter conditions of freezing temperatures and moderate to heavy snowfall expected from late October through late April. Spring and fall seasons are cool, with frequent periods of rain and heavy fog. Summer conditions typically prevail from July through late September, and generally the operating season is fairly short for field exploration and drilling (from late May following spring break-up to late September).

Environment Canada's 1981 to 2010 records for the coastal community of Isle aux Morts, located approximately 20 km south of the Cape Ray Property, show daily mean temperatures in August of 15.8°C and an average maximum August daily temperature of 19.0°C (see Table 5.1). The average daily winter maximum temperature in February is -2.4°C and the corresponding average minimum is -9.2°C.

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Table 5.1 Climate Data for Isle aux Morts, Newfoundland Environment Canada 1981-2010 Data.

1981 to 2010 Canadian Climate Normals station data

Temperature	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Year	Code
Daily Average (°C)	-5.0	-5.8	-3.0	1.5	5.9	10.1	14.0	15.8	12.4	7.5	2.9	-1.5	4.6	C
Standard Deviation	1.3	2.2	2.1	1.2	1.1	1.1	1.0	0.8	1.1	1.1	1.0	1.5	1.2	C
Daily Maximum (°C)	-1.7	-2.4	0.0	4.2	9.1	13.4	17.0	19.0	15.5	10.4	5.6	1.3	7.6	C
Daily Minimum (°C)	-8.3	-9.2	-6.1	-1.2	2.7	6.8	11.0	12.5	9.2	4.5	0.2	-4.2	1.5	C
Extreme Maximum (°C)	9.0	9.0	11.0	17.0	23.0	24.5	26.0	**27.5**	25.0	19.0	16.0	9.5		
Date (yyyy/dd)	2000/ 25	2000/ 29	1983/ 14	1998/ 30	1999/ 07	1983/ 21	1991/ 19	**1996/ 06**	2001/ 10	2003/ 12	2000/ 11	1993/ 14		
Extreme Minimum (°C)	-22.5	**-24.0**	-23.5	-13.0	-8.0	-4.0	4.0	4.5	2.0	-4.0	-11.0	-18.0		
Date (yyyy/dd)	1991/ 26	**1994/ 09**	1986/ 10	1994/ 03	2000/ 10	1988/ 03	1991/ 01	1989/ 29	1988/ 30	1993/ 27	1993/ 25	1984/ 27		

The extreme winter minimum is -24°C, recorded in February. Average yearly precipitation totals 1,653.1 mm, which includes 325.8 cm of snowfall. Very high winds with gusts >100 km/h can occur, often from southerly and southeasterly directions. Temperature and precipitation averages for Isle aux Morts are shown below in Figure 5.6.

Weather and site conditions during the spring break-up period (typically April to late May) can prevent some exploration activities from being carried out due to high water levels and remnant snow cover. Scheduling of field activities to avoid this period is generally advisable. Winter programs can be carried out with snowmobiles, with consideration given to snowfall and cold temperature conditions, as well as potentially high wind conditions with reduced visibility that generally accompany storm events. Coastal waters south of the Cape Ray Property are free of Arctic pack ice and are navigable all year around.

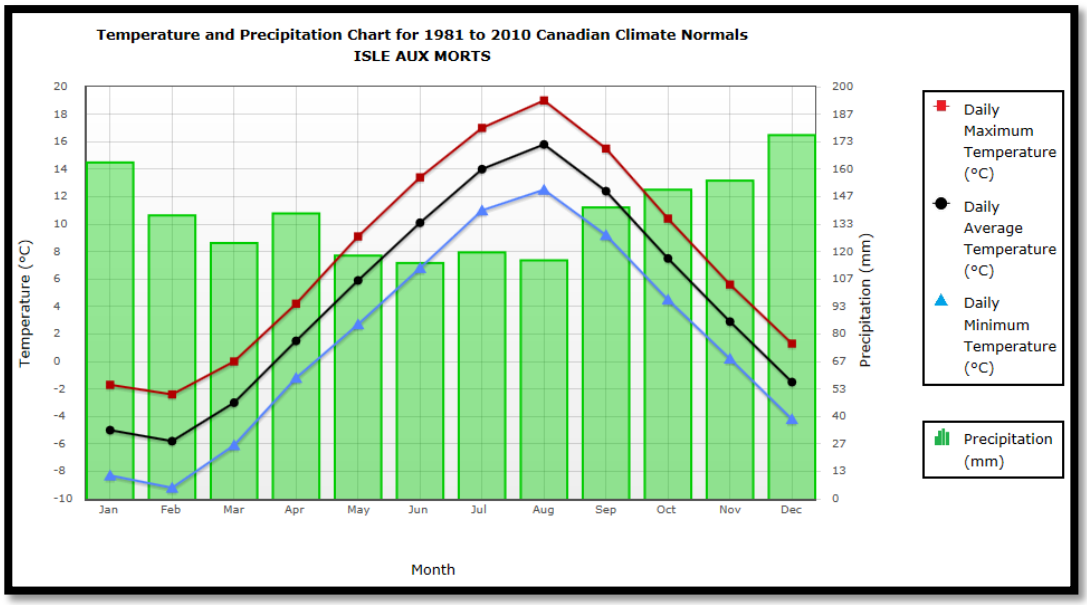

Figure 5.6 Temperature and Precipitation Averages for Isle aux Morts, Newfoundland Environment Canada.

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5.3 Local Resources and Infrastructure

Minimal infrastructure currently exists on the Cape Ray Property and evidence of previous exploration work including drilling, trenching, and bulk sampling are present. No electrical or telephone line infrastructure extends to the Cape Ray Property and the nearest electrical high voltage substation is located approximately 25 km to the southwest in the community of Channel-Port aux Basques. Portable camp facilities on the Cape Ray Property require the use of diesel or gas generators, and regular fuel and supply runs can be completed using the road access.

All areas presented in this economic assessment presented in this report occur on crown lands owned by the Province of Newfoundland and Labrador. Under terms of an agreement with the provincial government in the 1970's and 1980's, previous workers established access to adjacent crown lands in order to conduct scoping level underground metallurgical sampling and test work on the 04, 41, and 51 Zone deposits. The development of any future mining operations in this area must follow permitting requirements, and it is reasonable to assume that enhanced restrictions with respect to watershed issues would be encountered.

The deposit area is non-populated at present and surrounded by an abundance of barren land that would favorably contribute to mine development. There are a few seasonal cottages within the claim boundary (Figure 5.7). Access to government lands for the purpose of future mine development can be reasonably anticipated and would require negotiation of surface rights and other issues with the Newfoundland government. At the present time Benton Resource Inc. does not hold any mining leases within the project area.

Access to the provincial electrical transmission grid is available along the Trans-Canada Highway near Port aux Basques (25 km southwest and west of the site). Lower voltage distribution power lines run along the north side of Route 470 to the town of Isle Aux Morts and the installation of a three-phase 66kV electrical transformer station at a transmission line tap location along with a medium voltage overhead power line to the site may be necessary for the milling operations.

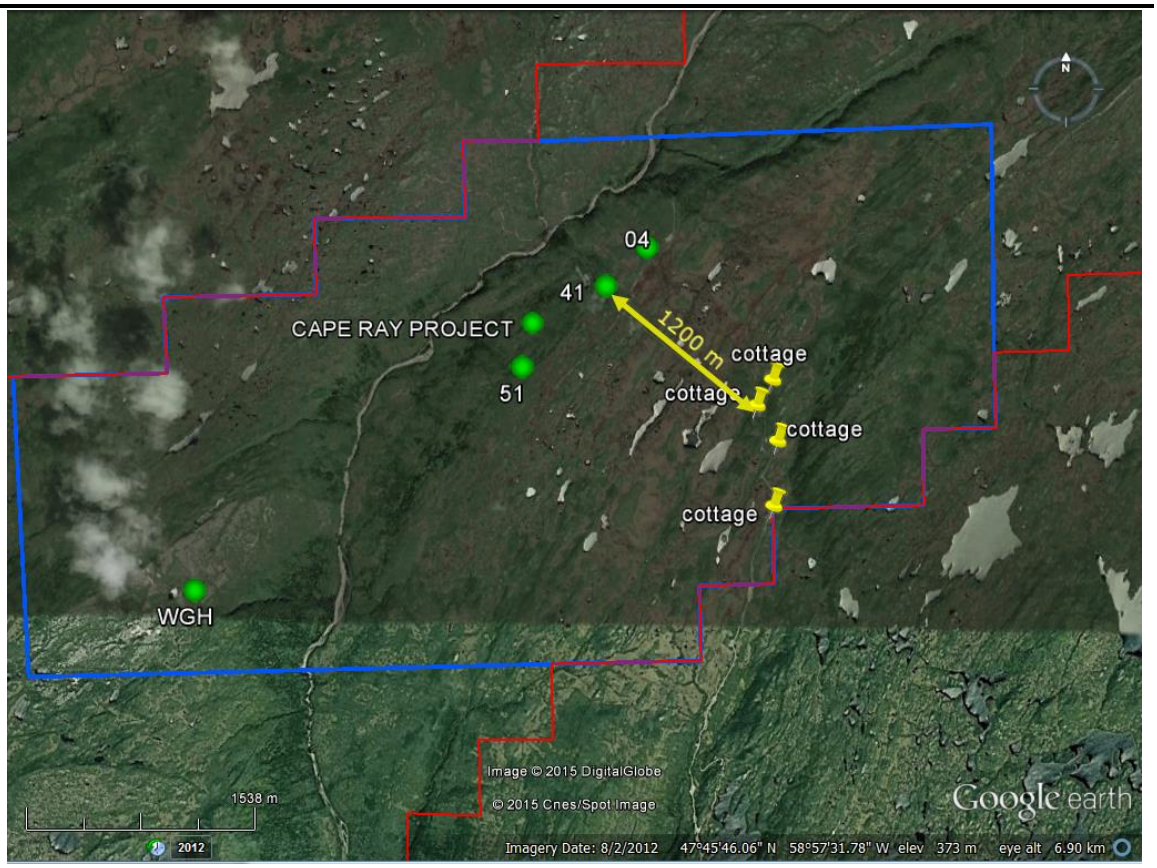

Figure 5.7 Proximity of Cottages in the Claim Area

The abundance of surface water resources from lakes on the Cape Ray Property is a positive factor for future development, as is the presence of deep water wharf facilities and commercial shipping infrastructure at nearby Port aux Basques (<50 km driving distance).

The current access road will require upgrades to simplify logistics to ready the property for production and to supply year round access. The road would also need to be extended to connect outlier deposits that are deemed economically viable for production.

5.4 Physiography

The Cape Ray Property is situated on an upland glaciated plateau with gentle to moderate relief and elevations typically ranging from 300 to 350 metres above sea level. Occasional ridges and peaks reach 450 metres above sea level. Rivers have cut deeply into the plateau forming steep, wooded valleys with an average elevation of 200 metres above sea level in the valley bottoms. The Cape Ray Property is characterized by boulder-strewn barren lichens and peat bogs interspersed with scattered crops of stunted spruce and fir, locally called tuckamore. These stunted spruce and fir forests also occur along the upper regions of river valleys, but grade into more open evergreen forest in the lower portion of the valleys.

The south-flowing Isle aux Morts, Grandy's, and La Poile river systems represent three major

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drainage systems that have formed the deeply incised river valleys in the upland barrens. The major drainage systems are fed by streams that drain into sparsely distributed, generally shallow lakes. Due to the sparse vegetation in the higher barren regions and normally saturated condition of the ground, surface run-off can produce abrupt changes in water levels in local streams and rivers.

The Cape Ray Fault Zone appears as a prominent V-shape valley (gulch) trending northeast across the area. South of the fault, valleys and ridges are often structurally controlled and generally trend northeast to southwest. The Cape Ray Property has been extensively glaciated and visible outcrops are minimal in areas of abundant glacial till and outwash. Depths to bedrock on the Cape Ray Property are typically only 6 to10 metres with generally minimal overburden present (glacial deposits).

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6. History

The Cape Ray property has been explored studied and has been the subject of academic and geological papers since the early 1970s. This section provides a brief history of past exploration on the property and is largely summarized by Ténière and Hilchey (2012) who produced an NI 43-101 technical report on the Cape Ray Au-Ag Property in the Isle aux Morts Area.

Note that the historical data is directly quoted from previous reports and portions of the data are in the imperial system due to the time of publication. For future sections of this report, all numbers for this project will be reported in the metric system.

6.1 Academic and Government Studies (Ténière & Hilchey, 2012)

Beginning in the early 1970's, the Geological Survey of Canada flew an aeromagnetic survey over the Cape Ray Property area, and Gillis (1972) published results of regional geological mapping covering the Port aux Basques map sheet (NTS 11O). Gillis defined the Cape Ray Fault Zone (CRFZ) as a prominent lineament that separated Precambrian to Ordovician mixed plutonic, meta-volcanic and meta-sedimentary rocks in the northwest from Silurian to Ordovician quartzo-feldspathic gneiss and schist in the southeast.

Brown (1972, 1974, 1976 and 1977) conducted 1:50,000 scale geological mapping over the Port aux Basques map sheet as part of M.Sc. and Ph.D. thesis projects, as well as for the Newfoundland and Labrador Geological Survey. He identified the metamorphic and structural complexities of the area adjacent to the Cape Ray Fault as well as the structural history of the Port aux Basques Gneiss Complex, the Cape Ray Plutonic Complex and the younger Windsor Point Group and their relationships with the Cape Ray Fault.

Between 1979 and 1980, while working for Riocanex, D.H.C. Wilton mapped the CRFZ from the area of the Cape Ray Gold Deposit (CRGD) to the coast. This work formed the basis of a Ph.D. thesis (Wilton, 1983a). Wilton (1983b, 1984 and 1985) and Wilton and Strong (1986) published several technical papers based on these studies.

Chorlton (1980, 1983 and 1984) and Chorlton and Dingwell (1981) also carried out geological mapping over part of the CRFZ. This work was conducted for the Newfoundland Department of Mines and Energy and also as part of a Ph.D. thesis. Chorlton and Dallmeyer (1986) carried out geochronological studies in the area.

Lake-bottom sediments collected during the late 1970's by the Newfoundland Department of Mines and Energy were analyzed for gold and related elements, the results of which, including those for the Cape Ray Gold Project area were published in Butler and Davenport (1980) and Davenport and Nolan (1987).

Metallogenic studies of the granitoids in the Cape Ray area were reported in Tuach (1986) and Tuach et al. (1988).

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During the period from 1989 to 1995 the Quebec Geoscience Centre of the Geological Survey of Canada conducted detailed geological mapping (1:20,000 scale maps) and metallogenic studies along a 60 km segment of the southwestern portion of the CRFZ (Dubé, 1990; Dubé et al., 1991, 1992, 1993, 1996; and Dubé and Lauzière, 1996a, b). The compiled results of these detailed studies are presented in Dubé and Lauzière (1997).

6.2 Mineral Exploration Summary

Summarized below in Table 6.1 is a brief history of geological and exploration work carried out in the area and around the Cape Ray Project as reported by Ténière and Hilchey (2012) and within Benton Resources Inc. news releases (after 2012).

Table 6.1 Exploration Summary by Year

Year(s)	Company(s)	Highlights
1953-1969	British Newfoundland Exploration Company (Brinex)	• Government of Newfoundland granted majority of current Cape Ray Property to Brinex. • Brinex performed stream sediment survey and discovered 'Gulch' showing located 2.7 km Northeast of 51 Zone.
1975-1976	Philips Management Inc. (Philips)	• Philips acquired large property along 40 km strike length of CRFZ. • Conducted large airborne electromagnetic and magnetic survey with mapping, prospecting, and stream sediment sampling. • Sulphide anomalies found in the 'H' and 'I' Brook showings within CRFZ.
1976-1977	Amax Minerals Exploration Ltd. (Amax)	• Amax optioned claims from Philips and completed 2 diamond drill holes (ddh) in 41 zone vicinity. • One hole encountered mineralisation at a depth of 29.57 m.
1977-1983	Rio Tinto Canada Exploration Ltd. (Riocanex)	• Riocanex optioned claims from Philips in 1977. • Carried out several geophysical and geochemical surveys as well as trenching and diamond drilling. • Drilled 172 ddh by 1980 (24,054 m). • Discovered the CRGD which includes the 04, 41 and 51 deposits and followed by WGH deposit. • Parent company Rio Algom did metallurgical testing and hired Acres Davy McKee Ltd. (ADM) to complete open pit pre-feasibility study (1981). • Rio Algom retained Lakefield Research of Canada Ltd. (Lakefield) to perform metallurgical testing on a bulk sample from the 41 zone. • Riocanex dropped the property in 1983 after determining the potential tonnage was insufficient to justify mine development.
1984-1985	New Venture Equities Ltd. (New Venture) partners with Mascot Gold Mines (Mascot)	• New Venture acquired the Cape Ray property in 1984. • Completed geological mapping, prospecting, and soil geochemical surveys. • No diamond drilling was completed. • Focus of sampling and mapping was the area southwest of CRGD. • Underground exploration program carried out on 41 deposit: • 152 m decline to depth of 27 m. • Vertical drifting along the 'A vein' • Discovered the discontinuous Au grades were due to structural complexity. • Grade control during an operation would require constant supervision. • Retained ADM to perform revised underground mining prefeasibility study (1984).
1986-1989	International Corona Resources Ltd. (Corona) with subsidiary Dolphin Explorations Ltd.	• Corona acquired Cape Ray after amalgamating with New Venture (1986). • Dolphin carried out extensive exploration and development work (1986-1989): • Compilation of previous work • Geological mapping, prospecting, rock, soil and lake sediment sampling.

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Year(s)	Company(s)	Highlights
	(Dolphin)	• VLFEM horizontal loop EM, and IP geophysical surveys • Diamond drilling 118 holes (23,310 m) • Discovery of the Sleeper and Big Pond zones Southeast of Windowglass Hill. • Continue underground exploration by: • Extending the decline additional 525 m to vertical depth of 72 m. • Completed 25 m of vertical drifting along 'A vein'. • 60 m of crosscuts, and 47 m of lateral development for the 307 tonne bulk sample (41 deposit) • Profitable results from first bulk sample prompted an extension: • 41 deposit decline was extended another 91 m to vertical depth of 82 m. • Test stoping on the 'A vein' between 72 m level and 27 m level completed with lateral development (211 m) and raising (77 m). • Stope recovered 3,228 tons of 'A vein' • Another sample of 262 tons was taken from 'C vein' from the 82 m level. • Retained Kilborn Ltd. To complete a mining feasibility study (1990).
1990-1996	American Gem Corporation	• Corona and Dolphin acquired by Homestake Mining Corp. (1992) and sold the Cape Ray property to American Gem Corporation • Property is dormant from late 1989 – 1996
1996-1998	Royal Oak Mines Ltd. (Royal Oak)	• Royal Oak purchased property from American Gem Corporation. • 17 ddh completed. • Site road survey and improvements. • 22.73 km of ground Mag-VLF (Sleeper and Big Pond). • Royal Oak filed for bankruptcy in 1999.
1999-2002	South Coast Ventures Ltd. (South Coast) , Alexander J. Turpin (Turpin) and Cornerstone Capital Resources Inc. (Cornerstone)	• Claims open for staking via draw system for expired mineral licences in late 2000. • South Coast staked over the Isle aux Morts, Big Pond-Sleeper, 04, and 41 Deposits. • Turpin staked over the 51 and WGH Deposits. • 2002 Terra Nova Gold options for 100% ownership of South Coast claims. • 2002 Cornerstone options for 100% ownership of Turpin Claims.
2003	Terra Nova Gold Corporation (Terra Nova)	• Expand existing IP survey. • Diamond drilling 20 holes completed.
2004	Terra Nova Gold and Tenacity Gold Mining Company.	• In December 2004 decided to relinquish options on Cape Ray to explore an opportunity in Mexico. Tenacity acquired the property.
2005-2011		• No exploration work completed.
2012	Cornerstone Capital Resources Inc. (Cornerstone)	• Cornerstone retained Mercator Geological Services to prepare an NI 43-101 resource update on the 51 and WGH deposits • Resource Estimate reported by Mercator shown below in Table 6.2.
2013	Benton Resources Inc. (Benton) and Tenacity Gold Mining Corporation (Tenacity)	• Benton purchased 100% interest in the claims held by Cornerstone (51 and WGH deposits). • Benton collected 150 kg mini bulk sample from 51 deposit for metallurgical testing. • Benton entered into option agreement for 100% purchase of the four claims held by Tenacity which include the 04, 41, Isle aux Morts and Big Pond deposits. • Benton also staked an additional 108 claims to hold 100% control of the 60 km strike length along the Cape Ray Fault. • Benton conducted geochemical sampling, geological mapping, and IP surveys.
2014-2015	Benton Resources Inc. (Benton)	• Diamond drill program to drill 17 holes (2000 m). • Completed final payment to Tenacity (04, 41, Isle aux Morts and Big Pond deposits) for 100% interest in 60 km strike length of Cape Ray. • 2015 trenching efforts

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The Crown holds all surface rights in the Cape Ray Project area. None of the property or adjacent areas is encumbered in any way. The area is not in an environmentally or archeologically sensitive zone and there are no aboriginal land claims or entitlements in this region of the province. There has been no commercial production from the property as of the time of this report.

Previous resource estimates for 04, 41 and 51 deposits were reported prior to the creation of NI 43-101 and are therefore non-compliant; this information has been left out of this report due to the age of the data without validation.(see Table 6.2).

Table 6.2 Cornerstone Resource Estimate Summary (Ténière & Hilchey, 2012)

Deposit	Tonnes	Au (g/t)	Ag (g/t)	Ounce Au	Ounce Ag	Resource Class	Cut-off Grade	Description
51 Zone	530,000	6.0	15.86	103,943	270,253	Inferred	2.5 g/t Au	43-101 compliant inferred resource
Windowglass Hill	1,310,000	2.14	12.27	90,131	516,781	Inferred	1.0 g/t Au	43-101 compliant inferred resource

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7. Geological Settings and Mineralisation

The Cape Ray property has been explored, studied and the subject of academic and geological papers since the early 1970s. This section provides a description of the geology and mineralisation of the southwestern Newfoundland area of interest. This was well summarized by Ténière and Hilchey (2012) who produced an NI 43-101 technical report on the Cape Ray Au-Ag Property in the Isle aux Morts Area and has been directly extracted from their report.

7.1 Regional Geology and Tectonic Setting

Southwestern Newfoundland is an area of complex Appalachian geology recording intense crustal shortening, variable metamorphism, and magmatism beginning in the Late Silurian, which is related to collision of the composite North American continent to the north and the Gander-Avalon composite terrane of Gondwanan affinity to the south (Dunning et al. 1990).

The Cape Ray Fault Zone (CRFZ) forms a structural boundary between two of four tectonostratigraphic zones, or terranes that define the geology of Newfoundland (Figure 7.1), these being the Dunnage Zone to the northwest and the Gander Zone to the southeast (Williams, 1979). The Dunnage Zone includes rocks originally deposited in the Late Precambrian – Early Paleozoic Iapetus Ocean, and in southwestern Newfoundland is represented by the Cape Ray Igneous Complex (CRIC) (Williams, 1979; Dubé & Lauzière, 1997). The Gander Zone is mainly comprised of deep water turbidite sedimentary rocks deposited at, or near the eastern side of the Iapetus Ocean. In southwestern Newfoundland these are represented by the Port aux Basques Gneiss (PABG) (Williams, 1979; Dubé & Lauzière, 1997). The CRFZ is therefore located at, or very close to the boundary between the Laurentian and Gondwanan terranes (Dubé et al. 1996). The CRFZ hosts the Cape Ray Gold Deposits (CRGD), which comprise one of the most important groups of vein-type gold deposits in the Appalachians, as well as other significant vein-type gold occurrences that include the Big Pond and, Isle aux Morts prospects, and the Windowglass Hill Deposit (WGHD).

The CRFZ is a zone of highly strained rocks approximately 100 km in length and several hundred m in width that includes three major geological domains (Figure 7.2): (1) the CRIC, (2) the Windsor Point Group (WPG), and (3) the PABG. The CRFZ separates the CRIC to the northwest from the PABG to the southeast (Dubé & Lauzière, 1996a) and WPG rocks, located between these two units, which coincide approximately with the trace of the CRFZ. All three units are intruded by a number of pre- to late-tectonic granitoid intrusions.

The CRIC is comprised mainly of large bodies of mafic to ultramafic intrusive rocks intruded by granitoid rocks (Cape Ray Granite, Cape Ray Tonalite, Red Rocks Granite), and is interpreted to have a U-Pb zircon age of 469 ± 2Ma (Mid-Ordovician) based on samples obtained from the Cape Ray Tonalite (Dubé et. al., 1996). The WPG unconformably overlies the CRIC and consists of bimodal volcanics and volcaniclastics with associated sedimentary rocks that are interpreted to be Ordovician to Devonian in age based on U-Pb radiometric dating (Dubé et. al., 1996).

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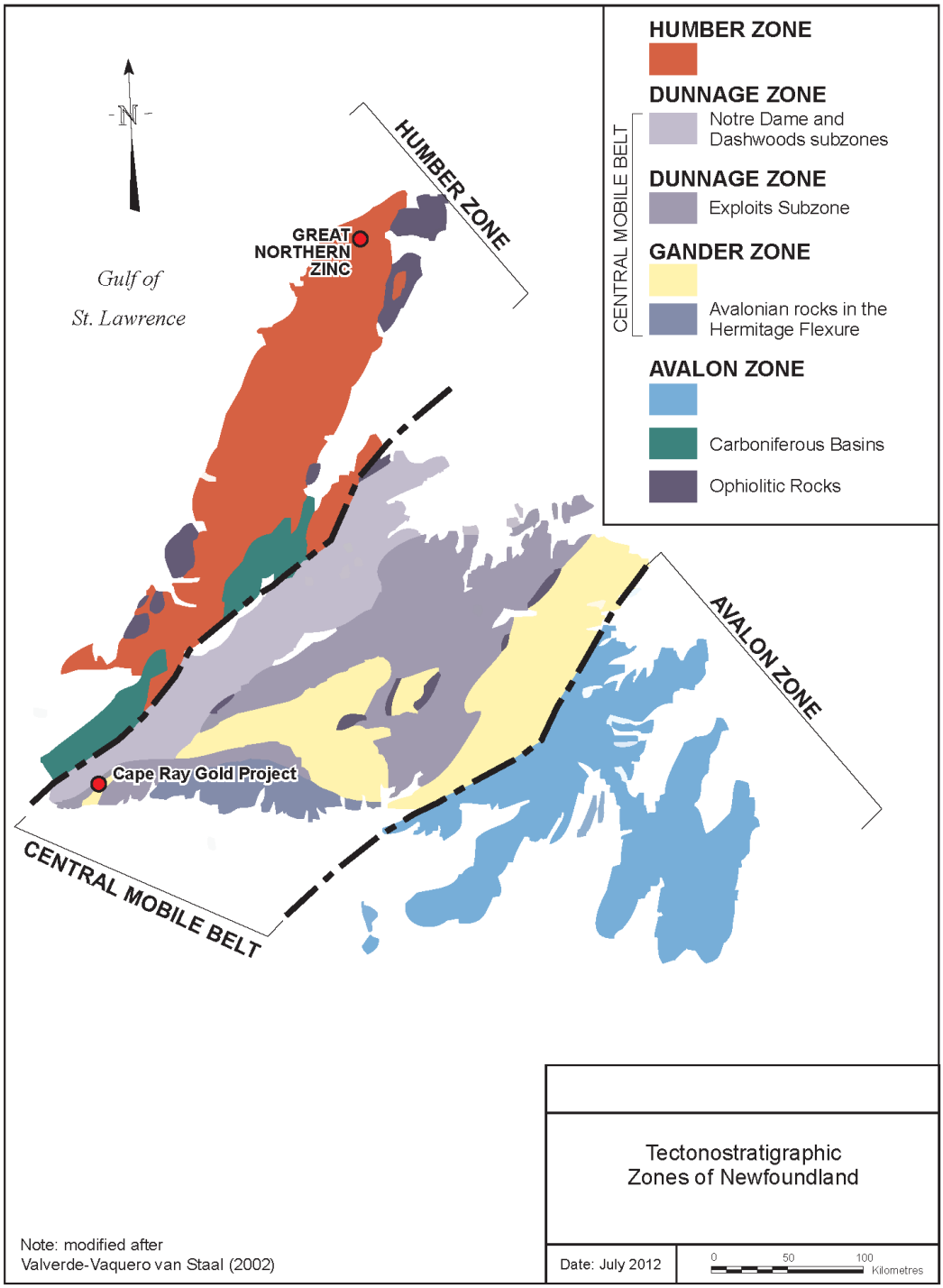

Figure 7.1 Tectonostragraphic Zones of Newfoundland (Ténière & Hilchey, 2012)

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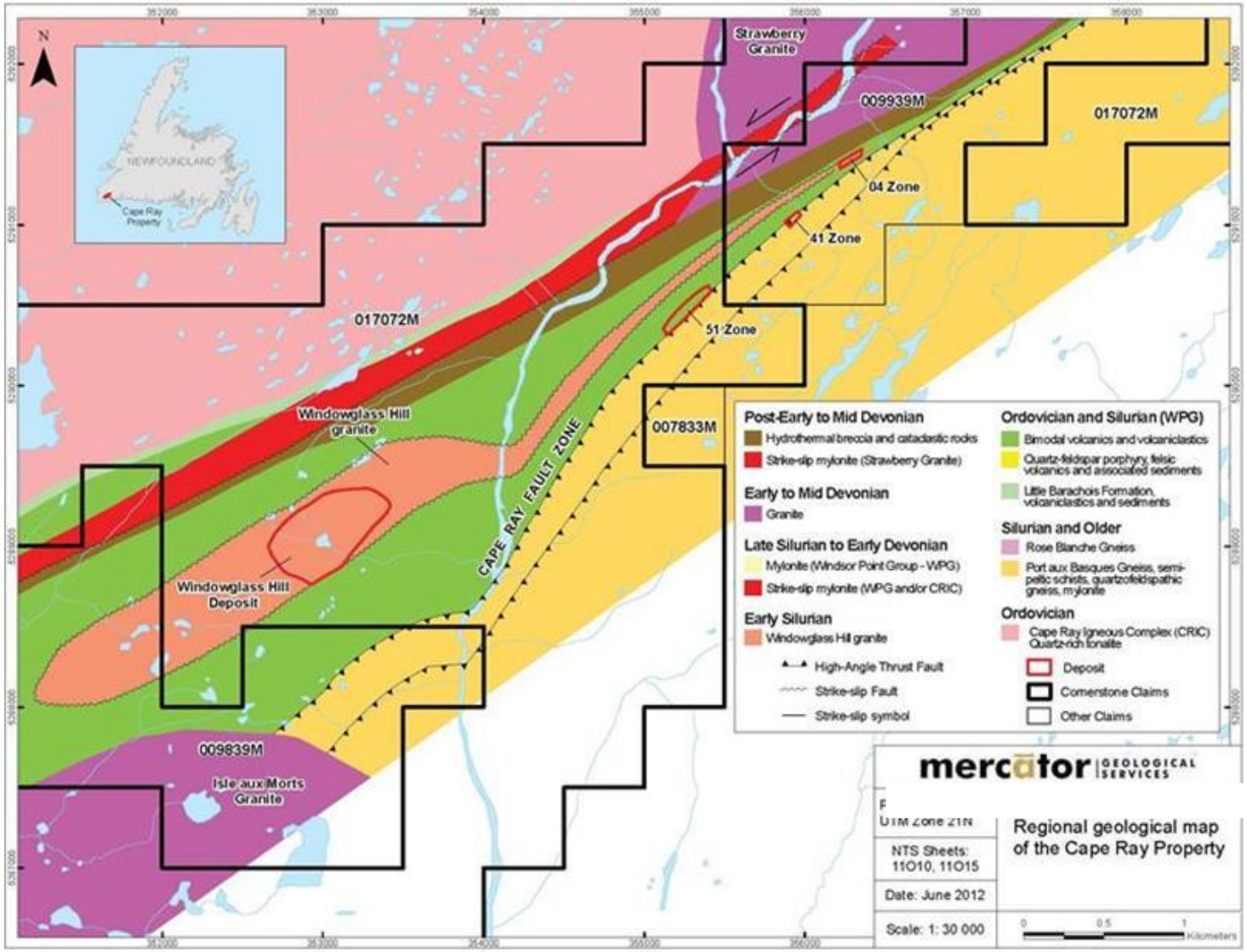

Figure 7.2 Bedrock Geology of the Cape Ray Property (Ténière & Hilchey, 2012)

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The PABG consists of high grade, kyanite-sillimanite-garnet, quartzo-feldspathic pelitic and granitic rocks intercalated with hornblende schists or amphibolites. The PABG is subdivided into three distinct lithological units: (1) the Grand Bay Complex (GBC); (2) the Port aux Basques Complex (PABC); and (3) the Harbour Le Cou Group. These units are intruded by large volumes of basic sheets and coeval Middle Ordovician granitoids, as represented by the Margaree and Kelby Cove orthogneiss bodies. The Grand Bay and Port aux Basques complexes are juxtaposed by the Late Silurian Isle aux Morts Fault Zone against the Harbour Le Cou Group rocks to the east, which dominantly consist of sillimanite and garnet-bearing, pyritiferous metasediments with amphibolitic intrusions and meta-pillow basalts in lower stratigraphic units (Schofield et al., 1998).

The CRFZ is well exposed along the south western coast of Newfoundland near the town of Channel-Port aux Basques (Dubé & Lauzière, 1996a). In this location, the Grand Bay Complex contains strongly foliated biotite and hornblende-rich psammites. Further inland to the northeast, it also includes garnet-rich metapelites and mylonitic felsic orthogneisses. Northeast of the GBC there is a wide mylonitic zone that separates WPG volcanic conglomerates from the biotite-hornblende psammites. These mylonites are characterized by a strong planar fabric and by mm-scale compositional layering defined by alternating silica-rich layers and sericite to chlorite-rich layers. The protolith of the mylonites is difficult to determine. However, the gradual transition between the conglomerates and the mylonites, and the local preservation of granitoid clasts within the mylonite zone along its northwest margin indicate that the mylonite zone is at least partly derived from the tonalite pebble conglomerate of the WPG. Metamorphic minerals present in the mylonitic foliation are typical of greenschist facies conditions and vary from biotite along the southeast margin of the mylonite zone, near the biotite and hornblende-rich rocks of the GBC, to dominantly chlorite ± sericite along the northwest margin (Dubé & Lauzière, 1996a).

Northwest of the mylonite zone, a central sedimentary domain and a northern volcanic domain of the WPG are present. The former consists of folded and transposed conglomerate interbedded with tuff, pebbly sandstone, black shale, greywacke, and siltstone. The volcanic domain is approximately 700 m thick and dominated by mafic volcanic rocks, including flows and gabbroic sills (Dubé et al. 1992). Felsic volcanic rocks occur mainly near the contact with the CRIC and include ignimbrites, rhyolitic flows, and associated pyroclastic and volcaniclastic – epiclastic rocks. Units of pillowed basalt are locally intercalated with the felsic volcanic rocks. A narrow sedimentary unit consisting of polymictic conglomerate, greywacke, limestone, and graphitic shale occurs between the felsic and mafic volcanic domains. The WPG is crosscut by the pretectonic and subvolcanic Late Silurian Windowglass Hill Granite dated at 424 ± 2Ma (Dubé et al. 1993, 1996) (Figure 7.2). All lithologies are cut by the late-tectonic Mid-Devonian Strawberry Hill (384 ± 2Ma) and Isle aux Morts (386 ± 3Ma) granites, composed of red coarse grained to porphyritic granite (Dubé et al., 1993, 1996).

The northwestern side of the coastal section is a quartz-bearing tonalite and/or a megacrystic granitic phase of the CRIC. The CRIC - WPG contact represents a deformed unconformity (Dubé et al. 1992, 1996).

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7.2 Structural Geology of the CRFZ

7.2.1 Introduction

The CRFZ is a major reverse-oblique fault zone that was active between Silurian and late Devonian time. Activity is largely attributed to major oblique collision between the composite Gondwanan and Laurentian terranes (Dubé & Lauzière, 1996a). Dubé and Lauzière (1997) describe the following structures that define six major deformation events (D1 to D6).

7.2.1.1 D1 - D2 Event

The first two generations (D1 and D2), correspond to amphibolite facies metamorphism and deformation and are restricted to the gneissic rocks of the PABG. The main fabric recognized in rocks of the PABG is a composite S1 – S2 gneissic layering. Peak metamorphic conditions were reached during the D2 event. An amphibolite-grade mylonite zone in rocks of the GBC is exposed along the coast near the WPG/GBC contact. It is characterized by a strong northeast oriented fabric (S2) and contains moderate to steep S-plunging mineral lineation (L2). Kinematic structural indicators point to reverse sinistral movement that occurred prior to 406 ± 4 Ma and 403 ± 4 Ma (Dubé et al., 1996).

7.2.1.2 D3 Event

D3 structures within the CRFZ affect WPG and GBC rocks and, to a lesser extent, CRIC intrusive rocks. The D3 increment of deformation occurred under greenschist-facies metamorphic conditions that resulted in prograde metamorphism in WPG rocks plus retrogression and overprinting of the amphibolite-grade mylonites and gneisses of the GBC. Two sets of structures are grouped under D3 and are interpreted to be related to a single progressive deformational event because of their similarity in deformation style, metamorphic conditions and orientation of structures (Dubé et al., 1996).

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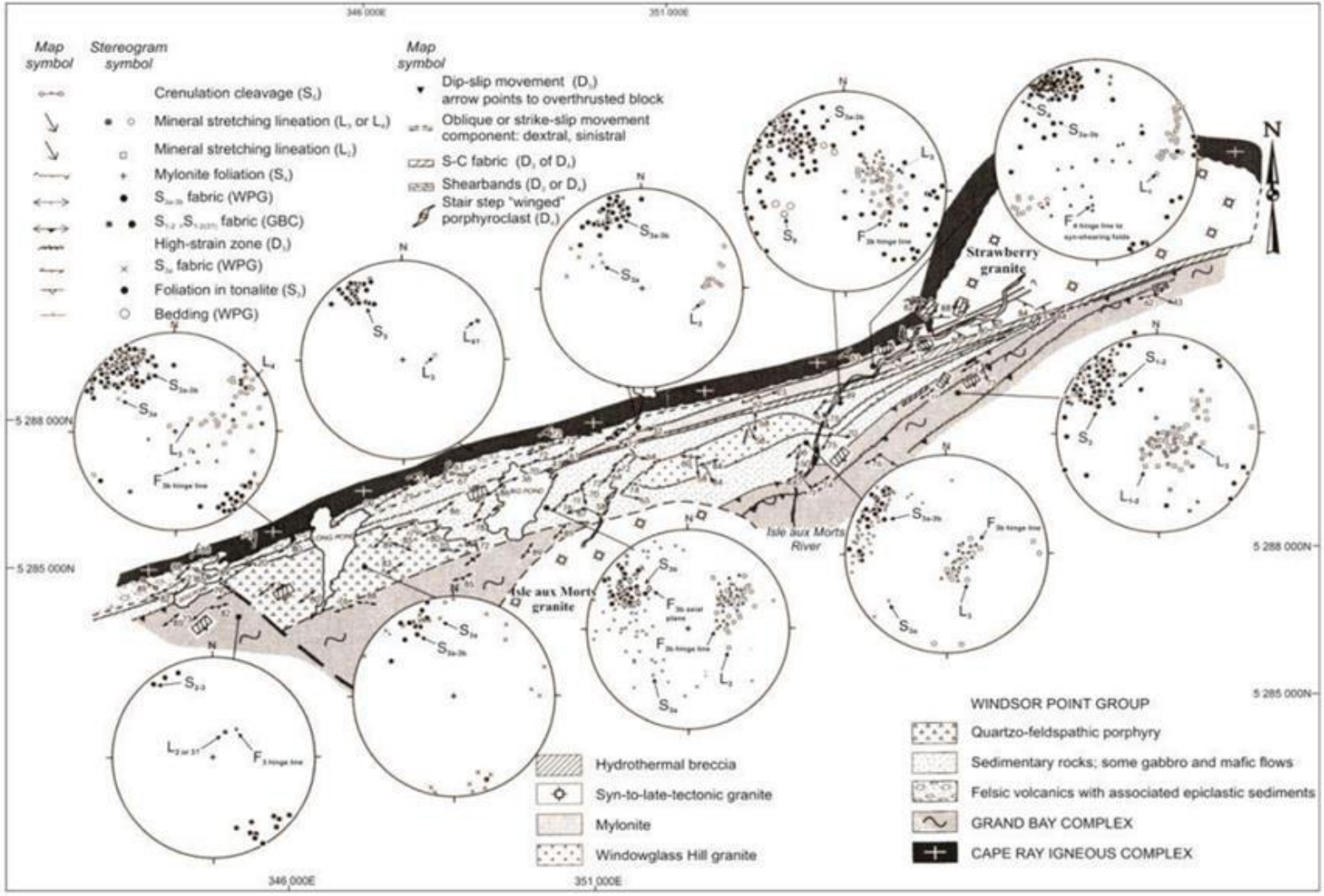

Figure 7.3 Geological and Structural Map of the Isle aux Morts River Area (Dubé & Lauzière, 1996a)

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Moderate-strain characterizes most of the WPG, although there is a northwest directional decrease in D3 deformation away from the WPG - GBC contact, where bedding and a transposed S_{3a} foliation are folded by a secondary S_{3b} foliation. In most of the area, S_{3b} is a penetrative fabric, axial planar to asymmetric F_{3b} folds. Commonly, S_{3a} is transposed into the S_{3b} fabric, producing a complex composite S_{3a-b} foliation. S_3 folding is also present in the PABG, where it produces complex interference patterns defined by the gneissosity. S_{3a} and S_{3b} commonly strike northeast to north-northeast and dip moderately southeast to steeply northwest. L_{3b} lineations plunge moderately east to east-southeast, although south-plunging stretching lineations were locally observed. Generally, F_3 hinges are sub-parallel to the stretching lineation and plunge steeply to moderately east-southeast. Increasing deformation during D_3 generated narrow to wide zones of high strain. These zones are well developed at the CRIC contact, but more intense at the PABG contact, where the zone varies in orientation from northeast-southwest to east-northeast in the flexure area of the fault zone (Dubé et al., 1996).

The D3 mylonite zone is made up of quartzo-feldspathic mylonites and schists produced either by extensive retrogression of the PABG, by shearing of the WPG rocks, or by shearing of the Rose Blanche Granite in the flexure area. Within the mylonites, the dominant fabric is a Stectonite characterized by a mylonitic fabric (S_{3b}), which is probably a composite S_{3a-b} foliation and/or a compositional layering sub-parallel to the shear zone. In the northeast-oriented segment of the fault, the mylonitic fabric contains a steeply to moderately east-plunging stretching lineation (L_3) and reverse-dextral kinematic structural indicators. In the east-west oriented segment of the fault, the S_3 fabric contains a moderate southwest-plunging to sub-horizontal stretching lineation, and dextral strike-slip to oblique-slip motion is indicated by a number of kinematic structural indicators. This segment of the fault is interpreted as a tear fault accommodating differential shortening rates on either side. The upper time limit of the reverse shearing event is constrained by the intrusion of the Early Devonian Isle aux Morts Granite (384 ± 2 Ma) that cuts across the reverse-oblique shear (Dubé et al., 1996).

7.2.1.3 D4 Event

Another greenschist facies mylonitic zone (D4) is developed at/or near the WPG/CRIC boundary in the Isle aux Morts River area. This zone can be traced for up to a few tens of metres northeast of the Isle aux Morts River where it is in part cut by the Strawberry Granite. This mylonite gradually thins west of Big Pond and persists as a high strain zone a few metres wide for up to 3km west of Dog Pond. It is characterized by a steeply dipping east-northeast foliation (S4) and compositional layering, and shallowly-plunging to sub-horizontal stretching lineations (L4). Kinematic structural indicators suggest both dextral and sinistral apparent motions.

7.2.1.4 D5 Event

During the D5 event, two sets of late axial-planar cleavages (S_5) and associated local folds and kinks (F_5) developed at moderate to high angles to the main S_3 fabric along the CRFZ. These developed in all the structural domains and are post-mylonitic.

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7.2.1.5 D6 Event

Late brittle faulting (D_6) is indicated by the presence of tectonic and hydrothermal breccias along the mylonite zone in the Isle aux Morts River area that also cut the Mid-Devonian Strawberry Hill Granite.

7.3 Geology of the Cape Ray Gold Deposits

The Cape Ray Gold Deposits (CRGD) consist of three main mineralised zones: the 04, 41 and 51 Zones. These have historically been referred to as the "Main Zone". They occur as quartz veins and vein arrays along a 1.8 km segment of the CRFZ at or near the tectonic boundary between the WPG and the PABG (Dubé & Lauzière, 1997). Gold bearing quartz veins at the three locations are collectively known as the "A vein", and are typically located at (41 and 51 Zones) or near (04 Zone) the southeast limit of a sequence of highly deformed and brecciated graphitic schist of the WPG. The graphitic schists host the mineralisation and form the footwall of the CRFZ. Graphitic schist is in fault contact with highly strained chloritic schists and quartz-sericite mylonites farther up in the hanging wall structural succession. The protolith of these mylonites is difficult to ascertain, but they appear to be partly or totally retrograded PABG lithologies. Other veins (C vein) are present in the structural footwall and represent secondary lodes hosted by more competent lithologies (Dubé & Lauzière, 1997). The mylonite zone in the hanging wall of the A vein is mainly composed of strongly sheared quartz-feldspar porphyries, quartzo-feldspathic gneisses and banded sediments, all of which are partly to completely retrograded to greenschist facies.

The contact between the mylonite zone and the graphitic schist is a zone of strong hydrothermal alteration attributed to emplacement of the mineralised quartz veins (Dubé & Lauzière, 1997). In the 41 and 51 Zones, this is commonly represented by a 10-25 m thick chlorite-biotite-magnetite schist unit. A 5-12 m thick chlorite-calcite schist frequently exists immediately adjacent to the mineralised A vein in the hanging wall. The chlorite-calcite schist is much wider (up to 47 m) in the 04 Zone and the chlorite-biotite-magnetite schist zone is not as well defined. Graphitic schist units intercalated with chloritic schists are present up to 30-40 m into the hanging wall above the A vein interval and indicate additional structural complexity.

In the CRGD area, a continuous sequence of banded, highly contorted, folded and locally brecciated graphitic schist with intercalations of chloritic and sericite-carbonate schists and banded mylonites constitutes the footwall and host of the mineralised A vein. The banded mylonites are characterized by cm-wide siderite-muscovite-quartz-rich bands within graphitic chlorite-quartz-muscovite schist. The mylonites are commonly spatially associated with local Au-mineralised quartz veins, vein breccias (C vein) and stringer zones. Chloritic and carbonatized sheared mafic rocks and carbonatized pale green metasedimentary rocks are also present. Locally, less altered sections within the chloritic schist of the 41 Zone suggest that the protoliths in at least part of these schists are sills and dykes of the Windowglass Hill Granite (Dubé & Lauzière, 1997). Graphitic gouge zones up to 6 m wide that host the A vein contain cm-wide fragments of wall rock, quartz veins and sulphides in a fine-grained matrix and locally show high-grade values of Au, Ag, Pb and Zn.

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The graphitic schist unit becomes strongly to moderately contorted and banded farther into the footwall of the fault zone, but cm- to m-wide graphitic and/or chloritic gouge is still common. The graphitic schist unit contains up to 60% quartz or quartz-carbonate veins. At least three mineralised quartz breccias veins or stockwork zones are present in the footwall of the 41 Zone and these are termed the C vein. The thickness of the graphitic-rich sequence ranges from 20-70m but averages 50-60 m in the CRGD area (Dubé & Lauzière, 1997).

7.4 Geology of the Windowglass Hill Deposit

Mineralisation in the WGHD occurs predominantly in a set of flat-lying sulphide-rich quartz veins hosted in the relatively un-deformed Windowglass Hill Granite near the CRFZ. The Windowglass Hill Granite is well exposed on the southwest side of the Isle aux Morts River and intrudes WPG rocks (Figure 7.3, Figure 7.4 and Figure 7.5). The intrusion is elongate in shape, measuring 2.7 x 0.6 km on the western side of the Isle aux Morts River and 100 m wide x 3.4 km long on the eastern side, where it is cut by a strike-slip mylonite zone. The Windowglass Hill Granite is composed of pink, aphanitic alkali feldspar granite, with minor plagioclase phenocrysts and <2% biotite (Dubé & Lauzière, 1997).

Deformation in the Windowglass Hill Granite is generally weak, but can be heterogeneous. Strain is most intense in centimetre to metre wide zones of shearing that separate relatively un-deformed domains characterized by jointing and locally by northeast- to north-northeast-trending fracture cleavage. The high strain zones are best developed at the Windowglass Hill Granite-WPG contact in the schists.

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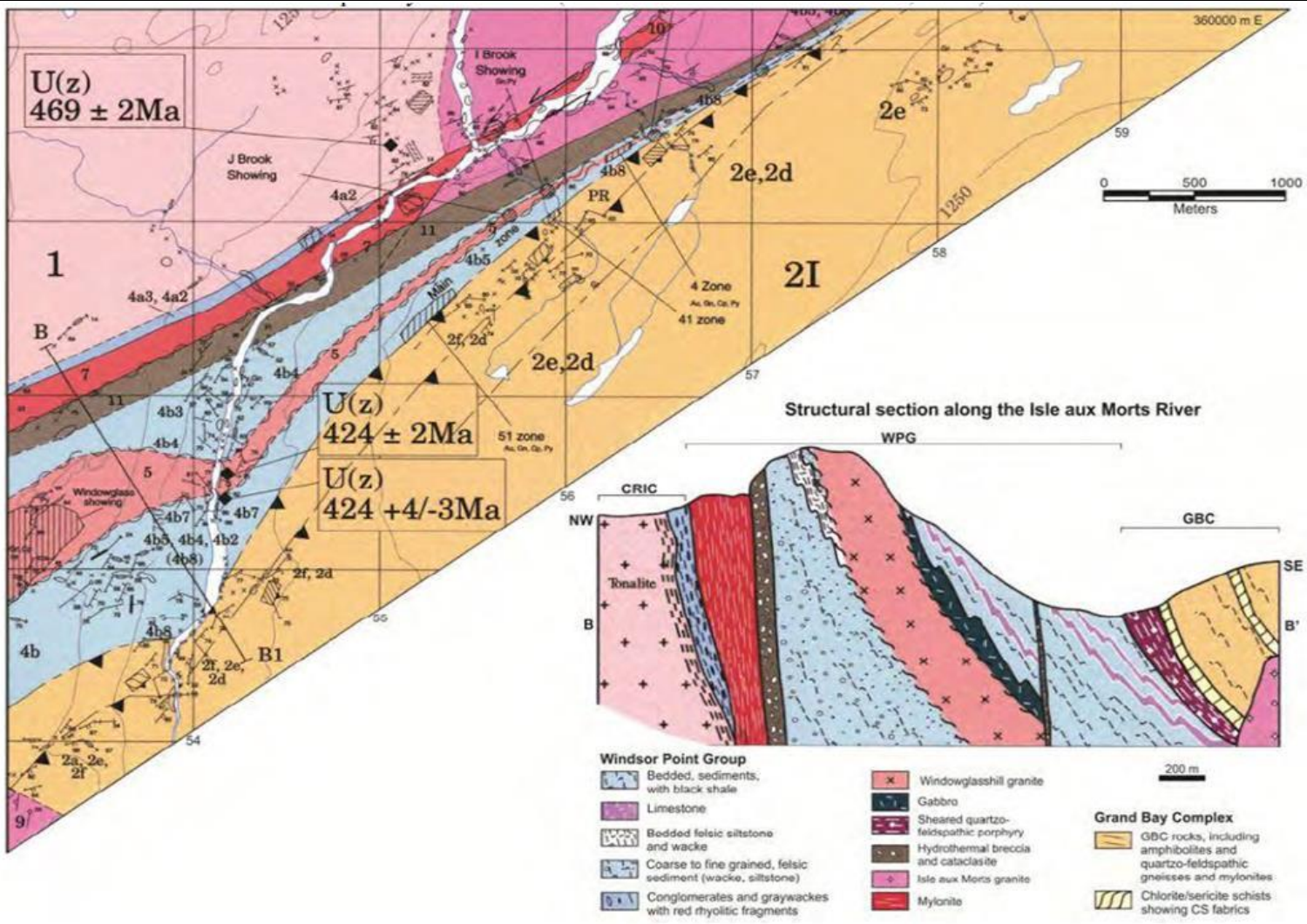

Figure 7.4 Structural Section of the Cape Ray Fault Zone (Dubé and Lauzière, 1995)

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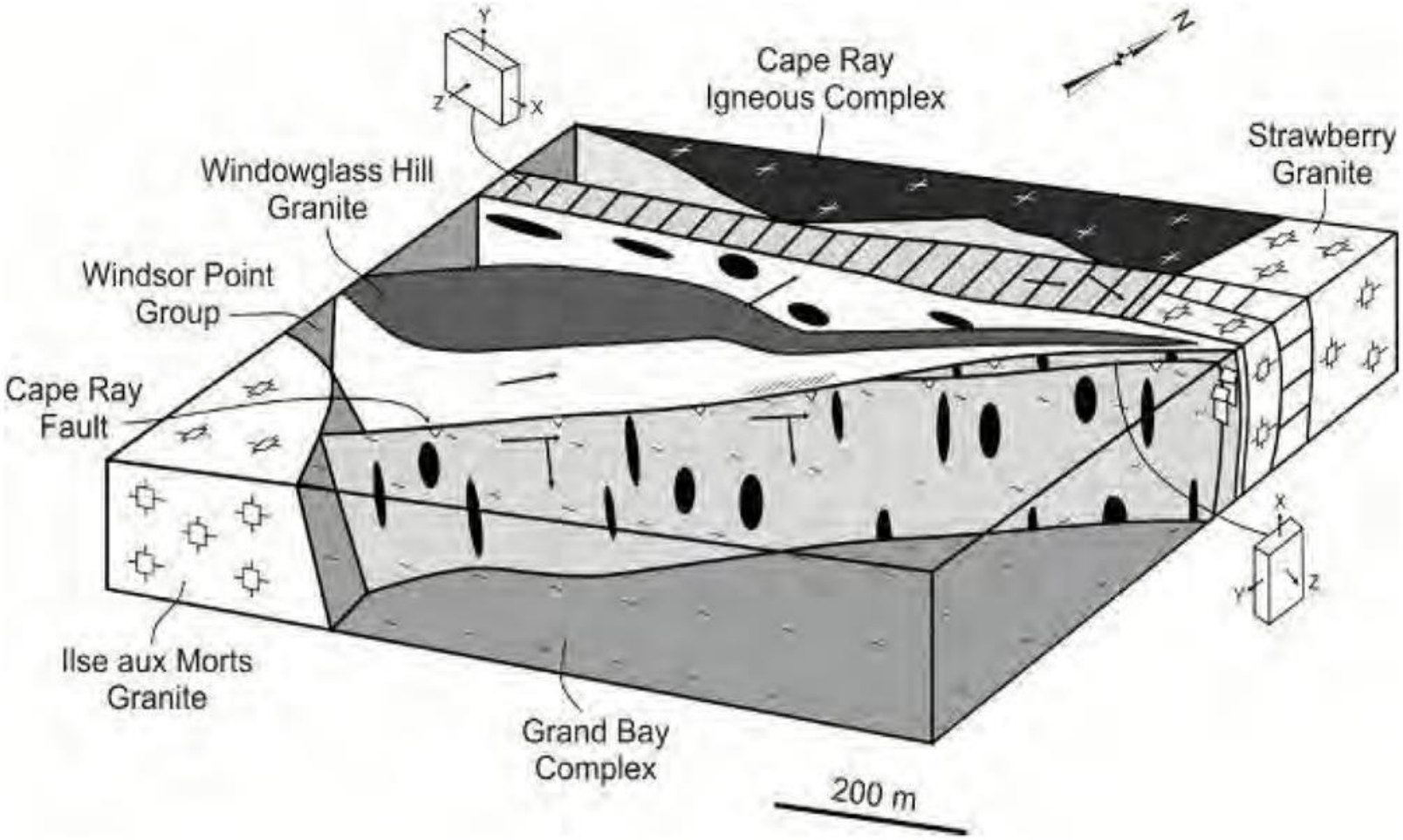

Figure 7.5 Block Diagram of Geological and Structural Elements (Dubé & Lauzière, 1996)

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7.5 Mineralisation

7.5.1 Cape Ray Gold Deposit

The CRGD consists of electrum-sulphide mineralisation that occurs in boudinaged quartz veins within an auxiliary shear zone (the "Main Shear") of the CRFZ. The boudinaged veins and associated mineralisation are hosted by chlorite-sericite and interlayered graphitic schists of the WPG (Table 7.1), with sulphides and associated electrum occurring as stringers, disseminations and locally discrete massive layers within the quartz bodies.

Table 7.1 Summary of Lithologies Structure and Mineralogy (modified after Dubé & Lauzière 1997)

Mineralised Zone	Cape Ray Gold Deposit (04, 41 & 51 Zones)	Windowglass Hill Deposit
Type of Mineral Occurrence	Quartz vein, vein breccia and fault fill veins (A and C veins)	Extension quartz veins; simple, multiple stages; veins up to 1m; banded structures and comb structures
Main Host Rock	Windsor Point Group: Graphitic schist	Windowglass Hill Granite; U-Pb age 424 ± 2 Ma
Zone Orientation and Geometry	3 mineralised zones within a northeast trending fault that dips moderately to the southeast (50-60º)	north-south veins; shallow, west dipping, associated with north trending shear zone
Mineralogy	-Electrum (Au, Ag (minor)); Au:Ag ratio 40:1 to 1:100; average 1:2.25 - 5-10% sulphides, chalcopyrite, galena, pyrite, sphalerite and minor pyrrhotite and arsenopyrite	Electrum (Au, Ag (minor)); Au:Ag ratio 1:1 to 1:350; average 1:99
Alteration	Moderate chloritization & carbonatization	Moderate pyritization, sericitization and local chloritization
Relative Timing	Syn- to late D3, pre S5 folding	Syn- to late D3

The 04, 41 and 51 Zones occur along a northeast-trending fault within the CRFZ system, which dips moderately (50-60°) to the southeast. These zones consist of complex tabular zones of quartz veins, fault gouge and wall rock fragments, range from several cm to a few m in width, and correlate laterally for up to 700 m along strike. In section, the 04 and 41 Zones show east-southeast to southeast plunges and locally show down-dip extension of up to 300 m (Dubé & Lauzière, 1997). The 51 Zone is not as consistently defined but both sub-horizontal and steeply plunging grade trends are present.

Within the CRGD, the A vein typically consists of milky white breccia veins with various proportions of angular clasts of wall rock (chlorite and graphitic schist) set in a matrix of quartz, and containing up to 40% sulphides (averaging 5-10%). The vein is typically up to 2 m wide with local wider pods. Fault gouge commonly borders mineralised veins and is superimposed on already highly strained rocks. The nature of the host rocks and superimposed deformation make

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this mineralised zone complex and variable in terms of vein continuity, width and grade.

Underground investigations by Dolphin (Arnold, 1988) within the 41 Zone noted that the A vein pinches and swells, and commonly branches out into smaller discrete veins or lenses. Similar observations were made by Wilton and Strong (1986) in a trench that exposed the A vein within the 51 Zone for over 61 m. They observed numerous quartz veins (up to 2.4 m thick and 21 m long) sub-parallel to the main fabric that show well developed pinch and swell structures. Wilton and Strong (1986) suggested that the podiform nature of the veins resulted from boudinage of a formally more continuous quartz vein. Outside of the main mineralised zone (30-50 m laterally), the A vein structure consists of either uneconomic quartz veining or fault gouge with little to no veining with traces of disseminated pyrite. Brittle deformation is superimposed on the veins as cataclastic features.

The C vein is typically located up to 30 m down-dip on the footwall of the A vein and is generally less deformed. Historical drill hole data indicate that the C vein is up to 15 m thick, and gold values are not as consistent or as high grade as those within the A vein (Dubé & Lauzière, 1997). Underground investigations of the 41 Zone showed the C vein to consist of a series of fault-fill mineralised quartz breccia intervals measuring up to 1.5 m in thickness. The transition between mineralised and barren quartz breccia veins is locally gradual, with the barren veins located in the central part of the mineralised quartz breccia. Fragments within the quartz breccia veins are typically angular and composed of graphitic schist and sericite-chlorite schist set in a matrix of hydrothermal quartz.

In the 51 Zone, the mineralised quartz is at the top of the graphitic schist unit within a gouge zone. The A vein is broken up into large lenses and breccia fragments of varying size. The graphitic schist in this zone varies from less than 15 m thick in the west to nearly 61 m thick in the east. Locally mineralised quartz veins and/or quartz stockwork intervals are found within the footwall schists. High grade mineralisation of good continuity is commonly restricted to the top of the graphitic unit (Ford, 1985).

Minerals of economic interest in the CRGD consist of varying amounts of galena, chalcopyrite, sphalerite and pyrite with associated electrum and free Au (Table 7.1). Pyrrhotite occurs as minor inclusions in other sulphide grains and the primary iron oxide is magnetite, mainly found in wall rock fragments rather than veins. In places, deformation has recrystallized galena and chalcopyrite, shattered the pyrite and removed sphalerite. All sulphides contain complex intergrowths of euhedral pyrite crystals, and sphalerite typically contains rounded chalcopyrite inclusions. Electrum occurs as minute grains generally less than 0.02 mm in size, but up to 0.8 mm and dominantly inter-grown with pyrite, and to a lesser extent with chalcopyrite, galena and sphalerite or some combination of these sulphides. Electrum also occurs as solitary grains within the quartz veins in regions of high sulphide concentrations. Paragenetic features indicate that pyrite formed first followed by simultaneous deposition of the other sulphide minerals (Wilton, 1983).

Electron microprobe analyses indicate an average FeS content of 5.14% in the sphalerite. Electrum contains 15-64 mole% Ag and Au:Ag ratios range from 0.55 to 5.83 in contrast to 0.61 to 0.76 for the WGHD (Wilton, 1983). Electrum grains with the highest Au:Ag ratios are generally

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isolated and not in contact with the sulphides. According to Wilton (1983), wall rocks to the electrum-bearing quartz veins experienced potassic alteration with light rare earth element depletion and minor elevation in chalcophile element concentrations. On the basis of sulphide mineral intergrowth, Wilton (1983) also estimated approximately 300°C to be the temperature of vein formation and concluded that sulphur isotope ratios indicate that sulphides are of magmatic origin and that Pb isotope ratios in galena have an oceanic crust/mantle isotopic signature. Based on field observations and detailed geochemical analyses, he also concluded that the electrum sulphide-bearing quartz veins were deposited by hydrothermal fluids evolved as a vapour phase from the Windowglass Hill Granite..

7.5.2 Windowglass Hill Deposit

Two sets of sulphide-bearing quartz veins at the WGHD cut the Windowglass Hill Granite, with the first set hosting the majority of the Au, Ag, Pb and Zn mineralisation. The mm- to cm-wide veins strike generally north-south, generally dip moderately (25°) to the west, parallel to a joint set in the granite, and are spaced from 1 cm to several tens of cm apart. The veins commonly show open space-filling textures, with idiomorphic quartz crystals developed sub parallel to a down-dip stretching lineation. Dubé and Lauzière (1997) concluded that vein development occurred during the D3 deformation event (see previous Table 7.1).

Pinching and swelling of the veins are common and the best sulphide mineralisation occurs in zones of maximum swell. Subhedral cubes of pyrite and coarse-grained galena, and anhedral masses of galena, chalcopyrite and rare sphalerite make up the sulphide component of the quartz veins. Within these veins, sulphide associations with elevated Au concentrations are as follows: (1) galena rich with pyrite and minor chalcopyrite; and (2) chalcopyrite-pyrite rich with minor galena. Haloes of 2-3% disseminated pyrite are present in the host granite, with best gold and sulphide mineralisation occurring in veins with the largest pyrite haloes (Wilton, 1983).

The second set of veins strike northeast and dip steeply southeast at approximately 80°, parallel to the main D1 fabric developed in schists surrounding the granites. These veins post-date those of the first set and are mineralised only where they intersect mineralised veins of the first set. Some mineralisation occurs in narrow linear zones that plunge at shallow angles to the southeast and correspond to lines of intersection between the two vein sets (Wilton, 1983).

Alteration haloes are best developed around the more extensively mineralised veins and intense alteration only occurs in very close proximity to the veins. The proximal alteration zone is defined by disseminated pyrite and moderate to intense sericitization of feldspar that gives a green colour and bleached aspect to the granitic rock. Sericite alteration occurs in both perthite and plagioclase crystals. Sericite associated with the mineralisation is generally coarser grained than the fine sericite attributed to regional metamorphism. Chloritization of feldspars is localized and appears to be associated with magnetite-bearing veins. Although alteration is present in close proximity to mineralised veins, deformation is limited and igneous textures are commonly preserved (Dubé & Lauzière, 1997).

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7.5.3 Timing of Mineralisation

According to Dubé and Lauzière (1997), the main Au-Ag mineralisation is post 424 ± 2 Ma, the age of the Windowglass Hill Granite that is cut by mineralised veins, post-peak metamorphism in the PABG (415 Ma), and post 407 ± 4 Ma, the cooling age of the hornblende as both the gneisses and the hornblende are retrograded at greenschist facies within the CRFZ structure hosting the gold deposits (see previous Table 7.1). The mineralisation also pre-dates the 386 ± 2 Ma Isle aux Morts Granite that cuts the Main Zone shear of the CRFZ. This time period appears to correspond to the development of D3 ductile structures, an interpretation that is also compatible with the structural setting of the mineralisation at the WGHD. Highly strained wall rock fragments enclosed within a mineralised quartz breccia vein in the CRFZ indicates that the mineralizing event is syn- to late-D3 ductile deformation (Dubé & Lauzière, 1997).

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8. Deposit Types

Descriptions within this section are taken directly from Ténière and Hilchey (2012). Extensive work by Dubé and Lauzière (1997) in the CRGD area includes a detailed discussion of the deposit style and possible analogies to Archean and younger gold deposits around the world. The following section summarizes the findings of Dubé and Lauzière (1997) and the reader is referred to this extensive work for greater detail.

8.1 Style of Mineralisation

The style of lode gold mineralisation in the CRGD has a number of characteristics in common with mesothermal gold deposits. The relationship of the different mineral zones with a major ductile fault zone, the nature of quartz veins, grade of metamorphism, and alteration style are all generally compatible with classic mesothermal lode gold deposits (Roberts, 1987). In addition, the temperature of the mineralised zone formation, or equilibration, estimated from arsenopyrite and other sulphide intergrowths appears to be around 300°C (Wilton & Strong, 1986) and is consistent with such an interpretation. However, the mineralogical assemblage within the mineralised zone, in particular the abundance of base metals, differs from typical mesothermal mineral assemblages, and the abundance of galena may suggest a lower temperature of formation.

The relatively high concentration of Ag seen in the CRGD is characteristic of gold deposits formed at shallower depths. Most mineralised zones at Cape Ray have elevated Ag contents compared to Au (see Au:Ag ratios in Table 7.1), as is typically seen in epithermal deposits (Dubé & Lauzière, 1997). Otherwise, the Cape Ray mineralised zones show few of the critical geological characteristics of epithermal deposits such as characteristic alteration mineral assemblages, crustiform vein textures, and formation in an extensional (volcanic) tectonic environment. Their structural and lithological characteristics are more analogous with greenstone belt deposits (mesothermal), although the limited amount of hydrothermal carbonate in the veins or the wall rocks contrasts with typical mesothermal gold deposits. Based on the above, it appears that the auriferous zones in the CRGD were formed at depths intermediate between these two hydrothermal environments (2-5 km depth, 200-300°C) (Dubé and Lauzière, 1997). A similar conclusion was proposed by Peters et al. (1990) for mélange-sediment hosted (slate to turbidite-hosted) gold deposits in the Hodgkinson goldfield in North Queensland, Australia. In that case, high Ag content may reflect the dominantly sedimentary nature of the rocks through which the fluids circulated and Dubé and Lauzière (1997) interpret a similar situation for the CRGD.

According to Dubé and Lauzière (1997), mineralisation at the WGHD is comparative to intrusion-related Korean-type gold-silver deposits (1:2 to 2:1 Au:Ag ratios), and to more silver rich epithermal deposits (1:10 to 1:200 Au:Ag ratios). The intensity and type of alteration, the banded structure of the veins, and some of the Au:Ag ratios are also similar to this type of deposit. Notable differences with the Korean-type deposits are the presence of abundant base metal sulphides and the more elevated silver content of some of the veins. Korean-type deposits are genetically associated with Early to Late Cretaceous granites and are characterized by a general deficiency of sulphide minerals (formed at 270°C and 1.25 km depth). Ag rich epithermal

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deposits associated with Late Cretaceous to Tertiary granites are characterized by different alteration (advanced argillic and silicification) and an extensional tectonic setting (deposition at <240°C at depth <0.75 km).

8.2 Analogy with Archean and younger gold deposits

Dubé and Lauzière (1997) note that the association of Au mineralisation with graphitic schists has been reported from Archean Au deposits such as the Hollinger and McIntyre mines in the Abitibi greenstone belt, where Au has precipitated at quartz vein margins injected within graphitic zones. They consider the CRGD to be potentially analogous with the Owl Creek deposit near Timmins, Ontario, where auriferous quartz veins are hosted by sheared graphitic schists that are interpreted as critical in the formation of the deposit. At Owl Creek gold occurs in epigenetic quartz veins and their pyritic wall rocks in two zones within a package of east striking, steeply north dipping, volcanic and sedimentary rocks. At the West Zone, 1.7 Mt of ore with a grade of 4.83 g/t Au (268,587 recovered ounces) were produced from an open pit centred on a wedge-shaped unit of basalt that occurs between two overturned, south facing units of greywacke and argillite (Coad et al., 2004). Basalt/greywacke contacts are locally marked by graphitic-carbonaceous argillite, strike-parallel faults and massive quartz veins, all of which are apparent in the Cape Ray area.

The CRGD also have some similarities to deposits of the Ashanti gold district in western Ghana, where mineralisation occurs in a major shear zone in Early Proterozoic rocks interpreted as a thrust fault juxtaposing tightly folded and metamorphosed metasedimentary rocks and less deformed and metamorphosed metavolcanic rocks (Dubé & Lauzière, 1997). The gold mineralisation occurs in shear zones characteristically associated with carbonaceous schists, and most of the mineralisation is hosted by quartz veins that contain significant amounts of base metal mineralisation such as galena, chalcopyrite, and sphalerite. As with the CRGD, mineralised bodies pinch and swell along both strike and dip. Notably, the Ashanti mineralised shear zone intervals are more extensive, with lengths up to 7.5 km and depth extents of 1.6 km or more. A comparison in deposit style and mineralisation can also be made with Paleozoic and younger rocks hosting significant gold deposits in structurally complex terranes. Dubé and Lauzière (1997) identified a gold-graphite association in the Ballarat goldfield in Australia, where auriferous quartz lodes are mainly restricted to an Ordovician stratigraphic horizon rich in graphitic slates, in the Otago schist of New Zealand that hosts numerous Au bearing quartz vein deposits, and in the Hodgkinson goldfield in Australia where Silurian-Devonian metasediments and mélange, including carbonaceous shale, host vein associated Au. Multiple injections of quartz and multiple shear zone movements are commonly encountered in these settings and, similar to Cape Ray, the quartz veins pinch and swell or occur in pods mixed with gouge within the host shear zones. Gold mineralisation also accompanied brittle deformation and is interpreted as contemporaneous with regional plutonism.

8.3 Tectonic-metallogenic model for CRFZ

Wilton (1983a) and Wilton and Strong (1986) proposed a genetic model in which the

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Windowglass Hill Granite exsolved a base and precious metal-rich vapour phase. The exsolution of this phase raised the internal pressure within the cooling magma and country rocks and hydraulically fractured brittle horizons such as the graphitic schists, which offered a structural, and possibly a geochemical trap for gold precipitation from the auriferous fluids slightly before the deformation stage. For reasons detailed below, Au and Ag mineralisation of the WGHD is unlikely to be genetically related to its granitic host. Textural features of the Windowglass Hill Granite indicate that emplacement took place relatively high in the crust, in a subvolcanic environment, prior to deformation related to the major thrusting event along the CRFZ. The veins are perpendicular to a stretching lineation (L3) interpreted as being produced during thrusting (D3), which suggests that emplacement of veining and Au-Ag mineralisation is associated with that deformation. The Windowglass Hill Granite, acting as a rigid body, would have been a preferential site for fracturing and extension, while the surrounding WPG rocks underwent ductile deformation, accommodating most of the strain. These fractures could have been conduits for hydrothermal fluid circulation resulting in Au-Ag mineralisation within the CRFZ during the main episodes of ductile deformation or during later phases of deformation. Dubé and Lauzière (1997) had separately noted that the Au-Ag mineralizing event in the Cape Ray area is contemporaneous with development of the ductile fabric produced by motion along the CRFZ.

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9. Exploration

Historical exploration and efforts have been outlined in Section 6 History. This section specifically details the efforts that Benton Resources Inc. has applied to the property.

9.1 2013 Exploration Program

From January 2013 to December 2013 Benton Resources Inc., completed an exploration program consisting of line-cutting, IP geophysical survey, prospecting/mapping, and geochemical rock/soil sampling. The line-cutting program totaled 65.4 line kilometres and the IP geophysical survey consisted of 18.1 line kilometres. The results of the 2013 IP survey are shown below in Figure 9.1 and Figure 9.2. A total of 96 rock samples and 588 soil samples were collected on the licence. A mini bulk sample was collected from an old trench site which exposed the 51 zone.

9.1.1 Prospecting/Mapping and Sampling Program

Benton personnel carried out the geochemical sampling over several periods between January and December of 2013. Samples were taken and locations recorded with hand-held GPS units.

A total of 96 rock samples (Figure 9.3) and 588 soil samples were collected within the licence 17072M (Figure 9.1). All rock/soil samples were assayed for gold by fire-assay at Eastern Analytical Laboratory Ltd. in Springdale, Newfoundland and a 30 element ICP was completed on all samples.

The majority of rock samples taken from the property were from the altered Windowglass Hill granite, quartz/quartz veins or quartz associated with granite, chloritic/graphitic schist and few from sediments. The selective grab samples ranged from trace up to 15-20% sulphides, mainly pyrite and galena with minor chalcopyrite, disseminated throughout the rock, but minor blebs of sulphide were noted. Results from the program returned highs of 69.7 g/t and 39.7 g/t gold from samples 118951 and 118761. Sample 118952 provided the highest silver value of 196.8 g/t. Generally, with the higher grade gold samples, analysis for silver and copper were also elevated.

An old trench (Harris 1980) which exposed the 51 zone was reopened by Benton in 2013 to collect a mini bulk sample. A total of 11 plastic pails were filled, each containing approximately 20 kg of rock chips. The pails were shipped to Met-Solve Laboratories in Langley, BC and tested for dense media separation (DMS) and gravity.

The soil sampling program was conducted on a cut grid at 60 and 120 metre line spacing and at 25 metre spacing along the individual lines. The program was very successful in outlining prospective target areas for both gold and silver. A total of 31 samples returned gold values greater than or equal to 100 ppb Au, with a high of 801 ppb Au for sample 69993..

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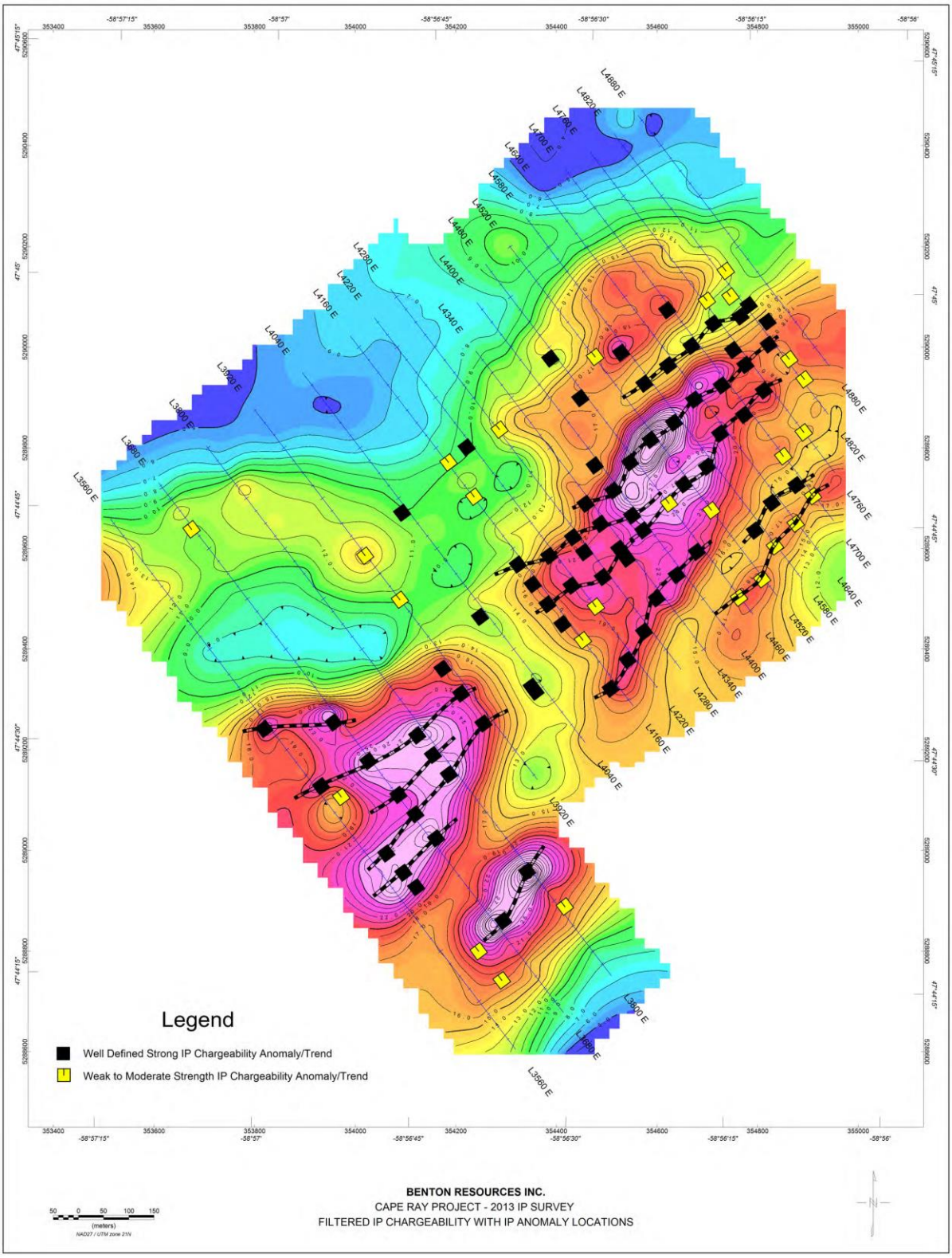

Figure 9.1 2013 IP Survey Result 1

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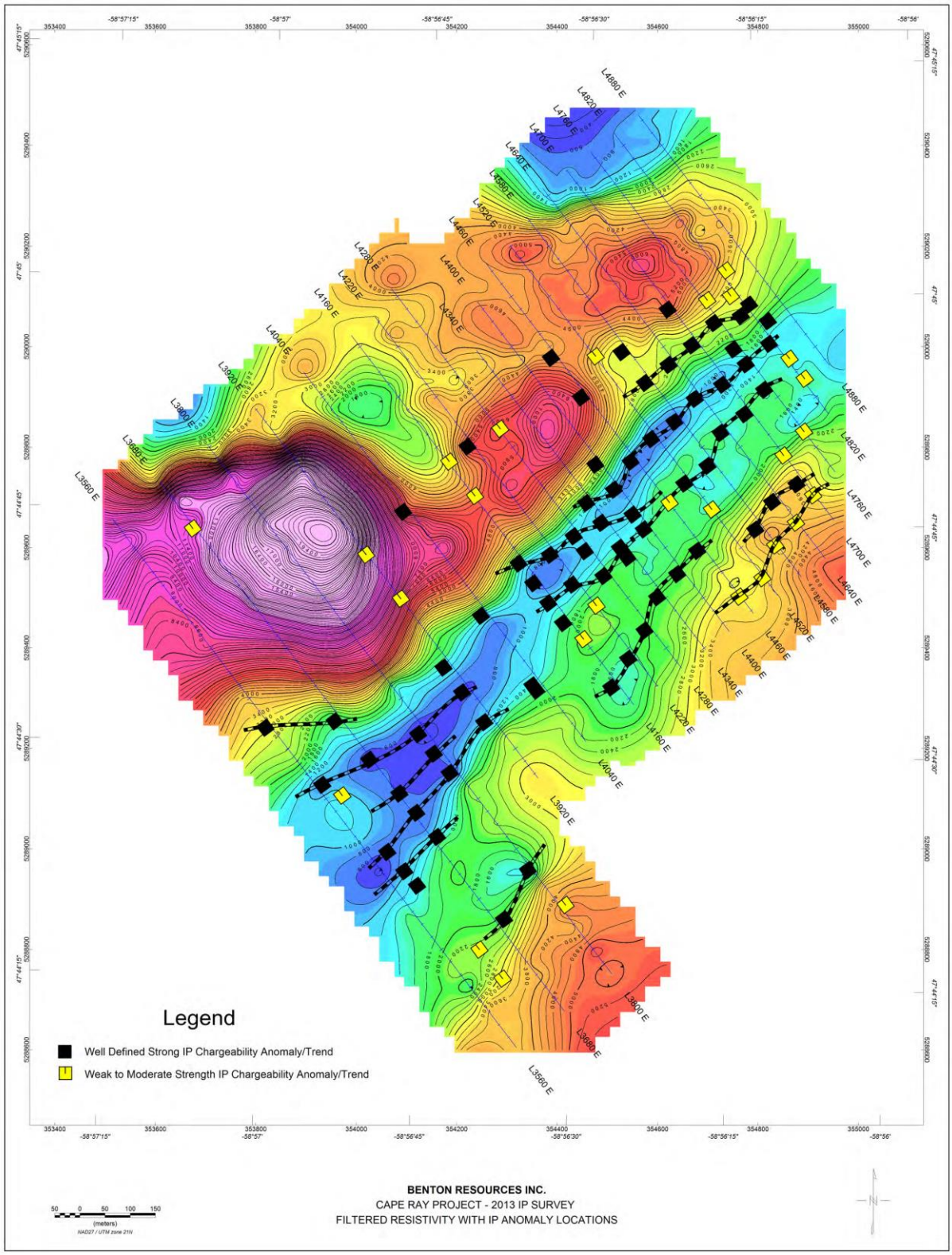

Figure 9.2 2013 IP Survey Result 2

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9.2 2014 Exploration Program

From January, 2014 to November, 2014, Benton Resources Inc. (Benton) of Rosslyn, Ontario, Canada, completed a diamond drill program, a bulk sampling program, a line-cutting program, a prospecting/mapping program, and a geochemical rock and soil sampling program based on previous exploration results on the Cape Ray Property.

9.2.1 Diamond Drill Program

Benton personnel Steve House, Steve Stares, Clint Barr, Paul Degagne, Nathan Sims, Calvin Keats and Todd Murray along with contractors Cabo Drilling Ltd. and West Bottom Drilling Ltd. carried out a diamond drill program over the period from June to October of 2014. A list of diamond drill holes and their locations which were completed in the 2014 drill campaign are shown below in Table 9.1.

Cabo Drilling Ltd. (Cabo) of Springdale, NL, carried out the first phase of diamond drilling using a Nodwell-mounted Boyles B15 diamond drill rig equipped to drill NQ sized core from June to August 2014. Drill moves were made via the Nodwell-mounted diamond drill rig. West Bottom Drilling Inc. completed the second phase of diamond drilling using a skid-mounted Duralite 500 diamond drill. Drill moves were completed using an excavator. Drill sites were spotted by Steve House, Clint Barr and Paul Degagne. Core was placed in wooden trays supplied by Cabo. Drill casings were left in the ground, capped and marked with wooden posts labelled with aluminum embossed tape. The collar coordinates were located with a Garmin GPSMAP78 unit by Steve House. Hole deviation was monitored with a Reflex instrument at nominal 60 metre intervals down the hole. All drill core was supervised/logged onsite by geologists, Steve House, Clint Barr, Paul Degagne and Nathan Sims. Assay samples were split using a rock saw by Cal Keats and Todd Murray, with half of the sample inserted in a plastic bag and securely sealed, and the other half returned to the core box. All core boxes were labelled with aluminum tags.

Samples were then sent to Eastern Analytical Laboratories Ltd. lab in Springdale, NL for sample analysis. All pulps were processed using Fire Assay for gold and an ICP analysis for another 34 elements. After the completion of the program, the drill core was stacked and left onsite at the Cape Ray property exploration camp.

A total of 1043 samples were submitted for analysis; 941 samples were of drill core, 51 were blanks (inserted by Benton personnel) and 51 were standards with a known Au content (inserted by Benton personnel) to ensure quality control. Of every 20 samples shipped to the assay lab there was one blank and one standard inserted as control.

Drill data was entered into a MS Excel spreadsheet. Drill logs and sections were created using GeoticLog 6 and Downhole Explorer software, respectively.

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Table 9.1 Drill Hole Collar Coordinates

Hole ID	UTM Northing	UTM Easting	Azimuth	Dip	Length	Area
PB14-378	5289320	354527	323	-45	112.00	Extension of 51 Zone
PB14-379	5289118	354222	323	-45	163.00	Extension of 51 Zone
PB14-380	5289204	354162	323	-45	125.00	Extension of 51 Zone
PB14-381	5289471	354124	323	-45	145.00	PW Zone
PB14-382	5289303	354239	323	-45	25.00	Extension of 51 Zone
PB14-383	5289239	354288	323	-45	226.00	Extension of 51 Zone
PB14-384	5289496	354393	323	-45	43.10	Extension of 51 Zone
PB14-385	5289768	354413	323	-45	208.00	PW Zone
PB14-386	5289433	354446	323	-45	230.00	Extension of 51 Zone
PB14-387	5289866	354566	322	-45	188.00	PW Zone
PB14-388	5289661	354576	322	-55	310.50	PW Zone
PB14-389	5289704	354576	323	-45	268.00	Extension of 51 Zone
PB14-390	5289891	354846	323	-45	186.00	Extension of 51 Zone
PB14-391	5289924	354891	323	-45	162.00	Extension of 51 Zone
PB14-392	5290009	354906	323	-45	125.00	Extension of 51 Zone
PB14-393	5291051	356077	322	-60	268.00	Zones 04 and 41
PB14-394	5291086	356143	322	-60	267.00	Zones 04 and 41
M14-01	5291308	356224	136	-65	87.00	Met hole on 04 Zone
M14-02	5291351	356189	136	-65	125.00	Met hole on 04 Zone
Total					**3,263.60**	

9.2.2 Cape Ray Drilling

A total of 19 holes were drilled at Cape Ray in 2014. All 19 drill holes were designed to test the potential for gold mineralisation in the Cape Ray area near existing outlined deposits (Figure 9.3). Three separate zones were targeted, the PW Zone (extension of Windowglass Hill zone), the extension of the 51 Zone and the 04 zone including 2 holes in between the 04 and 41 zones as well as two metallurgical holes on the 04 zone, based on previous soil/geophysical exploration results. Highlights of the 2014 drill results are displayed in Table 9.2 below.

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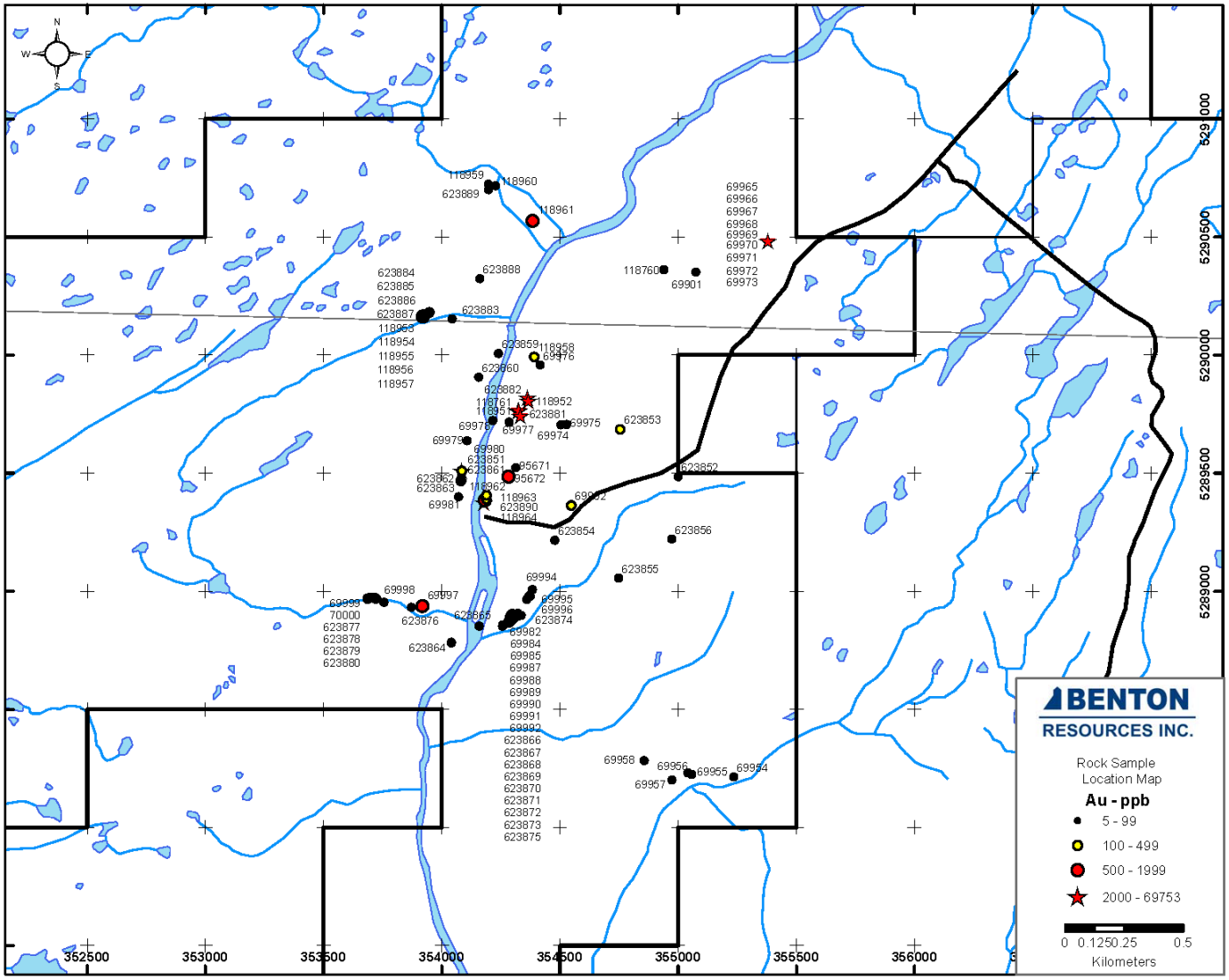

Figure 9.3 Rock Sample Location Map

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Table 9.2 Cape Ray Drill Highlights

Hole Number	Location	Azimuth (true)	Inclination	From (m)	To (m)	Width (m)	Au(g/t)	Ag(g/t)
PB14-378	Extension of 51 Zone – no significant assays	323	-45					
PB14-379	Extension of 51 Zone	323	-45	80.00	80.60	0.60	0.41	0.20
PB14-380	Extension of 51 Zone	323	-45	42.50	43.10	0.60	0.73	1.50
PB14-381	PW Zone – no significant assays	323	-45					
PB14-382	Extension of 51 Zone – no significant assays	323	-45					
PB14-383	Extension of 51 Zone	323	-45	187.40	188.00	0.60	1.49	0.20
	and			195.00	195.60	0.60	0.50	0.30
PB14-384	Extension of 51 Zone	323	-45	26.40	27.10	0.70	0.59	0.38
PB14-385	PW Zone	323	-45	61.00	139.50	78.50	0.42	2.00
	including			85.30	86.20	0.90	2.98	10.40
	Including			91.00	91.9	0.90	1.36	0.70
	Including			106.00	107.3	1.3	1.12	1.70
	including			107.30	107.55	0.25	25.38	82.90
	including			127.00	128.50	1.50	4.90	3.60
	including			134.50	135.60	1.10	1.63	14.20
PB14-386	Extension of 51 Zone	323	-45	134.00	135.50	1.50	0.54	0.70
PB14-387	PW Zone	322	-45	68.00	68.70	0.70	1.45	1.10
	and			77.65	121.00	43.35	0.83	4.00
	including			80.00	81.00	1.00	1.09	1.30
	including			87.30	87.93	0.63	1.50	32.50
	including			109.00	110.00	1.00	2.15	8.60
	including			117.50	118.00	0.50	27.53	80.00
	including			119.00	120.00	1.00	9.82	12.20
PB14-388	PW Zone – no significant assays	322	-55					
PB14-389	Extension of 51 Zone – no significant assays	323	-45					
PB14-390	Extension of 51 Zone	323	-45	52.40	53.00	0.60	0.47	1.20
PB14-391	Extension of 51 Zone	323	-45	111.90	112.90	1.00	0.86	0.80
PB14-392	Extension of 51 Zone – no significant assays	323	-45					
PB14-393	Zones 04 and 41	322	-60	131.2	132.2	1.00	2.18	5.2
	and			172.5	173.6	1.10	6.89	60.5
	and			254.6	257.2	2.60	0.57	13.0
PB14-394	Zones 04 and 41	322	-60	172.2	174.0	1.80	2.70	2.45
	and			182.0	182.5	0.50	2.20	5.20

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9.2.3 WGH Extension (PW Zone)

The PW Zone was defined based on a soil geochemical survey and prospecting followed by an IP geophysical survey. This zone is an extension of the Windowglass Hill Zone on the southwest side of the Isle aux Morts River, which is hosted in extensional, sub-horizontal quartz within the Windowglass Hill Granite.

Holes PB14-381, PB14-385 and PB14-387 to 388 were drilled to further define the PW Zone mineralisation. The best intercept was from hole PB14-387 with 0.5 m of 27.5 g/t Au and 80 g/t Ag. A wider mineralised interval grading 43.35m of 0.83 g/t Au and 4.0 g/t Ag was also intersected further uphole in PB14-387.

Hole PB14-385 returned a high gold value of 25.4 g/t Au and 82.9 g/t Ag over 0.25 m within a wider interval grading 0.42 g/t Au and 2.0 g/t Ag over 78.5 m.

9.2.4 Extension of the 51 Zone

At the 51 Zone, gold occurs mainly in galena and chalcopyrite-rich, fault-fill quartz breccia veins hosted by graphitic schist of the Windsor Point Group.

Drill holes PB14-378 to 380, PB14-382 to 384, PB14-386 and PB14-389 to 392 were drilled to determine the extension of the 51 zone based on a soil geochemical survey, prospecting and an IP geophysical survey. The best intercept was in hole PB14-383 with 0.6 m of 1.49 g/t Au and 0.2 g/t Ag. The sample was taken from a quartz vein which was 9.4 m thick, containing approximately 50% quartz vein material and chlorite schist. It contained 1-2% pyrite as disseminations, blebs, fracture fill and rare fine cubes. Three other quartz veined zones and 2 fault zones were also present in the hole, but did not return anomalous gold values.

9.2.5 04 and 41 Zones

At the 04 and 41 zones, mineralisation is mainly contained in boudined quartz veins within the graphitic and chloritic schists. Sulphides are mainly pyrite, chalcopyrite and galena.

Drill holes PB14-393 and 394 were drilled in between the 04 and 41 zones to determine continuity for the zones. The best intercept in PB14-393 was 1.10 m of 6.89 g/t Au and 60.5 g/t Ag. The best intercept in PB14-394 was 1.80 m of 2.70 g /t Au and 2.45 g/t Ag. Holes M14-01 and M14-02 were drilled for metallurgical purposes and all core was sent in for metallurgical analysis. Initial analysis was completed on the mineralised zones of the metallurgical holes with a best intercept of 13.0 m of 13.4 g/t Au, 22.0 g/t Ag, 0.82% Pb, 0.20% Cu and 0.98% Zn from hole M14-01 and 18.0 m of 15.2 g/t Au, 13.3 g/t Ag, 1.0% Pb, 0.2% Cu and 0.77% Zn from hole M14-02

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9.2.6 Bulk Sampling and Trenching Program

Two bulk sampling programs and one trenching program were completed by Benton, all supervised by Steve House. One bulk sampling program was in October 2013 and samples were sent to Met-Solve Laboratories for metallurgical testing. The second bulk sampling program, in October 2014 included drilling two diamond drill holes and sampling the complete core from the holes. Trenching and blasting was also completed in 2014 by Manuals Construction from Port aux Basques, NL. The trench samples were collected for use in the metallurgical testing program that was completed by ALS Laboratories in Kamloops, BC.

9.2.7 Line-cutting Program

A line-cutting program was completed by S.C.I. Exploration Ltd. from Miles Cove, NL, both supervised by Dennis Rowsell (Figure 9.4). The program started and was completed in September of 2014, 36.6 line kilometres were completed on three licenses. License 17072M had 7.33 line kilometres cut, 1.28 line kilometres for license 21559M and 28 line kilometres for license 9939M.

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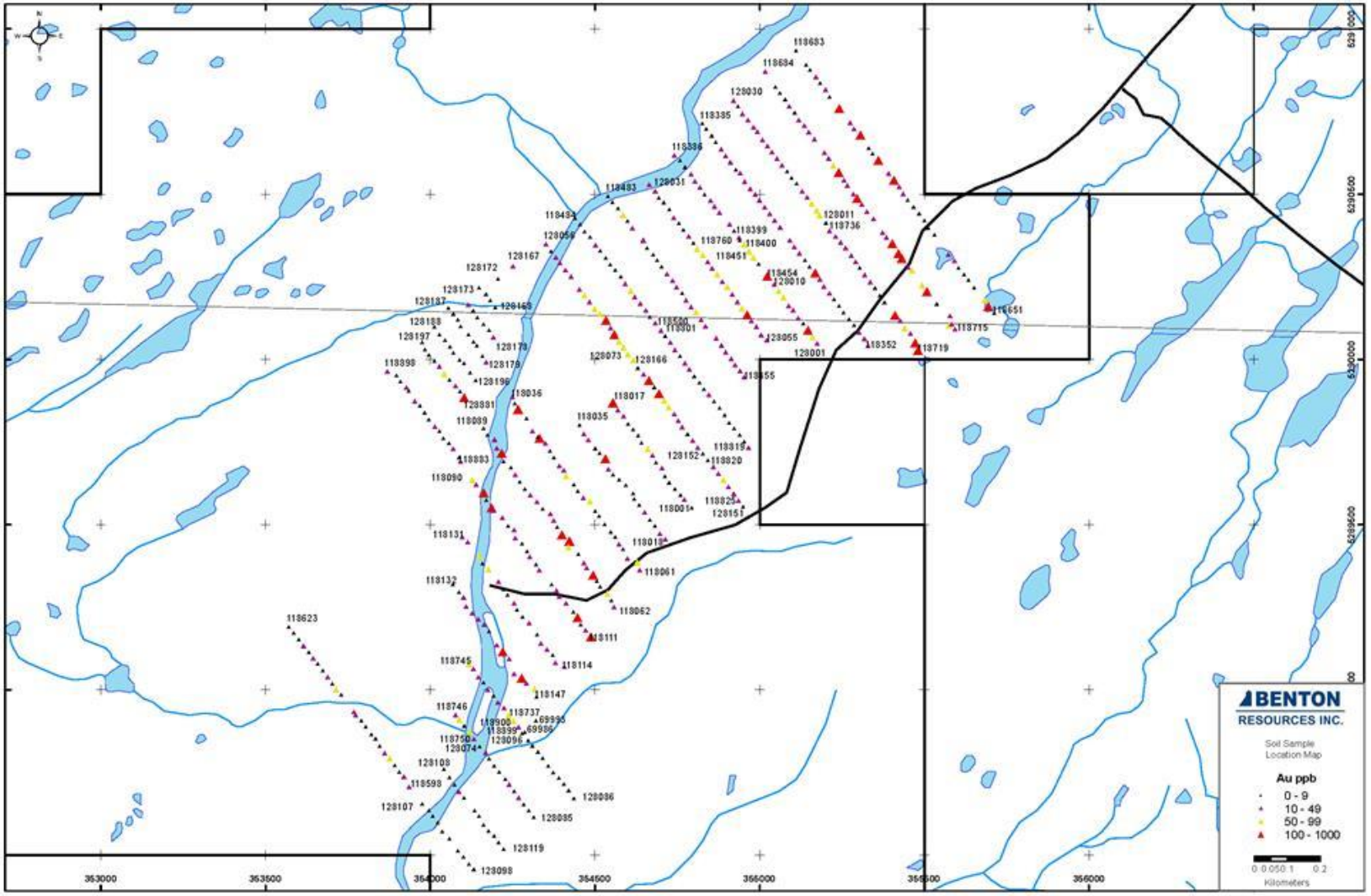

Figure 9.4 Soil Sample Location Map

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9.3 2015 Exploration Program

In August 2015 Manuals Construction from Port aux Basques, NL was contracted to trench along the 51 and 41 zones. The exposed zones were washed and geological mapping conducted. Mapping and trench photos can be viewed on the Benton web site at http://www.bentonresources.ca/

The 51 zone was exposed for a strike length of approximately 200 meters and a total of 104 saw-cut samples, ranging in length from 0.4 to 1.2 meters, were cut at intervals ranging from 7.5 to 15 meters in 23 separate section lines within the exposed trench.

The trenching efforts over the 41 zone consisted of two separate trenches over a strike length of approximately 125m, although flooding restricted exposure to about 85m. A total of 100 saw-cut samples, ranging in length were cut at intervals ranging from 3.0 to 16 meters in 15 separate section lines within the exposed trench. All saw cut samples were sent to Eastern Analytical Ltd. Springdale, NL for assay. A complete table of results for the trenching program is listed below.

Individual gold and silver assays were cut to 40g/t and 80g/t respectively, prior to calculating weighted composites.

