Technical Report on the Robinson Nevada Mining Company Operation

Ruth, Nevada
USA

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Prepared by:

Quadra Mining Ltd.

Table of Contents

Table of Contents

LIST OF FIGURES

LIST OF TABLES

1. Summary

1. Summary

The Robinson operation is an open pit copper and gold mine located in eastern Nevada approximately 7 miles (11 km) west of the town of Ely. The property is a mature mine site that has been actively mined from the late 1800's to 1978, from 1986 to 1999, and again from 2004 to the present. Modern milling and sulfide concentrating facilities were constructed by Magma Copper Company (Magma) and its successor, BHP Copper Inc. (BHP), and operated from 1996 to 1999. BHP discontinued mining at Robinson in mid-1999, and the property was placed under a Care and Maintenance program for economic reasons. The property was then purchased by Quadra Mining Ltd. (Quadra) in 2004 and mining and processing operations were re-initiated in the same year. The Robinson Nevada Mining Company (RNMC), a wholly owned subsidiary of Quadra, has been operating the property continually since 2004.

In general, the Robinson deposits are characterized as porphyry copper ± molybdenum ± gold systems that are associated with monzonitic rocks of Cretaceous age. Copper mineralization with by-product molybdenum ± gold is hosted in porphyry and in skarn that formed in calcareous rocks adjacent to mineralized porphyry. The principal hypogene sulfide minerals in the Robinson deposits are pyrite and chalcopyrite that occur as both dissemination and veinlets with quartz. Supergene enrichment resulted in chalcocite blankets up to 100 m thick. Weathering has remobilized the chalcocite which has resulted in considerable portions of the deposits containing a broad distribution of weak chalcocite mineralization that tends to mantle pyrite and chalcopyrite.

Not all gold is in direct association with the copper mineralization. Primary gold deposits are hosted by various calcareous sedimentary rocks and are generally located around the periphery of the copper deposits. Nevertheless, gold does occur as inclusions and fills fractures in the chalcopyrite grains. Gold also occurs in the 'leach cap' above the copper deposits and as free gold, often randomly attached to sulfides or silicates.

The post-mineralization structural history of the Robinson District is very complex; the district is situated along a Tertiary extensional zone. Geologic investigations over the years have identified at least seven major structural sets within the district itself, all of which appear to be normal faults with minor oblique-slip, each of which offset and rotated the previous set of faults.

Since purchasing the property from BHP, Quadra has been engaged in a 2.5 year exploration program that included both new drilling and re-assaying of existing core and pulps from historic drilling. As a result of this exploration program, RNMC updated the geologic model for the Tripp-Veteran and Ruth areas. Resources and reserves were re-estimated by Quadra and RNMC personnel using this updated geologic model. These resources and reserves are consistent with Canadian Institute of Mining, Metallurgy, and Petroleum (CIM) Standards on Mineral Resources and Reserves Definitions and Guidelines adopted August 20, 2000 by the Canadian Institute of Mining, Metallurgy and Petroleum and modified with adoption of the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" in 2005, and in accordance with the standards set out in NI 43-101. Resources are summarized in Table 1.1 and reserves in Table 1.2.

Table 1.1 Robinson Operation Mineral Resources

Total Robinson Measured plus Indicated Resource

Cutoff Cu%	Tons above cut x1000	Avg Grade Cu %	Contained lbs Cu x1000	Avg Grade Au oz/ton	Contained oz Au x1000
0.10	1,763,300	0.28	9,779,720	0.004	7,238
0.20	852,500	0.43	7,252,900	0.005	4,476
0.30	477,900	0.57	5,430,760	0.006	2,726
0.40	300,400	0.69	4,164,980	0.006	1,901
0.50	207,100	0.81	3,343,160	0.006	1,309
0.60	145,000	0.92	2,678,860	0.006	916
0.70	104,400	1.03	2,143,380	0.007	743
0.80	73,200	1.14	1,674,720	0.007	505
0.90	50,400	1.29	1,297,720	0.007	348
1.00	35,800	1.42	1,016,820	0.007	248

Table 1.2 Robinson Operation Mineral Reserve Estimate

Total Robinson

Reserve Classification	Ore Tons (000)	Cu Grade (%)	Au Grade (opt)	Contained Metal		Waste Tons (000)	Total Tons (000)	Strip Ratio
				Cu Tons (000)	Au oz (000)			
Proven	130,045	0.55%	0.007	711	884			
Probable	4,097	0.42%	0.005	17	21			
Proven and Probable	134,142	0.54%	0.007	728	905	413,200	547,342	3.08

The underlying data consists of over 10,000 drill holes that have been recorded in the district by numerous exploration campaigns conducted during the span of 100 years. Currently, 9,651 of these drill holes are included in the current District Central Drill Hole Database, (DCDHD). Historic drill-hole data was augmented with 335 additional RC, Becker and core drill holes completed by Quadra between 2006 and 2008. In addition, sample pulps from 1,047 of the available historic drill holes located within the active portions of the Tripp–Veteran Deposit and Ruth Deposit were selected by Quadra for complete modern-day re-analysis. The modern Quadra drilling and historic drill hole sample re-assay programs were subject to quality checks and review as described later in this document.

The underlying geologic model is complicated, reflecting the geologic history of the district and the need to incorporate milling parameters and other economic considerations into the model.

Quadra believes that, globally, the model is a good predictor of contained metal content, though the complicated geology could result in local variations. Reconciliation of model grades, mill grades, and blast-hole grades has proven this to be true in the Tripp-Veteran area.

Pit and waste stockpile designs were developed by RNMC personnel. Budget level estimates for operating and processing costs and metals recoveries used in the economic calculations were based on reported actual values for the most recent operating period, 2004 to 2008. These estimates assume truck haulage of concentrates to the railroad near East Wendover, Utah, rail haulage to Vancouver, Washington and ocean transportation to an overseas smelter. Since 2004, the majority of the concentrates have been sold to Pacific Rim smelters, but there have also been some sales to domestic US smelters.

As of the date of this report, mining is actively underway in the Tripp-Veteran areas, and the Tripp-Veteran deposits have been continuously mined since Quadra started operations in 2004. Nominal Mill capacity is approximately 45,000 tons per day, and maximum mining rate is expected to be approximately 255,000 tons per day over the life of mine.

Robinson has all of the regulatory permits in place to operate the mine and no new permits are required, unless there is a "substantial change" to what has already been approved by the agencies. Robinson has appropriate legal title to the land on which mining is taking place, and a list of property, patented, and unpatented claims controlled by the Robinson Operation can be obtained by contacting the Quadra corporate office in Vancouver, BC, Canada.

2. Introduction and Terms of Reference

2. Introduction and Terms of Reference

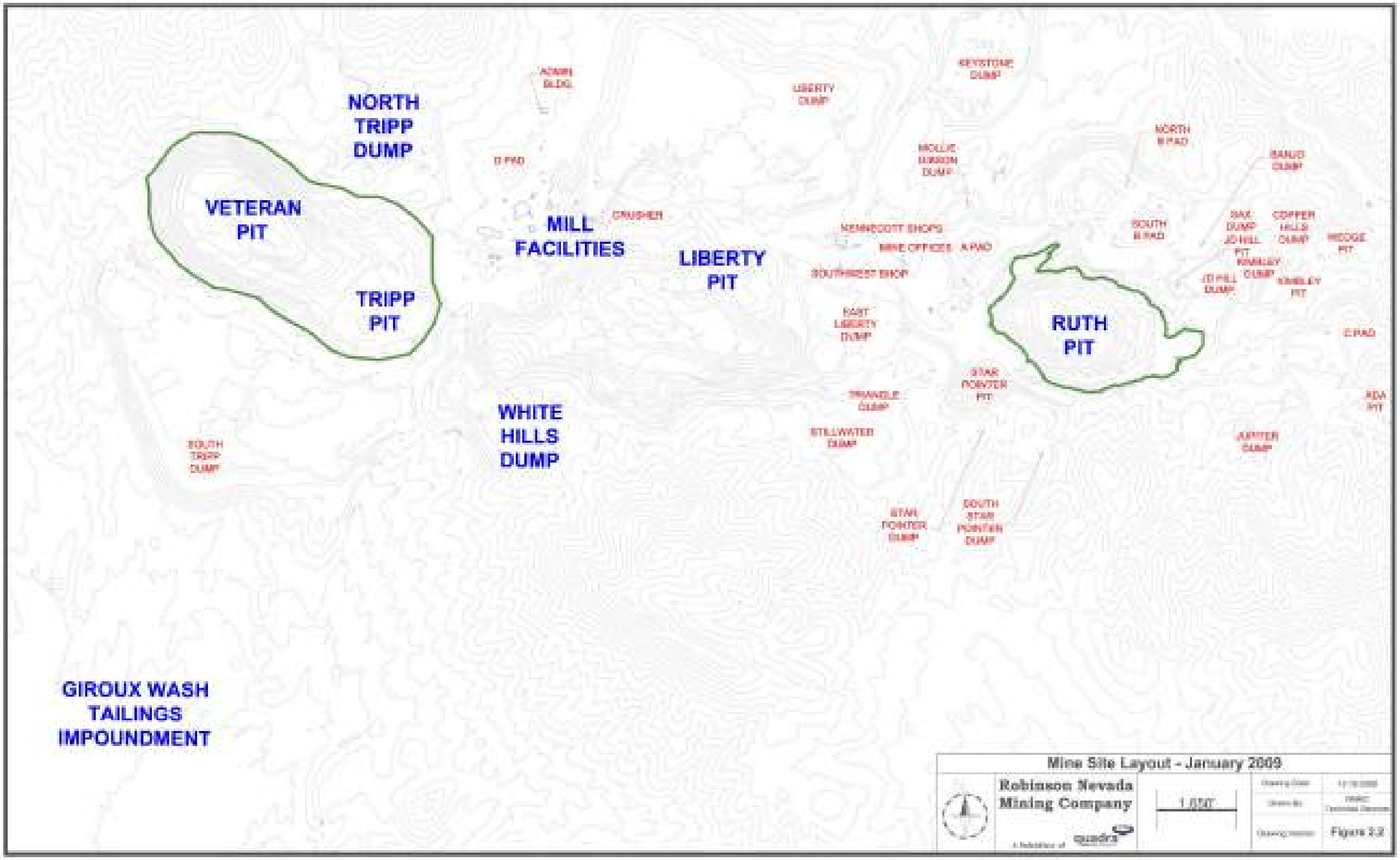
This document is a technical report on the Robinson mining operation, located in eastern Nevada, by Quadra Mining Ltd. (Quadra). Quadra has performed additional drilling and interpretation of in-place geology, mineralogy and metallurgical recovery data to prepare this independent technical report on the Robinson operation in accordance with National Instrument 43-101, *Standards of Disclosure for Mineral Projects*.

Quadra purchased 100% of the equity interests in each of BHP Nevada Mining Company (BNMC) and BHP Nevada Railroad Company (BNRC) which form, together with Robinson Holdings USA (RHUSA), the “Robinson Interests”. The operator at Robinson is Robinson Nevada Mining Company (RNMC), a wholly owned subsidiary of Quadra.

The Robinson operation is located in White Pine County, Nevada approximately 11 km west of the town of Ely (Figure 2.1). The property is a mature mine site that has been actively mined from the late 1800’s to 1978, from 1996 to 1999, and again from 2004 to the present. Mining at the site has been by both underground and surface methods with recent mining from the 1940’s onward by open pit methods. The site contains three major open pit areas: Tripp-Veteran, Liberty and Ruth pits (Figure 2.2). These pits occur within an area measuring approximately 14 km east to west and 8 km north to south. Modern milling and sulfide concentrating facilities, which operated from 1996 to 1999 and 2004 to the present, are situated at the site.

Except for historic resources and reserves, this report refers to the copper and gold resources and reserves at the Tripp-Veteran and Ruth Pits. As used in this report, the Ruth Pits refers to specific mining areas associated with the Ruth West, Ruth East, Kimbley and Wedge areas of the property.

Figure: 2.1 Project Location Map


NORTH TRIPP DUMP
VETERAN PIT
TRIPP PIT
MILL FACILITIES
LIBERTY PIT
RUTH PIT
WHITE HILLS DUMP
GIROUX WASH TAILINGS IMPOUNDMENT
ADMIN BLDG.
D PAD
CRUSHER
LIBERTY DUMP
KEYSTONE DUMP
MOLLIE GIBSON DUMP
KENNECOTT SHOPS
MINE OFFICES
A PAD
SOUTHWEST SHOP
EAST LIBERTY DUMP
NORTH B PAD
SOUTH B PAD
BANJO DUMP
COPPER HILLS DUMP
WEDGE PIT
KIMBLEY PIT
C PAD
ADA PIT
JUPITER DUMP
STAR POINTER PIT
TRIANGLE DUMP
STILLWATER DUMP
STAR POINTER DUMP
SOUTH STAR POINTER DUMP
SOUTH TRIPP DUMP
Mine Site Layout - January 2009
Robinson Nevada Mining Company

2.1 Definitions

Acronyms and abbreviations commonly used in this report are presented in the following section. Both imperial and metric units are used in this report depending on the source of the data referenced. The units used are identified in the text of this report but this may not apply to secondary documents and/or references quoted in this report. Therefore caution should be exercised in reviewing numerical values and corresponding units reported in this document since in some cases there are different units used within the same sections and tables.

2.1.1 Frequently Used Acronyms and Abbreviations

AA	atomic absorption spectrometry
Ag	silver
Au	gold
BCI	BHP Copper Inc.
BE	Bucyrus International Inc.
BHP	BHP Billiton Group (also may reference BCI)
BLM	United States Department of the Interior, Bureau of Land Management
BNMC	BHP Nevada Mining Company
BNRC	BHP Nevada Railroad Company
BOC	Barge Operating Channel
BWI	Bond Work Index
CAT	Caterpillar Inc.
CEB	Chalcocite Enrichment Blanket
cm	centimeter
Cu	copper
DCDHD	District Central Drill Hole Database
Dmt	dry metric tonnes
dst	dry short tons
expit	Material mined from inside a pit and transported to a processing plant or rock storage facility outside the pit, as opposed to material hauled from stockpiles or material hauled solely within the pit.
FA	Fire Assay
ft	feet
ft^3	cubic feet
$Fe_2(SO_4)_3$	ferrous sulfate
G&A	General and Administrative
g	grams
gpm	gallons per minute
g/t	grams per ton
ha	hectare
HCl	hydrochloric acid
$HClO_4$	perchloric acid

HF	hydrofluoric acid
HP	horsepower
HNO_3	nitric acid
H_2SO_4	sulfuric acid
ICP-AES	Inductively Coupled Plasma Atomic Emission Spectroscopy
ICP-ES	Inductively Coupled Plasma Emission Spectroscopy
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
ID^2	Inverse Distance Squared
in.	inches
JCR	Joint Condition Rating
JRC	Joint Roughness Coefficient
km	kilometer
kWh/ton	kilowatt hours per ton
lb	pound (2,000 lbs to 1 ton, 2,204.6 lbs to 1 tonne)
LOM	Life-of-Mine
m	meters
M	millions
Ma	millions of years ago
MDA	Mine Development Associates
mph	miles per hour
Magma	Magma Copper Company
Mo	molybdenum
NAG	Non-acid Generating
NPV	Net Present Value
NSM	Net Smelter Return
Opt	Troy ounces (12 oz to 1 pound) per ton
oz	troy ounce (12 oz to 1 pound)
Quadra	Quadra Mining Ltd.
QA/QC	Quality Assurance/Quality Control
QLT	Quick Leach Test, a method of assaying for acid soluble copper
PAG	Potentially Acid Generating
pdf	portable document format
ppb	parts per billion
ppm	parts per million
RC	reverse circulation drilling method
RQD	Rock Quality Designation
RNMC	Robinson Nevada Mining Company
RHUSA	Robinson Holdings USA
Robinson	Robinson Nevada Mining Company
SAG	Semi-autogenous Grinding
st	short (imperial) ton
stph	(short) tons per hour
ton	short (imperial) ton
tonne	metric ton
TD	total depth
tpd	(short) tons per day

tph	(short) tons per hour
US$	United States 2009 dollars
USGS	United States Geologic Survey
µm	micron
VBM	variable block model, Medsystem data file

3. Reliance on Other Experts

3. Reliance on Other Experts

This Technical Report is intended to be read as a whole, and individual chapters or sections should not be read or relied upon out of context. The Technical Report contains the expression of the professional opinions of Quadra, its employees, and consultants, and is based upon information available at the time of preparation. The quality of the information, conclusions and estimates contained herein are consistent with the intended level of accuracy as set out in this report, as well as the circumstances and constraints under which the report was prepared which are also set out herein.

Quadra (and the Qualified Persons for the purposes of this Technical Report) have relied on a number of other reports and statements made by various sources and the information, conclusions and recommendations contained herein are based on such reports and statements, including:

1. Data and information supplied to Quadra by BHP. BHP provided Quadra with an inventory of the available documentation for the property. Several of these reports and other documents were prepared by mining consulting firms on behalf of BHP and previous operators of the property. RNMC has used a number of these references in the preparation of this report. These sources are listed in Chapter 21 of this report and cited in the text.
2. Cornerstone Lands, of Tucson, Arizona, USA, Deconcini, McDonald, Yetwin & Lacy LLP, a law firm based in Tucson, Arizona, USA, and Gorsuch, Kirgis, Campbell, Walker and Grover, a law firm based in Denver, Colorado, USA provided opinions and information concerning corporate ownership and land tenure as described in Chapter 4..
3. Environmental and permitting review, opinions, and information were provided by Pat Gochnour, Gochnour & Associates, Inc., Parker CO, USA.
4. Information and data analyses regarding drill-hole data discussed in Chapter 13 Sample Preparation, Analysis and Security was provided by Dr. Jeffrey Jaacks, Geochemical Applications Intl. Inc., Centennial, CO, USA. Dr. Mark Osterberg, of Mine Mappers Ltd., Tucson, AZ, USA provided geologic modeling services and interpretations described in Chapter - 9, Mineralization, Chapter 10 - Exploration, Chapter 11 - Drilling, Chapter 12 - Sampling Methods and Approach, and Chapter 17 - Mineral Resource and Mineral Reserve Estimates.

4. Property Description and Location

4. Property Description and Location

The Robinson Mine site is located in White Pine County, Nevada approximately 11 km west of Ely (Figure 4.1), in the central Egan Range at an average elevation of 2,130 m. The Robinson site contains three major open pit areas: Tripp-Veteran, Liberty and Ruth pits. These pits occur within an area measuring approximately 14 km east to west and 8 km north to south. The existing ore processing facilities (sulfide copper flotation mill) are located between the Tripp-Veteran and Liberty pit areas (Figure 4.2). The tailings dam is located to the south of the Tripp-Veteran Pit.

The property has been actively mined for over 100 years and has the remnant dumps, structures, pits and other signs of long-term mining. Most recently the property was operated by BNMC from 1996 to 1999 and by Quadra from 2004 to the present.

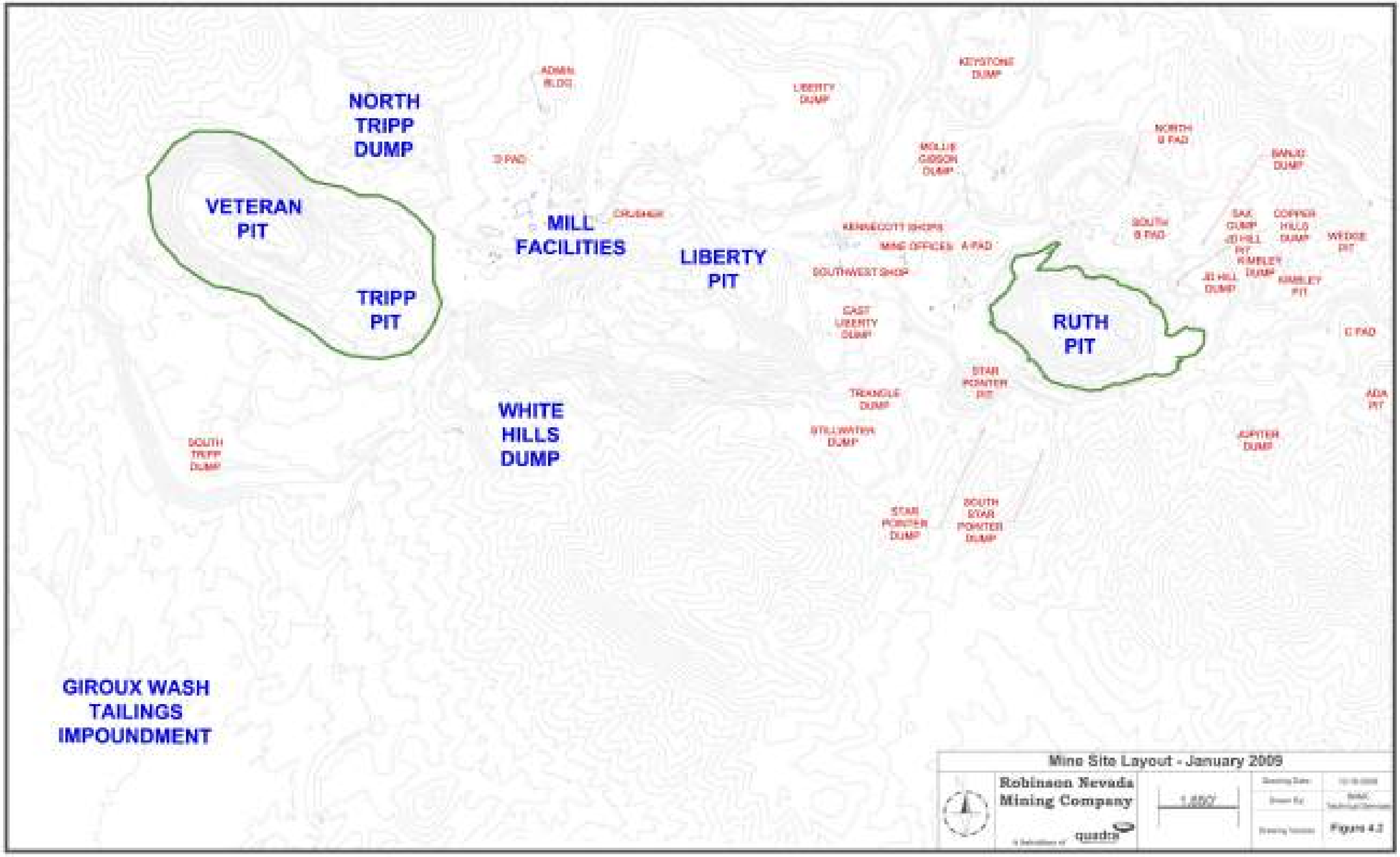
In 2008, Robinson acquired 241 acres (97.5ha) of private land (patented mining claims) on the South side of the Ruth pit area for use as a waste stockpile site for future Ruth pit mining. In 2007, Robinson acquired 1,708 acres (691.2 ha) from the patent of mill site claims at the tailing dam. In 2006, Robinson acquired 165 acres (66.7 ha) of private land (patented mining claims) on the East side of the property from Ely Gold. These acquisitions have consolidated and expanded RNMC's land position; a list of land controlled by RNMC is provided in Table 4.1.

Table 4.1:
Land Holdings

Number	Classification	Acres (approx.)
	Patented Lode & Mill Sites	9,458
664	Unpatented Lode Claims	9,417
596	Unpatented Mill Sites	2,690
	Private Lands	911
	TOTAL	22,476

Title opinions concerning the Robinson property were provided in 1991 (patented claims) and 1992 (unpatented claims) by Gorsuch, Kirgis, Campbell, Walker and Grover, a law firm based in Denver, Colorado. In 1999 and again in 2001, Gorsuch, Kirgis, Campbell, Walker and Grover prepared reports updating material changes from the 1991 report.

Figure: 4.1 Location Map


NORTH TRIPP DUMP
VETERAN PIT
TRIPP PIT
MILL FACILITIES
LIBERTY PIT
WHITE HILLS DUMP
RUTH PIT
GIROUX WASH TAILINGS IMPOUNDMENT
ADMIN BLDG
LIBERTY DUMP
KEYSTONE DUMP
MOLLIE GIBSON DUMP
NORTH B PAD
BANJO DUMP
CRUSHER
KENNECOTT SHOPS
MINE OFFICES
A-PAD
SOUTHWEST SHOP
SOUTH B PAD
COPPER HILLS DUMP
WEDGE PIT
KIMBLEY PIT
EAST LIBERTY DUMP
C PAD
STAR POINTER PIT
TRIANGLE DUMP
STILLWATER DUMP
ADA PIT
JUPITER DUMP
SOUTH TRIPP DUMP
STAR POINTER DUMP
SOUTH STAR POINTER DUMP
Mine Site Layout - January 2009
Robinson Nevada Mining Company
quadra
Figure 4.2

On August 6, 2003, John Lacy of Deconcini, McDonald, Yetwin & Lacy LLP, a law firm based in Tucson, Arizona issued a report entitled "BHP Nevada Mining Company; Review of Title Reports and Update Report for Robinson Mine, White Pine County, Nevada". The purpose of this report was to bring title forward from 1999 resulting in an updated title opinion. Issues requiring attention were identified and twelve recommendations made. Quadra received this title opinion prior to acquiring the Robinson Mine. Additional Mill site claims were staked in 2008 for the purpose of establishing Rapid Infiltration Basin sites, many of these claims will be dropped in 2009 once final site selection is completed.

A list of property, patented, and unpatented claims controlled by the Robinson Operation can be obtained by contacting the Quadra corporate office in Vancouver, BC, Canada. The reported annual costs for maintaining the unpatented lode and mill site claims are as follows:

Payment to the Bureau of land Management, Yearly Assessment fees of $150.00 per claim: Payment to White Pine County,		$150,500.00
Intent to hold ($8.50/claim) and recording fees:		$ 10,714.00
	Total:	$161,214.00

The Robinson operation is subject to a three percent net smelter royalty (NSR) currently payable to Royal Gold Inc. (Royal Gold). This royalty was formerly payable to Kennecott Minerals Company (Kennecott). This NSR was to be used in the first instance to fund a reclamation trust and indemnify Kennecott for environmental liabilities, including reclamation costs. The trust was funded with the three percent NSR up to $20 million, including interest and with credits for certain reclamation expenditures. Once the trust was fully funded pursuant to the provisions of the trust, the NSR royalty was sold by Kennecott to Royal Gold.

In addition to the Royal Gold royalty, Franco-Nevada Corporation (Franco-Nevada) is entitled to receive royalties from the production of the Robinson Mine. The royalties owing to Franco-Nevada consist of:

- A 10% royalty on net smelter returns on 51% of the production of gold from the Robinson Mine in excess of 60,000 troy ounces per calendar year;
- A royalty on 51% of copper production in excess of 130 million pounds of copper, payable in any calendar year in which the price of copper exceeds US$1.00 (adjusted for inflation from 1990) at the end the year (the "Trigger Price"), in an amount equal to US$0.05 per pound plus an incremental amount

equal to 40% of the amount by which the price of copper exceeds the Trigger Price; and

- A 0.225% royalty on net smelter returns of all minerals from the Robinson Mine.

Robinson operations historically (Kennecott and BHP) used rail to transport ores or concentrate. The Robinson Nevada Rail Road (RNRR) currently has a right-of-way from the United States Department of the Interior, Bureau of Land Management (BLM) for the section of the rail line that is on unpatented mineral claims owned by Robinson. This 5 km section of railroad is currently un-usable due to several factors which prevent it from ultimately connecting with Union Pacific Railroad at Shafter, Nevada. As a result, Robinson currently trucks concentrates from its operations to a rail trans-loading facility located in East Wendover, Utah. At that point, the material is loaded in rail cars for shipment to the Vancouver Boat Terminal, located in Vancouver, Washington for final shipment to smelters.

4.1 Environmental Considerations and Permits

The following information describing environmental considerations and permits for RNMC, has been provided by Pat Gochnour, of Gochnour & Associates, Inc.:

The Robinson Operation (Robinson) is part of an industry that is subject to the application of numerous laws, regulations, permits and licenses, as well as internal and external (community and regulatory) conditions that are designed to protect the environment. These expectations and regulatory provisions are in place to protect the quality of land, water, and air, and provide for cleanup and reclamation of impacted lands for future post mine land uses.

Environmental legislation provides for restrictions and prohibitions on spills, releases or emissions of various substances from operations. Evolving regulatory standards and expectations can result in increased litigation and/or increased capital, operating, compliance and remediation costs, all of which may have a material impact on existing, as well as future operations.

Robinson has developed environmental policies, standards and procedures that have demonstrated a strong commitment towards public health, welfare and the environment. In addition, Robinson has developed a management system that has allowed them to assesses risk based issues and formulate action plans in order to mitigate environmental risks.

At this time, there are no known environmental issues that would limit or preclude exploitation of the permitted resources.

Permits currently held and maintained by RNMC are listed in Table 4.2.

Table 4.2:
Robinson Nevada Mining Company - Robinson Mine Permits

Description of Permit	Permit Term/ Status
• Class II Air Quality Operating Permit No. AP1021-0373.02, issued by Nevada Division of Environmental Protection – Bureau of Air Pollution Control	Permit issued October 31, 2006 and expires October 31, 2011. • Permit modification (replacement pages for permit for spray painting – maintenance) June 18, 2007. • Permit modification (add System 38 – Pump House Generator) June 27, 2008. • Permit modification (move System 38 to Insignificant Activity List) August 29, 2008.
• Nevada Hazardous Materials Storage Permit No. 2917-7336, FDID No. 17856, issued by Nevada State Fire Marshal	Permit reissued March 1, 2008. Permit term: March 01, 2008 through February 28, 2009.
• Hazardous Waste Facility ID # NVD982440539, issued by Environmental Protection Agency	Permit has no expiration.
• General Permit for Stormwater Discharges Associated with Industrial Activity from Metals Mining Activities No. NVR300000, issued by NDEP-Bureau of Water Pollution Control	General Permit reissued and active June 1, 2007 and expires June 1, 2012. Pursuant to NDEP's letter dated June 1, 2007, existing permit holders needed to resubmit a Notice of Intent (NOI) within 90-days of June 1, 2007. RNMC submitted their NOI on August 30, 2007. It also required a one time submittal of: 1) Revision to the Stormwater Pollution Prevention Plan (SWPPP), which is due *within 6 months of the effective date of this* [reissued] *permit* (Part I.C.3.ii). RNMC submitted theirs on November 30, 2007. 2) Updated monitoring plan for sampling stormwater discharges from waste rock dumps and overburden piles to waters of the U.S., which is also due *within six months of the effective date of this* [reissued] *permit* (Part I.C.12.i.a). RNMC submitted theirs on November 30, 2007.
• Discharge Permit No. NEV94013 for Operation of Wastewater Treatment Facility, issued by NDEP – Bureau of Water Pollution Control	Permit No. NEV94013 effective February 23, 2007 and expires February 23, 2012.
• Water Pollution Control Permit No. NEV92105, issued by NDEP- Bureau of Mining Regulation and Reclamation	Permit effective May 5, 2008 and expires April 5, 2010. • Permit updated per minor permit modification for the Moly Circuit, effective August 25, 2005. • Permit updated per minor permit modification for the D-Pad Gold Heap Leach Expansion, effective August 30, 2006. • Permit updated per minor modification for Ruth Pit Expansion & Facility Overdumping on May 1, 2008.
• Dam Safety Permit J-413, issued by State Engineer's Office, Nevada Division of Water Resources	Permit has no expiration. Annual reports are due every August. NDWR letter dated August 5, 2004, approves the Permit transfer to Robinson Nevada Mining Company.
• Permit to Operate a Public Water System, Permit No. WP-0855-12NTNC	Reissued on October 9, 2008 and expires October 31, 2009.
• Mining Bioremediation Facility General Permit No. GNV 041995, issued by NDEP – Bureau of Mining Regulation and Reclamation	Permit has no expiration date. • NDEP-BMRR is presenting their *New Program and Guidance for Management of Petroleum Contaminated*

Description of Permit	Permit Term/ Status
	Soil (PCS) at Mine Sites at workshops in October 2008. • RNMC submitted a PCS Management Plan to NDEP-BMRR on September 29, 2006. It has yet to be reviewed.
• Programmatic Agreement - Treatment of Historic Properties During Mineral Development Bureau of Land Management, et al, August 1992	Basis of Agreement accepted by Robinson Nevada Mining Company, August 19, 2004.
• Radioactive Material License 17-11-0372-01, issued by Bureau of Health Protection Services, Radiological Health	Permit re-issued November 14, 2005 under Amendment No. 10 issued by Bureau of Health Protection and it expires November 30, 2010.
• BHP Nevada Mining Water Rights Woodburn and Wedge letter, June 21, 2004 to H. Ricci, Nevada State Engineer	Submittal addressing legal name change for thirty-two permits. Package includes Report of Conveyance and Abstract of Title. Receipt of notice from the Division of Water Resources, December 2, 2005 confirming RNMC as owner of record.
• Mining Operation Reclamation Permit No. 0021, issued by NDEP-Bureau of Mining Regulation and Reclamation	Permit has no expiration. Permit transfer notice dated May 11, 2004 submitted to NDEP pursuant to NAC 519A.215.1(a). Revised permit re-issued November 3, 2005 to RNMC. • Permit reissued for December 2006 reclamation plan update on June 4, 2007. • Permit reissued for July 13, 2007 reclamation plan update for Ruth development on August 14, 2007. • Permit reissued for February 11, 2008 reclamation plan update for 2007-2008 Exploration Drilling Program. • Permit reissued for May 19, 2008 to include Minor Modification for Initial Expansion of Facilities in Support of Future Ruth Pit Mining. • 3-Year update submitted on September 16, 2008. It is still under review by BMRR and BLM. It includes more Ruth Expansion and bonds Robinson through December 31, 2011.
• Liquefied Petroleum Gas License No. 5-4546-01, Class 5, issued by the Nevada Board for the Regulation of Liquefied Petroleum Gas	License reissued July 15, 2008 and expires July 31, 2009.
• Plan of Operations, Robinson Project No. N46-92-004P, Bureau of Land Management	Plan has no expiration.
• Hazardous Materials Certificate of Registration, Reg. No. 080408 003 011QS, Issued by U.S. Department of Transportation Pipeline and Hazardous Materials Safety Administration.	Issued August 4, 2008 and expires June 30, 2011.
• Industrial Artificial Ponds Permit No. S-26608 for Mill/Tailings, issued by Nevada Division of Wildlife	Permit issued July 1, 2004 and expires June 30, 2009.
• Industrial Artificial Ponds Permit No. S-26609 for A, B & C East Heaps, issued by Nevada Division of Wildlife	Permit issued July 1, 2004 and expires June 30, 2009.
• Industrial Artificial Ponds Permit No. S-26610 for D Pad Heap, issued by Nevada Division of Wildlife	Permit issued July 1, 2004 and expires June 30, 2009.
• Solid Waste Mining Site Class III Waiver, Application No. SWMI-17-62, issued by NDEP-Bureau of Waste Management	Application has no expiration. NDEP letter dated October 18, 2007 approving Class III Waivered Landfill permit Application No. F468 for onsite disposal of large mine equipment tires in waste rock dumps.

All of the operating permits listed above are current and valid.

The above referenced permits are necessary for continued operations. No new permits are required, unless there is a "substantial change" to what has already been approved by the agencies. Minor permit amendments have been necessary to address minor design changes at the Robinson Operation. Review and approval of these notifications and requests have gone very smoothly without jeopardizing the ability operate.

Reclamation and remediation for closure is an ongoing activity at the mine site. Approved activities include facility dismantling, solution management, treatment and disposal, grading of disturbed areas (excluding pits), placement of growth media and revegetation. Regulations require that operations must post "financial assurance" to cover the cost of remediation of current disturbance plus three years of planned disturbance/operation. Land disturbed prior to October 1, 1990 and which is no longer actively being used as part of the operation (NAC 519A.375) is grandfathered and no financial assurance deposit is required. As of November 2008, the financial assurance requirement was $40,168,096. Regulations require that reclamation estimates and financial assurance account for reclamation being performed by an independent third party. The amounts required for financial assurance and final closure have been calculated by an independent third party (SRK Consultants, Reno Office) proficient in Nevada regulatory requirements.

As a condition of permitting, Quadra has accepted responsibility for reclamation obligations and posted a financial instrument (Letter of Credit).

Current estimates of final closure (if all permitted facilities are constructed) are ~$85.3 million. As owner/operator of the Robinson Operation, Quadra has assumed responsibility for these activities. Quadra regularly performs a full review of the closure plans and obligations in conjunction with mine planning, with an emphasis of "mining for closure" disposal of waste.

5. Accessibility, Climate, Local Resources, Infrastructure and Physiography

5. Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Robinson Mine site is accessed via a public paved road that connects to US Highway 50, west of Ely, Nevada. Ely has an airport with only limited commercial airline service at the present time. The property has limited rail accessibility due to the current poor condition of the rail line that connects to the Union Pacific Railroad at Shafter, Nevada, located 200 km to the north of the mine site. The metal concentrate is transported by truck from the mine site to East Wendover, Utah, where rail trans-loading is undertaken. The concentrate is then railed to port facilities for sea transport.

The property directly borders the town of Ruth, Nevada. Precipitation falls regularly throughout the year, with an average annual precipitation of approximately 30 cm at Ruth. Historically, snow has been recorded in all months of the year except July and August. During the summer, the average temperature ranges from about 29ºC to 7ºC and in the winter, the range is 7ºC to -15ºC (www.wrcc.dri.edu/cgi-bin/climain.pl/nvruth). The topography is generally rugged at an average elevation of approximately 2,130 m. The area's vegetation consists mainly of sagebrush, piñon pine and juniper trees.

Continuous mining and processing of ore has been conducted at the mine site year-around since 2004. All mining and processing facilities at the Robinson Mine are in good working condition at the time of writing. In addition, commercial electrical power, telephone lines, and water supply infrastructure at the site are all operational. RNMC utilizes numerous water wells that supply sufficient water to meet the site's requirements during full operating conditions.

Presently, there are just over 500 workers employed by RNMC. During the 2004-present operating period, the majority of RNMC's employees resided in the Ely-Ruth area. A 2002 study indicated there are potentially 1,100 employees available within a 100-mile radius of the Robinson site, and as of December 2008, there were more applicants than positions available at RNMC.

6. History

6. History

The following is summarized from an internal BHP company report written in 2000, *Robinson Project, Internal Ore Reserve Report, FY2000*.

> *The Robinson District was founded in 1867, when several underground gold and silver mines were established. By the early 1900's, copper-gold-molybdenum ores were mined, with the first copper production in 1908. Up until 1958, there were numerous companies operating in the district, including Giroux Consolidated Mining Company, Nevada Consolidated Copper Company, Consolidated Coppermines Company, and the Nevada Mines Division of Kennecott Copper Corporation, who consolidated and controlled the district by 1958 through a series of purchases and buy-outs. The majority of production came from five large open pits, with lesser production from underground mines and smaller pits. Ore was hauled by rail approximately 22 miles to a mill and smelter at McGill, Nevada. Kennecott closed the mines in 1978, reportedly due to low copper prices and outdated mining and processing facilities.*
>
> *Production reported for the period 1908 to 1978 is more than 4 billion pounds copper, 2.7 million ounces of gold. Additional metal recovered included molybdenum, silver, lead, zinc, manganese, rhenium, palladium, and platinum.*
>
> *Through a series of leases with Kennecott, Silver King Mines and Pacific Silver Corporation (predecessors of Alta Gold Company) began mining a series of small gold-silver deposits in the district. Alta Gold subsequently entered into a joint-venture agreement with Echo Bay Mines and mined the deposits through 1991. Gold and silver were recovered using carbon-in-pulp milling and heap leaching. BHP reports that between 1986 and 1991, approximately 300,000 ounces of gold and 200,000 ounces of silver were produced from the Robinson District.*
>
> *In 1990, Magma Copper Company bought all mining rights from Kennecott and also entered into a joint-venture agreement with Alta Gold; in early 1991, Magma exercised its option to become operator of the gold-silver mines. By May of 1991, Magma had decommissioned the mill, bought Alta Gold's interest in the gold operations, and reduced Echo Bay's interest to a royalty. By October 1991, Magma acquired a 100% working interest in the district by buying the remainder of Alta Gold's interest in the joint venture. Gold production continued until 1993, with approximately 77,000 ounces of gold produced between 1991 and 1993.*

> *Magma began stripping the Liberty copper-gold pit in mid-1995 and commenced production in late-1995. In early-1996 Magma was acquired by BHP, who continued production from the Liberty pit and commissioned the mill, which operated at a throughput of approximately 40,000 tpd. Concentrates were shipped to BHP's San Manuel smelter for refining. Production from the Tripp pit began in early-1998. BHP discontinued mining at Robinson in mid-1999….*

Table 6.1 is a summary of 1996-1999 Robinson operation production as reported by BHP and all values in Table 6.1 are stated in metric units. The vast majority of mining during this period was from the Liberty deposit.

Table 6.1:
Summary of Robinson Operation Production (1996-1999)

		1996[1]	1997	1998	1999
Material mined	(000 tonnes)	19,414	69,116	78,686	87,857
Ore milled	(000 tonnes)	3,095	12,814	13,457	13,869
Average head grade					
Copper	(%)	0.456	0.546	0.622	0.561
Gold[2]	(g/t)	0.190	0.249	0.457	0.342

(1) 5 months to May 31
(2) reported in oz/tonne, converted to g/t

Magma Copper and BHP reported reserves for the property in annual reports, and summaries of some these estimates are reproduced in Table 6.2. Note that the 1993 reserves are in imperial units whereas the other reports are in metric units.

Table 6.2:
Historic Robinson Reserve Estimates

Magma Reported Reserves 1 January 1993

Tons (000's)	Copper grade %	Gold grade oz/t	Recoverable Cu lbs (000's)
201,384	0.605	0.11	1,911,148

BHP Reported Reserves 31 May 1997					
	Tonnes Millions	Copper grade %	Gold grade g/t	Recoverable (000's)	
				Tonnes Cu	Troy oz
Proved	213	0.55	0.27	1,044	1,273
Probable	7	0.50	0.24	33	39
Total	220	0.55	0.26	1,077	1,312

BHP Reported Reserves 31 May 1998					
	Tonnes Millions	Copper grade %	Gold grade g/t	Recoverable (000's)	
				Tonnes Cu	Troy oz
Proved	217	0.55	0.23	1,054	1,105
Probable	9	0.49	0.25	38	47
Total	226	0.54	0.23	1,092	1,152

BHP Reported Reserves 31 May 1999					
	Tonnes Millions	Copper grade %	Gold grade g/t	Recoverable (000's)	
				Tonnes Cu	Troy oz
Proved	168	0.59	0.26	851	603
Probable	8	0.51	0.26	31	26
Total	176	0.59	0.26	882	629

The reporting system and accuracy were not stated and the commodity prices used to establish the BHP reserves are not known. While it was not stated what specific classification system was used, the following statement describing reserves was included in the 1999 BHP Annual Report.

> *All reserve statistics for mineral reserves are quoted in terms of the product. This is the estimated quantity of material that can be profitably mined, processed and sold or consumed internally. Current recovery factors have been applied, as required by the rules of the Australian Stock Exchange, and a competent person has determined whether a reserve should be classified as marketable, proved or probable.*

In 2004, Quadra acquired the Robinson Property from BHP and began mining. Table 6.3 shows a summary of the estimated reserves as of June 8, 2004, which were reported in a technical report entitled "Technical Report on Robinson Operation, Ruth, Nevada, USA" dated June 30, 2004 (the "2004 Technical Report"). The 2004 Technical Report met CIM and NI43-101 standards in effect at the time (2004) and was filed on the SEDAR at www.sedar.com website. The reserves in Table 6.3 are stated in metric units, with the exception of contained gold ounces.

Table 6.3:
Historic Robinson Operation Reserve Estimates as of June 8, 2004

		Grades				Waste	Total	
		Total	Au	Contained Metal (000)		Tonnes	Tonnes	Strip
Total Robinson	**Ore Tonnes (000)**	**Cu %**	**g/tonne**	**Cu Tonnes**	**Au oz**	**(000)**	**(000)**	**Ratio**
Proved	128,433	0.686	0.287	881	1,187			
Probable	4,282	0.716	0.226	31	31			
Proved & Probable	**132,714**	**0.687**	**0.285**	**911**	**1,218**	**436,067**	**568,781**	**3.3**

RNMC has been operating the Robinson property continually since 2004. Production from 2004 through 2008 has been entirely from the Tripp-Veteran deposit. Table 6.4 is a summary of 2004-2008 Robinson operation production as reported by Quadra. This production is reported in short tons with gold grade reported in ounces per ton.

Table 6.4:
Summary of Robinson Operation Production (2004 – 2008)

	2004	2005	2006	2007	2008
Expit Material Mined (000 Tons)	19,927,256	78,349,386	82,112,883	81,211,174	79,566,712
Ore Milled (000 Tons)	3,625,811	15,164,511	15,278,412	15,620,982	15,257,229
Ore Head Grade (Total Cu)	0.508	0.545	0.607	0.627	0.679
Ore Head Grade (Au opt)	0.009	0.009	0.009	0.011	0.013

Expit: Material mined from inside a pit and transported to a processing plant or rock storage facility outside the pit, as opposed to material hauled from stockpiles or material hauled solely within the pit

After purchasing the property from BHP in 2004, Quadra undertook a 2.5 year exploration program that included both new drilling (Section 10.3) and a re-assaying program of existing core and drill pulps from historic drilling (Section 11.1.4).

7. Geologic Setting

7. Geologic Setting

7.1 Regional Geology

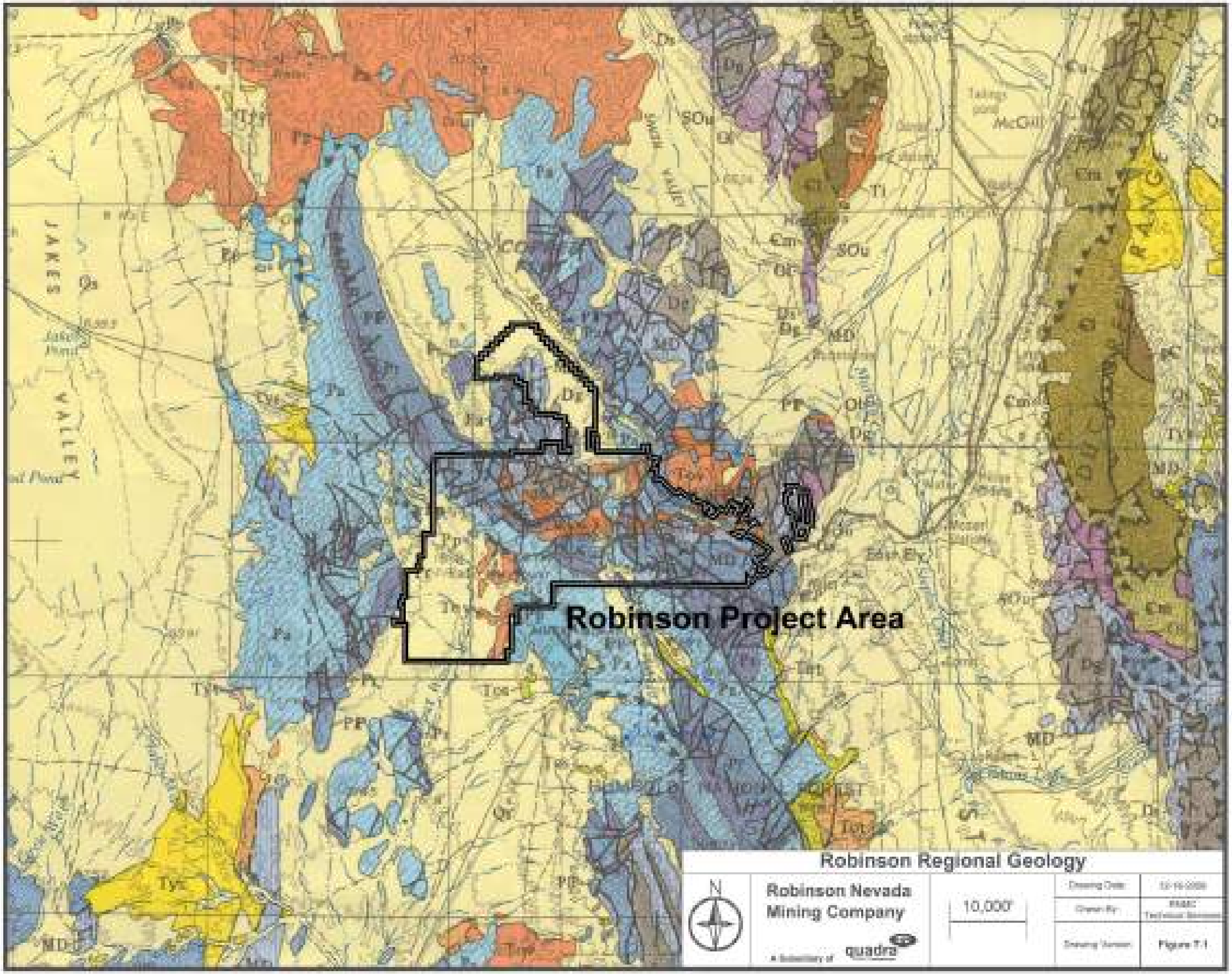
The Robinson District, in the Egan Range of east-central Nevada, is underlain by more than 11,000 feet (3,350 m) of miogeoclinal clastic and carbonate rocks, including the Devonian Guilmette Formation upward through the Permian Arcturus Formation. At approximately 111 Ma (McDowell and Kulp, 1967), a quartz monzonite porphyry intruded the sedimentary rocks. Faulting evidently was active either prior to, or concurrently with, porphyry emplacement. Hydrothermal alteration and mineralization associated with the intrusive event, in both the wall rocks and the intrusion itself, resulted in the copper and gold deposits at the Robinson property (Hose, Blake and Smith, 1976; Kliche, Knight, & Stevermer, undated). Figure 7.1 provides a map of the regional geologic setting surrounding the Robinson District.

During the early Tertiary, the district was overlain by conglomerate and lacustrine limestone of the Eocene Sheep Pass Formation, and by a series of rhyolitic volcanic rocks. Rhyolitic dikes and diatremes, also of Tertiary age, cut the strata.

The post-mineral structural history of the Robinson District is very complex. Tertiary extension resulted in complex, multiple stages of dismemberment and tilting. Sets of tilted normal fault blocks are themselves cut by several later series of normal faults, resulting in structural superposition. Faulting also caused mineralization that formed at varying elevations to be exposed at the surface, further complicating geologic interpretation (Albino, 1995).

Four general types of deposits have been mined in the area:

1. Copper±molybdenum±gold deposits in altered quartz monzonite porphyry. (mineralization occurs as disseminations and in quartz veinlets);
2. Carbonate-hosted copper±gold deposits adjacent to the porphyry. (includes both calc-silicate skarn deposits and silica-pyrite replacement deposits);
3. Disseminated gold deposits in limestone and calcareous sandstone peripheral to the copper mineralization. (these deposits are controlled by both stratigraphy and structure); and,
4. Supergene chalcocite deposits in both porphyry and sedimentary units that can be up to 100 m thick (Albino, 1995).


Robinson Project Area
JAKES VALLEY
McGill
Robinson Regional Geology
N
Robinson Nevada
Mining Company
quadra
10,000'
Figure 7-1

7.2 Property Geology

A significant body of literature has been written on the geology at the Robinson operation. The reader is referred to the numerous reports, documenting nearly 100 years of geologic study, for detailed descriptions of the geology and mineralization of the Robinson District (Bauer, Cooper, and Breitrick, 1960; Seedorff, Houhoulis, undated; Westra, Gerhard, 1982; Westra, Gerhard, 1976). In general, the Robinson deposits are characterized as porphyry Cu±Mo±Au systems that are associated with mildly acidic, monzonitic rocks of Cretaceous age. Copper mineralization, with molybdenum by-product, is hosted in porphyry and in skarn that formed in calcareous rocks adjacent to mineralized porphyry. Supergene enrichment resulted in chalcocite blankets up to 100 m thick. Gold deposits are hosted by various calcareous sedimentary rocks and are generally located around the periphery of the copper deposits.

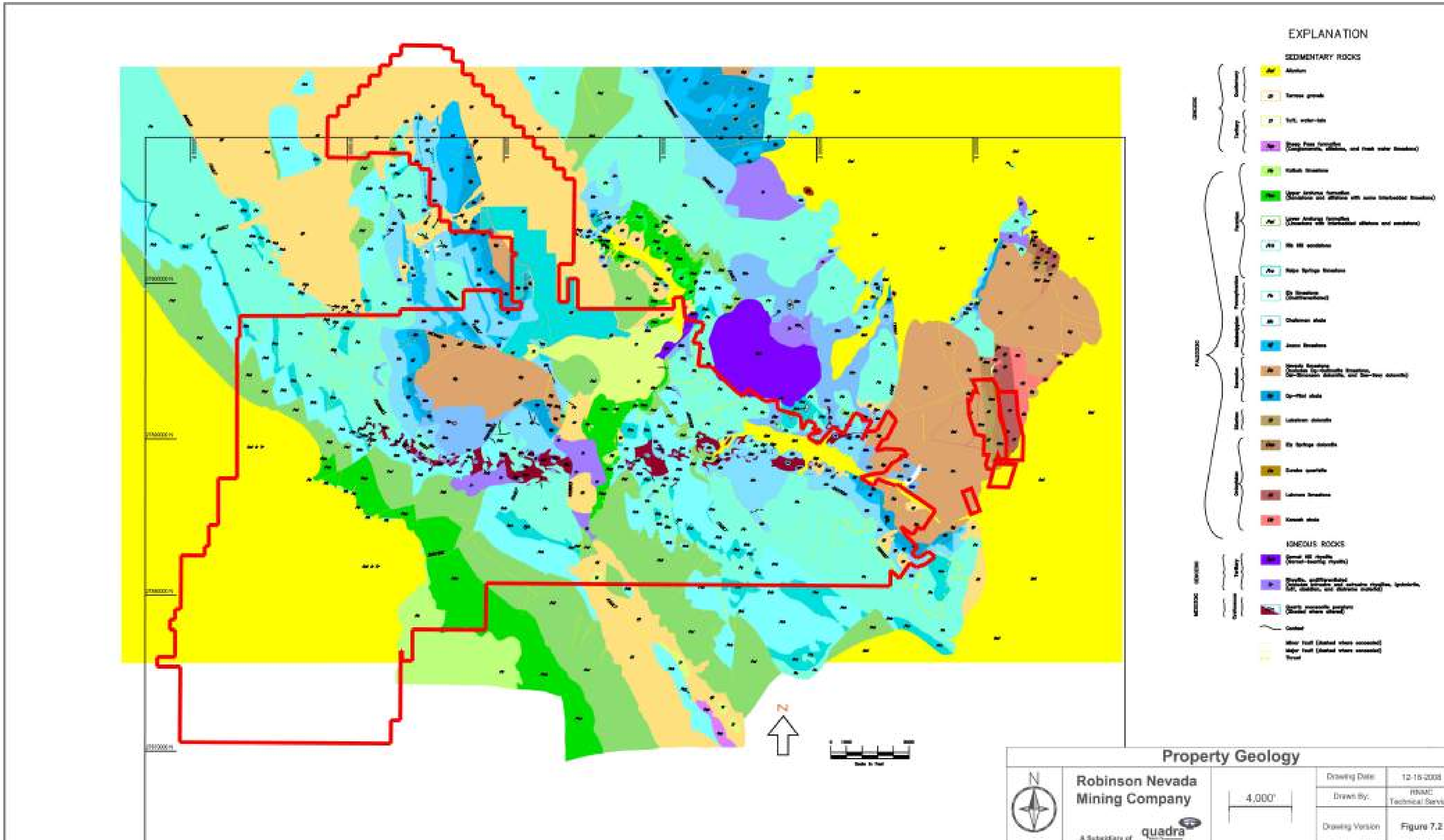
Sedimentary rocks exposed in the Robinson District range in age from Devonian to Tertiary. Copper skarn mineralization is primarily hosted in the Pennsylvanian Ely Limestone and in the upper portion of the Mississippian Chainman Shale. Known gold deposits occur in Mississippian to lower Permian strata, which are stratigraphically higher and lower than the copper mineralization. The Paleozoic rocks are moderately to (locally) strongly folded and complexly faulted. Figure 7.2 provides a detailed illustration of the Robinson property geology.

The porphyry and skarn copper mineralization are related to monzonitic intrusions that have been dated at approximately 111 Ma (early Late Cretaceous). Evidence suggests that the mineralized porphyry is associated with the Weary Flat pluton, which is found at deeper stratigraphic levels.

Primary copper mineralization occurs as chalcopyrite+pyrite associated with potassic and quartz-sericite alteration in the porphyry and with hydrous retrograde assemblages in skarn. Near-surface primary mineralization was overprinted by important quantities of chalcocite mineralization associated with Tertiary-aged supergene leaching and enrichment. While chalcocite enrichment is believed to have occurred before and during Tertiary extension, the current distribution of chalcocite reflects post-structural leaching and enrichment.

Primary gold mineralization is found in association with the primary copper mineralization but also occurs in calcareous sedimentary rocks peripheral to the porphyry. Gold is present in chalcopyrite and as free gold.

Unaltered, post-mineral Tertiary rocks overlay the mineralized strata and porphyritic intrusions and provide important timing constraints regarding emplacement of intrusives and post-mineral deformation of the host rocks. Tertiary rocks exposed in the Robinson


EXPLANATION
SEDIMENTARY ROCKS
IGNEOUS ROCKS
N
Property Geology
Robinson Nevada
Mining Company
A Subsidiary of quadra
4,000'
Drawing Date: 12-16-2008
Drawn By: RNMC Technical Services
Drawing Version
Figure 7.2

District include conglomerate and lacustrine limestone of the Eocene Sheep Pass Formation that is overlain by a series of rhyolitic volcanic rocks. Tertiary dikes and diatremes, generally of rhyolitic composition, also cut the strata.

Pre- and syn-mineralization structures that are documented and/or interpreted to be channels and conduits for porphyry-related hydrothermal solutions include steeply dipping normal faults with negligible offset; steeply dipping normal faults with up to several hundred feet of offset; and, a series of prominent, steeply dipping normal faults that are believed to have controlled the emplacement of the mineralizing porphyry system. Other Mesozoic-age structures include minimal displacement, low-angle faults that are generally sub-parallel to bedding and locally-controlled mineralization and alteration proximal to the porphyry intrusions.

The post-mineralization structural history of the Robinson District is very complex; the district is situated along a Tertiary extensional zone. Geologic investigations over the years have identified at least seven major structural sets within the district itself, all of which appear to be normal faults with minor oblique-slip and each of which off-set and rotated the previous set of faults. Normal faults range from low-angle (~5°) to high-angle, the majority of which are moderately dipping (40° to 50°).

The intrusive rocks within the Robinson District exhibit a wide range of alteration, including potassic, propylitic, intermediate argillic, sericitic, and advanced argillic. Carbonate rocks in contact with the porphyritic intrusions have been altered to calc-silicate assemblages, including hornfels, garnet-pyroxene skarn, and massive magnetite, that are spatially and, most likely, genetically related to the potassic alteration of the porphyry. Mineralized garnet-pyroxene skarn typically has copper and gold grades higher than the adjacent porphyry. Silica-pyrite alteration developed synchronous with sericitic alteration. The silica-pyrite alteration is economically important, as it is generally associated with anomalous gold mineralization.

8. Deposit Types

8. Deposit Types

In general, the Robinson deposits are characterized as porphyry Cu± Mo±Au systems that are associated with mildly acidic, monzonitic rocks of Cretaceous age. Copper mineralization is hosted in porphyry, skarns and replacements that formed in calcareous rocks adjacent to mineralized porphyry. Gold deposits are hosted by various calcareous sedimentary rocks and are generally located around the periphery of the copper deposits.

All of the district mineralization is genetically related to the porphyry system and a long list of deposit types could be generated depending upon occurrence of metal, associated alteration, rock types and structures. However, there are four types of deposits that have been mined, these are:

1. Copper±molybdenum±gold deposits in altered quartz monzonite porphyry (mineralization occurs as disseminations and in quartz veinlets);
2. Carbonate-hosted copper±gold deposits adjacent to the porphyry (includes both calc-silicate skarn deposits and silica-pyrite replacement deposits);
3. Disseminated gold deposits in limestone and calcareous sandstone peripheral to the copper mineralization (these deposits are controlled by both stratigraphy and structure); and,
4. Supergene chalcocite deposits in both porphyry and sedimentary units that can be up to 100 m thick.

Studies suggest that metal zonation is not as systematic as for other porphyry-related systems. In general, the deeper levels of the mineralizing system are characterized by a proximal zone of granular quartz + biotite + chalcopyrite, which is zoned laterally out to weak potassic alteration. Gold generally follows copper, with the highest grades of each metal occurring slightly inside the porphyry-sedimentary rock contact. Intermediate levels are characterized by the addition of argillic and sericitic alteration, superimposed on potassic alteration. There is an abundance of molybdenum that tends to occur outside of the porphyry and forms a 'halo' around the copper mineralization. The highest exposed levels of the system exhibit intense sericitic and advanced argillic alteration, with only moderately anomalous copper, but contain anomalous molybdenum, and are notably enriched in gold.

9. Mineralization

9. Mineralization

9.1 Copper Mineralization

Copper mineralization at Robinson occurs in monzonite porphyry intrusive rocks and associated marginal garnet skarns with varying quantities of pyroxene, magnetite and sulfide which formed in the surrounding upper Paleozoic wall rocks. Early stage potassium silicate alteration in porphyry and skarn mineralization in the wall rocks are typically overprinted by a widespread and commonly intensive high acid alteration event which altered porphyry and some host units to quartz-sericite-pyrite or even advanced argillic assemblages locally (James, 1976; Westra, 1982). Carbonate host rocks were altered to pyritic marbles or silica pyrite rock containing up to 20 volume percent pyrite locally during this late stage alteration event. Early stage potassium silicate mineralization and skarn was dominated by pyrite and chalcopyrite and is generally low in sulfide with average values ranging from trace to 3%. The high acid alteration event is dominated by up to 10% or 20% pyrite with pyrite:chalcopyrite ratios typically ranging from 10:1 to more than 1,000:1. Where present in the late stage alteration zones, the chalcopyrite may be relict from earlier stage alteration events. Chalcopyrite appears to be the sole hypogene copper mineral, with a few possible exceptions.

Skarn and porphyry ores form much of the current resource and reserve with grades ranging from cutoff (and below) to greater than 1.0%. Metallurgical recoveries in this material are generally good, but are complicated where weathering has partially altered the sulfides or silicate gangue.

Mid-Tertiary extension and normal faulting dissected and rotated the hypogene copper deposits to their current configuration and they have been extensively oxidized, leached and enriched. The oxidation and enrichment is extensive, generally pervasive and intense. Faults and fracture zones locally modify the morphology of the leached and enriched zones, but in many places the upper surface of the original enrichment blankets parallel the surface. In light of this, it seems likely that much of the weathering and enrichment took place after the mid-Tertiary faulting.

Before the period of gold mining from the late 1970's through the early 1990's, copper miners mainly exploited supergene chalcocite mineralization which developed best where porphyry hosted mineralization with intense quartz-sericite-pyrite alteration allowed for strong development of enrichment blankets. High grade, hypogene and enriched skarn deposits were also mined. These deposits, mined from the surface and in bulk tonnage block caves during the period from 1906 to the late 1970's, accounted for more than 300 million tons of material at a grade of roughly 1.0% (Smith, 1976). Some of this material, mined in the early days of the district, had significant tonnages with grades of 2% to 3% copper.

9.2 Copper Ore Types

Several ore types are segregated for planning and mining purposes:

- *Leached Cap* is intensely weathered and oxidized material with partial to complete destruction of protolith mineralogy and texture; leaving a relict, porous network of iron and manganese oxides, silica and clays. Leached Cap typically contains zones of relict copper sulfides and non acid soluble copper oxides as well as leachable copper oxides and gold. Mill recoveries when processing this material are typically very low. Leached Cap typically grades downward into Supergene Enrichment Blanket with depth.
- *Supergene Enrichment Blanket* is comprised of secondary copper sulfide and oxide minerals concentrated at the water table by the neutralization of downward flowing acidic, copper bearing meteoric fluids. This material is characterized by as much as a 2 or 3-fold increase in copper grade compared to underlying hypogene mineralization and is still an important ore type in the district; it has relatively good grades, but generally has lower mill recoveries due to complex and varying sulfide and gangue mineralogy.
- *Secondary Sulfide Ore* occurs in a mixed zone where primary hypogene sulfides are partially replaced by the secondary sulfides chalcocite and digenite. The zone is coincident with the zone of quartz-sericite-pyrite and locally silica-pyrite alteration. The secondary sulfide zone begins at the base of the Supergene Enrichment Blanket and grades downward into hypogene mineralization with the diminishment of secondary sulfide overprinting.
- *Hypogene Ore* is composed of chalcopyrite and magnetite bearing ores, hosted primarily within prograde, calc-silicate skarns and potassically altered monzonite porphyry intrusions.

9.3 Geology and Mineralization of the Tripp-Veteran Pit

Monzonite porphyry, historically known as the "ore porphyry" intrudes Chainman formation shales, Ely formation cherty limestones, Reipe Springs limestone and Rib Hill sandstone in the Tripp-Veteran open pit. The deep and proximal portions of the hydrothermal system are characterized by prograde garnet-pyroxene+/-magnetite skarns in the sedimentary and potassium silicate alteration assemblages in the intrusions.

Carbonate and fine grained clastic wall rocks on the distal margins of the ore deposit are recrystallized to marble and hornfels respectively. Shallower and proximal portions are characterized by strong hydrolytic overprinting of the prograde and potassium silicate assemblages, yielding retrograde skarn assemblages and intensive and extensive quartz-sericite-pyrite alteration in the intrusions. The nearer surface pyrite-rich retrograde and quartz-sericite-pyrite zones subsequently were oxidized and leached, and extensive chalcocite enrichment was superimposed on the system.

Oxidation, leaching and enrichment produced thick and rich chalcocite blankets within the area of the current Tripp-Veteran open pit. The supergene blanket gently plunges to the west and south and crudely honors primary stratigraphic and structural controls. It passes downward into chalcopyrite-magnetite bearing hypogene ores that persist past the pit design limits.

Unaltered rhyolite of mid-Tertiary age intrudes and occupies zones of tectonic weakness, particularly in the southeastern half of the open pit. Occasionally, incorporated, comminuted mineralized porphyry and skarn fragments were abundant enough to raise certain rhyolite bearing zones above ore-grade cut-off.

The distribution of rock types and their signature alteration assemblages are as follows. A moderately dipping, east-southeast striking, intact sequence of upper Ely formation cherty limestones, Reipe Springs limestone and Rib Hill sandstone are exposed on the western and southern benches of the open pit, generally above the 6700 elevation, on average. The northeastern wall of the pit is formed by the Footwall West fault, which separates the pit from the Weary Flat structural block - characterized by a north striking, westward facing, steeply dipping sequence of lowermost Ely formation, Chainman shale, Joana limestone, Pilot shale and Guilmette formation. Monzonite porphyry is present within both of these stratigraphic domains. The Weary Flat porphyritic monzonite body is confined to the Weary Flat structural block. Most of the economic mineralization in the area is to the southwest of the Footwall West fault.

The northwest trending Tripp-Veteran pit is separated from the Weary Flat structural block on the northeast by the steeply southwest dipping Footwall West fault, an ore boundary. Drill substantiated mineralization is present to the north of the Footwall West fault but lies outside the current pit design. In addition, the pit is crossed by the southeast dipping Pilot Knob fault, both of these faults have substantial post-mineral normal movement.

9.4 Geology and Mineralization of the Liberty Pit

The monzonite intrusion at Liberty divides Ely formation limestone and Rib Hill sandstone on the south from Chainman formation lithologies on the north. These rocks are separated from the Weary Flat block including weakly altered Weary Flat monzonite and Devonian through Mississippian stratigraphic section by the southeast dipping Footwall East fault, which places the upper Chainman and lower Ely formations down against Guilmette formation and Permian rocks down against Chainman; supporting offset of hundreds to thousands of feet of offset.

Supergene and hypogene sulfide ore bodies in the Liberty pit were hosted in altered porphyry and skarns in the Ely formation and upper parts of the Chainman formation. Rocks on the south side of the existing pit are now intensely weathered leached cap, probably after silica-pyrite altered limestones and calcareous sandstones of the Ely and Rib Hill formations, the few *in-situ* exposures of Chainman formation on the north are also intensely weathered leached cap.

9.5 Geology and Mineralization of the Ruth Pit

Chainman, Ely, Reipe Springs and Rib Hill formations were intruded by monzonite porphyry within the Ruth open pit, and historically mined mineralization was hosted in porphyry and surrounding skarns of the Ely and Chainman formations. The Chainman and lower part of the Ely formation are separated from strongly altered limestone and sandstones of the upper Ely and the overlying Permian units along the west dipping High Grade fault. Rocks above and below this fault are strongly altered, but the best hypogene mineralization lies in porphyry and, to a lesser extent skarn, below the fault. Grades above the fault are generally lower and considerably more erratic. Geologically, the Ruth ore body is separated from the Liberty on the west by the Eureka fault, a 35º east dipping fault with more than 3,000 feet of down to the east movement, and bounded on the east by the Queen fault, a 40º to 50º east dipping fault with 500 feet or more of normal movement. All of these faults offset intrusive rocks and hypogene alteration and mineralization and are mid-Tertiary in age.

Late stage hypogene quartz-sericite-pyrite and silica-pyrite alteration have largely obliterated any pre-existing early stage potassium silicate and skarn alteration above the High Grade fault and have strongly overprinted it below the fault. The leached cap-supergene enrichment blanket-hypogene sequence in the proximal portion and marble and hornfels in the distal portions of the orebody are similar to those described from the Tripp-Veteran open pit. However, the intensity of late stage alteration in the Ruth area is significantly greater and this has produced an orebody with significantly more quartz-sericite-pyrite and silica-pyrite alteration than found at Tripp-Veteran. In that regard, Ruth is much more similar to Liberty than Tripp-Veteran.

9.6 Disseminated Gold Deposits

The Robinson district originated as a gold producer in the 1880's and gold was again the primary commodity during the Alta Gold era of the 1980's and early 1990's. In addition, gold has been and remains an important by product of copper mining. The total production of the district surpassed more than 2,000,000 ounces in the 1990's (Tingley, 1998) and Quadra has produced more than 400,000 ounces since restarting production in 2004 (Quadra Mining Ltd. quarterly reports for 2005 through third quarter 2008).

Gold appears to be strongly anomalous in all parts of the Robinson hydrothermal system, and is clearly associated with copper mineralization, but a number of deposits have stood on their own as gold producers (Figure 9.1). These deposits are generally distal to copper mineralization and are hosted in sedimentary rocks. The Rib Hill sandstone is the most important host, but the Chainman and Pilot shales as well as the Ely formation also host important deposits. Of the eleven stand alone gold deposits recognized in the 1980's and 1990's, most are associated with silicification and hydrolytic alteration of the host rocks, although primary skarn is present at some localities. The most important deposit, the Star Pointer mine, lies in intensely decalcified and pyritized Rib Hill sandstones in the hanging wall of the High Grade fault above the Ruth open pit. The rocks at Star Pointer are now completely oxidized and strongly leached and it is not clear whether there was significant hypogene copper associated with the gold mineralization. All of the stand alone gold deposits mined in the 1980's and 1990's were thoroughly oxidized and were mined by small open pits and processed mainly by cyanide heap leaching.

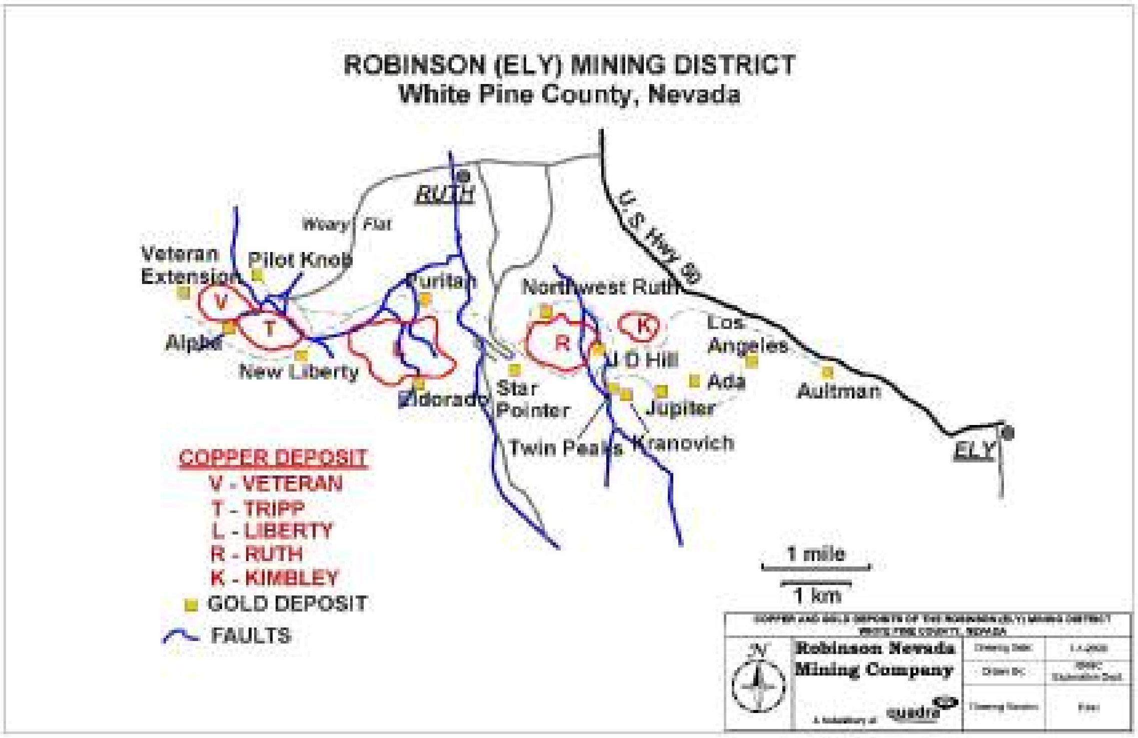


Figure 9.1: Copper and Gold Deposits of the Robinson (Ely) Mining District, White Pine County, Nevada

10. Exploration and Development

10. Exploration and Development

The first mines in the Robinson district began producing in the 1880's and the area had been thoroughly prospected by the time the first large scale copper mines were developed during the early 1900's. The discovery of new resources and the development of the understanding of district geology proceeded apace with the development of the understanding of the genesis and controls of leached cappings and chalcocite enrichment blankets in the early part of the last century and then with the increasingly sophisticated understanding of porphyry copper deposits after World War II. For a long time, district exploration has been largely indistinguishable from development and much of the discovery of resources has probably been more strongly driven by the simple expedient of extension drilling as it has by any conceptual ideas.

Kennecott, in the period of the 1950's through the late 1970's, did make a considerable effort to evaluate and expand the resources in the district with the XD (for exploration drilling) program, which resulted in the discovery of additional resources in the Kimbley, Wedge and Nelly areas east of the main Ruth deposit. The drill program was coordinated with a comprehensive effort to map the district geology and to detail the complex stratigraphy and structure. Geologic mapping and stratigraphic work done by Richard Breitrick and John Welsh of Kennecott resulted in a district geologic map and detailed understanding of the district stratigraphy (various Kennecott Copper Company internal reports). Further work on alteration and structure in the area around Tripp-Veteran was completed by Larry James and around Ruth by Gerhard Westra in the 1970's (James, 1976; Westra, 1982). The understanding of district structure was hindered by the general lack of knowledge about Tertiary extension throughout the Cordillera. These ideas were not well developed until the 1970's and continue to develop to the present day.

Alta Gold in the 1980's developed a number of stand-alone gold deposits, starting with the Star Pointer and including a number of smaller deposits throughout the district. Their approach was simple and practical, emphasizing drilling and assaying over geologic understanding; but was successful in developing and mining a number of additional deposits after the Star Pointer was exhausted. Later work by Magma and then BHP did not succeed in finding additional gold resources, but substantial development drilling was done to expand and confirm copper resources in the Tripp-Veteran, Liberty and Ruth.

Since 2004, work by Quadra has primarily concentrated on expanding the known resource and evaluating untested ground within and around the projected open pits. In 2008 several holes were completed on hypothetical targets which were developed on the basis of detailed geologic studies. A detailed airborne magnetic survey was also completed as an aid to three-dimensional geologic interpretations in the development of new conceptual targets.

10.1 Quadra Exploration and Development Summary

Robinson Nevada Mining Company has expended funds during the project years of 2006 through 2008 to provide for the continuation of the ongoing New Robinson Mine Optimization Project Capital exploration program (NRMOP). Expenditures are as follows; 2006 - $2.1 million, 2007 - $11.2 million and a forecast of $18.5 million for the ongoing 2008 project. These projects included four separate programs within the overall NRMOP Project; the Veteran Mine and Ruth Deposit Drill Programs, the District Sample Archive Re-Assay Program, the District-wide Database and Geologic Modeling Program, and the New Concept Drill Program.

The Veteran Mine and Ruth Deposit Drill Programs were designed to focus on delineation drilling, metallurgical drilling, geotechnical definition drilling and step-out drilling of mill-copper potential within and immediately adjacent to the existing infrastructure of the Tripp – Veteran mine and Ruth deposit areas and their logical extensions. Additional drilling resources were allocated for potential conversion of multi-metal resources (mill-copper, copper leach, gold-leach and molybdenum) into well constrained resources. The exploration staff assisted with a Becker hammer drill program of the Keystone dump in 2007.

The District Archive Sample Re-Assay Programs are being conducted to further delineate distribution of mineralogical and metallurgical characteristics within existing and potential mill-copper resources, as well as gold-leach, copper-leach, and molybdenum resources. To date, 1,047 district historic drill holes have been submitted for re-analysis utilizing 52 element ICP, Total Copper, Soluble Copper, Quick Leach Copper analysis, SAP analysis, molybdenum analysis, and a fire assay for gold.

The District-wide Database and Geologic Modeling Program has been focused upon generating a centralized database for the benefit of all operations departments. This new database will allow the geologists to refine or reinterpret the district geology in a comprehensive three-dimensional exploration model, a critical tool to generate new conceptual targets for testing. This program includes the re-logging, translation or reinterpretation of existing historic drill holes, remapping of all accessible pit exposures, remapping of the entire district adjacent to the ongoing operations and the detailed logging of the ongoing drilling, utilizing new standards of evaluation and data capture. The program has generated significant insight and wholesale changes to the interpretation of the deposit geology. These interpretational changes, as well as the expanded drilling, have led to increased resources and further refinement of the metallurgical characteristics of the deposits.

The New Concept Drill Program focuses on drill testing the new geologic concept targets derived from the interpretation of the District-wide Database and Geologic Modeling Program. These targets will be tested for the benefit of extension of current mine life or identification of new business opportunities. The program began the initial testing of two new district conceptual targets, Giroux Wash target and Taylor target, in the second quarter of 2008.

Drilling for these programs was performed by contract drillers under RNMC direct supervision. Sampling, assaying and drill core logging were conducted or supervised by RNMC employees.

10.2 The District-wide Database and Geologic Modeling Program

The NRMOP required the ability to compile and recapture existing data and new data into a centralized and standardized database. Ideally, this database will allow quick and simple use and evaluation of data across the entire district. The acQuire database system was selected and designed to import and preserve all existing drill hole data including, geological observations, analytical data and geotechnical data to form the District-wide Central Database. Two Data Entry Objects (DEOs) were constructed within this system to allow for the electronic capture of new data or re-logged drill hole data, both geological and geotechnical. The geological and analytical data derived from re-logging and re-assaying would not replace old data, but allow the best available data to be prioritized ahead of lesser quality data in any data extraction report.

Plans are underway to capture all historic drill logs and maps as PDF files. The scanning of the historic geologic logs representing 10,600 drill holes has been completed. These scanned files will be compared against the District-wide Central Database (acQuire database) files and any exceptions delineated. The scanning of the historic map database is 40% complete and is anticipated to be completed by Jan. 1st, 2010. The scanned historic maps will be geo-referenced and pertinent data such as original surface geologic contacts and structural observations selected from the images and compiled into a district GIS database.

As re-interpretation of geology, section by section, is completed on the respective drill holes, geologic codes are assigned back to each drill hole interval and will be imported into the acQuire database in a column called “As Interpreted”. This will allow for the systematic exporting of data consistent with the MineSite 3D Geologic and Ore Type models.

Re-evaluation of the district geology began in late 2005 with the contracting of MineMappers geological consultants to remap all pit exposures and nearby outcrops. In addition, all available quality drill holes were re-interpreted or re-logged to redefine the character of the lithology, alteration, mineralogy and metallurgy. Remapping of all surface exposures for the Tripp – Veteran mine through the Liberty and east through the Ruth deposit has been completed. Efforts are ongoing to combine these models into a coherent district three dimensional ore type and geologic model. This work has helped expand existing resources and also increase reserves.

Final translation and re-logging of historic drill hole logs and drill hole samples is ongoing, and is expected to be completed by midyear 2009. Detailed geologic mapping continues in the district within areas more distal from the existing resources. Ultimately, the three-dimensional geologic model will be of regional extent.

10.3 The New Concept Drill Program

Through the end of 2008 the NRMOP focused most resources on delineating and expanding the known ore reserves and resources of the Tripp – Veteran mine and Ruth deposit. True exploration drilling has been relatively minor. However, two conceptual targets have been tested recently: the Taylor mine target – a high grade skarn target north of the Tripp-Veteran pit; and the Giroux Wash target, a fault offset target to the south of Tripp-Veteran. Final assays and evaluation of the drill hole results are pending as of December 2008.

In the future, integration of the recent drilling and geochemical data with the historic and ongoing geologic database as well as the detailed aeromagnetic model will be the basis for developing new conceptual drill targets.

10.4 Geophysics

During the spring of 2008, the Quadra Exploration Group, contracting through Fugro Airborne Surveys Inc., flew a high resolution, detailed aeromagnetic survey over the Robinson district. The survey area was comprised of one block roughly 15 by 18 kilometers and covers most of the known prospects in the district. Flight lines were flown at 100 meter intervals in a north – south direction. Tie lines were flown perpendicular to flight lines. The survey encompassed approximately 2,826.8 line kilometers of traverse lines and 287.5 line kilometers of tie lines and was flown with a Eurocopter AS 350 helicopter.

In September 2008 an Induced Polarization (IP) test line, contracted through Zonge Geoscience (Reno, Nevada), was run over the eastern end of the Ruth Pit. The purpose of this line was to view the depth and extent of the sulfide system and also to determine if different sulfide species could be identified in the district using this method. Results are pending.

11. Drilling

11. Drilling

Well over 10,000 drill holes have been recorded in the Robinson district and the number of holes included in the current District Central Drill Hole Database, (DCDHD) is 9,651. Surface and underground drilling has been a steady process through much of the district history. Initial programs from the early 1900's through the 1950's were designed to define the high grade chalcocite enrichment blankets, and were largely completed with churn and minor core drilling. Later programs from the 1950's on had more core drilling and some standard rotary, but the churn drill was a common tool in the district up through the 1960's. Reverse circulation drilling was used almost exclusively in the 1970's through the 1990's, particularly in the search for gold.

11.1 Drilling and Re-Assaying by Quadra Mining Ltd.

Currently Quadra is using a mix of reverse circulation drilling and a lesser amount of core drilling. Reverse circulation is faster, cheaper and can give excellent assay information and adequate geologic information in many cases. Core is employed where needed for geotechnical information, metallurgical samples, and detailed geologic information. Preservation and re-assay of available historic drill samples from mostly pulps, but also including some coarse reject material, is part of the ongoing work to develop and extend the resource. Most of the drill holes selected for re-assay have poor quality data or are lacking information regarding gold and molybdenum.

11.1.1 Veteran Mine Drill Program

The New Robinson Mine Optimization Project (NRMOP) Drill Program has been designed to efficiently identify, test, and generate significant extensions to existing resources. The NRMOP Drill program also conducts metallurgical drilling, geotechnical drilling and condemnation drilling on planned dump areas. The NRMOP Drill Program to date has utilized reverse circulation, core and Becker type drill rigs. In addition, all drill holes are surveyed by standard gyroscopic downhole surveys and Colog conducted visual or acoustic televiewer logs of several of these holes.

Through 2006 to December 31, 2008, 113 reverse circulation holes have been completed in the Tripp – Veteran mine, comprising 124,036 feet of drilling. Twenty-three core holes have been completed in the Tripp – Veteran Mine, comprising 24,322 feet of drilling (Figure 11.1, and Table 11.1). These holes were planned and drilled to expand Quadra's knowledge of the geology and the extent of the ore body, to allow a better

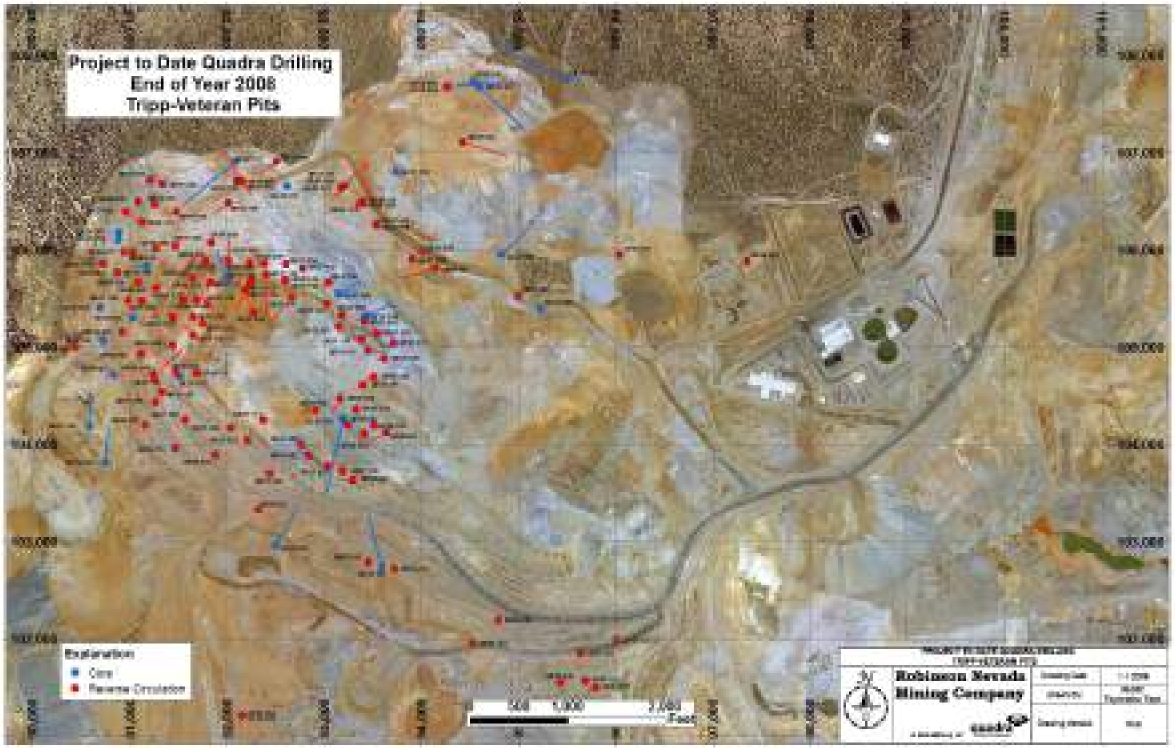


Figure 11.1: Tripp – Veteran Mine PTD Drilled Holes

Table 11.1:
Tripp- Veteran Mine Drilling

	Tripp Veteran Deposit	
	Footage	Holes
Reverse Circulation	124,036	113
Core	24,322	23
TOTAL	148,358	136

Tripp-Veteran								
	Geology		Metallurgy		Condemnation		Geotechnical	
	Footage	Holes	Footage	Holes	Footage	Holes	Footage	Holes
Reverse Circulation	118,232	107	5804	6	0	0	0	0
Core	20,605	18	0	0	0	0	3,718	5
TOTAL	138,837	125	5804	6	0	0	3,718	5

understanding of the metallurgical characteristics of the Tripp – Veteran ore, and the geotechnical constraints to be considered in pit highwall design.

11.1.2 Ruth Deposit Drill Program

To December 31, 2008, 126 reverse circulation holes have been completed in the Ruth deposit, comprising 142,489 feet of drilling; in addition 34 core holes totaling 39,448 feet have also been completed (Figure 11.2, Table 11.2). These holes were planned and drilled to expand Quadra's knowledge of the geology and the extent of the ore body and to improve the understanding of the metallurgical characteristics of the Ruth ore.

During July and August 2007, a dump drilling program was carried out in the Ruth area to determine if there might be economic value in the dumps currently burdening the pit design between the planned East Ruth pit and the planned Kimbley and Wedge pits (Figure 11.2 and Table 11.2). A total of 5,406 feet of drilling was completed in 39 holes for this program (Figure 11.2 and Table 11.2). The holes ranged in depth from 60 to 250 feet, with an average depth of 139 feet. Chalcopyrite and chalcocite were observed in nearly all of the 39 drill holes and some dumps contain substantial tonnages of low-grade sulfide material.

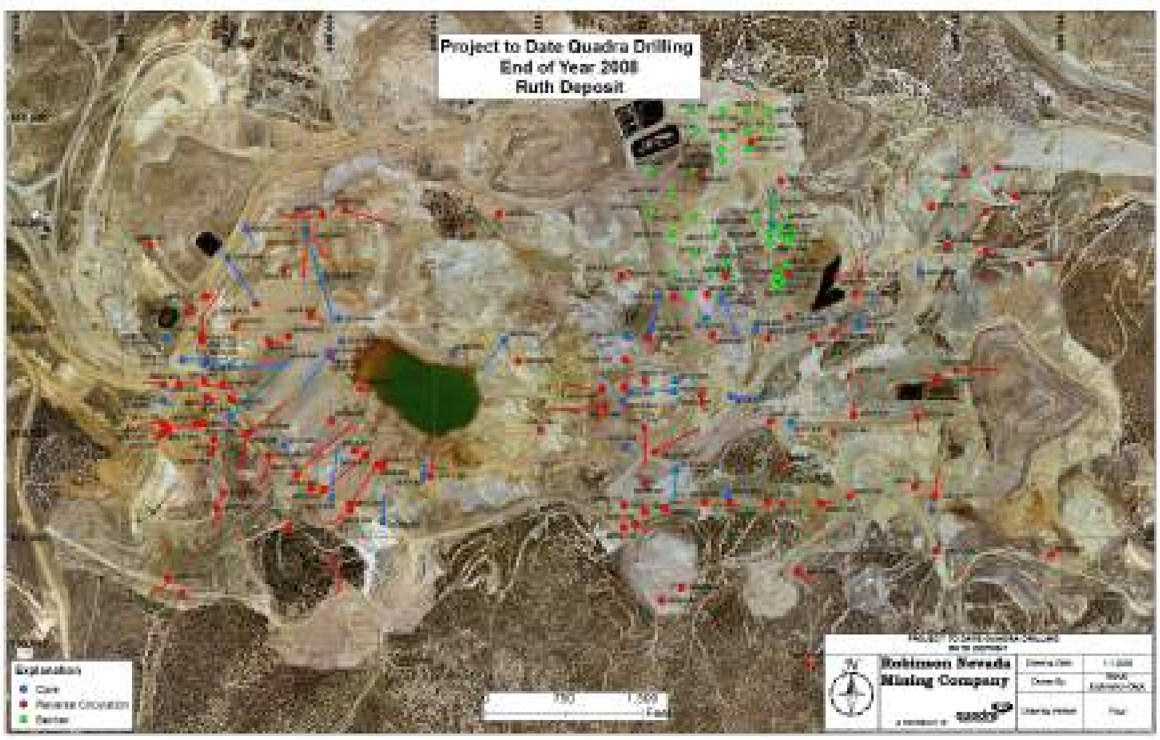


Figure 11.2: Ruth Deposit PTD Drilled Holes

**Table 11.2:
Ruth Deposit Drilling**

Ruth Deposit	Footage	Holes
Reverse Circulation	142,489	126
Core	39,448	34
Becker	5,406	39
TOTAL	187,343	199

	Geology		Metallurgy		Condemnation		Hydrologic		Geotechnical	
	Footage	Holes	Footage	Holes	Footage	Holes	Footage	Holes	Footage	Holes
Reverse Circulation	105,815	86	25474	30	7,280	5	3,920	5	0	0
Core	4,567	4	26982.5	22	0	0	0	0	7,898	8
Becker	5,406	39	0	0	0	0	0	0	0	0
TOTAL	115,788	129	52456.5	52	7,280	5	3,920	5	7,898	8

Project-to-date (December 31, 2008) drilling has included a total of 13 diamond drill core holes, drilled specifically for geotechnical purposes. Five of these holes were drilled at the Tripp-Veteran mine, comprising 3,718 feet of drilling (Figure 11.3 and Table 11.2); and eight were drilled at the Ruth deposit, comprising 7,898 feet of drilling (Figure 11.3 and Table 11.2). These holes were each logged with standard gyroscopic downhole survey techniques, and with Colog's visual or acoustic televiewer techniques to evaluate structural features in situ. The core and the televiewer logs were evaluated by Golder Associates, who then produced reports on highwall design recommendations. These holes were subsequently logged for geological information, but were not sampled or assayed.

11.1.3 Keystone Dump Drill Program

During June 2007, a drilling program was carried out on the Keystone dump to investigate the possibility that the dump might contain an economically viable mill-feed resource. Fifteen Becker hammer drill holes were completed from the surface of the Keystone dump down to native soil below the dump, comprising 2,557 feet of drilling. Hole depths ranged from 125 to 218 feet, with an average depth of 170 feet. Chalcopyrite and chalcocite were seen in all of the 15 drill holes and the dump does contain a substantial tonnage of low grade, mill feed material.

11.1.4 The District Sample Archive Re-Assay Program

Work began in 2005 to inventory and select drill holes for re-assay from the archives stored in buildings near the Deep Ruth headframe. Initial analysis comparisons indicated insignificant to no deterioration of sample quality while in storage. However, due to the

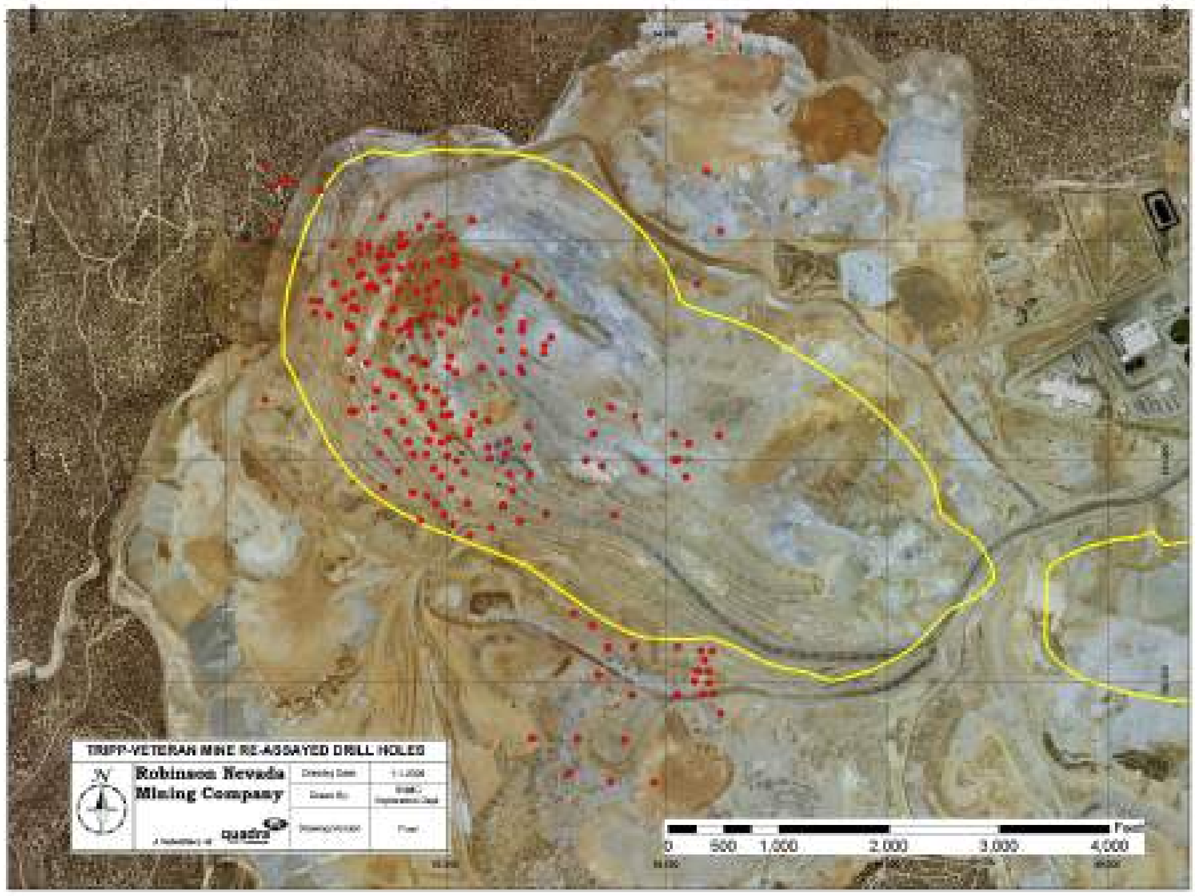


Figure 11.3: Tripp – Veteran Mine Re-assayed Drill Holes

significant vandalism and weather damage to the facility, it was decided that future work would require the complete reorganization and inventorying of the entire archive. Today, the entire contents of the Ruth headframe drill hole archives have been relocated, inventoried and stored in a new facility near the exploration offices. Final detailed inventorying of all core, pulps, and coarse rejects was nearing completion in December of 2008

To date, 1,047 district historic drill holes have been submitted for re-analysis utilizing 52 element ICP, Total Copper, Soluble Copper, Quick Leach Copper analysis, SAP analysis, molybdenum analysis, and a fire assay for gold.

