Report to:

Stratabound MINERALS CORP.

# Preliminary Economic Assessment on the Captain, CNE, and Taylor Brook VMS Deposits, New Brunswick, Canada

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TETRA TECH WARDROP

Report to:

# PRELIMINARY ECONOMIC ASSESSMENT ON THE CAPTAIN, CNE, AND TAYLOR BROOK VMS DEPOSITS, NEW BRUNSWICK, CANADA

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# GLOSSARY

## UNITS OF MEASURE

| | |
|---|---|
| Above mean sea level | amsl |
| Acre | ac |
| Ampere | A |
| Annum (year) | a |
| Billion | B |
| Billion tonnes | Bt |
| Billion years ago | Ga |
| British thermal unit | BTU |
| Centimetre | cm |
| Cubic centimetre | $cm^3$ |
| Cubic feet per minute | cfm |
| Cubic feet per second | $ft^3/s$ |
| Cubic foot | $ft^3$ |
| Cubic inch | $in^3$ |
| Cubic metre | $m^3$ |
| Cubic yard | $yd^3$ |
| Coefficients of Variation | CVs |
| Day | d |
| Days per week | d/wk |
| Days per year (annum) | d/a |
| Dead weight tonnes | DWT |
| Decibel adjusted | dBa |
| Decibel | dB |
| Degree | ° |
| Degrees Celsius | °C |
| Diameter | ø |
| Dollar (American) | US$ |
| Dollar (Canadian) | Cdn$ |
| Dry metric ton | dmt |
| Foot | ft |
| Gallon | gal |
| Gallons per minute (US) | gpm |

| | |
|---|---|
| Gigajoule | GJ |
| Gigapascal | GPa |
| Gigawatt | GW |
| Gram | g |
| Grams per litre | g/L |
| Grams per tonne | g/t |
| Greater than | > |
| Hectare (10,000 $m^2$) | ha |
| Hertz | Hz |
| Horsepower | hp |
| Hour | h |
| Hours per day | h/d |
| Hours per week | h/wk |
| Hours per year | h/a |
| Inch | in |
| Kilo (thousand) | k |
| Kilogram | kg |
| Kilograms per cubic metre | $kg/m^3$ |
| Kilograms per hour | kg/h |
| Kilograms per square metre | $kg/m^2$ |
| Kilometre | km |
| Kilometres per hour | km/h |
| Kilopascal | kPa |
| Kilotonne | kt |
| Kilovolt | kV |
| Kilovolt-ampere | kVA |
| Kilovolts | kV |
| Kilowatt | kW |
| Kilowatt hour | kWh |
| Kilowatt hours per tonne (metric ton) | kWh/t |
| Kilowatt hours per year | kWh/a |
| Less than | < |
| Litre | L |
| Litres per minute | L/m |
| Megabytes per second | Mb/s |
| Megapascal | MPa |
| Megavolt-ampere | MVA |
| Megawatt | MW |
| Metre | m |
| Metres above sea level | masl |
| Metres Baltic sea level | mbsl |
| Metres per minute | m/min |
| Metres per second | m/s |
| Metric ton (tonne) | t |
| Microns | µm |
| Milligram | mg |

| | |
|---|---|
| Milligrams per litre | mg/L |
| Millilitre | mL |
| Millimetre | mm |
| Million | M |
| Million bank cubic metres | $Mbm^3$ |
| Million bank cubic metres per annum | $Mbm^3/a$ |
| Million tonnes | Mt |
| Minute (plane angle) | ' |
| Minute (time) | min |
| Month | mo |
| Ounce | oz |
| Pascal | Pa |
| Centipoise | mPa·s |
| Parts per million | ppm |
| Parts per billion | ppb |
| Percent | % |
| Pound(s) | lb |
| Pounds per square inch | psi |
| Revolutions per minute | rpm |
| Second (plane angle) | " |
| Second (time) | s |
| Specific gravity | SG |
| Square centimetre | $cm^2$ |
| Square foot | $ft^2$ |
| Square inch | $in^2$ |
| Square kilometre | $km^2$ |
| Square metre | $m^2$ |
| Thousand tonnes | kt |
| Three Dimensional | 3D |
| Three Dimensional Model | 3DM |
| Tonne (1,000 kg) | t |
| Tonnes per day | t/d |
| Tonnes per hour | t/h |
| Tonnes per year | t/a |
| Tonnes seconds per hour metre cubed | $ts/hm^3$ |
| Volt | V |
| Week | wk |
| Weight/weight | w/w |
| Wet metric ton | wmt |
| Year (annum) | a |

## ABBREVIATIONS AND ACRONYMS

| | |
|---|---|
| Aboriginal Affairs Secretariat | AAS |

| | |
|---|---|
| acid mine drainage | AMD |
| acid rock drainage | ARD |
| ALS Group | ALS |
| Bathurst Joint Venture | BJV |
| Billiton Canada Ltd. | Billiton |
| Bathurst Mining Camp | BMC |
| Canadian Environmental Assessment Act | CEEA |
| Canadian Environmental Assessment | CEA |
| Canadian Institute of Mining | CIM |
| Canadian Water Quality Guidelines for the Protection of Aquatic Life | CCME |
| Captain, CNE, and Taylor Brook Properties | the Properties |
| cobalt | Co |
| Consensus Economic Energy and Metal Forecast | EMCF |
| copper equivalent | CuEQ |
| copper | Cu |
| Department of the Environment | DENV |
| electromagnetic | EM |
| Environmental Impact Assessment | EIA |
| Fisheries and Oceans Canada | DFO |
| Geographic Posiotioning System | GPS |
| gold | Au |
| harmful alteration, disruption or destruction | HADD |
| horizontal loop electromagnetic | HLEM |
| indicator kriging | IK |
| inductively coupled plasma atomic emission spectroscopy | ICP-AES |
| inductively coupled plasma mass spectroscopy | ICP-MS |
| Interim Sediment Quality Guidelines | ISQG |
| internal rate of return | IRR |
| inverse distance squared | ID2 |
| lead | Pb |
| Lerchs-Grossman | LG |
| life of mine | LOM |
| magnesium | MgO |
| National Instrument 43-101 | NI 43-101 |
| National Topographic System | NTS |
| net present value | NPV |
| net smelter return | NSR |
| New Brunswick Department of Natural Resources | NBDNR |
| Noranda Ltd. | Noranda |
| ordinary kriging | OK |
| preliminary economic assessment | PEA |
| present value | PV |
| pulse electromagnetic | PEM |
| Qualified Person | QP |
| quality assurance/quality control | QA/QC |
| Sabina Industries Ltd. | Sabina |

| | |
|---|---|
| silica | $SiO_2$ |
| silver | Ag |
| sodium isopropyl xanthate | SIPX |
| sodium sulphate | $Na_2SO_3$ |
| Standing Committee on Mining and the Environment | SCME |
| Stratabound Minerals Corp. | Stratabound |
| sulphur dioxide | $SO_2$ |
| Teck Corporation | Teck |
| Traditional Knowledge Study | TKO |
| Universal Transverse Mercator | UTM |
| vertical loop electromagnetic | VLEM |
| very low frequency | VLF |
| volcanogenic massive sulphide | VMS |
| Wardrop, a Tetra Tech Company | Tetra Tech |
| zinc equivalent | ZnEQ |
| zinc | Zn |

# 1.0 SUMMARY

## 1.1 INTRODUCTION

Stratabound Minerals Corp. (Stratabound) is a publicly listed and Canadian registered base metal exploration company that is actively exploring and developing base metal (lead (Pb), zinc (Zn), copper (Cu) and cobalt (Co) and gold (Au)) deposits in New Brunswick and Québec.

The Captain, Captain North Extension (CNE) and Taylor Brook properties (the Properties) are located in the Bathurst Mining District in Northumberland and Gloucester Counties, New Brunswick; approximately 45 km southwest of Bathurst. The two mineral claim blocks and mining lease for Captain and CNE are located at latitude 47°17’ N and longitude 65°53’ W. The mineral claim block for Taylor Brook, 12 km northwest of CNE, is located at latitude 47°21’ N and longitude 66°00’ W.

The Properties are comprised of three mineral claim blocks and one mining lease and cover approximately 3,792 ha. The mineral rights to the Properties are 100% held by Stratabound under prospector license number 13727.

Stratabound has retained Wardrop, a Tetra Tech Company (Tetra Tech) to produce a National Instrument 43-101 (NI 43-101) compliant resource estimate and preliminary economic assessment (PEA) for the Properties. Prior to Tetra Tech’s involvement, the Captain deposit had a NI 43-101 technical report completed by Mercator Geological Services.

This PEA conforms to the standards set out in NI 43-101 Standards of Disclosure for Mineral Projects and is in compliance with Form 43-101F1. The resource estimate in this report conforms to the Canadian Institute of Mining (CIM) Mineral Resource and Mineral Reserve definitions referred to in NI 43-101 Standards of Disclosure for Mineral Projects.

The designated Qualified Persons (QPs) for this report are:

- Daniel Coley, P.Eng., Senior Metallurgist with Tetra Tech
- Michael P. Cullen, P.Geo., Senior Geologist with Mercator Geological Services Limited (Mercator)
- Paul J. Daigle, P.Geo., Senior Geologist with Tetra Tech
- Daniel Gagnon, P.Eng., Senior Open Pit Engineer with Tetra Tech
- Mike McLaughlin, P.Eng., Project Manager with Tetra Tech

- Robert Morrison, P.Geo., Lead Resource Geologist with Tetra Tech
- Doug Ramsey, R.P. Bio. (BC), Manager - Environmental Assessment, Permitting, and Natural Resources with Tetra Tech.

Two site visits to the Properties were conducted by Tetra Tech. The first site visit was conducted by Paul Daigle between October 18 and 19, 2010. Mr. Daigle was accompanied during this site visit by Mr. John Duncan, Consulting Project Geologist for Stratabound, Mr. Kevin Vienneau, Consulting Mining Engineer for Stratabound, and Mr. Michael McLaughlin. The second site visit was conducted between November 29 and 30, 2010 by Mr. McLaughlin and Mr. Wenchang Ni, former Senior Mining Engineer with Tetra Tech. Mr. McLaughlin and Mr. Ni were accompanied by Mr. Vienneau.

Site visits to the Captain property were conducted by Mr. Cullen on August 6, 2008 and October 15, 2010. During the site visit on August 6, 2008, Mr. Cullen was accompanied by Mr. Matthew Harrington, Geologist with Mercator.

The Properties are defined by the mineral rights to 183 mineral claims and one mining lease, currently 100% held by Stratabound, and cover an area of approximately 3,792 ha. All claims and the mineral lease are in good standing. Tetra Tech is not aware of any environmental liabilities.

The Properties all lie within the Bathurst Mining Camp (BMC). *The BMC deposits formed in a sediment-covered back-arc continental rift, referred to as the Tetagouche-Exploits backarc basin, during periods when the basin was stratified with a lower anoxic water-column. The basin was subsequently intensely deformed and metamorphosed during multiple collisional events related to east-dipping subduction of the basin* (Goodfellow 2007).

## 1.2 GEOLOGY

### *1.2.1 CAPTAIN AND CNE DEPOSITS*

From Lemmon (2007):

> *The Tetagouche Group dominates the local geology in the CNE and Captain properties. This Group consists mainly of a voluminous suite of Middle Ordovician mafic and felsic volcanic rocks. Felsic volcanic rocks dominate and have compositions that range from dacite to rhyolite (Whitehead and Goodfellow 1978; Winchester and van Staal 1988). The Tetagouche Group is locally divided into several separate formations including (from oldest to youngest) the Patrick Brook, Nepisiguit Falls, Flat landing Brook, Canoe Landing Lake and Boucher Brook Formations.*
>
> *The Nepisiguit Falls Formation hosts the massive sulphide deposits at Heath Steele, Stratmat, (Taylor Brook) and Half Mile Lake and is divided into two*

*members composed mainly of volcanic and mainly sedimentary rocks, respectively. The volcanic member consists primarily of quartz-feldspar crystal tuffs (popularly referred to as quartz-feldspar porphyry or quartz feldspar augen schist) that exhibit characteristics of both lavas and pyroclastic rocks, suggesting unusual circumstances of eruption and emplacement. The sedimentary member comprises green feldspathic or quartzose wackes, siltstones, shales, and minor epiclastic rocks.*

*The Flat Landing Brook Formation is composed of aphyric or feldspar-phyric flows and domes, local felsic hyaloclastites and quartz-feldspar crystal tuffs, alkalic and tholeiitic mafic to intermediate intrusive and extrusive rocks, and minor sedimentary rocks. Where primary features are preserved, rhyolites typically exhibit perlitic fracture patterns and devitrification textures indicating that they were emplaced in a glassy state. Textural modifications resulting from alteration, shearing and dynamic metamorphism obscure and distort primary features and locally create apparent pyroclastic textures, especially in high-strain zones of actively moving flows. Flat Landing Brook rhyolites and crystal tuffs are chemically distinct from the Nepisiguit Falls Brook Formation and includes alkalic to tholeiitic extrusive and subvolcanic intrusive rocks that can generally be divided into separate suites based on geochemistry including:*

- *Tailings Lagoon tholeiitic gabbro and diabase*
- *Forty Mile Brook tholeiitic basalt; Otter Brook tholeiitic gabbros*
- *Tomogonops alkali gabbro*
- *Moody Brook andesite (comprising intermediate to mafic flows, tuffs and agglomerates).*

### 1.2.2 TAYLOR BROOK DEPOSIT

From Walker (1999):

*The Taylor Brook deposit is hosted within an intercalated sequence of felsic ash and lapilli tuff, aphyric to sparsely feldspar-phyric rhyolite flows, hyaloclastite, and minor sedimentary rocks of the Flat Landing Brook Formation. The Taylor Brook deposit has a strike length of approximately 650 m and a down-dip extent of greater than 600 m. The surface trace of the deposit is tadpole shaped with the thickest accumulation of sulphides at the shallow (less than 50 m depths in the western part of the deposit.*

*The sulphide zone comprises one to four stratabound horizons of heavily disseminated to semi-massive and massive sulphides interlayered with hydrothermally altered volcanic rocks. The upper and lower contacts of individual sulphide horizons vary from diffuse to sharp. Sulphide minerals include: pyrite, pyrrhotite, sphalerite, galena, and chalcopyrite.*

## 1.3 EXPLORATION AND DRILLING

Exploration over the Captain and Taylor Brook properties began in the 1950s. The CNE property was not actively explored until 1975. Stratabound began their exploration of the Properties in the late 1980s and has continued intermittently on all three until present.

### 1.3.1 *TAYLOR BROOK*

From 1988 to 2007, Stratabound undertook several exploration programs consisting of ground and airborne magnetic and electromagnetic (EM) geophysical surveys, prospecting, trenching, and several drilling programs. The most recent drilling was completed in 1997. In 2006, Stratabound completed magnetic, MaxMin and very low frequency (VLF) geophysical surveys. There has been no active exploration on Taylor Brook since 2009.

## 1.4 RESOURCE ESTIMATE

### 1.4.1 *CAPTAIN RESOURCE ESTIMATE*

Mercator completed an NI 43-101 compliant resource estimate for the Captain deposit effective December 28, 2010. The resource estimate is in accordance with the CIM Standards on Mineral Resources and Reserves. The Captain deposit was estimated using the Inverse Distance Squared (ID2) interpolation method (Cullen and Harrington 2011).

The following is taken from Cullen and Harrginton (2011):

> *Mineral resources in the Inferred, Indicated and Measured categories are included in the current resource estimate, with the largest percentage reported in the Indicated category. This reflects relative spacing and distribution of the 30 drillholes completed by Stratabound along a strike length of approximately 175 m in the immediate deposit area. The following resource category definitions apply to the current Captain estimate and reflect a progressively decreasing scale of certainty based on proximity of sample composites included in block grade assignments.*
>
> ***Measured Resource Category:*** *Blocks having seven or more contributing assay composites from three separate drillholes, with an average distance of 31.25 m (25% the major axis range) from all contributing composites, located 12.5 m or less (10% the major axis range) from the nearest contributing composite, and falling within a smoothed three dimensional model solid based on the noted block definition parameters.*

*__Indicated Resource Category:__ Blocks having seven or more contributing assay composites from three separate drillholes, with an average distance of 62.5 m (50% of the major axis range) or less from all contributing composites, located 41.7 m or less (33.3% of the major axis range) from the nearest sample, and not classified in the Measured category.*

*__Inferred Resource Category:__ All remaining valid blocks within the Peripheral Domain solid that have interpolated grades and are not included in the Indicated or Measured categories [fall into the Inferred category].*

The Measured, Indicated, and Inferred Mineral Resource Estimates for the Captain copper-cobalt deposit, at a 0.60 copper equivalent (CuEQ%) cut-off, are:

- a Measured Resource of 68 kt at 1.09 Cu% and 0.059 Co%
- an Indicated Resource of 938 kt at 1.03 Cu% and 0.050 Co%
- an Inferred Resource of 960 kt at 0.64 Cu% and 0.039 Co%.

The mineral resource estimates are presented in Table 1.1 and Table 1.2.

**Table 1.1 Measured and Indicated Resource Estimate for the Captain Deposit (modified from Cullen and Harrington, 2011)**

| *CuEQ% Cut-off | Density | Tonnes ('000) | Cu% | Co% | Au (g/t) |
|---|---|---|---|---|---|
| **Measured** | | | | | |
| 1.40 | - | 32 | 1.86 | 0.057 | 0.29 |
| 1.00 | - | 46 | 1.51 | 0.056 | 0.25 |
| **0.60** | **-** | **68** | **1.09** | **0.059** | **0.20** |
| **Indicated** | | | | | |
| 1.40 | - | 416 | 1.74 | 0.045 | 0.30 |
| 1.00 | - | 621 | 1.41 | 0.047 | 0.25 |
| **0.60** | **-** | **938** | **1.03** | **0.050** | **0.20** |
| **Measured + Indicated** | | | | | |
| 1.40 | - | 448 | 1.75 | 0.046 | 0.30 |
| 1.00 | - | 667 | 1.42 | 0.048 | 0.25 |
| **0.60** | **-** | **1,006** | **1.03** | **0.051** | **0.20** |

Note: *CuEQ% = Cu% + (Co% x 9.25) as used in previous resource estimate and based on comparable relative three-year copper and cobalt pricing and 100% recovery for both metals.

**Table 1.2 Inferred Resource Estimate for the Captain Deposit (modified from Cullen and Harrington, 2011)**

| *CuEQ% Cut-off | Density | Tonnes ('000) | Cu% | Co% | Au (g/t) |
|---|---|---|---|---|---|
| 1.40 | - | 162 | 1.47 | 0.040 | 0.24 |
| 1.00 | - | 298 | 1.18 | 0.038 | 0.20 |
| **0.60** | **-** | **960** | **0.64** | **0.039** | **0.12** |

Note: *CuEQ% = Cu% + (Co% x 9.25) as used in previous resource estimate and based on comparable relative three-year copper and cobalt pricing and 100% recovery for both metals.

### 1.4.2 CNE RESOURCE ESTIMATE

Tetra Tech completed an NI 43-101 compliant resource estimate for the CNE deposit effective May 12, 2011. The mineral resource for the CNE deposit is categorized as having Measured, Indicated and Inferred Resources, based on historical drilling and limited quality assurance/quality control (QA/QC) data, economic parameters, and on the drillhole sample support. The resource estimate was prepared using Datamine™ software Version 3.19.3638. Due to the geological complexity of the deposit, the resource was first estimated based on an Indicator Kriged (IK) probability model. Cells within the IK model which equalled to or surpassed a nominated probability of meeting or exceeding a predetermined cut-off grade were subsequently estimated using Ordinary Kriging (OK).

The final resource estimate was prepared using the OK interpolation method. No recoveries have been applied to the interpolated estimates as these were applied during the Gemcom Whittle™ pit optimization subsequent to the resource estimate.

Copper and lead-zinc-silver mineralization occur in two different, yet overlapping volumes within the CNE deposit. Thus a zinc equivalent (ZnEQ%) resource estimation was required to combine all metals for reporting.

The CNE resource statement was reported at a 3% ZnEQ cut-off. Based on the mine optimization and design work completed in the subsequent PEA, it was determined that the mined resource from the CNE deposit would be equitable to a 1.5% ZnEQ cut-off. These resources are presented in Table 1.3 to Table 1.5. The volume and grade at a ZnEQ of 1.5% is highlighted in the tables below to represent the CNE cut-off included in the PEA analysis based on the mine design optimization.

**Table 1.3 Measured Resource Estimate for the CNE Deposit by ZnEQ% Cut-off**

| ZnEQ% Cut-off | Tonnes | ZnEQ (t) | Zn (t) | Pb (t) | Cu (t) | Ag (oz) | Grade (ZnEQ%) | Grade (Zn%) | Grade (Pb%) | Grade (Cu%) | Grade (Ag g/t) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 0.5 | 39,149 | 2,270 | 2,181 | 719 | 26 | 77,845 | 5.80 | 5.57 | 1.84 | 0.07 | 61.85 |
| 1 | 38,535 | 2,266 | 2,180 | 719 | 24 | 77,760 | 5.88 | 5.66 | 1.87 | 0.06 | 62.76 |
| **1.5** | **37,710** | **2,255** | **2,174** | **719** | **21** | **77,489** | **5.98** | **5.77** | **1.91** | **0.06** | **63.91** |
| 2 | 36,507 | 2,234 | 2,156 | 715 | 19 | 76,577 | 6.12 | 5.90 | 1.96 | 0.05 | 65.24 |
| 2.5 | 34,367 | 2,185 | 2,113 | 705 | 16 | 74,692 | 6.36 | 6.15 | 2.05 | 0.05 | 67.60 |
| 3 | 30,802 | 2,086 | 2,018 | 681 | 13 | 71,225 | 6.77 | 6.55 | 2.21 | 0.04 | 71.92 |
| 3.5 | 27,264 | 1,971 | 1,906 | 651 | 10 | 67,098 | 7.23 | 6.99 | 2.39 | 0.04 | 76.55 |
| 4 | 24,386 | 1,863 | 1,800 | 622 | 7 | 63,166 | 7.64 | 7.38 | 2.55 | 0.03 | 80.57 |
| 4.5 | 21,296 | 1,733 | 1,669 | 587 | 6 | 58,878 | 8.14 | 7.84 | 2.76 | 0.03 | 85.99 |
| 5 | 18,905 | 1,619 | 1,555 | 554 | 5 | 55,166 | 8.57 | 8.23 | 2.93 | 0.03 | 90.76 |

**Table 1.4 Indicated Resource Estimate for the CNE Deposit by ZnEQ% Cut-off**

| ZnEQ% Cut-off | Tonnes | ZnEQ (t) | Zn (t) | Pb (t) | Cu (t) | Ag (oz) | Grade (ZnEQ%) | Grade (Zn%) | Grade (Pb%) | Grade (Cu%) | Grade (Ag g/t) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 0.5 | 344,521 | 14,451 | 12,751 | 4,666 | 506 | 554,454 | 4.19 | 3.70 | 1.35 | 0.15 | 50.06 |
| 1 | 309,598 | 14,190 | 12,703 | 4,650 | 414 | 548,985 | 4.58 | 4.10 | 1.50 | 0.13 | 55.15 |
| **1.5** | **277,044** | **13,782** | **12,566** | **4,596** | **302** | **537,992** | **4.97** | **4.54** | **1.66** | **0.11** | **60.40** |
| 2 | 247,430 | 13,262 | 12,250 | 4,501 | 213 | 518,904 | 5.36 | 4.95 | 1.82 | 0.09 | 65.23 |
| 2.5 | 218,639 | 12,612 | 11,737 | 4,340 | 155 | 491,661 | 5.77 | 5.37 | 1.99 | 0.07 | 69.94 |
| 3 | 191,346 | 11,862 | 11,095 | 4,142 | 108 | 462,042 | 6.20 | 5.80 | 2.16 | 0.06 | 75.11 |
| 3.5 | 167,326 | 11,082 | 10,408 | 3,918 | 71 | 430,519 | 6.62 | 6.22 | 2.34 | 0.04 | 80.03 |
| 4 | 145,202 | 10,255 | 9,647 | 3,674 | 45 | 397,964 | 7.06 | 6.64 | 2.53 | 0.03 | 85.25 |
| 4.5 | 122,954 | 9,310 | 8,744 | 3,376 | 32 | 359,655 | 7.57 | 7.11 | 2.75 | 0.03 | 90.98 |
| 5 | 103,865 | 8,403 | 7,887 | 3,070 | 23 | 324,111 | 8.09 | 7.59 | 2.96 | 0.02 | 97.06 |

**Table 1.5 Inferred Resource Estimate for the CNE Deposit by ZnEQ% Cut-off**

| ZnEQ% Cut-off | Tonnes | ZnEQ (t) | Zn (t) | Pb (t) | Cu (t) | Ag (oz) | Grade (ZnEQ%) | Grade (Zn%) | Grade (Pb%) | Grade (Cu%) | Grade (Ag g/t) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 0.5 | 27,211 | 628 | 501 | 212 | 36 | 23,273 | 2.31 | 1.84 | 0.78 | 0.13 | 26.60 |
| 1 | 22,360 | 592 | 489 | 211 | 25 | 22,427 | 2.65 | 2.19 | 0.94 | 0.11 | 31.20 |
| **1.5** | **16,517** | **520** | **453** | **199** | **10** | **19,912** | **3.15** | **2.74** | **1.20** | **0.06** | **37.50** |
| 2 | 12,419 | 449 | 401 | 177 | 4 | 17,317 | 3.62 | 3.23 | 1.42 | 0.03 | 43.37 |
| 2.5 | 9,197 | 377 | 341 | 144 | 3 | 14,377 | 4.10 | 3.71 | 1.57 | 0.03 | 48.62 |
| 3 | 6,854 | 313 | 290 | 116 | 2 | 11,647 | 4.57 | 4.23 | 1.69 | 0.02 | 52.86 |
| 3.5 | 5,118 | 256 | 242 | 93 | 1 | 9,198 | 5.01 | 4.72 | 1.81 | 0.01 | 55.90 |
| 4 | 3,461 | 194 | 185 | 70 | - | 6,436 | 5.62 | 5.34 | 2.03 | - | 57.85 |
| 4.5 | 2,443 | 151 | 141 | 57 | - | 4,468 | 6.18 | 5.77 | 2.34 | - | 56.89 |
| 5 | 2,003 | 130 | 121 | 50 | - | 3,460 | 6.50 | 6.03 | 2.51 | - | 53.72 |

### 1.4.3 TAYLOR BROOK RESOURCE ESTIMATE

There have been no recent exploration activities on Taylor Brook deposit since 2009 and Stratabound has not conducted any drilling on the Taylor Brook deposit since 1997.

Tetra Tech completed an NI 43-101 compliant resource estimate for the Taylor Brook deposit effective May 12, 2011. The mineral resource for the Taylor Brook deposit is categorized as having Indicated and Inferred Resources, based on historical drilling and limited QA/QC data, economic parameters, and on the drillhole sample support. The resource estimate was prepared using OK interpolation method. No recoveries have been applied to the interpolated estimates as these were applied during the Gemcom Whittle™ pit optimization subsequent to the resource estimate.

The mineral resource estimates for the Taylor Brook deposit at 1.60% ZnEQ% cut-off grade are:

- an Indicated Resource of 243,000 t at 1.69 Zn%, 0.85 Pb%, 0.02 Cu% and 33.42 g/t Ag
- an Inferred Resource of 102,000 t at 1.70 Zn%, 0.87 Pb%, 0.02 Cu% and 32.59 g/t Ag.

The OK resource estimates for the massive sulphide zones were made at ZnEQ% cut-off grades from 0.6 to 2.0 Zn% and are presented in Table 1.6 and Table 1.7. No recoveries have been applied to the interpolated estimates.

**Table 1.6 Indicated Resource Estimate for the Taylor Brook Deposit**

| ZnEQ% Cut-off | Density | Tonnes ('000 t) | Zn% | Pb% | Cu% | ZnEQ% | Ag (g/t) |
|---|---|---|---|---|---|---|---|
| 0.60 | 3.19 | 1,706 | 0.99 | 0.44 | 0.02 | 1.13 | 19.24 |
| 0.80 | 3.21 | 1,212 | 1.14 | 0.52 | 0.02 | 1.31 | 21.36 |
| 1.00 | 3.23 | 898 | 1.26 | 0.59 | 0.02 | 1.45 | 23.34 |
| 1.20 | 3.27 | 628 | 1.38 | 0.66 | 0.02 | 1.60 | 25.71 |
| 1.40 | 3.27 | 390 | 1.53 | 0.76 | 0.02 | 1.79 | 29.52 |
| **1.60** | **3.28** | **243** | **1.67** | **0.85** | **0.02** | **1.97** | **33.42** |
| 1.80 | 3.27 | 137 | 1.85 | 0.95 | 0.02 | 2.18 | 36.65 |
| 2.00 | 3.31 | 80 | 1.99 | 1.07 | 0.03 | 2.39 | 41.57 |

**Table 1.7 Inferred Resource Estimate for the Taylor Brook Deposit**

| ZnEQ% Cut-off | Density | Tonnes ('000 t) | Zn% | Pb% | Cu% | ZnEQ% | Ag (g/t) |
|---|---|---|---|---|---|---|---|
| 0.60 | 3.13 | 1,786 | 0.88 | 0.31 | 0.03 | 0.96 | 13.78 |
| 0.80 | 3.21 | 1,037 | 1.06 | 0.40 | 0.02 | 1.17 | 16.62 |
| 1.00 | 3.28 | 634 | 1.21 | 0.47 | 0.03 | 1.35 | 17.64 |
| 1.20 | 3.32 | 332 | 1.39 | 0.58 | 0.03 | 1.57 | 23.11 |
| 1.40 | 3.42 | 181 | 1.55 | 0.72 | 0.02 | 1.79 | 26.75 |
| **1.60** | **3.44** | **102** | **1.70** | **0.87** | **0.02** | **2.01** | **32.59** |
| 1.80 | 3.46 | 57 | 1.87 | 1.00 | 0.03 | 2.26 | 36.36 |
| 2.00 | 3.52 | 42 | 1.97 | 1.07 | 0.03 | 2.39 | 37.57 |

Tetra Tech concludes that the Taylor Brook deposit warrants further investigation and development.

Tetra Tech believes further exploration is warranted and recommends that additional drilling be conducted to further investigate and develop the known Taylor Brook deposit as the deposit has not been delineated laterally to the northwest and to determine the continuity of geology and grade at depth and laterally to the east. Tetra Tech recommends a definition drill program of approximately 2,900 m at an estimated cost of $350,000.

## 1.5 MINERAL PROCESSING AND METALLURGICAL TESTING

This report summarizes the various investigations and test work that have been undertaken. The summary recommendations are used to provide guidance on the chosen processing and milling option.

### *1.5.1 ORE COMPOSITION*

Table 1.8 lists the resource and grades for the following: a bulk sample mined in 1990, the Mining Campaign #1 in 1991, and the Mining Campaign #2 in 1992. The ore was toll milled at the Heath Steel mill in three separate batches in 1990, 1991 and 1992. As part of the toll mill contract, the ore assays were cut by 5% prior to introduction to the mill operation.

**Table 1.8 CNE Metal Assays**

| Year | Dry Tonnes | Assays | | | Assays (cut 5%) | | |
|---|---|---|---|---|---|---|---|
| | | Pb (%) | Zn (%) | Ag (g/t) | Pb (%) | Zn (%) | Ag (g/t) |
| 1990 | 11,444 | 5.57 | 12.00 | 173 | 5.29 | 11.40 | 164 |
| 1991 | 14,559 | 4.82 | 11.94 | 152 | 4.58 | 11.34 | 144 |
| 1992 | 13,619 | 2.83 | 7.65 | 98 | 2.69 | 7.27 | 93 |

The CNE deposit can be described as a relatively high-grade lead-zinc-silver deposit.

### 1.5.2 TEST WORK

The following test work programs have been executed on the CNE deposit.

#### CNE TRENCH SAMPLE

The mineral grain size data indicated a preliminary grind for metallurgical testing of 38 µm (400 mesh) or 53 µm (270 mesh) with a regrind to improve galena liberation.

As this work was preliminary and inconclusive, Tetra Tech recommends further investigation particularly of the gold and silver occurrences, as well as other typical mineral occurrences in the ore and gangue.

#### TEST WORK AT BRUNSWICK MINING AND SMELTING LAB

The plant testing was directed at understanding the reasons for and reducing the zinc in the lead concentrate.

The test work concluded that the bulk concentrate could be eliminated by the removal of 11% of the zinc from the copper-lead cleaner concentrate, which would reduce the amount of the zinc in the lead and other related circuits by approximately 6% of the total zinc in the feed.

#### BULK SAMPLE FLOTATION

Several process changes were explored, which included:

- an increased pH using lime instead of soda ash
- ceasing aeration
- stopping sulphur dioxide ($SO_2$)
- additions to the copper-lead circuit
- using less sodium isopropyl xanthate (SIPX) and more Aerofloat 241 (a more specific galena collector).

None of these initiatives resulted in an appreciable reduction in the zinc content of the lead concentrate. However, increased lime addition increased the amount of lead in the copper rougher tails.

#### MINING CAMPAIGN 2

From this work it was demonstrated that acceptable grades and recoveries are achievable (Table 1.9). Smelters typically find zinc grades above 53% acceptable.

In the absence of a deleterious element assessment, a statement regarding the extent of smelter penalties and treatment charges cannot be made.

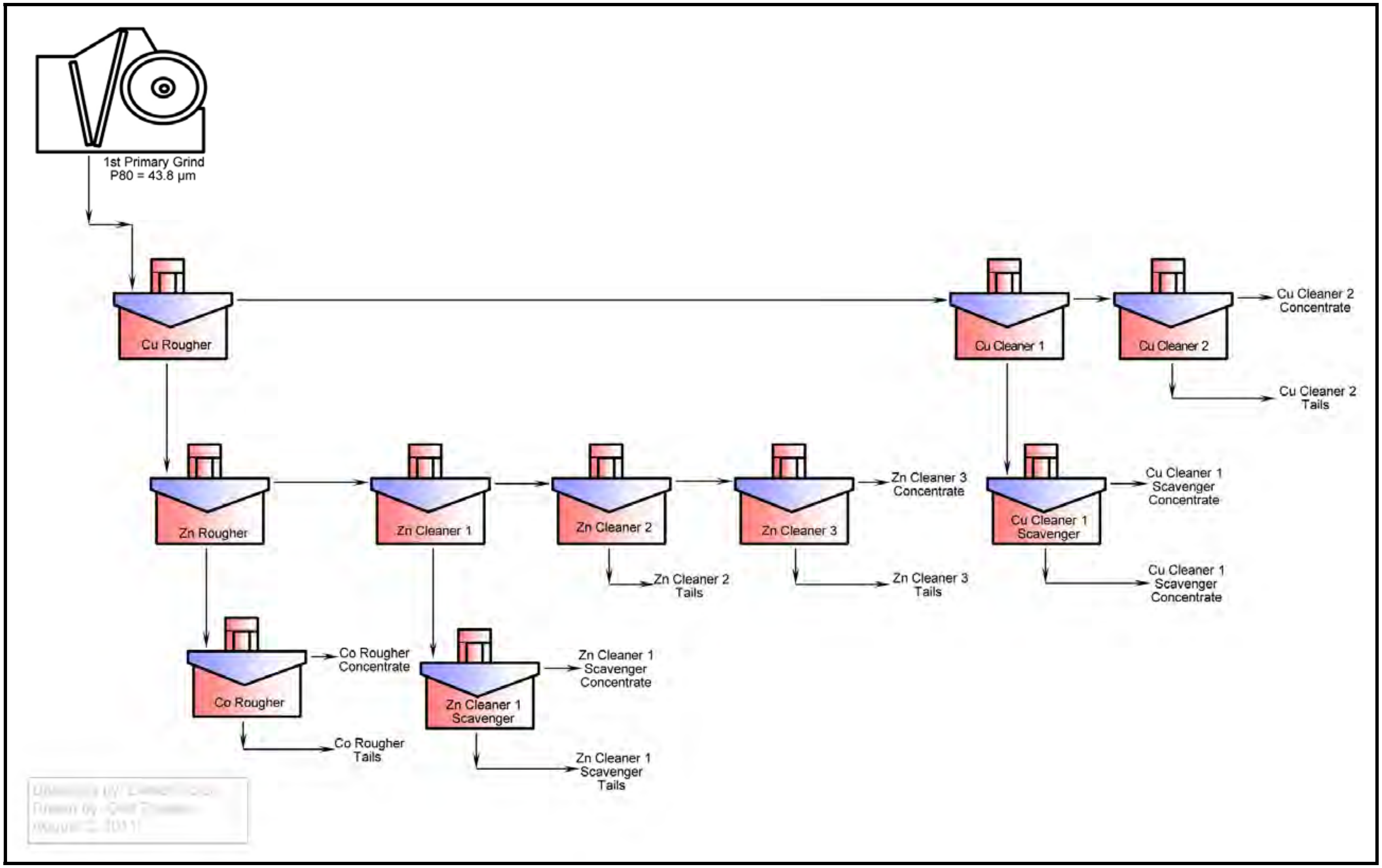
**Table 1.9 Achievable Zinc Grades and Recoveries**

| Year | Dry Tonnes | Zn (%) | |
|---|---|---|---|
| | | Grade | Recovery |
| 1990 | 1,716 | 53.4 | 72.5 |
| 1991 | 24,92 | 54.8 | 82.7 |
| 1992 | 1,352 | 54.6 | 74.6 |

### 1.5.3 *DELETERIOUS ELEMENTS ASSESSMENT*

Test work to identify the occurrence of deleterious elements is yet to be conducted. This is necessary to arrive at an estimation of smelter acceptance and associated penalties. This should be given serious consideration for the next phase of test work.

## Figure 1.1 Flow Sheet without Bulk Concentrate



### 1.5.4 ANALYSIS AND DISCUSSION

Based on the investigations carried out on the CNE deposit there is sufficient evidence to suggest that the process is technically viable.

While crushing and grindability work may be necessary for determining power consumption contribution, it has been established that the historical Heath Steele mill crushing circuit is sufficient for this ore at the proposed feed rates.

Additionally, the results from the Mining Campaign #2 flotation test work suggest that with a better collector/depressant selectivity, choice and optimization, metal recoveries and grades can be further improved.

## 1.6 MINING METHODS

The Taylor Brook, CNE, and Captain open pits were optimized using the Lerchs-Grossman (LG) pit algorithm. Detailed designs with catch berms and in-pit ramps were only completed for the CNE and Captain deposits. Afterwards, mine development and production schedules were developed. Due to the short mine life, a contract mine operation was selected. Information pertaining to the 3D geological block model that was used in the optimization is provided in Section 14 Mineral Resource Estimates.

For this study, Tetra Tech determined that the CNE deposit was the only deposit feasible to be mined at this time. The deposit is located close to surface and open pit mining methods will be used. Section 16 Mining Methods outlines work completed by Tetra Tech in evaluating an underground mining option, however based on this analysis the underground operation is not viable at this time.

The mine provides mill feed of resource at a rate of 1,000 t/d beginning the first year of the mine life. The study indicated that a resource of 325,021 t, grading of 1.76% Pb, 58.09 g/t Ag, 0.08% Cu, and 4.74% Zn is contained within the base case pit for the CNE deposit.

The optimizations for each of the three open pit deposits were evaluated based on a production rate of 500 t/d. Based on the optimization results the Taylor Brook deposit did not merit further evaluation and was not included in mine design considerations for the PEA. For the three deposits, two optimizations were completed based on the following options:

- an owner operated mill with a process operating cost of US$12.25/t processed
- a toll mill option with a process operating cost of US$18.90/t processed.

The results for the CNE deposit for optimization Option #1 are:

- Pit shell #31 generates the highest present value (PV) at $23.9 million for the specified case, based on a 500 t/d operation over a mine life of approximately two years. The selected base case pit shell contains 339,000 t of resource with average grades of:
  - 4.61% Zn
  - 1.72% Pb
  - 0.08% Cu
  - 57.37 g/t Au.

The results for the CNE deposit for optimization Option #2 are:

- Pit shell #30 generates the highest PV at $22 million for the specified case, based on a 500 t/d operation over a mine life of approximately 1.8 years. The selected base case pit shell contains 325,000 t of resource with average grades of:
  - 4.80% Zn
  - 1.79% Pb
  - 0.07% Cu
  - 59.72 g/t Au.

The initial mine design for the PEA is based on a combined Captain/CNE mine operation. The initial mine plan includes the mining of the CNE deposit first, followed by mining of the Captain deposit. The mines are independent open pit operations. Mine production rates of 500 t/d and 1,000 t/d were evaluated in the mine plan for this combined scenario. It was determined that the inclusion of the Captain deposit in the mine plan resulted in a negative impact on the cash flow analysis. Subsequently, only the CNE deposit was included in the mine plan for the economic analysis in the PEA. The mine plan utilizing a mine production rate of 1,000 t/d for the CNE deposit was carried forward for the remainder of the PEA evaluation.

The ultimate pit design for CNE and Captain deposits result in the following statistics (Table 1.10).

**Table 1.10 General Pit Statistics**

| Item | Size | |
|---|---|---|
| | CNE | Captain |
| Pit Top Elevation (approximate) | 171 m | 152 m |
| Pit Bottom Elevation | 100 m | 68 m |
| Pit Depth | 71 m | 84 m |
| Volume of Pit | 490,500 $m^3$ | 834,900 $m^3$ |
| Area of Pit Top | 30,480 $m^2$ | 36,110 $m^2$ |
| Perimeter at the Top of the Pit | 850 m | 770 m |

*table continues…*

| Item | Size | |
|---|---|---|
| | CNE | Captain |
| Length from East to West | 220 m | 215 m |
| Length from North to South | 175 m | 235 m |

The resource contained within the ultimate pit design for CNE and used in the economic analysis of the PEA is presented in Table 1.11.

**Table 1.11 Ultimate Pit Design Results - CNE**

| Item | Tonnes (t) | Zn% | Pb% | Cu% | Ag (g/t) |
|---|---|---|---|---|---|
| Measured Resource | 39,815 | 5.45 | 1.80 | 0.02 | 58.48 |
| Indicated Resource | 273,542 | 4.67 | 1.76 | 0.09 | 59.54 |
| **Total M+I Resource** | **313,357** | **4.77** | **1.76** | **0.08** | **59.40** |
| Inferred Resource | 11,664 | 3.79 | 1.77 | 0.05 | 22.71 |
| **Total Resource** | **325,021** | **4.74** | **1.76** | **0.08** | **58.09** |
| Waste Rock | 958,684 | - | - | - | - |
| Stripping Ratio | 2.95 | - | - | - | - |

Note: At cut-off grades of 0.6% Zn (in the Pb-Zn-Cu Zone), 0.35% Cu (in the Cu Zone), and 1.0% Zn (in the Pb-Zn Zone).

A mining resource recovery of 95% with an overall waste rock dilution of 5% was assumed.

Initially, production was scheduled to mine the CNE and Captain deposits consecutively. The final overall production schedule includes only the CNE deposit. A toll mill option was selected for the stand-alone CNE mine operation. The mine rate of 1,000 t/d was adopted and a production schedule of less than one year was developed for the CNE mine operation. Table 1.12 reflects the production schedule carried into the financial analysis.

**Table 1.12 Overall Ultimate Pit Production Schedule – CNE**

| Material | Units | Quarter | | | | Total |
|---|---|---|---|---|---|---|
| | | 1 | 2 | 3 | 4 | |
| Overbuden | tonnes | 243,829 | 78,928 | 2,533 | - | 325,290 |
| Waste | tonnes | 75,610 | 171,056 | 243,788 | 142,939 | 633,393 |
| Total Resource | tonnes | 32,987 | 102,441 | 106,105 | 83,488 | 325,021 |
| Lead | % | 2.25 | 2.06 | 1.73 | 1.24 | 1.76 |
| Silver | g/t | 60.12 | 60.31 | 55.32 | 58.07 | 58.09 |
| Copper | % | 0.06 | 0.07 | 0.06 | 0.14 | 0.08 |
| Zinc | % | 5.60 | 5.30 | 4.75 | 3.68 | 4.74 |
| Stripping Ratio | - | 9.68 | 2.44 | 2.32 | 1.71 | 2.95 |

The mining operations will be performed by a mining contractor who would be supplying the necessary equipment.

## 1.7 ENVIRONMENTAL

The CNE deposit is located near the boundary between Gloucester County and Northumberland County and is the only property included in the environmental assessment. At the current stage of planning, the project is expected to include an open pit mine and associated infrastructure on the site, and toll-milling of the ore at the Xstrata Brunswick 12 mill.

The high sulphide content identified in the deposit strongly suggests that waste rock produced from the property could be a potential source of acid mine drainage, as well as potential metal leaching (ML). Geochemical properties of the ore and waste rock have not been assessed to date.

A provincial mining lease has been maintained for the property and is still active. The project will be subject to the provincial environmental impact assessment (EIA) process and may also be subject to an environmental assessment under the *Canadian Environmental Assessment (CEA) Act*. A federal approval will be required if an approval from a federal agency, such as approval under the *Fisheries Act*, is identified. The environmental assessment and permitting process for a development in New Brunswick is managed by the Project Assessment Branch of the Department of the Environment (DENV).

New Brunswick's Environmental Impact Assessment Regulation falls under the *Clean Environment Act*. The filing of an EIA registration document will be required for the DENV to determine if any further EIA work is required prior to final environmental approval. The EIA registration document must describe the project, the existing environment potentially affected by the project, the anticipated environmental impacts, and any proposed mitigative measures that, if implemented, would lessen, eliminate, or avoid such impacts. Environmental approval can typically be accomplished for a project of this nature by the filing of an EIA registration document.

Some aquatic resources baseline data have been collected downstream of the property boundary, but there are no site-specific environmental baseline data. There is the potential to advance project environmental approvals without having a full-year of site-specific baseline data if an impact-avoidance approach is taken in project planning. This approach would include avoidance of the disturbance or destruction of fish habitat and of any raptor nests, locating all project facilities outside of wetlands, and a commitment to treating any discharges to water to a level that is consistent with the assimilative capacity of the receiving waters. A walk-away closure plan also will be required that ensures long-term secure closure of the site.

A mining and reclamation plan must be prepared and filed with the New Brunswick Department of Natural Resources (NBDNR) in accordance with Guide to the Development of a Mining and Reclamation Plan in New Brunswick (NBDNR 2005a). The mining and reclamation plans for the property must include an estimate for mine closure and reclamation costs and an anticipated schedule for these activities. Security may be required for the payment of cost with respect to protection, reclamation and rehabilitation of the environment as required by any provision of the *Mining Act* of New Brunswick, and these securities would be credited into the Mine Reclamation Fund.

Once the EIA process concludes and the Approval to Construct and Approval to Operate are issued, the remaining permits can be obtained. The permits and/or authorizations that may be required include an update to the mining lease that is in place for the CNE property under the *Mining Act*, and environmental permits such as development and building permits, approval to install storage for explosives, air quality and water treatment facilities.

The implementation of an effective community and Aboriginal engagement program is fundamental to the successful environmental permitting of mining projects. The purpose of this program is to ensure that all potentially affected persons, businesses, and communities have a full understanding of the project and an opportunity to share information with respect to concerns regarding potential effects, and so the proponent has an opportunity to explain how these concerns are addressed in the project design and operations. This program typically begins in the early stages of project planning and continues through the life of the project.

## 1.8 CAPITAL AND OPERATING COSTS ESTIMATES

The CNE deposit as a standalone mineable deposit is the only scenario presented in the capital and operating cost discussion and the financial analysis discussion within this PEA. The capital costs for the Captain deposit are not detailed in these sections as the deposit has not been included in further economic modeling.

The capital and operating costs are based on a contract mine operation and a toll mill processing operation. Capital is therefore not required for mine equipment or for the process plant. The capital costs for the CNE deposit are estimated at $6.9 million (2011 base year). A detailed summary of the direct and indirect capital costs is presented in Table 1.13. A 15% contingency has been applied to the direct capital costs. The indirect capital costs have been calculated as 1% of the direct capital costs and the Owner's cost has been calculated as 2% of the direct capital costs. Salvage value has been calculated as 10% of the direct capital costs for utilities, mobile equipment, infrastructure, and water treatment.

**Table 1.13 Direct and Indirect Capital Costs**

| Item | Amount ($) |
|---|---|
| **Direct Capital Costs** | |
| Site Development | 1,442,749 |
| Utilities | 2,084,900 |
| Mobile Equipment | 125,000 |
| Infrastructure | 200,000 |
| Water Treatment | 200,000 |
| Closure/Reclamation | 2,000,000 |
| Subtotal Direct Capital Costs | 6,052,649 |
| **Indirect Capital Costs** | |
| Indirect Costs | 60,726 |
| Owner's Costs | 121,453 |
| Contingency (15%) | 907,897 |
| Salvage | (260,990) |
| **Subtotal Indirect Capital Costs** | **828,487** |
| **Total Capital Costs** | **6,881,136** |

Due to a short life-of-mine (LOM), it is feasible and practical to use contract mining and a toll mill option for processing. The mining operating cost by quarter for the single year of operation is outlined in Table 1.14.

**Table 1.14 CNE Operating Mining Costs by Period**

| Item | Unit | Q1 | Q2 | Q3 | Q4 | Total |
|---|---|---|---|---|---|---|
| Total Operating Costs | $ | 1,072,769 | 2,217,172 | 2,377,387 | 1,728,575 | 7,395,902 |
| Total Resource Mined | $/t | 32.52 | 21.64 | 22.41 | 20.70 | 22.76 |
| Total Resource , Waste & Overburden Mined | $/t | 3.04 | 6.29 | 6.75 | 7.63 | 5.76 |

In the financial analysis used in this PEA, a surcharge for toll milling has been built into the process operating cost rather than the net smelter return (NSR) values used in the financial analysis. The process operating cost used in the financial analysis is $28.53. This represents an approximate 75 to 100% increase in operating cost for a typical 1,000 t/d mill with three concentrates. The financial analysis assumes this is adequate to represent the revenue generated from a toll mill operation.

## 1.9 ECONOMIC ANALYSIS

Table 1.15 details the metal prices used in the economic analysis for the CNE deposit PEA.

**Table 1.15 Metal Prices**

| Metal | Metal Price | Units |
|---|---|---|
| Zn | 1.22 | US$/lb |
| Cu | 3.62 | US$/lb |
| Pb | 1.10 | US$/lb |
| Ag | 22.74 | US$/oz |

The financial analysis is based on the open pit mine design and one-year production schedule for the CNE deposit as defined in Section 16 Mining Methods. Revenue contribution is calculated from the NSR for three concentrates. Silver contributes to the NSR value for the lead concentrate and the copper concentrate. The NSR values for these concentrates are presented in Table 1.16.

**Table 1.16 NSR Values**

| Concentrate | NSR Value (US$/dmt) |
|---|---|
| Zn | 965 |
| Cu | 2,030 |
| Pb | 1,763 |

In this analysis, the full value of the NSRs are carried into the financial analysis calculation, even though a toll mill operation is presented as the only viable option for the project. Typically, a toll mill contract will erode the revenue from the NSR paid to the supplier of ore to the mill. As a toll mill contract has not been established between Stratabound and an existing mill, a surcharge for toll milling has been built into the process operating cost rather than the NSR values in the financial analysis (see Sections 21 Capital and Operating Costs for details). The financial analysis is based on the assumption that this is adequate to represent the revenue generated from a toll mill operation.

For the one operating year the gross revenue is $37.6 million, the total operating costs, including toll mill surcharges in the process operating cost, is $16.8 million and total capital costs are $6.9 million. This results in a pre-tax cash flow of $14.0 million. The annual cash flow, when calculated quarterly with the application of various discount rates, is presented in Table 1.17.

**Table 1.17 Pre-tax Net Present Value (NPV) and Internal Rate of Return (IRR)**

| Item | Amount |
|---|---|
| Pre-tax & Pre-finance NPV @ 6% | $13,131,483 |
| Pre-tax & Pre-finance NPV @ 8% | $12,862,435 |
| Pre-tax & Pre-finance NPV @ 10% | $12,599,611 |
| Pre-tax & Pre-finance NPV @ 12% | $12,342,838 |
| Pre-tax & Pre-finance NPV @ 15% | $11,968,655 |
| Pre-tax & Pre-finance NPV @ 20% | $11,372,963 |
| Project IRR | 292% |

## 1.10 RECOMMENDATIONS

### *1.10.1 GEOLOGY*

#### CAPTAIN

Based on results of the resource estimation program completed by Mercator, the following recommendations are provided with respect to future exploration and resource delineation programs for the Captain deposit:

- The Captain deposit still remains open at depth and further assessment of the higher grade core of the deposit in this direction is warranted.
- Six drillholes totalling approximately 1,100 m should be completed to better define the deposit in specific areas.
- Selective borehole EM surveying should be undertaken at Captain.

#### CNE

Tetra Tech recommends the following recommendations for the CNE deposit:

- A detailed pit map to accurately delineate the volume which has been previously mined out should be completed.
- A re-assay program should be devised and executed to develop a more accurate gold database.

#### TAYLOR BROOK DEPOSIT

Tetra Tech recommends the following recommendations for the Taylor Brook deposit:

- A detailed review of the historical drill logs and drill core to standardize the lithology in the lithology database.
- A total of 24 drillholes are proposed for a total of 2,900 m of drilling. Eleven drillholes were located along the western edge of the deposit as there has been no drilling to determine the western extent of the massive sulphide zones. Eight drillholes are proposed along the southeast of the 1995 drillholes to ascertain continuity of geology and grade to an approximate depth of 150 m. Lastly, five drillholes are proposed between TBD95-7 and CM077-1 to determine the continuity of geology and grade to the east of the 1995 drilling. The estimated cost for this drill program is approximately $350,000.

### 1.10.2 Mining Operations and Geotechnical

Due to the strategy of contract mining employed in this study, an effort to determine the availability and cost for local contract mining is required to validate the cost assumptions used in this report.

Details on the wall slopes by rock types should be determined for the typical CNE orientations. A comprehensive hydrological study is required for the next level of study or for detail mine design and will be required to determine the dewatering requirements.

### 1.10.3 Mineral Processing

The key recommendations for mineral processing are:

- Engage potential mills in negotiation regarding contract terms. Currently in the economic analysis, a toll mill surcharge to the mine operator has been applied in the process operating costs.
- Metallurgical test work to determine the compatibility of the Stratabound ore to the selected host mill's respective ore will be required to confirm contract terms.

### 1.10.4 Environmental

Study requirements to be completed prior to the application of environmental permit:

- fish and fish habitat survey of all waterbodies directly involved in the project (i.e., mine site and any access road stream crossings)
- water quality survey of all waterbodies directly involved in the project (i.e., mine site and any access road stream crossings) and immediate downstream receiving water bodies
- wetland delineation throughout the footprint of disturbance

- acid rock drainage (ARD)/ML testing on all waste rock and ore rock types, including acid-base accounting and following humidity cell tests (minimum six month duration)
- groundwater quality/quantity study (potentially using existing or new exploration drillholes) to estimate quantity and quality of groundwater that will seep into the pit and require management
- public engagement sessions regarding the proposed project.

An estimated engineering cost for the recommended activities is $120,000. An estimated cost for required field and lab work is $80,000.

# 2.0 INTRODUCTION

Stratabound is a publicly listed and Canadian registered base metal exploration company that is actively exploring and developing base metal (lead, zinc, copper and cobalt) and gold deposits in New Brunswick and Québec.

The Captain, CNE and Taylor Brook properties are located in the Bathurst Mining District in Northumberland and Gloucester Counties, New Brunswick; approximately 45 km southwest of Bathurst. The two mineral claim blocks and mining lease for Captain and CNE are located at latitude 47°17’ N and longitude 65°53’ W, and the mineral claim block for Taylor Brook, 12 km northwest of CNE, is located at latitude 47°21’ N and longitude 66°00’ W.

The Properties are comprised of three mineral claim blocks and one mining lease and cover approximately 3,792 ha. The mineral rights to the properties are 100% held by Stratabound under prospector license number 13727.

## 2.1 TERMS OF REFERENCE

Stratabound has retained Tetra Tech to produce two NI 43-101 compliant resource estimates for the CNE and Taylor Brook deposits and PEA report for the Properties. Prior to Tetra Tech’s involvement with CNE and Taylor Brook, there have been no NI 43-101 compliant technical reports completed on these two properties. Mercator located in Dartmouth, Nova Scotia previously completed two NI 43-101 compliant technical reports and resource estimates on the Captain deposit with the effective dates of August 29, 2008 and December 9, 2010, respectively.

This technical report conforms to the standards set out in NI 43-101 Standards of Disclosure for Mineral Projects and is in compliance with Form 43-101F1. The resource estimates in this report conform to the CIM Mineral Resource and Mineral Reserve definitions referred to in NI 43-101 Standards of Disclosure for Mineral Projects.

The designated QPs for this report are:

- Daniel Coley, P.Eng., Senior Metallurgist with Tetra Tech
- Michael P. Cullen, P.Geo., Senior Geologist with Mercator
- Paul J. Daigle, P.Geo., Senior Geologist with Tetra Tech
- Daniel Gagnon, P.Eng., Senior Open Pit Engineer with Tetra Tech
- Mike McLaughlin, P.Eng., Project Manager with Tetra Tech

- Robert Morrison, P.Geo., Senior Resource Geologist with Tetra Tech.
- Doug Ramsey, R.P. Bio. (BC), Manager - Environmental Assessment, Permitting, and Natural Resources with Tetra Tech

Two site visits were conducted by Tetra Tech to the Properties. The first site visit was conducted by Paul Daigle between October 18 and 19, 2010. Mr. Daigle was accompanied during this site visit by Mr. John Duncan, Consulting Project Geologist for Stratabound, Mr. Kevin Vienneau, Consulting Mining Engineer for Stratabound, and Mr. Mike McLaughlin. The second site visit was conducted between November 29 and 30, 2010 by Mr. McLaughlin and Mr. Wenchang Ni, a former Senior Mining Engineer with Tetra Tech. They were accompanied by Mr. Vienneau from Stratabound.

Site visits to the Captain property were conducted by Mr. Cullen on August 6, 2008 and October 15, 2010. During the site visit on August 6, 2008, Mr. Cullen was accompanied by Mr. Matthew Harrington, Geologist, also with Mercator.

### 2.1.1 UNITS AND MEASURES

The unit and measures used in this report are metric and all units of cost are in Canadian dollars unless otherwise stated.

# 3.0 RELIANCE ON OTHER EXPERTS

Tetra Tech has relied upon others for information in this report. Tetra Tech has relied on information provided by Stratabound and the website of the NBDNR from their online database (website: New Brunswick-eClaims) for matters relating to property ownership, property titles, and environmental issues. Tetra Tech has not conducted an examination of land titles or mineral rights.

Information from third party sources is referenced in Section 21 References. Tetra Tech used information from these sources under the assumption that the information is accurate.

# 4.0 PROPERTY DESCRIPTION AND LOCATION

The Properties are defined by the mineral rights to 183 mineral claims and one mining lease, currently 100% held by Stratabound, and covers an area of approximately 3,792 ha.

## 4.1 LOCATION

The Properties are situated as shown in Figure 4.1 and Figure 4.2 below.

**Figure 4.1 Project Location Map**

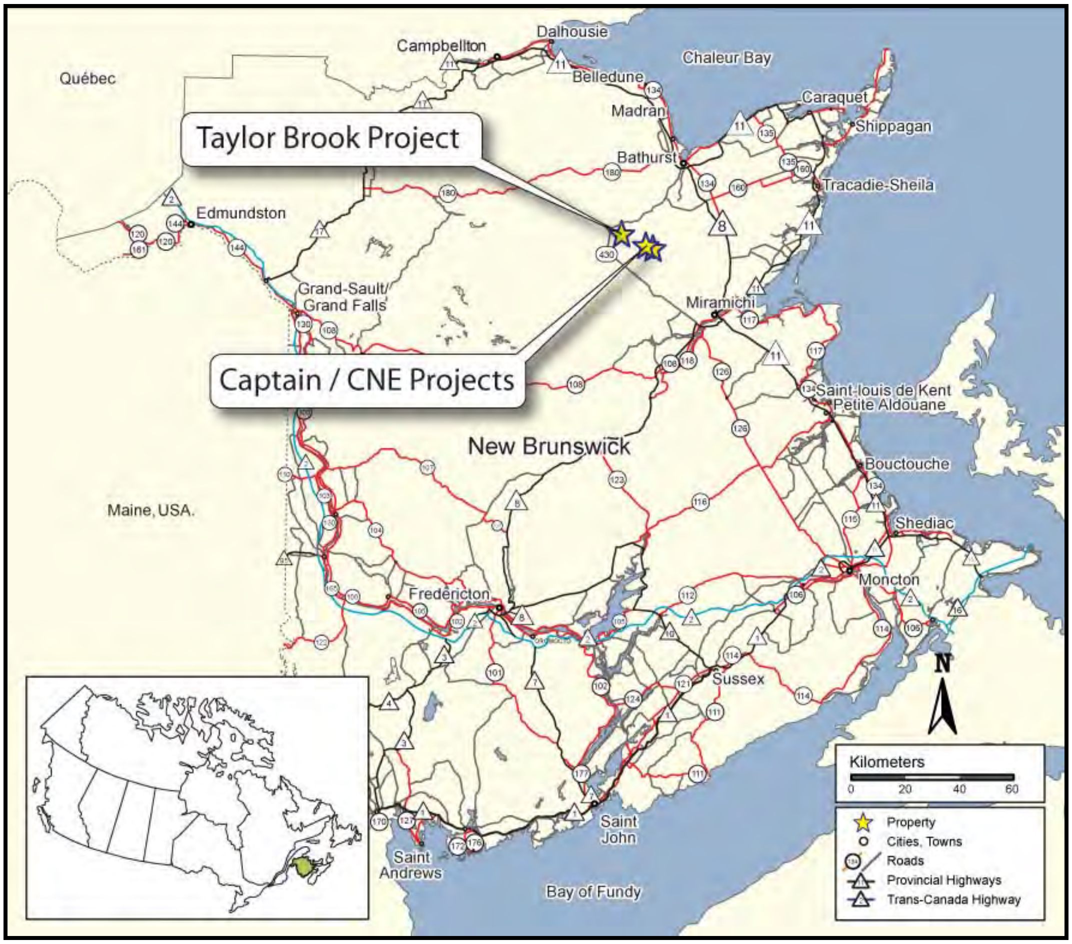
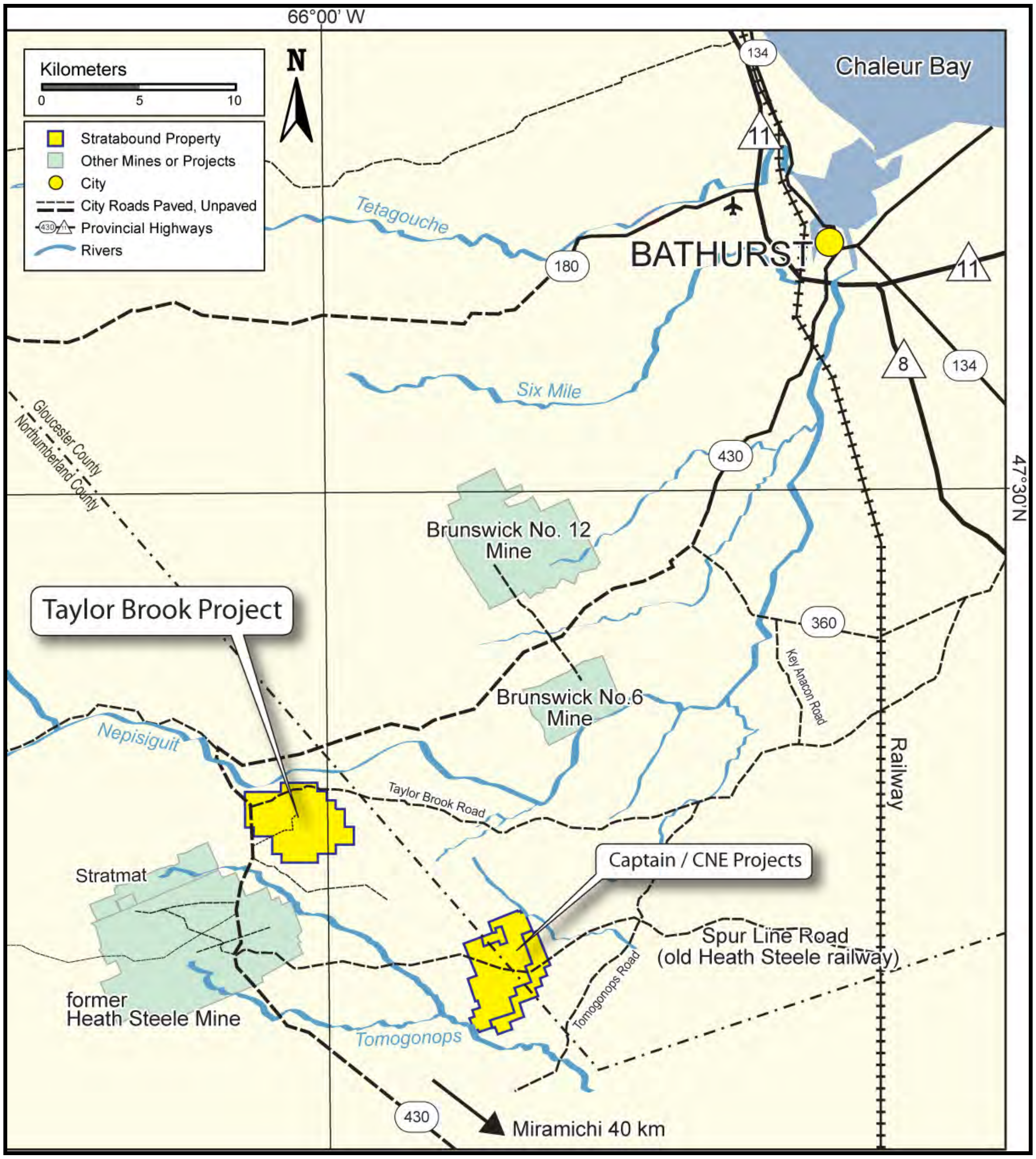


**Figure 4.2 Captain, CNE and Taylor Brook Property Location Map**




The Properties are located within National Topographic System (NTS) map sheets 21O/08 and 21P/05:

- at approximately 47°17’ N and 65°53’ W (Captain and CNE) and 47°21’ N and 66°00’ W (Taylor Brook) in northeast New Brunswick, in eastern Canada
- at approximately 170 km north of Fredericton, the provincial capital city of New Brunswick
- at approximately 40 km northwest from Miramichi

- at approximately 40 km southwest from Bathurst, New Brunswick
- on the border of the Counties of Northumberland and Gloucester
- in the Parishes of Bathurst and Northesk
- at approximately 19 km south (Captain and CNE) and 16 km southwest (Taylor Brook) of the operating Brunswick 12 Mine site
- at approximately 14 km east (Captain and CNE) and 8 km northeast (Taylor Brook) of the former Heath Steele Mine site
- at approximately 8 km south of the Nepisiguit River (Captain and CNE) and 3 km south of the Nepisiguit River (Taylor Brook).

## 4.2 PROPERTY DESCRIPTION

The Properties are comprised of the claims listed in Table 4.1 and illustrated in Figure 4.3 and Figure 4.4 below. Additional information on the mineral claims may be found in Appendix B.

**Table 4.1 Captain, CNE, and Taylor Brook Projects Mineral Claim Blocks**

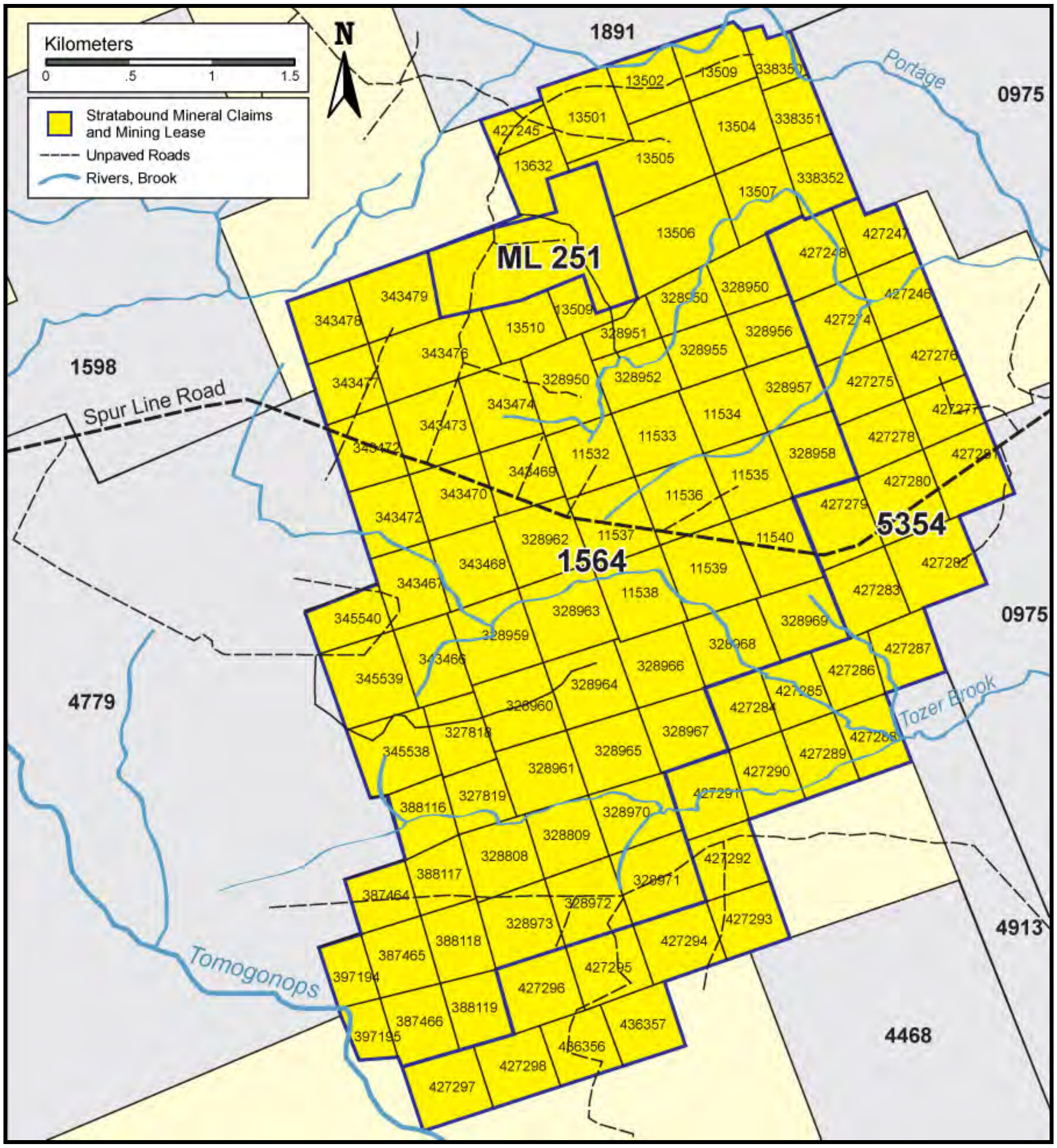
| Mineral Claim Name | Claim Number | Number of Claims | Area (ha) | Date Recorded | Date of Expiry |
|---|---|---|---|---|---|
| CNE Mining Lease | ML 251 | 1 | 65 | n/a | n/a |
| CNE Group | 1564 | 76 | 1,346 | Mar 3, 1987 | March 3, 2012 |
| Captain East Extension | 5354 | 30 | 481 | Feb 22, 2008 | February 22, 2012 |
| Taylor Brook | 1839 | 76 | 1,900 | Oct 27, 1984 | October, 27 2011 |
| **Total** | **-** | **183** | **3,792** | - | - |

Source: Government of New Brunswick, Ministry of Natural Resources

The mineral claims have been held in good standing by Stratabound within the past two decades. The surface rights to the Properties area are owned by the provincial government, known as Crown lands, and these have been licensed to Fornebu Lumber Company Inc. for timber.

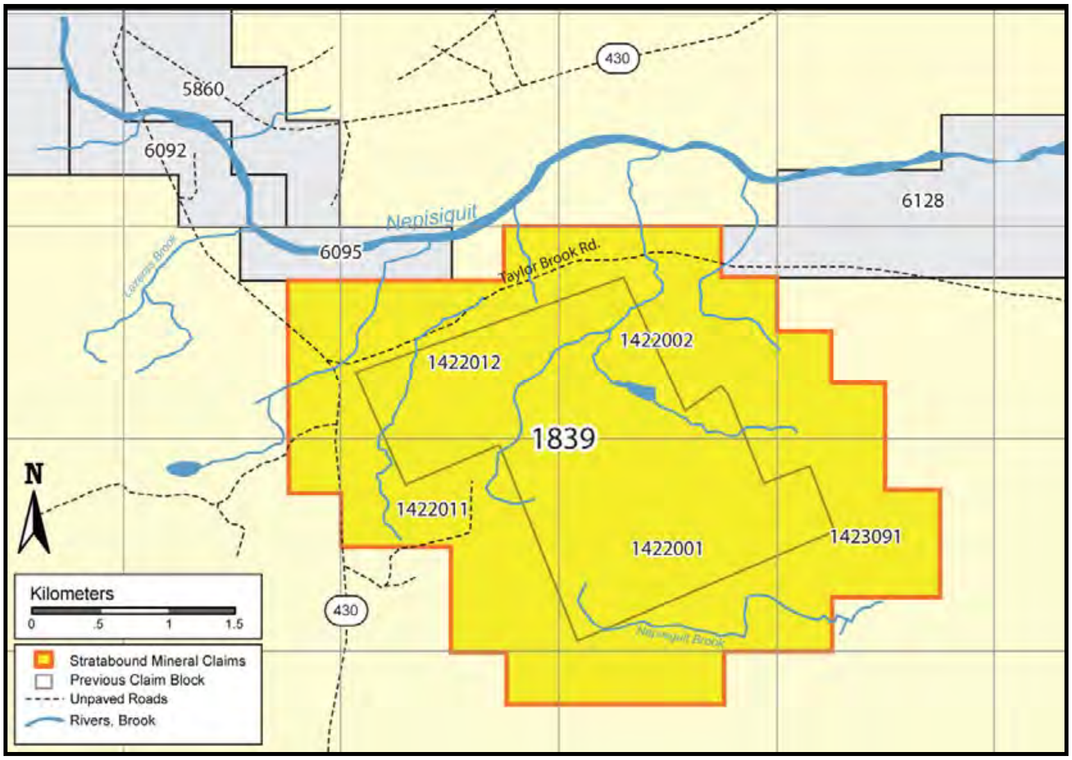
The mineralized zones of the Captain deposit occurs on the CNE Group mineral claims; the CNE deposit occurs on the CNE mining lease, and; the Taylor Brook deposit occurs on the Taylor Brook mineral claims. The Captain East Extension has been included here for completion as they are contiguous to the southeast to the CNE Group mineral claims. The Captain East Extension property is not subject to this report.

**Figure 4.3 Captain and CNE Property Mineral Claim Map**




In January 2010, the government of New Brunswick began to implement a new on-line map staking system (NB e-Claims) for the Province. This new claim acquisition system is based on a predefined New Brunswick grid system (Universal Transverse Mercator (UTM) coordinates). Historic mineral claims will require the conversion of existing ground staked claims to conform to the new format. In August 2011, Stratabound applied for the conversion of the Taylor Brook claim group. The previous Taylor Brook claim group (No. 1839) made up of 32 mineral claims (627 ha) was converted to the new format of mineral claims and is currently made up of 76 mineral claims covering 1,900 ha (Figure 4.4). These mineral claims are in good standing.

**Figure 4.4 Taylor Brook Property Mineral Claim Map**




It should be noted, that the conversion to new system has been voluntary to date. However, the Mining Recorder has asked the government of New Brunswick to end the voluntary conversion by January 1, 2012. After this time the Mining Recorder shall, within 90 days, convert all remaining ground staked mineral claims to map staked mineral claims and register them accordingly.

## 4.3 LIABILITIES

Tetra Tech is not aware of any environmental liabilities on the Properties.

## 4.4 PERMITS

Stratabound is not currently conducting any exploration activities on the Captain, CNE, and Taylor Brook deposits but expects to do so in the future. Permits will be required should Stratabound decide to establish exploration programs or operations on these properties.

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

## 5.1 ACCESSIBILITY

The Properties are situated 40 km southwest from the city of Bathurst, the business hub for northeastern New Brunswick. The Properties are most easily accessible from Bathurst or from Moncton, via Miramichi. There are, however, limited daily flights to Bathurst and more regular daily flights to Moncton.

From Bathurst airport, the Captain and CNE properties may be accessed by following:

- Highway 180 east for approximately 3 km to join
- Highway 11 south for approximately 7 km to join
- Highway 430 southwest for approximately 16 km to join
- Highway 360 southeast for approximately 6 km to join
- Key Anacon Road south for approximately 7 km to join
- Taylor Brook Road west for approximately 7 km to join
- Tomogonops Road south for approximately 6 km to join
- Spur Line Road (the old railway to the former Heath Steele Mine) for approximately 10 km to join
- a secondary road north for approximately 8 km to the Captain and CNE deposits.

From Bathurst airport, the Taylor Brook property may be accessed by following:

- Highway 180 east for approximately 3 km to join
- Highway 11 south for approximately 7 km to join
- Highway 430 southwest for approximately 45 km to continue on
- Highway 430 south for approximately 5.5 km to join
- an unmaintained dirt road for approximately 2 km to the Taylor Brook property boundary.

The drive from Bathurst to the Captain/CNE or Taylor Brook properties is roughly ¾ of an hour. The drive from Moncton to the Properties is approximately three hours.

## 5.2 CLIMATE

Northeast New Brunswick is situated within a northern temperate climate or a Warm Summer Continental (Dfb) climate on the Köppen climate classification system. This climate is generally characterized by cold winters and warm summers. Average winter temperatures vary between -17.2°C to -5.4°C with extreme minimums of -30°C to -38°C. Average summer temperatures vary between 12.9°C to 24.8°C (website: WorldClimate.com).

Average monthly precipitation (rain and snowfall) varies between 62 mm and 104 mm. Average yearly precipitation is 1,058 mm with an average yearly rainfall of 744 mm and average yearly snowfall of 314 mm (website: Environment Canada).

Conditions on the Properties during the spring breakup period may prevent some exploration activities from being carried out due to impassable access roads, high water levels in rivers and streams and unsafe ice conditions.

## 5.3 LOCAL RESOURCES

The city of Bathurst is the business and service hub for northeastern New Brunswick and is located approximately 50 km to the north and northeast of the Properties. Bathurst has a well developed mining infrastructure and a skilled workforce since the development of the large lead-zinc deposits (Brunswick 12 and Brunswick 6 mines) in the area in the mid-1950s.

Currently, the only operating mine and concentrator working at, or near, full capacity is Brunswick 12, owned and operated by Xstrata. This facility operates at a nominal rate of 10,000 t/d from current underground operations. Xstrata also owns Brunswick Mining and Smelting Limited, which operates a lead smelting facility at the port of Belledune located 35 km north of Bathurst. This smelter processes concentrate from the Brunswick 12 Mine.

Up until October 2008, Blue Note Mining Inc. operated a 3,000 t/d concentrator at their Caribou zinc mine, located 30 km north of the Properties.

## 5.4 INFRASTRUCTURE

The area has a well-developed network of paved highways and secondary roads. The Properties also have several unpaved logging roads that allow easy access to various areas on the Properties.

There is a railway that passes along the coast of New Brunswick with rail yards located in Bathurst and Belledune. Belledune also has a deep water port to permit the shipment of concentrate ore. Bathurst also has a regional airport with daily air service from Montreal.

Electricity for the region is produced from the electrical generating station located in Belledune, 35 km north of Bathurst. The Belledune station is coal powered and has a capacity of 458 MW. In 2009, the Caribou Wind Park was completed approximately 70 km west of Bathurst and roughly 60 km northwest of the Properties. This wind farm consists of thirty-three 3 MW wind turbines for an estimated capacity of 99 MW.

Water sources are abundant on and adjacent to all Properties.

## 5.5 PHYSIOGRAPHY

The Captain and CNE properties are situated on the eastern edge of the Miramichi highlands in an area of gently rising slope from east to west with relief varying from 100 to190 m. The Captain and CNE properties sit across the watershed that drains north to the Nepisiguit River and to the east and south to the Tomogonops River.

The Taylor Brook property is situated within the Miramichi highlands on a gently rolling plateau with elevations varying between 230 and 330 m. The Taylor Brook property boundary is approximately 1 km from the Nepisiguit River valley which descends to a low of 150 m.

The Properties are mostly covered by a layer of glacial till (up to 4 m thick) with limited rock outcroppings.

Vegetation throughout the Properties consists of boreal forest, populated mainly of spruce and balsam fir. Almost all of the Properties have been clear-cut in the past with areas of replanting and natural regrowth.

# 6.0 HISTORY

## 6.1 BATHURST MINING CAMP

The BMC is renowned for volcanogenic massive sulphide (VMS) copper-lead-zinc-silver deposits with initial discoveries made in the late 1800s. These discoveries included Keymet Mine (1880), Austin Brook (1898) and Brunswick 6 (1907) (Goodwin 1990). The advent of early geophysical techniques led to the discovery of the Brunswick 12 deposit in 1952 which established the significance of the BMC. Brunswick 12 has produced in excess of 120 Mt of ore (website: Xstrata 2011).

Since 1950, other significant deposits and mines in the BMC were Heath Steele, Wedge, Caribou, Key Anacon, and Chester. All of these deposits achieved short periods of production (Goodwin 1990).

## 6.2 CAPTAIN

The exploration of the Captain property began in the 1950s and has continued until the present time. Assessment reports are available to the public on the NBDNR website (website: NBDNR, Assessment Reports).

Table 6.1 summarizes the exploration activities conducted on the Captain deposit from past assessment reports and tabulations of historical work (e.g. Cullen and Harrington 2011).

**Table 6.1 Summary of Historical Exploration on the Captain Property**

| Year | Assessment File No. | Company | Work Performed |
|---|---|---|---|
| 1956 | 471123 | Captain Gold Mines Ltd. | • mapping, Vertical Loop Electromagnetic (VLEM) geophysical survey and 26 drillholes were completed (~5,578 m)<br>• discovery of the Captain deposit. |
| 1966 | 471123 | Captain Gold Mines Ltd. | • VLEM and soil geochemistry surveys were completed but no new targets identified |

*table continues…*

| Year | Assessment File No. | Company | Work Performed |
|---|---|---|---|
| 1966 | 471123 | Captain Gold Mines Ltd. | • fifteen additional drillholes were completed (~3277 m) and resource estimate reported |
| 1972 | 471126 | Captain Gold Mines Ltd. | • four additional drillholes (~462 m) plus Induced Polarization (IP) geophysical survey |
| 1977 | 472392 | Sabina Industries Ltd. | • airborne EM geophysical survey was flown and an anomaly detected over the Captain deposit as part of South Bathurst Joint Venture |
| 1979 | 472630 | Sabina Industries Ltd. | • Horizontal Loop Electromagnetic (HLEM) and VLEM geophysical surveys. Mapping and geological interpretation and three drillholes completed |
| 1982 | 472823 | Billiton Canada Ltd. | • stream sediment sampling |
| 1983 | 473076 | Brunswick Mining and Smelting Ltd. | • line cutting, VLF EM, ground magnetic geophysical surveys |
| 1984 | 473120 | B. Wilson | • VLF EM, ground magnetic surveys (south of the Captain deposit) |
| 1985 | 473203 | Brunswick Mining and Smelting Ltd. | • two drillholes (518.2 m) were completed to test the north and down-dip deposit extensions |
| 1986 | 473218 | Brunswick Mining and Smelting Ltd. | • soil sampling, Pulse Electromagnetic (PEM) geophysical survey |
| 1988 | 473515 | Stratabound | • data compilation, IP survey, line cutting. |
| 1989 | 473707 | Stratabound | • data compilation, gridding, and 12.4 km of IP survey and resource estimate reported |
| 1991 | 473999 | Stratabound | • grid establishment, VLF EM and IP surveys, four drillholes completed (621.1 m) |
| 1992 | 474198 | Teck Exploration Co. | • line cutting, magnetic and VLF geophysical surveys, geological mapping |
| 1994 | 474430 | Teck Exploration Co. | • one drillhole (261 m), downhole EM survey, soil sampling |
| 2004 | - | Eastmain Resources Ltd. | • two drillholes completed |
| 2008 | 476541 | Stratabound | • grid rehabilitation and 25 drillholes completed (5,098 m) |
| 2009 | 476720 | Stratabound | • grid establishment on the CNE group of claims, basal till sampling, and the completion of IP geophysical gradient survey |

### 6.2.1 HISTORICAL RESOURCE ESTIMATE

In 2008, Stratabound retained Mercator to carry out a NI 43-101 compliant resource estimate on the Captain deposit. Table 6.2 and Table 6.3 present the initial NI 43-101 compliant resource estimate taken from Cullen and Harrington (2009 and 2011) and presented here for completeness. The resource estimate for the Captain deposit, described in Section 14.1 Captain Resource Estimate, supersedes the following resource statements.

**Table 6.2 Historical NI 43-101 Compliant Measured and Indicated Resources for the Captain Cu-Co Deposit (compiled from Cullen and Harrington 2009 and 2011)**

| CuEQ% Cut-off | Resource Category | Tonnes ('000 t) | Cu% | Co% | Au (g/t) | CuEQ%* |
|---|---|---|---|---|---|---|
| **0.60** | **Measured (M)** | **53** | **1.14** | **0.061** | **0.21** | **1.70** |
| | **Indicated (I)** | **808** | **1.10** | **0.051** | **0.22** | **1.58** |
| | **M+I** | **861** | **1.10** | **0.052** | **0.22** | **-** |
| 0.80 | Measured (M) | 45 | 1.32 | 0.062 | 0.24 | 1.90 |
| | Indicated (I) | 660 | 1.30 | 0.051 | 0.25 | 1.77 |
| | M+I | - | - | - | - | - |
| 1.00 | Measured (M) | 38 | 1.50 | 0.061 | 0.26 | 2.06 |
| | Indicated (I) | 543 | 1.51 | 0.049 | 0.28 | 1.97 |
| | M+I | 581 | 1.51 | 0.050 | 0.28 | - |
| 1.20 | Measured (M) | 31 | 1.74 | 0.059 | 0.28 | 2.29 |
| | Indicated (I) | 466 | 1.67 | 0.048 | 0.30 | 2.11 |
| | M+I | - | - | - | - | - |
| 1.40 | Measured (M) | 25 | 1.99 | 0.060 | 0.31 | 2.54 |
| | Indicated (I) | 397 | 1.81 | 0.047 | 0.33 | 2.25 |
| | M+I | 422 | 1.82 | 0.048 | 0.33 | - |

*CuEQ % = Cu % + (Co % X 9.25). The 9.25 factor represents the relative price of cobalt compared to copper based on three year average metal pricing with no metal recovery factors applied.
' – ' indicates not available

**Table 6.3 Historical NI 43-101 Compliant Inferred Resources for the Captain Cu-Co Deposit (compiled from Cullen and Harrington, 2009 and 2011)**

| CuEQ % Cut-off | Resource Category | Tonnes ('000 t) | Cu% | Co% | Au (g/t) | CuEQ%* |
|---|---|---|---|---|---|---|
| **0.60** | Inferred | **681** | **0.60** | **0.039** | **0.12** | **0.96** |
| 0.80 | | 354 | 0.83 | 0.042 | 0.16 | 1.22 |
| 1.00 | | 192 | 1.14 | 0.040 | 0.21 | 1.51 |
| 1.20 | | 126 | 1.41 | 0.035 | 0.26 | 1.74 |
| 1.40 | | 94 | 1.57 | 0.034 | 0.29 | 1.89 |

*CuEQ % = Cu % + (Co % X 9.25). The 9.25 factor represents the relative price of cobalt compared to copper based on three year average metal pricing with no metal recovery factors applied.

RESOURCE ESTIMATE METHODOLOGY (CULLEN AND HARRINGTON 2009)

*The resource estimate was based on a three dimensional block model of the deposit developed by Mercator using Surpac© Version 6.1 deposit modeling software. Validated results for 25 diamond drillholes by Stratabound were used in the model and four higher grade mineralized zones were modeled separately, with grade interpolation in all cases being constrained within three dimensional solids reflecting cut-off parameters of 0.60% Cu or 0.05% Co over 4 m down hole sample composite lengths. A peripheral deposit constraint reflecting a 0.60% Cu Equivalent envelope was used to limit interpolation of Cu and Co grades outside the higher grade solids. Au values were interpolated in conjunction with Cu values. ID2 interpolation was used with a block size of 2 m x 2 m x 2 m and 1 m x 1 m x 1 m sub-blocking. Grade interpolation ellipse ranges were based on variogram analysis and specific gravity (SG) values were calculated from block grades using the linear regression formula SG = 2.85 + (0.1263 x CuEQ %).*

## 6.3 CNE

In 1977, the area of the CNE deposit was staked by McIntyre Mines Ltd. in response to a positive stream (silt) geochemical anomaly near the headwaters of two brooks north of the Captain deposit. Follow up soil geochemistry sampling delineated a significant anomaly in a swampy area drained by both brooks. Trenching at the borders of the swamp failed to identify the source of the anomaly (Frankland 1987; Arnold 1988)

In 1978, further basal till/soil geochemical sampling led to the discovery of the deposit by Metallgesellschaft Canada Ltd., with Sabina Industries Ltd. (Sabina) as an exploration partner. In 1980, the property was converted to a mining license and given to Sabina under an options agreement clause. The mining license subsequently lapsed in 1983, and Billiton Canada Ltd. (Billiton) took over the property.

The Billiton claims expired in 1986, and Noranda Ltd. (Noranda) optioned the property in 1987, drilling eleven holes (1,378.9 m) by the end of 1988. At the expiry of Noranda's options on the CNE property, the options were taken up by Stratabound.

**Table 6.4 Summary of Historical Exploration on the CNE Property**

| Year | Assessment File No. | Company | Work Performed |
|---|---|---|---|
| 1975 | 470427 | McIntyre Mines Inc. | • known as McDonough I Claims<br>• magnetic geophysical survey |
| 1976 | 470427 | McIntyre Mines Inc. | • HLEM geophysical survey |
| 1977 | | Metallgesellschaft / Sabina Industries Ltd. | • 35.3 km grid, geological mapping, nine trenches, detailed stream and swamp geochemical sampling, 403 soil samples, magnetic geophysical survey, MaxMin EM survey Horizontal shootback EM survey. |
| 1978 | | Metallgesellschaft / Sabina Industries Ltd. | • four hundred and twenty six soil and till samples 35 lithogeochemical samples, VLF EM survey, 69 pits and trenches (mapping and sampling), 17 auger holes, established detailed 21.9 km grid, detailed VLF EM and magnetic surveys, limited IP survey, 16 diamond drillholes (2,194.3 m) and structural analysis and ore microscopy |
| 1979 | | Metallgesellschaft / Sabina Industries Ltd. | • ten trenches (mapping and sampling), detailed IP survey, limited Mis-a-la-Masse survey, limited gravity survey, and two diamond drillholes (567.3 m). |
| 1980 | | Metallgesellschaft / Sabina Industries Ltd. | • CNE property convert to Mining Lease 555 and given to Sabina under the Captain Option Agreement perimeter clause |
| 1983 | - | - | • mining licence lapsed resulting in a staking rush by Billiton and Noranda<br>• CNE deposit located on Billiton mineral claim. |
| 1984 | 473061 | Billiton | • Line cutting<br>• Gravity survey (5.5 line km)<br>• Magnetic survey (51.5 line km)<br>• IP survey<br>• 1124 soil samples |
| 1985 | 473180 | Billiton | • soil geochemistry (1,124 samples) |

*table continues…*

| Year | Assessment File No. | Company | Work Performed |
|---|---|---|---|
| 1985 | 473181 | Billiton | • soil geochemistry (297 samples), 5.6 line km of line cutting, magnetic, VLF-EM, HEM geophysical surveys |
| 1987 | 473410 | Noranda | • Billiton claims expired. Restaked by Mr. N. Pitre. Noranda options the property from Mr. Pitre<br>• nine diamond drillholes completed (908.9 m) |
| 1988 | 473600 | Noranda | • two diamond drillholes completed (470.0 m)<br>• option agreement with Mr. Pitre discontinued<br>• Mr. Pitre options the property to Stratabound |
| 1989 | 430750 | Stratabound | • established 4.95 line km grid, five trenches (135 m), magnetic survey (4.95 line km), IP survey (2.3 ine km), 20 diamond drillholes (1,366 m.<br>• ore reserve calculations*<br>• two flotation tests, microscope analysis and density tests |
| 1990 | 473892 | Stratabound | • fourteen shallow BQ diamond drillholes (107 m); seven shallow NQ drillholes (35 m); one trench (40 m); preparation for bulk sample including road building (5 km); settling pond construction; water quality tests<br>• bench tests |
| 1990 | 473925 | Stratabound | • twenty six BQ diamond drillholes (1,966.7 m), resource estimate, flotation tests; lock cycle tests |
| 1990 | - | Stratabound | • Feasibility study (Goodwin, 1990) outlined 207,555 t of probable ore* |
| 1991 | 474044 | Stratabound | • 11,100 dry tonne bulk sample; geotechnical studies |
| 1991 | 474097 | Stratabound | • VLF and magnetic geophysical surveys; silt sample and rock sample survey; all undertaken to outline additional favourable zone associated with the CNE deposit |

*table continues…*

| Year | Assessment File No. | Company | Work Performed |
|---|---|---|---|
| 1992 - 1993 | 474381 | Stratabound (Teck option) | • CNE and Captain optioned to Teck<br>• three trenches (173 m); Mag and VLF (17.0 line km), MaxMin survey (14.1 line km) and gravity survey (21.2 line km); four diamond drillholes (1,025 m) |
| 1993 | 474430 | Stratabound (Teck option) | • three diamond drillholes (1,273 m) and bore hole transitent EM survey |
| 1995 | 474621 | Stratabound | • two BQ diamond drillholes (191 m) |
| 2009 | 476720 | Stratabound | • grid establishment on the CNE group of claims, basal till sampling, and the completion of IP geophysical gradient survey |

*'ore reserve calculations' are not NI 43-101 compliant

## 6.4 TAYLOR BROOK

Exploration work in the Taylor Brook property began in the mid 1950s, initiated by the discovery of the Brunswick 6 and 12 deposits.

**Table 6.5 Summary of Historical Exploration on Taylor Brook Property**

| Year | Assessment File No. | Company | Work Performed |
|---|---|---|---|
| 1956 | 470923 | American Metals Co. | • geophysical surveys |
| 1965 | 471431 | Mining Corp. of Canada | • geology and geophysical surveys |
| 1967 | 471681 | Mining Corp. of Canada | • geology and drilling |
| 1968 | 471682 | Mining Corp. of Canada | • trenching |
| 1977 | 471358 | Cominco | • geology and geophysical surveys |
| 1977 | 471363 | Cominco | • geology |
| 1977 | 472148 | Cominco | • geology and geophysical surveys |
| 1977 | 472233 | Cominco | • geology |
| 1977 | 472146 | J. Duffy | • geophysical surveys |
| 1978 | 472146 | Consolidated Morrison Ltd. | • geology, geochemistry and geophysical surveys |
| 1978 | 472357 | Consolidated Morrison Ltd. | • trenching and drilling |
| 1984 | 473096 | J.Duffy | • geology |
| 1985 | 473229 | J.Duffy | • geology |
| 1986 | 473316 | Granges | • drilling |

*table continues…*

| Year | Assessment File No. | Company | Work Performed |
|---|---|---|---|
| 1988 | 473514 | Stratabound | • HLEM survey, line cutting |
| 1989 | 473556 | Stratabound | • HLEM, IP surveys and trenching |
| 1995 | 473694 | Stratabound | • drilling |
| 1996 | 474341 | Stratabound | • drilling |
| 1997 | 474788 | Stratabound | • drilling |
| 1997 | 474984 | Stratabound | • drilling |
| 2004 | - | Stratabound | • airborne MegaTEM geophysical survey (part of regional survey conducted by Noranda/Falconbridge) |
| 2006 | 476390 | Stratabound | • grid re-establishment and line cutting, magnetic, MaxMin and VLF geophysical surveys, trenching and prospecting |

### 6.4.1 *Pre-Discovery, 1956 – 1968*

In 1956, American Metal Company conducted line cutting, geological mapping and magnetic and EM geophysical surveys over the area that hosts the Taylor Brook deposit. Later studies show that the deposit appears as a weak EM anomaly.

In 1965, Mining Corporation of Canada carried out line cutting, geological mapping, VLEM and magnetic geophysical surveys. The magnetic survey detected a 2 km linear feature correlating to a diabase dyke found on the Taylor Brook property.

In 1975, Cominco conducted line cutting, geological mapping, magnetic and IP geophysical surveys on their Stratmat property, 6 km to the southwest, where 1 km of magnetic survey covered a portion of the Taylor Brook property. The survey data showed no significant anomalies.

### 6.4.2 *Consolidated Morrison Ltd., 1977*

The Taylor Brook deposit was discovered in 1977 by Consolidated Morrison Ltd. (CML) in a follow up drill program to define a PEM geophysical anomaly. CML drilled seven diamond drillholes from 1977 to 1978.

### 6.4.3 *Granges, 1984*

In 1984, J. Duffy, of Bathurst, staked 13 claims over the Taylor Brook deposit. In 1985, J. Duncan attempted a downhole geophysical survey (Crone PEM) on the CML drillholes but was abandoned due to blocked drillholes.

In 1986, Granges drilled two diamond drillholes to test the Taylor Brook deposit at depth. Both drillholes returned significant results.

### 6.4.4 *STRATABOUND, 1987 – 2007*

In 1987 Stratabound optioned the Taylor Brook property from Messrs. Duffy and Jamieson and in 1988 Stratabound compiled all data over the Taylor Brook deposit.

Later in 1988, Stratabound carried out line cutting and a MaxMin geophysical survey. In 1989, Stratabound extended the MaxMin survey and conducted a limited IP geophysical survey. Five trenches were also excavated over the known deposit area.

In August 1991, Teck Corporation (Teck), now Teck Resources Ltd., optioned the property from Stratabound and drilled two diamond drillholes.

In 1995 to 1997, Stratabound completed 25 diamond drillholes on the Taylor Brook deposit for a total of 2,739.2 m. In total, 47 historic diamond drillholes have been completed on the Taylor Brook property with 35 of these over the Taylor Brook deposit. There has been no further drilling on the property since this drill program.

In 2004, Noranda Inc./Falconbridge Ltd. Bathurst Joint Venture (BJV), contracted Fugro Airborne Surveys to conduct a regional electro-magnetic airborne geophysical survey (MegaTEM) and a gravity survey. The airborne geophysical surveys consisted of over 32,000 line km. Stratabound, in agreement with the BJV, paid for 50% of the airborne survey over the Taylor Brook property, equalling 198.78 line km of data (Duncan 2007). The same report states that 32.706 line km were paid for covering the Taylor Brook deposit.

In 2006, Stratabound undertook several exploration activities over the Taylor Brook deposit that consisted of:

- establishing approximately 19 line km of the historic and new grid lines
- ground geophysical surveys consisting of 6 line km magnetic and VLF surveys and a 12 line km MaxMin survey
- a trenching program consisting of three trenches for a total of 345 m.

The three trenches (#4, #5, and #6), of approximately 115 m each, were excavated to define several geophysical anomalies defined by previous geophysical surveys. Trench #4, situated 400 m to the west of the Taylor Brook deposit, tested an IP anomaly and uncovered a zone of hydrothermal alteration with up to 5% pyrite. Trench #5 (Trench #6 on the Duncan 2007, map), is situated north-south across the centre of the Taylor Brook deposit, tested a magnetic high anomaly, but did not reveal the source of the anomaly. Trench #6 (Trench #5 on the Duncan 2007, map), situated 400 m southwest of Trench #5 over drillhole TBD95-16, tested an airborne gravity anomaly, however, did not expose any significant mineralization or outcrop (Duncan 2007).

# 7.0 GEOLOGICAL SETTING AND MINERALIZATION

## 7.1 REGIONAL GEOLOGY

Taken from Goodfellow (2007):

> *BMC deposits formed in a sediment-covered back-arc continental rift, referred to as the Tetagouche-Exploits backarc basin, during periods when the basin was stratified with a lower anoxic water-column. The basin was subsequently intensely deformed and metamorphosed during multiple collisional events related to east-dipping subduction of the basin.*
>
> *Four hydrothermal events spanning 12 to 14 million years have been recognized; the Chester (478 Ma), Caribou (472-470 Ma), Brunswick (469-468 Ma), and Stratmat (467-465 Ma) horizons. The Stratmat and Brunswick horizons both occur in the Tetagouche Group, whereas the Caribou and Chester horizons are hosted by the California Lake and Sheephouse Brook groups, respectively.*
>
> *…The Bathurst Mining Camp has been subdivided into four approximately coeval groups of volcanic and sedimentary rocks (Tetagouche, California Lake, Sheephouse Brook and Fournier groups)*

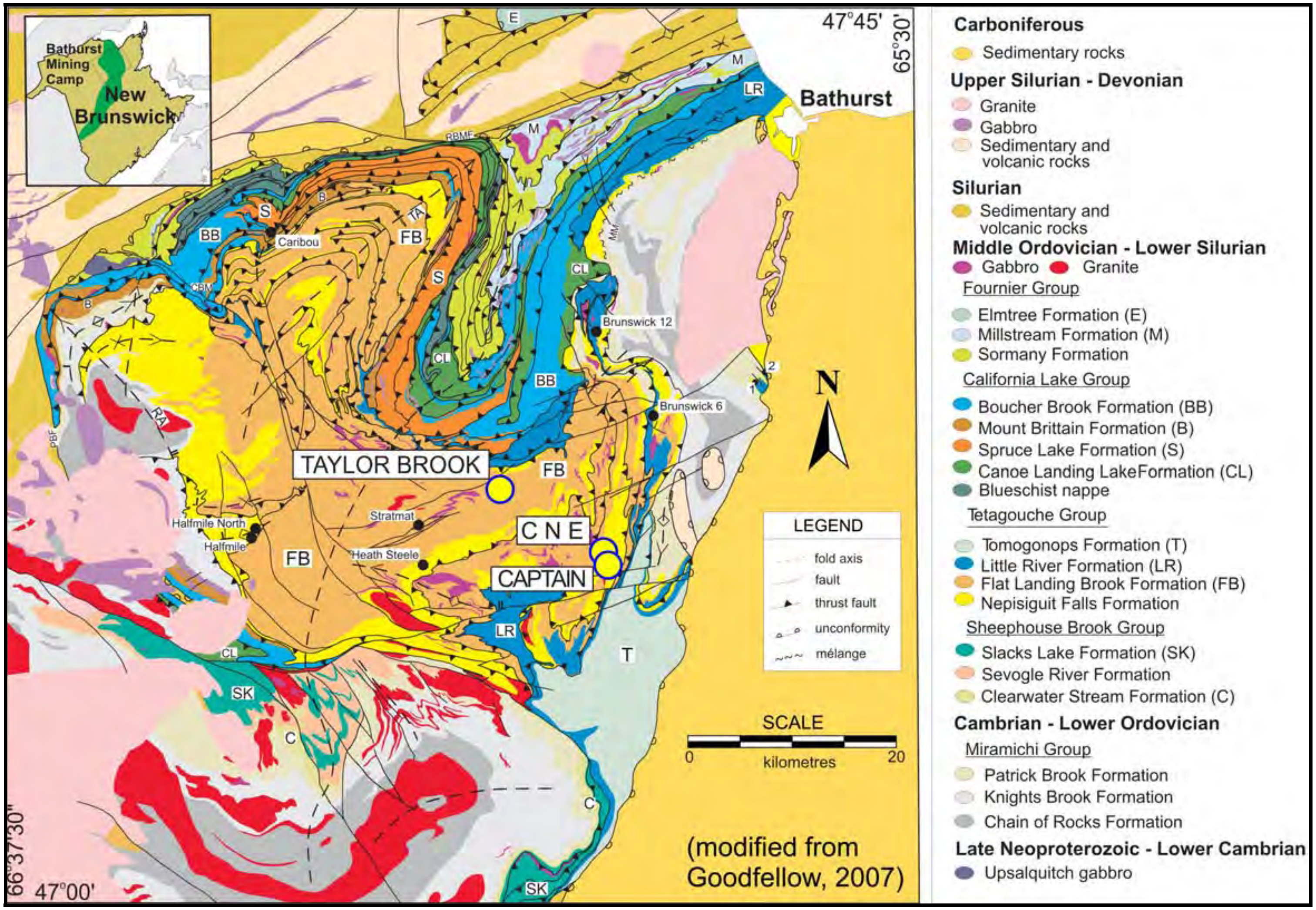
This portion of the report was taken from Davies et al 1985; van Staal and Langton 1990; and Daigle et.al. 2009.

The BMC is underlain by rocks of Ordovician age that are known as the Tetagouche Group and form part of the Miramichi Zone of northern New Brunswick. The Tetagouche Group is composed primarily of dacitic to rhyolitic volcanic rocks that have been subdivided into aphyric/feldspar-phyric rhyolites of the Flat Landing Brook Formation and quartz-feldspar porphyries of the Nepisiguit Falls Formation. These units are disconformably underlain by quartz-wackes and pelites of the Miramichi Group. Thin-bedded feldspathic wacke/shale and alkali basalts of the Boucher Brook Formation conformably overlie the felsic package. The Tetagouche Group rocks have been metamorphosed to the greenschist facies.

The Ordovician rocks of the BMC have undergone a complex history of polyphase folding and faulting. At least five deformational events are recognized based on overprinting relationships. $D_1$ is evidenced by a dominant, layering parallel foliation which is locally mylonitic and is interpreted to represent progressive deformation associated with thrusting. The $D_1$ structures are refolded by $F_2$ folds into tight structures defining flat and steep belts. These are refolded again by open recumbent $F_3$ folds. $F_4$ and $F_5$ folds and kinks at various scales overprint and refold the initial three phases. This deformation can be attributed to orogenic movements in the Appalachians during the Taconic and Acadian orogenies.

Regional geology of the BMC is illustrated in Figure 7.1.

**Figure 7.1 Regional Geology Map**



## 7.2 CNE AND CAPTAIN PROPERTY GEOLOGY

From Mercator (2008):

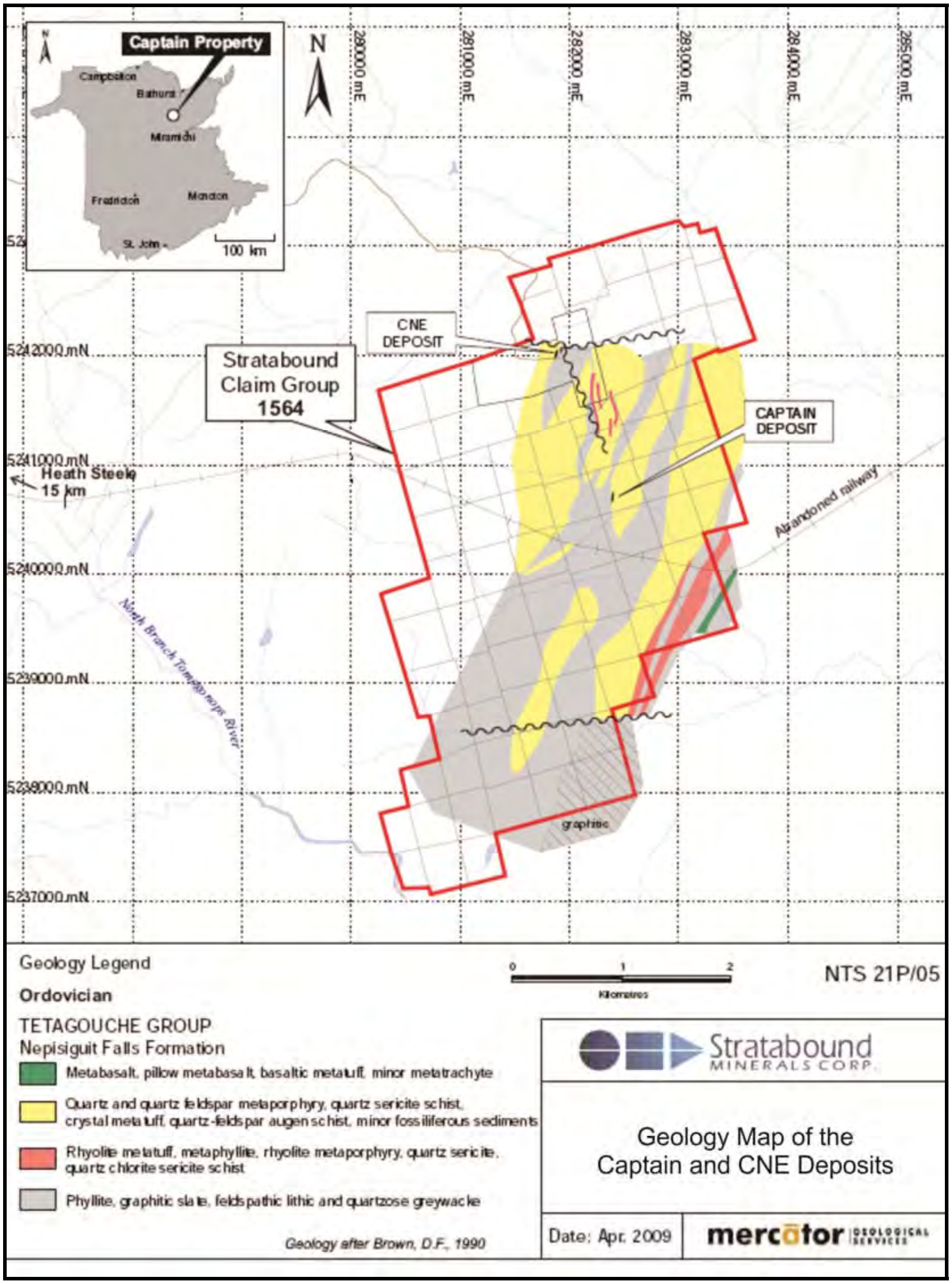
*Tupper et al. (1966) published a detailed description of the Captain deposit and property geology and since that time various other workers have reported on results of continued property investigations. Records of such appear in geological and geophysical reports accessible through the New Brunswick government's assessment reporting archive system. A detailed compilation of historic property exploration results was assembled by Brown (1991) and provides a useful interpretation of property geology that has not been significantly changed by results of work carried out since that time.*

Figure 7.2 *is based on the Brown (1990) interpretation and specifically reflects compiled results of property scale mapping, geophysical surveys and drilling. The original compilation plan from Brown (1988) incorporates a great deal more information than represented in Figure 7.2 and a copy of this document is included in Appendix E. Major lithologic contacts and structural grain are north-south trending in the immediate deposit area with bedding and dominant foliation dips being steep and to the west at 60 to 80 degrees. Cross-strike complexity of lithologic unit trends immediately north and west of the deposit suggest that a fold closure zone may be present in this area, the Captain deposit being located on the east limb of an indicated east verging synclinal fold. This is an important factor when considering both down-dip potential of the Captain deposit and its possible genetic relationship with the CNE massive sulphide deposit, located approximately 1,500 m to the northwest, potentially on the western limb of the same folded sequence. However, faulting that occurred after the major folding events complicated spatial relationships between the two deposits. The deposit occurs predominantly within Nepisiguit Falls Formation felsic augen schist unit, at or near its contact with underlying graphitic siliciclastic sedimentary rocks that occupy flanking positions within the folded zone (*Figure 7.2*).*

*At the deposit scale, transposition of volcanogenic sulphide "feeder stockwork" zones into near-parallelism with both related exhalative sulphide zones and the superimposed dominant regional foliation is seen in other BMC settings such as Heath Steel and Brunswick No. 12 (van Staal and van Staal 2003). In extreme cases, transposition of "feeder stockwork" zone mineralization may result in complete spatial separation from related exhalative massive sulphide mineralization.*

*Duncan (2008, personal communication) recognized this and suggested that the CNE deposit could represent part of a displaced exhalative facies of the Captain feeder stockwork system.*

**Figure 7.2 Captain and CNE Property Geology Map**




The following description is taken from Lemmon (2007):

> *The Tetagouche Group dominates the local geology in the CNE and Captain properties. This Group consists mainly of a voluminous suite of Middle*

*Ordovician mafic and felsic volcanic rocks. Felsic volcanic rocks dominate and have compositions that range from dacite to rhyolite (Whitehead and Goodfellow 1978; Winchester and van Staal 1988). The Tetagouche Group is locally divided into several separate formations including (from oldest to youngest) the Patrick Brook, Nepisiguit Falls, Flat landing Brook, Canoe Landing Lake and Boucher Brook Formations. These formations are discussed briefly below:*

*The Patrick Brook Formation consists of a dark grey to black quartzites, quartz wackes, and shales.*

*The Nepisiguit Falls Formation hosts the massive sulphide deposits at Heath Steele, Stratmat, (Taylor Brook), and Half Mile Lake and is divided into two members composed mainly of volcanic and mainly sedimentary rocks, respectively. The volcanic member consists primarily of quartz-feldspar crystal tuffs (popularly referred to as quartz-feldspar porphyry or quartz feldspar augen schist) that exhibit characteristics of both lavas and pyroclastic rocks, suggesting unusual circumstances of eruption and emplacement. The sedimentary member comprises green feldspathic or quartzose wackes, siltstones, shales, and minor epiclastic rocks.*

*The Flat Landing Brook Formation is composed aphyric or feldspar-phyric flows and domes, local felsic hyaloclastites and quartz-feldspar crystal tuffs, alkalic and tholeiitic mafic to intermediate intrusive and extrusive rocks, and minor sedimentary rocks. Where primary features are preserved, rhyolites typically exhibit perlitic fracture patterns and devitrification textures indicating that they were emplaced in a glassy state. Textural modifications resulting from alteration, shearing and dynamic metamorphism obscure and distort primary features and locally create apparent pyroclastic textures, especially in high-strain zones of actively moving flows. Flat Landing Brook rhyolites and crystal tuffs are chemically distinct from the Nepisiguit Falls Brook Formation and includes alkalic to tholeiitic extrusive and subvolcanic intrusive rocks that can generally be divided into separate suites based on geochemistry including:*

- *Tailings Lagoon tholeiitic gabbro and diabase*
- *Forty Mile Brook tholeiitic basalt; Otter Brook tholeiitic gabbros*
- *Tomogonops alkali gabbro*
- *Moody Brook andesite (comprising intermediate to mafic flows, tuffs and agglomerates).*

*Exposures of the Canoe Landing Lake Formation are very minor and consist of tholeiitic or alkalic pillow basalt with minor red shale or chert.*

*The Boucher Brook Formation consists of roughly equal amounts of dark grey to black, locally graphitic or manganiferous, locally magnetic, brick-red to dark grey shale and chert which occur at or near the base of the Boucher Brook Formation, or are intercalated with basalt.*

*Minor quantities of peralkaline felsic pyroclastic rocks locally occur near the base of the formation.*

*The felsic volcanic rocks comprise a heterogeneous mixture of flows, shallow intrusions (e.g. porphyries), pyroclastic and proximal epiclastic deposits (van Staal 1987). For mapping purposes, these rocks are generally divided into aphyric or feldspar-phyric rhyolite of the Flat Landing Brook Formation, and quartz and feldspar-phyric flow, pyroclastic and proximal epiclastic rocks of the Nepisiguit Falls formation). Field, geochemical and petrographic studies have indicated that a large proportion of the felsic volcanics previously interpreted as ash flows represent rhyolite flows (van Staal 1987, McCutcheon et.al. 1989). The large aerial extent of the rhyolite flows indicates that the felsic magma was relatively fluid, probably because it was dry and hot.*

*The felsic volcanic rocks are locally interbedded with tthin, but generally laterally extensive bodies of iron formation, jasper, and multicoloured (red, purple, green, and black) iron (Fe)/manganese (Mn)-rich phyllites. These metalliferous sediments are closely associated with most of the major base metal (Zn-Pb-Cu-Ag) massive sulphide depsoites in northern New Brunswick.*

*Locally felsic volcanics are also interbedded with minor bodies of tholeiitic basalt. Mineralization identified on the Caribou lake claim group to date includes sphalerite, chalcopyrite, and trace bornite with a matrix of rhyolitic tuff. It has been suggested that this mineralogy could indicate a proximal Stratmat type deposit.*

*Favourable Nepisiguit Falls Formation volcanic and sedimentary stratigraphy is present on the property and defines a north striking trend that reflects folding-related offset of the interpreted CNE stratigraphic interval, representing the locally interpreted Heath Steele – Brunswick No. 12 horizon (Lebel 1999).*

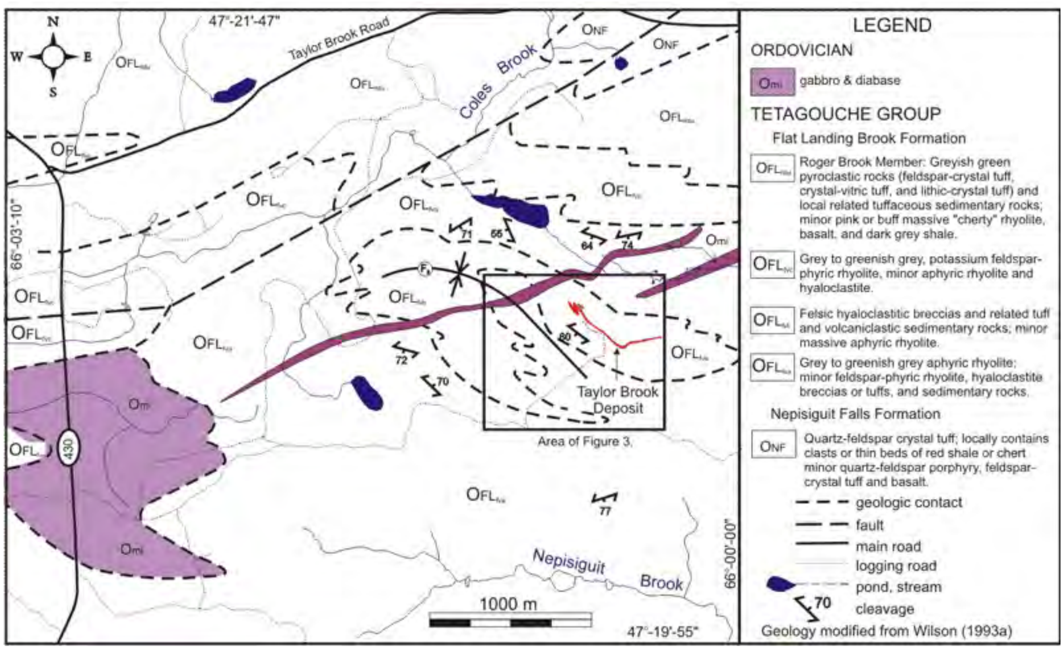
## 7.3 TAYLOR BROOK PROPERTY GEOLOGY

From Walker (1999):

*The Taylor Brook deposit is hosted within an intercalated sequence of felsic ash and lapilli tuff, aphyric to sparsely feldspar-phyric rhyolite flows, hyaloclastite, and minor sedimentary rocks of the Flat Landing Brook Formation. The deposit has a strike length of approximately 650 m and a down-dip extent of greater than 600 m. The surface trace of the deposit is tadpole shaped with the thickest accumulation of sulphides at the shallow (less than 50 m depths in the western part of the deposit.*

*The sulphide zone comprises one to four stratabound horizons of heavily disseminated to semi-massive and massive sulphides interlayered with hydrothermally altered volcanic rocks. The upper and lower contacts of* *individual sulphide horizons vary from diffuse to sharp. Sulphide minerals include: pyrite, pyrrhotite, sphalerite, galena, and chalcopyrite. Metal zonation, that is, zinc and lead-rich tops and copper enriched bases is locally developed at the scale of individual horizons or on the scale of total deposit thickness. The lead, zinc, and copper mineralization is consistent with the same ratios from other deposits in the Bathurst Mining Camp. A well-developed chalcopyrite-pyrrhotite stringer zone occurs below the stratabound sulphide horizon (north side of lens), and is particularly well developed at depth in the western part of the deposit. Hydrothermal alteration is developed in both footwall and hanging-wall rocks. Footwall alteration is denoted by development of moderate to locally abundant chlorite, whereas hanging-wall alteration is characterized by white micas or minor chlorite. The relative position of stratabound and stringer-type mineralization and variable footwall and hanging-wall alteration suggests the at the deposit dips to the south, is right-way up and is part of a proximal autochthonous system.*

**Figure 7.3 Taylor Brook Geology Map (Walker, 2007)**



# 8.0 DEPOSIT TYPES

The Tetagouche Group rocks host approximately 45 base metal-rich sulphide deposits and 95 mineral occurrences (Goodfellow 2007). These occurrences are typically closely associated with the felsic volcanic and epiclastic rocks of the Nepisiguit Falls and Flat Landing Brook Formations.

## 8.1 CAPTAIN AND CNE DEPOSITS

The Captain and CNE properties are host to two known delineated mineral deposits.

From Cullen and Harrington (2009):

*Sulphide deposit styles present in the BMC include (1) stratiform, laterally extensive and compositionally zoned bodies, (2) stratabound to stratiform, laterally restricted, mound-like bodies, often associated with vent complex stockwork sulphide zones, and (3) stratiform, poorly zoned to non-zoned sheet-like deposits that lack vent characteristics and reflect transport and reworking of previously deposited sulphides and host materials. The largest deposits such as Brunswick No. 12 and Heath Steele are associated with well developed siliciclastic sedimentary sections that accumulated after cessation of major volcanic episodes (Goodfellow 2007). The Brunswick No.12 and Heath Steele deposits are also marked by laterally extensive, zoned carbonate-oxide-sulphide iron formation units that extend substantially beyond deposit limits and form important stratigraphic marker intervals that are useful for exploration purposes.*

*The Captain* (and CNE) *base metal sulphide property is situated in a favourable and prospective stratigraphic interval of the Nepisiguit Falls Formation volcanics at the contact with overlying argillites and graphitic argillites. This position, along with documented presence of exhalative iron formation and base metal sulphide accumulations on the property, indicates close association of the Captain deposit host stratigraphy with the Brunswick No. 12 and Heath Steele stratigraphic intervals, thereby defining good potential for occurrence of base metal sulphide zones of similar affinity.*

The CNE deposit occurs with both a copper-rich zone and a lead-zinc zone that appear to be emplaced separately but cross-cutting each other.

## 8.2 TAYLOR BROOK DEPOSIT

The Taylor Brook property is situated approximately 6 km, along strike and northeast of the Stratmat deposits and is host to the Taylor Brook polymetallic (VMS deposit).

The western portion dips evenly to the southwest at approximately 45° and has been found to contain localized lead-zinc mineralization. Sphalerite and galena are the dominant economic minerals and the width and grades of the base metal mineralization are highly variable within the sulphide zone. Stratabound has focused its historic exploration on the western portion of the Taylor Brook deposit and has explored to depths up to 500 m. The pyritic tuffaceous rocks which host the Taylor Brook deposit appear to continue to the north-northwest of an east-west trending mafic dyke. The mapped distribution of the rock units from the 1996-1997 mapping programmes may indicate a broad F2-F3 fold axis in the Coles Brook area. This would suggest a stratigraphic correlation between the Taylor Brook sulphide zone and areas of weak pyritic mineralization of the mafic dyke (Duncan and Vienneau 2007).

The eastern portion of the Taylor Brook deposit has been drilled to shallow depths and has not been delineated to the east.

# 9.0 EXPLORATION

## 9.1 CAPTAIN, 2007-2008

The following sections are taken from Cullen and Harrington (2011).

### 9.1.1 2007-2008 DRILLING PROGRAMS

*In September of 2007 Stratabound initiated a detailed core drilling program on the Captain deposit to support a planned mineral resource estimate. The drilling program continued through May of 2008 and consisted of 25 drillholes totalling 5,098 meters of drilling. Stratabound established surveyed drill setups along lines spaced at 25 metre intervals along the length of the deposit and vertical holes testing the mineralization were completed from these setups. This constituted general re-drilling of the known deposit and provided good quality data for resource estimation work. Metals of interest included Cu, Zn, Pb, Au, Ag, Co and bismuth (Bi). The company also recognized that much of the historic core was no longer available for sampling, and that locating some historic collar locations was problematic, and that improved core recoveries expected from modern drilling procedures would be advantageous to deposit definition and evaluation studies.*

*Results of the 2007-2008 core drilling programs formed the basis of a NI 43-101 compliant mineral resource estimate (Cullen and Harrington 2008) prepared by Mercator for Stratabound in 2008 and details of these programs by Stratabound are presented in report section 10. Details of the 2008 resource estimate appear in report Section 14.1.*

### 9.1.2 OTHER 2008 PROGRAMS

*In addition to the core drilling program noted above, Stratabound also carried out a limited program of deep overburden sampling along selected transects across the Captain property. This program included surface pitting with a small excavator to recover base of overburden samples or recovery of such samples in areas of deeper overburden through use of a power auger attachment for the excavator. Anomalous results were locally returned from this work but are not considered material to the updated resource estimation program.*

### 9.1.3 DRILLING, 2010

*During 2010, Stratabound completed five additional diamond drill holes that were targeted to assess down plunge and along strike extensions of the mineralized* *zone. The most significant result was that hole CP10-26 extended the down plunge extension of core zone higher grade Cu mineralization. Other holes intersected lesser grades. Details of this program are presented in report section 10. Additionally, McCutcheon (2010) reported on a geological re-interpretation study of the property carried out on behalf of Stratabound and proposed that the Captain stockwork zone may be related to a currently undiscovered polymetallic massive sulphide deposit located near the overturned contact between Nepisiquit Falls Formation and Flat Landing Brook Formation strata elsewhere in the property*.

## 9.2 CNE, 2007-2009

Exploration work from 2007 to 2009 consisted of establishing a reference grid, an IP geophysical survey and a basal till sampling program.

### *9.2.1 Geophysics*

Exploration work conducted over the CNE deposit consisted of establishing a northwesterly grid over the area and basal till sampling.

In 2008-2009, a geophysical survey consisting of an IP gradient survey, was completed over the CNE deposit an area to the south of the deposit. The IP survey found that the CNE deposit correlated well with a chargeability high and resistivity low. This anomaly trends south 600 m and is open to the south. This south trending anomaly appears to be smaller with a lower intensity than the anomaly over the Captain deposit (Duncan 2009).

It was also noted in the IP geophysical survey that a weak anomaly was evident, approximately 1,300 m east of the CNE deposit. The anomaly trends over 1,200 m in a north-south direction and reflects similar IP response as the CNE anomaly. This anomaly is open to the south and appears not to have been drilled in the past (Duncan 2009).

Several satellite anomalies were noted to the north, east and south of the CNE deposit which may require some follow up exploration in the future.

### *9.2.2 Basal Till Sampling*

Basal till sampling program in the CNE area consisted of 391 samples.

Sampling within the mined pit area returned values of 350 ppm Zn, 0.68% Pb, 184 ppm Cu, 8 ppm Co and 8 ppb Au. Given the location where these samples were taken, the results, with the exception of lead, were not as expected. These low values indicate that the B and C soil horizons are poorly developed and/or were not present due to glacial till outwash or clay capping the bedrock (Duncan 2009).

## 9.3 Taylor Brook, 2006-2007

In 2006, Stratabound conducted several small scale exploration programs on the Taylor Brook property, the first of such programs since the last phase of exploration diamond drilling in 1996. Exploration included line cutting of a reference grid, ground geophysical surveys and trenching.

Stratabound began by re-establishing a reference grid over the Taylor Brook deposit of approximately 19 line km of line cutting followed by two ground geophysical surveys that consisted of 6 line km magnetic and VLF surveys and a 12 line km MaxMin survey.

In 2007, Stratabound undertook a trenching program over the Taylor Brook deposit area to determine the source of several recent and historical geophysical anomalies. The program consisted of 342 m in three trenches where several chip samples were collected but were not sent away for analysis.

The three trenches (#4, #5, #6) of approximately 115 m each, were excavated to define several geophysical anomalies defined by previous geophysical surveys. Trench #4, situated 400 m to the west of the Taylor Brook deposit, tested an IP anomaly and uncovered a zone of hydrothermal alteration with up to 5% pyrite. Trench #5 (Trench #6 on the Duncan, 2007, map), is situated north-south across the centre of the Taylor Brook deposit, tested a magnetic high anomaly, but did not reveal the source of the anomaly. Trench #6 (Trench #5 on the Duncan, 2007, map), situated 400 m southwest of Trench #5 over drill hole TBD95-16, tested an airborne gravity anomaly, however, did not expose any significant mineralization or outcrop (Duncan 2007).

There has been no further exploration conducted on the Taylor Brook property since 2007.

# 10.0 DRILLING

## 10.1 CAPTAIN

### 10.1.1 DRILLING PROGRAM, 2007-2008

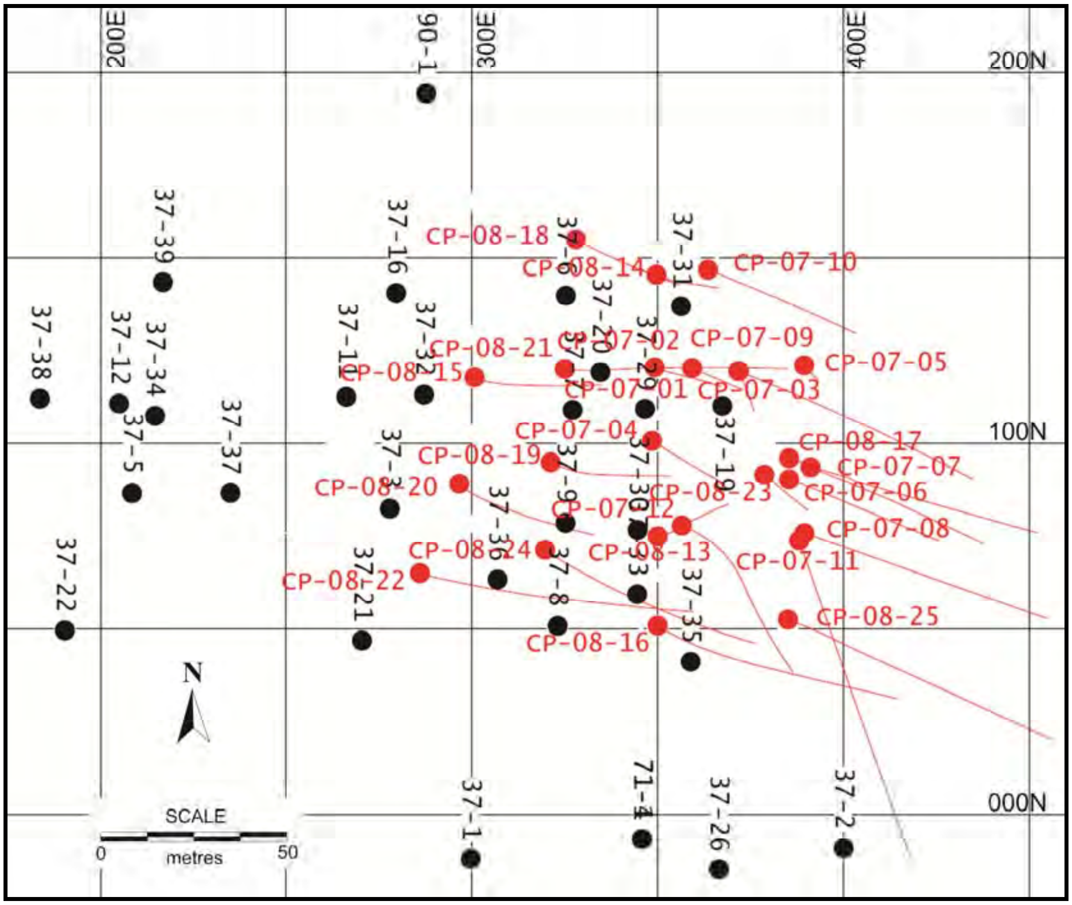
In September 2007, Stratabound conducted a diamond core drilling program on the Captain deposit to support a planned mineral resource estimate. The drilling program continued through May 2008 and consisted of 25 drillholes totalling 5,098 m of drilling. Stratabound established surveyed drill setups along lines spaced at 30 m intervals along the length of the deposit and vertical holes testing the mineralization were completed from these setups.

This drill program constituted a re-drilling of the known deposit and provided good quality and a verifiable dataset for resource estimation. Stratabound conducted the re-drilling of the Captain deposit because:

- much of the historic core was no longer available for sampling
- the location of some historic collar locations was problematic
- that improved core recoveries expected from modern drilling procedures would be advantageous to deposit definition and evaluation studies (Cullen 2008).

Results from this drill program formed the basis of the resource estimate completed by Mercator in October 2008.

**Figure 10.1 2007 – 2008 Drill Location Map (modified from Cullen and Harrington 2009)**



### 10.1.2 DRILLING PROGRAM, 2007-2010

The following is an excerpt taken from Cullen and Harrington (2011):

*Diamond drilling data from the Captain property used in the current mineral resource estimate consisted of 30 drillholes completed by Stratabound between September 2007 and July 2010. These total 7,308 meters of drilling and are supported by surveyed collar locations, down-hole orientation surveys, full analytical datasets and accessible core archives. Sample records and logs for numerous historic drillholes completed in the same deposit area were also reviewed but not used in the resource estimate due to (1) lack of comparably accurate collar coordinates, (2) lack of historic core sample analytical documentation, (3) non-availability of historic core for check sampling purposes and (4) presence of complete records, laboratory reports and core archives for the new 2007-2010 drilling dataset developed for the same deposit area. The recent Stratabound drilling programs were deemed sufficiently complete in regard to hole spacing and deposit coverage to meet resource estimation requirements.*

*Drilling program details are presented below under separate headings. In each case, associated information such as lithologic and sampling logs, assay results, laboratory reports, collar survey data and down hole survey data was assembled from digital and/or hard copy records and reports made available by Stratabound. Some digital compilation of historic drilling data had been completed by Stratabound staff prior to initiation of the resource report and this was also made available.*

### *10.1.3 LOGISTICS OF STRATABOUND 2007-2008 AND 2010 DRILLING PROGRAMS*

*Maritime Diamond Drilling Limited of Truro Nova Scotia provided contract drilling services for the 2007-2008 and 2010 drilling programs that recovered NQ size drill core measuring approximately 47.6 mm in diameter. Stratabound staff supervised on-site geological work under direct supervision of Mr. John Duncan, P. Geo., and also carried out core logging, sampling, interpretive and reporting functions. Conventional wire-line drilling equipment was utilized and the program was coordinated from the company's Bathurst exploration office. Drill core from the program was securely archived in racks at the company's Bathurst facility after sampling and logging.*

*Collar locations and elevations for all holes were established through instrument surveying to a local grid and elevation datum, with these subsequently converted to Universal Transverse Mercator (UTM) Northern Hemisphere, Zone 20, NAD 83 coordination. Compilation plans for past drillholes prepared by Stratabound were used to establish the spatial context of historic and recent holes for digitizing. Topographic relief in the deposit drilling area is limited to a few metres.*

*At Stratabound's request, local grid coordination was retained by Mercator for resource estimation purposes. This required standardized conversion of original west and south grid increments to (-) east and (-) north increments. All holes by Stratabound were tested for inclination and azimuthal variation using an electronic down-hole survey instrument. In contrast, acid test hole inclination results support some historic holes but in other instances only the initial inclination value set at the drill head is available.*

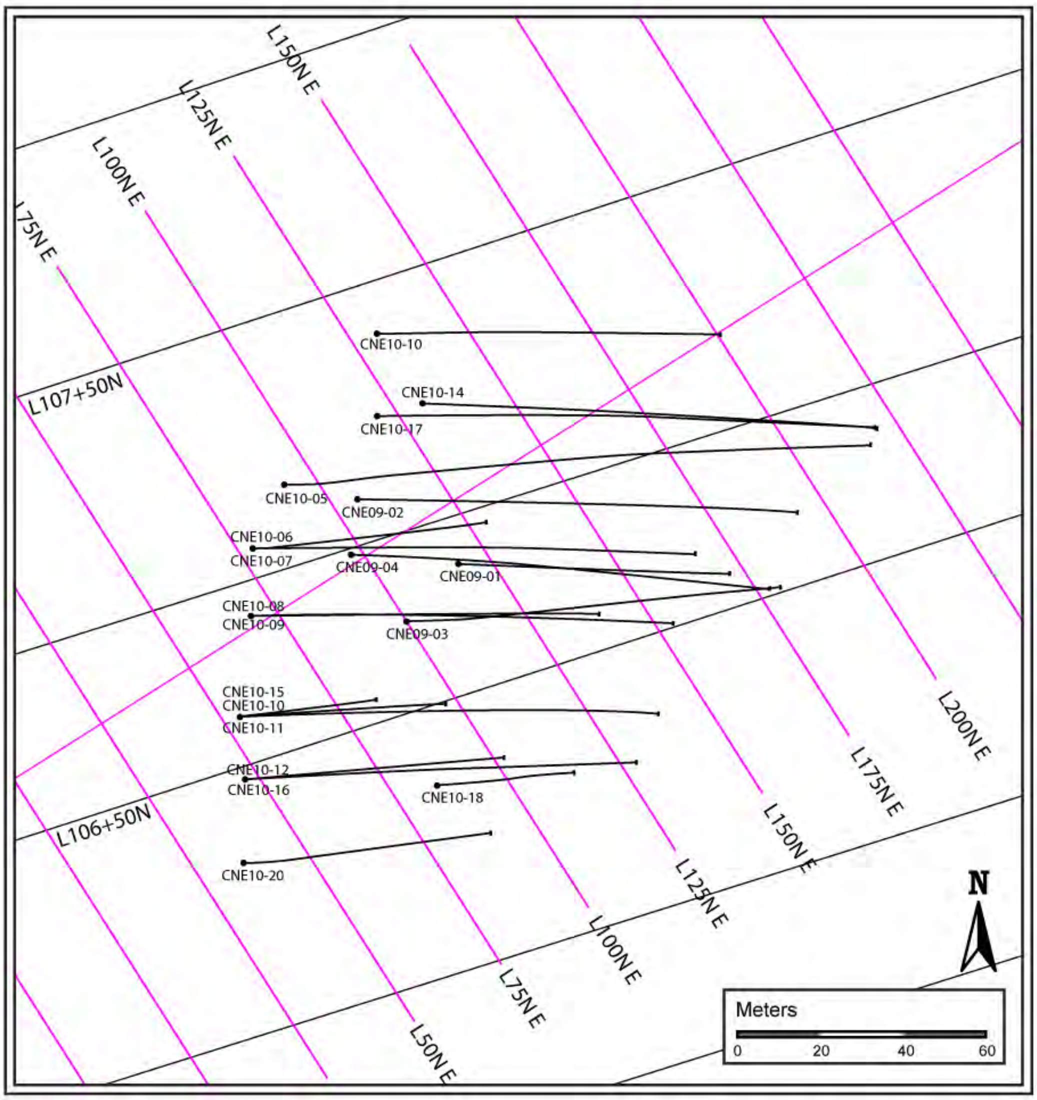
*After validation against project reporting records, all drilling coordination and down-hole survey data were compiled for respective programs in Microsoft Excel spreadsheets and then incorporated in a Microsoft Access database to support Mercator's resource estimation program. A tabulation of all compiled drillholes in the Captain deposit area, including collar coordinates plus azimuth, inclination and depth values, is included in Appendix D. Coordination values for both local and UTM grid systems are included in that tabulation and Map 2010-1 (Appendix D) presents drillhole locations.*

## 10.2 CNE, 2009-2010

In 2009 and 2010, Stratabound completed a 2,889.5 m diamond drill program from 20 diamond drillholes over the CNE deposit.

Based on a reinterpretation of the historical drill data, the 2009-2010 drill sites were oriented in an east/west direction. Figure 10.2 shows the historical and more recent drillholes in plan-view.

**Figure 10.2 CNE Deposit, 2009-2010 Drillhole Locations (Drillhole 10-13 not Marked; Located Approximately 200 m West)**



## 10.3 TAYLOR BROOK

There has been no drill programs conducted on the Taylor Brook property by Stratabound since 1996.

# 11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

## 11.1 CAPTAIN SAMPLE PREPARATION, ANALYSES AND SECURITY

This section has been taken from Cullen (2010).