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Table 9.3 Zone 51 Channel Sampling Results

Zone	Channel section	Section Length (meters)	Au (g/t)	Au cut (g/t)	Ag (g/t)	Ag cut (g/t)	CU%	Pb%	Zn%
51	A	1.50	9.19	9.19	5.10	5.10	0.03	0.08	0.04
51	B	2.30	1.98	1.98	33.69	33.69	0.03	0.84	3.29
51	C	3.00	0.57	0.57	2.20	2.20	0.01	0.07	0.04
51	D	2.70	6.79	6.79	45.36	33.87	0.17	0.85	0.03
51	E	5.70	6.14	6.14	12.43	12.43	0.17	0.13	0.11
51	F	5.35	8.70	8.70	27.12	27.12	0.12	0.90	0.23
51	G	6.80	11.27	11.27	46.37	26.89	0.22	0.95	0.35
51	H	5.20	8.55	8.55	46.19	39.46	0.23	1.96	0.77
51	I	6.10	0.95	0.95	18.78	18.78	0.06	0.63	0.14
51	J	4.30	11.32	11.32	32.78	32.43	0.21	1.40	0.60
51	K	4.90	5.46	5.46	11.51	11.51	0.06	0.53	0.27
51	L	5.00	2.92	2.92	6.77	6.77	0.06	0.30	0.41
51	M	3.70	0.83	0.83	4.23	4.23	0.03	0.16	0.05
51	N	2.50	0.02	0.02	1.22	1.22	0.01	0.01	0.03
51	O	3.00	5.85	5.85	24.69	24.69	0.32	1.37	0.11
51	P	3.60	3.37	3.37	13.84	13.84	0.28	0.27	0.10
51	Q	5.10	3.20	3.20	18.48	18.48	0.10	0.66	0.53
51	R	5.20	13.57	13.57	57.93	39.33	0.50	1.96	0.20
51	S	3.40	14.24	14.24	164.99	44.64	1.34	9.25	0.25
51	T	2.70	13.44	11.52	43.29	32.40	0.35	1.88	1.20
51	U	3.50	14.42	13.96	72.29	44.52	0.79	1.55	1.16
51	V	2.90	2.12	2.12	8.07	8.07	0.13	0.17	0.07
51	W	3.20	10.94	10.23	56.63	27.08	0.87	1.00	0.30

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Table 9.4 Zone 41 Channel Sampling Results

Zone	Channel section	Section Length (meters)	Au (g/t)	Au cut (g/t)	Ag (g/t)	Ag cut (g/t)	Cu %	Pb%	Zn%
41	A	8.40	3.50	3.50	14.14	14.14	0.12	0.31	0.27
	incl.	4.10	5.78	5.78	15.88	15.88	0.17	0.20	0.25
41	B	5.80	0.40	0.40	4.15	4.15	0.03	0.15	0.06
41	C	2.70	0.24	0.24	1.52	1.52	tr	0.04	0.07
41	D	6.00	0.68	0.68	2.39	2.39	0.03	0.14	0.13
41	E	1.60	31.17	25.34	69.47	37.93	0.86	2.80	0.39
41	F	12.60	0.34	0.34	2.23	2.23	0.02	0.02	0.01
41	G	3.40	0.59	0.59	7.84	7.84	0.05	0.10	0.04
41	H	6.20	0.08	0.08	4.14	4.14	0.01	0.04	0.01
41	I	4.00	0.41	0.41	3.78	3.78	0.04	0.11	0.07
41	J	3.00	0.34	0.34	9.57	9.57	0.10	0.10	0.04
41	K	7.00	0.22	0.22	2.99	2.99	0.03	0.04	0.03
	incl.	1.00	1.15	1.15	3.20	3.20	0.03	0.05	0.11
41	L	7.00	1.31	1.31	12.81	12.81	0.03	0.51	0.03
	incl.	2.00	3.48	3.48	35.90	35.90	0.04	1.66	0.01
41	M	7.90	1.94	1.94	3.98	3.98	0.02	0.05	0.02
	incl.	4.90	3.08	3.08	5.28	5.28	0.02	0.07	0.03
41	N	7.70	2.28	2.28	9.61	8.16	0.12	0.42	0.57
	incl.	3.70	4.73	4.72	19.08	16.07	0.23	0.86	1.16
41	O	7.50	tr	tr	0.79	0.79	0.01	0.02	0.03

10. Drilling

Drilling completed to date on the Cape Ray property is illustrated in Figure 10.1, a plan view that also shows the access road and the property outline. The first drill hole completed on the property was in 1977 by Riocanex. Between 1977 and 2006 there have been 512 holes drilled totalling 77,008.54 metres. Table 10.1 summarizes the drilling done to date by company and identifies which deposits the drilling was done on, while Table 10.2 summarizes the drilling by deposit. In 2014 Benton has drilled 19 holes totalling 3204.6 metres. This most recent drilling was not used for resource estimation purposes as reported in Section 14.

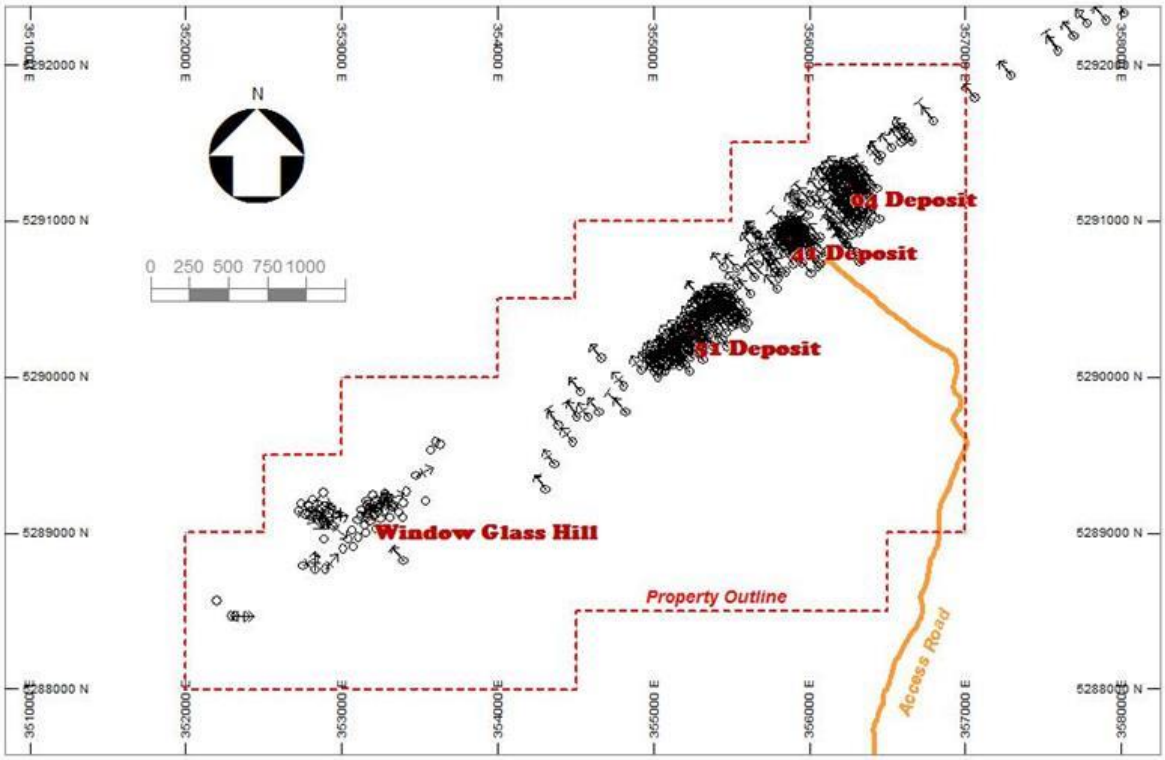

Figure 10.1 Plan of Diamond Drilling Completed to Date on the Cape Ray property

Companies that have been involved in exploring the property, including conducting the various diamond drilling campaigns, have included Rio Tinto Canada Exploration (Riocanex), Mascot Gold Mines Ltd., Dolphin Exploration Ltd., Tenacity Gold Mining Company, Terra Nova, Cornerstone Capital Resources Inc. and , most recently, Benton.

Core sizes were either BQ (36.5 mm) or NQ (47.6 mm). All historical sampling was done using mechanical splitting.

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Table 10.1 Summary of Diamond Drilling by Previous Operators for the Cape Ray Property

Year	Company	Number of Holes	Total Length (m)
1977-1980	Riocanex	172	24,053.54
	04	26	4,638.72
	41	40	6,441.77
	51	50	7,262.50
	Other	41	5,011.04
	WGH	15	699.52
1986-1987	Mascot Gold Mines	71	13,910.36
	04	10	3,780.18
	41	13	3,550.91
	51	6	1,274.98
	Other	8	1,856.40
	WGH	34	3,447.88
1988-1989	Dolphin Exploration	118	23,309.15
	04	36	9,390.24
	41	8	849.39
	51	67	12,063.72
	Other	7	1,005.79
1989-2003	Tenacity	86	7,574.72
	Big Pond	41	3,917.15
	Isle aux Morts	45	3,657.57
1996	Royal Oak Mine s	17	963.33
	51	16	765.16
	Other	1	198.17
2003-2004	Terra Nova	20	4,677.44
	04	19	4,496.34
	41	1	181.10
2004-2006	Cornerstone	28	2,520.00
	51	2	358.20
	WGH	26	2,161.80
Grand Total		**512**	**77,008.54**

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Table 10.2 Summary of Diamond Drilling by Deposit for the Cape Ray Property

Deposit	Number of Holes	Total Length
04	91	22,305.49
41	62	11,023.17
51	141	21,724.56
Windowglass	75	6,309.20
Big Pond	41	3,917.15
Isle aux Morts	45	3,657.57
Other	57	8,071.40
Grand Total	**512**	**77,008.54**

Drill hole deviation measurements were carried out by all of the operators on the Cape Ray property. Riocanex initially used the Tropari system and later holes used down hole single shot or multi-shot systems such as Sperry Sun or Reflex. Higher, but systematic deviation rates characterise deeper BQ size holes completed by Riocanex, while NQ holes generally remained on their intended inclination and azimuth. Cornerstone data do not typically show problematic deviation issues, but relatively soft hanging-wall lithologies such as graphitic schist present potential for higher deviation rates in deep holes cutting through the stratigraphy (Ténière & Hilchey, 2012).

Evaluating available resources for precious metal deposits, especially gold, is hampered by a number of risks. These include:

I. "High grade" actually being represented by minute quantities of a particular precious metal.
II. The use of smaller diameter drill core make representative sampling of such minute quantities difficult, especially when only a half of the core is actually submitted for analysis.
III. The use of manual splitting rather than the use of a diamond saw commonly results in biased sampling as more or less material than intended is included in the actual sample.
IV. The reliance of using only a 30 gram sample is based on the assumption that the material being analysed has been thoroughly homogenised. Ductile metals, such as gold, typically fail to be homogenised and thus the 30 gram sample commonly can be biased high or low.

As precious metals are difficult even under the best of conditions to be seen by the naked eye and as they may or may not be associated with other minerals that are easily recognised it is imperative that sampling not be selective. All drill core should be sampled to ensure mineralisation is not missed.

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The historical drilling done on the Cape Ray suffers from all of the above, in particular, the non-continuous sampling based solely on visual characterisation of the core. As a result gold-bearing rock may not have been sampled. By necessity all non-sampled intervals in-between sampled intervals must be assigned a zero grade which can introduce a negative bias.

Based on the drilling data compilation and validation processes, plus visual core assessments carried out during the site visit on September 22-25, 2014, Nordmin is of the opinion that documented drilling and related procedures such as drill hole location surveys, down hole orientation surveys, core logging, core sampling and sample handling carried out by these companies met industry standards of their respective periods and that associated results, as compiled and validated, are acceptable for resource estimation purposes.

Therefore the author is of the opinion that the sample preparation techniques, security and analytical procedures used were adequate.

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11. Sample Preparation, Analyses and Security

Riocanex did selective sampling of drill core based on geological criteria such as visible mineralisation or the presence of quartz veining. Samples were generally no greater than 1.5 m in length. The core was split on site with one half being placed into bags for analysis and the remaining half being returned to the core trays. The bagged samples were labeled and shipped to X-Ray Assay Laboratories (XRAL – now SGS Canada) and analysed for Au and Ag. If sulphides were apparent, Cu, Pb and Zn were also determined. Au and Ag were analysed by fire assay with an atomic adsorption finish method (FA-AA) and detection limits of 0.34 g/t for Au and 6.9 g/t for Ag. Cu, Pb and Zn were analysed by the X-Ray fluorescence method (XRF) with a detection limit of 0.01%. Riocanex also performed check analyses of select samples using 1/8th of the reject sample from X-Ray Assay Laboratories, which were sent to Swastika Laboratories Ltd. of Swastika, Ontario for analysis. Some of the archived drill core was moved to the Pasadena core facility upon Riocanex's withdrawal from the Cape Ray project in the early 1980's, where it is presently available for viewing (Ténière & Hilchey, 2012).

For the Dolphin drilling programs the samples were analysed using the fire assay (FA) method with a gravimetric finish to evaluate for Au and Ag; final results were reported in troy ounces per short ton (1 troy ounce/ton = 34.28 g/t). The screen metallic's method was used for selected drill core samples suspected of containing visible or coarse Au. All drill core samples were shipped directly to Vangeochem Lab Ltd. of North Vancouver, BC, TSL Laboratories of Mississauga, ON, or Eastern Analytical Ltd. for preparation and analysis. All laboratories were independent of Dolphin. In addition to precious metal assays, samples containing >1000 ppb Au were analysed for Cu, Pb and Zn (Ténière & Hilchey, 2012).

The Royal Oak drill core was logged and split on site; core samples shipped to Eastern Analytical Ltd for FA-AA analysis. Samples were also analysed using a 9-element inductively coupled plasma and mass spectrometer (ICP-MS) package that included Cu, Pb and Zn. After completion of the drilling program, the core was moved and stored at the Hope Brook mine site. None of the core has survived over time and none is available for viewing. Royal Oak reporting does not include details of sampling methodology for their Cape Ray drilling program (Ténière & Hilchey, 2012).

No information is available for the sampling and analysis methods used by Tenacity Gold Mining Company, Terra Nova and Cornerstone Capital Resources Inc.

Based on review of available documentation as summarized above, Nordmin is of the opinion that details of quality control and quality assurance methods employed by Riocanex, Dolphin and Royal Oak during the 1978 to 1996 periods on the Cape Ray property are not well documented and cannot be assessed in detail, but nothing in the data indicates that anything but industry standard practices at that time were not being followed.

Therefore the author is of the opinion that the sample preparation techniques, security and analytical procedures used were adequate.

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12. Data Verification

The Cape Ray project has utilised historical drilling dating back as far as 1977. Of the 512 holes on record a total of 93, or 18%, are located in the core storage facility operated by the Newfoundland Department of Natural Resources (NLDNR) and located in the town of Pasadena. Twenty of the holes are stored on site in cross piles.

Data verification has consisted of examining six holes representing a random selection of those available and locating a selection of holes in the field where the casing had not been removed. None of the available core is in good enough shape to allow a reasonable resampling through the collection of additional ¼ core samples (Figure 12.1). However, hole PB79-135 had been recently resampled by Benton Resources in early 2014 and the assay certificates were made available to Nordmin. A review of the certificates confirmed proper procedures had been followed including the use of blank samples and duplicates as well as standards. The core remaining in the core boxes shows that samples were collected by cutting the available core in half using a diamond saw. Figure 12.2 a) and b) show the comparison of the re-assays with the original assays on record. It is the opinion of Alan Aubut, the QP responsible for this report, that this work verifies the validity of the original sampling and analyses and that those results are suitable for use in resource estimation.

Figure 12.1 Examples of Core Available at the Core Storage Facility. a) PB79-135 and b) PB79-142.

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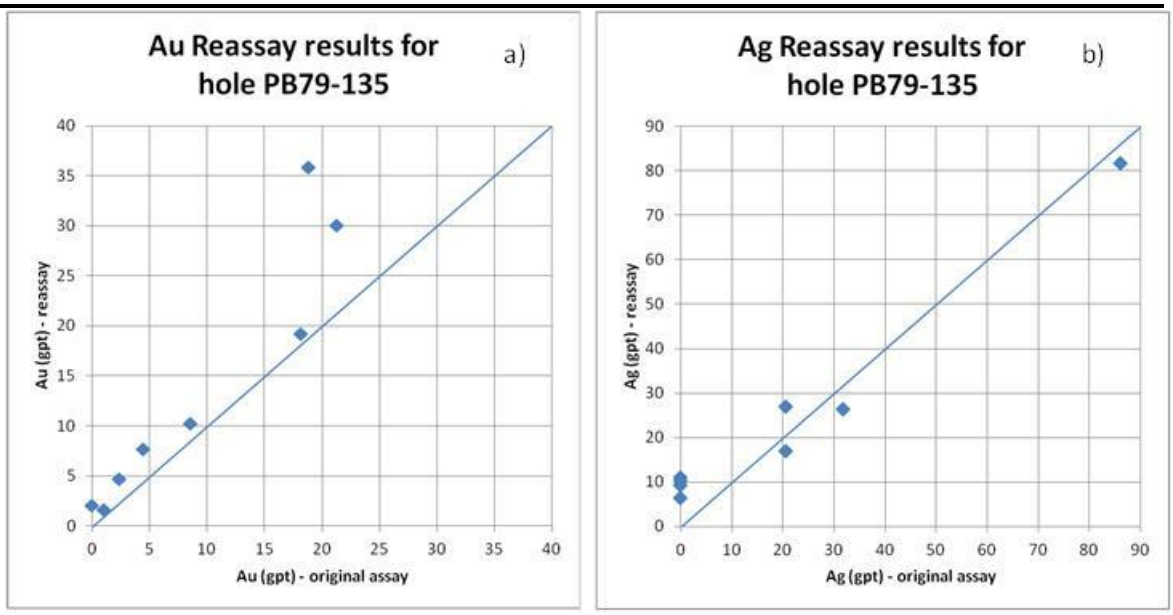

Figure 12.2 Comparison of Re-assays of Samples from Hole PB79-135. a) Au and b) Ag

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13. Mineral Processing and Metallurgical Testing

There are historical and recent bench test campaigns which were conducted on the ore bodies in the Cape Ray fault zone which have been summarized in the following:

13.1 Metallurgical Testing

1981, Rio Algom retained Lakefield Research of Canada Ltd. (Lakefield) to conduct metallurgical testing on a bulk sample from the Cape Ray 41-A vein (Wyslouzil & Yen, 1981).

1989, Dolphin Explorations Ltd., wholly owned by Corona Resources Ltd., retained Lakefield for bench testing on a composite made from Cape Ray 51 and Cape Ray 04 deposits drill core rejects (Salter & Williamson, 1989).

2013, Benton Resources commissioned Met-Solve Laboratories Inc. in Langley, BC for test work on dense media separation (DMS) and gravity with a trench sample from Cape Ray 51 deposit (Grewal & Lum, 2014).

2014, Nordmin Engineering Ltd. selected ALS Laboratories of Kamloops, BC, under partnership with Benton Resources, who conducted tests consisting of whole ore flotation, whole ore leach and gravity recoverable gold on Cape Ray 04 deposit, Cape Ray 51 deposit and a grab sample from a stockpile drawn from the Cape Ray 41 deposit (Sloan & Roulston, 2015).

Previous studies looked at recovering all valuable minerals with a high grade lead concentrate or recovering the gold and silver only. Economics of the day swayed the test program one way or another. The 2014 test work for the Nordmin and Benton partnership looked at whole ore flotation, whole ore leach and gravity recoverable gold as the baseline test work.

13.2 1981, Lakefield Research for Rio Algom

Calculated head grade of a sample taken from Cape Ray 41 A vein was 4.53 g/t Au, 24.4 g/t Ag, 0.32% Pb, 0.53% Zn and 0.28% Cu. The sample was subjected to 15 cyanide roll tests at a grind size of 86.8% passing 200 mesh (74 µm). Gold extraction was 97% and silver was 40% at a cyanide consumption rate of 0.7 kg/t and lime consumption of 1.4 kg/t. (Wyslouzil & Yen, 1981).

Three whole ore bench flotation tests were completed to produce a lead concentrate as illustrated in Table 13.1.

Table 13.1 1981 Lakefield Bench Scale Flotation Tests

Concentrate Grade					Recovery				
Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Cu (%)	Au (%)	Ag (%)	Pb (%)	Zn (%)	Cu (%)
45	1477	52.8	2.1	5.1	6.9	46	73.7	4.3	12.0

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An additional test of flotation of cyanide residue yielded 97% gold recovery and 84% silver recovery. Settling tests of cyanide leach residue displayed settling rates of 0.13-0.15 m/tonne of dry solids. (Wyslouzil & Yen, 1981)

13.3 1989, Lakefield Research for Corona Resources Ltd.

Samples for this campaign were made up of six drill core rejects from both the Cape Ray 04 and Cape Ray 51 deposit. A composite sample was created with the calculated head grade of 13.4 g/t Au, 30.8 g/t Ag, 1.41% Pb, 1.04% Zn, and 0.24% Cu (Salter & Williamson, 1989). The results of this testing campaign are as follows:

- The sample was subject to 12 cyanide roll tests at a grind size of 86% passing 200 mesh (74 µm).
- Gold extraction was 97% with a cyanide consumption rate of 0.6 kg/t and lime consumption rate of 1.0 kg/t lime.
- Settling test results of cyanide residue were 0.35 m2/tonne/day.
- Locked cycle tests were conducted to establish if recycling pre-aeration solution and barren solution would have an adverse effect on leach extraction. Once equilibrium was achieved 96.2% gold extraction was observed at a cyanide consumption rate of 0.4 kg/t.
- Cyanide destruction test revealed that both total and free cyanide levels can be reduced to less than 1 mg/L.

13.4 2013, Met-Solve Laboratories for Benton Resources

In 2013, Benton Resources sent a bulk trench sample from Cape Ray 51 deposit to Met-Solve Laboratories in Langley, British Columbia to investigate the potential for pre-concentration using dense media separation and gravity recovery. The calculated head grade of the sample was 8.0 g/t Au, 42.1 g/t Ag, 2.05% Pb, 0.68% Zn, and 0.23% Cu.

The sample was subject to heavy liquid separation to determine the specific gravity (SG) cut point of the sample at two different crush sizes (-10 mm and -6.7 mm); dense media separation (DMS) at two different SG cut points (2.83 and 2.93) and gravity concentration on products. A Bond ball work index test was completed and is summarized in Table 13.9 (Grewal & Lum, 2014).

13.4.1 Met-Solve Dense Media Separation Results

The dense media separation (DMS) testing was conducted using Sepro's Condor Multi-Stage pilot unit. The sample was crushed to -19 mm and screened at 1.3 mm. The +1.3 mm size fraction was subjected to the DMS. At the 2.93 cut point, the unit recovered 55.4% of the lead and 26.7% of the gold in a 5.8% mass yield. Lowering the SG cut point to 2.83 increased the recovery to 62.9% and 29.9% for lead and gold respectively, but also increased the mass yield to 9.1% resulting in the lower grades as illustrated in Table 13.2 (Grewal & Lum, 2014).

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Table 13.2 Met-Solve Dense Media Separation

Method	Products	Mass Yield	Pb Grade (%)	Pb Recovery (%)	Au Grade (g/t)	Au Recovery (%)
DMS Only	**Sink 1**	5.2	20	52.8	37.3	24.5
SG 2.93	**Sink 2**	0.6	9.3	2.6	32.3	2.3
	Total	5.8	19.0	55.4	36.8	26.8
	Undersize	15.2	2.6	20.2	9.0	17.2
	Floats	79.0	0.6	24.3	5.7	56.1
	Calc Head	100	2.0		8.0	
DMS Only	**Sink 1**	7.6	15.1	58.1	35.2	26.5
SG 2.83	**Sink 2**	1.5	6.3	4.8	22.6	3.4
	Total	9.1	13.7	62.9	33.1	29.9
	Undersize	15.0	2.6	20.0	9.0	13.4
	Floats	17.0	0.4	17.0	7.5	56.7
	Calc Head	100	2.0		10.1	100

13.4.2 Met-Solve Gravity Results

A two stage process was used to determine gravity recoverable gold. The first gravity stage used a Falcon L40 to produce a high grade gold product similar to batch recovery. The second stage used a Falcon L40 operating at high mass yields to simulate a continuous product. The Met-Solve Analytical Table was used to upgrade the product. Total recovery is summarized in Table 13.3. There is no value for lead, as gold was analysed to extinction by fire assay (Grewal & Lum, 2014).

Table 13.3 Met-Solve Gravity Results

Method	Products	Mass Yield	Pb Grade (%)	Pb Recovery (%)	Au Grade (g/t)	Au Recovery (%)
Falcon Only	Gold Conc	<0.01			127.2	68.3
	HMY Conc*	10.9	5.7	69.7	11.6	20.0
	Total	**10.9**	**n/a**	**69.7**	**n/a**	**88.3**

HMY = High Mass Yield

A combination of DMS with gravity was tested due to the low gold values reporting to the DMS sinks fraction. Gravity was conducted on the -1.3 mm fraction (screened prior to DMS) and the floats fraction. Met-Solve cautions that the mass yields for the gold is much lower than would be observed in the plant, due to the nature of the testing equipment. This combined test is summarized in Table 13.4.

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Table 13.4 Met-Solve Results from Staged DMS and Gravity Test work

Method	Products	Mass Yield	Pb Grade (%)	Pb Recovery (%)	Au Grade (g/t)	Au Recovery (%)
DMS+Gravity	**Sink 1**	5.2	20.0	55.9	37.3	24.3
SG2.93	**Sink 2**	0.6	9.3	2.8	32.3	2.2
	Conc Fines	0.4	43.0	9.8	139.8	9.3
	Conc Float	1.3	14.0	9.4	68.4	46.6
	Total	7.5	19.5	77.9	56.1	82.4
DMS+Gravity	**Sink 1**	7.6	15.1	60.6	35.2	32.9
SG2.83	**Sink 2**	1.5	6.3	5.0	22.6	4.2
	Conc Fines	0.4	43	9.6	139.8	9.0
	Conc Float	1	11.1	5.9	78.6	40.6
	Total	**10.5**	**14.6**	**81.1**	**51**	**86.7**

13.5 ALS Laboratories for Benton/Nordmin Partnership

Four samples, collected in September and October 2014, were sent to ALS Laboratory in Kamloops, BC. The Cape Ray 04 sample was composited of approximately 40 m from two drilled cores collected specifically for metallurgical testing. The composited philosophy for this sample was to maintain a higher grade for testing purposes. The Cape Ray 51 deposit sample was composited from a trench sample. Buckets were collected, assayed and a deposit grade sample was created. The Cape Ray 41 sample was composited from grab samples collected from a 3,000 tonne stockpile that was mined from an underground ramp and has been sitting on surface since 1989. Head grades are listed in Table 13.5 (Sloan & Roulston, 2015).

Table 13.5 ALS Assayed Sample Head Grades

	Au g/t	Ag g/t	Pb %	Zn %	Cu %	Fe %	S %
Cape Ray 04	11.5	12.0	0.87	0.58	0.17	4.2	0.89
Cape Ray 51	6.8	14.0	0.24	0.10	0.21	4.9	0.95
Cape Ray 41	2.6	21.5	0.58	0.34	0.23	5.6	1.32

13.5.1 ALS Flotation Summary

The Cape Ray 04 and the Cape Ray 51 composite samples were tested for flotation response. Grind size for the samples was 80% passing 95 µm for 04 and 80% passing 98 µm for the 51 deposit. The goal for this campaign was to determine if a separate bulk copper/lead concentrate and zinc concentrate could be possible for a potential market.

Bulk rougher recoveries for Cape Ray 04 were 96% for gold, 90% for silver, 96% for lead, 45% for

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zinc and 87% for copper. Zinc recovery in the zinc circuit was 52% because of the higher recovery in bulk rougher. Optimizing conditions could change the reporting stage for zinc and depress more in the bulk circuit (Sloan & Roulston, 1989).

Overall rougher and cleaner recoveries for the 51 deposit 95% for gold, 89% for silver, 60% for lead, 52% for zinc and 92% for copper.

Reagent scheme included zinc sulphate and sodium cyanide for primary grind and depression of sphalerite with 3418A as the copper and lead sulphide collector. Copper sulphate was used to activate the sphalerite in the zinc circuit, with SIPX as the collector. Lime was used for pH adjustment with targets of 9 in the bulk circuit and 11 in the zinc circuit.

A single batch cleaner test was completed on both the 04 and 51 samples, with the grind size and reagent suite mentioned above. Only a bulk flotation test was performed on the 51 sample and no flotation work was performed on the 41 sample due to the probability that the stockpile sample had been oxidized.

Both Cape Ray 04 and 51 showed good recovery for gold in the bulk rougher stage. For 04, this value was a 91% recovery and for the 51 sample, 75% gold was recovered (Sloan & Roulston, 2015).

13.5.2 ALS Gravity Summary

For each of Cape Ray 04, 51 and 41 a 30 kg sample was subjected to a gravity gold recovery test (GRG) using a lab scale Knelson concentrator with a 100 g cone. Three iterative sizes were used of 665 µm, 216 µm 59 µm and the concentrate was assayed to extinction. For the three samples, gravity recovery was between 73 and 86%. See Table 13.6 for the cumulative final stage recovery of gold and silver. Silver recovery was not as good with a range of 33-49% for the three samples (Sloan & Roulston, 2015).

Table 13.6 ALS Gravity Results, Concentrate Grades and Overall Recoveries

		Au grade	Ag grade	Au Rec	Ag Rec
	Wt%	g/t	g/t	%	%
Cape Ray 04	1.1	725.0	472.0	80.2	48.9
Cape Ray 51	1.0	586.0	633.0	86.2	38.9
Cape Ray 41	1.2	235.0	572.0	73.3	32.9

13.6 ALS Cyanide Leach Summary

Each sample as subjected to bench scale bottle roll cyanide leach test on whole ore. The samples were sparged with oxygen and lime was used for pH adjustment targets of 11-11.5. The samples were leached for a total of 48 hours with the liquor sampled at hour 2, 6, 24 and 48. Grind sizes (K80) were between 95-105 µm. (Sloan & Roulston, 2015).

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Initial sodium cyanide concentrations of 1,000 ppm were used for all samples. However, due to the higher consumption rates of the 51 deposit, three additional tests at 750 ppm, 500 ppm, and 250 ppm were conducted to observe the effect on gold and silver extraction. Table 13.7 shows the results of each of the tests. At 24 hours, there is a greater than 96% extraction of gold for all samples except the 250 ppm concentration. Gold and Silver extraction rates are summarized in Table 13.8

Table 13.7 ALS Gold and Silver Extraction

Cape Ray Deposit	NaCN Con'n ppm	Au Extraction at time (hour)				Ag Extraction at time (hour)			
		2	6	24	48	2	6	24	48
04	1000	36.0	91.2	96.9	97.6	38.0	65.0	70.6	72.2
04	1000	38.0	89.8	97.6	97.9	39.7	66.7	72.8	73.9
41	1000	55.9	89.1	96.9	97.0	35.6	42.8	48.9	51.9
41	1000	41.5	84.0	96.1	96.8	32.8	42.0	48.0	50.9
51	1000	25.3	85.5	97.9	98.9	34.1	51.9	59.2	63.6
51	1000	27.6	86.2	98.3	98.7	35.2	51.5	59.0	62.2
51	750	12.5	92.0	99.3	98.8	24.6	52.3	60.1	63.1
51	500	5.9	78.4	99.4	98.9	15.7	48.2	58.8	62.4
51	250	4.1	10.1	74.1	97.9	2.5	18.5	48.1	57.6
41	500	26.5	81.4	96.4	97.7	28.4	44.0	51.8	55.9
51	500	9.2	76.4	97.9	98.4	21.2	50.5	59.9	62.9
04	500	31.8	81.8	97.8	95.8	36.8	62.4	71.2	73.6

The highest silver extraction was with the Cape Ray deposit 04 at 70-74%, and the lowest was the Cape Ray 41 stockpile sample at 50-52% extraction. The Cape Ray deposit 51 achieved 62-64% extraction, even at the lower sodium cyanide concentrations. The last three tests in Table 13.7 include the addition of 200 ppm $PbNO_3$ to aid in increasing the silver recovery but did not show any significant effect. (Sloan & Roulston, 2015).

Reagent consumption rates for all the tests are listed in Table 13.8 below. Optimization work would be required to determine the appropriate relationship between leaching kinetics and cyanide consumption depending on individual sample or a blending strategy. Recoverable gold extraction with this test work is consistent with the test campaign conducted on the 41 deposit in 1981 for Rio Algom and a blend of 04+51 deposit 1989 for Corona Resources.

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Table 13.8 ALS Cyanide and Lime Consumption Rates

Deposit	NaCN Concentration ppm	NaCN Consumption kg/t	Lime Consumption kg/t
41SP	1000	1.4	0.7
41SP	1000	1.6	0.7
51	1000	2.3	1.2
51	1000	2.4	1.2
51	750	2.2	0.8
51	500	1.9	0.9
51	250	1.2	1.2
41	500	1.3	0.9
51	500	0.8	0.8
04	500	1.6	0.8

13.7 Comminution

Work indices have been calculated through the different testing campaign history and are summarized in Table 13.9 below. The ALS samples labeled as 41 index and 51 index were rocks of greater than 3" collected from the trench to be tested for crusher work index and abrasion index.

Table 13.9 Comminution Tests

	Sample	Bond Work Index	Bond Abrasion Index	Crusher Work Index
Sample Campaign	Description	kWh/tonne	Ai	kWh/tonne
1989 Lakefield	04 & 51 composite	15.6 *		
2013 Met-Solve	51	14.1		
2014 ALS	41 index	14.1	0.026	8.4
2014 ALS	51 index	14.6	0.185	7.1
2014 ALS	04	14.5		
2014 ALS	41	15.0		
2014 ALS	51	14.9		

13.8 Mineralogical Analysis

The 04 sample and the 51 sample were both subjected to Particle Mineral Analysis (PMA) using QEMSCAN at the ALS laboratory.

Chalcopyrite is the primary copper bearing mineral with minor amounts of bornite and chalcocite and trace amounts of covellite and tennantite/enargite (Sloan & Roulston, 2015).

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The sulphide content measured 2.7 wt% for the 04 and 2.2 wt% for the 51 deposit. Liberation of the copper minerals in the two-dimensional field was 41% for 90 µm K80 primary grind for the 04 deposit. This was higher for the 51 deposit which showed 70% liberation for copper minerals at 111 µm K80. This indicates that there is availability for optimization of the target grind size should flotation be the primary recovery method, as a target of 50-55% liberation is recommended for performance.

In the Cape Ray 04 composite, there was evidence of chalcopyrite disease, which is when very fine grains of chalcopyrite are dotted within a sphalerite particles making liberation of both minerals difficult at all grind sizes. Galena liberation was 55% for the sample.

13.9 Discussion

It is the author's opinion, based solely on geological and mineralogical principals, that the tests done to date have been reasonable and that they have resulted in favourable results that have successfully demonstrated that processing should not present any insurmountable challenges.

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14. Mineral Resource Estimates

14.1 Mineral Resource Estimation

Updated mineral resource estimates were completed for the Cape Ray 04, 41, 51, and Windowglass Hill gold-silver deposits belonging to Benton Resources Inc. With the exception of the Windowglass Hill deposit for which Inverse Distance was used, estimation was done using Multiple Indicator Kriging. Following is a description of the general process used for all of the deposits with specific details provided for each deposit later in this section.

14.1.1 Software Used

The software used in the modeling process, including data preparation was CAE Studio, Release 3.23.52.0 for the 04, Datamine Software Studio, Release 3.24.25.0 for the 41, 51 and the Windowglass Hill deposits.

14.1.2 Drill Hole Database

Initial data are contained within a set of Excel tables provided by Benton (master_collar_march 26 2014.xlsx, master_assay_march 24 2014.xlsx, master_litho_march 24 2014.xlsx and master_survey_march 24 2014.xlsx). These four files were used to construct a Microsoft Access database (Cape_Ray_Drillhole_Database-June 2014.accdb). A number of validation checks were done on this database and any identified issues were fixed using the original logs as control. The data were then imported into Studio as four tables: cr_collars.dm, cr_assays.dm, cr_litho.dm and cr_survey.dm. The collar file was then processed using the CAE Studio 3 CDTRAN routine to convert UTM coordinates to local coordinates. The survey table was also processed, using EXTRA, to change the azimuths to local grid azimuths. For those samples with only partial assays any missing values were set to 0. A lookup table was created summarizing all lithological codes and using the DECODE routine the lithology table was reprocessed. Using the collar, assay, lithology, and survey files as input the process HOLES3D was used to create two de-surveyed 3D drill holes files: cr_holes.dm (holes in the world coordinate system) and cr_holes_lg.dm (holes in the local coordinate system). Then, based on a code assigned identifying each of the 6 deposits known on the property, subsets were copied out for each of these deposits.

14.1.3 Compositing

Composited samples are weighted by specific gravity (SG) as it is a close approximation of density (mass per unit volume). The samples were composited to standard 1 m intervals using the CAE Studio 3 process COMPDH. The COMPDH process starts the composites at the beginning of the selected data interval and leaves any remainder at the end of the interval. This results in most holes having one sample with a length less than the established composite length. For grade estimation purposes, drill composites are treated like point data (i.e. their length is not used), thus the need to composite to a standard sample length to eliminate any sample bias. To avoid bias from a very short sample being treated the same as a standard sample any sample lengths that were less than 40% of the composite length were rejected.

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14.1.4 The Use of Unfolding

Mineral deposits typically vary in thickness along strike due to the non-uniform nature of the original deposition environment. Primary and secondary structural modifications also produce variations in strike and dip as well as thickness. The Cartesian coordinate system makes modeling of the natural geological chemical distribution within a mineral deposit difficult. To ensure that all interpolation takes place within a given geological domain, the domain is unfolded to a planar slab to make variogram calculation and grade interpolation easier. After interpolation has been carried out the samples are re-arranged to their original positions. This unfolding process first requires the generation of unfold strings that are used by CAE Studio 3 as a guide. These strings also include between section and within section tag strings to further constrain the unfolding process.

The unfolding routine used is based on a “proportional” concept, under which hanging wall and footwall surfaces of the domain are made flat and parallel to one another. The true along strike and down dip distances are retained but the across dip distances are first normalised to the distance across as a proportion of the total distance. Then this normalised value is multiplied by the average thickness of the mineral domain.

After being composited to uniform sample lengths the samples were unfolded using a custom script. Using another custom script the unfold string file was processed further this routine checks and validates the strings. The composited sample files and the validated unfold string file are then used as input to the CAE Studio 3 UNFOLD routine. The output files contain the samples in unfolded co-ordinate space. All subsequent processing was done on these files and utilized the new coordinate system consisting of UCSA, UCSB and UCSC (Across the Dip, Down the Dip and Along the Strike).

14.1.5 Block Size Determination

The block size used for resource estimation is usually a function of SMU, or Smallest Mining Unit, and is determined by taking into consideration the type of equipment that may be used during mining as it has a direct impact on the degree of selectivity that can take place. There is no point using a block size smaller than the smallest unit that can be physically mined selectively (usually a blast round). Another factor that needs to be considered is the degree of sampling detail. If samples are large and/or spaced far apart a small block size would be inappropriate.

14.1.6 Indicator Kriging

The purpose of block modeling is to provide a globally unbiased estimate based on discrete sample data. Geostatistical methods rely on mathematically modeling the autocorrelation of a regionalized variable, using variography. Then, using these mathematical models, weights are derived. These weights are applied to the samples used to derive the estimates while at the same time minimizing the estimation variance.

When the grade distribution is highly skewed such distributions often result in spatial estimation bias where a few outliers representing the high-grade tail of the distribution are difficult to

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characterize. These have a significant impact on both the mean and variance for the grade population even though they only represent a small fraction of it. Since traditional linear estimation methods, including Ordinary Kriging, often misrepresent these outliers resulting in a smoothing of the overall variability and therefore in over-estimation, some other approach is needed. An Indicator approach has proven successful in addressing the problem of highly skewed populations.

An Indicator approach involves coding the data into binary groupings using a series of grade cut-offs. For example, if a grade is above a certain cut-off it is assigned a code of 1 for that cut-off. Similarly, if below a cut-off, it is assigned a code of 0. Each cut-off corresponds initially to the grade of the upper limit of each portion of the sample data representing approximately 9.1% of the total metal. By multiplying the mean for each cut-off interval by its probability determined by Indicator Kriging and then summing them we can reconstitute the grade. Additional cut-offs may be required to better reflect changes in the distribution.

Once the parameter files needed for Multiple Indicator Kriging are constructed using a custom script is used to actually carry out the Multiple Indicator Kriging process. Each cell in the block model is discretised using a matrix of 3 x 3 x 3 points in the ABC (unfolded) coordinate system. The Kriging functions are interpolated at each discretisation point using the same search volume as the nearest neighbour interpolation, based on the grade variogram results. In case of local low sample density a nested search is implemented (the estimation is repeated on those blocks not estimated in the previous run and with the ranges increased to ensure enough samples are selected).

Estimated indicator functions range between 0 and 1. In theory, the interpolated values should not be higher than 1 or lower than 0. In practice, Ordinary Kriging can generate negative weights. These negative weights may cause the interpolated values to exceed the maximum or minimum values in the initial data set. It is important to check the magnitude of this problem. If deemed to be within acceptable tolerances then all of the negative values are set to 0.0 and all values above 1 are set to 1, during post processing of the indicator values (done using another custom script).

In addition to the previous check, it is important to ensure that the estimated proportion below one cut-off cannot be greater than the estimated proportion below a higher cut-off (Isaaks and Srivastava, 1989). For example the proportion below 1 g/t Au should be less than the proportion below 2 g/t Au. These order relation problems are to be expected when the indicator functions have different variograms but should not be more than approximately 1% of all of the estimates.

Post processing involves taking the weight proportion of material in each grade class and multiplying by the arithmetic average grade for each class calculated from the nearest neighbour model (grade class means). For the last class the median of this last class is typically used.

14.1.7 Block Model Validation

Verification of grade estimation is carried out in two ways: visually, and statistically. In the case of a visual check, interpolated estimates are loaded into sections and plans along with the original borehole data. Using contrasting colour schemes grades were tested. Any major

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discrepancy between the original information and the estimated block was analysed for possible processing error.

Major discrepancies were also looked for between the statistics of the sample composites, nearest neighbour model (de-clusterised statistics) and the ordinary kriged model. Specific statistics checked include reproduction of the global mean, as established by nearest neighbour modeling, and ensuring that all blocks were estimated.

14.1.8 Volume Variance Correction

The "averaging" process that goes on during interpolation within the block model tends to reduce the variance from its original level. Overall the mean for the entire population remains unaffected. However, since a cut-off grade is used to separate the above- and below-cut-off populations, their specific means are now affected due to this homogenization, or smoothing, of individual estimates. The interpolated mean can be lower or higher than the original mean depending upon whether the cut-off grade is above or below the original mean.

Regression methods such as Kriging may result in an over-smoothing or under-smoothing of the grade variability producing a block grade distribution with a variance that is lower or higher than expected. This expected variance can be calculated using Krige's relationship, which states that the dispersion variance for the samples within the deposit is the sum of the dispersion variance of samples within the blocks and the dispersion variance of the blocks within the deposit (for a more detailed explanation see Appendix D).

14.2 Cape Ray 04 Deposit

14.2.1 Resource Estimation Methodology

14.2.1.1 Data Validation

Core-drilling data were supplied as Microsoft Excel tables that included collar information, assays, lithology information, down hole survey information and specific gravity data. Sibley Basin Group compiled the data into a Microsoft Access database and completed data validation. Once validated this information was imported into CAE Studio 3 as four tables: a collar file, an assay file, a lithology file, and a survey file. As the data supplied were in UTM coordinates, a transformation was completed to make a copy of the drill hole file in local grid coordinates based on the collar location of hole PB80-156. The collar for this hole has a World Coordinate System (WCS) location of 356005.19E, 5290825.26N with a Local Coordinate System (LCS) location of 334.61W, 136.37S. Using the CAE Studio 3 HOLES3D routine a desurveyed drill hole file, cr_04_holes_lg.dm, was created. This drill hole file was last updated on July 14, 2014.

The CAE Studio 3 desurveying routine, HOLES3D, does a rigorous set of validation checks including checking for duplicate borehole numbers, missing survey data and overlapping sample intervals. If present, it generates a summary report with a list of all errors encountered. This file was checked to determine if any errors occurred and where appropriate corrections were made.

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Once it had been confirmed that no errors remained the drill hole file was then used for subsequent steps.

There are no density or specific gravity values in the data set provided, but a previous resource estimate for the project area (Ténière & Hilchey, 2012) had specific gravity (SG) values determined on 44 samples. From these an average value was calculated from 6 samples that fell below the 25th percentile and used as a global value. This same value of 2.84 g/cm3 has been used for this project.

14.2.1.2 Geological Domains

Experienced geologists had coded each rock unit based on core logging description. All of the holes are inclined and most intersected at least some portion of the deformation zone that hosts the gold mineralisation. Construction of the resource block model was controlled by building a wire frame that was then used to isolate related samples. No cut-off was used to limit the extent of this mineral envelope. The envelope for the mineral domain (see Figure 14.1) extends from an elevation of 296 metres above sea level down to a maximum depth of 59 metres below sea level. It is not a geological envelope as the deformation zone hosting the mineralisation cuts stratigraphy.

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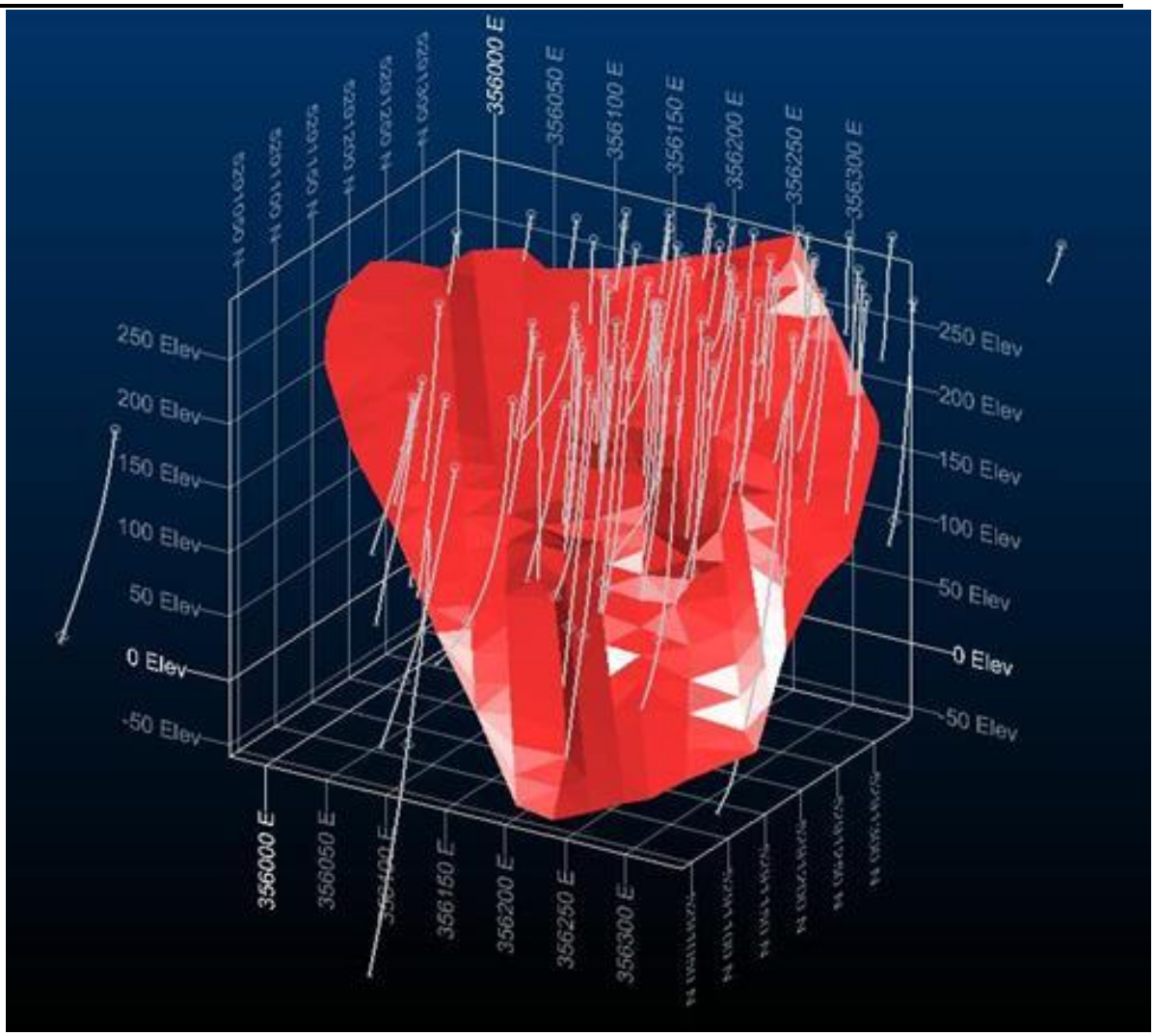

Figure 14.1 Isometric view of the Cape Ray 04 geological domain used

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A total of 66 holes have penetrated the mineral zone (see Table 14.1 and Figure 14.2).

Table 14.1 Summary of Diamond Drilling for the Cape Ray 04 Deposit

DDH	UTM East	UTM North	Elevation	Length	Grid East	Grid North	Azimuth	Dip
CR03-02	356375.2	5291164.4	328.7	228.7	6748.1	4905.8	322.4	-56
CR03-03	356252.3	5291290.4	319.5	103.7	6727.3	5080.6	322.4	-70
CR03-04	356331.2	5291168.4	329.4	225.6	6715.7	4935.7	324.4	-67
CR03-07	356305.2	5291008.4	334.6	396.6	6597.6	4824.6	322.4	-78
CR03-08	356276.2	5291312.4	318.9	60.1	6759.7	5083.5	322.4	-47
CR04-10	356319.2	5290924.4	338.1	285.1	6557.6	4749.5	322.4	-56
CR04-11	356223.3	5291039.4	326.8	154.0	6551.4	4899.1	322.4	-56
CR04-12	356233.3	5290902.4	332.5	277.7	6476.0	4784.3	322.4	-56
CR04-13	356216.3	5290890.4	332.3	86.9	6455.2	4785.2	322.4	-55
CR04-14	356213.3	5290890.4	332.3	272.3	6452.8	4787.0	322.4	-56
CR04-15	356200.3	5290924.4	330.2	232.0	6463.2	4821.9	322.4	-50
CR04-16	356238.2	5290850.4	333.1	273.5	6448.3	4740.1	322.4	-60
CR04-17	356322.3	5290737.4	331.1	528.0	6446.1	4599.3	322.4	-70
CR04-18	356306.3	5290796.4	334.5	393.9	6469.4	4655.8	320.4	-62
CR04-19	356259.3	5291152.4	317.4	105.2	6648.8	4966.9	322.4	-56
CR04-20	356068.3	5290721.4	337.1	307.0	6234.9	4741.2	322.4	-60
PB77-004	356220.1	5291284.7	318.6	122.0	6698.2	5095.7	322.4	-60
PB77-008	356232.6	5291293.3	319.5	90.9	6713.3	5094.8	322.4	-60
PB77-009	356196.7	5291265.8	319.2	98.8	6668.2	5094.9	322.4	-60
PB77-010	356248.6	5291250.0	324.1	137.2	6699.7	5050.8	322.4	-60
PB77-014	356224.2	5291231.8	324.4	182.9	6669.3	5051.2	322.4	-60
PB77-015	356165.2	5291255.2	317.4	115.9	6636.7	5105.6	322.4	-60
PB77-016	356125.8	5291255.0	314.6	152.4	6605.3	5129.5	322.4	-60
PB77-017	356092.6	5291247.1	313.1	111.6	6574.2	5143.4	322.4	-60
PB77-018	356281.7	5291210.3	328.0	201.2	6701.8	4999.1	322.4	-60
PB77-019	356200.9	5291214.1	325.0	178.4	6640.1	5051.3	322.4	-60
PB77-024	356346.0	5291326.1	325.0	174.7	6823.3	5051.8	322.4	-60
PB77-025	356070.7	5291181.6	318.6	164.6	6517.0	5104.8	322.4	-60
PB77-027	356246.5	5291156.5	330.2	244.5	6641.1	4977.8	322.4	-60
PB77-029	356293.0	5291195.4	328.7	235.7	6701.7	4980.4	322.4	-70
PB77-029A	356293.0	5291195.4	328.7	61.9	6701.7	4980.4	322.4	-60
PB77-035	356320.4	5291261.1	326.8	159.1	6763.5	5015.9	322.4	-65
PB77-038W	356272.8	5291120.2	331.4	63.7	6639.9	4933.0	322.4	-65
PB77-039	356229.1	5291084.5	333.2	299.4	6583.5	4931.4	322.4	-70
PB79-097	356450.1	5291387.2	319.2	91.5	6943.1	5036.9	322.4	-45
PB79-113	356343.4	5291023.5	334.8	418.3	6637.1	4813.4	322.4	-70

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DDH	UTM East	UTM North	Elevation	Length	Grid East	Grid North	Azimuth	Dip
PB79-141	356281.8	5290914.9	336.3	414.6	6522.1	4764.7	322.4	-70
PB86-174	356361.2	5291206.7	329.0	249.4	6762.7	4947.8	322.4	-70
PB86-178	356394.3	5291062.2	336.9	368.3	6701.0	4813.1	322.4	-55
PB87-194	356452.3	5291011.2	336.5	566.2	6715.9	4737.3	322.4	-70
PB87-198	356338.1	5291036.0	334.4	414.3	6640.4	4826.5	322.4	-70
PB87-201	356302.5	5291083.0	333.3	366.8	6640.9	4885.5	322.4	-70
PB87-203	356296.2	5291042.2	334.2	395.4	6611.0	4856.9	322.4	-70
PB87-205	356275.7	5291018.4	334.6	407.6	6580.2	4850.5	322.4	-70
PB87-207	356135.6	5291052.4	326.2	326.8	6489.8	4962.8	322.4	-70
PB87-210	356436.3	5291207.6	339.5	337.5	6822.9	4902.8	322.4	-70
PB88-242	356288.7	5291101.2	333.0	319.2	6640.9	4908.3	322.4	-70
PB88-243	356248.2	5291054.6	334.4	350.6	6580.4	4896.0	322.4	-70
PB88-244	356261.5	5291186.8	329.6	243.9	6671.5	4992.7	322.4	-70
PB88-245	356268.0	5291078.6	333.1	320.1	6610.8	4903.0	322.4	-70
PB88-246	356316.1	5291115.4	331.9	304.9	6671.3	4902.9	322.4	-70
PB88-247	356231.7	5291126.3	332.5	274.7	6611.0	4962.9	322.4	-70
PB88-248	356340.2	5291133.6	332.6	289.6	6701.5	4902.6	322.4	-70
PB88-249	356272.5	5291122.9	332.3	288.1	6641.4	4935.4	322.4	-70
PB88-250	356327.7	5291200.0	328.7	215.5	6732.1	4962.9	322.4	-65
PB88-251	356262.2	5291036.0	334.6	364.6	6580.2	4872.7	322.4	-70
PB88-252	356256.2	5290994.1	335.1	380.2	6550.0	4843.1	322.4	-70
PB88-253	356194.2	5291325.3	311.9	74.7	6702.5	5143.6	322.4	-55
PB88-254	356147.1	5291287.6	312.5	91.5	6642.1	5142.4	322.4	-61
PB88-255	356220.0	5291041.5	335.1	277.7	6550.1	4902.8	322.4	-70
PB88-257	356383.7	5290975.9	336.8	445.7	6640.0	4751.0	322.4	-70
PB88-260	356385.7	5291174.2	331.5	270.4	6762.4	4907.1	322.4	-70
PB88-290	356343.4	5291079.5	334.7	336.6	6671.1	4857.8	322.4	-70
PB88-299	356213.5	5291150.1	331.3	235.4	6611.1	4992.9	322.4	-70
PB88-302	356323.4	5291005.8	335.6	429.6	6610.4	4811.4	322.4	-70
PB88-303	356176.3	5291299.6	313.8	76.5	6672.6	5134.2	322.4	-55
PB88-306	356307.9	5290975.8	335.5	398.2	6579.8	4797.1	322.4	-70
PB88-311	356370.7	5291043.0	336.0	370.7	6670.6	4812.2	322.4	-70
PB88-316	356243.5	5291210.7	327.7	187.5	6671.8	5022.6	322.4	-60
PB88-317	356394.0	5291114.1	337.0	373.2	6732.4	4854.5	322.4	-70
PB88-323	356312.5	5291169.8	329.4	227.7	6701.6	4948.3	322.4	-60
PB88-326	356330.7	5290946.2	338.2	405.2	6579.9	4759.7	322.4	-60
PB89-330	356358.7	5291259.6	326.7	203.0	6793.0	4991.3	322.4	-65
PB89-333	356376.6	5291235.9	328.0	223.5	6792.7	4961.7	322.4	-68
PB89-335	356394.2	5291212.6	329.8	254.3	6792.5	4932.4	322.4	-70

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PB89-339	356297.2	5291290.3	325.9	148.5	6762.8	5053.1	322.4	-60
PB89-340	356166.4	5291212.3	323.5	154.6	6611.5	5070.9	322.4	-60
PB89-349	356314.7	5291217.1	328.0	190.5	6732.1	4984.4	322.4	-60
PB89-350	356296.2	5291241.8	327.7	168.6	6732.5	5015.2	322.4	-58
PB89-351	356326.2	5291252.3	327.3	168.9	6762.7	5005.3	322.4	-63
PB89-352	356320.5	5291310.1	325.5	128.0	6793.3	5054.7	322.4	-60
PB87-198	356338.1	5291036.0	334.4	414.3	6640.4	4826.5	322.4	-70
PB87-201	356302.5	5291083.0	333.3	366.8	6640.9	4885.5	322.4	-70
PB87-203	356296.2	5291042.2	334.2	395.4	6611.0	4856.9	322.4	-70

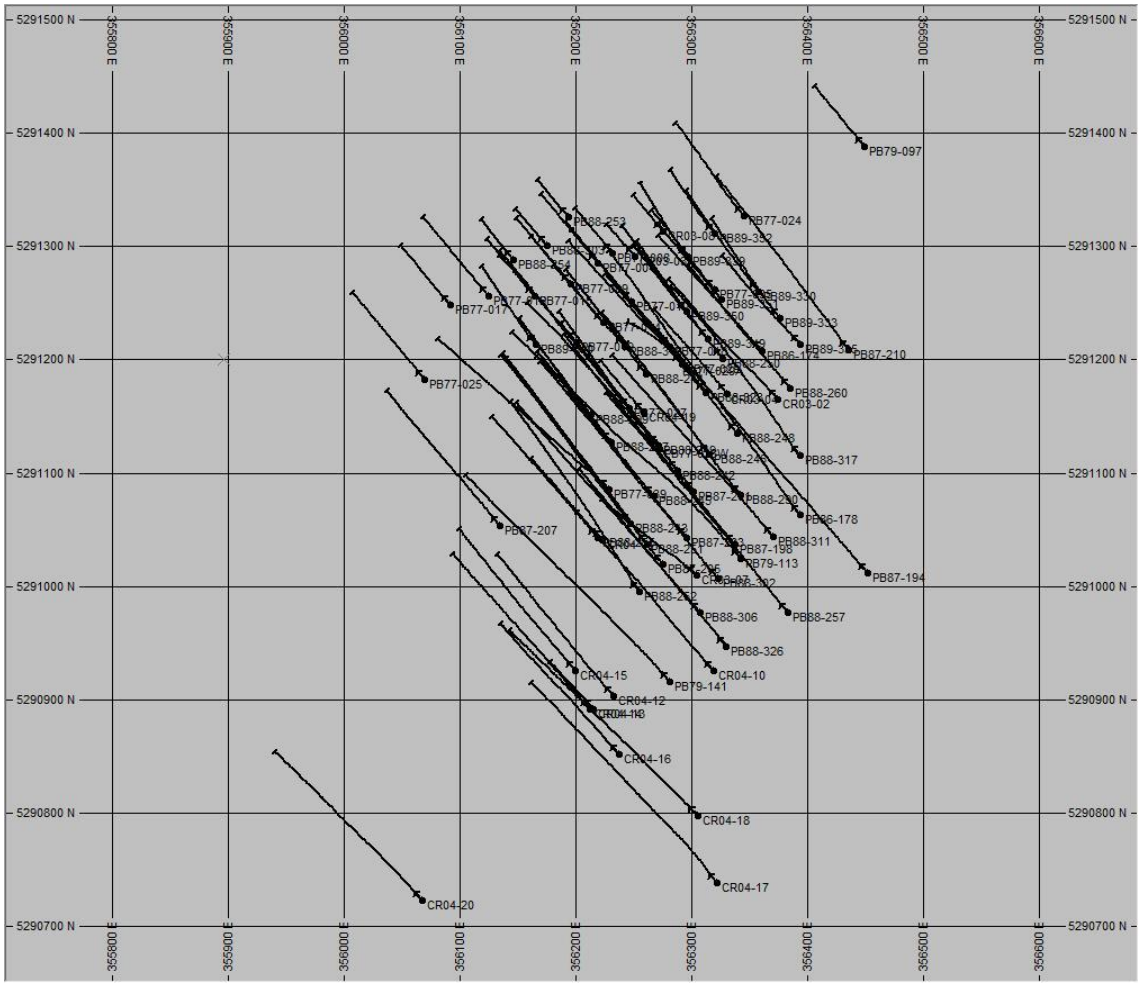

Figure 14.2 Plan view of Cape Ray 04 Deposit drilling.

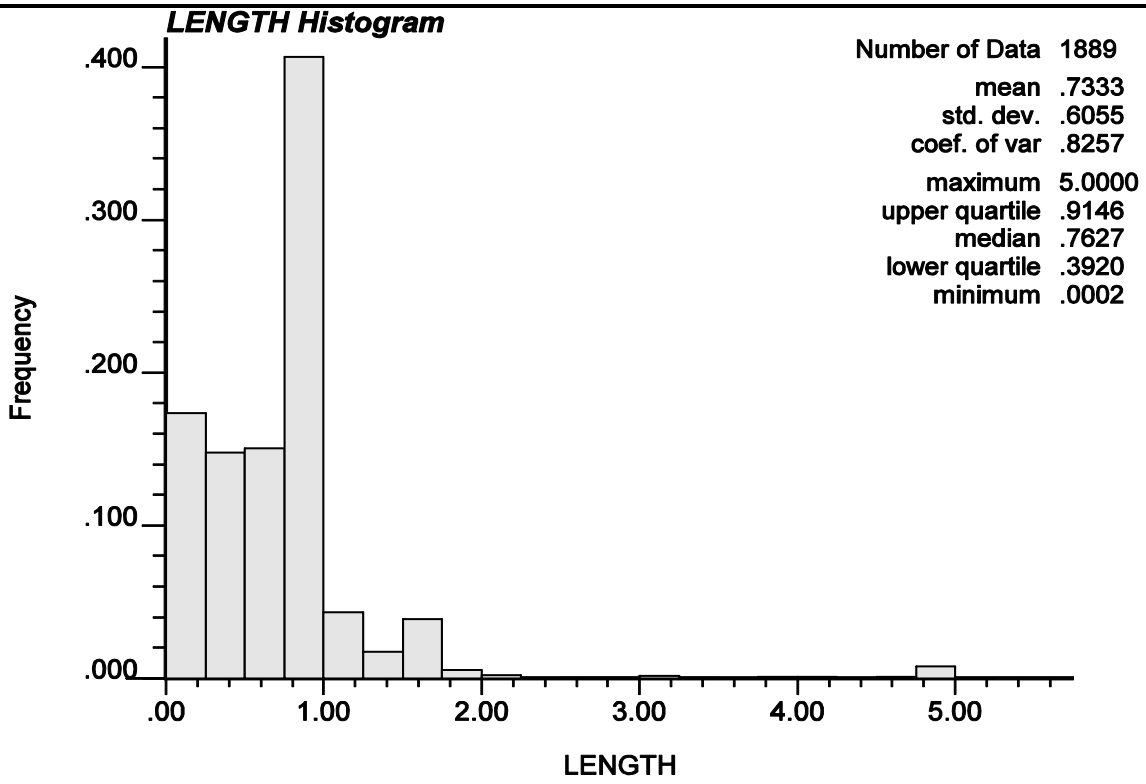

Figure 14.3 Histogram of Sample Lengths for the Cape Ray 04 Deposit

14.2.1.3 Drill Hole Database

For the Cape Ray 04 deposit the file used for resource estimation purposes, "cr_04_holes_lg.dm", contains information for 81 drill holes totalling 22,488 metres and with 5,544 samples collected.

14.2.1.4 Sample Selection

Working in cross section a set of mineral zone lines, or strings, was defined for the domain. These strings were drawn to enclose the Cape Ray 04 Au-Ag zone by snapping to the drill holes. The strings were then used to construct a mineral envelope wire frame for the domain. The envelope extends from 296 metres above mean sea level down to 59 metres below mean sea level. The borehole samples located within the mineral envelopes were captured using a custom script

14.2.1.5 Compositing

The captured samples have an average sample length of 0.73 metres (see Figure 14.3). It is expected that mining at the 04 deposit could be by underground mining methods. A block size that will allow a reasonable amount of selectivity using that mining method is 3 metres in all three dimensions. A 1 metre composite length represents a multiple of these blocks dimensions ensuring uniform support.

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The samples were composited to standard 1 m intervals using the CAE Studio 3 process COMPDH. The COMPDH process starts the composites at the beginning of the selected data interval and leaves any remainder at the end of the interval. This results in most holes having one sample with a length less than the established composite length. For grade estimation purposes, drill composites are treated like point data (i.e. their length is not used), thus the need to composite to a standard sample length to eliminate any sample bias. To avoid bias from a very short sample being treated the same as a standard sample any sample lengths that were less than 40% of the composite length were rejected.

14.2.1.6 Exploratory Data Analysis

A review of the drill hole database was done using histograms for all primary elements and X/Y scatter plots of element pairs (see Appendix A). Features watched for are outliers and irregularities in the element statistics. Univariate summary statistics for all elements are presented in Table 14.2 and a correlation matrix is presented in Table 14.3. For the latter a correlation coefficient of 0.7 or higher indicates a good linear relationship between the bivariate components. A positive coefficient indicates that with increasing concentration of one element there is a sympathetic increase in the other. A negative coefficient indicates that as one element increases the other decreases.

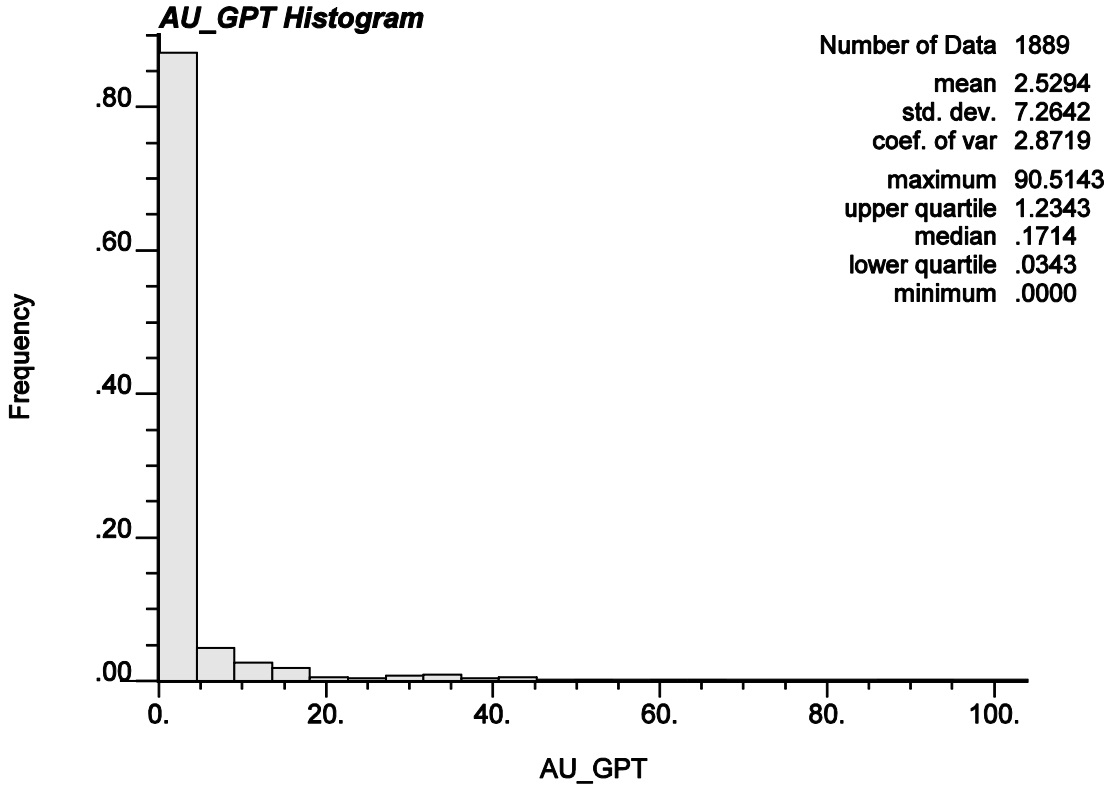

Figure 14.4 Histogram of Au for the Cape Ray 04 Deposit

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Table 14.2 Summary Univariate Statistics for the Cape Ray 04 Deposit

FIELD	NSAMPLES	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDDEV	SKEWNESS	KURTOSIS
AU_GPT	1890	0	90.51	2.55	53.54	7.32	5.25	36.57
AG_GPT	1890	0	274.29	8.42	455.17	21.33	4.96	31.86
CU%	1890	0	2.86	0.06	0.05	0.22	7.90	79.68
PB%	1890	0	12.50	0.30	0.79	0.89	7.42	76.69
ZN%	1890	0	11.10	0.26	0.79	0.89	7.78	75.12

The scatter plots for the element pairs do not indicate any serious issues. All show a cloud that have a crude linear relationship and this is confirmed by the correlation coefficients that are typically between 0.6 and 0.78 with the best being Pb and Zn with a correlation coefficient of 0.87.

Au, the primary element of interest, does have a few spurious values with a maximum value of 90.5 g/t. The histogram for Au (Figure 14.4) is a positively skewed distribution, typical of a lode gold deposit.

Table 14.3 Correlation Matrix for the Cape Ray 04 Deposit

	AU_GPT	AG_GPT	CU%	PB%	ZN%
AU_GPT	1				
AG_GPT	0.6848	1			
CU%	0.6078	0.6255	1		
PB%	0.7231	0.6598	0.7618	1	
ZN%	0.6554	0.6440	0.7557	0.8727	1

While there are several outliers for all elements this in itself is not unusual for this style of mineralisation. As a result the Exploratory Data Analysis found no issues with the drill hole database that would invalidate their use for resource estimation purposes.

14.2.1.7 Grade Variography

Prior to doing grade variography a custom script was used to prepare a variogram map for Au (see Figure 14.5) in order to check for a rotation in the primary direction of anisotropy. Directional lognormal variograms were calculated in 15° increments in the unfolded plane of the mineral zone. lognormal variograms provide a clearer picture of the spatial continuity as it reduces the influence of very large values on the calculation of the moment of inertia, common with highly positively skewed distributions such as with lode gold deposits (Isaaks and Srivastava, 1989).

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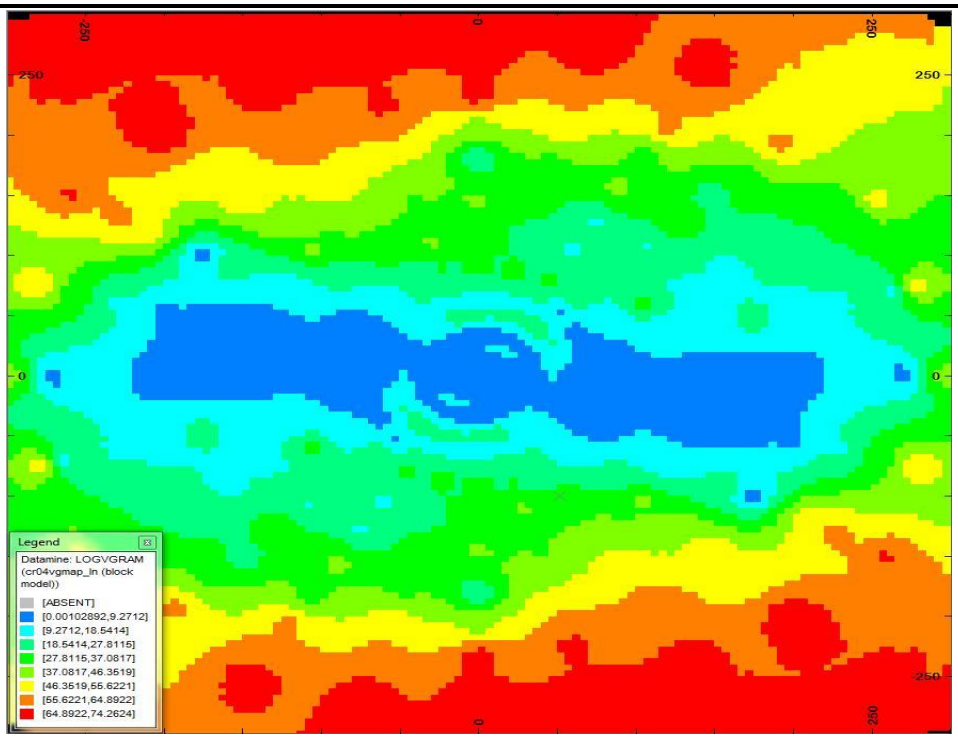

Figure 14.5 Variogram map for the main Cape Ray 04 Au-Ag Deposit

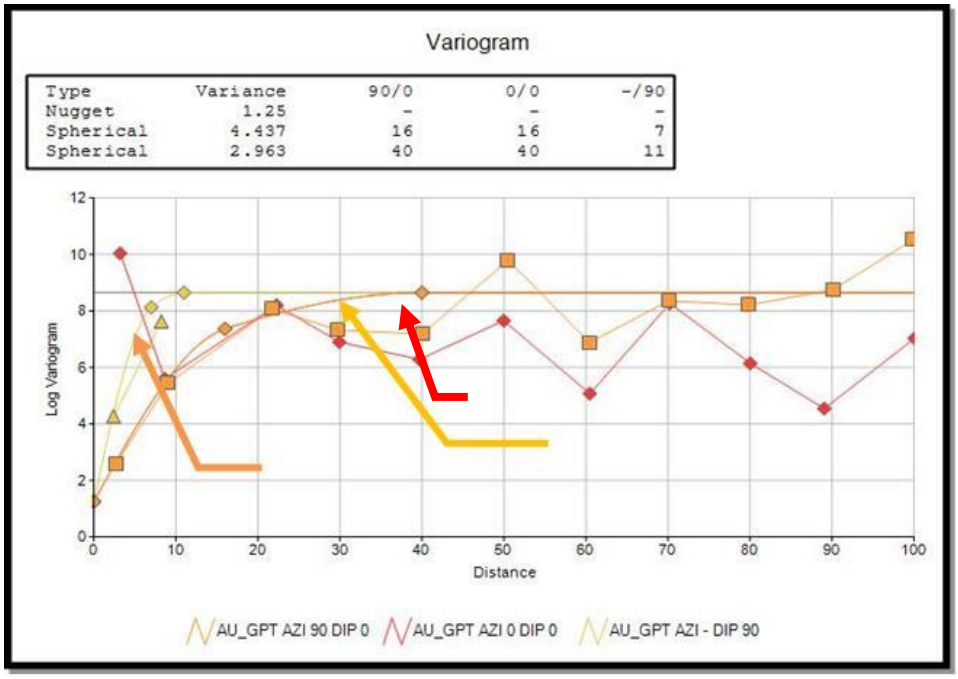

Figure 14.6 Experimental Variograms and fitted models for Au – Cape Ray 04 Deposit

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The variogram map does not show a rotation (plunge) around the UCSA (across dip) axis. It also shows that the along strike direction (UCSC – Y axis) is very poorly defined indicating the drill hole spacing in this direction is too far apart to properly define the variogram in this direction. The sill is reached at a range of approximately 50 metres. Due to the nature of the drilling (primarily on 25 metre centres) the along strike direction (the Y axis of the map) is poorly defined

The experimental grade variograms (see Figure 14.6) were calculated for the unfolded composited data sets using a custom script.

Typically, with inclined drilling, the across dip and down dip directions of the variogram are usually well defined due to the abundant sampling of the distance spectrum. This is indeed the case for the down dip direction. The along strike variogram is poorly defined due to the poor sampling resulting from the use of 25 metre sectional drilling.

In summary, there is a reasonable degree of confidence in the curves used to model the down dip and across strike directions. While not well defined there is just enough sampling in the along strike direction that it is reasonable to assume that the same ranges in that direction as those used in the down dip direction.

The ranges derived from the log variogram models are shown in Table 14.4.

Table 14.4 Log Variogram Model Parameters for the Cape Ray 04 Deposit

Variogram Models – Cape Ray 04	AU_GPT
Nugget	0.87
1st spherical structure range A	2
1st spherical structure range B	16
1st spherical structure range C	16
1st spherical structure sill	5.71
2nd spherical structure range A	5
2nd spherical structure range B	40
2nd spherical structure range C	40
2nd spherical structure sill	2.07
Total sill	**8.65**

14.2.1.8 Block Size Determination

For Cape Ray 04 deposit, due to the geometry and based on recommendations from mine engineering affiliated with this project, a block size suitable for evaluating underground mining methods as well as open pit mining was selected.

Another factor that needs to be considered is the degree of sampling detail. If samples are large and/or spaced far apart a small block size would be inappropriate. A block size of 3 metres by 3 metres by 3 metres was chosen as this meets the above criteria.

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A custom script was used to create the empty prototype model and then fill it with blocks using the mineral envelope for the domain wire frame. The resultant empty model was then regularized creating FILLVOL and VOIDVOL fields containing the volume for each block inside or outside the mineral domain wire frame.

14.2.1.9 Nearest Neighbour Block Model

A Nearest Neighbour (NN) estimated model (nn_cr04_mod.dm) was created for the mineral domain in order to determine the de-clustered mean for the data. This mean can then be used to validate the kriged global estimate as all methods of estimation should produce essentially the same global mean, if done correctly. The de-clustered mean is also used in assessing smoothing and, if necessary, calculating a variance correction of the kriged model.

Summary statistics comparing the nearest neighbour model to the sample file are presented in Table 14.5. Note how the mean values for the NN and IK models for most of the elements estimated are close to or are within 5% of one another, confirming that the mean has been reproduced. The two elements where the means are more than 5% apart are Cu and Zn and this is due to a scale issue; the absolute difference is small but when viewed as a ratio results in values outside our usually acceptable threshold.

Table 14.5 Summary Statistics for Samples and Models – Cape Ray 04 Deposit

FILE	FIELD	NRECORDS	NSAMPLES	MINIMUM	MAXIMUM	MEAN	% DIFF	VARIANCE	SKEWNESS	WGTFIELD
cr04_data1u	AU_GPT	1389	1389	0	53.03	1.69		20.448	4.95	LENGTH
nn_cr04_mod	AU_GPT	93966	93966	0	53.03	1.51		12.329	5.207	FILLVOL
ik_cr04_mod	AU_GPT	93966	93966	0	22.64	1.64	8.66	3.232	2.434	FILLVOL
cr04_data1u	AG_GPT	1389	1389	0	228.04	6.26		244.068	5.859	LENGTH
nn_cr04_mod	AG_GPT	93966	93966	0	228.04	5.48		200.256	7.563	FILLVOL
ik_cr04_mod	AG_GPT	93966	93966	0.09	81	5.7	4.12	34.683	2.379	FILLVOL
cr04_data1u	CU%	1389	1389	0	1.63	0.04		0.015	6.63	LENGTH
nn_cr04_mod	CU%	93966	93966	0	1.63	0.03		0.009	6.738	FILLVOL
ik_cr04_mod	CU%	93966	93966	0	0.55	0.04	7.07	0.002	2.456	FILLVOL
cr04_data1u	PB%	1389	1389	0	5.64	0.2		0.249	5.158	LENGTH
nn_cr04_mod	PB%	93966	93966	0	5.64	0.17		0.131	5.087	FILLVOL
ik_cr04_mod	PB%	93966	93966	0	1.76	0.18	5.4	0.028	1.726	FILLVOL
cr04_data1u	ZN%	1389	1389	0	5.19	0.17		0.209	5.695	LENGTH
nn_cr04_mod	ZN%	93966	93966	0	5.19	0.16		0.156	6.165	FILLVOL
ik_cr04_mod	ZN%	93966	93966	0	1.7	0.15	-5.42	0.021	1.98	FILLVOL

A visual inspection on a section-by-section and plan-by-plan basis comparing the input sample file with the resultant nearest neighbour file showed good correlation with the drill holes and proper spreading of the grade.

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14.2.1.10 Indicator Kriging Block Model

As noted previously, the grade distributions are highly skewed for the Cape Ray 04 deposit. These distributions often result in spatial estimation bias where a few outliers representing the high-grade tail of the distribution are difficult to characterize resulting in a significant impact on both the mean and variance for the grade population even though they only represent a small fraction of it. Since traditional linear estimation methods, including Ordinary Kriging, often misrepresent these outliers resulting in a smoothing of the overall variability and therefore in over-estimation, Indicator Kriging was used to model the Cape Ray 04 deposit. An Indicator approach has proven successful in addressing the problem of highly skewed populations

As discussed in 14.1.5, the indicator approach involves coding data into binary groupings using a series of grade cut-offs. The metal grade can be estimated by multiplying the mean of the binary grade cut-off groupings, by its probability, which is determined by Indicator Kriging, before summing them. Using this indicator approach, the metal estimated was interpolated for all of the 3 m x 3 m x 3 m cells contained within the block model for the Cape Ray 04 mineral domain.

After the parameter files needed for Indicator Kriging were constructed, a custom script was used carry out the Multiple Indicator Kriging process. Each cell in the block model was discretised using a matrix of 3 x 3 x 3 points in the ABC (unfolded) coordinate system. The Kriging functions were interpolated at each discretisation point using the same search volume as the nearest neighbour interpolation, based on the grade variogram results. In case of local low sample density a nested search was implemented.

14.2.1.10.1 Definition of Indicator Functions

The grade data for Cape Ray 04 are all highly skewed. To deal with such highly skewed frequency distributions of grade an indicator approach has been applied for grade estimation. The 1 m composites were coded according to the cut-offs determined from the de-clusterised cumulative statistics for all 5 grade fields using a custom script. The indicators used are listed inTable 14.6.

Initially each cut-off corresponds to the grade of the upper limit of each portion of the sample data representing approximately 9.1% of the total metal. As the low grade portions of the cumulative distribution functions are poorly sampled using this approach additional indicators were added to ensure proper sampling of the full distribution.

14.2.1.10.2 Indicator Variography

The indicator variograms for the Cape Ray 04 mineral domain were modelled using CAE Studio 3 variogram modeling tools.

One advantage of Indicator variograms is that they are often less erratic and easier to model than the grade variograms. This is because they do not suffer from having large differences between values.

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Table 14.6 Cape Ray 04 Deposit - Indicator Cut-offs Used

Indicator	AU_GPT	AG_GPT	CU%	PB%	ZN%
1	0.034	0.200	0.006	0.011	0.012
2	0.171	3.428	0.012	0.047	0.034
3	0.307	3.429	0.017	0.140	0.106
4	1.371	7.610	0.028	0.249	0.218
5	2.000	12.996	0.060	0.372	0.299
6	2.983	22.824	0.095	0.544	0.439
7	4.927	28.929	0.151	0.750	0.621
8	5.829	36.167	0.258	0.933	0.908
9	8.898	49.029	0.321	1.066	1.000
10	12.847	77.175	0.385	1.410	1.362
11	15.227	109.989	0.513	1.764	1.731
12	21.073		0.769	2.549	3.049
13	29.725				

14.2.1.10.3 Indicator Kriging

Custom scripts were used to create the parameter files needed for Indicator Kriging and to carry out the Indicator Kriging process. Each cell in the block model was discretised using a 3 x 3 x 3 point matrix. The indicator functions were interpolated at each discretisation point using the same search volume as the nearest neighbour interpolation, based on the grade variogram results. In case of local low sample density a nested search was implemented.

For the model 24.0% of the blocks were estimated in the first search, 33.3% in the second and 42.7% in the third.

14.2.1.10.4 Investigation of Kriging Results

In theory estimated indicator functions should range between 0 and 1, but as mentioned in Section 14.1.6, Ordinary Kriging can create negative weights that may cause the interpolated values to be outside of this range. The Cape Ray 04 deposit had no significant overlap issues caused by negative weights; all of the negative indicator values were set to 0.00 and all values over 1 were set to 1 using a custom script during post processing.

Results were also checked to ensure that indicator values decrease as the cut-off increases. Although order relations issues (probability increases as the grade cut-off increases) can be expected when different variograms are used for the indicator functions, they should not be more than approximately 1% of the estimates. In the case of Cape Ray 04 data there are no significant cumulative proportions issues.

14.2.1.10.5 Volume Variance Correction

Regression methods such as Kriging may result in an over-smoothing or under-smoothing of the grade variability producing a block grade distribution with a variance that is lower or higher than

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expected. For Cape Ray 04 the smoothing ratio for gold (3.22) and for silver (5.7) shows that there is some smoothing and as a result a correction was required.

The variance of the model for both elements has been increased and therefore more in line with the expected variance with respect to block size. As should be expected, the mean remains essentially unchanged by the transformation.

As they are relatively minor elements with very low average (mean) values it was determined that no corrections would be made for Cu, Pb, or Zn.

14.2.1.11 Model Verification

Validation procedures were carried out on the estimated block models including visually checking the sample file against estimated blocks. The sample grades were found to reasonably match the estimated block grades in the model.

A global statistical comparison of the global means of all estimations method was done. The difference between all the global means was found in most cases not to exceed approximately 5%, to be expected if the process was done correctly (see Table 14.5). Where differences do exceed this threshold it is easily explainable as a scale issue.

Other statistical checks that were done include the use of swath plots (see Appendix C). Swath plots compare the moving average of the mean for both the NN and IK models and the sample file using panels, or “swaths”, through the mineral envelope. As this is best done when the data is within a rectilinear volume the unfolded coordinates were used to define the swaths. The result is a curve for each data set. The curves for the models should inter-weave with the sample curve and the two model curves should be sympathetic with one another with no major deviations from one another. No issues were noted.

14.3 Cape Ray 41 Deposit

14.3.1 Resource Estimation Methodology

14.3.1.1 Data Validation

Core-drilling data were supplied as Microsoft Excel tables that included collar information, assays, lithology information, down hole survey information and specific gravity data. Sibley Basin Group compiled the data into a Microsoft Access database and completed data validation. Once validated this information was imported into CAE Studio 3 as four tables: a collar file, an assay file, a lithology file, and a survey file. As the data supplied were in UTM coordinates a transformation was completed to make a copy of the drill hole file in local grid coordinates based on the collar location of hole PB80-156. The collar for this hole has a World Coordinate System (WCS) location of 356005.19E, 5290825.26N with a Local Coordinate System (LCS) location of 334.61W, 136.37S. Using the CAE Studio 3 HOLES3D routine a desurveyed drill hole file, cr_41_holes_lg.dm, was created.

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In fall of 2015 the drill hole file was modified by incorporating surface channel samples that had been collected during the summer exploration program (see Section 9 for details). All 13 channels, comprised of a total of 88 samples, were treated as horizontal drill holes. This drill hole file incorporating the merged channel samples was last updated on October 01, 2015.

The Studio 3 desurveying routine, HOLES3D, does a rigorous set of validation checks including checking for duplicate borehole numbers, missing survey data and overlapping sample intervals. If present, it generates a summary report with a list of all errors encountered. This file was checked to determine if any errors occurred and where appropriate corrections were made. Once it had been confirmed that no errors remained the drill hole file was then used for subsequent steps.

There are no density or specific gravity values in the data set provided but a previous resource estimate for the project area (Ténière & Hilchey, 2012) had specific gravity (SG) values determined on 44 samples. From these an average value was calculated from 6 samples that fell below the 25th percentile and used as a global value. This same value of 2.84 g/cm3 has been used for this project.

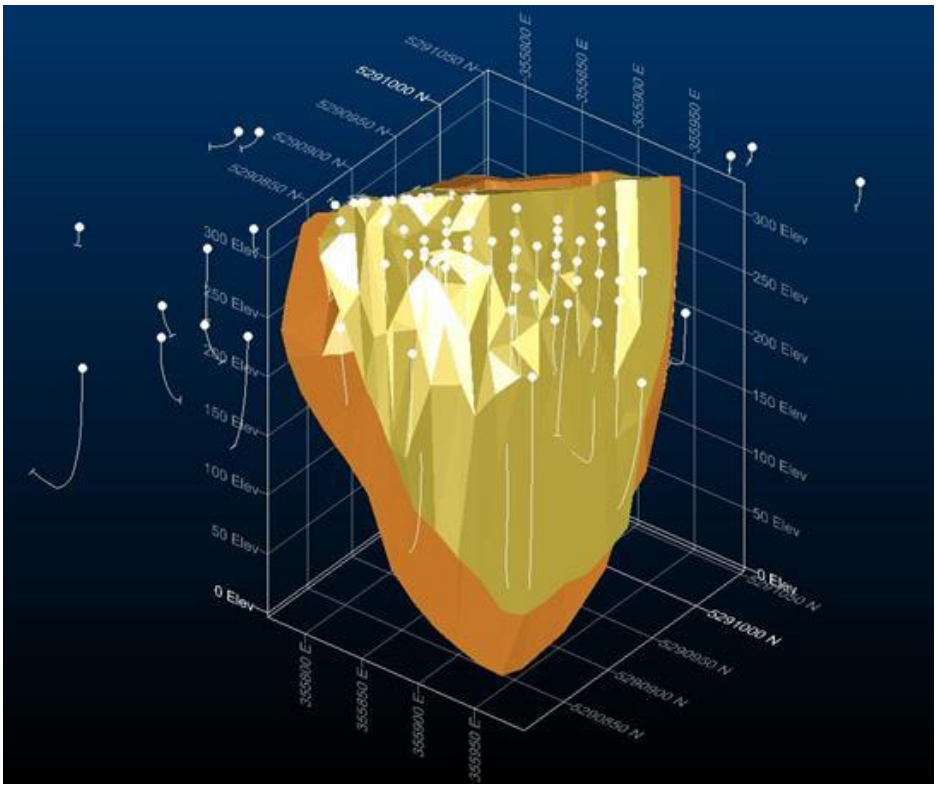

Figure 14.7 Isometric view of the Cape Ray 41 geological domains used

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14.3.1.2 Geological Domains

Experienced geologists had coded each rock unit based on core logging description. All of the holes are inclined and most intersected at least some portion of the deformation zone that hosts the gold mineralisation. Construction of the resource block model was controlled by building two wire frames, each representing a different mineralisation domain and these were then used to isolate related samples. No cut-off was used to limit the extent of this mineral envelope. The envelopes for the two mineral domains (see Figure 14.7) extend from an elevation of about 316 metres above sea level down to a maximum depth of approximately 4 metres below sea level. These are not geological envelopes as the deformation zone hosting the mineralisation cuts stratigraphy.

A total of 59 holes and 13 surface channels have tested the mineral zones (see Table 14.7 and Figure 14.8).

Table 14.7 Summary of Diamond Drilling for the Cape Ray 41 Deposit

DDH	UTM East	UTM North	Elevation	Length	Grid East	Grid North	Azimuth	Dip
CR03-06	355996.3	5290851.4	333.5	181.1	6256.9	4888.2	322.35	-69
PB78-040	355537.3	5290689.3	337.8	152.4	5794.1	5038.9	322.35	-45
PB78-041	355884.0	5290936.8	320.7	228.7	6219.8	5024.3	322.35	-45
PB78-042	355676.3	5290910.7	311.6	91.5	6039.2	5130.0	322.35	-45
PB78-043	355664.7	5290902.4	311.6	106.7	6024.9	5130.4	322.35	-45
PB78-044	355959.3	5291107.7	311.9	91.5	6383.6	5114.0	322.35	-45
PB78-045	355952.2	5291092.4	309.8	88.4	6368.6	5106.1	322.35	-45
PB78-046	355917.8	5290940.3	323.5	76.2	6248.8	5006.4	322.35	-45
PB78-047	355872.1	5290900.1	323.2	78.7	6188.0	5002.4	322.35	-45
PB78-048	355845.6	5290883.2	323.8	100.6	6156.7	5005.0	322.35	-45
PB78-049	355934.5	5290968.0	322.9	94.5	6278.9	5018.3	322.35	-45
PB78-053	355824.5	5290863.2	324.7	97.6	6127.8	5002.1	322.35	-45
PB78-054	355801.7	5290842.1	326.2	103.7	6096.9	4999.2	322.35	-45
PB78-055	355890.4	5290835.2	332.9	134.1	6163.0	4939.7	322.35	-60
PB78-056	355840.3	5290792.9	332.3	140.2	6097.6	4936.7	322.35	-60
PB78-059	355937.3	5290864.4	332.0	184.5	6218.0	4934.4	322.35	-60
PB78-060	356059.3	5291102.2	327.4	122.0	6459.6	5048.8	322.35	-45
PB78-061	355761.2	5290796.0	327.4	94.5	6036.7	4987.3	322.35	-45
PB78-067	355986.3	5290899.9	328.7	186.0	6278.5	4932.7	322.35	-60
PB78-068	355973.4	5290817.0	336.9	243.9	6217.8	4874.8	322.35	-70
PB78-069	356069.9	5290890.4	335.7	253.0	6339.1	4874.3	322.35	-60
PB78-070	356029.8	5290841.5	337.2	250.0	6277.5	4859.9	322.35	-60
PB78-071	355905.9	5290803.6	337.2	195.1	6156.1	4905.3	322.35	-60
PB79-086	355668.3	5290717.3	332.3	94.5	5915.1	4981.3	322.35	-45
PB79-087	355540.7	5290584.2	339.0	94.5	5732.8	4953.5	322.35	-45

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DDH	UTM East	UTM North	Elevation	Length	Grid East	Grid North	Azimuth	Dip
PB79-112	356086.3	5290765.7	341.2	350.6	6276.1	4765.4	322.35	-70
PB79-115	355956.2	5290726.3	342.1	333.8	6149.0	4813.3	322.35	-70
PB79-132	355598.6	5290410.7	353.6	280.4	5673.2	4780.6	322.35	-70
PB79-138	355793.3	5290560.9	350.3	384.1	5919.0	4781.3	322.35	-70
PB80-149	355895.2	5290867.2	329.9	143.3	6186.3	4962.1	322.35	-60
PB80-150	355906.2	5290906.1	325.9	132.0	6218.8	4986.3	322.35	-60
PB80-151	355947.3	5290900.7	327.7	139.9	6248.0	4957.1	322.35	-60
PB80-152	355960.6	5290935.0	325.3	137.2	6279.5	4976.1	322.35	-60
PB80-153	355863.5	5290813.9	331.4	134.1	6128.7	4939.2	322.35	-60
PB80-154	355956.3	5290843.7	334.5	179.9	6220.6	4906.4	322.35	-70
PB80-155	355956.3	5290843.7	334.5	202.4	6220.6	4906.4	322.35	-65
PB80-156	356005.2	5290825.3	337.8	216.5	6248.1	4862.0	322.35	-60
PB80-157	356011.0	5290916.7	331.4	185.4	6308.3	4931.0	322.35	-60
PB80-158	355923.9	5290831.8	335.1	179.6	6187.6	4916.6	322.35	-70
PB80-159	355935.3	5290918.1	326.2	129.3	6249.1	4978.2	322.35	-60
PB80-160	355948.3	5290926.8	325.6	118.9	6264.7	4977.2	322.35	-58
PB80-161	355930.2	5290900.9	327.7	118.9	6234.6	4967.6	322.35	-58
PB80-162	355973.0	5290944.8	324.4	118.9	6295.2	4976.4	322.35	-60
PB80-163	355902.8	5290885.1	328.7	114.0	6203.2	4971.7	322.35	-60
PB80-164	355927.2	5290928.1	325.0	100.6	6248.8	4991.0	322.35	-60
PB80-165	355957.0	5290888.0	329.0	137.2	6248.1	4941.1	322.35	-65
PB80-165A	355957.0	5290888.0	329.0	61.0	6248.1	4941.1	322.35	-60
PB80-166	355891.3	5290847.2	332.0	128.0	6171.1	4948.7	322.35	-60
PB80-167	355867.1	5290835.7	329.6	152.4	6144.9	4954.3	322.35	-60
PB80-168	355923.1	5290886.7	329.0	161.6	6220.3	4960.7	322.35	-60
PB80-169	356102.9	5290798.6	340.2	341.5	6309.3	4781.4	322.35	-70
PB86-171	355754.7	5290751.8	332.0	114.6	6004.6	4956.2	322.35	-60
PB86-172	355650.7	5290635.8	339.0	152.4	5851.5	4927.4	322.35	-45
PB86-175	355897.0	5290720.0	342.4	243.9	6098.2	4844.3	322.35	-55
PB86-180	356013.3	5290663.3	343.6	413.7	6155.9	4728.6	322.35	-70
PB86-181	356056.0	5290704.9	342.4	424.4	6215.1	4735.5	322.35	-70
PB87-185	355980.4	5290882.2	329.7	213.4	6263.0	4922.2	322.35	-65
PB87-186	355963.0	5290904.3	326.7	169.8	6262.7	4950.4	322.35	-50
PB87-187	355971.1	5290845.2	333.5	227.1	6233.2	4898.5	322.35	-45
PB87-188	355925.6	5290852.0	332.1	218.6	6201.2	4931.6	322.35	-65
PB87-189	355907.8	5290849.4	332.1	191.2	6185.5	4940.4	322.35	-65
PB87-190	355903.1	5290831.7	335.7	194.5	6171.0	4929.3	322.35	-60
PB87-192	355880.4	5290836.4	332.1	218.3	6155.9	4946.8	322.35	-50
PB87-193	355818.9	5290666.5	340.6	274.4	6003.6	4849.4	322.35	-47
PB87-195	355779.5	5290668.8	338.9	217.7	5973.7	4875.2	322.35	-45

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PB87-196	355945.7	5290951.9	323.8	157.6	6277.9	4998.6	322.35	-60
PB87-197	356004.9	5290899.0	331.1	297.9	6292.7	4920.6	322.35	-70
PB87-199	356023.0	5290875.4	334.3	355.8	6292.7	4890.9	322.35	-70
PB87-200	355806.3	5290633.6	342.5	317.1	5973.6	4831.0	322.35	-50
PB87-202	355853.5	5290671.3	343.0	327.7	6034.0	4832.2	322.35	-60
PB87-204	356039.3	5290755.8	341.5	468.6	6232.9	4786.1	322.35	-70
PB88-320	355522.3	5290505.6	341.6	85.4	5670.3	4902.3	322.35	-50
PB89-328	355864.0	5290857.8	328.3	107.0	6155.9	4973.7	322.35	-45
PB89-329	355888.5	5290875.4	329.1	104.3	6186.0	4972.7	322.35	-45
PB89-331	355841.3	5290863.5	324.1	91.5	6141.3	4992.0	322.35	-45
PB89-344	355873.7	5290845.4	329.5	123.5	6156.0	4958.0	322.35	-47
PB89-345	355889.6	5290842.0	332.5	135.7	6166.5	4945.7	322.35	-49
PB89-346	355879.9	5290862.3	329.3	107.6	6171.2	4967.6	322.35	-49
PB89-347	355869.0	5290876.6	326.8	99.1	6171.3	4985.6	322.35	-49
PB89-348	355858.1	5290891.0	323.6	80.8	6171.4	5003.6	322.35	-49
PB96-360	355623.5	5290528.3	346.0	198.2	5764.4	4858.7	322.35	-45
41A	355852.924	5290927.2	322	8.4			320	0
41B	355849.453	5290924.2	321	5.8			323	0
41D	355840.06	5290920	322	6			323	0
41E	355829.151	5290908	322.7	1.6			300	0
41F	355825.555	5290904	322.7	12.6			309	0
41G	355818.975	5290897.8	322	3.4			309	0
41H	355816.806	5290895.2	322	6.2			309	0
41I	355811.281	5290888.6	322.5	4			309	0
41J	355808.131	5290885.6	322.7	3			309	0
41L	355799.283	5290874.9	323.5	7			306	0
41M	355797.395	5290872.6	324	7.9			314	0
41N	355793.379	5290868.3	324	7.7			314	0
41O	355784.811	5290858.2	324	7.7			298	0

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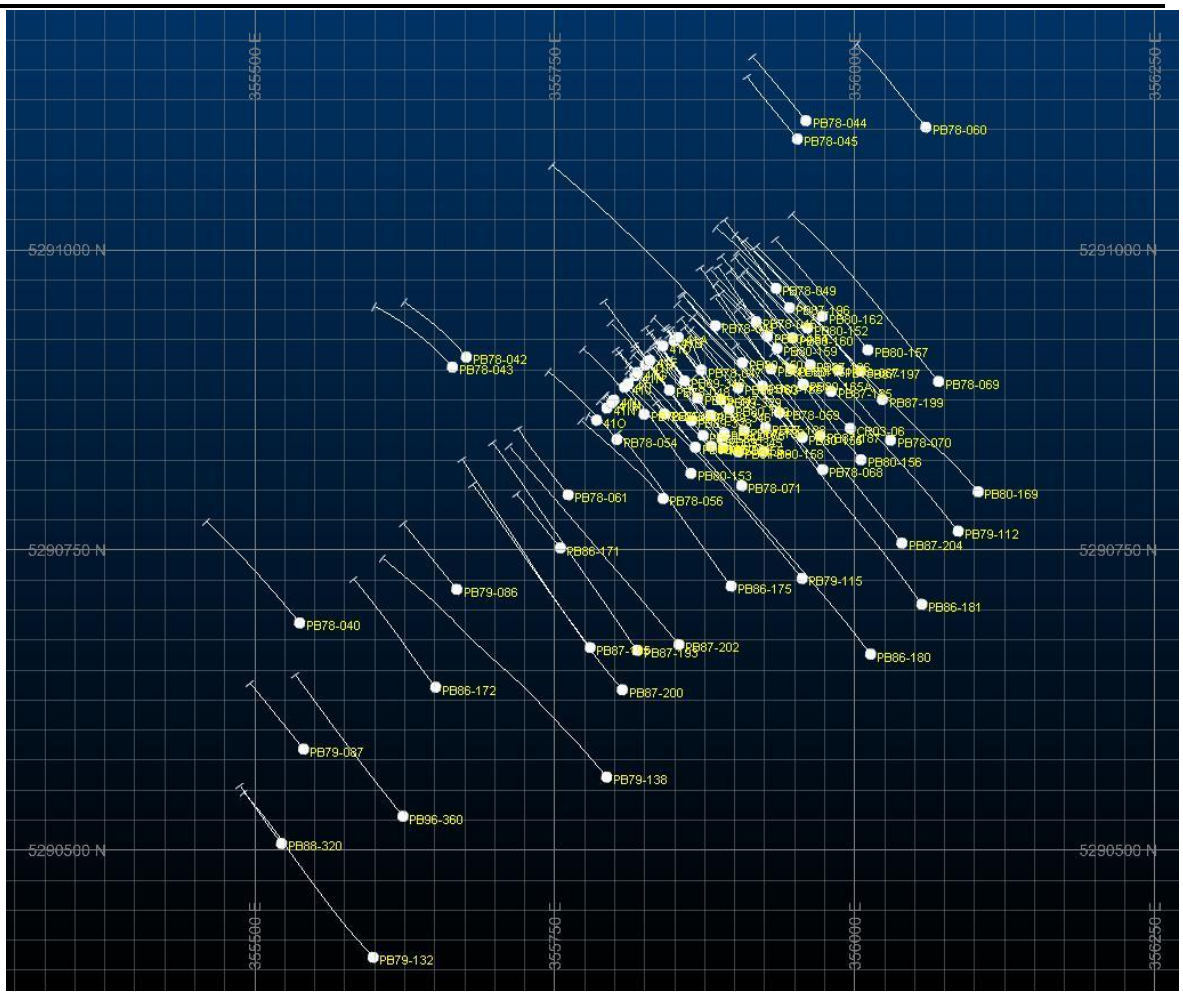

Figure 14.8 Plan view of Cape Ray 41 Deposit drilling

14.3.1.3 Drill Hole Database

For the Cape Ray 41 deposit the file used for resource estimation purposes, "cr_41_all_lg.dm", contains information for 81 drill holes totalling 14,497 metres and with 6,820 samples collected along with 13 channels totalling 81.3 metres and with 88 samples. The file was last updated October 1, 2015.