Three-hundred-and-two drill holes were selected from the Tripp – Veteran mine area, and 723 drill holes have been selected from the Ruth deposit area (Figure 11.3) for re-analysis and included into the district-wide central database. Tripp-Veteran re-assay work is complete and final re-assay results from the Ruth deposit are expected to be completed by January of 2009. An additional 22 drill holes were selected from the Taylor mine area, north of the Veteran mine to assist in evaluating a district target area.

All historic drill hole pulps have been submitted to Minerals Exploration & Environmental Geochemistry (MEG), Washoe Valley, Nevada for re-pulverization to modern industry standards, insertion of commercial QA/QC standards and blind coding for submittal to ALS Chemex in Vancouver, BC for analysis.

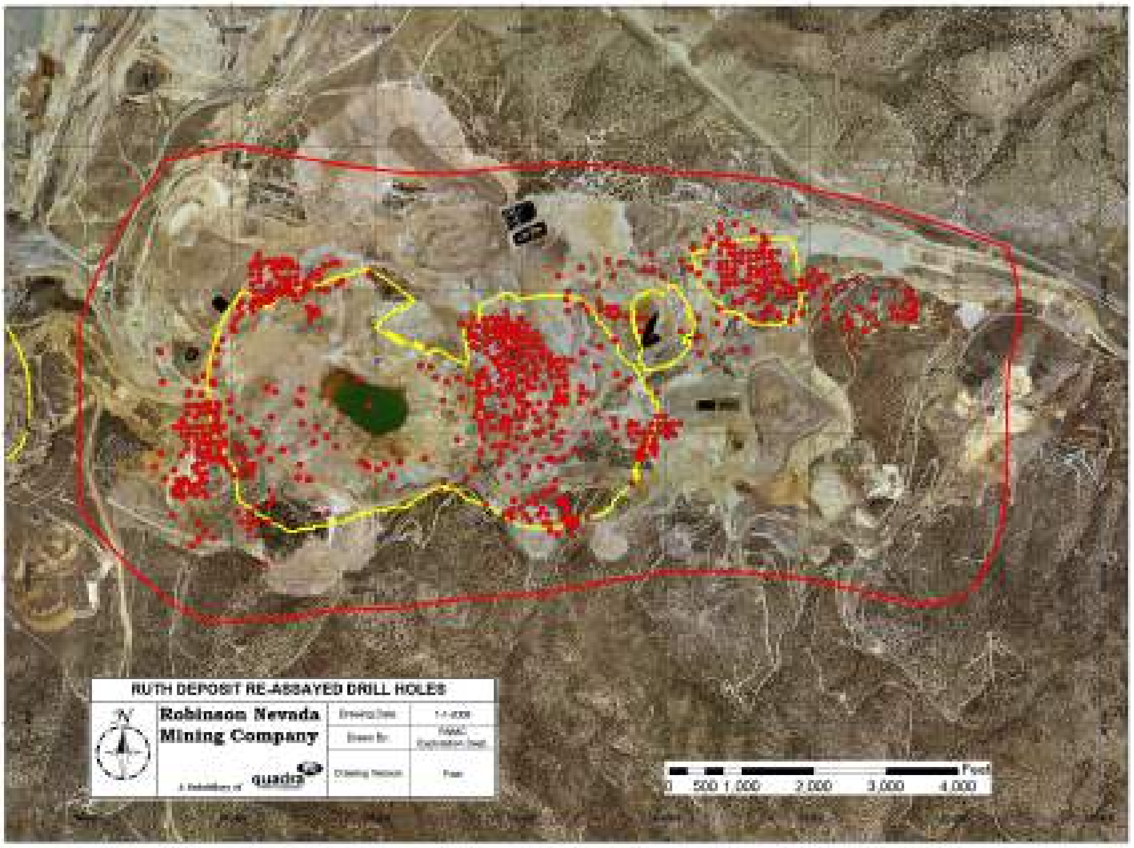


Figure 11.4: Ruth Deposit Re-assayed Drill Holes

12. Sampling Method and Approach

12. Sampling Methods and Approach

12.1 Historic Drill Sampling

Throughout the long history of the Robinson district, the ore body has been sampled by churn drilling, standard rotary drilling, reverse circulation rotary, Becker hammer drilling, core drilling, and probably other methods as well. Core holes have typically been sampled on geologic intervals with a maximum of five to ten feet for any interval. Most of the other methods were sampled on five foot, or in some cases, ten foot intervals. Sampling methods and details and sample preparation have not typically been documented as one might expect for such an old district. There may well be problems locally with the use of the old drilling data, which in some cases extends back to the very earliest years of copper mining. However, the long production history of the district and the overall reconciliation of production to the drill hole data indicates that the data is useable in the estimation of resources. Most of the very oldest drill holes are long since mined out and the current resources are largely based on drilling data of much more recent vintage.

Copper analyses are available for virtually all recorded drill holes, and re-assay tests on some holes suggest that this data is generally good with no significant bias (Chapter 14). Historical analytical information for molybdenum and gold is only available for some samples and is generally of much lower quality than for copper. This almost certainly reflects the importance of modern sample preparation methods as well as analytical methods.

12.2 Quadra Reverse Circulation Drill Sample Collection Procedures and Protocols

Samples from reverse circulation (RC) drilling are collected on five foot intervals by Layne Christensen Drilling using either a T-3 or Schramm 685 rig. Samples at the rig travel out of the drill hole and through Jones dry splitter or an adjustable Vezin rotary splitter, which is set to capture a larger or smaller percentage of the total volume of cuttings depending on sample recovery and groundwater flow. The splitter is leveled to work correctly and adjusted such that the sample can be captured in a five-gallon bucket (weight ranges from one to ten pounds of dried material). During dry drilling the splitter is cleared after each sample with a vibrator and air hose. The wet splitter is cleaned at every rod change. Drill rods are twenty feet long and the hole is blown clean with compressed air at the end of each rod length. Samples are caught by an assistant on five foot intervals and placed in a pre-labeled cloth sample bag. When no sample is recovered the bags are so marked.

A six or eight inch kitchen strainer is placed on top of the five-gallon bucket to capture a small volume of representative sample from each interval which is then used to fill chip trays for geologic logging. After chips are caught in the strainer they are initially shaken or rinsed over the splitter. Chips are then rinsed in relatively clean water and placed in the proper interval in the chip tray. Hand sorting of chips is not permitted, and when clay zones or alteration is encountered the clay is not washed out of the chips.

On a typical exploration drill hole, only one sample is taken per five foot interval using a cloth 11x17 inch bag. At the end of each 100 feet, an additional duplicate “field check” sample is collected. This sample is collected on the 0 to 5 foot interval, then the 95 to 100 foot sample and each succeeding x95 to x00 interval. The bag for this field duplicate sample has the same markings as its mate, but it is labeled with an “A” after the footage. When a hole is being drilled for dual purposes, such as exploration and metallurgy or exploration and environmental, two or three samples are collected per five foot interval. These samples are split out of the total cuttings return using the wet splitter in order to ensure that each sample is representative of the five foot interval.

Samples are generally picked up from the drill rigs twice per day by RNMC employees. Sample bags are laid out on site in correct order to simplify pick up and noting of missing samples. The duplicate at each 100 ft is placed in sequence with the remainder of the samples. When drilling wet in freezing temperatures plastic sheeting is used between layers of samples to avoid freezing together. The samples are transported back to the exploration office area where each sample bag is inventoried on a sample tracking sheet and placed on shelving in a heated storage container to dry. The storage container is locked at night.

After a hole has been completed, a sample toting sheet is prepared from the sample tracking sheet. RNMC employees then take the sample toting sheet and verify each sample is present, in numerical order, and labeled correctly as they place sample bags into re-usable plastic totes for shipment to ALS Chemex Laboratories. Clear plastic is used to line the bottom of the plastic tote, and a layer of clear plastic is placed between layers of sample bags. If more than one hole is included in a single tote, black plastic is placed between samples from the two holes. The completed sample toting sheet is returned to the office, and used to prepare a sample dispatch which will accompany the samples when they are shipped offsite to ALS Chemex.

When a sample tote is full, the lid is placed on it and a white square is spray painted onto the lid for labeling. If multiple holes are contained in the tote, all hole IDs are written on the tote lid. If a single hole fills more than one tote, the hole ID is written on each tote, along with the number of totes that hole is in (i.e. 1 of 2). Partial holes are not shipped to the laboratory. ALS Chemex picks up totes of samples and returns totes of sample rejects on a regular basis, as scheduled by RNMC.

12.3 Quadra Core Drilling and Sampling

Core drilling was done by Ruen Drilling of Clark Fork Idaho from 2006-2007 using a 1997 Christensen model CS 1500 drill. During 2008, Tona Tec Exploration, LLC of West Mapleton Utah was contracted for the drilling. They used a Boart-Longyear LF-90. Core was drilled using HQ and NQ sizes. Reducing from HQ to NQ took place only when problem areas were encountered or the hole was too deep to further lift the HQ rods. A five foot core barrel was used.

As the core is drilled, it is placed in core boxes at the drill rig. Each box is labeled with the project ID, the company name, the hole ID, and from-to depths of the core that box holds. Depths are measured and recorded by the drillers to the nearest tenth of a foot. Each drill run is separated by a block of wood labeled with the feet drilled and the feet recovered, or labeled as NR in intervals of no recovery. Mis-latches are recorded on similar blocks of wood placed in the core boxes.

Core is picked up from the drill rig twice per day by RNMC employees and is transported back to the exploration office. Core is stored inside until it has been logged and photographed. The geologist logging the core selects intervals to be sampled and enters them into the AcQuire database; then the geologist marks these intervals on the core. Sample intervals for assay are typically between six inches and five feet. The logging Geologist also identifies core to be collected for density grab samples. Samples do not cross lithologic contacts. Each box of core is photographed by RNMC employees before collecting density samples every 50 feet. Finally, a sample dispatch is prepared, and core is shipped to ALS Chemex Laboratories to be prepared for assay.

12.4 Becker Hammer Drill Sample Collection

Specific drilling at Keystone dump and dumps in the Ruth area was done with a Becker hammer drill, devised especially for drilling in unconsolidated materials such as sand and gravel deposits or mine dumps. The Becker drill utilizes a diesel powered pile hammer to drive a special double walled casing into the ground. The casing does not rotate. As the casing is driven into the ground, broken rock fragments and cuttings are returned by high pressure air which is pumped down between the casing walls and then returned up the center of the pipe. As the Becker rig drilled, the full volume of material taken out of the hole was collected by the drillers in a wheelbarrow. At the end of each five foot interval, the wheelbarrow was emptied on a ten foot square piece of HDPE liner. For the

Keystone Dump drilling program, the entire volume of the sample was shoveled into 5-gallon buckets, while a geologist at the drill rig logged the interval. For the dump drilling in the Ruth Deposit area, RNMC employees quartered the sample, then used a plastic spoon to fill a chip tray, taking one spoonful of material from each quadrant in turn. RNMC employees then switched to a shovel, and filled a sample bag a fourth of a shovelful at a time, again collecting the sample from each quadrant in turn. The sample bag was labeled in the same manner as Reverse Circulation samples. At the end of each 100 feet an additional duplicate "field check" sample bag is collected and labeled as appropriate. The remaining volume of sample was collected into one or more five-gallon buckets, for compositing and metallurgical analysis. At the end of each hole, and at the end of the day if a hole was not completed in one day, RNMC employees transported the sample bags back to the exploration office area where each sample bag was inventoried on a sample tracking sheet and placed on shelving in a heated storage container to dry. The storage container was locked at night.

For both dump drilling programs, the five-gallon buckets were labeled with permanent marker on the side and on the lid. Also, aluminum tags were scribed with the Hole ID, sample interval, and number of buckets in that interval and attached to the bucket handles. After collecting the sample from each interval, an RNMC employee swept the HDPE liner with a broom, and then with a dust mop, before the next interval was brought over.

For the Keystone dump drilling program, samples were handled and analyzed internally by RNMC's onsite lab. For the dump drilling located in the Ruth deposit area, samples were managed in the same manner as RC samples, and analyzed by American Assay Laboratories, which was doing all of the analytical work for the exploration department at that time.

13. Sample Preparation, Analyses and Security

13. Sample Preparation, Analyses and Security

13.1 Historical Database Summary

Prior to Alta Gold's involvement in the district in the 1980's, it appears that all of the drill sample assaying was performed by company laboratories either on-site or at the McGill facility. Since BHP-Magma's (Magma) acquisition of the district, the majority of drill samples have been assayed at commercial laboratories, including Bondar-Clegg, Monitor Labs, American Assay Laboratories (AAL), Rocky Mountain Geochemical, and ALS-Chemex Labs (ALS).

No written assay laboratory procedures could be located at the Robinson mine site. Magma personnel suggest no special sample preparation or assaying procedures were done for the drill samples when Magma operated the property (E. Seedorff, personal communication). Mine Development Associates (MDA) contacted AAL and confirmed that only 'standard' sample preparation and assaying procedures were performed on the Robinson drill samples. Other than blast hole sampling, MDA could find no documentation for drill hole sampling and assaying procedures. Likewise no procedures outlining sample security were found.

13.1.1 Historical Database Check Assays/Blank Samples

An internal company report, Robinson Project, Internal Reserve Report, FY2000, (Kliche, Knight, & Stevermer), describes check assaying and 'round-robin' procedures that were implemented by Magma. However, MDA could not locate any of the raw data or comparative data to evaluate. Numerous drill log folders contain more than one set of assays for the same holes, predominantly for holes drilled by Kennecott. A brief comparison of several holes showed only minor differences in total copper for a small percentage of sample intervals. However, this was not a large enough sample to be considered applicable to the entire dataset. In instances where there is more than one total copper assay, the original assay is posted in the database; multiple assays were not averaged to derive the final number used for modeling.

For the post-Alta Gold drilling, gold was analyzed by atomic absorption (AA) and often re-assayed by fire assay. More often than not, the fire assay value, if available, is used for estimation rather than the AA assay result. Again, multiple analytical results were not averaged to derive the final gold value.

MDA found no evidence of blank samples submitted to any lab from exploration/in-fill drilling programs. Ex-Magma personnel have verified that there was no blank sample program during their tenure in the district (E. Seedorff, personal communication).

13.2 Quadra Era Drill Hole Program and Re-Assay Program Summary

In August 2006, Quadra Mining Company initiated a new district-wide drill program and a re-assay program of historical archived district drill hole sample pulps, rejects, and core. Initial analysis was conducted utilizing AAL, Reno, NV and initial samples submitted utilized previously developed internally derived QA/QC standards, sourced from Robinson Nevada Mining Company from production drill holes within the Veteran Pit. Evaluation of QA/QC results and procedures and protocols by exploration staff in early 2007, aided by Dr. Jeffrey Jaacks, Geochemical Applications Intl. Inc., Centennial, Colorado, indentified the need to change to implementation of commercial standards, modify contractual analytical procedures and techniques, modify application of standards procedures and protocols, QA/QC monitoring procedures, and consequently analytical laboratories. In addition, all previously submitted Quadra series drill hole pulp samples whose AAL analysis indicated total copper grades greater then 1,000 ppm Cu, (one-half the cutoff grade used for modeling at RNMC) were resubmitted for analysis utilizing the new procedures and protocols.

ALS-Chemex, Reno, NV was chosen as the primary lab and Skyline Labs, Tucson, AZ as the secondary lab. Minerals Environmental Geochemical Lab, Washoe Valley, NV was engaged to provide sample preparation of historical archived sample material, QA/QC insertion and sequencing and blinding of samples on all samples submitted to ALS-Chemex. ALS-Chemex is an ISO 9001:2000 certified institution; the work conducted by Skyline Labs was performed and supervised by Arizona State Registered Assayers. The ALS-Chemex data are of acceptable quality for resource estimation.

Sample preparation and security for Quadra era drill samples was adequate for resource and reserve estimation. Historic samples were unmonitored for many years before Quadra acquired the property, but there is no reason to believe there were security problems prior to Quadra's acquisition of the project in 2004. No Quadra employee, officer, director, or associate was involved in any aspect of the sample preparation.

13.3 American Assay Laboratories Analyses

All 2006 to 2008 sample values in the resource estimate with greater than 1,000 ppm total copper were analyzed by ALS-Chemex with re-prepared pulps and acceptable standards. In that light, it serves no purpose to discuss the preparation, analysis and QA/QC of samples from American Assay.

13.4 ALS-Chemex Sample Preparation

During the course of the 2006 to 2008 drill programs, three types of samples were sent into three laboratories for sample preparation and analyses. This included drill core and reverse circulation samples collected from the current drilling programs and drill sample pulps assembled from company archives. In the original 2006-2007 program, the samples were sent to AAL in Reno. Starting in the fall of 2007, samples were sent to ALS-Chemex and to Mineral Exploration Geochemistry (MEG) laboratories for sample preparation and analysis. Drill core and reverse circulation cuttings were sent for initial preparation to ALS-Chemex at the Winnemucca sample preparation facility, and then forwarded to Minerals Exploration Geochemistry to be re-numbered and have blinded standards and blanks inserted into the sample stream of analytical pulps. After these blinded quality control samples were inserted into the sample stream, the re-numbered samples were re-submitted as a new job to ALS-Chemex for analysis.

Drill core and drill cuttings were sent to ALS-Chemex on shrink-wrapped pallets containing no more than 48 boxes of core, organized from collar to TD and labeled with the drill number and footage of the interval. Archival pulps were sent in boxes organized by drill hole and interval.

ALS-Chemex sawed the diamond drill core in half as per individual core hole sampling sheet directives from Robinson Nevada Mining Company (RNMC). The core was marked by Robinson Nevada Mining Company with each sample interval and a sawing guide line.

Drill core and cuttings were dried at 50ºC. Core and reverse circulation samples were crushed in a Boyd Crusher Rotary Splitter Device Combo jaw crusher to obtain a 70% passing 10 mesh (2 mm) crush sample. Ten percent of these samples were sieve-tested for compliance. Clean silica sand was used to clean the crusher between each sample. A 250g sub-sample was rotary split from the sample and pulverized to 85% passing 200 mesh (75 µm) for analysis using a Labtech LM-2 Pulverizer. Analytical pulps were forwarded to Minerals Exploration Geochemistry (in Reno) in pulp boxes, where MEG inserted blind blanks, standards, and duplicates and re-numbered the pulps. These pulps were then resubmitted to ALS-Chemex for analysis. The reject was bagged, placed onto a pallet and shrink-wrapped for transport back to RNMC.

Archival pulp samples were selected from the drill archives and sent to Minerals Exploration Geochemistry in Reno for sample preparation. These pulps were dried and re-pulverized to 85% passing 200 mesh. MEG inserted blind blanks, standards, and duplicates and numbered the pulps. These pulps were then submitted to ALS-Chemex for analysis.

13.5 ALS-Chemex Analytical Program

All samples were analyzed at ALS-Chemex using the following protocol:

1. *All Samples* - 0.25g - 4-acid digestion/ICP-AES/MS (HNO_3/HCl/HF/$HClO_4$) digestion with ICP-ES Finish for 48 elements (ALSC Method ME-MS61).
2. *All Samples* - 30g - Fire Assay with ICPAES Finish for low level (0.001-10 ppm) Au (ALSC Method Au-ICP21).
3. *For Samples with Au > 0.5 ppm* - 30g Fire Assay digestion with gravimetric finish for (0.05-10,000 ppm) Au (ALSC Method Au-GRA21).
4. *For Samples with Cu > 1,000 ppm (0.10%)*:
   a. *Total Cu* - 0.4g - 4-acid digestion with ICP Finish (ALSC Method Cu-OG62).
   b. *Acid Soluble Copper* - 0.25g - 15% H_2SO_4 digestion with atomic adsorption finish. (ALSC Method Cu-AA05q).
   c. *Cu Quick Leach Test (QLT)* - 1g - $H_2SO_4 + Fe_2(SO_4)_3$ digestion with atomic adsorption finish (ALSC Method Cu-AA08q).
   d. *Cu Hot Quick Leach Test (SAP)* - 0.25g - Hot $H_2SO_4 + Fe_2(SO_4)_3$ digestion with atomic adsorption finish (AAL ALSC Method Cu-AA08hq).
5. *For Samples with Mo > 50 ppm (0.005%):*
   a. Total Mo – 0.4g - 4 Acid Digestion with ICP Finish (ALSC Method Mo-OG62).

13.6 RNMC QAQC Protocol for ALS-Chemex Analyses

Five different certified reference materials and a blank of varying Cu and Au concentrations were obtained from CDN Resource Laboratories Ltd., in Delta, B.C. and were inserted in rotation at a rate of 2/20 samples, or 10%, including a lower grade and a higher grade sample within each group of 20 samples. In addition, every group of 20 samples included a field duplicate, increasing the overall percentage of quality control samples to 15%. As the project progressed, selected reference materials were introduced as the original reference material supply became exhausted. To date, the quality control database contains 400 to 2,500 determinations for each standard analyzed from a period between September 2007 and December 2008.

Check analyses are underway at Skyline Laboratories in Tucson, Arizona and need to be evaluated upon receipt.

13.7 Quality Control Acceptance Criteria

In reviewing the quality control information for all laboratories, the following criteria were used:

- Standard analyses measure accuracy and potential bias and should be within the mean ±2 standard deviations as determined and stated in certificate for the certified reference material. These values are listed in Table 13.1 for the standards used in this program. If any significant bias is observed using a particular lab and/or analytical method, the mean and standard deviation can be re-determined from the history of analyses for that particular method. At the 95% confidence interval, less than 5% of the analyses should exceed the certificate mean ±2 standard deviations. Certificate values are determined from total extraction methods. Therefore, any method which is not a total extraction method could result in a significant bias in the quality control results. Any sample that exceeds the mean ±2 standard deviations is outside of the acceptable limits and is classified as a failure.
- Blank analyses measure sample preparation contamination and should be within 5 times of the detection limit. Any sample that exceeds 5 times the detection limit is outside of the acceptable limits and is classified as a failure.
- Duplicate field sample analyses measure sampling reproducibility and sample duplicates should have a precision of 15%. Some reviewers state that the acceptance criteria for this level of precision should be 10%. This author disagrees with this "tight" or level of precision for sample preparation and has yet to find an example of sampling precision equal to or less than 10% to be achievable given the current level of sample preparation protocols. However, one could expect that 95% of the duplicate analyses should be within 15% of one another. Any sample duplicate pair that exceeds 15% precision is outside of the acceptable limits and is classified as a failure.
- Duplicate pulp analyses measure analytical reproducibility and should have a precision of 10%. Thus, one would expect that 95% of the pulp analyses are within 10% of each other. Conversations with ALS-Chemex quality control personnel reveal that the lab is generally able to obtain a precision of no better than 7% for duplicate analyses on pulp materials. This precision is derived from analyses on tens of thousands of pulp or analytical duplicates monitored over a period of years. Any analytical duplicate pair that exceeds 10% precision is outside of the acceptable limits and is classified as a failure.

- Check analyses are a measure of inter-lab reproducibility and are used as a verification of the original analytical data. Check analyses on pulps should exhibit better than 10% precision. This, of course, assumes that the laboratories are using the same analytical digestion protocol. The sample digestion protocol should be the same if no bias is to be observed. Any "pulp" check analysis duplicate pair that exceeds 10% precision is outside of the acceptable limits and is classified as a failure.

13.8 Certified Reference Materials

As previously mentioned, five certified reference materials were used in the 2006-2008 drill program (Table 13.1). These standards (CGS-08, CGS-12, CGS-16, CGS-16, and CM-01) were obtained from CDN Resource Laboratories Ltd., in Delta, B.C. The standards were prepared under the supervision of Dr. Barry Smee, by using bulk ore materials which were dried, crushed, pulverized, and then passed through a 200 mesh screen. The +200 material was discarded. The -200 material was blended for 5 days in a rotary mixer. After internal assaying to test for homogeneity, splits were taken and sent to 12 laboratories for round robin assaying using total digestion (4-acid for selected elements or 30g Fire Assay for gold) with an ICP or AA finish. The means and standard deviation values determined from round robin tests are given in Table 13.1 for each standard.

Table 13.1:
Certificate Analytical Values for CDN Standards

	Certified Values (mean ± 2 std dev)		
Standards	**Cu (ppm)**	**Au (g/t)**	**Mo (ppm)**
CDN-CGS-08	1050 ± 80	0.080 ± 0.012	
CDN-CGS-12	2650 ± 150	0.290 ± 0.040	
CDN-CGS-16	1120 ± 50	0.140 ± 0.046	
CDN-CGS-18	3190 ± 150	0.297 ± 0.040	
CDN-CM-01	8530 ± 200	1.850 ± 0.160	760 ± 80

13.9 ALS-Chemex – Internal QAQC Protocol

ALS-Chemex utilizes a QA/QC procedure which includes the placement of 2 standards, 3 replicates and one blank for each analytical batch of 84 samples in the Au-ICP21 method. The fire assay racks are capable of firing 84 samples within a given batch. The multielement – total digestion ME-MS-61 method utilizes an analytical protocol which includes the placement of 2 standards, 1 replicate and one blank for each analytical batch of 40 samples in the ME-MS61 method. Any quality control samples not meeting ALS-Chemex acceptability criteria triggers check analyses of selected samples within the batch.

13.10 Blanks at ALS-Chemex

As previously discussed, the field blank material is composed of "barren" limestone collected on the RNMC property. This blank contains low concentrations of copper (5.7 ppm) and molybdenum (1.8 ppm) (Table 13.2). Ninety-nine percent of the copper analyses exceed the 5 times detection limit threshold. Forty-two percent of the molybdenum analyses exceeded the 5 times detection limit threshold. Further use of this material as a blank is not recommended.

Table 13.2:
Summary Statistics for Blanks at ALS-Chemex

Blank Statistics			
Element	Au (ppm)	Cu (ppm)	Mo (ppm)
Method	*ICP-21*	*ME-MS61*	*ME-MS61*
Finish	*ICP*	*ICP*	*ICP*
Detection Limit =	0.001	0.2	0.05
Count =	3639	3626	3643
Min =	0.001	0.2	0.05
Max =	2.2300	107.0	777
Range =	2.2300	107.0	777
Mean =	0.0076	5.7	1.8
Median =	0.0030	3.6	0.2
Std Dev =	0.0796	7.0	31.9
Variance =	0.0063	49.4	1,018.4
Coeff of Variance =	10.4738	1.2	17.464
Standard Error =	0.0013	0.1	0.53
% RSD =	17.3625	2.0	28.93
Acceptability Criteria			
Statistic	Au (ppm)	Cu (ppm)	Mo (ppm)
xDL =	5	5	5
Warning Threshold =	0.005	1.0	0.25
# > 5x Detection Limit	706	3592	*1518*
% > 5x Detection Limit	19.4	99.1	*41.7*

13.11 Standards at ALS-Chemex

Four commercial copper-gold standards (CG prefix) and one copper-molybdenum standard (CM prefix) were used during the ALS-Chemex analytical program. Reference sample statistics for this portion of the RNMC database are presented in Table 13.3.

Table 13.3 includes the certificate values for each standard for gold, copper and molybdenum. There are between 400 and 2,600 determinations for each reference material. Statistics from current laboratory analyses are tabulated by element and analytical method. Total counts, as well as calculated means and standard deviations are included for each standard. The calculated means show no statistically significant difference from the means determined from the original round robin analyses of the certificate, with the exception of gold determinations for CGS-08, which contains 80 ppb gold. The standard deviations calculated from ALS-Chemex analyses are 4 to 7 times the original certificate standard deviations, reflecting a larger degree of variation as the number of determinations is increased.

Analytical bias ranges from -4.4 to 4.8 % depending upon the individual standard and the type of analytical method. One exception to this is the 30% bias for the low grade CGS-08 gold standard.

The OG-62 method uses a larger sample weight for the digestion; 0.4 grams as opposed to the 0.1 gram used for the MS-61 analytical method. The bias and the failure rates decrease with the increased sample weights. This is consistent with a larger sample digestion weight often resulting in a more accurate and precise determination.

Failure rates, or percent of samples outside the acceptable limits, are indicated in the far right column of Table 13.3. There is a dramatic improvement in the failure rates at ALS-Chemex using the ICP-21, MS-61 and OG-62 analytical methods when compared to the 2A method used at AAL. Once again, at the 95% confidence limit, one could expect that 5/100 standards would fail to pass the certificate mean ± 2 standard deviation acceptability criteria. This is the equivalent of a 5% failure rate. The calculated rates of the table range from 0.4 % to 49 %.

The OG-62 rates are generally improved over the MS-61 rates, particularly for the higher grade standards above 1,000 ppm. The failure rates for CGS-12, CGS-18 and CM-01 are within acceptable criteria for copper by the OG-62 method and molybdenum by the MS-61 and OG-62 analytical methods.

Table 13.3:
Reference Sample Statistics for analyses at ALS-Chemex Laboratory

Au (ppm)	Certificate Value (Mean + 2 SD)	Method	Count	Mean	Std Dev	% Mean Bias	% Outside Limit
CDN-CGS-08	0.080 + 0.012	ICP21	807	0.104	0.047	30.1	42.4
CDN-CGS-12	0.290 ± 0.040	ICP21	2142	0.290	0.102	0.0	7.7
CDN-CGS-16	0.140 ± 0.046	ICP21	1630	0.144	0.057	2.9	6.6
CDN-CGS-18	0.297 ± 0.040	ICP21	356	0.293	0.094	-1.3	26.2
CDN-CM-01	1.850 ± 0.160	ICP21	2147	1.830	0.24	-1.1	19.6
Cu (ppm)							
CDN-CGS-08	1050 + 80	MS61	798	1078	70	2.7	22.7
CDN-CGS-12	2650 ± 150	MS61	2133	2542	172	-4.1	5.0
CDN-CGS-16	1120 ± 50	MS61	1635	1137	95	1.5	11.4
CDN-CGS-18	3190 + 150	MS61	356	3141	304	-1.5	23.0
CDN-CM-01	8530 ± 200	MS61	2147	8151	758	-4.4	49.0
CDN-CGS-08	1050 + 80	OG-62	717	1100	50	4.8	29.4
CDN-CGS-12	2650 ± 150	OG-62	2133	2680	380	1.1	1.9
CDN-CGS-16	1120 ± 50	OG-62	1614	1150	80	2.7	3.2
CDN-CGS-18	3190 + 150	OG-62	356	3220	300	0.9	0.5
CDN-CM-01	8530 ± 200	OG-62	2109	8480	520	-0.6	4.7
Mo (ppm)							
CDN-CM-01	760 ± 80	MS61	2146	733	74	-3.6	1.2
		OG-62	2109	760	40	0.0	0.4

13.11.1 CGS-08

Control charts for the CGS-08 are presented in Figures 13.1 to 13.3. Gold analysis by the ICP-21 method is presented in Figure 13.1. Accuracy and precision for these analysis fails in selected batches after sequence number 450 (Figure 13.1, around January 15th, 2007). Forty-two percent of the standards are outside the acceptable error limits and biased to a higher grade. This is the reason for the 30% mean bias. Analyses prior to sequence number 450 show acceptable accuracy and precision.

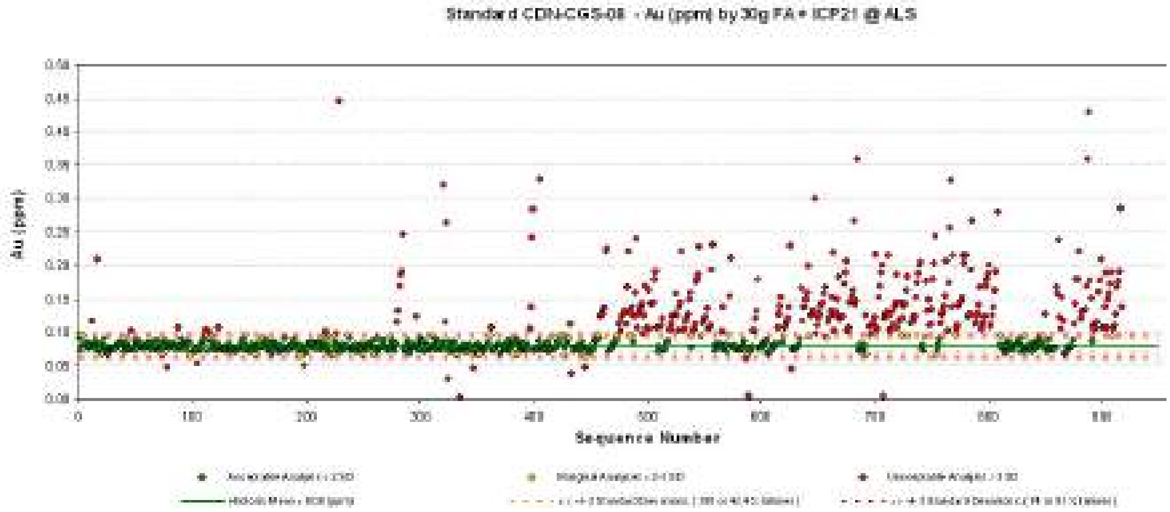


Figure 13.1: CGS-08 Au by 30g FA and ICP Finish (Method ICP21) at ALS-Chemex

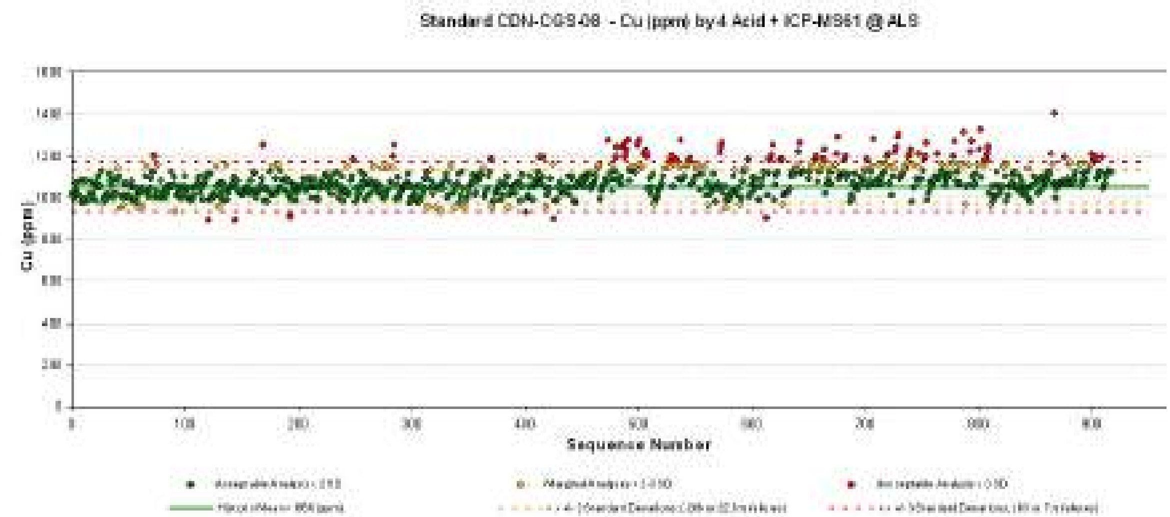


Figure 13.2: CGS-08 Cu by 4-Acid and ICP Finish (Method MS61) at ALS-Chemex

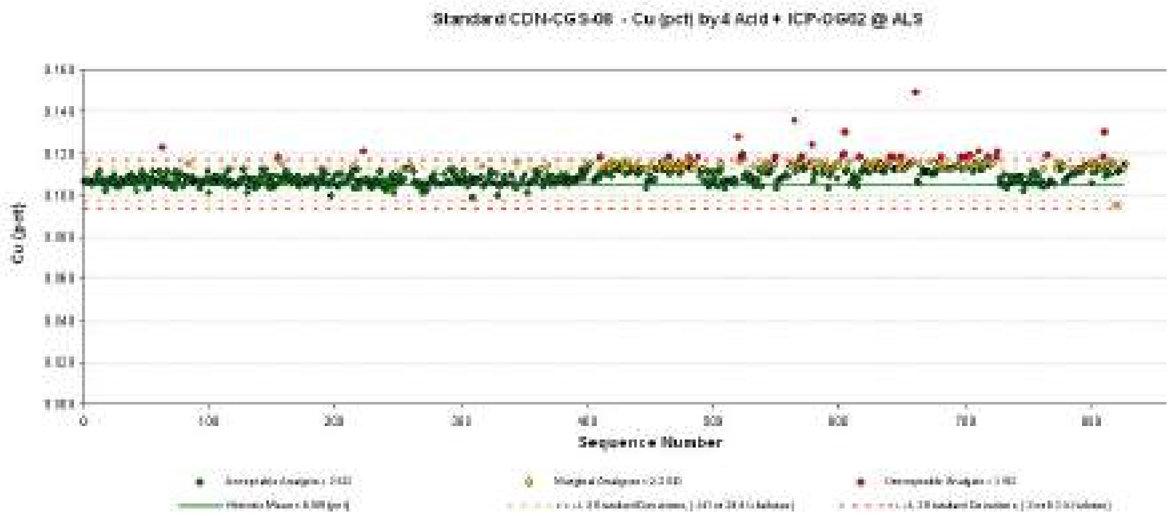


Figure 13.3: CGS-08 Cu by 4-Acid and ICP Finish (Method OG62) at ALS-Chemex

Copper by the MS61 method (screening analysis) is presented in Figure 13.2. Copper by the OG62 (high grade analysis) is shown in Figure 13.3. All samples with copper greater than 1,000 ppm are analyzed by both methods, as they are first screened using the MS61 method and then analyzed by the ore grade method (OG-62) if copper exceeds 1,000 ppm. Both plots show the same features. Until mid-sequence (sequence number 400-500), the determinations display good accuracy and precision. After sequence number 500 in Figure 13.2 and sequence number 400 in Figure 13. 3, the precision starts to open up (increase) and the accuracy starts to gently drift upwards. The trend of the upward drift is still within the acceptable limits of the reference material but a greater proportion of the analyses start to exceed the acceptable limits. The mean for the MS-61 data increases to 1,078 ppm and to 1,100 ppm for the OG-62 analyses, reflecting this upward drift.

13.11.2 CGS-12

Control charts for the CGS-12 are presented in Figures 13.4 to 13.6. Gold concentration of this standard is 290 ppb, which from the authors experience, is the lower end of obtainable accuracy and precision for gold analyses using this method. Copper concentration from the certificate is 2,650 ppm. Less than eight percent of the gold analyses, 5.0% of the copper by MS-61 analyses (Figure 13.5), and 1.9% of the copper by OG-62 analyses (Figure 13.6) exceed the outside limits. Gold and copper both show acceptable accuracy and precision at these concentration levels. There is no evidence of upward analytical drift observed earlier.

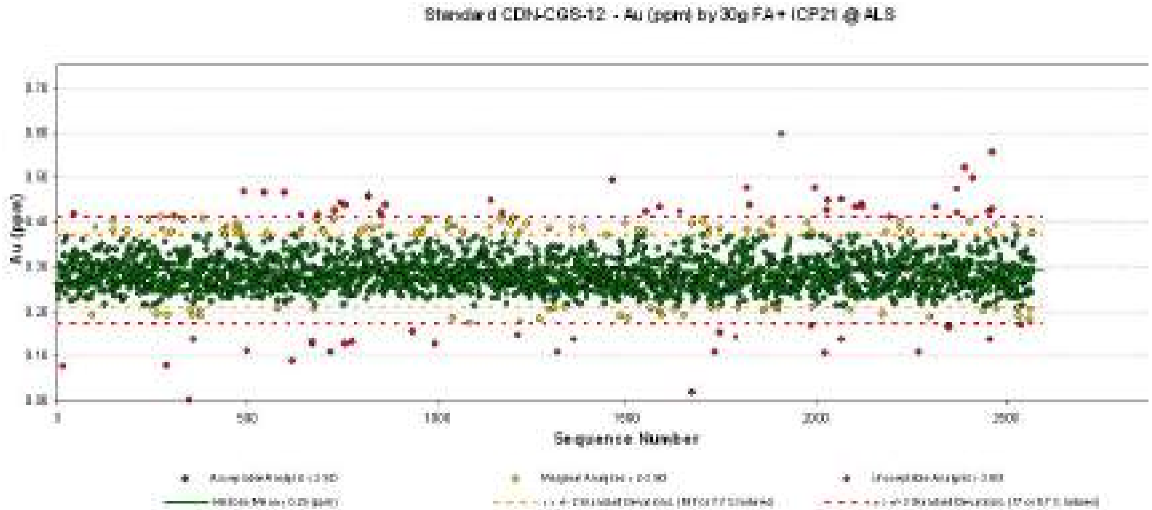


Figure 13.4: CGS-12 Au by 30g FA and ICP Finish (Method ICP21) at ALS-Chemex

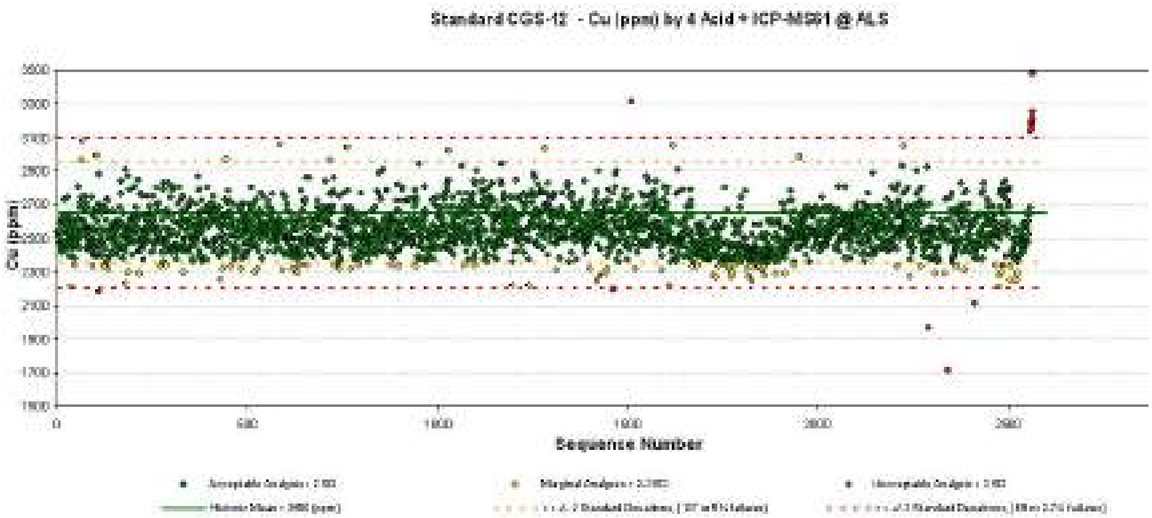


Figure 13.5: CGS-12 Cu by 4-Acid and ICP Finish (Method MS61) at ALS-Chemex

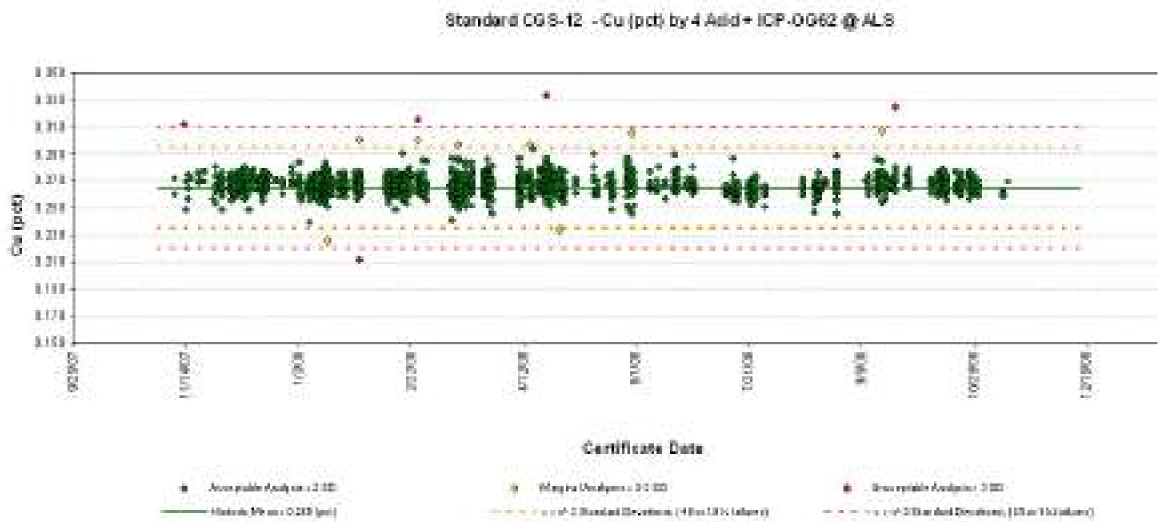


Figure 13.6: CGS-12 Cu by 4-Acid and ICP Finish (Method OG62) at ALS-Chemex

13.11.3 CGS-16

Control charts for the CGS-16 are presented in Figures 13.7 to 13.9. Gold concentration of this standard is 140 ppb. Under 6% of the analyses fall outside of the acceptable limits. A majority of these outlier analyses cluster about the 300 ppb range and indicate a set of possible misclassified standards as both CGS-08 and CGS-18 have gold concentrations in that range (Figure 13.7). If the analyses within the range of values for both of those standards are removed, only 2.3 % of the analyses fall outside the acceptable bounds and the mean bias decreases from 6.6% to 2.1 %.

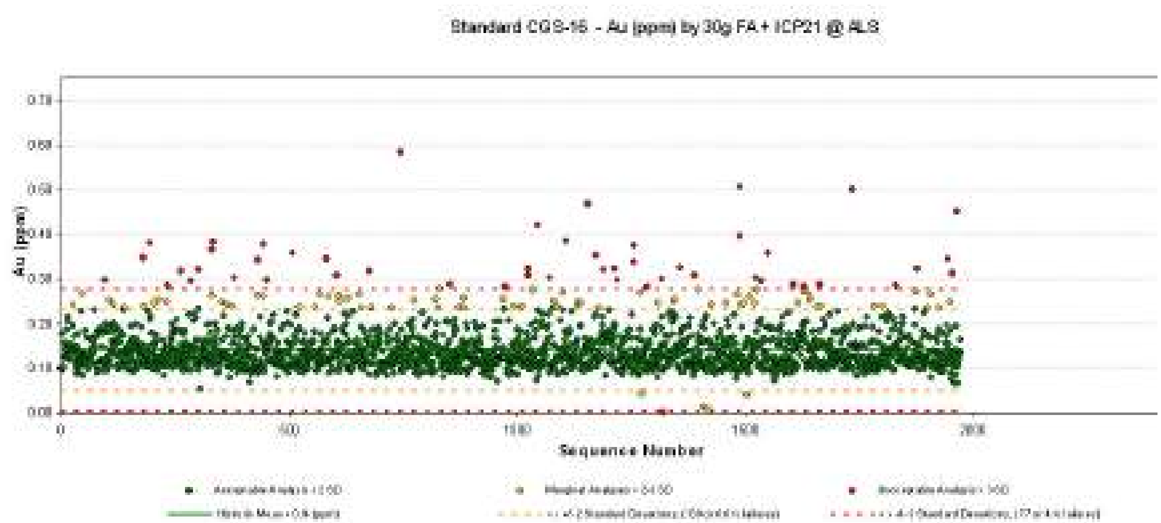


Figure 13.7: CGS-16 Au by 30g FA and ICP Finish (Method ICP21) at ALS-Chemex

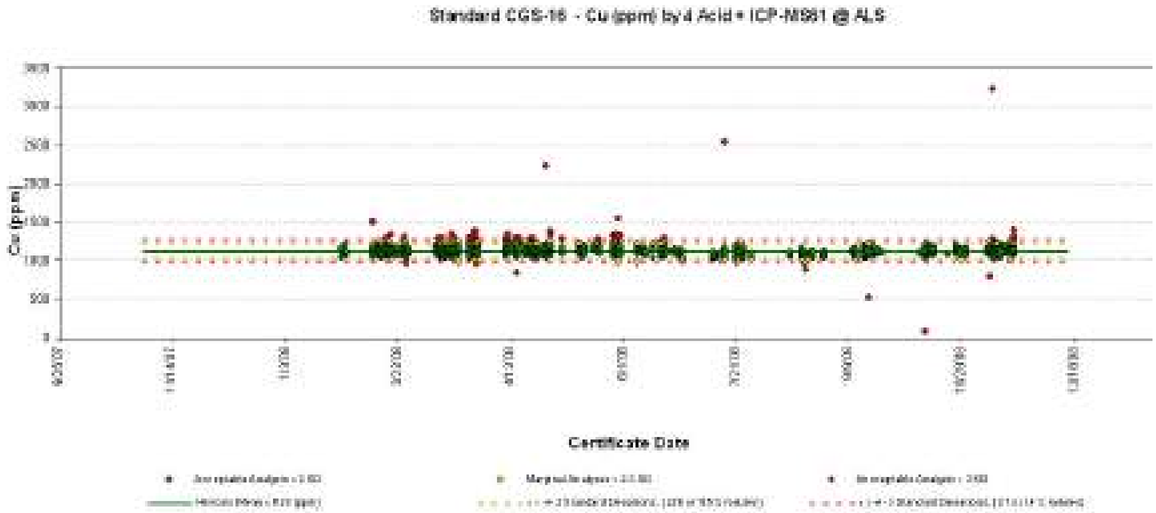


Figure 13.8: CGS-16 Cu by 4-Acid and ICP Finish (Method MS61) at ALS-Chemex

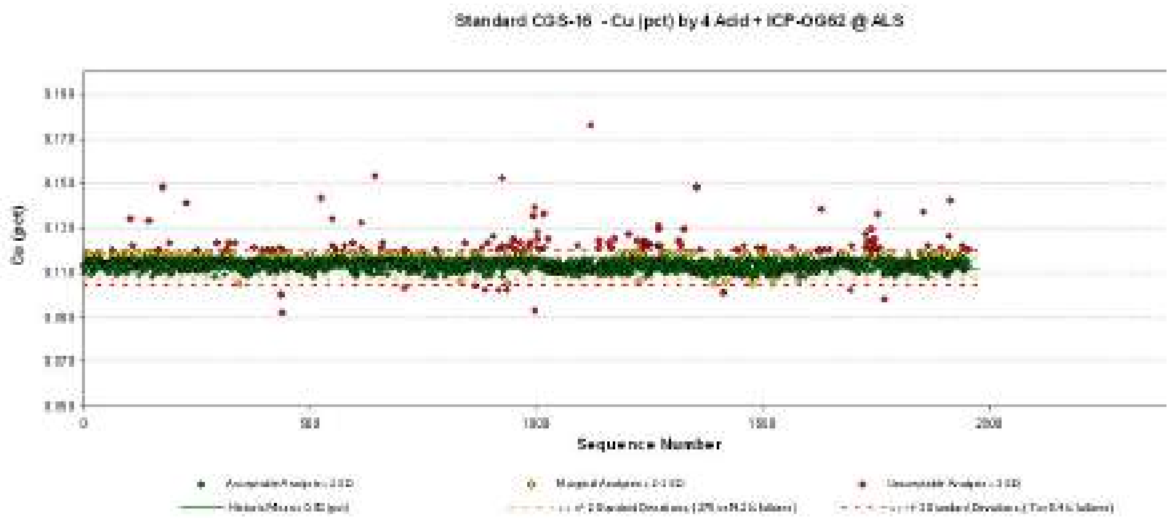


Figure 13.9: CGS-16 Cu by 4-Acid and ICP Finish (Method OG62) at ALS-Chemex

Copper concentration from the certificate is 1,120 ppm. Analyses for copper by method MS-61 exceed the outside limits by 11.4% (Figure 13.8). The certificate 2 standard deviation value is 50 ppm. If this is changed to 95 ppm, which is the calculated 1 standard deviation level for the current analyses by this method, the percentage of samples outside the acceptable limits drops to 1.4 percent. The same holds true for copper analyses by the OG62 method, where the number of outliers drops from 3.2 to 1.3

percent. No significant bias is observable in either analytical method. Gold and copper analysis for this standard display an acceptable accuracy and precision.

The current means and standard deviations are based upon means and standard deviations calculated from a total digestion on 100 samples analyzed at 10 labs. At this point in the RNMC program, after more than 1,000 determinations using the same digestion method at the same laboratory, the mean and standard deviation for the reference material needs to be re-calculated and re-incorporated into the quality control review.

13.11.4 CGS-18

Control charts for the CGS-18 are presented in Figures 13.10 to 13.12. The CGS-18 standard was introduced into the sample stream in October of 2008 after supplies of the CGS-12 standard were exhausted. Gold concentration of this standard is 297 ppb. Twenty-six percent of the analyses fall outside of the acceptable limits (Figure 13.10).

Copper concentration from the certificate is 3,190 ppm. Twenty-three percent of the analyses fall outside of the acceptable limits for the MS-61 method (Figure 13.11). Less than one percent (0.5 %) of the copper analyses from the OG-62 method fall outside of the acceptable limits (Figure 13.12). Mean bias is -1.5% and 0.9 % respectively.

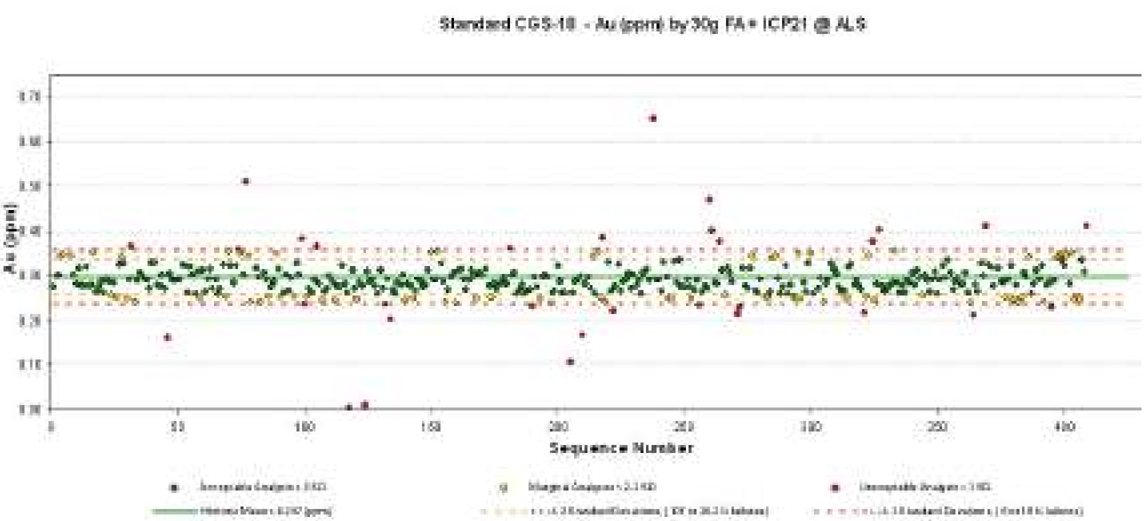


Figure 13.10: CGS-18 Au by 30g FA and ICP Finish (Method ICP21) at ALS-Chemex

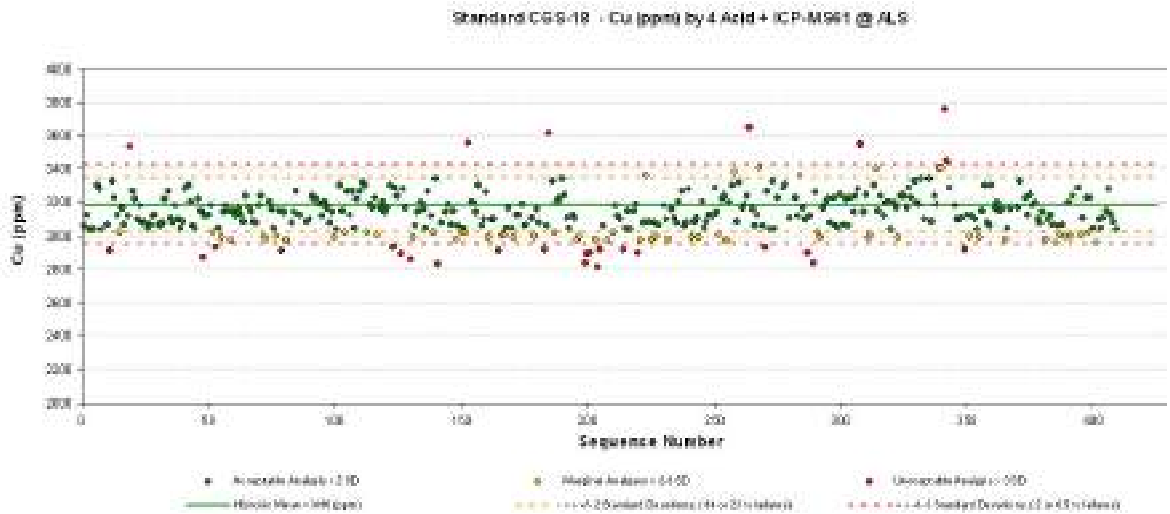


Figure 13.11: CGS-18 Cu by 4-Acid and ICP Finish (Method MS61) at ALS-Chemex

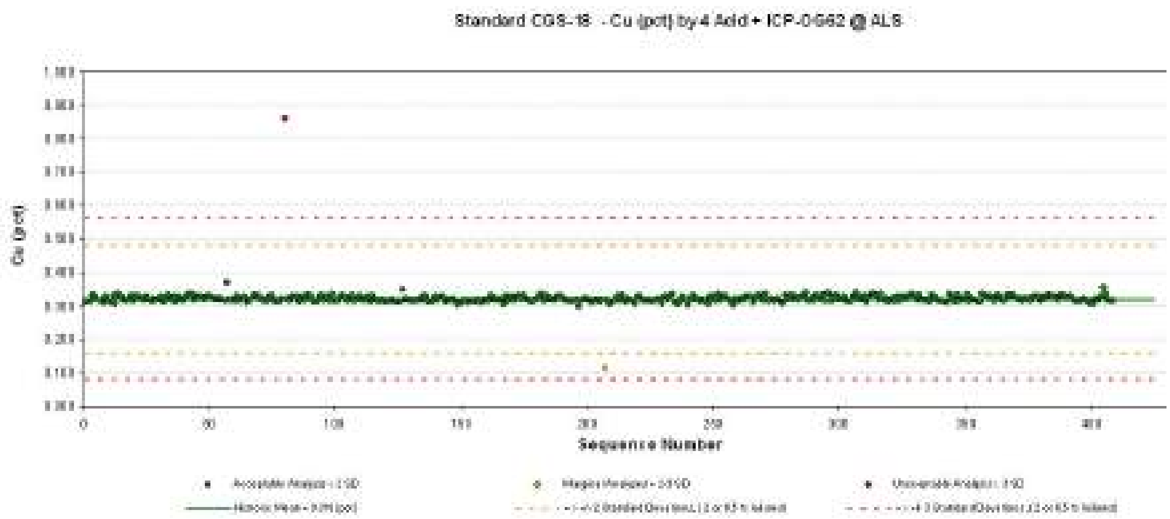


Figure 13.12: CGS-18 Cu by 4-Acid and ICP Finish (Method OG62) at ALS-Chemex

The standard deviations range from 2 (for copper) to 4 times (for gold) the standard deviation of the original certificate (Table 13.1). If the calculated standard deviation is used with a + 1 standard deviation acceptance criteria, the failure rate for gold falls from 26.2% to 1.5% and for copper falls from 23% to 0.5%. Most of the failures for both methods occur between the 2 and 3 standard deviation range, which suggests that the criteria needs to be updated and that the overall accuracy and precision of these analyses are acceptable when the method specific acceptance criteria are used.

13.11.5 CM-01

Control charts for the CM-01 are presented in Figures 13.13 to 13.17. The CM-01 standard was the only standard used in the analytical program that was certified for gold, copper and molybdenum. The summary statistics are located in Table 13.1.

Gold concentration of this standard is 1,850 ppb. Nineteen percent of the analyses fall outside of the acceptable limits. The mean bias is -1.1%. Copper concentration from the certificate is 8,530 ppm. Forty-nine percent of the MS-61 and 4.7 % of the OG-62 analyses fall outside of the acceptable limits. The mean bias is -4.4% for the MS-61 method and 0.9% for the OG-62 method. No trends are visible in these analyses.

As previously discussed, if the most obvious misclassified standards are reclassified and the calculated standard deviation for each method is substituted into the acceptance criteria, the percentage of samples outside of the acceptable limits drops from 19% to 2.2% for gold, 49% to 0.2% for copper by the MS-61 method, and 4.7% to 4.1 % for copper by the OG-62 method. One may observe the number of misclassified standards by looking at linear patterns of points extending across the control charts in all of the figures. A case in point is the line of points extending across the control chart at around 2,650 ppm Cu (Figure 13.14) or 0.265 ppm in (Figure 13.15). Overall, the accuracy and precision of the gold and copper are acceptable for the CM-01 standard.

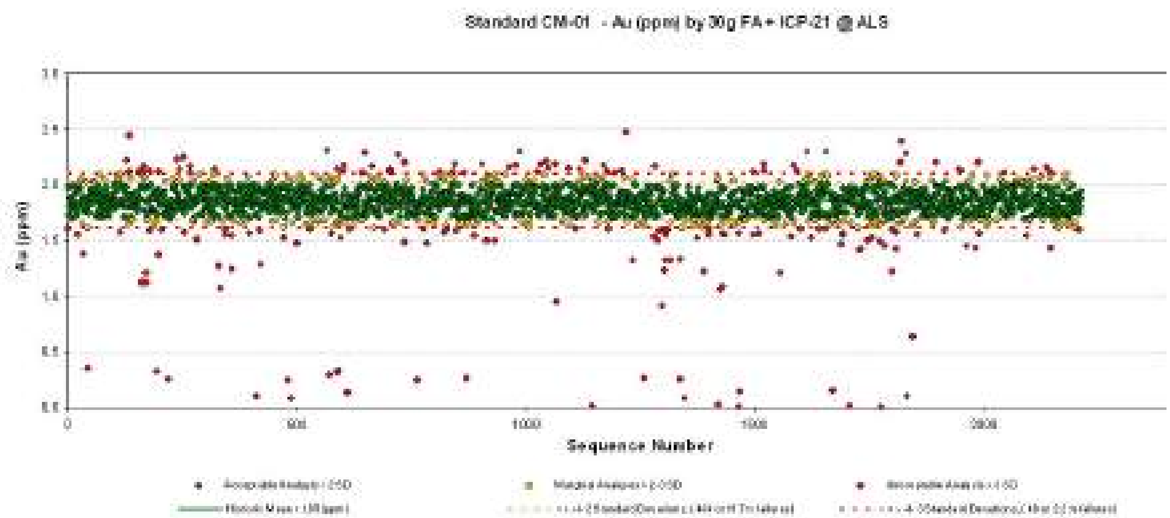


Figure 13.13: CM-01 Au by 30g FA and ICP Finish (Method ICP21) at ALS-Chemex

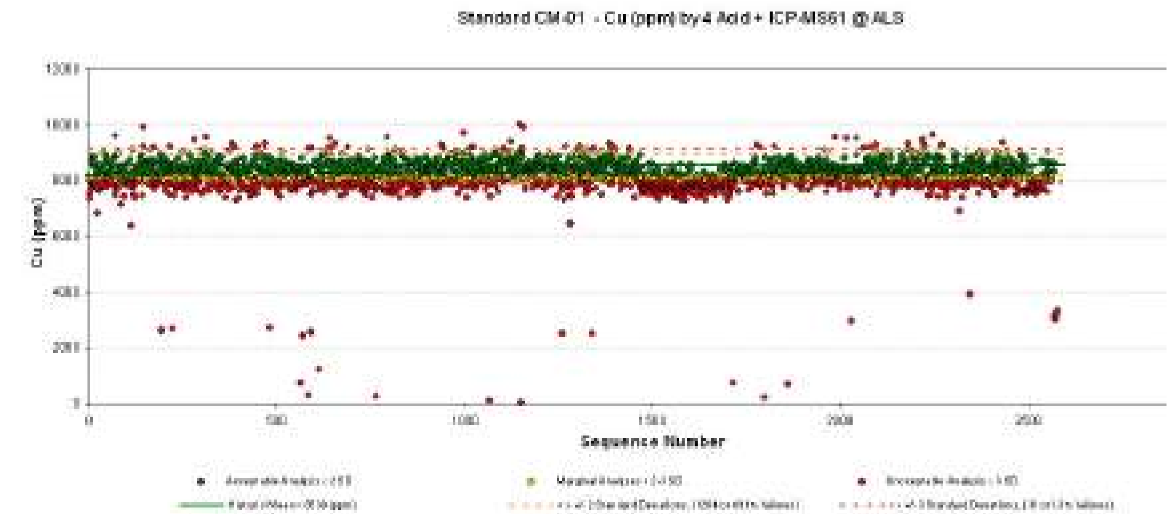


Figure 13.14: CM-01 Cu by 4-Acid and ICP Finish (Method MS61) at ALS-Chemex

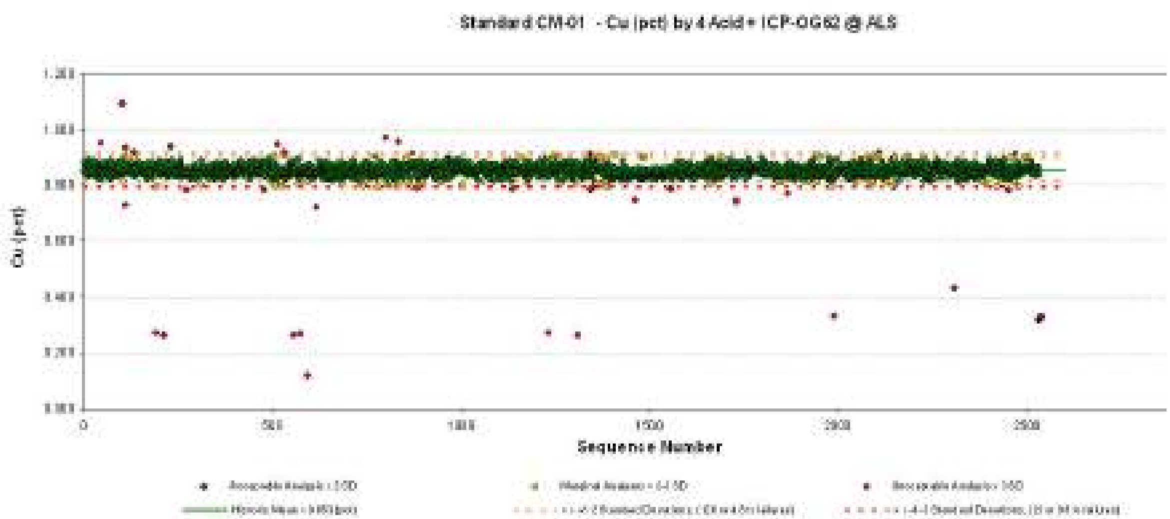


Figure 13.15: CM-01 Cu by 4-Acid and ICP Finish (Method OG62) at ALS-Chemex

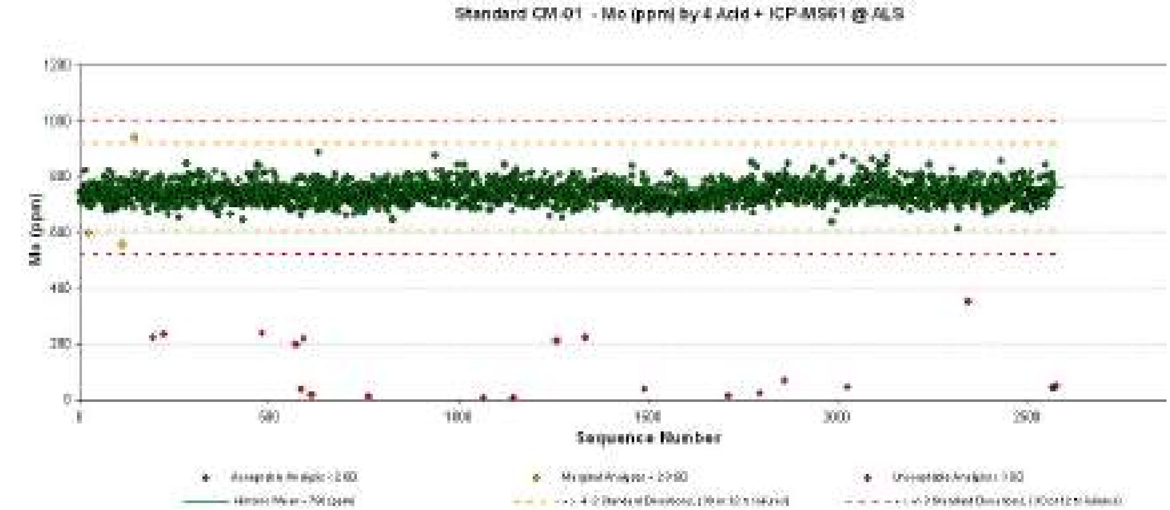


Figure 13.16: CM-01 Mo by 4-Acid and ICP Finish (Method MS61) at ALS-Chemex

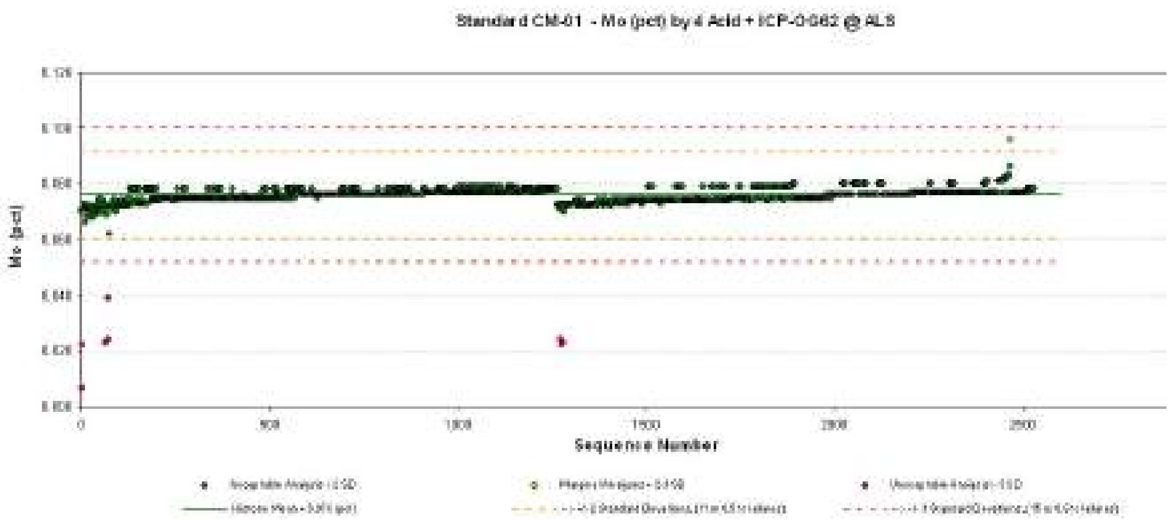


Figure 13.17: CM-01 Mo by 4-Acid and ICP Finish (Method OG62) at ALS-Chemex

Molybdenum concentration from the certificate is 760 ppm. One point two percent of the MS-61 and 0.4 % of the OG-62 analyses fall outside of the acceptable limits for Mo. The Mo data is accurate and precise.

Figure 13.17 shows Mo analyses of the CM-01 standard by the OG-62 Method. The points rise from 0.65 percent to the accepted value of 0.77 percent molybdenum. The pattern repeats around sequence number 1250. This is an artifact of the data structure. The data were sorted by date and by certificate. The points after sequence number 1250 still require input of the date information, thus the pattern appears to repeat itself. Once the data is updated with date information this pattern will overlay with the original sequence.

Figure 13.14 shows copper analysis of the CM-01 standard by the ALS Method MS-61. There is an unusual dip in the data starting from sequence number 1450 to sequence number 1750. This correlates with the time period from the beginning of July 2008 to mid-September 2008. During this period the values of the copper analyses dropped by 6 to 8% for standards CGS-12, CGS-16, and CM-01, which were submitted in the same sample stream. During that period, analysts at ALS-Chemex had modified the calibration protocol of the ICP instruments in order to obtain better precision in the MS-61 determinations. Copper and molybdenum analyses were affected by this change, resulting in a drop of 6 to 8% in the value of the analyses. ALS-Chemex was notified of this, conducted an internal review based on information provided by RNMC, and agreed to correct the problem and re-issue the certificates for analyses completed during this time period.

The OG-62 copper and molybdenum data does not show this discrepancy, as the instruments used for this method were not subjected to the calibration modification that occurred during that time period.

13.12 Field and Lab Duplicates at ALS-Chemex

The duplicates database for ALS-Chemex contains over 2,500 sample duplicates and up to 3,000 laboratory duplicates. Comparative statistics for both duplicate types are summarized in Table 13.4.

Table 13.4:
Summary Statistics for Field & Lab Duplicates at ALS-Chemex

Field Duplicates	**Au_ICP21_ppm**		**Cu_MS-61_ppm**		**Cu_OG-62_ppm**		**Mo_MS-61_ppm**		**Mo_OG-62_ppm**	
	Original	Dup	Original	Dup	Original	Dup	Original	Dup	Original	Dup
# of Analyses	2421		2422		805		2422		484	
Mean	0.1108	0.1162	1236	1247	0.3573	0.3618	36.3349	38.0805	0.012	0.013
Median	0.049	0.048	488	475	0.205	0.216	12.375	12.2	0.009	0.009
Min	0	0	0.9	0.9	0	0	0.09	0.08	0.001	0.001
Max	3.8	2.87	10000	10000	5.82	5.42	950	893	0.104	0.093
Variance	0.0396	0.0447	3871061	3939766	0.2272	0.2113	4296.035	5272.827	0.0001	0.0001
Coeff. of Var.	1.80	1.82	1.59	1.59	1.33	1.27	1.80	1.91	0.88	0.93
Std. Dev.	0.1989	0.2115	19681	1984	0.4767	0.4597	66	72	0.0106	0.0116
% Bias	4.92		0.96		1.23		4.8		3.89	
Corr. Coeff.	0.81		0.91		0.94		0.78		0.71	
% samples with Precision of < 10%	38.2		48.5		54.2		41.8		40.3	
% samples with Precision of < 15%	**48.9**		**62.5**		**67.0**		**56.2**		**53.9**	
% samples with Precision of < 20%	57.9		70.9		74.2		65.8		68.2	

Lab Duplicates	**Au_ICP21_ppm**		**Cu_MS-61_ppm**		**Cu_OG-62_ppm**		**Mo_MS-61_ppm**		**Mo_OG-62_ppm**	
	Original	Dup	Original	Dup	Original	Dup	Original	Dup	Original	Dup
# of Analyses	3050		2819		1137		2819		616	
Mean	0.131	0.131	1343	1338	0.3439	0.3448	36	36	0.012	0.012
Median	0.064	0.064	561	562	0.21	0.21	13.2	13.15	0.009	0.009
Min	0	0	1.5	1.6	0	0	0.09	0.09	0.001	0.001
Max	5.51	5.75	10000	10000	6.12	5.98	1030	1010	0.078	0.078
Variance	0.057	0.059	4177894	4175673	0.2085	0.2073	4748	4638	0.0001	0.0001
Coeff. of Var.	1.83	1.85	1.52	1.53	1.33	1.32	1.91	1.89	0.79	0.78
Std. Dev.	0.2394	0.2428	2043.99	2043.446	0.4566	0.4553	68.91	68.12	0.0094	0.0093
% Bias	0.26		-0.36		0.24		-0.02		-0.75	
Corr. Coeff.	1.00		1.00		1.00		1.00		0.99	
% samples with Precision of < 10%	**67.4**		**90.7**		**96.7**		**84.5**		**80.4**	
% samples with Precision of < 15%	76.9		95.7		98.9		91.6		90.6	
% samples with Precision of < 20%	83.5		97.3		99.3		94.8		96.4	

Field (sample) and laboratory (analytical) duplicates show a much improved precision between gold, copper and molybdenum duplicates with the change in analytical methods at ALS-Chemex. There is no statistically significant difference between means or standard deviations between the duplicate sets. Bias is less than 5% for the field duplicates and less than 1% for the laboratory duplicates.

Correlation coefficients range from 0.81 to 1.0. Correlation increases from field duplicates to lab duplicates. Lab or "analytical duplicates" show greater reproducibility than the field duplicates. Similarity of the means and standard deviations, the lack of any significant bias, and the high correlation indicate acceptable precision for both field and analytical samples during the program.

Different precision thresholds are listed in Table 13.4 for each duplicate type along with the corresponding percent of samples passing each threshold. One generally expects that 95% of the field duplicates will have a precision of <15% and 95% of lab or analytical duplicates would have a precision of <10%. The calculated percentages for both duplicate types are listed in bold red font.

One can see that field duplicates at the 15% precision threshold range from 48.9 % for gold analyzed using the ICP21 method to 67 % for copper analyzed using the OG-62 method (Field Duplicates, Table 13.4). Laboratory duplicates at the 10% precision threshold range from 67.4 % for gold analyzed using the ICP21 method to 96.7 % for copper analyzed using the OG-62 method (Lab Duplicates, Table 13.4). For copper field duplicates using the MS-61 method, 62.5% of the field duplicates are reproducible to within 15% of one another and 90.7% of the lab duplicates are reproducible to within 10% of one another. These percentages are influenced by a high percentage of field and laboratory duplicates at the lower end of the linear working range of the analytical method, where precision falls off as concentrations decrease. This can be observed in the scatter plots of Figures 13.18 to 13.22.

These measures of reproducibility indicate that overall precision is acceptable, particularly for samples above the mine cutoff grade of 2,000 ppm Cu.

Scatterplots of the gold, copper and molybdenum sample and lab duplicates are presented in Figures 13.18 to 13.22. The sample duplicates (field) are plotted on the left and the analytical (laboratory) duplicates on the right for each method and element. The data was log-transformed and plotted on log-log axes to be able to view the precision performance over the grade ranges exhibited in the analyses.

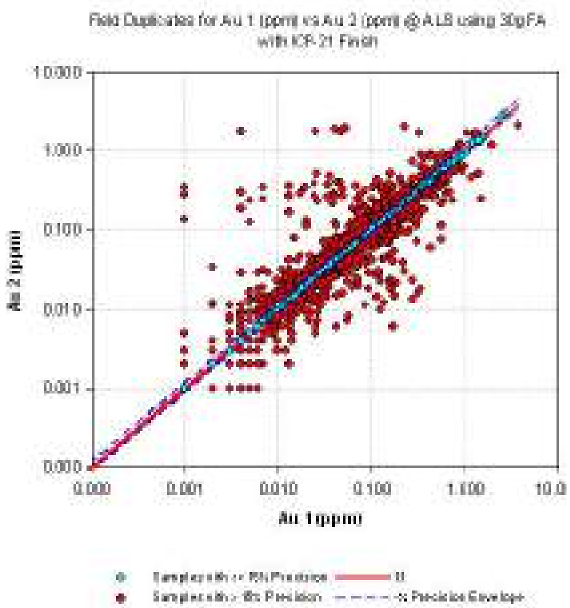


Figure 13.18a: Sample duplicates for Au analyzed with the ICP21 method at ALS-Chemex

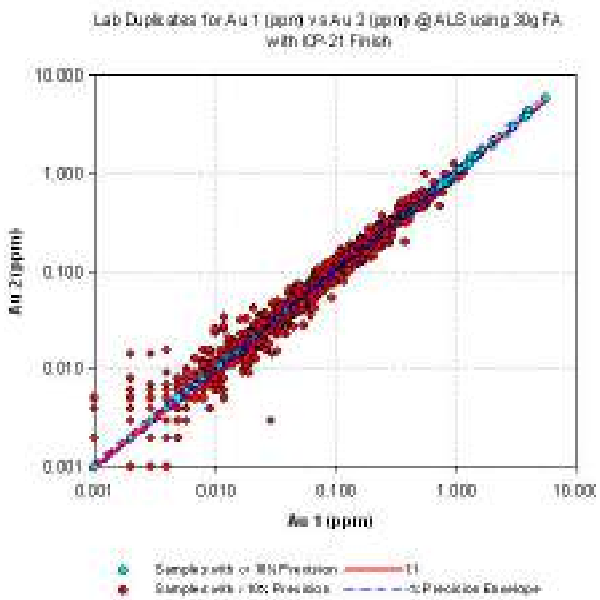


Figure 13.18b: Laboratory duplicates for Au analyzed with the ICP21 method at ALS-Chemex

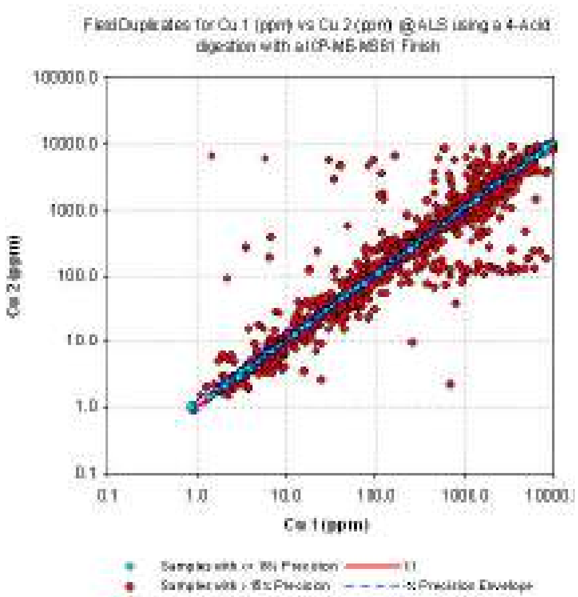


Figure 13.19a: Sample duplicates for Cu analyzed with the MS-61 method at ALS-Chemex

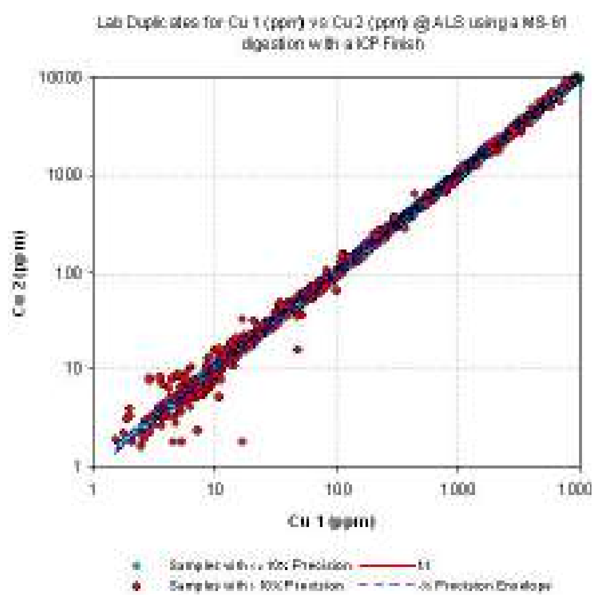


Figure 13.19b: Lab duplicates for Cu analyzed with the MS-61 method at ALS-Chemex

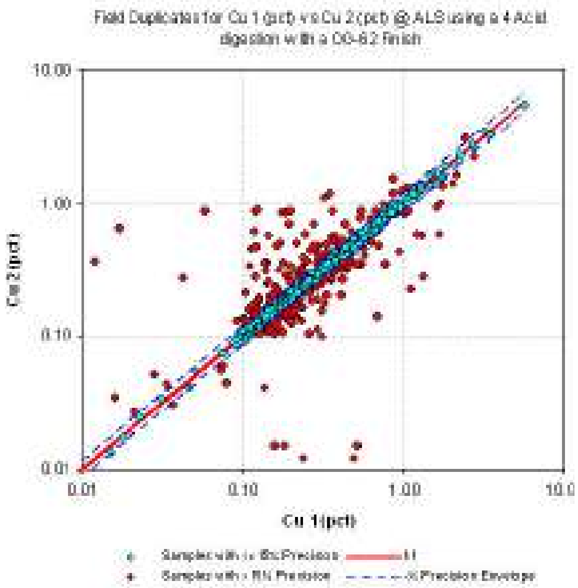


Figure 13.20a: Sample duplicates for Cu analyzed with the OG-62 method at ALS-Chemex

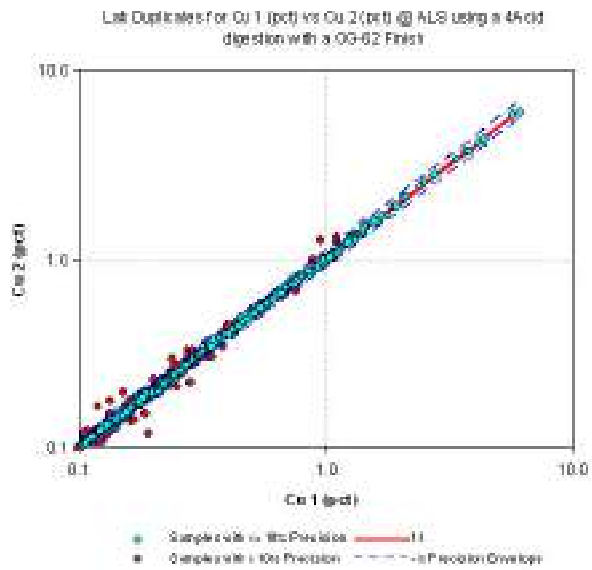


Figure 13.20b: Lab duplicates for Cu analyzed with the OG-62 method at ALS-Chemex

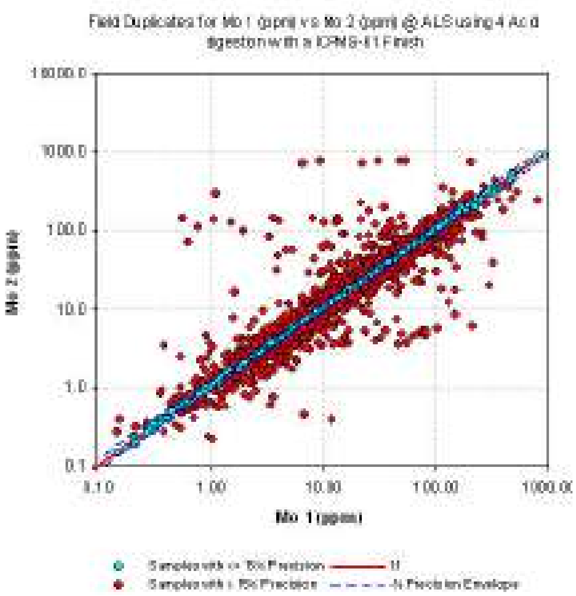


Figure 13.21a: Sample duplicates for Mo analyzed with the MS-61 method at ALS-Chemex

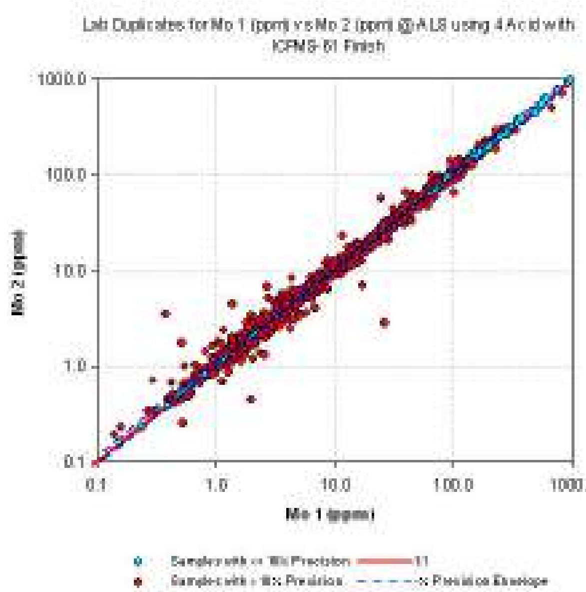


Figure 13.21b: Lab duplicates for Mo analyzed with the MS-61 method at ALS-Chemex

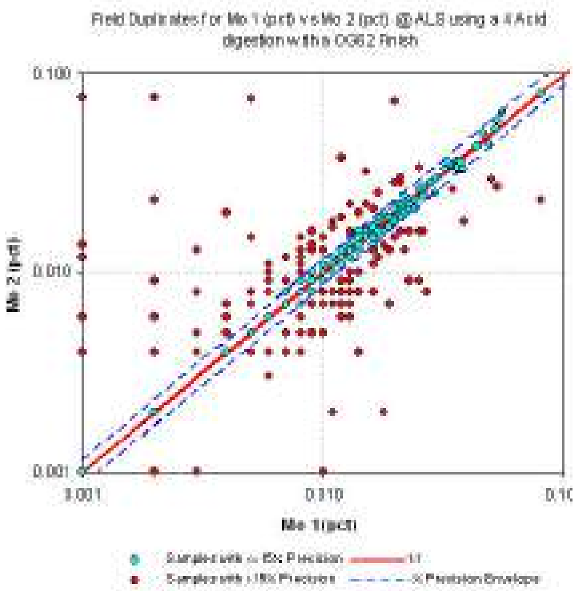


Figure 13.22a: Sample duplicates for Mo analyzed with the OG-62 method at ALS-Chemex

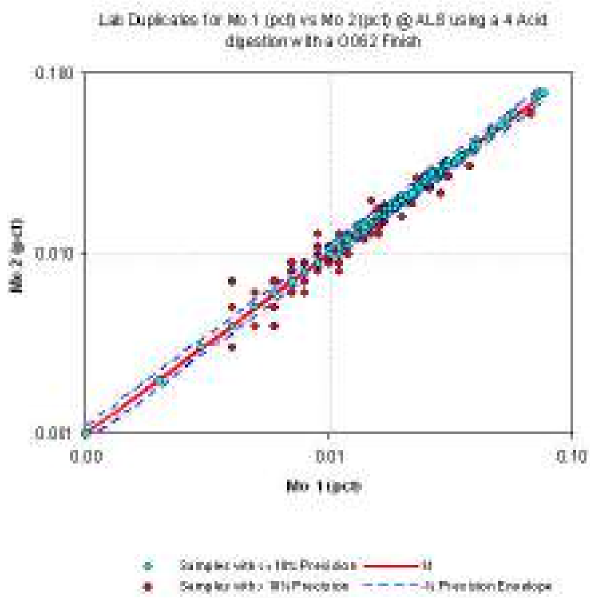


Figure 13.22b:Lab duplicates for Mo analyzed with the OG-62 method at ALS-Chemex

As one examines the scatter plots from sample duplicate on the left to analytical duplicate on the right, one can see that the precision improves. The reproducibility throughout the linear working range of the analytical methods is relatively consistent in the field duplicates and strongly co-linear in the laboratory duplicates for each analytical method.

There is linear array of points extending horizontally away from 1:1 line in the Au (ICP-21 Method, Figure 13.18a), Cu (MS61 Method, Figure 13.19a), Mo (MS61 Method, Figure 13.21a) for the field duplicate scatter plots. These features can be created by dilution points in the analytical protocol where a concentrated solution needs to be diluted while being analyzed or by transition of analyses between ICP-ES and ICP-MS determinations at ALS-Chemex. ALS-Chemex uses both instruments in the method, prescreening the ICP-MS analysis (which is used to obtain data below 100 ppm Cu), with the ICP-ES instrument (which is used to obtain data from 100 to 10,000 ppm Cu).

13.13 Check Samples from ALS-Chemex to Skyline

Approximately 2,000 pulp samples spanning the entire copper grade distribution were sent to Skyline Laboratories in Tucson, AZ. Skyline is analyzing these samples using the same analytical protocol for Au, Cu and Mo that was used by ALS-Chemex. The results are pending.

13.14 ALS-Chemex QC Summary

A rigorous quality assurance and quality control program was implemented when the analysis program was switched to ALS-Chemex. Standards and field duplicates were included into the sample stream at a rate of 15%. Laboratory duplicates were also monitored. All quality control samples were blinded to the analytical laboratory through a preparation protocol that involved sample preparation at ALS-Chemex, insertion of blinded standards and re-numbering of samples at MEG, and re-submittal to ALS-Chemex for analysis. Over 100,000 samples went through this program at ALS-Chemex. Four hundred to 3,000 determinations were performed on five certified standards submitted into the sample stream. Five hundred to 3,000 duplicate analyses were performed on field and laboratory duplicates.

Reference sample statistics and control charts show acceptable accuracy and precision for materials with gold concentrations above 300 ppb Au analyzed by the ICP-21 method. No significant bias is evident.

Reference sample statistics and control charts show acceptable accuracy and precision for materials with copper concentrations above 10 ppm. Copper analyzed by the OG-62 method shows greater accuracy and precision than the MS-61 method. However, both methods yield results with acceptable accuracy and precision. No significant analytical bias is evident. Accuracy and precision improve when misclassified materials are re-classified.

Field duplicates show acceptable precision, particularly at grades above 1 ppm Au and 10 ppm Cu. The precision improves when duplicates below these thresholds are removed from analysis. Duplicates with concentrations above these thresholds display strong correlation and no significant bias for all analytical methods.

Laboratory duplicates show excellent precision, at concentrations above 1 ppm Au and 10 ppm Cu. Duplicates with concentrations above these thresholds display strong correlation and strong co-linearity for all analytical methods. No significant bias exists.

The ALS-Chemex data are of acceptable quality for resource modeling.

13.15 Recommendations

At this point in the RNMC program, after more than 1,000 determinations using the same digestion method at the same laboratory, means and standard deviations for the reference material should be re-calculated and new standard statistics re-incorporated into future quality control reviews.

After this is completed, all standards with analyses exceeding the mean ± 3standard deviations should be identified and re-analyzed at ALS-Chemex. These standard failures should be bracketed by ± 5 samples to determine if the failures persist upon re-analysis.

Check analyses are underway at Skyline Laboratories in Tucson, Arizona and need to be evaluated upon receipt.

A final report containing the evaluation of the Skyline Laboratories analysis will be available at the Robinson Nevada Mining Company offices by February 1, 2009.

14. Data Verification

14. Data Verification

14.1 Historical Data Verification

The 2004 Technical Report on the Robinson Operation (Ristorcelli and Hardy, 2004) reviewed a number of aspects of the historical data available for the Robinson project, and concluded that there was evidence of isolated problems with recording or lack of structural data, density data, and downhole surveys; as well as a lack of information to evaluate the possibility of downhole contamination and historic check assay programs. These problems were viewed as relatively minor and normal for a district with such a long operating history. Indeed, the historical economic success of mines in the district is a strong argument for the reliability of the data that those mines relied on and the continued success for Quadra's Robinson operation also argues for the validity of that historical data. In their review of historical data, Ristorcelli and Hardy (2004) detailed the following areas.