### 11.1.1 STRATABOUND QA/QC PROGRAMS

#### INTRODUCTION

*Drill core sampling carried out by Stratabound during the 2007-2008 and 2010 Captain programs was subject to a Quality Control and Quality Assurance program administered by the company. This included submission of blind blank samples, duplicate split samples of quarter core, duplicate pulp splits, certified analytical standards and analysis of check samples at a third party commercial laboratory. Additionally, internal laboratory reporting of quality control and assurance sampling was monitored by Stratabound staff on an on-going basis during the course of the project. Results of the 2007-2008 Quality Control and Quality Assurance programs were discussed by Cullen and Harrington (2009) and were found to be acceptable for purposes of the 2009 Captain deposit resource. The authors accept this conclusion as remaining valid with respect to the updated Captain resource estimate program described in this report. On this basis, the following discussion of Quality Control and Quality Assurance program components is restricted to the new datasets that pertain to the five new drill holes completed in 2010 by Stratabound. A copy of the Quality Control and Quality Assurance report section from Cullen and Harrington (2009), which deals specifically with the 2007-2008 datasets, appears in Appendix D.*

#### CERTIFIED REFERENCE STANDARD PROGRAMS

*Stratabound used four certified reference standards during the course of the 2007-2008 drilling programs (CDN-HZ-2, CDN-HLLC, CDN-CGS-10 and CDN-WMS-1a), all of which were obtained from CDN Laboratories of Vancouver, BC. The 2010 program used 6 standards, these being CDN-HZ-2, CDN-GS-2B, NI-114, NI-116, OREAS-18-Pb and OREAS-15-Pb. As noted above, results for the 2007-2008 program are discussed in Cullen and Harrington (2009) and are considered acceptable for resource estimate use. Details of the 2009 program appear in Appendix D and 2010 program results are discussed below. Logistics of the programs are comparable.*

*In total, 26 certified samples were submitted for analysis during the 2010 program. Each sample consisted of a pre-packaged, prepared sample pulp weighing approximately 50 grams that was systematically inserted into the laboratory sample shipment sequence by Stratabound staff. Records of certified standard insertion were maintained as part of the core sampling and logging protocols and samples were submitted at a nominal frequency of one for every 35 samples submitted.* Table 11.1 *presents certified mean ± 2 standard deviation ranges for the various standards used in 2010 and also shows that not all metals of interest are covered by any one certified sample. These control limits were applied for report purposes.*

**Table 11.1 Certified Standard Tabulation for 2010 Drilling Program**

| *Standard* | **Accepted %Cu* | **Accepted g/t Au* | **Accepted %Co* | *Number Used* |
|---|---|---|---|---|
| CDN-HZ-2 | 1.36% ± 0.06% | 0.124 g/t ± 0.024 g/t | NA | 4 |
| CDN-GS-2B | NA | 2.03 g/t ± 0.12 g/t | NA | 4 |
| OREAS-15Pb | NA | 1.06 g/t ± 0.06 g/t | NA | 3 |
| OREAS-18Pb | NA | 3.63 g/t ± 0.14g/t | NA | 4 |
| NI-114 | 0.45%±.031% | NA | 0.037% ± 0.0038 | 6 |
| NI-116 | 0.78%±.027% | NA | 0.058% ± 0.0048 | 6 |
| PB 139 | 0.37%±.014% | NA | NA | 2 |

**CDN, NI and PB ranges reflect certified mean ± 2 standard deviations for laboratory data sets and OREAS materials reflect specified mean ± 2 standard deviations performance limits*

*Limited availability of certain reference materials was encountered during the 2010 drilling program and resulted in introduction of new standards not used in the earlier 2007-2008 program. Only the CDN HZ-2 standard is common to both programs. This material was used for drill holes CP10-26, CP10-28 and CP10-29. Standard NI-114 was used for hole CP10-26 and standard NI-116 was used for holes CP10-26, CP10-27, CP10-28 and CP10-29. 27. Standard PB-139 was used only in holes CP10-28 and CP10-29 and standards OREAS 15PB and OREAS 18PB were used only in hole CP10-30. Stratabound staff advised that inconsistency of coverage resulted from limited availability of reference materials and, in the case of hole CP10-30, unintended substitution of the OREAS15B and OREAS18PB Au standards.*

### *2010 Results for Reference Standard CDN-HZ-2*

*This standard was used in both the 2007-2008 and 2010 drilling programs and was obtained from Canadian Resource Laboratories Ltd. of Delta, BC. In total, 40 samples of the CDN-HZ-2 standard were analyzed during the two programs, but only four of these pertain to the 2010 drilling addressed in this report. Cu and Au results for the 2010 program are presented below in Figure 11.1 (Cu) and Figure 11.2 (Au) and those pertaining to 2007-2008 work are included in the Cullen and Harrington (2009) excerpt that appears in Appendix C.*

**Figure 11.1 Standard CDN-HZ-2 Results: Cu**

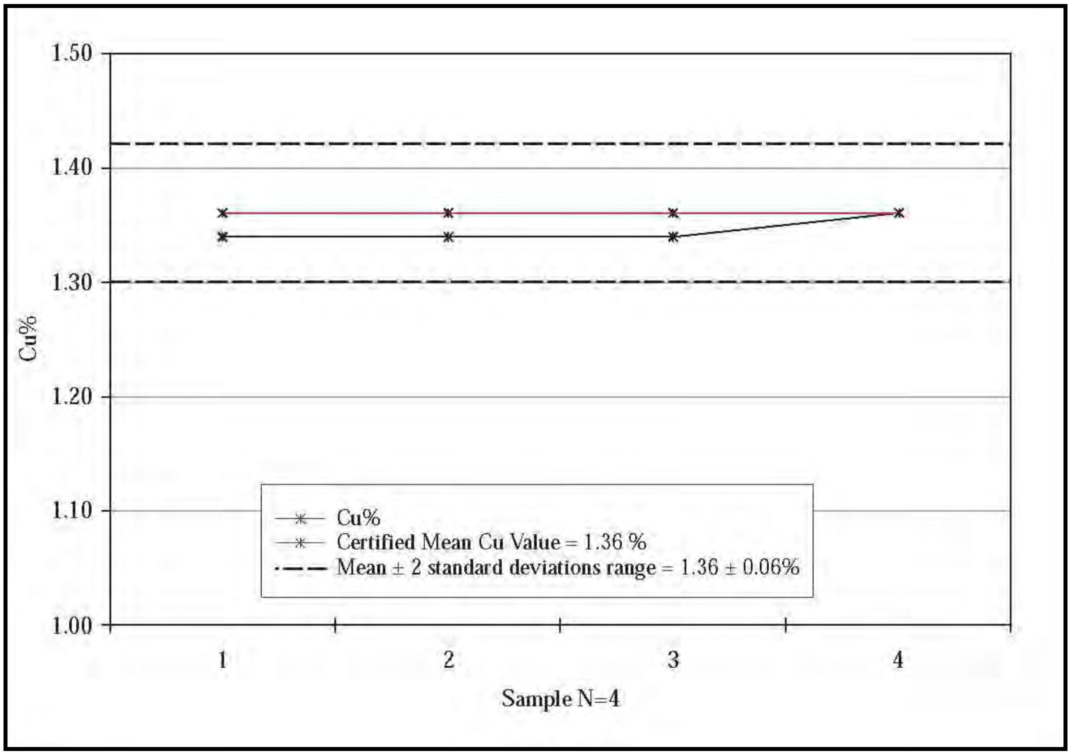
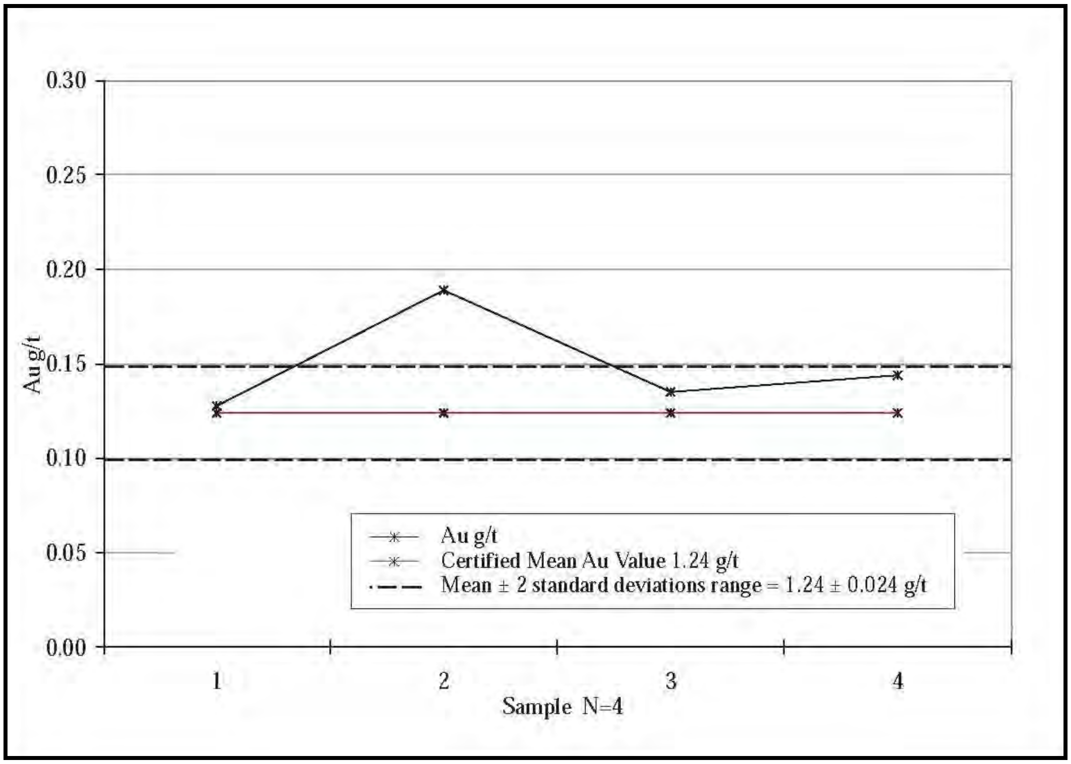



**Figure 11.2 Standard CDN-HZ-2 Results: Au**



*Cu results for 2010 have a mean value of 1.35% and are grouped closely around the certified reference value of 1.36±.06%. All results fall within ±2 standard deviations of the certified mean value for the standard. This is in contrast with results from the 2007-2008 program that show a low bias relative to the certified mean. Au results have a mean value of 0.149 g/t, and group above the standard's mean value of 0.124 g/t. The data set mean value exceeds the upper range limit by .001 g/t but three of four contributing samples fall within the mean plus 2 standard deviations control range for the standard. The remaining value of 0.189g/t exceeds the control range and is responsible for the 2010 mean falling outside these limits. In comparison to 2007-2008 Au results for the CDN-HZ-2 standard, the limited 2010 data set does not show continuation of a low bias trend noted by Cullen and Harrington (2009). Descriptive statistics for CDN HZ-2 results appear in Appendix D.*

### *2010 Results for Reference Standard NI-114*

*This material was supplied by WCM Sales Ltd. of Burnaby, BC and a total of 6 standard samples were analyzed during the 2010 drilling program. Compiled 2010 results for Cu and Co are presented below in Figure 11.3 (Cu) and Figure 11.4 (Co). The mean Cu value for 2010 drilling of 0.43% falls within the mean ± 2 standard deviations control range for the standard, which is 0.45 ± 0.03%, and all data plot systematically below the certified mean. The mean Co value of 0.029% is below the mean ±2 standard deviations control range for the standard, which is 0.037 ±.0038%, and data show a consistent trend at this level reflecting slight under reporting below the lower control range limit.*

*The reporting trends for NI-114 indicate a slight low bias for Cu and more pronounced low bias for Co in comparison to certified values and ranges. However, results for other standards within the same sample stream show closer agreement with reference material mean value ranges. Low n values probably contribute to the low reporting trends but analytical matrix effects and other factors may also contribute. Descriptive statistics for standard NI-114 appear in Appendix D.*

**Figure 11.3 Standard NI-114 Results: Cu**

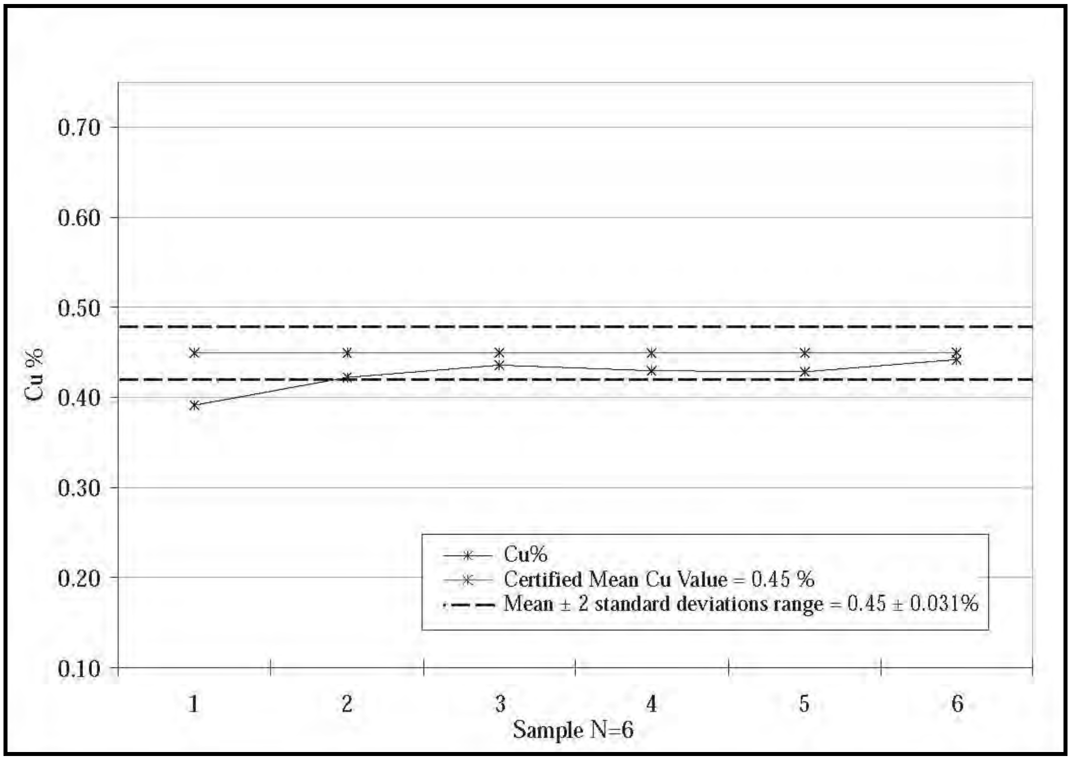
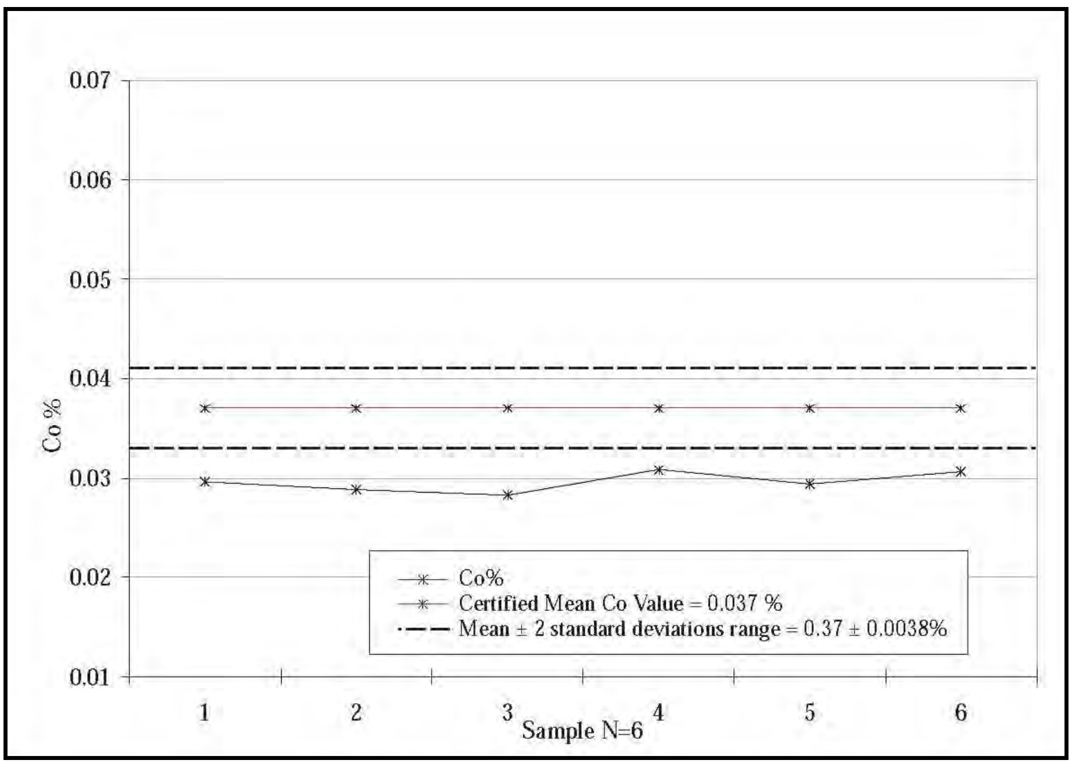



**Figure 11.4 Standard NI-114 Results: Co**



#### *2010 Results for Reference Standard NI-116*

*This Cu, Ni and Co certified reference material was supplied by WCM Sales Ltd. of Burnaby, BC and was used in all 2010 drill hole sampling except that associated with hole CP10-030. In total, six analyses for Cu and Co were returned and results for these are presented in Figure 11.5 (Cu) and Figure 11.6 (Co). Cu values have a mean of 0.68% which is below the 0.753% Cu mean plus ± 2 standard deviations lower control range limit for the material. The 2010 data group consistently around a 0.70% Cu trend line.*

*Co values for the 2010 program also show a low bias, with the 2010 mean value of 0.046% being below the 0.053% mean plus ± 2 standard deviations lower control range limit for the material. All values group around a 0.045% Co trend line that defines a systematic trend of under-reporting.*

*As in the case for NI-114, no clear explanation exists for the low reporting trends noted for Cu and Co. However, the systematic nature of trends for both Co and Cu may indicate that sample matrix effects are a contributing factor, since other standards in the same sample stream, such as CDN HZ-2, returned more acceptable results. Descriptive statistics for standard NI-116 results appear in Appendix D.*

#### *2010 Results for Reference Standard CDN GS-2B*

*This certified reference material for Au was supplied by Canadian Resource Laboratories Ltd. of Delta, BC and a total of 4 standard samples were analyzed during the 2010 drilling program. Au results for 2010 are presented in Figure 11.7 and have a mean value of 1.89 g/t which closely approximates the lower control range limit for the standard. The certified Au value and mean ± 2 standard deviations control range for the material is 2.03 ±0.12%g/t. Two of the four samples fall within the standard's control range. Descriptive statistics for standard CDN GS-2B results appear in Appendix D.*

#### *2010 Results for Reference Standard OREAS-15PB*

*This certified reference material for Au was supplied by Ore research & Exploration Proprietary Ltd. of Bayswater, Victoria, Australia and a total of 3 standard samples were analyzed during the 2010 drilling program. Au results for 2010 are presented in Figure 11.8 and have a mean value of 0.96 g/t. All but one sample reported below the mean ± 2 standard deviations control range for the standard, which is 1.06 ± .06 g/t. The mean Au value for 2010 reflects systematic under-reporting of Au grades below this range. Descriptive statistics for standard OREAS-15PB appear in Appendix D.*

**Figure 11.5 Standard NI-116 Results: Cu**

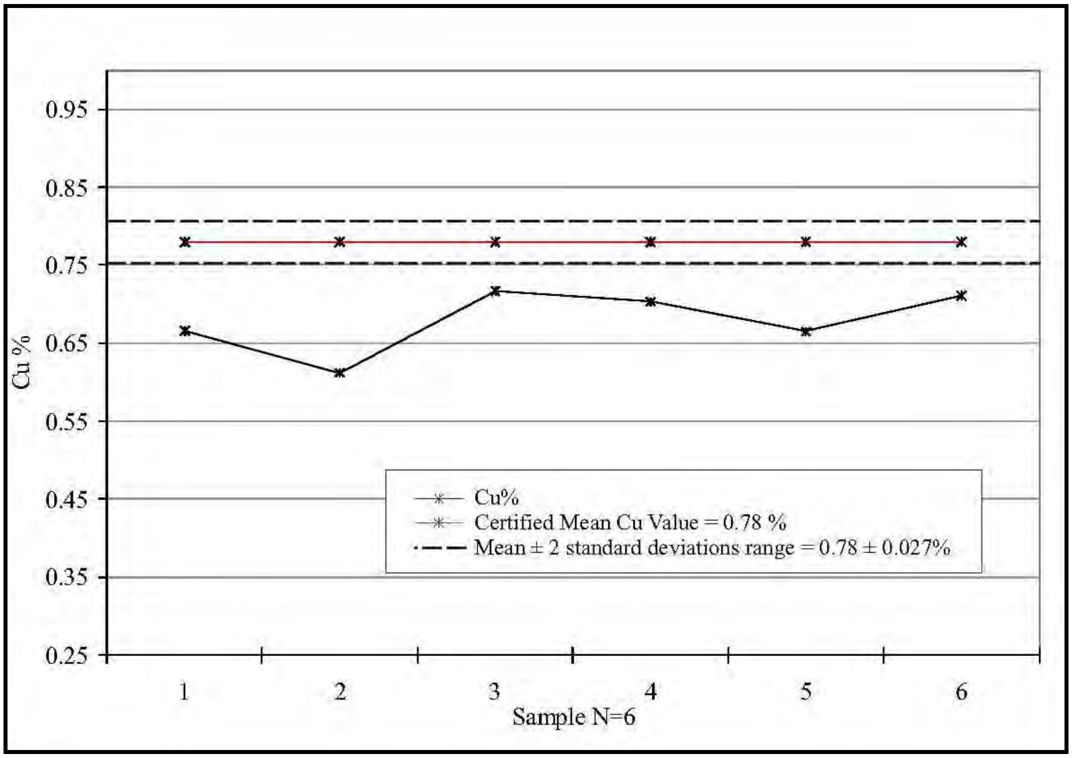



**Figure 11.6 Standard NI-116 Results: Co**

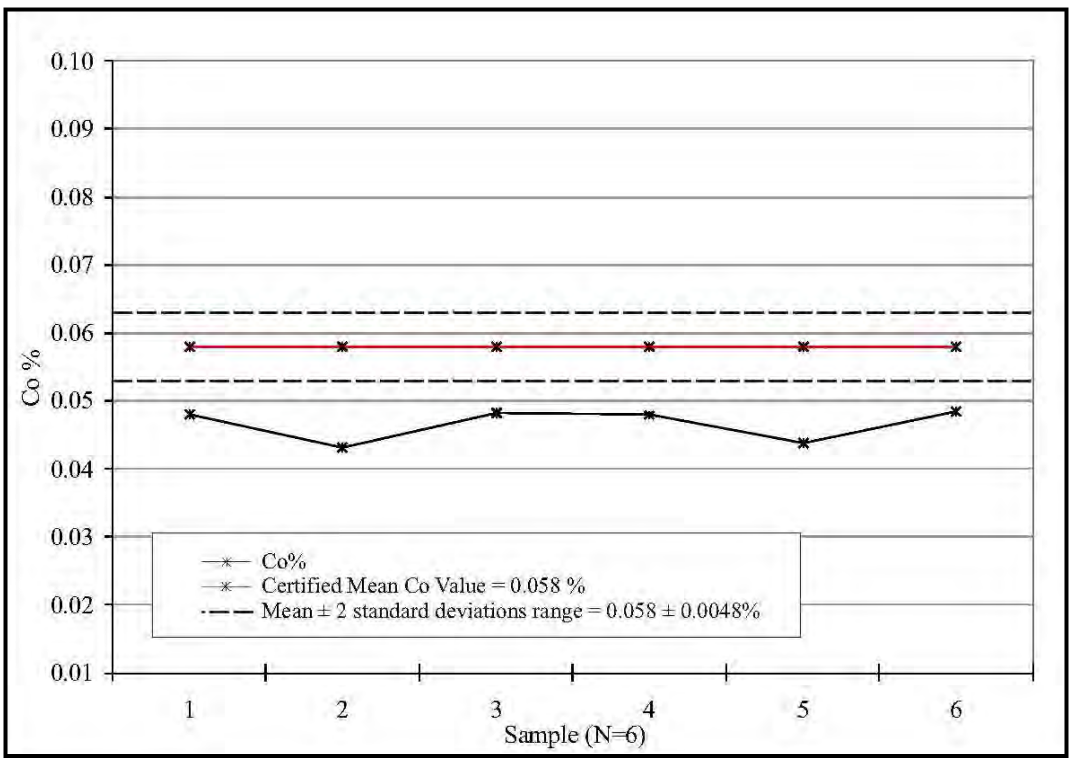


**Figure 11.7 Standard CDN GS-2 Results: Au**

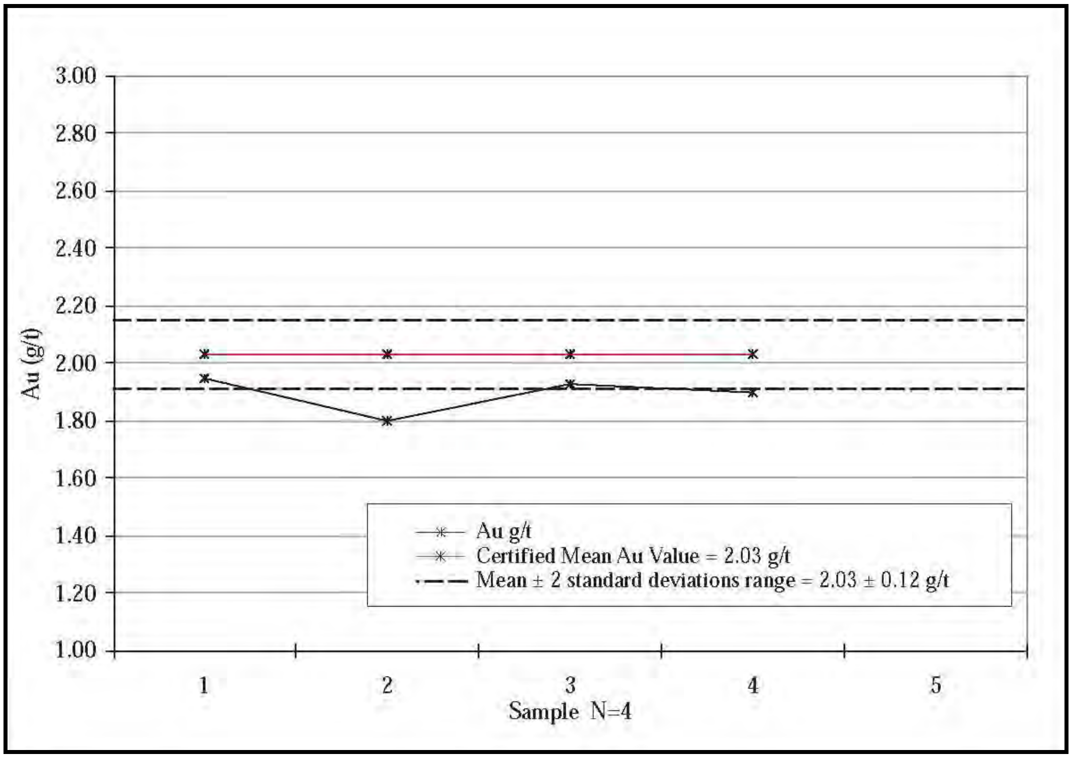
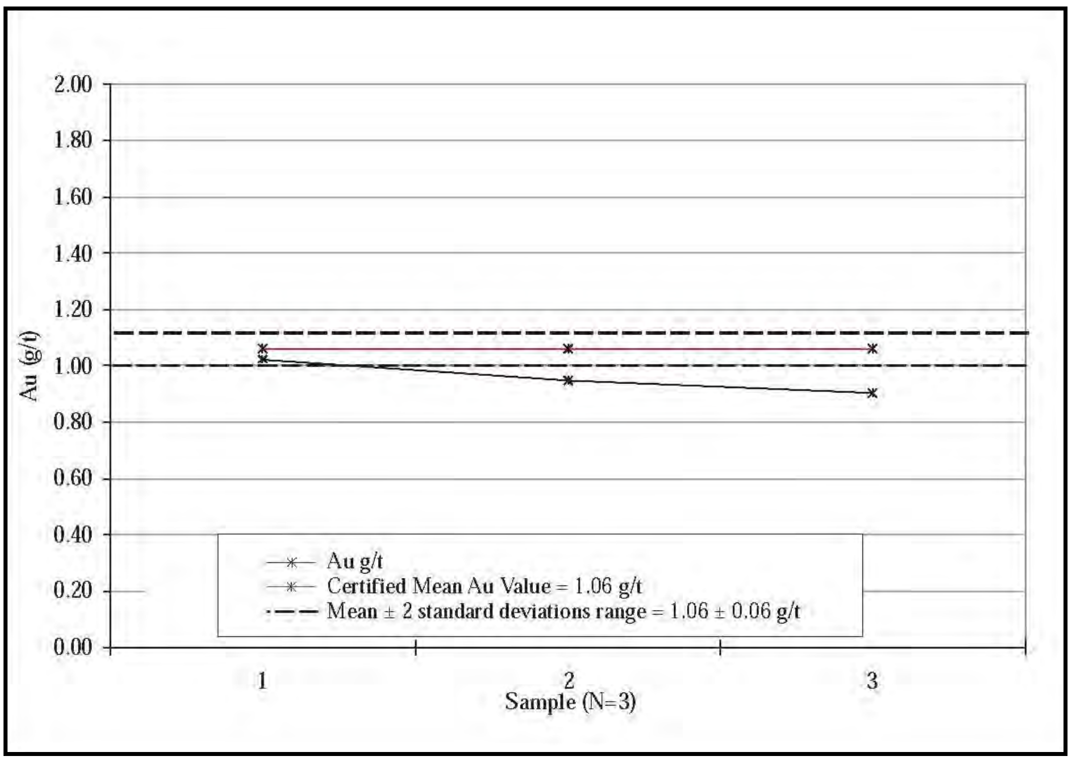



**Figure 11.8 Standard OREAS 15PB: Au**



#### *2010 Results for Reference Standard OREAS-18PB*

*This certified reference material for Au was also supplied by Ore Research & Exploration Proprietary Ltd. of Bayswater, Victoria, Australia and a total of 4 standard samples were analyzed during the 2010 drilling program. Results for 2010 are presented in Figure 11.9 and have a mean Au value of 3.00 g/t. All results fall substantially below the 3.63 ± .14 g/t. mean plus ± 2 standard deviations control range for the standard. The mean Au value for 2010 reflects systematic under reporting of the average Au grade and none of the 2010 samples falls within 2 standard deviations of the standard's mean value. Descriptive statistics for standard OREAS-18PB appear in Appendix D.*

#### *2010 Results for Reference Standard PB-139*

*This certified reference material for Cu, Pb, Zn and Ag was supplied by WCM Sales Ltd. of Burnaby, BC. Only the certified Cu value is pertinent to this report and a total of 3 standard samples were analyzed during the 2010 drilling program. Results for 2010 are presented in Figure 11.10 and have a mean Cu value of 0.35%. Both samples report slightly below the 0.36% lower limit for the standard's mean ± 2 standard deviations control range and reflect slight under-reporting of average Cu grades. Descriptive statistics for standard PB-139 results appear in Appendix D.*

#### *Summary Comments on 2010 Certified Standards Program*

*Very small populations characterize all certified reference material datasets associated with the 2010 drilling program and incomplete reference material coverage exists for Cu, Au and Co with respect to the five 2010 holes. The latter resulted from samples for hole CP10-27 being submitted without inclusion of any Au reference materials and those for hole CP10-30 being submitted without Cu or Co reference materials. In both instances the laboratory quality control and quality assurance datasets reported with each analytical batch were relied upon to provide coverage of the metals not included by Stratabound. In all instances, laboratory datasets returned acceptable results.*

**Figure 11.9 Standard OREAS 18 PB: Au**

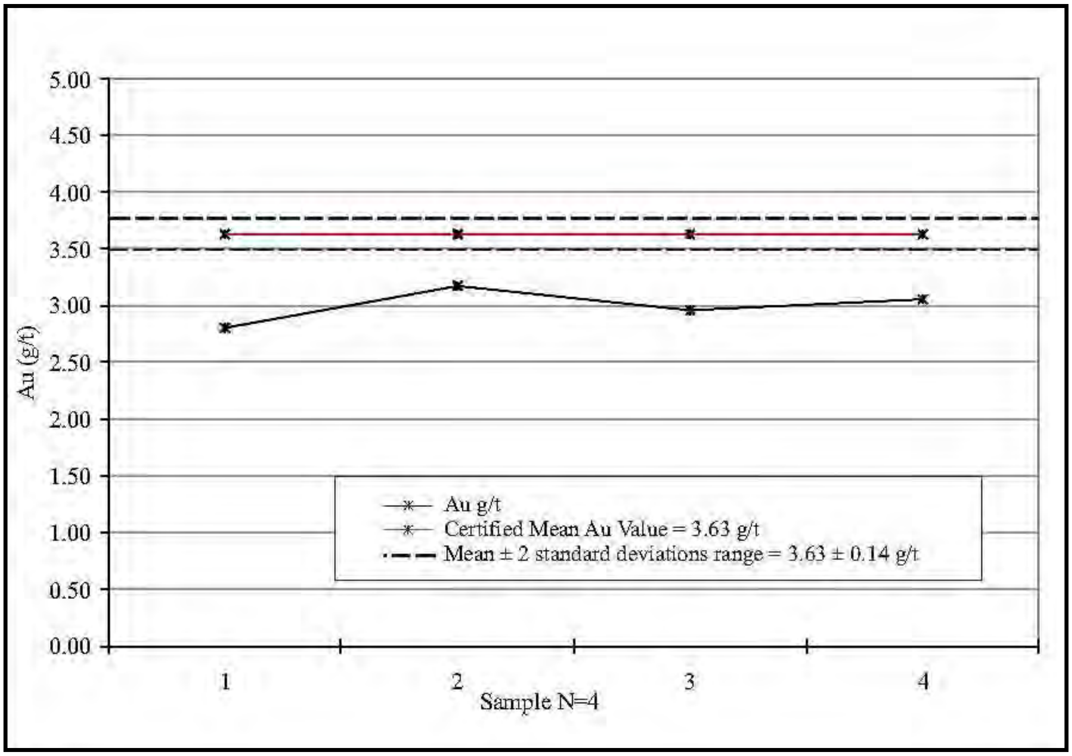
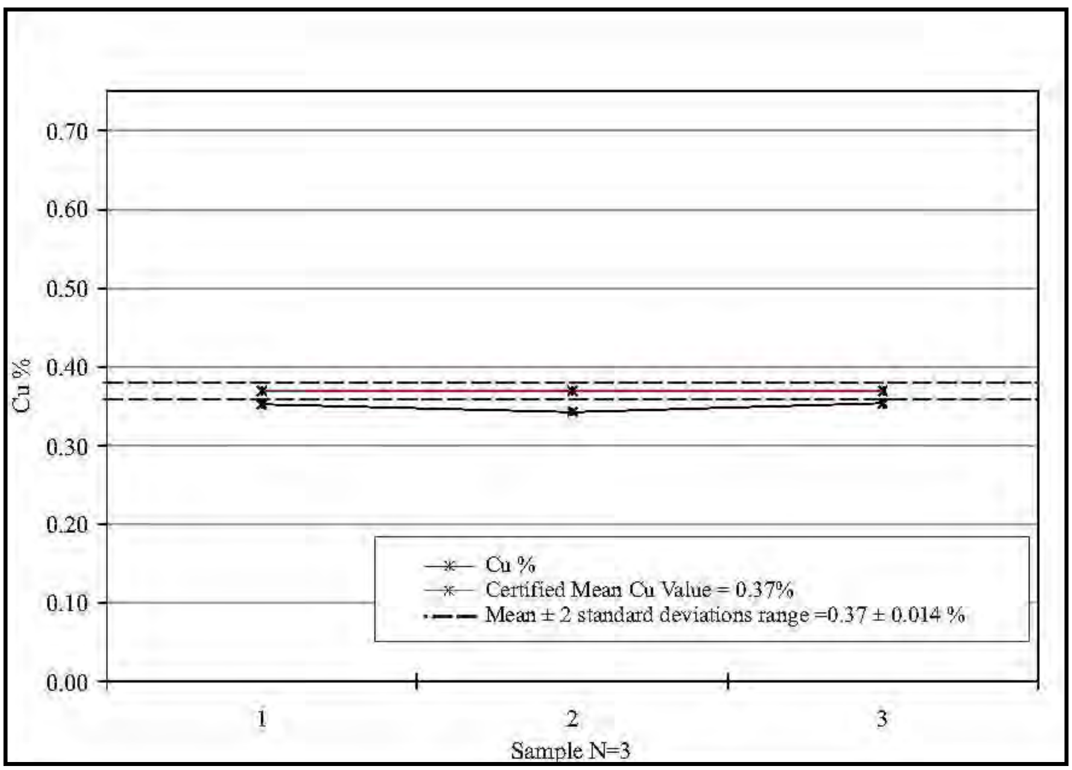



**Figure 11.10 Standard PB-139: Cu**



*Based upon review of individual 2010 data set trends described above, plus associated graphic distribution patterns, it is apparent that systematic under reporting of Cu, Co and Au values occurred with respect to certified reference materials OREAS-18PB, OREAS 15PB, CDN GS-2, NI-114, NI-116 and PB-139. Reasonable precision of results is typically seen but accuracy varies within two standard deviations of associated means. In contrast, higher degrees of both precision and accuracy characterize results for the CDN-HZ-2 standard used in conjunction with standards PB-139, NI-116 and NI-114 for samples from 3 of the 2010 holes. This suggests that sample preparation and analytical methods used in the 2010 program were better matched to matrix and composition characteristics of CDN HZ-2 than to those of the other materials. Further to this point, review of reference material support documents showed that CDN HZ-2 standard does not contain gold-bearing arsenopyrite, while the NI-114 and NI-116 standards contain sulphides, including arsenopyrite. It is possible that differential extraction of Au from auriferous sulphides characterized these two materials relative to CDN HZ-2 and that this contributed to systematic under-reporting of Au levels for standards NI-114 and NI-116 used in the 2010 Stratabound program. Other mineralogical factors may account for variation seen in the Cu and Co datasets but these are not immediately apparent from reference material descriptions.*

*Notwithstanding trends described above, combined results of certified reference material programs carried out by both the project laboratory and Stratabound are considered sufficiently consistent to support use of associated datasets for current resource estimation purposes. However, measures should be taken by Stratabound to (1) prevent future submission of sample shipments lacking in full reference material coverage for the metal assemblage Cu, Co and Au, (2) further assess the under-reporting trends that characterise some reference materials used during 2010, and (3) assess the possibility of a project specific reference material being prepared from mineralized Captain deposit bedrock, the purpose being to more closely match matrix characteristics of reference materials and deposit samples.*

### 11.1.1(c) Blank Sample Programs

#### *Introduction*

*Blank samples of comparable weight to normal 0.5 meter half core samples were systematically inserted into the laboratory sample stream by Stratabound staff during the 2007-2008 program, with 222 such samples being submitted for analysis. This approximates an insertion rate of 1 blank per 20 core samples submitted. Two blank sample lithologies were used by Stratabound during this period, these being (1) visibly non-mineralized gabbro drill core from a local intrusion and (2) calcareous siltstone drill core from Stratabound's Elmtree gold project, located approximately 24 kilometers northwest of Bathurst. Company records do not systematically identify lithology of inserted blank samples but from discussions with Stratabound staff it was determined that gabbro samples were* *used in the early part of the drilling program, being replaced later by the calcareous siltstone. Cullen and Harrington (2009) discussed results of the blank sample program and concluded that program results were sufficiently consistent to indicate that no major cross-contamination issues were present in the data set. However, they also noted that a more geochemically consistent materials be used for future sampling programs.*

*During the 2010 program the nominal 1 in 20 frequency for blank sample insertion rate was maintained and sample material was switched to a screened commercially available blasting sand product. In total, 55 blank samples were submitted along with core samples from the five drill holes completed in 2010 and results for these are discussed below. Analytical results for these are discussed below. Use of a pre-sized sand product for blank sample purposes is not optimal, since exposure of such to primary crushing stages of the sample preparation circuit is minimal in comparison to rock or core materials within the same sample stream. A substantially coarser granular product or a coherent, homogenous rock material should be used in future programs at Captain.*

#### *2010 Blank Sample Cu Results*

*The average Cu value for the 55 blank samples is 22.9 ppm with an analytical detection limit of 0.5 ppm and maximum value of 132 ppm. Most data group below the 40 ppm level and 6 grade spikes exceeding 50 ppm are present (Figure 11.11). Grade spikes are typically defined by only one sample and show lower frequency of occurrence in the central zone of the dataset. This may indicate that two sand sources with slightly differing trace metal contents were used for the blank samples. The Au dataset does not show a similar trend but Co values are in part similarly distributed. No substantial cross contamination effects with respect to Cu levels are interpreted to be present in the blank sample Cu data set.*

#### *2010 Blank Sample Au Results*

*The average Au value returned for the 55 blank samples is 4.5 ppb with an analytical detection limit of 5 ppb applicable. The Au data set mean falls below the stated detection limit because samples that returned “less than detection limit” values were entered as 2.5 ppb for purposes of analysis. Only two samples exceeded the detection limit, both registering 10 ppb (Figure 11.12). Based on these results, no significant cross-contamination effect is interpreted to be present in the Au data set.*

**Figure 11.11 2010 Blank Sample Results: Cu**

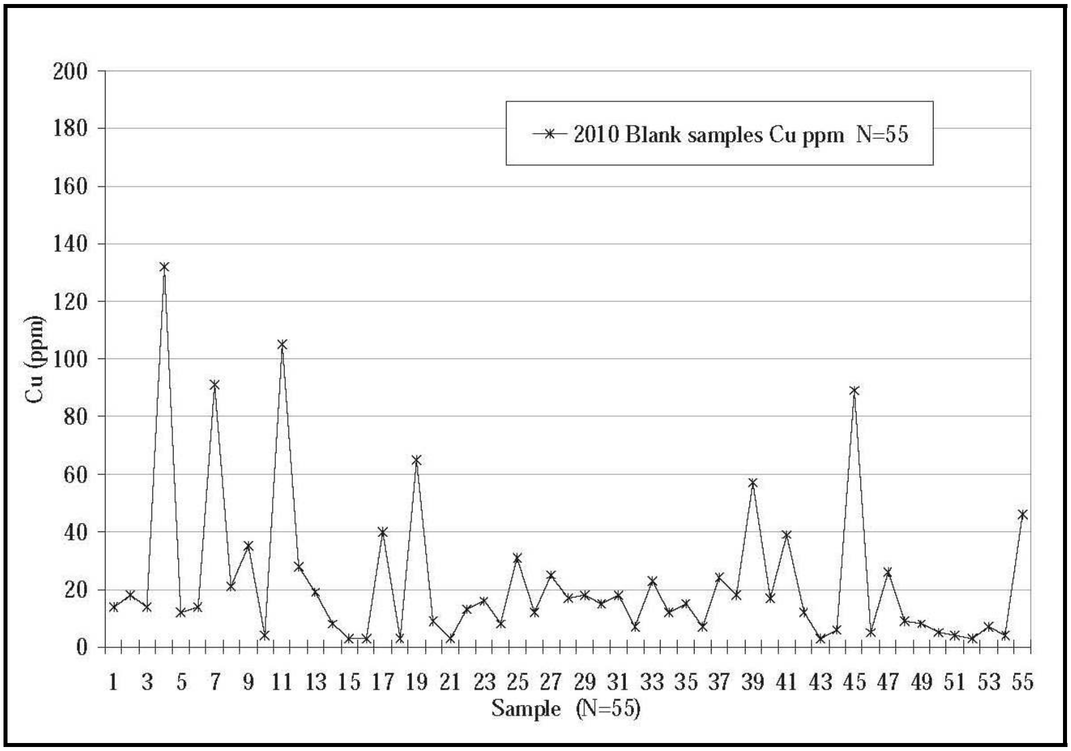
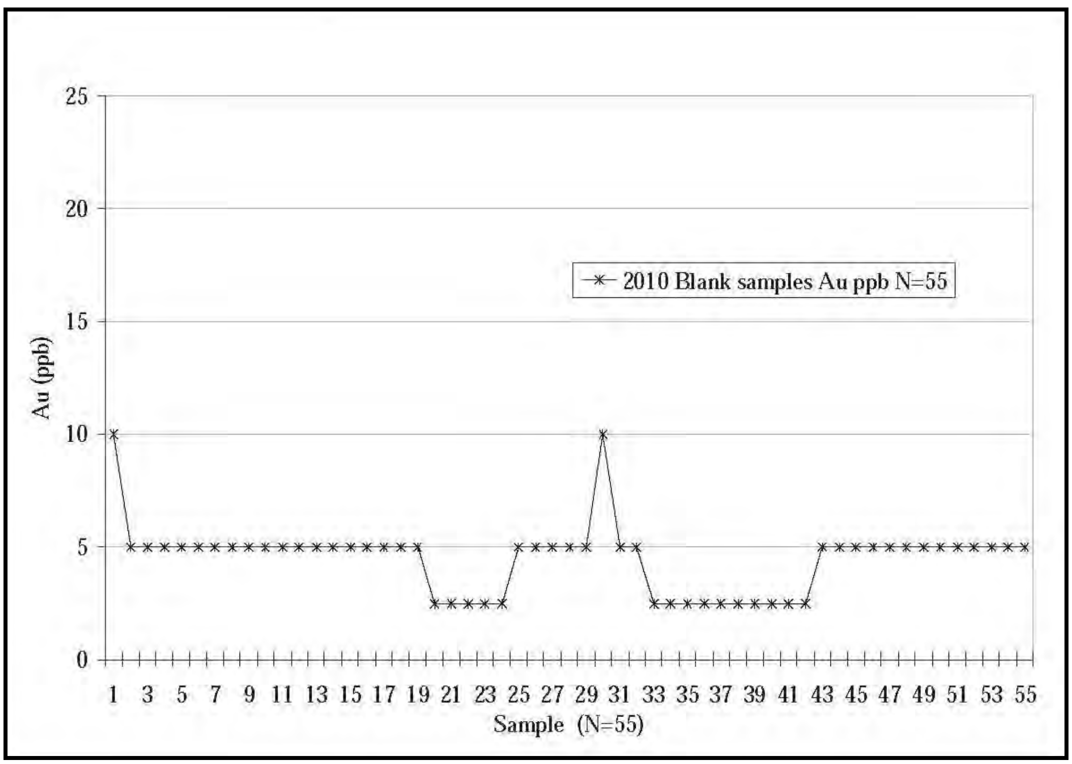



**Figure 11.12 2010 Blank Sample Results: Au**



*2010 Blank Sample Co Results*

*The average Co value returned for the 55 blank samples is 13 ppm with an analytical detection limit of 1.0 ppm applicable. With the exception of the survey high of 58 ppm, all samples returned values of 13 ppm or less (Figure 11.13). Spiking of values is more prevalent in the first half of the data set but this trend does not exactly coincide with that mentioned earlier for Cu. Based on these results, and the overall low levels of Co detected, no significant cross-contamination effect is interpreted to be present in the Co data set.*

### 11.1.1(D) Pulp Duplicate Split Programs

*Splits of coarse sample pulp material from the initial core sample preparation stream core were prepared for analysis as duplicate splits at a nominal frequency of every 25$^{th}$ sample submitted for both the 2007-2008 and 2010 programs. In total, results for 189 pairs were returned for the 2007-2008 program and the 2010 program included 45 pairs. Program results for Cu, Au and Co are presented in Figure 11.14 (Cu), Figure 11.15 (Co) and Figure 11.16 (Au) and 2010 data pairs for the three metals support correlation coefficients of 0.94 (Cu), 0.92 (Au) and 0.98 (Co). Cu and Au results for the 2010 program show generally more scatter than those from 2007-2008 while Co data define a slight trend toward reporting of lower values in the duplicate split. The Co and Au data sets provide coverage throughout the range of grades occurring within the deposit but that for Cu lacks substantial coverage above the 0.75% Cu level. These trends reflect the fact that only one of the 2010 drill holes (CP10-26) intersected substantial widths of mineralization grading in excess of 1% Cu.*

### 11.1.1(E) Quarter Core Duplicate Programs

*In addition to analysis of duplicate splits of core sample pulps, Stratabound carried out a program of quarter core sampling to check on variation of results between half core sample components. In total, 105 samples were analysed for the 2007-2008 program and 19 samples were analysed for the 2010 program. Cullen and Harrington (2009) discussed results of the 2007-2008 work and found results to be generally acceptable. Details of this program are included in the 2009 report excerpt included in Appendix C. Results for Cu, Au and Co associated with the 2010 program are presented below in Figure 11.17 (Cu), Figure 11.18 (Co) and Figure 11.19 (Au). Correlation coefficients for Cu, Co and Au are 0.92, 0.93 and 0.86 respectively.*

**Figure 11.13 2010 Blank Sample Results: Co**

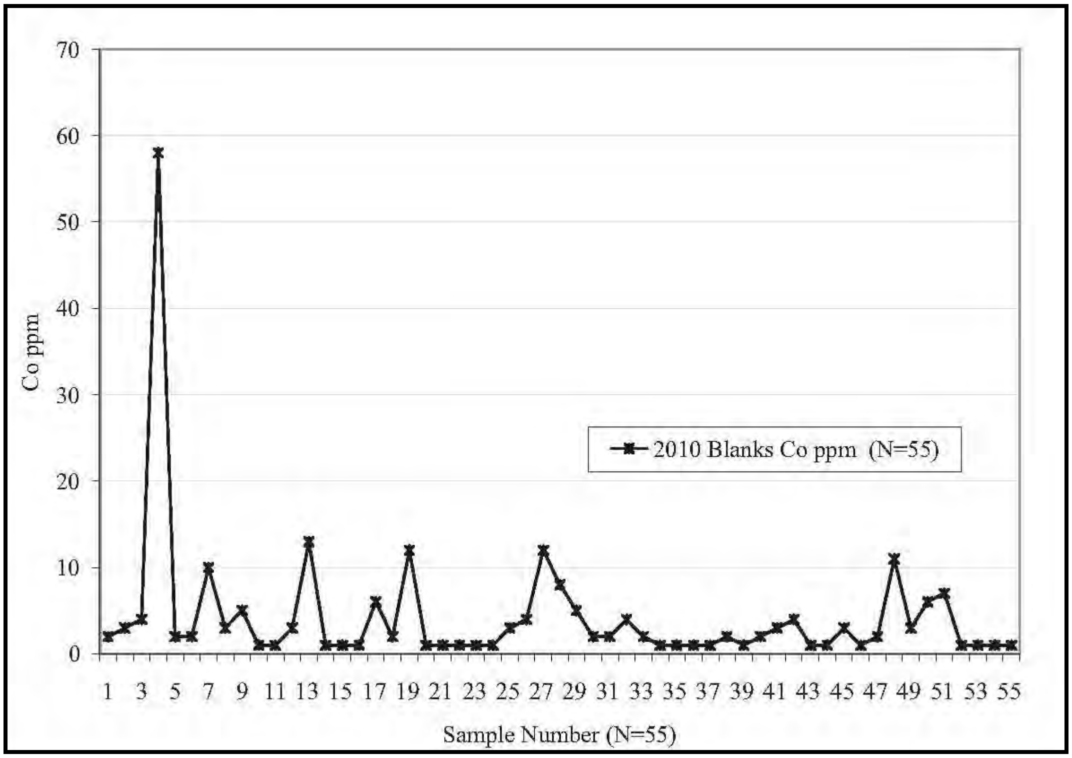
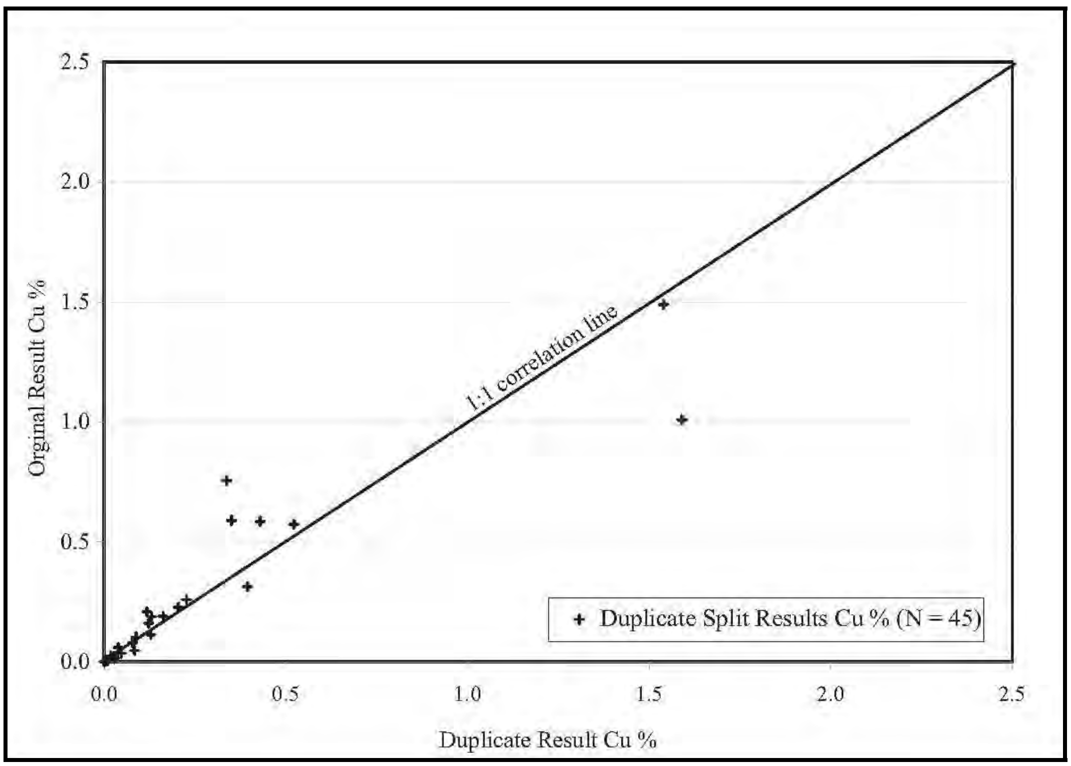
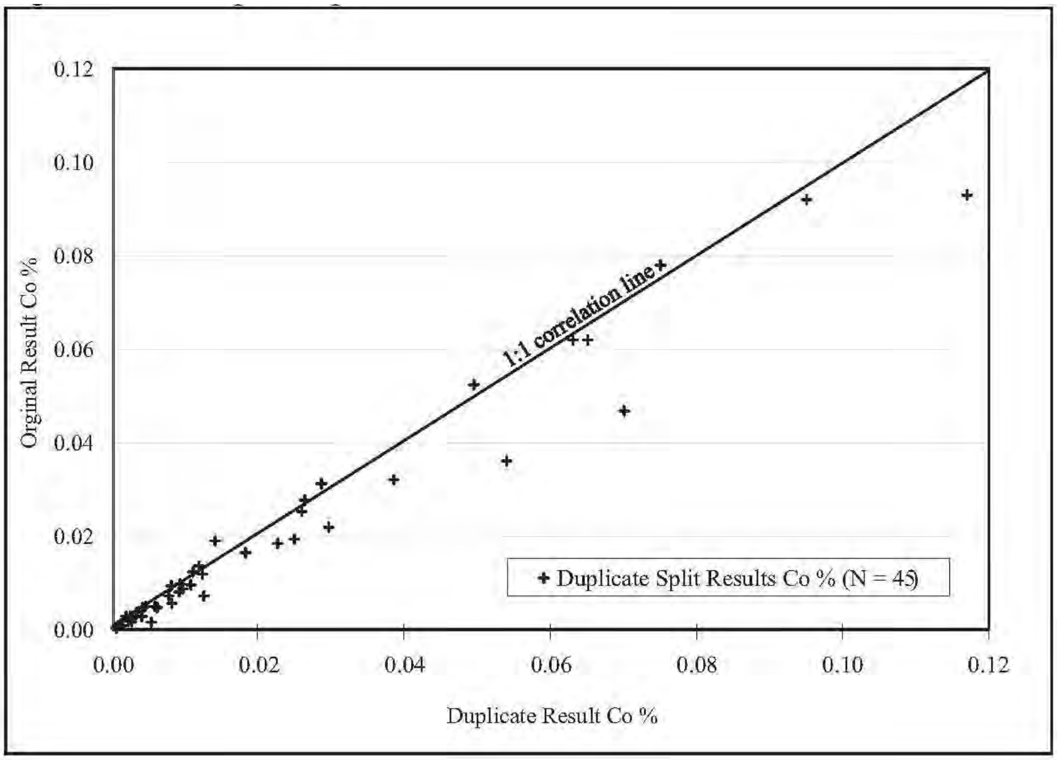



**Figure 11.14 2010 Duplicate Split Results: Cu**



### Figure 11.15 2010 Duplicate Split Results: Co




### Figure 11.16 2010 Duplicate Split Results: Au

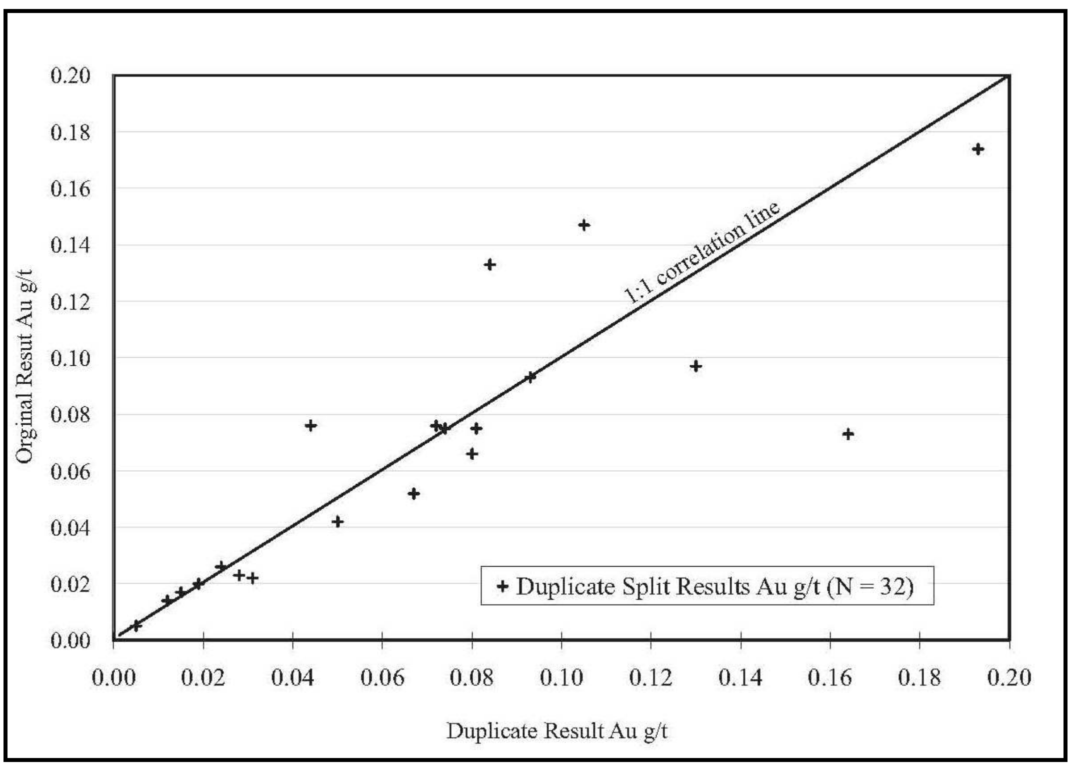


**Figure 11.17 Duplicate Quarter Core Split Results: Cu**

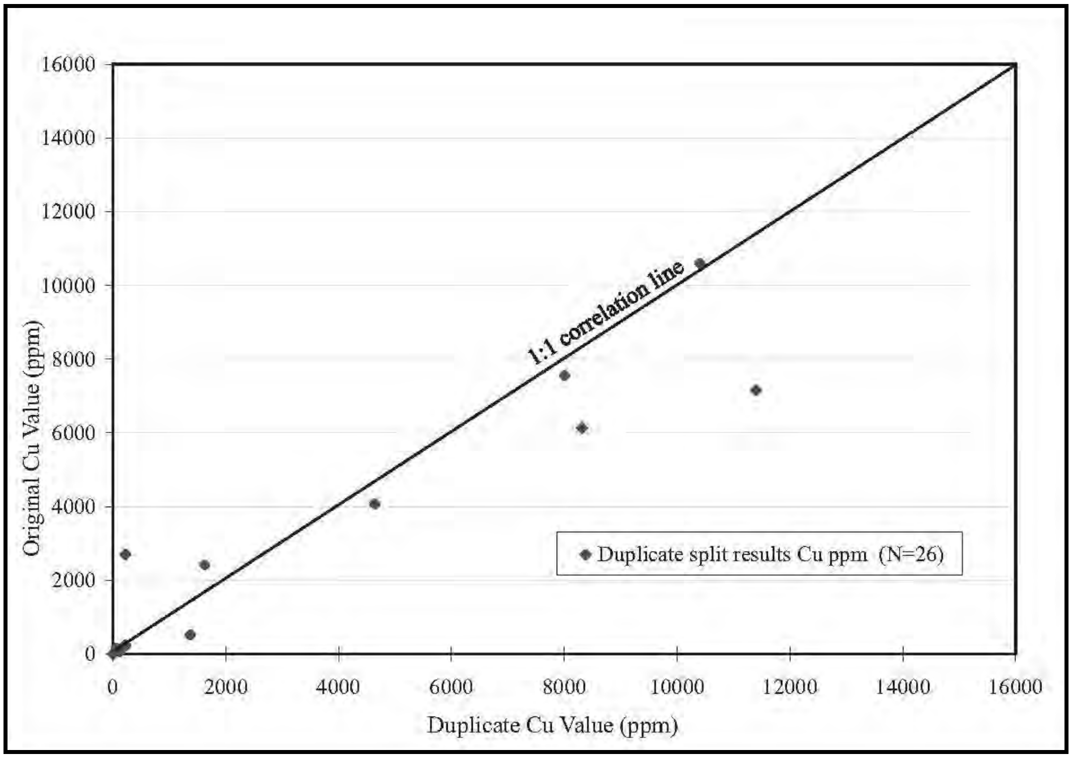



**Figure 11.18 Duplicate Quarter Core Split Results: Co**

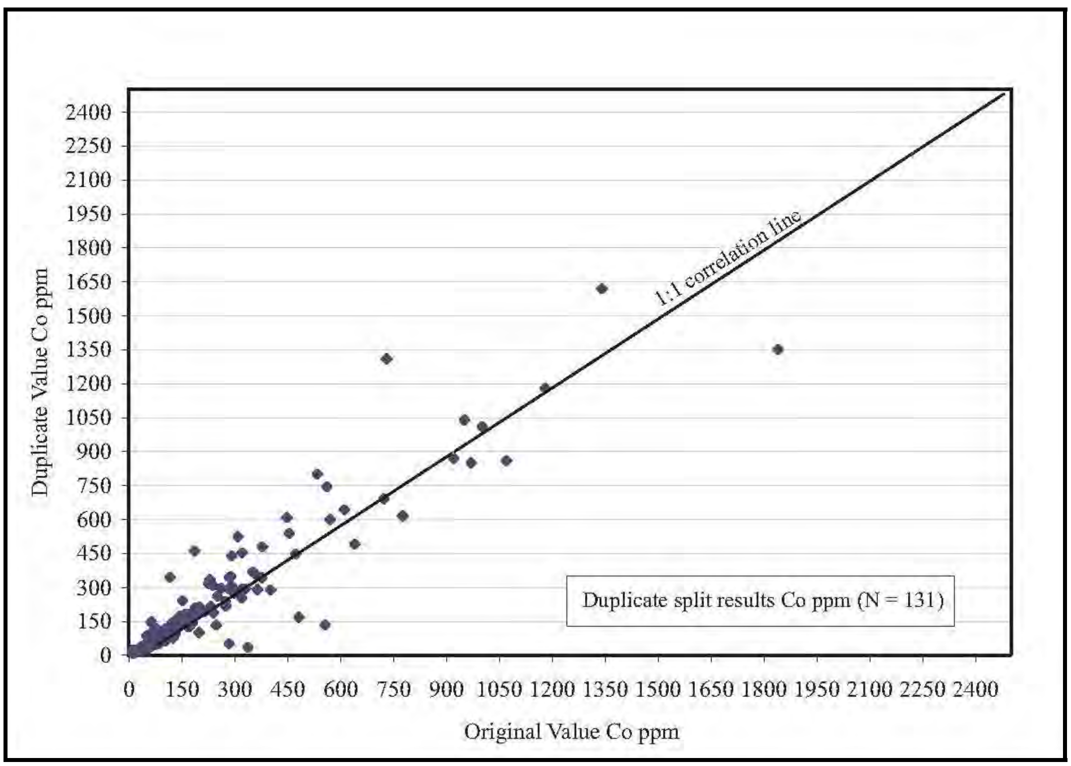
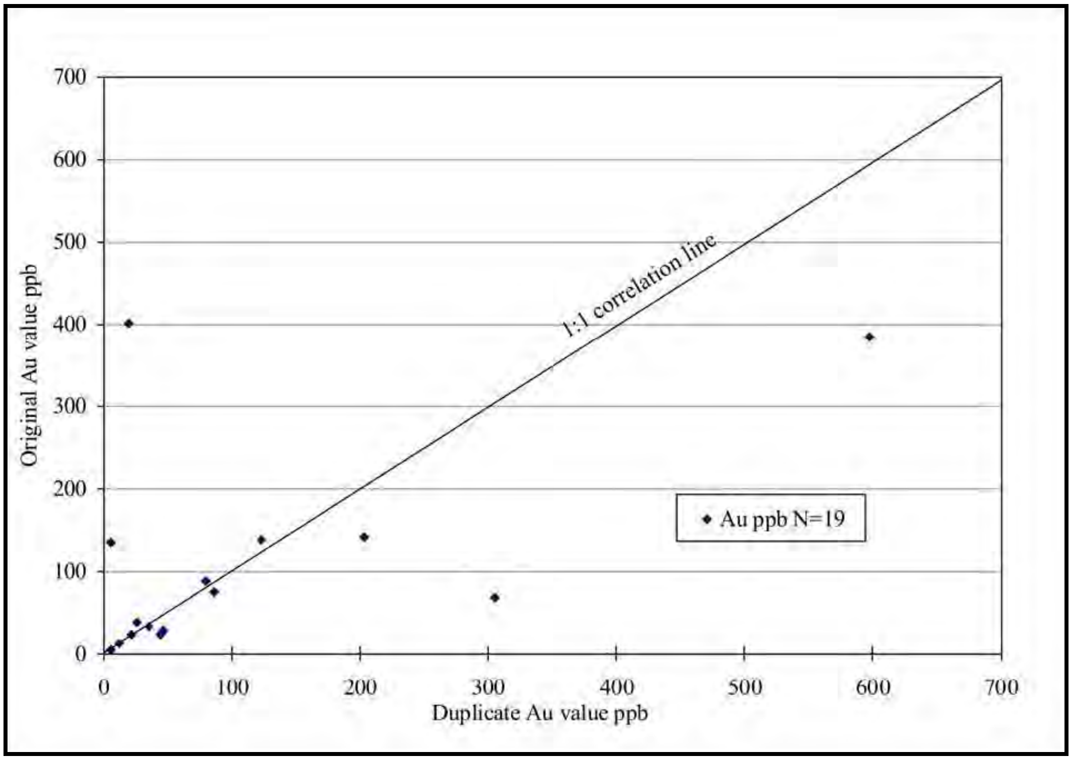


**Figure 11.19 Duplicate Quarter Core Split Results: Au**




## 11.1.1(F) CHECK SAMPLE PROGRAMS

### *Stratabound Programs*

*Stratabound incorporated collection of third party check samples in both the 2007-2008 and 2010 drilling programs, with a pulp split prepared from every 25th sample for this purpose. Full results were not available for the 2007-2008 program at the time of reporting by Cullen and Harrington (2009) and data assessment at that time was based on check samples collected by Mercator in combination with results for the certified standards used during the program. For the 2010 program a total of 42 check samples of prepared pulp were analyzed at AGAT Laboratories Inc., an ISO accredited commercial analytical firm with corporate headquarters located in Calgary, AB.*

**Figure 11.20 Stratabound Check Sample Results: Cu**

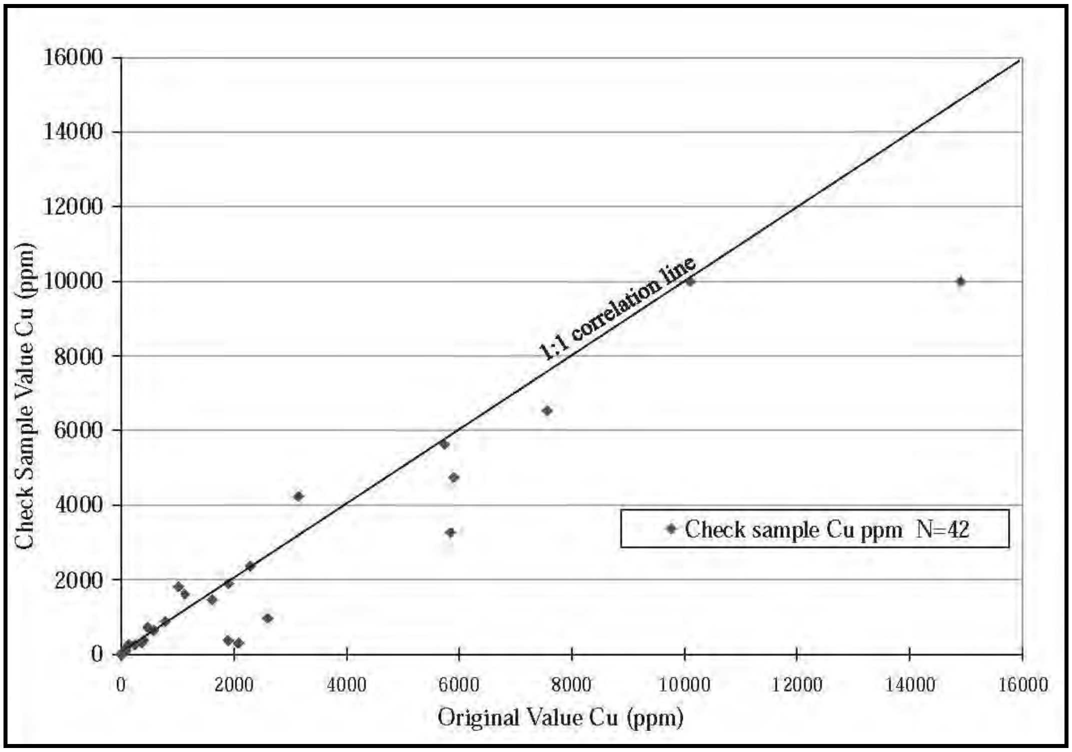



**Figure 11.21 Stratabound Check Sample Results: Co**

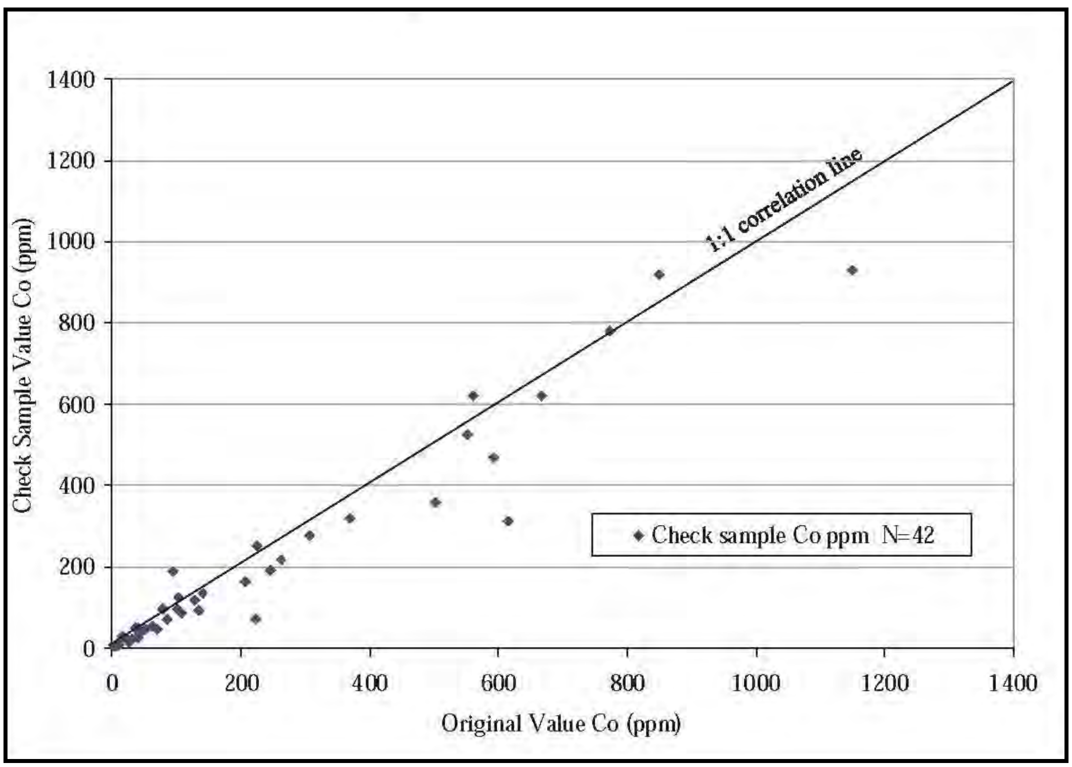
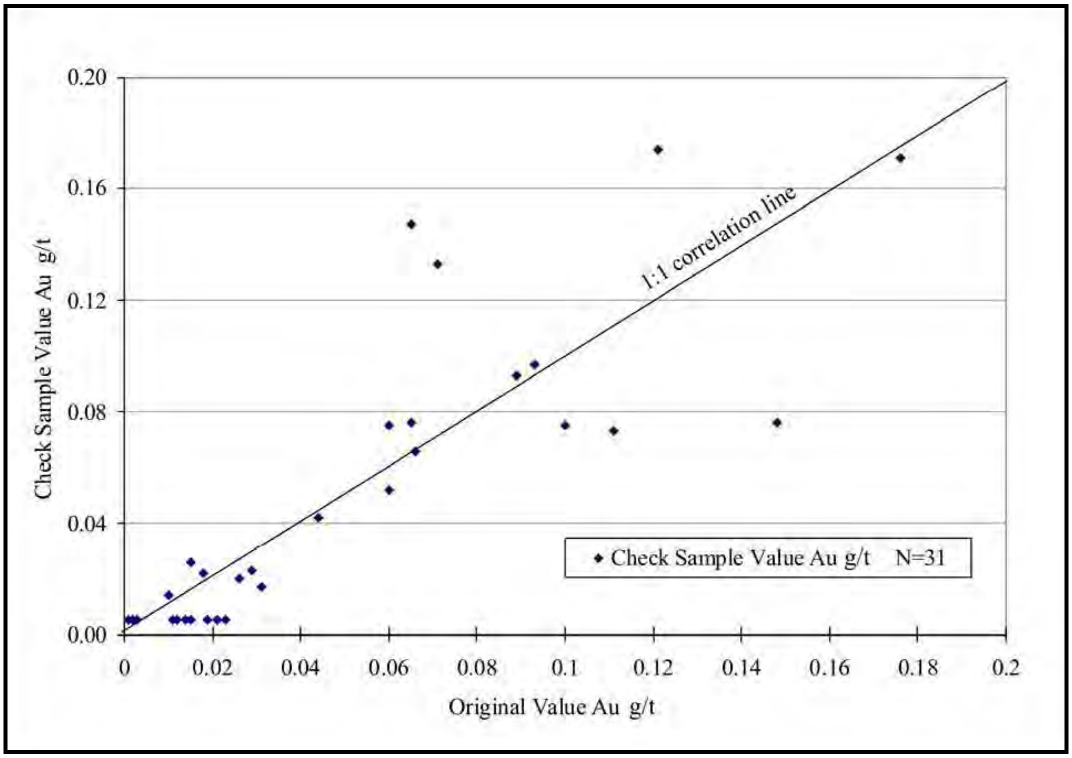


**Figure 11.22 Stratabound Check Sample Results: Au**




*Check sample analytical results for Cu, Au and Co appear in Figure 11.20 (Cu), Figure 11.21 (Co) and Figure 11.22 (Au) and support correlation coefficients for Cu, Co and Au are 0.97, 0.98 and 0.88 respectively. Higher correlation for the Cu and Co datasets is consistent with results for the quarter core sampling program discussed above, with Au in both cases showing greatest tendency to differ between splits. In contrast, Cu and Co results for the 2010 program compare more closely between laboratories. Check analyses from AGAT define a slight low bias relative to original values and Au results above approximately 0.070g/t show increasing inconsistency between sample pairs. Overall results of the 2010 check sample program are interpreted to show acceptable confirmation of Stratabound's dataset mineralization levels for Cu and Co and to more broadly confirm the gold range represented in the original drill core data set.*

## 11.2 CNE Sample Preparation, Analysis and Security

### *11.2.1 Core Logging and Sampling*

Core logging begins on arrival of the core boxes to Stratabound's office. The drilling company has a key to the garage door at the back of Stratabound's office and leaves the core boxes inside the core storage area. Core boxes are labelled by the drillers in permanent marker with metre markers inside the box.

Drill core is brought into the core logging area of the office where it is rough-logged by Stratabound's geologist. Metreage and sample intervals are marked by the geologist in red or yellow grease pencil. The core is subsequently logged in detail. Neither core photographs, nor density measurements are taken at this point.

Two sample tags are printed with the sample interval and drillhole number on paper and then laminated. One sample tag is stapled into the core box, at the start of the sample interval; the other is stapled to a sample bag. Sample tags are made for duplicates, blanks, and standards and also stapled to the sample bags. Core box labels are then printed, laminated and stapled on the ends of each core box.

Once the drill core has been logged in detail, the core is taken to the core-cutting station where the core is sawed in two. One-half remains in the core box and the other half is placed in the sample bag with the corresponding sample tag. The bag is left unsealed until the entire hole is sampled. Duplicates are taken by the lab, where standards and blanks are inserted at regular intervals.

Once the drillhole is completely sampled, the sample bags are weighed, sealed and placed in cardboard boxes for shipping.

Upon completion of the sampling of the drill core, the core boxes are stored in the core racks of Stratabound's office to be transferred at a later time to the government core storage facility in Madran, located approximately 25 km north of Bathurst.

The drill logging and sampling procedures are conducted to industry standards. Both the core-logging area and core-cutting areas are kept clean. The rock saw is kept clean by collecting (scrapping) the slimes from the cutting table, but it was not clear whether this was done after each sample, or whether it was cleaned with water after every sample to avoid contamination.

### *11.2.2 DATABASE*

All data is compiled and reviewed by Mr. Vienneau. Mr. Vienneau is the only person with access to the database.

### *11.2.3 SAMPLE, ANALYSIS AND CHAIN OF CUSTODY*

During the drill program at CNE, drill core samples were sent out regularly to Eastern Analytical Laboratories (Eastern) in Springdale, Newfoundland and Labrador. Unsealed sample bags are sorted and recorded quality control to ascertain standards, blanks, and duplicates are included in the shipment. Each sample bag is weighed and sealed prior to being placed in cardboard boxes, which are themselves subsequently sealed. The sample manifest is placed in the final box before it is sealed. Batches of samples are made by drillhole, so a complete drillhole is sent to Eastern as a single shipment.

The sealed boxes are taken by Stratabound personnel to a local trucking company, Armour Transportation System (Armour) for shipment to Eastern. The Armour office is located one kilometre southeast of Stratabound's office.

#### 11.2.4 SAMPLE PREP AND ANALYSIS

At Eastern, samples are crushed and pulverized where pulps and rejects are separated. The pulps undergo an Aqua Regia Digestion and are analyzed for 30 elements by Inductively Coupled Plasma analysis. Gold is analyzed by Fire Assay and Atomic Absorption.

# 12.0 DATA VERIFICATION

## 12.1 CAPTAIN DATA VERIFICATION, 2008-2010

### 12.1.1 REVIEW AND VALIDATION OF PROJECT DATA SETS (CULLEN AND HARRINGTON, 2011)

*Stratabound files consisting of core sample records, lithologic logs, laboratory reports and associated drill hole information for all 2007-2008 and 2010 holes, plus those for historic drill holes compiled by Stratabound staff and made available to Mercator, were reviewed by Mercator. Historic drilling records and reports on all other past property exploration were also accessed through the provincial government assessment report archive and reviewed to assess completeness of Stratabound files and records. After initial spot checking of digital records supplied by Stratabound against source documents it was determined that a comprehensive review and validation of the entire digital dataset should be completed. Mercator completed such review, which consisted of checking all database entries including collar coordinates, down hole survey values, hole depths, lithocodes and assay entries against the original source hard copy or digital drill logs or assay documents. Stratabound staff provided technical input and support as required for this process. Any record inaccuracies revealed during the checking process were corrected and a new, validated Microsoft Access® database created that was considered acceptable for resource estimation purposes. Record checking was facilitated by, but not limited to, use of automated validation routines that detect data entry errors associated with sample records, drill hole depths, lithocodes intervals, and collar or down hole survey tables.*

*The validated database resulting from completion of the programs identified above is considered by the authors to be acceptable with respect to support of the current resource estimate.*

### 12.1.2 CAPTAIN SITE VISITS AND CORE REVIEW (CULLEN AND HARRINGTON, 2011)

#### *2008 SITE VISIT*

*On August 25th, 2008, [Mercator] visited the Captain property as well as Stratabound's Bathurst office and core logging facility. During that time discussions regarding the property were held with Mr. John Duncan, P. Geo., project manager for Stratabound, plus other members of the company's technical and professional staff. Drill cores from several representative diamond drill holes completed during* *the 2007-2008 Captain program were viewed, and three were selected for quarter core sampling and photography. The company's logging, sampling, security, record keeping and quality control/quality assurance procedures and protocols were discussed with staff.*

*During the core inspection and review process, several previously sampled core intervals representative of the copper grade range seen in the deposit were selected from drill holes CP07-01, CP07-03 and CP07-19 as part of the Mercator check sampling program. Stratabound staff carried out quarter core sampling of these archived half core samples under direction of the authors, who retained secure possession of the resulting bagged and labelled samples thereafter. A suite of 17 archived laboratory sample pulps from a thirteen separate drill holes from the 2007-2008 core program were also recovered from company archives to complement the program.*

*A site visit to the Captain deposit was also completed on August 26$^{th}$ by the authors, accompanied by Stratabound staff. At that time several outcrops of Nepisiguit Falls Formation volcanics were viewed along the access road (abandoned rail-bed) leading to the property and the area of 2007-2008 Captain deposit delineation drilling was accessed. A survey plan of Stratabound drill collars was available during the site visit and visual field checks were completed against hole numbers, locations and casing orientations against mapped database records. UTM (Zone 20, NAD 83) coordinates for several collars were obtained using a Garmin E-trek handheld Global Positioning System (GPS) instrument and these were recorded for later checking of database drill collar location coordinates.*

*Observations regarding character of forest cover, site elevations, surface drainage, road/drill pad features, exploration grid conditions and coordination, and general access road conditions were noted during the site visit (Figure 12.1 and Figure 12.2). Assessment report records for the property show that the on-site a core facility containing most pre-1980 Captain drill core had been destroyed by fire prior to the Stratabound's acquisition of the property and this eliminated use of such historic core in check or re-sampling programs. The old core facility was however, viewed during the site visit and representative samples of main core lithologies from the property were assembled from drill core fragments present at the site (Figure 12.3 and Figure 12.4).*

**Figure 12.1 View of Captain Deposit Area Showing 2010 Program Drilling Set-up**

**Figure 12.2 View of Captain Deposit Showing Low Relief and Forest Cover**

**Figure 12.3 Historic Drill Core at Site of Burned Core Facility on Captain Property**

**Figure 12.4 Historic Drill Core at Site of Burned Core Facility on Captain Property**

*2010 SITE VISIT*

*A second site visit was carried out by [Mercator] on October 15, 2010 in support of the updated resource estimation program described in this report. At that time Mr. John Duncan, P. Geo., and manager of Stratabound's programs at Captain, confirmed that field, security, core facility and sampling procedures and protocols adopted for the 2007-2008 drilling program were maintained for the five holes completed in 2010. Selected Stratabound drill cores from the 2010 program were subsequently reviewed at the New Brunswick government's Madran core storage facility and a suite of 16 check samples were collected for analysis by ALS Chemex in Vancouver, BC. After completion of the core inspection a site visit was carried out at which time all 2010 drill collars were located and a general inspection of site conditions was carried out. Full remediation of the 2010 drill sites had not been carried out but it was understood that completion of such work was being scheduled by Stratabound. A handheld GPS unit was used to collect collar coordinates for all five holes completed in 2010.*

*Results of the two site visit were consistent with those anticipated, based on consideration of prior exploration program reporting, discussions with Stratabound staff and review of Stratabound file materials.*

### *12.1.3 MERCATOR CHECK SAMPLE PROGRAM*

*During both the 2008 and 2010 site visits carried out by Mercator, sample pulps and/or quarter or half core samples were obtained for purposes of independent check sample analysis. In total, nine quarter core samples and 17 pulp split samples were obtained for the 2007-2008 program and results for these were considered acceptable by Cullen and Harrington (2009). Details of the 2007-2008 program appear in Appendix B. For purposes of the 2010 program, 16 half core samples were collected from drill holes CP10-26 through CP10-29 and these were submitted for preparation to ALS Chemex Ltd. in Sudbury, ON with final analysis carried out by ALS Chemex Ltd. in Vancouver, BC. A sample of certified reference material CDN HZ-2 plus a blank sample consisting of non-mineralized marble were included in the 2010 sample stream to for quality control and quality assurance purposes.*

*Sample intervals of archived drill core were selected and marked by Mercator and then photographed prior to being placed in labelled plastic bags for shipment to the laboratory. Core intervals taken for check sample purposes were clearly identified by explanatory tags secured in the core boxes for archival reference purposes. All core sampling was work was carried out at the NBDNR core storage facility at Madran, near Bathurst, under supervision of [Mercator], with assistance from Stratabound staff. Effort was made to obtain representative samples across the deposit grade range as represented in the 2010 drill holes. After standard crushing and pulverization, assay quality determinations for copper, lead, zinc and silver were obtained using four acid digestion with inductively coupled plasma atomic emission spectroscopy (ICP-AES) or AAS finish (OG62 Code) and gold was* *determined using fire assay pre-concentration of a 30 g split followed by ICP-AES finish (the ICP-21 Code) Specific gravity measurements were by pycnometer (GRA08b Code). The Mercator check sample dataset for cobalt was not available for consideration at the time of report preparation due to a laboratory instruction omission.*

*Mercator check sample results are compared to original Stratabound data set values in Figure 12.5 (copper) and Figure 12.6 (gold). Correlation coefficients between datasets for copper and gold are 0.99 and 0.90, respectively, indicating that copper levels between the sample pairs are in very close agreement while paired gold values show strong, but lesser, correlation. This trend is consistent with earlier comparisons of quarter core and duplicate split-sample populations and indicates that gold is less systematic in distribution than copper at the scale of individual half core splits. No obvious laboratory reporting bias is present between the copper and gold check sample data sets and results are interpreted as showing general confirmation of Stratabound's dataset mineralization levels.*

**Figure 12.5 Check Sample Results Copper**

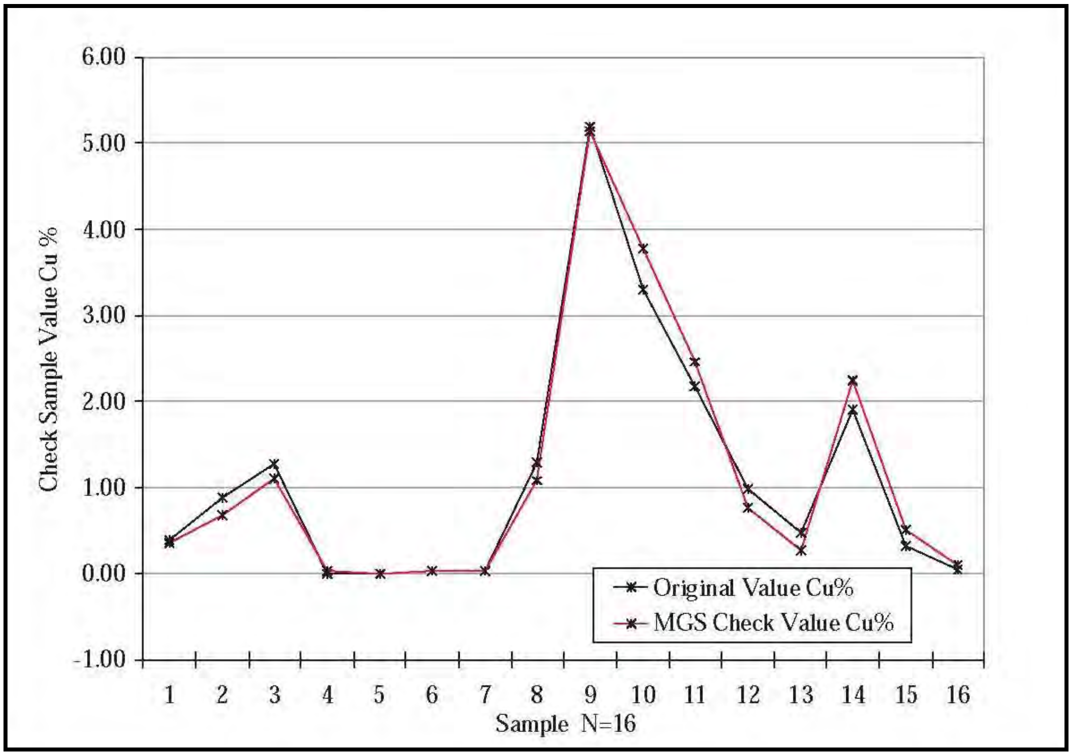
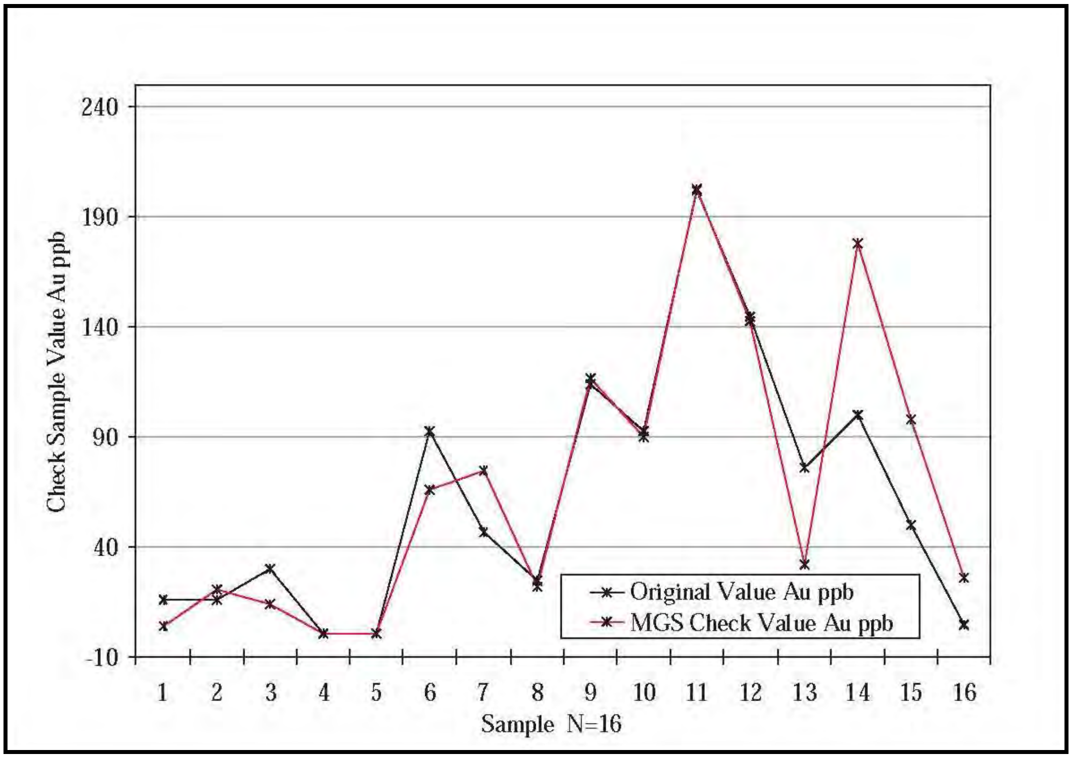


**Figure 12.6 Check Sample Results Gold**




## 12.2 CNE AND TAYLOR BROOK DATA VERIFICATION, 2010

### *12.2.1 TETRA TECH SITE VISITS, 2010*

Two site visits were conducted by Tetra Tech to the CNE and Taylor Brook properties. The first site visit was conducted by Mr. Daigle, Senior Geologist and Mr. McLaughlin, Project Manager, both employees of Tetra Tech, between October 18 and 19, 2010. Mssrs. Daigle and McLaughlin were accompanied during this site visit by Mr. Duncan, Consulting Project Geologist for Stratabound and Mr. Kevin Vienneau, Consulting Mining Engineer for Stratabound.

The second site visit was conducted between November 29 and 30, 2010, by Mssrs. Wenchang Ni, P.Eng., former Senior Engineer for Tetra Tech and Mr. McLaughlin. They were accompanied on this visit by Mr. Vienneau from Stratabound.

### *12.2.2 CORE LOGGING AND STORAGE FACILITY*

Stratabound's office is situated on the southwest end of Bathurst. The office has a large warehouse area that houses the core logging and storage facility (Figure 12.7 and Figure 12.8). During the various drill programs, Stratabound transfers their logged and sampled drill core to the provincial core storage in Madran as space becomes limited.

**Figure 12.7 Stratabound Core Logging Area**

**Figure 12.8 Stratabound Core Storage Area**

### 12.2.3 CHECK SAMPLE ANALYSIS

During Tetra Tech's site visit in October 2010, six samples were collected for check sample analysis. Three samples were collected from the 1988, 2009 and 2010 drill core from the CNE deposit and three samples were collected from the 1987 and 1995 drill core from the Taylor Brook deposit.

The CNE check samples from the 2009 and 2010 drill program were available at Stratabound's office and core logging facility. These samples were taken from the halved drill core and quartered by the rock saw on site, placed in a sample bag and sealed by Tetra Tech. The sample collected from the 1988 drill core was collected at the Madran core storage facility. The Taylor Brook check samples were collected at the Madran core storage facility where the historic drill core is kept.

The drill core was examined before samples were collected and matched to the drill logs; photos were taken of the intervals where the samples were collected, and; samples were placed in sample bags, marked with a sample number and sealed. The samples were kept in the care of Tetra Tech until the samples were sent to ALS Group (ALS), based in Sudbury, Ontario, for analysis.

At ALS, the samples were prepared by crushing the samples where 70% is reduced to less than 2 mm; a 250 g split is taken and pulverized to greater than 85% passing through 75 microns (ALS code: PREP-31). The samples were then dissolved by a four acid "near-total" digestion and analysed for 48 elements by inductively coupled plasma mass spectroscopy (ICP-MS) or ICP-AES (ALS code: ME-MS61a). The comparison of the original assay analysis and the check sample analysis are shown in Table 12.1 and Table 12.2 below for the CNE and Taylor Brook.

**Table 12.1 CNE Check Sample Comparison**

| Tetra Tech Sample No. | Stratabound Sample No. | Drillhole | Sample Interval From (m) | To (m) | Interval (m) |
|---|---|---|---|---|---|
| CNE 88-5 | 19-20 | CNE 88-5 | 19.00 | 20.00 | 1.00 |
| CNE 09-02 | CNE09-02-46.00 | CNE 09-02 | 46.00 | 46.50 | 0.50 |
| CNE 10-08 | CNE10-08-66.50 | CNE 10-08 | 66.50 | 67.00 | 0.50 |
| **Tetra Tech Sample No.** | | **Zn (ppm)** | **Pb (ppm)** | **Cu (ppm)** | **Ag (ppm)** |
| CNE 88-5 | | 73,800 | 30,000 | 230 | 121 |
| CNE 09-02 | | 3,910 | 110 | 19,450 | 13 |
| CNE 10-08 | | 62,600 | 35,200 | 260 | 155 |
| **Stratabound Sample No.** | | **Zn (ppm)** | **Pb (ppm)** | **Cu (ppm)** | **Ag (ppm)** |
| CNE 88-5 | | 84,800 | 52,000 | 200 | 62.3 |
| CNE 09-02 | | 3,900 | 32 | 19,100 | 17.6 |
| CNE 10-08 | | 60,000 | 37,000 | 309 | 333.6 |
| Difference (ppm) | | -11,000 | -22,000 | 30 | 58.7 |
| | | 10 | 78 | 350 | -4.6 |
| | | 2600 | -1800 | -49 | -178.6 |
| Difference (%) | | -15% | -73% | 13% | 49% |
| | | 0% | 71% | 2% | -35% |
| | | 4% | -5% | -19% | -115% |

**Table 12.2 Taylor Brook Check Sample Comparison**

| Tetra Tech Sample No. | Stratabound Sample No. | Drillhole | Sample Interval From (m) | To (m) | Interval (m) |
|---|---|---|---|---|---|
| TBD 95-5 | 7765 | TBD 95-5 | 38.70 | 39.70 | 1.00 |
| TBD 95-12 | 9775 | TBD 95-12 | 111.93 | 112.93 | 1.00 |
| CMO 87-2 | 193 | CMO 78-2 | 227' | 227.8' | 0.8' |
| **Tetra Tech Sample No.** | | **Zn (ppm)** | **Pb (ppm)** | **Cu (ppm)** | **Ag (ppm)** |
| TBD 95-5 | | 23000 | 8,820 | 310 | 60 |
| TBD 95-12 | | 23600 | 11,400 | 150 | 60 |
| CMO 87-2 | | 3010 | 1,080 | 420 | 9 |
| **Stratabound Sample No.** | | **Zn (ppm)** | **Pb (ppm)** | **Cu (ppm)** | **Ag (ppm)** |
| 7765 | | 32300 | 7,500 | 300 | 59.7 |
| 9775 | | 19900 | 7,000 | 100 | 51.8 |
| 193 | | 270 | 410 | 410 | 5.5 |
| Difference (ppm) | | -9300 | 1,320 | 10 | 0.3 |
| | | 3700 | 4,400 | 50 | 8.2 |
| | | 2740 | 670 | 10 | 3.5 |
| Difference (%) | | -40% | 15% | 3% | 0% |
| | | 16% | 39% | 33% | 14% |
| | | 91% | 62% | 2% | 39% |

Results from the check analysis are not intended to match the assay grades but to observe the relative abundance of metals within the drill core in comparison to the original assay results. The samples collected by Tetra Tech were taken from intervals with an abundance of lead and zinc sulphide minerals and are not considered representative of the deposits.

Several assay results from the check samples, notably from the historic drill core, appear to be lower than historically determined. This may be due to the outdoor storage of the historic drill core at the Madran core storage facility which may cause some sulphides to break down from the drill core. Overall, however, results for lead and zinc are elevated as expected within massive sulphide zones. Variation in assay analysis may also be due to inconsistent concentration of sulphides in the drill core or by differing assay analysis methods.

# 13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

This metallurgical review focuses on the CNE deposit, which is classified as a predominantly lead-zinc-silverore bodyand the Captain deposit which is copper-cobalt gold ore body. Both are hosted by Ordovician volcanics in the BMC, northern New Brunswick.

This report summarizes the various investigations and test work that has been undertaken. The summary recommendations are used to provide guidance on the chosen processing and milling option.

## 13.1 ORE COMPOSITION

Table 13.1 lists the resource and grades for the following: a bulk sample mined in 1990, the Mining Campaign #1 in 1991, and the Mining Campaign #2 in 1992. The ore was toll milled at the Heath Steel mill in three separate batches in 1990, 1991 and 1992. As part of the toll mill contract, the ore assays were cut by 5% prior to introduction to the mill operation.

**Table 13.1 CNE Resource and Grades**

| | | Assays | | | Assays (cut 5%) | | |
|---|---|---|---|---|---|---|---|
| Year | Dry Tonnes | Pb% | Zn% | Ag (g/t) | Pb% | Zn% | Ag (g/t) |
| 1990 | 11,444 | 5.57 | 12.00 | 173 | 5.29 | 11.40 | 164 |
| 1991 | 14,559 | 4.82 | 11.94 | 152 | 4.58 | 11.34 | 144 |
| 1992 | 13,619 | 2.83 | 7.65 | 98 | 2.69 | 7.27 | 93 |

With the exception of the Mining Campaign #2 results, the deposit can be described as a relatively high grade lead-zinc-silver deposit.

Table 13.2 lists the Captain head assays as determined by metallurgical lab work conducted by RPC labs on June 2, 2008.

**Table 13.2 Captain Head Assays**

| Head Assays | Assays | | | | | |
|---|---|---|---|---|---|---|
| | Fe% | Cu% | Zn% | Co% | Bi% | Ag (ppm) |
| Co Zone (Average) | 19.79 | 0.95 | 0.025 | 0.13 | 0.017 | 2.5 |
| Cu-Co Zone (Average) | 23.18 | 1.31 | 0.75 | 0.040 | 0.027 | 13 |

The respective metal grades for copper, zinc and cobalt are relatively low.

## 13.2 MINERALOGICAL INVESTIGATIONSAND TEST WORK

The following individual reports were reviewed and are summarized in the section.

### *13.2.1 MINERALOGICAL EXAMINATION OF TRENCH SAMPLE FROM THE CNE MASSIVE SULPHIDE DEPOSIT – JULIE MELLUISH – MARCH 1989*

This examination was carried out using semi-quantitative optical microprobe and image analysis techniques.

The ore contained significant amounts of sphalerite, pyrite and galena. Minor amounts of arsenopyrite were also observed (~0.5%). Chalcopyrite, cassiterite, and tetrahedrite were only present in trace amounts. The mineralization was described as “fairly coarse-grained massive sulphide primarily containing massive sphalerite, coarse equant pyrite, coarse irregular galena, and gangue with minor amounts of arsenopyrite, and trace amounts; of chalcopyrite, cassiterite, and argentian tetrahedrite. There were no gold-bearing minerals identified, and it was therefore determined that the sample contains 7.6% galena, 47.8%, sphalerite, 0.3% chalcopyrite, 33.1% pyrite, and 11.2% gangue mineral”.

The mineral grain size dataindicated a preliminary grind for metallurgical testing of 38 µm (400 mesh) or 53 µm (270 mesh) with a regrind to improve galena liberation.

### *13.2.2 MINERALOGY AND PETROLOGY OF SAMPLES FROM THE CNE DEPOSIT PART 1 – LECH LEWCZUK – RESEARCH AND PRODUCTIVITY COUNCIL – JULY 23, 1990*

The objective of this work was as follows:

- a search for covellite in the samples of massive sulphides
- definition of gold occurrences,describing relations between chalcopyrite and pyrite in the samples of massive sulphides
- identification of chert-like lithologies
- electron microprobe investigation of chlorites in selected samples.

This work was focussed on gold occurrences as a follow up from the previous report. The following text is from the report:

*Gold was not identified in the samples of massive sulphides (Lenses #2-4 and 3). It was also noted that distinctly zonal pyrite in samples from drill hole CNED 88-9 (12.4, 13.5, 22.5 and 24.3 m) contains abundant inclusions of heavy metals and/or heavy metal sulphides). Mineral compositions of these inclusions were not identified due to their small size (<1 micrometer).*

### 13.2.3 REPORT ON TEST WORK CARRIED OUT FOR STRATABOUND MINERALS AND BRUNSWICK MINING AND SMELTING- RICHARD A.J. MAY – AUGUST 20, 1990

This test work arose from an agreement between Stratabound and Heath Steel Mines (a joint venture of Brunswick Mining and Smelting Corporation Limited and Noranda Minerals Corporation) to mill a 12,000 t sample of Stratabound CNE lead-zinc-silver ore, whichtook place from September 4 to September 9, 1990.

A bulk sample of the CNE ore was sent to Brunswick Mining and Smelting, a representative portion of this sample was used for laboratory testing, to determine any potential recovery issues. Among the parameters investigated were the fineness of grind required, the use of $SO_2$ in the grind, and the use of aeration prior to flotation. Collector dosage, and the need for regrind, were also tested, in the absence of prior laboratory investigations. This test data was used as a baseline for the definition of flotation parameters including reagent dosages, grinding and flotation times.

The results indicated that the respective concentrate grades of lead and zinc achievable were 42% and 55%, respectively. The respective lead and zinc recoveries were 71% and 77% reporting to their concentrates. A major concern arising from this investigation was the magnitude of the zinc metal reporting to the lead concentrate(13%).

Lead grade and recoveries of the respective metals are relatively low, by today's (2011) standards. It would be reasonably expected that grades and recoveries would be improved in a current process.

### 13.2.4 FLOTATION OF A BULK SAMPLE OF STRATABOUND MINERAL'S CNE ORE AT HEATH STEELE MINES CONCENTRATOR [PROJECT1000 – RICHARD MAY AND ASSOCIATES, SEPTEMBER 1990]

The plant testing was directed at understanding the reasons for and reducing the zinc in the lead concentrate. It was also decided to bypass the copper circuit as the copper content of the ore was too low to justify attempting to produce a copper concentrate. Instead a bulk concentrate was produced, mainly because of the metal recovery challenges.

Several process changes were explored, which included an increased pH, using lime instead of soda ash, ceasing aeration, stopping $SO_2$ additions tothe copper-lead circuit and using less SIPX and more Aerofloat241 (a more specific galena collector).

None of these initiatives resulted in appreciable reduction in the zinc content of the lead concentrate. However,increased lime addition increasedthe amount of lead in the copper rougher tails.

Arising from this work, Richard May and Associates concluded that additional aeration was not required and $SO_2$ in the copper lead rougher was not worth pursuing.

Reagent optimization efforts for galena and sphalerite were not successful. Sodium sulphite ($Na_2SO_3$) at 200 g/t, sodium dichromate at 200 g/t combined with a pH increase to 11.1 with lime did not produce the desired result as a lead depressant.

Conditioning with $SO_2$ to pH 7.0 and 6.0 respectively, followed by SIPX addition, showed some positive results. Lead grade was improved by 2% and zinc grade improved by 3%. The results are shown in Table 13.3.

**Table 13.3 Results of Conditioning with $SO_2$ to pH 7.0 and 6.0, respectively**

| | | Assay (%) | | | Distribution (%) | | |
|---|---|---|---|---|---|---|---|
| | **Wt%** | **Pb** | **Zn** | **Fe** | **Pb** | **Zn** | **Fe** |
| **pH 7.0** | | | | | | | |
| Concentrate | 64.15 | 34.7 | 20.5 | 10.1 | 92.3 | 69.1 | 58.2 |
| Tail | 35.85 | 5.2 | 16.4 | 12.9 | 7.7 | 30.9 | 41.8 |
| **Total** | **100.0** | **24.2** | **19.0** | **11.1** | **100.0** | **100.0** | **100.0** |
| **pH 6.0** | | | | | | | |
| Concentrate | 59.8 | 36.8 | 17.7 | 11.0 | 93.3 | 57.7 | 62.0 |
| Tail | 40.2 | 3.9 | 19.3 | 10.1 | 6.7 | 42.3 | 38.0 |
| **Total** | **100.0** | **23.6** | **18.3** | **10.7** | **100.0** | **100.0** | **100.0** |

The report also concluded that the bulk concentrate could be eliminated by the removal of 11% of the zinc from the copper-lead cleaner concentrate, which would reduce the amount of the zinc in the lead and other related circuits by approximately 6% of the total zinc in the feed.

Two potential problems were highlighted:

- greater than 2% lead in the zinc concentrate
- approximately 15% zinc in the lead concentrate.

A mineralogical examination failed to improve the understanding of free sphalerite. This may indicate the existence of a polymorph of sphalerite such as wurtzitein the lead concentrate.

LABORATORY TEST WORK

During the 1990 plant trial, several process conditions were varied in an attempt to improve recovery and/or grade.

- pH was varied in try and affect a better separation of lead and zinc
- zanthate ratio to dithiophosphate in the collector was varied
- sodium sulphite was tested as an alternative to $SO_2$.but was not any better.

None of these tests lasted more than approximately three hours and were all inconclusive.

A further four tests were therefore undertaken in the Heath Steele laboratory. The first used MIBC and SIPX (a more selective xanthate) to improve the lead recovery rate, and the second tried to float the zinc ahead of the lead, but no benefit was found. The final two tests were more successful in using the $SO_2$ to reduce the pH in the copper cleaner circuit (from 7 to 6), better separations of the copper, lead and zinc were obtained, it was predicted that further testing might have improved metal separation to the extent that the “bulk” concentrate might be eliminated.

A sample density of mill feed of 3.1 was also determined.

As an overall conclusion and recommendation, Richard May and Associates, concludedthat the test work was limited and recommended that additional effort was required todefine the use of $SO_2$, which is still incomplete. Additional variation tests with talc were also required to define some remaining processingchallenges.

### *13.2.5 STRATABOUND MINERALS CORPORATION CNE DEPOSIT NEW BRUNSWICK FEASIBILITY STUDY- J. A.GOODWIN – DECEMBER 31, 1990, VOLUME 1 AND 2*

The Goodwin Feasibility report pre-dates NI 43-101 and consequently is not NI 43-101 compliant, and the use of terms such as possible and probable reserves is not compliant with current CIM and NI 43-101 standards. The data reported in this section uses the same terms used in the Goodwin Feasibility report.

The Goodwin report combined the findings of the various studies and test work on the CNE deposit that had been completed. Included in this report was a tabling of probable and possible reserves for lead zinc and silver, along with probable reserves of copper and gold.

**Table 13.4 CNE Lead/Zinc Ore Reserves**

| Type | Tonnes | Pb% | Zn% | Ag (oz/t) |
|---|---|---|---|---|
| Probable | 190,506 | 2.65 | 7.24 | 2.58 |
| Possible | 17,049 | 3.95 | 9.01 | 3.76 |
| **Total** | **207,555** | **2.76** | **7.38** | **2.68** |

**Table 13.5 CNE Copper/Gold Ore Reserves (South Zone only)**

| Type | Tonnes | Cu% | Au (oz/t) |
|---|---|---|---|
| Probable | 30,850.3 | 1.27 | 0.02 |
| **Total** | **30,850.3** | **1.27** | **0.02** |

PRELIMINARY TESTING

From the preliminary test work it was concluded that the best results were obtained from a sample ofthe following composition in Table 13.6.

**Table 13.6 Ore Composition**

| Pb% | Zn% | Ag% |
|---|---|---|
| 3.81 | 6.03 | 0.025 |

The resulting testsindicated 81.3% Pb, 43.1% Zn and 100% of the silver reporting tothe lead rougher concentrate, while 11.5% Pb and 54.9% Zn went to the zinc concentrate and 7.2% Pb and 2.0% Zn to the rougher tails.

The mineralogical report stated that gold was not recognized in any massive sulphide sample and presented no definitive occurrence path.

This study re-iterated the concern for the proportion of the zinc reporting to the lead concentrate. The conclusions and recommendations were in line with those of the previous authors.

### 13.2.6 MINING CAMPAIGN 2 – FLOTATION OF STRATABOUND MINERAL'S CNE ORE AT HEATH STEELE MINE CONCENTRATOR APRIL 6-8, 1992 AND MAY 30-JUNE 2, 1992 – RICHARD MAY AND ASSOCIATES

In this study, the overall metallurgical results from the bulk plant trial, in 1990, Mining Campaign#1 in 1991 and Mining Campaign #2 in 1992 were summarized.

**Table 13.7 Met Results for Bulk Sample ('90), mining Campaign #1 ('91) and Mining Campaign #2 ('92)**

| Mill Feed | Dry Tonnes | Assays | | | Assays (cut 5%) | | |
|---|---|---|---|---|---|---|---|
| | | Pb% | Zn% | Ag (g/t) | Pb% | Zn% | Ag (g/t) |
| 1990 | 11,444 | 5.57 | 12.00 | 173 | 5.29 | 11.40 | 164 |
| 1991 | 14,559 | 4.82 | 11.94 | 152 | 4.58 | 11.34 | 144 |
| 1992 | 13,619 | 2.83 | 7.65 | 98 | 2.69 | 7.29 | 93 |

The results indicate that the plant feed for Campaign #2 was significantly different to that used in the previous investigations.

For this work the bulk concentrate is included to provide information on ore distribution only. Based on the foregoing, Tetra Tech believes that with appropriate process enhancements this concentrate can be eliminated from any process step.

**Table 13.8 Lead Concentrate Results**

| Lead Concentrate | Dry Tonnes | Pb | | Ag | | Au |
|---|---|---|---|---|---|---|
| | | Grade (%) | Recovery (%) | Grade (g/t) | Recovery (%) | Grade (%) |
| 1990 | 824 | 46.0 | 64.5 | 1,456 | 66.0 | - |
| 1991 | 875 | 50.0 | 65.7 | 1,408 | 58.6 | - |
| 1992 | 442 | 50.5 | 61.0 | 1,510 | 53.0 | 3.13 |

The presence of gold in this deposit is an anomaly and requires further investigation on its occurrence before further consideration. Its occurrence is not yet sufficiently defined. Additionally, without any determination of deleterious elements no statement can be made regarding the extent of smelter penalties and treatment charges, however,according to statements from Stratabound, no smelter penalties were incurred in the 1990, 1991and 1992 mill runs at the Heath Steele mill.

**Table 13.9 Zinc Concentrate Results**

| Zinc Concentrate | Dry Tonnes | Zn | |
|---|---|---|---|
| | | Grade (%) | Recovery (%) |
| 1990 | 1,716 | 53.4 | 72.5 |
| 1991 | 2,492 | 54.8 | 82.7 |
| 1992 | 1,352 | 54.6 | 74.6 |

From this work it was demonstrated that acceptable grades and recoveries are achievable. Smelters typically find zinc grades above 53% acceptable. In the absence of a deleterious element assessment, a statement regarding the extent of smelter penalties and treatment charges cannot be made.

**Table 13.10 Bulk Concentrate Results**

| | | Pb | | Zn | | Ag | |
|---|---|---|---|---|---|---|---|
| **Lead Concentrate** | **Dry Tonnes** | **Grade (%)** | **Recovery (%)** | **Grade (%)** | **Recovery (%)** | **Grade (g/t)** | **Recovery (%)** |
| 1990 | 208 | 16.4 | 5.8 | 43.1 | 7.1 | 278 | 3.1 |
| 1991 | 291 | 19.2 | 8.4 | 32.5 | 5.7 | 424 | 5.9 |
| 1992 | 300 | 16.9 | 13.8 | 30.6 | 9.3 | 939 | 9.4 |

### *13.2.7 Mineral Beneficiation Tests on Stratabound Minerals Corp.'s Captain Cu-Co Deposit- Final Report: Reference No.: PET-J1710 (Rev 01) – Ross Gilders, Leo Cheong and Feng Gao – RPC – March 24, 2010*

The 2010 investigation carried out by RPC was directed at defining the Captain copper-cobalt and cobalt zone mineralization, target grind and scoping of copper-cobalt-zinc concentration by sequential rougher/cleaning flotation.

#### Test Work

Separate flotation circuits were designed for the samples from respective zones; the copper-cobalt zone and the cobalt zone.

**Figure 13.1 Open Circuit Flotation Flow Sheet as Tested Copper-Cobalt Zone**

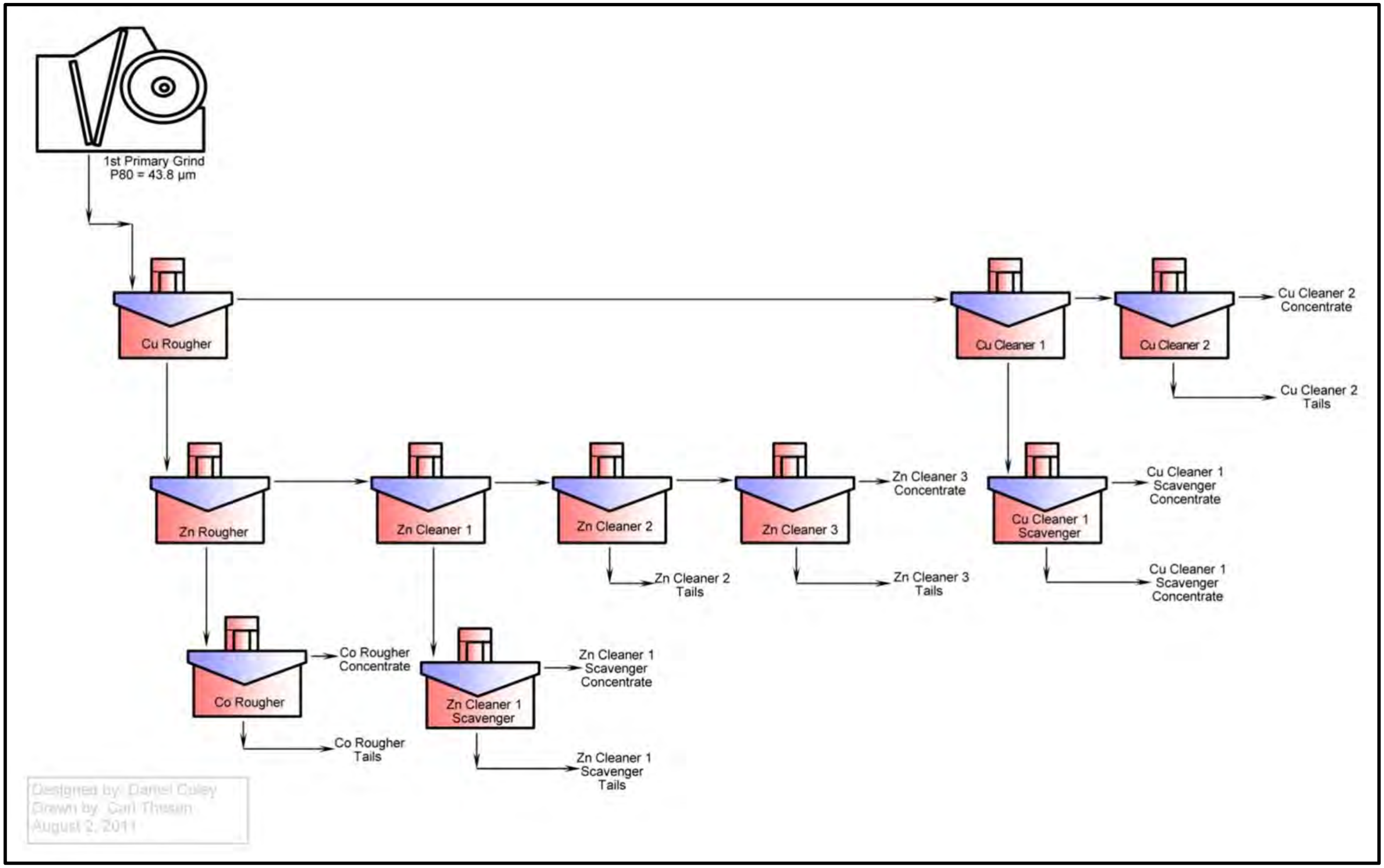
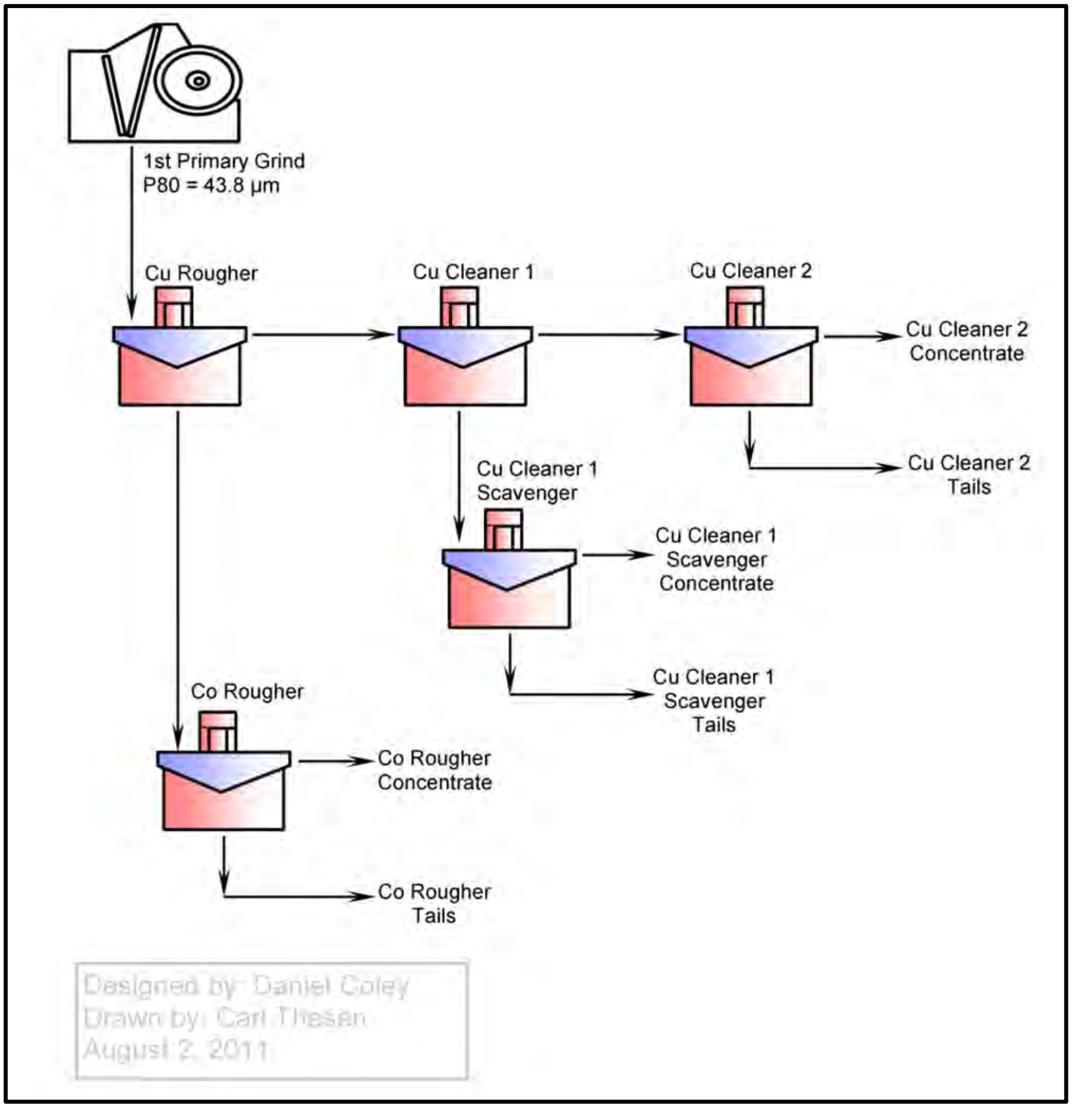


**Figure 13.2 Open Circuit Flotation Flow Sheet as Tested – Co Zone**




### MINERALIZATION

The sulphide minerals in both zones consist of ~30% pyrite, chalcopyrite, sphalerite with minor galena and bismuth/bismuthinite. Cobalt occurs as a solid solution in pyrite (an isomorphic substitution in a pyrite lattice).

### MINERAL RESULTS ON CO ROUGHER CONCENTRATE

Because the cobalt rougher concentrate was basically a pyrite concentrate with solid solution of cobalt, there is practically no hope in achieving a “concentrate grade” beyond 0.5%.

Most of the mineral examinations were semi quantitative and optical and revealed relatively low cobalt occurrences (0.041% (preliminary flotation tests)).

A number of challenges were faced during this work as follows:

- For the copper-cobalt zone, the particle size of rougher flotation feed did not have much effect on the copper and cobalt circuit, but the zinc rougher flotation performance benefits from finer size fractions. Additionally, two complete rougher flotation tests were performed at the optimum rougher conditions from the scoping tests.
- Zinc cleaner concentrate grade was poor at 27.7% with only ~71% recovery even after three stage cleaning.
- For the cobalt zone,additional cobalt extraction techniques were investigated using hot sulphuric acid, ferric sulphate, bacterial and high pressure acid leaching. The hot acid leaching achieved 55% cobalt extraction, while the autoclave achieved 100% of all metals(cobalt and iron).

#### SUMMARY FINDINGS

The cobalt rougher grade could not be concentrated beyond ~0.5% necessitating a two stage roast/leach to extract the cobalt. Autoclave leaching of the calcine did achieve 100% extraction of the cobalt, but at a prohibitive cost.

Rougher and cleaner flotation achieved copper grades of ~28% with recoveries of 94% for both copper-cobalt and cobalt zones.

The zinc in the copper-cobalt zone head grade was low~0.7%. Cleaning could not produce concentrate grades above27%.

## 13.3 DELETERIOUS ELEMENTS ASSESSMENT

A review of the 1992 lead, zinc and bulk concentrate smelter agreements from the Heath Steele smelter indicate a potential penalty of $2/t if arsenic exceeded 0.1%, or $0.50/t for each 0.01%that bismuth was over 0.015%. It has been reported verbally that no penalties were deducted; however, as emission limits are now more strictly enforced this should be reviewed and testwork should focus on minimising any deleterious materials. The 1992 concentrate was not assayed for arsenic (or if it was no results were published) but high arsenic assays have been reported in some of the CNE 2010 drill core.

Except for the concentrates of the Captain copper-cobalt and the cobalt zones (RPC 2010) the investigation of deleterious components in the concentrates was very limited.

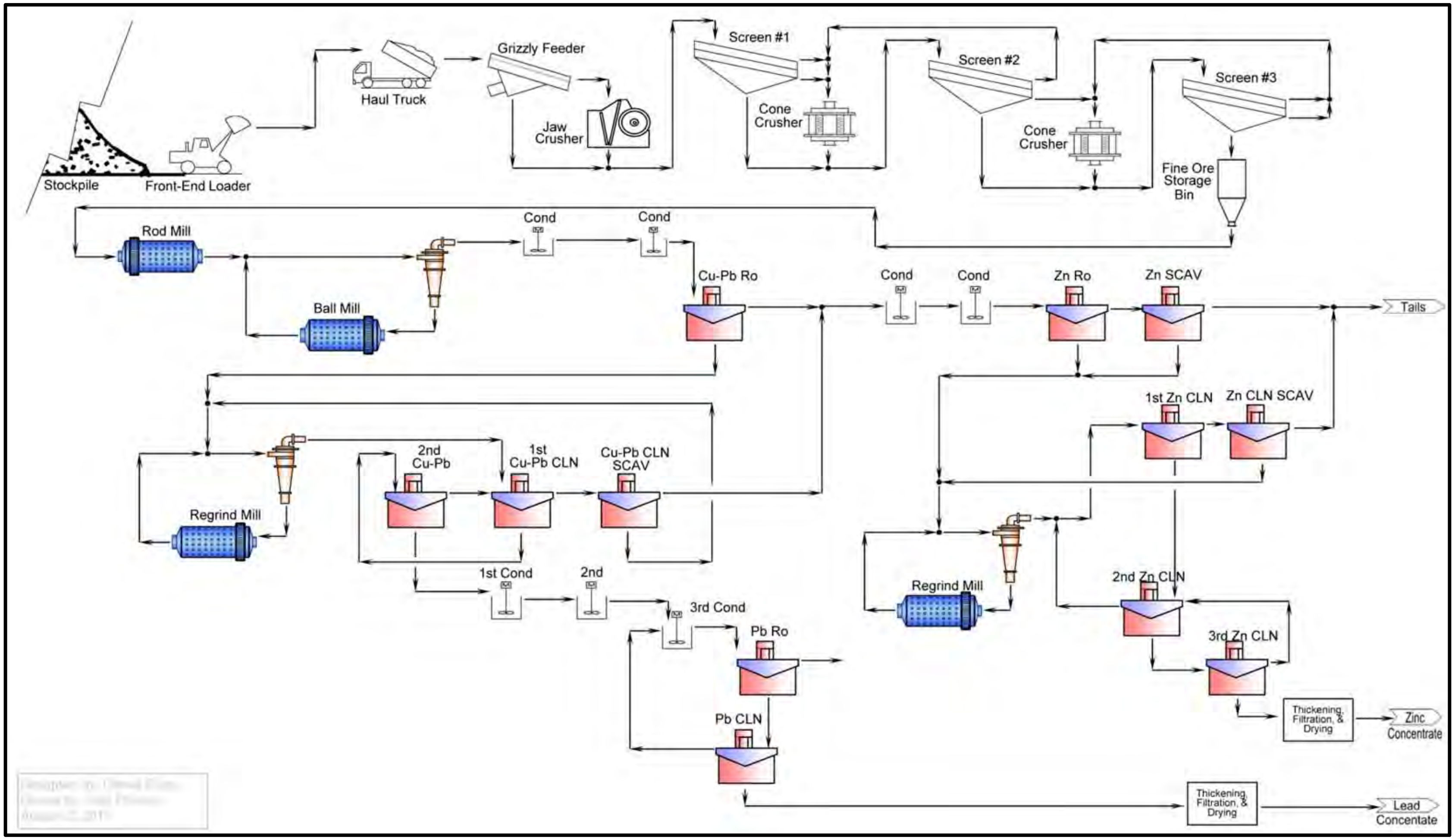
Since this original agreement was drafted in 1992, a number of changes to smelter terms have been introduced. The occurrence of antimony, silica ($SiO_2$), magnesia

(MgO) bismuth and mercury should all be verified as possible concentrate contaminants, and testwork undertaken to minimize the risk.

## 13.4 PROCESS FLOWSHEET DESCRIPTION

The flow sheet proposed is a modification of that for CNE Mining Campaign #2 with the exception of the bulk concentrate circuit. With the low recoveries and current metal prices the copper-cobalt and cobaltzone flotation circuitsrequired for the Captain deposit are not currently worth pursuing and have been excluded from the financial analysis. It is envisaged that with reagent enhancements, the proposed flow sheet will produce economical grades and recoveries.

**Figure 13.3 Needs Figure Caption**



### PROCESS DESCRIPTION

The ore is recovered from the stock pileby a front end loader and transported in a haul truck tothe primary crusher. The crushing circuit consists of a primary jaw crusher in open circuit followed by two stages of cone crushing in closed circuit (i.e., after crushing the product is screened and the oversize is returned tothe cone crushers). The crushed ore is then stored in fine ore bins.

From these fine ore bins, the ore is fed to a rod mill in open circuit followed bytwo stages of ball milling operated in closed circuit.

The ground pulp is aerated, and then conditioned in four banks of flotation cells and is then conditioned in two stages of conditioning. This conditioned pulp is fed to the copper lead rougher flotation circuit, consisting of four banks of flotation cells.

The copper-lead rougher tailings form part of the feed to the zinc rougher circuit while the copper-lead rougher concentrate is reground and thencleaned in two stages.

Further treatment of the copper-lead rougher tails is carried out in a scavenger circuit, from which a concentrate is recycled to the rougher circuit. Tailings from the copper-lead cleaner circuits are combined with the copper-lead rougher tailsand fed to the zinc circuit.

Feed to the zinc circuits is composed of the combined copper-lead rougher and cleaner scavenger tails. The zinc rougher feed is conditioned in two stages, and the recovered zinc rougher concentrate is refloated. The zinc scavenger tails pass to the plant tails, while the zinc scavenger concentrate is combined with the zinc rougher concentrate and sent to the zinc regrind mill.

Zinc rougher concentrate is cleaned in three stages, with the concentrates advancing to the next, and the tails returning to the previous stage. Tailings from the zinc first stage cleaner go to the regrind circuit and the tails join the plant tails.

All the final concentrates are thickened, filtered and dried prior to shipment.

## 13.5 ANALYSIS AND DISCUSSION

Based on these investigations carried out on the CNE deposit there is sufficient evidence to suggest that the process is technically viable.

While crushing and grindability work may be necessary for determining power consumption contribution, it has been established that the historical Heath Steele mill crushing circuit is sufficient for this ore at the proposed feed rates.

Additionally, the results from the Mining Campaign #2 flotation test work suggest that with a better collector/depressant selectivity, choice and optimization,metal recoveries and grades can be further improved.

However, caution is required in the tracking of deleterious elements from ore sampling to concentrate, since very little work has been carried out to date.

To properly account for and reconcile the ore grades, a sampling protocol is necessary. If the material is either blended or fed independently to a receiving mill, the integrity of the feed composition must be ensured. This will enhance any process predictability, trouble-shooting or optimization effort. Should the approach to feeding preparation be blending, a range of feed blends need to be investigated.

The Captain deposit (copper-cobalt and cobalt zones), based on the RPC test work, requires further investigation before any statement on viability. The reason for this is the relatively low mineral occurrence of the copper and zinc and the intricate inclusion of the cobalt in the pyrite. This should include a complete cost benefit analysis.

# 14.0 MINERAL RESOURCE ESTIMATES

## 14.1 CAPTAIN RESOURCE ESTIMATE

### 14.1.1 INTRODUCTION

The following description of the Captain resource estimate was taken from Cullen and Harrington (2011).

The resource estimate for the Captain deposit was estimated by Mercator in November 2010. Tetra Tech received this resource estimate and has reviewed it and included it within this NI 43-101 compliant resource report.

The effective date of the resource estimate for the Captain deposit is December 10, 2010.

### 14.1.2 GENERAL

*The definition of mineral resource and associated mineral resource categories used in this report are those recognized under National Instrument 43-101 and set out in the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves Definitions and Guidelines (the CIM Standards). Assumptions, metal threshold parameters, capping factors and deposit modeling methods associated with this estimate are discussed below in report sub-sections 14.1.3 through 14.1.18.*

### 14.1.3 GEOLOGICAL INTERPRETATION USED IN RESOURCE ESTIMATION

*As previously discussed in report section 7.2, for resource modeling purposes, the Captain deposit is considered to be a plunging zone of disseminated, stringer, sub-massive and massive sulphide mineralization measuring approximately 150 m in surface strike length, a maximum of 50 m in width and having a drilling-defined plunge extent of at least 400 m. In simplified perspective, it geometrically approximates an elongate, flattened cigar plunging south at 70° to 80° within a folded, north-northeast striking stratigraphic sequence that shows west dips of 70° to 85° in the immediate deposit area.*

*The deposit and host rocks are strongly deformed and quartz augen schist volcanic host lithologies are both foliated and strongly chloritized or sericitized. Evidence of sulphide remobilization along the penetrative regional foliation is present in many areas and transposition of bedding features, sulphide stringers and veins along this foliation is well represented in drill core. A well-developed halo of disseminated pyritic sulphide (5% to 15%) surrounds the deposit. Cu, Co and Au are the main* *metals of economic interest, but significant levels of Ag are also present locally. With local exceptions, Zn and Pb concentrations are generally low (<1.0%) and do not meet grade levels of primary economic interest at the deposit scale. However, these metals plus Ag may be recoverable as by-products of primary processing directed toward concentration of the deposit's Cu, Co and Au components. Strongest gold and silver grades generally follow spatial trends defined by highest copper grades, but highest cobalt levels are not typically coincident.*

### 14.1.4 METHODOLOGY OF RESOURCE ESTIMATION

#### OVERVIEW OF ESTIMATION PROCEDURE

*The revised Captain deposit mineral resource estimate is based on a three dimensional block model developed using Gemcom Surpac ® Version 6.1 modelling software. The model was developed from composited results of 5,583 drill core samples from a total of 30 separate drillholes completed by Stratabound during the company's 2007–2008 and 2010 drilling programs. Prior to deposit modelling, a complete set of vertical cross sections through the deposit were produced from the project database and used to develop manual geological interpretations through the deposit, at a nominal section spacing of 25 m. To identify grade domains within the deposit, analytical results from core sampling, composited to a support length of 4.0 m, were represented on the interpreted geological sections and correlated into zones of higher grade for which acceptable grade continuity could be demonstrated.*

*Based on inspection of higher grade Cu and Co areas within the deposit, two higher grade domains for each metal were developed. These were based on wireframe outlines developed from interpretation of the 4.0 m assay composite data presented on the deposit cross sections. Section-based wireframes for each metal, representing a minimum grade envelope of 0.60%/4.00 m for Cu and 0.05%/4.00 m for Co, were joined to develop unique three dimensional grade domain solids. A peripheral solid was also developed that encompassed the two Cu solids and two Co solids, thereby providing definition of a fifth grade domain comprised of uncorrelated higher grade intercepts as well as lower grade material present outside the higher grade solids. Au was found to typically occur in conjunction with elevated Cu levels and was evaluated accordingly. No lithological solids were developed for density assignment within the block model, since the relationship between sulphide concentration and enhanced metal grades was determined to be the best guide for assignment of block density values.*

*Composites of 1 m support length were developed from the raw core-sample assay data set for grade interpolation purposes. Experimental variograms were assessed at various lags for Cu and Co based on the 1 m composite sample populations and normal variograms were then developed using spherical models at a series of lag distances. Search ellipse orientation and range values for inverse distance squared grade interpolation within the block model were based on these results, in combination with results of the geological section interpretations.*

*Cu and Au values were interpolated into the block model using common parameters and Co was separately interpolated. Block model grade interpolation was constrained by the grade domain solids to prevent inappropriate smoothing of high grade values into adjacent lower grade areas. The peripheral grade solid reflecting Cu and Co distribution constituted the limiting constraint for block grade estimation. After interpolation of metal grades, a Cu % Equivalent (CuEQ%) value was calculated for each block using the formula Cu Equivalent % = Cu % + (Co % * 9.25). This factor was originally established by Cullen and Harrington (2009) and based on 3-year trailing average market metal pricing at the time of resource estimation. A subsequent review of 3-year metal pricing current to November 2010 showed that while market values for both metals had increased since reporting of the last resource estimate, relative pricing of the two had stayed at closely comparable levels. More specifically, a [CuEQ%] of Cu% + (Co% *9.28) was calculated in November 2010 based on updated 3-year trailing average pricing. In light of the relatively minor difference between the two calculated equivalence factors, the factor used by Cullen and Harrington (2009) was retained for the current estimate to provide consistency with earlier modeling and reporting. Since definitive metallurgical recovery attributes were not available for the deposit at the time of the current resource estimation program, recovery factors for equivalence calculation purposes were assumed to be 100%.*

#### DENSITY

*Unique density values were calculated for each resource block based using the regression equation "Block density (g/cm$^3$) = 2.85 g/cm$^3$ + [0.123 *(Block Cu% grade + Block Co% grade)]". This formula was developed from metal grade and density data returned for a total of 68 Stratabound samples from combined 2007-2008 and 2010 drilling programs. This equation differs from that used by Cullen and Harrington (2009) in the previous Captain resource estimate by use of summed Cu% and Co% values instead of the calculated [CuEQ%] parameter.*

*After assessment of model results, Measured, Indicated and Inferred category tonnage and grade resource estimates for Cu, Co, and Au were calculated at 0.60%, 0.80%, 1.00%, 1.20% and 1.40% [CuEQ%] threshold values. For validation purposes, estimate results were assessed on a cross sectional basis against the earlier geological and grade interpretation sections and the deposit was re-modeled for check purposes using Nearest Neighbour interpolation methods.*

*Report subsections 14.3.2 through 14.3.14 provide further information regarding the resource estimation procedures and parameters summarized above.*

### 14.1.5 DATA VALIDATION

*Results from 30 drillholes completed by Stratabound in the 2007-2008 and 2010 programs were compiled and imported into Gemcom Surpac ® Version 6.1 for resource estimation. Validation checks on overlapping intervals, inconsistent drillhole identifiers, improper lithological assignment, unreasonable assay value* *assignment, and missing interval data were performed. Checking of database analytical entries was also carried out against laboratory records supplied by Stratabound.*

*Historic drilling information was excluded from the current resource estimate but 58 historic drillholes are known to exist on the property, many of which do not intersect the current Captain deposit outline. Holes from the historic program are characterized by poorly constrained collar locations, limited analytical data sets and incomplete or non-existent core archives. In contrast, the Stratabound drillholes were found to provide reliable, survey controlled and complete coverage of the deposit area and to be supported by modern analytical data sets based on continuous sampling through mineralized intervals. In light of these factors, Mercator elected to base both the current and previous resource estimates solely on the higher quality Stratabound data set that had been verified through record validation, site visit and check sampling.*

#### 14.1.6 *DATA DOMAINS AND SOLIDS MODELLING*

*Down-hole assay composites at 4 m support length were developed to define grade domains used in block model grade interpolation. Mercator interpreted and developed three dimensional wireframe solid models of two higher grade Cu domains, based on a 0.60% Cu over 4 m cut-off, and two higher grade Co domains, based on a 0.05% Co over 4 m cut-off (Figure 14.1, Figure 14.2, and Figure 14.3). The Cu domains, identified as "Cu Domain A" and "Cu Domain B" for report purposes, occur as two tabular bodies of higher grade Cu mineralization that form the core of the deposit. Au values of economic interest are also generally found in these areas of higher Cu grades. The Cu domains in part coincide with two similarly oriented domains of higher grade Co mineralization that were termed "Co Domain A" and "Co Domain B" for report purposes. All four higher grade metal domains generally conform to the north striking, steeply west-dipping trends of major lithologic units in the deposit area and in combination define the steeply plunging, elongate zone of Captain mineralization.*

**Figure 14.1 High Grade Cu Domain A (Red) and Cu Domain B (Blue) Surpac Solid Models**

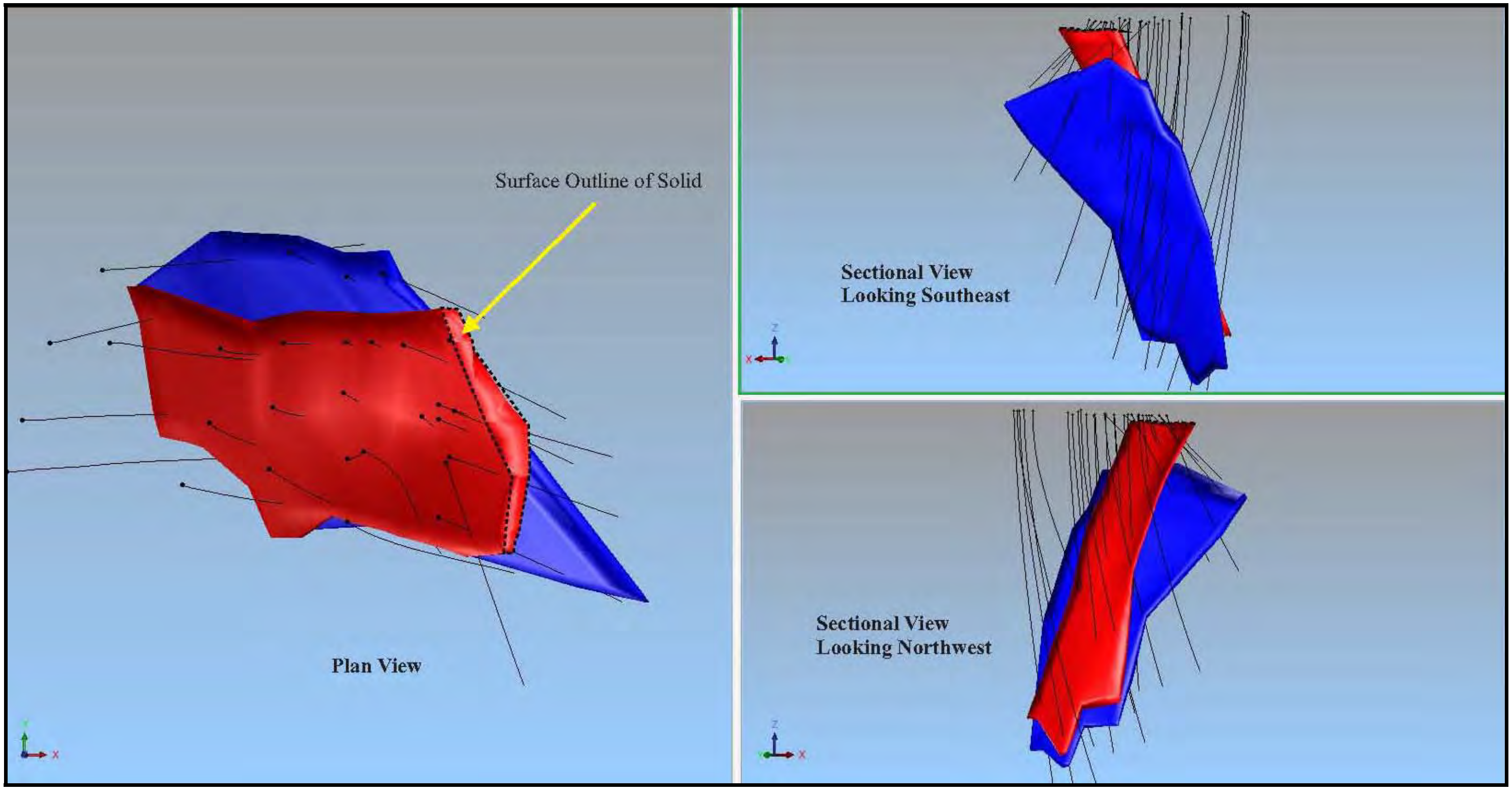


**Figure 14.2 High Grade Co Domain A (Gold) and Co Domain B (Blue) Surpac Solid Models**

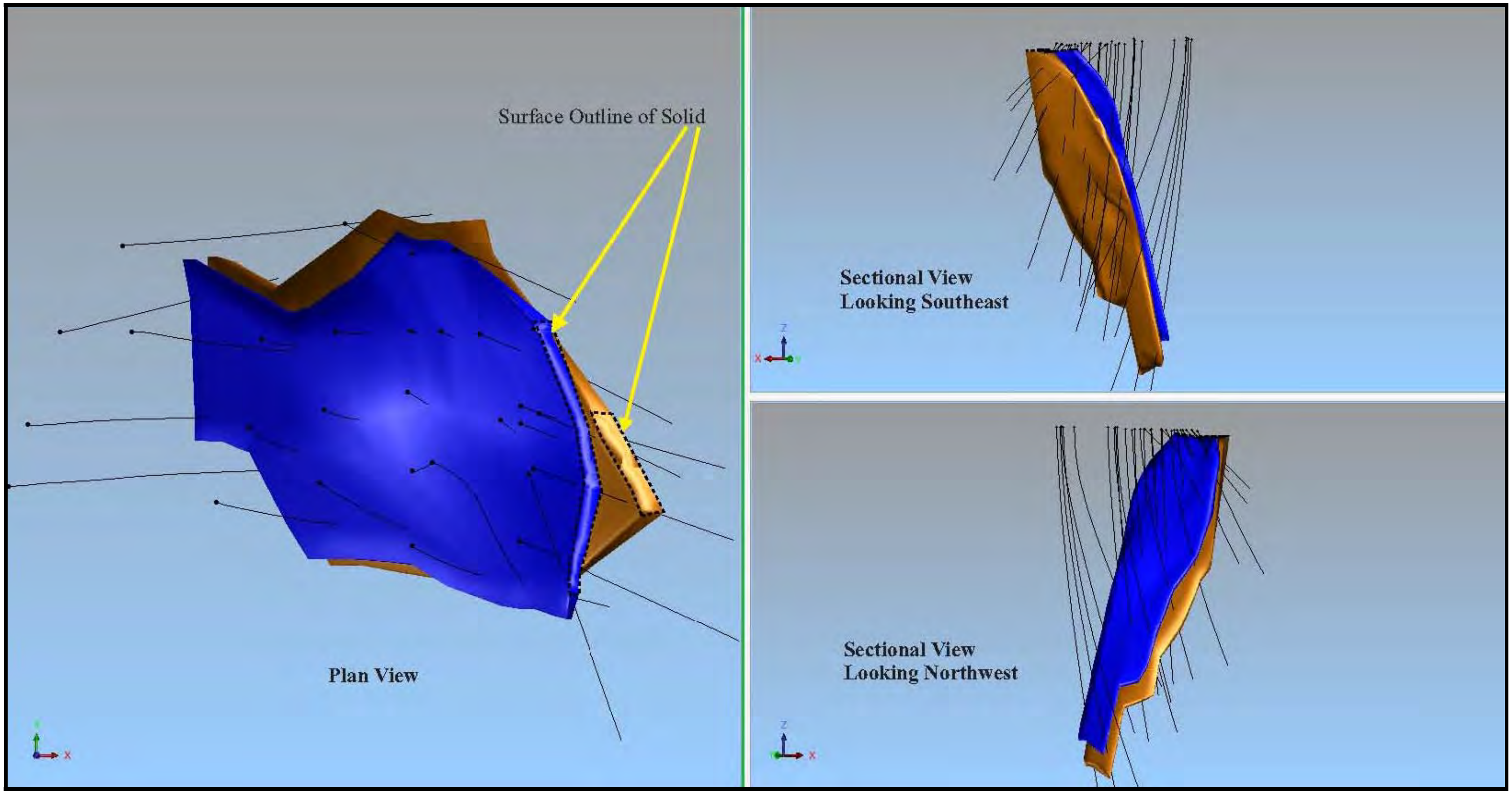
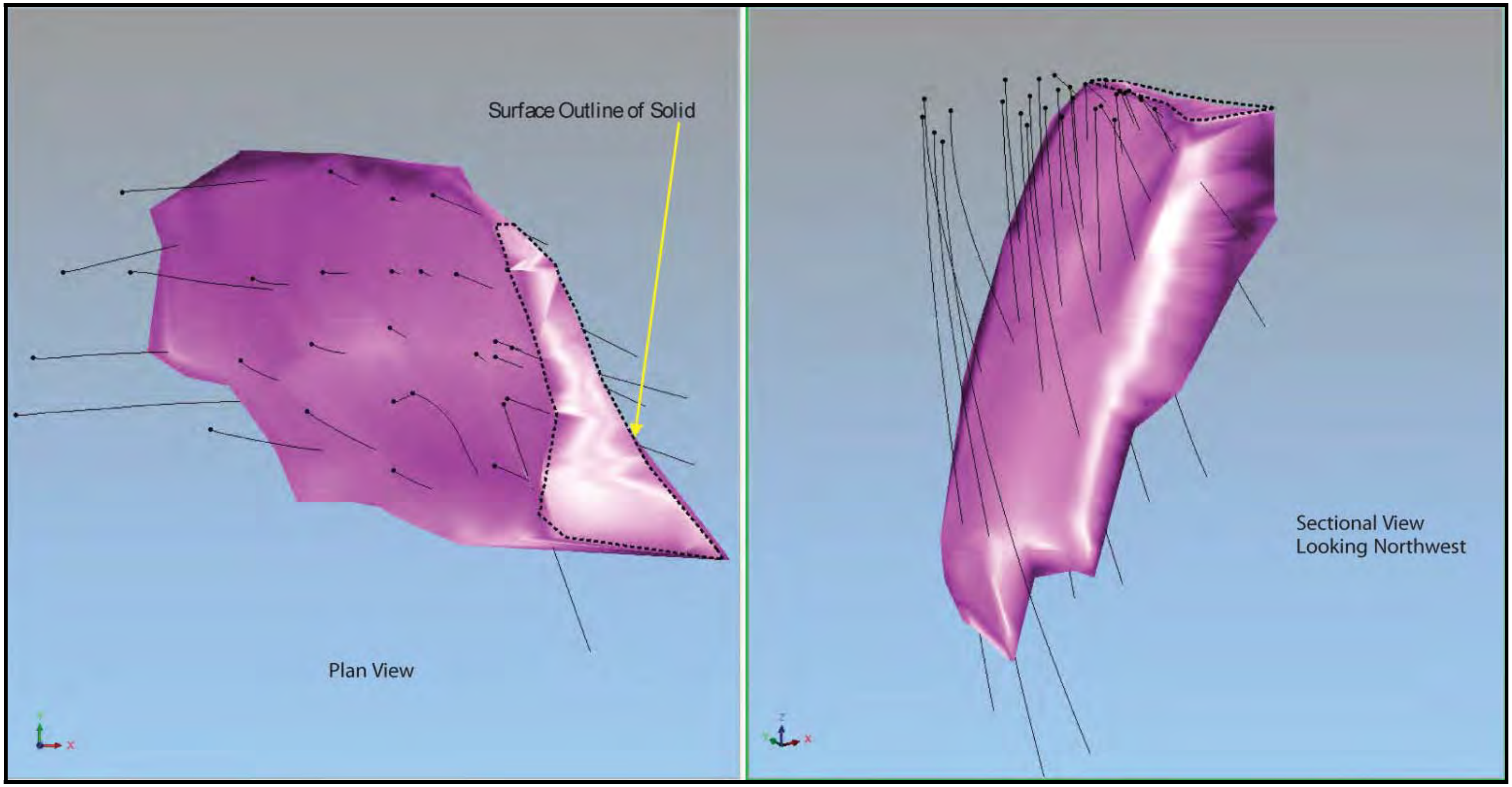


**Figure 14.3 Cu Eq Peripheral Envelope (Purple) Surpac Three Dimensional Solid Model**



*Solids were truncated up-dip by an interpreted base of overburden digital terrain model and projected 50 m down-dip from last drillhole intersections occurring on 25 m spaced cross sections. In situations where dip correlation was limited by drillholes that were non-mineralized or below acceptable grade parameters, solids were projected half the distance between the last intersection and the constraining drillhole. Solid strike extents near surface were closed at 12.5 m or half the distance to the nearest non-mineralized drillhole from the midpoint of the last drillhole intersection. Solid strike extents at deeper levels were projected up to 50 m from the endpoints of the last intersections in areas where mineralized continuity was otherwise demonstrated.*

*Contact plots were developed to validate the basis for creating higher grade metal domain solids. These were based on 1 m down-hole assay composites and graphically characterize distance-grade relationships that mark contacts between higher grade model solids and surrounding lower grade domains. Composites were binned in 1 m distance increments away from the specified domain contact, with mean metal grade and composite population size (n) plotted for comparison. In this form of contact plot, distances from all areas of the solid are assembled, and reflect both up-hole and down-hole contacts of the specified domain. For example, in Figure 14.4, "Bin 1" in the contact plot shows the mean grade of all the composites within Cu Domain A that are 1 m away from both the upper and lower contacts. In contrast, "Bin -1" shows the mean grade of all composites 1 meter away from the upper and lower contacts that are external to the domain and, therefore represent samples taken in the adjacent, lower grade Peripheral Domain. Figure 14.5 to Figure 14.7 present contact plots for the remaining three high grade domains. These figures are based on data from 25 of the 30 drillholes used in the current resource estimate, but review of grade distribution in the remaining five holes did not reveal any conflict with the trends portrayed. In all plots, breaks in metal grade are apparent across the solid limits, indicating that grade restriction should be applied to prevent inappropriate spatial smoothing of metal values during interpolation.*

### 14.1.7 *DRILL CORE ASSAY COMPOSITES AND STATISTICS*

*The drill core assay dataset used in the resource estimate contains 5583 core sample records, exclusive of quality control and quality assurance samples. Approximately 73% of the samples measured 0.5 m or less in length with most remaining samples being 1 m in length. A few sample length outliers are also present and these range between 0.20 m and 0.50 m in length and between 1 m and 3 m in length. Tabulated results of a rank and percentile analysis and a frequency distribution plot for all sample lengths are included in Appendix C.*