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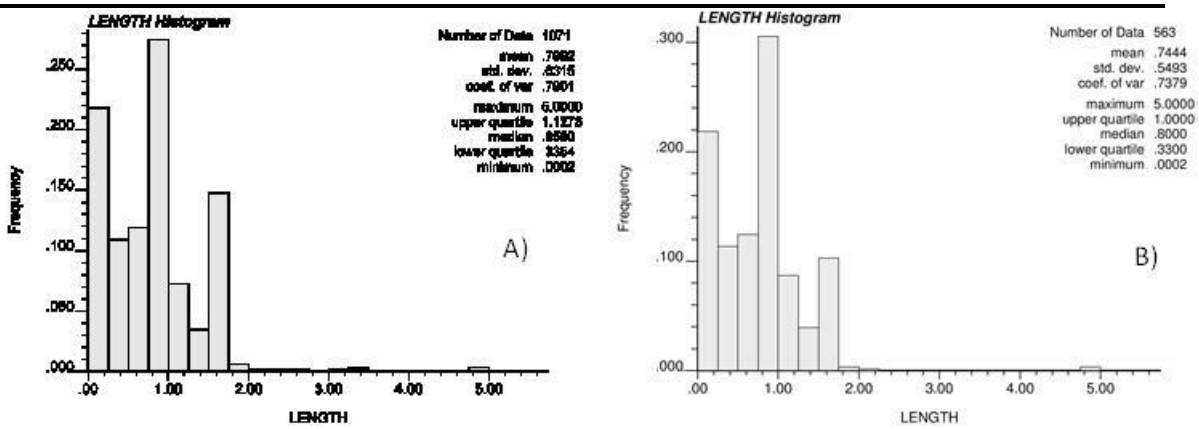

Figure 14.9 Histograms of sample length for the Cape Ray 41 Deposit. A) Domain 1 and B) Domain 2

14.3.1.4 Sample Selection

Working in cross section a set of mineral zone, lines, or strings were defined for the two domains. These strings were drawn to enclose the Cape Ray 41 Au-Ag zone by snapping to the drill holes. The strings were then used to construct a mineral envelope wire frame for the domain. The envelope extends from 316 metres above mean sea level down to about 4 metres below mean sea level. The borehole samples located within the mineral envelopes were captured using a custom script.

14.3.1.5 Compositing

For Domain 1 the captured samples have an average sample length of 0.86 metres and for Domain 2, 0.74 metres (see Figure 14.19). It is expected that mining at the 41 deposit could be by underground mining methods. A block size that will allow a reasonable amount of selectivity using that mining method is 3 metres in all three dimensions. A 1 metre composite length represents a multiple of these blocks dimensions ensuring uniform support.

The samples for both domains were composited to standard 1 m intervals using the CAE Studio 3 process COMPDH. The COMPDH process starts the composites at the beginning of the selected data interval and leaves any remainder at the end of the interval. This results in most holes having one sample with a length less than the established composite length. For grade estimation purposes, drill composites are treated like point data (i.e. their length is not used), thus the need to composite to a standard sample length to eliminate any sample bias. To avoid bias from a very short sample being treated the same as a standard sample any sample lengths that were less than 40% of the composite length were rejected.

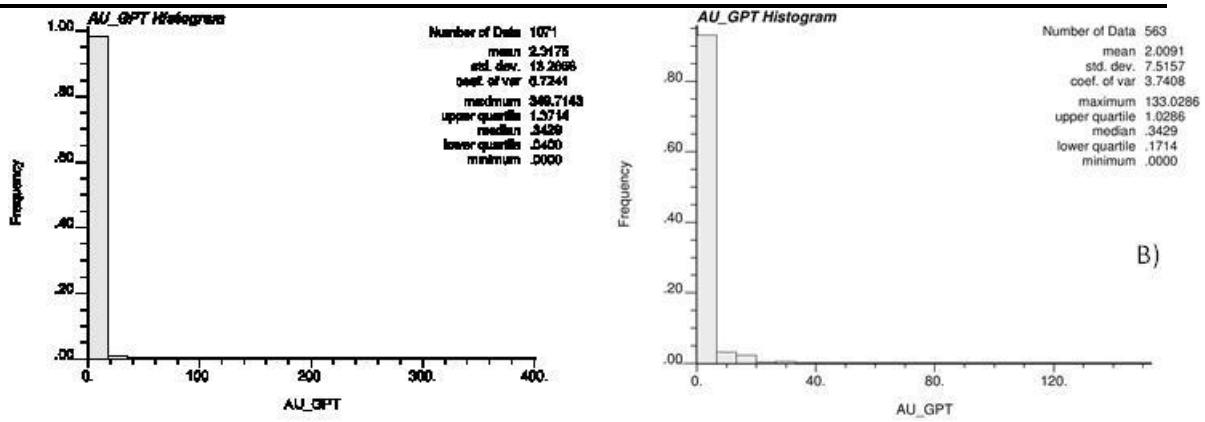

Figure 14.10 Histograms of Au for Cape Ray 41 Deposit. A) Domain 1 and B) Domain 2

14.3.1.6 Exploratory Data Analysis

A review of the drill hole database was done using histograms for all primary elements and X/Y scatter plots of element pairs (see Appendix A). Features watched for are outliers and irregularities in the element statistics. Univariate summary statistics for all elements are presented in Table 14.8 and correlation matrices are presented in Table 14.9. For the latter a correlation coefficient of 0.7 or higher indicates a good linear relationship between the bivariate components. A positive coefficient indicates that with increasing concentration of one element there is a sympathetic increase in the other. A negative coefficient indicates that as one element increases the other decreases.

The scatter plots for the element pairs do indicate issues as several of the elements, especially for Domain 1, have significant outliers. Indicator Kriging, the method chosen for estimation as it is the one Kriging method that handles highly skewed distributions the best, usually can handle the tail values, but due to the number and magnitude of these outliers something needed to be done as the histograms have significant gaps. Using a visual inspection of the scatter plots trimming values were selected and these outliers trimmed to these new values.

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Table 14.8 Summary Univariate Statistics for the Cape Ray 41 Deposit

Domain 1								
FIELD	**NSAMPLES**	**MINIMUM**	**MAXIMUM**	**MEAN**	**VARIANCE**	**STANDDEV**	**SKEWNESS**	**KURTOSIS**
AU_GPT	860	0	85.30	1.42	21.35	4.62	10.70	152.71
AG_GPT	860	0	276.54	4.99	313.64	17.71	7.99	89.18
CU%	860	0	8.86	0.04	0.08	0.28	26.09	802.75
PB%	860	0	7.63	0.12	0.22	0.46	9.39	114.01
ZN%	860	0	5.65	0.08	0.16	0.40	8.93	91.59
Domain 2								
FIELD	**NSAMPLES**	**MINIMUM**	**MAXIMUM**	**MEAN**	**VARIANCE**	**STANDDEV**	**SKEWNESS**	**KURTOSIS**
AU_GPT	428	0	40	1.409	14.81	3.85	5.91	42.94
AG_GPT	428	0	120	6.158	261.22	16.16	4.35	21.54
CU%	428	0	1.91	0.06	0.03	0.16	5.42	40.04
PB%	428	0	6.3	0.156	0.20	0.45	6.65	62.66
ZN%	428	0	1.39	0.071	0.03	0.19	4.53	22.85

Table 14.9 Correlation Matrices for the Cape Ray 41 Deposit`

Domain 1					
	AU_GPT	**AG_GPT**	**CU%**	**PB%**	**ZN%**
AU_GPT	1				
AG_GPT	0.4974	1			
CU%	0.1754	0.3803	1		
PB%	0.4568	0.6339	0.2144	1	
ZN%	0.3446	0.4777	0.1535	0.8808	1
Domain 1					
	AU_GPT	**AG_GPT**	**CU%**	**PB%**	**ZN%**
AU_GPT	1				
AG_GPT	0.8266	1			
CU%	0.8359	0.844	1		
PB%	0.805	0.7721	0.7918	1	
ZN%	0.6625	0.5641	0.5594	0.7182	1

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The trimming limits chosen are shown in Table 14.10

For Domain 1 all element pairs evaluated show a cloud and some have a crude linear relationship which is confirmed by the correlation coefficients that are, except for Pb and Zn, below 0.5. Pb and Zn have a correlation coefficient of 0.88 and exhibit a good linear correlation between the two elements.

For Domain 2 all element pairs evaluated, except for Zn-Au, Zn-Ag and Zn-Cu, have correlation coefficients greater than 0.7. These results are better than Domain 1 but this is more likely due to the prevalence of outliers in the Domain 1 data set that have had a negative influence on the correlation coefficient calculation because a visual examination of the scatter plots for Domain 1 show similar relationships to the scatter plots for Domain 2.

Au, the primary element of interest, does have a few spurious values with a maximum value of 349 g/t for Domain 1 and 133 g/t for Domain 2. The histograms for Au (Figure 14.10) are positively skewed distributions, typical of a lode gold deposit.

Exploratory data analysis showed that, other than the issue of the outliers, there are no issues with the drill hole database that would invalidate their use for resource estimation purposes.

Table 14.10 Sample Trimming Statistics – Cape Ray 41 Deposit

	Maximum Value	Trimming Value Used	#Samples Trimmed
Domain 1			
AU_GPT	349.7	40	7
AG_GPT	552	140	6
CU%	10.1	2	3
PB%	16.5	5	5
ZN%	8.23	4	7
Domain 2			
AU_GPT	133	40	3
AG_GPT	160.5	120	3
ZN%	3.3	1.6	6

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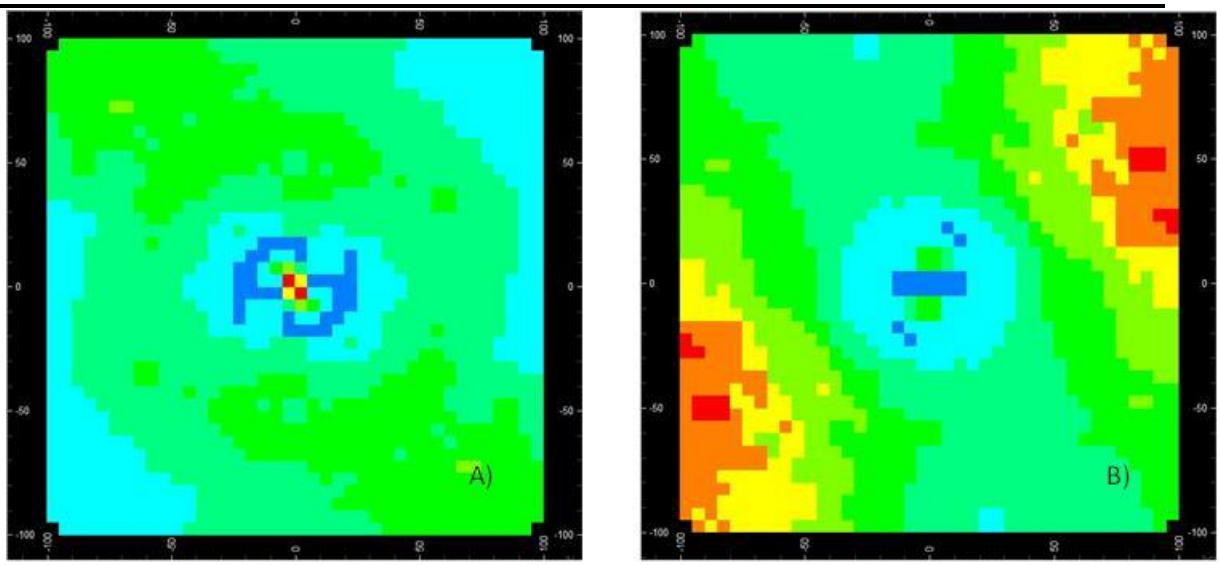

Figure 14.11 Variogram maps for the Cape Ray 41 Au-Ag Deposit. A) Domain 1 and B) Domain 2

14.3.1.7 Grade Variography

Prior to doing grade variography a custom script was used to prepare a variogram map for Au (see Figure 14.11) in order to check for a rotation in the primary direction of anisotropy. Directional lognormal variograms were calculated in 15° increments in the unfolded plane of the mineral zone. lognormal variograms provide a clearer picture of the spatial continuity as it reduces the influence of very large values on the calculation of the moment of inertia, common with highly positively skewed distributions (Isaaks and Srivastava, 1989).

The variogram maps do show a rotation (plunge) around the UCSA (across dip) axis for both domains of approximately -15° (counter-clockwise). They also show that the along strike direction (UCSC – Y axis) and down dip directions (UCSB – X axis) have similar ranges. The sill in both cases is reached at a range of approximately 50 metres.

The experimental grade variograms for the two domains (see Figure 14.12 and Figure 14.13) were calculated for the unfolded composited data sets using a custom script.

Typically, with inclined drilling, the across dip and down dip directions of the variogram are usually well defined due to the abundant sampling of the distance spectrum. This is indeed the case for both the down dip and along strike directions.

Table 14.11 Log Variogram Model Parameters for the Cape Ray 41 Deposit

	Domain 1	Domain 2
Variogram Models – Cape Ray 41	**AU_GPT**	**AU_GPT**
Nugget	2.511	0.577
1st spherical structure range A	1	1
1st spherical structure range B	1	40
1st spherical structure range C	18	28
1st spherical structure sill	1.490	0.266
2nd spherical structure range A	5	5
2nd spherical structure range B	45	50
2nd spherical structure range C	40	50
2nd spherical structure sill	3.385	4.932
Total sill	7.386	5.775

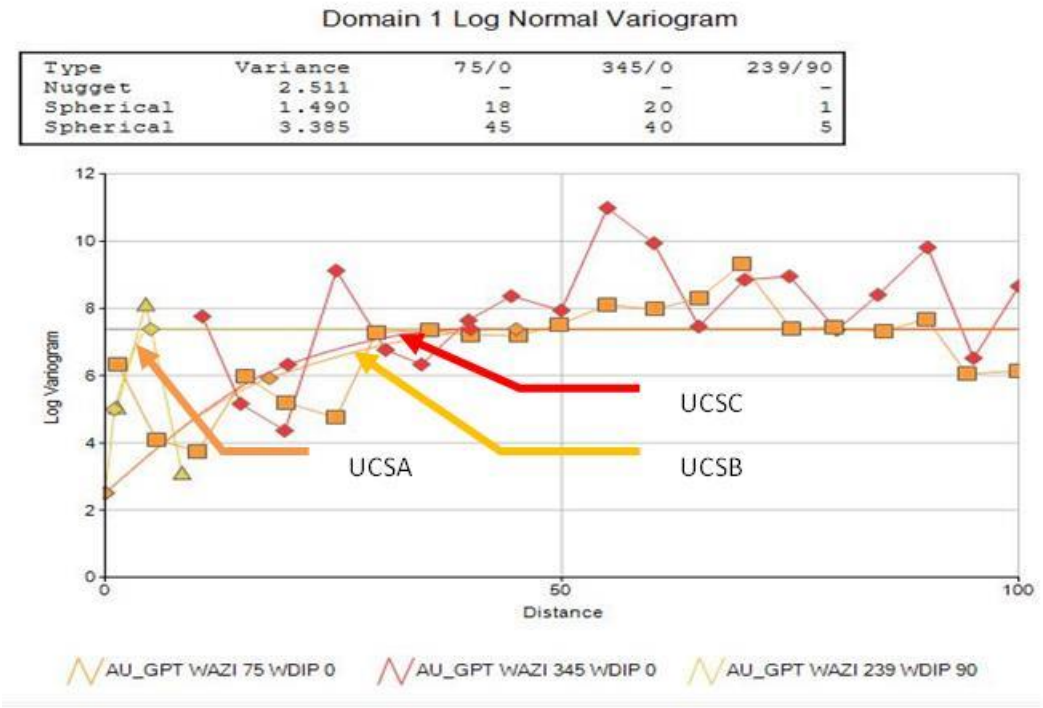

Figure 14.12 Experimental variograms and fitted models for Au – Cape Ray 41 Deposit Domain 1

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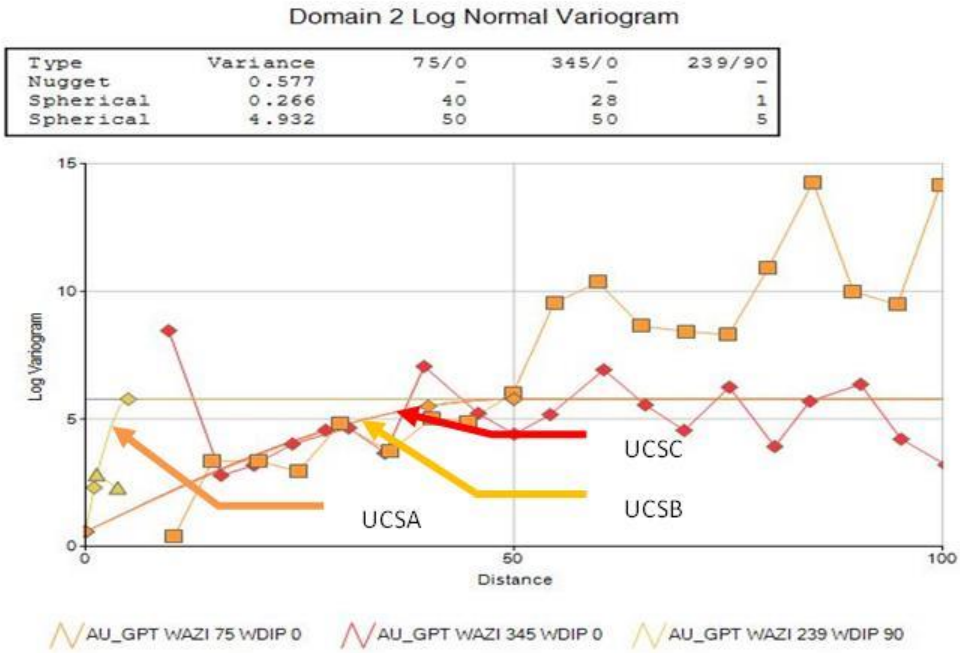

Figure 14.13 Experimental variograms and fitted models for Au – Cape Ray 41 Deposit Domain 2

In summary, there is a reasonable degree of confidence in the curves used to model all three directions.

The ranges derived from the log variogram models are shown in Table 14.11.

14.3.1.8 Block Size Determination

For the Cape Ray 41 deposit, due to the geometry and based on recommendations from mine engineering affiliated with this project, a block size suitable for evaluating underground mining methods as well as open pit mining was selected.

A block size of 3 metres by 3 metres by 3 metres was chosen as this meets the above criteria.

A custom script was used to create the empty prototype models and then fill them with blocks using the mineral envelopes for the domain wire frames. The resultant empty models were then regularized creating FILLVOL and VOIDVOL fields containing the volume for each block inside or outside the mineral domain wire frame.

14.3.1.8.1 Nearest Neighbour Block Model

Nearest Neighbour (NN) estimated models (nn_cr41d1_mod.dm and nn_cr41d2_mod.dm) were created for the two mineral domains in order to determine the de-clustered mean for the data. This mean can then be used to validate the kriged global estimate as all methods of estimation should produce essentially the same global mean, if done correctly. The de-clustered mean is also used in assessing smoothing and, if necessary, calculating a variance correction of the kriged model.

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Summary statistics comparing the nearest neighbour model to the sample file are presented in Figure 14.11. The mean values for the NN and IK models should normally be within 5% of one another if there are enough samples to confirm that the mean has been reproduced. In the case of the Cape Ray 41, for Domain 1, they all are close to or less than 5%. For Domain 2 they are a bit higher than 5% but less than 10% and appears to be due to a scale issue; the absolute difference is small but when viewed as a ratio results in values outside our usually acceptable threshold.

A visual inspection on a section-by-section and plan-by-plan basis comparing the input sample file with the resultant nearest neighbour file showed good correlation with the drill holes and proper spreading of the grade.

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Table 14.12 Summary Statistics for Samples and Models – Cape Ray 41 Deposit

Domain 1										
FILE	**FIELD**	**NRECORDS**	**NSAMPLES**	**MINIMUM**	**MAXIMUM**	**MEAN**	**% DIFF**	**VARIANCE**	**SKEWNESS**	**WGTFIELD**
cr04_data1u	AU_GPT	860	860	0	85.3	1.42		21.35	10.7	LENGTH
nn_cr04_mod	AU_GPT	30163	30163	0	30.63	1.15		6.6	7.306	FILLVOL
ik_cr04_mod	AU_GPT	30163	30163	0.02	8.49	1.23	6.35	0.674	1.674	FILLVOL
cr04_data1u	AG_GPT	860	860	0	276.54	4.99		313.644	7.986	LENGTH
nn_cr04_mod	AG_GPT	30163	30163	0	140	3.77		138.999	5.397	FILLVOL
ik_cr04_mod	AG_GPT	30163	30163	0.08	43.29	3.79	0.53	11.716	1.637	FILLVOL
cr04_data1u	CU%	860	860	0	8.86	0.04		0.078	26.094	LENGTH
nn_cr04_mod	CU%	30163	30163	0	1.76	0.04		0.012	5.607	FILLVOL
ik_cr04_mod	CU%	30163	30163	0	0.45	0.04	-0.97	0.001	2.046	FILLVOL
cr04_data1u	PB%	860	860	0	7.63	0.12		0.216	9.388	LENGTH
nn_cr04_mod	PB%	30163	30163	0	4.05	0.11		0.151	7.546	FILLVOL
ik_cr04_mod	PB%	30163	30163	0.01	1.43	0.11	3.39	0.017	2.187	FILLVOL
cr04_data1u	ZN%	860	860	0	5.65	0.08		0.163	8.933	LENGTH
nn_cr04_mod	ZN%	30163	30163	0	3.24	0.08		0.113	7.137	FILLVOL
ik_cr04_mod	ZN%	30163	30163	0.01	1.54	0.08	-1.07	0.013	2.675	FILLVOL
Domain 2										
FILE	**FIELD**	**NRECORDS**	**NSAMPLES**	**MINIMUM**	**MAXIMUM**	**MEAN**	**% DIFF**	**VARIANCE**	**SKEWNESS**	**WGTFIELD**
cr41_d2_data1u	AU_GPT	428	428	0	40	1.41		14.807	5.911	LENGTH
nn_cr41d2_mod	AU_GPT	15416	15416	0	40	1.5		16.495	5.487	FILLVOL
ik_cr41d2_mod	AU_GPT	15416	15416	0.01	15.2	1.64	9.5	2.993	1.911	FILLVOL
cr41_d2_data1u	AG_GPT	428	428	0	120	6.16		261.224	4.349	LENGTH
nn_cr41d2_mod	AG_GPT	15416	15416	0	120	6.08		308.1	4.243	FILLVOL
ik_cr41d2_mod	AG_GPT	15416	15416	0.11	45.8	6.61	8.82	37.675	1.142	FILLVOL
cr41_d2_data1u	CU%	428	428	0	1.91	0.06		0.025	5.416	LENGTH
nn_cr41d2_mod	CU%	15416	15416	0	1.91	0.06		0.025	5.149	FILLVOL
ik_cr41d2_mod	CU%	15416	15416	0.01	0.42	0.07	6.84	0.003	1.452	FILLVOL
cr41_d2_data1u	PB%	428	428	0	6.3	0.16		0.204	6.654	LENGTH
nn_cr41d2_mod	PB%	15416	15416	0	6.3	0.16		0.196	6.444	FILLVOL
ik_cr41d2_mod	PB%	15416	15416	0.02	1.45	0.17	6.9	0.024	1.734	FILLVOL
cr41_d2_data1u	ZN%	428	428	0	1.39	0.07		0.035	4.533	LENGTH
nn_cr41d2_mod	ZN%	15416	15416	0	1.39	0.08		0.038	4.184	FILLVOL
ik_cr41d2_mod	ZN%	15416	15416	0.01	0.82	0.08	9.39	0.007	2.235	FILLVOL

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14.3.1.8.2 Indicator Kriging Block Models

As noted previously, the grade distributions for the Cape Ray 41 Deposit are highly skewed. Such distributions often result in spatial estimation bias where by a few outliers representing the high-grade tail of the distribution are difficult to characterize. These have a significant impact on both the mean and variance for the grade population even though they only represent a small fraction of it. Since traditional linear estimation methods, including Ordinary Kriging, often misrepresent these outliers resulting in a smoothing of the overall variability and therefore in over-estimation, some other approach is needed. An Indicator approach has proven successful in addressing the problem of highly skewed populations and so Multiple Indicator Kriging was used to model the Cape Ray 41 deposit.

As discussed in Section 14.1.6, the indicator approach involves coding data into binary groupings using a series of grade cut-offs. The metal grade can be estimated by multiplying the mean of the binary grade cut-off groupings, by its probability, which is determined by Indicator Kriging, before summing them. Using this indicator approach, the metal estimated was interpolated for all of the 3 m x 3 m x 3 m cells contained within the block model for the Cape Ray 41 mineral domain.

After the parameter files needed for Indicator Kriging were constructed, a custom script was used carry out the Multiple Indicator Kriging process. Each cell in the block model was discretised using a matrix of 3 x 3 x 3 points in the ABC (unfolded) coordinate system. The Kriging functions were interpolated at each discretisation point using the same search volume as the nearest neighbour interpolation, based on the grade variogram results. In case of local low sample density a nested search was implemented.

14.3.1.8.3 Definition of Indicator Functions

The grade data for Cape Ray 41 are all highly skewed. To deal with such highly skewed frequency distributions of grade an indicator approach has been applied for grade estimation. The 1 m composites were coded according to the cut-offs determined from the de-clusterised cumulative statistics for all 5 grade fields using a custom script. The indicators used are listed in in Table 14.13.

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Table 14.13 Cape Ray 41 Deposit - Indicator Cut-offs Used

Domain 1					
Indicator	**AU_GPT**	**AG_GPT**	**CU%**	**PB%**	**ZN%**
1	0.171	0.2	0.02	0.085	0.044
2	0.343	0.34	0.048	0.181	0.188
3	0.686	4.1	0.07	0.285	0.355
4	0.985	7.395	0.19	0.372	0.51
5	1.567	10.395	0.197	0.53	0.519
6	2.202	17.226	0.25	0.695	1.14
7	2.957	21.943	0.354	1.159	1.33
8	3.245	28.566	0.487	2.251	2.09
9	3.771	46.006	0.732	3.064	2.622
10	4.541	63.738	0.775	3.569	3.006
11	6.678	75.757			
12	18.479	86.099			
Domain 2					
Indicator	**AU_GPT**	**AG_GPT**	**CU%**	**PB%**	**ZN%**
1	0.077	0.248	0.035	0.086	0.03
2	0.502	1.86	0.076	0.13	0.073
3	0.686	4.22	0.104	0.25	0.111
4	1.158	6.629	0.163	0.371	0.162
5	1.976	11.354	0.267	0.44	0.249
6	4.482	17.11	0.394	0.772	0.37
7	7.16	22.497	0.522	1.097	0.508
8	8.509	33.19	0.568	1.444	0.596
9	13.71	40.139	0.599	1.893	0.837
10	18.493	52.42	0.814	1.905	1.116
11	19.486	65.106			
12	24.961	78.2			

Initially each cut-off corresponds to the grade of the upper limit of each portion of the sample data representing approximately 9.1% of the total metal. As the low grade portions of the cumulative distribution functions are poorly sampled using this approach additional indicators were added to ensure proper sampling of the full distribution.

14.3.1.8.4 Indicator Variography

The indicator variograms for the Cape Ray 41 mineral domain were modelled using CAE Studio 3 variogram modeling tools.

One advantage of Indicator variograms is that they are often less erratic and easier to model than the grade variograms. This is because they do not suffer from having large differences between values.

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14.3.1.8.5 Indicator Kriging

A custom script was used to create the parameter files needed for Indicator Kriging. This was then followed by the use of another custom script to actually carry out the Indicator Kriging process. Each cell in the block model was discretised using a matrix of 3 x 3 x 3 points. The indicator functions were interpolated at each discretisation point using the same search volume as the nearest neighbour interpolation, based on the grade variogram results. In case of local low sample density a nested search was implemented.

For the Domain 1 model 35.0% of the blocks were estimated in the first search, 26.3% in the second and 38.7% in the third. The Domain 2 model estimated 35.0% of the blocks in the first search, 26.3% in the second and 38.7% in the third.

14.3.1.8.6 Investigation of Kriging Results

In theory estimated indicator functions should range between 0 and 1, but as mentioned in 14.1.6, Ordinary Kriging can create negative weights that may cause the interpolated values to be outside of this range. The Cape Ray 41 deposit had no significant overlap issues caused by negative weights; all of the negative indicator values were set to 0.00 and all values over 1 were set to 1 using a custom script during post processing.

Results were also checked to ensure that indicator values decrease as the cut-off increases. Although order relations issues (probability increases as the grade cut-off increases) can be expected when different variograms are used for the indicator functions, they should not be more than approximately 1% of the estimates. In the case of Cape Ray 41 data there are no significant cumulative proportions issues.

14.3.1.8.7 Volume Variance Correction

For Cape Ray 41 Domain 1 the smoothing ratio for gold is 3.62, for silver 3.84, for Cu is 4.27, for Pb is 3.56 and for Zn it is 3.46. All show that there is some smoothing and as a result a correction was required. Similarly for Domain 2 the smoothing ratio for gold is 3.23, for silver 4.96, for Cu is 4.45, for Pb is 5.46 and for Zn it is 2.78.

The variance of the model for all elements has been increased and therefore more in line with the expected variance with respect to block size. As should be expected, the mean remains essentially unchanged by the transformation.

14.3.1.9 Model Verification

Validation procedures were carried out on the estimated block models including visually checking the sample file against estimated blocks. The sample grades were found to reasonably match the estimated block grades in the model.

A global statistical comparison of the global means of all estimations method was done. The difference between all the global means was found in most cases to be close to or less than

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approximately 5%, but none were over 10% indicating the process completed correctly (see Table 14.12). Where differences do exceed 5% it is easily explainable as a scale issue.

Other statistical checks that were done include the use of swath plots (see Appendix C). Swath plots compare the moving average of the mean for both IK and NN models and the sample file using panels, or "swaths" through the mineral envelope. As this is best done when the data is within a rectilinear volume the unfolded coordinates were used to define the swaths. The result is a curve for each data set. The curves for the models should inter-weave with the sample curve and the two model curves should be sympathetic with one another with no major deviations from one another. No issues were noted.

14.4 Cape Ray 51 Deposit

14.4.1 Resource Estimation Methodology

14.4.1.1 Data Validation

Core-drilling data were supplied as Microsoft Excel tables that included collar information, assays, lithology information, down hole survey information and specific gravity data. Sibley Basin Group compiled the data into a Microsoft Access database and completed data validation. Once validated this information was imported into CAE Studio 3 as four tables: a collar file, an assay file, a lithology file, and a survey file. As the data supplied were in UTM coordinates a transformation was completed to make a copy of the drill hole file in local grid coordinates based on the collar location of hole PB80-156. The collar for this hole has a World Coordinate System (WCS) location of 356005.19E, 5290825.26N with a Local Coordinate System (LCS) location of 334.61W, 136.37S. Using the CAE Studio 3 HOLES3D routine a desurveyed drill hole file, cr_51_holes_lg.dm, was created.

In fall of 2015 the drill hole file was modified by incorporating surface channel samples that had been collected during the summer exploration program (see Section 9 for details). All 23 channels, comprised of a total of 104 samples, were treated as horizontal drill holes. This drill hole file incorporating the merged channel samples was last updated on September 10, 2015.

The CAE Studio 3 desurveying routine, HOLES3D, does a rigorous set of validation checks including checking for duplicate borehole numbers, missing survey data and overlapping sample intervals. If present, it generates a summary report with a list of all errors encountered. This file was checked to determine if any errors occurred and where appropriate corrections were made. Once it had been confirmed that no errors remained the drill hole file was then used for subsequent steps.

There are density or specific gravity values in the data set provided, but a previous resource estimate for the project area (Ténière & Hilchey, 2012) had specific gravity (SG) values determined on 44 samples. From these an average value was calculated from 6 samples that fell below the 25th percentile and used as a global value. This same value of 2.84 g/cm3 has been used for this project.

14.4.1.2 Geological Domains

Experienced geologists had coded each rock unit based on core logging description. All of the holes are inclined and most intersected at least some portion of the deformation zone that hosts the gold mineralisation. Construction of the resource block model was controlled by building a wire frame that was then used to isolate related samples. No cut-off was used to limit the extent of this mineral envelope. The envelope for the mineral domain (see Figure 14.14) extends from an elevation of 331 metres above sea level down to a maximum depth of about 71 metres below sea level. It is not a geological envelope as the deformation zone hosting the mineralisation cuts stratigraphy.

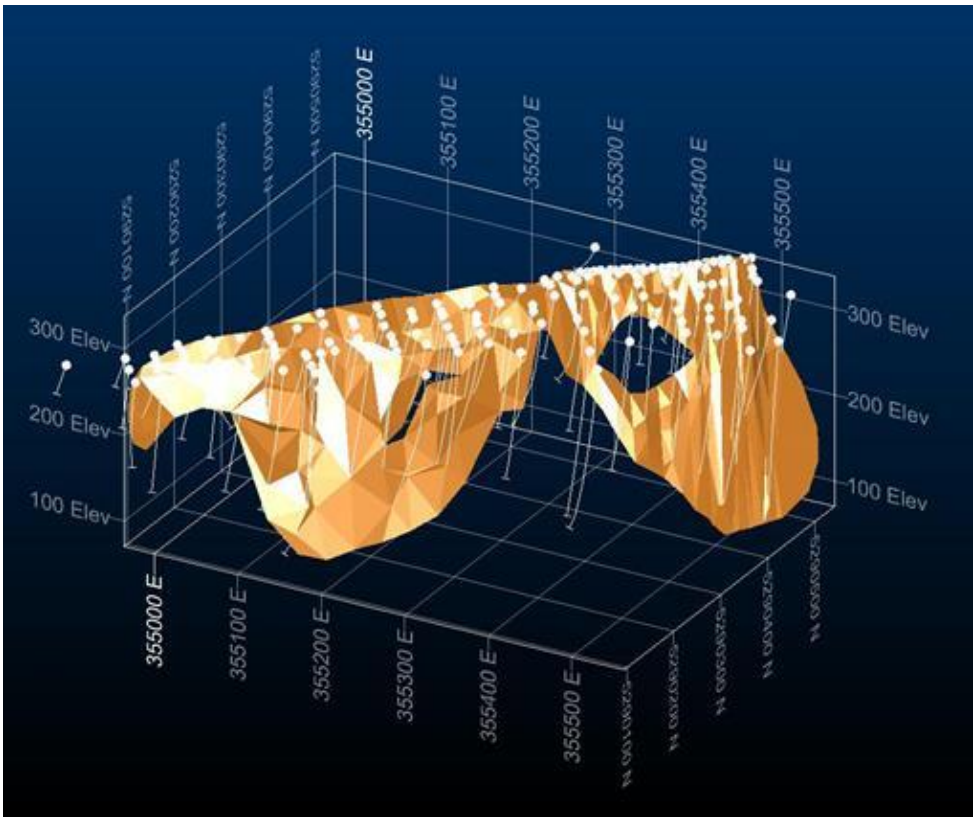

Figure 14.14 Isometric view of the Cape Ray 51 geological domain used.

A total of 106 holes have penetrated the mineral zone (see Figure 14.15)

14.4.1.3 Drill Hole Database

For the Cape Ray 51 deposit the file used for resource estimation purposes, "*cr_51_all_lg.dm*", contains information for 137 drill holes totalling 21,097 metres and with 1,956 samples collected and 23 channels totalling 91.7 metres.

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14.4.1.4 Sample Selection

Working in cross section a set of mineral zone lines, or strings, was defined for the domain. These strings were drawn to enclose the Cape Ray 51 Au-Ag zone by snapping to the drill holes. The strings were then used to construct a mineral envelope wire frame for the domain. The envelope extends from 331 metres above mean sea level down to about 71 metres below mean sea level. The borehole samples located within the mineral envelopes were captured using a custom script.

14.4.1.5 Compositing

The captured samples have an average sample length of 0.77 metres (see Figure 14.16). A block size that will allow a reasonable amount of selectivity is 3 metres in all three dimensions. A 1 metre composite length represents a multiple of these blocks dimensions ensuring uniform support.

Table 14.14 Summary of Diamond Drilling for the Cape Ray 51 Deposit

DDH	UTM East	UTM North	Elevation	Length	Grid East	Grid North	Azimuth	Dip
PB77-033	355288.6	5290518.8	315.2	129.9	5493.0	5055.1	322.35	-45
PB78-050	354822.0	5289771.5	296.0	220.7	4667.9	4746.2	322.35	-45
PB78-051	355006.4	5290083.7	306.9	91.4	5004.2	4881.6	322.35	-45
PB78-052	355257.5	5290358.6	329.5	85.3	5370.8	4946.9	322.35	-45
PB78-057	355029.2	5290101.8	308.8	91.4	5033.3	4882.1	322.35	-45
PB78-058	354981.2	5290066.6	304.2	91.4	4973.8	4883.4	322.35	-45
PB78-062	355074.1	5290146.0	316.4	91.4	5095.9	4889.9	322.35	-45
PB78-063	355119.2	5290186.4	321.6	94.5	5156.3	4894.5	322.35	-45
PB78-066	355078.4	5290037.6	323.1	176.8	5033.3	4801.3	322.35	-60
PB79-088	355445.1	5290511.5	337.1	94.5	5612.7	4954.0	322.35	-45
PB79-089	355357.4	5290423.9	335.9	85.3	5489.8	4937.9	322.35	-45
PB79-090	355164.0	5290224.5	326.1	91.4	5214.9	4897.4	322.35	-45
PB79-091	355167.5	5290117.9	332.5	182.9	5152.9	4810.7	322.35	-60
PB79-092	354925.3	5290040.3	294.4	88.8	4913.5	4896.6	322.35	-45
PB79-093	355031.5	5289998.8	317.3	170.7	4972.5	4799.1	322.35	-60
PB79-094	354810.7	5289940.8	278.0	77.7	4762.0	4887.5	322.35	-45
PB79-095	354583.6	5289738.1	244.1	76.2	4458.4	4864.9	322.35	-45
PB79-096	354366.6	5289438.0	217.0	91.4	4103.5	4758.9	322.35	-45
PB79-098	355405.3	5290460.4	338.3	91.4	5550.0	4937.7	322.35	-45
PB79-099	355308.9	5290387.0	332.2	90.9	5428.8	4938.1	322.35	-45
PB79-100	355394.7	5290372.4	342.9	164.6	5488.1	4874.3	322.35	-60
PB79-101	355444.9	5290408.4	344.4	173.7	5549.8	4872.3	322.35	-60
PB79-102	355213.2	5290262.1	331.3	111.3	5276.9	4897.3	322.35	-45
PB79-103	355200.2	5290173.3	335.0	170.7	5212.5	4834.8	322.35	-60

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DDH	UTM East	UTM North	Elevation	Length	Grid East	Grid North	Azimuth	Dip
PB79-104	355113.4	5290089.3	326.1	173.7	5092.5	4821.0	322.35	-60
PB79-105	355245.8	5290323.8	331.0	91.4	5340.3	4926.4	322.35	-45
PB79-106	355252.8	5290204.1	340.2	179.8	5273.0	4827.2	322.35	-60
PB79-107	355287.0	5290269.7	341.7	167.6	5340.1	4858.4	322.35	-60
PB79-108	355327.0	5290318.0	342.6	149.4	5401.2	4872.4	322.35	-60
PB79-110	355320.4	5290103.7	350.8	345.2	5265.5	4706.4	322.35	-70
PB79-111	355362.3	5290274.0	348.1	228.6	5402.4	4816.0	322.35	-70
PB79-114	355232.9	5290032.2	344.1	356.6	5152.6	4702.9	322.35	-70
PB79-129	355512.8	5290423.6	348.4	192.0	5612.9	4843.1	322.35	-60
PB79-130	355384.7	5290310.7	343.2	195.1	5442.6	4831.5	322.35	-60
PB79-131	354667.5	5290117.6	249.3	121.9	4756.0	5114.9	322.35	-45
PB79-133	354533.1	5289902.2	240.5	135.6	4518.2	5025.8	322.35	-45
PB79-135	355458.7	5290443.8	343.2	149.4	5582.3	4892.0	322.35	-60
PB79-136	355382.7	5290441.0	336.5	96.9	5520.3	4936.0	322.35	-45
PB79-137	355496.1	5290392.8	347.8	207.3	5580.9	4828.8	322.35	-60
PB79-139	355034.4	5290040.5	315.5	161.5	5000.2	4830.4	322.35	-55
PB79-140	355086.7	5290075.2	322.8	172.2	5062.8	4826.1	322.35	-55
PB79-142	355098.3	5290164.5	318.8	106.7	5126.4	4889.8	322.35	-45
PB79-143	355120.8	5290132.8	325.8	164.6	5124.9	4851.0	322.35	-55
PB79-144	355116.8	5290239.8	316.1	51.8	5186.9	4938.3	322.35	-45
PB79-145	355175.5	5290155.4	332.2	152.4	5182.1	4835.6	322.35	-55
PB79-146	355166.8	5290271.4	322.5	70.1	5245.8	4932.9	322.35	-45
PB79-147	355207.2	5290215.0	333.5	140.2	5243.5	4863.6	322.35	-55
PB79-148	355141.5	5290100.6	330.1	234.7	5121.7	4812.8	322.35	-60
PB80-170	355320.6	5290219.7	349.3	230.7	5336.4	4828.8	322.35	-70
PB86-173	355505.5	5290479.1	344.4	193.2	5638.2	4891.4	331.35	-70
PB86-177	354652.7	5289771.3	259.4	136.9	4533.4	4849.2	322.35	-45
PB86-179	354509.9	5289743.7	237.7	184.4	4403.3	4914.2	327.85	-55
PB87-215	355338.7	5290204.2	348.9	253.9	5341.2	4774.9	322.35	-45
PB87-218	355278.6	5290178.3	344.4	228.9	5277.8	4791.0	322.35	-55
PB87-222	355206.1	5290070.6	339.6	277.7	5154.7	4749.7	322.35	-60
PB88-256	355320.8	5290171.0	349.0	274.3	5306.8	4759.5	322.35	-65
PB88-258	355381.5	5290196.4	352.2	293.2	5370.4	4742.8	322.35	-65
PB88-259	355337.3	5290301.5	344.6	170.7	5399.4	4853.0	322.35	-55
PB88-261	355418.6	5290391.2	343.7	173.7	5518.4	4874.7	322.35	-55
PB88-262	355003.4	5290034.7	310.3	148.1	4972.0	4844.7	322.35	-55
PB88-263	355358.2	5290370.6	340.4	135.9	5458.0	4895.1	322.35	-55
PB88-264	355168.9	5290069.1	334.5	264.9	5124.2	4771.1	322.85	-65
PB88-265	355061.8	5290109.7	316.2	125.0	5064.0	4868.6	322.35	-55

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PB88-266	355039.9	5290138.7	309.1	85.7	5064.3	4904.9	323.55	-55
PB88-267	355106.7	5290150.2	320.8	140.2	5124.3	4873.4	322.35	-55
PB88-268	355262.5	5290299.6	335.7	129.5	5338.8	4897.1	322.35	-55
PB88-269	355182.6	5290123.9	334.4	206.7	5168.5	4806.3	322.35	-60
PB88-270	355185.6	5290247.0	327.8	107.0	5245.8	4902.1	322.35	-50
PB88-271	355244.7	5290269.8	335.5	130.2	5306.6	4884.2	322.35	-55
PB88-272	355042.9	5290082.6	313.8	123.8	5032.5	4858.6	322.35	-55
PB88-273	355274.2	5290231.3	341.3	187.8	5306.6	4835.8	322.35	-60
PB88-274	355051.1	5290022.5	317.9	198.1	5002.4	4805.9	322.35	-65
PB88-275	355326.1	5290267.0	345.3	199.6	5369.5	4832.5	322.35	-60
PB88-276	355019.9	5290062.9	310.7	121.9	5002.3	4857.0	322.35	-45
PB88-277	355010.4	5290125.4	302.4	62.5	5032.8	4912.3	322.35	-55
PB88-278	355332.9	5290354.5	339.3	131.4	5428.1	4897.8	322.35	-55
PB88-279	355290.6	5290313.1	338.3	126.8	5369.4	4890.7	322.35	-55
PB88-280	355091.8	5290069.9	323.1	184.4	5063.6	4818.8	322.35	-60
PB88-281	355287.4	5290364.6	332.5	78.0	5398.2	4933.5	322.35	-45
PB88-282	355328.5	5290410.1	332.4	73.5	5458.5	4944.5	322.35	-45
PB88-283	355349.4	5290237.4	348.7	211.5	5370.0	4794.8	322.35	-55
PB88-284	355089.8	5290172.5	317.2	100.0	5124.5	4901.4	322.35	-45
PB88-286	355138.8	5290208.1	322.4	97.8	5185.1	4899.8	322.35	-50
PB88-287	355175.2	5290209.4	328.3	121.9	5214.7	4878.6	322.35	-55
PB88-288	355427.7	5290477.2	337.7	75.0	5578.0	4937.4	322.35	-50
PB88-289	355293.1	5290206.9	344.4	218.2	5306.7	4804.9	322.35	-55
PB88-291	355226.3	5290243.4	334.5	129.5	5275.9	4874.5	322.35	-50
PB88-292	355470.3	5290419.4	345.3	145.1	5576.7	4865.6	322.35	-55
PB88-293	355230.4	5290188.2	338.4	195.4	5245.6	4828.2	322.35	-65
PB88-294	355511.6	5290407.6	348.6	208.8	5602.2	4831.1	322.35	-60
PB88-295	355297.5	5290251.8	343.6	200.0	5337.5	4837.8	322.35	-60
PB88-296	355487.8	5290349.6	348.7	243.8	5548.1	4799.6	322.35	-60
PB88-298	355307.2	5290188.4	346.9	263.0	5306.6	4781.6	322.35	-65
PB88-300	355192.0	5290088.8	336.9	254.8	5154.6	4772.7	322.35	-70
PB88-301	355465.8	5290278.5	350.1	283.2	5487.2	4756.5	328.10	-70
PB88-304	355239.0	5290127.1	341.9	250.9	5215.2	4774.5	322.35	-65
PB88-305	355440.5	5290238.9	349.7	319.4	5443.1	4740.5	322.35	-65
PB88-308	355412.1	5290276.5	350.0	210.3	5443.5	4787.6	322.35	-60
PB88-310	355258.7	5290151.2	343.5	228.6	5245.4	4781.6	322.35	-65
PB88-313	355572.7	5290389.7	353.6	221.3	5639.7	4779.7	322.35	-60
PB88-314	355368.4	5290407.4	337.5	76.2	5488.5	4918.1	322.35	-55
PB88-315	355225.1	5290295.5	329.1	88.7	5306.7	4916.6	322.35	-50
PB88-318	355481.9	5290509.4	340.3	67.1	5640.7	4929.9	322.35	-45

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PB88-319	355394.6	5290421.7	339.6	76.2	5518.0	4913.5	322.35	-55
PB88-321	355419.9	5290439.3	341.0	76.2	5548.8	4912.1	322.35	-55
PB88-322	355555.8	5290461.0	348.3	149.4	5669.8	4846.6	322.35	-58
PB88-324	355493.1	5290492.1	342.2	82.3	5639.0	4909.4	322.35	-55
PB88-325	355173.1	5290112.1	333.0	227.4	5153.8	4802.8	322.35	-65
PB89-332	355507.7	5290323.4	351.6	262.1	5547.9	4766.7	322.35	-70
PB89-334	355463.9	5290331.0	347.8	205.7	5517.7	4799.4	326.35	-60
PB89-336	355511.7	5290366.9	349.7	206.4	5577.5	4798.7	322.35	-60
PB89-338	355555.9	5290357.6	353.9	264.6	5607.0	4764.5	322.35	-70
PB89-341	355475.3	5290192.7	356.2	379.5	5442.6	4682.6	322.35	-70
PB89-342	355244.4	5290169.7	341.0	218.9	5245.4	4805.0	325.35	-65
PB89-343	355193.8	5290134.5	335.5	213.4	5183.8	4807.8	324.35	-65
PB89-353	355592.7	5290308.9	357.1	331.0	5606.5	4703.4	322.35	-70
PB89-354	355542.7	5290378.6	350.8	228.6	5609.2	4789.1	322.35	-70
PB89-355	355540.3	5290331.9	352.6	271.9	5578.9	4753.5	322.35	-70
PB89-356	355606.9	5290344.1	356.4	314.6	5639.1	4722.7	322.35	-70
PB96-361	355424.2	5290359.3	345.1	187.5	5503.4	4846.0	322.35	-60
PB96-363	355412.8	5290488.3	332.9	27.5	5572.9	4955.3	321.35	-52
PB96-364	355294.7	5290400.9	328.9	27.4	5426.0	4957.8	322.35	-52
PB96-365	355409.0	5290475.5	332.2	28.7	5562.1	4947.4	322.35	-52
PB96-366	355301.2	5290416.5	327.9	14.4	5440.7	4966.2	322.35	-52
PB96-367	355397.1	5290489.6	330.5	30.5	5561.3	4965.9	322.35	-52
PB96-368	355330.3	5290429.2	330.2	29.6	5471.5	4958.5	322.35	-52
PB96-369	355316.3	5290420.7	329.1	26.5	5455.3	4960.4	322.35	-46
PB96-370	355357.6	5290447.6	331.6	31.7	5504.4	4956.6	322.35	-50
PB96-371	355378.0	5290466.3	331.6	30.5	5532.0	4959.0	322.35	-52
PB96-372	355368.8	5290460.0	330.7	20.4	5520.8	4959.6	322.35	-46
PB96-373	355438.1	5290490.1	337.4	37.2	5594.1	4941.2	322.35	-50
PB96-374	355411.9	5290450.5	338.3	60.1	5549.2	4925.8	322.35	-46
PB96-375	355464.4	5290456.5	340.5	91.8	5594.5	4898.7	322.35	-53
PB96-376	355346.5	5290443.3	330.7	29.6	5492.9	4959.9	322.35	-46
PB96-377	355278.1	5290325.8	338.3	92.1	5367.1	4908.3	322.35	-53
51Z-04-01	355126.0	5290072.0	328.0	215.8	5092.0	4799.6	316.55	-65
51Z-04-02	355251.0	5290263.0	335.0	142.4	5307.4	4875.0	325.95	-62
51A	355463.8	5290539.8	336.0	1.5			316.0	0.0
51B	355458.5	5290534.4	336.0	2.3			316.0	0.0
51C	355446.3	5290522.5	337.1	3.0			327.0	0.0
51D	355440.3	5290518.3	337.0	2.7			327.0	0.0
51E	355434.6	5290512.3	336.3	5.7			327.0	0.0
51F	355427.5	5290508.0	335.0	5.3			327.0	0.0

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51G	355420.4	5290502.4	333.6	6.8			323.0	0.0
51H	355414.5	5290499.7	332.5	5.2			323.0	0.0
51I	355407.8	5290494.1	331.4	6.1			327.0	0.0
51J	355400.4	5290490.1	331.0	4.3			327.0	0.0
51K	355395.5	5290486.6	331.0	4.9			326.0	0.0
51L	355390.1	5290482.1	331.0	5.0			326.0	0.0
51M	355381.3	5290477.5	330.5	3.7			323.0	0.0
51N	355372.7	5290472.5	330.3	2.5			323.0	0.0
51O	355368.0	5290468.3	330.3	3.0			326.0	0.0
51P	355360.7	5290465.1	330.0	3.6			326.0	0.0
51Q	355355.3	5290461.0	330.0	5.1			334.0	0.0
51R	355347.9	5290458.4	328.0	5.2			334.0	0.0
51S	355342.5	5290456.1	328.5	3.4			323.0	0.0
51T	355333.7	5290451.0	328.0	2.7			323.0	0.0
51U	355328.8	5290445.7	328.0	3.5			323.0	0.0
51V	355324.0	5290440.4	328.0	2.9			323.0	0.0
51W	355317.4	5290436.2	328.0	3.2			323.0	0.0

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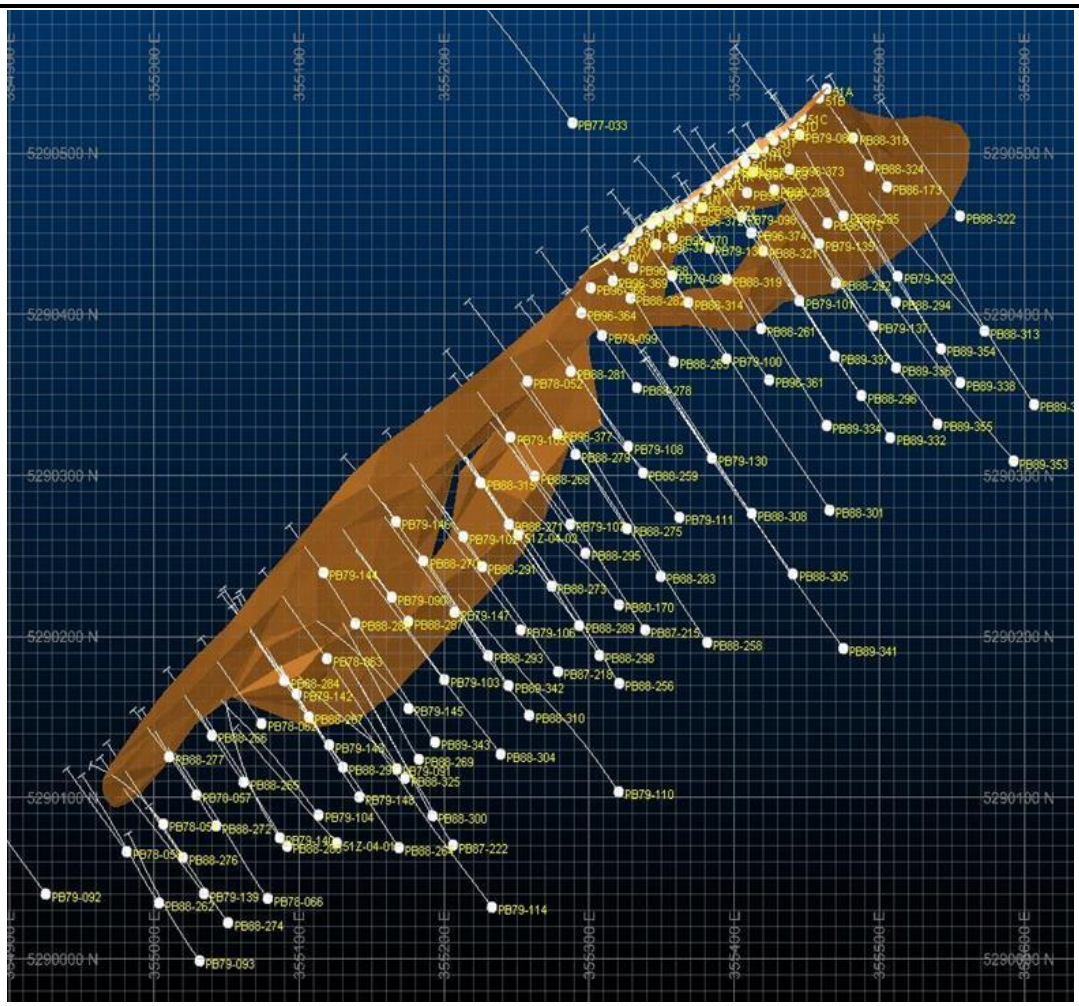

Figure 14.15 Plan view of Cape Ray 51 Deposit drilling

The samples were composited to standard 1 m intervals using the CAE Studio 3 process COMPDH. The COMPDH process starts the composites at the beginning of the selected data interval and leaves any remainder at the end of the interval. This results in most holes having one sample with a length less than the established composite length. For grade estimation purposes, drill composites are treated like point data (i.e. their length is not used), thus the need to composite to a standard sample length to eliminate any sample bias. To avoid bias from a very short sample being treated the same as a standard sample any sample lengths that were less than 40% of the composite length were rejected.

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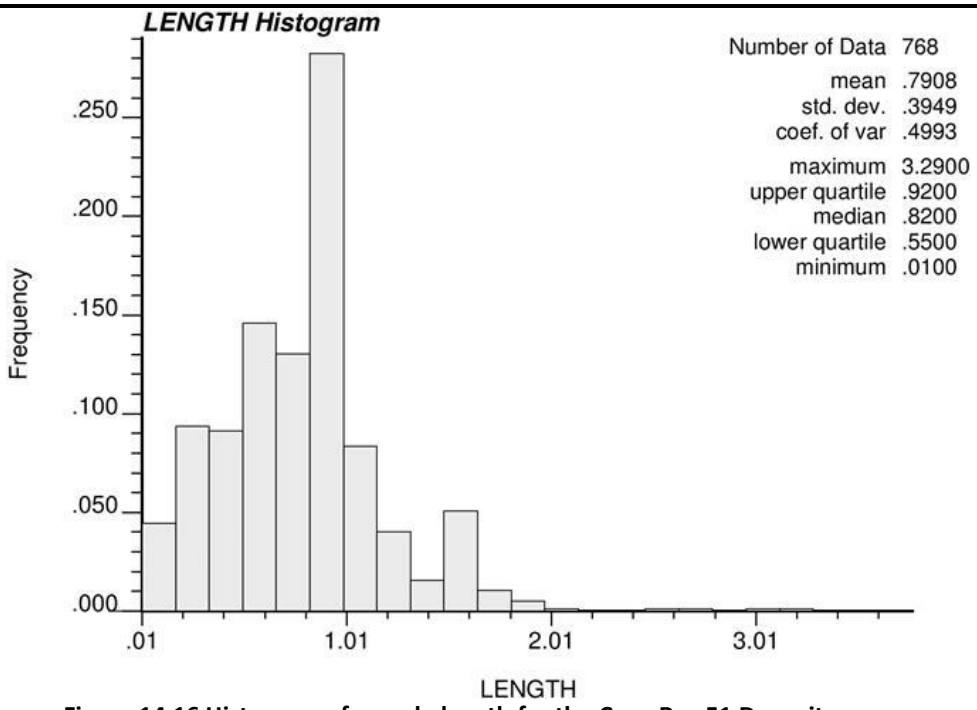

Figure 14.16 Histogram of sample length for the Cape Ray 51 Deposit

14.4.1.6 Exploratory Data Analysis

A review of the drill hole database was done using histograms for all primary elements and X/Y scatter plots of element pairs (see Appendix A). Features watched for are outliers and irregularities in the element statistics. Univariate summary statistics for all elements are presented in Table 14.15 and a correlation matrix is presented in Table 14.16. For the latter a correlation coefficient of 0.7 or higher indicates a good linear relationship between the bivariate components. A positive coefficient indicates that with increasing concentration of one element there is a sympathetic increase in the other. A negative coefficient indicates that as one element increases the other decreases.

The scatter plots for the element pairs do indicate issues as all of the elements have significant outliers. Indicator Kriging, the method chosen with which to do estimation as it is the one Kriging method that handles highly skewed distributions the best, usually can handle the tail values, but because of the number and magnitude of these outliers something needed to be done as the histograms have significant gaps. Using a visual inspection of the scatter plots trimming values were selected and these outliers trimmed to these new values. The trimming limits chosen are shown in Table 14.17.

Au, the primary element of interest, has some spurious values with a maximum value of 300 g/t. The histogram for Au (Figure 14.17) is a positively skewed distribution, typical of a lode gold deposit.

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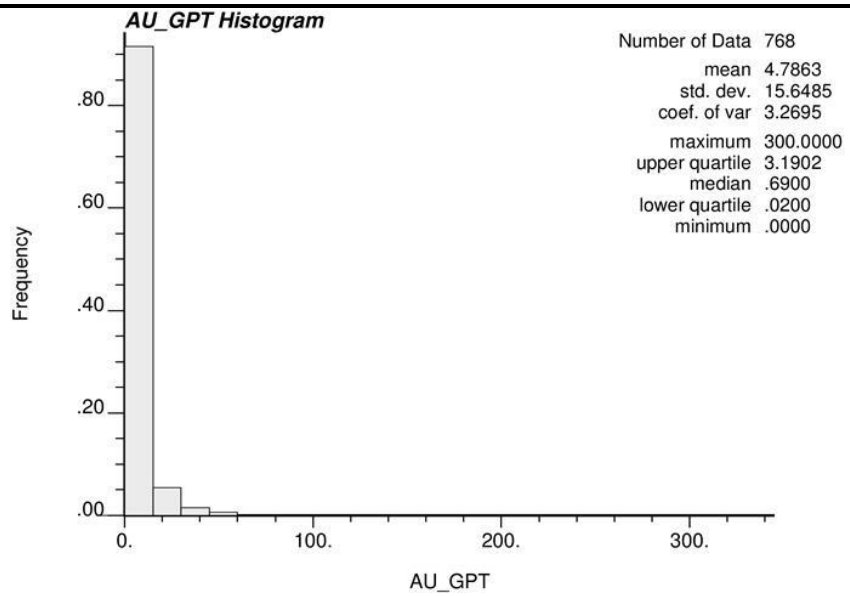

Figure 14.17 Histogram of Au for Cape Ray 51 Deposit

Table 14.15 Summary Univariate Statistics for the Cape Ray 51 Deposit

FIELD	NSAMPLES	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDDEV	SKEWNESS	KURTOSIS
AU_GPT	769	0	300.00	4.43	183.89	13.56	10.87	190.22
AG_GPT	769	0	452.00	13.27	1428.42	37.79	5.88	47.69
CU%	769	0	3.88	0.08	0.08	0.29	7.50	71.48
PB%	769	0	26.20	0.30	2.12	1.46	11.83	186.13

Exploratory data analysis showed that, other than the issue of the outliers, there were no issues with the drill hole database that would invalidate their use for resource estimation purposes.

Table 14.16 Correlation Matrix for the Cape Ray 51 Deposit

	AU_GPT	AG_GPT	CU%	PB%	ZN%
AU_GPT	1				
AG_GPT	0.6631	1			
CU%	0.471	0.6429	1		
PB%	0.4691	0.6463	0.8129	1	
ZN%	0.3832	0.4777	0.5806	0.7897	1

Table 14.17 Sample Trimming Statistics – Cape Ray 51 Deposit

	Maximum Value	Trimming Value Used	#Samples Trimmed
AU_GPT	300	40	15
AG_GPT	384	80	33
CU%	3.1	0.6	25
PB%	10.3	2	33
ZN%	5.11	1.7	13

14.4.1.7 Grade Variography

Prior to doing grade variography a custom script was used to prepare a variogram map for Au (see Figure 14.18) in order to check for a rotation in the primary direction of anisotropy. Directional lognormal variograms were calculated in 15° increments in the unfolded plane of the mineral zone. lognormal variograms provide a clearer picture of the spatial continuity as it reduces the influence of very large values on the calculation of the moment of inertia, common with highly positively skewed distributions such as with nickel sulphide deposits (Isaaks and Srivastava, 1989).

The variogram map shows a rotation (plunge) around the UCSA (across dip) axis of approximately 15° counter clockwise. It also shows that the along strike direction (UCSC – Y axis) is very poorly defined indicating the drill hole spacing in this direction is too far apart to properly define the variogram in this direction. The sill is reached at a range of approximately 40 metres. Due to the nature of the drilling (primarily on 30 metre centres) the along strike direction (the Y axis of the map) is poorly defined.

The experimental grade variograms (see Figure 14.19) were calculated for the unfolded composited data sets using a custom script.

Typically, with inclined drilling, the across dip and down dip directions of the variogram are usually well defined due to the abundant sampling of the distance spectrum. This is indeed the case for the down dip direction. The along strike variogram is poorly defined due to the poor sampling resulting from the use of 30 metre sectional drilling..

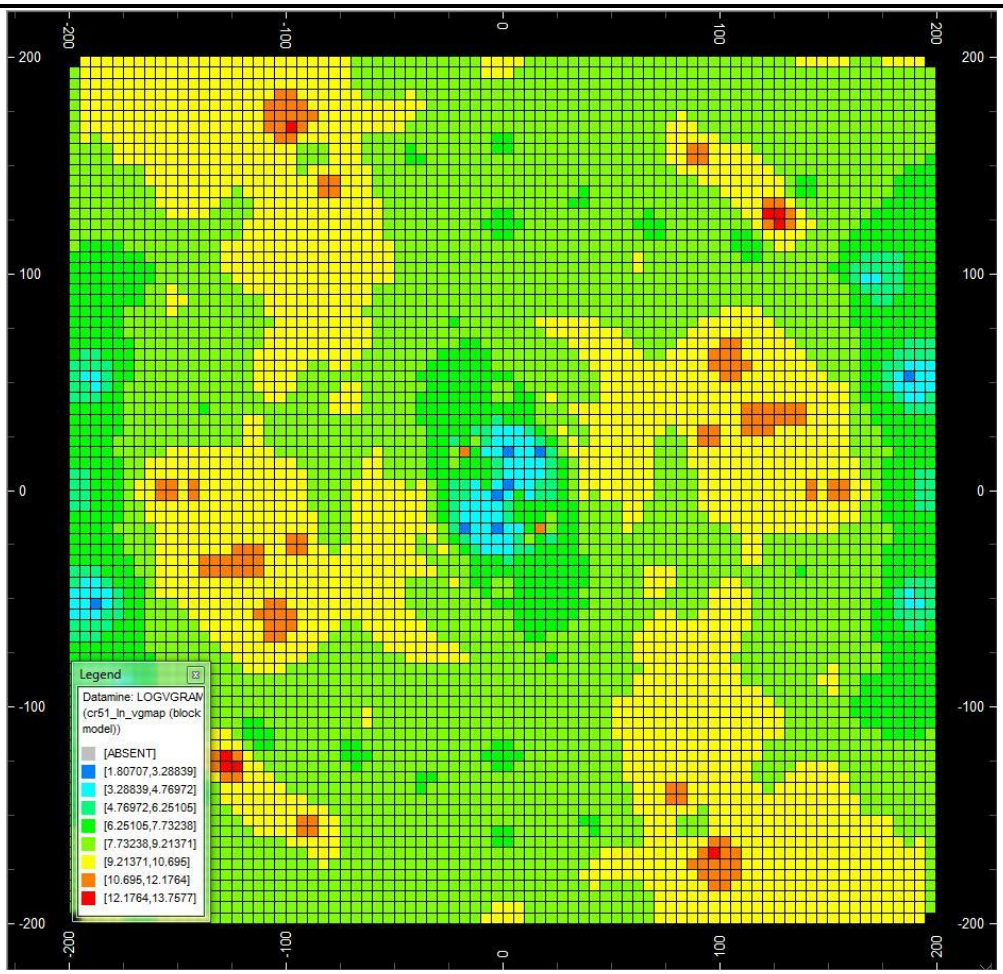

Figure 14.18 Variogram map for the Cape Ray 51 Au-Ag Deposit

In summary, there is a reasonable degree of confidence in the curves used to model the down dip and across strike directions. There is not enough sampling in the along strike direction, so it was assumed to have the same ranges in that direction as those used in the down dip direction.

The ranges derived from the log variogram models are shown in Table 14.18.

14.4.1.8 Block Size Determination

For the Cape Ray 51 deposit, due to the geometry and based on recommendations from mine engineering affiliated with this project, a block size suitable for evaluating underground mining methods as well as open pit mining was selected.

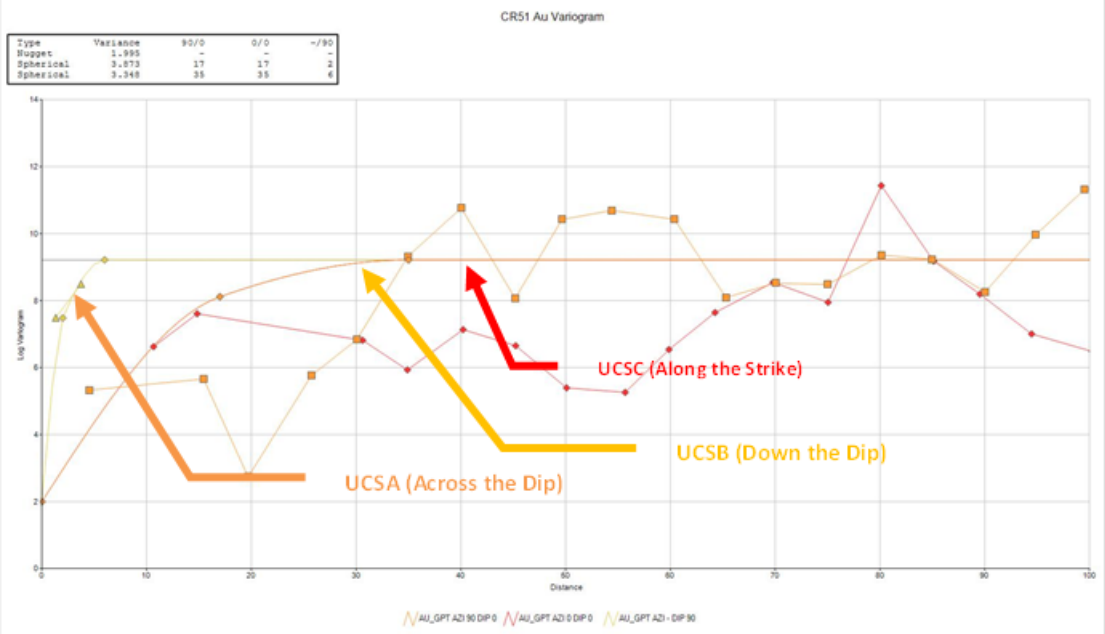

Figure 14.19 Experimental variograms and fitted models for Au – Cape Ray 51

Table 14.18 Log Variogram Model Parameters for the Cape Ray 51 Deposit

Variogram Models – Cape Ray 51	AU_GPT
Nugget	10.06
1st spherical structure range A	2
1st spherical structure range B	17
1st spherical structure range C	17
1st spherical structure sill	19.53
2nd spherical structure range A	6
2nd spherical structure range B	35
2nd spherical structure range C	35
2nd spherical structure sill	16.89
Total sill	**46.48**

A block size of 3 metres by 3 metres by 3 metres was chosen as this meets the above criteria.

A custom script was used to create the empty prototype model and then fill it with blocks using the mineral envelope for the domain wire frame. The resultant empty model was then regularized creating FILLVOL and VOIDVOL fields containing the volume for each block inside or outside the mineral domain wire frame.

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Table 14.19: Summary Statistics for Samples and Models – Cape Ray 51 Deposit

FILE	FIELD	NRECORDS	NSAMPLES	MINIMUM	MAXIMUM	MEAN	% DIFF	VARIANCE	SKEWNESS	WGTFIELD
cr51_lg_data1u	AU_GPT	613	613	0	40	3.84		48.943	2.669	LENGTH
nn_cr51_lg_mod	AU_GPT	35080	35080	0	40	3.47		57.403	3.373	FILLVOL
ik_cr51_lg_cor	AU_GPT	35080	35080	0	67.09	3.68	6	19.585	2.847	FILLVOL
cr51_lg_data1u	AG_GPT	613	613	0	80	10.04		330.943	2.434	LENGTH
nn_cr51_lg_mod	AG_GPT	35080	35080	0	80	8.9		297.563	2.79	FILLVOL
ik_cr51_lg_cor	AG_GPT	35080	35080	0.02	113.71	9.25	3.96	81.181	2.33	FILLVOL
cr51_lg_data1u	CU%	613	613	0	0.6	0.05		0.015	3.02	LENGTH
nn_cr51_lg_mod	CU%	35080	35080	0	0.6	0.04		0.01	3.558	FILLVOL
ik_cr51_lg_cor	CU%	35080	35080	0	0.99	0.03	-8.3	0.003	4.17	FILLVOL
cr51_lg_data1u	PB%	613	613	0	2	0.17		0.172	3.039	LENGTH
nn_cr51_lg_mod	PB%	35080	35080	0	2	0.11		0.119	3.76	FILLVOL
ik_cr51_lg_cor	PB%	35080	35080	0	2.62	0.1	-13.26	0.031	4.21	FILLVOL
cr51_lg_data1u	ZN%	613	613	0	1.7	0.08		0.057	4.186	LENGTH
nn_cr51_lg_mod	ZN%	35080	35080	0	1.7	0.06		0.046	4.563	FILLVOL
ik_cr51_lg_cor	ZN%	35080	35080	0	1.82	0.05	-19.06	0.01	4.943	FILLVOL

14.4.1.9 Nearest Neighbour Block Model

A Nearest Neighbour (NN) estimated model (nn_cr51_lg_mod.dm) was created for the mineral domain in order to determine the de-clustered mean for the data. This mean can then be used to validate the kriged global estimate as all methods of estimation should produce essentially the same global mean, if done correctly. The de-clustered mean is also used in assessing smoothing and, if necessary, calculating a variance correction of the kriged model.

Summary statistics comparing the nearest neighbour model to the sample file are presented in Table 14.19. Note how the mean values for the NN and IK models for most of the elements estimated are close to or are within 5% of one another for Au and Ag confirming that the mean has been reproduced. Cu, Pb and Zn all means more than 5% apart but this is due to a scale effect in that the absolute values are small yet the percentage difference is large.

A visual inspection on a section-by-section and plan-by-plan basis comparing the input sample file with the resultant nearest neighbour file showed reasonable correlation with the drill holes and proper spreading of the grade.

14.4.1.10 Indicator Kriging Block Models

As noted previously, the grade distributions are highly skewed for the Cape Ray 51 deposit. These distributions often result in spatial estimation bias where a few outliers representing the high-grade tail of the distribution are difficult to characterize resulting in a significant impact on both the mean and variance for the grade population even though they only represent a small fraction of it. Since traditional linear estimation methods, including Ordinary Kriging, often

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misrepresent these outliers resulting in a smoothing of the overall variability and therefore in over-estimation, Indicator Kriging was used to model the Cape Ray 51 deposit. An Indicator approach has proven successful in addressing the problem of highly skewed populations

As discussed in 14.1.5, the indicator approach involves coding data into binary groupings using a series of grade cut-offs. The metal grade can be estimated by multiplying the mean of the binary grade cut-off groupings, by its probability, which is determined by Indicator Kriging, before summing them. Using this indicator approach, the metal estimated was interpolated for all of the 3 m x 3 m x 3 m cells contained within the block model for the Cape Ray 51 mineral domain.

After the parameter files needed for Indicator Kriging were constructed, a custom script was used carry out the Multiple Indicator Kriging process. Each cell in the block model was discretised using a matrix of 3 x 3 x 3 points in the ABC (unfolded) coordinate system. The Kriging functions were interpolated at each discretisation point using the same search volume as the nearest neighbour interpolation, based on the grade variogram results. In case of local low sample density a nested search was implemented.

14.4.1.10.1 Definition of Indicator Functions

The grade data for Cape Ray 51 are all highly skewed. To deal with such highly skewed frequency distributions of grade an indicator approach has been applied for grade estimation. The 1 m composites were coded according to the cut-offs determined from the de-clusterised cumulative statistics for all 5 grade fields using a custom script. The indicators used are listed in Table 14.20

Initially each cut-off corresponds to the grade of the upper limit of each portion of the sample data representing approximately 9.1% of the total metal. As the low grade portions of the cumulative distribution functions are poorly sampled using this approach additional indicators were added to ensure proper sampling of the full distribution.

14.4.1.10.2 Indicator Variography

The indicator variograms for the Cape Ray 51 mineral domain were modelled using CAE Studio 3 variogram modeling tools.

One advantage of Indicator variograms is that they are often less erratic and easier to model than the grade variograms. This is because they do not suffer from having large differences between values.

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Table 14.20 Cape Ray 51 Deposit - Indicator Cut-offs Used

Indicator	AU_GPT	AG_GPT	CU%	PB%	ZN%
1	0.326	2.14	0.01	0.02	0.01
2	1.03	6.86	0.059	0.215	0.09
3	1.718	8.23	0.1	0.343	0.18
4	4.368	12.892	0.13	0.633	0.28
5	6.892	17.83	0.15	0.828	0.48
6	10.944	22.71	0.202	0.922	0.561
7	13.03	32.207	0.235	0.99	0.71
8	18.86	45.714	0.276	1.16	0.799
9	25.83	58.058	0.362	1.393	1.07
10	28.78	78.025	0.495	1.5	1.418
11	33.024				
12	36.425				

14.4.1.10.3 Indicator Kriging

A custom script was used to create the parameter files needed for Indicator Kriging. This was then followed by the use of another custom script to actually carry out the Indicator Kriging process. Each cell in the block model was discretised using a matrix of 3 x 3 x 3 points. The indicator functions were interpolated at each discretisation point using the same search volume as the nearest neighbour interpolation, based on the grade variogram results. In case of local low sample density a nested search was implemented.

For the model 7.5% of the blocks were estimated in the first search, 52.1% in the second and 40.4% in the third.

14.4.1.10.4 Investigation of Kriging Results

In theory estimated indicator functions should range between 0 and 1, but as mentioned in Section 14.1.6, Ordinary Kriging can create negative weights that may cause the interpolated values to be outside of this range. The Cape Ray 51 deposit had no significant overlap issues caused by negative weights; all of the negative indicator values were set to 0.00 and all values over 1 were set to 1 using a custom script during post processing.

Results were also checked to ensure that indicator values decrease as the cut-off increases. Although order relations issues (probability increases as the grade cut-off increases) can be expected when different variograms are used for the indicator functions, they should not be more than approximately 1% of the estimates. In the case of Cape Ray 51 data there are no significant cumulative proportions issues.

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14.4.1.10.5 Volume Variance Correction

For Cape Ray 51 the smoothing ratio for gold (9.98) and for silver (11.1) shows that there is smoothing and as a result a correction was required.

The variance of the model for both elements has been increased and therefore more in line with the expected variance with respect to block size. As should be expected, the mean remains essentially unchanged by the transformation.

As they are relatively minor elements with very low average (mean) values it was determined that no corrections would be made for Cu, Pb or Zn.

14.4.1.11 Model Verification

Validation procedures were carried out on the estimated block models including visually checking the sample file against estimated blocks. The sample grades were found to reasonably match the estimated block grades in the model.

A global statistical comparison of the global means of all estimations method was done. The difference between all the global means was found in most cases not to exceed approximately 5%, to be expected if the process was done correctly (see Table 14.19). Where differences do exceed this threshold it is easily explainable as a scale issue.

Other statistical checks that were done include the use of Swath plots (see Appendix C). Swath plots compare the moving average of the mean for both NN and IK models and the sample file using panels, or “swaths” through the mineral envelope. As this is best done when the data is within a rectilinear volume the unfolded coordinates were used to define the swaths. The result is a curve for each data set. The curves for the models should inter-weave with the sample curve and the two model curves should be sympathetic with one another with no major deviations from one another. No issues were noted.

14.5 Cape Ray Windowglass Hill Deposit

14.5.1 Resource Estimation Methodology

14.5.1.1 Data Validation

Core-drilling data were supplied as Microsoft Excel tables that included collar information, assays, lithology information, down hole survey information and specific gravity data. Sibley Basin Group compiled the data into a Microsoft Access database and completed data validation. Once validated this information was imported into CAE Studio 3 as four tables: a collar file, an assay file, a lithology file, and a survey file. As the data supplied were in UTM coordinates a transformation was completed to make a copy of the drill hole file in local grid coordinates based on the collar location of hole PB80-156. The collar for this hole has a World Coordinate System (WCS) location of 356005.19E, 5290825.26N with a Local Coordinate System (LCS)

location of 334.61W, 136.37S. Using the CAE Studio 3 HOLES3D routine a desurveyed drill hole file, cr_wg_holes_lg.dm, was created. This drill hole file was last updated on July 12, 2014.

The CAE Studio 3 desurveying routine, HOLES3D, does a rigorous set of validation checks including checking for duplicate borehole numbers, missing survey data and overlapping sample intervals. If present, it generates a summary report with a list of all errors encountered. This file was checked to determine if any errors occurred and where appropriate corrections were made. Once it had been confirmed that no errors remained the drill hole file was then used for subsequent steps.

There are no density or specific gravity values in the data set provided, but a previous resource estimate for the project area (Ténière & Hilchey, 2012) had specific gravity (SG) values determined on 44 samples. From these an average value was calculated from 6 samples that fell below the 25th percentile and used as a global value. This same value of 2.84 g/cm3 has been used for this project.

14.5.1.2 Geological Domains

Experienced geologists had coded each rock unit based on core logging description. All of the holes are inclined and most intersected at least some portion of the deformation zone that hosts the gold mineralisation. Construction of the resource block model was controlled by building two wire frames, each representing a different mineralisation domain and these were then used to isolate related samples. No cut-off was used to limit the extent of this mineral envelope. The envelopes for the two mineral domains (see Figure 14.20) extend from an elevation of about 354 metres above sea level down to a maximum depth of approximately 175 metres above sea level. These are not geological envelopes as the deformation zone hosting the mineralisation cuts stratigraphy.

A total of 33 holes have penetrated the mineral zones (see Figure 14.20 and Figure 14.23).

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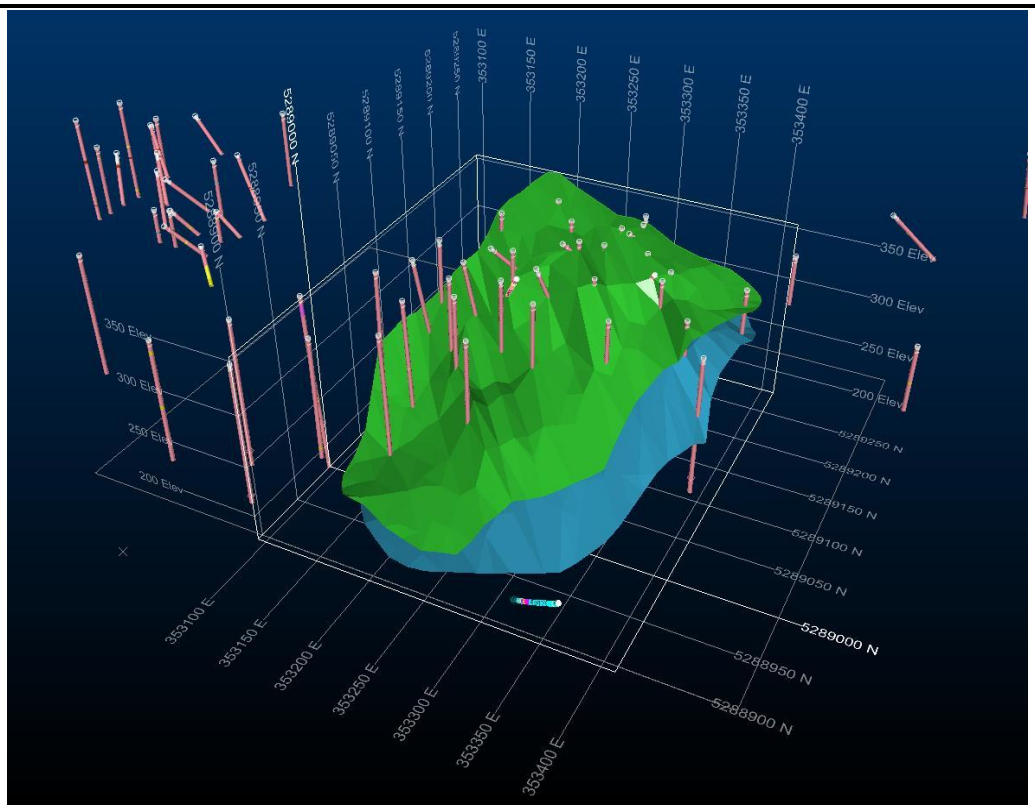

Figure 14.20 Isometric view of the Cape Ray Windowglass Hill geological domains used

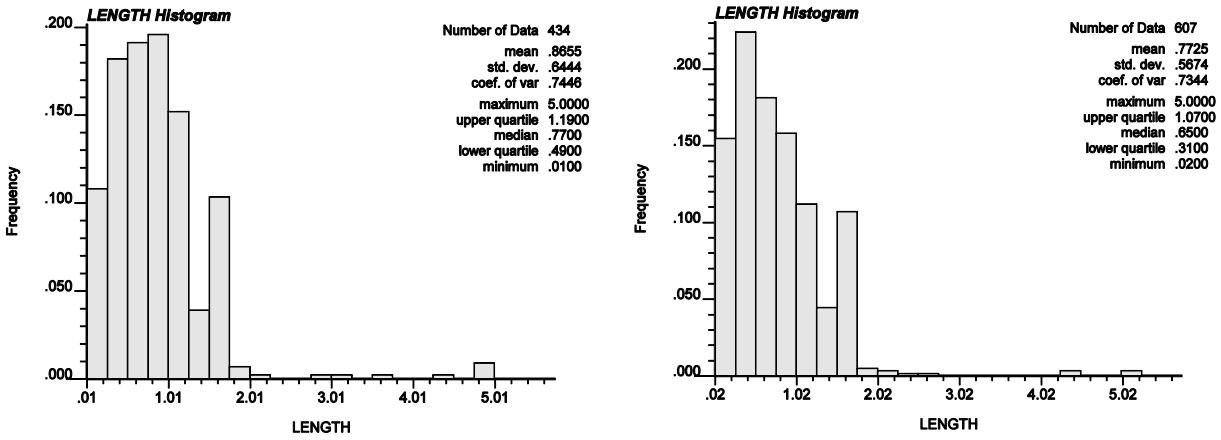

Figure 14.21 Histograms of sample length for the Cape Ray Windowglass Hill Deposit. A) Lower Domain and B) Upper Domain

14.5.1.3 Drill Hole Database

For the Windowglass Hill deposit the file used for resource estimation purposes, "*cr_wg_holes_lg.dm*", contains information for 75 drill holes totalling 6,308 metres and with 1,901 samples collected.

14.5.1.4 Sample Selection

Working in cross section sets of mineral zone lines, or strings, were defined for the two domains. These strings were drawn to enclose the Cape Ray Windowglass Hill Au-Ag zone by snapping to the drill holes. The strings were then used to construct a mineral envelope wire frame for the domain. The envelope extends from 354 metres above mean sea level down to about 175 metres above mean sea level. The borehole samples located within the mineral envelopes were captured using a custom script.

14.5.1.5 Compositing

For the Lower Domain the captured samples have an average sample length of 0.87 metres and for the Upper Domain, 0.77 metres (see Figure 14.21). To be consistent with the block size used for other deposits in the area 3 metres in all three dimensions was used. A 1 metre composite length represents a multiple of these blocks dimensions ensuring uniform support.

The samples for both domains were composited to standard 1 m intervals using the CAE Studio 3 process COMPDH. The COMPDH process starts the composites at the beginning of the selected data interval and leaves any remainder at the end of the interval. This results in most holes having one sample with a length less than the established composite length. For grade estimation purposes, drill composites are treated like point data (i.e. their length is not used), thus the need to composite to a standard sample length to eliminate any sample bias. To avoid bias from a very short sample being treated the same as a standard sample any sample lengths that were less than 40% of the composite length were rejected

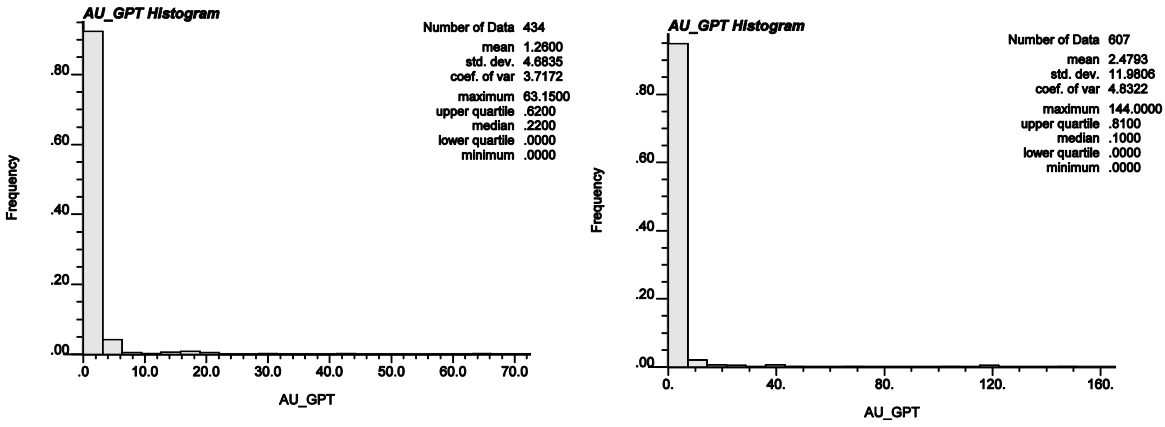

Figure 14.22 Histograms of Au for Cape Ray Windowglass Hill Deposit. A) Lower Domain and B) Upper Domain

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Table 14.21 Summary of Diamond Drilling for the Cape Ray Windowglass Hill Deposit

DDH	UTM East	UTM North	Elevation	Length	Grid East	Grid North	Azimuth	Dip
PB78-072	353270.9	5289222.2	346.3	31.4	3102.9	5254.7	0	-90.0
PB78-073	353266.9	5289192.3	343.2	30.5	3081.5	5233.4	0	-90.0
PB78-074	353282.0	5289251.3	346.9	30.5	3129.4	5271.1	0	-90.0
PB78-075	353279.8	5289217.7	345.0	30.5	3107.2	5245.7	76.35	-45.0
PB78-076	352876.6	5289154.6	356.3	61.0	2748.9	5441.1	0	-90.0
PB78-077	352921.2	5289098.2	338.6	30.5	2750.0	5369.2	86.35	-45.0
PB78-078	352920.4	5289098.2	338.6	30.5	2749.3	5369.7	0	-90.0
PB78-079	352887.5	5289074.8	342.9	33.5	2709.0	5371.2	57.35	-50.0
PB78-080	352886.8	5289074.2	342.9	36.6	2708.1	5371.2	0	-90.0
PB78-081	352900.2	5289050.4	341.7	45.7	2704.2	5344.0	57.35	-45.0
PB78-082	352855.9	5289108.3	352.7	48.8	2704.3	5416.9	57.35	-45.0
PB78-083	352942.3	5289047.2	333.8	36.6	2735.7	5315.9	131.35	-45.0
PB78-084	353233.4	5289178.4	342.9	38.7	3046.5	5242.8	54.35	-45.0
PB78-085	353177.7	5289149.7	338.9	39.6	2984.8	5253.9	54.35	-45.0
PB79-134	354392.6	5289686.3	217.9	175.3	4275.3	4940.1	322.35	-45.0
PB86-182	353323.7	5289167.5	341.1	83.2	3111.5	5179.1	322.35	-45.0
PB86-183	353219.9	5289115.9	337.1	92.4	2997.7	5201.5	322.35	-45.0
PB86-184	353397.2	5288820.2	304.5	153.3	2958.4	4858.9	322.35	-45.0
PB87-206	353245.0	5289187.9	343.6	78.3	3061.5	5243.2	0	-90.0
PB87-208	353305.7	5289197.2	341.9	47.9	3115.3	5213.7	0	-90.0
PB87-209	353397.8	5289189.7	334.3	46.3	3183.8	5151.7	0	-90.0
PB87-211	353367.8	5289129.3	329.9	47.9	3123.2	5122.0	0	-90.0
PB87-212	353280.5	5289140.4	340.1	52.1	3060.7	5184.0	0	-90.0
PB87-213	353200.5	5289240.8	348.3	66.1	3058.3	5312.3	0	-90.0
PB87-214	353333.6	5289179.7	340.1	66.1	3126.8	5182.8	0	-90.0
PB87-216	353539.3	5289204.5	311.8	63.1	3305.1	5077.2	0	-90.0
PB87-217	353418.0	5289263.9	334.1	47.9	3245.0	5198.2	0	-90.0
PB87-219	353332.7	5289161.8	340.8	47.9	3115.1	5169.2	0	-90.0
PB87-220	353285.4	5289237.2	346.0	57.0	3123.5	5257.8	0	-90.0
PB87-221	353197.5	5289153.5	339.7	111.9	3002.9	5244.9	0	-90.0
PB87-223	353185.1	5289069.8	335.4	305.1	2942.1	5186.0	0	-90.0
PB87-224	352877.2	5289065.3	345.8	32.6	2695.0	5369.9	0	-90.0
PB87-225	352788.1	5289181.2	371.4	63.1	2694.9	5516.1	0	-90.0
PB87-226	352768.6	5289116.3	363.4	69.2	2639.9	5476.5	0	-90.0
PB87-227	352885.1	5289257.5	366.3	78.3	2818.4	5517.6	0	-90.0
PB87-228	353012.6	5288896.8	343.8	115.8	2699.9	5153.8	0	-90.0
PB87-229	353226.4	5289212.3	347.3	130.2	3061.6	5273.9	0	-90.0
PB87-230	352889.1	5288958.7	355.4	118.0	2639.6	5278.1	0	-90.0

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DDH	UTM East	UTM North	Elevation	Length	Grid East	Grid North	Azimuth	Dip
PB87-231	353161.9	5289201.6	345.2	136.3	3003.9	5304.7	0	-90.0
PB87-232	353040.7	5288958.4	335.8	147.2	2759.7	5185.6	0	-90.0
PB87-233	353129.9	5289143.3	338.7	138.7	2943.0	5277.9	0	-90.0
PB87-234	353077.0	5288910.8	332.6	136.3	2759.5	5125.6	0	-90.0
PB87-235	353103.2	5289079.0	332.7	120.4	2882.7	5243.2	0	-90.0
PB87-236	353107.6	5288971.0	329.8	118.0	2820.4	5154.8	0	-90.0
PB87-237	353166.5	5289094.7	334.9	130.2	2942.4	5217.1	0	-90.0
PB87-238	353250.3	5289084.7	338.1	142.3	3002.9	5158.2	0	-90.0
PB87-239	353311.9	5289097.3	332.1	121.9	3059.4	5130.6	0	-90.0
PB87-240	353222.5	5289021.1	331.6	148.4	2942.1	5124.6	0	-90.0
PB87-241	353394.3	5289094.9	322.9	134.7	3123.4	5078.6	0	-90.0
WGH-04-01	352892.0	5289172.0	358.0	65.8	2771.8	5445.5	118.35	-57.0
WGH-04-02	352790.0	5289170.0	369.3	54.9	2689.6	5506.0	126.35	-45.0
WGH-04-03	352815.0	5289208.0	369.0	45.1	2732.6	5521.0	101.35	-45.0
WGH-04-04	352752.0	5288788.0	362.0	50.3	2426.9	5226.1	56.35	-45.0
WGH-04-05	352830.0	5288760.0	361.0	64.6	2471.8	5156.4	1.35	-45.0
WGH-04-06	352898.0	5288768.0	358.0	54.9	2530.6	5121.4	39.35	-45.0
WGH-04-07	352318.0	5288464.0	314.0	70.1	1885.4	5233.3	91.35	-45.0
WGH-04-08	352201.0	5288563.0	309.0	12.8	1852.8	5383.0	91.35	-45.0
WGH-04-09	352293.0	5288465.0	310.0	51.2	1866.2	5249.3	91.35	-45.0
WGH-04-10	353479.0	5289362.0	342.0	66.5	3353.1	5238.9	61.35	-45.0
WGH-04-11	353209.0	5289108.0	337.0	130.2	2984.3	5201.8	1.35	-90.0
WGH-04-12	353124.0	5289106.0	336.0	150.3	2915.6	5251.9	53.35	-80.0
WGH-04-13	352839.0	5289117.0	354.0	54.3	2696.2	5434.2	141.35	-70.0
WGH-04-14	352823.0	5289138.0	360.0	50.6	2696.3	5460.6	136.35	-55.0
WGH-04-15	353228.0	5289137.0	338.0	129.9	3017.0	5213.2	53.35	-70.0
WGH-04-16	352790.0	5289119.0	361.0	50.3	2658.6	5465.6	141.35	-50.0
WGH-06-17	352741.0	5289184.0	378.0	101.0	2659.3	5547.0	126.35	-88.0
WGH-06-18	352725.0	5289138.0	374.0	100.0	2618.6	5520.2	126.35	-80.0
WGH-06-19	352792.0	5289169.0	368.5	80.0	2690.6	5504.0	126.35	-88.0
WGH-06-20	353606.0	5289583.0	352.0	50.0	3588.4	5336.9	51.35	-80.0
WGH-06-21	353572.0	5289529.0	352.0	70.0	3528.6	5314.8	51.35	-80.0
WGH-06-22	353069.0	5289016.0	334.0	179.0	2817.2	5214.0	61.35	-88.0
WGH-06-23	353154.0	5289001.0	328.7	138.0	2875.5	5150.4	61.35	-88.0
WGH-06-24	353147.0	5289050.0	332.0	161.0	2899.8	5193.5	61.35	-88.0
WGH-06-25	353166.0	5289121.0	336.9	131.0	2958.1	5238.3	61.35	-80.0
WGH-06-26	353632.0	5289565.0	337.0	50.0	3598.1	5306.8	46.35	-80.0

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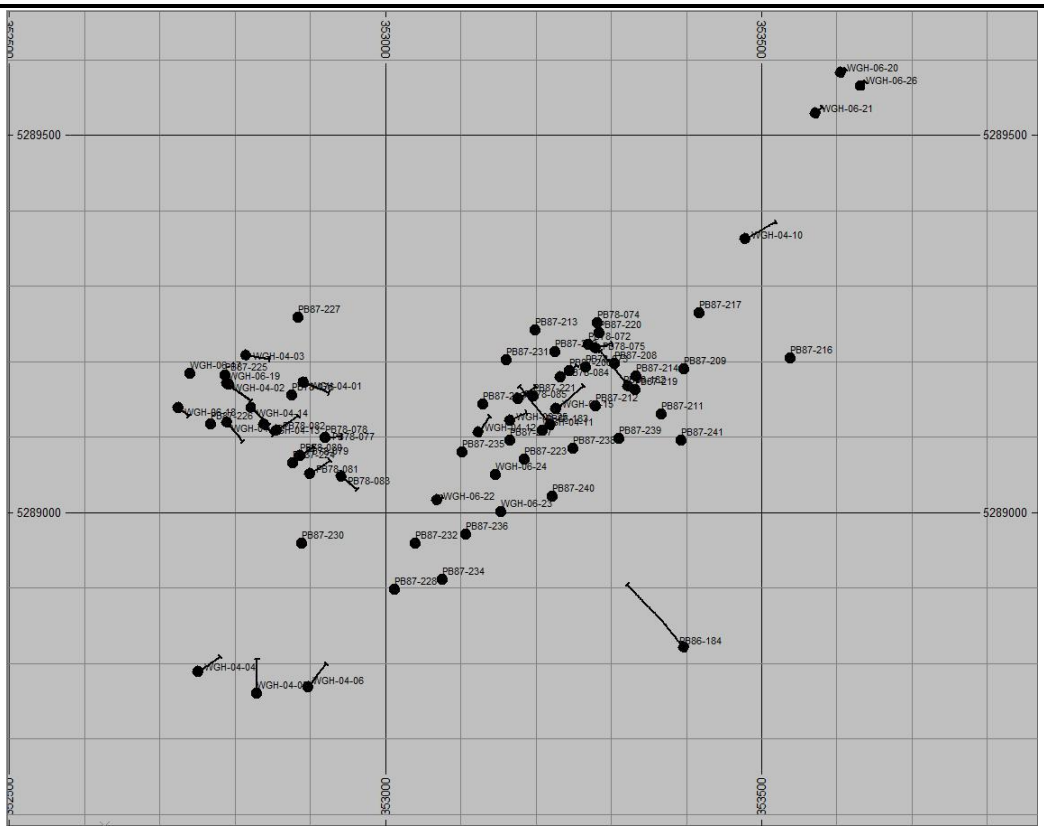

Figure 14.23 Plan view of Cape Ray Windowglass Hill Deposit drilling

Table 14.22 Summary Univariate Statistics for the Cape Ray Windowglass Hill Deposit

Upper Domain								
FIELD	**NSAMPLES**	**MINIMUM**	**MAXIMUM**	**MEAN**	**VARIANCE**	**STANDDEV**	**SKEWNESS**	**KURTOSIS**
AU_GPT	608	0	144.00	2.48	143.31	11.97	8.52	79.77
AG_GPT	608	0	688.40	14.00	3084.49	55.54	7.40	66.91
CU%	608	0	3.14	0.02	0.02	0.15	16.94	337.46
PB%	608	0	19.40	0.14	1.72	1.31	12.21	162.72
ZN%	608	0	1.64	0.01	0.01	0.08	15.96	288.35
Lower Domain								
FIELD	**NSAMPLES**	**MINIMUM**	**MAXIMUM**	**MEAN**	**VARIANCE**	**STANDDEV**	**SKEWNESS**	**KURTOSIS**
AU_GPT	435	0	63.15	1.26	21.89	4.68	8.41	88.77
AG_GPT	435	0	426.34	10.52	1370.63	37.02	6.26	48.68
CU%	435	0	0.00	0.00				
PB%	435	0	0.00	0.00				
ZN%	435	0	0.00	0.00				

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14.5.1.6 Exploratory Data Analysis

A review of the drill hole database was done using histograms for all primary elements and X/Y scatter plots of element pairs (see Appendix A). Features watched for are outliers and irregularities in the element statistics. Univariate summary statistics for all elements are presented in Table 14.22 and correlation matrices are presented in Table 14.23. For the latter a correlation coefficient of 0.7 or higher indicates a good linear relationship between the bivariate components. A positive coefficient indicates that with increasing concentration of one element there is a sympathetic increase in the other. A negative coefficient indicates that as one element increases the other decreases.