1. Geologic Data Audit
2. Database Audit
3. Drill Collar Audit
4. Downhole Survey Audit
5. Data Entry Audit
6. Other Audits
7. Sampling and Assay Audit
8. Drill Methods/Downhole Contamination Audits
9. Geotechnical Data Audits

14.2 Sample and Data Conservation Efforts

During the period of care and maintenance prior to Quadra's acquisition of the Robinson mine, the historic pulp, reject, and core archives were located in two unsecured dilapidated buildings in the area of the Deep Ruth shaft. Some of this material was exposed to weathering, and some of the shelving for pulps was vandalized. However, most pulp and coarse reject samples survived intact, and since 2006, Quadra has undertaken to recover, conserve, and properly store all of this material in sea shipping containers which are transportable, lockable and weatherproof.

Literally tens of thousands of geologic and engineering maps, sections, diagrams, drill logs and reports have been produced throughout the history of the Robinson district and many of these are irreplaceable. Some of this material was stored in safe quarters in the mill building during the period of care and maintenance, but much was stored in a badly ventilated, leaky vault near the Ruth pit. The material was safe from vandalism, but during the period from the late 1990's to 2005, there was marked damage and deterioration due to mold. This material has been stored in more secure conditions and is still in the process of being stabilized and scanned into digital form.

14.3 Quadra Database Development

The drilling database at Robinson consists of 9,651 drill holes, drilled from the earliest periods of mining activity in the Robinson district, through December 31, 2008. They were drilled for a number of different companies and organizations and include churn, reverse circulation, rotary and core holes. Significant variability in logging quality, assay methodology and quality control, collar and down-hole surveys is notable. During the Magma Copper-BHP tenure in the district, data from these holes were entered into a simple spreadsheet database, and then subsequently converted into a Microsoft Access database.

Since acquiring the property in 2004, Quadra has established and maintained the District Central Drill Hole Database, (DCDHD) utilizing an AcQuire database structure, containing all relevant analytical data, geologic logging information, drill hole surface and down hole surveys, and topographic information.

14.3.1 Geology Validation

For geologic modeling, there is a wealth of historical data, but it is common that some of the interpretations and data sources may disagree, particularly where completed by different people from different eras. A data hierarchy was developed to assign a quality weight for geological data, including drilling, so that an appropriate degree of reliability and consistency might be established for downstream users of geological data. Table 14.1 lists rank and criteria for the data hierarchy.

Table 14.1:
Listing of Rank and Criteria for Historical Data Hierarchy

Rank	Criteria
1	Bench face and outcrop mapping.
2	Core and rotary holes with down hole deviation surveys, detailed lithologic data sufficient to reliably assign lithology and PCD systematic codes.
3	Core and rotary holes without down hole deviation surveys, detailed lithologic data sufficient to assign some lithology and PCD systematic codes.
4	Rotary holes with brief summarized lithological data that may be useful, in concert with that from adjacent holes, to assign lithology codes.
5	Holes of uncertain provenance, lacking lithological data, possibly with historic geochemistry.

All available paper drill logs have been scanned and converted to portable document format (pdf) and archived on the Robinson Nevada Mining Company servers. These files are currently undergoing review by qualified, experienced geologists. The data is being captured in a Microsoft Access database and subsequently transferred into the DCDHD. In cases where multiple geologic and analytical data are available, the database captures all existing data sets and establishes a data extraction prioritization sequence to the respective data, based upon the data hierarchy listed above. Lithology, alteration, mineralization, collar survey and downhole survey information are recorded as accurately and precisely as the historic scanned documents allow. Codes are subsequently assigned using a series of query's and manual coding designed to identify the lithology, structure, alteration, mineralization and ore type.

After data are entered into the Microsoft Access database, drill logs and graphic geologic sections are produced. These logs and sections are then reviewed for data omissions, gaps, overlaps and inconsistencies. If such errors are identified, the original logs are reviewed and edits are applied to the modern database. Logs and sections are then reviewed a second time, and if full rectification is not possible from the original logs, further study, up to the point of re-logging archived chips and core or re-drilling a duplicate modern hole, is used to rectify drill log and interpretational problems.

These procedures have produced a robust drill hole database that permits users to select high quality data and to omit, with clearly defined, unprejudiced guidelines, data that is considered inadequate.

14.3.2 Assay Validation

The assays in the DCDHD are proofed by comparing an export of Total Cu, Sol Cu, Cu QLT, Cu SAP, Au MS61, Au overgrade, Mo MS61, and MO overgrade from the database with electronic assay certificates from ALS Chemex. The comparison is conducted for each assay type and each interval. This comparison is done electronically and includes historical samples that have been re-assayed as well as modern Quadra "Q" series holes.

A second check of the assays in the DCDHD is also completed. A random five percent of all assays returned from ALS Chemex have the paper assay certificate pulled and compared on an interval by interval basis for each of the elements listed above.

For the historical sample re-assays, the interval data that exists in the central database does not always match the interval of the pulp envelopes that exists. This may be due to missing samples, past composite sampling for some elements or for several other reasons. These gaps and overlaps in the assay extraction are handled by a script run on the assay file in the DCDHD prior to extraction.

The historical database contained analyses for MoS_2 in the field for molybdenum, while modern samples are analyzed for Mo. The historical data is of insufficient quality and density to be usable for the purpose of calculating molybdenum resources and reserves, but is certainly usable for some mine planning purposes. The historical data is converted to Mo by multiplying the value of each record by 59.94 percent (the weight percentage of Mo in molybdenite).

14.3.3 Downhole Survey Validation

Quadra era drillholes are routinely surveyed with a gyroscopic survey tool supplied by IDS Survey, Elko, Nevada. An electronic comparison of the survey data in the database versus electronic copies of the downhole survey certificates is completed for all Quadra series drilling. A review of five percent of all downhole surveys as entered in the database are checked against the respective original paper certificates from the outside downhole survey contractor to determine if the surveys were imported into the database correctly. No errors were found in the period 2006-2008.

14.3.4 Collar Survey Validation

Drill holes are designed in Mine Sight software and the designed drill hole coordinates are recorded into the DCDHD, (AcQuire Database). These design coordinates are used to locate the proposed drill holes on the ground utilizing GPS. When site conditions require the designed drill hole to be moved, new as-built coordinates are captured with the GPS and the correction recorded into the Drill Hole Data Book and into the DCDHD. Upon completion of the drill hole, the Mine Operations Survey Department surveys in the actual location of the completed drill hole, which has a marked stake left in the cemented and abandoned drill hole collar. The Survey Department records the final survey as part of their official daily records and transmits the drill hole collar survey data electronically to the Exploration Department Database Manager. The Database Manager imports the final drill hole collar survey data into the DCDHD and conducts a comparison against the as-built survey data. If an error occurs, the Survey Department is contacted to resolve the error. An annual electronic review of all Quadra series collar coordinates is done by comparing the DCDHD against the Mine Survey Departments coordinates for each hole. An annual review of five percent of the drill hole collar surveys, as entered in the data base, are checked against the electronic records from the Mine Survey Department. No errors have been found in the period 2006-2008.

14.3.5 Drill Methods/Downhole Contamination Audits

Drill rig operators and sample technicians are instructed in Quadra's standardized core and RC drill rig sample drilling operation, sampling procedures and protocols on a routine basis. Site procedural audits are conducted and rig geologist who are actively logging drill hole core and RC sample chips during drilling operations, inspect core on a twice daily basis, measure each drill core run for accurate core block measurements, evaluate consistent and proper alignment of core in the core boxes, measure accurate depth of drill holes and monitor RC drill sample chip quality for downhole contamination indications. Final assay grades are compared against detailed geologic logging anticipated copper grade estimates for intervals, and if substantive differences are noted, the assay lab is contacted to re-run the analysis. A review of potential downhole contamination, utilizing software to evaluate cyclicity and decay for the entire DCDHD, is pending.

14.3.6 Geotechnical Data Audits

Please refer to the geotechnical discussion in Chapter 23.1.2.1.

14.4 Data Verification Summary and Conclusions

The critical conclusions made in the 2004 Technical Report for Robinson (Ristorcelli and Hardy, 2004); namely, that the historical drill hole database for copper assays and collar survey was clean and of sufficient accuracy to be a reliable base for a mining operation, has been borne out by the operational results of the last several years. Data acquired over the last several years by Quadra has been subjected to a higher standard of QA/QC scrutiny and database verification than at any time in the past, and a number of deficiencies in the database have been addressed – including the addition of much more and higher quality data on rock density, as well as molybdenum and gold content.

In past models, interpreted mineralization and alteration have been coded into the model based on interpreted geology from sections and plans. This was necessary owing to the long history of drilling in the district, and the uneven and variable interpretation of geology which was inevitable as ideas progressed through the years. In the past, these reinterpreted geologic codings were not distinguished from the original data. Recent drilling and modeling is confronted with the same problem of reinterpreting drill hole geology and changing codings for many historical drill holes. With the assistance of MineMappers consultants, a more thorough and systematic re-mapping, re-logging, reinterpretation and systematice coding of the entire district geology and drill hole geologic database is being completed. This reinterpreted coding is now distinguished from, but does not replace, earlier interpretations in the database.

15. Adjacent Properties

15. Adjacent Properties

The Robinson district is large, covering more than 30 square miles, and Quadra's property position includes all known major historic producers. Quadra is the only currently operating mine in the district. RNMC regularly reviews and documents the ownership status of adjacent lands. In the past four years, RNMC has acquired some additional lands, adjacent to the mine property, as patented mill site claims and via purchase from other land owners. This is documented in Chapter 4.

The Taylor and Ward Mountain mines, some fifteen to twenty miles to the southeast, were active as recently as the 1990's. Barrick's Bald Mountain gold mine, some 60 miles to the northeast of Robinson, is believed to be the nearest operating mine at the time of writing.

16. Mineral Processing and Metallurgical Testing

16. Mineral Processing and Metallurgical Testing

This chapter has been co-authored by David Newhook, P. Eng., of Quadra Mining Ltd. and Mark O'Brien, Chief Metallurgist, of Robinson Nevada Mining Company.

16.1 Summary

A review of the metallurgical data from the operation of the Robinson mill by Robinson Nevada Mining Company has been completed with respect to predictions and modeling for the current resource estimates. The main elements in the model are plant throughput, copper recovery, copper concentrate grade and gold recovery.

The Robinson mill consists of a crushing circuit with one 60' x 89' gyratory crusher with a nominal capacity of 2,500 TPH, crushing to minus 6" material. The grinding circuit consists of one 32' x 14.75' (9.75 m x 4.50 m), 10,000 horsepower (HP) variable speed semi-autogenous grinding (SAG) mill followed by two 20' x 30.5' (6.1 m x 9.3 m) 8750 horsepower closed circuit ball mills. The rougher flotation circuit consists of two parallel rows of ten 3,000 cubic feet flotation cells followed by cleaning in six 10' x 38' (3.05 m x 11.58 m) column flotation cells in a parallel configuration. No regrinding is currently being performed. The concentrates are thickened and filtered prior to shipment. The tailings are thickened and are transported by gravity flow to the tailings impoundment.

Mill feed is from the Veteran Pit, with a transition to the Ruth Pit expected in the next several years. The data used to project future capacity and metal recoveries has been developed primarily from Veteran Pit operating data. The Ruth Pit metallurgical testing is in progress and was not used in this report. Table 16.1 provides a summary of actual plant performance.

Table 16.1:
Summary of Actual Plant Performance (2005 – 2008)

Year	Plant Throughput (stpd)	Cu Recovery (%)	Copper Concentrate Grade (%)	Au Recovery (%)
2005	41,547	76.4%	25.2%	53.2%
2006	41,859	65.4%	24.6%	53.7%
2007	42,797	67.4%	25.5%	59.0%
2008	41,684	77.6%	29.0%	68.5%

Plant design throughput estimates indicate a nominal throughput rate of 35,000 short tons per day (stpd) with an average bond work index (BWI) of 8.6 kWh/ton. Limited data was available for the Veteran Pit which indicated that the bond work index is on average very consistent with the design criteria.

Based on the various differences within the three ore types, the Robinson operation has successfully implemented a system of blending material feeding the primary crusher through to the concentrator. This has improved the stability of the processing plant, enabling more consistent operation and better metallurgical performance with respect to copper recovery and copper concentrate grade.

The Robinson mill has two Falcon concentrators for gold separation which contribute minimally to the overall gold recovery.

16.2 General

David Newhook, P. Eng. Quadra Mining Ltd., has reviewed the metallurgical section for the 43-101 report. David Newhook has reviewed a number of reports pertaining to Robinson operations and has completed the following assessment.

The design criteria for the existing Robinson milling operation were established in 1993. This was followed with the engineering and construction of the mill and mine facilities by BHP (Magma Copper) and the subsequent process plant commissioning in February 1996.

The primary elements of the design criteria established for the mill were:

- Nominal feed rate 35,000 stpd
- Percent Run Time 90%
- Average feed grade 0.605% Cu
- Average concentrate grade 28% Cu
- Average copper recovery 85%

The primary elements of the operational assumptions established for the mill during the remaining life-of-mine are:

- Nominal feed rate 45,000 stpd
- Percent Run Time 93%
- Average feed grade 0.54% Cu
- Average concentrate grade 26% Cu
- Average gold recovery 49%
- Average copper recovery 77%

As previously discussed, the supply of ore to the mill during 2007 - 2008 was primarily from the Veteran pit. Ore deliveries during 2007 were a mixture of hypogene material early in the year followed by supergene material in the later part of the year. A blend of supergene and hypogene materials were the primary ore delivered during 2008.

The mine plan has pit sequencing designed to achieve the blend requirements of supergene and hypogene material types. Therefore, the transition from the Veteran pit to the Ruth pit is anticipated to produce similar results with similar blending requirements.

The process operational assumptions for molybdenum recovery from the copper concentrate are based on actual plant operational data. The molybdenum plant operates when there is sufficient molybdenum present in the copper/molybdenum concentrate to warrant plant operation.

The Robinson process facilities have historically not achieved design production. Robinson Nevada Mining Company however, has had great success in achieving production targets in spite of the high variability in the ore.

16.3 Mineralogy

Operation of the mill during mining of supergene material from the Veteran Pit was subjected to wide swings of head grade and copper mineralogy which led to poor recovery and concentrate grade. At times, chalcocite also led to lower concentrate grades in the range of 17% - 20%, postulated to be a result of rimming of pyrite by chalcocite.

Robinson Nevada Mining Company has identified three main ore type categories:

- leach cap;
- supergene (which includes the chalcocite enrichment blanket); and
- hypogene.

Leach cap material is not delivered to the mill. During 2007, ore was delivered to the mill as mined. The variability of the operation and recovery was such that blending of the Supergene with hypogene material became necessary. The current mine plan recognizes the significant differences in ore material mineralogy and the subsequent processing requirements. Blending of the two principal ores (supergene and hypogene) was initiated late in Q4/07 and is being actively practiced to date.

Recent preliminary metallurgical scoping tests from the Ruth, Kimbley and Wedge deposits indicate that they are generally consistent with the material being mined presently from the Veteran deposit. This is also supported by past Ruth operating and testwork data. As such, it is expected that the Ruth, Kimbley, and Wedge supergene material will require blending similar to the material(s) currently being processed.

16.4 Grinding

The Robinson mill grinding circuit consists of one 32' x 14.75' (9.75 m x 4.50 m), 10,000 horsepower (HP) variable speed semi-autogenous grinding (SAG) mill followed by two 20' x 30.5' (6.1 m x 9.3 m) 8750 horsepower closed circuit ball mills. The power split between the SAG mill and ball mills has been weighted significantly towards the ball mills, 37%: 63%: SAG: Ball. The 2008 BWI test results, while limited, are consistent with the mill design criteria (Table 16.2).

**Table 16.2:
Summary of 2008 Bond Work Index Test Results
for Veteran Pit Material**

Material	Design	2008 Veteran Pit
Porphyry	7.2 – 10.8	8.95
Skarn	7.1 – 8.6	7.8
Mixed Skarn & Porphyry	7.7 – 9.1	8.4

The ore hardness in the Ruth, Kimbley, and Wedge deposits is expected to remain generally consistent with current levels from the Veteran pit. This will facilitate RNMC's ability to achieve above design throughput. Additional milling capacity was achievable in 2008 but was restricted to enable the mine to reach its strip tonnage targets.

Historically the Robinson mill grinding circuit availability has been less than 90%. Through the extensive maintenance program, the grinding circuit run time is over 93% for 2008.

16.5 Copper Flotation

The rougher flotation circuit consists of two parallel banks of 3,000 cubic feet (ft^3) cells, which are followed by cleaning in six 10 foot x 38 foot (3.05 m x 11.58 m) column flotation cells in a parallel configuration. Concentrate is thickened and filtered prior to shipment. The tailings are thickened and are transported by gravity flow to the tailings impoundment.

In 2008, RNMC conducted six weeks of on-site flotation cell testing with a pilot plant. As a result of that testing, it was determined that recovery of ~30% of the copper currently reporting to tailing could be recovered. Capital has been included in the 2009 budget for the installation of 4 @ 160 cubic meter (m^3) flotation cells to treat the rougher tail flow. The average tail head grade is approximately 0.15% copper. Based on this test, it was determined that a conservative increase in copper recovery of 10 million pounds per year was likely. This increase has been included in the current production schedule as has the capital required to add the cells.

The primary issues with copper flotation results are the fluctuating ore types and associated mineralization. As mining progresses through the chalcocite enrichment blanket and the supergene zone, the recoveries and concentrate grades improve significantly. Once in the hypogene zone, the recoveries and concentrate grades improve considerably. Table 16.3 illustrates the individual ore type's general metallurgical results and does not reflect active ore blending. It should be noted that these results are based on operational experience and the plant performance can change significantly based on the actual blend of the components being delivered to the mill.

Table 16.3:
Summary of Metallurgical Results by Individual Ore Type

Ore Type	Copper Concentrate	Recovery	Comment
Leach Cap			Material classified as waste and not delivered to the mill.
Supergene	17% - 28%	70% - 75%	Metallurgical results improves with depth
Hypogene	25% - 30%	75% - 80%	

Within the supergene ore type, the chalcocite enrichment material can exhibit a recovery as low as 50% and a concentrate grade in the low teens. Similarly, due to the highly altered material, clays can exist in the supergene in varying degrees, yielding recoveries between 45 to 50%.

Several modeling equations were developed by Kenneth Edmiston in 1993 to predict recovery depending on material characteristics. RNMC continues to evaluate operating data to improve these correlations to project future performance.

The mine plan has pit sequencing designed and scheduled to achieve the blend requirements of supergene and hypogene material types. Therefore, the transition from the Veteran pit to the Ruth pit is anticipated to produce similar results with similar blending requirements

16.6 Gold Recovery

Gold recovery at the Robinson mill has been evaluated continuously since plant start-up as an opportunity to improve the economics of the operation. The Robinson mill has two Falcon concentrators for gold separation which contribute minimally to the overall gold recovery.

Gold recovery appears to be related to copper mineralogy, with lower recoveries obtained when the mineralogy includes chalcocite. The current copper collector being used (Flomin 7931) appears to enhance gold recovery.

17. Mineral Resource and Mineral Reserve Estimates

17. Mineral Resource and Mineral Reserve Estimates

This chapter has been co-authored by Scott Hardy, P. Eng., Manager, Technical Services of Quadra Mining Ltd., Steven Johnson, Manager, Geologic Services of Quadra Mining Ltd., and Juris Ore, Technical Services Superintendent, of Robinson Nevada Mining Company.

17.1 General Resource Comments

Robinson resources were re-estimated in 2008 to reflect better understanding of the deposits, and to include additional in-fill and exploration drilling completed in 2006 through 2008. While reinterpretation of mineralogical zones and geology was undertaken, much of the underlying geologic interpretation previously performed by BHP, Magma, and MDA was retained, with the new interpretations verifying most of the earlier work.

The resource modeling was done by RNMC and Quadra personnel under the supervision of Scott Hardy, P. Eng., Manager of Technical Services of Quadra. Mr. Hardy is a designated Qualified Person as defined by National Instrument 43-101.

Section 17.1 provides a summary of a few points common to both the Veteran and Ruth resources with details of each resource estimate discussed separately in Sections 17.2 and 17.3, respectively. Finally, the Robinson Tripp-Veteran and Ruth deposit mineral reserves are discussed in Section 17.4.

17.1.1 Density

The feasibility study completed by The Winters Company in 1991, and updated in 1994, contains tonnage factors for 15 different rock types, including tonnage factors for caved ground, dumps, and slides. The report states that densities were calculated for 50 rock samples to determine tonnage factors, but contains no supporting documentation as to the sample location or methodology used to derive the specific gravity determinations. The tonnage factors utilized in these resources are provided in Table 17.1.

**Table 17.1:
Tonnage Factors by Rock Type**

Rock Type	Tonnage Factor ft^3/ton
Mississippian Joana Limestone	10.5
Mississippian Chainman Shale	11.0
Pennsylvanian Ely Limestone	11.0
Permian Rib Hill Sandstone	13.0
Cretaceous monzonite porphyry	13.0
Cretaceous monzonite	13.0
Tertiary rhyolite	16.0
Permian Riepe Springs Limestone	12.0
dump or rubble	17.0
caved ground	17.0
Permian Arcturus Formation	13.0
Devonian-Mississippian Pilot Shale	13.0
Kaibab Formation	12.0
Devonian Guilmette Limestone	12.0
undefined	12.0

These values are used in the current resource models for Veteran and Ruth based on the BHP lithologic interpretations. Reconciliation results indicate a reasonable match between reported tonnages and modeled tonnages but work to better define rock densities is ongoing. See Section 23.1.2.3 for discussions on reconciliation.

17.1.2 Acid Soluble Copper Models

Acid soluble copper (referred to as soluble copper) is not currently considered an economic resource by itself in the deposits because it is generally low grade. Nevertheless, soluble copper grades are estimated as a guide to identifying material types and for use in calculating mill recovery. In general, higher soluble copper grades indicate more oxides and the presence of leach cap and supergene materials.

There are several major issues associated with soluble copper estimation, including:

- significantly fewer samples of soluble copper are available than for total copper;
- historic assaying methods are not documented well enough to identify the assaying procedures used; and
- in many historic drill holes the soluble copper data was a composite of many samples over several hundred feet, thus rendering it useless for grade estimation.

Nevertheless, soluble copper grades have been estimated in the resource models with the understanding that results are not likely to be accurate as discussed below.

Through the reconciliation process it has been determined that there is a potentially significant bias in soluble copper assay grades. The soluble copper assay values, composites, and resulting estimated block grades are about 50% higher on average than the reconciled Veteran blast-hole and mill reported soluble copper grades. The reasons for the differences are not apparent but are being investigated. One of the confounding issues is that the apparent bias is observed in the newer Robinson drill data as well as the historic data, which may eliminate assay method and sample age as factors in the bias.

The most significant consequences of higher soluble copper estimates are that predictions of copper recovery may be lower than actual recovery, and material types may be misinterpreted. This would result in more material being reclassified from ore to waste. The magnitude of the situation is estimated to be small but the situation is under observation until resolution can be obtained.

17.1.3 Resource

The Robinson resource was classified into Inferred, Indicated and Measured categories in compliance with Canadian National Instrument 43-101 and the "CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines," issued in 2000 and modified with adoption of the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" in 2005. The CIM mineral resource definitions are reproduced below.

Mineral Resource

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-

economic and governmental factors. The phrase 'reasonable prospects for economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralization that under realistically assumed and justifiable technical and economic conditions might become economically extractable. These assumptions must be presented explicitly in both public and technical reports.

Inferred Mineral Resource

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

Indicated Mineral Resource

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions.

<u>*Measured Mineral Resource*</u>

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

The definition of the different resource classes is the same for both the Veteran and Ruth deposits and is based on the distance to the nearest copper sample and the number of copper composites. These criteria are provided in Table 17.2.

Table 17.2:
Classification of Robinson Resource

Class	Distance	Minimum number Of composites
Measured	0 – 150 ft	2
Indicated	150 – 250 ft	2
Inferred	0 – 250 ft > 250 ft	1

The Veteran resource is given in Tables 17.3 through 17.5. Note that the Tripp Pit resource is not included in the tabulation because it is viewed as being mined out at this point in time. The Ruth resource is provided in Tables 17.6 through 17.8, and the entire Robinson resource is presented in Tables 17.9 through 17.12. Note that the resource includes the Robinson reserve.

Table 17.3:
Veteran Measured Resource

Cutoff Cu%	Tons above cut x1000	Avg Grade Cu %	Contained lbs Cu x1000	Avg Grade Au oz/ton	Contained oz Au x1000
0.10	267,000	0.34	1,815,600	0.006	1,602
0.20	163,700	0.47	1,538,780	0.007	1,146
0.30	101,900	0.60	1,222,800	0.008	815
0.40	63,600	0.75	954,000	0.008	509
0.50	42,100	0.91	766,220	0.008	337
0.60	29,200	1.07	624,880	0.008	234
0.70	22,000	1.20	528,000	0.008	176
0.80	16,300	1.36	443,360	0.007	114
0.90	12,300	1.54	378,840	0.007	86
1.00	9,900	1.68	332,640	0.007	69

Table 17.4:
Veteran Indicated Resource

Cutoff Cu%	Tons above cut x1000	Avg Grade Cu %	Contained lbs Cu x1000	Avg Grade Au oz/ton	Contained oz Au x1000
0.10	87,300	0.22	384,120	0.005	437
0.20	30,300	0.39	236,340	0.006	182
0.30	15,400	0.53	163,240	0.007	108
0.40	7,300	0.73	106,580	0.006	44
0.50	4,500	0.92	82,800	0.006	27
0.60	3,000	1.11	66,600	0.005	15
0.70	2,300	1.26	57,960	0.005	12
0.80	1,800	1.39	50,040	0.005	9
0.90	1,400	1.52	42,560	0.005	7
1.00	1,200	1.63	39,120	0.005	6

Table 17.5:
Veteran Inferred Resource

Cutoff Cu%	Tons above cut x1000	Avg Grade Cu %	Contained lbs Cu x1000	Avg Grade Au oz/ton	Contained oz Au x1000
0.10	20,000	0.15	60,000	0.004	80
0.20	3,400	0.43	29,240	0.006	20
0.30	1,200	0.82	19,680	0.007	8
0.40	1,000	0.91	18,200	0.006	6
0.50	600	1.19	14,280	0.006	4
0.60	400	1.50	12,000	0.005	2
0.70	300	1.80	10,800	0.003	1
0.80	300	1.93	11,580	0.003	1
0.90	300	1.96	11,760	0.003	1
1.00	300	1.96	11,760	0.003	1

Table 17.6:
Ruth Measured Resource

Cutoff Cu%	Tons above cut x1000	Avg Grade Cu %	Contained lbs Cu x1000	Avg Grade Au oz/ton	Contained oz Au x1000
0.10	972,000	0.30	5,832,000	0.004	3,888
0.20	514,400	0.44	4,526,720	0.005	2,572
0.30	303,000	0.58	3,514,800	0.005	1,515
0.40	201,000	0.69	2,773,800	0.006	1,206
0.50	143,100	0.79	2,260,980	0.006	859
0.60	103,500	0.89	1,842,300	0.006	621
0.70	74,400	0.98	1,458,240	0.007	521
0.80	51,700	1.08	1,116,720	0.007	362
0.90	35,100	1.20	842,400	0.007	246
1.00	23,900	1.31	626,180	0.007	167

Table 17.7:
Ruth Indicated Resource

Cutoff Cu%	Tons above cut x1000	Avg Grade Cu %	Contained lbs Cu x1000	Avg Grade Au oz/ton	Contained oz Au x1000
0.10	437,000	0.20	1,748,000	0.003	1,311
0.20	144,100	0.33	951,060	0.004	576
0.30	57,600	0.46	529,920	0.005	288
0.40	28,500	0.58	330,600	0.005	143
0.50	17,400	0.67	233,160	0.005	87
0.60	9,300	0.78	145,080	0.005	47
0.70	5,700	0.87	99,180	0.006	34
0.80	3,400	0.95	64,600	0.006	20
0.90	1,600	1.06	33,920	0.006	10
1.00	800	1.18	18,880	0.007	6

Table 17.8:
Ruth Inferred Resource

Cutoff Cu%	Tons above cut x1000	Avg Grade Cu %	Contained lbs Cu x1000	Avg Grade Au oz/ton	Contained oz Au x1000
0.10	267,700	0.19	1,017,260	0.003	803
0.20	82,700	0.32	529,280	0.004	331
0.30	29,800	0.47	280,120	0.004	119
0.40	11,400	0.67	152,760	0.004	46
0.50	6,900	0.80	110,400	0.004	28
0.60	4,500	0.95	85,500	0.004	18
0.70	3,500	1.04	72,800	0.005	18
0.80	2,000	1.25	50,000	0.005	10
0.90	1,300	1.49	38,740	0.005	7
1.00	900	1.69	30,420	0.005	5

Table 17.9:
Robinson Measured Resource

Cutoff Cu%	Tons above cut x1000	Avg Grade Cu %	Contained lbs Cu x1000	Avg Grade Au oz/ton	Contained oz Au x1000
0.10	1,239,000	0.31	7,647,600	0.004	5,490
0.20	678,100	0.45	6,065,500	0.005	3,718
0.30	404,900	0.59	4,737,600	0.006	2,330
0.40	264,600	0.70	3,727,800	0.006	1,715
0.50	185,200	0.82	3,027,200	0.006	1,195
0.60	132,700	0.93	2,467,180	0.006	855
0.70	96,400	1.03	1,986,240	0.007	697
0.80	68,000	1.15	1,560,080	0.007	476
0.90	47,400	1.29	1,221,240	0.007	332
1.00	33,800	1.42	958,820	0.007	237

Table 17.10:
Robinson Indicated Resource

Cutoff Cu%	Tons above cut x1000	Avg Grade Cu %	Contained lbs Cu x1000	Avg Grade Au oz/ton	Contained oz Au x1000
0.10	524,300	0.20	2,132,120	0.003	1,748
0.20	174,400	0.34	1,187,400	0.004	758
0.30	73,000	0.47	693,160	0.005	396
0.40	35,800	0.61	437,180	0.005	186
0.50	21,900	0.72	315,960	0.005	114
0.60	12,300	0.86	211,680	0.005	62
0.70	8,000	0.98	157,140	0.006	46
0.80	5,200	1.10	114,640	0.006	29
0.90	3,000	1.27	76,480	0.006	17
1.00	2,000	1.45	58,000	0.006	12

Table 17.11:
Robinson Inferred Resource

Cutoff Cu%	Tons above cut x1000	Avg Grade Cu %	Contained lbs Cu x1000	Avg Grade Au oz/ton	Contained oz Au x1000
0.10	287,700	0.19	1,077,260	0.003	883
0.20	86,100	0.32	558,520	0.004	351
0.30	31,000	0.48	299,800	0.004	128
0.40	12,400	0.69	170,960	0.004	52
0.50	7,500	0.83	124,680	0.004	31
0.60	4,900	0.99	97,500	0.004	20
0.70	3,800	1.10	83,600	0.005	18
0.80	2,300	1.34	61,580	0.005	11
0.90	1,600	1.58	50,500	0.005	7
1.00	1,200	1.76	42,180	0.005	5

**Table 17.12:
Robinson Measured plus Indicated Resource**

Cutoff Cu%	Tons above cut x1000	Avg Grade Cu %	Contained lbs Cu x1000	Avg Grade Au oz/ton	Contained oz Au x1000
0.10	1,763,300	0.28	9,779,720	0.004	7,238
0.20	852,500	0.43	7,252,900	0.005	4,476
0.30	477,900	0.57	5,430,760	0.006	2,726
0.40	300,400	0.69	4,164,980	0.006	1,901
0.50	207,100	0.81	3,343,160	0.006	1,309
0.60	145,000	0.92	2,678,860	0.006	916
0.70	104,400	1.03	2,143,380	0.007	743
0.80	73,200	1.14	1,674,720	0.007	505
0.90	50,400	1.29	1,297,720	0.007	348
1.00	35,800	1.42	1,016,820	0.007	248

17.2 Veteran Mineral Model

17.2.1 General Comments

Re-estimation of the Veteran resource was completed in May of 2008 utilizing historic drill-hole data, Robinson exploration and in-fill drill holes, pit mapping conducted in the Veteran Pit, and updated geologic interpretations. The Quadra drilling in-filled areas where historic drilling was missing or additional metallurgical data was needed, where verification of existing drilling was desired, or near the edges of the deposit where the historic drill data was less prevalent.

One of the main reasons re-estimation was performed was that some key areas of the deposit were found to be geologically different than previously modeled. The situation proved to be complex in that the classification of ore types could not be made accurately using the data available in the historic drill-hole database. In addition, as mining progressed further from the better-drilled portions of the deposit, differences between the interpreted geology and actual geology became apparent.

Because the Tripp Pit (southeastern portion of the deposit) had essentially been mined out by the time this work was undertaken it was decided to re-estimate only the Veteran deposit (northwestern portion). The existing 2004 MDA resource estimate was retained in the Tripp area but is not reported as part of this resource.

Quadra has been mining in the Tripp-Veteran deposit since 2004 and as such, there are considerable numbers of blast holes and significant production data available to verify estimated grades. This information was used to adjust estimation parameters in order to better represent reported mining production.

Copper, both total and soluble, and gold grades were estimated for the resource. Other elements and metals; iron, zinc, molybdenum, and QLT (Quick Leach Test) values are in the drilling database but were not estimated due to the limited spatial extent of the data.

17.2.2 Deposit Geology and Mineral Zones

For the purposes of the Veteran resource, Quadra chose to model only the mineralized zones (ore types) considered most significant to the production of copper and gold. These zones are defined as hypogene, chalcocite enrichment blanket (CEB), and leach cap. Additionally, there are trivial volumes of rhyolite in this portion of the deposit, which were modeled but for which no grades were estimated. There are dumps and fill materials defined in the model but grades were not estimated into them. Anything not defined as the above mentioned materials, essentially the large volume of material surrounding the main deposit (dominantly hypogene), was called "other" and estimated separately.

The definitions of the mineral zones are basically the same as the mineral zones used in the previous resource model. The exception being the chalcocite enrichment blanket that now includes most materials containing visible chalcocite. Construction of the zones was performed by geologists from the exploration group, the site, the corporate office, and consultants. Cross sections were constructed perpendicular to the strike of the deposit, mineral zones interpreted which were then converted into 3-D computer solids. The deposit strikes approximately 315°, and the sections were drawn on an azimuth of 45°. The mineral zones used in the model are described in Table 17.13

Table 17.13:
Veteran Mineral Zones

Code	Zone
1	Hypogene porphyry
2	Chalcocite-enriched blanket
3	Leach Cap
9	Dump or fill
10	Other

Computer solids outlining dumps and areas of fill were built from drill data and topographic data. Likewise, digitized underground maps were used to build solids representing the underground workings in the deposit area. The resulting solids were used to code the block model along with the mineral zone solids. The dumps and underground voids were excluded from grade estimation and not included in resource tabulations.

Other geologic features including lithology and structural features are largely unchanged from the BHP geologic model. For the most part, these interpretations have been found to be reasonably accurate but were changed where pit mapping or new drilling indicate significant differences between interpretations and actual field conditions.

17.2.3 Copper Sample Data

Drill data appropriate for modeling was extracted from the master database, maintained in Acquire® software, and imported into MineSight® software. The assay intervals were then coded from the 3-D zone solids built from the sectional interpretation. The dumps and underground workings solids were coded into the assay intervals as well as the mineralized zones.

Three historic holes were found with incorrect or unverifiable collar locations or down-hole dips. These holes were excluded in the modeling efforts.

Total copper statistics by zone are provided in Table 17.14 and probability plots can be found in Appendix D.

Table 17.14:
Veteran Total Copper Sample Statistics by Mineral Zone (%, length weighted)

Zone	N (feet)	Mean	Std Dev	CV	Min	Max
Hypogene (1)	94,034.6	0.389	0.502	1.290	0.000	13.200
Chalcocite (2)	22,111.0	1.050	1.552	1.478	0.000	27.200
Leach Cap (3)	138,933.4	0.179	0.503	2.810	0.000	15.150
Other (10)	137,565.7	0.114	0.569	4.991	0.000	22.460

Copper grades were not capped. Grade distribution plots, included in Appendix D, revealed a few high-grade outliers but their influence is limited by neighboring lower-grade samples. This decision is supported by the blast-hole grade reconciliation in which the average resource model grades are lower than blast-hole grades and lower than reported mill feed grades. See Section 23.1.2.3 for a discussion of reconciliation work and Appendix D for grade distribution plots.

17.2.4 Copper Composite Data

During the data review, 101 assay samples were identified that had interval lengths greater than 50 ft (double the composite length of 25 ft). The interval lengths ranged from 58 ft to 385 ft. Of these samples, 85 had total copper grades of 0.01%. It is likely that these 0.01% Cu values were entered into the database instead of nil values. The remaining long samples consisted of 11 Quadra samples taken for metallurgical testing and 5 historic samples having total copper grades greater than 0.01%. All of the samples longer than 50 ft were excluded from compositing.

The total copper grade composites were created by compositing assays into 25 ft lengths producing two composites per 50 ft bench. The influence of composites less than 25 ft, which commonly occur at the end of drill holes, was reduced by length-weighting during grade estimation. Composites were coded from the zone solids in order to partially smooth the boundaries between material types.

Composite statistics by zone are provided in Table 17.15 and probability plots can be found in Appendix D.

Table 17.15:
Veteran Composite Statistics by Mineral Zone (%, length weighted)

Zone	N (feet)	Mean	Std Dev	CV	Min	Max
Hypogene (1)	93,490.5	0.380	0.409	1.075	0.000	9.154
Chalcocite (2)	22,140.3	1.007	1.227	1.227	0.000	8.929
Leach Cap (3)	149,884.1	0.161	0.394	2.442	0.000	11.950
Other (10)	174,582.2	0.087	0.379	4.378	0.000	8.798

17.2.5 Copper Block Model

The 3-D mineral solids were used to code the mineral zone codes into blocks in the MineSight® block model. Included in the coding were the codes for dumps, fill, and rhyolite. Coding of blocks was verified by comparison of the 3-D solids volumes with block volumes as well as visual checks made on section and plan. Several iterations were made during which the zone solids were adjusted to correct minor geometry issues.

The size and location of the block model were unchanged from the 2004 MDA model in order to maintain compatibility. The model dimensions are summarized in Table 17.16.

Table 17.16:
Tripp-Veteran Model Dimensions

(in feet)	Minimum	Maximum	Block size	number of blocks
Easting	89000	98000	50	180
Northing	100000	108000	50	160
Elevation	5500	7500	50	40

17.2.6 Copper Estimation

Once composites were created, geostatistical analyses were undertaken. Correlograms were constructed from the composites by domain, in numerous directions, dips, and tilts. In general the correlograms exhibited excellent structures to which spherical models were fitted. These models provided the base parameters for grade estimation using kriging.

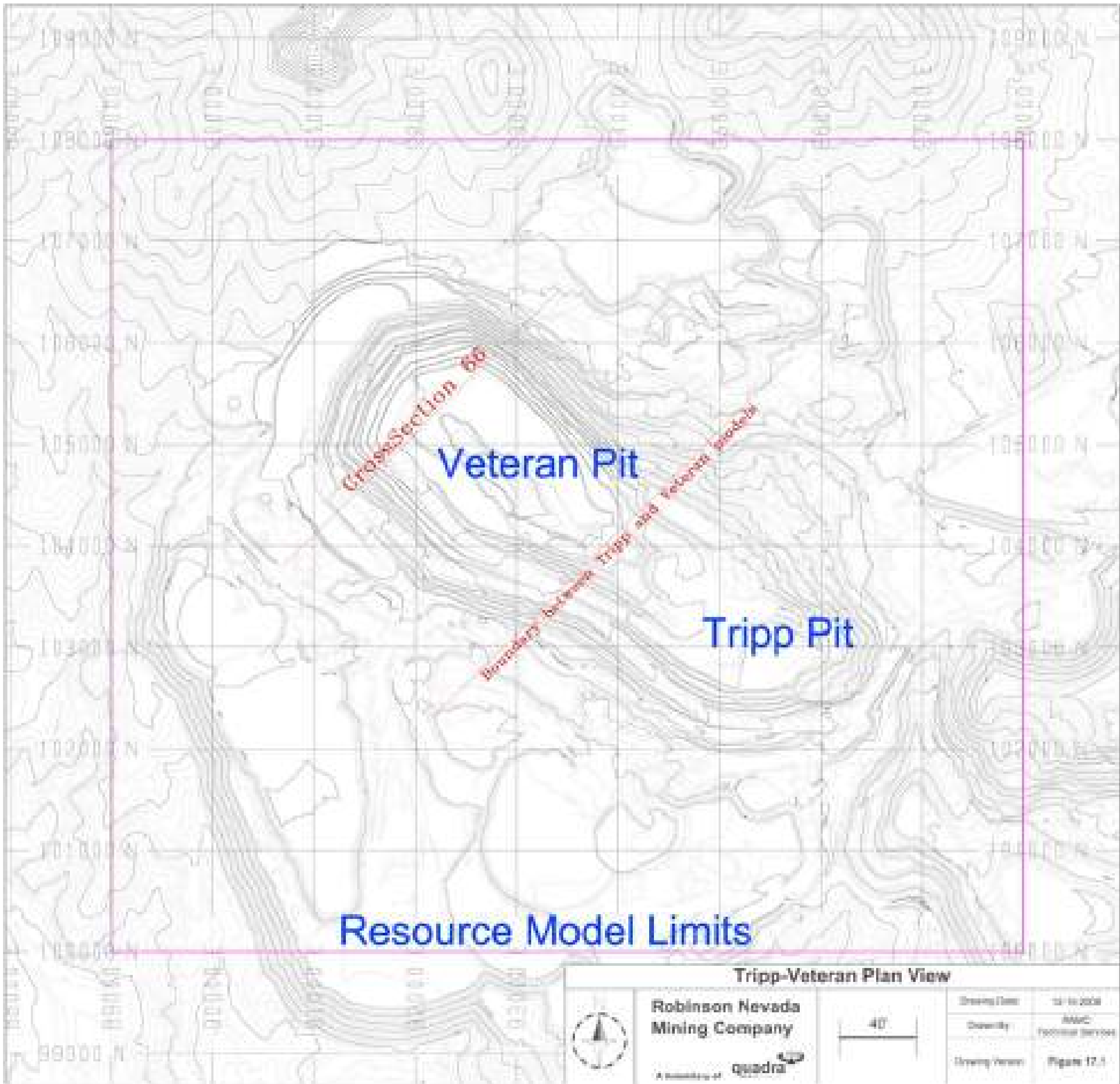
MineSight® Data Analyst was used for the majority of this work. The variograms and estimation parameters are provided in Appendix A.

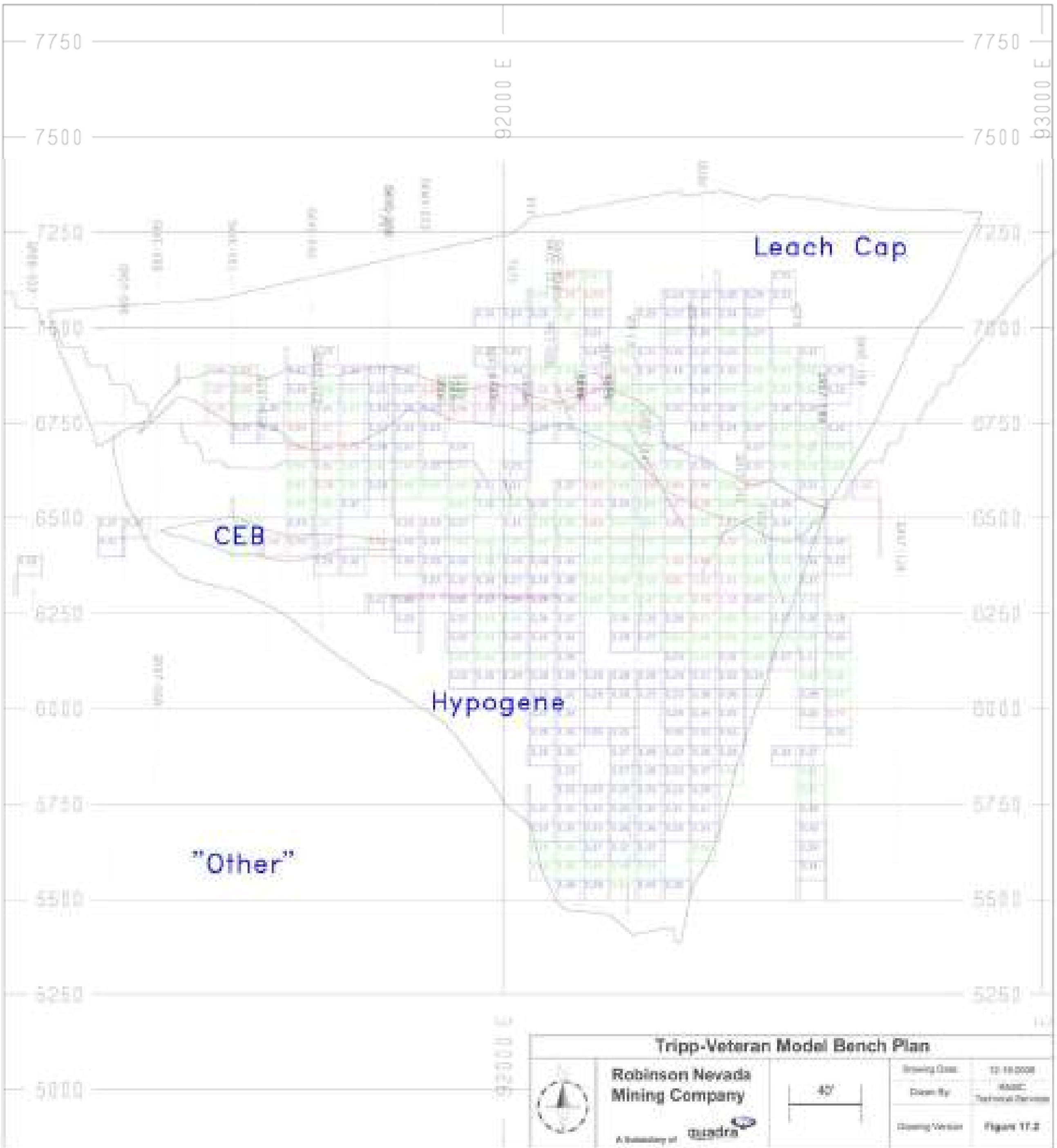
The copper grades were estimated using kriging. Nearest neighbor (polygonal) and inverse distance methods were used to validate the estimate. All three methods yielded similar global results. Each mineral zone was estimated separately using only composites from the specific zone.

Because there are a significant number of blast holes in the modeled area, it was decided to adjust the estimation parameters so that the resource estimate reasonably predicted the blast holes if possible. An iterative process of changing a parameter or parameters and then comparing the resource model grades with blast-hole grades was used to establish the final kriging parameters. With some effort, it was possible to obtain a reasonable prediction of the blast-hole grades while still honoring the composite variogram models. Blast holes that were available as of April 30, 2008 were used in this work. During the period after the resource model was completed in May 2008 and the writing of this report, several thousand blast holes have been added to the database and reconciliation results are different from the results used to establish the estimation parameters. The reconciliation is reviewed on a regular basis and if the resource model is found to be significantly different from reported values, changes will be made to the estimation parameters and the model re-estimated.

A general plan map is shown in Figure 17.1 and a cross section and bench plan of the model and composites are shown in Figures 17.2 and 17.3.

Descriptive statistics by zone of the model are shown in Table 17.17.


Veteran Pit
Tripp Pit
Resource Model Limits
CrossSection 66
Boundary between Tripp and Veteran Models
Tripp-Veteran Plan View
Robinson Nevada
Mining Company
quadra
Figure 17.1


7750
7500
7250
7000
6750
6500
6250
6000
5750
5500
5250
5000
92000 E
93000 E
Leach Cap
CEB
Hypogene
"Other"
Tripp-Veteran Model Bench Plan
Robinson Nevada
Mining Company
40'
Figure 17.2

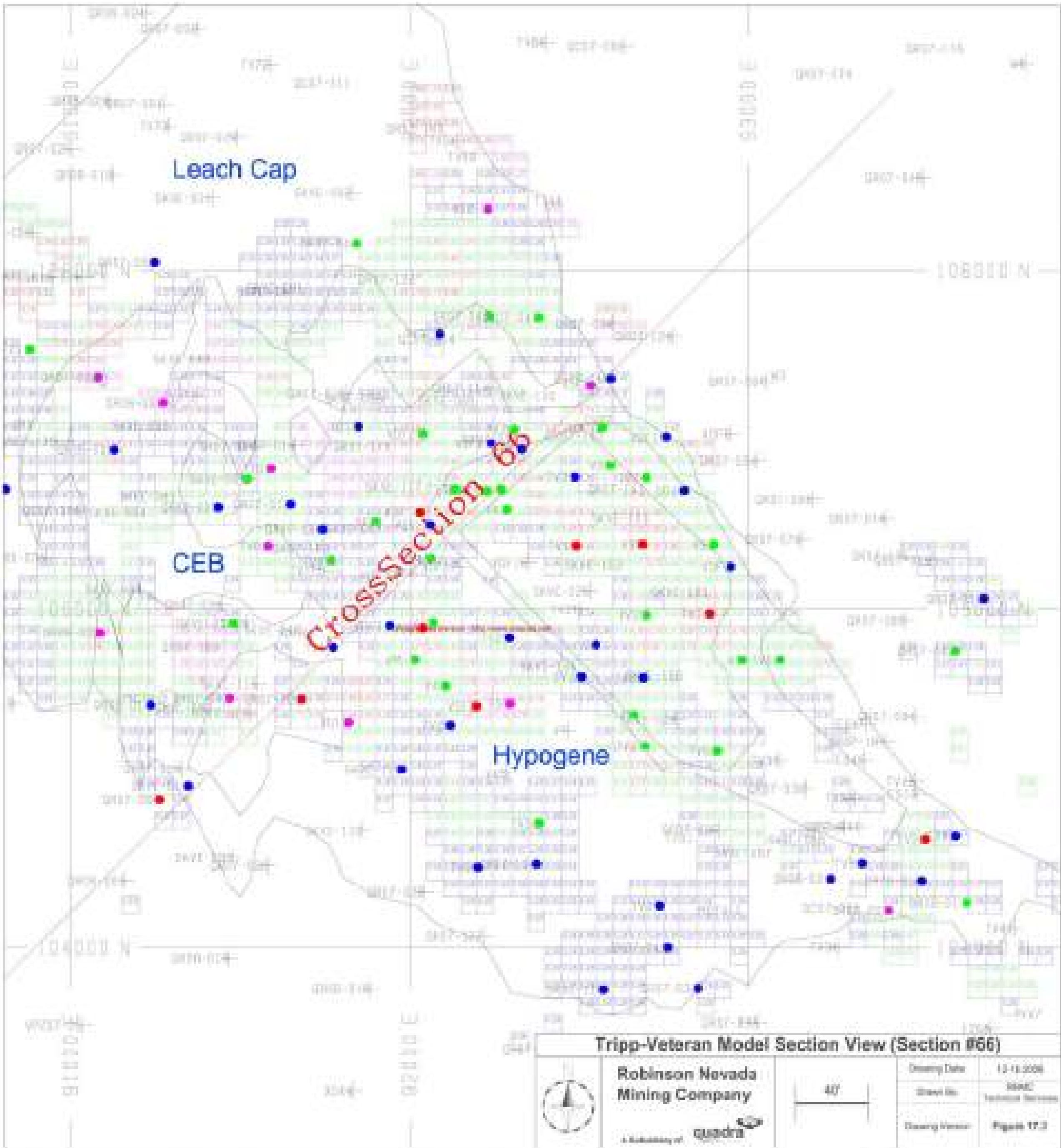

Leach Cap
CEB
CrossSection 66
Hypogene
106000 N
104000 N
Tripp-Veteran Model Section View (Section #66)
Robinson Nevada
Mining Company
40'

Table 17.17:
Veteran Model Total Copper Statistics by Zone (%)

Zone	N (blocks)	Mean	Std Dev	CV	Min	Max
Hypogene (1)	27,475.0	0.2832	0.263	0.927	0.001	6.296
Chalcocite (2)	2,749	0.614	0.671	1.092	0.000	6.878
Leach Cap (3)	33,696.0	0.144	0.228	1.582	0.000	4.056
Other (10)	256,911	0.043	0.123	2.834	0.000	8.528

17.2.7 Gold General Comments

A review of gold grade distributions was made and from that review it was determined that the distribution of gold values were not influenced by the copper mineral zones. As with the previous model, it was decided to estimate gold grades with a single set of parameters throughout the entire deposit. Dump, fill, and rhyolite did not have grades estimated.

17.2.8 Gold Sample Data

The Gold assay statistics are provided by zone in Table 17.18.

Table 17.18:
Statistics for Gold Assays by Zone (oz/t, length weighted)

Zone	N (feet)	Mean	Std Dev	CV	Min	Max
Hypogene (1)	87,366.2	0.009	0.010	1.111	0.000	0.193
Chalcocite (2)	17,411.0	0.010	0.014	1.400	0.000	0.261
Leach Cap (3)	123,883.4	0.006	0.011	1.833	0.000	0.435
Other (10)	146,397.8	0.004	0.011	2.750	0.000	0.636

17.2.9 Gold Composite Data

Gold grades were composited using the same method as total copper. No capping was applied again due to restricted influence of higher-grade samples and the indication from reconciliation that the resource gold grades are lower than reported mill and blast-hole grades.

Gold composite statistics are provided in Table 17.19.

Table 17.19:
Statistics for Gold Composites by Zone (oz/t, length weighted)

Zone	N (feet)	Mean	Std Dev	CV	Min	Max
Hypogene (1)	85,478.6	0.009	0.008	0.931	0.000	0.082
Chalcocite (2)	16,885.1	0.010	0.011	1.029	0.000	0.156
Leach Cap (3)	119,336.8	0.006	0.008	1.274	0.000	0.230
Other (10)	144,735.8	0.004	0.009	2.237	0.000	0.350

17.2.10 Gold Estimation

Gold grades were estimated into the copper block model using kriging and verified by nearest neighbor (polygonal) and inverse distance methods. The entire deposit was estimated as a single unit except for dumps, fill material, and the rhyolite which were not estimated. The variograms and estimation parameters can be found in Appendix B. Basic statistics of the gold estimate are given in Table 17.20.

Table 17.20:
Statistics for Gold Model by Zone (oz/t)

Zone	N (blocks)	Mean	Std Dev	CV	Min	Max
Hypogene (1)	28,044.0	0.007	0.005	0.721	0.000	0.052
Chalcocite (2)	2,771	0.009	0.006	0.629	0.001	0.052
Leach Cap (3)	32,340.0	0.005	0.005	0.944	0.000	0.072
Other (10)	149,741	0.003	0.004	1.250	0.000	0.073

17.2.11 Soluble Copper Model

Soluble copper data is limited, about half the number of samples as total copper, and generally not as reliable as the total copper data. There is little documentation describing the assay methods used to determine the soluble copper grades in the historic database. There are also many situations where several hundred feet of drill samples have been combined and assayed as single samples. Considering these situations, the accuracy of a soluble copper model would be low. However, the re-assaying done by Robinson combined with the new drilling (2006-2008) provided higher-confidence data in areas within or near the existing and planned pit. An estimate of soluble copper grades in those areas could be relied upon with more confidence than in areas with only historic data.

The soluble copper estimate was performed in the same manner as the total copper estimate with the exception that the ratio of soluble copper to total copper was calculated and stored in the assay database. The ratios were then composited in the same way as the total copper assays were. The ratio was then estimated into the model blocks and the soluble copper grades calculated in each block by multiplying the ratio by the estimated total copper grade. The ratios were limited to a maximum of 1.00 so that the resulting soluble copper grades would not exceed the total copper grades.

Statistics for the soluble assays, composites, and model blocks are given in Table 17.21.

Table 17.21:
Statistics for Soluble Copper Grades by Zone

Soluble Copper Assays (%, length weighted)

Zone	N (feet)	Mean	Std Dev	CV	Min	Max
Hypogene (1)	66,569.1	0.044	0.112	2.545	0.000	4.020
Chalcocite (2)	16,102.0	0.194	0.340	1.753	0.000	4.420
Leach Cap (3)	70,427.1	0.082	0.380	4.634	0.000	11.800
Other (10)	25,073.2	0.066	0.495	7.500	0.000	20.010

Soluble Copper Composites (%, length weighted)

Zone	N (feet)	Mean	Std Dev	CV	Min	Max
Hypogene (1)	64,728.5	0.044	0.090	2.043	0.000	1.556
Chalcocite (2)	15,647.3	0.189	0.261	1.381	0.000	1.724
Leach Cap (3)	66,266.3	0.074	0.252	3.405	0.000	5.720
Other (10)	24,393.3	0.055	0.259	4.719	0.000	7.579

Soluble Copper Blocks (%)

Zone	N (blocks)	Mean	Std Dev	CV	Min	Max
Hypogene (1)	27,475	0.024	0.051	2.139	0.000	1.578
Chalcocite (2)	2,749	0.153	0.262	1.709	0.000	2.911
Leach Cap (3)	33,696	0.042	0.110	2.616	0.000	2.591
Other (10)	256,911	0.006	0.051	8.845	0.000	2.769

Reconciliation in the Veteran Pit shows very poor soluble copper correlation between the resource model, blast holes and mill feed. Averages for 2008 (through October) indicate that the blast-hole soluble copper grades are about 30% lower than mill feed grades and the resource model soluble copper grades are about 50% higher than mill feed grades. At the time of this writing there has been no clear resolution to this situation and it remains under review. Estimation parameters and variograms are presented in Appendix C.

17.3 Ruth Mineral Model

17.3.1 General Comments

The resource and reserve estimation for the Ruth deposit was completed in June of 2008. The estimate included total copper, soluble copper and gold values. Molybdenum was also estimated but not reported as a resource or reserve. No mining has taken place by Quadra in the Ruth deposit to date; however, previous owners have mined portions of the deposit using both open pit and underground methods. The previous resource and reserve work on the Ruth deposit completed in 2004 by MDA was available for this study, along with a significant amount of data added by Robinson. Robinson is involved in drilling a series of reverse circulation and core drill holes in the Ruth area in addition to re-assaying samples from historic drill holes that were in storage at the mine site. All the pre-existing data, as well as any new drill data and geologic information completed by June of 2008, was used in this resource-reserve study. As additional data is collected from the Ruth deposit, the resource calculation will be updated.

Detailed geologic interpretations for the Ruth deposit were completed by previous owners. These included models of lithology, mineral content, and alteration. Robinson is in the process of refining this information with new drilling, re-assaying of historic samples, and surface mapping. Current mining by Robinson in the adjacent Tripp-Veteran deposit is also adding to our understanding of the underlying geologic controls.

17.3.2 Deposit Geology Pertinent to Resource Estimation

The geology of the Ruth deposit is largely controlled by the emplacement of the Cretaceous quartz monzonite porphyry in a series of sedimentary rocks. The geology is complicated by both pre- and post-mineral faulting as well as minor post-mineral rhyolite intrusions. The spatial location of the ore body correlates well with the porphyritic intrusion and the units directly adjacent to it.

The Ruth deposit has a well-developed surface leach cap and underlying chalcocite-enrichment zone (supergene), both of which are situated directly over the copper hypogene mineralization. These features have impact on both metal content and proper metallurgical treatment of the ore; therefore, a proper understanding of their location and relationships is critical to successfully modeling and mining the deposit. Other characteristics of the geology, such as structure, alteration, and mineralization, are also important in understanding the ore body. It was noted in modeling the chalcocite enrichment zone that greater concentrations of chalcocite did occur in the classic tabular or blanket form but also correlated with areas of structural disturbance, which was likely due to open-space and ground water movement.

The copper mineral domains used for resource modeling were defined by the supergene zone overlying the hypogene copper occurrence. The mineral domains (ore-types) were broken down into the leach cap, the chalcocite-enrichment blanket, and the hypogene zone. Surrounding areas not classified as one of those zones were grouped together in the "other" category. Each of these domains was interpreted in section, and then a 3-D solid was created and used to code both the drill data and the block model.

The gold mineralization in the Ruth deposit does not correlate directly with the copper mineralization or the supergene-hypogene zones. There is a broader relationship evident between the gold and copper occurrence, with higher gold values often adjacent to or overlapping copper mineralization. Although some gold is associated with nearly all of the copper resource, zones of higher grade gold values can be seen near the margins of the intrusive. Previous operators have mined these areas based on gold content alone. No boundaries were used to constrain the gold modeling in the Ruth deposit.

17.3.3 Resource

The resource estimate for the Ruth deposit utilized a newly created ore-type model to control the copper grade estimation and assign metallurgical characteristics used in ore processing and economic determinations. The gold resource was also determined but no geologic boundaries were used in that estimation. The grade estimations for both copper and gold used the inverse distance squared algorithm with anisotropic search ellipses where appropriate. The resource classification was based on the distance to the nearest sample and the number of samples used for the block being estimated. MineSight® mining software was used for all modeling, estimation, and reporting.

17.3.4 Geologic Model

Previous work on the Ruth deposit utilized various geologic features to model the deposit, including lithologic boundaries, mineralogical and alteration zones, and structural blocks. The most current work completed by MDA in 2004 used a mineralogical model to bound the deposit. The model was a combination of supergene-hypogene zoning, lithology, and grade distribution. The model completed in June of 2008 by Quadra uses an ore-type model to control copper estimation. The ore-type model separated the leach cap, chalcocite-enriched blanket, and hypogene areas as separate and distinct zones. The drill data and block model were coded by the 3-D solids that represent this interpretation. For copper, these boundaries represent major breaks in ore grade, mineralogy, and metallurgical characteristics that determine profitability.

The ore-type model was developed based on historic data, new drilling, re-assaying of previous drilling, surface mapping, pit mapping, and information from current mining. The model addresses both grade and metallurgical issues relevant to processing and scheduling. The model was initially built in cross sections and refined until 3-D solids were created that could then be used to code drill holes and blocks. The modeling of the central chalcocite-enrichment zone was critical due to metallurgical implication. The zone was defined using new detailed logging that specifically identified chalcocite as a key mineral. With the addition of QLT (quick leach test) assays to the database, we were able to use QLT and soluble copper assays to better define areas of chalcocite enrichment. Cross section relationships, pit mapping, logging, grade tenor, and QLT-soluble copper assays were all used to determine the zone boundaries. Three-dimensional solids were also created for the historic dumps and underground workings in order to separate these areas for resource estimation and reporting.

There are several areas in development that could impact the future Ruth resource, although the impact will be limited.

- **New Drilling:** On-going drilling of both reverse circulation and core drill holes in the Ruth deposit will improve grade estimation, geologic interpretation, and metallurgical classification.
- **Historic Drilling:** By re-assaying historic drill-hole samples the database will be improved, providing more accurate analytical work, and increased confidence in the data set. Additional data is being collected (such as QLT) which can improve the modeling process.
- **Surface and Drill Data:** Consistent logging of both new and old drill holes in conjunction with geologic mapping will provide a more cohesive understanding of the geologic controls. Updated ore-type, lithology, and alteration models can then be completed.

17.3.5 Mineral Model – Copper

17.3.5.1 General Comments – Copper

Copper grades correlate well with mineralogy, which can be spatially understood based on the leach cap, chalcocite-enriched, and hypogene zones. No reserves exist in the upper leach cap where copper oxides and halloysite can be found. The transitional chalcocite-enriched zone is defined by secondary chalcocite and higher copper grades. Both copper oxide and sulfide can be found in this zone. The hypogene zone is defined by sulfide minerals with common pyrite and chalcopyrite. Copper grades and mineralogy are more uniform and consistent in the hypogene zone. The mineralization in the Ruth deposit was modeled using the zones listed in Table 17.22

Table 17.22:
Copper Mineral Zones

Code	Zone
1	Hypogene porphyry
2	Chalcocite-enriched blanket
3	Leach cap
4	Other

17.3.5.2 Copper Sample Data

The drill-hole database maintained by Quadra employees at the Robinson mine was reviewed and exported with only data that had been determined to be complete and accurate. The assay files were then coded from the ore-type solids. The codes from the ore-type solids were back-loaded to the drill-hole and block model files. This was done by initially coding all blocks as zone 4 (other) and then overwriting those codes with codes 1, 2, and 3 respectively from the ore-type solids. The drill-hole intervals logged as "dump" were then used to exclude the areas with dump material from the estimation and resource reporting. The statistics by ore-type zone are given in Table 17.23.

Table 17.23:
Descriptive Statistics for Copper Assays by Zone

Ruth
Copper Assays (%, length-weighted)

Code	Zone	N	Mean	CV	Min	Max
1	Hypogene porphyry	48,242	0.36	1.27	0.00	9.60
2	Chalcocite-enriched blanket	25,744	0.74	1.85	0.00	40.46
3	Leach cap	41,930	0.14	3.61	0.00	21.57
4	Other	8,601	0.13	2.48	0.00	8.26
	Dumps	dump assays excluded-not modeled				

Based on the statistics of each zone and the grade distribution plots, included in Appendix H, outlier grades were capped. The capping grades used for each deposit are provided in Table 17.24.

Table 17.24:
Capped Grades for Copper Assays by Zone

Ruth
Copper Assays

Code	Zone	N	Cap Grade (%Cu)
1	Hypogene porphyry	39	5.00
2	Chalcocite-enriched blanket	68	12.00
3	Leach cap	33	9.00
4	Other	26	4.00

17.3.5.3 Copper Composite Data

Copper data was composited with geologic matching codes so composites were broken at modeled ore-type boundaries. Twenty-five foot down-hole composites were used and were length-weighted to account for any length variation. The capped assays values were used in compositing. Descriptive statistics of the copper composite grades are given in Table 17.25. Grade distribution plots of the composites by zone are included in Appendix H for the Ruth deposit.

Table 17.25:
Descriptive Statistics for Copper Composites by Zone

Ruth
Copper Composites (%, length-weighted)

Code	Zone	N	Mean	CV	Min	Max
1	Hypogene porphyry	12,152	0.35	1.15	0.00	5.00
2	Chalcocite-enriched blanket	6,486	0.75	1.51	0.00	12.00
3	Leach cap	12,475	0.15	2.97	0.00	9.00
4	Other	2,821	0.14	1.83	0.00	4.00
	dumps	dump assays excluded-not modeled				

17.3.5.4 Copper Block Model

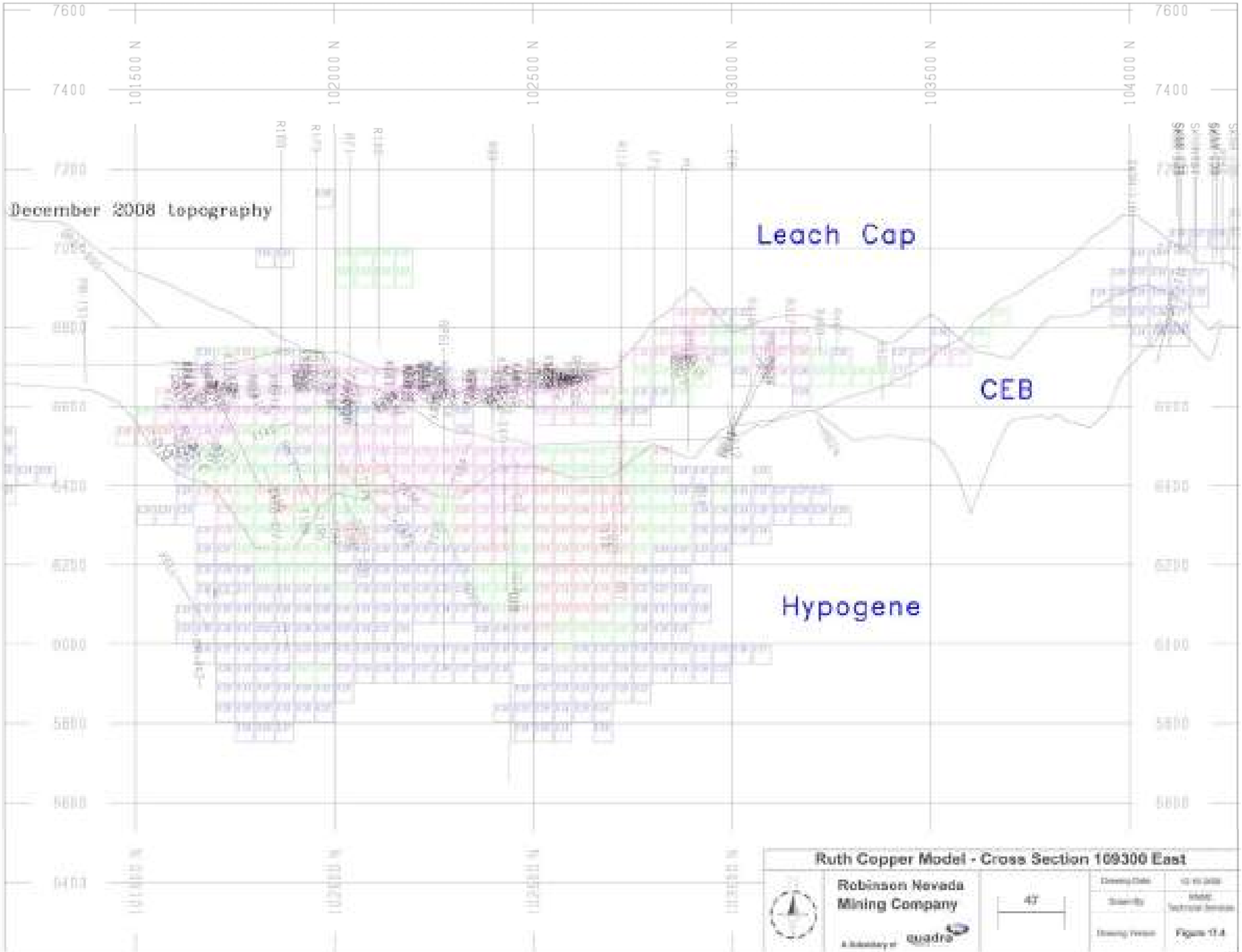
A block model was created with fifty foot square blocks and fifty foot benches. The blocks were coded with current topography and dumps. The geologic zones were added which included ore-type zones, structure, lithology, and alteration. The rock densities established from previous work were used for the Ruth model. Density tests are underway on new core samples to confirm and refine the existing density values. The ore-type block codes were assigned to the block model based on the 3-D solids previously created for the ore-type model. The areas consisting of dumps were coded using a 3-D solid created by mine staff that incorporated current surveys and records from previous mining. Coding and estimation were done for all areas determined to be below the pre-mining surface. Resource reporting was based on the current topography and in-situ rock, so only in-place material was considered to be resource.

17.3.5.5 Copper Estimation

Variograms were calculated from the composite file using varying directions and lag distances. They were calculated separately for each ore-type zone and used to determine anisotropic search parameters for the estimation of copper grades. They are provided in Appendix E. The inverse distance squared algorithm (ID^2) was used for copper grade estimation and anisotropic search ellipses were used where appropriate. Estimation parameters for the Ruth deposit are given in Appendix E.

A cross section of the model is shown in Figure 17.4 and a plan shown in Figure 17.5.

Descriptive statistics by zone of the block model are shown in Table 17.26.


December 2008 topography
Leach Cap
CEB
Hypogene
7600
7400
101500 N
102000 N
102500 N
103000 N
103500 N
104000 N
Ruth Copper Model - Cross Section 109300 East
Robinson Nevada Mining Company

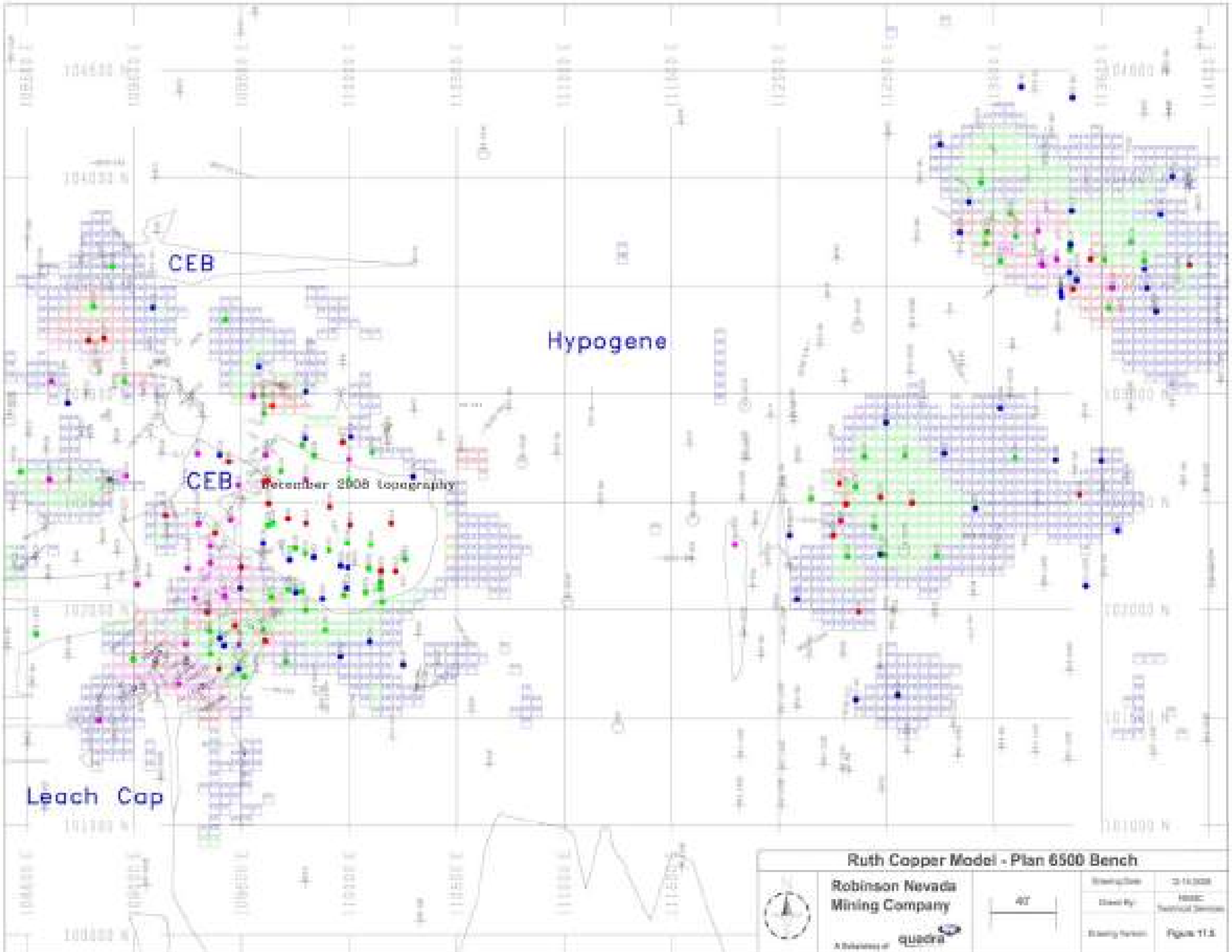

CEB
Hypogene
CEB
December 2008 topography
Leach Cap
Ruth Copper Model - Plan 6500 Bench
Robinson Nevada
Mining Company
quadra

Table 17.26:
Descriptive Statistics for Copper Composites by Zone

Model Total Copper Statistics by Zone (%)

Code	Zone	N	Mean	CV	Min	Max
1	Hypogene porphyry	211,394	0.16	1.13	0.00	2.73
2	Chalcocite-enriched blanket	16,038	0.26	1.32	0.00	6.98
3	Leach cap	75,324	0.10	1.38	0.00	2.93
4	Other	90,152	0.05	2.40	0.00	2.58
	dumps	dump assays excluded-not modeled				

17.3.5.6 Copper Estimation Checks

As a model validation check a comparison was made between composites, polygonal, and ID^2 grade frequencies. The ID^2 model shows some smoothing but overall provided a reasonable match. The model checks are presented in Appendix H.

17.3.6 Mineral Model – Gold

17.3.6.1 General Comments – Gold

The gold database was exported from the database maintained by Quadra employees at the Robinson Mine. The historic gold and copper data can exist in the database exclusive of one another. For example many of the historic drill holes were assayed for copper and not gold, while a previous gold operator assayed for gold and not copper. There are distinct higher grade gold areas in the Ruth deposit, some of which were mined by a previous operator (Alta Gold) that do not correlate with higher copper values. There is a spatial relationship between the gold and the porphyry but the gold does not directly correlate with copper concentrations. The gold assay values are statistically higher in the upper leach cap and chalcocite-enriched zone relative to the hypogene grades. The gold values do increase above the hypogene zone with higher mean and maximum grades. The lower gold values in the hypogene area suggest secondary enrichment in the overlying oxidized units or simply higher density drilling in the gold deposits. The gold mineralization controls are not as obvious as those for copper and therefore were not broken out by zone. The gold assays for the Ruth deposit showed several population breaks based on the grade distribution plots. The population breaks identified for the gold assays mineral domains are given in Table 17.27.

Table 17.27:
Population Breaks for Gold

Ruth	
Gold (oz/t)	0 - 0.004 - 0.02 - 0.15 - 0.4

17.3.6.2 Gold Sample Data

The gold assay statistics by zone are given in Table 17.28. No gold values were reported in the areas that were mined out or determined to be dumps. The higher gold values associated with the hypogene zone are of note, as well as an increase in the coefficient of variation.

Table 17.28:
Descriptive Statistics for Gold Assays by Zone

Ruth
Gold Assays (oz/t, length-weighted)

Code	Zone	N	Mean	CV	Min	Max
1	Hypogene porphyry	42,020	0.004	1.75	0.000	0.784
2	Chalcocite-enriched blanket	16,960	0.005	1.20	0.000	0.183
3	Leach cap	64,781	0.009	3.00	0.000	1.660
4	Other	23,554	0.006	3.50	0.000	1.550
	Dumps	dump assays excluded-not modeled				

Capping for gold assay grades was based on population distributions with a maximum value of 0.5 opt allowed. Gold capped values are shown in Table 17.29.

Table 17.29:
Capped Grades for Gold Assays by Zone

Ruth
Gold Assays

Code	Zone	N	Gold Grade (oz/t)
1	Hypogene porphyry	2	0.500
2	Chalcocite-enriched blanket	0	0.500
3	Leach cap	33	0.500
4	Other	6	0.500

17.3.6.3 Gold Composition Data

The capped gold grades were composited to 25 ft down-hole. Like the copper composites, gold composites that were less than 25 ft in length were length-weighted for use in the resource estimation. Descriptive statistics of the gold composite grades are given in Table 17.30.

**Table 17.30:
Descriptive Statistics for Gold Composites by Zone**

Ruth

Gold Composites (oz/t, length-weighted)

Code	Zone	N	Mean	CV	Min	Max
1	Hypogene porphyry	10,368	0.004	1.28	0.000	0.208
2	Chalcocite-enriched blanket	4,221	0.005	0.92	0.000	0.064
3	Leach cap	14,451	0.006	1.55	0.000	0.364
4	Other	4,936	0.004	3.30	0.000	0.351
	dumps	dump assays excluded-not modeled				

17.3.6.4 Gold Block Model

The gold block model is the same block model built for copper. Current mining practice is to mine gold as a secondary product to copper, and therefore it is always mined with copper. The gold model is defined within the existing copper metallurgical/mineralogy model and uses all the other geologic and economic criteria applied to copper.

17.3.6.5 Gold Estimation

Gold in the Ruth deposit was estimated with an inverse distance squared algorithm using anisotropic searches where appropriate. Variograms were calculated from the composite file with varying lag lengths and directions. Representative variograms and gold estimation parameters for Ruth are given in Appendix F.

As a model validation check a comparison was made between composites and ID^2 grade frequencies the results are presented in Appendix H.

Gold descriptive statistics by zone of the model are shown in Table 17.31.

**Table 17.31:
Descriptive Statistics for Gold Model by Zone**

Model Gold Statistics by Zone (oz/t)

Code	Zone	N	Mean	CV	Min	Max
1	Hypogene porphyry	210,411	0.00	1.24	0.00	0.11
2	Chalcocite-enriched blanket	16,172	0.00	0.74	0.00	0.04
3	Leach cap	75,285	0.00	1.51	0.00	0.15
4	Other	79,363	0.00	2.10	0.00	0.11
	dumps	dump composites excluded-not modeled				

17.3.7 Soluble Copper Block Model

Soluble copper values are available for part of the Ruth dataset. Only 52% of the samples with total copper assays also have soluble copper determinations. The soluble copper values provide an approximation of copper occurring as oxides, which is valuable in predicting mill recovery and material types. Additional soluble copper data is being collected with new drilling and the re-assay of existing samples. The quality of the historic soluble copper assays is not documented but the newer data compares reasonably well and confirms the overall tenor of the historic grades.

The estimation for soluble copper was made by first calculating the ratio of soluble copper to total copper in the assay database with a maximum ratio of 1.0 allowed so the soluble copper grade will not exceed the total copper value. The ratio was composited over the same intervals as copper and gold and interpolated using the ID^2 algorithm. The interpolated ratio was then multiplied by total copper values to get the block estimate for soluble copper. The modeled soluble copper values should only be considered a reasonable approximation and not a precise determination.

Statistics for the soluble copper assays, composites, and model blocks are given in Table 17.32.

Table 17.32:
Statistics for Soluble Copper Grades by Zone

Ruth

Soluble Copper Assays (%, length-weighted)

Code	Zone	N	Mean	CV	Min	Max
1	Hypogene porphyry	29,349	0.05	3.04	0.00	7.32
2	Chalcocite-enriched blanket	12,060	0.14	2.81	0.00	13.00
3	Leach cap	18,077	0.08	2.68	0.00	5.71
4	Other	5,439	0.09	3.38	0.00	7.36

Soluble Copper Composites (%, length-weighted)

Code	Zone	N	Mean	CV	Min	Max
1	Hypogene porphyry	7,930	0.05	2.49	0.00	3.77
2	Chalcocite-enriched blanket	3,370	0.13	2.22	0.00	5.86
3	Leach cap	4,894	0.09	2.11	0.00	3.66
4	Other	1,718	0.09	2.89	0.00	5.03

Soluble Copper Blocks (%)

Code	Zone	N	Mean	CV	Min	Max
1	Hypogene porphyry	182,312	0.02	1.36	0.00	1.21
2	Chalcocite-enriched blanket	15,320	0.06	1.84	0.00	2.02
3	Leach cap	77,333	0.05	1.69	0.00	1.31
4	Other	45,245	0.04	2.55	0.00	2.06
	dumps	dump blocks excluded-not modeled				

Soluble copper variograms and estimation parameters are in Appendix G, and model checks are in Appendix H.

17.4 Mineral Reserves

Robinson Tripp-Veteran and Ruth deposit mineral reserves were developed by RNMC personnel under the supervision of Scott Hardy, P. Eng, Manager of Technical Services of Quadra Mining Ltd. Mr. Hardy is a designated Qualified Person as defined by National Instrument 43-101. Historic methodology, parameters and criteria were used in order to be consistent with past estimates and company policy.

The reserves were estimated using a projected (based on anticipated mining) end of year 2008 topography as a starting surface. The updated resource model incorporating results from 2006, 2007, and 2008 exploration drilling was utilized in calculating the reserve estimate. The reserves reflect anticipated metal prices, recoveries, and operating costs. Table 17.33 summarizes the ore reserves as of January 1, 2009.

Table 17.33:
Veteran & Ruth Copper & Gold Mineral Reserves as of January 1, 2009

Tripp-Veteran Area

Reserve Classification	Ore Tons (000)	Cu Grade (%)	Au Grade (opt)	Contained Metal		Waste Tons (000)	Total Tons (000)	Strip Ratio
				Cu Tons (000)	Au oz (000)			
Proven	38,564	0.49%	0.010	189	395			
Probable	575	0.35%	0.007	2	4			
Proven and Probable	39,139	0.49%	0.010	191	399	100,912	140,051	2.58

Ruth Area

Reserve Classification	Ore Tons (000)	Cu Grade (%)	Au Grade (opt)	Contained Metal		Waste Tons (000)	Total Tons (000)	Strip Ratio
				Cu Tons (000)	Au oz (000)			
Proven	90,896	0.57%	0.005	518	484			
Probable	3,522	0.43%	0.005	15	17			
Proven and Probable	94,418	0.57%	0.005	533	501	312,288	406,706	3.31

Stockpiles

Reserve Classification	Ore Tons (000)	Cu Grade (%)	Au Grade (opt)	Contained Metal		Waste Tons (000)	Total Tons (000)	Strip Ratio
				Cu Tons (000)	Au oz (000)			
Proven	585	0.68%	0.009	4	5			
Probable								
Proven and Probable	585	0.68%	0.009	4	5	0	585	0.00

Total Robinson

Reserve Classification	Ore Tons (000)	Cu Grade (%)	Au Grade (opt)	Contained Metal		Waste Tons (000)	Total Tons (000)	Strip Ratio
				Cu Tons (000)	Au oz (000)			
Proven	130,045	0.55%	0.007	711	884			
Probable	4,097	0.42%	0.005	17	21			
Proven and Probable	134,142	0.54%	0.007	728	905	413,200	547,342	3.08

17.5 Cutoff Strategy

The reserves are based on a cutoff of 3.5 recoverable pounds per ton. Recovery was calculated based on the modified Edmiston equations. Estimated future commodity prices and historic mining costs were used to calculate the economic cutoff grade. Table 17.34 presents a summary of the mining costs used for the cutoff calculations.

Table 17.34:
Cutoff Calculation Economic Parameters

G&A ($/ore ton)	$1.433
Mining ($/mined ton)	$1.416
Processing ($/ore ton)	$3.322
Ground Freight ($/con ton)	$85.513
Ocean Freight ($/con ton)	$65.729
Smelting ($/con ton)	$46.514
Refining ($/Cu lbs)	$0.500

The cutoff grade used for these reserves is based solely on copper value, and does not include by-product credits. In addition to the economic cut-off, the life-of-mine plan uses a floating cutoff strategy intended to maximize near-term copper production. In each time period, all ore blocks are sorted by value (based on recoverable copper only) and the highest value blocks available during the period are routed as ore to the concentrator. The remaining blocks which are above the economic cutoff are routed to a stockpile for processing later in the mine life. It is important to note that the stockpiles section included in the reserve statement only includes material currently in the ore stockpiles and does not include material sent to stockpiles in the future.

17.6 Model Adjustments

As previously stated, the updated resource model incorporating results from 2006, 2007, and 2008 exploration drilling was utilized in calculating the reserve estimate. Due to the potential for variable metallurgical performance, ore tonnage reductions were applied to certain material types in the deposits as follows. Tonnage reductions were applied to individual blocks in the resource model by reducing ore tons by the specified amount and increasing waste tons in the block by the same amount.

In the Veteran deposit tonnage of supergene ore was reduced by 18%. This tonnage reduction is based on historic quantities of Supergene “metallurgical waste” in the Veteran Pit. (“Metallurgical waste” is defined as material which meets economic cutoffs, but is routed as waste due to poor recovery, concentrate grade, or other undesirable metallurgical characteristics.) In addition to the supergene tonnage reduction, all material in the Veteran area above the 6900 bench was classified as waste.

In the Ruth deposit the following reductions were applied.

"Wood ore", all benches, all materials ("Wood ore" is ore in or near underground workings that may contain wood or other items from underground mining.)

Ore tons reduced by 50%
Copper Grades reduced by 30%

Benches above 6500 ft elevation

Hypogene	20% Ore Tonnage Reduction
Supergene	30% Ore Tonnage Reduction

Benches between 6500 and 6000 ft elevation (inclusive)

Hypogene	20% Ore Tonnage Reduction on grades <0.5% TCu No Ore Tonnage reduction on grades >= 0.5% TCu
Supergene	20% Ore Tonnage Reduction

Benches below 6000 ft elevation

Hypogene	No Reductions
Supergene	20% Ore Tonnage Reduction

17.7 Pit Designs

The Veteran area pit design was updated from the previous reserve design (January 1, 2008) to include an additional pushback on the West side of the Veteran Pit. The Ruth area pit designs and phases (including Kimbley and Wedge) were updated to reflect changes in the Ruth Resource Model and to improve operational flexibility. Additional discussion of the pit designs and life-of-mine production schedule can be found in Section 23.

18. Other Relevant Data and Information

18. Other Relevant Data and Information

Quadra started mining at the Robinson property on June 11, 2004 and concentrator operations commenced in late August 2004, with first concentrate production occurring on September 4, 2004. Mining commenced in the Tripp pit, and progressed to the adjacent Veteran pit, where the two pits have been merged. RNMC's mining and processing operations have been continuous from mid-2004 to the present. Annual production rates have varied from approximately 125 million to 160 million pounds of copper per year.

Initially in 2004, mining was undertaken by a contract miner but was taken over by Quadra in late 2005. RNMC has made significant modifications to the processing circuit to improve the functionality and performance of the process plant that included; a change to the semi-autogenous grinding (SAG) feed; installation of a gravity gold recovery system (Falcon Concentrator); addition of a molybdenum recovery circuit; addition of lime slaker mill; and, addition of different filter presses.

Water for the Robinson operations is obtained from pit dewatering and locally-installed groundwater wells. The water rights are sufficient for all of RNMC's mining and processing operations and all environmental permits are in good standing.

There is no other relevant data or information that has major significance to the property known at this time.

19. Interpretation and Conclusions

19. Interpretations and Conclusions

The Robinson operation is a complicated property with a lengthy history and unique challenges. RNMC has been operating the property at a profit for four and a half years and has successfully dealt with the challenges encountered to date. Mining is transitioning to the Ruth area and it is likely that several new challenges will arise. The most significant issues moving forward can be summarized as follows:

- *Metallurgy* Due to the poor historic performance of the Ruth deposit during the operation of the property by Kennecott, efforts are currently underway to improve both the performance and the modeling of that metallurgy. The Ruth deposit has been drilled and available core and pulps have been re-assayed and are being re-processed for modeling of the Ruth pit metallurgy. Testing for recovery and concentrate grade is ongoing and results are being correlated with geologic interpretations to develop accurate metallurgical models.

- *Dewatering* Ruth area dewatering requirements have been estimated with conservative methods that may have overstated/understated the quantity of water required for dewatering the aquifer adjacent to the Ruth pit. Additional work is underway to more clearly define both the quantity and quality of water to be encountered and used/discharged back to the aquifer.

- *Resource Model Performance* During 2008 the ore body model predicted more ore at a lower grade than was actually found. Forecast pounds of recovered copper were lower than the recovered copper pounds reported by the mill. Going forward, this trend is likely to continue at least in the Veteran pit until refinements are made to the model.

20. Recommendations

20. Recommendations

This Technical Report has provided a detailed description of the current status of the Robinson Mine operations. The following recommendations are made to RNMC:

- Actively continue metallurgical test work of materials in the Ruth area;
- Continue to evaluate the resource models and reconciliation with mill grades and update the estimation parameters as needed to better estimate metals grades;
- Explore alternative methods for dewatering the Ruth pit; and
- Evaluate the resource model rock density values.

21. References

21. References

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22. Certificate of Authors

22. Certificate of Authors

I, Scott Hardy, P.E., do hereby certify that:

1. I am currently employed as Manager, Technical Services by:

   Quadra Mining Ltd.
   Suite 2414, Four Bentall Centre
   1055 Dunsmuir Street
   Vancouver, BC Canada V7X 1K8

2. I graduated with a Bachelor of Science degree in General Engineering from Oregon State University in 1978 and Bachelor of Science degree in Geology from the University of Wyoming in 1984.

3. I am a Registered Professional Engineer in the state of Nevada (#11891) and a member of the Society for Mining, Metallurgy, and Exploration, Inc.

4. I have worked as an engineer for a total of 24 years since my graduation from university.

5. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.

6. I am responsible for the preparation of Chapters 1 through 6, and 17 through 23 of this technical report titled: “Technical Report on Robinson Nevada Mining Company Operation, Ruth, Nevada, USA” (“Technical Report”) dated February 12, 2009. I have visited the project numerous times between May 2000 and November 2008. Most recently, I visited the Robinson project from November 18, 2008 to November 20, 2008.

7. I have had prior involvement with the property that is the subject of this Technical Report. That involvement was evaluating production options, evaluating different mining methods, reviewing and reporting on historic resources, sampling, mining engineering work, and preparation of the report titled “Data Audit, Resource Estimate, and Ore Reserves of BHP’s Southwest Copper Property Robinson operation Ruth, Nevada” dated September 28, 2000. I was responsible for preparation of some portions of a technical report on the property titled “Technical Report on Robinson operation, Ruth, Nevada” dated January 31, 2004.

8. As at the date hereof, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

9. I am an employee of the issuer, and therefore not independent of the issuer within the meaning of section 1.4 of NI 43-101.

10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 12th day of February 2009.

Signature of Qualified Person

Scott Hardy
Print Name of Qualified Person

I, David Newhook, P. Eng., do hereby certify that:

1. I am currently employed as Vice President, Project Evaluations by:

   Quadra Mining Ltd.
   Suite 2414, Four Bentall Centre
   1055 Dunsmuir Street
   PO Box 49185
   Vancouver, BC V7X 1K8

2. The title of this report is, “Technical Report on Robinson Nevada Mining Company Operation, Ruth, Nevada, USA”, dated February 12, 2009.

3. I graduated with a Bachelor of Engineering – Metallurgical, from McGill University in 1991.

4. I am a registered Professional Engineer in the province of British Columbia (#22303). I have worked in the mining/metallurgical industry since 1991 both in technical and management positions.

5. I visited the site on December 16, 2008.

6. I am responsible for Chapter 16 – Mineral Processing & Metallurgical Testing of this “Technical Report on Robinson Nevada Mining Company Operation, Ruth, Nevada, USA” (Technical Report) dated February 12, 2009.

7. I am an employee of the issuer, and therefore not independent of the issuer within the meaning of section 1.4 of NI 43-101.

8. I have not had any prior involvement with this property.

9. I have read the definition of a “Qualified Person” set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of this NI 43-101. I certify that this report has been prepared in compliance with the NI 43-101.

10. As at the date hereof, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

11. I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and

any publication by them including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 12th day of February, 2009.

Signature of Qualified Person

David Newhook
Print Name of Qualified Person

I, Patrick L. Fahey, Professional. Geologist, do hereby certify that I am currently employed as Vice President, Exploration by:

Quadra Mining Ltd.
Suite 2414, Four Bentall Centre
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PO Box 49185
Vancouver, BC V7X 1K8

I further certify that:

1. I graduated with a Bachelor of Arts degree in Geology from California State University at Chico in 1974; and a Master of Science degree in Geology from the University of Washington in 1979.
2. I am a Registered Professional Geologist in the state of Arizona (#38,731), a Fellow of the Society of Economic Geologists, and a member of the Geological Society of America, and the Geological Society of Nevada.
3. I have worked as a geologist or attended graduate school for a total of 34 years since my graduation with a Bachelors degree.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "non-independent qualified person" for the purposes of NI 43-101.
5. I supervised the preparation of Chapters 7 through 15 of this technical report titled "Technical Report on Robinson Nevada Mining Company Operation, Ruth, Nevada, USA", (Technical Report), February 12, 2009.
6. I visited the project as a consultant or employee of Quadra Mining Ltd. at least 10 times during the period between April 2005 and December, 2008 for periods of from one to ten days. Most recently, I visited the site from December 5, 2008 to December 11, 2008.
7. Prior to 2005, I worked as a geologist in the Robinson district many times, including much of the period from the spring of 1991 to the summer of 1995, while a consultant and employee for Magma Copper Company, which was then the owner of the Robinson project.
8. As at the date hereof, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information

that is required to be disclosed to make the Technical Report not misleading.

9. I am Vice President, Exploration of Quadra Mining Ltd. the corporation that owns the Robinson property and therefore I am not independent of Quadra within the meaning of section 1.4 of NI 43-101.

10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 12th day of February, 2009.