**Figure 14.4 Contact Plot for Cu Domain A**

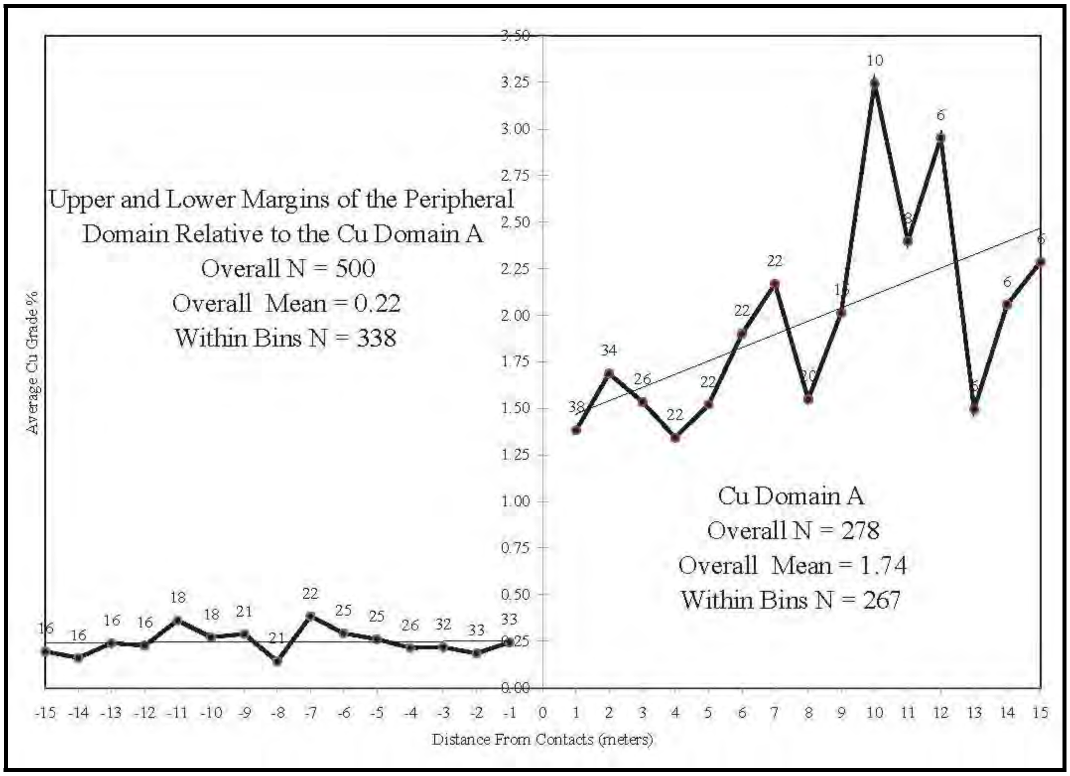



**Figure 14.5 Contact Plot for Cu Domain B**

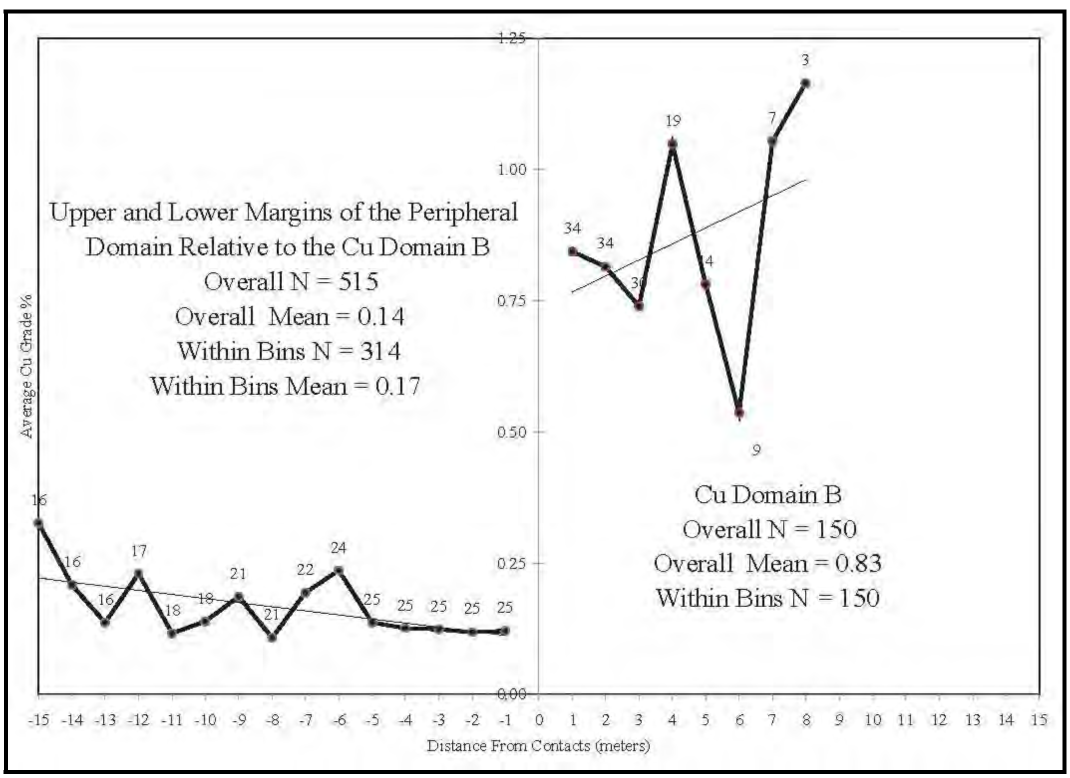


**Figure 14.6 Contact Plot for Co Domain A**

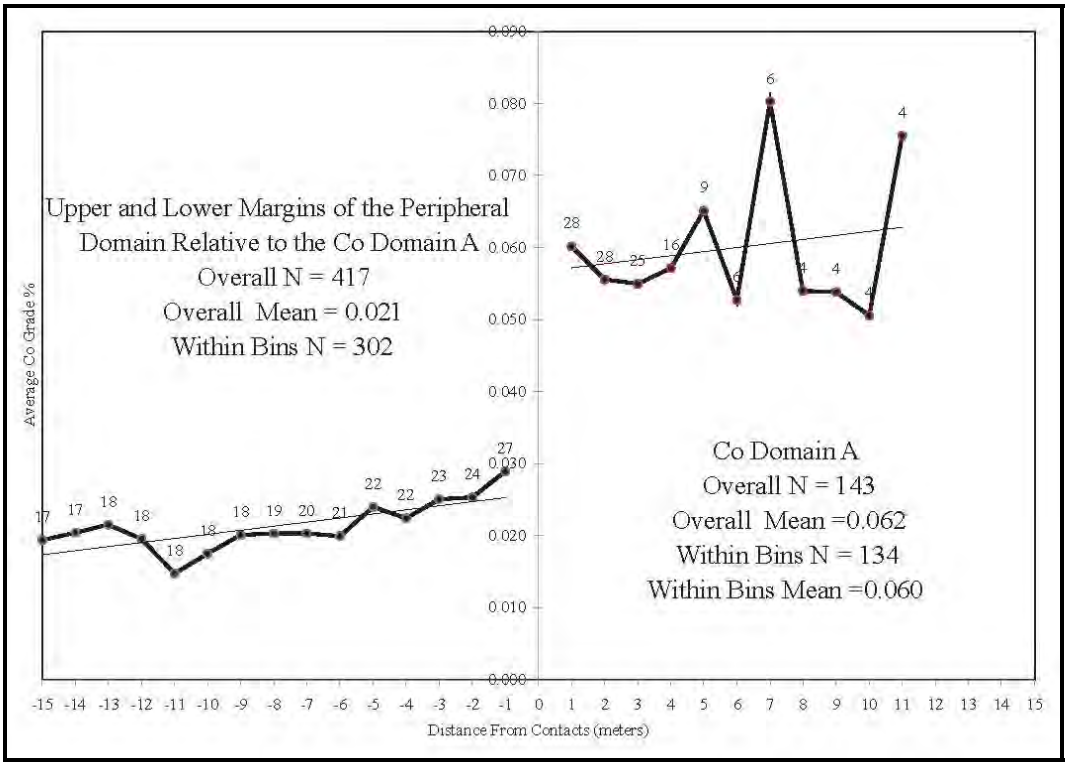
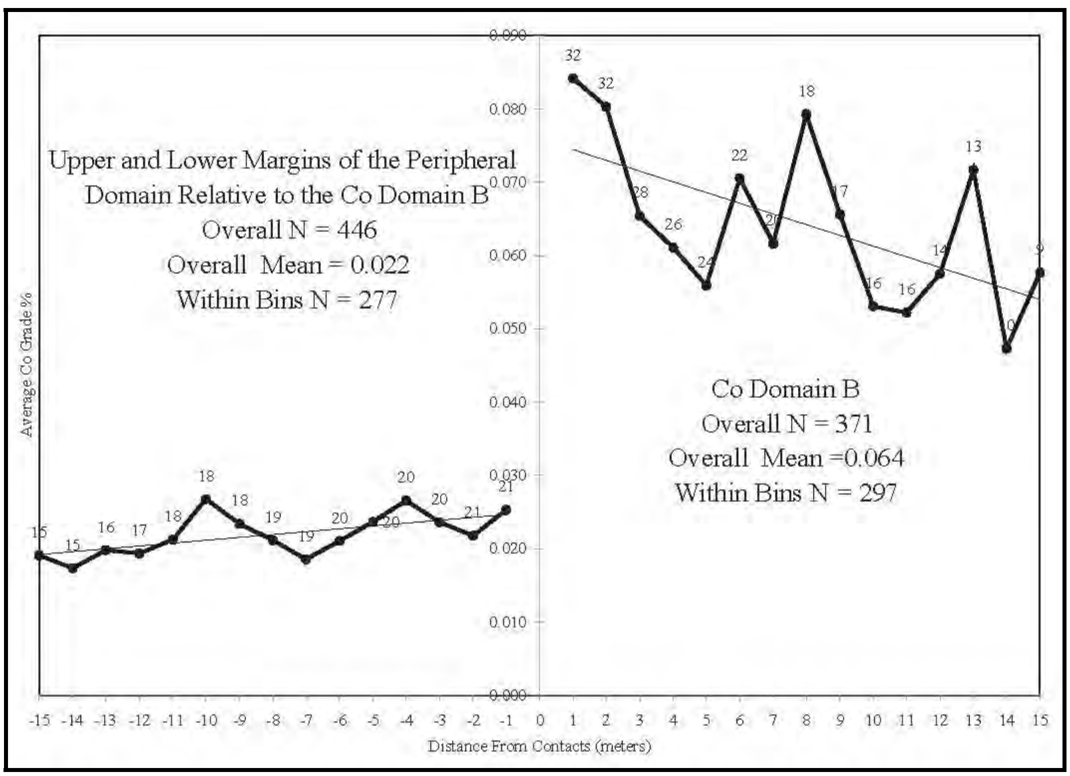



**Figure 14.7 Contact Plot for Co Domain B**



*A 1 m composite length was chosen for resource estimation purposes, which is twice the length of the majority of supporting core samples in the dataset. A unique set of 1 m down hole assay composites was created for each of the four higher grade metal domains, with sample data constrained inside each of the respective grade solids. A fifth set of assay composites was produced for drillhole intervals within the Peripheral Domain solid, with no internal limitations applied based on intersections with the higher grade domain solids. No lithological constraints were imposed on down-hole compositing, since core observations and contact plots showed mineralization to commonly cross lithologic contacts at the scale of one or more composite sample lengths and to be more accurately defined by the assay-based grade solids.*

*Descriptive statistics were calculated for the 1 m assay composite datasets prepared for each of the five grade interpolation domains: these being Cu Domain A, Cu Domain B, Co Domain A, and Co Domain B and the Peripheral Domain. Results are presented in Table 14.1. Cu Domain A composites are highest, with a mean grade of 1.53 % Cu and the smaller Cu Domain B population has a lower mean grade of 0.83% Cu. Co Domain A and Co Domain B populations show similar mean grades of 0.060% Co and 0.061% Co respectively. Cumulative frequency plots for 1 m assay composites of Cu, Co, and Au for all domains appear in Appendix C.*

**Table 14.1 Cu, Co, and Au Statistics for 1 m Composites**

| | All Deposit Domains Including Peripheral | | | Cu Domain A | | Cu Domain B | | Co Domain A | Co Domain B |
|---|---|---|---|---|---|---|---|---|---|
| **Parameter** | **Cu%** | **Co%** | **Au (g/t)** | **Cu%** | **Au (g/t)** | **Cu%** | **Au (g/t)** | **Co%** | **Co%** |
| Number of Samples | 1642 | 1642 | 1642 | 343 | 343 | 206 | 206 | 202 | 443 |
| Mean | 0.524 | 0.035 | 0.114 | 1.530 | 0.274 | 0.830 | 0.141 | 0.060 | 0.061 |
| Maximum | 8.750 | 0.321 | 1.415 | 8.750 | 1.415 | 7.450 | 1.070 | 0.293 | 0.321 |
| Minimum | 0.001 | 0.001 | 0.010 | 0.001 | 0.005 | 0.001 | 0.010 | 0.009 | 0.002 |
| Variance | 0.882 | 0.001 | 0.029 | 1.887 | 0.066 | 1.006 | 0.025 | 0.001 | 0.002 |
| Standard Deviation | 0.939 | 0.034 | 0.170 | 1.374 | 0.256 | 1.003 | 0.159 | 0.032 | 0.042 |
| Coefficient of Variation | 1.793 | 0.956 | 1.496 | 0.896 | 0.935 | 1.208 | 1.130 | 0.542 | 0.687 |

### 14.1.8 HIGH GRADE CAPPING OF ASSAY COMPOSITE VALUES

*A limited number of high Cu and Co grades are present in the 1 m assay composite datasets and these can be identified in the cumulative frequency curves, probability plots and grade distribution histograms included in Appendix D. The maximum Cu grade for a 1 m composite is 8.75%, the maximum Co grade for a 1 m composite is 0.321% and the maximum Au grade for a 1 m composite is 1.415 g/t. All of these are geologically reasonable values that can be expected to occur over measurable and correlatable areas within a volcanogenic massive sulphide deposit's sulphide* *stockwork system. This point is illustrated by the Figure 14.8 sectional view through Cu Domain A that shows hole to hole dip correlation of Cu values exceeding 4%. Additionally, relatively low variation coefficients for Cu, Co and Au results characterize the respective data sets.*

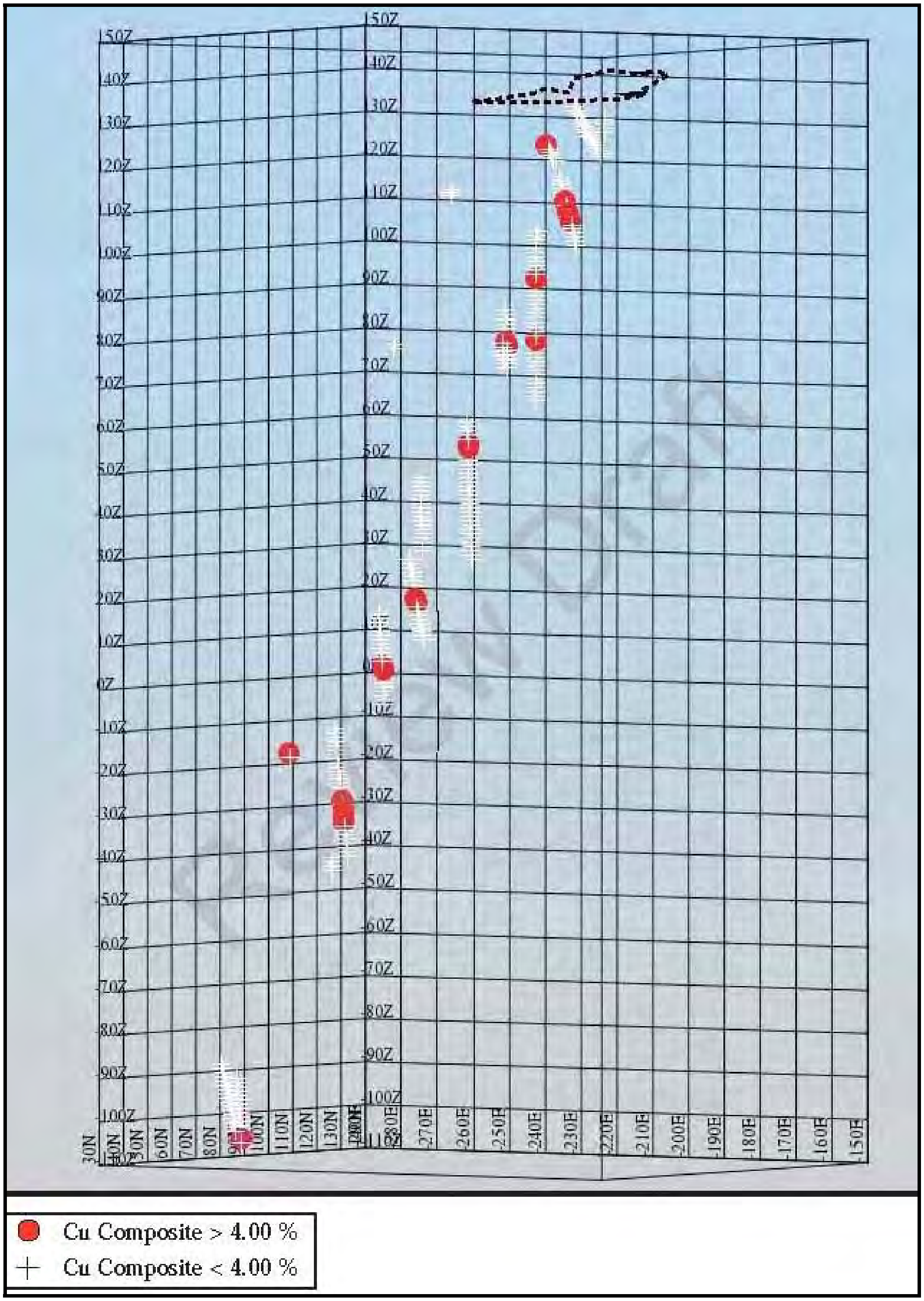
*After considering the above, and recognizing that interpolation of high metal values within the block model is strongly constrained by the high grade Cu and Co grade domain solids, it was determined that no capping of assay composite values of Cu, Co and Au was necessary.*

### 14.1.9 *Calculation of Cu Equivalent and Cut-off Values*

*A Cu equivalent value was calculated by Cullen and Harrington (2009) for the previous resource estimate block model and was based on relative market pricing for Cu and Co for a 3-year period ending in September 2008. Metal recovery factors of 100% were incorporated, since definitive results of metallurgical work on the deposit had not been reported at the effective date of that report. Three year trailing average metal prices were used to define the equivalence equation at that time as Cu Equivalent % = Cu % + (9.25 x Co %).*

*For current resource estimation purposes, the earlier Cu Equivalent equation was reviewed in light of updated 3-year trailing average metal pricing figures calculated from November 2010. This resulted in an equation of [CuEQ%] = Cu % + (9.28 x Co %) based on a Cu average of $ 2.95/lb and Co average of $ 27.40/lb. While market values for both metals have increased since reporting of the last resource estimate, relative pricing of the two has remained closely comparable. Based on this similarity, the equivalency equation used by Cullen and Harrington (2009) was retained for the current estimate to provide consistency with earlier modeling and resource reporting. Additionally, since definitive metallurgical recovery attributes had still not been established at the current resource effective date, Cu and Co recovery factors for Cu Equivalent calculation purposes were assumed to be 100%, as was the case for the previous resource estimate.*

**Figure 14.8 Sectional View of Cu Domain A with High Grade Cu (>4.0%) Correlation**



*Cu equivalent values were deemed necessary to provide an assay-based parameter upon which grade threshold or cut-off values representing contributions from both of these widely distributed metals could be based. While a range of local metal value contributions is represented within the block model data set, Cu predominates on the total deposit scale.*

*Cut-off values were selected to support modeling of the deposit above the 0.60% Cu Equivalent level, but this is not an economic cut-off value based on a project economic analysis. Such analysis has not been completed to date for the Captain deposit. However, geometry and grade distribution characteristics of the deposit, combined with very thin overburden cover, are interpreted as indicating that any future economic assessment will include study of open pit development potential. The 0.60% Cu Equivalent cut-off provides a reasonable base case for such assessment. Higher cut-off values used in the current resource statement will similarly be useful in assessing underground development potential.*

### 14.1.10 *VARIOGRAPHY*

#### INTRODUCTION

*Variography described below was developed for purposes of the earlier resource estimate block model reported by Cullen and Harrington (2009). This model incorporated assay data for 25 of the 30 drillholes now available for deposit assessment, and on that basis, is also considered representative of the expanded project data set used for the current study. Results of variography developed for the earlier block model were directly incorporated in the current resource estimation program and a detailed description of the earlier work by Cullen and Harrington (2009) appears below.*

#### METHODOLOGY

*Manually derived models of geology and grade trends provided definition of metal distribution trends that broadly conform to the north-northwest strike and steep west-southwest dip exhibited by major lithologic contacts in the deposit area. To assess spatial aspects of grade distribution within this recognized orientation corridor, Cullen and Harrington (2009) calculated experimental variograms for Cu % and Co % at various lags and 1 m down-hole assay composite support within the deposit's Peripheral Domain. This provided access to the total population of assay composites that defined the deposit in the previous resource block model. All experimental variograms were assessed at 10° and 11.25° increments within a plane corresponding to a steep dip of 65° to 70°towards 239° Azimuth (or 149° strike with a 65° to 70°dip). Each of the grade domains was similarly assessed, but these did individually provide consistently useful results, possibly due to the relatively small sample composite populations that define each domain.*

*Promising experimental variogram results for Cu were returned from the Peripheral Domain dataset (total deposit - Figure 14.9 and Figure 14.10) and application of* *spherical model attributes provided definition of a primary axis (major axis) of continuity with a range of 125 m along azimuth 283° with a plunge of 57° to the west. Spherical models were developed for the Cu % primary axis at lags of 10 m and 15 m. A secondary axis of continuity (semi-major axis) was identified with a range of 50 m at azimuth 153° and having a 22° plunge to the southeast. Acceptable spherical models were developed for the Cu % secondary continuity axis at lags of 10 m and 15 m (Figure 14.11 and Figure 14.12). A well-defined third (minor) axis of continuity could not be resolved, but a down-hole experimental variogram for Cu % supports a continuity range of 16 m.*

*Experimental variograms developed for Co assay composites identified a primary continuity axis with a range of 110 m at azimuth 265$^o$ with a 68$^o$ plunge to the west. Spherical models were then developed for the primary axis at lags of 15 m and 20 m (Figure 14.13 and Figure 14.140 A sub-horizontal secondary axis of continuity was identified with a range of 75 m along azimuth of 175° and spherical models were developed for the secondary axis at lags of 15 m and 20 m (Figure 14.15). The third (minor) axis of continuity could not be resolved, but a down-hole experimental variogram for Co% shows continuity to a range of 25 m (Figure 14.16).*

### DISCUSSION OF RESULTS

*Experimental variograms for both Cu and Co assay composites for the total deposit, as defined by the Peripheral Domain solid, show that the primary direction of grade continuity for both metals occurs in the down dip direction with ranges of up to 125 m. The secondary direction of continuity is oriented approximately along strike with ranges between 50 m and 75 m and low angle to horizontal plunges. These results were reviewed in light of the manual grade interpretation sections prepared earlier and were found to be geologically reasonable and acceptable. On this basis, ellipsoids for grade interpolation were oriented along the dip and strike directions of respective grade interpolation domains. A down-dip range of 125 m and along-strike range of 50 m were used for all grade interpolations except those carried out for Co Domain B and a small subzone of Cu Domain A, where modified values were assigned. The minor axis of interpolation ellipsoids, with the same exceptions noted above, was assigned a range of 12.5 m after consideration of down hole variogram model ranges and their relative orientations with respect to mineralized zone true thicknesses.*

**Figure 14.9 Variogram Model for Primary Continuity Axis: 10 m Lag – Cu%**

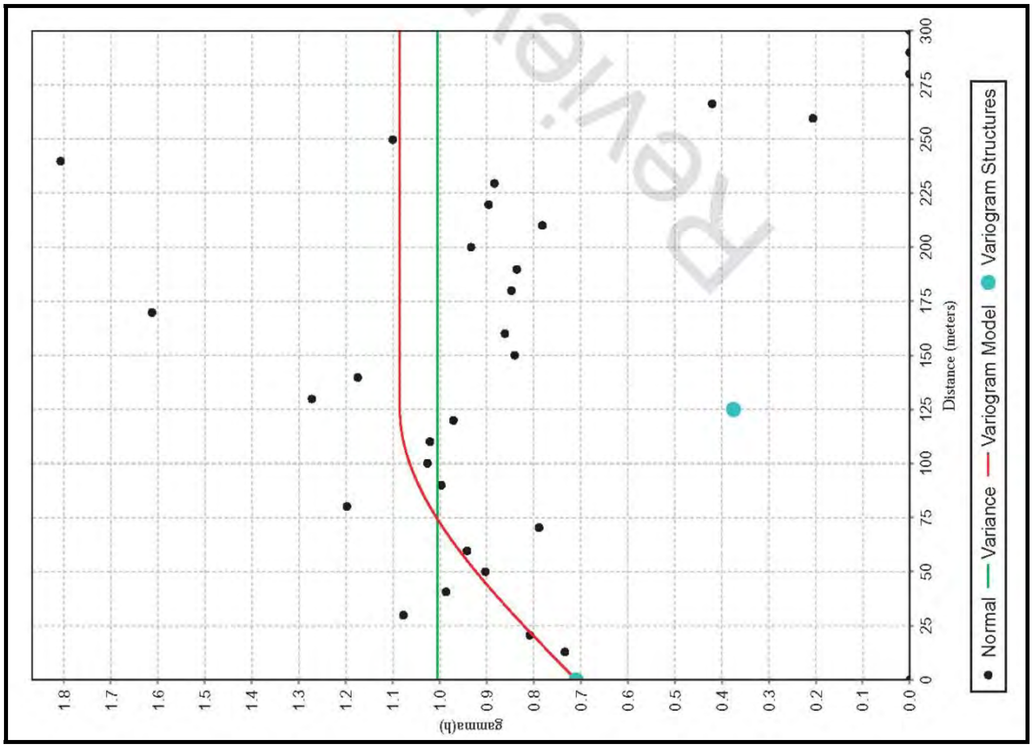



**Figure 14.10 Variogram Model for Primary Continuity Axis: 15 m Lag – Cu%**

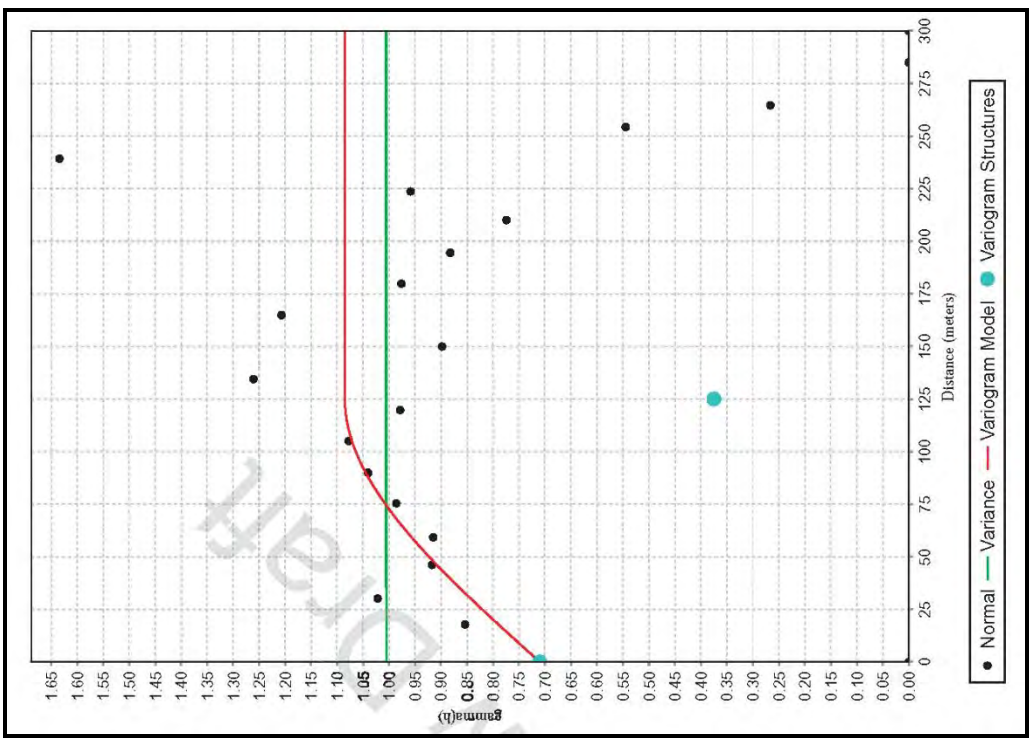


**Figure 14.11 Variogram Model for Secondary Continuity Axis: 10 m Lag – Cu%**

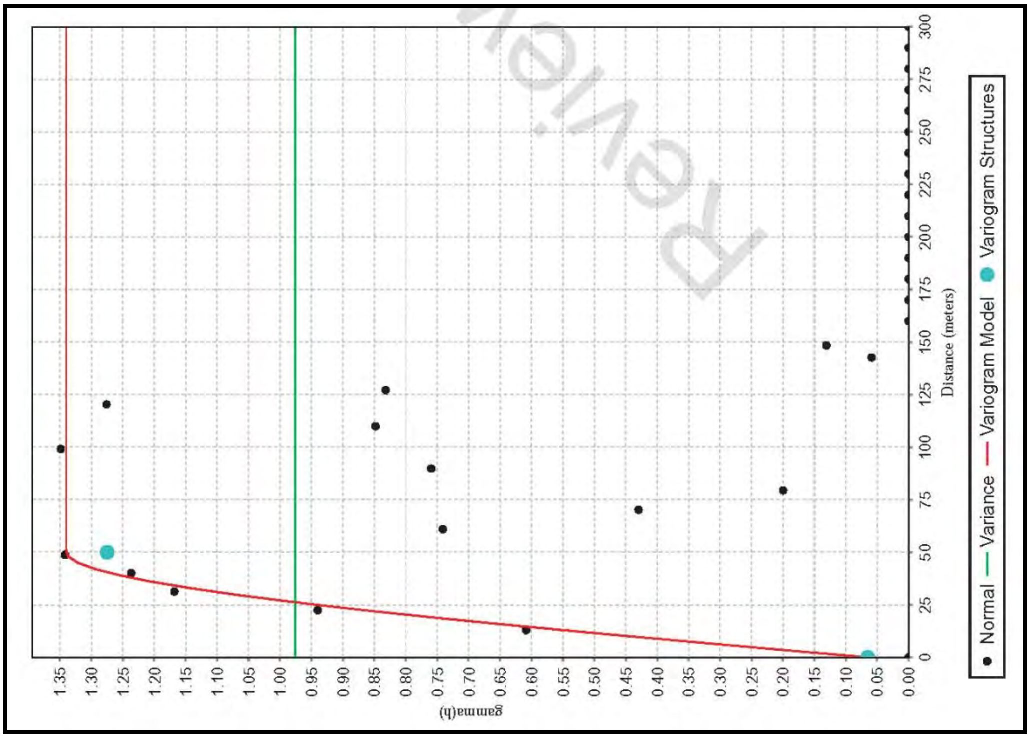



**Figure 14.12 Variogram Model for Secondary Continuity Axis: 15 m Lag – Cu%**

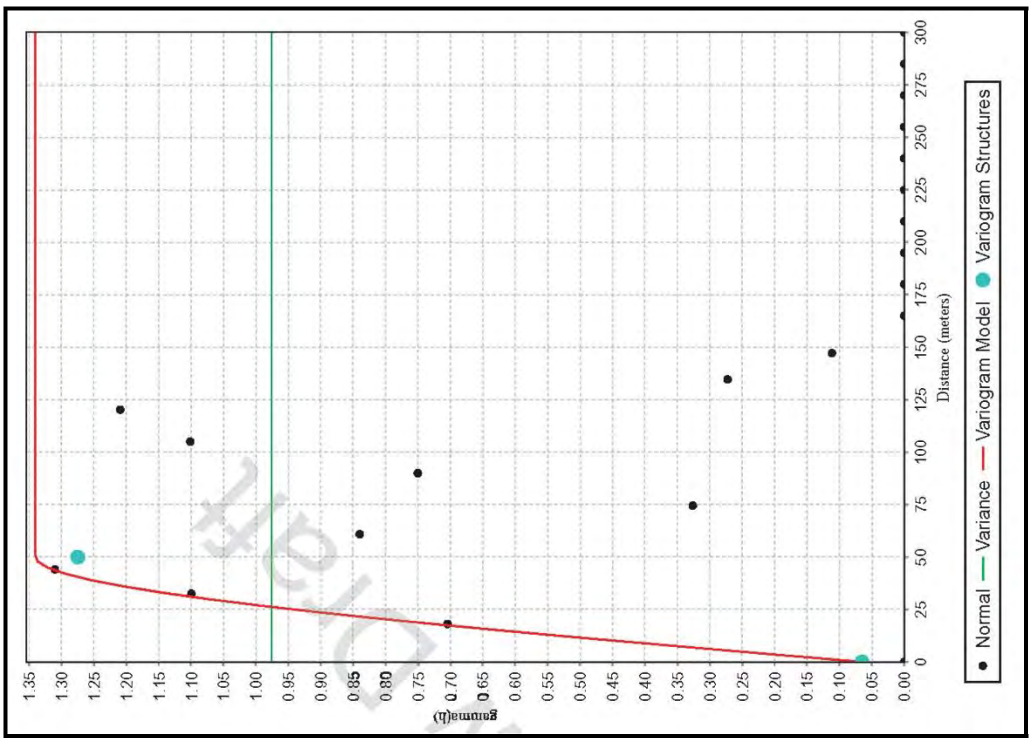


**Figure 14.13 Variogram Model for Primary Continuity Axis: 20 m Lag – Co%**

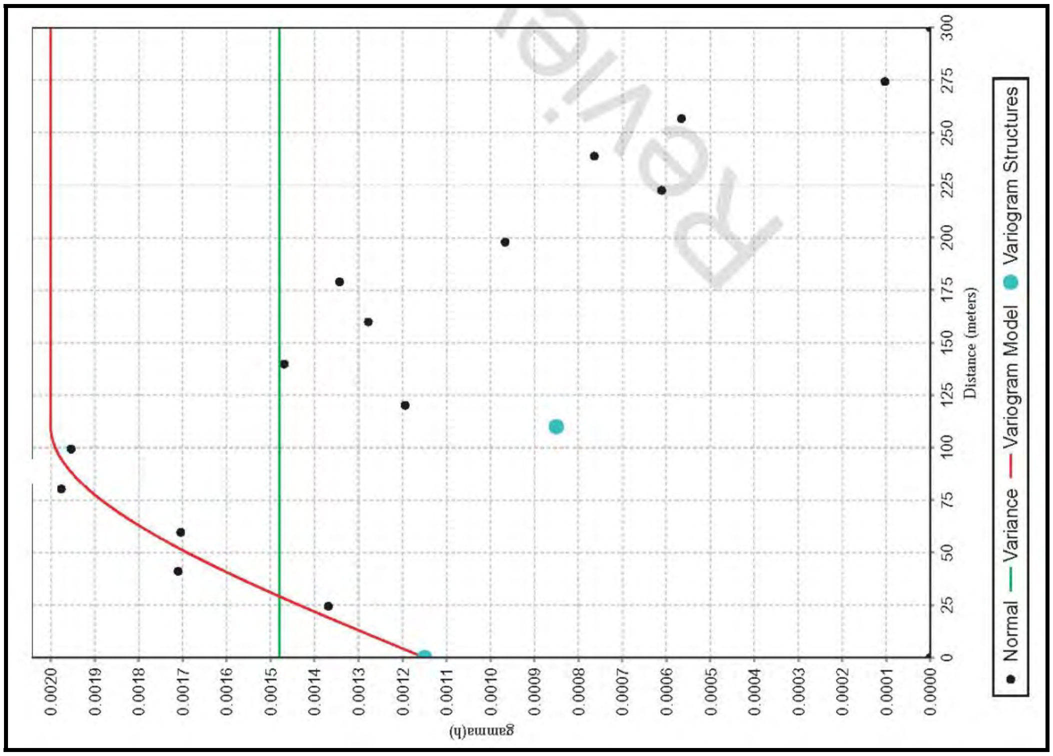



**Figure 14.14 Variogram Model for Secondary Continuity Axis: 15 m Lag – Co%**

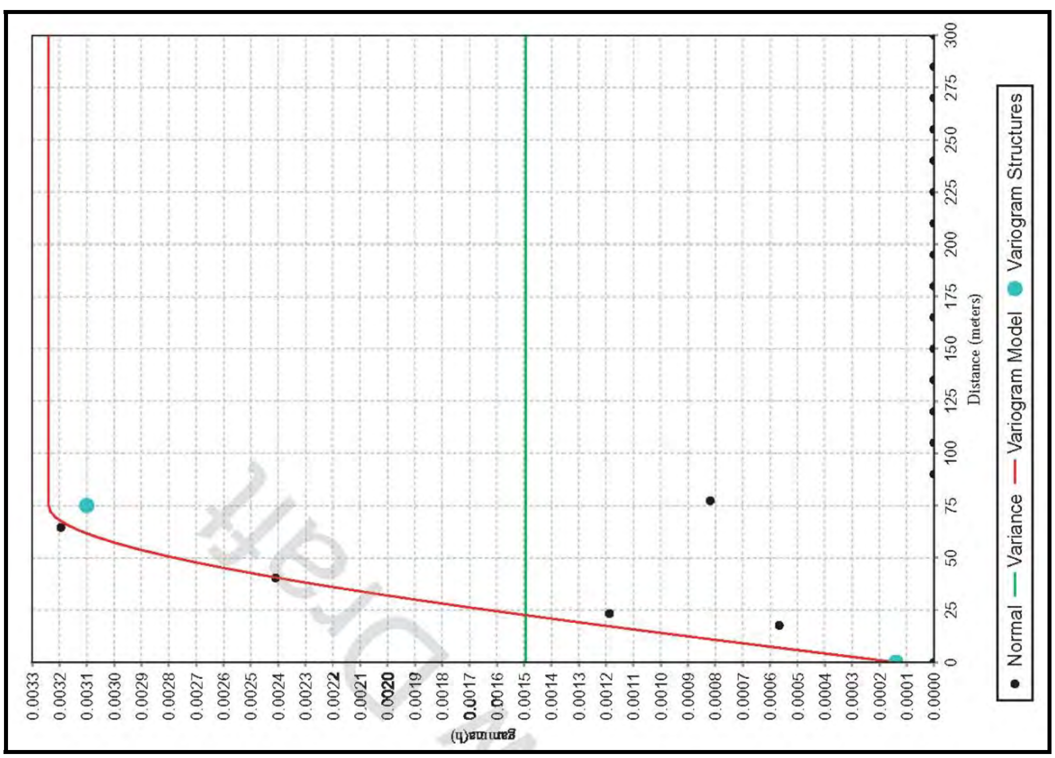


**Figure 14.15 Variogram Model for Secondary Continuity Axis: 20 m Lag – Co%**

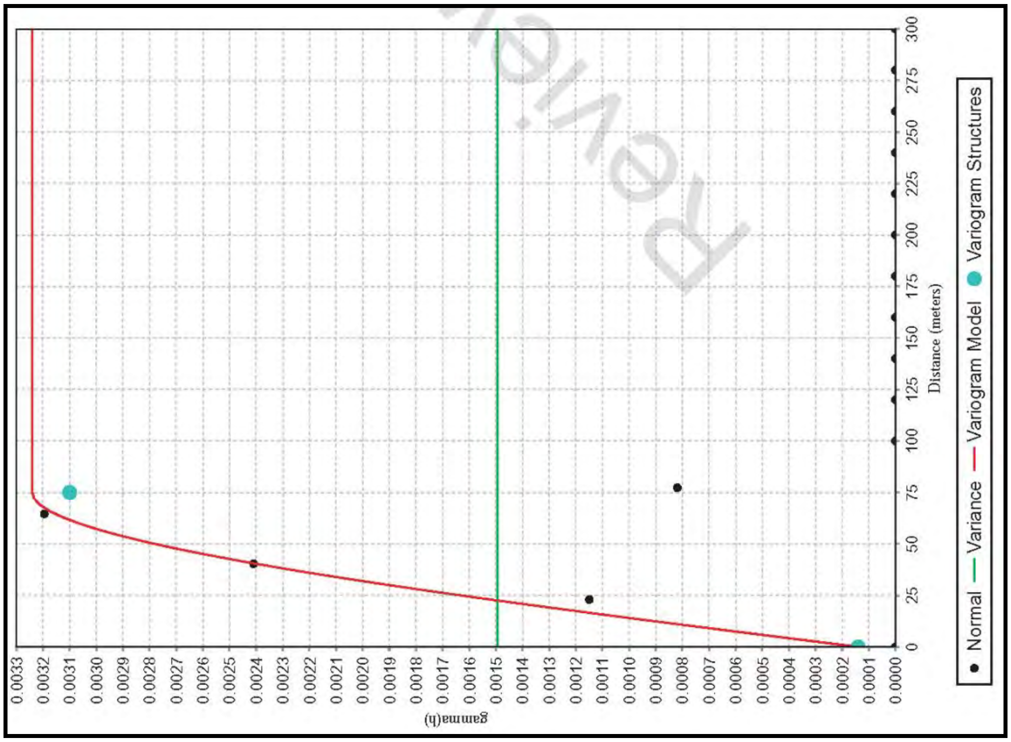
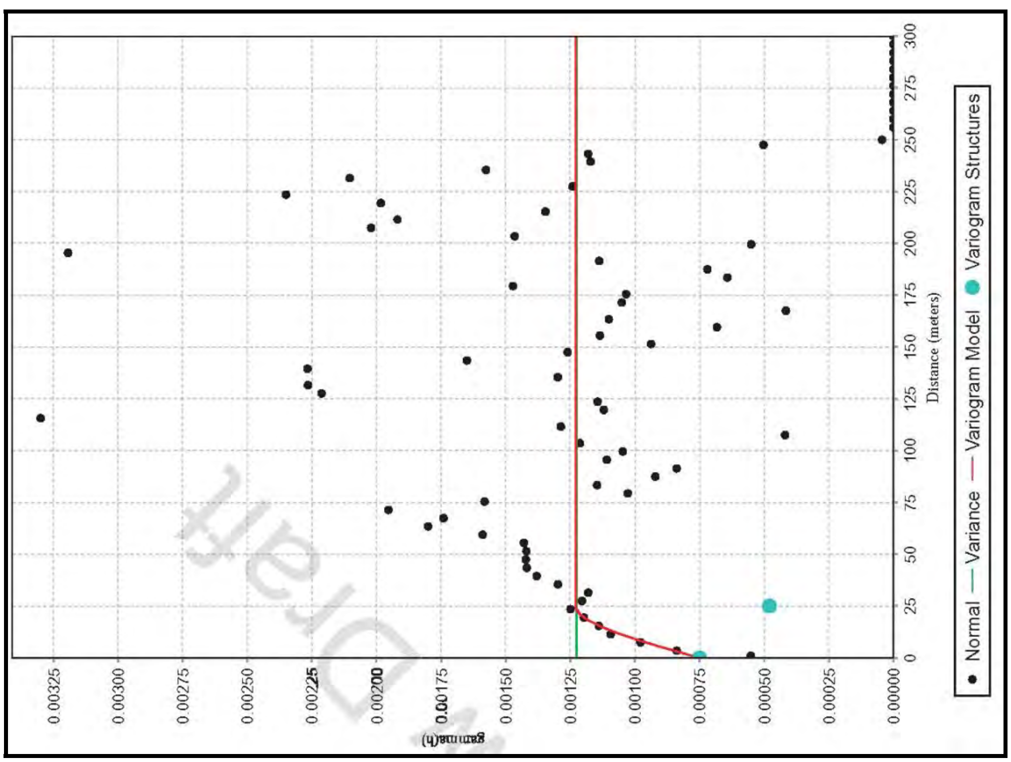



**Figure 14.16 Down Hole Variogram Model for Minor Continuity Axis: Co%**



*In Co Domain B, ellipsoid axial ranges of 125 m (major axis), 83 m (semi-major axis) and 12.5 m (minor axis) were used to address wider drillhole spacing and in a small interpolation sub-domain of Cu Domain A, ellipsoid axial ranges of 125 m (major axis), 50 m (semi-major axis) and 16 m (minor axis) were used to accommodate local geometric irregularity of the solid. Interpolation ellipsoid orientations and ranges for all domains are summarized below in Table 14.2 using the Surpac reference framework.*

**Table 14.2 Ellipsoid Ranges and Orientations for Interpolation Domains**

| *Interpolation Domain* | *Azimuth (Axis 1)* | *Plunge (Axis 2)* | *Dip (Axis 3)* | *Ranges (m) (Major/Semi-major/mMinor)* |
|---|---|---|---|---|
| *Peripheral – Above Cu Solids* | *270* | *-70* | *25* | *125/50/12.5* |
| *Peripheral – Below Cu Solids* | *290* | *-55* | *55* | *125/50/12.5* |
| *Peripheral – Between Cu Solids* | *270* | *-65* | *35* | *125/50/12.5* |
| *Peripheral – Above Co Solids* | *265* | *-67.5* | *15* | *125/50/12.5* |
| *Peripheral – Below Co Solids* | *265* | *-67.5* | *15* | *125/50/12.5* |
| *Peripheral – Between Co Solids* | *265* | *-67.5* | *15* | *125/50/12.5* |
| *Co Solid A* | *270* | *-65* | *25* | *125/50/12.5* |
| *Co Solid B* | *260* | *-70* | *15* | *125/50/12.5* |
| *Cu Solid A – Sub-domain 1* | *265* | *-67.5* | *20* | *125/50/12.5* |
| *Cu Solid A – Sub-domain 2* | *265* | *-67.5* | *-15* | *125/50/16* |
| *Cu Solid B – Sub-domain 1* | *290* | *-55* | *70* | *125/83/12.5* |
| *Cu Solid B – Sub-domain 2* | *290* | *-55* | *45* | *125/83/12.5* |

### 14.1.11 *SETUP OF DECEMBER 2010 THREE DIMENSIONAL BLOCK MODEL*

*At the request of Stratabound, the Captain block model was developed using the company's local grid coordination system combined with a sea level elevation datum. A series of surveyed control points were used to develop a transformation between the local coordinate system and UTM NAD 83 (Zone 19) coordination and a listing of collar coordinates calculated in both systems appears in Appendix C. Block model extents were defined in the local grid as being from minus 400 m East to minus 100 m East and from 0 m North to plus 200 m North. The model extends in elevation from minus 400 m to plus 160 m relative to sea level datum, with the nominal topographic surface defined by drill collar elevations being at approximately 148 m above sea level.*

*Standard block size for the model was 2 m x 2 m x 2 m (X,Y,Z) and a minimum sub-block size of 1 m x 1 m x 1 m was permitted to better fit the model at grade domain, overburden surface and peripheral solid limits. In contrast to the earlier model by Cullen and Harrington (2009), no block rotation was applied. Topographic surface and base of overburden digital terrain models were established using surveyed drill collar coordinates plus drillhole lithocodes, with the base of overburden surface used to define an upper limit of the deposit in areas where mineralization is interpreted to extend to the bedrock surface.*

### 14.1.12 RESOURCE ESTIMATION

*ID2 grade interpolation was used to assign block grades within the updated Captain block model and the same approach was used for the previous estimate by Cullen and Harrington (2009). Earlier trials had shown that ID2 interpolation provided appropriate grade distribution within the Captain model, which is characterized by multiple discrete interpolation domains that correspond with recognizable metal grade domains.*

*As reviewed earlier, interpolation ellipsoid orientation and range values used in the estimation reflect a combination of trends determined from variography that were checked against sectional geology and grade distribution interpretations developed for the deposit. The trends and ranges of the major, semi-major and minor axes of grade interpolation ellipsoids used in the various grade domains were described previously in report Section 14.3.7 and Table 14.2. The maximum number of contributing composites used to estimate a block grade was set at 12 and the minimum number of contributing composites was set at 1. A maximum of 4 composites per drillhole was allowed for block grade estimation and block discretization was set at 1Y x 1X x 1Z. Block model grades for Cu and Co were separately interpolated, with Au interpolated in conjunction with Cu.*

*Multiple interpolation domains were used, with a combination of composite and block constraints. Block grades in Cu Domain A and Cu Domain B were first interpolated using respective assay composite files, with Cu and Au interpolated in the same passes. Two sub-domains were established in each case to accommodate local geometric irregularities of the solids, with grade interpolation in each carried out using slightly differing ellipsoid orientations. In all cases, the entire assay composite population for the full domain was available for sub-domain grade interpolation purposes.*

*After completion of the above, Cu and Au grades were interpolated for all blocks occurring in the Peripheral Domain solid and outside the two high grade Cu solids. This was completed sequentially by first interpolating blocks occurring above (up-hole) of the high grade Cu solids followed by blocks occurring between the high grade Cu solids, and finally, by the remaining blocks occurring below (down-hole) the high grade Cu solids. Assay composites were constrained during the Peripheral Domain interpolation pass by excluding those assay composites in the high grade solids and in the other Peripheral Domain subzones.*

*Co grades were interpolated next for all blocks occurring in the Co Domain A and Co Domain B solids followed by interpolation of all remaining blocks in the Peripheral Domain solid. The sequential methodology used for Cu-Au interpolation in the Peripheral Domain solid was also followed for Co interpolation. After completion of all grade interpolation runs, Cu Equivalent values for all blocks were calculated from the interpolated Cu and Co block grades using the previously discussed equation CuEQ% = Cu% +(Co% x 9.25).*

*A complete set of block model cross sections at 25 m grid section spacing and block model level plans at 25 m elevation increments were prepared to document block model grade distribution relationships. Representative examples of these appear in Appendix E and Figure 14.17, Figure 14.18, and Figure 14.19 present perspective views of the block model showing CuEQ%, Cu % and Co % grade parameters. Spatial aspects of the higher grade Cu core zone comprised of Cu Domain A and Cu Domain B, plus the sub-parallel zones of higher Co grade modeled as Co Domain A and Co Domain B are apparent in these figures.*

**Figure 14.17 Block Model Perspective View – CuEQ%**

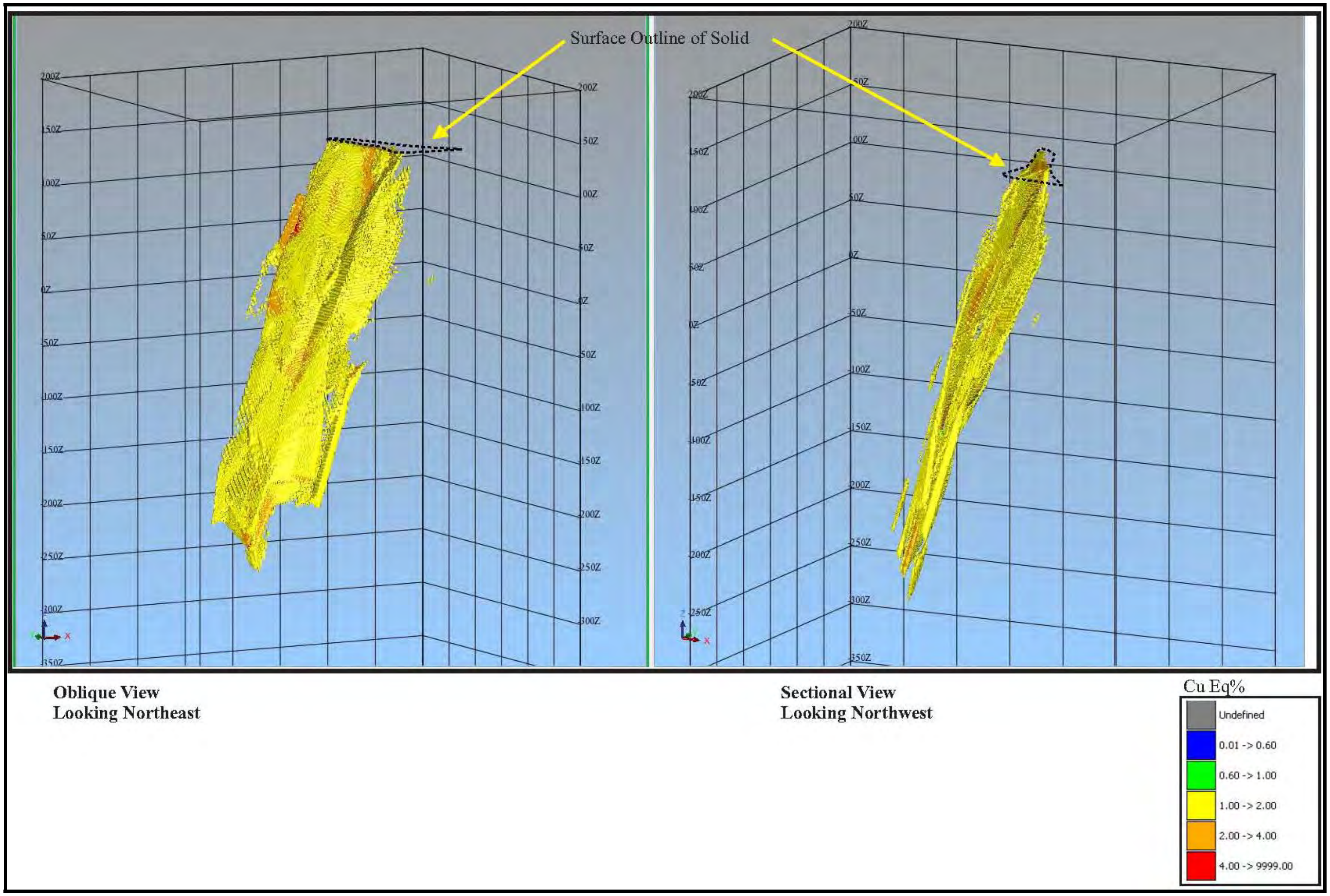


**Figure 14.18 Block Model Perspective View – Cu%**

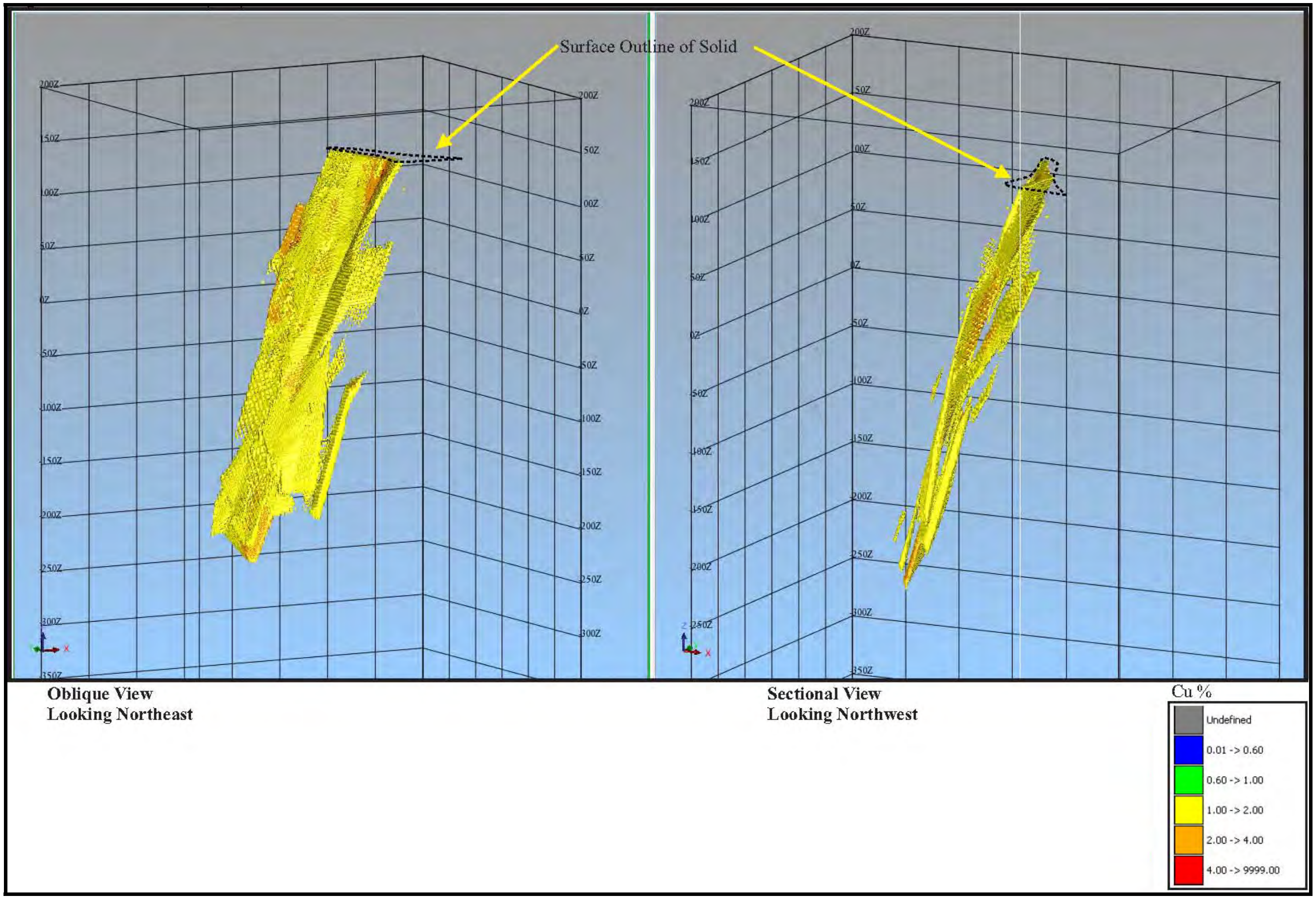
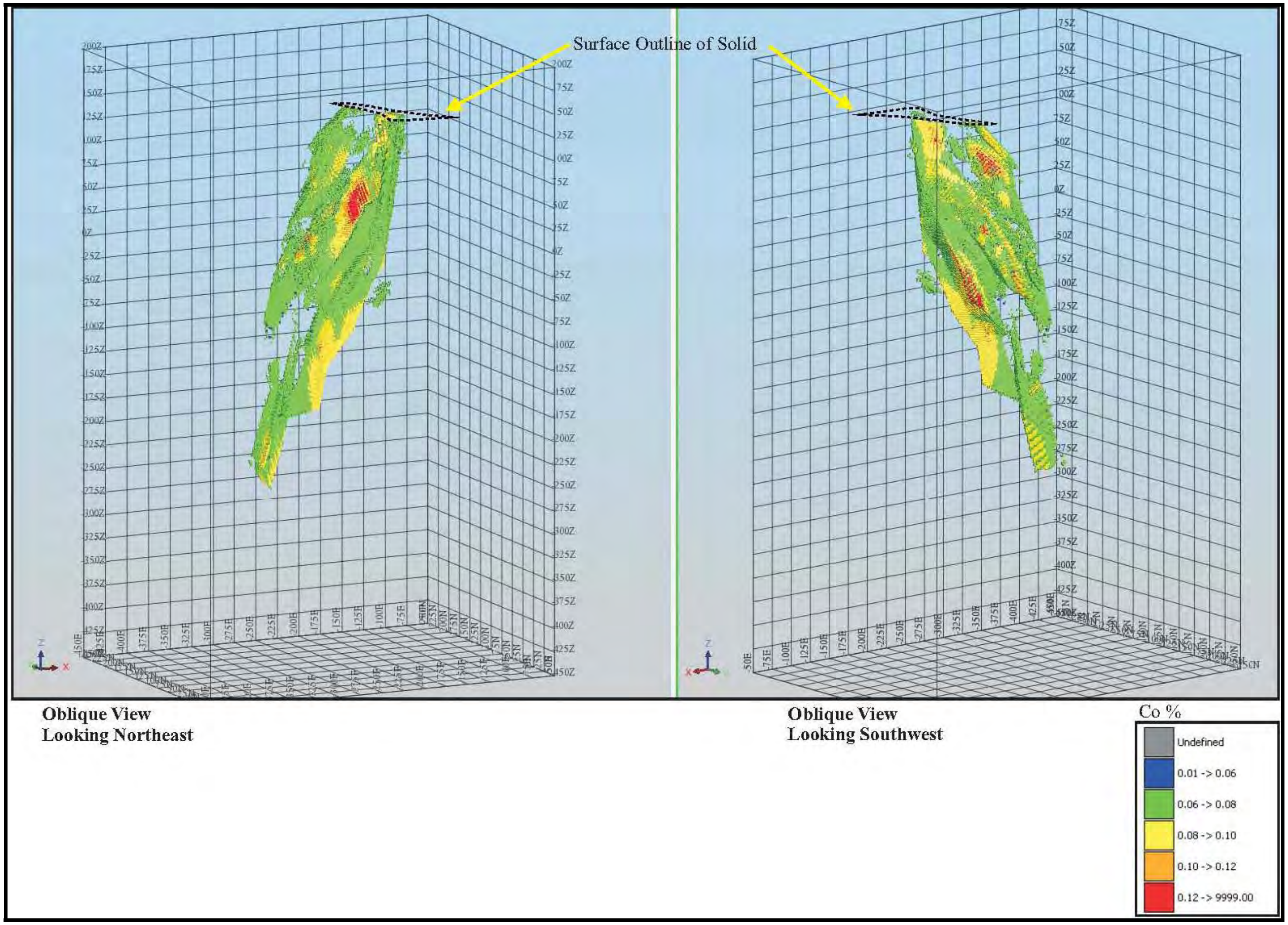


**Figure 14.19 Block Model Perspective View – Co%**



### 14.1.13 DENSITY DETERMINATIONS

*Density values used in the block model were derived from laboratory determinations carried out on Stratabound core sample pulps. The data set contains a total of 68 pyncnometer determinations carried out on check sample pulps from both Stratabound drilling programs. ALS Chemex and AGAT Laboratories Ltd., both ISO accredited firms, provided associated laboratory services. No checking of determinations by other methods was carried out, but values from the two laboratories show consistency of range for comparable rock types. The current project dataset includes 42 more determinations than used by Cullen and Harrington (2009) and covers a broader range of Cu grades and mineralized rock types. Descriptive statistics for the data set used in the current resource estimate appear in Table 14.3*

**Table 14.3 Density Value Descriptive Statistics**

| Parameter | Value |
|---|---|
| Mean | 2.99 g/cm$^3$ |
| Standard Deviation | 0.29 |
| Range | 1.46 |
| Minimum | 2.73 |
| Maximum | 4.19 |
| Confidence Interval 95% | 0.071 |
| Number | 68 |

*Cullen and Harrington (2009) plotted density values against corresponding Cu, Co and CuEQ grades and a generally systematic increase of SG with increasing metal grades was noted. This represents the geological association of higher total metal concentrations with increasing amounts of total sulphide mineralization, which is typically dominated by increasing pyrite content. With this geological model in mind, a linear regression curve was established using Microsoft Excel® to better define the relationship between metal grades and density value. The associated regression equation is density (g/cm$^3$) = 2.83 g/cm$^3$ + (0.147 x (Cu% + Co%)) and differs from that used by Cullen and Harrington (2009) through use of combined Cu and Co analyses rather than a calculated Cu Equivalent value.*

### 14.1.14 RESOURCE CATEGORY DEFINITIONS

*Definitions of mineral resource and associated mineral resource categories used in this report are those recognized under NI 43-101 and set out in the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves Definitions and Guidelines (the CIM Standards).*

### 14.1.15 *Resource Category Parameters Used in Current Estimate*

*Mineral resources in the Inferred, Indicated and Measured categories are included in the current resource estimate, with the largest percentage reported in the Indicated category. This reflects relative spacing and distribution of the 30 drillholes completed by Stratabound along a strike length of approximately 175 m in the immediate deposit area. The following resource category definitions apply to the current Captain estimate and reflect a progressively decreasing scale of certainty based on proximity of sample composites included in block grade assignments.*

***Measured Resource Category:*** *Blocks having seven or more contributing assay composites from three separate drillholes, with an average distance of 31.25 m (25% the major axis range) from all contributing composites, located 12.5 m or less (10% the major axis range) from the nearest contributing composite, and falling within a smoothed three dimensional model solid based on the noted block definition parameters.*

***Indicated Resource Category:*** *Blocks having seven or more contributing assay composites from three separate drillholes, with an average distance of 62.5 m (50% of the major axis range) or less from all contributing composites, located 41.7 m or less (33.3% of the major axis range) from the nearest sample, and not classified in the measured category.*

***Inferred Resource Category:*** *All remaining valid blocks within the Peripheral Domain solid that have interpolated grades and are not included in the Indicated or Measured categories.*

### 14.1.16 *Statement of Resource Estimate*

*Block grade, block specific gravity and block volume parameters for the Captain deposit were estimated through the methods described in preceding sections of this report. Subsequent application of the resource category parameters set out above resulted in the mineral resource estimate statement presented in Table 14.4. Results are in accordance with the CIM Standards as well as disclosure requirements of NI 43-101.*

**Table 14.4 Measured and Indicated Resources for the Captain Cu-Co Deposit; Effective 8 December, 2010 (Cullen and Harrington, 2010)**

| CuEQ% Cut-off | Resource Category | Tonnes ('000 t) | Cu% | Co% | Au (g/t) |
|---|---|---|---|---|---|
| **0.60** | **Measured** | **68** | **1.09** | **0.059** | **0.20** |
| | **Indicated** | **938** | **1.03** | **0.050** | **0.20** |
| | **Measured + Indicated** | **1,006** | **1.03** | **0.051** | **0.20** |
| 1.00 | Measured | 46 | 1.51 | 0.056 | 0.25 |
| | Indicated | 621 | 1.41 | 0.047 | 0.25 |
| | Measured + Indicated | 667 | 1.42 | 0.048 | 0.25 |
| 1.40 | Measured | 32 | 1.86 | 0.057 | 0.29 |
| | Indicated | 416 | 1.74 | 0.045 | 0.30 |
| | Measured + Indicated | 448 | 1.75 | 0.046 | 0.30 |

Note: *CuEQ% = Cu % + (Co % X 9.25). The 9.25 factor represents the relative price of Co compared to Cu based on 3 year average metal pricing with no metal recovery factors applied.

**Table 14.5 Inferred Resources for the Captain Cu-Co Deposit; Effective 8 December, 2010 (Cullen and Harrington, 2010)**

| CuEQ% Cut-off | Resource Category | Tonnes ('000 t) | Cu% | Co% | Au (g/t) |
|---|---|---|---|---|---|
| 0.60 | Inferred | 960 | 0.64 | 0.039 | 0.12 |
| 1.00 | | 298 | 1.18 | 0.038 | 0.20 |
| 1.40 | | 162 | 1.47 | 0.040 | 0.24 |

Note: *CuEQ % = Cu % + (Co % X 9.25). The 9.25 factor represents the relative price of Co compared to Cu based on 3 year average metal pricing with no metal recovery factors applied.

### 14.1.17 MODEL VALIDATION

#### VISUAL COMPARISON TO GEOLOGICAL SECTIONS

*Results of block modeling were compared on a section by section basis with corresponding manually interpreted geological and grade distribution sections prepared prior to model development. This showed block model grade patterns to generally conform to the trend of a north-northwest striking, steeply west-southwest dipping deposit envelope and to locally transgress lithological unit boundaries. This is consistent with the recognized volcanogenic massive sulphide stockwork character of Cu, Co, and Au mineralization interpreted for the Captain deposit. Isolation of higher grade Cu and Co values within corresponding solids is apparent and excessive smoothing of grade to surrounding lower grade areas does not appear to have occurred. Overall, results of the visual inspection show an acceptable degree of consistency between the block model and the independently derived sectional interpretations.*

### Comparison of Composite Database and Block Model Grades

*Descriptive statistics were calculated for the drillhole composite values used in the block model grade interpolations and these were compared to values calculated for the individual blocks in the block model for each grade domain (Table 14.6). The mean drillhole composite grades for the various domains were found to compare acceptably with corresponding grades of the block model, thereby providing a general check on the model with respect to the underlying assay composite population. The Peripheral Domain figures illustrate effect of the high percentage of lower grade Cu and Co mineralization external to the higher grade domain solids. Results of the composite and block grade comparison show acceptable agreement between the data sets.*

**Table 14.6** ***Comparison* of Drillhole Composite Grades and Block Model Grades**

| ***Total Deposit*** | ***Block Model*** | | | ***1 m Composites*** | | |
|---|---|---|---|---|---|---|
| *Metal* | *Cu%* | *Co%* | *Au (g/t)* | *Cu%* | *Co%* | *Au (g/t)* |
| *Mean Grade* | *0.503* | *0.034* | *0.109* | *0.524* | *0.035* | *0.114* |
| *Variance* | *0.379* | *0.0005* | *0.012* | *0.882* | *0.001* | *0.029* |
| *Standard Deviation* | *0.615* | *0.021* | *0.111* | *0.939* | *0.034* | *0.17* |
| *Coefficient of Variation* | *1.224* | *0.633* | *1.019* | *1.793* | *0.956* | *1.496* |
| *Number of Samples* | *686033* | *686033* | *686033* | *1642* | *1642* | *1642* |
| ***Cu Domain A*** | ***Block Model*** | | | ***1 m Composites*** | | |
| *Metal* | *Cu%* | *Au (g/t)* | | *Cu%* | *Au (g/t)* | |
| *Mean Grade* | *1.458* | *0.262* | | *1.53* | *0.274* | |
| *Variance* | *0.455* | *0.023* | | *1.887* | *0.066* | |
| *Standard Deviation* | *0.675* | *0.150* | | *1.374* | *0.256* | |
| *Coefficient of Variation* | *0.463* | *0.573* | | *0.896* | *0.935* | |
| *Number of Samples* | *120653* | *120653* | | *343* | *343* | |
| ***Cu Domain B*** | ***Block Model*** | | | ***1 m Composites*** | | |
| *Metal* | *Cu%* | *Au (g/t)* | | *Cu%* | *Au (g/t)* | |
| *Mean Grade* | *0.820* | *0.135* | | *0.83* | *0.141* | |
| *Variance* | *0.210* | *0.005* | | *1.006* | *0.025* | |
| *Standard Deviation* | *0.458* | *0.073* | | *1.003* | *0.159* | |
| *Coefficient of Variation* | *0.558* | *0.541* | | *1.208* | *1.13* | |
| *Number of Samples* | *97595* | *97595* | | *206* | *206* | |
| ***Co Domain A*** | ***Block Model*** | | | ***1 m Composites*** | | |
| *Metal* | *Co%* | | | *Co%* | | |
| *Mean Grade* | *0.055* | | | *0.06* | | |
| *Variance* | *0.0002* | | | *0.001* | | |
| *Standard Deviation* | *0.014* | | | *0.032* | | |

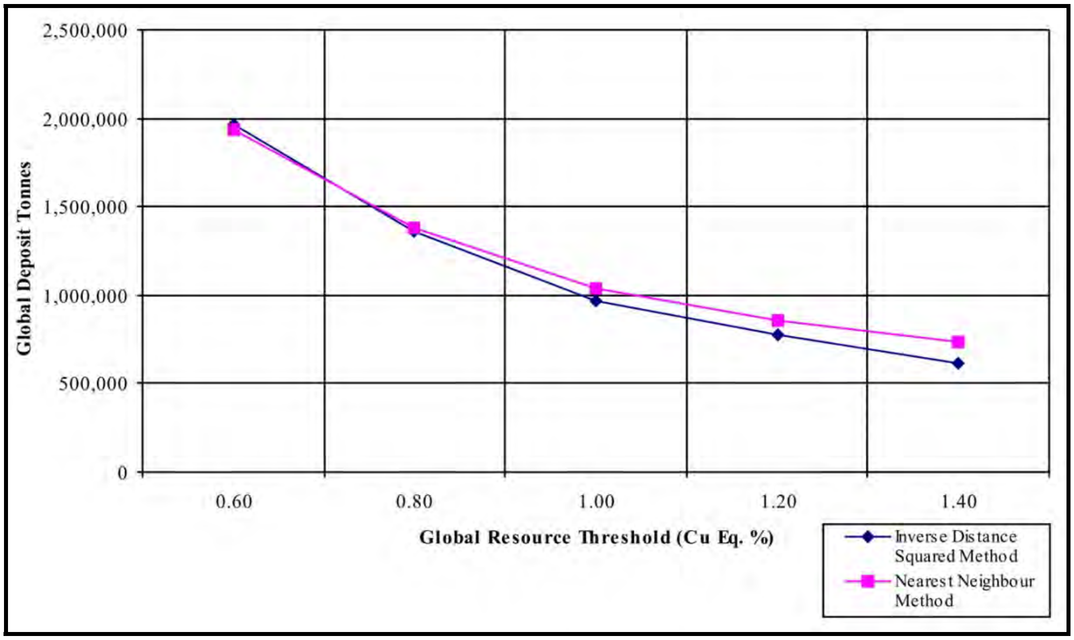
*table continues…*

| Co Domain A | Block Model | 1 m Composite |
|---|---|---|
| Coefficient of Variation | 0.254 | 0.542 |
| Number of Samples | 77971 | 202 |
| **Co Domain B** | **Block Model Grades** | **1 m Composites** |
| Metal | Co% | Co% |
| Mean Grade | 0.061 | 0.061 |
| Variance | 0.0003 | 0.002 |
| Standard Deviation | 0.019 | 0.042 |
| Coefficient of Variation | 0.303 | 0.687 |
| Number of Samples | 137318 | 443 |

### Comparison With Nearest Neighbour Interpolation Model

*The ID2 resource model for the Captain deposit was checked using Nearest Neighbour (NN) interpolation methodology. Single drillhole composites for each interpolation sub-domain, assigned at the mid-point of the intersection between the drillhole and the respective domain boundaries, were used for the NN check model. Search ellipse parameters and other applicable parameters were the same as those used in the ID2 model. Figure 14.20 shows that deposit tonnage and average grades at the lowest thresholds are directly comparable with slightly higher tonnage and grades occurring in the NN model at thresholds above 1.0 CuEQ%. This results from strong spatial dependence of new tonnage and grade on results of a single new drillhole, CP10-26, that is the deepest hole to intersect the high grade Cu domain to date. ID2 results are lower in this area due to averaging with lower grade holes throughout the same area. Both approaches fill available model space defined by the Peripheral, Cu and Co domain solids and this is directly reflected in lowest threshold results. The NN model provides a de-clustered result relative to the ID2 method but also results in locally less realistic grade distribution trends. Results of the two methods are considered sufficiently consistent to provide an acceptable check on the preferred ID2 methodology.*

**Figure 14.20 Grade/Tonnage Curves for ID2 and NN Models**




### 14.1.18 *PREVIOUS RESOURCE OR RESERVE ESTIMATES*

#### HISTORIC RESOURCE ESTIMATES PRIOR TO STRATABOUND PROGRAMS

*Records show that at least seven separate Captain deposit resource estimates had been referenced prior to 2008, with the earliest being completed by Captain Gold Mines Ltd. in 1965 (Table 14.7). All of these were based on results of historic core drilling programs completed on the property prior to the 1988. Cullen and Harrington (2009) note that records are incomplete or limited for most of the estimates, but in all cases, conventional methods of polygon-based resource calculation are assumed. The Whaley (1988) estimate for Stratabound incorporated polygonal end area calculations with volume projection to midpoints between cross sections. In all instances, previous resource estimates are considered historic in nature and not compliant with either NI-43-101 or the CIM Standards. As such, they should not be relied upon.*

**Table 14.7 *Non NI 43-101 Compliant Historic Resource Estimated for the Captain Deposit**

| *Year* | *Source* | *Category* | *Tons* | *Cu (%)* | *Ag (oz/ton)* | *Au (oz/ton)* | *Co %* |
|---|---|---|---|---|---|---|---|
| *1965* | *Ritchie (1965)* | *Unclassified* | *349,330* | *1.63* | *---* | *---* | *---* |
| *1965* | *Including* | *Unclassified* | *262,000* | *---* | *0.22 oz/t* | *---* | *---* |
| *1965* | *Including* | *Unclassified* | *300,450* | | *---* | *0.015* | *---* |
| *1971* | *Northern Miner, June 3, 1971* | *Unclassified* | *343,000* | *1.99* | *0.28* | *0.015* | *---* |
| *1974* | *GSC Memoir 371* | *Unclassified* | *802,000* | *1.15* | *---* | *---* | *---* |

*table continues…*

| Year | Source | Category | Tons | Cu (%) | Ag (oz/ton) | Au (oz/ton) | Co % |
|---|---|---|---|---|---|---|---|
| 1976 | Northern Miner, June 24th | Unclassified | 802,000 | 1.15 | --- | 0.032 | --- |
| 1976 | National Mineral Inventory | Unclassified | 802,000 | 1.15 | --- | 0.032 | --- |
| 1976 | Williams? | Unclassified | 215,000 | 1.99 | 0.28 | 0.017 | --- |
| 1983 | NBDNR Information Circular 83-2 | Unclassified | 343,000 | 1.99 | 0.28 | 0.017 | --- |
| 1983 | National Mineral Inventory | Unclassified | 311,000 | 1.99 | .31 | 0.019 | --- |
| 1988 | Whaley (1988) | Unclassified | 197,220 | 2.12 | --- | --- | --- |
| 1988 | Including | Unclassified | 145,680 | --- | 0.3 oz Ag | --- | --- |
| 1988 | Including | Unclassified | 180,160 | --- | --- | 0.019 | --- |

*Cullen and Harrington (2009) noted that historic estimates were based drillholes that provide less systematic coverage of the deposit than the Stratabound 2007-2008 program. The Stratabound program also included continuous core sampling through the total mineralized zone, with systematic analysis of Cu, Pb, Zn, Ag, Au, and Co for all core samples plus selected Bi analyses. This contrasts with sampling of the earlier programs that was focused on higher grade intervals marked by visible chalcopyrite and/or massive and semi-massive sulphide.*

### RESOURCE ESTIMATE REPORTED BY CULLEN AND HARRINGTON (2009)

*A mineral resource estimate for the Captain deposit was prepared for Stratabound by Mercator in late 2008 and was reported by Cullen and Harrington (2009). Results of this estimate are presented in Table 14.8 and are in compliance with NI 43-101. Associated methodology is directly comparable to that described in the current report, which differs through expansion of the earlier deposit model by addition of data from five drillholes completed in 2010 by Stratabound.*

**Table 14.8 NI 43-101 Compliant Captain Resource Estimate - Effective Oct. 29th 2008**

| CuEQ%* Threshold | Classification | Rounded Tonnes | CuEQ%* | Cu% | Co% | Au (g/t) |
|---|---|---|---|---|---|---|
| 0.60 | Inferred | 681,000 | 0.96 | 0.60 | 0.039 | 0.12 |
| | Indicated | 808,000 | 1.58 | 1.10 | 0.051 | 0.22 |
| | Measured | 53,000 | 1.70 | 1.14 | 0.061 | 0.21 |
| 0.80 | Inferred | 354,000 | 1.22 | 0.83 | 0.042 | 0.16 |
| | Indicated | 660,000 | 1.77 | 1.30 | 0.051 | 0.25 |
| | Measured | 45,000 | 1.90 | 1.32 | 0.062 | 0.24 |
| 1.00 | Inferred | 192,000 | 1.51 | 1.14 | 0.040 | 0.21 |
| | Indicated | 543,000 | 1.97 | 1.51 | 0.049 | 0.28 |
| | Measured | 38,000 | 2.06 | 1.50 | 0.061 | 0.26 |

*table continues....*

| CuEQ%* Threshold | Classification | Rounded Tonnes | CuEQ%* | Cu% | Co% | Au (g/t) |
|---|---|---|---|---|---|---|
| 1.20 | Inferred | 126,000 | 1.74 | 1.41 | 0.035 | 0.26 |
| | Indicated | 466,000 | 2.11 | 1.67 | 0.048 | 0.30 |
| | Measured | 31,000 | 2.29 | 1.74 | 0.059 | 0.28 |
| 1.40 | Inferred | 94,000 | 1.89 | 1.57 | 0.034 | 0.29 |
| | Indicated | 397,000 | 2.25 | 1.81 | 0.047 | 0.33 |
| | Measured | 25,000 | 2.54 | 1.99 | 0.060 | 0.31 |

Notes: *CuEQ% = Cu% + (Co% x 9.25) based on three year trailing average market pricing of Cu and Co.

*Results of the two estimation programs are comparable with respect to both tonnage and grade parameters, with global metal grades of the updated estimate being slightly lower than those of the earlier estimate and global tonnage being higher. These results reflect substantial down plunge extension of the mineralized zone through 2010 drilling, but less drilling intercept exposure to higher grade metal domains. Results of the two programs are interpreted to be mutually consistent.*

## 14.2 CNE RESOURCE ESTIMATE

### 14.2.1 *INTRODUCTION*

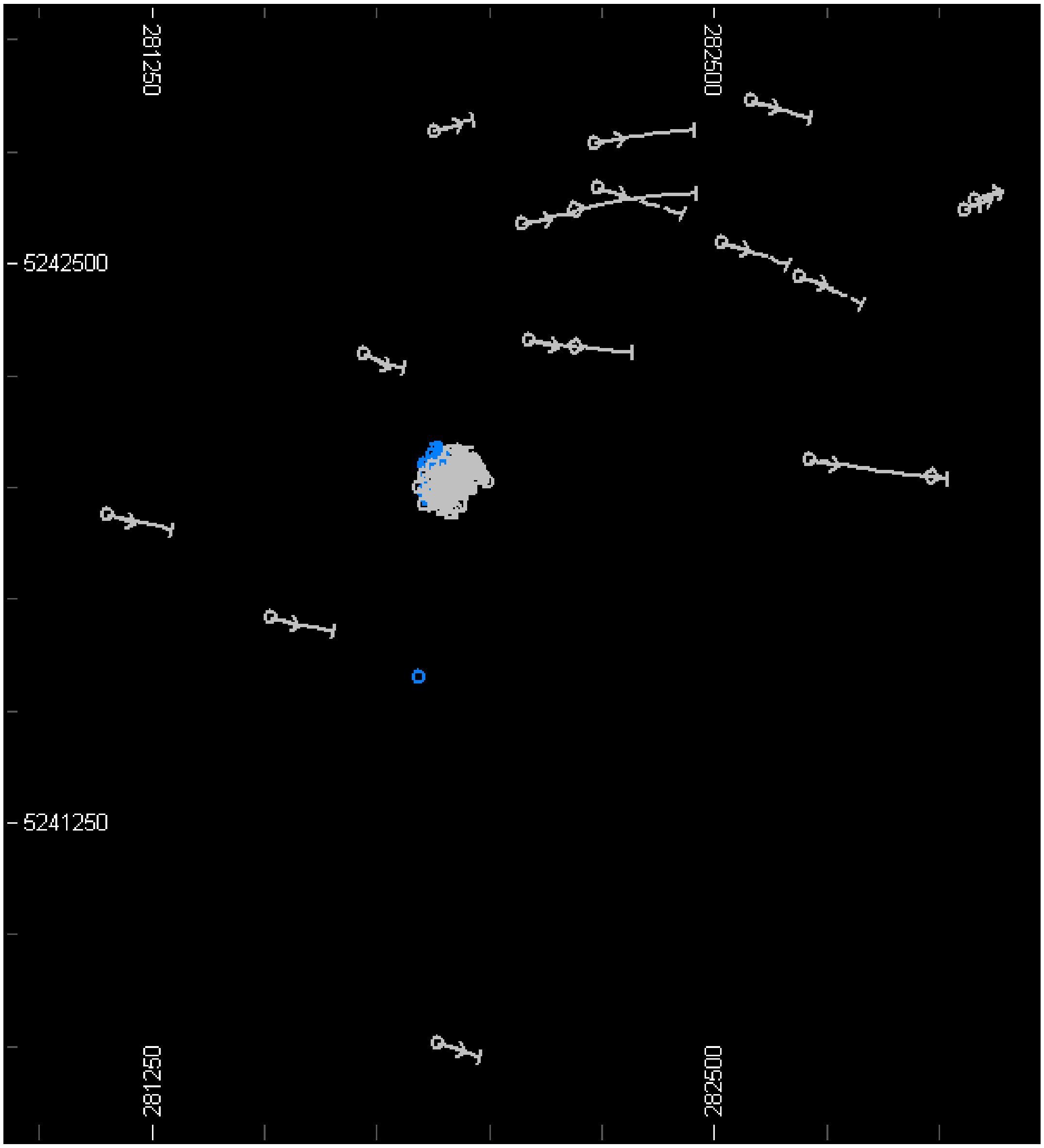
Tetra Tech has estimated an NI 43-101 compliant resource of a zinc-lead-copper massive sulphide that is the CNE VMS deposit.

The effective date of this resource estimate is May 10, 2011.

### 14.2.2 *DATABASE*

Stratabound supplied all of the digital data for the CNE resource estimate, dated September 2010. Stratabound compiled the historical drillhole data from previous assessment reports which are publicly available on the NBDNR website. This data was imported into Datamine$^{TM}$ Studio 3 (Version 3.19.3638.0) Resource Software package in the form of a drill collar file, a down hole survey file and an assay file; all in a comma-separated format (.csv file). From these files, a comprehensive drillhole file was generated (Figure 14.21). The macro used in the block model estimation may be found in Appendix F.

**Figure 14.21 Plan View of Datamine TM Drillhole File for the CNE Deposit**



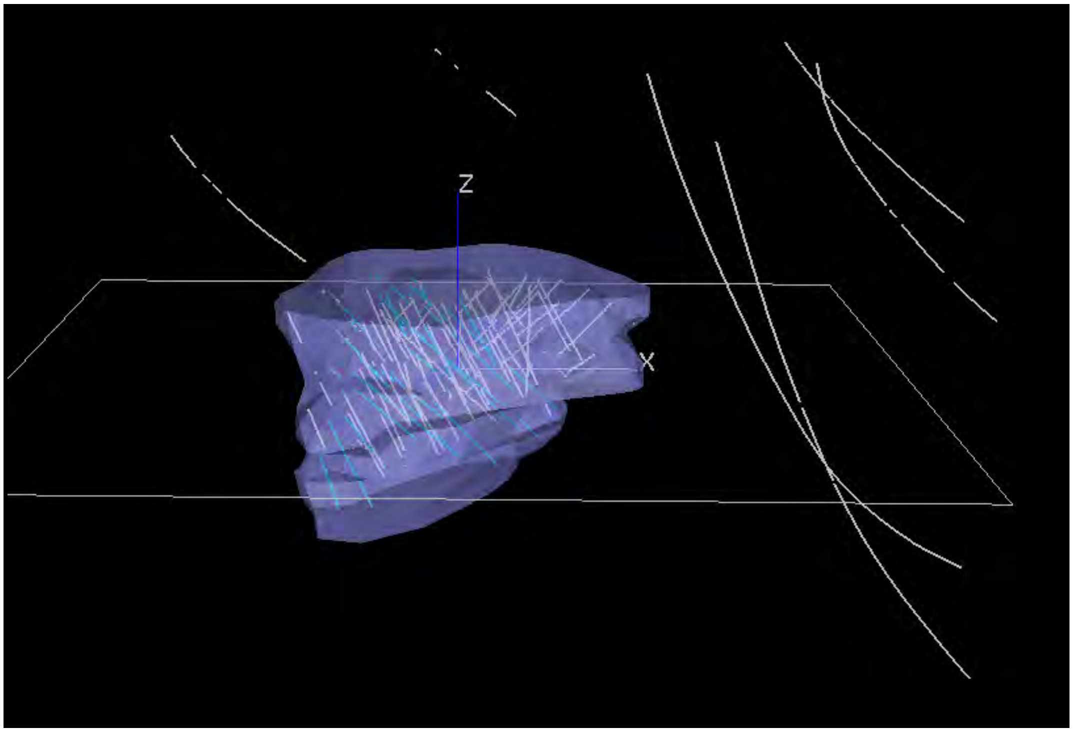

Note: The actual CNE deposit is located at the central area of clustered drillholes in the centre of the image.

The data set underwent a preliminary verification to determine the reliability of the data. It was found to contain downhole survey errors, collar data with no dip and azimuth at surface (AT = zero), and some gold assays were entered into the database erroneously. As a result, all data points were re-checked against original logs, where possible.

Data was restricted to the volume immediately adjacent to and surrounding the CNE mineralization. This was achieved by designing an appropriate wireframe in

Datamine™ (Figure 14.22). This volume also defined the limits of interpolated and extrapolated blocks in the resource model.

**Figure 14.22 Wireframe Defining Drillholes for Resource Modelling and Limits of Interpolation and Extrapolation in the Block Model, Isometric View Looking North**




### 14.2.3 *SPECIFIC GRAVITY*

A re-sampling and re-analysis was conducted on the 2009 and 2010 diamond drill core for the CNE deposit. This included specific gravity measurements. The reanalysis was completed by ACME Laboratories of Vancouver, BC (VAN10004978 Certificate 2).

Specific gravity measurements were taken on several distinct lithologies and mineralized sections of core of the CNE deposit.

The average of the specific gravity measurements were assigned to the indicator wireframes and specific lithologies (Table 14.9).

**Table 14.9 Gravity and Rock Codes**

| Wireframe | Specific Gravity |
|---|---|
| Air | 0 |
| Overburden | 2.00 |
| Country and Host Rock (un-mineralized) | 2.82 |
| High Grade Copper | 3.23 |
| Low Grade Zinc | 3.37 |
| High Grade Zinc | 3.86 |

### 14.2.4 DOMAINS OF CNE

#### DRILLING

Unlike many deposits which have well-defined geological controls to mineralization, the CNE deposit has proven to be too complex to employ classic geological interpretations. To accommodate this complexity, drill densities were increased and drill orientations modified. The drillhole data (coloured by Zn%) are depicted in the following images (Figure 14.23 and Figure 14.24). Note that major grid markers in both images are 50 m. Thus, drilling density can be as close as every 5 to 7 m.

**Figure 14.23 Plan View of Total CNE Drilling**

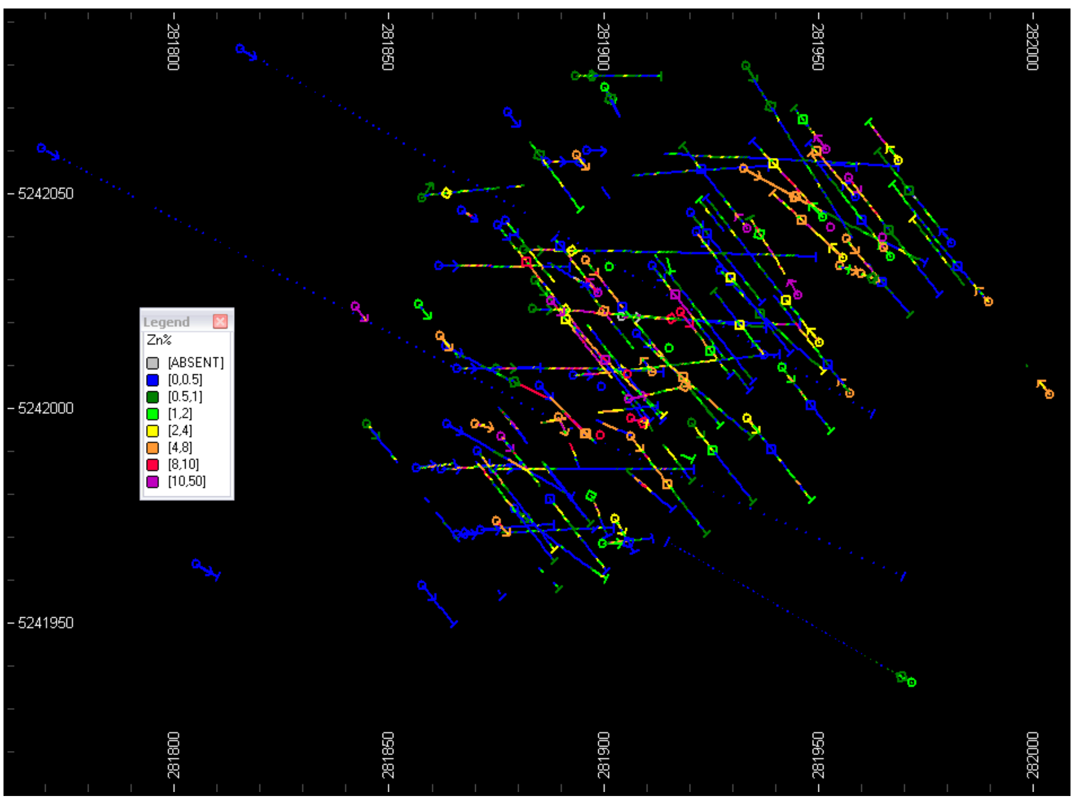
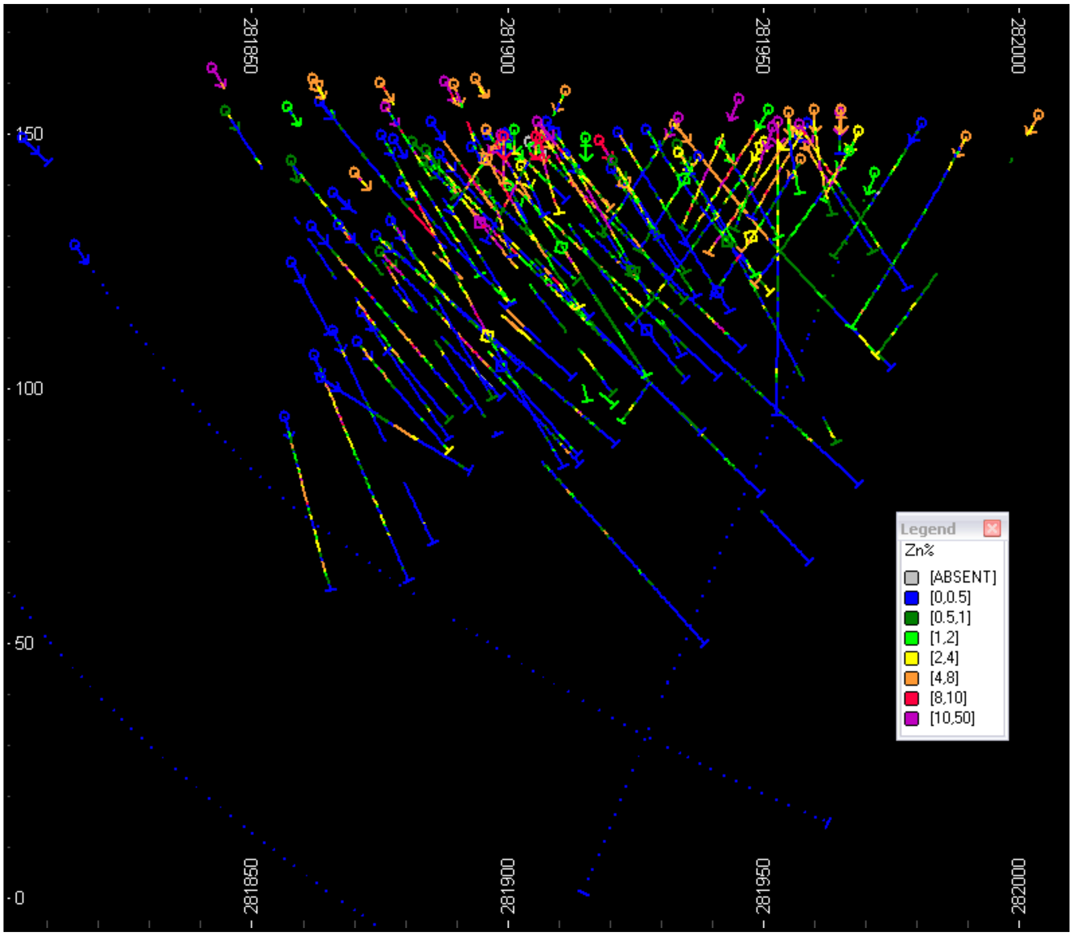


**Figure 14.24 West-east Section View of Total CNE Drilling**




At this resolution, and with drill fences oriented both east-west and northwest-southeast, it is impractical to model geological constraints for mineralization (e.g. massive sulphides, heavy disseminated sulphides, stringer sulphides) by classical sectional interpretation. Thus Tetra Tech employed Indicator Kriging to determine the most appropriate blocks to interpolate grade. The process of Indicator Kriging is discussed further in the text.

Mineralization at CNE falls into three broad categories:

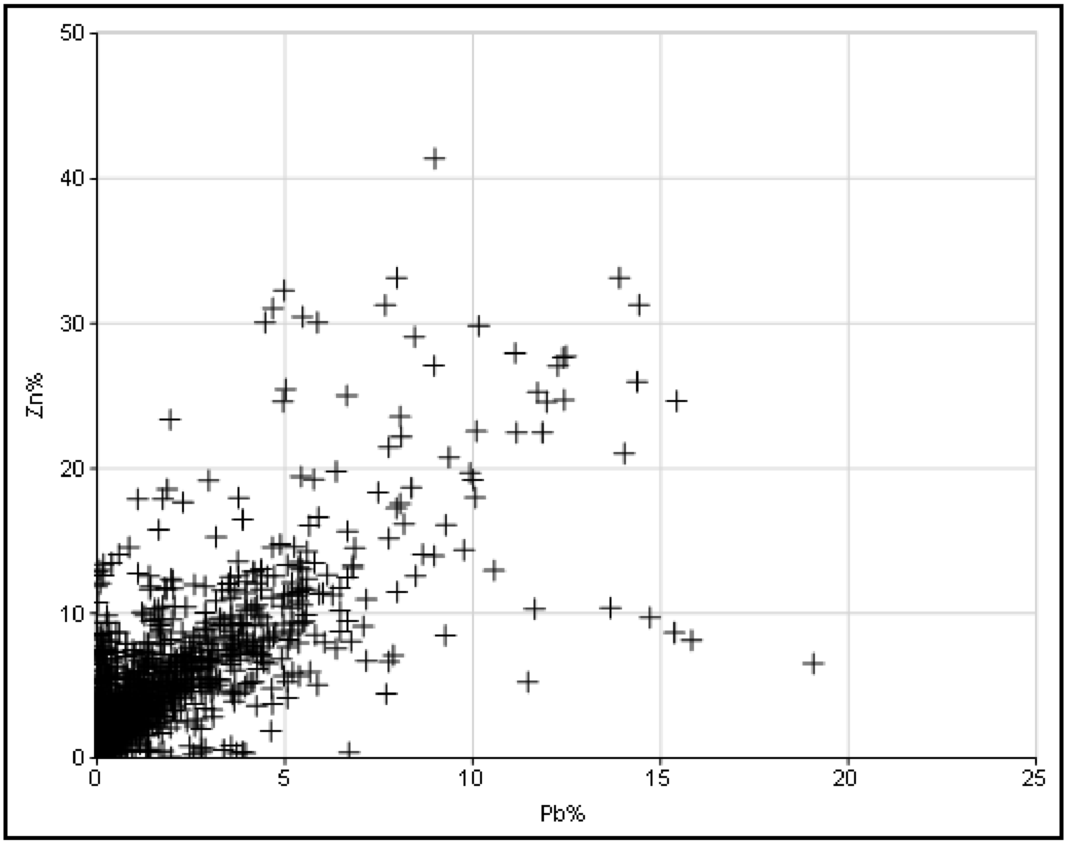
- zinc-lead-silver mineralization
- copper-cobalt mineralization
- gold mineralization.

These three categories are commonly, but not necessarily spatially independent of each other.

### LEAD, ZINC AND SILVER

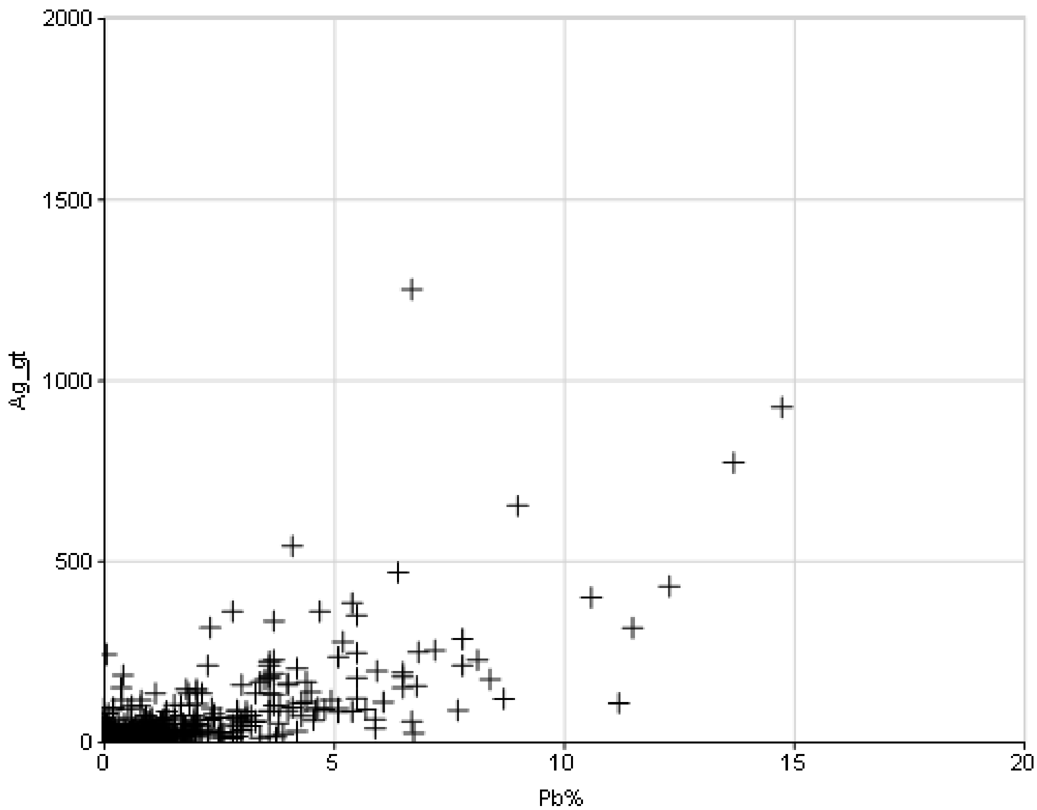
Lead and zinc show a strong correlation due to the intimate relationship of sphalerite and galena in the massive sulphide mineralization. This correlation is demonstrates in the scatterplot below (Figure 14.25).

**Figure 14.25 CNE Lead vs Zinc Assay Scatterplot**




As is expected, silver shows the strongest correlation with lead, as demonstrated in the scatterplot below (Figure 14.26). Note that there are significantly fewer silver assays to compare with lead and/or zinc.

**Figure 14.26 CNE Lead vs Silver Scatterplot**

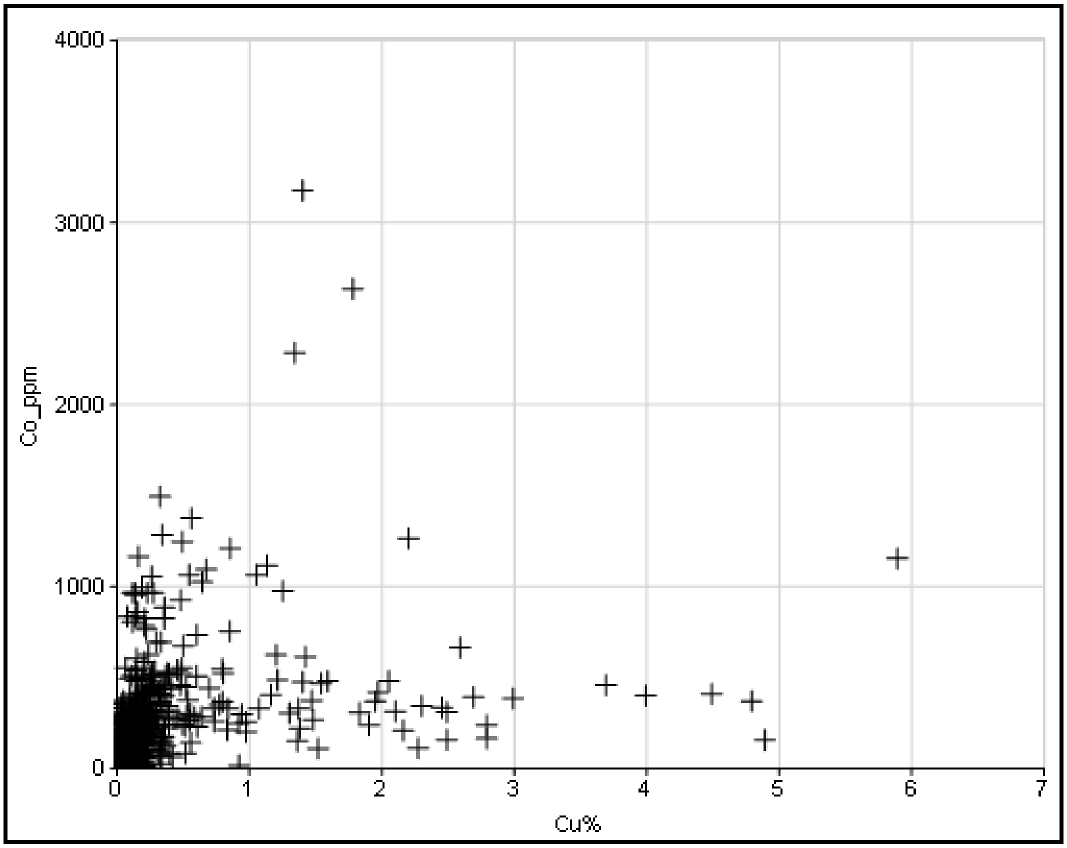



Lead, zinc and silver mineralization, in particular the high-grade, massive sulphide component, is extremely difficult to follow from section to section. In the process of deformation, massive sulphides can behave very fluid, and form contorted lensoid shapes which can be subsequently disengaged through local faulting.

### COPPER AND COBALT

Copper and cobalt show a positive correlation (Figure 14.27). The positive correlation is most prominent at lower grade concentrations. Higher grade concentrations are probably influenced by secondary processes (e.g. folding and metamorphism), independent of primary metal concentrating process (e.g. hydrothermal VMS activity).

**Figure 14.27 CNE Copper vs Cobalt Scatterplot**

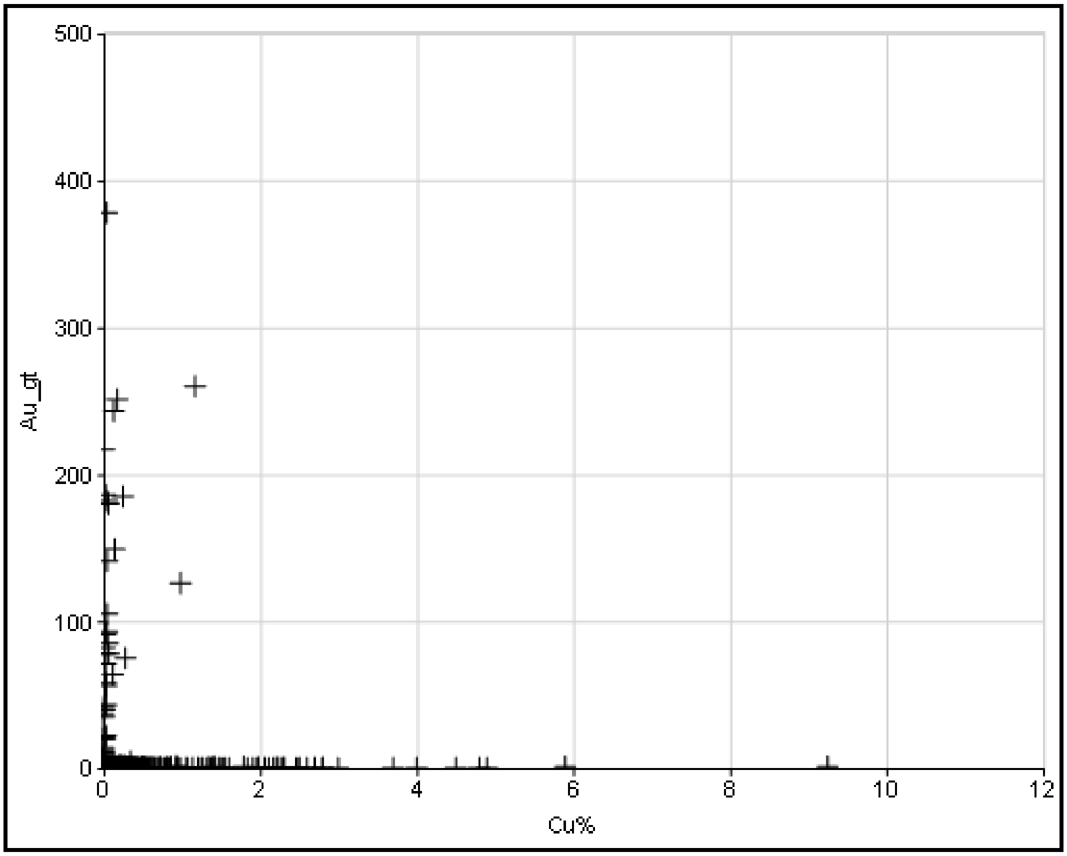



GOLD

In contrast to both copper-cobalt and lead-zinc-silver, gold occurs independently; probably more closely associated with pyrite mineralization. Of all metals, gold is usually most likely to be associated with copper (e.g. copper-gold porphyry deposits and IOCG deposits). This association does not exist, even though some high-grade gold assays have been recorded (see scatterplot Figure 14.28).

Gold's distribution within the CNE deposit is much more (spatially) evenly distributed, in contrast to both the copper-cobalt and zinc-lead-silver systems. Its sporadic occurrence, combined with an incomplete database for gold, prevented interpolation of the metal within the deposit at this stage. However, the metal should not be discounted. It represents a future "up-side" to the deposit if it can be more accurately modelled.

**Figure 14.28 CNE Gold vs Copper Scatterplot**




### INDICATOR KRIGING

IK is used to select blocks within the model for interpolation of grade. For the CNE deposit, two domains were recognized demonstrating sufficient mineralization continuity to warrant interpolation; the lead-zinc-silver domain, and the copper-cobalt domain. IK, in particular, is very applicable in domains where the geology and data are too (spatially) complex to model by traditional (sectional) means. This is because the IK process relies on the spatial distribution of data across all three dimensions, whereas complex sectional interpretation is most effective in two dimensions (e.g. east-west, north-south or planar).

## 14.2.5 *STATISTICS AND EXPLORATORY DATA ANALYSIS*

### DATA STATISTICS

Statistics were generated from data comprising the CNE volume of interest as determined by IK. These represent assay data from 68 drillholes. These statistics are tabulated below (Table 14.10).

Over 99.3% of the drillhole sample data were assayed for lead, zinc, silver and copper, whereas less than 50% of the sample data were assayed for cobalt and gold.

The lack of data for the minor metals, cobalt and gold resulted in treating resource classification for these metals differently than for the base metals (copper, lead and zinc) and silver.

**Table 14.10 Data Statistics Showing Raw Drillhole Data (Raw), Raw Data with Capping Applied (tc) and Capped and Composited Data (tc-c)**

| Field | | Number of Records | Number of Samples | Number of Missing Values | Minimum | Maximum | Mean | Variance | Standard Deviation | Standard Error | Skewness | Kurtosis | Geomean |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| raw | Zn% | 3880 | 3874 | 6 | 0 | 41.31 | 2.337 | 16.678 | 4.084 | 0.066 | 3.694 | 18.215 | 0.760 |
| raw | Pb% | 3880 | 3873 | 7 | 0 | 19.1 | 0.755 | 3.211 | 1.792 | 0.029 | 4.065 | 21.143 | 0.067 |
| raw | Ag_gt | 3880 | 3738 | 142 | 0 | 1250.2 | 29.04 | 4,740 | 68.85 | 1.126 | 7.680 | 91.06 | 9.655 |
| raw | Cu% | 3880 | 3854 | 26 | 0 | 25 | 0.147 | 0.399 | 0.631 | 0.010 | 21.277 | 692.192 | 0.032 |
| raw | Co_ppm | 3880 | 1618 | 2262 | 1 | 3170 | 108 | 44,834 | 212 | 5.264 | 5.615 | 52.336 | 35.822 |
| raw | Au_gt | 3880 | 2031 | 1849 | 0 | 378 | 2.128 | 320.737 | 17.909 | 0.397 | 12.618 | 186.490 | 0.083 |
| raw | Length | 3880 | 3880 | 0 | 0.1 | 27.9 | 1.155 | 6.186 | 2.487 | 0.040 | 8.662 | 79.087 | 0.824 |
| tc | Zn% | 3880 | 3874 | 6 | 0 | 33 | 2.334 | 16.503 | 4.062 | 0.065 | 3.620 | 17.143 | 0.760 |
| tc | Pb% | 3880 | 3873 | 7 | 0 | 16 | 0.755 | 3.184 | 1.784 | 0.029 | 3.998 | 20.051 | 0.067 |
| tc | Ag_gt | 3880 | 3738 | 142 | 0 | 500 | 28.02 | 3,301 | 57.46 | 0.940 | 4.627 | 27.32 | 9.64 |
| tc | Cu% | 3880 | 3854 | 26 | 0 | 3 | 0.131 | 0.119 | 0.345 | 0.006 | 5.638 | 36.281 | 0.032 |
| tc | Co_ppm | 3880 | 1618 | 2262 | 1 | 1500 | 106 | 35,780 | 189 | 4.703 | 3.627 | 16.766 | 35.784 |
| tc | Au_gt | 3880 | 2031 | 1849 | 0 | 120 | 1.608 | 126.166 | 11.232 | 0.249 | 9.094 | 85.579 | 0.083 |
| tc | Length | 3880 | 3880 | 0 | 0.1 | 27.9 | 1.155 | 6.186 | 2.487 | 0.040 | 8.662 | 79.087 | 0.824 |
| tc-c | Zn% | 4343 | 4319 | 24 | 0 | 32.844 | 3.450 | 16.074 | 4.009 | 0.061 | 2.293 | 9.651 | 1.403 |
| tc-c | Pb% | 4343 | 4316 | 27 | 0 | 16 | 1.251 | 3.095 | 1.759 | 0.027 | 2.362 | 9.499 | 0.201 |
| tc-c | Ag_gt | 4343 | 4238 | 104 | 0 | 500 | 46.31 | 3,319 | 57.61 | 0.885 | 2.344 | 9.881 | 19.02 |
| tc-c | Cu% | 4343 | 4144 | 199 | 0 | 3 | 0.116 | 0.082 | 0.286 | 0.004 | 6.139 | 45.442 | 0.038 |
| tc-c | Co_ppm | 4343 | 878 | 3465 | 2 | 1500 | 100 | 29,073 | 171 | 5.754 | 3.326 | 15.217 | 35.357 |
| tc-c | Au_gt | 4343 | 2276 | 2067 | 0 | 120 | 0.863 | 53.888 | 7.341 | 0.154 | 13.251 | 183.056 | 0.145 |
| tc-c | Length | 4343 | 4343 | 0 | 0.5 | 1 | 0.997 | 0.001 | 0.035 | 0.001 | -11.800 | 144.545 | 0.996 |

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### OUTLIER MANAGEMENT

The primary minerals within the CNE deposit include copper, cobalt, lead, zinc, silver and gold. The assay statistics of these metals were interrogated to investigate the need for controlling the outlier (high-grade) sample populations. Outliers invariably conform to massive sulphide positions, especially in the cases of copper, lead and zinc. Thus domaining of the massive sulphide positions with the use of indicator wireframes will to some extent control the influence of these outliers. Extreme values, however, require “capping”, also referred to as “top-cutting”.

Note that cobalt was determined to not be of economic value at this time.

The copper-cobalt scatter plot (Figure 14.27) clearly demonstrates extreme cobalt values above 1500 ppm, whereas copper indicates a clear break in grade continuity above 3%. Thus capping values of 1500 ppm and 3% Cu were chosen for the sample assays of the raw data.

Both lead and zinc show good grade continuity with the exception of extreme values of 18% Pb and 41% Zn (see Figure 14.25). Thus caps of 16% Pb and 33% Zn were applies to the raw sample assay data.

The continuity of grade in the silver assays begins to become erratic at approximately 500 g/t (see Figure 14.26). Thus this value was chosen as the top-cut value.

Gold shows little, if any, correlation with any of the other metals. However, gold records some exceptionally high values up to 378 g/t. Subsequent discussions with Stratabound suggest that there may be assay errors with the database. However, as gold is not interpolated in this model, this is not an immediate issue, but should be addressed for any subsequent model updates.

The continuity of the gold grade appears to become less cohesive above 120 g/t. Thus this value was chosen as the top-cut for gold for the drillhole database for future reference.

The raw assay values were capped respectively as tabulated below (Table 14.11).

**Table 14.11 Top-cuts (capping) Applied to the Raw Assay Data File**

| Metal | Top-cut |
|---|---:|
| Copper | 3% |
| Cobalt | 1500 ppm |
| Lead | 16% |
| Zinc | 33% |
| Silver | 500 g/t |
| Gold | 120 g/t |

### SAMPLE COMPOSITING

Drillhole assay samples range in lengths from 0.1 m to 27.9 m in length. The average length is 1.155 m. With a cell block size of only 2 $m^3$, a composited sample length of 1 m is considered appropriate to retain resolution and to maximize the number of samples available for interpolation. In such cases, where a large drillhole length was bulk sampled and a single assay generated, the sample lengths are divided into 1 m sections with identical assays.

### 14.2.6 SPECIFIC GRAVITY

A re-sampling and reanalysis was conducted on 29 samples of the 2009 and 2010 diamond drill core (rock pulp) for the CNE deposit. This included specific gravity measurements. The reanalysis was completed by ACME Laboratories of Vancouver, BC (VAN10004978 Certificate 2) and the specific gravity measurements were conducted by the same lab. Specific gravity of the distinct lithologies was taken on several distinct lithologies and mineralized sections of core of the CNE deposit. Results are tabulated below (Table 14.12).

**Table 14.12 CNE Density Measurements**

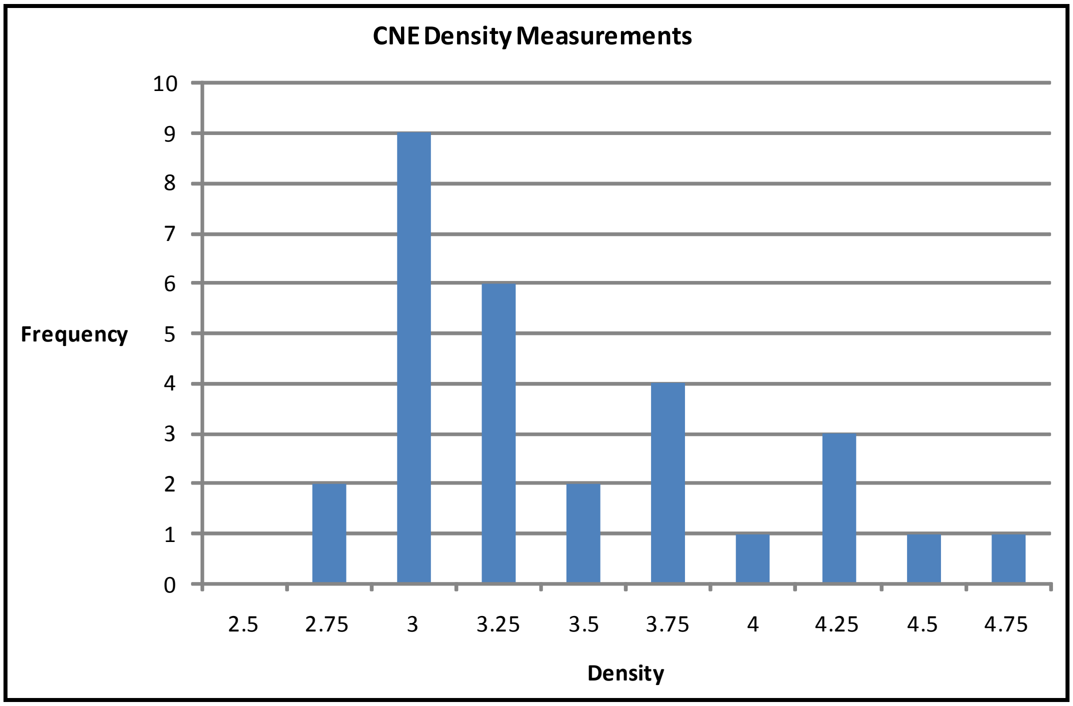
| Grade or Rock Code | Drillhole | Density |
|---|---|---|
| Zn/Pb ore | CNE-09-01 | 3.59 |
| Zn/Pb ore | CNE-09-02 | 3.6 |
| Zn/Pb ore | CNE-09-04 | 2.51 |
| Zn/Pb med | CNE-09-02 | 4.74 |
| Zn/Pb med | CNE-10-10 | 2.98 |
| Zn/Pb high | CNE-09-03 | 4.32 |
| Zn/Pb high | CNE-09-03 | 4.04 |
| Zn/Pb low | CNE-09-03 | 2.84 |
| Zn/Pb low | CNE-09-03 | 3.51 |
| Zn/Pb low | CNE-09-04 | 3.76 |
| Cu/Co high | CNE-09-02 | 3.18 |
| Cu/Co high | CNE-09-02 | 3.22 |
| Cu/Co med | CNE-09-02 | 2.94 |
| Cu/Co med | CNE-09-02 | 3.07 |
| Cu/Co med | CNE-09-02 | 3.41 |
| Cu/Co low | CNE-09-02 | 3.15 |
| Cu/Co low | CNE-09-02 | 3.07 |
| Low Cu high Co | CNE-09-04 | 3.22 |
| Low Cu high Co | CNE-09-04 | 3.37 |
| Med Cu high Co | CNE-09-04 | 4.12 |
| Semi mass py/Co | CNE-10-14 | 3.61 |

*table continues...*

| Grade or Rock Code | Drillhole | Density |
|---|---|---|
| Semi mass py/Co | CNE-10-14 | 4.21 |
| FFA rhy | CNE-10-05 | 2.85 |
| FFA rhy | CNE-10-15 | 2.82 |
| FFA rhy | CNE-10-12 | 2.74 |
| FTA ash-tuff | CNE-10-05 | 2.8 |
| FTA ash-tuff | CNE-10-13 | 2.79 |
| FTC xtl tuff | CNE-10-05 | 2.79 |
| FTC xtl tuff | CNE-10-05 | 2.95 |

Density variability can also be depicted as a histogram (see below, Figure 14.29).

**Figure 14.29 CNE Density Measurements Histogram**




From inspection, density within mineralized rocks is a function of sulphide content, notwithstanding the prevalence of un-mineralized pyrite. However, insufficient density measurements preclude the ability to directly estimate specific gravity into the CNE block model. To accommodate density, average values were assigned to individual blocks, after grade interpolation, depending on rock type (un-mineralized blocks) or metal content (mineralized blocks). The assigned densities and their conditions are tabulated below (Table 14.13).

**Table 14.13 Specific Gravity and Rock Codes**

| Wireframe | Specific Gravity | Rock Code |
|---|---|---|
| Air | 0 | 0 |
| Overburden | 2.00 | 1 |
| Country and Host Rock (unmineralized) | 2.80 | 2 |
| High Grade Copper (> 4.5% Cu) | 3.23 | 2 |
| Low Grade Zinc (>2% and <4.5% Zn) | 3.37 | 2 |
| High Grade Zinc (>4.5% Zn) | 3.86 | 2 |

### 14.2.7 *SPATIAL ANALYSIS - VARIOGRAPHY*

Variography was conducted using Datamine™ software. Samples used for variography are a function of geological interpretation (domaining), assaying, data capping and compositing.

Variography was performed on copper, zinc and lead. An incomplete dataset, relative to copper, lead and zinc, for silver and cobalt, precluded respective variography. No variography was performed on gold as there was no intention of estimating gold into the block model.

Variograms are presented in Appendix G, and a summary table of variogram parameters is presented below (Table 14.14).

Downhole variograms, using a lag distance equal to the composite length, were created for each of the separate domains to determine the intrinsic sample variance ("nugget"). Lag distances for constructing experimental variograms were typically short, reflecting the close-spaced data populations. The number of lags varied, but usually 10 to 20 lags sufficed to cover most of the deposit. All variograms utilized 30° directional increments and +15° tolerance to optimize orientations.

Modelled variograms provided a "best fit" of the experimental variograms while maintaining same statistics (e.g. experimental total variance is equal to the modelled total variance).

## Table 14.14 Variogram Parameters

| Description | VANGLE1 | VANGLE2 | VANGLE3 | VAXIS1 | VAXIS2 | VAXIS3 | NUGGET |
|---|---|---|---|---|---|---|---|
| Zn_ind_3 | 0 | 0 | 120 | 3 | 2 | 1 | 0.02 |
| Cu_ind_02 | -30 | 0 | 60 | 3 | 2 | 1 | 0.215 |
| Zn_ok | -60 | 0 | 30 | 3 | 2 | 1 | 0.14 |
| Pb_ok | -60 | 0 | 120 | 3 | 2 | 1 | 0.278 |
| Cu_ok | -30 | 0 | 60 | 3 | 2 | 1 | 0.1 |
| Ag_ok | -30 | | 120 | 3 | 2 | 1 | 0.4 |

| Description | ST1PAR1 | ST1PAR2 | ST1PAR3 | ST1PAR4 | ST2PAR1 | ST2PAR2 | ST2PAR3 | ST2PAR4 |
|---|---|---|---|---|---|---|---|---|
| Zn_ind_3 | 12 | 12 | 15 | 0.5 | 12 | 35 | 18 | 0.48 |
| Cu_ind_02 | 5 | 5 | 7 | 0.518 | 6 | 15 | 18 | 0.267 |
| Zn_ok | 12 | 20 | 16 | 0.86 | - | - | - | - |
| Pb_ok | 11 | 31 | 25 | 0.722 | - | - | - | - |
| Cu_ok | 12 | 11 | 16 | 0.9 | - | - | - | - |
| Ag_ok | 10 | 15 | 20 | 0.3 | 20 | 30 | 40 | 0.3 |

### 14.2.8 THE CNE BLOCK MODEL

The parameters which define the CNE block model are as follows:

Model Origin: 281750 mE, 5241860 mN, 20 mRL. Note that the model origin as defined in Datamine™ is the lowest southwestern point of the block model. The model is not rotated.

Parent cell size: 2 m x 2 m x 2 m. Sub-cells are not used except to better define the surface (air-overburden interface). It is believed that the 2 $m^3$ cell-size provides the best resolution for mineralization, given the overall close data (assay) density.

Number of cells to the east: 175; number of cells to the north: 150; number of cells in rising elevation: 80.

The total volume represented by the CNE block model is 16.8 $Mm^3$, which includes fresh rock, overburden and “air”.

#### WIREFRAMES

Wireframes generated to model the geology of the deposit include a topographic “surface” DTM wireframe and a DTM wireframe to define the base of overburden; the interface between overburden and fresh rock. These wireframes were constructed using drillhole collar point data and downhole logging data respectively.

A wireframe was constructed to define the portion of the deposit which had been previously mined out as a small open pit to the northeast of the deposit. This wireframe was constructed as a DTM based on point survey data. There are no accompanying flitch plans available to better define the positions of pit walls and extent of the pit floor. Thus the resultant wireframe must be considered a “minimum” total pit volume. This wireframe is used to define “mined out” material and is used to assist in ore reconciliation.

As mentioned previously, a closed wireframe volume was designed to define and control the extent of viable blocks available for grade interpolation and extrapolation. The enclosing volume of this wireframe is 1,001,381 4 $m^3$, which represents 6% of the total block model volume.

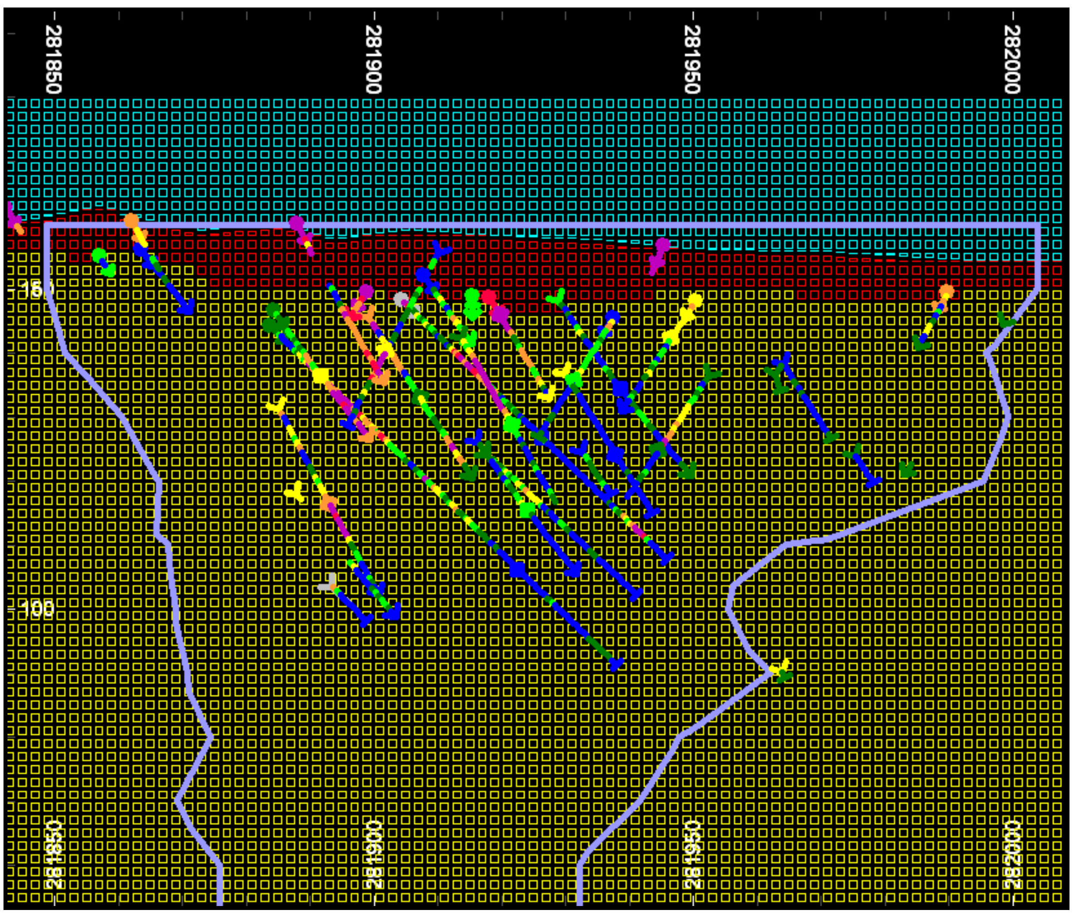
The details of wireframes used in the block model are tabulated below (Table 14.15).

**Table 14.15 Wireframes of the CNE Deposit Block Model**

| Name | Description | Type |
|---|---|---|
| Cne_topo2tr/pt | Defines topographic surface | DTM based on drill collar survey data |
| Overbtr/pt | Defines base of overburden | DTM based on downhole drill logging data |
| Pittr/pt | Defines mined out portion of the deposit | DTM based on pit survey point data |
| Boundarytr/pt | Defines extent of mineralization | Closed wireframe volume based on proximal drillhole data |

The following image depicts the block model at 5242020 mN, showing blocks coloured by rock type (blue is air, red is overburden and yellow is fresh rock). It also outlines the “boundary” wireframe which limits grade extrapolation. Drillholes are coloured by zinc with hotter colours reflecting higher grades. Note sub-cells (in the z-direction) are limited to the surface interface.

**Figure 14.30 Cross Section of Block Model at 5242020 mN Coloured by Rock Type**



### INTERPOLATION PLAN

The interpolation plan for the resource block model consists of three parts; the estimation parameter file (designated as .epar file in Datamine™), the search parameter file (designated as .spar file in Datamine™), and the variogram parameter file (designated as .vpar in Datamine™). The variogram parameter file is tabulated above (Table 14.14), whereas the estimation and search parameter files are tabulated in Table 14.16 and Table 14.17, respectively.

**Table 14.16 Estimation Parameter File**

| Description | VALUE_IN | VALUE_OU | SREFNUM | NUMSAM_F | SVOL_F | VAR_F |
|---|---|---|---|---|---|---|
| Zn_ind_3 | Zn% | Zn_ind | 1 | NUMSAM | SVOL | KVAR |
| Cu_ind_02 | Cu% | Cu_ind | 1 | - | - | - |
| Zn_ok | Zn% | Zn% | 1 | NUMSAM | class_zn | KVAR |
| Pb_ok | Pb% | Pb% | 1 | - | - | - |
| Ag_ok | Ag_gt | Ag_gt | 3 | - | - | - |
| Zn_id | Zn% | Zn_id | 1 | - | - | - |
| Pb_id | Pb% | Pb_id | 2 | - | - | - |
| Zn_nn | Zn% | Zn_nn | 1 | - | - | - |
| Pb_nn | Pb% | Pb_nn | 2 | - | - | - |
| Cu_ok | Cu% | Cu% | 1 | NUMSAM | class_cu | KVAR |
| Ag_nn | Ag_gt | Ag_nn | 3 | - | - | - |
| Cu_id | Cu% | Cu_id | 1 | - | - | - |
| Cu_nn | Cu% | Cu_nn | 1 | - | - | - |

| Description | MINDIS_F | IMETHOD | POWER | TOL | Cut-off |
|---|---|---|---|---|---|
| Zn_ind_3 | MINDIS | 3 | 2 | 0.01 | 3 |
| Cu_ind_02 | - | 3 | 2 | 0.01 | 0.2 |
| Zn_ok | MINDIS | 3 | 2 | 0.01 | 0 |
| Pb_ok | - | 3 | 2 | 0.01 | 0 |
| Ag_ok | - | 3 | 2 | 0.01 | 0 |
| Zn_id | - | 2 | 2 | 0.01 | 0 |
| Pb_id | - | 2 | 2 | 0.01 | 0 |
| Zn_nn | - | 1 | 2 | 0.01 | 0 |
| Pb_nn | - | 1 | 2 | 0.01 | 0 |
| Cu_ok | MINDIS | 3 | 2 | 0.01 | 0 |
| Ag_nn | - | 1 | 2 | 0.01 | 0 |
| Cu_id | - | 2 | 2 | 0.01 | 0 |
| Cu_nn | - | 1 | 2 | 0.01 | 0 |

Note: SREFNUM is the search reference number. IMETHOD 3 is Ordinary Kriging. IMETHOD 2 id Inverse Distance (squared). IMETHOD 1 is Nearest Neighbour. POWER only applies to IMETHOD 2. CUTOFF is for Indicator Kriging. See Table 14.18 for other attribute definitions.

**Table 14.17 Search Parameter File**

| Definition | SREFNUM | SDIST1 | SDIST2 | SDIST3 | SANGLE1 | SANGLE2 | SANGLE3 |
|---|---|---|---|---|---|---|---|
| Zn_ind_3 | 1 | 12 | 35 | 18 | 0 | 0 | 120 |
| Cu_ind_02 | 1 | 4 | 12 | 14 | -30 | 0 | 60 |
| Zn_ok | 1 | 6 | 14 | 10 | -60 | 0 | 30 |
| Pb_ok | 1 | 6 | 14 | 10 | -60 | 0 | 120 |
| Zn_id_nn | 1 | 6 | 14 | 10 | 0 | 0 | 120 |
| Pb_id_nn | 2 | 6 | 14 | 10 | 0 | 0 | 120 |
| Ag_ok | 3 | 6 | 14 | 10 | -30 | 0 | 120 |
| Cu_ok | 1 | 4 | 12 | 14 | -30 | 0 | 60 |
| Ag_nn | 3 | 6 | 14 | 10 | -30 | 0 | 120 |

| Definition | SAXIS1 | SAXIS2 | SAXIS3 | MINNUM1 | MAXNUM1 | SVOLFAC2 |
|---|---|---|---|---|---|---|
| Zn_ind_3 | 3 | 2 | 1 | 12 | 20 | 1.5 |
| Cu_ind_02 | 3 | 2 | 1 | 12 | 20 | 2 |
| Zn_ok | 3 | 2 | 1 | 12 | 20 | 2 |
| Pb_ok | 3 | 2 | 1 | 12 | 20 | 2 |
| Zn_id_nn | 3 | 2 | 1 | 12 | 20 | 2 |
| Pb_id_nn | 3 | 2 | 1 | 12 | 20 | 2 |
| Ag_iok | 3 | 2 | 1 | 12 | 20 | 2 |
| Cu_ok | 3 | 2 | 1 | 12 | 20 | 2 |
| Ag_nn | 3 | 2 | 1 | 12 | 20 | 2 |

| Definition | MINNUM2 | MAXNUM2 | SVOLFAC3 | MINNUM3 | MAXNUM3 | MAXKEY |
|---|---|---|---|---|---|---|
| Zn_ind_3 | 8 | 20 | 2 | 6 | 20 | 3 |
| Cu_ind_02 | 8 | 20 | 3 | 6 | 20 | 3 |
| Zn_ok | 8 | 20 | 3 | 6 | 20 | 3 |
| Pb_ok | 8 | 20 | 3 | 6 | 20 | 3 |
| Zn_id_nn | 8 | 20 | 3 | 6 | 20 | 3 |
| Pb_id_nn | 8 | 20 | 3 | 6 | 20 | 3 |
| Ag_ok | 8 | 20 | 3 | 6 | 20 | 3 |
| Cu_ok | 8 | 20 | 3 | 6 | 20 | 3 |
| Ag_nn | 8 | 20 | 3 | 6 | 20 | 3 |

Note: SREFNUM is the search reference number. SDIST (1-3) are sample search radii in metres. SANGLE (1-3) are rotation angles around SAXIS (1-3) (1 = x-axis, 2 = y-axis and 3 = z-axis). MINNUM (1-3), MAXNUM (1-3) and SVOLFAC (1-3) are the minimum number of samples, the maximum number of samples with respect to the search volume expansion factor (search pass). MAXKEY is the maximum number of samples allowed per drillhole for cell estimation.

### BLOCK ATTRIBUTES

The following table (Table 14.18) details the attributes estimated and calculated into the CNE block model.

### Table 14.18 CNE Block Model Attributes List

| Attribute | Description | Unit | Details |
|---|---|---|---|
| IJK | Unique parent cell code | Integer | Sub-cells may have the same code |
| XC | Cell Centroid (X) | m | Easting |
| YC | Cell Centroid (Y) | m | Northing |
| ZC | Cell Centroid (Z) | m | Elevation |
| XINC | Cell dimension (X) | m | Easting |
| YINC | Cell dimension (Y) | m | Northing |
| ZINC | Cell dimension (Z) | m | Elevation |
| ZONE | Not used | | Datamine feature |
| density | Specific gravity | g/cc | Estimate |
| rock | rock code | Integer | 0=air, 1=overburden, 2=fresh |
| mstatus | Mined status | Integer | 1=mined out, 2=not mined |
| class_zn | Zn domain resource classification | Integer | 1=Measured, 2=Indicated, 3=Inferred |
| class_cu | Cu domain resource classification | Integer | 1=Measured, 2=Indicated, 3=Inferred |
| Zn_ind | Initial Zn indicator | 0-1 | Not used |
| Pb% | Pb Ordinary Kriged estimate | % | Zn-Pb-Ag domain |
| Ag_gt | Ag Ordinary Kriged estimate | ppm | Zn-Pb-Ag domain |
| Cu% | Cu Ordinary Kriged estimate | % | Cu-Co domain |
| Ag_nn | Ag Nearest Neighbour estimate | ppm | Zn-Pb-Ag domain |
| Cu_ind | Initial Cu indicator | 0-1 | Not used |
| Zn% | Zn Ordinary Kriged estimate | % | Zn-Pb-Ag domain |
| PRAB1 | Proportion of cell above cut-off | 0-1 | Used in Indicator Kriging |
| NUMSAM | Number of samples | Integer | Used in estimate |
| SVOL | Search Volume | Integer | Estimation pass (1-3) |
| MINDIS | Minimum sample distance | Function | Samples estimation distances |
| KV | Kriging Variance | Function | Estimation variance |
| GRAB1 | Cell grade above cut-off | Estimate | Calculated as average between cut-offs |
| ZN3 | Zn 3% Indicator | 0-1 | Used to flag Zn (3%) Indicator cells |
| CU2 | Cu 0.2% Indicator | 0-1 | Used to flag Cu (0.2%) Indicator cells |
| Zn_id | Zn Inverse Distance estimate | % | Zn-Pb-Ag domain |
| Pb_id | Pb Inverse Distance estimate | % | Zn-Pb-Ag domain |
| Zn_nn | Zn Nearest Neighbour estimate | % | Zn-Pb-Ag domain |
| Pb_nn | Pb Nearest Neighbour estimate | % | Zn-Pb-Ag domain |
| ZN4 | Zn 4% Indicator | Integer | Used to flag Zn (4%) Indicator cells |
| Cu_id | Cu Inverse Distance estimate | % | Cu-Co domain |
| Cu_nn | Cu Nearest Neighbour estimate | % | Cu-Co domain |
| XMORIG | Block model origin | m | Easting |
| YMORIG | Block model origin | m | Northing |
| ZMORIG | Block model origin | m | Elevation |
| NX | Number cells | Integer | Easting |
| NY | Number cells | Integer | Northing |
| NZ | Number cells | Integer | Elevation |

### BLOCK MODEL GEOLOGY

The geology of the deposit was assigned by means of two wireframes; the topographic surface wireframe and the base of overburden wireframe. The "rock" attribute of zero was assigned to "air" cells (cells above the topographic surface); one was assigned to overburden and two was assigned to fresh rock.

### INDICATOR BLOCK MODELLING

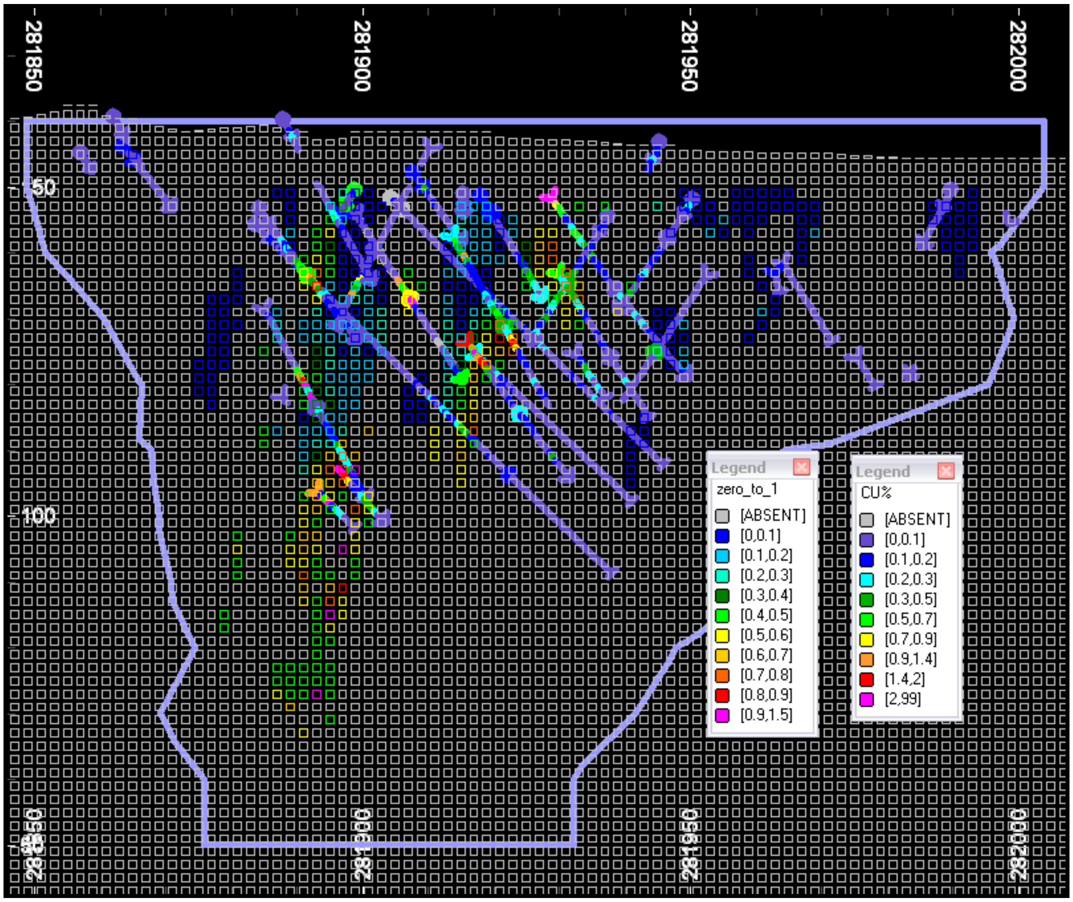
Indicator modelling is ideally suited to deposits, like CNE, which are geologically complex with an abundance of data. These deposits are commonly too complex to manually wireframe estimation domains. The Indicator process involves assigning a probability, which can be equally interpreted to be a proportion, of cells which can achieve or exceed a given cut-off.

For the zinc-lead-silver domain, the assigned indicator cut-off is 3% Zn. For the copper-cobalt domain, the assigned indicator is 0.2% Cu. This method is used to select cells for grade estimation. It is not used to assign grades to cells.

Composited drillhole zinc and copper data are assigned a value of one or zero depending on whether the meet and exceed the cut-off (one, or 100%) or the fall below the cut-off (zero, or 0%). Variography is then performed on the conditioned data, and appropriate, respective variograms modelled. The modelled variography is then used in conjunction with sample search and selection parameters to interpolate probability (or proportionality) into the cells. Thus each cell is assigned a number between zero and one which defines the probability of that cell meeting or exceeding the nominated cut-off. Alternatively, the same number can be considered a proportion of the block which meets or exceeds the cut-off.

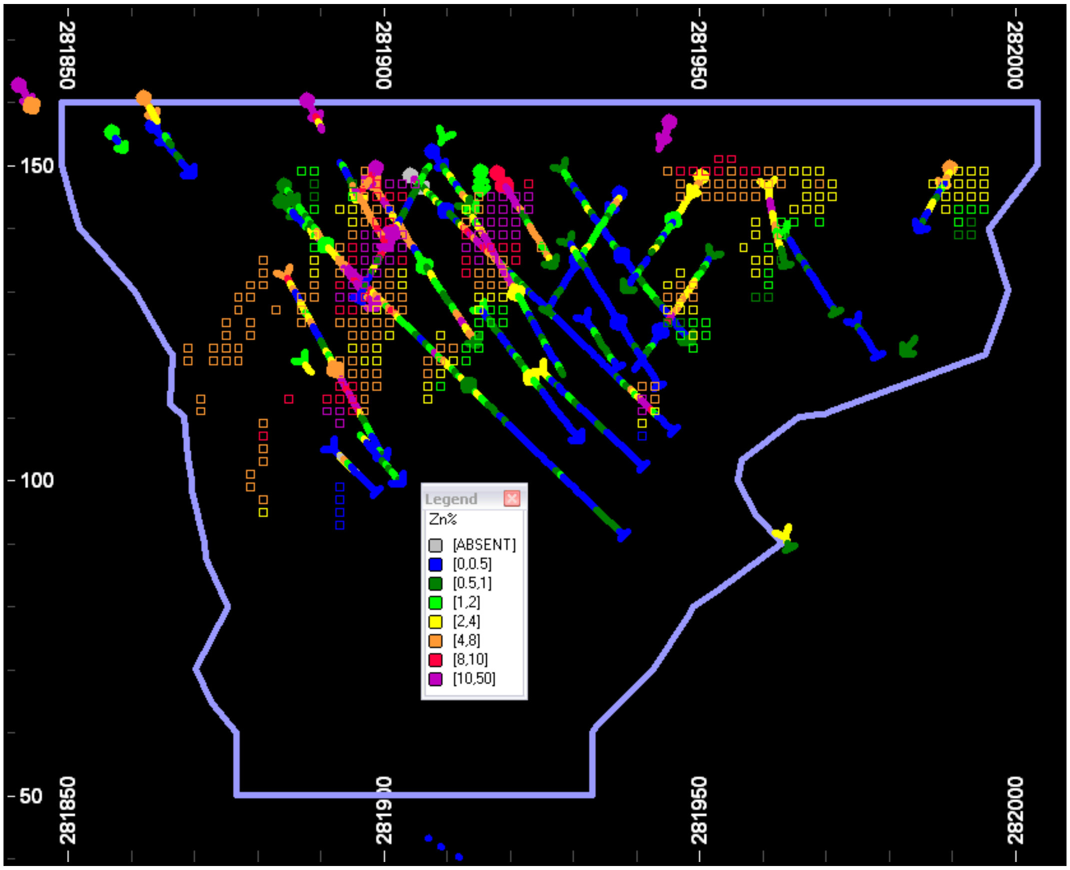
An example section of the indicator model is presented in Figure 14.31.

**Figure 14.31 Block Model Cross Section at 5242020 mN Showing Blocks Coloured by Probability of Cu Greater or Equal to 0.2% (Drillholes are Coloured by Grade Cu%)**




Some cells share both the copper-cobalt domain and the zinc-lead-silver domain. Where they do, each cell records the details of all interpolation estimates. The cells of the zinc-lead-silver domain are depicted below (Figure 14.32) as a function of estimated grade. Compare the distribution of zinc-lead-silver domain blocks with those of the copper-cobalt domain (Figure 14.31).

**Figure 14.32 Block Model Cross Section at 5242020 mN showing Zn-Pb-Ag Cells Coloured by Estimated Zn Grade (Drillholes are also Coloured by Zn Grade)**




## 14.2.9 RESOURCE ESTIMATION

Selected cells, based on the results of the IK with respect to assigned cut-off grades, were estimated using OK, ID2 and NN interpolation methods. Zinc, lead and copper were estimated using all three methods. Silver was estimated using NN and OK. Cobalt was only estimated using ID2 and NN methods due to insufficient samples to generate confident variography. The NN and ID2 estimation methods were used in block model validation for copper, zinc, lead and silver.

Copper and cobalt were estimated into selected (as per assigned cut-off) cells of the copper indicator block model. However, cobalt was never reported. Lead, zinc and silver were estimated into selected (as per assigned cut-off) cells of the zinc indicator block model. Some cells shared both the copper-cobalt domain and the zinc-lead-silver domain. Only these cells had estimated assigned for all five metals.

For each of the three estimation methods, three sequential estimation passes were run, based on increasing sample search distances accompanied by decreasing minimum sample requirements for successful cell interpolation. These are defined in the search parameter file tabulated in Table 14.17.

No metal was estimated into the overburden (rock = 1).

### 14.2.10 *Block Model Validation*

#### Statistics

The model statistics, with respect to each interpolation method (OK, ID2 and NN) for each of the estimated metals are tabulated below (Table 14.19).

The statistics correspond to grade estimated into cells above a minimal cut-off (1% Zn for zinc-lead cells and 0.1% Cu for copper cells). In general, there is a 2.3% difference between the OK mean and the ID2 mean for zinc and lead interpolations, a 0.3% difference between the OK mean and the ID2 mean for copper and a difference of 0.5% in the mean silver grade between OK and NN. ID2 was not used for estimating silver.

**Table 14.19 Block Model Statistics for Zn, Pb and Cu**

| Field | Zn% | Zn_id | Zn_nn | Pb% | Pb_id | Pb_nn | Cu% | Cu_id | Cu_nn |
|---|---|---|---|---|---|---|---|---|---|
| NRECORDS | 2,126,184 | 2,126,184 | 2,126,184 | 2,126,184 | 2,126,184 | 2,126,184 | 2,126,184 | 2,126,184 | 2,126,184 |
| NSAMPLES | 12,281 | 12,281 | 12,281 | 12,281 | 12,281 | 12,281 | 4,585 | 4,585 | 4,585 |
| NMISVALS | 2,113,903 | 2,113,903 | 2,113,903 | 2,113,903 | 2,113,903 | 2,113,903 | 2,121,599 | 2,121,599 | 2,121,599 |
| NUMTRACE | 12,281 | 12,224 | 12,224 | 12,270 | 12,224 | 12,143 | 4,585 | 4,585 | 4,576 |
| MINIMUM | 1.001 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.100 | 0.074 | 0.000 |
| MAXIMUM | 25.567 | 26.498 | 32.844 | 17.647 | 12.417 | 16.000 | 2.535 | 2.614 | 3.000 |
| **MEAN** | **4.868** | **5.344** | **6.245** | **1.803** | **1.980** | **2.330** | **0.558** | **0.551** | **0.550** |
| VARIANCE | 10.358 | 8.764 | 24.551 | 2.308 | 1.914 | 5.680 | 0.121 | 0.112 | 0.430 |
| STANDDEV | 3.218 | 2.960 | 4.955 | 1.519 | 1.384 | 2.383 | 0.348 | 0.334 | 0.656 |
| STANDERR | 0.029 | 0.027 | 0.045 | 0.014 | 0.012 | 0.022 | 0.005 | 0.005 | 0.010 |
| SKEWNESS | 1.887 | 1.753 | 2.094 | 1.723 | 1.524 | 1.992 | 1.551 | 1.513 | 1.874 |
| KURTOSIS | 5.244 | 5.199 | 6.674 | 4.429 | 4.214 | 5.864 | 3.001 | 3.119 | 3.081 |
| GEOMEAN | 4.036 | 4.719 | 4.321 | 1.242 | 1.531 | 1.073 | 0.471 | 0.467 | 0.238 |
| SUMLOG | 17,134 | 18,966 | 17,890 | 2,662 | 5,204 | 851 | -3,452 | -3,495 | -6,566 |
| MEANLOG | 1.395 | 1.552 | 1.464 | 0.217 | 0.426 | 0.070 | -0.753 | -0.762 | -1.435 |
| LOGVAR | 0.376 | 0.255 | 1.138 | 0.923 | 0.653 | 2.853 | 0.335 | 0.332 | 2.617 |
| LOGESTMN | 4.871 | 5.360 | 7.634 | 1.971 | 2.121 | 4.467 | 0.557 | 0.551 | 0.881 |

Note: Zn%, Pb% and Cu% are results of Ordinary Kriging estimate. Zn_id, Pb_id and Cu_id are the results of Inverse Distance estimate. Zn_nn, Pb_nn and Cu_nn are the results of Nearest Neighbour estimate.