Unlike the other deposits in the area (04, 41 and 51), base metal sulphides are not typically associated with the gold and silver mineralisation and as a result only 42 samples have assays for Cu, Pb and Zn out of the total of 1043 collected. All of the base metal assays are only within the Upper Domain.

The scatter plots for the element pairs do indicate issues as Au and Ag for both the Upper and Lower domains, have significant outliers. Due to the number and magnitude of these outliers something needed to be done as the histograms have significant gaps. Using a visual inspection of the scatter plots trimming values were selected and these outliers trimmed to these new values. The trimming limits chosen are shown in Table 14.24

Both domains lack good correlation coefficients thus do not have a good linear correlation between the Au and Ag.

Table 14.23 Correlation Matrix for the Cape Ray Windowglass Hill Deposit

Upper Domain					
	AU_GPT	**AG_GPT**	**CU%**	**PB%**	**ZN%**
AU_GPT	1				
AG_GPT	0.4767	1			
CU%	0.3433	0.3151	1		
PB%	0.2473	0.5021	0.34	1	
ZN%	0.085	0.0497	0.2553	0.0832	1
Lower Domain					
	AU_GPT	**AG_GPT**	**CU%**	**PB%**	**ZN%**
AU_GPT	1				
AG_GPT	0.6269	1			
CU%	0	0	1		
PB%	0	0	0	1	
ZN%	0	0	0	0	1

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Table 14.24 Sample Trimming Statistics – Cape Ray Windowglass Hill Deposit

	Maximum Value	Trimming Value Used	#Samples Trimmed
Lower Domain			
AU_GPT	63.15	40	2
AG_GPT	426.34	160	7
Upper Domain			
AU_GPT	144	40	9
AG_GPT	688.4	160	13

Au, the primary element of interest, does have a few spurious values with a maximum value of 144 g/t for the Upper Domain and 63.16 g/t for the Lower Domain. The histograms for Au (Figure 14.22) are positively skewed distributions, typical of a lode gold deposit. Ag shows similar results with significant outliers.

Exploratory data analysis showed that, other than the issue of the outliers, there were no issues with the drill hole database that would invalidate their use for resource estimation purposes.

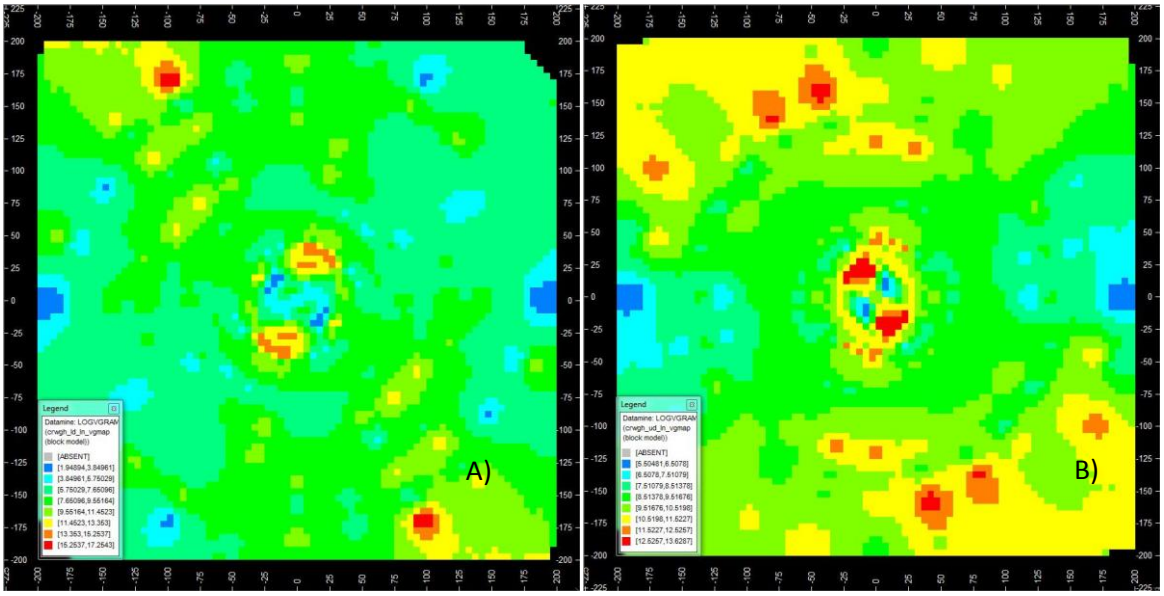

Figure 14.24 Variogram maps for the Cape Ray Windowglass Hill Au-Ag Deposit. A) Lower Domain and B) Upper Domain

14.5.1.7 Grade Variography

Prior to doing grade variography a custom script was used to prepare a variogram map for Au (see Figure 14.24) in order to check for a rotation in the primary direction of anisotropy. Directional lognormal variograms were calculated in 15° increments in the unfolded plane of the mineral zone. lognormal variograms provide a clearer picture of the spatial continuity as it reduces the influence of very large values on the calculation of the moment of inertia, common

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with highly positively skewed distributions (Isaaks and Srivastava, 1989).

The variogram maps do not show a rotation (plunge) around the UCSA (across dip) axis for either domains. They also show that the along strike direction (UCSC – Y axis) and down dip directions (UCSB – X axis) have similar ranges. The sill in both cases is reached at a range of approximately 50 metres.

The experimental log grade variograms for the two domains (see Figure 14.25 and Figure 14.26) were calculated for the unfolded composited data sets using a custom script.

Typically, with inclined drilling, the across dip and down dip directions of the variogram are usually well defined due to the abundant sampling of the distance spectrum. This is not the case for Windowglass Hill as the along strike direction (UCSC), and for that matter the down dip (UCSB) are not well defined due to the relatively wide spacing of the drill holes.

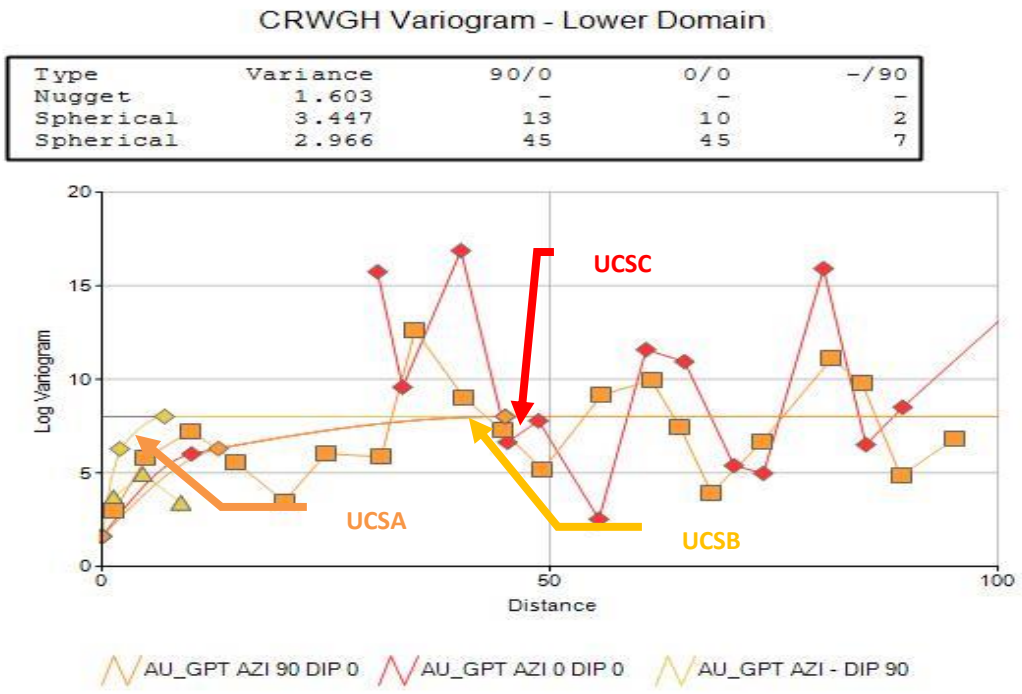

Figure 14.25 Experimental variograms and fitted models for Au – Cape Ray Windowglass Hill Deposit Lower Domain

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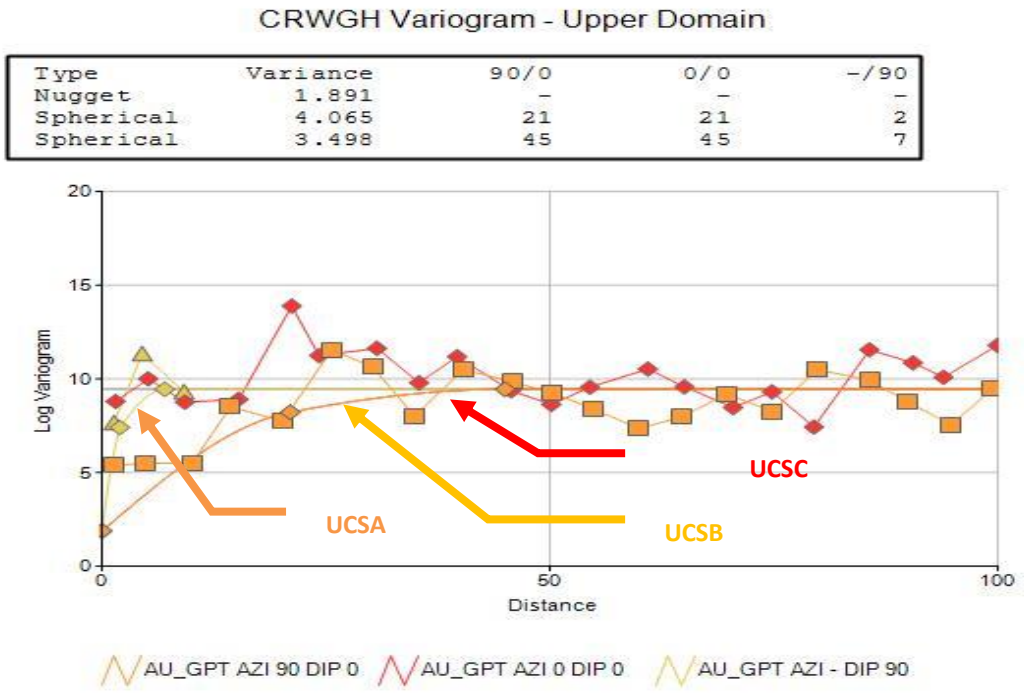

Figure 14.26 Experimental variograms and fitted models for Au – Cape Ray Windowglass Hill Deposit Upper Domain

In summary, there is a poor degree of confidence in the curves used to model all three directions.

The ranges derived from the log variogram models are shown in Table 14.25.

Table 14.25 Log Variogram Model Parameters for the Cape Ray Windowglass Hill Deposit

Variogram Models Cape Ray Windowglass Hill	Lower Domain AU_GPT	Upper Domain AU_GPT
Nugget	1.603	1.891
1st spherical structure range A	2	2
1st spherical structure range B	13	21
1st spherical structure range C	10	21
1st spherical structure sill	3.447	4.065
2nd spherical structure range A	7	7
2nd spherical structure range B	21	45
2nd spherical structure range C	21	55
2nd spherical structure sill	2.966	3.498
Total sill	**8.016**	**9.454**

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14.5.1.8 Block Size Determination

For the Cape Ray Windowglass Hill deposit, due to the geometry and based on recommendations from mine engineering affiliated with this project, a block size suitable for evaluating underground mining methods as well as open pit mining was selected.

A block size of 3 metres by 3 metres by 3 metres was chosen as this meets the above criteria.

A custom script was used to create the empty prototype models and then fill them with blocks using the mineral envelopes for the domain wire frames. The resultant empty models were then regularized creating FILLVOL and VOIDVOL fields containing the volume for each block inside or outside the mineral domain wire frame.

14.5.1.9 Nearest Neighbour Block Model

Nearest Neighbour (NN) estimated models (nn_crwg_ld.dm and nn_crwg_ud.dm) were created for the two mineral domains in order to determine the de-clustered mean for the data. This mean can then be used to validate the kriged global estimate as all methods of estimation should produce essentially the same global mean, if done correctly. The de-clustered mean is also used in assessing smoothing and, if necessary, calculating a variance correction of the kriged model.

Summary statistics comparing the nearest neighbour model to the sample file are presented in Table 14.26. Note how the mean values for the NN and IK models for Au and Ag are typically greater than 5% of one another but this is primarily due to a scale issue; the absolute difference is small but when viewed as a ratio results in values outside our usually acceptable threshold.

A visual inspection on a section-by-section and plan-by-plan basis comparing the input sample file with the resultant nearest neighbour file showed good correlation with the drill holes and proper spreading of the grade.

14.5.1.10 Inverse Distance Block Model

As noted previously, the grade distributions are highly skewed. Such distributions often result in spatial estimation bias where a few outliers representing the high-grade tail of the distribution are difficult to characterize. These have a significant impact on both the mean and variance for the grade population even though they only represent a small fraction of it. While Indicator Kriging is the preferred method for handling these types of distributions in this case the sample density is too small making modeling of each indicator very difficult. In addition, as so many samples have zero grade since they were not assayed, a large number of indicators would be required to properly sample the distribution. As a result it was decided that the use of Inverse Distance is appropriate as only Inferred Resources can be estimated with this poorly reprehensive data set.

An inverse distance approach involves assigning weights based solely on, as the name implies, the inverse of the distance to the sample from the point being estimated. To reduce the effects

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of geometric bias an octant search method was applied.

The parameter files needed for Inverse Distance were constructed. This was then followed by the using of a custom script to actually carry out the Inverse Distance estimation process. In case of local low sample density a nested search was implemented.

Table 14.26 Summary Statistics for Samples and Models – Cape Ray Windowglass Hill Deposit

Lower Domain										
FILE	**FIELD**	**NRECORDS**	**NSAMPLES**	**MINIMUM**	**MAXIMUM**	**MEAN**	**% DIFF**	**VARIANCE**	**SKEWNESS**	**WGTFIELD**
crwg_ld_data1u	AU_GPT	384	384	0	18.13	0.63		2.813	6.449	LENGTH
nn_crwg_ld	AU_GPT	49001	49001	0	18.13	0.54		1.958	6.611	FILLVOL
id_crwg_ld	AU_GPT	49001	49001	0	5.89	0.63	16.29	0.318	2.527	FILLVOL
crwg_ld_data1u	AG_GPT	384	384	0	114.18	4.7		147.01	5.457	LENGTH
nn_crwg_ld	AG_GPT	49001	49001	0	114.18	4.66		132.872	5.341	FILLVOL
id_crwg_ld	AG_GPT	49001	49001	0.01	44.93	5.45	17.04	19.675	1.777	FILLVOL
Upper Domain										
FILE	**FIELD**	**NRECORDS**	**NSAMPLES**	**MINIMUM**	**MAXIMUM**	**MEAN**	**% DIFF**	**VARIANCE**	**SKEWNESS**	**WGTFIELD**
crwg_ud_data1u	AU_GPT	472	472	0	17.1	0.77		3.694	5.015	LENGTH
nn_crwg_ud	AU_GPT	41535	41535	0	17.1	0.6		2.525	6.348	FILLVOL
id_crwg_ud	AU_GPT	41535	41535	0	5.23	0.65	9.84	0.293	1.995	FILLVOL
crwg_ud_data1u	AG_GPT	472	472	0	127.06	5.4		186.856	5.176	LENGTH
nn_crwg_ud	AG_GPT	41535	41535	0	127.06	6.54		302.821	4.801	FILLVOL
id_crwg_ud	AG_GPT	41535	41535	0	69.12	6.79	3.85	45.971	2.809	FILLVOL

14.5.1.10.1 Volume Variance Correction

For Windowglass Hill's Lower Domain the smoothing ratio for gold is 2.79 and for silver 3.03. The ratios show that there is some smoothing and as a result a correction was required. Similarly for the Upper Domain the smoothing ratio for gold is 4.01 and for silver 3.07.

The variance of the model for both elements has been increased and therefore more in line with the expected variance with respect to block size. As should be expected, the mean remains essentially unchanged by the transformation.

14.5.1.10.2 Model Verification

Validation procedures were carried out on the estimated block models including visually checking the sample file against estimated blocks. The sample grades were found to reasonably match the estimated block grades in the model.

A global statistical comparison of the global means of all estimations method was done. The difference between all the global means was found in most cases to exceed approximately 5%,

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to be expected if the process was done correctly (see Table 14.26). This is easily explainable as due to a scale issue since the absolute difference (i.e. 0.09 g/t for Au in the Lower Domain) is not significant.

Other statistical checks that were done include the use of swath plots (see Appendix C). Swath plots compare the moving average of the mean for both models and the sample file using panels, or "swaths" through the mineral envelope. As this is best done when the data is within a rectilinear volume the unfolded coordinates were used to define the swaths. The result is a curve for each data set. The curves for the models should inter-weave with the sample curve and the two model curves should be sympathetic with one another with no major deviations from one another. No issues were noted.

14.6 Mineral Resource Reporting

14.6.1 Resource Classification

Classification of resources is all about confidence in the estimate. As the variograms are not well defined in all three primary directions for the Cape Ray deposits we therefore have a low to moderate confidence in the Kriging equations.

The next factor that needs to be addressed is the quantity and spatial location of the data actually used in the estimation process. To assist in this a nested approach was used whereby the first search utilised a very rigorous set of criteria, any blocks not estimated would then be evaluated by the second search that used somewhat less rigorous criteria and blocks remaining that were not estimated would utilise the third search that used very loose criteria just to ensure that all remaining blocks were estimated. As a result resource classification can be assigned based on which search a block was estimated with. Thus, if estimated during the first search as it has the most rigorous criteria and therefore the highest confidence in the estimate, then these blocks could be classified as Measured Resources. And if estimated during the second search which uses less rigorous criteria for selecting samples then they could be classified as Indicated Resources as it has moderate confidence in the estimate. Those blocks estimated during the third search use the least rigorous criteria and therefore have low confidence in the estimate and would be classified as Inferred Resources.

An octant search (the search ellipsoid is divided into 8 equal segments based on the primary axis planes) was utilised. It is used to reduce spatial bias by ensuring samples are selected all around the point being estimated. The minimum number of octants was set to 5 for the first two searches. But blocks on the edge of the mineral domain would automatically fail to be estimated during the first and second searches even though all other parameters, including minimum number of samples were met. To overcome this issue wireframe surfaces were manually constructed that isolated areas of high over all confidence from areas of moderate confidence from areas of low confidence (measured, indicated and inferred). The blocks were then recoded based on what confidence domain they are within and all subsequent analysis based on these codes.

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As a general rule of thumb a minimum value of approximately $100 per tonne is usually needed for a deposit to be considered potentially economic for mining using underground methods. If economies of scale can be attained through the application of open pit mining methods the price threshold can be considerably lower.

As it is still very early in the development stage and as mining and processing studies have not yet been finalised it is inappropriate to apply any sort of "mine design" as such would imply that any contained resources can be considered "reserves" and this is not correct and is very misleading. As a result the mineral resources reported here are based only on blocks above the cut-off grade. The only constraints used are the application of a series of grade cut-offs and the resource classification based on confidence in the estimate assigned to the blocks.

In all cases the block modeling was only constrained by the definition of the respective mineral envelopes, none of which extend any further than approximately 50 metres past the available drilling. Mineral resources are reported unconstrained by any pit shell for surface mining methods or any preliminary underground mining outline for underground methods. See Appendix E for resource classification definitions.

14.6.1.1 Determination of Cut-off Grade

While a series of cut-off grades can be used to generate tonnage and grade curves, which demonstrate the sensitivity to grade, a decision needs to be made on selecting one cut-off grade for reporting purposes to demonstrate reasonable prospects for eventual economic extraction. An accepted method is to use a cut-off grade that is currently being used for a similar deposit in a similar location or that has been used for similar style of mineralisation. It is the qualified person's opinion that the cut-off grade for reporting mineral resources does not need to reflect any specific assumed metal price, metallurgical recovery, mining and processing costs or general and administrative costs.

In the case of the Cape Ray deposits, two, 51 and Windowglass Hill, have been previously reported by Ténière and Hilchey (2012) and they used 2.5 grams per tonne as a cut-off for reporting mineral resources potentially minable using underground mining methods and 0.5 grams per tonne for reporting mineral resource potentially minable using open pit mining methods. In both cases they used a gold price of US$1400 per ounce, a CAD$105 exchange rate and a gold recovery of 95%.

Brown (2012), in a non-publicly released report that included a cut-off analysis, used a price of US$1600 per ounce of Au, 90% Au recovery and a number of assumed mining related costs. The cut-offs derived were 1.5 grams per tonne for mineral resources potentially minable using underground mining methods and 0.65 grams per tonne for mineral resources potentially minable using open pit methods.

Based on consultation with Nordmin mining engineers it was felt that using a cut-off grade of 4.0 grams per tonne for reporting mineral resources that are potentially minable using underground mining methods should be used. For appraising mineral resources that are potentially minable using open pit methods a cut-off grade of 1.0 grams per tonne has been recommended.

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14.6.1.2 Cape Ray 04 Deposit

The drilling density is such that reasonable variogram models could be defined for the lower grade indicators, but this is not the case for the higher value indicators that have ranges less than the average drill hole spacing of 25 metres. Even though 26.6% of the blocks were estimated in the first search, which normally could be considered measured resources, the reduced confidence in the upper indicator models requires that these be downgraded to Indicated. In addition, blocks estimated during the third search that are on the edges of the domain actually had enough data available that these blocks should have been estimated in the first or second search. A sectional review was made of the model and based on these two criteria a wireframe surface was constructed to isolate those blocks that should be considered Indicated resources and those that should be Inferred resources. The model was then evaluated using this wireframe surface and the blocks classified based on whether they are inside (Indicated) or outside (Inferred).

Resources that potentially could be mined by underground methods and using a 4.0 grams per tonne (g/t) cut-off, consist of a total of 0.46 million tonnes at a grade of 6.97 g/t Au, 16.93 g/t Ag, 0.11% Cu, 0.59% Pb and 0.54% Zn of Indicated Resources and 0.08 million tonnes at a grade of 6.41 g/t Au, 12.11 g/t Ag, 0.03% Cu, 0.24% Pb and 0.16% Zn of Inferred Resources.

Resources that potentially could be mined by open pit methods and using a 1.0 grams per tonne (g/t) cut-off, consist of a total of 1.58 million tonnes at a grade of 3.45 g/t Au, 11.07 g/t Ag, 0.06% Cu, 0.34% Pb and 0.31% Zn of Indicated Resources and 1.10 million tonnes at a grade of 2.29 g/t Au, 4.00 g/t Ag, 0.07% Cu, 0.42% Pb and 0.35% Zn of Inferred Resources.

All of the resources should be upgradable through gravity separation and possibly flotation. These resources are blocks above cut-off and have had no mineability criteria applied to them.

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Table 14.27 Classification of In-Situ Resources for the Cape Ray 04 Deposit, using a range of Au cut-offs

Cut-off	Classification	TONNES	AU_GPT	AG_GPT	CU%	PB%	ZN%
0.5	Measured Resources						
0.5	Indicated Resources	2.23	2.66	9.57	0.06	0.28	0.25
0.5	Meas. & Ind. Resources	2.23	2.66	9.57	0.06	0.28	0.25
0.5	Inferred Resources	1.40	1.96	3.71	0.03	0.21	0.14
1.0	Measured Resources						
1.0	Indicated Resources	1.58	3.45	11.07	0.06	0.34	0.31
1.0	Meas. & Ind. Resources	1.58	3.45	11.07	0.06	0.34	0.31
1.0	Inferred Resources	1.10	2.29	4.00	0.03	0.24	0.16
1.5	Measured Resources						
1.5	Indicated Resources	1.22	4.11	12.12	0.07	0.39	0.35
1.5	Meas. & Ind. Resources	1.22	4.11	12.12	0.07	0.39	0.35
1.5	Inferred Resources	0.76	2.74	4.61	0.03	0.29	0.19
2.0	Measured Resources						
2.0	Indicated Resources	0.93	4.85	13.71	0.08	0.45	0.41
2.0	Meas. & Ind. Resources	0.93	4.85	13.71	0.08	0.45	0.41
2.0	Inferred Resources	0.49	3.30	5.47	0.03	0.33	0.22
2.5	Measured Resources						
2.5	Indicated Resources	0.73	5.58	15.03	0.09	0.50	0.46
2.5	Meas. & Ind. Resources	0.73	5.58	15.03	0.09	0.50	0.46
2.5	Inferred Resources	0.30	3.94	6.78	0.04	0.37	0.26
3.0	Measured Resources						
3.0	Indicated Resources	0.61	6.15	15.80	0.10	0.54	0.49
3.0	Meas. & Ind. Resources	0.61	6.15	15.80	0.10	0.54	0.49
3.0	Inferred Resources	0.17	4.90	8.98	0.05	0.40	0.31
3.5	Measured Resources						
3.5	Indicated Resources	0.53	6.58	16.34	0.10	0.57	0.52
3.5	Meas. & Ind. Resources	0.53	6.58	16.34	0.10	0.57	0.52
3.5	Inferred Resources	0.12	5.69	10.70	0.06	0.41	0.32
4.0	Measured Resources						
4.0	Indicated Resources	0.46	6.97	16.93	0.11	0.59	0.54
4.0	Meas. & Ind. Resources	0.46	6.97	16.93	0.11	0.59	0.54
4.0	Inferred Resources	0.08	6.41	12.11	0.07	0.42	0.35
4.5	Measured Resources						
4.5	Indicated Resources	0.40	7.43	17.51	0.11	0.61	0.57
4.5	Meas. & Ind. Resources	0.40	7.43	17.51	0.11	0.61	0.57
4.5	Inferred Resources	0.06	7.13	13.59	0.08	0.44	0.38

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There is moderate confidence in the lateral continuity of the mineralisation and so those resources classified as Indicated and/or Measured resources can be used for a pre-feasibility or feasibility mining study.

Table 14.27 presents tonnes and grade for each Resource Classification using a range Au cut-offs for the Cape Ray 04 deposit.

Figure 14.27 presents the Au tonnes-grade curves for the Cape Ray 04 Au-Ag Deposit and helps illustrate the effect of different cut-offs on available resources. The mining and processing methods chosen will determine what proportion can be converted to reserves as these do not take into consideration mineability and dilution.

While resource estimation has been done for Au, Ag, Cu, Pb and Zn the poly-metallic nature of the mineralisation may require that a Net Smelter Returns (NSR) evaluation be done to properly evaluate the economics of recovering any of the identified resources.

Table 14.28 Classification of In-Situ Resources for the Cape Ray 41 Deposit, using a range of Au cut-offs

Cut-off	Classification	TONNES	AU_GPT	AG_GPT	CU%	PB%	ZN%
0.5	Measured Resources						
0.5	Indicated Resources	1.00	2.00	6.88	0.06	0.19	0.12
0.5	Meas. & Ind. Resources	1.00	2.00	6.88	0.06	0.19	0.12
0.5	Inferred Resources	0.27	1.92	6.02	0.06	0.19	0.13
1.0	Measured Resources						
1.0	Indicated Resources	0.60	2.85	9.58	0.08	0.27	0.16
1.0	Meas. & Ind. Resources	0.60	2.85	9.58	0.08	0.27	0.16
1.0	Inferred Resources	0.13	3.18	9.49	0.07	0.32	0.22
1.5	Measured Resources						
1.5	Indicated Resources	0.40	3.68	12.20	0.10	0.34	0.20
1.5	Meas. & Ind. Resources	0.40	3.68	12.20	0.10	0.34	0.20
1.5	Inferred Resources	0.10	3.81	11.02	0.09	0.38	0.26
2.0	Measured Resources						
2.0	Indicated Resources	0.30	4.29	13.98	0.12	0.39	0.23
2.0	Meas. & Ind. Resources	0.30	4.29	13.98	0.12	0.39	0.23
2.0	Inferred Resources	0.08	4.35	12.07	0.10	0.43	0.29
2.5	Measured Resources						
2.5	Indicated Resources	0.24	4.84	15.50	0.13	0.44	0.25
2.5	Meas. & Ind. Resources	0.24	4.84	15.50	0.13	0.44	0.25
2.5	Inferred Resources	0.07	4.74	12.89	0.11	0.47	0.31
3.0	Measured Resources						
3.0	Indicated Resources	0.19	5.35	17.05	0.14	0.48	0.28
3.0	Meas. & Ind. Resources	0.19	5.35	17.05	0.14	0.48	0.28

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Cut-off	Classification	TONNES	AU_GPT	AG_GPT	CU%	PB%	ZN%
3.0	Inferred Resources	0.06	5.14	13.70	0.12	0.51	0.34
3.5	Measured Resources						
3.5	Indicated Resources	0.16	5.84	18.52	0.16	0.52	0.30
3.5	Meas. & Ind. Resources	0.16	5.84	18.52	0.16	0.52	0.30
3.5	Inferred Resources	0.05	5.53	14.46	0.13	0.54	0.36
4.0	Measured Resources						
4.0	Indicated Resources	0.13	6.34	20.05	0.17	0.56	0.32
4.0	Meas. & Ind. Resources	0.13	6.34	20.05	0.17	0.56	0.32
4.0	Inferred Resources	0.04	5.83	15.02	0.13	0.56	0.38
4.5	Measured Resources						
4.5	Indicated Resources	0.10	6.91	21.80	0.19	0.61	0.35
4.5	Meas. & Ind. Resources	0.10	6.91	21.80	0.19	0.61	0.35
4.5	Inferred Resources	0.03	6.39	16.51	0.15	0.60	0.38

14.6.1.3 Cape Ray 41 Deposit

The drilling density is such that reasonable variogram models could be defined for the lower grade indicators, but this is not the case for the higher value indicators that have ranges less than the average drill hole spacing of 15 metres. Even though approximately a third of the blocks for both domains were estimated in the first search, which normally could be considered measured resources, the reduced confidence in the upper indicator models requires that these be downgraded to Indicated. In addition, blocks estimated during the third search that are on the edges of the domain actually had enough data available that these blocks should have been estimated in the first or second search. A sectional review was made of the model and based on these two criteria a wireframe surface was constructed to isolate those blocks that should be considered Indicated resources and those that should be Inferred resources. The model was then evaluated using this wireframe surface and the blocks classified based on whether they are inside (Indicated) or outside (Inferred). See Appendix E for resource classification definitions.

Resources that potentially could be mined by underground methods and using a 4.0 grams per tonne (g/t) cut-off, consist of a total of 0.13 million tonnes at a grade of 6.34 g/t Au, 20.05 g/t Ag, 0.17% Cu, 0.56% Pb and 0.32% Zn of Indicated Resources and 0.04 million tonnes at a grade of 5.83 g/t Au, 15.02 g/t Ag, 0.13% Cu, 0.56% Pb and 0.38% Zn of Inferred Resources.

Resources that potentially could be mined by open pit methods and using a 1.0 grams per tonne (g/t) cut-off, consist of a total of 0.60 million tonnes at a grade of 2.85 g/t Au, 9.58 g/t Ag, 0.08% Cu, 0.27% Pb and 0.16% Zn of Indicated Resources and 0.13 million tonnes at a grade of 3.18 g/t Au, 9.49 g/t Ag, 0.07% Cu, 0.32% Pb and 0.22% Zn of Inferred Resources.

All of the resources should be upgradable through gravity separation and possibly flotation. These resources are blocks above cut-off and have had no mineability criteria applied to them.

There is moderate confidence in the lateral continuity of the mineralisation and those resources classified as Indicated and/or Measured resources can be used for a pre-feasibility or feasibility mining study.

Table 14.28 presents tonnes and grade for each Resource Classification using a range Au cut-offs for the Cape Ray 41 deposit.

Figure 14.28 presents the Au tonnes-grade curves for the Cape Ray 41 Au-Ag Deposit and helps illustrate the effect of different cut-offs on available resources. The mining and processing methods chosen will determine what proportion can be converted to reserves as these do not take into consideration mineability and dilution.

While resource estimation has been done for Au, Ag, Cu, Pb and Zn the poly-metallic nature of the mineralisation may require that a Net Smelter Returns (NSR) evaluation be done to properly evaluate the economics of recovering any of the identified resources.

14.6.1.4 Cape Ray 51 Deposit

The drilling density is such that reasonable variogram models could be defined for the lower grade indicators, but this is not the case for the higher value indicators that have ranges less than the average drill hole spacing of 30 metres. Even though approximately 54% of the blocks were estimated in the first and second searches, which normally could be considered measured and indicated resources, the reduced confidence in the variograms, especially for the along strike direction, and many of the upper indicator models requires that these all be downgraded to Inferred. See Appendix E for resource classification definitions.

Resources that potentially could be mined by underground methods and using a 4.0 grams per tonne (g/t) cut-off, consist of a total of 0.40 million tonnes at a grade of 8.39 g/t Au, 16.34 g/t Ag, 0.04% Cu, 0.14% Pb and 0.07% Zn of Inferred Resources.

Resources that potentially could be mined by open pit methods and using a 1.0 grams per tonne (g/t) cut-off, consist of a total of 0.88 million tonnes at a grade of 5.08 g/t Au, 11.82 g/t Ag, 0.04% Cu, 0.12% Pb and 0.06% Zn of Inferred Resources.

All of the resources should be upgradable through gravity separation and possibly flotation. These resources are blocks above cut-off and have had no mineability criteria applied to them.

There is a low confidence in the lateral continuity of the mineralisation and so these resources cannot be used for a pre-feasibility or feasibility mining study.

Table 14.29 presents tonnes and grade for each Resource Classification using a range Au cut-offs for the Cape Ray 51 deposit.

Figure 14.27 presents the Au tonnes-grade curves for the Cape Ray 51 Au-Ag deposit and helps illustrate the effect of different cut-offs on available resources. At this time there is too low a confidence in these resources that none can be converted to reserves.

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While resource estimation has been done for Au, Ag, Cu, Pb and Zn the poly-metallic nature of the mineralisation may require that a Net Smelter Returns (NSR) evaluation be done to properly evaluate the economics of recovering any of the identified resources.

Table 14.29 Classification of In-Situ Resources for the Cape Ray 51 Deposit, using a range of Au cut-offs

`	Classification	TONNES	AU_GPT	AG_GPT	CU%	PB%	ZN%
0.5	Inferred Resources	1.01	4.52	10.82	0.04	0.11	0.06
1	Inferred Resources	0.88	5.08	11.82	0.04	0.12	0.06
1.5	Inferred Resources	0.78	5.57	12.64	0.04	0.12	0.06
2	Inferred Resources	0.69	6.07	13.42	0.04	0.12	0.06
2.5	Inferred Resources	0.61	6.61	14.13	0.04	0.12	0.06
3	Inferred Resources	0.53	7.16	14.86	0.04	0.13	0.07
3.5	Inferred Resources	0.46	7.76	15.57	0.04	0.13	0.07
4	Inferred Resources	0.40	8.39	16.34	0.04	0.14	0.07
4.5	Inferred Resources	0.34	9.02	17.07	0.04	0.15	0.08

14.6.1.5 Cape Ray Windowglass Hill Deposit

The drilling density is such that reasonable variogram models could not be defined. As a result the Inverse Distance method was used for estimation. As this method uses arbitrary weighting based solely on the distance from the point being estimated and the location of the samples it results in a lower confidence in the estimate. A nested search method was used such that any blocks estimated in the first search normally could be considered measured resources. But the reduced confidence in estimation through the use of Inverse Distance requires that all resources, regardless of which search resulted in the estimate be downgraded to Inferred. See Appendix E for resource classification definitions.

Resources that potentially could be mined by underground methods and using a 4.0 grams per tonne (g/t) cut-off, consist of a total of 0.07 million tonnes at a grade of 5.63 g/t Au and 22.22 g/t Ag of Inferred Resources.

Resources that potentially could be mined by open pit methods and using a 1.0 grams per tonne (g/t) cut-off, consist of a total of 1.80 million tonnes at a grade of 1.38 g/t Au and 8.49 g/t Ag of Inferred Resources.

All of the resources should be upgradable through gravity separation and possibly flotation. These resources are blocks above cut-off and have had no mineability criteria applied to them.

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Table 14.30 Classification of In-Situ Resources for the Cape Ray Windowglass Hill Deposit, using a range of Au cut-offs

Cut-off	Classification	TONNES	AU_GPT	AG_GPT
0.5	Inferred Resources	1.80	1.38	8.49
1	Inferred Resources	0.88	2.09	10.14
1.5	Inferred Resources	0.50	2.76	12.06
2.0	Inferred Resources	0.31	3.41	14.08
2.5	Inferred Resources	0.21	3.98	16.09
3.0	Inferred Resources	0.14	4.51	17.97
3.5	Inferred Resources	0.10	5.05	19.91
4	Inferred Resources	0.07	5.63	22.22
4.5	Inferred Resources	0.05	6.14	24.05

There is a low confidence in the lateral continuity of the mineralisation and so these resources cannot be used for a pre-feasibility or feasibility mining study.

Table 14.30 presents tonnes and grade for each Resource Classification using a range Au cut-offs for the Windowglass Hill deposit.

Figure 14.28presents the Au tonnes-grade curves for the Cape Ray Windowglass Hill Au-Ag Deposit and helps illustrate the effect of different cut-offs on available resources. The mining and processing methods chosen will determine what proportion can be converted to reserves as these do not take into consideration mineability and dilution.

While resource estimation has been done for Au and Ag the poly-metallic nature of the mineralisation may require that a Net Smelter Returns (NSR) evaluation be done to properly evaluate the economics of recovering any of the identified resources.

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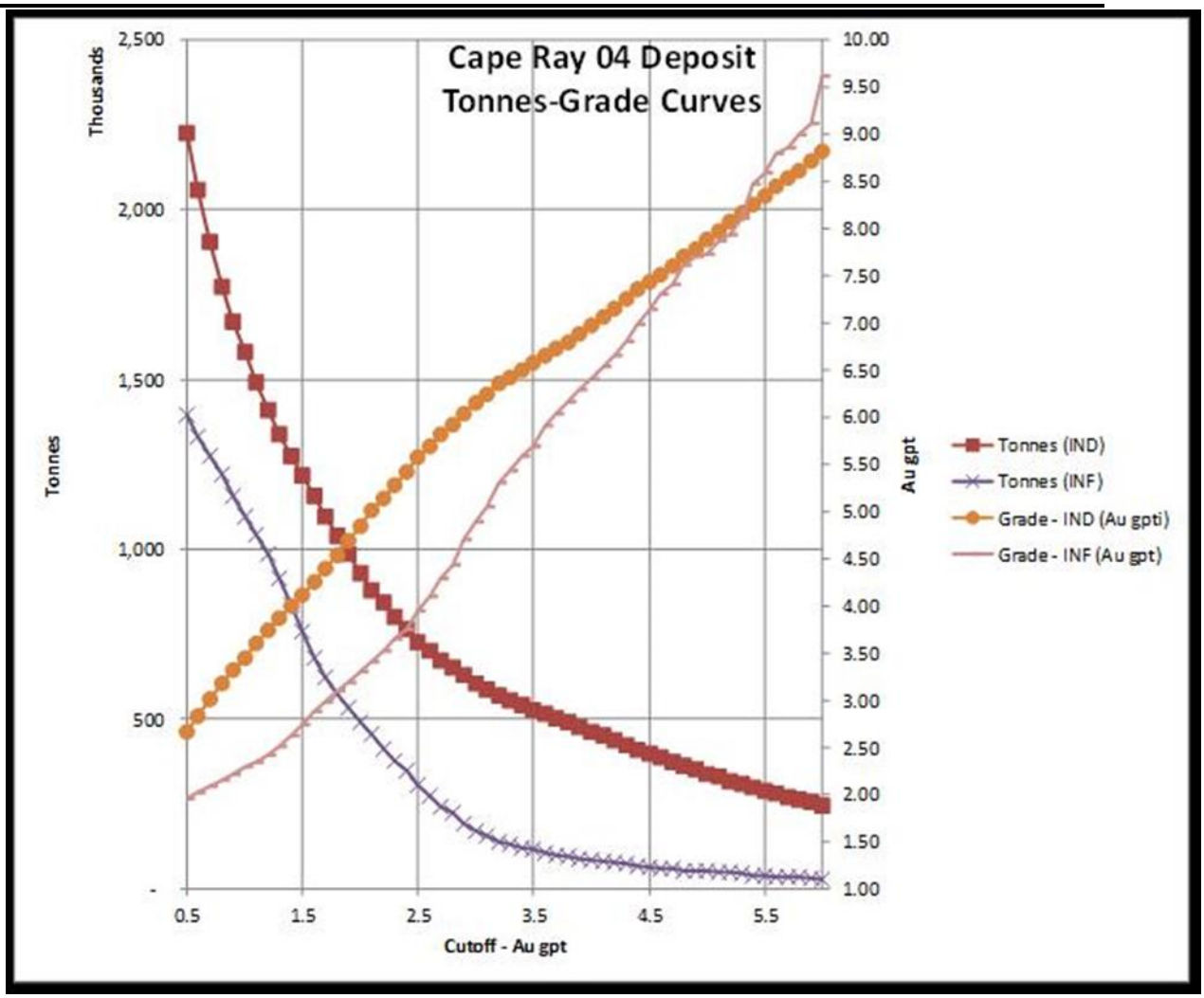

Figure 14.27 Au Tonnes-Grade curves for the Cape Ray 04 Au-Ag Deposit

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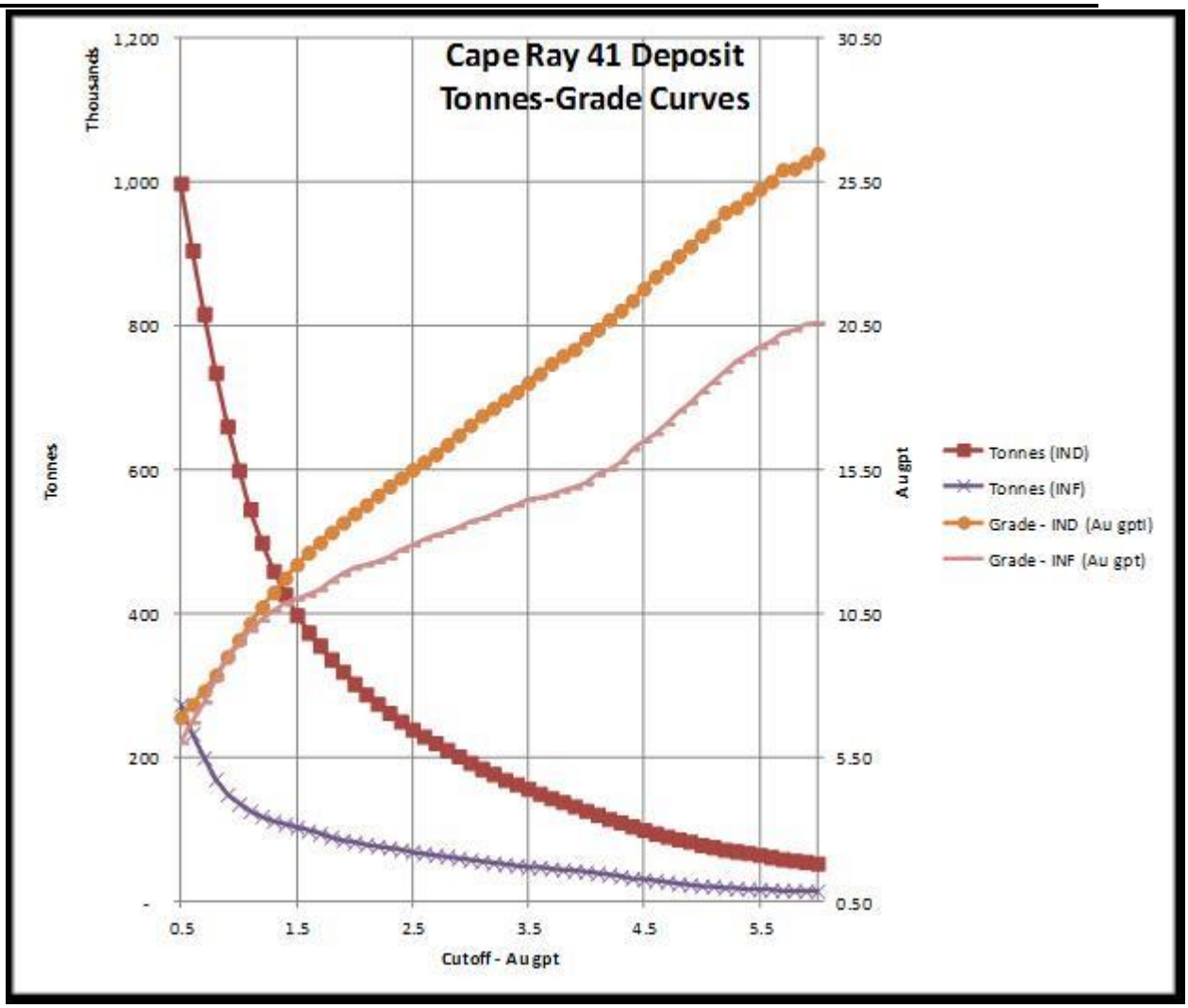

Figure 14.28 Au Tonnes-Grade curves for the Cape Ray 41 Au-Ag Deposit

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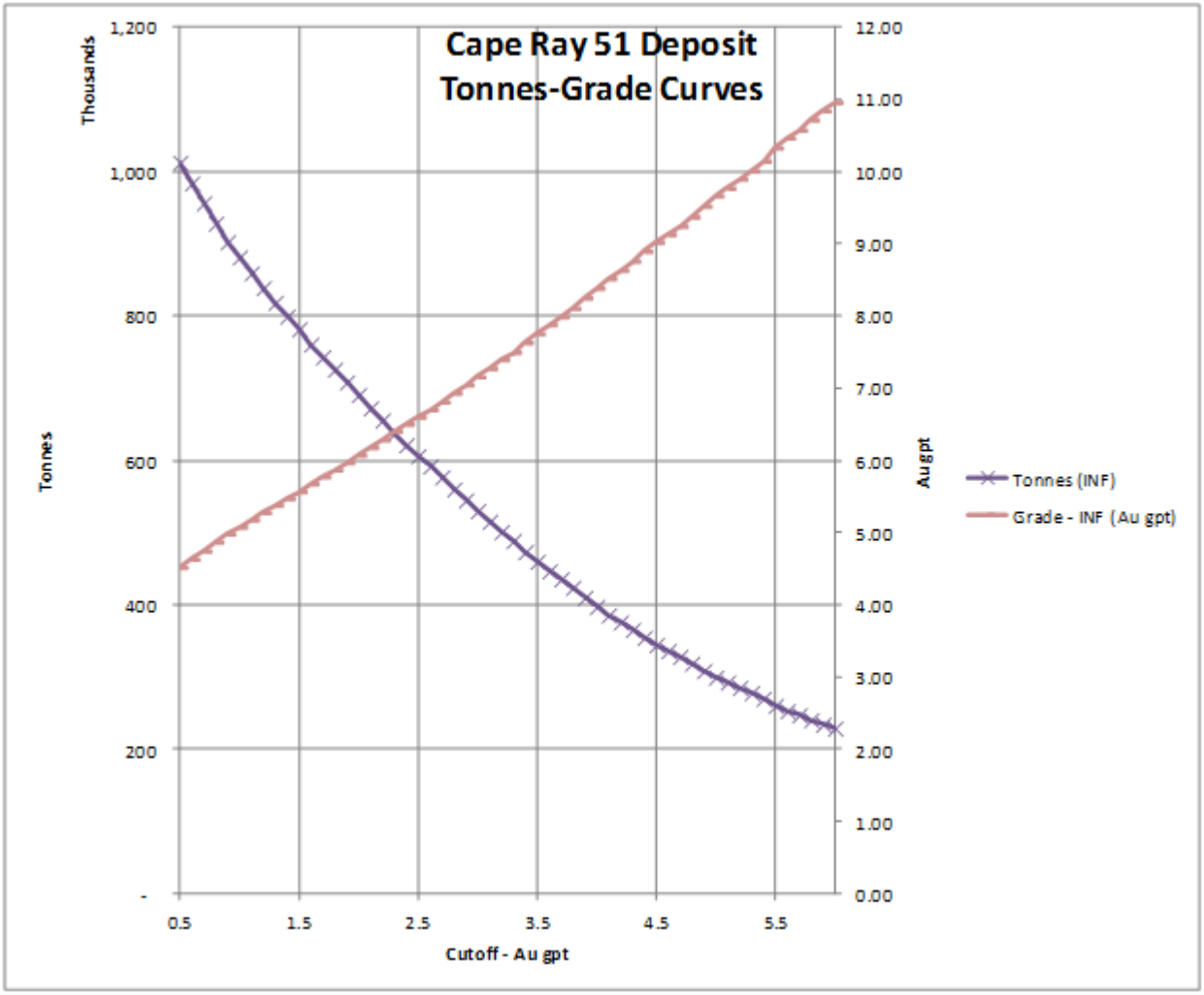

Figure 14.29 Au Tonnes-Grade curves for the Cape Ray 51 Au-Ag Deposit

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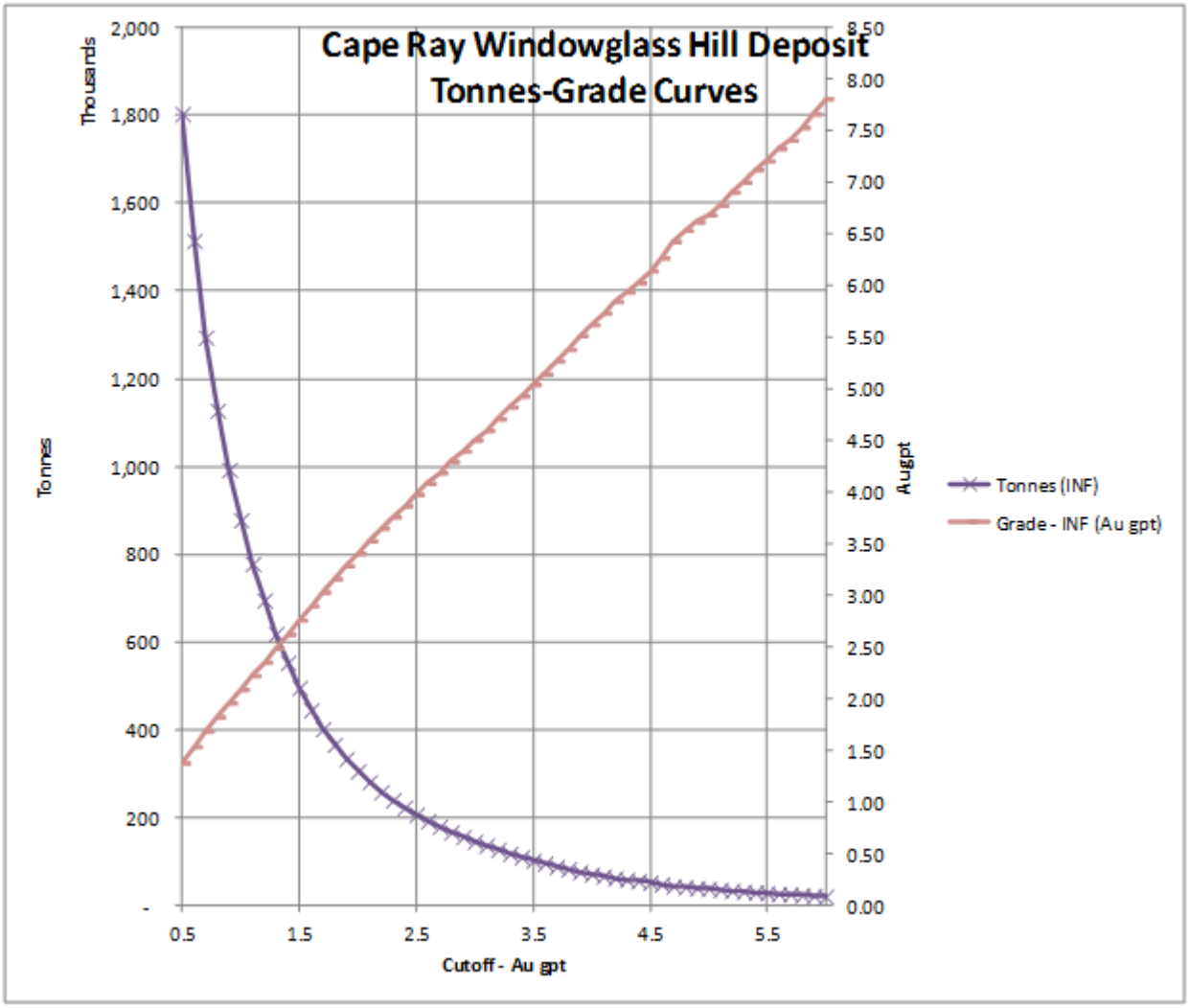

Figure 14.30 Au Tonnes-Grade curves for the Cape Ray Windowglass Hill Au-Ag Deposit

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15. Mineral Reserve Estimates

This section is not applicable to this report.

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16. Mining Methods

16.1 Introduction

This PEA is based on the updated mineral resource estimate completed by Sibley Basin Group Ltd as reported in Section 14.

The PEA for the 04, 41, 51, and WGH deposits includes inferred mineral resources that are too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

The mining methods, mine design, mine plan and costing was prepared by Nordmin's mining group and reviewed by PRB Mining Services. PRB found the work suitable for the scope of this PEA study and is the QP for said sections.

The relative location and geometry of the 41 and WGH deposit to surface warrant an open pit mine scenario. The bottom of the 41 deposit below the pit would be mined underground using a longhole stope method with cemented and un-cemented rock fill.

The 51 deposit proposed plan incorporates a small surface trench with the remainder of the deposit mined underground using a longhole stope method with cemented and un-cemented rock fill. Due to the geotechnical constraints and dips of the 04 deposit two mining methods are considered

- In the shallow dipping area (45^0) a cut and fill drift method is proposed.
- In the steeply dipping area (60^0) sub levels shrink (SLS) technique is proposed. The SLS method applies to only 7% of the 04 deposit's resources.

Each underground deposit would be accessed by a ramp from surface, with broken rock transported to surface in underground haul trucks.

The proposed Cape Ray production schedule would begin with the open pit mining of the 41 deposit. The 41 open pit would take approximately 2 years to complete and generate approximately 500,000 tonnes of mill feed and 6.4 MT of waste that would be made available for underground backfill. The 41 pit would supply initial feed to the mill and surface stockpile. The 04 underground deposit ramp would start shortly after mill feed from the 41 open pit is established. The 04 underground capital development would take one year to complete with production from the 04 SLS and cut & fill beginning in year 2. Production from the 04 underground would take 2.5 years producing approximately 480,000 tonnes of mill feed.

Once the 41 open pit is complete, towards the end of year 2, the 41 pit crew and equipment would move to the WGH deposit. The WGH open pit would take approximately 3 years to mine producing approximately 325,000 tonnes of mill feed and generating 1.9 MT of waste rock. The WGH open pit mining would be complete early in year 6 and the contractor pit crews and

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equipment would demobilize at this time.

With the 04 underground deposit mined out in year 4, the underground crews and equipment would move to the small area under the 41 open pit. Once the 41 underground development is completed, the crews and equipment would move to the 51 deposit. The 41 underground would start production and generate mill feed in year 5. The 51 underground deposit would be complete in year 6. The 51 and 41 underground deposit would generate approximately 350,000 tonnes and 45,000 tonnes of mill feed respectively over a 2 year period. A 6 year life of mine is envisaged.

The proposed plan is to operate the open pits for 6 months of the year, from May to November producing approximately 1,500 t/d of mill feed with the surplus stockpiled by the mill. The annual open pit production is anticipated to be 285,000 tonnes per year.

The underground deposits would be mined year round with an anticipated production rate of 600 t/d for an annual production rate of 174,000 tonnes.

The total feed to the mill over the 6 year mine life would be 1.7 MT and would be processed by the mill at a rate of 850 t/d or 285,000 tonnes per year (see Table 16.1).

Table 16.1 Production schedule (Potential Mineable Resources, in thousands)

Production Schedule	Yr. -2	Yr. -1	Yr. 1	Yr. 2	Yr. 3	Yr. 4	Yr. 5	Yr. 6	Recovered Diluted Tonnes Total	Au Avg (g/t)	Ag Avg (g/t)
Pit 41	-	-	288	215	-	-	-	-	503	3.12	10.31
U.G 04	-	-	-	33	227	219	-	-	479	5.99	14.07
Windowglass	-	-	-	10	59	90	134	35	328	2.24	2.68
U.G. 51	-	-	-	-	-	-	128	219	347	7.29	14.21
Other (41 U.G. / 51 Trench)	-	-	-	-	-	-	44	-	44	6.89	5.24
Total	**-**	**-**	**288**	**258**	**286**	**309**	**306**	**254**	**1,701**		

16.2 Geotechnical Analysis

The PEA level geotechnical investigation work for selection of an underground mining method and required ground support as well as preliminary open pit parameters was performed by Mine Design Engineering. The recommendations formed the basis for the mine methods and selected methodology used in this technical report (PEA).

The full geotechnical analysis report - MDENG Technical Report 0316-R0814-D02 is available for review.

Mine Design Engineering (MDEng) of Kingston, Ontario, has conducted analyses based on geotechnical information collected from previous studies and reports and observations of core from previous drilling campaigns. A summary of the results are as follows:

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16.2.1 Geotechnical Site Characterization

Rock conditions on site are very poor to good (mean Q rating between 0.6 and 32), in moderately strong rock (mean UCS 25-50 MPa). Structural conditions are assumed to be consistent across site, dominated by foliation dipping at approximately 55° to the southeast and a sub-horizontal set (J1) dipping at about 20 degrees to the northwest.

16.2.1.1 Open Pit Geotechnical Design

Southeast facing slopes may support a maximum inter-ramp pit angle of 45° by empirical assessment. It has been assumed that benches would be 10m high; and while it is likely that benches can be drilled vertically, it is possible that the upper 5 m of southeast facing slopes would fail back to foliation, producing an effective bench face at 71°. These benches should be designed with 10 m catch bench width; this allows for a final effective bench width of 6.5 m after the hypothesized instability.

On all other pit slopes, an inter-ramp angle of 52° is recommended; a bench width of 7.8 m is recommended to produce an inter-ramp angle of 52°, assuming 10 m high, near-vertical bench faces. This pit slope angle is also recommended for the Windowglass Hill deposit.

For the shallow (about 50 m) Windowglass Hill pit, an inter-ramp angle of 57° is recommended; assuming 10 m high, vertical benches, a minimum catchment width of 6.5 m is suggested by rule-of-thumb calculations.

Further site investigation is recommended to establish geotechnical domains, and provide structural orientation data to be used in more detailed analyses.

16.2.1.2 Underground Geotechnical Designs

Empirical methods were used to estimate appropriate stope sizes for the given ground conditions and make recommendations for mining method based on stope size restrictions.

Two mining methods have been put forward for consideration by this study:

Underhand Drift and Fill (assuming the full width will be selectively mined, with transition zone left in place).This mining method was not used in the mine design. A cut and fill drift with shotcrete, bolt and screen support was used in the mine design.

Long Hole Open Stoping (assuming the full width of the geological structure, plus the transition zone will be mined from HW and FW).

- Endwall instability will be a major risk to longhole stoping.
- Mine design will require small sub-level intervals
- 04 Deposit Recommended Hydraulic Radii: (Unsupported HW/FW: 3.5, Supported HW/FW: 5)
- 41 Deposit Recommended Hydraulic Radii: (Unsupported HW/FW: 3, Supported HW/FW:

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All development will require:

- Short rounds to avoid uncontrolled caving of the back.
- In-cycle shotcrete to provide early confinement to the back and sidewalls.
- Bolts and weld wire mesh screen should be installed over the shotcrete. Swellex bolts length should be ½ the drift span (dictated by deposit thickness), with spacing equal to no greater than ½ the bolt length (a maximum practical support spacing is 1.2 x 1.2 m).
- For spans exceeding 6 m Super Swellex (24 tonne) should be utilized. Where development exceeds 5-6 m it will likely be necessary to develop a primary drive, and then slash out the sidewall(s) after support has been installed in the primary drive.

16.3 Underground Mining

16.3.1 Underground Mine Design

The selected stope cut-off grade for underground mining was 4.0 g/tonne Au. This was selected based on the parameter from Table 1.3 and Table 21.3. The block model was set to a cut-off grade of 4.0 g/t Au and a preliminary mine design was created.

All the operating costs are based on Canadian norm prices from contractors, suppliers and other similar type Canadian projects, for consumables and parts. The cost of power is based on rates charged by Newfoundland Labrador Hydro for similar sized power consumers in the province.

The 04 deposit would be developed from the ramp at approximately 20 m vertical intervals. In the steep dipping area (60^0) a sub levels shrink (SLS) technique is envisaged. The SLS method applies to only 7% of the 04 deposit's resources. This sub level shrink (SLS) method developed at Mt. Wright in Queensland is still being used today. The SLS method eliminates the need for cemented rock fill.

The majority (93%) of the 04 zone is shallow dipping (45^0). It is envisioned that a cut and fill drift mine method would be used in that area. Development in the mineralized zone would incorporate shotcrete with swellex bolts and screen over the shotcrete for support. The developed sill cut would be backfilled tight to the drift back with unconsolidated waste rock prior to taking the sill cut above. With the zone dipping at 45^0 the cut and fill mining could commence on multiple horizons. All muck would be loaded on to underground 30 tonne haul trucks and hauled-up the ramp, to surface. With the SLS stopes mined out waste rock fill would be drawn from the SLS stopes for fill in the cut and fill drifts. Pit waste rock would be introduced from surface to the fill raise to keep the SLS stopes full. Figure 16.1 represents the 04 underground SLS stopes and the cut & fill drifts.

The 51 deposit would have levels developed from the ramp at approximately 18 m vertical intervals. This small sub level interval will reduce the stopes hydraulic radius (HR) to control hanging wall failure. Each level could be developed from the ramp to the mineralized zone. All muck would be loaded on to underground haul trucks and hauled-up the ramp to surface.

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The mining method proposed would be longhole open stoping with placement of consolidated rock fill (CRF) in the primary stopes and unconsolidated waste fill in the secondary stopes. Figure 16.2 represents the proposed 51 underground development and longhole stopes.

The 41 open pit would be mined first to generate early resource tonnes to the mill and supply waste rock for underground fill. On completion of the 41 open pit, the 41 underground mining would commence. The 41 underground deposit could be mined using longhole stopes with cemented and un-cemented rock fill. Figure 16.3 represents the proposed 41 underground mine.

The mine access ramps for each deposit would facilitate movement of equipment, rock, manpower and materials to and from the underground mine.

Most underground support facilities would be located on surface including the maintenance shop, main washrooms and the fuel farm. A push-pull ventilation arrangement would utilize raises at each extremity of the mineralized zones for fresh or return air. The ramp system would also be used to ventilate the mine.

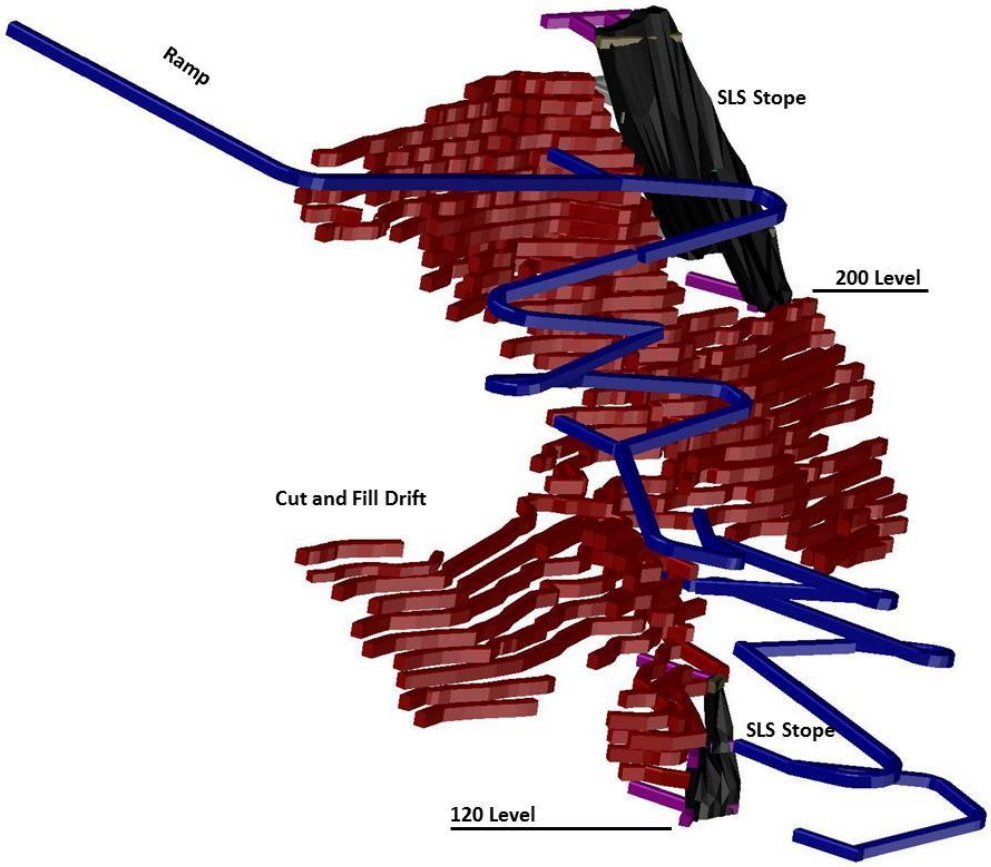

Figure 16.1 Representation of the 04 SLS stopes and Cut & Fill Drifts.

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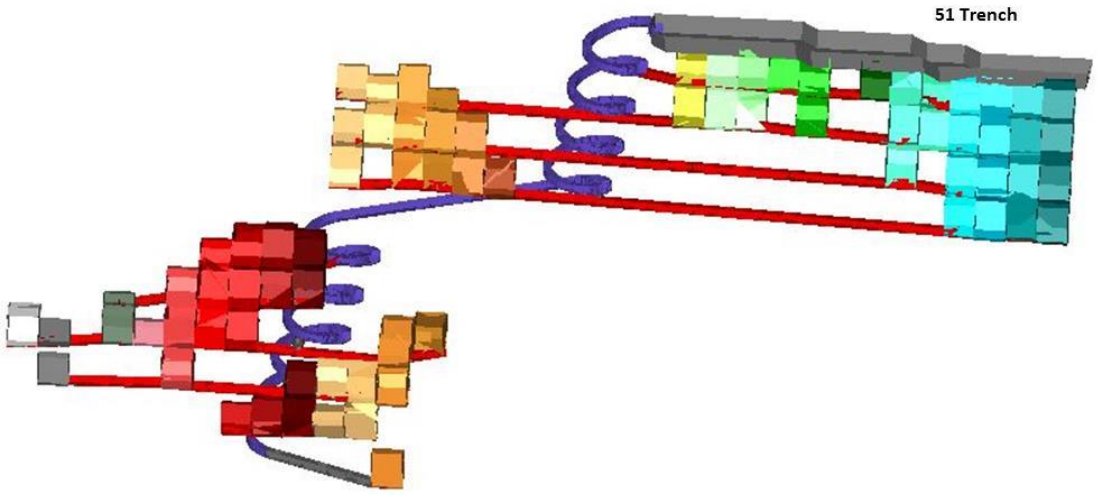

Figure 16.2 Representation of the 51 underground LH Stopes.

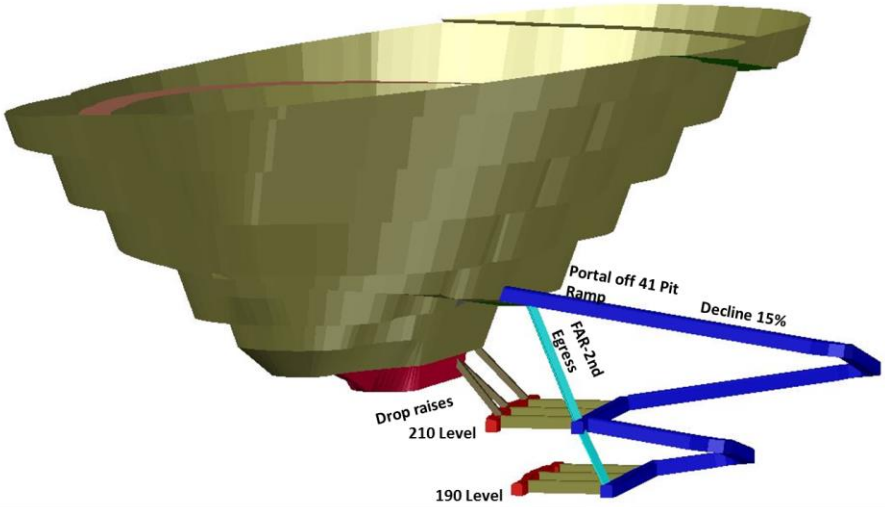

Figure 16.3 Representation of the 41 Underground Stopes below the 41 Pit.

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16.3.2 Underground Mine Development and Infrastructure

16.3.2.1 Access Ramp

The access ramp for the 04 and 51 deposit would be developed from surface at a grade of 15% to 18% respectively. The access ramp for the 41 deposit would be developed at 15% from within the pit. The required length of ramp for each deposit is approximately as follows:

- 04 Ramp 1,582 meters at 15%
- 41 Ramp 446 meters at 15%
- 51 Ramp 1,119 meters at 18%

Remuck bays would be provided every 150 m along the ramp. The ramp would have dimensions of 4.5 m by 4.5 m to accommodate the underground haul trucks and also to provide a reasonable cross-sectional area for ventilation. Safety bays would be spaced at 30 m intervals, as required. Services located in the main ramp would include compressed air line, water line, discharge water line, electrical power cables, blasting line and a leaky feeder cable.

The 4.5 m wide by 5.5 m high muck bays would be located on each mucking horizon. These loading areas would accommodate a Load-Haul-Dump (LHD) loading a 30 tonne truck.

16.3.2.2 Level Development

Levels would be developed with access from the main ramp to the SLS zones and LH stopes. The levels would consist of a haulage drift in waste running the entire strike length of the zones to be mined. Drawpoints on the bottom SLS mucking horizon would be at 10 meter spacing. Drawpoints on all other mucking horizons would be at 20 meter intervals

Access drives to the cut and fill drifts would be 3.5m wide and 3.8m high and would be excavated at a grade of -10% , 0%, and +10%. Figure 16.5 shows the typical cut and fill drift access.

Access to the ventilation raise and 2nd egress would be developed on each level. Truck loading stations would be located at intersections of the haulage drift and ramp access. The loading area would have the backs excavated to a height to facilitate an LHD loading 30 tonne underground haul trucks. Other level development would include temporary electrical sub-stations, sumps and refuge stations.

Services installed in the haulage drift would include 51 mm water line, 102 mm compressed airline, electric cable, blasting line and a leaky feeder cable. At each active mucking stope an auxiliary ventilation fan would be located in the haulage drift with ventilation ducting hung into the draw point for ventilation.

All ramp and level waste development would be performed by the mining contractor using 2 boom electric hydraulic drill jumbos, LHDs, 30 tonne haul trucks, scissor lift/bolters and other rubber tired diesel-powered support equipment.

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16.3.3 Underground Service Infrastructure.

16.3.3.1 Process Water

Process water would be used for underground activities and muck conditioning.

16.3.3.2 Compressed Air

Compressed air would be supplied at 110 PSI from two electric centrifugal air compressors. The compressors would be connected to the 102 mm schedule 40 pipe in the main ramp with distribution to each haulage drift and stope draw point.

16.3.3.3 Collection Sumps

Process water collection sumps would be located throughout the mine and would be staged pumped along the main ramp back up to the surface collection pond.

16.3.3.4 Refuge Station

Refuge stations would be set up near each of the deposits. The refuge stations would be 15 m long by 4 m wide and 3.8 m high; they would be located off the haulage drifts. The refuge stations ground support would consist of 1.8 m resin grouted rebar bolts installed on a 1.2 m by 1.2 m pattern with welded wire mesh screen, on the back and walls to the drift sill. The floor would be poured concrete. The access wall would be shotcrete, and a steel man door would be installed for entry. The refuge station would contain potable water, fire clay, rescue and first aid equipment, a fire extinguisher, an eyewash station, emergency rations and a stretcher. The refuge station would also serve as a heated lunch room with basic amenities. The refuge station would be vented through intake and exhaust ventilation ducts which can be sealed off in an emergency. A compressed air line would also service the refuge station. The 41 underground deposit due to the short mine duration would have a portable refuge station.

16.3.3.5 Explosives Magazine

An explosives magazine would be located off the main ramp and would meet the Newfoundland Mines Act and Regulations requirements. The powder magazine would also be equipped with sufficient auxiliary ventilation to adequately ventilate the area.

16.3.3.6 Detonator Magazine

A detonator magazine would be located off the main ramp and would meet the Newfoundland Mines Act and Regulations requirements. The detonator magazine would also be equipped with sufficient auxiliary ventilation to adequately ventilate the area.

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16.3.3.7 Materials Handling and Storage Area

With access readily available to surface via the ramp system, the main storage of materials would be in a surface storage building or surface laydown area. Smaller amounts of material would be stored underground in idle headings. Fuel and lubricants would be transported underground by the lube truck which would service underground equipment at the face. Mobile equipment destined for surface would stop at the surface fuel farm for fuel and lubes prior to heading back underground.

16.3.3.8 Latrines

The heated washrooms on surface would be easily accessible from the mine due to the relatively short distance from the work headings to surface. Underground toilets of a temporary nature would be provided. They would be of portable design and the catchment containers would be checked daily and exchanged as required. The used catchment containers would be transported to the sewage treatment area where they would be dumped and cleaning before they are returned into service.

16.3.3.9 Communications and Controls System

The main emergency notification system would be a stench gas that is injected into the main fresh air ventilation system and the compressed air lines. A mine radio system would be installed to allow communication on surface and underground through the leaky feeder cable.

16.3.4 Mine Production Methods

It is envisaged that three mining methods would be used to mine the 04, 51 and 41 underground deposits. The 51 and 41 underground deposit would be mined by conventional longhole stoping. The 04 deposit would be mine by a cut and fill drift method and a small test area would be mine by sub level shrink stoping (SLS). Table 16.2 below shows the tonnes and grade by deposit and mining method envisaged.

Table 16.2 Underground deposits and mine methods (in thousand)

Mining Method	Deposit 04			Deposit 51			Deposit 41		
	Tonnes	Au g/t	Ag g/t	Tonnes	Au g/t	Ag g/t	Tonnes	Au g/t	Ag g/t
Longhole	-	-	-	347	7.29	14.21	39	6.1	0.92
Cut & Fill	445	5.99	14.07	-	-	-	-	-	-
SLS	34	5.99	14.07	-	-	-	-	-	

16.3.4.1 Longhole Stoping

Longhole open stoping could be the primary method used underground to mine the 51 and 41 deposits. Stopes would have a nominal dimension of 20 m on strike by 3m depth with sub level intervals of 18m. The 41 underground sub level interval of 20 m is anticipated.

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Mineral extraction would commence from the bottom stope in each area and continue upwards. Primary stopes would be filled with cemented rock fill (CRF) and secondary stopes with un-cemented rock fill.

Longhole open stoping requires an undercut and top sill drift to be developed for each stope. The undercut sill is the extraction level from which stopes are mucked out. The top sill is initially the drilling horizon. After a stope has been backfilled, the top sill is used as a mucking horizon for the longhole stope directly above the backfilled stope.

Mucking and backfilling activities would be accomplished utilizing remote controlled load-haul-dump (LHD) units.

With the 41 underground a sump at the bottom of the 41 pit would collect runoff water that is diverted away from the stope breakthrough. Runoff water would be pumped to the collection pond outside the pit.

16.3.4.1.1 Stope Sequencing

Stopes would be sequenced from the bottom of the zone with primary stopes filled with CRF and secondary stopes with waste rock. Figure 16.4 shows a typical stope sequence in a long section view.

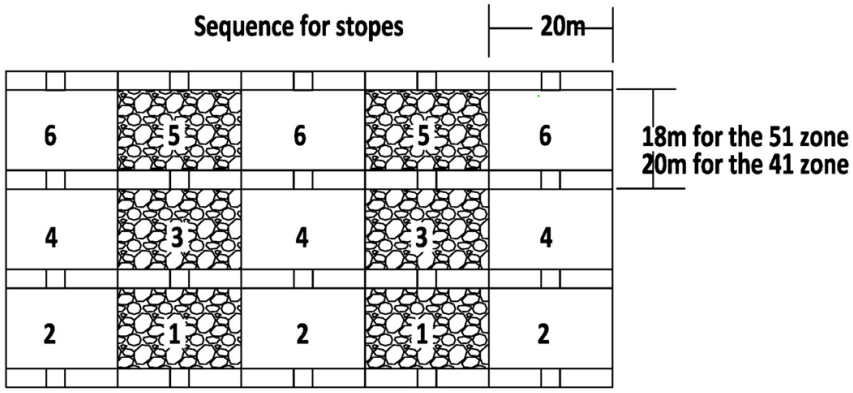

Figure 16.4 Stope Sequence

16.3.4.1.2 Mining Dilution

The longhole open stopes are mined from the bottom up with primary stopes filled with cemented rock fill and secondary stopes with waste rock.

The following methodology is used to estimate the dilution from all sources. The dilution estimate is from the stope perimeter walls sloughing. All dilution is considered unplanned dilution from stope wall failures, blast over break or backfill mixed in the mineralized muck from the stope floor.

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The maximum extent of hanging wall, footwall and side wall slough into the stopes is controlled by the short sub level interval. Backfill dilution would also come from mucking off the waste filled stope below. Emphasis would be placed on mucking operation to minimize this dilution. Dilution in the range of 20% can be expected with longhole mining.

16.3.4.1.3 Mining Recovery & Loss

Recovery of (diluted) resource tonnes (RDT) is expected to be 95%. The losses can occur from inefficient drilling or hole deviation, blasting in stope corners and walls, difficult remote mucking in stope corners and edges, or abandoning a stope due to excessive dilution from wall failure. Mineralized rock would be mucked until the stope is empty or deemed waste under geology control.

16.3.4.2 Cut and Fill Drifting

The majority (93%) of the 04 deposit dips at 45° and would be mined with a cut and fill drift method. The drifts would be 5m high and 4m wide and would follow the mineralized zone. These drifts would be shotcreted then bolted and screened with 1.8m swellex bolts on a 1.2mx1.2m pattern and 6 gauge mine screen. The drifts would be back filled with waste rock. A "rammer jammer" may be utilized to push the waste rock tight to the cut and fill drift back. Waste fill would come from the mined out SLS stopes, rock fill, and lateral development waste. Figure 16.5 shows a typical -10%, 0% and +10% access drift to the cut and fill drive.

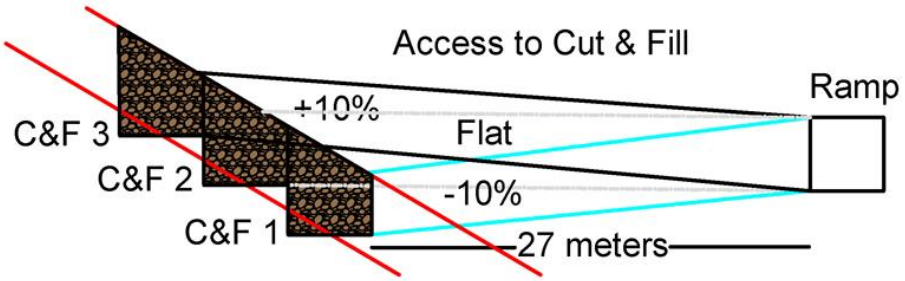

Figure 16.5 Cut and Fill Access

16.3.4.2.1 Stope Sequencing

Cut and Fill drift mining would be from two horizons, the 200 level up and the 120 level up. Access drives into the cut and fill drift would be centrally located allowing two mining faces in the cut and fill drift, one in each direction. A third horizon could be prepared to secure production if necessary. A sill pillar would be left between horizons. The thickness of the sill pillar would depend on geological structures but is anticipated to be less than 5 m thick.

16.3.4.2.2 Mining Dilution

Cut and fill mining dilution is expected in the 15% range as selective mining and shotcrete support would help control drift dilution. Drift mucking off the waste sill must be controlled to minimize dilution.

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16.3.4.2.3 Mining Recovery and Loss

Recovery of (diluted) PMR tonnes is expected to be 85% in the cut and fill mining.

16.3.4.3 Sub Level Shrink (SLS)

Two small areas within the 04 deposit dip at 60 degrees and a possible test mining method envisaged is sub level shrink (SLS).

The SLS stopes would be mined from the top down. The stope would be drilled from the level below. The fanned upholes would be drilled, loaded and blasted a few rings at a time until all of the stopes on the level are blasted. After each blast an LHD would clean up the blast area and muck some of the blasted rock swell. Blast holes would be collar primed with long period delay caps. Long delays are essential in this choke blast environment. Controlled mucking is critical to ensure the stope is full of muck at all times. This muck supports the footwall (FW) and hanging wall (HW) reducing wall sluffing.

Mucking initially is to remove blasted rock and to confirm breakage by drawing down rock from the bottom level. Once the SLS zone is completely mined, drilled and blasted, a controlled amount of muck would be drawn from each draw point on the bottom level. The draw points on the bottom muck level would be approximately 10 m apart. The muck would be drawn down evenly across the whole level and waste rock would be added on the top level to keep the stopes full. The majority of mucking would be done manually. To maximize muck extraction from the SLS stopes the bottom mucking level draw point would access the stopes from the footwall side. An even drawn down of muck would reduce the amount of waste rock mixed in the mineralized muck. A muck sampling program along with a geologist inspection of the draw point muck would determine when the SLS stope is mucked out.

Waste rock used for fill would be generated from the lateral development cycle and waste rock from the mining of the 41 open pit. A fill raise from the top SLS stope over cut to surface would be used to deliver pit waste to the stope. An LHD would distribute waste rock on the top level in the SLS stope as required. Figure 16.6 shows top down mining with blasted SLS stopes. Also a typical array of uphole fan drilling and mining direction. Figure 16.7 represents the SLS stope blasting complete and Figure 16.8 the muck draw down and waste rock fill placement.

16.3.4.3.1 Stope Sequencing

SLS stopes would be sequenced from the top down. Drilling and blasting would retreat on a level from one end of the SLS stope longitudinally. Figure 16.6 shows the typical stope retreat in a long section view.

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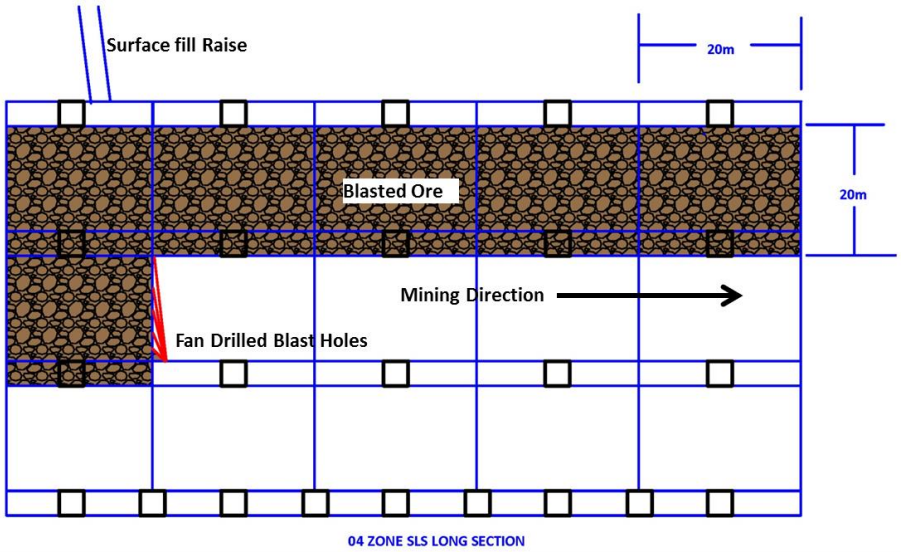

Figure 16.6 SLS Stope Mining Direction with fanned blast holes.

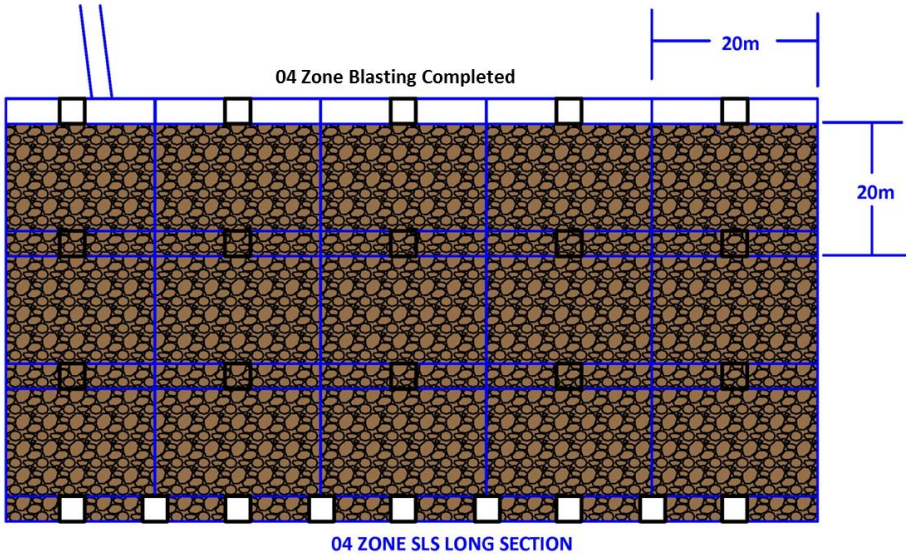

Figure 16.7 SLS Stope Blasting Completed

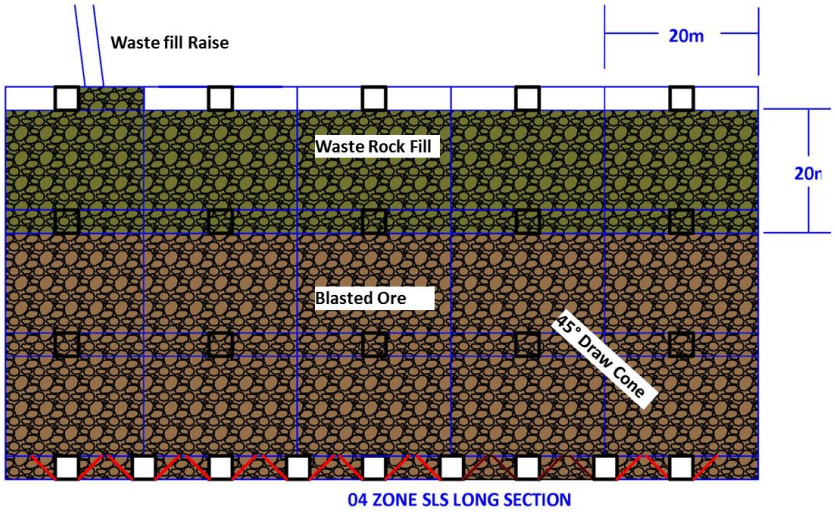

Figure 16.8 SLS Muck Draw Down and Waste Filling Above

16.3.4.3.2 Mining Dilution

The restraining force of the broken rock in the SLS stopes minimizes hanging wall, footwall and side wall slough into the stope. Dilution would occur as muck is drawn down creating friction on the stope walls. A small amount of dilution may occur with uneven draw down causing waste rock fill mixing in with mineralize muck. Emphasis would need to be placed on the mucking operation to minimize this dilution. Dilution in the range of 20% could be expected with SLS mining.

16.3.4.3.3 Mining Recovery and Loss

Recovery of (diluted) resource tonnes (RDT) is expected to be 80% with the SLS method. Mineralized material left between the drawpoints would account for the majority of loss (see Figure 16.9).

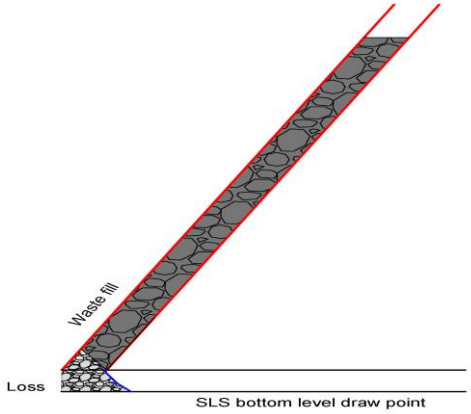

Figure 16.9 Typical cross-section of the SLS mucking drawpoint

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16.3.5 Backfill Distribution System

Underground development waste rock would be generated as follows: 04 zone 363, 000t, 51 zone 147,000t, and 41 zone 35,000t.

Pit waste would be generated as follows: 41 Pit 6,363,000t, WGH Pit 1,900,000t and 51 Trench 26,000t.

Pit waste generated: 8,286,000 tonnes

Underground waste generated: 545,000 tonnes

Total Waste generated: 8,831,000 tonnes

Table 16.3 below shows the rock fill tonnes required underground per deposit.

Table 16.3 Waste fill tonnes required (in thousands)

Underground Deposits	Rockfill Required Tonnes
04	321
51	233
41	26
Total Rockfill	**580**

16.3.5.1 The 04 deposit backfill methodology.

On surface, waste rock would be dumped into the backfill raise and travel down the raise to the top of the SLS stope. From the top SLS sill waste rock would be distributed to the SLS stope requiring fill with an LHD.

The cut and fill drifts would be backfilled with development waste and waste rock from the filled SLS stope. Once the development crew completes the first lift they would backfill this drift then commence with the sill cut above.

16.3.5.2 The 51 and 41 deposit backfill methodology.

On surface, waste rock would be dumped into the backfill raise and travel down the raise to a designated backfill station. A portable concrete slurry batching plant would be set up on surface near the 51 underground deposit. If cemented rock fill (CRF) is required cement slurry would be sent underground via a drilled hole to the backfill station and cement slurry sprayed on a truck loaded with waste rock. The CRF would then be trucked and placed into the stope.

16.3.6 Ventilation

The ventilation system is required to provide clean air of adequate volume to remove airborne contaminants from the underground work headings. The air flow requirements were based on the Newfoundland Occupational Health and Safety Act and Regulations dated January 17, 2012

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which states that ASHRAE, ACGIH requirement to be observed. Table 16.4 presents the equipment ventilation requirements totalling 144m³/s (310,000 CFM) of airflow at peak production.

Table 16.4 Equipment Ventilation Requirements Calculation

Unit	Quantity	Engine Size kW	Utilization %	Unit Airflow (m³/s)
Production 4.5" ITH drill	1	0	0%	0.0
Production Top hammer drill 2" to 3.5"	1	74	10%	0.7
LHD ST7 Production	2	144	95%	24.6
LHD ST7 Development	1	144	45%	5.8
30 tonne Haul Truck Production	2	298	95%	51.0
30 tonne Haul Truck-Development	1	298	50%	13.4
Development Jumbo	2	110	20%	4.0
Mechanical bolter	2	62	20%	2.2
Shotcrete spray carrier	2	110	30%	5.9
Scissor Lift	2	62	20%	2.2
Fuel - Lube truck	1	110	20%	2.0
Grader	1	193	50%	8.7
Forklift	1	110	50%	5.0
Mechanics Vehicles	2	45	25%	2.0
Personnel Vehicles	3	45	25%	3.0
Subtotal (m³/s)				130.5
Contingency 10%				13.1
Total (m³/s)				**143.6**

16.3.6.1 Ventilation Plan

Air distribution would be dependent on the quantity and location of diesel equipment deployed throughout the mine. Vent regulators, air lock doors and auxiliary vent fans would control and direct air volumes and flow to the required areas.

The primary ventilation system proposed is a push-pull system which would consist of high pressure exhaust fans and low pressure intake fans located on surface. Low pressure fans would push fresh air down the fresh air raise. This air would be heated in cold weather conditions. The high pressure exhaust fans would draw air from underground up the exhaust raise to surface. The ventilation network was not simulated. The fans selected and costs are typical for the size of these underground operations.

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16.3.7 Life of Mine Development Plan and Schedules

Pre-production mine development and construction for underground requires approximately 1.6 years that would be performed by a mining contractor. Initially one crew would drive the main ramp for the 04 deposit from surface with the second crew starting as soon as the first access level is reached. The two development crews would drive the ramp system, SLS haulage and the cut and fill drifts. The 51 U/G would follow in sequence as manpower and equipment could be released from the 04 deposit. The anticipated life of mine for underground mining is 4 years operating 350 days per year at the following mining rates.

Table 16.5 Expected Production Rates

Single Headings	**Rate**
Ramp and lateral development	5.2m/d
Multiple Headings	
Cut and Fill development (2 crews D/S, 2 crews N/S)	14.4m/d (Production)
Cut and Fill backfilling	600t/d

The 41 underground deposit would be mined once the 41 open pit is complete. The 41 deposit would be mined in conjunction with the 51 underground deposit. The 41 underground is only 39,000 tonnes of resource. The underground development schedule is shown in Table 16.6.

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Table 16.6 Mine Development Schedule

Development Schedule	Yr. -2	Yr. -1	Yr. 1	Yr. 2	Y3	Yr. 4	Yr. 5	Yr. 6	Total
04 Underground									
Ramp Development (m)	-	-	1,582	-	-	-	-	-	1,582
Level Development Waste and SLS sill (m)	-	-	238	134	-	-	-	-	372
Raise Development (m)	-	-	65	200	-	-	-	-	265
C&F Access Development waste (m)	-	-	-	1,686	569	-	-	-	2,255
Deposit Development with Shotcrete (m)		-	-	-	5,040	2,328	-	-	7,368
Total (m)									**11,842**
41 Underground (m)									
Ramp Development (m)	-	-	-	-	-	446	-	-	446
Level Development waste (m)	-	-	-	-	-	201	-	-	201
Level Development Sill (m)	-	-	-	-	-	160	-	-	160
Raise Development	-	-	-	-	-	143	-	-	143
Total (m)									950
51 Underground (m)									
Ramp Development (m)	-	-	-	-	-	800	319	-	1,119
Level Development waste (m)	-	-	-	-	-	-	-	-	
Level Development Sill (m)	-	-	-	-	-	-	-	-	
Raise Development(m)	-	-	-	-	-	-	-	-	
Total									**1,119**
Grand Total									**13,911**

16.3.8 Life of Mine Production Plan and Schedule

The mill production schedule is based on processing 850 tonnes of resource per day for 335 days per year, or 285,000 tonnes per year. The mill would have a capacity of 950 t/d based on 85% availability providing some flexibility to the operation.

The open pit mine would operate 183 days/yr. or 6 months of the year at a rate of approximately 1,500 t/d PMR. This mineralized muck would feed the mill and surface stockpile.

The underground mine would operate 350 days/yr. at a rate of approximately 600 t/d.

The production schedule by deposit for the underground and open pits is presented in Table 16.7. A total production of 1,7MT of resource material would be mined over the 6 years.

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Table 16.7 Production Schedule Underground and Open Pit by Deposit (in thousands)

Production Schedule	Yr. -2	Yr. -1	Yr. 1	Yr. 2	Yr. 3	Yr. 4	Yr. 5	Yr. 6	Recovered Diluted Tonnes
Resources									
Pit 41	-	-	288	215	-	-	-	-	503
U.G 04	-	-	-	33	227	219	-	-	479
Windowglass	-	-	-	10	59	90	134	35	328
U.G. 51	-	-	-	-	-	-	128	219	347
Others (41 U.G. / 51 Trench)	-	-	-	-	-	-	44	-	44
Potential mineable resource									
Start of Period	-	-	1,701	1,413	1,155	869	560	254	-
Mined	-	-	288	258	286	309	306	254	1,701
End of Period	-	-	1,413	1,155	869	560	254	-	-
Production Potential mineable resource									
Mined	-	-	288	258	286	309	306	254	1,701
Processed (850 t/d) for 335 days	-	-	285	261	286	309	306	254	1,701
Stockpile	-	-	3	-	-	-	-	-	-

16.3.9 Underground Equipment

16.3.9.1 Underground Mine Fleet

The underground mining equipment would primarily consist of rubber tired diesel powered equipment. Some of this equipment would be shared during the development and production phase. At full development capacity the equipment utilized is shown in Table 16.8.

Table 16.8 Required Equipment for Development.

Development	Quantity
2 boom electric/hydraulic jumbos	2
LHD's ST7	3
30 tonne haul trucks	3
mechanical bolters, or scissor lifts dedicated to ground support	2
scissor lifts (service installation and loading)	2

A fuel / lube truck would be used to service the underground mobile fleet. The main ramp would be maintained with one grader. Material received would be transported underground with a 2 yard forklift. Five man carriers would be utilize to move mining and support service crews into and out of the mine. These man carriers would also be used to move maintenance and support staff into and out of the mine. Two shotcrete spray units would be required during the cut and fill drift mining. Drills would be required for longhole production drilling and deep swellex bolt support. The full complement of underground equipment is listed in Table 16.9.

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Table 16.9 Underground Mobile Equipment List

Unit	Quantity
Production 4.5" ITH drill	1
Production Top hammer drill 2" to 3.5"	1
ST7 LHD	3
30t Haul Truck	3
Development Jumbo	2
Mechanical bolter	2
Shotcrete spray carrier	2
Scissor Lift	2
Fuel - Lube truck	1
Grader	1
Forklift	1
Mechanics Vehicles	2
Personnel Vehicles	3

16.3.10 Underground Manpower

Manpower estimates for the underground mining operation is approximately 110 mine contractors and up to 16 company staff. These figures include mine supervision, management, engineering and geology staff. Table 16.10 shows proposed manpower and rotation schedule.

Table 16.11 shows the total underground manpower by year over the project life.

Table 16.10 Summary Company - Contractor Manpower

Underground manpower	#/day	#/night	Total	Schedule
800 MT/day				
Administration				
Project manager	1	0	2	dayshift 14/14
Engineering - Geology	4	2	12	dayshift 14/14
Clerk/buyer	1	0	2	dayshift 14/14
HR				
Safety & Training Coordinator - contractor	1	0	2	dayshift 14/14
U/G Supervision				
Captain - contractor	1	0	2	dayshift 14/14
Supervisor - contractor	2	2	6	day/night shift 28/14
Sub Total			**26**	
Service - General				
Construction - Timber man - contractor	2	2	8	dayshift 14/14
Service man contractor	1	1	3	day/night shift 28/14
Subtotal			**11**	

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Underground manpower	#/day	#/night	Total	Schedule
U/G mechanical services and fixed equipment				
Master mechanic - contractor	1	0	2	dayshift 14/14
Mechanical - contractor	2	2	6	day/night shift 28/14
U/G and surface electrical services				
Master electrician - contractor	1	0	2	dayshift 14/14
Electrician - contractor	1	1	3	day/night shift 28/14
Surface mechanical services				
Surface Labourer				day/night shift 28/14
Machine doctor - contractor	1	0	2	dayshift 14/14
Subtotal			**15**	
U/G Crews				
Dev Crew A - contractor				
Jumbo man	1	1	3	day/night shift 28/14
Miner	2	2	6	day/night shift 28/14
Shotcrete	4	4	12	day/night shift 28/14
Mucker	1	1	3	day/night shift 28/14
Dev Crew B Production Cut and Fill - contractor				
Jumbo man	1	1	3	day/night shift 28/14
Miner	2	2	12	day/night shift 28/14
Shotcrete	4	4	12	day/night shift 28/14
Mucker	1	1	3	day/night shift 28/14
Subtotal			**54**	
Production Crew - contractor				
Longhole Driller	1	1	3	day/night shift 28/14
Longhole C/B Driller	1	0	2	dayshift 14/14
Blasters	2	0	4	dayshift 14/14
Cable Bolters	2	0	4	dayshift 14/14
Muckers	1	1	3	day/night shift 28/14
Sub Total			16	
Alimak Raise Crew - contractor				
Raise miners	2	0	4	day/night shift 14/14
Subtotal			**4**	
Total			**126**	

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Table 16.11 Life of Project Manpower

Position	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Management	10	14	14	14	14	14
Engineering, Geology	8	12	12	12	8	8
Underground Services	11	11	11	11	11	3
Underground Mechanical -Electrical	6	15	15	15	6	6
Development	12	12	0	12	12	0
Production Included C&F Dev	0	36	59	29	16	16
Alimak Raising	4	0	0	4	4	0
Total	**51**	**100**	**111**	**97**	**71**	**47**

16.4 Open Pit

16.4.1 Introduction

The 41 deposit, WGH and the trench in deposit 51 would be mined by open pit method. Based on the limited geotechnical data available the overall pit slope would be at a maximum of 48°. The 41 deposit is less than 1.0 km from the proposed mill site and would be mined first for mill feed. The 51 trench and the WGH pit would follow the 41 pit. The WGH is approximately 3.5km from the mill. A steel bridge would also be required to cross the Isle Aux Morts River to access the WGH pit. Refer to the Section 18 mill site and pit locations.