Signature of Qualified Person

Patrick L. Fahey
Print Name of Qualified Person

23. Additional Requirements for Technical Reports on Development Properties and Production Properties

23. Additional Requirements for Technical Reports on Development Properties and Production Properties

This chapter has been co-authored by Scott Hardy, P. Eng., Manager, Technical Services of Quadra Mining Ltd., and Juris Ore, Technical Services Superintendent of Robinson Nevada Mining Company.

23.1 Mining Operations

23.1.1 Pit Design and Mineable Reserves

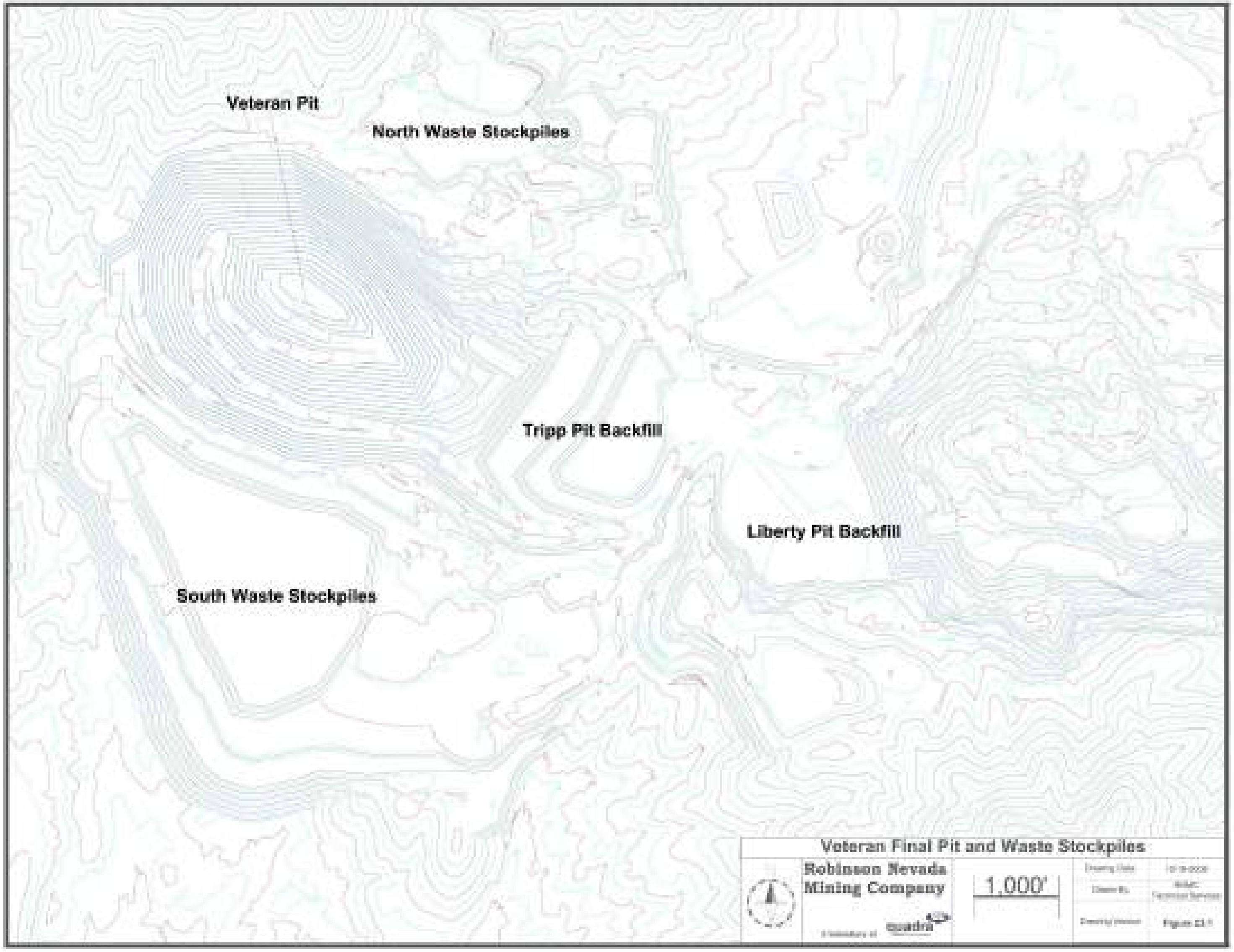
Pit design work and Reserve estimation have been performed by RNMC using internal sources. A discussion of the mineral resources and reserves is found in Section 17.

Final pit and waste stockpiles were designed using MineSight© software. Figure 23.1 and Figure 23.2 are maps of the final pit and waste stockpile designs before reclamation activities.

23.1.2 Mine Planning

RNMC mine planning is designed to provide the best possible ore feed to the concentrator on an annual basis. Mining from the pits is currently scheduled to last into 2017. As of January 1, 2009, RNMC has approximately 585,000 tons of ore in stockpiles. Periodically throughout the life-of-mine plan, low-grade ore will be stockpiled for later processing. This material is above the economic cutoff grade but is stockpiled to improve the efficiency of mine operations, maximize the grade processed by the concentrator, and facilitate blending. By the end of mine life, the stockpiles will contain approximately 16 million tons. As long as economic conditions remain the same or improve, these stockpiles will be processed and the concentrator will continue to run until 2017.

Mine operations occur at several distinct pits that exist on the property and include the Veteran-Tripp pit complex, the Ruth West, Ruth East, Kimbley and Wedge pit areas. At this time, there is no identified resource or reserve associated with the Liberty pit area. Table 23.1 is a summary of the mine production schedule for the life of the mine.


Veteran Pit
North Waste Stockpiles
Tripp Pit Backfill
Liberty Pit Backfill
South Waste Stockpiles
Veteran Final Pit and Waste Stockpiles
Robinson Nevada
Mining Company
1,000'

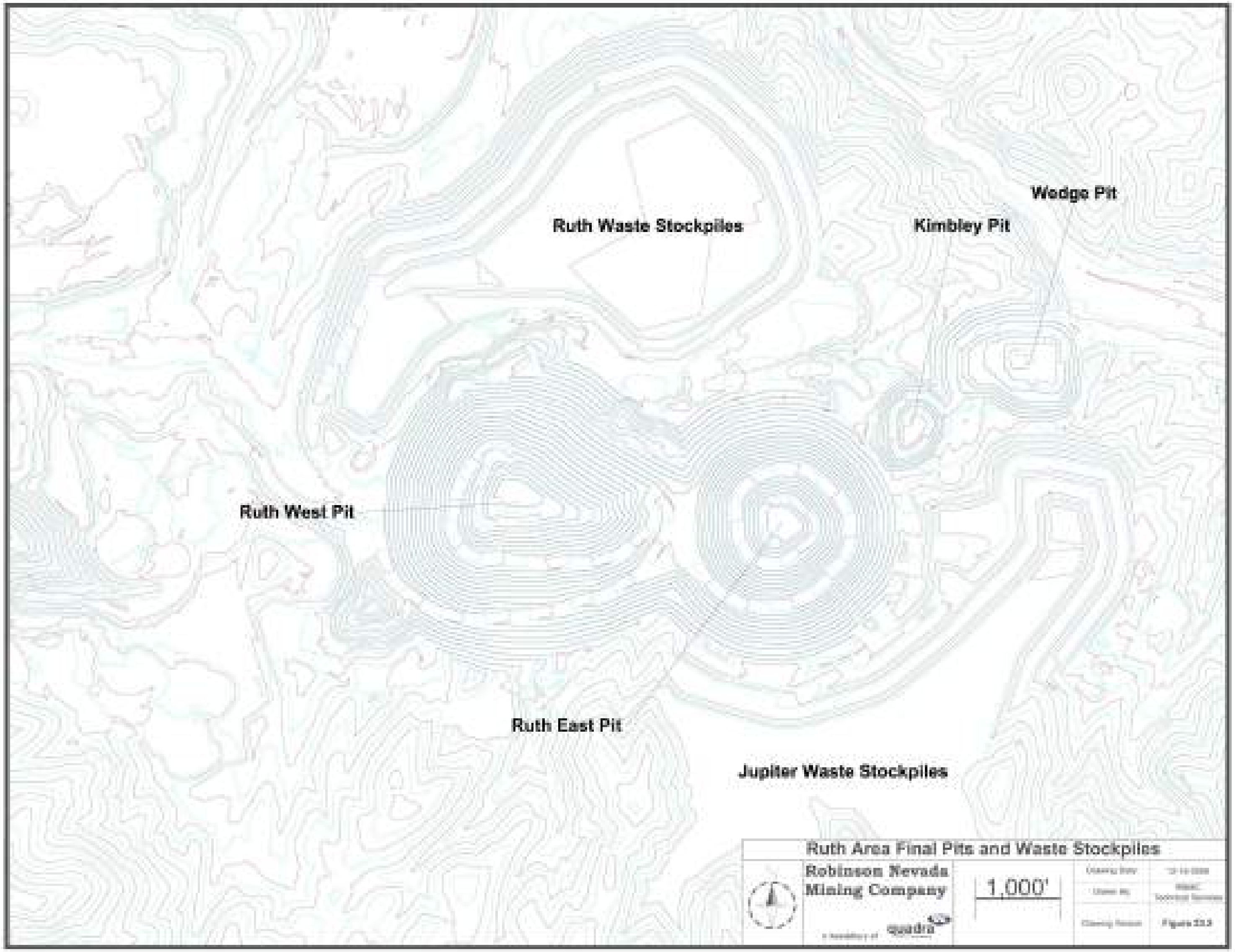

Wedge Pit
Ruth Waste Stockpiles
Kimbley Pit
Ruth West Pit
Ruth East Pit
Jupiter Waste Stockpiles
Ruth Area Final Pits and Waste Stockpiles
Robinson Nevada Mining Company
1,000'
quadra

Table 23.1:
Life-of-Mine Production Schedule

Total Expit Mining	2009	2010	2011	2012	2013	2014	2015	2016	Total
Ore tons mined (000)	16,645	17,530	17,832	16,425	16,425	22,228	16,425	10,047	133,557
Cu Grade (%)	0.61	0.54	0.54	0.57	0.52	0.47	0.65	0.40	0.54
Au Grade (opt)	0.008	0.009	0.006	0.006	0.005	0.005	0.008	0.005	0.006
Waste tons mined (000)	51,567	50,967	47,728	68,000	65,939	49,197	67,646	12,156	413,200
Strip Ratio	3.10	2.91	2.68	4.14	4.01	2.21	4.12	1.21	3.09
Total Tons Mined (000)	68,212	68,497	65,560	84,425	82,364	71,425	84,071	22,203	546,757

Waste stockpiles will be located adjacent to the pits being mined and are segregated by material type (PAG- Potentially Acid Generating and NAG-Non-Acid Generating) so that PAG materials are isolated and encapsulated for future reclamation. Some waste will be backfilled into mined out pits. Some of the waste from Veteran is currently being put in the Tripp pit. All waste rock disposal is regulated under the Waste Rock Management Plan.

23.1.2.1 Slope Stability

Call and Nicholas Inc., performed an extensive geotechnical study based on drill core obtained in 1991 from holes drilled along the periphery of the Ruth Pit and inside the Tripp-Veteran pit. Geotechnical test work included uniaxial compression strength tests, Brazilian disk tension tests, small scale direct shear tests, and intact rock shear strength tests in order to model and predict slope stability in each of the proposed pits.

Golder Associates performed a geotechnical study based on drill core obtained in 2007 and 2008 for holes drilled along the periphery and inside the Veteran Pit, Ruth West Pit, and Ruth East Pit. The data collected includes core recovery, Joint Condition Rating (JCR), Rock Quality Designation (RQD), strength index, discontinuity data, Joint Roughness Coefficient (JRC), and discontinuity orientations. Geotechnical test work included triaxial test and Atterberg limit tests in order to model and predict slope stability in each of the proposed pits.

23.1.2.2 Ore and Waste Management

The resource model has variable density based on material type, which can include: rock type, alteration, mineralization, and geologic interpretation. Due to the nature of the RNMC mix of porphyry-skarn deposit type, the mining is highly selective. All blast holes are assayed and those blocks of mineralized material are also tested in the metallurgical lab to assess the metallurgical performance of material to verify that it is of ore tenor.

Ore control is based on assay and metallurgical testing and the various blocks of material are specifically flagged (marked) in the field. Each loading unit operator is provided with a map of the ore and waste type boundaries and assigned specific tasks on a daily basis. Blending, when required is accomplished using two loading units.

All blast holes are sampled and the material is analyzed in two labs depending on the material type. All samples are sent to the assay lab (managed by the processing division) where metals content is determined as well as providing determinations for other characteristics. Those samples with sufficient copper content are identified and representative samples are processed in the metallurgical lab to determine suitability as ore feed to the concentrator.

The ore-body model contains information on ore grades and metallurgical performance. This information includes geological as well as metallurgical information and is used to model pit slope angles, anticipated mill feed characteristics and concentrator performance factors.

23.1.2.3 Resource Model Reconciliation

Two comparisons are discussed here, the first is a review of the resource model performance by Quadra corporate personnel and the second is the regular monthly reconciliation done by RNMC personnel. The site reconciliation focuses on tons and grades above an economic cutoff whereas the corporate review focuses on how well the resource model predicts all grades. At this point in time, only total copper grade reconciliation will be covered as gold and soluble copper reconciliations are not complete.

<u>Global Grade Reconciliation</u>

All blast holes in the Veteran Pit drilled between mine startup in 2004 and November 8, 2008 were incorporated into a copy of the resource block model. Average blast-hole grades were calculated for each block in the model that contained at least one blast hole.

Straight averaging of the blast-hole grades falling inside a block was used. The block size is 50 ft x 50 ft x 50 ft. The minimum number of blast holes in a block in the mined area is 1 and the maximum is 16; the average number of blast holes per block is 4.3.

Table 23.2 provides summary statistics of Veteran Pit blast-hole grades and resource model grades mined between 2004 and November 2008. Table 23.3 contains similar information but is limited only to grades above a total copper cutoff of 0.25%. The 0.25% copper cutoff was chosen to approximate an economic cutoff without the added complications of variable recoveries, different material types, and gold values. Figure 23.3 is a histogram of the blast holes and associated statistical data.

Table 23.2:
Veteran Total Copper Grade Reconciliation Statistics

	Blast Holes	Model	Composites
Mean	0.376	0.344	0.497
Standard Error	0.003	0.003	0.019
Median	0.317	0.277	0.331
Mode	0.004	0.010	0.010
Standard Deviation	0.326	0.345	0.714
Sample Variance	0.106	0.119	0.510
Kurtosis	7.998	25.913	35.910
Skewness	1.916	3.381	5.022
Range	4.068	5.623	8.339
Minimum	0.000	0.001	0.001
Maximum	4.068	5.624	8.340
Sum	4287.66	3922.54	705.89
Count	11394	11394	1420
CV*	0.865	1.001	1.437

*CV = Coefficient of Variation (Std dev/mean)

**Table 23.3:
Veteran Total Copper Grade
Reconciliation Statistics > 0.25% TCu**

	Blast Holes	Model	Composites
Mean	0.553	0.558	0.753
Standard Error	0.004	0.005	0.028
Median	0.469	0.468	0.504
Mode	0.273	0.402	0.455
Standard Deviation	0.303	0.350	0.819
Sample Variance	0.092	0.123	0.670
Kurtosis	12.498	34.898	27.802
Skewness	2.593	4.400	4.611
Range	3.818	5.374	8.090
Minimum	0.250	0.250	0.250
Maximum	4.068	5.624	8.340
Sum	3815.32	3369.22	648.66
Count	6900	6041	862
CV	0.547	0.628	1.088

*CV = Coefficient of Variation (Std dev/mean)

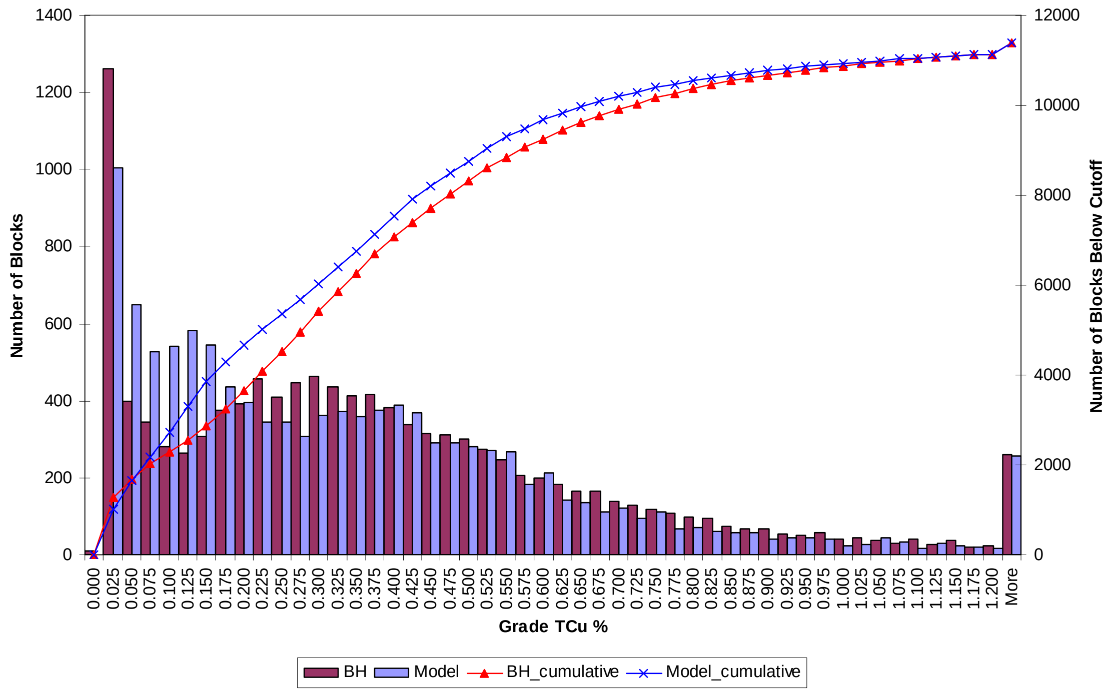


Figure 23.3: Histogram of Veteran Pit Model and Blast-Hole Total Copper Grades

Overall, the resource model under-predicts blast-hole grades by approximately 9%. However, above a 0.25% total copper cutoff, the average model grade is within 1% of the average blast-hole grade. The resource model predicts about 12% lower volume than the blast holes above the 0.25% cutoff. This would effectively result in the resource model under predicting the volume of ore-grade material in the deposit, assuming that the total copper grade was the only criteria used for ore control. This amount is somewhat more conservative than ideal and efforts are being made to improve the model performance.

Note that the composite grade average is significantly higher than either blast holes or model grades (this is likely a result of clustered composite data).

Monthly Ore Reconciliation

The Robinson Technical Services Group reconciles reported mill production, the ore-control model (*e.g.* blast holes), and the resource model on a monthly basis. This work is focused on how well the grade models predict the amount and grade of material processed by the mill.

This reconciliation includes factors not directly related to the resource model, including economic cutoff grades, recoveries, and the use of blast holes to define ore-waste boundaries. Ore is distinguished by a cutoff in recoverable copper pounds per ton of material. The calculation includes total copper grade, soluble copper grade, and material type from which process recovery is determined by a fairly involved set of equations.

The overall conclusion reached by review of the data from January 2008 through October 2008 is that the resource model has predicted the overall contained units of metal extremely well; although this is a result of under-predicting grade and over-predicting tons.

Results of 2008 monthly comparisons through October between the mill, blast holes, and resource model are summarized in Table 23.4.

Table 23.4:
Veteran Total Copper Grade Reconciliation January-October 2008

Source	Tons	Total Cu %	Soluble Cu %	Au oz/ton	Contained Cu lbs
Mill	12,860,031	0.674	0.026	0.013	173,353,218
Blast holes	13,452,584	0.655	0.018	0.014	176,228,850
Resource Model	14,834,119	0.580	0.051	0.011	172,075,780
Differences					
BH - Mill	592,553	(0.019)	(0.008)	0.001	2,875,633
Model - BH	1,381,535	(0.075)	0.033	(0.003)	(4,153,070)
Model - Mill	1,974,088	(0.094)	0.025	(0.002)	(1,277,437)
Differences %					
BH - Mill	5%	-3%	-31%	8%	1.7%
Model - BH	10%	-11%	183%	-21%	-2.4%
Model - Mill	13%	-16%	49%	-18%	-0.7%

One complication in interpreting this data is the presence of material that is above cutoff, but cannot be sent to the mill due to poor metallurgical performance. At this time, most of this material cannot be identified in the resource model prior to performing flotation tests on blast-hole cuttings. There is approximately one million tons of material that fit this description in the resource model in the area mined during 2008. This material is included in Table 23.4 in the Resource Model but was excluded from the "Mill" and "Blast holes" entries because the material was never sent to the mill.

Further reconciliation work is recommended and is underway, the results of which should be used to adjust modeling parameters to better predict mill tons and grades. Part of these analyses should be verification of densities used in the model.

23.1.3 Mining Equipment

The mining equipment fleet consists of five large loading units: a BE-495 shovel, a P&H 2300 shovel, a Hitachi EX-5500 shovel, a Hitachi EX-3500 shovel and a LeTourneau L-1850 front end loader. There is a smaller front end loader, a CAT 992 that is used for utility and clean-up operations.

The haulage fleet consists of sixteen CAT 793D (240 ton class) trucks and six CAT 785 (150 ton class) trucks. Haulage requirements vary by year, driven mostly by the haulage profile. When requirements diminish, the 785 trucks are idled first.

The drilling fleet consists of one Atlas Copco Pit Viper, one BE-49RII, and two Atlas Copco DML drills. There is also a mobile Atlas Copco secondary blasting drill. A third DML drill was leased for the pioneering work for the Ruth pit.

Blasting operations are currently performed by SW Energy, a blasting contractor. This work is performed under the direction of RNMC personnel and these services include placement of blasting agents, placement of timing and booster devices and accessories, stemming of holes and blast initiation and clearing.

Support equipment consists of a fleet of mixed dozers (CAT D-10 and Komatsu 375A units), CAT motorgraders, Rubber Tired Dozers, water trucks, fuel and lube trucks and a trackhoe. The entire major mine equipment currently at the site is held under capital or operating leases. There are also light vehicles such as a tire handler, service trucks, forklifts, welding trucks, pick-up trucks and man-vans.

Equipment demand, combined with operations and maintenance schedules are used to predict costs and manpower requirements.

23.1.4 Mine Operating Costs

The mine division is under the supervision of the Mine Manager. This division includes mine operations, mine maintenance and technical services. These groups perform all the functions required to mine, with the exception of loading and blasting services, which are performed with a contractor.

Technical services and mine operations personnel direct all activities of the blasting contractor and design the blasts and determine the priorities for those blasts. The contractor provides the explosives and powder trucks, priming, loading, stemming and firing services. The blast design is optimized to assure adequate fragmentation as well as to minimize damage to pit highwalls.

The mine is operating with four crews which work rotating shifts of 12 hours each. This allows the mine to operate 24 hours per day, continuously. The crews each have personnel trained for all the functions required on an operating shift: loading, haulage, drilling and all support functions. A training department has been utilized to perform initial training and is now focused on improving skill sets of the mine operators.

Maintenance is also provided on a 24 hour basis. Maintenance activities include all preventative maintenance activities that include oil filtering and changes, air filter changes, tire changes, component repairs and replacement, welding and machining activities. This program has evolved to the point where reliability centered maintenance is used, based on oil sampling, vibration and temperature analysis to predict component life.

Technical services group activities include: blast design, power distribution and pipeline location design, geology, ore control, surveying, slope monitoring and geotechnical support, mine planning, and statistical support. This group maintains the ore-body model and performs reconciliations of mined tonnages, material delivery locations and reconciliations of the model predicted to actual performance.

Mine costs are variable due to strip ratio, haulage profile, wet/dry conditions (dewatering costs), diesel fuel and lubricant costs, wear steel and parts costs, tire costs and blasting agent and accessory costs. The mine budgets are based on anticipated costs for repairs and operating costs based on quotations or estimates for the items listed above. Mine operating cost predictions are based on over four years of operational experience.

23.1.5 Mine Capital Costs

Mine capital costs are based on the purchase of new and used equipment from vendors for all future fleet equipment requirements. In addition, there are general construction and material removal activities that are capitalized. This type of work can include, but is not limited to: drilling of wells, removal of tailing and other material from the Ruth pit, construction of roads, pipelines, power lines, fencing, fuel stations and other fixtures specifically associated with mine activities.

At the discretion of the company, capital equipment may also be leased and the costs of those leases reflected in operating costs rather than as capital costs.

Mine equipment has an established life based on operating hours, and duty life. Some equipment will last the life of the mine, while other equipment will be replaced during the life of the mine.

The estimated mine capital costs are approximately US$13.5 million dollars over the life of mine.

23.1.6 Mine Facilities

Mine facilities include change rooms and offices needed to support mining activities. There are two change room and office complexes currently at site: one located near the Ruth pit and one located nearer the shops, warehouse and mineral processing facilities. There is currently a single large mine equipment repair shop, adjacent to the warehouse.

Additional mine related facilities include: a wash bay, fuel docks, dewatering facilities and pipelines, water spouts for filling water trucks, specific storage areas for drill steel, ground engaging tools (bits and teeth), explosives/blasting agent storage, spare parts and parking facilities. Each pit will also be equipped with a slope monitoring station(s) dependent upon needs.

23.1.7 Dewatering System

There are three distinct types of mine dewatering systems currently in use at the Robinson mine.

When the water flow is minimal and results from small perched water zones within the rock, such as currently encountered in the Tripp and Veteran pits, in-pit dewatering holes are drilled.

When the pits have small pit lakes and/or receive flows in excess of the evaporation rate (*e.g.*, Ruth, Liberty, and Kimbley pits), pumping systems and pipelines have been or will be constructed to pipe that mine impacted water to the concentrator for use as processing water. The Ruth and Liberty pit are currently equipped with barge or sump pumps and piping systems.

The third type of dewatering system is to dewater aquifers in advance of mining activities. This type of dewatering was anticipated in the Magma EIS and is currently under construction to allow for local depression of the water table and to depressurize the pit walls in the vicinity of the Ruth pit complex. This water is pumped from large capacity (+1000 gpm) pumps and is currently being used for process water and has potential to be used for mine potable water in the near future.

23.1.8 Processing Operations

The processing division is under the supervision of the Processing Manager. This division includes mill operations, mill maintenance, technical services, and surface maintenance. Mill operations are staffed with four crews working rotating 12 hour shifts. The crews each have personnel trained for all of the functions required for an operating shift. The mill technical services include the analytical and metallurgical laboratories.

23.1.8.1 Processing Facilities

Ore is hauled from the pits and truck-dumped into the primary crusher. The crushing circuit at Robinson consists of one 60' x 89' gyratory crusher with a nominal capacity of 2,500 TPH, crushing to minus 6” material.

The Robinson mill grinding circuit consists of one 32' x 14.75' (9.75 m x 4.50 m), 10,000 horsepower (HP) variable speed semi-autogenous grinding (SAG) mill followed by two 20' x 30.5' (6.1 m x 9.3 m) 8750 horsepower closed circuit ball mills.

The rougher flotation circuit consists of two parallel rows of ten 3,000 ft^3 flotation cells followed by cleaning in six 10' x 38' (3.05 m x 11.58 m) column flotation cells in a parallel configuration. No regrinding is currently being performed. The concentrates are thickened and filtered prior to shipment.

The tailings impoundment is located south of the Veteran Pit in Giroux Wash. The dam is a "center-lift" design, constructed out of tailings material. Tailings are thickened prior to deposition in the tailings impoundment. The barge operating channel (BOC) is used to reclaim water from the tailings dam for use in the mill.

23.1.8.2 Process Operating Costs

Mill operating costs per pound are variable due to ore hardness, ore grade, and recovery. There are several key drivers to the mill operating costs, including manpower, electricity, steel price, and reagent costs. The mill operating budgets are based on anticipated costs of repairs and materials/supplies from quotations and estimates based on over four years of operating experience.

23.1.8.3 Process Capital Costs

Total life-of-mine processing capital is estimated at approximately US$36 million. The majority of the capital expenditure is related to periodic increases in the height of the dam to maintain the necessary freeboard. In addition, the capital forecast includes expenditures for future process improvements as well as replacement of support equipment such as loaders, small haul trucks, and light vehicles.

23.1.9 Administration

Administrative functions at RNMC include accounting, purchasing/warehousing, safety, environmental, information technology, and human resources.

23.1.9.1 Administration Capital Costs

Total life-of-mine General and Administrative capital is estimated at approximately US$51 million. The majority of the capital expenditure is related to expenditures for Ruth Development including dewatering/depressurization of the pits in advance of mining, and removal of the Alta Gold tailings from the Ruth West pit.

23.2 Recoverability

Table 23.5 contains a summary of the life-of-mine metal plan. A discussion of the metallurgical performance of Robinson's ore and expectations for future performance is found in Chapter 16.

Table 23.5:
Life-of-Mine Metal Production Summary

	2009	2010	2011	2012	2013	2014	2015	2016	2017	Total
Ore Tons Milled (000)	16,064	16,334	16,425	16,425	16,425	16,425	16,425	15,958	3,661	134,142
Cu Grade (%tcu)	0.62	0.56	0.57	0.57	0.52	0.54	0.65	0.35	0.38	0.54
Au Grade (opt)	0.009	0.010	0.006	0.006	0.005	0.006	0.008	0.006	0.007	0.007
Cu Production (000 lbs)	138,987	144,730	144,733	143,683	134,025	143,190	166,317	89,927	16,664	1,122,256
Gold Production (000 oz)	99	109	66	33	33	33	49	48	14	484

23.3 Markets

The copper market has experienced a severe downturn in demand over the past 6 months, largely due to the US-led housing crisis followed by a world-wide financial credit crisis. These factors have further culminated into a severe de-leveraging process that has affected all industries, but particularly resource based enterprises. The copper market has recently experienced the typical increase in copper warehouse inventories followed by copper metal price reductions. Production cutbacks to counteract a decreasing market demand are being done and planned within the base metal industry. However, the long term fundamental market outlook for copper remains strong and an average copper price of US$2.00 per pound has been assumed for planning purposes.

23.4 Contracts

Robinson currently has in place contracts for concentrate transportation and handling, smelting, refining, electric power, fuel, explosives, consumables, and other miscellaneous items necessary to operate the mine. These contracts are all within industry norms.

23.5 Environmental Considerations

The current environmental liability at the Robinson Mine is approximately US$85 million. This estimate is based on closure cost estimates compiled by RNMC internal personnel and 3rd party consultants and represents costs anticipated to ensure permanent closure of the facility upon cessation of mining operations. Robinson is in the process of updating the cost estimates to include some additional disturbance areas necessary to execute the current mine plan. These disturbance areas include additional waste rock dumps and dewatering disposal facilities. Closure cost estimation work should be completed by the end of Q1 2009. At this time, RNMC does not foresee any additional obligations that would materially increase the closure cost. Section 4.1 contains a description of the environmental considerations and permits.

23.6 Taxes

A summary of applicable taxes and royalties is provided in this section. Specifically, the Nevada Net Proceeds, Corporate Taxes and Royalties are described below.

Nevada Net Proceeds

RNMC is subject to the Nevada Net Proceeds of Minerals Tax, an ad valorem property tax assessed on minerals mined or produced in Nevada when they are sold or removed from the state. If the net proceeds of the mine in the taxable year total $4 million or more, the tax rate is 5%. If the net proceeds of the mine in the taxable year is less than $4 million, a graduated rate based on the percentage of net proceeds of gross proceeds is applied.

RNMC's royalty holders are also subject to the Nevada Net Proceeds of Minerals Tax. They are taxed at 5% and have no allowable deductions.

Corporate Taxes

RNMC is subject to U.S. Federal tax (the higher, in any given year, of the regular tax and the alternative minimum tax). The State of Nevada does not have an income tax.

The federal regular tax rate is assumed to remain constant at 35% over the life-of-mine.

Royalties

The Robinson operation is subject to a three percent net smelter royalty (NSR) currently payable to Royal Gold Inc. (Royal Gold). This royalty was formerly payable to Kennecott Minerals (Kennecott). This NSR was to be used in the first instance to fund a reclamation trust and indemnify Kennecott for environmental liabilities, including reclamation costs. The trust was funded with the three percent NSR up to $20 million, including interest and with credits for certain reclamation expenditures. Once the trust was fully funded pursuant to the stipulations, the NSR royalty was sold by Kennecott to Royal Gold.

In addition to the Royal Gold Royalty, Franco-Nevada Corporation, (Franco-Nevada), is entitled to receive royalties from the production of the Robinson Mine. The royalties owing to Franco-Nevada consist of:

- A 10% royalty on net smelter returns on 51% of the production of gold from the Robinson Mine in excess of 60,000 troy ounces per calendar year;

- A royalty on 51% of copper production in excess of 130 million pounds of copper, payable in any calendar year in which the price of copper exceeds US$1.00 (adjusted for inflation from 1990) at the end the year (the "Trigger Price"), in an amount equal to US$0.05 per pound plus an incremental amount equal to 40% of the amount by which the average price of copper during the year exceeds the Trigger Price; and
- A 0.225% royalty on net smelter returns of all minerals from the Robinson Mine.

The mine is currently not subject to any Federal Royalty from production from Federal lands, which is from time-to-time considered by Congress. None of the production is currently coming from, or is planned to come from, unpatented Federal lands.

23.7 Economic Analysis and Payback

An economic analysis of the Robinson Operation was developed by Quadra using the production schedule presented in Section 23.1.2, current operating costs, estimated future operating costs, estimated capital costs, and estimated reclamation and closure costs. A long-term copper price of US$2.00/lb and gold price of US$800/oz were used in the analysis. The measurement of Robinson's economic viability that Quadra chose to use is net present value (NPV).

The amount of capital required for the remainder of the mine life is estimated to be approximately US$100 million. Capital is included in 2009 to increase mill flotation capacity and the additional recovered copper is included in the cash flow (See Chapter 16). A schedule of estimated capital expenditures over the life-of-mine is presented below in Table 23.6.

Table 23.6:
Schedule of Capital Expenditures

Year	(US$ Millions)
2009	$30
2010	$26
2011	$10
2012	$18
2013	$4
2014	$4
2015	$4
2016	$4
2017	$0
Total	**$100**

Total closure and reclamation costs as of January 1, 2009 have been estimated at US$85 million, based on existing mine plans. Operating costs for the mining operation and processing are described in Section 23.1.4 and 23.1.6, respectively. Offsite costs include inland freight costs, ocean freight, and concentrate treatment and refining charges. Average forecasted life-of-mine unit operating costs are summarized in Table 23.7 below.

Table 23.7:
Forecast Life-of-Mine Unit Operating Costs

Area	Life-of-Mine Cost
Mine (per ton mined)	$1.42
Processing (per ton milled)	$3.75
G&A (per ton milled)	$1.24
Offsite costs (per lb. of Cu produced)	$0.49

An estimate of annual pre-tax cash flow for the Robinson Operation is presented in Table 23.8.

Table 23.8:
Life-of-Mine Pre Tax Cash Flow

	2009	2010	2011	2012	2013	2014	2015	2016	2017
Pre-Tax Cash Flow US($000)	$48,850	$70,379	$58,429	$15,620	$12,036	$32,532	$70,360	$37,322	$3,928

Closure costs and bonding payments continue after mining has ceased and, while not included in the cash flow table (Table 23.8), are included in the NPV calculations.

The economic analysis indicates that based on the stated assumptions, the Robinson Operation should generate a positive NPV (at both 0% and 8% discounting rates) for Quadra if the long term copper price is at or above $2.00/lb. Sensitivities to copper price were estimated with the result that NPV (at 8% discount rate) reaches zero if the copper price is reduced by 15%. If copper price is increased by 15% NPV increases 101%. In reality, the price of copper and gold as well as other costs will vary in both the short and long term, and the other assumption values may change, all changing the economics.

23.8 Mine Life

Mine plans have been developed that provide for an economic mine life to 2017 for mining. The resource model predicts that there will be enough material stockpiled during the mine life to allow the mill to operate into the year 2017 by processing those stockpiles, assuming appropriate metal prices. This is discussed in greater detail in Section 23.1.

To-date, exploration activities have been focused on the margins and logical extensions of known ore bodies adjacent to the above pits. Exploration efforts have been primarily focused on drilling adjacent to and within those pits to look for expansions of those pits either laterally or vertically. Some exploration has also looked at nearby features within the existing permitted mining area but not immediately associated with a specific pit.

In addition, the site has performed some remote sensing to delineate potential hidden or previously unknown potential exploration sites associated with the Robinson District. An aggressive re-assaying of older drill-hole pulps and core has also been completed and is being input into the ore body model.

It is possible that certain higher copper prices could result in expansions of many of the existing pits and known deposits.

Appendix A:
Veteran Estimation Parameters and Variograms – Total Copper

VETERAN TOTAL COPPER
Estimation Parameters

Kriging Zone 1 (Hypogene)	Pass 1	Pass 2
Minimum # of Composites	2	2
Maximum # of Composites	10	5
Maximum # of Composites per hole	3	3
Search distances (ft)	300/250/250	200/140/140
Search directions	120°/10°/-80°	120°/10°/-80°
Restriction on high grade		
Grade (Cu%)	none	0.30
Distance (ft)	none	130

Nugget	Model	Sill	Range 1	Range 2	Range 3	Rotn	Dipn	Dipe
0.28	First spherical model	0.49	150	105	95	120	10	-80
	Second spherical model	0.21	900	900	750	120	0	0

Kriging Zone 2 (CEB)	Pass 1	Pass 2
Minimum # of Composites	2	2
Maximum # of Composites	10	7
Maximum # of Composites per hole	3	3
Search distances (ft)	200/200/300	100/100/100
Search directions	30°/5°/8°	30°/5°/8°
Restriction on high grade		
Grade (Cu%)	none	1.50
Distance (ft)	none	60
First pass maximum grade 0.30%Cu		

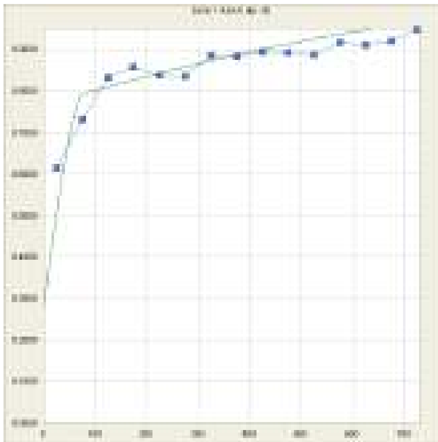
Nugget	Model	Sill	Range 1	Range 2	Range 3	Rotn	Dipn	Dipe
0.51	First spherical model	0.18	50	20	750	30	5	8
	Second spherical model	0.27	280	130	565	0	23	-42

Kriging Zone 3 (Leach Cap)	Pass 1
Minimum # of Composites	2
Maximum # of Composites	7
Maximum # of Composites per hole	3
Search distances (ft)	400/270/300
Search directions	330°/-5°/-50°
Restriction on high grade	
Grade (Cu%)	none
Distance (ft)	none

Nugget	Model	Sill	Range 1	Range 2	Range 3	Rotn	Dipn	Dipe
0.33	First spherical model	0.26	25	270	85	300	0	-20
	Second spherical model	0.14	395	130	300	355	-10	-85
	Third spherical model	0.23	750	750	400	80	42	-38

Kriging Zone 10 (other)	Pass 1	Pass 2
Minimum # of Composites	2	2
Maximum # of Composites	12	7
Maximum # of Composites per hole	3	3
Search distances (ft)	750/350/750	50/80/300
Search directions	120°/-24°/9°	27°/7°/-24°
Restriction on high grade		
Grade (Cu%)	none	none
Distance (ft)	none	none
First pass maximum grade 0.30%Cu		

Nugget	Model	Sill	Range 1	Range 2	Range 3	Rotn	Dipn	Dipe
0.38	First spherical model	0.41	45	84	310	27	7	-24
	Second spherical model	0.21	750	520	750	120	-24	10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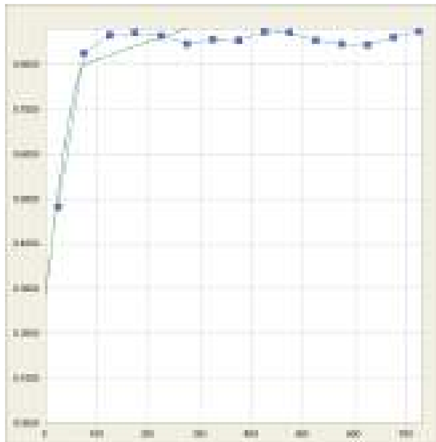
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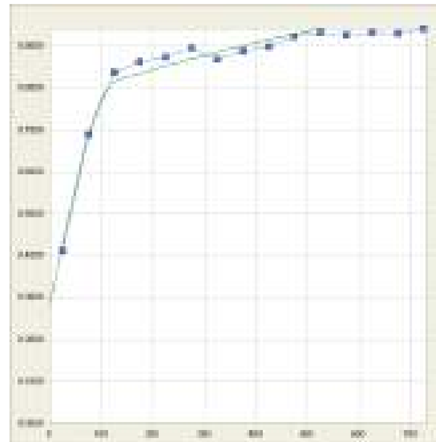
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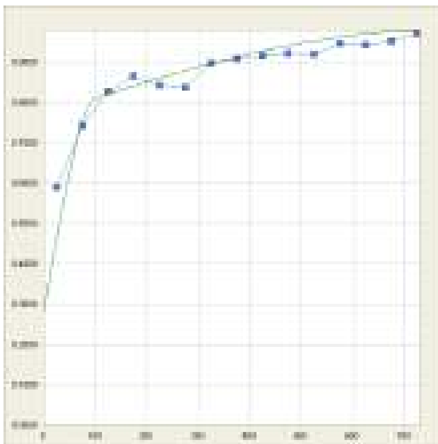
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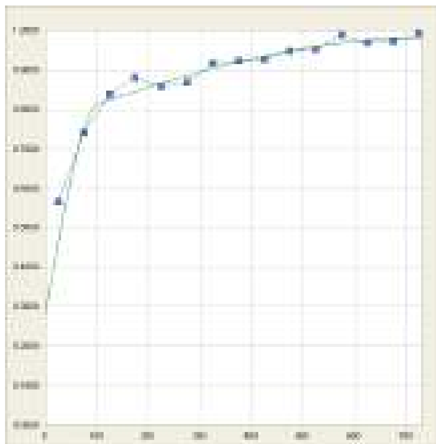
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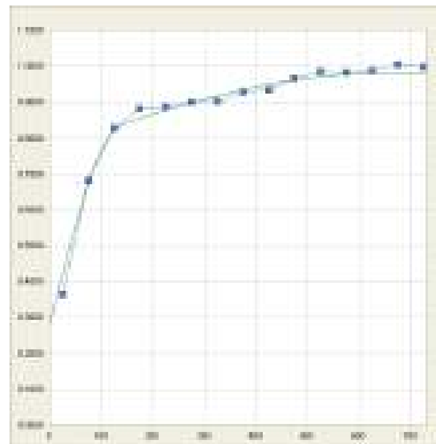
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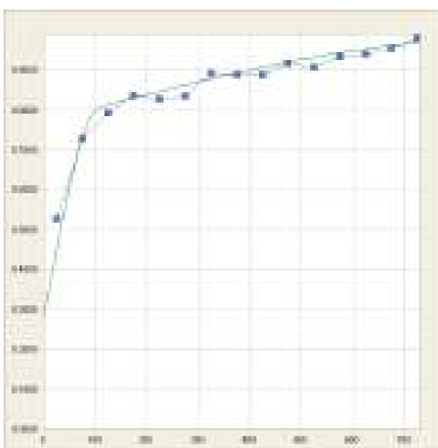
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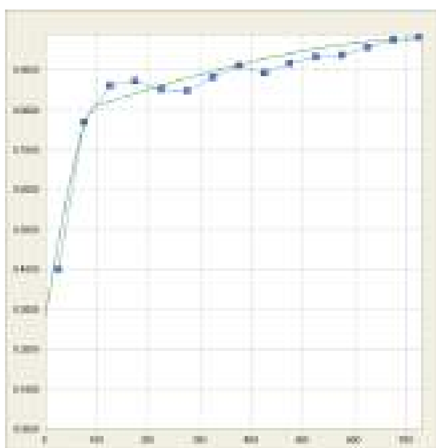
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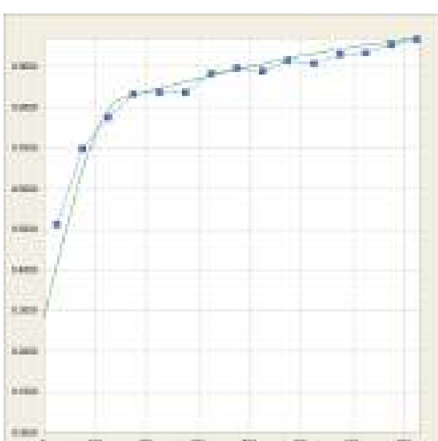
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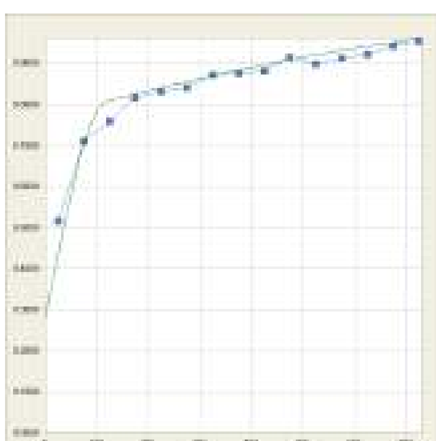
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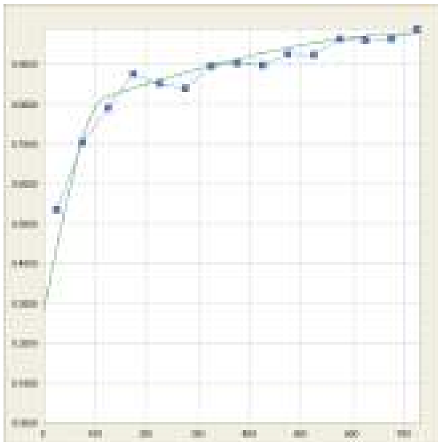
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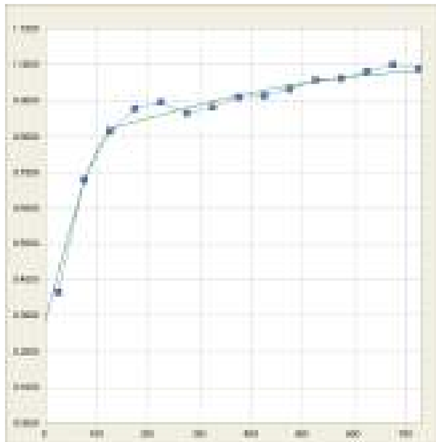
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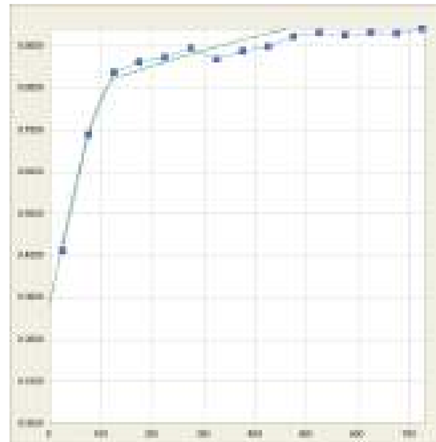
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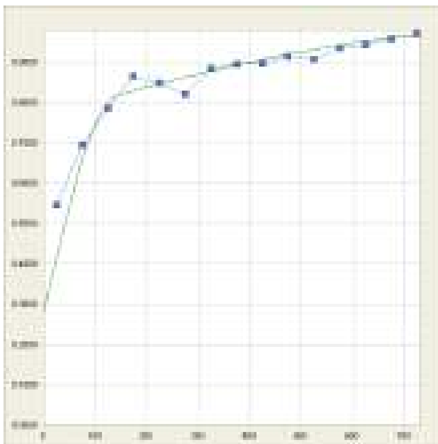
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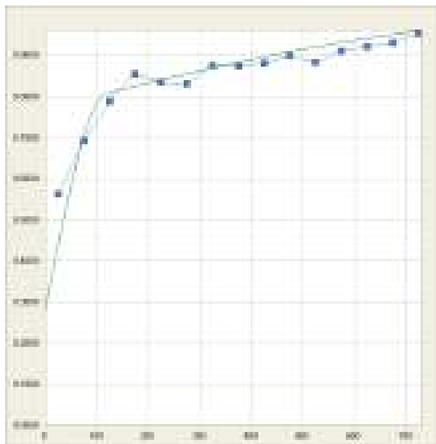
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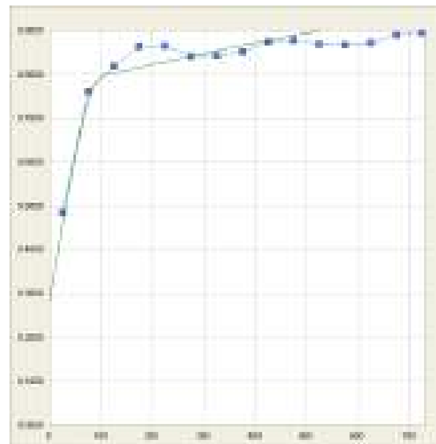
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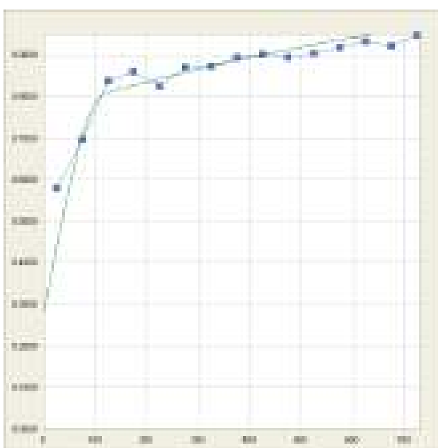
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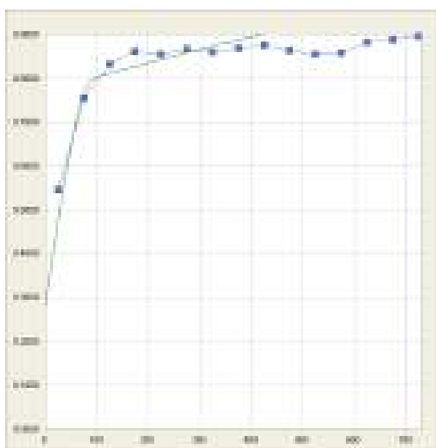
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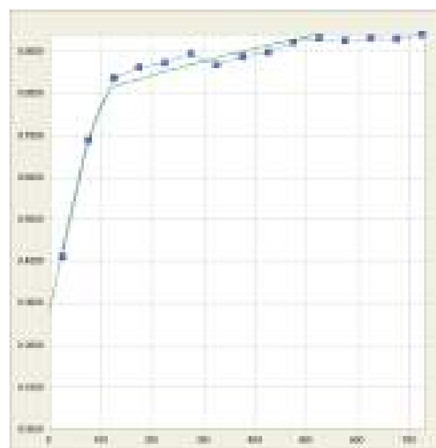
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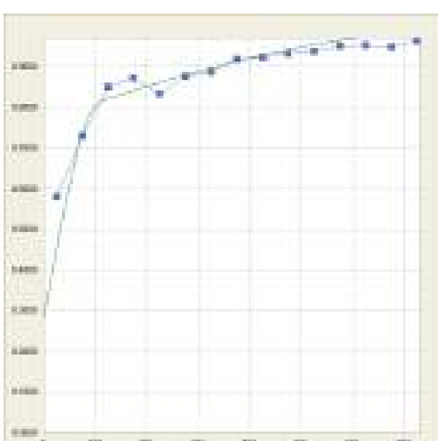
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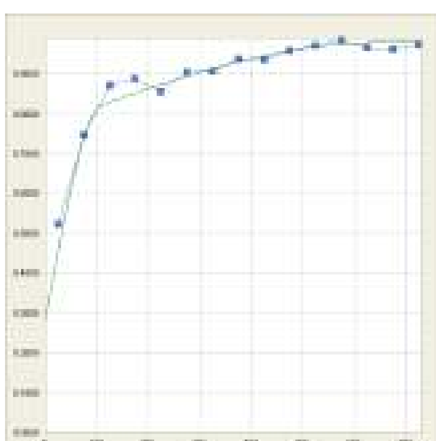
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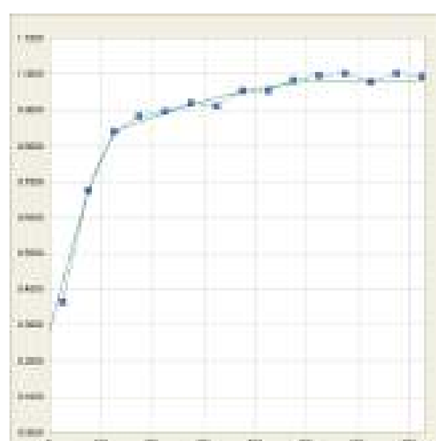
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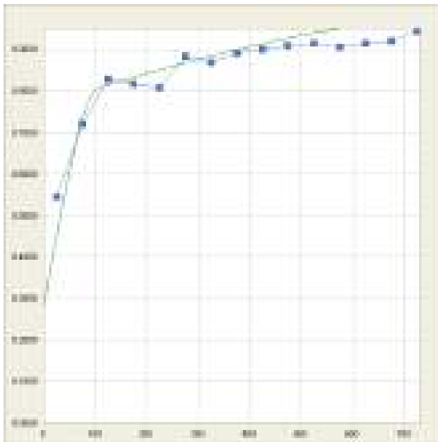

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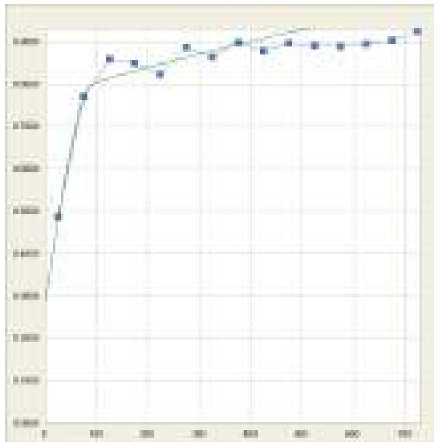

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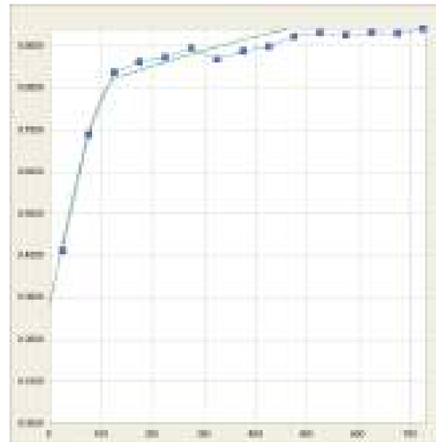

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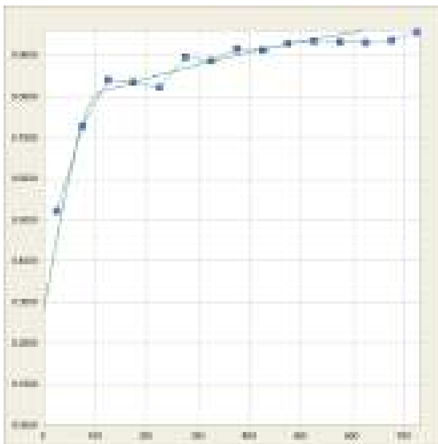

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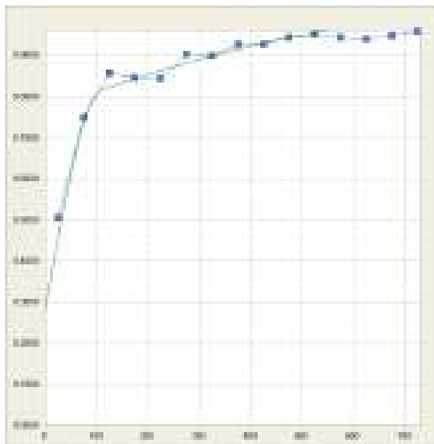

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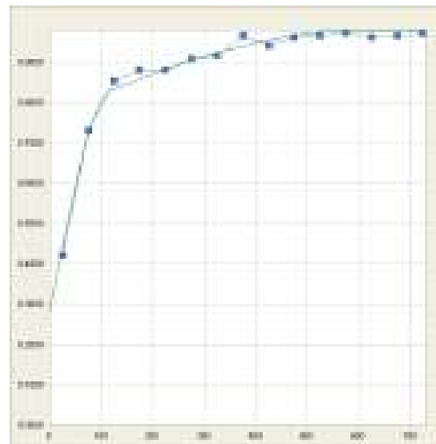

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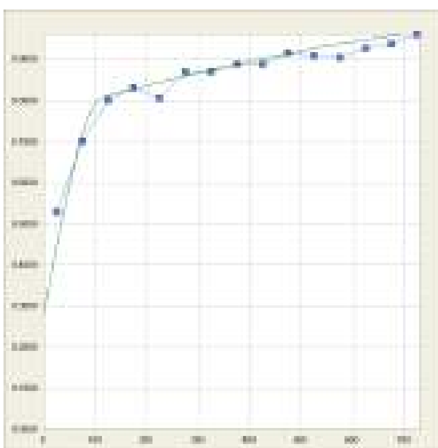

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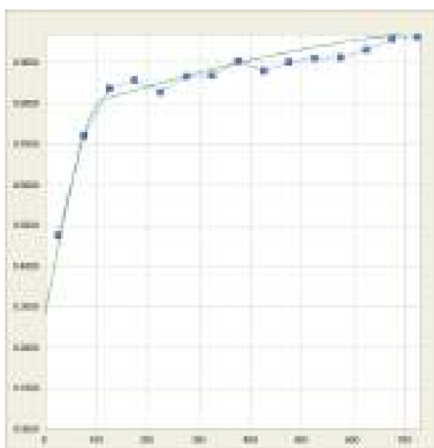

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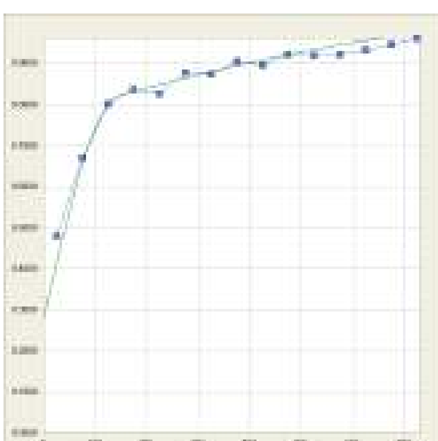

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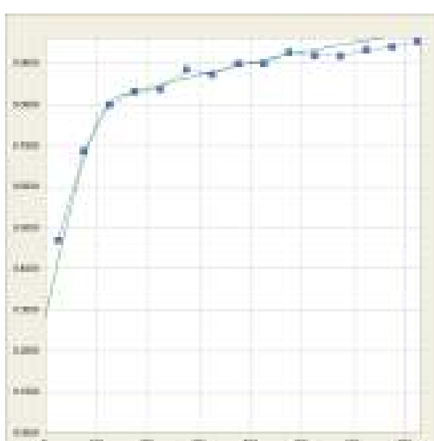

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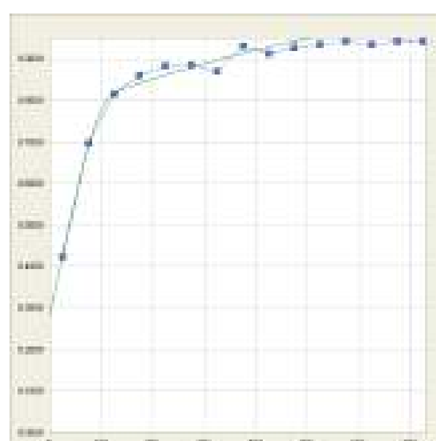

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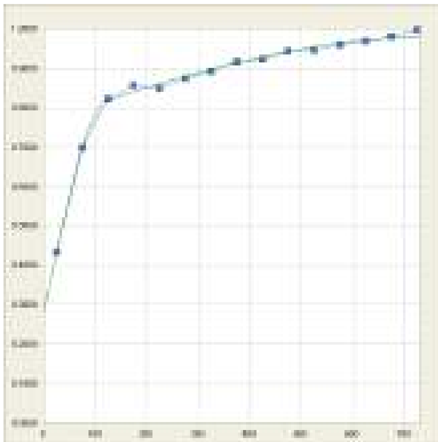
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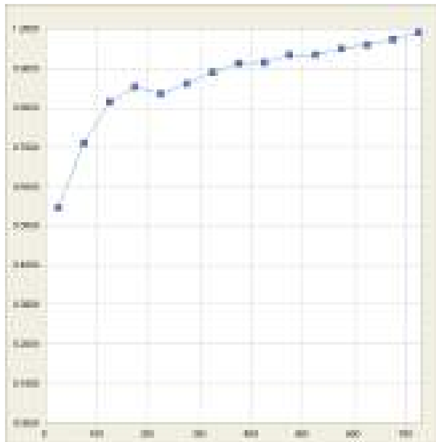
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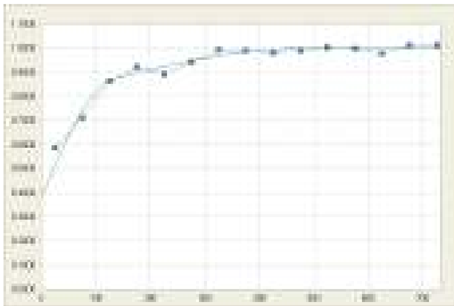

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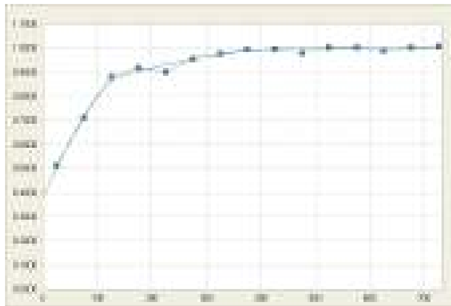

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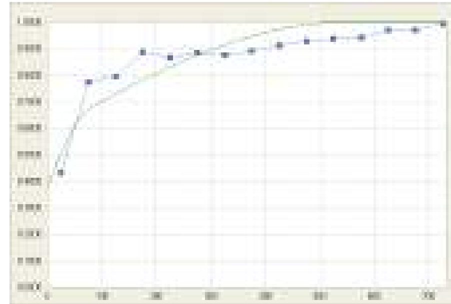

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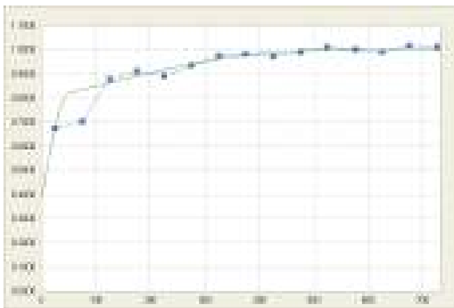

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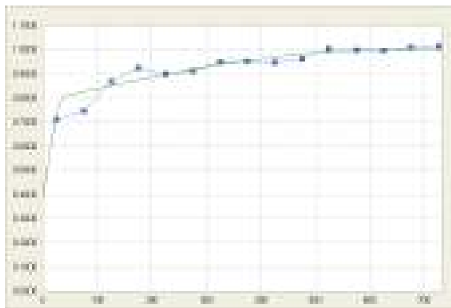

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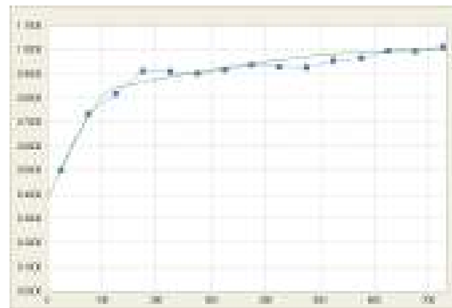

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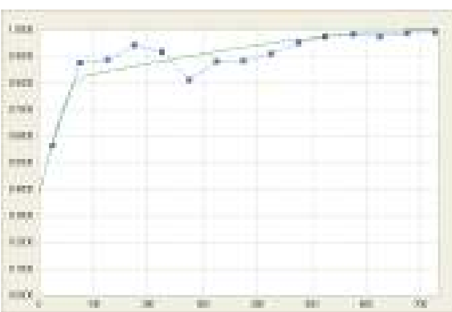

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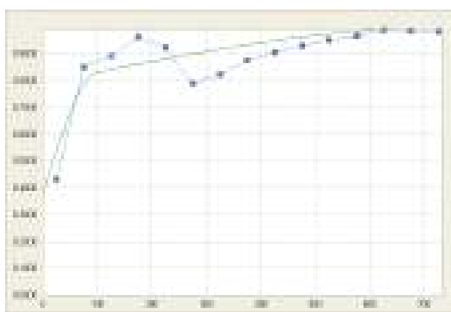

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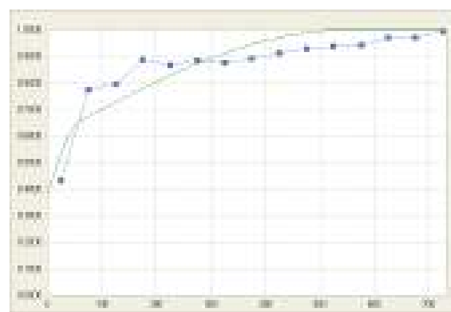

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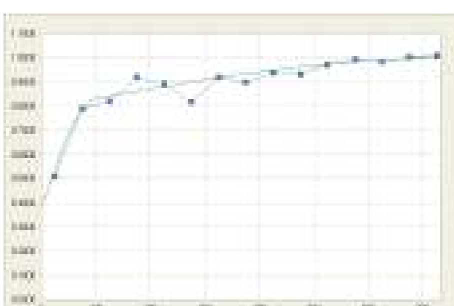

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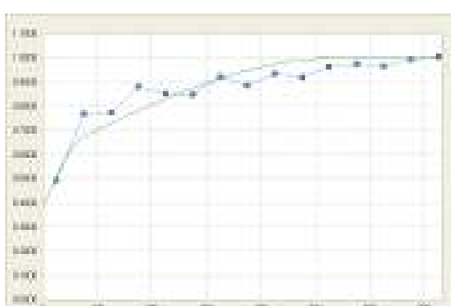

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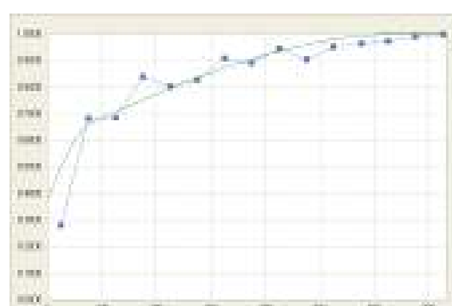

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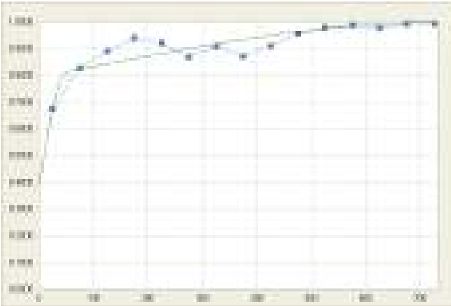

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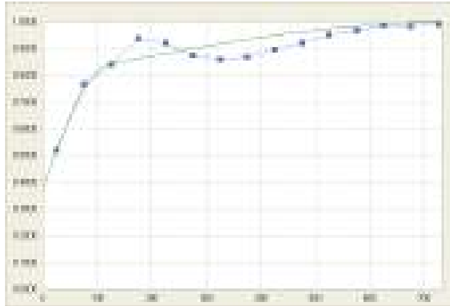

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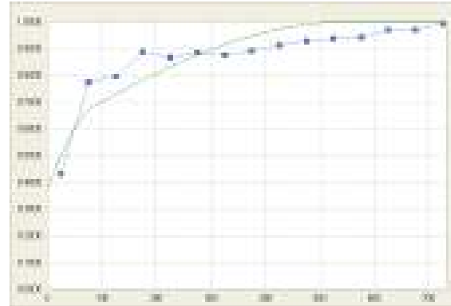

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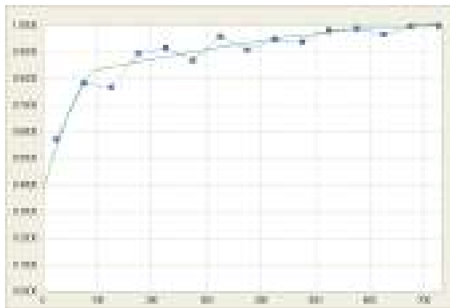

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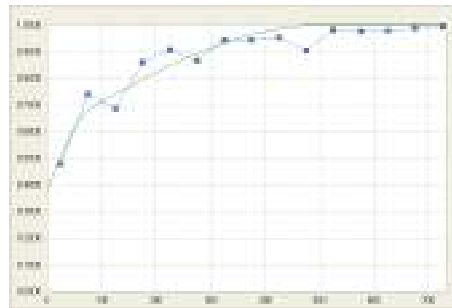

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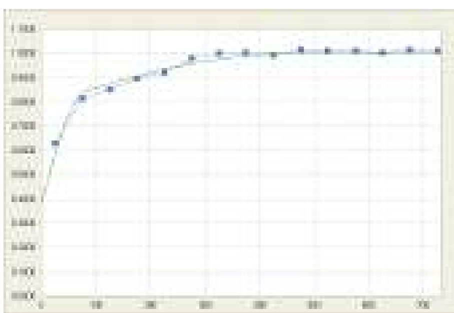

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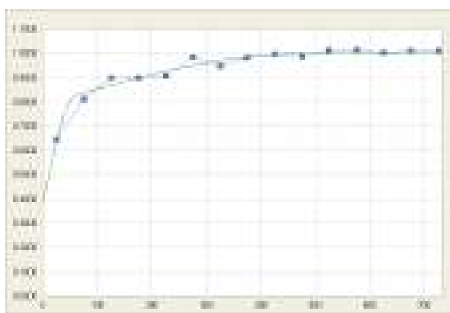

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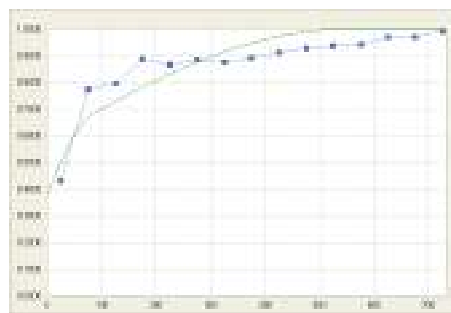

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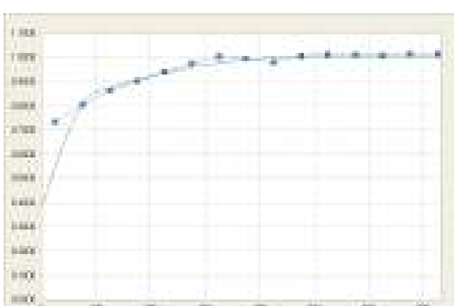

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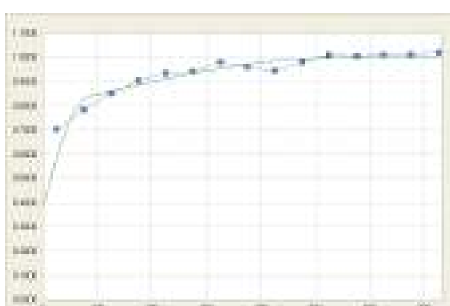

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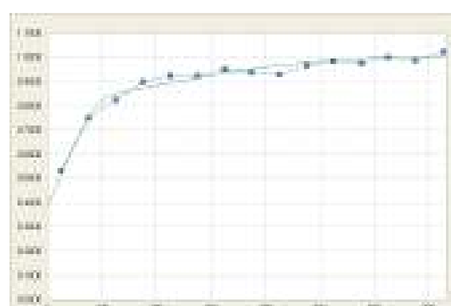

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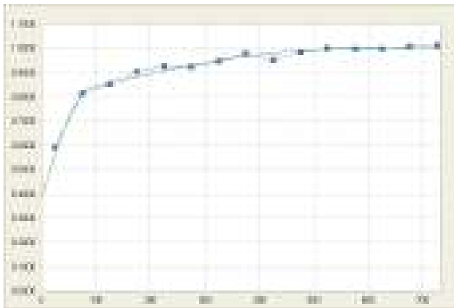

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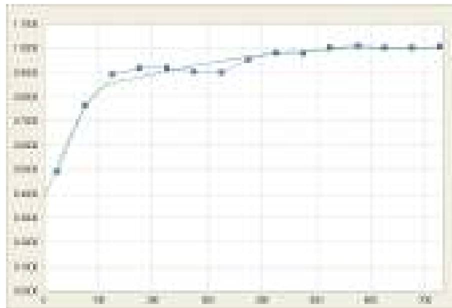

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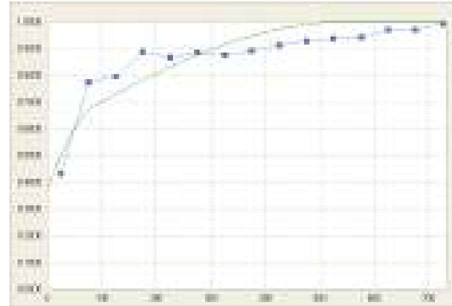

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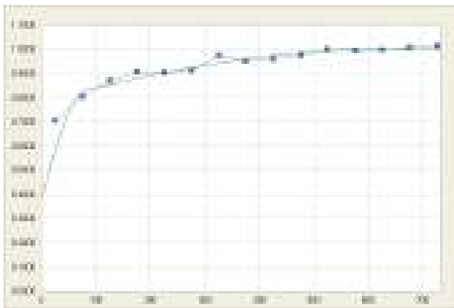

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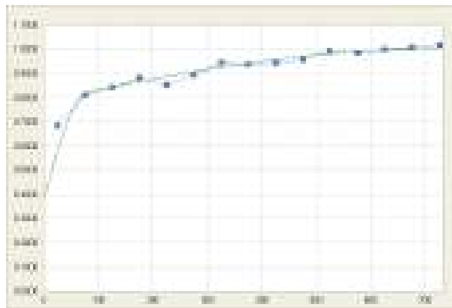

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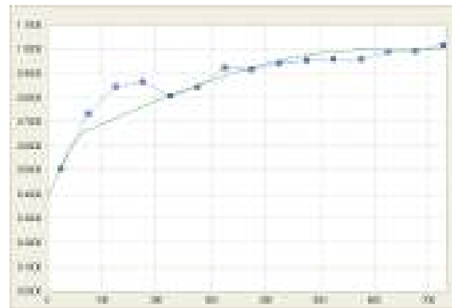

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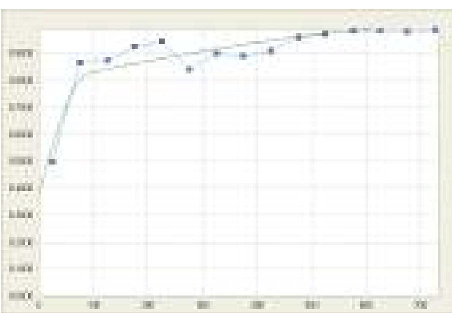

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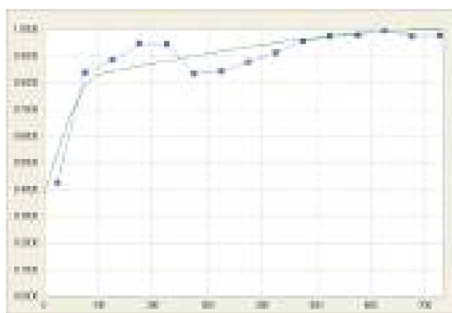

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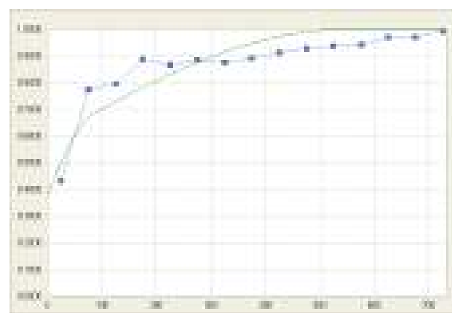

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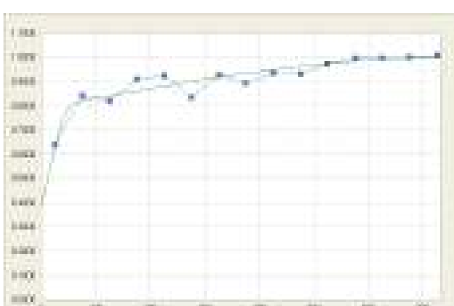

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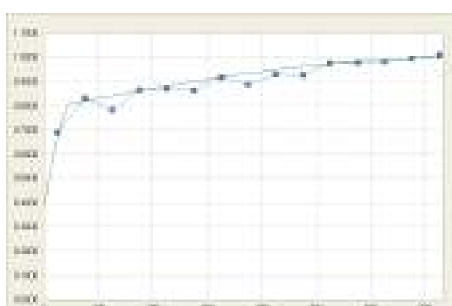

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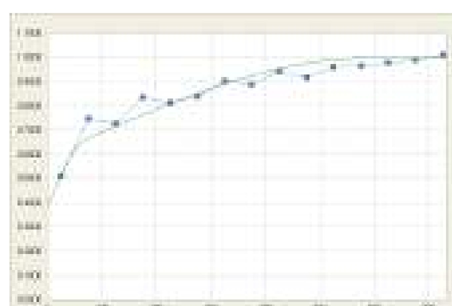

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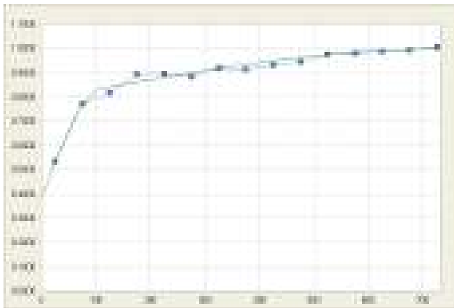

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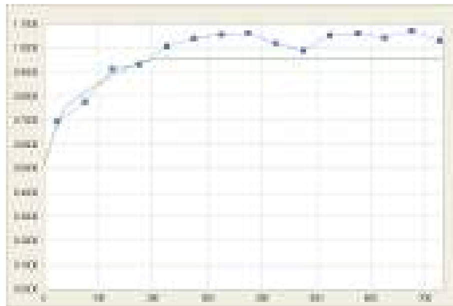

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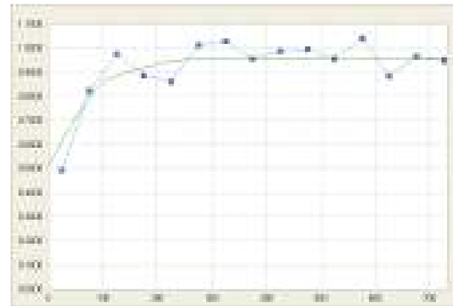

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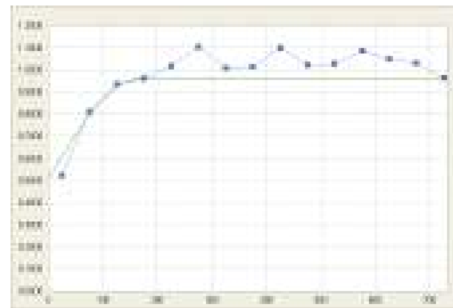

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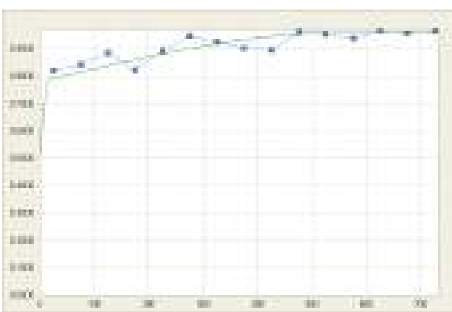

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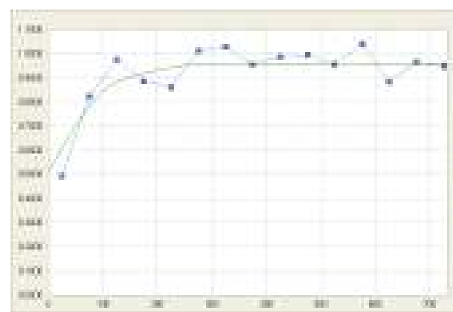

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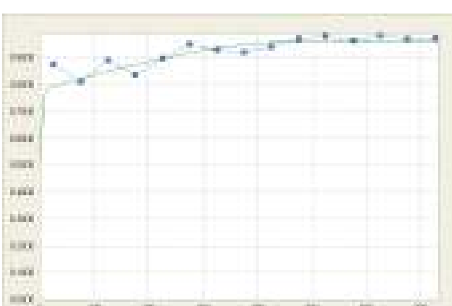

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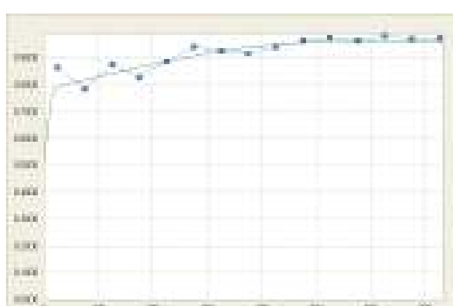

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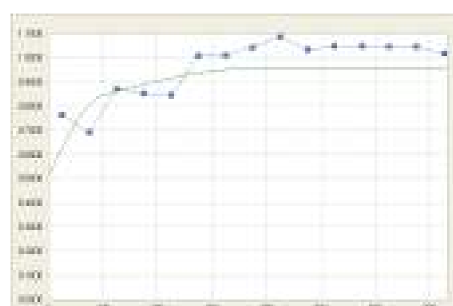

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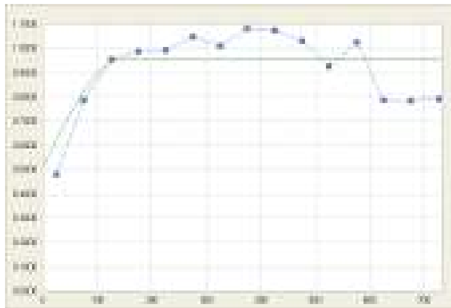

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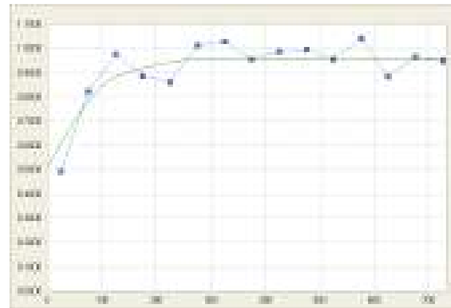

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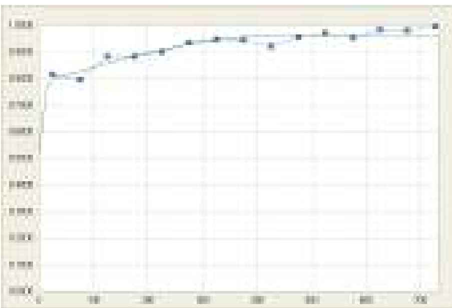

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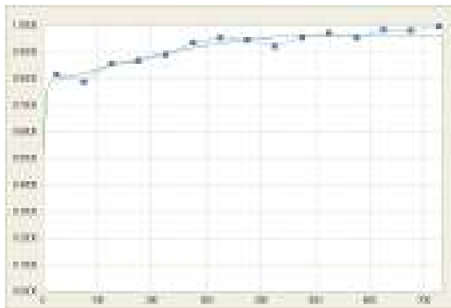

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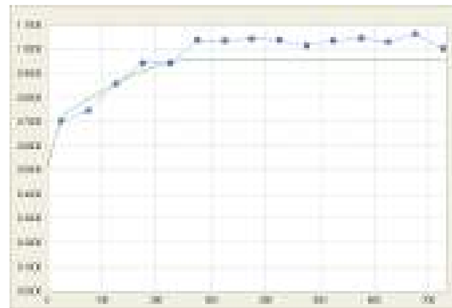

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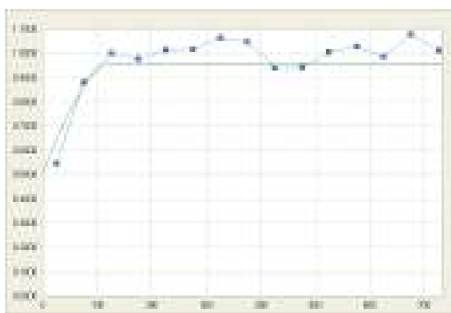

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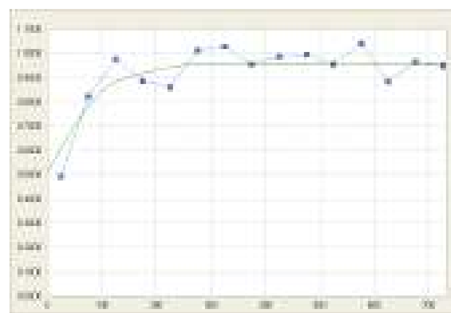

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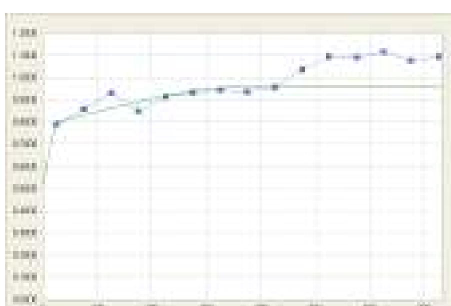

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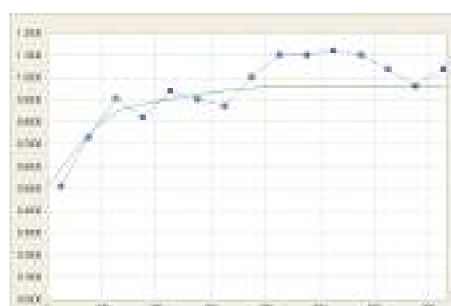

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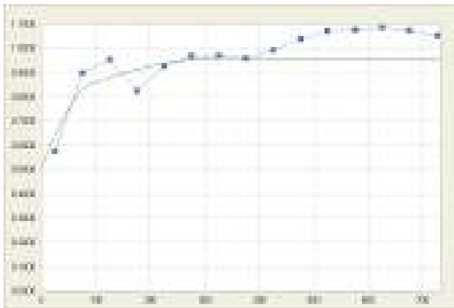

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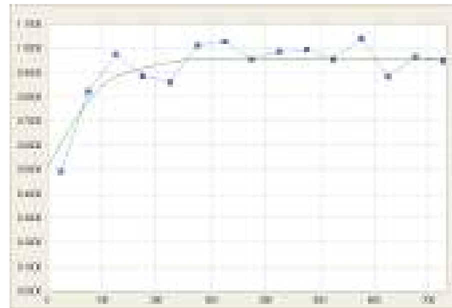

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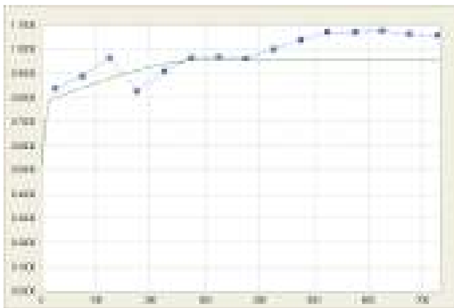

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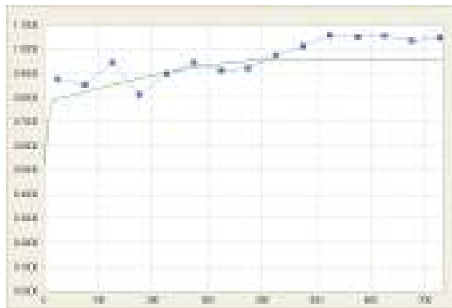

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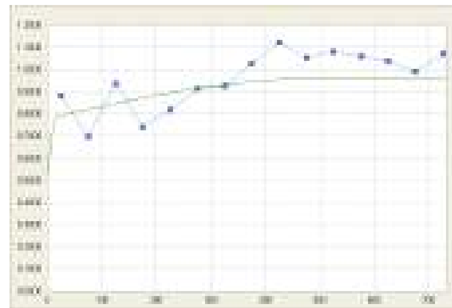

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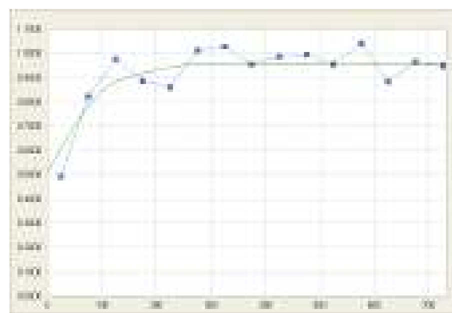

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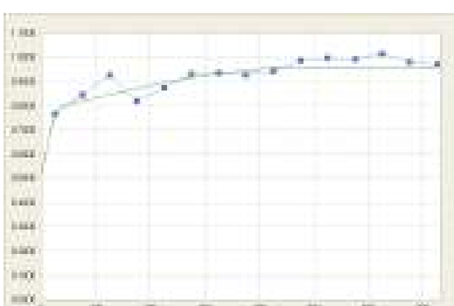

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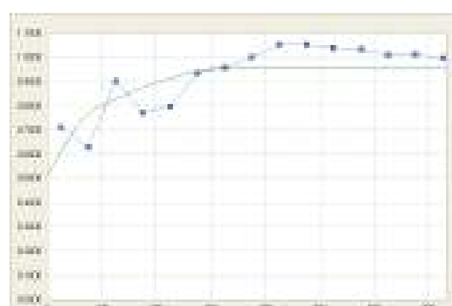

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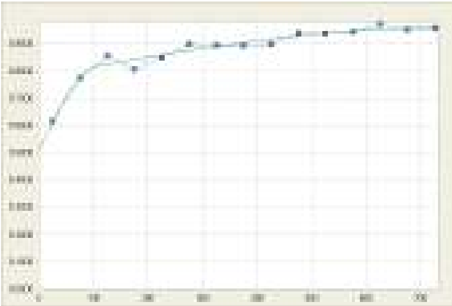

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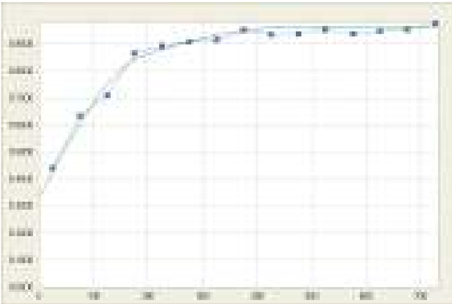

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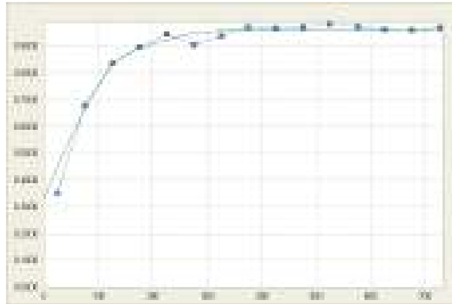

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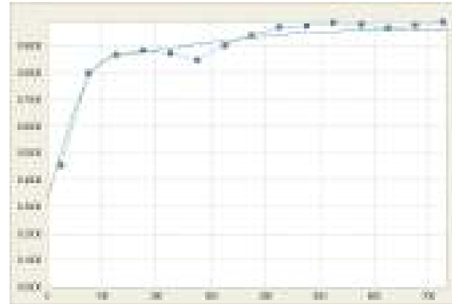

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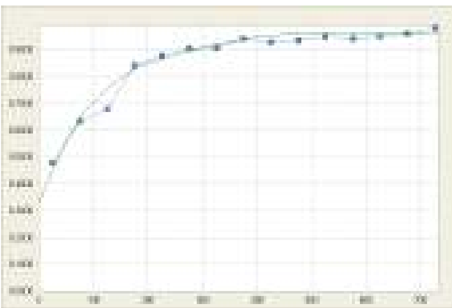

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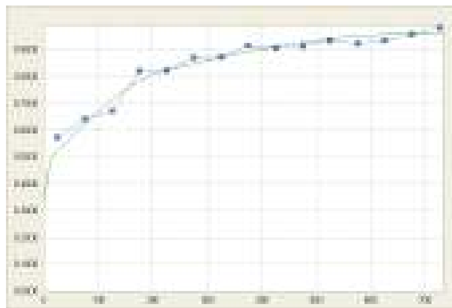

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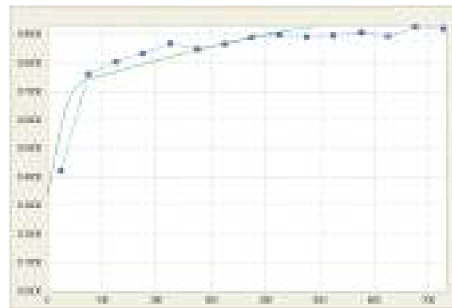

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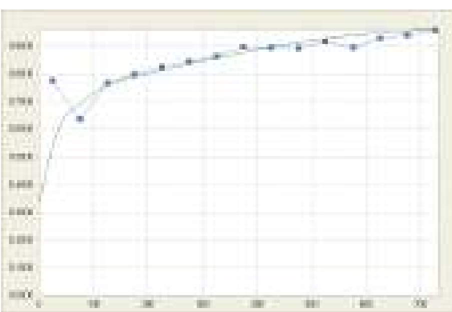

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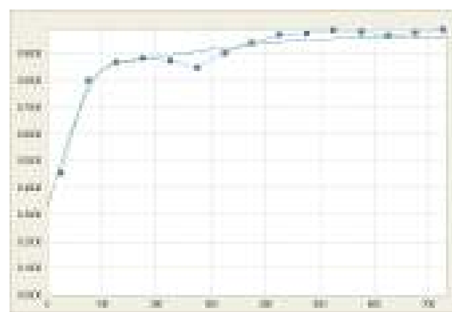

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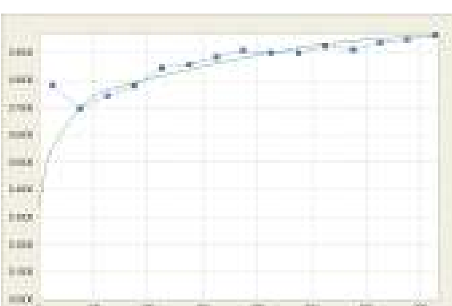

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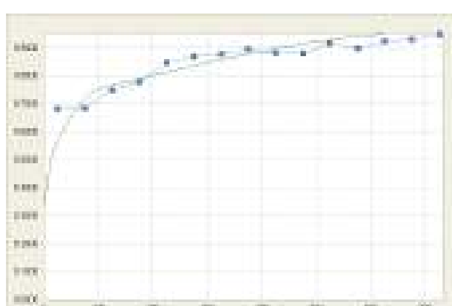

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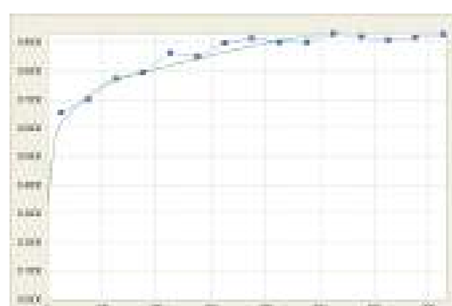

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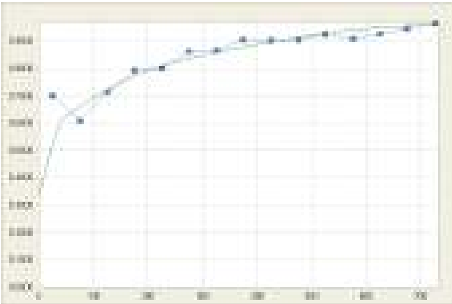

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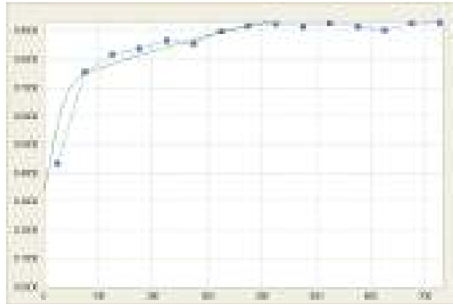