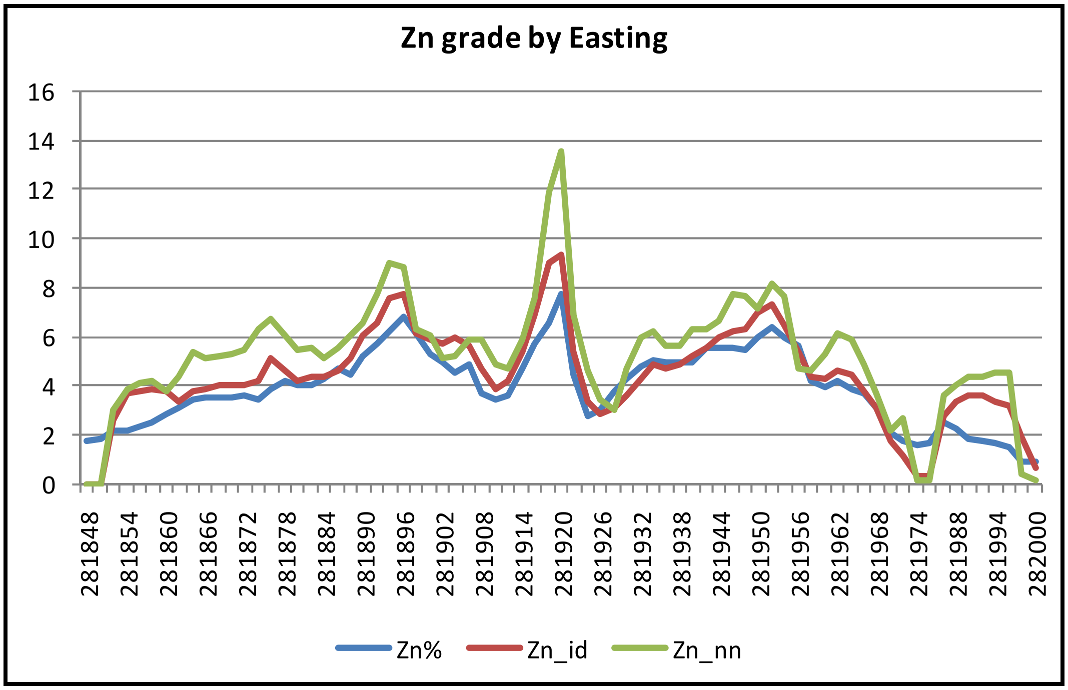
**Table 14.20 Block Model Statistics for Ag (continued)**

| Field | Ordinary Kriging | Nearest Neighbour |
|---|---|---|
| NRECORDS | 2,126,184 | 2,126,184 |
| NSAMPLES | 17,164 | 17,164 |
| NMISVALS | 2,109,020 | 2,109,020 |
| NUMTRACE | 12,829 | 12,706 |
| MINIMUM | -14.789 | 0.000 |
| MAXIMUM | 310 | 500 |
| RANGE | 325 | 500 |
| TOTAL | 819,460 | 970,827 |
| MEAN | 47.74 | 56.56 |
| VARIANCE | 2,331.78 | 6,162.87 |
| STANDDEV | 48.289 | 78.504 |
| STANDERR | 0.369 | 0.599 |
| SKEWNESS | 1.303 | 2.495 |
| KURTOSIS | 1.852 | 7.908 |
| GEOMEAN | 49.353 | 43.497 |
| SUMLOG | 50,020 | 47,936 |
| MEANLOG | 3.899 | 3.773 |
| LOGVAR | 0.563 | 1.397 |
| LOGESTMN | 65.415 | 87.464 |

### Swath Plots

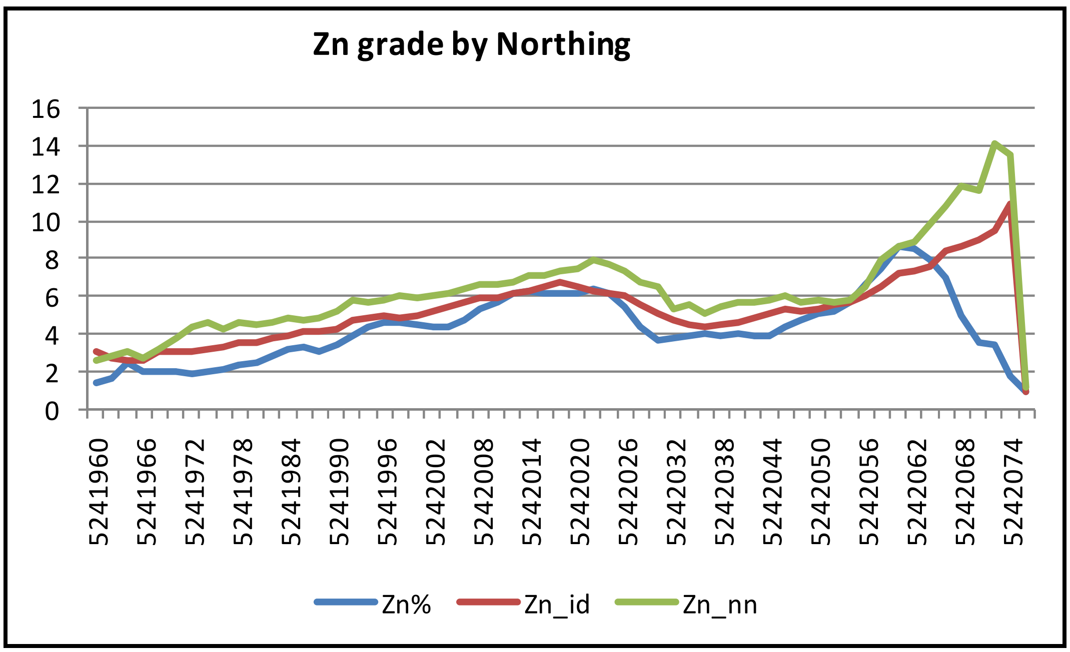
Swath plots, showing estimated cell grade variations across eastings, northings and elevations, were generated for the OK, ID2 and NN interpolation methods. These plots are depicted below. The plots demonstrate reasonable correlations for all interpolation techniques. Only zinc OK and lead OK, with respect to elevation, deviate significantly from the ID and NN estimates. Silver swath plots do not include ID2.

Figure 14.33 Zn Grade by Easting



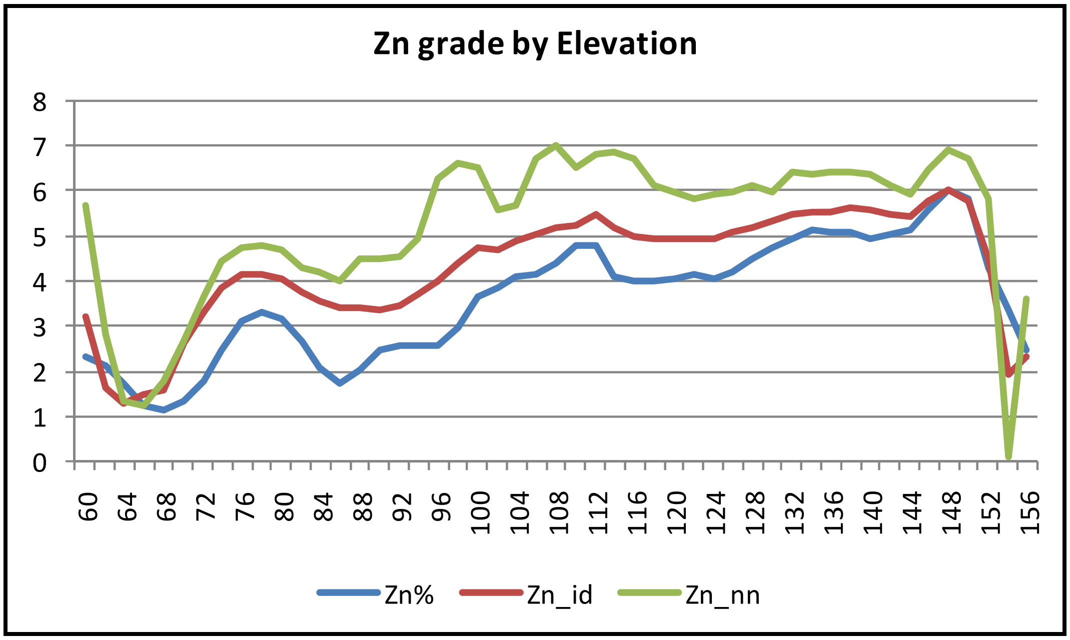

Note: Vertical axis in Zn%. Zn% - Ordinary Kriging; Zn_id – Inverse Distance; Zn_nn – Nearest Neighbour

Figure 14.34 Zn Grade by Northing



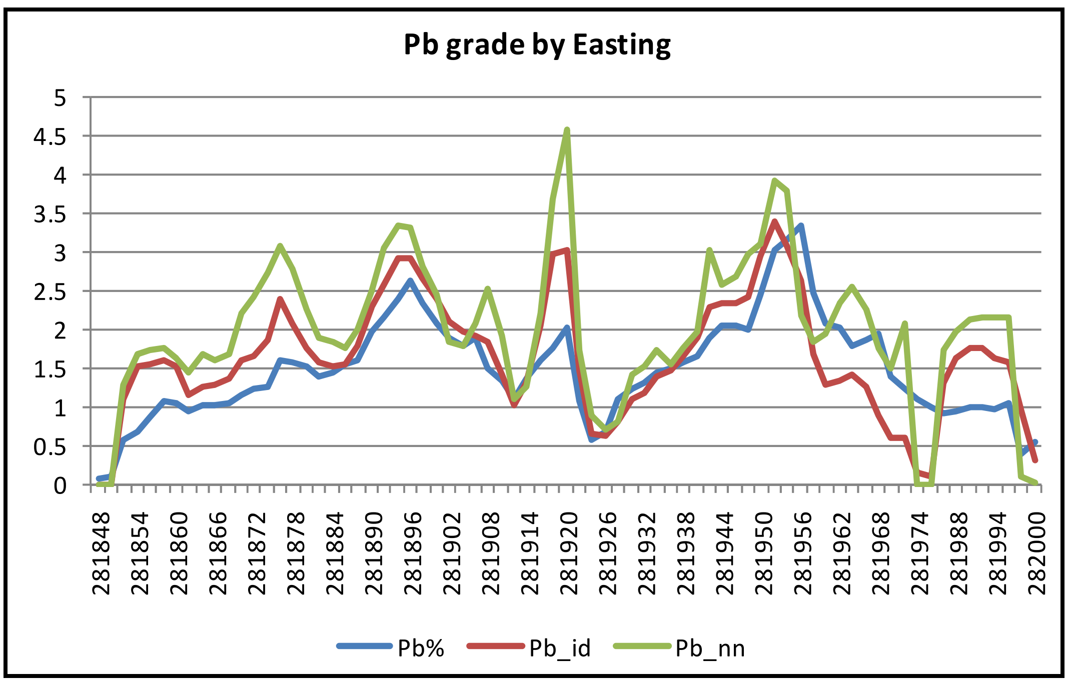

Note: Vertical axis in Zn%. Zn% - Ordinary Kriging; Zn_id – Inverse Distance; Zn_nn – Nearest Neighbour

Figure 14.35 Zn Grade by Elevation



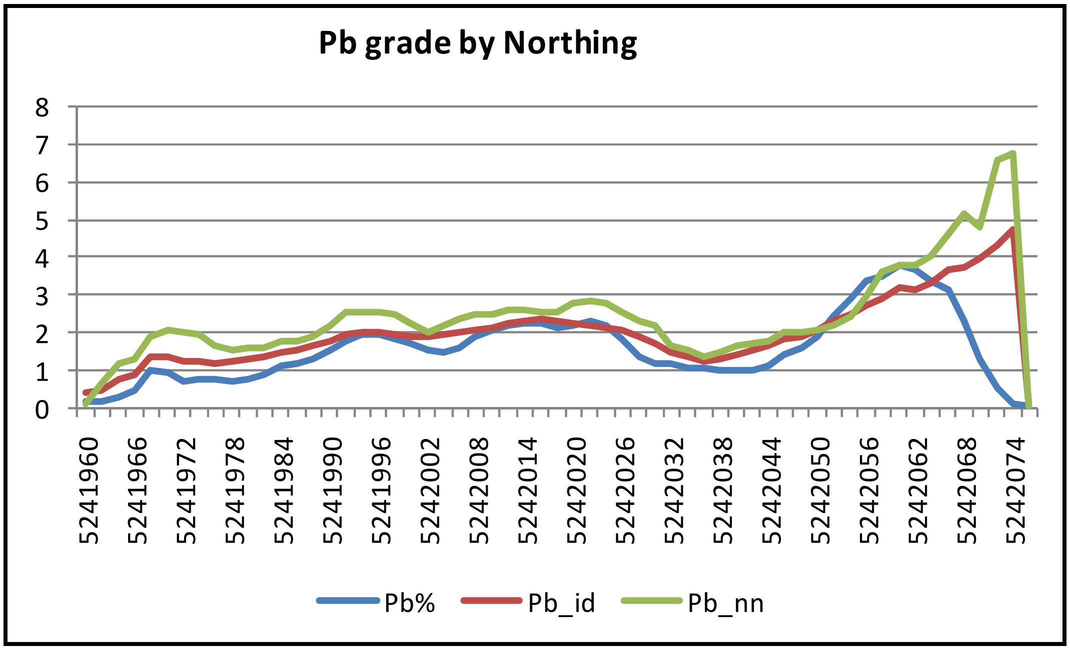

Note: Vertical axis in Zn%. Zn% - Ordinary Kriging; Zn_id – Inverse Distance; Zn_nn – Nearest Neighbour

Figure 14.36 Pb Grade by Easting



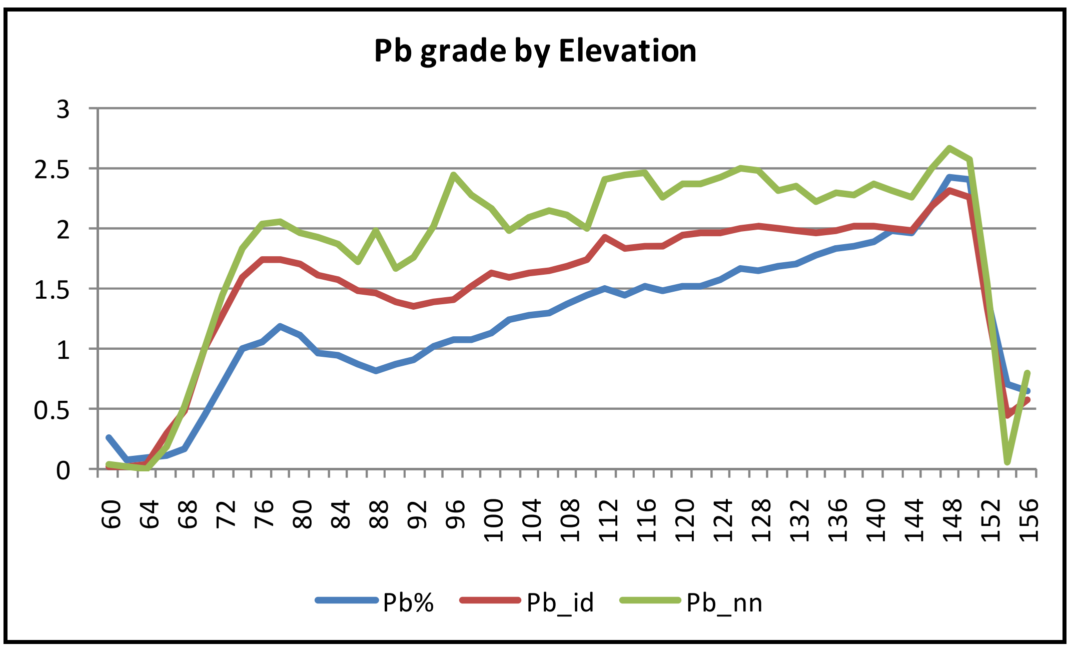

Note: Vertical axis in Pb%. Pb% - Ordinary Kriging; Pb_id – Inverse Distance; Pb_nn – Nearest Neighbour

Figure 14.37 Pb Grade by Northing



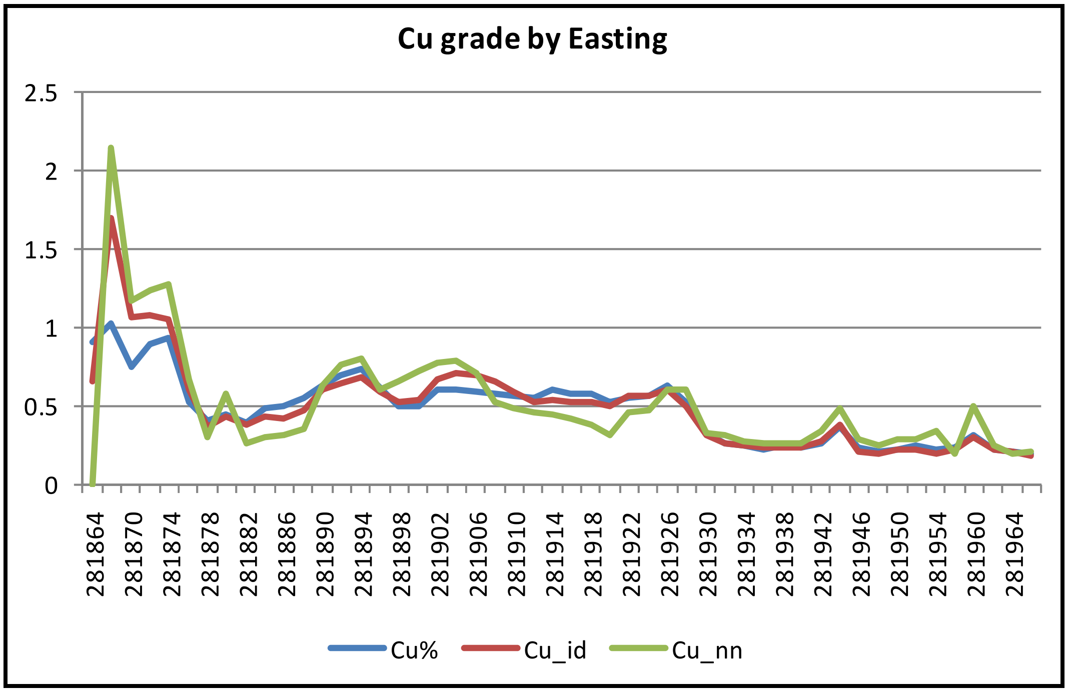

Note: Vertical axis in Pb%. Pb% - Ordinary Kriging; Pb_id – Inverse Distance; Pb_nn – Nearest Neighbour

Figure 14.38 Pb Grade by Elevation



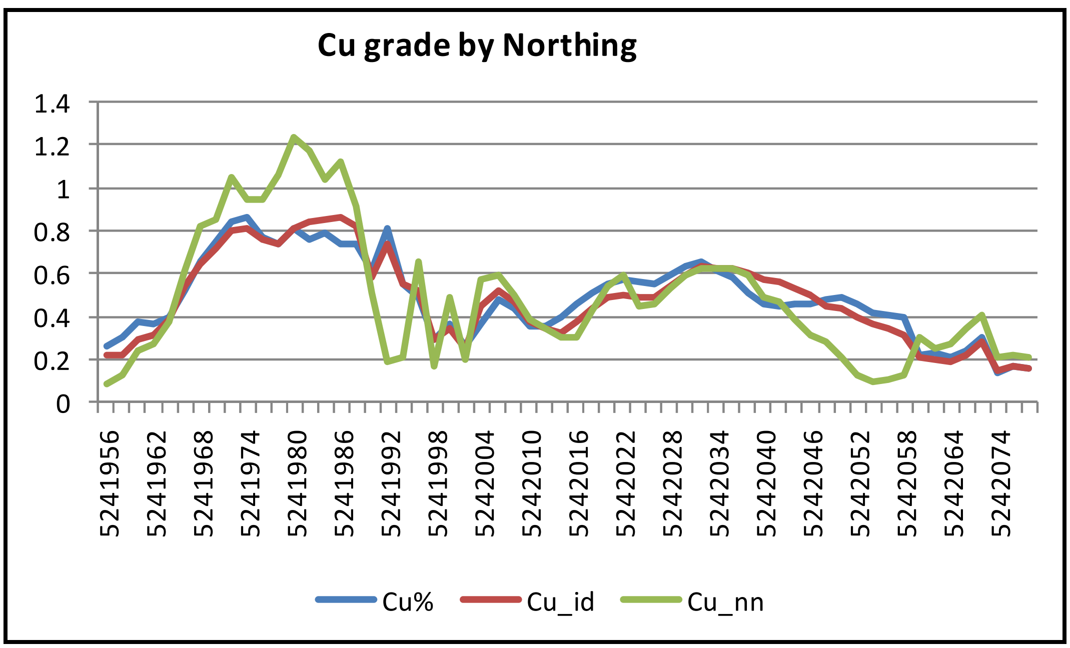

Note: Vertical axis in Pb%. Pb% - Ordinary Kriging; Pb_id – Inverse Distance; Pb_nn – Nearest Neighbour

Figure 14.39 Cu Grade by Easting




Note: Vertical axis in Cu%. Cu% - Ordinary Kriging; Cu_id – Inverse Distance; Cu_nn – Nearest Neighbour

Figure 14.40 Cu Grade by Northing




Note: Vertical axis in Cu%. Cu% - Ordinary Kriging; Cu_id – Inverse Distance; Cu_nn – Nearest Neighbour

**Figure 14.41 Cu Grade by Elevation**

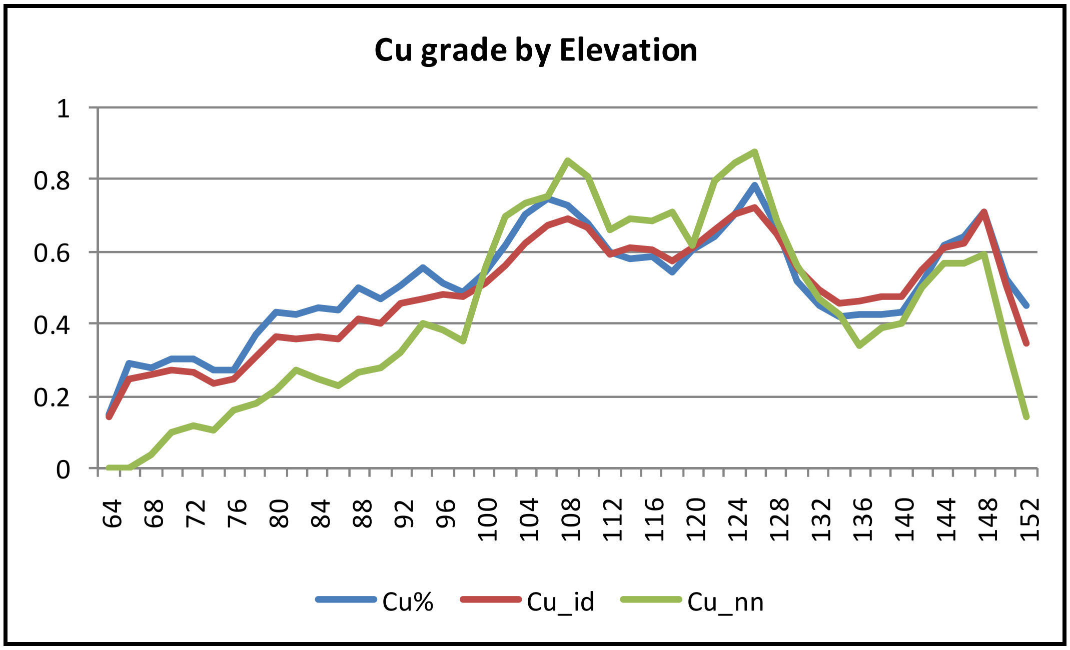



**Figure 14.42 Ag Grade by Easting**

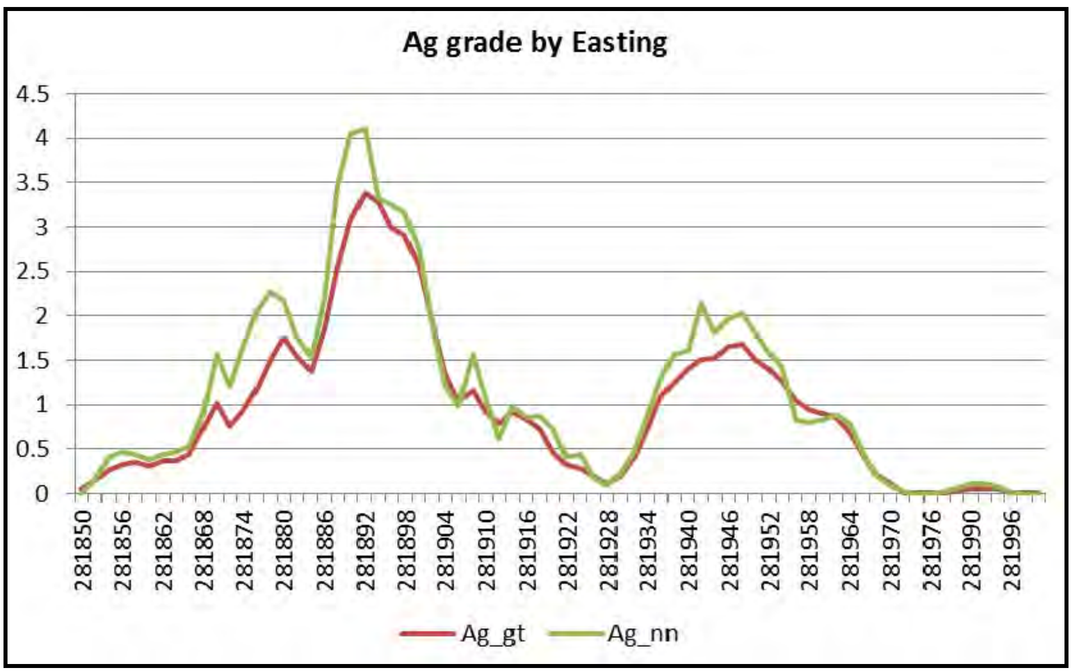


**Figure 14.43 Ag Grade by Northing**

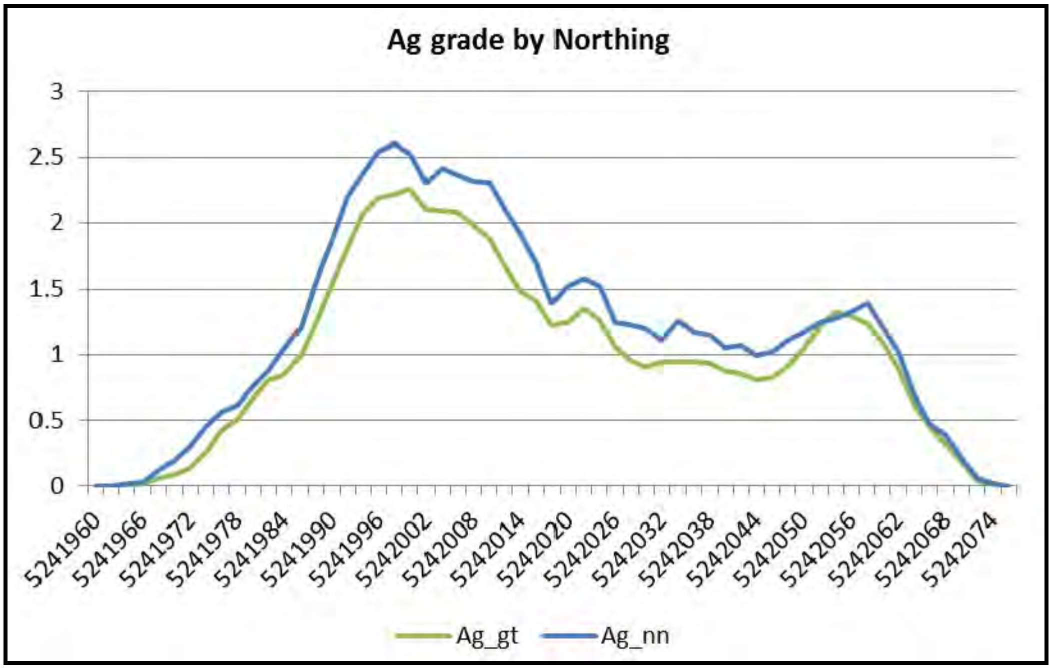
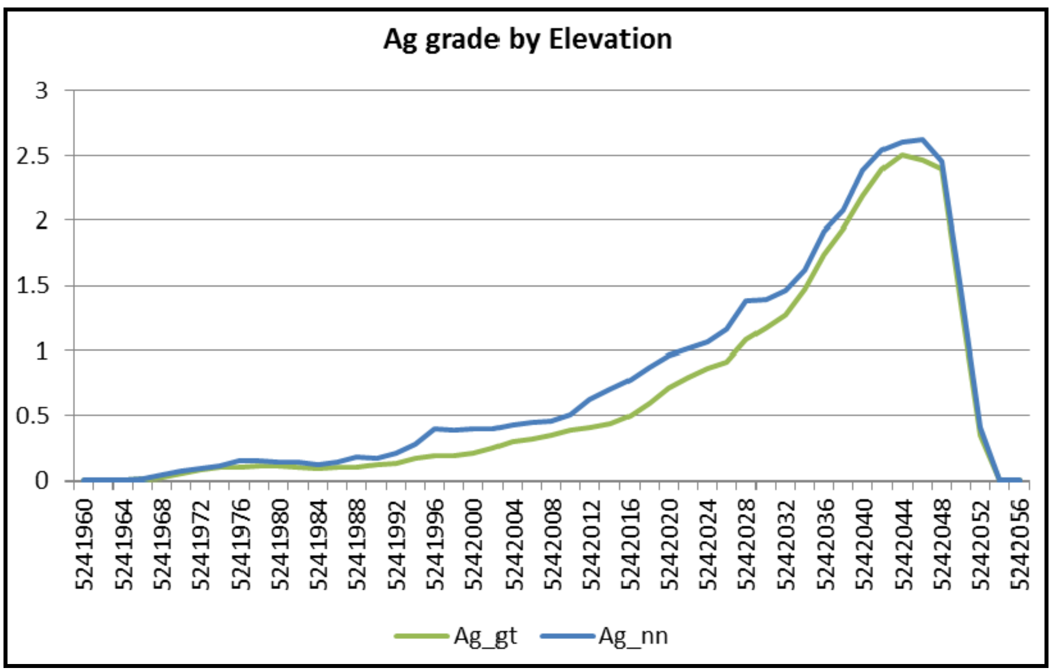



**Figure 14.44 Ag Grade by Elevation**



### 14.2.11 MINERAL RESOURCE CLASSIFICATION

#### MEASURED, INDICATED AND INFERRED

NI 43-101 relies on the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines for the definition of Mineral Resources. Among other things, for a mineralized body to be considered Mineral Resources, it must be demonstrated that under reasonable technical assumptions, it must have “reasonable prospects for economic extraction”. One of the principal conditions for “reasonable prospects for economic extraction” is met by application of a cut-off grade. Prior mining experience in the Bathurst district has developed and provided generally acceptable cut-off grades for mill ore feed.

Mineral resource classification was largely based on the grade interpolation plan. From surface exposure through open pit mining in the north-western part of the deposit, Measured and Indicated classifications were assigned. This is specifically a requirement for Measured classification according to CIM.

The search parameters used to control resource classification are recorded in Table 14.17. In summary, blocks that can be estimated by fulfilling sample searches in the first pass are allocated *Measured* status; second pass is nominated as *Indicated*, and *Inferred* cells are assigned to the third pass.

As there are two mineralized systems estimated within the deposit; a copper-cobalt system and a zinc-lead-silver system, two resource classification attributes were assigned: “cu_class” and “class_zn”. Measured, Indicated and Inferred status is assigned to copper, lead and zinc depending on search pass employed for estimation. These resource classification attributes are tabulated below (Table 14.21).

**Table 14.21 Resource Classification Attributes in the Block Model**

| Metal | cu_class | class_zn | Description |
|---|---|---|---|
| Zn | - | 1 | Measured |
| | - | 2 | Indicated |
| | - | 3 | Inferred |
| Pb | - | 1 | Measured |
| | - | 2 | Indicated |
| | - | 3 | Inferred |
| Cu | 1 | - | Measured |
| | 2 | - | Indicated |
| | 3 | - | Inferred |
| Ag | - | 1 | Measured |
| | - | 2 | Indicated |
| | - | 3 | Inferred |

The following figures (Figure 14.45 and Figure 14.46) depict the spatial relations between the copper-cobalt system and the zinc-lead-silver system with respect to resource classification at 140 m elevation.

**Figure 14.45 Zn-Pb-Ag Resource Classification for Cells at 140 m RL**

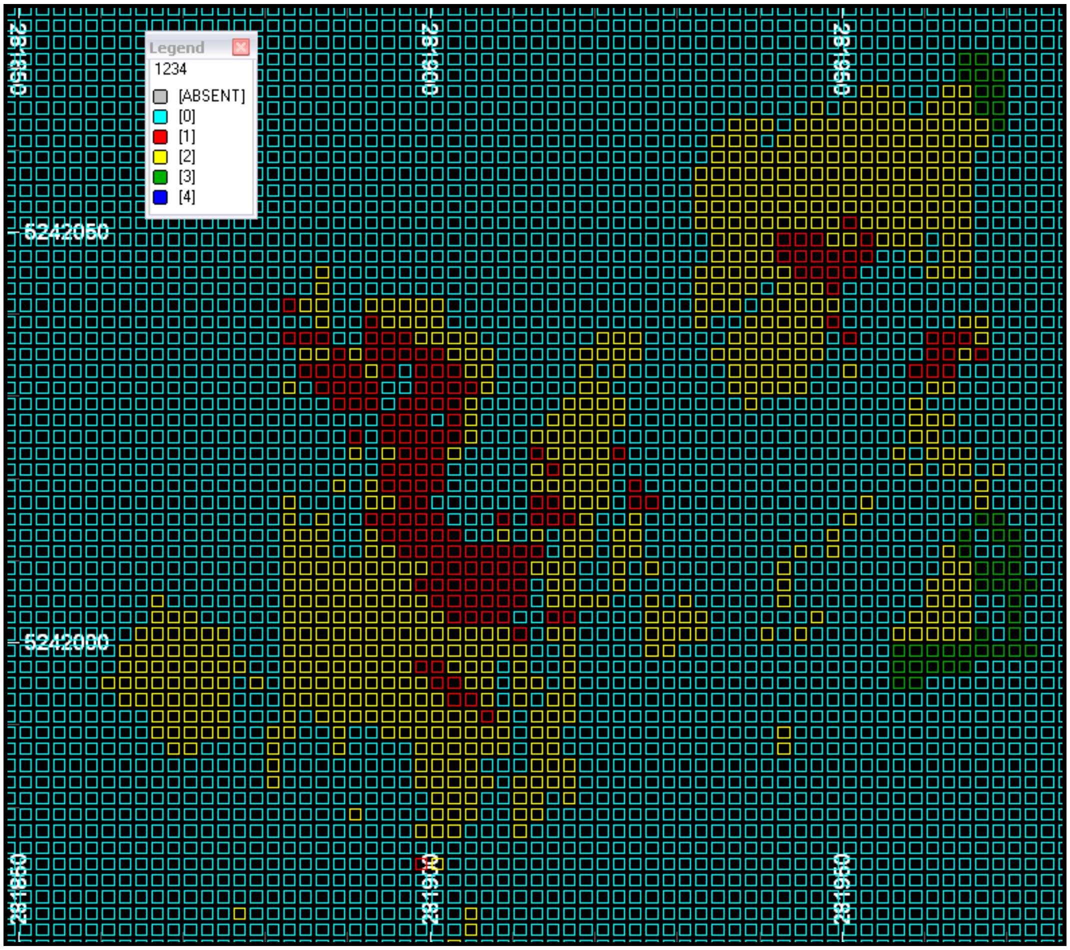
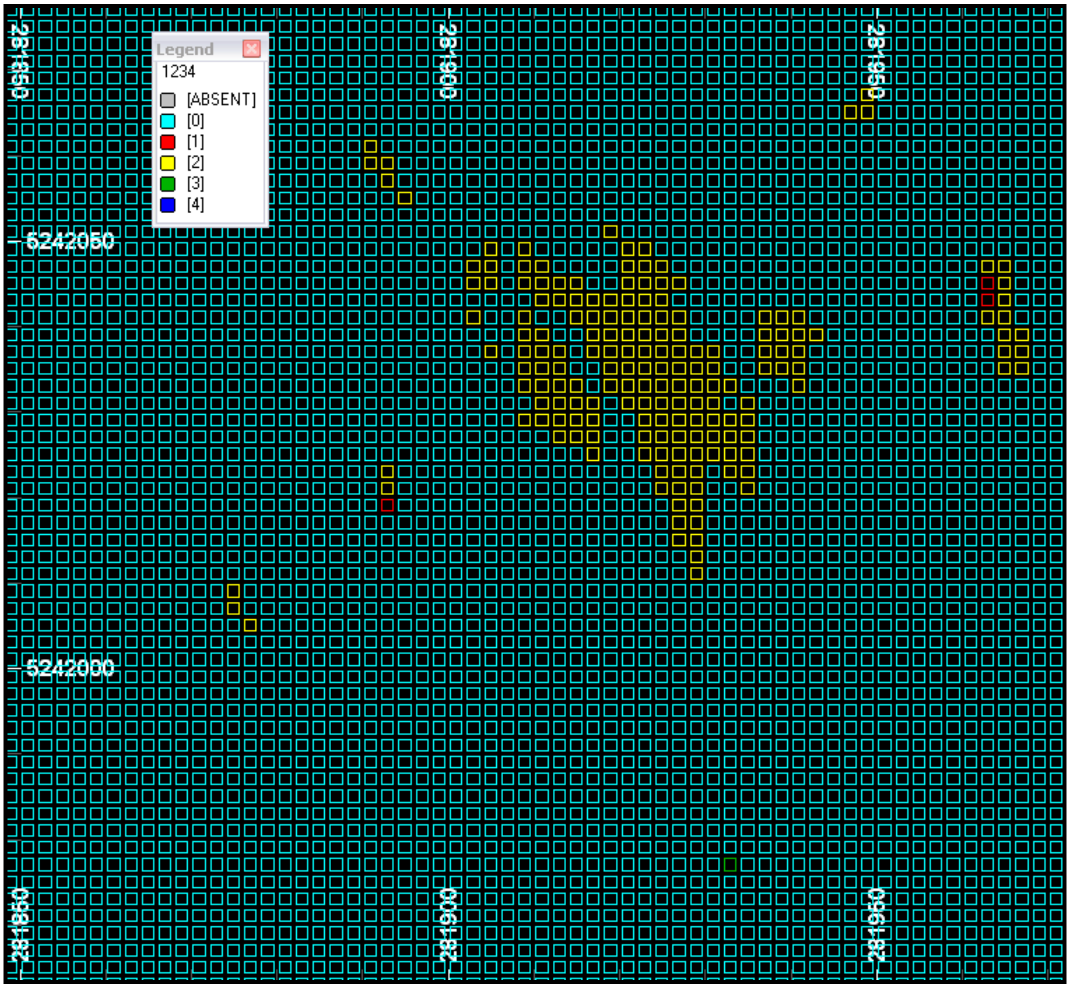


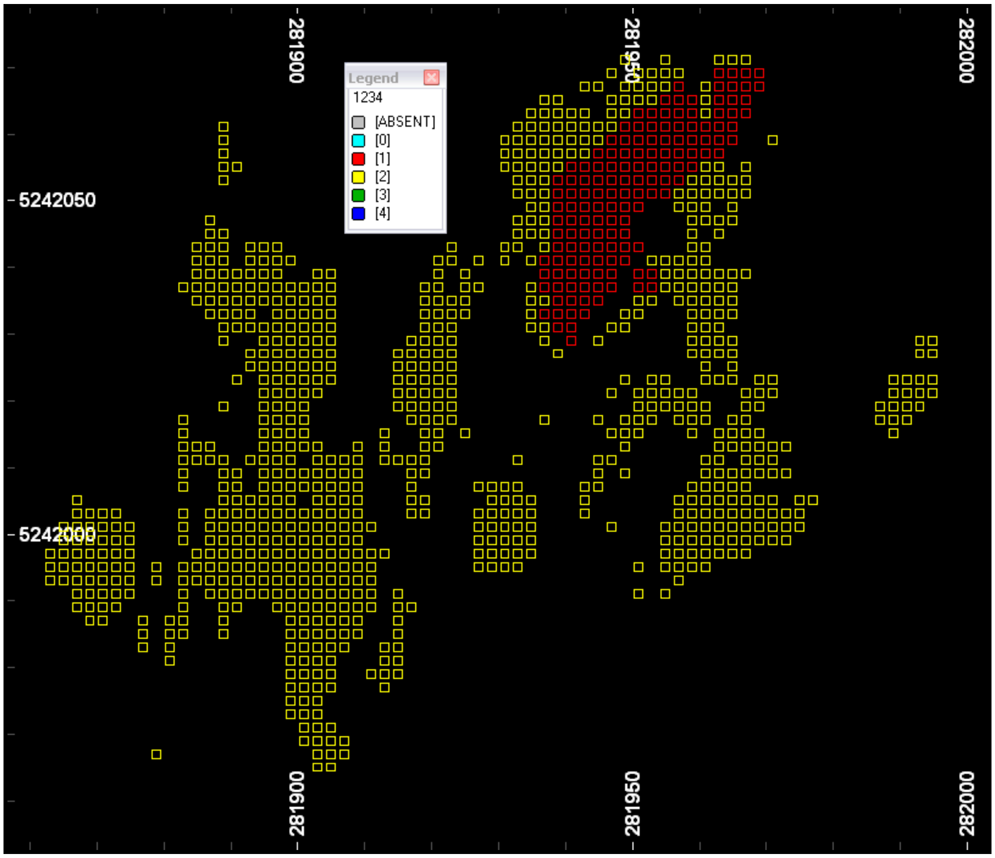
**Figure 14.46 Cu Resource Classification for Cells at 140 m RL**




## MINING STATUS

The CNE block model has a volume which has been mined out. The attribute used to identify previously mined and un-mined cells is *mstatus*. The cells which have been mined out are designated as “mstatus=1”. Cells which have not been mined out are designated “mstatus=2”. Figure 14.47 illustrates the *mstatus* cells with respect to only mineralized cells.

**Figure 14.47 Mining Status at 140 m Elevation**




Grade has been estimated into the mined-out cells as well as the un-mined cells. This information has been further documented and commented on in the section on reconciliation (Section 14.2.14).

### 14.2.12 MINERAL RESOURCE TABULATION

Table 14.22 to Table 14.24 record the mineral inventory for CNE. These tables correspond to the un-mined portion of block model mineralization (i.e. it does not include material which has already been extracted from the small open pit to the northwest). The mined portion of the CNE mineralization is reported in the following section on reconciliation (Section 14.2.14).

#### ZINC EQUIVALENT CUT-OFF

At the request of Stratabound, Tetra Tech reported the resource estimate in a zinc equivalent (ZnEQ%) cut-off. ZnEQ% is often used in polymetallic deposits to value all other metals in the deposit as an equivalent to zinc. ZnEQ% is calculated based on metal value and metal recovery.

The following parameters and equation were employed in determining the ZnEQ% values:

*ZnEQ% = (((Zn Price * Zn Grade * 22.04622 * Zn Recovery) + (Pb Price * Pb Grade * 22.04622 * Pb Recovery) * (Cu Price* Cu Grade * 22.04622 * Cu Recovery)) / (Zinc Price)) / 22.04622*

| | | | |
|---|---|---|---|
| Zn Price: | $1.06 | Zn Recovery: | 76.50% |
| Pb Price: | $0.99 | Pb Recovery: | 80.75% |
| Cu Price: | $3.01 | Cu Recovery: | 82.03% |

lbs per tonne: 2,204.622

ZINC, LEAD, SILVER AND COPPER

The CNE mineral inventory for copper, zinc, silver and lead, as reported by resource category (Measured, Indicated and Inferred), are presented in Table 14.22 to Table 14.24. These metals are reported as a function of zinc equivalent cut-offs. This accommodates cells with copper, but no zinc, as well as cells with zinc, but no copper.

### 14.2.13 *MINERAL INVENTORY GRADE – TONNAGE CURVES AND GRADE – METAL CURVES*

Figure 14.48 to Figure 14.51 depict the grade-tonnage and grade-metal curves for the different metal by resource classification for the CNE deposit. These figures correspond to the un-mined portion of block model mineralization (i.e. it does not include material which has already been extracted from the small open pit to the northwest.

### Table 14.22 CNE Mineral Inventory by Measured Resource Classification

| ZnEQ% Cut-off | Tonnes | ZnEQ (t) | Zn (t) | Pb (t) | Cu (t) | Ag (oz) | Grade (ZnEQ%) | Grade (Zn%) | Grade (Pb%) | Grade (Cu%) | Grade (Ag g/t) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 0.5 | 39,149 | 2,270 | 2,181 | 719 | 26 | 77,845 | 5.80 | 5.57 | 1.84 | 0.07 | 61.85 |
| 1 | 38,535 | 2,266 | 2,180 | 719 | 24 | 77,760 | 5.88 | 5.66 | 1.87 | 0.06 | 62.76 |
| **1.5** | **37,710** | **2,255** | **2,174** | **719** | **21** | **77,489** | **5.98** | **5.77** | **1.91** | **0.06** | **63.91** |
| 2 | 36,507 | 2,234 | 2,156 | 715 | 19 | 76,577 | 6.12 | 5.90 | 1.96 | 0.05 | 65.24 |
| 2.5 | 34,367 | 2,185 | 2,113 | 705 | 16 | 74,692 | 6.36 | 6.15 | 2.05 | 0.05 | 67.60 |
| 3 | 30,802 | 2,086 | 2,018 | 681 | 13 | 71,225 | 6.77 | 6.55 | 2.21 | 0.04 | 71.92 |
| 3.5 | 27,264 | 1,971 | 1,906 | 651 | 10 | 67,098 | 7.23 | 6.99 | 2.39 | 0.04 | 76.55 |
| 4 | 24,386 | 1,863 | 1,800 | 622 | 7 | 63,166 | 7.64 | 7.38 | 2.55 | 0.03 | 80.57 |
| 4.5 | 21,296 | 1,733 | 1,669 | 587 | 6 | 58,878 | 8.14 | 7.84 | 2.76 | 0.03 | 85.99 |
| 5 | 18,905 | 1,619 | 1,555 | 554 | 5 | 55,166 | 8.57 | 8.23 | 2.93 | 0.03 | 90.76 |

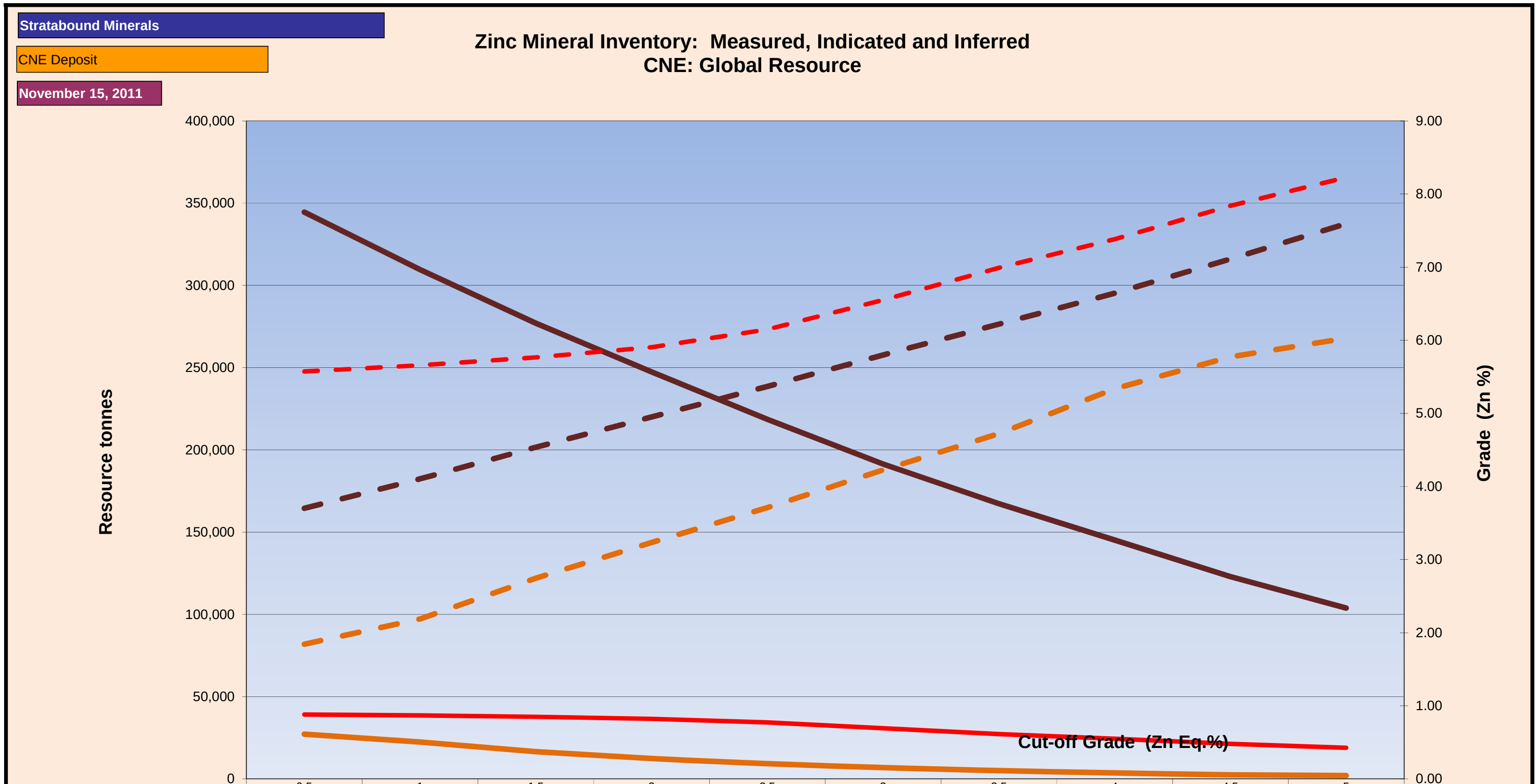
### Table 14.23 CNE Mineral Inventory by Indicated Resource Classification

| ZnEQ% Cut-off | Tonnes | ZnEQ (t) | Zn (t) | Pb (t) | Cu (t) | Ag (oz) | Grade (ZnEQ%) | Grade (Zn%) | Grade (Pb%) | Grade (Cu%) | Grade (Ag g/t) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 0.5 | 344,521 | 14,451 | 12,751 | 4,666 | 506 | 554,454 | 4.19 | 3.70 | 1.35 | 0.15 | 50.06 |
| 1 | 309,598 | 14,190 | 12,703 | 4,650 | 414 | 548,985 | 4.58 | 4.10 | 1.50 | 0.13 | 55.15 |
| **1.5** | **277,044** | **13,782** | **12,566** | **4,596** | **302** | **537,992** | **4.97** | **4.54** | **1.66** | **0.11** | **60.40** |
| 2 | 247,430 | 13,262 | 12,250 | 4,501 | 213 | 518,904 | 5.36 | 4.95 | 1.82 | 0.09 | 65.23 |
| 2.5 | 218,639 | 12,612 | 11,737 | 4,340 | 155 | 491,661 | 5.77 | 5.37 | 1.99 | 0.07 | 69.94 |
| 3 | 191,346 | 11,862 | 11,095 | 4,142 | 108 | 462,042 | 6.20 | 5.80 | 2.16 | 0.06 | 75.11 |
| 3.5 | 167,326 | 11,082 | 10,408 | 3,918 | 71 | 430,519 | 6.62 | 6.22 | 2.34 | 0.04 | 80.03 |
| 4 | 145,202 | 10,255 | 9,647 | 3,674 | 45 | 397,964 | 7.06 | 6.64 | 2.53 | 0.03 | 85.25 |
| 4.5 | 122,954 | 9,310 | 8,744 | 3,376 | 32 | 359,655 | 7.57 | 7.11 | 2.75 | 0.03 | 90.98 |
| 5 | 103,865 | 8,403 | 7,887 | 3,070 | 23 | 324,111 | 8.09 | 7.59 | 2.96 | 0.02 | 97.06 |

### Table 14.24 CNE Mineral Inventory by Inferred Resource Classification

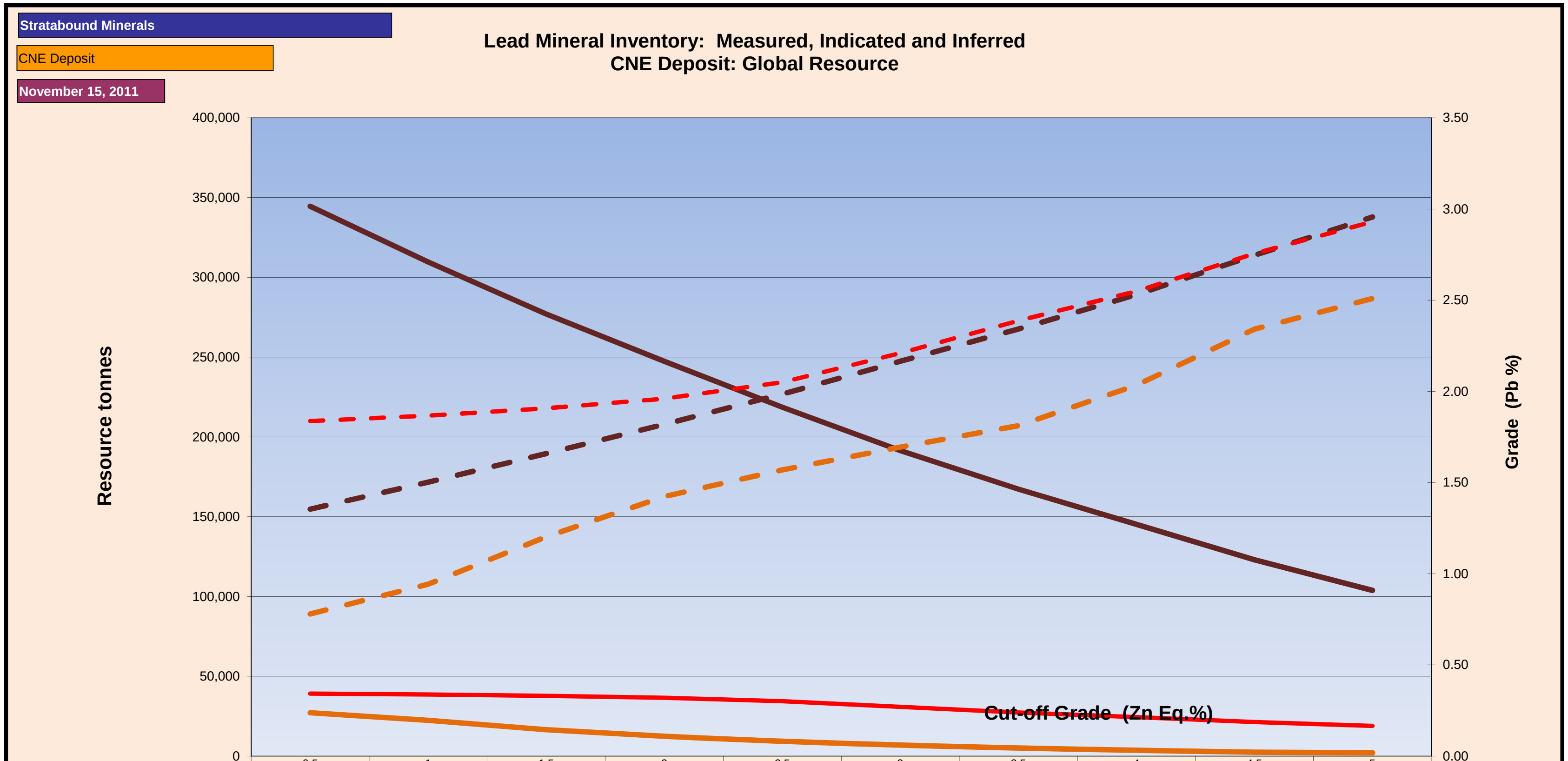
| ZnEQ% Cut-off | Tonnes | ZnEQ (t) | Zn (t) | Pb (t) | Cu (t) | Ag (oz) | Grade (ZnEQ%) | Grade (Zn%) | Grade (Pb%) | Grade (Cu%) | Grade (Ag g/t) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 0.5 | 27,211 | 628 | 501 | 212 | 36 | 23,273 | 2.31 | 1.84 | 0.78 | 0.13 | 26.60 |
| 1 | 22,360 | 592 | 489 | 211 | 25 | 22,427 | 2.65 | 2.19 | 0.94 | 0.11 | 31.20 |
| **1.5** | **16,517** | **520** | **453** | **199** | **10** | **19,912** | **3.15** | **2.74** | **1.20** | **0.06** | **37.50** |
| 2 | 12,419 | 449 | 401 | 177 | 4 | 17,317 | 3.62 | 3.23 | 1.42 | 0.03 | 43.37 |
| 2.5 | 9,197 | 377 | 341 | 144 | 3 | 14,377 | 4.10 | 3.71 | 1.57 | 0.03 | 48.62 |
| 3 | 6,854 | 313 | 290 | 116 | 2 | 11,647 | 4.57 | 4.23 | 1.69 | 0.02 | 52.86 |
| 3.5 | 5,118 | 256 | 242 | 93 | 1 | 9,198 | 5.01 | 4.72 | 1.81 | 0.01 | 55.90 |
| 4 | 3,461 | 194 | 185 | 70 | - | 6,436 | 5.62 | 5.34 | 2.03 | - | 57.85 |
| 4.5 | 2,443 | 151 | 141 | 57 | - | 4,468 | 6.18 | 5.77 | 2.34 | - | 56.89 |
| 5 | 2,003 | 130 | 121 | 50 | - | 3,460 | 6.50 | 6.03 | 2.51 | - | 53.72 |

**Figure 14.48 Zn Grade-Tonnage Curves for the Measured, Indicated and Inferred Resources**



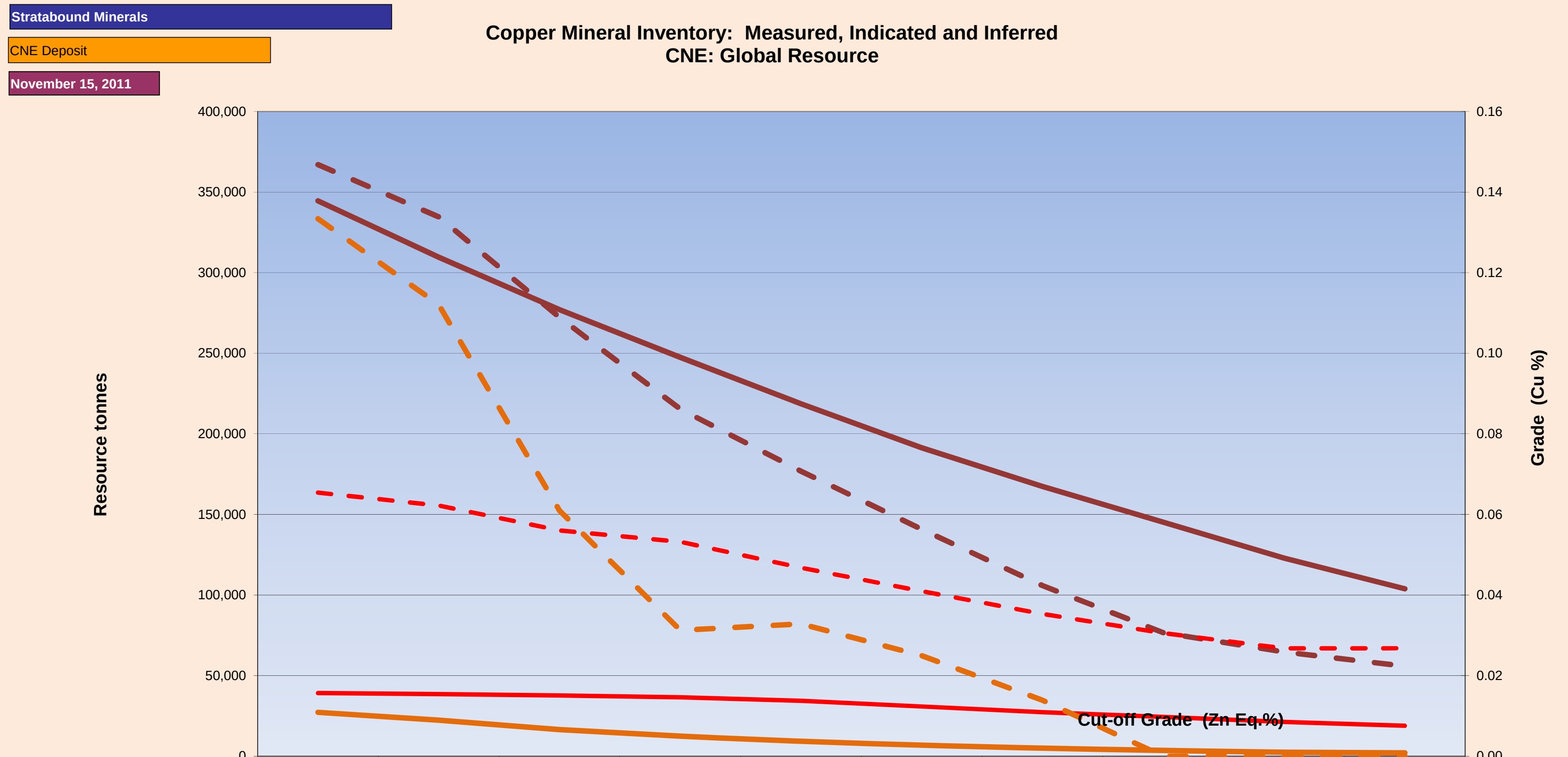

| | 0.5 | 1 | 1.5 | 2 | 2.5 | 3 | 3.5 | 4 | 4.5 | 5 |
|---|---|---|---|---|---|---|---|---|---|---|
| Indicated Tonnes | 344,521 | 309,598 | 277,044 | 247,430 | 218,639 | 191,346 | 167,326 | 145,202 | 122,954 | 103,865 |
| Inferred Tonnes | 27,211 | 22,360 | 16,517 | 12,419 | 9,197 | 6,854 | 5,118 | 3,461 | 2,443 | 2,003 |
| Measured Tonnes | 39,149 | 38,535 | 37,710 | 36,507 | 34,367 | 30,802 | 27,264 | 24,386 | 21,296 | 18,905 |
| Indicated Grade Zn% | 3.70 | 4.10 | 4.54 | 4.95 | 5.37 | 5.80 | 6.22 | 6.64 | 7.11 | 7.59 |
| Inferred Grade Zn% | 1.84 | 2.19 | 2.74 | 3.23 | 3.71 | 4.23 | 4.72 | 5.34 | 5.77 | 6.03 |
| Measured Grade Zn% | 5.57 | 5.66 | 5.77 | 5.90 | 6.15 | 6.55 | 6.99 | 7.38 | 7.84 | 8.23 |

**Figure 14.49 Pb Grade-Metal Curves for the Measured, Indicated and Resources**




| | 0.5 | 1 | 1.5 | 2 | 2.5 | 3 | 3.5 | 4 | 4.5 | 5 |
|---|---|---|---|---|---|---|---|---|---|---|
| Indicated Tonnes | 344,521 | 309,598 | 277,044 | 247,430 | 218,639 | 191,346 | 167,326 | 145,202 | 122,954 | 103,865 |
| Inferred Tonnes | 27,211 | 22,360 | 16,517 | 12,419 | 9,197 | 6,854 | 5,118 | 3,461 | 2,443 | 2,003 |
| Measured Tonnes | 39,149 | 38,535 | 37,710 | 36,507 | 34,367 | 30,802 | 27,264 | 24,386 | 21,296 | 18,905 |
| Grade_Pb% | 1.35 | 1.50 | 1.66 | 1.82 | 1.99 | 2.16 | 2.34 | 2.53 | 2.75 | 2.96 |
| Grade_Pb% | 0.78 | 0.94 | 1.20 | 1.42 | 1.57 | 1.69 | 1.81 | 2.03 | 2.34 | 2.51 |
| Measured Grade Pb% | 1.84 | 1.87 | 1.91 | 1.96 | 2.05 | 2.21 | 2.39 | 2.55 | 2.76 | 2.93 |

## Figure 14.50 Cu Grade-Tonnage Curves for the Measured, Indicated and Inferred Resources




| | 0.50 | 1.00 | 1.50 | 2.00 | 2.50 | 3.00 | 3.50 | 4.00 | 4.50 | 5.00 |
|---|---|---|---|---|---|---|---|---|---|---|
| Indicated Tonnes | 344,521 | 309,598 | 277,044 | 247,430 | 218,639 | 191,346 | 167,326 | 145,202 | 122,954 | 103,865 |
| Inferred Tonnes | 27,211 | 22,360 | 16,517 | 12,419 | 9,197 | 6,854 | 5,118 | 3,461 | 2,443 | 2,003 |
| Measured Tonnes | 39,149 | 38,535 | 37,710 | 36,507 | 34,367 | 30,802 | 27,264 | 24,386 | 21,296 | 18,905 |
| Indciated Grade Cu% | 0.15 | 0.13 | 0.11 | 0.09 | 0.07 | 0.06 | 0.04 | 0.03 | 0.03 | 0.02 |
| Inferred Grade | 0.13 | 0.11 | 0.06 | 0.03 | 0.03 | 0.02 | 0.01 | - | - | - |
| Measured Grade Cu% | 0.07 | 0.06 | 0.06 | 0.05 | 0.05 | 0.04 | 0.04 | 0.03 | 0.03 | 0.03 |

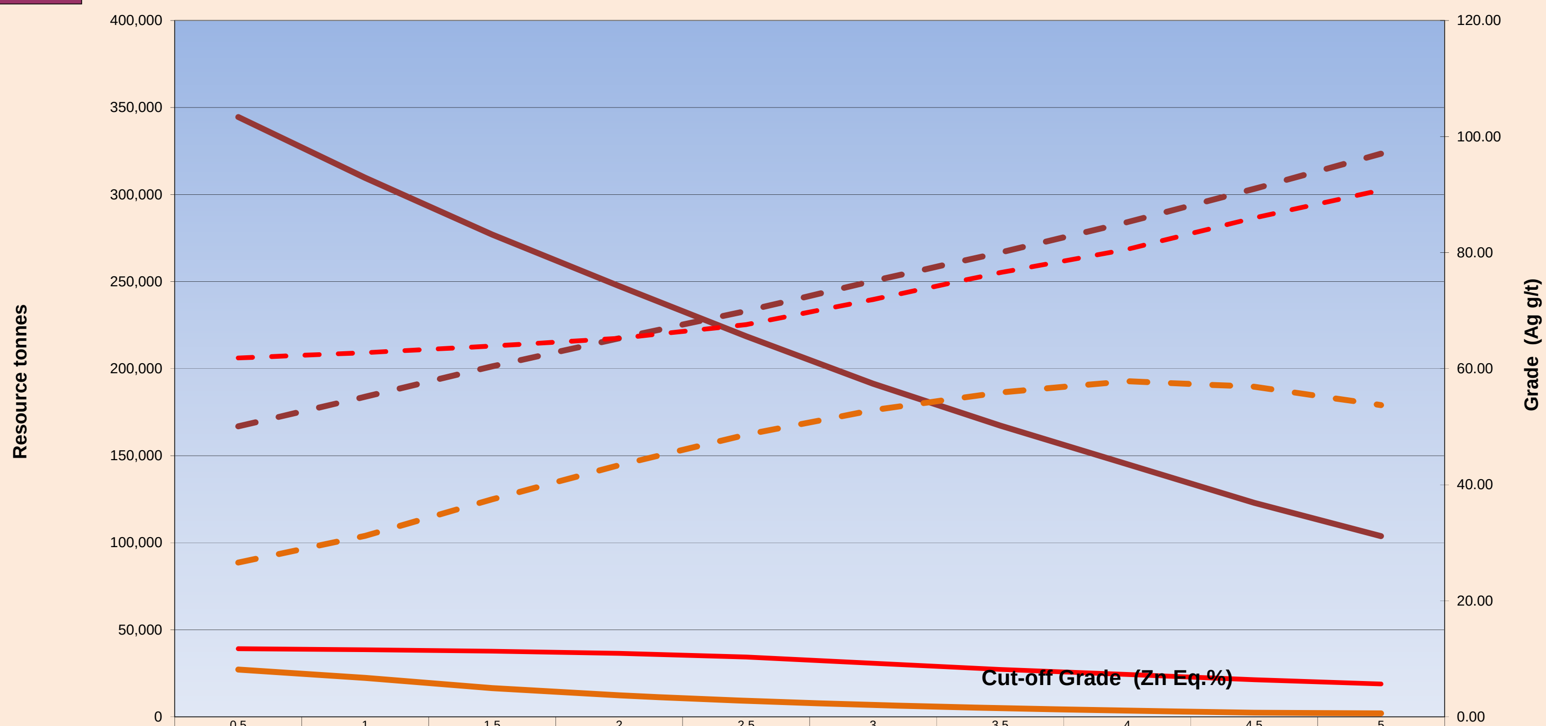
## Figure 14.51 Ag Grade-Tonnage Curves for Measured, Indicated and Inferred Resources

Stratabound Minerals

CNE Deposit

November 15, 2011

Silver Mineral Inventory: Measured, Indicated and Inferred
CNE: Global Resource




| | 0.5 | 1 | 1.5 | 2 | 2.5 | 3 | 3.5 | 4 | 4.5 | 5 |
|---|---|---|---|---|---|---|---|---|---|---|
| Indicated Tonnes | 344,521 | 309,598 | 277,044 | 247,430 | 218,639 | 191,346 | 167,326 | 145,202 | 122,954 | 103,865 |
| Inferred Tonnes | 27,211 | 22,360 | 16,517 | 12,419 | 9,197 | 6,854 | 5,118 | 3,461 | 2,443 | 2,003 |
| Measured Tonnes | 39,149 | 38,535 | 37,710 | 36,507 | 34,367 | 30,802 | 27,264 | 24,386 | 21,296 | 18,905 |
| Indicated Grade Ag gt | 50.06 | 55.15 | 60.40 | 65.23 | 69.94 | 75.11 | 80.03 | 85.25 | 90.98 | 97.06 |
| Inferred Grade Ag gt | 26.60 | 31.20 | 37.50 | 43.37 | 48.62 | 52.86 | 55.90 | 57.85 | 56.89 | 53.72 |
| Measured Grade Ag gt | 61.85 | 62.76 | 63.91 | 65.24 | 67.60 | 71.92 | 76.55 | 80.57 | 85.99 | 90.76 |

### 14.2.14 RESOURCE COMPARISON AND RECONCILIATION

#### RESOURCE COMPARISON

JA Goodwin (2003) calculated the CNE reserves for the 1990 Stratabound Minerals Corp. Feasibility Study.

Goodwin defined the March 1990 reserves using a cut-off of 6% and 8% combined lead and zinc with a minimum 4% Zn over a full level block. The copper cut-off was a copper-gold combined equivalent of 1% or greater. At these cut-offs, Goodwin calculated the deposit contained 350,000 t and 470,000 t, respectively.

In October 1990, Goodwin revised his reserve estimate based on better information. In a final feasibility study for the CNE deposit, JA Goodwin declared an estimated resource of 207,555 t of probable ore with an average grade of 7.38% Zn, 2.76% Pb and 2.68 oz/t Ag. An additional 30,850 t of probable copper reserves were also declared with an average grade of 1.25% Cu and 0.02 g/t Au.

These two reserve calculations (March 1990 and October 1990) were subsequently reported by DF Brown and KDA Whaley as "probable reserves" (October 1990) and "geological reserves" (March 1990). The geological reserves were quoted by Brown and Whaley as 352,302 t with average grades of 2.0% Pb, 5.6% Zn and 2.0 oz/t Ag. These "geological reserves" may represent Goodwin's reserve calculations with a cut-off of 6% combined lead and zinc.

In comparison, this CNE resource estimate, including material mined in the pit, at cut-offs of (a) 2% Zn and (b) 3% Zn, record (a) 293,885 t at an average zinc grade of 5.62%, lead grade of 2.04% and silver grade of 65.23 g/t, and (b) 241,241 t at an average zinc grade of 6.30%, lead grade of 2.29% and silver grade of 74.66 g/t. Table 14.25 demonstrates that the resource estimates, considering discrepancies in reporting and interpolating, are very similar.

Note that reporting of Inferred material with Indicated and Measured material is not permitted as per CIM Standards on Mineral Resources and Reserves Definitions and Guidelines. In this instance, Table 14.26 presents a comparison with the historical resource estimate.

**Table 14.25 Zn-Pb-Ag Comparison Between this Resource Tabulation and Goodwin (1990)**

| **Class_zn** | 1+2+3 | 1+2+3 | 1+2 | 1+2 | Goodwin | Goodwin |
|---|---|---|---|---|---|---|
| **Cut-off** | 2% Zn | 3% Zn | 2% Zn | 3% Zn | 8% Pb + Zn | 6% Pb + Zn |
| **Tonnes** | 305,394 | 248,437 | 293,885 | 241,241 | 207,555 | 352,302 |
| **Grade Zn%** | 5.56 | 6.26 | 5.62 | 6.30 | 7.38 | 5.60 |
| **Grade Pb%** | 2.03 | 2.29 | 2.04 | 2.29 | 2.76 | 2.00 |
| **Grade Ag (g/t)** | 64.32 | 74.01 | 65.23 | 74.66 | 83.36 | 62.21 |
| **Metal Zn (t)** | 16,980 | 15,553 | 16,519 | 15,198 | 15,318 | 19,729 |
| **Metal Pb (t)** | 6,214 | 5,690 | 6,009 | 5,532 | 5,729 | 7,046 |
| **Metal Ag (oz)** | 612,798 | 544,915 | 595,480 | 533,267 | 556,265 | 704,638 |

Note: 1 = Measured, 2 = Indicated and 3 = Inferred (Resource Classification).

Similarly, the copper domain total tonnes is sensitive to the cut-off employed. Goodwin assigned a cut-off based on combined copper and gold. Gold was not estimated into this resource model due to lack of samples and lack of correlation. The comparison of the 1990 reserve figures of Goodwin and the total mineral inventory of this resource is tabulated below.

**Table 14.26 Cu Comparison between this Resource Tabulation and Goodwin (1990)**

| **Cut-off** | **Class_cu** | **Tonnes** | **Metal (t)** | **Grade Cu%** | **Grade Au (g/t)** |
|---|---|---|---|---|---|
| Cu 0.6% | 1+2+3 | 35,542 | 334.3 | 0.94 | n/a |
| Cu 0.8% | 1+2+3 | 19,884 | 226.4 | 1.14 | n/a |
| Cu + Au 1% | Goodwin | 30,850 | 391.8 | 1.27 | 0.62 |

Note: 1 = Measured, 2 = Indicated and 3 = Inferred (Resource Classification).

### RECONCILIATION

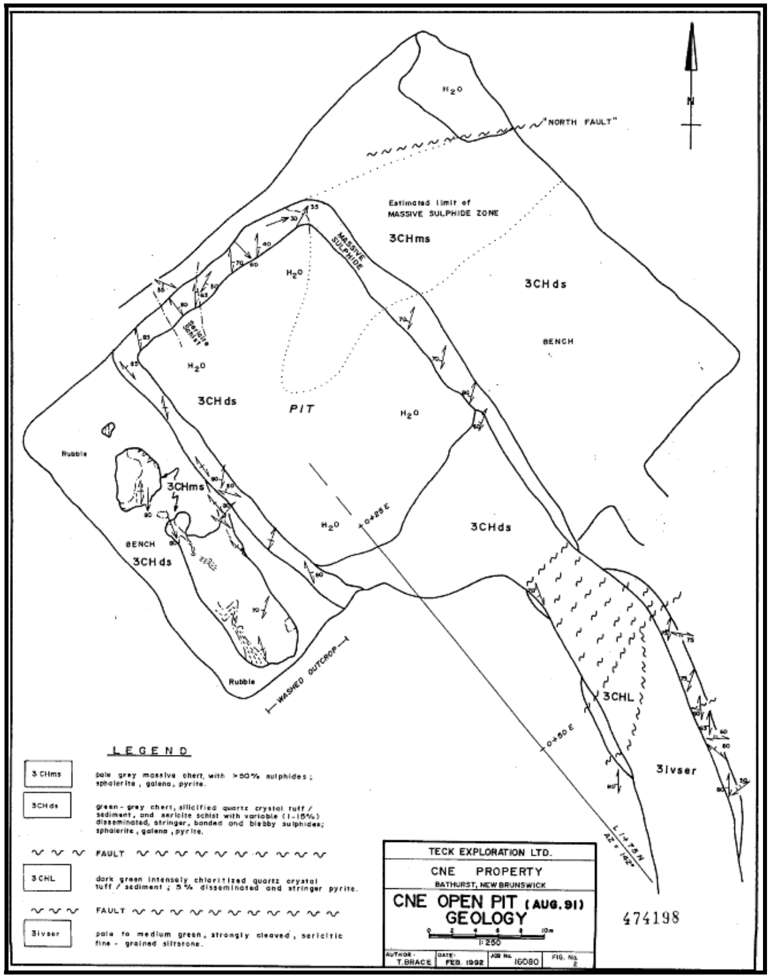
From August 1990 to 1992, approximately 40,000 t of zinc-lead-silver ore had been mined from a small pit to the northwest of the CNE deposit. The ore was processed at the Heath Steele mill. Detailed in-pit mapping was completed by Teck Exploration in 1992 (Figure 14.52). It revealed complex folding and faulting of volcaniclastic rocks and cherts within zones of semi-massive sulphides.

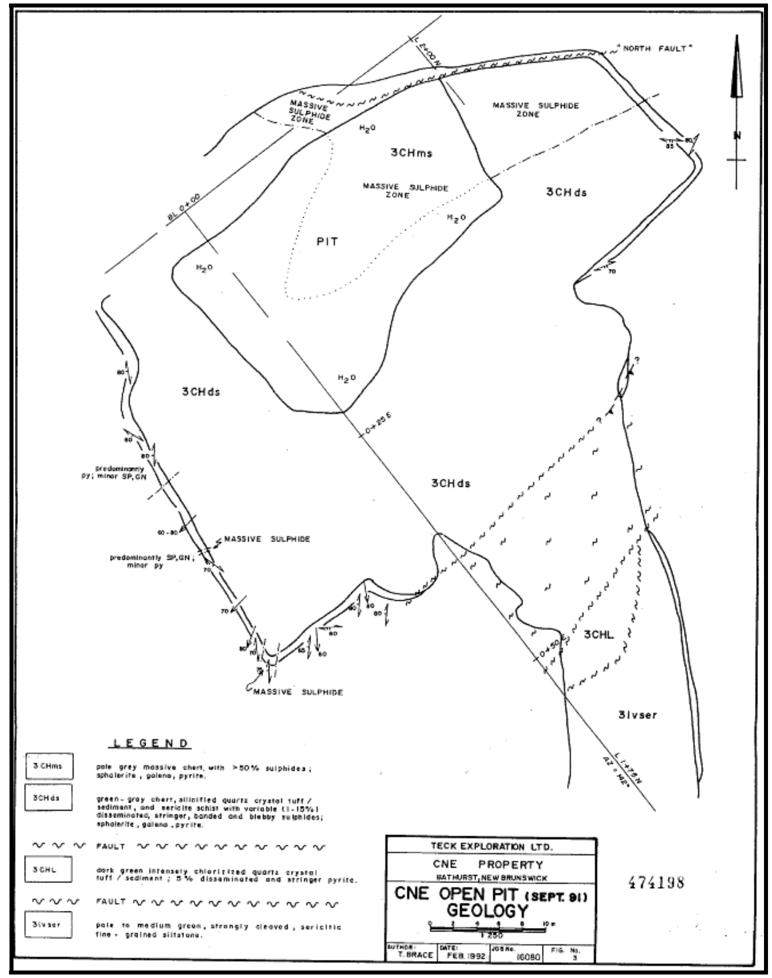
The ore from this pit was treated at the Heath Steel Mines mill. A total of 39,622 t was treated at an average grade of 10.11% Zn, 4.14% Pb and 132 g/t Ag. In comparison, the current resource model estimates a within-pit resource of 30,004 t at an average grade of 7.44% Zn, 3.20% Pb and 50.7 g/t Ag.

The difference in total tonnes may in part reflect the actual versus modelled pit shell. The actual pit had square walls and a flat floor (Figure 14.52). The paucity of survey points did not permit an accurate recreation of this original configuration. Thus the in-pit volume modelled is invariably less than the volume that was actually mined out.

DK Brown (personal communication) also noted that significantly more massive sulphides were excavated within the pit than was originally modelled in Goodwin's 1990 reserve estimate. This significant increase in ore tonnes and grade reflected in the complexity of the deposit geology as noted by Teck Exploration. Massive sulphide lenses may be much more prevalent in ore zones than can be adequately modelled with current drillhole sample spacing. This represents an "upside" to be considered in evaluating CNE ore zones.

**Figure 14.52 CNE Pit Geology Plans by Teck Dated August 1991 (left) and September 1991 (right)**



## 14.3 TAYLOR BROOK RESOURCE ESTIMATE

### 14.3.1 *INTRODUCTION*

Tetra Tech has estimated an NI 43-101 compliant resource of a massive sulphide horizon that is the Taylor Brook VMS deposit. The deposit has been interpreted as a single main massive sulphide zone with five smaller lenses for a total of six wireframe models.

The effective date of the Taylor Brook resource estimate is May 12, 2011.

### 14.3.2 *DATABASE*

Stratabound supplied all of the digital data for the resource estimate. Stratabound compiled the historical drillhole data from previous assessment reports which are publicly available on the NBDNR website. This data was imported into Gemcom GEMS 6.2.4 Resource Evaluation software package.

The dataset included a collar (header), survey, assay and lithology files. The dataset underwent a preliminary verification to determine the reliability of the data. The dataset included 47 drillholes, where 12 of which lie outside the interpreted deposit, therefore only 35 drillholes were used for the deposit interpretation and resource estimation. It should be noted that the drillholes were selectively assayed across the massive sulphide intervals.

Initial verification found that some conversions in the header file from feet to metres were inconsistent and that some downhole survey data needed to be edited as several readings of the downhole survey were noted as being wrong or missing. The header and downhole survey files file were corrected and verified. The assay file was determined to be correct. The lithology file was correct, however, some editing was required to break down lithology codes to individual lithology, texture and alteration codes.

*Manual checks on the database were made to remove any obvious errors prior to statistical treatments (such as negative values).*

### 14.3.3 *SPECIFIC GRAVITY*

There are no known specific gravity readings in the Taylor Brook database. Specific gravity readings of the distinct lithologies were taken from the CNE deposit as the lithological units belong to the same geological formation (Nepisiguit Falls Formation). Table 14.27 below shows the specific gravities used in the resource model.

**Table 14.27 Specific Gravity and Rock Codes**

| Lithology | Specific Gravity | Rock Code |
|---|---|---|
| Air | 0 | 0 |
| Overburden | 2.0 | 8 |
| Quartz Vein | 2.65 | 101 |
| Country Rock | 2.8 | 9 |
| Rhyolite | 2.8 | 101 |
| Rhyolite Lapilli Tuff | 2.8 | 101 |
| Rhyolite Crystal Tuff | 2.9 | 101 |
| Massive Sulphide | 3.6 | 101 |

### 14.3.4 *EXPLORATORY DATA ANALYSIS*

Exploratory Data Analysis is the application of statistical tools to understand the characteristics of the values in the database. The tools used are descriptive statistics, histograms, probability plots and box plots.

The statistics on the raw assay data are presented in Table 14.28. Statistics on values greater than zero are shown.

**Table 14.28 Raw Assay Statistics (No Zeroes)**

| | Length (m) | Zn% | Pb% | Cu% | Ag (g/t) | Au (g/t) |
|---|---|---|---|---|---|---|
| Count | 764 | 757 | 645 | 525 | 711 | 131 |
| Min | 0.09 | 0.005 | 0.005 | 0.001 | 0.340 | 0.031 |
| Max | 5.00 | 11.510 | 9.670 | 0.340 | 487.200 | 1.714 |
| Mean | 0.91 | 0.839 | 0.477 | 0.025 | 18.699 | 0.236 |
| Std Dev | 0.41 | 1.605 | 1.016 | 0.036 | 38.871 | 0.271 |
| Variance | 0.17 | 2.575 | 1.033 | 0.001 | 1510.958 | 0.074 |
| CV | 0.45 | 1.913 | 2.132 | 1.452 | 2.079 | 1.149 |

Since gold has only been sporadically analyzed during the various drill campaigns there is very limited data available. Therefore gold has been omitted from this resource estimate.

#### CAPPING

Cumulative probability Parrish plots were used to determine whether capping was required and the capping level. Typically, a change or break in the probability curve (i.e. separation of data points) indicates distinct populations of data. Outlying high values occur in the highest percentiles of a cumulative probability plot where separations in the data are common. The break in the curve was used in determining the capping level of the various grade elements.

Capping levels were established at 10% for Zn, 7% for Pb, 0.3% for Cu, and 200 g/t for Ag. Examples of the cumulative frequency plots for the raw uncapped data at Taylor Brook Deposit are given Figure 14.53 and Figure 14.54. All other plots are in Appendix H. The Parrish plots are presented in Appendix I.

Figure 14.53 Histogram and Cumulative Probability Plot for Zn

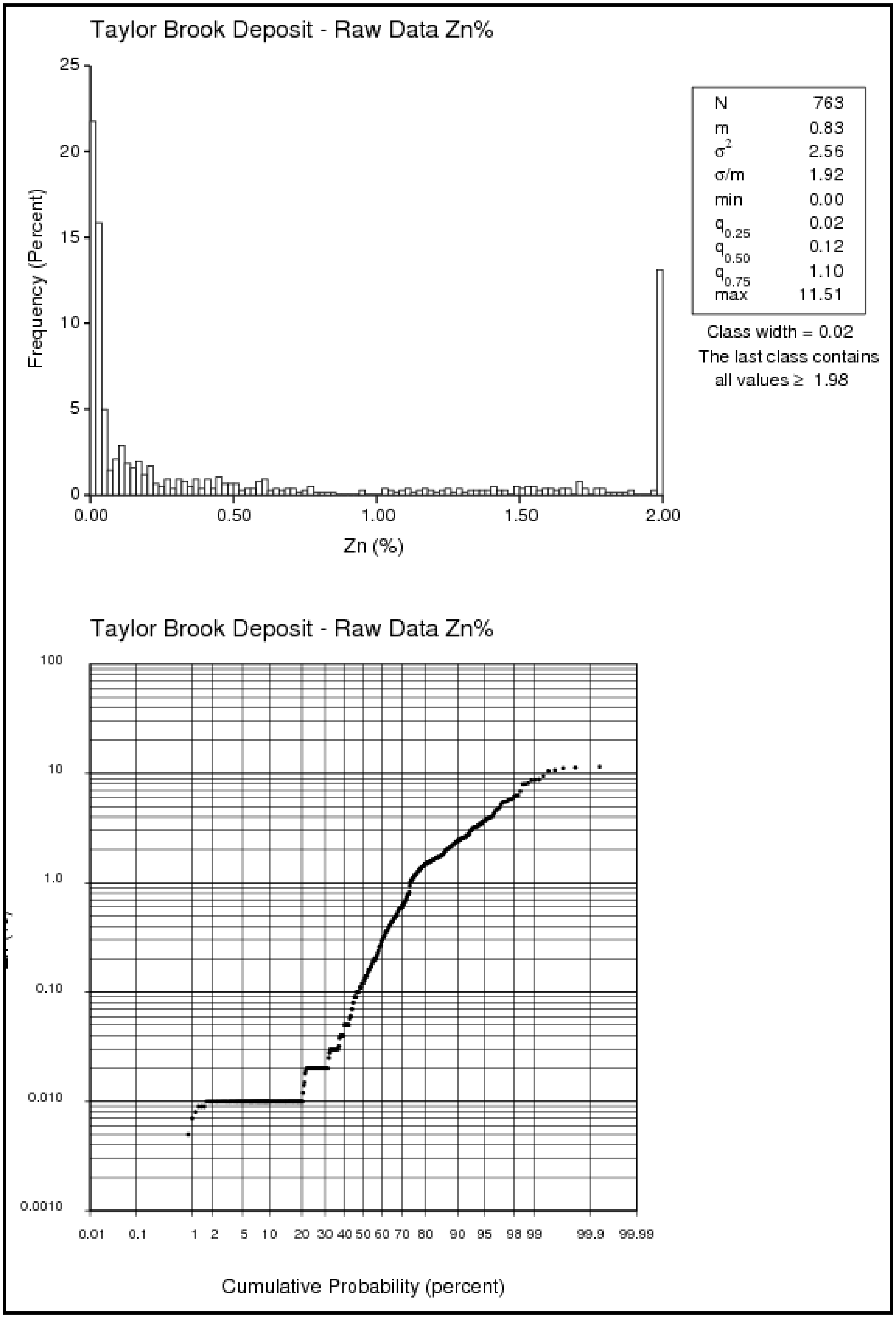


**Figure 14.54 Histogram and Cumulative Probability Plot for Pb**

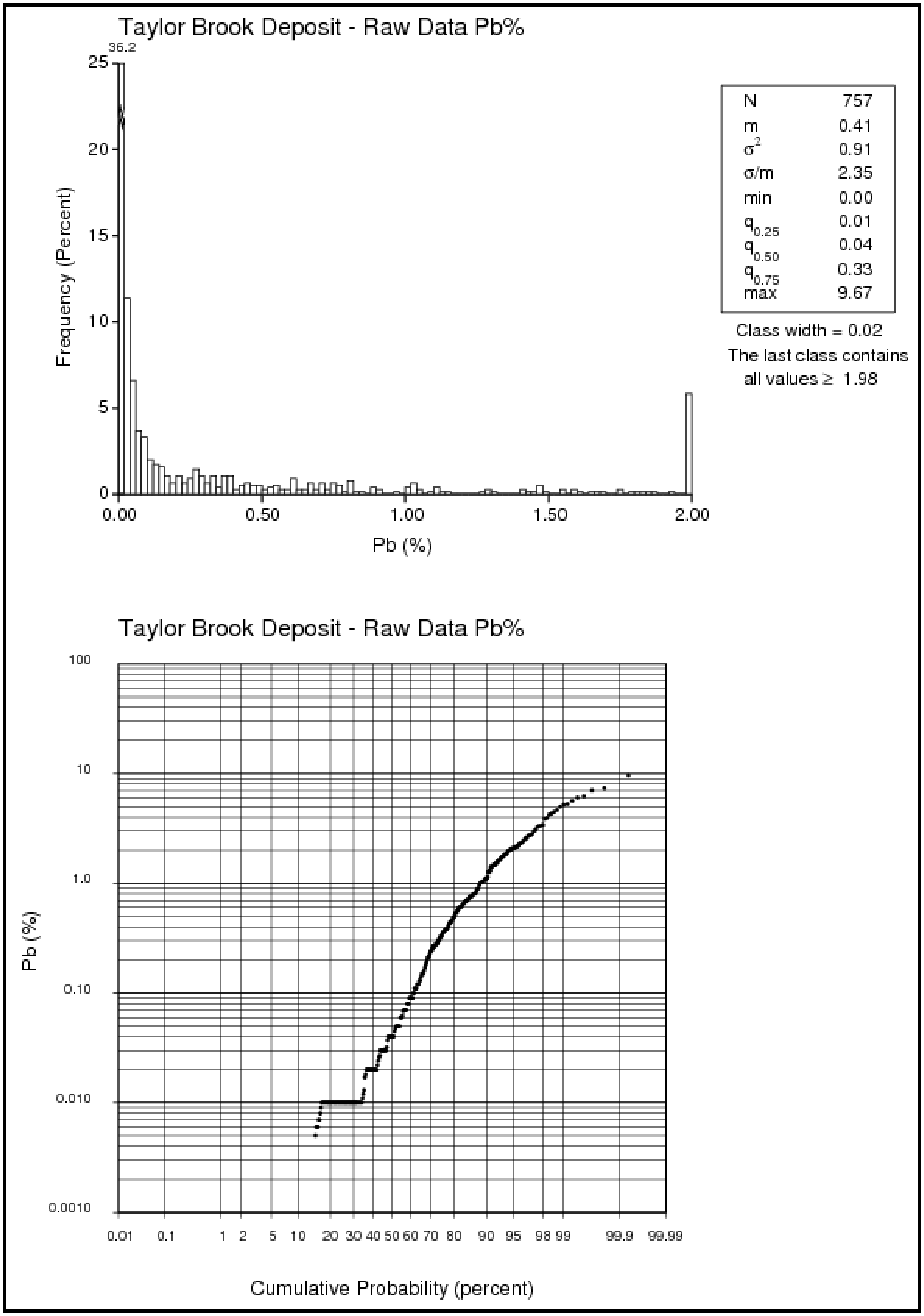


Table 14.29 summarizes the capping effects for the dataset and Table 14.30 summarizes the statistics of the capped assay values (no zeroes).

**Table 14.29 Capping Levels Summary**

| Element | Capped Value | Number of Samples Capped |
|---|---|---|
| Zn% | 10.00 | 5 |
| Pb% | 7.00 | 3 |
| Cu% | 0.30 | 1 |
| Ag (g/t) | 200.00 | 6 |

**Table 14.30 Capped Assay Statistics (No Zeroes)**

| | Length (m) | Zn% | Pb% | Cu% | Ag (g/t) |
|---|---|---|---|---|---|
| Count | 764 | 757 | 645 | 525 | 711 |
| Min | 0.09 | 0.005 | 0.005 | 0.001 | 0.340 |
| Max | 5.00 | 10.000 | 7.000 | 0.300 | 200.000 |
| Mean | 0.91 | 0.832 | 0.472 | 0.025 | 17.607 |
| Std Dev | 0.41 | 1.563 | 0.980 | 0.035 | 29.889 |
| Variance | 0.17 | 2.442 | 0.960 | 0.001 | 893.377 |
| CV | 0.45 | 1.878 | 2.076 | 1.430 | 1.698 |

### COMPOSITES

Composite lengths were created on 1 m intervals to ensure a minimum of six composite samples for each block (in 6 m x 6 m x 6 m blocks) and to maintain a large enough population of data points for a decent estimation of blocks.

Table 14.31 presents the statistics for 1 m composites (no zeroes).

**Table 14.31 1 m Composite Statistics (No Zeroes)**

| | Length (m) | Zn% | Pb% | Cu% | Ag (g/t) |
|---|---|---|---|---|---|
| Count | 592 | 537 | 591 | 453 | 565 |
| Min | 0.021 | 0.005 | 0.007 | 0.001 | 0.340 |
| Max | 1.000 | 6.220 | 10.000 | 0.300 | 200.000 |
| Mean | 0.969 | 0.378 | 0.762 | 0.023 | 16.606 |
| Std Dev | 0.138 | 0.738 | 1.326 | 0.032 | 26.534 |
| Variance | 0.019 | 0.545 | 1.759 | 0.001 | 704.038 |
| CV | 0.142 | 1.951 | 1.741 | 1.440 | 1.598 |

### 14.3.5 GEOLOGICAL INTERPRETATIONS

#### SOLID WIREFRAMES

Three dimensional (3D) rings were created along drill fences for all six wireframes using a combination of raw assay values and lithological boundaries of massive sulphides within the felsic volcanic units. Shoulder samples outside the massive sulphide zones were included where appropriate. The 3D rings at the ends of each solid were copies at a distance equal to half the distance between the last two drillholes. The 3D rings were joined together by tie lines and the solids were created and validated. Rock codes were assigned to the solid wireframes as shown in Table 14.32.

**Table 14.32 Wireframe Solid Names and Explanation**

| Rock Code | Lithological Code | Zone |
|---|---|---|
| 8 | OB | Overburden |
| 101 | MS | Massive Sulphide zones |
| 9 | CR | Country Rock |

Figure 14.55 is a perspective view of the multiple solids used for the resource estimate looking towards the north. Figure 14.56 is the same view looking to the northwest. This figure presents the four mineralized zones in the footwall, with three zones in the west portion of the deposit and one in the east); and one mineralized zone in the hanging wall in the west.

#### TOPOGRAPHIC SURFACE

To create the topographic surface over the Taylor Brook deposit, all drill collar elevations from the 47 drillholes were used to create a surface between these points.

**Figure 14.55 Taylor Brook Deposit Solids and Drillholes (Looking North)**

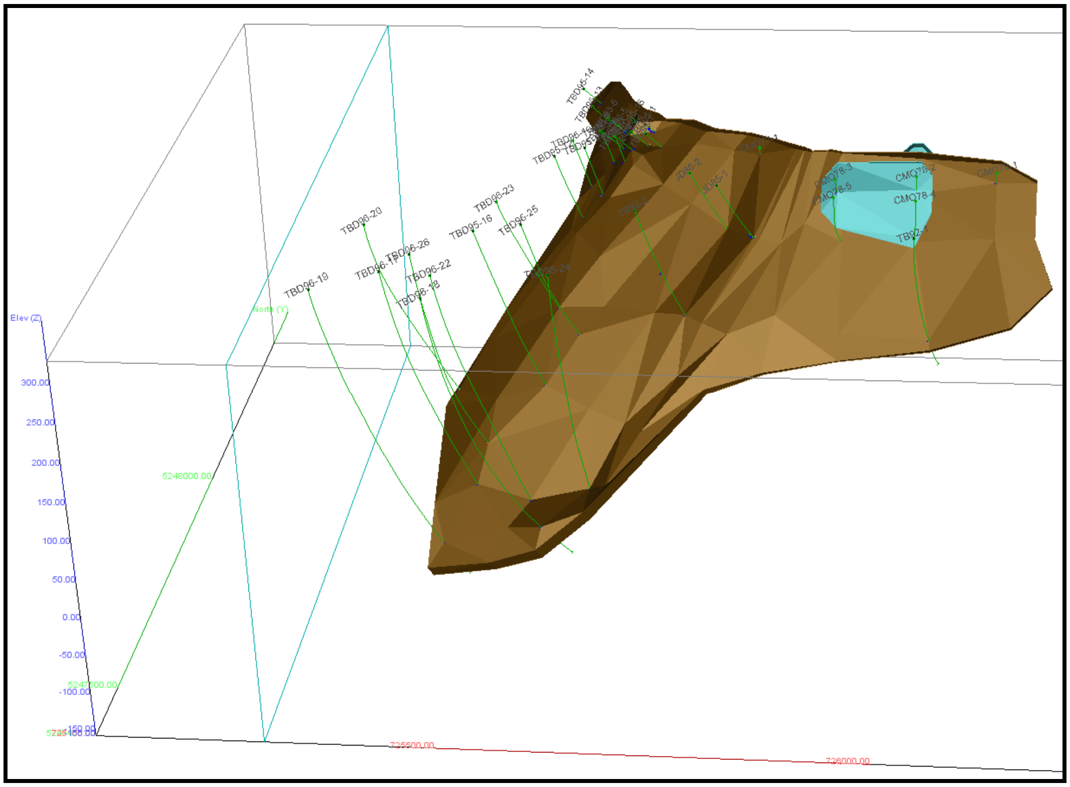
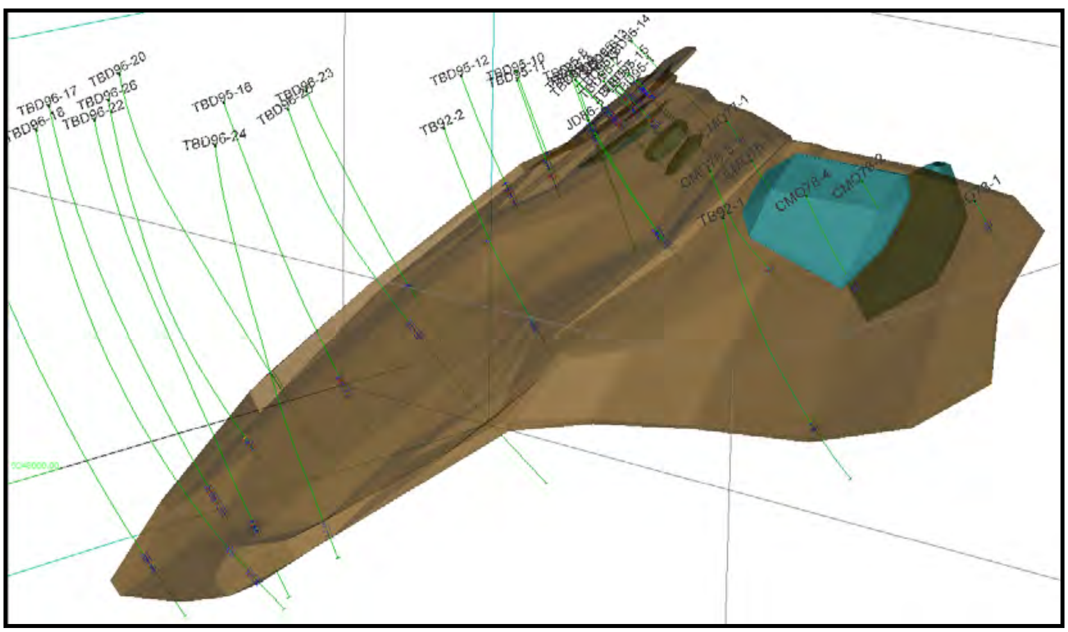



**Figure 14.56 Taylor Brook Deposit Solids and Drillholes (Looking Northeast)**



### 14.3.6 BLOCK MODEL

A single block model was created to cover all six solid wireframes that make up the Taylor Brook deposit. Table 14.33 shows the GEMS coordinates for the block model.

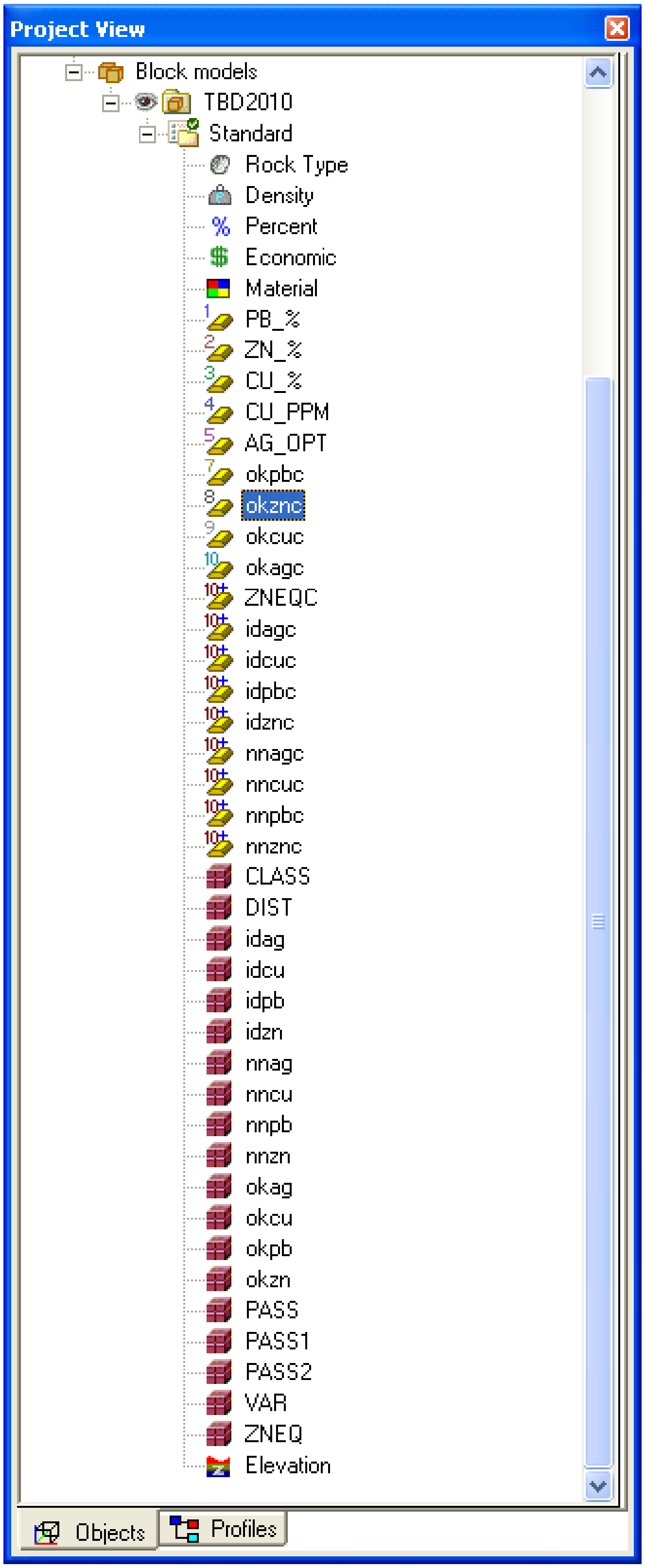
**Table 14.33 Block Model Origin for the Taylor Brook Deposit**

| Element | Minimum | Maximum |
|---|---|---|
| Easting (X) | 725100 | 726396 |
| Northing (Y) | 5247400 | 5248396 |
| Elevation (Z) | -156 | 330 |

#### BLOCK MODEL SIZE

A block size of 6 m x 6 m x 6 m was used to estimate the resources. These parameters were chosen as mining units that may be mined by open pit methods. A screen capture of the block model folders that were created for the block model is shown in Figure 14.57.

**Figure 14.57 Block Model Folders for the Taylor Brook Project**



### 14.3.7 INTERPOLATION AND SPATIAL ANALYSIS

The interpolation methods used for populating the block model and determining resource classification were OK, ID2, and NN.

Three passes were used to interpolate blocks using the OK and ID2 methods; NN used only a single pass. The maximum number of samples per drillhole is limited to three in the first pass (PASS0) and a maximum of four in the second and third pass (PASS1 and PASS2). This is summarized in Table 14.34.

**Table 14.34 Number of Composites and Drillholes Used per Pass**

| Profile Name | No. of Composites | | Maximum No. of Samples per Drillhole | No. of Drillholes | |
|---|---|---|---|---|---|
| | Minimum | Maximum | | Minimum | Maximum |
| PASS0 | 7 | 12 | 3 | 3 | 4 |
| PASS1 | 5 | 15 | 4 | 2 | 4 |
| PASS2 | 5 | 15 | 4 | 2 | 4 |

A detailed list of parameters for the search ellipses for this resource estimate is shown in Table 14.35.

**Table 14.35 Search Ellipse Parameters**

| Profile Name | Rotation | | | Range | | | Search Type |
|---|---|---|---|---|---|---|---|
| | About Z | About X | About Z | X (m) | Y (m) | Z (m) | |
| PASS0 | -45 | 45 | 0 | 50 | 25 | 8 | Ellipsoidal |
| PASS1 | -10 | 45 | -10 | 120 | 60 | 20 | Ellipsoidal |
| PASS2 | -10 | 45 | -10 | 200 | 120 | 30 | Ellipsoidal |

### 14.3.8 ZINC EQUIVALENT CUT-OFF

At the request of Stratabound, Tetra Tech reported the resource estimate in a zinc equivalent (ZnEQ%) cut-off. ZnEQ% is often used in polymetallic deposits to value all other metals in the deposit as an equivalent to zinc. ZnEQ% is calculated based on metal value and metal recovery.

Metallurgical and smelter recoveries are based on previous projects for similar deposits in the Bathurst mining camp.

The following parameters and equation were employed in determining the ZnEQ% values:

**ZnEQ%** = (((Zn Price * Zn Grade * 22.04622 * Zn Recovery) + (Pb Price * Pb Grade * 22.04622 * Pb Recovery) * (Cu Price* Cu Grade * 22.04622 * Cu Recovery)) / (Zinc Price)) / 22.04622

| | | | |
|---|---|---|---|
| Zn Price: | $1.06 | Zn Recovery: | 76.50% |
| Pb Price: | $0.99 | Pb Recovery: | 80.75% |
| Cu Price: | $3.01 | Cu Recovery: | 82.03% |

lbs per tonne: 2,204.622

### 14.3.9 *MINERAL RESOURCE CLASSIFICATION*

The mineral resource for the Taylor Brook deposit is categorized as having an Indicated and Inferred Resource, based on historic drillhole data and limited QA/QC data, economic parameters, and on the drillhole sample support.

The mineral resource was classified into Indicated or Inferred Resources based on the number of drillholes and distance. Individual blocks were classified as Indicated if the block was populated within the first pass and if the distance to the nearest point was less than 30 m, or within the second pass, if two holes were located within 50 m of the block center. The blocks were classified as Inferred if only one hole was within 50 m on the first pass and if two holes were located within 80 m. Any Indicated blocks on the edge of the deposit with only one a single drillhole as support were classified as inferred. Any remaining blocks remained unclassified.

The mineral resource estimates for the Taylor Brook deposit, at 1.60% ZnEQ% cut-off grade is: an Indicated Resource of 243,000 t at 1.69 Zn%, 0.85 Pb%, 0.02 Cu% and 33.42 g/t Ag; and an Inferred Resource of 102,000 t at 1.70 Zn%, 0.87 Pb%, 0.02 Cu% and 32.59 g/t Ag. The OK resource estimates for the massive sulphide zones were made at ZnEQ% cut-off grades from 0.6 Zn% to 2.0 Zn% and presented in Table 14.36 and Table 14.37. No recoveries have been applied to the interpolated estimates as these were applied during the Whittle pit optimization subsequent to the resource estimate.

**Table 14.36 Indicated Resource Estimate for the Taylor Brook Deposit**

| ZnEQ%* Cutoff | Density | Tonnes ('000 t) | Zn% | Pb% | Cu% | ZnEQ%* | Ag (g/t) |
|---|---|---|---|---|---|---|---|
| 0.60% | 3.19 | 1,706 | 0.99 | 0.44 | 0.02 | 1.13 | 19.24 |
| 0.80% | 3.21 | 1,212 | 1.14 | 0.52 | 0.02 | 1.31 | 21.36 |
| 1.00% | 3.23 | 898 | 1.26 | 0.59 | 0.02 | 1.45 | 23.34 |
| 1.20% | 3.27 | 628 | 1.38 | 0.66 | 0.02 | 1.60 | 25.71 |
| 1.40% | 3.27 | 390 | 1.53 | 0.76 | 0.02 | 1.79 | 29.52 |
| 1.60% | 3.28 | 243 | 1.67 | 0.85 | 0.02 | 1.97 | 33.42 |
| 1.80% | 3.27 | 137 | 1.85 | 0.95 | 0.02 | 2.18 | 36.65 |
| 2.00% | 3.31 | 80 | 1.99 | 1.07 | 0.03 | 2.39 | 41.57 |

*ZnEQ%: based on zinc, lead and copper only

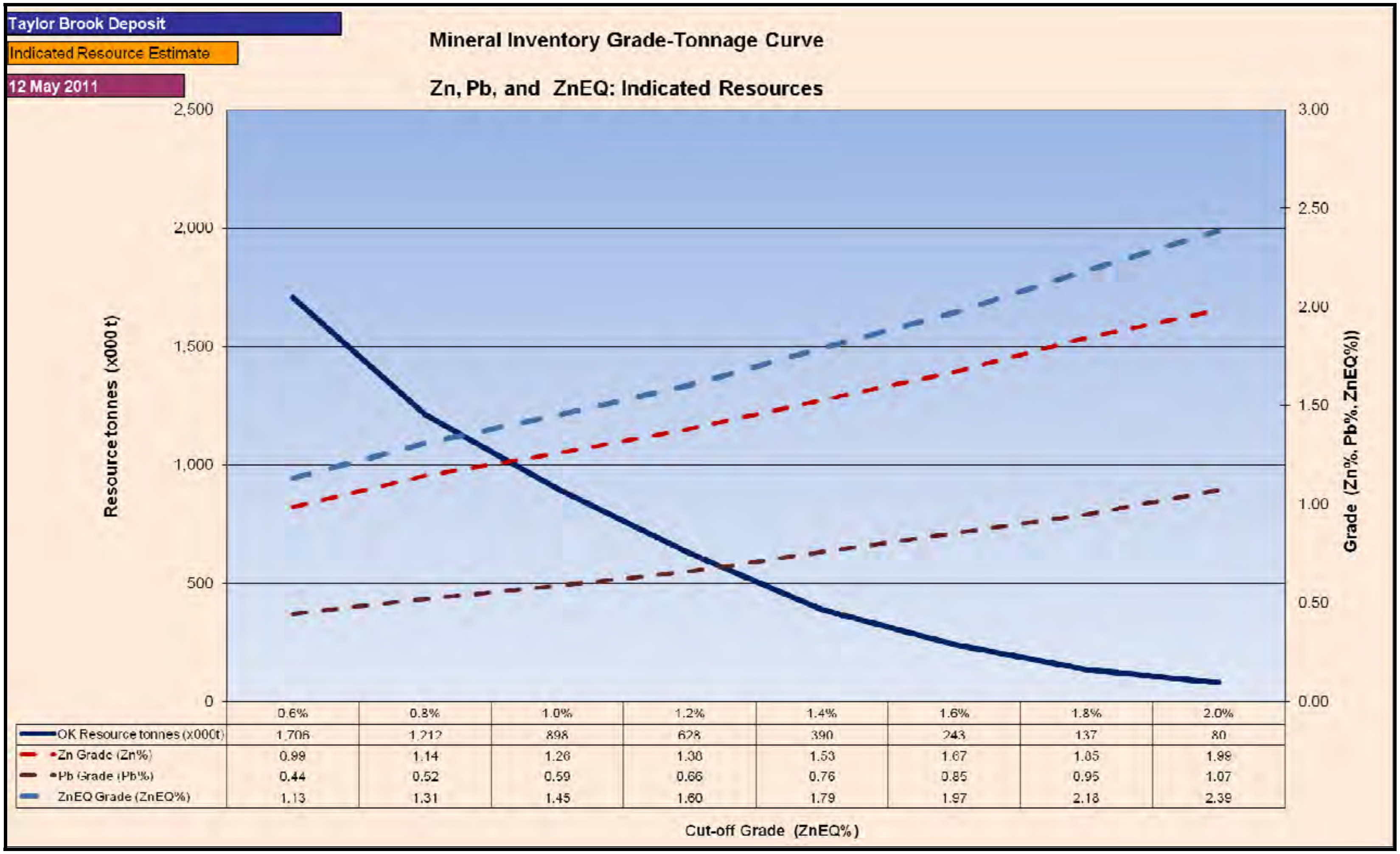
**Table 14.37 Inferred Resource Estimate for the Taylor Brook Deposit**

| ZnEQ%* Cut-off | Density | Tonnes ('000 t) | Zn% | Pb% | Cu% | ZnEQ%* | Ag (g/t) |
|---|---|---|---|---|---|---|---|
| 0.60% | 3.13 | 1,786 | 0.88 | 0.31 | 0.03 | 0.96 | 13.78 |
| 0.80% | 3.21 | 1,037 | 1.06 | 0.40 | 0.02 | 1.17 | 16.62 |
| 1.00% | 3.28 | 634 | 1.21 | 0.47 | 0.03 | 1.35 | 17.64 |
| 1.20% | 3.32 | 332 | 1.39 | 0.58 | 0.03 | 1.57 | 23.11 |
| 1.40% | 3.42 | 181 | 1.55 | 0.72 | 0.02 | 1.79 | 26.75 |
| 1.60% | 3.44 | 102 | 1.70 | 0.87 | 0.02 | 2.01 | 32.59 |
| 1.80% | 3.46 | 57 | 1.87 | 1.00 | 0.03 | 2.26 | 36.36 |
| 2.00% | 3.52 | 42 | 1.97 | 1.07 | 0.03 | 2.39 | 37.57 |

*ZnEQ%: based on zinc, lead and copper only

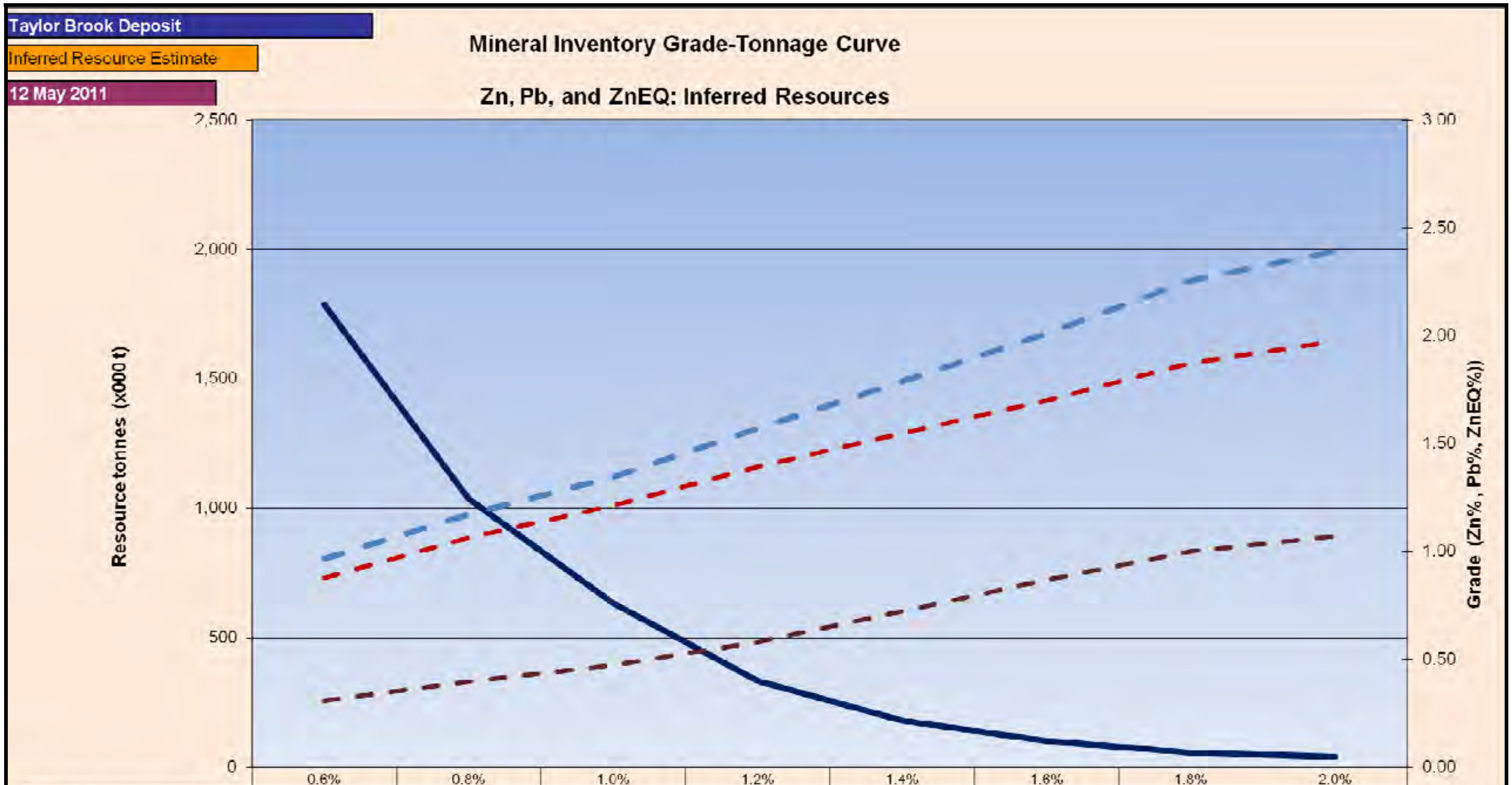
Grade Tonnage curves for Indicated and Inferred Resources for zinc and lead, at the stated ZnEQ% cut-off grades, are presented Figure 14.58 and Figure 14.59.

**Figure 14.58 Grade Tonnage Curves for Indicated Resources (Zn, Pb, and ZnEQ)**




| | 0.6% | 0.8% | 1.0% | 1.2% | 1.4% | 1.6% | 1.8% | 2.0% |
|---|---|---|---|---|---|---|---|---|
| OK Resource tonnes (x000t) | 1,706 | 1,212 | 898 | 628 | 390 | 243 | 137 | 80 |
| Zn Grade (Zn%) | 0.99 | 1.14 | 1.26 | 1.38 | 1.53 | 1.67 | 1.85 | 1.99 |
| Pb Grade (Pb%) | 0.44 | 0.52 | 0.59 | 0.66 | 0.76 | 0.85 | 0.95 | 1.07 |
| ZnEQ Grade (ZnEQ%) | 1.13 | 1.31 | 1.45 | 1.60 | 1.79 | 1.97 | 2.18 | 2.39 |

**Figure 14.59 Grade Tonnage Curves for Indicated Resources (Zn, Pb, and ZnEQ)**




| | 0.6% | 0.8% | 1.0% | 1.2% | 1.4% | 1.6% | 1.8% | 2.0% |
|---|---|---|---|---|---|---|---|---|
| OK Resource tonnes (x000 t) | 1,786 | 1,037 | 634 | 332 | 181 | 102 | 57 | 42 |
| Zn Grade (Zn%) | 0.88 | 1.06 | 1.21 | 1.39 | 1.55 | 1.70 | 1.87 | 1.97 |
| Pb Grade (Zn%) | 0.31 | 0.40 | 0.47 | 0.58 | 0.72 | 0.87 | 1.00 | 1.07 |
| ZnEQ Grade (ZnEQ%) | 0.96 | 1.17 | 1.35 | 1.57 | 1.79 | 2.01 | 2.26 | 2.39 |

Cut-off Grade (ZnEQ%)

### 14.3.10 VALIDATION

#### DEPOSIT VOLUME COMPARISON

The block model volumes were validated against the wireframe volumes and all differences were found to be within a tolerance of less than 1.00%. The results of these comparisons are shown in Table 14.38.

**Table 14.38 Volume Comparison between Wireframe Solid Models and Block Models**

| | Wireframe Total ($m^3$) | Block Model ($m^3$) | Difference ($m^3$) | Difference (%) |
|---|---|---|---|---|
| Main | 3,132,194 | 3132,215 | 21 | <0.01 |
| East1 | 71,053 | 71,083 | 30 | <0.01 |
| East2 | 86,290 | 86,302 | 12 | <0.01 |
| West1 | 10,227 | 10,202 | 25 | 0.25 |
| West2 | 12,009 | 11,950 | 59 | 0.49 |
| West3 | 4,756 | 4,789 | 33 | 0.70 |
| **Total** | **3,316,529** | **3,316,541** | **12** | **<0.01** |

#### STATISTICS COMPARISON

A comparison was made between the composited values and those populated into the block model. The final resource estimate is based on the OK values, and both ID2 and NN methods were run as a validation method. The comparison of the mean grades for zinc, lead, copper and silver shows no extraneous values. The results of the comparison are shown in Table 14.39.

**Table 14.39 Comparison of the Mean Grades**

| | Zn% | Pb% | Cu% | Ag (g/t) |
|---|---|---|---|---|
| OK | 0.191 | 0.508 | 0.021 | 10.096 |
| ID2 | 0.194 | 0.527 | 0.023 | 10.325 |
| NN | 0.190 | 0.465 | 0.021 | 10.150 |
| 1 m Composites | 0.378 | 0.762 | 0.023 | 16.606 |

#### VISUAL COMPARISON

A visual comparison was also made between estimated block grades and the surrounding composite grades used for estimation. This ascertains that estimated grades do not exceed the maximum grade of any one specific 1 m composite.

### SWATH PLOTS

Swath Plots were created for each estimated capped metal grade by bench, by column and by row for each interpolation method as a visual comparison of the precision of the interpolation methods. Figure 14.60, Figure 14.61 and Figure 14.62 show the swath plots for Zn% by bench, column and row respectively.

**Figure 14.60 Swath Plots for Zn% by Bench**

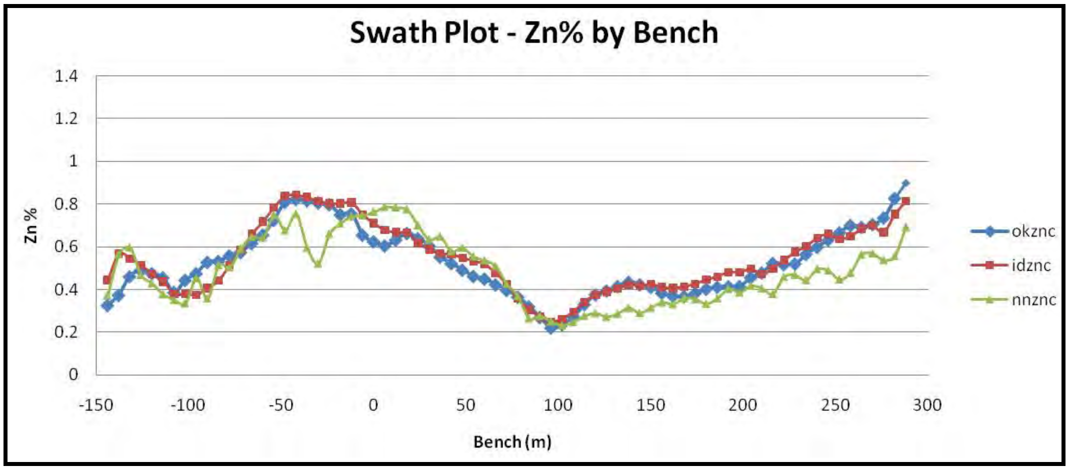



**Figure 14.61 Swath Plots for Zn% by Column**

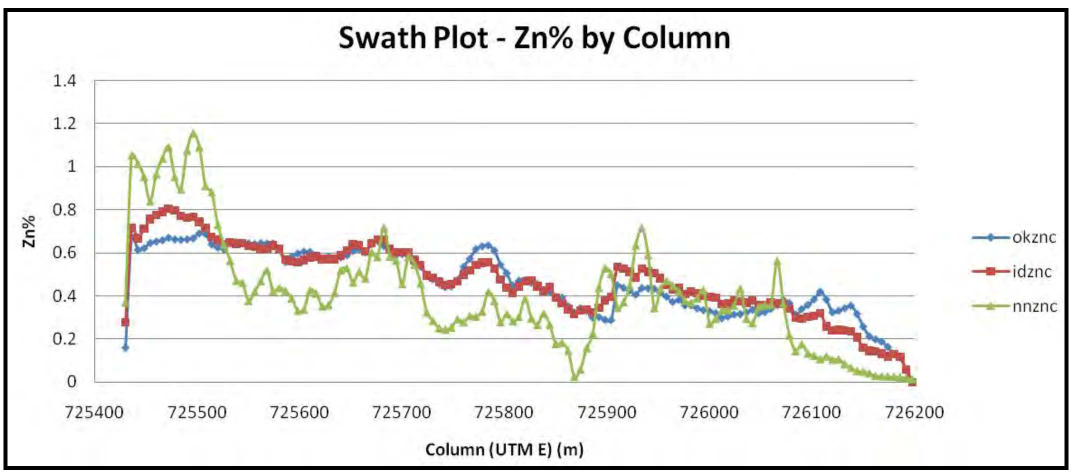
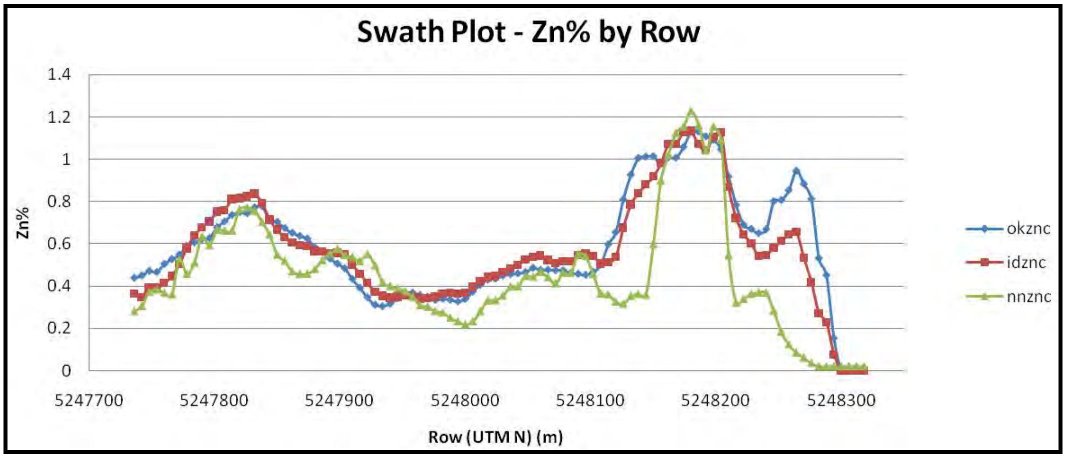


**Figure 14.62 Swath Plots for Zn% by Row**



# 15.0 MINERAL RESERVE ESTIMATES

This section is not applicable to the Captain, CNE or Taylor Brook deposits as an economic assessment at a Prefeasibility level has not been completed for any of the deposits.

# 16.0 MINING METHODS

## 16.1 OVERVIEW

Tetra Tech initiated an open pit design for Stratabound's Taylor Brook, CNE, and Captain properties. For each property, the open pit was designed using a two-stage approach. The first stage identified an optimum pit shell using the LG pit optimization method. In the second stage, phase mining and production schedules were developed, equipment selections were performed, and the capital and operating costs were estimated. The second stage was performed for the CNE and Captain properties as a combined ore feed. CNE was then analyzed as a standalone open pit mine operating without Captain. Taylor Brook did not proceed to the second stage due to poor results from the first stage Whittle™ analysis.

This study analyzed two milling options for a combined Captain/CNE mine operation:

- an owner-operator milling scenario
- a toll mill scenario.

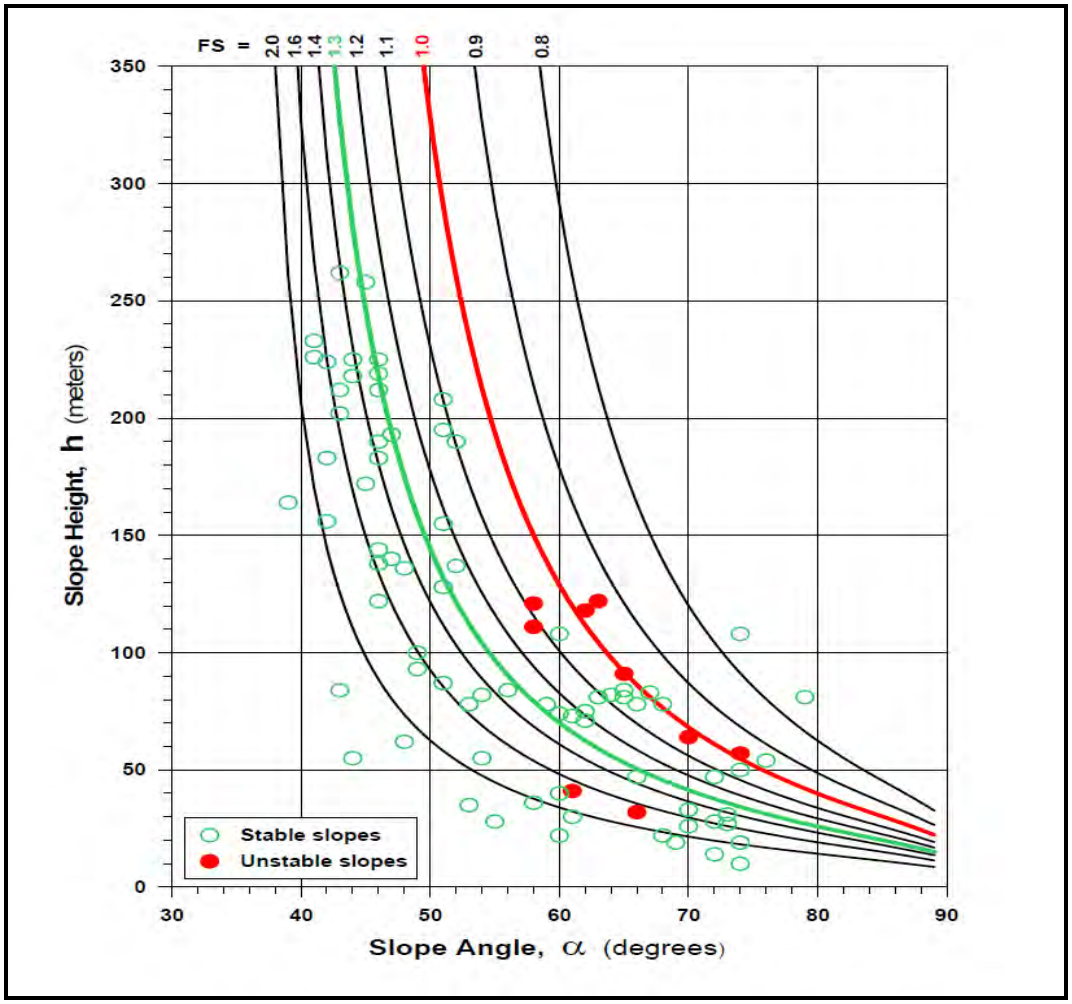
Potential toll mill options include Xstrata's Brunswick 12 mill or the Caribou mill. The original Whittle™ analysis was based on a milling rate of 500 t/d. Ultimately, a toll mill option for CNE as a standalone mine operation was recommended with a milling rate of 1,000 t/d. The mining operations will be performed by a mining contractor utilizing conventional open pit mining methods.

For the standalone CNE open pit model, the ultimate pit design for the project contains 313,357 t of Measured and Indicated Resource at an average diluted grade of 4.77% Zn, 1.76% Pb, 0.08% Cu, and 59.40g/t Ag and 11,664 t of Inferred Resource at an average diluted grade of 3.79% Zn, 1.77% Pb, 0.05% Cu, and 22.71 g/t Ag. The overall stripping ratio is 2.95 t/t (waste/resource). A total of 958,684 t of waste material will be moved over the one-year mine life.

## 16.2 OVERALL OPEN PIT SLOPE ANGLE

Since the required geotechnical data is not available for determining the pit slope angle, Tetra Tech utilized an overall pit slope angle of 20° in overburden and 45°otherwise. The use of these angles is based on conservative estimates from previous experience. Figure 16.1 indicates a 45° overall pit slope is a conservative stable slope to start evaluations. As geotechnical data becomes available, pit slopes could potentially steepen and improve the mine plan and economic evaluation of study.

**Figure 16.1 Cases of Stable and Failed Rock Slopes Conditions**




Note: Reference Paper - "A Risk Evaluation Approach for Pit Slope Design", figure provided by Hoek & Bray, 1974.

## 16.3 PIT OPTIMIZATION

### *16.3.1 PIT OPTIMIZATION PROCEDURES*

Pit optimization was performed to determine the optimum pit limits and evaluate the resource contained within the pit at the highest PV. Tetra Tech used 3D LG algorithm of Gemcom Whittle™ 4.4 commercial software to perform the pit optimization for this project. The results from the optimization are used as a guide for pit creation and phase design.

A 3D geological block model and other required economical and operational variables were used as input parameters of the LG algorithm. These variables include overall pit slope angle, mining cost, milling cost, metal prices, concentrate treatment costs and other parameters as discussed in Section 16.4.2. The LG algorithm progressively identifies economic blocks, taking into account waste stripping, that results in a highest possible total value mined within the open pit shell, subject to the specified pit slope angles.

Tetra Tech conducted the following design steps during the pit optimization stage for each of the Taylor Brook, CNE, and Captain properties:

- Conducted pit optimization:
  - Optimization 1 - owner-operated milling scenario, 500 t/d mine rate
  - Optimization 2 - toll-milling scenario, 500 t/d mine rate.
- Evaluated preliminary production schedules based on best, worse and specified cases.
- Applied operation parameters to the production schedule.
- Selected an optimized (base case) pit shell that represents the highest PV of the specified case.
- For Optimization 2, the process operating cost was increased to represent the expected premium on operating cost pass through for a toll mill agreement. Capital costs are not included in Whittle™ PV calculations. Optimization 1 will have the burden of a mill capital cost, Optimization 2 will not.

### 16.3.2 PIT OPTIMIZATION PARAMETERS

For the optimization, the required parameters were selected by Tetra Tech and Stratabound in evaluating the most economic open pit profile. Although these parameters are not necessarily final, a reasonable degree of accuracy is required, since the analysis is an iterative process. The economic and operating parameters are given in Table 16.1. The economic parameters used at the time of pit optimization do not necessarily conform to those stated in the economic model.

**Table 16.1 Economic Parameters of Pit Optimization**

| Items | | Units | Captain Deposit Value | Captain Deposit Comments | Taylor Brook Deposit Value | Taylor Brook Deposit Comments | CNE Deposit Value | CNE Deposit Comments |
|---|---|---|---|---|---|---|---|---|
| Exchange Rate | | Cdn$:US$ | 1.023 | Tetra Tech forecast, Nov 2010 | 1.023 | Tetra Tech forecast, Nov 2010 | 1.023 | Tetra Tech forecast, Nov 2010 |
| Discount Rate | | % | 8 | Based on Tetra Tech financial model procedure on March 18, 2010 | 8 | Based on Tetra Tech financial model procedure on March 18, 2010 | 8 | Based on Tetra Tech financial model procedure on March 18, 2010 |
| Metal Prices (Market) | Zinc | US$/dry lb | - | Tetra Tech Forecast, Jan 2011 | 1.06 | Tetra Tech Forecast, Jan 2011 | 1.06 | Tetra Tech Forecast, Jan 2011 |
| | Lead | US$/dry lb | - | | 0.99 | | 0.99 | |
| | Copper | US$/dry lb | 3.01 | | 3.01 | | 3.01 | |
| | Cobalt | US$/dry lb | 18.51 | | 18.51 | | 18.51 | |
| | Silver | US$/dry lb | - | | 19.73 | | 19.73 | |
| | Gold | US$/dry lb | - | | 1,221 | | 1,221 | |
| Metal Recoveries | Zinc | % | - | - | 76.5% | 90% mill x 85% smelter | 76.5% | 90% mill x 85% smelter |
| | Lead | % | - | - | 80.8% | 85% mill x 95% smelter | 80.8% | 85% mill x 95% smelter |
| | Copper | % | 89.8% | 96% mill x 93.4% smelter | 82.0% | 85% mill x 96.5% smelter | 82.0% | 85% mill x 96.5% smelter |
| | Cobalt | % | - | - | - | - | - | - |
| | Silver | % | - | - | 45.0% | 50% mill x 90% smelter | 45.0% | 50% mill x 90% smelter |
| | Gold | % | - | - | - | - | - | - |

*table continues…*

| Items | | Units | Captain Deposit | | Taylor Brook Deposit | | CNE Deposit | |
|---|---|---|---|---|---|---|---|---|
| | | | Value | Comments | Value | Comments | Value | Comments |
| Concentrate Costs | Ocean Freight Charge - Europe or N. America (Zn) | US$/wet tonne | - | - | 0.00 | - | 0.00 | - |
| | Ocean Freight Charge - Europe or N. America (Pb) | US$/wet tonne | - | - | 40.00 | - | 40.00 | - |
| | Ocean Freight Charge - Mainland China (Cu) | US$/wet tonne | - | - | 115.00 | - | 115.00 | - |
| Concentrate Costs | Stevedoring Charge (Zn) | US$/wet tonne | - | - | 0.00 | - | 0.00 | - |
| | Stevedoring Charge (Pb) | US$/wet tonne | - | - | 15.00 | - | 15.00 | - |
| | Stevedoring Charge (Cu) | US$/wet tonne | - | - | 15.00 | - | 15.00 | - |
| Concentrate Costs | Representation – Zn | US$/wet tonne | - | - | 0.00 | - | 0.00 | - |
| | Representation – Pb | US$/wet tonne | - | - | 3.00 | - | 3.00 | - |
| | Representation – Cu | US$/wet tonne | - | - | 3.00 | - | 3.00 | - |
| Operating Costs | Mining Cost (OB) | US$/t | 1.10 | - | 1.10 | - | 1.10 | - |
| | Mining Cost (Waste) | US$/t | 2.36 | - | 2.28 | - | 2.28 | - |
| | Mining Cost (Resource) | US$/t | 2.36 | - | 2.84 | - | 2.84 | - |
| | Re-handle of Total Resource Production | % | 100 | - | - | - | - | - |
| | Stockpile Re-handling for Resource(Filling and Hauling) | US$/t resource | 0.49 | - | - | - | - | - |
| | Ore Hauling | US$/t | 1.25 | - | - | - | - | - |
| | Processing - Scenario #1 Owner/Operator | US$/t processed | 12.25 | - | 12.25 | - | 12.25 | - |
| | Processing - Scenario #2 Toll Milling | US$/t processed | 18.90 | - | 18.90 | - | 18.90 | - |
| | G & A Time Cost | US$/tonne resource | 1.96 | - | 1.96 | - | 1.96 | - |

*table continues…*

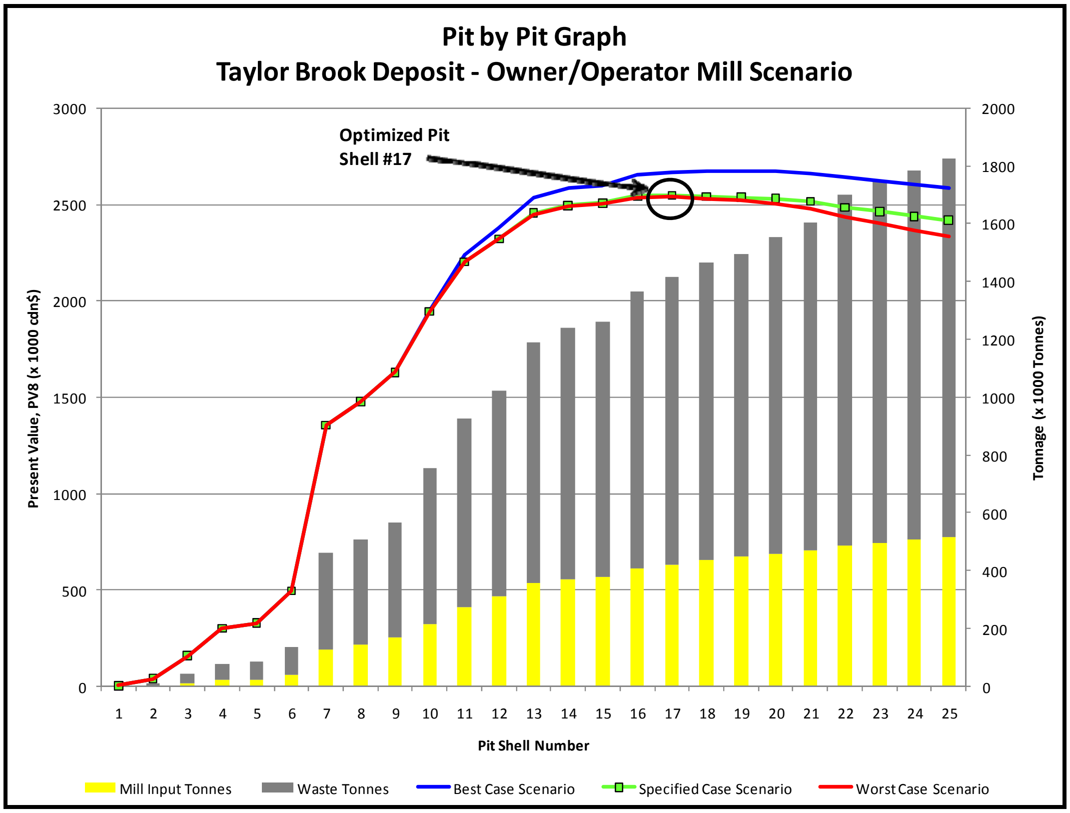
| Items | | Units | Captain Deposit | | Taylor Brook Deposit | | CNE Deposit | |
|---|---|---|---|---|---|---|---|---|
| | | | Value | Comments | Value | Comments | Value | Comments |
| Selling Cost (Concentrate Hauling, Smelting and Refinery Charge) | Zn | US$/lb | - | - | 0.297 | - | 0.297 | - |
| | Pb | US$/lb | - | - | 0.245 | - | 0.245 | - |
| | Cu | US$/lb | 0.546 | - | 0.546 | - | 0.546 | - |
| | Co | US$/lb | - | - | - | - | - | - |
| | Ag | US$/oz | - | - | 0.75 | - | 0.75 | - |
| | Au | US$/oz | - | - | - | - | - | - |
| Overall Pit Slope Angle | Overburden | degrees | 20 | - | 20 | - | 20 | - |
| | Rock | degrees | 45 | - | 45 | - | 45 | - |
| Mining Recovery | | % | 95 | - | 95 | - | 95 | - |
| Mining Dilution | | % | 5 | - | 5 | - | 5 | - |
| Milling Throughput | | t/d | 500 | - | 500 | - | 500 | - |

### 16.3.3 PIT OPTIMIZATION RESULTS

#### TAYLOR BROOK

A series of nested pit shells were generated by varying the revenue factor. Figure 16.2, Figure 16.3 Table 16.2 and Table 16.3 demonstrate the relationship between the resource contained within the pit shell and the PV for each of the nested pit shells for the two optimization scenarios. The specified case represents the most practical operation scenario.

**Figure 16.2 Taylor Brook Nested Pits and Present Values – Optimization 1**




Note: Replicated from chart generated in the Whittle™ 4.4 commercial software

**Table 16.2 Taylor Brook Optimization Results of Nested Pits – Optimization 1**

| Final Pit | Revenue Factor | Open Pit Cash Flow $ Discounted ('000) | | | Resource Tonnes ('000) | Waste Tonnes '000) | Strip Ratio | Grade | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Best | Worst | Specified | | | | Zn% | Pb% | Cu% | Ag% |
| 1 | 0.64 | 4 | 4 | 4 | 0 | 0 | 2.21 | 1.66 | 0.74 | 0.02 | 39.56 |
| 2 | 0.66 | 38 | 38 | 38 | 2 | 6 | 2.88 | 1.36 | 0.74 | 0.02 | 39.89 |
| 3 | 0.68 | 158 | 158 | 158 | 11 | 32 | 3.04 | 1.32 | 0.69 | 0.01 | 35.43 |
| 4 | 0.70 | 301 | 301 | 301 | 22 | 56 | 2.57 | 1.32 | 0.63 | 0.01 | 33.07 |
| 5 | 0.72 | 327 | 327 | 327 | 24 | 63 | 2.69 | 1.34 | 0.64 | 0.02 | 32.78 |
| 6 | 0.74 | 494 | 494 | 494 | 40 | 96 | 2.40 | 1.26 | 0.59 | 0.01 | 32.96 |
| 7 | 0.76 | 1,355 | 1,355 | 1,355 | 127 | 333 | 2.63 | 1.31 | 0.62 | 0.02 | 29.70 |
| 8 | 0.78 | 1,476 | 1,476 | 1,476 | 143 | 363 | 2.53 | 1.30 | 0.61 | 0.02 | 28.95 |
| 9 | 0.80 | 1,629 | 1,629 | 1,629 | 169 | 397 | 2.35 | 1.28 | 0.58 | 0.02 | 28.03 |
| 10 | 0.82 | 1,952 | 1,945 | 1,945 | 215 | 541 | 2.51 | 1.27 | 0.59 | 0.02 | 27.39 |
| 11 | 0.84 | 2,239 | 2,202 | 2,202 | 274 | 651 | 2.38 | 1.23 | 0.55 | 0.02 | 26.78 |
| 12 | 0.86 | 2,380 | 2,322 | 2,319 | 313 | 708 | 2.26 | 1.20 | 0.52 | 0.02 | 26.19 |
| 13 | 0.88 | 2,537 | 2,455 | 2,449 | 356 | 834 | 2.34 | 1.19 | 0.51 | 0.02 | 26.28 |
| 14 | 0.90 | 2,582 | 2,495 | 2,488 | 371 | 868 | 2.34 | 1.18 | 0.51 | 0.02 | 26.12 |
| 15 | 0.92 | 2,598 | 2,508 | 2,501 | 378 | 884 | 2.34 | 1.18 | 0.51 | 0.02 | 25.99 |
| 16 | 0.94 | 2,654 | 2,545 | 2,536 | 409 | 957 | 2.34 | 1.16 | 0.49 | 0.02 | 25.69 |
| **17** | **0.96** | **2,668** | **2,548** | **2,539** | **421** | **994** | **2.36** | **1.16** | **0.49** | **0.02** | **25.47** |
| 18 | 0.98 | 2,675 | 2,541 | 2,531 | 437 | 1,030 | 2.36 | 1.15 | 0.48 | 0.02 | 25.18 |
| 19 | 1.00 | 2,676 | 2,538 | 2,521 | 447 | 1,050 | 2.35 | 1.15 | 0.48 | 0.02 | 25.12 |
| 20 | 1.02 | 2,672 | 2,530 | 2,504 | 457 | 1,096 | 2.40 | 1.14 | 0.48 | 0.02 | 25.09 |
| 21 | 1.04 | 2,663 | 2,516 | 2,479 | 469 | 1,134 | 2.42 | 1.14 | 0.47 | 0.02 | 25.03 |
| 22 | 1.06 | 2,639 | 2,485 | 2,432 | 488 | 1,213 | 2.49 | 1.13 | 0.46 | 0.02 | 24.84 |

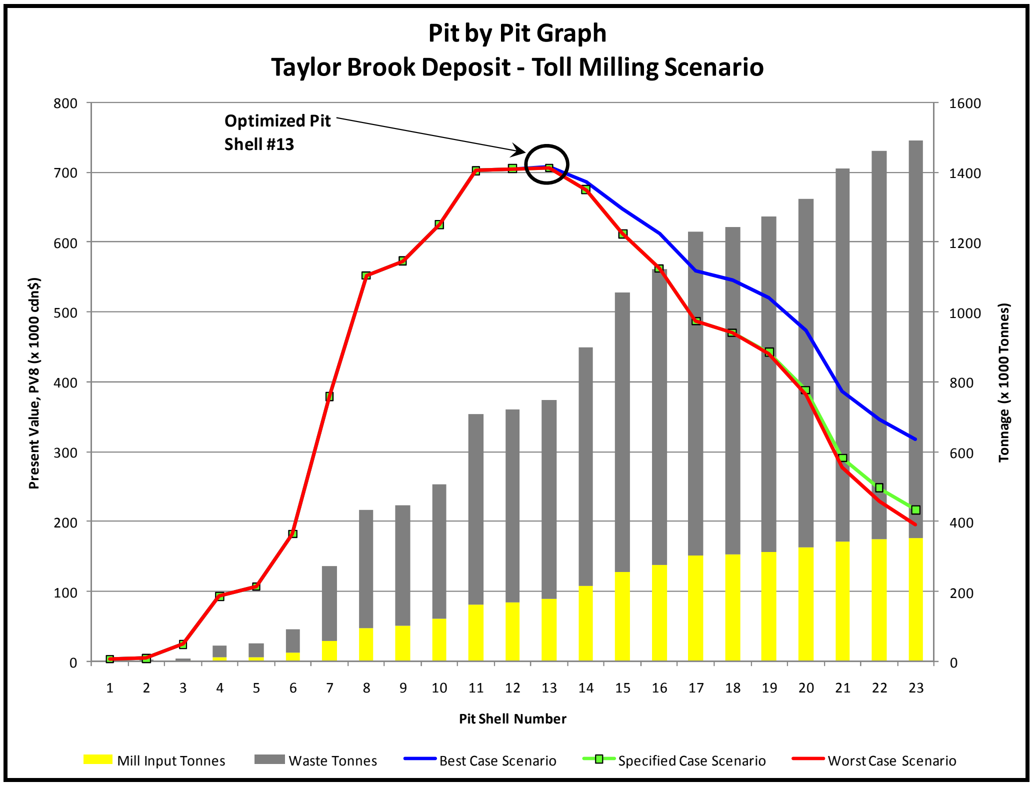
*table continues…*

| Final Pit | Revenue Factor | Open Pit Cash Flow $ Discounted ('000) | | | Resource Tonnes ('000) | Waste Tonnes '000) | Strip Ratio | Grade | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Best | Worst | Specified | | | | Zn% | Pb% | Cu% | Ag% |
| 23 | 1.08 | 2,623 | 2,464 | 2,404 | 497 | 1,248 | 2.51 | 1.13 | 0.46 | 0.02 | 24.66 |
| 24 | 1.10 | 2,602 | 2,439 | 2,367 | 508 | 1,278 | 2.52 | 1.12 | 0.46 | 0.02 | 24.70 |
| 25 | 1.12 | 2,583 | 2,417 | 2,335 | 516 | 1,308 | 2.54 | 1.12 | 0.45 | 0.02 | 24.61 |
| 26 | 1.14 | 2,536 | 2,367 | 2,270 | 532 | 1,359 | 2.55 | 1.11 | 0.45 | 0.02 | 24.24 |
| 27 | 1.16 | 2,495 | 2,324 | 2,217 | 545 | 1,399 | 2.57 | 1.10 | 0.44 | 0.02 | 24.21 |
| 28 | 1.18 | 2,474 | 2,302 | 2,189 | 550 | 1,426 | 2.59 | 1.10 | 0.44 | 0.02 | 24.24 |

The following features are identified:

- Pit shell #17 generates the highest PV at Cdn$2.5 million for the specified case, based on a 500 t/d operation over a mine life of approximately 2.5 years. The selected base case pit shell contains 421,000 t of resource with average grades of:
  - 1.16% Zn
  - 0.49% Pb
  - 0.02% Cu
  - 25.47g/t Au
- For pit shells larger than pit #17, although the resource is progressively increased, PV is gradually decreased. The decrease in cash flow results from resource grades gradually decreasing, stripping ratio gradually increasing, and the time value of money (discount rate).
- Pit #17 is an optimized pit shell or base case pit shell.
- The pit optimization study includes inferred geological resources that are considered too speculative geologically to be considered reserves and there is no certainty that the present values as determined by the Whittle™ analysis will be realized.