16.4.2 Cut-off Grade

Cut-off grade criterion represents the breakeven point of costs compared to revenue. As such the cut-off criterion varies over the life of the project with variations of costs and revenues. The following table provides a calculation of breakeven cut-off grades for direct operating costs. The engineered pit designed for this estimate was based on an optimized pit shell using a gold price US$ 1,200 oz. and silver price was US$ 14/oz. with the exchange rate of $1.25. Table 16.12 shows the block cell value and cut-off grade calculation.

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Table 16.12 Block cell value and cut-off grade calculation

Parameters	
Block Cell Dimension (m)	3x3x3
Block Cell Volume m^3	27
Block Cell Density	2.84
Block Cell tonne	76.68
Mine Recovery	95%
Mining Dilution	10%
Mill Recovery	97%
Au Price $/gm	47.15
Mining Cost/tonne (PMR)	$8.19
Mining Cost/tonne (waste)	$2.92
Milling Cost/tonne	$27.95
Processing Cost / Block cell	$2,240
Mining Cost / Block cell	$691
Au grade (g/tonne)	0.88
Au (gm)	62.2
Revenue	$2,931
Net Value	$1.4
Net Revenue $/tonne	5.23
Cut-off grade used g/t	**1.0**

16.4.3 Ultimate pit optimization

The ultimate pit design for the 41 deposit and WGH was carried out using the block model files ik_cr51_mod and id_crwg_utm. The block model files were transferred into the Maxipit® ver 4.24.75.0 software package. The model was checked, including a review of the block grades to ensure the integrity of the transfer. The gold and silver grade data was provided in the resource block model and are the main items used for pit-shell optimization. The block size for the model is 3 x 3 x 3 metres. The following data in the block model were utilized.

- Gold and Silver grade g/t
- Resource category (Indicated and Inferred)
- Density
- FILLVOL – volume of the mineral zone in the block.
- VOIDVOL – volume of the block outside the mineral zone.

The development of the ultimate pit design for the deposit 41 and WGH was carried out in two consecutive steps as follows:

- Step 1: Determination of the economic pit-shell using the Maxipit ® pit optimizer, based on operating costs and revenue data along with pit slope requirements; and
- Step 2: Preparation of the ultimate pit design in the CAE Studio 5D Planner® ver 14.24.49.0 by incorporating in the pit shell the final ramp and proper pit slope, as well as the benching arrangement.

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16.4.3.1 Pit shell parameters

The pit shell optimization process uses the LG 3D algorithm in Maxipit ® and is considered as the standard routine in the industry. The pit optimizer generates an optimal economic pit-shell from a 3D block model. The basic optimization principle of the algorithm operates on a net value calculation for the block in the model, i.e. revenue from sales less total operating costs. The combination of these values and slope requirements allow the optimizer to search for the pit shell with the highest undiscounted cash flow.

The general overall pit-shell slopes used in the optimization was 48° in the hanging wall and footwall. Since the pit optimization program does not produce an ultimate pit design with a final haul ramp, geotechnical berms and a proper benching arrangement, angles are taken to be slightly steeper than the final pit design slopes, to ensure that the ultimate pit design slopes are respected. Overall 95% mine recovery and 10% dilution was added during the pit shell optimization

16.4.3.2 Ultimate Pit Design Parameters

In order to maximize the recovery inside the ultimate pit design the optimized pit shell obtained from the Maxipit® optimization process was used as main guides. The resulting ultimate pit design includes practical geometry that is required in an operational mine, such as the haul road to access all the benches, recommended pit slopes with berms, proper benching configuration and smoothed pit walls. The bench face angles was 70° and the inter ramp angles vary between 42° to 48°. The resulting berm widths created by the slope configuration in each sector was 10 metres wide. The final pit was designed using a double benching configuration, with a berm every 18 metres of vertical height. The pit design was expanded at every 9 metres to match the height of the blocks in the resource model.

The final ramp was designed to a 12 metre width in order to support haulage cycles using the expected truck widths. This is based on the industry standard for the running width of a haul road to be approximately three times the width of the largest equipment, not including additional allocation for the drainage ditch and safety berms. For the last stretch of the ramp near the pit bottom, the haul road is steeper in order to reduce the stripping ratio. The ramp gradient used was 12.5%. Figure 16.10 and Figure 16.11 show cross-section and longitudinal section of the pit and block model. Figure 16.12 displays 3D view of pit.

Table 16.13 Ultimate pit design parameters

Parameters	
Rock Face Slope Angle	70 °
Rock Inter-Ramp Angle	42°-48°
Bench Height	9m
Bench Arrangement	Double benched
Ramp Width	12
Haulage Ramp Grade	12.5%

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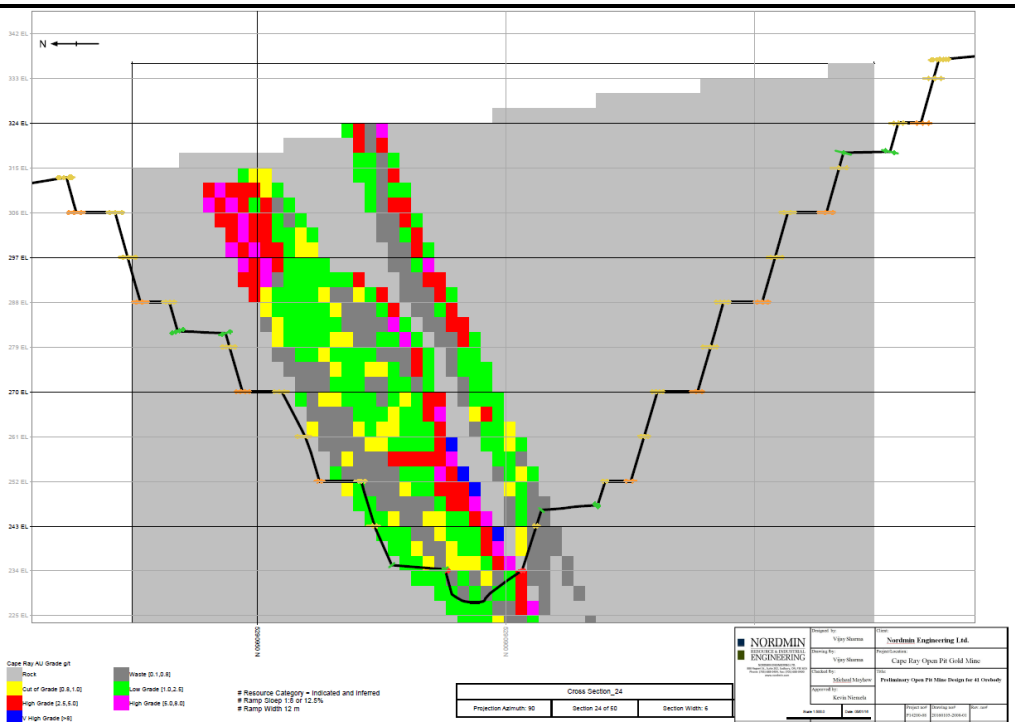

Figure 16.10 Cross section of deposit 41 pit design and block model

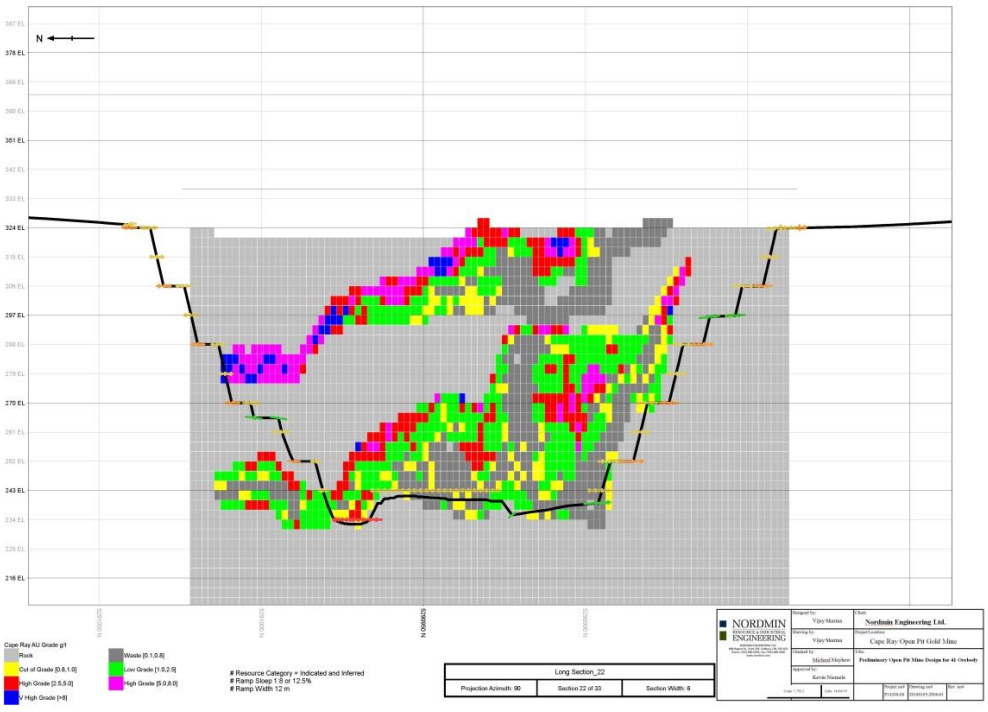

Figure 16.11 Longitudinal section of deposit 41 pit design and block model

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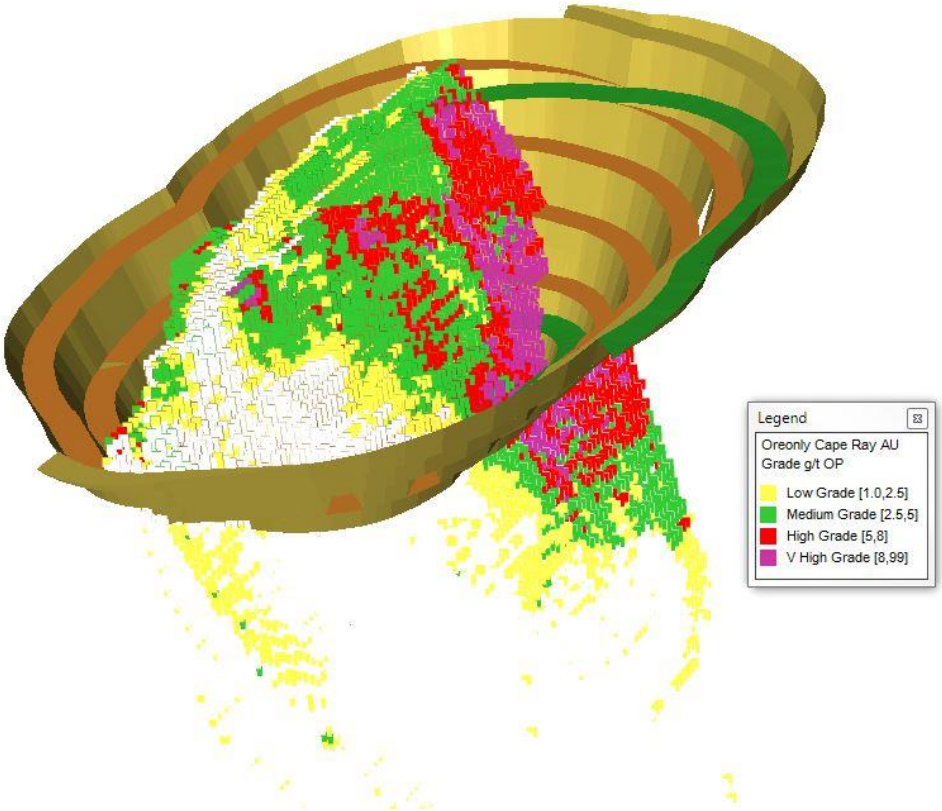

Figure 16.12 Ultimate Pit Design in 3-Dimensional View with Block Model

Table 16.14 Pit Design and potential resources for deposit 41, WGH and a trench in deposit 51 (in thousands)

Deposit	Mining Method : Cut-off Grade	Resource (indicated + inferred)			Potential mineable resource				
		Tonne	Au oz.	Ag oz.	Tonne	Au oz.	Ag oz.	Waste	Stripping Ratio
41	Open Pit: 1.0 g/t	734	69	226	503	50	167	6,362	12.65
	Underground: 4.0 g/t	166	33	101	39	8	1	-	
51	Trench: 1.0 g/t	881	144	335	5	2	6	26	5.37
	Underground: 4.0 g/t	397	107	209	347	81	159	-	
WGH	Open Pit: 1.0 g/t	878	59	286	328	24	63	1,597	4.86
	Underground: 4.0 g/t	70	13	50	-	-	-	-	
Total	Open Pit: 1.0 g/t	2,493	272	847	836	77	236	7,985	
	Underground: 4.0 g/t	634	153	360	386	89	160	-	

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16.4.4 Open Pit Operations

The proposed open pits are conventional operations that use demonstrated technology and equipment for drilling, blasting, loading and haulage. A mine contractor would supply the needed manpower and equipment for the open pit operation. The pit contractor would also manage and supply all explosives and blasting accessories. The pit contract crews and equipment would also construct the haul road.

Due to the extreme snow fall conditions in this area it is envisaged that open pit mining operations would be limited to 6 months of the year. The anticipated open pit production rate would be approximately 1,500 t/d, with muck transported to the mill and surface stockpile. During the winter months mill feed would come from underground and the surface stockpile.

Table 16.15 shows the proposed contractor open pit equipment:

Table 16.15 Open pit Equipment

Unit	#
40t haul trucks (CAT 770)	5
excavators (CAT 345)	2
loader (CAT 990)	1
drills (Sandvik DR540)	2
dozers(CATD8)	1

16.4.5 Open Pit Infrastructure

16.4.5.1 Dewatering

Conventional sumps and pumps would be used to dewater the pits during the warmer weather months. The pit water would be stored in a collection pond and treated prior to discharge if required.

16.4.5.2 Water Management Infrastructure

All open pit sites require water management infrastructure. This includes construction of collection ponds, ditching to collect contact water from the site, and berms to divert outside run off from entering the sites. Contact water would be directed to collection ponds by gravity feed and ditching. These collection ponds would allow sediments to settle. Water would be tested before being released into the environment or treated if required.

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16.4.5.3 Waste Dump Designs

It is assumed for this PEA that all waste rock from open pit mining is potentially acid generating (PAG) material and would be dealt with accordingly. The 51 trench waste, 25,900 tonnes, would be hauled to the 41 pit waste dump. Overburden material would be segregated from the waste rock and stockpiled for mine closure.

16.4.6 41 Pit and WGH Pit

The 6.10 Mt of waste material mined from the 41 pit and 51 trench would result in 2.79M loose cubic meters (LCM) of waste material to be placed in the waste dump east of the 41 pit. The 1.92 Mt of waste material mined from the WGH pit would result in 0.88M LCM of waste material to be placed in the waste dump west of the pit. Refer to Section 18 for the envisaged waste dump location relative to the respective pit.

Stockpiling of PMR material on a pad at the mill sites is envisaged. The PMR stockpile and waste dumps would be designed to tie in with a water collection pond.

16.4.7 Open Pit Manpower

Table 16.16 Open Pit Manpower

Description	Position	Men/ Day
Superintendent	Surface G&A	1
Blasting Foreman	Surface G&A	1
Maintenance Superintendent	Surface Maintenance	1
Total Mine Salaried		**3**
Drillers	Drilling	2
Blaster	Blasting	1
Blaster Helper	Blasting	1
Total Drilling and Blasting		**4**
Excavator Operator	Loading	2
Total Loading		**2**
Truck Driver	Haulage	6
Total Hauling		**6**
Dozer Operator	Roads & Dumps	2
Grader Operator	Roads & Dumps	1
Utility Driver	Roads & Dumps	2
Total Support		**5**
Lead Mechanic	Surface Maintenance	1
Heavy Equipment Mech.	Surface Maintenance	4
Light Vehicle Mech.	Surface Maintenance	2
Total Mine Maintenance		**7**
Grand Total Mine Operations		**27**

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17. Recovery Methods

17.1 Conceptual Process Flowsheet Summary

Run of mine PMR would be delivered to the mill feed stockpile at a rate of 1000 t/d for six to seven months of the year from the open pit while underground production would continue throughout the year. The proposed processing rate is 36 t/h or 850 t/d with the mill operating at 92 % availability 365 days per year. ROM (Run of Mine), at– 203.2 mm, is first reduced to –25.4mm in a single jaw crusher. Further size reduction to–12.7mm is achieved in a series of two gyratory cone crushers operating in closed circuit with a double deck screen. The crushed product is stockpiled ahead of the grinding circuit and is then fed to grinding via a primary grinding mill feed conveyor.

Size reduction from – 10 mm to 80 % passing 100 microns is achieved in the grinding circuit. Crushed product is fed to two parallel primary ball mills at 36 t/h. Primary mill discharge is directed by a screen and the screen underflow reports to a pair of gravity concentrators for recovery of any coarse free gold. The screen oversize reports back to the primary mill feed. The gravity concentrator tailing reports to the secondary grinding circuit for further size reduction before reporting to the CIL circuit thickener.

Prior to cyanide leaching of the contained gold in the PMR, the pulp is thickened to 45 % solids by weight using the CIL pre-leach thickener. The thickened pulp is advanced in the CIL circuit at the full process flow rate of 36 t/h solids. In the CIL circuit gold is recovered from the solids using whole cyanidation and the gold is transferred to the surface of the activated carbon. The activated carbon is separated from the main pulp flow by a combination of pumping and screening. The loaded carbon is washed with hydrochloric acid solution to remove the carbonates. Gold is then removed from the loaded carbon using pressure stripping. The gold is transferred from the surface of the loaded carbon to solution and from there is recovered to marketable form using the electrowinning process.

The stripped carbon is regenerated in a reactivation kiln before being reintroduced to the process. Fine carbon is constantly eliminated (and recovered) from the process to avoid gold loss, with fresh carbon being continuously added to the process.

The cyanide contained in the tailings from the CIL circuit is eliminated in a cyanide destruction tank with SO_2-air process. Once the cyanide is destroyed, the tailings are transported to the tailings pond for disposal.

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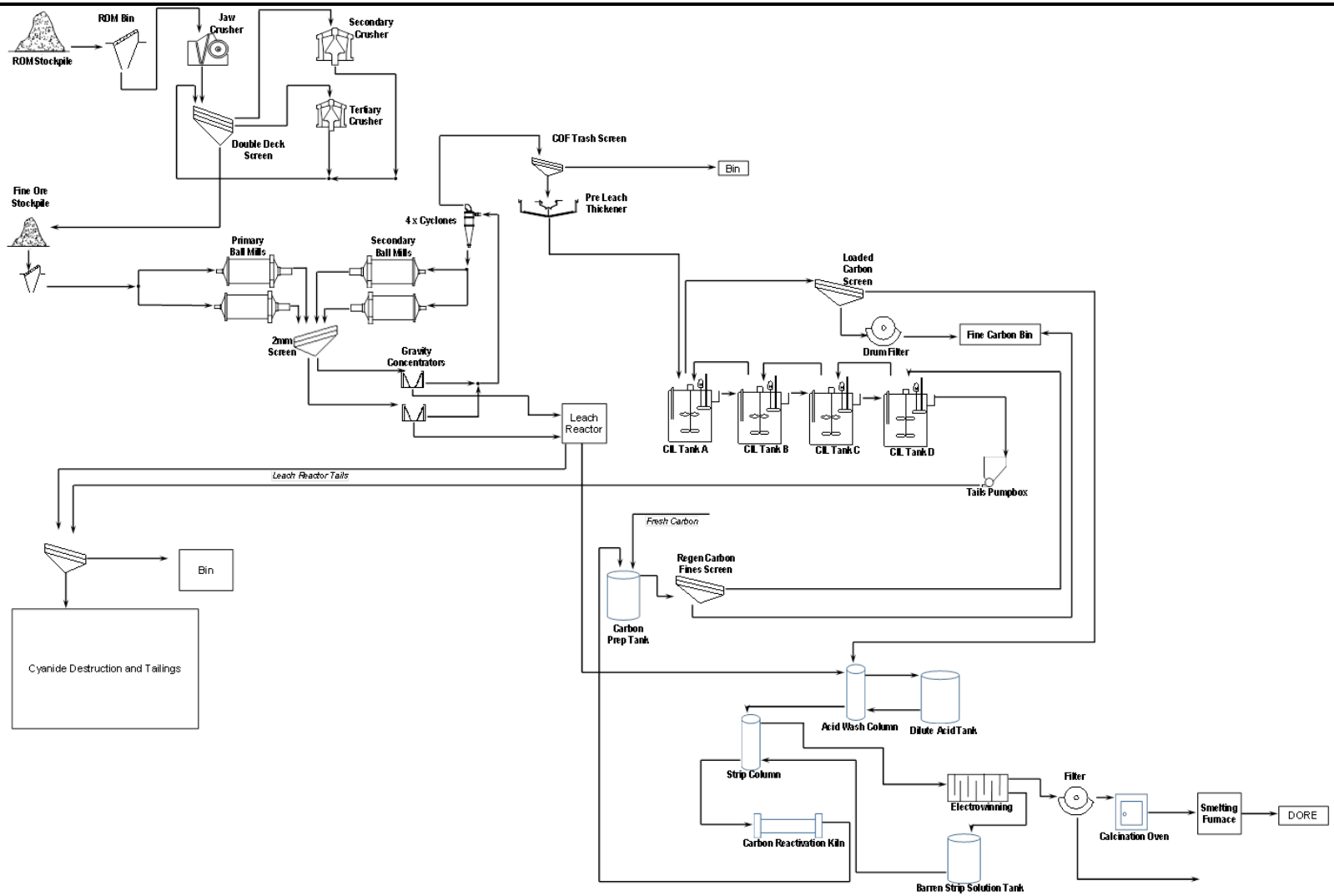

Figure 17.1 Conceptual Flowsheet

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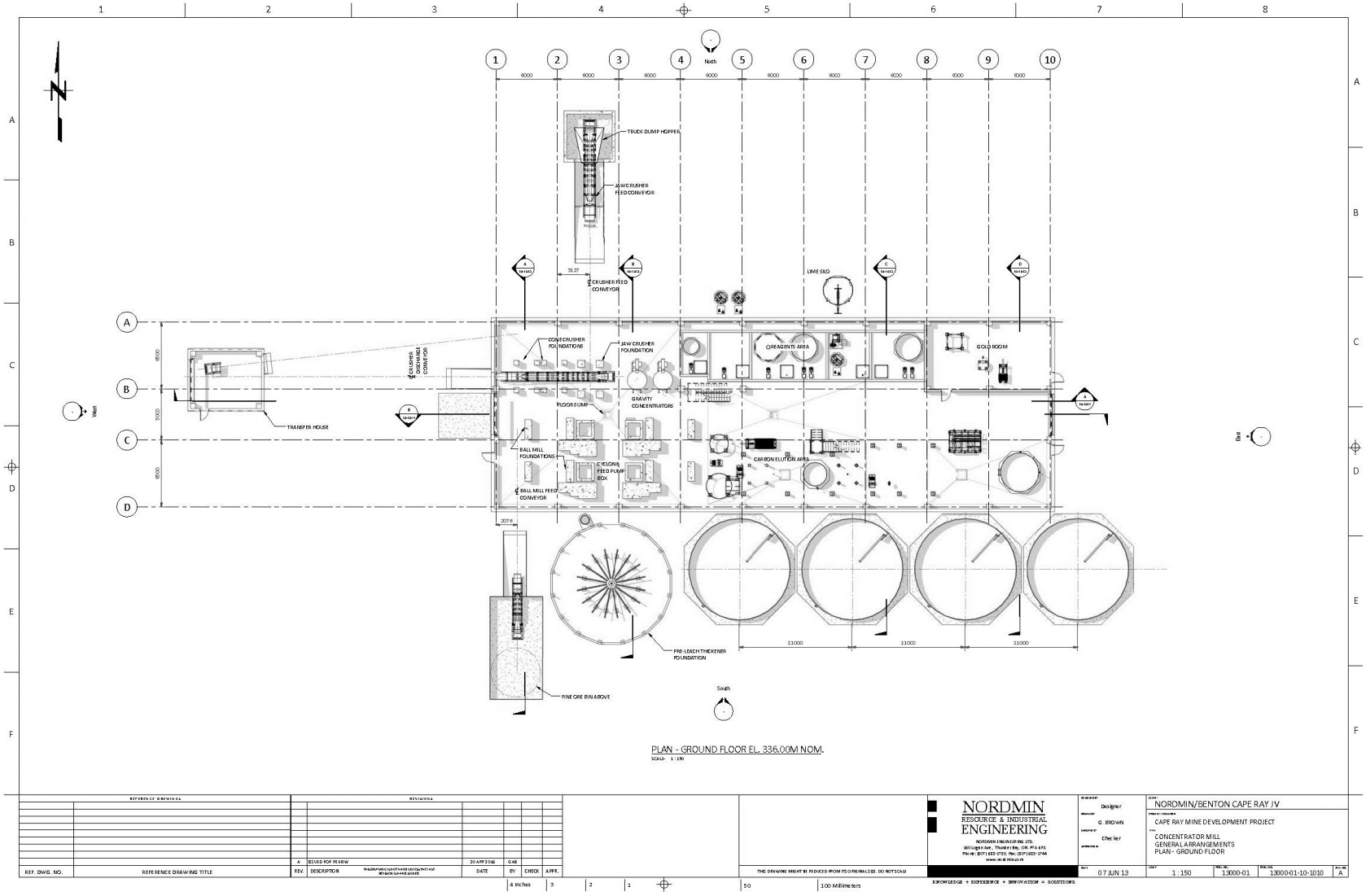

Figure 17.2 Concentrator Mill General Arrangements Plan -Ground Floor

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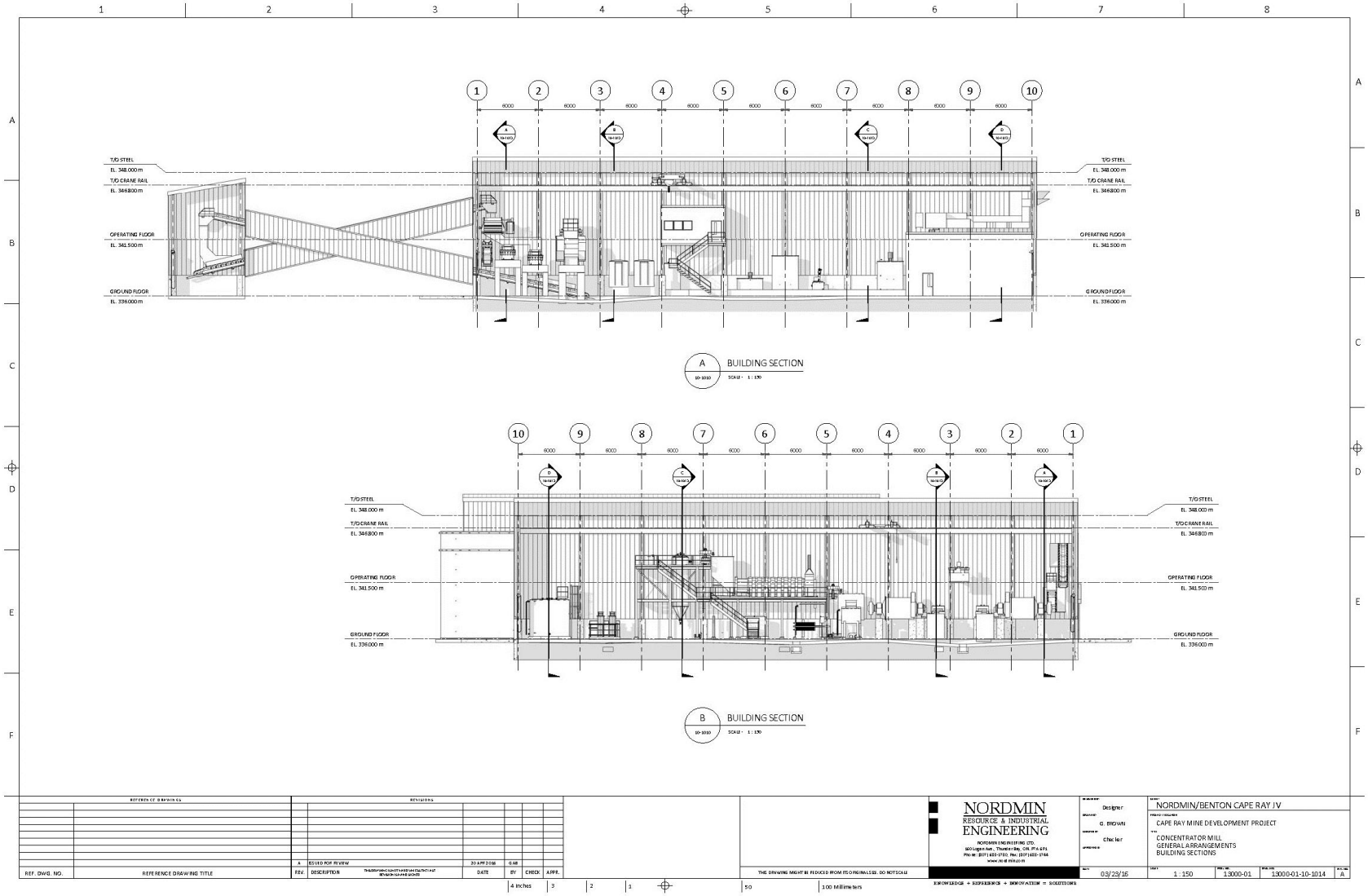

Figure 17.3 Concentrator Mill General Arrangements Building Sections

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17.2 Process Description

17.2.1 Material Handling

Run of mine will be trucked to the mill site which will be in the rage of one to three kilometers. It will be stockpiled through the summer months and processed year round. The crushing circuit was designed with a nominal 76tph capacity, allowing for further flexibility and a de-coupling between crushing and processing plant.

Primary crushing will comprise a 20 tonne capacity feed hopper with 42" vibrating grizzly feeder and 30'x42' jaw crusher driven by 150 hp motor and will be fed by a front end loader.

Crushed PMR will drop on to a 36" conveyor and travel under a cross-belt magnet for tramp metal removal and over a single idler belt scale for monitoring the processed tonnage. The jaw crusher product will be fed to a double deck 5'x12' inclined vibrating screen.

Screen oversize will report to a secondary crusher, intermediate product will report to a tertiary crusher. The screen undersize, (-10mm) will be stockpiled outside the processing plant in an uncovered pile. The secondary crusher will operate in open circuit and the tertiary crusher will operate in closed circuit with the vibrating screen. The discharges from all three crushers will be conveyed back to the screen feed conveyor. A Sepro Blackhawk 100 crusher with 100 kW direct drive motor has been recommended for both secondary and tertiary crushing duty.

17.2.2 Grinding and Thickening

The fine PMR surge pile reclaim conveyor will discharge onto a fixed speed primary grinding feed conveyor. The conveyor will have a single idler belt scale and discharge into a splitting discharge chute to feed material to the two primary ball mill feed conveyors.

The grinding circuit will have a total of four 2.1mx 6.0m, 330 kW drive, tyre-mounted grinding mills. Two primary mills will operate in parallel with two more identical mills for secondary grinding duty. The primary mills will be in open circuit and the secondary mills in closed circuit with classifying cyclones. The discharge from all four mills will be pumped to a 2mm scalping screen. Oversize material will be returned to the primary ball mills and the fines will report to two Falcon SB 1350 gravity concentrators.

The gravity tailings will be pumped to the cyclones. The cyclone overflow, with a target p80 of 100µm will report to a trash screen to remove wood chips, plastic etc., and to a primary thickener ahead of the CIL circuit. The cyclone underflow will be split two ways and delivered by gravity back to the two secondary mills.

The gravity concentrate will drop into a secure containment and will be pumped to the Sepro SLR 3000 intensive leach reactor where it will be subject to leaching under high cyanide and dissolved oxygen. Clear pregnant leach solution from the SLR will be pumped to the secure electrowinning circuit located in the elution section of the plant. Tailings from the SLR will be

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returned to the secondary mill discharge sump. In this configuration, the entire ball mill discharge stream can be processed through the gravity concentrators.

The tails underflow target is 35% solids by weight at a pH of 10.5 reporting to the CIL circuit. The pulp pH is controlled by lime addition in the grinding circuit and additional lime added in the CIL as required. .

17.2.3 Carbon-in-Leach

The thickened slurry from the preleach thickener will be pumped to a series of four 7m diameter by 12m high agitated tanks, providing 30h residence time. The inter-tank flow will be by gravity launders equipped with inter-stage screening to maintain the counter current flow of activated carbon

Leaching is achieved with sodium cyanide (NaCN) which will be added at the front end of the leach circuit. pH will be monitored throughout the leach and absorption circuit. Manual sampling will be used to monitor and control free cyanide levels in the leach circuit.

The underflow from the pre-leach thickener is pumped to the first of four CIL tanks. Slurry from this tank overflows into the second and flows by gravity through the circuit. Cyanide solution and lime are added, as required, for gold dissolution and pH control. Gold in the solution is adsorbed onto the activated carbon. Slurry from the last CIL leach tank passes over a safety screen prior to cyanide destruction. Air is injected in the bottom of each tank for gold dissolution. Interconnecting tank launders are arranged so that any tank in series can be bypassed without having to shut down the entire CIL circuit

Loaded carbon is pumped counter current to the slurry flow through the CIL tanks in order to increase gold loading. The carbon-forwarding pump of the first CIL tank transfers the slurry on to the loaded carbon screen to recover the loaded carbon from the slurry. The undersize from the loaded carbon screen is filtered and sold to recover the contained gold. Fresh and regenerated carbon is added into the last CIL tank.

17.2.4 Elution and Carbon Reactivation

Loaded carbon recovered by the screen reports to the acid wash column of the elution circuit. The acid solution is prepared in the dilute acid tank and the pumped through the acid wash column. The spent acid is then neutralized with caustic. The carbon is transferred from the acid wash column to the strip column for gold desorption. During desorption the gold is transferred from the surface of the activated carbon into the strip solution. This process is carried out at elevated temperature and pressure and is normally a 8 - 12 hours cycle. A series of heat exchangers and a heater in the barren strip solution tank are used to bring the strip solution to operating temperature before entering the strip column. The pregnant solution then goes to the electrowinning cells in the refinery for gold recovery.

The stripped carbon is drawn from the strip column and reports to the carbon reactivation kiln. After reactivation, the carbon is discharged into the carbon pre-preparation tank. The carbon

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from the carbon pre-preparation tank is pumped and screened out to remove and recover fine carbon and then drops by gravity to the last CIL tank. Fresh carbon is added in the carbon pre-preparation tank on a regular basis to compensate for the fine carbon removal.

17.2.5 Electrowinning and Refinery

The pregnant solution from the strip column flows first by gravity to the electrowinning flash tank and then the electrowinning cells, where the gold is plated on cathodes. The barren solution from the electrowinning cells is recovered in a pump box and pumped back to the barren strip solution tank in the carbon elution circuit.

In the electrowinning cells, the stainless wool cathodes are cleaned with high pressure water and the gold sludge sinks to the bottom of the cells. The gold sludge is then pumped to a filter-press to remove the excess water. The filtrate is pumped to the Tank A launder in the CIL circuit.

The filtered gold sludge is sent to the calcination oven to remove excessive humidity. The dried gold sludge is then mixed with flux and is fed to the electric induction furnace. Once the gold is melted, it is poured into the doré moulds. The doré bar is recovered for shipment.

17.2.6 Cyanide Destruction

Slurry from the last CIL tank reports across a safety screen (size) in order to capture any carbon that might inadvertently escape through the last interstage screen and report to tailing. The carbon is recovered at the oversize and into a bin.

The screen undersize flows by gravity into the cyanide destruction tank feed pump box and is pumped to the cyanide destruction tank. Air and liquid SO2 is added at the bottom of the cyanide destruction tank. Once cyanide destruction is complete, the tailings are discharged in to a buffer tank for filtering before disposing in the tailings containment area.

17.2.7 Tailings Management

Tailings from mill operations (after CN destruction process) will be handled by conventional methods and disposed of in an engineered Tailings Management Facility (TMF). The tailings and waste water will be pumped to the TMF via a slurry pipeline at 35% solids.

The TMF supernatant pond and waste water treatment plant will be designed to withstand a 30 day duration, 1 in 100 year rain on snow event. The TMF will be equipped with an engineered spillway to prevent a potential dam failure due to overtopping during a Probable Maximum Precipitation (“PMP”) event.

The preliminary TMF design has storage capacity of 1.2M cubic metres of tailings solids, with the possibility to increase in the future by raising the height, concurrent with any requirements for amendments to necessary approvals. The TMF surface footprint is not anticipated to be materially changed as a result of any future raises. The TMF will be constructed, operated, maintained and monitored in accordance with Guide to the Management of Tailings Facilities

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(MAC, 1999) and Environment Canada (2009).

At closure, the TMF and plant sites will be covered with an engineered, low-permeability dry cover that will minimize infiltration of water. Seepage collection around the TMF will continue post closure until the seepage decreases and no longer poses a risk to the environment.

17.2.8 Reagents

Sodium Cyanide (NaCN) is used for gold leaching in the CIL circuit and for carbon stripping in the carbon elution circuit. This will be supplied in tote bags and will be mixed on site.

Flocculent: used in the thickeners to aid in settling rates. This will be supplied in tote bags and mixed on site.

Hydrochloric acid (HCl) is used for carbon acid wash. This will be supplied in drums and pumped to a storage tank.

Sulphur dioxide (SO_2) is used as an oxidising agent in the cyanide destruction process. This will be delivered by 1 tonne totes.

Lime is used to control the pH in the grinding circuit, CIL and cyanide destruction circuit. It will be delivered in bulk and stored in a lime silo on site.

Copper sulphate ($CuSO_4$) is used as a catalyst in the cyanide destruction process. It will be delivered in totes and mixed on site.

Sodium Hydroxide (NaOH) is used for carbon stripping and after the carbon acid wash to neutralize the residual acid in the dilute acid tank and the acid wash column. This will be supplied in drums.

Anti-scalent will be used to reduce the calcium carbonate buildup in the water system. This will be supplied in drums and added to the process water tank.

Grinding balls will be delivered in totes or large containers for use in the ball mills. 3.5” balls will be used in the primary mills and a mixed ball charge of 1.25” and 2” will be used in the secondary mills.

The two cone crushers will require liners. Standard liner configuration will be required for the secondary crusher and shorthead standard configuration of liners for secondary crushing.

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17.3 Utilities

Utilities required will include a fresh water system; reclaim water system from the tailings impoundment and process water.

Potable water will be required for emergency showers and washroom facilities and fire system.

Compressed air is required for the CIL circuit, and cyanide destruction circuit as well as instrumentation.

17.4 Capital and Operating Costs

Based on the test work completed on the deposits, Table 17.1 describes considerations used for circuit design.

Table 17.1 Circuit Design Parameters

Design Parameters	Value	Units
Rock Hardness	14.5	kWh/t
Ball Mill Feed Size k80	19	mm
CIL Circuit Feed Size k80	100	µm
CIL Circuit Feed Density	45	%
CIL Residence Time	30	h
CIL Au Recovery	97	%
CIL Ag Recovery	45	%
Gravity Au Recovery	25	%
Gravity Ag Recovery	5	%
Overall Au Recovery	97	%

The equipment was selected based on 850 tonnes per operating day and an availability of 92%.

Capital costs are broken out by equipment required for each area in the processing plant and associated construction costs.

The flowsheet was developed on gravity and bottle roll leach testing. There was no filtration test work completed on tailings or effective cyanide destruction values. Carbon absorption studies were also not conducted. Further exploration for proper equipment sizing would be required to move forward with this project.

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Table 17.2 Equipment Included in the Capital Costs (in thousands)

AREA	UNIT DESCRIPTION	QUANTITY	$US
Crushing			1,967
	42'x17' vibrating grizzly feeder	1	
	3042 Jaw crusher; drive motor, frame and control tower	1	
	Discharge conveyor 36"x35' long; rollers and magnet	1	
	Screen feed conveyor; 30"x 60'long;	1	
	Skid mounted 5'x12' double deck screening plant and chutes	1	
	Screen undersize conveyor; 36"x20'long;	1	
	Secondary cone crusher feed conveyors; 30"x 30' long	2	
	Tertiary cone crusher feed conveyor; 30"x 60' and chutes	1	
	Blackhawk cone crusher - secondary crusher duty	1	
	Blackhawk cone crusher - tertiary crusher duty	1	
	Crusher discharge conveyor; 30" x 50'	1	
	Recycle conveyor; 30"x40'	1	
	Surge pile stacking conveyor 30"x100'	1	
	Reclaim system from surge pile	1	
Grinding and Gravity			3,207
	Primary feed conveyor; 30"x60'	1	
	Ball mill feed conveyors; 30"x40'	2	
	Primary ball mills	2	
	Primary ball mill discharge pumps	2	
	Vibrating screen deck - 5'x12'	1	
	Sepro Falcon SB1350 SFC - semi batch concentrators	2	
	Cyclone feed pumps	2	
	Hydrocyclone cluster (4) and splitter box		
	Secondary ball mills	2	
	Secondary ball mill discharge pumps	4	
	Sepro SLR3000 Leach Reactor; pumps; control panel	1	
	Oxygen generation plant	1	
Cil plant			2,550
	Trash screen; 4'x12'	1	
	Pre-Leach thickener	1	
	Thickener UF pumps	2	
	CIL tanks and agitators	4	
	Vibrating screens 4'x8': carbon sizing, loaded carbon and carbon safety screen	3	
	CIL transfer pumps	2	
	Cyanide destruction agitators	2	
Reagent Prep Area			282
	Mixing Tanks (CN and Lime Addition)	2	
Tailings Pumping Station			474
	Pumps and Tank	2	
Assay lab			750
Carbon Batch Containerized Gold Desorption And Recovery Plant			2,616
	1 ton loaded carbon management system: acid wash, carbon elution, electrowinning, gold room, carbon regeneration and PLC and security	1	
Engineering and Design Package			2,008
	Subtotal $USD		**13,854**
	CAD:US Exchange		**0.75**
	Total CAD$		**18,472**

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Table 17.3 Construction and Engineering Capital Costs (in thousand)

Construction and Engineering Services	Cost ($CAD)
Mining and Area Prep	1,100
Civil/Structural Installations	1,985
Power Connection to Mill	1,200
Mechanical Installation	2,350
Electrical and Instrumentation	1,250
Contractor Indirect Costs	788.5
Owner Indirect Costs	591.4
Commissioning	263.6
Contingency (0%)	0
Total for Services and Infrastructure	**9,528**

Total capital for the mill equipment has been estimated at $ 18,472,200 and construction and engineering services has been estimated at $9,528,447 for a total of $28,000,647 capital cost for the processing facility. There is no contingency included in this estimate, as the economics of the project have an over-arching contingency in place.

Table 17.4 Consumable Operating Costs (in thousand)

Consumable	Consumption Rate	Unit	Unit Cost CAD$	Unit	Annual Cost	Monthly Cost
3.5" Balls	0.6	kg/t	1500	$/tonne	257	21
Mixed Ball (1.25" & 2")	0.5	kg/t	1500	$/tonne	214	18
Crusher Liners	0.2	kg/t	1750	$/tonne	75	6
Ball Mill Liners	0.8	kg/t	800	$/tonne	173	14
Cyanide	2.0	kg/t	4.375	$/kg	2,494	208
Lime	1.0	kg/t	0.25	$/kg	71	6
Caustic	0.075	kg/t	0.6875	$/kg	15	1
Carbon	0.07	kg/t	4.375	$/kg	87	7
Safety & Mtc Supplies	1		1.5	$/tonne	428	36
Power	59	kWh/t	0.085	$/kwh	1,429	119
Tailings storage	850	t/d	0.35	$/tonne	100	8
Total annual cost					**5,342**	**445**

Cost per tonne	$18.74
Daily operating tonnage	850 tonne per op day
Annual operating tonnage	285,000 Avg. tonne per year

The operating cost per tonne is proposed to be $18.74, not including manpower.

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17.5 Manpower

The concentrator will be set up for operating 24 hours per day; requiring four shifts of operators. The functionality of the crews will include three operators, a millwright and an electrician and an assayer for each crew. There will be a metallurgist, seven days a week on day shifts only. Administration would work 40 hours per week on a Monday – Thursday schedule and would consist of a superintendent, maintenance and electrical foreman as well as a clerk.

Table 17.5 Manpower Requirements for Mill Operation

	Number	Hr/Shift	Shift/Mo	Base $/yr.	Base $/hr	Burden %	Burden $	TOTAL $/yr.	Total $/mo.
Mill Administration									
Mill Superintendent	1		21	140,000		20	28,000	168,000	14,000
Maintenance Foreman	1		21	105,000		20	21,000	126,000	10,500
Electrical Foreman	1		21	105,000		20	21,000	126,000	10,500
Metallurgist / Supervisor	1	12	15	75,000		20	15,000	90,000	7,500
Assayer	1		15	55,000		20	11,000	66,000	5,500
Clerk	1		21	48,000		20	9,600	57,600	4,800
Operations									
Lead Hand / Concentrator Operators	4	12	15		32	20	55,296	331,776	27,648
Crusher Operators / Loader	8	12	15		28	20	96,768	580,608	48,384
Leach/CIP Operator	4	12	15		28	20	48,384	290,304	24,192
Maintenance									
Millwrights	2	12	15		34	20	29,376	176,256	14,688
Electricians	2	12	15		34	20	29,376	176,256	14,688
Total	**26**							**2,188,800**	**182,400**
Cost per tonne	285,000	tons						7.68	$/tonne

Manpower adds an additional $7.68/t on top of consumable operating cost of $18.74/t provides a total of $26.42/t.

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18. Project Infrastructure

Currently there is no permanent infrastructure at the Cape Ray Property. There is evidence of previous exploration work including drilling, trenching, and bulk sampling via a portal and drift at the 41 Deposit. Electrical and telephone lines do not extend to the Cape Ray Property with the nearest high voltage electrical substation located approximately 25 km southwest in the community of Channel-Port aux Basques Figure 18.1

Presently, mobile portable camp trailers on the Cape Ray Property require the use of gas generators for electricity and propane for heat and cooking. Due to the small quantities required, fuel is brought to site in portable tanks via the access road. To support activity on the property, a sewage system and water supply system is required along with the appropriate certificate of approvals.

All areas covered by the PEA presented in this technical report occur on crown land owned by the Province of Newfoundland. The development of any future mining operations in this area must follow provincial permitting requirements.

The deposit area is surrounded by an abundance of barren land that is not populated except for a few seasonal cottages within the claim boundary. Considering the local economy and the ideal location, it is anticipated that the Newfoundland government would be favorable to the development of a new mining project, provided all of the environmental, socio-economic, and permitting issues were addressed properly.

The abundance of surface water resources from lakes on the Cape Ray Property is a positive factor for future development, as is the presence of deep water wharf facilities and commercial shipping infrastructure at nearby Channel - Port aux Basques (<50 km driving distance) (Ténière & Hilchey, 2012).

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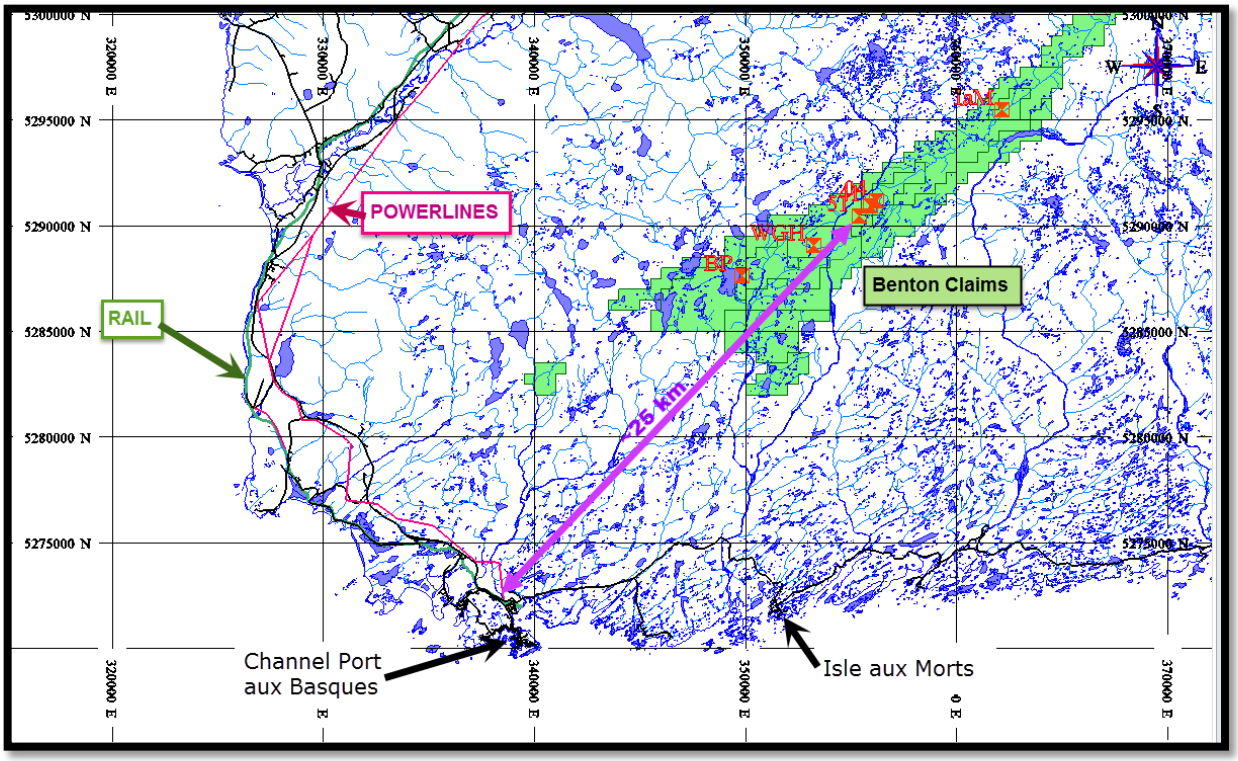

Figure 18.1 Cape Ray Property nearby Infrastructure

The main infrastructure required for the Cape Ray project includes: (Figure 18.2 Mine Site and Complex.)

- Processing Plant (Mill)
- Tailing Management Facility (TMF) with Treatment plant and CN Destruction plant.
- Road and culvert upgrades
- BRIDGE_Mabey C200 bridge x 5.25m width, c/w epoxy aggregate steel decks.
- Electrical and Communications
- Support Services
- Waste and material pad

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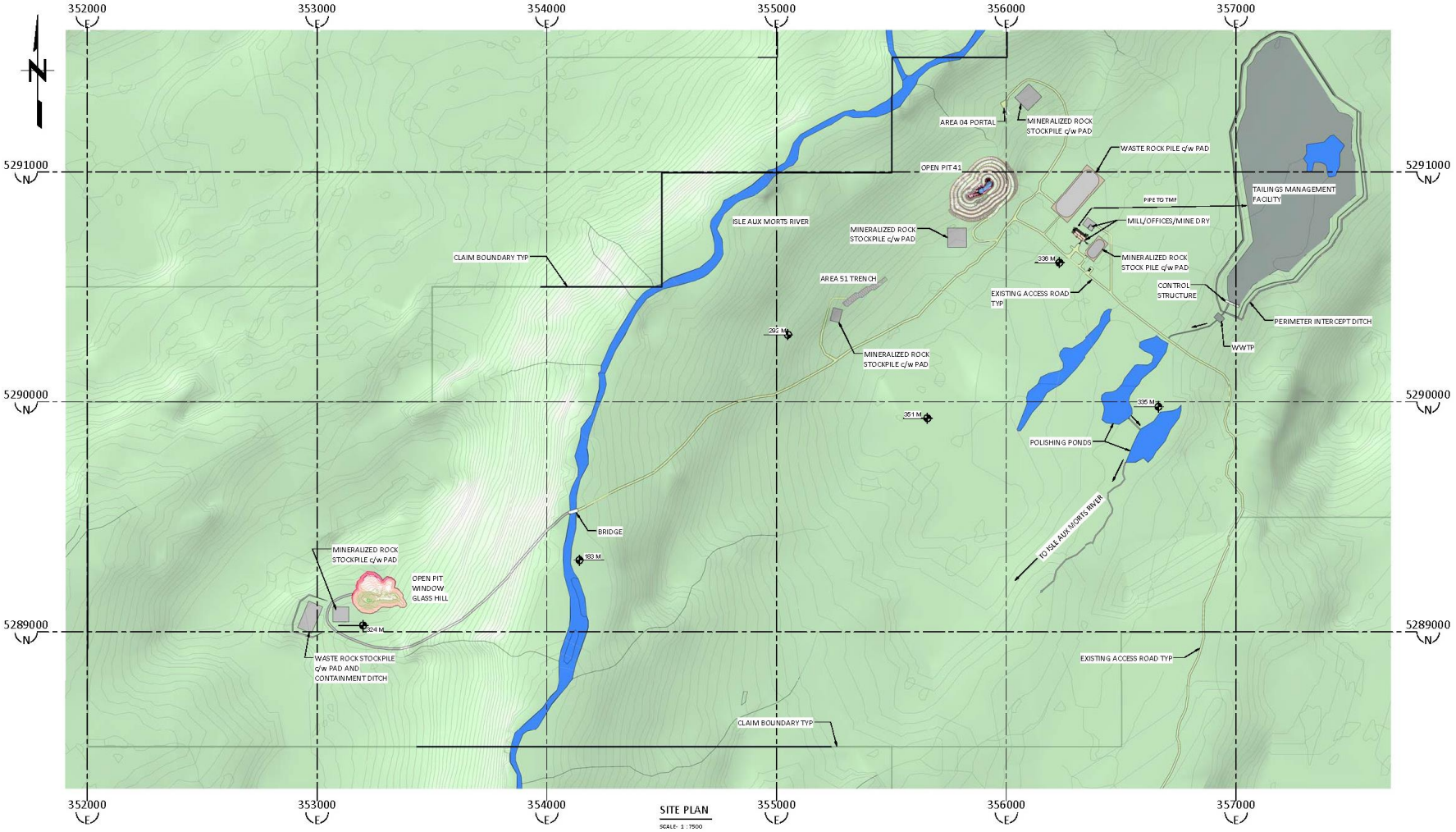

Figure 18.2 Mine Site and Complex

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18.1 Processing Plant

The Mill would process muck at a rate of 850 tonnes per day with an anticipated annual capacity of 285,000 tonnes. During the winter months, underground operations would supply feed to the mill with supplemental feed drawn from the stockpile, outside the mill. The mill would house the assay lab, dry, office, first aid station and a lunch room.

Process facility design and recovery are covered in greater detail in Sections 13 and 17.

18.2 Tailings management Facility (TMF)

The TMF and effluent treatment plant will be design to withstand a 100 year rain or snow event. The TMF would cover an area of approximately 38 hectares and provides adequate storage capacity for the approximately 1.2M m^3 of tailings anticipated to be produced over the projected mine life. The average rate of deposition of tails is 285k t/yr. at a density of 1.5 t/m^3. The TMF construction of the dams and buttresses will utilize mine waste rock or quarried rock from a nearby location. Tailings would be deposited from spigots located along the ridge of the TMF. Process water would be drawn from the TMF area. If required to control water levels in the TMF, effluent water would be treated and discharged to the polishing pond. Figure 18.2 show the TMF location.

18.3 Site Access Road

The existing 22 km road to the Cape Ray project would require upgrades to enable site construction and year round mill access. This includes widening parts of the road, changing some of the existing culverts, and cutting and filling in some of the steep sections of road.

The final 22km road would be a single lane gravel road with pull overs. The road would be maintained to supply year round access to the mill site. Figure 18.3 shows the location of the existing road into the Cape Ray site.

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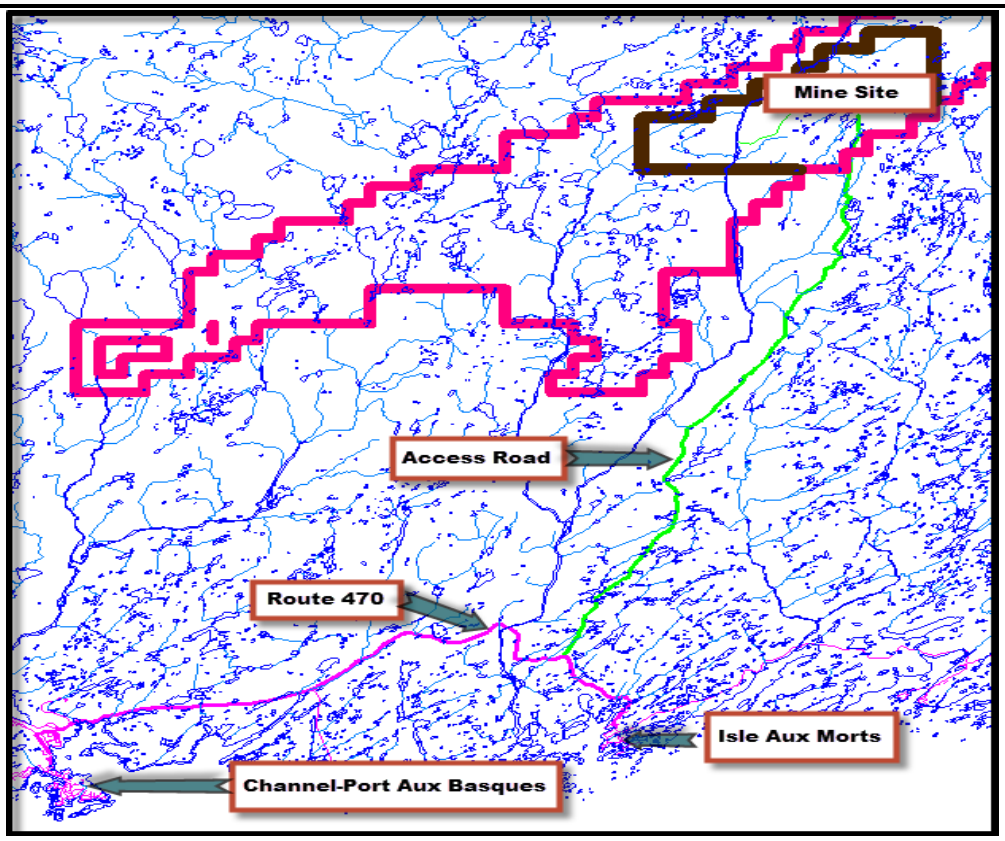

Figure 18.3 Cape Ray Road access off Route 470 to the proposed Mine Site & Complex

18.4 Electrical and Communications

18.4.1 Mine Site Power

Access to the Newfoundland Power electrical transmission grid is available along Route 470 near Isle aux Morts. The installation of a three-phase 66kV electrical transformer station and a transmission line tap would need to be installed.

The utility that owns the 66kV line is actually Newfoundland Power (a Fortis company) that connects to Newfoundland and Labrador Hydro further up the line near Port aux Basques at a higher voltage. The site would be a customer of Newfoundland Power.

A medium voltage 13.8 kV overhead power line to the proposed 5MVA, 4160 Volts substation at the mine site would supply the needed power for the milling and underground mining operations. A 1MVA genset will supply emergency power to the site during a power outage. A genset will also be used for electricity needed for the open pit operation.

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18.4.2 Communications

Communication lines would be run in conjunction with the electrical pole line to the mine site. A radio system would be installed to establish mine site surface and underground communication. Typical land line telephones will be used for contact on and off site.

18.5 Propane

Propane storage tanks of a suitable size would be located close to the area it serves. Propane units would be used for mine air heating, surface building heat and hot water.

18.6 Fuel Storage

A 20,000 litre double wall above ground fuel tank would store and dispense fuel as required. Open pit haul trucks would fuel at the fuel storage site along with underground haul trucks. A service fuel truck would also deliver fuel to underground and open pit drill equipment.

18.7 Process and potable water

Process water in the mill and water accumulating in the TMF would be recirculated back into the mill process water supply system which minimizes the amount of fresh water pumped from the lake.

An insulated pump house would draw fresh lake water from a nearby source at a volume of 1,150 cu.m per day (800 L/min). This water will be utilized as process make up water to the mill and as a supply of raw water for treatment to serve as potable water.

18.8 Sewage Treatment Facility

A small sewage treatment facility will be required to service the entire mine site.

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18.9 Mine Services

The mine site would have an office, first aid station, security, mine rescue room, mine dry and process plant. A parking area for employee vehicles and service vehicles would be located near the office.

A warehouse and cold storage would be attached to the process plant. A 20m x 30m dome style shop would service the open pit and underground mine equipment.

An explosive and detonator storage would be located a safe distance from buildings and mine operations.

Waste rock generated from the 41 pit and 51 trench would be stored on the waste pad north west of the TMF.

Site water and open pit water would be controlled by diversion ditches and berms. Open pit water would report to the existing catchment and sediment pond. A water treatment plant would treat water discharged to the environment if required. The mine complex is depicted in Figure 18.4.

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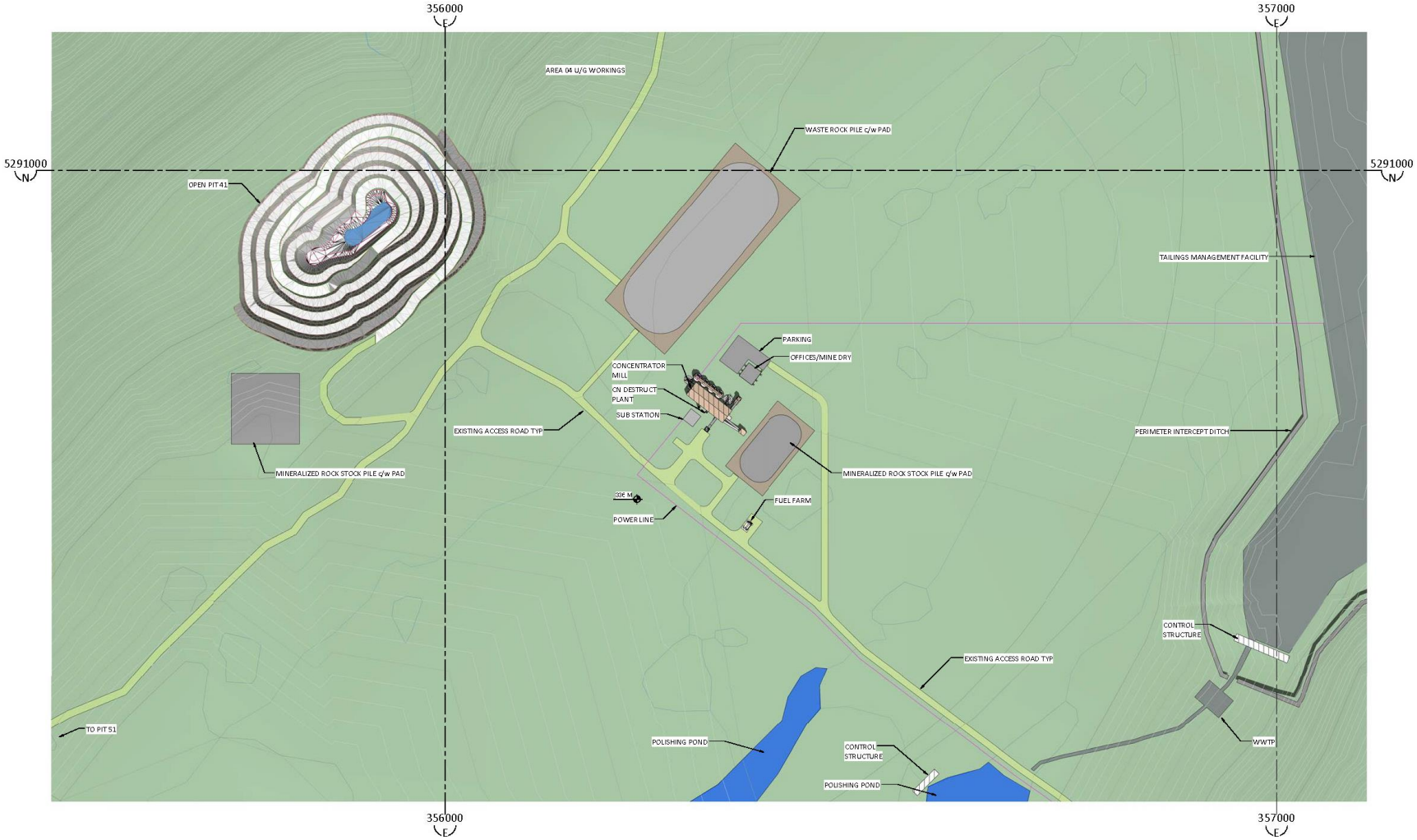

Figure 18.4 Proposed Mine Site Complex.

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19. Market Studies and Contracts

Gold and silver will be the only commodities produced from Cape Ray. These commodities are freely traded, at prices that are widely known, and the prospects for the sale of any production are virtually assured. Prices for these commodities are usually quoted in U.S. dollars per troy ounce. Figure 19.1 and Figure 19.2 show the fluctuation of the price of gold and silver and the American dollar to the Canadian dollar exchange rate over the last five years. The PEA base case scenario uses a price of 1,200 $US per ounce gold, 14 $US per ounce silver, and an exchange rate of 1.25 Canadian dollars per American dollar.

Figure 19.1 5 Year Gold Price Charts

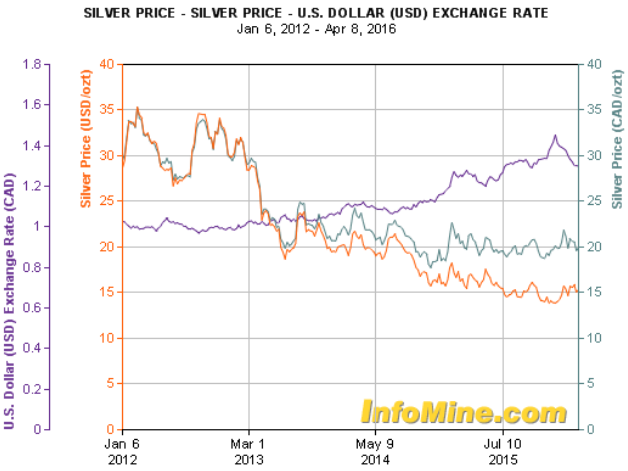

Figure 19.2 5 Year Silver Price Charts

Nordmin is aware that while a project of this size does not require absolute knowledge of the intricacies associated with markets and contracts, the joint venture partnership with Benton is well aware of the risks and benefits provided through different strategies.

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The use of outside consultants, knowledgeable in the areas of marketing, finance and law, as well as freight and contract management..

Doré will be produced at the Cape Ray Mill. The doré will be transported via a contracted security company from the mill to Deer Lake then to the contracted refinery. Agreements will be required with a refinery to sell doré and produce bullion.

The cost associated with doré transportation from the mine site to the commercial refinery and insurance used in the economic analysis is $5/oz.

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20. Environmental Studies, Permitting and Social Impacts

For the environmental studies, permitting and socio-economic evaluation sections of this report the following documents were reviewed:

- An Environmental Preview Report for the Proposed Access Road to the Cape Ray Exploration Site. S. Fudge and Associates Ltd. August 1988.
- Fisheries Resource Component Study – Environmental Impact Statement. LeDrew, Fudge and Associates Ltd. April 1990.
- Raptor Resource Component Study for the Cape Ray Environmental Impact Statement. LeDrew, Fudge and Associates. April 1990.
- Ecological Land Classification Component Study for the Cape Ray Environmental Impact Statement. August 1990.
- Wildlife – Caribou Component Study for the Cape Ray Environmental Impact Statement. LeDrew, Fudge and Associates Ltd. August 1990.

Limitations

The list of documents above, which were accessed through an Inter-Library loan from Memorial University, were the only ones available on site conditions at Cape Ray after an exhaustive literature search. Other documents reviewed but not listed here included Government of Newfoundland and Labrador EA Guidance Manuals, Environmental Acts and Regulations as well as personal communications with regulatory officials in Newfoundland and Labrador Environment and Conservation and Department of Natural Resources.

20.1 Environmental Studies and Permitting

Access to the Cape Ray site is by gravel road which is off the main highway just outside of the town of Isle aux Morts and is approximately 22 km long. The main watershed of the site area is the Isles aux Morts River Watershed, which has provincial monitoring stations recording watershed data. All of the deposits which are being considered for mining fall in this watershed shown in Figure 20.1. The Isle aux Morts River is also a Scheduled Atlantic Salmon River by the Newfoundland and Labrador Department of Natural Resources.

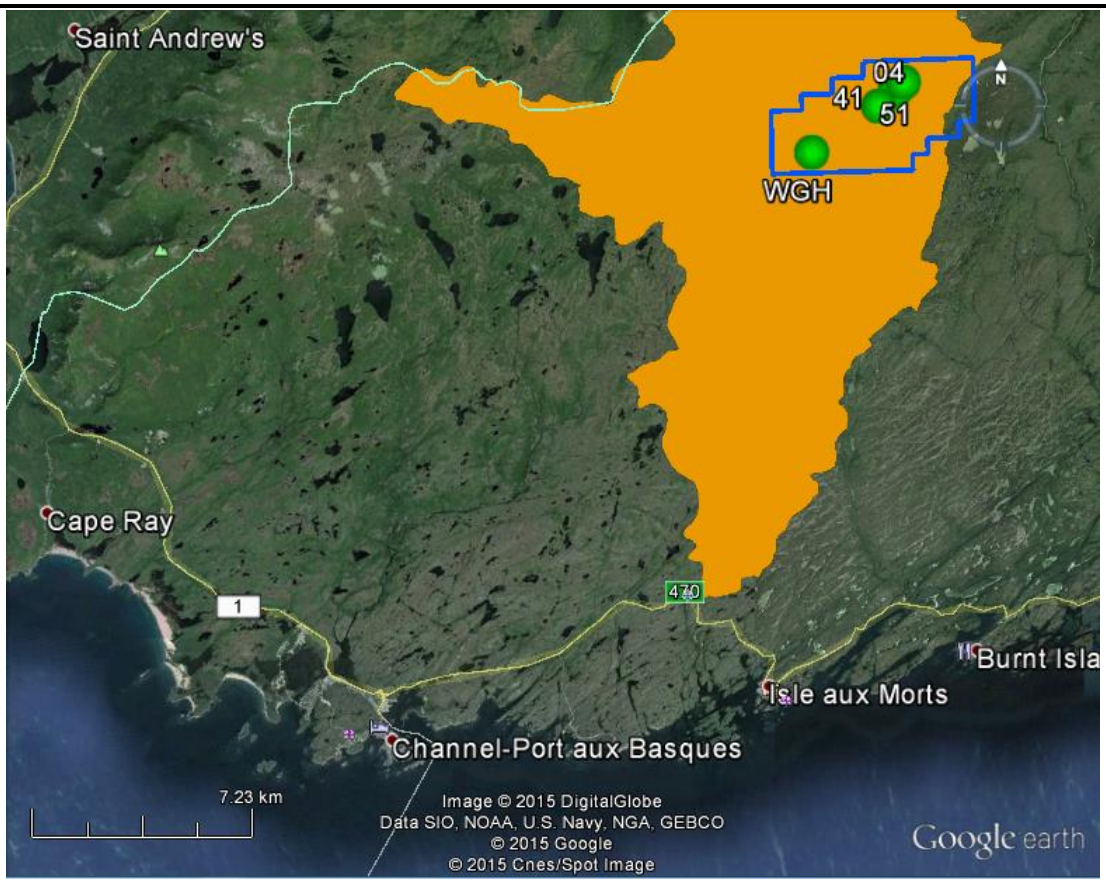

Figure 20.1 Location of Deposits within Isles Aux Morts River Watershed

20.2 Existing Environment and Past Environmental Studies

The following sections will briefly describe the physical, biological and socio-economic environments within the area affected by Project activities. The collection of empirical data for all of these environmental aspects is planned for the 2016/17 season as part of the EA process; however, some data on bio-physical baseline conditions was obtained from historical publications submitted to the Newfoundland government by the previous owners and listed in Section 6 History. This information is summarized below. For those aspects where no baseline data exists, brief plans for the collection of this data are outlined.

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20.3 Natural Environment

20.3.1 Regional Climate

The study area has a moderate climate with a maritime influence based on its proximity to the sea (North Altantic). Freeze up usually does not occur until December. There are variable weather conditions and frequent freeze/thaw cycles throughout the winter. The prevailing winds are west to northwesterly in autumn and winter and southwesterly in spring and summer.

Climate data for the periods 1981 to 2010 is shown in Table 5.1 and Figure 5.6 in Section 5. This data was obtained from the Environment Canada Climate Data website from climate monitoring stations located in Isle aux Morts which is the village most proximate to the Project site.

The climate data for the period of record analyzed shows that show daily mean temperatures in August of 15.8 °C and an average maximum August daily temperature of 19.0 °C (see Table 5.1) In summary, the average daily winter maximum temperature in February is -2.4 °C and the corresponding average minimum is -9.2 °C. The extreme winter minimum is -24 °C, recorded in February. Average yearly precipitation totals 1,653.1 mm, which includes 325.8 cm of snowfall. Very high winds with gusts >100 km/h can occur, often from southerly and southeasterly directions. Illustrative details were covered in Figure 5.6.

20.3.2 Air Quality and Noise

There are no historical records for air quality and noise in the regional area surrounding the Project Site. As part of the Environmental Assessment process and Environmental Baseline data collection program, these surveys would be conducted as per previously issued Environmental Impact Statement (EIS) Guidelines for similar mineral development projects in Newfoundland. The surveys would take place in 2015 or 2015 and the data included in the EIS or Environmental Permitting Regulations (EPR) submissions to the government of Newfoundland.

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20.3.3 Geology and Topography

There is little historical data available on the Geology and Topography of the study area other what could be obtained from some of the component studies performed by the previous owners of the Site. In summary, the regional area surrounding the Project site was glaciated during the advance of the Wisconsin ice sheet. The study area is located on the southern slopes of the Long Range Mountains and is characterized by broad wooded valleys, gently sloping organic deposits and rugged bedrock dominated uplands. The dominant surficial characteristics are bedrock concealed by vegetation, barren outcrops or lichen covered rock, thin bog cover, minor thin till deposits and the occasional drumlinoid feature. The deeply incised river valleys contain glaciofluvial deposits and alluvial deposits derived from post glacial erosional and sedimentary processes.

The rugged and often strongly sloping topography varies in elevation from approximately 140 m along the valley bottom of the Isle aux Morts River to nearly 480 m along the eastern boundary of the regional study area. .

20.3.4 Vegetation and Soils

The following information was gleaned from various component study reports undertaken by the previous owners. Nordmin intends to augment this data with baseline field surveys as part of the EA process. These studies are in the planning stages at present and would be implemented during the 2016/17 field seasons.

Most of the regional study area is occupied by dwarf shrub heath. Species composition varies depending on snow cover and exposure. Sheep laurel (Kalmia angustifolia) heath is the most abundant dwarf shrub vegetation in this ecoregion. In areas with deep snow cover, this heath is augmented by rhodora (Rhododendron canadense). Exposed knolls and summits are generally occupied by heath, dominated by arctic-alpine species such as diapensia (Diapensia lapponica), pink crowberry (Empetrum eamesii) and tundra bilberry (Vaccinium uliginosa).

Wind pruned stands of black spruce (Picea mariana) and balsam fir (Abies balsamea) known as tuckamoor are generally found on well drained sites with moderate wind exposures. These stands are characteristic of the Long Range Barrens ecoregion. Fully developed forests are restricted to sheltered valleys. In the Southern Long Range subregion, these forests are composed mainly of balsam fir, with small quantities of mountain white birch (Betula cardifolia) and black spruce. These stands are densely stocked and are often severely affected by moose browsing and damage from exposure to wind and ice.

Peatlands vary in abundance depending on the topography. Peatlands are extensive in the Southern Long Range sub region but are not the dominant feature of the landscape. The most common peatlands are oligotrophic patterned fens.

Ferro-Humic Podzols and Humo –Ferric Podzols are the characteristic soils of the study area. These soils are course textured, extremely stony and bouldery. They are usually very shallow, but bedrock outcrops are uncommon. Soils of the study area are unsuitable for agricultural

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purposes by virtue of their shallowness and wetness, and lack of organic matter and nutrients. Organic soils in the form of Typic Mesisols are derived from sedge peats and occur throughout the study area. Orthic Regosols occur on the lower terraces along the rivers and streams. These soils are by definition poorly developed and have poor moisture holding capacity because of their coarse textures and are susceptible to inundation because of their location.

20.3.5 Wetlands

There are presently no historical records for the wetlands that occur in the regional area surrounding the Project Site. As part of the EA process and Environmental Baseline data collection program, these surveys would be conducted as per previously issued EIS Guidelines for similar mineral development projects in Newfoundland. The surveys would take place in 2016 or 2017 and the data included in the EIS or EPR submissions to the government of Newfoundland.

20.3.6 Regional Hydrology

Monthly average flow rates for the Isle aux Morts River were obtained from the Water Survey of Canada (WSC) hydrometric data base. The WSC maintains a hydrometric gauging station on the Isle aux Morts River just below the Highway 470 bridge (Sta 02ZB001). Hydrometric data is available for this station dating back to 1962, however, only the period from 2000 to 2013 (the last year available) was analyzed for trending purposes for this report. Table 20.1 summarizes the mean monthly flow rates (m3/sec) for this period.

Figure 20.2 and Figure 20.3 illustrates the flow trends over this same period of record. As illustrated, the maximum flow rates occur in April and May and minimum flow rates generally occur in January and February and again in July/August.

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Table 20.1 Monthly Mean Discharge Volumes of the Isle aux Morts River (m3/s), 2003 - 2013

Year	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual Mean
2003	20.10	15.90	4.51	42.20	29.80	11.70	9.08	3.43	7.30	17.80	14.10	12.80	15.70
2004	2.18	0.92	0.74	29.60	31.40	14.90	7.00	3.46	12.60	10.30	20.30	19.20	12.70
2005	4.19	3.09	2.26	19.20	23.50	4.07	4.46	4.69	18.00	11.30	20.40	12.20	10.60
2006	13.50	3.23	1.17	25.50	11.70	6.38	9.81	10.90	9.14	23.90	29.80	8.49	12.80
2007	9.18	1.76	7.65	12.40	30.40	10.50	13.60	17.90	16.20	10.10	20.80	3.74	12.90
2008	3.13	9.38	4.06	22.50	30.30	7.13	2.70	11.70	14.40	12.20	22.90	20.60	13.40
2009	2.93	1.47	6.16	40.70	27.20	11.20	3.03	10.50	5.85	11.40	19.10	11.30	12.60
2010	4.36	1.98	2.35	22.40	10.10	8.83	11.10	3.38	12.00	21.50	15.60	19.40	11.10
2011	4.26	3.11	12.00	27.70	37.50	17.80	8.42	4.98	4.37	13.30	13.80	14.70	13.50
2012	9.38	9.12	4.74	39.30	17.30	2.01	5.02	5.86	26.60	17.00	11.70	20.90	14.00
2013	3.34	12.00	11.70	27.30	32.00	6.81	8.96	7.80	14.90	8.06	36.10	9.84	14.90
2011	1.73	1.73	2.03	2.20	2.41	1.93	1.83	1.70	1.65	1.97	1.96	1.99	1.93
2012	1.90	1.78	1.79	2.31	1.96	1.48	1.63	1.71	2.12	2.01	1.90	2.05	1.89
2013	1.69	2.29	1.98	2.11	2.24	1.75	1.79	1.74	1.96	1.82	2.19	2.01	1.96
Mean	5.85	4.84	4.51	22.53	20.56	7.61	6.32	6.41	10.51	11.62	16.48	11.37	10.71
Min	1.69	0.92	0.74	2.11	1.96	1.48	1.63	1.70	1.65	1.82	1.90	1.99	1.89
Max	20.10	15.90	12.00	42.20	37.50	17.80	13.60	17.90	26.60	23.90	36.10	20.90	15.70

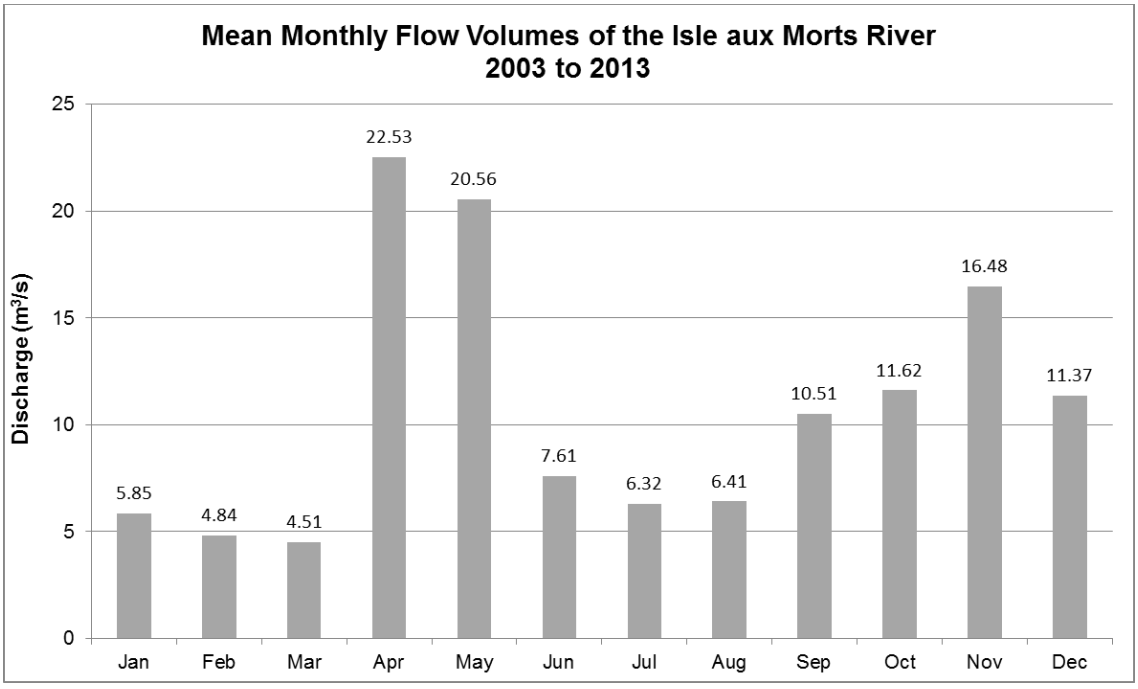

Figure 20.2 Mean Monthly Flows for Isle aux Morts River for 2003-2013

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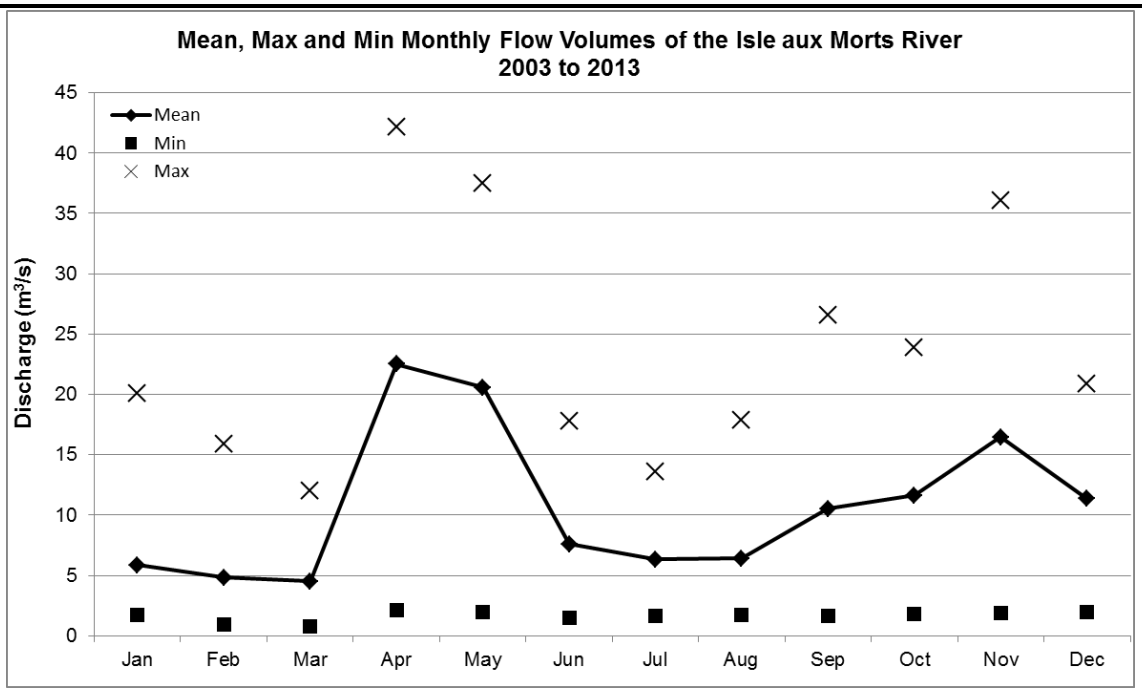

Figure 20.3 Mean, Max and Min Monthly Flows for Isle aux Morts River 2003-2013

The drainage area for the Isle aux Morts river system is 205 km2 and as can be seen from the hydrometric data in Figure 20.4 the river tends to be quite “flashy” with mean daily flows which can increase up to 150% or more from one day to the next. This is due mainly to the very shallow soils in the study area, exposed bedrock and very little moisture retention capability from these materials.

As illustrated, the largest number of peak floods occurs during the months of April and May, then again in November and December.

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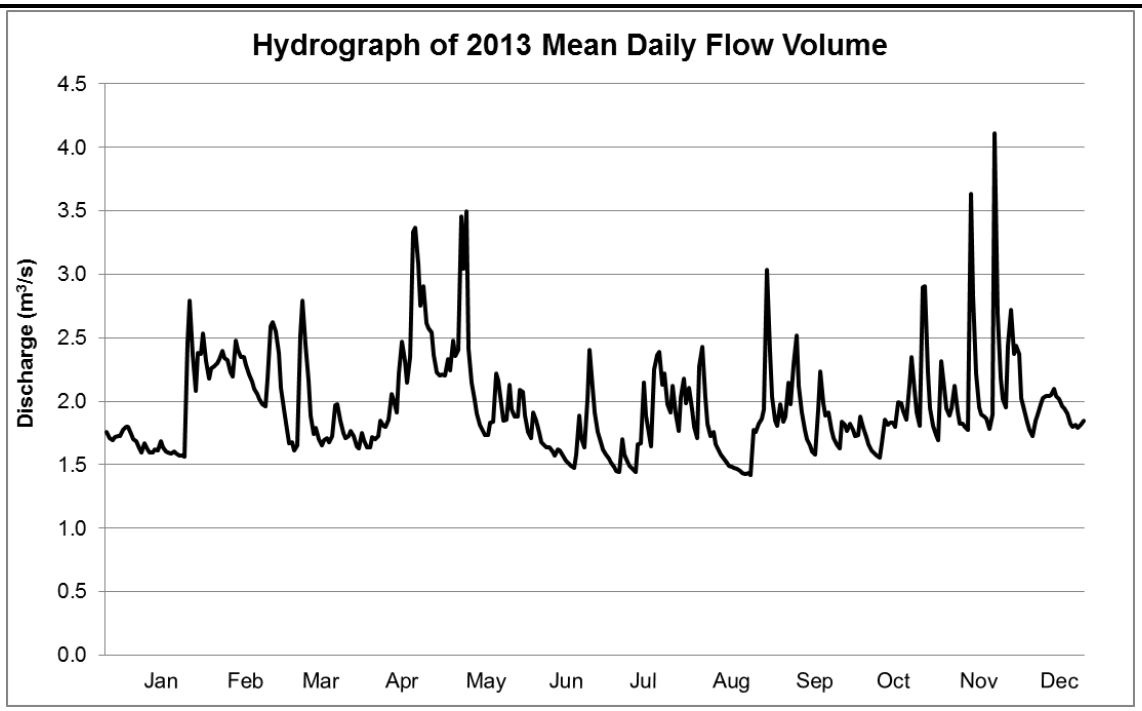

Figure 20.4 Isle aux Morts River Hydrograph 2013

20.3.6.1 Site Hydrology and Water Quality

There appear to be only two small sub-watersheds on the property that would potentially be affected by project activities, and the camp sits on the height of land that separates them. Surface water runoff north of the camp flows directly to the river via a small stream. Surface water to the south of the camp collects in a series of four small interconnected bogs that also eventually report to the river but take a much more elongated flow path. The Isle aux Morts River is directly north of the property and difficult to access. From visual observations made from a distance it appeared to have a relatively high and fast flow with a series of riffles and pools along the length of river adjacent to the property.

This small bogs south of the camp reports to a series of larger ponds that eventually report to the Isle aux Morts River by way of Jack Henry's Brook 3.8 km downstream. These smaller ponds could be utilized as polishing ponds or emergency ponds in the event of an unplanned discharge. Control structures would be constructed at the outlet of the tailings storage pond and the emergency pond.

Currently, ground water from the existing underground workings (flooded adit), flows into a perimeter collection ditch then into a series of settling ponds which then discharge to a small un-named brook which flows into the Isle aux Morts River. Water samples were obtained by Nordmin staff in September of 2014 from several locations along this collection system for the purposes of assessing water quality. Samples were obtained from the portal entrance, north drainage ditch, south drainage ditch and the polishing pond overflow which reports to the Isle

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aux Morts River. The results of this sampling campaign are summarized in Table 20.2. Water quality currently discharging from the property is good and there were only minor exceedances of CCME (Canadian Council of Ministers of the Environment) criteria for the protection of aquatic life but no exceedances of the Newfoundland and Labrador Environmental Control and Sewage Regulation 65/03. A broader baseline surface water monitoring program was undertaken in 2015 during exploration activities and again there were no exceedances of Regulation 65/03 criteria. These results are shown below in Table 20.3, Table 20.4 and Table 20.5. Location of the water quality monitoring stations is shown in Figure 20.5.

Table 20.2 2014 Water Quality Summary

Physical Tests	MDL (1)	UNITS	Reg 65/03 (2)	CEQG-PAL (3)	PORTAL	NORTH DRAINAGE DITCH WP87	POND OVERFLOW CULVERT WP90	SOUTH DRAINAGE DITCH WP93
					22-SEP-14 @ 16:30	22-SEP-14 @ 16:45	22-SEP-14 @ 17:00	22-SEP-14 @ 17:30
Hardness (as CaCO3)	0.51	mg/L			35.2	121	50.6	23.5
pH	0.1	pH			7.42	7.72	7.54	7.29
Total Suspended Solids	2	mg/L	30 (4)		<2.0	<2.0	5.6	<2.0
Anions and Nutrients								
Alkalinity, Total (as CaCO3)	2	mg/L			27	67.3	31.5	13.6
Ammonia, Total (as N)	0.02	mg/L	2		<0.020	<0.020	<0.020	<0.020
Chloride (Cl)	0.1	mg/L		120	4.3	4.17	4.16	4.68
Fluoride (F)	0.03	mg/L		0.120	<0.030	0.032	<0.030	<0.030
Nitrate (as N)	0.03	mg/L	10		0.47	0.249	0.121	<0.030
Nitrite (as N)	0.02	mg/L			<0.020	<0.020	<0.020	<0.020
Total Kjeldahl Nitrogen	0.25	mg/L			0.51	0.38	0.46	0.54
Phosphorus (P)	0.003	mg/L			0.0113	0.008	0.0118	0.0115
Sulfate (SO4)	0.3	mg/L			5.83	54.8	18.3	5.22
Cyanides								
Cyanide, Total	0.002	mg/L	0.025		<0.0020	<0.0020	<0.0020	<0.0020
Organic/Inorganic Carbon								
Total Organic Carbon	1	mg/L			13.3	21.3	16.7	16.1
Total Metals								
Aluminum (Al)	0.005	mg/L			0.112	0.0428	0.0843	0.139
Antimony (Sb)	0.0006	mg/L			<0.00060	<0.00060	<0.00060	<0.00060
Arsenic (As)	0.001	mg/L	0.5	0.005	<0.0010	<0.0010	0.0011	<0.0010
Barium (Ba)	0.01	mg/L	5		0.028	0.026	0.01	<0.010
Beryllium (Be)	0.001	mg/L			<0.0010	<0.0010	<0.0010	<0.0010
Bismuth (Bi)	0.001	mg/L			<0.0010	<0.0010	<0.0010	<0.0010
Boron (B)	0.05	mg/L	5	1.5	<0.050	<0.050	<0.050	<0.050
Cadmium (Cd)	0.000017	mg/L	0.05	0.09	0.000113	0.000201	0.000107	0.000244
Calcium (Ca)	0.2	mg/L			12.7	43.9	16.9	7.81
Chromium (Cr)	0.001	mg/L			<0.0010	<0.0010	<0.0010	<0.0010
Cobalt (Co)	0.0005	mg/L			<0.00050	<0.00050	<0.00050	<0.00050
Copper (Cu)	0.001	mg/L	0.3	0.002 to 0.00278	**0.0109**	**0.0037**	**0.006**	**0.0052**
Iron (Fe)	0.02	mg/L	10	0.300	0.138	**0.341**	0.202	0.124
Lead (Pb)	0.001	mg/L	0.2	0.001 to 0.00406	<0.0010	<0.0010	**0.0022**	**0.0021**
Lithium (Li)	0.05	mg/L			<0.050	<0.050	<0.050	<0.050
Magnesium (Mg)	0.02	mg/L			0.819	2.68	2.05	0.979
Manganese (Mn)	0.001	mg/L			0.0239	0.295	0.0513	0.0077
Mercury (Hg)	0.00001	mg/L	0.005	0.000026	<0.000010	<0.000010	<0.000010	<0.000010
Molybdenum (Mo)	0.001	mg/L			<0.0010	0.002	<0.0010	<0.0010
Nickel (Ni)	0.002	mg/L	0.5	0.025 to 0.11048	<0.0020	<0.0020	<0.0020	<0.0020
Potassium (K)	0.5	mg/L			0.73	1.42	0.7	<0.50
Selenium (Se)	0.001	mg/L	0.01	0.001	<0.0010	<0.0010	<0.0010	<0.0010
Silicon (Si)	1	mg/L			1.3	1.3	<1.0	<1.0
Silver (Ag)	0.0001	mg/L	0.05	0.0001	<0.00010	<0.00010	<0.00010	<0.00010
Sodium (Na)	0.1	mg/L			2.82	2.48	2.53	2.75
Strontium (Sr)	0.001	mg/L			0.0408	0.116	0.0597	0.0273
Tellurium (Te)	0.001	mg/L			<0.0010	<0.0010	<0.0010	<0.0010
Thallium (Tl)	0.0003	mg/L		0.0008	<0.00030	<0.00030	<0.00030	<0.00030
Tin (Sn)	0.001	mg/L			<0.0010	<0.0010	<0.0010	<0.0010
Titanium (Ti)	0.002	mg/L			<0.0020	<0.0020	<0.0020	<0.0020
Tungsten (W)	0.01	mg/L			<0.010	<0.010	<0.010	<0.010
Uranium (U)	0.005	mg/L		0.015	<0.0050	<0.0050	<0.0050	<0.0050
Vanadium (V)	0.001	mg/L			<0.0010	<0.0010	<0.0010	<0.0010
Zinc (Zn)	0.003	mg/L	0.5	0.030	0.0042	0.0136	0.0081	0.0125
Zirconium (Zr)	0.001	mg/L			<0.0010	<0.0010	<0.0010	<0.0010
Radiological Parameters								
Ra-226	0.01	Bq/L			<0.0100	<0.0100	<0.0100	<0.0100

NOTES:

1) MDL - Laboratory Method Detection Limit, ALS Environmental laboratory, Thunder Bay, Ontario.

2) Newfoundland and Labrador, Regulation 65/03 under the Environmental Control Water and Sewage Regulations, 2003 under the Water Resources Act (O.C. 2003-231)

3) Canadian Council of Ministers of the Environment (CCME), Water Quality Guidelines for the Protection of Freshwater Aquatic Life.

4) If water is being abstracted from a water course, used, treated and subsequently returned to the same water course, the total suspended solids data mean that the effluent should not contain more than 30 mg/L more than was in the water originally abstracted.

5) The sample concentrations were not equal to or greater than the Reg.65/03 crietria concentrations .

6) Shaded, bold values indicate concentrations equal to or greater than the CEQG-PAL criteria.

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Table 20.3 2015 Water Quality Summary – Pre-exploration Activities

				Sample ID	WA-01	WA-02	WA-03	WA-04	WA-05	WA-6	WA-7	WA-8
				Sample Date	08/06/2015	08/06/2015	08/06/2015	08/07/2015	08/07/2015	08/09/2015	08/09/2015	08/09/2015
				Lab ID	6849481	6849490	6849448	6849499	6849507	6858413	6858416	6858426
PARAMETER	**UNITS**	**RDL [1]**	**NLR 65/03 [2]**	**CEQG-PAL [3]**								
General Paramters & Nutrients												
Alkalinity as CaCO3	mg/L	5			<5	10	<5	65	24	7	<5	16
pH				6.5 to 9.0	**5.1**	**6.1**	**4.2**	7.3	7.1	6.7	6.5	7.2
Total Organic Carbon	mg/L	0.5			31.6	15.3	8.9	4.3	7.0	14.0	15.3	4.7
Total Phosphorous as P	mg/L	0.03			0.06	0.05	0.74	<0.03	<0.03	<0.03	<0.03	<0.03
Total Suspended Solids	mg/L	5	30 [4]		23	8	<5	8	18	<5	<5	<5
Total Dissolved Solids	mg/L	5	1000 [4]		40	34	24	150	68	32	32	32
Total Kjeldahl Nitrogen as N	mg/L	0.4			2.6	2.1	0.9	3.2	1.8	0.8	0.7	1.3
Ammonia as N	mg/L	0.05	2	190 to 0.2 [5]	0.06	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Bromide	mg/L	0.3		-	<0.3	<0.3	<0.3	<0.3	<0.3	<0.3	<0.3	<0.3
Chloride	mg/L	1		120	2	3	2	3	2	3	2	4
Fluoride	mg/L	0.1		0.12	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	**0.3**
Nitrate as N	mg/L	0.05		13	<0.05	<0.05	0.07	<0.05	0.18	0.09	<0.05	<0.05
Nitrite as N	mg/L	0.05		0.06	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Sulphate	mg/L	2			<2	<2	24	38	12	<2	2	2
Hardness	mg/L	0.7			6.1	19.2	15.8	158.0	50.4	13.5	11.0	25.1
Total Cyanide	mg/L	0.002	0.025	0.005 [6]	<0.002	<0.002	<0.002	<0.002	0.003	<0.002	<0.002	<0.002
Total Metals												
Aluminum	ug/L	5		5 to 100 [7]	**1470**	**286**	**201**	37	**179**	**197**	**244**	63
Antimony	ug/L	2			<2	<2	<2	<2	<2	<2	<2	<2
Arsenic	ug/L	2	500	5	**17**	2	<2	<2	2	<2	<2	<2
Barium	ug/L	5	5000		<5	9	17	15	13	12	11	<5
Beryllium	ug/L	2			<2	<2	<2	<2	<2	<2	<2	<2
Bismuth	ug/L	2			<2	<2	<2	<2	<2	<2	<2	<2
Boron	ug/L	5	5000	1500	<5	<5	<5	<5	<5	<5	<5	<5
Cadmium	ug/L	0.017	50	0.09	**0.145**	0.049	0.039	**1.020**	**0.119**	0.020	0.025	<0.017
Chromium	ug/L	1			35.00	<1	<1	<1	<1	<1	<1	<1
Cobalt	ug/L	1			6.00	<1	<1	<1	<1	<1	<1	<1
Copper	ug/L	2	30	2.0 to 3.61 [8]	**5.0**	<2	<2	**4.0**	**4.0**	<2	2.0	<2
Iron	ug/L	50	10000	300	**5470**	**1760**	**522**	265	**303**	**2700**	**975**	216
Lead	ug/L	0.5	200	1 to 5.97 [8]	**3**	<0.5	<0.5	<0.5	**4**	1	<0.5	<0.5
Manganese	ug/L	2			620	209	176	260	27	29	15	9
Mercury	mg/L	0.00005	0.005	0.000026	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005
Molybdenum	ug/L	2		73	4	<2	<2	<2	<2	<2	<2	<2
Nickel	ug/L	2	500	25 to 150 [8]	11.00	<2	<2	<2	<2	<2	<2	<2
Selenium	ug/L	1	10	1	<1	<1	<1	<1	<1	<1	<1	<1
Silver	ug/L	0.1	50	0.25	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Strontium	ug/L	5			<5	13	12	158	38	9	8	17
Thallium	ug/L	0.1		0.8	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Tin	ug/L	2			<2	<2	<2	<2	<2	<2	<2	<2
Titanium	ug/L	2			22	2	2	<2	3	4	4	<2
Tungsten	mg/L	0.050			<0.050	<0.050	<0.050	<0.050	<0.050	<0.010	<0.010	<0.010
Uranium	ug/L	0.1		15	0.1	<0.1	<0.1	0.8	0.2	<0.1	<0.1	<0.1
Vanadium	ug/L	2			2.00	<2	<2	<2	<2	<2	<2	<2
Zinc	ug/L	5	500	30	15	8	8	**113**	<5	**130**	<5	<5
Calcium	mg/L	0.1			1.3	5.7	5.0	53.5	17.2	4.1	3.1	7.9
Magnesium	mg/L	0.1			0.7	1.2	0.8	5.9	1.8	0.8	0.8	1.3
Potassium	mg/L	0.1			0.5	0.3	0.3	0.6	0.4	0.4	0.4	0.5
Sodium	mg/L	0.1			2.6	3.0	3.1	4.1	2.9	3.2	3.1	4.7
Radionuclides												
Radium 226 [9]	Bq/L	0.005			<0.005	<0.005	<0.005	-	-	<0.005	<0.005	<0.005

Notes:
1. RDL = Reportable Detection Limit provided by AGAT Laboratories, Dartmouth, Nova Scotia.
2. NLR 65/03 = Newfoundland and Labrador Regulation 65/03 Environmental Control Water and Sewage Regulations, 2003 under the Water Resources Acts (O.C. 2003-231).
3. CEQG-PAL = Canadian Council of Ministers of the Environment: Canadian Environmental Quality Guidelines for the Protection of Freshwater Aquatic Life.
4. If water is being abstracted from a water course, used, treated and subsequently returned to the same water course, these solids data mean that the effluent should not contain more than 1000 or 30 milligrams per litre more than was in the water originally abstracted.
5. Total Ammonia guideline values based on pH and temperature measurements (consult CEQG factsheet for more details).
6. Total Cyanide guideline applicable to free fraction of cyanide, presented here for comparison purposes only.
7. Total Aluminum guideline based on pH values.
8. Total Lead, Copper and Nickel guidelines calculated using sample hardness values.
9. Radium ^{226} analysis subcontracted to Saskatchewan Research Council Environmental Analytical Laboratories, Saskatoon, Saskatchewan.
10. No results exceeded the respective NLR Reg.65/03 concentrations.
11. Shaded cells with bold values indicate results above the respective CEQG-PAL guidelines.