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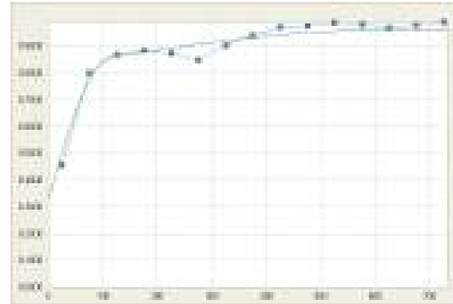

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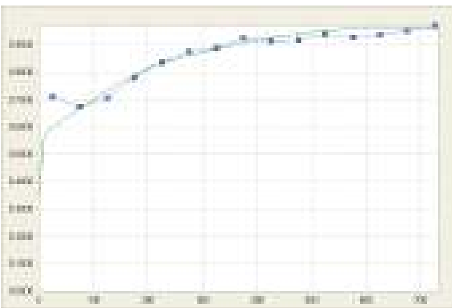

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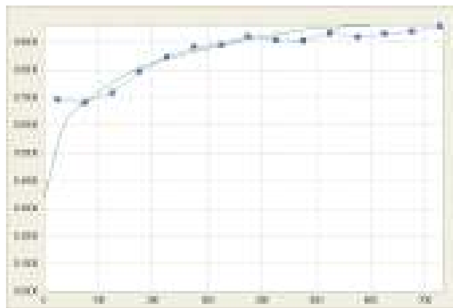

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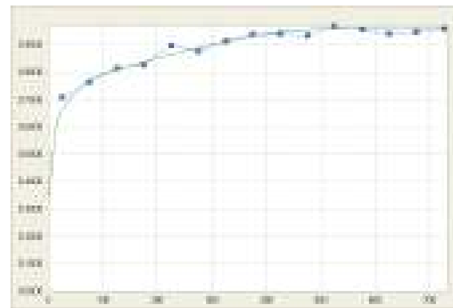

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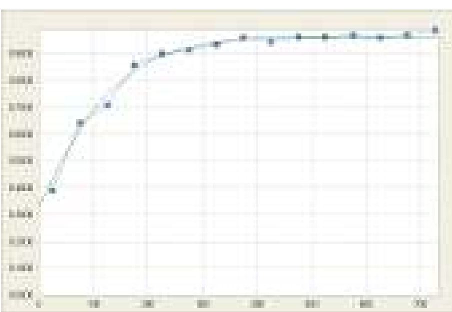

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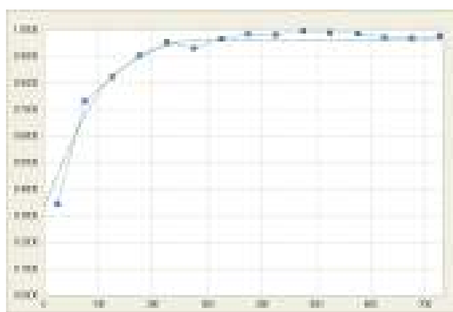

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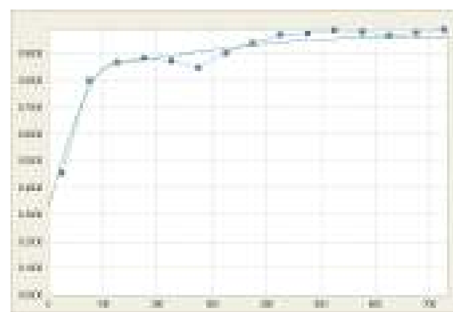

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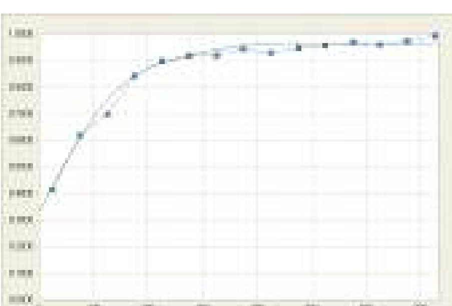

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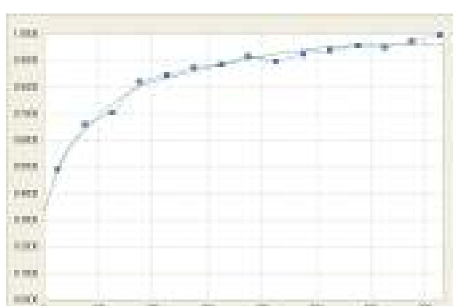

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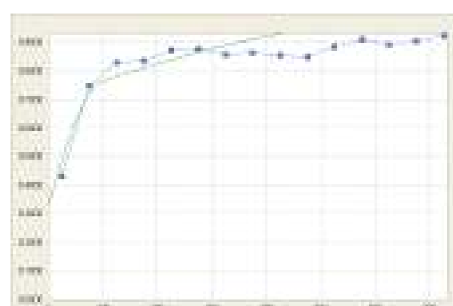

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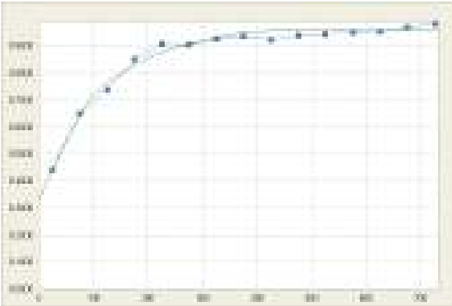

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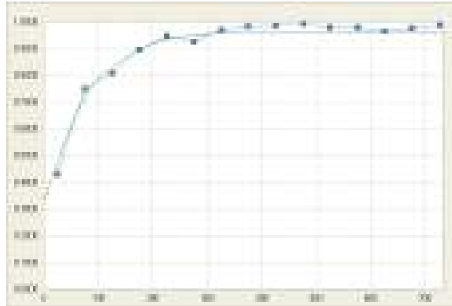

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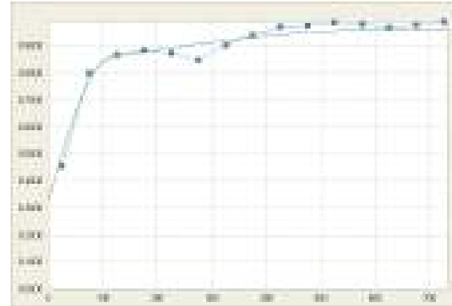

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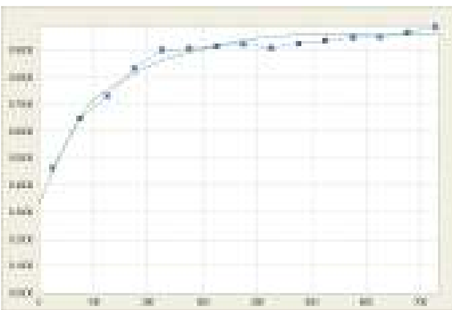

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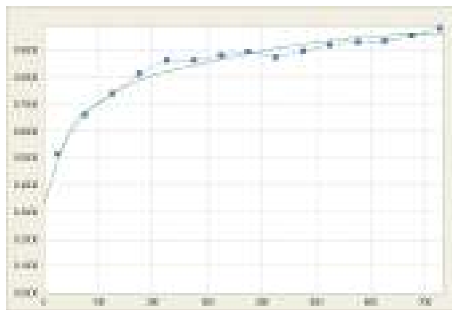

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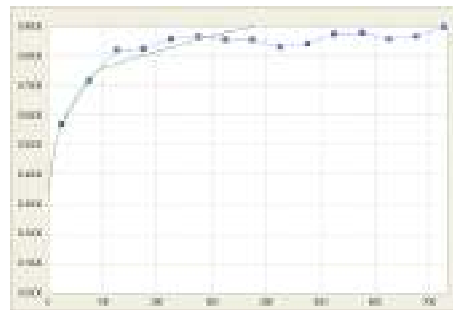

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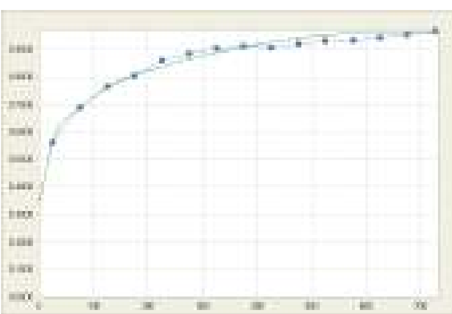

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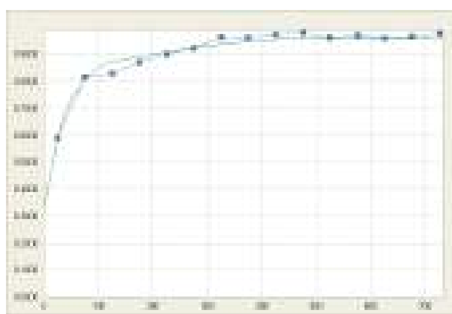

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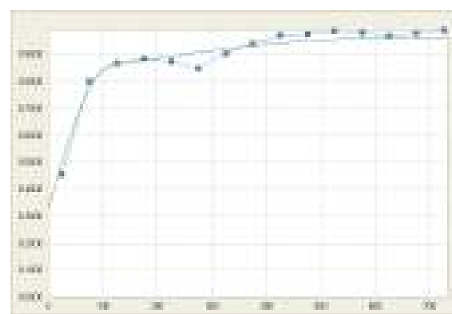

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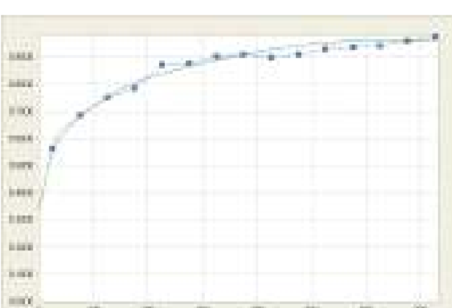

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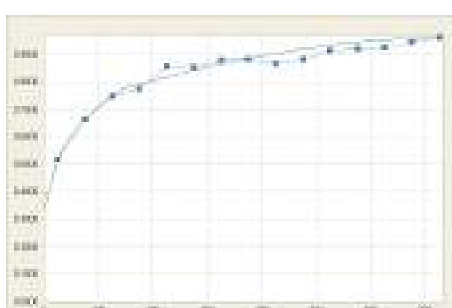

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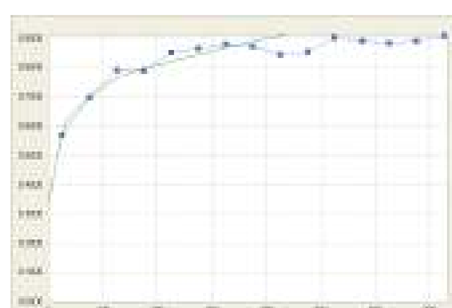

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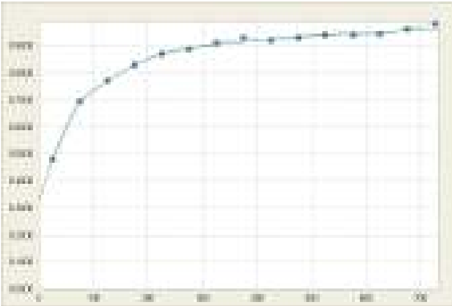

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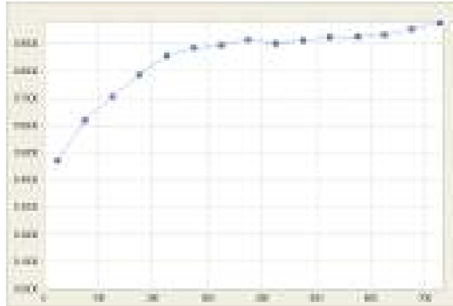

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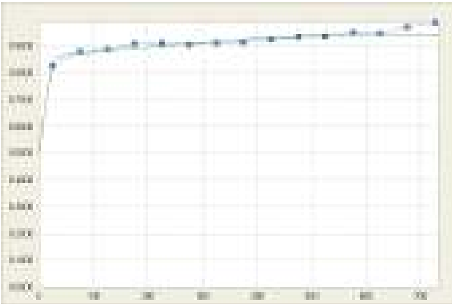
Appendix B: Veteran Estimation Parameters and Variograms – Gold



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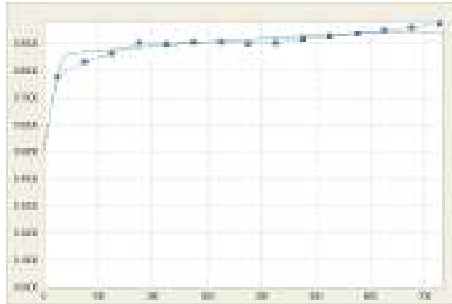

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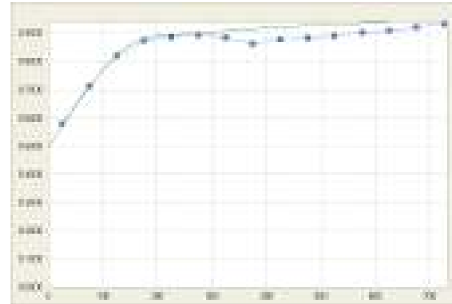

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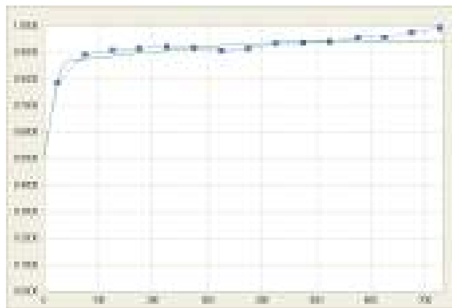

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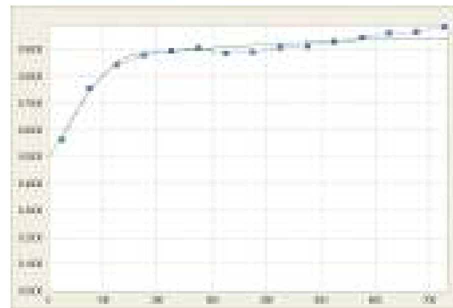

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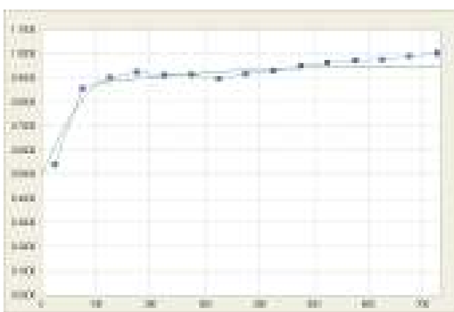

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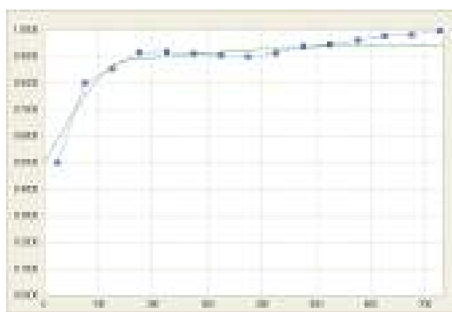

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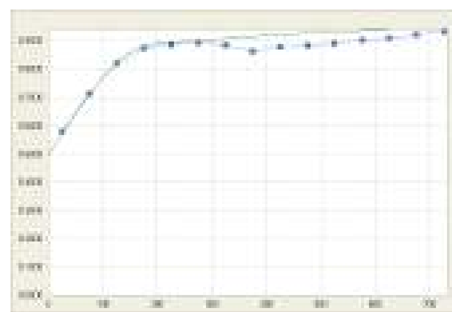

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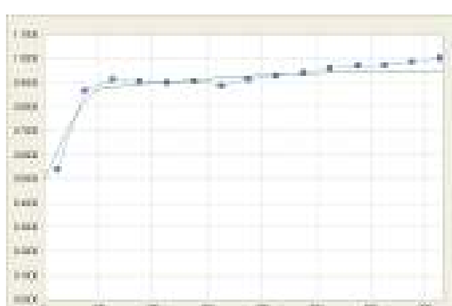

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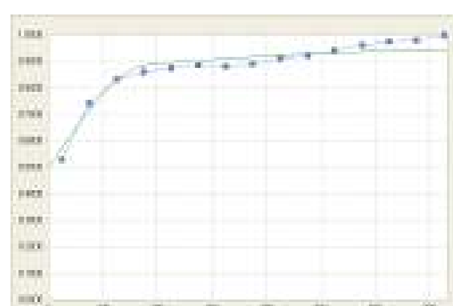

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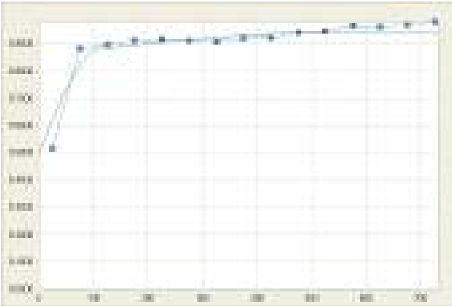
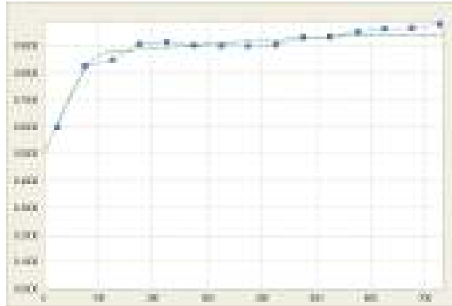
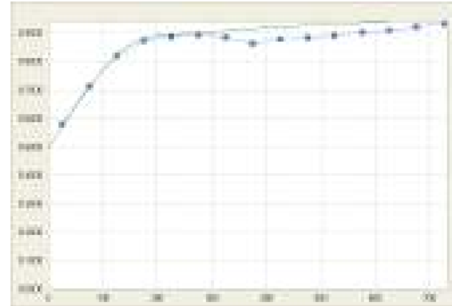

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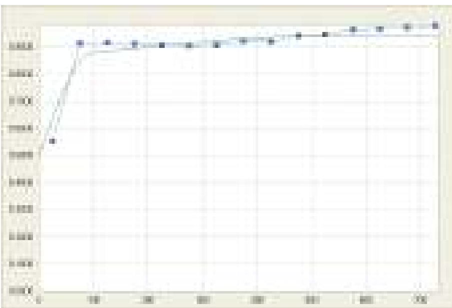
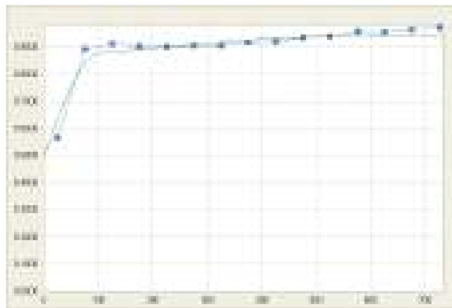
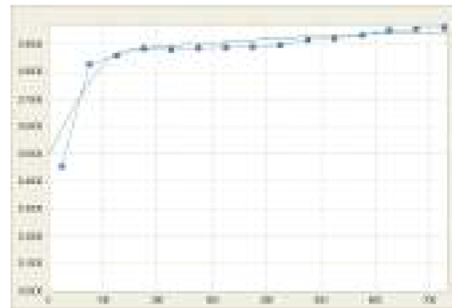

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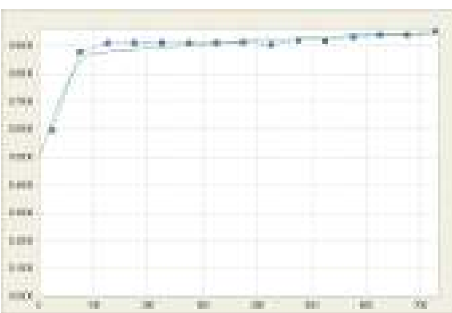
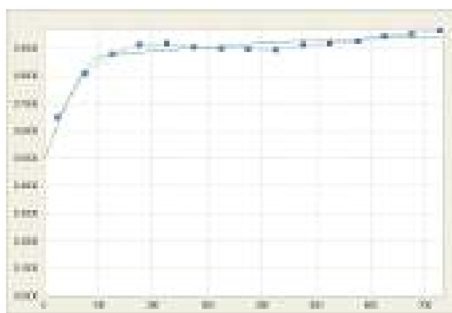
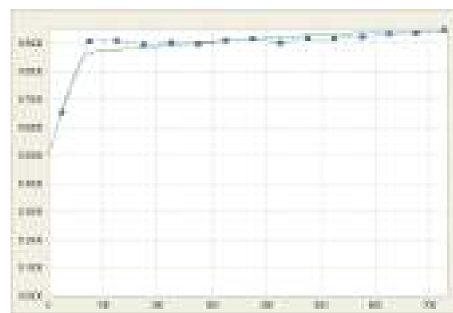

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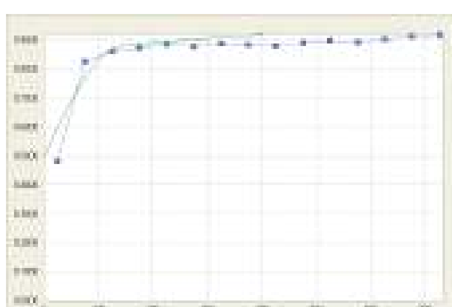


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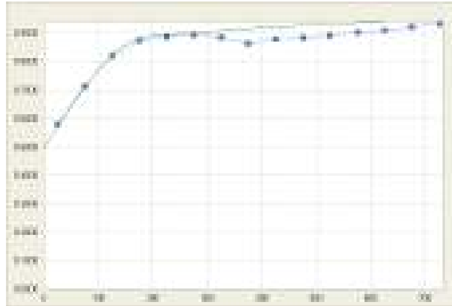

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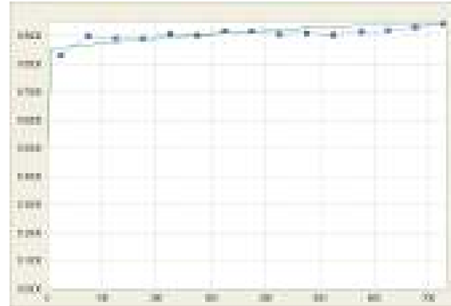

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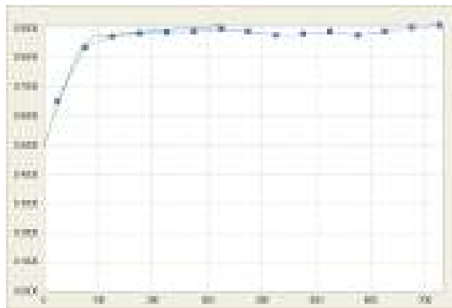

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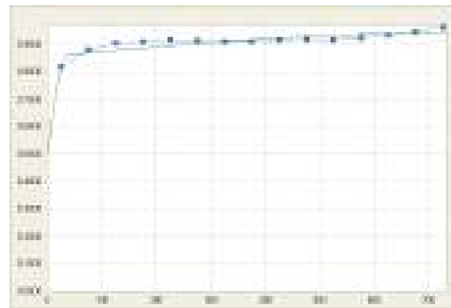

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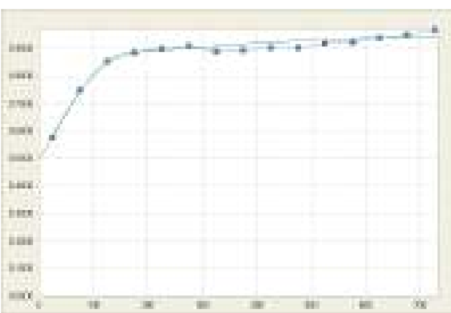

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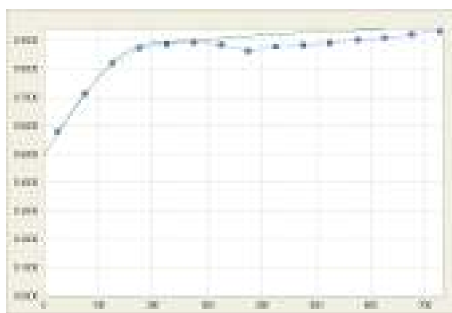

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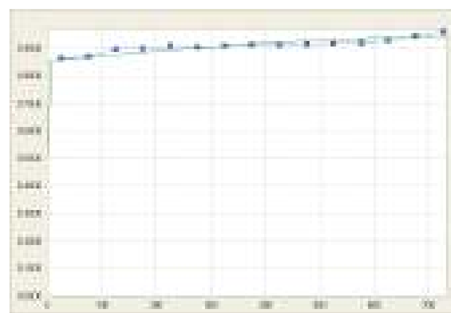

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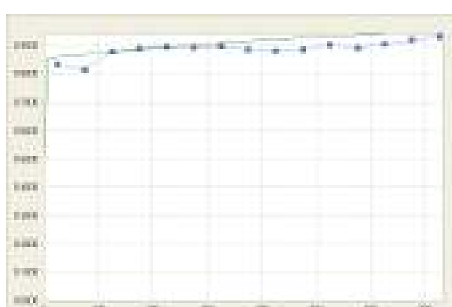

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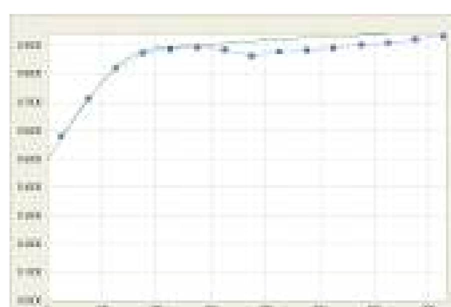

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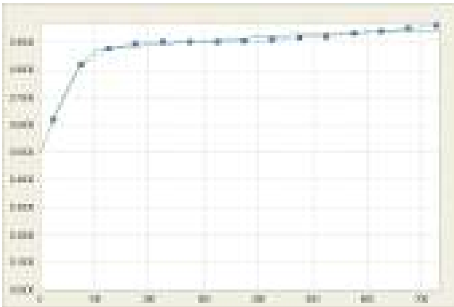

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au_ZONE_1,2,3,10_hrz_global.var

VETERAN GOLD
Estimation Parameters

Kriging All Zones	Pass 1
Minimum # of Composites	2
Maximum # of Composites	8
Maximum # of Composites per hole	3
Search distances (ft)	350/220/350
Search directions	220°/6°/-8°
Restriction on high grade	
Grade (Cu%)	none
Distance (ft)	none

Nugget	Model	Sill	Range 1	Range 2	Range 3	Rotn	Dipn	Dipe
0.50	First spherical model	0.36	98	62	100	223	6	-8
	Second spherical model	0.09	615	750	750	211	3	-55

Appendix C: Veteran Estimation Parameters and Variograms – Soluble Copper

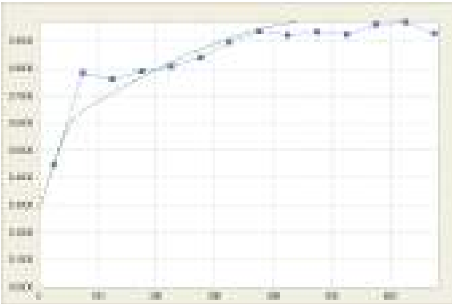
VETERAN SOLUBLE COPPER RATIO
Estimation Parameters

Kriging Zone 1 (Hypogene)	Pass 1	Nugget	Model	Sill	Range 1	Range 2	Range 3	Rotn	Dipn	Dipe
Minimum # of Composites	2	0.28	First spherical model	0.43	100	32	179	204	15	-8
Maximum # of Composites	10		Second spherical model	0.28	700	556	700	129	5	-43
Maximum # of Composites per hole	3									
Search distances (ft)	150/60/250									
Search directions	200°/15°/-8°									
Restriction on high grade										
Grade (Cu%)	none									
Distance (ft)	none									

Kriging Zone 2 (CEB)	Pass 1	Nugget	Model	Sill	Range 1	Range 2	Range 3	Rotn	Dipn	Dipe
Minimum # of Composites	1	0.31	First spherical model	0.52	231	200	164	0	46	-37
Maximum # of Composites	7		Second spherical model	0.17	26	16	693	193	2	-5
Maximum # of Composites per hole	3									
Search distances (ft)	250/200/180									
Search directions	0°/46°/-28°									
Restriction on high grade										
Grade (Cu%)	none									
Distance (ft)	none									

Kriging Zone 3 (Leach Cap)	Pass 1	Nugget	Model	Sill	Range 1	Range 2	Range 3	Rotn	Dipn	Dipe
Minimum # of Composites	1	0.29	First spherical model	0.53	40	80	209	84	10	6
Maximum # of Composites	7		Second spherical model	0.18	618	248	503	136	28	-6
Maximum # of Composites per hole	3									
Search distances (ft)	250/200/180									
Search directions	0°/46°/-28°									
Restriction on high grade										
Grade (Cu%)	none									
Distance (ft)	none									

Kriging Zone 10 (other)	Pass 1	Nugget	Model	Sill	Range 1	Range 2	Range 3	Rotn	Dipn	Dipe
Minimum # of Composites	1	0.08	First spherical model	0.97	114	233	233	96	-4	-7
Maximum # of Composites	7									
Maximum # of Composites per hole	3									
Search distances (ft)	150/250/350/									
Search directions	96°/-4°/-7°									
Restriction on high grade										
Grade (Cu%)	none									
Distance (ft)	none									



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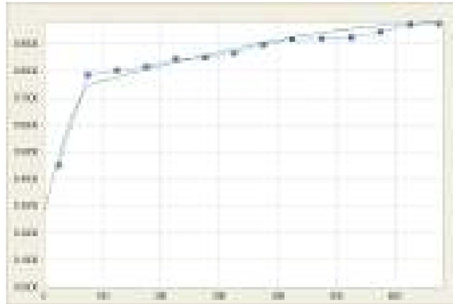

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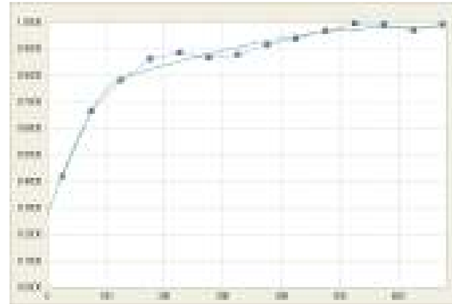

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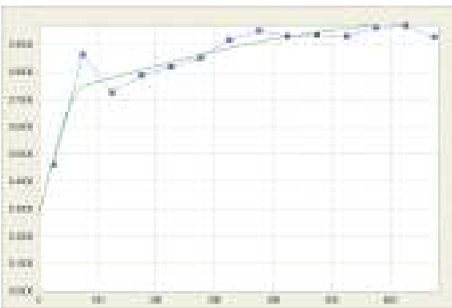

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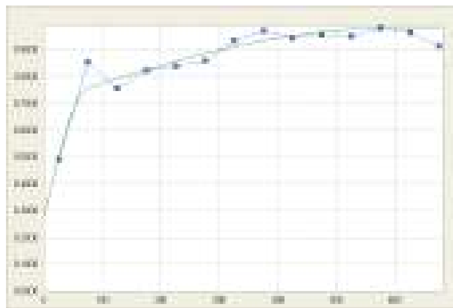

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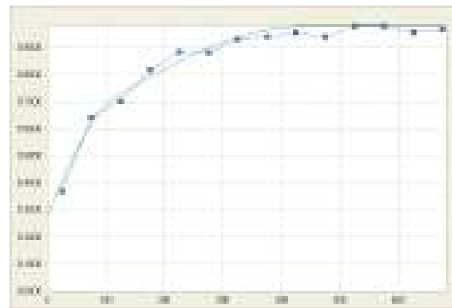

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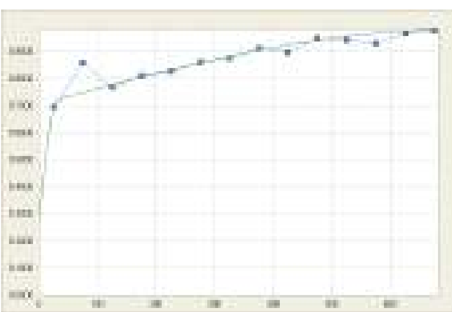

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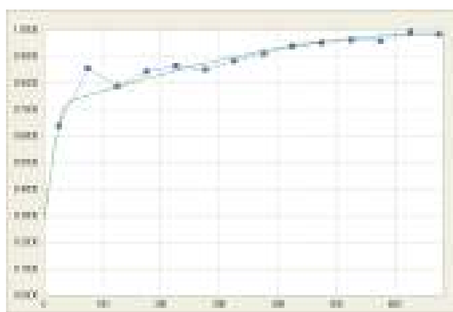

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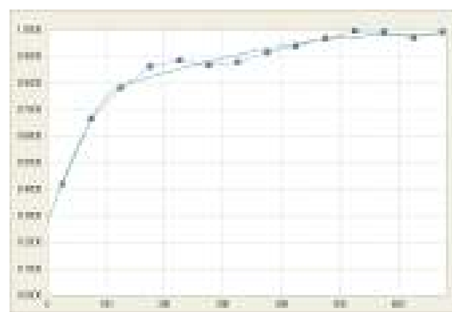

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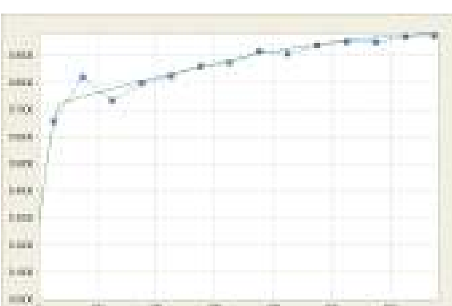

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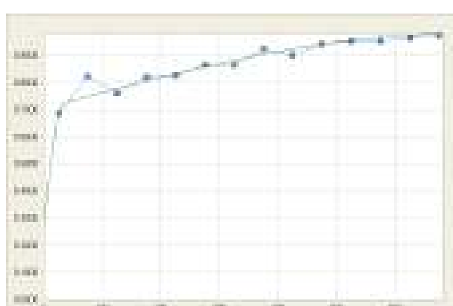

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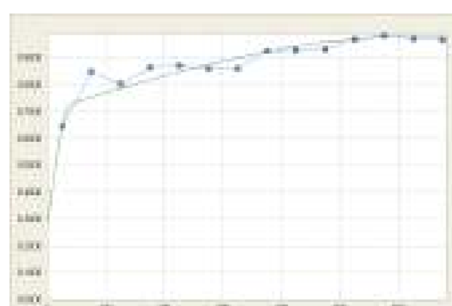

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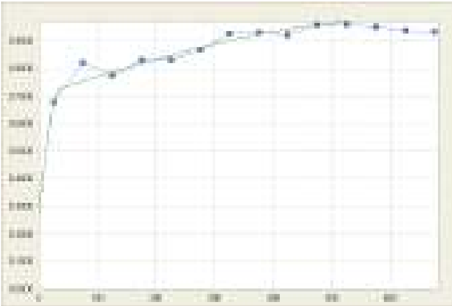

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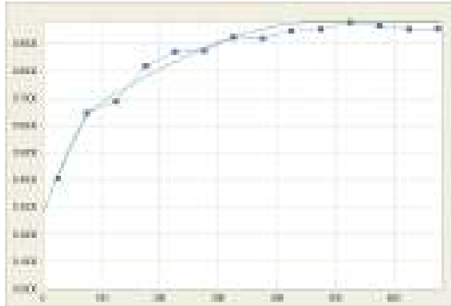

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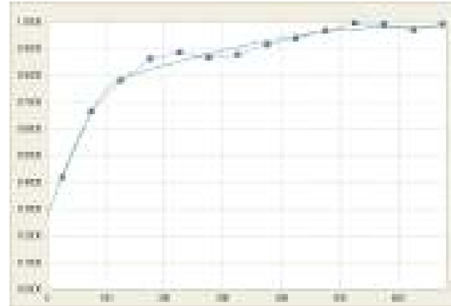

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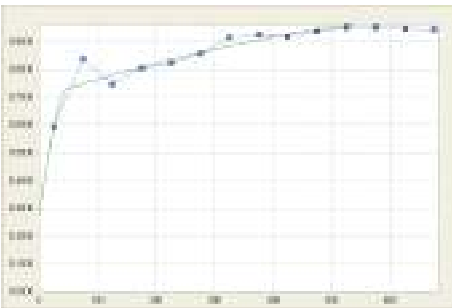

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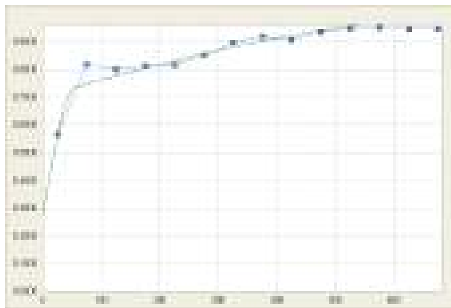

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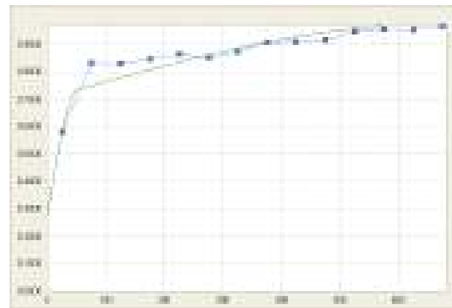

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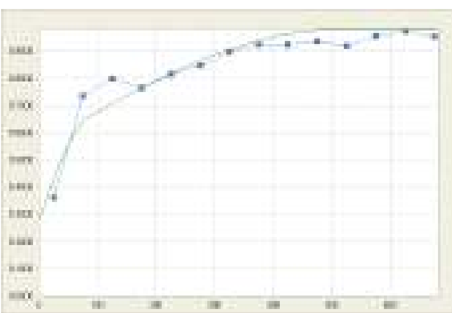

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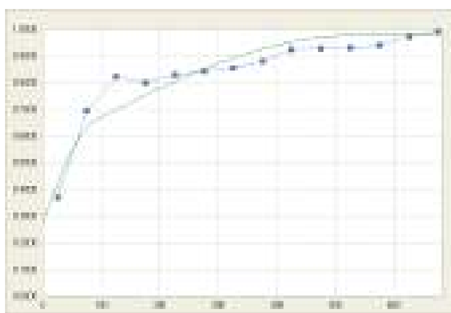

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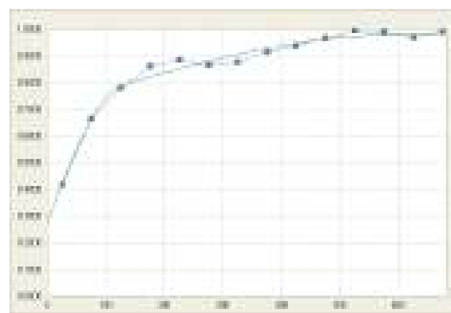

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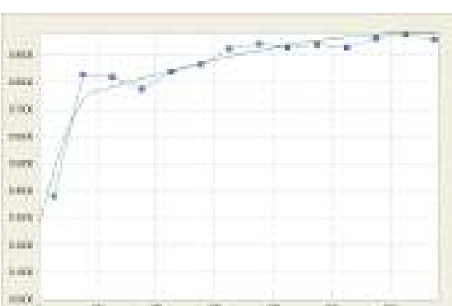

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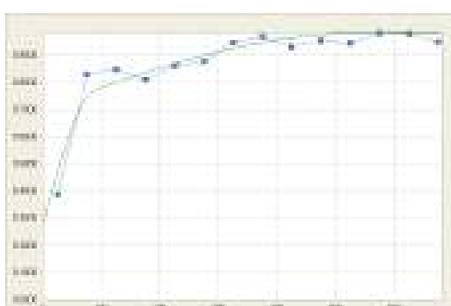

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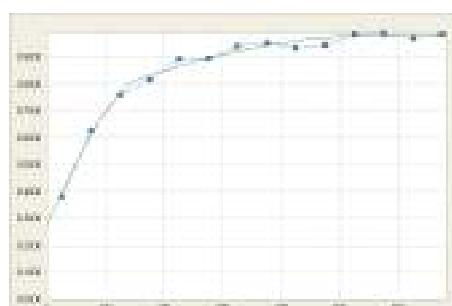

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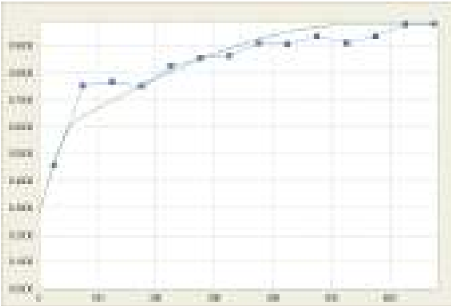

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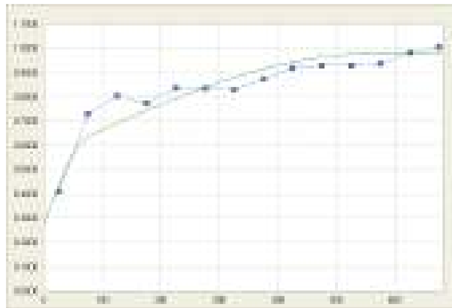

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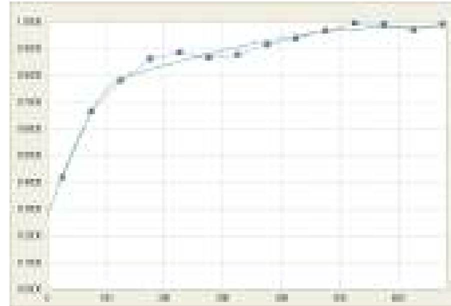

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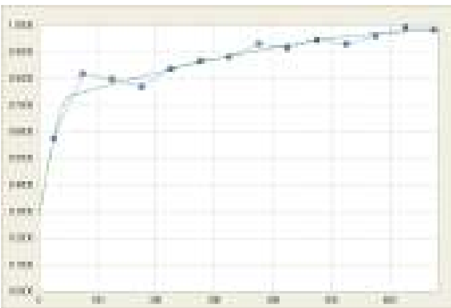

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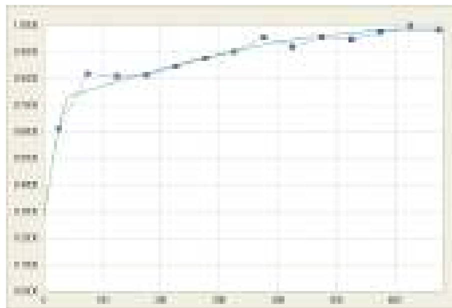

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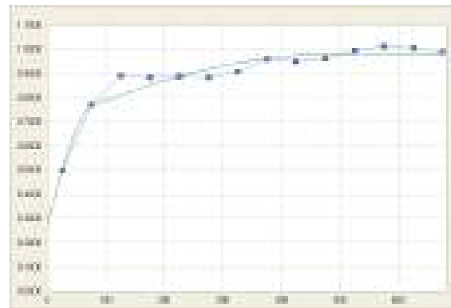

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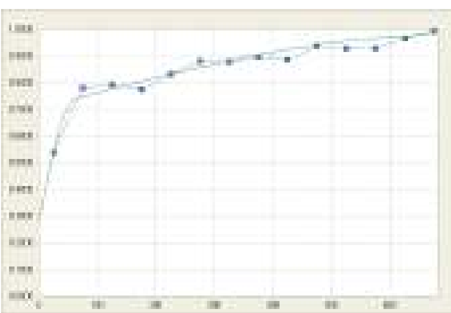

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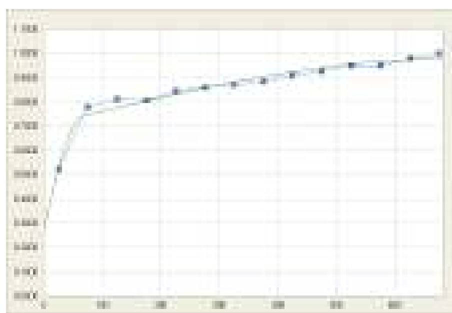

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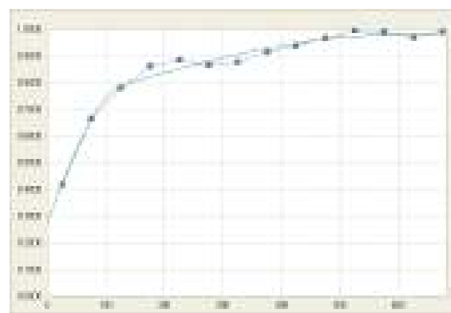

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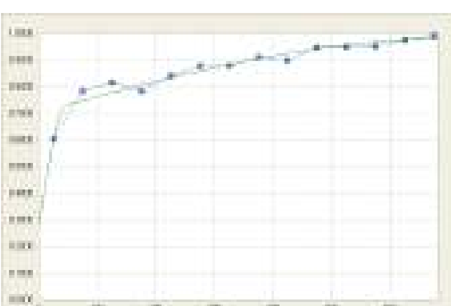

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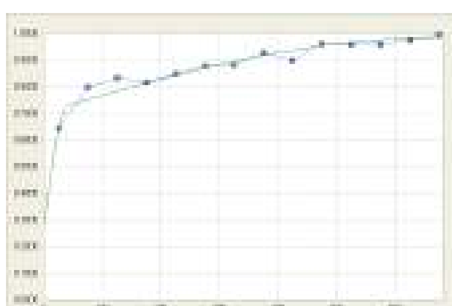

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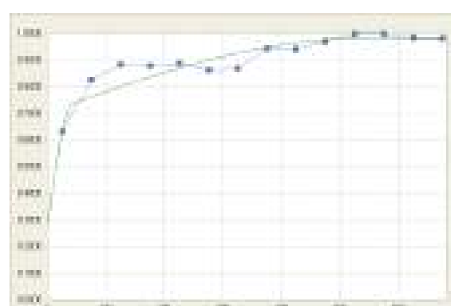

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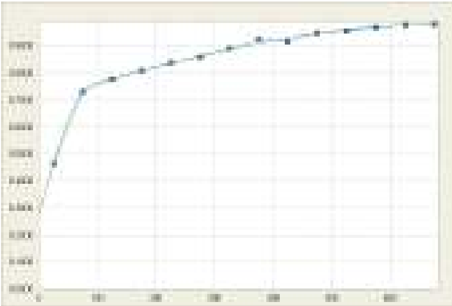

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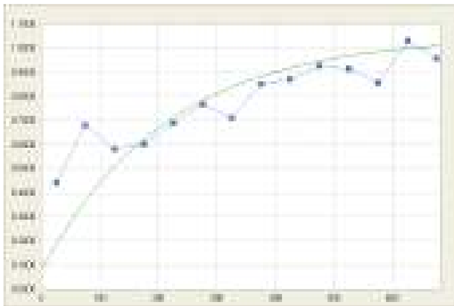

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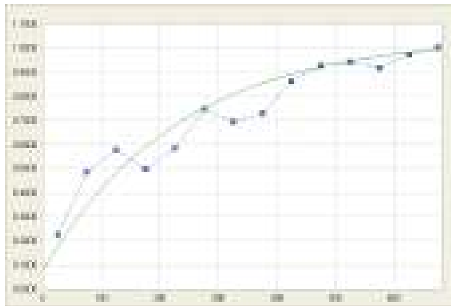

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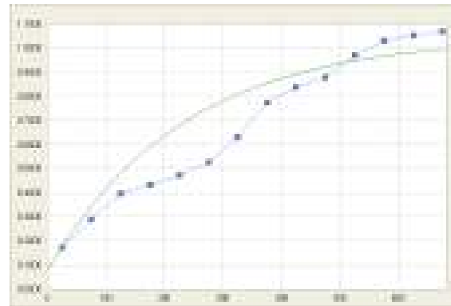

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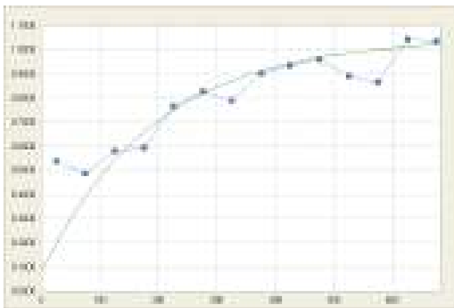

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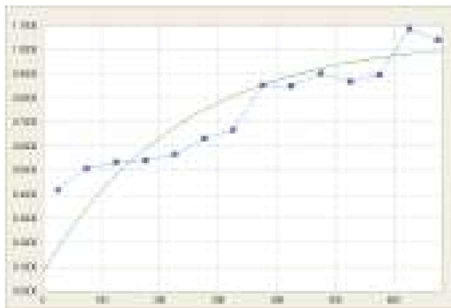

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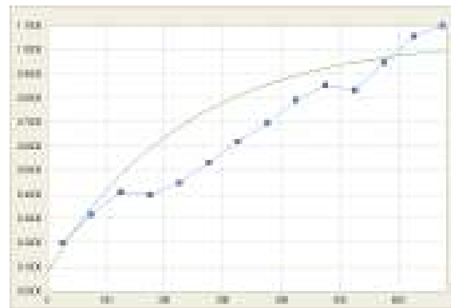

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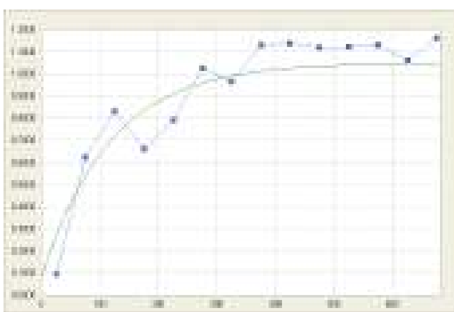

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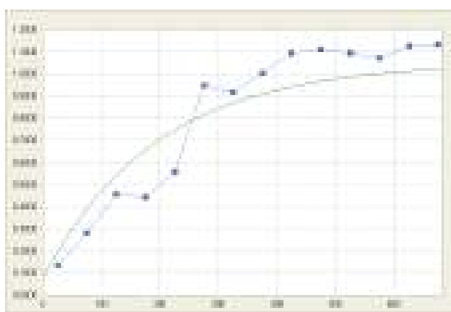

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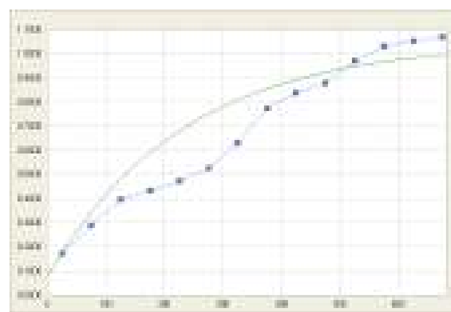

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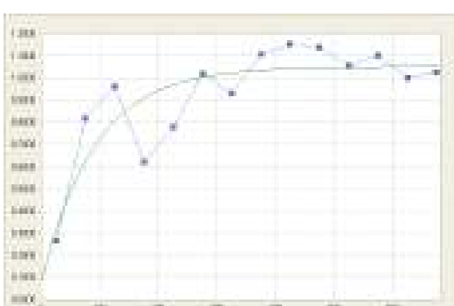

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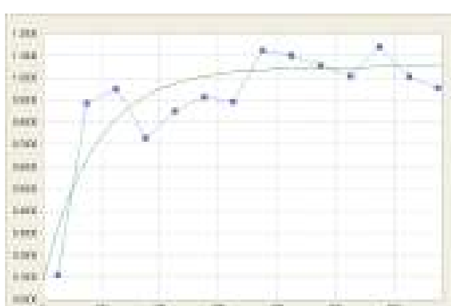

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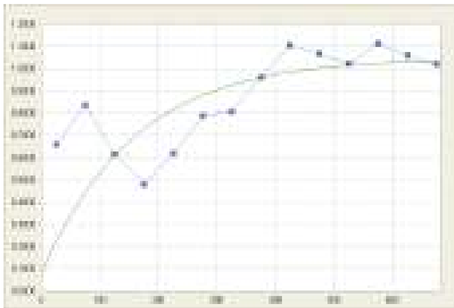
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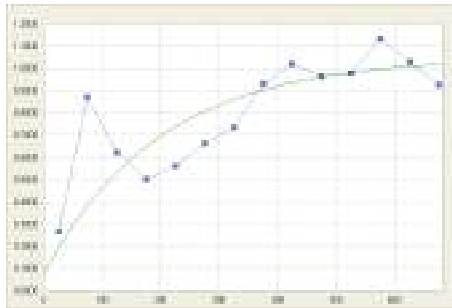
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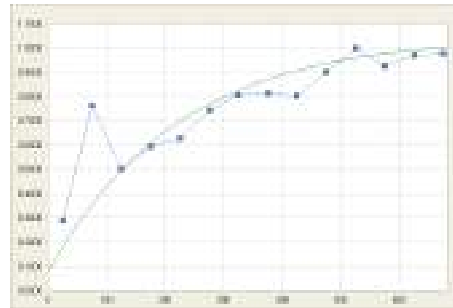
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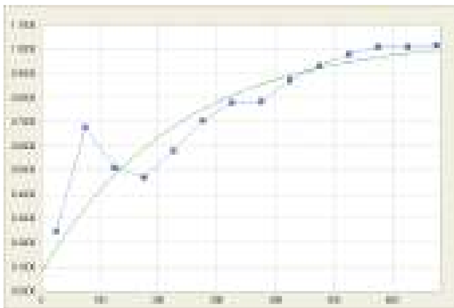
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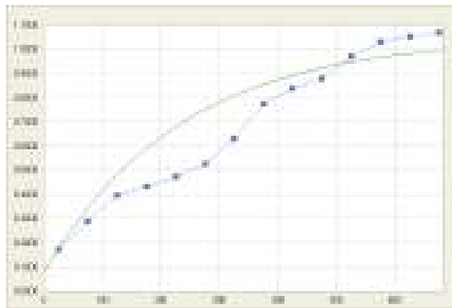
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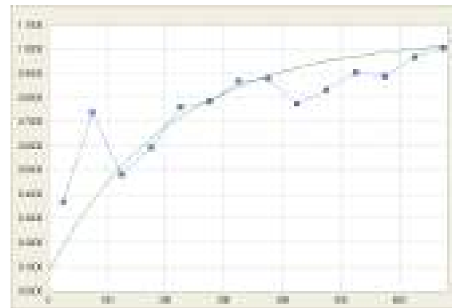
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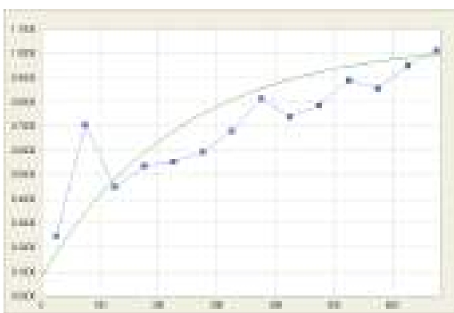
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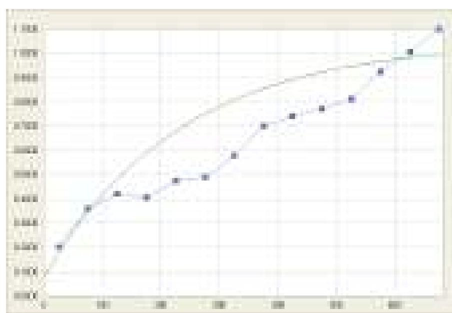
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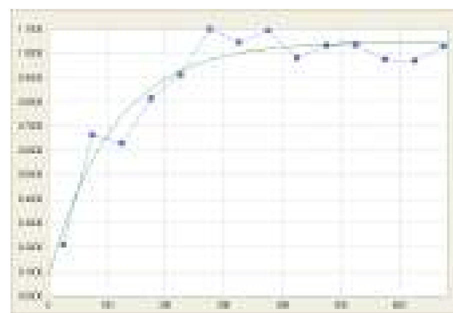
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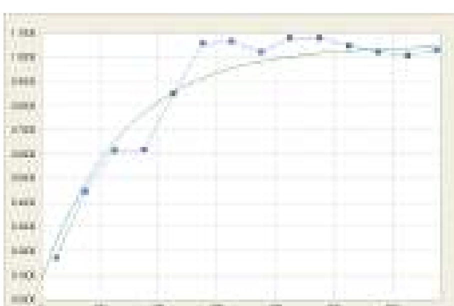
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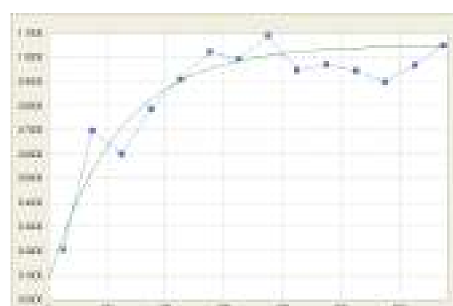
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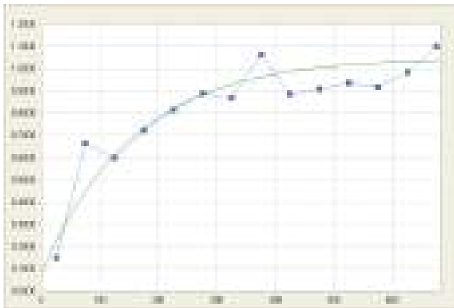

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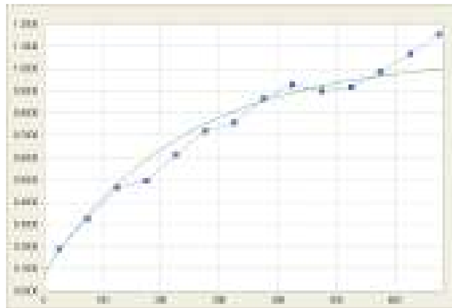

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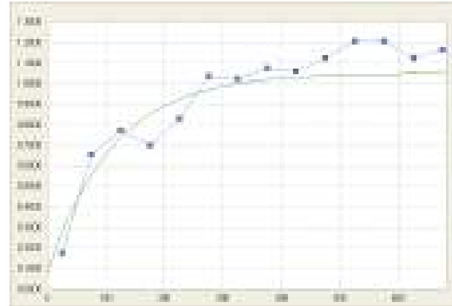

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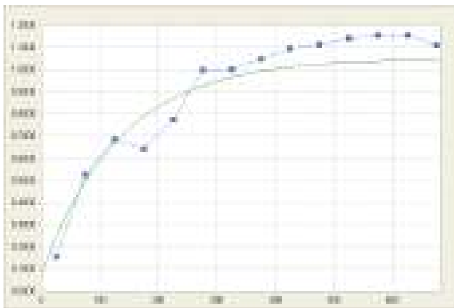

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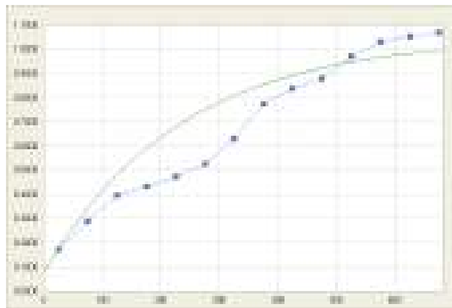

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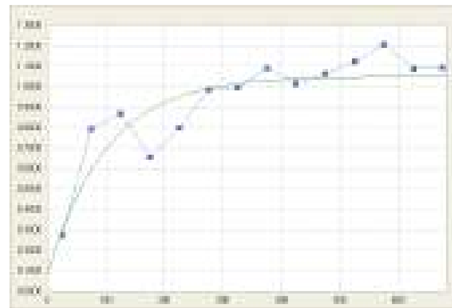

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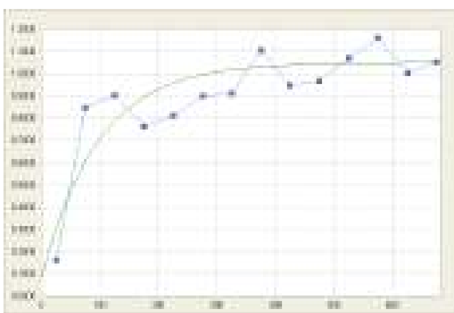

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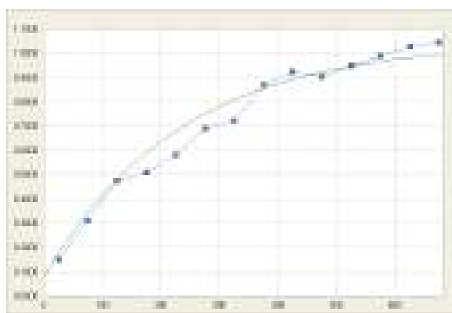

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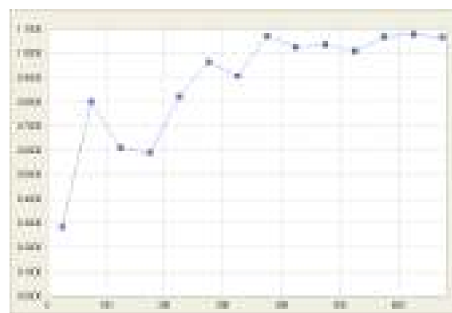

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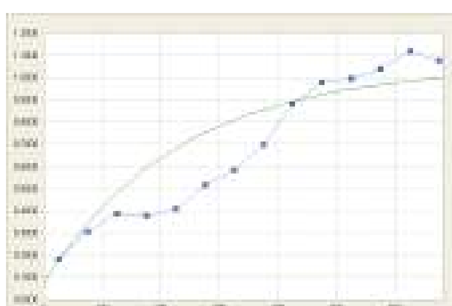

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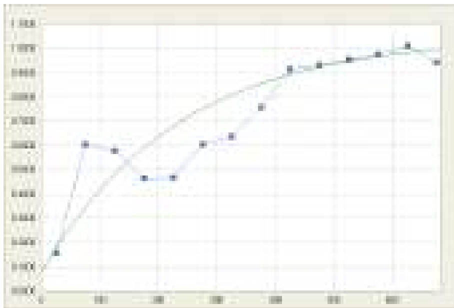

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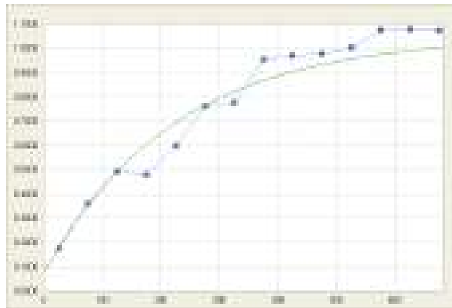

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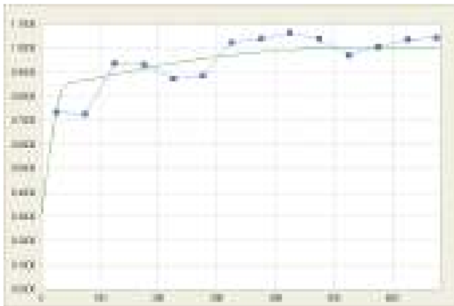
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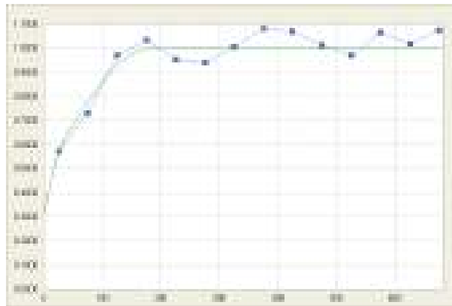
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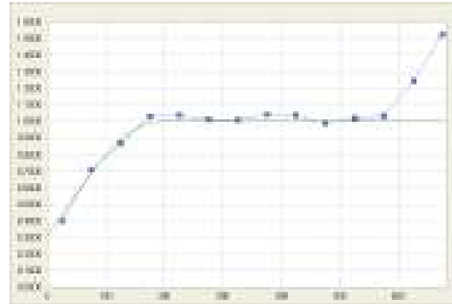
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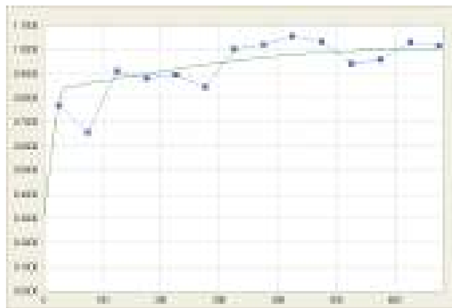
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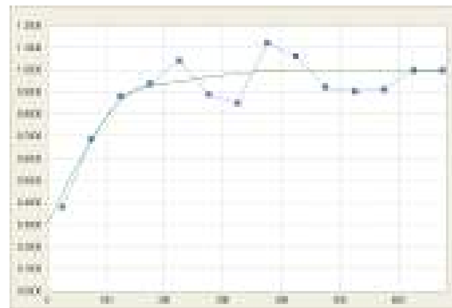
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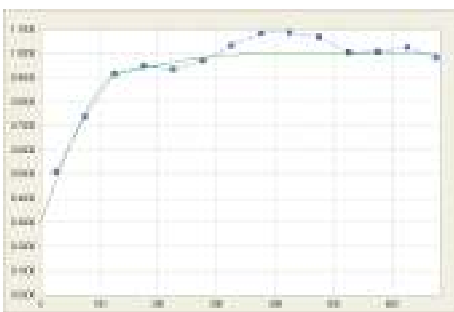
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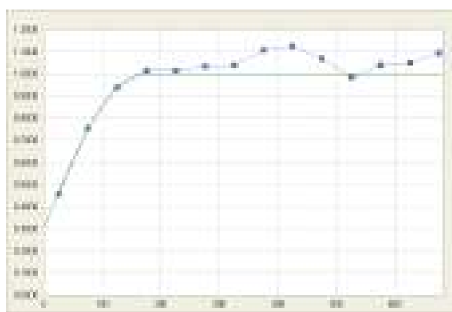
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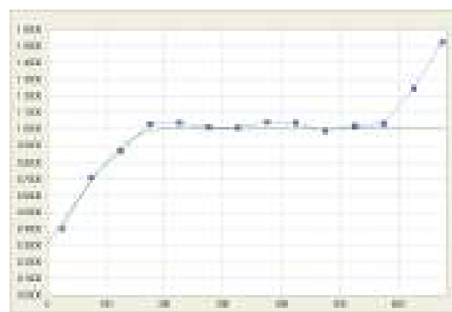
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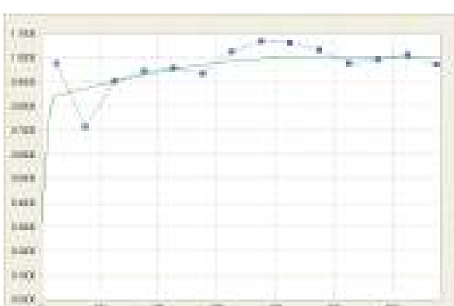
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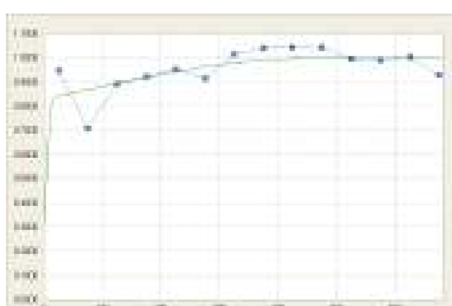
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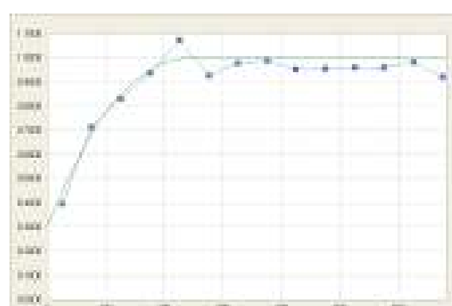
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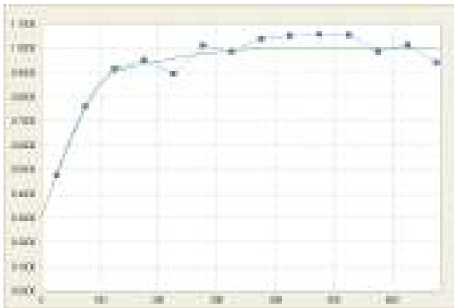

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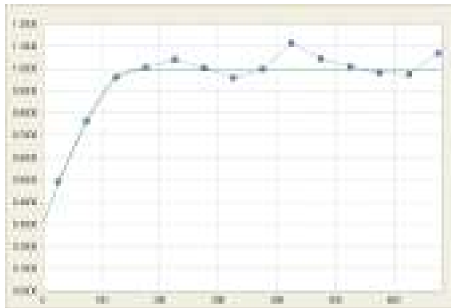

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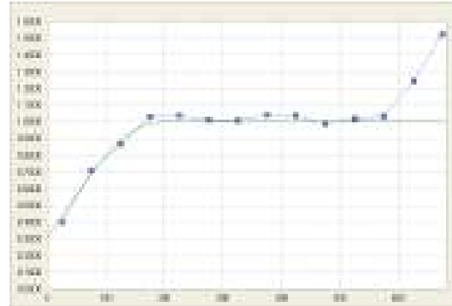

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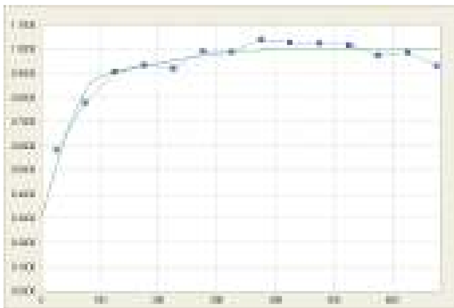

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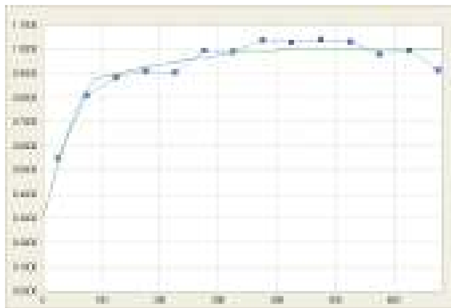

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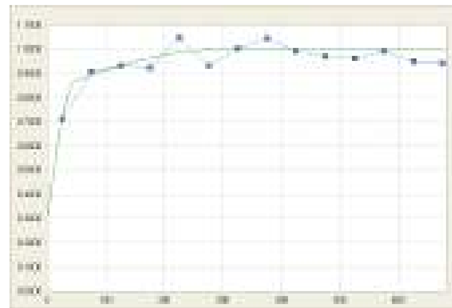

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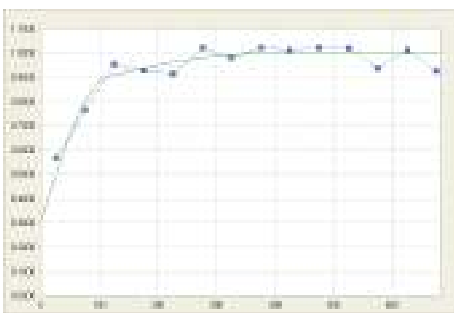

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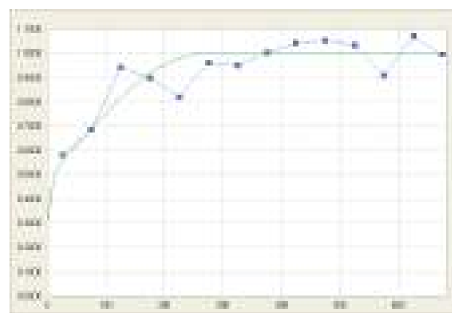

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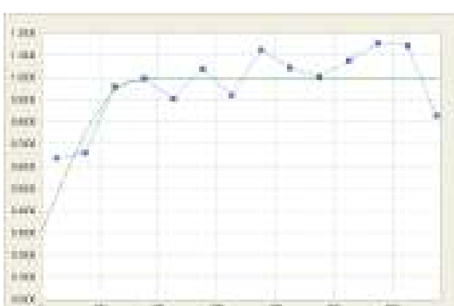

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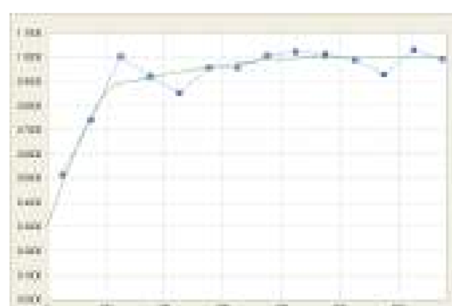

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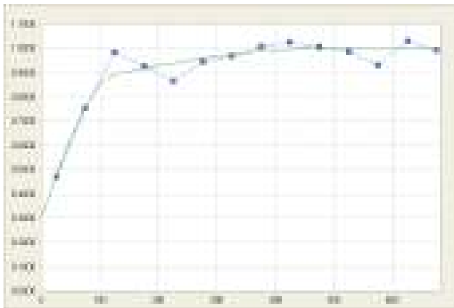

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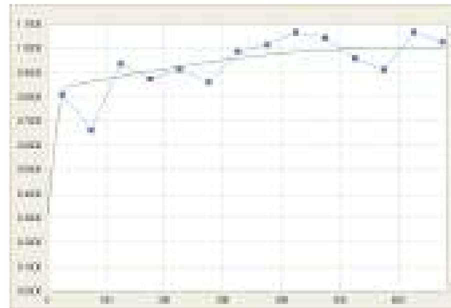

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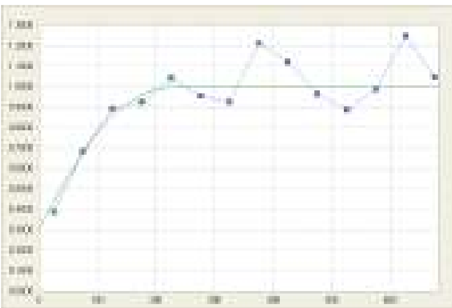

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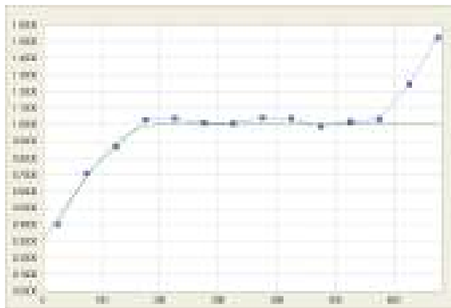

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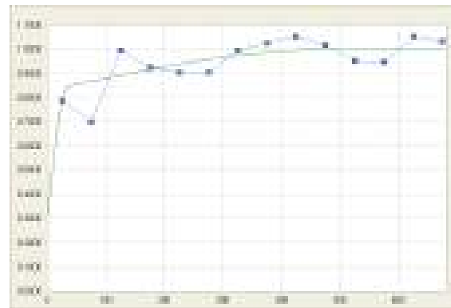

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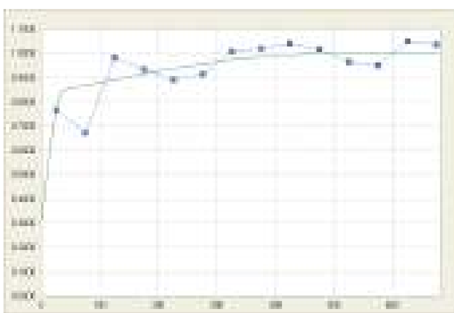

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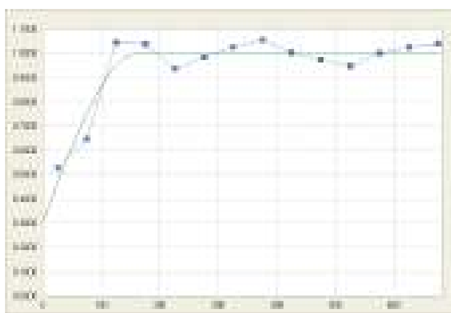

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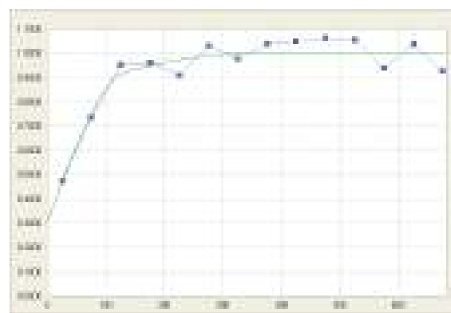

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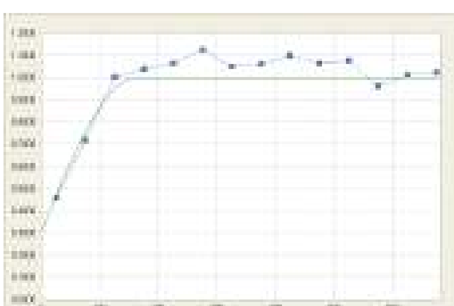

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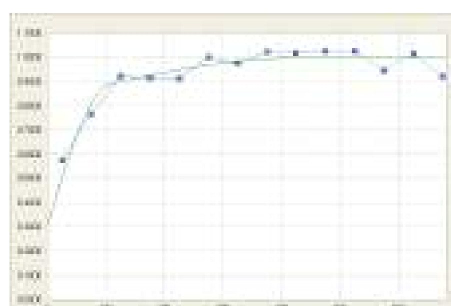

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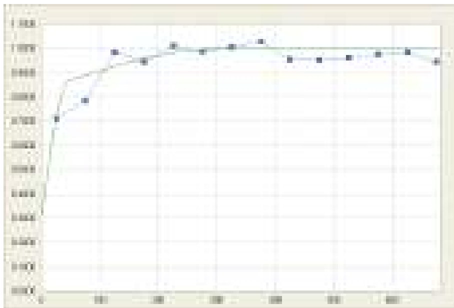

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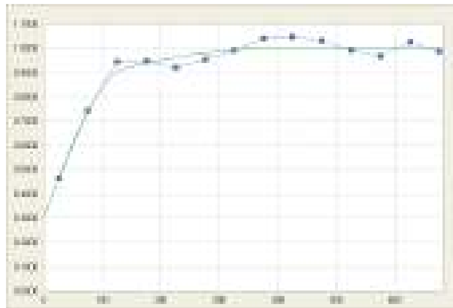

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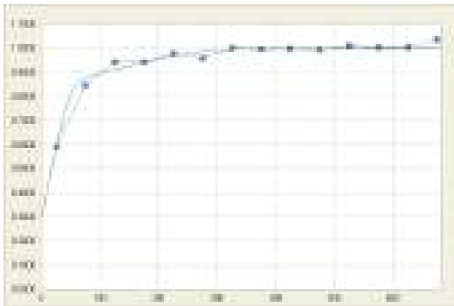
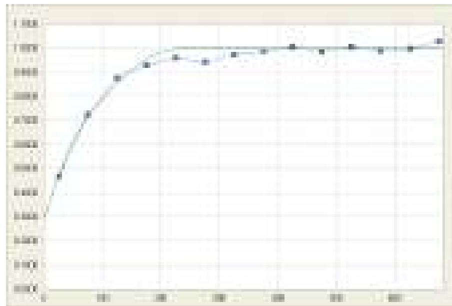
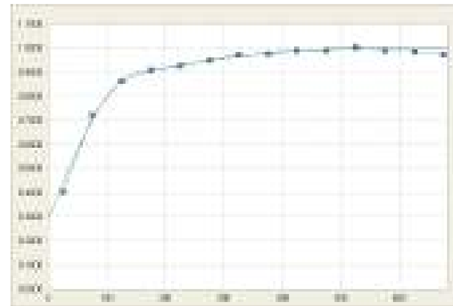


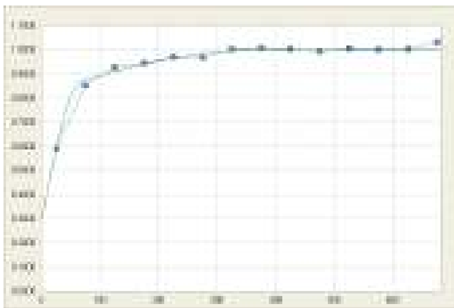
oxrat_ZONE_3_0_-30.var



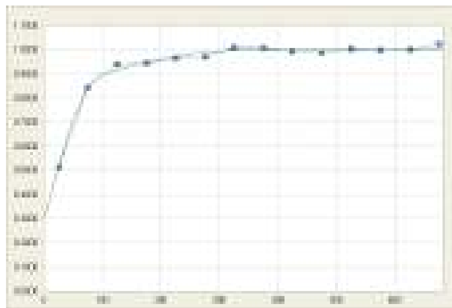
oxrat_ZONE_3_0_-60.var



oxrat_ZONE_3_0_-90.var



oxrat_ZONE_3_0_0.var



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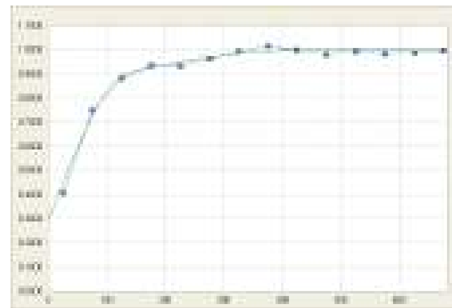
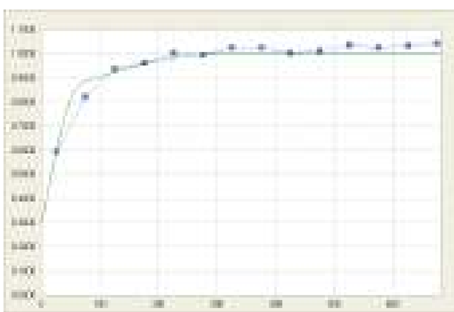


oxrat_ZONE_3_0_60.var



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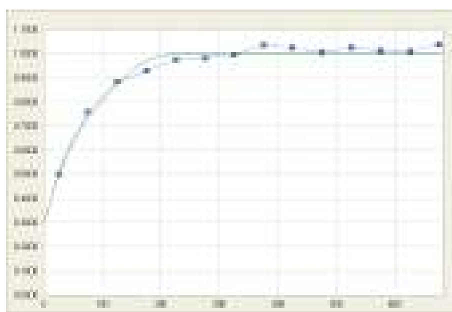
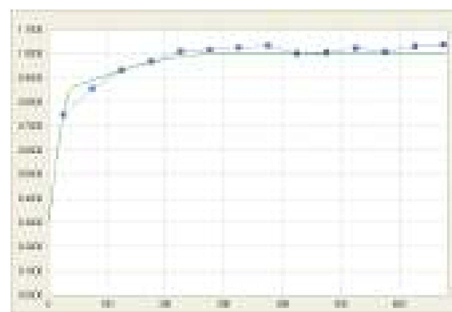


oxrat_ZONE_3_30_-60.var



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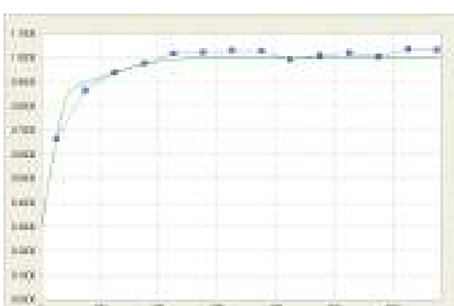
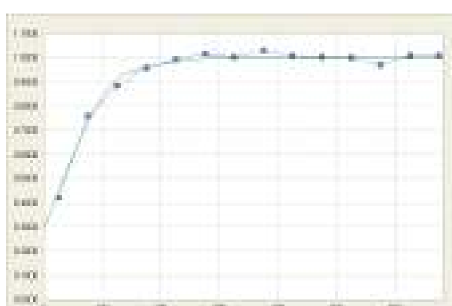


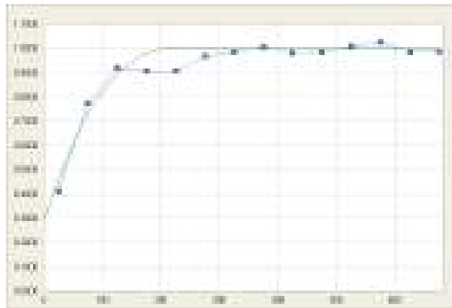
oxrat_ZONE_3_30_30.var



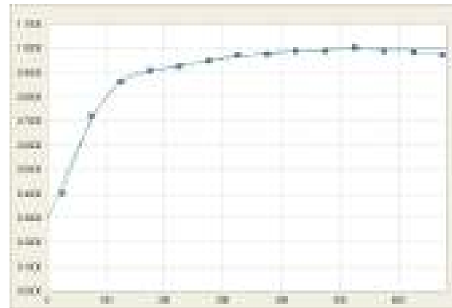
oxrat_ZONE_3_30_60.var

oxrat_ZONE_3_hrz_global.var

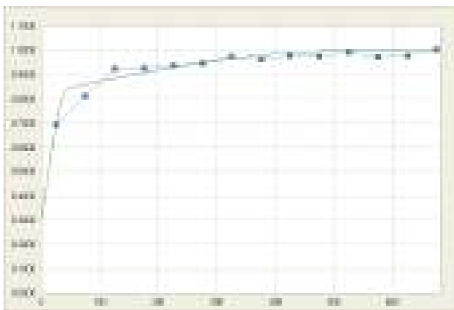
oxrat_ZONE_3_120_-30.var



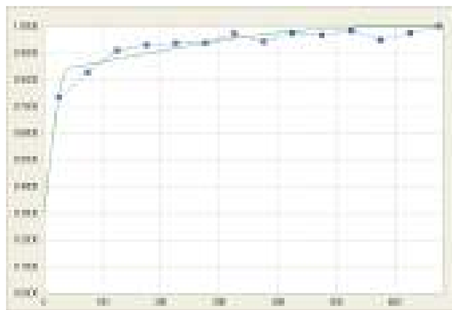
oxrat_ZONE_3_120_-60.var



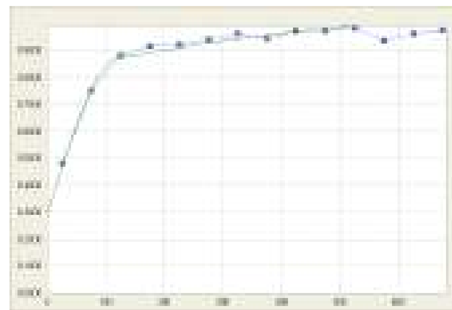
oxrat_ZONE_3_120_-90.var



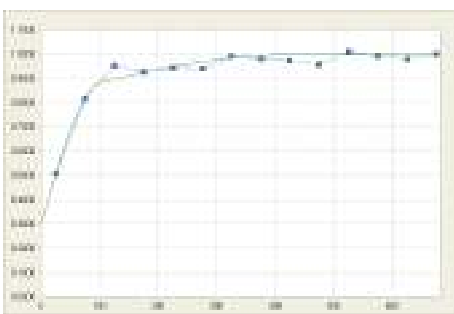
oxrat_ZONE_3_120_0.var



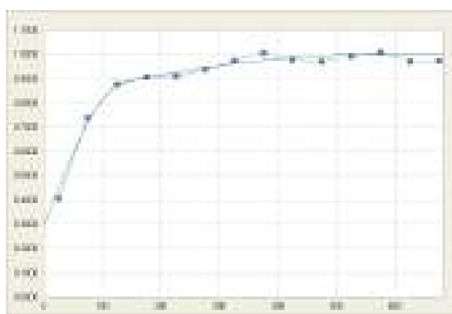
oxrat_ZONE_3_120_30.var



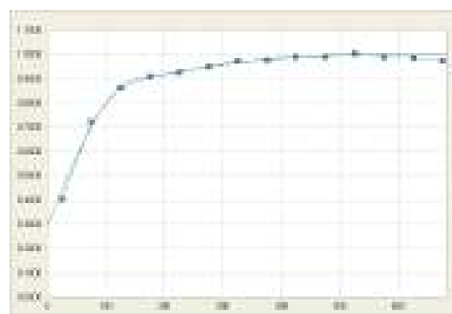
oxrat_ZONE_3_120_60.var



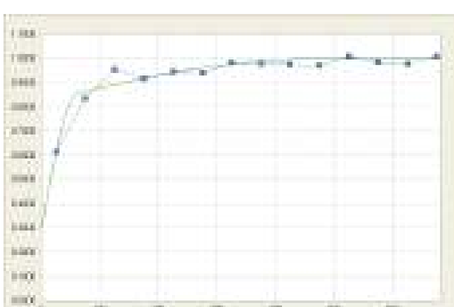
oxrat_ZONE_3_150_-30.var



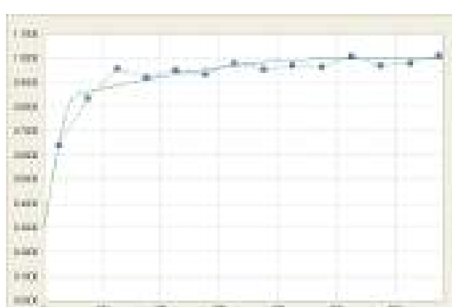
oxrat_ZONE_3_150_-60.var



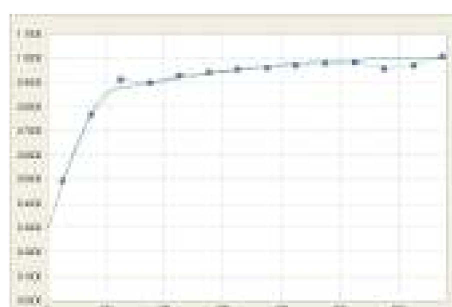
oxrat_ZONE_3_150_-90.var



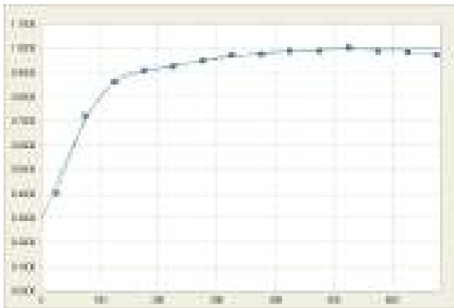
oxrat_ZONE_3_150_0.var



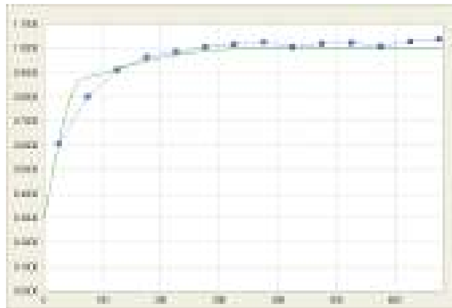
oxrat_ZONE_3_150_30.var



oxrat_ZONE_3_150_60.var



oxrat_ZONE_3_30_-90.var



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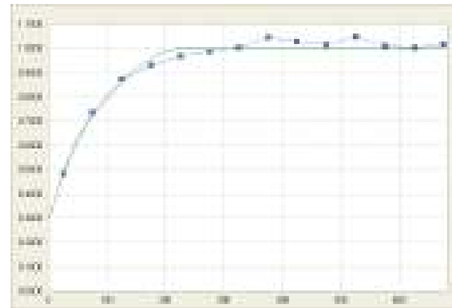
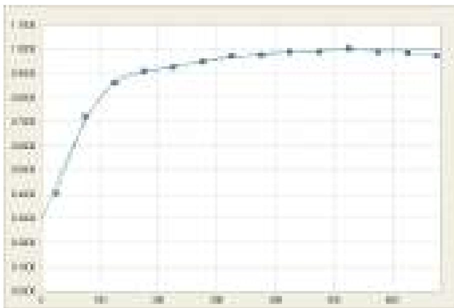
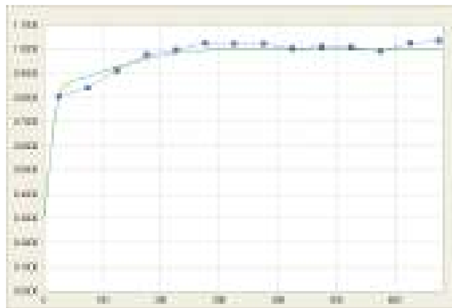


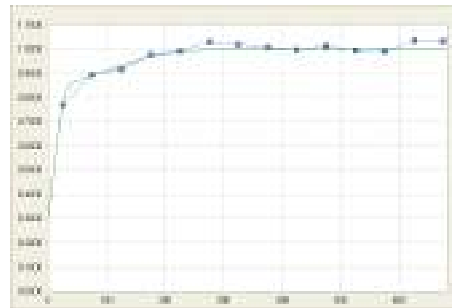
oxrat_ZONE_3_60_-60.var



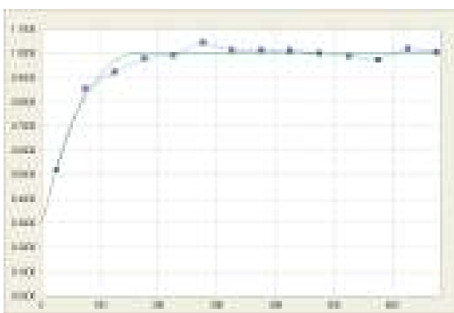
oxrat_ZONE_3_60_-90.var



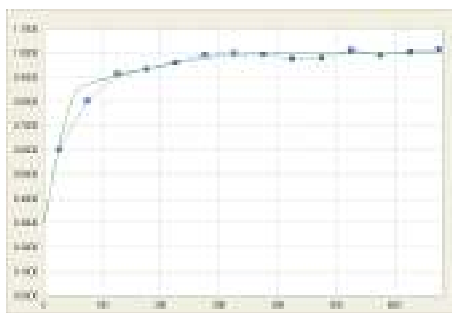
oxrat_ZONE_3_60_0.var



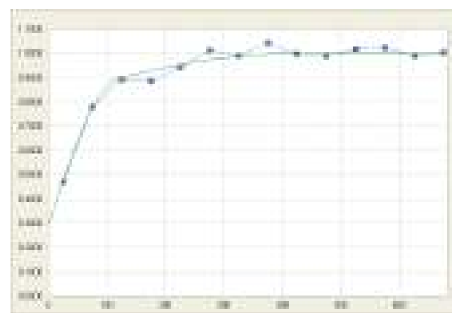
oxrat_ZONE_3_60_30.var



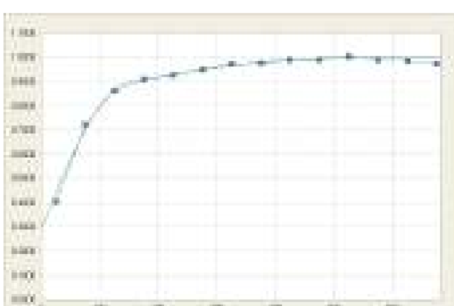
oxrat_ZONE_3_60_60.var



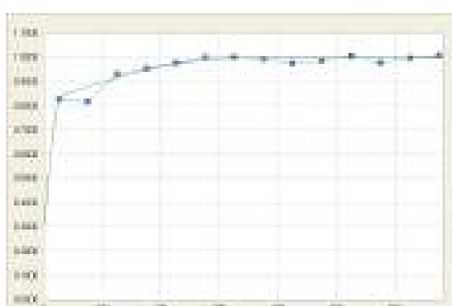
oxrat_ZONE_3_90_-30.var



oxrat_ZONE_3_90_-60.var

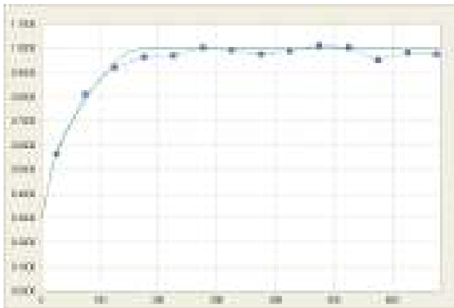


oxrat_ZONE_3_90_-90.var

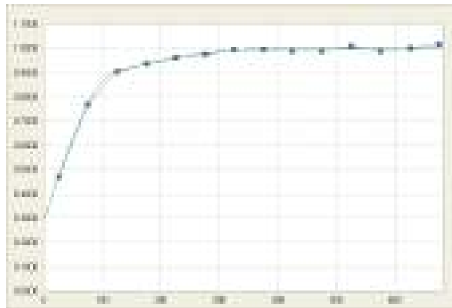


oxrat_ZONE_3_90_0.var

oxrat_ZONE_3_90_30.var



oxrat_ZONE_3_90_60.var



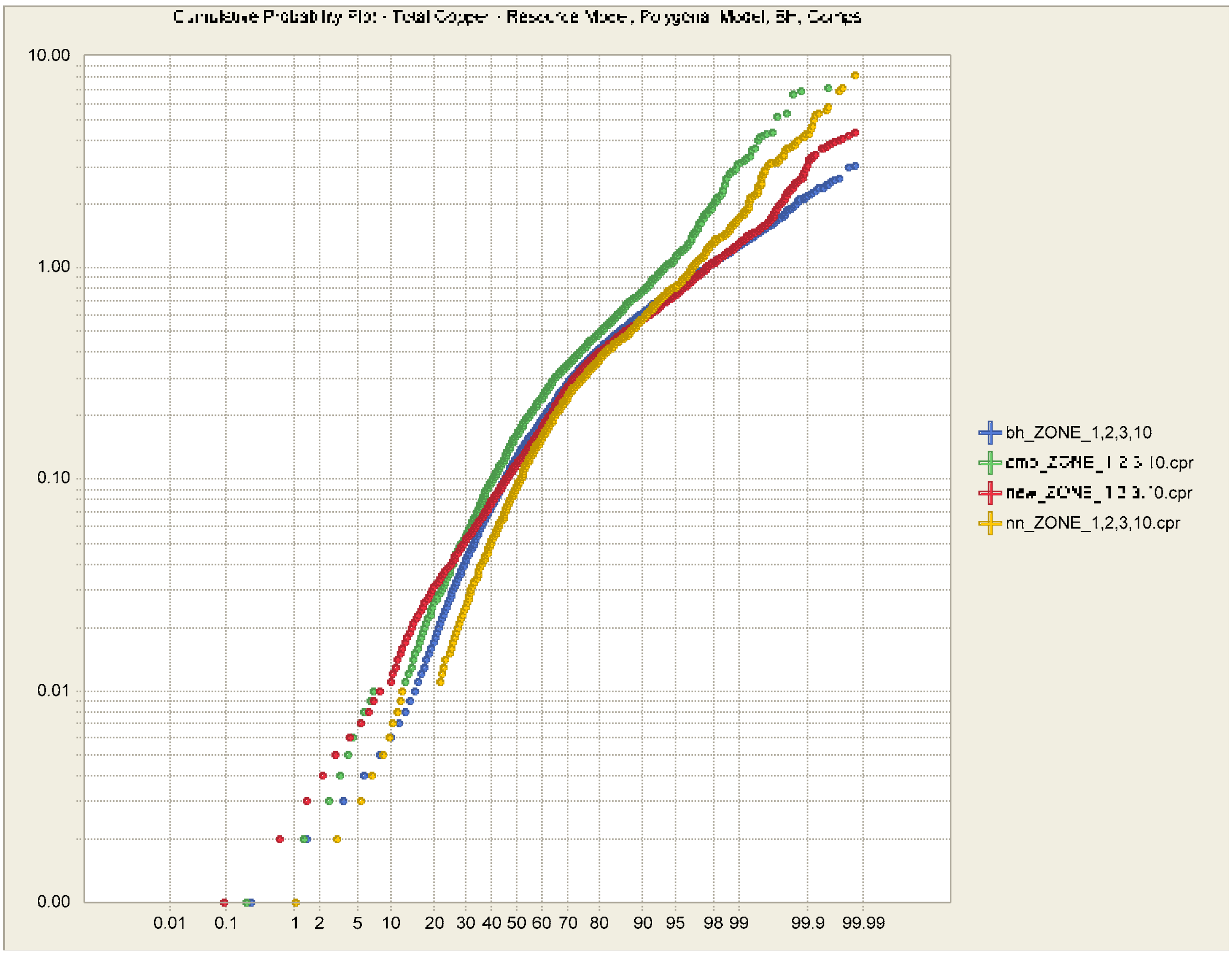
oxrat_ZONE_3_global.var

Appendix D:
Veteran Grade Distribution Plots
Including Blast Hole Data

Cumulative Probability Plots
Veteran Deposit
(Blast-hole data through April 2008)