**Figure 16.3 Taylor Brook Nested Pits and Present Values - Optimization 2**




Note: Replicated from chart generated in the Whittle™ 4.4 commercial software

**Table 16.3 Taylor Brook Optimization Results of Nested Pits – Optimization 2**

| Final Pit | Revenue Factor | Open Pit Cash Flow $ Discounted ('000) | | | Resource Tonnes ('000) | Waste Tonnes ('000) | Strip Ratio | Grade | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Best | Worst | Specified | | | | Zn% | Pb% | Cu% | Ag% |
| 1 | 0.74 | 3 | 3 | 3 | 0 | 0 | 2.21 | 1.66 | 0.74 | 0.02 | 39.56 |
| 2 | 0.78 | 4 | 4 | 4 | 0 | 1 | 3.64 | 1.58 | 0.80 | 0.02 | 43.53 |
| 3 | 0.80 | 24 | 24 | 24 | 2 | 6 | 2.93 | 1.37 | 0.74 | 0.02 | 40.10 |
| 4 | 0.82 | 93 | 93 | 93 | 10 | 33 | 3.42 | 1.39 | 0.73 | 0.01 | 36.86 |
| 5 | 0.84 | 107 | 107 | 107 | 11 | 39 | 3.44 | 1.38 | 0.72 | 0.01 | 37.18 |
| 6 | 0.86 | 182 | 182 | 182 | 23 | 69 | 3.06 | 1.39 | 0.67 | 0.02 | 33.51 |
| 7 | 0.88 | 379 | 379 | 379 | 59 | 214 | 3.59 | 1.43 | 0.67 | 0.02 | 33.30 |
| 8 | 0.90 | 552 | 552 | 552 | 94 | 339 | 3.61 | 1.43 | 0.68 | 0.02 | 32.59 |
| 9 | 0.92 | 573 | 573 | 573 | 101 | 345 | 3.41 | 1.41 | 0.68 | 0.02 | 32.01 |
| 10 | 0.94 | 625 | 625 | 625 | 123 | 382 | 3.12 | 1.38 | 0.66 | 0.02 | 30.44 |
| 11 | 0.96 | 702 | 702 | 702 | 163 | 545 | 3.35 | 1.39 | 0.66 | 0.02 | 29.68 |
| 12 | 0.98 | 705 | 705 | 705 | 168 | 554 | 3.30 | 1.38 | 0.66 | 0.02 | 29.42 |
| **13** | **1.00** | **707** | **706** | **706** | **179** | **567** | **3.17** | **1.37** | **0.64** | **0.02** | **28.60** |
| 14 | 1.02 | 686 | 675 | 675 | 216 | 682 | 3.15 | 1.33 | 0.62 | 0.02 | 28.61 |
| 15 | 1.04 | 647 | 612 | 612 | 254 | 803 | 3.17 | 1.31 | 0.59 | 0.02 | 28.11 |
| 16 | 1.06 | 613 | 562 | 562 | 275 | 848 | 3.08 | 1.29 | 0.57 | 0.02 | 27.40 |
| 17 | 1.08 | 558 | 487 | 487 | 301 | 928 | 3.08 | 1.28 | 0.56 | 0.02 | 27.38 |
| 18 | 1.10 | 545 | 470 | 470 | 305 | 939 | 3.08 | 1.27 | 0.56 | 0.02 | 27.37 |
| 19 | 1.12 | 521 | 442 | 440 | 312 | 962 | 3.08 | 1.27 | 0.55 | 0.02 | 27.09 |
| 20 | 1.14 | 474 | 388 | 381 | 325 | 998 | 3.07 | 1.26 | 0.54 | 0.02 | 27.02 |
| 21 | 1.16 | 387 | 291 | 277 | 342 | 1,070 | 3.13 | 1.25 | 0.53 | 0.02 | 26.62 |
| 22 | 1.18 | 347 | 248 | 229 | 348 | 1,114 | 3.20 | 1.25 | 0.53 | 0.02 | 26.58 |
| 23 | 1.20 | 317 | 217 | 196 | 353 | 1,139 | 3.23 | 1.25 | 0.53 | 0.02 | 26.42 |

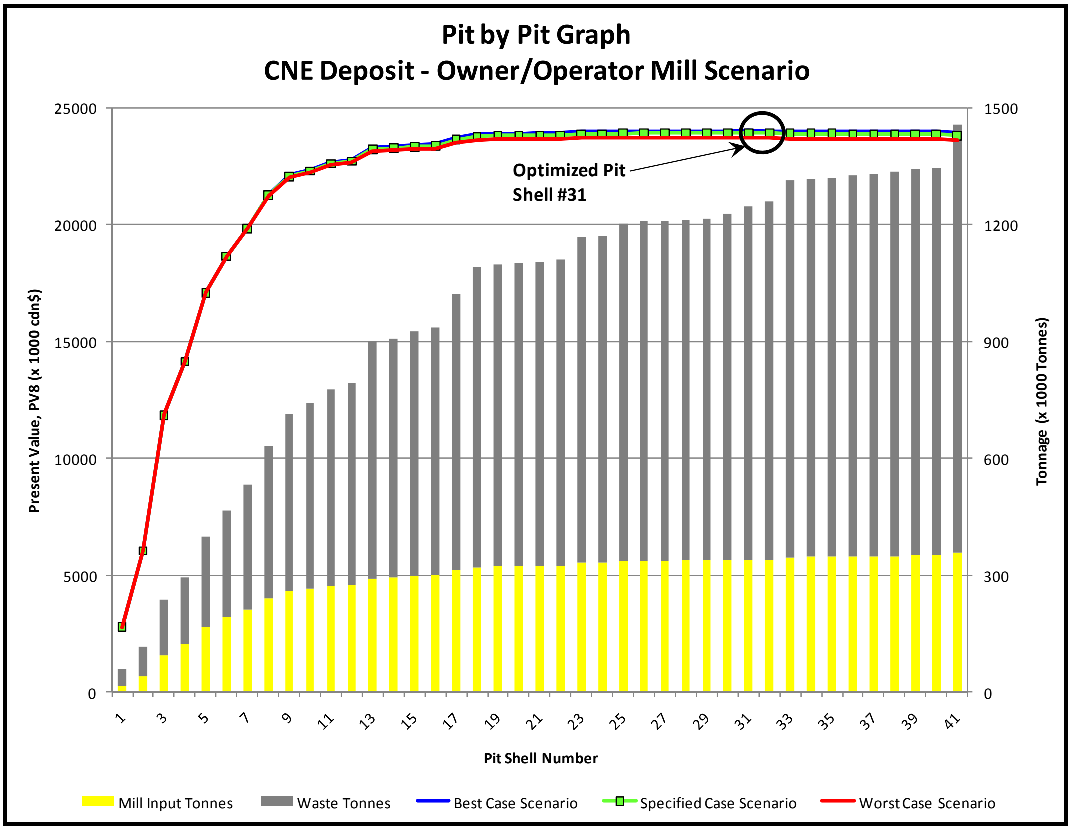
The following features are identified:

- Pit shell #13 generates the highest PV at Cdn$0.7 million for the specified case, based on a 500 t/d operation over a mine life of approximately one-year. The selected base case pit shell contains 179,000 t of resource with average grades of:
  - 1.37% Zn
  - 0.64% Pb
  - 0.02% Cu
  - 28.60 g/t Ag
- For pit shells larger than pit #13, although the resource is progressively increased, PV is gradually decreased. The decrease in cash flow results from resource grades gradually decreasing, stripping ratio gradually increasing, and the time value of money (discount rate).
- Pit #13 is an optimized pit shell or base case pit shell.
- The pit optimization study includes inferred geological resources that are considered too speculative geologically to be considered reserves and there is no certainty that the present values as determined by the Whittle analysis will be realized.

### CNE

A series of nested pit shells were generated by varying the revenue factor. Figure 16.4, Figure 16.5, Table 16.4 and Table 16.5 demonstrate the relationship between the resource contained within the pit shell and the PV for each of the nested pit shells for the two optimization scenarios. The specified case represents the most practical operation scenario.

**Figure 16.4 CNE Nested Pits and Present Values – Optimization 1**




Note: Replicated from chart generated in the Whittle™ 4.4 commercial software

## Table 16.4 CNE Optimization Results of Nested Pits – Optimization 1

| Final Pit | Revenue Factor | Open Pit Cash Flow $ Discounted ('000) Best | Worst | Specified | Resource Tonnes ('000) | Waste Tonnes ('000) | Strip Ratio | Grade Pb (%) | Ag (g/t) | Cu (%) | Zn (%) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 0.32 | 2,772 | 2,772 | 2,772 | 15 | 43 | 2.79 | 4.04 | 136.92 | 0.01 | 8.70 |
| 2 | 0.34 | 6,042 | 6,042 | 6,042 | 39 | 78 | 2.03 | 3.48 | 113.92 | 0.01 | 7.98 |
| 3 | 0.36 | 11,848 | 11,848 | 11,848 | 93 | 143 | 1.54 | 2.81 | 85.10 | 0.05 | 7.05 |
| 4 | 0.38 | 14,132 | 14,132 | 14,132 | 121 | 174 | 1.43 | 2.61 | 79.12 | 0.06 | 6.57 |
| 5 | 0.40 | 17,083 | 17,083 | 17,083 | 166 | 233 | 1.40 | 2.37 | 71.90 | 0.05 | 6.04 |
| 6 | 0.42 | 18,654 | 18,649 | 18,649 | 193 | 273 | 1.42 | 2.25 | 68.98 | 0.05 | 5.79 |
| 7 | 0.44 | 19,856 | 19,832 | 19,832 | 212 | 321 | 1.51 | 2.18 | 67.01 | 0.05 | 5.66 |
| 8 | 0.46 | 21,314 | 21,273 | 21,250 | 240 | 392 | 1.63 | 2.07 | 64.65 | 0.05 | 5.47 |
| 9 | 0.48 | 22,127 | 22,073 | 22,028 | 259 | 454 | 1.75 | 1.99 | 63.81 | 0.05 | 5.31 |
| 10 | 0.50 | 22,352 | 22,295 | 22,243 | 264 | 478 | 1.81 | 1.97 | 63.72 | 0.05 | 5.27 |
| 11 | 0.52 | 22,669 | 22,607 | 22,546 | 272 | 506 | 1.86 | 1.95 | 63.13 | 0.05 | 5.22 |
| 12 | 0.54 | 22,788 | 22,724 | 22,654 | 276 | 517 | 1.87 | 1.93 | 62.72 | 0.05 | 5.18 |
| 13 | 0.56 | 23,302 | 23,228 | 23,135 | 292 | 608 | 2.08 | 1.88 | 62.01 | 0.06 | 5.04 |
| 14 | 0.58 | 23,344 | 23,268 | 23,171 | 294 | 613 | 2.08 | 1.87 | 61.78 | 0.06 | 5.03 |
| 15 | 0.60 | 23,411 | 23,334 | 23,233 | 297 | 628 | 2.12 | 1.86 | 61.72 | 0.06 | 5.01 |
| 16 | 0.62 | 23,457 | 23,378 | 23,271 | 299 | 636 | 2.13 | 1.86 | 61.41 | 0.06 | 4.98 |
| 17 | 0.64 | 23,719 | 23,633 | 23,504 | 312 | 711 | 2.28 | 1.81 | 60.25 | 0.07 | 4.87 |
| 18 | 0.66 | 23,876 | 23,785 | 23,642 | 320 | 770 | 2.41 | 1.79 | 59.59 | 0.07 | 4.80 |
| 19 | 0.68 | 23,903 | 23,810 | 23,661 | 322 | 776 | 2.41 | 1.78 | 59.28 | 0.07 | 4.78 |
| 20 | 0.70 | 23,909 | 23,816 | 23,664 | 323 | 778 | 2.41 | 1.78 | 59.18 | 0.07 | 4.77 |
| 21 | 0.72 | 23,915 | 23,822 | 23,667 | 323 | 780 | 2.41 | 1.78 | 59.08 | 0.07 | 4.77 |
| 22 | 0.76 | 23,924 | 23,830 | 23,674 | 324 | 785 | 2.42 | 1.78 | 58.99 | 0.07 | 4.76 |

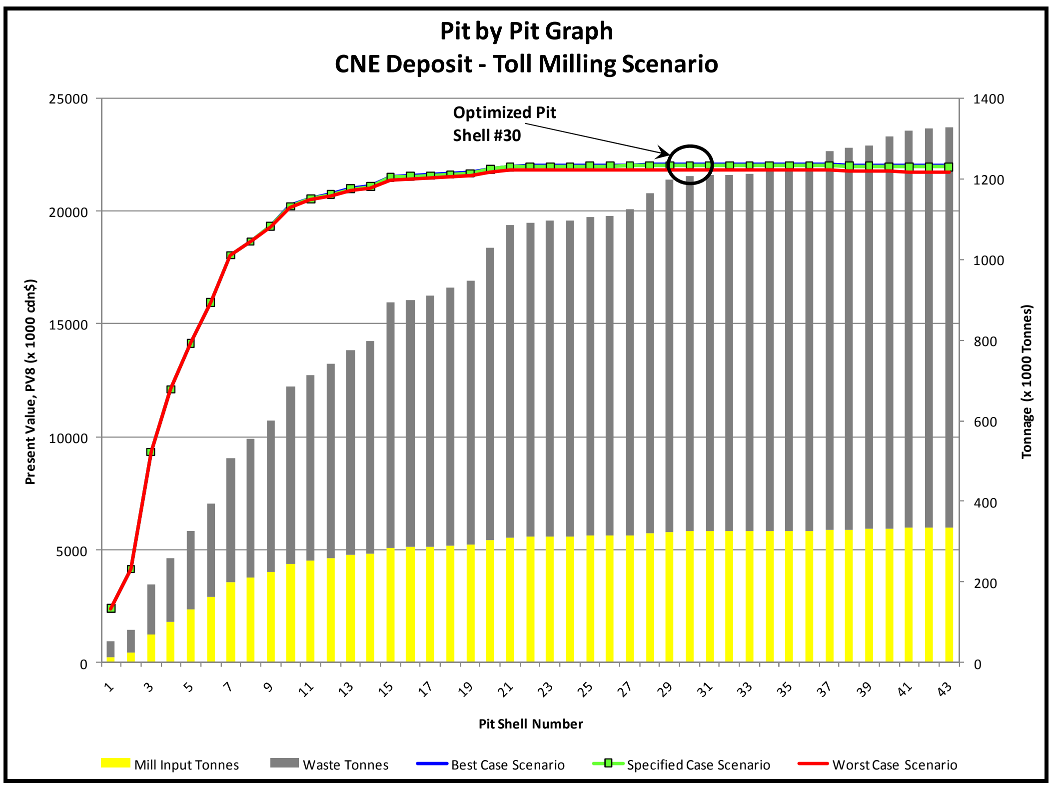
*table continues…*

| Final Pit | Revenue Factor | Open Pit Cash Flow $ Discounted ('000) Best | Worst | Specified | Resource Tonnes ('000) | Waste Tonnes ('000) | Strip Ratio | Grade Pb (%) | Ag (g/t) | Cu (%) | Zn (%) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 23 | 0.78 | 23,979 | 23,881 | 23,711 | 331 | 836 | 2.53 | 1.75 | 58.34 | 0.08 | 4.69 |
| 24 | 0.80 | 23,982 | 23,884 | 23,712 | 332 | 838 | 2.52 | 1.75 | 58.20 | 0.08 | 4.68 |
| 25 | 0.82 | 24,008 | 23,907 | 23,729 | 336 | 866 | 2.58 | 1.73 | 57.81 | 0.08 | 4.64 |
| 26 | 0.84 | 24,012 | 23,911 | 23,732 | 336 | 873 | 2.60 | 1.73 | 57.84 | 0.08 | 4.64 |
| 27 | 0.86 | 24,013 | 23,912 | 23,732 | 336 | 873 | 2.60 | 1.73 | 57.79 | 0.08 | 4.64 |
| 28 | 0.88 | 24,013 | 23,912 | 23,731 | 337 | 875 | 2.60 | 1.73 | 57.72 | 0.08 | 4.63 |
| 29 | 0.90 | 24,014 | 23,912 | 23,730 | 337 | 879 | 2.61 | 1.73 | 57.65 | 0.08 | 4.63 |
| 30 | 0.92 | 24,014 | 23,911 | 23,724 | 339 | 888 | 2.62 | 1.72 | 57.40 | 0.08 | 4.61 |
| **31** | **0.94** | **24,016** | **23,913** | **23,723** | **339** | **907** | **2.67** | **1.72** | **57.37** | **0.08** | **4.61** |
| 32 | 0.96 | 24,015 | 23,912 | 23,719 | 340 | 920 | 2.70 | 1.72 | 57.27 | 0.08 | 4.60 |
| 33 | 0.98 | 24,003 | 23,896 | 23,693 | 346 | 968 | 2.80 | 1.69 | 56.56 | 0.09 | 4.54 |
| 34 | 1.00 | 24,003 | 23,895 | 23,692 | 347 | 971 | 2.80 | 1.69 | 56.55 | 0.09 | 4.54 |
| 35 | 1.02 | 24,002 | 23,895 | 23,691 | 347 | 972 | 2.80 | 1.69 | 56.54 | 0.09 | 4.54 |
| 36 | 1.04 | 23,998 | 23,890 | 23,683 | 348 | 978 | 2.81 | 1.69 | 56.41 | 0.09 | 4.53 |
| 37 | 1.06 | 23,995 | 23,886 | 23,679 | 348 | 983 | 2.82 | 1.69 | 56.33 | 0.09 | 4.52 |
| 38 | 1.08 | 23,991 | 23,882 | 23,673 | 349 | 986 | 2.83 | 1.69 | 56.22 | 0.09 | 4.51 |
| 39 | 1.10 | 23,985 | 23,875 | 23,665 | 350 | 991 | 2.83 | 1.68 | 56.04 | 0.09 | 4.50 |
| 40 | 1.16 | 23,983 | 23,873 | 23,662 | 351 | 996 | 2.84 | 1.68 | 55.99 | 0.09 | 4.50 |
| 41 | 1.20 | 23,927 | 23,817 | 23,597 | 357 | 1,101 | 3.09 | 1.66 | 55.38 | 0.10 | 4.44 |

The following features are identified:

- Pit shell #31 generates the highest PV at Cdn$23.9 million for the specified case, based on a 500 t/d operation over a mine life of approximately 2.0 years. The selected base case pit shell contains 339,000 t of resource with average grades of:
  - 4.61% Zn
  - 1.72% Pb
  - 0.08% Cu
  - 57.37 g/t Au
- For pit shells larger than pit #31, although the resource is progressively increased, PV is gradually decreased. The decrease in cash flow results from resource grades gradually decreasing, stripping ratio gradually increasing, and the time value of money (discount rate).
- Pit #31 is an optimized pit shell or base case pit shell.
- The pit optimization study includes inferred geological resources that are considered too speculative geologically to be considered reserves and there is no certainty that the present values as determined by the Whittle analysis will be realized.

**Figure 16.5 CNE Nested Pits and Present Values - Optimization 2**




Note: Replicated from chart generated in the Whittle™ 4.4 commercial software

### Table 16.5 CNE Optimization Results of Nested Pits – Optimization 2

| Final Pit | Revenue Factor | Open Pit Cash Flow $ Discounted ('000) | | | Resource Tonnes ('000) | Waste Tonnes ('000) | Strip Ratio | Grade | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Best | Worst | Specified | | | | Pb (%) | Ag (g/t) | Cu (%) | Zn (%) |
| 1 | 0.34 | 2,402 | 2,402 | 2,402 | 13 | 40 | 2.97 | 4.15 | 137.75 | 0.01 | 8.94 |
| 2 | 0.36 | 4,133 | 4,133 | 4,133 | 25 | 55 | 2.26 | 3.99 | 118.00 | 0.00 | 8.74 |
| 3 | 0.38 | 9,325 | 9,325 | 9,325 | 71 | 122 | 1.73 | 2.99 | 91.53 | 0.05 | 7.51 |
| 4 | 0.40 | 12,093 | 12,093 | 12,093 | 102 | 157 | 1.54 | 2.74 | 82.50 | 0.06 | 6.95 |
| 5 | 0.42 | 14,140 | 14,140 | 14,140 | 132 | 195 | 1.48 | 2.56 | 77.16 | 0.06 | 6.48 |
| 6 | 0.44 | 15,964 | 15,964 | 15,964 | 162 | 233 | 1.44 | 2.41 | 72.52 | 0.05 | 6.15 |
| 7 | 0.46 | 18,045 | 18,038 | 18,038 | 199 | 308 | 1.55 | 2.25 | 69.29 | 0.04 | 5.83 |
| 8 | 0.48 | 18,667 | 18,656 | 18,649 | 210 | 346 | 1.65 | 2.20 | 68.41 | 0.05 | 5.76 |
| 9 | 0.50 | 19,361 | 19,345 | 19,325 | 225 | 376 | 1.67 | 2.15 | 67.11 | 0.04 | 5.64 |
| 10 | 0.52 | 20,237 | 20,214 | 20,171 | 245 | 440 | 1.80 | 2.07 | 65.53 | 0.05 | 5.49 |
| 11 | 0.54 | 20,572 | 20,546 | 20,490 | 254 | 460 | 1.81 | 2.03 | 65.02 | 0.05 | 5.41 |
| 12 | 0.56 | 20,758 | 20,731 | 20,668 | 258 | 482 | 1.87 | 2.02 | 64.96 | 0.05 | 5.38 |
| 13 | 0.58 | 21,020 | 20,990 | 20,917 | 266 | 510 | 1.92 | 1.99 | 64.45 | 0.05 | 5.33 |
| 14 | 0.60 | 21,121 | 21,091 | 21,013 | 269 | 529 | 1.97 | 1.98 | 64.31 | 0.05 | 5.31 |
| 15 | 0.62 | 21,531 | 21,495 | 21,393 | 284 | 610 | 2.15 | 1.92 | 63.63 | 0.05 | 5.17 |
| 16 | 0.64 | 21,565 | 21,529 | 21,422 | 286 | 614 | 2.15 | 1.92 | 63.40 | 0.05 | 5.16 |
| 17 | 0.66 | 21,601 | 21,565 | 21,455 | 287 | 622 | 2.16 | 1.92 | 63.29 | 0.05 | 5.15 |
| 18 | 0.68 | 21,656 | 21,619 | 21,505 | 290 | 639 | 2.21 | 1.91 | 63.18 | 0.05 | 5.13 |
| 19 | 0.70 | 21,702 | 21,664 | 21,545 | 292 | 655 | 2.24 | 1.90 | 63.14 | 0.05 | 5.11 |
| 20 | 0.72 | 21,891 | 21,850 | 21,711 | 303 | 725 | 2.39 | 1.87 | 62.13 | 0.06 | 5.02 |
| 21 | 0.74 | 21,997 | 21,953 | 21,798 | 310 | 776 | 2.50 | 1.84 | 61.34 | 0.06 | 4.94 |
| 22 | 0.76 | 22,011 | 21,967 | 21,809 | 312 | 780 | 2.50 | 1.84 | 61.09 | 0.07 | 4.92 |

*table continues…*

| Final Pit | Revenue Factor | Open Pit Cash Flow $ Discounted ('000) | | | Resource Tonnes ('000) | Waste Tonnes ('000) | Strip Ratio | Grade | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Best | Worst | Specified | | | | Pb (%) | Ag (g/t) | Cu (%) | Zn (%) |
| 23 | 0.78 | 22,019 | 21,975 | 21,815 | 313 | 783 | 2.50 | 1.83 | 60.95 | 0.07 | 4.91 |
| 24 | 0.80 | 22,023 | 21,978 | 21,816 | 313 | 784 | 2.50 | 1.83 | 60.83 | 0.07 | 4.91 |
| 25 | 0.82 | 22,029 | 21,984 | 21,819 | 314 | 790 | 2.51 | 1.83 | 60.74 | 0.07 | 4.90 |
| 26 | 0.84 | 22,034 | 21,989 | 21,819 | 315 | 792 | 2.51 | 1.83 | 60.55 | 0.07 | 4.89 |
| 27 | 0.86 | 22,043 | 21,998 | 21,825 | 316 | 808 | 2.55 | 1.82 | 60.60 | 0.07 | 4.88 |
| 28 | 0.88 | 22,059 | 22,013 | 21,833 | 320 | 843 | 2.63 | 1.81 | 60.19 | 0.07 | 4.84 |
| 29 | 0.90 | 22,070 | 22,022 | 21,835 | 323 | 876 | 2.71 | 1.79 | 59.86 | 0.07 | 4.81 |
| **30** | **0.92** | **22,071** | **22,023** | **21,833** | **325** | **882** | **2.72** | **1.79** | **59.72** | **0.07** | **4.80** |
| 31 | 0.94 | 22,071 | 22,022 | 21,832 | 325 | 883 | 2.72 | 1.79 | 59.64 | 0.07 | 4.79 |
| 32 | 0.96 | 22,070 | 22,022 | 21,830 | 325 | 883 | 2.72 | 1.79 | 59.59 | 0.07 | 4.79 |
| 33 | 0.98 | 22,070 | 22,022 | 21,830 | 325 | 886 | 2.72 | 1.79 | 59.57 | 0.07 | 4.79 |
| 34 | 1.00 | 22,069 | 22,021 | 21,826 | 326 | 904 | 2.77 | 1.79 | 59.53 | 0.08 | 4.78 |
| 35 | 1.02 | 22,067 | 22,019 | 21,824 | 327 | 909 | 2.78 | 1.78 | 59.48 | 0.08 | 4.78 |
| 36 | 1.04 | 22,067 | 22,018 | 21,823 | 327 | 910 | 2.79 | 1.78 | 59.45 | 0.08 | 4.78 |
| 37 | 1.06 | 22,054 | 22,004 | 21,802 | 329 | 940 | 2.86 | 1.78 | 59.32 | 0.08 | 4.76 |
| 38 | 1.08 | 22,046 | 21,996 | 21,790 | 330 | 947 | 2.87 | 1.77 | 59.08 | 0.08 | 4.75 |
| 39 | 1.10 | 22,043 | 21,993 | 21,785 | 331 | 952 | 2.88 | 1.77 | 59.03 | 0.08 | 4.74 |
| 40 | 1.12 | 22,027 | 21,976 | 21,765 | 333 | 971 | 2.92 | 1.76 | 58.78 | 0.08 | 4.72 |
| 41 | 1.16 | 22,012 | 21,961 | 21,746 | 335 | 984 | 2.94 | 1.75 | 58.48 | 0.08 | 4.69 |
| 42 | 1.18 | 22,004 | 21,952 | 21,735 | 335 | 991 | 2.95 | 1.75 | 58.36 | 0.08 | 4.69 |
| 43 | 1.20 | 22,001 | 21,950 | 21,733 | 336 | 992 | 2.96 | 1.75 | 58.33 | 0.08 | 4.68 |

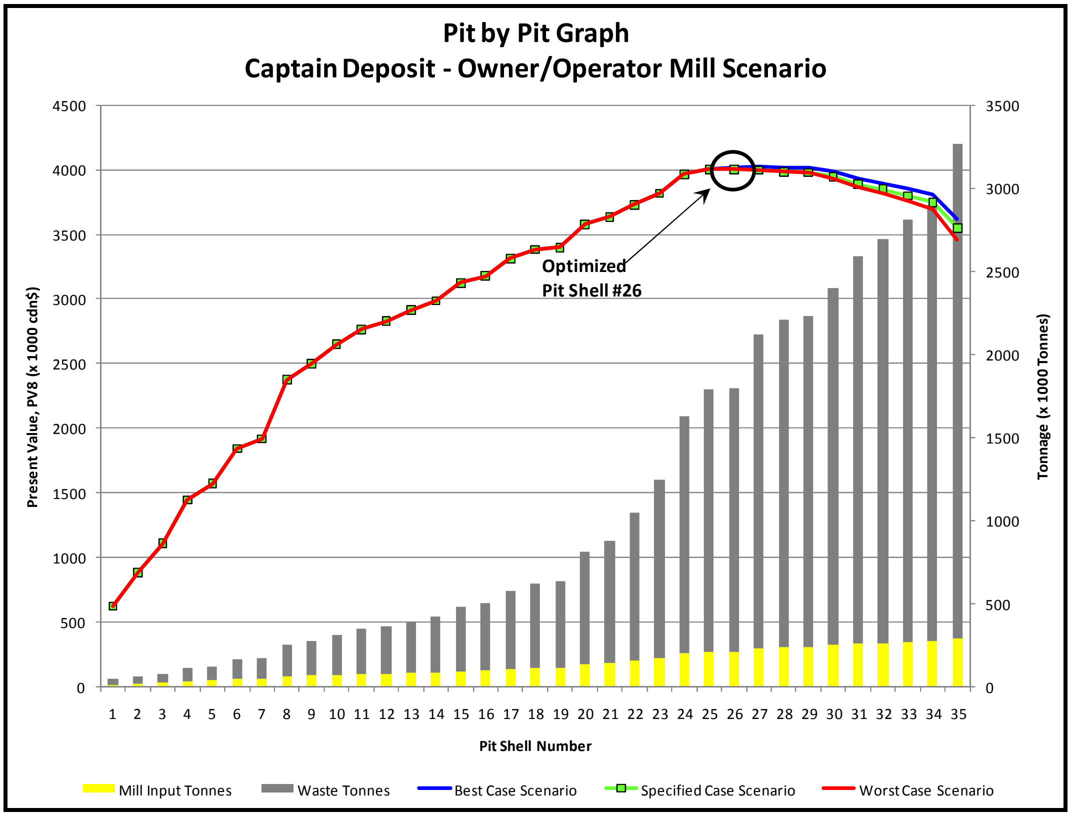
The following features are identified:

- Pit shell #30 generates the highest PV at Cdn$22.0 million for the specified case, based on a 500 t/d operation over a mine life of approximately 1.8 years. The selected base case pit shell contains 325,000 t of resource with average grades of:
  - 4.80% Zn
  - 1.79% Pb
  - 0.07% Cu
  - 59.72 g/t Ag
- For pit shells larger than pit #30, although the resource is progressively increased, PV is gradually decreased. The decrease in cash flow results from resource grades gradually decreasing, stripping ratio gradually increasing, and the time value of money (discount rate).
- Pit #30 is an optimized pit shell or base case pit shell.
- The pit optimization study includes inferred geological resources that are considered too speculative geologically to be considered reserves and there is no certainty that the present values as determined by the Whittle™ analysis will be realized.

### CAPTAIN

A series of nested pit shells were generated by varying the revenue factor. Figure 16.6, Figure 16.7, Table 16.6 and Table 16.7 demonstrate the relationship between the resource contained within the pit shell and the PV for each of the nested pit shells for the two optimization scenarios. The specified case represents the most practical operation scenario.

**Figure 16.6 Captain Nested Pits and Present Values – Optimization 1**




Note: Replicated from chart generated in the Whittle™ 4.4 commercial software

**Table 16.6 Captain Optimization Results of Nested Pits - Optimization 1**

| Final Pit | Revenue Factor | Open Pit Cash Flow $ Discounted ('000) | | | Resource Tonnes ('000) | Waste Tonnes ('000) | Strip Ratio | Grade | |
|---|---|---|---|---|---|---|---|---|---|
| | | Best | Worst | Specified | | | | Cu (%) | Co (g/t) |
| 1 | 0.40 | 621 | 621 | 621 | 11 | 36 | 3.31 | 1.65 | 0.03 |
| 2 | 0.42 | 882 | 882 | 882 | 16 | 44 | 2.75 | 1.57 | 0.03 |
| 3 | 0.44 | 1,106 | 1,106 | 1,106 | 21 | 56 | 2.65 | 1.53 | 0.03 |
| 4 | 0.46 | 1,444 | 1,444 | 1,444 | 30 | 78 | 2.59 | 1.45 | 0.03 |
| 5 | 0.48 | 1,570 | 1,570 | 1,570 | 34 | 87 | 2.58 | 1.43 | 0.03 |
| 6 | 0.50 | 1,840 | 1,840 | 1,840 | 41 | 118 | 2.90 | 1.42 | 0.03 |
| 7 | 0.52 | 1,915 | 1,915 | 1,915 | 43 | 124 | 2.86 | 1.40 | 0.03 |
| 8 | 0.54 | 2,372 | 2,372 | 2,372 | 58 | 192 | 3.34 | 1.36 | 0.03 |
| 9 | 0.56 | 2,501 | 2,501 | 2,501 | 63 | 211 | 3.38 | 1.34 | 0.03 |
| 10 | 0.58 | 2,647 | 2,647 | 2,647 | 68 | 243 | 3.58 | 1.33 | 0.03 |
| 11 | 0.60 | 2,766 | 2,766 | 2,766 | 72 | 271 | 3.75 | 1.32 | 0.03 |
| 12 | 0.62 | 2,826 | 2,826 | 2,826 | 75 | 286 | 3.80 | 1.31 | 0.03 |
| 13 | 0.64 | 2,914 | 2,914 | 2,914 | 80 | 312 | 3.91 | 1.30 | 0.03 |
| 14 | 0.66 | 2,986 | 2,986 | 2,986 | 83 | 335 | 4.02 | 1.29 | 0.03 |
| 15 | 0.68 | 3,123 | 3,123 | 3,123 | 92 | 386 | 4.21 | 1.26 | 0.03 |
| 16 | 0.70 | 3,179 | 3,179 | 3,179 | 96 | 407 | 4.25 | 1.25 | 0.03 |
| 17 | 0.72 | 3,312 | 3,312 | 3,312 | 106 | 471 | 4.46 | 1.22 | 0.03 |
| 18 | 0.76 | 3,382 | 3,382 | 3,382 | 111 | 508 | 4.58 | 1.21 | 0.03 |
| 19 | 0.78 | 3,399 | 3,399 | 3,399 | 113 | 521 | 4.63 | 1.21 | 0.03 |
| 20 | 0.80 | 3,579 | 3,579 | 3,579 | 130 | 677 | 5.20 | 1.18 | 0.03 |
| 21 | 0.84 | 3,635 | 3,635 | 3,635 | 137 | 736 | 5.38 | 1.17 | 0.03 |
| 22 | 0.86 | 3,731 | 3,731 | 3,731 | 152 | 893 | 5.87 | 1.15 | 0.03 |
| 23 | 0.88 | 3,818 | 3,818 | 3,818 | 169 | 1,076 | 6.37 | 1.14 | 0.04 |
| 24 | 0.92 | 3,967 | 3,966 | 3,966 | 196 | 1,428 | 7.28 | 1.13 | 0.04 |
| 25 | 0.94 | 4,011 | 4,004 | 4,004 | 207 | 1,579 | 7.63 | 1.13 | 0.04 |

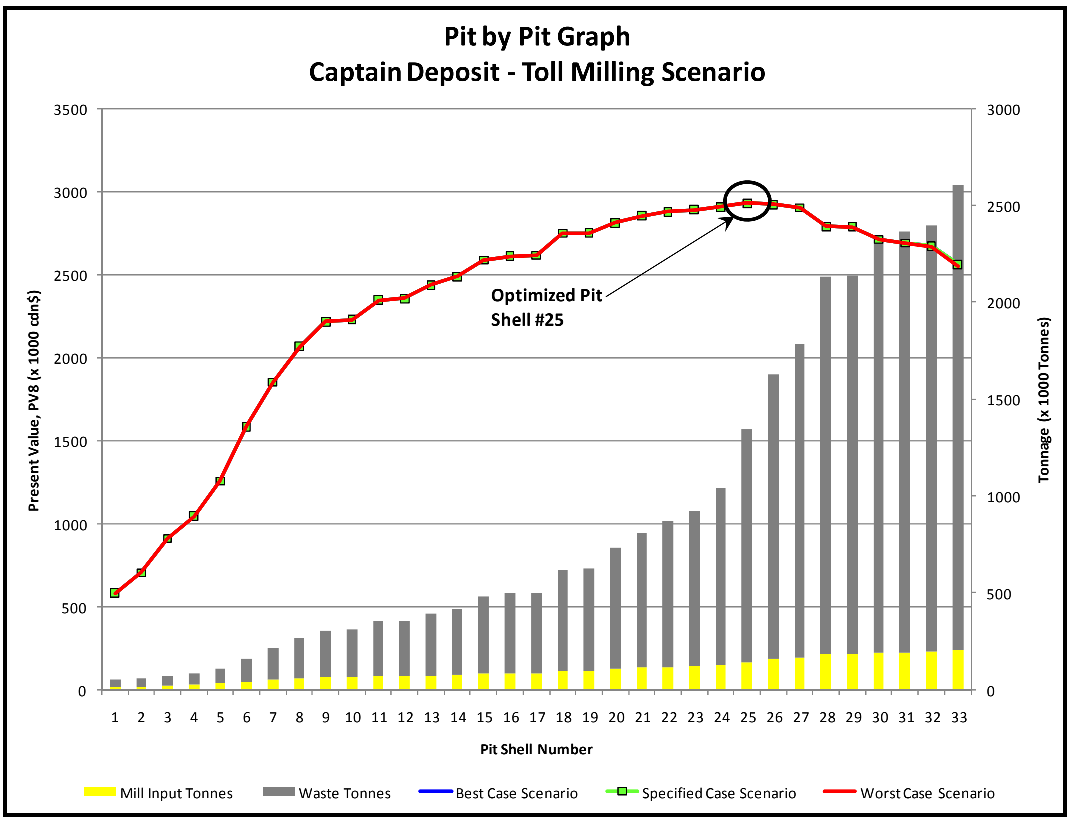
*table continues…*

| Final Pit | Revenue Factor | Open Pit Cash Flow $ Discounted ('000) | | | Resource Tonnes ('000) | Waste Tonnes ('000) | Strip Ratio | Grade | |
|---|---|---|---|---|---|---|---|---|---|
| | | Best | Worst | Specified | | | | Cu (%) | Co (g/t) |
| **26** | **0.98** | **4,012** | **4,005** | **4,005** | **208** | **1,588** | **7.64** | **1.13** | **0.04** |
| 27 | 1.00 | 4,024 | 3,998 | 3,998 | 229 | 1,887 | 8.23 | 1.12 | 0.03 |
| 28 | 1.02 | 4,018 | 3,984 | 3,984 | 235 | 1,971 | 8.39 | 1.12 | 0.03 |
| 29 | 1.04 | 4,015 | 3,980 | 3,979 | 237 | 1,991 | 8.41 | 1.12 | 0.03 |
| 30 | 1.08 | 3,987 | 3,946 | 3,936 | 247 | 2,154 | 8.73 | 1.11 | 0.03 |
| 31 | 1.10 | 3,933 | 3,888 | 3,865 | 256 | 2,334 | 9.13 | 1.11 | 0.03 |
| 32 | 1.12 | 3,898 | 3,849 | 3,817 | 261 | 2,432 | 9.30 | 1.11 | 0.03 |
| 33 | 1.16 | 3,852 | 3,800 | 3,757 | 267 | 2,547 | 9.54 | 1.11 | 0.03 |
| 34 | 1.18 | 3,804 | 3,749 | 3,697 | 273 | 2,624 | 9.60 | 1.10 | 0.03 |
| 35 | 1.20 | 3,615 | 3,550 | 3,460 | 290 | 2,975 | 10.25 | 1.10 | 0.03 |

The following features are identified:

- Pit shell #26 generates the highest PV at Cdn$4.0 million for the specified case, based on a 500 t/d operation over a mine life of approximately 1.2 years. The selected base case pit shell contains 208,000 t of resource with average grades of:
  - 1.13% Cu
- For pit shells larger than pit #26 although the resource is progressively increased, PV is gradually decreased. The decrease in cash flow results from resource grades gradually decreasing, stripping ratio gradually increasing, and the time value of money (discount rate).
- Pit #26 is an optimized pit shell or base case pit shell.
- The pit optimization study includes inferred geological resources that are considered too speculative geologically to be considered reserves and there is no certainty that the present values as determined by the Whittle™ analysis will be realized.

**Figure 16.7 Captain Nested Pits and Present Values - Optimization 2**




Note: Replicated from chart generated in the Whittle™ 4.4 commercial software

### Table 16.7 Captain Optimization Results of Nested Pits - Optimization 2

| Final Pit | Revenue Factor | Open Pit Cash Flow $ Discounted ('000) | | | Resource Tonnes ('000) | Waste Tonnes ('000) | Strip Ratio | Grade | |
|---|---|---|---|---|---|---|---|---|---|
| | | Best | Worst | Specified | | | | Cu% | Co (g/t) |
| 1 | 0.46 | 580 | 580 | 580 | 11 | 38 | 3.31 | 1.65 | 0.03 |
| 2 | 0.48 | 702 | 702 | 702 | 14 | 42 | 2.99 | 1.62 | 0.03 |
| 3 | 0.50 | 908 | 908 | 908 | 19 | 53 | 2.81 | 1.58 | 0.04 |
| 4 | 0.52 | 1,045 | 1,045 | 1,045 | 23 | 61 | 2.69 | 1.54 | 0.03 |
| 5 | 0.54 | 1,254 | 1,254 | 1,254 | 29 | 80 | 2.78 | 1.50 | 0.03 |
| 6 | 0.58 | 1,584 | 1,584 | 1,584 | 38 | 120 | 3.12 | 1.48 | 0.03 |
| 7 | 0.60 | 1,852 | 1,852 | 1,852 | 48 | 164 | 3.43 | 1.44 | 0.03 |
| 8 | 0.62 | 2,067 | 2,067 | 2,067 | 55 | 208 | 3.78 | 1.43 | 0.03 |
| 9 | 0.64 | 2,216 | 2,216 | 2,216 | 61 | 245 | 4.02 | 1.43 | 0.03 |
| 10 | 0.66 | 2,228 | 2,228 | 2,228 | 61 | 248 | 4.04 | 1.43 | 0.03 |
| 11 | 0.68 | 2,346 | 2,346 | 2,346 | 67 | 286 | 4.30 | 1.42 | 0.03 |
| 12 | 0.70 | 2,355 | 2,355 | 2,355 | 67 | 287 | 4.27 | 1.41 | 0.03 |
| 13 | 0.72 | 2,437 | 2,437 | 2,437 | 72 | 319 | 4.44 | 1.40 | 0.03 |
| 14 | 0.74 | 2,489 | 2,489 | 2,489 | 75 | 343 | 4.58 | 1.39 | 0.03 |
| 15 | 0.76 | 2,586 | 2,586 | 2,586 | 81 | 396 | 4.87 | 1.37 | 0.03 |
| 16 | 0.78 | 2,611 | 2,611 | 2,611 | 83 | 413 | 4.96 | 1.37 | 0.03 |
| 17 | 0.80 | 2,617 | 2,617 | 2,617 | 84 | 417 | 4.99 | 1.37 | 0.03 |
| 18 | 0.84 | 2,748 | 2,748 | 2,748 | 96 | 523 | 5.45 | 1.34 | 0.03 |
| 19 | 0.86 | 2,751 | 2,751 | 2,751 | 96 | 527 | 5.48 | 1.34 | 0.03 |
| 20 | 0.88 | 2,813 | 2,813 | 2,813 | 105 | 624 | 5.95 | 1.33 | 0.04 |
| 21 | 0.90 | 2,853 | 2,853 | 2,853 | 111 | 697 | 6.30 | 1.32 | 0.03 |
| 22 | 0.92 | 2,879 | 2,879 | 2,879 | 115 | 756 | 6.56 | 1.32 | 0.04 |
| 23 | 0.94 | 2,890 | 2,890 | 2,890 | 118 | 803 | 6.81 | 1.32 | 0.04 |
| 24 | 0.96 | 2,908 | 2,908 | 2,908 | 126 | 913 | 7.28 | 1.31 | 0.04 |
| **25** | **0.98** | **2,931** | **2,931** | **2,931** | **142** | **1,201** | **8.45** | **1.32** | **0.04** |

*table continues…*

| Final Pit | Revenue Factor | Open Pit Cash Flow $ Discounted ('000) | | | Resource Tonnes ('000) | Waste Tonnes ('000) | Strip Ratio | Grade | |
|---|---|---|---|---|---|---|---|---|---|
| | | Best | Worst | Specified | | | | Cu% | Co (g/t) |
| 26 | 1.00 | 2,922 | 2,922 | 2,922 | 158 | 1,467 | 9.31 | 1.31 | 0.04 |
| 27 | 1.08 | 2,904 | 2,904 | 2,904 | 165 | 1,621 | 9.82 | 1.32 | 0.04 |
| 28 | 1.10 | 2,791 | 2,791 | 2,791 | 182 | 1,947 | 10.71 | 1.31 | 0.04 |
| 29 | 1.12 | 2,788 | 2,788 | 2,788 | 182 | 1,955 | 10.73 | 1.31 | 0.04 |
| 30 | 1.14 | 2,712 | 2,712 | 2,709 | 190 | 2,132 | 11.22 | 1.31 | 0.04 |
| 31 | 1.16 | 2,691 | 2,691 | 2,687 | 192 | 2,174 | 11.33 | 1.31 | 0.04 |
| 32 | 1.18 | 2,673 | 2,673 | 2,669 | 194 | 2,204 | 11.38 | 1.31 | 0.04 |
| 33 | 1.20 | 2,563 | 2,563 | 2,551 | 202 | 2,402 | 11.92 | 1.31 | 0.04 |

The following features are identified:

- Pit shell #25 generates the highest PV at Cdn$2.9 million for the specified case, based on a 500 t/d operation over a mine life of approximately 0.8 years. The selected base case pit shell contains 142,000 t of resource with average grades of:
  - 1.32% Cu
- For pit shells larger than pit #25, although the resource is progressively increased, PV is gradually decreased. The decrease in cash flow results from resource grades gradually decreasing, stripping ratio gradually increasing, and the time value of money (discount rate).
- Pit #25 is an optimized pit shell or base case pit shell.
- The pit optimization study includes inferred geological resources that are considered too speculative geologically to be considered reserves and there is no certainty that the present values as determined by the Whittle™ analysis will be realized.

### 16.3.4 Ultimate Pit Design Parameters

Based on the optimization results and current geological models, Tetra Tech recommends that the detailed pit design should be based on Pit shell #31 for CNE and Pit shell #26 for Captain, from Optimization 1 scenario, to maximize PV.

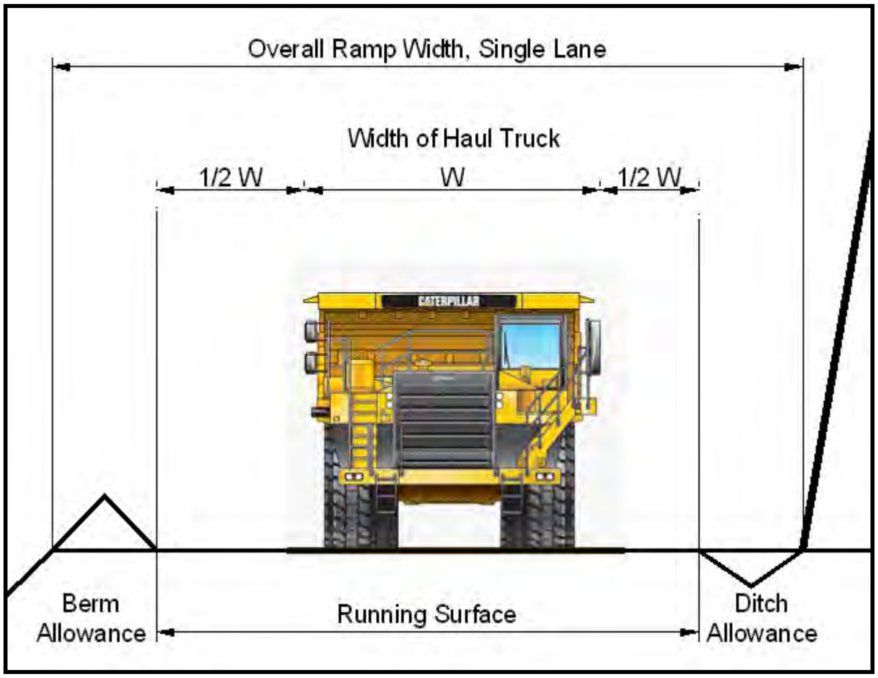
Pit design criteria for Captain and CNE includes:

| | |
|---|---|
| Bench Height | 8 m |
| Face Angle | 80° |
| Benching | double |
| Safety Berm Width | 12 m |
| Overall Pit Slope (without inclusion of ramp) | 45° |
| Ramp Width: | |
| Single Lane | 10 m |
| Double Lane | 15 m |

#### Road Width

In-pit ramps are designed with an overall ramp width of 15 m for double lane traffic with a maximum gradient of 10%. A 2.6 m wide and 0.87 m high safety berm and an internal 1.5 m wide water ditch will be provided for two lane traffic to accommodate 36 tonne haul trucks, as shown in Figure 16.8 and Table 16.8.

**Figure 16.8 Ramp Width Design – Concept**



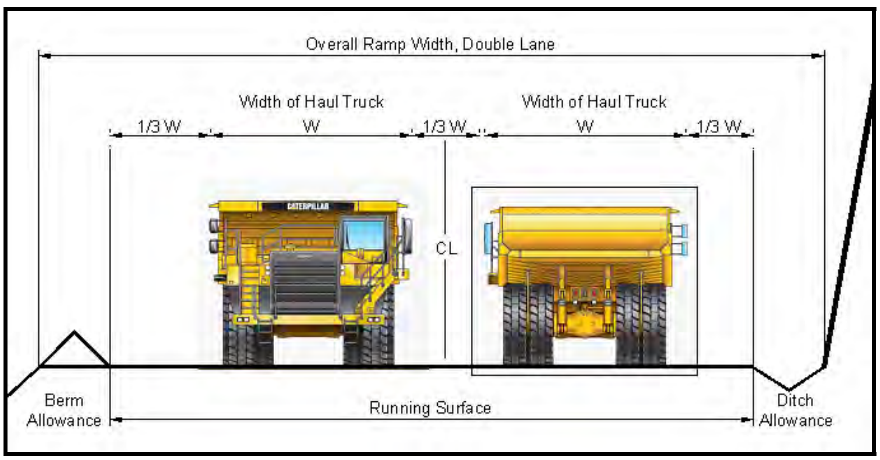



**Table 16.8 Pit Design Ramp Width Calculation**

| Truck Parameters (Double Lane) | |
|---|---|
| Operating Width | 3.35 m |
| Double Lane (3) x Operating Width | X 3 |
| Road Width | 10.0 m |
| **Berm** | |
| Tire Overall Diameter | 1.75 m |
| Height (1/2 of largest tire) | 0.875 m |
| Slope (H:V) | 1.5 :1 |
| Berm Width | 2.6 m |
| **Ditch** | |
| Depth | 0.5 m |
| Slope (H:V) | 1.5:1 |
| Ditch Width | 1.5 m |
| Total Road Width | 14.2 m |

| Truck Parameters (Single Lane) | |
|---|---|
| Operating Width | 3.35 m |
| Single Lane (2.00) x Operating Width | X 2 |
| Road Width | 6.7 m |
| **Berm** | |
| Tire Overall Diameter | 1.75 m |
| Height (1/2 of largest tire) | 0.875 m |
| Slope (H:V) | 1.5 :1 |
| Berm Width | 2.6 m |
| **Ditch** | |
| Depth | 0.5 m |
| Slope (H:V) | 1.5 : 1 |
| Ditch Width | 1.5 m |
| Total Road Width | 10.8 m |

PUSHBACK WIDTH

Due to the short mine life of the CNE deposit, no production phasing was required, thus no pushback width was calculated for phase development.

### 16.3.5 ULTIMATE PIT DESIGN

Based on the above parameters, the ultimate pit designs for the CNE and Captain properties results in the following:

**Table 16.9 General Pit Statistics**

| Item | Size | |
|---|---|---|
| | CNE | Captain |
| Pit Top Elevation | approx. 171 m | approx. 152 m |
| Pit Bottom Elevation | 100 m | 68 m |
| Pit Depth | 71 m | 84 m |
| Volume of Pit | 490,500 $m^3$ | 834,900 $m^3$ |
| Area of Pit Top | 30,480 $m^2$ | 36,110 $m^2$ |
| Perimeter at the Top of the Pit | 850 m | 770 m |
| Length from East to West | 220 m | 215 m |
| Length from North to South | 175 m | 235 m |

**Figure 16.9 Ultimate Pit Design - CNE**

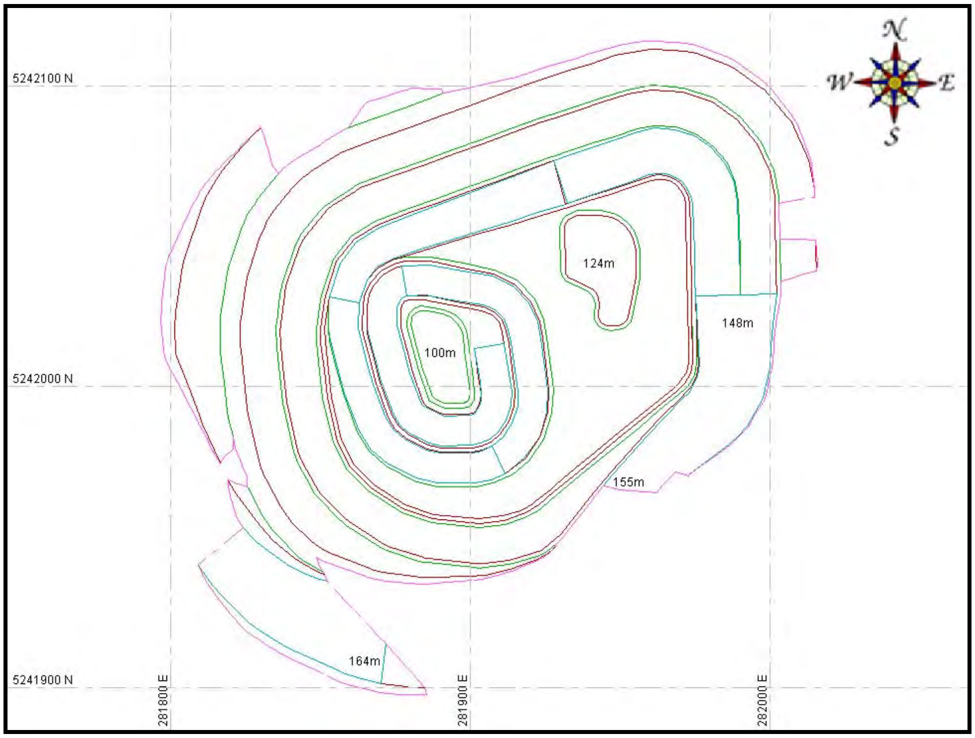



**Figure 16.10 3D Rendered Image of CNE Pit Design and Resource Body**

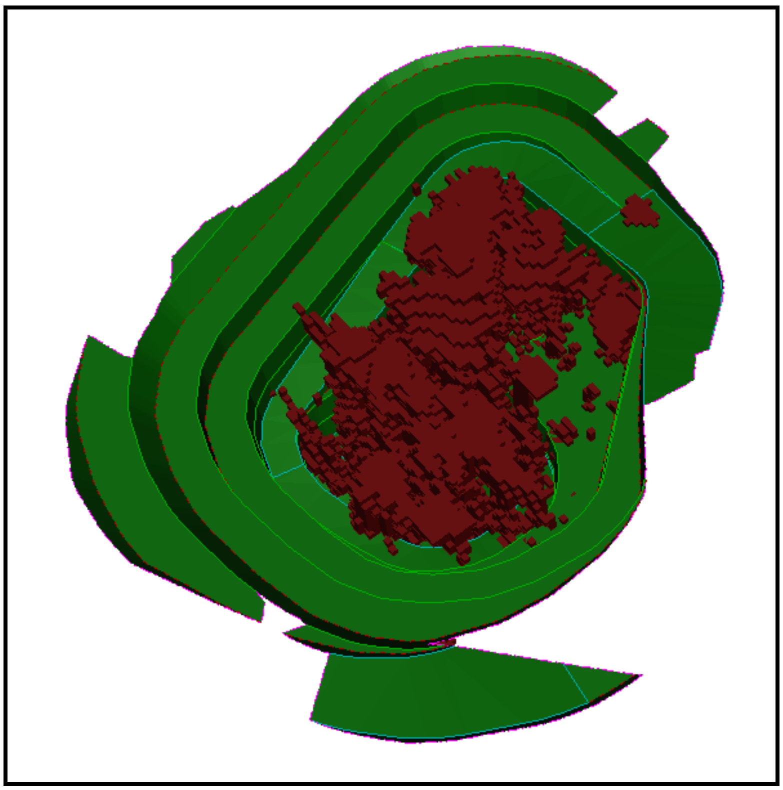


**Figure 16.11 Ultimate Pit Design - Captain**

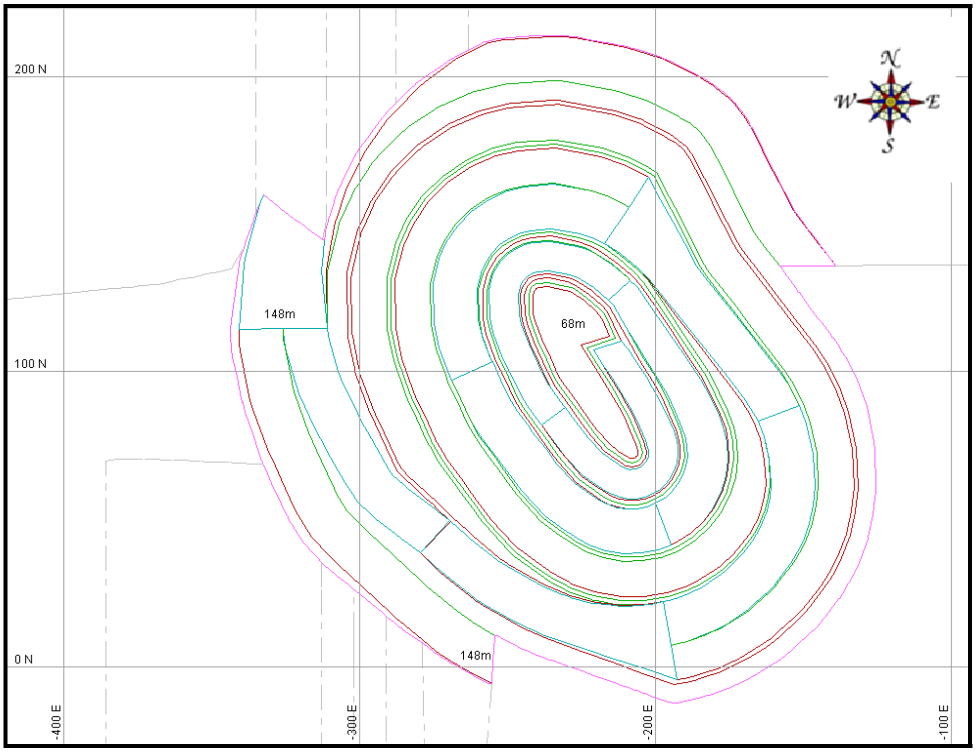
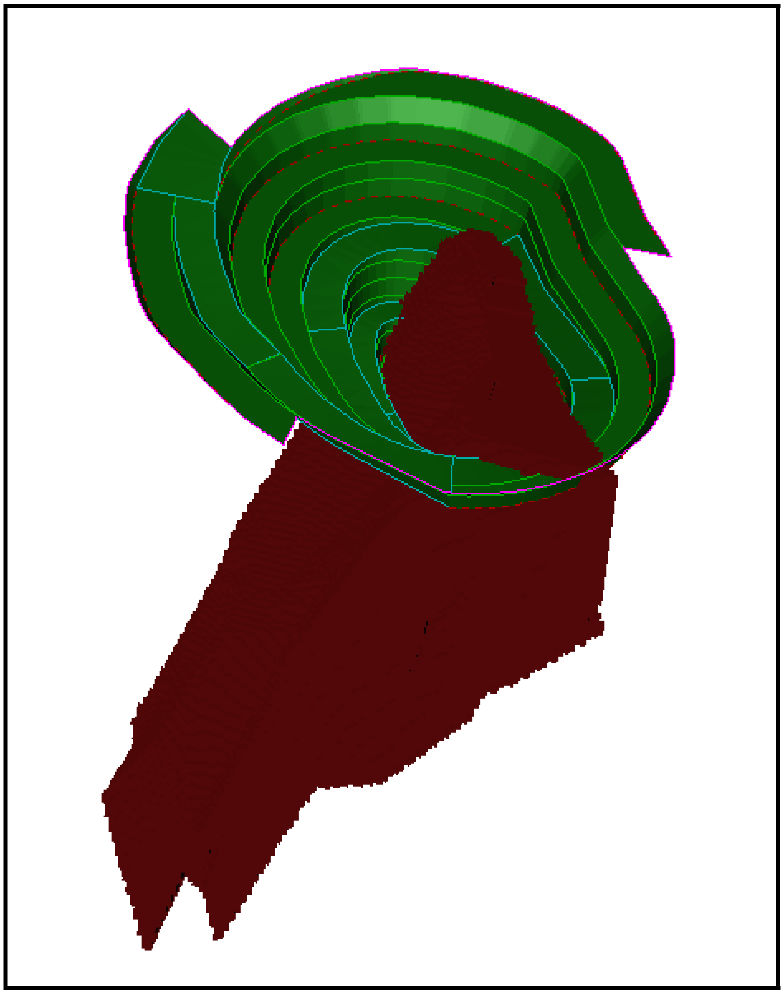


**Figure 16.12 3D Rendered View of Captain Pit Design and Resource Body**



### 16.3.6 RESOURCE CONTAINED WITHIN PIT DESIGN

#### CNE

The ultimate pit design results for CNE are presented in Table 16.10.

**Table 16.10 Ultimate Pit Design Results - CNE**

| Item | Tonnes | Zn (%) | Pb (%) | Cu (%) | Ag (g/t) |
|---|---|---|---|---|---|
| Measured Resource | 39,815 | 5.45 | 1.80 | 0.02 | 58.48 |
| Indicated Resource | 273,542 | 4.67 | 1.76 | 0.09 | 59.54 |
| **Total M+I Resource** | **313,357** | **4.77** | **1.76** | **0.08** | **59.40** |
| Inferred Resource | 11,664 | 3.79 | 1.77 | 0.05 | 22.71 |

*table continues…*

| Item | Tonnes | Zn (%) | Pb (%) | Cu (%) | Ag (g/t) |
|---|---|---|---|---|---|
| **Total Resource** | **325,021** | **4.74** | **1.76** | **0.08** | **58.09** |
| Waste Rock | 958,684 | - | - | - | - |
| Stripping Ratio | 2.95 | - | - | - | - |

Note: At cut-off grades of 0.6% Zn (in the Pb-Zn-Cu Zone), 0.35% Cu (in the Cu-Zone), and 1.0% Zn (in the Pb-Zn Zone).

A mining resource recovery of 95% with an overall waste rock dilution of 5% was assumed.

### CAPTAIN

As presented in Table 16.11, the ultimate pit design for Captain contains:

**Table 16.11 Ultimate Pit Design Results - Captain**

| Item | Tonnes | Cu (%) | Co (%) | Au (g/t) | CuEQ (%) |
|---|---|---|---|---|---|
| Measured Resource | 23,786 | 1.61 | 0.05 | 0.25 | 2.06 |
| Indicated Resource | 140,869 | 1.33 | 0.04 | 0.26 | 1.69 |
| **Total M+I Resource** | **164,656** | **1.37** | **0.04** | **0.26** | **1.75** |
| Inferred Resource | 75,300 | 0.54 | 0.02 | 0.13 | 0.75 |
| **Total Resource** | **239,954** | **1.11** | **0.04** | **0.22** | **1.43** |
| Waste Rock | 1,971,245 | - | - | - | - |
| Stripping Ratio | 8.22 | - | - | - | - |

Note: At cut-off grades of 0.35% Cu.

A mining resource recovery of 95% with an overall waste rock dilution of 5% was assumed.

## 16.4 MINE DEVELOPMENT AND PRODUCTION SCHEDULE

The mine development was designed to meet the following objectives:

- enable the mining of high grade resource as early as possible
- effectively reduce stripping ratio in the initial mining stage
- balance the stripping ratio over the period of the mine life
- maintain minimum mining width between two working phases.

### 16.4.1 Mine Development

Referring to Figure 16.5 (CNE Optimizations) and Figure 16.7 (Captain Optimizations), there is little variance between the Whittle™ optimization results for best and worst case scenarios. This indicates that there is little influence on the selection of the nested pits for scheduling purposes. Therefore, in the cases of the Captain and CNE properties, one mineable phase has been identified to develop ultimate pit.

### 16.4.2 Production Schedule

A mill throughput of 500 t/d allows for an annual production of 175,000 t based on 350 days per year. Tetra Tech developed the production schedule having a mine life of approximately 3.2 years in this study.

Initially, it was scheduled to mine the CNE and Captain properties consecutively. Table 16.12 outlines the complete mining schedule period by period showing waste and resource tonnage mined by resource deposit.

It was determined during the financial analysis that Captain was not a positive contributor to the overall cash flow (refer to Section 22 for further details). Therefore, the overall production schedule includes only CNE property. A toll mill option was selected for the stand alone CNE mine operation. The mine rate was increased to 1,000 t/d and a new production schedule of less than 1 year was developed for the CNE mine operation. Table 16.13 reflects the production schedule carried into the Financial Analysis.

**Table 16.12 Total Material Mined by Property by Year – CNE and Captain**

| Material | Units | Year | | | | | Total |
|---|---|---|---|---|---|---|---|
| | | 1 | 2 | 3 | 4 | 5 | |
| **Overburden** | | | | | | | |
| CNE | tonnes | 325,009 | 281 | 0 | 0 | 0 | 325,290 |
| Captain | tonnes | 0 | 404,099 | 299 | 0 | 0 | 404,399 |
| **Total** | **tonnes** | **325,009** | **404,381** | **299** | **0** | **0** | **729,689** |
| **Waste** | | | | | | | |
| CNE | tonnes | 488,720 | 144,673 | 0 | 0 | 0 | 633,393 |
| Captain | tonnes | 0 | 669,107 | 878,113 | 19,626 | 0 | 1,566,847 |
| **Total** | **tonnes** | **488,720** | **813,780** | **878,113** | **19,626** | **0** | **2,200,240** |
| **Resource** | | | | | | | |
| CNE | tonnes | 175,000 | 150,021 | 0 | 0 | 0 | 325,021 |
| Captain | tonnes | 0 | 24,979 | 175,000 | 39,975 | 0 | 239,954 |
| **Total Resource** | **tonnes** | **175,000** | **175,000** | **175,000** | **39,975** | **0** | **564,975** |
| **Stripping Ratio** | | | | | | | |
| CNE | | 4.65 | 0.97 | 0.00 | 0.00 | 0.00 | 2.95 |

*table continues…*

| Material | Units | Year | | | | | Total |
|---|---|---|---|---|---|---|---|
| | | 1 | 2 | 3 | 4 | 5 | |
| Captain | - | 0.00 | 42.96 | 5.02 | 0.49 | 0.00 | 8.22 |
| **Overall** | **-** | **4.65** | **6.96** | **5.02** | **0.49** | **0.00** | **5.19** |

**Table 16.13 Overall Ultimate Pit Production Schedule - CNE**

| Material | Units | Quarter | | | | Total |
|---|---|---|---|---|---|---|
| | | 1 | 2 | 3 | 4 | |
| Overburden | tonnes | 243,829 | 78,928 | 2,533 | | 325,290 |
| Waste | tonnes | 75,610 | 171,056 | 243,788 | 142,939 | 633,393 |
| Total Resource | tonnes | 32,987 | 102,441 | 106,105 | 83,488 | 325,021 |
| Lead | % | 2.25 | 2.06 | 1.73 | 1.24 | 1.76 |
| Silver | g/t | 60.12 | 60.31 | 55.32 | 58.07 | 58.09 |
| Copper | % | 0.06 | 0.07 | 0.06 | 0.14 | 0.08 |
| Zinc | % | 5.60 | 5.30 | 4.75 | 3.68 | 4.74 |
| Stripping Ratio | - | 9.68 | 2.44 | 2.32 | 1.71 | 2.95 |

### 16.4.3 *Pit Water Handling*

The progressive development of the open pit will result in increasing water infiltration from precipitation and groundwater inflows. As the pit deepens and increases in footprint, it will be necessary to control water inflow through the construction of in-pit dewatering systems such as drainage ditches, sumps, pipelines and pumps.

## 16.5 Mine Equipment Selection

### 16.5.1 *Major Equipment Selection*

The mining operations will be performed by a mining contractor who would be supplying the necessary equipment. However, for a project this size (1,000 t/d and 3:1 strip ratio), a typical mining equipment fleet would include the following equipment:

**Table 16.14 Proposed Mine Equipment Fleet - Typical**

| Equipment Fleet | Size | Units |
|---|---|---|
| Haul Trucks | 36.0 tonne | 5 |
| Shovels | 2.3 $m^3$ | 1 |
| Loader | 3.0 $m^3$ | 1 |

*table continues…*

| Equipment Fleet | Size | Units |
|---|---|---|
| Drills | 6.35 cm | 3 |
| Track Dozers | 60 kW | 3 |
| Graders | 115 kW | 1 |
| Water Truck | 50 ton | 1 |
| Backhoe Excavator | 1.5 $m^3$ | 1 |
| Service Truck | 1,800 kg gvw | 1 |
| Light Vehicles (Pick-up Trucks) | 0.5 ton crew cab | 3 |

### 16.5.2 DRILLING AND BLASTING PARAMETERS

#### FINAL PIT WALL BLASTING

The preservation of rock mass integrity is to allow for the development of the steepest wall slope by applying careful blasting methods. A pre-shear and buffer blasting practice should be implemented adjacent to the final pit walls to minimize damage to the final pit walls due to blasting.

# 17.0 RECOVERY METHODS

Information regarding the recoverability of the valuable component minerals and a description or flow sheet of any current or proposed process plant is included in Section 13 Mineral Processing and Metallurgical Testing.

# 18.0 PROJECT INFRASTRUCTURE

It is proposed to execute the CNE mine plan utilizing a contract mining operation for a limited one year duration based on a 1,000 t/d production rate. It is assumed that the mining contractor will be self-sufficient for office trailers, work force trailers, and equipment supply and maintenance. As well, it is proposed that the mined resource will be transported to a toll mill destination for storage. On site ore storage will be limited to approximately 5,000 t.

Site development infrastructure will include early site clearing and grubbing as well as excavation, fill and granular surface preparation for an area of approximately 3.5 ha. Perimeter fencing will total 1.8 km.

Site roads will be constructed of fine and coarse rock material and will include 15 m wide haul roads complete with a 1.5 m wide water ditch. The haul roads will be provided for two lane traffic to accommodate 36 t haul trucks. Access roads for small trucks and miscellaneous operational traffic will be 8 m in width. Hard areas will be constructed for a mobile crusher and ore stockpile area.

Waste rock dumps will be required on site and adequate ditching to collect water run-off will be required. On-site water systems will include mine dewatering. Waste rock seepage, surface contaminated water and mine dewater will be directed to collection ponds. All collected water will be treated via a waste water treatment plant located on site prior to release to the environment.

On site power will be supplied by diesel generators. No grid power from the province of New Brunswick is required for the CNE mine operations. Electrical distribution on site will be completed by overhead wires with pole mounted transformers as required.

# 19.0 MARKET STUDIES AND CONTRACTS

This Section is not applicable to this report.

# 20.0 ENVIRONMENTAL

## 20.1 INTRODUCTION

This section of the PEA identifies and examines environmental considerations associated with the project, including:

- environmental setting
- environmental assessment and permitting process
- community and Aboriginal engagement.

## 20.2 ENVIRONMENTAL SETTING

The CNE property is located near the boundary between Gloucester County and Northumberland County. Bathurst is the closest city in Gloucester County and Miramichi is the closest city in Northumberland County. The property is located approximately 14 km east of the former Heath Steele Mine site and approximately 19 km south of the operating Brunswick 12 Mine site. Site access is via the Newcastle-Heath Steele road (Route 430) and 17 km of gravel logging road (Spurline Road).

The property is within the Acadian Forest Region and in the Tomogonops Ecodistrict of the Northern Uplands Ecoregion of the Atlantic Maritime Ecozone (NBDNR 2007). The climate is characterized by a cool moist climate with moderate temperatures formed from the Atlantic Ocean (AAFC and EC 1996). The ecodistrict is dominated mostly by conifer stands due to the prevalence of acidic soil (NBDNR 2007). Common species include black spruce, red spruce, and balsam fir. Wetlands in the ecodistrict are dominated by cedar and black spruce. Other species in the ecodistrict that are less common include beech, white pine, yellow birch, sugar maple, and eastern hemlock (AAFC and EC 1996, NBDNR 2007). Sections of the property have been clear-cut in the past with areas of replanting and natural re-growth, the time of the clearing and the areas have not been clarified at this time.

This ecoregion provides habitat for moose, black bear, white-tailed deer, red fox, snowshoe hare, raccoon, striped skunk, eastern chipmunk, porcupine, bobcat, fisher, coyote, wolves, beaver, ruffed grouse, bobcat, marten, and northern flying squirrel. Common bird species include whip-poor-will, blue jay, eastern bluebird and rose-breasted grosbeak (AAFC and EC 1996, NBDNR 2007).

### 20.2.1 WATER RESOURCES

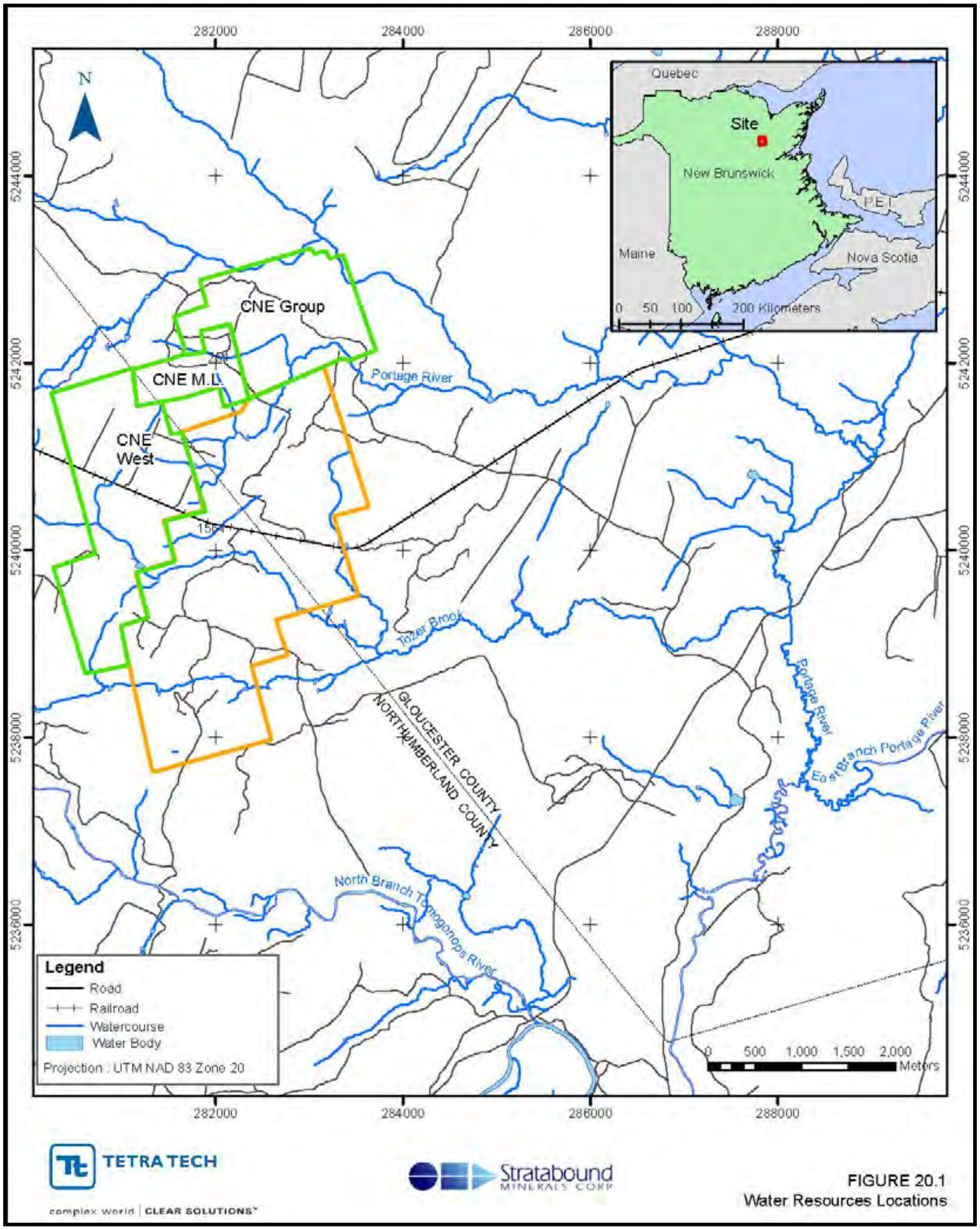
The two tributaries of the Portage River flow northeast across the property to join Mckay Brook and on to the Miramichi River (Figure 20.1). One tributary crosses the northeast side and the other crosses the southeast side of the property. A baseline environmental sampling program was completed in 2007 for the Captain deposit, located south of the property. The program included water quality, sediment quality, fish and fish habitat, and fish tissue sampling from the Portage River, located east of the property, and Tozer Brook, located southeast of the property (Jacques Whitford 2008). Two of the sampling stations were located on Portage River, approximately 150 M and 1,650 m downstream and east of the property. The other sampling stations were located on Tozer Brook south of the property and on the Portage River at a tributary southeast of the property.

The sampling program involved a single sampling period in October 2007. All sites had a neutral pH (6.8-7.1) and parameter concentrations were generally within the Canadian Water Quality Guidelines for the Protection of Aquatic Life (CCME) with a few exceptions. These exceptions included aluminum, cadmium, copper, lead, and iron which occurred at concentrations at or above the CCME guidelines at one or more of the five sampling locations. Sediment quality exceeded applicable CCME Interim Sediment Quality guidelines (ISQG) criteria for arsenic, cadium, lead, mercury and zinc, at all or one of the five locations. Parameters without CCME criteria (e.g., tin, beryllium) were similar across all sampling sites. The higher concentration of parameters in the sediment was attributed to the natural mineralization in the area (Jacques Whitford 2008).

Fish were observed at all sampling locations downstream from the property, including brook trout, Atlantic salmon, common shiner, black nose dace, white sucker and fivespine stickleback (Jacques Whitford 2008). None of these populations are listed by the *New Brunswick Species at Risk Act* or COSEWIC (NBDNR, 2000).

Based on the above findings, there is the potential for fish to occur in both of the streams that traverse the property, and stream water quality is such that it has little or no capacity to assimilate an additional contaminant load. The project is located in the headwaters of an important Atlantic salmon river and project planning will therefore need to accommodate this environmental sensitivity.

**Figure 20.1 Water Resource Locations**




### 20.2.2 GEOCHEMISTRY

The high sulphide content identified in the deposit strongly suggests that waste rock produced from the property could be a potential source of acid mine drainage (AMD) as well as potential metal leaching (ML). Geochemical properties of the ore and waste rock have not been assessed to date. Acid-base accounting to screen for

AMD potential and confirmatory humidity cell tests on representative samples of waste rock are required. All waste rock lithologies need to be screened for AMD potential. Subsequent mine site water management, waste rock management, and closure requirements will be developed on the basis of the geochemical test work.

### 20.2.3 PROJECT DESCRIPTION

A provincial mining lease has been maintained for this property and is still active. The property was originally staked in 1977, with subsequent exploration involving drilling and trenching. The property was remediated to a site resembling that of a greenfield site in the 1990s (C. Neumann, pers. comm.). A detailed description of the site history is described in Section 6.3.

At the current stage of planning, the project is expected to include an open pit mine and associated infrastructure on the property, and toll-milling of the ore at the Xstrata Brunswick 12 mill. The planned mining rate is 1,000 t/d, with slightly less than a one year mine life, and a stripping ratio of approximately 3:1. The total overburden production over the life of the mine is 325,290 Mt and the waste rock production over the LOM is estimated at 633,393 t. The current project schedule indicates contract mining would start late in 2012, with mine site development completed prior to the start of mining operations. No ore processing will be conducted on site and no tailings management area will be required for the project.

Site facilities will include a mine office/dry, truck shop, waste rock disposal area, overburden storage, and an ore storage pad that may include provision for a mobile crusher. The access road will connect to Spurline Road and/or Tomogonops Road. The alignment and design of the access road will be developed later in project planning. If the final route crosses watercourses, then construction of bridges or installation of culverts will be required. Pit water management will be required but pumping quantities and quality and treatment requirements have not been developed at this stage in planning.

#### WATER QUALITY MANAGEMENT

Effluents discharged to surface water from mining activities (e.g., runoff from waste rock) must at minimum meet MMER regulations (Table 20.1; Government of Canada 2002). Discharge from the project site should not cause a parameter increase in a receiving water body, outside the mixing zone, that would cause a receiving water guideline to be exceeded. Site-specific approvals of effluent quality may be assigned after application to the New Brunswick Ministry of the Environment if simple compliance with the MMER regulations will not provide adequate protection of receiving water quality. Effluent discharges also need to be managed to prevent harmful changes to downstream fish habitat. This can most effectively be managed by locating any discharge well upstream of any important fish habitat.

**Table 20.1 Metal Mining Effluent Regulations, SOR/2002-222 – Authorized Limits of Deleterious Substances**

| Item | Deleterious Substances | Maximum Authorized Monthly Mean Concentrations | Maximum Authorized Concentration in a Composite Sample | Maximum Authorized Concentration in a Grab Sample |
|---|---|---|---|---|
| 1 | Arsenic | 0.50 mg/L | 0.75 mg/L | 1.00 mg/L |
| 2 | Copper | 0.30 mg/L | 0.45 mg/L | 0.60 mg/L |
| 3 | Lead | 0.20 mg/L | 0.30 mg/L | 0.40 mg/L |
| 4 | Nickel | 0.50 mg/L | 0.75 mg/L | 1.00 mg/L |
| 5 | Zinc | 0.50 mg/L | 0.75 mg/L | 1.00 mg/L |
| 6 | Total Suspended Solids | 15.00 mg/L | 22.50 mg/L | 30.00 mg/L |
| 7 | Radium 226 | 0.37 Bq/L | 0.74 Bq/L | 1.11 Bq/L |

Note: All concentrations are total values.
Current version as posted between Apr 3, 2009 and Apr 15, 2009. SOR/2006-239, s. 24.

## 20.3 ENVIRONMENTAL ASSESSMENT AND PERMITTING

### 20.3.1 PROVINCIAL PROCESS

#### ENVIRONMENTAL IMPACT ASSESSMENT REGULATION

The environmental assessment and permitting process for a development in New Brunswick is managed by the Project Assessment Branch of the DENV. New Brunswick's *Environmental Impact Assessment Regulation* falls under the *Clean Environment Act*.

Commercial extraction and/or processing of a mineral is a Class 1 undertaking under the Act's *Environmental Impact Assessment Regulation*. The filing of an EIA registration document will be required for DENV to determine if any further environmental impact assessment work is required prior to final environmental approval. The EIA registration document must describe the project, the existing environment potentially affected by the project, the anticipated environmental impacts and any proposed mitigative measures that, if implemented, would lessen, eliminate, or avoid such impacts. Environmental approval can typically be accomplished for a project of this nature by the filing of an EIA registration document. A requirement for a full environmental impact statement (EIS) to obtain environmental approval is unlikely, but cannot be ruled out with the current level of project planning.

The review of the registration document includes a public consultation component. The onus is on the proponent to conduct the consultation and to demonstrate to the Department that the potentially affected public and other stakeholders have been given meaningful opportunity to comment. The public consultation/involvement must be conducted in accordance with the minimum standards outlined in the EIA Registration Guide. Evidence of the conduct of public consultation must be provided to the Department within 60 days of project registration.

The Aboriginal Affairs Secretariat (AAS) requires proponents to consult with any and all Aboriginal groups that may be impacted by the project. A Traditional Knowledge Study (TKS) is required by AAS unless an agreement with interested Aboriginal groups is planned.

The Minister will make a decision regarding the project within 30 days of the receipt of sufficient information regarding the project, which includes the EIA registration document and documentation of public and stakeholder concerns.

If, after the Determination/Comprehensive Review, the Minister approves the project to proceed under the *Clean Environment Act*, certificates for Approval to Construct and Approval to Operate will be issued, and other permits necessary for operation can be applied for through the appropriate licensing agency.

Some aquatic resources baseline data have been collected downstream of the property boundary but there are no site-specific environmental baseline data. There is the potential to advance project environmental approvals without having a full-year of site-specific baseline data if an impact-avoidance approach is taken in project planning. This approach would include avoidance of the disturbance or destruction of fish habitat and of any raptor nests, location of all project facilities outside of wetlands, and a commitment to treating any discharges to water to a level that is consistent with the assimilative capacity of the receiving waters. A walk-away closure plan also will be required that ensures long-term secure closure of the site.

Selective site-specific data collection will be necessary to support the preparation of an EA registration document and this work should be undertaken. At a minimum, this data collection should include ground-truthing of wetland delineation, and site-specific water and sediment quality and fish sampling. Partial documentation of rare plant species in area of planned disturbance could be accomplished with surveys according to the known occurrences for the bird breeding season. Surveys of bird species occurrence and the presence/absence of raptor nests on the site would be required to be included. Assessment of site-specific hydrogeological conditions may be necessary to plan dewatering requirements for the open pit, assess the potential effect of the project on local groundwater resources which can be important in maintaining baseflow in local streams; and to predict the filling rate of the mine-out pit. This information can be collected as part of an active exploration drilling program. Geochemical characterization of the waste rock and of the rock types that will form the walls of the final open pit also is needed to design the closure plan and predict water quality in the pit lake that will form after mining.

### *Permitting*

Once the EIA process is concluded and the Approval to Construct and Approval to Operate have been issued, the remaining permits can be obtained. Stratabound has advised that The Mining Lease for the CNE property was successfully renewed prior to September 2011 for an additional 20-year term.

### *Mining Lease*

A mining lease under the *Mining Act* is in place for the CNE property. This lease allows for access to the mine reserve; however, mining activities cannot be initiated under this lease alone. The area under the current mining lease can be increased if Stratabound owns the adjacent mineral claims (C. Neumann, Department of Natural Resources, pers. comm.). Any proposed changes to the active mining leases must be submitted to the Standing Committee on Mining and the Environment (SCME), and will require the proponent to submit a Feasibility Study (or an appropriate presentation of the economic viability of the project) and a detailed Mining and Reclamation Plan, including an estimate of required financial security (C. Neumann, pers. comm.). The SCME operates under the direction of the Minister of Natural Resources.

### *Other Provincial Permits*

Other project related approvals may include development and building permits, approval to install storage for explosives, and water treatment facilities. Additional Crown Land leases may be required during construction at the site, including leases for access roads and utilities. For example, if construction of access roads and utility poles are required, a Licence of Occupation will need to be issued during the construction. The length of this permit cannot exceed twenty years.

Approvals for water use and effluent release are issued under the *Clean Environment Act* and the *Water Quality Regulation*. Applications must be submitted at least 90 days prior to construction. After application submission, the applicant may be required to publish a notice of the application in The Royal Gazette or other newspaper. Objections must be filed with the Minister within thirty days of such publication and objections must filed within sixty days of the filing of the application for the approval. The water quality of the effluent will have to meet at minimum the MMER criteria; however, more stringent New Brunswick specific criteria may be required on the basis of site-specific conditions.

Air quality operating approvals may be required for sources of particulate matter emissions under the *Clean Air Act,* if emissions are expected to be greater than 10 t/a. The approval specifies operating conditions and emission limits. Approvals are classified according to the volume of emissions released. The air quality approvals are issued through the Department of Environment and can be valid for up to five years.

If a water well is to be installed, the well must be installed in compliance with the *Water Well Regulation* under the *Clean Water Act.* The well must be installed by a certified well driller and a detailed water well driller's report must be submitted to the Department of Environment after installation. Tests for water yield will be required to determine the rate of yield of a well, with results reported to the Department of Environment.

A Watercourse and Wetland Alteration Permit may be required if gravel is to be removed from the bed or bank of a watercourse, if a bridge or culvert are installed, or if soil is disturbed or trees are removed within 30 m of a watercourse. This is required under the *Watercourse and Wetland Alteration Regulation* under the *Clean Water Act*. The application for the permit must be submitted at least 60 days prior to start of work.

Approval and License for Petroleum Storage maybe required under the *Petroleum Product Storage and Handling Regulation* under the *Clean Environment Act*. An approval will be required for all storage systems with a total capacity of at least two thousand litres.

The federal *Explosives Act* regulates the manufacturing, testing, sale, storage, transportation and importation of explosives, and may trigger the CEAA (see Section 22.4.4). Specifically, by virtue of the *Explosive Act*, Natural Resources Canada (NRCan) could become involved in the environmental assessment of projects by issuing, if required, a licence for the manufacture (i.e., factory) and/or storage (i.e., magazine) of explosives (Section 7(1)(a) of the *Explosives Act*) (NRCan 2009).

#### *Reclamation and Closure Plan*

A Mining and Reclamation Plan must be prepared and filed with NBDNR in accordance with *Guide to the Development of a Mining and Reclamation Plan in New Brunswick* (NBDNR 2005a). The Mining and Reclamation Plans for the property must include an estimate for mine closure and reclamation costs and an anticipated schedule for these activities. These two items are considered by the Minister of Natural Resources in determining the reclamation security that will be required from the proponent and held by the Province to cover the cost of performing site reclamation if the proponent is unable to fulfill its obligation. Security may be required for the payment of cost with respect to protection, reclamation and rehabilitation of the environment as required by any provision of the *Mining Act* of New Brunswick, and these securities would be credited into the Mine Reclamation Fund.

The reclamation security may be released once reclamation is complete; however, if a site requires long-term water treatment and/or site maintenance, all or part of the reclamation security may be withheld according to the expected annual operational cost and estimated duration of treatment.

It is recognized that a practical reclamation strategy may change from the inception of a mining project due to unforeseen changes in the mining plan, site conditions or improving technology. A proponent may be required to revise and resubmit the reclamation strategy in the event that a project deviates from the documented development plan.

### 20.3.2 FEDERAL PROCESS

#### CANADIAN ENVIRONMENTAL ASSESSMENT ACT

The Canadian Environmental Assessment (CEA) Agency is included in the distribution of EIA Registration document by DENV and in turn distributes the document to federal authorities such as Environment Canada, Health Canada, Fisheries and Oceans Canada, Natural Resources Canada, and Transport Canada. During this time, the federal agencies will determine if a federal environmental assessment is necessary. A federal environmental assessment is triggered when a federal authority determines it must provide a license, permit or an approval that enables a project to be carried out (such as an Authorization under the *Fisheries Act* or *Navigable Waters Protection Act*). No requirement for a federal approval has been identified at this time. However, the most likely trigger is a requirement for access road crossing of a fish bearing stream involving a culvert. In the event that a federal environmental review under CEAA is required, a screening level assessment would be conducted. That review would cover all components of the project even though the federal approval would apply only to one or more specific project components.

#### FISHERIES ACT

Fisheries and Oceans Canada (DFO) is responsible for protecting fish and fish habitat in Canada. Under section 35(1) of the federal *Fisheries Act*, "no person shall carry on any work or undertaking that results in the harmful alteration, disruption or destruction (HADD) of fish habitat" except when authorized by DFO, as contemplated in subsection 35(2) or through regulations under the *Fisheries Act* (DFO 2002). Authorization of a potential HADD requires the preparation by the proponent, and acceptance by DFO, of a habitat compensation plan which is capable of fully replacing the habitat that is being lost or altered.

#### NAVIGABLE WATERS PROTECTION ACT

In general, navigable waters include all bodies of water that are capable of being navigated by any type of floating vessel for transportation, recreation or commerce (Transport Canada 2010). New construction happening in, on, over, under, through or across navigable waters with the potential for interfering with navigation must be reviewed under the *Navigable Waters Protection Act* (NWPA), which is administered by Transport Canada. An application under subsection 10(1) NWPA is also required for the repair, rebuilding or alteration of existing work (Transport Canada 2010).

### METAL MINING EFFLUENT REGULATIONS

The proposed project will be required to comply with the MMER once the mine is in commercial production. The MMER, administered by Environment Canada, were developed under section 36 of the *Fisheries Act* to regulate the discharge of water to the environment. The MMER apply to both existing and new mines. The MMER require the development of an environmental effects monitoring program and annual reporting for performance monitoring.

## 20.4 COMMUNITY AND ABORIGINAL ENGAGEMENT

A Community and Aboriginal engagement program is necessary and should include local communities as well as Aboriginal communities within a 100 km radius of the property. Stakeholders may include town/county councils and organizations, elected officials, business owners, resource users, economic development agencies, and environmental organizations, primarily from within the counties of Gloucester and Northumberland. Public input in relation to the conceptual project description, must be sought, addressed, and documented during the provincial Determination Review. Engagement must be conducted, at minimum, according to *the Guide to Environmental Impact Assessment in New Brunswick*. Community engagement opportunities must be provided by Stratabound throughout the environmental assessment process to provide project specific information and seek out issues, concerns, and ideas for inclusion in the environmental impact statement.

First Nations consultation on part of the Province is handled by the AAS. The AAS should be consulted early in the process to help determine which Mi'kmaq communities or organizations (e.g., the New Brunswick Aboriginal Peoples Council, North Shore Micmac District Council) should be included in Stratabound's public engagement program.

The AAS will require a TKS to be completed prior to approval of the project. The objective of the TKS is to gather information regarding traditional uses of the land within the project footprint, to identify the locations of any interests that need protection, identify any impacts to traditional or current First Nations interests and culture, and to provide solutions to mitigate any negative impacts. At a minimum a TKS will take several months to complete depending on the willingness of the surrounding communities to share information and the availability of Elders, in most cases, to supply the information. AAS may decide that the requirement for a TKS may be waived if an agreement is reached between the proponent and the First Nations with respect to mitigation measures such that Aboriginal and Treaty rights are not affected by the project.

# 21.0 CAPITAL AND OPERATING COSTS

## 21.1 OVERVIEW

Tetra Tech evaluated the capital costs using the following scenarios:

- Captain and CNE as a combined mined deposit, contract mining and toll milling option only
- CNE as a standalone mined deposit, contract mining and toll milling option only.

The capital and operating costs for Scenario 2 were determined to be the optimal scenario based on financial analysis. In Scenario 1, the combined Captain and CNE mine option is determined to have a total of 565,000 t of potentially mineable resource. The evaluation of this combined option has proven that the inclusion of the Captain deposit has a negative impact on the project financials, as the capital costs for the combined option are approximately twice the cost of the CNE standalone option. The CNE deposit as a standalone mineable deposit is the only scenario presented in the capital and operating discussion and the financial analysis discussion within this PEA. The capital costs for the Captain deposit are not detailed in this section as the deposit has not been included in further economic modeling.

For Scenario 2, the CNE deposit is determined to have a total of 325,000 t of potentially mineable resource. The CNE deposit was analyzed using mine rates of 500 t/d and 1,000 t/d. The LOM for each rate was determined to be less than two years and one year, respectively. The analysis using a mine rate of 1,000 t/d provided a better cash flow and economic analysis than using a mine rate of 500 t/d. Due to the short LOM of less than one-year based on the optimal mine rate of 1,000 t/d, the only viable option for the project financial evaluation is based on an open pit contract mine operation and a toll mill option for processing. This is reflected in the presented capital and operating costs. The capital costs are reduced by eliminating the direct purchase of all required mine equipment as well as all metallurgical processing facilities. Incurring capital costs for mining equipment and processing plant requirements for a one year LOM is not economic.

The capital costs for the CNE deposit are estimated at $6.9 million (2011 base year). A detailed summary of the direct and indirect capital costs is presented in Table 21.1. A 15% contingency has been applied to the direct capital costs. The indirect capital costs have been calculated as 1% of the direct capital costs and the Owner's cost has been calculated as 2% of the direct capital costs. Salvage value has been calculated as 10% of the direct capital costs for utilities, mobile equipment, infrastructure, and water treatment.

**Table 21.1 Direct and Indirect Capital Costs**

| Item | Amount ($) |
|---|---|
| **Direct Capital Costs** | |
| Site Development | 1,442,749 |
| Utilities | 2,084,900 |
| Mobile Equipment | 125,000 |
| Infrastructure | 200,000 |
| Water Treatment | 200,000 |
| Closure/Reclamation | 2,000,000 |
| **Subtotal Direct Capital Costs** | **6,052,649** |
| **Indirect Capital Costs** | |
| Indirect Costs | 60,726 |
| Owner's Costs | 121,453 |
| Contingency (15%) | 907,897 |
| Salvage | (260,990) |
| **Subtotal Indirect Capital Costs** | **828,487** |
| **Total Capital Costs** | **6,881,136** |

## 21.2 DIRECT CAPITAL COSTS

The direct capital cost breakdown is presented in Table 21.2. No permanent building or infrastructure will be constructed and electrical power will be supplied by on site generators as minimal site development is planned. Waste rock will be stored on site and water discharge from the site will be controlled and treated through a series of retention ponds and water treatment. Water treatment equipment will be leased or purchased as used equipment. A fueling system will be set up on site with dual fuel storage tanks and a dispensing system.

**Table 21.2 Direct Capital Costs**

| Item | Amount ($) |
|---|---|
| **Site Development Total** | **1,442,749** |
| Early Site Development | 17,500 |
| Site Development | 467,500 |
| Site Roads and Hard Standing Areas | 857,749 |
| Rock Material Dumps | 100,000 |

*table continues…*

| Item | Amount ($) |
|---|---|
| **Site Utilities Total** | **2,084,900** |
| Electrical Distribution | 450,000 |
| Power Supply | 750,000 |
| Water and Sewer Systems | 519,900 |
| Site Fuelling System | 365,000 |
| **Mobile Equipment Total** | **125,000** |
| Water/Sanding/Plowing Truck | 125,000 |
| **Infrastructure** | **200,000** |
| Modular Facilities | 200,000 |
| **Water Treatment Non Process Buildings Total** | **200,000** |
| WTP | 200,000 |
| **Total CNE Direct Capital Costs** | **4,052,649** |

## 21.3 INDIRECT CAPITAL COSTS

The indirect capital cost breakdown consists of the following major category descriptions:

- Indirect Cost $60,526
  - construction facilities
  - commissioning and start-up
  - professional fees – Engineering, Procurement and Construction Management (EPCM)
- Owner's Cost $121,053
  - permitting and licensing
  - miscellaneous taxes, insurance
- Contingency $907,897
  - set at 15%
- Salvage ($260,990)
  - set at 10% of direct capitals costs to spend on utilities, mobile equipment, infrastructure, and water treatment.

The strategy, as directed by Stratabound, involves a contract mining operation and toll milling, effectively eliminating most of the typical indirect and Owner's costs for mine construction projects. As the LOM presented for the CNE deposit is slightly less than one year, at a mine rate of 1,000 t/d, this strategy is reasonable and is reflected in the capital costs for the project. The indirect costs are estimated to be 1% of the total direct capital cost and the Owner's costs are estimated to be 2%.

## 21.4 Operating Costs

Due to a short LOM, it is feasible and practical to use contract mining and a toll mill option for processing. In the financial analysis used in this PEA, a surcharge for toll milling has been built into the process operating cost rather than the NSR values in the financial analysis. The process operating cost used in the financial analysis is $28.53. This represents an approximate 75 to 100% increase in operating cost for a typical 1,000 t/d mill with three concentrates. The financial analysis assumes this is adequate to represent the revenue generated from a toll mill operation. Unit costing for mine operations was supplied by local contracting companies and those details are presented in Table 21.3.

Unit Cost Breakdown (Table 21.3):

- Onsite crushing breakdown:
  - $0.50/t crushing
  - $0.50/t conveying
  - $0.30/t loading
  - $0.52/t added 40% for fuel, infrastructure, and mobilization/demobilization
  - total cost of $1.82 per resource tonne was used.
- The hauling to the toll mill cost is based on:
  - $0.08/t/km
  - 100 km round trip
  - total cost of $8.00 per resource tonne was used.

**Table 21.3 CNE Unit Operating Mining Costs ($/t)**

| Item | Q1 | Q2 | Q3 | Q4 |
|---|---|---|---|---|
| Overburden | 1.10 | 1.10 | 1.10 | 1.10 |
| Waste Rock | 2.28 | 2.28 | 2.28 | 2.28 |
| Resource | 2.84 | 2.84 | 2.84 | 2.84 |
| On-site Crushing (6” minus) | 1.82 | 1.82 | 1.82 | 1.82 |
| Resource Hauling to Mill | 8.00 | 8.00 | 8.00 | 8.00 |

In comparison to the cost models from Tetra Tech’s database, the mine operating costs provided for the CNE operation were deemed to be low. Therefore, Tetra Tech increased all mine operating costs by 25%. The revised operating cost by tonne of material mined and by tonne of resource mined is shown at the bottom of Table 21.4.

The total mine operating costs are estimated at $7.4 million which equates to $5.76/t of material mined or $22.76/t of ore milled. The breakdown of the operating costs is shown in Table 21.4.

**Table 21.4 CNE Operating Mining Costs by Period**

| Item | Unit | Q1 | Q2 | Q3 | Q4 | Total |
|---|---|---|---|---|---|---|
| Overburden | $ | 268,212 | 86,821 | 2,786 | 0 | 357,819 |
| Waste Rock | $ | 172,391 | 390,009 | 555,837 | 325,900 | 1,444,137 |
| Resource | $ | 93,682 | 290,933 | 301,337 | 237,106 | 923,060 |
| Onsite Crushing (6" minus) | $ | 60,036 | 186,443 | 193,111 | 151,948 | 2,600,168 |
| Ore Hauling to Mill | $ | 263,894 | 819,531 | 848,838 | 667,905 | 648,289 |
| **Total** | **$** | **858,215** | **1,773,737** | **1,901,909** | **1,382,860** | **5,916,722** |
| Cost Per Total Resource Mined | $/t | 26.02 | 17.31 | 17.92 | 16.56 | 18.20 |
| Cost Per Total Material Mined | $/t | 2.44 | 5.03 | 5.40 | 6.11 | 4.61 |
| Scale Factor | % | 125 | 125 | 125 | 125 | 125 |
| **Total** | **$** | **1,072,769** | **2,217,172** | **2,377,387** | **1,728,575** | **7,395,902** |
| Revised Cost Per Total Resource Mined | $/t | 32.52 | 21.64 | 22.41 | 20.70 | 22.76 |
| Revised Cost Per Total Material Mined | $/t | 3.04 | 6.29 | 6.75 | 7.63 | 5.76 |

**Table 21.5 CNE Mining Schedule by Quarter (1,000 t/d)**

| Item | Unit | Q1 | Q2 | Q3 | Q4 | Total |
|---|---|---|---|---|---|---|
| Overburden | t | 243,829 | 78,928 | 2,533 | - | 325,290 |
| Waste Rock | t | 75,610 | 171,056 | 243,788 | 142,939 | 633,393 |
| Total Waste | t | 319,439 | 249,985 | 246,321 | 142,939 | 958,684 |
| Resource | t | 32,987 | 102,441 | 106,105 | 83,488 | 325,021 |
| Lead | % | 2.25 | 2.06 | 1.73 | 1.24 | 1.76 |
| Silver | g/t | 60.12 | 60.31 | 55.32 | 58.07 | 58.09 |
| Copper | % | 0.06 | 0.07 | 0.06 | 0.14 | 0.08 |
| Zinc | % | 5.60 | 5.30 | 4.75 | 3.68 | 4.74 |
| Stripping Ratio | - | 9.68 | 2.44 | 2.32 | 1.71 | 2.95 |

### OPEN PIT OPERATIONS MANPOWER

The mine will operate 24 hours per day for 325 days in Year 1. This will exhaust the 325,000 t of ore in the deposit.

Since the mine is considered contractor operated, the manpower working schedule will be determined by the contractor to meet production requirements.

The labour details presented in Table 21.6 represent manpower requirements for a typical owner operated 1,000 t/d open pit mining operation at a 4:1 waste tonnes to resource tonnes stripping ratio.

### Table 21.6 Manpower Requirements for 4:1 Stripping Ratio

| Position | No. |
|---|---|
| **Hourly Personnel Requirements** | |
| Drillers | 5 |
| Blasters | 2 |
| Excavator Operators | 3 |
| Truck Drivers | 7 |
| Equipment Operators | 7 |
| Utility Operators | 3 |
| Mechanics/Electricians | 5 |
| Labourers/Maintenance | 8 |
| **Total Hourly Personnel** | **40** |
| **Salaried Personnel Requirements** | |
| Manager | 1 |
| Superintendent | 0 |
| Foreman | 2 |
| Engineer | 1 |
| Geologist | 0 |
| Supervisor | 1 |
| Technician | 3 |
| Accountant | 0 |
| Clerk | 1 |
| Personnel | 0 |
| Secretary | 1 |
| Warehouse | 0 |
| **Total Salaried Personnel** | **10** |
| **Total Labour** | **50** |

Source: 2009 InfoMine USA, Inc.

# 22.0 ECONOMIC ANALYSIS

## 22.1 METAL PRICING

Currently, Tetra Tech's metal prices are set quarterly. The prices are based on the Consensus Economic Energy and Metal Forecast Group (EMCF) of London. This group provides a quarterly long term forecast (5 to 10 years) for a variety of metals, based on a selection of analysts. The EMCF then averages the analysts' projections in a single consensus forecast. Tetra Tech considers these forecasts to be independent, transparent, consistent, and generally aligned with the timelines for potential construction and operation of mine opportunities considered in most economic studies.

To set the metal prices, Tetra Tech uses the average of three quarterly reports. The reason for averaging three periods is to avoid any single outlier forecast, and to smooth any large fluctuation between quarterly forecasted prices.

Table 22.1 details the metal prices used in the economic analysis for the CNE deposit PEA.

**Table 22.1 Metal Prices**

| Metal | Metal Price | Units |
|---|---|---|
| Zn | 1.22 | US$/lb |
| Cu | 3.62 | US$/lb |
| Pb | 1.10 | US$/lb |
| Ag | 22.74 | US$/oz |

## 22.2 FINANCIAL ANALYSIS

The financial analysis considered a total of 325,000 t of resource from the CNE deposit. The financial analysis is based on the open pit mine design and schedule as defined in Section 16.0 Mining Methods. Revenue contribution is calculated from the NSR for three concentrates. Silver contributes to the NSR value for the lead concentrate and the copper concentrate. The NSR values for these concentrates are shown in Table 22.2.

**Table 22.2 NSR Values**

| Concentrate | NSR Value (US$/dmt) |
|---|---|
| Zn | 965 |
| Cu | 2,030 |
| Pb | 1,763 |

In this analysis, the full value of the NSRs are carried into the financial analysis calculation, even though a toll mill operation is presented as the only viable option for the project. Typically, a toll mill contract will erode the revenue from the NSR paid to the supplier of ore to the mill. In the financial analysis used in this PEA, a surcharge for toll milling has been built into the process operating cost rather than the NSR values in the financial analysis. The process operating cost used in the financial analysis is $28.53. This represents an approximate 75 to 100% increase in operating cost for a typical 1,000 t/d mill with three concentrates. The financial analysis is based on the assumption that this is adequate to represent the revenue generated from a toll mill operation.

The LOM for the project is less than 1 year based on a production rate of 1,000 t/d. A detailed report of the financial analysis and cash flow by quarter is provided in Appendix J.

Due to the deposit size and the resulting short duration of the LOM, the financial analysis is based on a contract mine operation and toll mill processing. The cashflow generated from the CNE deposit will not support capital spending for either the mine operations or the processing plant operations. The mining operating costs is based on a similar size quarry operation in the New Brunswick BMC area. The process operating costs considers an assumed premium for toll mill surcharges on a 1,000 t/d mill throughput.

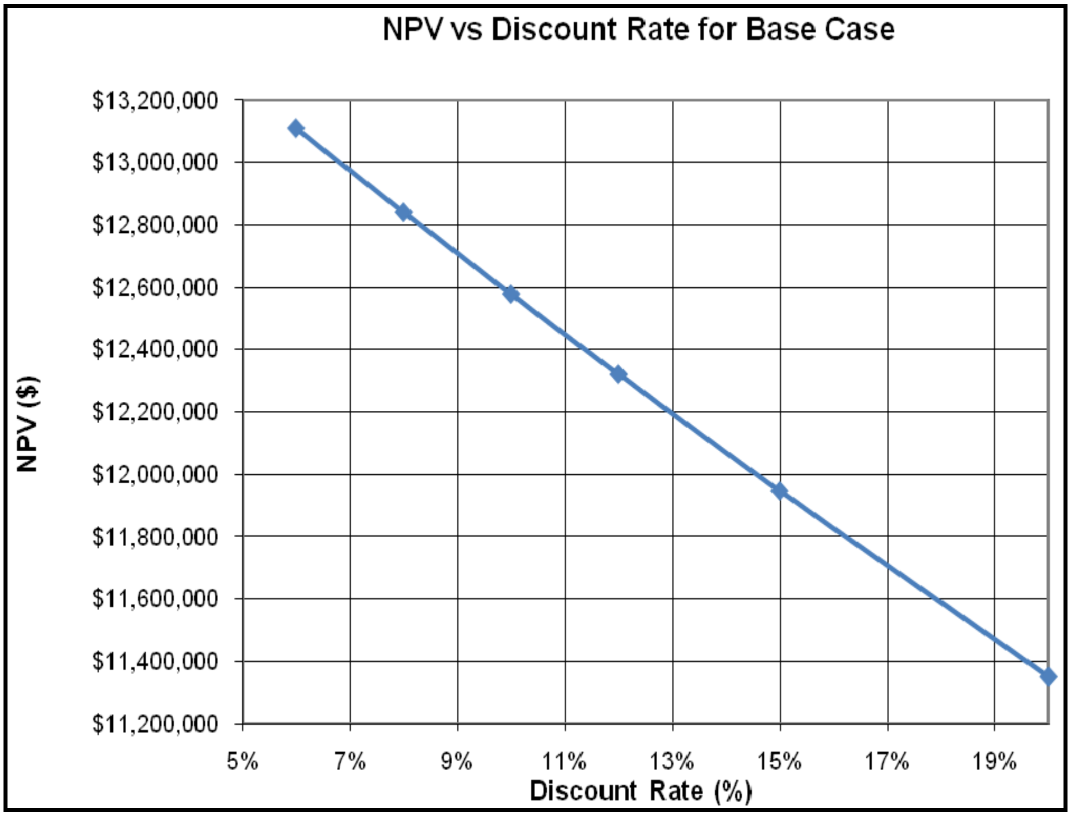
The NPV represents a return based on the single year of operation. For the operating year the gross revenue is $37.6 million, the total operating costs including toll mill surcharges in the process operating cost is $16.8 million and total capital costs are $6.9 million. This results in a pre-tax cash flow of $14.0 million. The annual cash flow, when calculated quarterly with the application of various discount rates is shown in Table 22.3.

**Table 22.3 Pre-tax NPV and IRR**

| Item | Amount |
|---|---|
| Pre-tax & Pre-finance NPV @ 6% | $13,131,483 |
| Pre-tax & Pre-finance NPV @ 8% | $12,862,435 |
| Pre-tax & Pre-finance NPV @ 10% | $12,599,611 |
| Pre-tax & Pre-finance NPV @ 12% | $12,342,838 |
| Pre-tax & Pre-finance NPV @ 15% | $11,968,655 |
| Pre-tax & Pre-finance NPV @ 20% | $11,372,963 |
| Project IRR | 292% |

Figure 22.1 shows the NPV for the various discount rates for the base case scenario (no sensitivities applied), as defined by the metal prices in Table 22.1 and NSR values in Table 22.2. The base case represents the NSR values at different discount rates without the application of any sensitivity analysis.

**Figure 22.1 NPV vs. Discount Rate for Base Case**



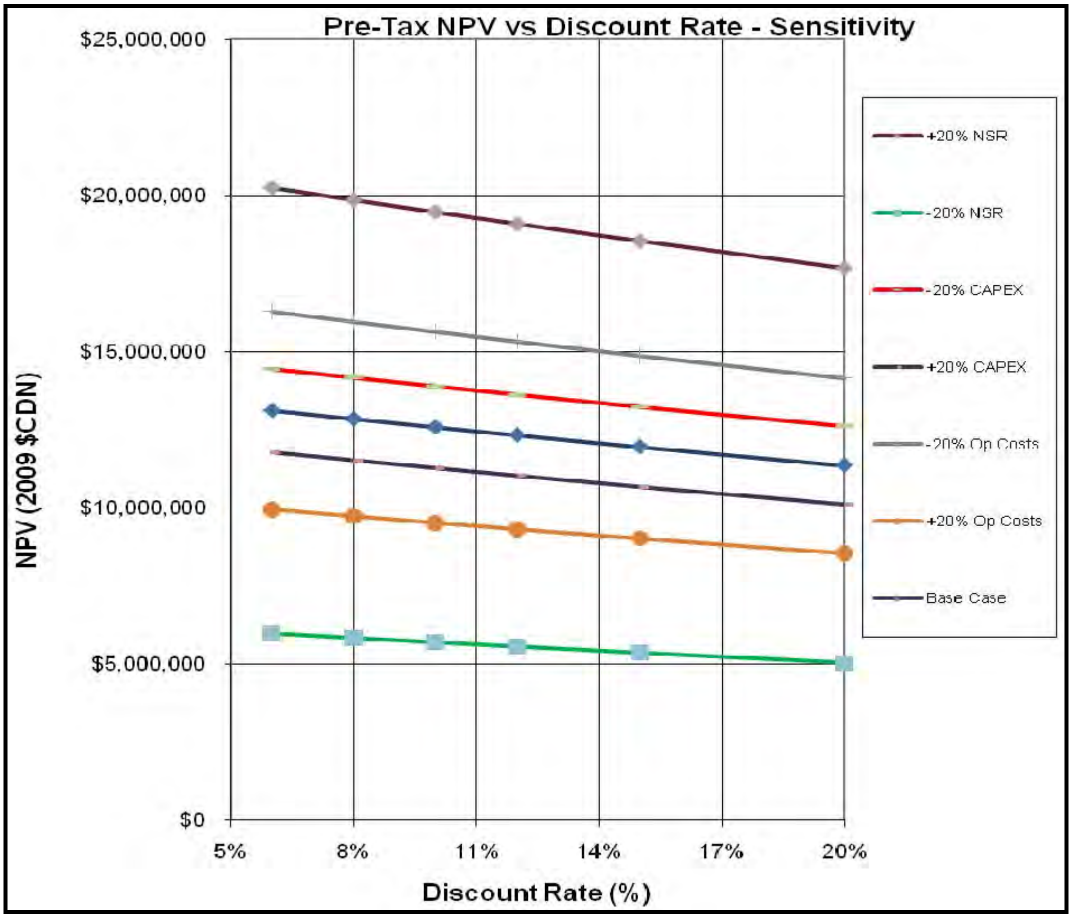

## 22.3 SENSITIVITY ANALYSIS

Tetra Tech ran several sensitivity analyses to determine the effect on key financial statistics if the following basic parameters change:

- the operating costs increased or decreased by 20%
- the capital costs increased or decreased by 20%
- the NSR increased or decreased by 20%
- the NSR for zinc only increased or decreased by 20%.

The pre-tax results are shown in Figure 22.2.

**Figure 22.2 Capital Cost, Operating Cost and NSR Sensitivity**




Based on the sensitivity analysis results, it is clear that the project is most sensitive to variation of the NSR value, much less sensitive on operating costs and least sensitive on capital costs. The sensitivity results are summarized in Table 22.4 and Table 22.5. The IRR sensitivity is shown in Figure 22.3.

Moving forward, an attempt should be made to determine toll mill contract terms in order to accurately predict the revenue and cost streams used in this financial analysis.

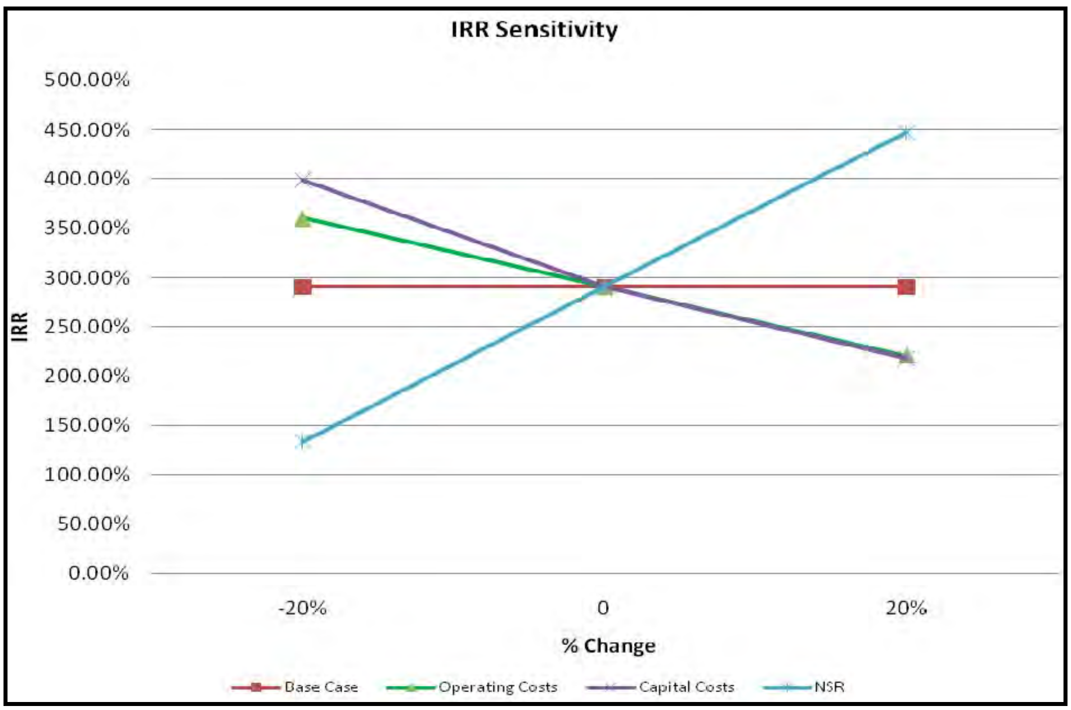
### Table 22.4 Operating Costs and Capital Costs Sensitivity

| | Discount Rate (%) | NPV ($) | NPV Difference ($) | IRR (%) |
|---|---|---|---|---|
| **Base Case** | | | | |
| | 6 | $13,131,483 | - | 290 |
| | 8 | $12,862,435 | - | |
| | 10 | $12,599,611 | - | |
| | 12 | $12,342,838 | - | |
| | 15 | $11,968,655 | - | |
| | 20 | $11,372,963 | - | |
| **Operating Cost** | | | | |
| Increase 20% | 6 | 9,936,212 | ($3,195,271) | 221 |
| | 8 | 9,724,119 | ($3,138,316) | |
| | 10 | 9,516,890 | ($3,082,721) | |
| | 12 | 9,314,391 | ($3,028,447) | |
| | 15 | 9,019,231 | ($2,949,424) | |
| | 20 | 8,549,172 | ($2,823,791) | |
| Decrease 20% | 6 | 16,283,965 | $3,152,482 | 360 |
| | 8 | 15,958,100 | $3,095,665 | |
| | 10 | 15,639,820 | $3,040,209 | |
| | 12 | 15,328,909 | $2,986,071 | |
| | 15 | 14,875,912 | $2,907,257 | |
| | 20 | 14,154,936 | $2,781,973 | |
| **Capital Cost** | | | | |
| Increase 20% | 6 | 11,773,833 | ($1,357,650) | 218 |
| | 8 | 11,518,945 | ($1,343,490) | |
| | 10 | 11,269,962 | ($1,329,649) | |
| | 12 | 11,026,720 | ($1,316,118) | |
| | 15 | 10,672,275 | ($1,296,380) | |
| | 20 | 10,108,065 | ($1,264,898) | |
| Decrease 20% | 6 | 14,446,344 | $1,314,861 | 400 |
| | 8 | 14,163,274 | $1,300,839 | |
| | 10 | 13,886,748 | $1,287,137 | |
| | 12 | 13,616,581 | $1,273,743 | |
| | 15 | 13,222,868 | $1,254,213 | |
| | 20 | 12,596,043 | $1,223,080 | |

**Table 22.5 NSR Sensitivity**

| | Discount Rate (%) | NPV ($) | NPV Difference ($) | IRR (%) |
|---|---|---|---|---|
| Decrease 20% | 6 | 5,977,939 | ($7,153,544) | 134 |
| | 8 | 5,833,732 | ($7,028,703) | |
| | 10 | 5,692,826 | ($6,906,785) | |
| | 12 | 5,555,131 | ($6,787,707) | |
| | 15 | 5,354,420 | ($6,614,235) | |
| | 20 | 5,034,773 | ($6,338,190) | |
| Increase 20% | 6 | 20,242,238 | $7,110,755 | 447 |
| | 8 | 19,848,487 | $6,986,052 | |
| | 10 | 19,463,883 | $6,864,272 | |
| | 12 | 19,088,170 | $6,745,332 | |
| | 15 | 18,540,723 | $6,572,068 | |
| | 20 | 17,669,335 | $6,296,372 | |

**Figure 22.3 IRR Sensitivity**



# 23.0 ADJACENT PROPERTIES

There are no significant properties adjacent to the Captain, CNE or Taylor Brook properties.

On July 29, 2010, Stratabound entered into an option agreement with Commander Resources Ltd. whereby Stratabound can acquire up to a 65% interest in the adjoining mineral claim (Claim No. 1891, Nepisiguit Brook) to the north of Stratabound's CNE Group mineral claim. The mineralization on this mineral claim resembles Stratabound's Captain deposit which is located along strike, 4.5 km to the south (Stratabound Press Release July 2010).

# 24.0 OTHER RELEVANT DATA AND INFORMATION

This Section is not applicable to this report.

# 25.0 INTERPRETATIONS AND CONCLUSIONS

## 25.1 GEOLOGY

### 25.1.1 THE CNE DEPOSIT

Tetra Tech has estimated an NI 43-101 compliant resource of massive sulphide mineralization that is the CNE VMS deposit. Stratabound supplied all of the digital data for the CNE resource estimate, dated September 2010. Stratabound compiled the historical drillhole data from previous assessment reports which are publicly available on the NBDNR website. This data was imported into Datamine™ Studio 3 (Version 3.19.3638.0) resource software, and from Datamine™, final resource figures were generated.

The block model was composed of cells measuring 2 m x 2 m x 2 m. Sub-cells are not used except to better define the surface (air-overburden interface). It is believed that the 2 $m^3$ cell-size provides the best resolution for mineralization, given the overall close data (assay) density. The total volume represented by the CNE block model is 16.8 $Mm^3$, which includes fresh rock, overburden and “air”.

Indicator modelling is ideally suited to deposits, like CNE, which are geologically complex with an abundance of data. For the zinc-lead-silver domain, the assigned indicator cut-off is 3% Zn. For the copper domain, the assigned indicator is 0.2% Cu. This method was used to select cells for grade estimation; it was not used to assign grades to cells. OK was used to assign final grade to cells for the base metals; $ID^2$ was used for silver.

At a 1.5% ZnEQ cut-off, the CNE deposit reports a Measured Resource of 30.7 kt at 5.77% Zn, 1.91% Pb, 0.06% Cu and a silver grade of 63.91 g/t; an Indicated Resource of 277.0 kt at 4.54% Zn, 1.66% Pb, 0.11% Cu and a silver grade of 60.40 g/t; and an Inferred Resource of 16.5 kt at 2.74% Zn, 1.20% Pb, 0.06% Cu and a silver grade of 37.50 g/t.

## 25.2 MINERAL PROCESSING AND METALLURGICAL TESTING

The historical data shows that ore from the Stratabound deposits were processed by Heath Steele for an extended period. However, as described above, the CNE deposit is a geologically complex ore-body.

Arsenic was found in some samples, whereas there is no comment about deleterious elements in the historical operations.

With smelter emissions more closely monitored than they were twenty years ago, it would be beneficial to update the metallurgical test work, and to review procedures to depress any potentially deleterious elements. The ore from CNE will be toll milled, therefore depending on the process flow sheet for the prospective toll mill, the updated test work may be a requirement for verification of ore compatibility and mill optimization for the toll mill operation.

## 25.3 MINING OPERATIONS

Tetra Tech initiated an open pit design for Stratabound's Taylor Creek, CNE, and Captain properties. The Taylor Creek property did not proceed beyond the initial stage of optimization using the LG pit optimization method. The CNE and Captain properties were analyzed beyond the initial optimization stage to a second stage requiring the development of a mining plan and production schedules. For the CNE and Captain properties, equipment selections were performed and the capital and operating costs were estimated. Initially the second stage of analysis was performed for the CNE and Captain properties as sequential ore feeds. Due to negative cashflow contributions from Captain, CNE was analyzed as a standalone open pit mine operating without Captain.

For the standalone CNE open pit model, the ultimate pit design for the project contains 313,357 t of Measured and Indicated Resource at an average diluted grade of 4.77% Zn, 1.76% Pb, 0.08% Cu, and 59.40 g/t Ag and 11,664 t of Inferred Resource at an average diluted grade of 3.79% Zn, 1.77% Pb, 0.05% Cu, and 22.71 g/t Ag. The overall stripping ratio is 2.95 t/t (waste/resource). A total of 958,684 t of waste material will be moved over the one-year mine life.

## 25.4 CAPITAL AND OPERATING COSTS ESTIMATES

### *25.4.1 CAPITAL COSTS*

The analysis of the standalone CNE deposit using a mine rate of 1,000 t/d provided the best cash flow and economic analysis than other considered scenarios. The project financial evaluation is based on an open pit contract mine operation and a toll mill option for processing. The capital costs are reduced by eliminating the direct purchase of all required mine equipment as well as all metallurgical processing facilities.

For the CNE deposit, the capital costs are estimated at $6.9 million (2011 base year). The direct capital costs are $6.05 million and the indirect capital costs are $0.83 million.

### *25.4.2 OPERATING COSTS*

Due to a short LOM, it is feasible and practical to use contract mining and a toll mill option for processing. The process operating cost accounts for the premium to be paid for toll milling and is calculated at $28.53. Unit costing for mine operations was supplied by local contracting companies was adjusted based on relative mine operating costs from Tetra Tech's database. The mine operating cost supplied by local contractors was $18.20/t of ore mined, however the adjusted operating cost used in the PEA was $22.76/t ore mined.

The mine will operate 24 hours per day for 325 days in Year 1. This will exhaust the 325,000 t of ore in the deposit. Since the mine is considered contractor operated, the manpower working schedule will be determined by the contractor to meet production requirements.

## 25.5 ECONOMIC ANALYSIS

Table 25.1 details the metal prices used in the economic analysis for the CNE deposit PEA.

**Table 25.1 Metal Prices**

| Metal | Metal Price | Units |
|---|---|---|
| Zn | 1.22 | US$/lb |
| Cu | 3.62 | US$/lb |
| Pb | 1.10 | US$/lb |
| Ag | 22.74 | US$/oz |

Revenue contribution is calculated from the NSR for three concentrates. Silver contributes to the NSR value for the lead concentrate and the copper concentrate. The NSR values for these concentrates are shown in Table 25.2.

**Table 25.2 NSR Values**

| Concentrate | NSR Value (US$/dmt) |
|---|---|
| Zn | 965 |
| Cu | 2,030 |
| Pb | 1,763 |

For the operating year the gross revenue is $37.6 million, the total operating costs including toll mill surcharges is $16.8 million and total capital costs are $6.9 million. This results in a pre-tax cash flow of $14.0 million.

In conclusion, the CNE deposit is a positive short-term venture with potential for a large single year profit relative to capital outlay. This result is dependent on successfully acquiring a toll mill contract and contract mining rate as used in this report. The values used in this report have not been tested in the open market and present the greatest risk to attain the single year pre-tax cash flow of $14.0 million.

At this point, Taylor Brook and Captain deposits are not economically viable.

# 26.0 RECOMENDATIONS

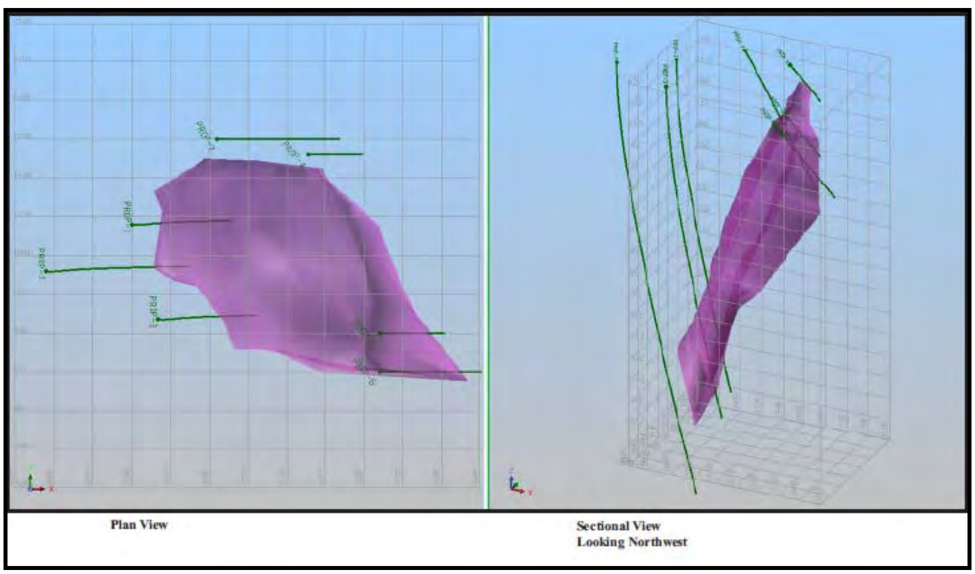
## 26.1 GEOLOGY

### 26.1.1 CAPTAIN

Based on results of the resource estimation program completed by Mercator, the following recommendations are provided with respect to future exploration and resource delineation programs for the Captain deposit:

- The Captain deposit still remains open at depth and further assessment of the higher grade core of the deposit in this direction is warranted. Completion of a single exploratory drillhole measuring approximately 500 m in length is recommended, with this designed to target the core zone approximately 50 m vertically below the CP10-30 intercept (see Figure 26.1).

**Figure 26.1 Target Areas for Further Drilling**




- Two drillholes totalling approximately 770 m in combined length should be completed between CP10-26 and shallower holes to better define this area of the deposit.
- Four near surface drillholes totalling approximately 380 m in length should be completed to better define near surface immediate strike extensions of the deposit.

- Results of the current Captain resource estimate should be included in any future economic assessment studies of Stratabound assets in the BMC.
- The recommendation of Cullen and Harrington (2009) that selective bore hole EM surveying be undertaken at Captain remains valid at the current report date. Strong off-hole anomalies delineated by such work should be considered high priority drilling targets.
- The recommendation of Cullen and Harrington (2009) that further assessment of deposit metallurgical characteristics be carried out also remains valid at the current report date.

### 26.1.2 CNE

Tetra Tech recommends the following for the CNE deposit:

- If an accurate consideration for the previously mined resource is required by Stratabound prior to developing a mine plan for CNE, a detailed pit map to accurately delineate the volume which has been mined out should be completed. Some plans have been drawn to date, but have not been translated into a wireframe and incorporated into the block model.

### 26.1.3 TAYLOR BROOK DEPOSIT

Tetra Tech recommends the following for the Taylor Brook deposit:

- A detailed review of the historical drill logs and drill core to standardize the lithology in the lithology database. This will facilitate geological interpretation should future exploration drilling be carried out.
- Any further development of the Taylor Brook deposit should include a drill program. The Taylor Brook deposit appears to have a nucleus of massive suphides concentrated in the northwest of the deposit that has been established in the 1995 drill campaign. Tetra Tech believes that further drilling is warranted in this area.

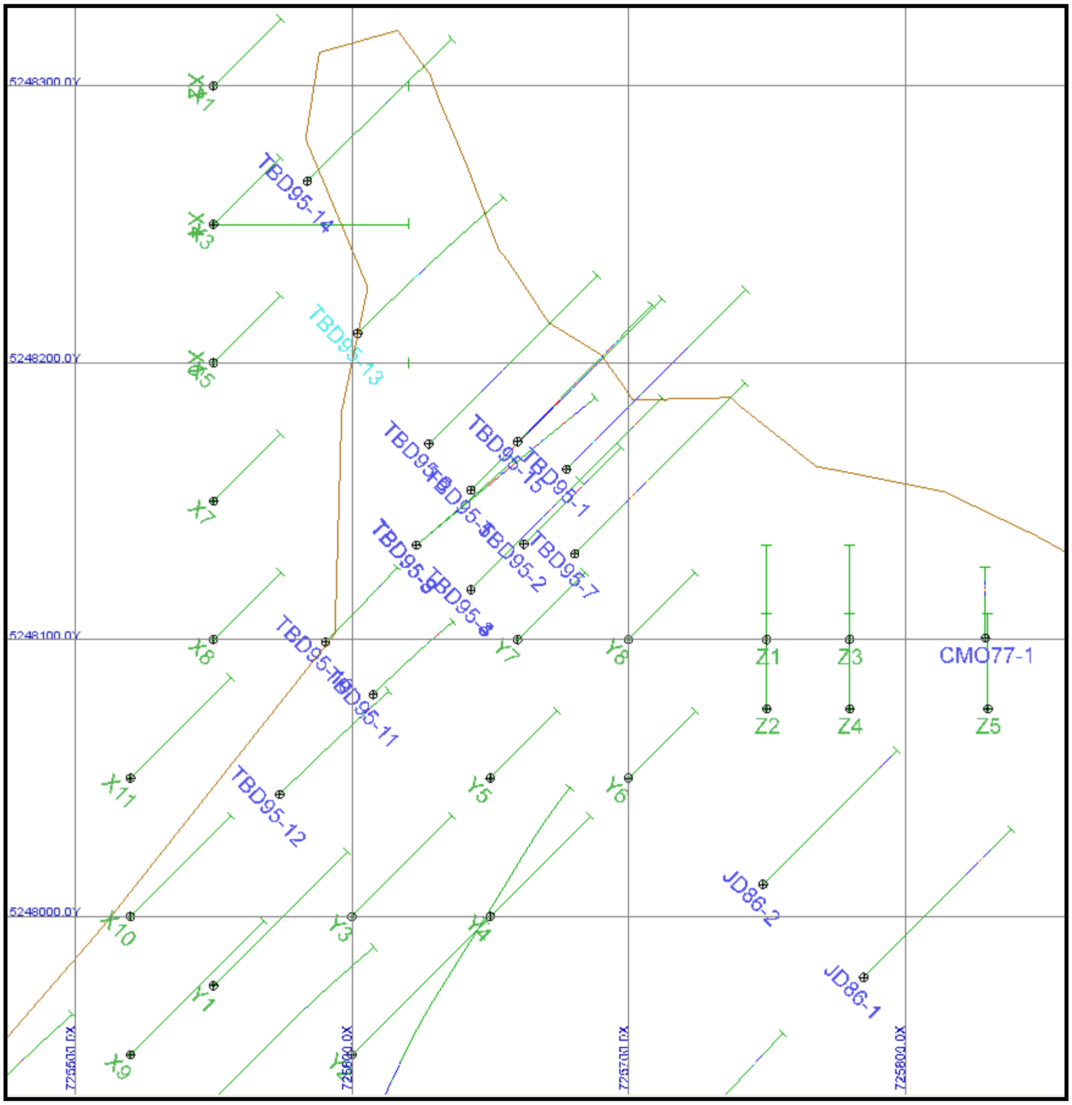
At the request of Stratabound, Tetra Tech has compiled a summary list of proposed drillhole locations that are listed in Table 26.1. Twenty-four drill holes are proposed for a total of 2,900 m of drilling. The locations of these proposed drillholes are shown in Figure 26.2. Eleven drillholes were located along the western edge of the deposit as there has been no drilling to determine the western extent of the massive sulphide zones. Eight drill holes are proposed along the southeast of the 1995 drillholes to ascertain continuity of geology and grade to an approximate depth of 150 m. Lastly, five drillholes are proposed between TBD95-7 and CM077-1 (Figure 26.2) to determine the continuity of geology and grade to the east of the 1995 drilling. The estimated cost for this drill program is approximately Cdn$350,000.

**Table 26.1 Proposed Drillhole Locations for Taylor Brook Property**

| Target Area | Proposed Drillhole | UTM E | UTM N | Azimuth (°) | Dip (°) | Approximate Length (m) |
|---|---|---|---|---|---|---|
| West | X1 | 725550 | 5248300 | -70 | 45 | 100 |
| | X2 | 725550 | 5248300 | -45 | 90 | 100 |
| | X3 | 725550 | 5248250 | -70 | 45 | 100 |
| | X4 | 725550 | 5248250 | -45 | 90 | 100 |
| | X5 | 725550 | 5248200 | -70 | 45 | 100 |
| | X6 | 725550 | 5248200 | -45 | 90 | 100 |
| | X7 | 725550 | 5248150 | -70 | 45 | 100 |
| | X8 | 725550 | 5248100 | -70 | 45 | 100 |
| | X9 | 725520 | 5247950 | -70 | 45 | 200 |
| | X10 | 725520 | 5248000 | -70 | 45 | 150 |
| | X11 | 725520 | 5248050 | -70 | 45 | 150 |
| Centre | Y1 | 725550 | 5247975 | -70 | 45 | 200 |
| | Y2 | 725600 | 5247950 | -70 | 45 | 200 |
| | Y3 | 725600 | 5248000 | -70 | 45 | 150 |
| | Y4 | 725650 | 5248000 | -70 | 45 | 150 |
| | Y5 | 725650 | 5248050 | -70 | 45 | 100 |
| | Y6 | 725700 | 5248050 | -70 | 45 | 100 |
| | Y7 | 725660 | 5248100 | -70 | 45 | 100 |
| | Y8 | 725700 | 5248100 | -70 | 45 | 100 |
| Eastt | Z1 | 725750 | 5248100 | -70 | 0 | 100 |
| | Z2 | 725750 | 5248075 | -70 | 0 | 100 |
| | Z3 | 725780 | 5248100 | -70 | 0 | 100 |
| | Z4 | 725780 | 5248075 | -70 | 0 | 100 |
| | Z5 | 725830 | 5248075 | -70 | 0 | 100 |
| **Total** | - | - | - | - | | **2,900** |

Drillholes X1 to X6 are shown with two drill directions and are proposed as such to determine more precisely the direction of sulphide mineralization in the northwest arm of the Taylor Brook deposit.

**Figure 26.2 Proposed Drillhole Locations on the Taylor Brook Deposit**




## 26.2 MINERAL PROCESSING

The project is based on toll milling only the CNE ore. Toll milling is the only viable option as the project does not generate sufficient cash flow to consider an owner constructed and operated mill. The key recommendations for mineral processing are:

- Engage potential mills in negotiations regarding contract terms. Currently in the economic analysis, a toll mill surcharge to the mine operator has been applied in the process operating costs.

- Metallurgical test work to determine the compatibility of the Stratabound ore to the selected host mill's respective ore will be required to confirm contract terms.
- Regarding the Captain deposit, metallurgical test work is required to determine the process flow sheet for the recovery of cobalt. Currently insufficient test data is available to determine cobalt liberation and processing, and as a result cobalt was not included in the mine optimization and mine plan. The Captain deposit was not economically feasible and was excluded from the economic analysis of this PEA.

## 26.3 MINING OPERATIONS

The study indicated that a resource of 325,290 t, grading 1.76% Pb, 58.09 g/t Ag, 0.08% Cu, and 4.74% Zn is contained within the base case pit for the CNE deposit. This is sufficient for a one-year mine life at a 1,000 t/d production rate.

With an updated geological model, a re-optimization of the pit and pit design should be completed.

A further geotechnical assessment should be undertaken to determine final pit wall slope parameters. This will be a requirement prior to pit construction.

Due to the strategy of contract mining employed in this study, an effort to determine the availability and cost for local contract mining is required to validate the cost assumptions used in this report.

## 26.4 GEOTECHNICAL

The following items are recommended action items to satisfy the geotech design requirements:

- Details on the wall slopes by rock types should be determined for the typical CNE orientations.
- The stability of the waste dump should be determined to ensure a proper design while minimizing the footprint. This needs to be assessed for the next level of study.
- A comprehensive geotechnical study is required for the next level of study or for detail mine design, since the pit slope angle was based on conservative estimates from previous experience.
- A comprehensive hydrological study is required for the next level of study or for detail mine design and will be required to determine the dewatering requirements and the surface water management plan for the site.

## 26.5 ENVIRONMENTAL

The project will commence upon approval under the *Clean Environment Act* as well as the approval of the additional required permits. The approval application will need to be submitted by January 2012 to facilitate completion of permitting by the end of Q3 2012.

The completion of environmental approvals on this schedule will require demonstration (in the Environmental Determination submission) that the project is not a likely cause of significant adverse environmental impacts and, in particular, will not involve the alteration, disturbance, or destruction of fish habitat. It also will be necessary to ensure the project avoids interaction with wetlands. The quantity and quality of mine water and runoff from waste rock dump(s) also needs to be quantified and demonstrated that none of the pit walls or the waste dumps will be a source of acid rock drainage or metal leaching, either because of the rock characteristics or planned closure method.

Development of the environmental determination document will require site-specific environmental information that must be collected before winter in order to meet the review timeline. The following is the absolute minimum information requirement. Additional information may be requested by New Brunswick Environment or other agencies but, in our experience, the following will address the key requirements.

Study requirements:

- fish and fish habitat survey of all waterbodies directly involved in the project (i.e. mine site and any access road stream crossings)
- water quality survey of all waterbodies directly involved in the project (i.e., mine site and any access road stream crossings) and immediate downstream receiving water bodies
- wetland delineation throughout the footprint of disturbance
- ARD/ML testing on all waste rock and ore rock types, including acid-base accounting and following humidity cell tests (minimum six month duration)
- groundwater quality/quantity study (potentially using existing or new exploration drillholes) to estimate quantity and quality of groundwater that will seep into the pit and require management.

Stratabound should plan on continuing the water quality work into the open water season of 2012 and may be required by New Brunswick Environment to conduct some terrestrial survey work in spring/summer 2012. However, the most important information requirements are listed above. An estimated engineering cost for the recommended activities is $120,000. An estimated cost for required field and lab work is $80,000.

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# APPENDIX A

## CERTIFICATES OF QUALIFIED PERSONS

I, Daniel Coley, MBA, P.Eng., of Toronto, Ontario, do hereby certify:

- I am a Senior Process Engineer with Tetra Tech WEI Inc. with a business address at 900-330 Bay Street, Toronto, Ontario, M5H 2S8.
- This certificate applies to the technical report entitled Preliminary Economic Assessment on the Captain, CNE, and Taylor Brook VMS Deposits, New Brunswick, Canada, dated November 23, 2011 (the “Technical Report”).
- I am a graduate of the University of the West Indies (B.Sc. Chemical Engineering, 1991) and Nova South Eastern University (MBA, 1991). I am a member in good standing of the Professional Engineers of Ontario (License #100132643) and the Association of Professional Engineers and Geoscientists of Saskatchewan (License #16618). My relevant experience is 20 years of experience in mining minerals and chemical processes covering design and engineering integration for processes to include base metals, rare earth elements, alumina, potash, boron, graphite, sulphuric acid, potash, water purification and management, reagent systems design. EPCM experience covers scoping studies prefeasibility studies, process design criteria development, feasibility studies, detailed design, construction management, commissioning and start-up, operator training and process system design validation. Experience also extends to project costing and economic analyses.]. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- I have not visited the Property that is the subject of this Technical Report.
- I am responsible for Sections 1.5, 1.10.3, 13.0, 17.0, 25.2, 26.2 and 27.0 (Metallurgy) of the Technical Report.
- I am independent of Stratabound Minerals Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the Sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 23rd day of November, 2011 at Toronto, Ontario.

*“Original document signed and sealed by Daniel Coley, MBA, P.Eng.”*

Daniel Coley, MBA, P.Eng.
Senior Process Engineer
Tetra Tech WEI Inc.

I, Daniel Gagnon, P.Eng., of Sudbury, Ontario, do hereby certify:

- I am a Senior Open Pit Mining Engineer with Tetra Tech WEI Inc. with a business address at 101-957 Cambrian Heights, Sudbury ON, P3C 5M6.
- This certificate applies to the technical report entitled Preliminary Economic Assessment on the Captain, CNE, and Taylor Brook VMS Deposits, New Brunswick, Canada, dated November 23, 2011 (the "Technical Report").
- I am a graduate of Laurentian University (B.Sc. Mining Engineering, 1993). I am a member in good standing of the Association of Professional Engineers Ontario (License #100101928). My relevant experience includes 18 years of open pit mining experience with a strong mine planning background and considerable operational experience in base metals, precious metals, and coal. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not visited the Property that is the subject of this Technical Report.
- I am responsible for Sections 1.6, 15.0, 16.0, 25.3, 26.3, and 26.4 of the Technical Report.
- I am independent of Stratabound Minerals Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the Sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 23rd day of November, 2011 at Toronto, Ontario.

*"Original document signed and sealed by Daniel Gagnon, P.Eng."*

Daniel Gagnon, P.Eng.
Senior Open Pit Mining Engineer
Tetra Tech WEI Inc.

I, Doug Ramsey, R.P. Bio (BC), of Vancouver, BC, do hereby certify:

- I am a Manager – Environmental Assessment, Permitting, and Natural Resources with Tetra Tech, with a business address at 800-555 West Hastings St., Vancouver, BC, V6B 1M1.
- This certificate applies to the technical report entitled Preliminary Economic Assessment on the Captain, CNE, and Taylor Brook VMS Properties, New Brunswick, Canada, dated November 23, 2011 (the "Technical Report").
- I am a graduate of the University of Manitoba, Winnipeg, Manitoba (B.Sc. (Hons), Zoology, 1979, and M.Sc. Zoology, 1985). I am a member in good standing of the College of Applied Biology, British Columbia, as a Registered Professional Biologist (#1581). My relevant experience is 29+ years of experience as an environmental consultant working in environmental permitting and 23 years of experience in the environmental permitting monitoring and closure of mining projects. My mining permitting and planning experience includes NB, NL,PQ, ON, MB, SK, NWT, YK, and BC and includes coal, gold, base metal, rare earth elements, and potash. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not visited the Property that is the subject of this Technical Report.
- I am responsible for Sections 1.7, 1.10.4, 20.0, 26.5, and 27.0 (Environmental) of the Technical Report.
- I am independent of Stratabound Minerals Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the Sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 23rd day of November, 2011 at Vancouver, British Columbia.