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Table 20.4 2015 Water Quality Summary - During Exploration Activities

				Sample ID	WA-02	WA-06	WA-08
				Sample Date	08/18/2015	08/18/2015	08/18/2015
				Lab ID	6877204	6877212	6877219
PARAMETER	UNITS	RDL [1]	NLR 65/03 [2]	CEQG-PAL [3]			
General Paramters & Nutrients							
Alkalinity as CaCO3	mg/L	5			20	5	15
pH				6.5 to 9.0	**6.3**	**6.4**	7.1
Total Organic Carbon	mg/L	0.5			14.5	17.1	5.2
Total Phosphorous as P	mg/L	0.03			0.35	0.10	0.04
Total Suspended Solids	mg/L	5	30 [4]		<5	<5	<5
Total Dissolved Solids	mg/L	5	1000 [4]		162	52	48
Total Kjeldahl Nitrogen as N	mg/L	0.4			2.1	1.2	0.6
Ammonia as N	mg/L	0.05	2	190 to 0.2 [5]	<0.05	<0.05	<0.05
Bromide	mg/L	0.3		-	<0.3	<0.3	<0.3
Chloride	mg/L	1		120	3	2	4
Fluoride	mg/L	0.1		0.12	<0.1	<0.1	**0.2**
Nitrate as N	mg/L	0.05		13	<0.05	<0.05	<0.05
Nitrite as N	mg/L	0.05		0.06	<0.05	<0.05	<0.05
Sulphate	mg/L	2			<2	<2	<2
Hardness	mg/L	0.7			25.9	11.2	22.8
Total Cyanide	mg/L	0.002	0.025	0.005 [6]			
Total Metals							
Aluminum	ug/L	5		5 to 100 [7]	**184**	**179**	76
Antimony	ug/L	2			<2	<2	<2
Arsenic	ug/L	2	500	5	2	<2	<2
Barium	ug/L	5	5000		8	14	<5
Beryllium	ug/L	2			<2	<2	<2
Bismuth	ug/L	2			<2	<2	<2
Boron	ug/L	5	5000	1500	<5	<5	<5
Cadmium	ug/L	0.017	50	0.09	0.080	<0.017	<0.017
Chromium	ug/L	1			<1	<1	<1
Cobalt	ug/L	1			1.00	<1	<1
Copper	ug/L	2	30	2.0 to 3.61 [8]	<2	<2	<2
Iron	ug/L	50	10000	300	**843**	**2570**	**312**
Lead	ug/L	0.5	200	1 to 5.97 [8]	1	1	<0.5
Manganese	ug/L	2			707	36	32
Mercury	mg/L	0.00005	0.005	0.000026	<0.00005	<0.00005	<0.00005
Molybdenum	ug/L	2		73	<2	<2	<2
Nickel	ug/L	2	500	25 to 150 [8]	<2	<2	<2
Selenium	ug/L	1	10	1	<1	<1	<1
Silver	ug/L	0.1	50	0.25	<0.1	<0.1	<0.1
Strontium	ug/L	5			21	10	25
Thallium	ug/L	0.1		0.8	<0.1	<0.1	<0.1
Tin	ug/L	2			<2	<2	<2
Titanium	ug/L	2			2	4	<2
Tungsten	mg/L	0.050			<0.010	<0.010	<0.010
Uranium	ug/L	0.1		15	<0.1	<0.1	<0.1
Vanadium	ug/L	2			<2	2.00	<2
Zinc	ug/L	5	500	30	10	6	<5
Calcium	mg/L	0.1			7.9	3.5	7.3
Magnesium	mg/L	0.1			1.5	0.6	1.1
Potassium	mg/L	0.1			<0.5	<0.5	<0.5
Sodium	mg/L	0.1			3.2	2.7	4.3
Radionuclides							
Radium 226 [9]	Bq/L	0.005			<0.005	<0.005	<0.005

Notes:

1. RDL = Reportable Detection Limit provided by AGAT Laboratories, Dartmouth, Nova Scotia.
2. NLR 65/03 = Newfoundland and Labrador Regulation 65/03 Environmental Control Water and Sewage Regulations, 2003 under the Water Resources Acts (O.C. 2003-231).
3. CEQG-PAL = Canadian Council of Ministers of the Environment Canadian Environmental Quality Guidelines for the Protection of Freshwater Aquatic Life.
4. If water is being abstracted from a water course, used, treated and subsequently returned to the same water course, these solids data mean that the effluent should not contain more than 1000 or 30 milligrams per litre more than was in the water originally abstracted.
5. Total Ammonia guideline values based on pH and temperature measurements (consult CEQG factsheet for more details).
6. Total Cyanide guideline applicable to free fraction of cyanide, presented here for comparison purposes only.
7. Total Aluminum guideline based on pH values.
8. Total Lead, Copper and Nickel guidelines calculated using sample hardness values.
9. Radium ^{226} analysis subcontracted to Saskatchewan Research Council Environmental Analytical Laboratories, Saskatoon, Saskatchewan.
10. No results exceeded the respective NLR Reg.65/03 concentrations.
11. Shaded cells with bold values indicate results above the respective CEQG-PAL guidelines.

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Table 20.5 2015 Water Quality Summary – Post Exploration Activities

				Sample ID	WA-02	WA-03	WA-05	WA-06	WA-07	WA-08	41-DS	Portal
				Sample Date	08/26/2015	08/26/2015	08/26/2015	08/26/2015	08/26/2015	08/26/2015	08/26/2015	08/21/2015
				Lab ID	6906554	6906563	6906268	6908174	6906571	6906866	6906270	6906937
PARAMETER	**UNITS**	**RDL [1]**	**NLR 65/03 [2]**	**CEQG-PAL [3]**								
General Paramters & Nutrients												
Alkalinity as CaCO3	mg/L	5			16	22	-	7	6	11	-	123
pH				6.5 to 9.0	6.6	6.8	-	6.5	6.5	7.2	-	7.8
Total Organic Carbon	mg/L	0.5			21.0	13.0	-	21.1	20.3	9.6	-	2.5
Total Phosphorous as P	mg/L	0.03			0.04	<0.03	<0.03	0.07	0.04	<0.03	<0.03	<0.03
Total Suspended Solids	mg/L	5	30 [4]		<5	<5	<5	<5	<5	<5	<5	15
Total Dissolved Solids	mg/L	5	1000 [4]		<5	<5	32	<5	<5	<5	96	118
Total Kjeldahl Nitrogen as N	mg/L	0.4			0.8	0.6	-	1.2	1.0	1.1	-	0.4
Ammonia as N	mg/L	0.05	2	190 to 0.2 [5]	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Bromide	mg/L	0.3		-	<0.3	<0.3	-	<0.3	<0.3	<0.3	-	<0.3
Chloride	mg/L	1		120	3	5	-	2	3	4	-	5
Fluoride	mg/L	0.1		0.12	<0.1	<0.1	-	<0.1	<0.1	**0.2**	-	**0.3**
Nitrate as N	mg/L	0.05		13	<0.05	0.07	0.06	<0.05	<0.05	<0.05	0.09	<0.05
Nitrite as N	mg/L	0.05		0.06	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Sulphate	mg/L	2			<2	<2	-	<2	<2	<2	-	11
Hardness	mg/L	0.7			26.2	36.2	-	10.0	11.0	17.1	-	164.0
Total Cyanide	mg/L	0.002	0.025	0.005 [6]	0.002	<0.002	<0.002	0.003	<0.002	<0.002	<0.002	<0.002
Total Metals												
Aluminum	ug/L	5		5 to 100 [7]	**228**	**336**	245	**248**	**254**	98	212	**754**
Antimony	ug/L	2			<2	<2	<2	<2	<2	<2	<2	<2
Arsenic	ug/L	2	500	5	3	2	<2	<2	<2	<2	3	4
Barium	ug/L	5	5000		8	33	18	14	13	<5	13	66
Beryllium	ug/L	2			<2	<2	<2	<2	<2	<2	<2	<2
Bismuth	ug/L	2			<2	<2	<2	<2	<2	<2	<2	<2
Boron	ug/L	5	5000	1500	<5	<5	<5	6	<5	<5	5	<5
Cadmium	ug/L	0.017	50	0.09	**0.109**	**0.152**	**0.185**	<0.017	0.023	<0.017	**0.340**	**0.441**
Chromium	ug/L	1			<1	<1	<1	<1	<1	<1	<1	1.00
Cobalt	ug/L	1			2.00	2.00	<1	<1	<1	<1	<1	2.00
Copper	ug/L	2	30	2.0 to 3.61 [8]	<2	**3.0**	**4.0**	<2	<2	<2	**4.0**	3.0
Iron	ug/L	50	10000	300	**1230**	**1800**	**840**	**2560**	**1200**	**490**	**313**	**1520**
Lead	ug/L	0.5	200	1 to 5.97 [8]	1	**3**	**3**	1	1	<0.5	**7**	**13**
Manganese	ug/L	2			2190	2180	650	47	25	36	201	797
Mercury	mg/L	0.00005	0.005	0.000026	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005
Molybdenum	ug/L	2		73	<2	<2	<2	<2	<2	<2	6	5
Nickel	ug/L	2	500	25 to 150 [8]	<2	<2	<2	<2	<2	<2	<2	3.00
Selenium	ug/L	1	10	1	<1	<1	<1	<1	<1	<1	<1	<1
Silver	ug/L	0.1	50	0.25	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Strontium	ug/L	5			23	36	114	11	11	20	336	665
Thallium	ug/L	0.1		0.8	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Tin	ug/L	2			<2	<2	<2	<2	<2	<2	<2	<2
Titanium	ug/L	2			3	10	6	5	4	<2	7	15
Tungsten	mg/L	0.050			<0.010	<0.010		<0.010	<0.010	<0.010		<0.010
Uranium	ug/L	0.1		15	<0.1	0.1	0.4	<0.1	<0.1	<0.1	1.5	2.5
Vanadium	ug/L	2			<2	<2	<2	<2	<2	<2	<2	3.00
Zinc	ug/L	5	500	30	7	8	10	6	<5	<5	6	13
Calcium	mg/L	0.1			8.0	12.2	-	3.0	3.1	5.2	-	58.5
Magnesium	mg/L	0.1			1.5	1.4	-	0.6	0.8	1.0	-	4.3
Potassium	mg/L	0.1			0.2	0.5	-	0.2	0.4	0.3	-	1.1
Sodium	mg/L	0.1			3.0	3.7	-	2.7	2.9	3.8	-	6.1
Radionuclides												
Radium 226 [9]	Bq/L	0.005			0.010	0.010	-	<0.005	0.006	0.008	-	0.030

Notes:
1. RDL = Reportable Detection Limit provided by AGAT Laboratories, Dartmouth, Nova Scotia.
2. NLR 65/03 = Newfoundland and Labrador Regulation 65/03 Environmental Control Water and Sewage Regulations, 2003 under the Water Resources Acts (O.C. 2003-231).
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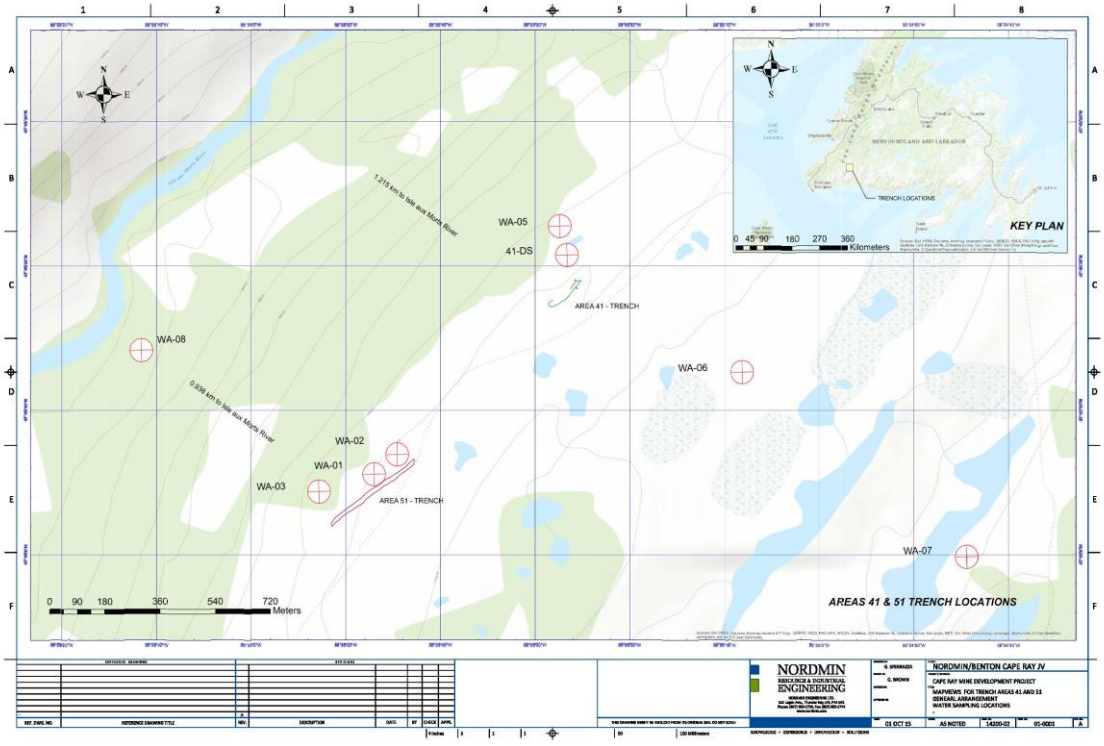

Figure 20.5 Surface Water Sampling Locations during 2015 Exploration Program

20.3.7 Hydrogeology

There are no historical records for hydrogeology in the regional area surrounding the Project Site. As part of the EA process and Environmental Baseline data collection program, these surveys would be conducted as per previously issued EIS Guidelines for similar mineral development projects in Newfoundland. The surveys would take place in 2016 or 2017 and the data included in the EIS or EPR submissions to the government of Newfoundland.

20.3.8 Fish and Fish Habitat

Information on fish and fish habitat within the surface waters of the regional study area was gleaned from component study reports prepared by the previous owners of the site. Although this data is dated (1989/90) it is still of relevance and would be augmented by further fishery baseline studies to be conducted by Nordmin at the site in 2016 and 2017 as part of the EA and permitting process.

Historical data indicates that species reported to be present in the Isle aux Morts River, which is a Scheduled Salmon River, are; Atlantic salmon(Salmo salar), brook trout (Salvelinus fontinalis), and American eel (Anguilla rostrate). The river was considered to be completely obstructed 3.2 km from the mouth until the two falls were channelized by the Department of Fisheries and Oceans (DFO) in 1961 and is now considered unobstructed clear to the headwaters.

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Salmon angling data for 2012 for the Isle aux Morts River, collected by DFO revealed that a total of 177 individual fish were either captured and retained, or released (small and large) over a total of 289 rod/days. This resulted in a catch-per-unit-effort (CPUE) of 0.61. The river is considered to be an important angling river by DFO although fishing effort is restricted to the lower 5 km of river. The removal of the obstructions in 1961 has not dramatically changed the success rate or harvest. The 2012 angling data was compared to historical catch and CPUE data (1952 – 1990) and revealed that individual numbers of fish caught or released has remained fairly constant (mean = 172 vs. 177), while the CPUE has increased (mean = 0.42 vs. 0.61). This may be due to the fact that commercial fishing for salmon was suspended in 1984.

Brook trout are fished in the watershed but mostly in ponds and brooks near the lower reaches of the river.

Fish community data was collected in 1989 by the previous owners as part of the component studies and involved back pack electrofishing of the small brooks and ponds as well as in the river and overnight gill net sets in the deeper ponds. This effort produced 123 Atlantic salmon, 2 brook trout and 1 American eel. These were all captured in the Isle aux Morts River and the lower reaches of Jack Henry's Brook. Most of the salmon captured (52%) were young of the year (age 0+) with the remainder (36%) being either yearlings (age 1+) or 2 year old fish (12%).

The two brook trout that were captured were taken in the lower reaches of Jack Henry's Brook just upstream of the confluence with the Isle aux Morts River. Age of the specimens was estimated to be 0+ and 1+. The single American eel was captured in the river.

Gill net sets in the deeper ponds to the south of the proposed mine site did not produce any fish. Electroshocking of the interconnecting streams, shoreline areas and shallower sections of these ponds also did not produce any fish.

Level of effort was not recorded so a CPUE could not be calculated for these surveys.

20.3.8.1 Fish Habitat

Fish habitat mapping was also undertaken as part of the fishery resource component studies conducted in 1989 by the previous owners and is summarized below. Habitat mapping was completed on the entire length of the Isle aux Morts River from the mouth at the Gulf of St. Lawrence to the headwaters upstream of the proposed mine site. The river forks into two branches approximately 6 km upstream from the mouth; the Main Stem branch and the Big Pond branch.

The main stem of the Isle aux Morts comprises 12 sections of river each of which can be categorized into 5 main habitat types. They are (moving from upstream to downstream):

i. Section 1 – described as wide well defined channel with steep banks for the first four kilometers of the river. Shoreline is mainly bedrock and heavily wooded with mature spruce and alder that provide some instream cover. Average water depth is one metre or more with some cobble and gravel bars that would be exposed during low flow

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periods. This section of river has some holding pools and is subject to the highest fishing pressure as the upstream areas are difficult to access on foot. The spawning habitat type could be characterized as Type I and II good for spawning and rearing of salmonid species.

ii. Section 2 – described as a narrow stretch of rapids and chutes with a substrate of mainly bedrock. The banks are steep and also bedrock with steep vertical cliffs. There are many rapids, riffles and chutes along these sections but they do not pose as an obstacle to upstream migration. The spawning habitat types can be characterized as Type II, good for rearing and III, mostly falls or rapids which are not optimal areas for spawning or rearing.

iii. Section 3 – described as a wide shallow stretch of river with some slow riffle sections and an average depth of less than one metre. The average width is 20 m with low banks composed of boulders, cobble and sod with a gentle backslope. The substrate in this section of the river is mainly cobble, gravel and boulders with some exposed bars. There is limited shoreline cover and the spawning habitat is Type II and some Type I areas.

iv. Section 4 – this section is described as having several braided channels with large wooded islands in the middle of the river. The water depth is shallow (< 1m) and is boulder strewn with large banks that would be exposed during low flow periods. The shoreline areas are gently sloping and treed with spruce and alder providing some shade and overhead cover. River substrate here is mainly cobble and gravel which provide good areas for spawning and rearing for salmonid species (Type I and II).

v. Section 5 – this section is defined as being boulder strewn with an ill-defined streambed. The banks are barren plateau areas indicative of the upper watershed areas of the river with boulders and rock providing the only cover. This section is poor spawning and rearing habitat and is probably not utilized by anadromous salmon or trout. This section is typical of the numerous rivulets and small streams that drain the highland regions of this watershed.

The Big Pond branch of the Isle aux Morts River comprises habitats similar to those described above, with some Type I and II spawning habitats at the section closest to the confluence with the main stem. Upstream of this point is the limit for migrating salmonids due to physical barriers however there are still good rearing areas for trout.

20.3.9 Wildlife

Little information exists on the wildlife common to the Project site and surrounding areas however, some information was obtained from the Environmental Preview Report for the proposed mine access road that was prepared by the previous owners in 1988. Most of this information is anecdotal in nature or based on personal communication with local residents or outfitters. Some information was gathered from game harvesting records as well. This information is summarized below.

20.3.9.1 Big Game

Three species of big game are known to inhabit the Project site and surrounding areas. They are: black bear (Ursus americanus), woodland caribou (Rangifer tarandus), and moose (Alces alces).

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Although no detailed information is available on black bear populations in the area, sightings are common but they are not considered to be abundant in the Project area.

Woodland caribou are by far the most abundant big game in the Project area and surrounding territories however the two major herds (La Poile, and Long range barrens) listed in the information available showed that they exist mainly 50 -60 km and 30 -40 km respectively from the proposed mine site. However, some of these caribou migrate south to the Burnt Island Pond area to calve in the spring which is approximately 10 km east of the proposed mine site.

Although some moose habitat does exist in the Project area it is considered marginal and confined to the river valleys where there is some forest cover. Moose herd numbers are unknown but 1988 harvest records show that the two local outfitters harvested 15 moose each during the season.

20.3.9.2 Small Game and Furbearing Mammals

Very little data exists on small game and furbearing mammals in the immediate Project Area but information obtained from historical reports suggest that the area provides habitat for the following species: arctic hare *(Lepus arcticus*), willow ptarmigan (*Lagopus lagopus*), snowshoe hare (*Lepus americanus*), rock ptarmigan (*Lagopus mutus*), red fox (*Vulpes vulpes),* muskrat (*Ondatra gibethicus*), mink (*Mustela vison*), ermine (*Mustela erminea*), beaver (*Castor Canadensis*), and otter (*Lutra Canadensis*).

20.3.9.3 Avifauna

There is some historical information available on raptor resources in the Project area from the component study report prepared by the previous owners and this is summarized below. Information on waterfowl and passerines is not available other than anecdotal references in some reports that the area may be used by migrating waterfowl in the spring and fall and that waterfowl production in the proposed Project area is not considered significant.

Aerial and ground surveys for large raptors were conducted by the previous owners in 1989 as part of the resource component studies. Aerial surveys covering 100 % of the study area and ground surveys covering approximately 30 km of ground within an 8 km radius of the proposed mine site and along the access road were conducted during early June of 1989. Ground surveys involved the use of tape recorded raptor calls to evoke a vocal response if any raptors were present.

The results of the surveys revealed the presence of only one species of raptor. Two osprey (*Pandion haliaetus*) were sighted during one of the ground surveys. No other raptors were sighted or responded to any of the raptor calls.

Although only one species of raptor was noted during these surveys, the Project area would likely be frequented by other raptor species for hunting , nesting or overwintering such as Bald Eagle (*Haliaeetus leucocephalus*), Northern Harrier (Circus cyaneus), American Rough-legged Hawk (Buteo lagopus), Northern Goshawk (Accipiter gentiles), Sharp-shinned Hawk (Accipiter

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striatus), American Kestrel (Falco sparverius), Merlin (Falco columbarius), Great Horned Owl (Bubo virginianus), Northern Hawk Owk (Surnia ulula), Short-eared Owl (Asio flammeus), and Boreal Owl (Aegolius funereus).

20.4 Human Environment

20.4.1 Historic and Heritage Resources

There are no known designated environmentally sensitive or cultural heritage sites within the Project lands. Aboriginal consultation, as well as biological and archaeological assessment work that are planned as part of the EA process will identify environmentally sensitive sites and cultural heritage sites. An objective of the on-going consultation process with the engaged Aboriginal communities is anticipated to result in the identification of such sites so that they can be considered for preservation during the EA and Project planning process.

A Stage 1 Archaeological Assessment of the Project lands and road corridor will be completed as part of the baseline studies to be undertaken as part of the EA process. This assessment would be conducted by a licensed archaeologist, in accordance with requirements of the Government of Newfoundland, with full participation and involvement by the surrounding First Nations to the extent that they are willing to participate.

20.4.2 Aboriginal, Community Consultation and Social Requirements

Aboriginal and community engagement will be an integral part of the Environmental Assessment process should the project move forward. Consultation plans will be developed for Aboriginal groups, local communities and other interested persons including NGO's and implemented to provide feedback on social and community impacts from the project.

20.5 Potential Environmental Constraints to Development

The following sections describe possible constraints to development of the property and, are based solely on visual observations and literature available on the site. These may or may not be mitigated through good management and sustainable development practices but this would require further investigation and collection of empirical data from the site though baseline environmental assessment studies.

20.5.1 Proximity to Isle aux Morts River

As stated previously, previous studies have shown that this river is utilized by Atlantic salmon for spawning purposes and is a Scheduled Salmon River. The present mine workings (adit, ore/waste rock pile, collection ditching and ponds) are within 900 metres of the river and are currently discharging directly into it. The location of the open pit mines would likely be within 500 metres of the river. If spawning (and early life cycle stage) salmon are utilizing the section of the river proximate to the site, water would likely have to be treated and pumped into the watershed

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south of the mine site. This may be a requirement regardless of salmon activity but would depend on the quality of the water discharging from the adit and coming into contact with the ore/waste rock piles. Geochemistry data suggests the waste rock is potentially acid generating.

20.6 Environmental Assessment and Permitting

The Cape Ray Project represents the construction and operation of a proposed 850 tonne per day gold mine and mill along with the construction/upgrading of an existing 22 km access road to the site from highway 470. As such, the project will be subject to a Provincial and Federal Environmental Assessment process as well as various permitting requirements. These are outlined briefly in the following sections.

20.6.1 Environmental Assessment Processes and Requirements

In the province of Newfoundland and Labrador, proposed resource development projects may be subject to provincial and/or environmental assessment legislation.

Under the provincial *Environmental Protection Act (EPA, 2002),* the proponent of a project that could have significant effects on the natural, social or economic environment is required to register this project or undertaking for examination under the provincial EA process. Under the EPA the project, or undertaking is subject to Part 10 of the EPA and pursuant to Section 33(2) of the associated *Environmental Assessment Regulations (EAR, 2013)* which states:

33(2) “An undertaking that will be engaged in the mining, beneficiating and preparing of a mineral as defined in the Mineral Act whether or not these operations are to be performed in conjunction with a mine or at mills that will be operated separately shall be registered.”

Following a 45 day public and governmental review of this EA Registration, the Minister of Environment and Conservation will make a determination whether the project may proceed, subject to terms and conditions or if further EA review and obligations are warranted.

Based on the Ministerial review, there are four possible outcomes following the 45 day review period. These are:

1. The Project may be released – subject to terms and conditions set by the Minister;
2. The Project will require an Environmental Preview Report (EPR) – in which additional information is required that is not contained in the Project Registration;
3. The Project will require an Environmental Impact Statement (EIS) – will be required where significant potential negative environmental effects are anticipated or there is significant public concern regarding the undertaking; or,
4. The Project may be rejected – this may occur if unacceptable environmental effects are indicated or the undertaking is not in the public interest.

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All decisions will be announced in the EA Bulletin within ten days of notification to the proponent.

Under *CEAA 2012,* if the project or undertaking is captured by the Regulations Designating Physical Activities, a project description must be submitted which then undergoes a 45 day screening process to determine if a federal EA is required.

20.6.2 Permitting Requirements

Permits that will potentially be required for exploration work include license to operate a temporary work camp in accordance with Section 10 of the Lands Act (1991) issued by the Department of Municipal and Intergovernmental Affairs, a Surface Lease and Mineral Exploration Approval both issued by the Newfoundland Department of Natural Resources, Mineral Development Division. A Water Use Licence may also be required from the Newfoundland Department of the Environment and Conservation, Water Resources Division, as well as a Certificate of Approval for Septic System for water use and disposal for project site facilities.

Permits that will potentially be required for the construction and development of a mining and production facility are outlined in the Table below including and Federal Approvals

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Table 20.6 Summary of Anticipated Applicable Acts and Regulations

Level of Government	Approval Potentially Required	Responsible Agency	Legislation/Act
Federal	Authorization under Section 35 or 36 of the Fisheries Act	Department of Fisheries and Oceans and Environment Canada	Fisheries Act
		Transport Canada	Navigation Protection Act Transportation of Dangerous Goods Act
	Environmental Assessment, Compliance Standards; Permits may be required	Environment Canada	Migratory Birds Convention Act Canadian Environmental Protection Act Canadian Environmental Assessment Act 2012 Species at Risk Act Federal Policy on Wetland Conservation
	Explosives User Magazine Licence	Natural Resources Canada	Explosives Act
	Compliance Standards; Permit may be required	Engineering Services Division, Service Newfoundland	National Fire Code National Building Code
Provincial	Surface Lease – Areas to be developed for mining should be held under a Surface Lease.	Mineral Development Division, Department of Natural Resources	Mining Act
	Mineral Exploration Approval – An application for approval must contain a detailed plan and description of the exploration activities	Mineral Development Division, Department of Natural Resources	Mineral Act and Mineral Regulations
	Environmental Assessment Water Use Licence – Water withdrawals for use at temporary camps or during construction and operation. Certificate of Approval for Water withdrawal system of > 4,500 L/d. Water supply for construction or operations activities.	Department of Environment and Conservation	Water Resources Act Environmental Protection Act Environmental Assessment Regulations
	Certificate of Approval for Septic System > 4,546 L/d and for drilled wells. Certificate of Approval for Storage and Handling of Gasoline and Associated Products Fuel cache Permit	Engineering Services Division, Service Newfoundland	Environmental Protection Act
	Quarry Permit – to dig, excavate and extract quarry borrow material for construction purposes	Mineral Lands Division, Department of Natural Resources	Quarry Materials Act and Regulations
	Cutting Permit Operating Permit Permit to Burn	Forestry Branch, Department of Natural Resources	Forestry Act and Regulations
	Certificate of Approval for installation of a water supply and Sewage system.	Water Resources Division, Department of Environment and Conservation.	Health and Community Services Act and Regulations
	Permit for Storage, Handling, Use or Sale of Flammable and Combustible Liquids	Engineering Services Division, Service Newfoundland	Fire Protection Act
Municipal	Approval for Waste Disposal – the operation of a waste disposal site may require municipal approval. License to operate a temporary work camp.	Community Council	Municipal Plan and Development Regulations

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The specific approvals required will be known once the Environmental Assessment (EA) process has been completed. It is premature to prepare a precisely detailed approvals list until each aspect of the project is better defined.

20.7 Tailings and Water Management Requirements

20.7.1 Waste Rock

The volume of waste rock produced is highly dependent the final pit design, project economics, and deposit geology. Waste rock material will be stockpiled near the pit it was excavated from on an engineered pad if it is acid generating. The potential for acid generation for waste rock materials is unknown. There is a test program currently underway to determine acid generation potential. Any potentially acid generating (PAG) materials will require a NPAG (non-potentially acid generating)pad and either need to be relocated back to the pit once it is complete, or covered to reduce the acid generation potential. Some waste rock materials which are non-acid generating will be required to build roads, pads for buildings, and laydown areas.

20.7.2 Tailings Management

Tailings from mill operations (after CN destruction process) will be handled by conventional methods and disposed of in an engineered Tailings Management Facility (TMF) via a slurry pipeline. A flocculent will be added to the tailings stream to promote settling of solids. Decanted water from the TMF will then pass through the Wastewater Treatment Plant (WWTP) which will treat to provincial and federal effluent criteria (process described in Section 20.8). Treated effluent will then report to a series of polishing ponds before discharging to the environment via a natural stream that eventually discharges to the Isle aux Morts River. The approximate linear distance of stream between the polishing pond discharge point to the river is four kilometres. Water quality monitoring stations will be established at fixed points along this stream and sampled on a monthly basis.

Run off from the watershed surrounding the TMF will be collected in a perimeter intercept ditch which will be constructed around the base of the TMF. The interception ditches will be excavated in bedrock or glacial till lined with rock to mitigate erosion. All runoff water from these intercept ditches will report to the polishing ponds.

20.8 Water and Wastewater Management and Treatment

20.8.1 Open Pit Dewatering

During the life of the open pit, a sump (that is founded in bedrock) will be carried in the floor of the pit to contain runoff, direct precipitation and the groundwater seepage that will enter the pit because much of the excavation will be done below the water table. The sump will serve as a primary treatment pond to achieve the treatment listed below, if required.

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- Settling of suspended solids and potential chemical conditioning (coagulant and/or flocculant addition). Metals will not be precipitated in the sump.
- Potential addition of commercially available ammonia oxidizing bacteria to reduce the concentration of ammonia that would be present due to the use of explosives.
- Potential attenuation of hydrocarbons using sorbent materials and/or the use of hydrocarbon degrading enzymes.
- The sump will also serve as a surge pond to equalize the flows of water being pumped from the pit to the treatment system and/or process water tanks (for recycling) and/or release to the environment.

The potential for the pit shell to have a strong hydraulic connection (i.e. through overburden) to proximal surface watercourses or aquifers is interpreted as low. However, additional field investigations will be conducted to determine the potential for seepage from these features to enter the pit so that adaptive management plans can be developed and implemented as may be required during the life of the Project. The adaptive management plan may include, but not necessarily be limited to the measures listed in the bullets listed below.

- Low-permeability cut off walls installed in overburden to reduce the hydraulic connectivity between the pit and the water feature.
- Targeted grouting program to install a grout curtain in bedrock to reduce groundwater seepage entering the pit.
- Groundwater interception wells to intercept overburden and/or bedrock groundwater before it enters the pit.
- Groundwater and/or surface water interception trenches to intercept overburden groundwater and surface water before it enters the pit.

As the development of open pit(s) is not anticipated to create a strong hydraulic connection with any surface water features or aquifers, no dewatering wells surrounding the pit are planned. Sometimes this strategy is employed at “wet pits” to create a drawdown cone surrounding open pit developments to intercept groundwater before it enters the pit and typically allow direct discharge of this water to the environment.

A hydrogeological model will be developed to predict the volume of groundwater that will enter the workings as part of the environmental assessment and permitting process.

20.8.2 Water Treatment

Wastewater (i.e. mine water, grey water, treated sewage from domestic sewage systems, backwash water from potable water treatment plants, collected runoff that has contacted process materials, wash water, etc.) will be consolidated in the Tailings Management Facility (“TMF”). This water will be recycled for use in the mine and the mill. Surplus water in the TMF will be treated to meet provincial effluent criteria as well as federal requirements pursuant to the *Metal Mining Effluent Regulations*. Treatment steps and water treatment strategies that are currently anticipated are summarized in the bullets below.

- Control of contaminants at the source through good housekeeping practices that

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minimize hydrocarbon.

- Implementation of an ammonia control program to reduce ammonia in mine water. Components of the ammonia control program would include, but not necessarily be limited to, low solubility product selection, minimization of spillage, promote efficient blasting practices.
- Deployment of hydrocarbon degrading enzymes in sumps and in the TMF as needed.
- Use of ammonia oxidizing bacteria in sumps and in the TMF as needed.
- Use of coagulants and/or flocculants (i.e. chemical conditioning) to remove total suspended solids either in a pond or in a clarifier. Sludge from this treatment step would be placed in the TMF.
- Additional chemical conditioning (i.e. metal precipitants that work at neutral pH) and/or pH adjustment to precipitate metals as low solubility hydroxides. Sludge from this treatment step would be consolidated and managed to minimize any environmental risk from potential metal leaching.
- Filtration, following the chemical conditioning step, to remove suspended solids and metal precipitates.
- Final pH adjustment through the addition of carbon dioxide or acid (i.e. citric, hydrochloric or sulphuric) to within regulatory limits.
- Aeration or biological treatment of residual ammonia to oxidize any nitrites or other contaminants that could be present.

Any treatment steps that are deemed unnecessary based on operating experience and empirical evidence, would be by-passed.

Cyanide would be destroyed in tailings slurry before it leaves the mill building envelope. Although additional engineering is required, this would likely be achieved using the well-established S02-air process. The by-product of this process of most concern would be ammonia, which would be produced through the hydrolysis of cyanate.

20.8.3 Potable Water

Raw water from a drilled well will be filtered and disinfected in accordance with regulatory requirements and good engineering practices. Water will report to a surge tank and then distributed to areas of the Project site where potable water is required. Backwash water from the potable water treatment systems will report to the TMF.

20.9 Mine decommissioning and closure

The objectives of mine closure activities, as outlined in the Closure Plan are to return the site to a natural condition as soon as possible upon cessation of mining activities and to ensure the long term chemical and physical stability of the site post-closure.

Remediation and reclamation activities are designed to minimize and reduce the duration of any impacts to the local receiving environment from past mining activities. These include the closing and rehabilitation of the mine and tailings management facilities and the immediate dismantling

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of all unnecessary infrastructure.

A monitoring program will also be implemented post-closure to ensure the long term performance of the restoration measures, and to put into effect, remedial action if necessary. The monitoring program will also include continuous characterization and control of the surface waters throughout the various project sites as well as the quality of the soils and the sustainability of the permafrost after closure and restoration activities have been completed. Campaigns for the collection of surface water and soil samples, as well as visual inspections of reclamation sites will be implemented.

The monitoring program will be carried out over a ten year period after closure of the mine.

Total cost for all closure activities is estimated to be $4,400,000. A financial guarantee will be paid annually to the Minister of Finance of Newfoundland and Labrador.

20.9.1 Mine Closure and Decommissioning

The rehabilitation measures that are being developed are intended for walk-away closure. Rehabilitation activities will be completed within approximately 36 months from the commencement of closure and the major activities are summarized below in general chronological order.

- Buildings, trailers, intermodal shipping containers, storage tanks, equipment and any chemicals/consumables will be removed and salvaged, recycled or disposed of in accordance with applicable legislation. Concrete foundations will be demolished to grade as is necessary and used to backfill local depressions.
- Hydrocarbon contaminated soil will be identified and remediated in accordance with applicable legislation.
- Equipment in the pit workings will be purged of all operating fluids and salvaged to the maximum extent practical. Consumables will be removed from the pit workings and salvaged.
- The open pits will be backfilled with clean NAG rock, overburden replaced and surface areas will be revegetated with native plant species.
- The haulage roads between the pits and on surface along with waste rock and material pads will be removed and the areas revegetated with native plant species.
- Pipelines (water, compressed air) on the site will be flushed with fresh water or air, plugged and left in place. Fuel pipelines (propane, diesel) will be decommissioned with legislative requirements and good engineering standards as applicable.
- The domestic sewage disposal system components will be salvaged. The septic tank will be purged of its contents by a licensed hauler and backfilled with locally available soil and/or rock.
- Remaining liquid and solid waste at the Project site will be removed for recycling or disposal with licensed contractors in accordance legislative requirements.
- Rock, sludge and tailings that pose a risk of acid generation and/or metal leaching will be consolidated during the life of the mine and managed in accordance with best

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industry practice.

- Areas of the Project site that are overlain with (clean) development rock will be scarified and any modest embankments will be sloped to a minimum of 3H:1V for long-term physical stability. Accumulations of soil sized particles in the crevasses of rock embankments will be planted with native tree seedlings in accordance with established silvicultural practices. Native soil from the established stockpiles on the site will be placed over these prepared areas and the area will be re-vegetated.
- The power line, electrical distribution infrastructure and substation will be removed and re-used to the extent practical.
- Site roads will be rehabilitated in general accordance with standard road abandonment practices as removed from use. This will include, but not necessarily be limited to removing and replacing culverts with erosion proof channels, and scarifying road surfaces to encourage re-vegetation.
- The long-term chemical and physical stability monitoring program will be continued to completion, in accordance with the regulatory requirements.

The access road beyond the first stream crossing is owned by the Crown and is outside the scope of the Project site closure.

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21. Capital and Operating Costs

Preparation of the capital cost estimates are based on the Nordmin's philosophy that emphasizes accuracy over contingency and uses defined and proven project execution strategies. The estimates were developed by using first principles and applying direct applicable project experience, and by avoiding the use of general industry factors. Virtually all of the estimate inputs are derived from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans as defined in the PEA. The target accuracy of the capital cost estimate is in the range of +20%/-20%.

The following cost estimates are detailed within this section:

- Initial Capital Cost – Major costs incurred in constructing the Mill and Tailing Management Facilities (TMF).
- Sustaining Capital Cost – Expenditres incurred during operations for waste development, underground set-up and underground infrastructure for deposit 04, 51 and 41.
- Closure and Reclamation Cost – Costs incurred to permanently close and reclaim the site

The following capital costs are not discussed in this section:

- Sunk Costs – Not considered in this PEA.
- Owners Reserve – Not considered in this PEA.

Contractors performed preproduction mining, it is assumed that Contractors will be used for all deposits. The specific scope and execution plans described in this PEA. Any deviations from these plans will affect the capital costs.

The following costs are not included in the capital cost estimate:

- HST
- Schedule acceleration costs
- Schedule delays and associated costs, such as those caused by:
    - Unexpected site conditions
    - Latent ground conditions
    - Force majeure
    - Permit applications
- Development fees and approval costs beyond those specifically identified
- Cost of any disruption to normal operations
- Foreign currency changes from project exchange rates
- Commodity specific escalation rates
- Event risk
- Cost associated with third party delays
- Sunk costs
- Escalation – all cost data is presented in Canadian 2015 dollars.

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Certain items within the operating costs begin during the pre-production phase and continue through the life of the mine. All costs incurred during the pre-production phase have been capitalized and are part of the capital cost estimate under sustaining capital. Operating costs have been compiled in accordance with industry standards.

The pre-production capital expenditures have an internal contingency factor of 10%. The estimates are based on budget pricing from suppliers for critical components, consultants, contractors and a review of other Canadian projects. Smaller equipment and facilities component costs were factored based on industry norms for the type of facility being constructed and, where possible, adjusted to reflect local conditions.

Table 21.1 Pre Production Capital Expenditures (in thousands)

Pre-Production Capital	Contingency	Yr-2 ($)	Yr-1 ($)	Yr. 1 ($)	Yr. 2 (S)	Yr. 3 (S)	Yr. 4 (S)	Yr. 5 (S)	Yr. 6 (S)	Total (S)
Permitting	10%	1,359	1,359	-	-	-	-	-	-	2,718
Roadwork-Quote from Adams Construction	10%	506	143	-	-	-	-	-	-	649
Over burden Removal	10%	220	275	-	-	-	-	-	-	495
Surface Infrastructure-Open Pit/Mill	10%	-	2,524	-	-	-	-	-	-	2,524
Material and Waste Pads(3)-Mine & Mill	10%	165	275	-	-	-	-	-	-	440
Dewatering/trench/sumps	10%	-	275	-	-	-	-	-	-	275
Processing Plant	10%	10,481	20,319	-	-	-	-	-	-	30,801
Tailings	10%	-	3,520	-	-	-	-	-	-	3,520
Surface Equipment	10%	-	825	-	-	-	-	-	-	825
Water Treatment Plants/Testing	10%	385	1,650	-	-	-	-	-	-	2,035
Power Distribution to Mill	10%	385	770	-	-	-	-	-	-	1,155
Working Capital	10%	-	550	-	-	-	-	-	-	550
Engineering for Capital	10%	-	802	-	-	-	-	-	-	802
Mine Closure	0%	-	-	-	-	-	-	-	4,400	4,400
Total		**13,502**	**33,287**	**-**	**-**	**-**	**-**	**-**	**4,400**	**51,189**

The sustaining capital expenditures by deposit do not have any contingency. Included in these expenditures are the initial cost to start-up each deposit as well as ramp development for the underground mine. In the open pit the expenditures include initial setup as well as overburden and first pioneer benching (see Table 21.2).

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Table 21.2 Sustaining Capital Expenditures by Deposit (in thousands)

Deposit	Yr-2 ($)	Yr-1 ($)	Yr. 1 ($)	Yr. 2 ($)	Yr. 3 ($)	Yr. 4 ($)	Yr. 5 ($)	Yr. 6 ($)	Total ($)
Pit41	-	556	-	-	-	-	-	-	556
U.G.04	-	-	13,669	2,857	-	-	-	-	16,526
Windowglass	-	-	-	3,149	-	-	-	-	3,149
U.G.51	-	-	-	-	2,103	7,514	-	-	9,617
Other (41U.G./51Trench)	-	-	-	-	-	3,840	-	-	3,840
Total	**-**	**556**	**13,669**	**6,007**	**2,103**	**11,355**	**-**	**-**	**33,689**

All the operating costs are based on typical Canadian prices from contractors, suppliers and other similar type Canadian projects, for consumables and parts. The cost of power is based on rates charged by Newfoundland and Labrador Hydro for similar sized power consumers and services in the province. Labour costs for the operating period are based on the manpower schedules presented for each department and their associated labour costs. The costs include a burden component of 25-30%. All costs are quoted in constant 2015 Canadian Dollars.

Table 21.3 Underground Operating Costs

Underground Operating Costs	
Mining Block - Total Tonnes	$49.72/tonne
Development	$4,675/metre
PMR Development with Shotcrete	$94/tonne
Treatment Plants (All inclusive)	$1.10/tonne
Processing	$26.42/tonne

Table 21.4 Open pit Operating Costs

Open pit Operating Costs	$/tonne
Mining Block - Total Waste Tonnes	3.66
Mining Block-Total PMR Tonnes	6.81
Transportation of PMR	2.00
Crushing	1.45
Treatment Plants (All inclusive)	1.10
Processing	26.42

Both operating costs include the expenditures for direct mining related labour and supervision. Geology and engineering staff costs are provided as G&A within each mining method. The vision of this PEA is to use contractors for both Open Pit and underground operations. The processing cost is expected to be $26.42 per tonne of potentially economic mineralization based on preliminary discussions with custom milling operations. A breakdown of the processing cost is presented in Section 17.

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22. Economic Analysis

The PEA is based on Indicated and Inferred Mineral Resources. The Inferred Resources are preliminary and generally considered to be too speculative to be categorized as Mineral Reserves beyond a Preliminary Economic Analysis. Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure.

An engineering economic model was developed to estimate annual cash flows and sensitivities. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed to approximate the true investment value. It must be noted the tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

For the cash-flow, gold price of US$1,200/oz, silver price of US$14/oz and exchange rate US$=CAD$1.25 were considered. The processing plant recovery for gold is 97% and for silver 45% utilized throughtout the project life.

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Table 22.1 Cash-flow

Description	Au Avg. g/t	Ag Avg. g/t	Unit	Yr. -2	Yr. -1	Yr. 1	Yr. 2	Yr.3	Yr. 4	Yr. 5	Yr. 6	Total
Potential Mineable Resources												
Pit 41	3.12	10.31	**tonnes**	503	503	503	215					
U.G 04	5.99	14.07	**tonnes**	479	479	479	479	446	219			
Window Glass	2.24	2.68	**tonnes**	328	328	328	328	318	259	169	35	
U.G. 51	7.29	14.21	**tonnes**	347	347	347	347	347	347	347	219	
Other (41 U.G. / 51 Trench)	6.89	5.24	**tonnes**	44	44	44	44	44	44	44		
Mined			tonnes	0	0	288	258	286	309	306	254	**1,701**
Total Remaining resources			**tonnes**	**1,701**	**1,701**	**1,413**	**1,155**	**869**	**560**	**254**		
Gold Produced			Ounces			**28**	**28**	**47**	**47**	**48**	**52**	**250**
Silver Produced			Ounces			**42**	**40**	**49**	**48**	**35**	**46**	**260**
Revenue												
Gold - Revenue			CAD $			41,586	42,111	69,803	70,789	71,917	78,389	374,596
Silver - Revenue			CAD $			744	693	849	841	610	812	4,549
Total Revenue			**$**			**42,329**	**42,805**	**70,653**	**71,630**	**72,527**	**79,201**	**379,145**
Dore Refining Cost			$5/ounce			139	140	233	236	240	261	1,249
Net Revenue			**$**			**42,191**	**42,664**	**70,420**	**71,394**	**72,288**	**78,940**	**377,896**
Operating Costs			**Contingency**									
Total Operating Cost			0%			**23,938**	**29,826**	**26,913**	**25,143**	**49,817**	**48,956**	**204,593**
EBITDA						**18,252**	**12,838**	**43,507**	**46,251**	**22,471**	**29,984**	**173,304**
Capital Expenditures												
Sustaining Capital Expenditures by Deposit			**Contingency**									
Pit 41			0%		**556**							**556**
U.G. 04			0%			**13,669**	**2,857**					**16,526**
Window Glass			0%				**3,149**					**3,149**
U.G. 51			0%					**2,103**	**7,514**			**9,617**
Other (41 U.G. / 51 Trench)			0%						**3,840**			**3,840**
Major Pre-Production Capital			**Contingency**									
Total Capital Expenditures			10%	**13,502**	**33,843**	**13,669**	**6,007**	**2,103**	**11,355**		**4,400**	**84,878**
Pre-Tax Cash flow			**$**	**-13,502**	**-33,843**	**4,584**	**6,831**	**41,405**	**34,896**	**22,471**	**25,584**	**88,426**
Pre-Tax Cumulative Cash flow			$	-13,502	-47,345	-42,761	-35,930	5,475	40,371	62,842	88,426	
Tax			29%					**942**	**10,120**	**6,517**	**7,419**	**24,998**
After-Tax Cash flow				-13,502	-33,843	4,584	6,831	40,463	24,776	15,954	18,164	**63,428**
After-Tax Cumulative Cash flow				-13,502	-47,345	-42,761	-35,930	4,533	29,309	45,263	63,428	

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The economic analysis of the base case scenario produces a pre-tax net present value discounted at 7% of $48.4M and a rate of return of 29%.

The preproduction capital cost and the sustaining capital costs are estimated to $51.2M and $33.7M respectively for a total capital cost of $84.9M.

The total operating cost for the project is estimated to $M or $per oz. The all-in costs are estimated to be $ per Oz. A summary of the expected parameters used for the financial analysis is presented in Table 22.2

Table 22.2 Economic Analysis

	Pre-Tax	After-Tax
Net Revenue	$378M	$378M
Undiscounted Cash-flow	$88.4M	$63.4M
NPV (7%)	$48.4M	$32.6M
NPV (10%)	$36.7M	$23.6M
IRR	29%	24%
Payback Period	3 years	
Life of Mine	6 years	

The potentially mineable resource is estimated and shown in Table 22.3.

Table 22.3 Potential Mineable Resources (in thousands)

Deposit	Au Avg. (g/t)	Ag Avg. (g/t)	Tonnes
Pit 41	3.12	10.31	503
U.G 04	5.99	14.07	479
Windowglass	2.24	2.68	328
U.G. 51	7.29	14.21	347
Other (41 U.G. / 51 Trench)	6.89	5.24	44
Total			**1,701**

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Table 22.4 Potential minable resource parameters

	Underground	Open Pit
	%	%
Resource (diluted)	80	95
Mine Dilution	15	10
Mine Recovery	5	5

Table 22.5 Major Pre-Production Capital Expenditures

Description	Contingency	$ M
Permitting	10%	$2.72
Processing Plant	10%	$30.80
Tailings	10%	$3.52
Mine Closure	0%	$4.40
Total Capital Expenditures		$41.44

Table 22.6 Total sustaining and capital expenditures

Expenditures	$ M
Deposit	
Pit 41	$0.56
U.G. 04	$16.53
Windowglass Hill	$3.15
U.G. 51	$9.62
Other (41 U.G. / 51 Trench)	$3.84
Total Sustaining Capital	**$33.69**
Permitting	$2.72
Road Work	$0.65
Overburden Removal	$0.50
Surface Infrastructure – Open Pit/Mill	$2.52
PMR and Waste Pads (3) – Mine & Mill	$0.44
Dewatering/Trench/Sumps	$0.28
Processing Plant	$30.80
Tailings	$3.52
Surface Equipment	$0.83
Water Treatment Plants/Testing	$2.04
Power Distribution to Mill	$1.16
Working Capital	$0.55
Engineering for Capital	$0.80
Mine Closure (Paid by Asset Sales)	$4.40
Total Capital	**$51.19**
Total of Sustaining and Capital	**$84.88**

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Table 22.7 Sensitivity Table

Variable	Pre-Tax NPV @ 7% ($millions)	IRR%	After-Tax NPV @ 7% ($millions)	IRR %
Gold Price - $1,400 U.S.	90	44	62.3	35
Gold Price - $1,000 U.S.	7	11	3	9
Silver Price - $17.00 U.S.	49	29	33	24
Silver Price - $11.00 U.S.	47.8	29	32	24
Exchange Rate = $1.33	65	35	44.6	29
Exchange Rate = $1.18	33.6	23	22	19
Capital Costs +20%	40.3	24	26.5	19
Capital Costs -20%	54	32	36.3	26
Sustaining Capital Costs +20%	43.3	27	29	22
Sustaining Capital Costs -20%	56.5	36	39	30
Operating Costs +20%	22	18	13.7	15
Operating Costs -20%	75	39	51.7	32

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23. Adjacent Properties

In addition to the Nordmin and Benton JV claims which hold the deposits of 04, 41, 51, and WGH, Benton also holds claims on either side of the JV area. Benton's additional claims house the known Isle aux Morts (IaM), and Big Pond (BP) deposits as shown below in Figure 23.1.

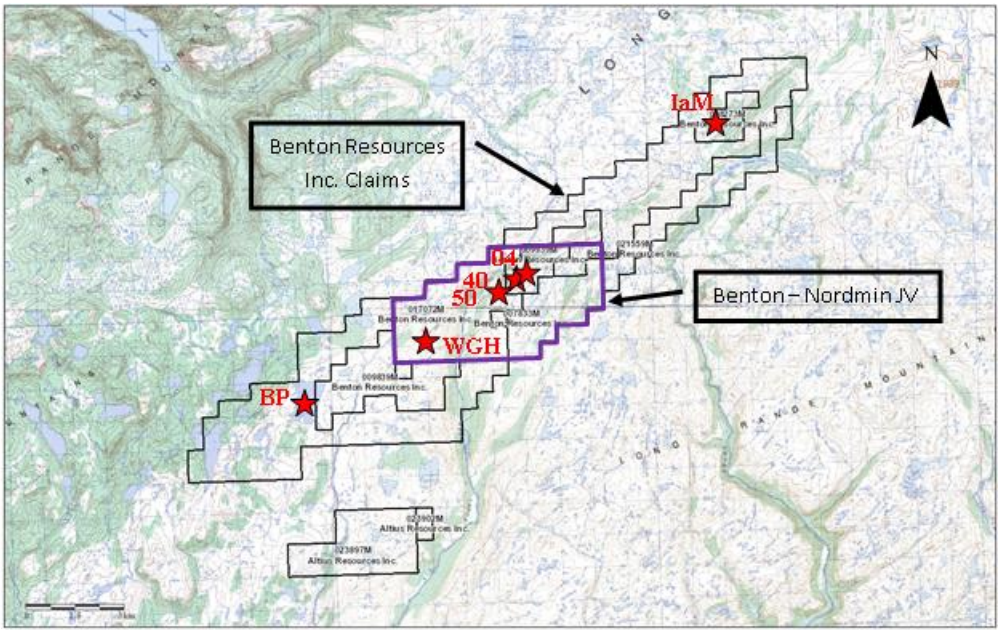

Figure 23.1 Benton Entire Claim Block for the Cape Ray Project

There are no other claim holders with claims directly bordering Benton's in the Cape Ray area.

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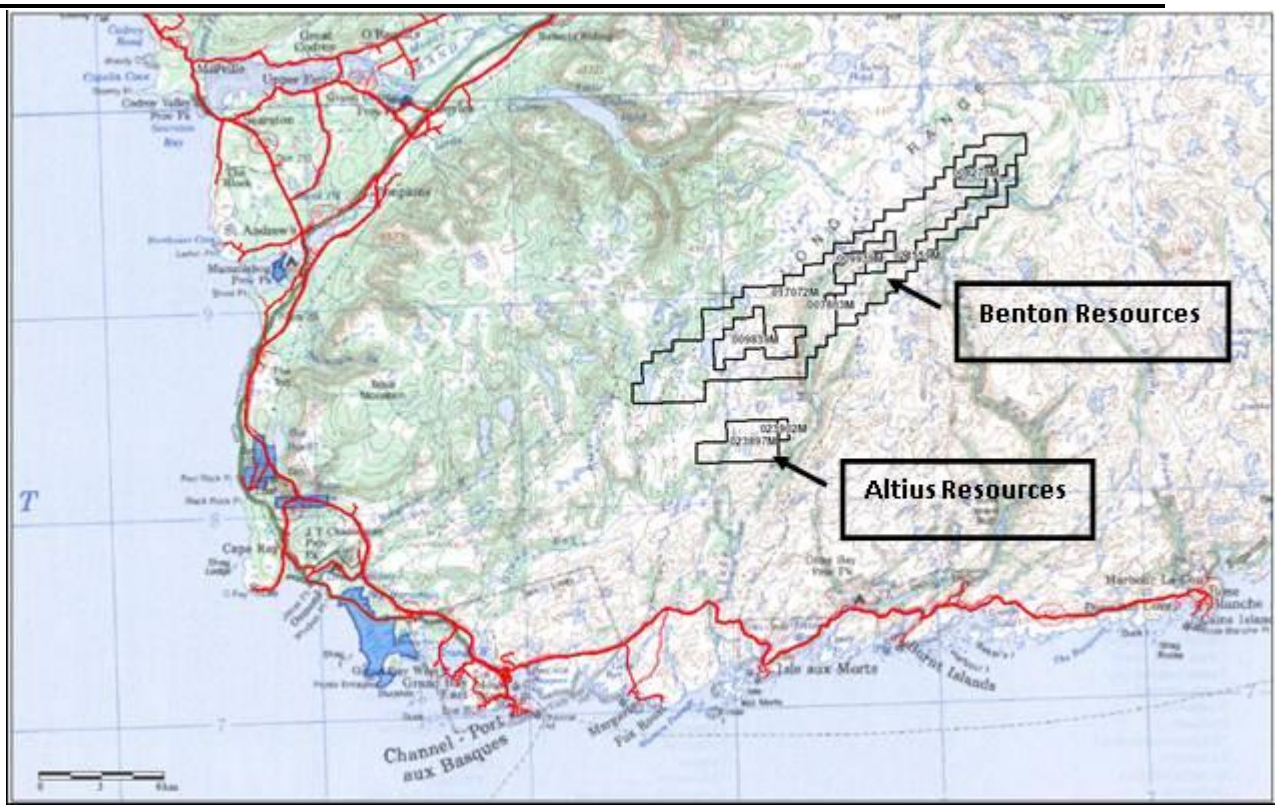

Figure 23.2 closest Nearby Claim Holders .Geoscience Atlas 2016.

The only nearby property to that held by Benton Resources are two mining leases to the south held by Altius Resources Inc. as shown in Figure 23.3.

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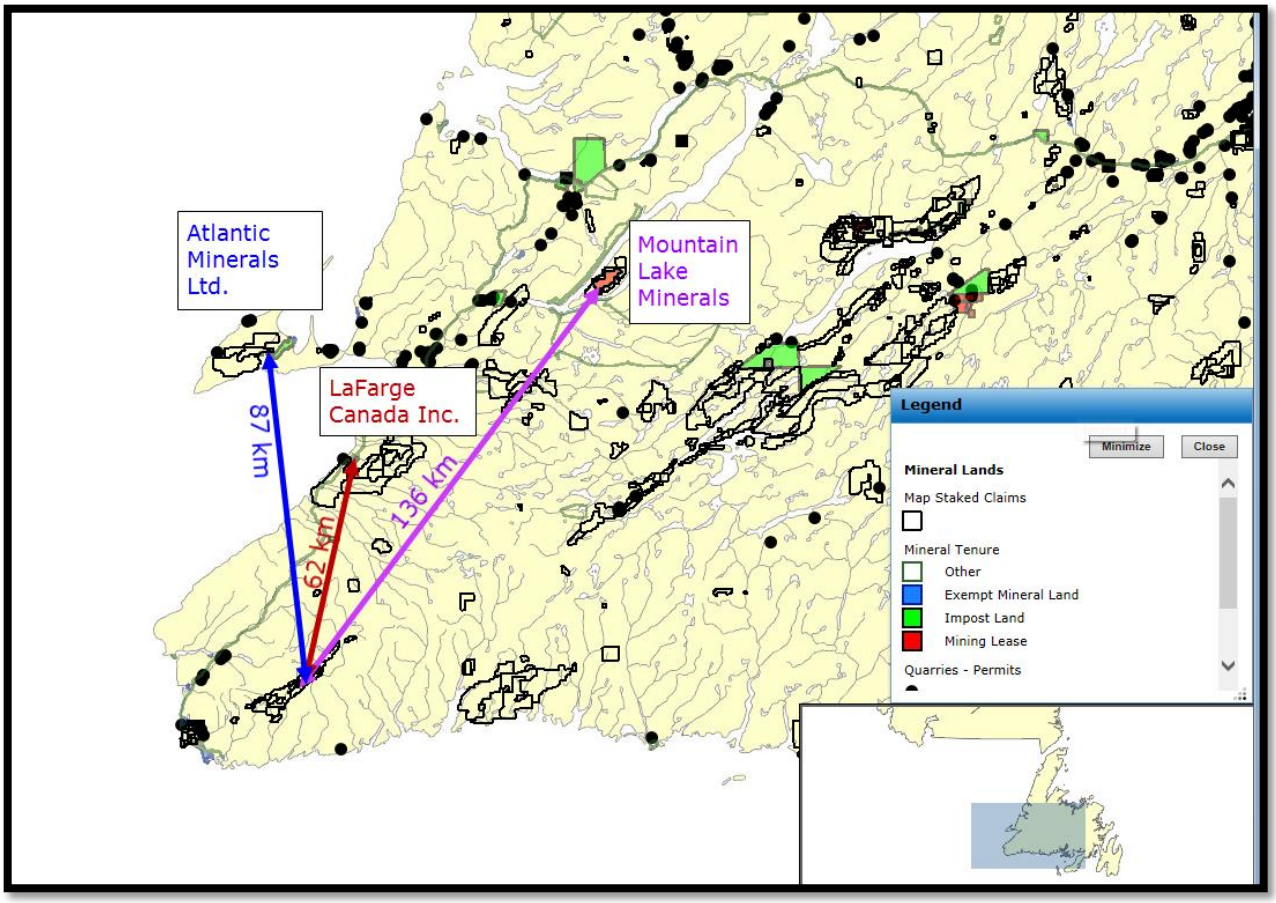

Figure 23.3 Nearest Mining Lease. Geoscience Atlas 2015.

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24. Other Relevant Data and Information

To the best of the authors' knowledge there is no other relevant data, additional information or explanation necessary to make the Report understandable and not misleading.

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25. Interpretations and conclusions

The four deposits in the Cape Ray Gold Zone include 04, 41, 51 and Windowglass Hill are the topics of interest in this report and are included in the joint venture agreement between Nordmin and Benton.

Results of the current NI 43-101 compliant mineral resource estimate for the 04, 41, 51 and Windowglass Hill Deposits are presented in Table 25.1 and Table25.2 below. The effective date of this mineral resource estimate is March 7, 2016. As it is still very early in the development stage and as mining and processing studies have not yet been finalised it is inappropriate to apply any sort of "mine design" as such would imply that any contained resources can be considered "reserves" and this is not correct and is very misleading. As a result the resources reported here are only blocks above cut-off and have had no mineability criteria applied to them.

Table 25.1 The Estimated Resources for the Cape Ray 04, 41, and 51 Deposits at a Gold cut-off grade of 4.0 g/t

Deposit	Classification	Tonnes (Mt)	Au g/t	Ag g/t	Cu%	Pb%	Zn%	Contained Au oz.	Contained Ag oz.
04	Measured Resources								
04	Indicated Resources	0.46	6.97	16.93	0.11	0.59	0.54		
04	Meas.&Indicated Resources	0.46	6.97	16.93	0.11	0.59	0.54	103,000	250,000
04	Inferred Resources	0.08	6.41	12.11	0.07	0.42	0.35	16,500	31,000
41	Measured Resources								
41	Indicated Resources	0.13	6.34	20.05	0.17	0.56	0.32		
41	Meas.&Indicated Resources	0.13	6.34	20.05	0.17	0.56	0.32	26,500	84,000
41	Inferred Resources	0.04	5.83	15.02	0.13	0.56	0.38	7,500	19,000
51	Measured Resources								
51	Indicated Resources								
51	Meas.&Indicated Resources								
WGH	Measured Resources								
WGH	Indicated Resources								
WGH	Meas.&Indicated Resources								
WGH	Inferred Resources	0.07	5.63	22.22				12,500	50,000

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Table25.2 The Estimated Resources for the Cape Ray 04, 41, 51 and Windowglass Hill Deposit at a Gold cut-off grade of 1.0g/t

Deposit	Classification	Tonnes (Mt)	Au g/t	Ag g/t	Cu%	Pb%	Zn%	Contained Au oz.	Contained Ag oz.
04	Measured Resources								
04	Indicated Resources	1.58	3.45	11.07	0.06	0.34	0.31		
04	Meas.&Indicated Resources	1.58	3.45	11.07	0.06	0.34	0.31	175,000	562,000
04	Inferred Resources	1.10	2.29	4.00	0.03	0.24	0.16	81,000	141,500
41	Measured Resources								
41	Indicated Resources	0.6	2.85	9.58	0.08	0.27	0.16		
41	Meas.&Indicated Resources	0.6	2.85	9.58	0.08	0.27	0.16	55,000	185,000
41	Inferred Resources	0.13	3.18	9.49	0.07	0.32	0.22	13,000	39,500
51	Measured Resources								
51	Indicated Resources								
51	Meas.&Indicated Resources								
51	Inferred Resources	0.88	5.08	11.82	0.04	0.12	0.06	144,000	334,500
WGH	Measured Resources								
WGH	Indicated Resources								
WGH	Meas.&Indicated Resources								
WGH	Inferred Resources	1.80	1.38	8.49				80,000	491,000

H) The mineral resources are estimated in conformance with the Canadian Institute of Mining, Metallurgy and Petroleum – Standards on Mineral Resources and Reserves – Definitions and Guidelines, as referenced in NI 43-101

I) Assay composites were capped at 40 g/t Au.

J) A density value of 2.84 g/cm^3 was used for all deposits.

K) The rounding of tonnes as required by NI 43-101 reporting guidelines may result in apparent differences between tonnes, grade and contained ounces.

L) Mineral resources that are not mineral reserves do not have demonstrated economic viability. This estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing and other relevant issues.

M) The quantities and grades of reported inferred mineral resources are uncertain in nature and further exploration may not result in their upgrading to Indicated or Measured status.

N) Mineral resources are reported unconstrained by any pit shell for surface mining methods or any preliminary underground mining outline for underground methods.

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Historical metallurgical test campaigns align well with recent bench scale test campaigns. Under Benton's operation, samples were selected for metallurgical testing. At a lab scale level, extractable gold can be as high as 97% and extractable silver between 50 and 70% with cyanide leach. Gravity recoverable gold has shown potential to be greater than 80%. Further work is required for flowsheet development and optimization.

Based on the overall size and characteristics of the mineralized zones known to date, the Cape Ray Fault Zone has the demonstrated potential for occurrence of potentially mineable gold resources. However due to the geologic uncertainty, definition of the zone limits and grade trends require further drill definition at closely spaced intervals.

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26. Recommendations

It is of the opinion of the author that the property has potential to host mineable gold deposits. Based on the findings of this technical report, it is recommended that additional drilling be completed so that the existing Inferred Resources can be upgraded to Indicated Resources status as defined by NI 43-101.

Nordmin is proposing a two phase approach such that results from Phase 1 will be the baseline for Phase 2.

Phase 1 consists of completing surface stripping and mapping with infill drilling down to the 150 m level and beginning environmental baseline studies. The environmental estimate includes all labour, equipment and materials to carryout hydrology and surface water baseline studies. The location of diamond drill holes that meet the requirements of the geotechnical study would be incorporated

Phase 2 would include deeper drilling to provide inputs to the geological model down to the 300 m level as well as some infrastructure upgrades and continued environmental work. The road in to the property requires some cut and fill to reduce steep sections and a crushed rock cap to be a passable single lane road. Fulfilling the recommended drilling should provide sufficient information for a resource update and pre-feasibility study (PFS). A proposed budget for Phase 1 and 2 is shown below in Table 26.1.

Table 26.1 Proposed Budget

Phase1	**Estimated Cost (CAD$)**
Drilling down to 150 m level plus infill drilling	$4,175
Environmental baseline studies, hydrology and oriented core*	$660
Contingency (10%)	$483
Subtotal	**$5,318**
Phase 2	
Drilling down to 300 m level plus infill drilling and oriented core	$7,350
Upgrade to road infrastructure	$560
Updated mineral resource estimate and Prefeasibility study*	$400
Contingency (10%)	$831
Subtotal	**$9,141**
Total	**$14,460**

*Provided by Nordmin Engineering in conjunction with existing JV agreement

Note: Estimated costs are inclusive of drilling rates, stand by and work rates, camp support, consumables, laboratory charge and supervision.

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28. Certificates

Certificate of Qualifications

I, Alan James Aubut, do hereby certify the following:

I am a co- author of this technical document titled “Preliminary Economic Assessment Technical Report, for the Cape Ray Property: 04, 41, 51, and Windowglass Hill Deposits, 58° 55’ 47” W Longitude ,47° 45’ 3” N Latitude”, (the report), with an effective date of March 7, 2016.

I take responsibility for sections 1-12, 14, 20 and 23-26 of the report.

I visited the site on September 22, 2014.

As of the effective date of the report, the report is to the best of my knowledge, information and belief contains all scientific and technical information that is required to be disclosed in order to make the report not misleading.

Based on the following training and experience I am a “qualified person” for the purposes of NI43-101:

I am a graduate of Lakehead University, in Thunder Bay, Ontario with the degree of Honours Bachelor of Science, Geology (1977).

I am a graduate of the University of Alberta, in Edmonton, Alberta with the degree of Master of Science, Geology (1979).

I hold an Applied Geostatistics Citation through the Faculty of Extension of the University of Alberta, in Edmonton, Alberta.

I have been actively practicing geology since 1979.

I have been practicing mineral resource estimation since 2000.

I am independent of Benton Resources Inc. as defined by Section 1.5 of NI 43-101 and do not expect to become an insider, associate or employee of the issuer.

I am independent of Nordmin Engineering Ltd. as defined by Section 1.5 of NI 43-101 and do not expect to become an insider, associate or employee of the company.

I am currently a member in good standing of the Association of Professional Geoscientists of Ontario.

From 2000 to 2009 I was a member in good standing of the Association of Professional Engineers and Geoscientists of Manitoba.

I am a member of the Society of Economic Geologists.

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I operate under the business name of Sibley Basin Group Geological Consulting Services Ltd., a business independent of Benton Resources Inc. and do not expect to become an insider, associate or employee of the issuer.

The business address of Sibley Basin Group Geological Consulting Services Ltd. is:

Sibley Basin Group

PO Box 304

300 First St. West

Nipigon, ON

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[Signed] [Sealed]

Alan Aubut

April 16, 2016

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Certificate of Qualifications

I, Paul Bonneville, do hereby certify the following:

- I am a co- author of this technical document titled "Preliminary Economic Assessment Technical Report, for the Cape Ray Property: 04, 41, 51, and Windowglass Hill Deposits, 58° 55' 47" W Longitude ,47° 45' 3" N Latitude", (the report), with an effective date of March 7, 2016.
- I take responsibility for sections 15, 16, 18, 19, 21 and 22 of the report.
- As of the effective date of the report, the report is to the best of my knowledge, information and belief contains all scientific and technical information that is required to be disclosed in order to make the report not misleading.
- Based on the following training and experience I am a "qualified person" for the purposes of NI43-101:
- I am independent of Benton Resources Inc. as defined by Section 1.5 of NI 43-101 and do not expect to become an insider, associate or employee of the issuer.
- I am independent of Nordmin Engineering Ltd. as defined by Section 1.5 of NI 43-101 and do not expect to become an insider, associate or employee of the company.
- I am currently a member in good standing of the Order of Engineers of Quebec.
- I operate under the business name of PRB Mining Services Inc., a business independent of Benton Resources Inc. and do not expect to become an insider, associate or employee of the issuer.
- The business address of PRB Mining Services is:
  152 rue Perreault, Suite 100
  Val d'Or, QC
  Canada

Paul Bonneville

April 20, 2016

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Certificate of Qualifications

- I, John Folinsbee, do hereby certify the following:
- I am a co- author of this technical document titled "Preliminary Economic Assessment Technical Report, for the Cape Ray Property: 04, 41, 51, and Windowglass Hill Deposits, 58° 55' 47" W Longitude ,47° 45' 3" N Latitude", (the report), with an effective date of March 7, 2016.
- I take responsibility for sections 13 and 17 of the report.
- As of the effective date of the report, the report is to the best of my knowledge, information and belief contains all scientific and technical information that is required to be disclosed in order to make the report not misleading.
- Based on the following training and experience I am a "qualified person" for the purposes of NI43-101:
- I am independent of Benton Resources Inc. as defined by Section 1.5 of NI 43-101 and do not expect to become an insider, associate or employee of the issuer.
- I am independent of Nordmin Engineering Ltd. as defined by Section 1.5 of NI 43-101 and do not expect to become an insider, associate or employee of the company.
- I am currently a member in good standing of the Professional Engineers and GeoScientists of British Columbia, Professional Engineers Ontario and CIMM.
- I operate under the business name of Heads Ore Tails Metallurgical Consulting, a business independent of Benton Resources Inc. and do not expect to become an insider, associate or employee of the issuer.
- The business address of is:

Heads Ore Tails Metallurgical Consulting Inc

3726 Overlander Drive

Kamloops British Columbia

V2B 8M4

[Signed] [Sealed]

John Folinsbee

April 16, 2016

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Appendices

Appendix A Exploratory Data Analysis

Appendix B Block Model Plans and Sections

Appendix C Model Validation

Appendix D Variance Correction Theory

Appendix E Resource Classification

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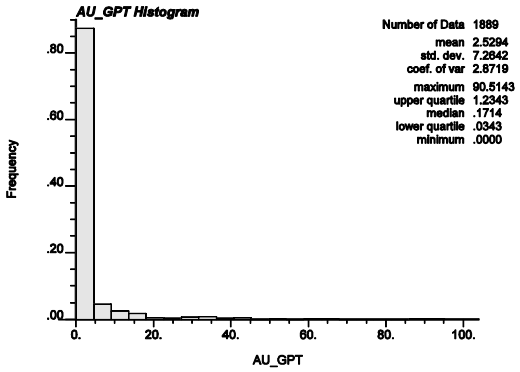
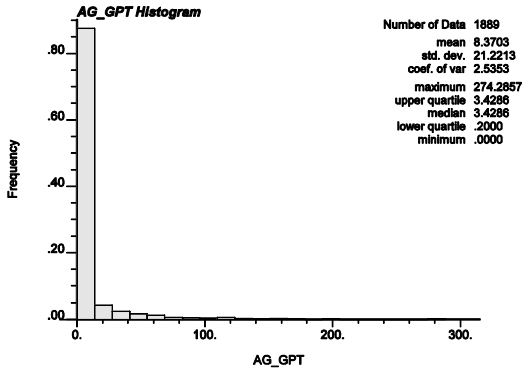
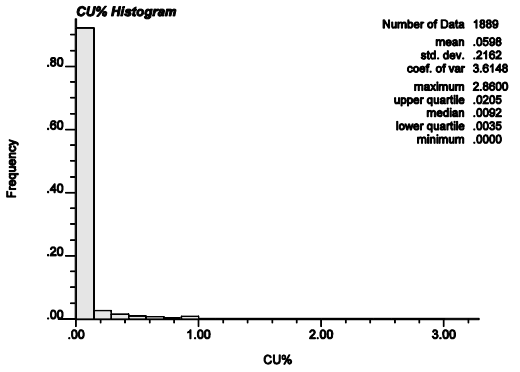
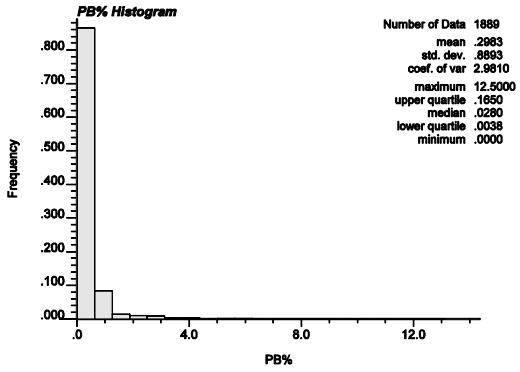
Appendix A Exploratory Data Analysis

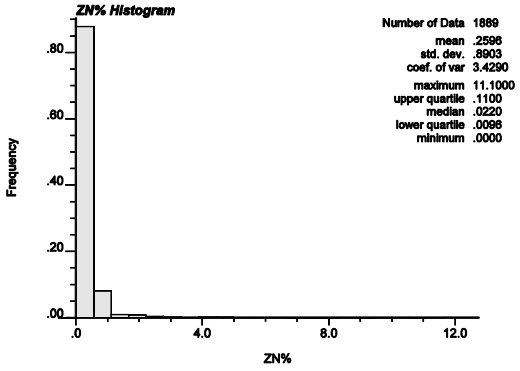
04 Deposit Histograms











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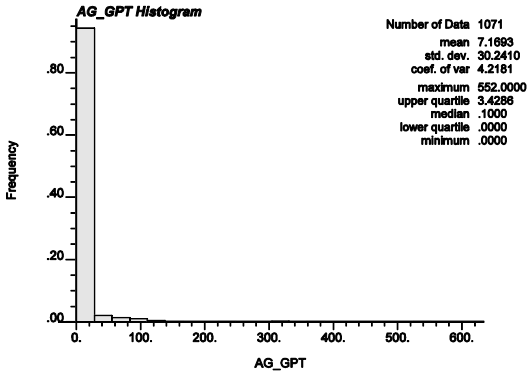
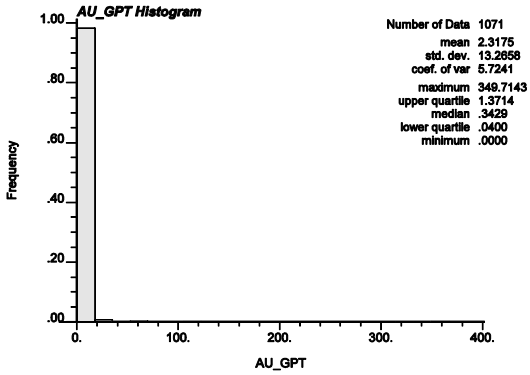
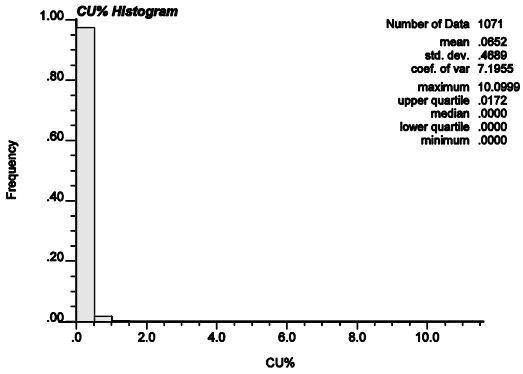
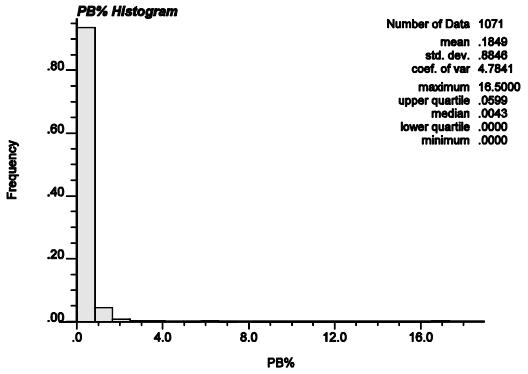
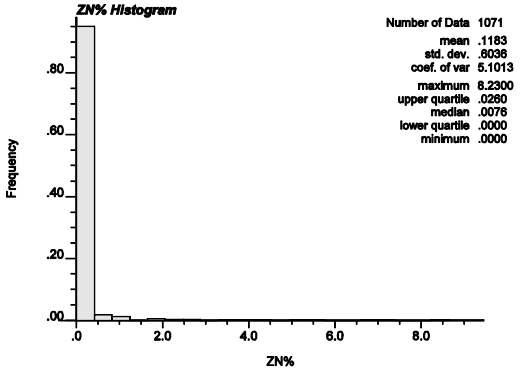
41 Deposit Histograms–Domain1











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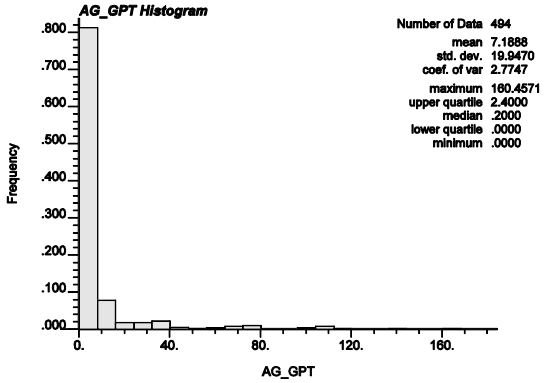
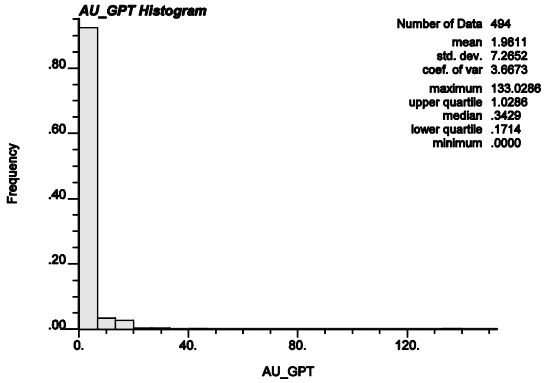
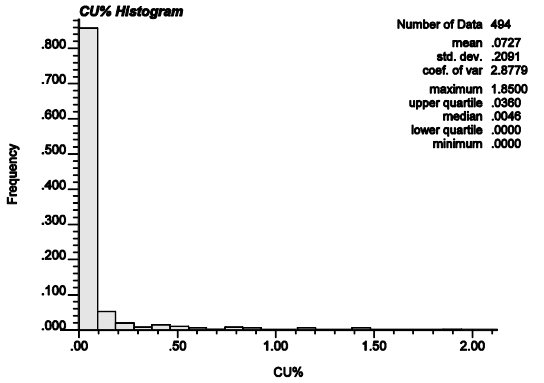
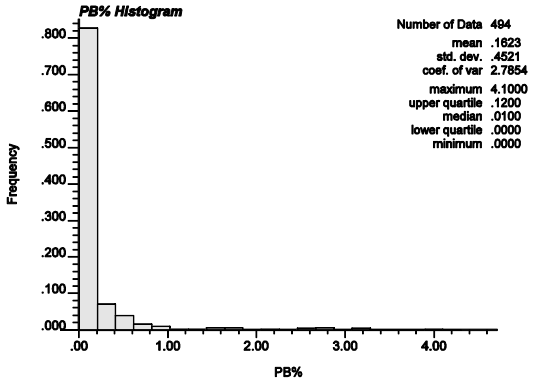
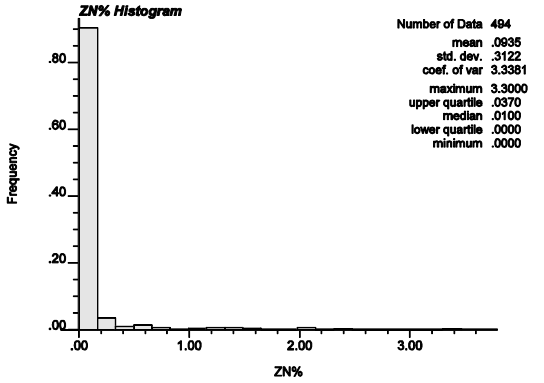
41 Deposit Histograms–Domain2











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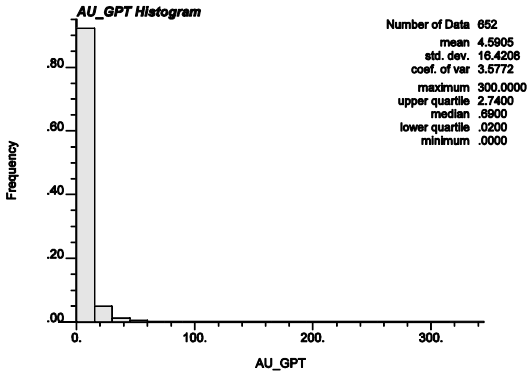
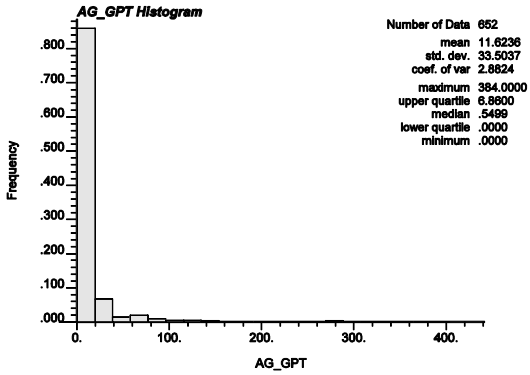
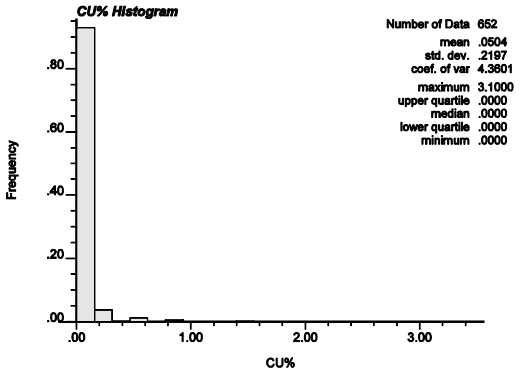
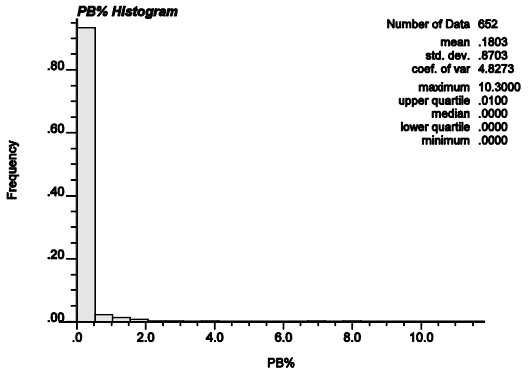
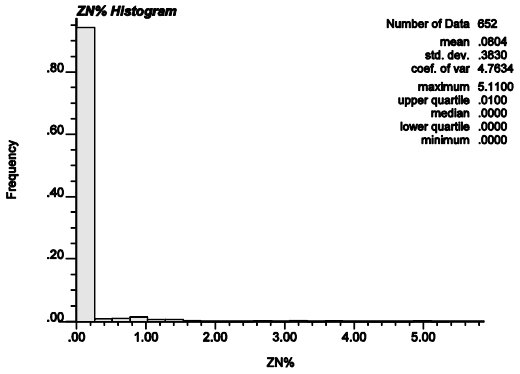
51Deposit Histograms











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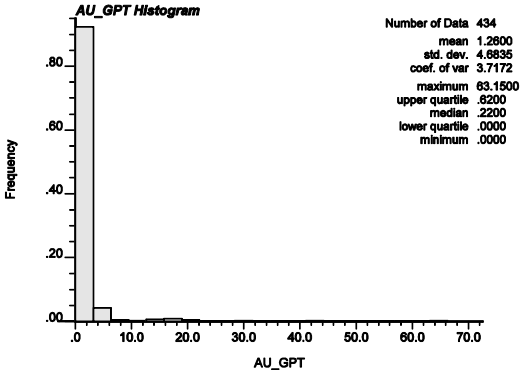
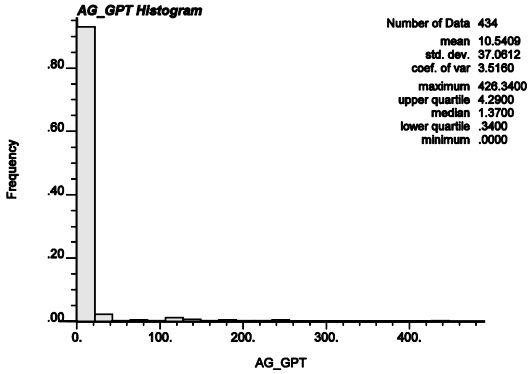
Windowglass Hill Deposit Histograms–Lower Domain





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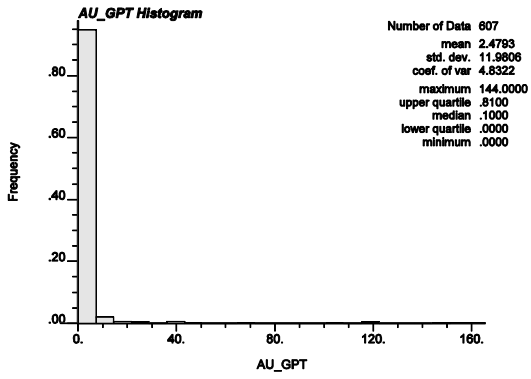
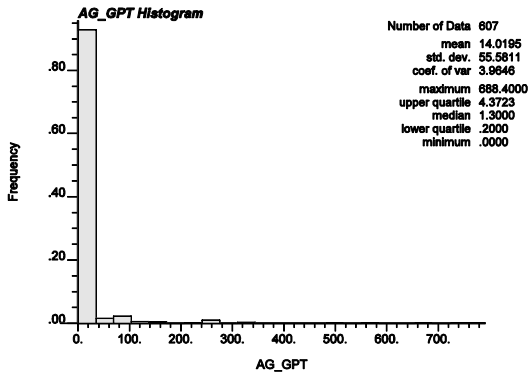
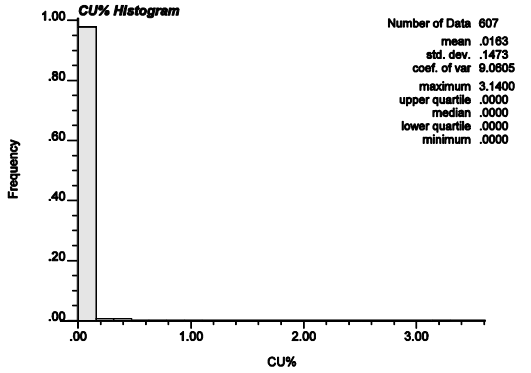
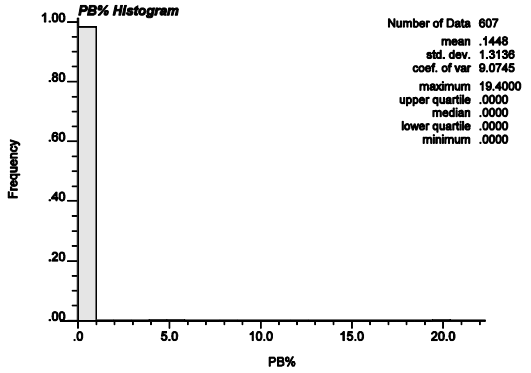
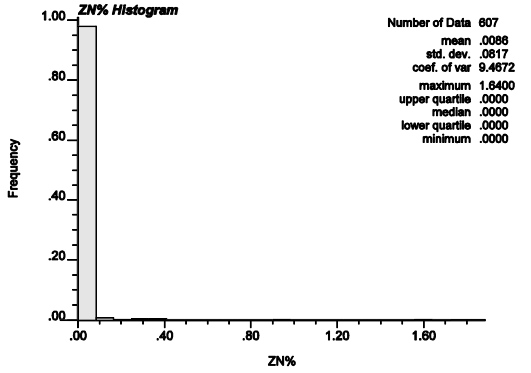
Windowglass Hill Deposit Histograms–Upper Domain











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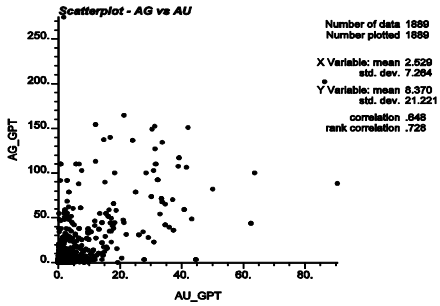
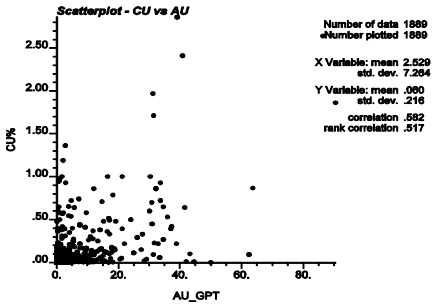
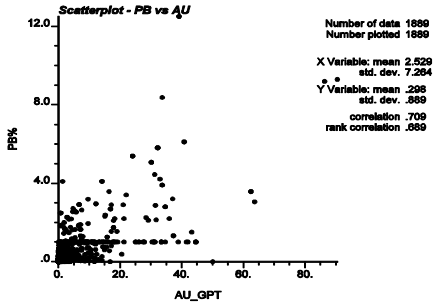
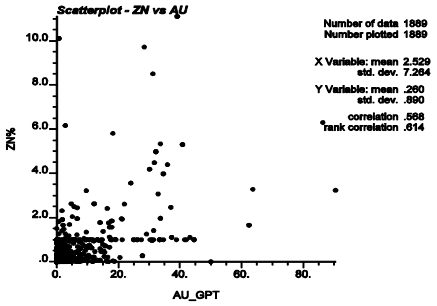
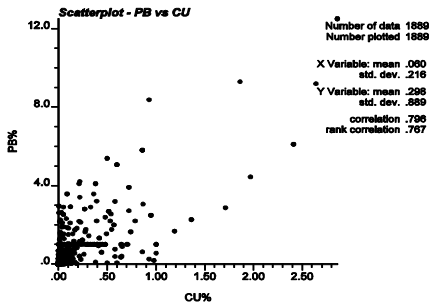
04 Deposit Scatter Plots

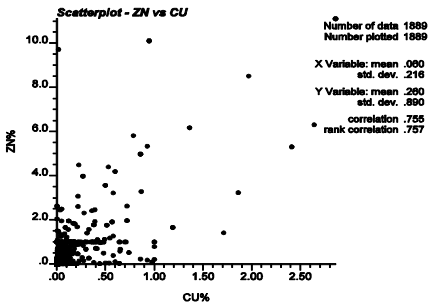


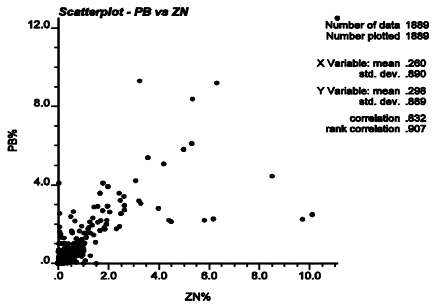












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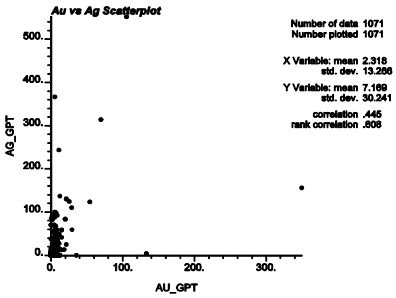
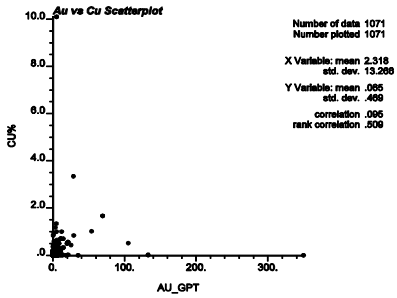
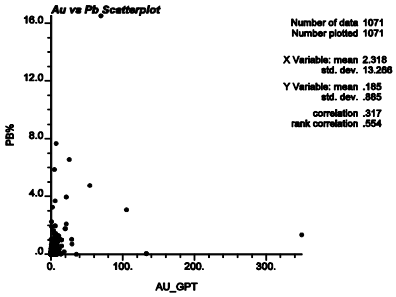
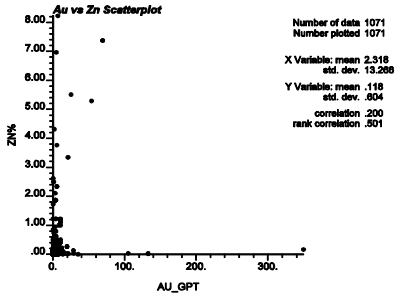
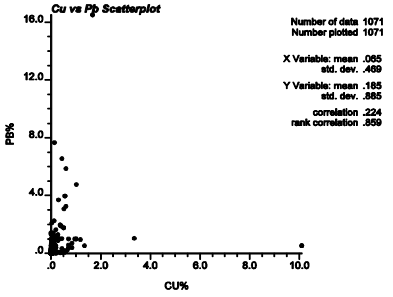
41Deposit Scatter Plots–Domain1

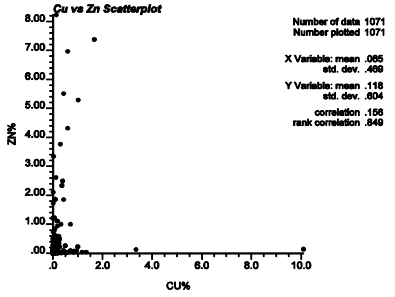


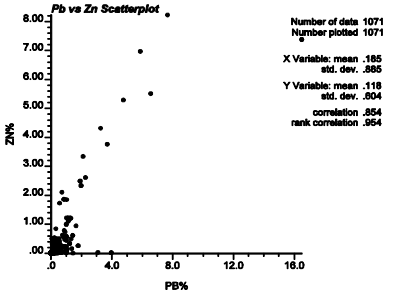












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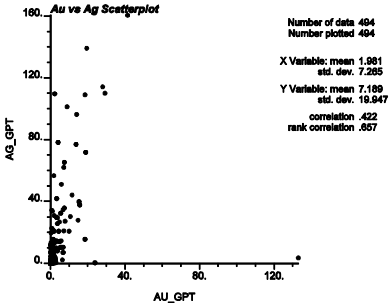
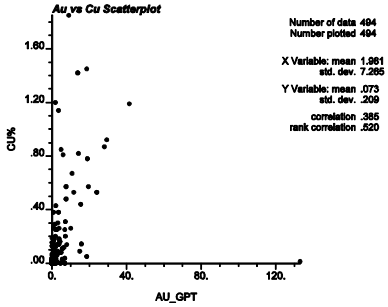
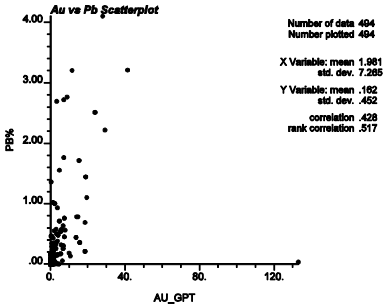
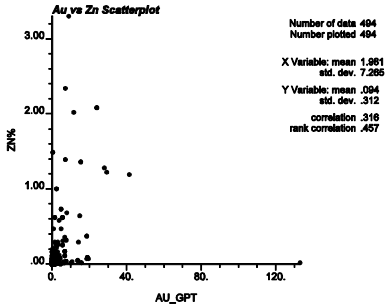
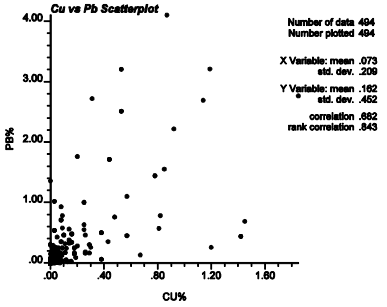
41Deposit Scatter Plots–Domain2

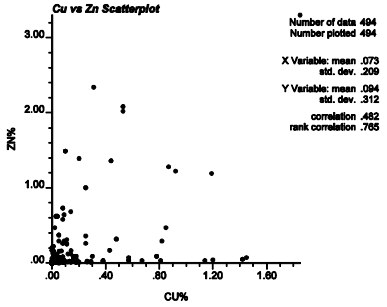


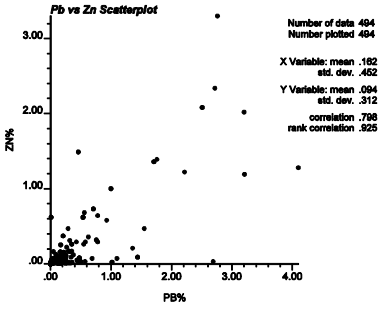












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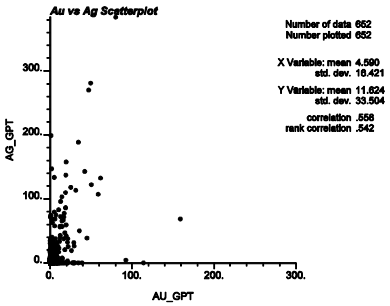
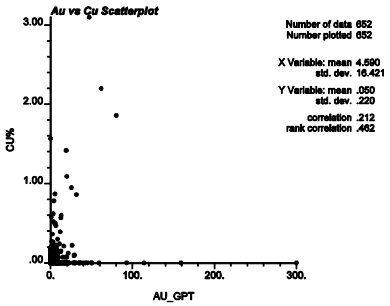
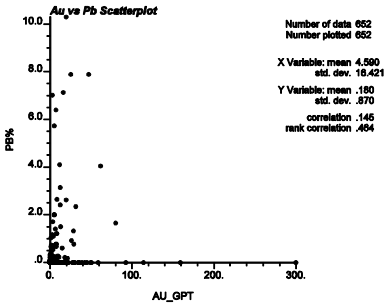
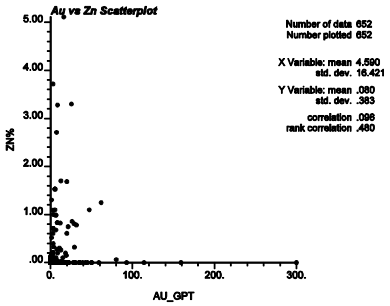
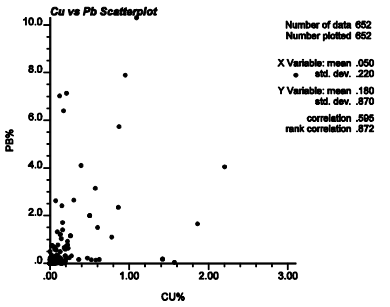
51Deposit Scatter Plots

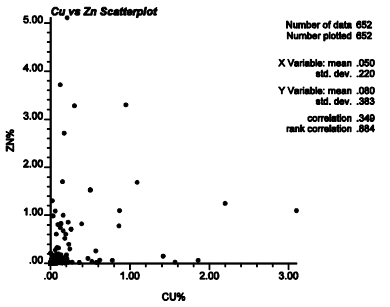


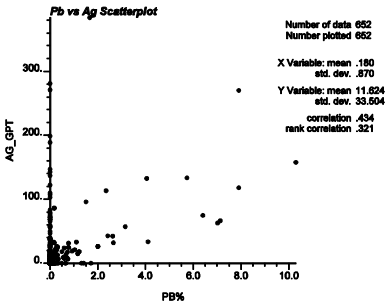


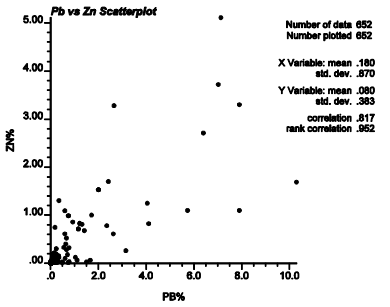












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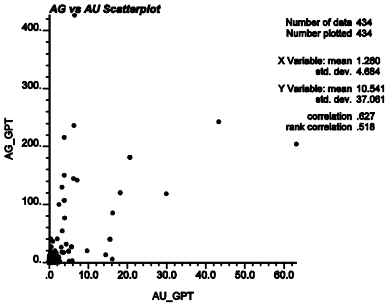
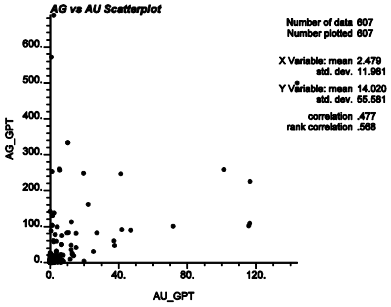
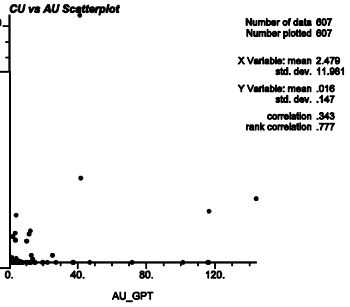
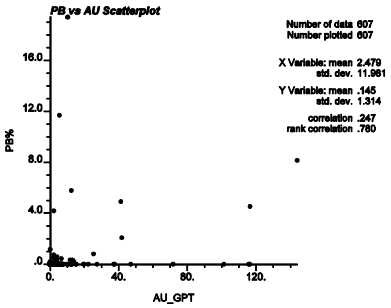
Windowglass Hill Deposit Scatter Plots

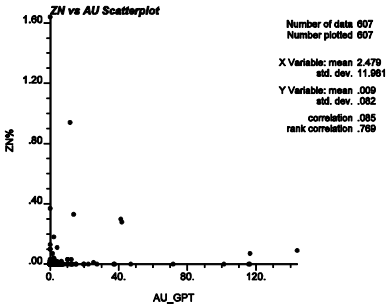




Scatter plots for the Lower Domain are on the Left and for the Upper Domain, on the Right.