Plot show the following grades all zones combined:

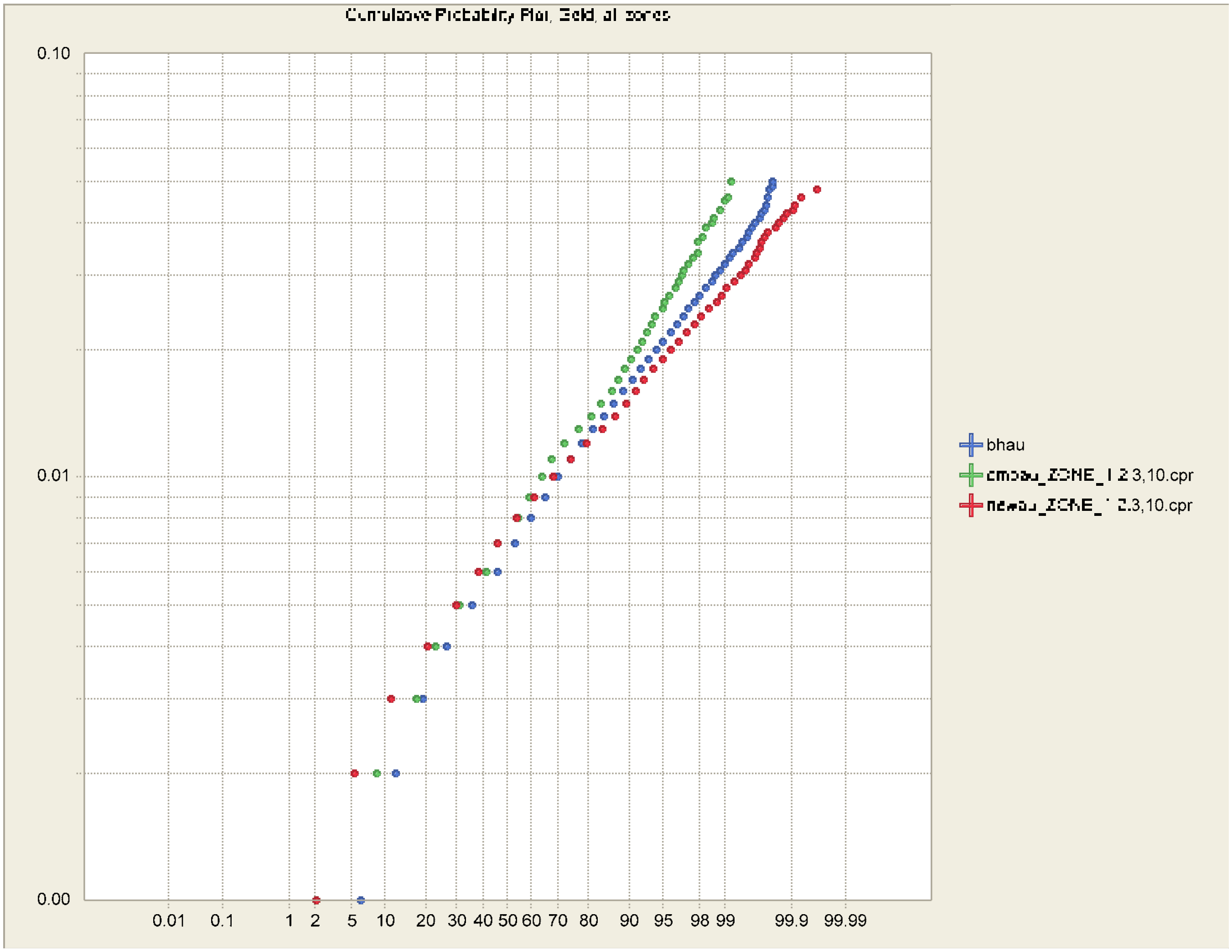
Item	**Label on chart**
Resource Model Total Copper	(new_ZONE)
Blast Hole Total Copper	(bh_ZONE)
Composites Total Copper	(cmp_ZONE)
Polygonal (nearest neighbor) Total Copper	(nn_ZONE)



Appendix D

Plot show the following grades all zones combined:

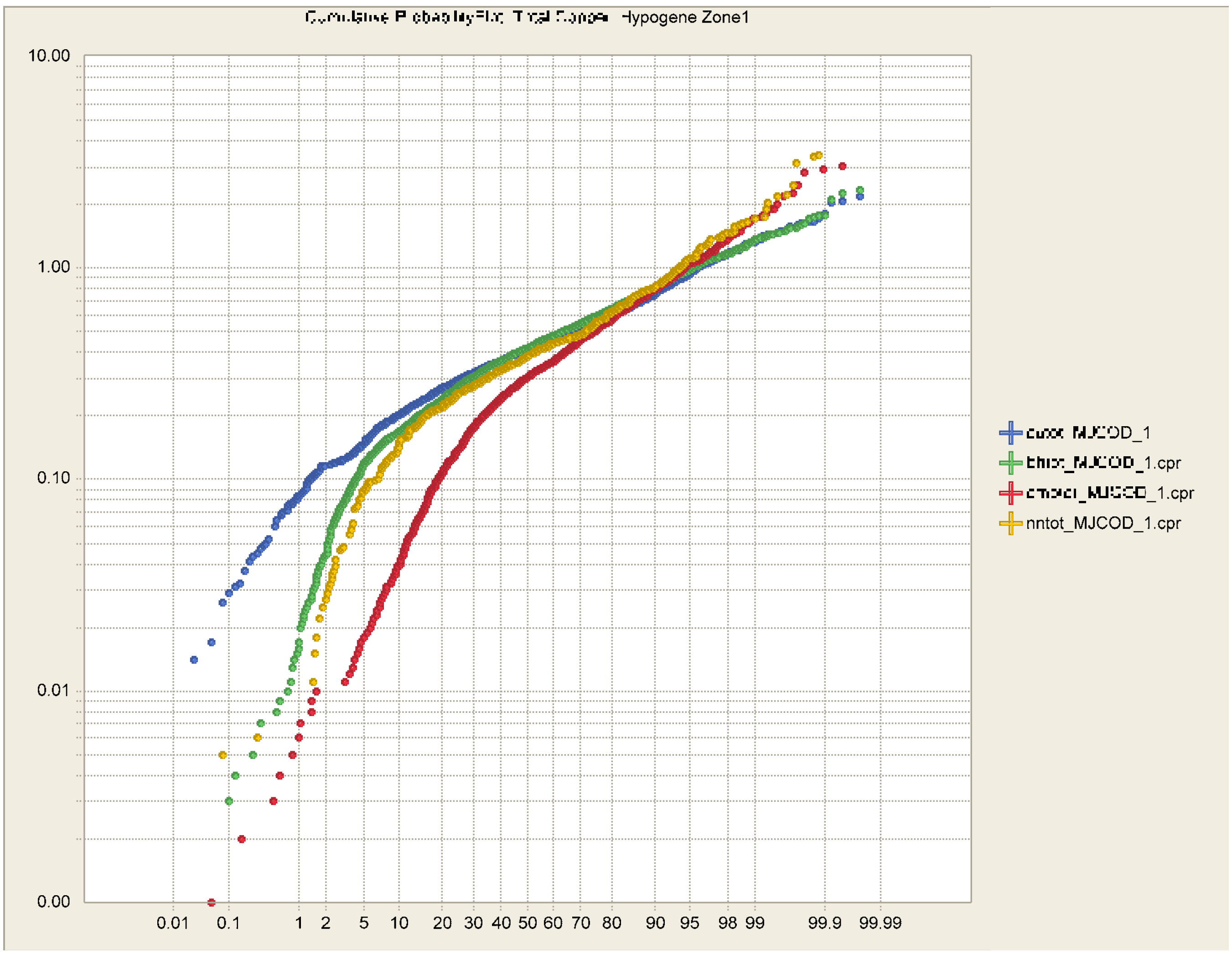
Item	**Label on chart**
Resource Model Gold	(newau_ZONE)
Blast Hole Gold	(bhau)
Composites Gold	(cmpau_ZONE)



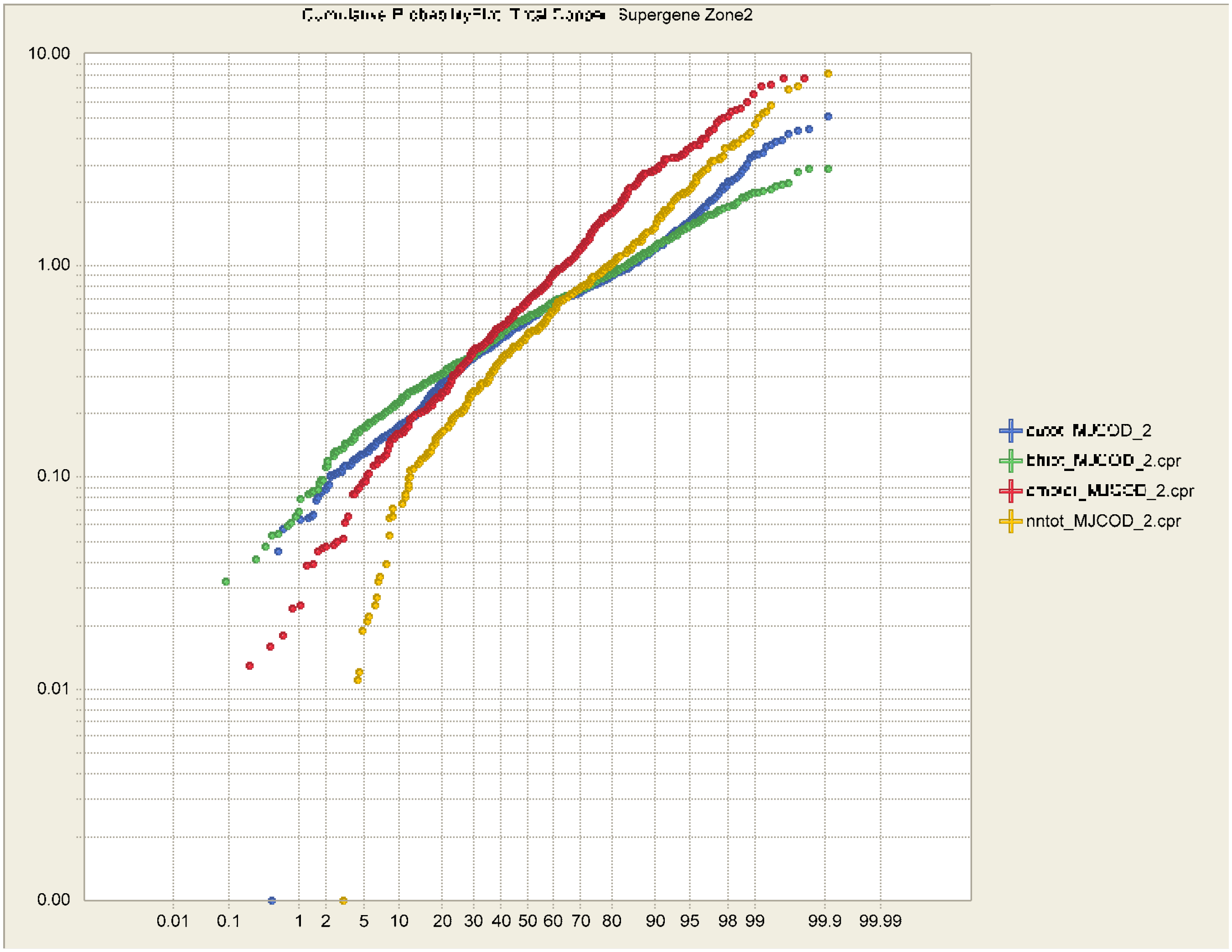
Cumulative Probability Plots
Veteran Deposit

Plots show the following grades by zone:

Item	**Label on chart**
Resource Model Total Copper	(cutot_MJCOD)
Blast Hole Total Copper	(bhtot_MJCOD)
Composites Total Copper	(cmptot_MJCOD)
Polygonal (nearest neighbor) Total Copper	(nntot_MJCOD)

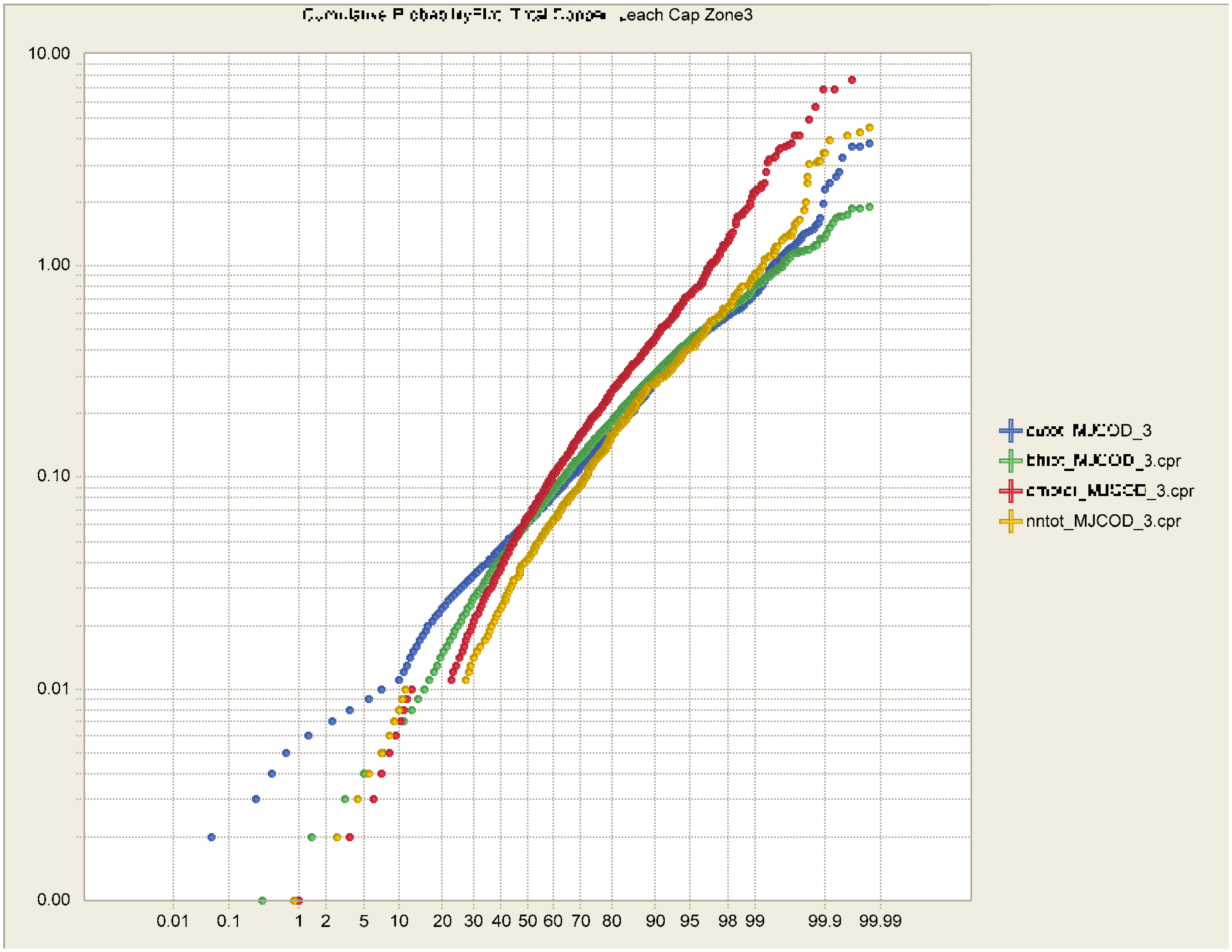


Appendix D



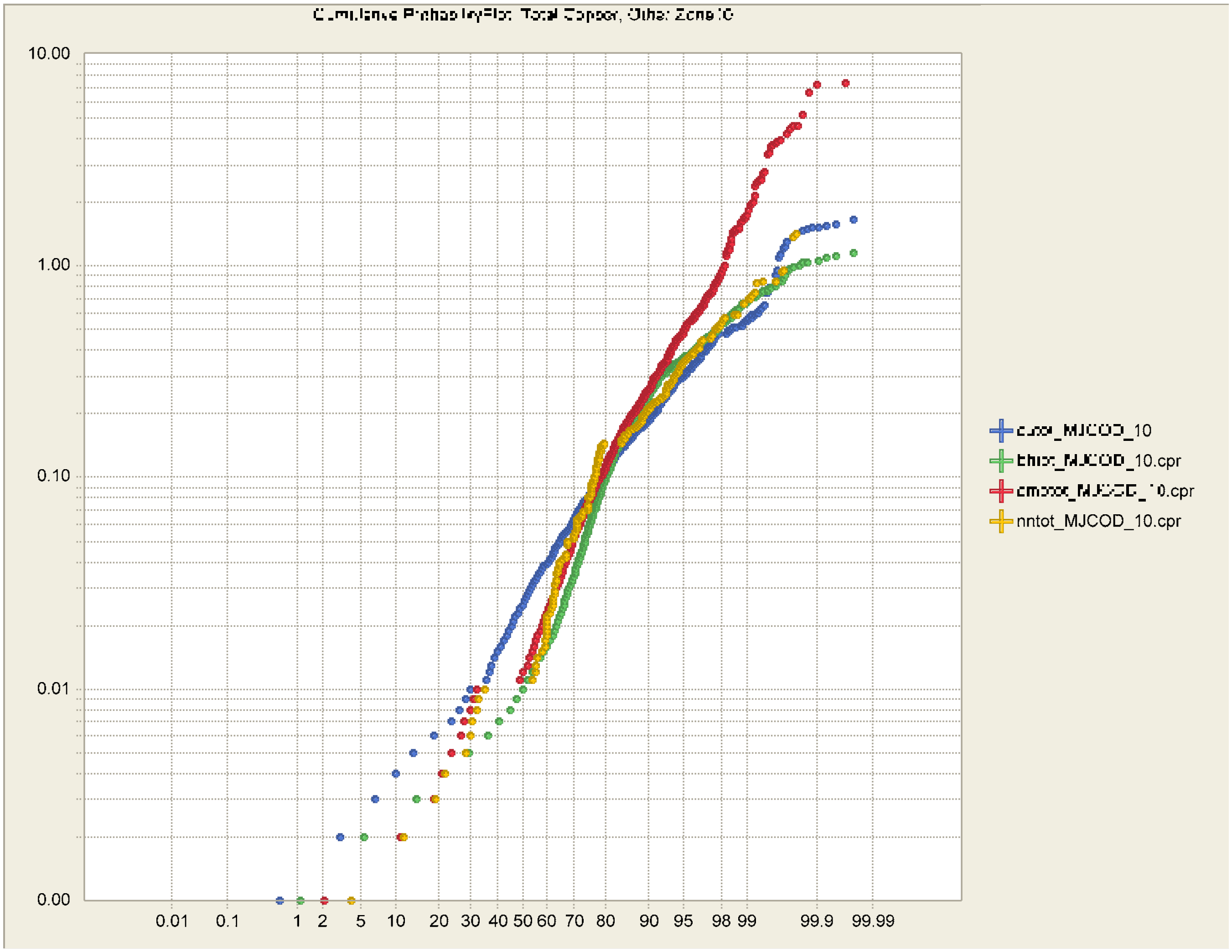
Supergene Zone2
10.00
1.00
0.10
0.01
0.00
0.01
0.1
1
2
5
10
20
30
40
50
60
70
80
90
95
98
99
99.9
99.99
nntot_MJCOD_2.cpr

Appendix D



Leach Cap Zone3
10.00
1.00
0.10
0.01
0.00
0.01
0.1
1
2
5
10
20
30
40
50
60
70
80
90
95
98
99
99.9
99.99
nntot_MJCOD_3.cpr

Appendix D

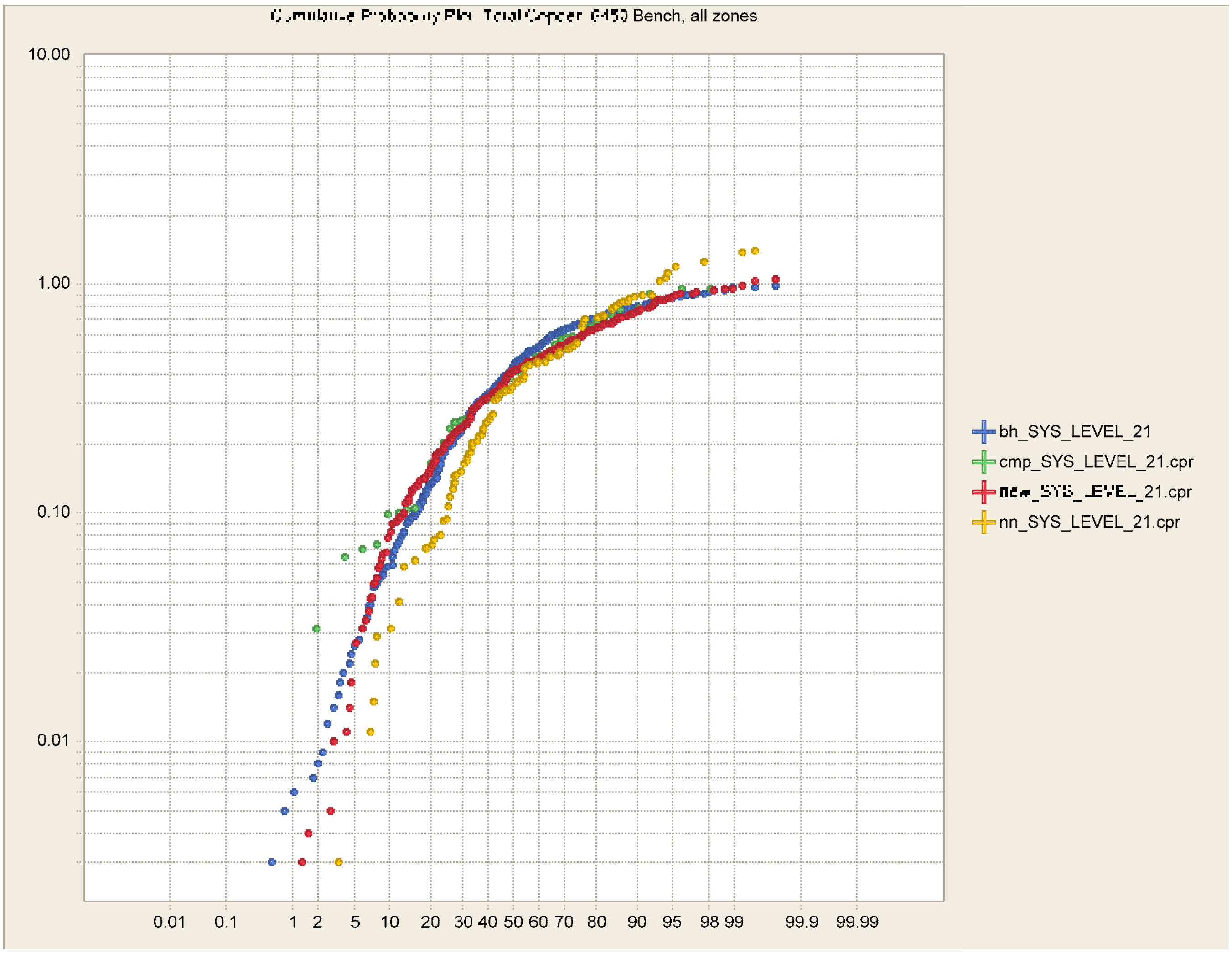


10.00
1.00
0.10
0.01
0.00
0.01
0.1
1
2
5
10
20
30
40
50
60
70
80
90
95
98
99
99.9
99.99
nntot_MJCOD_10.cpr

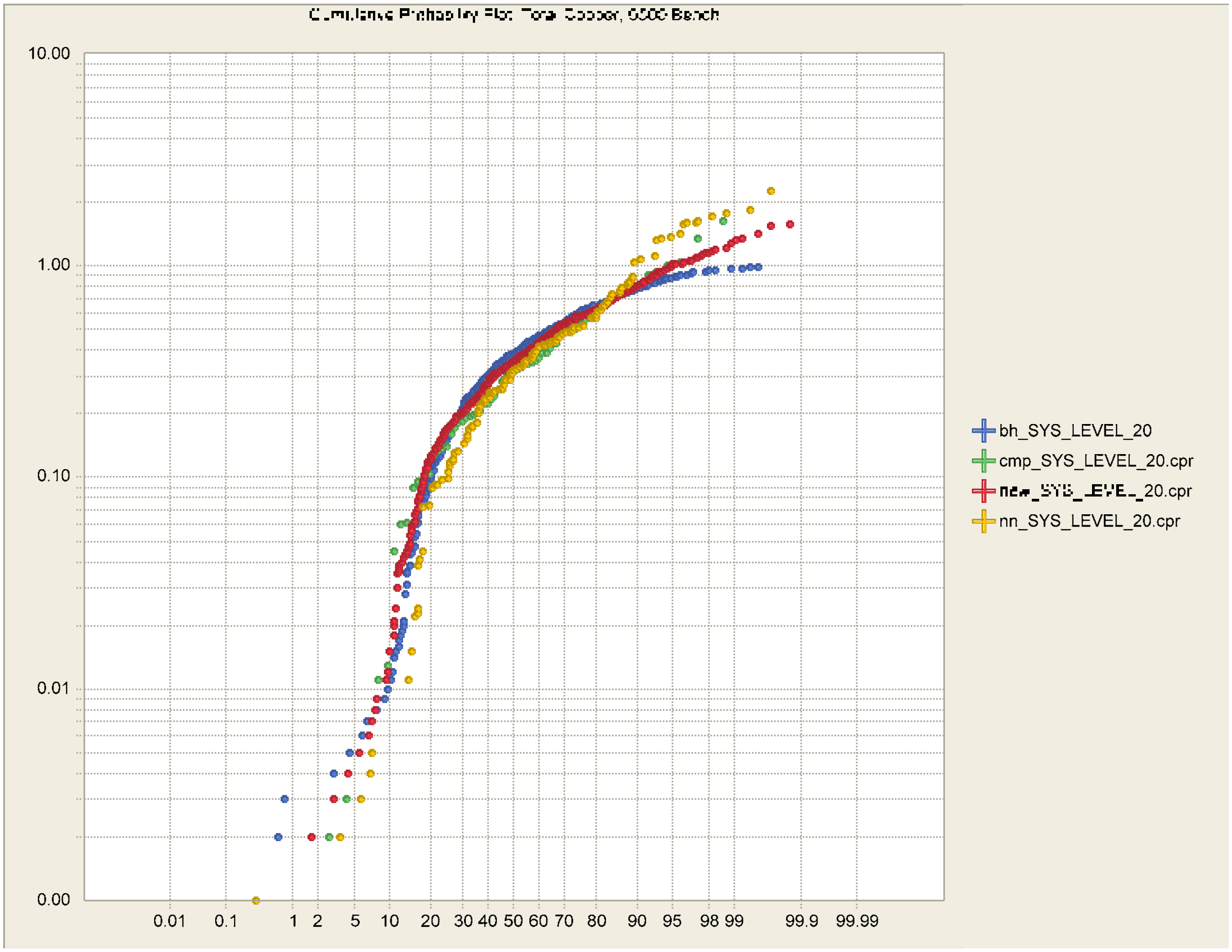
Cumulative Probability Plots
Veteran Deposit

Plots show the following grades by bench:

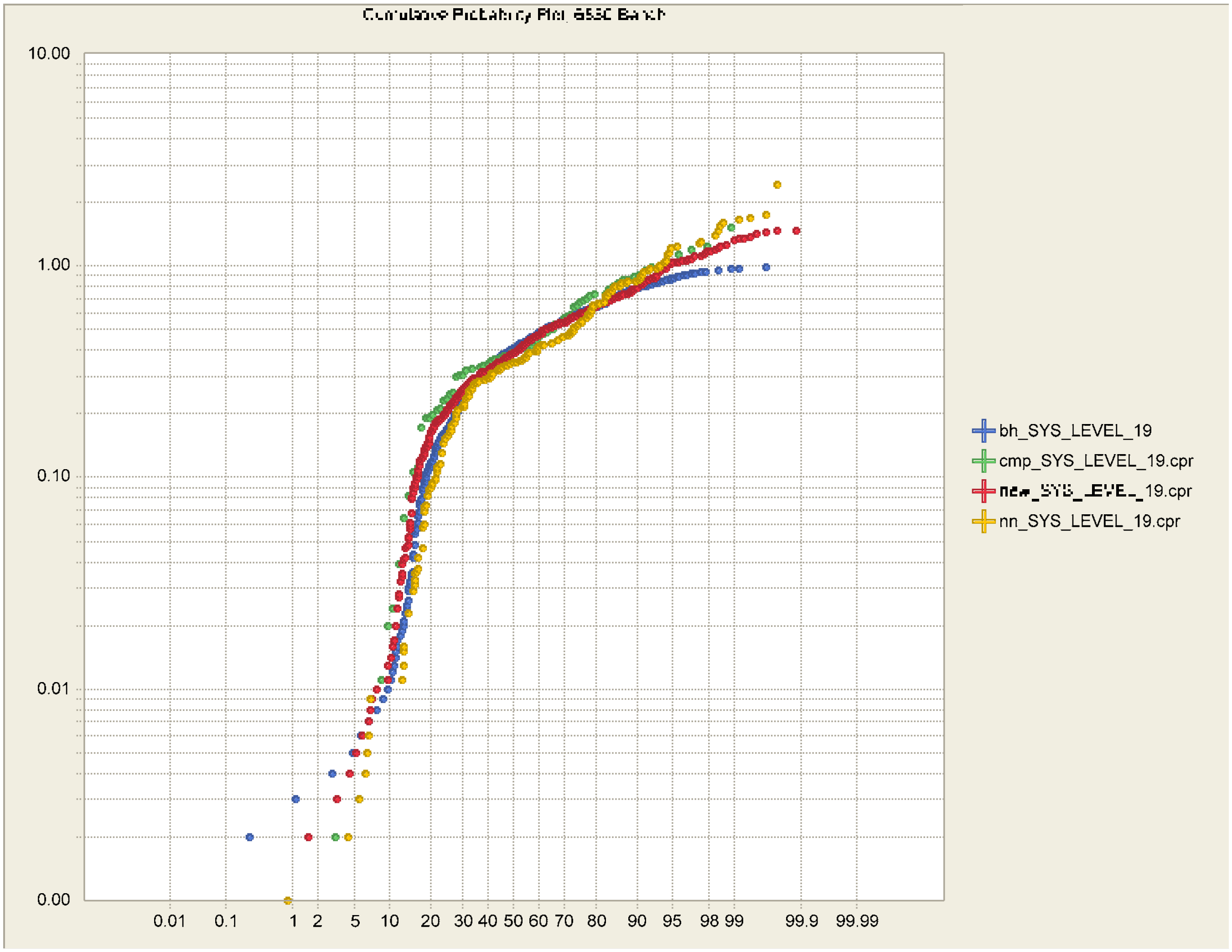
Item	**Label on chart**
Resource Model Total Copper	(new_SYS_LEVEL)
Blast Hole Total Copper	(bh_SYS_LEVEL)
Composites Total Copper	(cmp_SYS_LEVEL)
Polygonal (nearest neighbor) Total Copper	(nn_SYS_LEVEL)



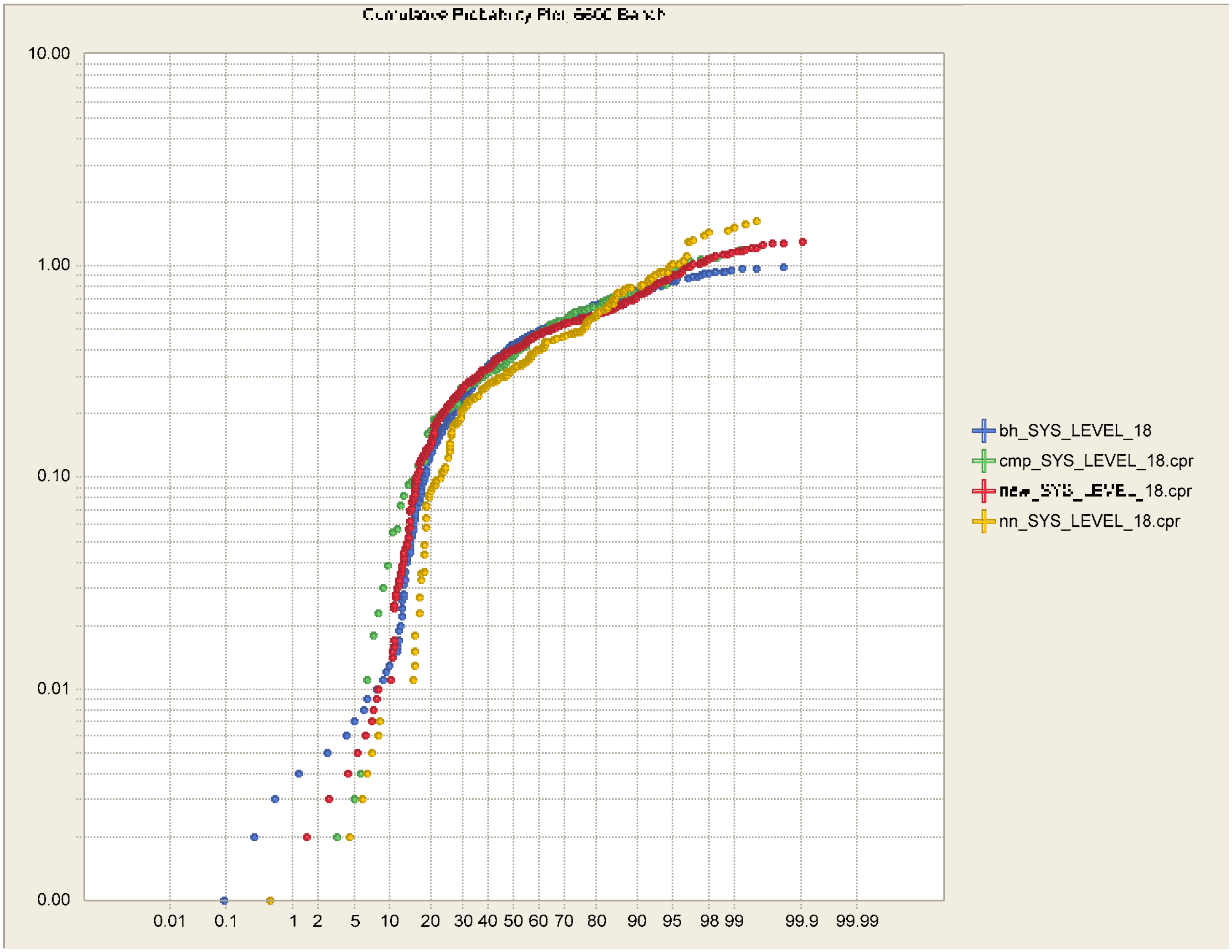
Appendix D


10.00
1.00
0.10
0.01
0.00
0.01
0.1
1
2
5
10
20
30
40
50
60
70
80
90
95
98
99
99.9
99.99
bh_SYS_LEVEL_20
cmp_SYS_LEVEL_20.cpr
new_SYS_LEVEL_20.cpr
nn_SYS_LEVEL_20.cpr

Appendix D

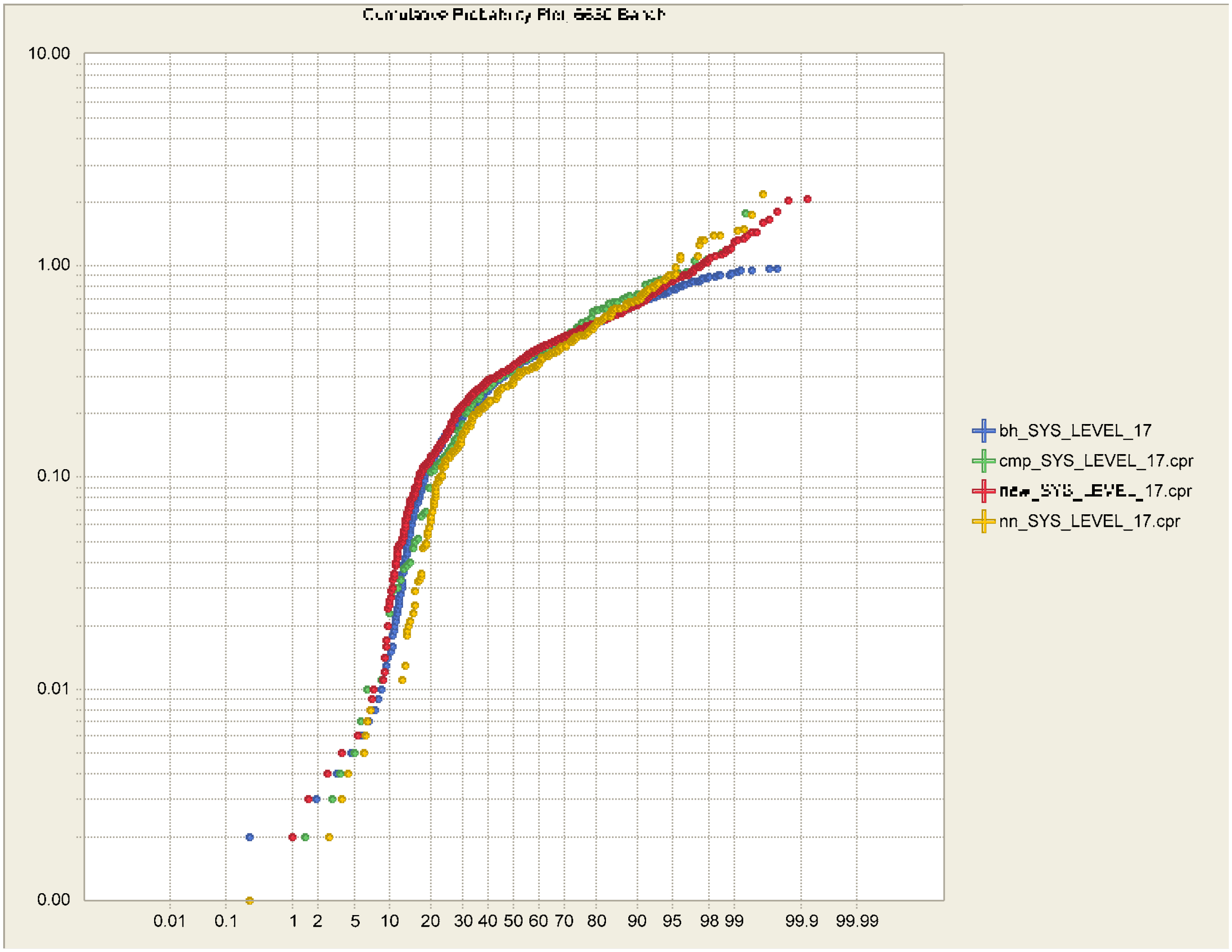

Cumulative Probability Plot, 6530 Bench
10.00
1.00
0.10
0.01
0.00
0.01
0.1
1
2
5
10
20
30
40
50
60
70
80
90
95
98
99
99.9
99.99
bh_SYS_LEVEL_19
cmp_SYS_LEVEL_19.cpr
new_SYS_LEVEL_19.cpr
nn_SYS_LEVEL_19.cpr

Appendix D



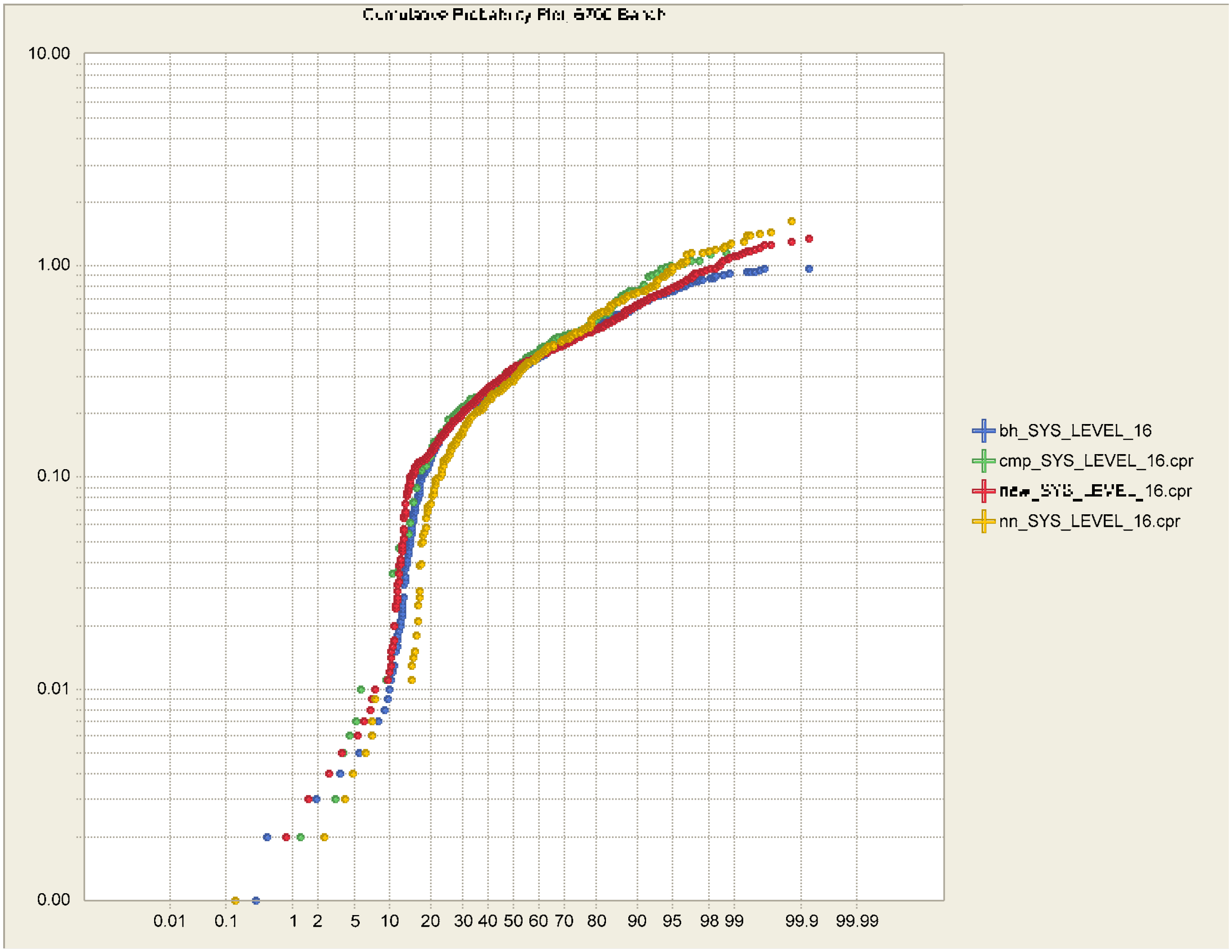
10.00
1.00
0.10
0.01
0.00
0.01
0.1
1
2
5
10
20
30
40
50
60
70
80
90
95
98
99
99.9
99.99
bh_SYS_LEVEL_18
cmp_SYS_LEVEL_18.cpr
nn_SYS_LEVEL_18.cpr

Appendix D



Cumulative Probability Plot
10.00
1.00
0.10
0.01
0.00
0.01
0.1
1
2
5
10
20
30
40
50
60
70
80
90
95
98
99
99.9
99.99
bh_SYS_LEVEL_17
cmp_SYS_LEVEL_17.cpr
nn_SYS_LEVEL_17.cpr

Appendix D



10.00
1.00
0.10
0.01
0.00
0.01
0.1
1
2
5
10
20
30
40
50
60
70
80
90
95
98
99
99.9
99.99
bh_SYS_LEVEL_16
cmp_SYS_LEVEL_16.cpr
nn_SYS_LEVEL_16.cpr

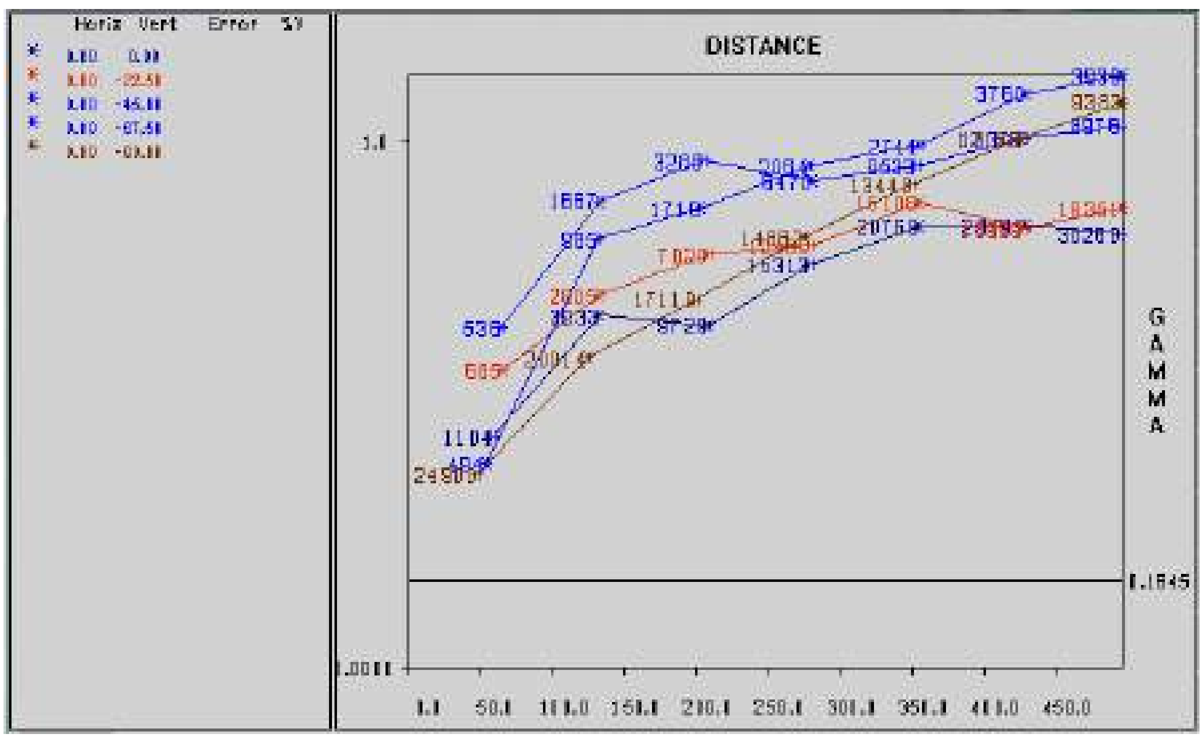
Appendix E: Ruth Estimation Parameters and Variograms – Total Copper

RUTH TOTAL COPPER ESTIMATION PARAMETERS

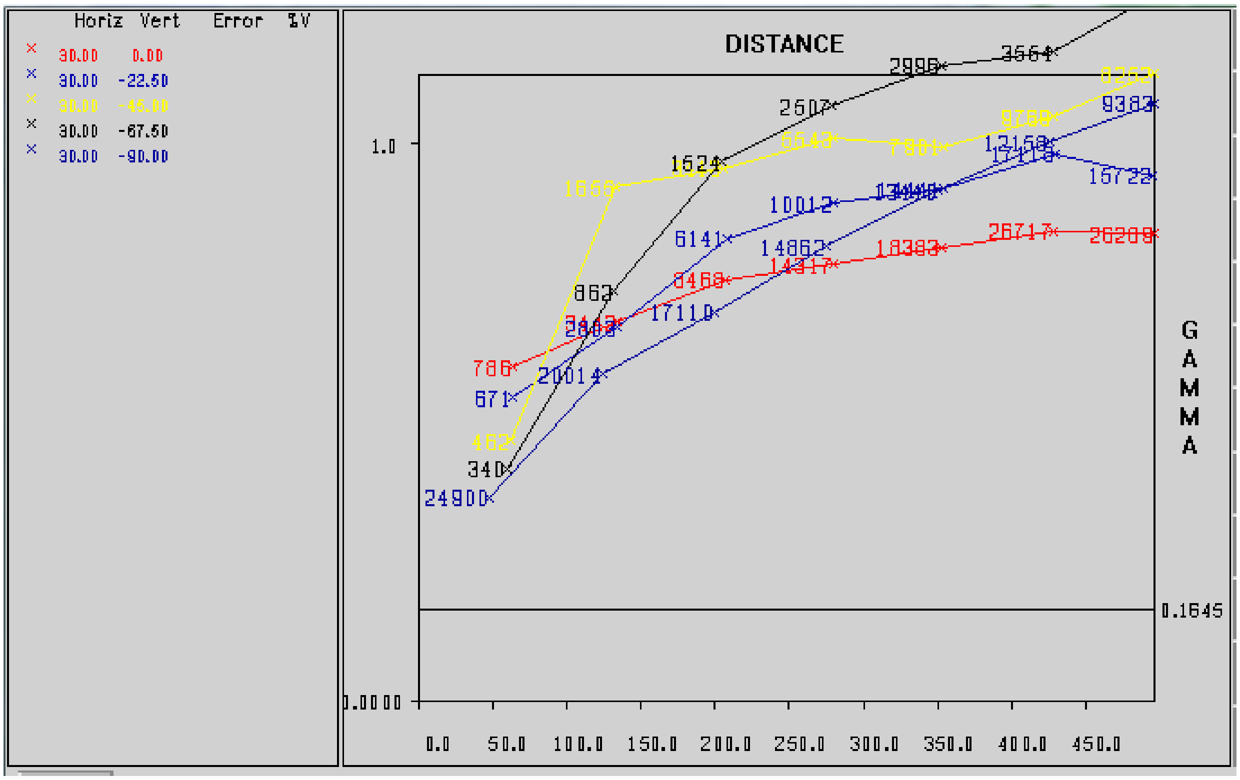
	Zone 1 (Hypogene)	Zone 2 (CEB)	Zone 3 (Leach Cap)	Zone 4 (Other)
Algorithm	ID2	ID2	ID2	ID2
Minimum # composites	1	1	1	1
Maximum # composites	12	12	12	12
Maximum # composites per hole	4	4	4	4
Search distance (ft) X,Y,Z	300,300,300	400,300,250	300,250,200	400,300,250
Search directions Meds Az,plunge, dip	0,0,0,	100,0,0	100,0,0	80,0,0
Geologic matching zone	1	2	3	4
Cap Grade (on assays)	5.0	12.0	9.0	4.0
Composites omitted	dump	dump	dump	dump

Variography

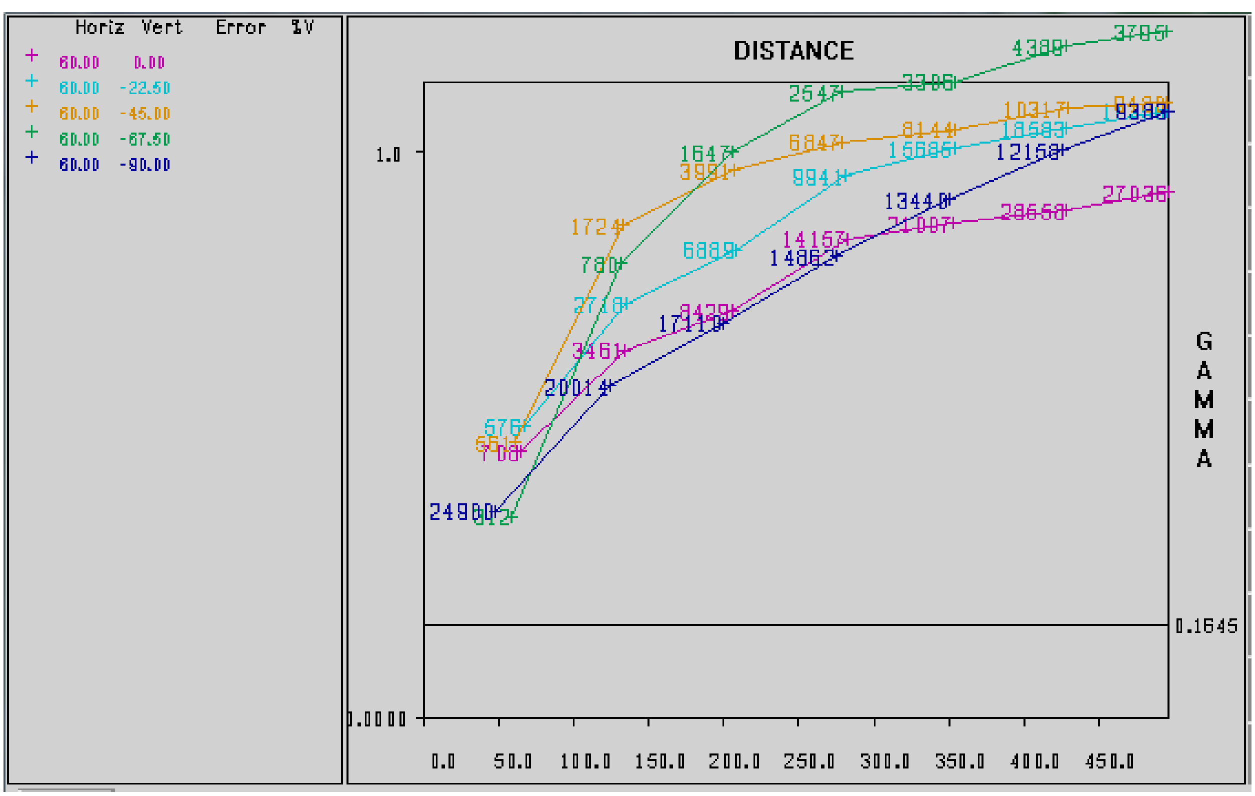
Total Copper, Zone1, Azimuth 0, Dip 0,-22,-45,-67,-90, Correlograms



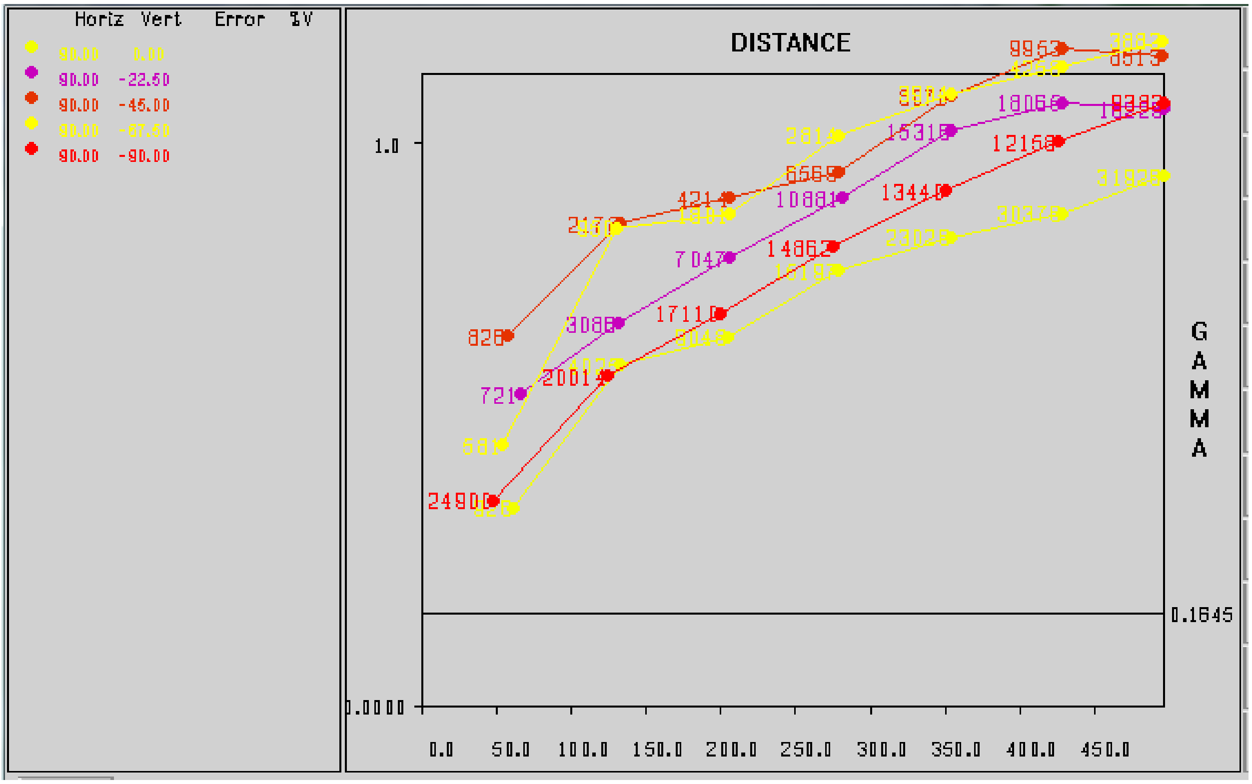
Total Copper, Zone1, Azimuth 30, Dip 0,-22,-45,-67,-90, Correlograms



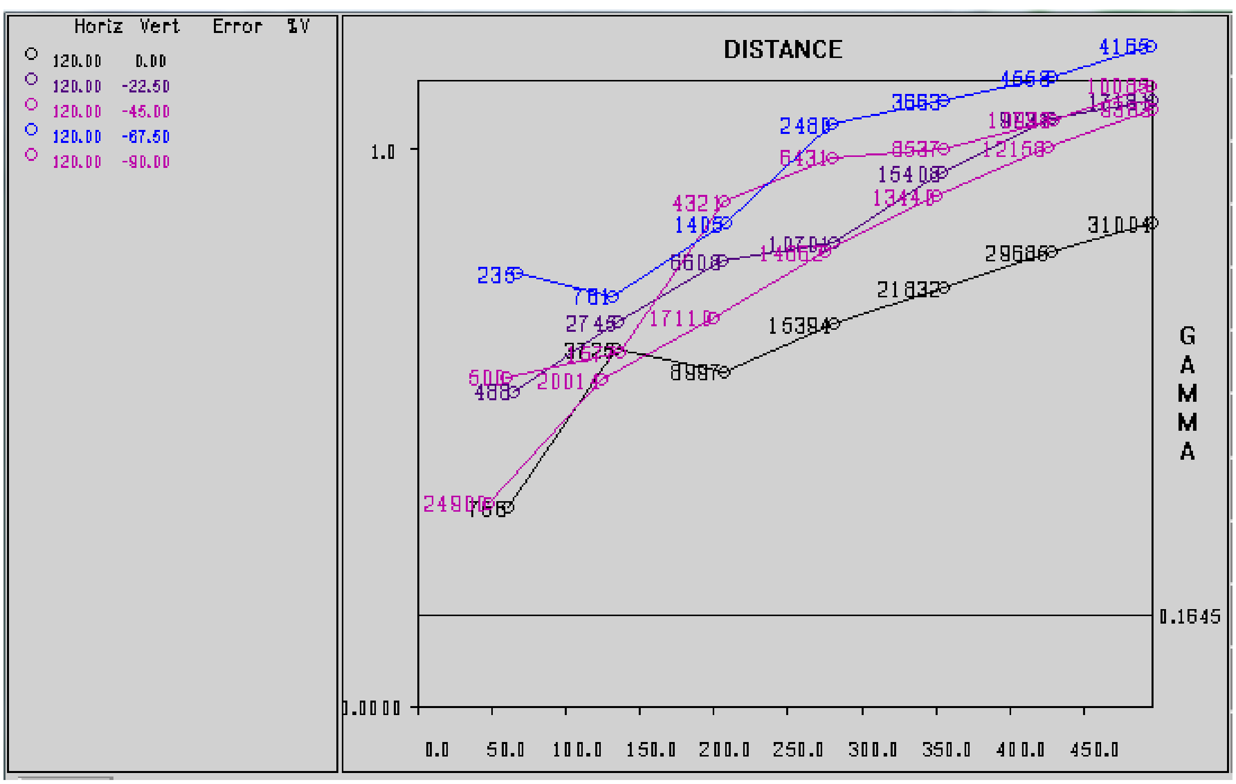
Total Copper, Zone1, Azimuth 60, Dip 0,-22,-45,-67,-90, Correlograms



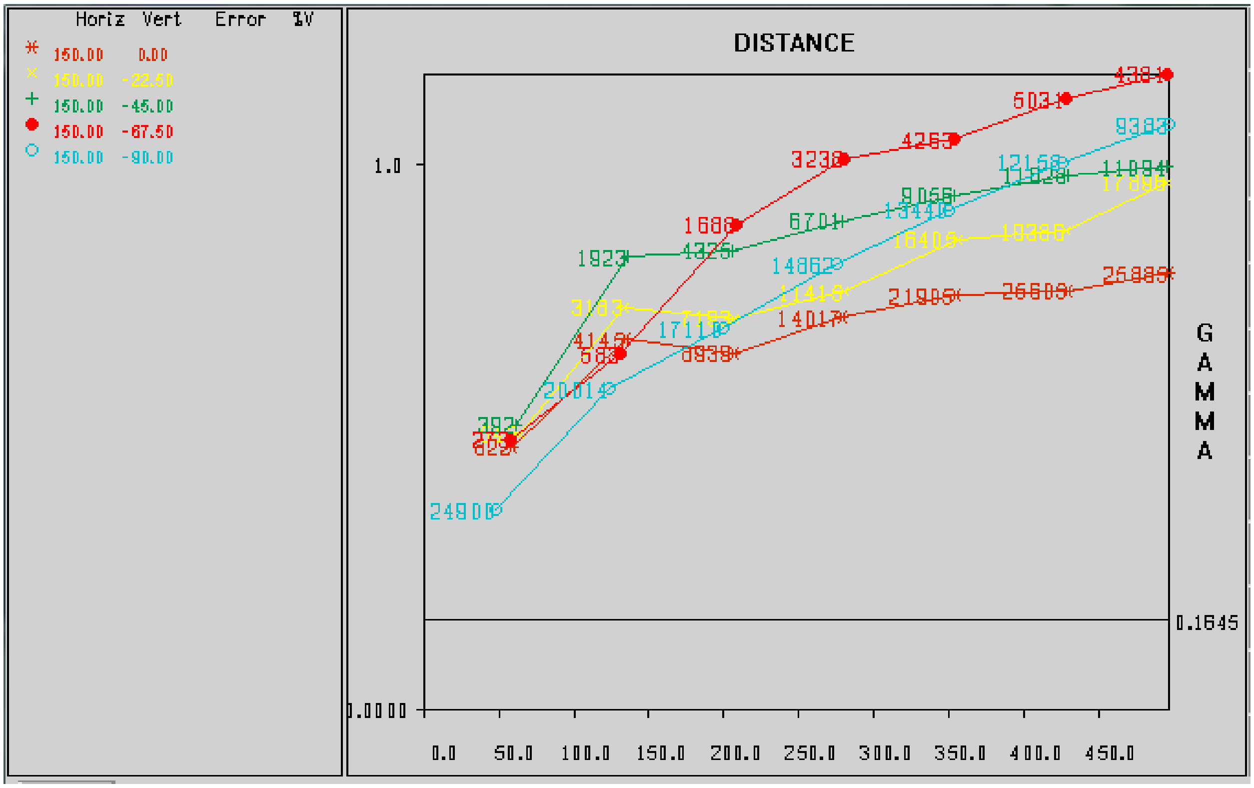
Total Copper, Zone1, Azimuth 90, Dip 0,-22,-45,-67,-90, Correlograms



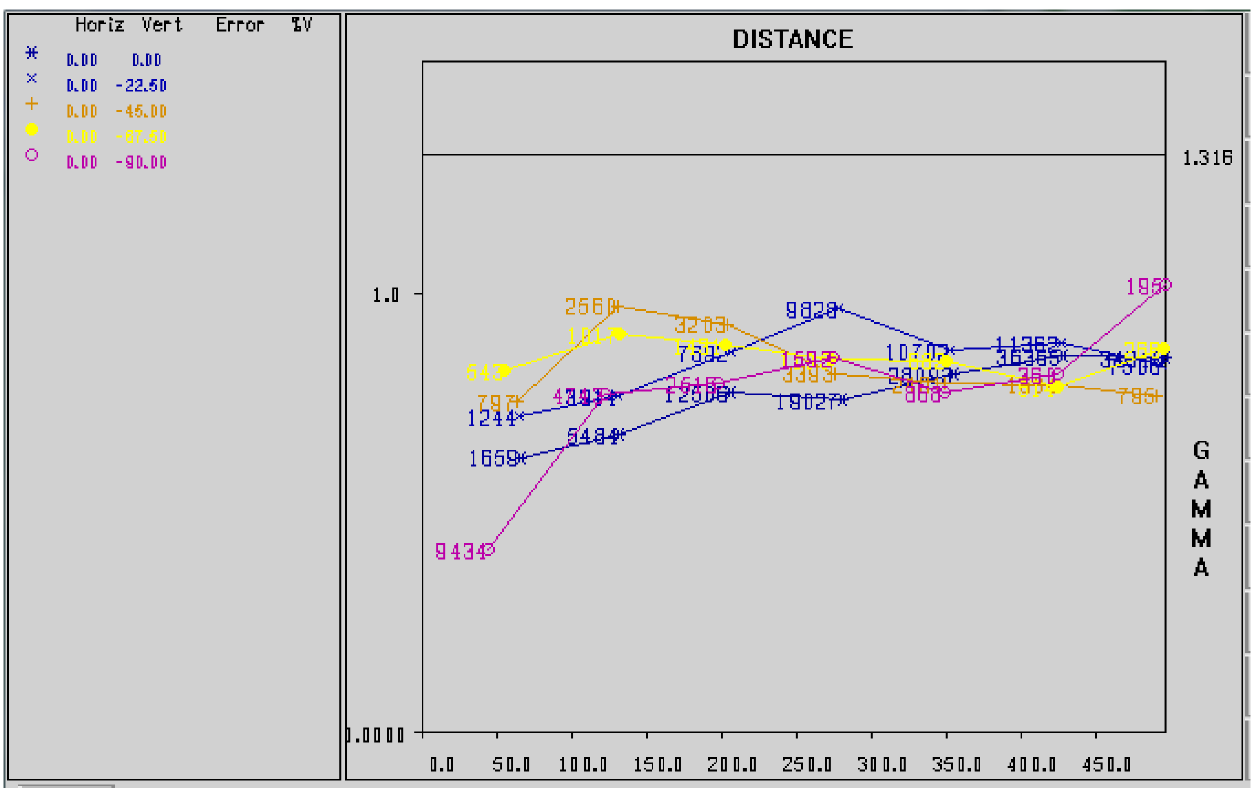
Total Copper, Zone1, Azimuth 120, Dip 0,-22,-45,-67,-90, Correlograms



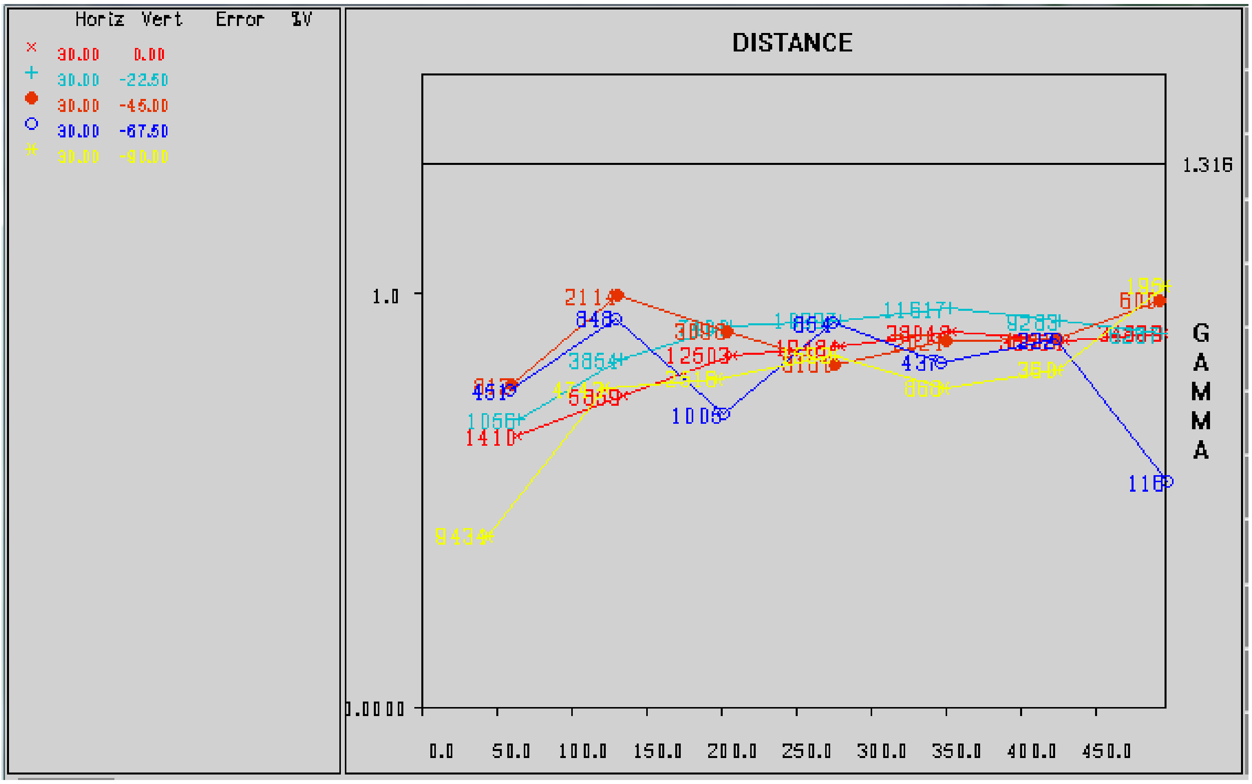
Total Copper, Zone1, Azimuth 120, Dip 0,-22,-45,-67,-90, Correlograms



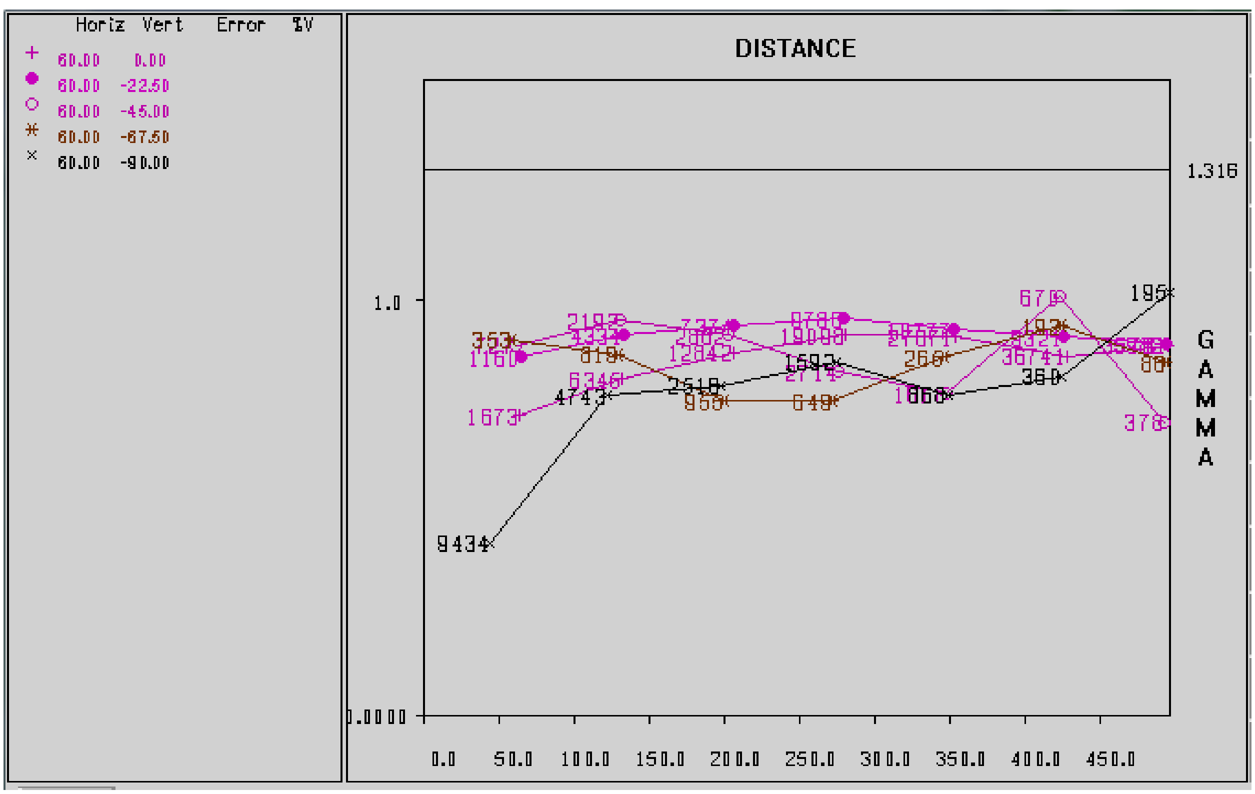
Total Copper, Zone2, Azimuth 0, Dip 0,-22,-45,-67,-90, Correlograms



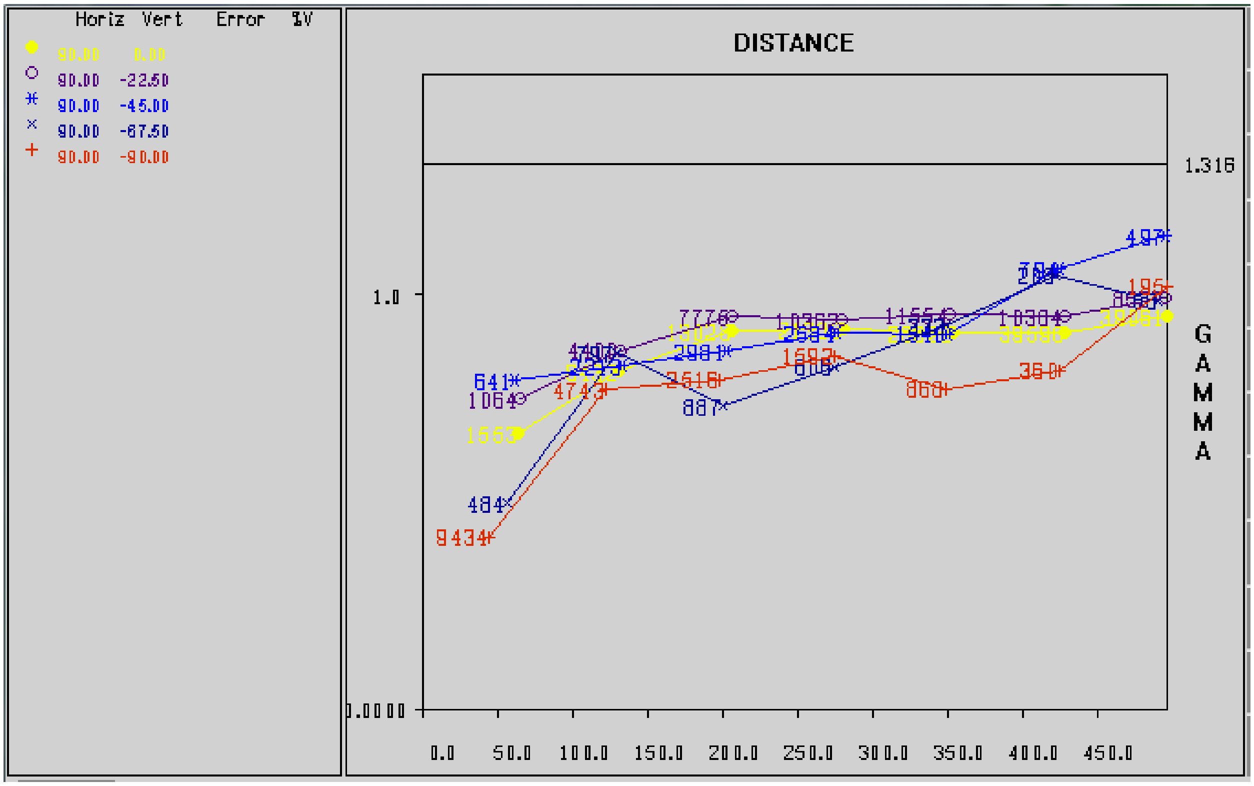
Total Copper, Zone2, Azimuth 30, Dip 0,-22,-45,-67,-90, Correlograms



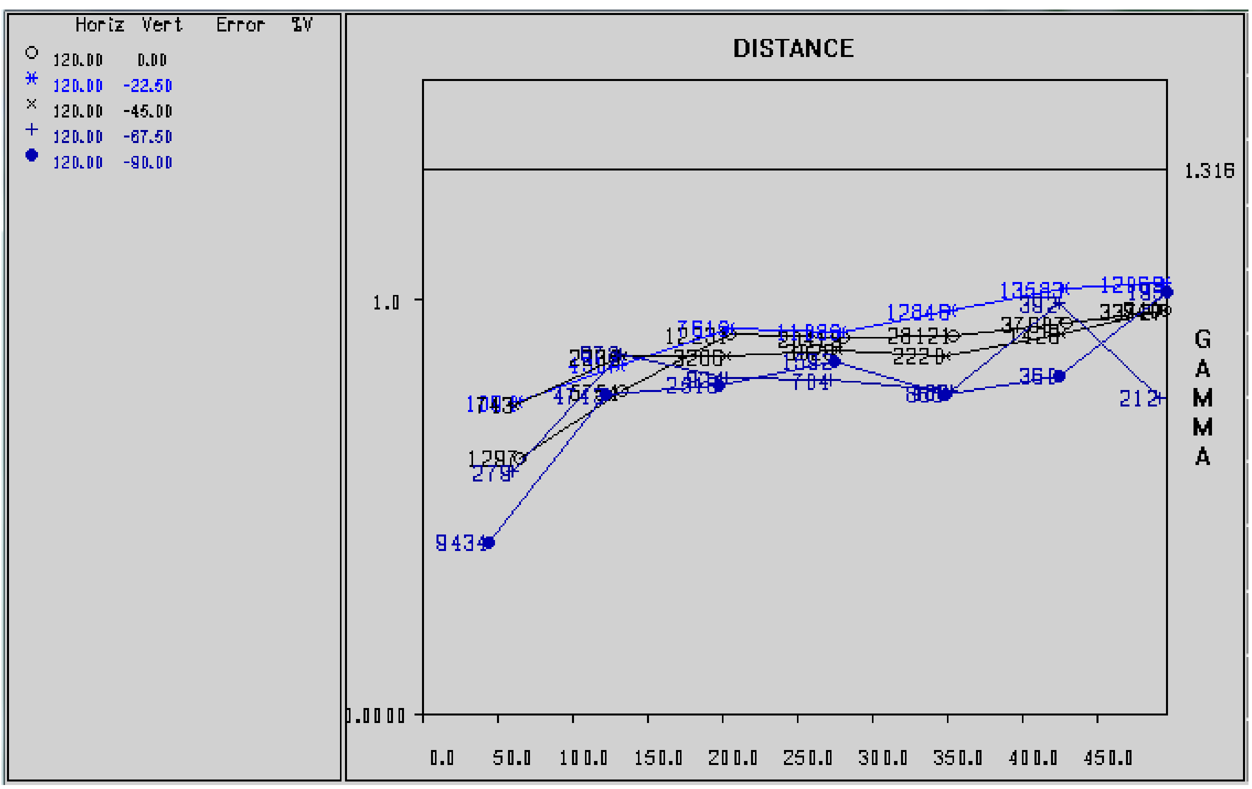
Total Copper, Zone2, Azimuth 60, Dip 0,-22,-45,-67,-90, Correlograms



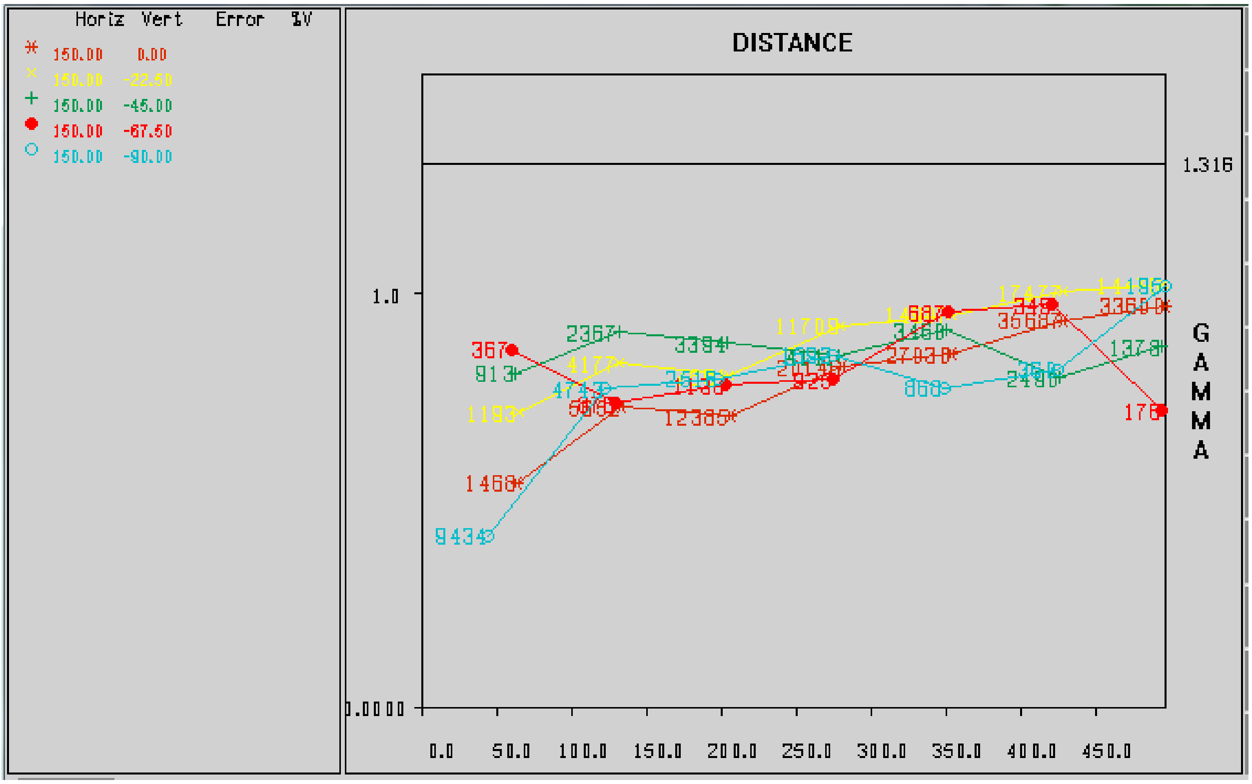
Total Copper, Zone2, Azimuth 90, Dip 0,-22,-45,-67,-90, Correlograms



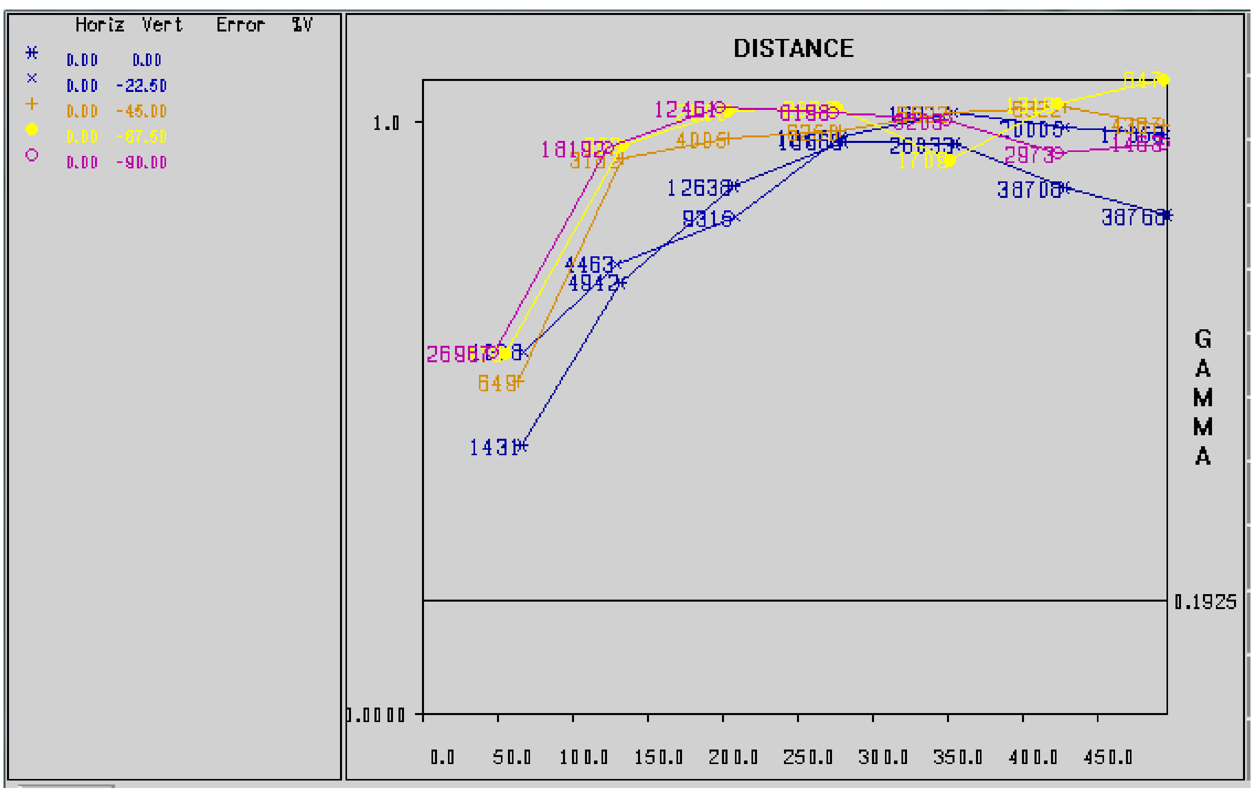
Total Copper, Zone2, Azimuth 120, Dip 0,-22,-45,-67,-90, Correlograms



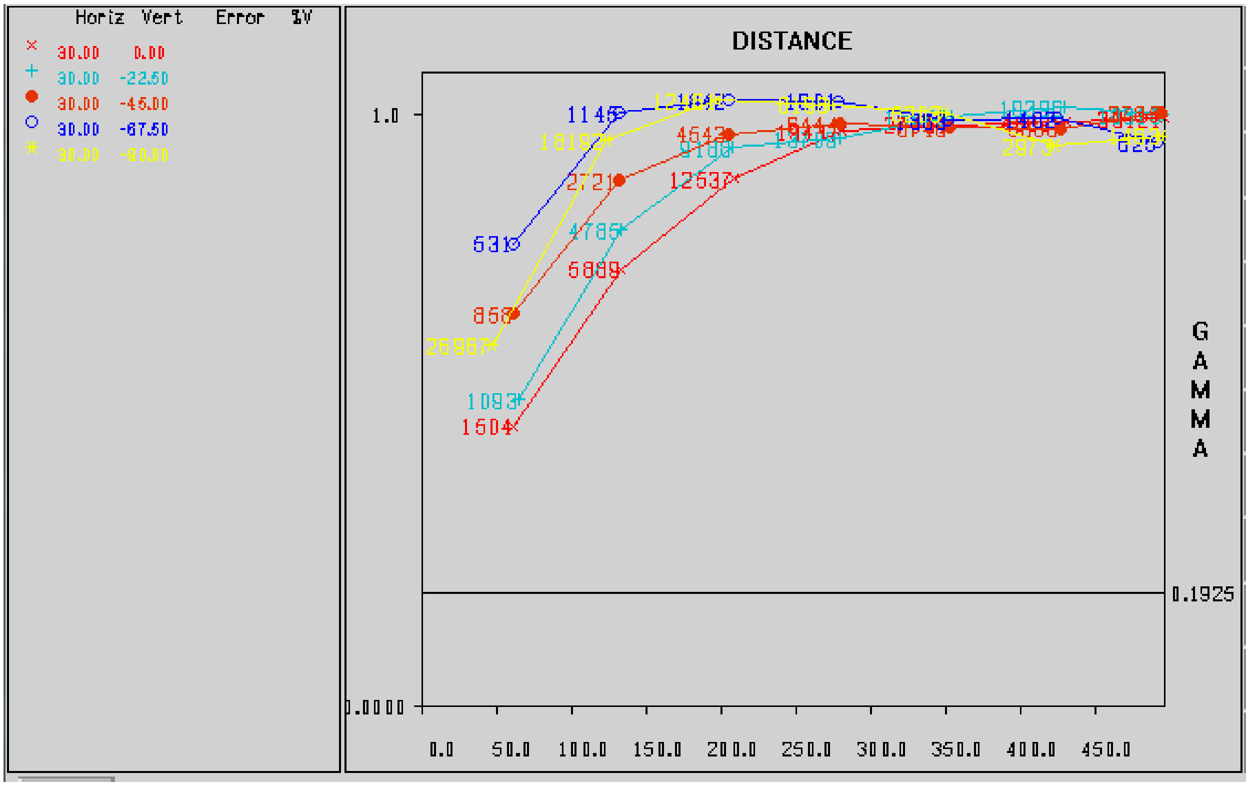
Total Copper, Zone2, Azimuth 150, Dip 0,-22,-45,-67,-90, Correlograms



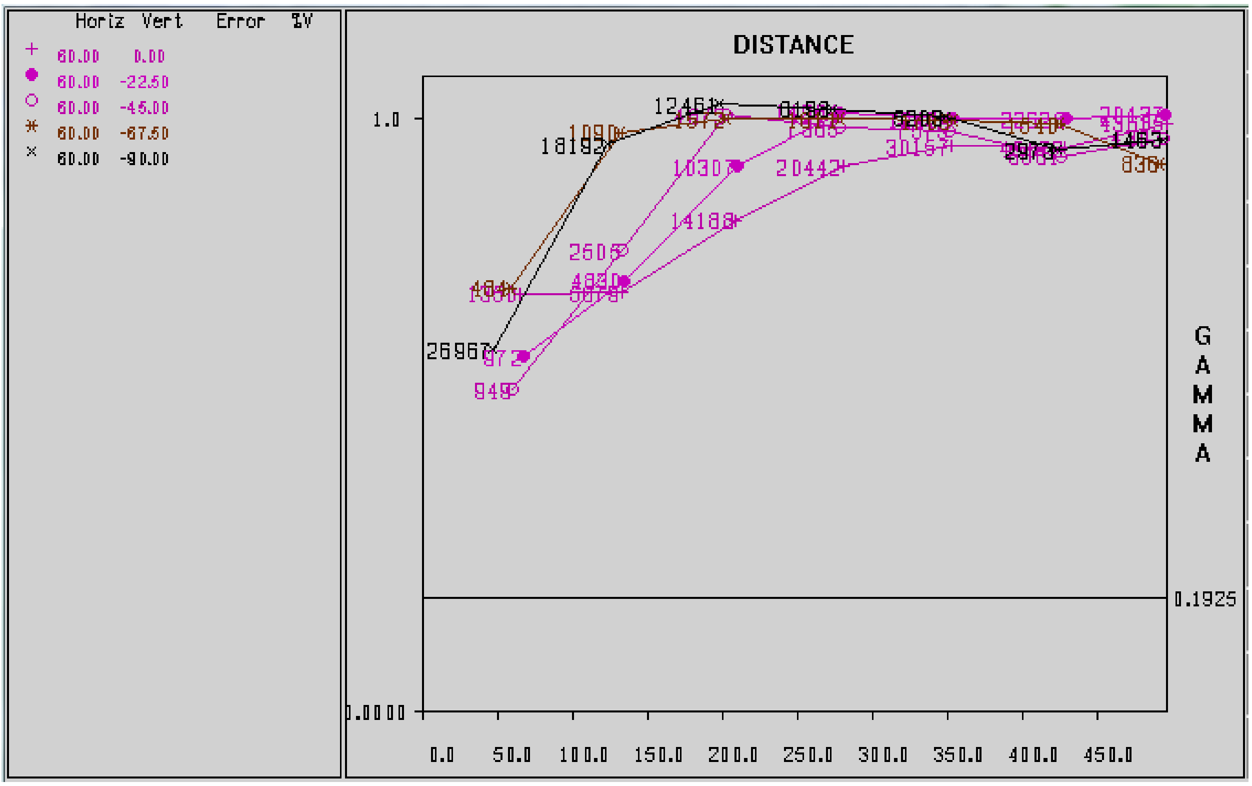
Total Copper, Zone3, Azimuth 0, Dip 0,-22,-45,-67,-90, Correlograms