*"Original document signed and sealed by Doug Ramsey, R.P. Bio. (BC)"*

Doug Ramsey, R.P. Bio. (BC)
Manager – Environmental Assessment,
Permitting, and Natural Resources
Tetra Tech

I, Mike McLaughlin, P.Eng., of Barrie, Ontario, do hereby certify:

- I am a Project Manager with Tetra Tech WEI Inc. with a business address at 900-330 Bay Street, Toronto, Ontario, M5H 2S8.
- This certificate applies to the technical report entitled Preliminary Economic Assessment on the Captain, CNE, and Taylor Brook VMS Deposits, New Brunswick, Canada dated November 23, 2011 (the “Technical Report”).
- I am a graduate of McMaster University, B.Eng. in Mechanical Engineering, 1990. I am a member in good standing of the Association of Professional Engineers Ontario (License #10084932). My relevant experience includes +14 years of engineering experience, successfully managing projects involving front end mining resource models and economic evaluation studies. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent person inspection of the Property was on November 29, 2010 for two days.
- I am responsible for Sections 1.1, 1.8, 1.9, 1.10.2, 2.0, 3.0, 18.0, 19.0, 21.0, 22.0, 24.0, 25.4, and 25.5 of the Technical Report.
- I am independent of Stratabound Minerals Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the Sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 23rd day of November, 2011 at Toronto, Ontario.

*“Original document signed and sealed by Mike McLaughlin, P.Eng.”*

Mike McLaughlin, P.Eng.
Project Manager
Tetra Tech WEI Inc.

I, Paul Daigle, P.Geo., of Toronto, Ontario, do hereby certify:

- I am a Senior Geologist with Tetra Tech WEI Inc. with a business address at 900-330 Bay Street, Toronto, Ontario, M5H 2S8.
- This certificate applies to the technical report entitled Preliminary Economic Assessment on the Captain, CNE, and Taylor Brook VMS Deposits, New Brunswick, Canada dated November 23, 2011 (the “Technical Report”).
- I am a graduate of Concordia University, (B.Sc. Geology, 1989). I am a member in good standing of the Association of Professional Geoscientists of Ontario (Registration #1592) and the Association of Professional Engineers and Geoscientists of Saskatchewan (Registration #10665). My relevant experience includes over 19 years of experience in a wide variety of geological settings. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was October 18, 2010 for two days.
- I am responsible for Sections 1.2, 1.3, 1.4.3, 1.10.1 (Taylor Brook), 4.0, 5.0, 6.0, 7.0, 8.0, 9.0, 10.0, 11.0, 12.0, 14.3, 23.0, 26.1.3, and 27.0 (Geology) of the Technical Report.
- I am independent of Stratabound Minerals Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the Sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 23rd day of November, 2011 at Toronto, Ontario.

*“Original document signed and sealed by Paul Daigel, P.Geo.”*

Paul Daigle, P.Geo.
Senior Geologist
Tetra Tech WEI Inc.

I, Robert Sinclair Morrison, Ph.D., MAusIMM (CP), P.Geo., of Toronto, Ontario, do hereby certify:

- I am a Lead Resource Geologist with Tetra Tech WEI Inc., with a business address at Suite 900, 330 Bay Street, Toronto, ON, M5H 2S8.
- This certificate applies to the technical report entitled Preliminary Economic Assessment on the Captain, CNE, and Taylor Brook VMS Deposits, New Brunswick, Canada, dated November 23, 2011 (the "Technical Report").
- I am a graduate of Acadia University, (B.Sc. 1981) and University of Adelaide (Ph.D. 1990). I am a member in good standing of the Australasian Institute of Mining and Metallurgy (#11212), and I am registered as a Chartered Professional in Geology with the Australasian Institute of Mining and Metallurgy since 2004. I am a member in good standing of the Association of Professional Geoscientists of Ontario (#1839) since 2010. My relevant experience with respect to base metal deposits includes three years as Senior Resource Geologist with BHP Billiton for their Olympic Dam Expansion Project in South Australia. My relevant experience with respect to deposit geology, ore body modelling and resource estimation includes 10 years with WMC Resources and Gold Fields Ltd as an Extensional Exploration Geologist, Senior Project Geologist, Resource Evaluation Geologist and Senior Resource Evaluation Geologist at the St Ives Gold Mine. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not visited the Property that is the subject of this Technical Report.
- I am responsible for Sections 1.4.2, 1.10.1 (CNE), 14.2, and 25.1.1, and 26.1.2 of the Technical Report.
- I am independent of Stratabound Minerals Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the Sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 23rd day of November, 2011 at Toronto, Ontario

*"Original document signed and sealed by Robert Sinclair Morrison, Ph.D., MAusIMM (CP), P.Geo."*

Robert Sinclair Morrison, Ph.D.,
MAusIMM (CP), P.Geo.
Lead Resource Geologist
Tetra Tech WEI Inc.

**mercator** GEOLOGICAL SERVICES

I, Michael P. Cullen, P.Geo., of Dartmouth, Nova Scotia, do hereby certify:

- I am a Senior Geologist with Mercator Geological Services Limited with a business address at 65 Queen Street, Dartmouth, Nova Scotia, B2Y 1GA.
- This certificate applies to the technical report entitled Preliminary Economic Assessment on the CNE, Captain, and Taylor Brook VMS Deposits, New Brunswick, Canada, dated November 23, 2011 (the "Technical Report").
- I am a graduate of Dalhousie University, (M.Sc. (Geology), 1984) and Mount Allison University (B.Sc. Honours (Geology), 1980). I am a member in good standing of the Association of Professional Geoscientists of Nova Scotia (Registration #064), the Newfoundland and Labrador Professional Engineers and Geoscientists (Registration #05058), and the Association of Professional Engineers and Geoscientists of New Brunswick (Registration #L4333). My relevant experience includes (1) co-authoring two previous resource estimates for the Captain Cu-Co deposit, these having effective dates of October 29th, 2008 and December 8th, 2010, (2) over 30 years of experience relating to study and evaluation of geology and mineral deposits of the Appalachian system and other areas of Canada and abroad, (3) specific experience relating to implementation, evaluation and/or review of various volcanogenic massive sulphide exploration programs carried out in the Bathurst Mining District of New Brunswick, Canada, which contains the Captain Cu-Co deposit.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was October 15, 2010 for 1 day.
- I am responsible for Sections 1.4.1, 1.10.1 (Captain), 14.1, and 26.1.1 of the Technical Report.
- I am independent of Stratabound Minerals Corp. as defined by Section 1.5 of the Instrument.
- Other than participation as a co-author in the two resource estimates noted above, I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the Sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 23rd day of November, 2011 at Dartmouth, Nova Scotia.

LICENCED PROFESSIONAL GEOSCIENTIST
Province of New Brunswick
L4333 2011
Michael P. Cullen
Signature
Nov 23, 2011
Date
GÉOSCIENTIFIQUE TITULAIRE DE PERMIS
Province du Nouveau-Brunswick

Michael P. Cullen, P.Geo.
Senior Geologist

# APPENDIX B

## MINERAL CLAIMS

## Search

Current time: 14 Oct 2011, 4:47:08 PM

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### Mineral Claim Detail

| | |
|---|---|
| **Right Number** | 251 |
| **Claim Type** | Mineral |
| **Claim Sub Type** | Lease |
| **Title Type** | Mineral Lease |
| **Expiry Date** | 2011-08-22 |
| **Issue Date** | 1991-08-23 |
| **Termination Type** | Expired |
| **Termination Date** | 2011-08-23 |
| **Claim Name** | CNE |
| **Status** | Cancelled |
| **NTS Sheet** | |
| **Owners** | |

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Current time: 14 Oct 2011, 4:45:07 PM

## Search

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### Mineral Claim Detail

**Right Number** 1564
**Claim Type** Mineral
**Claim Sub Type** Claim
**Title Type** Mineral Claim Group
**Expiry Date** 2012-03-03
**Issue Date** 1987-03-03
**Claim Name** C.N.E. Group
**Status** Active
**NTS Sheet**

**Owners** 13727 STRATABOUND MINERALS CORP. 100%

**Claim Events:**

| Submitter | Event | Effective Date |
|---|---|---|
| 12193 | Renewal ---109 | 2010-02-24 |
| 12193 | Renewal ---3542 | 2011-02-09 |

**Number of Units** 76
**Work Applied** $967,172.01
**Work Required** $690,487.17
**Excess Work** $276,684.85

**Units:**

| Unit Id | Expiry Date | Total Applied | Work Required | Excess Work |
|---|---|---|---|---|
| 011532 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 011533 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 011534 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 011535 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 011536 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 011537 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 011538 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 011539 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 011540 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 013501 | 2012-03-03 | $17,903.55 | $10,450.00 | $7,453.55 |
| 013502 | 2012-03-03 | $17,903.55 | $10,450.00 | $7,453.55 |
| 013503 | 2012-03-03 | $17,903.55 | $10,450.00 | $7,453.55 |
| 013504 | 2012-03-03 | $17,903.55 | $10,450.00 | $7,453.55 |
| 013505 | 2012-03-03 | $17,903.55 | $10,450.00 | $7,453.55 |
| 013506 | 2012-03-03 | $17,903.55 | $10,450.00 | $7,453.55 |
| 013507 | 2012-03-03 | $17,903.55 | $10,450.00 | $7,453.55 |
| 013509 | 2012-03-03 | $17,903.55 | $10,450.00 | $7,453.55 |
| 013510 | 2012-03-03 | $17,903.55 | $10,450.00 | $7,453.55 |
| 013632 | 2012-03-03 | $17,903.55 | $10,450.00 | $7,453.55 |
| 327818 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 327819 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328808 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328809 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328950 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328951 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328952 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328953 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328954 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328955 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328956 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328957 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328958 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328959 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328960 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328961 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |

| | | | | |
|---|---|---|---|---|
| 328962 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328963 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328964 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328965 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328966 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328967 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328968 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328969 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328970 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328971 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328972 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 328973 | 2012-03-03 | $11,657.55 | $10,156.41 | $1,501.14 |
| 338350 | 2012-03-03 | $17,903.55 | $10,450.00 | $7,453.55 |
| 338351 | 2012-03-03 | $17,903.55 | $10,450.00 | $7,453.55 |
| 338352 | 2012-03-03 | $17,903.55 | $10,450.00 | $7,453.55 |
| 343466 | 2012-03-03 | $11,657.55 | $9,153.12 | $2,504.43 |
| 343467 | 2012-03-03 | $11,657.55 | $9,153.12 | $2,504.43 |
| 343468 | 2012-03-03 | $11,657.55 | $9,153.12 | $2,504.43 |
| 343469 | 2012-03-03 | $11,657.55 | $9,153.12 | $2,504.43 |
| 343470 | 2012-03-03 | $11,657.55 | $9,153.12 | $2,504.43 |
| 343471 | 2012-03-03 | $11,657.55 | $9,153.12 | $2,504.43 |
| 343472 | 2012-03-03 | $11,657.55 | $9,153.12 | $2,504.43 |
| 343473 | 2012-03-03 | $11,657.55 | $9,153.12 | $2,504.43 |
| 343474 | 2012-03-03 | $11,657.55 | $9,153.12 | $2,504.43 |
| 343476 | 2012-03-03 | $11,657.55 | $9,153.12 | $2,504.43 |
| 343477 | 2012-03-03 | $11,657.55 | $9,153.12 | $2,504.43 |
| 343478 | 2012-03-03 | $11,657.55 | $9,153.12 | $2,504.43 |
| 343479 | 2012-03-03 | $11,657.55 | $9,153.12 | $2,504.43 |
| 345538 | 2012-03-03 | $11,657.55 | $9,153.12 | $2,504.43 |
| 345539 | 2012-03-03 | $11,657.55 | $9,153.12 | $2,504.43 |
| 345540 | 2012-03-03 | $11,657.55 | $9,153.12 | $2,504.43 |
| 387464 | 2012-03-03 | $11,657.55 | $3,400.00 | $8,257.55 |
| 387465 | 2012-03-03 | $11,657.55 | $3,400.00 | $8,257.55 |
| 387466 | 2012-03-03 | $11,657.55 | $3,400.00 | $8,257.55 |
| 388116 | 2012-03-03 | $11,657.55 | $3,400.00 | $8,257.55 |
| 388117 | 2012-03-03 | $11,657.55 | $3,400.00 | $8,257.55 |
| 388118 | 2012-03-03 | $11,657.55 | $3,400.00 | $8,257.55 |
| 388119 | 2012-03-03 | $11,657.55 | $3,400.00 | $8,257.55 |
| 397194 | 2012-03-03 | $11,657.55 | $3,400.00 | $8,257.55 |
| 397195 | 2012-03-03 | $11,657.55 | $3,400.00 | $8,257.55 |
| 427245 | 2012-03-03 | $11,657.55 | $1,800.00 | $9,857.55 |

**Back** **New Search** **Print**

**Current time:** 14 Oct 2011, 4:41:40 PM

## Search

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### Mineral Claim Detail

**Right Number** 1839
**Claim Type** Mineral
**Claim Sub Type** Claim
**Title Type** Mineral Exploration Agreement
**Expiry Date** 2011-10-27
**Issue Date** 1984-10-27
**Claim Name** Taylor Brook
**Status** Active
**NTS Sheet**
**Owners** 13727 STRATABOUND MINERALS CORP. 100%

**Claim Events:**

| Submitter | Event | Effective Date |
|---|---|---|
| 12193 | Report of Work ---265 | 2010-03-09 |
| 12193 | Renewal ---266 | 2010-03-09 |

**Number of Units** 76
**Work Applied** $6,291.96
**Work Required** $0.00
**Excess Work** $6,291.96
**Units:**

| Unit Id | Expiry Date | Total Applied | Work Required | Excess Work |
|---|---|---|---|---|
| 1422001A | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422001B | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422001C | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422001D | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422001E | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422001F | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422001G | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422001H | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422001I | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422001J | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422001K | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422001L | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422001M | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422001N | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422001O | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422001P | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422002A | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422002B | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422002C | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422002D | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422002E | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422002F | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422002G | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422002H | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422002I | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422002J | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422002K | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422002L | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422002M | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422002N | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422002O | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422011A | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422011B | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422011G | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422011H | 2011-10-27 | $82.79 | $0.00 | $82.79 |

| | | | | |
|---|---|---|---|---|
| 1422011I | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422011J | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422011K | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422011L | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422011M | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422011N | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422011O | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422011P | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422012A | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422012B | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422012C | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422012D | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422012E | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422012F | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422012G | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422012H | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422012I | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422012J | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422012K | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422012L | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422012P | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422021P | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422022A | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422022H | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1422022I | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1423091D | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1423091E | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1423091F | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1423091G | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1423091J | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1423091K | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1423091L | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1423091M | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1423091N | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1423092C | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1423092D | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1423092E | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1522010M | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1522010N | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1522010O | 2011-10-27 | $82.79 | $0.00 | $82.79 |
| 1522020P | 2011-10-27 | $82.79 | $0.00 | $82.79 |

**Back** **New Search** **Print**

## Search

**Current time:** 14 Oct 2011, 4:46:04 PM

**Back** **New Search** **Print**

**Mineral Claim Detail**

**Right Number** 5354
**Claim Type** Mineral
**Claim Sub Type** Claim
**Title Type** Mineral Claim Group
**Expiry Date** 2012-02-22
**Issue Date** 2008-02-22
**Claim Name** Captain East Ext.
**Status** Active
**NTS Sheet**
**Owners** 13727 STRATABOUND MINERALS CORP. 100%

**Claim Events:**

| Submitter | Event | Effective Date |
|---|---|---|
| Joe MacIntosh | Reinstatement ---186 | 2010-03-03 |
| 12193 | Renewal ---188 | 2010-03-03 |
| 12193 | Renewal ---3543 | 2011-02-09 |

**Number of Units** 30
**Work Applied** $19,098.00
**Work Required** $13,500.00
**Excess Work** $5,598.00

**Units:**

| Unit Id | Expiry Date | Total Applied | Work Required | Excess Work |
|---|---|---|---|---|
| 427246 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427247 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427248 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427274 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427275 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427276 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427277 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427278 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427279 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427280 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427281 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427282 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427283 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427284 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427285 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427286 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427287 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427288 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427289 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427290 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427291 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427292 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427293 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427294 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427295 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427296 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427297 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 427298 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 436356 | 2012-02-22 | $636.60 | $450.00 | $186.60 |
| 436357 | 2012-02-22 | $636.60 | $450.00 | $186.60 |

**Back** **New Search** **Print**

# APPENDIX C

## CAPTAIN DRILLING DOCUMENTS

**Stratabound Minerals Corp.** Captain Project Drill Collars

| Hole ID | Hole Azimuth | Dip | Length (m) | Grid X (m) | Grid Y (m | Elevation (m) | Date |
|---|---|---|---|---|---|---|---|
| CP-07-01 | 110 | -88 | 215 | -250.8 | 120.42 | 148.54 | 2007 |
| CP-07-02 | 115 | -86 | 179 | -240.7 | 120.25 | 148.58 | 2007 |
| CP-07-03 | 156 | -88 | 144 | -228.21 | 119.38 | 148.16 | 2007 |
| CP-07-04 | 121 | -86 | 230 | -251.54 | 100.62 | 148.3 | 2007 |
| CP-07-05 | 138 | -88 | 101 | -210.54 | 120.97 | 147.46 | 2007 |
| CP-07-06 | 130 | -85 | 134 | -221.27 | 91.55 | 147.56 | 2007 |
| CP-07-07 | 111 | -45 | 71 | -208.83 | 93.56 | 147.64 | 2007 |
| CP-07-08 | 111 | -45 | 98 | -210.47 | 75.86 | 147.4 | 2007 |
| CP-07-09 | 111 | -45 | 95 | -228.21 | 119.38 | 148.16 | 2007 |
| CP-07-10 | 111 | -45 | 60.2 | -236.43 | 146.83 | 148.43 | 2007 |
| CP-07-11 | 161 | -45 | 125 | -211.9 | 73.82 | 147.44 | 2007 |
| CP-07-12 | 111 | -60 | 101 | -243.5 | 77.89 | 147.91 | 2007 |
| CP-08-13 | 110 | -88 | 297 | -250 | 75 | 148.3 | 2008 |
| CP-08-14 | 110 | -88 | 225 | -250.32 | 145.5 | 148.52 | 2008 |
| CP-08-15 | 110 | -88 | 311 | -299.3 | 117.74 | 150.62 | 2008 |
| CP-08-16 | 110 | -88 | 284 | -250 | 50.8 | 147.57 | 2008 |
| CP-08-17 | 110 | -70 | 171 | -214.6 | 96 | 147.3 | 2008 |
| CP-08-18 | 110 | -88 | 287 | -272 | 155 | 149.33 | 2008 |
| CP-08-19 | 110 | -88 | 311.6 | -278.8 | 94.9 | 148.37 | 2008 |
| CP-08-20 | 110 | -88 | 368 | -303.4 | 89.1 | 151.01 | 2008 |
| CP-08-21 | 110 | -88 | 326 | -275 | 120 | 148.55 | 2008 |
| CP-08-22 | 110 | -88 | 365 | -314 | 65 | 152.11 | 2008 |
| CP-08-23 | 110 | -70 | 115 | -214.6 | 90.3 | 147.3 | 2008 |
| CP-08-24 | 110 | -88 | 299 | -280.3 | 71.3 | 149.02 | 2008 |
| CP-08-25 | 110 | -70 | 185 | -214.9 | 52.4 | 147.38 | 2008 |
| CP-09-26 | 90 | -88 | 534 | -375.67 | 90 | 151.55 | 2009 |
| CP-10-27 | 90 | -88 | 506 | -381.81 | 70 | 152.01 | 2010 |
| CP-10-28 | 90 | -88 | 326 | -341.81 | 120 | 151.53 | 2010 |
| CP-10-29 | 90 | -88 | 374 | -344.81 | 148 | 152.42 | 2010 |
| CP-10-30 | 90 | -88 | 471 | -365.13 | 120 | 151.72 | 2010 |

**Stratabound Minerals Corp** Captain Project Local Grid Control Points

| Control Point ID | Grid X (m) | Grid Y (m) | UTM NAD83 X (m) | UTM NAD83 Y (m) |
|---|---|---|---|---|
| 200701 | 250.8 | 120.42 | 282358.029 | 5240728.729 |
| 200702 | 240.7 | 120.25 | 282367.338 | 5240724.843 |
| 200703 | 228.21 | 119.38 | 282378.647 | 5240719.42 |
| 200704 | 251.54 | 100.62 | 282350.046 | 5240710.588 |
| 200705 | 210.54 | 120.97 | 282395.29 | 5240714.527 |
| 200706 | 221.27 | 91.55 | 282374.835 | 5240690.994 |
| 200707 | 208.83 | 93.56 | 282387.144 | 5240688.271 |
| 200708 | 210.47 | 75.86 | 282379.086 | 5240672.43 |
| 200709 | 228.21 | 119.38 | 282386.351 | 5240717.337 |
| 200710 | 236.43 | 146.83 | 282381.134 | 5240747.969 |
| 200711 | 211.9 | 73.82 | 282377.007 | 5240671.063 |
| 200712 | 243.5 | 77.89 | 282349.131 | 5240686.498 |
| 122 | 125 | 120 | 282474.874 | 5240682.61 |
| 130 | 200 | 100 | 282394.539 | 5240688.156 |
| 138 | 175 | 60 | 282410.712 | 5240655.629 |
| 139 | 150 | 60 | 282433.886 | 5240646.191 |
| 142 | 250 | 60 | 282341.265 | 5240684.026 |
| 155 | 275 | 120 | 282335.252 | 5240737.357 |
| 158 | 325 | 120 | 282288.769 | 5240755.703 |
| 159 | 350 | 120 | 282265.108 | 5240764.138 |
| 160 | 375 | 120 | 282241.884 | 5240773.378 |
| 161 | 350 | 210 | 282298.626 | 5240847.451 |
| 162 | 325 | 210 | 282321.697 | 5240838.255 |
| 163 | 300 | 210 | 282344.749 | 5240828.878 |
| 164 | 275 | 210 | 282368.22 | 5240820.327 |
| 165 | 250 | 210 | 282391.41 | 5240811.357 |
| 166 | 175 | 210 | 282461.346 | 5240784.589 |
| 167 | 150 | 210 | 282484.814 | 5240776.209 |
| 168 | 125 | 210 | 282507.79 | 5240767.236 |
| 169 | 400 | 180 | 282243.083 | 5240840.956 |
| 170 | 350 | 180 | 282289.575 | 5240822.427 |
| 171 | 325 | 180 | 282313.035 | 5240813.514 |
| 172 | 300 | 180 | 282336.029 | 5240803.792 |
| 173 | 275 | 180 | 282359.197 | 5240794.335 |
| 174 | 200 | 180 | 282427.76 | 5240766.869 |
| 175 | 175 | 180 | 282451.107 | 5240758.315 |
| 176 | 150 | 180 | 282474.144 | 5240748.64 |
| 177 | 400 | 150 | 282229.331 | 5240808.166 |
| 178 | 350 | 150 | 282276.526 | 5240790.632 |
| 179 | 325 | 150 | 282300.047 | 5240781.397 |
| 180 | 175 | 150 | 282439.835 | 5240727.785 |
| 181 | 150 | 150 | 282463.071 | 5240719.117 |
| 182 | 100 | 150 | 282509.517 | 5240701.149 |
| 183 | 75 | 150 | 282532.807 | 5240691.944 |
| 184 | 50 | 150 | 282555.916 | 5240683.047 |

| 2011 Proposed Drill Hole Locations for Captain Project - December 2010 | | | | | | |
|---|---|---|---|---|---|---|
| Hole # | Grid y (m) | Grid x (m) | Elevation z (m) | End Depth (m) | Azimuth | Dip |
| Prop-1 | 120 | -350 | 150 | 385 | 110 | -88 |
| Prop-2 | 90 | -405 | 150 | 500 | 110 | -88 |
| Prop-3 | 59 | -333 | 150 | 385 | 110 | -88 |
| Prop-4 | 165 | -236 | 148 | 60 | 110 | -45 |
| Prop-5 | 50 | -190 | 150 | 60 | 110 | -45 |
| Prop-6 | 25 | -190 | 147 | 100 | 110 | -45 |
| Prop-7 | 175 | -295 | 150 | 160 | 110 | -60 |
| TOTAL | | | | 1650 | | |

Note: See Report Figure 60 (page 97) for hole locations

# TECHNICAL REPORT ON A MINERAL RESOURCE ESTIMATE

# STRATABOUND MINERALS CORP. CAPTAIN PROPERTY

GLOUCESTER AND NORTHUMBERLAND COUNTIES
NEW BRUNSWICK, CANADA

Effective Date: October 29th, 2008

Chapter 13 Excerpt From Cullen and Harrington (2009)

Latitude 47° 17' 01''N
Longitude 65° 52' 30''W

Prepared For: Stratabound Minerals Corp.
Prepared By: Michael P. Cullen, Senior Geologist, P. Geo.
Matthew Harrington, Geologist
Mercator Geological Services Limited

mercator GEOLOGICAL SERVICES

## 13.3 Quality Control and Quality Assurance (QA/QC)

### *13.3.1 Stratabound 2007-2008 Program*

#### *13.3.1.1 Introduction*

Drill core sampling carried out by Stratabound during the 2007-2008 Captain program was subject to a Quality Control and Quality Assurance program administered by the company. This included submission of blind blank samples, duplicate split samples of quarter core, duplicate pulp splits, certified analytical standards and analysis of check samples at a third party commercial laboratory. Additionally, internal laboratory reporting of quality control and assurance sampling was monitored by Stratabound staff on an on-going basis during the course of the project. Details of programs noted above are presented below under separate headings.

#### *13.3.1.2 Certified Standard Samples*

Stratabound used four certified standards during the course of the 2007-2008 drilling program (CDN-HZ-2, CDN-HLLC, CDN-CGS-10 and CDN-WMS-1a), all of which were obtained from CDN Laboratories of Vancouver, BC. In total, 120 certified samples were submitted for analysis, with each sample consisting of a pre-packaged, prepared sample pulp weighing approximately 50 grams that was systematically inserted into the laboratory sample shipment sequence by Stratabound staff. Records of certified standard insertion were maintained as part of the core sampling and logging protocols and samples were submitted at a nominal frequency of one for every 35 samples submitted. Table 5 presents certified values and error ranges for the four standards used during the 2007-2008 program.

**Table 5: Certified Standard Tabulation**

| Standard | Certified Cu (%) | Certified Au (g/t) | Certified Co (%) |
|---|---|---|---|
| CDN-HZ-2 | 1.36% ± 0.06% | 0.124 g/t ± 0.024 g/t | NA |
| CDN-HLLC | 1.49 % ± 0.06% | 0.83 g/t ±0.12 g/t | NA |
| CDN-CGS-10 | 1.55% ± 0.07% | 1.73 g/t ± 0.15 g/t | NA |
| CDN-WMS-1a | 1.396% ± 0.014% | NA | 0.145 ± 0.002 |

Table 5 shows that three standards provide coverage for copper and gold values whereas one standard provides coverage for copper and cobalt. Provisional values for various other metals are also provided but are not pertinent to this report.

*Results for CDN-HZ-2*

In total, 36 samples of the CDN-HZ-2 standard were analyzed and compiled results of this work for Cu and Au are presented below in Figure 14 (Cu) and Figure 15 (Au). Cu results are grouped about the lower certified error range limit for the standard and define a consistent low bias factor of approximately 0.06% Cu when considered against the accepted mean value for the standard. This approximates under-reporting of the average copper grade by a factor of approximately 4.4%. Au results are similarly grouped around the lower error limit for the standard and define an average low bias factor of approximately 0.02 g/t when considered against the accepted mean value for the standard. The reason for under-reporting of the metal grades is not clear, but comparable trends are not present in results for the other standards used during the drilling program. Notably, all reported values fall within the total range of values reported for the sample materials in support documentation supplied by CDN Laboratories Ltd. The common trend of under-reporting for both metals and lack of a comparable trend in results for the other certified standards used for the 2007-2008 program suggests that the source of observed variation may lie in composition of sample material itself. However, for both Cu and Au values, variation from certified mean values occurs at levels that would not have significant economic impact on resource grades. No systematic change in grade levels in the dataset is notable against time. .

*Results for CDN-HLLC*

In total, 33 samples of the CDN-HLLC standard were analyzed and compiled results of this work are presented for Cu and Au below in Figure 16 (Cu) and Figure 17 (Au). Cu results group closely around the mean value and show no systematic trend variations through time. Values are closely grouped about the certified mean of 1.49% ± 0.06% Cu and only one value plots outside sample error limits. The Cu result for this sample exceeded the standard's upper error limit by 0.05% Cu but is not considered a substantial data quality issue. Au results are similarly grouped about the certified value of 0.83 g/t ± 0.12g/t and only one value falls outside the error limits. This sample returned a value that was 0.34 g/t lower than the lower error limit for the standard but is not supported by similarly anomalous results in either Cu or Co. No explanation for the lower than expected result is apparent.

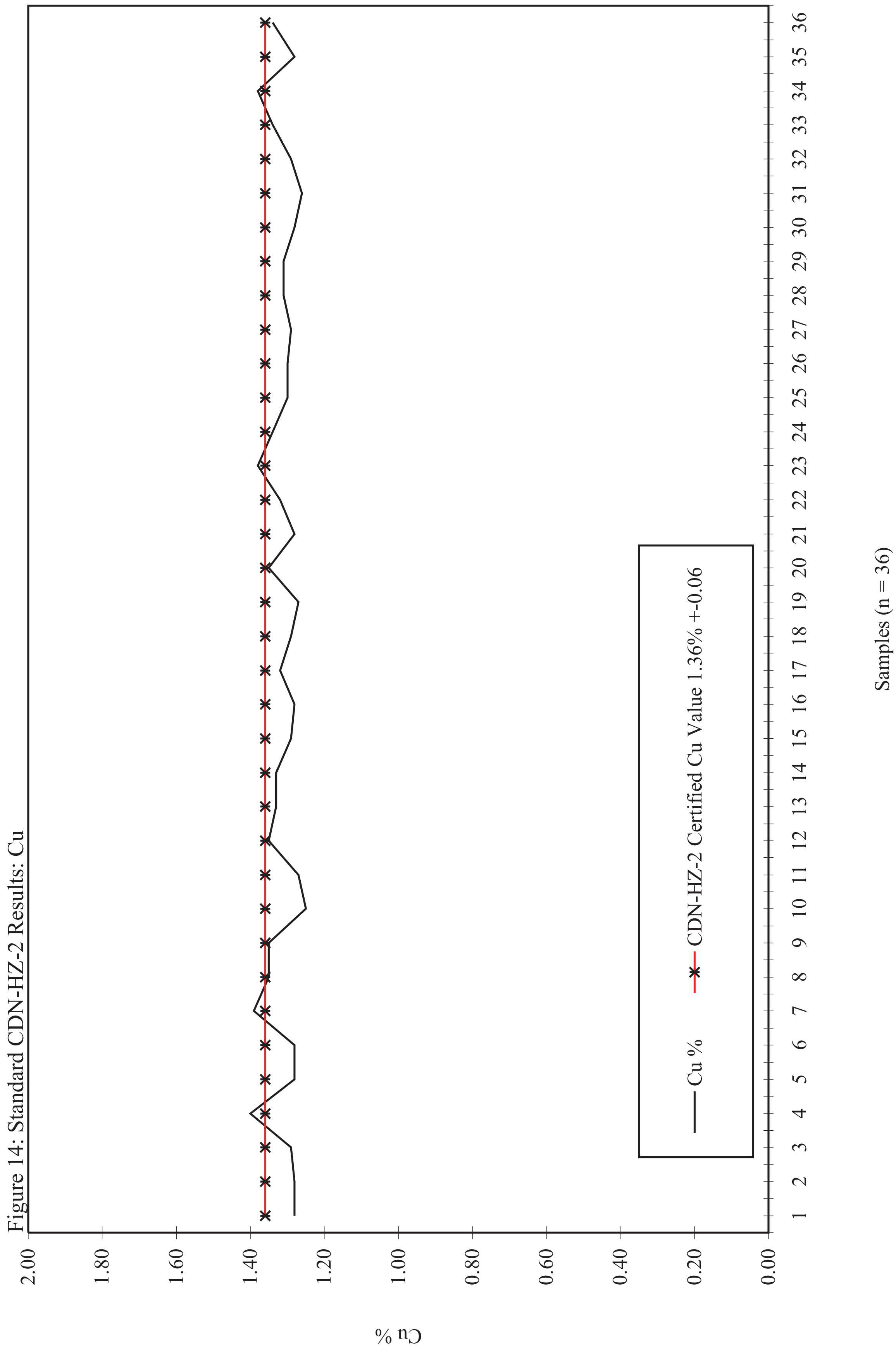
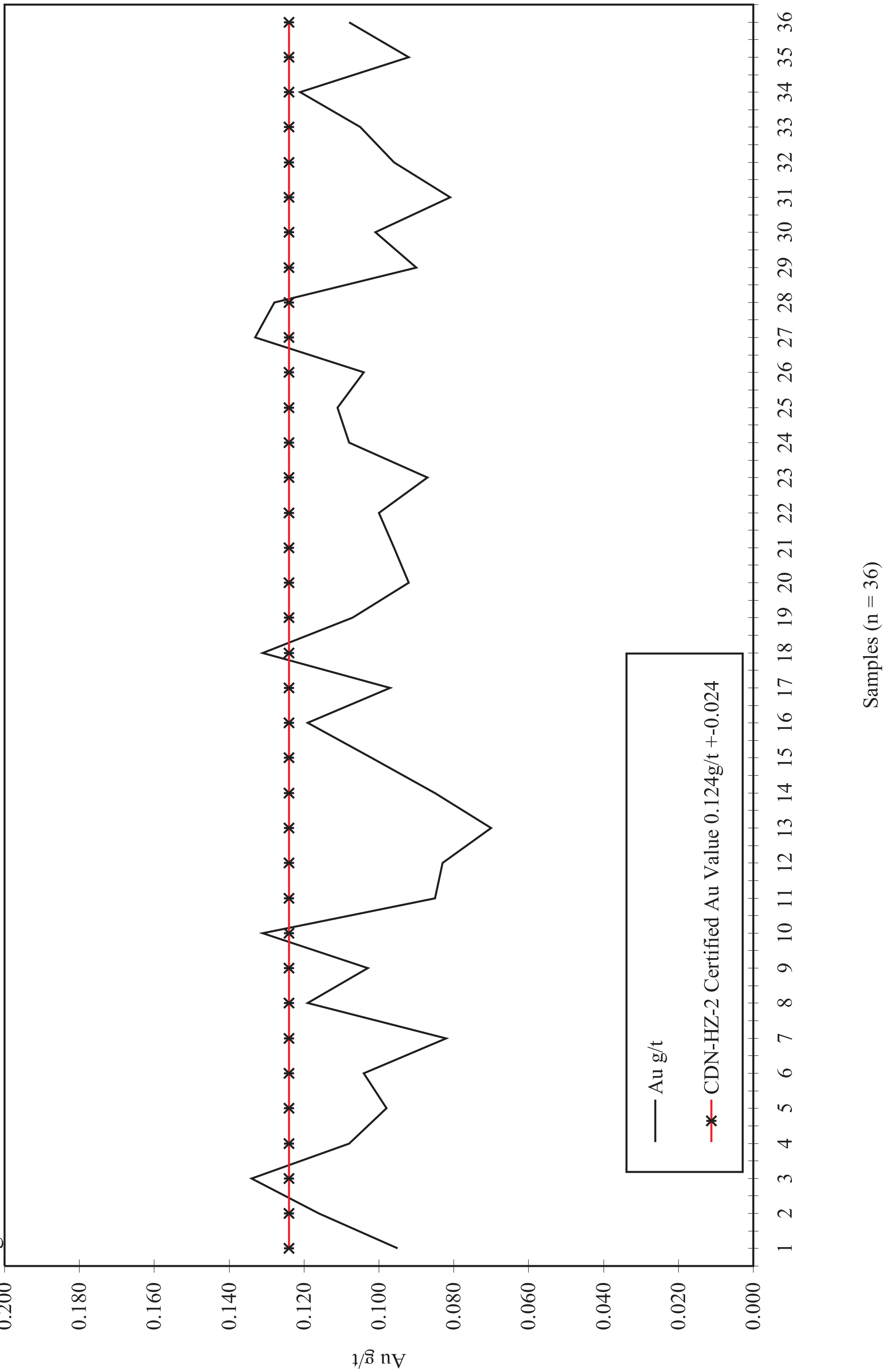



Figure 14: Standard CDN-HZ-2 Results: Cu

Figure 15: Standard CDN-HZ-2 Results: Au



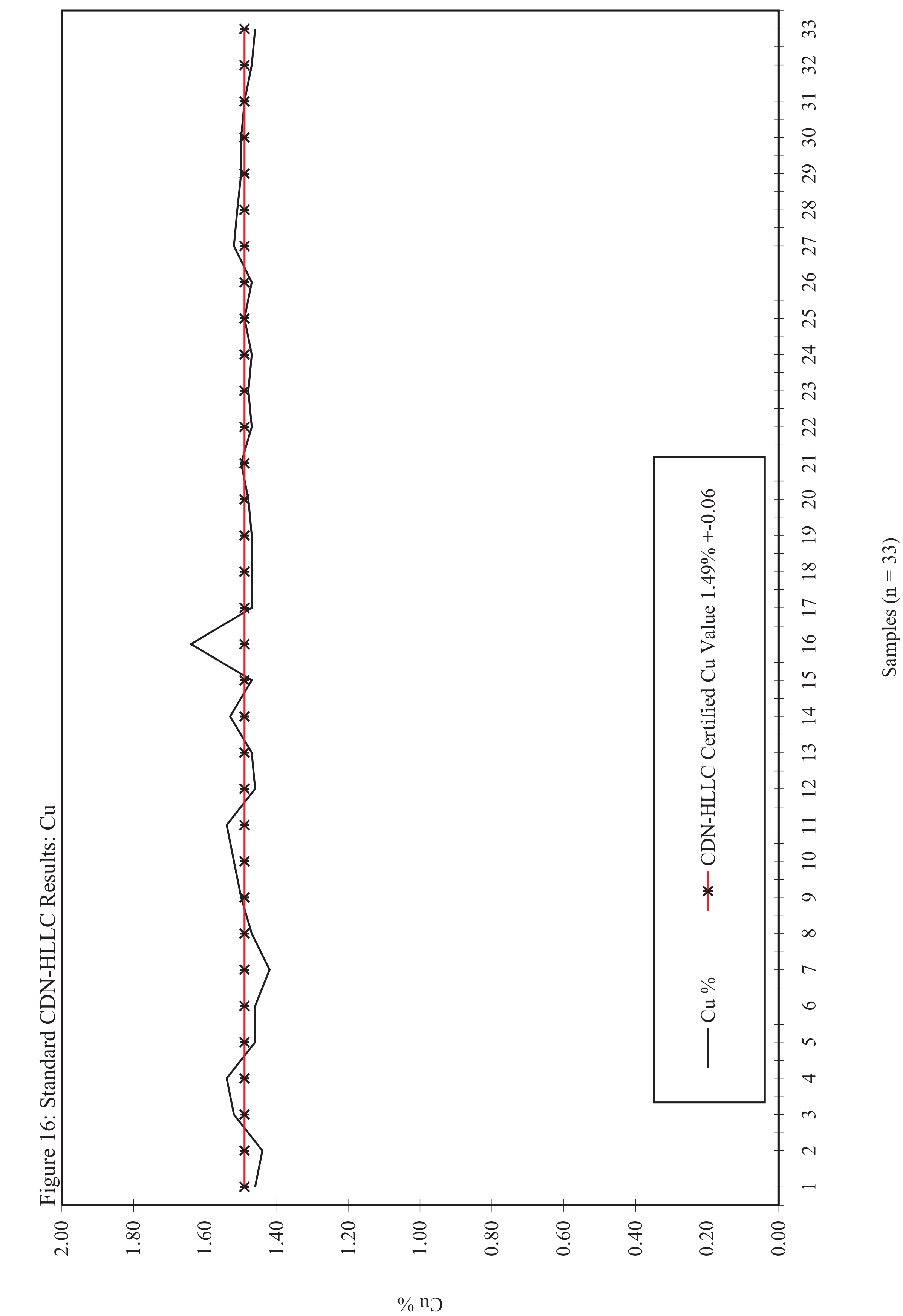



Figure 16: Standard CDN-HLLC Results: Cu

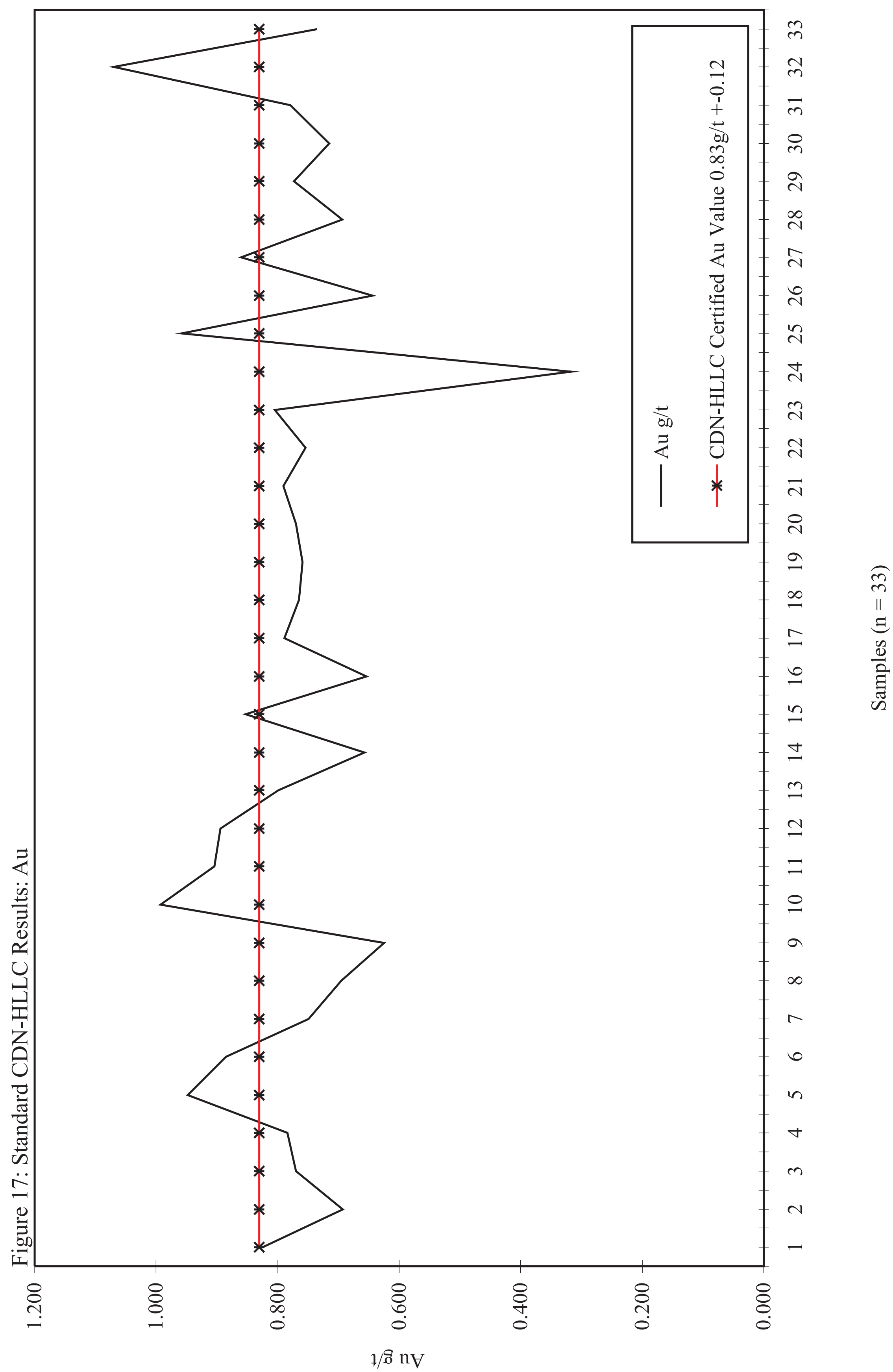



Figure 17: Standard CDN-HLLC Results: Au

*Results for CGS-10*

In total, 42 samples of the CGS-10 standard were analyzed and compiled results of this work for Cu and Au are presented below in Figure 18 (Cu) and Figure 19 (Au). Au results were returned for only 33 of the standards submitted, but no explanation for this was present at the report date. Cu results for the first 14 drill holes group around the mean certified value of 1.55% Cu and analytical services for these samples were provided by SGS. Values for the remaining 11 drill holes show a clear tendency to group around the standard's lower error limit of 1.48% Cu and Eastern Analytical provided laboratory services for these drill holes. This defines a background shift between laboratories in the order of 0.05 % Cu to 0.07% Cu and a low bias tendency in results for the last 11 drill holes. Au results for the CGS-10 standard report close to the lower error limit of 1.58 g/t and samples from the last 11 drill holes that were analyzed by Eastern Analytical Limited and define a lower trend than those from the first 14 drill holes analyzed by SGS. These results define a case of under-reporting of Au values when considered against the certified standard value of 1.73g/t ±0.07g/t. An explanation for the low Cu bias may lie in differing sample digestion protocols used by the labs. The SGS peroxide fusion procedure used in ore grade analysis may liberate more metal than the multi-acid approach used by Eastern Analytical. However, Fire Assay pre-concentration and analytical procedures for Au determination are comparable. Consistency of results within each laboratory population suggests that a constant factor separates the two.

*Results for WMS-1a*

This standard was selected due to its certified Co and Cu values, but was only used in the last 7 drill holes of the project. As a result, only 8 sets of Cu and Co analytical results were obtained and these are presented in Figure 20(Cu) and Figure 21 (Co). All analytical work was completed at Eastern Analytical Limited and Cu values show a consistent low bias that exceeds the 1.382% Cu lower error margin of the sample. The average Cu value returned was 1.320% which is 0.076% below the certified mean value. However, all values fall within 10% of the certified mean Cu value of 1.396% ± 0.014%. In contrast to the Cu trend, Co values show a positive bias and consistently plot above the certified mean value. With the exception of two samples, all results fall within ± 10% of the mean value and the two exceptions exceed the 10% value by 0.045% or less. The average Co value returned for the 8 samples is 0.154%, which is approximately 6% higher than the certified mean value of 0.145% ± 0.002%. As in the previous case, no obvious explanation exists for the variation trends noted for Cu and Co. However, the generally systematic nature of bias for each metal indicates an origin associated with contrasting digestion methods or instrumental factors.

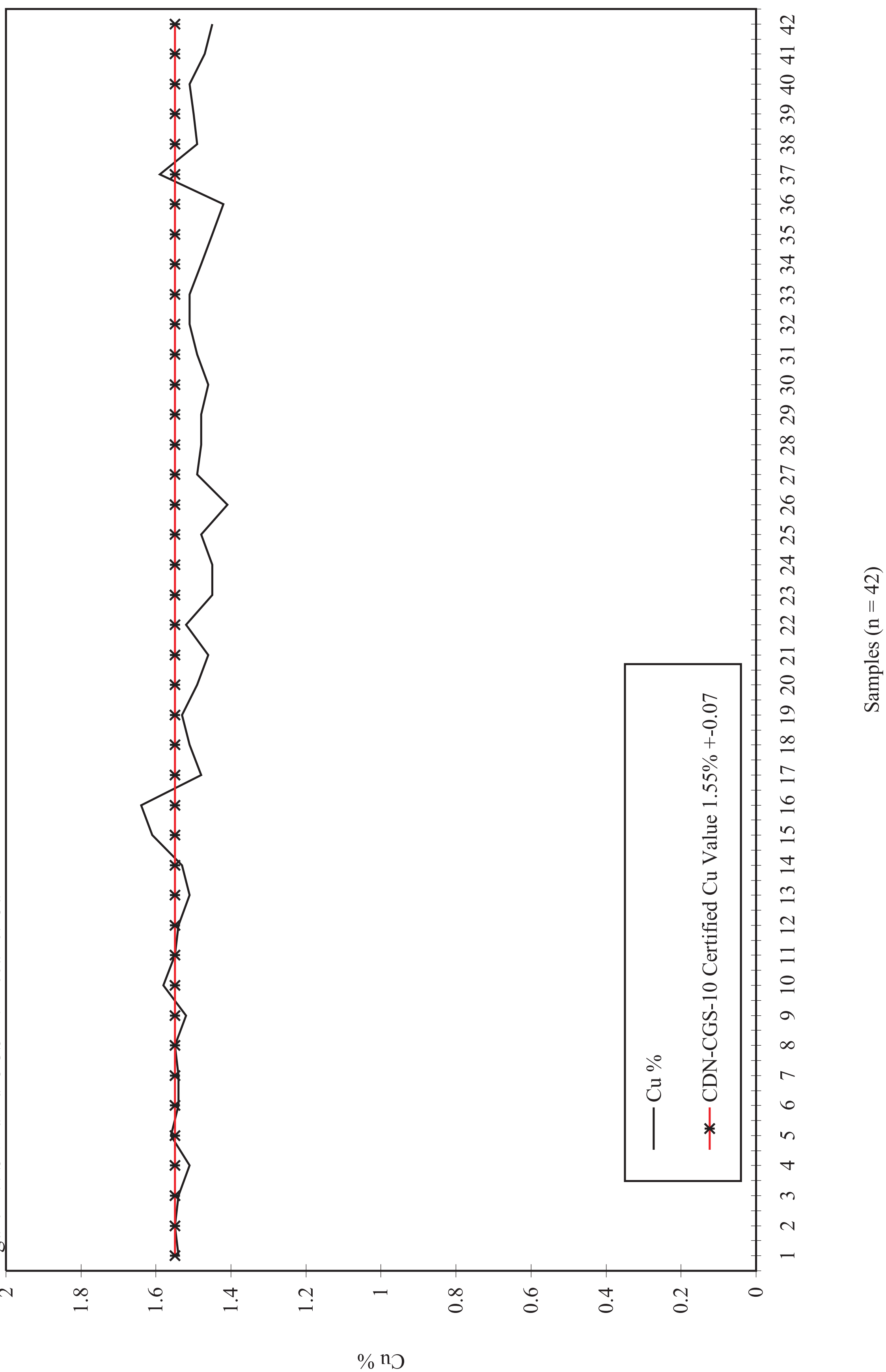



Figure 18: Standard CGS-10 Results: Cu

Figure 19: Standard CGS-10 Results: Au

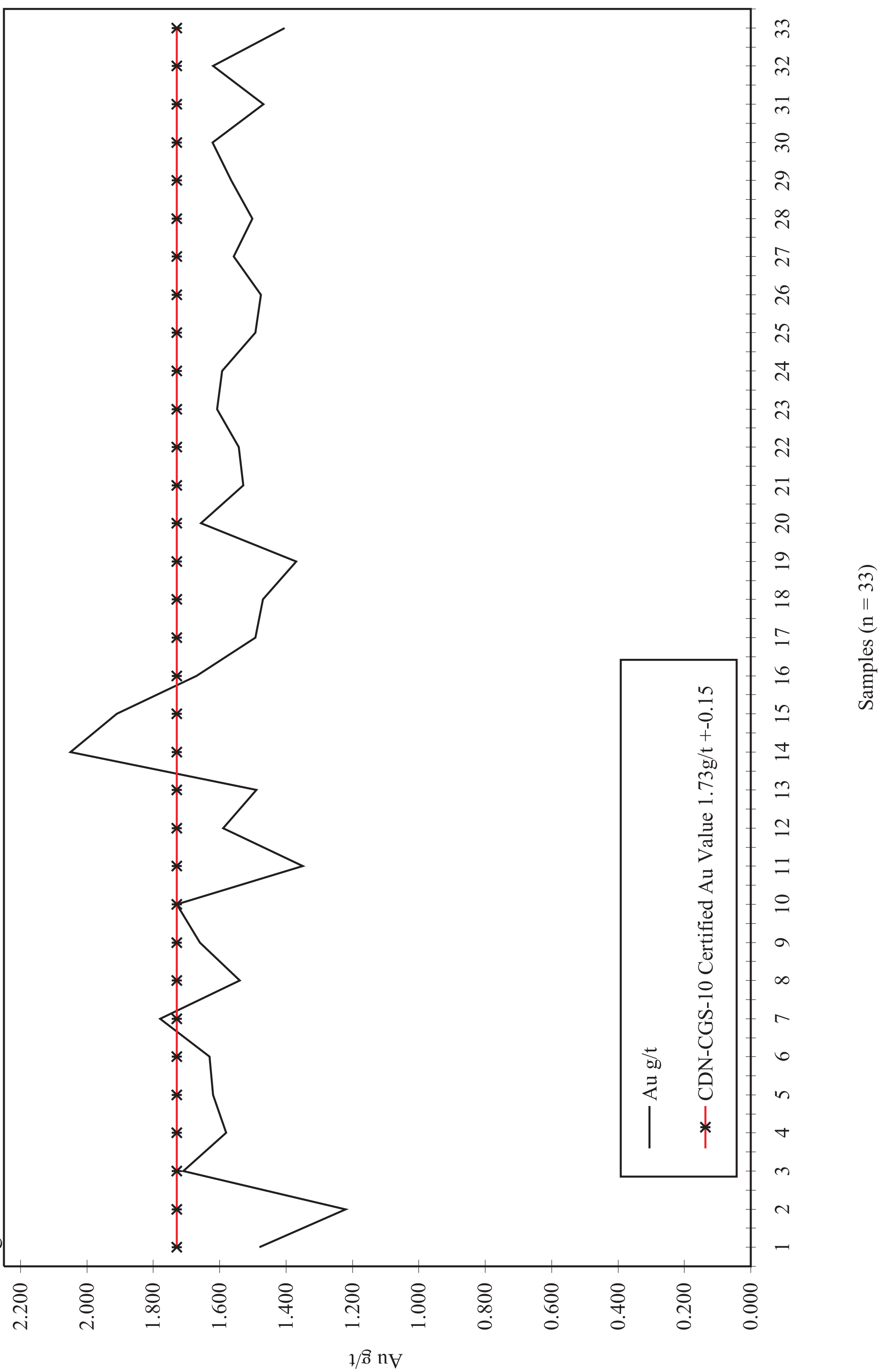
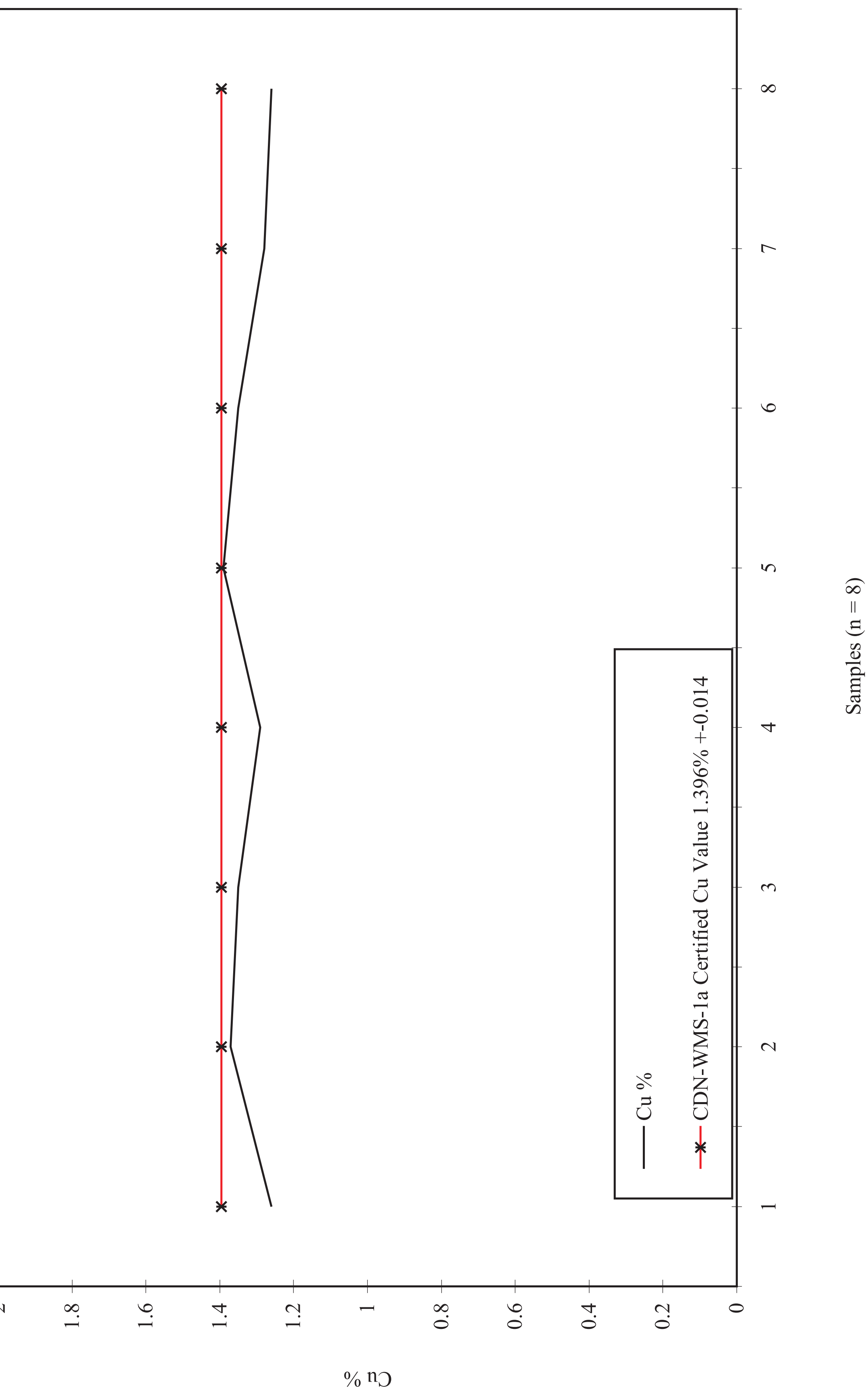


Figure 20: Standard WMS-1a Results: Cu



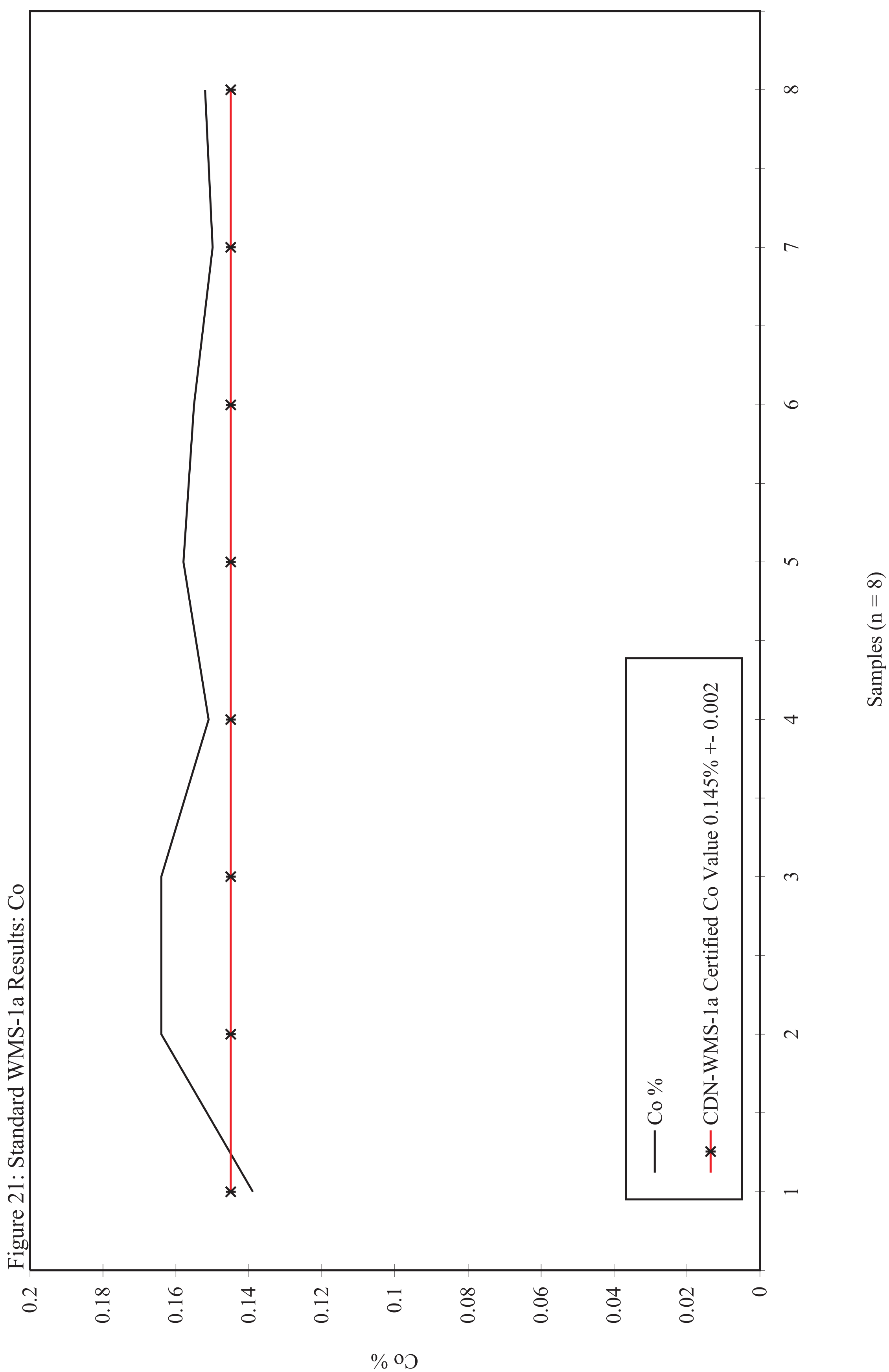


Figure 21: Standard WMS-1a Results: Co

### 13.3.1.3 Blind Blank Samples

*Introduction*

Blank samples of comparable weight to normal 0.5 meter half core samples were systematically inserted into the laboratory sample stream by Stratabound staff, with 222 such samples submitted during the 2007-2008 program. This approximates an insertion rate of 1 blank per 20 core samples submitted. Two blank sample lithologies were used by Stratabound, these being (1) visibly non-mineralized gabbro drill core from a local intrusion and (2) calcareous siltstone drill core from Stratabound's Elmtree gold project, located approximately 24 kilometers northwest of Bathurst. Company records do not systematically identify lithology of inserted blank samples but from discussions with Stratabound staff it was determined that gabbro samples were used in the early part of the drilling program, being replaced later by the calcareous siltstone.

Separate geochemical signatures would be expected for the differing blank sample lithologies and to aid in identifying such signatures all data were first ordered in time sequence of drilling and metal levels considered. The anticipated result was that gabbro samples would show higher Cu and Co background levels than those of the calcareous siltstone and thereby provide distinction between lithology groups. Review of metal distribution graphs showed that Cu values average 300 ppm in the first 7 drill holes and are followed by a less well-defined population averaging 97 ppm that continues to near the end of drill hole CP-08-17. Remaining holes to CP-08-25 are marked by still lower Cu values, averaging 53 ppm. These results are interpreted as indicating that two geochemically distinct groups of gabbroic material had been used, followed by the calcareous siltstone. Co distribution shows a break at the end of drill hole CP-08-016 sampling, where higher Co values averaging 80 ppm and 50 ppm that mark the gabbro association are seen to abruptly drop to values in the 10 ppm to 25 ppm range that mark calcareous siltstone samples. Based on these results, three background levels for Cu, Co and Au within the blank sample population are recognized in the following discussions.

*Blank Sample Cu Results*

The average Cu value of blank samples is 136 ppm with an analytical detection limit of 0.5 ppm applicable to ALS Chemex data and 1.0 ppm applicable to Eastern Analytical data. Three Cu domains are present and represent the two gabbro and one calcareous siltstone lithology suites used by Stratabound. Samples in the first gabbro domain have a mean Cu value of 300 ppm and those in the second have a mean value of 97 ppm. In contrast, the mean Cu value for calcareous siltstone is 53 ppm. Grade spikes occur in all three domains and generally exceed background values by 100 ppm to 300 ppm. Since Cu at the levels present, including spike highs, would not be unusual in any of the blank material lithologies, it is difficult to distinguish natural Cu variation from that resulting from possible cross-contamination related to sample preparation.

Taken as a whole, Cu results are interpreted as being somewhat difficult to interpret but not indicating presence of a significant and systematic contamination issue in the core sample dataset. Cu results are presented in Figure 22.

*Blank Sample Au Results*

The average Au value returned for blank samples is 9.3 parts per billion (ppb) with an analytical detection limit of 1 ppb applicable to ALS Chemex data and 5 ppb applicable to the Eastern Analytical dataset. Three samples returned values exceeding 51 ppb and of these only one exceeded 101 ppb, this being the maximum value reported of 324 ppb. In addition to these outliers, elevated results in the 25 ppb to 55 ppb range mark short sequences of consecutive blank samples. These intervals may reflect higher background levels of Au in these core samples sourced from the Elmtree Au property, where structurally focused hydrothermal alteration has affected substantial volumes of country rock. Au values alone do not provide clear definition of the gabbro and siltstone sample populations. In context of potential cross-contamination impact on resource estimate grades, the Au content variation range represented in the blank sample data set would not have a significant impact on resource grades. Au results are presented in Figure 23.

*Blank Sample Co Results*

The average Co value of blank samples is 43.52 ppm with an analytical detection limit of 0.1 ppm applicable to ALS Chemex data and 1.0 ppm applicable to Eastern Analytical data. The three lithologic domains described earlier are clearly represented in the Co data set, with the first gabbro domain having a mean Co value of 82.1 ppm, the second having a mean value of 49.6 ppm and the calcareous siltstone averaging 17.3 ppm. The maximum value returned was 615 ppm and occurs as a single site spike in the first gabbro domain. The next highest values are single site spikes of 172 ppm and 208 ppm and occur in the second gabbro domain. Strongly anomalous Co results are not present in preceding or succeeding core samples, as might be expected in a case of simple preparation stream cross-contamination. With the exception of the

Figure 22: Blank sample results: Cu

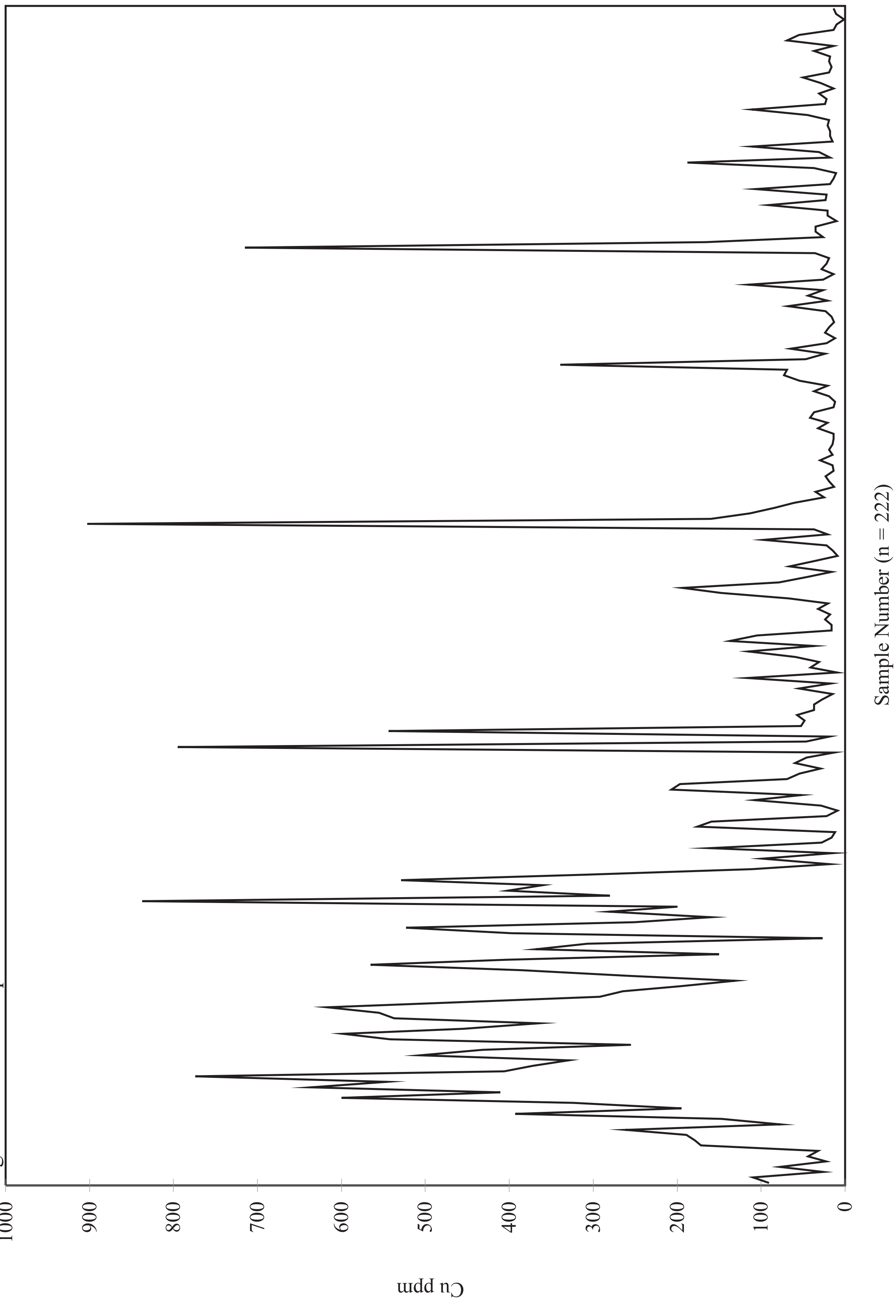
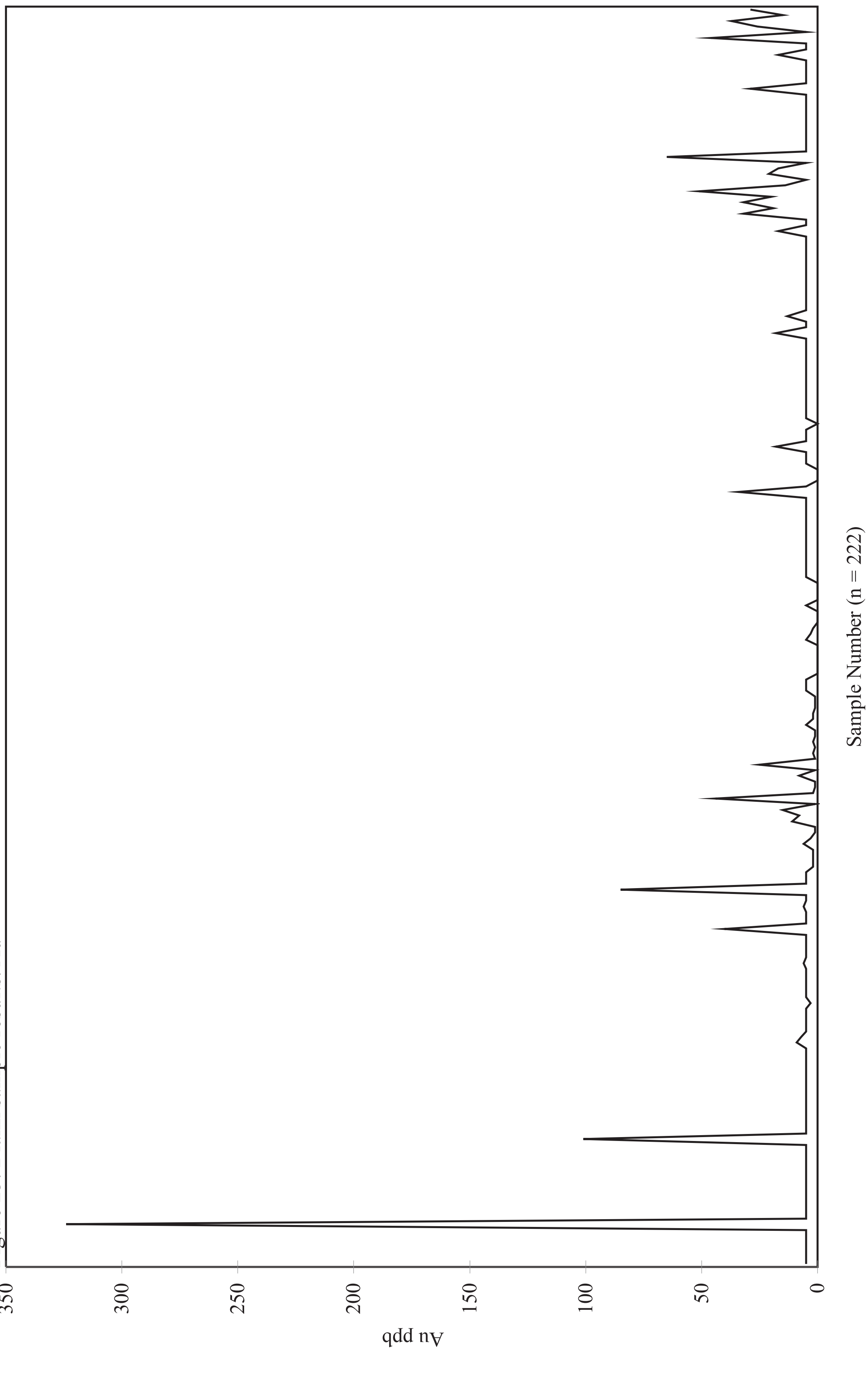


Figure 23: Blank sample results: Au



samples noted above, which account for 1.3% of the total dataset, other results range above and below the background values for each lithology domain on the order of a few tens of ppm. Based on these trends, Co results for the blank sample population are not interpreted as indicating presence of problematic cross contamination in the dataset. In an isolated instance such as that of the 615 ppm sample, it is difficult to determine whether this value represents an anomalously mineralized gabbro sample or contamination encountered during sample preparation. Co results are presented in Figure 24.

#### *13.3.1.4 Pulp Duplicate Sample Splits*

Splits of coarse reject material from the initial core sample preparation stream core were prepared for analysis as duplicate splits at a nominal frequency of every 25$^{th}$ sample submitted. In total, results for 189 pairs were returned. Program results for Cu, Au and Co are presented in Figure 25 (Cu), Figure 26 (Au) and Figure 27 (Co) and data pairs for the three metals support correlation coefficients of 0.987 (Cu), 0.983 (Au) and 0.984 (Co). The Au coefficient reflects removal of a single outlier sample pair of 2190 ppb vs 258 ppb. If this outlier pair is included, the Au coefficient drops to 0.870. Based on these results, precision of these and associated data set samples is considered acceptable for resource estimation purposes. No clear explanation for the outlier is apparent but it is clearly atypical of the remaining data set.

#### *13.3.1.5 Quarter Core Duplicate Samples*

In addition to analysis of duplicate splits of core sample pulps, Stratabound carried out a program of quarter core sampling to check on variation of results between half core sample components. In total, 105 samples were reviewed by Mercator and results for Cu, Au and Co are presented below in Figure 28 (Cu), Figure 29 (Au) and Figure 30(Co). Correlation coefficients for Cu, Au and Co are 0.94, 0.93 and 0.91 respectively and plotted results show that agreement between sample pairs varies with both grade level and metal. Co results show best correlation between samples and this is interpreted to reflect its known association with fine grained pyrite mineralization that is homogenously distributed at the sample scale. In contrast, Au and Cu results show poorer correlation between samples and are interpreted to reflect greater spatial variability in style and distribution of higher grade stringer sulphides at the scale of a 0.5 meter core sample. This defines a higher nugget effect than that seen for Co. The quarter core sample set also shows generally lower Au and Cu grades in the in the 0.5% to 1% Cu range but the few samples present above 1% Cu show better agreement.

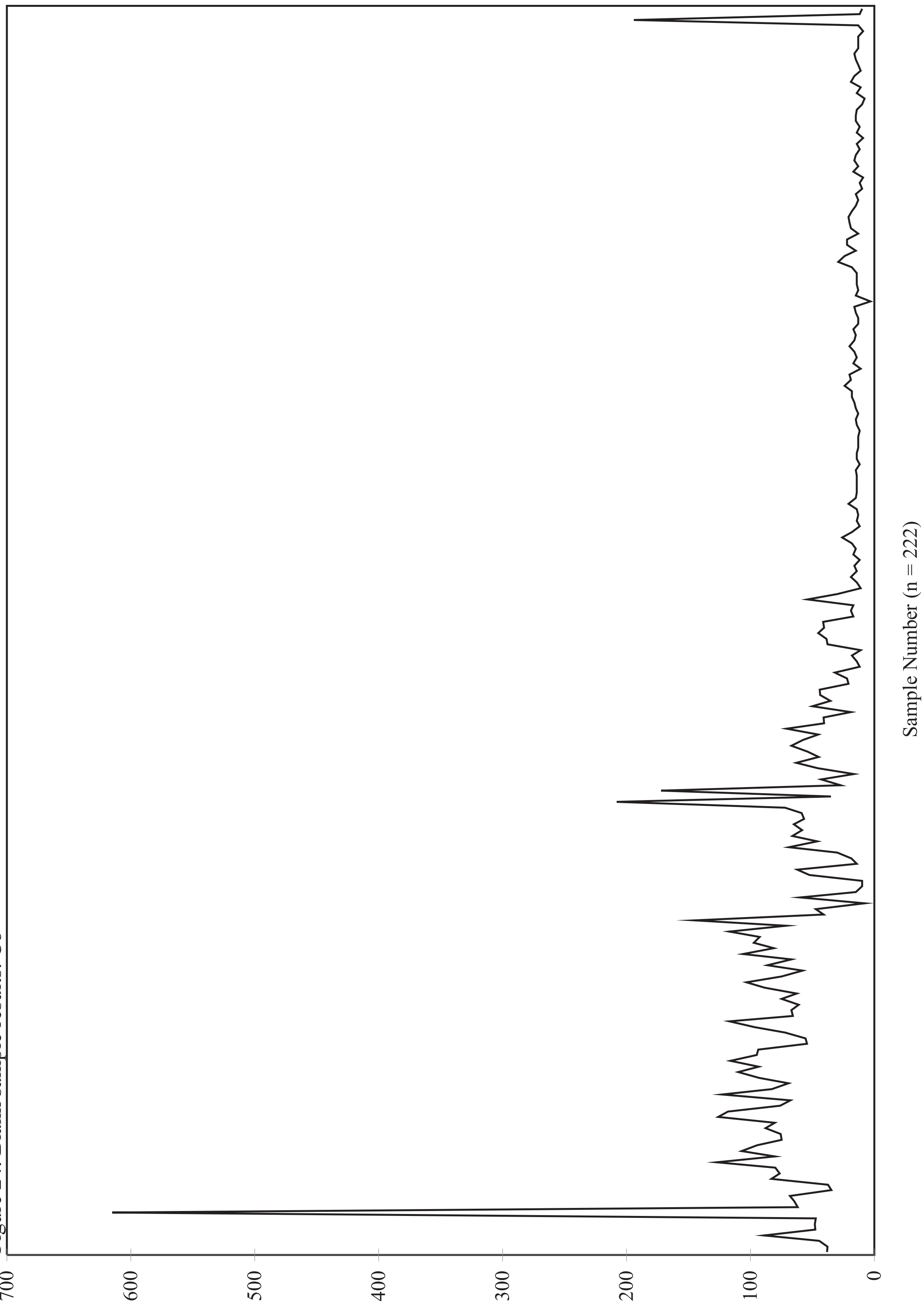



Figure 24: Blank sample results: Co

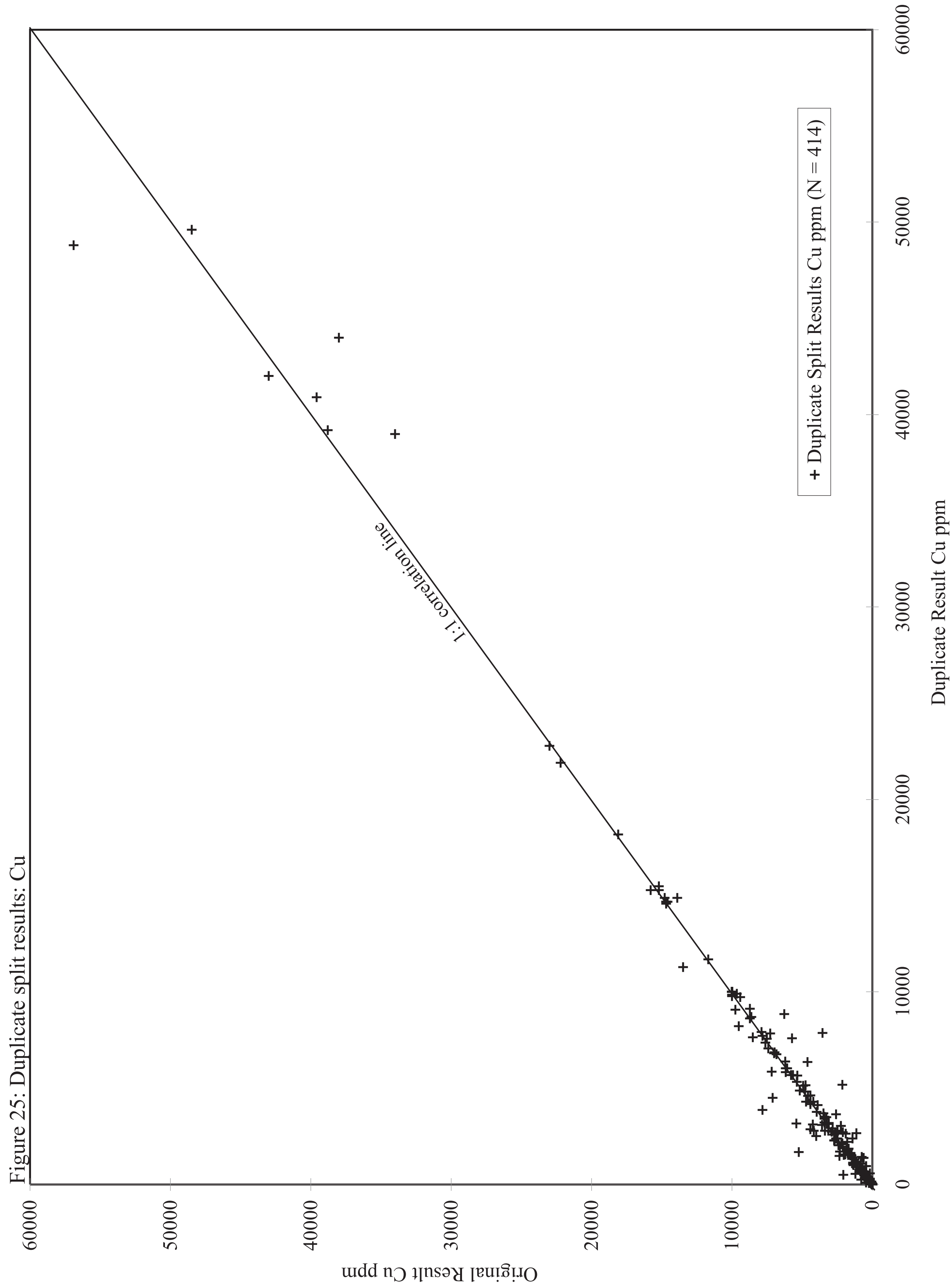



Figure 25: Duplicate split results: Cu

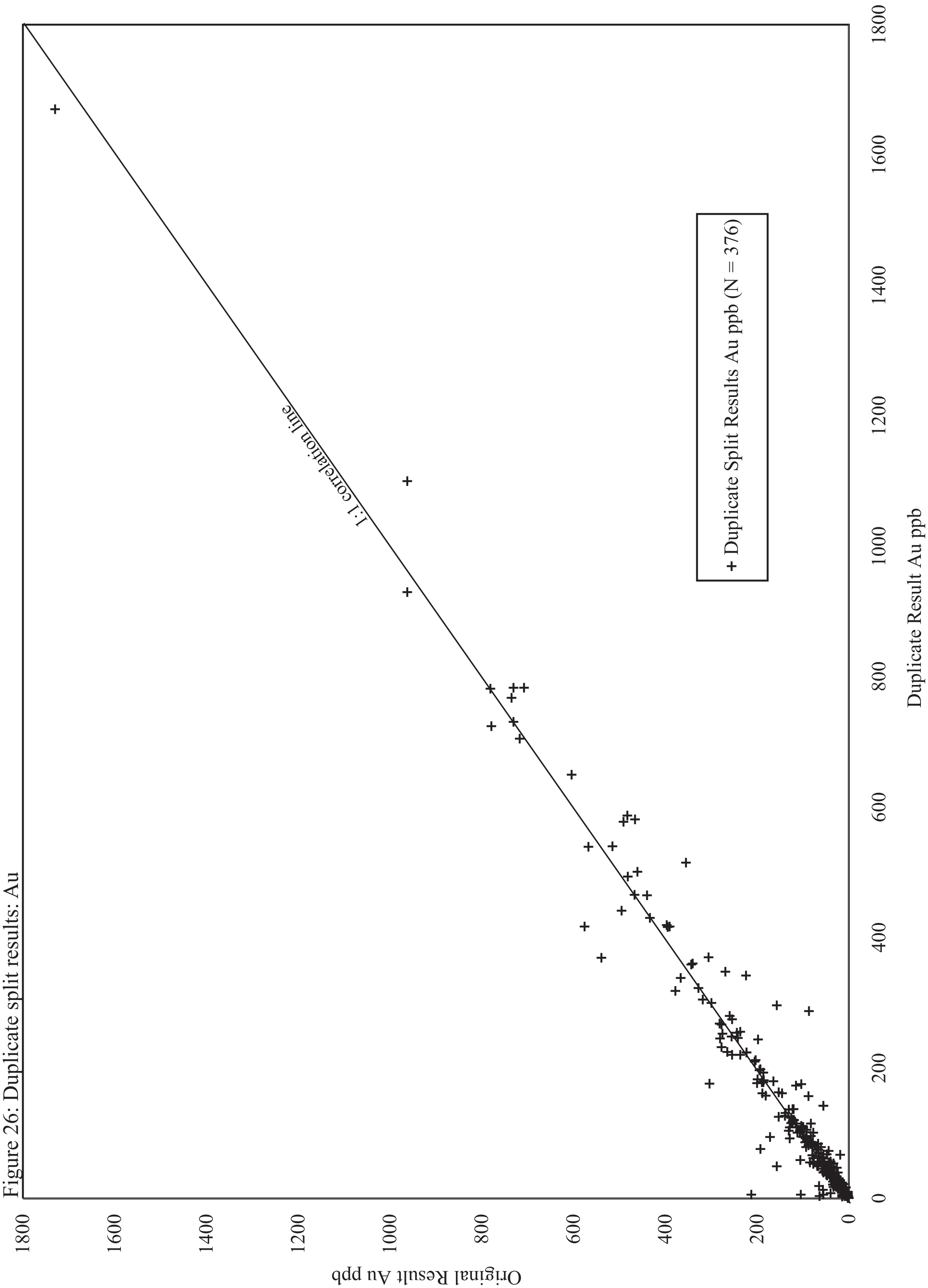



Figure 26: Duplicate split results: Au

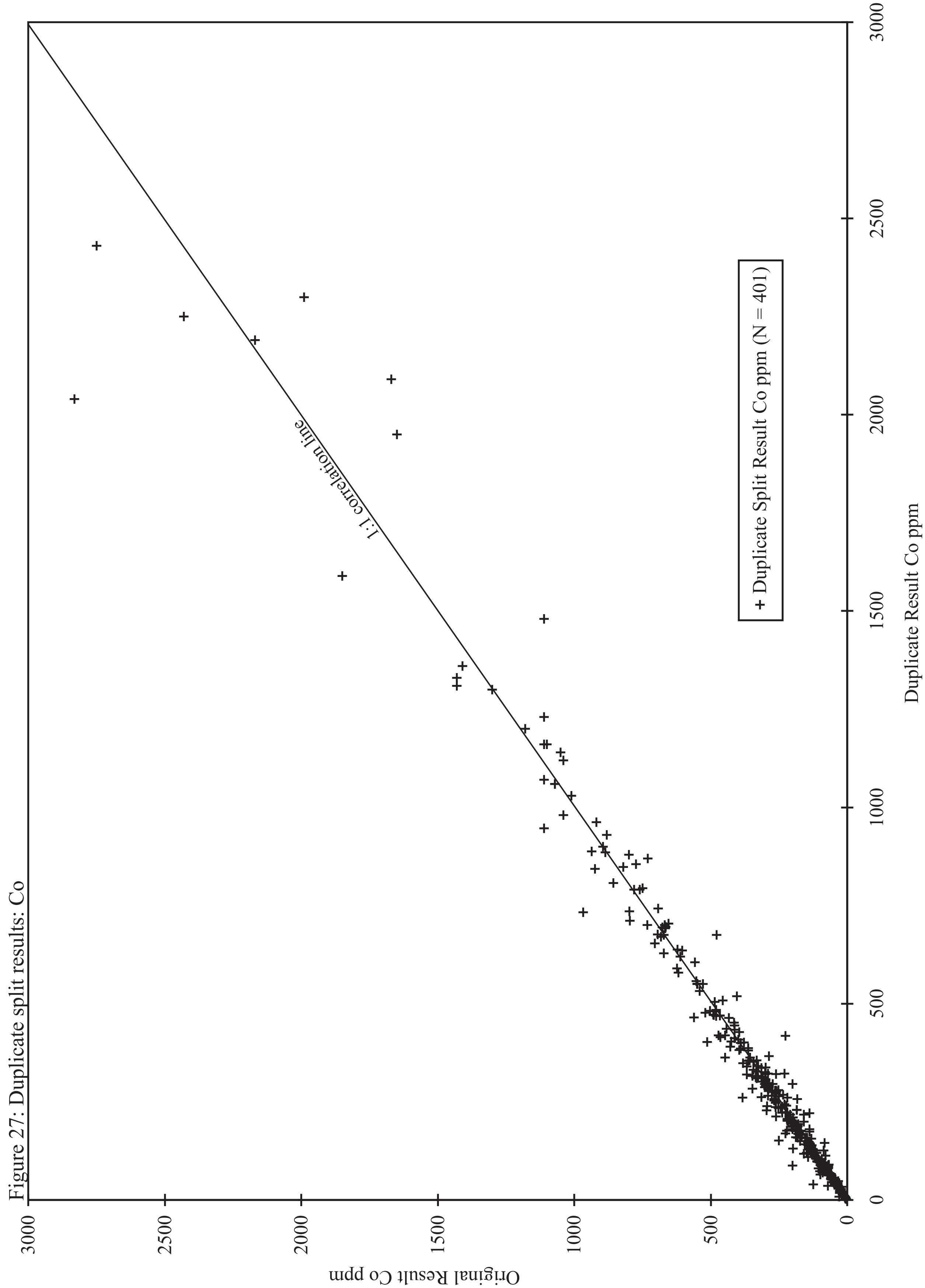
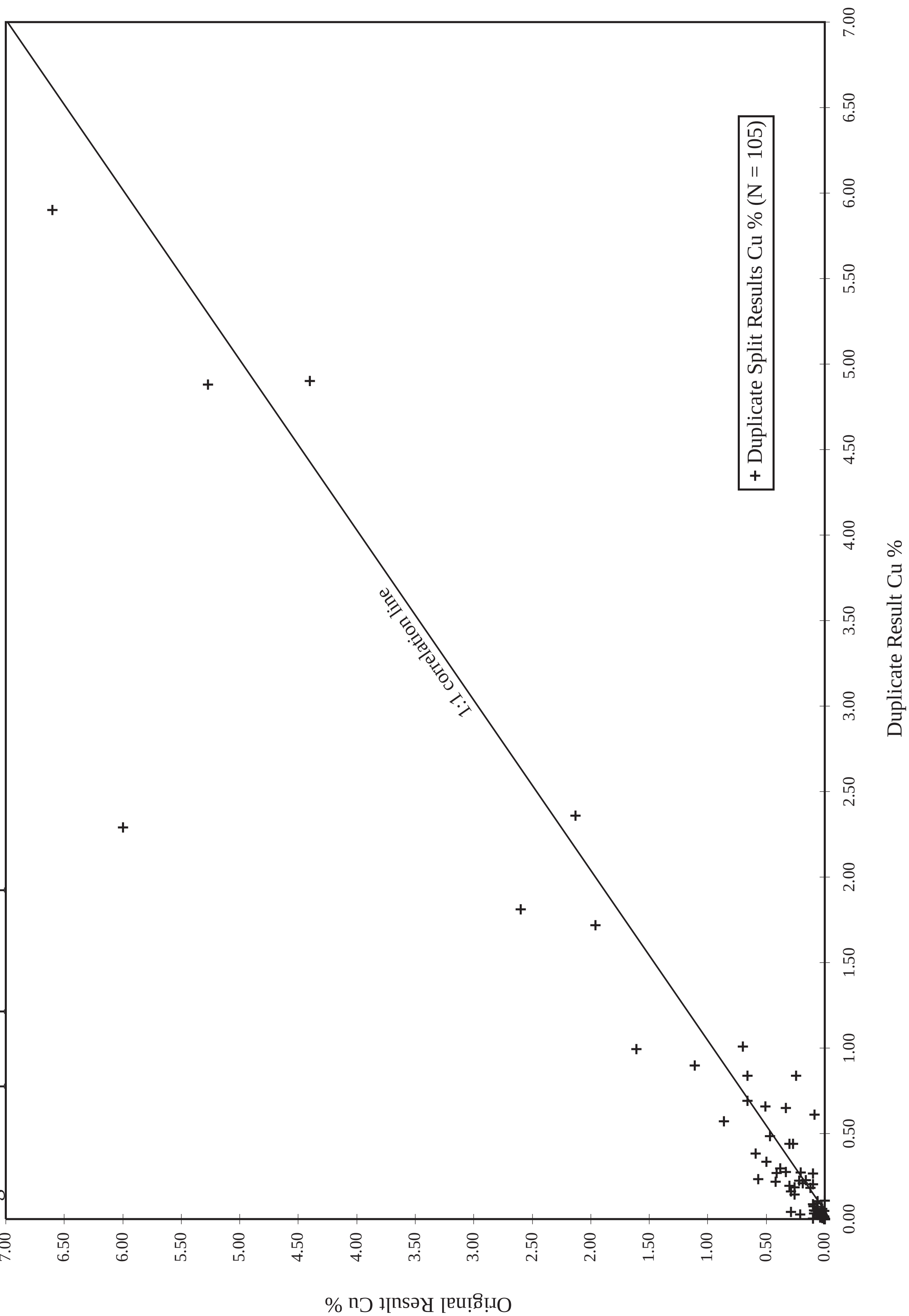



Figure 27: Duplicate split results: Co

Figure 28: Duplicate quarter core split results: Cu



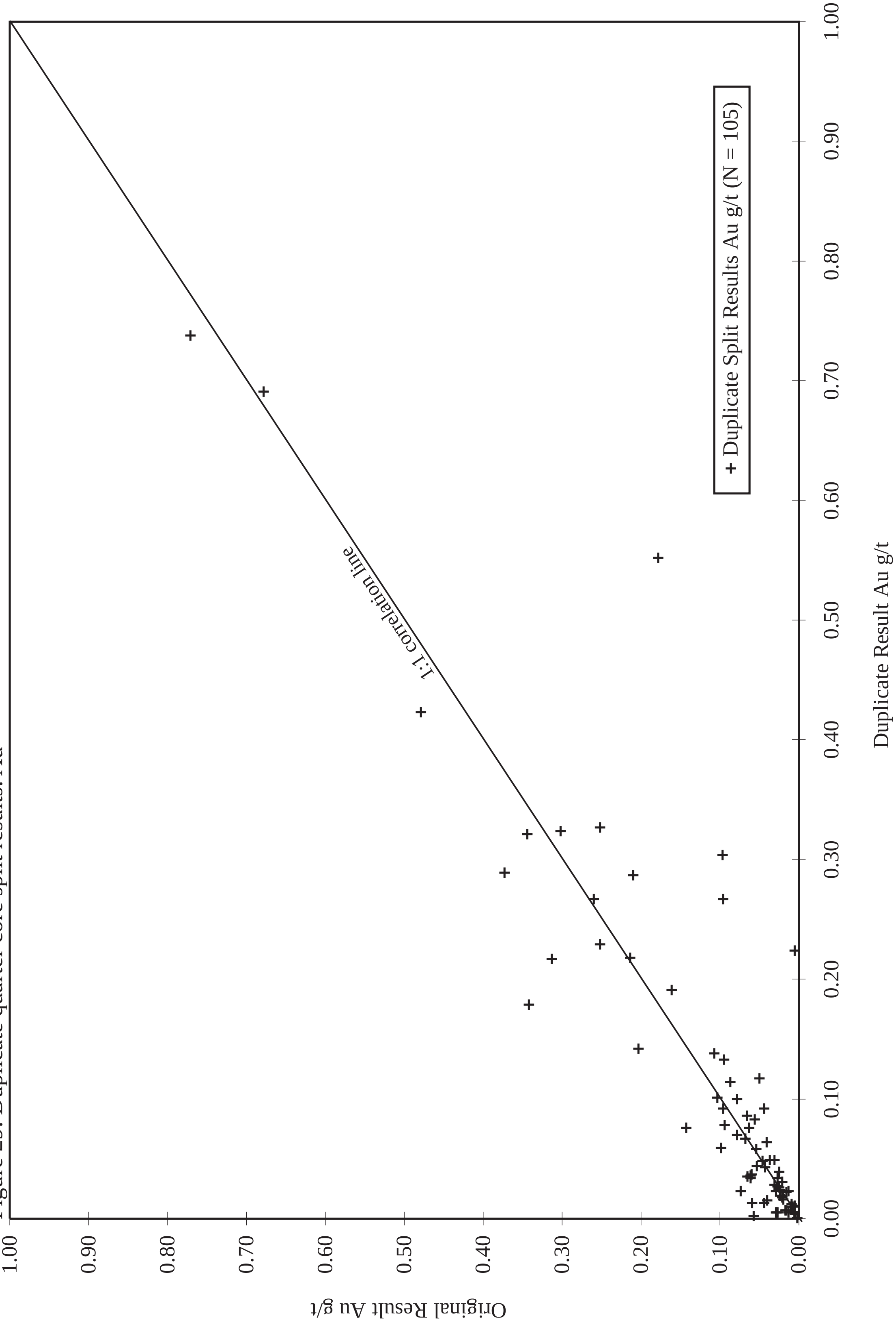
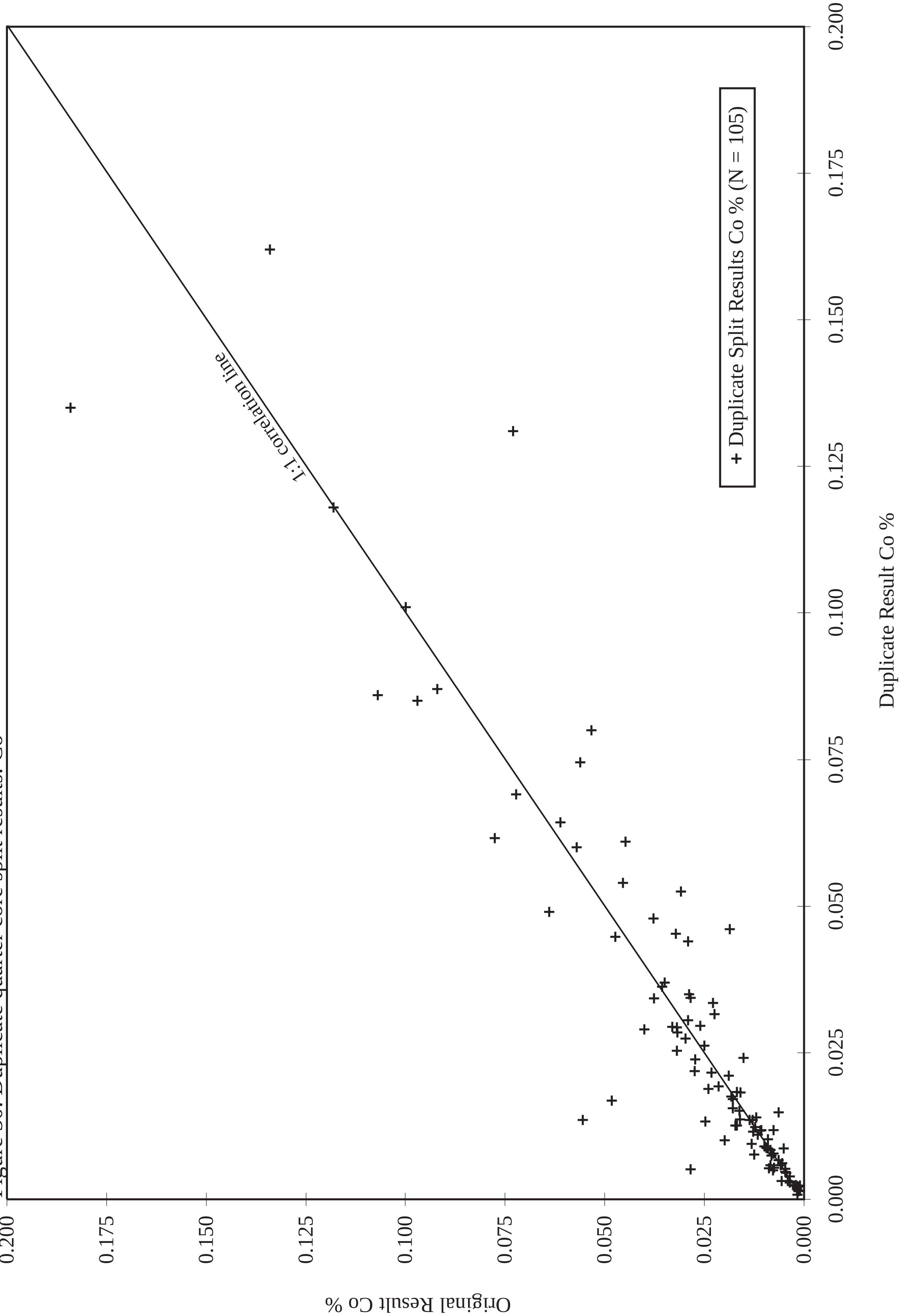



Figure 29: Duplicate quarter core split results: Au

Figure 30: Duplicate quarter core split results: Co



# APPENDIX D

## Captain Resource Estimate Support Documents

| CDN-HZ-2B 2010 | | | CDN-HZ-2B 2010 | |
|---|---|---|---|---|
| *CDN HZ-2 Au g/t* | | | *CDN HZ-2 Cu %* | |
| | | | | |
| Mean | 0.149 | | Mean | 1.345 |
| Standard Error | 0.014 | | Standard Error | 0.005 |
| Median | 0.140 | | Median | 1.340 |
| Standard Deviation | 0.027 | | Standard Deviation | 0.010 |
| Sample Variance | 0.001 | | Sample Variance | 0.000 |
| Kurtosis | 2.879 | | Kurtosis | 4.000 |
| Skewness | 1.670 | | Skewness | 2.000 |
| Range | 0.061 | | Range | 0.020 |
| Minimum | 0.128 | | Minimum | 1.340 |
| Maximum | 0.189 | | Maximum | 1.360 |
| Sum | 0.596 | | Sum | 5.380 |
| Count | 4.000 | | Count | 4.000 |
| Confidence Level(95.0%) | 0.044 | | Confidence Level(95.0%) | 0.016 |

| NI 114 | | | NI 114 | |
|---|---|---|---|---|
| *NI 114 Cu %* | | | *ni114 Co %* | |
| | | | | |
| Mean | 0.425 | | Mean | 0.029 |
| Standard Error | 0.007 | | Standard Error | 0.000 |
| Median | 0.429 | | Median | 0.029 |
| Standard Deviation | 0.018 | | Standard Deviation | 0.001 |
| Sample Variance | 0.000 | | Sample Variance | 0.000 |
| Kurtosis | 3.336 | | Kurtosis | 0.706 |
| Skewness | -1.670 | | Skewness | 0.702 |
| Range | 0.051 | | Range | 0.003 |
| Minimum | 0.392 | | Minimum | 0.028 |
| Maximum | 0.442 | | Maximum | 0.031 |
| Sum | 2.550 | | Sum | 0.147 |
| Count | 6.000 | | Count | 5.000 |
| Confidence Level(95.0%) | 0.019 | | Confidence Level(95.0%) | 0.001 |

| NI 116 | | | NI 116 | |
|---|---|---|---|---|
| *NI116 Cu %* | | | *NI116 Co %* | |
| | | | | |
| Mean | 0.679 | | Mean | 0.046 |
| Standard Error | 0.016 | | Standard Error | 0.001 |
| Median | 0.684 | | Median | 0.048 |
| Standard Deviation | 0.040 | | Standard Deviation | 0.003 |
| Sample Variance | 0.002 | | Sample Variance | 0.000 |
| Kurtosis | 0.346 | | Kurtosis | -3.127 |
| Skewness | -0.966 | | Skewness | -0.626 |
| Range | 0.105 | | Range | 0.005 |
| Minimum | 0.612 | | Minimum | 0.043 |
| Maximum | 0.717 | | Maximum | 0.048 |
| Sum | 4.075 | | Sum | 0.231 |
| Count | 6.000 | | Count | 5.000 |
| Confidence Level(95.0%) | 0.042 | | Confidence Level(95.0%) | 0.003 |

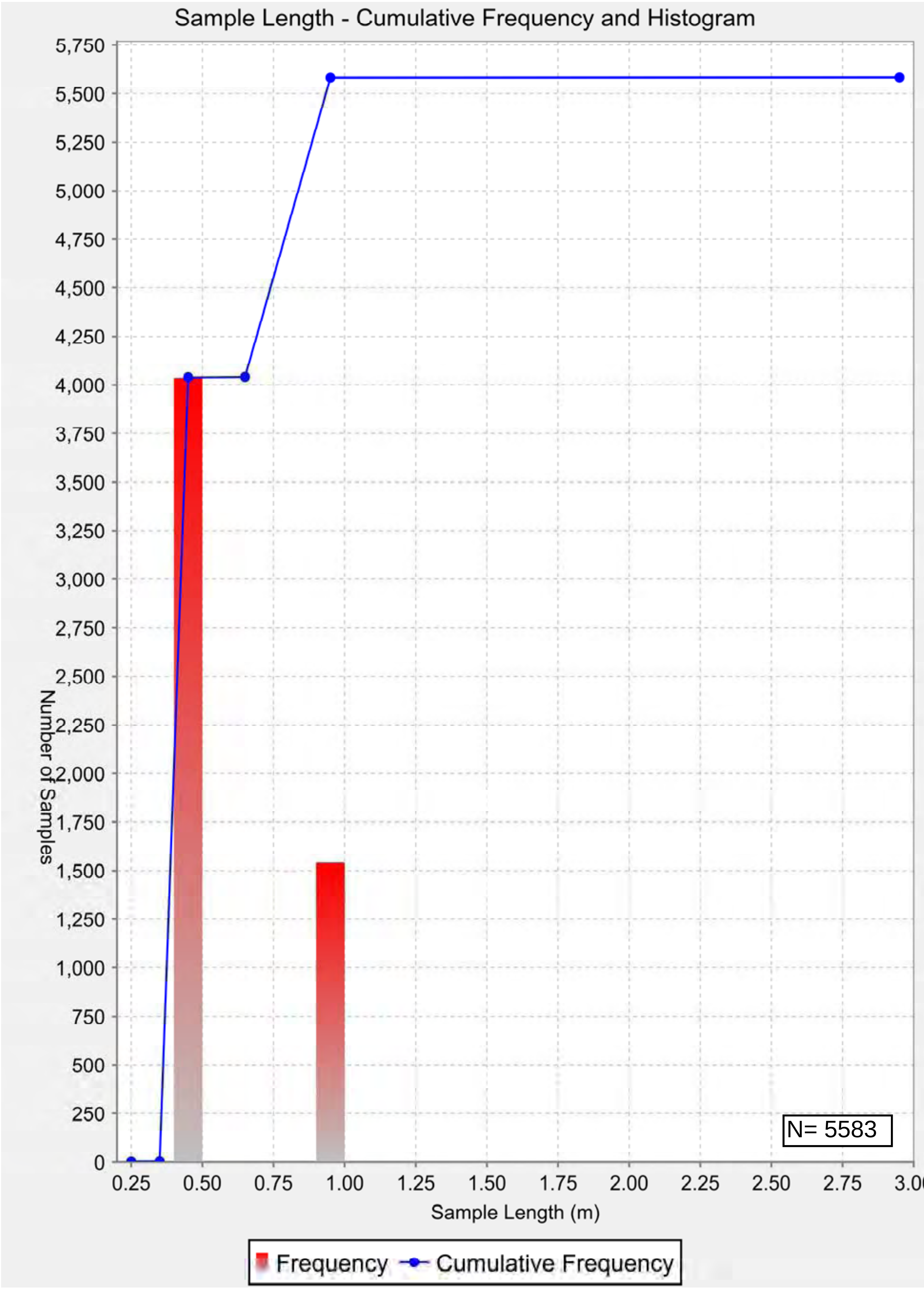
| OREAS 15 Pb | | | OREAS 18 Pb | |
|---|---|---|---|---|
| *OREAS 15Pb Au g/t* | | | *OREAS 18Pb Au g/t* | |
| | | | | |
| Mean | 0.958 | | Mean | 2.999 |
| Standard Error | 0.035 | | Standard Error | 0.077 |
| Median | 0.946 | | Median | 3.009 |
| Standard Deviation | 0.061 | | Standard Deviation | 0.154 |
| Sample Variance | 0.004 | | Sample Variance | 0.024 |
| Kurtosis | #DIV/0! | | Kurtosis | -0.058 |
| Skewness | 0.852 | | Skewness | -0.316 |
| Range | 0.120 | | Range | 0.366 |
| Minimum | 0.904 | | Minimum | 2.807 |
| Maximum | 1.024 | | Maximum | 3.173 |
| Sum | 2.874 | | Sum | 11.997 |
| Count | 3.000 | | Count | 4.000 |
| Confidence Level(95.0%) | 0.151 | | Confidence Level(95.0%) | 0.246 |

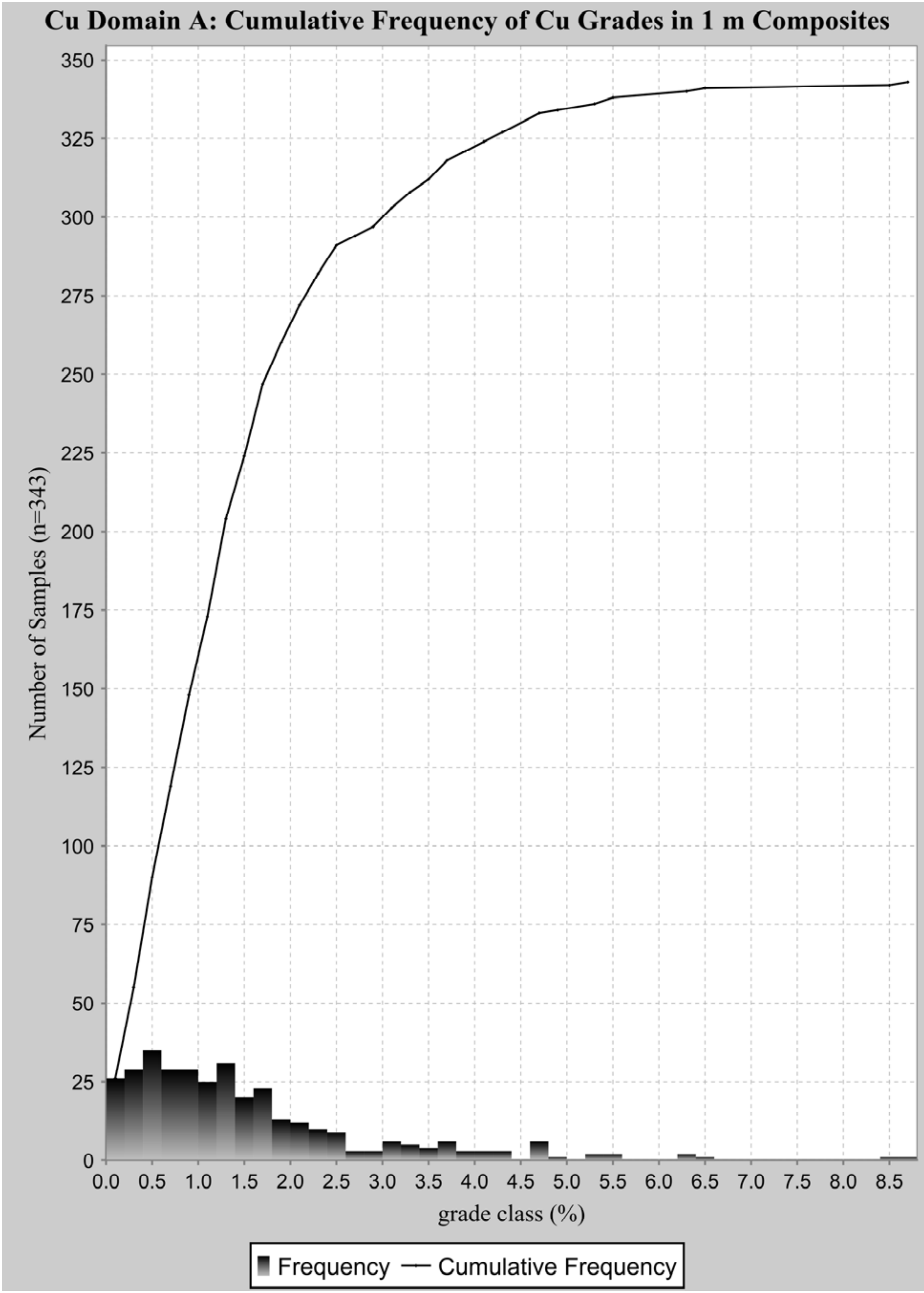
| PB 139 | | | CDN-GS-2B | |
|---|---|---|---|---|
| *Pb 139 Cu%* | | | *CDN GS-2B Au* | |
| | | | | |
| Mean | 0.350 | | Mean | 1.893 |
| Standard Error | 0.003 | | Standard Error | 0.033 |
| Median | 0.353 | | Median | 1.914 |
| Standard Deviation | 0.006 | | Standard Deviation | 0.065 |
| Sample Variance | 0.000 | | Sample Variance | 0.004 |
| Kurtosis | #DIV/0! | | Kurtosis | 2.029 |
| Skewness | -1.704 | | Skewness | -1.477 |
| Range | 0.010 | | Range | 0.145 |
| Minimum | 0.343 | | Minimum | 1.800 |
| Maximum | 0.354 | | Maximum | 1.945 |
| Sum | 1.049 | | Sum | 7.572 |
| Count | 3.000 | | Count | 4.000 |
| Confidence Level(95.0%) | 0.014 | | Confidence Level(95.0%) | 0.104 |

# Stratabound Minerals Corp. Captain Deposit - December 2010

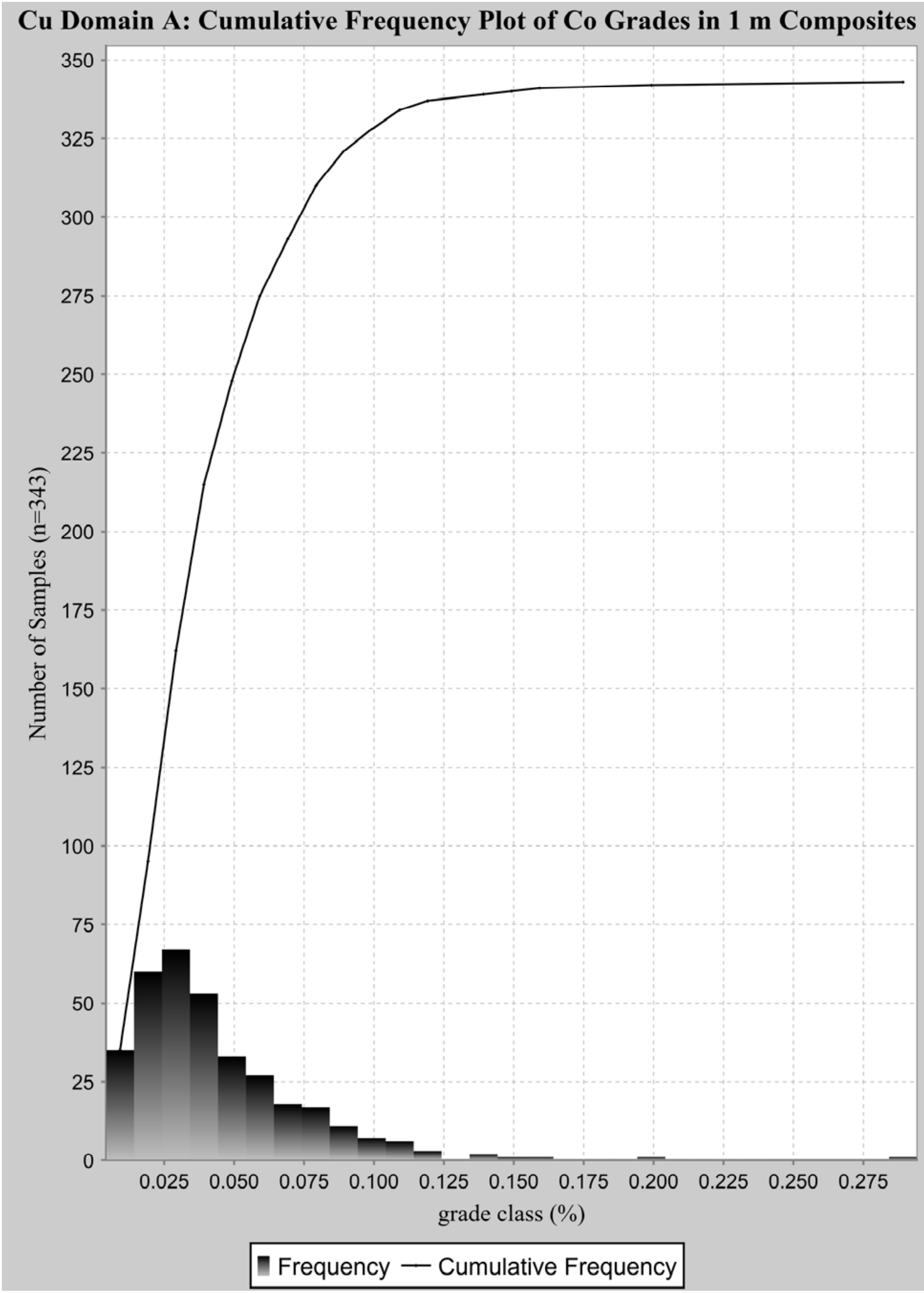
| **Rank And Percentile Report On Core Sample Lengths** | | | | | | | |
|---|---|---|---|---|---|---|---|
| | | | | | | | |
| Number of samples | 5583.000 | | | | | | |
| Minimum value (m) | 0.200 | | | | | | |
| Maximum value (m) | 3.000 | | | | | | |
| | Ungrouped Data | | | | | | |
| Mean | 0.638 | | | | | | |
| Median | 0.500 | | | | | | |
| Geometric Mean | 0.606 | | | | | | |
| Variance | 0.051 | | | | | | |
| Standard Deviation | 0.226 | | | | | | |
| Coefficient of variatior | 0.354 | | | | | | |
| Skewness | 1.169 | | | | | | |
| Kurtosis | 4.061 | | | | | | |
| Natural Log Mean | -0.502 | | | | | | |
| Log Variance | 0.097 | | | | | | |
| | Core Length Class (m) | | | | | | |
| 1.0 Percentile | 0.500 | | 35.0 Percentile | 0.500 | | 69.0 Percentile | 0.500 |
| 2.0 Percentile | 0.500 | | 36.0 Percentile | 0.500 | | 70.0 Percentile | 0.500 |
| 3.0 Percentile | 0.500 | | 37.0 Percentile | 0.500 | | 71.0 Percentile | 0.500 |
| 4.0 Percentile | 0.500 | | 38.0 Percentile | 0.500 | | 72.0 Percentile | 0.500 |
| 5.0 Percentile | 0.500 | | 39.0 Percentile | 0.500 | | 73.0 Percentile | 1.000 |
| 6.0 Percentile | 0.500 | | 40.0 Percentile | 0.500 | | 74.0 Percentile | 1.000 |
| 7.0 Percentile | 0.500 | | 41.0 Percentile | 0.500 | | 75.0 Percentile | 1.000 |
| 8.0 Percentile | 0.500 | | 42.0 Percentile | 0.500 | | 76.0 Percentile | 1.000 |
| 9.0 Percentile | 0.500 | | 43.0 Percentile | 0.500 | | 77.0 Percentile | 1.000 |
| 10.0 Percentile | 0.500 | | 44.0 Percentile | 0.500 | | 78.0 Percentile | 1.000 |
| 11.0 Percentile | 0.500 | | 45.0 Percentile | 0.500 | | 79.0 Percentile | 1.000 |
| 12.0 Percentile | 0.500 | | 46.0 Percentile | 0.500 | | 80.0 Percentile | 1.000 |
| 13.0 Percentile | 0.500 | | 47.0 Percentile | 0.500 | | 81.0 Percentile | 1.000 |
| 14.0 Percentile | 0.500 | | 48.0 Percentile | 0.500 | | 82.0 Percentile | 1.000 |
| 15.0 Percentile | 0.500 | | 49.0 Percentile | 0.500 | | 83.0 Percentile | 1.000 |
| 16.0 Percentile | 0.500 | | 50.0 Percentile ( | 0.500 | | 84.0 Percentile | 1.000 |
| 17.0 Percentile | 0.500 | | 51.0 Percentile | 0.500 | | 85.0 Percentile | 1.000 |
| 18.0 Percentile | 0.500 | | 52.0 Percentile | 0.500 | | 86.0 Percentile | 1.000 |
| 19.0 Percentile | 0.500 | | 53.0 Percentile | 0.500 | | 87.0 Percentile | 1.000 |
| 20.0 Percentile | 0.500 | | 54.0 Percentile | 0.500 | | 88.0 Percentile | 1.000 |
| 21.0 Percentile | 0.500 | | 55.0 Percentile | 0.500 | | 89.0 Percentile | 1.000 |
| 22.0 Percentile | 0.500 | | 56.0 Percentile | 0.500 | | 90.0 Percentile | 1.000 |
| 23.0 Percentile | 0.500 | | 57.0 Percentile | 0.500 | | 91.0 Percentile | 1.000 |
| 24.0 Percentile | 0.500 | | 58.0 Percentile | 0.500 | | 92.0 Percentile | 1.000 |
| 25.0 Percentile | 0.500 | | 59.0 Percentile | 0.500 | | 93.0 Percentile | 1.000 |
| 26.0 Percentile | 0.500 | | 60.0 Percentile | 0.500 | | 94.0 Percentile | 1.000 |
| 27.0 Percentile | 0.500 | | 61.0 Percentile | 0.500 | | 95.0 Percentile | 1.000 |
| 28.0 Percentile | 0.500 | | 62.0 Percentile | 0.500 | | 96.0 Percentile | 1.000 |
| 29.0 Percentile | 0.500 | | 63.0 Percentile | 0.500 | | 97.0 Percentile | 1.000 |
| 30.0 Percentile | 0.500 | | 64.0 Percentile | 0.500 | | 98.0 Percentile | 1.000 |
| 31.0 Percentile | 0.500 | | 65.0 Percentile | 0.500 | | 99.0 Percentile | 1.000 |
| 32.0 Percentile | 0.500 | | 66.0 Percentile | 0.500 | | | |
| 33.0 Percentile | 0.500 | | 67.0 Percentile | 0.500 | | | |

## Stratabound Minerals Corp. Captain Deposit - December 2010

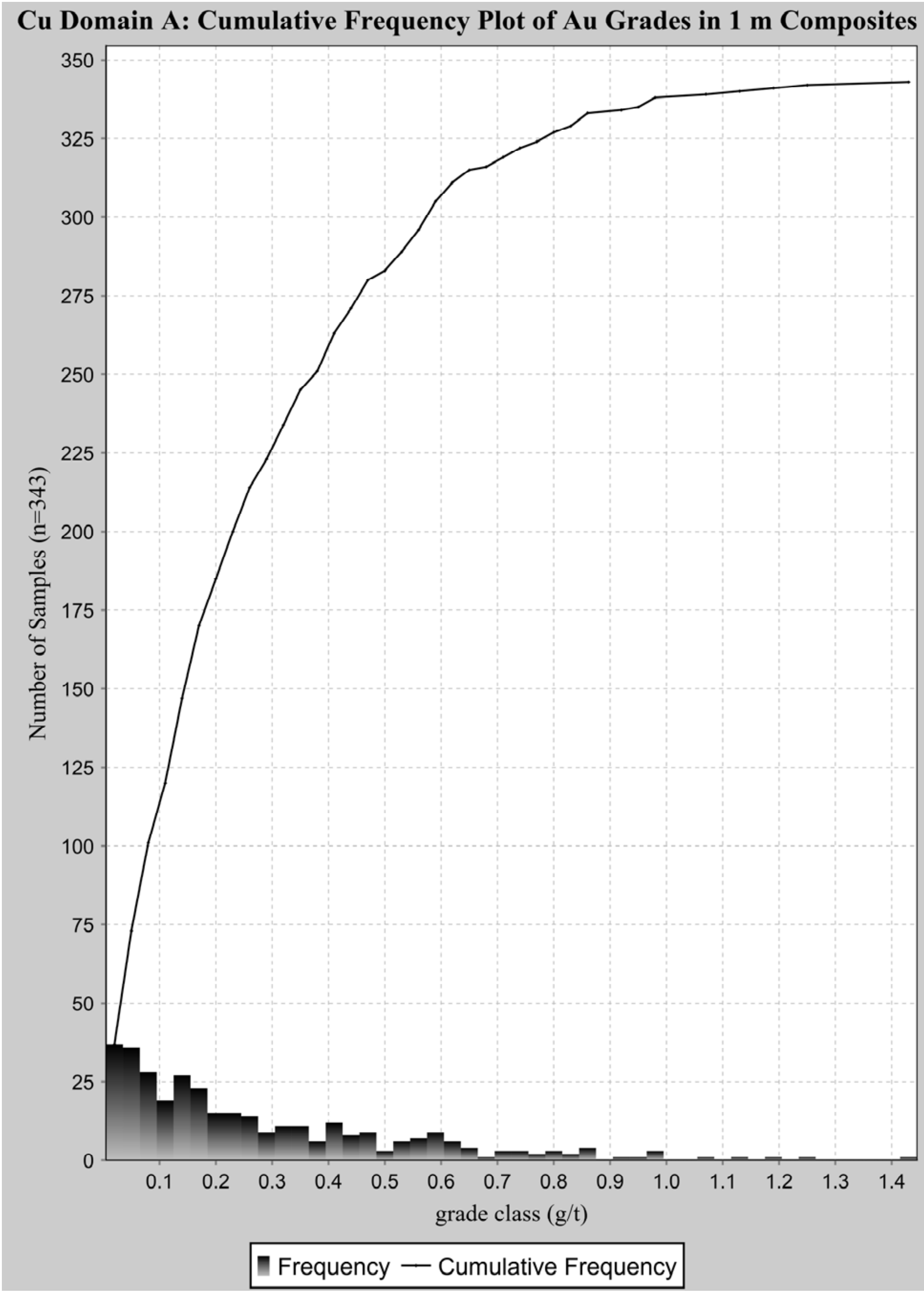




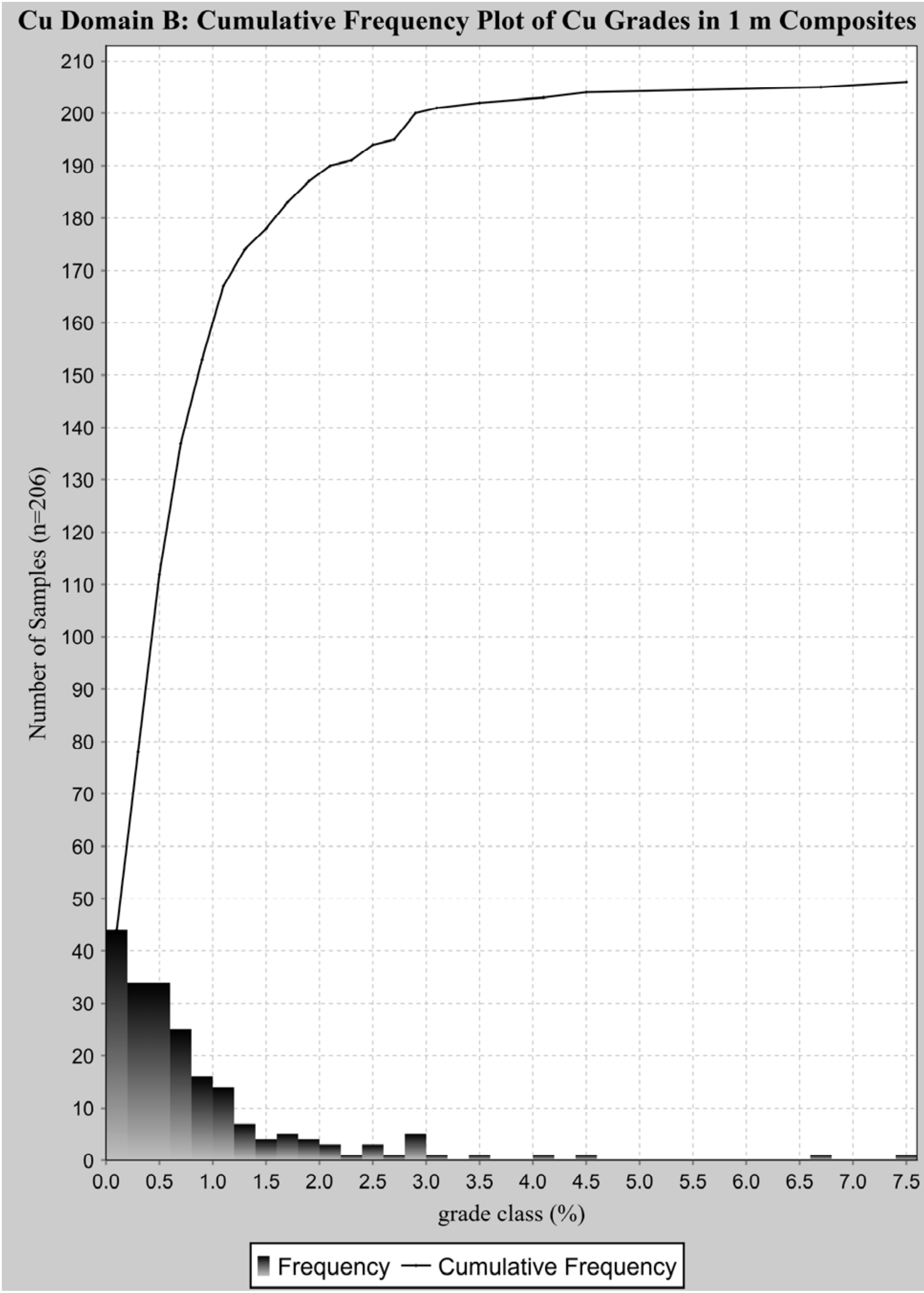
Cu Domain A: Cumulative Frequency of Cu Grades in 1 m Composites
Number of Samples (n=343)
350
325
300
275
250
225
200
175
150
125
100
75
50
25
0
0.0 0.5 1.0 1.5 2.0 2.5 3.0 3.5 4.0 4.5 5.0 5.5 6.0 6.5 7.0 7.5 8.0 8.5
grade class (%)
Frequency
Cumulative Frequency

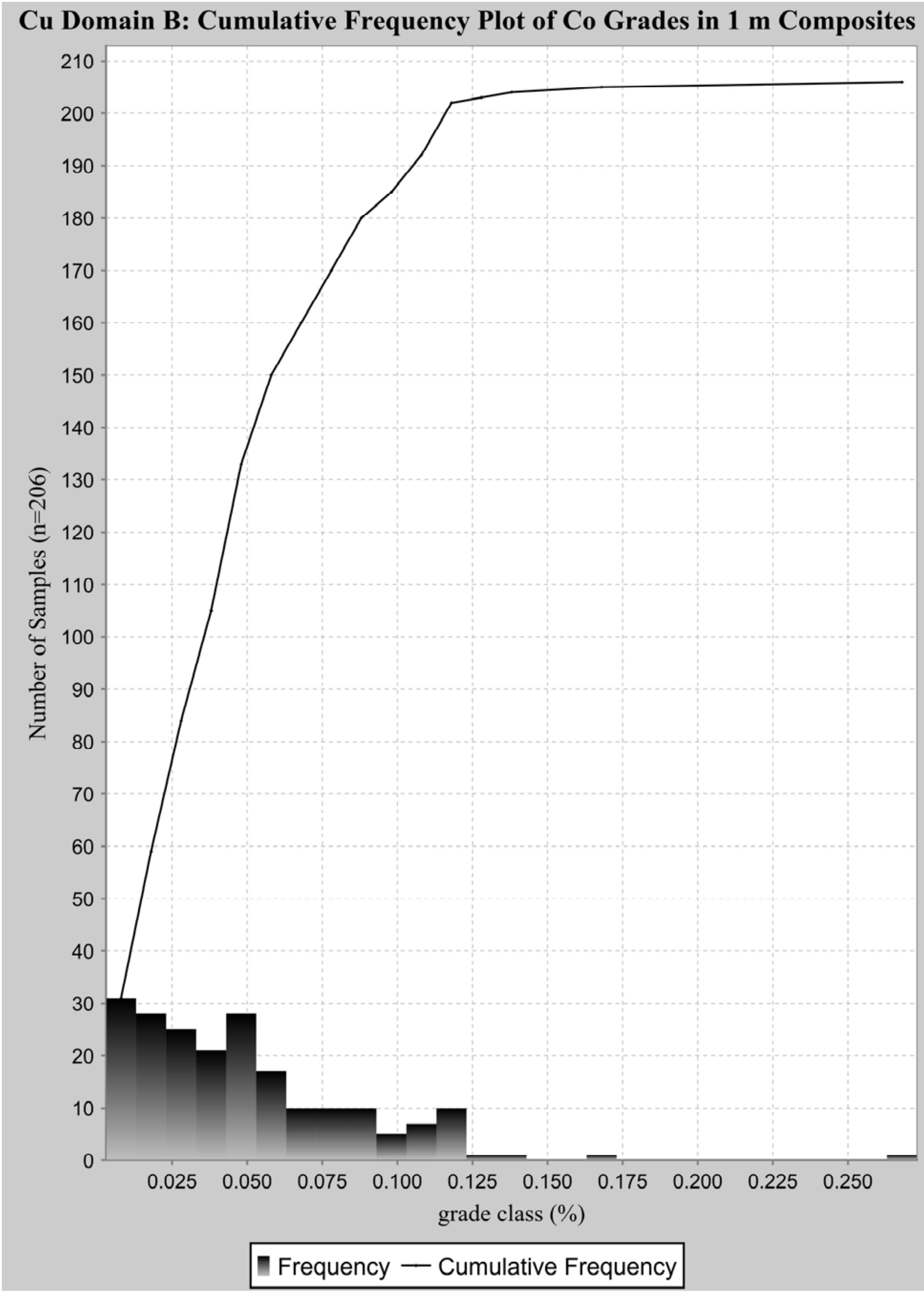


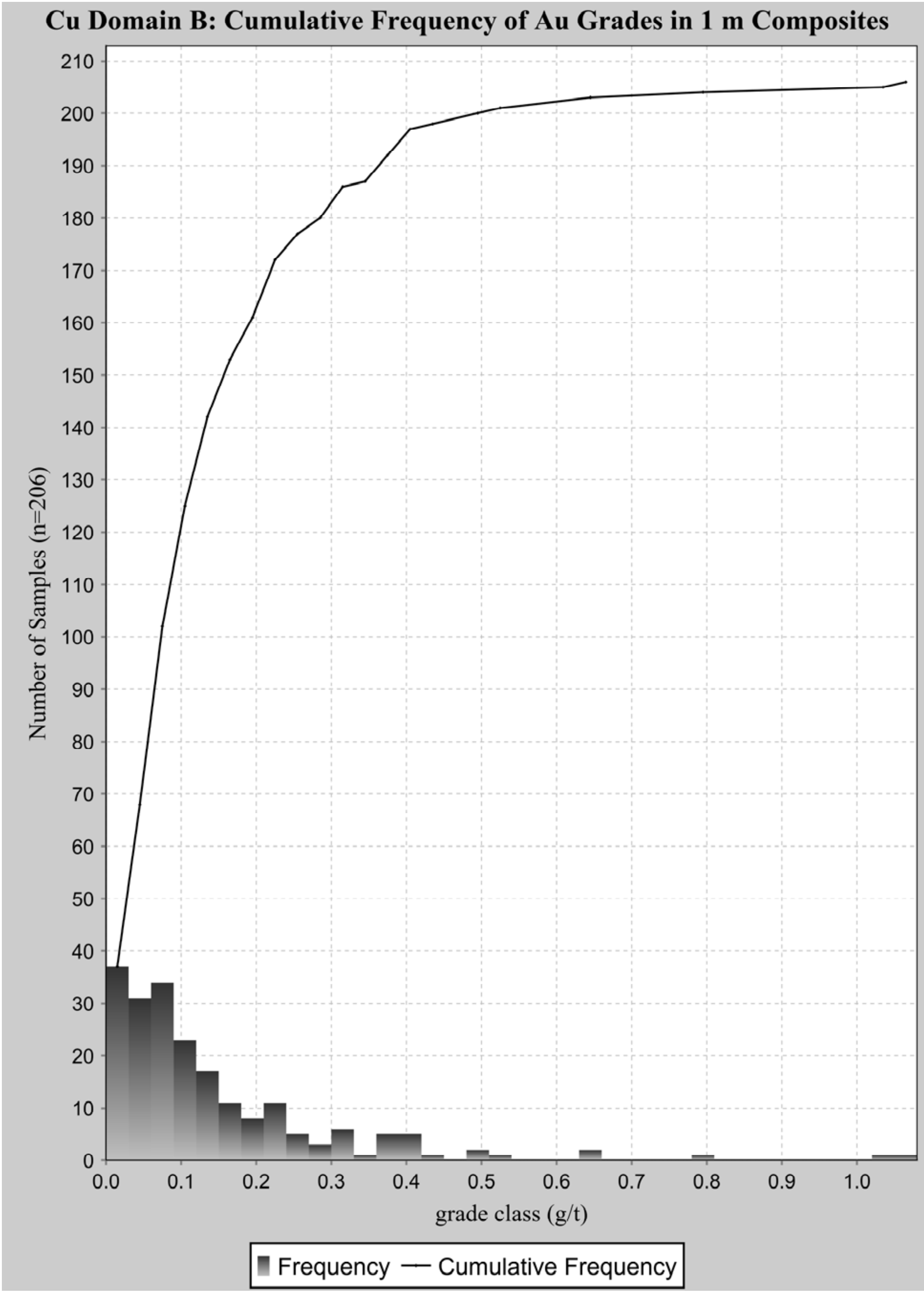
Cu Domain A: Cumulative Frequency Plot of Co Grades in 1 m Composites
Number of Samples (n=343)
0
25
50
75
100
125
150
175
200
225
250
275
300
325
350
0.025
0.050
0.075
0.100
0.125
0.150
0.175
0.200
0.225
0.250
0.275
grade class (%)
Frequency
Cumulative Frequency



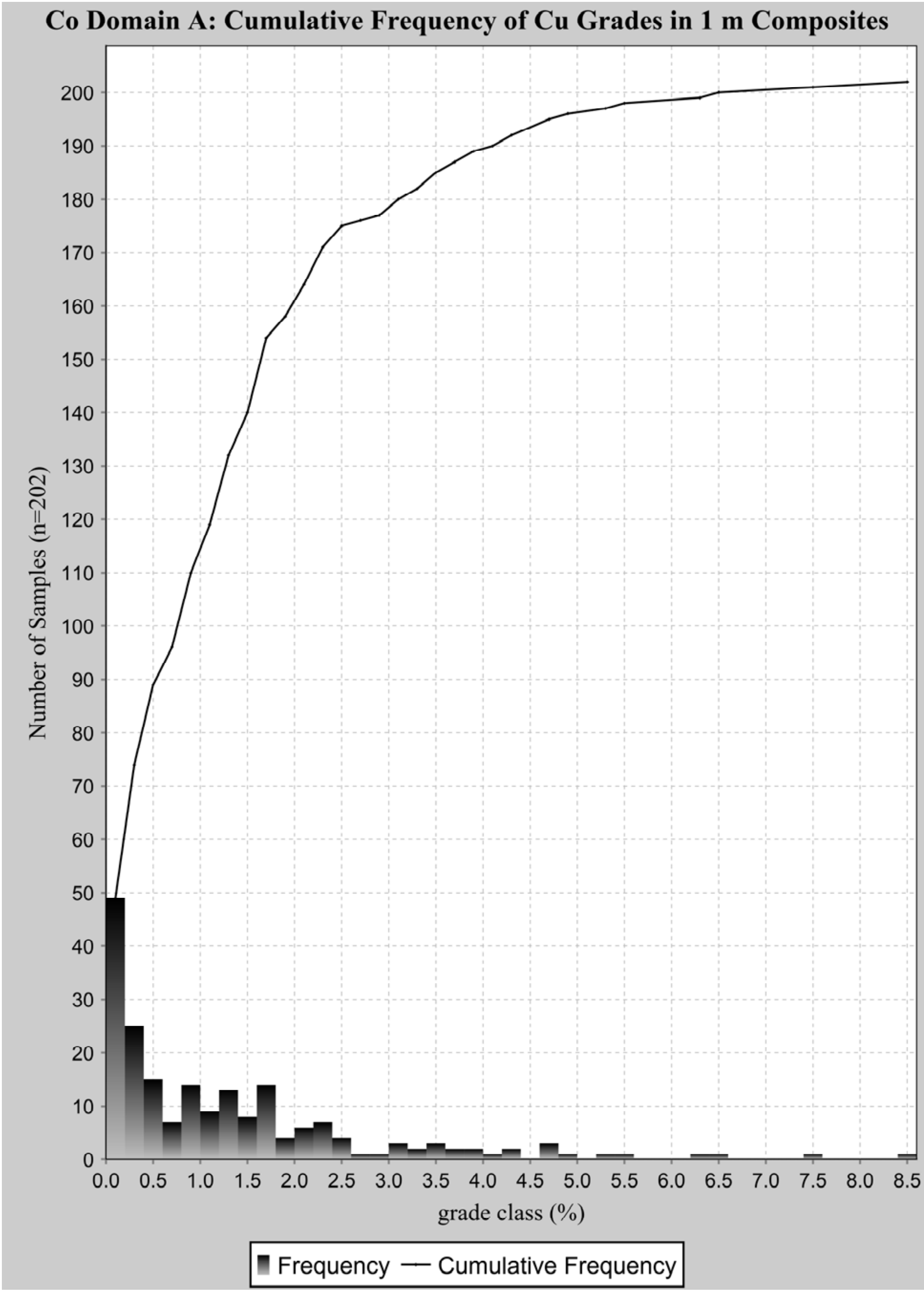
Cu Domain A: Cumulative Frequency Plot of Au Grades in 1 m Composites
Number of Samples (n=343)
350
325
300
275
250
225
200
175
150
125
100
75
50
25
0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
1.0
1.1
1.2
1.3
1.4
grade class (g/t)
Frequency
Cumulative Frequency


Cu Domain B: Cumulative Frequency Plot of Cu Grades in 1 m Composites
Number of Samples (n=206)
210
200
190
180
170
160
150
140
130
120
110
100
90
80
70
60
50
40
30
20
10
0
0.0
0.5
1.0
1.5
2.0
2.5
3.0
3.5
4.0
4.5
5.0
5.5
6.0
6.5
7.0
7.5
grade class (%)
Frequency
Cumulative Frequency

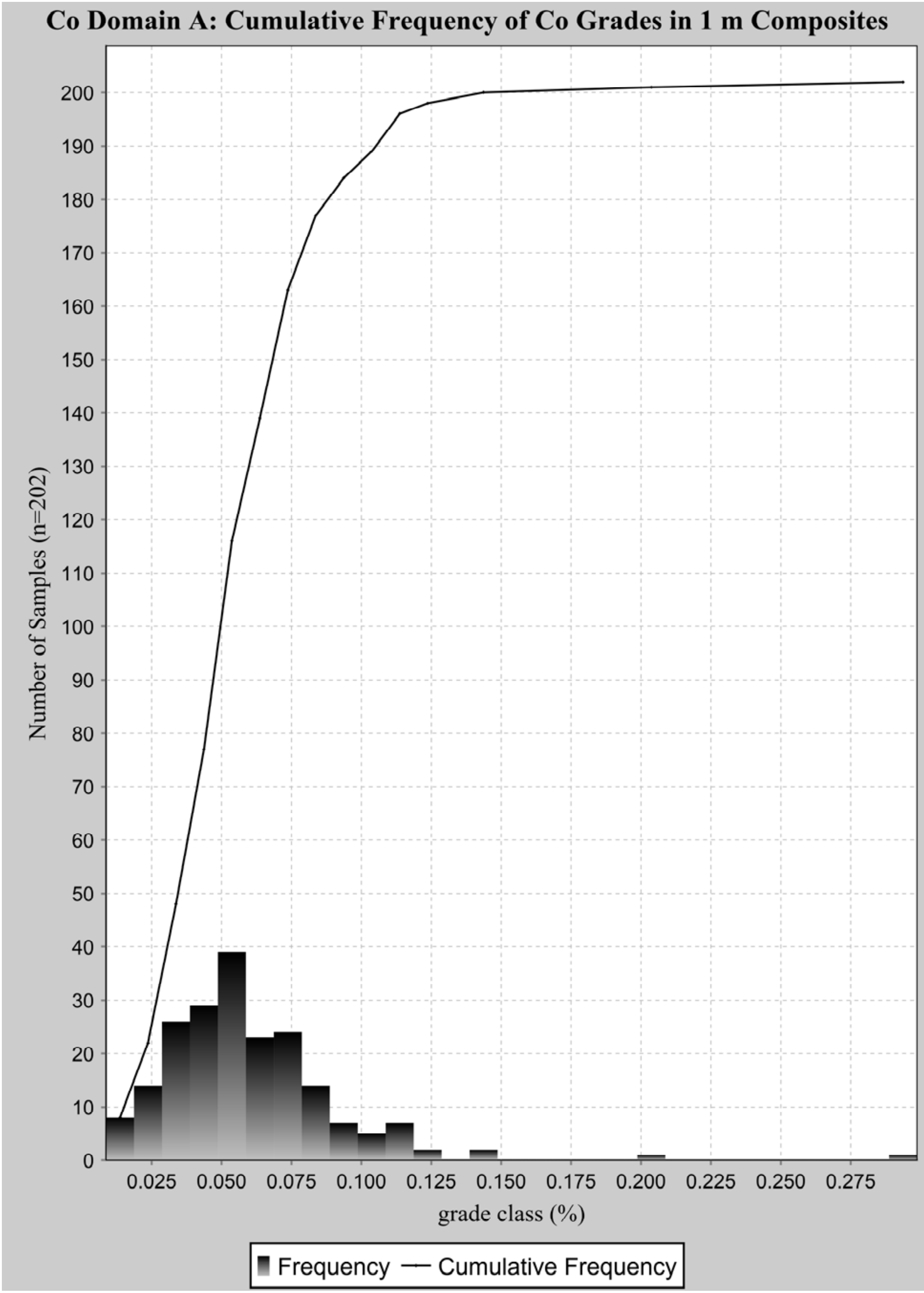

Cu Domain B: Cumulative Frequency Plot of Co Grades in 1 m Composites
Number of Samples (n=206)
210
200
190
180
170
160
150
140
130
120
110
100
90
80
70
60
50
40
30
20
10
0
0.025
0.050
0.075
0.100
0.125
0.150
0.175
0.200
0.225
0.250
grade class (%)
Frequency
Cumulative Frequency

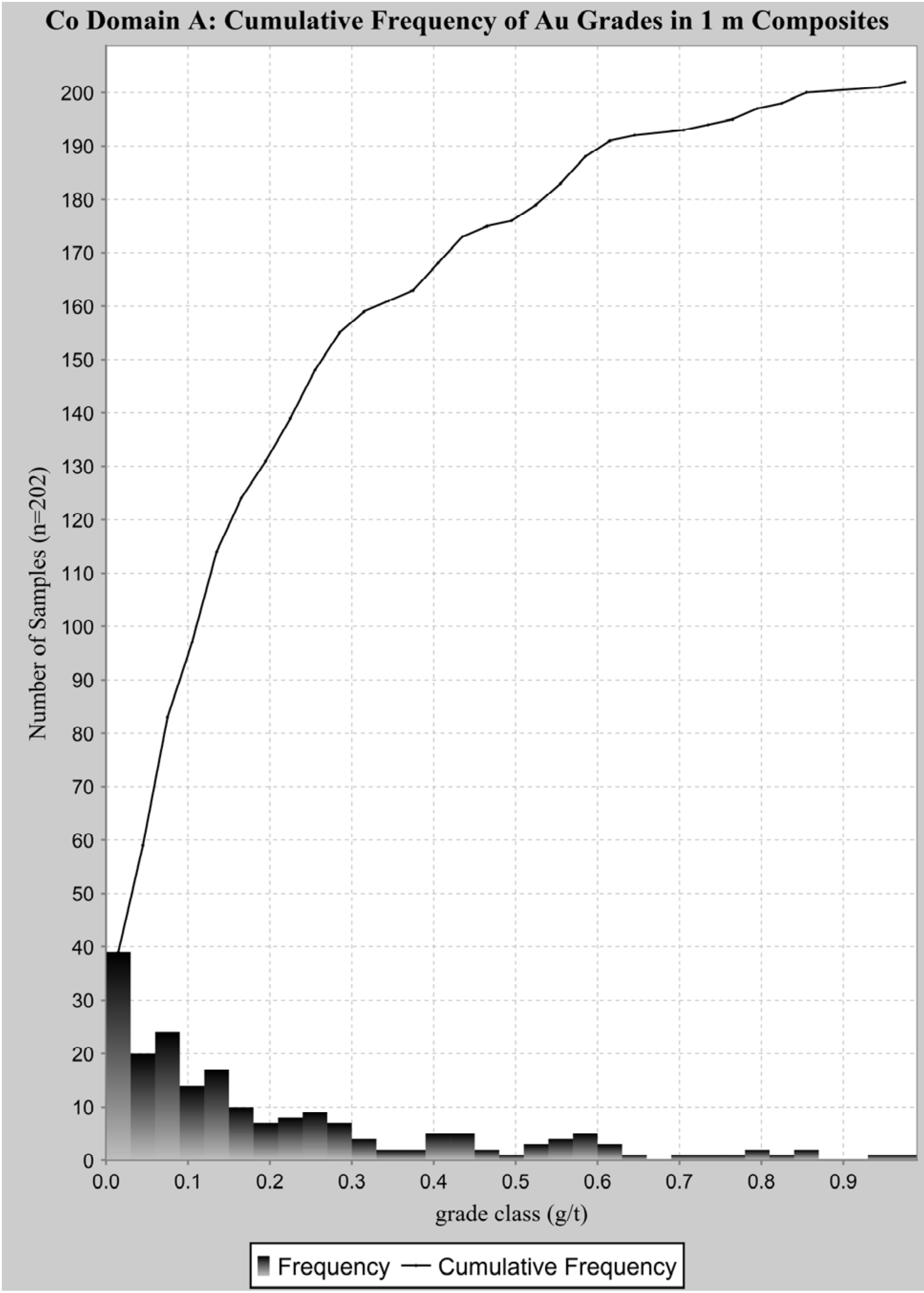


Cu Domain B: Cumulative Frequency of Au Grades in 1 m Composites
Number of Samples (n=206)
210
200
190
180
170
160
150
140
130
120
110
100
90
80
70
60
50
40
30
20
10
0
0.0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
1.0
grade class (g/t)
Frequency
Cumulative Frequency