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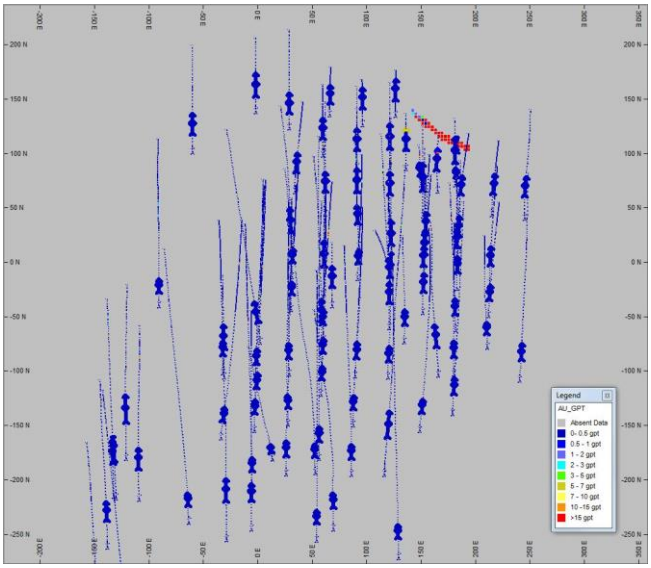
Appendix B Block Model Plans and Sections

04 Deposit NN MODELS Plan Views

275 Elev.



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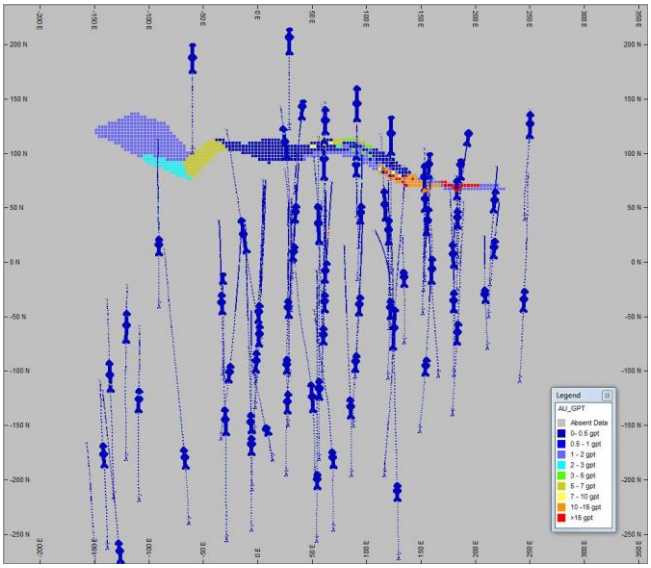

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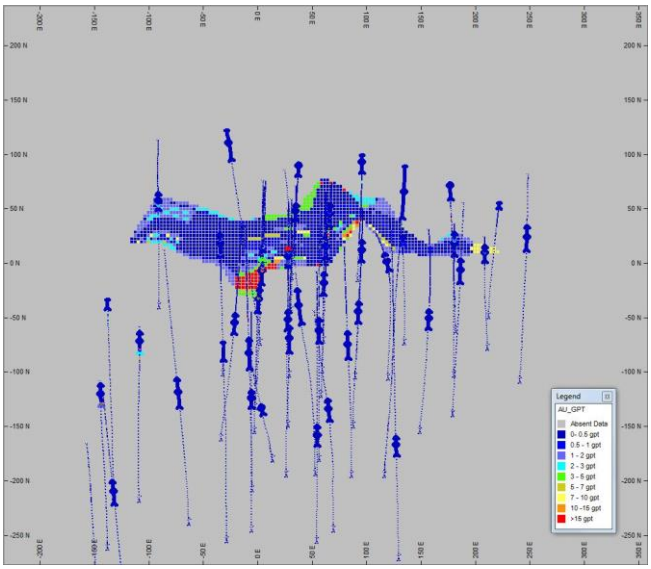
125 Elev.



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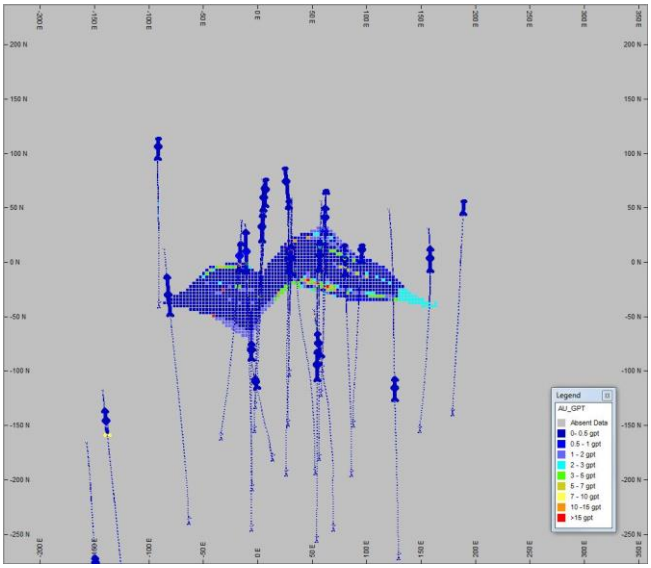

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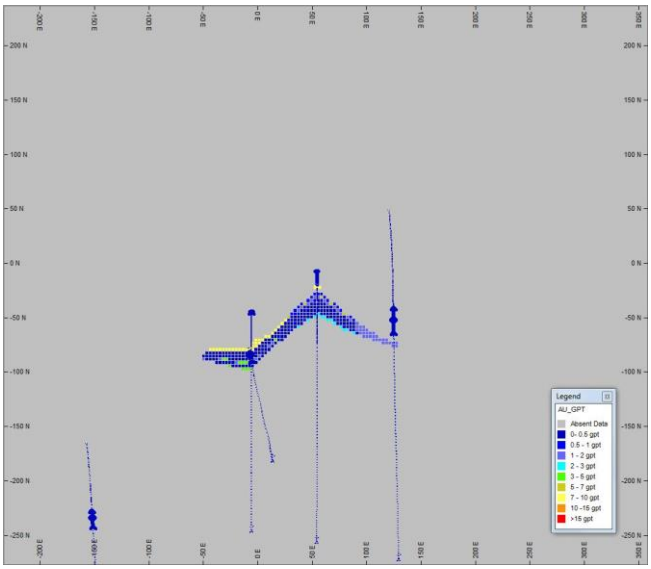

04 Deposit IK Models: Plan views

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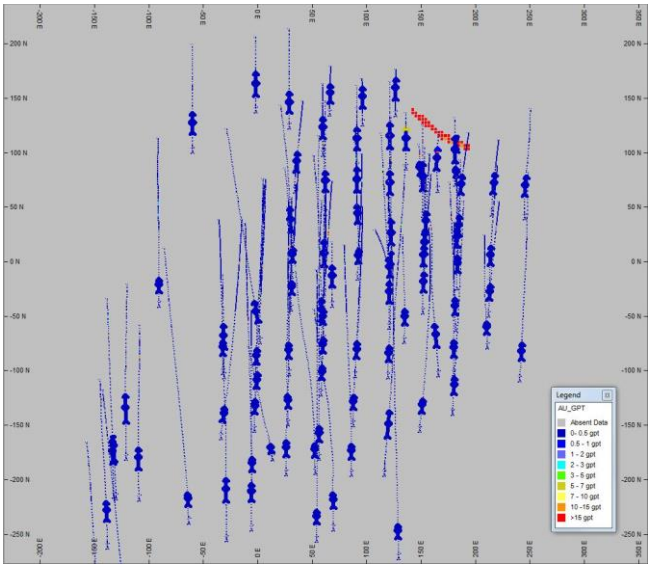

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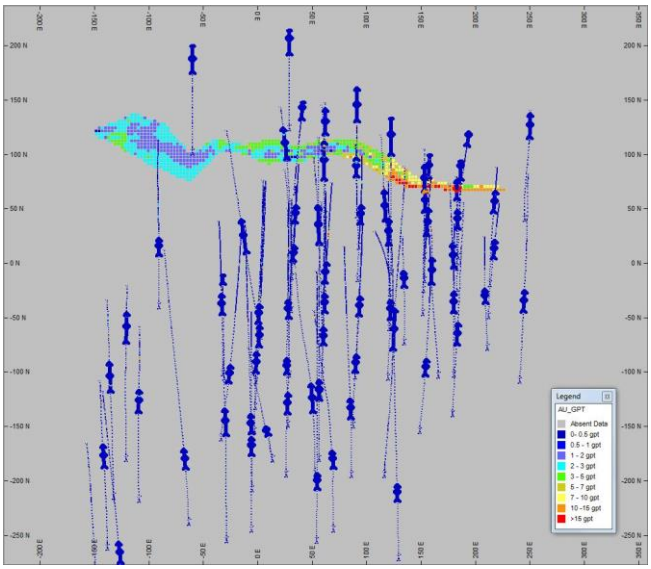
200 Elev.



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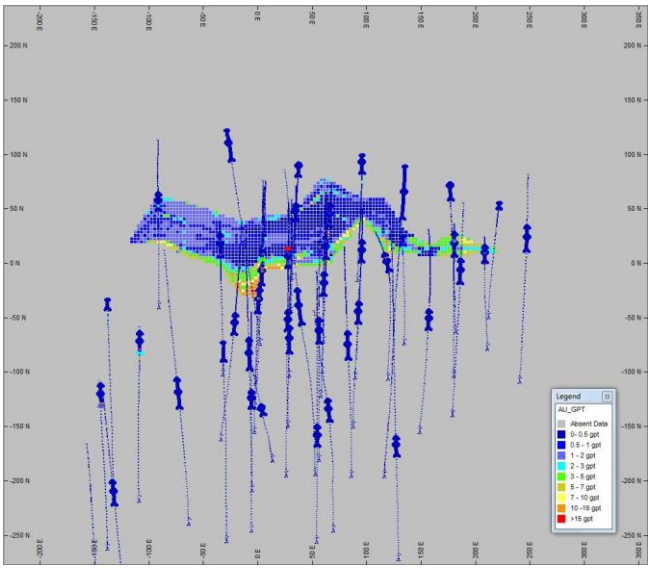

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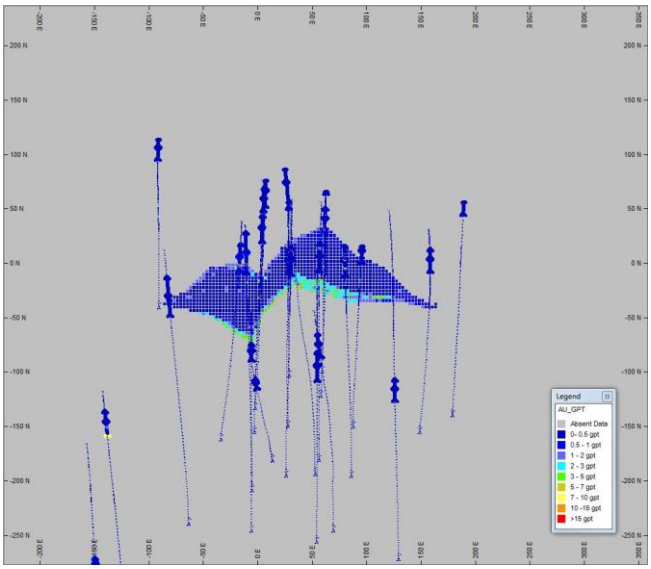
050 Elev.



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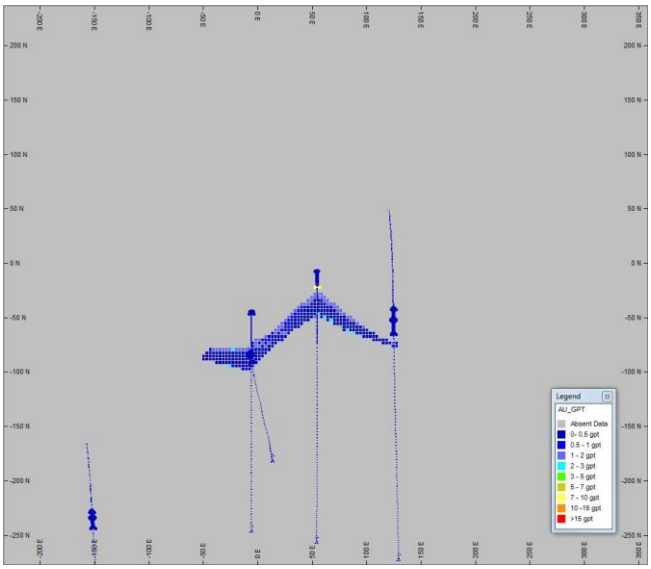

04 Deposit NN Model – N-S Sample Sections

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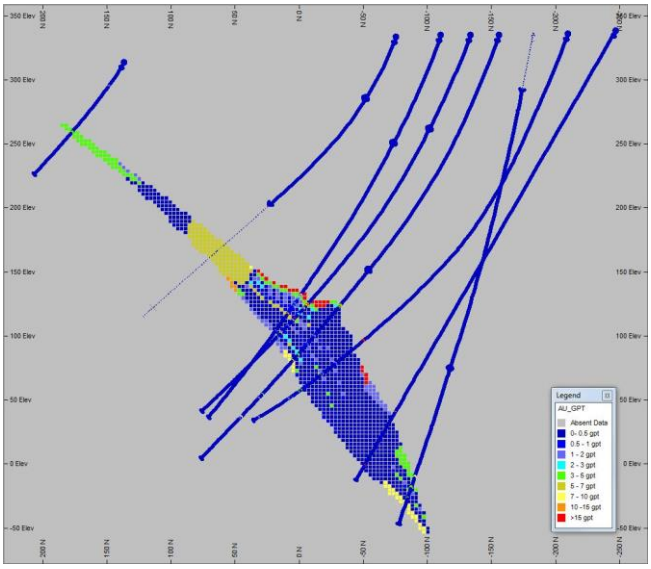

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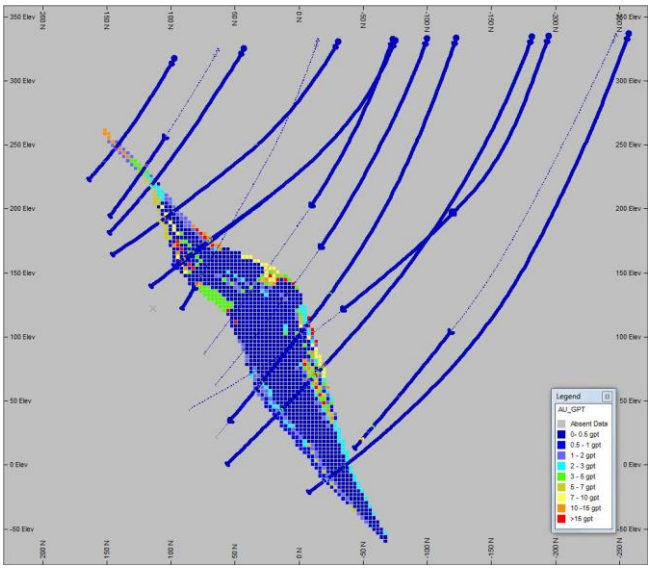

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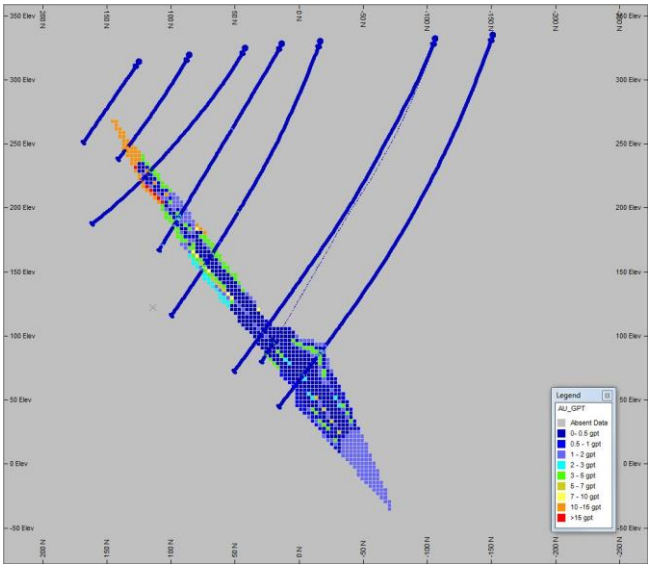
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AU_GPT
Absent Data
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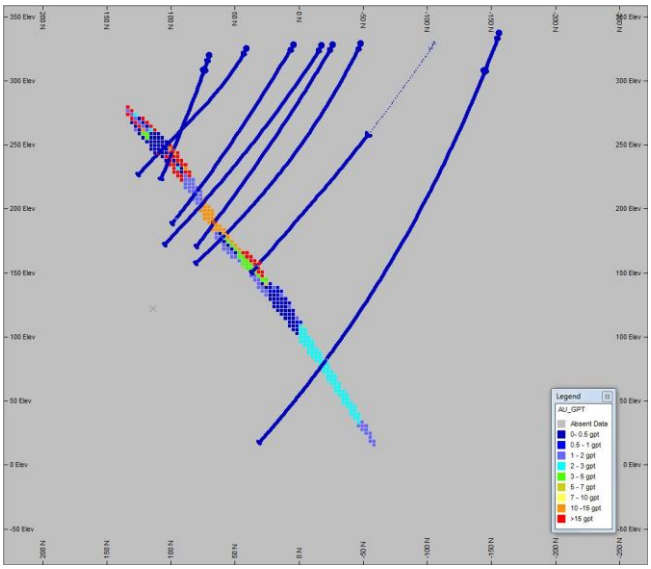
0.5 - 1 gpt
1 - 2 gpt
2 - 3 gpt
3 - 5 gpt
5 - 7 gpt
7 - 10 gpt
10 - 15 gpt
>15 gpt

150 E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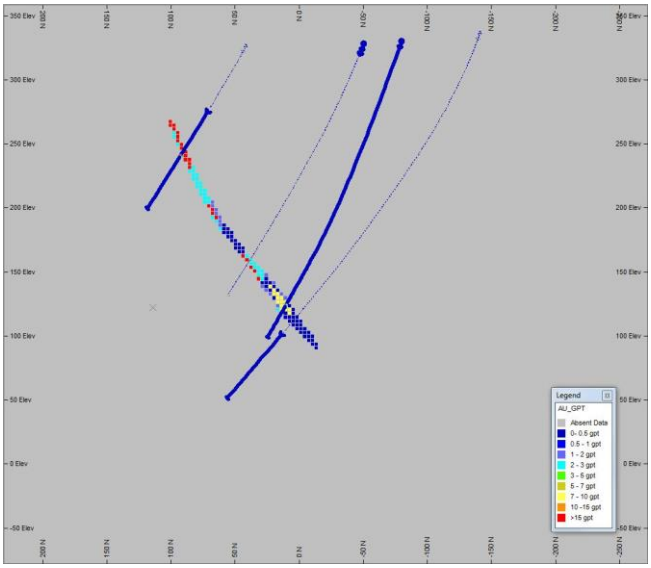

04 Deposit IK Models – N-S Sample Sections

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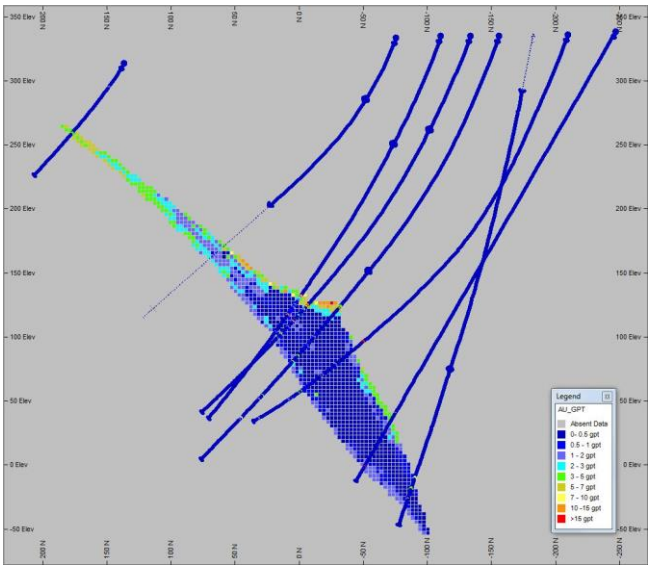

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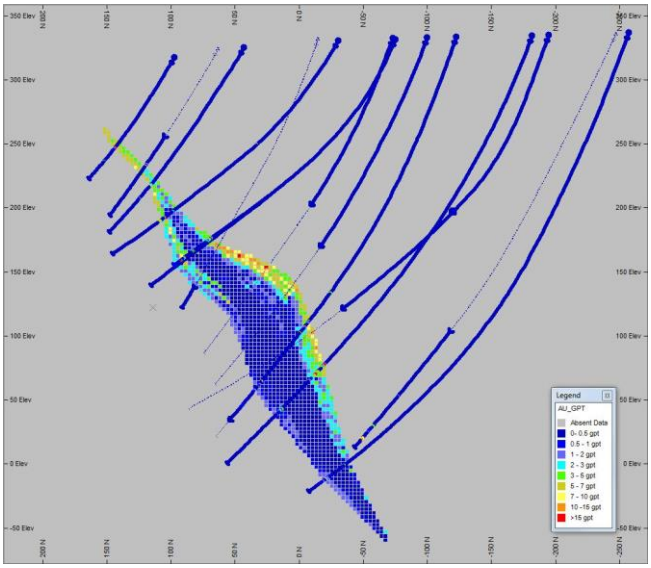
50 E



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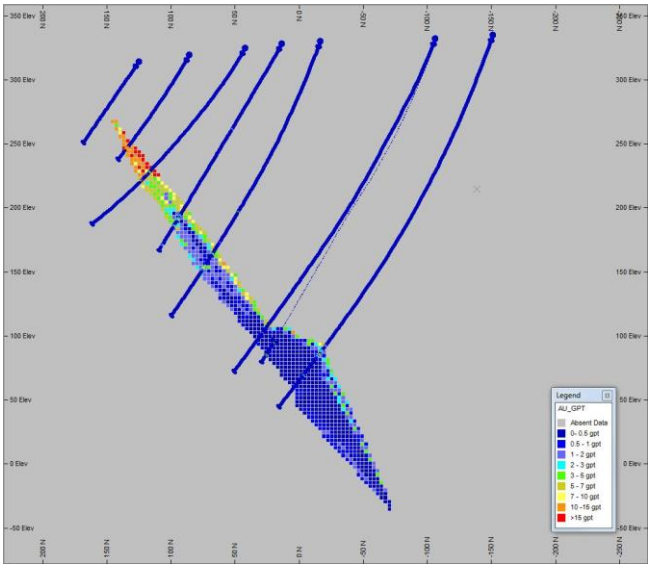

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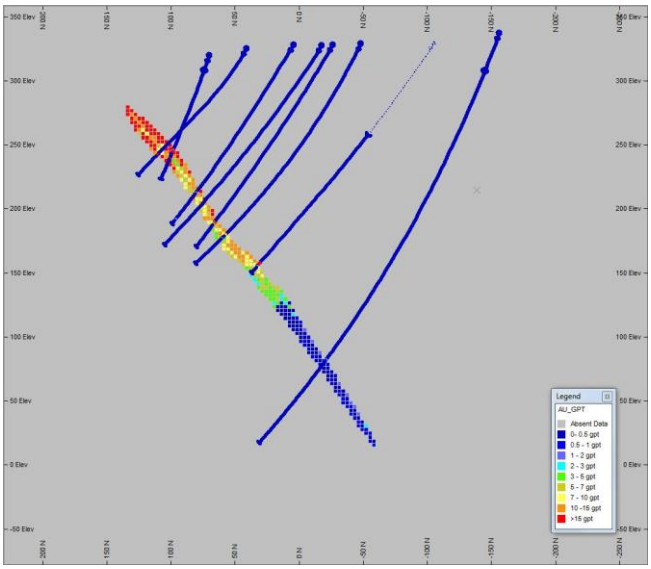
150 E



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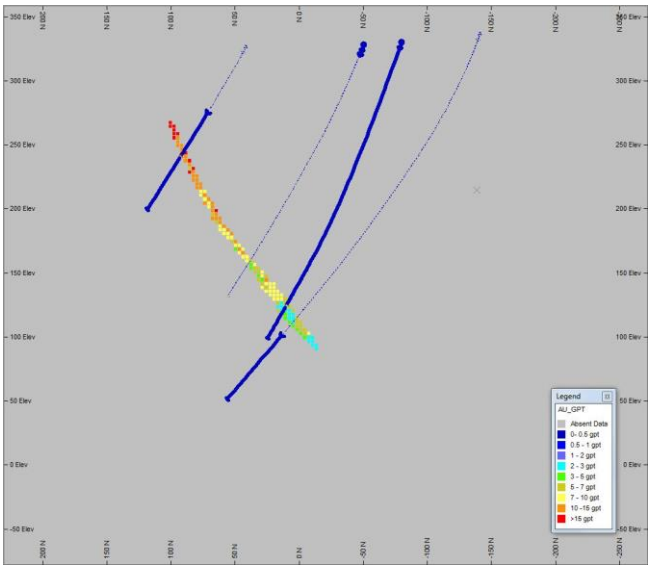

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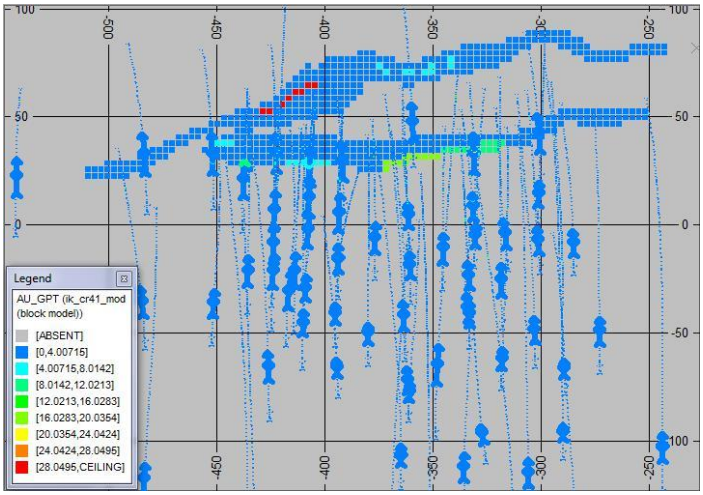
41 Deposit NN Models Plan views

300 Elev.



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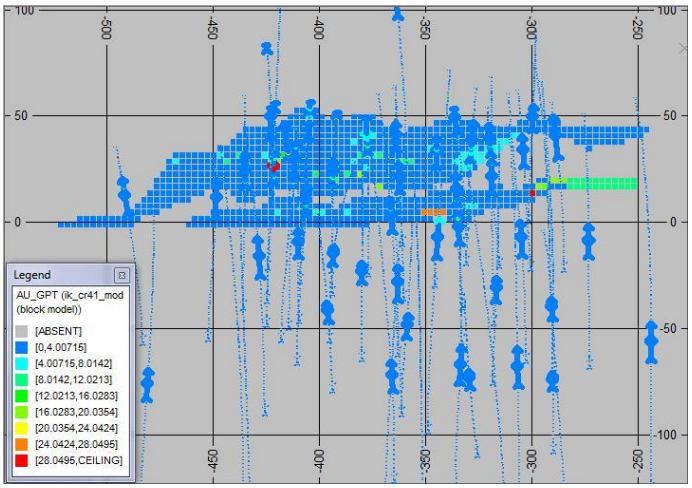

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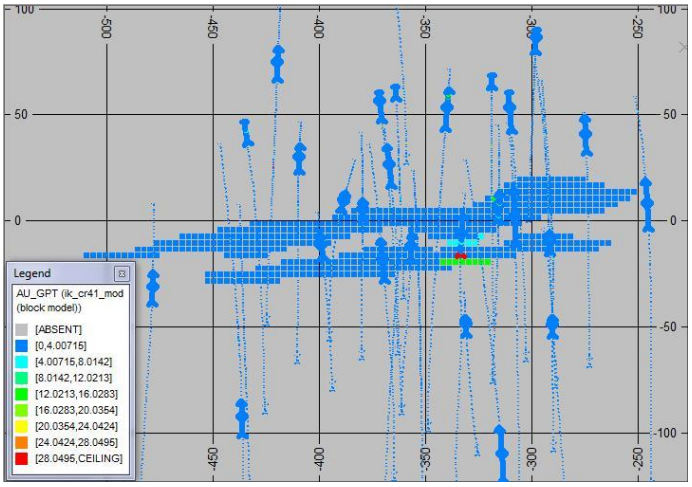

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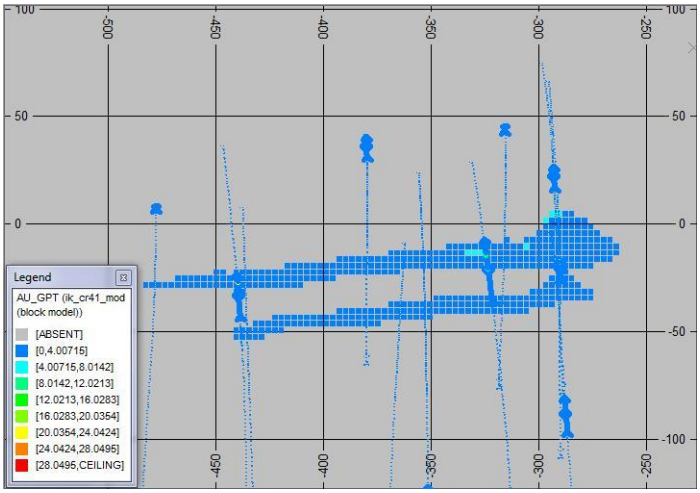

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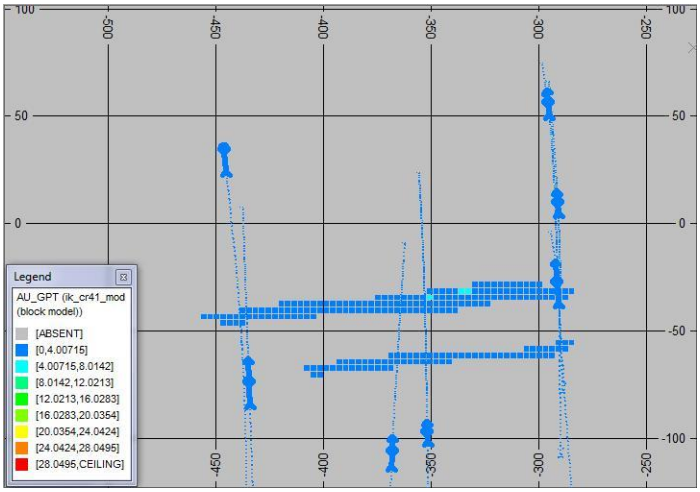

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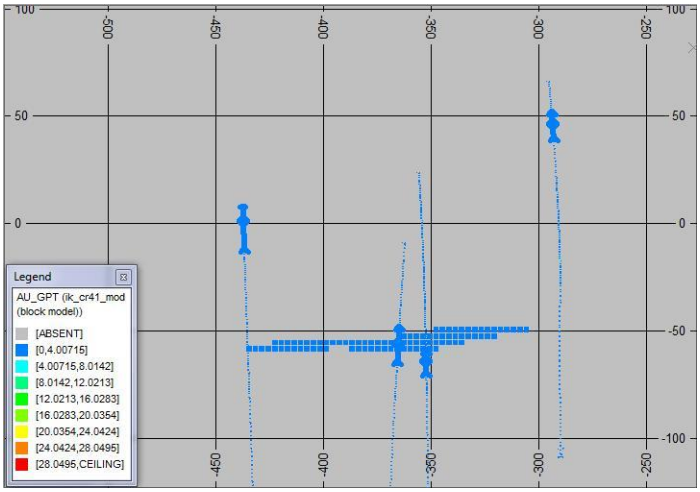

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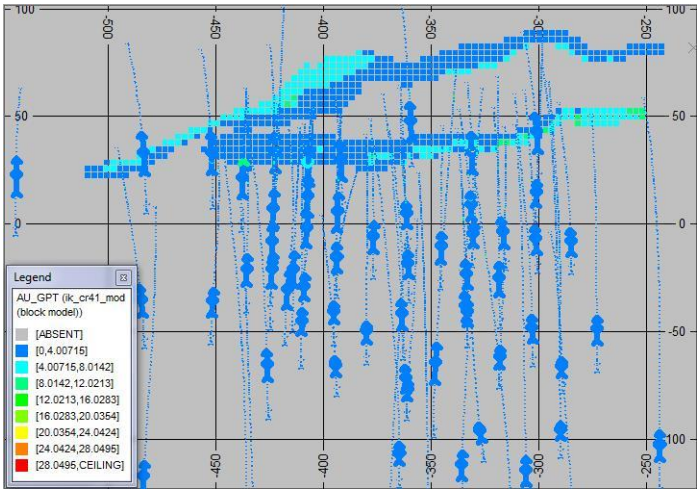
41 Deposit IK Models: Plan views

300 Elev.



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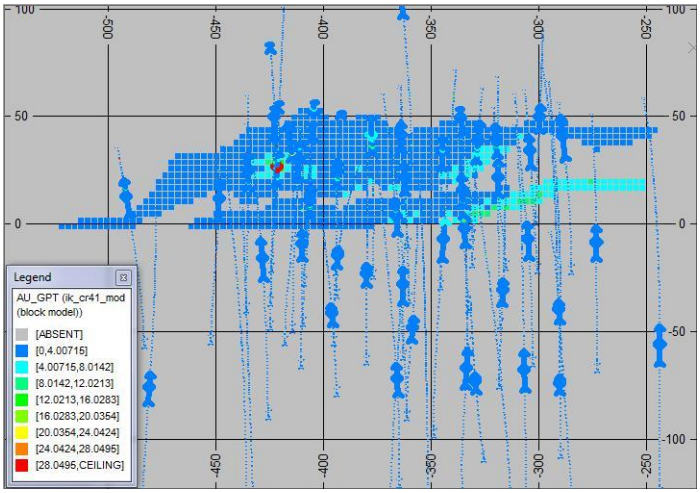

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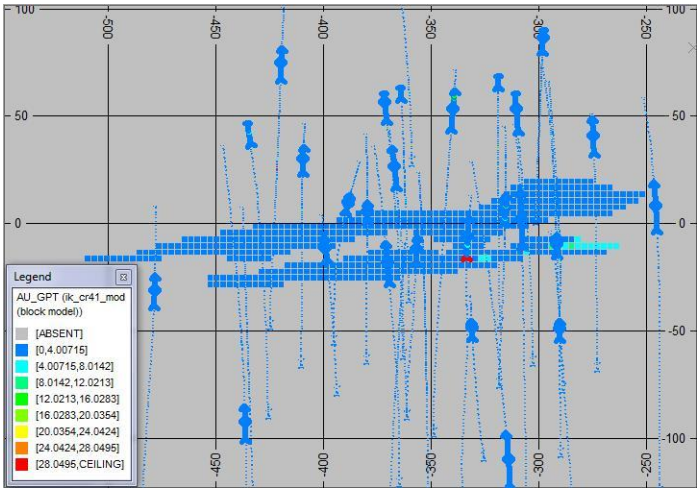
200 Elev.



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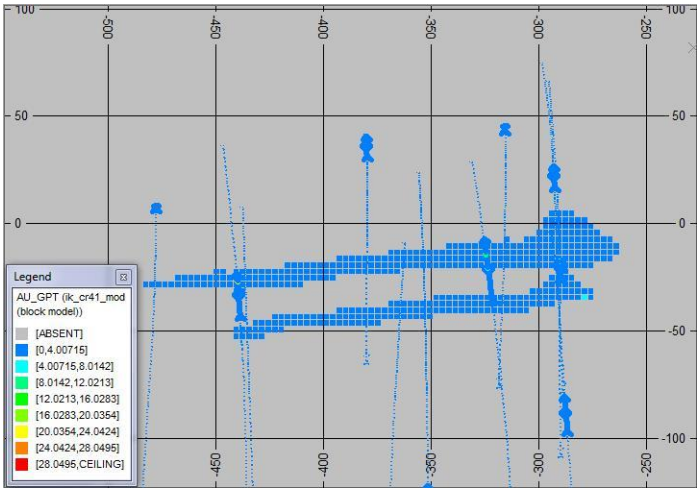

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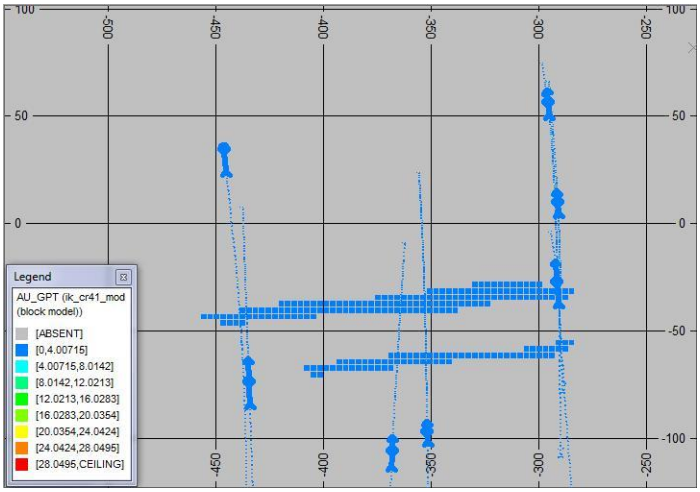
100 Elev.



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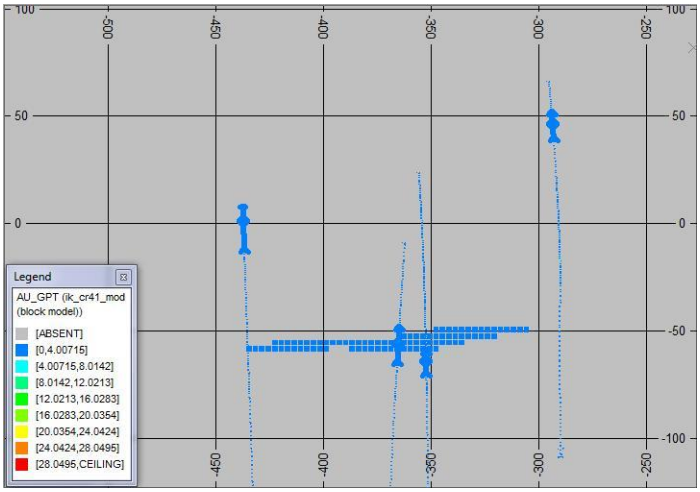

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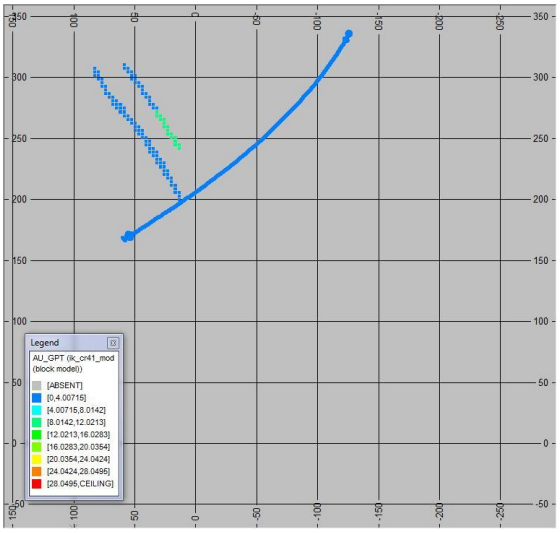
41 Deposit NN Model – N-S Sample Sections

250W



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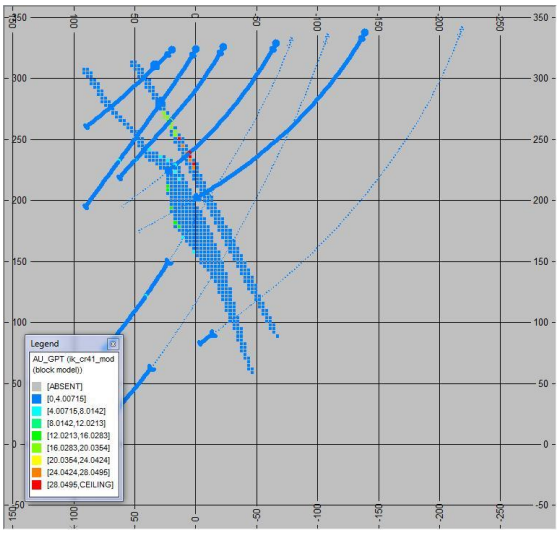

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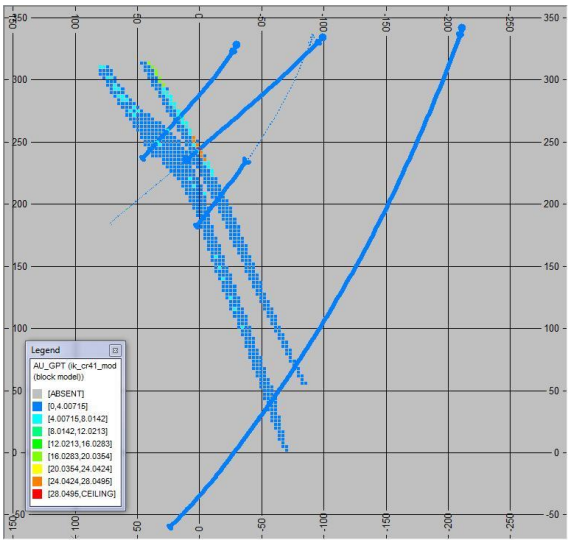
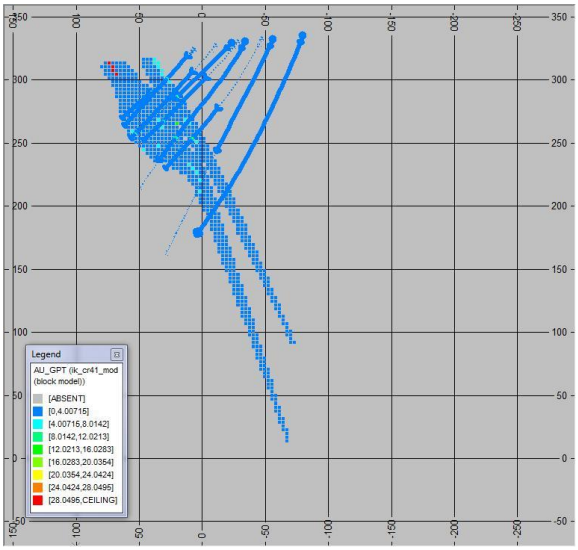
350W



400W



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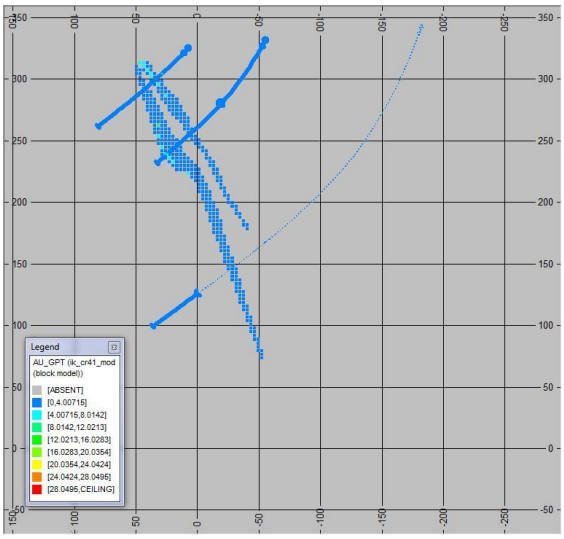
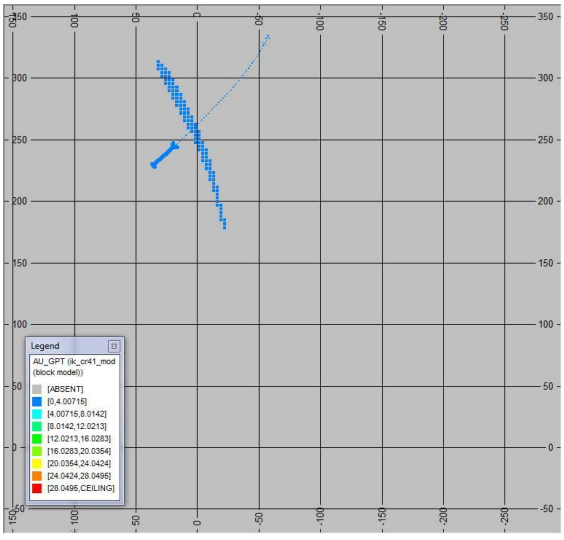
450W



500W



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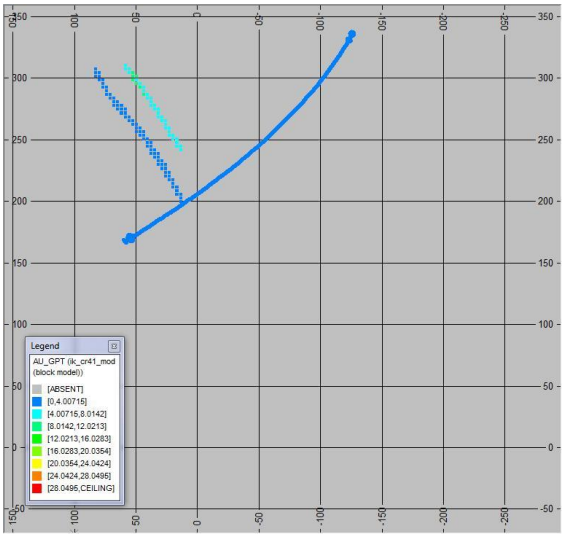
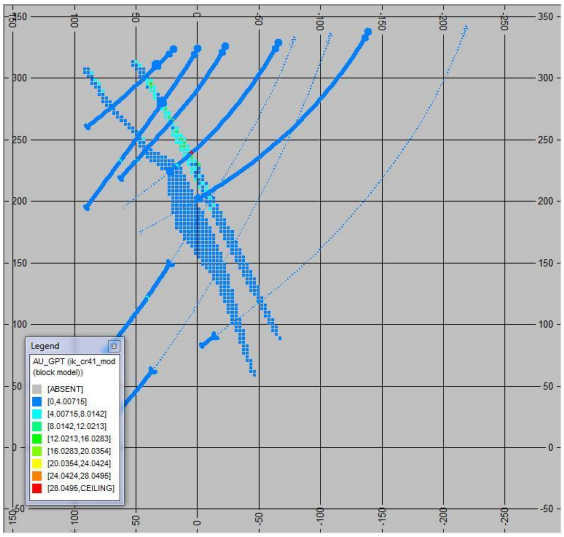
41 Deposit IK Models – N-S Sample Sections

250W



300W



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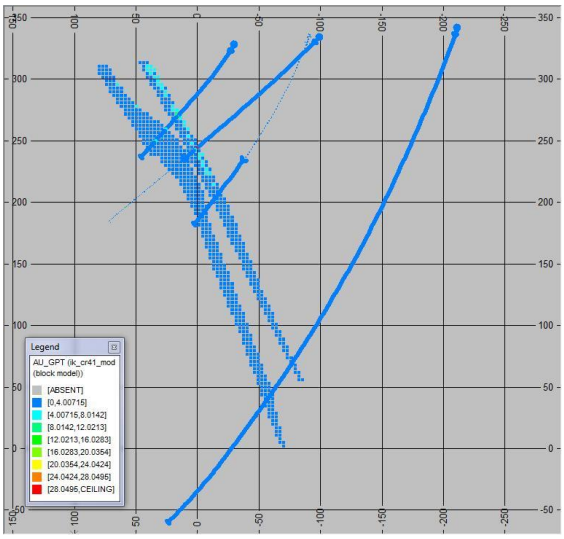
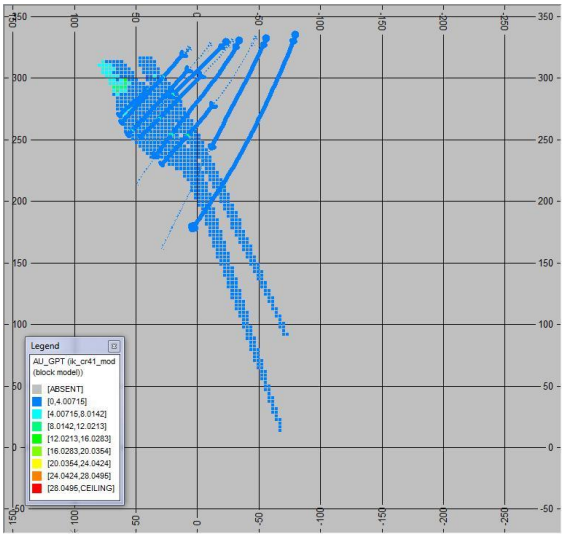
350W



400W



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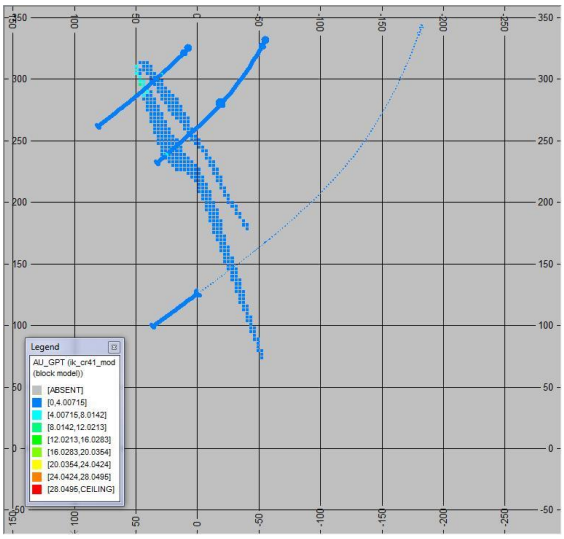
450W



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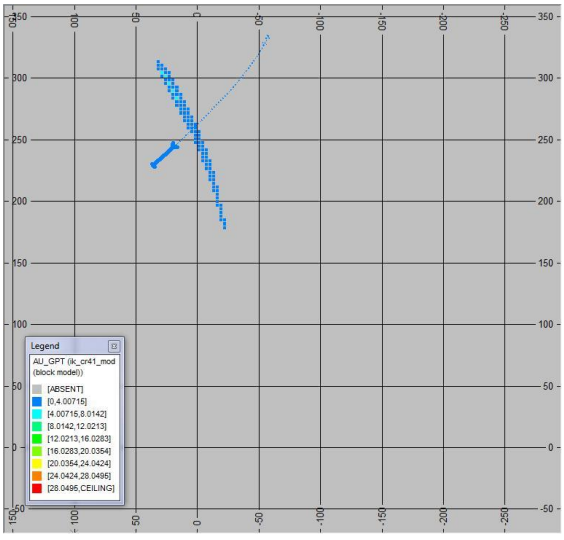

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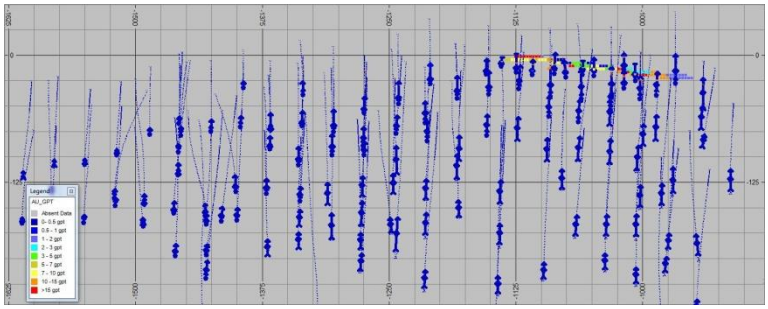
51 Deposit NN Models Plan views

325 Elev.



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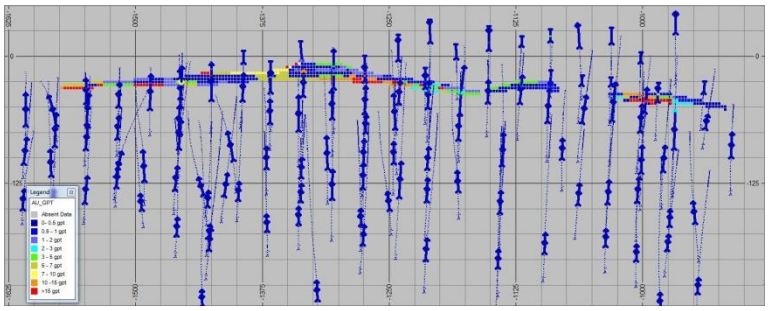
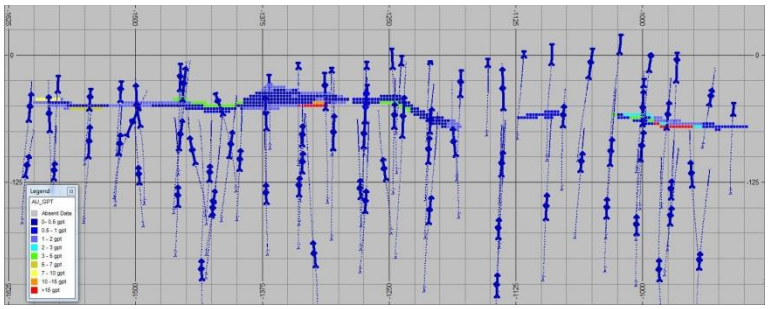


225 Elev.



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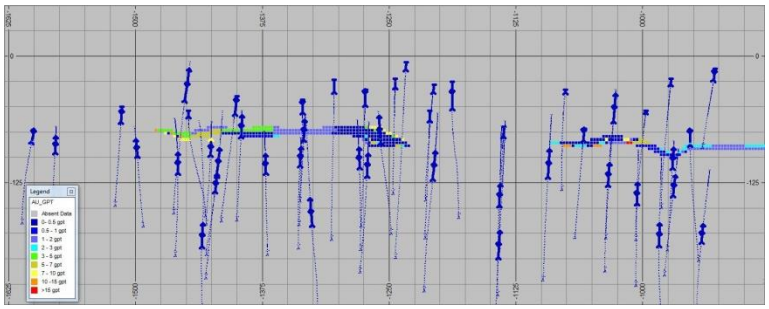
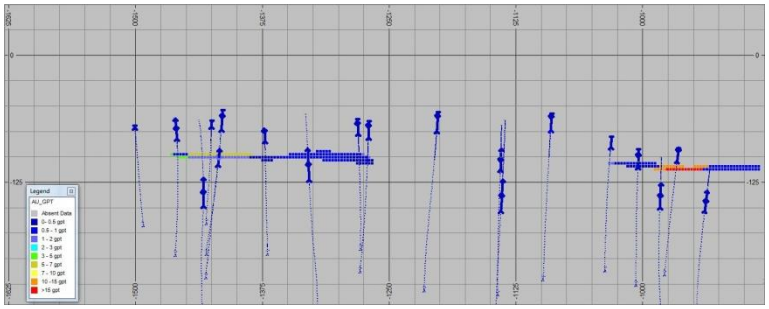
175 Elev.

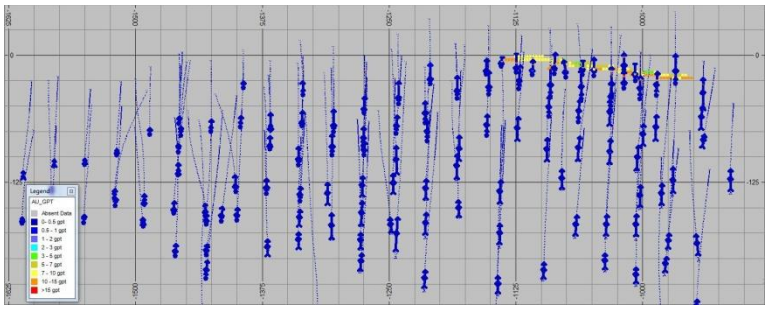


125 Elev.



51 Deposit IK Models: Plan views

325 Elev.



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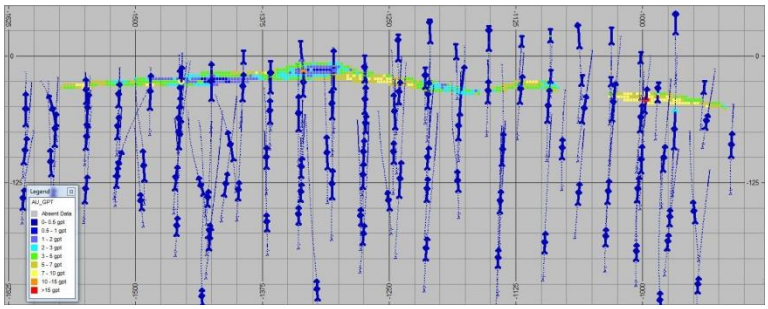
275 Elev.



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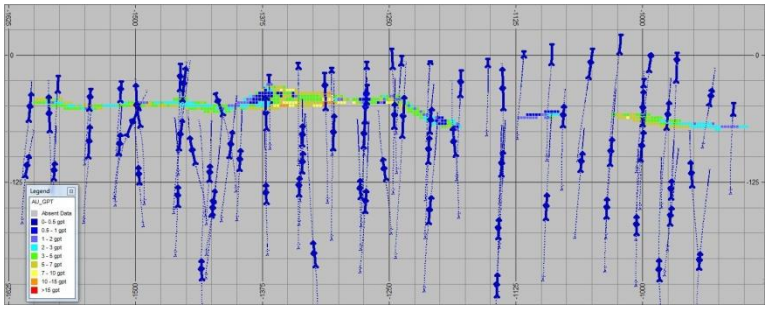

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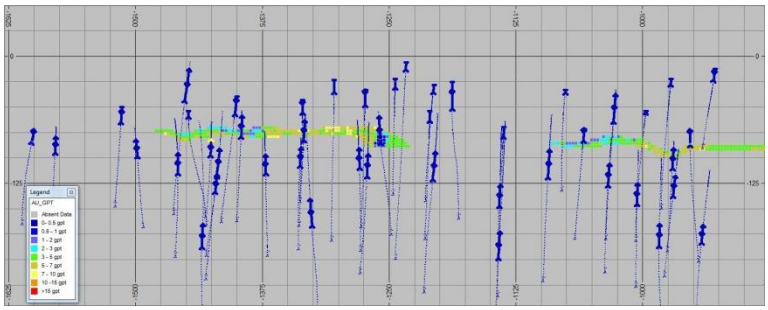
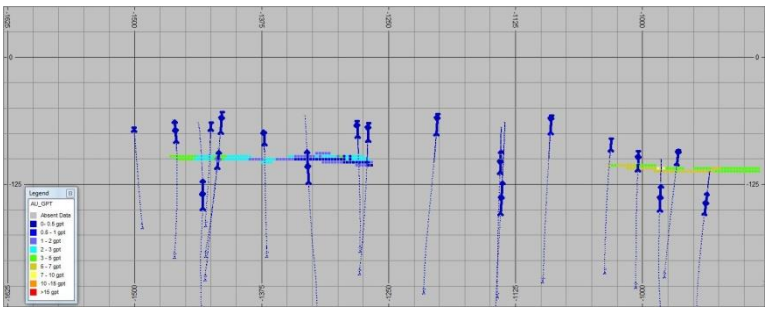


125 Elev.



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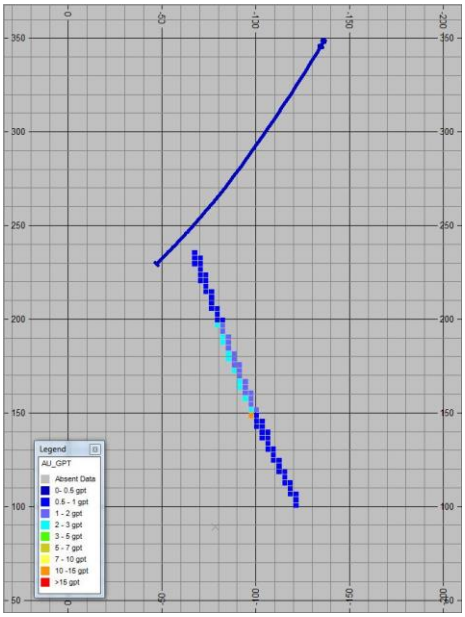
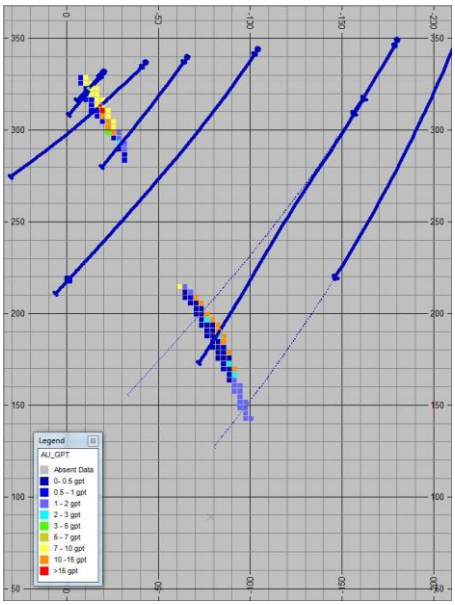
51 Deposit NN Model – N-S Sample Sections

900 W



1050 W



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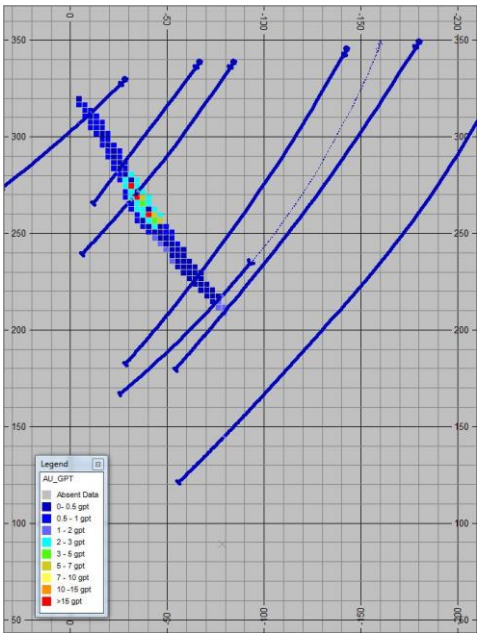
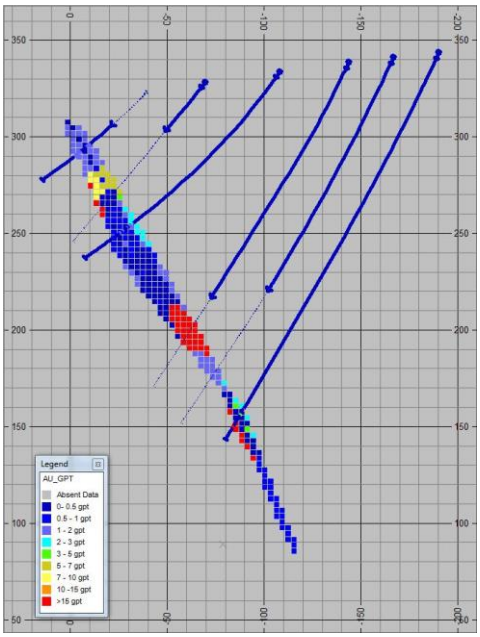
1200 W



1350 W



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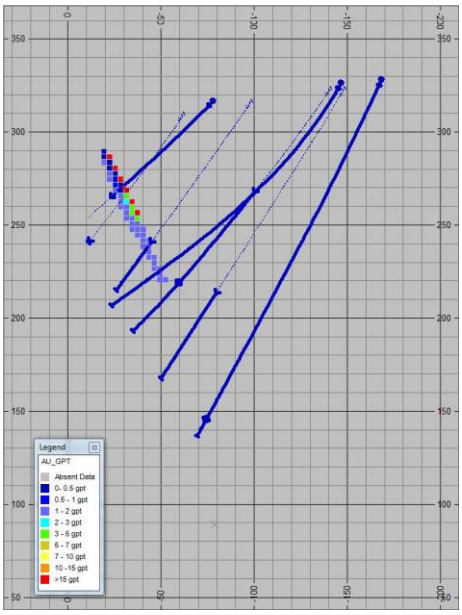
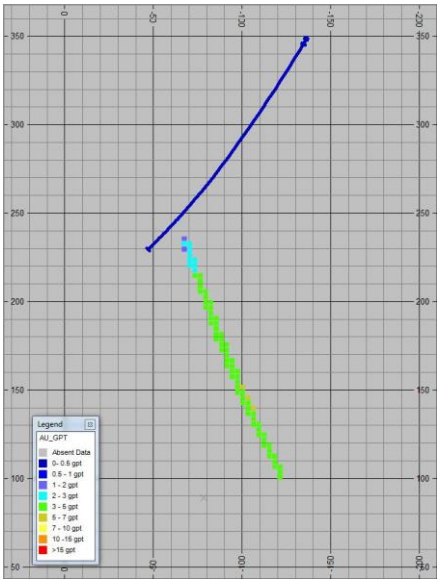
1500 W



51 Deposit IK Models – N-S Sample Sections

900 W



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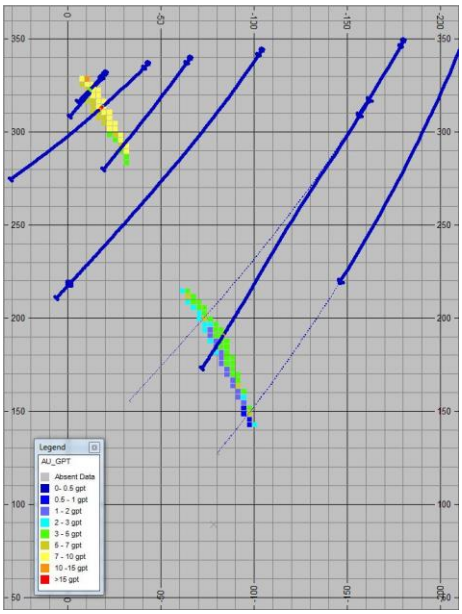
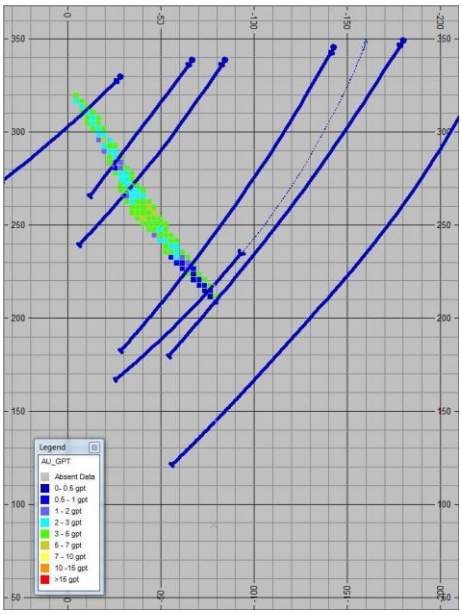
1050 W



1200 W



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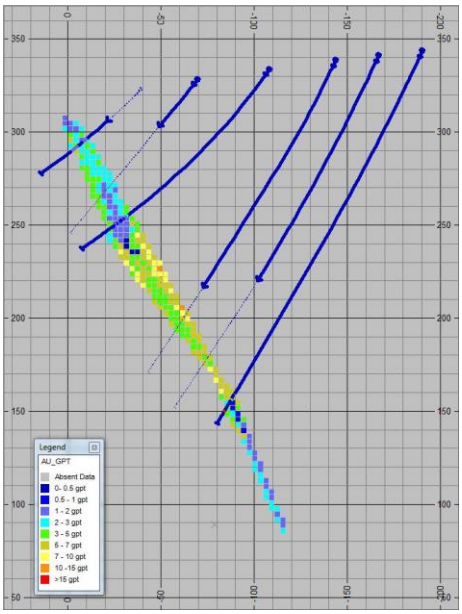
1350 W



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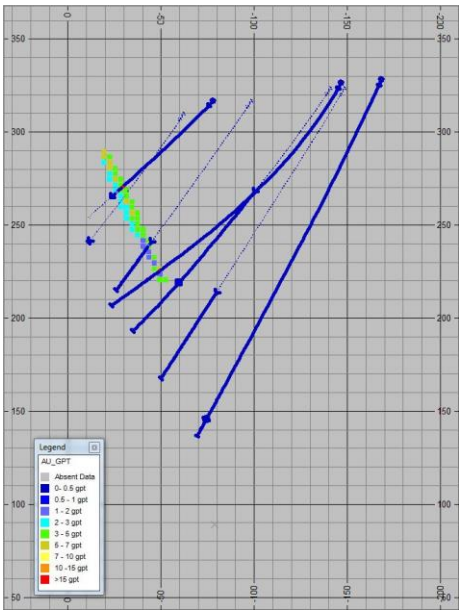

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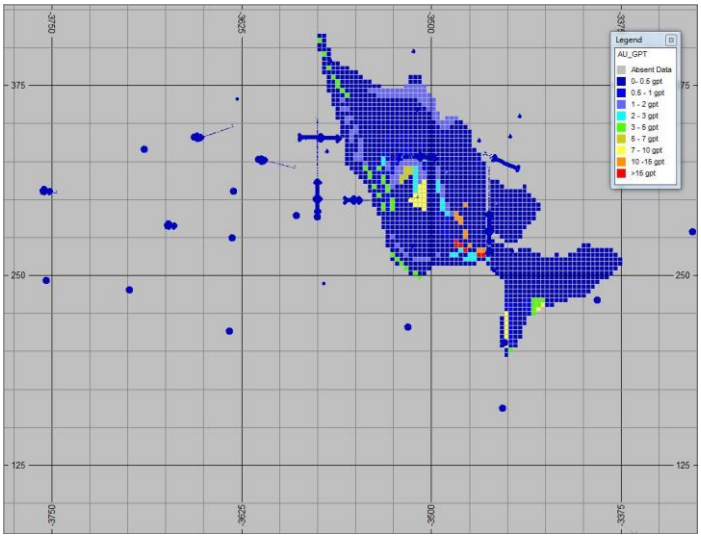
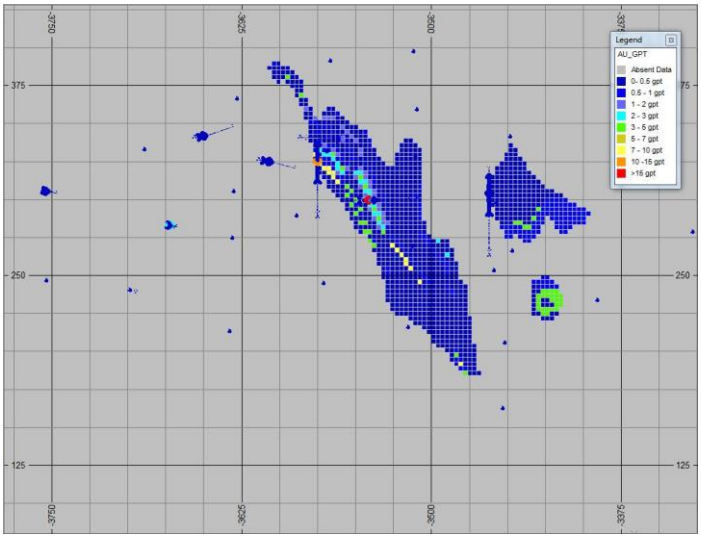
Windowglass Hill Deposit NN Models Plan views

325 Elev.



300 Elev.



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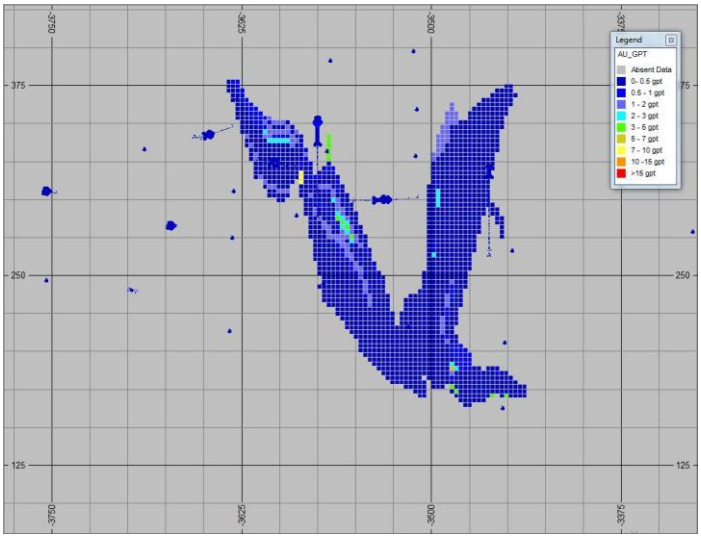
275 Elev.



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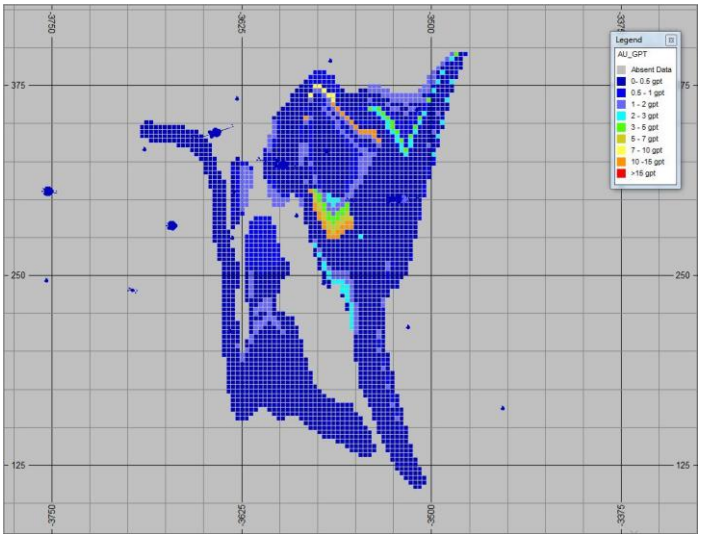

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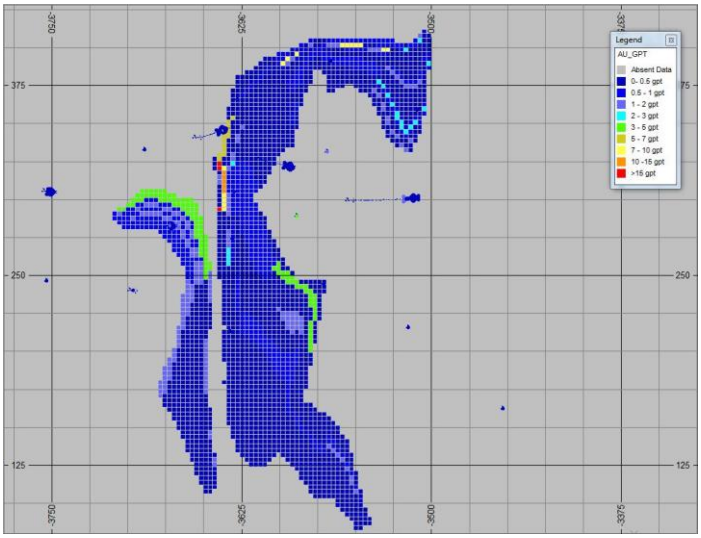
225 Elev.



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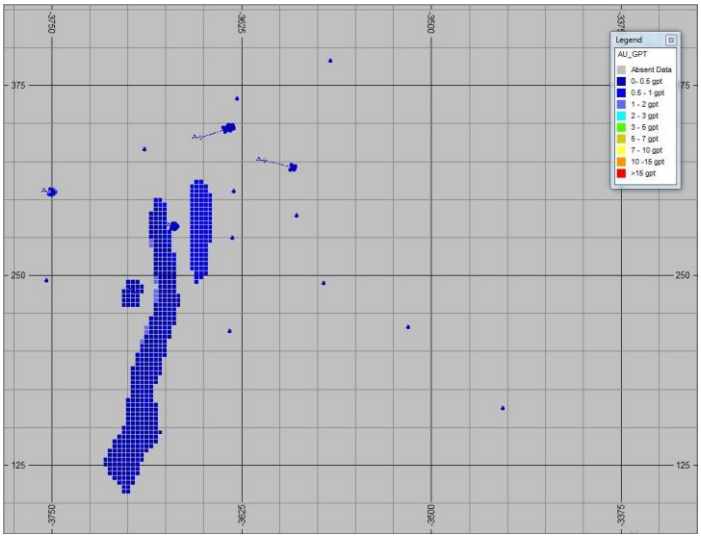

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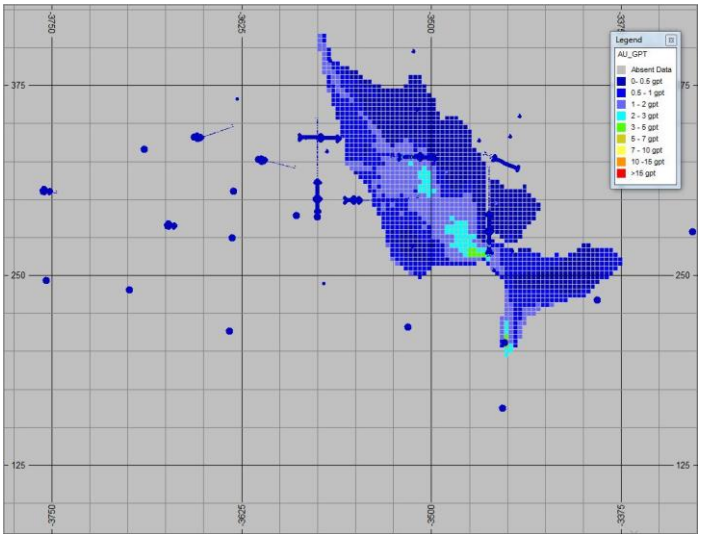
Windowglass Hill Deposit ID Model: Plan views

325 Elev.



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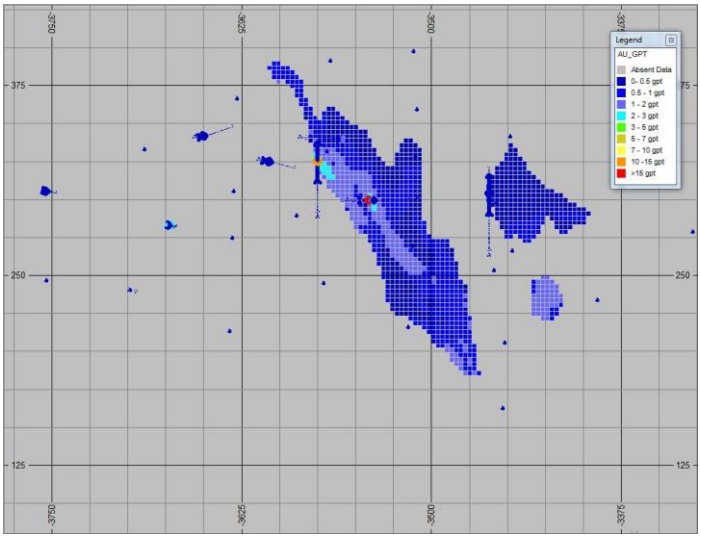

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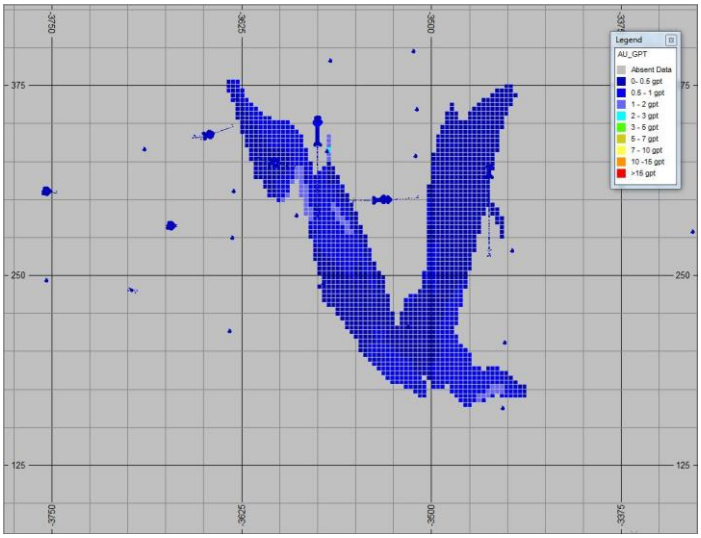
275 Elev.



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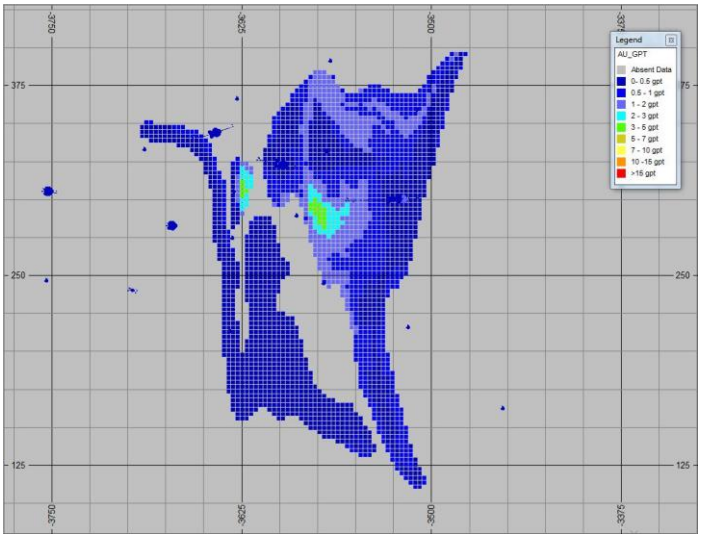

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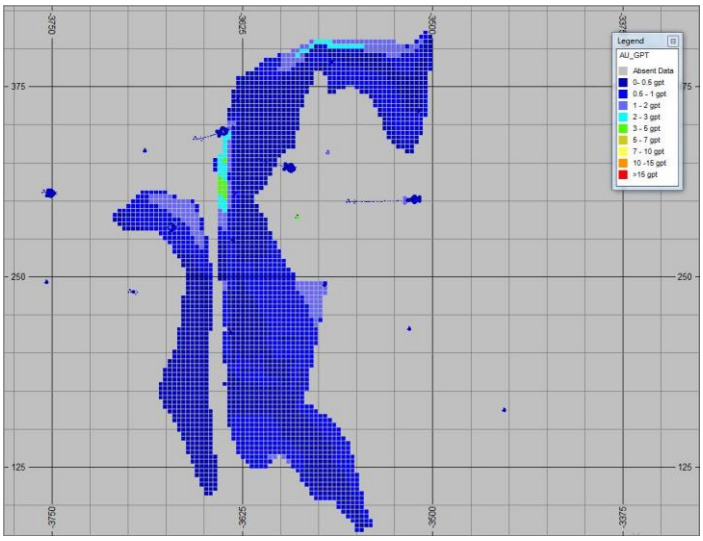
225 Elev.



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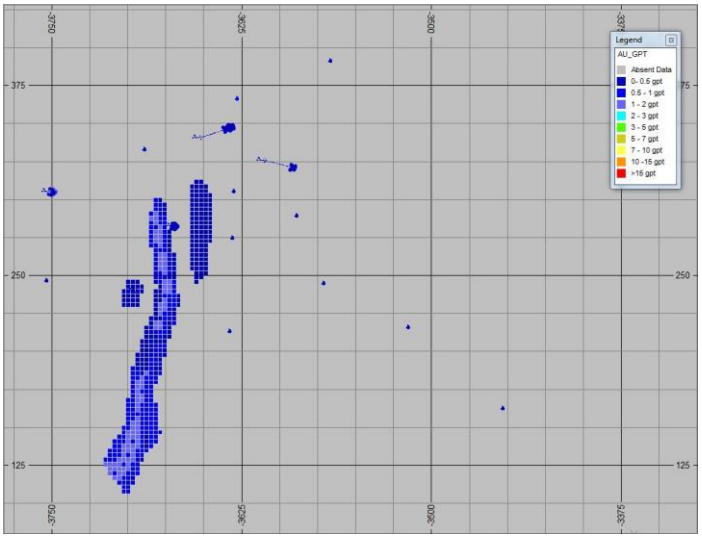
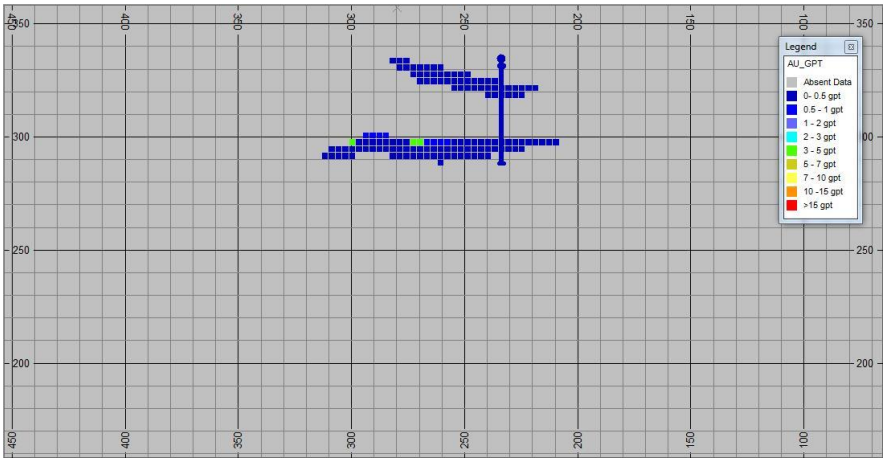
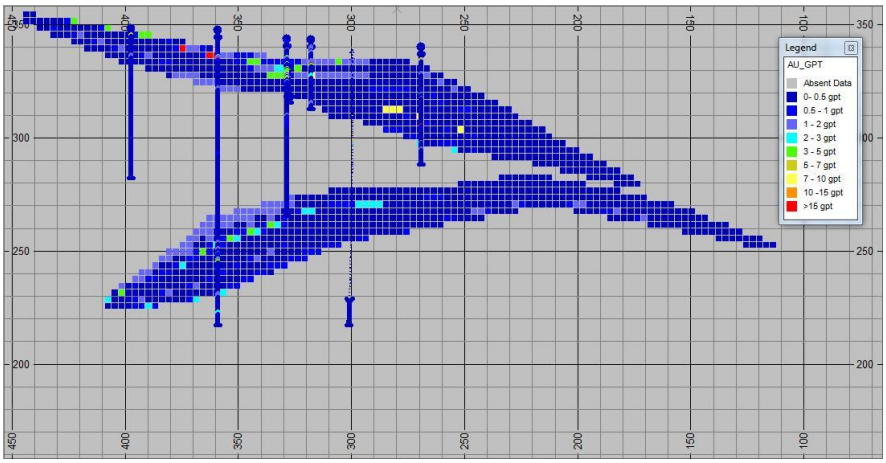


Windowglass Hill NN Model – N-S Sample Sections

3400W



3500W



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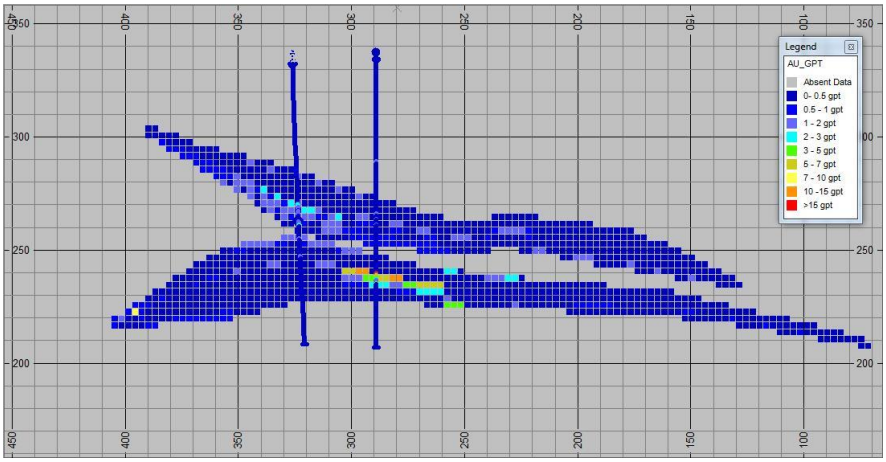
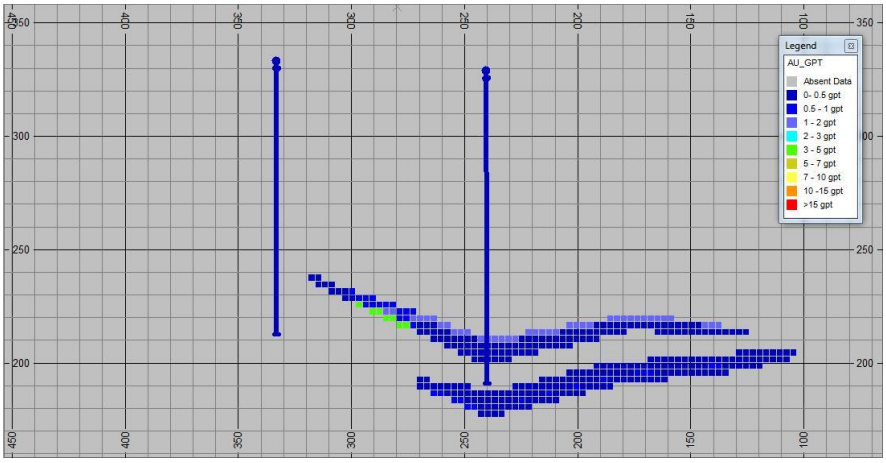
3600W

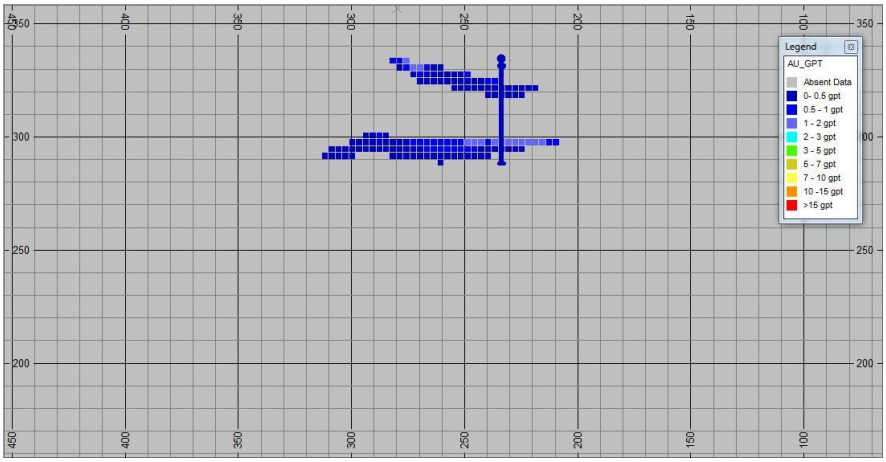


3700W

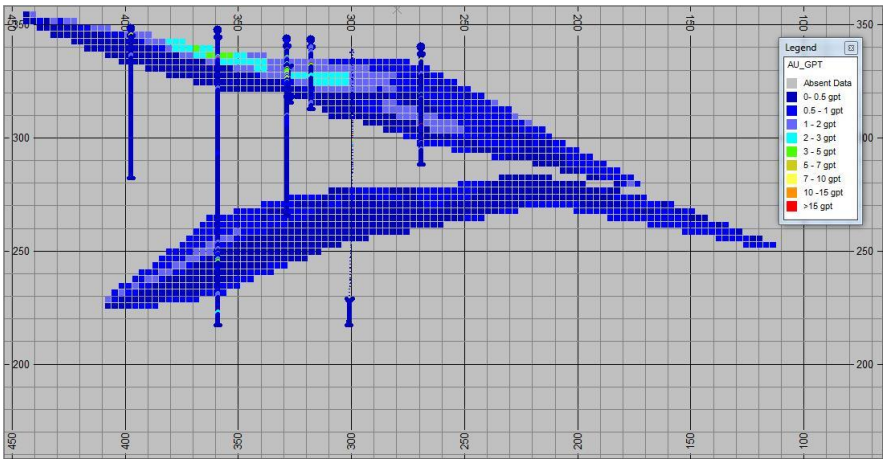


Windowglass Hill Deposit ID Model – N-S Sample Sections

3400W



3500W



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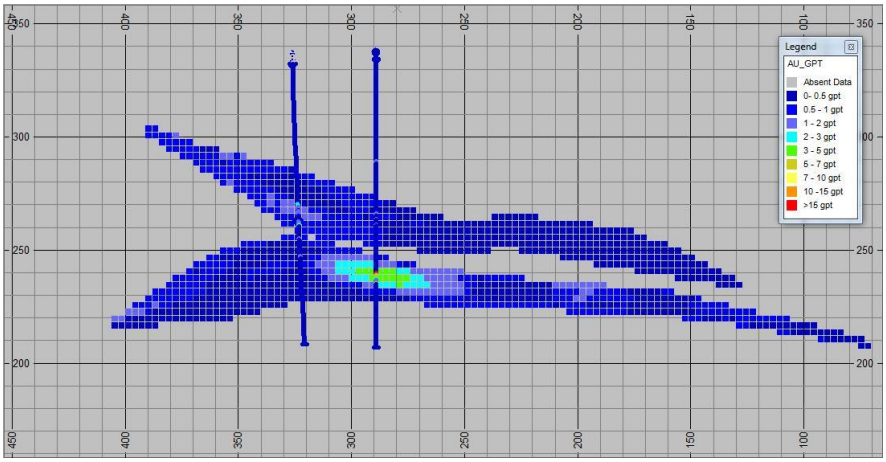
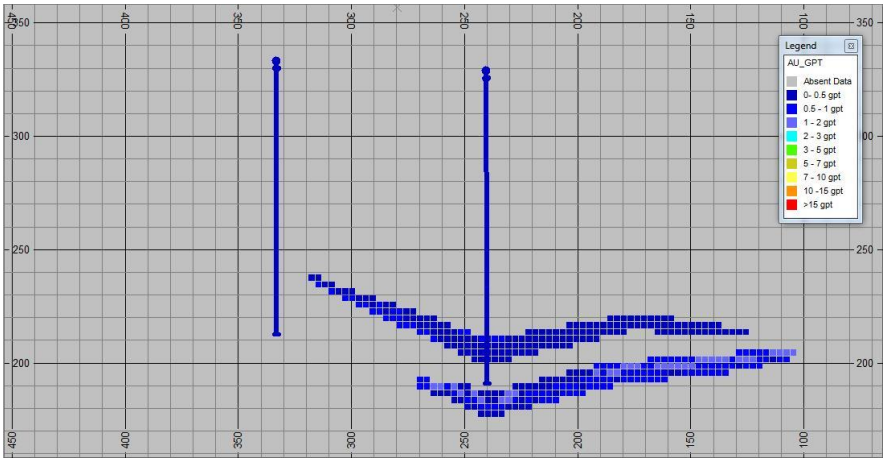
3600W



3700W



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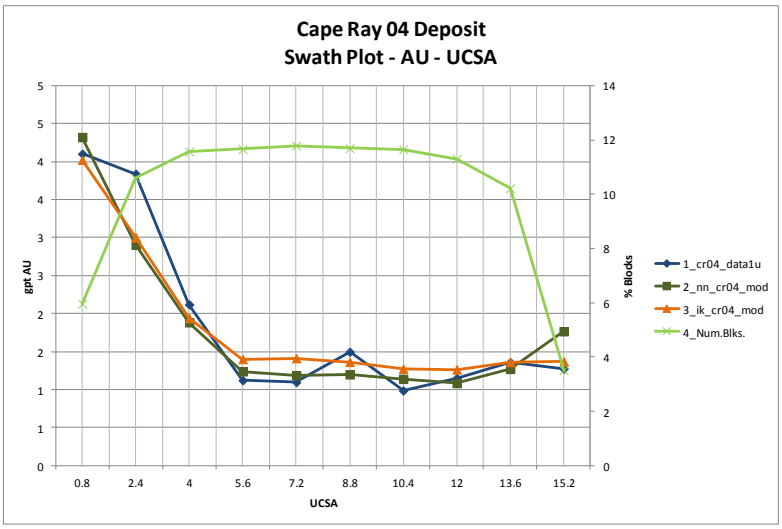
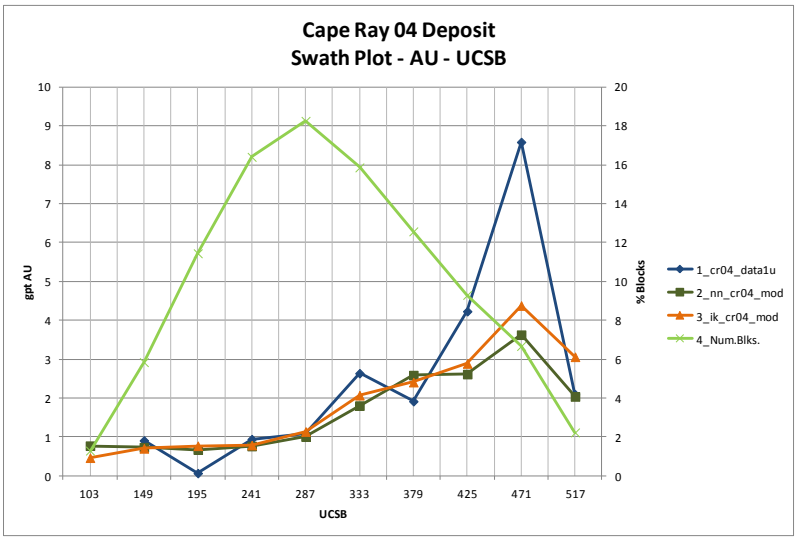
Appendix C Model Validation

04 Deposit Swath Plots – Au





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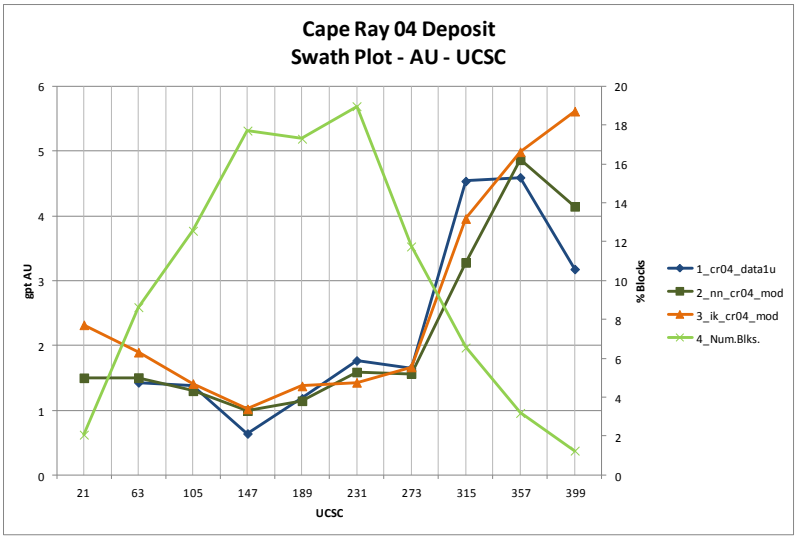
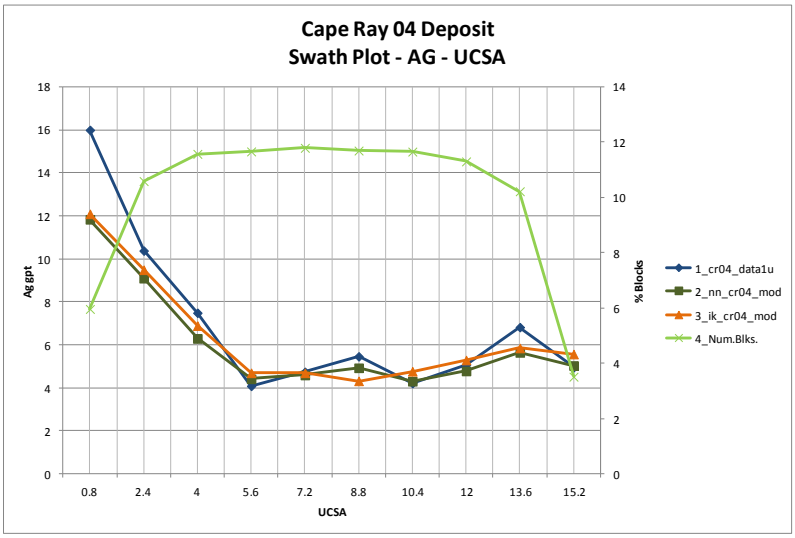