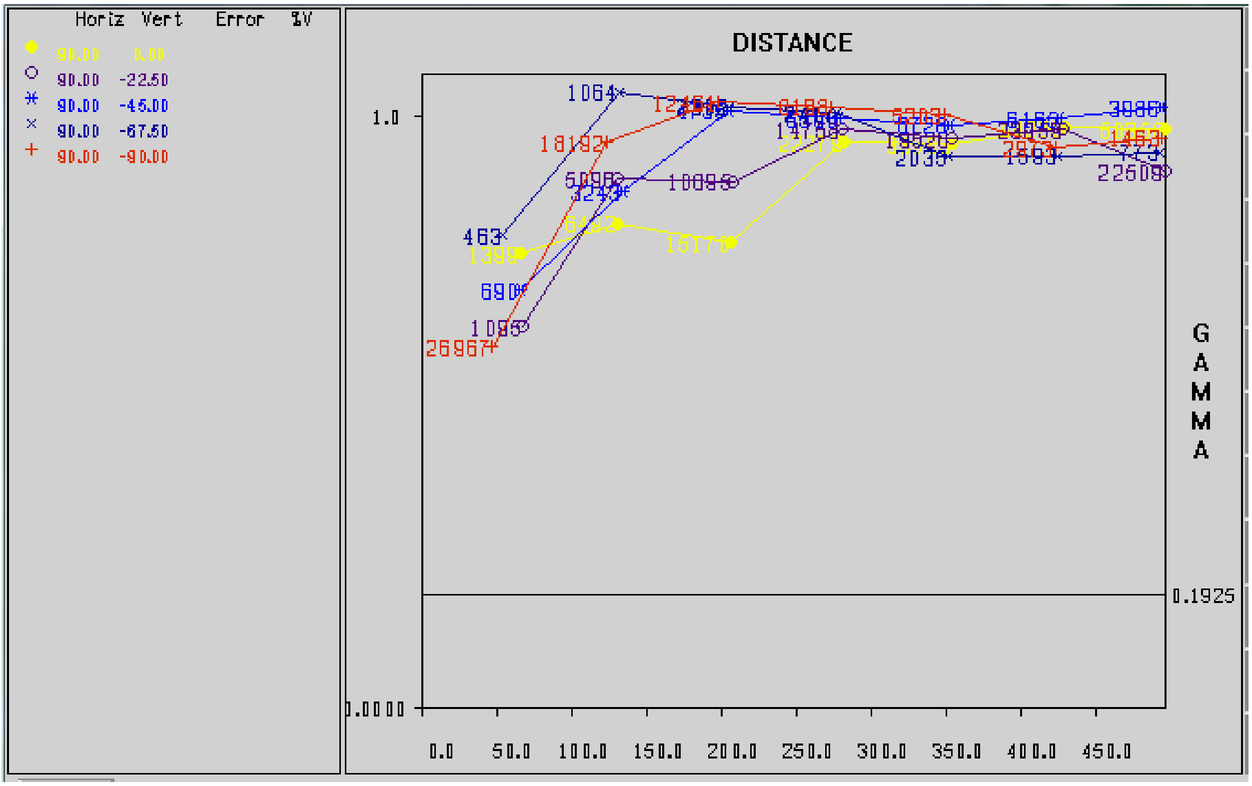
Total Copper, Zone3, Azimuth 30, Dip 0,-22,-45,-67,-90, Correlograms



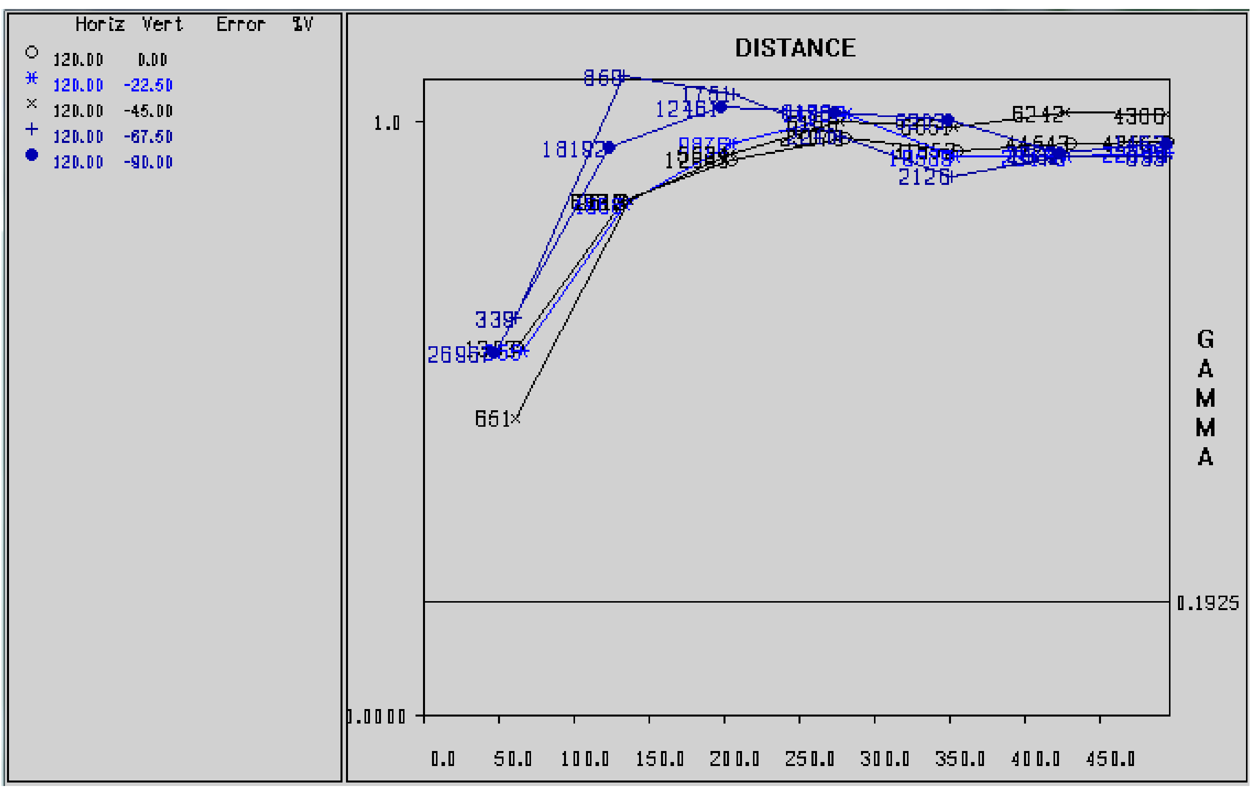
Total Copper, Zone3, Azimuth 60, Dip 0,-22,-45,-67,-90, Correlograms



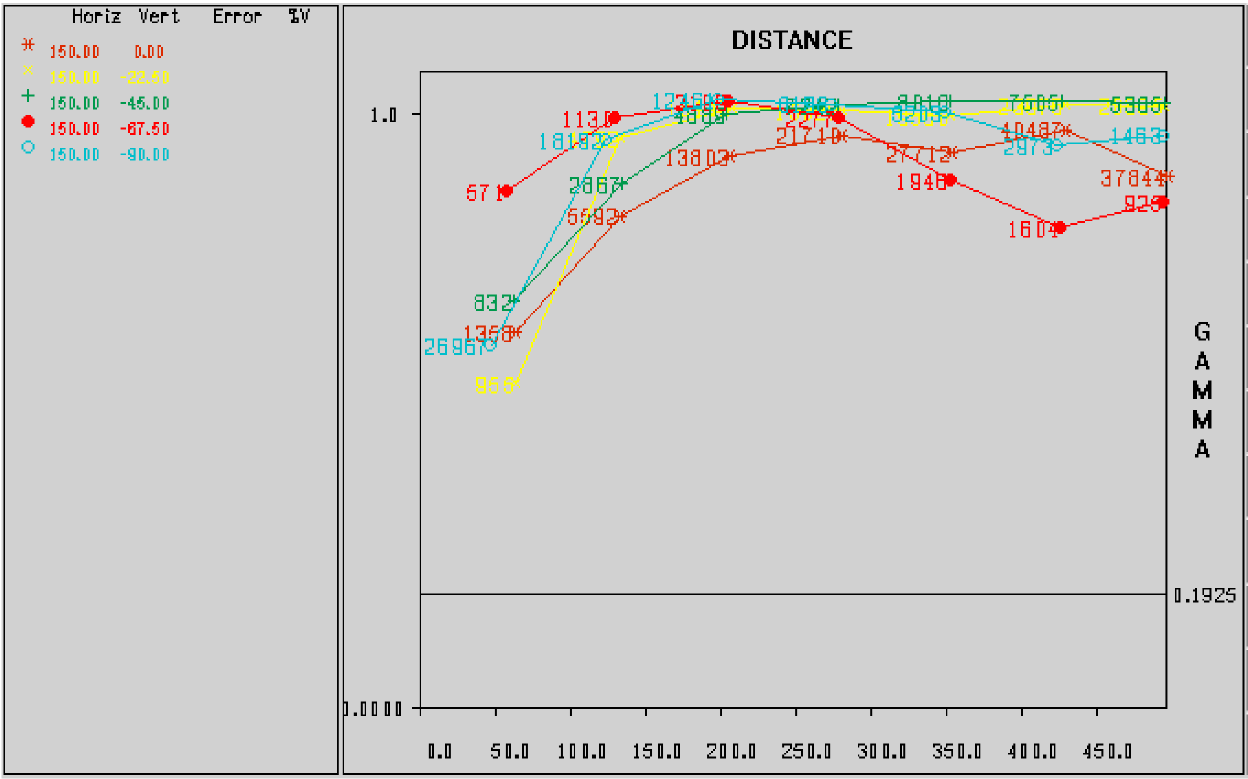
Total Copper, Zone3, Azimuth 90, Dip 0,-22,-45,-67,-90, Correlograms



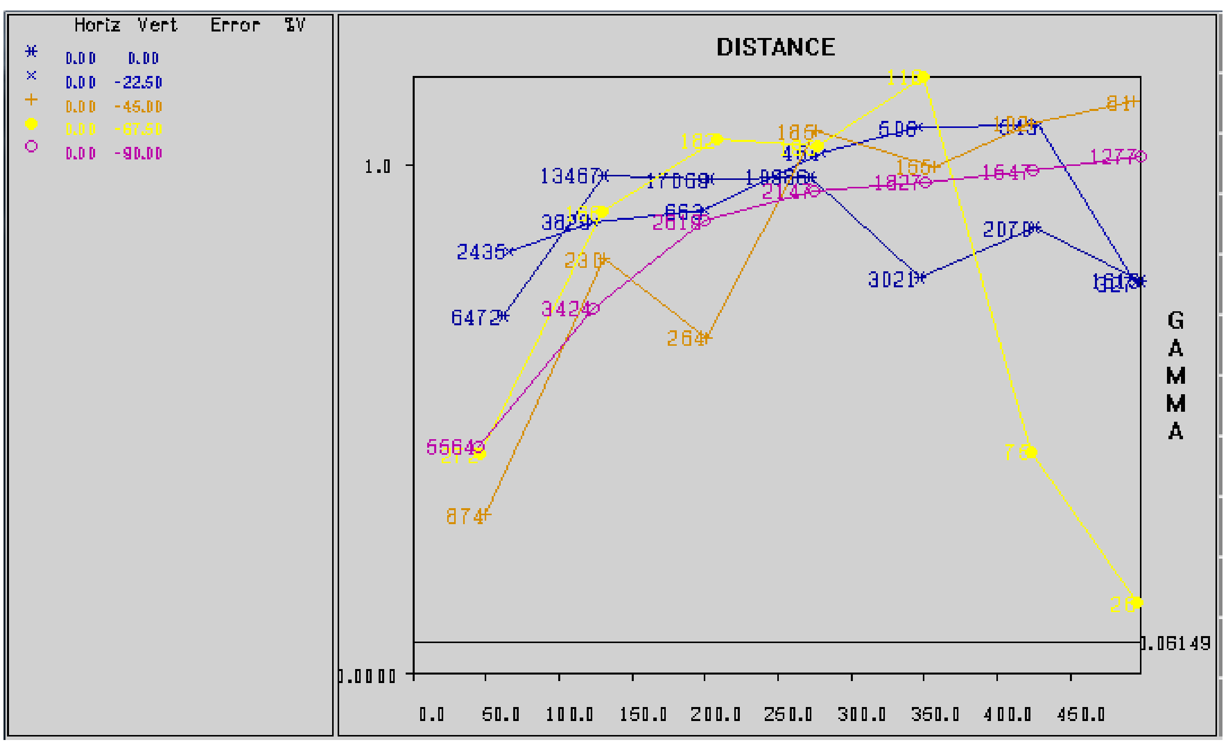
Total Copper, Zone3, Azimuth 120, Dip 0,-22,-45,-67,-90, Correlograms



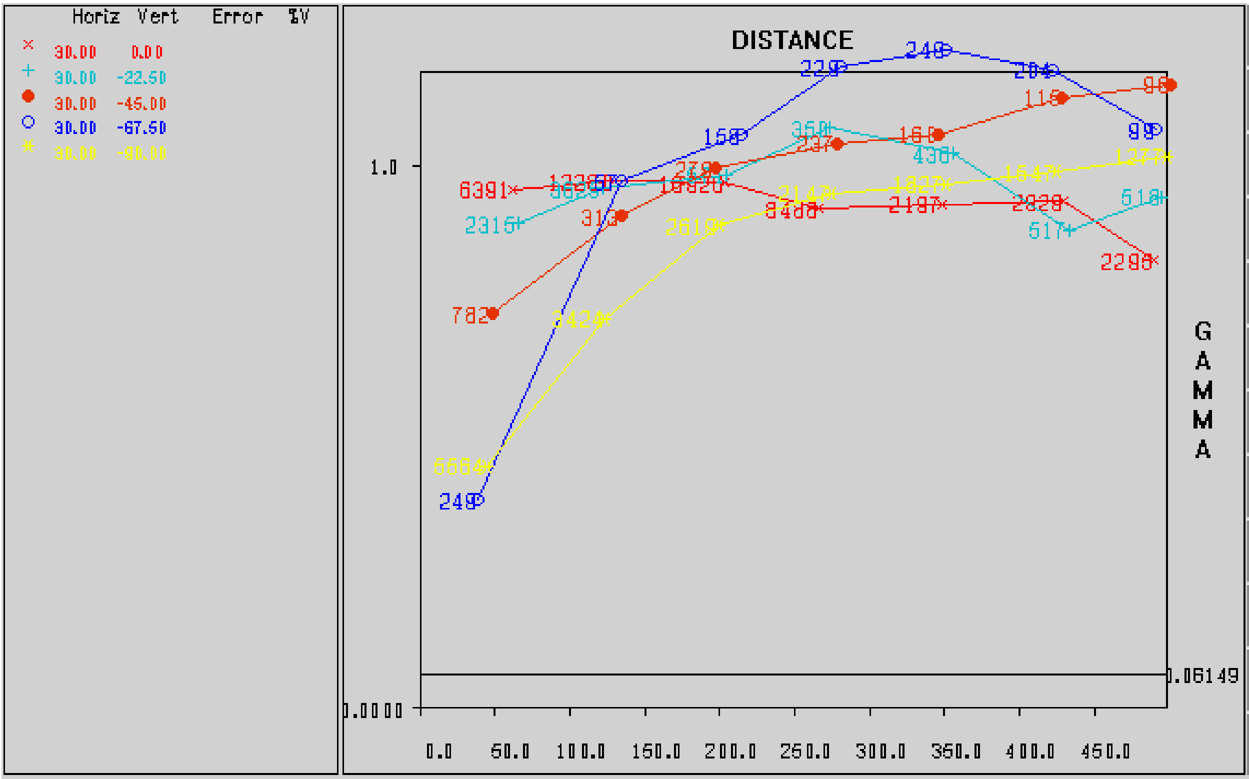
Total Copper, Zone3, Azimuth 150, Dip 0,-22,-45,-67,-90, Correlograms



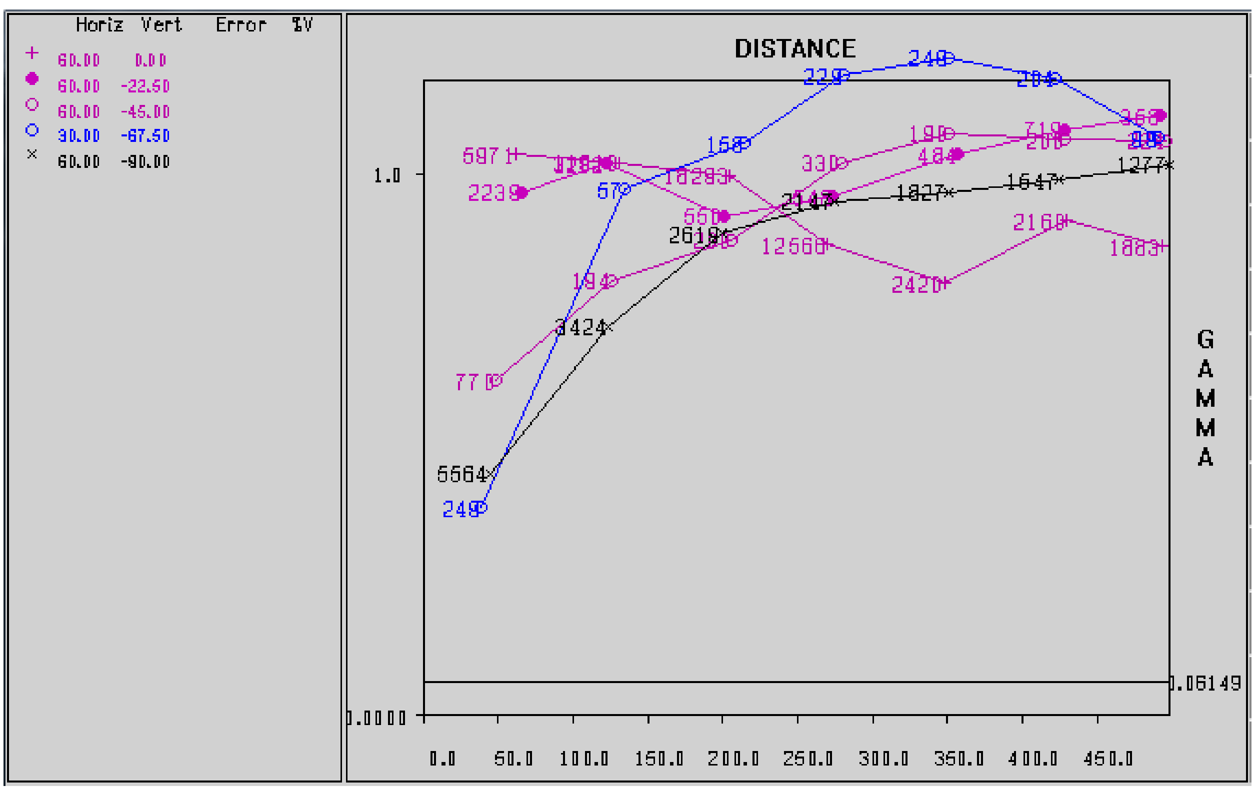
Total Copper, Zone4, Azimuth 0, Dip 0,-22,-45,-67,-90, Correlograms



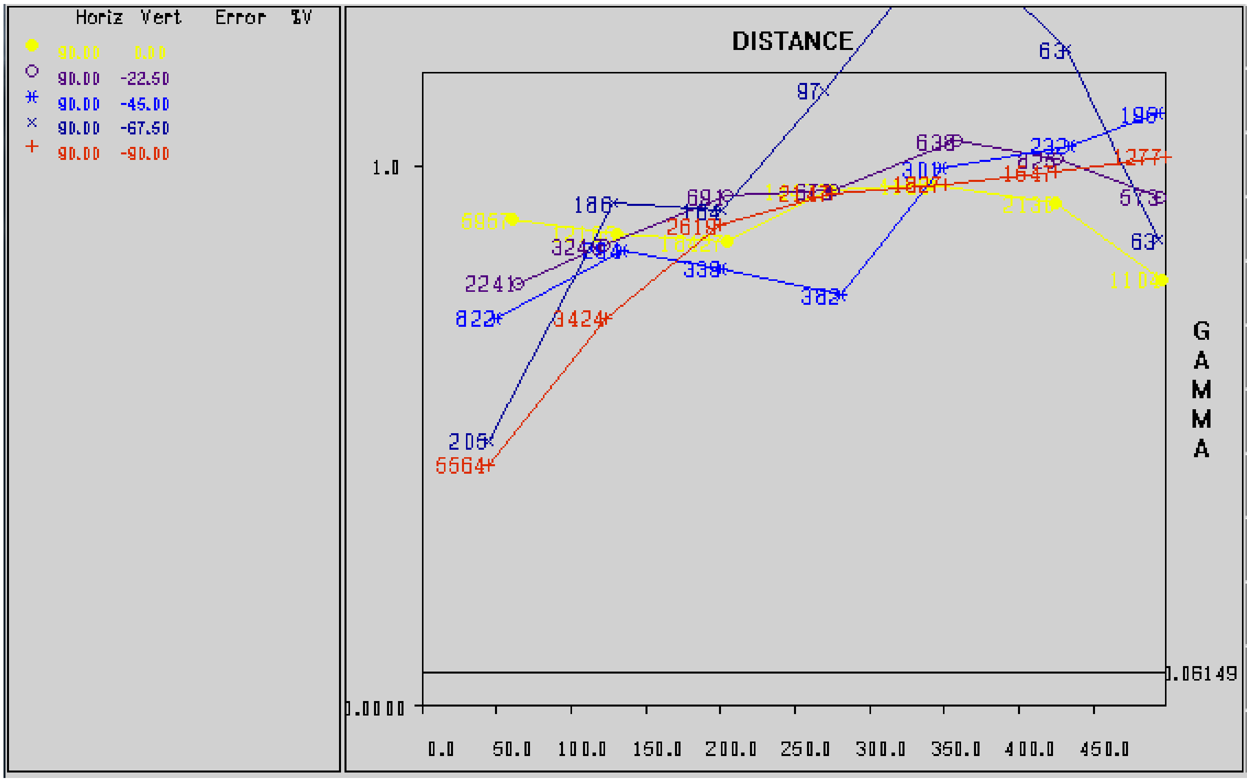
Total Copper, Zone4, Azimuth 30, Dip 0,-22,-45,-67,-90, Correlograms



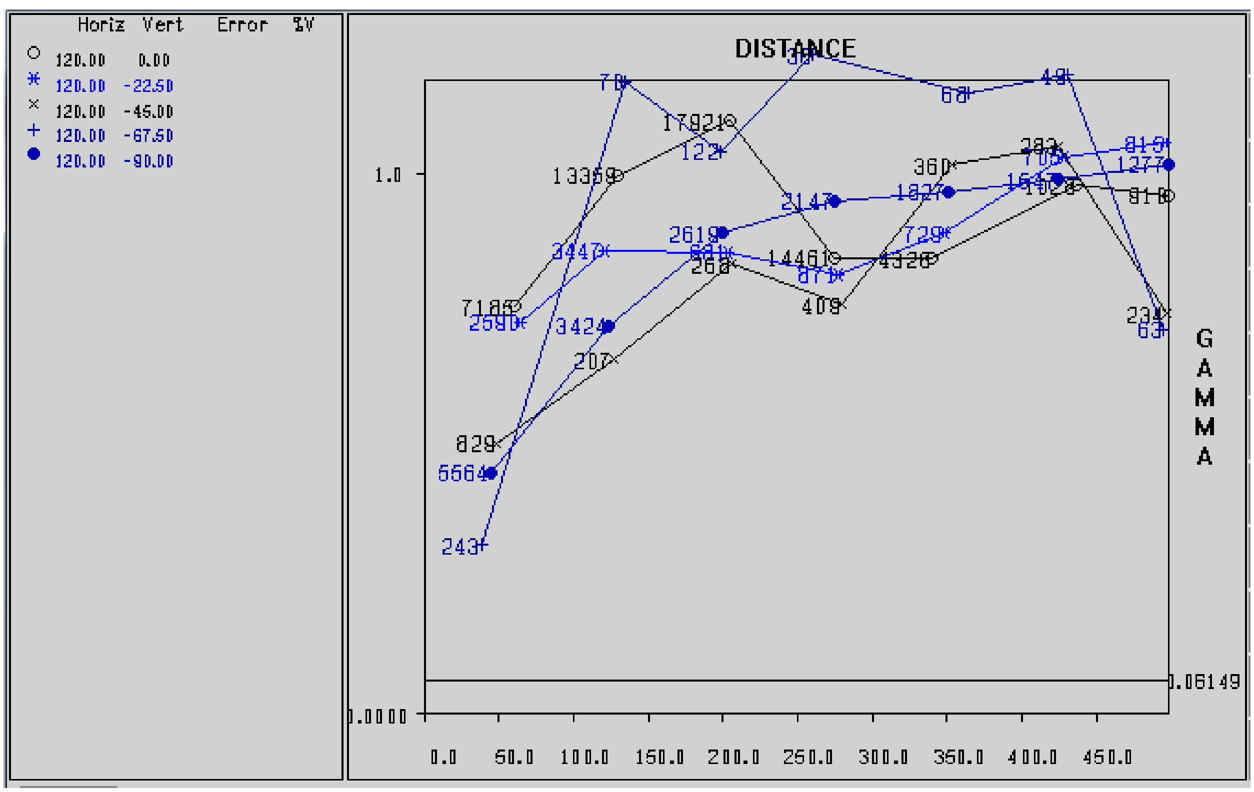
Total Copper, Zone4, Azimuth 60, Dip 0,-22,-45,-67,-90, Correlograms



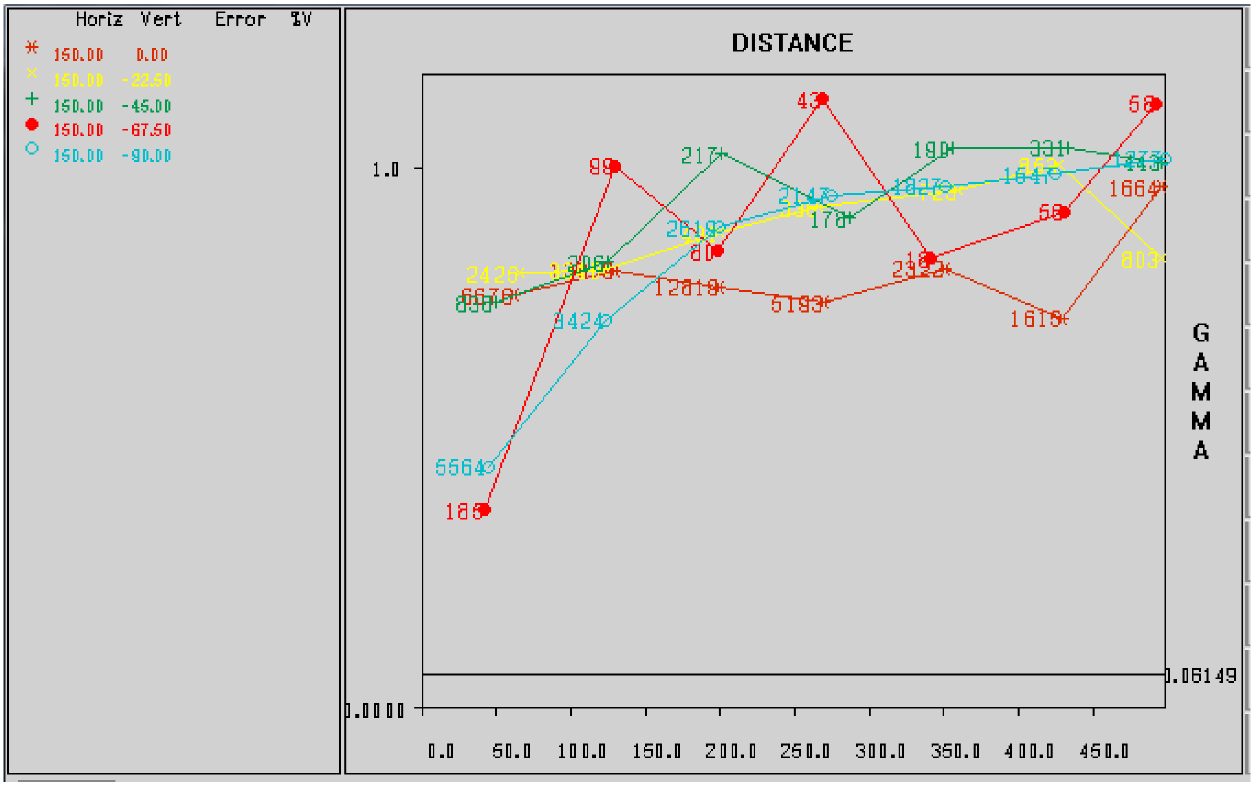
Total Copper, Zone4, Azimuth 90, Dip 0,-22,-45,-67,-90, Correlograms



Total Copper, Zone4, Azimuth 120, Dip 0,-22,-45,-67,-90, Correlograms



Total Copper, Zone4, Azimuth 150, Dip 0,-22,-45,-67,-90, Correlograms



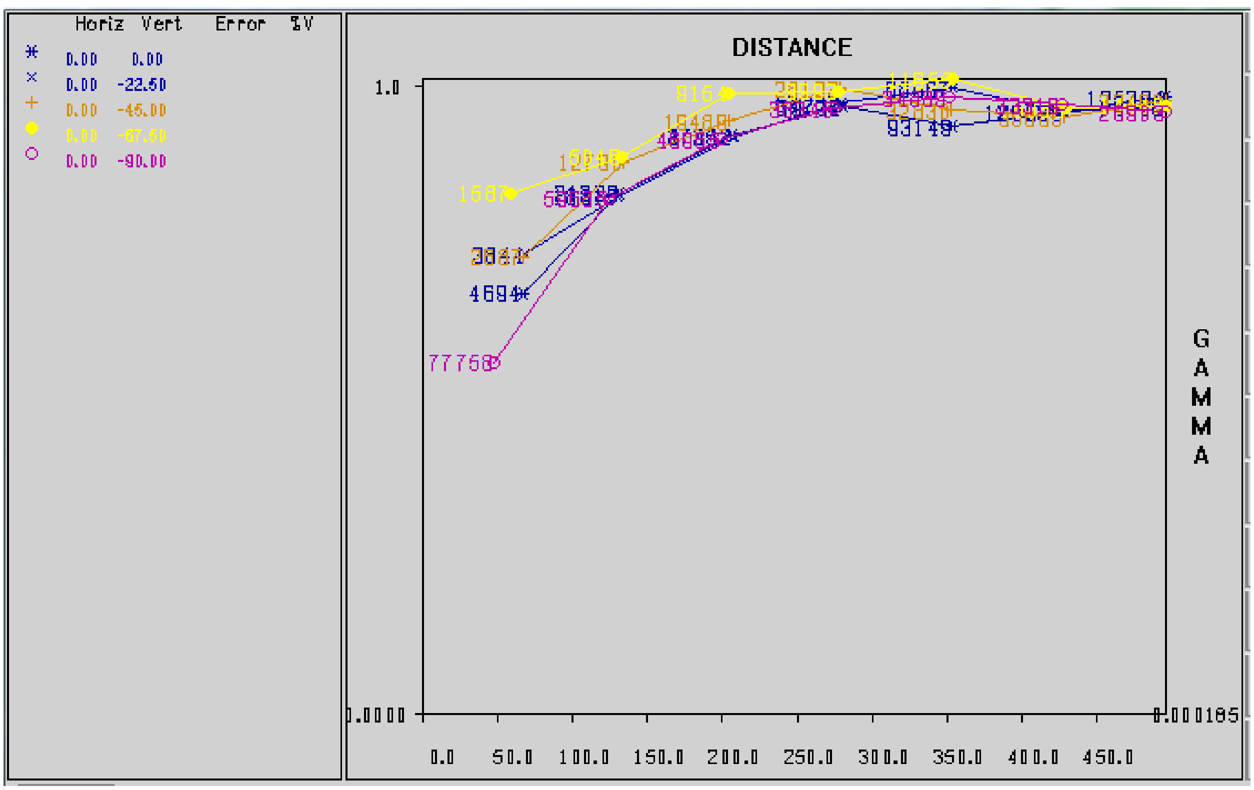
Appendix F: Ruth Estimation Parameters and Variograms – Gold

RUTH GOLD ESTIMATION PARAMETERS

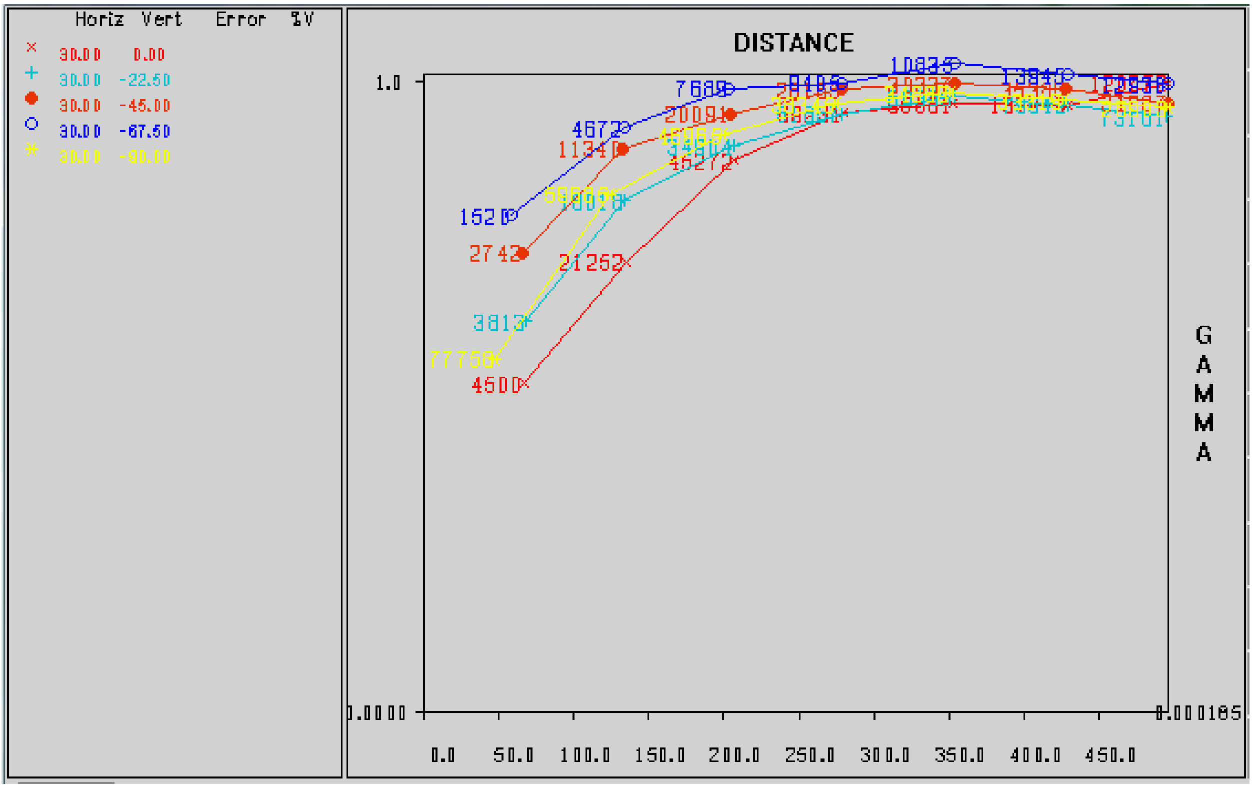
	Zone 1 (Hypogene)
Algorithm	ID2
Minimum # composites	1
Maximum # composites	12
Maximum # composites per hole	4
Search distance (ft) X,Y,Z	400,300,250
Search directions Az,Dip,Rotn	80,0,0
Geologic matching zone	none
Cap Grade (on assays)	0.5
Composites omitted	dump

Variograghy

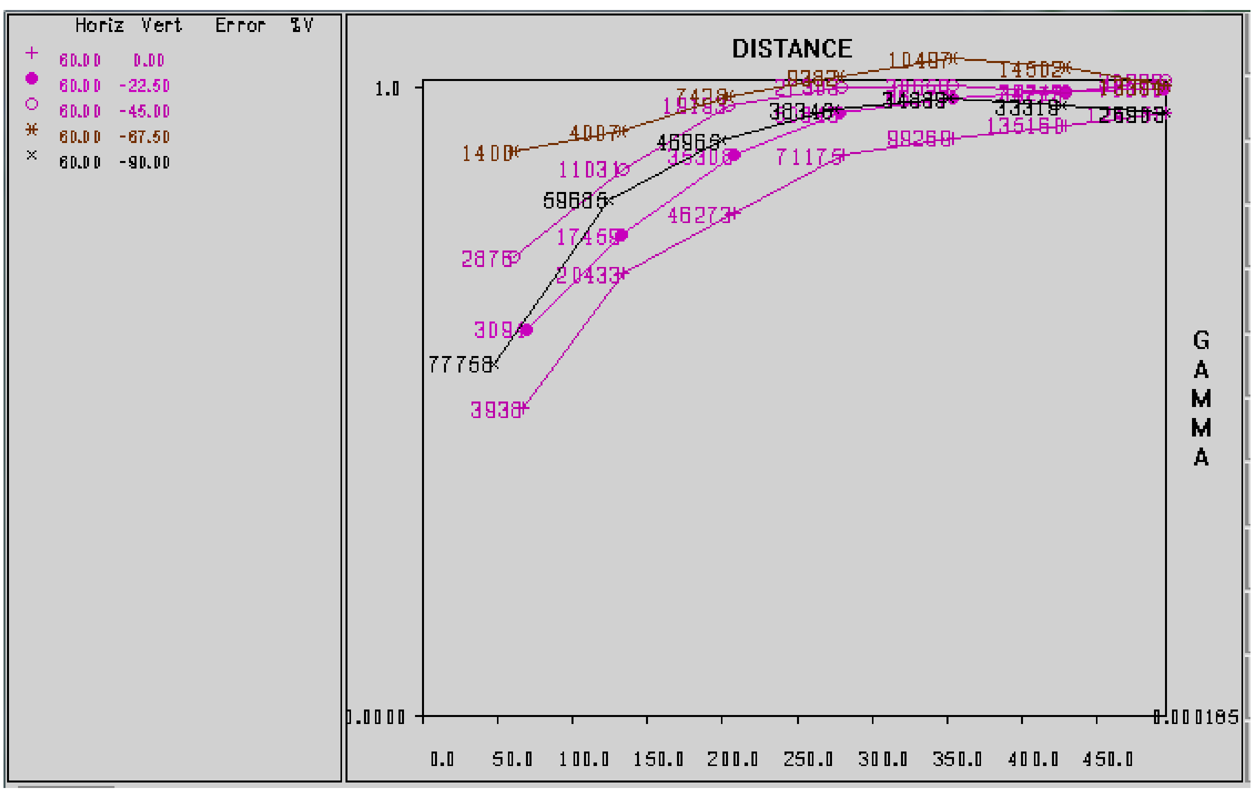
Gold, Zones (1,2,3,4) Azimuth 0, Dip 0,-22,-45,-67,-90, Correlograms



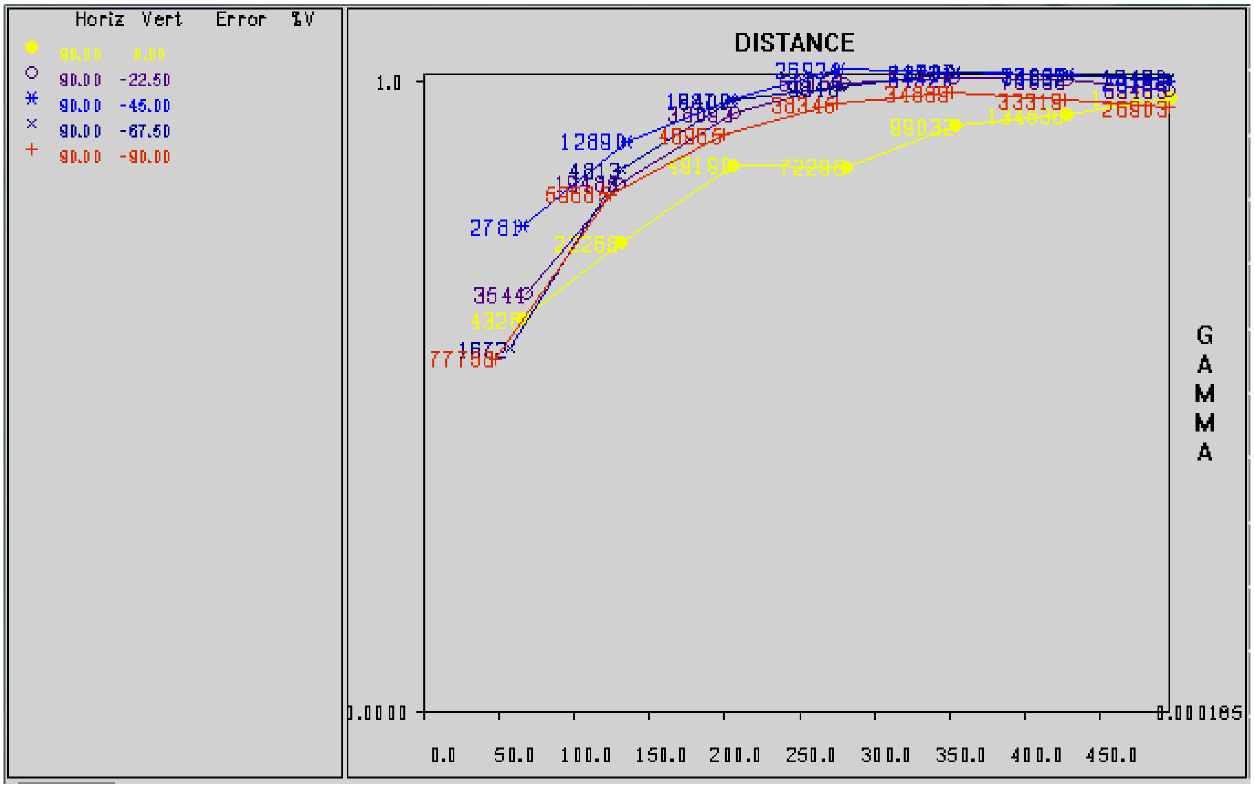
Gold, Zones (1,2,3,4) Azimuth 30, Dip 0,-22,-45,-67,-90, Correlograms



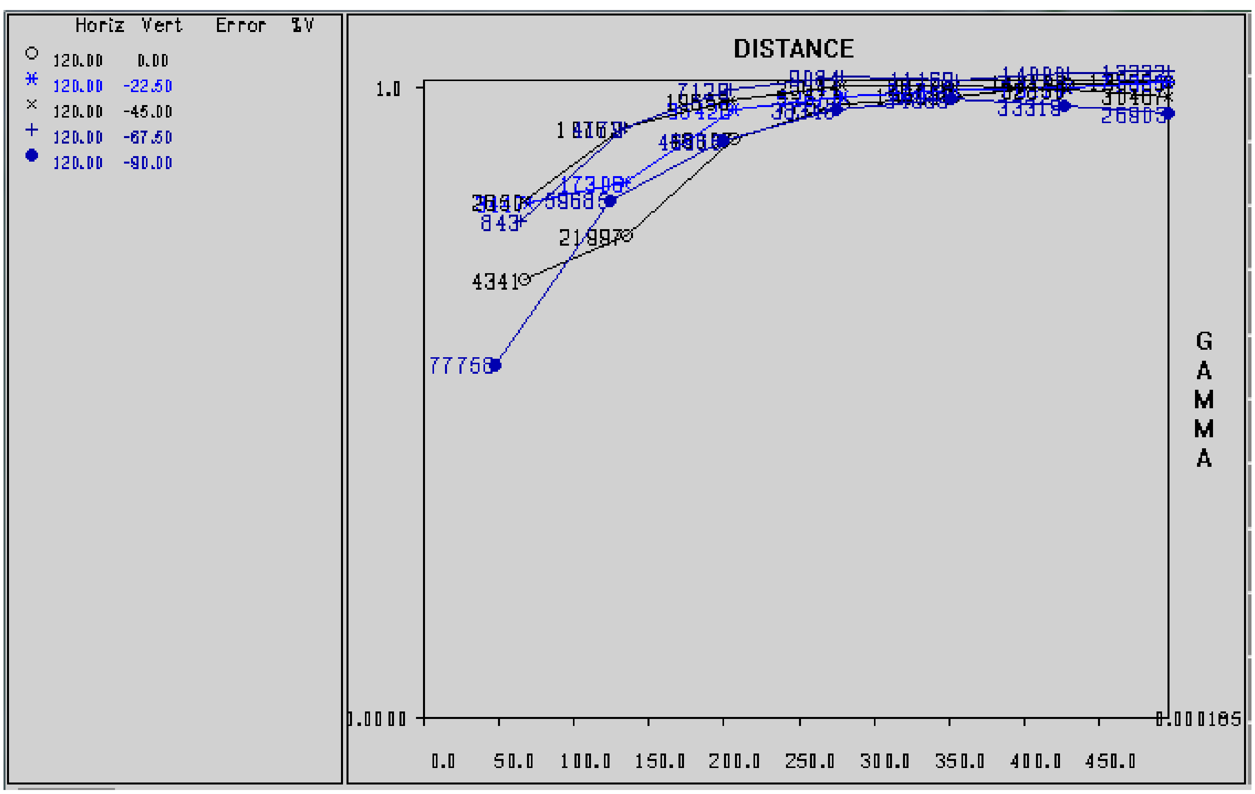
Gold, Zones (1,2,3,4) Azimuth 60, Dip 0,-22,-45,-67,-90, Correlograms



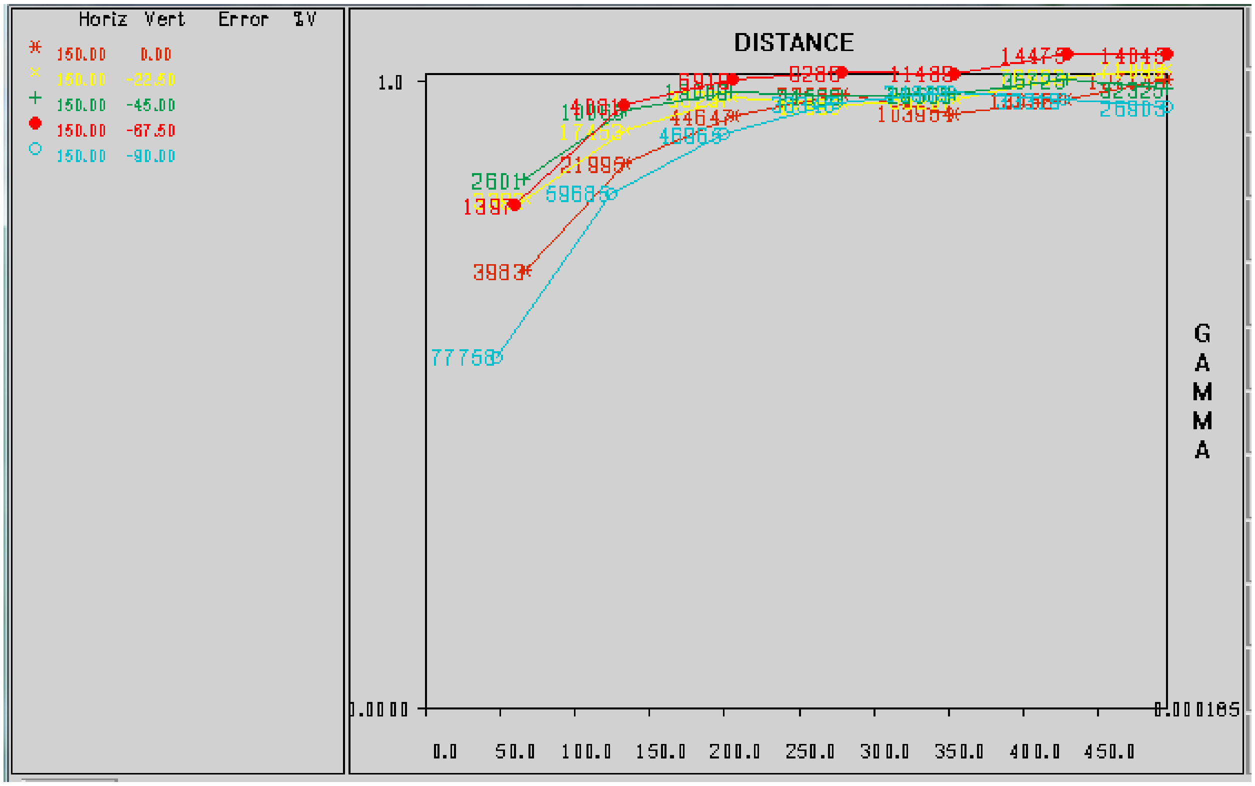
Gold, Zones (1,2,3,4) Azimuth 90, Dip 0,-22,-45,-67,-90, Correlograms



Gold, Zones (1,2,3,4) Azimuth 120, Dip 0,-22,-45,-67,-90, Correlograms



Gold, Zones (1,2,3,4) Azimuth 150, Dip 0,-22,-45,-67,-90, Correlograms



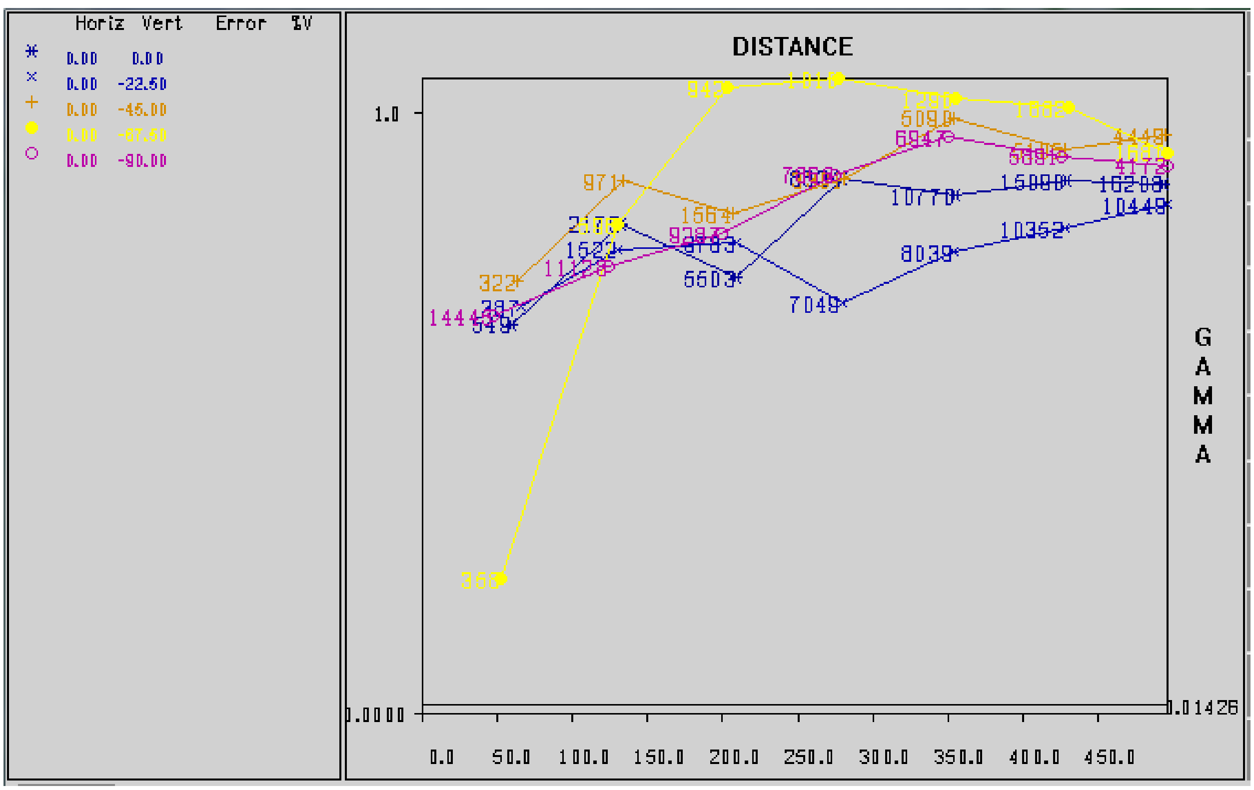
Appendix G: Ruth Estimation Parameters and Variograms – Soluble Copper

RUTH SOLUBLE COPPER ESTIMATION PARAMETERS

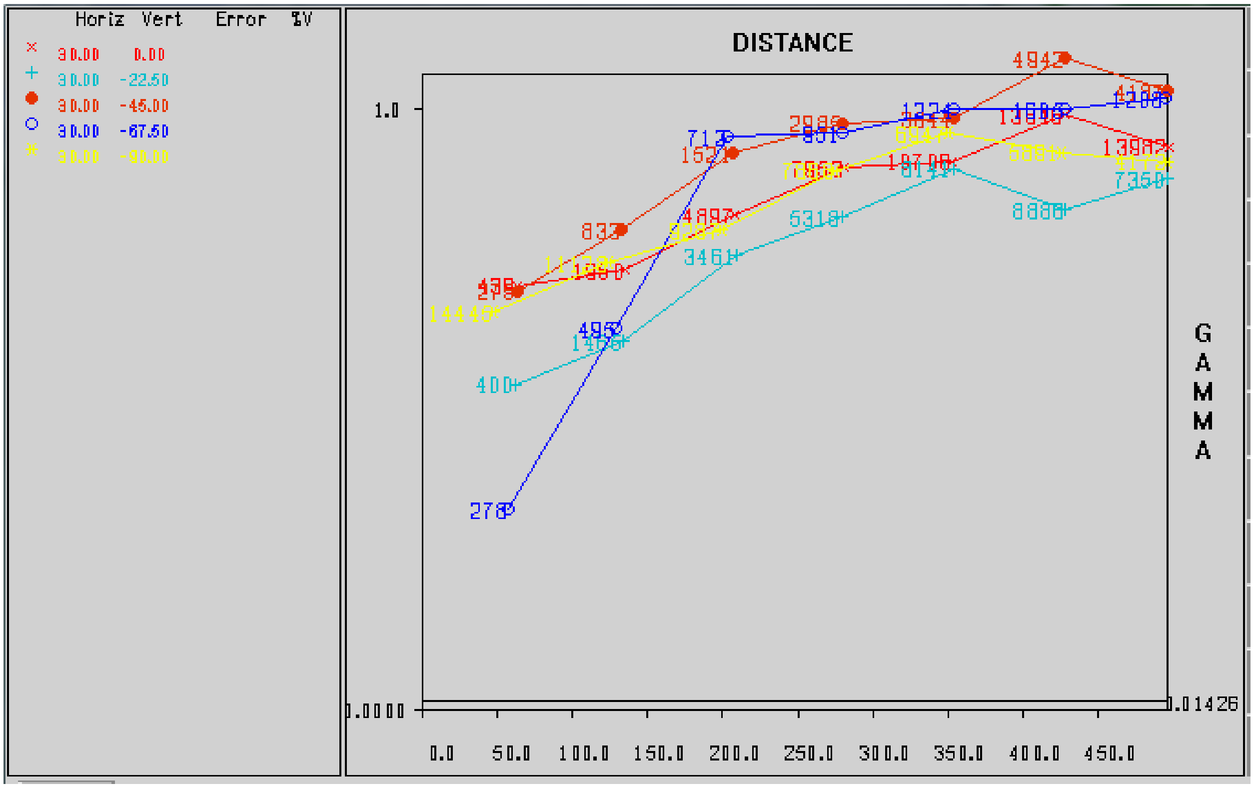
	Zone 1 (Hypogene)	Zone 2 (CEB)	Zone 3 (Leach Cap)	Zone 4 (Other)
Algorithm	ID2	ID2	ID2	ID2
Minimum # composites	1	1	1	1
Maximum # composites	12	12	12	12
Maximum # composites per hole	4	4	4	4
Search distance (ft) X,Y,Z	300,300,300	400,300,250	300,250,200	400,300,250
Search directions Az,Dip,Rotn	0,0,0,	100,0,0	100,0,0	80,0,0
Geologic matching zone	1	2	3	4
Cap on solcu/tcu ratio	1	1	1	1
Composites omitted	dump	dump	dump	dump

Variography

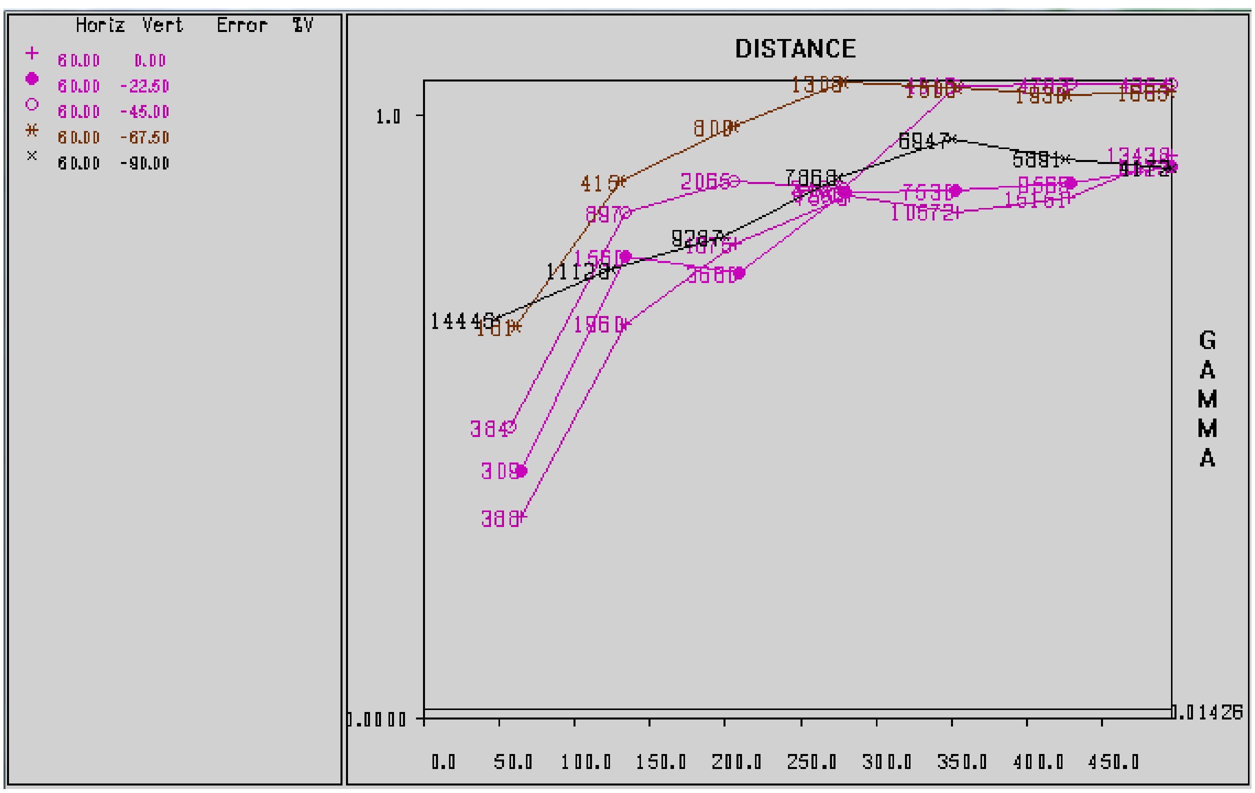
Soluble Copper, Zone1, Azimuth 0, Dip 0,-22,-45,-67,-90, Correlograms



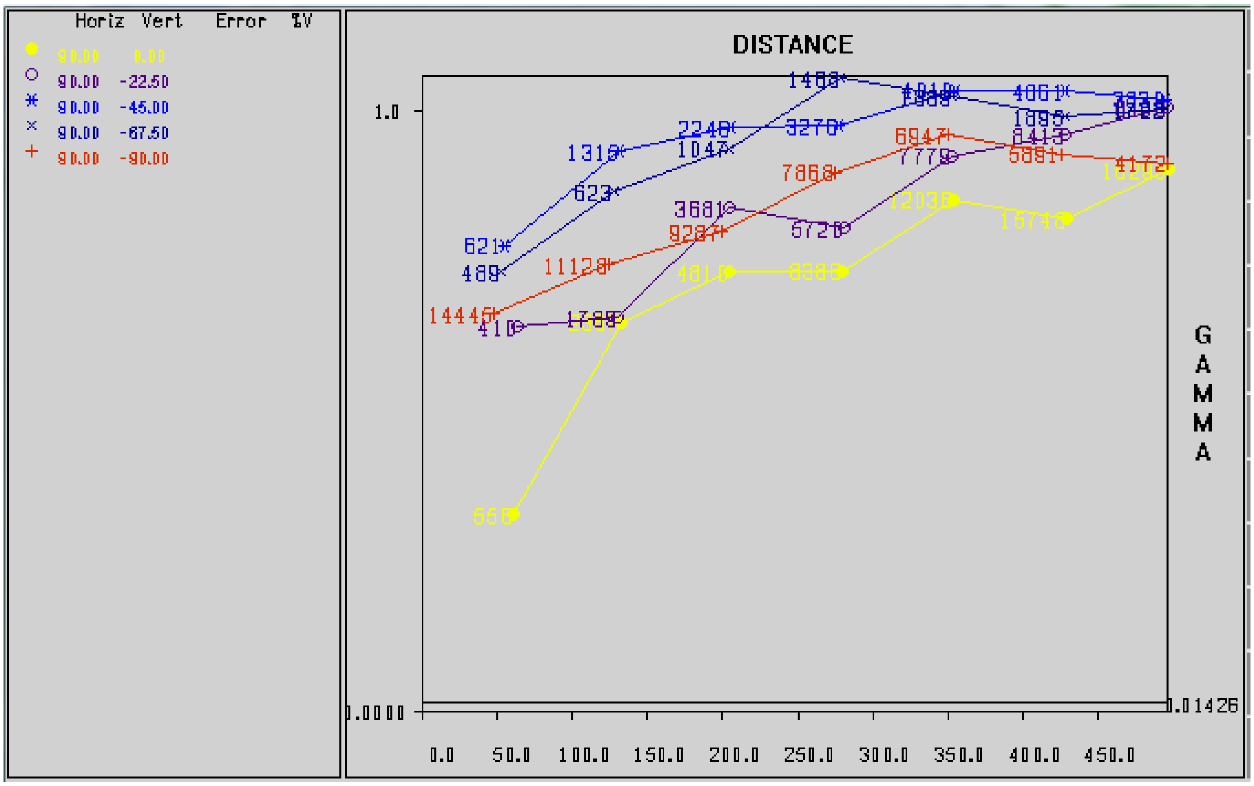
Soluble Copper, Zone1, Azimuth 30, Dip 0,-22,-45,-67,-90, Correlograms



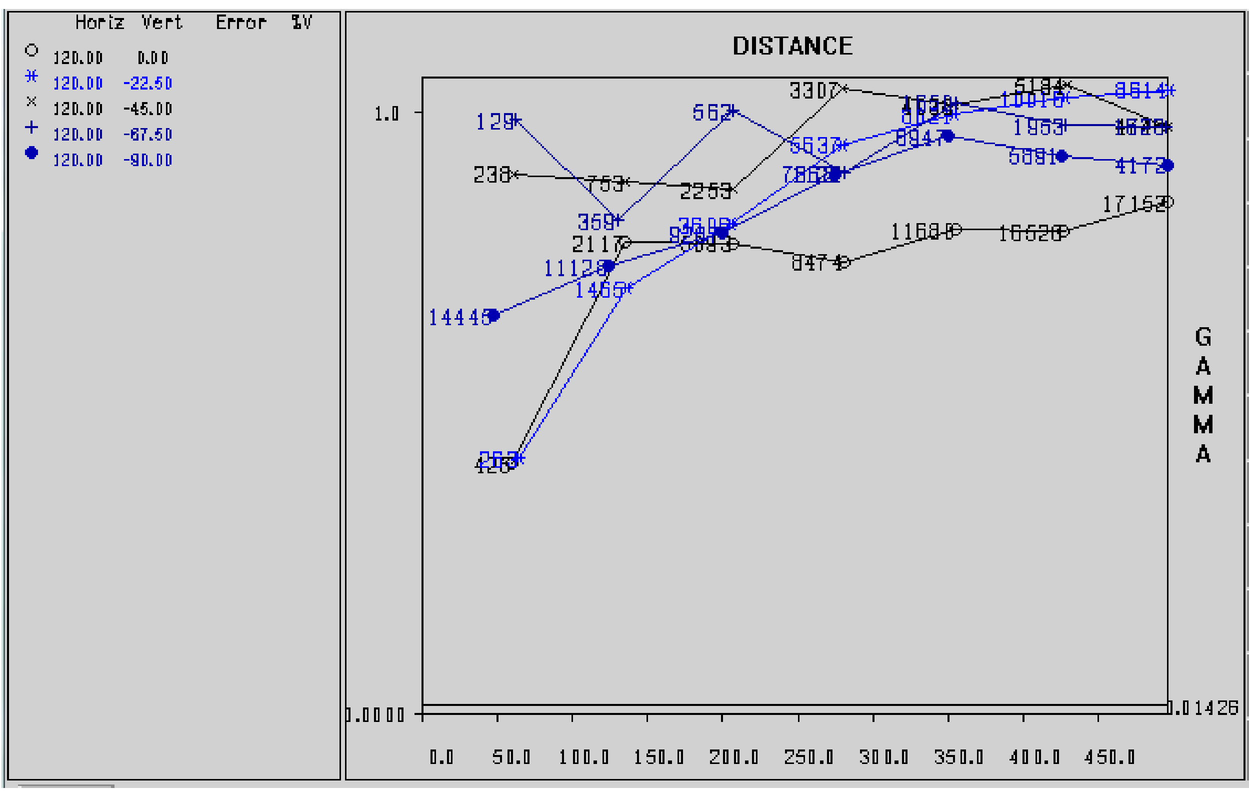
Soluble Copper, Zone1, Azimuth 60, Dip 0,-22,-45,-67,-90, Correlograms



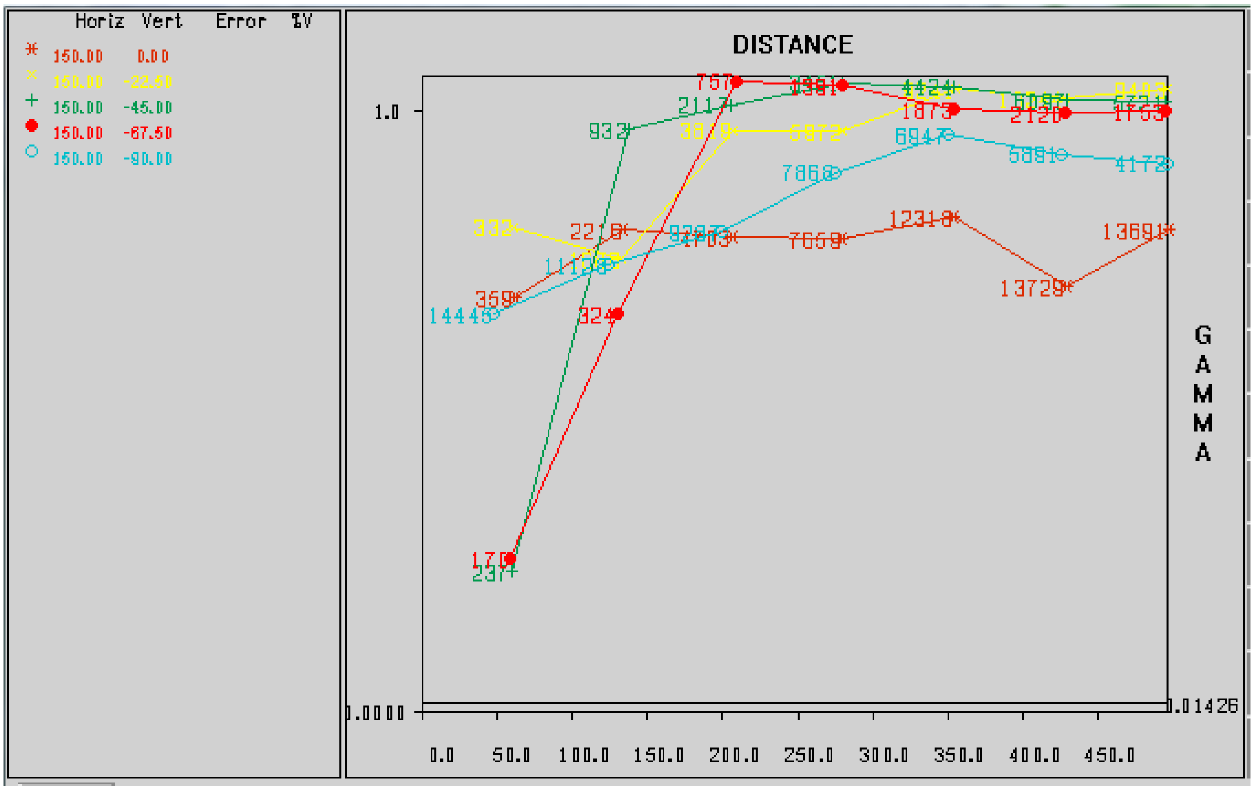
Soluble Copper, Zone1, Azimuth 90, Dip 0,-22,-45,-67,-90, Correlograms



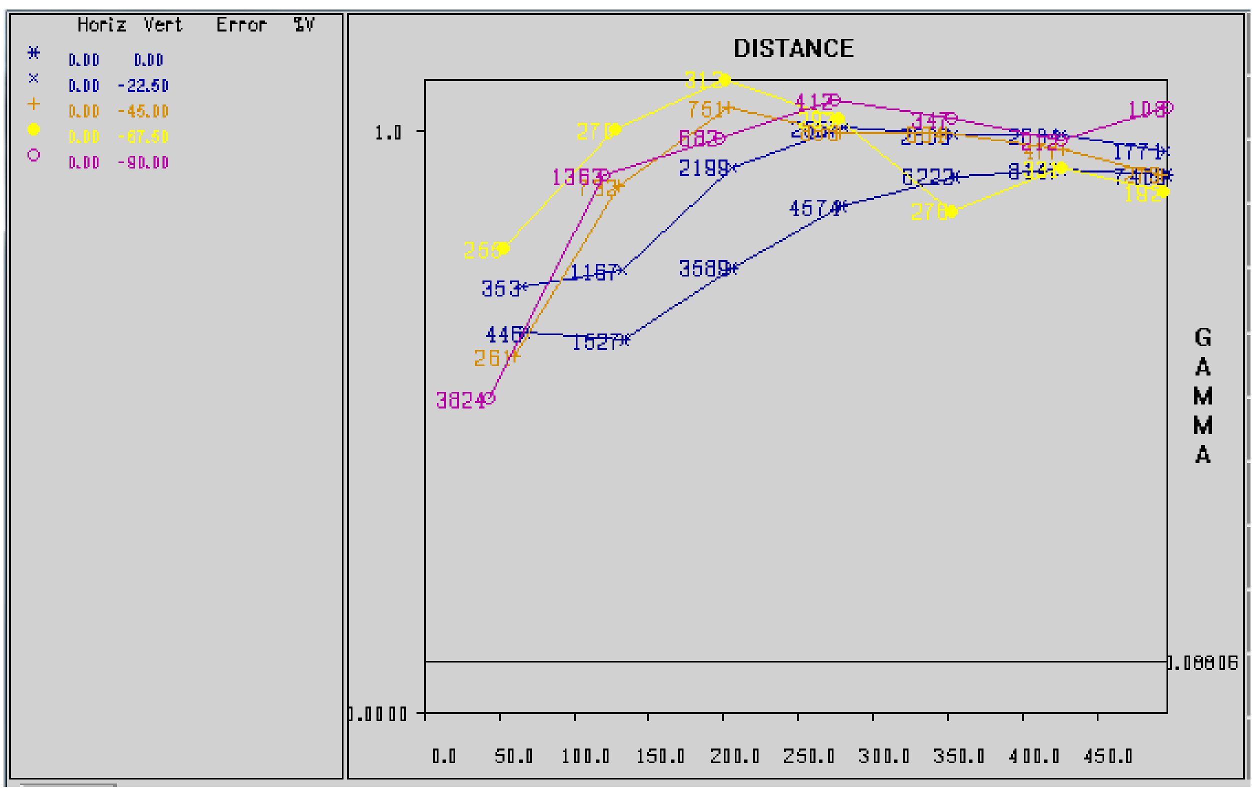
Soluble Copper, Zone1, Azimuth 120, Dip 0,-22,-45,-67,-90, Correlograms



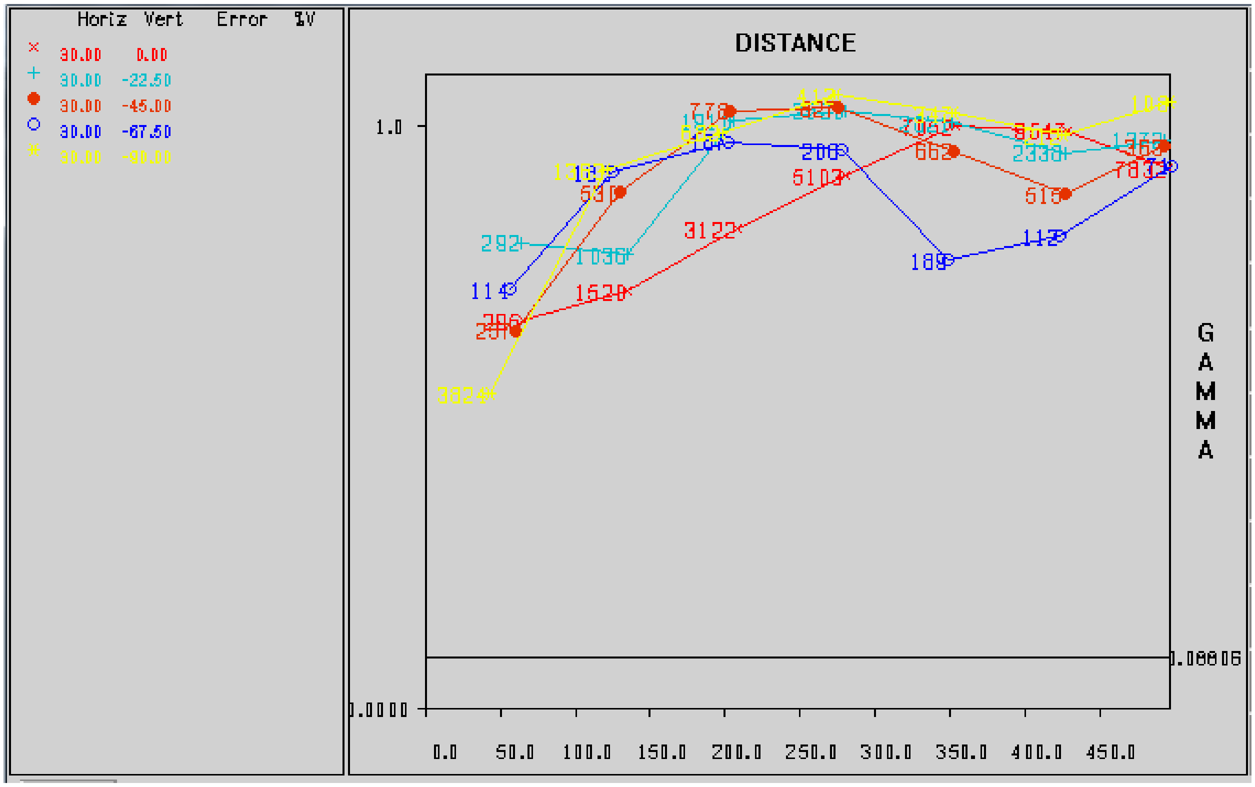
Soluble Copper, Zone1, Azimuth 120, Dip 0,-22,-45,-67,-90, Correlograms



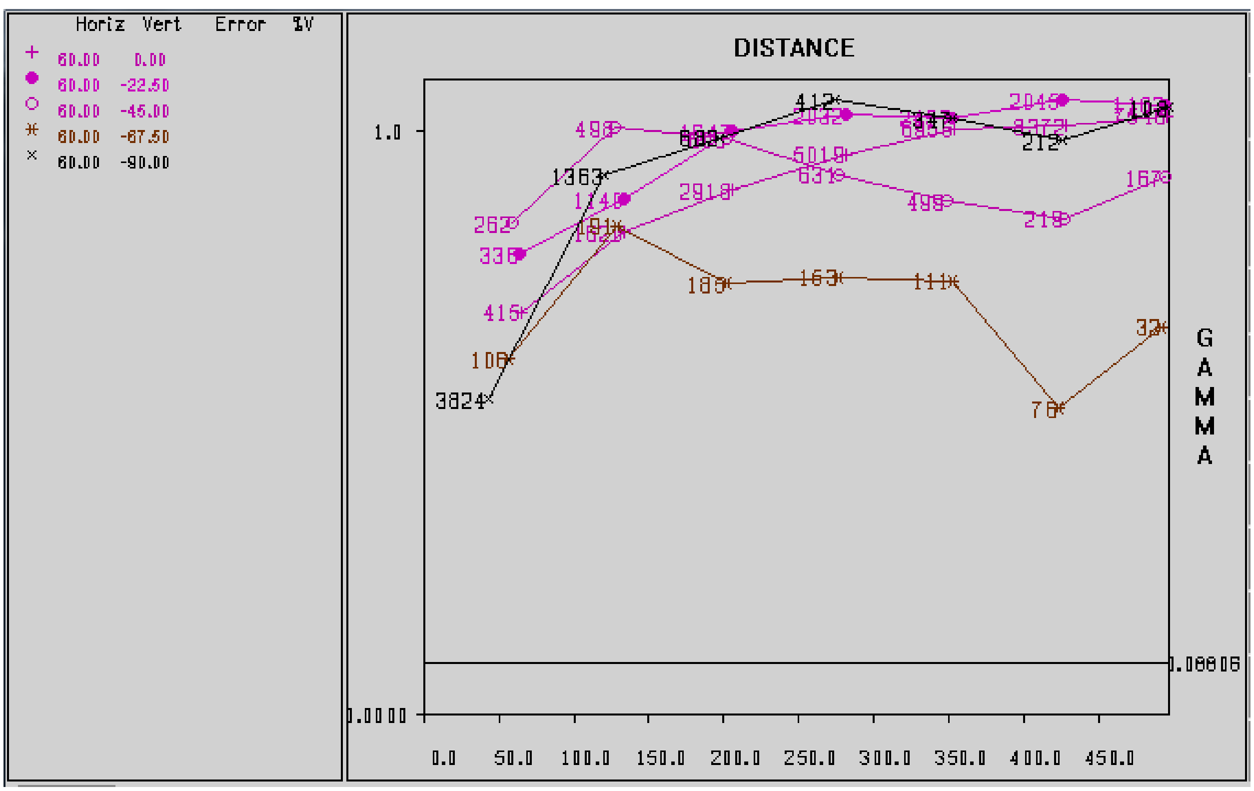
Soluble Copper, Zone2, Azimuth 0, Dip 0,-22,-45,-67,-90, Correlograms



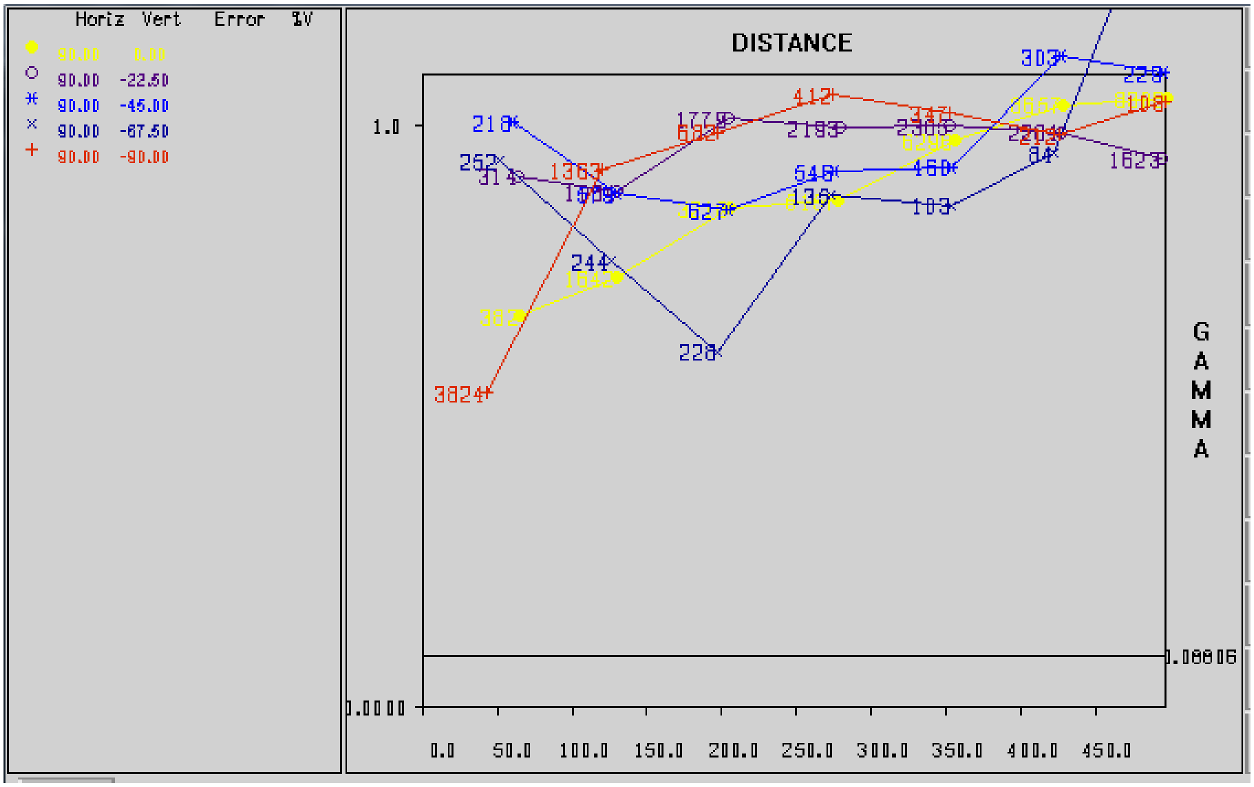
Soluble Copper, Zone2, Azimuth 30, Dip 0,-22,-45,-67,-90, Correlograms



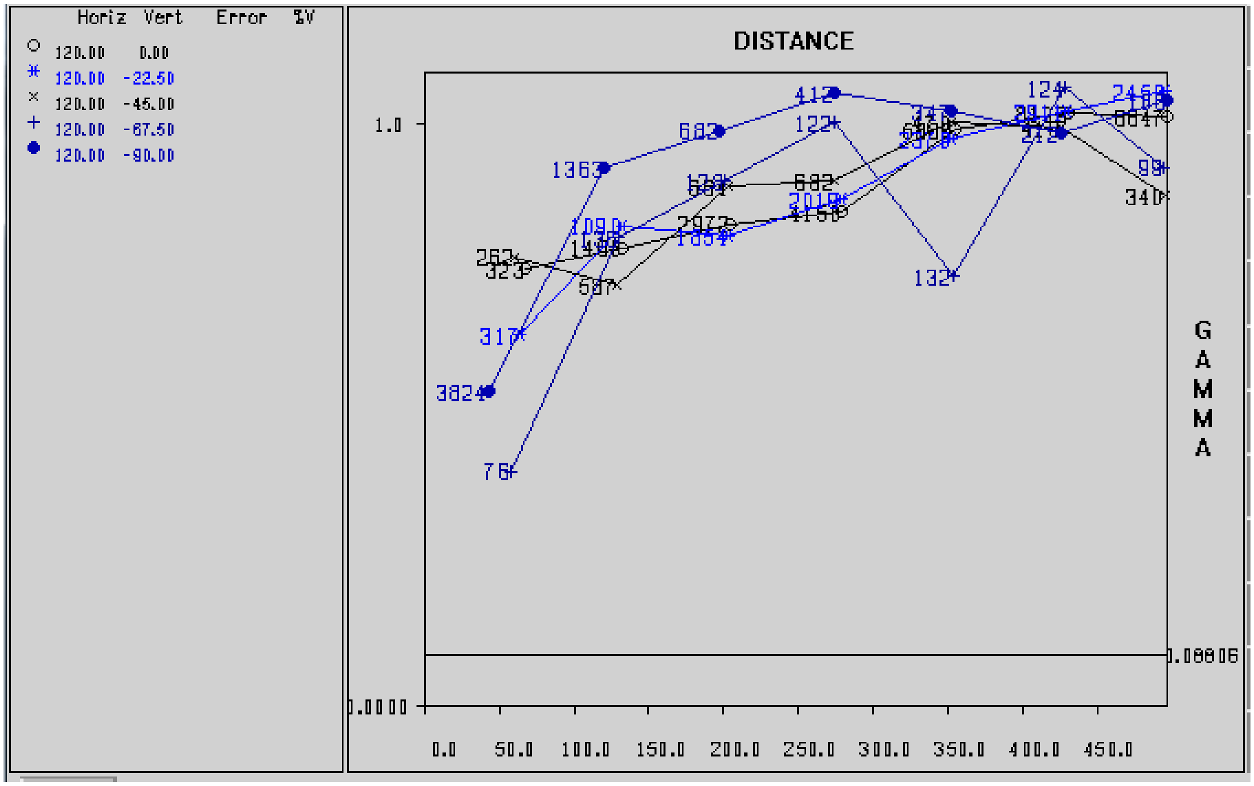
Soluble Copper, Zone2, Azimuth 60, Dip 0,-22,-45,-67,-90, Correlograms



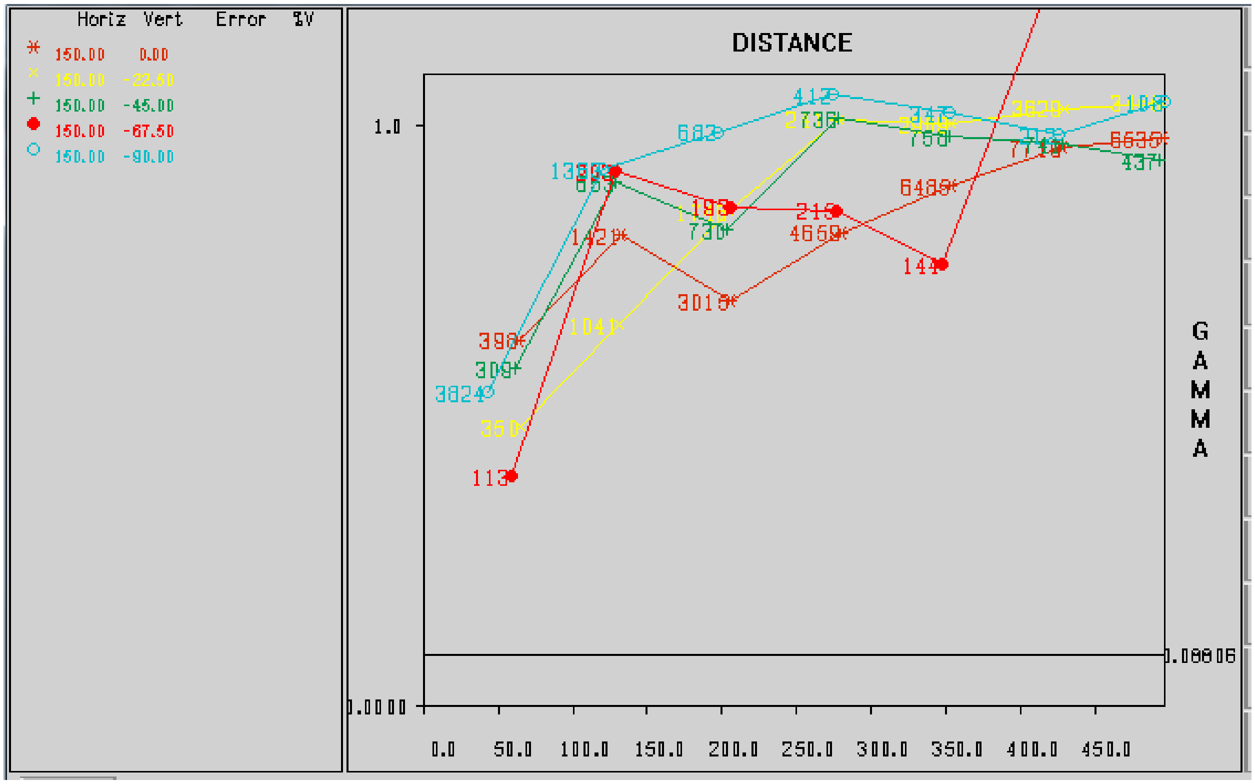
Soluble Copper, Zone2, Azimuth 90, Dip 0,-22,-45,-67,-90, Correlograms



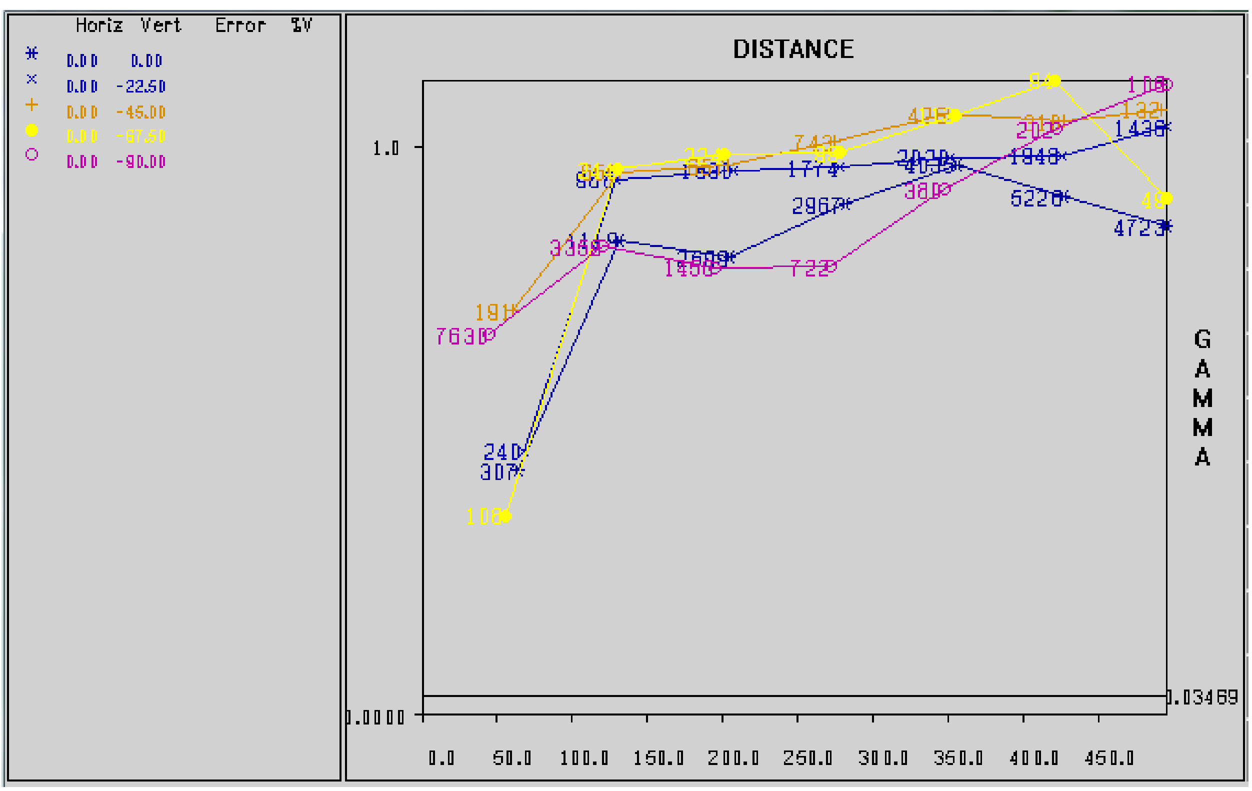
Soluble Copper, Zone2, Azimuth 120, Dip 0,-22,-45,-67,-90, Correlograms



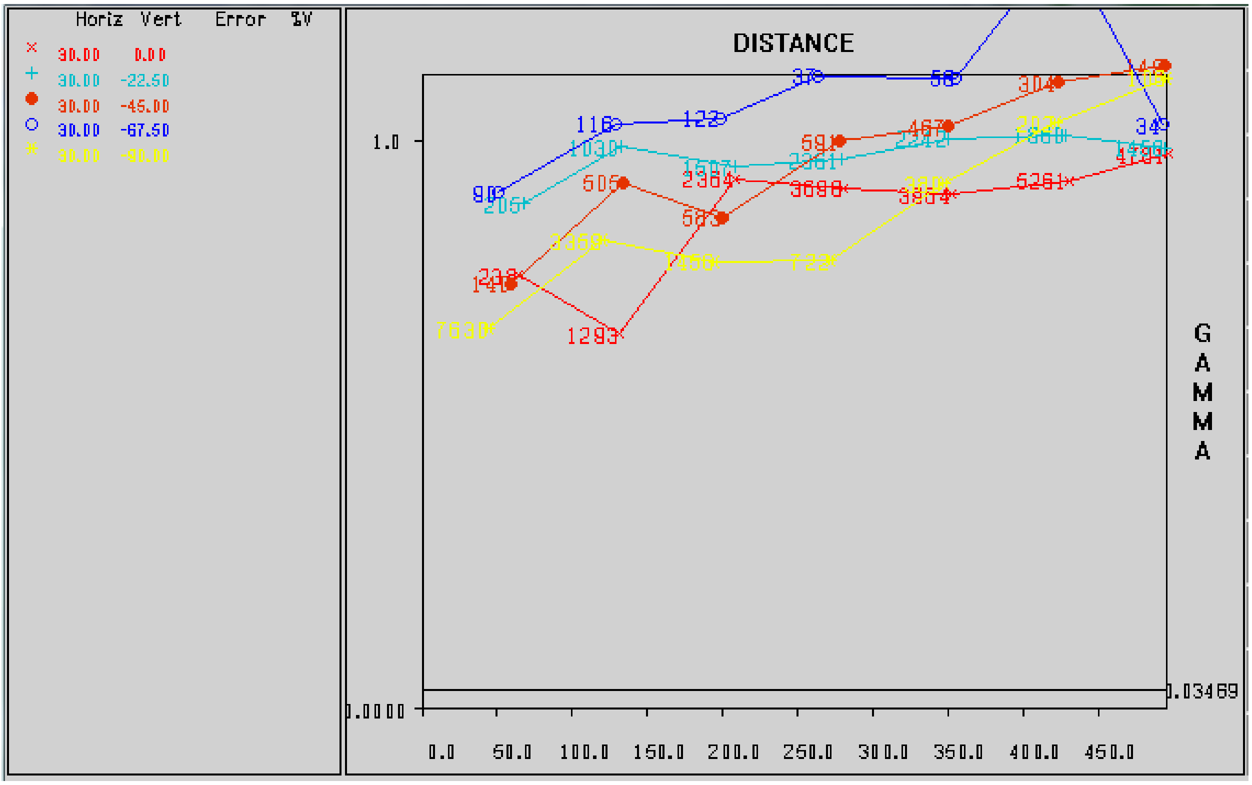
Soluble Copper, Zone2, Azimuth 150, Dip 0,-22,-45,-67,-90, Correlograms



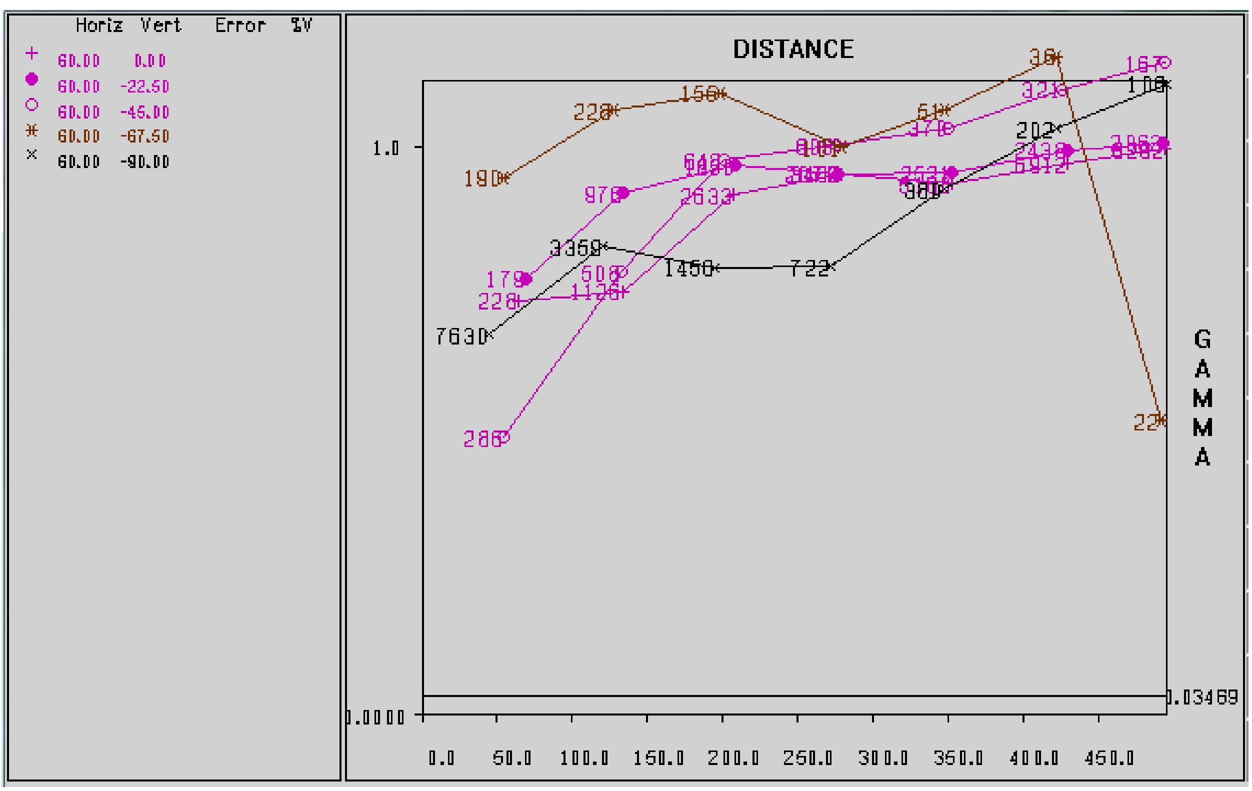
Soluble Copper, Zone3, Azimuth 0, Dip 0,-22,-45,-67,-90, Correlograms



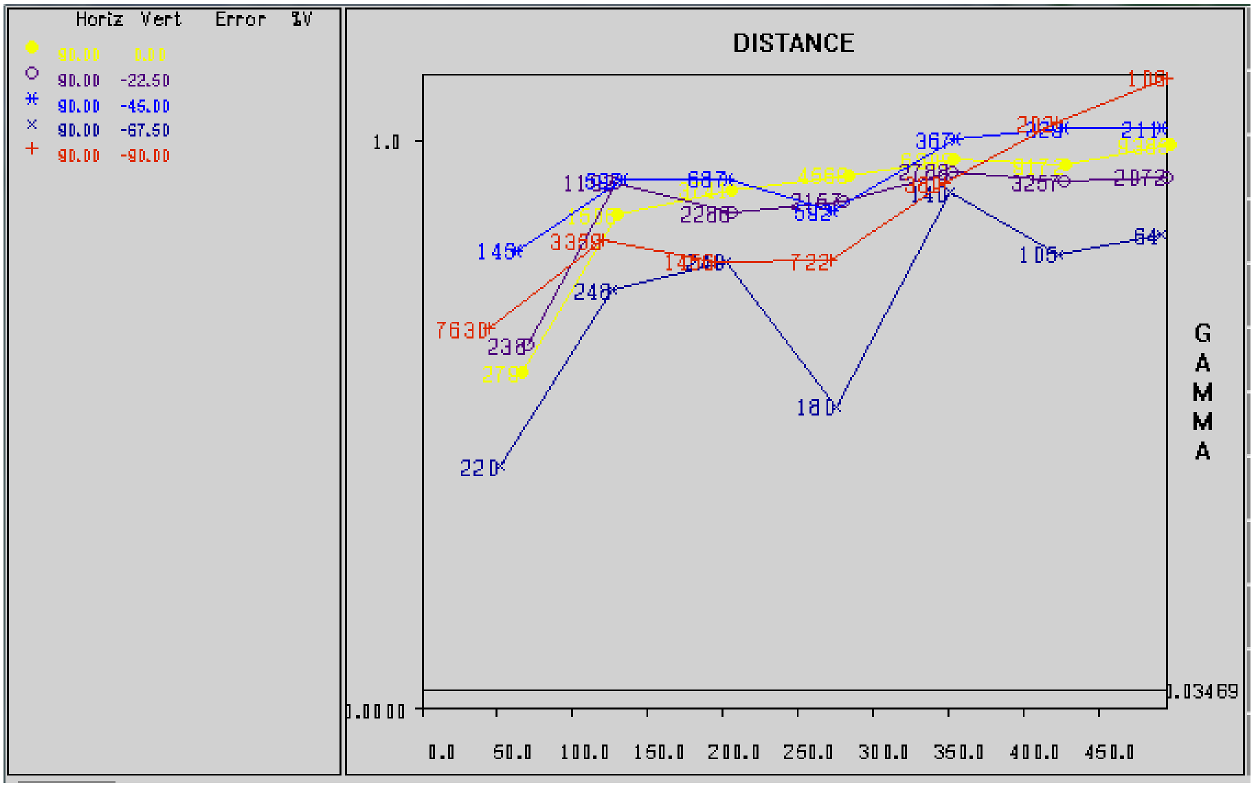
Soluble Copper, Zone3, Azimuth 30, Dip 0,-22,-45,-67,-90, Correlograms



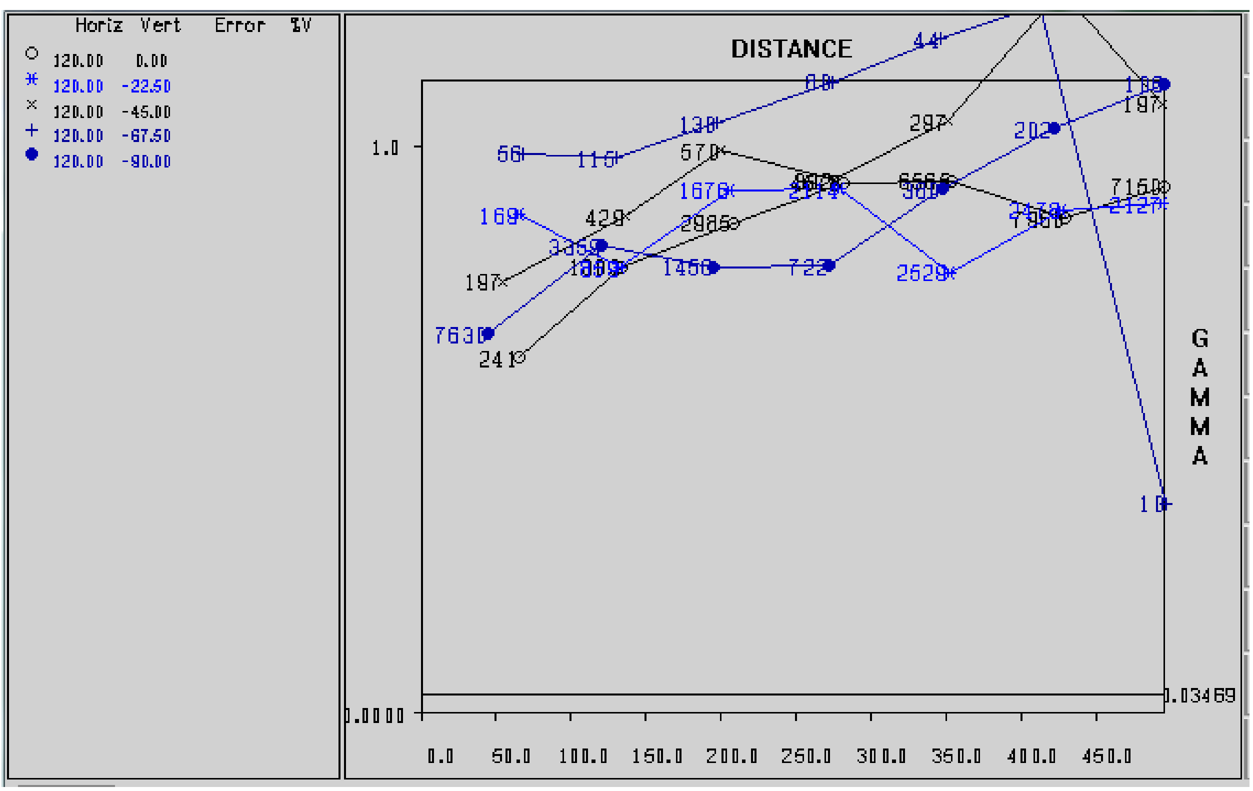
Soluble Copper, Zone3, Azimuth 60, Dip 0,-22,-45,-67,-90, Correlograms



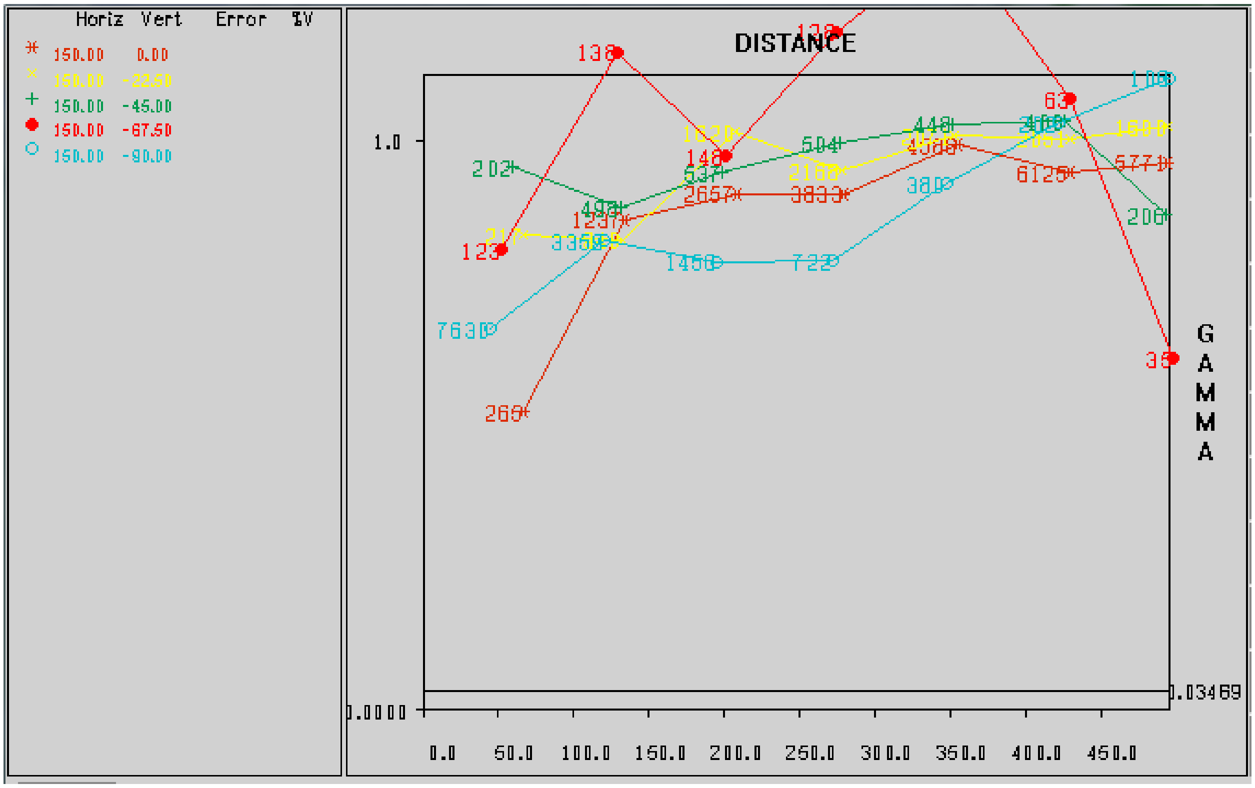
Soluble Copper, Zone3, Azimuth 90, Dip 0,-22,-45,-67,-90, Correlograms



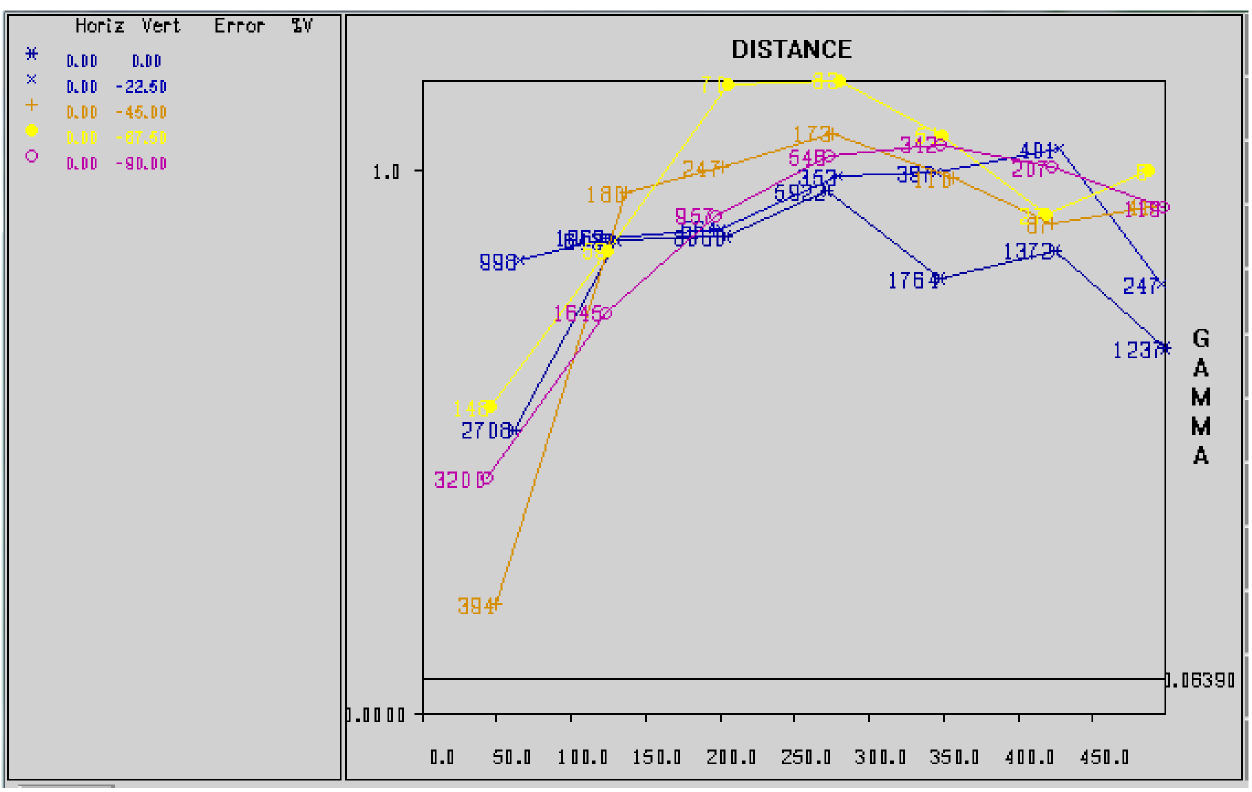
Soluble Copper, Zone3, Azimuth 120, Dip 0,-22,-45,-67,-90, Correlograms



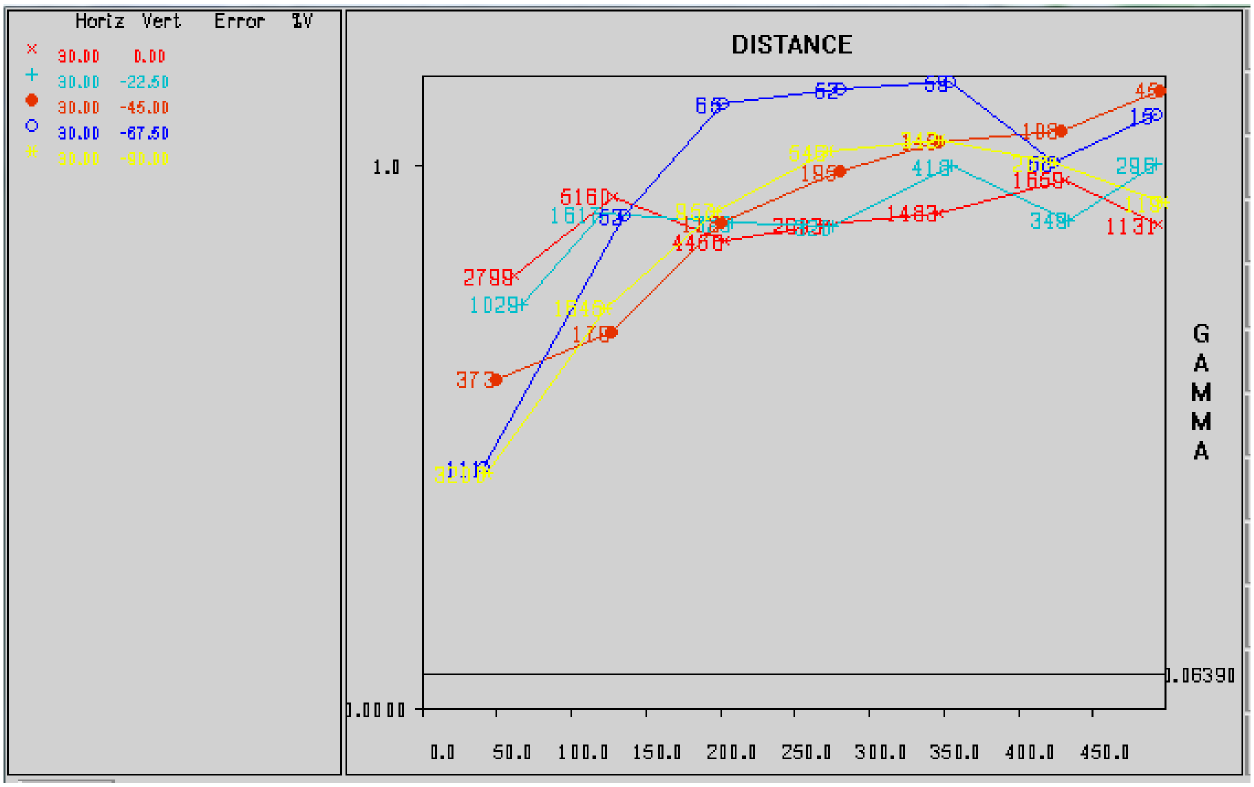
Soluble Copper, Zone3, Azimuth 150, Dip 0,-22,-45,-67,-90, Correlograms



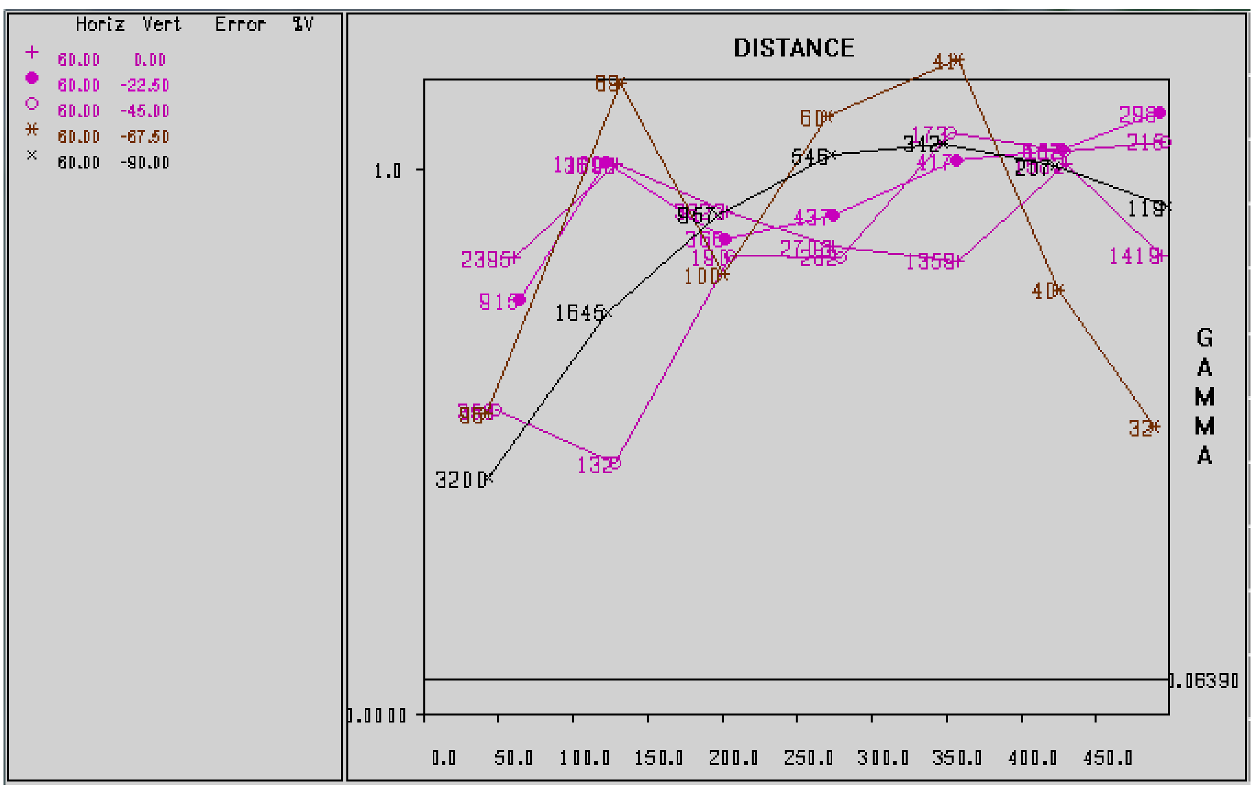
Soluble Copper, Zone4, Azimuth 0, Dip 0,-22,-45,-67,-90, Correlograms



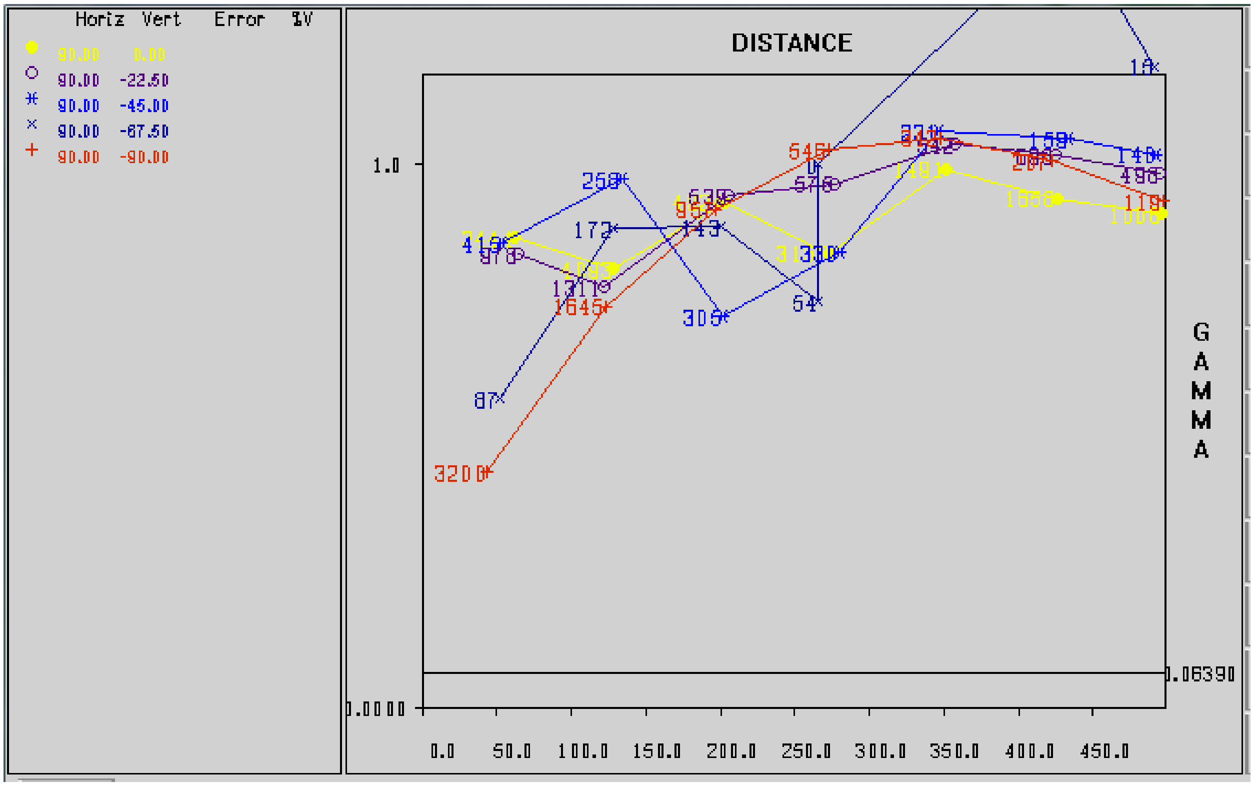
Soluble Copper, Zone4, Azimuth 30, Dip 0,-22,-45,-67,-90, Correlograms



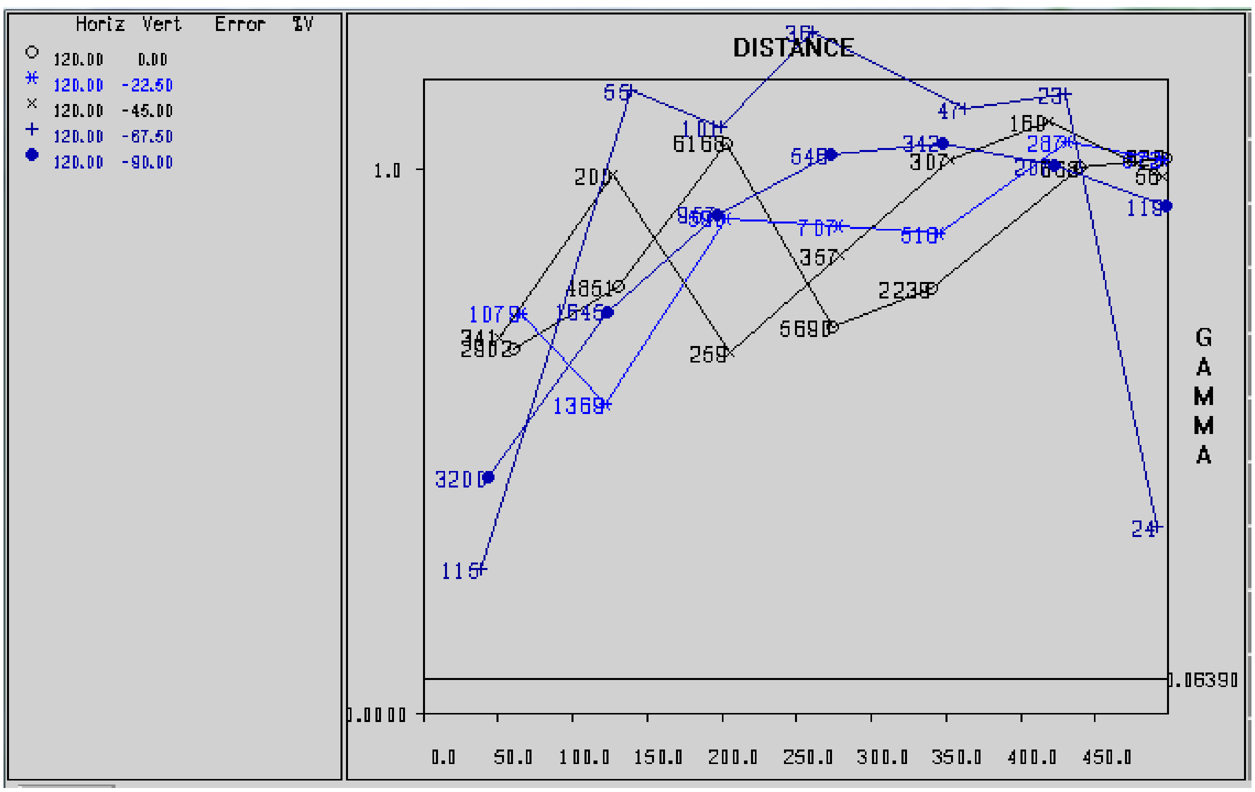
Soluble Copper, Zone4, Azimuth 60, Dip 0,-22,-45,-67,-90, Correlograms



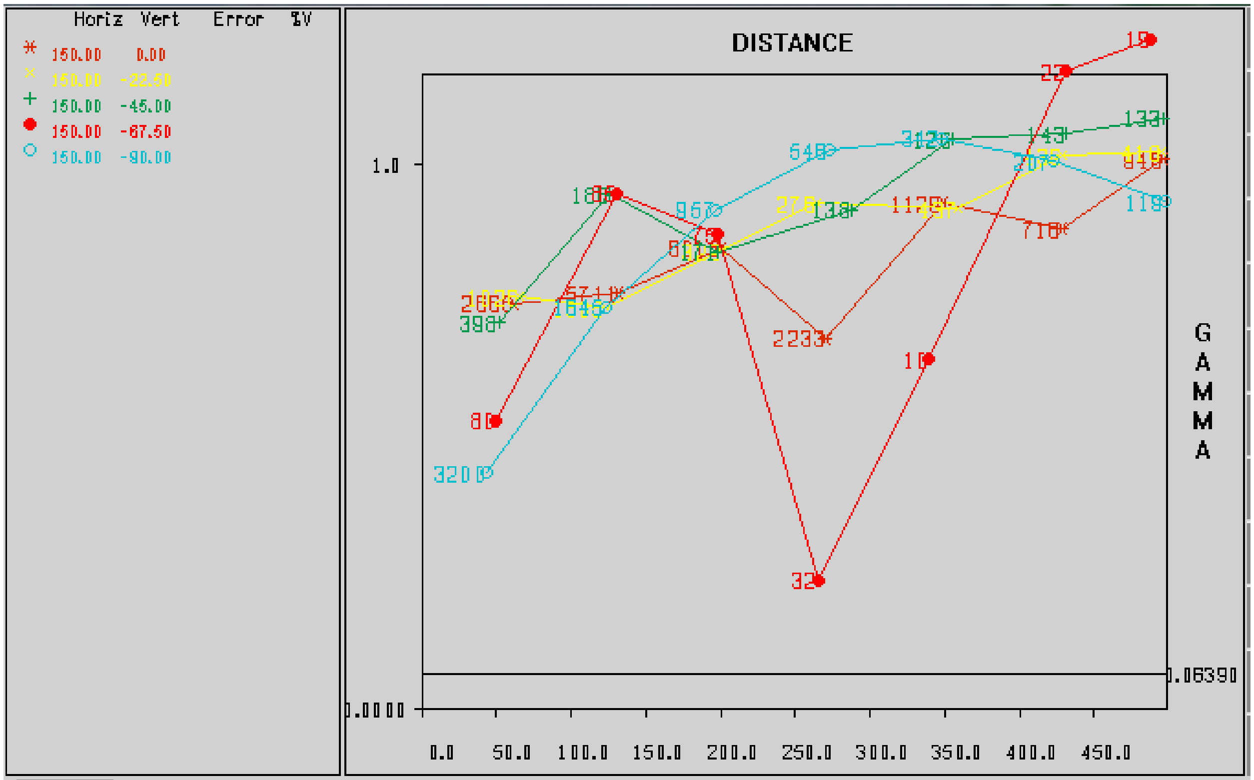
Soluble Copper, Zone4, Azimuth 90, Dip 0,-22,-45,-67,-90, Correlograms



Soluble Copper, Zone4, Azimuth 120, Dip 0,-22,-45,-67,-90, Correlograms



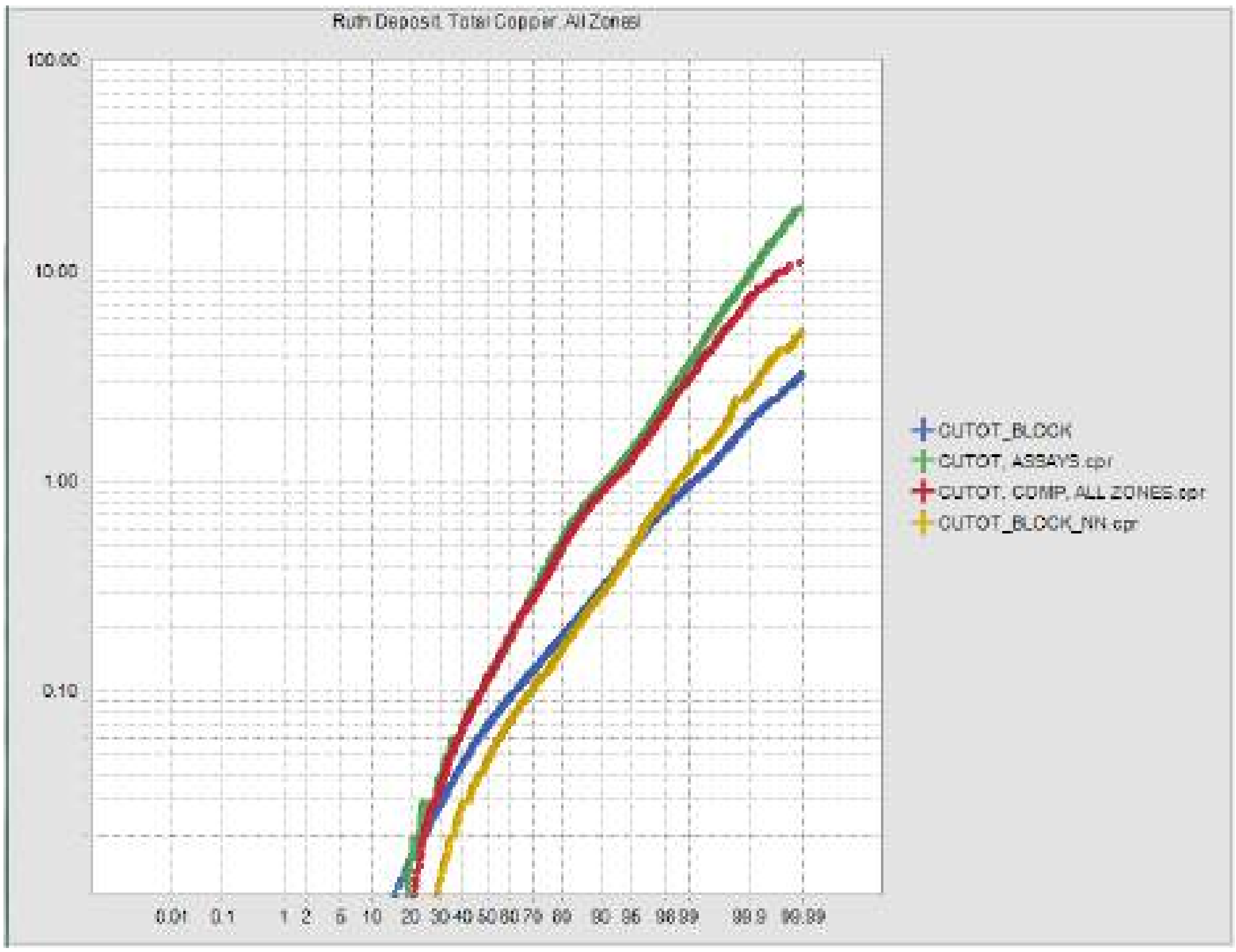
Soluble Copper, Zone4, Azimuth 150, Dip 0,-22,-45,-67,-90, Correlograms



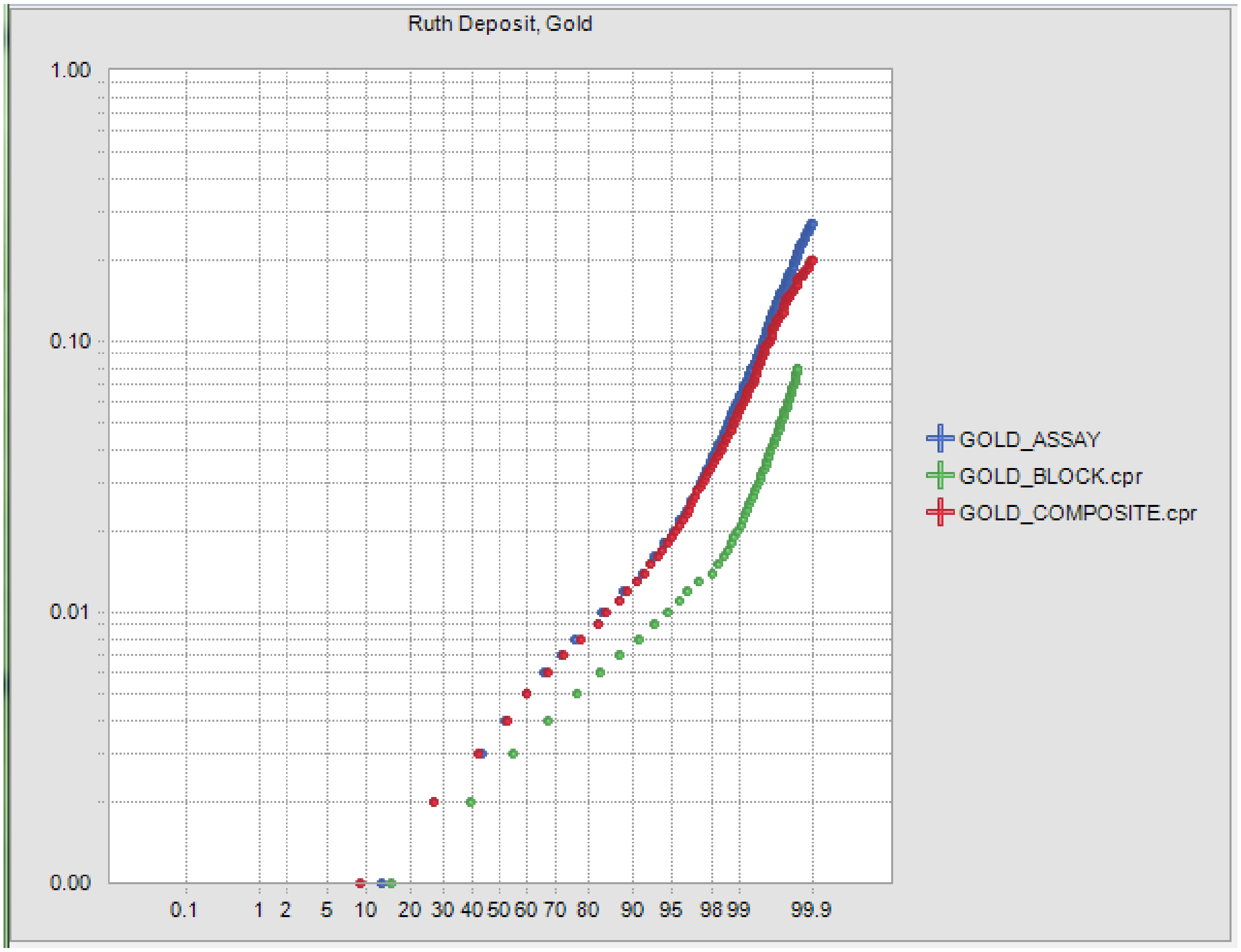
Appendix H: Ruth Distribution Plots and Model Checks

Ruth Deposit
Cumulative Probability Plots

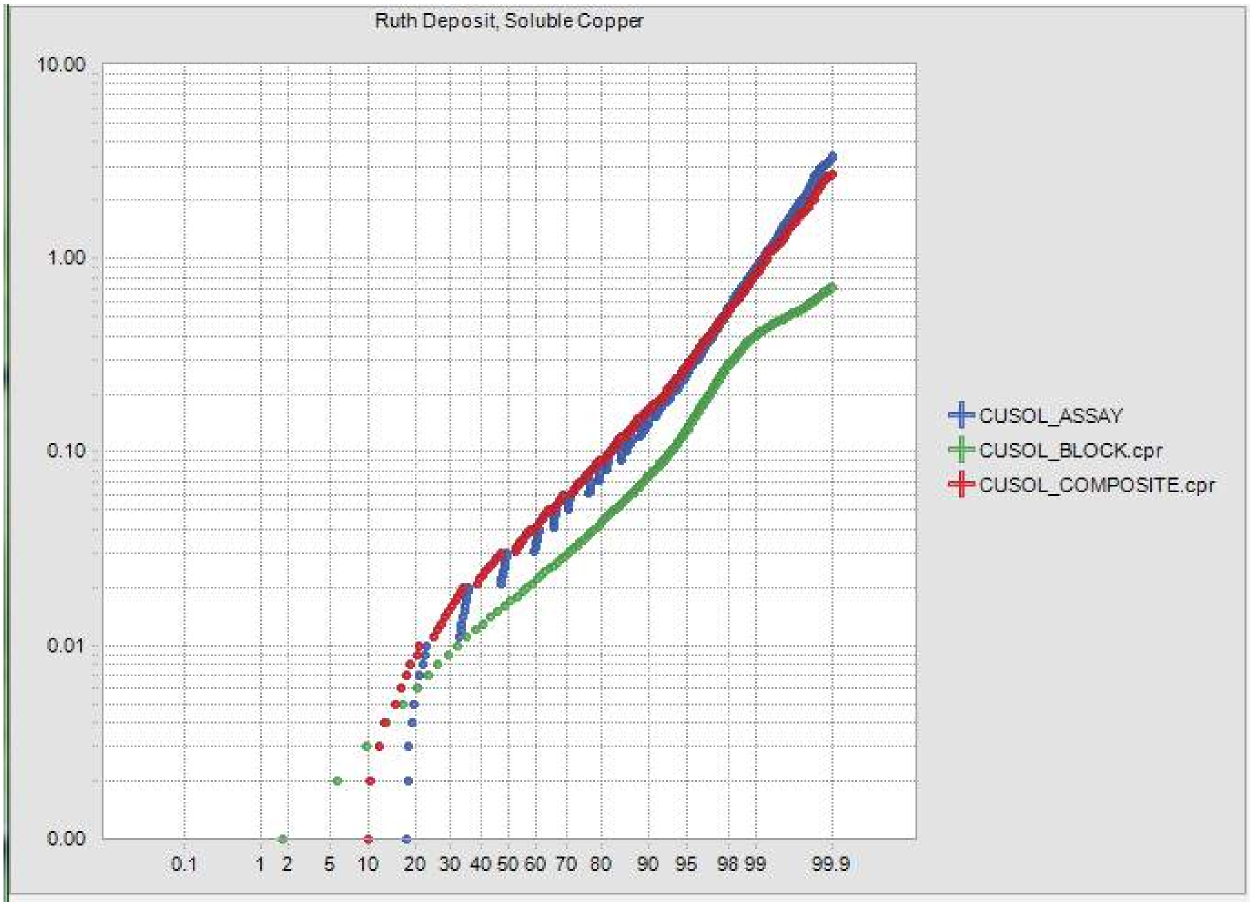
Item	**Label on chart**
Resource Model Total Copper	(CUTOT_BLOCK)
Polygonal (nearest neighbor) Total Copper	(CUTOT, BLOCK,NN)
Composites Total Copper	(CUTOT,COMP,ALLZONES)
Assays Total Copper	(CUTOT,ASSAYS)



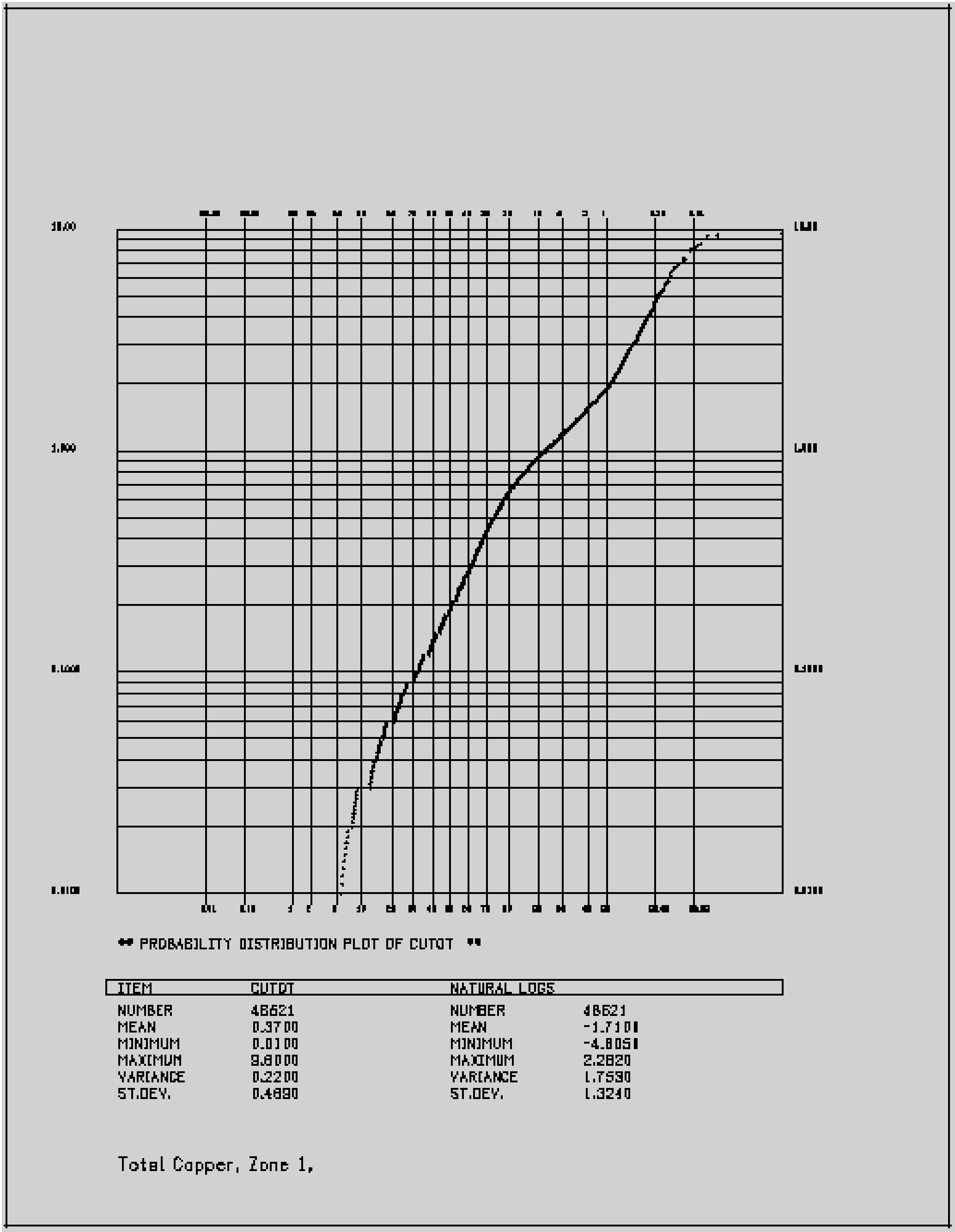
Item	**Label on chart**
Assays Gold	(GOLD_ASSAY)
Resource Model Gold	(GOLD_BLOCK)
Composite Gold	(GOLD_COMPOSITE)



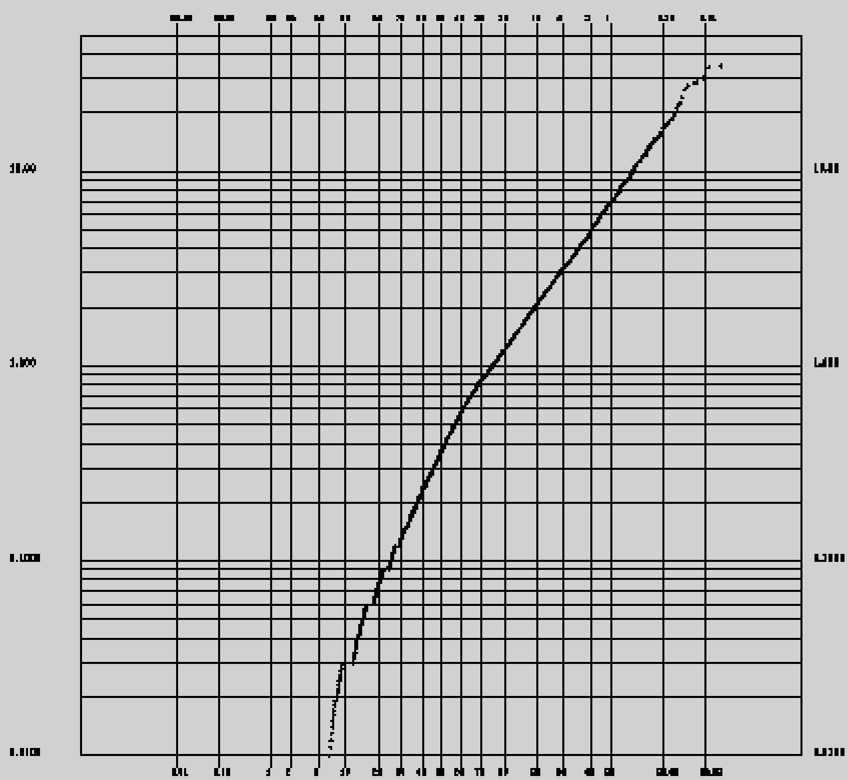
Item	**Label on chart**
Assays Soluble Copper	(CUSOL_ASSAY)
Resource Model Soluble Copper	(CUSOL_BLOCK)
Composite Soluble Copper	(CUSOL_COMPOSITE)



Ruth Deposit
Cumulative Probability Plot, Assays, Total Copper, Zone 1



Ruth Deposit
Cumulative Probability Plot, Assays, Total Copper, Zone 2

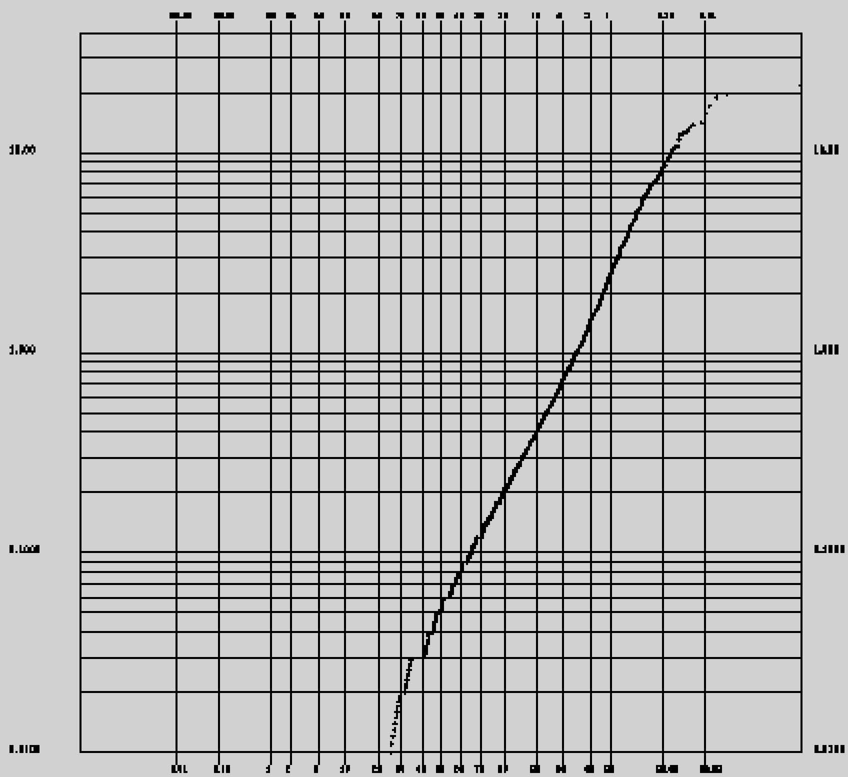


** PROBABILITY DISTRIBUTION PLOT OF CUTOT **

ITEM	CUTOT	NATURAL LOGS	
NUMBER	26434	NUMBER	26434
MEAN	0.8480	MEAN	-1.2091
MINIMUM	0.0100	MINIMUM	-4.6051
MAXIMUM	40.4800	MAXIMUM	3.7000
VARIANCE	2.3230	VARIANCE	2.8480
ST.DEV.	1.5240	ST.DEV.	1.6270

Total Copper, Zone 2.

Ruth Deposit
Cumulative Probability Plot, Assays, Total Copper, Zone 3

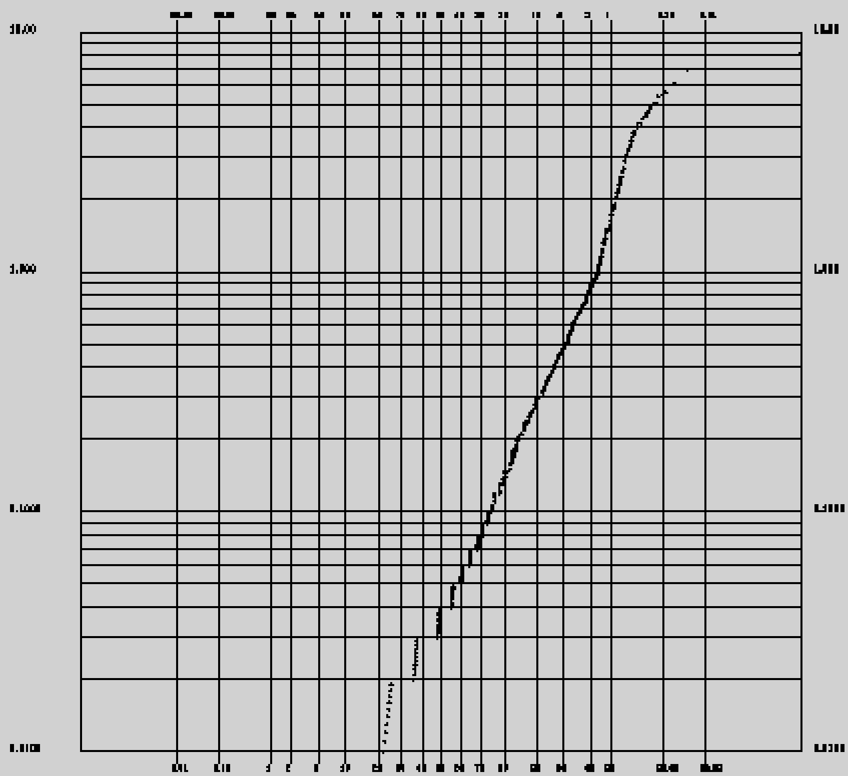


** PROBABILITY DISTRIBUTION PLOT OF CUTOT **

ITEM	CUTOT	NATURAL LOGS	
NUMBER	40431	NUMBER	40431
MEAN	0.1960	MEAN	-2.8731
MINIMUM	0.0100	MINIMUM	-4.6051
MAXIMUM	21.5700	MAXIMUM	3.0710
VARIANCE	0.3840	VARIANCE	2.1130
ST.DEV.	0.6200	ST.DEV.	1.4540

Total Copper, Zone 3

Ruth Deposit
Cumulative Probability Plot, Assays, Total Copper, Zone 4

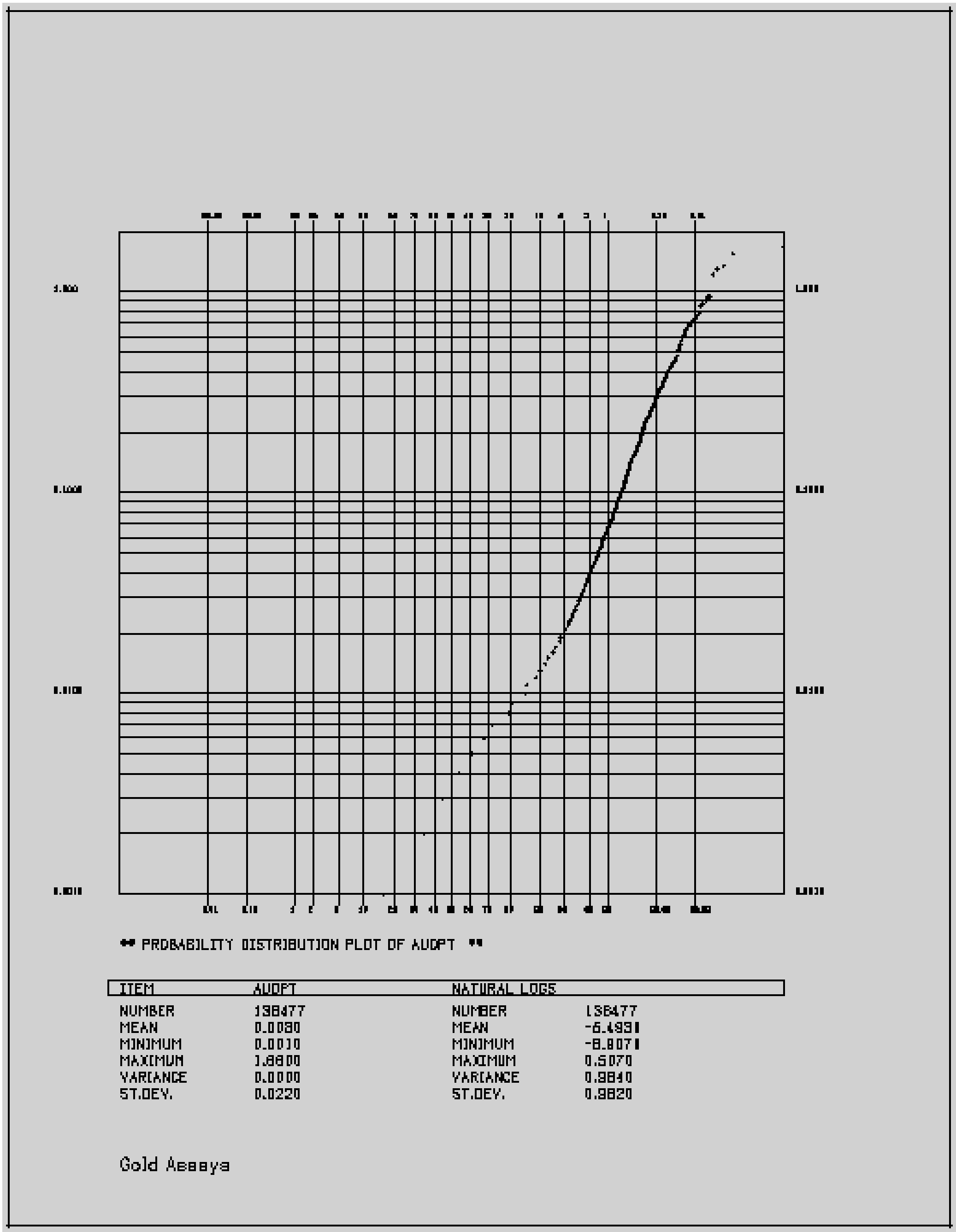


** PROBABILITY DISTRIBUTION PLOT OF CUTOT **

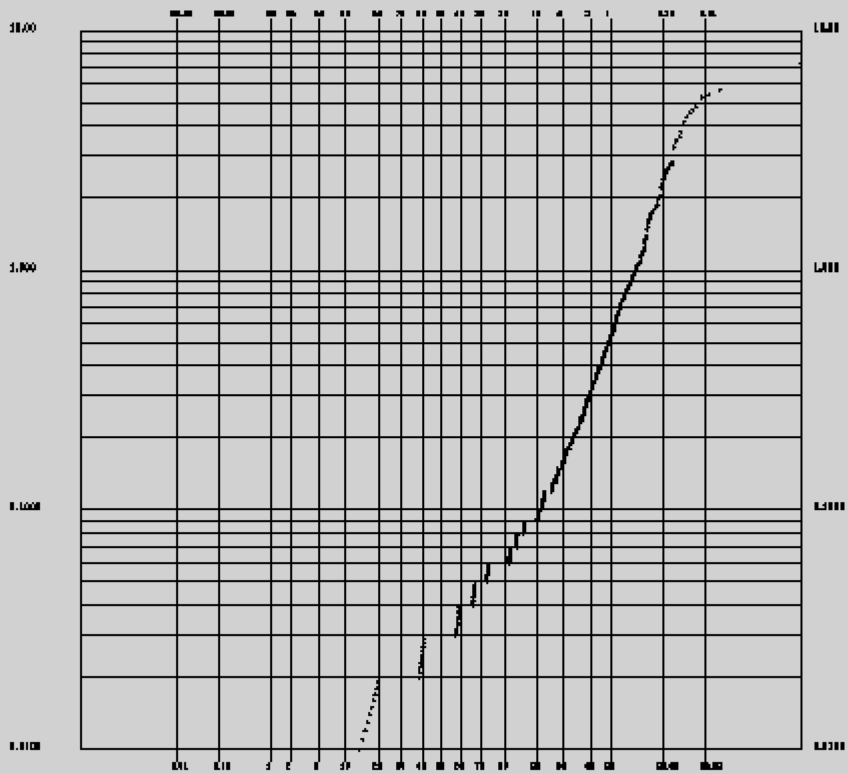
ITEM	CUTOT	NATURAL LOGS	
NUMBER	8872	NUMBER	8872
MEAN	0.1380	MEAN	-3.0821
MINIMUM	0.0100	MINIMUM	-4.6051
MAXIMUM	8.2800	MAXIMUM	2.1110
VARIANCE	0.1740	VARIANCE	1.6710
ST.DEV.	0.4170	ST.DEV.	1.2930

Total Copper, Zone 4

Ruth Deposit
Cumulative Probability Plot, Assays, Gold



Ruth Deposit
Cumulative Probability Plot, Assays, Soluble Copper, Zone 1

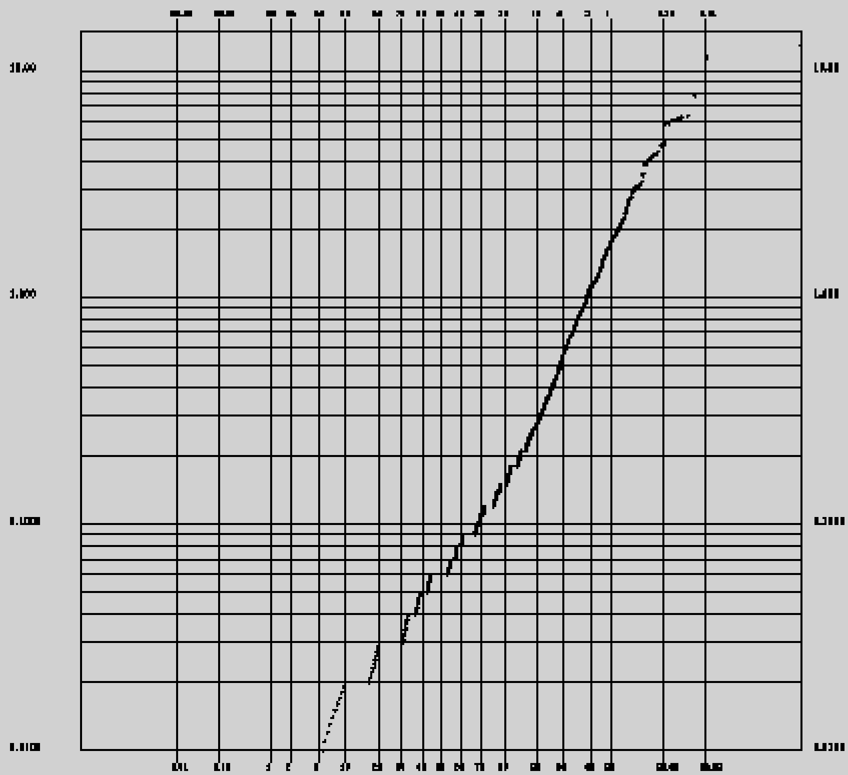


** PROBABILITY DISTRIBUTION PLOT OF CUSOL **

ITEM	CUSOL	NATURAL LOGS	
NUMBER	24203	NUMBER	24203
MEAN	0.0590	MEAN	-3.3981
MINIMUM	0.0100	MINIMUM	-4.6051
MAXIMUM	7.3200	MAXIMUM	1.9910
VARIANCE	0.0270	VARIANCE	0.7740
ST.DEV.	0.1650	ST.DEV.	0.8800

Soluable Copper Assays, Zone 1

Ruth Deposit
Cumulative Probability Plot, Assays, Soluble Copper, Zone 2

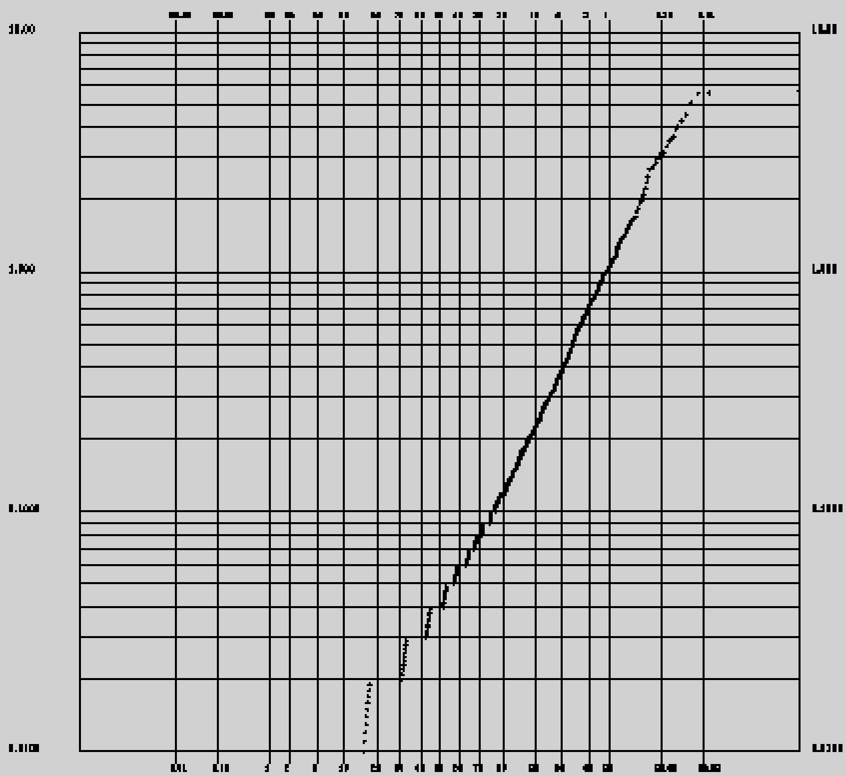


** PROBABILITY DISTRIBUTION PLOT OF CUSOL **

ITEM	CUSOL	NATURAL LOGS	
NUMBER	10884	NUMBER	10884
MEAN	0.1520	MEAN	-2.7081
MINIMUM	0.0100	MINIMUM	-4.6051
MAXIMUM	13.0000	MAXIMUM	2.5650
VARIANCE	0.1590	VARIANCE	1.2540
ST.DEV.	0.3990	ST.DEV.	1.1200

Soluable Copper Assays, Zone 2

Ruth Deposit
Cumulative Probability Plot, Assays, Soluble Copper, Zone 3

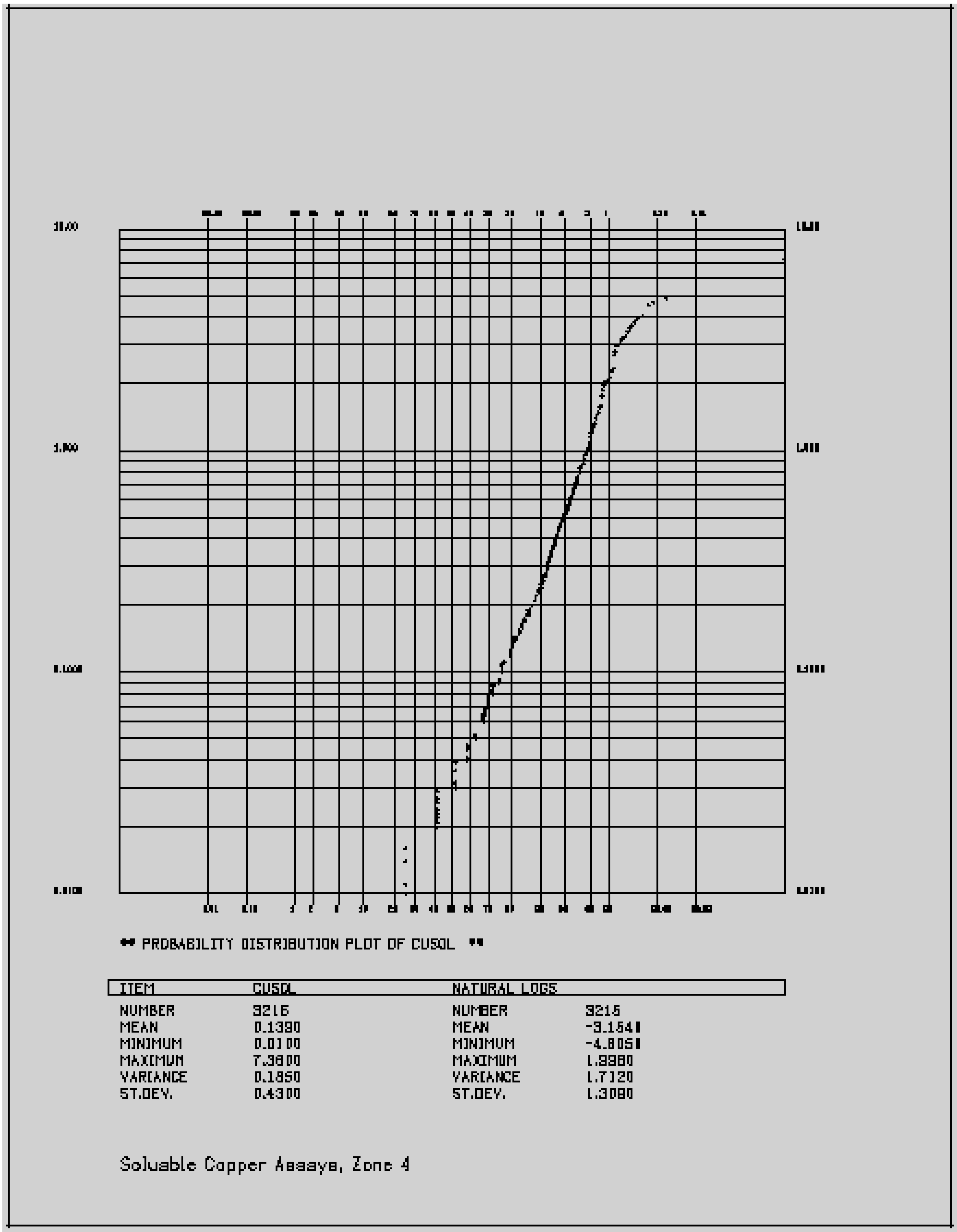


** PROBABILITY DISTRIBUTION PLOT OF CUSOL **

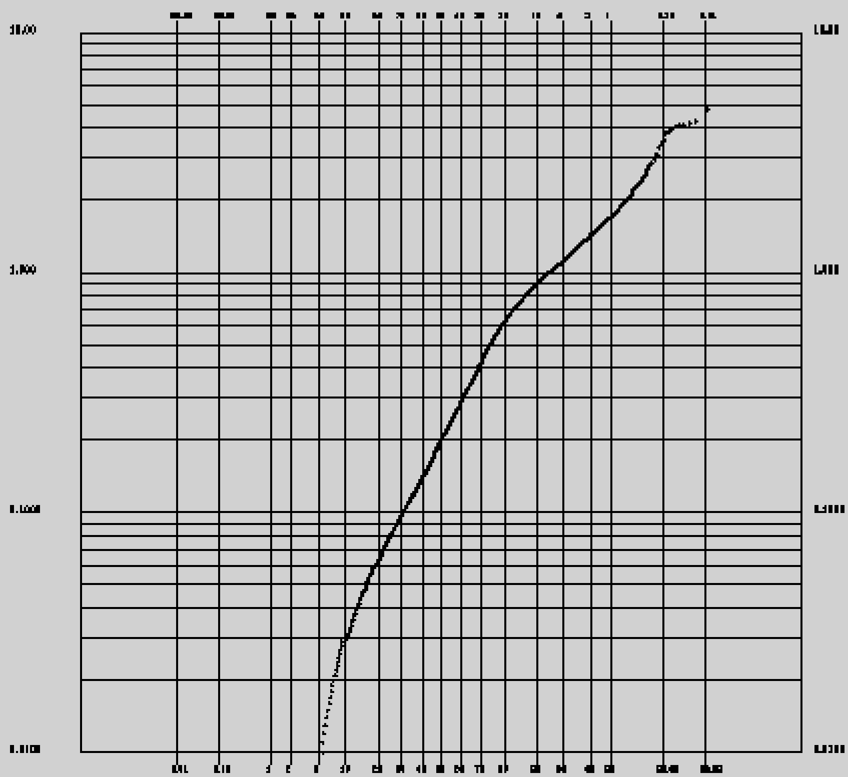
ITEM	CUSOL	NATURAL LOGS	
NUMBER	14888	NUMBER	14888
MEAN	0.1080	MEAN	-3.0481
MINIMUM	0.0100	MINIMUM	-4.6051
MAXIMUM	5.7100	MAXIMUM	1.7420
VARIANCE	0.0570	VARIANCE	1.2980
ST.DEV.	0.2400	ST.DEV.	1.1390

Soluable Copper Assays, Zone 3

Ruth Deposit
Cumulative Probability Plot, Assays, Soluble Copper, Zone 4



Ruth Deposit
Cumulative Probability Plot, Composites, Total Copper, Zone 1

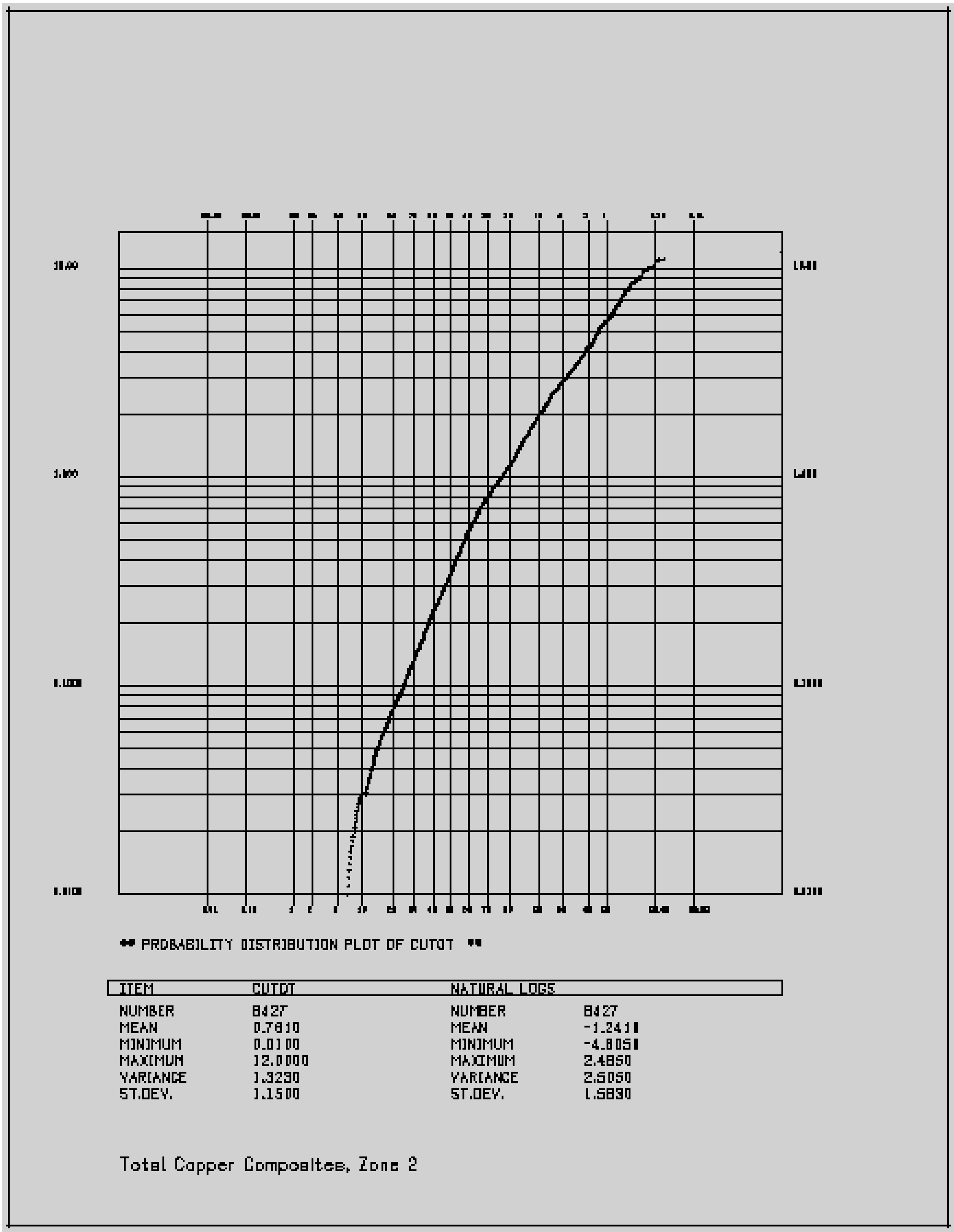


** PROBABILITY DISTRIBUTION PLOT OF CUTOT **

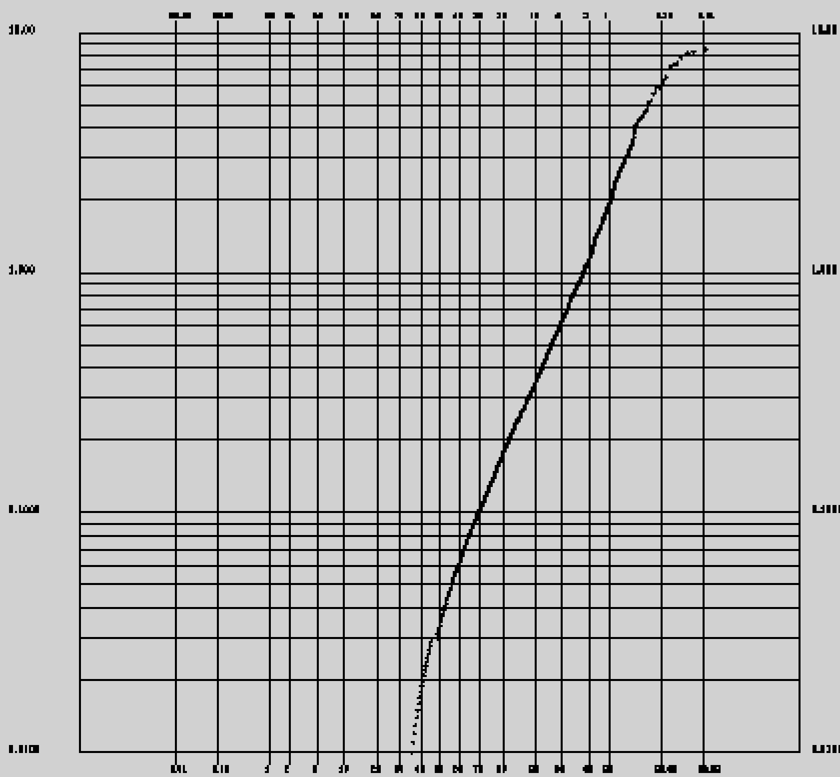
ITEM	CUTOT	NATURAL LOGS	
NUMBER	12012	NUMBER	12012
MEAN	0.3560	MEAN	-1.7041
MINIMUM	0.0100	MINIMUM	-4.6051
MAXIMUM	5.0000	MAXIMUM	1.6090
VARIANCE	0.1630	VARIANCE	1.6810
ST.DEV.	0.4040	ST.DEV.	1.2960

Total Copper Composites, Zone 1

Ruth Deposit
Cumulative Probability Plot, Composites, Total Copper, Zone 2



Ruth Deposit
Cumulative Probability Plot, Composites, Total Copper, Zone 3

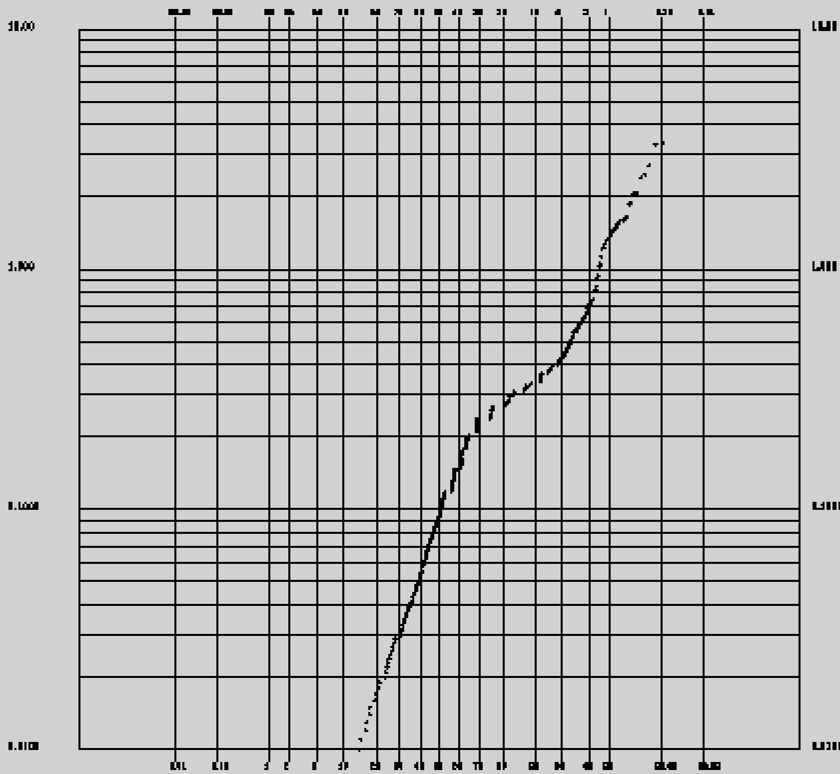


** PROBABILITY DISTRIBUTION PLOT OF CUTOT **

ITEM	CUTOT	NATURAL LOGS	
NUMBER	11885	NUMBER	11886
MEAN	0.1570	MEAN	-3.1241
MINIMUM	0.0100	MINIMUM	-4.8051
MAXIMUM	9.0000	MAXIMUM	2.1970
VARIANCE	0.2040	VARIANCE	2.1880
ST.DEV.	0.4520	ST.DEV.	1.4720

Total Copper Composites, Zone 3

Ruth Deposit
Cumulative Probability Plot, Composites, Total Copper, Zone 4

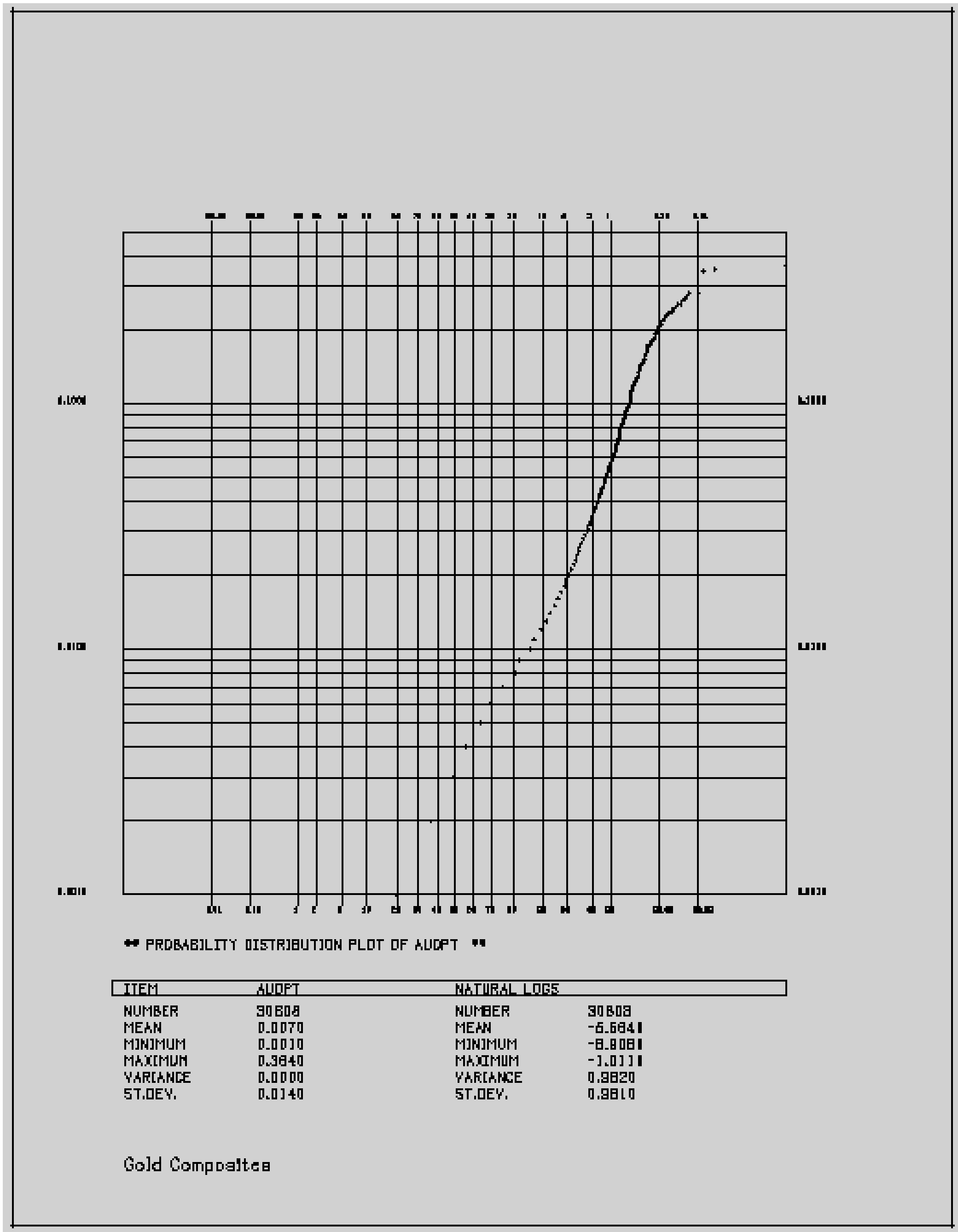


** PROBABILITY DISTRIBUTION PLOT OF CUTOT **

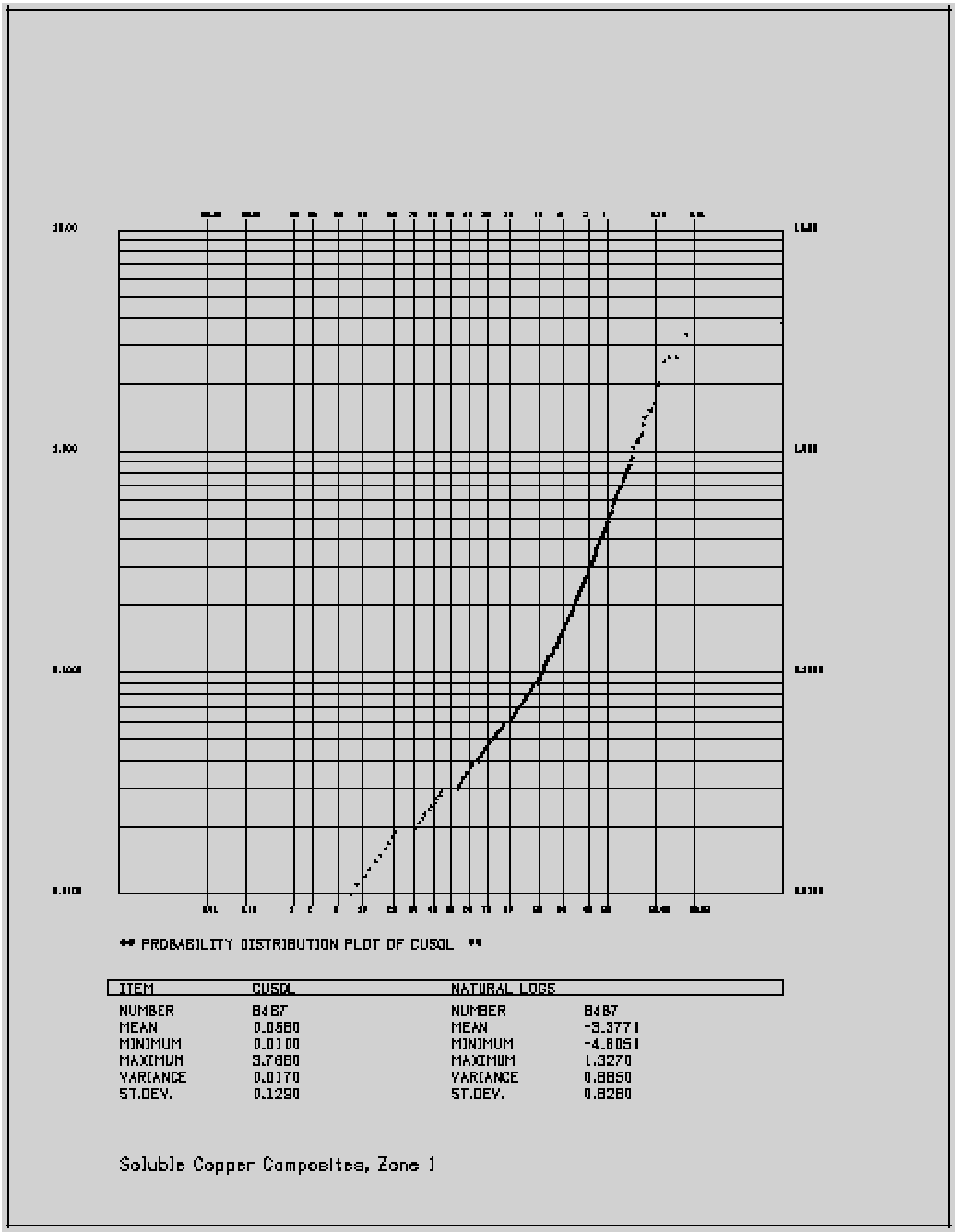
ITEM	CUTOT	NATURAL LOGS	
NUMBER	2238	NUMBER	2238
MEAN	0.1700	MEAN	-2.6501
MINIMUM	0.0100	MINIMUM	-4.6051
MAXIMUM	4.0000	MAXIMUM	1.3860
VARIANCE	0.0720	VARIANCE	1.8000
ST.DEV.	0.2680	ST.DEV.	1.3420

Total Copper Composites, Zone 4

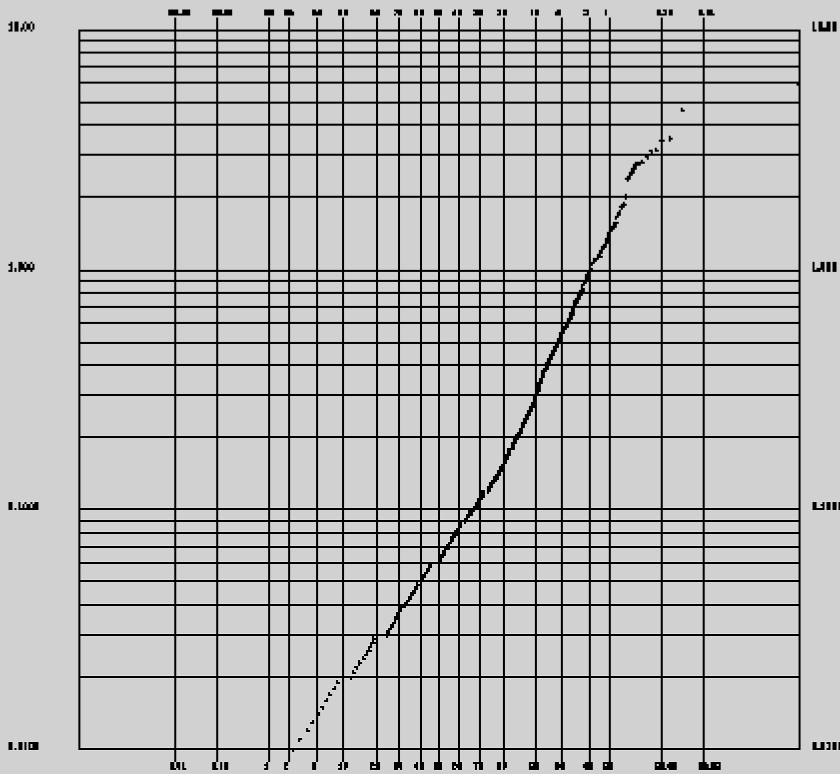
Ruth Deposit
Cumulative Probability Plot, Composites, Gold



Ruth Deposit
Cumulative Probability Plot, Composites, Soluble Copper, Zone 1



Ruth Deposit
Cumulative Probability Plot, Composites, Soluble Copper, Zone2

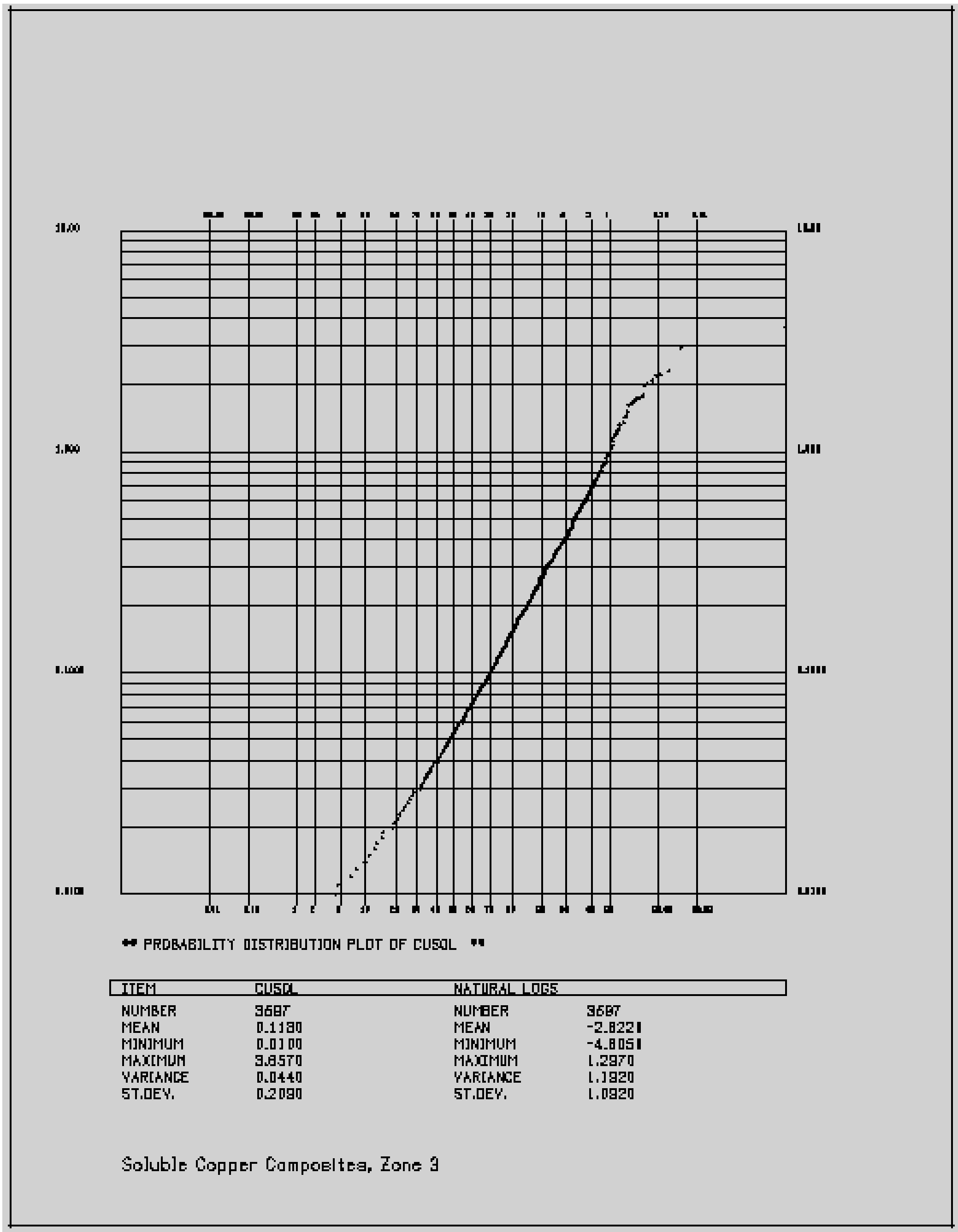


** PROBABILITY DISTRIBUTION PLOT OF CUSOL **

ITEM	CUSOL	NATURAL LOGS	
NUMBER	3073	NUMBER	3073
MEAN	0.1480	MEAN	-2.8511
MINIMUM	0.0100	MINIMUM	-4.6051
MAXIMUM	5.9840	MAXIMUM	1.7880
VARIANCE	0.0950	VARIANCE	1.1480
ST.DEV.	0.3080	ST.DEV.	1.0720

Soluble Copper Composites, Zone 2

Ruth Deposit
Cumulative Probability Plot, Composites, Soluble Copper, Zone 3



Ruth Deposit
Cumulative Probability Plot, Composites, Soluble Copper, Zone 4