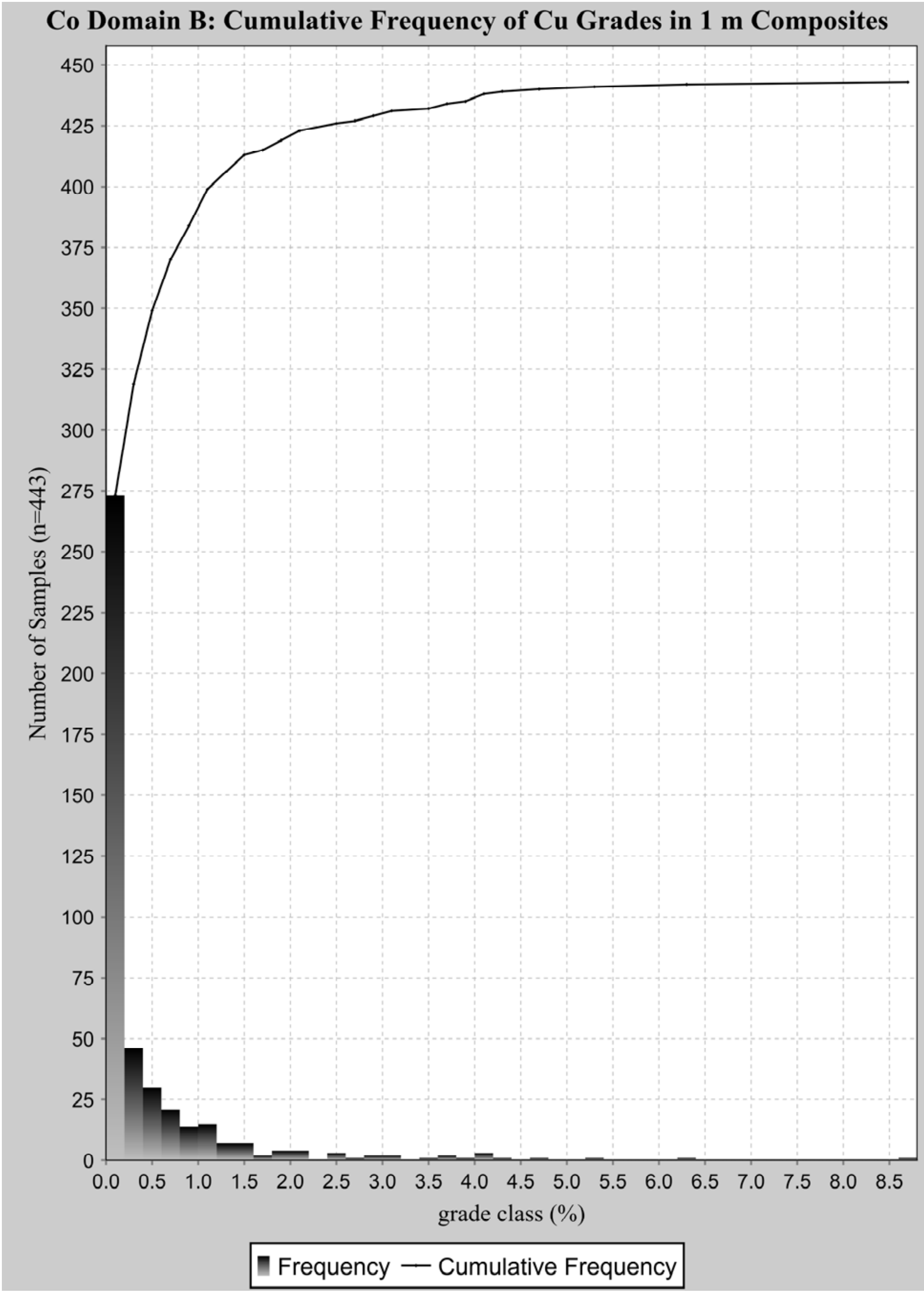


Co Domain A: Cumulative Frequency of Cu Grades in 1 m Composites
Number of Samples (n=202)
0
10
20
30
40
50
60
70
80
90
100
110
120
130
140
150
160
170
180
190
200
0.0
0.5
1.0
1.5
2.0
2.5
3.0
3.5
4.0
4.5
5.0
5.5
6.0
6.5
7.0
7.5
8.0
8.5
grade class (%)
Frequency
Cumulative Frequency

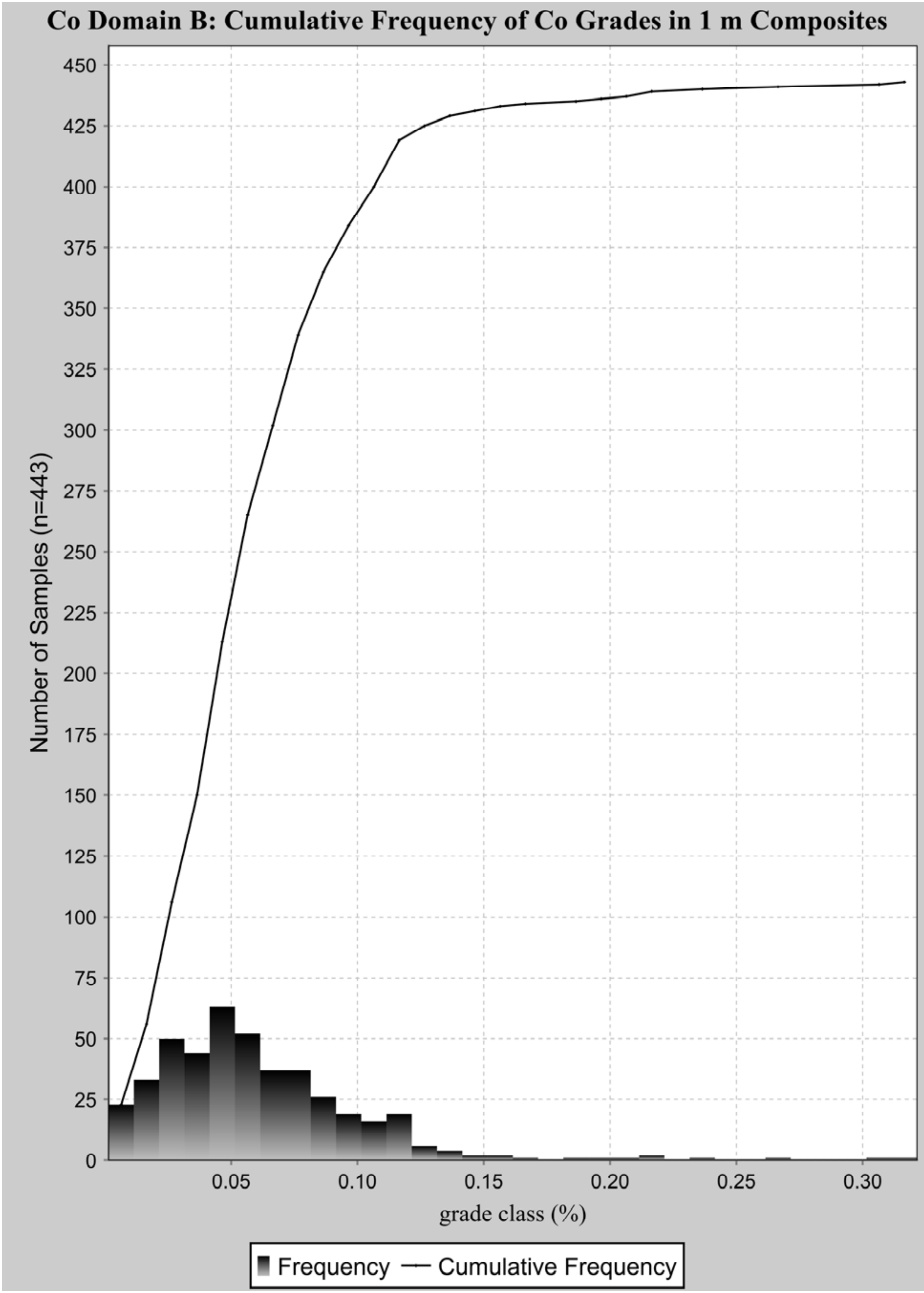

Co Domain A: Cumulative Frequency of Co Grades in 1 m Composites
Number of Samples (n=202)
0
10
20
30
40
50
60
70
80
90
100
110
120
130
140
150
160
170
180
190
200
0.025
0.050
0.075
0.100
0.125
0.150
0.175
0.200
0.225
0.250
0.275
grade class (%)
Frequency
Cumulative Frequency



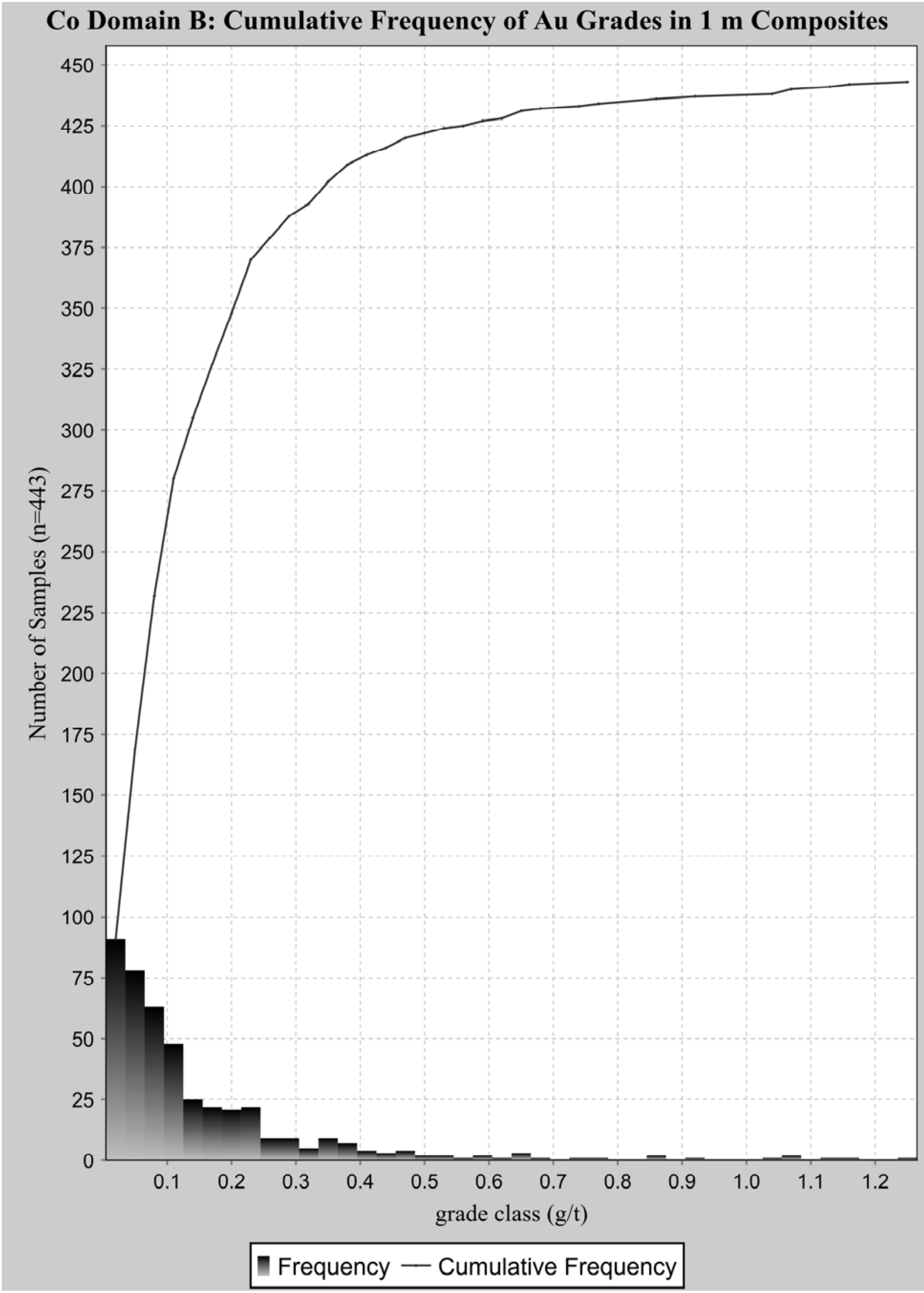
Co Domain A: Cumulative Frequency of Au Grades in 1 m Composites
Number of Samples (n=202)
0
10
20
30
40
50
60
70
80
90
100
110
120
130
140
150
160
170
180
190
200
0.0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
grade class (g/t)
Frequency
Cumulative Frequency



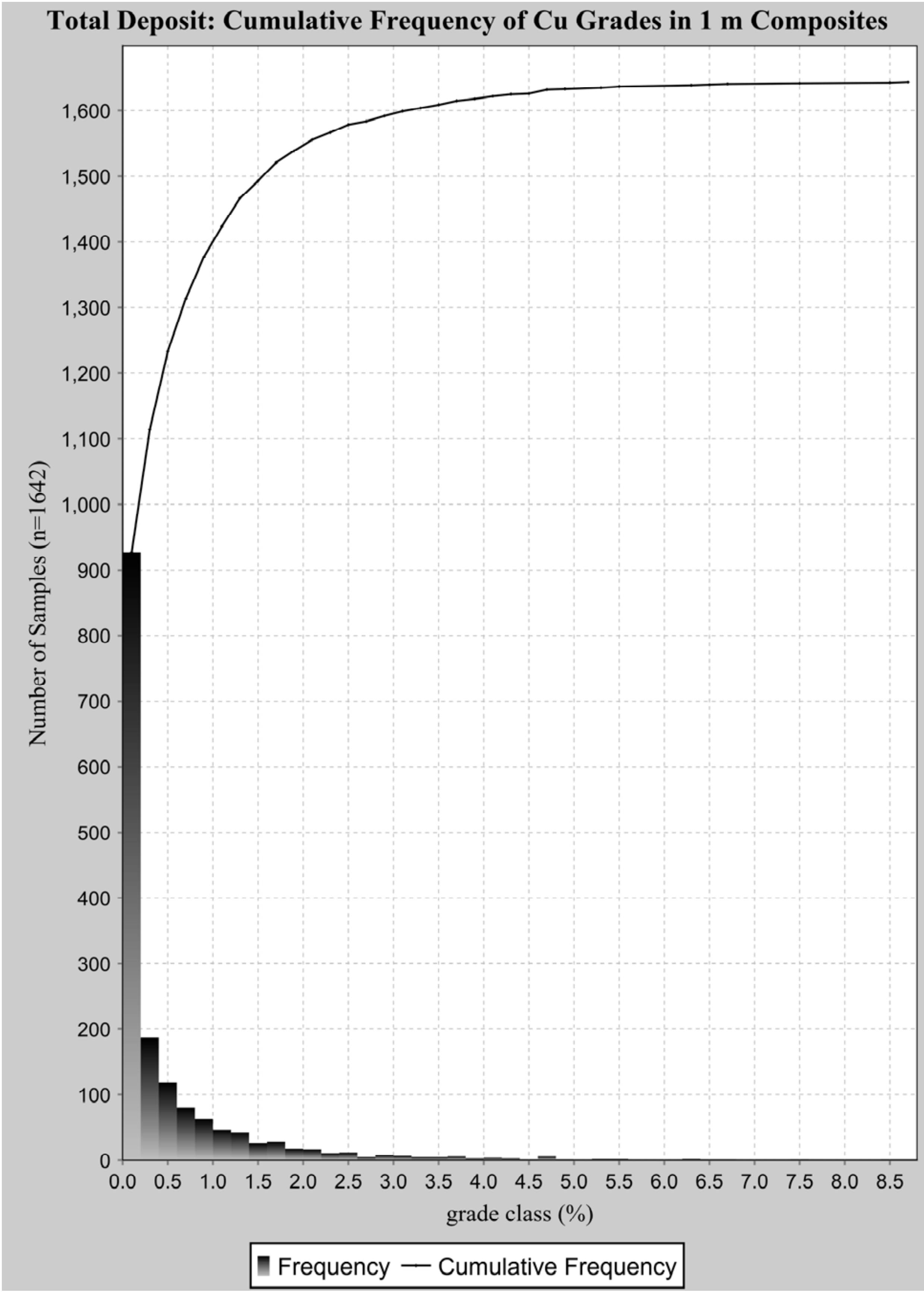
Co Domain B: Cumulative Frequency of Cu Grades in 1 m Composites
Number of Samples (n=443)
0
25
50
75
100
125
150
175
200
225
250
275
300
325
350
375
400
425
450
0.0
0.5
1.0
1.5
2.0
2.5
3.0
3.5
4.0
4.5
5.0
5.5
6.0
6.5
7.0
7.5
8.0
8.5
grade class (%)
Frequency
Cumulative Frequency

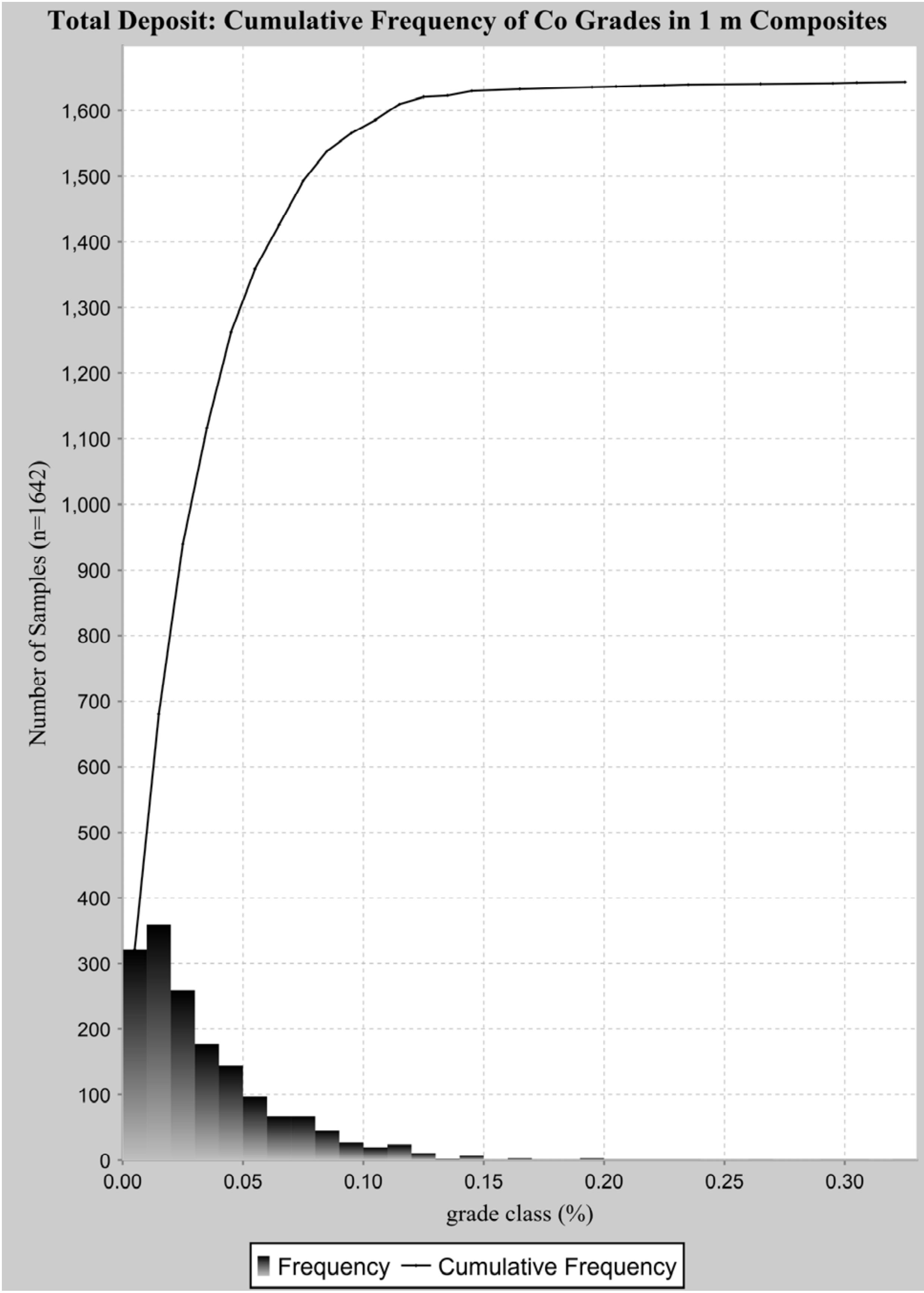


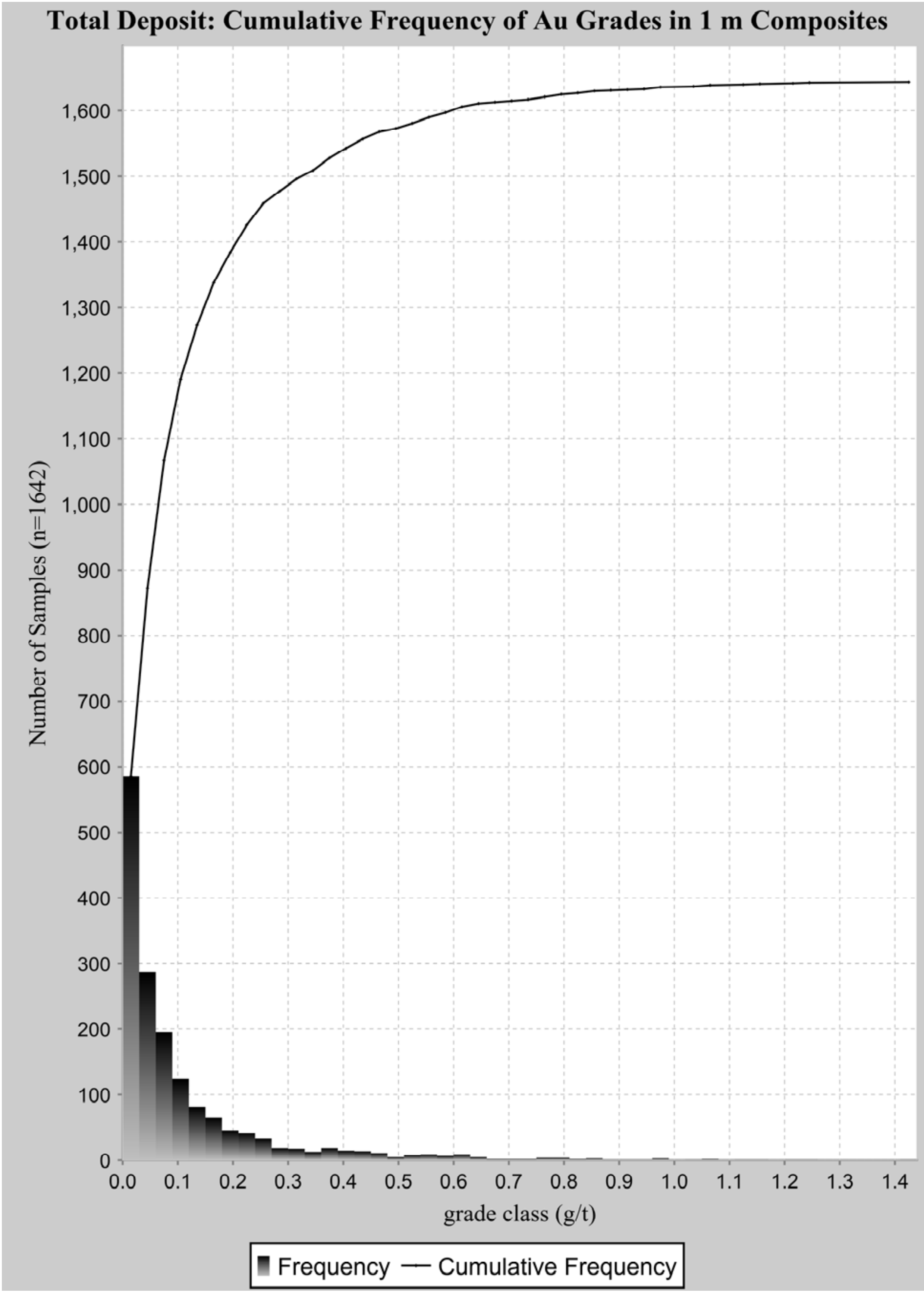
Co Domain B: Cumulative Frequency of Co Grades in 1 m Composites
Number of Samples (n=443)
450
425
400
375
350
325
300
275
250
225
200
175
150
125
100
75
50
25
0
0.05
0.10
0.15
0.20
0.25
0.30
grade class (%)
Frequency
Cumulative Frequency



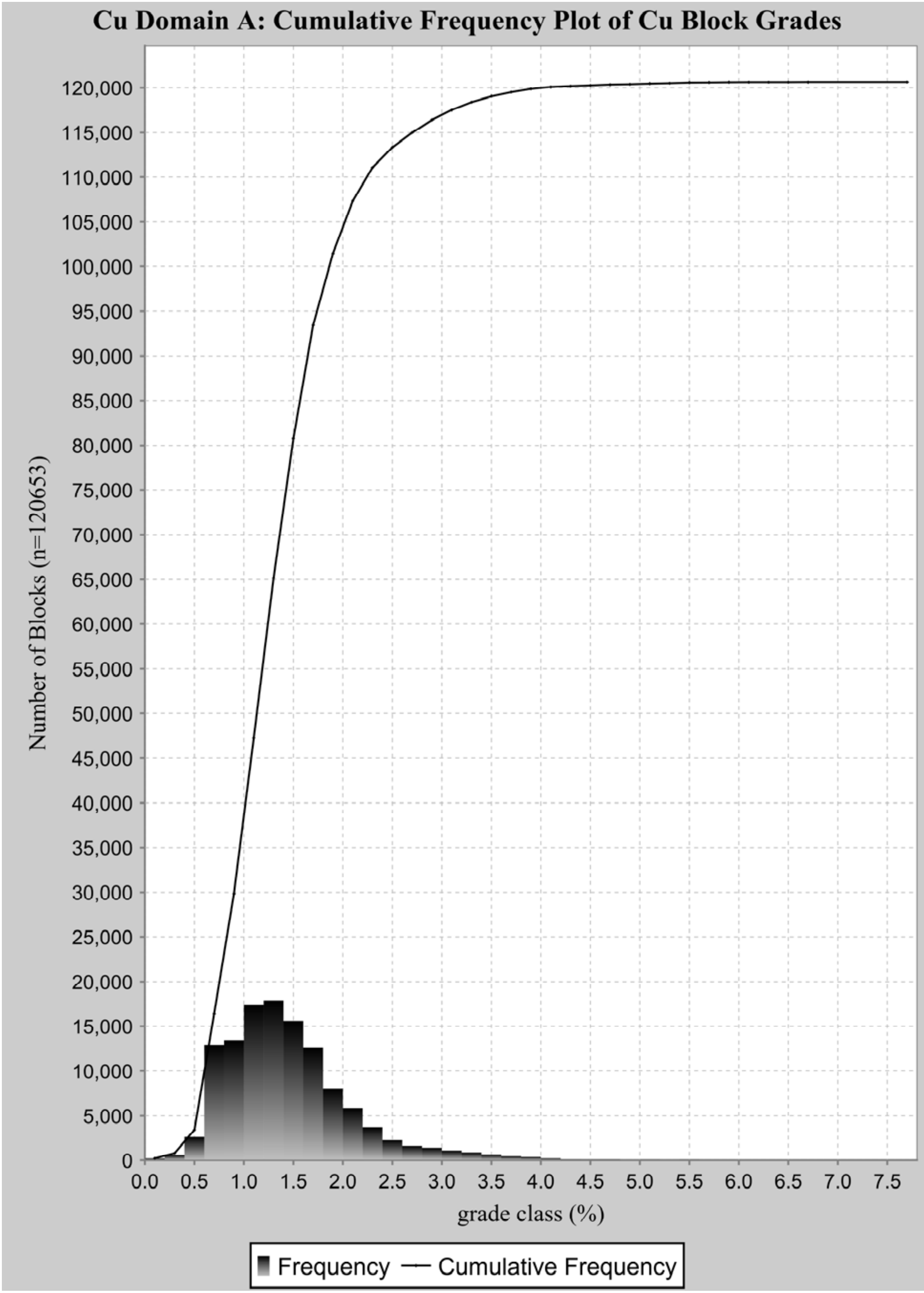
Co Domain B: Cumulative Frequency of Au Grades in 1 m Composites
450
425
400
375
350
325
300
275
250
225
200
175
150
125
100
75
50
25
0
Number of Samples (n=443)
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
1.0
1.1
1.2
grade class (g/t)
Frequency
Cumulative Frequency


Total Deposit: Cumulative Frequency of Cu Grades in 1 m Composites
Number of Samples (n=1642)
0
100
200
300
400
500
600
700
800
900
1,000
1,100
1,200
1,300
1,400
1,500
1,600
0.0 0.5 1.0 1.5 2.0 2.5 3.0 3.5 4.0 4.5 5.0 5.5 6.0 6.5 7.0 7.5 8.0 8.5
grade class (%)
Frequency
Cumulative Frequency

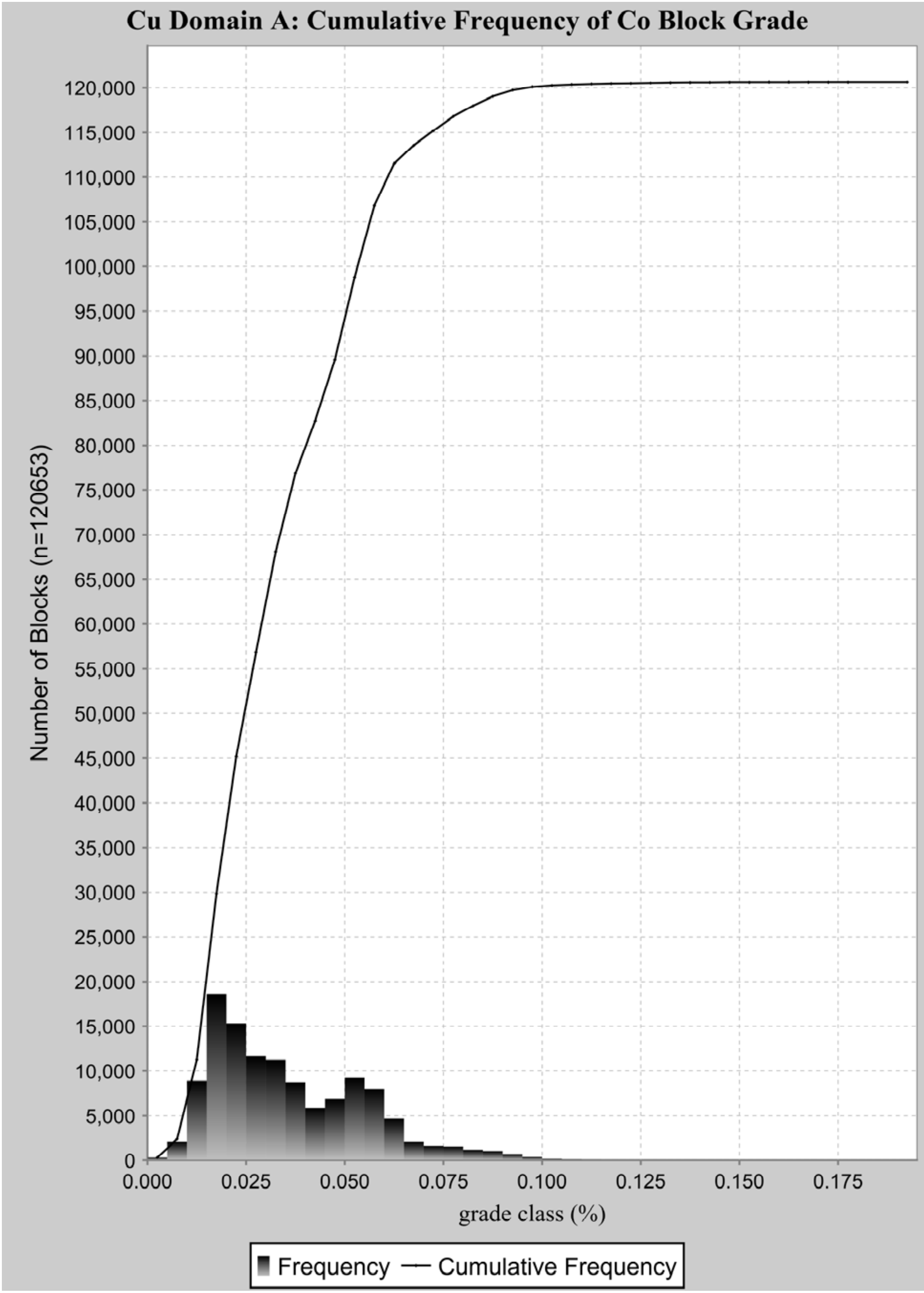

Total Deposit: Cumulative Frequency of Co Grades in 1 m Composites
Number of Samples (n=1642)
1,600
1,500
1,400
1,300
1,200
1,100
1,000
900
800
700
600
500
400
300
200
100
0
0.00
0.05
0.10
0.15
0.20
0.25
0.30
grade class (%)
Frequency
Cumulative Frequency



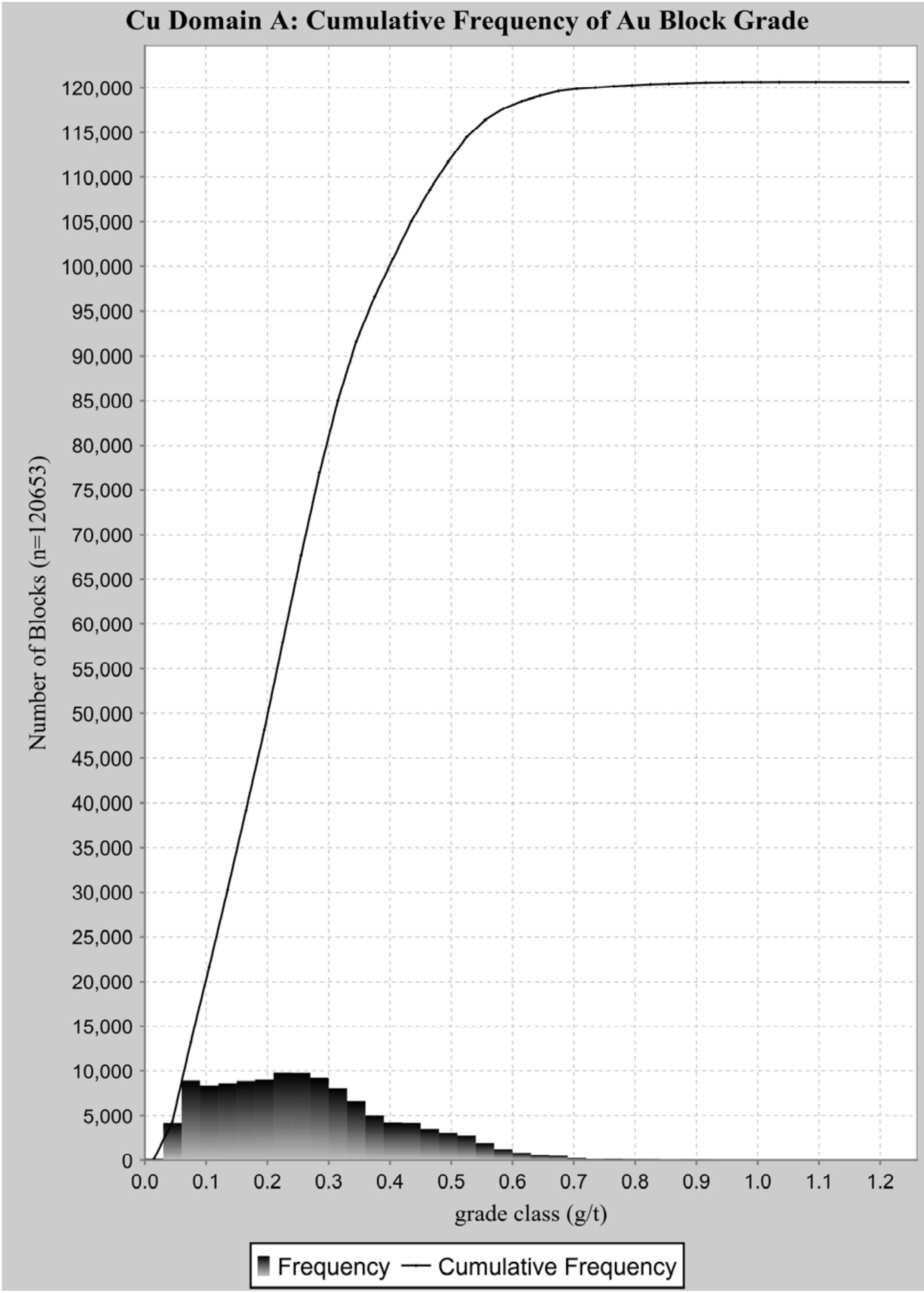
Total Deposit: Cumulative Frequency of Au Grades in 1 m Composites
Number of Samples (n=1642)
1,600
1,500
1,400
1,300
1,200
1,100
1,000
900
800
700
600
500
400
300
200
100
0
0.0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
1.0
1.1
1.2
1.3
1.4
grade class (g/t)
Frequency
Cumulative Frequency



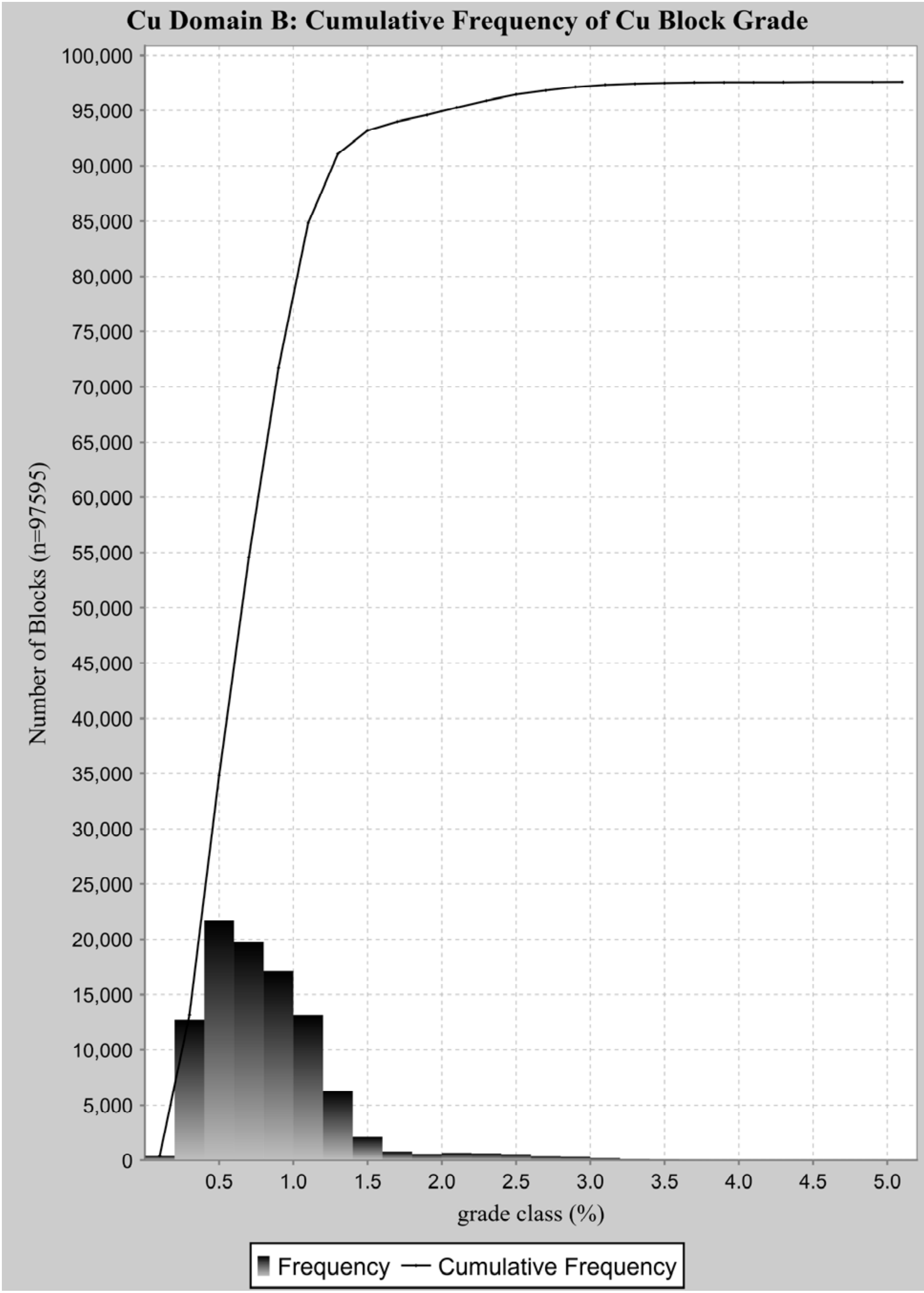
Cu Domain A: Cumulative Frequency Plot of Cu Block Grades
Number of Blocks (n=120653)
120,000
115,000
110,000
105,000
100,000
95,000
90,000
85,000
80,000
75,000
70,000
65,000
60,000
55,000
50,000
45,000
40,000
35,000
30,000
25,000
20,000
15,000
10,000
5,000
0
0.0 0.5 1.0 1.5 2.0 2.5 3.0 3.5 4.0 4.5 5.0 5.5 6.0 6.5 7.0 7.5
grade class (%)
Frequency
Cumulative Frequency

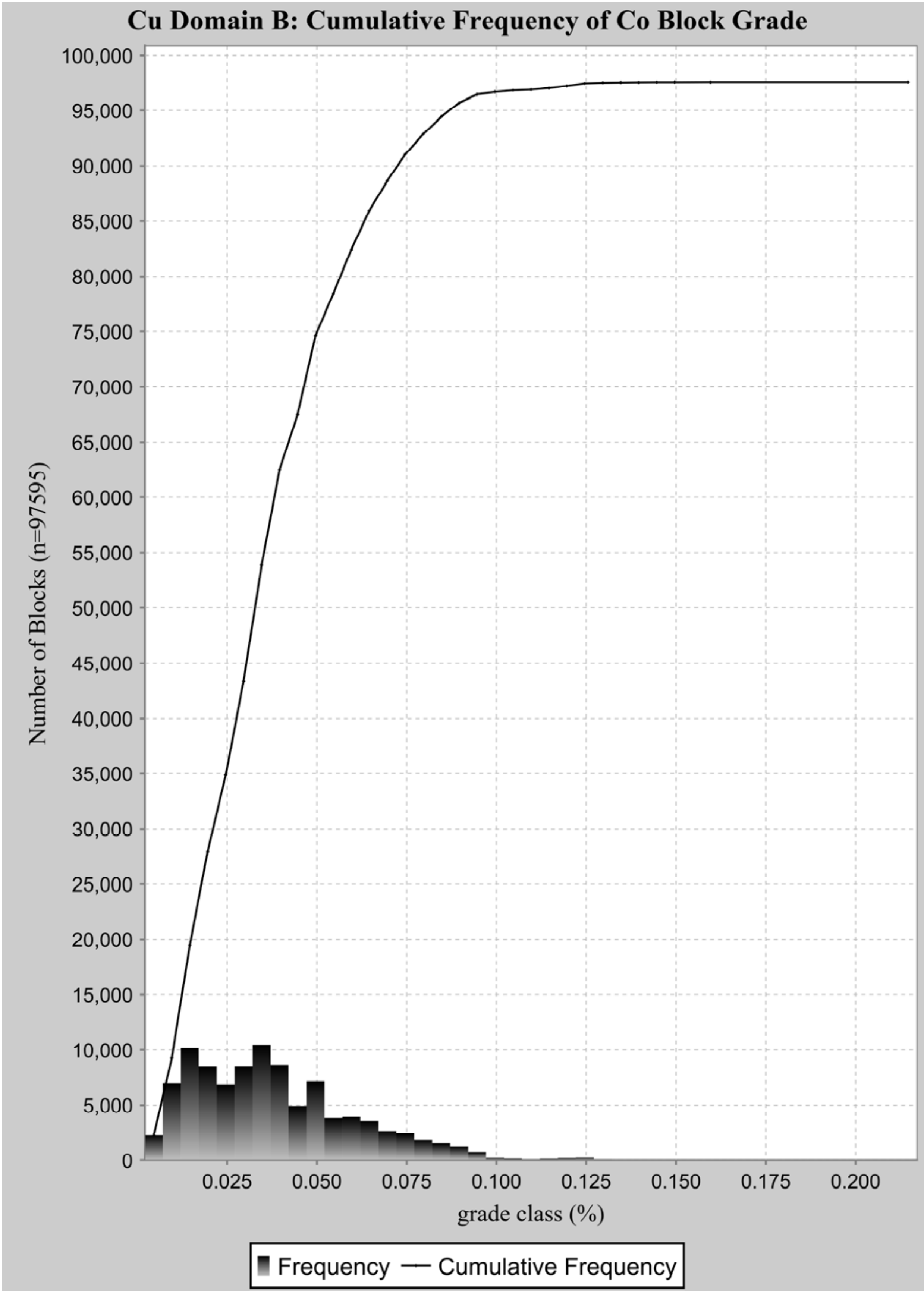


Cu Domain A: Cumulative Frequency of Co Block Grade
Number of Blocks (n=120653)
120,000
115,000
110,000
105,000
100,000
95,000
90,000
85,000
80,000
75,000
70,000
65,000
60,000
55,000
50,000
45,000
40,000
35,000
30,000
25,000
20,000
15,000
10,000
5,000
0
0.000
0.025
0.050
0.075
0.100
0.125
0.150
0.175
grade class (%)
Frequency
Cumulative Frequency

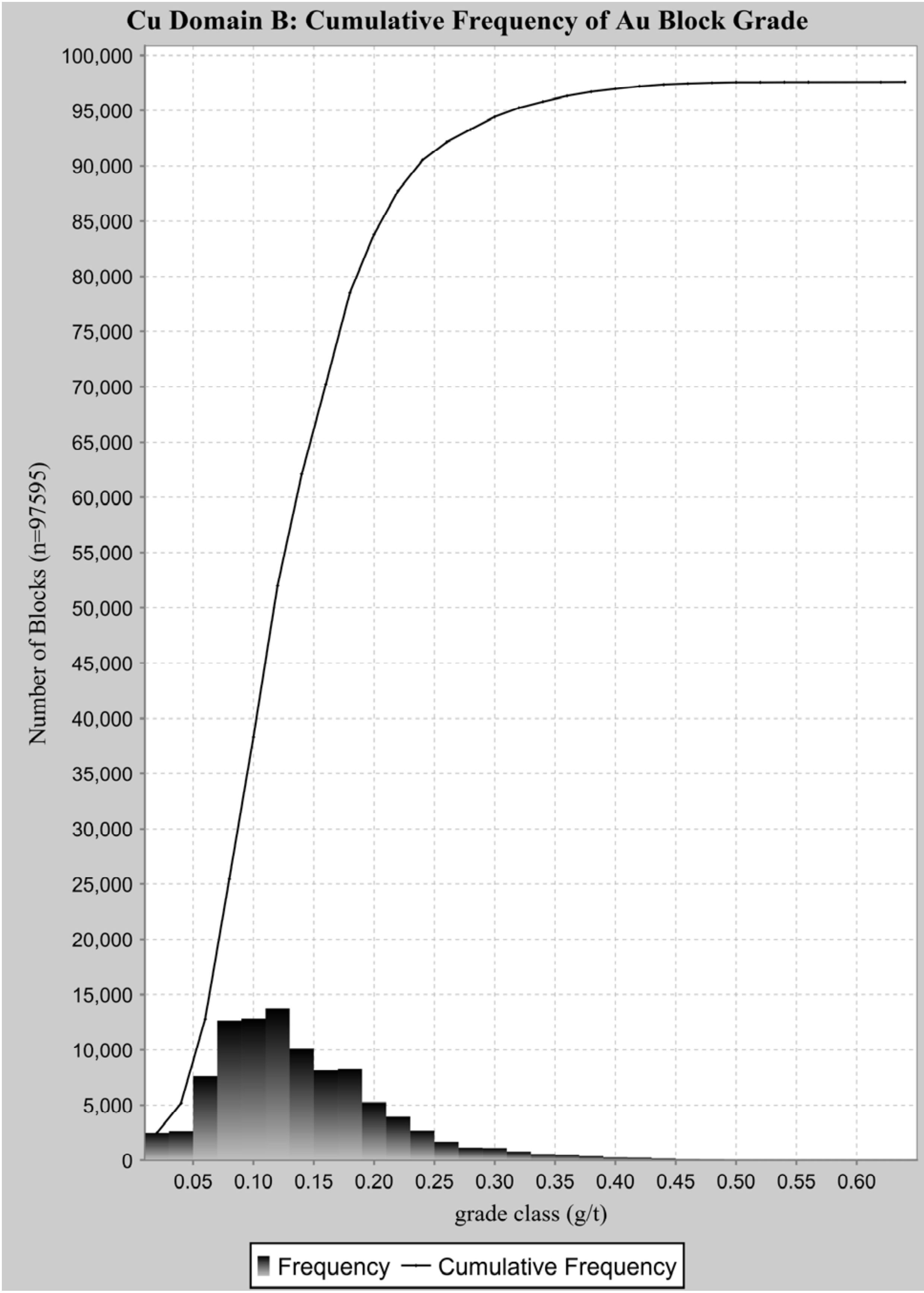


Cu Domain A: Cumulative Frequency of Au Block Grade
Number of Blocks (n=120653)
120,000
115,000
110,000
105,000
100,000
95,000
90,000
85,000
80,000
75,000
70,000
65,000
60,000
55,000
50,000
45,000
40,000
35,000
30,000
25,000
20,000
15,000
10,000
5,000
0
0.0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
1.0
1.1
1.2
grade class (g/t)
Frequency
Cumulative Frequency

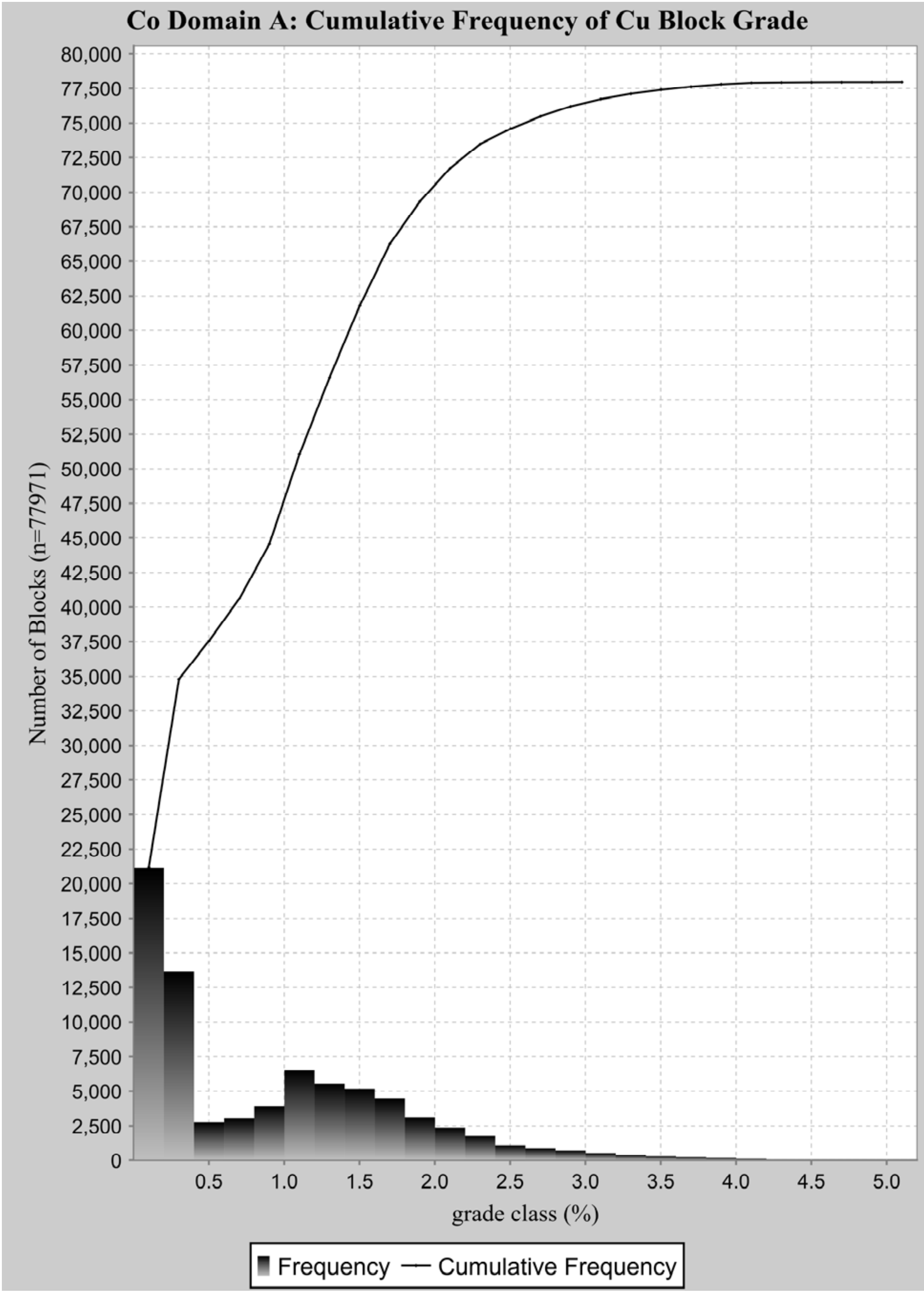

Cu Domain B: Cumulative Frequency of Cu Block Grade
Number of Blocks (n=97595)
100,000
95,000
90,000
85,000
80,000
75,000
70,000
65,000
60,000
55,000
50,000
45,000
40,000
35,000
30,000
25,000
20,000
15,000
10,000
5,000
0
0.5
1.0
1.5
2.0
2.5
3.0
3.5
4.0
4.5
5.0
grade class (%)
Frequency
Cumulative Frequency



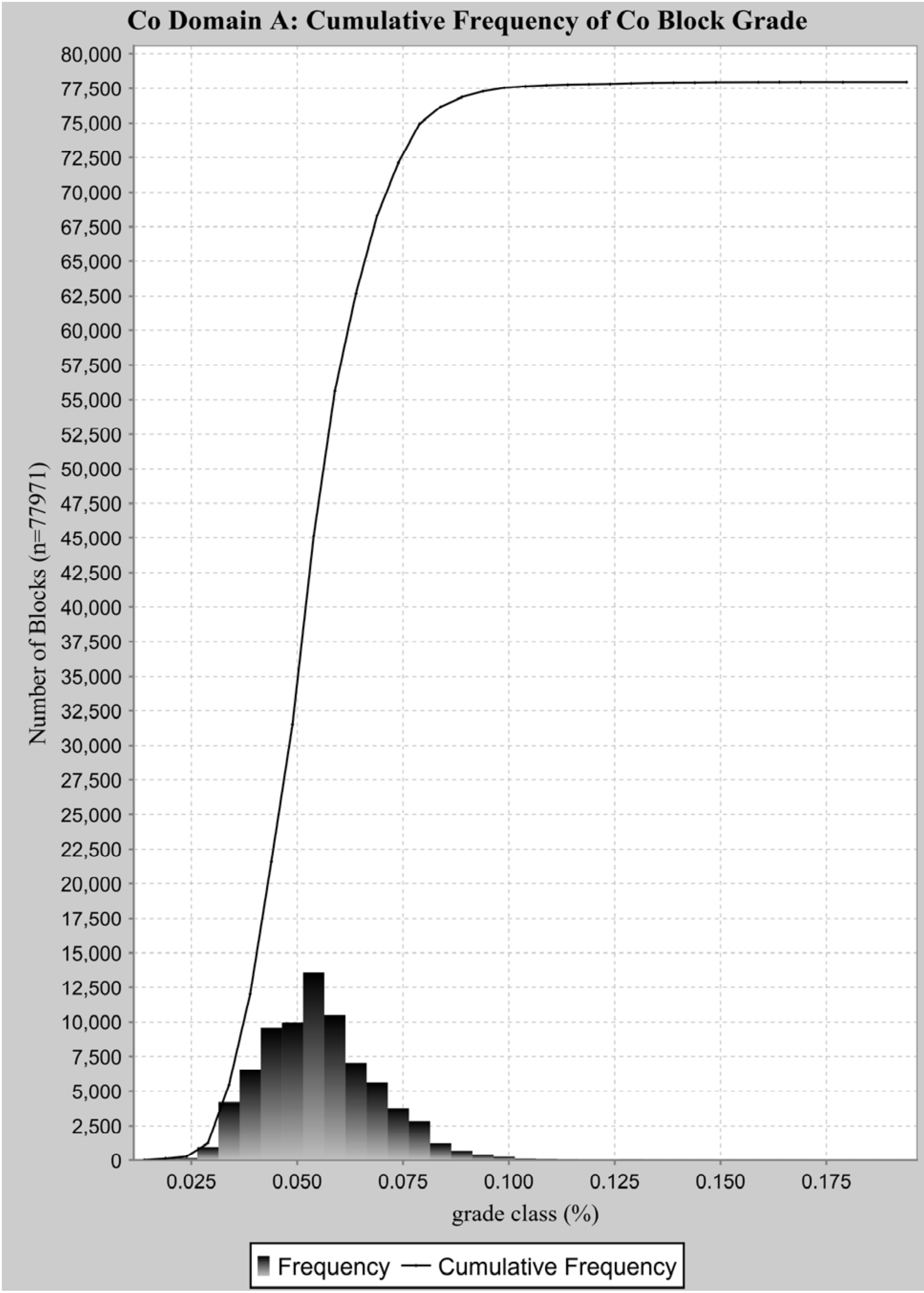
Cu Domain B: Cumulative Frequency of Co Block Grade
Number of Blocks (n=97595)
100,000
95,000
90,000
85,000
80,000
75,000
70,000
65,000
60,000
55,000
50,000
45,000
40,000
35,000
30,000
25,000
20,000
15,000
10,000
5,000
0
0.025
0.050
0.075
0.100
0.125
0.150
0.175
0.200
grade class (%)
Frequency
Cumulative Frequency

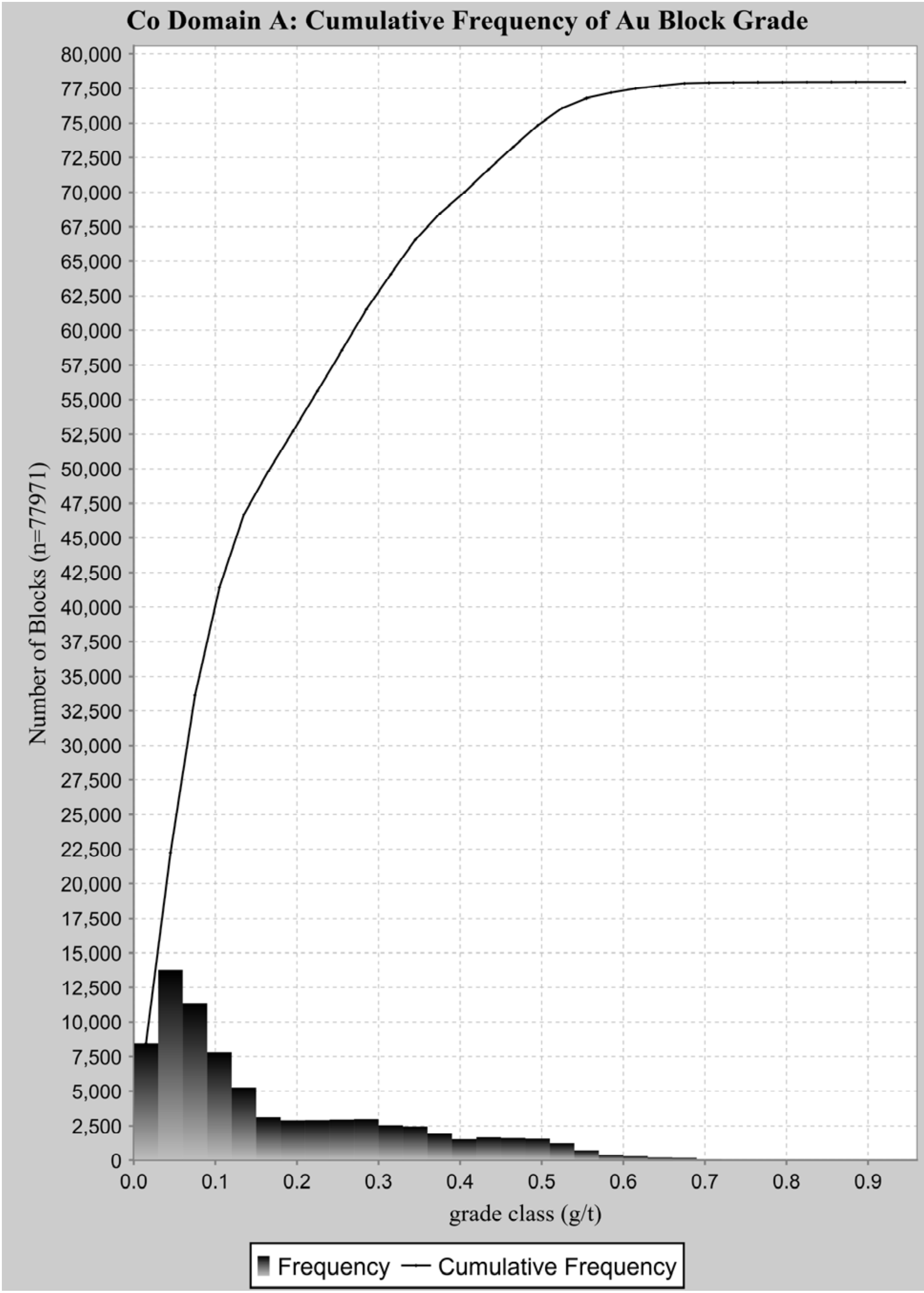


Cu Domain B: Cumulative Frequency of Au Block Grade
Number of Blocks (n=97595)
100,000
95,000
90,000
85,000
80,000
75,000
70,000
65,000
60,000
55,000
50,000
45,000
40,000
35,000
30,000
25,000
20,000
15,000
10,000
5,000
0
0.05
0.10
0.15
0.20
0.25
0.30
0.35
0.40
0.45
0.50
0.55
0.60
grade class (g/t)
Frequency
Cumulative Frequency

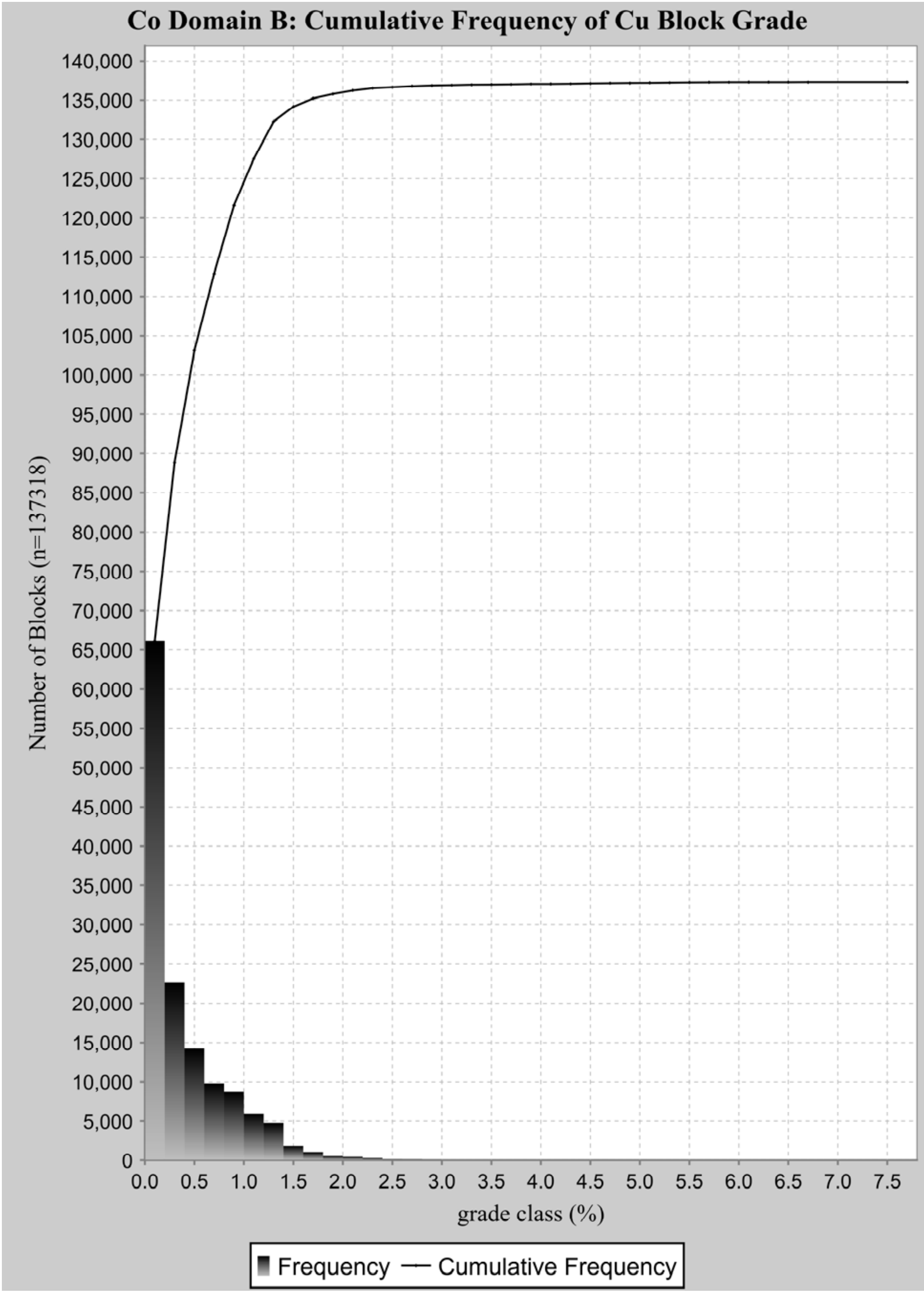


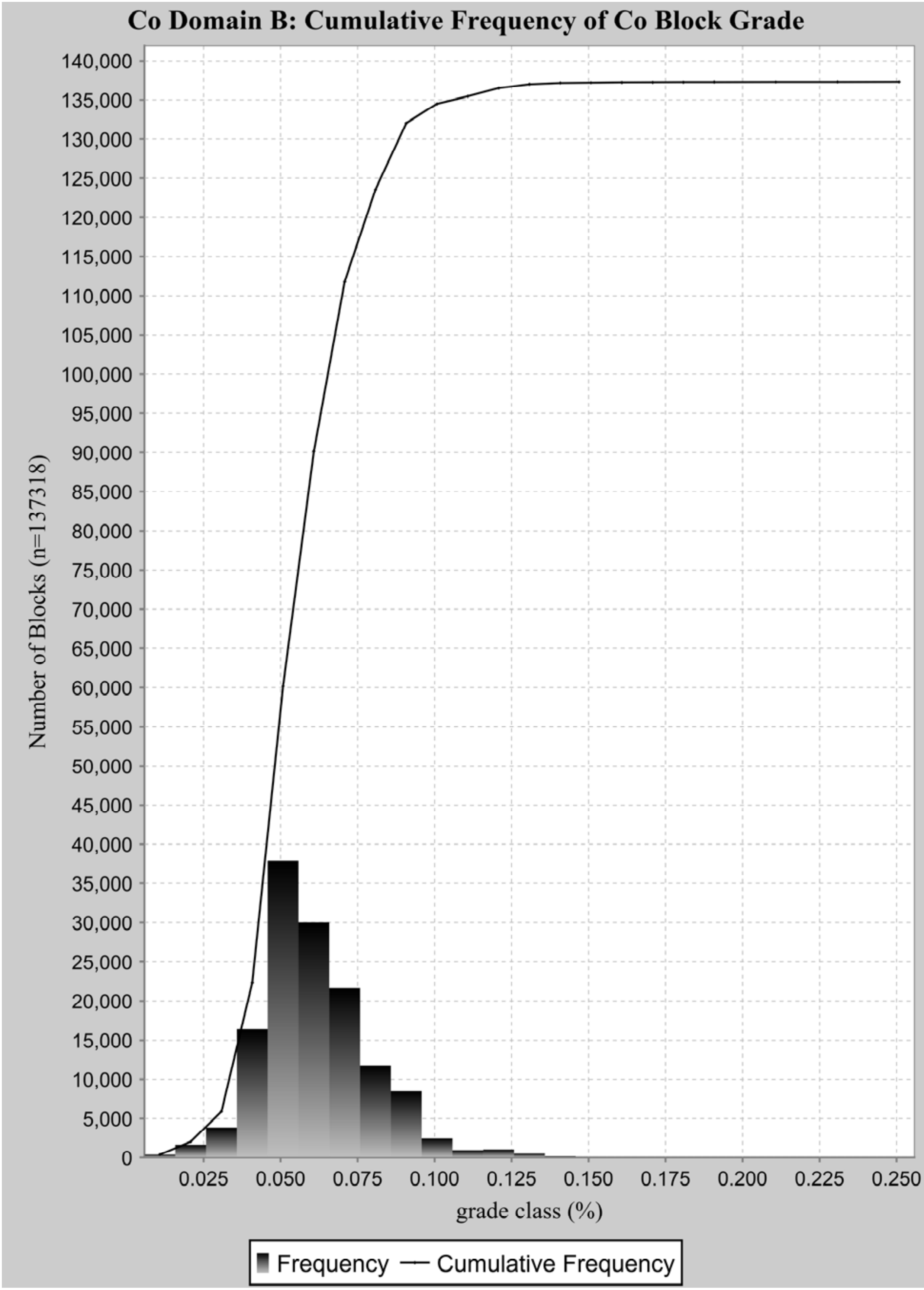
Co Domain A: Cumulative Frequency of Cu Block Grade
Number of Blocks (n=77971)
80,000
77,500
75,000
72,500
70,000
67,500
65,000
62,500
60,000
57,500
55,000
52,500
50,000
47,500
45,000
42,500
40,000
37,500
35,000
32,500
30,000
27,500
25,000
22,500
20,000
17,500
15,000
12,500
10,000
7,500
5,000
2,500
0
0.5
1.0
1.5
2.0
2.5
3.0
3.5
4.0
4.5
5.0
grade class (%)
Frequency
Cumulative Frequency


Co Domain A: Cumulative Frequency of Co Block Grade
Number of Blocks (n=77971)
80,000
77,500
75,000
72,500
70,000
67,500
65,000
62,500
60,000
57,500
55,000
52,500
50,000
47,500
45,000
42,500
40,000
37,500
35,000
32,500
30,000
27,500
25,000
22,500
20,000
17,500
15,000
12,500
10,000
7,500
5,000
2,500
0
0.025
0.050
0.075
0.100
0.125
0.150
0.175
grade class (%)
Frequency
Cumulative Frequency

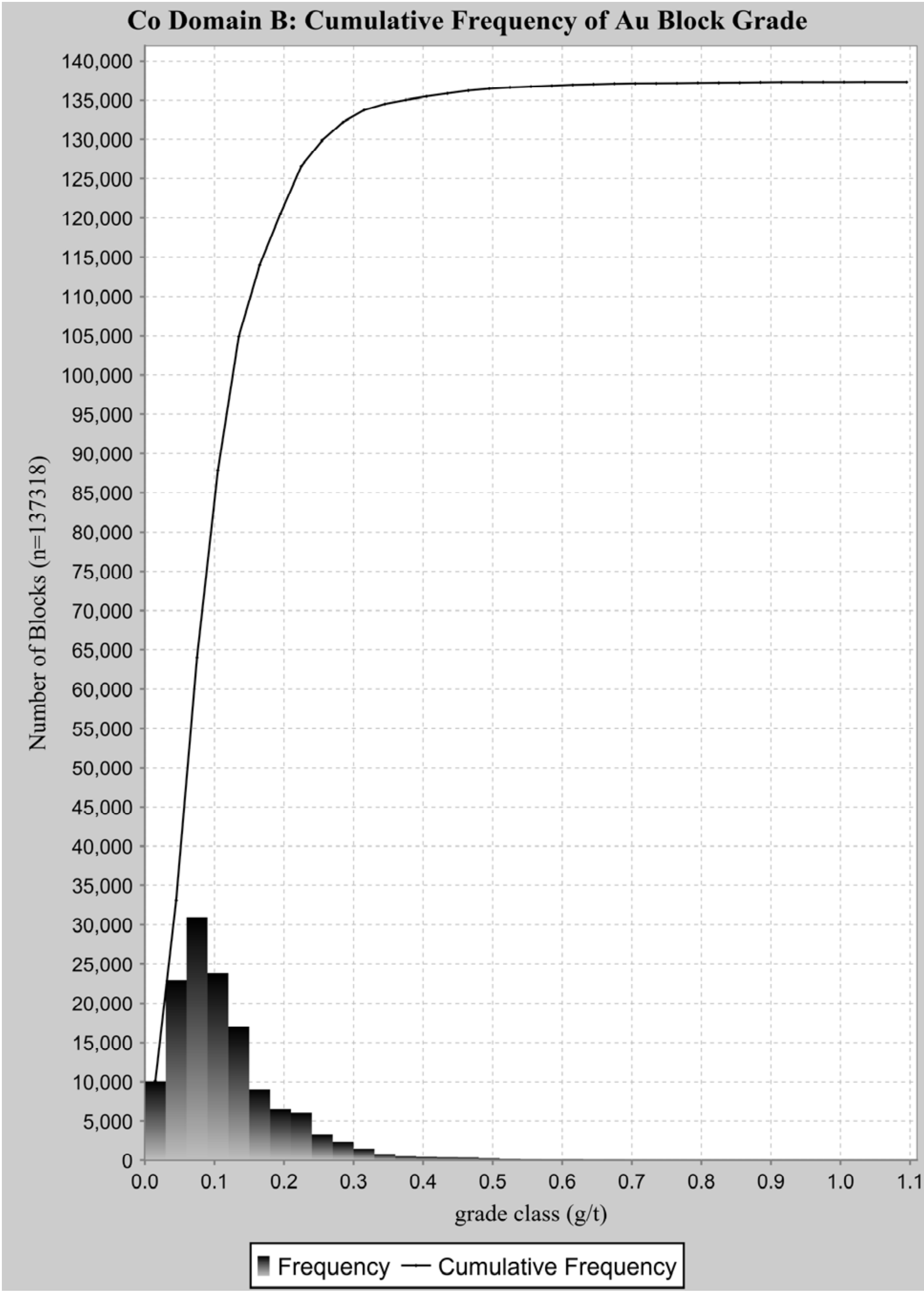


Co Domain A: Cumulative Frequency of Au Block Grade
Number of Blocks (n=77971)
grade class (g/t)
Frequency
Cumulative Frequency
0
2,500
5,000
7,500
10,000
12,500
15,000
17,500
20,000
22,500
25,000
27,500
30,000
32,500
35,000
37,500
40,000
42,500
45,000
47,500
50,000
52,500
55,000
57,500
60,000
62,500
65,000
67,500
70,000
72,500
75,000
77,500
80,000
0.0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9

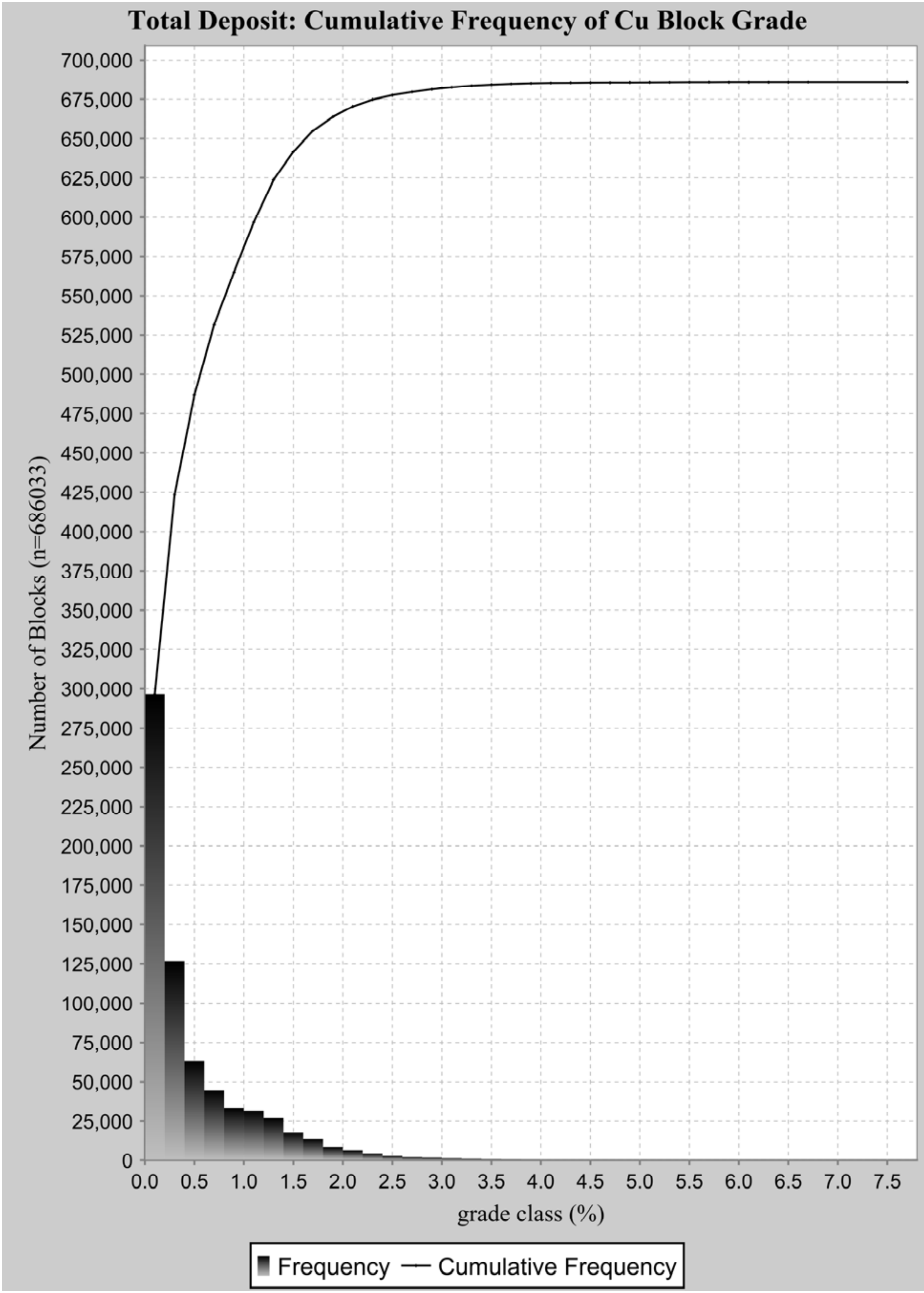

Co Domain B: Cumulative Frequency of Cu Block Grade
Number of Blocks (n=137318)
0
5,000
10,000
15,000
20,000
25,000
30,000
35,000
40,000
45,000
50,000
55,000
60,000
65,000
70,000
75,000
80,000
85,000
90,000
95,000
100,000
105,000
110,000
115,000
120,000
125,000
130,000
135,000
140,000
0.0 0.5 1.0 1.5 2.0 2.5 3.0 3.5 4.0 4.5 5.0 5.5 6.0 6.5 7.0 7.5
grade class (%)
Frequency
Cumulative Frequency

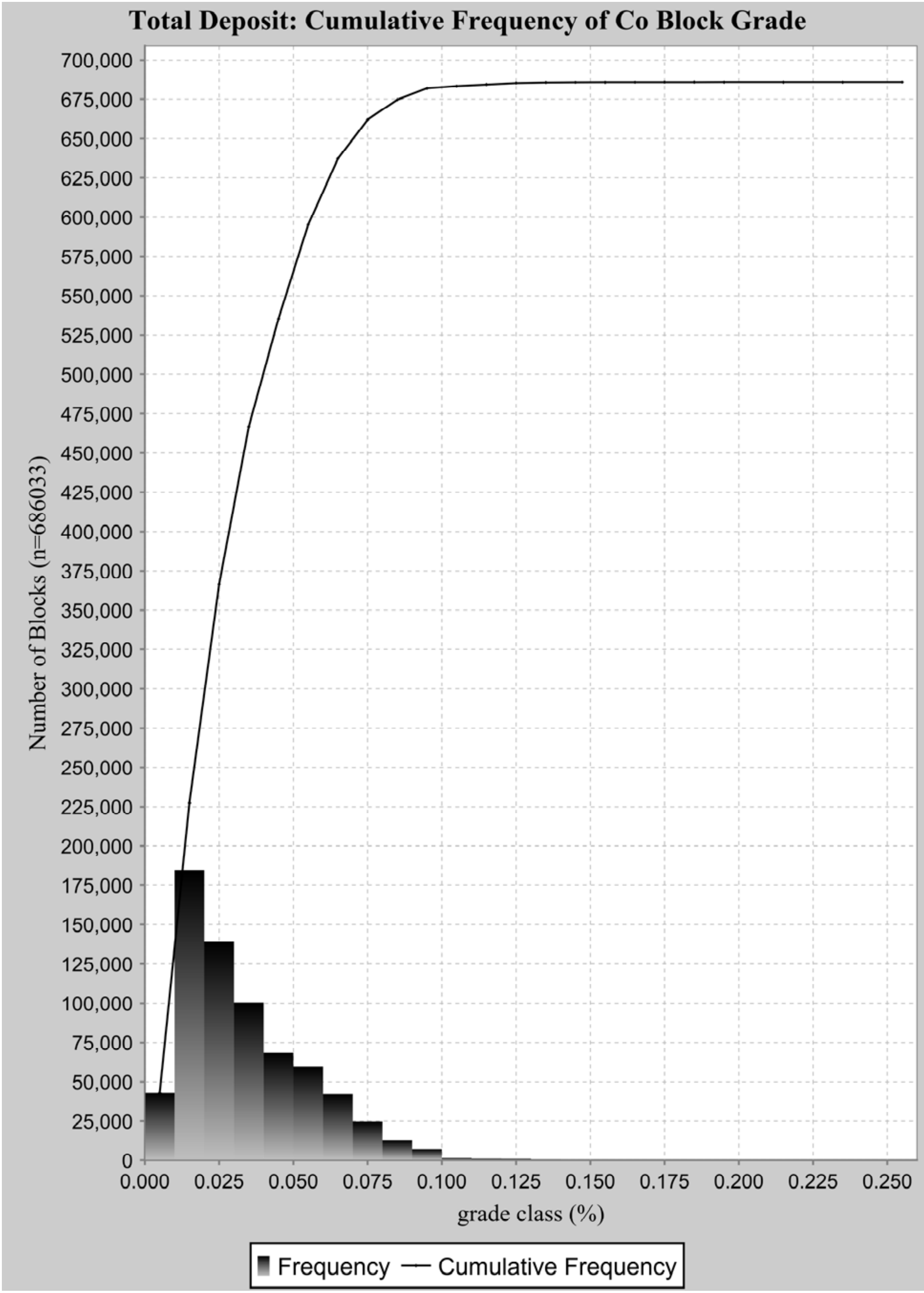


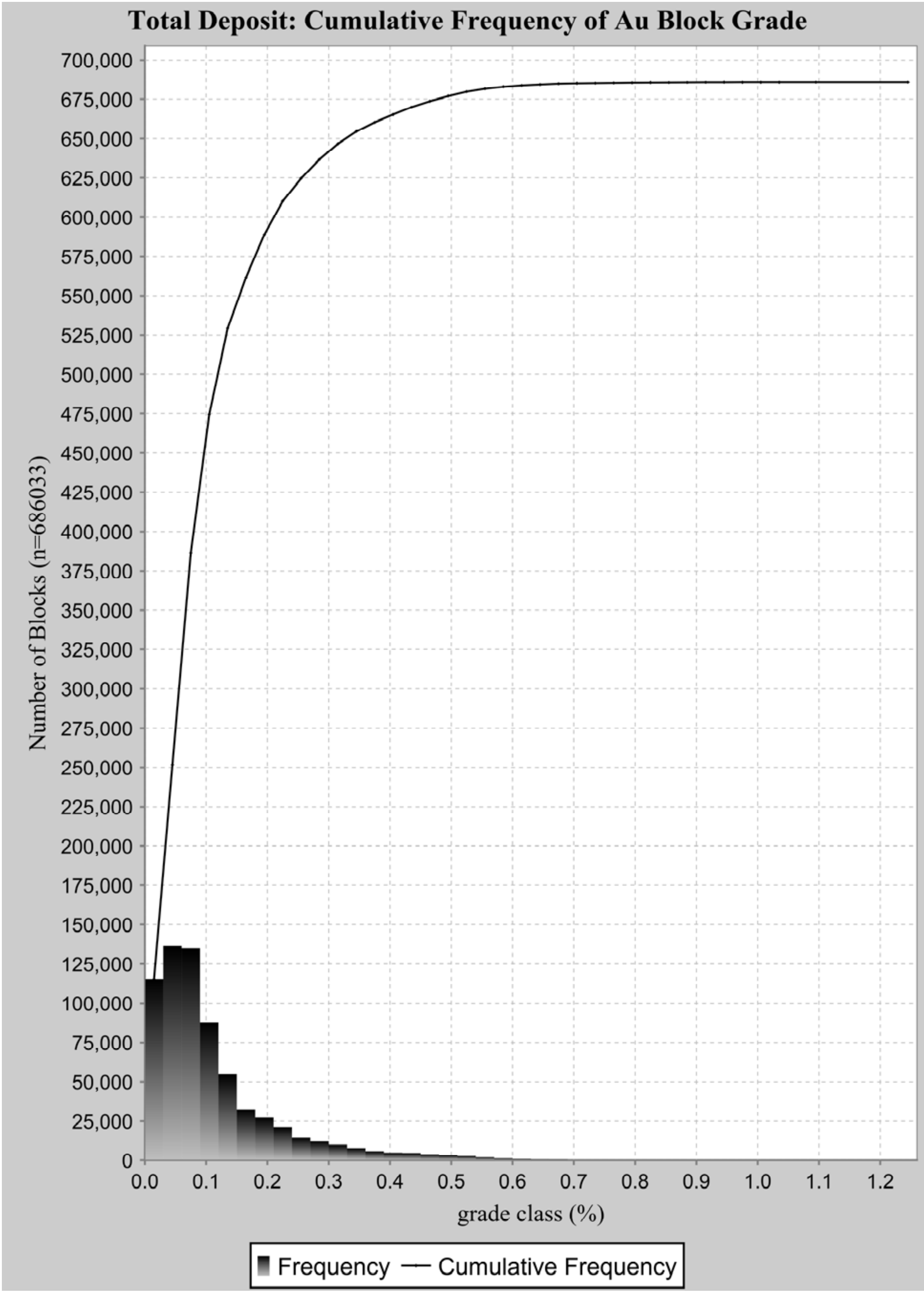
Co Domain B: Cumulative Frequency of Co Block Grade
Number of Blocks (n=137318)
140,000
135,000
130,000
125,000
120,000
115,000
110,000
105,000
100,000
95,000
90,000
85,000
80,000
75,000
70,000
65,000
60,000
55,000
50,000
45,000
40,000
35,000
30,000
25,000
20,000
15,000
10,000
5,000
0
0.025
0.050
0.075
0.100
0.125
0.150
0.175
0.200
0.225
0.250
grade class (%)
Frequency
Cumulative Frequency



Co Domain B: Cumulative Frequency of Au Block Grade
Number of Blocks (n=137318)
140,000
135,000
130,000
125,000
120,000
115,000
110,000
105,000
100,000
95,000
90,000
85,000
80,000
75,000
70,000
65,000
60,000
55,000
50,000
45,000
40,000
35,000
30,000
25,000
20,000
15,000
10,000
5,000
0
0.0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
1.0
1.1
grade class (g/t)
Frequency
Cumulative Frequency

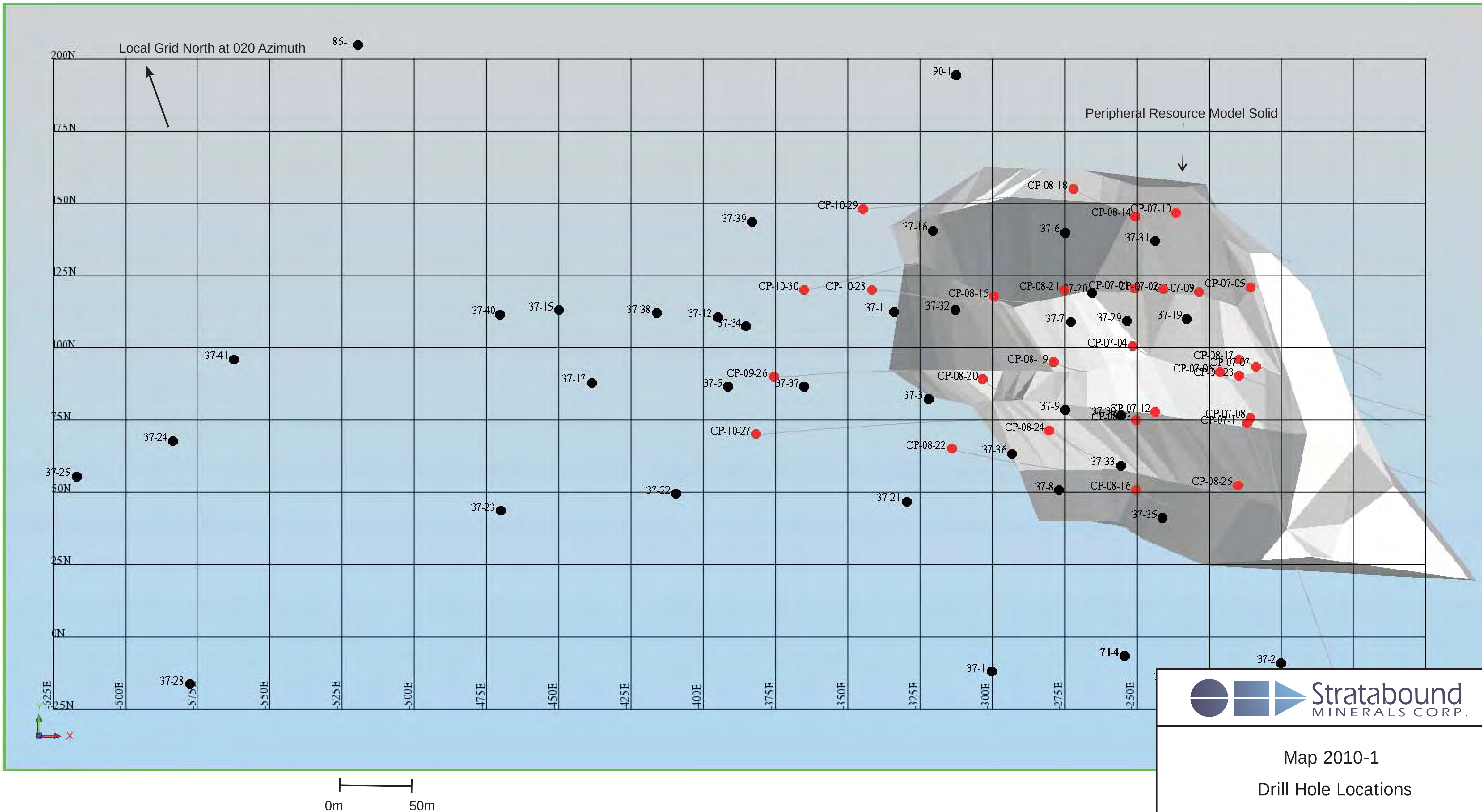

Total Deposit: Cumulative Frequency of Cu Block Grade
Number of Blocks (n=686033)
700,000
675,000
650,000
625,000
600,000
575,000
550,000
525,000
500,000
475,000
450,000
425,000
400,000
375,000
350,000
325,000
300,000
275,000
250,000
225,000
200,000
175,000
150,000
125,000
100,000
75,000
50,000
25,000
0
0.0 0.5 1.0 1.5 2.0 2.5 3.0 3.5 4.0 4.5 5.0 5.5 6.0 6.5 7.0 7.5
grade class (%)
Frequency
Cumulative Frequency


Total Deposit: Cumulative Frequency of Co Block Grade
Number of Blocks (n=686033)
700,000
675,000
650,000
625,000
600,000
575,000
550,000
525,000
500,000
475,000
450,000
425,000
400,000
375,000
350,000
325,000
300,000
275,000
250,000
225,000
200,000
175,000
150,000
125,000
100,000
75,000
50,000
25,000
0
0.000
0.025
0.050
0.075
0.100
0.125
0.150
0.175
0.200
0.225
0.250
grade class (%)
Frequency
Cumulative Frequency


Total Deposit: Cumulative Frequency of Au Block Grade
Number of Blocks (n=686033)
700,000
675,000
650,000
625,000
600,000
575,000
550,000
525,000
500,000
475,000
450,000
425,000
400,000
375,000
350,000
325,000
300,000
275,000
250,000
225,000
200,000
175,000
150,000
125,000
100,000
75,000
50,000
25,000
0
0.0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
1.0
1.1
1.2
grade class (%)
Frequency
Cumulative Frequency

# APPENDIX E

## Captain Plans and Sections



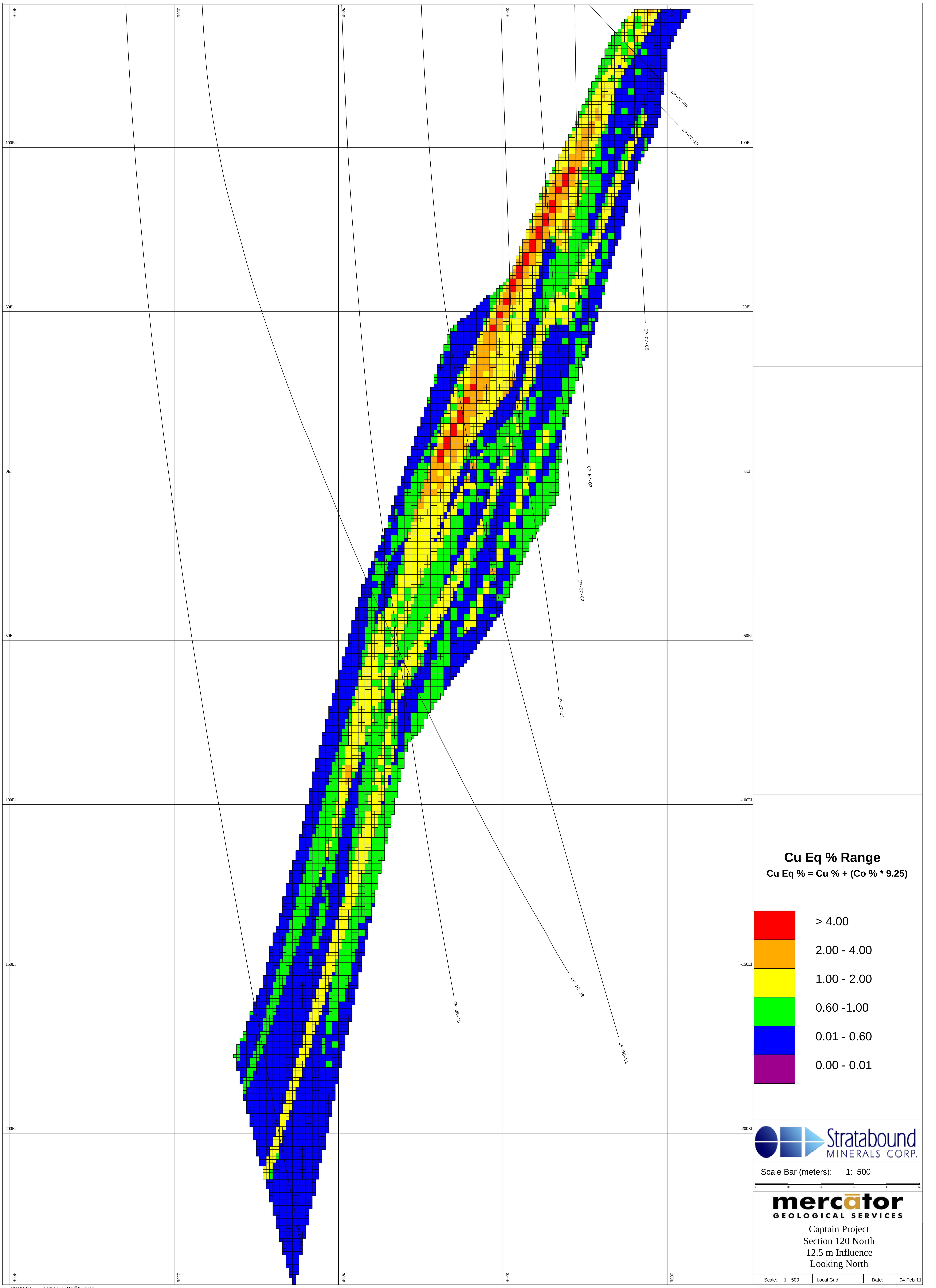

Stratabound MINERALS CORP.

Map 2010-1

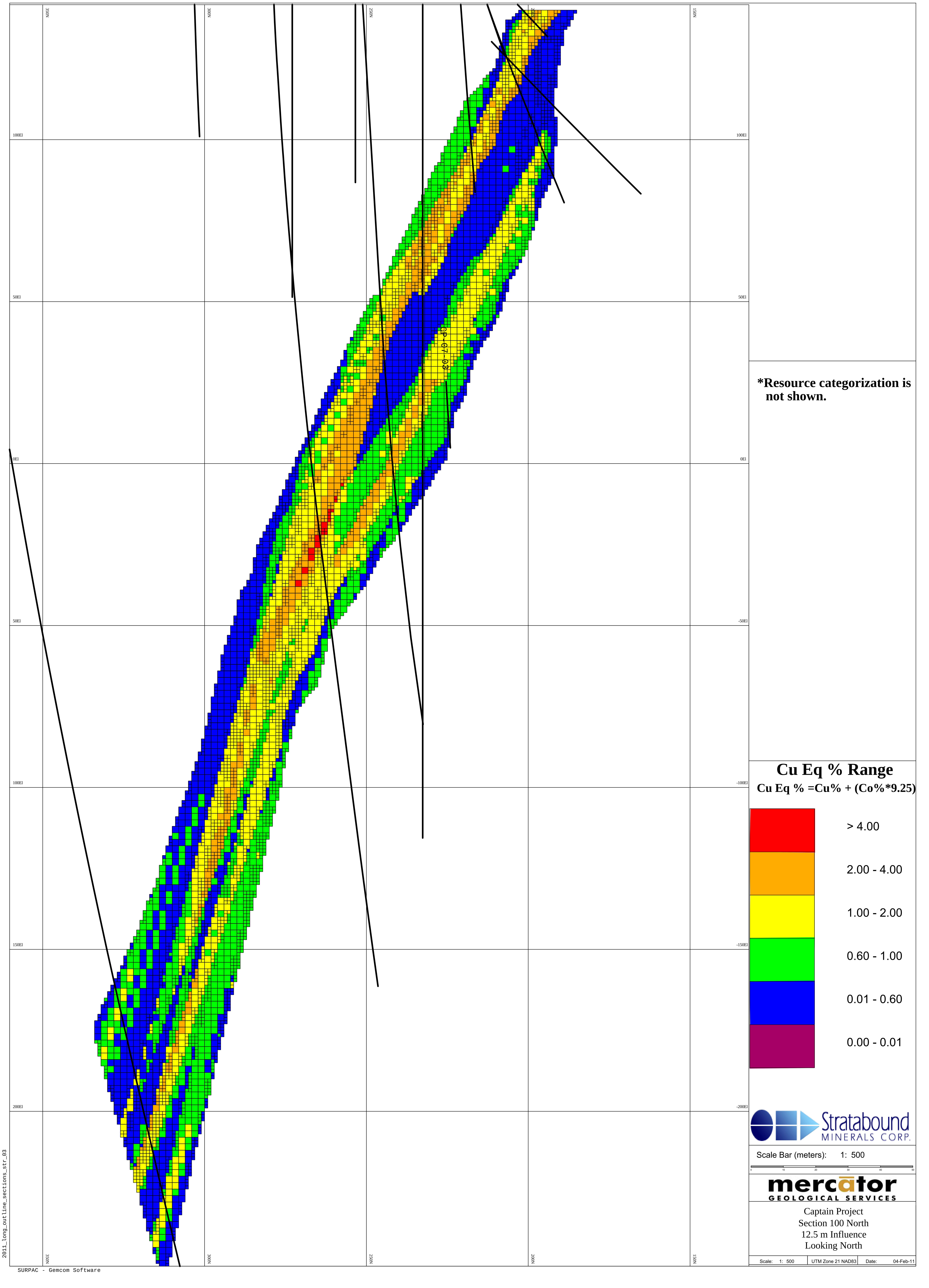
Drill Hole Locations

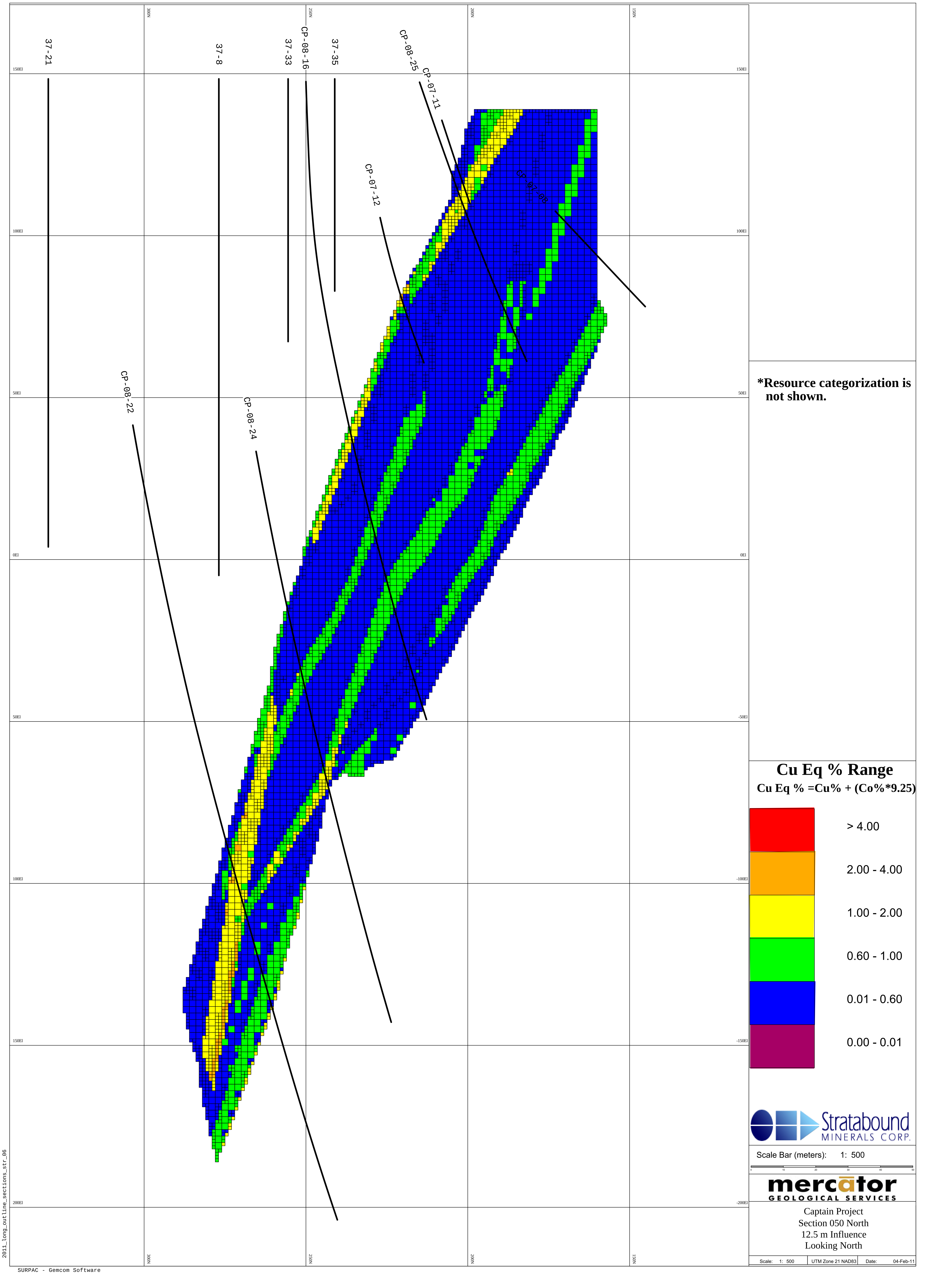
Date: Feb. 2011

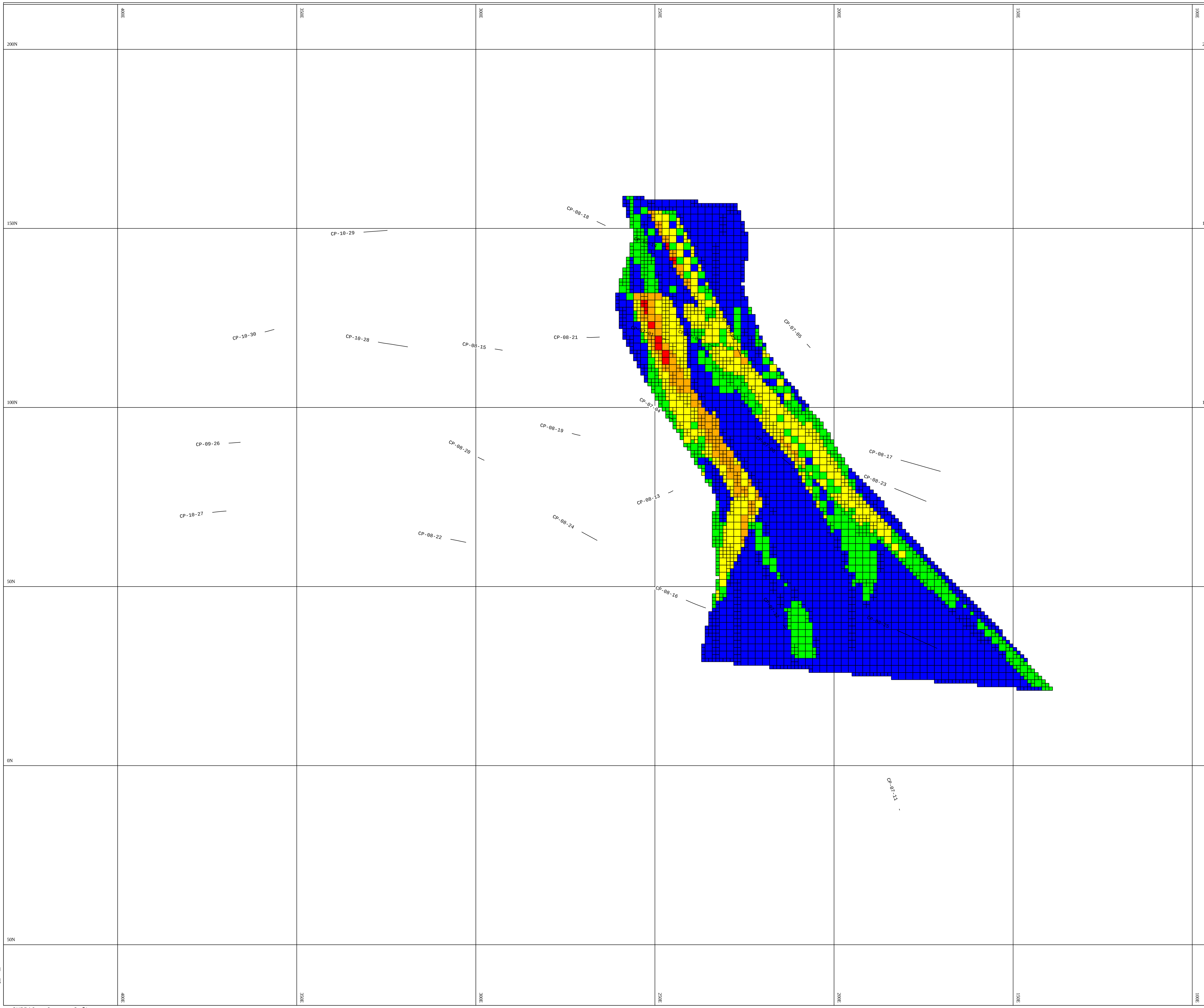
mercator GEOLOGICAL SERVICES



SURPAC - Gemcom Software


*Resource categorization is not shown.
Cu Eq % Range
Cu Eq % =Cu% + (Co%*9.25)
> 4.00
2.00 - 4.00
1.00 - 2.00
0.60 - 1.00
0.01 - 0.60
0.00 - 0.01
Scale Bar (meters): 1: 500
Captain Project
Section 100 North
12.5 m Influence
Looking North
Scale: 1: 500
UTM Zone 21 NAD83
Date: 04-Feb-11
SURPAC - Gemcom Software
2011_long_outline_sections_str_03


37-21
37-8
37-33
CP-08-16
37-35
CP-08-25
CP-07-11
CP-07-12
CP-07-08
CP-08-22
CP-08-24
*Resource categorization is not shown.
Cu Eq % Range
Cu Eq % =Cu% + (Co%*9.25)
> 4.00
2.00 - 4.00
1.00 - 2.00
0.60 - 1.00
0.01 - 0.60
0.00 - 0.01
Stratabound MINERALS CORP.
Scale Bar (meters): 1: 500
mercator GEOLOGICAL SERVICES
Captain Project
Section 050 North
12.5 m Influence
Looking North
Scale: 1: 500
UTM Zone 21 NAD83
Date: 04-Feb-11
2011_long_outline_sections_str_06
SURPAC - Gemcom Software




SURPAC - Gemcom Software

**Cu Eq % Range**

**Cu Eq % = Cu % + (Co % * 9.25)**

- > 4.00
- 2.00 - 4.00
- 1.00 - 2.00
- 0.60 -1.00
- 0.01 - 0.60
- 0.00 - 0.01

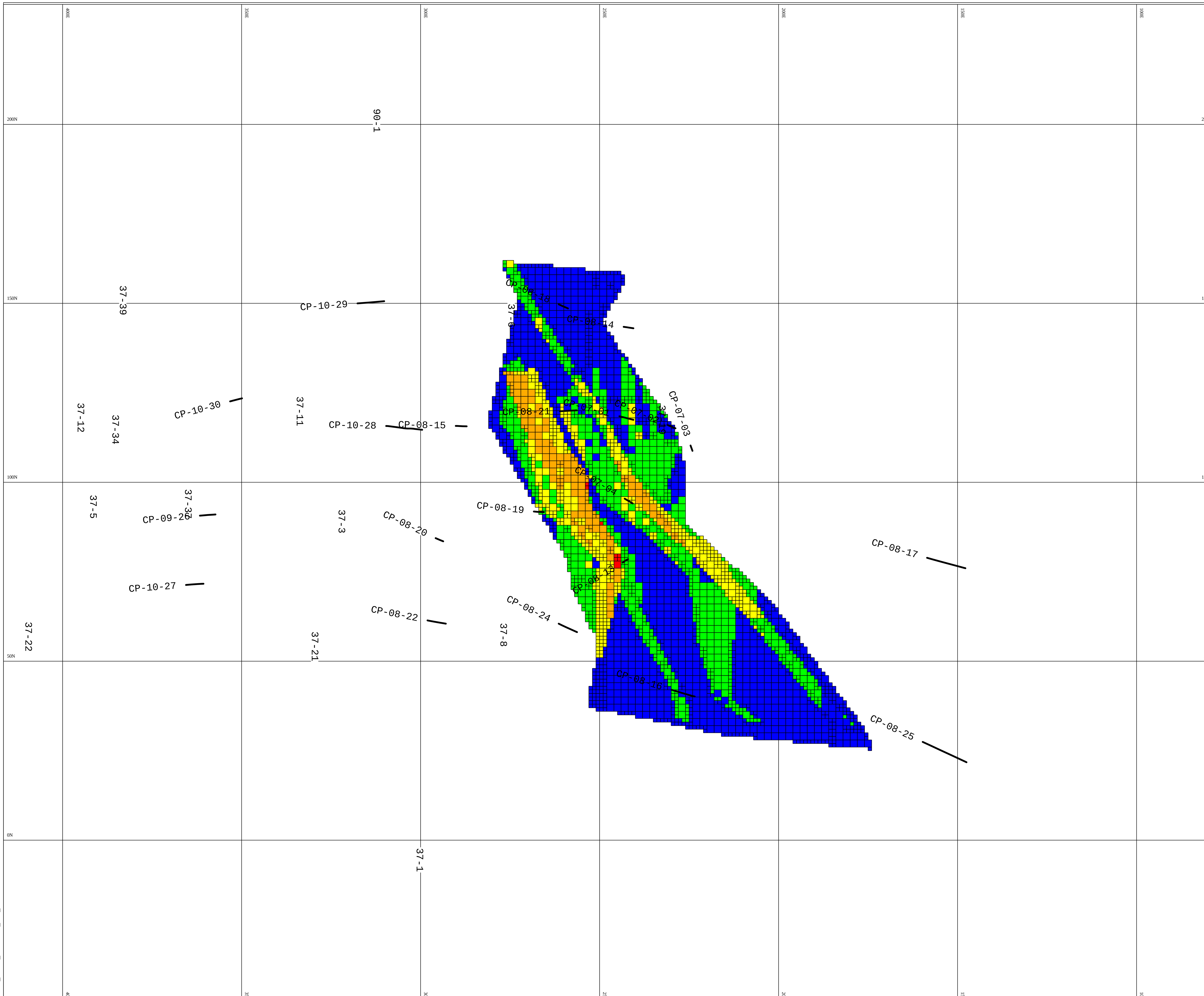
Stratabound MINERALS CORP.

Scale Bar (meters): 1: 500

mercator GEOLOGICAL SERVICES

Stratabound Minerals Corp
Captain Project
Plan View
Level Plan -50 m El

| Scale: 1: 500 | Local Grid | Date: 04-Feb-11 |
|---|---|---|




*Resource categorization is not shown.

**Cu Eq % Range**

**Cu Eq % =Cu% + (Co%*9.25)**

| | |
|---|---|
| red | > 4.00 |
| orange | 2.00 - 4.00 |
| yellow | 1.00 - 2.00 |
| green | 0.60 - 1.00 |
| blue | 0.01 - 0.60 |
| purple | 0.00 - 0.01 |

Stratabound MINERALS CORP.

Scale Bar (meters): 1: 500

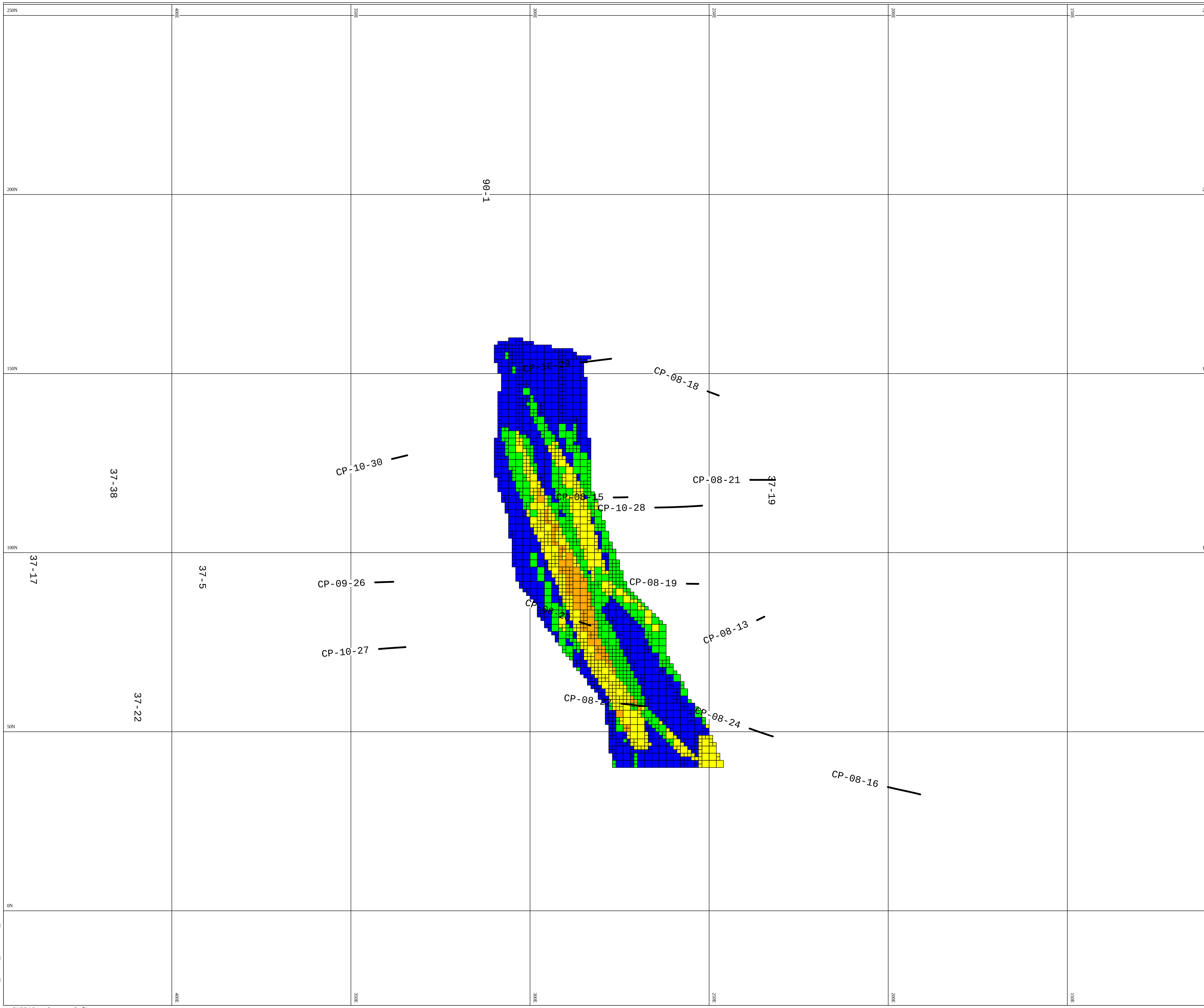
mercator GEOLOGICAL SERVICES

Stratabound Minerals Corp.
Captain Project
Plan View
Level Plan 0 m El

Scale: 1: 500 | Local Grid | Date: 04-Feb-11

2011_level_sections_str_06

SURPAC - Gemcom Software




2011_level_sections_str

SURPAC - Gemcom Software

***Resource categorization is not shown.**

## Cu Eq % Range

**Cu Eq % =Cu% + (Co%*9.25)**

- > 4.00
- 2.00 - 4.00
- 1.00 - 2.00
- 0.60 - 1.00
- 0.01 - 0.60
- 0.00 - 0.01

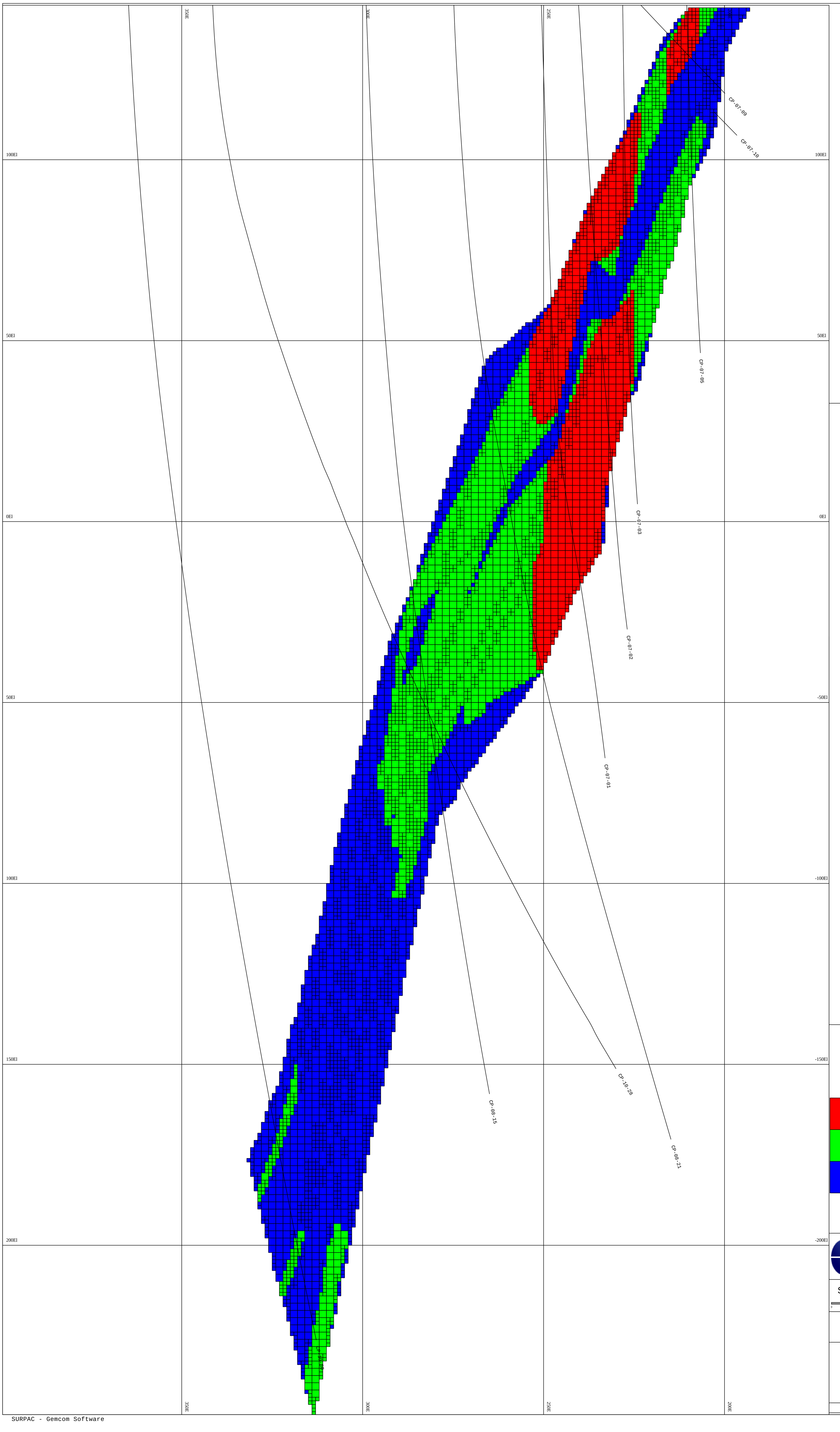
Stratabound MINERALS CORP.

Scale Bar (meters): 1: 500

mercator GEOLOGICAL SERVICES

Stratabound Minerals Corp.
Captain Project
Plan View
Level Plan -100 m El

| Scale: 1: 500 | Local Grid | Date: 04-Feb-11 |
|---|---|---|




**Resource Categorization**

Measured

Indicated

Inferred

Stratabound Minerals Corp.

Scale Bar (meters): 1: 500

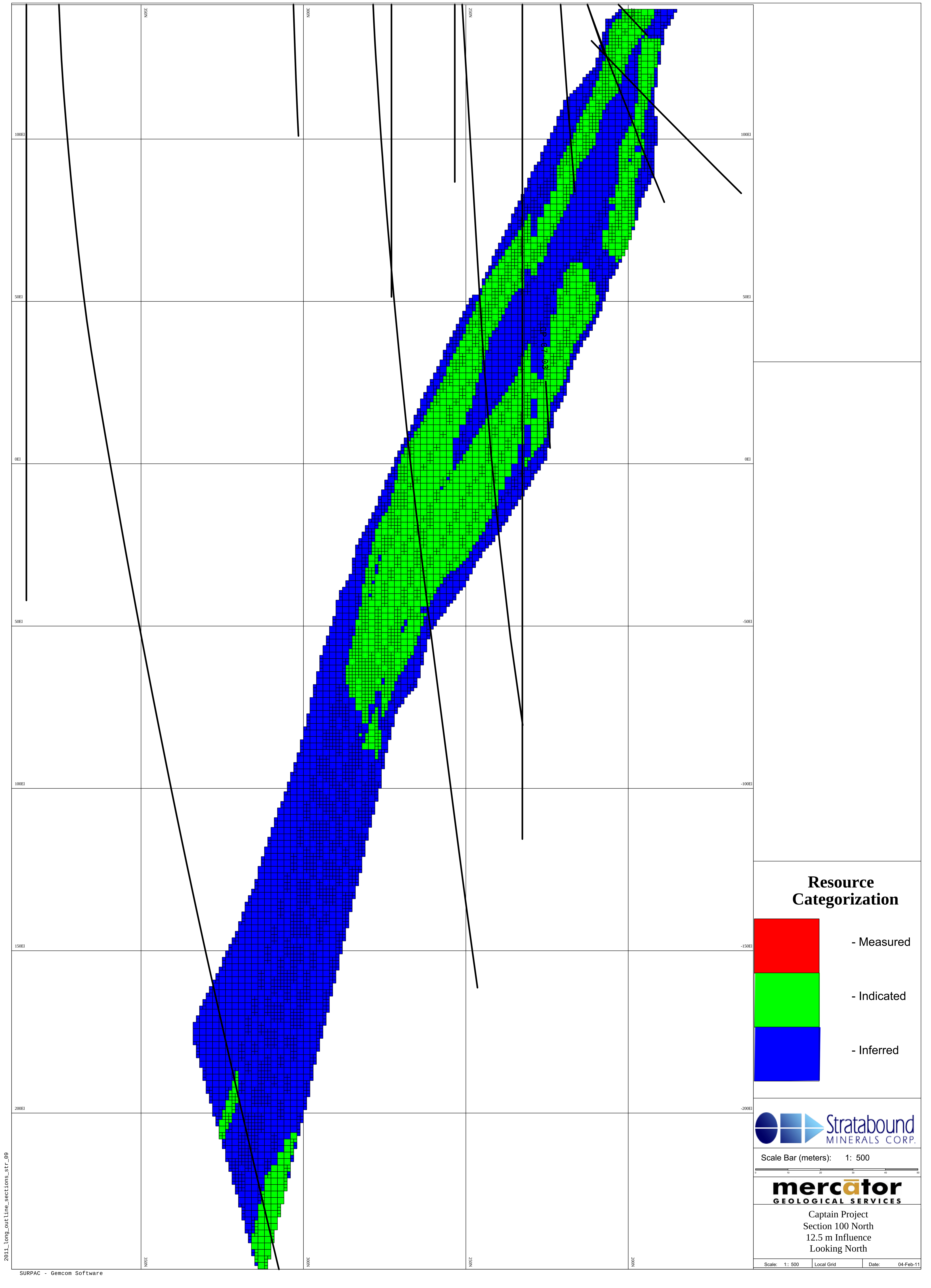
mercator GEOLOGICAL SERVICES

Captain Project
Section 120 North
12.5 m Influence
Looking North

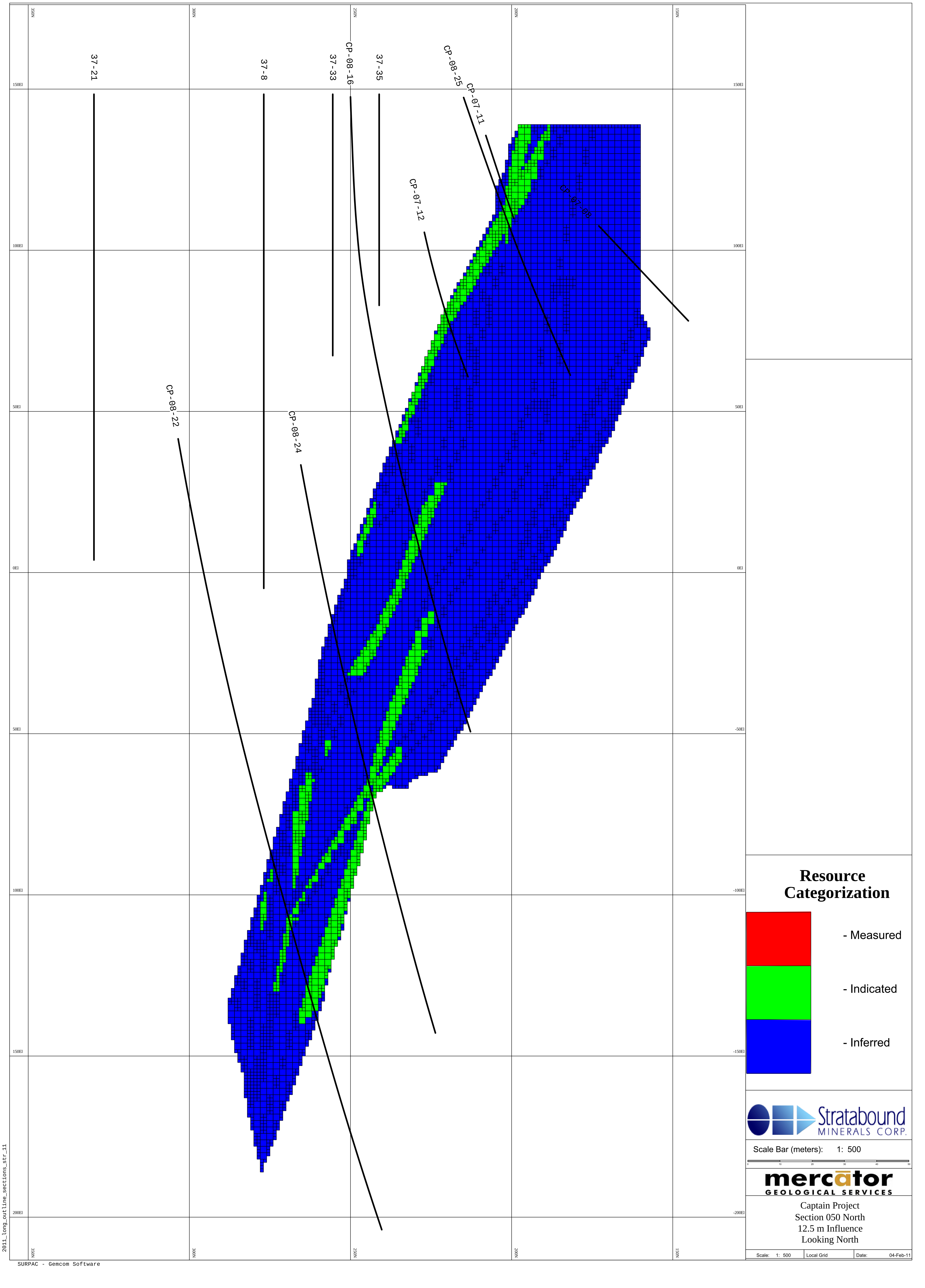
| | | |
|---|---|---|
| Scale: 1: 500 | Local Grid | Date: 04-Feb-11 |

categ_120_section

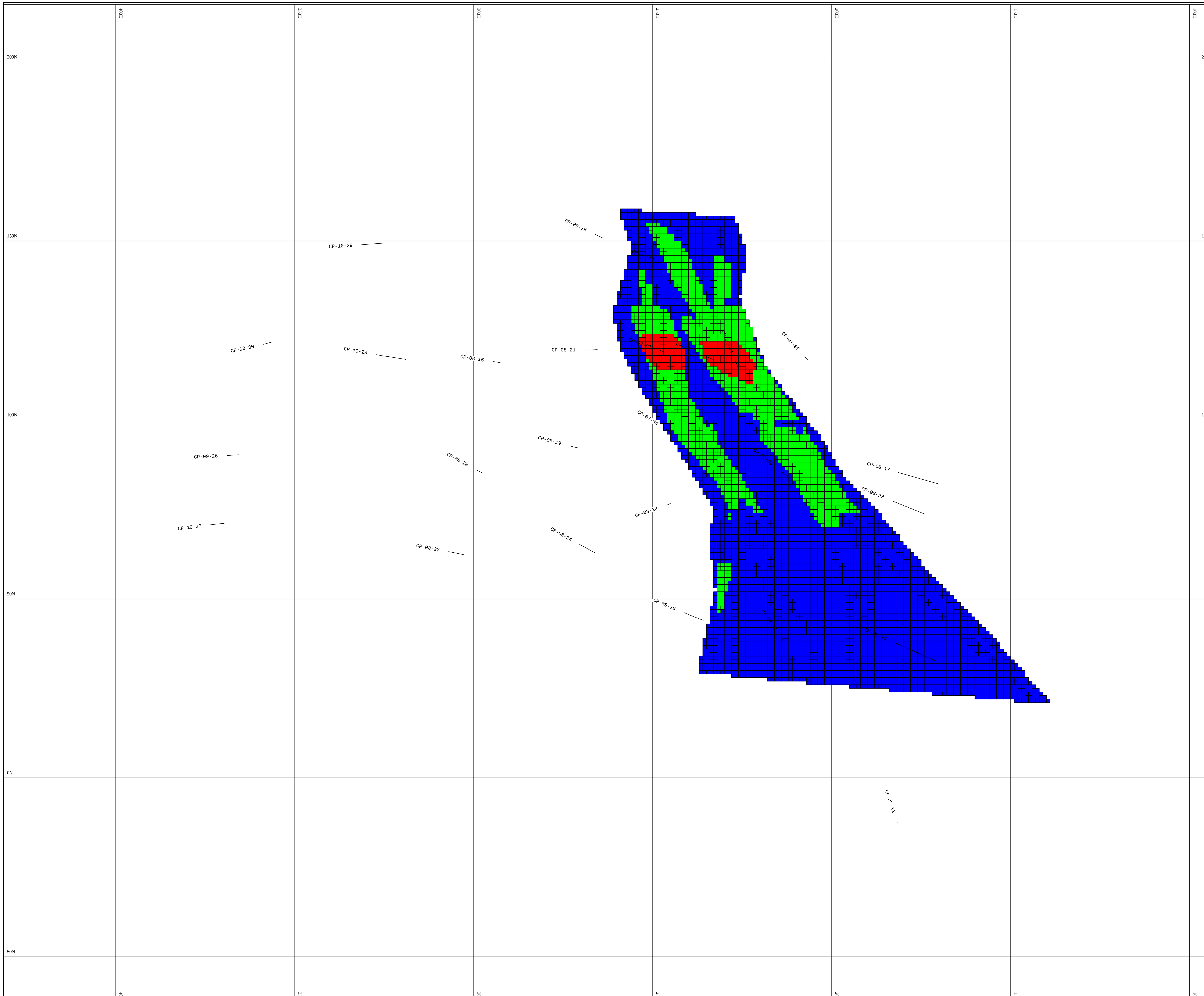
SURPAC - Gemcom Software



Resource Categorization
- Measured
- Indicated
- Inferred
Stratabound MINERALS CORP.
Scale Bar (meters): 1: 500
mercator GEOLOGICAL SERVICES
Captain Project
Section 100 North
12.5 m Influence
Looking North
Scale: 1:: 500
Local Grid
Date: 04-Feb-11
2011_long_outline_sections_str_09
SURPAC - Gemcom Software



37-21
37-8
37-33
CP-08-16
37-35
CP-08-25
CP-07-11
CP-07-12
CP-08-22
CP-08-24
Resource Categorization
- Measured
- Indicated
- Inferred
Stratabound
MINERALS CORP.
Scale Bar (meters): 1: 500
mercator
GEOLOGICAL SERVICES
Captain Project
Section 050 North
12.5 m Influence
Looking North
Scale: 1: 500
Local Grid
Date: 04-Feb-11
SURPAC - Gemcom Software
2011_long_outline_sections_str_11




SURPAC - Gemcom Software

## Resource Categorization

Measured

Indicated

Inferred

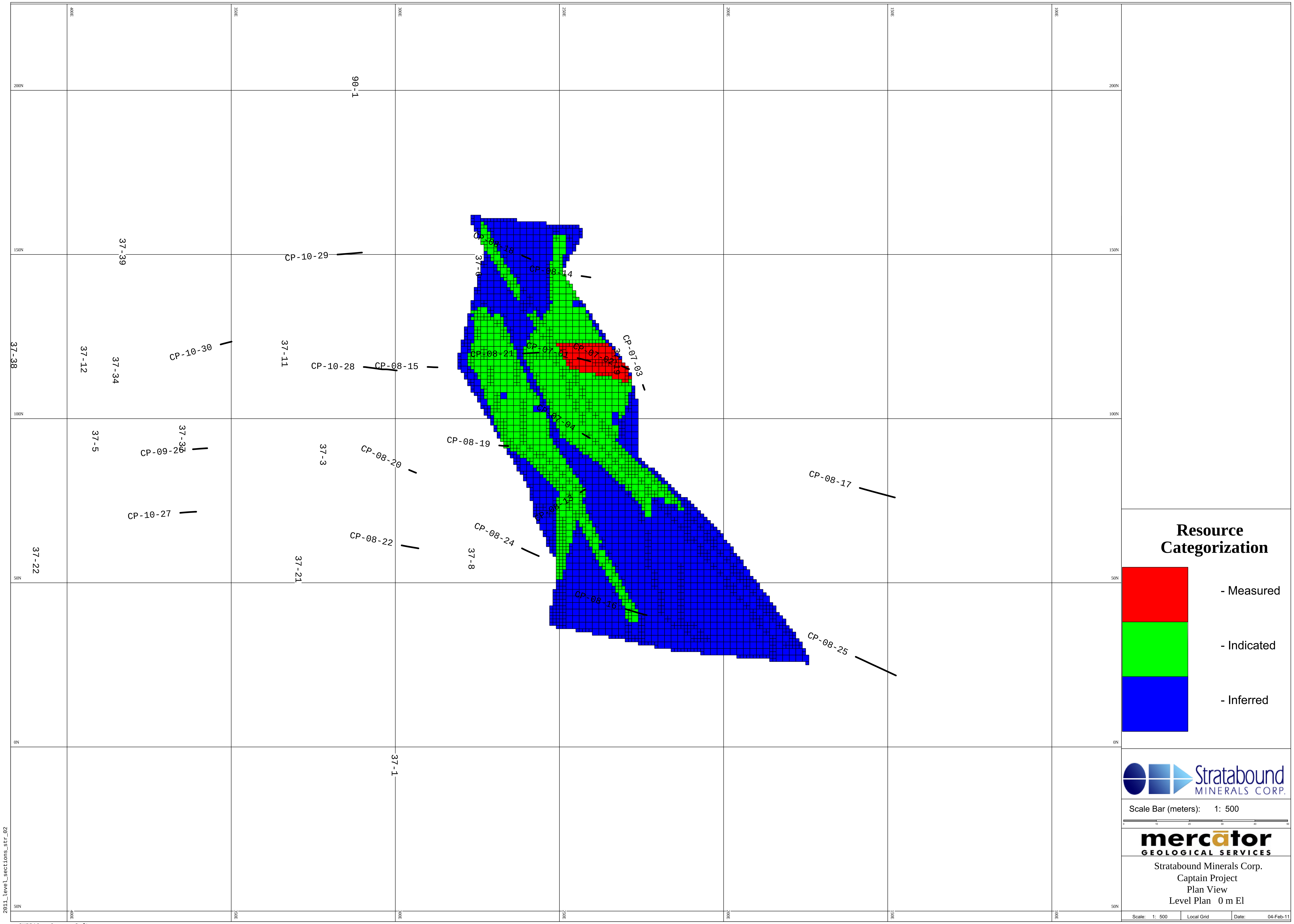
Stratabound MINERALS CORP.

Scale Bar (meters): 1: 500

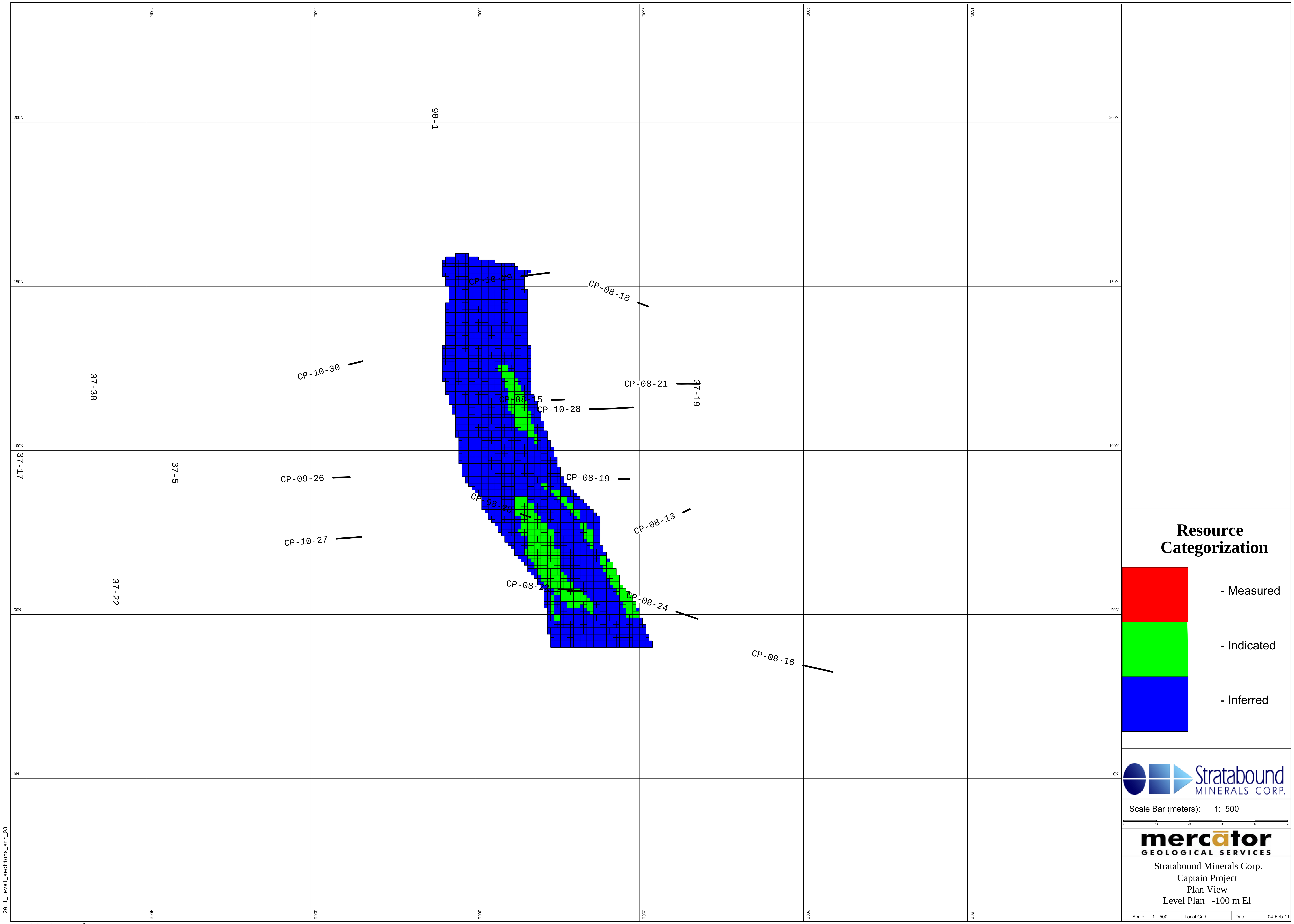
mercator GEOLOGICAL SERVICES

Stratabound Minerals Corp
Captain Project
Plan View
Level Plan 50 m El

| Scale: 1: 500 | Local Grid | Date: 04-Feb-11 |
|---|---|---|



SURPAC - Gemcom Software



90-1
CP-10-29
CP-08-18
CP-10-30
37-38
CP-08-21
37-19
CP-08-15
CP-10-28
37-17
37-5
CP-09-26
CP-08-19
CP-08-20
CP-08-13
CP-10-27
37-22
CP-08-22
CP-08-24
CP-08-16
Resource Categorization
- Measured
- Indicated
- Inferred
Stratabound MINERALS CORP.
Scale Bar (meters): 1: 500
mercator GEOLOGICAL SERVICES
Stratabound Minerals Corp.
Captain Project
Plan View
Level Plan -100 m El
Scale: 1: 500
Local Grid
Date: 04-Feb-11
2011_level_sections_str_03
SURPAC - Gemcom Software


NEW BRUNSWICK
FREDERICTON
CLAIM GROUP
CNE DEPOSIT
CAPTAIN DEPOSIT
CANADIAN NATIONAL RAILWAY (ABANDONED)
SPRIG BROOK GRID
STRATABOUND MINERALS CORP
N PITRE OPTION
Tozer Brook
Os3 (graphitic)
Interpreted from IP Graphitic sediments
Potential Pb-Zn-Ag Trend (Possible CNE Trend)
Possible Captain Trend
Captain Trend
Extremely weak trend interpreted as graphitic or weakly pyritic
NOTE
Trenches excavated in the Sprig Brook Grid were undertaken by Billiton Canada Ltd. in 1983. No data filed.
LEGEND
ORDOVICIAN AND OLDER (?)
TETAGOUCHE GROUP
Omv Metabasalt, pillow metabasalt, basaltic metatuff, minor metatrachyte
Ofv2 Quartz and quartz feldspar metaporphyry, quartz sericite schist, meta quartz-chlorite sericite schist, crystal metatuff, minor fossiliferous sediments
Ofv1 Rhyolite metatuff, metarpylite, rhyolite metaporphyry, quartz sericite, quartz chlorite sericite schist
Os3 Phyllite, graphitic slate, feldspathic lithic and quartzose greywacke
SYMBOLS
Geological contact (approximate)
IP anomaly axis, metal factor indicated (1990)
IP anomaly axis, metal factor indicated (Stratabound Minerals Corp., 1988)
Drill hole
C-90-001 Rock sample location with identification number
C-A-001 Silt sample location with identification number
Trench location
Soil anomaly
Fault
GEOLOGICAL ASSOCIATION OF CANADA
D. F. BROWN
FELLOW
Figure 8
KW K. D. A. Whaley
EXPLORATION GEOLOGIST
for
CONSOLIDATED RHODES RESOURCES LTD.
CAPTAIN PROPERTY
GLOUCESTER & NORTHUMBERLAND COUNTIES, N.B.
Prepared by : D.F. Brown, October, 1990
473999
COMPILATION
SCALE 1 : 5000
100 0 100 200 300 400 500 Metres
N.T.S. 21 P/5 W
10

# APPENDIX F

## CNE MACRO

```
#################################################
#
#    DATAMINE MACRO FOR STRATABOUND RESOURCES
#    CNE DEPOSIT, NEW BRUNSWICK, CANADA
#
#    BY:       ROBERT MORRISON
#    WARDROP ENGINEERING INC. A TETRA TECH COMPANY
#        330 BAY STREET, SUITE 900
#        TORONTO, ONTARIO, CANADA
#
#    DATE:     DECEMBER 1, 2010, VERSION 1.A
#        January 16, 2011, VERSION 1.B
#        February 5, 2011, VERSION 1.C
#        October 27, 2011, VERSION 1.D
#
##################################################

##################################################
#
#    FILES REQUIRED TO RUN MACRO:
#        ____________________________
#
#    OVERBTR/PT       OVERBURDEN WIREFRAME DTM
#    CNE_TOPO2TR/PT  ADJUSTED TOPOGRAPHY DTM WIREFRAME
#    PITTR/PT         FORMER PIT WIREFRAME
#    cne_collar_final    drill collar data
#    cne_survey_check    downhole survey data
#    cne_assays1      main assay data
#    cne_assay_extra   recently discovered assay data
#    BOUNDARYTR/PT LIMIT OF CNE DRILLING SOLID
#    WIREFRAME
#
#################################################

#################################################
#
#    OUTPUT FILES:
#        _____________
#
#    holes2-d    raw drillhole file
#    holes1_tc-d RAW DRILLHOLE DATA WITH TOP CUTS
#    APPLIED
#    holes1_tc-c CAPPED DRILLHOLE DATA COMPOSITED TO
#    1M INTERVALS
#    zn_45_tc-c  AS ABOVE BUT WITHIN ZN 4.5% WIREFRAME
#    zn_45_stats STATISTICS OUTPUT FOR ABOVE
#    zn_2_tc-c   AS ABOVE BUT WITHIN ZN 2.0% WIREFRAME
#    zn_2_stats  STATISTICS OUTPUT FOR ABOVE
#    cu_05_tc-c  AS ABOVE BUT WITHIN CU 0.5% WIREFRAME
#    cu_05_stats STATISTICS OUTPUT FOR ABOVE

#    block-m     primary block model
#    cne_ind3-m  Zn and Cu indicator model
#    cne_grade3-m    grade estimated model
```

```
#    cne_tonnage3-m  model with assigned density
#    mined_3_40-m    mined-out blocks
#    not_mined_3_40-m    not mined blocks (remaining
#    resource)

#    rock = 0    Air
#    rock = 1    Overburden
#    rock = 2    host rock (volcanics,
#    volvaniclastics, etc)

#    mstatus = 2 Not mined out
#    mstatus = 1 Mined out (open pit)

#    Grade Variables
#    _______________

#    Zn% - Zinc Percent
#    Pb% - Lead Percent
#    Cu% - Copper Percent
#    Co_id - Cobalt ppm
#    Ag_id - Silver ppm

#    Resource Classification
#    _______________________
#
#    class_cu = 1 - Measured (copper)
#    class_cu = 2 - Indicated (copper)
#    class_cu = 3 - Inferred (copper)
#
#    class_zn = 1 - Measured (zinc, lead)
#    class_zn = 2 - Indicated (zinc, lead)
#    class_zn = 3 - Inferred (zinc, lead)
#
#    density = density (tonnes per metres cubed)

#    Indicator Probability
#    _____________________

#    ZN3 = 1 - 40% probability of block >= 3% Zn
#    CU2 = 1 - 40% probability of block >= 2% Cu
#
####################################################

!START BLOCKS
!PROTOM    &OUT(proto_222-m),@ROTMOD=0.0
    Y
    N
    281750
    5241860
    20
    2
    2
    2
    175
    150
    80
```

```
!TRIFIL     &PROTO(proto_222-
m),&WIRETR(cne_topo2tr),&WIREPT(cne_topo2pt),
            &MODEL(xx2-
m),@MODLTYPE=4.0,@MAXDIP=0.0,@SPLITS=0.0,
            @PLANE='XY
',@XSUBCELL=1.0,@YSUBCELL=1.0,@ZSUBCELL=1.0,
            @RESOL=0.0

!TRIFIL     &PROTO(proto_222-
m),&WIRETR(cne_topo2tr),&WIREPT(cne_topo2pt),
            &MODEL(xx3-
m),@MODLTYPE=3.0,@MAXDIP=0.0,@SPLITS=0.0,
            @PLANE='XY
',@XSUBCELL=1.0,@YSUBCELL=1.0,@ZSUBCELL=1.0,
            @RESOL=0.0

!ADDMOD     &IN1(xx2-m),&IN2(xx3-m),&OUT(xx1-
m),@TOLERNCE=0.001

!EXTRA      &IN(xx1-m),&OUT(xx2-m),@APPROX=0.0
     density=0
     rock=0
     mstatus=0
     class_zn=0
     class_cu=0
     Zn_ind=0
     Pb%=0
     Ag_id=0
     Cu%=0
     Co_id=0
     Cu_ind=0
     Zn%=0
     GO

!SELWF      &IN(xx2-m),&WIRETR(overbtr),&WIREPT(overbpt),
            &OUT(xx1-
m),*X(XC),*Y(YC),*Z(ZC),@SELECT=2.0,@PLANE='XY  ',
            @EXCLUDE=0.0,@TOLERANC=0.001
!SORTX      &IN(xx1-m),&OUT(xx3-m),*KEY1(IJK),@BINS=5.0,
            @ORDER=1.0
!EXTRA      &IN(xx3-m),&OUT(xxrock-m),@APPROX=0.0
     density=2.8
     rock=2
     mstatus=2
     GO

!SELWF      &IN(xx2-
m),&WIRETR(cne_topo2tr),&WIREPT(cne_topo2pt),
            &OUT(xx1-
m),*X(XC),*Y(YC),*Z(ZC),@SELECT=2.0,@PLANE='XY  ',
            @EXCLUDE=0.0,@TOLERANC=0.001
!SORTX      &IN(xx1-m),&OUT(xx3-m),*KEY1(IJK),@BINS=5.0,
            @ORDER=1.0
!EXTRA      &IN(xx3-m),&OUT(xxoverb-m),@APPROX=0.0
     density=2.00
     rock=1
     mstatus=2
     GO

!SELWF      &IN(xx2-
m),&WIRETR(cne_topo2tr),&WIREPT(cne_topo2pt),
            &OUT(xx1-
m),*X(XC),*Y(YC),*Z(ZC),@SELECT=1.0,@PLANE='XY  ',
            @EXCLUDE=0.0,@TOLERANC=0.001
!SORTX      &IN(xx1-m),&OUT(xx3-m),*KEY1(IJK),@BINS=5.0,
            @ORDER=1.0
!EXTRA      &IN(xx3-m),&OUT(xxair-m),@APPROX=0.0
     density=0
     rock=0
```

```
    GO

!ADDMOD    &IN1(xxoverb-m),&IN2(xxrock-m),&OUT(XX4-
m),@TOLERNCE=0.001

!SELWF     &IN(xx4-m),&WIRETR(pittr),&WIREPT(pitpt),
           &OUT(xx1-
m),*X(XC),*Y(YC),*Z(ZC),@SELECT=1.0,@PLANE='XY  ',
           @EXCLUDE=0.0,@TOLERANC=0.001
!EXTRA     &IN(xx1-m),&OUT(xx5-m),@APPROX=0.0
    mstatus=1
    GO

!ADDMOD    &IN1(xx4-m),&IN2(xx5-m),&OUT(xx6-
m),@TOLERNCE=0.001
!ADDMOD    &IN1(xx6-m),&IN2(xxair-m),&OUT(xx7-
m),@TOLERNCE=0.001

!SORTX     &IN(xx7-m),&OUT(BLOCK-m),*KEY1(IJK),@BINS=5.0,
           @ORDER=1.0
#!END

!START HOLES
###########################################################
#
#    CREATE FRILLHOLR FILE FROM
#    ASSAY, SURVEY AND COLLAR DATA
#
############################################################
#

!HOLES3D
&COLLAR(cne_collar_final),&SURVEY(cne_survey_check),
           &SAMPLE1(cne_assay_oct_2011),&OUT(holes1-
d),*BHID(BHID),

*XCOLLAR(XCOLLAR),*YCOLLAR(YCOLLAR),*ZCOLLAR(ZCOLLAR),
       *FROM(FROM),*TO(TO),*AT(AT),*BRG(AZIMUTH),*DIP(DIP),
       @SURVSMTH=1.0,@ENDPOINT=0.0,@DIPMETH=-1.0

!HOLES3D
&COLLAR(cne_collar_final),&SURVEY(cne_survey_check),
           &SAMPLE1(cne_assay_extra),&OUT(holes3-
d),*BHID(BHID),

*XCOLLAR(XCOLLAR),*YCOLLAR(YCOLLAR),*ZCOLLAR(ZCOLLAR),

*FROM(FROM),*TO(TO),*AT(AT),*BRG(AZIMUTH),*DIP(DIP),
           @SURVSMTH=1.0,@ENDPOINT=0.0,@DIPMETH=-1.0

!HOLMER    &IN1(holes3-d),&IN2(holes1-d),&OUT(xxholes2-d),
           *BHID(BHID),*FROM(FROM),*TO(TO)

!SORTX     &IN(xxholes2-d),&OUT(holes_2011-
d),*KEY1(BHID),*KEY2(FROM),@BINS=5.0,
           @ORDER=1.0
#!END
##########################################################
#
#    APPLY CAPS (TOP CUTS) TO RAW DRILLHOLE DATA

#########
#
#    COLLECT DRILLHOLES WITHIN JUST THE AREA OF INTEREST
#
#    Zn top-cut = 33%
#    Cu top-cut = 3%
#    Pb top-cut = 16%
#
```

```
#############################################

!SELWF      &IN(holes_2011-d),&WIRETR(boundarytr),
            &WIREPT(boundarypt),&OUT(raw_cne-d),*X(X),*Y(Y),
            *Z(Z),@SELECT=3.0,@EXCLUDE=0.0,@TOLERANC=0.001

!EXTRA      &IN(raw_cne-d),&OUT(holes1_tc-d),@APPROX=0.0
    IF(Cu%>3)
    Cu%=3
    end
    IF(Zn%>33)
    Zn%=33
    end
    if(Co_ppm>1500)
    Co_ppm=1500
    end
    if(Au_gt>120)
    Au_gt=120
    end
    if(Ag_gt>500)
    Ag_gt=500
    end
    if(Pb%>16)
    Pb%=16
    end
    GO

#########
#
#    COMPOSITE CAPPED DRILLHOLE FILE TO 1M INTERVALS
#

!COMPDH     &IN(holes1_tc-d),&OUT(holes1_tc-
c),*BHID(BHID),*FROM(FROM),
            *TO(TO),@INTERVAL=1.0,@START=0.0,@MODE=0.0
!END
####
#
#    STATS FOR RAW VS COMPOSITED AND CAPPED

#!STATS      &IN(raw_cne-d),&OUT(raw_stats-
d),*F1(Zn%),*F2(Pb%),

*F3(Ag_gt),*F4(Cu%),*F5(Co_ppm),*F6(Au_gt),*F7(LENGTH)

#!STATS      &IN(holes1_tc-d),&OUT(cap_stats-
d),*F1(Zn%),*F2(Pb%),

*F3(Ag_gt),*F4(Cu%),*F5(Co_ppm),*F6(Au_gt),*F7(LENGTH)

#!STATS      &IN(holes1_tc-c),&OUT(cap_stats-
c),*F1(Zn%),*F2(Pb%),

*F3(Ag_gt),*F4(Cu%),*F5(Co_ppm),*F6(Au_gt),*F7(LENGTH)

##########################################################

##########################################################
#
#    GENERATE BLOCK MODEL FOR CREATING A
#    INDICATOR/PROBABILITY WIREFRAME based on a 3% Zn
Indicator
#
##############################################

!START INDIC

!COPY       &IN(block-m),&OUT(xx1-m),rock=2.0
```

```
!SELWF      &IN(xx1-
m),&WIRETR(boundarytr),&WIREPT(boundarypt),
            &OUT(xx2-
m),*X(XC),*Y(YC),*Z(ZC),@SELECT=3.0,@PLANE=-,
            @EXCLUDE=0.0,@TOLERANC=0.001

!INDEST     &PROTO(xx2-m),&IN(holes1_tc-
c),&SRCPARM(zn_ind_3_spar),
            &ESTPARM(zn_ind_3_epar),&MODEL(xx3-m),

&VMODPARM(zn_ind_3_vpar),*X(X),*Y(Y),*Z(Z),*KEY(BHID),

@DISCMETH=1.0,@XPOINTS=3.0,@YPOINTS=3.0,@ZPOINTS=2.0,

@XDSPACE=1.0,@YDSPACE=1.0,@ZDSPACE=1.0,@PARENT=1.0,

@MINDISC=1.0,@COPYVAL=0.0,@FVALTYPE=1.0,@FSTEP=1.0,

@XMIN=281750.0,@XMAX=282100.0,@YMIN=5241860.0,@YMAX=5242160.
0,

@ZMIN=20.0,@ZMAX=180.0,@XSUBCELL=1.0,@YSUBCELL=1.0,

@ZSUBCELL=1.0,@LINKMODE=3.0,@UCSAMODE=2.0,@UCSBMODE=3.0,

@UCSCMODE=2.0,@PLANE=1.0,@TOLRNC=0.0,@GRMETHOD=3.0,
            @PGFIELDS=1.0,@ORDER=3.0

!EXTRA      &IN(xx3-m),&OUT(xx4-m),@APPROX=0.0
    if(PRAB1>=0.4)
    ZN3=1
    else
    ZN3=0
    END
    if(PRAB1>=0.0)
    PRAB1=0.0
    END
    GO

!SORTX      &IN(xx4-m),&OUT(xx5-m),*KEY1(IJK),@BINS=5.0,
            @ORDER=1.0

!INDEST     &PROTO(xx5-m),&IN(holes1_tc-
c),&SRCPARM(cu_ind02_spar),
            &ESTPARM(cu_ind02_epar),&MODEL(xx2-m),

&VMODPARM(cu_ind02_vpar),*X(X),*Y(Y),*Z(Z),*KEY(BHID),

@DISCMETH=1.0,@XPOINTS=3.0,@YPOINTS=3.0,@ZPOINTS=2.0,

@XDSPACE=1.0,@YDSPACE=1.0,@ZDSPACE=1.0,@PARENT=1.0,

@MINDISC=1.0,@COPYVAL=0.0,@FVALTYPE=1.0,@FSTEP=1.0,

@XMIN=281750.0,@XMAX=282100.0,@YMIN=5241860.0,@YMAX=5242160.
0,

@ZMIN=20.0,@ZMAX=180.0,@XSUBCELL=1.0,@YSUBCELL=1.0,

@ZSUBCELL=1.0,@LINKMODE=3.0,@UCSAMODE=2.0,@UCSBMODE=3.0,

@UCSCMODE=2.0,@PLANE=1.0,@TOLRNC=0.0,@GRMETHOD=3.0,
            @PGFIELDS=1.0,@ORDER=3.0

!EXTRA      &IN(xx2-m),&OUT(xx4-m),@APPROX=0.0
    if(PRAB1>=0.4)
    CU2=1
    else
    CU2=0
```

```
      END
      GO

!SORTX      &IN(xx4-m),&OUT(cne_ind3-m),*KEY1(IJK),@BINS=5.0,
            @ORDER=1.0
#!END
####################################################
#
#    ESTIMATION RUNS
#
############################

!START EST

!COPY       &IN(cne_ind3-m),&OUT(xx1-m),ZN3=1.0

#    Zn Pb and Ag Ordinary Kriged Estimate
############################

!ESTIMA     &PROTO(xx1-m),&IN(holes1_tc-
c),&SRCPARM(zn_3_ok_spar),
            &ESTPARM(zn_3_ok_epar),&MODEL(xx5-m),
        &VMODPARM(zn_3_ok_vpar),*X(X),*Y(Y),

*Z(Z),*KEY(BHID),@DISCMETH=1.0,@XPOINTS=3.0,@YPOINTS=3.0,

@ZPOINTS=3.0,@XDSPACE=1.0,@YDSPACE=1.0,@ZDSPACE=1.0,

@PARENT=1.0,@MINDISC=1.0,@COPYVAL=0.0,@FVALTYPE=1.0,

@FSTEP=1.0,@XMIN=281750.0,@XMAX=282100.0,@YMIN=5241860.0,

@YMAX=5242160.0,@ZMIN=20.0,@ZMAX=180.0,@XSUBCELL=1.0,

@YSUBCELL=1.0,@ZSUBCELL=1.0,@LINKMODE=3.0,@UCSAMODE=2.0,

@UCSBMODE=3.0,@UCSCMODE=2.0,@PLANE=1.0,@TOLRNC=0.0,
            @GRMETHOD=3.0,@PGFIELDS=0.0,@ORDER=3.0


#    ZN PB AG = NN AND ID
############################

!ESTIMA     &PROTO(xx5-m),&IN(holes1_tc-
c),&SRCPARM(id_nn_spar),
            &ESTPARM(id_nn_epar),&MODEL(ZNPBAG-m),*X(X),*Y(Y),

*Z(Z),*KEY(BHID),@DISCMETH=1.0,@XPOINTS=3.0,@YPOINTS=3.0,

@ZPOINTS=3.0,@XDSPACE=1.0,@YDSPACE=1.0,@ZDSPACE=1.0,

@PARENT=1.0,@MINDISC=1.0,@COPYVAL=0.0,@FVALTYPE=1.0,

@FSTEP=1.0,@XMIN=281750.0,@XMAX=282100.0,@YMIN=5241860.0,

@YMAX=5242160.0,@ZMIN=20.0,@ZMAX=180.0,@XSUBCELL=1.0,

@YSUBCELL=1.0,@ZSUBCELL=1.0,@LINKMODE=3.0,@UCSAMODE=2.0,

@UCSBMODE=3.0,@UCSCMODE=2.0,@PLANE=1.0,@TOLRNC=0.0,
            @GRMETHOD=3.0,@PGFIELDS=0.0,@ORDER=3.0

#    Cu and Co = OK, NN AND ID
############################

!COPY       &IN(ZNPBAG-m),&OUT(XX9-M),CU2=1.0
!COPY       &IN(cne_ind-m),&OUT(XX8-M),CU2=1.0
!ADDMOD     &IN1(xx8-m),&IN2(xx9-m),&OUT(xx3-
m),@TOLERNCE=0.001
```

```
!ESTIMA     &PROTO(XX3-m),&IN(holes1_tc-c),&SRCPARM(cne_spar),
            &ESTPARM(cne_epar),&MODEL(xx7-m),
            &VMODPARM(cne_vpar),*X(X),*Y(Y),*Z(Z),*KEY(BHID),

@DISCMETH=1.0,@XPOINTS=2.0,@YPOINTS=2.0,@ZPOINTS=2.0,

@XDSPACE=1.0,@YDSPACE=1.0,@ZDSPACE=1.0,@PARENT=1.0,

@MINDISC=1.0,@COPYVAL=0.0,@FVALTYPE=1.0,@FSTEP=1.0,

@XMIN=281750.0,@XMAX=282100.0,@YMIN=5241860.0,@YMAX=5242160.
0,

@ZMIN=20.0,@ZMAX=180.0,@XSUBCELL=1.0,@YSUBCELL=1.0,

@ZSUBCELL=1.0,@LINKMODE=3.0,@UCSAMODE=2.0,@UCSBMODE=3.0,

@UCSCMODE=2.0,@PLANE=1.0,@TOLRNC=0.0,@GRMETHOD=3.0,
            @PGFIELDS=0.0,@ORDER=3.0

!ADDMOD     &IN1(ZNPBAG-m),&IN2(xx7-m),&OUT(xx2-
m),@TOLERNCE=0.001
!SORTX      &IN(xx2-m),&OUT(cne_grade3-
m),*KEY1(IJK),@BINS=5.0,
            @ORDER=1.0

####
#
#    FOR CALCULATING OUT DENSITY...
#    SELECT ONLY BLOCKS WITH ZN HIGH-GRADE
#    (e.g. ZN = 4.5%)
#



!EXTRA      &IN(cne_grade3-m),&OUT(xx1-m),@APPROX=0.0

    if(Cu%>=4.5)
    density=3.23
    end
    if(Zn%>=2.0)
    density=3.37
    end
    if(Zn%>=4.5)
    density=3.86
    end
    GO

!SORTX      &IN(xx1-m),&OUT(xxok-m),*KEY1(IJK),@BINS=5.0,
            @ORDER=1.0

!EXTRA      &IN(xxok-m),&OUT(cne_tonnage3-m),@APPROX=0.0
    if(Zn%==absent())
    Zn%=0
    end
    if(Zn_id==absent())
    Zn_id=0
    end
    if(Zn_nn==absent())
    Zn_nn=0
    end
    if(Pb%==absent())
    Pb%=0
    end
    if(Pb_id==absent())
    Pb_id=0
    end
    if(Pb_nn==absent())
```

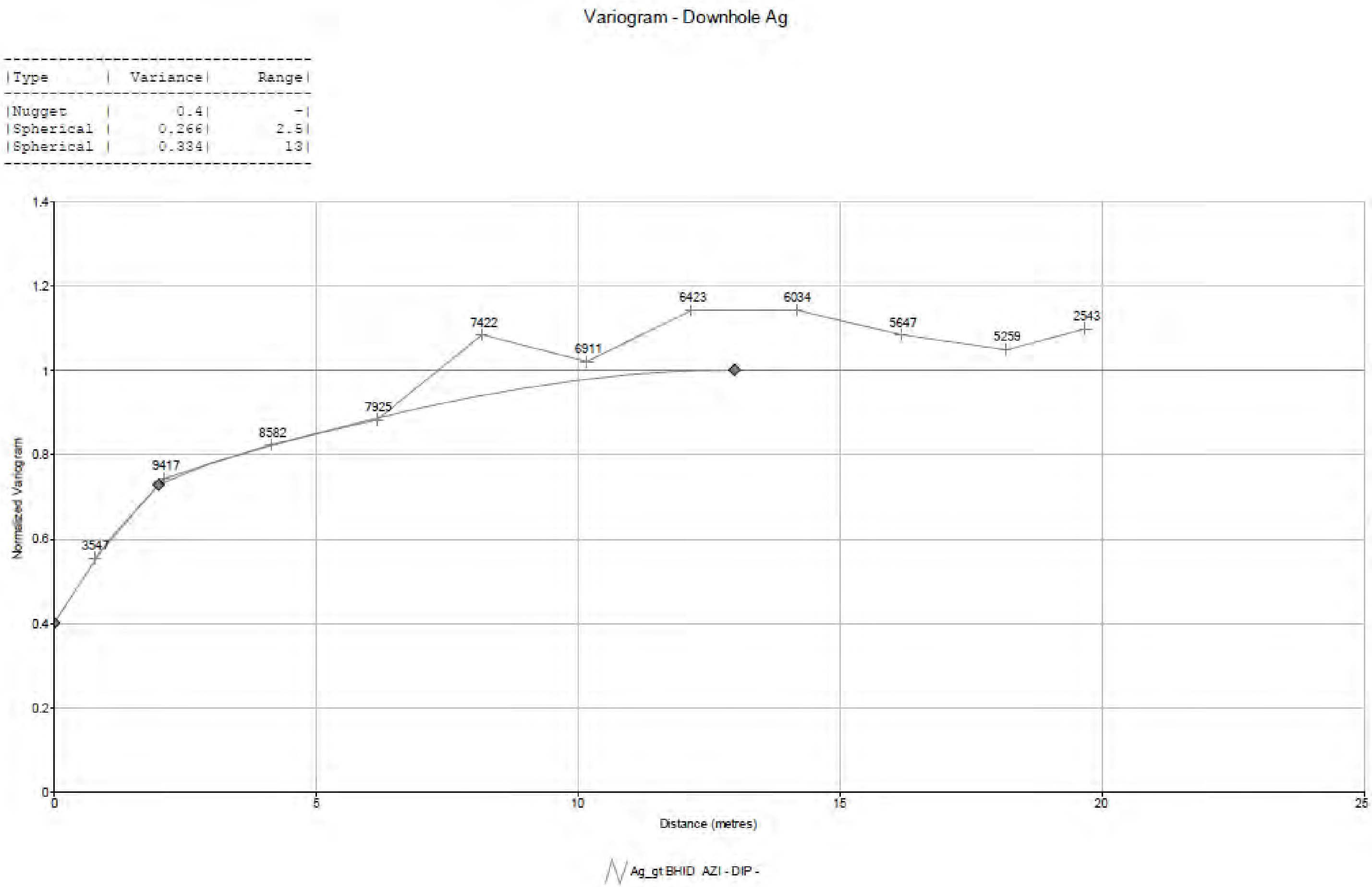
```
      Pb_nn=0
      end
      if(Cu%==absent())
      Cu%=0
      end
      if(Co_id==absent())
      Co_id=0
      end
      if(Ag_id==absent())
      Ag_id=0
      end
      if(Cu_id==absent())
      Cu_id=0
      end
      if(Cu_nn==absent())
      Cu_nn=0
      end
      if(Ag_gt==absent())
      Ag_gt=0
      end
      if(Ag_nn==absent())
      Ag_nn=0
      end
      GO

!COPY      &IN(cne_tonnage3-m),&OUT(not_mined_3_40-
m),mstatus=2
!COPY      &IN(cne_tonnage3-m),&OUT(mined_3_40-m),mstatus=1

!ADDMOD    &IN1(block-m),&IN2(cne_tonnage3-m),&OUT(xx2-
m),@TOLERNCE=0.001
!SORTX     &IN(xx2-m),&OUT(cne_final3-
m),*KEY1(IJK),@BINS=5.0,
           @ORDER=1.0

!LISTDR XX?,&OUT(XX)
!DELETE &IN(XX),@CONFIRM=0.0
!END

##################################################
!START price

!EXTRA     &IN(not_mined_3_40-m),&OUT(xx2-m),@APPROX=0.0
      Zn_1 = (Zn%*0.765)
      Zn_2 = (Pb%*0.8075*0.99*22.0462)
      Zn_3 = (Cu%*3.01*0.8203*22.0462)
      Zn_4 = Zn_2+Zn_3
      Zn_eq = Zn_1+(Zn_4/1.06/22.0462)
      GO

!EXTRA     &IN(xx2-m),&OUT(xx3-m),@APPROX=0.0
      if (class_cu==3)
      class=3
      end
      if (class_cu==2)
      class=2
      end
      if (class_cu==1)
      class=1
      end
      if (class_cu==absent())
      class=0
      end
      if (class_zn==3)
      class=3
      end
      if (class_zn==2)
      class=2
      end
```

```
      if (class_zn==1)
      class=1
      end
      if (class_zn==absent())
      class=0
      end
GO

!SORTX      &IN(xx3-m),&OUT(zn_eq_oct_2011-
m),*KEY1(IJK),*KEY2(mstatus),
            @BINS=10.0,@ORDER=1.0
!START TONGRAD
!TONGRAD   &IN(zn_eq_oct_2011-
m),&OUT(xx1),&CSVOUT(zn_eq_nov_2011),

*F1(Zn_eq),*F2(Zn%),*F3(Pb%),*F4(Cu%),*F5(Ag_gt),*KEY1(class
),

*DENSITY(density),@FACTOR=1.0,@DENSITY=1.0,@COGSTEP=1.5
!END
```

# APPENDIX G

## CNE Variography

**Figure 1 - Silver Downhole Variography**



## Figure 2 - Silver Variography

Variogram - Ag - 2011

| Type | Variance | 60/0 | 150/60 | 330/30 |
|---|---|---|---|---|
| Nugget | 0.4 | - | - | - |
| Spherical | 0.109 | 21 | 19 | 25 |
| Spherical | 0.491 | 28 | 28 | 45 |

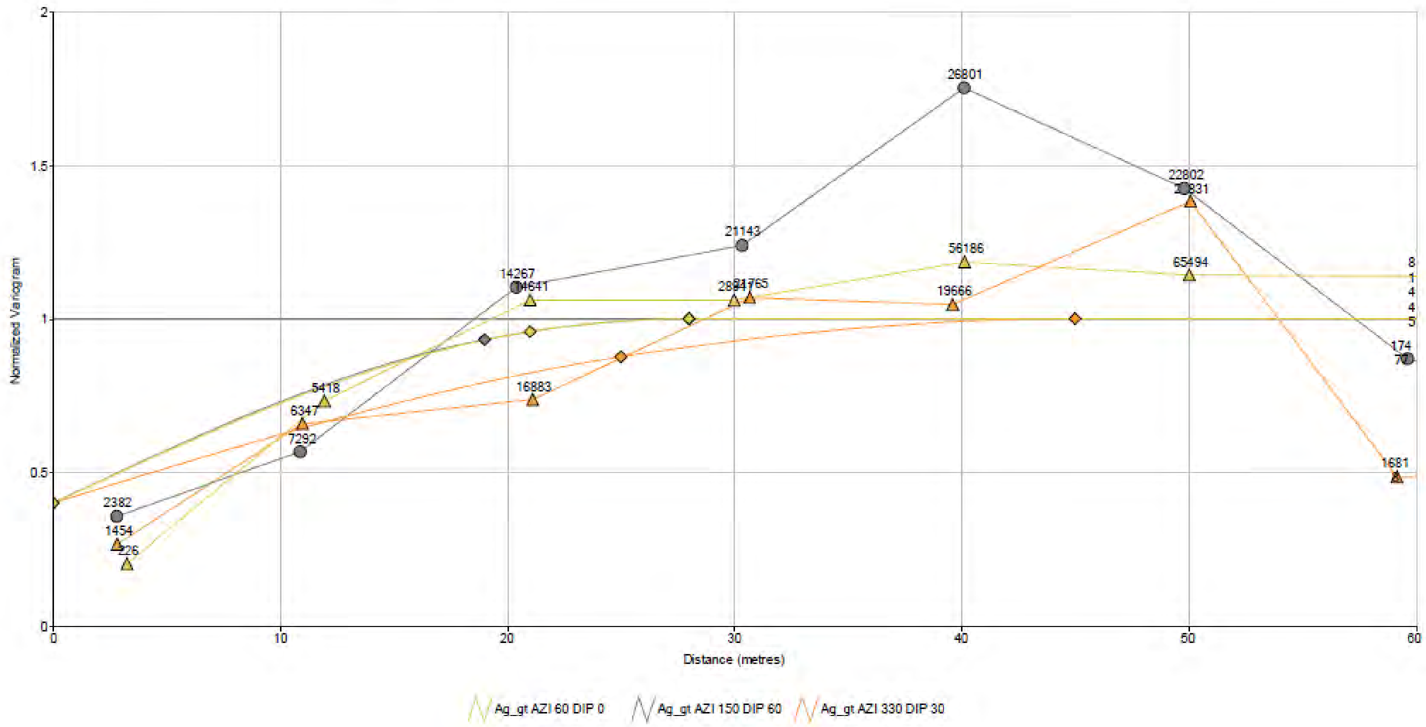


**Figure 3 - Copper Downhole Variography**

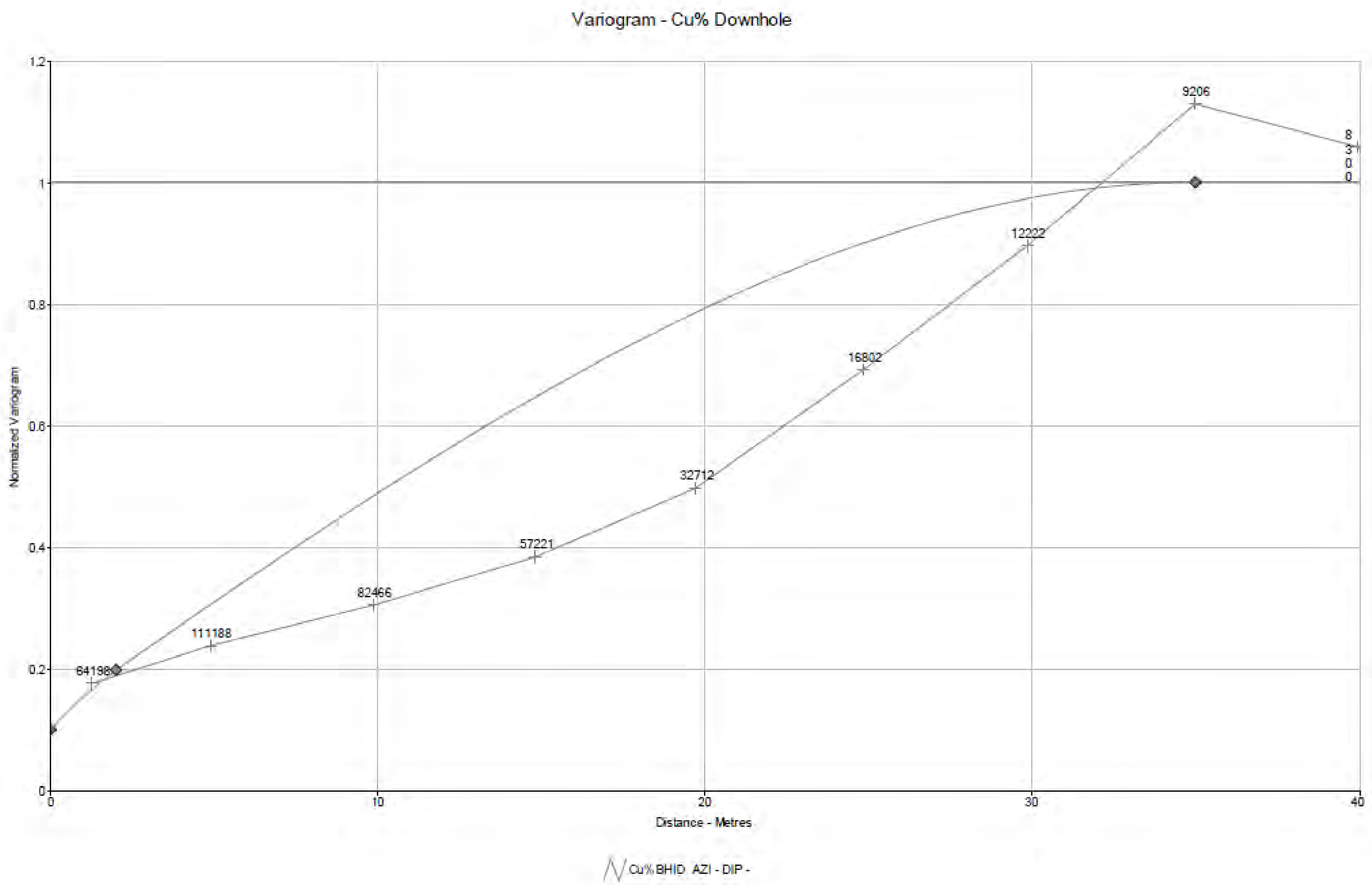
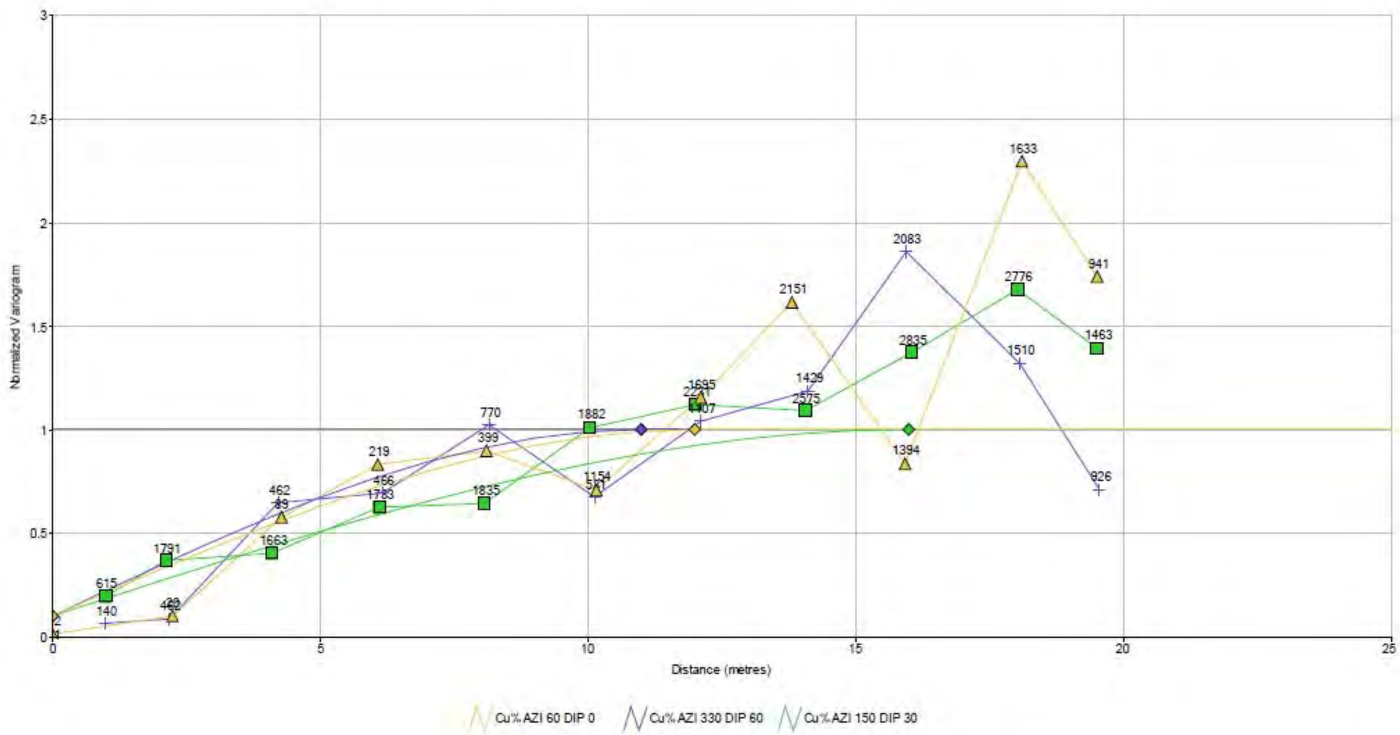


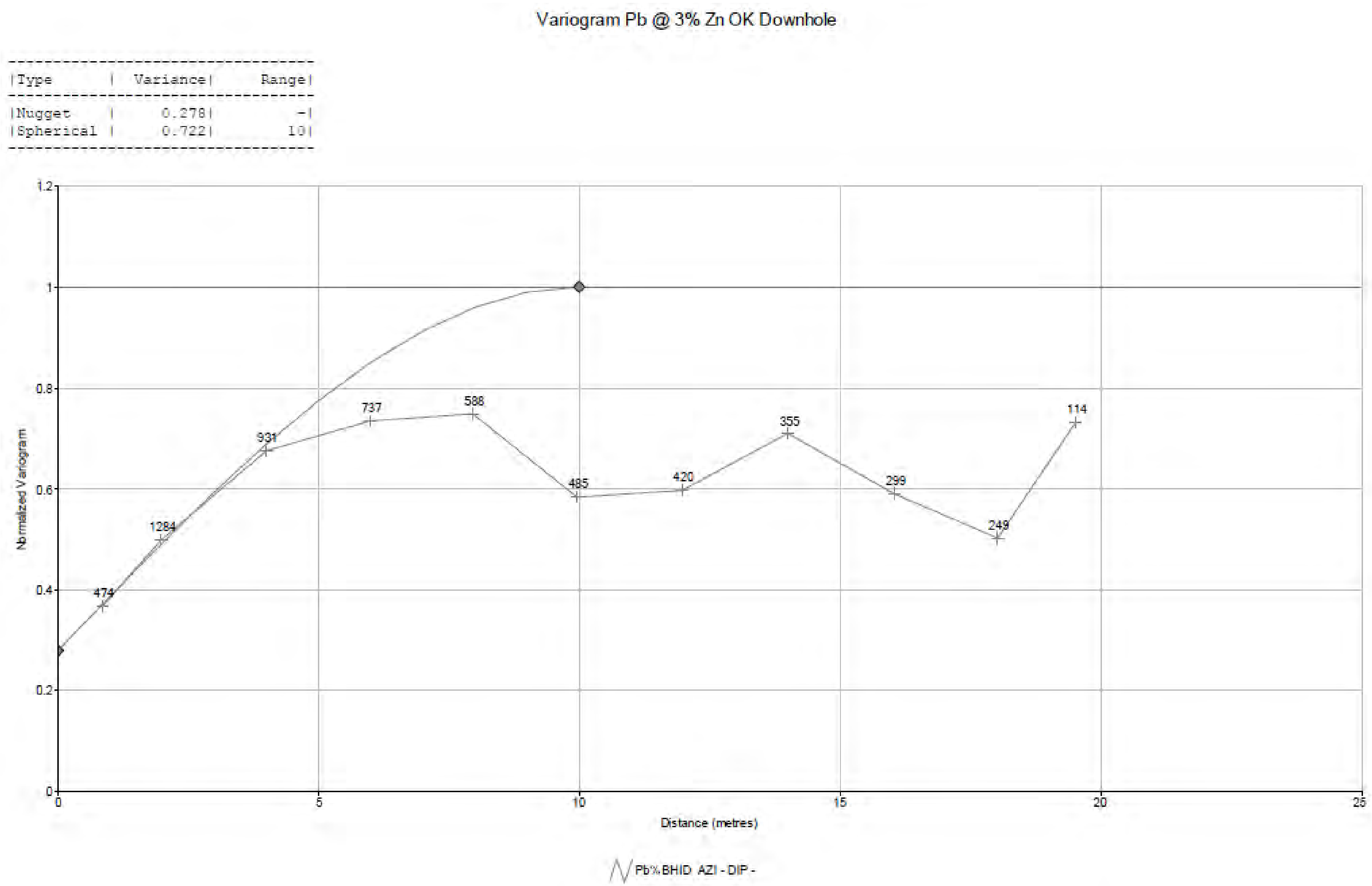
**Figure 4 - Copper Variography**

Variogram Cu for OK

| Type | Variance | 60/0 | 330/60 | 150/30 |
|---|---|---|---|---|
| Nugget | 0.1 | - | - | - |
| Spherical | 0.900 | 12 | 11 | 16 |



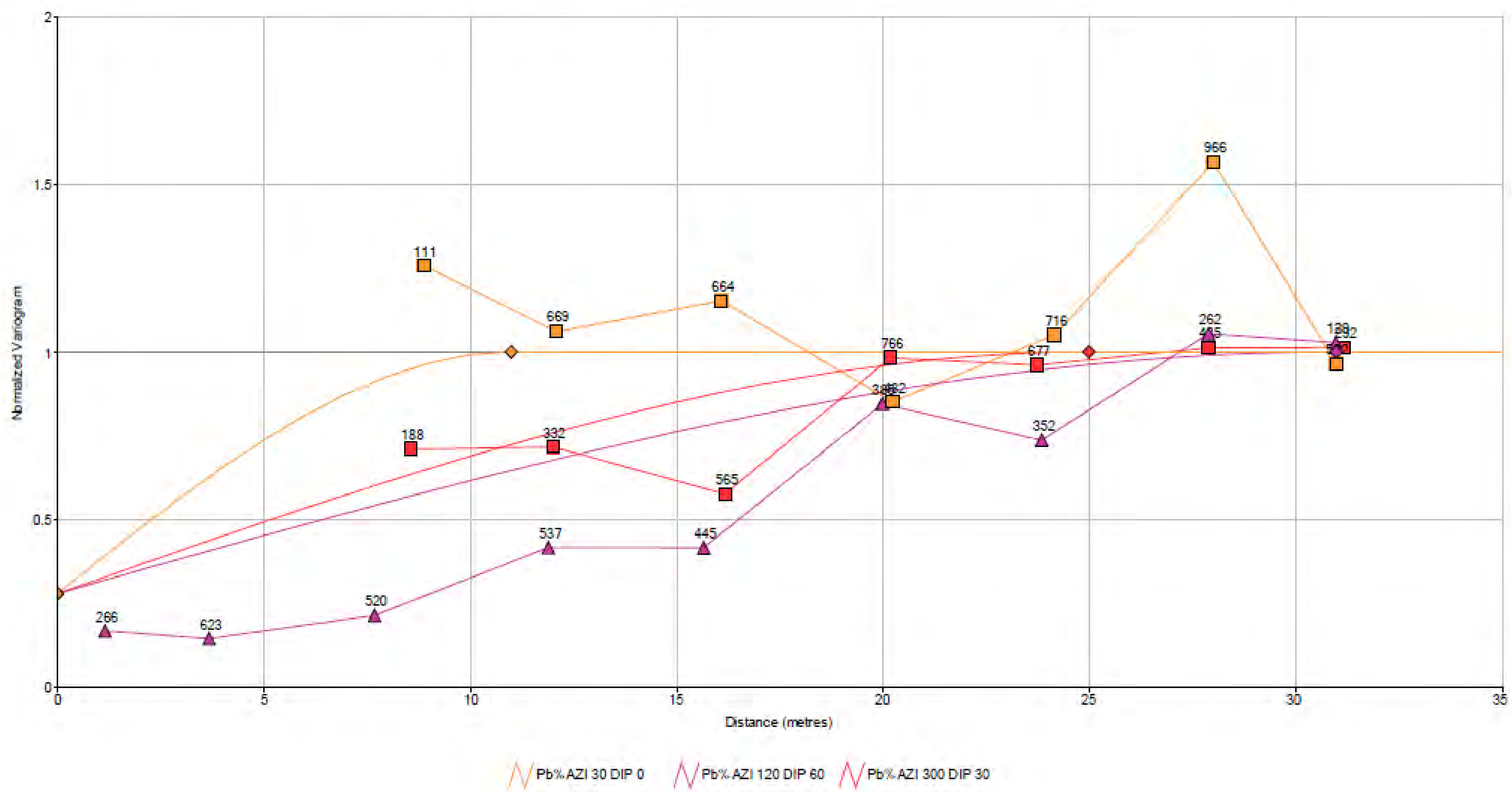
## Figure 5 - Lead Downhole Variography



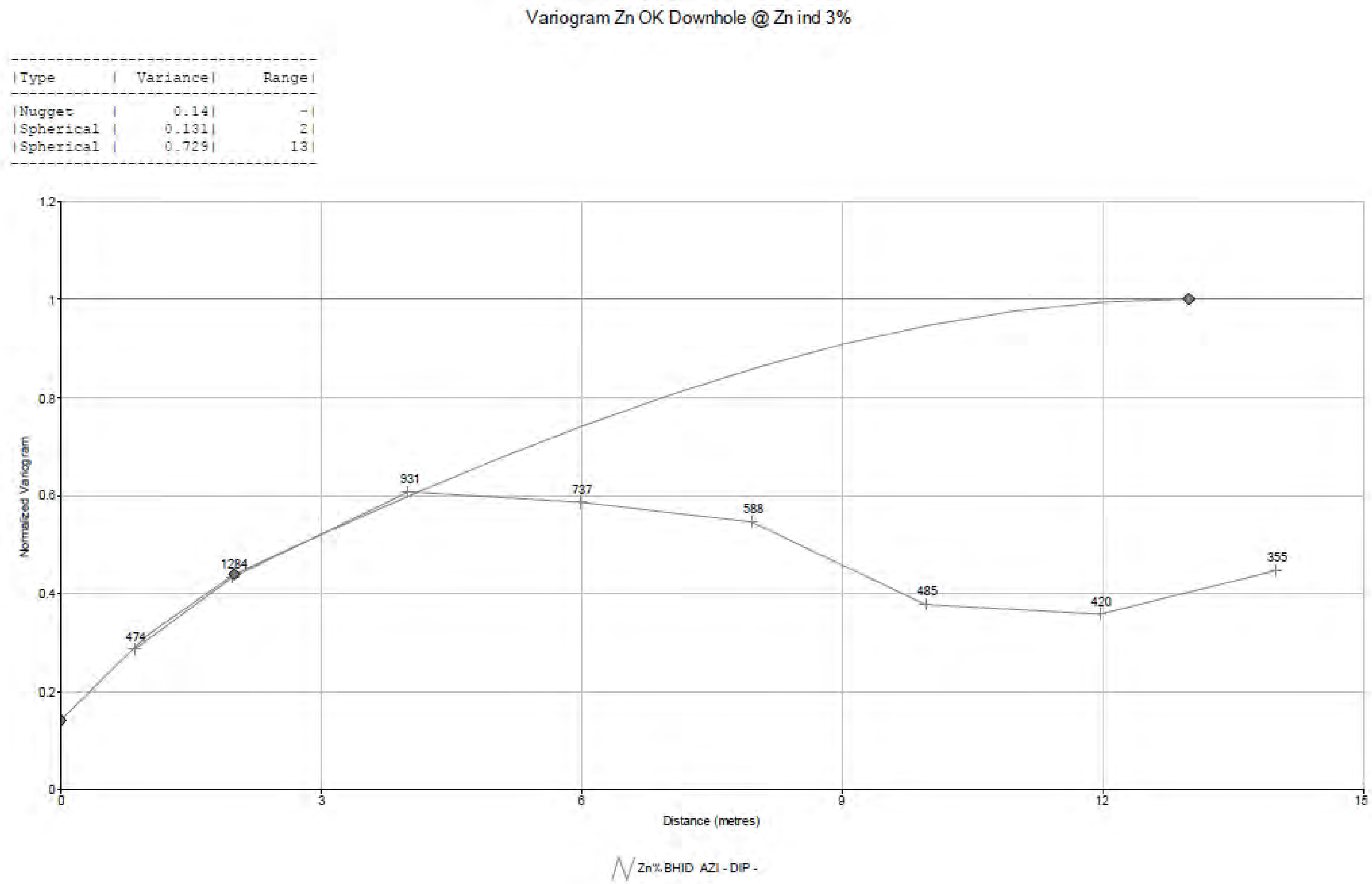
## Figure 6 - Lead Variography

Variogram Pb OK @ 3% Zn cut-off

| Type | Variance | 30/0 | 120/60 | 300/30 |
|---|---|---|---|---|
| Nugget | 0.278 | - | - | - |
| Spherical | 0.722 | 11 | 31 | 25 |



## Figure 7 - Zinc Downhole Variography



## Figure 8 - Zinc Variogram

Variogram Zn 3% OK

| Type | Variance | 30/0 | 300/30 | 120/60 |
|---|---|---|---|---|
| Nugget | 0.14 | - | - | - |
| Spherical | 0.860 | 12 | 20 | 16 |

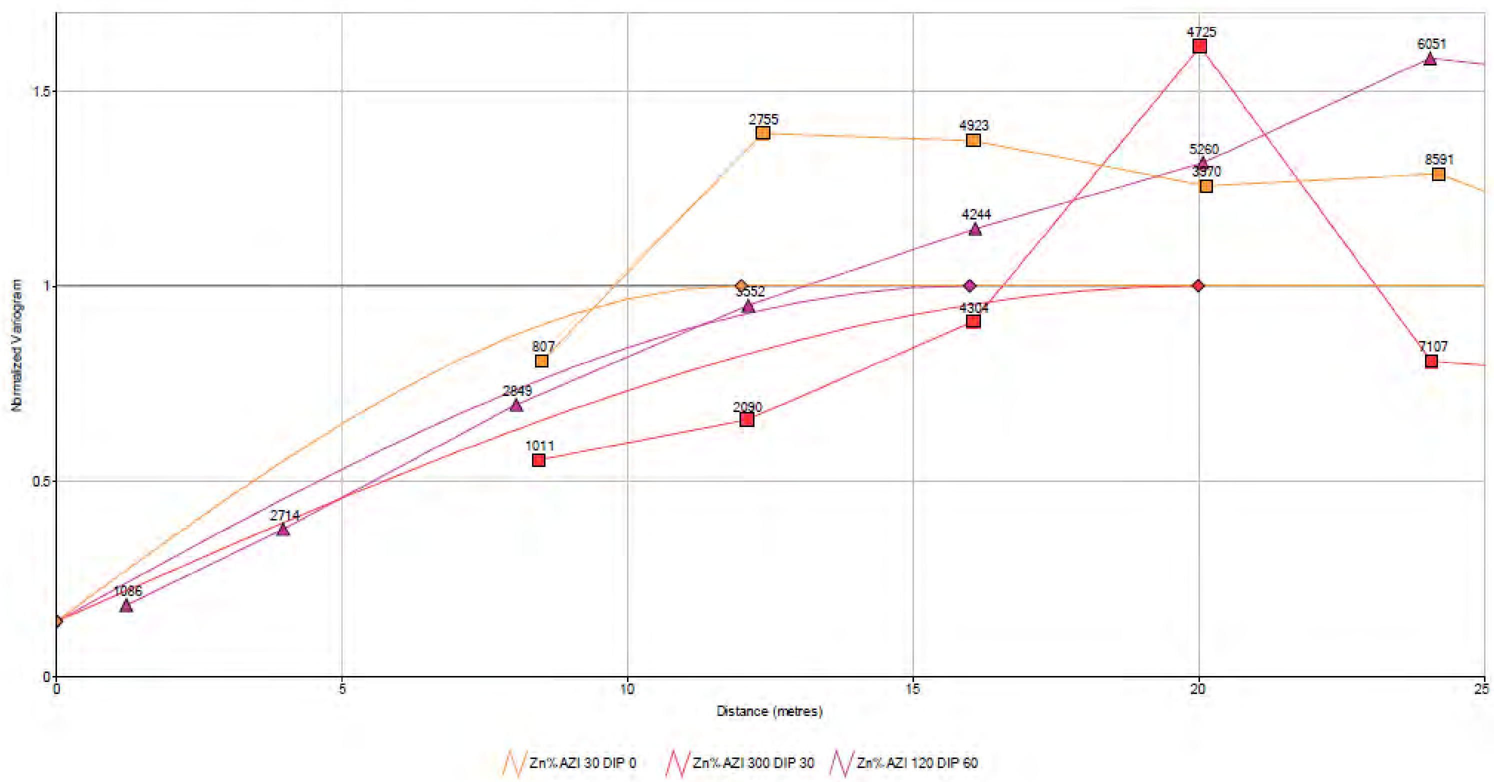


**Figure 9 - Zn 3% Downhole Indicator Variogram**

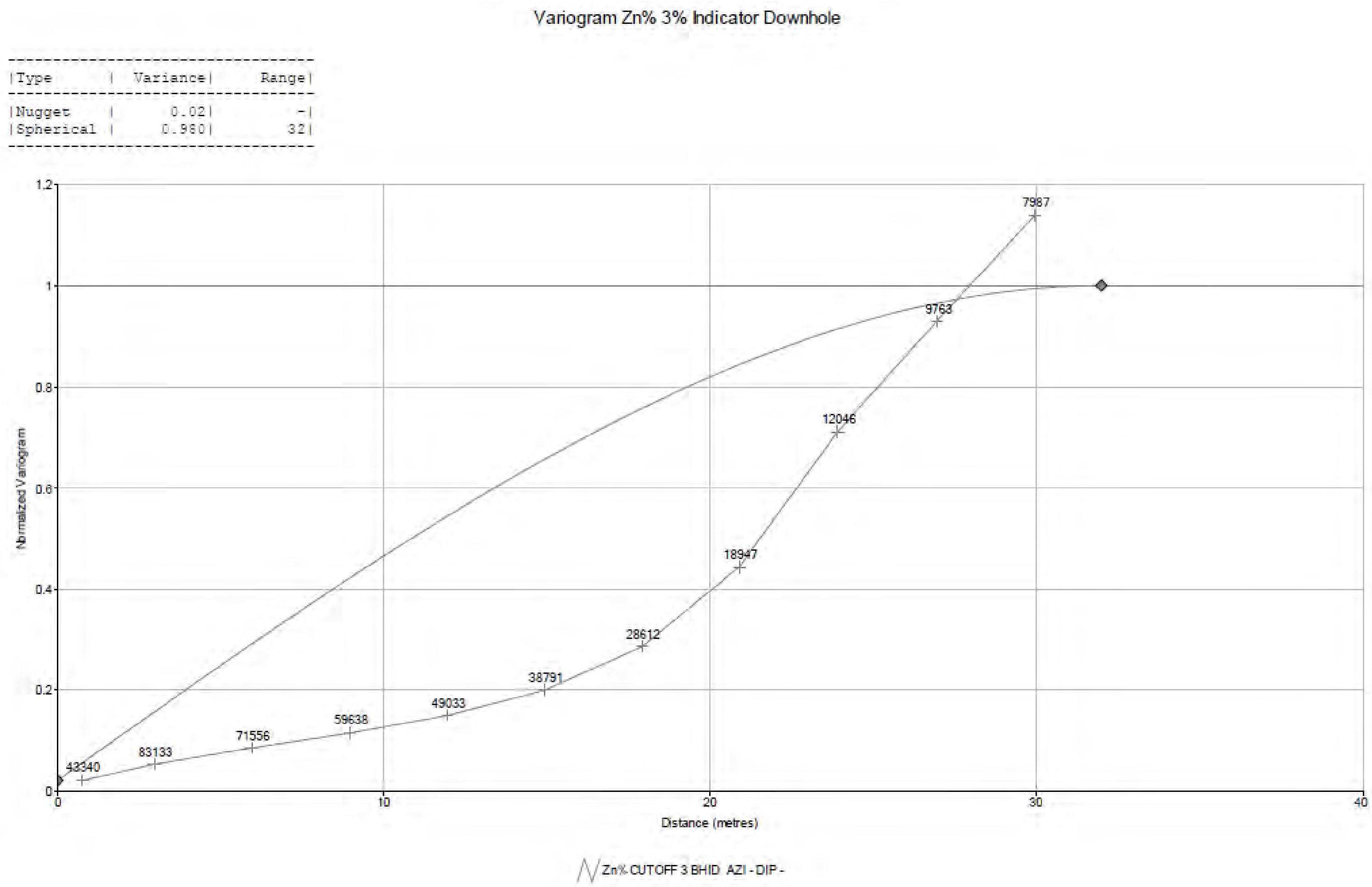
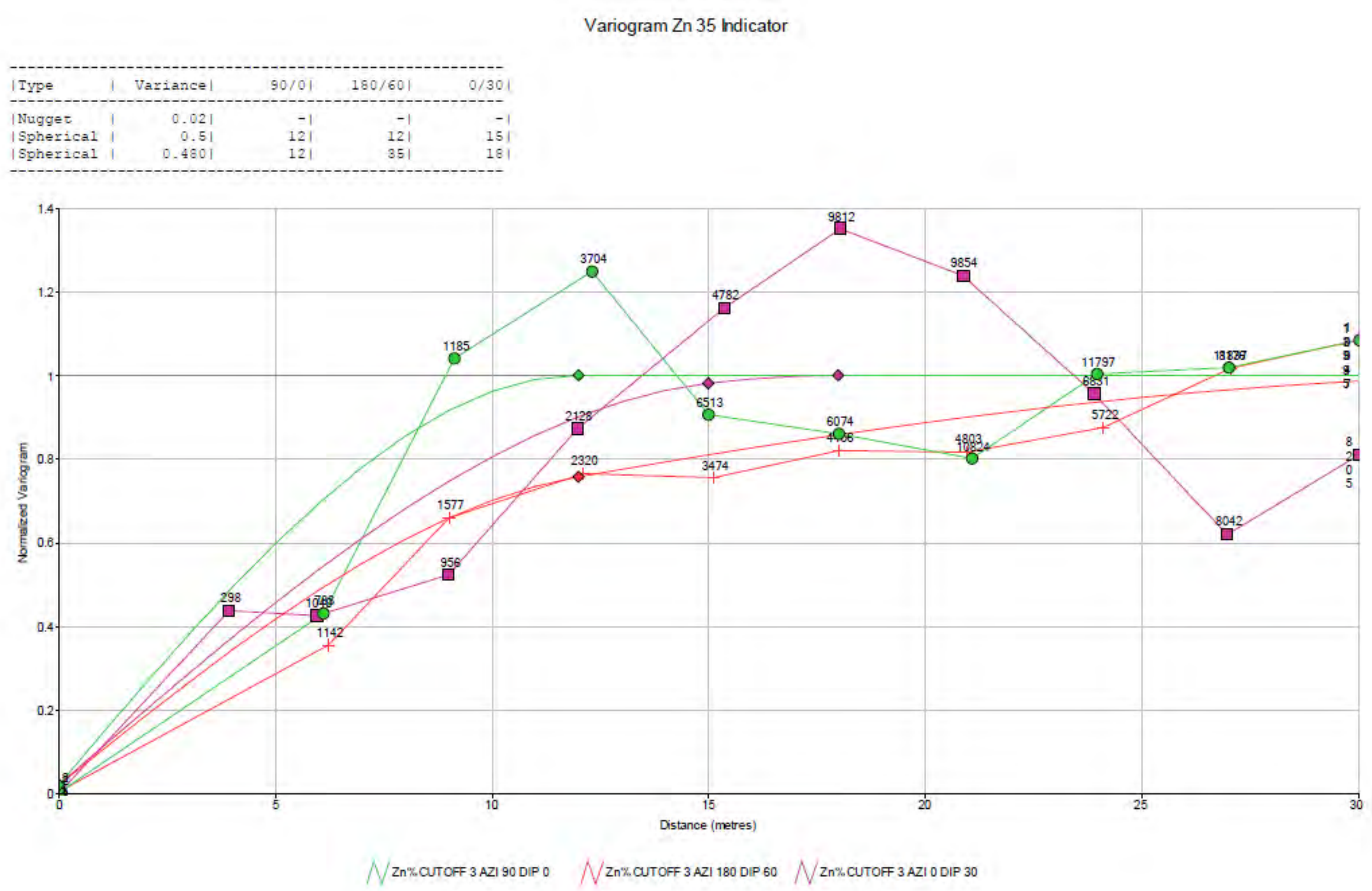
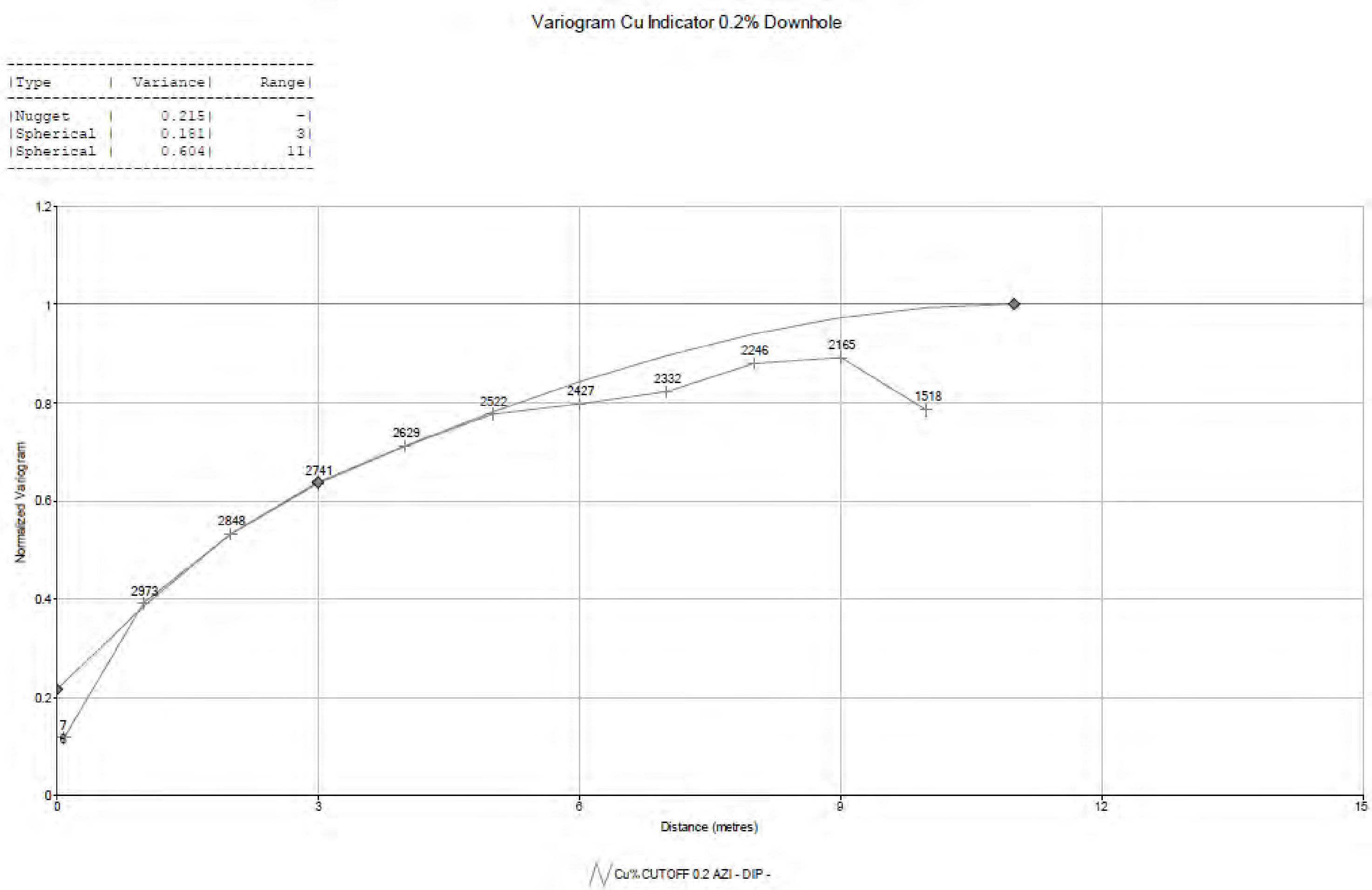


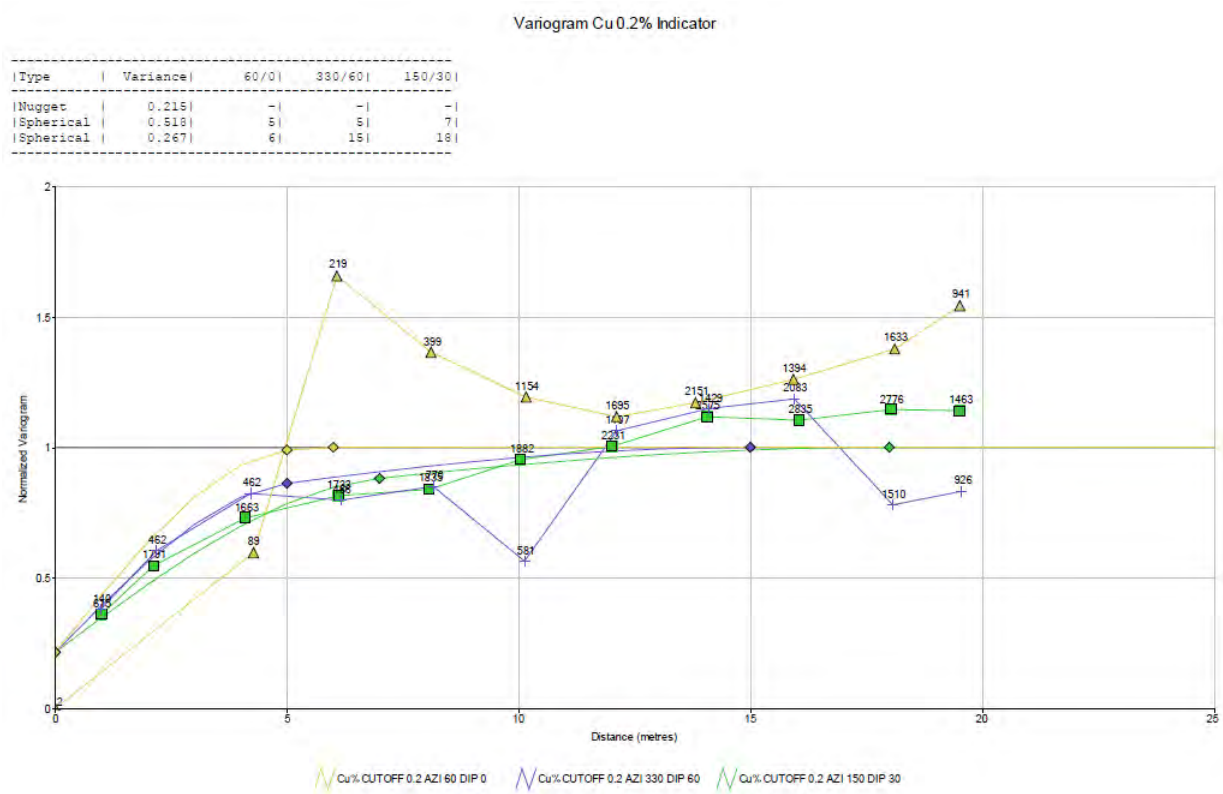
**Figure 10 - Zn 3% Indicator Variogram**



## Figure 11 - Cu 0.2% Downhole Indicator Variogram

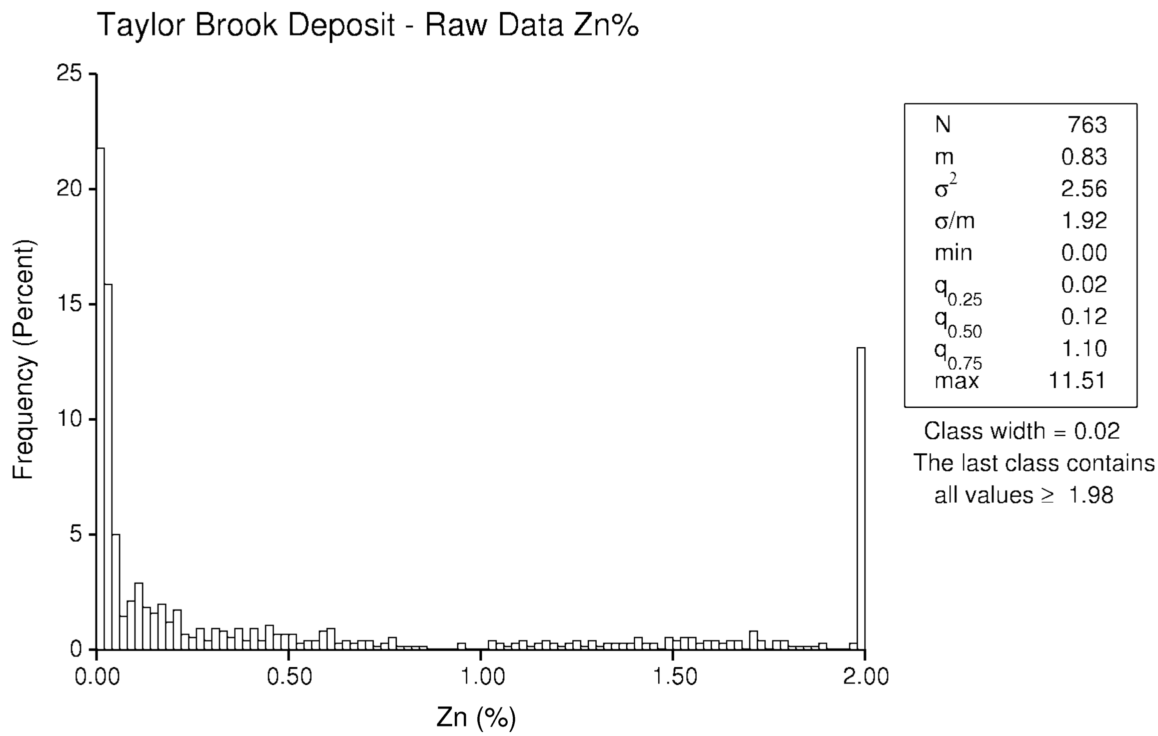


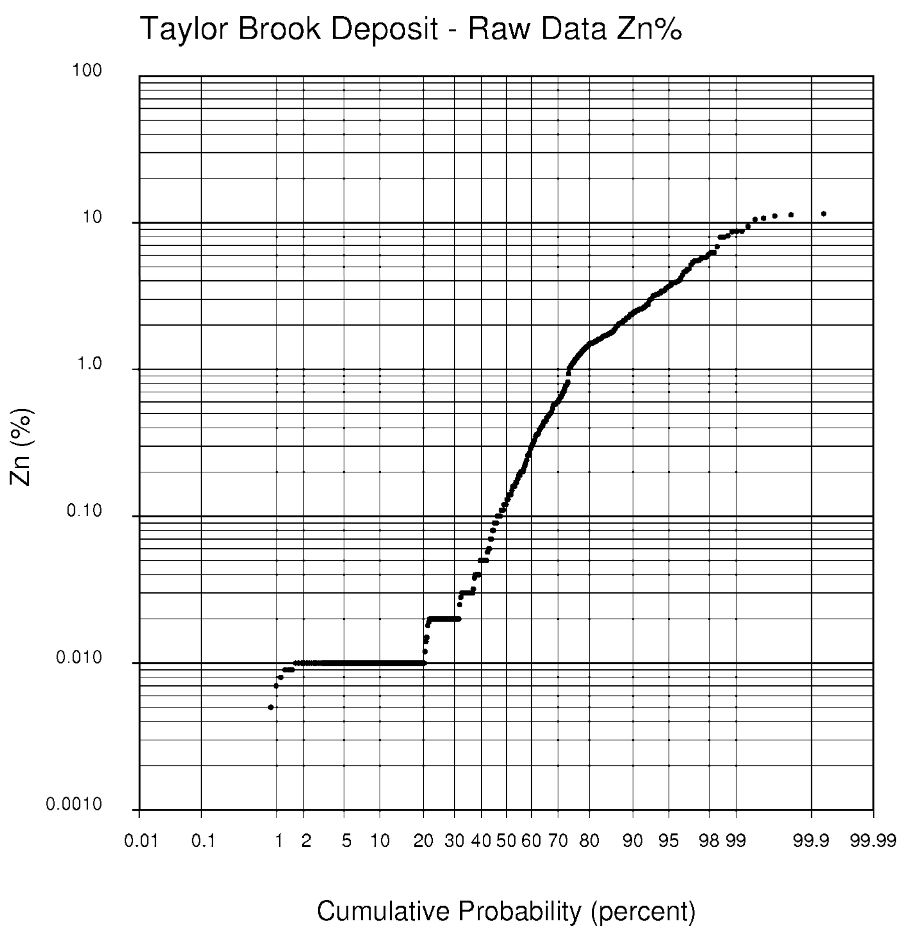
## Figure 12 - Cu 0.2% Indicator Variogram



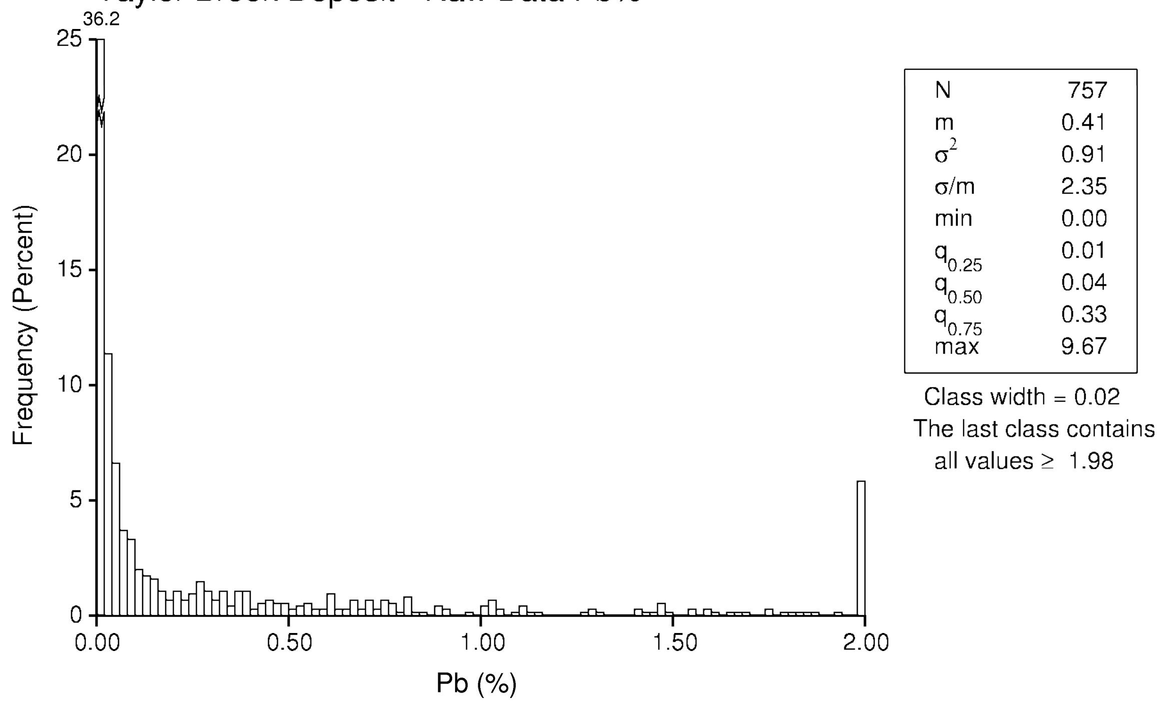
# APPENDIX H

## TAYLOR BROOK HISTORGRAMS

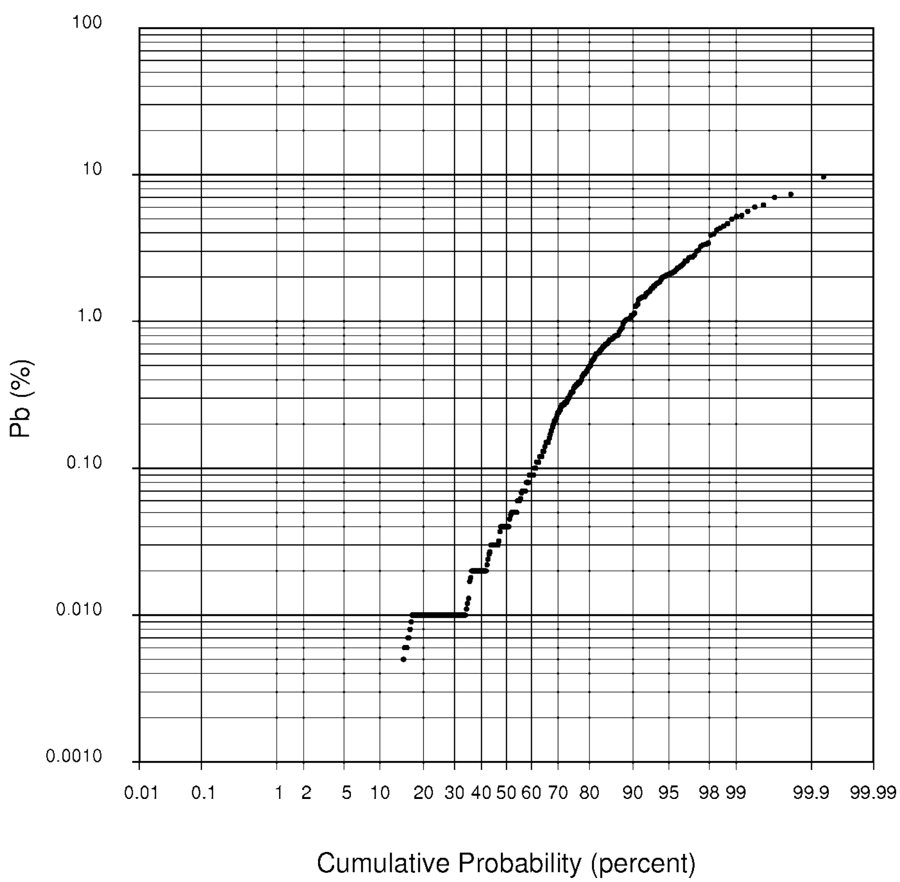

Taylor Brook Deposit - Raw Data Zn%
Frequency (Percent)
Zn (%)
0.00
0.50
1.00
1.50
2.00
0
5
10
15
20
25
N 763
m 0.83
σ2 2.56
σ/m 1.92
min 0.00
q0.25 0.02
q0.50 0.12
q0.75 1.10
max 11.51
Class width = 0.02
The last class contains
all values ≥ 1.98

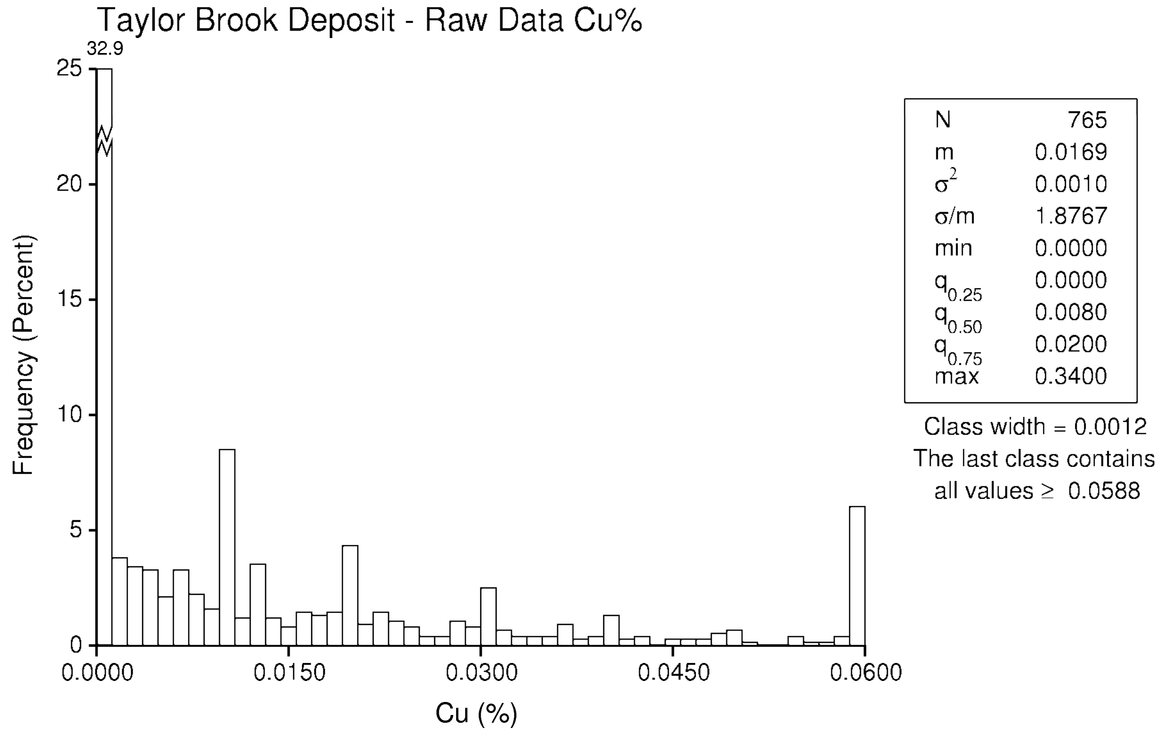

Taylor Brook Deposit - Raw Data Zn%
Zn (%)
100
10
1.0
0.10
0.010
0.0010
0.01 0.1 1 2 5 10 20 30 40 50 60 70 80 90 95 98 99 99.9 99.99
Cumulative Probability (percent)

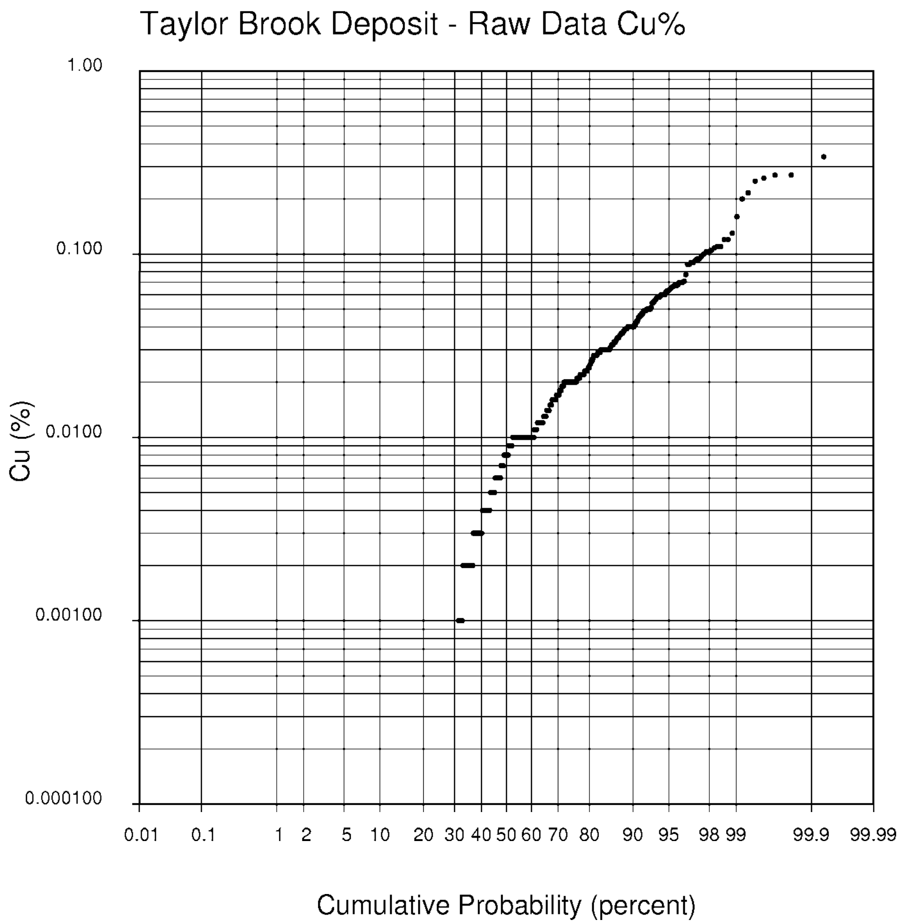
Taylor Brook Deposit - Raw Data Pb%

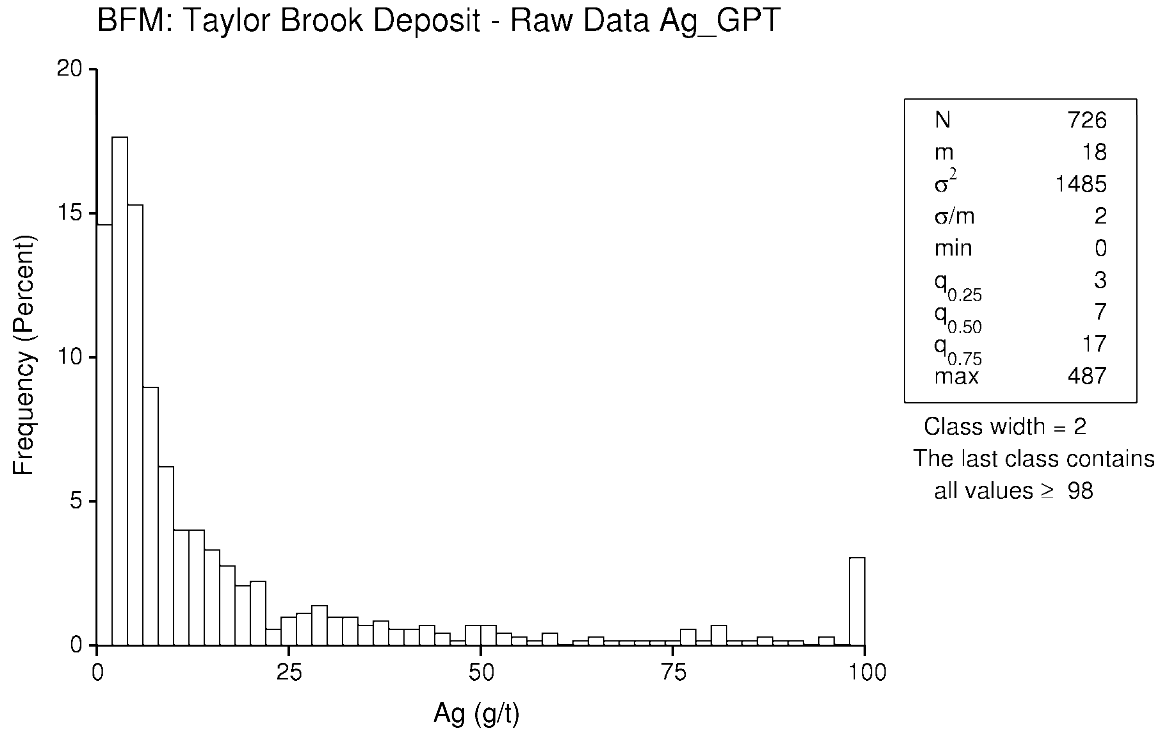



Taylor Brook Deposit - Raw Data Pb%

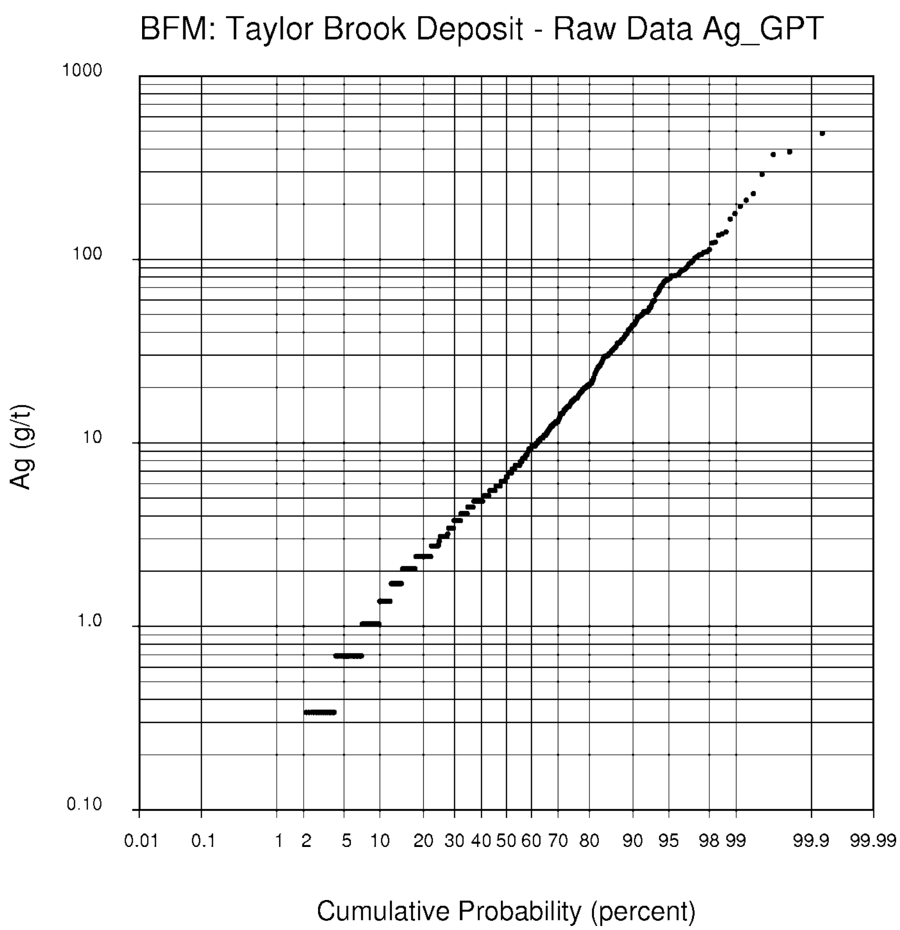



Taylor Brook Deposit - Raw Data Cu%
32.9
25
20
15
10
5
0
Frequency (Percent)
0.0000
0.0150
0.0300
0.0450
0.0600
Cu (%)
N 765
m 0.0169
$\sigma^2$ 0.0010
$\sigma$/m 1.8767
min 0.0000
$q_{0.25}$ 0.0000
$q_{0.50}$ 0.0080
$q_{0.75}$ 0.0200
max 0.3400
Class width = 0.0012
The last class contains
all values ≥ 0.0588

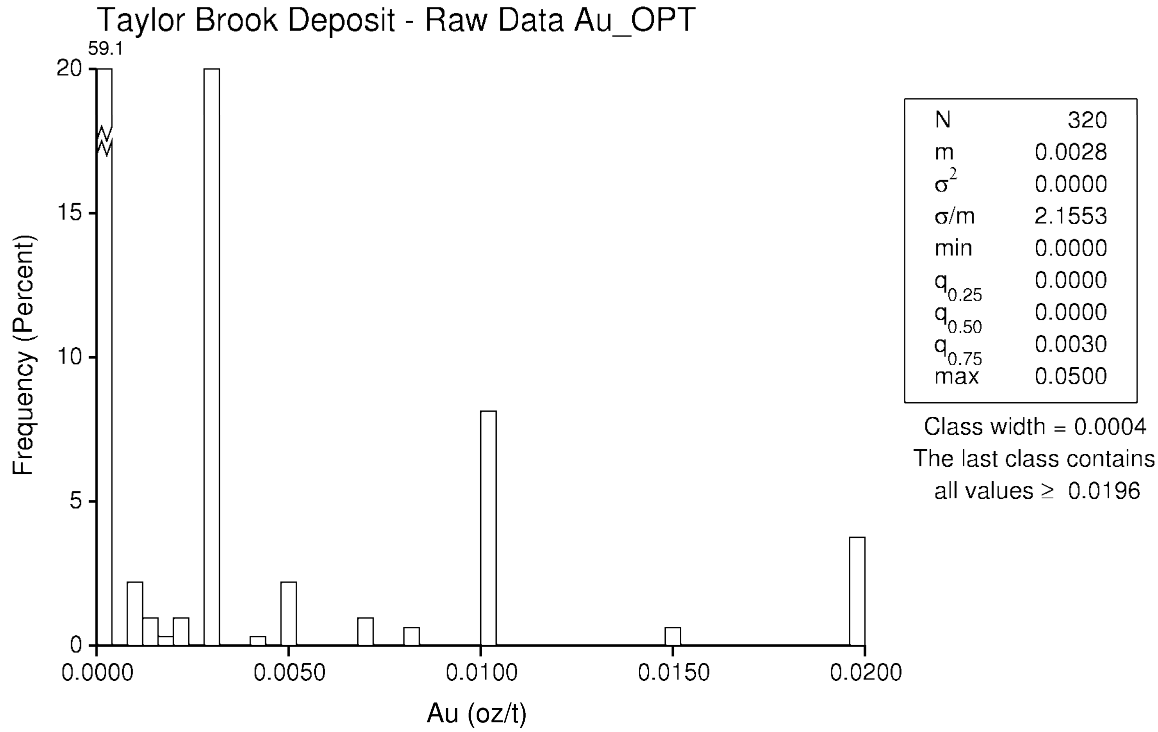

Taylor Brook Deposit - Raw Data Cu%
1.00
0.100
0.0100
0.00100
0.000100
Cu (%)
0.01
0.1
1
2
5
10
20
30
40
50
60
70
80
90
95
98
99
99.9
99.99
Cumulative Probability (percent)



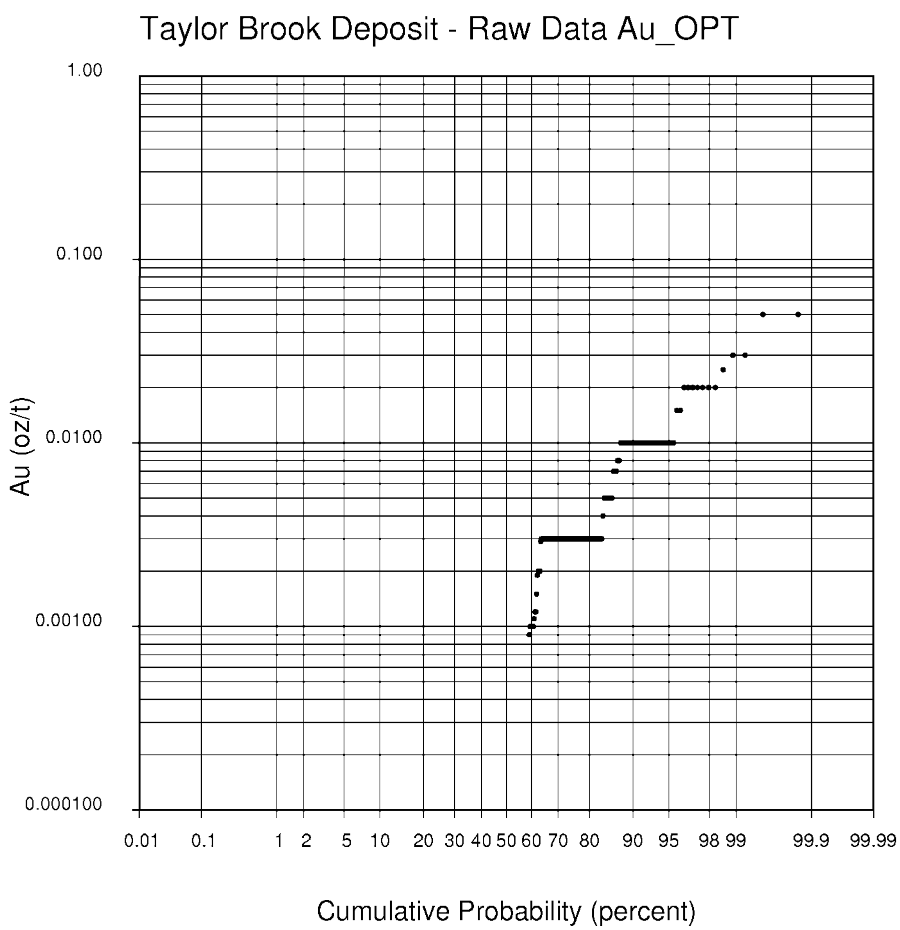
BFM: Taylor Brook Deposit - Raw Data Ag_GPT
Frequency (Percent)
Ag (g/t)
N 726
m 18
σ² 1485
σ/m 2
min 0
q0.25 3
q0.50 7
q0.75 17
max 487
Class width = 2
The last class contains
all values ≥ 98

BFM: Taylor Brook Deposit - Raw Data Ag_GPT
Ag (g/t)
Cumulative Probability (percent)



Taylor Brook Deposit - Raw Data Au_OPT
59.1
20
15
10
5
0
Frequency (Percent)
0.0000
0.0050
0.0100
0.0150
0.0200
Au (oz/t)
N 320
m 0.0028
σ2 0.0000
σ/m 2.1553
min 0.0000
q0.25 0.0000
q0.50 0.0000
q0.75 0.0030
max 0.0500
Class width = 0.0004
The last class contains
all values ≥ 0.0196

Taylor Brook Deposit - Raw Data Au_OPT
1.00
0.100
0.0100
0.00100
0.000100
Au (oz/t)
0.01 0.1 1 2 5 10 20 30 40 50 60 70 80 90 95 98 99 99.9 99.99
Cumulative Probability (percent)

# APPENDIX I

## Taylor Brook Capping Reports

# TBD_Zn

## Decile Analysis

| From | To | Count | ------------ Element------------ Mean | Min | Max | Total Metal | Percent | Capping Note |
|---|---|---|---|---|---|---|---|---|
| Decile | | | | | | | | |
| 0 | 10 | 75 | 0.009 | 0.000 | 0.010 | 0.66 | 0.14 | |
| 10 | 20 | 75 | 0.010 | 0.010 | 0.012 | 0.69 | 0.14 | |
| 20 | 30 | 76 | 0.020 | 0.013 | 0.020 | 1.50 | 0.31 | |
| 30 | 40 | 75 | 0.034 | 0.020 | 0.050 | 2.46 | 0.51 | |
| 40 | 50 | 75 | 0.087 | 0.050 | 0.130 | 6.23 | 1.29 | |
| 50 | 60 | 76 | 0.200 | 0.130 | 0.310 | 14.04 | 2.90 | |
| 60 | 70 | 75 | 0.454 | 0.310 | 0.605 | 31.73 | 6.56 | |
| 70 | 80 | 76 | 1.071 | 0.610 | 1.490 | 67.83 | 14.02 | |
| 80 | 90 | 75 | 1.835 | 1.490 | 2.410 | 108.25 | 22.37 | |
| 90 | 100 | 76 | 4.667 | 2.440 | 11.510 | 250.55 | 51.77 | >40 >2.3x -- >50 <3x |
| Percentile | | | | | | | | |
| 90 | 91 | 8 | 2.494 | 2.440 | 2.560 | 13.51 | 2.79 | |
| 91 | 92 | 7 | 2.604 | 2.570 | 2.670 | 20.95 | 4.33 | |
| 92 | 93 | 8 | 2.869 | 2.690 | 3.040 | 17.55 | 3.63 | |
| 93 | 94 | 7 | 3.221 | 3.170 | 3.290 | 19.78 | 4.09 | |
| 94 | 95 | 8 | 3.476 | 3.300 | 3.660 | 23.26 | 4.81 | |
| 95 | 96 | 7 | 3.854 | 3.700 | 3.960 | 25.04 | 5.17 | |
| 96 | 97 | 8 | 4.585 | 4.030 | 5.170 | 31.15 | 6.44 | |
| 97 | 98 | 7 | 5.600 | 5.380 | 5.810 | 27.72 | 5.73 | |
| 98 | 99 | 8 | 7.283 | 6.090 | 8.660 | 29.20 | 6.03 | |
| 99 | 100 | 8 | 10.263 | 8.720 | 11.510 | 42.40 | 8.76 | <10 <1.75x -- <15 <2x |
| Total | | | | | | | | |
| 0 | 100 | 754 | 0.842 | 0.000 | 11.510 | 483.95 | 100.00 | |

Interpretation notes:

Capping is warranted if

The last decile has more than 40 percent of metal; or,
the last decile contains more than 2.3 times the metal quantity contained in the one before last; or,
the last centile contains more than 10 percent of metal; or,
the last centile contains more than 1.75 times the metal quantity contained in the one before last.

Exception will be made if all following conditions are met:
The last decile has more than 50 percent metal; and,
the last decile contains more than 3 times the metal quantity contained in the one before last; and,
the last centile contains more than 15 percent of the metal; and,
the last centile contains more than 2 times the metal quantity contained in the one before last.

# TBD_Pb

## Decile Analysis

| From | To | Count | ------------ Element------------ Mean | Min | Max | Total Metal | Percent | Capping Note |
|---|---|---|---|---|---|---|---|---|
| Decile | | | | | | | | |
| 0 | 10 | 75 | 0.000 | 0.000 | 0.000 | 0.00 | 0.00 | |
| 10 | 20 | 76 | 0.005 | 0.000 | 0.010 | 0.36 | 0.16 | |
| 20 | 30 | 75 | 0.010 | 0.010 | 0.010 | 0.75 | 0.34 | |
| 30 | 40 | 76 | 0.015 | 0.010 | 0.020 | 1.01 | 0.45 | |
| 40 | 50 | 75 | 0.030 | 0.020 | 0.040 | 1.96 | 0.88 | |
| 50 | 60 | 76 | 0.061 | 0.040 | 0.090 | 4.19 | 1.88 | |
| 60 | 70 | 76 | 0.145 | 0.090 | 0.240 | 10.49 | 4.71 | |
| 70 | 80 | 75 | 0.345 | 0.240 | 0.490 | 23.01 | 10.32 | |
| 80 | 90 | 76 | 0.751 | 0.490 | 1.100 | 49.46 | 22.19 | |
| 90 | 100 | 76 | 2.688 | 1.110 | 9.670 | 131.66 | 59.07 | >40 >2.3x -- >50 <3x |
| Percentile | | | | | | | | |
| 90 | 91 | 7 | 1.216 | 1.110 | 1.310 | 7.05 | 3.17 | |
| 91 | 92 | 8 | 1.444 | 1.400 | 1.470 | 8.68 | 3.89 | |
| 92 | 93 | 8 | 1.586 | 1.490 | 1.680 | 8.93 | 4.01 | |
| 93 | 94 | 7 | 1.774 | 1.690 | 1.840 | 10.23 | 4.59 | |
| 94 | 95 | 8 | 1.995 | 1.860 | 2.080 | 10.71 | 4.80 | |
| 95 | 96 | 7 | 2.177 | 2.090 | 2.310 | 12.64 | 5.67 | |
| 96 | 97 | 8 | 2.522 | 2.320 | 2.730 | 13.96 | 6.26 | |
| 97 | 98 | 7 | 3.073 | 2.740 | 3.340 | 16.53 | 7.42 | |
| 98 | 99 | 8 | 4.230 | 3.410 | 4.990 | 19.67 | 8.82 | |
| 99 | 100 | 8 | 6.549 | 5.200 | 9.670 | 23.26 | 10.44 | >10 <1.75x -- <15 <2x |
| Total | | | | | | | | |
| 0 | 100 | 756 | 0.407 | 0.000 | 9.670 | 222.87 | 100.00 | |

Interpretation notes:

Capping is warranted if

The last decile has more than 40 percent of metal; or,
the last decile contains more than 2.3 times the metal quantity contained in the one before last; or,
the last centile contains more than 10 percent of metal; or,
the last centile contains more than 1.75 times the metal quantity contained in the one before last.

Exception will be made if all following conditions are met:
The last decile has more than 50 percent metal; and,
the last decile contains more than 3 times the metal quantity contained in the one before last; and,
the last centile contains more than 15 percent of the metal; and,
the last centile contains more than 2 times the metal quantity contained in the one before last.

# TBD_Cu

## Decile Analysis

| From | To | Count | ------------ Element------------ Mean | Min | Max | Total Metal | Percent | Capping Note |
|---|---|---|---|---|---|---|---|---|
| Decile | | | | | | | | |
| 0 | 10 | 76 | 0.000 | 0.000 | 0.000 | 0.00 | 0.00 | |
| 10 | 20 | 76 | 0.000 | 0.000 | 0.000 | 0.00 | 0.00 | |
| 20 | 30 | 77 | 0.000 | 0.000 | 0.000 | 0.00 | 0.00 | |
| 30 | 40 | 76 | 0.002 | 0.000 | 0.003 | 0.13 | 1.23 | |
| 40 | 50 | 77 | 0.005 | 0.003 | 0.008 | 0.37 | 3.53 | |
| 50 | 60 | 76 | 0.010 | 0.008 | 0.010 | 0.67 | 6.31 | |
| 60 | 70 | 76 | 0.013 | 0.010 | 0.017 | 0.92 | 8.76 | |
| 70 | 80 | 77 | 0.020 | 0.017 | 0.024 | 1.38 | 13.05 | |
| 80 | 90 | 76 | 0.032 | 0.024 | 0.040 | 2.02 | 19.17 | |
| 90 | 100 | 77 | 0.086 | 0.040 | 0.340 | 5.06 | 47.95 | >40 >2.3x -- <50 <3x |
| Percentile | | | | | | | | |
| 90 | 91 | 8 | 0.042 | 0.040 | 0.045 | 0.21 | 1.98 | |
| 91 | 92 | 7 | 0.047 | 0.045 | 0.049 | 0.22 | 2.10 | |
| 92 | 93 | 8 | 0.050 | 0.049 | 0.051 | 0.36 | 3.38 | |
| 93 | 94 | 8 | 0.056 | 0.054 | 0.058 | 0.43 | 4.08 | |
| 94 | 95 | 7 | 0.061 | 0.059 | 0.063 | 0.41 | 3.86 | |
| 95 | 96 | 8 | 0.066 | 0.063 | 0.068 | 0.41 | 3.87 | |
| 96 | 97 | 8 | 0.078 | 0.070 | 0.090 | 0.50 | 4.77 | |
| 97 | 98 | 7 | 0.096 | 0.090 | 0.103 | 0.61 | 5.76 | |
| 98 | 99 | 8 | 0.113 | 0.103 | 0.130 | 0.57 | 5.39 | |
| 99 | 100 | 8 | 0.246 | 0.160 | 0.340 | 1.35 | 12.77 | >10 >1.75x -- <15 >2x |
| Total | | | | | | | | |
| 0 | 100 | 764 | 0.017 | 0.000 | 0.340 | 10.56 | 100.00 | |

Interpretation notes:

Capping is warranted if
The last decile has more than 40 percent of metal; or,
the last decile contains more than 2.3 times the metal quantity contained in the one before last; or,
the last centile contains more than 10 percent of metal; or,
the last centile contains more than 1.75 times the metal quantity contained in the one before last.

Exception will be made if all following conditions are met:
The last decile has more than 50 percent metal; and,
the last decile contains more than 3 times the metal quantity contained in the one before last; and,
the last centile contains more than 15 percent of the metal; and,
the last centile contains more than 2 times the metal quantity contained in the one before last.

# TBD_Ag

## Decile Analysis

| | | | ------------ Element------------- | | | Total | | |
|---|---|---|---|---|---|---|---|---|
| **From** | **To** | **Count** | **Mean** | **Min** | **Max** | **Metal** | **Percent** | **Capping Note** |
| Decile | | | | | | | | |
| 0 | 10 | 31 | 0.429 | 0.010 | 3.100 | 9.60 | 8.21 | |
| 10 | 20 | 31 | 0.053 | 0.020 | 0.070 | 1.21 | 1.03 | |
| 20 | 30 | 31 | 0.095 | 0.070 | 0.120 | 2.70 | 2.31 | |
| 30 | 40 | 32 | 0.134 | 0.120 | 0.150 | 4.24 | 3.62 | |
| 40 | 50 | 31 | 0.158 | 0.150 | 0.180 | 4.89 | 4.18 | |
| 50 | 60 | 31 | 0.205 | 0.180 | 0.240 | 5.46 | 4.66 | |
| 60 | 70 | 32 | 0.307 | 0.240 | 0.370 | 6.75 | 5.77 | |
| 70 | 80 | 31 | 0.481 | 0.380 | 0.580 | 12.32 | 10.53 | |
| 80 | 90 | 31 | 0.985 | 0.580 | 1.530 | 22.76 | 19.45 | |
| 90 | 100 | 32 | 2.650 | 0.081 | 14.210 | 47.11 | 40.25 | >40 <2.3x -- <50 <3x |
| Percentile | | | | | | | | |
| 90 | 91 | 3 | 1.817 | 1.590 | 2.220 | 3.57 | 3.05 | |
| 91 | 92 | 3 | 2.337 | 2.260 | 2.380 | 2.77 | 2.36 | |
| 92 | 93 | 4 | 3.255 | 2.790 | 3.950 | 7.66 | 6.54 | |
| 93 | 94 | 3 | 6.167 | 4.840 | 8.480 | 11.84 | 10.12 | |
| 94 | 95 | 3 | 0.084 | 0.081 | 0.085 | 0.21 | 0.18 | |
| 95 | 96 | 3 | 0.120 | 0.091 | 0.175 | 0.55 | 0.47 | |
| 96 | 97 | 3 | 0.222 | 0.209 | 0.229 | 0.78 | 0.67 | |
| 97 | 98 | 3 | 0.301 | 0.271 | 0.324 | 1.86 | 1.59 | |
| 98 | 99 | 3 | 0.527 | 0.383 | 0.613 | 1.85 | 1.58 | |
| 99 | 100 | 4 | 9.266 | 0.704 | 14.210 | 16.02 | 13.69 | >10 >1.75x -- <15 >2x |
| Total | | | | | | | | |
| 0 | 100 | 313 | 0.554 | 0.010 | 14.210 | 117.04 | 100.00 | |

Interpretation notes:

Capping is warranted if

The last decile has more than 40 percent of metal; or,
the last decile contains more than 2.3 times the metal quantity contained in the one before last; or,
the last centile contains more than 10 percent of metal; or,
the last centile contains more than 1.75 times the metal quantity contained in the one before last.

Exception will be made if all following conditions are met:
The last decile has more than 50 percent metal; and,
the last decile contains more than 3 times the metal quantity contained in the one before last; and,
the last centile contains more than 15 percent of the metal; and,
the last centile contains more than 2 times the metal quantity contained in the one before last.

# TBD_Au

## Decile Analysis

| From | To | Count | Mean (Element) | Min (Element) | Max (Element) | Total Metal | Percent | Capping Note |
|---|---|---|---|---|---|---|---|---|
| Decile | | | | | | | | |
| 0 | 10 | 31 | 0.000 | 0.000 | 0.000 | 0.00 | 0.00 | |
| 10 | 20 | 32 | 0.000 | 0.000 | 0.000 | 0.00 | 0.00 | |
| 20 | 30 | 32 | 0.000 | 0.000 | 0.000 | 0.00 | 0.00 | |
| 30 | 40 | 32 | 0.000 | 0.000 | 0.000 | 0.00 | 0.00 | |
| 40 | 50 | 32 | 0.000 | 0.000 | 0.000 | 0.00 | 0.00 | |
| 50 | 60 | 32 | 0.000 | 0.000 | 0.001 | 0.00 | 0.64 | |
| 60 | 70 | 32 | 0.002 | 0.001 | 0.003 | 0.08 | 10.75 | |
| 70 | 80 | 32 | 0.003 | 0.003 | 0.003 | 0.10 | 12.81 | |
| 80 | 90 | 32 | 0.006 | 0.003 | 0.010 | 0.17 | 21.50 | |
| 90 | 100 | 32 | 0.017 | 0.010 | 0.050 | 0.42 | 54.30 | >40 >2.3x -- >50 <3x |
| Percentile | | | | | | | | |
| 90 | 91 | 3 | 0.010 | 0.010 | 0.010 | 0.02 | 2.26 | |
| 91 | 92 | 3 | 0.010 | 0.010 | 0.010 | 0.01 | 1.86 | |
| 92 | 93 | 3 | 0.010 | 0.010 | 0.010 | 0.04 | 4.53 | |
| 93 | 94 | 3 | 0.010 | 0.010 | 0.010 | 0.03 | 3.24 | |
| 94 | 95 | 4 | 0.010 | 0.010 | 0.010 | 0.05 | 6.68 | |
| 95 | 96 | 3 | 0.012 | 0.010 | 0.015 | 0.04 | 5.29 | |
| 96 | 97 | 3 | 0.018 | 0.015 | 0.020 | 0.06 | 8.05 | |
| 97 | 98 | 3 | 0.020 | 0.020 | 0.020 | 0.06 | 7.63 | |
| 98 | 99 | 3 | 0.022 | 0.020 | 0.025 | 0.03 | 3.69 | |
| 99 | 100 | 4 | 0.040 | 0.030 | 0.050 | 0.09 | 11.06 | >10 >1.75x -- <15 >2x |
| Total | | | | | | | | |
| 0 | 100 | 319 | 0.003 | 0.000 | 0.050 | 0.77 | 100.00 | |

Interpretation notes:

Capping is warranted if
The last decile has more than 40 percent of metal; or,
the last decile contains more than 2.3 times the metal quantity contained in the one before last; or,
the last centile contains more than 10 percent of metal; or,
the last centile contains more than 1.75 times the metal quantity contained in the one before last.

Exception will be made if all following conditions are met:
The last decile has more than 50 percent metal; and,
the last decile contains more than 3 times the metal quantity contained in the one before last; and,
the last centile contains more than 15 percent of the metal; and,
the last centile contains more than 2 times the metal quantity contained in the one before last.

# APPENDIX J

## FINANCIALS

| CNE-TB MINERALS ECONOMIC PARAMETERS FOR PIT OPTIMIZATION | | | | | | |
|---|---|---|---|---|---|---|
| ITEMS | | UNITS | UNITS | Whittle #3<br>CNE-TB<br>(April 27, 2011) | Whittle #2B<br>CNE-TB<br>(March 21, 2011) | Whittle #2A<br>CNE-TB<br>(March 21, 2011) |
| Exchange Rate | | Cdn$:US$ | Cdn$:US$ | 1.023 | 1.023 | 1.023 |
| Discount Rate | | % | % | 8 | 8 | 8 |
| Metal Price (Market) | Zinc (Zn) | US $/drylb | US $/drylb | 1.22 | 1.06 | 1.06 |
| | Lead (Pb) | US $/drylb | US $/drylb | 1.10 | 0.99 | 0.99 |
| | Copper (Cu) | US $/drylb | US $/drylb | 3.62 | 3.01 | 3.01 |
| | Cobalt (Co) | US $/drylb | US $/drylb | 18.32 | 18.51 | 18.51 |
| | Silver (Ag) | US $/oz | US $/oz | 22.74 | 19.73 | 19.73 |
| | Gold (Au) | US $/oz | US $/oz | 1,258 | 1,221 | 1,221 |
| Metal Recoveries | Zinc (Zn) | % | % | 76.5% | 76.5% | 76.5% |
| | Lead (Pb) | % | % | 80.8% | 80.8% | 80.8% |
| | Copper (Cu) | % | % | 82.0% | 82.0% | 82.0% |
| | Cobalt (Co) | % | % | 0.0% | 0.0% | 0.0% |
| | Silver (Ag) | % | % | 45.0% | 45.0% | 45.0% |
| | Gold (Au) | % | % | | | |
| Concentrate Grade | Zinc (Zn) | % | % | 50% | 50% | 50% |
| | Lead (Pb) | % | % | 47% | 47% | 47% |
| | Copper (Cu) | % | % | 28% | 28% | 28% |
| | Cobalt (Co) | % | % | | | |
| | Gold (Au) | % | % | | | |
| | Silver (Ag) | % | % | | | |
| Concentrate Costs | Transport Land (Zinc (Zn)) | US $/wet tonne | US $/wet tonne | $7.50 | $7.50 | $7.50 |
| | Transport Land (Lead (Pb)) | US $/wet tonne | US $/wet tonne | $7.50 | $7.50 | $7.50 |
| | Transport Land (Copper (Cu)) | US $/wet tonne | US $/wet tonne | $7.50 | $7.50 | $7.50 |
| | Ocean Freight Charge - Europe or N. America (Zinc (Zn)) | US $/wet tonne | US $/wet tonne | $0.00 | $0.00 | $0.00 |
| | Ocean Freight Charge - Europe or N. America (Lead (Pb)) | US $/wet tonne | US $/wet tonne | $40.00 | $40.00 | $40.00 |
| | Ocean Freight Charge - Europe or N. America (Cobalt (Co)) | | | | | |
| | Ocean Freight Charge - Mainland China (Copper (Cu)) | US $/wet tonne | US $/wet tonne | $115.00 | $115.00 | $115.00 |
| | Stevedoring Charge (Zinc (Zn)) | US $/wet tonne | US $/wet tonne | $0.00 | $0.00 | $0.00 |
| | Stevedoring Charge (Lead (Pb)) | US $/wet tonne | US $/wet tonne | $15.00 | $15.00 | $15.00 |
| | Stevedoring Charge (Cobalt (Co)) | | | | | |
| | Stevedoring Charge (Copper (Cu)) | US $/wet tonne | US $/wet tonne | $15.00 | $15.00 | $15.00 |
| | Representation - Zinc (Zn) | US $/wet tonne | US $/wet tonne | $0.00 | $0.00 | $0.00 |
| | Representation - Lead (Pb) | US $/wet tonne | US $/wet tonne | $3.00 | $3.00 | $3.00 |
| | Representation - Lead (Co) | | | | | |
| | Representation - Copper (Cu) | US $/wet tonne | US $/wet tonne | $3.00 | $3.00 | $3.00 |
| | Miscellaneous - Zinc (Zn) | US $/wet tonne | US $/wet tonne | $10.00 | $10.00 | $10.00 |
| | Miscellaneous - Lead (Pb) | US $/wet tonne | US $/wet tonne | $10.00 | $10.00 | $10.00 |
| | Miscellaneous - Lead (Co) | | | | | |
| | Miscellaneous - Copper (Cu) | US $/wet tonne | US $/wet tonne | $5.00 | $5.00 | $5.00 |
| Refining Charge | Lead (Pb) Concentrate - Ag refining charge | US $/oz | US $/oz | $0.75 | $0.75 | $0.75 |
| | Copper (Cu) Concentrate | US $/tonne dry | US $/tonne dry | $80.00 | $80.00 | $80.00 |
| | Cobalt (Co) Concentrate | | | | | |
| Moisture Content | Zinc (Zn) | % | % | 8% | 8% | 8% |
| | Lead (Pb) | % | % | 8% | 8% | 8% |
| | Copper (Cu) | % | % | 8% | 8% | 8% |
| | Cobalt (Co) | % | % | | | |
| Treatment Charges | Zinc (Zn) | US $/tonne dry | US $/tonne dry | $275 | $275 | $275 |
| | Lead (Pb) | US $/tonne dry | US $/tonne dry | $140 | $140 | $140 |
| | Copper (Cu) | US $/tonne dry | US $/tonne dry | $60 | $60 | $60 |
| | Cobalt (Co) | US $/tonne dry | US $/tonne dry | | | |
| Royalty Costs | Concentrate Agent Fee (Zinc (Zn)) | US $/tonne dry | US $/tonne dry | $25 | $25 | $25 |
| | Concentrate Agent Fee (Lead (Pb)) | US $/tonne dry | US $/tonne dry | $25 | $25 | $25 |
| | Concentrate Agent Fee (Copper (Cu)) | US $/tonne dry | US $/tonne dry | $25 | $25 | $25 |
| | Third Party NSR (% of total revenue less treatment & transport costs): | Zinc (Zn) | Zinc (Zn) | 1% | 1% | 1% |
| | Third Party NSR (% of total revenue less treatment & transport costs): | Lead (Pb) | Lead (Pb) | 1% | 1% | 1% |
| | Third Party NSR (% of total revenue less treatment & transport costs): | Cobolt (Co) | Cobolt (Co) | 0% | 0% | 0% |
| | Third Party NSR (% of total revenue less treatment & transport costs): | Copper (Cu) | Copper (Cu) | 1% | 1% | 1% |
| **Operating Cost** | Mining Cost (OB) | US $/t | US $/t | 1.10 | 1.10 | 1.10 |
| | Mining Cost (Waste) | US $/t | US $/t | 2.28 | 2.28 | 2.28 |
| | Mining Cost (Ore) | US $/t | US $/t | 2.84 | 2.84 | 2.84 |
| | Re-handled of total ore production | % | % | 100% | 100% | 100% |
| | Stockpile Rehandling for Ore (filling and hauling) | US $/ore t | US $/ore t | 0.49 | 0.49 | 0.49 |
| | Crushing | US $/t | US $/t | - | - | - |
| | Hauling | US $/t | US $/t | 1.09 | 1.09 | 1.09 |
| | Processing | US $/t processed | US $/t processed | 18.90 | 18.90 | 12.25 |
| | G & A + Time Cost | US $/tonne ore | US $/tonne ore | 1.96 | 1.96 | 1.96 |
| **Selling Cost** ( concentrate hauling, smelting and refinery charge) | Zinc (Zn) | US $/lb | US $/lb | - | 0.297 | 0.297 |
| | Lead (Pb) | US $/lb | US $/lb | - | 0.245 | 0.245 |
| | Copper (Cu) | US $/lb | US $/lb | - | 0.546 | 0.546 |
| | Cobalt (Co) | US $/lb | US $/lb | | | |
| | Silver (Ag) | US $/oz | US $/oz | - | 0.750 | 0.750 |
| | Gold (Au) | US $/oz | US $/oz | | | |
| Overall Pit Slope Angle | OB | Degree | Degree | 20 | 20 | 20 |
| | ore | Degree | Degree | 45 | 45 | 45 |
| | Waste | Degree | Degree | 45 | 45 | 45 |
| Mining Recovery | Should be completed in Whittle running | % | % | 95% | 95% | 95% |
| Mining Dilution | Should be completed in Whittle running | % | % | 5% | 5% | 5% |
| Mill Throughput | | t/d | t/d | | | |
| Mining Capacity | | t/d | t/d | | | |

## Stratabound CNE Final Pit Designs - Schedule_B (Optimization #2A)

| | | Quarter 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Days => | 87.5 | 87.5 | 87.5 | 87.5 | 87.5 | 87.5 | 87.5 | 87.5 | 87.5 | 87.5 | 87.5 | |
| OB CNE | tonnes | 243,829 | 78,928 | 2,533 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 325,290 |
| OB Captain | tonnes | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Total OB | tonnes | 243,829 | 78,928 | 2,533 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 325,290 |
| Waste CNE | tonnes | 75,610 | 171,056 | 243,788 | 142,939 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 633,393 |
| Waste Captain | tonnes | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Total Waste | tonnes | 75,610 | 171,056 | 243,788 | 142,939 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 633,393 |
| Ore CNE | tonnes | 32,987 | 102,441 | 106,105 | 83,488 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 325,021 |
| Ore CNE - Lead | % | 2.25 | 2.06 | 1.73 | 1.24 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 1.761 |
| Ore CNE - Silver | g/t | 60.12 | 60.31 | 55.32 | 58.07 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 58.087 |
| Ore CNE - Copper | % | 0.06 | 0.07 | 0.06 | 0.14 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.082 |
| Ore CNE - Cobalt | g/t | 3.18 | 11.60 | 16.99 | 37.92 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 19.267 |
| Ore CNE - Zinc | % | 5.60 | 5.30 | 4.75 | 3.68 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 4.735 |
| Ore Captain | tonnes | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Ore Captain - Copper | % | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | #DIV/0! |
| Ore Captain - Cobalt | % | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | #DIV/0! |
| Ore Captain - Gold | g/t | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | #DIV/0! |
| Total Ore | tonnes | 32,987 | 102,441 | 106,105 | 83,488 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 325,021 |
| Stripping Ratio CNE | | 9.68 | 2.44 | 2.32 | 1.71 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 2.95 |
| Stripping Ratio Captain | | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Total Stripping Ratio | | 9.68 | 2.44 | 2.32 | 1.71 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0 | 2.95 |
| | | 319,439 | 249,985 | 246,321 | 142,939 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 958,684 |

**NOTE: Re-schedule by quarters with only CNE pit. Mine entire pit within one year.**
**File: Stratabound #2A-CNE Schedule May 18th 2011_B.xlsx**

## Stratabound - CNE Design vs. Whittle - (Whittle Run #2A)

| | Total Waste (Tonnes) | Total Ore (Tonnes) | Lead (Pb) (%) | Silver (Ag) (g/t) | Copper (Cu) (%) | Cobalt (Co) (g/t) | Zinc (Zn) (%) | Cobalt (Co) (%) | Gold (Au) (g/t) | Stripping Ratio |
|---|---|---|---|---|---|---|---|---|---|---|
| Design-CNE | 958,684 | 325,021 | 1.76 | 58.09 | 0.08 | 19.27 | 4.74 | | | 2.95 |
| Whittle-CNE (Pit 31) | 907,000 | 339,000 | 1.72 | 57.37 | 0.08 | 20.85 | 4.61 | | | 2.67 |
| Variance-CNE | 51,684 | -13,979 | 0.04 | 0.72 | 0.00 | -1.58 | 0.13 | 0.00 | 0.00 | 0.28 |

## Stratabound CNE Final Pit Designs - Schedule_B (Optimization #2A) - using percentage

| | | Quarter 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Days => | 87.5 | 87.5 | 87.5 | 87.5 | 87.5 | 87.5 | 87.5 | 87.5 | 87.5 | 87.5 | 87.5 | |
| Total OB | tonnes | 243,829 | 78,928 | 2,533 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | | 325,290 |
| Total Waste | tonnes | 75,610 | 171,056 | 243,788 | 142,939 | 0 | 0 | 0 | 0 | 0 | 0 | | 633,393 |
| Total Ore | tonnes | 32,987 | 102,441 | 106,105 | 83,488 | 0 | 0 | 0 | 0 | 0 | 0 | | 325,021 |
| Stripping Ratio | | 9.68 | 2.44 | 2.32 | 1.71 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | | 2.95 |
| | | | | | | | | | | | | | |
| Total Material Mined | tonnes | 352,426 | 352,426 | 352,426 | 226,427 | 0 | 0 | 0 | 0 | 0 | 0 | | 1,283,705 |
| Mining OPEX-OB | $/tonne | $1.10 | $1.10 | $1.10 | $1.10 | $0.00 | $0.00 | $0.00 | $0.00 | $0.00 | $0.00 | | $1.10 |
| Mining OPEX-Waste | $/tonne | $2.28 | $2.28 | $2.28 | $2.28 | $0.00 | $0.00 | $0.00 | $0.00 | $0.00 | $0.00 | | $2.28 |
| Mining OPEX-Ore | $/tonne | $2.84 | $2.84 | $2.84 | $2.84 | $0.00 | $0.00 | $0.00 | $0.00 | $0.00 | $0.00 | | $2.84 |
| Onsite Crushing (6" minus) | $/tonne | $1.82 | $1.82 | $1.82 | $1.82 | $0.00 | $0.00 | $0.00 | $0.00 | $0.00 | $0.00 | | $1.82 |
| Ore Hauling to Mill | $/tonne | $8.00 | $8.00 | $8.00 | $8.00 | $0.00 | $0.00 | $0.00 | $0.00 | $0.00 | $0.00 | | $8.00 |
| Mining OPEX | $ | $858,215 | $1,773,737 | $1,901,909 | $1,382,860 | $0 | $0 | $0 | $0 | $0 | $0 | | $5,916,722 |
| Mining CAPEX | $ | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | | $0 |
| Total Mining Cost | $ | $858,215 | $1,773,737 | $1,901,909 | $1,382,860 | $0 | $0 | $0 | $0 | $0 | $0 | | $5,916,722 |

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Prorated Annual Mining Cost | $/tonne (ore) | $26.02 | $17.31 | $17.92 | $16.56 | Mining Cost/ore tonne | 18.20 |
| Prorated Annual Mining Cost (125%) | $/tonne (ore) | $32.52 | $21.64 | $22.41 | $20.70 | | |
| Fractional Ore Tonnes | | 0.10 | 0.32 | 0.33 | 0.26 | | |
| Mining Cost per Period (125%) | $/tonne (ore) | 3.30 | 6.82 | 7.31 | 5.32 | Mining Cost/ore tonne (125%) | $22.76 |

**Cost By Quarterly Period**

| | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Mining OPEX-OB | $/tonne | $268,212 | $86,821 | $2,786 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | | $98,235 |
| Mining OPEX-Waste | $/tonne | $172,391 | $390,009 | $555,837 | $325,900 | $0 | $0 | $0 | $0 | $0 | $0 | | $364,483 |
| Mining OPEX-Ore | $/tonne | $93,682 | $290,933 | $301,337 | $237,106 | $0 | $0 | $0 | $0 | $0 | $0 | | $230,142 |
| Onsite Crushing (6" minus) | $/tonne | $60,036 | $186,443 | $193,111 | $151,948 | $0 | $0 | $0 | $0 | $0 | $0 | | $147,486 |
| Ore Hauling to Mill | $/tonne | $263,894 | $819,531 | $848,838 | $667,905 | $0 | $0 | $0 | $0 | $0 | $0 | | $648,289 |
| Total Mining OPEX Cost | $ | $858,215 | $1,773,737 | $1,901,909 | $1,382,860 | $0 | $0 | $0 | $0 | $0 | $0 | | $5,916,722 |

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Prorated Annual Mining Cost | $/tonne mined | $2.44 | $5.03 | $5.40 | $6.11 | Cost/total material mined | $4.61 |
| Prorated Annual Mining Cost (125%) | $/tonne mined | $3.04 | $6.29 | $6.75 | $7.63 | Cost/total material mined (125%) | $5.76 |
| Total Mining OPEX Cost (125%) | $ | $1,072,769 | $2,217,172 | $2,377,387 | $1,728,575 | Total Mining OPEX Cost (125%) | $7,395,902 |

Project No: **1088890100**

Project Name: **Stratabound CNE Cashflow Analysis**

Client: **Stratabound Minerals**

## Stratabound Minerals Corp.

| ITEM | VALUE | UNITS | TOTAL | YEAR -1 | 1st Quarter | 2nd Quarter | 3rd Quarter | 4th Quarter |
|---|---|---|---|---|---|---|---|---|
| **MINING** | | | | | | | | |
| Ore CNE | | mt | **325,021.0** | - | 32,987 | 102,441 | 106,105 | 83,488 |
| Ore Captain | | mt | **0.0** | - | - | - | - | - |
| Total Ore | | mt | **325,021.0** | - | 32,987 | 102,441 | 106,105 | 83,488 |
| **PROCESSING** | | | | | | | | |
| **CNE Ore Grades** | | | | | | | | |
| Zinc | 4.74 | % | | - | 5.60 | 5.30 | 4.75 | 3.68 |
| Copper | 0.08 | % | | - | 0.06 | 0.07 | 0.06 | 0.14 |
| Lead | 1.76 | % | | - | 2.25 | 2.06 | 1.73 | 1.24 |
| Silver | 58.09 | g/t | | - | 60.12 | 60.31 | 55.32 | 58.07 |
| **Concentrate Grades** | | | | | | | | |
| Zinc | 54.6% | % | | - | 54.6% | 54.6% | 54.6% | 54.6% |
| Copper | 28.0% | % | | - | 28.0% | 28.0% | 28.0% | 28.0% |
| Lead | 50.5% | % | | - | 50.5% | 50.5% | 50.5% | 50.5% |
| **Process Recoveries** | | | | | | | | |
| Zinc | 80% | % | | - | 80% | 80% | 80% | 80% |
| Copper | 94% | % | | - | 94% | 94% | 94% | 94% |
| Lead | 70% | % | | - | 70% | 70% | 70% | 70% |
| Silver | 54% | % | | - | 54% | 54% | 54% | 54% |
| **Concentrates Produced** | | | | | | | | |
| Zinc Concentrate | | mt | 22,551.4 | - | 2,704 | 7,962 | 7,390 | 4,496 |
| Copper Concentrate | | mt | 897.7 | - | 69 | 231 | 200 | 398 |
| Lead Concentrate | | mt | 7,934.4 | - | 1,027 | 2,928 | 2,547 | 1,432 |
| **Total** | | **mt** | **31,383.4** | **-** | **3,801** | **11,120** | **10,137** | **6,325** |
| **Metals Contained in Concentrates** | | | | | | | | |
| Zinc | | **mt** | 12,313.0 | | 1,476.6 | 4,347.1 | 4,034.8 | 2,454.6 |
| Copper | | **mt** | 251.4 | | 19.4 | 64.5 | 56.1 | 111.3 |
| Lead | | **mt** | 4,006.9 | | 518.6 | 1,478.7 | 1,286.4 | 723.2 |
| Silver | | **mt** | 10.2 | | 1.07 | 3.34 | 3.17 | 2.62 |
| **Metal Prices** | | | | | | | | |
| Zinc | $1.22 | US$/lb | | | $ 1.22 | $ 1.22 | $ 1.22 | $ 1.22 |
| Copper | $3.62 | US$/lb | | | $ 3.62 | $ 3.62 | $ 3.62 | $ 3.62 |
| Lead | $1.10 | US$/lb | | | $ 1.10 | $ 1.10 | $ 1.10 | $ 1.10 |
| Silver | $22.74 | US$/oz | | | $ 22.74 | $ 22.74 | $ 22.74 | $ 22.74 |
| **GROSS METAL VALUES IN CONCENTRATES** | | | | | | | | |
| Zinc | | US$ | $33,108,298 | | $ 3,970,339 | $ 11,688,745 | $ 10,848,995 | $ 6,600,218 |
| Copper | | US$ | $2,005,443 | | $ 154,746 | $ 514,999 | $ 447,680 | $ 888,017 |
| Lead | | US$ | $9,714,223 | | $ 1,257,243 | $ 3,584,982 | $ 3,118,775 | $ 1,753,223 |
| Silver | | US$ | $7,453,567 | | $ 782,925 | $ 2,439,275 | $ 2,317,184 | $ 1,914,183 |
| Gold | | US$ | $0 | | | | | |
| **Total** | | US$ | **$52,281,531** | | **$ 6,165,254** | **$ 18,228,002** | **$ 16,732,634** | **$ 11,155,641** |
| **NET SMELTER RETURN (NSR)** | | | | | | | | |
| Zinc NSR | $ 965.00 | US$/dmt con. shipped | $21,762,065 | | $ 2,609,702 | $ 7,683,005 | $ 7,131,038 | $ 4,338,320 |
| Copper NSR | $2,030.00 | US$/dmt con. shipped | $1,822,335 | | $ 140,617 | $ 467,977 | $ 406,805 | $ 806,936 |
| Lead NSR | $ 1,763.00 | US$/dmt con. shipped | $13,988,293 | | $ 1,810,406 | $ 5,162,305 | $ 4,490,976 | $ 2,524,607 |
| **Total Net Smelter Return US$** | | | **$37,572,693** | | **$ 4,560,725** | **$ 13,313,287** | **$ 12,028,818** | **$ 7,669,862** |
| Exchange Rate US$/CAN$ | 1.012 | CAN$ | | | 1.012 | 1.012 | 1.012 | 1.012 |
| **Total Net Smelter Return** | | CAN$ | **$38,023,565** | | **$ 4,615,453** | **$ 13,473,047** | **$ 12,173,164** | **$ 7,761,901** |
| **Total Net Smelter Return** | | CAN$/dmt | **$ 1,212** | | **$ 1,214** | **$ 1,212** | **$ 1,201** | **$ 1,227** |
| Allowance Marketing & Insurance | 0.05 | % | $19,012 | | $ 2,308 | $ 6,737 | $ 6,087 | $ 3,881 |
| **NET MINE RETURN** | | | **$38,004,553** | | **$ 4,613,146** | **$ 13,466,310** | **$ 12,167,078** | **$ 7,758,020** |
| **Net Mine Return $/t** | | CAN$/dmt | **$ 1,211** | | **$ 1,214** | **$ 1,211** | **$ 1,200** | **$ 1,227** |
| Royalty estimate | 1.0 | % | $380,045.53 | | $ 46,131 | $ 134,663 | $ 121,671 | $ 77,580 |
| **NET REVENUE(CAN $)** | | $ | **$37,624,508** | | **$ 4,567,014** | **$ 13,331,647** | **$ 12,045,407** | **$ 7,680,440** |
| **OPERATING COSTS (per t ore milled) per Qtr** | | | | | | | | |
| 100% Mining | $0.00 | CAN$/t | $7,395,902 | | $ 1,072,769 | $ 2,217,172 | $ 2,377,387 | $ 1,728,575 |
| 100% Processing | $28.53 | CAN$/t | $9,272,167 | | $ 941,042 | $ 2,922,438 | $ 3,026,946 | $ 2,381,741 |
| 100% G & A | $0.30 | CAN$/t | $97,506 | | $ 9,896 | $ 30,732 | $ 31,831 | $ 25,046 |
| 100% Ore Haulage from Mine to Mill | $0.00 | CAN$/t | $0 | | $ - | $ - | $ - | $ - |
| **SubTotal Operating Cost** | **$28.83** | CAN$/t | **$16,765,576** | | **$ 2,023,707** | **$ 5,170,342** | **$ 5,436,164** | **$ 4,135,362** |
| | | | $20,858,932.21 | | | | | |
| **CAPITAL COSTS** | | | | | | | | |
| **Direct Costs** | | | | | | | | |
| 100% Site Development CNE | | CAN$ | $1,442,749 | $1,442,749 | $0 | $0 | $0 | $0 |
| 100% Utilities | | CAN$ | $2,084,900 | $2,084,900 | $0 | $0 | $0 | $0 |
| 100% Mobile Equipment | | CAN$ | $125,000 | $125,000 | $0 | $0 | $0 | $0 |
| 100% Infrastructure | | CAN$ | $200,000 | $200,000 | $0 | $0 | $0 | $0 |
| 100% Water Treatment Plant | | CAN$ | $200,000 | $200,000 | $0 | $0 | $0 | $0 |
| 100% Closure / Reclamation Costs | | CAN$ | $2,000,000 | $0 | $0 | | $0 | $2,000,000 |
| **SubTotal Direct Costs** | | CAN$ | **$6,052,649** | **$ 4,052,649** | **$ -** | **$ -** | **$ -** | **$ 2,000,000** |
| **Indirect Costs** | | | | | | | | |
| 100% Indirect Costs | 1% | CAN$ | $60,526 | $40,526.49 | $ - | $ - | $ - | $ 20,000 |
| 100% Owners Costs | 2% | CAN$ | $121,053 | $81,052.98 | $0.00 | $0.00 | $0.00 | $40,000 |
| 100% Contingency | 15% | CAN$ | $907,897 | $607,897.33 | $ - | $ - | $ - | $ 300,000 |
| 100% Salvage of / Plant / Site / Working capital | 10% | CAN$ | -$260,990 | $0.00 | $ - | $ - | $ - | $ (260,990) |
| **SubTotal Indirect Costs** | | CAN$ | **$828,487** | **$ 729,477** | **$ -** | **$ -** | **$ -** | **$ 99,010** |
| **Total Capital Cost** | | CAN$ | **$6,881,136** | **$ 4,782,126** | **$ -** | **$ -** | **$ -** | **$ 2,099,010** |
| **PRE-TAX CASH FLOW** | | | | | | | | |
| Net Revenue | | CAN$ | **$37,624,508** | **$ -** | **$ 4,567,014** | **$ 13,331,647** | **$ 12,045,407** | **$ 7,680,440** |
| Operating Cost | | CAN$ | **$16,765,576** | **$ -** | **$ 2,023,707** | **$ 5,170,342** | **$ 5,436,164** | **$ 4,135,362** |
| Capital Costs | | CAN$ | **$6,881,136** | **$ 4,782,126** | **$ -** | **$ -** | **$ -** | **$ 2,099,010** |
| **TOTAL PRE-TAX CASH FLOW** | | CAN$ | **$13,977,797** | **(4,782,126)** | **$2,543,307** | **$8,161,306** | **$6,609,243** | **1,446,067** |
| **ACCUMULATED CASH FLOW** | | CAN$ | | **(4,782,126)** | **(2,238,819)** | **5,922,487** | **12,531,729** | **13,977,797** |
| **PAYBACK** | | | | | **-** | **2nd Quarter** | **-** | **-** |

| | |
|---|---|
| **PRE-TAX & PRE FINANCE NPV @ 6%** | **$13,131,483** |
| **PRE-TAX & PRE FINANCE NPV @ 8%** | **$12,862,435** |
| **PRE-TAX & PRE FINANCE NPV @ 10%** | **$12,599,611** |
| **PRE-TAX & PRE FINANCE NPV @ 12%** | **$12,342,838** |
| **PRE-TAX & PRE FINANCE NPV @ 15%** | **$11,968,655** |
| **PRE-TAX & PRE FINANCE NPV @ 20%** | **$11,372,963** |

| | |
|---|---|
| **PROJECT INTERNAL RATE OF RETURN (IRR)** | **292.29%** |

Project No: **1088890100**

Project Name: **Stratabound CNE Cashflow Analysis**
Client: **Stratabound Minerals**

| Category Number | Area Number | Package Number | Description | Unit | Quantity | Unit cost | Total (CAD) | |
|---|---|---|---|---|---|---|---|---|
| **A** | | | **Site Development CNE & Captain** | | | | **$ 2,885,498** | |
| | 100 | | *Electrical Grid Power Connection* | | | | | |
| | *200* | | *Early Site Development* | | | | | |
| | | 10 | Clearing, Grubbing and Removals | ha | 3.5 | $10,000 | $ 35,000 | $17,500 |
| | 250 | | *Plant Site Development* | | | | | |
| | | 10 | Excavation/ fill and granular surface | ha | 3.50 | $190,000.00 | $ 665,000 | |
| | | 20 | Fencing and gate | m | 1800.0 | $150 | $ 270,000 | |
| | | | | | | | $ 935,000 | $467,500 |
| | *400* | | *Site Roads & Hard Standing Areas* | | | | | |
| | | *10* | Haul Roads c/w dykes - 30m (coarse/fine rock, | m | 2500.0 | $529 | $ 1,322,115 | |
| | | 20 | Access Roads c/w dykes - 8m (coarse/fine rock | m | 750.0 | $250 | $ 187,500 | |
| | | *30* | Hard Areas - crusher, stockpile, office | m2 | 25,000 | $8.24 | $ 205,882 | |
| | | | | | | | $ 1,715,498 | $857,749 |
| | *500* | | *Rock Material Dumps* | | | | | |
| | | 10 | Overburden Dump/Ditching | lump sum | 1 | $50,000 | $ 50,000 | |
| | | 20 | Waste Rock Dump/Ditching | lump sum | 1 | $50,000 | $ 50,000 | |
| | | 30 | Waste Rock Seepage Collection Ditch/Pond | lump sum | 1.0 | $100,000 | $ 100,000 | |
| | | | | | | | $ 200,000 | $100,000 |
| **B** | | | **Site Utilities CNE & Captain** | | | | **$ 4,169,800** | **$ 2,084,900** |
| | *100* | | *230 kV/13.8 kV Sub-Station and Switchgear* | | | | | |
| | *200* | | *Electrical Distribution* | | | | | |
| | | | | m | 3000.0 | $300 | $ 900,000 | $450,000 |
| | *300* | | *Power* | | | | | |
| | | 10 | Diesel Gen Sets (1500KW) | lot | 1.0 | $1,250,000 | $ 1,250,000 | |
| | | 20 | Power Distribution & Transfer Switches | lot | 1.0 | $250,000 | $ 250,000 | |
| | | | | | | | $ 1,500,000 | $750,000 |
| | *400* | | *Water Systems* | | | | | |
| | | 10 | Water balance piping system c/w heat tracing/t | m | 500.0 | $500 | $ 250,000 | |
| | | 20 | Potable Water Systems | m | 400.0 | $350 | $ 140,000 | |
| | | 30 | Mine dewatering pipe/pump System | m | 1200.0 | $350 | $ 420,000 | |
| | | 40 | Fire Protection Distribution c/w heat tracing/pur | m | 400.0 | $262 | $ 104,800 | |
| | | | | | | | $ 914,800 | $457,400 |
| | *500* | | *Sewer System Improvements* | | | | | |
| | | 10 | Septic Systems - Remote | lot | 1.0 | $125,000 | $ 125,000 | |
| | | 20 | Collection Piping, pumps and manholes | m | 0.0 | $163 | $ - | |
| | | | | | | | $ 125,000 | $62,500 |
| | *600* | | *Site Fuelling System* | | | | | |
| | | 10 | Civil foundation | lot | 1.0 | $100,000 | $ 100,000 | |
| | | 20 | Dispensing Equipment | lot | 1.0 | $80,000 | $ 80,000 | |
| | | 30 | Mechanical Piping | lot | 1.0 | $150,000 | $ 150,000 | |
| | | 40 | Storage Tank | lot | 2.0 | $200,000 | $ 400,000 | |
| | | | | | | | $ 730,000 | $365,000 |
| **C** | | | **Water Treatment** | | | | **$ 200,000** | |
| | *100* | | *Tailings Containment* | | | | | |
| | | 10 | Tailings Cut Off Trench - Excavation | m3 | 0 | $0 | $ - | |
| | | 20 | Tailings Cut Off Trench - Fill | m3 | 0 | $0 | $ - | |
| | | 30 | Tailings Dam | m3 | 0 | $0 | $ - | |
| | | 40 | Emergency Spillway | lump sum | 0.0 | $0 | $ - | |
| | | 50 | Tailings Seepage Collection Ditch/Pond | lump sum | 0.0 | $0 | $ - | |
| | | 60 | Relocate Tailings Pond 2 Discharge Channel | lump sum | 0.0 | $0 | $ - | |
| | | | | | | | $ - | |
| | *200* | | *Tailings Pumping* | | | | | |
| | | 10 | Reclaim Barge c/w pumps walkway and access | unit | 0.0 | $0 | $ - | |
| | | 20 | Tailings Piping pumps and accessories | m | 0.0 | $0 | $ - | |
| | | 30 | Reclaim Water Pipeline | m | 0.0 | $0 | $ - | |
| | | | | | | | $ - | |
| | *300* | | *Tailings Water Treatment Systems* | | | | | |
| | | 10 | Effluent Treatment System | unit | 1.0 | $200,000 | $ 200,000 | |
| | | | | | | | $ 200,000 | |
| **D** | | | **Mobile Equipment** | | | | **$ 125,000** | |
| | *100* | | *General Trucks and Vehicles* | | | | | |
| | | 10 | Water/Sanding/Plowing Truck | unit | 1.0 | $250,000 | $ 125,000 | |
| | | 20 | Lowboy, Tractor | unit | 0.0 | $0 | $ - | |
| | | 30 | Lube/Fuel, Mechanics, Welding Tool Carrier Tr | unit | 0.0 | $0 | $ - | |
| | | 40 | Light Plants, Crew back Pick-ups, Pick-ups & B | unit | 0.0 | $0 | $ - | |
| | | 50 | Forklifts | unit | 0.0 | $0 | $ - | |
| | | 60 | Man Buses | unit | 0.0 | $0 | $ - | |
| | | | | | | | $ 125,000 | |
| | *200* | | *Support Equipment* | | | | | |
| | | 10 | Shovels | | | | $ - | |
| | | 20 | Drills | | | | $ - | |
| | | | | | | | $ - | |
| | *300* | | *Emergency Vehicles* | | | | | |
| | | 10 | Shovels | | | | $ - | |
| | | 20 | Drills | | | | $ - | |
| | | | | | | | $ - | |
| | *400* | | *Open Pit Mining Equipment* | | | | | |
| | | 10 | Shovels | | | | $ - | |
| | | 20 | Drills | | | | $ - | |
| | | 30 | Blasting Skid Steer Loaders | | | | $ - | |
| | | 40 | Tracked Dozer, Rubber Tired Dozer & Grader | | | | $ - | |
| | | 50 | Large Front End Loader | | | | $ - | |

| | | | | | | |
|---|---|---|---|---|---|---|
| | 60 | Haulage Trucks | | | | $ - |
| | 70 | Backhoes | | | | $ - |
| | | | | | | $ - |
| **E** | | **Ore Processing Facilities** | | | | |
| **F** | | **Infrastructure** | | | | **$ 200,000** |
| *100* | | *Miscellaneous Facilities* | | | | |
| | 10 | Truck Weigh Scale | unit | 0.0 | $0 | $ - |
| | 20 | Unheated Warehouse | unit | 0.0 | $0 | $ - |
| | | | | | | $ - |
| *200* | | *General Maintenance Building* | | | | |
| | 10 | Building c/w roll up doors/foundation | unit | 0.0 | $0 | $ - |
| | 20 | Truck Repair Equipment & Crane | unit | 0.0 | $0 | $ - |
| | | | | | | $ - |
| *300* | | *Modular Facilities* | | | | |
| | 10 | Guard House | lot | 1.0 | $20,000 | $ 20,000 |
| | 20 | Dry/Shift Change Building c/w washroom and laur | lot | 0.0 | $0 | $ - |
| | 30 | Kitchen & Cafeteria | lot | 0.0 | $0 | $ - |
| | 40 | Mine Office Complex | lot | 1.0 | $180,000 | $ 180,000 |
| | | | | | | $ 200,000 |

***Overhead Costs***

| | | | |
|---|---|---|---|
| | | **Indirect Costs** | |
| *X1000* | | *Construction Facilites* | |
| | X1100 | Temporary Power & Utilites | $ - |
| | X1200 | Temporary Construction Equipment | $ - |
| | X1300 | Site Construction Communications | $ - |
| | X1400 | Storage Facilities | $ - |
| | X1500 | Site Maintenance During Construction | $ - |
| | X1600 | Construction Fuel | |
| | | | $ - |
| *X2000* | | *Commissioning and Startup* | |
| | X3100 | Owner's Commissioning Team | $ - |
| | X3200 | Third Party Consultants | $ - |
| | X3300 | Vendor Representatives | $ - |
| | | | $ - |
| *X3000* | | *Spares and First Fill* | |
| | X4100 | Spare Parts | $ - |
| | X4200 | First Fill | $ - |
| | | | $ - |
| *X5000* | | *Professional Fees* | |
| | X5100 | Engineering, Procurement & Construction Management | $ - |
| | | | $ - |
| | | **Owner's Costs** | |
| *Y1000* | | *Site* | |
| | Y1100 | Computing | |
| | Y1200 | Project Photographs | |
| | Y1300 | Safety Equipment | |
| | Y1400 | Surveying | |
| *Y2000* | | *Miscellaneous* | |
| | Y2100 | Taxes and Duties | |
| | Y2200 | Project Funding or Financing Costs | |
| | Y2300 | Builders Risk Insurance | |
| | Y2400 | General Liability Insurance | |
| | Y2500 | Project Legal Costs | |
| | Y2600 | Environmental Monitoring | |
| | Y2700 | Goodwill and Local infrastructure Contributions | |
| *Y3000* | | *Systems and Training* | |
| | Y3100 | Manufacturing Executing System (MES) | |
| | Y3200 | IT Systems/Business System | |
| | Y3300 | Mining Modelling Software | |
| | Y3400 | Process Control Software | |
| | Y3500 | Systems Training - MES, PLC | |
| | Y3600 | OEM Training Costs | |
| *Y4000* | | *Employee Staffing* | |
| | Y4100 | Employee Training | |
| | Y4200 | Staff Relocation Costs | |
| | Y4300 | Accomodation and Living Expenses | |
| | Y4400 | Recruiting Costs | |
| *Y5000* | | *Approvals* | |
| | Y5100 | Technical Advisory Committee - Government | |
| | Y5200 | Tailings Storage Facility Authorization | |
| | Y5300 | De-Watering Permit | |
| | Y5400 | Water Rights License | |
| *Y6000* | | *Permitting and Licensing* | |
| | Y6100 | Environmental Permit | |
| | Y6200 | Quarry Permit | |
| | Y6300 | Mineral Lease for Mining Permits | |
| | Y6400 | Groundwater Exploration Permits | |
| | Y6500 | Manitoba Conservation Work Permit | |
| | Y6600 | Building Permits | |
| | Y6700 | Licenses | |
| *Y7000* | | *Closure Costs* | |
| | Y7100 | Closure Plant Permit | |
| | Y7200 | Closure costs | |