04 Deposit Swath Plots – Ag



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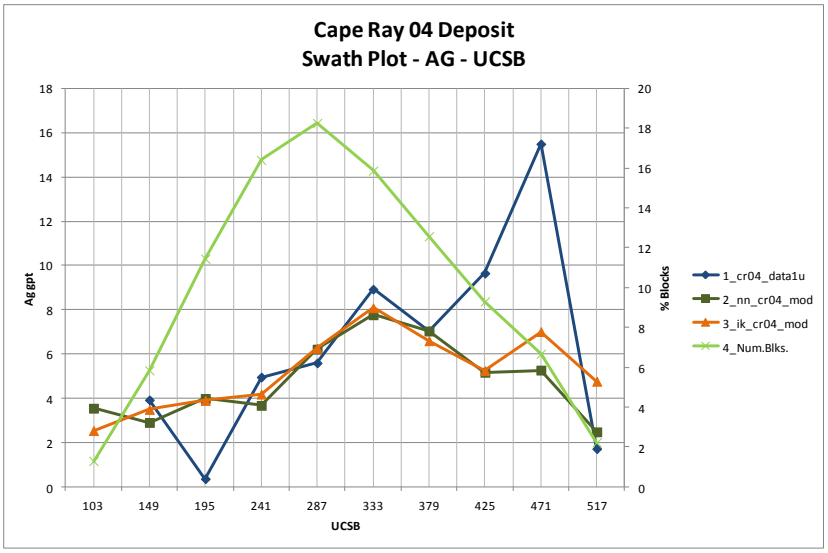

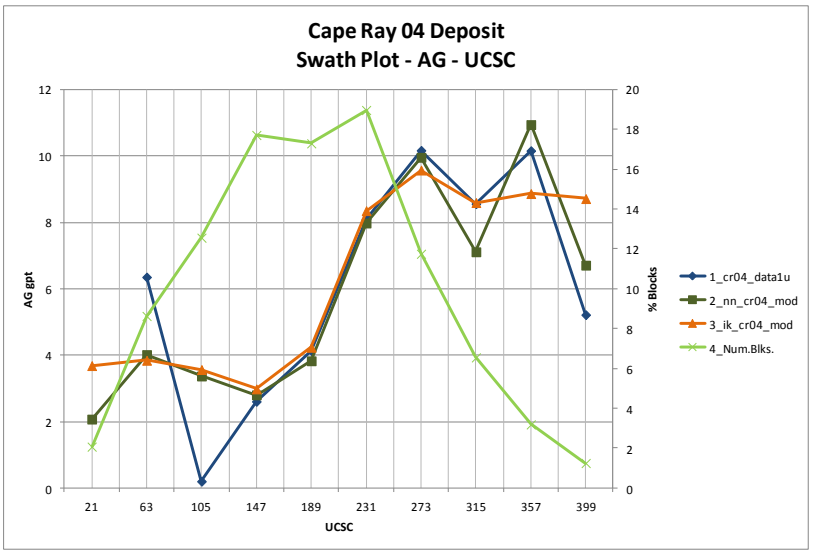

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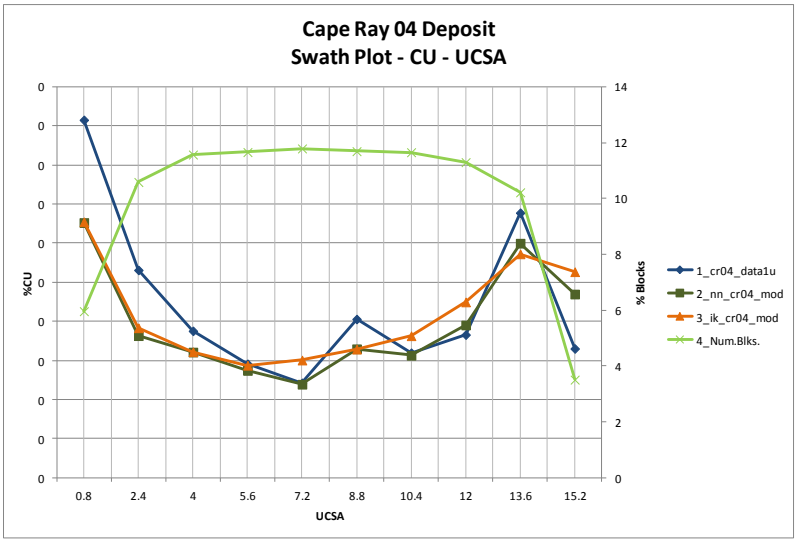
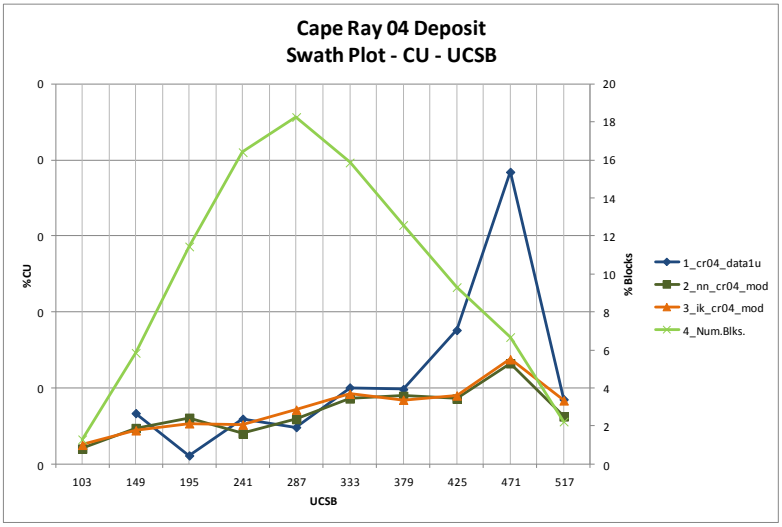
04 Deposit Swath Plots – Cu





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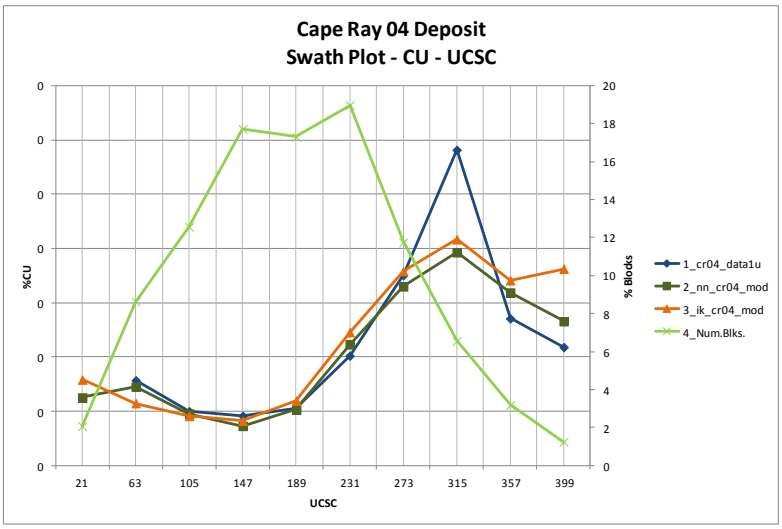
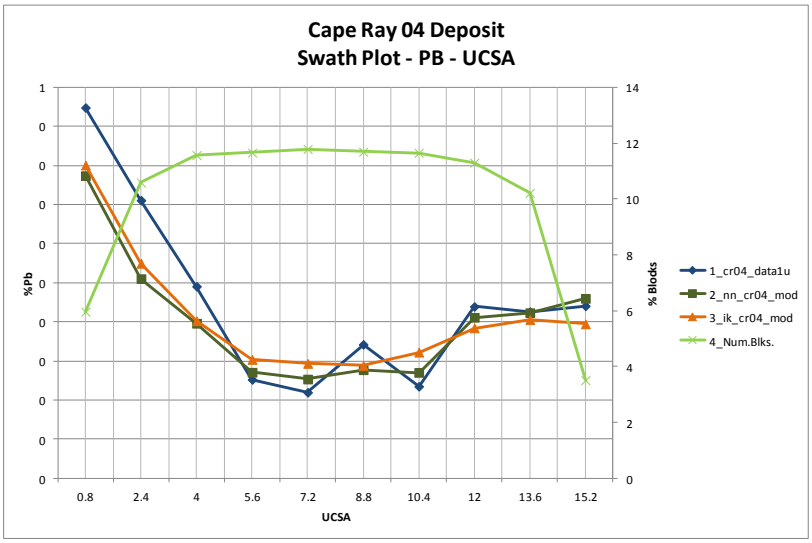


04 Deposit Swath Plots – Pb



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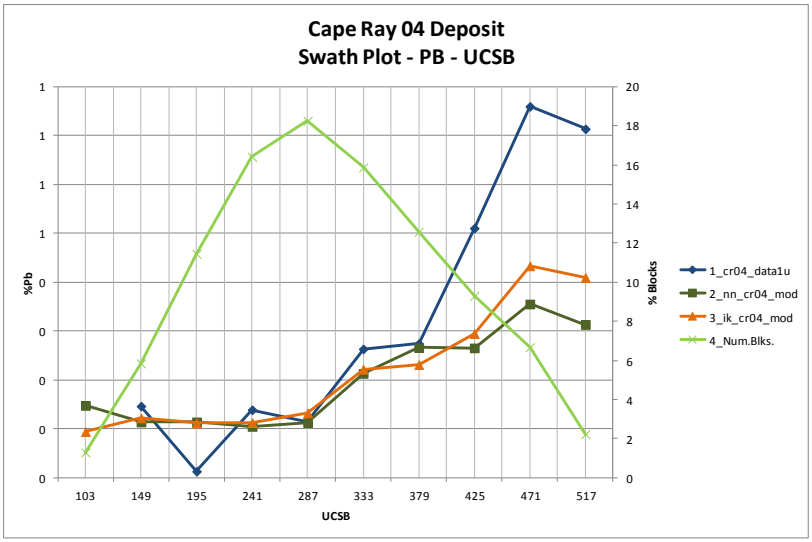

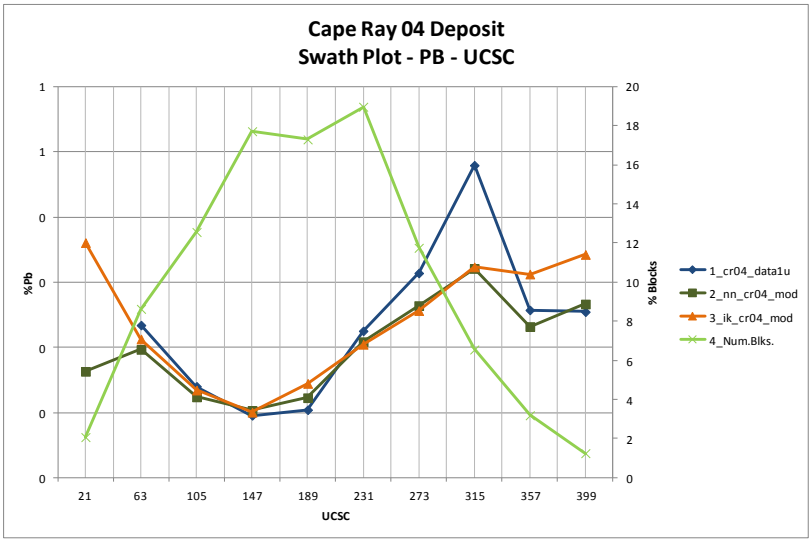

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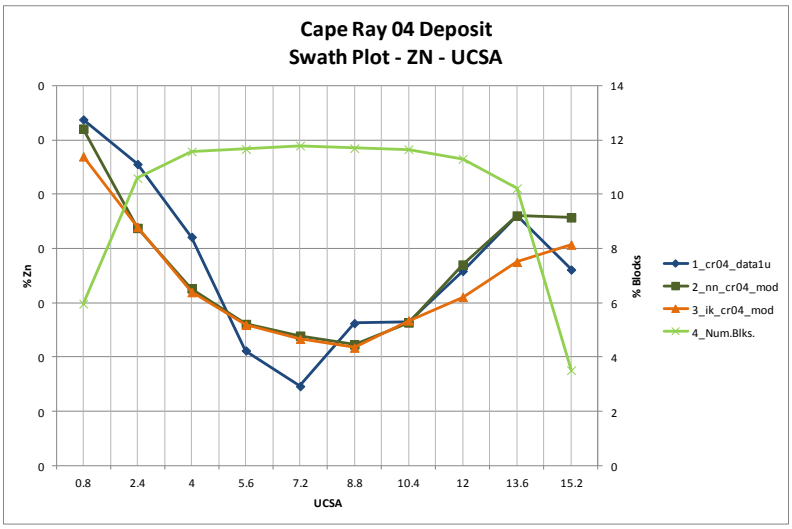
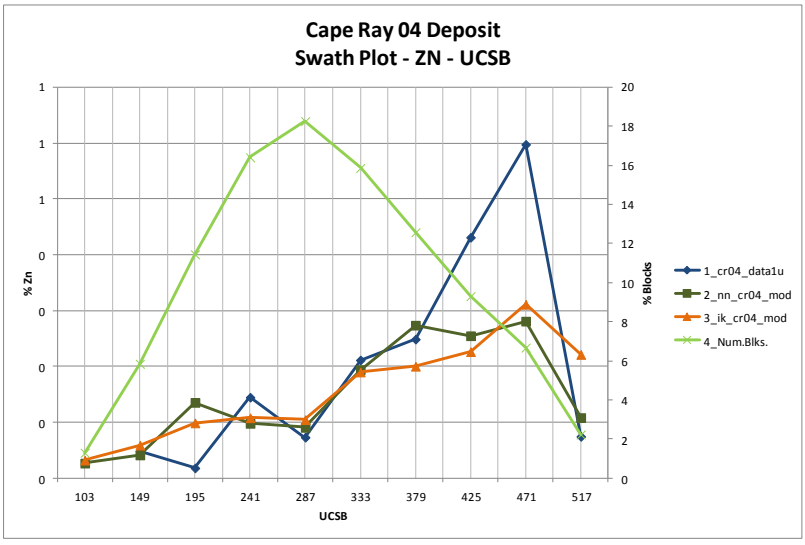
04 Deposit Swath Plots – Zn





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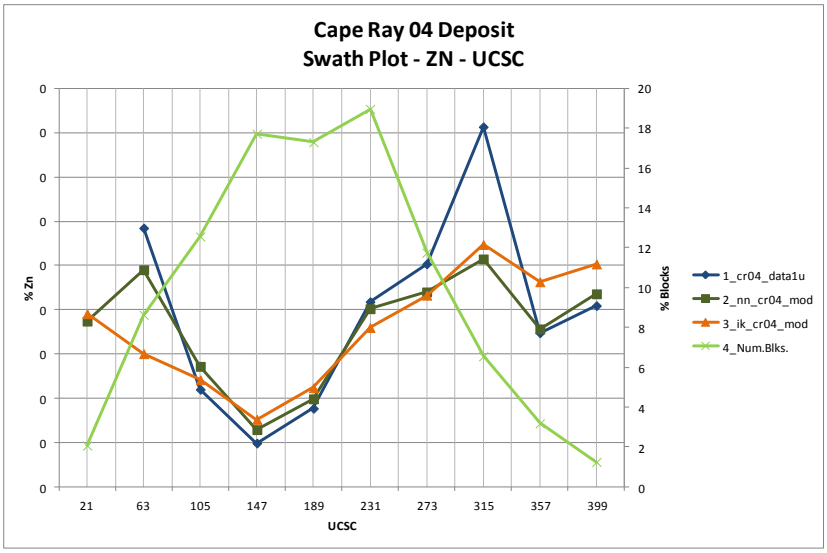
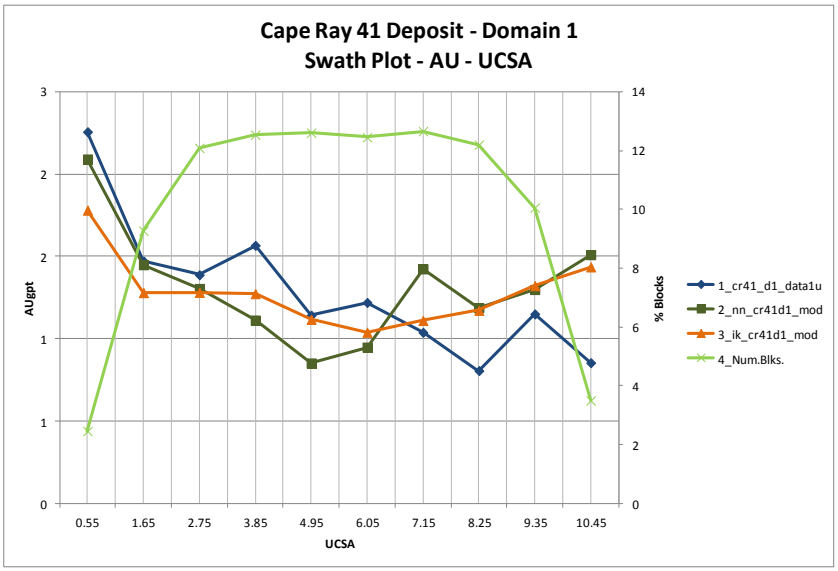


41 Deposit Swath Plots – Au – Domain 1



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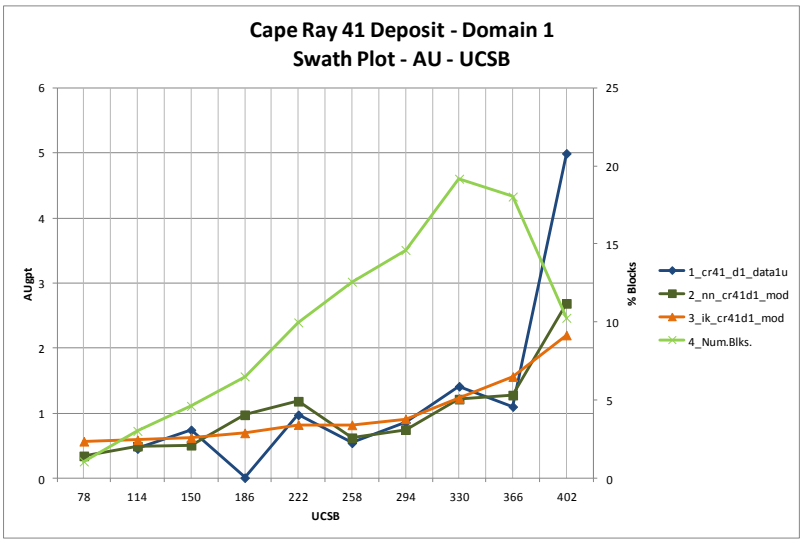
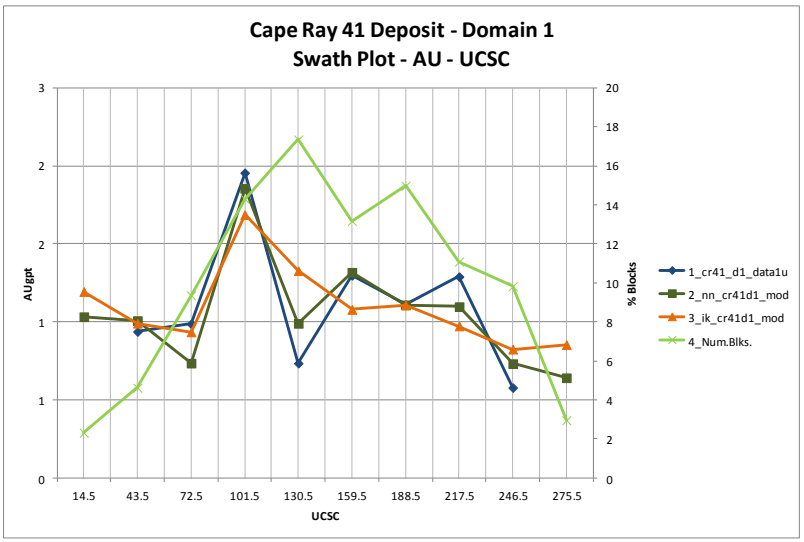
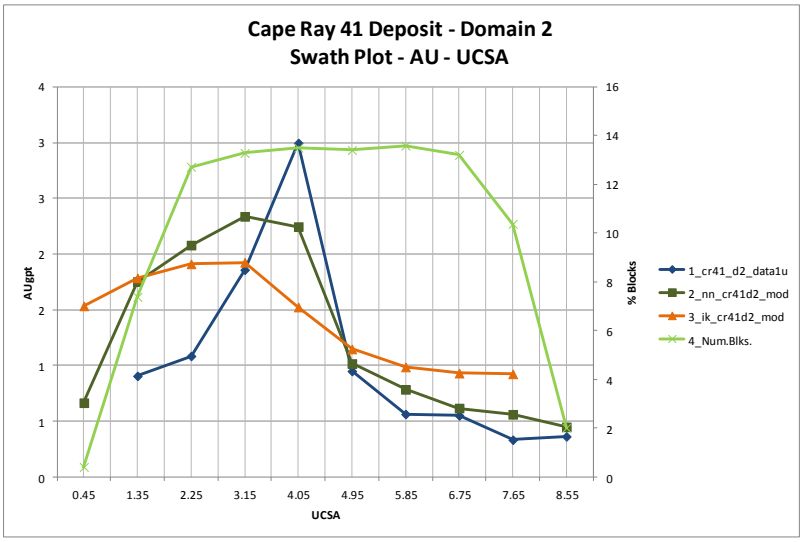
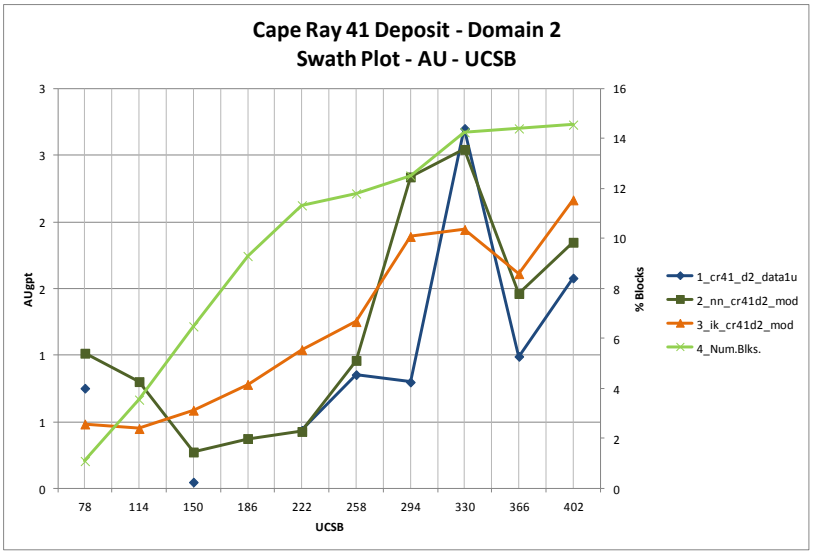




41 Deposit Swath Plots – Au – Domain 2





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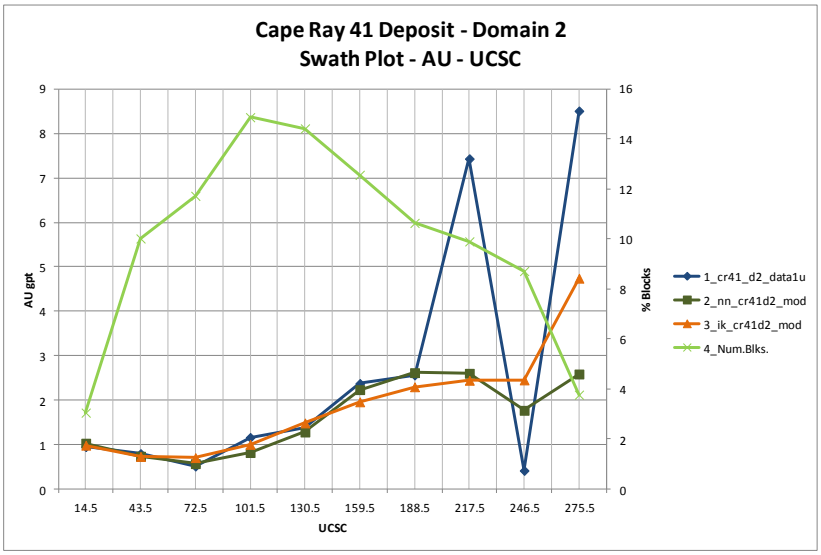
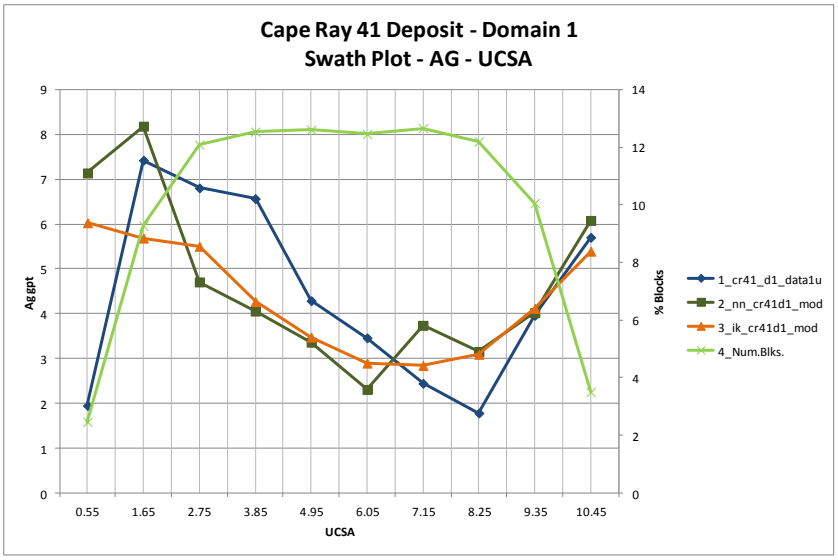


41 Deposit Swath Plots – Ag – Domain 1



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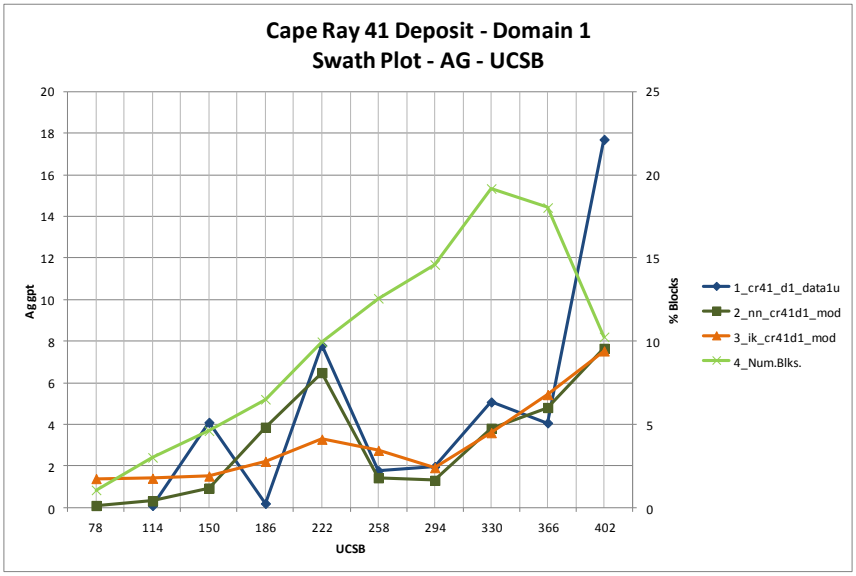

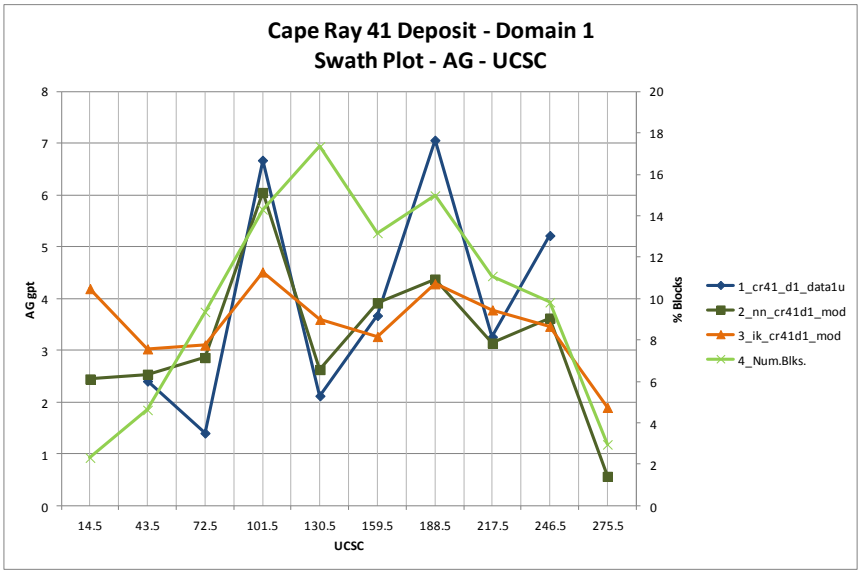

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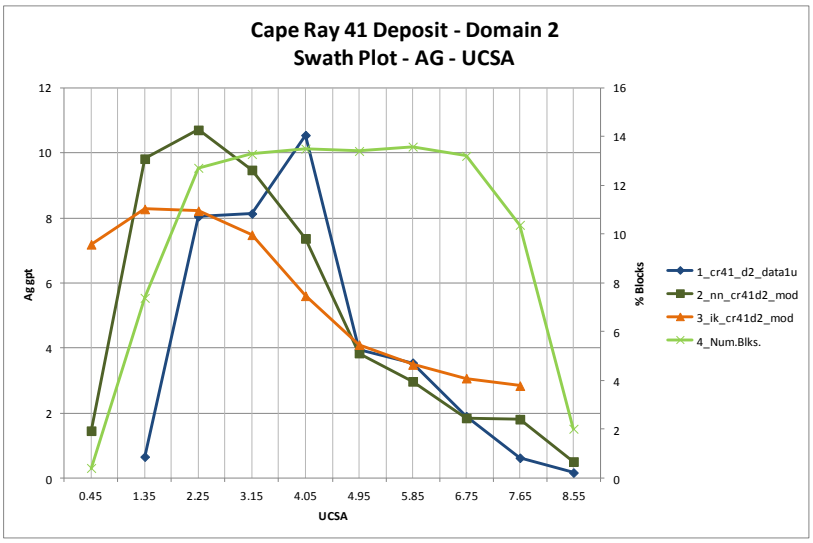
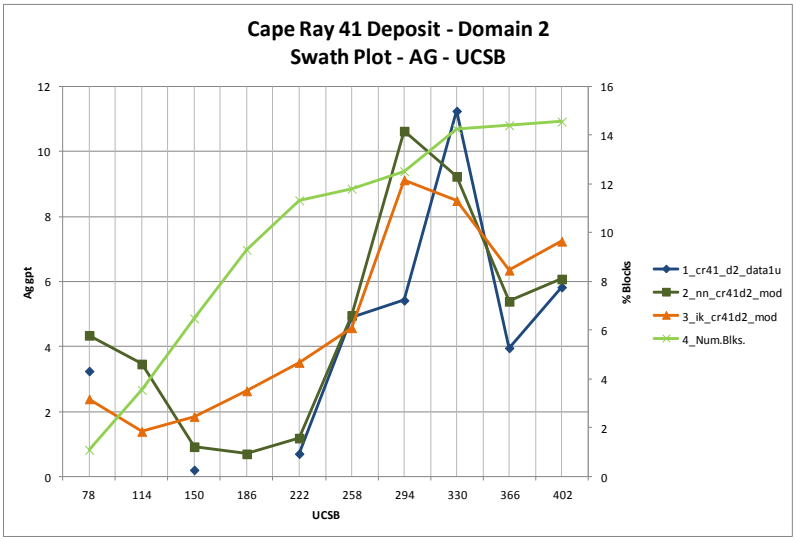
41 Deposit Swath Plots – Ag – Domain 2





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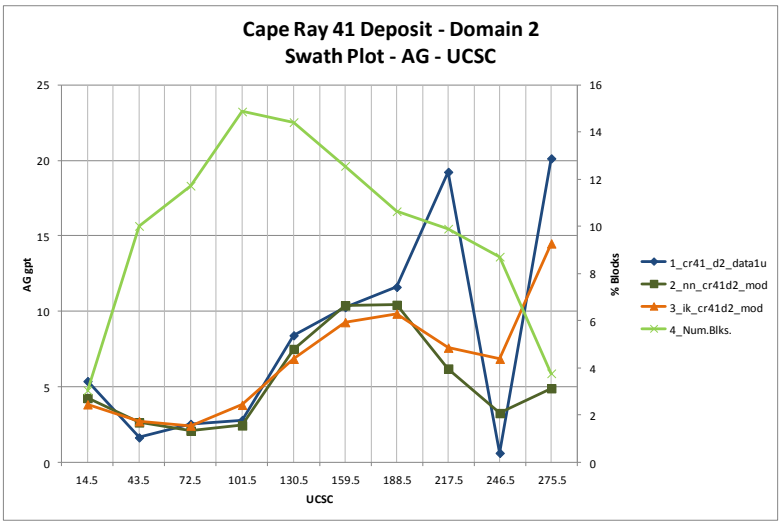
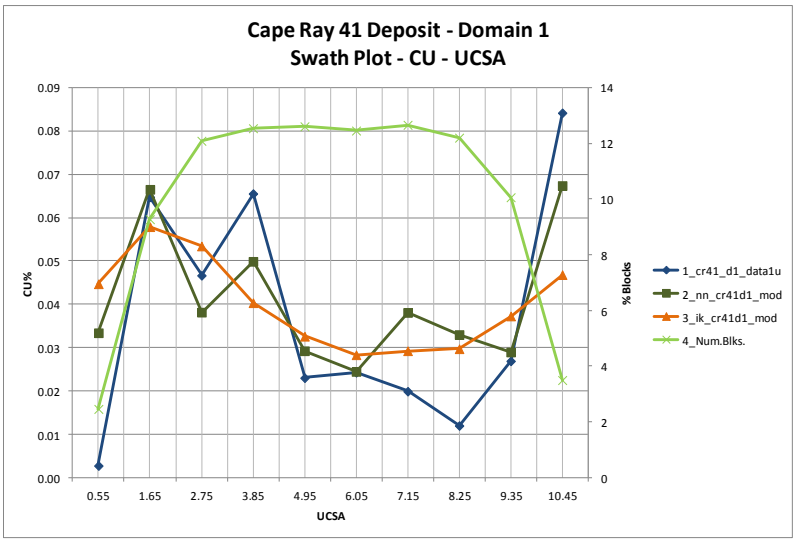


41 Deposit Swath Plots – Cu – Domain 1



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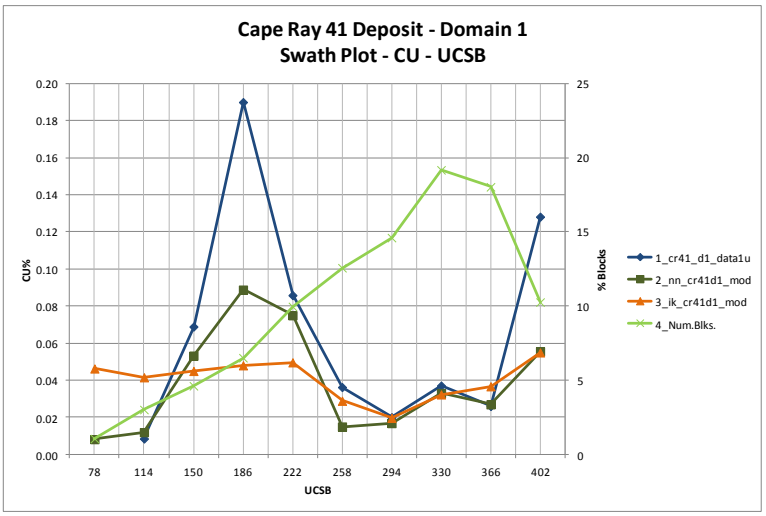
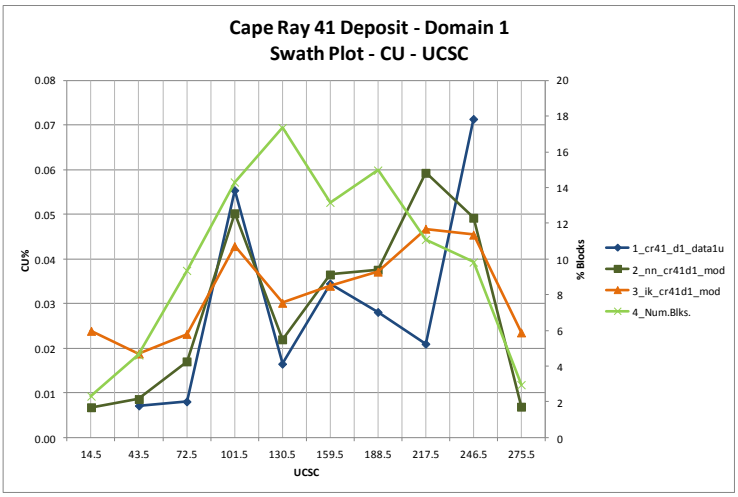
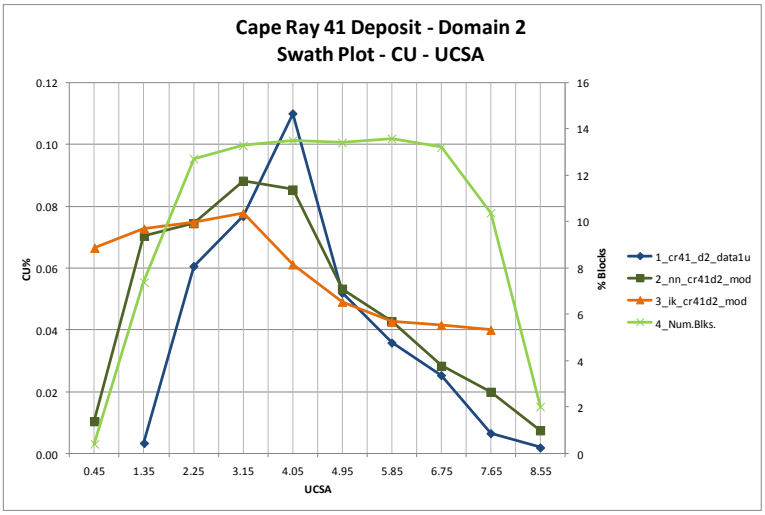
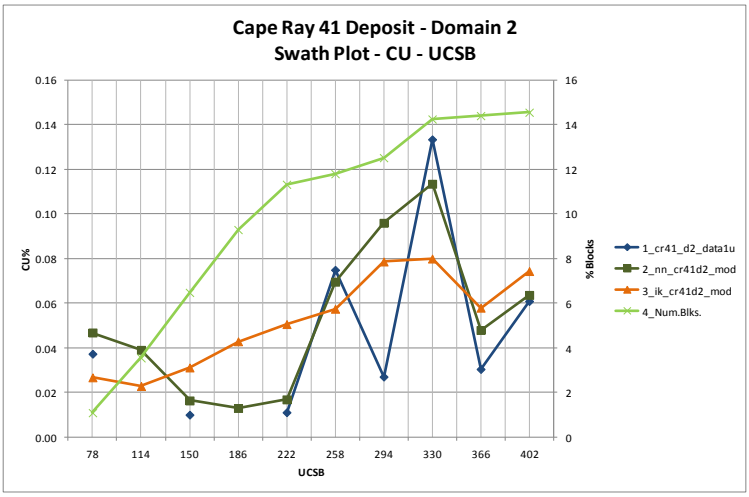
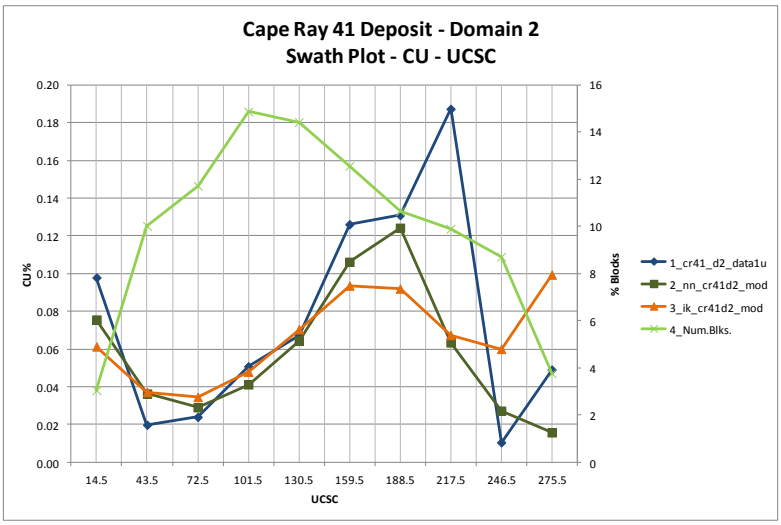




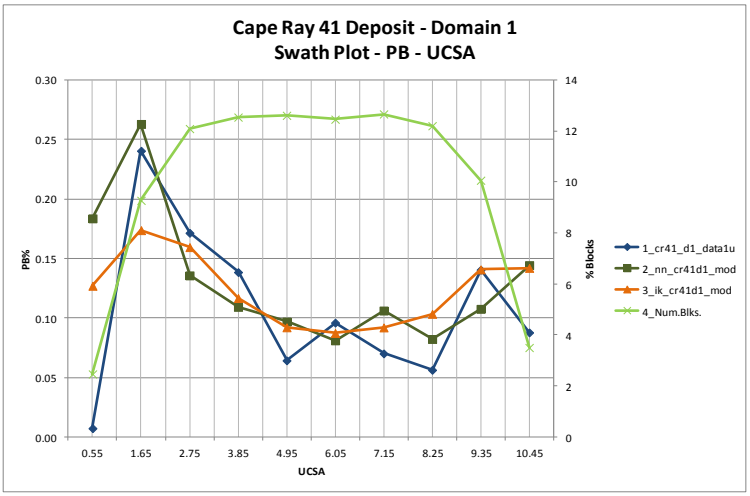
41 Deposit Swath Plots – Cu – Domain 2







41 Deposit Swath Plots – Pb – Domain 1



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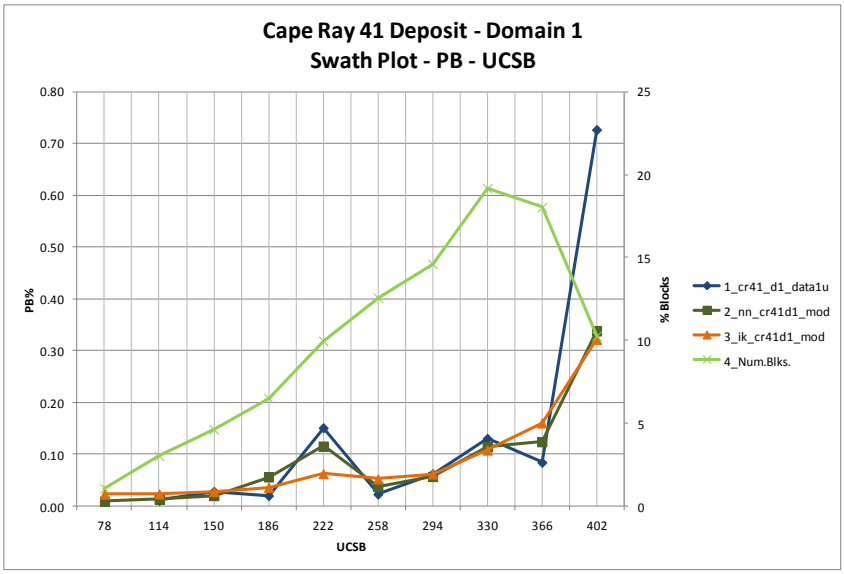

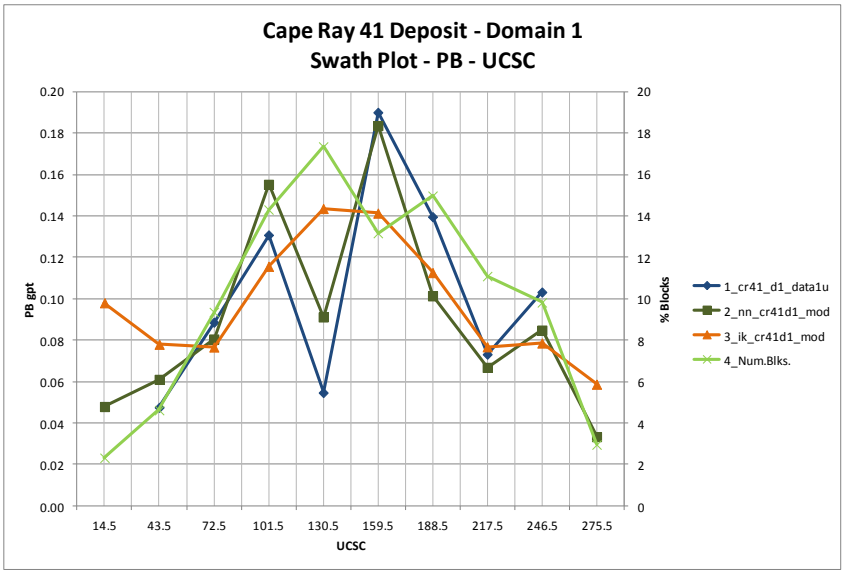

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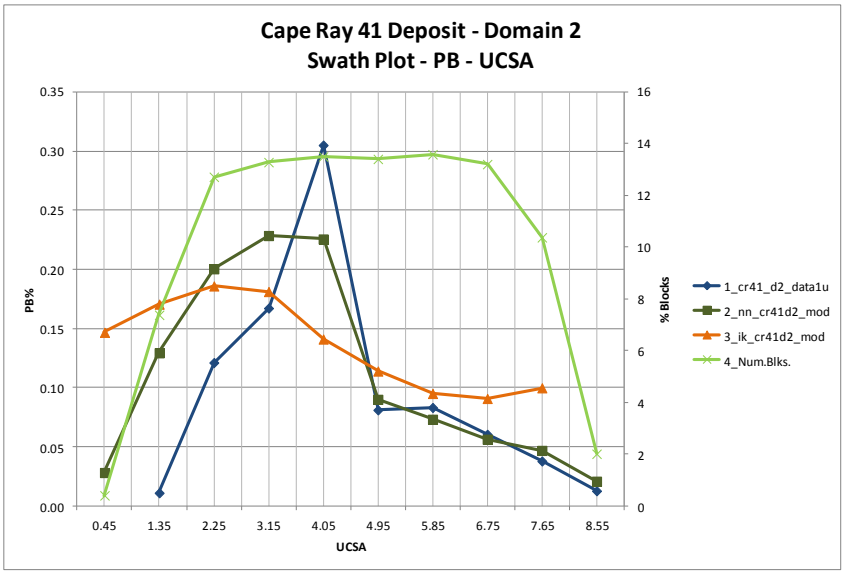
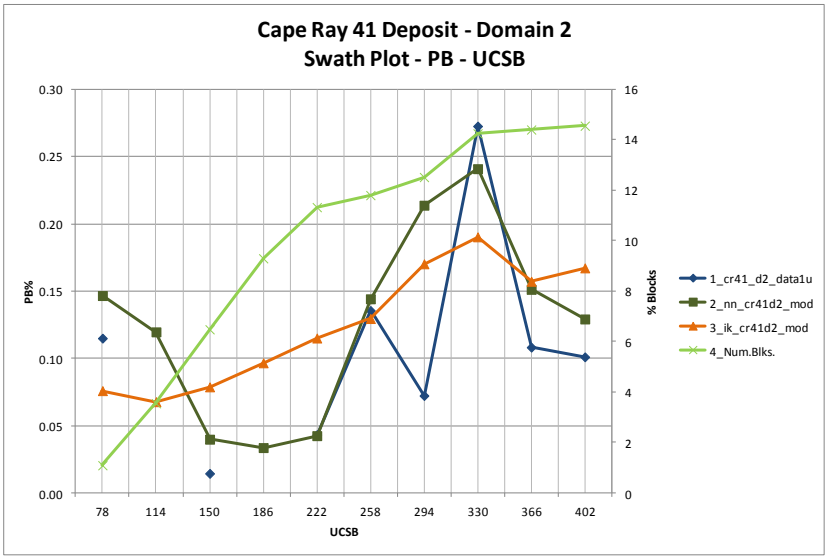
41 Deposit Swath Plots – Pb – Domain 2





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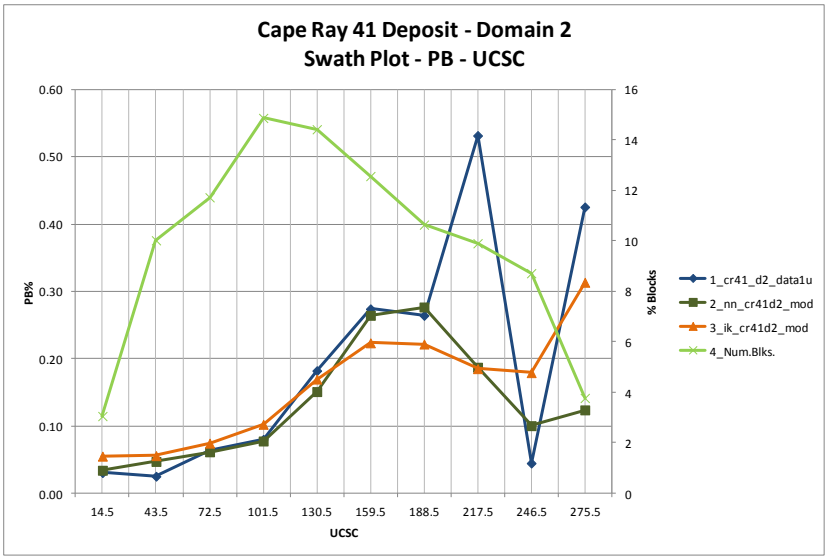
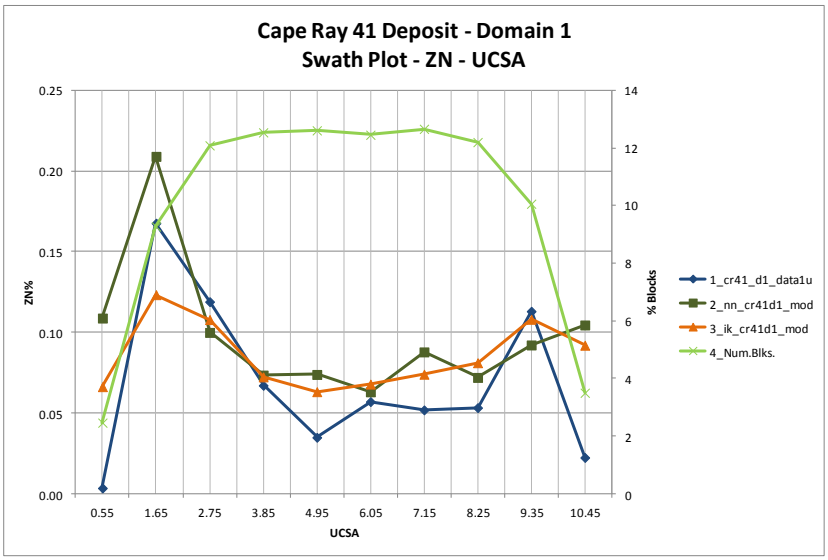


41 Deposit Swath Plots – Zn – Domain 1



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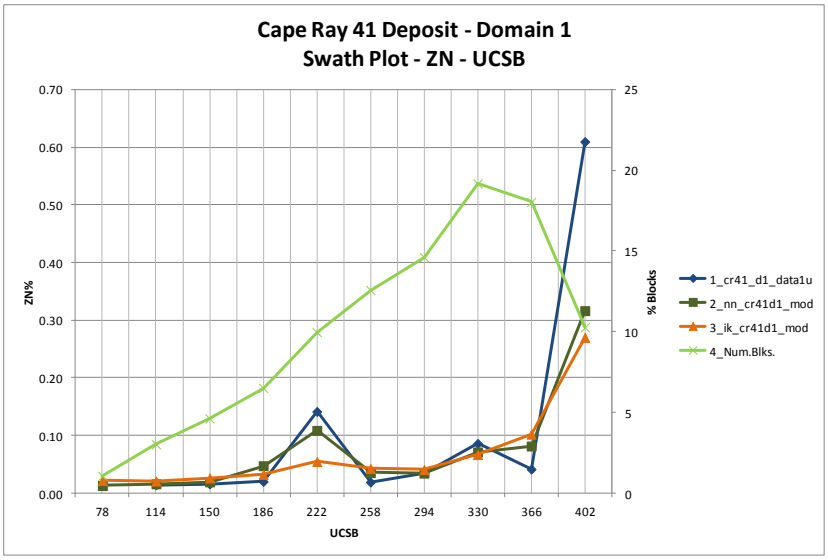

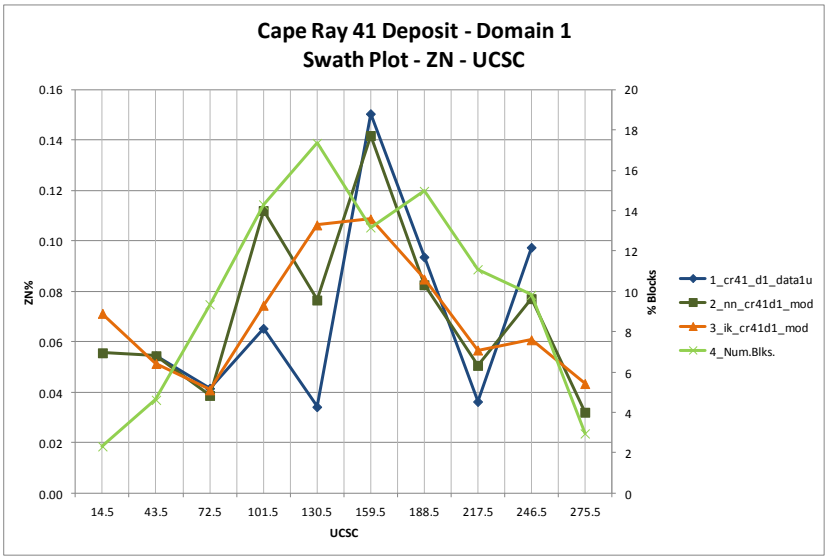

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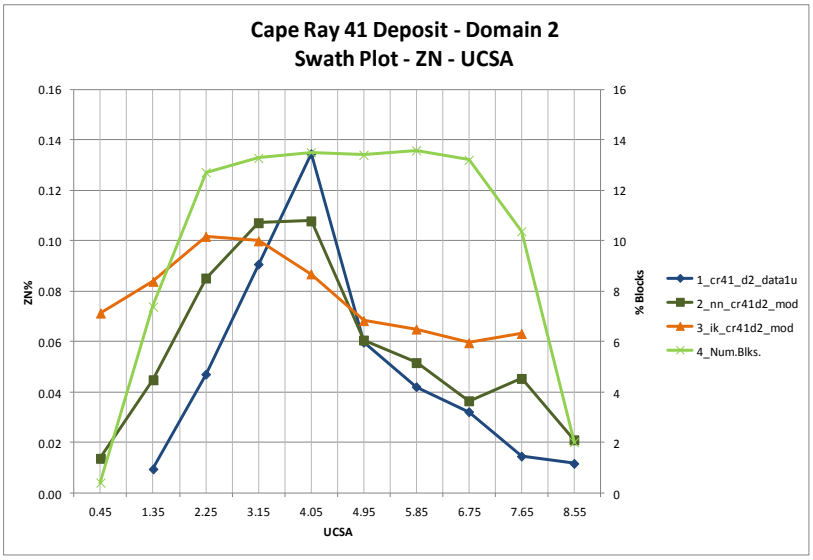
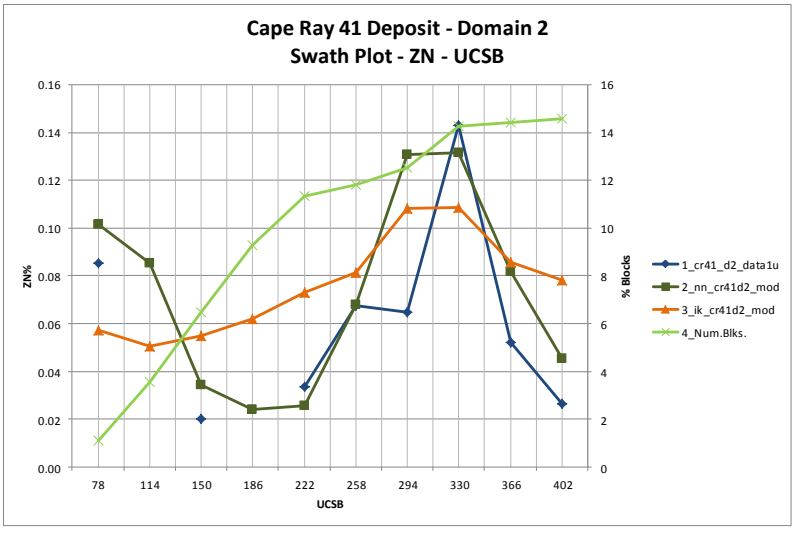
41 Deposit Swath Plots – Zn – Domain 2





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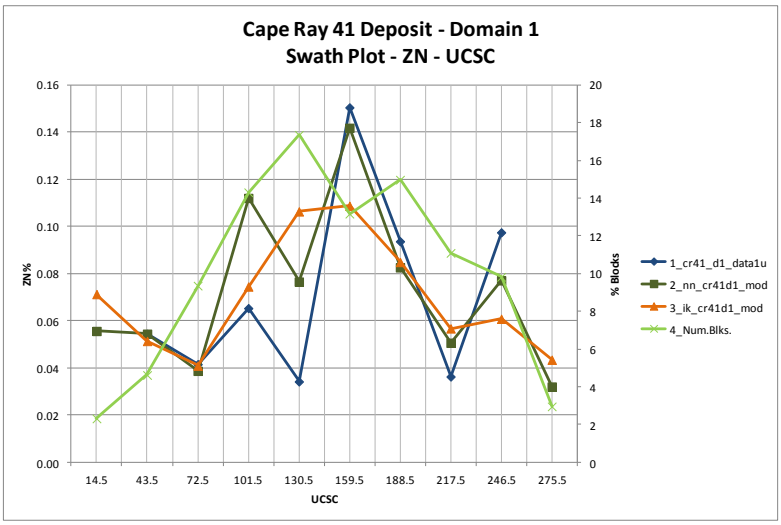
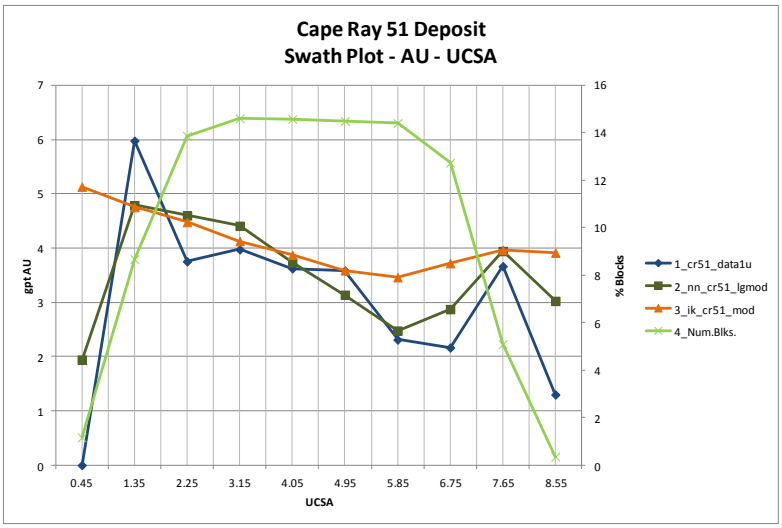


51 Deposit Swath Plots – Au



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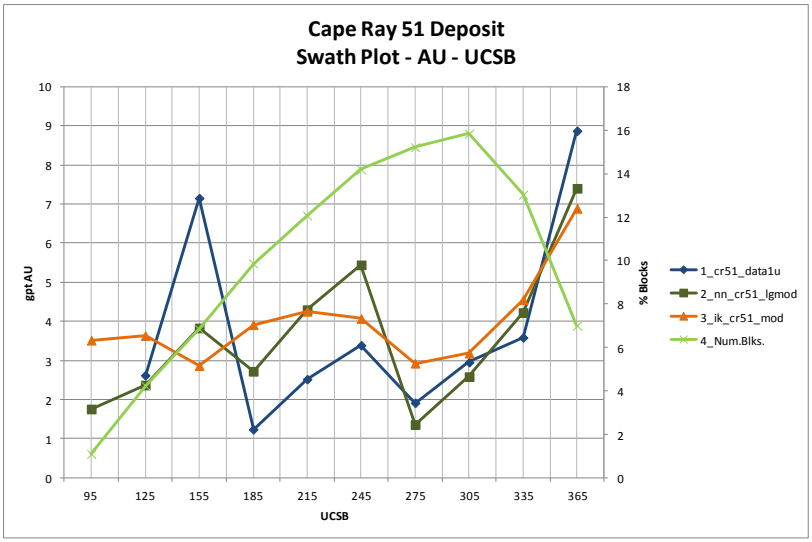
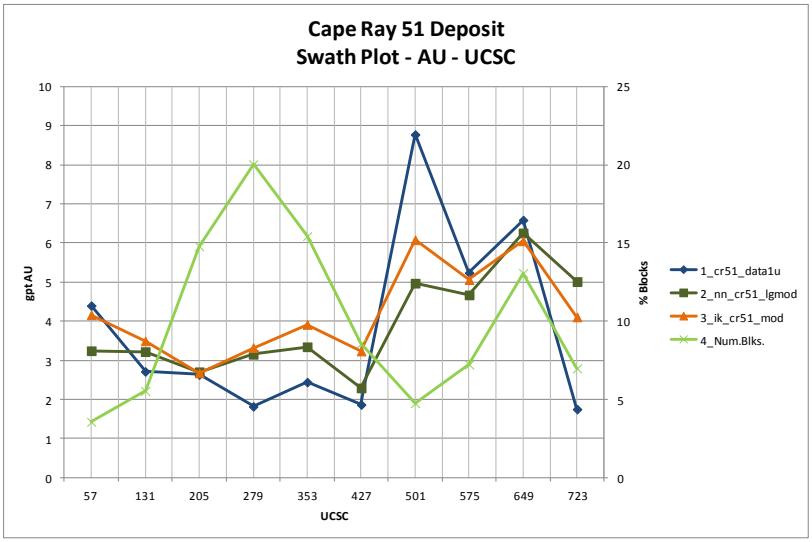
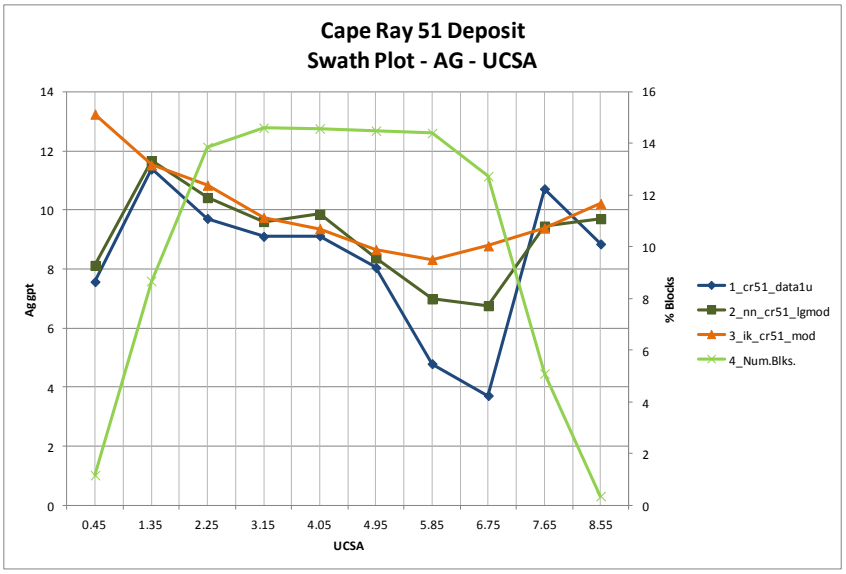
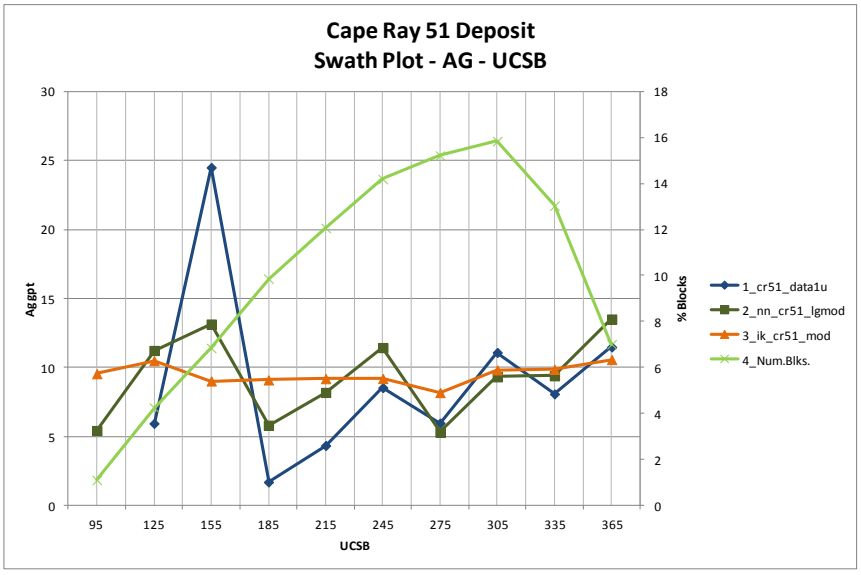




51 Deposit Swath Plots – Ag





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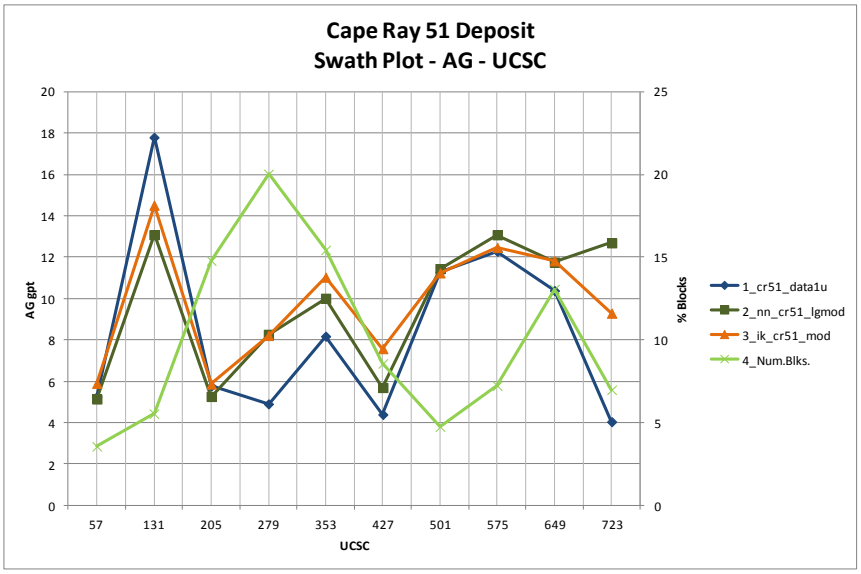

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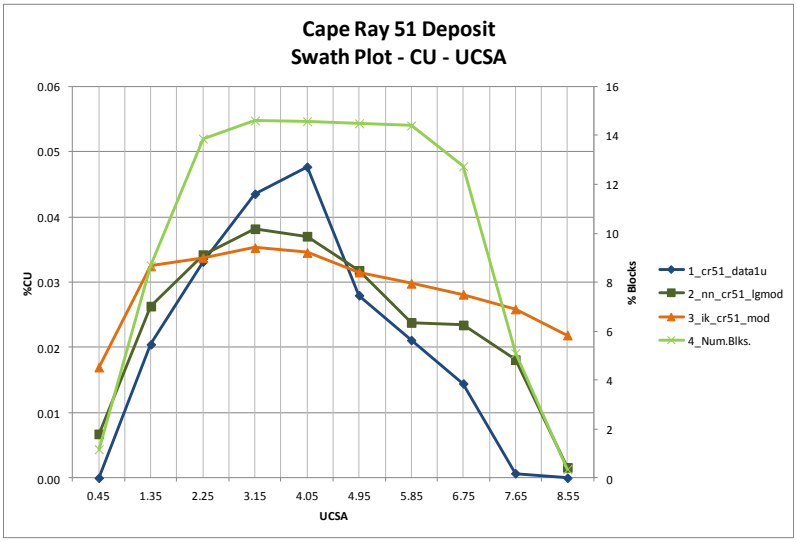

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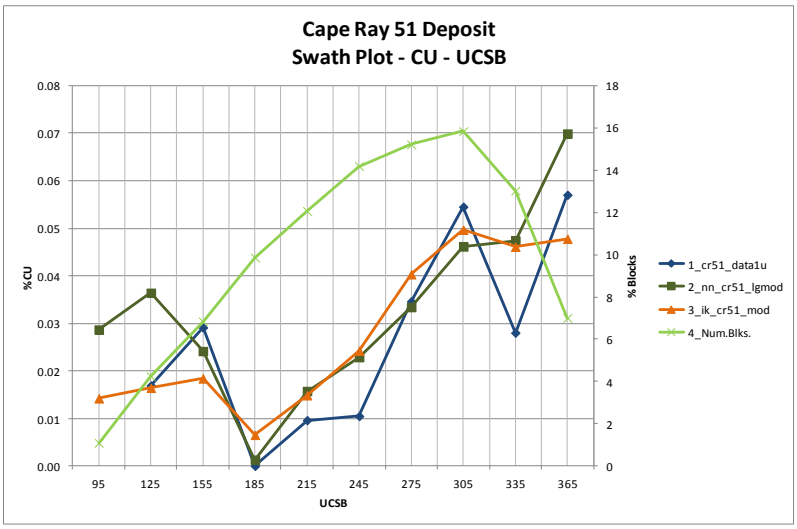

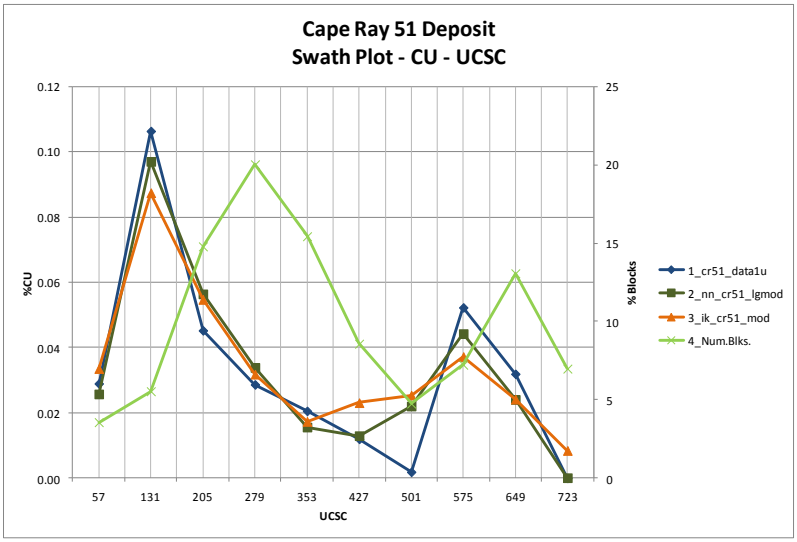

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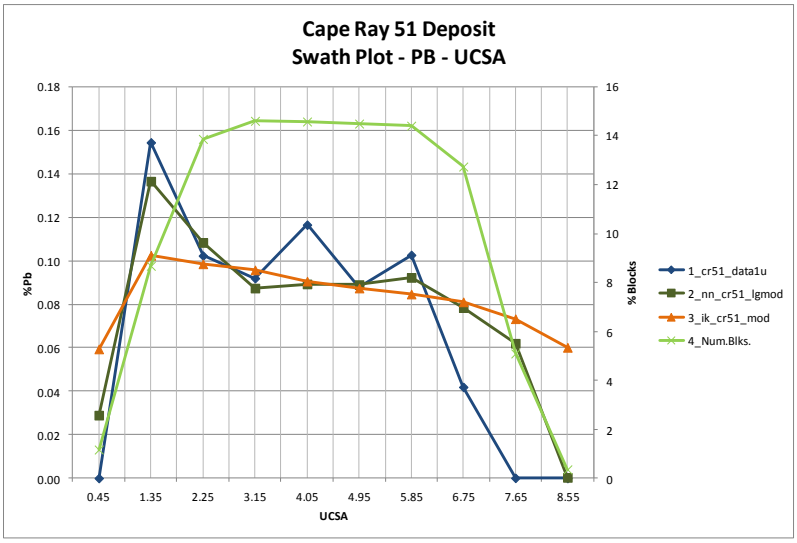
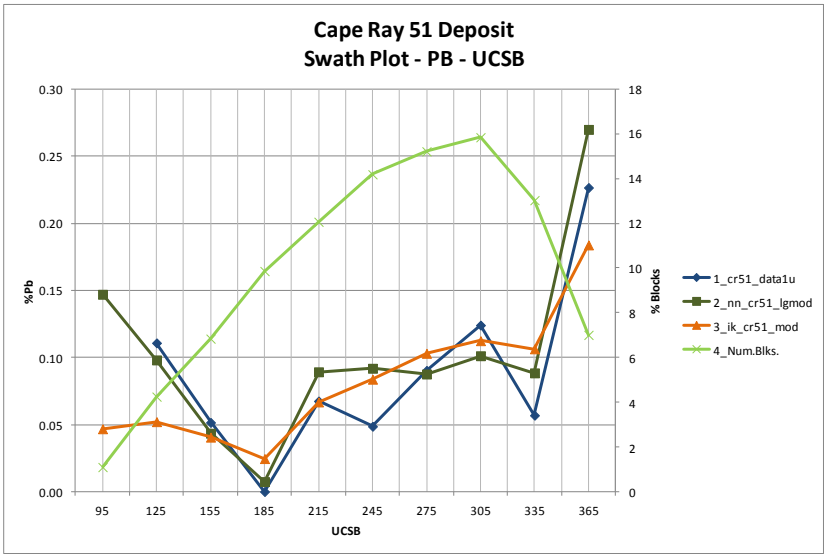
51 Deposit Swath Plots – Pb





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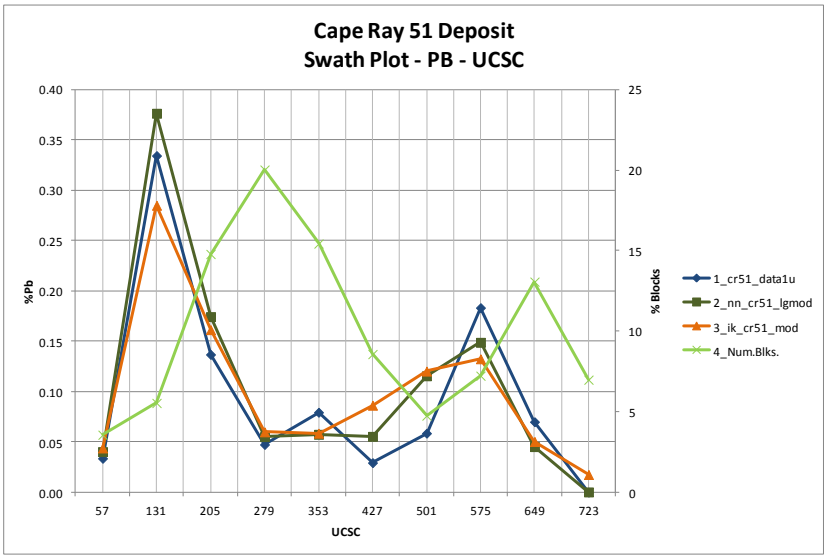

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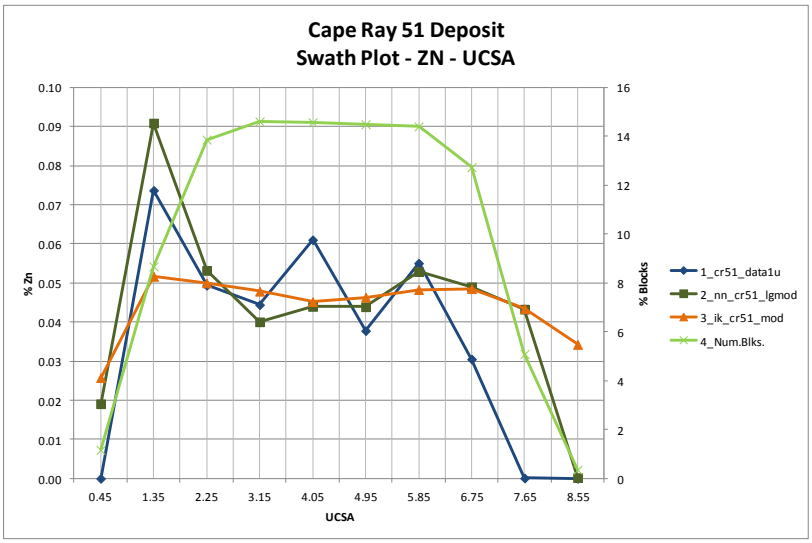

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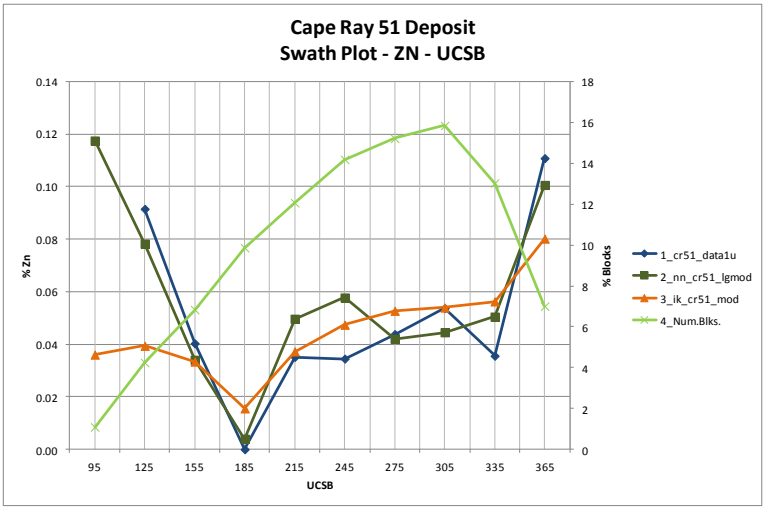

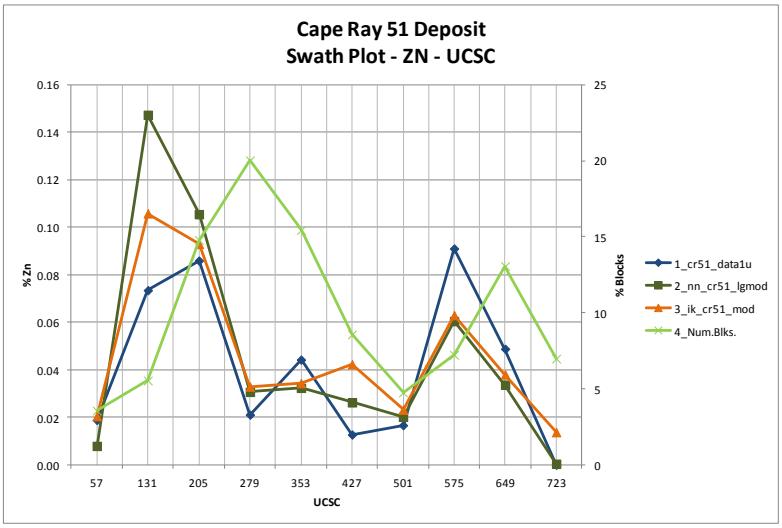

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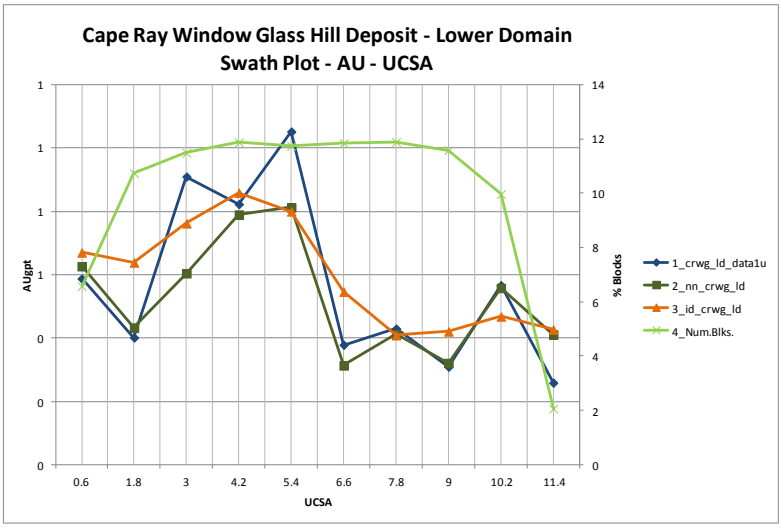
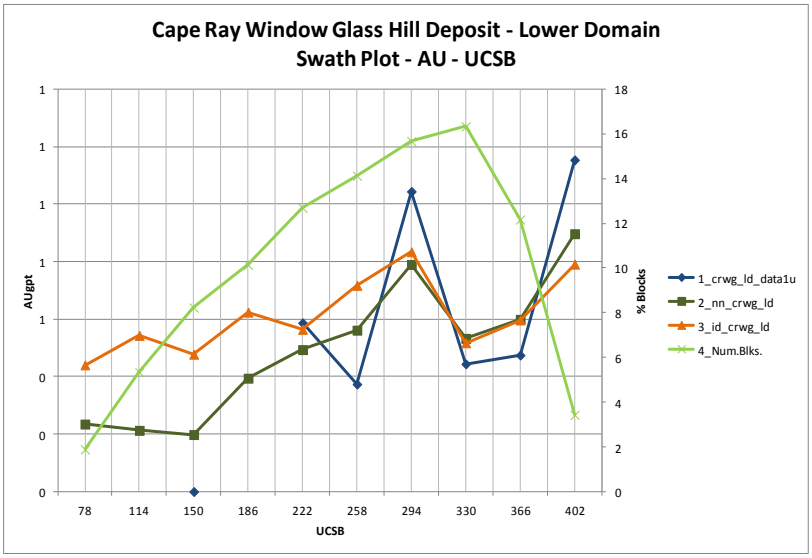
Windowglass Hill Deposit Swath Plots – Au – Lower Domain





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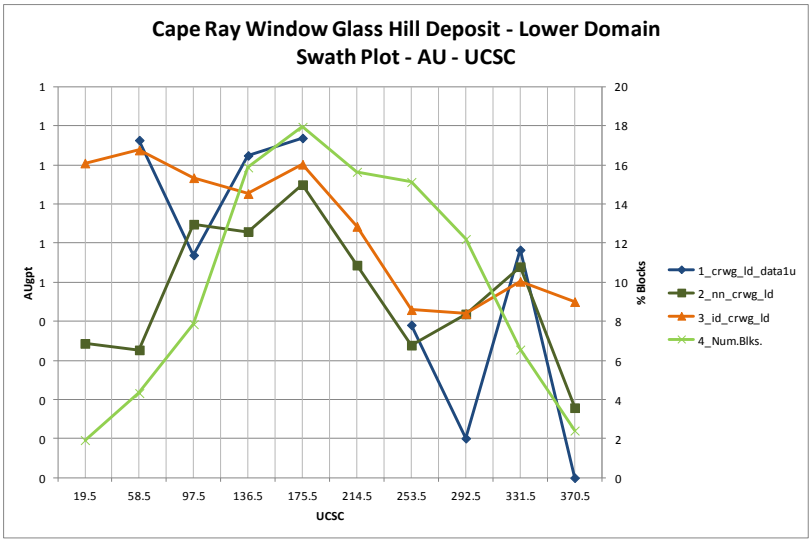
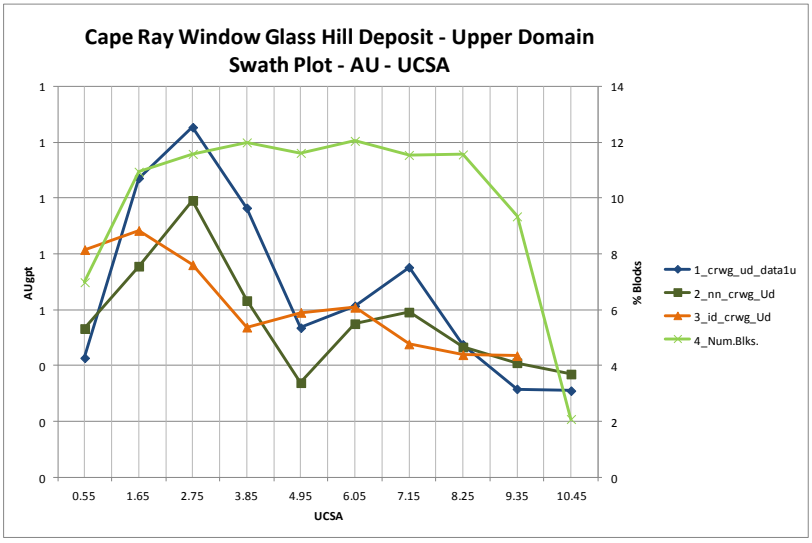


Windowglass Hill Deposit Swath Plots – Au – Upper Domain



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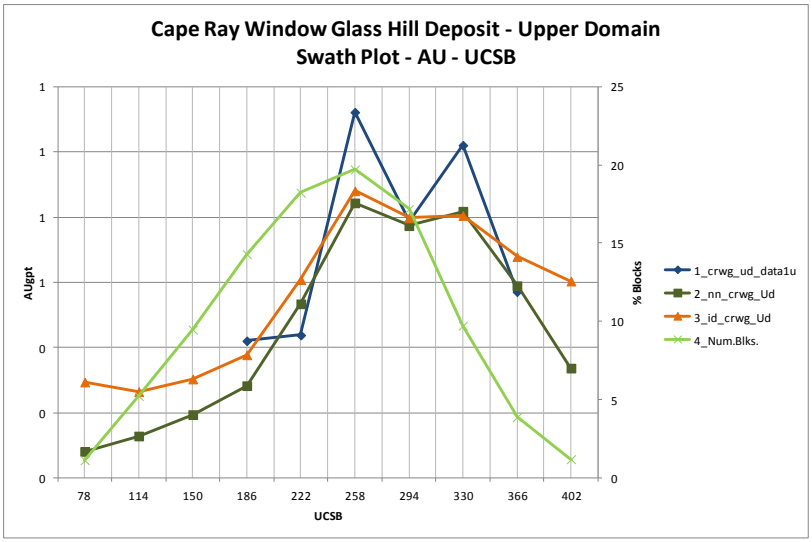
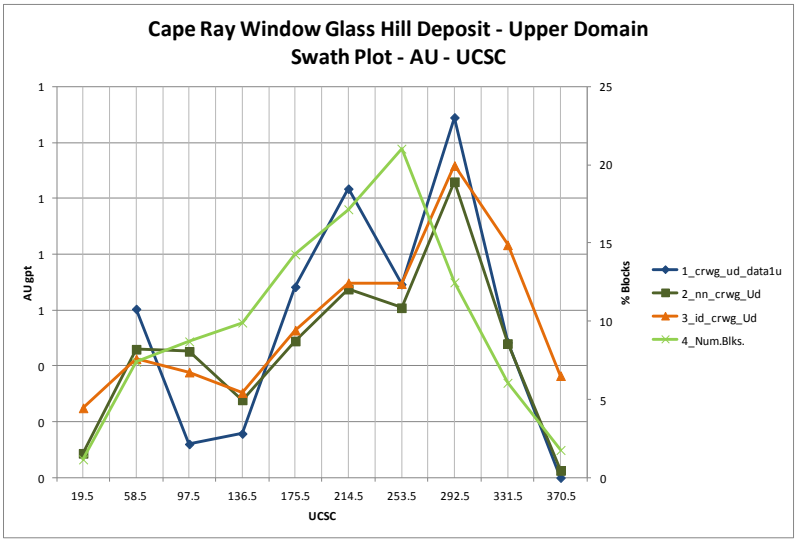
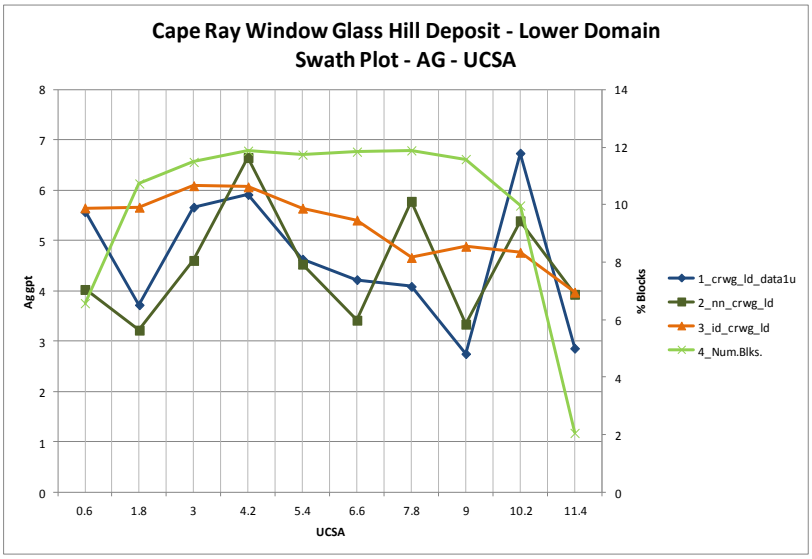
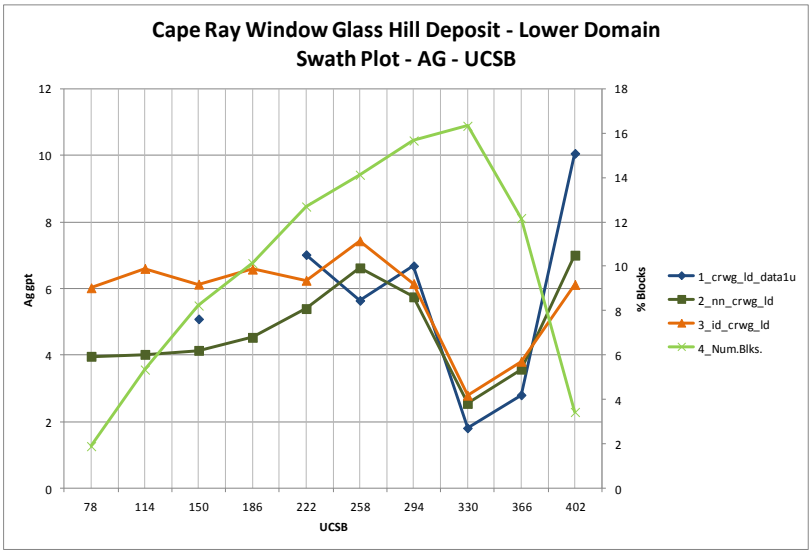




Windowglass Hill Deposit Swath Plots – Ag – Lower Domain





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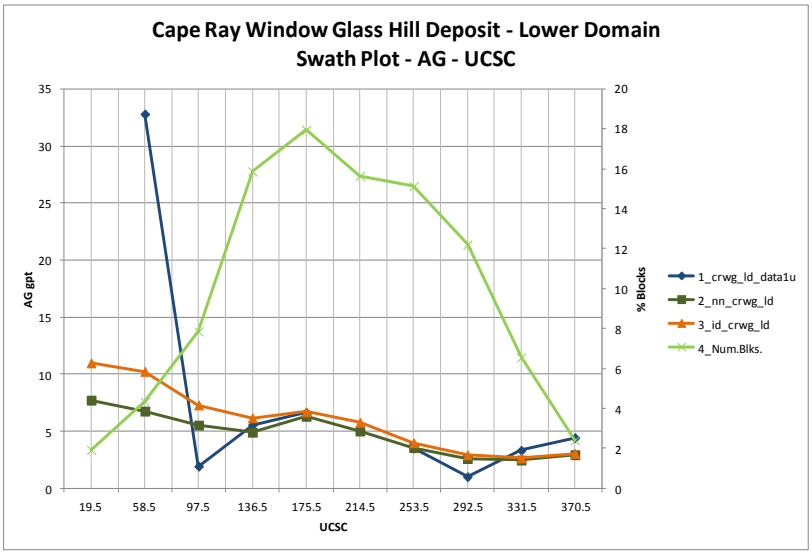
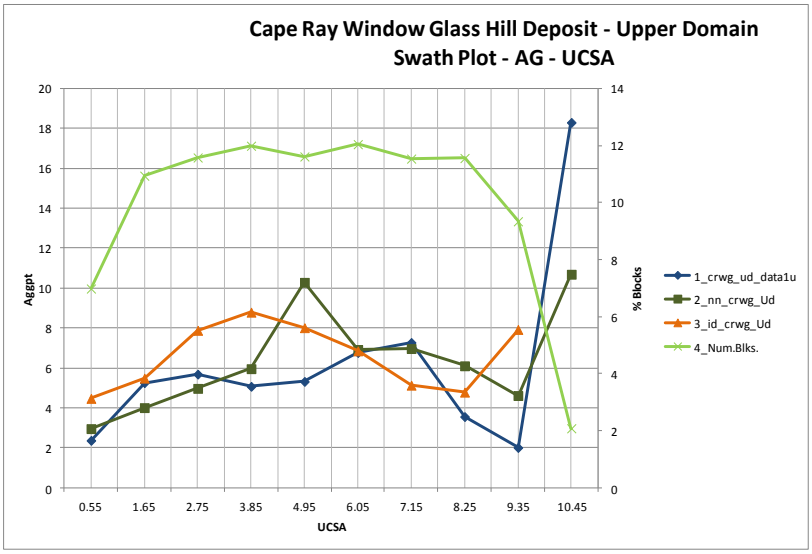


Windowglass Hill Deposit Swath Plots – Ag – Upper Domain



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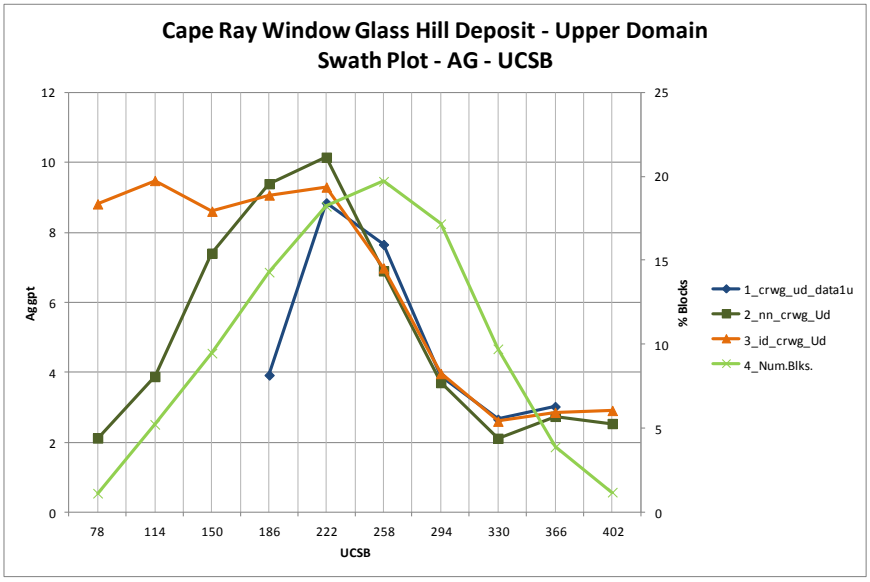

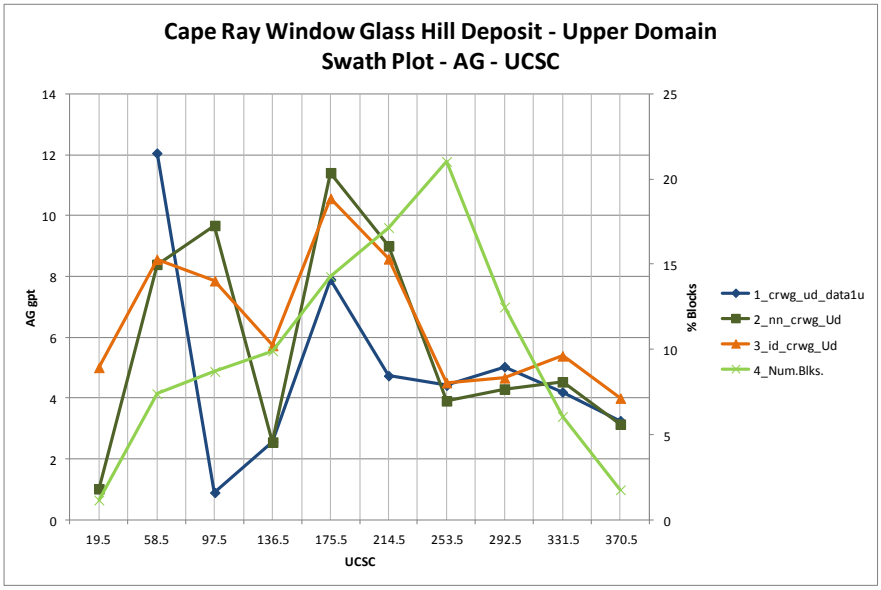

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Appendix D Variance Correction Theory

The "averaging" process that goes on during interpolation within the block model tends to reduce the variance from its original level. Overall the mean for the entire population remains unaffected. However, since a cut-off grade is used to separate the above- and below-cut-off populations, their specific means are now affected due to this homogenization, or smoothing, of individual estimates. The interpolated mean can be lower or higher than the original mean depending upon whether the cut-off grade is above or below the original mean.

Regression methods such as Kriging may result in an over-smoothing or under-smoothing of the grade variability producing a block grade distribution with a variance that is lower or higher than expected. This expected variance can be calculated using Krige's relationship which states that the dispersion variance for the samples within the deposit is the sum of the dispersion variance of samples within the blocks and the dispersion variance of the blocks within the deposit. This relationship can be written as:

$D^2(\bullet,A) = D^2(\bullet,\nu) + D^2(\nu,A)$ Eq. 5.1

Where • are the samples, v are the blocks and A is the deposit.

In terms of average variance (known as gamma bar and calculated using the CAE Studio 3 FFUNC) the dispersion variance for samples within blocks can be written as:

$$D^2(\bullet,\nu) = \bar{\gamma}(\nu,\nu) - \bar{\gamma}(\bullet,\bullet)$$ Eq. 5.2

As $\bar{\gamma}(\bullet,\bullet) = 0$ (the variance of a sample with itself equals 0) we can rewrite the above as:

$$D^2(\bullet,\nu) = \bar{\gamma}(\nu,\nu)$$ Eq. 5.3

And by substituting into the first equation we get:

$D^2(\bullet,A) = \bar{\gamma}(\nu,\nu) + D^2(\nu,A)$ Eq. 5.4

As the left side is equivalent to the sample variance (σ^2) we can reorder so that we can determine the block variance in terms of the average variogram of blocks to blocks (gamma bar) and the sample variance:

$D^2(\nu,A) = \sigma^2 - \bar{\gamma}(\nu,\nu)$ Eq. 5.5

Of these 3 terms we can get the sample variance (σ^2) from our variogram and we can calculate the gamma bar value, using as input our variogram model and our block size. Using this information, and a change of support model we can correct for any differences due to over or

under smoothing.

Gamma Bar is the calculated value of what *theoretically* should be the "Sample variance in Block" (V_b). As this value is based on the variogram we know that it is directly proportional to the variogram sill, which can be considered equivalent to the “Sample Variance in Deposit” (V_d). Knowing these two values we can then determine what **proportion of the total variance** is represented by the "Sample Variance in deposit" (V_d) using the following relationship:

$$P_{VB} = \frac{(V_d) - (V_b)}{(V_d)} = \frac{[\text{Variogram Sill}] - [\text{Gamma Bar}]}{[\text{Variogram Sill}]} \qquad \text{Eq. 5.6}$$

Using the declusterised variance for our mineral zone (V_d) we can then determine the variance that **should** be attributable to the “Variance between blocks” using the following relationship:

$V_{B\text{-theoretical}} = P_{VB} * V_d$ Eq. 5.7

Now that we have the theoretical “Variance between blocks” we can then compare this with the actual “Variance between blocks”. The latter is the variance of our Kriged model using our new support. By dividing the theoretical by the actual we get the smoothing ratio:

$$\text{Smoothing ratio} = \frac{V_{B\text{-theoretical}}}{V_{B\text{-actual}}} \qquad \text{Eq. 5.8}$$

$$q' = aq^b$$

$$a = \frac{m}{\sqrt{f \times CV^2 + 1}} \left(\frac{CV^2 + 1}{m} \right)^b$$

$$b = \sqrt{\frac{\ln (f \times CV^2 + 1)}{\ln (CV^2 + 1)}}$$

$$f = \frac{\sigma^2_{theoretical}}{\sigma^2_{actual}} \qquad CV = \frac{\sigma}{m}$$

A smoothing ratio less than 0.8 or greater than 1.2 require a variance correction. A smoothing ratio between 0 and 0.5 or greater than 4 could reveal errors in the data or in the models and necessitates further investigation.

There are two common methods of correcting for smoothing: the Affine correction (see Equation 5.9 – **Isaaks & Srivastava, 1989**) and the Indirect Lognormal Shortcut method (see Equation 5.10 – **Isaaks & Srivastava, 1989**). The former is best used for normal distributions and expands or contracts the distribution symmetrically about the mean and preserves the general shape of the original distribution. The Indirect Log Normal Shortcut on the other hand is best, as the name implies, for adjusting highly skewed distributions that approach being log normal. Unlike the Affine, which can result in negative values, the Log Normal Shortcut reduces the skewness of the distribution as the variance is reduced yet the minimum will always be 0.

$$q' = \sqrt{f} * (q - m) + m$$

Eq. 5.9

where $f = \frac{\sigma_{theoretical}}{\sigma_{actual}}$

For the current project the Indirect Lognormal Shortcut method was used

Eq. 5.10

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Appendix E Resource Classification

The following is an extract from the CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted May 10, 2014.

"Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase 'reasonable prospects for eventual economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cut-off grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing.

Inferred Mineral Resource

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

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An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.

Indicated Mineral Resource

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Pre-Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

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Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit."